STARFIELD RESOURCES INC.

PRELIMINARY ECONOMIC ASSESSMENT OF THE FERGUSON LAKE PROJECT, NUNAVUT, CANADA

NI 43-101 Report

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1 SUMMARY

EXECUTIVE SUMMARY

Roscoe Postle Associates Inc. (RPA) was retained by Starfield Resources Inc. (Starfield) to update the Preliminary Economic Assessment (PEA) of the Ferguson Lake Project (the Project) prepared by RPA (formerly Scott Wilson RPA) in December 2008. The Project is located in Nunavut, Canada. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects.

Starfield is an advanced exploration company focused on development of the Ferguson Lake nickel-copper-cobalt-platinum-palladium property. The property covers approximately 274,000 ha and has been explored with 625 diamond drill holes.

Currently, the major assets and facilities associated with the Project are:

- Mineral Resources held 100% by Starfield, subject to a 3% Net Smelter Return (NSR).
- Exploration camp on the western shore of Ferguson Lake.
- 830 m gravel airstrip adjacent to the exploration camp.
- Small scale construction and exploration support equipment on site.

The PEA is based on the following general assumptions:

- Mine production quantities of 40.3 Mt, based on Mineral Resources of the Main West Zone and the West Extension, modified by the addition of dilution and extraction factors.
- Production rate of 6,000 tonnes per day (tpd), 2.1 million tonnes per year (tpa).
- Initial mining of approximately two million tonnes from an open pit over the first two years.
- Subsequent underground mining by mechanized cut and fill methods, in both the Main West and West Extension zones simultaneously.
- Underground access to the Main West and West Extension zones by ramp and shaft, respectively.
- Backfill using waste rock material, from underground waste development, open pit waste, surface waste sources, and tailings from the concentrator.

- Concentrator at the mine site comprising crushing, grinding, and flotation circuits.

- Pumping of sulphide concentrate via a 285 km pipeline, to a process plant site, assumed to be located near Arviat, Nunavut.

- Processing at Arviat, Nunavut, comprising a two-stage leaching followed by solvent extraction (SX), ion exchange (IX) to purify the solution, then SX and electrowinning (EW) to recover the following products:
  - Nickel, copper, and cobalt as LME-grade metal
  - 93% sulphuric acid
  - Hematite
  - Gangue material

- Power generated at the Arviat plant site through burning of H_2S gas, and capture of process heat, used to power the process and, via transmission line, the mine site.

- Metal products and acid moved to market via bulk shipping.

- Gangue and hematite storage at Arviat process site by dry stacking.

Mineral Resources for this updated PEA were estimated by RPA. The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the mine schedules and economic forecasts on which this PEA is based will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

All economic forecasts in this report are as of Q4 2011, and are without escalation, unless otherwise noted.

PRELIMINARY ECONOMIC ASSESSMENT UPDATE

This report is an update of a PEA completed by RPA (formerly Scott Wilson RPA), dated December 15, 2008. The following areas were updated from the 2008 report:

- Mineral Resource estimate, as presented in Section 14.

- Update of metallurgical testwork results and revision to process design as discussed in Sections 13 and 17 respectively.

- Capital and operating cost modifications, as a result of new metallurgical testwork and process design, updates to the mine design, new labour costs, and freight assumptions.

- Cash flow update, with expanded discussion of product price sensitivities and markets.

Participants in the PEA include Michael Dry, Ph.D., process design consultant of Arithmetek Inc., Ernie Burga, P.Eng., and Phil Evans, P.Eng., of Andeburg Consulting Services (ACSI), process design consultants, and Randy Knapp, P.Eng., consultant with SENES Consultants Limited (SENES).

ECONOMIC ANALYSIS

A pre-tax Cash Flow Projection has been generated from the Life of Mine (LOM) production schedule, capital and operating cost estimates, and is summarized in Table 1-1. A summary of the key criteria is provided below. The cash flow considers the Project from the construction decision and does not include allowances for permitting, pre-feasibility, or feasibility studies.

ECONOMIC CRITERIA

PHYSICALS

- Mine life: 20 years
- Pre-production period: Three years (construction to start of production)
- Operations: 350 days per year
- Production rate:
  - Initial production from open pit of 2,000 tpd, 726,000 tpa, in Year 1
  - Combined production from open pit and underground of 6,000 tpd, 2,100,000 tpa, in Year 2
  - Production solely from underground of 6,000 tpd in Year 3 to end of mine life
- Total mill feed: 40.3 million tonnes
  - 0.61% Ni
  - 0.98% Cu
  - 0.07% Co
  - 36.3% Fe
  - 19.98%S
- Metallurgical recovery (two-stage leach and SX/IX/EW circuits):
  - 94% for Ni
  - 96% for Cu
  - 90% for Co

- Typical annual production of saleable products:
  - 25.6 million pounds of Ni
  - 41.6 million pounds of Cu
  - 2.9 million pounds of Co
  - 0.85 million tonnes of sulphuric acid
  - 0.74 million tonnes of hematite

REVENUE

- Sales prices: Base Case
  - US$10.00 per pound Ni
  - US$3.50 per pound Cu
  - US$20.00 per pound Co
  - US$85 per tonne acid
  - US$125 per tonne hematite
- Exchange rate: US$1.00 = C$1.00
- Payable products:
  - 100% for Ni
  - 100% for Cu
  - 100% for Co
  - 100% for sulphuric acid
  - 100% for hematite
  - No credit for Pt, Pd
- Sales charges:
  - US$0 for Ni (saleable cathode)
  - US$0 for Cu (saleable cathode)
  - US$0 for Co (saleable cathode)
  - US$70 per tonne (metal shipping)
  - US$84 per tonne (acid shipping)
  - US$70 per tonne (hematite shipping)
- Average NSR value: $239 per tonne
- Net Revenue by product:
  - Nickel 51.9%
  - Copper 29.5%
  - Cobalt 11.5%
  - Acid 0.2%
  - Hematite 6.9%

COSTS

- Operating costs:
  - Total mining: $43.71 per tonne milled
  - Processing: $25.31 per tonne milled
  - Site services: $ 8.26 per tonne milled
  - G&A: $ 11.16 per tonne milled
  - Total (average): $88.43 per tonne milled

- Capital costs: Life of Mine
  - Pre-production $1,387.5 million
  - Ongoing UG Mine $ 112.7 million
  - Sustaining: $ 44.2 million
  - Closure: $ 35.9 million
  - Total: $1,580.2 million

- Royalties: 2.0% NSR (after buy-out of 1% NSR)

- Average unit costs (per pound of equivalent nickel, net of byproduct credits):
  - Operating: US$3.39
  - Capital: US$1.72
  - Total: US$5.11

TABLE 1-1 PRE-TAX CASH FLOW SUMMARY
Starfield Resources Inc. - Ferguson Lake

			Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
PHYSICALS	Days Mined						350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350
Main West Open Pit	Ore Mined	tpd					2,073	2,983																		
	To Mill	'000 t	1,770				726	1,044																		
	To Dump	'000 t	1,783			300	731	1,052																		
	Ni	%	0.51				0.51	0.51																		
	Cu	%	0.82				0.82	0.82																		
	Pd	g/t	-				0.00	0.00																		
	Pt	g/t	-				0.00	0.00																		
	Co	%	0.06				0.06	0.06																		
	Fe	%	31.98				31.98	31.98																		
	S	%	16.64				16.64	16.64																		
Main West Underground	Production	tpd					-	2,143	3,301	3,301	3,299	3,301	3,301	3,299	3,300	3,300	3,300	3,300	3,299	3,299	3,300	3,301	3,301	3,301	4,280	5,092
		tpa	22,511,582				-	749,961	1,155,344	1,155,191	1,154,639	1,155,360	1,155,447	1,154,814	1,155,124	1,154,903	1,154,869	1,155,027	1,154,823	1,154,604	1,155,054	1,155,454	1,155,463	1,155,213	1,498,139	1,782,153
	Tonnes	'000 t	22,512				-	750	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,498	1,782
	Ni	%	0.59				-	0.67	0.61	0.61	0.62	0.60	0.61	0.59	0.52	0.51	0.52	0.63	0.60	0.54	0.56	0.63	0.67	0.64	0.54	0.56
	Cu	%	0.88				-	1.02	1.18	1.10	1.03	1.07	1.00	0.82	0.68	0.66	0.63	0.77	0.69	0.72	0.80	0.99	1.13	0.89	0.73	0.86
	Pd	g/t	-																							
	Pt	g/t	-																							
	Co	%	0.07				-	0.08	0.07	0.07	0.07	0.07	0.07	0.07	0.06	0.06	0.06	0.08	0.07	0.07	0.07	0.07	0.07	0.08	0.06	0.07
	Fe	%	35.01				-	39.35	36.57	36.84	37.16	36.42	36.87	35.88	32.74	32.24	32.81	35.05	33.76	31.68	32.49	36.64	39.32	37.64	32.96	32.34
	S	%	19.17				-	21.96	20.08	20.25	20.47	19.96	20.26	19.61	17.53	17.22	17.54	19.60	18.60	17.14	17.68	20.28	21.94	20.90	17.85	17.62
West Extension Underground	Production	tpd					-	899	2,701	2,699	2,701	2,700	2,699	2,701	2,699	2,699	2,700	2,699	2,699	2,699	2,701	2,699	2,699	2,699	1,719	-
		tpa	16,033,618				-	314,707	945,303	944,798	945,210	945,050	944,698	945,374	944,629	944,508	945,106	944,530	944,541	944,545	945,312	944,616	944,503	944,595	601,594	-
	Tonnes	'000 t	16,034				-	315	945	945	945	945	945	945	945	945	945	945	945	945	945	945	945	945	602	-
	Ni	%	0.66				-	0.67	0.62	0.58	0.71	0.68	0.73	0.59	0.63	0.60	0.69	0.72	0.70	0.77	0.67	0.61	0.61	0.61	0.64	-
	Cu	%	1.14				-	1.27	1.12	1.14	1.20	1.39	1.30	0.99	1.23	1.08	1.01	1.55	1.30	1.29	1.05	0.90	0.75	0.92	0.99	-
	Pd	g/t	-																							
	Pt	g/t	-																							
	Co	%	0.08				-	0.08	0.07	0.07	0.09	0.08	0.09	0.07	0.07	0.07	0.09	0.09	0.09	0.09	0.08	0.07	0.07	0.07	0.08	-
	Fe	%	38.64				-	39.32	37.22	35.52	40.90	39.46	41.48	35.63	37.74	36.63	39.99	40.91	40.33	42.86	39.20	36.83	36.85	37.05	37.79	-
	S	%	21.48				-	21.91	20.47	19.35	23.02	22.02	23.42	19.48	20.82	20.06	22.38	23.05	22.64	24.44	21.87	20.26	20.23	20.36	20.92	-
							34%																			85%
Total Mill Feed	Production	tpd					2,073	6,025	6,002	6,000	6,000	6,001	6,000	6,001	5,999	5,998	6,000	5,999	5,998	5,998	6,001	6,000	6,000	5,999	5,999	5,092
		tpa	40,314,779				725,578	2,108,668	2,100,646	2,099,989	2,099,849	2,100,411	2,100,145	2,100,188	2,099,753	2,099,411	2,099,975	2,099,557	2,099,363	2,099,148	2,100,365	2,100,070	2,099,966	2,099,808	2,099,734	1,782,153
	Tonnes	'000 t	40,315				726	2,109	2,101	2,100	2,100	2,100	2,100	2,100	2,100	2,099	2,100	2,100	2,099	2,099	2,100	2,100	2,100	2,100	2,100	1,782
	Ni	%	0.61				0.51	0.59	0.61	0.60	0.66	0.63	0.66	0.59	0.57	0.55	0.59	0.67	0.65	0.65	0.61	0.62	0.64	0.63	0.57	0.56
	Cu	%	0.98				0.82	0.96	1.15	1.12	1.11	1.22	1.13	0.89	0.93	0.85	0.80	1.12	0.96	0.97	0.92	0.95	0.96	0.90	0.81	0.86
	Pd	g/t	-				0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Pt	g/t	-				0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Co	%	0.07				0.06	0.07	0.07	0.07	0.08	0.07	0.08	0.07	0.07	0.06	0.07	0.08	0.08	0.08	0.07	0.07	0.07	0.07	0.07	0.07
	Fe	%	36.32				31.98	35.69	36.87	36.25	38.84	37.79	38.95	35.77	34.99	34.21	36.04	37.68	36.72	36.71	35.51	36.73	38.21	37.37	34.34	32.34
	S	%	19.98				16.64	19.32	20.26	19.85	21.61	20.89	21.69	19.55	19.01	18.50	19.72	21.15	20.42	20.42	19.57	20.27	21.17	20.66	18.73	17.62
METAL RECOVERIES	Ni	%	94%			94.1%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%
	Cu	%	96%			95.8%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%
	Pd	%	90%			90.0%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
	Pt	%	90%			90.0%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
	Co	%	90%			89.8%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
					Payable	Deduct																				
PAYABLE METAL	Ni	'000 lbs	511,667		100%		7,643	25,864	26,615	25,999	28,816	27,635	28,932	25,632	24,730	23,964	25,848	29,208	28,091	28,113	26,533	27,148	28,075	27,434	24,827	20,560
	Cu	'000 lbs	832,206		100%		12,563	42,649	51,112	49,470	49,092	53,932	50,327	39,640	41,101	37,716	35,660	49,781	42,785	43,125	40,624	42,032	42,391	40,091	35,748	32,370
	Pd	oz	-		0%		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Pt	oz	-		0%		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Co	'000 lbs	57,462		100%		829	2,834	2,951	2,897	3,235	3,086	3,159	2,905	2,753	2,700	3,021	3,442	3,331	3,162	2,963	2,978	2,982	3,105	2,806	2,324
PRODUCTS	Nickel	'000 lbs	511,667		100%	Ni metal	7,643	25,864	26,615	25,999	28,816	27,635	28,932	25,632	24,730	23,964	25,848	29,208	28,091	28,113	26,533	27,148	28,075	27,434	24,827	20,560
	Copper	'000 lbs	832,206		100%	Cu metal	12,563	42,649	51,112	49,470	49,092	53,932	50,327	39,640	41,101	37,716	35,660	49,781	42,785	43,125	40,624	42,032	42,391	40,091	35,748	32,370
	Pd	oz	-																							
	Pt	oz	-																							
	Cobalt	'000 lbs	57,462		100%	Co metal	829	2,834	2,951	2,897	3,235	3,086	3,159	2,905	2,753	2,700	3,021	3,442	3,331	3,162	2,963	2,978	2,982	3,105	2,806	2,324
	Sulphuric Acid	tonnes	17,026,812				305,260	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	752,965
	Elemental Sulphur	tonnes	-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Hematite	tonnes	14,706,868				263,668	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	650,372
	PGE Residue	tonnes	4,677,541				83,860	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	206,852
					24-Nov-11	Oct 2011 memo																				
SALES PRICES	Exchange Rate	C$/US$			1.00	1.00																				
	Acid	US$/t	$ 85.00		$ 85.00	80.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00
	Ni	US$ / lb	$ 10.00		$ 10.00	9.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00
	Cu	US$ / lb	$ 3.50		$ 3.50	3.15	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50
	Co	US$ / lb	$ 20.00		$ 20.00	17.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00
	Hematite	US$/t	$ 125.00		$ 125.00	100.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00

			Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
GROSS REVENUE	Acid	C$'000	$ 1,447,279				$ 25,947	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 64,002
	Ni	C$'000	$ 5,116,674				$ 76,428	$ 258,637	$ 266,146	$ 259,990	$ 288,162	$ 276,354	$ 289,324	$ 256,319	$ 247,301	$ 239,639	$ 258,485	$ 292,082	$ 280,914	$ 281,132	$ 265,327	$ 271,476	$ 280,754	$ 274,344	$ 248,265	$ 205,596
	Cu	C$'000	$ 2,912,722				$ 43,969	$ 149,271	$ 178,890	$ 173,146	$ 171,821	$ 188,761	$ 176,146	$ 138,739	$ 143,853	$ 132,006	$ 124,809	$ 174,233	$ 149,749	$ 150,936	$ 142,182	$ 147,111	$ 148,367	$ 140,319	$ 125,117	$ 113,295
	Co	C$'000	$ 1,149,243				$ 16,585	$ 56,682	$ 59,012	$ 57,938	$ 64,693	$ 61,710	$ 63,179	$ 58,097	$ 55,068	$ 54,001	$ 60,415	$ 68,843	$ 66,614	$ 63,246	$ 59,268	$ 59,556	$ 59,636	$ 62,106	$ 56,113	$ 46,481
	Hematite	C$'000	$ 1,838,358				$ 32,958	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 81,296
	TOTAL	**C$'000**	**$ 12,464,276**				**$ 195,888**	**$ 635,781**	**$ 675,239**	**$ 662,265**	**$ 695,867**	**$ 698,016**	**$ 699,840**	**$ 624,346**	**$ 617,413**	**$ 596,836**	**$ 614,899**	**$ 706,348**	**$ 668,468**	**$ 666,505**	**$ 637,968**	**$ 649,333**	**$ 659,949**	**$ 647,959**	**$ 600,686**	**$ 510,670**
				% Moisture																					
SALES CHARGES	Acid Shipping	C$'000	$ 1,430,252		$ 84.00 US$/t		$ 25,642	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 63,249
	Metal Shipping	C$'000	$ 44,495	0%	$ 70.00 US$/t		$ 668	$ 2,265	$ 2,562	$ 2,488	$ 2,576	$ 2,688	$ 2,617	$ 2,165	$ 2,178	$ 2,044	$ 2,049	$ 2,617	$ 2,356	$ 2,362	$ 2,226	$ 2,291	$ 2,332	$ 2,243	$ 2,012	$ 1,754
	Hematite Shipping	C$'000	$ 1,163,313	13%	$ 70.00 US$/t		$ 20,856	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 51,444
	TOTAL	C$'000	**$ 2,638,060**				**$ 47,166**	**$ 137,397**	**$ 137,694**	**$ 137,620**	**$ 137,708**	**$ 137,820**	**$ 137,749**	**$ 137,297**	**$ 137,310**	**$ 137,176**	**$ 137,181**	**$ 137,749**	**$ 137,488**	**$ 137,494**	**$ 137,358**	**$ 137,423**	**$ 137,464**	**$ 137,375**	**$ 137,144**	**$ 116,448**
NET SMELTER RETURN	NSR	C$'000	$ 9,826,216				$ 148,722	$ 498,384	$ 537,546	$ 524,645	$ 558,159	$ 560,197	$ 562,091	$ 487,049	$ 480,103	$ 459,660	$ 477,718	$ 568,599	$ 530,980	$ 529,011	$ 500,609	$ 511,910	$ 522,485	$ 510,584	$ 463,542	$ 394,222
	Royalty	C$'000	$ 196,524		2%		$ 2,974	$ 9,968	$ 10,751	$ 10,493	$ 11,163	$ 11,204	$ 11,242	$ 9,741	$ 9,602	$ 9,193	$ 9,554	$ 11,372	$ 10,620	$ 10,580	$ 10,012	$ 10,238	$ 10,450	$ 10,212	$ 9,271	$ 7,884
	NSR after Royalty	**C$'000**	**$ 9,629,692**				**$ 145,748**	**$ 488,416**	**$ 526,795**	**$ 514,152**	**$ 546,995**	**$ 548,993**	**$ 550,850**	**$ 477,308**	**$ 470,501**	**$ 450,467**	**$ 468,164**	**$ 557,227**	**$ 520,360**	**$ 518,431**	**$ 490,597**	**$ 501,672**	**$ 512,035**	**$ 500,373**	**$ 454,271**	**$ 386,337**
	Unit	**C$/t milled**	**$ 238.86**				**$ 201**	**$ 232**	**$ 251**	**$ 245**	**$ 260**	**$ 261**	**$ 262**	**$ 227**	**$ 224**	**$ 215**	**$ 223**	**$ 265**	**$ 248**	**$ 247**	**$ 234**	**$ 239**	**$ 244**	**$ 238**	**$ 216**	**$ 217**
					Unit Rates																				
OPERATING COSTS	Open Pit Mining	C$'000	$ 13,618		3.83 C$/t moved		$ 5,584	$ 8,034	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	UG Mining	C$'000	$ 1,748,534		45.36 C$/t			$88,406	$113,563	$96,751	$98,538	$94,304	$95,633	$96,052	$91,074	$89,207	$89,219	$89,210	$89,206	$89,202	$89,226	$89,220	$89,218	$89,215	$89,214	$82,073
	Process	C$'000	$ 1,020,246		25.31 C$/t		$ 18,362	$ 53,364	$ 53,161	$ 53,144	$ 53,141	$ 53,155	$ 53,148	$ 53,149	$ 53,138	$ 53,130	$ 53,144	$ 53,134	$ 53,129	$ 53,123	$ 53,154	$ 53,146	$ 53,144	$ 53,140	$ 53,138	$ 45,101
	Site Services	C$'000	$ 333,025		8.26 C$/t		$ 5,994	$ 17,419	$ 17,353	$ 17,347	$ 17,346	$ 17,351	$ 17,349	$ 17,349	$ 17,345	$ 17,342	$ 17,347	$ 17,344	$ 17,342	$ 17,340	$ 17,350	$ 17,348	$ 17,347	$ 17,346	$ 17,345	$ 14,722
	G&A	C$'000	$ 449,765		11.16 C$/t		$ 8,095	$ 23,525	$ 23,436	$ 23,428	$ 23,427	$ 23,433	$ 23,430	$ 23,430	$ 23,426	$ 23,422	$ 23,428	$ 23,423	$ 23,421	$ 23,419	$ 23,432	$ 23,429	$ 23,428	$ 23,426	$ 23,425	$ 19,882
	TOTAL	**C$'000**	**$ 3,565,189**				**$ 38,035**	**$ 190,748**	**$ 207,513**	**$ 190,671**	**$ 192,452**	**$ 188,243**	**$ 189,560**	**$ 189,981**	**$ 184,983**	**$ 183,101**	**$ 183,138**	**$ 183,111**	**$ 183,098**	**$ 183,084**	**$ 183,163**	**$ 183,144**	**$ 183,137**	**$ 183,127**	**$ 183,122**	**$ 161,778**
		C$/t milled	**$ 88.43**				**$ 52.42**	**$ 90.46**	**$ 98.79**	**$ 90.80**	**$ 91.65**	**$ 89.62**	**$ 90.26**	**$ 90.46**	**$ 88.10**	**$ 87.22**	**$ 87.21**	**$ 87.21**	**$ 87.22**	**$ 87.22**	**$ 87.21**	**$ 87.21**	**$ 87.21**	**$ 87.21**	**$ 87.21**	**$ 90.78**
	Cash Cost (net of byproduct credits)	**US$/lb Ni Eq.**	**$ 3.39**				**$ 2.14**	**$ 3.59**	**$ 3.66**	**$ 3.41**	**$ 3.24**	**$ 3.15**	**$ 3.16**	**$ 3.66**	**$ 3.60**	**$ 3.72**	**$ 3.58**	**$ 3.01**	**$ 3.22**	**$ 3.23**	**$ 3.41**	**$ 3.34**	**$ 3.27**	**$ 3.35**	**$ 3.69**	**$ 3.85**
OPERATING CASH FLOW		**US$'000**	**$ 6,064,502**				**$ 107,713**	**$ 297,668**	**$ 319,282**	**$ 323,480**	**$ 354,544**	**$ 360,750**	**$ 361,289**	**$ 287,328**	**$ 285,518**	**$ 267,365**	**$ 285,026**	**$ 374,116**	**$ 337,262**	**$ 335,346**	**$ 307,434**	**$ 318,528**	**$ 328,898**	**$ 317,246**	**$ 271,149**	**$ 224,559**
CAPITAL COSTS	Mine	C$'000	$ 151,358	$ -	$ 24,863	$ 59,790	$ 66,705	$ -	$ -																	
	Process	C$'000	$ 788,068	$ 2,649	$ 467,732	$ 317,687	$ -	$ -	$ -																	
	Infrastructure	C$'000	$ 94,490	$ 15,758	$ 44,308	$ 34,423	$ -	$ -	$ -																	
	Indirects	C$'000	$ 269,684	$ 26,315	$ 105,259	$ 111,795	$ 26,315	$ -	$ -																	
	Contingency	C$'000	$ 196,551	$ 19,655	$ 78,620	$ 78,620	$ 19,655	$ -	$ -																	
	Reclamation	C$'000	$ 35,889																							$ 35,889
	Sustaining Capital	C$'000	$ 44,194					$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	
	TOTAL	**C$'000**	**$ 1,580,232**	**$ 64,377**	**$ 720,783**	**$ 602,316**	**$ 112,674**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 35,889**
NET PRE-TAX CASH FLOW	**Per Year**	**C$'000**	**4,484,270**	**(64,377)**	**(720,783)**	**(602,316)**	**(4,961)**	**295,212**	**316,827**	**321,025**	**352,088**	**358,295**	**358,834**	**284,873**	**283,063**	**264,910**	**282,571**	**371,661**	**334,806**	**332,891**	**304,979**	**316,073**	**326,443**	**314,790**	**268,694**	**188,671**
	Cumulative	**C$'000**		**(64,377)**	**(785,160)**	**(1,387,476)**	**(1,392,437)**	**(1,097,224)**	**(780,398)**	**(459,372)**	**(107,284)**	**251,011**	**609,845**	**894,718**	**1,177,780**	**1,442,691**	**1,725,262**	**2,096,923**	**2,431,729**	**2,764,620**	**3,069,599**	**3,385,672**	**3,712,115**	**4,026,905**	**4,295,599**	**4,484,270**
Payback	Years	**5.3**	**5.3**				1.0	1.0	1.0	1.0	1.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Weeks	**276**	**276**				52	52	52	52	52	16	0	0	0	0	0	0	0	0	0	0	0	0	0	0
UNIT PRODUCTION COSTS	Operating (net of byproduct credits)	US$/lb Ni					2.14	3.59	3.66	3.41	3.24	3.15	3.16	3.66	3.60	3.72	3.58	3.01	3.22	3.23	3.41	3.34	3.27	3.35	3.69	3.85
	Life of Mine Capital	US$/lb Ni																								
	TOTAL	US$/lb Ni																								
	IRR			**16.8%**																						
NET PRESENT VALUE	**C$ Millions**	**5%**		**$1,872**																						
		8%		**$1,061**																						
		10%		**$692**																						
		15%		**$123**																						

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CASH FLOW ANALYSIS

Considering the Project on a stand-alone basis, the undiscounted pre-tax cash flow is $4,484 million over the 20 year mine life. Operating cash flow averages $303 million per year, and simple payback of pre-production capital is achieved after 5.3 years of operation.

The pre-tax Internal Rate of Return (IRR) is 16.8%. Pre-tax Net Present Value (NPV) of the project at various discount rates is as follows:

- Pre-tax NPV @ 5% $1,872 million
- Pre-tax NPV @ 8% $1,061 million
- Pre-tax NPV @ 10% $ 692 million
- Pre-tax NPV @ 15% $ 123 million

In RPA's opinion, a discount rate of 10% is appropriate for Base Case evaluation at this stage of the Project. The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the mine schedules and economic forecasts on which this report is based will be realized.

CURRENT PRICE CASE

The Project evaluated at current metal prices and exchange rate (November 22, 2011) yields an undiscounted pre-tax cash flow of $3.3 billion, and IRR of 13.6%. The following sales prices and exchange rate were used:

- US$8.05 per pound Ni
- US$3.36 per pound Cu
- US$14.50 per pound Co
- US$100 per tonne hematite
- US$100 per tonne acid
- US$/C$ = 1.05

At the following discount rates, the pre-tax NPV using current prices and exchange is as follows:

- Pre-tax NPV @ 5% $1,286 million

- Pre-tax NPV @ 8% $642 million
- Pre-tax NPV @ 10% $352 million
- Pre-tax NPV @ 15% ($90) million

Current prices are lower than those projected for the LOM cash flow estimates.

SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities to head grades, product prices, operating costs, capital costs, and exchange rate. The results are shown in Figure 1-1 and Table 1-2. The cash flow was estimated over a +20% to -20% variation from the Base Case for each variable.

The Project cash flow is most sensitive to changes in the product prices, followed by exchange rate, head grade, capital cost, and operating cost.

FIGURE 1-1 SENSITIVITY ANALYSIS

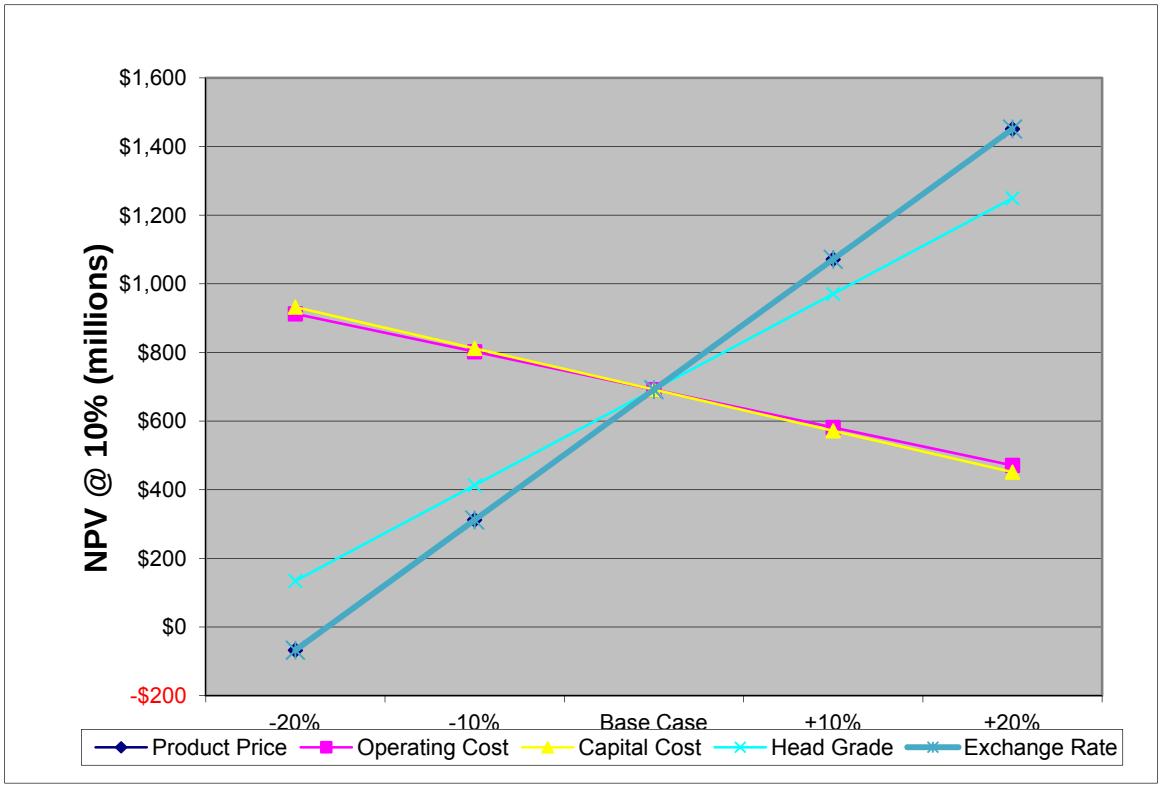

TABLE 1-2 SENSITIVITY ANALYSES
Starfield Resources Inc – Ferguson Lake

Parameter Variables	Units	-20%	-10%	Base	+10%	+20%
Product Price	US$/lb Ni	8.0	9.0	10.00	11.0	12.0
Exchange Rate	US$/C$	0.8	0.9	1.00	1.10	1.20
Head Grade	% Ni	0.49	0.55	0.61	0.67	0.73
Operating Cost	$/t	70.75	79.59	88.43	97.28	106.12
Capital Cost	$ millions	1,264	1,422	1,580	1,738	1,896

Pre-Tax NPV @ 10%	Units	-20%	-10%	Base	+10%	+20%
Product Price	$ millions	-67	312	692	1,072	1,451
Exchange Rate	$ millions	-67	312	692	1,072	1,451
Head Grade	$ millions	134	413	692	971	1,249
Operating Cost	$ millions	913	802	692	581	471
Capital Cost	$ millions	932	811	692	572	452

RPA notes that while Table 1-2 shows variations to product price in relation to nickel only, sensitivity factors were applied to all payable products. An expanded sensitivity analysis for the Project is presented in Section 22 of this report.

INTERPRETATION AND CONCLUSIONS

RPA reviewed the Project at the level of a PEA and concludes that under the Base Case assumptions, the Project has potential economic viability. In RPA's opinion, Starfield should continue work to advance the Project, by gathering information and undertaking studies in advance of completing a Pre-feasibility Study followed by a Feasibility Study.

Separate comments on the various aspects of the Project follow.

RESOURCES

Mineral Resources estimated for the Project include an Indicated Resource of 15.8 million tonnes grading 0.65% Ni, 0.99% Cu, and 0.07% Co and an Inferred Resource of 30.2 million tonnes grading 0.67% Ni, 1.00% Cu, and 0.05% Co. Mineral Reserves have not been estimated.

Approximately 57% of the Mineral Resources included in the PEA are Inferred Resources from the Main West and West Extension zones. Infill drilling will be required to upgrade some portion of these resources to the Measured or Indicated categories.

Exploration work to date has focused on massive sulphide portions of the deposit, with early work sampling only for nickel and copper. Resource estimates for the East Zone lack cobalt and precious metal grade estimates, as a high proportion of the drilling in the zone was completed in the early programs. The East Zone was not included in the mine schedule for this PEA.

Recent drilling programs have tested low-sulphide and/or base metal, high-platinum group element (PGE) potential, largely located below the massive sulphide lenses, with results that indicate the presence of potentially economic quantities of PGEs. The PGEs are not considered in the present Project economics.

MINING

Mechanized cut and fill mining is proposed to provide the best balance between productivity, required to achieve the production rate of 6,000 tpd, and selectivity, required to minimize dilution in the shallow-dipping mineralized lenses.

The significant horizontal extent of the deposit and the relatively narrow lenses will require a large quantity of level and internal ramp development to provide sufficient cut and fill faces for a continuous 6,000 tpd operation. In RPA's opinion, the proposed production rate is in the upper range of achievable rates, given the deposit geometry as presently understood.

The underground design parameters in this report are preliminary and further information is required for additional study and evaluation. Some of the key areas to be considered are:

- Data on geothermal gradient and rock temperatures
- Information on stability of rock when permafrost thaws
- Groundwater conditions (if any)
- Geotechnical information to assess stability of openings

There is potential to extend the mine life in future studies, by inclusion of the East Zone or other mineralized areas on the property.

PROCESSING AND TESTWORK

The recommended process will consist of:

- A concentrator producing a bulk sulphide concentrate at the mine site
- A concentrate slurry pipeline between the mine and the hydrometallurgical plant
- A hydrometallurgical plant located in Arviat

Testwork that was carried out from 2001 to 2008 by Kingston Process Metallurgy, SGS Minerals Services (SGS Lakefield), and NeoFerric Technologies Inc. (NeoFerric) has generated a flowsheet on which to assess the economic viability of the Project at the scoping level. Additional testwork was completed in 2009 at McGill University and in 2010 at SGS Lakefield to assess oxidation and hydrolysis steps. This testwork indicated that continuous oxidation of ferrous chloride solution was successful at 120°C and produced a precipitate consisting of 97% hematite. Continuous hydrolysis of the ferric chloride solution was successful in producing a precipitate with 69% iron and 0.9% chloride and hydrochloric acid at 22% strength.

During 2011, a flotation test program was carried out to assess if the Ferguson Lake ore was amenable to upgrading by means of flotation prior to further treatment at the hydrometallurgical process plant. A series of bench scale flotation tests established that it was possible to recover essentially all the sulphide minerals into a bulk sulphide concentrate while also rejecting about 80% of the gangue minerals.

The two major continuous leaching campaigns have been successfully carried out in a continuous mini-pilot plant at SGS Lakefield. The iron oxidation and hydrolysis steps have also been successfully tested in continuous mini-plant campaigns at SGS Lakefield.

The process oxidation and hydrolysis steps are novel, and are not commercially proven unit operations. Fundamental measurements of heat capacity, thermal conductivity, viscosity, and other properties are required in order to determine the required heat transfer coefficients to size heat transfer surfaces, or preferably, direct measurements of heat transfer at a larger scale than has been completed thus far.

The oxidation step appears to be controlled by the rate at which oxygen is transferred from the gas phase to the bulk liquid phase. This implies that agitation is extremely important, and further work is required to properly scale up the oxidation step.

The process has the potential to generate stable, non-toxic tailings, and most importantly, may recover a sufficient quantity of the intrinsic energy in the ore to make it self-sustaining in energy usage. This means that there may be no need to import fuel oil for generating power on-site, which would otherwise be necessary in such a remote location. The energy recovered is potentially sufficient to operate both the hydrometallurgical process plant, as well as the mine, concentrator, and concentrate pipeline pump stations.

There are a number of uncertainties in the process flowsheet where it has been necessary to make assumptions. The most important of these are the iron and sulphur contents of the ore, and its accompanying mineralogy, and secondly, the lack of fundamental data in the scientific literature to enable, in particular, a rigorous energy balance to be carried out.

No formal account has been taken of the precious metals content of the ore, since no process for their recovery has yet been properly evaluated, however, it is clear that their recovery is mandated, even as a relatively low grade concentrate, since they will comprise a significant revenue stream.

There is also potential to process low-base metal, high-PGE material, not currently included in Mineral Resources. Low-sulphur-content material will not provide the energy advantages of the massive sulphide material, however, there may be value in considering a process unit that can recover PGEs from both low-sulphide material and from the secondary leach residue.

The process risk assessment has been updated to reflect new work since the previous risk exercise, and has been reviewed by Arithmetek, Inc. (Arithmetek) and RPA. Results of this assessment show that the majority of the process has low associated risks. Medium-risk items include the oxidation and hydrolysis components of the proposed process as these areas have not yet been scaled up to commercial plant size, however, these risks can likely be mitigated through further testwork. The iron hydrolysis and HCl

recovery circuit is a technique originally developed by PORI Inc. in the 1970s, and piloted by SMS Siemag for four years since 2002.

ACID

The Project will produce a large volume of high quality sulphuric acid that will be shipped to market over a short shipping season. In order to achieve the forecast revenue from acid, it will be important for the Project to develop a long term customer base that can handle and store the large volume of product. It will also be important to identify a source of the large-volume specialty ships required to transport the acid from the process site to market during the short shipping season window. The PEA includes acid storage facilities with a capacity of nine months per year of acid production.

ENVIRONMENT

The proposed process features significant advantages over conventional industry operations for similar projects. The process includes potential power generation from the sulphur contained in the ore, which would otherwise be generated by diesel combustion (annual requirements in the order of 100 million litres of diesel), with attendant greenhouse gas emissions.

The Project includes some environmental management issues beyond those considered typical for northern mines, such as large-scale acid storage, handling and transportation, and concentrate transportation by slurry pipeline. Ongoing investigation, examination of alternatives, and best-practice design with due consideration for the environment must remain a high priority as the Project is advanced, if permitting is to proceed smoothly.

MARKETING

The prices for certain products and particularly the sulphuric acid are difficult to assess over a long term. The acid price is specific to individual client-supplier agreements that can vary significantly from region to region. RPA has assumed an acid price based on available information, however, the actual price realized may vary significantly. Equally, acid transportation costs are difficult to determine and the cost was assumed based on recent prices that may also vary considerably over the Project life.

RECOMMENDATIONS

RPA makes the following recommendations to proceed with the assessment of the potential development of the Project.

1. Complete sampling necessary to evaluate cobalt, platinum, palladium, iron, and sulphur grades for the East Zone. This will require additional diamond drilling.

2. Conduct a sampling program to evaluate iron and sulphur grades, and mineralogy for the Main West Zone and the West Extension. A program of iron and sulphur analysis was completed for part of the Main West Zone but not for the West Extension Zone. Determine expected variations and iron and sulphur process feed grades.

3. Carry out infill drilling to upgrade Inferred Resources to Indicated Resources, and to generate material for metallurgical testing. The infill drilling has only been partially achieved but no material has been generated for additional metallurgical testing. Additional drilling is recommended in the West Zone of the property, as follows:
    a. Selected holes in the eastern part of the West Zone for a total of 2,000 m.
    b. Selected holes in the western half of the West Zone to reduce drill spacing to 30 m and to provide the basis for upgrading resources in this area to indicated. Approximately 5,000 m is estimated.

4. Carry out evaluation of other zones of mineralization on the property, including drilling on the 119 Extension Zone, to follow up previous results as follows:
    a. Two holes on section, up dip and down dip, of hole FL11-432
    b. A series of holes to test for zone development between holes FL11-432 and FL04-174
    c. Two holes on section, up dip and down dip, of hole FL04-200
    d. A series of holes to test for a potential zone southwest of hole FL04-200

5. Evaluate Mineral Resources for the East Zone and the West Extension using block modelling, once additional drill and sample results are available.

6. Carry out interpretation of the low-base metal, high-PGE mineralization and relationship with adjacent massive sulphide intercepts Drill test the interpretation and if warranted complete updated resource estimate.

7. Complete outcrop stripping, or trenching, or other development to facilitate exposure of mineralization and confirm/evaluate the geometry and continuity of massive sulphide lenses. This could be completed in conjunction with advanced Project development study.

8. Gather geotechnical information and permafrost characteristics, for use in underground mine design.

9. Complete a trade-off evaluation of shaft hoisting vs. ramp haulage for transportation of ore to surface, to determine the optimum split between the two options.

10. Evaluate mining of low-base metal, high-PGE mineralization, generally located below the massive sulphide lenses, as an addition to the current mining scenario.

11. Evaluate processing scenarios for PGE recovery from gangue material, and from low-base metal, high-PGE material, as a separate circuit within the current process.

12. Generate fundamental data (water and hydrochloric acid vapour pressure, water and chloride activity, boiling point elevation, solubility data) for more accurate process design.

13. Carry out primary and secondary leach tests at a higher temperature and/or salt concentration consistent with the output of the mass/energy balance to ensure that it is possible to do so.

14. Carry out testing for heat transfer to determine fundamental measurements of heat capacity, thermal conductivity, viscosity, etc., to determine heat transfer coefficients required to size the heat transfer surfaces or, preferably, direct measurements of heat transfer.

15. Carry out testing for materials of construction, particularly for heat transfer surfaces as the fluids are aggressive and titanium or tantalum will be required.

16. Complete additional work for the oxidation step, as agitation will be extremely important to ensure this step is properly scaled up.

17. Audit the mass/energy balance to ensure the most efficient use of combustion gas energy.

18. Test methods for concentrating the platinum and palladium content of the secondary leach residue. Include possibility of concurrent processing of low-sulphide, high-PGE feed.

19. Run a pilot plant program, integrating all process units into a continuous operation.

20. Undertake a detailed marketing study to identify specific customers for the sulphuric acid product from the Project.

21. Carry out a transport and logistics study to identify carriers capable of meeting acid shipping capacity requirements.

22. Evaluate logistics for movement of materials to site for construction and operation.

23. Continue baseline environmental programs.

24. Review comparable Arctic pipelines, to assess best practices for design and operation.

25. Continue to investigate alternative process site locations and slurry pipeline routes.

26. Continue discussions for an impact benefit agreement (IBA) with local communities in the Project area.

TECHNICAL SUMMARY

PROPERTY DESCRIPTION AND LOCATION

The Ferguson Lake Project currently consists of 273 mineral claims and 10 mining leases covering an area of 274,211.6 ha (677,590.6 acres) in the Kivalliq region of southern Nunavut Territory some 240 km west of Rankin Inlet and 160 km south-southwest of Baker Lake. Ferguson Lake, central to the large property area, is midway between Yathkyed and Kaminuriak Lakes. The property currently measures approximately 90 km in an east-west direction and approximately 50 km north-south.

LAND TENURE

All of the mineral claims are contiguous and extend east, west, south, and northwest of Ferguson Lake between approximate UTM coordinates 6,940,000 mN and 6,990,000 mN, 565,000 mE and 655,000 mE, Zone 14.

Land use permits, including an Exploration Permit, Right of Way for winter transport, a Water Licence, and a Commercial Lease, enable exploration work to be conducted over the entire property. These permits have been issued or renewed by the Kivalliq Inuit Association (KIA) for parts of the property covering Inuit-owned lands.

Starfield owns a 100% interest in the mineral claims comprising the Project, subject to a 3% NSR on potential future mineral production. Starfield has the right to purchase 1% of the NSR for $1 million for a period of 180 days following receipt of a positive feasibility study recommending commercial production, and the PEA cash flow model includes the assumption that this purchase has been completed.

HISTORY

Copper-nickel mineralization at Ferguson Lake was discovered by Inco in 1950. A 3,000 km^2 concession was granted in late 1950, and work over the ensuing five years included construction of a 90-person all-season camp, airborne and surface geophysics, geological mapping, and diamond drilling.

Much of the Inco drilling was directed to mineralized zones east and west of Ferguson Lake and the intervening area beneath the lake (East Zone, West Zone, and Central or Lake Zone). The remainder of the drilling tested other targets within and outside the original concession area. A 10-ton bulk sample, extracted from the West Zone in 1953, was transported to Copper Cliff, Ontario, for mill testing.

Homestake Mineral Development Company (Homestake) acquired claims and prospecting permits around the existing Inco mining lease in 1986. A comprehensive exploration program in 1987 consisted of reconnaissance geological mapping and collection of 339 rock and 266 soil samples, mainly from the known East and West mineral zones. Homestake's mineral claims in the area subsequently lapsed, and the Inco mining lease expired on June 17, 1992.

In 1999, Starfield optioned the property and completed airborne and surface geophysical surveys, detailed geological mapping, prospecting and surface sampling, preliminary environmental baseline studies and several diamond drilling programs. Diamond drilling completed by Starfield at Ferguson Lake between 1999 and 2005 amounted to more than 108,000 m in 251 holes. Metallurgical testwork on a variety of processing options was carried out during this time.

GEOLOGY

REGIONAL GEOLOGY

The Project is situated in the Western Churchill Province, within the northwestern Hearne domain, which is made up principally of Archean metavolcanic and metasedimentary rocks and extensive gneissic terranes. The widespread Archean gneissic rocks are intruded by Archean granodiorites, quartz monzonites and a variety of mafic intrusions including diorites and gabbros. Late Archean intrusions include the east- to northeast-trending Kazan dykes, which consist of variably metamorphosed gabbros and hornblendites.

Early Proterozoic gabbros and slightly younger diabase dykes cut all older rocks as do late Proterozoic syenites and lamprophyres. Biotite-rich mafic dykes, prevalent within the property area, may be related to local intrusive events.

LOCAL AND PROPERTY GEOLOGY

Rocks in the Project area consist of multiply deformed Archean gneissic metavolcanic and metasedimentary rocks, and associated tonalitic, granitic and gabbroic rocks, which have been metamorphosed to upper amphibolite facies. Variably deformed Proterozoic rocks include late gabbros, and syenite plutons and associated lamprophyre dykes. Massive sulphide mineralization is described as being hosted by a hornblendite layer conformable with the regional stratigraphy.

Nickel-copper-cobalt plus PGE mineralization at Ferguson Lake is hosted mainly by fine- to coarse-grained gabbros which include hornblendites. Three of the mineral zones (East, Central, and West) are at least spatially related to the same gabbro unit which is between 10 m and 600 m thick and has been traced by intermittent exposures and by diamond drilling over a strike length of more than 12 km east and west of Ferguson Lake. This and the other gabbro units hosting the several other mineral zones dip moderately to steeply north and are generally conformable with enclosing hornblende-rich gneisses.

Better grades of base and precious metal mineralization are present within massive to semi-massive sulphide lenses, pods, and stringers which consist of between 80% and 90% pyrrhotite and lesser chalcopyrite, some pyrite, and very fine-grained pentlandite. Better grades are contained within zones having thicknesses of between two and tens of metres.

MINERAL RESOURCES

Table 1-3 summarizes Mineral Resources for the Ferguson Lake Project.

TABLE 1-3 MINERAL RESOURCES – NOVEMBER 30, 2011

Starfield Resources Inc – Ferguson Lake

INDICATED RESOURCES

Zone	Tonnes (Mt)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Fe %	S %
Main West Pit	1.1	0.63	0.97	0.07	0.22	1.54	36.34	19.95
Main West UG	14.7	0.65	0.99	0.08	0.25	1.55	38.16	20.97
Total Indicated	**15.8**	**0.65**	**0.99**	**0.07**	**0.25**	**1.55**	**38.04**	**20.90**

INFERRED RESOURCES

Zone	Tonnes (Mt)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Fe %	S %
Main West Pit	0.2	0.57	0.90	0.07	0.17	1.40	33.96	18.40
Main West UG	5.9	0.59	0.82	0.07	0.20	1.34	36.15	19.59
West Zone Ext.	14.7	0.71	1.23	0.08	0.31	1.88	41.63	23.14
East Zone	9.4	0.65	0.76	NE	NE	NE	38.41	21.16
Total Inferred	**30.2**	**0.67**	**1.00**	**0.05**	**0.19**	**1.18**	**39.49**	**21.79**

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources were estimated at NSR cut-off values of $53 and $97 per tonne for open pit and underground, respectively.
3. Mineral Resources were estimated using average long-term metal prices of US$9.00, US$3.15, and US$17.00 per pound of nickel, copper, and cobalt, respectively, a US$/C$ exchange rate of 1.00, and a royalty of 2% NSR.
4. Metallurgical recovery assumptions were 91% for nickel, 96% for copper, and 90% for cobalt.
5. Platinum, palladium, and cobalt not estimated for the East Zone as the East Zone contains a higher proportion of historical Inco holes for which cobalt or individual platinum or palladium grades have not been determined.

This Mineral Resource estimate has been completed by James Lavigne, P.Geo., Associate Consulting Geologist, RPA, and Qualified Person as defined in NI 43-101.

The geostatistical block model resource estimate was completed and reported in the December 25, 2008 Technical Report by RPA (then Scott Wilson RPA). Subsequently, in July 2009, RPA updated the 2008 resource estimate to incorporate drilling completed by Starfield subsequent to the preparation of the 2008 report. The drilling, and therefore the resource update, was restricted to a part of the Ferguson Lake deposit. For the purposes of the present PEA, the following work has been completed on the updated block model:

1. Iron and sulphur values have been added to the compiled block model based on nickel/iron and nickel/sulphur regression analyses.

2. Block NSR values have been updated based on current metal prices and updated project technical parameters.

3. The block model has been assessed, and resources reported, based on updated cut-off NSR values for open pit and underground resources.

The Mineral Resource estimate has been completed in compliance with the requirements of NI 43-101 and the definitions set out by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council in November 2010.

MINING OPERATIONS

Indicated and Inferred Resources in the Main West and West Extension zones were considered in the PEA. East Zone Mineral Resources were not included, as they are lower grade, lack estimates for cobalt grades, and are physically isolated from the Main West Zone and West Extension (on the other side of Ferguson Lake).

OPEN PIT

Investigation of open pit mining indicated that potential was limited, as the relatively narrow mineralized lenses (average thickness is ten metres) and shallow dip (40°) cause stripping ratios to increase rapidly with depth. Despite the limited scope to which it applies, open pit mining was included in the PEA, to provide process feed while underground development is completed. The open pit consists of 1.8 Mt of ore, at grades of 0.51% Ni, 0.82% Cu, and 0.06% Co, and a stripping ratio of 1:1 including pre-stripping of 300,000 tonnes of waste in year minus one.

UNDERGROUND

A production rate of 6,000 tpd based on 350 days per year was selected. The Main West Zone and West Extension will essentially be operated as two separate mines, nominally producing 3,000 tpd each. Mechanized cut and fill stoping was selected as the underground mining method. The Main West Zone will be accessed by ramp and the West Extension by shaft.

Ramping is planned to be in the footwall, linking main levels that will be 60 m apart vertically, dividing mineralized lenses into stoping blocks. Sublevels will separate the

blocks into 20 m vertical portions. Stope access crosscuts will be driven 40 m to the ore from the main and sublevels, on 200 m centres along strike. The crosscuts will be used to access each 20 m vertical portion of a block starting with a minus 20% grade down-ramp, which will be slashed after each lift, until it ends as a plus 20% grade up-ramp. The 200 m spacing will yield two 100-m-long stope cuts, running east and west from the access crosscut. Each cut will be four metres high, with each access crosscut servicing five cuts. Ore will be loaded into trucks either in the stope or at the junction of the stope crosscut and the footwall level. Approximately 25 active cuts will be required at any one time, to achieve 6,000 tpd ore production. Waste fill will be delivered by LHD and trucked from ongoing development or from waste raises, which will be driven to surface and connect to the main levels at convenient locations.

Dilution of 15% and mining extraction of 95% were applied to resource totals, after subtraction of open pit mining. Resulting underground production quantities of 38.6 Mt, at grades of 0.62% Ni, 0.98% Cu, and 0.07% Co, were scheduled in a Life of Mine Plan (LOMP).

MINERAL PROCESSING AND METALLURGICAL TESTING

Testwork that was carried out from 2001 to 2008 by Kingston Process Metallurgy, SGS Lakefield, and NeoFerric has generated a flowsheet on which to assess the economic viability of the Project at the scoping level. Additional testwork was completed in 2009 at McGill University and in 2010/2011 at SGS in Lakefield, Ontario (SGS) to assess oxidation and hydrolysis. This testwork indicated that continuous oxidation of ferrous chloride solution was successful at 120°C and produced a precipitate consisting of 97% hematite. Continuous hydrolysis of ferric chloride solution was successful in producing a precipitate with 69% iron and 0.9% chloride and hydrochloric acid at up to 36% strength.

During 2011, a flotation test program was carried out to assess if the Ferguson Lake ore was amenable to upgrading by means of flotation prior to further treatment at the hydrometallurgical process plant. A series of bench scale flotation tests established that it was possible to recover essentially all the sulphide minerals into a bulk sulphide concentrate while also rejecting about 80% of the gangue minerals.

The two major continuous leaching campaigns have been successfully carried out in a continuous mini-pilot plant at SGS Lakefield. The iron oxidation and hydrolysis steps

have also been successfully tested in continuous mini-plant campaigns at SGS Lakefield.

There has been sufficient laboratory work carried out to enable definition, at the conceptual level, a technically-viable preliminary flowsheet. The major unit operations (primary leach, secondary leach, ferrous iron oxidation and hydrolysis) have been successfully tested in continuous mode at SGS, thereby demonstrating the technical competence of these unit operations. For hydrolysis, a very similar circuit has been extensively piloted by SMS Siemag, leading to a decision to build a commercial plant.

The process oxidation and hydrolysis steps are novel, and are not commercially proven unit operations. Fundamental measurements of heat capacity, thermal conductivity, viscosity, and other properties are required in order to determine the required heat transfer coefficients to size heat transfer surfaces, or preferably, direct measurements of heat transfer at a larger scale than has been completed thus far.

The materials of construction will require further development, particularly for the heat transfer surfaces. The fluids under consideration are rather aggressive and materials normally used for heat exchange surfaces, e.g. steel, would be dissolved. SGS has completed preliminary corrosion testing, and it appears that higher grades of titanium may be suitable for the oxidation stage, and, tantalum may be required for the hydrolysis stage. It is expected that verification in this area will require substantial work.

The oxidation step appears to be controlled by the rate at which oxygen is transferred from the gas phase to the bulk liquid phase. This implies that agitation is extremely important, and further work is required to properly scale-up the oxidation step.

The process has evolved from the results of metallurgical testwork done over the past decade. As experimental data became available it was incorporated into a process model (numerically rigorous mass/energy balance) that was used to develop the operating and capital cost estimates used in this and the preceding evaluation of the process. The latest embodiment of the process model indicates that overall metal extractions of 91% for nickel, 96% for copper, and 90% for cobalt are achievable using the proposed process.

The process has the potential to generate stable, non-toxic tailings, and most importantly, recover a sufficient quantity of the intrinsic energy in the ore to make it self-sustaining in energy usage. This means that there may be no need to import fuel oil for generating power on-site, which would otherwise be necessary in such a remote location. The energy recovered is potentially sufficient to operate both the hydrometallurgical process plant, as well as the mine, concentrator, and concentrate pipeline pump stations.

There are a number of uncertainties in the process flowsheet, where it has been necessary to make assumptions. The most important of these are the iron and sulphur contents of the ore, and its accompanying mineralogy, and secondly, the lack of fundamental data in the scientific literature to enable, in particular, a rigorous energy balance to be carried out.

Both iron and sulphur are related to the energy balance. In the former, the amount of sulphur in the ore, particularly as the mineral pyrrhotite, determines the amount of hydrogen sulphide gas liberated, which in turn determines the amount of energy liberated and the amount of sulphuric acid produced. The amount of iron in the feed determines how much hydrochloric acid is needed, which in turn determines how much hydrolysis and acid recycle is needed.

No formal account has been taken of the precious metals content of the ore, since no process for their recovery has yet been properly evaluated. However, it is clear that their recovery is mandated, even as a relatively low-grade concentrate, since they will comprise a significant revenue stream.

There is also potential to process low-base metals, high-PGE material, not currently included in Mineral Resources. Low-base metals-content material will not provide the energy advantages of the massive sulphide material, however, there may be value in considering a process unit that can recover PGEs from both low-base metal material and from the secondary leach residue.

The process risk assessment has been updated to reflect new work since the previous risk exercise, and has been reviewed by Arithmetek and RPA. Results of this assessment show that the majority of the process has low associated risks. Medium-risk

items include the oxidation and hydrolysis components of the proposed process as these areas have not yet been scaled up to commercial plant size, however, these risks can likely be mitigated through further testwork. The iron hydrolysis and HCl recovery circuit is a technique originally developed by PORI Inc. in the 1970s, and piloted by SMS Siemag for four years since 2002.

The process flowsheet is based on the following unit operations:

- The run of mine (ROM) ore will be crushed and processed in the on-site concentrator and the concentrate will then be transported by pipeline to the hydrometallurgical processing facility located near Arviat.

- The concentrate slurry will be dewatered and filtered ahead of the process plant.

- Primary leaching of the sulphide concentrate will be carried out with hydrochloric acid (HCl) at a temperature of approximately 115°C, producing a ferrous chloride ($FeCl_2$) solution, hydrogen sulphide (H_2S) off-gas, and a residue containing virtually the base metals.

- The hydrogen sulphide (H_2S) off-gas from the primary leach will be f cooled to condense most of the water and any residual HCl, prior to burning in air to generate high-strength sulphur dioxide (SO_2), which will be sent to an acid plant to produce 93% sulphuric acid. This will permit the recovery of a substantial amount of the intrinsic energy in the ore as high pressure steam, for use both directly in the flowsheet and also for electric power generation.

- The solids from the primary leach will be treated in a secondary leach with oxygen at about five atmospheres, under conditions that dissolve the nickel, copper and cobalt while initially dissolving but then re-precipitating the iron as goethite. The sulphide sulphur in the primary leach residue will be converted to elemental sulphur.

- The slurry from the secondary leach will be filtered, the filtrate proceeding to further processing and the filter cake contacted with recycled strong hydrochloric acid to re-dissolve the goethite. The resulting slurry will be filtered and the filter cake washed with water. The washed filter cake will leave the process as a residue containing the precious metals. The filtrate, a concentrated solution of ferric chloride, will be sent to the oxidation/hydrolysis section of the process.

- The ferrous chloride ($FeCl_2$) solution will be combined with other recycled streams containing iron chloride, concentrated by evaporation, then oxidized with oxygen to initially precipitate one-third of the iron as hematite, and generate a ferric chloride ($FeCl_3$) solution.

- The slurry from the oxidation step will be subjected to a hydrolysis step, in which it is heated and most of the ferric chloride is converted to hematite and gaseous hydrochloric acid (about 35% HCl). The acid will be recycled.

- The slurry from the hydrolysis step will be filtered hot in a specialized filter and the filtrate is returned to either the oxidation step or the beginning of the hydrolysis step to control the solids content of the slurry in the hydrolysis step.

- The hot filter cake from the hydrolysis step will be quenched and diluted with recycled solutions from downstream in the circuit. The resulting slurry will be filtered and the filter cake washed with water. The washed filter cake will leave the process as a by-product hematite. The filtrate will be recycled to the evaporation step ahead of the oxidation stage.

- Zinc will be removed from the filtrate from the secondary leach by ion exchange (IX), after which copper and cobalt recovery from the secondary leach solution will occur through anionic SX. A second stage of SX will then separate the copper from the cobalt, and each metal will be recovered using SX and electrowon from a sulphate medium to generate LME-grade metal.

- Nickel will then be recovered by conventional sulphate SX, and electrowon to produce LME-grade metal.

- The raffinate from the nickel SX step will be combined with the strip solution from the zinc ion exchange step and evaporated at reduced pressure the steam produced being condensed and the condensate recycled, and the final concentrated chloride brine being contacted with concentrated sulphuric acid, generating hydrochloric acid that will be recycled and a sulphate effluent that will leave the process.

LIFE OF MINE PLAN

The LOMP was completed on an annual basis, as summarized in Table 1-4. Construction at the mine and process sites is scheduled to begin three years before production start-up, after supplies and materials are shipped to Arviat the previous summer. This is a major issue and items required on site for activity in a given year must be moved by sea in the summer to a staging area for subsequent movement to the site in a brief window between February and March of the following year. Mine site supplies will be transported to site by winter Cat train.

YEAR -3

Construction materials and equipment for the camp and site buildings must be shipped to a staging site in the north and then to site in the winter.

YEAR -2

Construction of camp facilities and site buildings will begin at both mine and process sites. Open pit and underground mining equipment will be purchased and transported to

Ferguson Lake. Slurry pipeline and powerline installation will be undertaken during the winter. Preparations for mine development and shaft sinking will commence.

YEAR -1

Construction of all surface facilities will be completed by the end of the year. Ramping and development in the Main West Zone, and shaft sinking in the West Extension will begin. Pre-stripping of the open pit will be completed.

YEAR 1

Production will begin, with feed from open pit mining at a production rate of 2,000 tpd. Development and stope preparation in the Main West Zone will be completed. Ramping and development will follow completion of shaft sinking in the West Extension. The process plant is scheduled to ramp up to full production of 6,000 tpd over the course of the year, averaging approximately 34% of full capacity.

YEAR 2

Production from the Main West Zone will begin at the start of the year. Open pit mining will be completed towards the end of the year, to be replaced by underground mining from the West Extension.

YEAR 3+

From Year 3 onwards, production will be split between the Main West Zone, at 3,300 tpd, and the West Extension, at 2,700 tpd. The Main West Zone is scheduled to be exhausted in Year 20 and the West Extension, in Year 19.

TABLE 1-4 LIFE OF MINE PLAN
Starfield Resources Inc. - Ferguson Lake

			Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
PHYSICALS	Days Mined						350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350
Main West Open Pit	Ore Mined	tpd					2,073	2,983																		
	To Mill	'000 t	1,770				726	1,044																		
	To Dump	'000 t	1,783			300	731	1,052																		
	Ni	%	0.51				0.51	0.51																		
	Cu	%	0.82				0.82	0.82																		
	Pd	g/t	-				0.00	0.00																		
	Pt	g/t	-				0.00	0.00																		
	Co	%	0.06				0.06	0.06																		
	Fe	%	31.98				31.98	31.98																		
	S	%	16.64				16.64	16.64																		
Main West Underground	Production	tpd					-	2,143	3,301	3,301	3,299	3,301	3,301	3,299	3,300	3,300	3,300	3,300	3,299	3,299	3,300	3,301	3,301	3,301	4,280	5,092
	Tonnes	'000 t	22,512				-	750	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,498	1,782
	Ni	%	0.59				-	0.67	0.61	0.61	0.62	0.60	0.61	0.59	0.52	0.51	0.52	0.63	0.60	0.54	0.56	0.63	0.67	0.64	0.54	0.56
	Cu	%	0.88				-	1.02	1.18	1.10	1.03	1.07	1.00	0.82	0.68	0.66	0.63	0.77	0.69	0.72	0.80	0.99	1.13	0.89	0.73	0.86
	Pd	g/t	-																							
	Pt	g/t	-																							
	Co	%	0.07				-	0.08	0.07	0.07	0.07	0.07	0.07	0.07	0.06	0.06	0.06	0.08	0.07	0.07	0.07	0.07	0.07	0.08	0.06	0.07
	Fe	%	35.01				-	39.35	36.57	36.84	37.16	36.42	36.87	35.88	32.74	32.24	32.81	35.05	33.76	31.68	32.49	36.64	39.32	37.64	32.96	32.34
	S	%	19.17				-	21.96	20.08	20.25	20.47	19.96	20.26	19.61	17.53	17.22	17.54	19.60	18.60	17.14	17.68	20.28	21.94	20.90	17.85	17.62
West Extension Underground	Production	tpd					-	899	2,701	2,699	2,701	2,700	2,699	2,701	2,699	2,699	2,700	2,699	2,699	2,699	2,701	2,699	2,699	2,699	1,719	-
	Tonnes	'000 t	16,034				-	315	945	945	945	945	945	945	945	945	945	945	945	945	945	945	945	945	602	-
	Ni	%	0.66				-	0.67	0.62	0.58	0.71	0.68	0.73	0.59	0.63	0.60	0.69	0.72	0.70	0.77	0.67	0.61	0.61	0.61	0.64	-
	Cu	%	1.14				-	1.27	1.12	1.14	1.20	1.39	1.30	0.99	1.23	1.08	1.01	1.55	1.30	1.29	1.05	0.90	0.75	0.92	0.99	-
	Pd	g/t	-																							
	Pt	g/t	-																							
	Co	%	0.08				-	0.08	0.07	0.07	0.09	0.08	0.09	0.07	0.07	0.07	0.09	0.09	0.09	0.09	0.08	0.07	0.07	0.07	0.08	-
	Fe	%	38.64				-	39.32	37.22	35.52	40.90	39.46	41.48	35.63	37.74	36.63	39.99	40.91	40.33	42.86	39.20	36.83	36.85	37.05	37.79	-
	S	%	21.48				-	21.91	20.47	19.35	23.02	22.02	23.42	19.48	20.82	20.06	22.38	23.05	22.64	24.44	21.87	20.26	20.23	20.36	20.92	-
							34%																			85%
Total Mill Feed	Production	tpd					2,073	6,025	6,002	6,000	6,000	6,001	6,000	6,001	5,999	5,998	6,000	5,999	5,998	5,998	6,001	6,000	6,000	5,999	5,999	5,092
	Tonnes	'000 t	40,315				726	2,109	2,101	2,100	2,100	2,100	2,100	2,100	2,100	2,099	2,100	2,100	2,099	2,099	2,100	2,100	2,100	2,100	2,100	1,782
	Ni	%	0.61				0.51	0.59	0.61	0.60	0.66	0.63	0.66	0.59	0.57	0.55	0.59	0.67	0.65	0.65	0.61	0.62	0.64	0.63	0.57	0.56
	Cu	%	0.98				0.82	0.96	1.15	1.12	1.11	1.22	1.13	0.89	0.93	0.85	0.80	1.12	0.96	0.97	0.92	0.95	0.96	0.90	0.81	0.86
	Pd	g/t	-				0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Pt	g/t	-				0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Co	%	0.07				0.06	0.07	0.07	0.07	0.08	0.07	0.08	0.07	0.07	0.06	0.07	0.08	0.08	0.08	0.07	0.07	0.07	0.07	0.07	0.07
	Fe	%	36.32				31.98	35.69	36.87	36.25	38.84	37.79	38.95	35.77	34.99	34.21	36.04	37.68	36.72	36.71	35.51	36.73	38.21	37.37	34.34	32.34
	S	%	19.98				16.64	19.32	20.26	19.85	21.61	20.89	21.69	19.55	19.01	18.50	19.72	21.15	20.42	20.42	19.57	20.27	21.17	20.66	18.73	17.62

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INFRASTRUCTURE

The Project site requires infrastructure to support open pit and underground mining operations, crushing and grinding mill facilities, and slurry preparation and pumping facilities. Site roads, camp facilities, back-up power generation, electrical distribution, mobile equipment, and buildings will be required.

Between the mine and process sites, a 285 km ground-level slurry pipeline will be routed to avoid large bodies of water. There will be thickening and storage tankage at Ferguson Lake, two pump stations at equal intervals along the pipeline, and a receiving terminal and tankage at the process site. A power transmission line will follow the pipeline route, allowing power generated at the process site to be used to power the mine, concentrator, and slurry pumps. All process water requirements at the mine site will be provided from Ferguson Lake.

Only winter road access has been included along the pipeline route. Construction and maintenance will take place during the winter, using Cat trains for transportation of heavy material and equipment. For purposes of emergency repair, Starfield will develop comprehensive spill management plans including procedures for access by season, which may include the use of Cat trains, all-terrain vehicles, or helicopters.

An initial proposed location for the hydrometallurgical processing facilities is near Arviat, Nunavut. Final site location will be determined following a program of community consultation, and consideration of alternatives such as Baker Lake or Rankin Inlet. The plant facility will house all of the necessary circuits required by the proposed metallurgical process, in addition to a receiving terminal for the concentrate slurry, a power plant, a control room, water treatment and water storage, fuel day tanks, and an administration building. Infrastructure items outside of this footprint include an acid tank farm, fuel storage, port facilities, hematite/gangue storage, and warehousing facilities.

The process will produce approximately 53 MW of power from four process turbines. Process equipment will consume 19 MW of power, leaving approximately 11 MW of surplus power after transmitting approximately 23 MW to the mine site, to power the mine, concentrator, and slurry pumps. Diesel generation at both sites will provide power for construction, process start-up, and emergency back-up.

As a by-product of the metal production process, the Project will produce approximately 0.85 Mt of 93% pure sulphuric acid per year. Acid will be stored in a tank farm at the plant site during the winter months. The tank farm will comprise 27 tanks, with a total storage capacity of 0.7 Mt of acid.

LOGISTICS

The Ferguson Lake site is 250 km southwest of Rankin Inlet. To date, heavy equipment and site facilities have been brought to the site by Cat trains, over the frozen ground and lakes in winter only, via permitted right of way. The proposed operational scenario does not include an all-season road between the mine and the hydrometallurgical processing site, and material transportation to the Ferguson Lake site will continue in the current manner. An owner-operated fleet of reliable Cat units will be required to develop and support the Project.

Air transport of critical or lightweight items will be possible, however, no cost allowance for that has been included in the PEA, beyond regular flights for transport of personnel.

The hydrometallurgical process facilities will be located in close proximity to port facilities. A fleet of surface vehicles will be used to transport supplies and materials from the port to the process site. Currently, port facilities are limited in Arviat and other potential process sites. Construction of dedicated deep-water docking, handling, and storage facilities will be required.

Supplies and materials will arrive by ship, in the ice-free season between July and mid-October. Items for the Ferguson Lake site will be stored at the process site, and staged for transportation by Cat train in winter. Purchasing and warehousing will be managed from the mine and process sites, by a warehouse manager, buyer, and warehousemen. Coordination of sea lift logistics will require consultants located in large shipping centres in the south.

LME-grade metal products and dried hematite precipitate will be transported by ship to buyers.

An annual total of 0.85 Mt of sulphuric acid will be shipped during the 3.5 months shipping season between July and mid-October. During the shipping season, the acid will be pumped to the wharf in order to load the ships.

The shipping requirement for acid is significant, in that the total annual volume must be moved in 15 weeks. For the PEA, it has been assumed that the average distance to market will be 12,000 km, or approximately Brazil, requiring approximately 8 x 50,000 tonnes acid capable tankers in operation during the shipping season.

The total manpower during the mine life will be approximately 700 employees, with an average of 367 on site to cover the daily operations. The work schedule will include two 11 hour shifts per day and a two week in two week out rotation.

ENVIRONMENTAL CONSIDERATIONS

Starfield does not yet hold any of the permits required to operate a mine. A production lease, a water licence, and numerous additional permits and approvals will be required for the Project.

Baseline environmental studies were initially conducted in 1999 and have been expanded upon over the years. A more comprehensive baseline program was conducted by Rescan Environmental Services Ltd. (Rescan) in 2007, however, the program was halted pre-maturely in 2009.

Key environmental management issues associated with the current Project concepts consist of:

- Aquatic Resources

  At the mine:

  - Storage facility for the flotation tailings (tailings may be mildly reactive). The siting of this facility may impact on aquatic habitats.
  - Storage facility for the waste rock (some of the waste rock is likely acid generating)
  - Water usage for grinding and slurrying of ore concentrate
  - Stream diversion at open pit site and tailings management facility

  Along the pipeline

  - Water crossings, spill containment and management

At the Hydrometallurgical Process Plant

- Cooling water usage.
- Marine habitat impacts (siting, operational and transportation).

- Proximity to sensitive and protected wildlife and archaeological areas
  - Slurry pipeline route relative to Caribou Protection Area.
  - Other Project sites (mine and port facilities).

- Acid storage and handling
  - Tank farm footprint.
  - Land transport, transhipment, and ocean transport.

- Storage of process products/by-products (hematite, sulphur, residue)

- Use/management of hazardous chemicals (H_2S, HCl, etc.)

Port facility locations have not been considered in detail. Further studies will involve marine habitat impact studies, with due consideration for bathymetry, fish habitat, harvest and fishing areas, baseline conditions, and protected areas. The use of established shipping routes, to the extent possible, may also minimize new environmental impacts.

RPA notes that the slurry pipeline concept consists of widely-used, industry-standard technology, and recommends a review of comparable pipelines currently in operation in the Arctic (North America, Scandinavia, Russia), to determine best practices for design and operation from an environmental viewpoint.

The acid storage tank farm includes an area allowance for spill containment. Further design work may reduce the overall footprint by increasing berm heights or taking advantage of natural topography. Acid handling, including transport from the process plant to the storage tanks, from the storage tanks to the port facility, and transhipment to ocean freighter, will require consideration of the following:

- Safety and Human Health.
- Spill Response and Management plans for land, fresh water, and marine environments.
- Transportation of Dangerous Goods regulations.

There is potential for the hydrometallurgical process plant to generate excess power, which could be sold to the local utility at considerably lower costs than is possible using diesel generation, which is the principle current method in the communities of the Kivalliq

Region. This would provide both an economic (lower power costs) and environmental benefit (reduced diesel combustion).

Under the Nunavut Land Claims Agreement, an Inuit Impact and Benefit Agreement (IIBA) will have to be negotiated with the Kivalliq Inuit Association. Likely included in the IIBA will be cash payments, scholarships, business opportunities, and commitments to hire and train residents of the Kivalliq region (Rankin Inlet, Baker Lake, Arviat, Chesterfield Inlet and Whale Cove).

Agreements with the Kivalliq Inuit Association for a Mining Lease as well as an Inuit Impact and Benefits Agreement will likely have to be negotiated as well as a Socio Economic Agreement with the Government of Nunavut.

A Community Consultation Plan involving meetings with the general public, community and business leaders, Community Land and Resources Committees, elders' councils, schools, political and regulatory agencies as well as Non-Government Organizations has been implemented, and will continue throughout the life of the Project.

RPA recommends that Starfield formalize arrangements with contractors to maximize employment and business opportunities for those within the Kivalliq Region. As well, Starfield should begin discussions with Nunavut Arctic College regarding co-operative opportunities with students as the project advances.

A site reclamation plan will be required as part of the design and permit application. In this area, the general expectation would be that all facilities would be removed from the site and that surface disturbance would be modified to minimize the impact upon wildlife and other land users. As part of the Project design, the area of disturbance will be minimized and there will be as much ongoing rehabilitation work as possible concurrent with operations.

Facilities that cannot be used in the future will be demolished and removed from site for salvage to the extent possible. The conceptual closure plan for the residual facilities is as follows:

Mine Site

- Cover and reclamation of the flotation tailings area
- Disposal and burial of acidic waste rock in the pit
- Capping of the underground mine
- Disposal/burial of inert construction rubble in the open pit or in an approved surface landfill.
- Allow the open pit to flood. Provide safety berms as required.
- Reclaim all disturbed areas

Pipeline

- Remove all pipelines and spill ponds
- Reclaim disturbed areas

Port Site

- Cover and reclamation of the residue storage pile and any residual hematite by-products
- Reclaim disturbed areas

Site monitoring of conditions and water quality would continue for a period of years after the cessation of operations and removal of facilities.

A reclamation bond will be required to provide funding for the closure plan. In general the bond will be expected to provide full coverage for a government run reclamation program. It may be possible to provide some of the funds in stages during construction, however, it is forecast that the bulk of the funds will need to be in place when operations commence.

The estimated cost for mine reclamation is $35.9 million. This cost will increase if tailings are reactive and require special covers.

CAPITAL COSTS

A pre-production capital cost of $1,500 million is estimated for the construction work required to bring the Project into production, including open pit pre-stripping, mining and process equipment purchase and transportation to site, construction of facilities and infrastructure at Ferguson Lake and the process site, and installation of all services required to support the Project. The estimate includes pre-production costs incurred over three years, with no allowance for escalation. Completion of underground mine

development extends through the first year of production, and costs for that work, totalling $113 million, have been included in the pre-production capital.

Costs for advancing the Project to construction stage, such as resource upgrade drilling, metallurgical testwork, process development, permitting, environmental studies, feasibility studies, and project financing, are not included.

Total Project capital costs are summarized in Table 1-5.

TABLE 1-5 CAPITAL COST SUMMARY
Starfield Resources Inc. - Ferguson Lake

Item	Pre-production ($ millions)	Ongoing ($ millions)	Total ($ millions)
Mine	84.7	66.7	151.3
Process Facilities	788.1	-	788.1
Infrastructure	94.5	-	94.5
Indirects	243.4	26.3	269.7
Contingency	176.9	19.7	196.6
Subtotal	**1,387.5**	**112.7**	**1,500.0**
Sustaining Capital	-	44.2	44.2
Reclamation	-	35.9	35.9
Total	**1,387.5**	**192.8**	**1,580.2**

Note: Totals may not add due to rounding.

OPERATING COSTS

Life of Mine operating costs average $178 million per year. Unit operating costs are summarized in the following table:

TABLE 1-6 UNIT OPERATING COST SUMMARY
Starfield Resources Inc – Ferguson Lake Project

Item	Cost
Mining – Open Pit ($/t moved)	3.83
Mining – Open Pit ($/t open pit ore)	6.54
Mining – Underground ($/t ug ore)	45.36
Total Mining ($/t milled)	43.71
Processing ($/t milled)	25.31
Site Services ($/t milled)	8.26
G&A ($/t milled)	11.16
Life of Mine Plan Total	**88.43**

Note: Totals may not add due to rounding.

Site services include surface operations and infrastructure maintenance, while G&A includes costs specific to remote operations, such as flights for personnel and freight for supplies and consumables.

2 INTRODUCTION

Roscoe Postle Associates Inc. (RPA) was retained by Starfield Resources Inc. (Starfield), to prepare a Preliminary Economic Assessment (PEA) of the Ferguson Lake Project (the Project), near Rankin Inlet, Nunavut, Canada. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects.

Starfield is an advanced exploration and emerging early-stage development company, focused on its 100%-owned Ferguson Lake nickel-copper-cobalt-platinum-palladium property. The property covers approximately 274,000 ha and Starfield has completed 162,000 m of diamond drilling in 445 holes since 1999.

Currently, the major assets and facilities associated with the Project are:

- Mineral Resources held 100% by Starfield, subject to a 3% NSR.
- An exploration camp on the western shore of Ferguson Lake.
- 830 m gravel airstrip adjacent to the exploration camp.
- Small scale construction and exploration support equipment on site.

The PEA is based on the following general assumptions:

- Mine production quantities of 40.3 Mt, based on Mineral Resources of the Main West and West Extension zones, modified by the addition of dilution and extraction factors.
- Production rate of 6,000 tonnes per day, 2.1 million tonnes per year.
- Initial mining of 1.8 million tonnes from an open pit over first two years.
- Subsequent underground mining by mechanized cut and fill methods, in both the Main West and West Extension zones simultaneously.
- Underground access to the Main West and West Extension zones by ramp and shaft, respectively.
- Backfill using waste rock material, from underground waste development, open pit waste, surface waste sources, and tailings from the concentrator.
- Pumping of concentrate ore via a 285 km pipeline, to a proposed process plant site located near Arviat, Nunavut.

- Processing via two-stage leaching followed by solvent extraction (SX), ion exchange (IX) to purify the solution, then SX and electrowinning (EW) to recover the following products:
  - Nickel, copper, and cobalt as LME-grade metal
  - 93% sulphuric acid
  - Hematite
  - Gangue material

- Power generated at the plant site through burning of H_2S gas, and capture of process heat, used to power the process and, via transmission line, the mine site.

- Metal products and acid moved to market via bulk shipping.

- Gangue storage at the Arviat process plant by dry stacking.

The process proposed in the PEA has been previously tested by NeoFerric Technologies Inc. (NeoFerric) and at SGS in Lakefield. Process unit operations have been demonstrated in continuous operation at a laboratory scale, and pilot plant demonstration of the complete process is planned as the Project is advanced.

This report is considered by RPA to meet the requirements of a PEA as defined in Canadian NI 43-101 regulations. The economic analysis contained in this report is based, in part, on Inferred Resources and is preliminary in nature. Inferred Resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the mine schedules and economic forecasts on which this PEA is based will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

All economic forecasts in this report are as of Q4 2011, without escalation, unless otherwise noted.

PRELIMINARY ECONOMIC ASSESSMENT UPDATE

This report is an update of a PEA completed by RPA (formerly Scott Wilson RPA), dated December 15, 2008. The bulk of the current report is unchanged, with the following exceptions, described in the appropriate sections of this report:

- Updated Mineral Resource estimate, as presented in Section 14.

- Discussion of drill results since the last Mineral Resource estimate.

- Updated metallurgical testwork results and revision to process design as discussed in Sections 13 and 17, respectively.

- Capital and operating cost modifications, as a result of new metallurgical testwork and process design.

- Cash flow update, with expanded discussion of product price sensitivities and markets.

SOURCES OF INFORMATION

Site visits were previously carried out by R. Dennis Bergen, P. Eng., Associate Mining Engineer with RPA, who visited the site from June 26 to 28, 2007, and Jamie Lavigne, P.Geo., who visited the site twice within the period of September to November 2006. For the current, updated PEA, no site visits were carried out.

During the course of the initial PEA, through to the current update, discussions were held with the following personnel from Starfield and consultants to Starfield:

- André Douchane, President & CEO, Starfield
- Michael Moran, Director of Engineering, Starfield
- Trifon Brakov, Project Geologist, Starfield
- Tyler Power, Project Geologist, Starfield
- Phil Evans, Process Design Consultant, ACSI
- Ernie Burga, Process Design Consultant, ACSI
- Michael Dry, Process Design Consultant, Arithmetek
- Carl White, Process Design Consultant, NeoFerric
- Nick Carter, P. Eng, Geological Consultant
- Allison Rippin Armstrong, Environmental and Permitting Consultant
- Denise Lockett, Community & Government Relations Consultant
- Janet Stritychuk, JM Stritychuk & Associates Inc.
- Randall Knapp, P.Eng., Principal Environmental Engineer, SENES

Michael J. Dry, Ph.D., of Arithmetek, Inc. (Arithmetek) contributed to Section 13 and parts of Section 17 of the report, while Phil Evans, P.Eng., and Ernie Burga, P.Eng., of Andeburg Consulting Services Inc., (ACSI) contributed to Section 17. Randall A. Knapp, P. Eng., of SENES Consultants Limited (SENES) contributed to Sections 20 and 21 regarding environmental aspects and costs related to them. Holger Krutzelmann, P. Eng., of RPA is a Qualified Person under NI 43-101 with overall responsibility for the metallurgical, processing, socio-economic, permitting, and environmental aspects of this report.

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 27 References.

LIST OF ABBREVIATIONS

Units of measurement used in this report conform to the Imperial system. All currency in this report is US dollars (US$) unless otherwise noted.

μ	micron
°C	degree Celsius
°F	degree Fahrenheit
μg	microgram
A	ampere
a	annum
bbl	barrels
Btu	British thermal units
C$	Canadian dollars
cal	calorie
cfm	cubic feet per minute
cm	centimetre
cm^2	square centimetre
d	day
dia.	diameter
dmt	dry metric tonne
dwt	dead-weight ton
ft	foot
ft/s	foot per second
ft^2	square foot
ft^3	cubic foot
g	gram
G	giga (billion)
Gal	Imperial gallon
g/L	gram per litre
g/t	gram per tonne
gpm	Imperial gallons per minute
gr/ft^3	grain per cubic foot
gr/m^3	grain per cubic metre
hr	hour
ha	hectare
hp	horsepower
in	inch
in^2	square inch
J	joule
k	kilo (thousand)
kcal	kilocalorie
kg	kilogram
km	kilometre
km/h	kilometre per hour
km^2	square kilometre
kPa	kilopascal
kVA	kilovolt-amperes
kW	kilowatt
kWh	kilowatt-hour
L	litre
L/s	litres per second
m	metre
M	mega (million)
m^2	square metre
m^3	cubic metre
min	minute
MASL	metres above sea level
mm	millimetre
mph	miles per hour
MVA	megavolt-amperes
MW	megawatt
MWh	megawatt-hour
m^3/h	cubic metres per hour
opt, oz/st	ounce per short ton
oz	Troy ounce (31.1035g)
ppm	part per million
psia	pound per square inch absolute
psig	pound per square inch gauge
RL	relative elevation
s	second
st	short ton
stpa	short ton per year
stpd	short ton per day
t	metric tonne
tpa	metric tonne per year
tpd	metric tonne per day
US$	United States dollar
USg	United States gallon
USgpm	US gallon per minute
V	volt
W	watt
wmt	wet metric tonne
yd^3	cubic yard
yr	year

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by Roscoe Postle Associates Inc. (RPA) for Starfield Resources Inc. (Starfield). The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to RPA at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other information supplied by Starfield and other third party sources.

For the purpose of this report, RPA has relied on land tenure and ownership information provided by Starfield and its mineral lands consultant, JM Stritychuk & Associates Inc.. RPA has relied on opinions provided by Arithmetek, ACSI, and SENES, for information provided in Sections 13, 17, 20, 21, and the summaries included in this report. RPA has not researched property title or mineral rights for the Ferguson Project and expresses no opinion as to the ownership status of the property.

RPA has relied on Starfield for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Fergusson Project.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk.

4 PROPERTY DESCRIPTION AND LOCATION

LAND TENURE

The Ferguson Lake Project currently consists of 273 mineral claims and 10 mining leases covering an area of 274,211.6 ha (677,590.6 acres) in the Kivalliq region of southern Nunavut Territory some 240 km west of Rankin Inlet and 160 km south-southwest of Baker Lake (Figures 4-1 and 4-2). Ferguson Lake, central to the large property area, is midway between Yathkyed and Kaminuriak Lakes. The property currently measures approximately 90 km in an east-west direction and approximately 50 km north-south.

All of the mineral claims are contiguous and extend east, west, south, and northwest of Ferguson Lake between approximate UTM coordinates 6,940,000 mN and 6,990,000 mN, 565,000 mE and 655,000 mE, Zone 14).

Parts of the current property were initially located in 1997 by way of one Prospecting Permit covering the northwest quarter of NTS map-area and three contiguous mineral claims. Additional mineral claims were located in 1998 and 1999 to cover the area of the Prospecting Permit which expired February 1, 2000. A number of these claims were allowed to lapse following detailed prospecting in 2001. The mineral claims acquired by Starfield prior to 2003 included a former Mining Lease previously held by Canadian Nickel Company, Ltd., a subsidiary of Inco Ltd (Inco).

Most of the mineral claims comprising the expanded property area were obtained between January and November of 2005. The current property has been decreased from a total of 540 claims (1,287,778.63 acres) in 2008 to the current claims as a number of claims in the northern part of the property were dropped.

Mineral claims in Nunavut are valid for two years from the recording date and may be renewed for an additional year by completing representation (assessment) work in the amount of $4.00/acre within the initial two-year period. Annual work in the amount of $2.00/acre is required to renew the claims beyond the third year. Representation work for the various mineral claims has been routinely filed on an annual basis since 1999.

Land use permits, including an Exploration Permit, Right of Way for winter transport, a Water Licence, and a Commercial Lease, enable exploration work to be conducted over the entire property. These permits have been issued or renewed by the Kivalliq Inuit Association for parts of the property covering Inuit-owned lands and by Indian and Northern Affairs Canada for Crown lands. The Commercial Lease was negotiated between the Kivalliq Inuit Association and Starfield Resources on July 23, 2007 for the main project area (expires July 23, 2012). Preliminary environmental studies, conducted by Rescan Environmental Services Ltd. (Rescan) on behalf of Starfield, have been ongoing since 1999.

The majority of exploration work completed since 1999 has been directed to several mineral zones east and west of Ferguson Lake in the central property area (Figures 4-2 and 4-3).

Starfield Resources Inc. entered into an option agreement in February, 1999 to purchase a 100% interest in the mineral claims comprising the Ferguson Lake Project from the Ferguson Lake Syndicate in exchange for an initial cash payment, the issuance of common shares and scheduled work commitments. The issuance of additional common shares to the Ferguson Lake Syndicate was based on incurred exploration expenditures. Starfield's current 100% earned interest in the property is subject to a 3% net smelter royalty (NSR) on potential future mineral production, a 3% gross overriding royalty on any diamond production and a $25,000 annual advance royalty payment. Starfield Resources Inc. has the right to purchase 1% of the NSR for $1 million, for a period of 180 days following receipt of a positive feasibility study recommending commercial production (the PEA cash flow model includes the assumption that this purchase has been completed).

On October 5, 2009 Starfield announced the signing of a Joint Venture Agreement with Thanda Resources Inc. (Thanda), a privately held diamond exploration company. The agreement gives Thanda the right to earn an initial 50% participating interest in a joint venture that will focus on the exploration and development of the diamond potential at the Ferguson Lake property by incurring cumulative exploration expenses of $6.5 million over a three year period. After the initial $6.5 million investment by Thanda, it may choose to commence a prefeasibility study. Upon completion of a prefeasibility study, Starfield may choose to either retain a 30% interest in the project by paying its pro-rata

share of the costs of the prefeasibility study and future development, or convert to a 3% gross revenue royalty. Starfield reports to RPA that Thanda is currently in default of the Joint Venture Agreement.

Figure 4-1

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Location Map

November 2011

Source: James G. Lavigne et al., 2007.

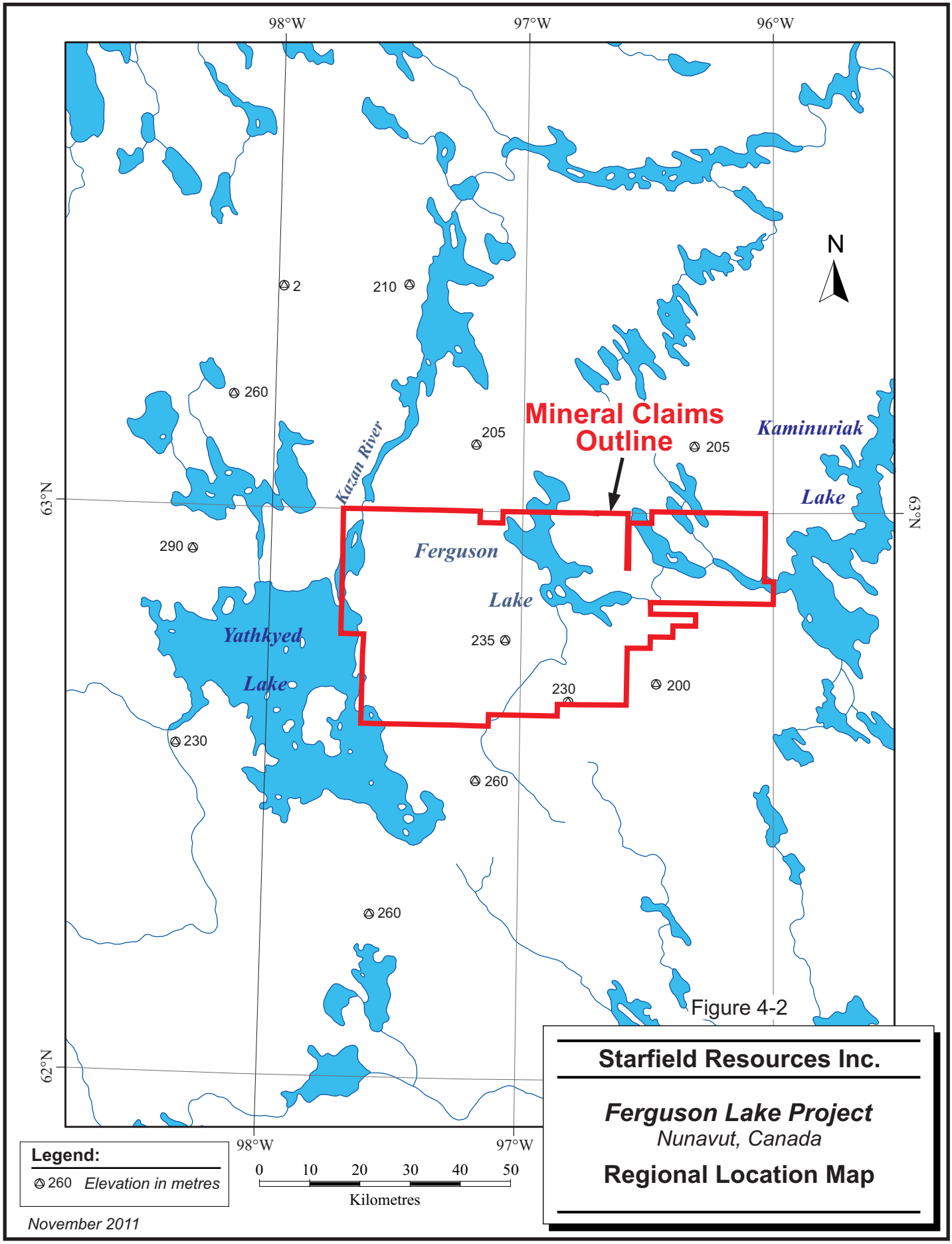

Figure 4-2

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Regional Location Map

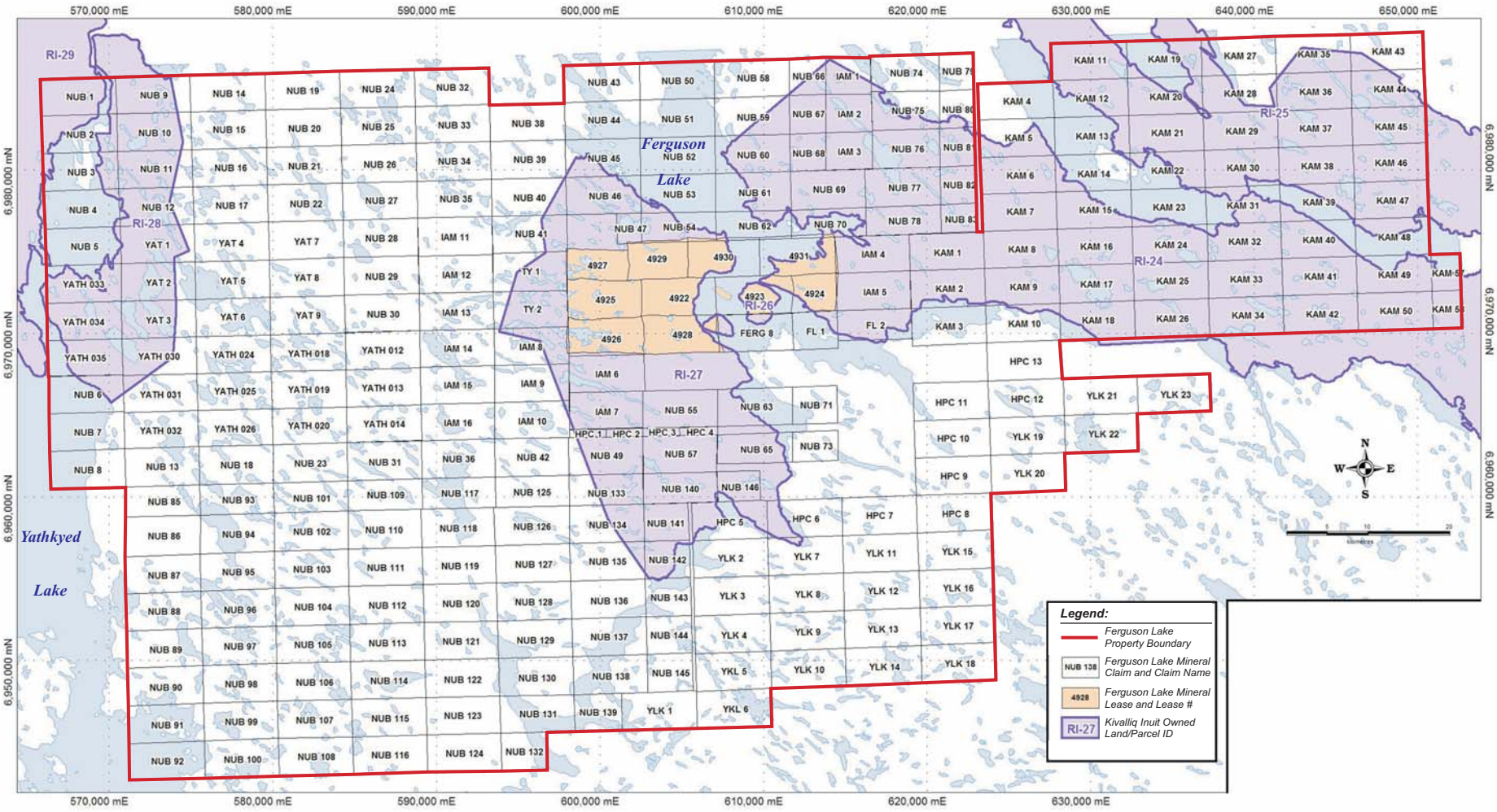

Figure 4-3

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Claim Map

November 2011

Source: Starfield Resources Inc.,2011.

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5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

ACCESSIBILITY

Access to the property is by air from Rankin Inlet, Baker Lake, Thompson, or Churchill, all of which have scheduled airline service and offer a number of facilities. A 500 m dirt airstrip on the large island in central Ferguson Lake is capable of handling wheel-equipped aircrafts. This airstrip is adjacent to the old camp, a private fishing lodge, which was leased, then modified and expanded over the years to accommodate exploration personnel. A new camp was constructed on the mainland on the western shore of Ferguson Lake in 2006 with occupancy commencing in mid 2007. A new, gravel airstrip was constructed in the fall of 2008 from blasted material obtained from a new quarry adjacent to the new camp.

CLIMATE

The area experiences a sub-arctic climate characterized by long winters (October through April) with mean temperatures of -30° C; and a short summer season with mean temperatures in the 15° C range from July through mid-September. Mineral exploration work is most conveniently carried out during the summer months and between March and May when geophysical surveys and diamond drilling can make use of ice-covered lakes.

LOCAL RESOURCES

Limited supplies and services are available in Rankin Inlet and Baker Lake and both of these communities have served as staging points for the supply of Ferguson Lake operations. In addition, staging points for recent programs have been Thompson, Manitoba, 765 km south of Ferguson Lake and Yellowknife, Northwest Territories, 900 km to the west. Both of these communities, with populations of about 15,000, are accessible by highway, have scheduled airline service, and are major exploration and mining supply centres. Previous and current programs involved shipping supplies, equipment and fuel by larger aircraft to an ice airstrip established on Ferguson Lake, and

fuel and other supplies have also been transported to the property by winter Cat train from Baker Lake, Rankin Inlet, and Arviat.

Communications in this remote location are excellent, made possible by satellite, which provides for telephone and high speed internet connections.

INFRASTRUCTURE

Beyond the isolated settlements there is no infrastructure in this area. Although there is no current all weather road link from southern Canada to any community in Nunavut, (and no community-to-community all-weather-road) the Nunavut government has been studying the possibility of extending an all weather road into the territory from northern Manitoba. The potential route would be approximately 150 km east of Ferguson Lake, making a “spur road” to Ferguson Lake a possibility.

Diesel generated electrical power has been used for past mining operations in the area including Cullaton Lake which is 200 km south of Ferguson Lake. Annual re-supply to the communities of Rankin Inlet, Arviat and Baker Lake is done by commercial barge service from Churchill Manitoba or Montreal, Quebec. Daily flights into Rankin Inlet from Southern Canada bring routine supplies and passenger service.

PHYSIOGRAPHY

The Yathkyed-Ferguson-Kaminuriak lakes area is one of low relief, featuring numerous smaller lakes and a few large river systems, notably the Kazan and Ferguson Rivers. Yathkyed and Ferguson lakes are 141 m and 114 m above sea level, respectively. Maximum elevations in the general area range from 200 m to 275 m. Elevations within the current property average less than 200 m and range from slightly less than 100 metres at the property’s eastern boundary to 290 m north of Yathkyed Lake. The orientation of Ferguson and a number of smaller lakes reflects the dominant southeasterly glacial direction. Bedrock is fairly well exposed on numerous low hills and ridges, but in lower areas bedrock may be obscured by between 6 m and 25 m of glacial debris.

The terrain is typical of the barren grounds. Vegetation consists principally of moss, lichen, dwarf birch and Labrador tea. Wildlife includes caribou, Arctic foxes, Muskox, and barren ground grizzly bears. The tree line is 150 km south of Ferguson Lake.

6 HISTORY

Canadian Nickel Company Ltd. (Canico), then the exploration arm of Inco, discovered copper-nickel mineralization at Ferguson Lake in 1950 (McGill, 1955). A 3,000 km^2 concession was granted in late 1950, and work over the ensuing five years included construction of a 90-person all-season camp, airborne and surface geophysics, geological mapping and diamond drilling (Nicholson 2007).

The East Zone, near the east shore of Ferguson Lake (Figure 6-1) was initially tested by 1950s Inco drilling which consisted of 7,115 m in 56 holes completed over a zone strike length of 2,500 m.

An Inferred Resource in the West Zone, consisting of combined copper-nickel grades greater than 1.5%, was initially explored by an unknown number of shallow Inco holes completed between 1951 and 1955. The moderately north-dipping zone was tested by original inclined drill holes at approximately 60-m intervals along north-south section lines 122 m apart. By drilling on section lines midway between the original sections or at 60 m spacings, this program was intended to provide detailed information regarding the down-dip and lateral continuity of the north dipping sulphide horizon(s), in addition to confirmation of original copper and nickel grades, and a more precise assessment of platinum group element and cobalt grades (original Inco results included limited precious metals grades and no cobalt analyses). Further, the program would also provide ample material for metallurgical testwork.

The results of 1950s Inco drilling identified a West Zone resource of 6.4 million tonnes grading 0.87% copper and 0.75% nickel (Nicholson 2007). This is a historical resource estimate as defined in NI 43-101 and cannot be relied upon.

Nearly three-quarters of the total drilling was directed to mineralized zones east and west of Ferguson Lake and the intervening area beneath the lake (East Zone, West Zone and Central or Lake Zone). The remainder of the drilling tested other targets within and outside the original concession area. Standard drilling techniques recovered EX-size (2.23 cm diameter) core (Nicholson 2007).

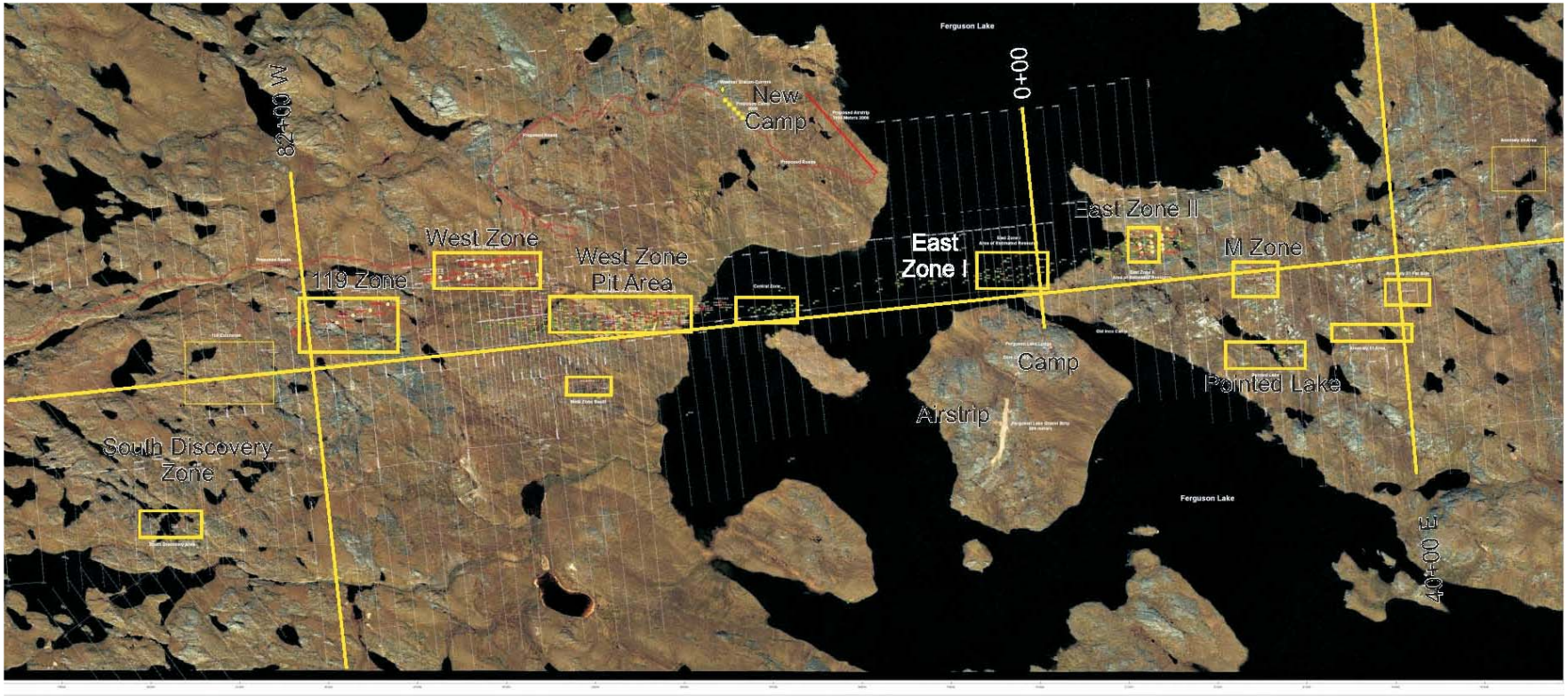

Figure 6-1

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Key Map

November 2011

Source: James G. Lavigne et al., 2007.

www.rpacan.com

Most drilling was on 120 m to 150 m-spaced north-south sections; the distance between holes along these sections averaged about 70 m. Hole lengths averaged 150 m to 180 m, with the deepest hole being 640 m. This drilling did not test the zones below a vertical depth of about 240 m (Carter, May 2006).

A 10-ton bulk sample, extracted from the West Zone in 1953, was transported to Copper Cliff, Ontario for mill testing (Nicholson 2007).

A central 20-claim area (4,180 ha) of the original concession was taken to mining lease in 1957; this was subsequently reduced by 50% in 1978.

Esso Minerals Canada optioned the property from Inco in 1980, and extracted a nine-tonne bulk sample, apparently to test the sulphur content for potential acid generation, to be used for uranium processing at another property under investigation at that time by Esso, elsewhere in the District of Keewatin (Cameron, 1987).

Homestake Mineral Development Company (Homestake) was aware of Pt and Pd values in the area of Ferguson Lake in 1981 (Cameron, 1987), and acquired claims and prospecting permits around the existing Inco mining lease in 1986. A comprehensive exploration program in 1987 consisted of reconnaissance geological mapping and, with Inco's permission, the collection of 339 rock and 266 soil samples mainly from the known East and West mineral zones. Details of most of the work undertaken by Homestake are available in a DIAND assessment report authored by Cameron.

Homestake's mineral claims in the area subsequently lapsed, and the Inco mining lease expired June 17, 1992. A Prospecting Permit covering part of the original Inco property was issued in early February, 1997 and the FERG 1-3 mineral claims were staked in mid-September of the same year. Ten rock and four soil samples were collected from the East and West Zones and from one of the other known mineralized zones at that time (Carter, 1998a).

A 1998 field program, carried out on behalf of the Ferguson Lake Syndicate between mid-August and early September 1998, included the re-establishment of survey control at several points along the 1950s Inco baseline, prospecting and the collection and

analyses of rock samples from the East and West Zones, in addition to several other mineralized zones (Carter, 1998b).

In February, 1999 Starfield entered into an option agreement with the Ferguson Lake Syndicate for the Project. The details of the agreement are contained in Section 4 of this report.

7 GEOLOGICAL SETTING AND MINERALIZATION

REGIONAL GEOLOGY

The Ferguson Lake Project is situated in the Western Churchill Province, an Archean craton which is divided into the lithologically distinct Rae and Hearne domains by the northeast-trending Snowbird Tectonic Zone (Figure 7-1).

Ferguson Lake, 100 km east of the Snowbird Tectonic Zone, is more precisely within the northwestern Hearne domain which is made up principally of Archean metavolcanic and metasedimentary rocks and areally extensive gneissic terranes derived from both Archean volcanosedimentary and plutonic rocks and early Proterozoic plutonic rocks (Miller, 2005b). The northwestern Hearne domain is bounded by northeast-trending, regional shear zones including the Tulemalu Fault Zone (part of the Snowbird Tectonic Zone) on the north and by the northeastern extension of the Tyrrell Shear Zone on the southeast.

The Ferguson Lake area includes the most northerly extension of the northeast-trending Yathkyed greenstone belt (Martel and Sandeman, 2004) here mainly present as strongly deformed, gneissic Archean supracrustal and intrusive rocks and variably deformed Proterozoic plutons and dykes. The deformed sequences are metamorphosed to upper amphibolite facies. Protoliths of the older supracrustal rocks are comparatively rare. Where seen, they consist principally of mafic metavolcanics with cherty iron formations and lesser intermediate to felsic metavolcanics and clastic metasedimentary rocks.

The widespread Archean gneissic rocks are intruded by Archean granodiorites, quartz monzonites and a variety of mafic intrusions including diorites and gabbros. Late Archean intrusions include the east- to northeast-trending Kazan dykes (Eade, 1986) which consist of variably metamorphosed gabbros and hornblendites.

Early Proterozoic (Tulemalu dykes - Eade, 1986) gabbros and slightly younger diabase dykes cut all older rocks as do late Proterozoic syenites and lamprophyres. The Martell Syenite (Bell, 1971; Eade, 1986) is an example. A large (13 km by 5 km) pluton of Martell Syenite, centred on Uligattilik Hill several km east of the Ferguson Lake Project,

is characterized by a positive magnetic anomaly on published airborne magnetic survey results for map-area 65I. As described by Bell (1971), this pluton consists of massive, uniform, biotite-pyroxene-amphibole syenite in which apatite is a common accessory mineral. Biotite-rich mafic dykes, prevalent within the property area, may be related to this intrusive event.

The Western Churchill Province, because of its diverse geological environments which span a 1.5-billion-year interval, is host to a variety of mineral deposit types (Miller, 2005b). Known mineral deposits, prospects and occurrences include mafic- to ultramafic-related magmatic nickel-copper-cobalt-PGE, orogenic (mesothermal) lode gold, volcanic-hosted massive sulphides, syngenetic and epigenetic uranium deposits and prospects, quartz-carbonate veins containing precious metals and diamonds associated with Phanerozoic kimberlite intrusions.

Mineral exploration efforts in the general area in the recent past have been directed to orogenic iron formation-hosted gold mineralization, and volcanic hosted massive sulphides in the Yathkyed and Ennadai-Kaminak-Rankin greenstone belts, and to the search for diamonds over a much broader area. Past mining operations in this part of Nunavut include the Cullaton Lake gold mine and the North Rankin Nickel Mines Ltd. underground mine (within the present community of Rankin Inlet) which operated between 1957 and 1962. Production amounted to 460,000 tonnes of ore with recovered grades of 2.1% nickel and 0.6% copper. Sulphide mineralization consisted of pentlandite, chalcopyrite, pyrrhotite and pyrite near the base of a serpentinized pyroxenite sill intruding Archean metavolcanic rocks.

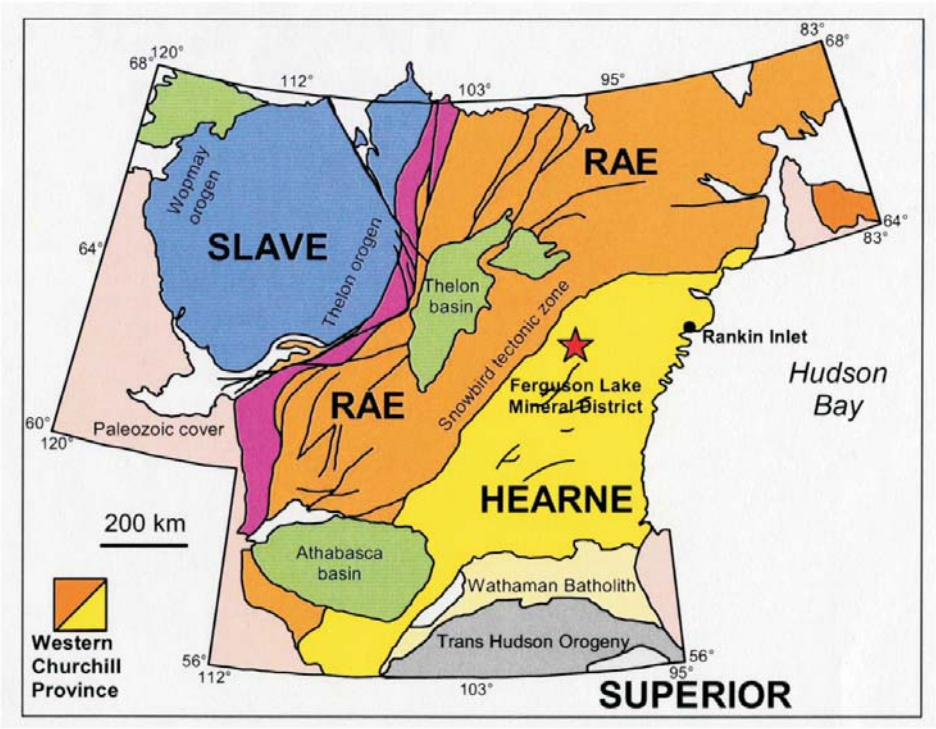

Figure 7-1
Regional Geology

Ferguson Lake
Regional Tectonic Setting
(Source: Hanmer, et al., 2004)

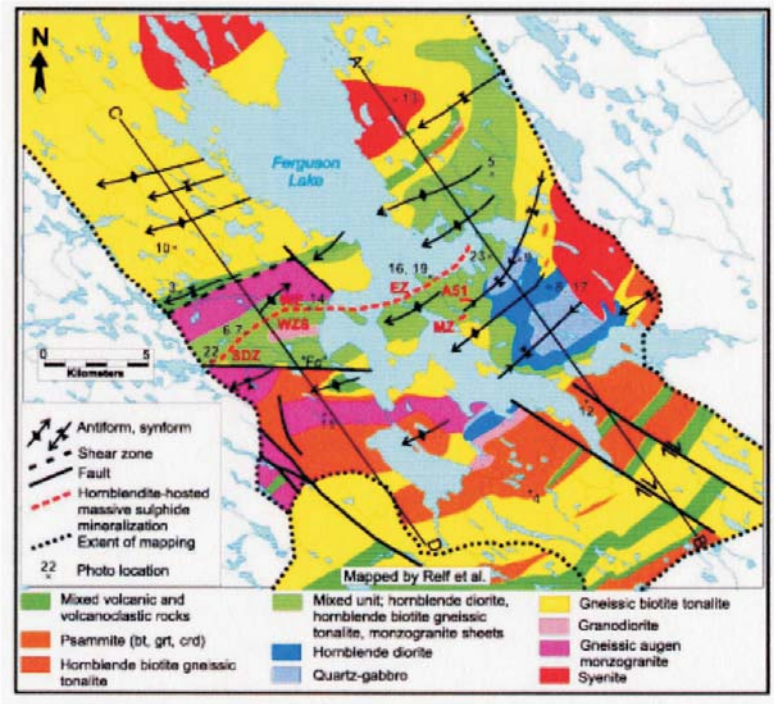

Figure 7-2
Property Geology

Generalized Geological Setting,
Central Ferguson Lake Property
(Source: Martel & Sandeman, 2004)

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Regional & Property Geology

November 2011

LOCAL AND PROPERTY GEOLOGY

The oldest rocks in the southern and northeastern parts of the Ferguson Lake Project are east- to northeast-trending, fine- to medium-grained amphibolites which are the metamorphic products of original mafic and intermediate volcanic rocks of Archean age. These supracrustal rocks, which contain sulphide, oxide and silicate banded iron formations in a number of localities, are interlayered with more widespread quartz-feldspar-biotite (hornblende) gneisses and paragneisses. All the units have been intruded by Archean tonalities, granite gneisses and smaller, complex, coarse-grained pegmatite bodies. A variety of younger (Proterozoic) dykes, sills and irregular intrusions cut the older rocks.

Pronounced layering in the supracrustal rocks trends east-northeast to northeast and dips moderately to steeply north. Medium- to coarse-grained, massive to weakly foliated gabbros, containing +60% hornblende and termed hornblendites in earlier reports, mainly occur within, and are conformable with, the layering in amphibolite-hornblende-biotite gneiss sequences. Petrographic studies suggest that these hornblende-rich gabbros, which are the principal host rocks for base metal sulphides and platinum group elements, may be metamorphic products of original tholeiitic mafic or ultramafic (pyroxenite-peridotite) intrusions.

All of the foregoing lithologic units, including the host gabbros, are cut by younger (mid-Proterozoic) gabbros and diabases and by late Proterozoic syenites, quartz-feldspar porphyries and fine-grained, locally biotite-rich mafic dykes.

Younger syenites, part of the Martell syenites, and distinctly post-mineral mafic dykes are also evident in many of the diamond drill holes completed to date. Larger bodies of this syenite occur near the east shore of Ferguson Lake. A larger body underlying Uligattilik Hill borders the eastern property boundary.

A structural mapping program in the areas of the East and West zones (Henderson, 1999), indicated that most of the foregoing lithologic units, including the granitic intrusive rocks but excluding the younger gabbro, diabase and mafic dykes and syenite plutons, were subjected to high grade metamorphism and deformation. Intricate folding of the gneissic rocks and the hornblendites (gabbros) has produced antiform and synform

structures which are particularly evident in the area east of Ferguson Lake. The East and West mineralized zones were interpreted as being within the south limb of a recumbent, doubly-plunging synform or canoe-shaped structure modified by numerous faults and shear zones which offset the various lithologic units.

Continuing geological studies over the past four years have provided more information regarding the setting and nature of the mineralized zones in the central part of the Ferguson Lake Project. These include 1:20,000-scale bedrock mapping under the auspices of the Canada-Nunavut Geoscience Office in 2003 (Martel and Sandeman, 2004), and more recent studies in the general area by Miller (2005a, b).

As illustrated in Figure 7-2 and described by Martel and Sandeman (2004), “rocks in the Ferguson Lake map area consist of multiply-deformed Archean gneissic metavolcanic and metasedimentary rocks, and associated tonalitic, granitic and gabbroic rocks, which have been metamorphosed to upper amphibolite facies. Variably deformed Proterozoic rocks include late gabbros, and syenite plutons and associated lamprophyre dykes.” Massive sulphide mineralization is described as being hosted by a hornblendite layer conformable with the regional stratigraphy. Three generations of deformation are recognized and the authors report an early phase foliation in the host hornblendite unit and suggest that it and the contained sulphide mineralization were subjected to two subsequent phases of folding with the first of these represented by the numerous northeast-southwest structures shown on Figure 7-1. As such, mineralization at Ferguson Lake is inferred to be of Archean age.

Miller (2005a) suggests that the host gabbroic units in both East and West sulphide zones at Ferguson Lake are a component of metamorphosed compositional layering. This compositional layering in the East Zone is represented by mesocratic through leucocratic gabbro and into anorthosite which is commonly garnet-bearing. At the West Zone, compositional layering ranges from pyroxenite through mesocratic and leucocratic gabbro. Miller further suggests that original magma underwent fractional crystallization within several chambers to form layered mafic and ultramafic rocks and proposes that the Ferguson Lake Intrusion might best be termed the Ferguson Lake Intrusive Complex.

Miller also suggests that the host rocks and contained mineralization are of Archean age but correctly points out that there are no geochronological dates to substantiate this hypothesis.

The main gabbro unit, which hosts all of the known sulphide zones including the East, Central (lake) and West Zones (Figure 7-2), exhibits a fair degree of continuity and predictability over an east-west strike length of more than 12 km. This linear feature, which significantly trends only slightly north of east as opposed to the dominant northeast structural trend of the surrounding area, suggests that the host intrusion post-dates much of the intense Archean deformation and metamorphism evident in the surrounding gneissic rocks.

It should noted that recently completed drilling on the 119 Zone Extension indicates that the contained massive sulphide mineralization, the like other mineralized zones, is associated with coarse-grained hornblendites.

MINERALIZATION

Nickel-copper-cobalt-PGE mineralization at Ferguson Lake is hosted mainly by fine- to coarse-grained gabbros which include hornblendites. Three of the mineral zones (East, Central, and West) are at least spatially related to the same gabbro unit which is between 10 m and 600 m thick and has been traced by intermittent exposures and by diamond drilling over a strike length of more than 12 km east and west of Ferguson Lake. This and the other gabbro units hosting the several other mineral zones dip moderately to steeply north and are generally conformable with enclosing hornblende-rich gneisses (Carter, May 2006).

Better grades of base and precious metals mineralization are present within irregularly shaped, laterally persistent, massive to semi-massive sulphide lenses, pods and stringers which consist of between 80% and 90% pyrrhotite and lesser chalcopyrite, some pyrite and very fine-grained pentlandite. Locally, multiple “stacked” lenses occur vertically over several metres to tens of metres. Rounded magnetite grains, up to one centimetre in size, are a common constituent of the sulphide lenses. Better grades are contained within zones having thicknesses of between two and tens of metres.

Sulphide matrix breccias, featuring one centimetre to two centimetre subrounded mafic clasts, are a common feature of massive sulphide zones. Net textures have been noted in some of the stringer and fracture-filling sulphide sections. The sulphide zones occur mainly in the upper structural hangingwall portion of the of north-dipping gabbro units and to a lesser degree as remobilized lenses within hangingwall and footwall gneisses. The sulphide-rich zones are marked on surface by prominent gossans up to 25 m wide and several hundred metres long.

The South Discovery Zone, a sulphide-bearing gabbro unit some three kilometres southwest of the 119 Zone has an exposed northeast strike length of 800 m and is offset by two parallel, northwest striking faults. Tow holes drilled to test the central part of the zone in 1999 intersected 1.5 m to 2.0 m intervals grading 0.30% Cu to 0.62% Cu, 0.11% to 0.50% Ni, 0.02% Co to 0.10% Co, and 0.13 g/t PGE to 0.67 g/t PGE, while core hole FL10-429 drilled in 2010 intersected 1.09 m averaging 0.15% Cu, 0.12% Ni, and 0.02% Co.

The principal focus of previous and current work has been directed to the West Zone which has been traced by intermittent bedrock exposures, geophysical surveys and by a considerable amount of diamond drilling over a zone length of several kilometres.

Other mineral zones identified east of Ferguson Lake include the M Zone, discovered in 2000 and situated one kilometre southeast of East Zone II (Figure 7-2). This zone was drilled by ten inclined holes to test a blind, gently north-dipping UTEM conductor. The drilling confirmed that the conductive zone was due to the presence of sulphide minerals within a gabbro, a setting similar to the other known mineral zones. Assays results were not particularly impressive. Further surface geophysical surveys were conducted over this zone in 2004 and three additional holes, drilled to test a new interpretation of geophysical signatures, returned results similar to those obtained from earlier drilling.

Located one kilometre south of the M Zone is Anomaly 51, also known as the Pointed Lake Zone. This zone includes a northeast trending, 50 m wide gossanous gabbro exposed for over 2,500 m of strike length. It contains pods and stringers of massive pyrrhotite with lesser chalcopyrite and pyrite, particularly in its western half. In the 1950s, this zone was tested over 1,700 m of strike length by nine widely-spaced, inclined diamond drill holes totalling 1,094 m of drilling. Several holes contained narrow intervals

of copper-nickel mineralization. Better grades (over 1% combined copper-nickel) are associated with narrow intervals (0.15 m to 2.29 m) of massive pyrrhotite and pyrite hosted by gabbro in most holes drilled. No results for PGEs were reported. Nickel values were generally higher than copper as opposed to results from other zones at Ferguson Lake.

This zone (and its potential extensions) was further tested in 2000 and 2004 by eight holes which were drilled in several locations to test isolated UTEM conductors north and east of the area previously drilled. Only narrow sulphide intervals were encountered.

Most exploratory work to date (and 80% of diamond drilling since 1999) has been directed to several mineral zones west of Ferguson Lake. Those zones of apparent lesser importance include the West Zone South, which is intermittently exposed over a strike length of approximately one kilometre. Associated with a gossanous gabbro, this zone is subparallel to the West Zone and located one kilometre to the south. Previous surface sampling returned relatively high PGE values of between 540 ppb Pt and 1,170 ppb Pt and 1,250 ppb Pd to 4,500 ppb Pd. One Inco hole and five holes drilled in 1999 and 2000, designed to test a strong UTEM conductor and a coincident magnetic signature, intersected narrow (0.3 m to 1.5 m) intervals of copper-nickel-PGE mineralization. Better copper-nickel grades are restricted to stringer and massive pyrrhotite, pyrite and chalcopyrite; three narrow mineralized intervals in hole FL00-23 contained PGE values of 2 g/t to 3 g/t, further confirming earlier surface sampling results.

The South Discovery Zone, a sulphide-bearing gabbro unit some three kilometres southwest of 119 Zone has an exposed northeast strike length of 800 m and is offset by two parallel, northwest-striking faults. Two holes drilled to test the central part of the zone in 1999 intersected 1.5 m to 2.0 m intervals grading 0.30% Cu to 0.62% Cu, 0.11% Ni to 0.50% Ni, 0.02% Co to 0.10% Co, and 0.13 g/t PGE to 0.67 g/t PGE.

The Central or Lake Zone, underlying Ferguson Lake between the East and West Zones, was tested by 35 vertical and inclined drill holes in the 1950s. The majority of holes completed intersected at least some copper-nickel values and ten holes contained significant results. One of the better holes (10515), drilled near the centre of the lake north of, and between the large and small islands, intersected 17.4 m grading more than 2% combined copper-nickel and 1.51 g/t PGE. Two holes drilled down-dip (10518,

10520) returned no significant values, however, hole 9924, drilled 250 m further west, intersected 3.4 m of 0.54% Cu, 0.79% Ni, and 5.31 g/t PGE. There is a clustering of better holes immediately north of the small island; three of these have five metre or greater intervals with grades of up to 1.96% Cu, 1.01% Ni, and 1.95 g/t PGE.

Near the west shore of Ferguson Lake, hole 11310 (on grid section 36+00W) cut 13.5 m grading 0.78% Cu, 0.88% Ni, and 1.44 g/t PGE, in addition to a lower 5.8 m interval containing 1.36% Cu, 0.50% Ni, and 0.69 g/t PGE. Two holes were drilled on ice-covered Ferguson Lake by Starfield in the spring of 2000 in an attempt to further test this part of the zone some 40 m down-dip of the previous intersections. The first hole was abandoned in lake-bottom sediments; the second hole intersected gabbro host rocks over three intervals but base and precious metals value were low. This zone was further tested by ten holes in 2003 and 2004 as part of the definition drilling program undertaken in the eastern part of the West Zone. A degree of continuity of grades in sulphide lenses was identified and a part of this zone has been incorporated into the revised resource estimates for the West Zone.

The West Zone proper has been the focus of most of the exploratory programs undertaken between 1999 and 2008 during which time the zone was tested by more than 150 holes plus five wedge holes (62,000 m) drilled over three kilometres of strike length. Much of this drilling was designed to expand the zone both to depth and along strike with a number of deeper holes testing the deeper, western part of the zone between sections 52+00W and 68+00W in 2001. Most of these holes intersected several ten metre intervals containing combined copper-nickel greater than 1%, most of which included significant sections of combined copper-nickel grades greater than 2%, in addition to PGE grades averaging more than 2 g/t.

Much of the drilling undertaken on the West Zone between 2002 and 2004 was directed to definition drilling in 74 relatively shallow holes, totalling 20,200 m, in the eastern part of the zone between sections 39+00W and 51+000W to better define near-surface sulphide mineralization.

The nature and scope of the 2002–2004 drilling programs consisted of collaring drill holes between existing holes such that this part of the West Zone has now been drilled on sections at an average of 30 m spacing or less (Figure 7-3). Most holes drilled

intersected two or three (and in some cases, up to six), parallel sulphide lenses containing grades of at least 1% combined copper plus nickel and over minimum hole lengths of two metres. Intervals of lower (or zero) grades between the parallel lenses range from two metres to as much as 65 m. Most of the inclined holes drilled to date are essentially normal to the moderately north-dipping sulphide lenses and apparent true widths of the various sulphide lenses range from two metres to a maximum of 45 m with an overall average of seven metres. The down-dip extent of the various sulphide lenses is variable and ranges from 20 m to 250 m. Many of the drill sections feature a well mineralized sulphide lens in one hole that may or may not continue through holes drilled up or down-dip of the mineralized lens. The best up and down-dip continuity of sulphide lenses is apparent between the following sections: 41+45W and 42+05W, 42+90W and 43+50W, 44+50W and 47+60W and 48+20W and 51+20W. A drill section illustrating the distribution of sulphide lenses and the low sulphide PGE mineralization described below is included as Figure 7-4.

In addition to better defining the sulphide zone, this program of definition drilling was also successful in identifying a unique style of low-sulphide PGE mineralization hosted by footwall gabbros some 50 m below the sulphide horizon(s). This style of mineralization was first recognized at depth in hole FL01-104 (section 63+00W) below the massive sulphide lens where a 0.50 m interval contained 9.86 g/t palladium and 1.44 g/t platinum accompanied by low copper, nickel and cobalt values. The shallower, eastern part of the West Zone, is characterized by dispersed biotite alteration and fine-grained disseminated pyrite. Palladium values in these zones range from 100 ppb to 500 ppb over several tens of metres within which higher PGE grades of up to tens of grams per tonne occur over narrow intervals.

The distribution of low sulphide, footwall mineralization containing PGE values of more than several grams per tonne over minimum hole lengths of one metre appears to be somewhat erratic. These occur at vertical depths of between 100 m and 200 m below surface with four distinct clustering's between sections 40+50W and 41+50W, 42+45W and 43+500W, 45+50 and 46+00W and 48+50W and 49+00W.

Holes drilled in 2007 and 2008 were designed to test the low sulphide PGE footwall mineralization as well as infill within intersections in massive sulphide within the West Zone.

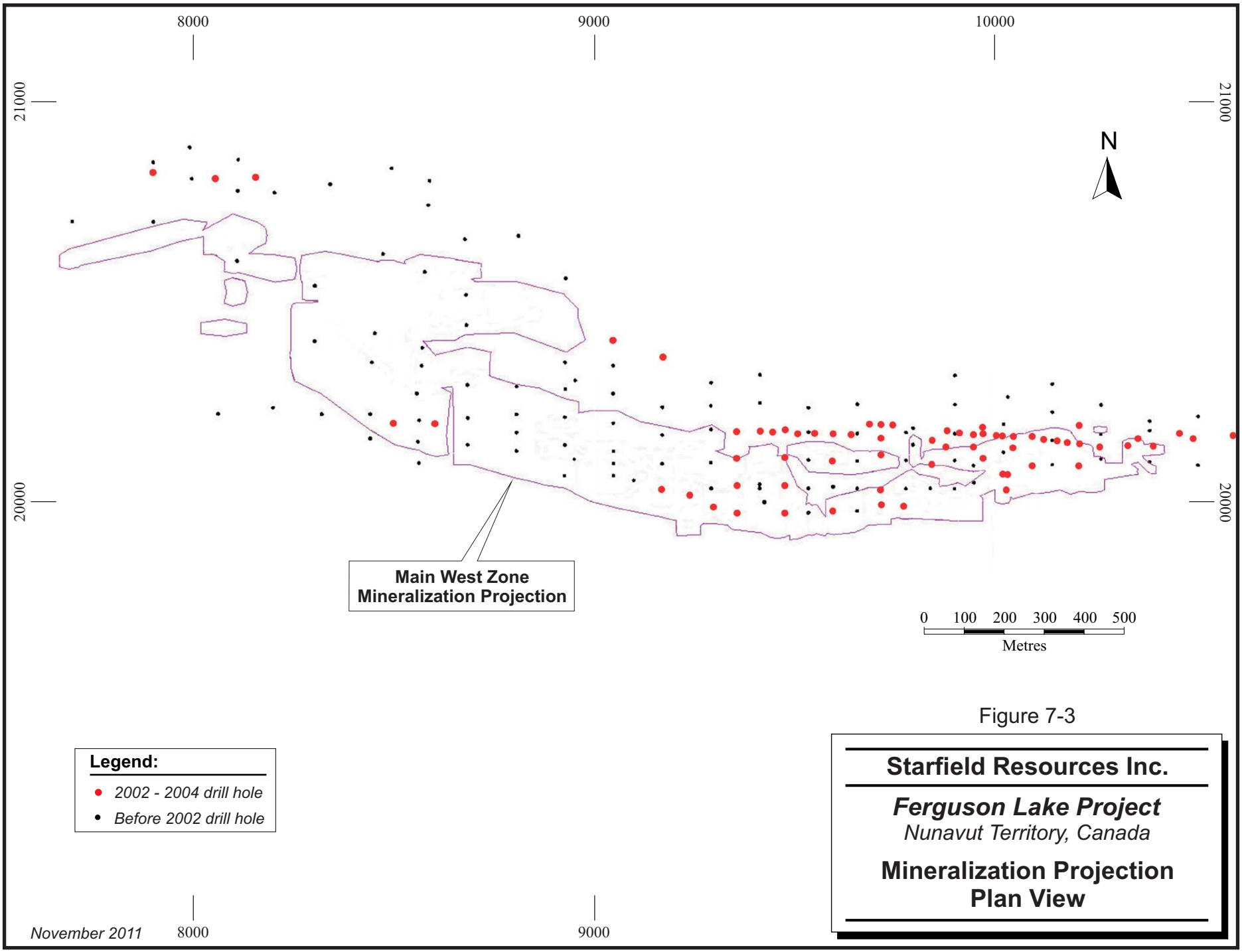

Figure 7-3

Starfield Resources Inc.

Ferguson Lake Project

Nunavut Territory, Canada

Mineralization Projection
Plan View

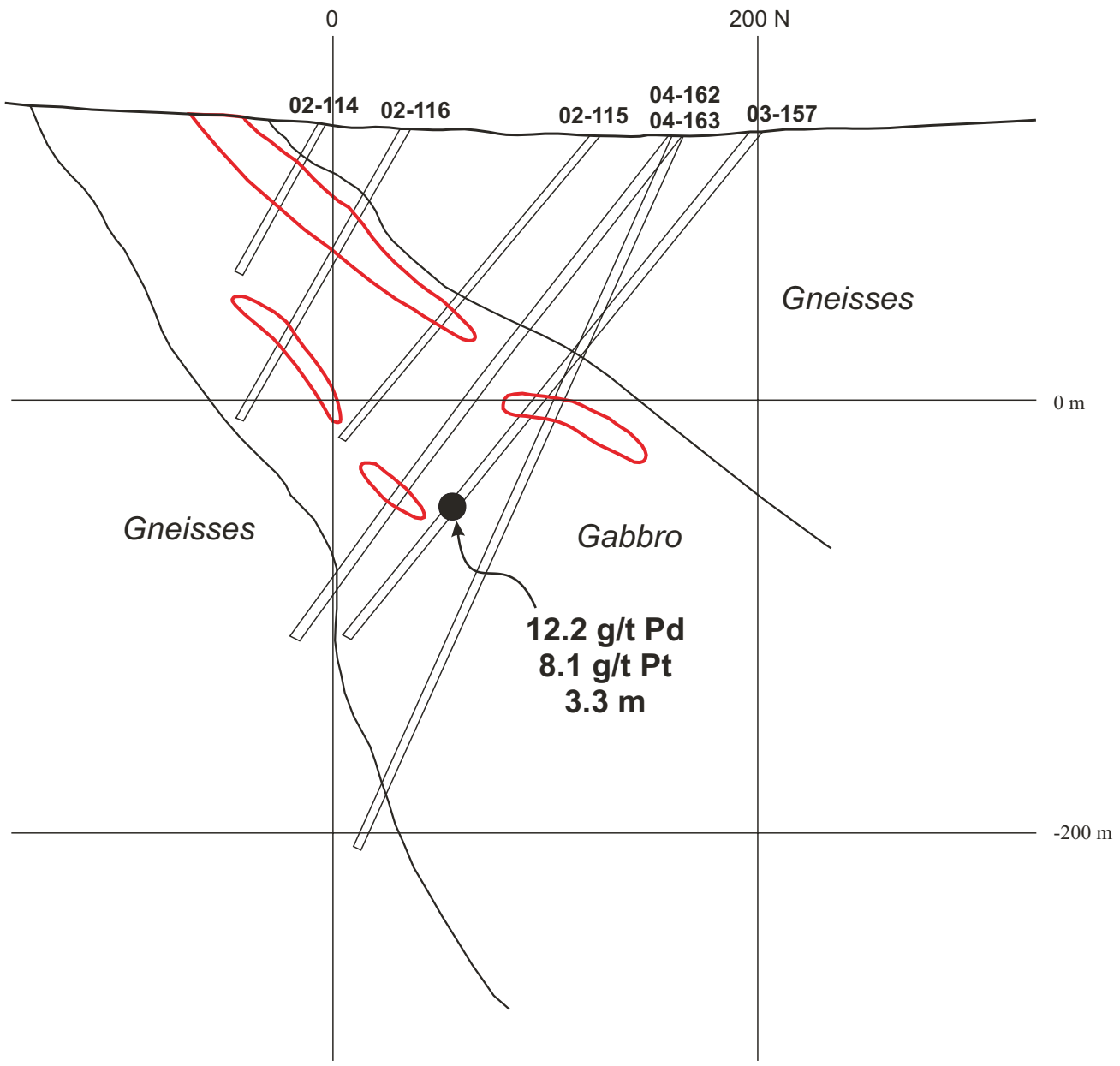

Figure 7-4

Starfield Resources Inc.

Ferguson Lake Project

Nunavut Territory, Canada

Massive Sulphide and Low Sulphide PGE Mineralization

Legend:

+1% Cu+Ni Contour

0 20 40 60 80 100

Metres

November 2011

Source: N.C. Carter, 2006.

The 119 Zone, between sections 76+00W and 80+00W or between 800 and 1200 m west of previous West Zone drilling, was discovered in 2002 by drilling a continuation of the deep-seated UTEM conductor and a coincident inverted magnetic anomaly. The zone includes parallel massive sulphide lenses featuring base and precious metal grades that are generally higher than those previously encountered in West Zone.

Eight holes totalling 10,433 m were drilled to test the 119 Zone in 2002. Six of the holes intersected two or more, apparently parallel sulphide lenses; the other two holes were drilled in the footwall of the zone. Characteristics of the individual massive sulphide lenses in the 119 Zone included a marked stratification of sulphide minerals with a fining to coarsening of grain size from top to bottom.

The 119 Extension Zone, situated about one kilometre southwest of the 119 Zone proper, was identified as a conductive zone by a deep penetrating UTEM survey over a southwest trending magnetic feature which suggested a linkage between 119 and South Discovery zones. Two holes and two wedges off the first hole (04-174) tested this zone in 2004. The first hole and the two wedges intersected intervals of massive sulphides over hole lengths of between 1.65 and 8.10 m at hole depths exceeding 1,200 m. The best values, in excess of 1.5% copper plus nickel, were obtained from the second wedge hole. A second deep hole (04-200) collared several hundred metres southwest, intersected sulphides over a 30-metre hole length at depths approaching 1,200 m; combined copper and nickel values were generally less than 1% except over hole intervals of less than one metre.

Core hole FL11-432, which was drilled in 2011 to a final depth of 1,229 m (4,032 ft), is located on the 119 Zone Extension approximately mid-point between holes FL04-174 and FL04-200 intersected three lenses of massive sulphide mineralization between 1,096.5 m and 1143.9 m. These intersections confirm the continuity of the massive sulphide mineralization between holes FL04-174 and FL04-200, as well as extend ore grade mineralization approximately 350 m (1,148 ft) southwest of FL04-174. The key mineralization intercepts are as follows:

- 1,096.49 m-1,098.52 m: 2.03 m - 0.03 g/t Pt, 1.28 g/t Pd, 1.19% Cu, 0.58% Ni, 0.07% Co.
- 1,106.45 m-1,112.10 m: 5.65 m - 0.32 g/t Pt, 1.73 g/t Pd, 1.23% Cu, 0.64% Ni, 0.07% Co.

- 1,118.70 m-1,124.66 m: 5.96 m - 0.11 g/t Pt, 1.86 g/t Pd, 1.33% Cu, 0.73% Ni, 0.04% Co.
- 1,125.60 m-1,126.09 m: 0.63 m - 0.47 g/t Pt, 4.93 g/t Pd, 0.68% Cu, 0.26% Ni, 0.03% Co.

8 DEPOSIT TYPES

The various mineral zones identified to date in that part of the property bordering Ferguson Lake are magmatic nickel-copper sulphide deposits which also contain cobalt and PGE values (Carter, May 2006). These zones are hosted by mafic (and ultramafic) intrusive rocks which are principally in the form of fine- to coarse-grained gabbros. The nature of the Ferguson Lake deposits, as currently understood, appear to conform with Eckstrand's (1996) subtype 27.1d - "other tholeiitic intrusion-hosted nickel-copper" deposits which are described as being associated with mafic and ultramafic phases of differentiated intrusions. Sulphide minerals present in this deposit subtype include abundant pyrrhotite with subordinate pentlandite, chalcopyrite and pyrite which are present as massive lenses, sulphide matrix breccias, net-textured fracture fillings and as disseminations. Nickel-copper ratios range from 3:1 to less than 1:1 (Eckstrand, 1996).

Intrusion hosted nickel-copper deposits are the primary target on the Ferguson Lake property, however, more recent exploration efforts on the property have also been directed at diamond deposits hosted by kimberlites. In addition, Starfield has evaluated some geophysical anomalies for their gold and base metal potential. In these cases, the deposit types are iron formation hosted gold deposits and volcanogenic massive sulphide (VMS) base metal deposits, respectively.

9 EXPLORATION

Starfield entered into an agreement with the Ferguson Lake Syndicate in February 1999 and undertook a two-phase exploratory program in the spring and summer of the same year (Carter, 1999a, b). Work completed included the establishment of 170 km of survey grid off an east-west baseline with north-south crosslines at 200 m spacings, airborne and surface geophysical surveys, detailed geological mapping, prospecting and surface sampling, preliminary environmental baseline studies and 3,918.5 m of diamond drilling in 19 holes.

Exploratory programs in 2000, undertaken between April and early December, consisted of expanded geophysical surveys (UTEM and magnetics) over 170 km of grid, and the testing of six mineralized zones by 15,600 m of diamond drilling in 49 holes (Carter, 2000b; 2001a).

The 2001 program was designed to confirm and expand upon results obtained from drilling on the West Zone in late 2000. Several holes from this program, which tested the zone to depth, returned markedly higher copper and nickel grades accompanied by enhanced concentrations of cobalt and PGEs. The initial objective of the 2001 program was to explore this higher grade zone to depth and along strike, and to further test the strong UTEM conductor which indicated continuity of the West Zone to depth and along strike for a considerable distance to the west. Drilling consisted of 28 inclined holes for a total of 18,570 metres. Further interpretation of UTEM geophysical data for the M Zone, east of Ferguson Lake, suggested that the previous drill holes may have been too shallow to properly test this zone. An additional five holes (1,871 m) were completed on this zone and four holes (635 metres) were drilled to test a UTEM conductor immediately east of the East Zone II resource area. Total diamond drilling completed between April and October of 2001 amounted to 21,076 m in 37 holes. Prospecting and bedrock sampling was also undertaken to further assess a number of additional prospective areas within the large Ferguson Lake claim block between July and September of 2001. Additional sampling of a number of West Zone drill holes was undertaken in October and early November to further investigate the potential for higher grades of palladium and platinum which had been encountered in the last two holes (FL101 and FL104) completed on the West Zone (Carter, 2002).

Work in 2002, consisting mainly of diamond drilling, was directed entirely to the West Zone and its western extension. The program on the West Zone proper consisted of 10,952 m of drilling and included five wedge holes off hole FL01-101, an additional deep hole on section 64+00W and definition drilling (39 inclined holes) in the eastern part of the zone. The initial objective of this definition drilling was to further define near-surface sulphide resources over a 1,000 m section of the eastern part of West Zone ("Pit Area") where an inferred resource with combined copper-nickel grades greater than 1.5% was initially explored by a number of shallow Inco holes. These holes were completed in the early 1950s at 60 m intervals along north-south section lines 122 m apart. The 2002 drilling, on section lines midway between the original sections or at 60 m spacings, was intended to:

- Provide detailed information regarding the down-dip and lateral continuity of the north-dipping sulphide horizon(s).
- Confirm the original copper and nickel grades.
- Provide a more precise assessment of platinum group element and cobalt grades (original Inco results included only limited precious metals grades and no cobalt analyses).
- Provide ample material for anticipated metallurgical testwork.

Eight holes totalling 10,433 m drilled between 800 m and 1,200 m west of previous West Zone drilling resulted in the discovery of the 119 Zone. Other work completed as part of the 2002 program included the collection and analyses of 138 soil samples from the eastern part of the West Zone and various metallurgical studies. Results of the 2002 program are contained in Carter (2003).

An exploration program in 2003 included the completion of nine drill holes totalling 2,667 m to further test both massive sulphide lenses and low-sulphide platinum-palladium mineralized horizons in footwall gabbro in the "Pit Area" or eastern portion of the West Zone.

The 2003 program also included 55.5 line km of UTEM geophysical survey over a trend of magnetic anomalies interpreted to extend southwest from the 119 Zone. A geological mapping program of the then property area was undertaken by Geological Survey of Canada personnel (Martel and Sandeman, 2004).

Exploration work on the Ferguson Lake Project in 2004 included 23,018 m of diamond drilling in 52 holes plus various airborne, surface and geophysical surveys. The West Zone and its extensions were the focus of much of the 2004 work but the area east of Ferguson Lake also received some attention. Similar to the two previous seasons, the bulk of diamond drilling (more than 65%) was directed to additional definition drilling of near surface sulphide mineralization in the West Zone "Pit Area". Nearly 14,000 m of drilling was completed in 41 inclined holes in this area between sections 35+67W and 48+80W. Several of the easternmost holes were drilled beneath Ferguson Lake while the remainder served to increase the density of drill hole spacing within previously drilled areas. In addition to further assessing the structural and grade continuity of the massive sulphide lenses, this program provided additional data pertaining to the nature and distribution of platinum group element mineralization associated with low sulphide zones developed in gabbroic footwall rocks.

Four holes (1,800 m) were drilled in the western half of the West Zone. Two deep holes and two wedges (3,950 m) were drilled to test the apparent southwest extension of the 119 Zone. Drilling (1,700 m) completed east of Ferguson Lake included three holes on the M Zone and two holes in the area of Anomaly 50 as part of a joint venture with Wyn Developments Inc.

Geophysical surveys undertaken in 2004 included a helicopter-borne VTEM and magnetic survey completed by Geotech Ltd. over a 43 km^2 area containing known mineralized zones east and west of Ferguson Lake in the central part of the property. A surface and borehole pulse electromagnetic survey using the time domain method in various parts of the property was carried out by Crone Geophysics and Exploration Ltd. Referred to as SQUID (Superconducting Quantum Interference Devices), the system has the capability of identifying conductive targets at depths exceeding 700 m.

Other work completed as part of the 2004 program included ongoing metallurgical testwork on drill core samples directed to the recoveries of platinum group elements.

Exploration work in 2005 included 16,861 m of diamond drilling in 29 holes plus surface and airborne geophysical surveys. Ten deep holes (11,213 m) were drilled on three sections to test the "gap" between the 119 Zone and the known western limits of the West Zone. The 119 Zone itself was tested by an additional two holes (2,116 m). A total

of 17 holes (3,523 m) were completed in the West Zone "Pit Area". Eight holes were drilled in an attempt to further define footwall PGE mineralization while the remaining nine holes (1,140 m) were drilled at various azimuths to provide information on the continuity and consistency of base and precious metal grades for a geostatistical study (Clarke, 2006).

Geophysical surveys undertaken in 2005 included 9,624 line km of helicopter-borne VTEM and magnetics, and surface and borehole time domain pulse electromagnetic surveys on various parts of the property by Crone Geophysics and Exploration Ltd.

Other work completed in 2005 included ongoing metallurgical testwork on drill core samples directed to the recovery of base metals and platinum group elements.

Much of the 2006 exploratory program at Ferguson Lake was directed at detailed infill diamond drilling of the West Zone Pit, West Zone Mid and Pit Extension, West Zone Main and East Zone II (Nicholson 2007). Some 24,330 metres of core was recovered from 110 holes as shown in Table 9-1.

TABLE 9-1 SUMMARY OF 2006 DIAMOND DRILLING
Starfield Resources Inc – Ferguson Lake

Zone	No. of Holes	Total Metres
West Zone Pit	53	8,884
West Zone Mid	21	3,771
West Zone Main	21	10,078
East Zone II	15	1,597

Two holes, totalling 211 m, were drilled to collect samples for metallurgical work, and 19 shallow holes totalling 234 m were drilled for geotechnical purposes. An additional six holes, totalling 657 m, were drilled at azimuth grid north to test a south dipping magnetic anomaly (termed the Clarke Plate) located in the West Zone Pit Area. In summary, 2006 diamond drilling at Ferguson Lake totalled 24,987 m.

REGIONAL AND OTHER PROGRAMS

In addition to the previously mentioned airborne geophysical survey, preliminary prospecting and till sampling were undertaken in 2005 and 2006, predominantly north

and south of the central property area. In total, some 1,532 till and 739 rock samples were collected. In addition, a new all season camp was constructed 1.5 km north of the West Zone. The mineral claims hosting the identified mineral resources, the new camp facilities and the proposed new gravel airstrip have been legally surveyed by McElhanney Land Surveys Ltd. (McElhanney) in preparation for application for mining lease status for this portion of the property. As well, Rescan Environmental Services Ltd. (Rescan) continued ongoing water quality baseline sampling, soil and botanical surveys, project site planning and archaeological investigations (Nicholson 2007).

In December 10, 2008, Starfield announced the discovery of a micro-diamond by SGS in one of the 1,135 till samples taken from the property during the 2005 to 2006 period.

In early 2009 a 5,193 line-km DIGHEM airborne geophysical survey was performed over the south-central portion of the Ferguson Lake property. Numerous anomalies indicative of possible kimberlitic bodies were identified for ground work follow-up. A total of ten ground grids were surveyed by ground magnetics. Seven diamond shallow core holes were completed to test for the source of the geophysical anomalies. This part of the 2009 exploration program was conducted by or for Thanda as part of the Joint Venture Agreement and formed their initial diamond exploration work on the Ferguson Lake property. An eighth diamond drill hole designed to test a BIF possessing weak base metal values in rock chip samples and associated with the intersection of two conductivity anomalies was also completed as part of Starfield's ongoing exploration in the Y Lake area.

During the 2010 exploration program, drilling operations were initiated on September 2 and completed on September 27. During the program, six holes totalling 1,126 m were attempted and five were completed. Four of these holes are located in the Y Lake area, while two holes (FL10-428 and FL10-429) tested the South Discovery Zone down dip of the 1999 drill holes.

The 2011 exploration program consisted of three holes totalling 1,866 m. During this program one hole, FL11-430, was completed in the western part of the West Zone; one hole, FL11-431, tested a gossanous zone associated with a conductivity anomaly located southwest of the South Discovery Zone and one hole, FL11-432, further evaluated the deep massive sulphide mineralization in the 119 Zone Extension.

10 DRILLING

Diamond drilling completed by Starfield at Ferguson Lake between 1999 and 2011 has amounted to more than 162,000 m in 445 holes. The contractor for all drilling programs to date has been Major Midwest Drilling. NQ-size (4.76 cm diameter) and some BQ thin-wall size (4.20 cm diameter) holes were drilled. Drill recoveries have been excellent, generally close to 100%.

Locations and details of holes drilled up to 2006 are included in the report authored by Nicholson (2007). The drill programs completed at Ferguson Lake since 2006 are described in the following subsections.

2006 DRILLING

WEST ZONE PIT DEFINITION

Fifty-three holes were drilled in the West Zone Pit Area to provide additional detailed information on massive sulphide lenses in the Pit Area for modelling and resource calculation purposes, and also to further assess footwall PGE-enriched, low-sulphide mineralization. Currently, resources are restricted to areas with greater than 50% sulphide content.

Results of the 53 holes confirmed the intermittent presence of footwall PGE mineralization, along a discrete horizon at vertical depths between 100 m and 200 m below surface. Base and precious metal grades from the main massive sulphide lenses were consistent with previous results.

WEST ZONE MID AND PIT EXTENSION

A total of 21 holes were drilled with six drilled in the West Zone Mid and 15 holes in the Pit Extension.

MAIN WEST ZONE

Twenty-one holes ranging in depth from 341 m to 620 m were drilled at 60 m intervals to infill an area where previous drilling had intersected massive sulphide lenses containing reasonably good base and precious metal grades over hole lengths of up to tens of metres.

The results are consistent with those previously obtained from the Main West Zone and provide additional information for modelling and resource calculation purposes. Of note, in terms of footwall PGE mineralization, is the 0.97 m hole length of 8.37 g/t Pt and 1.53 g/t Pd in hole FL06-280.

EAST ZONE II

Fifteen shallow holes were drilled on sections at 60 m spacing in order to infill the original Inco sections that were drilled at 120 m spacing. The original, widely-spaced Inco drilling indicated a potential for open pit resources centered on section line 13+41E.

CLARKE PLATE

Six holes were drilled to investigate a south dipping magnetic anomaly (termed the Clarke Plate) located in the West Zone Pit Area. Contrary to virtually all other holes drilled to date, these holes were drilled azimuth north in order to intersect the south dipping anomaly. All holes intersected magnetic post mineral mafic dykes, which are the likely source of the modeled magnetic anomaly. Hole FL06-351 intersected West Zone semi-massive sulphide mineralization in a partial down-dip manner, and thus does not represent true thickness of the mineralization.

2007 DRILLING

Late in 2007, Starfield initiated a drill program to further define and evaluate the footwall PGE mineralization that was discovered approximately 50 m below and parallel to the massive sulphide resource described in the latest resource estimate. The program target was to diamond-drill between 12,000 m and 15,000 m, or roughly 45 holes at an average length of 300 m.

Phase 1 of the drilling campaign was completed from September to November 2007, with 19 holes being drilled (Table 10-1) and fire assay results being disclosed in February 2008. These holes are located in the Main West Zone and ranged in depth from 200 m to 489 m, totalling 5,836 m of drilling for 2007. Two of the holes in late November had to be left short, to be completed in the 2008 drilling season.

TABLE 10-1 SUMMARY OF 2007 DIAMOND DRILLING
Starfield Resources Inc – Ferguson Lake

Zone	No. of Holes	Total Metres
Main West Zone	17 (+ 2 left short)	5,836

2008 DRILLING

Phase 2 of the drill program was undertaken in 2008. This phase of the drill program included 19,092 m of diamond drilling (43 new holes, plus completion of two holes from the 2007 program), drilled from April to November 2008, which are summarized in Table 10-2. Approximately 13,214 m (32 holes) were drilled in the Main West Zone, while the remaining 2,005 m (two holes) were drilled in the deeper West Extension Zone. The program aimed to both upgrade Mineral Resources to the Indicated category, and to further evaluate the footwall PGE mineralization beneath the massive sulphide lenses. In addition to the drilling completed in the Main West Zone and West Extension Zone, an additional 3,873 m (9 holes) were completed on other exploration targets in the Ferguson Lake area, not directly associated with these two zones.

TABLE 10-2 SUMMARY OF 2008 DIAMOND DRILLING
Starfield Resources Inc – Ferguson Lake

Zone	No. of Holes	Total Metres
Main West Zone	32 (+ 2 completed from 2007)	13,214
West Extension	2	2,005
Exploration – other targets	9	3,873
Total	**45**	**19,092**

Fire assay results were available for 32 of the holes by November 10, 2008.

Figure 10-1 illustrates the collar locations of previous drilling campaigns (up to and including the 2006 drilling) as well the locations in the Main West Zone and West Extension Zone of the diamond-drill hole collars in 2007 and 2008 drilling campaigns. Not all diamond-drill hole collars are illustrated exclusively in Figure 10-1, as two holes were drilled from one collar and two other collars had three holes drilled from each in the 2008 campaign.

2009 DRILLING

Drilling on the Ferguson Lake claims during 2009 included seven holes drilled by Thanda in the Y lake area to follow up surface diamond exploration and one hole, FL09-423, drilled by Starfield also in the Y Lake area.

Thanda tested seven kimberlite targets with five vertical holes and one inclined hole for a total of 570.9 m. Results showed the rocks comprising the background geology of the Ferguson Lake property record a complex magnetic history. The drilling also found that the rounded, highly magnetic targets identified by the Fugro airborne survey do not necessarily represent kimberlite pipes, but are associated with a wide variety of magnetite and/or pyrrhotite-bearing rocks. Drilling on the M2 anomaly did, however, intersect a thin kimberlitic dike at a depth of 6.3 m. A 2.5 kg core sample was shipped to SGS Canada Inc. to establish whether it was diamondiferous but was found to be barren.

FL09-423 was completed at a depth of 407 m (1,335 ft) targeted the intersection of two converging sets of conductors derived from the Fugro Airborne Survey. This hole encountered several thin horizons of cherty laminated metasediments containing disseminated sulphides and thin sulphide laminae. The most strongly mineralized interval, 338 m to 341 m, consisted of sub-massive to massive sulphides (pyrrhotite, pyrite, and minor chalcopyrite).

2010 DRILLING

Drilling operations were initiated on September 2 and completed on September 27, 2010. During the program six holes totaling 1,126 m were attempted and five were completed. Four of these holes are located in the Y Lake area, while two holes (FL10-428 and FL10-429) are located on the South Discovery Zone.

The Y Lake area is located 20 km to 25 km south of the Ferguson Lake deposit, and is underlain by a mixed package of Archean rock, including banded iron formations, comprising the northeast continuation of the Yathkyed greenstone belt. A project wide VTEM survey conducted in 2005 and a detailed follow-up DIGHEM EM survey of the Y Lake area flown in 2009 have defined a series of potentially significant, locally branching conductive zones more than 25 km in length that are centered on the south end of Y

Lake. Based on ground reconnaissance and limited widely spaced drilling, these conductors are directly related to sulphide mineralization. Widely spaced rock chip sampling along this trend has defined areas containing strongly anomalous copper, zinc, cobalt, gold, and silver.

In the Y Lake area, FL10-424 was drilled, as an offset, to further evaluate the gold mineralization intersected in drill hole FL09-423, while holes FL10-425 and FL10-426 were designed to evaluate a potentially significant sequence of banded iron formations located approximately 0.8 km west of FL10-424. These iron formations occur near a meta-sedimentary/meta-volcanic contact from which strongly anomalous copper, silver, and gold values had been obtained in previous rock chip sampling, and are represented by a moderate-strong geophysical response on the 2009 DIGHEM EM survey.

Hole FL10-427, which is located near the eastern shore of Y Lake approximately 8 km west of the FL10-425/426 pattern, was designed to test a sequence of sulphide-bearing iron formations several metres in width located approximately 400 m north of FL08-418.

Drill hole FL10-428, which is located on an inferred horst 1.5 km southwest of the 119 zone extension, was designed to test a coincident ground and airborne conductivity anomaly believed to be related to multiple horizons of Cu-Ni sulphide mineralization within a concealed gabbroic intrusive. This assumption is based on the presence of a gossanous zone located along the eastern edge of the geophysical anomaly on which two shallow 1999 drill holes intersected Cu-Ni mineralization near the surface. This gossan was referred to by previous workers as the South Discovery zone. Since drilling problems necessitated the abandonment of drill hole FL10-428 before reaching its planned depth, another hole, FL10-429 was completed to test the previously described target.

2011 DRILLING

The 2011 exploration program consisted of three holes for a total of 1,866 m. During this program, one hole, FL11-430, was completed in the western part of the West Zone; one hole, FL11-431, tested a gossanous zone associated with a conductivity anomaly located southwest of the South Discovery Zone; and one hole, FL11-432, further evaluated the deep massive sulphide mineralization in the 119 Zone Extension.. Significant results from holes FL11-430 and 432 are contained in Table 10-3.

TABLE 10-3 SUMMARY OF 2011 DIAMOND DRILLING RESULTS

Starfield Resources Inc – Ferguson Lake

Hole	From (m)	To (m)	Length (m)	Cu (%)	Ni (%)	Co (%)	Pt (g/t)	Pd (g/t)
FL11-430	104.60	107.55	2.95	1.07	0.91	0.09	0.14	1.93
	111.50	119.00	7.5	1.97	0.66	0.08	0.40	1.77
	134.00	140.00	6.0	0.60	0.13	0.04	0.13	0.40
FL11-432	1,096.49	1,098.52	2.03	1.19	0.58	0.07	0.03	1.28
	1,106.45	1,112.10	5.65	1.23	0.64	0.07	0.32	1.73
	1,118.70	1,124.66	5.96	1.33	0.73	0.04	0.11	1.86
	1,125.46	1,126.09	0.63	0.68	0.26	0.03	0.47	4.93

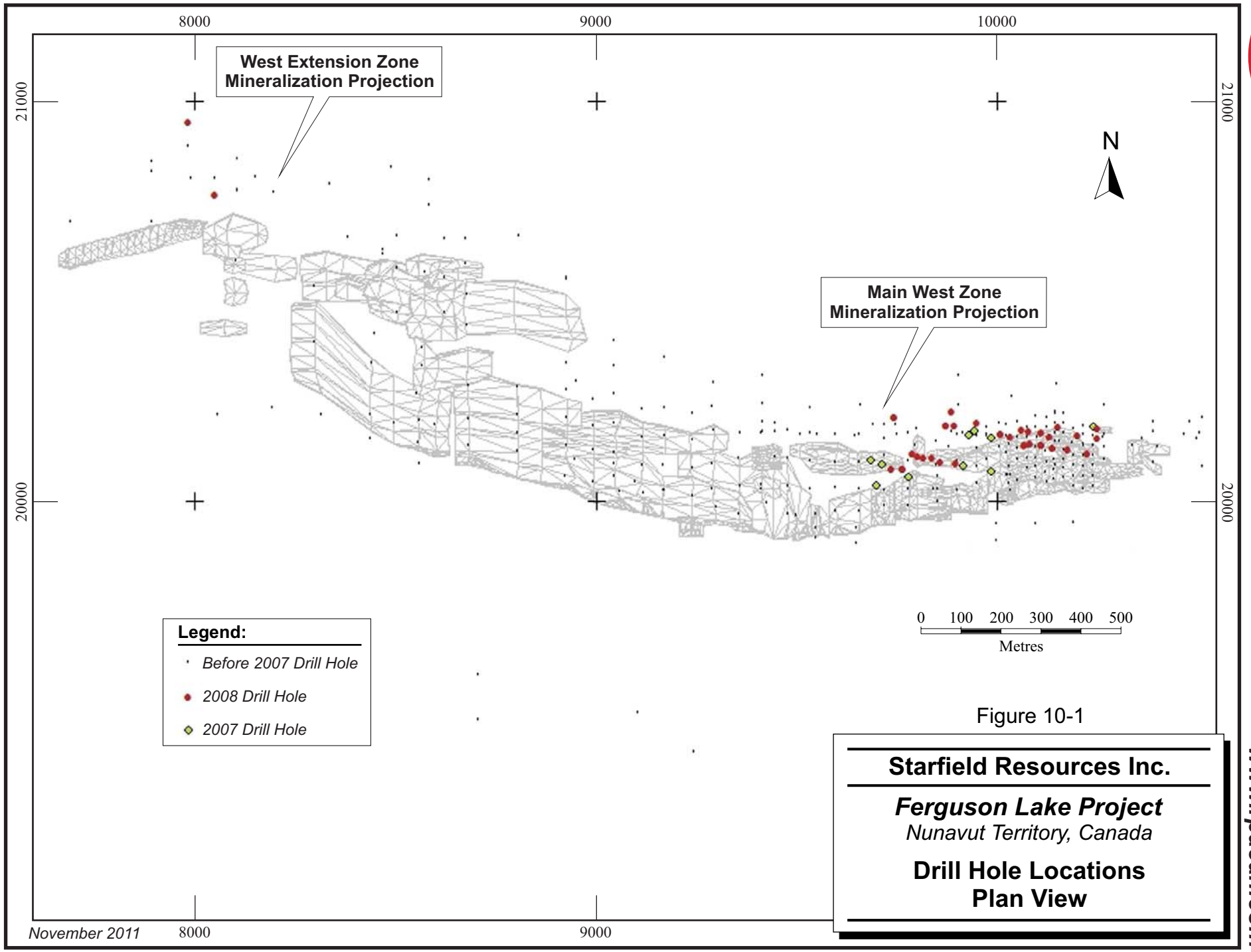

Figure 10-1

Starfield Resources Inc.

Ferguson Lake Project

Nunavut Territory, Canada

Drill Hole Locations
Plan View

www.rpacan.com

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Previous diamond drilling, core logging, and sampling at the Project, up to and including the 2007 drill campaign, was supervised and performed by John Nicholson, P.Geo., and Brian Game, P.Geo., of Nicholson and Associates, Geological Management Consultants, both Qualified Persons in accordance with NI 43-101. Since 2007, work on the Project has been completed by Starfield under the supervision of Ray Irwin, P. Geo. and Qualified Person as defined in NI 43-101.

NQ and BQ thin-wall sized core samples are logged and marked for sampling and subsequently halved by diamond saw with one-half of the core comprising the sample and one-half retained as a rock record in core boxes stored for future reference at the Ferguson Lake camp. The one-half core comprising a sample is tagged, secured and bagged for air shipment from site to the sample preparation laboratories in Vancouver.

Up to 2009, samples were prepared at Acme Analytical Laboratories Ltd. (Acme), and its predecessor Bondar Clegg, in Vancouver, an ISO accredited laboratory which participates in proficiency testing and quality assurance and control procedures for sample preparation and analysis. Acme issues signed Certificates of Analysis and Assay Reports. The one-half drill core samples from sample intervals of generally one metre in length are crushed, riffle split and pulverized prior to analysis. Splits of massive sulphide samples weighing between 10 g and 15 g are then fire-assayed for platinum and palladium. The doré bead is digested and then platinum and palladium are determined by inductively coupled plasma emission spectroscopy (ICP-ES) (Group 6). The massive sulphide samples are also assayed for copper, nickel and cobalt whereby 0.3 g to 1.0 g are digested by four-acid decomposition and then analyzed by ICP-ES (Group 7TD).

Low-sulphide PGE samples are analyzed at Acme where a 30 g sample is digested by aqua regia and then inductively coupled plasma mass spectroscopy (ICP-MS) analysis is conducted for a suite of 51 elements plus platinum and palladium (Group 1F-MS). This geochemical ultratrace method allows for a screening of the samples prior to assay determinations being implemented. All samples containing greater than 500 ppb palladium and/or 100 ppb Pt as determined by ICP-MS are then forwarded for 1AT (29.2

g) fire assay determination for platinum and palladium (Group 6). All samples containing greater than 5,000 ppm Cu and/or 4,000 ppm Ni are sent for four-acid ICP-ES assay determinations (Group 7TD).

Since 2009, samples have been shipped to the ALS Chemex sample preparation laboratory in Yellowknife via Rankin Inlet. Assays and analyses are completed at ALS Chemex in Vancouver using similar methods to those used at Acme.

12 DATA VERIFICATION

During the RPA site visit in June 2007, Mr. R. Dennis Bergen, P.Eng., Associate Principal Mining Engineer of RPA (then Scott Wilson RPA), collected three core samples from areas identified by Starfield's geologist as portions of the Mineral Resource estimate. The results of analyses conducted on the three samples are shown in Table 12-1.

TABLE 12-1 SAMPLE RESULTS
Starfield Resources Inc – Ferguson Lake

	Pt (ppb)	Pd (ppb)	Co (%)	Cu (%)	Ni (%)
Method	FAI313	FAI313	ICP90Q	ICP90Q	ICP90Q
Detection	10	1	0.01	0.01	0.01
38257	<10	2720	0.1	0.77	0.91
38258	40	2350	0.11	0.94	0.88
38259	860	2100	0.08	0.79	0.73
DUP-38257	10	2930	0.11	0.78	0.92

The samples collected are not representative of the whole deposit, nor were they sufficient samples collected to provide a statistical comparison with Starfield's samples. The analyses do indicate the presence of copper, nickel, cobalt and palladium in quantities similar to those in the resource estimate and the presence of platinum was noted.

During the period of September to November 2006, Carter visited the Ferguson Lake Project on two occasions for a total of fifteen days on site. During these visits, a number of 2006 as well as historical drill holes were examined and compared with drill log descriptions and assay data. It was concluded that the geological logging and sample descriptions are representative of the lithological units and distribution of sulphide mineralization. The sampling is appropriate for the deposit type and the assay data is consistent with the sulphide abundance, textures and mineralogy. Survey procedures used at Ferguson Lake for drill hole location and down hole trajectory are completed to exploration industry standards. It was concluded on the basis of the field visits that the drill hole exploration data collection procedures utilized at Ferguson Lake would support resource estimation to NI 43-101 standards.

The diamond drill programs have been managed for Starfield since 1999 by Nicholson and Associates, a professional geological services company based in Vancouver and managed by Professional Geologists. Nicholson and Associates were responsible for data collection, data base management, drilling and analytical QA/QC, and data verification for the Ferguson Lake project. Preparation of the 2008 Mineral Resource estimate relied upon Nicholson and Associates for the provision of the verified data base for resource modelling. Aspects of drilling, sample security, assaying, data handling and verification are contained in the appropriate sections of this report. Verification of Inco data has been referenced in the report by Carter (2006) filed on SEDAR.

Quality control of core samples is maintained by routinely analyzing a number of sample blanks, duplicates, and control reference standards of a similar matrix and content as samples provided. Selected high-grade samples are routinely subjected to repeat assay determinations. Inter-laboratory checks have been an ongoing part of the Ferguson Lake program since 1999. As noted in the previous section, the facilities of Acme have been used for Project analyses since 2002. Bondar Clegg was the laboratory of record in 2000 and 2001 and check analyses were performed by ALS Chemex. Since the merger of these two firms in late 2001, most of the analytical work has been undertaken by Acme while ALS Chemex has been used for necessary check analyses.

A number of inter-laboratory checks of samples have been undertaken over the past several years. An example is the 2004 analytical work undertaken by ALS Chemex of 24 core sample pulps initially analyzed by Acme. In summary, the results for copper are virtually identical while Acme's nickel values are in general almost 10% higher. Palladium values determined by the two laboratories correspond reasonably well but there are apparent difficulties in reproducing consistently similar platinum values. Two of the holes drilled in the West Zone Pit Area in 2002 twinned original Inco holes. Comparisons between these holes are tabulated in Table 12-2; note that no cobalt values were available for Inco holes and only palladium values were broken out of total precious metals in these holes. Consequently, the listed palladium and platinum values reported for the Inco holes incorporate the property-wide palladium to platinum ratio for sulphide mineralization of about 6:1.

TABLE 12-2 COMPARISON OF INCO DATA TO 2002 DATA
Starfield Resources Inc – Ferguson Lake

Section	Hole No.	Width (m)	Cu (%)	Ni (%)	Pd (g/t)	Pt (g/t)
42+68W	FL02-136	21.83	1.05	0.54	1.39	0.21
	9473(Inco)	23.92	0.93	0.91	1.86	0.30
51+20W	FL02-130	18.72	0.65	0.52	1.17	0.17
	9475 (Inco)	15.42	0.75	0.72	1.67	0.25

The results are in reasonably good agreement; nickel and PGE values are higher in the original Inco holes while the recent holes contained higher copper values.

In 2006, a total of 5,894 samples of drill core were taken and sent to Acme for geochemical analysis. A total of 137 of these samples were selected for check analysis by ALS Chemex. This represents approximately 2.36% of the samples assayed by Acme. In summary, nickel, copper, cobalt and palladium values analyzed by ALS Chemex are, in general, within 10% of those analyzed by Acme. There are, and historically have been, difficulties in reproducing consistently similar platinum values. This inconsistency is due to known nugget effects, which occur when sampling for platinum.

N.C. Carter, Ph.D., P. Eng., undertook a thorough review of Inco's 1950s drilling results in late 1999 and expressed confidence in those results as well as the sampling methods and sample preparation and analytical procedures used in the more recent drilling programs.

All sample results have been transmitted by the laboratory directly to the designated person at Starfield who dispersed the results to the writers for initial review of results and the calculation of weighted average grades for the mineralized intervals encountered in the holes drilled to date. These data have been subsequently reported to Starfield on a timely basis.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

METALLURGICAL TESTWORK (2001 TO 2007)

Initial testing and development of the proposed process was completed by Dr. Harris and Mr. White in the laboratories of Kingston Process Metallurgy, Kingston, Ontario. Early work on the primary leach circuit focused on demonstrating the technical viability of carrying out a relatively low-temperature hydrolytic precipitation of iron as hematite and/or magnetite, in the absence of any base addition, with the accompanying recovery of strong HCl.

SGS Mineral Services (SGS) conducted testwork on a series of drill core samples in 2001, and completed another phase of testing in 2006, which included an evaluation of grinding and flotation on three selected drill-hole composites, and chemical and mineralogical characterization, flotation, and PLATSOL™ testing of a pyrrhotite-depleted first stage leach residue sample.

In 2007, NeoFerric coordinated and supervised a leaching test program at SGS. The primary objectives were to test each stage of leaching in a continuous mode, to generate a sample of final leach residue for a subsequent metallurgical test program and to examine options for precious metal extraction.

In 2008 the testwork continued at the NeoFerric laboratory in Montreal. The magnesium chloride matrix, which was the basis of the original proprietary technology, was changed to a ferrous chloride matrix. Leaching in simple hydrochloric acid alone was demonstrated to be effective, and to have benefits in terms of capital and operating costs. Testing of a semi-continuous circuit for iron hydrolysis, precipitation of hematite and hydrochloric acid recovery was carried out. Ion exchange (IX) was replaced by solvent extraction (SX) in the nickel, copper, and cobalt recovery steps.

SAMPLES

Initial testwork included ten contiguous, quartered drill core samples collected from a drill hole which tested the massive sulphide horizon at depth near the known western limits of the Main West Zone in 2001. These samples were submitted to SGS and a

composite of eight of the ten samples was prepared for subsequent mineralogical examination and flotation testwork.

During the same period, a number of quartered drill core samples were collected from several parts of the deposit, including 18 holes in the shallower part of the Main West Zone, two holes in the Main West Zone at depth near the western limit, and two holes drilled to test the West Extension. These samples were submitted to SGS in 2002.

The testwork program in 2006 used the drill core samples that were submitted in 2002. From 2002 to 2006, the samples were freezer-stored at SGS' Lakefield site on Starfield's behalf. Three composites were prepared, representing approximately four kilometres of strike length. The composites originated from the following sources:

- Drill core samples from holes FL02-139 and 119: called 119 Composite.
- Drill core samples from hole FL02-105A: called 6400 Composite.
- Drill core samples from holes FL02-106, 107, and 136: called 4268 Composite.

The three composites were prepared by combining the entire mass from each of its component drill cores. For each composite, the sample was stage crushed to -6 mesh, and ten kilograms of each was set aside for Bond Work Index (BWI) testing. The remaining material was crushed to -10 mesh, blended, and split into two kilogram test charges for flotation testwork. All flotation test charges were nitrogen pumped and freezer-stored until required.

A head sample of each composite was riffled out from one of the flotation test charges and used for head analysis. Table 13-1 shows the chemical analyses of the composite head samples as per testwork completed by SGS.

TABLE 13-1 ANALYSIS OF TEST SAMPLES (2006)
Starfield Resources Inc. - Ferguson Lake Project

Element	Unit	Sample ID		
		Comp 119	Comp 6400	Comp 4268
Cu	%	1.22	1.28	0.87
Ni	%	0.96	0.89	0.85
Ni_{sulph}	%	0.66	0.65	0.69
Fe	%	56.0	54.9	56.4
S	%	27.5	28.9	31.8
SO_4	%	0.4	0.4	0.4
Pt	g/t	1.54	0.14	0.67
Pd	g/t	1.94	2.16	2.44
Au	g/t	0.08	0.06	0.06
Bi	g/t	2	2	2
Te	g/t	4	4	4

MINERALOGY

Mineralogical studies were completed in 2001 by SGS with electron microprobe analyses. It was found that nickel in solid solution in pyrrhotite ranged between 0.31 wt% Ni to 0.52 wt% Ni. As shown in Table 13-1, sulphide nickel accounts for approximately 70% of the total nickel in the head samples. As such, the nickel in solid solution amounts to a maximum of one third of the nickel in the sample. Based on the appointment of nickel, sulphur and copper between pentlandite, pyrrhotite, chalcopyrite and pyrite, the mineral composition of the three samples was calculated and reported by SGS in their July 2007 report as outlined in Table 13-2. The proportions are based on the chemical analysis of the materials, and are preliminary in nature.

TABLE 13-2 ESTIMATED MINERALOGY
Starfield Resources Inc. - Ferguson Lake Project

Sample ID	Estimated Composition (%)					
	Chalcopyrite	Pyrrhotite	Pentlandite	Magnetite	Pyrite	NSG*
Comp 119	3.5	73.5	1.9	14.2	1.3	5.7
Comp 6400	3.7	66.1	1.8	18.3	1.3	8.9
Comp 4268	2.5	63.6	1.7	22.4	1.3	8.4

*Non-Sulphide Gangue

For NeoFerric's initial laboratory testwork, core samples were crushed and ground to 100% passing 100 mesh (150 microns). Chemical analysis, x-ray diffraction (XRD), and size analysis were performed on these samples. The chemical analysis showed that copper was mainly concentrated in the finest fraction of the sample, whereas nickel, iron, manganese and cobalt were fairly evenly distributed. XRD showed the presence of a number of well-defined minerals, with pyrrhotite being the most abundant. The size analysis indicated a bimodal distribution in the samples.

GRINDING

A sample from each of the three composites was submitted for BWI testwork in 2006 to obtain a measure of variability in ore hardness in terms of ball mill grinding. The closing screen size for the BWI test was 150 mesh, which is appropriate for a target grind P_{80} in the order of 100 microns. Table 13-3 shows the results of these tests.

TABLE 13-3 BOND WORK INDEX
Starfield Resources Inc. - Ferguson Lake Project

Sample ID	Bond Work Index (kWh/t)
Composite 119	10.6
Composite 6400	11.0
Composite 4268	10.3

Average BWI for these three samples was found to be 10.6 kWh/t. There was little difference in hardness between the three composite samples, and the Ferguson Lake mineralization was considered to be medium-soft to medium.

FLOTATION

In 2002, flotation tests were undertaken to identify potential process routes that could produce saleable concentrates. Fifteen tests indicated that flotation has application for separating chalcopyrite, pentlandite and platinum group metals (PGEs) from the gangue pyrrhotite. Best metallurgical results were obtained from material ground to -58 µm, and one of the tests yielded a combined copper-nickel concentrate grade of 20.4% with copper and nickel recoveries of 97% and 32%, respectively. Palladium and platinum recoveries were 86% and 48%, respectively, indicating a strong correlation between copper and palladium grades and recoveries, and a similar correlation between nickel and platinum.

In 2006 and 2007, flotation tests were performed on samples of the 119 Composite. The copper-nickel separation flowsheet was investigated as part of the flotation testwork program. It was found that two products could be produced – a high grade copper concentrate grading approximately 30% copper, and a low grade bulk copper/nickel concentrate. Overall recoveries for copper, nickel, platinum and palladium were estimated to be approximately 95%, 46.5% (derived from 70% pentlandite recovery), 28% and 81%, respectively.

Nickel, which carries the majority of economic value in the deposit, showed the poorest response to flotation testwork. The nickel-bearing sulphide, pentlandite, was found to be of a small size in the massive sulphide mineralization, making it less amenable to standard flotation recovery circuits. Low nickel recoveries, and environmental issues concerning reactive sulphide tailings generated from this process method, caused the consideration of flotation for base metal recovery to be abandoned.

PRIMARY LEACH TESTING

Early testwork was comprised of initial laboratory testing, and two series of large-scale tests. The large-scale tests were conducted at Corem (formerly Centre de Recherche Minérales du Gouvernement du Québec), in Quebec City.

In general, it was found that the extraction of iron, cobalt and sulphur increased slowly with time, and rapidly with acid addition. Copper, and to a lesser extent nickel, were the most reluctant to go into solution under these reducing conditions. Like copper, nickel was somewhat refractory to leaching in a reducing environment, although stronger acid

additions did lead to some dissolution. Iron and sulphur extractions were the highest, with cobalt following. The data suggested the presence of multiple cobalt minerals, some leaching more easily than others. The easily leachable iron and high degree of sulphur elimination were also reflected in the residue fall, with 68% to 82% of the original weight fed to the reactor being removed.

The two main parameters that were investigated in the preliminary tests were the HCl addition and time. Most of the reaction seemed to be virtually instantaneous, therefore time was found to be a relatively insignificant factor. With respect to acid addition, the results suggested that a minimum addition of 600 kg/t was required, and that benefits may be achieved by increasing this. Leaching different size fractions was also tested, and as expected, metal extraction increased with decreasing particle size.

Since the residue fall from the primary leach was very small, at least five tests were required to generate sufficient residue to conduct secondary leach tests. As such, it was decided to carry out some large-scale (50 kg) tests at the facilities of Corem to generate sufficient leach residue for secondary leaching, and to confirm the observations of the primary leach on a larger scale. Seven tests were also carried out using the same head sample to check the consistency of the leaching process. The results confirmed those of the small-scale tests.

SECONDARY LEACH TESTS

A number of different lixiviant systems were tested for the secondary leach. Predominant amongst these was a mixture of ferrous and ferric chlorides which occur as a normal course of leaching.

The analysis of the primary leach solids used as feed to the secondary leach was variable, analyzing 17.7% to 20.2% iron, 1.06% to 2.58% nickel, 3.60% to 5.08% copper, and 0.28% to 0.36% cobalt. The results from the secondary leach tests led to the following conclusions:

- Apparent from the primary leach, the HCl alone is inadequate for leaching the base metals.
- The addition of a powerful oxidant, such as hypochlorite, was insufficient solely with HCl to result in adequate base metal recovery.

- Ferric chloride on its own was also unsuccessful in producing adequate base metal recovery.

- Increasing the $MgCl_2$ content from 360 g/L to 400 g/L had a beneficial effect on base metal leaching. The reason for doing this was to allow increased temperature of the reaction.

- Increasing the temperature from 105°C to near boiling, at 118°C, also had a very beneficial effect.

The results showed that recoveries above 98% were achievable for both copper and nickel with the high $MgCl_2$ concentration at the higher temperature of 118°C. Speculated to be a result of multiple phases, cobalt recovery was approximately 70%, which was lower than expected.

BASE METAL RECOVERY TESTING

Base metal concentrations in the secondary leach filtrate were relatively low, making IX to be thought a more attractive proposition than SX.

The secondary leach filtrate was a high chloride brine, predominantly magnesium and ferrous chloride, containing small amounts of sulphate. At high chloride concentration, copper and cobalt are all present in solution as anionic chloro-complexes, and as such, can be easily removed together by IX or SX using an anion exchange resin or solvent. Nickel remains as a cation, and can easily be removed by a cation exchanger. Loading profiles for a selection of resins were developed to track the exit concentration of copper and nickel against the bed volumes of solution passed.

Copper recovery from the anion exchange resin strip solution was successfully tested using relatively inexpensive and readily available iminodiacetate cation exchange resins, which are suited for closed circuit operation with copper EW. Strip solutions over 60 g/L copper were achieved with some resins. Analysis showed that a highly pure copper product can be generated from the secondary leach liquor via the IX/EW route.

OTHER HYDROMETALLURGICAL TESTWORK

An initial laboratory scoping leach test program was carried out by Intec Ltd. (Intec) in 2005. The results and conclusions of this work are summarized below:

- The oxidation of pyrrhotite to form elemental sulphur at atmospheric pressure and 85°C was demonstrated.

- Nickel extractions of greater than 90% were observed from whole ore at a grind size of P_{80} in the order of 30 microns, and a retention time in the range of 10 to 15 hours. Copper and cobalt extractions were greater than 80%, but palladium extraction was only 60%. The platinum grade of the sample provided, which was 0.10 g/t, was too low to be of interest.

- Magnesium extraction was typically less than 15%.

- Sufficient acid was generated in the leach to compensate for the acid consuming minerals in the mineralized material, such as calcite.

- Metathesis for the recovery of copper and nickel/cobalt sulphide by-products was found to be impractical.

- Sulphidization using sodium hydrosulphide (NaHS) was shown to work, but the use of H_2S would have been more appropriate.

- Sulphidization using SO_2 should be investigated as it could be easily generated by burning of the leach residue.

- As every effort should be made to upgrade the mineralized material, the very low grade of this material will require highly efficient leach residue washing.

The limited test results from Intec suggest that hydrometallurgical leach processing has potential, but poor recovery of copper, cobalt and precious metals would result in a substantial amount of unrecovered metal value. Aspects of the process appeared to be impractical without more testing of Ferguson Lake massive sulphides. The business plans of Intec no longer supported custom hydrometallurgical testing. As such, the Intec Process was not pursued further.

In 2006 and 2007, SGS carried out PLATSOL™ testwork on a sample of primary first stage leach residue produced from the hydrometallurgical leaching. The PLATSOL™ process requires concentrate as a feed, so the first stage leach processing using the in-house Ferguson Lake Process is mandatory in this case. The initial PLATSOL™ results were positive and further testing was completed.

Testwork performed on the primary leach residue sample indicated high dissolution of copper and nickel species, but variable dissolution of gold, platinum and palladium. Carbon species in the residue may have been the cause of lower recoveries of gold, platinum and palladium into solution, as it is inferred that these elements are preferentially re-absorbing into the graphitic and/or organic carbon species in the solids residue. Alternatively, a reductant metal present may be carried over from the previous

processing which could be allowing re-precipitation of these metals back into the residue.

It was recommended by SGS that further PLATSOL™ testing be performed. By applying a hot pre-leach with approximately 100 g/L sulphuric acid for up to four hours, the presence of any metallic reducing species may be eliminated. In addition, work carried out on a separate project indicated that the preg-robbing of gold chloride on carbon in an autoclave could be suppressed by strong acid. The resulting leach residue would then be re-leached under high acid PLATSOL™ conditions to evaluate the extent of base metal and PGE dissolution.

The initial PLATSOL™ results were encouraging, and demonstrated that another type of hydrometallurgical process may be suitable, albeit on a concentrate and not on whole ore.

A metallurgical test program that was carried out in 2007 by SGS examined the response of the samples to the proposed multi-stage leach process, including a primary leach for the removal of iron and sulphur, and a secondary leach for the dissolution of pay-metals, which are nickel, copper, and cobalt. The primary leach was first examined in a series of batch bench scale tests, and then by several mini-plant campaigns, which examined the operation in a continuous mode.

The results of this test program are summarized below, and a full description of the work performed by SGS can be found in the report entitled “An Investigation of The Hydrometallurgical Treatment of Massive Sulphide Ore Samples from Ferguson Lake”, dated February 19, 2008.

NeoFerric coordinated and supervised the work by SGS. The key conclusions from the test program are summarized below:

- A bench scale test program examining the primary leach process on selected ore samples showed iron extractions of approximately 87%. Leaching of pay-metals was observed, with average extractions of copper, nickel, and cobalt of 3%, 15%, and 7%, respectively.

- Two campaigns of continuous tests were performed to further evaluate the primary leach process. In general, iron extraction was higher, and dissolution of pay-metals was also slightly higher.

- The evaporation/oxidation process was tested in a single large batch, and results indicated that approximately 30% of the iron was precipitated in the form of hematite.

- The second stage leach was tested in and results showed almost complete dissolution of pay-metals. Minimum metal extractions for copper, nickel, and cobalt were estimated to be 96%, 94%, and 89%, respectively, for this stage of the process.

- Individual extractions determined by testwork for each stage were incorporated into process models, for calculation of overall metal extractions of 97% for nickel, 97% for copper, and 89% for cobalt.

Two bench scale tests were performed on a set of solution and solid samples received from Corem in 2007. Corem's earlier work for NeoFerric included production of residue and leach liquor for further testing. A composite of the solution was made by removing and mixing 1,000 mL from each drum. A rough composite of the solids was made by randomly selecting seven filter cake bags from the shipping drum and cutting a representative sample from each. The samples each weighed approximately 350 wet grams, amounting to a total composite weigh of approximately 2.4 kg.

After the first series of pilot plant tests, additional material was required to continue the test program. A new composite, designated Sept 07 Comp, was prepared from the 2002 ore samples.

Head analyses of the composites used, as reported by SGS, are summarized in Table 13-4. A comparison to the grades used in the Preliminary Economic Assessment is also shown.

TABLE 13-4 HEAD ANALYSIS OF TEST SAMPLES (2007)
Starfield Resources Inc. - Ferguson Lake Project

Element	Unit	Sample ID				Preliminary Economic Assessment Head Grade
		Comp 119	Comp 6400	Corem Residue	Comp Sept07	
Cu	%	1.18	1.22	2.57	0.78	0.95
Ni	%	0.92	0.86	1.87	0.90	0.60
Co	%	0.12	0.11	0.21	0.78	0.07
Fe	%	54.8	52.8	31.1	50.6	-
S	%	30.1	-	30.0	27.4	-
S(=)	%	-	-	-	27.6	-
C(t)	%	-	-	0.37	-	-

NeoFerric noted that metal extraction is not sensitive to variations in copper, nickel, or cobalt feed grades, and concluded that differences in grade between samples used for testwork and mill feed grades will not have a material effect on the overall process economics.

PRIMARY LEACH CIRCUIT – BATCH TESTING

A series of batch bench-scale test were performed prior to mini-plant trials to ensure that the test samples would respond as anticipated, and to gauge the extent and rate of H_2S gas generation. Three different feed samples were tested in a series of six tests that were designed to examine the effects of leach pulp density.

Tests LX-1 to LX-5 were performed in a 3,000 mL tall form resin kettle, fitted with an air-tight lid and chilled condenser. The last test, LX-6, was performed in one of the mini-plant reactors to gauge any possible difference due to the reactor geometry and setup. Tables 13-5 and 13-6 summarize the test results.

TABLE 13-5 BATCH TEST - CALCULATED VS. DIRECT HEAD

Starfield Resources Inc. - Ferguson Lake Project

Test No.	Feed Sample	Calculated Head				Direct Head			
		Fe (%)	Cu (%)	Ni (%)	Co (%)	Fe (%)	Cu (%)	Ni (%)	Co (%)
LX-1	Corem Residue	34.0	2.58	1.75	0.28	31.1	2.57	1.87	0.21
LX-2	Comp 6400	52.6	1.31	0.88	0.10	52.8	1.22	0.86	0.11
LX-3	Corem Residue	25.5	2.54	1.77	0.21	31.1	2.57	1.87	0.21
LX-4	Comp 6400	50.8	1.13	0.76	0.08	52.8	1.22	0.86	0.11
LX-5	Comp 119	59.9	1.07	0.9	0.10	54.8	1.18	0.92	0.12
LX-6	Comp 119	52.0	1.14	0.87	0.10	54.8	1.18	0.92	0.12

TABLE 13-6 BATCH TEST - CALCULATED RECOVERY

Starfield Resources Inc. - Ferguson Lake Project

Test No.	Feed Sample	Calculated Recovery (%)			
		Fe (%)	Cu (%)	Ni (%)	Co (%)
LX-1	Corem Residue	68.1	30.5	24.4	41.6
LX-2	Comp 6400	86.9	4.7	15.9	8.1
LX-3	Corem Residue	49.9	8.6	8.2	7.4
LX-4	Comp 6400	86.6	3.6	14.8	8.1
LX-5	Comp 119	86.9	1.9	15.3	6.0
LX-6	Comp 119	83.7	0.5	1.8	0.7

Tests performed in the resin kettle (LX-2, LX-4, and LX-6) showed iron extractions of 87%. Concurrent pay-metal leaching was also observed, potentially due to partial oxidation of the samples. The results of the test showed similar results to those obtained by NeoFerric, and supported the fact that continuous tests could be conducted with confidence.

The tests performed on the Corem residue and Corem leach liquor (LX-1 and LX-3) gave significantly different results from the tests using the mineralized material. Test LX-1 showed significant co-leaching of pay-metals. In this initial test, a nitrogen gas “blanket” was not used, and thus the ingress of oxygen may have contributed to the observed results. Iron leaching in LX-3 was lower than that achieved by leaching the ore. However, this was an expected result as the Corem material had been partially leached.

The results of test LX-6 indicated that the leach performance in the mini-plant reactor was similar to that obtained in the laboratory, with slight reductions in metal recovery for all metals.

Leach tests were run for four hours in all cases, and samples of pulp were withdrawn, filtered, and assayed to gauge leach kinetics during the tests. The tests suggested that iron leaching was complete after 120 minutes in all cases, and that dissolution of nickel only started once iron leaching was essentially complete.

PRIMARY LEACH CIRCUIT – CONTINUOUS TESTING

The continuous leaching tests were performed in a continuous stirred tank reactor train of six reactors. An initial series of continuous tests were performed over a period of five days, with acid dosage and residence time being the main parameters that were altered during the tests. Leach kinetics were measured by collecting composites of the feed and discharge throughout the tests, in addition to periodic sampling across the reactor train.

The analyses showed reasonably uniform solid composition throughout the runs. However, not all the sulphur was present as sulphide, which suggested that some minor oxidation of the ore had occurred. Such oxidation could account for the levels of nickel seen to leach.

The discharge solution analysis showed significant levels of sodium and calcium, which originated from either the reagent grade $MgCl_2$ and/or from partial dissolution of gangue minerals. The analysis also indicated that mostly ferrous iron was present in the leach solution, as expected.

The approximate percentage of metal reporting to the solution phase in the Reactor 6 discharge was estimated using the measured discharge slurry, solution densities, dried solid weights, and the solids and solution analysis. Results indicated that 76% to 89% of the iron was in the solution phase, and the nickel distribution was generally less than 10%. The rate of metal dilution was gauged, and indicated that iron dissolution was complete after 80 to 120 minutes.

The magnesium concentration in the discharge solution was lower than intended. This problem was a result of failure to account for the dilution by the large HCl addition as

well as the density of the feed slurry. Due to this error, an additional series of five continuous primary leach tests were performed using three different feed samples. As before, concentrations of iron and nickel in the discharge were tracked through the tests, and leach kinetics were performed. Operating data indicated that the magnesium content was in the target range. Iron concentrations changed as the target pulp densities in the different tests were changed. Periodically, the nickel concentration in solution increased, which seemed to be related to the maintenance of the balance between H_2S evolution and the addition of nitrogen gas. Leach conditions were maintained essentially reducing (<250 mV) throughout the campaign.

Table 13-7 shows the approximate percentage of metal reporting to the solution phase for all continuous tests performed.

TABLE 13-7 PRIMARY LEACH - METAL PARTITION TO SOLUTION
Starfield Resource Inc. - Ferguson Lake Project

Test		Element				
		Cu	Ni	Co	Fe	Mg
PP1	Sept 5 23:00	1.6	4.7	3.3	89.5	99.5
PP1	Sept 6 13:00	5.0	5.6	4.1	84.8	99.4
PP2	Sept 6 23:00	15.7	9.7	5.4	82.0	99.0
PP2	Sept 7 13:00	8.7	7.1	4.0	76.3	99.1
PP3	Sept 8 3:00	5.9	13.9	4.8	87.0	99.6
PP4	Sept 10 23:00	3.2	8.0	4.1	79.0	98.9
PP6	Sept 12 15:00	-	-	-	-	-
PP7	Sept 25 17:00	7.8	19.5	7.4	91.7	99.6
PP8	Sept 26 23:00	6.3	8.9	3.7	88.7	99.8
PP9+PP10	Sept 27 23:00	16.5	20.6	8.1	89.4	99.5
PP11	Sept 28 11:00	29.8	44.4	20.1	96.3	99.7

NeoFerric notes that tests PP1 to PP6 are from the earlier series when the magnesium chloride was lower than intended in the discharge solution. The results from the latest series of tests show more dissolution of iron in the solution phase compared to the earlier tests that were performed. The dissolution of pay metals was also higher, particularly for test PP11.

Metal dissolution kinetics from the second set of tests showed a wider variation than the first. The most rapid dissolution was observed in PP11, where the feed rate was increased. In this test, the iron dissolution was complete after approximately 60 minutes of leaching. The first set of tests indicated that dissolution was complete after approximately 80 to 120 minutes.

EVAPORATION AND IRON OXIDATION

A portion of the filtrate solution from tests PP7 to PP11 was selected for use in the evaporation and iron oxidation stage to produce a leach solution for the second stage leach. The continuous mini-plant was initially intended to perform the evaporation and oxidation process. However, the apparatus did not have enough power to evaporate the required volume of water in a reasonable amount of time. As such, this option was abandoned, and a single large scale batch process was devised.

A volume of solution was heated and boiled to evaporate the water to produce a resulting solution that was one-third of the starting volume. The solution had a final temperature of approximately 140°C, and the process took 34 hours and a combined power of 3.5 kW to complete. The solution was oxidized with the slow addition of 25% hydrogen peroxide solution, which required an additional 16 hours. The oxidized solution was then diluted back to the original solution volume and allowed to cool prior to filtration.

The precipitated iron solids did not settle well, and filtered poorly. This was contrary to what had been observed previously, and was almost certainly due to the less than ideal way in which the oxidation tests were ultimately carried out. Several days were required to filter all of the oxidized material. The filter cake solids were washed once by re-pulping in acidic de-ionized water, and then displacement washed several times.

Chemical analyses of the iron precipitate and filtrate solution showed that approximately 29% of the iron was precipitated during the oxidation process. X-ray diffraction analysis of the iron precipitate showed the presence of hematite, with possible trace quantities of gypsum, quartz, and potassium feldspar. Size analysis showed the solids to be very fine, with 80% of the particles being finer than eight microns.

The iron content of the precipitate was lower than the theoretical content of hematite, indicating the presence of other compounds. As the precipitate was difficult to wash, there is a possibility that residual ferric and magnesium chloride salts were present. It may also be possible that waters of crystallization were present in the hematite, which has been reported in NeoFerric's laboratory trials.

SECONDARY STAGE LEACH – BATCH TESTING

The first leach vessel in the reactor train was used to perform a single batch test (LX-8) using the planned conditions for the continuous mini-plant test. The primary leach residue was mixed with a $MgCl_2$ solution to make an approximate 50% solids pulp. The oxidized solution was then added to a leach solution to a solid weight ratio of 15.7:1, and the mixture was heated to the operating temperature and held for six hours.

Samples were removed from the leach every 30 minutes to evaluate the rate of reaction. The second stage leach converts ferric iron to ferrous iron, and chemical analysis of the leach solution indicated that at the pump density tested, the reaction almost consumed all of the ferric iron during the six hour period.

SECONDARY STAGE LEACH – CONTINUOUS TESTING

The Starfield mini-plant apparatus was used to test the second stage leach in a continuous mode. The leach operated for approximately 29 hours. As the initial batch test indicated that all of the ferric iron was consumed when the primary leach residue solids were mixed with the oxidized solution at a ratio of 15.7:1, a slightly higher ratio of 21:1 was used for the continuous test. The flow rates of the feed slurry and oxidized solution were adjusted so that the process had a residence time of six hours.

During the test, composite samples of the feed slurry were collected and chemical analyses of the feed solids and feed solutions were performed. As primary leach residues were not washed, metal concentrations in the feed solution were present due to some carried residual primary leach solution.

Temperature, electromotive force (EMF) and ferrous iron concentrations were monitored during the continuous test. It was found that the temperature remained steady through the test, near the target value of 118°C. The EMF of the pulps remained slightly oxidized for most of the test, with all of the reactors generally above 450 mV (vs.

Ag/AgCl electrode). As determined by an operator titration, ferrous iron concentration in Reactor 6 peaked at above 27 g/L. Subsequent chemical analysis of the discharge solution indicated that 20% to 30% of the iron present was still in the ferric state, indicating that the supply of oxidized solution was adequate for the process.

Chemical analyses of all of the second stage leach products showed that the final leach residues contain very low concentrations of copper, nickel and cobalt, indicating very high extraction of these metals. Leach kinetics suggested that the reaction was essentially complete in Reactor 5, or after approximately five hours of leaching.

An estimate of the metal extraction obtained in the second stage leach is shown in Table 13-8. The calculation is based on the conservative assumption that there is minimal weight loss in the leach. In order to calculate the exact metal extractions, the weight loss in this process needs to be determined accurately.

TABLE 13-8 SECOND STAGE LEACH – ESTIMATE OF METAL EXTRACTION
Starfield Resources Inc. - Ferguson Lake Project

	Assay			
	Cu (%)	Ni (%)	Co (%)	Fe (%)
Second Stage Feed	2.75	2.55	0.32	20.00
Second Stage Residue	0.11	0.15	0.03	7.28
Minimum Extraction	96	94	89	64

THICKENING TESTWORK

Scoping-level tests were performed on samples from the continuous primary leach using Zetag 7623, a low weight cationic flocculent. The final settled density of the solids was relatively low, in the range of 23% to 40% solids, and the unit thickener area required was in the range of 0.33 m^2/tonne/day to 0.10 m^2/tonne/day. It is noted that these were not exhaustive tests, and were carried out to give an indication of the final pulps behaviour.

An additional set of tests were performed on the discharge from the second stage leach. Magnafloc 10, a low weight anionic flocculent, was used and floc screening tests indicated better results than the previously used Zetag product. The settled density of

the solids was low, in the range of 23% to 25%. The required unit thickener area was calculated to be 0.05 m^2/tonne/day to 0.06 m^2/tonne/day.

FILTRATION TESTWORK

Batch tests simulating the use of a belt filter were performed on a slurry sample from the evaporation and oxidation stage. Filtering proved to be very difficult without any pre-treatment, so the work was not pursued further. Poor filtration was likely due to the very fine nature of the precipitate, as well as the presence of some hydro hematite which is known to have poor filtration characteristics.

A sample of the thickened stage two leach solids was also tested using the process to simulate a belt filter. Five tests were performed to examine the effect of the ratio of form to dry time. For the tests performed to produce an 11 mm thick cake, the unit area required for filtration was approximately 200 kg/m^2/hour to 300 kg/m^2/hour. The rate is dependent on the final moisture content that is desired in the cake. The moisture content of the test production ranged between 14% and 20%, which is fairly low for a product from a belt filter.

METALLURICAL TESTWORK (2008)

Since the preparation and publication of the previous PEA (dated December 15, 2008), NeoFerric has carried out and compiled results of additional testwork. The magnesium chloride matrix was abandoned and replaced by a simple hydrochloric acid primary leach. NeoFerric found that all the data generated with magnesium are equally applicable to acid only, both in the primary and secondary leach circuits. The primary leach and secondary leach, both of which were piloted at SGS Lakefield in 2007, performed exactly the same without magnesium as they did with it in the high chloride matrix.

Abandoning magnesium is a significant simplification, as iron was found to behave much better during hydrolysis, making the hydrolysis and acid recovery circuits much easier and more efficient to operate, comparable to the PORI Process that was operated for several years in the 1970s.

The metals recovery circuit was changed to primarily solvent extraction, with only zinc removal being a dedicated ion exchange circuit. Solvent extraction is better suited to

handling high metal throughputs, and because unit metal loading on the solvents is much higher than it is with ion exchange, the resulting circuits are significantly smaller.

Additionally, results generated in 2008 have shown that:

1. Hydrochloric acid can be recovered at a strength of 34%. This is significantly better than most conventional acid recovery processes currently being operated, such as pyrohydrolysis, which is used in the steel industry. The SMS Siemag plant currently being built, however, has also been shown to generate >30% HCl.

2. Up to 90% of the iron can be precipitated as highly crystalline hematite. The hematite produced is "black hematite," which is a very pure form of hematite. The iron content of the precipitates has varied from 67% to 71% iron. The precipitates show very low chloride (<0.2%), or base metals, making this material potentially a very suitable feed for a steelmaking process.

3. The precipitated solids have a moisture content of 10%, demonstrating that they filter and wash very well. This significantly aids the overall water balance of the hydrometallurgical plant.

4. Since no base metals were detected in the final iron solids, it will be possible to remove most of the iron from the circuit without incurring any loss of base metals. This compares very favourably with existing hydrometallurgical processes.

In September and October 2008, NeoFerric tested a semi-continuous circuit for iron hydrolysis, precipitation and hydrochloric acid recovery. Preliminary results demonstrated that:

1. The procedure can be effectively and efficiently translated from the initial batch mode of testing to continuous operation.

2. Hydrochloric acid, ranging from 30% to 35% in strength, has been continuously, steadily, and consistently produced over extended periods of several hours.

3. The process generates a highly crystalline black hematite, with a coarse texture that filters very well, and an iron content of 68% to 71%, with no detectable base metals and very low chloride content (<0.2%).

METALLURGICAL TESTWORK (2009)

During 2009, McGill University (McGill), under the supervision of Hatch Limited (Hatch), carried out additional batch and semi- continuous testwork for the Project. The purpose of the testwork was to verify that using synthetic leach solutions with a composition more representative of that expected in the commercial plant, along with improvements to the

mini-plant equipment, the target iron oxidation and hydrolysis extents of reaction for the commercial plant could be achieved.

OXIDATION

Based on agitator design fundamentals, Hatch recommended increasing the power input, to increase the oxygen mass-transfer rate and thus improve oxidation rates and extents. The main conclusions drawn from the oxidation testing done at McGill were:

1. The rate of iron oxidation achieved was well below the 75% ferrous iron oxidation that was thought to be required in the commercial plant.

2. The absence of baffles and a lower bath volume than expected, due to solution boil down, negatively impacted the iron oxidation results achieved.

3. The presence of high ferrous iron concentrations in the iron oxidation step may have been detrimental to the rate of iron oxidation.

4. Testwork showed that the presence of $FeCl_2$ to be unfavourable in the hydrolysis step.

HYDROLYSIS

The McGill testwork on hydrolysis was performed to:

1. Verify that scaling can be reduced by the addition of seed.

2. Evaluate hydrolysis using solution compositions similar to those predicted for the commercial plant.

Previous hydrolysis experiments performed at McGill, using pure ferric systems, demonstrated that over 75% precipitation of ferric could be achieved after six hours retention time. The commercial set-point was 75% of total iron (ferric as hematite and ferrous as magnetite), and to obtain HCl off gas concentration of approximately 11 molar (35%) HCl. The work done at McGill on hydrolysis generated the following conclusions:

1. Trying to offset a lower extent of reaction in the oxidation circuit by allowing more ferrous iron to advance and then re-circulating to get the total oxidation required, resulted in more difficulty in the hydrolysis step. The presence of ferrous iron in the hydrolysis stage appeared to reduce the rate of iron hydrolysis.

2. The amount of evaporation taking place in both the iron oxidation and iron hydrolysis steps was greater than originally predicted by the process modeling.

3. To achieve an acceptable rate of iron hydrolysis, adequate product acid strength and a good quality iron product, the hydrolysis reaction temperature must be maintained at or above 180°C.

4. The rate of iron hydrolysis in the testwork seemed to be limited by the rate of heat transfer into the reactor.

METALLURGICAL TESTWORK (2010)

The following confirmatory testwork was carried out in 2010 at SGS in Lakefield.

OXIDATION

A highly concentrated ferrous chloride solution (see Table 13-9 for a typical feed solution composition), which also contained ferric chloride and base metal chlorides, was subjected to oxidation using oxygen as reagent. This was completed in order to convert ferrous chloride to iron (III) oxide (hematite) and ferric chloride, according to the following reaction:

$$6FeCl_2 + 1.5O_2 \rightarrow 4FeCl_3 + Fe_2O_3$$

The feed solution, and the initial solutions in the pre-heater and the reactor, were synthetic solutions made up from solid reagents and melted in their respective vessels at the start of each test. Each composition was based on process modelling results provided to SGS by Starfield or its consultants. A photograph of the arrangement is shown in Figure 13-1. A summary of the test conditions is tabulated in Table 13-10, while a summary of the main results from the short oxidation tests is presented in Table 13-11.

TABLE 13-9 OXIDATION TESTS – FEED SOLUTION COMPOSITION
Starfield Resources Inc. – Ferguson Lake Project

Cations	mg/L
Fe[II]	199,024
Fe[III]	25,424
Cu	9,344
Ni	6,726
Co	590
Mg	5,280

TABLE 13-10 OXIDATION TESTS – TEST CONDITIONS
Starfield Resources Inc. – Ferguson Lake Project

Condition	Unit	Test 1	Test 3	Test 4	Test 5	Test 6	Test 7
Target feed	mL/min	4	4	2.9	4	4	4
Boiler temperature	°C	140	120	110	120	110	120
Reactor temperature	°C	150	120	140	130	110	120
Impeller type	-	Rushton turbine	Pitch-blade	Pitch-blade	Pitch-blade	Pitch-blade	Rushton turbine
Reactor condensate	-	Drawn off	Back to reactor	Drawn off	Drawn off	Back to reactor	Back to reactor

TABLE 13-11 OXIDATION TEST RESULTS
Starfield Resources Inc. – Ferguson Lake Project

Condition	Unit	Test 1	Test 3	Test 4	Test 5	Test 6	Test 7
Run time	hours	6.1	8	5.1	9	10	10.1
Average feed flow	g/min	6	5.5	4.8	6.2	5.4	
	mL/min	4	3.8	3.4	4.3	3.9	4.7
Retention time	hours	11	12	12	9	6	7
Turnovers	-	0.6	0.7	0.4	1	1.7	1.4
Feed	g	2,190	2,640	1,535	3,337	3,266	4,072
Product	g pulp	307	1,438	749	1,642	2,996	2,159
	g/min pulp	0.8	3	2.4	3	5	3.6
	g solids	0	162	127	174	172	232
Feed	% Fe(II)	78.6	73.8	97	82.4	84	80.2
Reactor - start	% Fe(II)	6.1	6.2	28.5	11.6	8.2	12.8
Discharge	% Fe(II)	8.6	12.6	22.7	18.6	28.6	17.8
Reactor - end	% Fe(II)	11.8	7.8	20	24	19.3	8.4
% Fe(II) Oxidized	%	85.1	85.9	60.4	71	60.4	90.5
Oxidation rate	g/min Fe(II)	0.65	0.62	0.46	0.57	0.42	0.8
Total Fe in discharge	g/kg	196	193	341	213	155	188
Fe Accountability	%	57	107	273	96	121	74
O_2 addition	mL/min	322	354	1250	361	353	369
O_2 utilization	%	9.3	8	1.7	7.2	5.4	9.9
Solids	%Fe	N/A	67.3	62.7	69.6	64.7	68.3
	%Cl	N/A	2.4	3.7	1.7	3.4	2.1
	D_{80}, µm					12.57	9.52

The solids from Test 3 were kept wet for use as seed for the long-run test. A sub-sample from the solids was submitted for X-Ray Diffraction (XRD – Table 13-12), which showed that the solids formed by the oxidation reaction were predominantly hematite and essentially all iron oxide.

TABLE 13-12 X-RAY DIFFRACTION RESULTS
Starfield Resources Inc. – Ferguson Lake Project

Mineral	Test 7 PPT-Reactor-Final
(wt %)	
Hematite	97.3
Magnetite	1.7
Siderite	0.8
Muscovite	0.2
TOTAL	100

An extended oxidation test was done to evaluate the effects of selected process conditions under steady operation. The boiler and reactor temperatures were held at 120ºC for the entire test. The feed solution composition and the oxygen-to-feed ratio were kept the same throughout the test. Different combinations of feed rate and stirrer speed were used. Discharge samples were taken every hour, solution samples were analyzed individually and solid samples were combined and analyzed at the end of each operating condition tested. Table 13-13 lists the process conditions tested and Table 13-14 summarizes the results.

TABLE 13-13 EXTENDED OXIDATION TEST CONDITIONS
Starfield Resources Inc. – Ferguson Lake Project

Condition	Unit	Cond 1	Cond 2	Cond 3	Cond 4	Cond 5	Cond 6
Start		4/27/11 8:30	4/27/11 20:30	4/28/11 8:30	4/28/11 20:50	4/29/11 9:30	4/29/11 21:30
Finish		4/27/11 20:30	4/28/11 8:30	4/28/11 20:50	4/29/11 9:30	4/29/11 21:30	4/30/11 14:30
Target feed	mL/min	4	4	4	6	8	8
Reactor rpm	rpm	1,000 rpm	750 rpm	1,250 rpm	1,250 rpm	1,250 rpm	1,250 rpm
Reactor condensate		Back to reactor	Back to reactor	Back to reactor	Back to reactor	Back to reactor	Drawn off

TABLE 13-14 EXTENDED OXIDATION TEST RESULTS
Starfield Resources Inc. – Ferguson Lake Project

Condition	Unit	Cond 1	Cond 2	Cond 3	Cond 4	Cond 5	Cond 6
Run time	hours	12	12	12.33	12.67	12	17
Average feed flow	g/min	5.7	5.5	5.5	8	10.9	10.4
	mL/min	3.9	4	3.7	5.5	7.6	7.2
Retention time	hours	9.8	8.9	10	6.7	4.3	5.9
Turn-overs	-	1.2	1.3	1.2	1.9	2.8	2.9
Feed	g	4,088	3,885	3,886	5,912	7,289	10,497
Product	g pulp	2,533	2,281	2,450	4,051	5,002	6,535
	g/min pulp	3.5	3.2	3.3	5.3	6.9	6.4
	g solids	122	147	190	302	336	468
Feed	% Fe(II)	0.911	0.849	0.921	0.921	0.931	0.905
Reactor - start	% Fe(II)	0.074	0.188	0.367	0.097	0.263	0.313
Discharge	% Fe(II)	0.222	0.316	0.214	0.251	0.365	0.227
Reactor - end	% Fe(II)	0.188	0.367	0.097	0.263	0.313	0.157
% Fe(II) Oxidized	%	0.738	0.621	1.01	0.681	0.551	0.865
Oxidation rate	g/min Fe(II)	0.55	0.44	0.68	0.68	0.6	1.19
Total Fe in discharge	g/kg	180	175	174	184	173	180
Fe Accountability	%	0.97	0.91	1	1.1	1.26	0.92
O_2 addition	mL/min	349	350	379	546	726	734
O_2 utilization	%	0.072	0.058	0.082	0.06	0.04	0.07
Solids	%Fe	0.671	0.666	0.671	0.663	0.67	0.666
	%Cl	0.02	0.02	0.022	0.026	0.027	0.026
	D_{80}, µm	13.14	17.43	19.52	19.72	18.65	18.52

The extended test was run for 78 hours. At the end of the test the three vessels (feed reservoir, boiler and reactor) were inspected and no visible damage to any of the vessels was observed. Also, there were no solids on the reactor walls or any crystallized solids in the feed reservoir or in the boiler. The precipitates produced throughout the test contained an average of 66.8% iron and 2.3% chloride, without any significant difference between each condition's solids. A particle size distribution analysis was done for each condition's solids, the overall result averaging a d_{80} of 17.8 microns.

Conditions 1, 2, and 3 were run at the same flows and 1,000, 750, and 1,250 RPM agitator speeds respectively. The extents of ferrous iron oxidation achieved under each condition were 73%, 62%, and 100% with oxygen utilizations of 7.2%, 5.8%, and 8.2% respectively. In Condition 4 and 5, the agitator speed was maintained at 1,250 RPM and the feed flow was increased to 6 mL/min and 8 mL/min, respectively. The extent of oxidation decreased to 68.1% and 55.1% respectively. Condition 4 had the same oxidation rate (0.68 g Fe^{2+}/min) as Condition 3, which could indicate that the reactor was

working “at full capacity” in Condition 3, and 0.68 g Fe^{2+}/min is the maximum oxidation rate achievable at those conditions. Condition 6 used the same parameters as Condition 5 except that the reactor condensate was drawn off. This caused a significant improvement in the extent of oxidation, which increased from 55.1 % in Condition 5 to 86.5 % in Condition 6.

HYDROLYSIS

A set of highly concentrated, ferric chloride solutions containing ferrous chloride and base metal chlorides were subjected to hydrolysis, using water as reagent, to convert the ferric chloride to hematite (Fe_2O_3) and to recover gaseous hydrochloric acid. The overall chemical reaction involved is as follows:

$$2FeCl_3 + 3H_2O \rightarrow Fe_2O_3 + 6HCl$$

A photograph of the arrangement is shown in Figure 13-2. The feed and initial bath solutions were made up from solid reagents and melted in their respective vessels at the start of each test. Each composition was predicted from process modelling work provided to SGS by Starfield Resources. A summary of the test conditions is presented in Table 13-15.

A summary of the short hydrolysis test results is shown in Table 13-16. Test 8 is the first of the hydrolysis tests. Test 10 was run to investigate the relation between boiler temperature and acid strength in the boiler condensate. The test started with the boiler at 120ºC, and temperature was gradually increased up to 180ºC. Samples from the boiler condensate were titrated for free acid. Figure 13-3 shows that free acid increased exponentially as the temperature was increased. It also shows that to avoid hydrolysis in the boiler, the boiler temperature should be kept at or below 140ºC. A detailed discussion of the conditions used and the results obtained in the series of tests listed in Tables 13-17 and 13-18 can be found in Appendix 1 (SGS report on project number 12323-001, April 2011).

Figure 13-1: **Oxidation Test Experimental Arrangement**

Figure 13-2: **Hydrolysis Test Arrangement**

TABLE 13-15 HYDROLYSIS TEST CONDITIONS
Starfield Resources Inc. – Ferguson Lake Project

Condition	Cation/ Unit	Test 8	Test 9	Test 10	Test 11	Test 12	Test 13	Test 14	Test 15	Test 16	Test 17	Test 18	Test 19
Date		06-May-10	14-May-10	19-May-10	01-Jun-10	08-Jun-10	15-Jun-10	17-Jun-10	24-Jun-10	29-Jun-10	05-Jul-10	13-Jul-10	15-Jul-10
Feed Composition, g/L	Fe(II)	2.7	2.7	2.7	2.7	2.7	-	-	-	2.7	2.7	2.7	38.8
	Fe(III)	213	213	213	213	213	213	213	213	213	213	213	177
	Cu	12.6	12.6	12.6	12.6	12.6	12.6	-	-	10	10	10	10
	Ni	9.1	9.1	9.1	9.1	9.1	9.1	9.1	-	-	9.1	9.1	9.1
	Co	0.8	0.8	0.8	0.8	0.8	0.8	0.8	-	-	0.8	0.8	0.8
	Mg	7.2	7.2	7.2	7.2	7.2	7.2	7.2	-	-	7.2	7.2	7.2
	H_2O	699	699	699	699	699	705	732	782	755	705	705	727
Starting Hydrolysis Bath Composition, g/L	Fe(II)	12.7	12.7	12.7	12.7	12.7	-	-	-	-	4.1	4.1	59.8
	Fe(III)	414	414	414	414	414	414	414	414	335	328	328	272
	Cu	64.1	64.1	64.1	64.1	64.1	64.1	-	-	-	15.4	15.4	15.4
	Ni	46.1	46.1	46.1	46.1	46.1	46.1	46.1	-	-	14	14	14
	Co	4.1	4.1	4.1	4.1	4.1	4.1	4.1	-	-	1.3	1.3	1.3
	Mg	36.3	36.3	36.3	36.3	36.3	36.3	36.3	-	-	11.1	11.1	11.1
Target feed	mL/min	2	2	2	2	2	2	2	2	2	2	2	2
Boiler temperature	°C	150	170	Variable	140	140	140	140	140	140	140	140	140
Reactor temperature	°C	180	180	180	180	180	180	180	180	180	180	190	180
Seeding	-	No	No	No	Yes	Yes	Yes	Yes	Yes	Yes	Yes	Yes	Yes
	g				98	95	93	83	88	88	92	87	85
Boiler condensate		Drawn off	Back to reactor	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off

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TABLE 13-16 SHORT HYDROLYSIS TEST RESULTS
Starfield Resources Inc. – Ferguson Lake Project

Condition	Unit	Test 8	Test 9	Test 10	Test 11	Test 12	Test 13	Test 14	Test 15	Test 16	Test 17	Test 18	Test 19
Date		06-May-10	14-May-10	19-May-10	01-Jun-10	08-Jun-10	15-Jun-10	17-Jun-10	24-Jun-10	29-Jun-10	05-Jul-10	13-Jul-10	15-Jul-10
Run time	hours	5	9	8	8	8	8	8	8.1	8	8	8	6
Average feed flow rate	g/min	3.9	3.6	3.6	3.4	3.2	3	2.9	3.2	3.1	3.7	3.6	4.2
	mL/min	2.6	2.4	2.4	2.3	2.2	2	2	2.2	2.1	2.4	2.4	2.8
Average water addition	mL/min	0.8	0.83	0.38	0.38	Nil	Nil	Nil	Nil	0.38	0.39	0.39	0.39
Total water addition	mole	13.3	24.7	10	10.2	0	0	0	0	10.2	10.4	10.3	7.8
Retention time	hours	16	12	14	7	10	12	12	15	37	12	34	17
Turn-overs	-	0.3	0.8	0.6	1.1	0.8	0.6	0.7	0.5	0.2	0.7	0.2	0.4
Feed	g	1110	1992	1691	1653	1536	1449	1395	1526	1494	1773	1734	1494
Feed	% Fe(III)	98.5	98.7	98.4	98.3	96.4	96.4	96.4	100	98.6	98.6	98.8	83.3
	g Fe(III)	174.3	294.6	259.7	247.3	221.2	216.4	202.9	220.7	228.1	276.2	280.4	186.9
	mole Fe(III)	3.1	5.3	4.6	4.4	4	3.9	3.6	4	4.1	4.9	5	3.3
	mole H_2O	30.4	50.7	44	43.2	40.7	37.8	39.1	46.5	42.5	46.1	45.6	40.2
Product	g H_2O boiler cond	198	517	788	358	396	345	334	445	422	391	373	397
	mole H_2O boiler cond	11	29	44	20	22	19	19	25	23	22	21	22
	g reactor pulp	414	968	744	1523	991	824	847	698	306	865	351	453
	g/min pulp	1.4	1.8	1.6	3.2	2.1							
	g solids (incl seed)	128	161	163	225	365	138	136	206	282	238	416	169
	g solids produced	128	161	163	127	270	46	53	118	194	145	329	84
	mole Fe solids	1.2	1.4	1.4	1.1	3	0.5	0.6	1.4	2.4	1.7	4	1
	g/kg H_2O	294	265	364	398	314	309	338	334	334	334	334	334
	g HCl	124	83	36	46	45	36	47	126	229	185	309	103
	mole HCl vapor	3.4	2.3	1	1.3	1.2	1	1.3	3.5	6.3	5.1	8.5	2.8
	mole H_2O w HCl	38.5	19.9	6.3	7.3	6.8	5.9	6.9	13.9	26.8	22.7	27.7	16.6
Reactor - start	% Fe(III)	96.4	96.7	96.6	98.8	116.2	99.3	100	100	100	98.8	98.8	82

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Condition	Unit	Test 8	Test 9	Test 10	Test 11	Test 12	Test 13	Test 14	Test 15	Test 16	Test 17	Test 18	Test 19
Discharge	% Fe(III)	96.1	97	97	98.6	97.4	99	99.7	100	97.8	97.7	97.4	80.8
Reactor - end	% Fe(III)	96.7	96.8	97.1	97.7	99.8	99.2	99.4	100	98.2	98.1	97	78.1
% Fe(III) hydrolyzed	%	79.4	68.6	72.7	8.5	31.6	30.8	33.2	36.1	62.7	41.7	86.5	58.4
Fe Accountability	%	59	59	58	109	145	82	88	99	98	94	95	80
HCl average strength	%	15.2	18.9	24.3	26.2	26.9	25.6	27.3	33.5	32.2	31.1	38.3	25.6
HCl maximum strength	%	20.9	19.9	25.8	28	30.6	27	31.5	38.1	37.6	34.3	39.3	28.4
Solids	%Fe	52.5	49.7	49	49.8	62	56.1	67	65.8	68.2	65.8	67.6	68.4
	%Cu	0.023	0.043	0.072	0.03	0.03	0.05	0.02	0	0.003	0.02	0.02	0.02
	%Cl	16.5	17.7	12.8	9.9	7.6	10.9	1.3	1.2	1.2	1.7	1.3	1
	% hematite			12.3	21.6	72.2	35.5	98	96.4	N/A	N/A	N/A	N/A
	% akaganeite			44	48.9	10.8	53						
	% iron oxide chloride			28	21.4	15.3	7.8		1.1				
	Rokuhnite			13					1.9				
	D_{80}, µm	44.6	35.8	40.2	22.6	48.7	26.7	15.6	17.6	24.1	17.7	18.1	15.6

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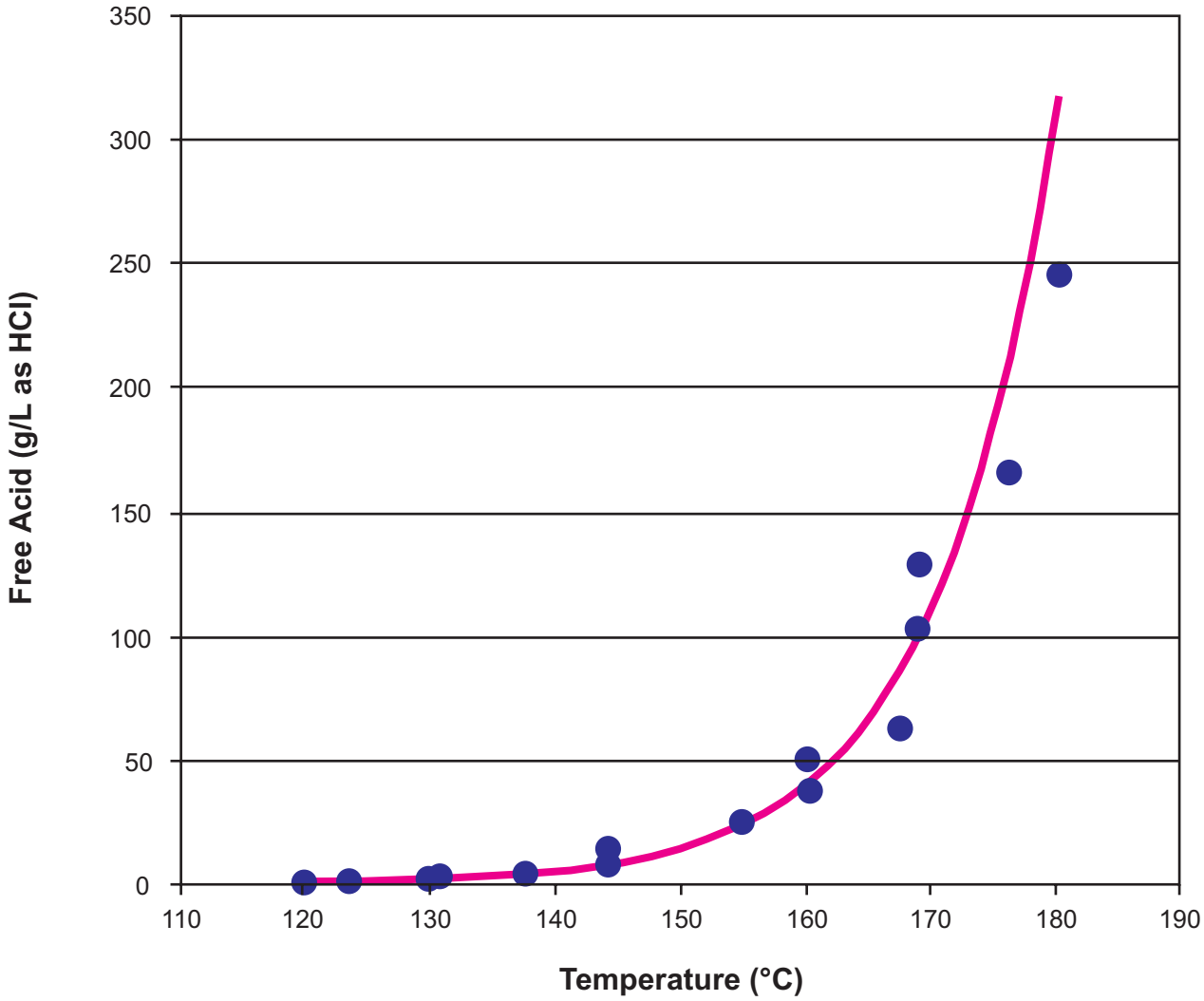


Figure 13-3

Legend:
- *Free Acid*
- *Expon. (Free acid)*

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Boiler Temperature Vs. Condensate Acid Strength

November 2011

Source: SGS Minerals Services, 2011.

Next, an extended test was carried out to demonstrate steady operation of the hydrolysis step. The boiler temperature was set to 140ºC for the entire test. A summary of the parameters used in the extended hydrolysis test is shown in Table 13-17.

TABLE 13-17 EXTENDED HYDROLYSIS TEST PARAMETERS
Starfield Resources Inc. – Ferguson Lake Project

Condition	Run Time	Avg Feed Flows		Water	Temperature	Reactor Condensate		Pure HCl
	h	g/min	mL/min	mL/min	°C	g/min	avg %HCl	mg/min
1	13.3	2.6	1.8	0.48	180	1.35	22.9	256
2	12.6	2.9	2	0.89	180	1.57	21.3	335
3	11.9	3.2	2.2	0.74	180	1.5	20.5	307
4	12.7	5.6	3.7	0.82	180	1.69	20.6	350
5	11.3	5.4	3.6	1.03	185	2.26	24.8	564
6	14	8.5	5.7	1.4	183	2.98	22.4	679

Figure 13-4 depicts temperature trends, as well as feed and boiler condensate flows. Condensate flow was very steady, just increasing when feed flow was increased. Figure 13-5 depicts condensate hourly amounts and free acid levels. Free acid values were between 1.5 g/L and 6 g/L (below 0.6%). No solids were found at the end of the test which confirmed hydrolysis did not take place in the boiler.

FIGURE 13-4 BOILER TEMPERATURE AND FLOWS

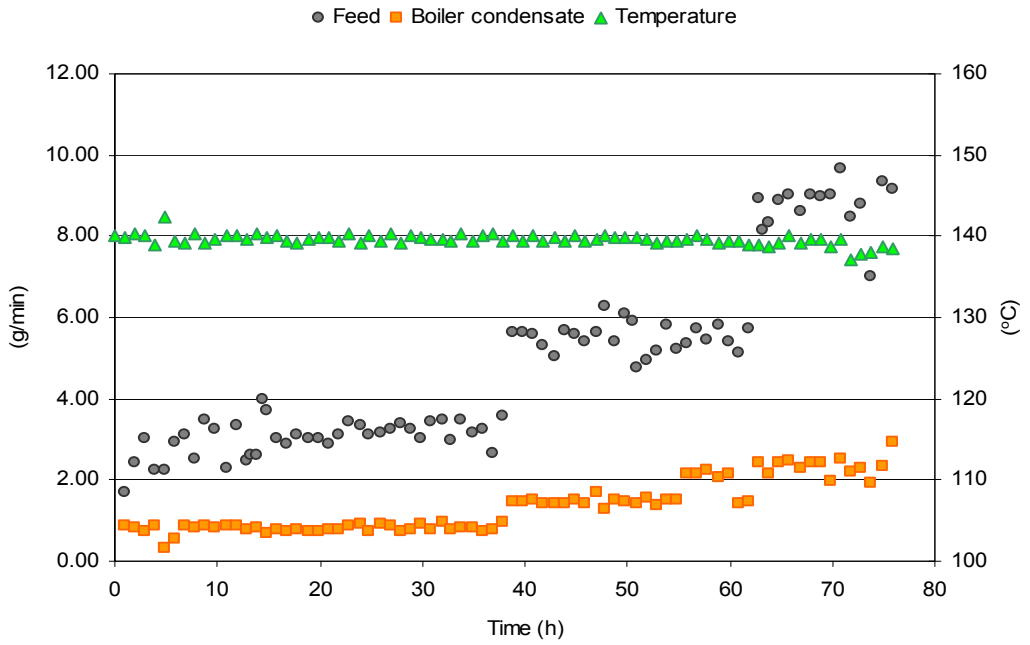
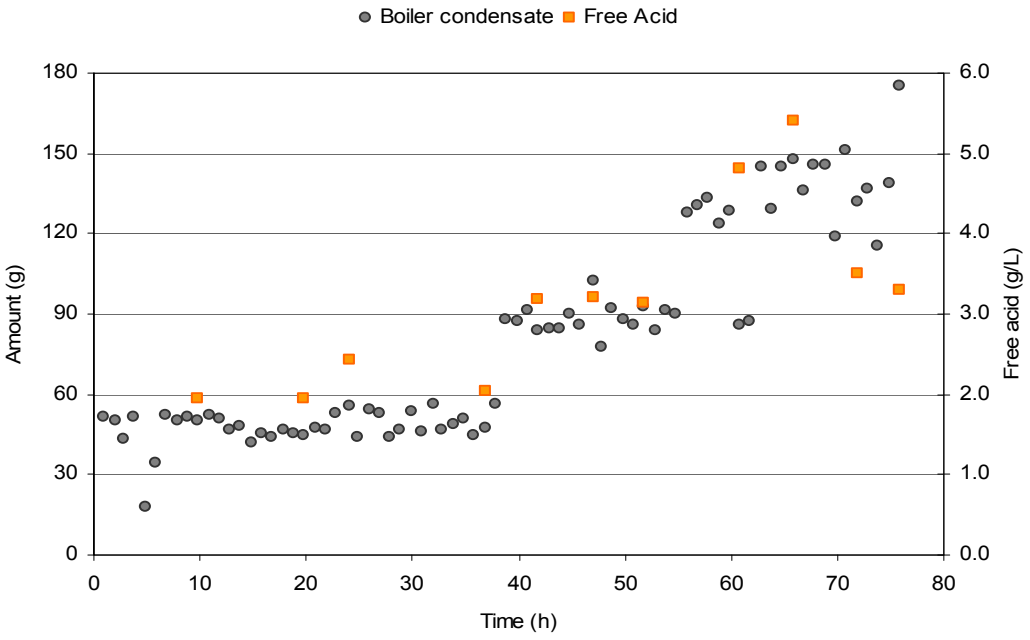


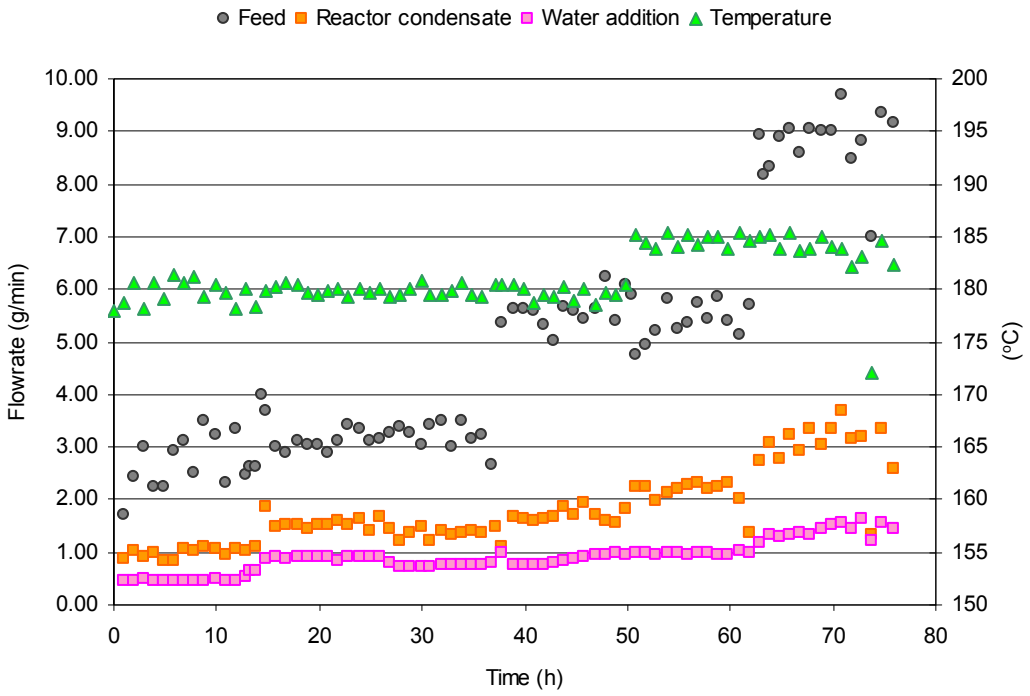
FIGURE 13-5 BOILER CONDENSATE AND FREE ACID



Operating conditions were varied in the reactor to investigate the effect of water addition, temperature and feed flow on the hydrolysis performance. Figure 13-6 shows the main reactor parameter trends. Since it was not possible to determine the exact amount of solids inside the reactor for every condition, hydrolysis performance was calculated from

the acid production (two moles of hydrolysed ferric chloride equals one mole of hematite and six moles of HCl).

FIGURE 13-6 REACTOR OPERATING PARAMETERS



As shown in Figures 13-7 and 13-8, when feed flow was increased acid production also increased. It is of interest to note that although acid production did increase it did not increase in the same proportion as the feed flow and water increases, which could indicate that residence time was a factor in the hydrolysis performance.

FIGURE 13-7 COMPARISON OF CONDITIONS 1, 2 AND 3

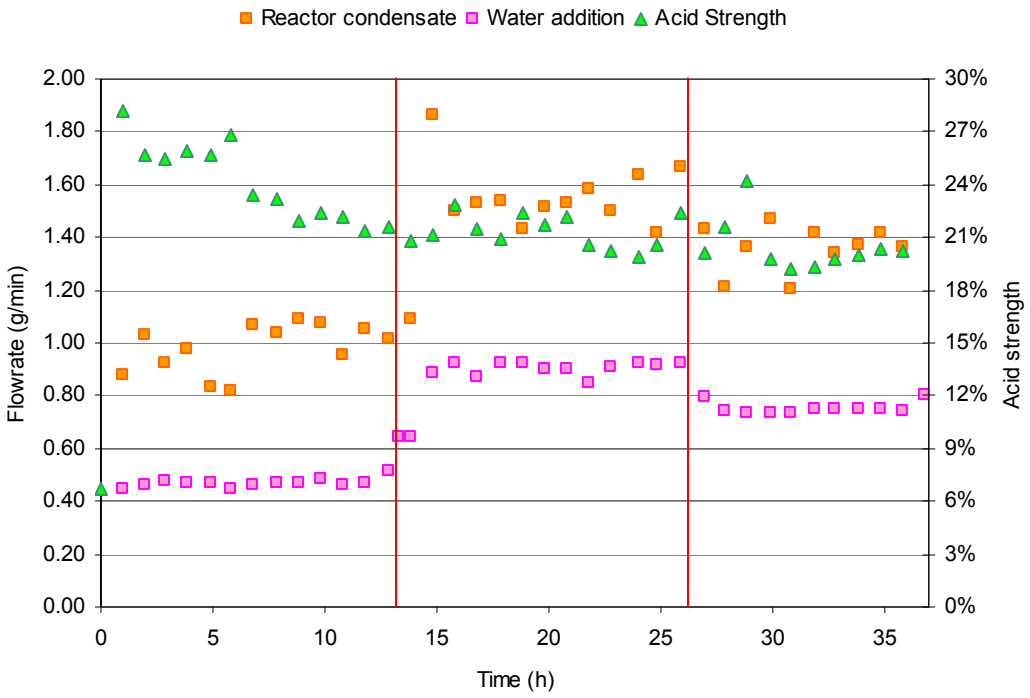


FIGURE 13-8 COMPARISON OF CONDITIONS 1 AND 4

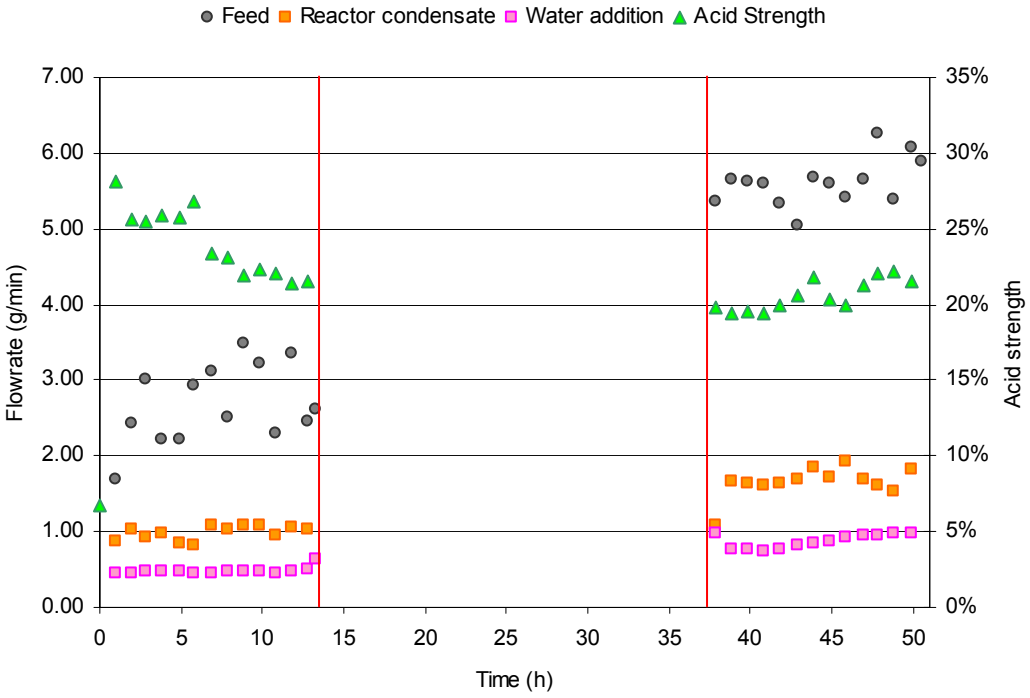


Figure 13-9 demonstrates that increasing the reactor temperature from 180ºC to 185ºC while keeping the water addition constant resulted in an increased reactor condensate production rate (from 1.69 g/min to 2.26 g/min). Figure 13-10 shows an increase in acid strength (20.6% to 24.8%) in addition to an increase in HCl production (0.35 g/min to 0.56 g/min. Figure 13-11 depicts the main parameters from Conditions 5 and 6. As shown in Figure 13-12, the acid production rate increased when the feed flow was increased while keeping acid strength fairly constant.

FIGURE 13-9 COMPARISON #1 - CONDITIONS 4 AND 5

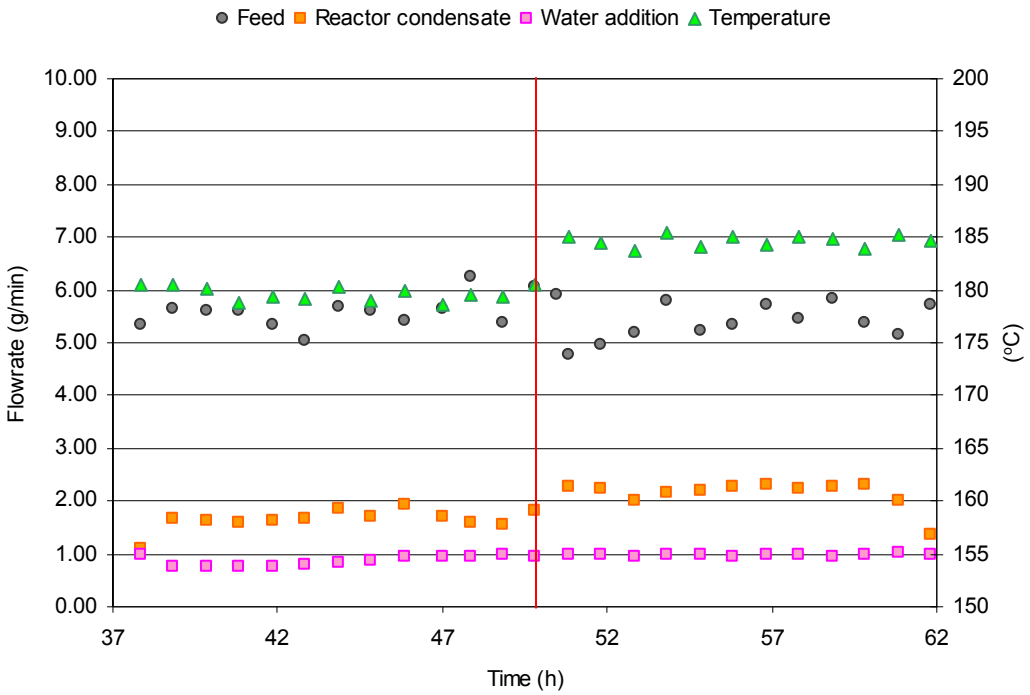


FIGURE 13-10 COMPARISON #2 - CONDITIONS 4 AND 5

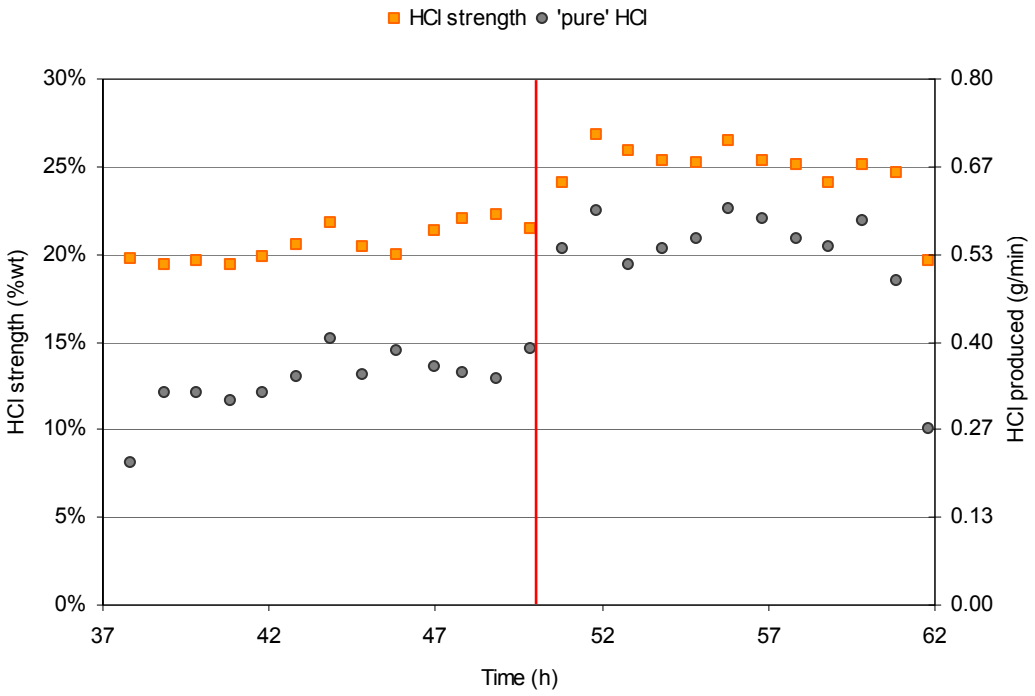


FIGURE 13-11 COMPARISON #1 - CONDITIONS 5 AND 6

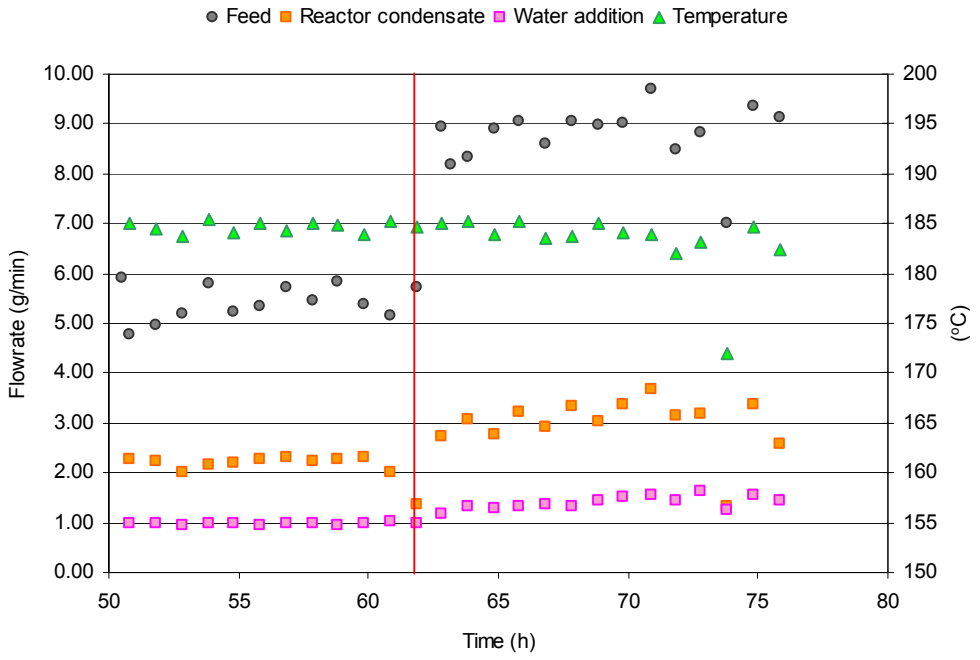
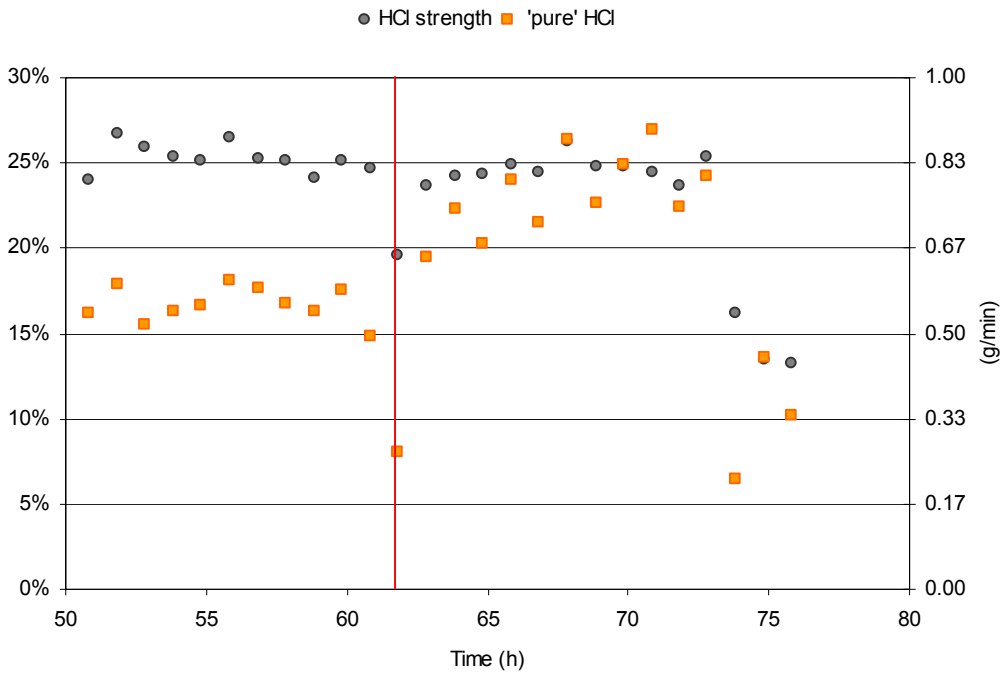


FIGURE 13-12 COMPARISON #2 - CONDITIONS 5 AND 6



Although XRD analysis confirmed solids produced in hydrolysis were mostly hematite (97% of total solids), the chloride content was still high (around 1%). Based on that, and the suspicion that chloride was not part of the crystal structure but crystallized chloride salts, a series of tests was done to “wash out” the chloride from the solids. These tests entailed re-pulping the solids from hydrolysis in solutions of hydrochloric acid of varying concentration and temperature.

A summary of hematite washing tests is shown in Table 13-18, using different hydrochloric acid solutions and temperatures. Neither temperature, nor acid, produced significant differences regarding chloride content in the solids after the tests. The chloride content was reduced to around 0.6-0.7%.

TABLE 13-18 HEMATITE WASHING TESTS
Starfield Resources Inc. – Ferguson Lake Project

Test ID	Initial Solids			HCl	Temperature	Final Solution	Final Solids	Raw extraction
	Source	% H_2O	% Cl	g/L	°C	mg/L Fe	% Cl	
CW1	Final Reactor Solids	11.1	0.9	12	55	884	0.67	0.25
CW2	Cond 4 Solids	8.9	1.14	56	52.7	16,100	0.62	0.46
CW3	Condition 5 Solids	12.5	0.9	106	56.2	40,400	0.74	0.18
CW4	Cond 4 Solids	8.9	1.14	56	94.1	23,000	0.67	0.42
Average	10.4	1					0.67	0.33

CONCLUSIONS PERTAINING TO 2010 OXIDATION TESTWORK

1. Continuous oxidation of a ferrous chloride solution was successful at 120ºC, producing a precipitate consisting of 97% hematite.

2. Increasing the oxidation reactor temperature to above 120ºC caused solids to accumulate on the reactor walls and decreasing the reactor temperature below 120ºC decreased the oxidation efficiency.

3. Both pitch-blade and Rushton turbine impellers were tested successfully. Under otherwise similar conditions, the Rushton turbine led to slightly higher oxidation efficiency, however, it could not be tested in the long-run test due to a corrosion problem. A custom, Teflon-only, Rushton turbine impeller was designed and constructed, and is recommended for use in further testing.

4. Although adequate oxygen dispersion was accomplished, the sparging operation was problematic due to continuous plugging of the sparger. For future lab tests or scale-up, a new sparger should be designed. Oxygen utilization should also be optimized by testing lower oxygen addition rates.

CONCLUSIONS PERTAINING TO 2010 HYDROLYSIS TESTWORK

1. Continuous hydrolysis of a ferric chloride solution was successful, producing a precipitate with 69% iron and 0.9% chloride, and hydrochloric acid at 22% strength.

2. Water addition is a key factor in hydrolysis performance, affecting solids composition, extent of hydrolysis, acid production, and acid strength. Increasing the amount of water increased the extent of hydrolysis and the amount of acid produced at the cost of lower acid strength. The exact water addition should be based on an evaluation of acid strength versus the extent of hydrolysis (or amount of hematite and acid produced).

3. Increasing hydrolysis temperature led to an increased extent of hydrolysis, amount of acid produced, and acid strength, however, it also increased slurry viscosity and "line plugging" problems increased significantly. Increasing temperature above 185ºC could make hydrolysis "challenging" from an operational point of view.

4. Acid washing of the hematite solids was able to reduce the chloride content from 1% to about 0.6%. A more specific study of the hematite structure is recommended.

METALLURGICAL TESTWORK (2011)

FLOTATION

A flotation test program was carried out to determine whether or not the Ferguson Lake mineralization is amenable to upgrading by means of flotation prior to further treatment in the hydrometallurgical process. The objectives were:

1. Recover essentially all of the sulphide minerals into a bulk flotation concentrate while rejecting most of the gangue minerals.
2. Generate a bulk concentrate for further hydrometallurgical development work.

Gangue minerals that do not dissolve in the hydrometallurgical process report to the leach residue with the PGEs and lower the PGE grade of the residue. The gangue minerals that do dissolve in the leach increase the acid requirement in the leach.

Full details of this work are available in Appendix 1 (SGS report on Project 12323-002, January 17, 2011).

A series of bench scale flotation tests established that it is possible to recover essentially all the sulphide minerals into a bulk sulphide concentrate while also rejecting about 80 percent of the gangue minerals. Table 13-19 lists the distribution of precious metals measured in these tests.

TABLE 13-19 PGE LOSSES TO ROUGHER TAILS
Starfield Resources Inc. – Ferguson Lake Project

	Mass, %	Au, g/t	Pt, g/t	Pd, g/t
Rougher Tails	20	0.02	0.03	0.22
Feed	100	0.08	0.2	2.28
Recovery		95	97	98.1

A bulk flotation test was done, using the flotation parameters found in the bench tests, to produce flotation concentrate for further work. Table 13-20 shows the results of the bulk flotation test.

TABLE 13-20 BULK FLOTATION MASS BALANCE
Starfield Resources Inc. – Ferguson Lake Project

Product	Weight		Assays, %			% Distribution		
	kg	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc	184.7	84	1.7	1.03	35.5	99.6	97.8	99.1
Rougher Tail	35.2	16	0.04	0.12	1.71	0.4	2.2	0.9
Head (calc.)	219.9	100	1.43	0.88	30.1	100	100	100
Head (assay)			1.48	0.87	29			

PRIMARY LEACH

The primary leach was tested in two continuous mini-pilot campaigns. The objective of this work was to produce primary leach solution (PLS) from actual Ferguson Lake ore to prove the viability of oxidation and hydrolysis processes using actual PLS and primary leach residue for testing the Secondary Leaching process.

Samples of Ferguson Lake ore and flotation concentrate were subjected to batch hydrochloric acid leaching in which iron, in the form of pyrrhotite, is leached producing ferrous chloride and hydrogen sulphide (see the equation below).

$$Fe_7S_8 + 14HCl_{(ia)} = 7FeCl_{2(ia)} + 7H_2S_{(g)} + S$$

Since the overall process uses hydrogen sulphide as its energy source, the sulphur balance is important, hence hydrogen sulphide scrubbing was also optimized. Copper chloride, copper sulphate, and sodium hydroxide were used for H_2S scrubbing. The scrubbing system consisted of three, in-series, four litre vessels (see Figure 13-13). The exhaust gas line was connected to a vacuum pump to “pull” the gas throughout the scrubbing system. The third vessel was always filled with a solution of 100 g/L of sodium hydroxide, while the first two vessels were tested with different solutions. Based on the equations below, hydrogen sulphide reacted with the scrubbing media, producing either a solid, or aqueous sulphur compound. The compound was analyzed for sulphur to calculate the amount of hydrogen sulphide produced.

$$2H_2S_{(g)} + 2CuCl_{2(ia)} = Cu_2S + S + 4HCl_{(ia)}$$

$$2H_2S_{(g)} + 2CuSO_{4(ia)} = Cu_2S + S + 2H_2SO_4$$

$$H_2S_{(g)} + NaOH_{(ia)} = NaSH_{(ia)} + H_2O$$

FIGURE 13-13 SCRUBBING SYSTEM FOR BENCH TESTING

Table 13-21 depicts the different parameters tested in each bench test. From L1 to L5, Ferguson Lake ore was used as feed, while L6 used flotation concentrate instead of ore. In L1, L2, and L3, different scrubbing reagents were tested, and from that point on, sodium hydroxide was used for the remainder of the tests. Also, a continuous scrubbing test was run to simulate the scrubbing of an off gas produced from continuous leaching.

TABLE 13-21 BENCH TEST CONDITIONS
Starfield Resources Inc. – Ferguson Lake Project

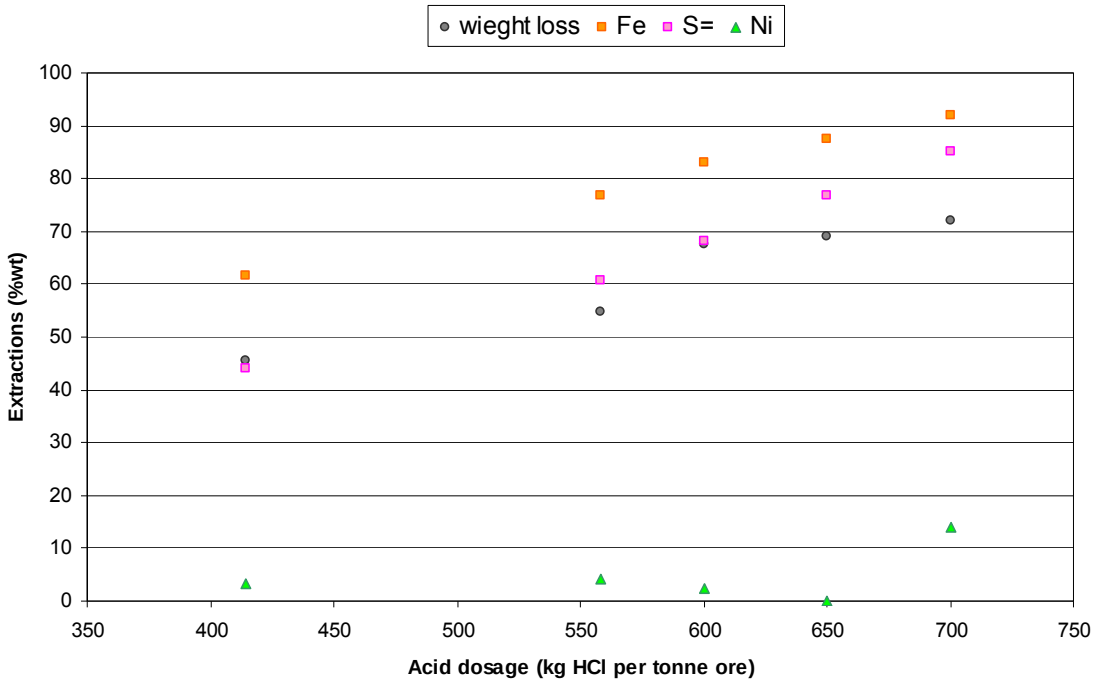
Test ID	Feed	HCl dose kg/t	Scrubbing media
L1	Ferguson Lake ore	558	$CuCl_2$
L2	Ferguson Lake ore	415	$CuSO_4$
L3	Ferguson Lake ore	650	NaOH
L4	Ferguson Lake ore	600	NaOH
L5	Ferguson Lake ore	700	NaOH
L6	Float Concentrate	650	NaOH

A summary of the bench test results is tabulated in Table 13-22. Each test was run for approximately three to four hours, depending on how fast the acid was added. Kinetic samples (solids and solution) were obtained as soon as all the acid was added and one hour after acid addition. Each test was stopped two hours after the acid addition was completed. Figure 13-14 is a plot of the extractions achieved against acid addition.

TABLE 13-22 BENCH TEST RESULTS
Starfield Resources Inc. – Ferguson Lake Project

Test ID	HCl Dose	Extraction - %wt (Cu-tie Method)							Free Acid
	kg/t	Mass	Fe	Ni	Co	Mg	S	S=	g/L
L1	558	54.7	77	3	1	12	54	61	
L2	415	45.6	62	4	0	27	29	44	0
L3	649	69.2	87	0	0	25	59	77	1
L4	599	67.5	83	2	0	23	66	68	1
L5	699	71.9	92	14	0	29	55	85	8
L6	650	78.2	89	11	20	54	74	83	1

FIGURE 13-14 BENCH TESTS (L1 TO L5 EXTRACTIONS)



PILOT PLANT

A schematic of the leaching circuit is shown in Figure 13-15 and a photograph of the circuit is included in Figure 13-16. Flotation concentrate was mixed with PLS which was recycled from the filter. A synthetic solution was used to simulate the PLS at start-up. The feed system consisted of two tanks: one feeding to the circuit, while the next feed batch was being made up in the other. The slurry was pumped from the bottom of the feed tank to a pre-heater tank located atop the circuit, in which the pulp was heated up to 105°C, then overflowed by gravity to the first reactor (R1).

The leaching circuit consisted of four stirred-tank reactors (R1, R2, R3, and R4) with a total of two hours residence time. Each reactor (as well as the pre-heater) was made from glass with an active volume of 1.5 L and an internal diameter of 97 mm. Each reactor was equipped with four baffles (13 mm wide and one-third of the reactor height, located at the bottom of the tank) as well as an agitator with a 55 mm diameter pitch-blade-type impeller (the impeller and the shaft were made of Teflon). A vapour condenser was connected at the top of each reactor, and a line from the top of the condensers was directed to the scrubbing system. The temperature in each reactor was controlled by an electrical heating mantle. Nitrogen was added to each reactor, the pre-heater, and the feed tanks in order to avoid any oxidation. Due to the foaming observed in the bench tests, hydrochloric acid addition was split into two and added to R1 and R2 tanks.

The slurry was transferred through the leaching circuit by gravity to a Buchner funnel-type filter. Filtrate was recycled back to the circuit and/or storage for further testwork. Wet cakes were washed and stored for further testwork.

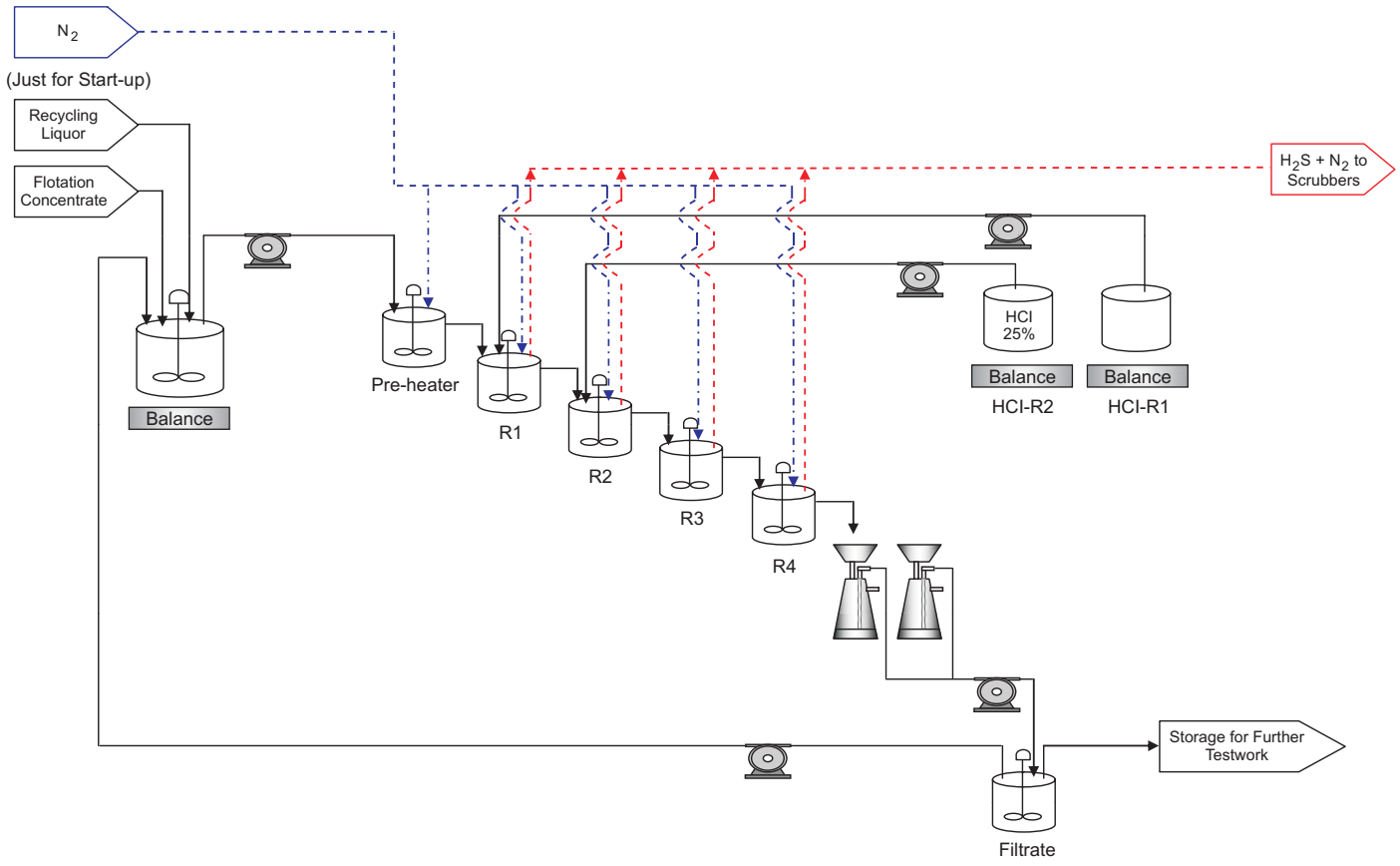


Figure 13-15

Starfield Resources Inc.

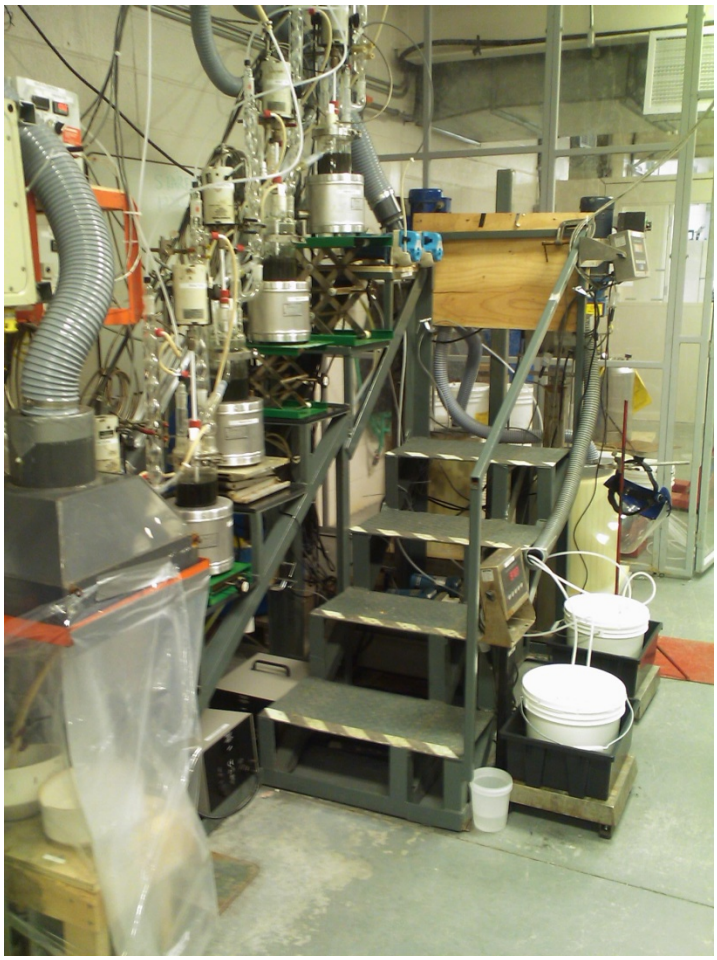
Ferguson Lake Project

Nunavut, Canada

Leaching Circuit Diagram

November 2011 ***Source: SGS Minerals Services, 2011.***

FIGURE 13-16 PHOTOGRAPH OF LEACHING CIRCUIT



SCRUBBING CIRCUIT

A schematic of the scrubbing operation is shown in Figure 13-17 and a photograph of the circuit is included in Figure 13-18. A mixture of hydrogen sulphide, nitrogen, and probably some air and water was pulled out of the leaching circuit and directed to the scrubbing circuit by a large peristaltic pump. The scrubbing circuit consisted of two counter-current columns full with a solution of 100 g/L sodium hydroxide. Each column was made of clear PVC with an active volume of 14 L and a diameter of 102 mm. Gas coming from the leaching circuit was pumped to the bottom of Column 2, proceeding all the way to the top of Column 2, and finally to the bottom of Column 1. Clean gas left Column 1 through the top of the column and was directed to the main plant scrubber. A solution of 100 g/L sodium hydroxide was pumped in counter-current to the gas flow, i.e., it was pumped to Column 1, then from Column 1 to Column 2, and finally to a drum for analysis and disposal.

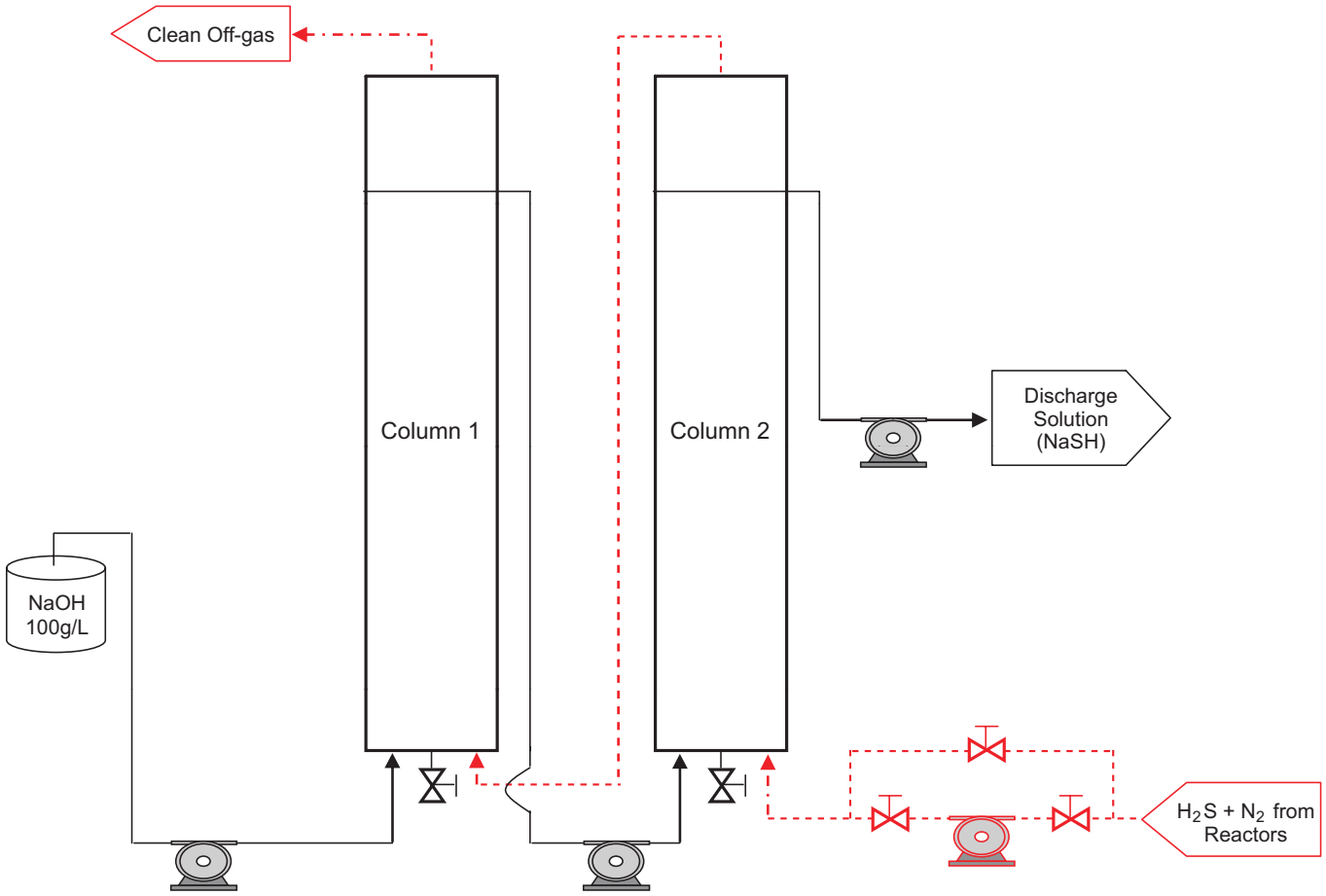


Figure 13-17

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada
Scrubbing Flowsheet

November 2011 ***Source: SGS Minerals Services, 2011.***

FIGURE 13-18 PHOTOGRAPH OF SCRUBBING CIRCUIT

The circuit was run in two campaigns: PP1 and PP2. Each campaign was run for one week and there were two weeks of down-time between the campaigns to allow detailed, data analysis.

The bulk flotation concentrate was divided into 11 charges. Each charge was used to make up a feed batch and a sample from each batch was assayed. Flotation concentrate solids compositions are shown in Tables 13-23 and 13-24.

TABLE 13-23 PP1 CONCENTRATE CHARGES ASSAYS (%WT)
Starfield Resources Inc. – Ferguson Lake Project

Charges	Fe	Ni	Cu	Co	Mg	S	S=
Charge 1	55.9	1.01	1.72	0.1	0.070	33.6	33.3
Charge 2	56.2	1.02	1.68	0.1	0.088	33.2	33.2
Charge 3	56.0	1.02	1.70	0.1	0.083	33.4	33.4
Charge 4	56.7	1.04	1.74	0.1	0.080	33.9	33.4
Charge 5	56.0	1.03	1.74	0.1	0.075	34.3	33.0
Average	56.2	1.02	1.72	0.1	0.079	33.7	33.3
Min	55.9	1.01	1.68	0.1	0.070	33.2	33.0
Max	56.7	1.04	1.74	0.1	0.088	34.3	33.4
Rel.STD	1%	1%	2%	0%	9%	1%	1%

TABLE 13-24 PP2 CONCENTRATE CHARGES ASSAYS (%WT)
Starfield Resources Inc. – Ferguson Lake Project

Charges	Fe	Ni	Cu	Co	Mg	S	S=
Charge 6	56.4	1.02	1.67	0.10	0.070	34.6	33.7
Charge 7	56.6	1.04	1.64	0.10	0.070	34.8	34.5
Charge 8	58.0	1.06	1.72	0.11	0.079	36.6	34.4
Charge 9	56.6	1.03	1.72	0.10	0.078	36.0	33.2
Charge 10	54.7	1.03	1.81	0.10	0.081	36.2	32.3
Charge 11	55.4	1.01	1.67	0.10	0.073	35.9	33.8
Average	56.3	1.03	1.71	0.10	0.075	35.7	33.7
Min	54.7	1.01	1.64	0.10	0.070	34.6	32.3
Max	58.0	1.06	1.81	0.11	0.081	36.6	34.5
Rel.STD	2%	2%	4%	4%	6%	2%	2%

In addition to a sample from each concentrate charge, the feed pulp was characterized by 12-hour composite samples taken just ahead of the pre-heater tank. The feed liquor and solids compositions are displayed in Tables 13-25 and 13-26 for PP1 and in Tables 13-27 and 13-28 for PP2.

TABLE 13-25 PP1 FEED SOLUTION ASSAYS (MG/L)
Starfield Resources Inc. – Ferguson Lake Project

Date	Fe	Fe(II)	Ni	Cu	Co	Mg
13-Oct AM	198,000	191,870	102	102	<5	1,820
13-Oct PM	188,700	187,300	121	243	<5	1,370
14-Oct AM	212,000	210,570	116	229	6	1,322
14-Oct PM	212,300	208,000	153	246	8	1,198
15-Oct AM	223,800	212,600	68	177	<5	899
15-Oct PM	229,635	203,800	117	208	7	845
16-Oct AM	227,200	213,100	179	254	9	802
Average	213,091	203,891	122	208	6	1,179
Min	188,700	187,300	68	102	<5	802
Max	229,635	213,100	179	254	9	1,820
Rel.STD	7%	5%	29%	26%	25%	31%

TABLE 13-26 PP1 FEED SOLIDS ASSAYS (%WT)
Starfield Resources Inc. – Ferguson Lake Project

Date	Fe	Ni	Cu	Co	Mg	S	S=
13-Oct AM	55.4	0.97	1.73	0.10	0.075	33.6	31.8
13-Oct PM	54.8	1.04	1.67	0.11	0.056	36.4	33.7
14-Oct AM	57.8	1.02	1.65	0.10	0.074	35.6	31.9
14-Oct PM	56.6	1.02	1.66	0.10	0.067	34.2	30.6
15-Oct AM	55.8	1.06	1.69	0.10	0.072	35.1	30.7
15-Oct PM	58.4	1.07	1.72	0.11	0.014	35.2	34.3
16-Oct AM	56.4	1.06	1.83	0.11	0.063	35.0	34.9
Average	56.5	1.03	1.71	0.10	0.060	35.0	32.6
Min	54.8	0.97	1.65	0.10	0.014	33.6	30.6
Max	58.4	1.07	1.83	0.11	0.075	36.4	34.9
Rel.STD	2%	3%	4%	5%	36%	3%	5%

TABLE 13-27 PP2 FEED SOLUTION ASSAYS (MG/L)
Starfield Resources Inc. – Ferguson Lake Project

Date	Fe	Fe(II)	Ni	Cu	Co	Mg
01-Nov PM	239,600	217,800	310	89	8	494
02-Nov AM	234100	234,100	445	132	12	435
02-Nov PM	239,900	217,400	656	281	21	474
03-Nov AM	236,500	213,700	694	332	22	481
03-Nov PM	236,200	215,700	597	449	23	397
04-Nov AM	230,900	214,300	472	363	15	354
04-Nov PM	232,400	198,800	406	344	12	314
05-Nov AM	214,400	199,600	374	258	10	270
05-Nov PM	218,100	203,260	436	382	14	295
Average	231,344	212,740	488	292	15	390
Min	214,400	198,800	310	89	8	270
Max	239,900	234,100	694	449	23	494
Rel.STD	4%	5%	27%	40%	36%	22%

TABLE 13-28 PP2 FEED SOLIDS ASSAYS (%WT)
Starfield Resources Inc. – Ferguson Lake Project

Date	Fe	Ni	Cu	Co	Mg	S	S=
01-Nov PM	57.0	1.07	1.81	0.11	0.065	35.8	32.5
02-Nov AM	57.9	1.04	1.67	0.10	0.064	36.5	33.4
02-Nov PM	57.8	1.06	1.75	0.10	0.053	37.5	34.5
03-Nov AM	56.4	1.11	1.8	0.11	0.067	37.3	34.1
03-Nov PM	56.6	1.07	1.85	0.11	0.063	37.5	35.2
04-Nov AM	56.1	1.02	1.7	0.10	0.070	37.1	34.8
04-Nov PM	56.7	1.01	1.71	0.10	0.069	36.0	32.8
05-Nov AM	55.8	1.07	2.02	0.11	0.085	36.9	33.5
05-Nov PM	56.2	1.02	1.76	0.10	0.072	36.9	32.4
Average	56.7	1.05	1.79	0.10	0.068	36.8	33.7
Min	55.8	1.01	1.67	0.10	0.053	35.8	32.4
Max	57.9	1.11	2.02	0.11	0.085	37.5	35.2
Rel.STD	1%	3%	6%	5%	13%	2%	3%

Start-up parameters for the circuit are provided in Table 13-29

TABLE 13-29 START-UP PARAMETERS
Starfield Resources Inc. – Ferguson Lake Project

Parameters	Feed mL/min	Temperature °C	N_2 mL/min	Acid g/min	NaOH mL/min
Feed Tank	20	-	300	-	-
Pre-heater	-	105	100	13	-
R1	-	105	100	13	-
R2	-	105	100	-	-
R3	-	105	100	-	-
R4	-	105	100	-	-
Scrubber	-	-	-	-	38

Tables 13-30 and 13-31 present 12-hour, average flows and pulp densities for the relevant streams used in the mass balance and efficiency calculations. All flows and pulp densities are measured values. Feed percentage solids were calculated at the time each feed batch was made up. Wet cakes (leach residue) were collected in 12-hour periods and not as a continuous stream.

TABLE 13-30 PP1 AVERAGE FLOWS AND DENSITIES
Starfield Resources Inc. – Ferguson Lake Project

Date	Availability	Feed			Acid addition, g/min (25%HCl)		Filtrate	
	%	Slurry		Solids Density	R1	R2	Flow Rate	Density
		mL/min	g/mL	%wt			mL/min	g/mL
13-Oct AM	100	20	1.673	0.280	11	11	48	1.39
13-Oct PM	100	20	1.474	0.280	12	12	30	1.422
14-Oct AM	100	20	1.665	0.280	17	0	29	1.426
14-Oct PM	100	21	1.712	0.280	26	0	40	1.425
15-Oct AM	96	21	1.757	0.280	22	0	30	1.438
15-Oct PM	94	21	1.726	0.280	21	0	33	1.441
16-Oct AM	100	21	1.757	0.280	26	0	32	1.432

TABLE 13-31 PP2 AVERAGE FLOWS AND DENSITIES
Starfield Resources Inc. – Ferguson Lake Project

Date	Availability	Feed			Acid addition, g/min (25%HCl)		Filtrate	
	%	Slurry		Solids Density	R1	R2	Flow Rate	Density
		mL/min	g/mL	%wt			mL/min	g/mL
01-Nov PM	94	20	1.707	0.280	23	0	37	1.433
02-Nov AM	100	20	1.736	0.279	23	0	43	1.448
02-Nov PM	97	21	1.443	0.274	23	0	28	1.407
03-Nov AM	97	21	1.725	0.275	24	0	39	1.410
03-Nov PM	92	23	1.410	0.280	23	0	41	1.385
04-Nov AM	100	22	1.694	0.280	29	0	40	1.387
04-Nov PM	100	20	1.721	0.280	29	0	44	1.384
05-Nov AM	100	20	1.648	0.278	28	0	42	1.367
05-Nov PM	97	20	1.712	0.277	27	0	37	1.382

Temperature control was steady throughout both campaigns, as is shown in Tables 13-32 and 13-33.

TABLE 13-32 PP1 AVERAGE TEMPERATURES
Starfield Resources Inc. – Ferguson Lake Project

Date	Temperature (°C)				
	Pre-heater	R1	R2	R3	R4
13-Oct AM	104.6	104.7	104.8	104.9	104.5
13-Oct PM	104.8	104.1	105.5	105.3	105.5
14-Oct AM	104.4	104.1	105.2	105.5	105.7
14-Oct PM	105.3	102.7	105.0	105.3	106.0
15-Oct AM	104.5	103.2	105.0	105.0	105.7
15-Oct PM	104.8	102.0	104.9	105.2	105.3
16-Oct AM	104.9	104.1	105.2	105.3	105.5

TABLE 13-33 PP2 AVERAGE TEMPERATURES
Starfield Resources Inc. – Ferguson Lake Project

Date	Temperature (°C)				
	Pre-heater	R1	R2	R3	R4
01-Nov PM	104.2	103.8	105.1	105.4	104.9
02-Nov AM	104.7	103.6	105.3	102.8	104.8
02-Nov PM	106.8	104.3	105.0	105.4	105.0
03-Nov AM	106.6	101.4	105.0	105.2	105.0
03-Nov PM	106.9	102.9	105.0	105.4	104.5
04-Nov AM	105.4	100.0	104.9	105.3	104.9
04-Nov PM	105.8	100.1	104.8	104.3	104.6
05-Nov AM	106.4	101.1	104.4	105.1	104.5
05-Nov PM	106.4	101.1	105.3	104.8	104.6

The average acid addition was 570 kg/t and 609 kg/t for PP1 and PP2, respectively. Daily values are shown in Tables 13-34 and 13-35. Due to crystallization in the filtrate, and its high iron concentration, it was decided to "dilute" the filtrate. This was accomplished by adding water to the acid. (This will most probably not be necessary at larger scale-in this work hear losses were significant because of the relatively small flows and high surface-to-volume ratios that are unavoidable at the scale of this work.)

TABLE 13-34 PP1 AVERAGE ACID CONSUMPTION
Starfield Resources Inc. – Ferguson Lake Project

Date	Ore	HCl @ 25%wt (g/min)			Dosage
	g/min	R1	R2	Total	kg/t ore
13-Oct AM	10.0	11	11	22	540
13-Oct PM	8.0	12	12	24	741
14-Oct AM	10.0	17	0	17	421
14-Oct PM	10.5	26	0	26	612
15-Oct AM	9.9	22	0	22	563
15-Oct PM	10.8	21	0	21	485
16-Oct AM	10.4	26	0	26	627
Average	9.9	19	3	22	570
Min	8.0	11	0	17	421
Max	10.8	26	12	26	741
Rel.STD	9%	33%	171%	14%	18%

TABLE 13-35 PP2 AVERAGE ACID CONSUMPTION
Starfield Resources Inc. – Ferguson Lake Project

Date	Ore	HCl @ 25%wt (g/min)				Dosage
	g/min	R1	R2	Total	% wt	kg/t ore
01-Nov PM	10.6	23	0	23	25.6	549
02-Nov AM	9.3	23	0	23	25.6	625
02-Nov PM	7.2	23	0	23	23.7	764
03-Nov AM	10.4	24	0	24	23.5	537
03-Nov PM	9.4	23	0	23	23.5	578
04-Nov AM	11.1	29	0	29	21.9	569
04-Nov PM	10.6	29	0	29	21.7	595
05-Nov AM	9.5	28	0	28	22.7	679
05-Nov PM	11.0	27	0	27	23.6	583
Average	9.9	25	0	25	23.5	609
Min	7.2	23	0	23	21.7	537
Max	11.1	29	0	29	25.6	764
Rel.STD	12%	11%	0%	11%	6%	12%

Overall leach extractions in the leaching circuit were calculated for both campaigns are listed in Table 13-36. These extractions were derived on a mass balance basis for the entire campaign and calculated as:

$$Extraction = \left(1 - \frac{Me\ Units\ in\ Residue}{Me\ Units\ in\ PLS + Me\ Units\ in\ Residue}\right) \times 100\%$$

TABLE 13-36 METAL EXTRACTIONS FOR PP1 AND PP2
Starfield Resources Inc. – Ferguson Lake Project

Campaign	Weight Loss	Fe	Ni	Cu	Co	Mg	S
PP1	0.70	0.73	0.06	0	0.02	0.50	0.63
PP2	0.76	0.83	0.12	0.01	0.05	0.38	0.69

Weight loss and iron extraction data were in line with expectations derived from the bench tests. Higher weight loss and Fe extraction in PP2 could be related to a higher acid addition in that campaign. Copper extraction was almost nil in both campaigns (1% in PP2). Cobalt and nickel extractions were relatively low. This confirmed that most of the base metals remain in the residue and can be extracted in the secondary leach. The sulphur extraction only reflects the sulphur converted to hydrogen sulphide, without taking into account the sulphide converted to elemental sulphur. Extractions were also calculated for the 12-hour compositing periods by a copper-tie method as follows:

$$Cu-tieExtraction = 1 - \frac{Me_{res}}{Me_{feed}} \times \frac{Cu_{feed}}{Cu_{res}}$$

Copper-tie-based extractions are plotted in Figures 13-19 and 13-20 for PP1 and PP2, respectively. The average extraction determined by this method was 6.7% for Ni and 3.4% for Co during PP1, and 12.9% for Ni and 2.3% for Co during PP2.

FIGURE 13-19 PP1 COPPER-TIE EXTRACTIONS

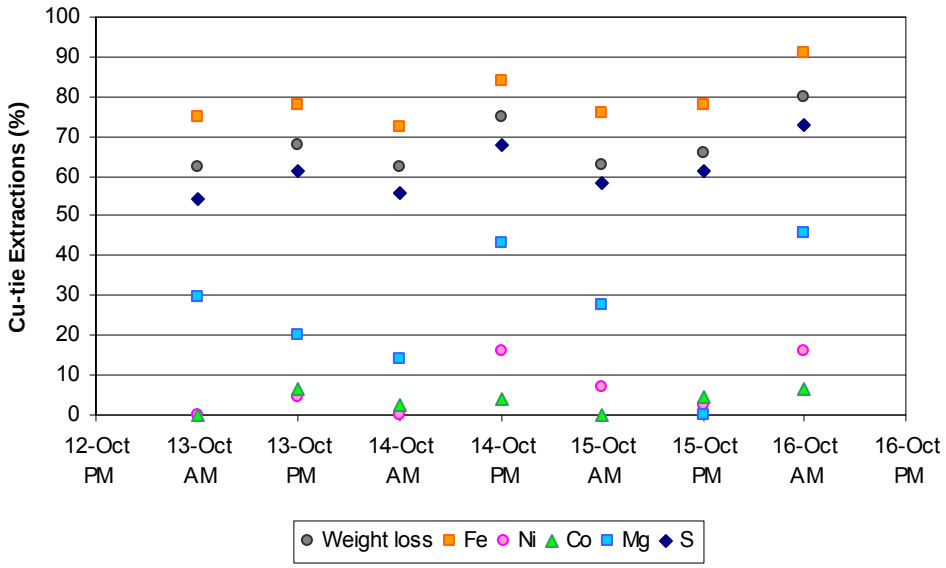
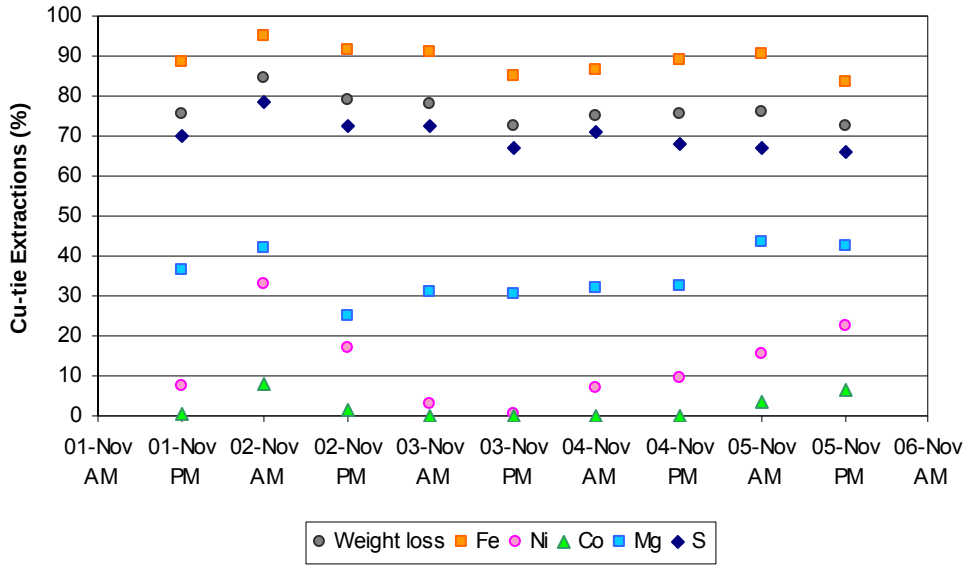


FIGURE 13-20 PP2 COPPER-TIE EXTRACTIONS



The leach discharge was characterized by 12-hour composite samples. Solution samples were taken after the filtration unit, and the filter cake was collected every 12 hours, washed, sampled, and the sample dried and analyzed. Filtrate and residue 12-hour average assays are shown in Tables 13-37 and 13-38 for PP1 and Tables 13-39 and 13-40 for PP2, respectively.

TABLE 13-37 PP1 FILTRATE ASSAYS (MG/L)
Starfield Resources Inc. – Ferguson Lake Project

Date	Fe	Fe(II)	Ni	Cu	Co	Mg
13-Oct AM	197,800	192,410	29	62	<5	1050
13-Oct PM	213,800	209,900	35	5	<5	890
14-Oct AM	219,800	219,800	<5	<5	<5	787
14-Oct PM	230,500	230,500	298	6	7	573
15-Oct AM	236,000	226,400	119	5	<5	550
15-Oct PM	248,550	207,400	<5	<5	<5	495
16-Oct AM	224,500	214,200	327	16	13	442
Average	224,421	214,373	117	15	6	684
Min	197,800	192,410	5	5	5	442
Max	248,550	230,500	327	62	13	1,050
Rel.STD	7%	6%	119%	143%	47%	33%

TABLE 13-38 PP1 RESIDUE (FILTER CAKE) COMPOSITION (%WT)
Starfield Resources Inc. – Ferguson Lake Project

Date	Fe	Ni	Cu	Co	Mg	S	S^{2-}
13-Oct AM	36.6	3.01	4.58	0.28	0.14	40.5	35.8
13-Oct PM	37.4	3.10	5.21	0.32	0.14	44.0	42.6
14-Oct AM	42.9	2.82	4.4	0.26	0.17	41.9	34.6
14-Oct PM	35.9	3.38	6.56	0.38	0.15	43.8	33.2
15-Oct AM	36.0	2.64	4.53	0.28	0.14	39.3	28.7
15-Oct PM	37.9	3.07	5.07	0.31	0.11	40.3	36.0
16-Oct AM	25.8	4.41	9.09	0.51	0.17	47.6	24.7
Average	36.1	3.20	5.63	0.33	0.146	42.5	33.7
Min	25.8	2.64	4.4	0.26	0.110	39.3	24.7
Max	42.9	4.41	9.09	0.51	0.170	47.6	42.6
Rel.STD	14%	18%	30%	26%	14%	7%	17%

TABLE 13-39 PP2 FILTRATE ASSAYS (MG/L)
Starfield Resources Inc. – Ferguson Lake Project

Date	Fe	Fe(II)	Ni	Cu	Co	Mg
01-Nov PM	237,400	217,800	484	<5	8	370
02-Nov AM	217,800	214,420	945	187	34	319
02-Nov PM	211,300	203,400	510	178	19	291
03-Nov AM	217,500	209,500	164	9	<5	282
03-Nov PM	203,800	197,500	230	<5	<5	210
04-Nov AM	207,000	199,600	216	<5	<5	211
04-Nov PM	206,800	195,800	255	7	<5	184
05-Nov AM	188,200	182,000	429	59	16	189
05-Nov PM	200,500	197,670	588	76	25	206
Average	210,033	201,966	425	59	14	251
Min	188,200	182,000	164	5	5	184
Max	237,400	217,800	945	187	34	370
Rel.STD	7%	5%	58%	127%	79%	26%

TABLE 13-40 PP2 RESIDUE (FILTER CAKE) COMPOSITION (%WT)
Starfield Resources Inc. – Ferguson Lake Project

Date	Fe	Ni	Cu	Co	Mg	S	S^{2-}
01-Nov PM	26.9	4.07	7.43	0.45	0.17	43.9	28.5
02-Nov AM	19.5	4.49	10.80	0.59	0.24	50.8	27.4
02-Nov PM	23.7	4.20	8.37	0.47	0.19	49.5	23.6
03-Nov AM	23.5	4.90	8.20	0.50	0.21	46.4	29.1
03-Nov PM	30.9	3.87	6.74	0.41	0.16	45.3	28.5
04-Nov AM	30.2	3.78	6.77	0.41	0.19	43.0	24.3
04-Nov PM	25.2	3.72	6.96	0.41	0.19	47.1	30.1
05-Nov AM	21.6	3.76	8.39	0.44	0.20	50.8	29.5
05-Nov PM	33.7	2.87	6.40	0.34	0.15	45.8	25.1
Average	26.1	3.96	7.78	0.45	0.19	47.0	27.3
Min	19.5	2.87	6.40	0.34	0.15	43.0	23.6
Max	33.7	4.90	10.80	0.59	0.24	50.8	30.1
Rel.STD	18%	14%	17%	16%	14%	6%	9%

Profile samples were taken from each tank once a day during the two campaigns. For PP1, after R2 (approximately one hour residence time), most of the compositions remained almost the same as shown in Figure 13-21. For PP2 (Figure 13-22), not only did they remain the same after R2, but nickel, copper, and cobalt compositions began decreasing after R2, which could indicate some of the base metals were leached after R2.

FIGURE 13-21 PP1 AVERAGE PROFILE SOLIDS COMPOSITION

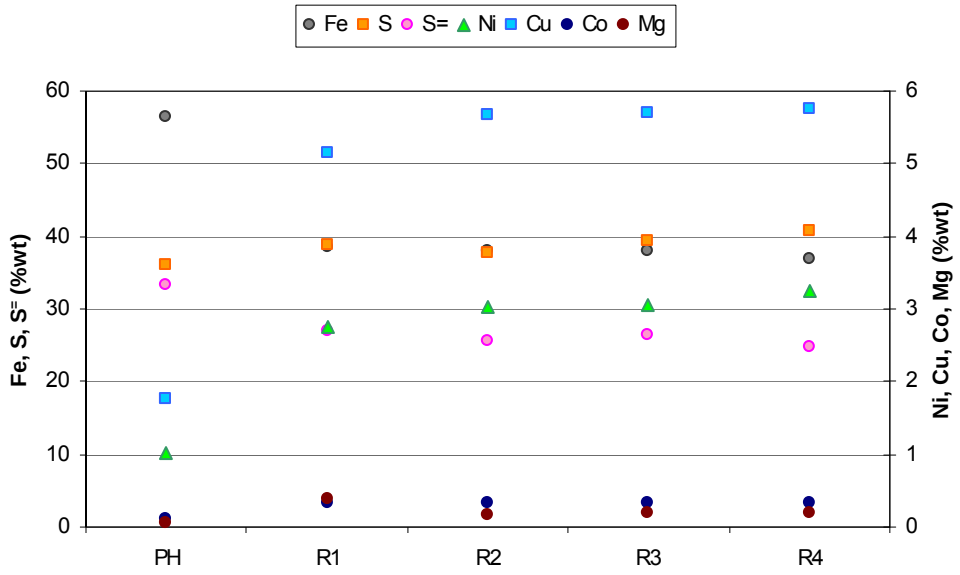
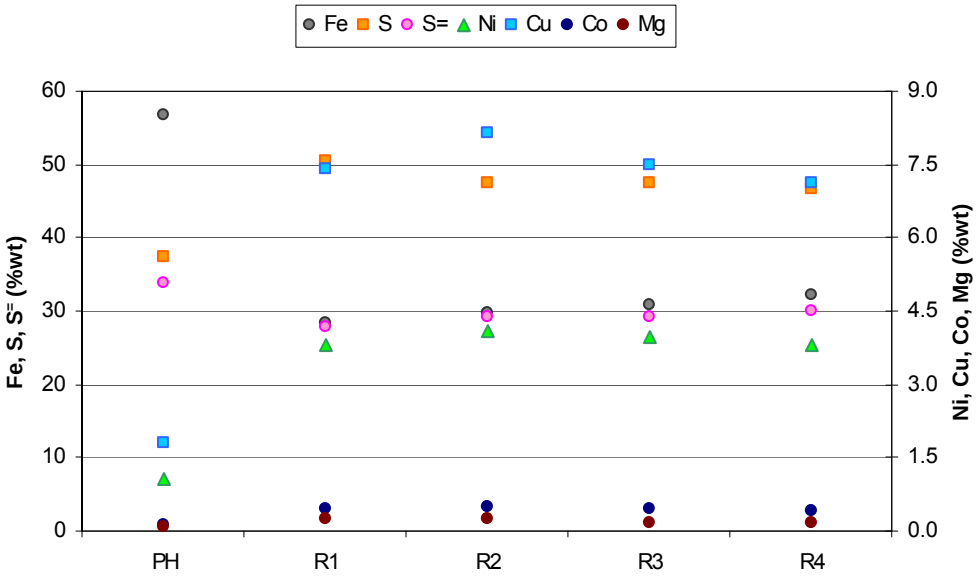


FIGURE 13-22 PP2 AVERAGE PROFILE SOLIDS COMPOSITION



Overall extraction efficiencies in the scrubbing circuit were calculated for both campaigns and are listed in Table 13-41. These extractions are derived from a mass balance for the entire campaign and are calculated as:

$$Efficiency = \frac{S_{SCRUBBING-LIQUOR}}{S_{OFF-GAS}} \times 100\%$$

TABLE 13-41 SCRUBBING EFFICIENCY
Starfield Resources Inc. – Ferguson Lake Project

Campaign	S in off-gas kg	S in scrubber liquor kg	Efficiency %
PP1	10.94	10.7	98
PP2	14.66	12.01	82

Scrubbing efficiency for PP1 was 98%, however, in PP2 it decreased to 82%. There is no apparent reason for the difference between campaigns, although there were some irregularities in PP2 operation. The hose in the peristaltic pump broke twice and the backup system was put on line, but that does not fully explain the difference between PP1 and PP2 scrubbing efficiencies. (This scrubbing system, though, is purely an artefact of the testwork, required because H_2S is highly toxic and cannot be emitted to the atmosphere. In the commercial operation, the H_2S will be converted to sulphuric acid in a conventional acid plant.)

Samples of the Ferguson Lake ore, flotation concentrate, flotation tails, and the primary leach residue were analyzed for PGE. As is shown in Table 13-42, all the PGE remained with the flotation concentrate after the flotation process, as well as with the primary leach residue after the leaching.

TABLE 13-42 PGE SURVEY
Starfield Resources Inc. – Ferguson Lake Project

Sample	Au*	Ag	Pt	Pd	Ru	Rh	Ir
Ferguson Lake ore	0.08	2.00	0.20	2.28	0.06	0.02	<0.02
Flotation Concentrate	0.20	2.10	0.20	3.17	0.04	0.04	<0.02
Flotation Tails	0.03	<0.50	0.05	0.20	0.04	0.04	<0.02
Primary Ore Leach Liquor	<0.01	1.63	<0.01	<0.01	-	-	-
Primary Ore Leach Residue	0.25	7.80	1.12	9.84	0.05	0.11	<0.02

*mg/L for liquor, g/t for solids

A sample of the flotation concentrate used as feed for the primary leach was subjected to liquid-solid separation testing. Also, a sample from the leach residue was collected during the pilot plant and subjected to liquid-solid separation testing. The testwork program is summarized in Table 13-43 indicating the measurements performed on each sample.

TABLE 13-43 LIQUID-SOLID SEPARATION TESTWORK PROGRAM SUMMARY
Starfield Resources Inc. – Ferguson Lake Project

Sample Identification	Testing Performed
B-3 concentrate	Flocculent Selection, Static Settling, Vacuum Filtration, Pressure Filtration, Rheology
Leach Residue	Flocculent Selection, Static Settling, Vacuum Filtration, Rheology

Particle size measurements were done on representative aliquots retrieved from the individual samples. The measured k_{80} for each sample is given in Table 13-44. The other measurements are detailed in the SGS reports on that work.

TABLE 13-44 SAMPLE CHARACTERIZATION
Starfield Resources Inc. – Ferguson Lake Project

Sample	k80 (microns)
B-3 concentrate	49.46
Leach Residue	78.38

INTERIM CONCLUSIONS FROM 2011 TESTWORK

1. Primary leaching was successfully run in a continuous, mini pilot plant configuration, producing the necessary liquor to move forward with oxidation and hydrolysis testing using liquor produced from real Ferguson Lake material.

2. Acid consumption was optimized in a series of bench tests and confirmed in the pilot plant run. Acid dosage around 600 kg of hydrochloric acid per tonne of solids was sufficient to obtain 73% iron extraction and 63% sulphide extractions while minimizing base metal extraction (6% nickel and 2% cobalt). Acid dosage in excess of 600 kg/t could lead to base metals leaching.

3. A PGE survey done on all the streams in this process showed that PGE remained with the primary leach residue.

4. Liquid-solid separation testing with samples of the flotation concentrate and the leach residue gave the information required for sizing the solid-liquid separation equipment in the process.

OXIDATION/HYDROLYSIS (FERGUSON LAKE FEED)

The primary leach solution (PLS) from the leach campaigns (processed from the bulk sulphide concentrate made from Ferguson Lake ore) was used in two further continuous mini-plant campaigns. The first of these tested the continuous oxidation step on the real feed solution and the second tested the oxidation and hydrolysis steps together, in continuous mode, on real feed. Figure 13-23 shows the experimental arrangement used for the first of these mini-plant campaigns – the arrangement is the same as was used in the work using synthetic feed.

The results achieved in this campaign are summarized in Figure 13-24. Full details of the results are presented in SGS's report on the campaign in Appendix 2.

At four hours into the run, the oxygen flow was increased from 217 to 346 mL/min. At 28 hours, the boiler and the oxidation reactor temperature set-points were increased slightly. Just before 50 hours, the feed flow was increased from 3 to 4 mL/min. At 62 hours, the oxidation bath temperature was raised from 117°C to 120°C. At 74 hours, the oxidation bath temperature was set back down to 117°C and the feed was dropped back to 3 mL/min.

At the end of the campaign, the amount of ferrous iron in the discharge from the oxidation reactor was approximately 15% of the total iron, i.e., the extent of oxidation achieved was 84%. No significant changes were observed between this campaign and the previous ones using synthetic feed.

In the next continuous mini-plant campaign, the oxidation and hydrolysis steps were run continuously, and simultaneously. Figure 13-25 is a diagram illustrating the configuration used in this campaign. Figure 13-26 shows the ratio of ferrous iron over total iron achieved in each of the three reactors. Figure 13-27 shows the amounts and strength of the hydrochloric acid produced in the hydrolysis step. Figure 13-28 shows the extent of hydrolysis achieved, as a function of the ferric chloride formed in the oxidation reactors to hematite and gaseous hydrochloric acid.

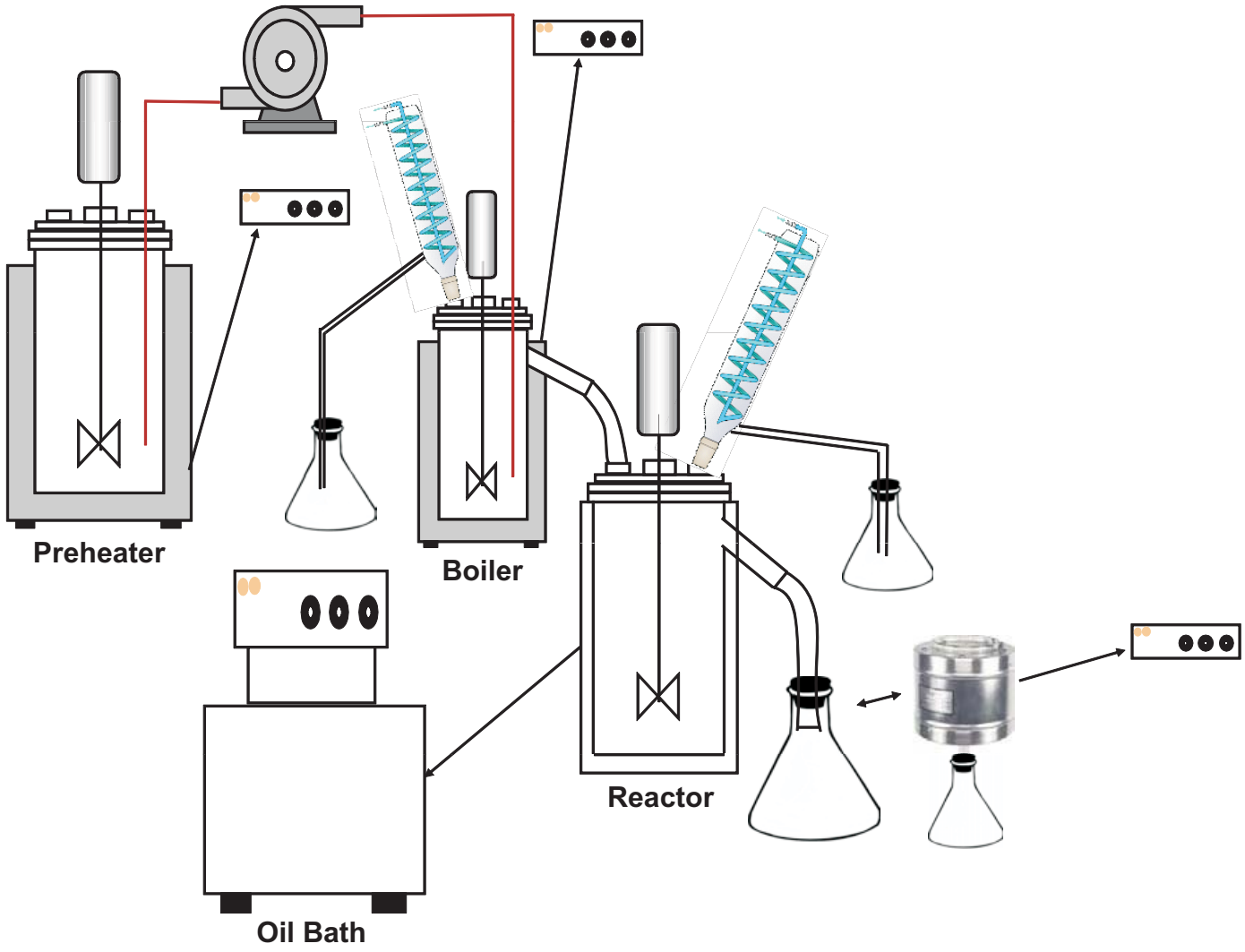


Figure 13-23

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Continuous Oxidation Arrangement

SGS Minerals Services, 2011.

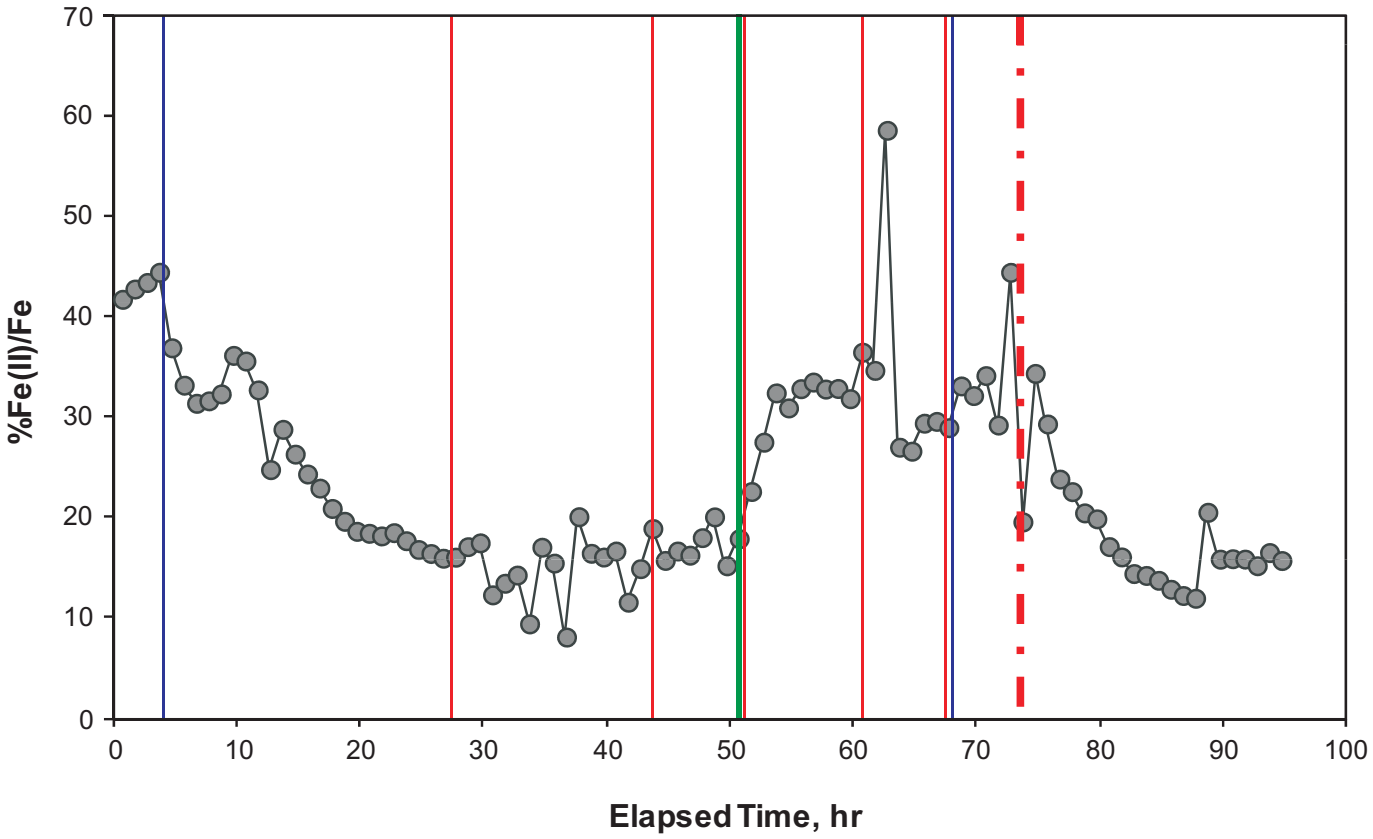


Figure 13-24

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Oxidation of Real Feed, (Ferrous Iron / Total Iron)

November 2011

SGS Minerals Services, 2011.

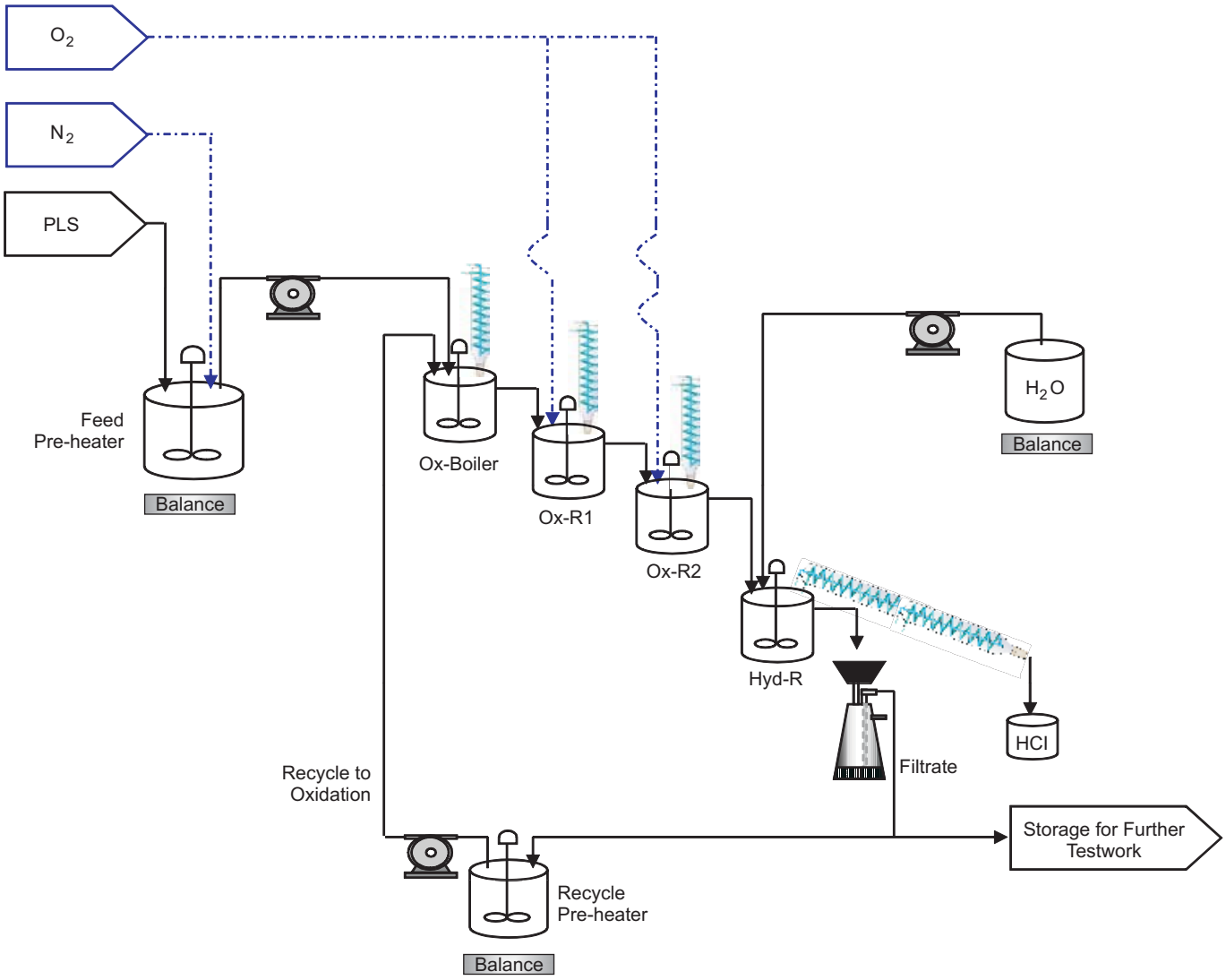


Figure 13-25

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Oxidation and Hydrolysis of Real Feed

FIGURE 13-26 CONTINUOUS OXIDATION AND HYDROLYSIS (FERROUS IRON/TOTAL IRON)

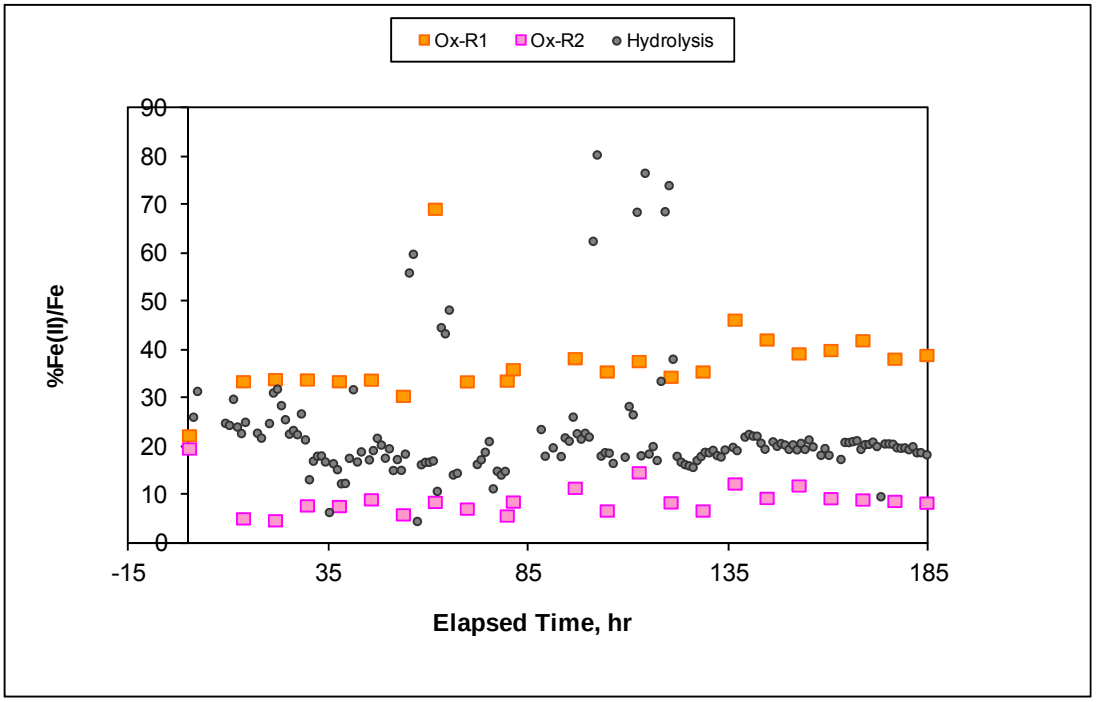


FIGURE 13-27 CONTINUOUS OXIDATION AND HYDROLYSIS (HCL PRODUCTION AND STRENGTH)

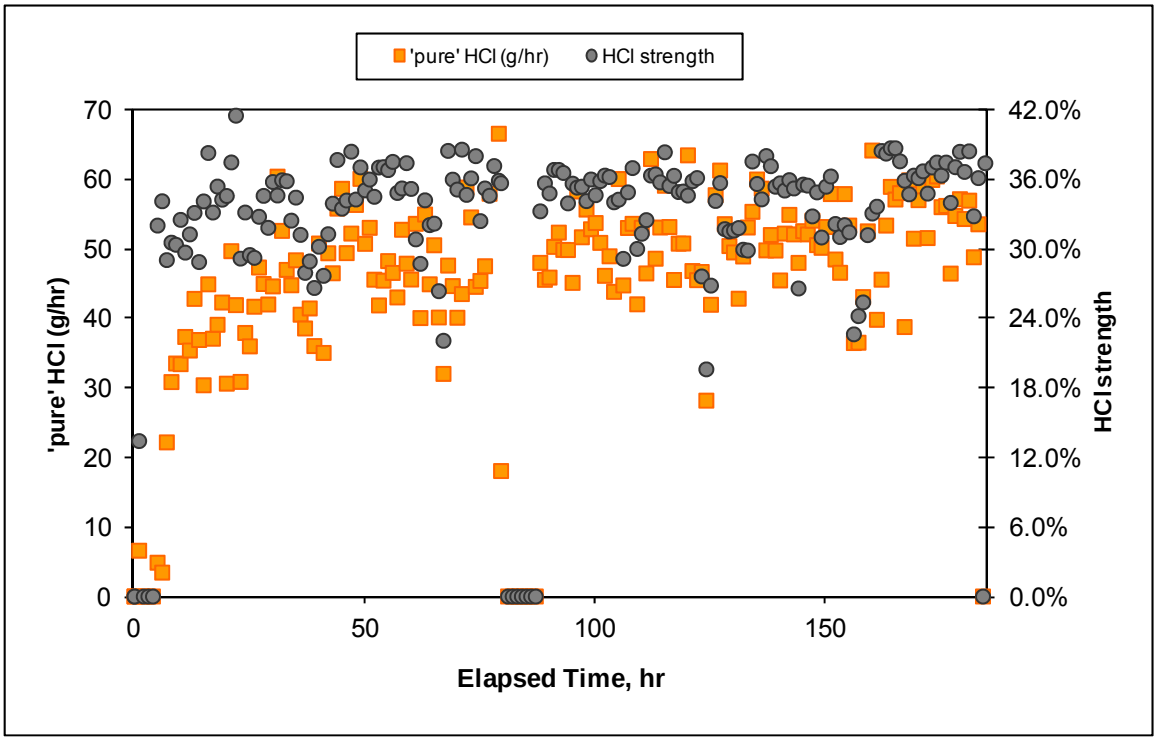
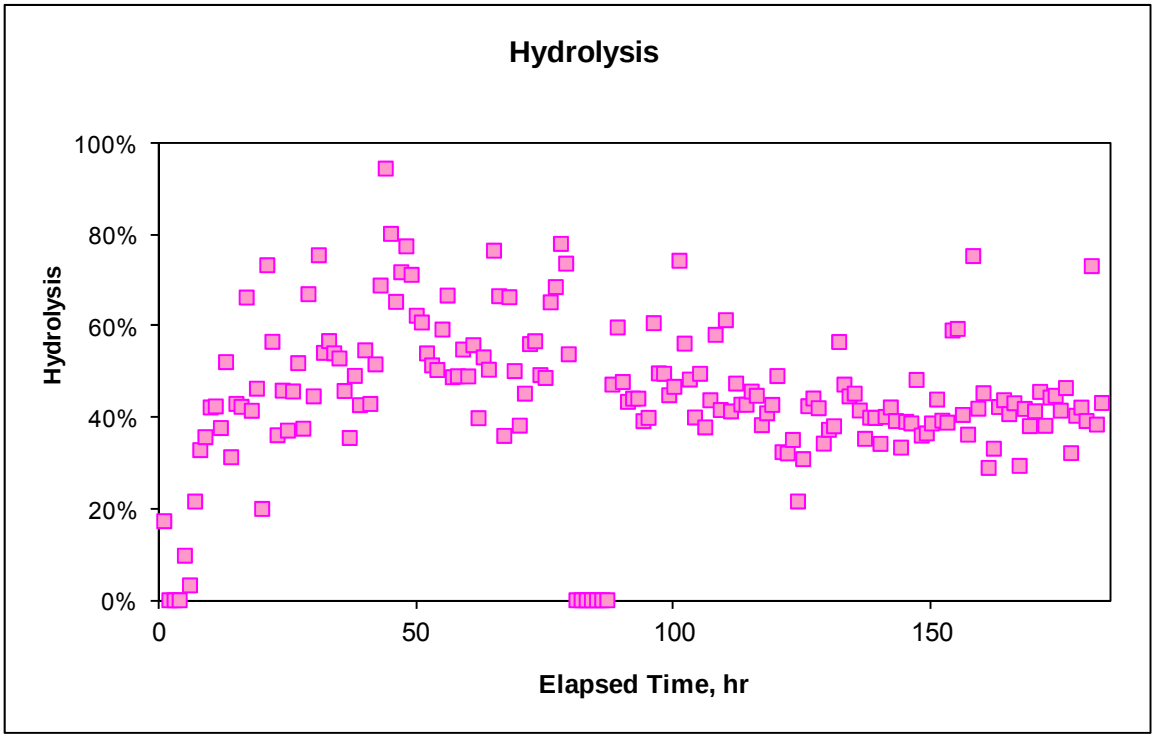


FIGURE 13-28 CONTINUOUS OXIDATION AND HYDROLYSIS (EXTENT OF HYDROLYSIS)



Full results of the above work are presented in SGS's report in Appendix 2. The key outcomes of this work are that the extent of oxidation of the ferrous iron was 90% and the extent of hydrolysis was approximately 40%.

The process model assumed an extent of ferrous iron oxidation of 75%, thus the aforementioned results exceeded the target level of oxidation. The process model assumed approximately 30 to 40% for the extent of hydrolysis because too high a percentage hydrolysis would cause the solids level of the slurry in the hydrolysis reactor to become too high and thus lose its flow characteristics.

The hot filtration step after hydrolysis separates the bulk of the unreacted ferric chloride from the hematite, thus permitting most of the unreacted ferric and ferrous chloride to be returned to the start of the oxidation-hydrolysis reactor train. In this way, the per-pass values of approximately 90% for oxidation and 40% for hydrolysis become almost complete overall conversions.

SECONDARY LEACH

The residue from the continuous mini-plant campaigns on the leached Ferguson Lake concentrate was used to test the secondary leach. Table 13-45 lists the results of batch leach tests in which the primary residue was leached with ferric chloride from the continuous oxidation/hydrolysis work. Full results are presented in the relevant SGS report in Appendix 2.

TABLE 13-45 RESULTS OF SECONDARY LEACH TESTS USING FERRIC CHLORIDE
Starfield Resources Inc. – Ferguson Lake Project

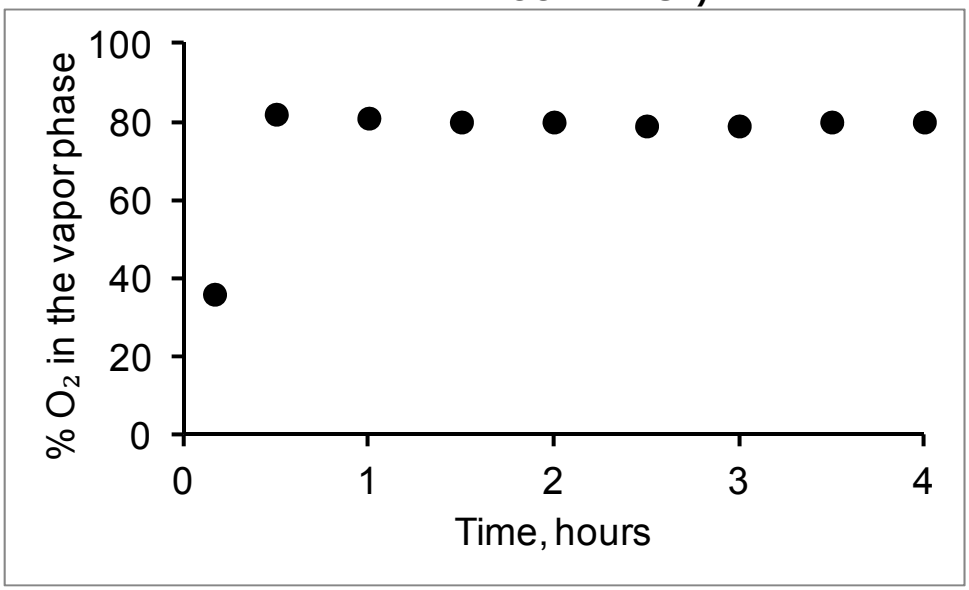
Condition	Test	1	2	3	4	5	6	7	8	9	10
Pulp density	%wt	10%	7%	20%	14%	14%	7%	14%	14%	13%	11%
Temperature	°C	92	93	98	92	102	100	106	102	101	100
Run time	hours	4	4	5	6	6	6	6	6	6	6
Fe(III)/feed ratio	g/kg	1138	1655	828	817	759	1748	764	634	780	780
Weight losses	%wt	39%	32%	46%	38%	45%	48%	42%	35%	41%	26%
Extraction, %	Fe	70.3	71.3	72.1	67.6	86.6	90.1	89.0	68.2	76.4	51.7
	Cu	56.7	44.3	80.9	75.4	93.0	92.0	97.3	87.9	97.2	86.0
	Co	61.5	54.2	83.5	87.1	90.3	85.2	89.5	87.4	91.2	84.8
	Ni	76.9	73.0	89.3	91.1	96.3	93.7	96.5	90.6	93.2	84.9
	Mg	5.6	**-23.0**	15.9	20.1	1.1	6.5	14.2	18.2		
Residue composition - %wt	Fe	10.9	11.1	15.8	15.8	7.3	5.8	6.1	16.8	13.4	21.5
	Cu	3.0	6.0	2.5	2.7	0.9	1.1	0.3	1.5	0.4	1.4
	Co	0.16	0.27	0.12	0.08	0.07	0.11	0.08	0.09	0.06	0.09
	Ni	0.9	1.5	0.7	0.5	0.2	0.4	0.2	0.6	0.5	0.8
	Mg	1.5	0.3	0.2	0.2	0.3	0.3	0.2	0.2		
	S	49.0	66.1	71.3	68.2	80.6	80.7	83.9	68.8	73.5	65.2

A sample of the primary residue was also subjected to a test in which it was leached at 120°C under an oxygen over-pressure of 5.6 atmospheres. Table 13-46 lists the results obtained and Figure 13-29 shows the measured percentage of oxygen in the vapour phase during the test. From the measured oxygen in the vapour phase, it appears that the dissolution reactions were complete after about three hours. The cobalt and nickel extractions were relatively high, at 89%. The copper extraction was lower and the iron extraction was substantially depressed, at 29%, due to re-precipitation of the dissolved iron as the dissolution of the base metals depleted the available chloride. Of the iron remaining in solution, 96% was in the ferric form.

TABLE 13-46 PERCENTAGE EXTRACTIONS, PRESSURE LEACHING THE PRIMARY RESIDUE
Starfield Resources Inc. – Ferguson Lake Project

Fe	Cu	Co	Ni	S
29	60	89	89	0

FIGURE 13-29 PRESSURE LEACH OF PRIMARY RESIDUE (OXYGEN IN VAPOUR PHASE)



The importance of the single pressure leach test that was completed is that it gave high extractions of nickel and cobalt while depressing the extraction of iron. In the overall process, the solution from the secondary leach passes through a solvent extraction step in which ferric chloride is removed prior to the purification of the remaining solution for separation and recovery of the copper, cobalt, and nickel.

In the step in which the ferric chloride is removed by solvent extraction, the ferric chloride is extracted into the organic phase and the loaded organic phase is stripped with water, generating a loaded strip solution of ferric chloride that is recycled. The more the ferric chloride is extracted from the secondary leach solution, the greater the amount of water needed to strip the loaded solvent, and hence, the greater the amount of water recycled, leading to a greater evaporation load on the overall process. Minimizing the amount of ferric iron in the secondary leach solution is important in the overall energy balance of the process.

SOLVENT EXTRACTON OF FERRIC CHLORIDE

The solution from the secondary leach tests in which ferric chloride was used to leach primary residue was used in laboratory bench testing to define the process conditions required for the solvent extraction of ferric chloride from the secondary leach solution. A number of organic extractants and diluents were evaluated, and pure TBP was found to be the most suitable. Figure 13-30 shows the isotherm generated for the extraction step and Figure 13-31 shows the corresponding stripping isotherm. This step in the process has yet to be tested in continuous mode, but solvent extraction is a commercially established operation and the results generated are confidently expected to translate into continuous operation.

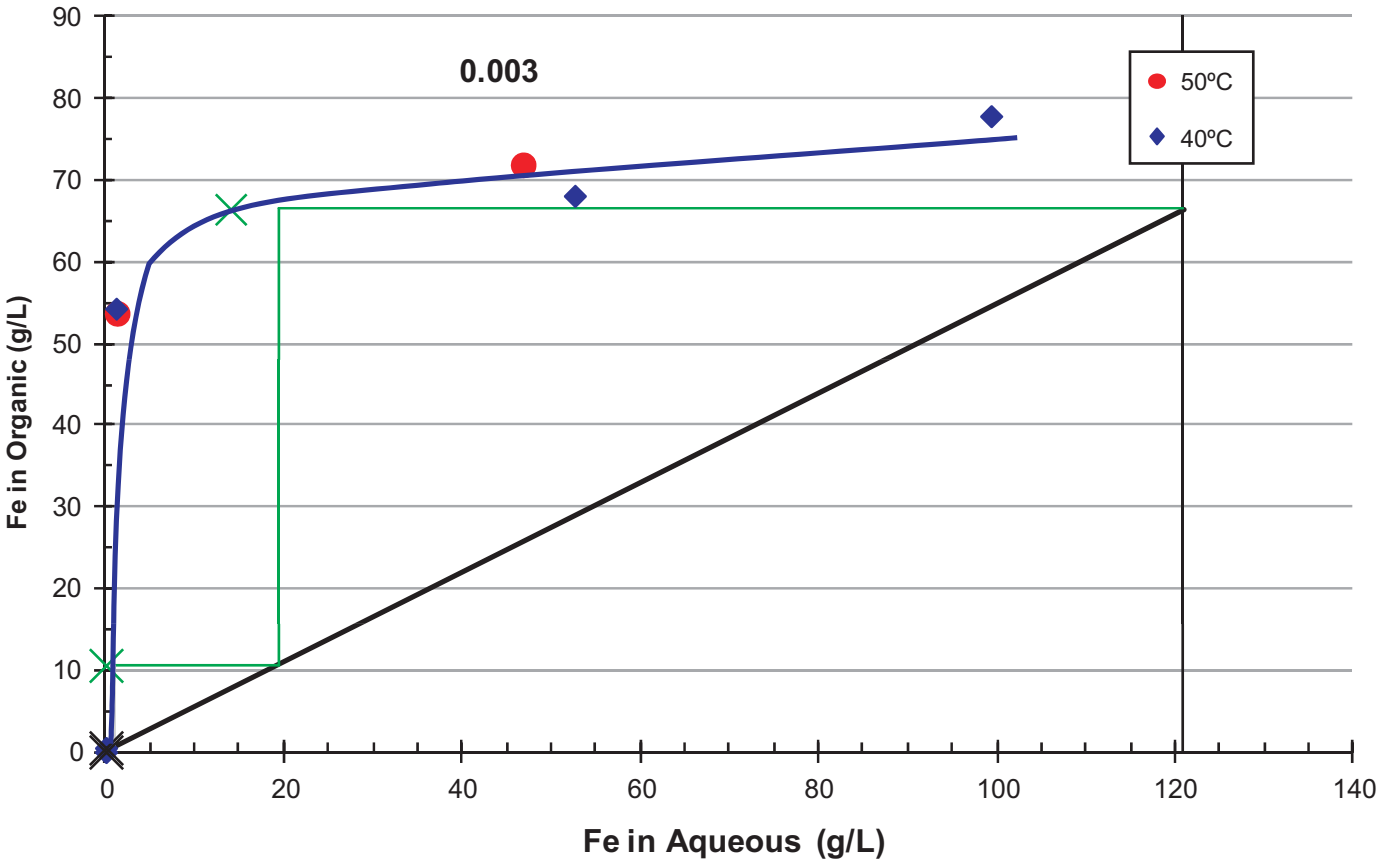


Figure 13-30

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Extraction Isotherm for $FeCl_3$ Solvent Extraction

November 2011

Source: SGS Minerals Services, 2011.

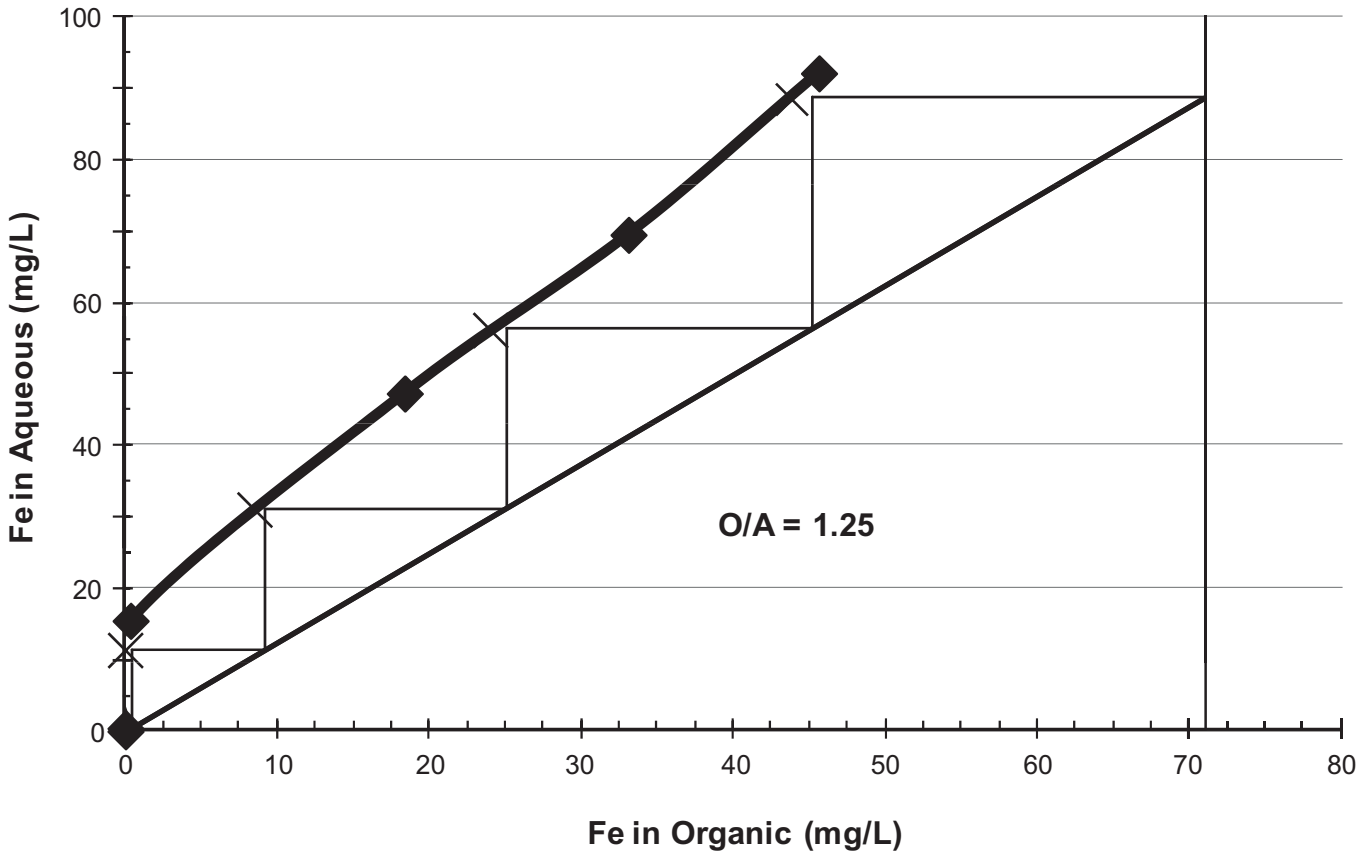


Figure 13-31

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Strip Isotherm for $FeCl_3$ Solvent Extraction

November 2011

Source: SGS Minerals Services, 2011.

CONCLUSIONS

GENERAL

The process, potentially, will not generate any acid rock drainage. It has the potential to generate stable, non-toxic tailings, and most importantly, may recover a sufficient quantity of the intrinsic energy in the ore to make it self-sustaining in energy usage. This means that there may be no need to import any fuel oil for generating power on-site, which would otherwise be necessary in such a remote location, where there is no local infrastructure. The energy recovered is potentially sufficient to operate both the hydrometallurgical process plant, as well as the mine, concentrator, and concentrate pipeline pump stations.

The process has evolved from the results of metallurgical testwork done over the past decade. As experimental data became available it was incorporated into a process model (numerically rigorous mass/energy balance) that was used to develop the operating and capital cost estimates used in this and the preceding evaluation of the process. The latest embodiment of the process model indicates that overall metal extractions of 91% for nickel, 96% for copper, and 90% for cobalt are achievable using the proposed process.

METALLURGICAL TESTWORK AND SCALE-UP

There has been sufficient laboratory work carried out to enable definition, at the conceptual level, a technically-viable preliminary flowsheet. The major unit operations (primary leach, secondary leach, oxidation and hydrolysis) have been operated continuously at SGS, thereby demonstrating the technical competence of these unit operations. For hydrolysis, a very similar circuit has been extensively piloted by SMS Siemag, leading to a decision to build a commercial plant.

The process oxidation and hydrolysis steps are novel, and are not commercially proven unit operations. Fundamental measurements of heat capacity, thermal conductivity, viscosity, and other properties are required in order to determine the required heat transfer coefficients to size heat transfer surfaces, or preferably, direct measurements of heat transfer at a larger scale than has been completed thus far.

The materials of construction will require further testwork, particularly for the heat transfer surfaces. The fluids under consideration are rather aggressive and materials

normally used for heat exchange surfaces, e.g. steel, will be dissolved. SGS has completed preliminary corrosion testing, and it appears that higher grades of titanium may be suitable for the oxidation stage and tantalum may be required for the hydrolysis stage. It is expected that verification in this area will require substantial work.

The oxidation step appears to be controlled by the rate at which oxygen is transferred from the gas phase to the bulk liquid phase. This implies that agitation is extremely important, and further work is required to properly scale-up the oxidation step.

ENERGY BALANCE

As noted throughout this report, there are a number of uncertainties, where it has been necessary to make reasonable assumptions. The most important of these are the iron and sulphur contents of the ore, and its accompanying mineralogy, and secondly, the lack of fundamental data in the scientific literature to enable, in particular, a rigorous energy balance to be carried out.

Both of these are related to the energy balance. In the former, the amount of sulphur in the ore, particularly as the mineral pyrrhotite, determines the amount of hydrogen sulphide gas liberated, which in turn determines the amount of energy liberated and the amount of sulphuric acid produced. The amount of iron in the feed determines how much hydrochloric acid is needed, which in turn determines how much hydrolysis and acid recycle is needed.

PLATINUM AND PALLADIUM

No formal account has been taken of the precious metals content of the ore, since no process for their recovery has yet been properly evaluated, however, it is clear their recovery is mandated, even as a relatively low-grade concentrate, since they will comprise a significant revenue stream.

There is also potential to process low-base metal, high-PGE material, not currently included in Mineral Resources. Low-base metal-content material will not provide the energy advantages of the massive sulphide material, however, there may be value in considering a process unit that can recover PGEs from both low base metal material and from the secondary leach residue.

PROCESS RISK ASSESSMENT

NeoFerric previously identified and evaluated risks for each step of the process. This assessment has been updated to reflect new work since the previous risk exercise, and has been reviewed by Arithmetek and RPA. The level of risk presented is purely subjective, but is based on both testwork observations, combined with operational experience with similar commercial and pilot plant operations.

The assessment shows that the majority of the process has low associated risks. Medium-risk items include the oxidation and hydrolysis components of the proposed process as these areas have not yet been scaled up to commercial plant size. The iron hydrolysis and HCl recovery circuit is a technique originally developed by PORI Inc. in the 1970s, and piloted by SMS Siemag for four years since 2002.

TABLE 13-47 PROCESS RISK ASSESSMENT
Starfield Resources Inc. – Ferguson Lake Project

Unit Operation	Comments	Risk
Primary leach	1. Significant laboratory testing on various cores. 2. Continuous testing at SGS, reproduced laboratory data. 3. Leaching of ores and concentrates is common in the industry. 4. Chloride processes – whilst not as common as those in sulphate, materials are now well-tested	Low
Primary leach slurry S/L separation	1. Final solids settle and filter very well, both in laboratory and on a continuous basis. 2. S:L separation testing has been done.	Low
Primary leach off-gas treatment	1. H_2S is a toxic, noxious gas. 2. Commonly used in the extractive metallurgical industry to precipitate metal sulphides, therefore handling and OHSA guidelines are well known. 3. Claus reactors for treating H_2S gas, recovering energy and generating either elemental S or SO_2 are very common in the oil & gas industry. 4. Wet sulphuric acid plants using H_2S as feed are commercially operated.	Low

Unit Operation	Comments	Risk
Ferrous iron oxidation and primary hematite precipitation	1. Ferrous iron oxidation with oxygen or air is very well known in both chloride and sulphate media. 2. Reaction is mass transfer controlled. 3. Abundant knowledge on how to effectively introduce a non-soluble gas into a liquid to effect a reaction. 4. Similar reaction piloted by PORI Inc. in the early 1970s. 5. Continuous testing. At SGS has given high conversion of ferrous to ferric chloride. 6. Scale-up has yet to be addressed.	Medium
Primary hematite S/L separation	1. Hematite solids are crystalline, settling and filtering very well. 2 S:L separation testing required. 3. Similar materials of construction to those used for the Magnola plant will be used. These performed well.	Low
Secondary leach	1. Ferric chloride/oxygen leaching of sulphide concentrates has been piloted at many locations in the past. None has achieved more than demonstration stage, but this was not because of the leaching process itself, rather in downstream operations, notably iron control and case-specific economics. 2. Reaction proceeds very well in the laboratory. 3. Continuous testing of ferric chloride leaching at SGS confirmed laboratory data. 4. Materials of construction for this duty are well known.	Low
Secondary leach slurry S/L separation	1. Solids settle and filter very well in the laboratory. 2. S:L separation testing required. 3. Similar materials of construction to those used for the Magnola plant will be used. These performed well.	Low
Iron hydrolysis, HCl recovery and primary hematite precipitation	1. This is a technique, originally developed by PORI Inc. in the 1970s, and piloted by SMS Siemag for four years since 2002, designed to have low energy requirements, control iron in the circuit and regenerate super-azeotropic HCl for recycle to the leaching process. 2. Water is used as a reagent, not as a solvent, which is unusual in hydrometallurgical processes. 3. No other reagents are required. 4. Process is a hybrid of molten salt and concentrated solution chemistry. 5. Considerable laboratory-batch and mini-plant continuous development work has led to confidence that the technique can achieve its goals. 6. Materials of construction and scale-up need to	Medium

Unit Operation	Comments	Risk
	be addressed. Preliminary coupon corrosion testing indicates that suitable materials do exist. 7. The off-gas, being strong HCl is similar to that experienced in hydrochloric acid plants and pyrohydrolysis reactors. These have functioned well for years, and similar materials will be used.	
Secondary hematite S/L separation	1. Hematite solids are crystalline, settling and filtering very well. 2. S:L separation testing required. In continuous mini-plant testing, hot vacuum filtration has worked well. 3. Similar materials of construction to those used for the Magnola plant could also be used. These performed well.	Low
Primary ion exchange separation of Zn	1. Resins have been identified and tested in the laboratory. Resins have been included in the Vale Goro plant, by Polymet and Baja Mining for similar applications in their processes. 2. Additional laboratory loading and stripping profiles are necessary with real leach solutions.	Medium
SX of Cu and Co	1. Reagent has been used for many years at Kristiansand in Norway.	Low
SX/EW of Cu	1. Conventional technology used in the copper industry.	Low
SX/EW of Co	1. Conventional technology used in the cobalt industry.	Low
SX/EW of Ni	1. SX is planned using conventional reagents. Testing is not complete. 2. Process has been used commercially in Australia. The issue of Ca is currently being investigated.	Low
Removal of Mn	1. Techniques are known and have been tested on other projects. 2. Verification for this project is required.	Low

RECOMMENDATIONS

Testwork has generated a flowsheet on which to assess the economic viability of the Project, at a scoping level. The leaching, oxidation and hydrolysis unit operations have been successfully carried out in a continuous mini-pilot plant operation at SGS Lakefield. It is clear, however, that in carrying out this study, there are a number of areas which will require much better definition in order to advance the process design:

- Determine input Fe and S contents of the ore with a greater degree of certainty, including expected variations within the process feed.

- Generate fundamental data (water and hydrochloric acid vapour pressure, water and chloride activity, boiling point elevation, solubility data) for more accurate process design.

- Carry out primary and secondary leach tests at a higher temperature and/or salt concentration consistent with the output of the mass/energy balance to ensure that it is possible to do so.

- Audit the mass/energy balance to ensure most efficient use of combustion gas energy.

- Test methods for concentrating the platinum and palladium content of the secondary leach residue. Include possibility of concurrent processing of low-sulphide, high-PGE feed.

- More detailed SX/IX testing on secondary leach liquor.

- Run a pilot plant program, integrating all process units into a continuous operation.

14 MINERAL RESOURCE ESTIMATE

SUMMARY

Mineral Resources, as summarized in Table 14-1, have been estimated by Jamie Lavigne, P. Geo., Associate Consulting Geologist, RPA.

TABLE 14-1 MINERAL RESOURCES - NOVEMBER 30, 2011

Starfield Resources Inc – Ferguson Lake

Zone	Tonnes (Mt)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Fe %	S %
INDICATED RESOURCES								
Main West Pit	1.1	0.63	0.97	0.07	0.22	1.54	36.34	19.95
Main West UG	14.7	0.65	0.99	0.08	0.25	1.55	38.16	20.97
Total Indicated	**15.8**	**0.65**	**0.99**	**0.07**	**0.25**	**1.55**	**38.04**	**20.90**
INFERRED RESOURCES								
Main West Pit	0.2	0.57	0.90	0.07	0.17	1.40	33.96	18.40
Main West UG	5.9	0.59	0.82	0.07	0.20	1.34	36.15	19.59
West Zone Ext.	14.7	0.71	1.23	0.08	0.31	1.88	41.63	23.14
East Zone	9.4	0.65	0.76	NE	NE	NE	38.41	21.16
Total Inferred	**30.2**	**0.67**	**1.00**	**0.05**	**0.19**	**1.18**	**39.49**	**21.79**

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources were estimated at NSR cut-off values of $53 and $97 per tonne for open pit and underground, respectively.
3. Mineral Resources were estimated using average long-term metal prices of US$9.00, US$3.15, and US$17.00 per pound of nickel, copper, and cobalt, respectively, a US$/C$ exchange rate of 1.00, and a royalty of 2% NSR.
4. Metallurgical recovery assumptions were 91% for nickel, 96% for copper, and 90% for cobalt.
5. Platinum, palladium and cobalt not estimated for the East Zone as the East Zone contains a higher proportion of historical INCO holes for which cobalt or individual platinum or palladium grades have not been determined.

Mineral Resources that included all diamond drilling up to 2006 were estimated by Lavigne and were reported by Nicholson (2007). Scott Wilson RPA previously reviewed this Mineral Resource estimate and concluded that the estimate was in compliance with

the requirements of NI 43-101 and the definitions set out by the CIM Standards on Mineral Resources and Reserves Definitions and Guidelines adopted by the CIM Council in December 2005 (Clow et al., 2008).

Starfield completed additional drilling in 2007 and 2008. A primary focus for the 2007 and 2008 drilling was the exploration for low-sulphide PGE mineralization within the near surface part of the Main West Zone. The Mineral Resource estimate presented in Clow et al. (2008) was updated for this part of the Ferguson Lake deposit using interpretation and modelling parameters revised to incorporate the low-sulphide, PGE style of mineralization. The Mineral Resource update was completed in July 2009 by Geoffrey T. Chinn, P.Geo., Scott Wilson RPA Associate Geologist.

New exploration data collected by Starfield since the 2008 diamond drilling has not been incorporated into the current Mineral Resource estimate. Exploration and drilling work completed since 2008 is discussed in Sections 9 and 10. For the purposes of the present PEA, the following work has been completed:

1. The model reported by Clow et al. (2008), and the Mineral Resource update subsequently completed by RPA, have been compiled into a single block model.
2. Iron and sulphur values have been added to the compiled block model based on nickel/iron and nickel/sulphur regression analyses.
3. Block NSR values have been updated based on current metal prices and updated Project technical parameters.
4. The block model has been assessed, and resources reported, based on updated cut-off NSR values for open pit and underground resources.

This work has been completed by Jamie Lavigne, P.Geo, Associate Consulting Geologist, RPA and Qualified Person as defined in NI 43-101, is responsible for the preparation and reporting of the current Mineral Resource estimate.

As per NI 43-101, Mineral Reserve estimates can only be based on the results of a preliminary feasibility study or feasibility study of a mineral project. Thus, no reserves have been estimated for the Project. Also, Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability.

DRILL HOLE DATABASE

Mineral Resources have been estimated for the Project exclusively from diamond drill hole data. No systematic excavations or surface sampling have been completed that contribute to the resource estimation database. A total of 608 diamond drill holes are in the Starfield Ferguson Lake database which has been used to estimate Mineral Resources. The total number of holes and metres drilled, by drill campaign, are contained in Table 14-2

TABLE 14-2 DRILLING SUMMARY BY YEAR
Starfield Resources Inc. – Ferguson Lake Project

Year	Company	No. of Holes	Metres
1953	Inco	173	26,385
1999	Starfield	19	3,923
2000	Starfield	48	15,533
2001	Starfield	38	21,500
2002	Starfield	53	24,957
2003	Starfield	9	2,721
2004	Starfield	55	23,018
2005	Starfield	29	16,973
2006	Starfield	116	24,951
2007	Starfield	19	6,139
2008	Starfield	49	18,813
Total		**608**	**184,913**

SAMPLE AND ASSAY DATA

The assay database includes a total of 36,557 samples. All of the drill core samples collected by Starfield up to and including the 2011 drill campaign have been analyzed for Ni, Cu, Co, Pt, and Pd. In the 2007 and 2008 drill campaigns, samples from drill holes that targeted the low sulphide PGE style of mineralization, totalling 1,980 samples, only had Pt and Pd analyses. Starfield has collected whole rock and trace element geochemical data on selected samples and this includes Fe and S analyses from selected samples in 2006, 2007, and 2008. The Inco samples were initially only analyzed for Ni and Cu. Subsequent to the Inco field programs at Ferguson Lake, Inco re-analyzed selected core samples for Pt, Pd, and Au, however, Inco has only reported this data as a total (i.e., Pt + Pd + Au equals total PGE). The total number of samples and metres of core analyzed by year is contained in Table 14-3.

TABLE 14-3 SAMPLE DATA BY DRILL CAMPAIGN
Starfield Resources Inc. – Ferguson Lake Project

Year	Company	No. of Samples	Total Metres
1953	Inco	2,350	3,277
1999	Starfield	912	680
2000	Starfield	2,351	2,172
2001	Starfield	1,868	1,792
2002	Starfield	3,396	3,602
2003	Starfield	1,107	1,415
2004	Starfield	4,746	6,075
2005	Starfield	2,099	2,504
2006	Starfield	5,927	5,897
2007	Starfield	2,713	3,764
2008	Starfield	9,088	12,681
Total		**36,557**	**43,858**

During the course of logging, the percent abundance of sulphide minerals was systematically recorded for each sample. This included the minerals pyrite, pyrrhotite and chalcopyrite. This data is included in the Starfield sample database.

LITHOLOGY

Logging by both Inco and Starfield included the identification and documentation of rock types. This includes the major distinctions between the intrusion, the country rocks to the intrusion, and massive sulphides. The country rocks are broken out into several types of amphibolites, gneisses and granitic rocks. The intrusion is mainly described as gabbro with lesser hornblendite. Hornblendite is usually spatially associated with massive sulphide. Massive sulphide was broken out as a rock type in the lithological description of the core. Mafic to felsic dykes have been logged as cross cutting the country rocks and the intrusion. RPA notes that differences in rock type coding between closely spaced holes, particularly those from different drill campaigns indicate inconsistent identification of the major rock types.

SPECIFIC GRAVITY

The Ferguson Lake drill hole database contains 1,248 specific gravity (SG) measurements taken over 1,229 m of core, all from the Starfield 2006 drill campaign.

The SG was determined using the water immersion method and was completed at ACME laboratories.

During 2002, the specific gravity (SG) was determined on 100 core samples. The samples were selected grab samples of core collected to be representative of the major rock types including massive and semi-massive sulphide.

DATABASE

The drill hole and assay data for the 2006 Resource Estimate was received from Nicholson and Associates as a series of spread sheet files. For the resource estimate update, drill hole data was received from Trifon Barakov, a former Starfield geologist, as a series of spreadsheets. In both cases, the data was imported into a resource estimation software package for the purposes of geological and resource modeling. Imported data imported drill hole collar locations, down hole survey data, rock type data and sample data including sample identification and analytical data for Ni, Cu, Co, Pt, Pd, Fe, and S.

ESTIMATION METHODOLOGY

DESCRIPTION OF DOMAINS

Massive sulphide intersections on the Project cover a strike length of approximately 15 km. For the purposes of resource estimation, the property has been divided into three zones: the East Zone, the Main West Zone, and the West Extension Zone (Figure 14-1). Drill spacing, and therefore the amount of geological and assay information, is variable across the zones and accordingly, different approaches to Mineral Resource estimation have been utilized. A general approach of geological modelling and geostatistical block model estimation has been used for the Main West Zone whereas a polygonal approach has been employed for the East and West Extension Zones.

MAIN WEST ZONE

The central zone, the Main West Zone, has been the target of the most detailed drilling, geostatistical study, and is the area from which past Measured and Indicated Resources have been reported. Drill hole spacing in the Main West Zone is variable from 25 m (locally 13 m spacing) in selected test areas of the zone (i.e. in the “Pit Area”) to a

maximum of approximately 100 m. Part of the Main West Zone was the object of a past geostatistical resource study and estimation (Clark, 2006).

The Main West Zone has been subdivided into three domains each comprised of a series of wireframes:

1) The 1000 Domain
2) The 2001 Domain
3) The 3001 Domain

The distribution of the domains comprising the Main West Zone is illustrated in figure 14-2. The 1000 domain series wireframes constitute four individual horizons referred to as the 1001, 1002, 10031, and 10032 sub-domains. These sub-domains are located near the surface at the east end of the Main West Zone. The 1000 domain is the part of the Main West Zone that was the object of a past geostatistical resource study and estimation (Clark, 2006) and was the focus of drilling in 2007 and 2008. The resource update completed by RPA was restricted to the 1000 domain within the Main West Zone.

The drill density and historical work in the Main West Zone supports resource estimation using a geostatistical block model methodology based on a geological interpretation of the intrusion and mineralization. The general procedure, or procedural sequence, utilized to estimate the Main West Zone is summarized in Table 14-4.

TABLE 14-4 ESTIMATION METHODOLOGY – MAIN WEST ZONE

Starfield Resources Inc – Ferguson Lake Project

Procedure	Description
Geological Modelling	Sectional interpretation of geology and mineralization guided by the upper intrusive contact; bulking out intercepts to a minimum mining width; correlation of mineralized domain between sections, creation of geological wireframes for mineralized domains, interpretation and wireframe creation of intrusion; creation of surface and overburden wireframe models
Statistics and Geostatistics	Statistical analyses of assay data; evaluation of assay populations within and related to geological and mineralization solids, calculation and evaluation of composites for grade interpolation; variography
Block Modelling	Creation of block model; grade estimation of blocks (interpolation) using Ordinary Kriging; estimation of block density; estimation of percent of block within (and outside of) geological or mineralogical domains (wireframes); various block manipulations to combine high grade and low grade models and to calculate block NSR values
Economics and Classification	Application of a dollar value cutoff based on economic, technical and operating parameters to establish resource; evaluation of block model based on various criteria to establish resource classification

In general, the geological modelling of mineralization is based on the interpretation of massive sulphide where it is defined as being logged sulphide abundances greater than 50%. For the update in the 1000 domain to include the 2007 and 2008 drilling, the interpretation was expanded to include low-sulphide and interpretation of high-sulphide and low-sulphide intercepts was based on $80/tonne NSR and 3.0 g/t PdEq respectively. In both cases, narrow intersections were bulked out to three metre minimum mining widths.

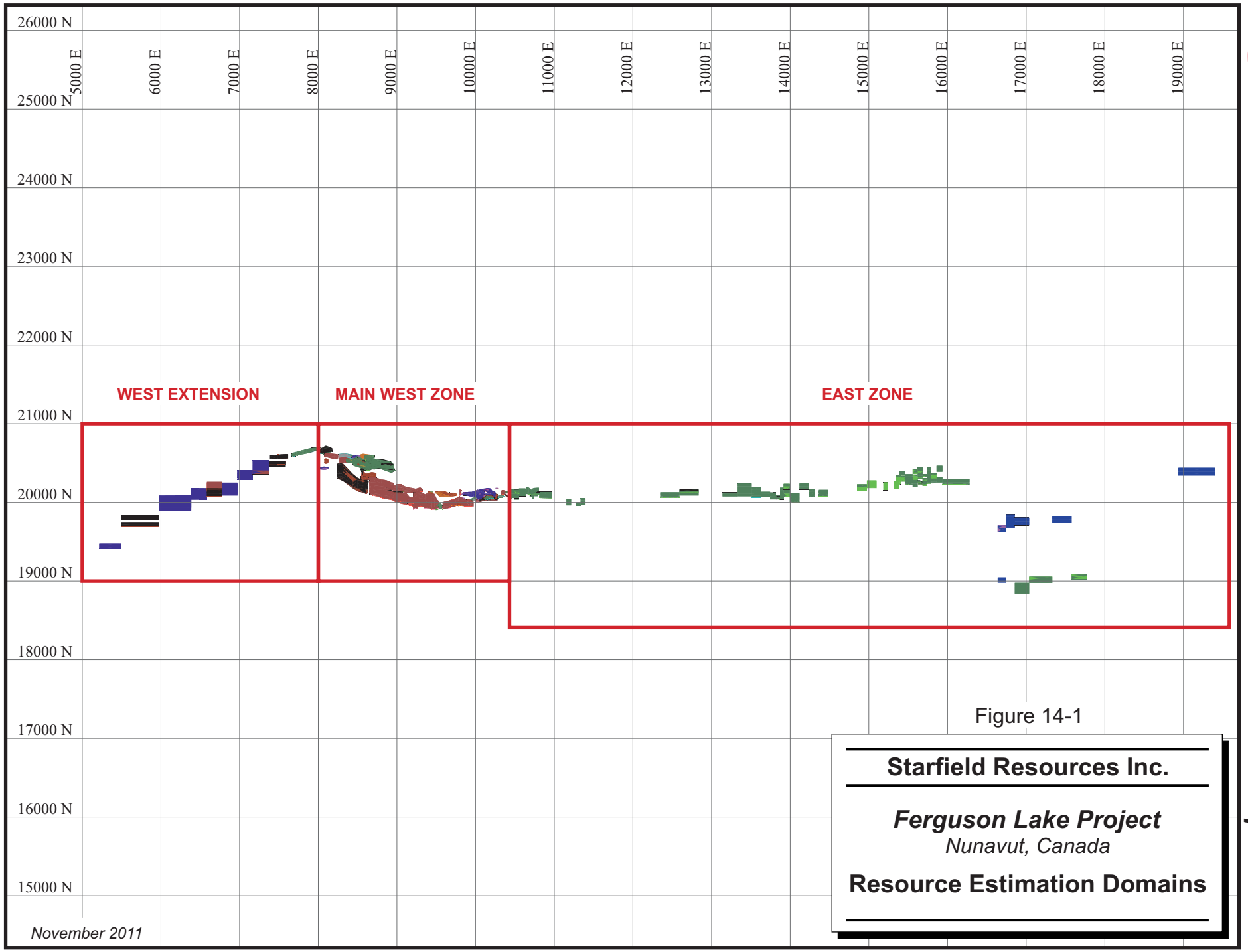


Figure 14-1

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Resource Estimation Domains

November 2011

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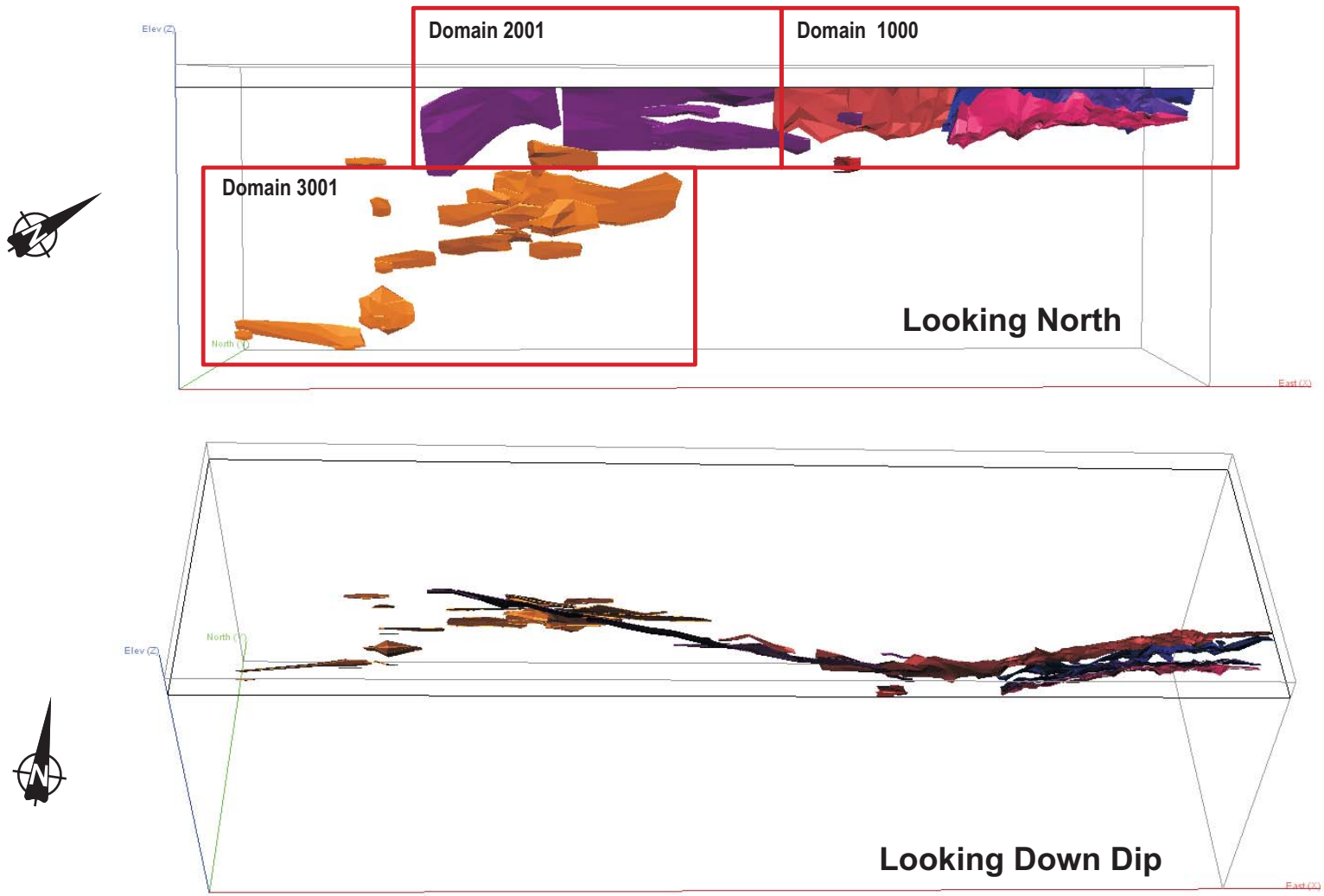


Figure 14-2

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Main West Zone
Isometric View

www.rpacan.com

WEST EXTENSION AND EAST ZONES

Drilling in the West Extension has intersected mineralization along strike to the west of the Main West Zone, approximately 600 m to 1,200 m below surface. Drilling has been completed on 200-metre-spaced sections with holes on section spaced irregularly but averaging approximately 100 m apart. All of the drilling in the West Extension has been completed by Starfield. Similarly, the East Zone is characterized by comparatively widely spaced drilling, however, it does include clusters of more detailed drilling in specific target areas. The East Zone contains a higher proportion of Inco drill holes. Given the drill hole spacing, detailed geological models were not created for the West Extension or East Zones, and a polygonal approach to estimation was utilized. The general procedures used are summarized in Table 14-5.

TABLE 14-5 ESTIMATION METHODOLOGY – WEST EXTENSION AND EAST ZONES

Starfield Resources Inc – Ferguson Lake Project

Procedure	Description
Definition of Intersections	Define massive sulphide intersections; application of dollar value cutoff and minimum width parameters to massive sulphide intersections to define economic intersections
Calculate Composites	Calculation of grades for complete intersections
Sectional Interpretation	Interpretation of mineralization polygons on drill section
Solid (volume) Creation	Extrusion of polygons to section limits
Block Modeling	Creation of block model for each of the West Extension and East Zone, estimation of grade into blocks utilizing Nearest Neighbor interpolation method; estimation of density and tonnage calculation based on percentage of block within the extruded polygon

MAIN WEST ZONE

MAIN WEST ZONE GEOLOGICAL MODEL

Diamond drill holes in the Main West Zone are commonly characterized by multiple intersections of massive sulphide, which can be of a highly variable width. Massive sulphide within the Main West Zone has been interpreted as a series of stacked lenses

or sheets with a moderate north dip and approximate east-west strike (Carter, 2006). This interpretation is consistent with and supported by the following observations:

- Dominant and strong magnetic linear with slightly variable but dominant east-west strike and north dipping asymmetry.
- Gabbro-sulphide contact locally mapped in outcrop with an east-west strike and coincident with a magnetic high (magnetic susceptibility data from drill core does not indicate any other significant magnetic expression other than pyrrhotite).
- Model “plates” (three dimensional geophysical interpretations), based on ground electromagnetic surveys, strike east-west and dip moderately to the north.
- Selected holes drilled from north to south (as opposed to all drilling at Ferguson Lake which has been at azimuth 180° grid, approximately 175° true) have intersected massive sulphide down dip supporting the geometry of the lenses with an east-west strike, north dip and limited thickness.
- This orientation of massive sulphide sheets with average east-west strike and north dips with much shorter downhole ranges is consistent with the variography by Clark (2006).

Massive sulphide intersections interpreted on section are based on the occurrence of greater than 50% sulphide as estimated on a sample-by-sample basis and incorporate inclusions of weakly mineralized to barren host rock. A minimum width of two metres was used for the massive sulphide intersections.

The update of the 1000 Domain, to include the 2007 and 2008 drilling includes low sulphide intersections interpreted on section which contain less than 50% sulphides and may contain almost no sulphides. Low-sulphide mineralization may contain high concentrations of Pd and Pt mineralization and in places this mineralization is interpreted to represent the lateral continuation of sulphide horizons. The update interpretation included the application of economic criteria and a three metre minimum vertical mining width which was applied to mineralized intersections which were diluted to this width where necessary.

Based on position and orientation within the Main West Zone, the lenses and sheets comprise three main domains: 1000, 2001, and 3001 (Figure 14-2). The 1000 Domain consists of four wireframes namely the 1001, 1002, 10031, and 10032 located in the near surface position at the east end of the Main West Zone. This is the area that has

historically been referred to as the “Pit Area”. The lenses comprising the Domain 1000 series wireframes strike east-west and dip moderately to the north. The 1001, 1002, 10031, and 10032 sub-domains have been designated from positions relative to the upper gabbro contact; such that the 1001 horizon occurs adjacent to the upper gabbro contact, the 1002 horizon occurs below the 1001 horizon, the 10031 horizon occurs below the 1002 horizon, and the 10032 sub-domain is located in the footwall position. The 2001 Domain consists dominantly of a comparatively thinner sheet with a consistent 103° strike. The 3001 Domain consists of a number of lenses of variable thickness with an east-west strike, similar to the 1000 Domain wireframes.

MAIN WEST ZONE ASSAY AND COMPOSITE DATA

A total of 3,546 samples occur within the Main West Zone massive sulphide wireframes. The samples have an average length of 0.95 m and a median length of 1.00 m. Samples outside of the massive sulphide, occurring within the gabbro zone, have an average length of 1.15 m and a median length of 1.06 m. The assay samples were composited to one metre length and the assays and composites are presented and compared in Table 14-6 for all domains in the Main West Zone. Statistics for the Main West Zone, Domain 1000 (based on the revised interpretation to include the 2007 and 2008 drilling and expanded to include the low-sulphide PGE mineralization) are contained in Table 14-7. All residual composites less than 0.3 m have been removed from the data base for the purposes of interpolation and are not included in the tabulation below.

Prior to compositing, assays were evaluated for grade capping. Capping levels were guided by a number of criteria including histograms, percent of total contained metal capped, and percent of assays capped. For most metals, minimal capping of only the most obvious outliers was required. Capping for platinum was more involved owing to its skewed distribution. Grade capping was applied only to the 1000 Domain and not to the 2001 or 3001 Domains. Capping levels and metrics applied to the 1000 Domain are shown in Table 14-8.

Based on the statistics from Clow et al. (2008) for the Main West Zone (Table 14-6) the 1000 and 3001 domains have similar Ni, Cu, and Pd values (slightly higher Cu in the 3001 domain) and the values in the 1000 and 3001 domains are distinct from the

average values in the 2001 domain. A consistent increase in Ni grade from the 1001 sub-domains to the 1003 sub-domain (i.e., from hangingwall to footwall) in the 1000 domain and a decrease in the Cu-Ni ratio is also apparent.

TABLE 14-6 MAIN WEST ZONE ASSAY AND COMPOSITE STATISTICS

Starfield Resources Inc – Ferguson Lake Project

Lens	Metal	Assay Statistics					Composite Statistics				
		n	Min	Max	Mean	Median	n	Min	Max	Mean	Median
1000*	Cu	2091	0.00	13.30	1.00	0.89	1955	0.00	5.38	0.97	0.91
1000*	Ni	2091	0.00	1.77	0.73	0.86	1955	0.00	1.45	0.75	0.83
1000*	Co	1827	0.00	0.80	0.09	0.09	1692	0.00	0.66	0.09	0.09
1000*	Pt	1827	0.00	5.49	0.27	0.18	1692	0.00	4.84	0.27	0.20
1000*	Pd	1827	0.00	6.06	1.71	1.87	1692	0.00	6.06	1.74	1.85
2001	Cu	431	0.00	6.30	0.78	0.65	372	0.11	6.26	0.78	0.69
2001	Ni	431	0.00	1.60	0.54	0.55	372	0.02	1.60	0.57	0.56
2001	Co	297	0.00	0.80	0.07	0.06	239	0.00	0.44	0.07	0.06
2001	Pt	297	0.00	1.35	0.20	0.16	239	0.00	1.23	0.21	0.17
2001	Pd	297	0.00	4.69	1.20	1.25	239	0.06	4.69	1.28	1.26
3001	Cu	1024	0.00	9.08	1.14	0.97	931	0.00	5.80	1.14	1.02
3001	Ni	1024	0.00	1.17	0.65	0.79	931	0.00	1.13	0.67	0.75
3001	Co	1024	0.00	0.41	0.08	0.09	931	0.00	0.23	0.08	0.08
3001	Pt	1024	0.00	4.83	0.31	0.18	931	0.00	3.78	0.31	0.22
3001	Pd	1024	0.00	3.59	1.63	1.88	931	0.01	3.23	1.68	1.85

* To maintain consistency in the tabulation, data for the 1000 zone is from Clow et al., 2008. Statistics for the updated interpretation of the 1000 zone is contained in table 14-4

TABLE 14-7 1000 DOMAIN ASSAY AND COMPOSITE STATISTICS

Starfield Resources Inc – Ferguson Lake Project

	Assays				Composites			
	Domain 1001	Domain 1002	Domain 10031	Domain 10032	Domain 1001	Domain 1002	Domain 10031	Domain 10032
Contained Samples	1617	955	503	497	1549	959	574	539
Nickel:								
Analyzed	1606	947	467	472	1537	948	523	509
Zero Grade	11	8	36	25	12	11	51	30
Minimum	0.001	0.003	0.004	0.003	0.000	0.000	0.000	0.000
Maximum	1.770	1.450	1.409	1.820	1.400	1.450	1.409	1.392
Mean	0.568	0.621	0.367	0.455	0.566	0.608	0.325	0.413
Standard Deviation	0.432	0.396	0.459	0.496	0.391	0.375	0.415	0.456
Coefficient of Variation	0.762	0.638	1.251	1.089	0.691	0.618	1.277	1.104
Copper:								
Analyzed	1603	945	465	467	1534	948	522	505
Zero Grade	14	10	38	30	15	11	52	34
Minimum	0.001	0.001	0.001	0.001	0.001	0.000	0.000	0.000
Maximum	13.300	13.047	4.140	11.070	4.892	5.625	2.756	5.245
Mean	0.902	1.027	0.372	0.455	0.852	0.956	0.326	0.399
Standard Deviation	0.850	0.997	0.499	0.739	0.628	0.707	0.428	0.574
Coefficient of Variation	0.943	0.970	1.343	1.626	0.737	0.739	1.314	1.440
Cobalt:								
Analyzed	1363	891	436	447	1537	946	523	507
Zero Grade	254	64	67	50	12	13	51	32
Minimum	0.001	0.001	0.001	0.001	0.000	0.000	0.000	0.001
Maximum	0.800	0.315	0.348	0.322	0.400	0.296	0.243	0.322
Mean	0.070	0.071	0.045	0.057	0.070	0.070	0.041	0.053
Standard Deviation	0.058	0.047	0.058	0.064	0.049	0.044	0.051	0.059
Coefficient of Variation	0.828	0.667	1.300	1.120	0.708	0.626	1.240	1.115
Palladium:								
Analyzed	1366	898	472	469	1549	957	574	537
Zero Grade	251	57	31	28	0	2	0	2
Minimum	0.005	0.005	0.005	0.005	0.001	0.000	0.006	0.005
Maximum	3.630	11.380	35.070	32.330	3.630	8.599	10.000	10.000
Mean	1.211	1.572	1.512	1.902	1.257	1.553	1.365	1.709
Standard Deviation	0.856	1.014	2.327	2.819	0.825	0.950	1.416	1.655
Coefficient of Variation	0.706	0.645	1.539	1.482	0.656	0.612	1.038	0.968

Platinum:								
Analyzed	1360	898	472	466	1545	957	574	537
Zero Grade	257	57	31	31	4	2	0	2
Minimum	0.001	0.001	0.001	0.001	0.001	0.000	0.001	0.001
Maximum	2.860	4.500	16.970	17.010	2.758	4.000	4.000	4.000
Mean	0.181	0.289	0.524	0.513	0.163	0.272	0.424	0.416
Standard Deviation	0.239	0.428	1.340	1.402	0.198	0.354	0.639	0.722
Coefficient of Variation	1.325	1.482	2.558	2.732	1.216	1.301	1.509	1.735
Sulphur:								
Analyzed	164	64	35	37	1537	948	523	509
Zero Grade	1453	891	468	460	12	11	51	30
Minimum	1.400	2.260	3.530	3.890	0.698	0.000	0.001	0.001
Maximum	35.290	35.600	36.190	36.970	37.990	38.780	38.140	39.610
Mean	22.241	25.643	26.279	25.868	17.936	19.293	10.535	12.995
Standard Deviation	10.145	8.789	11.162	10.689	11.200	10.939	11.846	12.926
Coefficient of Variation	0.456	0.343	0.425	0.413	0.624	0.567	1.124	0.995
Iron:								
Analyzed	212	77	39	48	1537	948	523	509
Zero Grade	1405	878	464	449	12	11	51	30
Minimum	5.180	6.510	6.980	6.880	1.301	0.001	0.005	0.003
Maximum	58.200	60.700	59.950	63.200	60.790	61.560	60.930	60.627
Mean	36.931	43.981	42.278	42.486	33.042	35.195	21.613	25.217
Standard Deviation	15.846	15.410	18.471	16.761	16.864	16.600	17.808	19.413
Coefficient of Variation	0.429	0.350	0.437	0.395	0.510	0.472	0.824	0.770

TABLE 14-8 MAIN WEST 1000 DOMAIN ASSAY CAPPING
Starfield Resources Inc. – Ferguson Lake Project

Metal	Capping Level	Upper Percentile	Metal Uncut	Metal Cut	Metal %
Cu %	6.0	0.4%	2,529	2,510	-0.75%
Ni %	1.5	0.2%	1,779	1,779	-0.03%
Co %	0.4	0.1%	187	187	-0.15%
Pd g/t	10.0	0.7%	4,540	4,414	-2.77%
Pt g/t	4.0	2.4%	1,020	929	-8.95%

GEOSTATISTICS

For the Mineral Resource estimate reported in Clow et al., 2008 spatial analysis of the Main West Zone assay data was completed by FSS Canada. Down hole variograms were used to establish the nugget effect and the range of correlation perpendicular to the strike and dip of the massive sulphide lenses. Strike and dip variograms were used to establish ranges across strike and down dip. Insufficient data exist to evaluate the massive sulphide domains on an individual basis. Thus, the analysis is based on the

total massive sulphide assay population. The results of the variography are summarized in Table 14-9.

TABLE 14-9 SUMMARY OF VARIOGRAM MODEL PARAMETERS ALL DOMAINS MAIN WEST ZONE
Starfield Resources Inc – Ferguson Lake Project

Metal	Nugget	Sill	Range			Model Type*
			X	Y	Z	
Co	0.5	0.5	70	140	10	Exp
Cu	0.45	0.55	100	200	9	Exp
Ni	0.5	0.5	70	140	10	Exp
Pd	0.3	0.7	70	140	8	Exp
Pt	0.5	0.5	50	100	9	Exp

Ranges: x=across strike, y=down dip, z=across the lens *Exp = Exponential

With the increase in drilling density, spatial analysis of the Main West Zone Domain 1000 series assay data was completed by RPA for the Mineral Resource update. Downhole semi-variograms were used to establish the nugget effect and the range of correlation perpendicular to the massive sulphide lenses. Strike and dip directional semi-variograms were used to establish ranges in those directions. The 2007 and 2008 infill drilling results together with the updated wireframes provide sufficient data to model the 1001 sub-domain and, to a lesser extent, the 1002 and 10032 sub-domains on individual basis using three dimensional semi-variograms. Modelled semi-variograms suggest strike and dip ranges between 10 m and 25 m, rather than the 50 m to 200 m ranges reported and used in the previous estimate. The reduced ranges are attributed to the following differences:

1. Use of the recent infill drilling, completed at 12.5 m spacings
2. Restriction of composites to individual sub-domains
3. Inclusion of both high-sulphide and low-sulphide intercepts to characterize grade variation
4. Use of larger and more continuous wireframes

An indication of the average grade range is provided by the omnidirectional semi-variogram shown in Figure 14-3. The average range is about 12 m. Directional semi-variograms in Figures 14-4 and 14-5 have less quality due to the limited number of sample pairs available within 10 m of each other; therefore, model results are considered approximate.

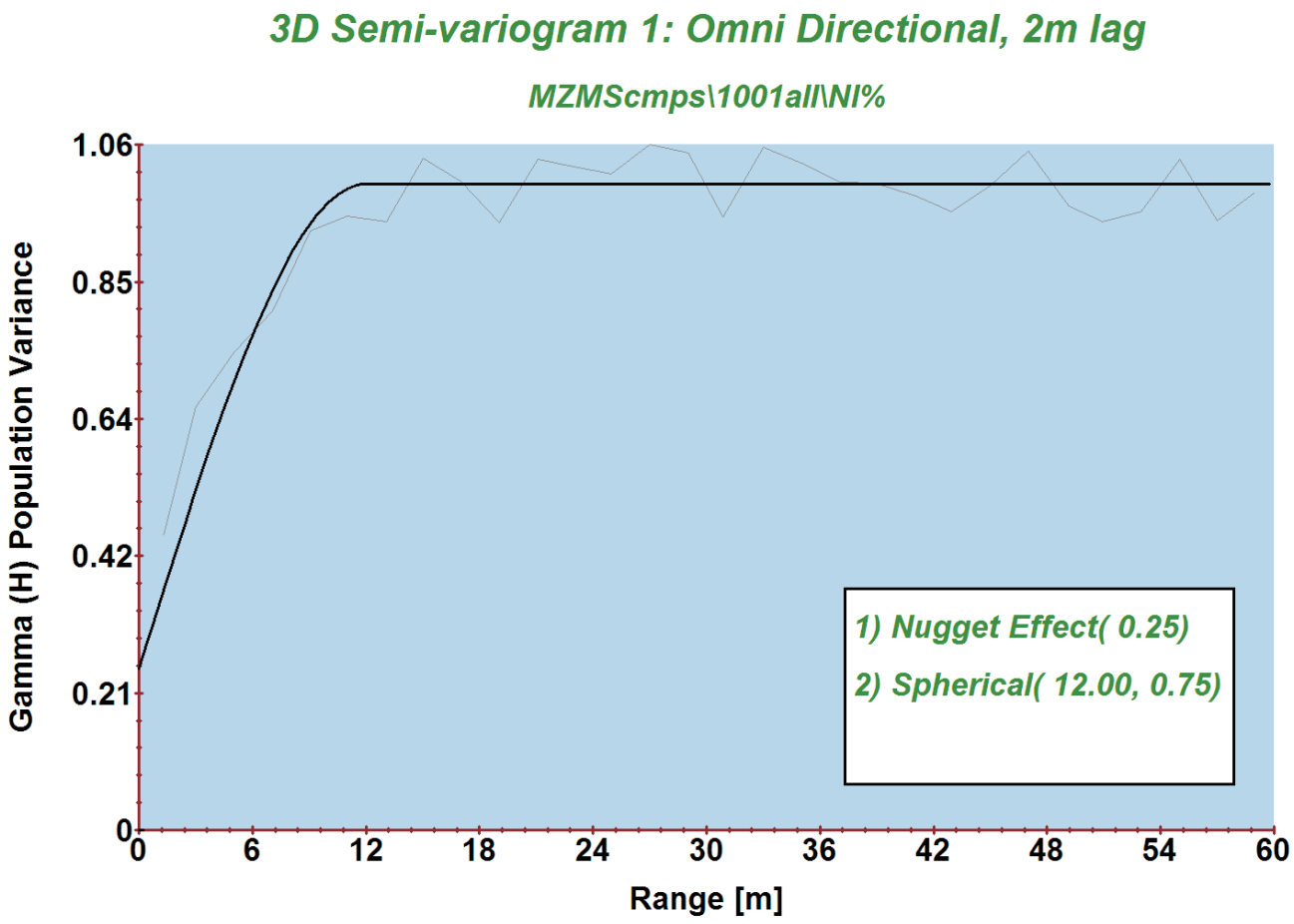


Figure 14-3

Starfield Resources Inc.

Ferguson Lake Project
Nunavut Territory, Canada

Domain 1001
Omini Directional Variogram

November 2011

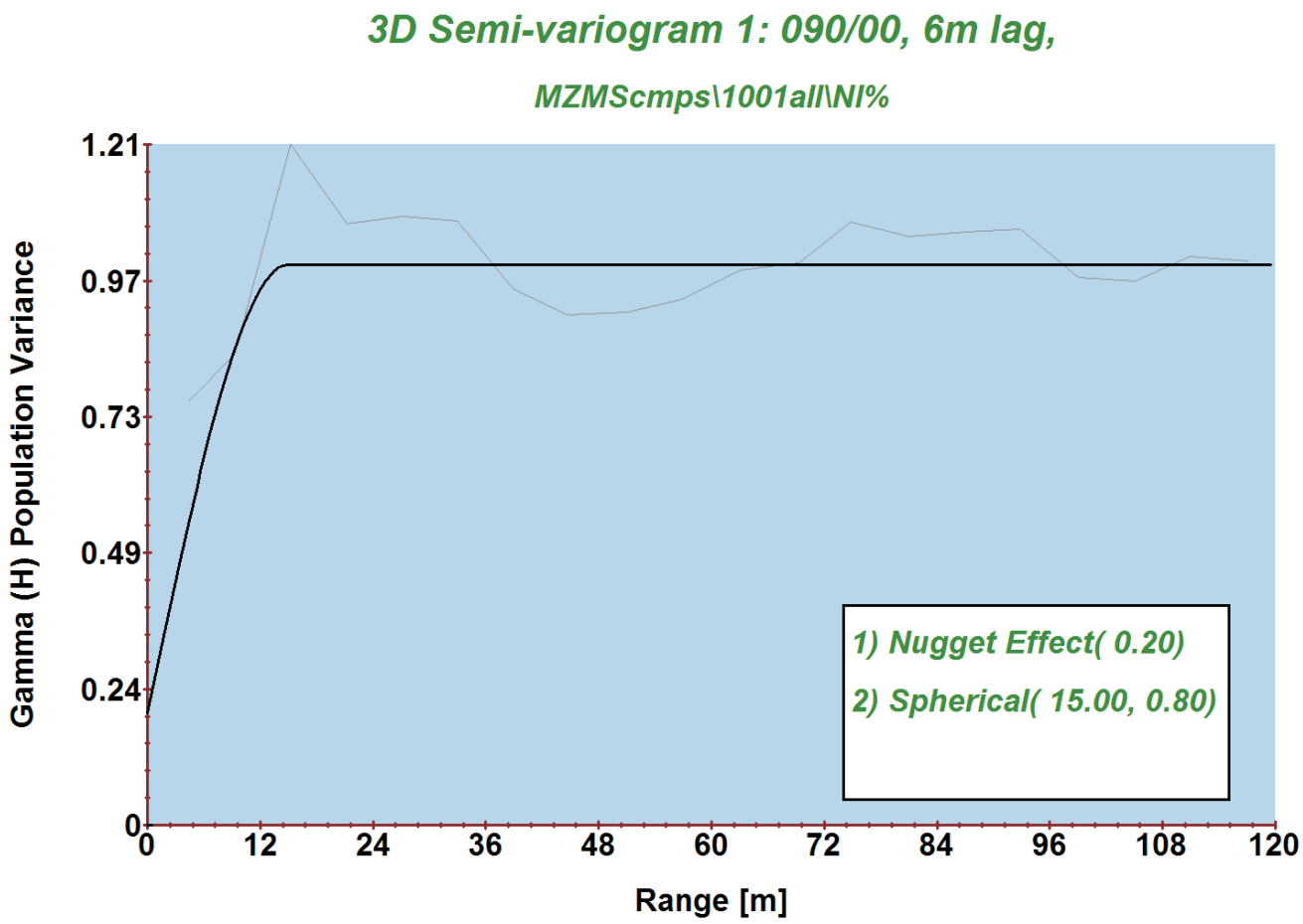


Figure 14-4

Starfield Resources Inc.

Ferguson Lake Project
Nunavut Territory, Canada

Domain 1001
Strike Direction Variogram

November 2011

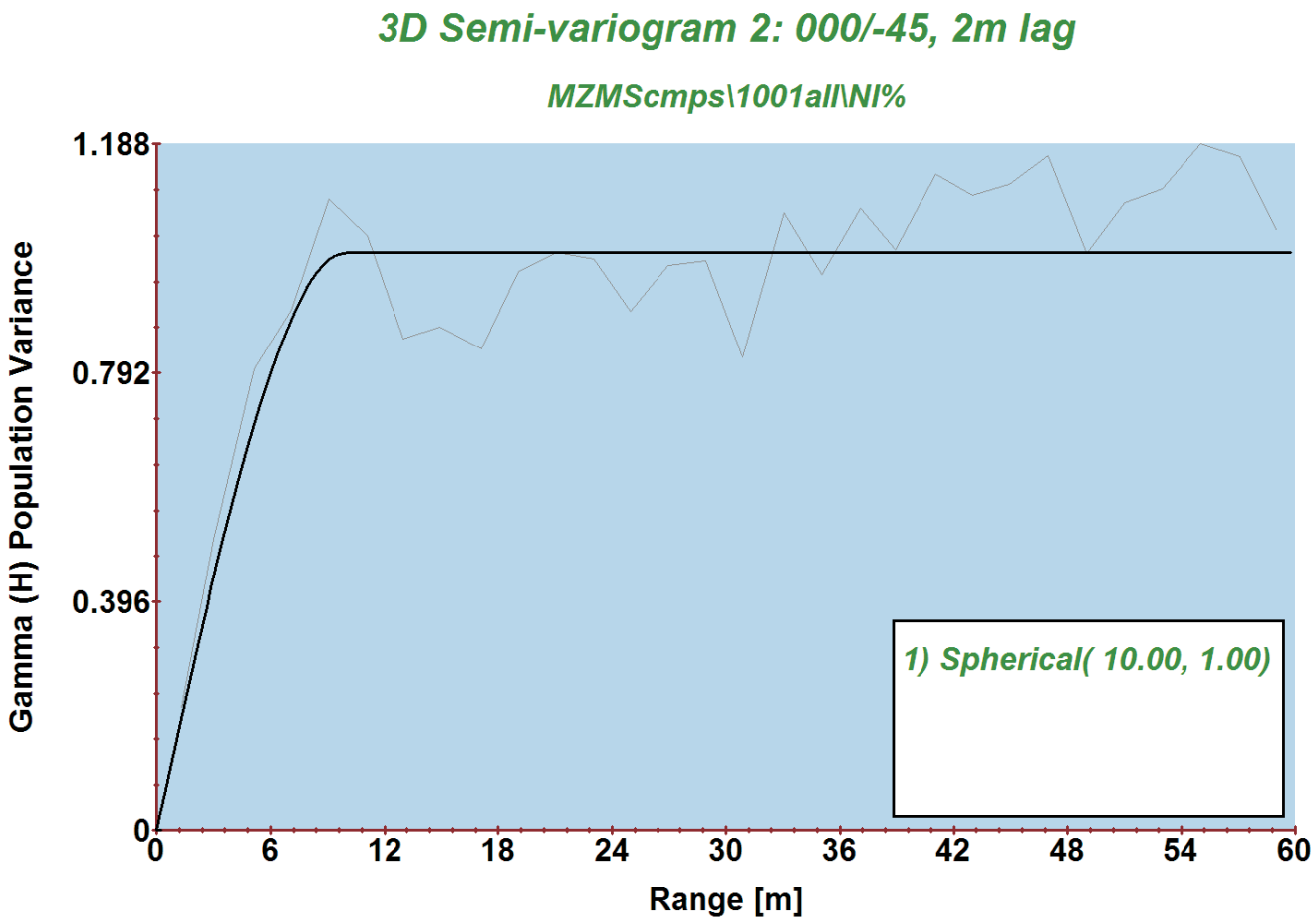


Figure 14-5

Starfield Resources Inc.

Ferguson Lake Project
Nunavut Territory, Canada

Domain 1001
Dip Direction Variogram

November 2011

Notwithstanding data limitations, semi-variogram models for each metal and each sub-domain generally point to the ranges summarized in Table 14-10.

TABLE 14-10 DOMAIN 1001 VARIOGRAM MODEL RESULTS
Starfield Resources Inc. – Ferguson Lake Project

			Range (m)			
Metal	**Nugget**	**Sill**	**X**	**Y**	**Z**	**Model Type***
Co	0.2	0.8	18	18	5	Spherical
Cu	0.2	0.8	20	10	6	Spherical
Ni	0.2	0.8	15	10	8	Spherical
Pd	0.2	0.8	13	10	8	Spherical
Pt	-	-	-	-	-	-
Omni	0.3	0.8	12	12	12	Spherical
Used	**0.2**	**0.8**	**20**	**15**	**5**	**Spherical**

Note: Ranges: x=along strike (090/00), y=down dip (000/-45), z=across the lens (180/-45) *Sph = Spherical

For the purposes of grade interpolation and considering their similarities, variograms were generalized into a single model that was applied to all metals and to all sub-domains. Variogram modelling results are consistent with on-section interpretations that showed generally good geological continuity of mineralized horizons with significant short range (<15 m) metal grade, metal distribution, mineralized thickness, and sulphur content variations along strike and down dip.

BLOCK MODEL

A block model was created to combine the results of the previous block model estimates and to complete updates to the model to support the current PEA. The block locations and dimensions are summarized in Table 14-11.

TABLE 14-11 BLOCK MODEL LOCATIONS AND DIMENSIONS

Starfield Resources Inc – Ferguson Lake Project

Parameter	Value
Minimum X	5000
Minimum Y	19300
Minimum Z	-1100
Model Rotation	0
No. Columns	1200
No. Rows	290
No. Levels	250
Column Width	5
Row Height	5
Level Height	5

GRADE ESTIMATION

The block model grades for Ni, Cu, Co, Pt, and Pd have been derived from the resource update for the 1000 Domain wireframes completed by Chinn and the Resource model reported in Clow et al. (2008) for the 2001 and 3001 domains.

1000 DOMAIN

A combination of ordinary kriging and inverse distance weighting interpolation methods were used to estimate block grades within each Main West Zone Domain 1000 wireframes. Based on database characteristics (partial analysis, zero-grade intercepts), as well as geological, statistical and geostatistical evaluations, the requirements for grade interpolation include:

- Use of capped values for Ni, Cu, Co, Pd, Pt, S, and Fe, including zero-grade composites for Ni and Cu from partially analyzed samples, from all holes.
- Use of some regressed values for Co, Pd, Pt, S, and Fe, from all holes.
- Exclusion of unsampled wireframe intercepts; consider that part of the hole as being undrilled.
- Same search orientations and ranges for the 1001, 1002, 10031, and 10032 sub-domains.
- Individual composite grade data for each of the 1001, 1002, 10031, and 10032 sub-domain interpolations, i.e., hard boundaries between sub-domains.

- Same kriging parameters (sill, nugget, ranges) for all metals.
- Same inverse distance weighted parameters (power, ranges) for each metal.

Multiple interpolation passes were used to assign grade to blocks within the wireframes. All combined, the multiple interpolation pass requirement together with interpolation requirements for each metal produced a large number of interpolation profiles. Interpolation parameters for each of four passes are outlined in Table 14-12.

The first pass interpolates grade within semi-variogram ranges in high data density areas. The second pass fills additional blocks within semi-variogram ranges by relaxing octant requirements and thereby capturing composites at greater distance, and interpolating block grades outside semi-variogram ranges in high data density areas by arithmetic averaging. The third pass interpolates remaining grade blocks within the semi-variogram ranges. Finally, the fourth pass fills blocks outside semi-variogram ranges in areas of few composites, well outside the variogram range.

TABLE 14-12 INTERPOLATION PARAMETERS FOR DOMAIN 1000
Starfield Resources Inc. – Ferguson Lake Project

Pass No.	Metal	Interp. Method	Nugget	Sill	Search Distances			Search Orientation		
					x	y	z	x	y	z
1	All	OK	0.2	0.8	20	15	5	080/00	350/-40	170/-50
2	All	OK	0.2	0.8	40	30	10	080/00	350/-40	170/-50
3	All	OK	0.2	0.8	20	15	5	080/00	350/-40	170/-50
4	All	ID^2	-	-	80	60	20	080/00	350/-40	170/-50

Notes:
1. Search type for pass 1 and 2 was octant; two was the minimum for interpolation. Search type for pass 3 and 4 was ellipsoidal.
2. For pass 1 the minimum number of samples was four and maximum number of samples was twelve. For pass 2, 3, 4 the minimum number of samples was two and maximum number of samples was twelve.
3. Search orientations are dip directions and dip for the principal axes.
4. OK = Ordinary kriging.
5. ID^2 = Inverse distance weighted squared.

2001 AND 3001 DOMAINS

For the 2000 and 3000 wireframes in the Main West Zone, ordinary kriging was used to estimate block grade in the massive sulphide. Based on the database as well as geological, statistical and geostatistical evaluation, requirements for grade interpolation in massive sulphide for the 2001 and 3001 domains include:

- Use of Ni and Cu from all of the Starfield and Inco holes but only use of the Pt, Pd and Co data from the Starfield holes. For interpolation of Pt, Pd, and Co, the Inco holes are considered not to be drilled.
- Individual search orientations and ranges for the 2001 and 3001 domains.
- Individual composite grade data for each of the 2001 and 3001 Domains interpolations.
- Individual kriging parameters (sill, nugget, ranges) for each of Ni, Cu, Co, Pt and Pd.

TABLE 14-13 INTERPOLATION PARAMETERS FOR DOMAINS 2001 AND 3001

Starfield Resources Inc – Ferguson Lake Project

Zone	Metal	Interp. Method	Nugget	Sill	Search Distances			Search Orientation		
					x	y	z	x	y	z
2001	Co	OK	0.5	0.5	70	140	10	103/00	013/-45	013/45
2001	Cu	OK	0.45	0.55	100	200	9	103/00	013/-45	013/45
2001	Ni	OK	0.5	0.5	70	140	10	103/00	013/-45	013/45
2001	Pd	OK	0.3	0.7	70	140	8	103/00	013/-45	013/45
2001	Pt	OK	0.5	0.5	50	100	9	103/00	013/-45	013/45
3001	Co	OK	0.5	0.5	70	140	10	090/00	000/-50	000/40
3001	Cu	OK	0.45	0.55	100	200	9	090/00	000/-50	000/40
3001	Ni	OK	0.5	0.5	70	140	10	090/00	000/-50	000/40
3001	Pd	OK	0.3	0.7	70	140	8	090/00	000/-50	000/40
3001	Pt	OK	0.5	0.5	50	100	9	090/00	000/-50	000/40

Notes:
1. Search type for all metals and zones was octant and minimum number of octants for interpolation was three.
2. Minimum number of samples was four and maximum number of samples was 20.
3. Search orientations are dip directions and dip for the principle axes.
4. OK = Ordinary Kriging

Multiple interpolation passes were required to assign a grade to all blocks in the massive sulphide models. As a result of the requirement of multiple interpolation passes combined with interpolation requirements for individual metals and domains, a large number of kriging profiles were created. The interpolation parameters for the first pass for each metal in the 2001 and 3001 domains are contained in Table 14-13. The second pass interpolation was the same as the first pass but with search ranges doubled. In the

third pass, the search range remained double the first pass and the number of octants required to interpolate a block was reduced from three to two. The fourth pass was based on an ellipsoidal search with the number of samples required for assignment of block grade reduced to two. For each metal, approximately 90% of the blocks were assigned a grade in the first two passes. As an example, the nickel block model is shown on the drill sections in Figure 14-4.

TONNAGE FACTOR AND TONNAGE ESTIMATE

The SG data from the 2006 drill campaign showed good correlation between SG and nickel grades for samples both inside and outside the massive sulphide domains within the gabbro. The linear regression equations for these populations were calculated as:

- For the low-grade gabbro, SG = 1.1712 * Ni + 2.8968.
- For the massive sulphide, SG = 1.3638 * Ni + 2.9435.

These formulas were used to estimate individual block density based on interpolated Ni grade in the 2000 and 3000 zone wireframes.

The SG data for the 1000 zone was isolated and re-assessed with the update to incorporate the 2007 and 2008 drilling. The linear regression equation and goodness of fit parameter to nickel grades was found to be:

- SG = (1.3796*[NI%]+2.8935)
- R^2 = 0.95

The graph of Ni (%) vs SG is illustrated in Figure 14-6. The specific gravity regression equation was used to assign density to individual assay samples. In turn, these values were composited and then interpolated utilizing the Ni interpolation parameters.

Block tonnage is calculated as the block volume (125 m^3) multiplied by the fraction of the block within the domain wireframe (%/100) multiplied by the density (tonnes/m^3).

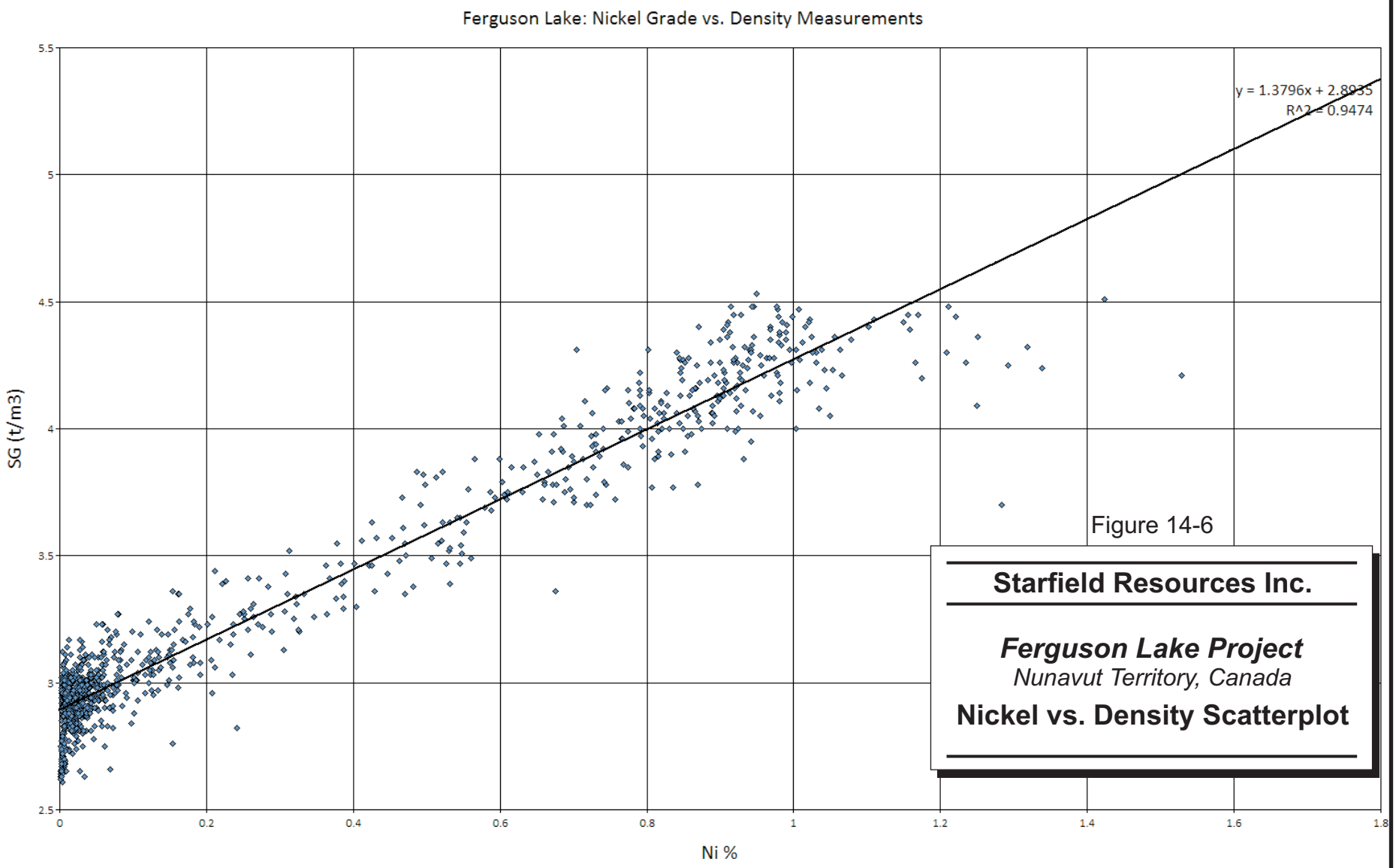


Figure 14-6

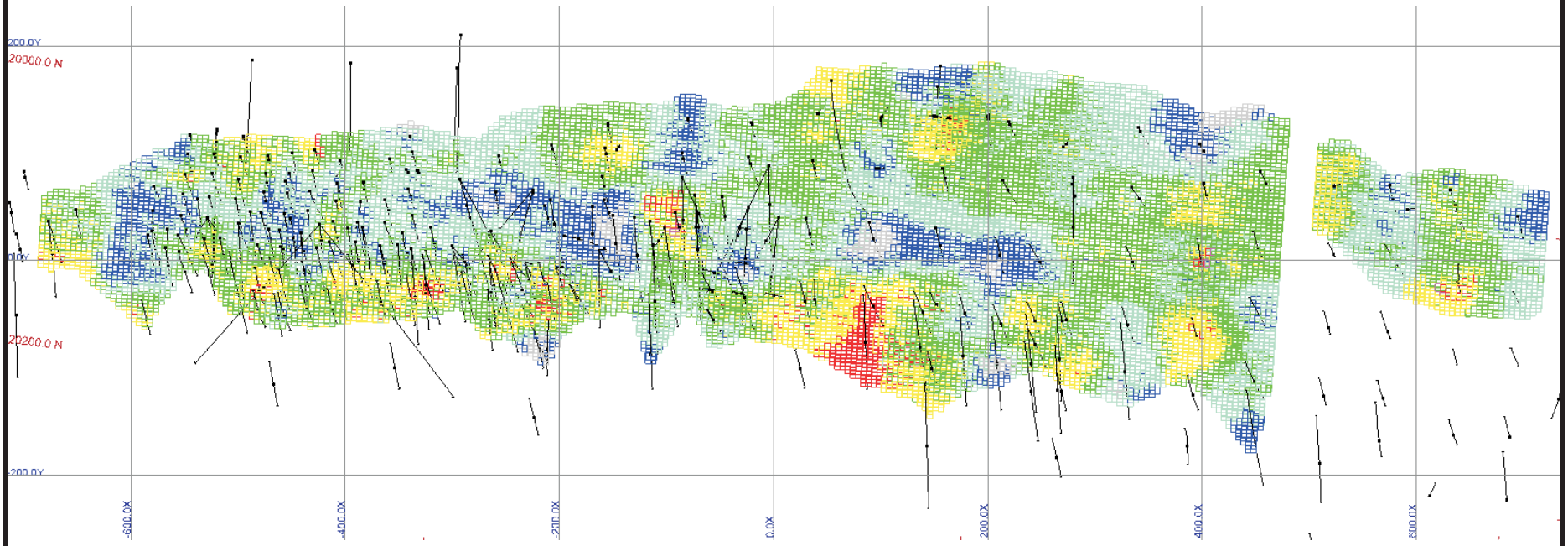
Starfield Resources Inc.

Ferguson Lake Project

Nunavut Territory, Canada

Nickel vs. Density Scatterplot

www.rpacan.com



Ni: Ni %

0.01	0.10
0.10	0.25
0.25	0.50
0.50	0.75
0.75	1.00
1.00	1.25
1.25	100.00

Figure 14-7

Starfield Resources Inc.

Ferguson Lake Project

Nunavut Territory, Canada

Domain 1001 Longsection Ni

RPA

www.rpacan.com

IRON AND SULPHUR ESTIMATIONS

Evaluation of assays and geochemical analytical data indicate strong correlations of nickel with iron and nickel with sulphur. Iron and sulphur have been estimated for each block in the block model based on regression analyses of nickel/iron and nickel/sulphur. Analyses of iron and sulphur have been largely from samples derived from the 1000 domain within the Main West Zone. The regression formulae used to calculate iron and sulphur contents in the block model are contained in Table 14-14.

TABLE 14-14 MAIN WEST 1000 ASSAY REGRESSION
Starfield Resources Inc. – Ferguson Lake Project

Metal	Regression To Nickel Equations	R^2
Fe	$(-16.068*[NI\%]^2+61.16*[NI\%]+6.6565)$	0.95
S	$(-7.5463*[NI\%]^2+37.168*[NI\%]+0.7477)$	0.88

RESULTS

ECONOMIC EVALUATION

As per NI 43-101, a Mineral Resource is “an inventory of mineralization that under realistically assumed and justifiable technical and economic conditions might become economically extractable”. The economic and technical parameters used to evaluate the massive sulphide block model are summarized in Table 14-15. No economic, environmental, legal, socio-economic, or governmental factors are known to RPA that would impact the estimate of Mineral Resources at Ferguson Lake.

Using the metal price, exchange rate, royalty payable, and recovery data, a dollar value was calculated for each block or partial block in massive sulphide (i.e., block NSR value). Resource cut-off value is established by the estimated operating costs. In order to assess the potential for open pit resources, RPA assessed the 2008 Whittle shells using a cut-off grade of $53/tonne. Blocks with NSR equal to or greater than $53/tonne within the 2008 Whittle shells have been tabulated as open pit resource. Blocks outside of the pit shells were assessed and those with NSR value of $97/tonne or greater have been tabulated as underground resources.

TABLE 14-15 NSR CUT-OFF VALUE INPUTS
Starfield Resources Inc. – Ferguson Lake Project

Economic and Recovery Factors		
Metal	**Price per pound**	**Recovery**
Nickel	US$9.00	91%
Copper	US$3.15	96%
Cobalt	US$17.00	90%
Iron	Note 1	74%
Sulphur	Note 1	67%
Product	**Price per tonne**	
Sulphuric Acid (93%)	US$80.00	
Hematite (62%)	US$100.00	
C$/US$ Exchange Rate: 1.00		
Royalty Payable: 2% Net NSR		
Operating Costs		
Breakeven underground mining cost per tonne milled		$97.00
Pit mining cost per tonne milled		$53.00

Notes:

1. NSR cut-off prices for pure iron and sulphur derived from iron and sulphur molecular weights in their respectively hematite and acid products.

CLASSIFICATION

Resource blocks in the Main West Zone have been classified as Indicated Resources and Inferred Resources. During block interpolation, the following data were recorded:

- Number of samples used to interpolate each block
- Number of octants used for the interpolation of each block
- Block variance for Ni grade estimation

Classification of a Mineral Resource as Indicated was based in part on consideration of the block model of these statistics in individual blocks, neighbouring blocks, and groups of blocks. This classification was also based on:

- Regular and tighter drill hole spacing
- Contours of sulphide thickness and therefore interpretation of thicker and apparently more continuous massive sulphide

The tabulation of Indicated and Inferred Resources in the Main West Zone is contained in Table 14-16.

TABLE 14-16 MINERAL RESOURCES, MAIN WEST ZONE

Starfield Resources Inc – Ferguson Lake Project

INDICATED RESOURCES

Zone	Tonnes (000t)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Fe Grade	S %
1000 Pit	1,100	0.63	0.97	0.22	0.07	1.54	36.33	19.95
2000 Pit	20	0.59	0.72	0.13	0.11	1.65	37.11	20.07
Total Pit	**1,120**	**0.63**	**0.97**	**0.22**	**0.07**	**1.54**	**36.34**	**19.95**
1000 UG	5,500	0.62	0.85	0.19	0.08	1.46	35.32	19.33
2000 UG	2,200	0.56	0.79	0.20	0.06	1.22	35.62	19.11
3000 UG	7,100	0.69	1.16	0.32	0.08	1.73	41.15	22.81
Total UG	**14,700**	**0.65**	**0.99**	**0.25**	**0.08**	**1.55**	**38.16**	**20.97**
Total Indicated	**15,820**	**0.65**	**0.99**	**0.25**	**0.07**	**1.55**	**38.04**	**20.90**

INFERRED RESOURCES

Zone	Tonnes (000 t)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Fe Grade	S %
1000 Pit	200	0.57	0.92	0.18	0.07	1.38	33.60	18.22
2000 Pit	20	**0.59**	**0.72**	**0.13**	**0.11**	**1.65**	**37.11**	**20.07**
Total Pit	**220**	**0.57**	**0.90**	**0.17**	**0.07**	**1.40**	**33.96**	**18.40**
1000 UG	1,200	0.58	0.78	0.15	0.07	1.38	33.61	18.30
2000 UG	3,400	0.59	0.71	0.15	0.07	1.20	36.54	19.75
3000 UG	1,300	0.63	1.19	0.36	0.07	1.71	38.72	21.16
Total UG	**5,900**	**0.59**	**0.82**	**0.20**	**0.07**	**1.34**	**36.15**	**19.59**
Total Inferred	**6,120**	**0.59**	**0.83**	**0.19**	**0.07**	**1.34**	**36.08**	**19.55**

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources were estimated at NSR cut-off values of $53 and $97 per tonne for open pit and underground, respectively.
3. Mineral Resources were estimated using average long-term metal prices of US$9.00, US$3.15, and US$17.00 per pound of nickel, copper, and cobalt, respectively, a US$/C$ exchange rate of 1.00, and a royalty of 2% NSR.
4. Metallurgical recovery assumptions were 91% for nickel, 96% for copper, and 90% for cobalt.

WEST EXTENSION AND EAST ZONES

ASSAYS AND INTERSECTIONS

The West Extension and the East zones were estimated and reported in Clow et al. (2008).

Massive sulphide intersections in diamond drill holes in the West Extension and East zones were evaluated for economic potential using a cut-off value of $75 for evaluation of intersections in the East Zone and the cut-off value of $110 for evaluation of intersections in the West Extension. Intersections above the cut-off value were incorporated into the Resource estimate. In both the West Extension and East Zone, a horizontal and vertical width of 2.5 m was used as a minimum width. As the West Extension only includes holes drilled by Starfield all of the Ni, Cu, and Co assay data were used to calculate intersection values.

In the East Zone where Inco holes do not include associated cobalt assay values, an average grade of 0.08% Co was used. This value is based on the average block grade from the Main West Zone.

POLYGONS AND SOLID EXTRUSION

Polygons were created on section limited by the defined intersections. The polygons were oriented to honour locally interpreted geological trends. The polygons were extruded to section limits which are half the distance to the next drill section. In three cases, two in the East Zone and one in the West Extension Zone, wireframes were created to handle closely and irregularly spaced holes. These wireframes represent less than 1% of the total volume of polygons created for the West Extension and East Zone.

BLOCK MODEL

DESCRIPTION

Two block models were created – one for each of the East Zone and the West Extension. The block model locations and dimensions are contained in Table 14-17.

TABLE 14-17 WEST EXTENSION AND EAST ZONE BLOCK MODEL DETAILS
Starfield Resources Inc – Ferguson Lake Project

	West Extension	East
Minimum X	5000	10300
Minimum Y	19300	18800
Minimum Z	-1100	-155
Model Rotation	0	0
No. of columns	600	1830
No. of rows	270	341
No. of levels	130	59
Column width	5	5
Row height	5	5
Level height	5	5

GRADE ESTIMATION

Nearest Neighbour interpolation was used to estimate grade in the block models. A number of search ellipses were constructed to closely approximate the major dip orientations of the interpreted sectional polygons. The use of specific search orientations minimizes the occurrence of blocks in one polygon being the "nearest neighbour" to an intersection in another polygon. As the West Extension only contains Starfield holes and therefore contains data for all of Ni, Cu, Co, Pt, and Pd, the grades of all metals were estimated into blocks. However, the East Zone mostly contains Inco drill holes and therefore only Ni and Cu block grades have been determined for the East Zone.

TONNAGE FACTOR AND TONNAGE ESTIMATE

The regression equation established and presented above for the Main West Zone 2001 and 3001 domains was utilized to establish block density for the West Extension and East Zone. Density was calculated based on an interpolated Ni grade. As for the Main West Zone, tonnage was estimated through the multiplication of the block volume within the extruded wireframe and the density where the volume is determined by the Gemcom percent block model.

RESULTS AND CLASSIFICATION

All blocks, by definition, are classified as a Mineral Resource as they occur within a domain defined by cut-off criteria. Based on the widely spaced drilling, all blocks in the

West Extension are classified as Inferred Resources. Similarly, the East Zone is generally characterized by relatively widely spaced drilling. Specific targets have been tested with closer spaced drilling in the East Zone, however, there is evidence in the East Zone of variable orientations of massive sulphide, and the controls on the orientation are not known. Thus, all blocks in the East Zone have been classified as Inferred Resources (Table 14-18).

TABLE 14-18 MINERAL RESOURCES – WEST EXTENSION AND EAST ZONES
Starfield Resources Inc – Ferguson Lake Project

Zone	INFERRED RESOURCES							
	Tonnes (Mt)	Ni %	Cu %	Co %	Pt g/t	Pd g/t	Fe %	S %
West Zone Ext.	14.7	0.71	1.23	0.08	0.31	1.88	41.63	23.14
East Zone	9.4	0.65	0.76	NE*	NE*	NE*	38.41	21.16
Total Inferred	**24.1**	**0.69**	**1.04**	**0.05**	**0.19**	**1.14**	**40.37**	**22.37**

NE - not estimated for the East Zone.

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources were estimated at NSR cut-off values of $53 and $97 per tonne for open pit and underground, respectively.
3. Mineral Resources were estimated using average long-term metal prices of US$9.00, US$3.15, and US$17.00 per pound of nickel, copper, and cobalt, respectively, a US$/C$ exchange rate of 1.00, and a royalty of 2% NSR.
4. Metallurgical recovery assumptions were 91% for nickel, 96% for copper, and 90% for cobalt.
5. Platinum, palladium and cobalt not estimated for the East Zone as the East Zone contains a higher proportion of historical INCO holes for which cobalt or individual platinum or palladium grades have not been determined.

VALIDATION

A number of procedures have been used to validate the estimation procedure and results:

- Sectional inspection of block models, drill hole grades, and wireframe limits was performed. The block grades and distribution of grades are consistent with the drill hole composite grades and the interpolation parameters. The volumes of blocks determined to be inside/outside rock type models was consistent in sectional inspection.

- Assay, composite, intersection and block statistics have been compared. The results of the comparisons are consistent with similar means and trends in median and variance consistent with increasing support from sample grades, to composite grades, to intersection grades, through to block grades.

- Wireframe volumes were compared with block model volumes and are generally in good agreement.

- Selected sub-regions of the Main West Zone were estimated using an inverse distance-squared interpolation. Local differences are explicable by the different interpolation method. The average model grades are acceptably similar.

- Tabulation of block model grade trends within and between the domains is consistent with assay statistics.

15 MINERAL RESERVE ESTIMATE

There are no Mineral Reserves at the Ferguson Lake Project.

16 MINING METHODS

Indicated and Inferred Resources used in the PEA are shown in Table 16-1. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Previously separated into open pit and underground categories, RPA notes that Mineral Resources were separated based on an arbitrary planar boundary, and are not representative of current operational concepts. Therefore, the two estimates were considered on a combined basis. Production quantities for the PEA consist of approximately 57% Inferred Resources.

TABLE 16-1 MINERAL RESOURCES FOR PEA
Starfield Resources Inc. – Ferguson Lake Project

Zone	Tonnes (Mt)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)
Indicated						
Main West Zone	**15.8**	**0.65**	**0.99**	**0.07**	**0.25**	**1.55**
Inferred						
Main West Zone	**6.1**	**0.59**	**0.83**	**0.07**	**0.19**	**1.34**
West Extension	14.7	0.71	1.23	0.08	0.31	1.88
Total Inferred	**20.8**	**0.67**	**1.11**	**0.08**	**0.27**	**1.72**

Notes:
1. CIM definitions were followed for Mineral Resources.
2. Mineral Resources were estimated at NSR cutoff values of $53 per tonne and $97 per tonne for open pit and underground, respectively
3. Mineral Resources were estimated using average long-term metal prices of US$9.00, US$3.15, and US$17.00 per pound of nickel, copper, and cobalt, respectively, a US$/C$ exchange rate of 1.0, and a royalty of 2% NSR.
4. Metallurgical recoveries were assumed to be 91% for nickel, 96% for copper, and 90% for cobalt.

East Zone Mineral Resources have not been considered for the purposes of this PEA, as they are lower grade, lack estimates for cobalt grades, and are physically isolated from the Main West Zone and West Extension (on the other side of Ferguson Lake). There is potential to extend the mine life in future studies, by inclusion of the East Zone.

LIFE OF MINE PLAN

To advance the Project, further studies and permitting work will be necessary in many areas, culminating in completion of a Feasibility Study. A decision to proceed with construction will allow financing and finalization of permitting. Construction timing will be dependent, to a certain extent, on logistics, as materials and equipment must be transported to the mine and process sites in stages constrained by seasonal weather conditions. This is a major issue and items required on site for activity in a given year must be moved by sea in the summer to a staging area for subsequent movement to the site in a brief window between February and March of the following year. Long-lead time items, such as grinding mills, may have to be ordered in advance of a final decision to proceed.

In RPA's opinion, there are too many uncertainties in the above process to predict a production start-up date for the Project with any accuracy. The Project Life of Mine Plan (LOMP) was therefore completed on an annual basis, using generic time periods (Year 1, Year 2, etc.).

SCHEDULE

Construction at the mine and process sites is scheduled to begin three years before production start-up, after supplies and materials are shipped to the process site the previous summer. Mine site supplies will be transported to site by winter Cat train.

YEAR -3

Construction materials and equipment for the camp and site buildings must be shipped to a staging site in the north and then to the Project site in the winter. Some site preparation work will be carried out at both sites.

YEAR -2

Construction of camp facilities and site buildings will begin at both mine and process sites. Open pit and underground mining equipment will be purchased and transported to Ferguson Lake. Slurry pipeline and powerline installation will be undertaken during the winter. Preparations for mine development and shaft sinking will commence.

YEAR -1

Construction of all surface facilities will be completed by the end of the year. Ramping and development in the Main West Zone, and shaft sinking in the West Extension will begin. Pre-stripping of the open pit will be completed.

YEAR 1

Production will begin, with feed from open pit mining at a production rate of 2,000 tpd. Development and stope preparation in the Main West Zone will be completed. Ramping and development will follow completion of shaft sinking in the West Extension. The process plant is scheduled to ramp up to full production of 6,000 tpd over the course of the year, averaging approximately 34% of full capacity.

YEAR 2

Production from the Main West Zone will begin at the start of the year. Open pit mining will be completed towards the end of the year, to be replaced by underground mining from the West Extension.

YEAR 3+

From Year 3 onwards, production will be split between the Main West Zone, at 3,300 tpd, and the West Extension, at 2,700 tpd. The Main West Zone is scheduled to be exhausted in Year 20, and the West Extension in 2019.

The LOMP is shown in Table 16-2.

Production grades were estimated using the following methodology:

- The resource model was completed using 5 m x 5 m x 5 m blocks with grades for Ni, Cu, Co, Fe, and S as well as the NSR values and densities. The resource model is a percent model with the percentage of ore estimated in each individual block.
- The resource model blocks were diluted to 100% ore.
- A cut-off value of $53/t (open pit) and $97/t (underground) NSR was applied to the blocks.
- The blocks above $97/t NSR were reported within the stoping units measuring 200 m long by 20 m high, which resulted in tonnage and grade estimates.

- Mining dilution and extraction factors were applied to the tonnage to get the final diluted and extracted tonnage and grade. Dilution for underground mining was added at 10% of average resource grade.

- The tonnes and grade were used to produce a production schedule of the 200 m by 20 m stoping units.

- The stoping units were determined by considering the access level intervals of 20 m and the fact that mining will take place 100 m either side of the central access to the orebody. Therefore access from one location at a grade of -20 % up to +20% will provide for five cuts of four metres in height.

TABLE 16-2 LIFE OF MINE PLAN
Starfield Resources Inc. - Ferguson Lake

			Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
PHYSICALS	Days Mined						350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350
Main West Open Pit	Ore Mined	tpd					2,073	2,983																		
	To Mill	'000 t	1,770				726	1,044																		
	To Dump	'000 t	1,783			300	731	1,052																		
	Ni	%	0.51				0.51	0.51																		
	Cu	%	0.82				0.82	0.82																		
	Pd	g/t	-				0.00	0.00																		
	Pt	g/t	-				0.00	0.00																		
	Co	%	0.06				0.06	0.06																		
	Fe	%	31.98				31.98	31.98																		
	S	%	16.64				16.64	16.64																		
Main West Underground	Production	tpd					-	2,143	3,301	3,301	3,299	3,301	3,301	3,299	3,300	3,300	3,300	3,300	3,299	3,299	3,300	3,301	3,301	3,301	4,280	5,092
	Tonnes	'000 t	22,512				-	750	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,498	1,782
	Ni	%	0.59				-	0.67	0.61	0.61	0.62	0.60	0.61	0.59	0.52	0.51	0.52	0.63	0.60	0.54	0.56	0.63	0.67	0.64	0.54	0.56
	Cu	%	0.88				-	1.02	1.18	1.10	1.03	1.07	1.00	0.82	0.68	0.66	0.63	0.77	0.69	0.72	0.80	0.99	1.13	0.89	0.73	0.86
	Pd	g/t	-																							
	Pt	g/t	-																							
	Co	%	0.07				-	0.08	0.07	0.07	0.07	0.07	0.07	0.07	0.06	0.06	0.06	0.08	0.07	0.07	0.07	0.07	0.07	0.08	0.06	0.07
	Fe	%	35.01				-	39.35	36.57	36.84	37.16	36.42	36.87	35.88	32.74	32.24	32.81	35.05	33.76	31.68	32.49	36.64	39.32	37.64	32.96	32.34
	S	%	19.17				-	21.96	20.08	20.25	20.47	19.96	20.26	19.61	17.53	17.22	17.54	19.60	18.60	17.14	17.68	20.28	21.94	20.90	17.85	17.62
West Extension Underground	Production	tpd					-	899	2,701	2,699	2,701	2,700	2,699	2,701	2,699	2,699	2,700	2,699	2,699	2,699	2,701	2,699	2,699	2,699	1,719	-
	Tonnes	'000 t	16,034				-	315	945	945	945	945	945	945	945	945	945	945	945	945	945	945	945	945	602	-
	Ni	%	0.66				-	0.67	0.62	0.58	0.71	0.68	0.73	0.59	0.63	0.60	0.69	0.72	0.70	0.77	0.67	0.61	0.61	0.61	0.64	-
	Cu	%	1.14				-	1.27	1.12	1.14	1.20	1.39	1.30	0.99	1.23	1.08	1.01	1.55	1.30	1.29	1.05	0.90	0.75	0.92	0.99	-
	Pd	g/t	-																							
	Pt	g/t	-																							
	Co	%	0.08				-	0.08	0.07	0.07	0.09	0.08	0.09	0.07	0.07	0.07	0.09	0.09	0.09	0.09	0.08	0.07	0.07	0.07	0.08	-
	Fe	%	38.64				-	39.32	37.22	35.52	40.90	39.46	41.48	35.63	37.74	36.63	39.99	40.91	40.33	42.86	39.20	36.83	36.85	37.05	37.79	-
	S	%	21.48				-	21.91	20.47	19.35	23.02	22.02	23.42	19.48	20.82	20.06	22.38	23.05	22.64	24.44	21.87	20.26	20.23	20.36	20.92	-
							34%																			85%
Total Mill Feed	Production	tpd					2,073	6,025	6,002	6,000	6,000	6,001	6,000	6,001	5,999	5,998	6,000	5,999	5,998	5,998	6,001	6,000	6,000	5,999	5,999	5,092
	Tonnes	'000 t	40,315				726	2,109	2,101	2,100	2,100	2,100	2,100	2,100	2,100	2,099	2,100	2,100	2,099	2,099	2,100	2,100	2,100	2,100	2,100	1,782
	Ni	%	0.61				0.51	0.59	0.61	0.60	0.66	0.63	0.66	0.59	0.57	0.55	0.59	0.67	0.65	0.65	0.61	0.62	0.64	0.63	0.57	0.56
	Cu	%	0.98				0.82	0.96	1.15	1.12	1.11	1.22	1.13	0.89	0.93	0.85	0.80	1.12	0.96	0.97	0.92	0.95	0.96	0.90	0.81	0.86
	Pd	g/t	-				0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Pt	g/t	-				0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Co	%	0.07				0.06	0.07	0.07	0.07	0.08	0.07	0.08	0.07	0.07	0.06	0.07	0.08	0.08	0.08	0.07	0.07	0.07	0.07	0.07	0.07
	Fe	%	36.32				31.98	35.69	36.87	36.25	38.84	37.79	38.95	35.77	34.99	34.21	36.04	37.68	36.72	36.71	35.51	36.73	38.21	37.37	34.34	32.34
	S	%	19.98				16.64	19.32	20.26	19.85	21.61	20.89	21.69	19.55	19.01	18.50	19.72	21.15	20.42	20.42	19.57	20.27	21.17	20.66	18.73	17.62

UNDERGROUND MINE DEVELOPMENT DETAIL

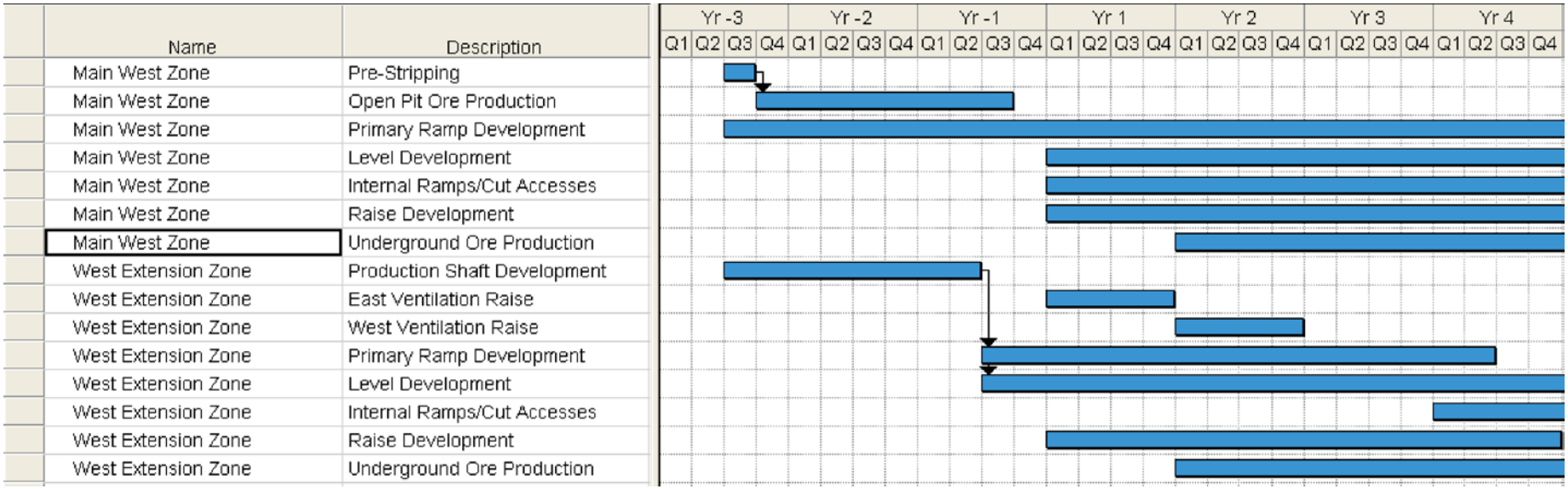
The Main West Zone access ramp will be driven 140 m below surface at a minus 15% grade. Starting at approximately 80 m below surface, main levels will be located some 40 m from the footwall of the deposit, on the 60 m level and 0 m level. From the main levels, stope access crosscuts will be driven from internal ramps on 200 m centres, at a minus 20% grade, to intersect the deposit for the initial cuts in several locations. Approximately 15 stoping blocks are required to be ready by the start of Year 2.

The West Extension will be accessed with a shaft that will be sunk to a total depth of 1,200 m. In addition to level access from shaft stations, a central ramping system will inter-connect the main levels, and provide additional flexibility for ore and waste haulage. Main level development will begin at the -600 m level, and continue down to the -1,020 m level. The development layout in the West Extension will be otherwise similar to that of the Main West Zone.

The shaft is scheduled to complete sinking by the end of Year 1. Level development off shaft stations will begin in advance of shaft completion, and five stoping blocks will be developed on the -600 m and -660 m levels, to be ready for production towards the end of Year 2. Additionally, the mine infrastructure development for services, refuge stations, ventilation and backfill distribution will be completed. Main ramp and level development in the footwall of both zones will continue to add more available work places for future production.

A pre-production development schedule is shown in Figure 16-1.

The average block of stopes, 200 m long by 60 m high, contain from 200,000 to 350,000 tonnes. Ongoing development for a block of this size includes two 40-m-long crosscuts and approximately 300 m of access ramp from the main level drift to the upper and lower sublevels (40 m vertical) for a total of 380 m of development.



Note: Level development includes a 10% allowance for excavation of underground facilities.

Figure 16-1

Starfield Resources Inc.

Ferguson Lake Project
Nunavut Territory, Canada

Preliminary Preproduction Development Schedule

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MINING METHOD SELECTION

OPEN PIT

Open pit mining potential was investigated by floating cone analysis, using Whittle software, run on the resource block model. The Main West Zone lies close to surface along a considerable proportion of the strike length, starting near the shoreline of Ferguson Lake, and extending to the west. Initial pit optimizations indicated that approximately 10% of the Main West Mineral Resources would be economic to mine using open pit methods, as opposed to underground methods. The relatively narrow mineralized lenses (average thickness is ten metres) and shallow dip (40°) cause stripping ratios to increase rapidly with depth.

From the site inspection, there appear to be few major issues related to potential open pit mining, although the pit crest will need to be offset from the shoreline of Ferguson Lake, and a seasonal stream will need to be diverted, as it runs across the surface of the deposit. Managing dilution through tight grade control may require small equipment to achieve selectivity in ore mining.

Despite the limited scope to which it applies, open pit mining has been included in the Base Case for the PEA, as it can provide process feed while underground development is completed. Detail on the open pit design is included below.

UNDERGROUND

The vertical extent of the deposit and the relatively narrow dimension (compared to the strike length) favour the use of underground mining methods for most of the deposit. The deposit includes single and stacked lenses which vary from several metres to over 15 m in width. A review of sections indicates that the average deposit dip is in the 40° range, with deviations of ±10°. In general, mining methods dependent upon gravity for muck flow, such as longhole stoping, are at a disadvantage, since the dip is shallow.

Underground mining methods that rely on caving were not considered appropriate for Ferguson Lake based upon the rock strength and competency. Longitudinal longhole stoping was considered, but has been set aside due to the shallow dip and the expectation that there would be high ore losses on the footwall or excessive dilution in

blasting a suitable footwall to permit the flow of ore. Dilution in the footwall could potentially be in the range of 20% to 30% to permit muck flow by gravity, in addition to the dilution that will occur over the normal course of mining.

An open stoping method without fill was also considered, but in RPA's opinion, safety concerns related to working in an open stope with a hanging wall dipping at 40° eliminated that option.

Appropriate options for underground mining of the Ferguson Lake deposit are:

- Cut and fill mining
- Transverse long hole stopes in areas where the deposit is over 15 m wide

Mechanized cut and fill mining is considered to be the most suitable method for the deposit. As there are only limited areas where the width exceeds 15 m, the purchase of specialized equipment to support and develop two separate mining methods is not warranted. Additional details in the comparison between cut and fill and transverse stoping are outlined below.

A mechanized mining system will provide increased flexibility to move manpower and equipment among workplaces spread over various locations along strike and dip. Mechanized equipment also provides greater efficiency in supporting higher production rates, along with ongoing mine development. Mechanized equipment will be sized to minimize manpower requirements, while still having suitable equipment fleets to provide flexibility and reliability consistent with the remote location.

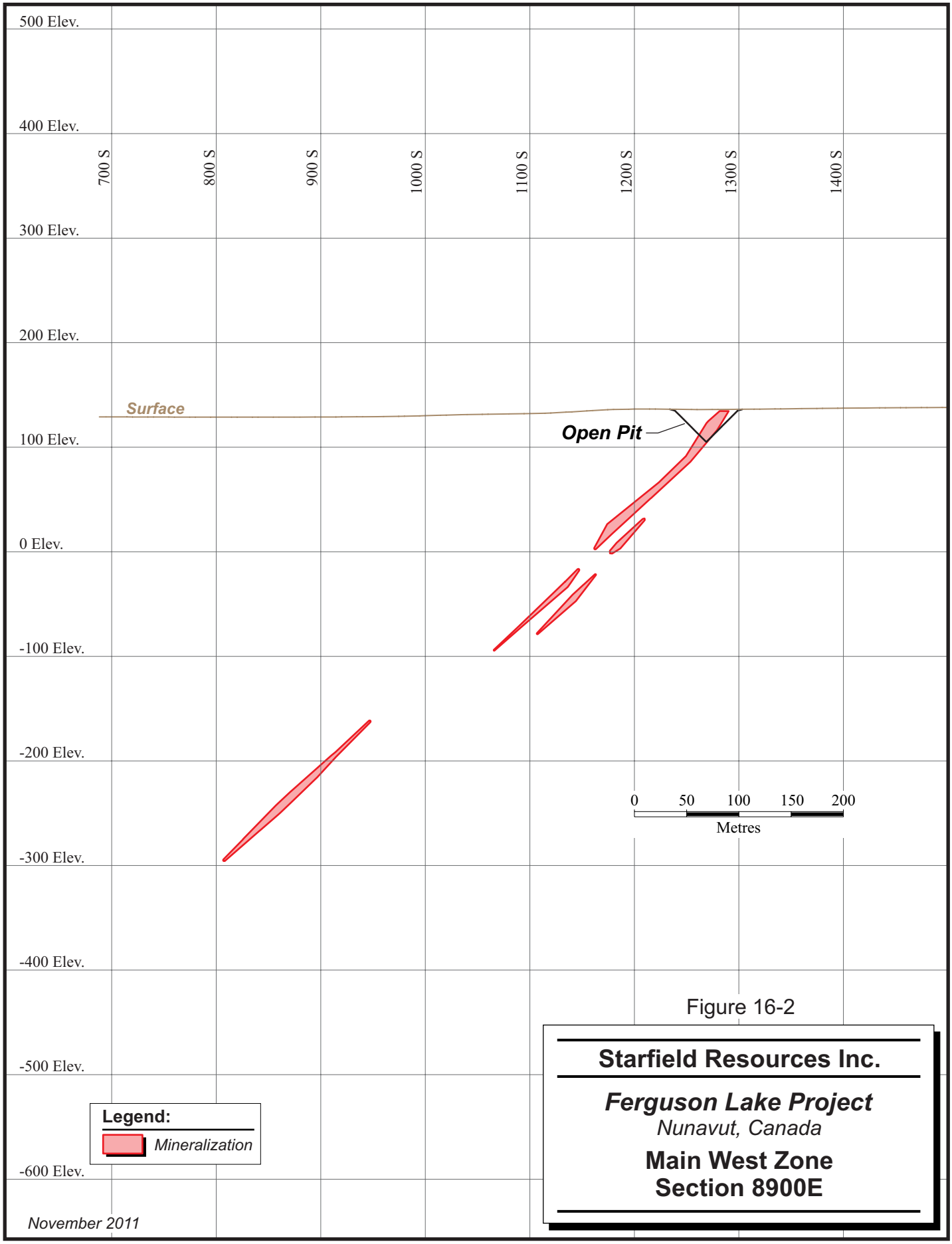

Figure 16-2

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Main West Zone
Section 8900E

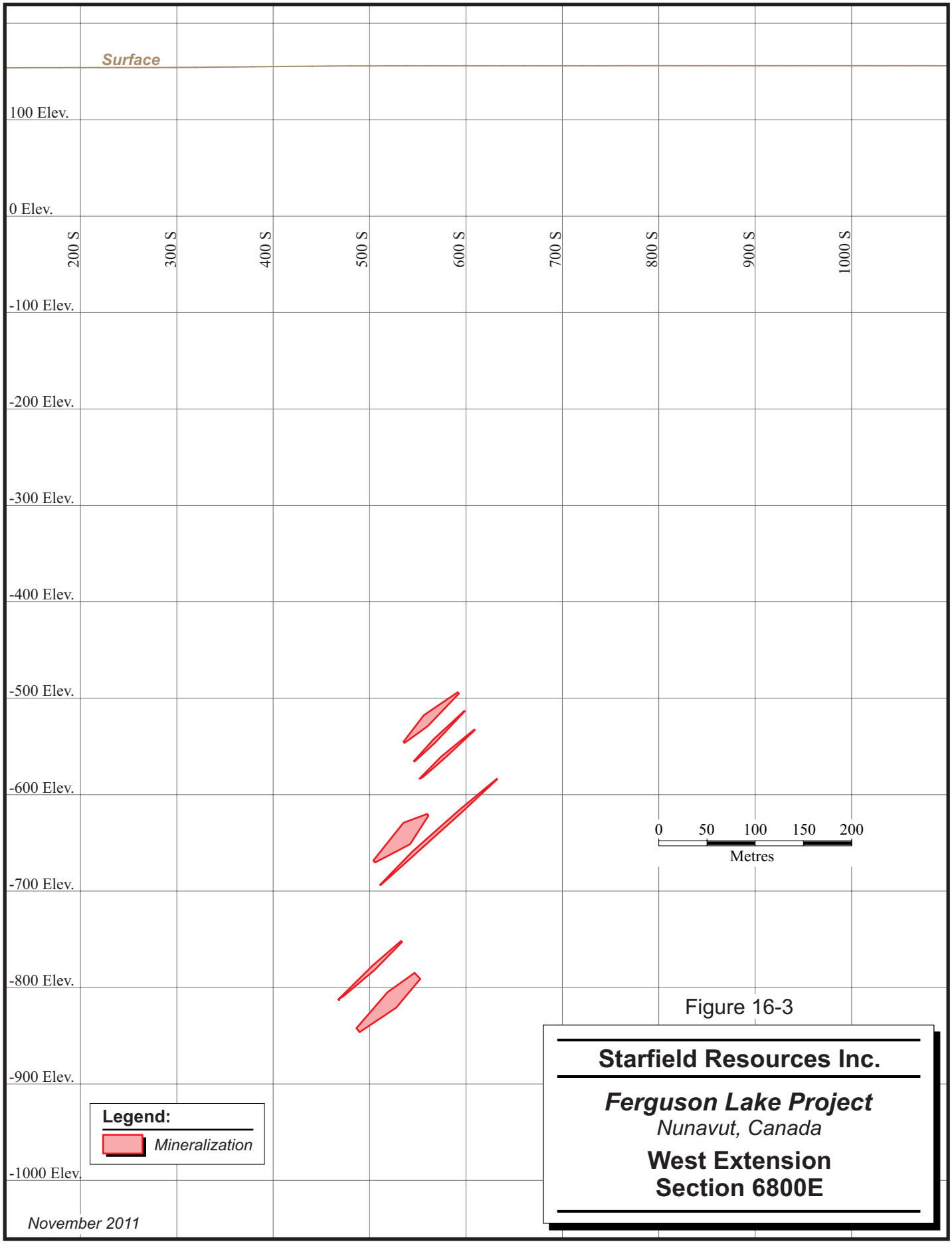

Figure 16-3

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

West Extension
Section 6800E

For this study, RPA has selected mechanized cut and fill stoping as the underground mining method, as described below:

- Drift along strike with parallel headings for extraction along zone, and backfill after each pass to provide support before taking adjacent passes.
- Mine progressively up-dip to permit breasting, reducing drilling and blasting efforts.
- Reduce dilution through the use of low-grade zones to be left as pillars, or used as fill in the immediate area.
- Backfill using waste rock and tailings, to minimize costs and enhance quick cycling of headings.
- Minimum mining width of four metres.

OPEN PIT DESIGN

RPA evaluated open pit potential by comparing the Net Smelter Return (NSR) to expected operating costs, through floating cone analysis using Whittle pit optimization software.

Traditionally, the term NSR is used for sulphide ores requiring smelting. In this report, the term is used to represent the same concept for hydrometallurgical treatment instead of smelting. NSR is a total value per tonne of ore, after metallurgical recovery and off-site deductions. Factors were developed to calculate NSR (in units of $ per tonne), using Ni, Cu, and Co grades as inputs. Table 16-3 outlines the NSR parameters used in the optimization exercise.

TABLE 16-3 NSR FACTORS
Starfield Resources Inc. – Ferguson Lake Project

Parameter	Value
Metallurgical Recovery (%):	
Nickel	91
Copper	96
Cobalt	90
Iron	74
Sulphur	69
Metal Price (US$/lb) & /t*:	
Nickel	9.00
Copper	3.15
Cobalt	17.00
*Sulphuric Acid (93%)	80.00
*Hematite (62%)	100.00
Exchange Rate ($/US$)	1.0
Royalty (%)	2.0
NSR Multipliers ($/%/t):	
Nickel	176.43
Copper	64.57
Cobalt	329.20
Iron	0.53
Sulphur	1.07

Calculation of NSR Factors includes an allowance for transportation of final products, and assumes that metals contained in those products are 100% payable, with no further processing charges. RPA notes that the NSR multipliers were developed using early metallurgical data. Since then, further process simulations have been completed, and the updated recoveries have been used for the purposes of cash flow modelling. The NSR factors were applied to block grades, generating a unit value for each block in the resource model.

Table 16-4 outlines the mining and milling cost inputs used for the Whittle optimization process. RPA did not redo the Whittle pit optimization for the present study as the change in NSR values, higher operating costs, and lower nickel recovery were expected to result in similar pit shells.

TABLE 16-4 WHITTLE INPUTS
Starfield Resources Inc. – Ferguson Lake

Parameter	Value
Pit slopes angles	45º
Open Pit Mining Cost ($/t moved)	2.90
Processing, Power and G&A Cost ($/t)	34.89

Further constraints were added to the model to account for the close proximity of the lake. It was assumed that the pit crest would be a minimum of 30 m from the lake's edge.

The Whittle pit shells were truncated at the bottom, to provide a minimum 20 m width for equipment operating on the lowest bench. Block size of 5.0 m by 5.0 m by 5.0 m was considered to be the minimum selective mining unit, and grades were diluted to full blocks. Table 16-5 outlines the results from the optimization exercise.

TABLE 16-5 OPEN PIT VOLUMETRICS (DILUTED)
Starfield Resources Inc. – Ferguson Lake Project

	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (g/t)	Pd (g/t)
Ore	1,770	0.51	0.82	0.06	NA	NA
Waste	2,083					
Total	**3,853**					
Stripping Ratio	1.18					

Platinum and palladium grades were not estimated in the present study, as prospects for their recovery have not yet been proven from recent testwork.

Figure 16-4 shows the pit limits, with respect to the zones of mineralization. The optimal pit has a total strike length of approximately 1.3 km, and has an average depth of 30 m.

RPA notes that the near surface mineralization in the East Zone may have open pit potential, however, it was not considered for the purposes of this PEA.

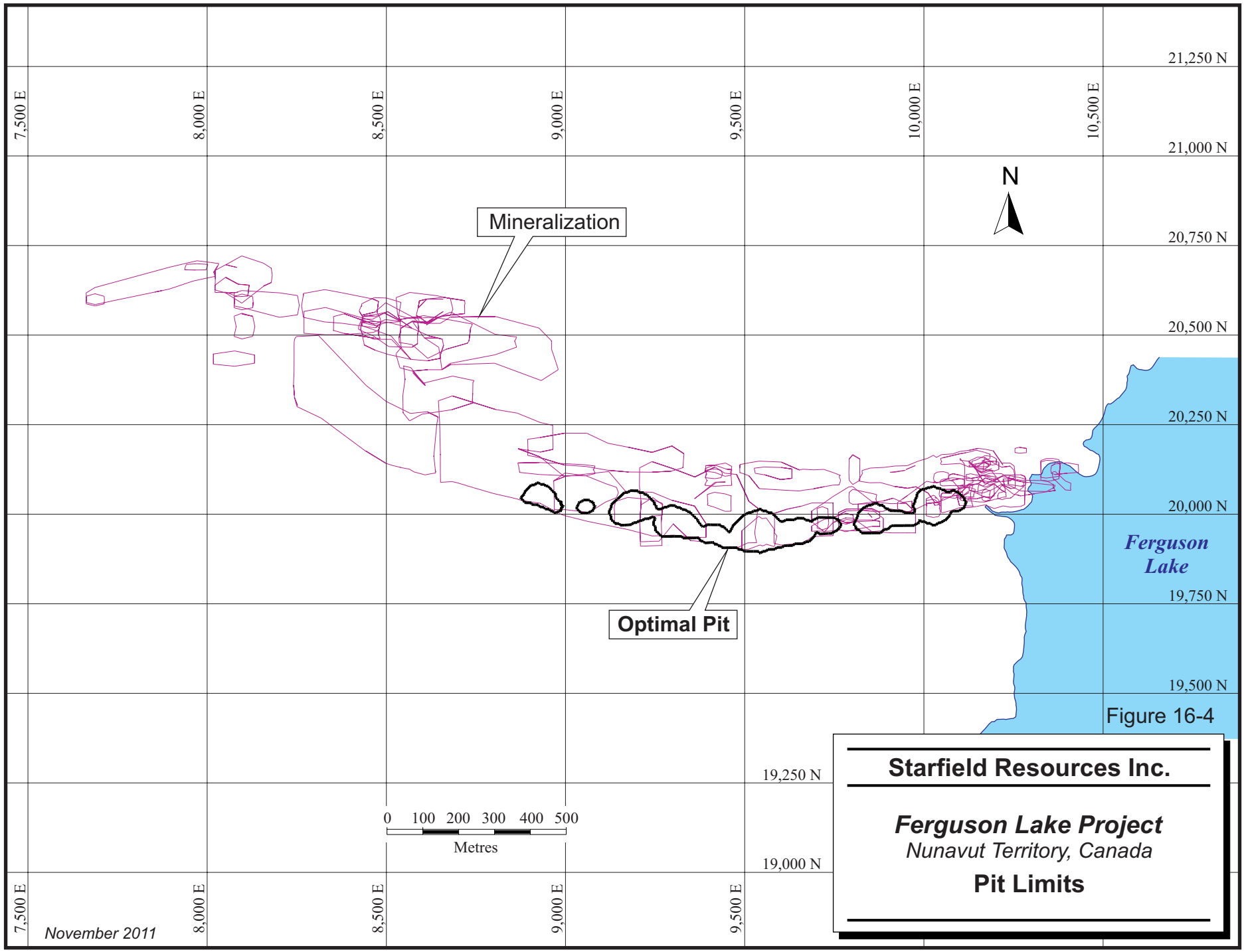

Figure 16-4

Starfield Resources Inc.

Ferguson Lake Project

Nunavut Territory, Canada

Pit Limits

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OPEN PIT MINING EQUIPMENT

Small-scale open pit mining equipment will be appropriate for the Project. The total size of the open pit is not large, and small equipment will allow better selectivity, an important consideration given the deposit geometry. Improved selectivity will help to reduce dilution. Table 16-6 summarizes the proposed equipment fleet for the open pit operation.

TABLE 16-6 OPEN PIT MINE EQUIPMENT FLEET
Starfield Resources Inc – Ferguson Lake Project

Number	Type	Description
1	Loader	950 F and 980 F
1	Excavator	Cat E320
5	Trucks	Cat 25 t Articulated
1	Dozer	Cat D4 and D8
1	Drill	Electric Hydraulic Blasthole Drill
1	Grader	140G
2	Utility	Service vehicle - Pickups

The ore and waste will be drilled and blasted with ANFO using a surface crawler-mounted electric hydraulic drill rig, drilling holes of approximately 100 mm diameter. Ore and waste will be loaded with a 980 size loader and there will be 950 size loaders on site to serve as backup in case of a failure with the larger loader. The small excavator on-site will provide additional back up for the loading fleet.

Ore and waste will be hauled in 25-tonne articulated dump trucks similar to the two already owned by Starfield. This size of truck will be the most flexible for use in the pit and in other aspects of the construction and ongoing operations.

In addition to the loader and trucks, there will be a requirement for one blasthole drill, one dozer, and service vehicles to support the open pit operation.

UNDERGROUND DESIGN

The Main West Zone, which covers a strike length of about 2,700 m, has a vertical extent of 850 m. The West Extension, which continues west of the Main West Zone at depth, has a strike length of approximately 2,400 m and extends an additional 400 m

vertically. Combined, the two zones extend over a strike length of 5,100 m, and cover a vertical extent of 1,200 m. The deposit is interpreted as having one or more parallel mineralized zones that vary in width and length along strike. Selection of the most favourable structures will be necessary as mine planning progresses. This may lead to the exclusion of some narrow structures and/or the inclusion of some dilution between structures that are combined for mining.

An analysis of NSR values for the various resource blocks was performed to better evaluate mine planning. It was found that the highest value material coincided with the mid-elevations of the western portion of the Main West Zone and mid-elevations of the eastern portion of the West Extension Zone. These mining areas were targeted for early extraction in the LOMP in order to optimize Project NPV. Such consideration should also be given to other zones that are included in future studies.

MINE ACCESS

The bulk of the tonnage for the Main West Zone is located over a vertical extent of approximately 600 m starting from surface. Mine access and material handling for this zone is assumed to be by ramp. The West Extension is found between 600 m and 1,200 m below surface. Shaft access will be required to reduce travel time for crews and for getting the ore to surface. Both zones will have internal ramping systems connecting the mining levels. As mining and development progress, the Main West Zone ramp will connect to the West Extension ramps and levels, allowing either the shaft or the ramp to be used to get ore to surface.

RPA notes that 600 m vertical distance approaches the maximum for truck haulage achieved in operations to date, and recommends that the split between ramp and shaft be investigated further in future studies.

The West Extension shaft will be six metres in diameter, and lined with concrete. Shaft sinking will proceed using conventional shaft-sinking techniques, assumed to be carried out by contractor. The shaft will be served by a ground-mounted single-drum hoist, powered by a 1.6 MW high-speed motor. A skip with a 13 tonne payload will be used. Hoisting capacity will be 325 tonnes per hour, or 4,000 tonnes per day, based on a shaft utilization of 13 hours per day, with an availability factor of 95%. Hoisting capacity in

excess of the planned production rate has been included, to ensure that the hoisting system does not become a bottleneck on production.

The layout of conveyances within the shaft is based on a single skip system for the extraction of ore, counterbalanced with a cage for carrying men and materials. A three metre by three metre envelope within the shaft configuration has been included to allow lowering of equipment by slinging it beneath the conveyance.

PERMAFROST

Preliminary indications from drilling are that permafrost extends to 300 m below surface at this location. Further analysis and testing is necessary to determine rock temperatures, permafrost gradient, and rock stability for permafrost, thawed permafrost, and below the permafrost. For this PEA, RPA has assumed that the rock is stable in a thawed state, so summer refrigeration of mine air will not be necessary. The mine will not be heated, except for local areas such as lunch rooms and shop offices.

Based on the assumption of the mine operating in an unheated condition, it will be necessary to choose a dust suppression method for drilling. Two choices are brine circulation and pumping, as used at Xstrata's Raglan Mine in Ungava, Quebec, or on-board dry dust collectors on drilling equipment, as used at both Polaris Mine and Nanisivik Mine in Nunavut. RPA notes that mining at Ferguson Lake is anticipated to extend both within and below the permafrost, as is the case at Raglan Mine, so the brine system is likely to be appropriate.

PRODUCTION RATE

A production rate of 6,000 tpd based on 350 days per year was selected for evaluation of the Project. The Main West Zone and West Extension will essentially be operated as two separate mines, nominally producing 3,000 tpd each. Various rules of thumb, of which Taylor's Law is an example, indicate that 3,000 tpd for each mine is near the maximum recommended production rate.

RPA notes that the significant horizontal extent of the deposit and the relatively narrow lenses will require a large quantity of level and internal ramp development to provide sufficient cut and fill faces for a continuous 6,000 tpd operation. To start underground

production at 6,000 tpd, pre-production development requires approximately two main levels to be developed in each of the Main West Zone and West Extension. To maintain this rate through the mine life, ongoing development requires a new main level to be developed with each passing year.

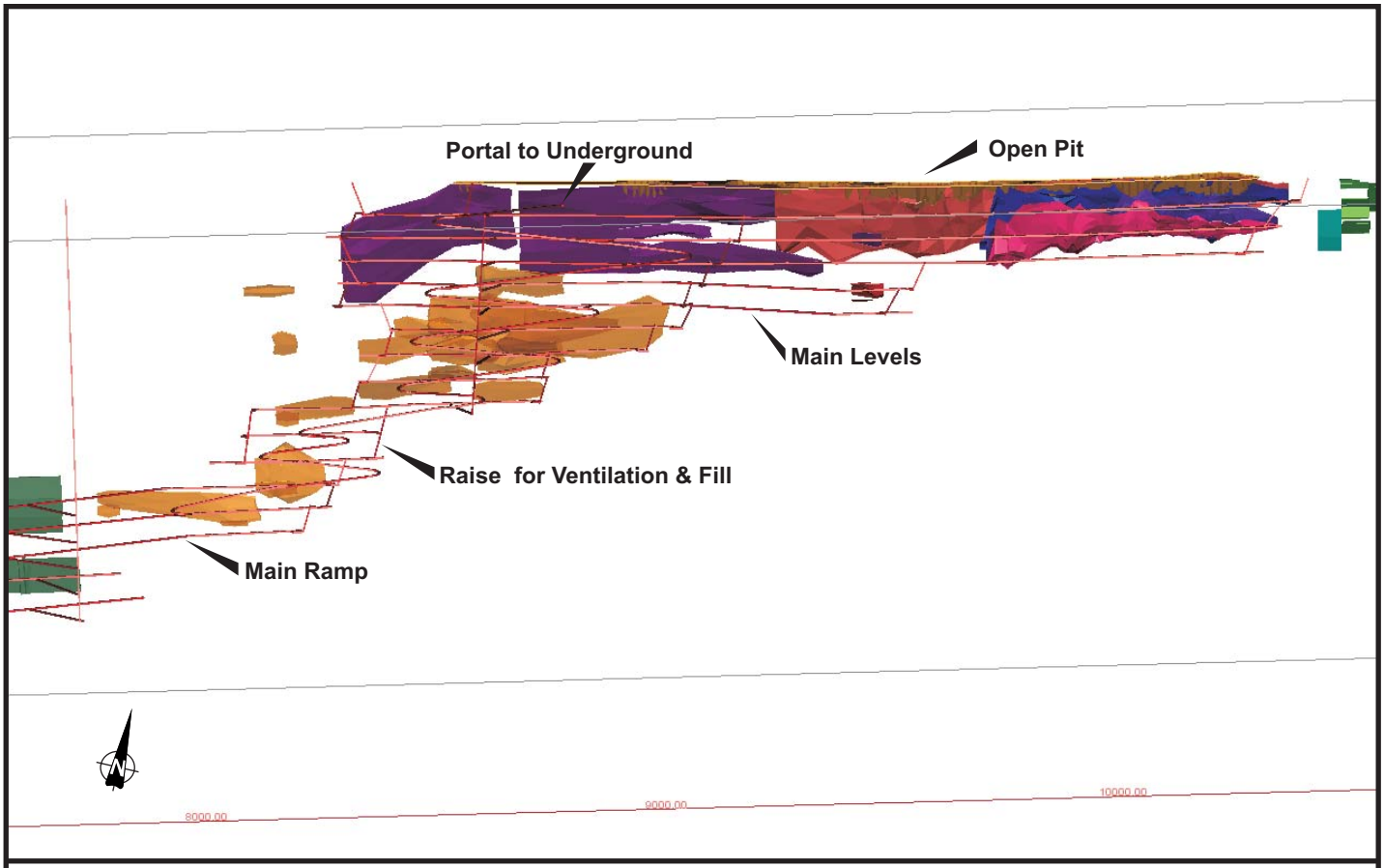
In RPA's opinion, the proposed 6,000 tpd production rate is in the upper range of achievable rates for such a deposit, and recommends that it be reviewed as mine plans are developed in more detail.

Isometric views of the mine and orebody for the Main West Zone and West Extension are shown in Figures 16-5 and 16-6, respectively.

CUT AND FILL STOPING

All underground mining is assumed to be by mechanized cut and fill stoping with rock fill from pit waste, underground waste development and surface waste sources. Stoping will be done with two-boom electric hydraulic jumbos for drilling breasts, LHDs for mucking and loading trucks, and trucks for ore haulage to ore passes or to surface.

Ramping is planned to be in the footwall, linking main levels that will be 60 m apart vertically (Figure 16-7), dividing lenses into stoping blocks. Sublevels will separate the blocks into 20 m vertical portions. Stope access crosscuts will be driven 40 m to the ore from the main and sublevels, on 200 m centres along strike. The crosscuts will be used to access each 20 m vertical portion of a block starting with a minus 20% grade down-ramp, which will be slashed after each lift, until it ends as a plus 20% grade up-ramp. The 200 m spacing will yield two 100 m long stope cuts, running east and west from the access crosscut. Each cut will be four metres high, with each access crosscut servicing five cuts. Ore will be loaded into trucks either in the stope or at the junction of the stope crosscut and the footwall level. The stopes will be ventilated by fans and ducting taking fresh air from the levels. Waste fill will be delivered by LHD and trucked from ongoing development or from waste raises, which will be driven to surface and connect to the main levels at convenient locations.



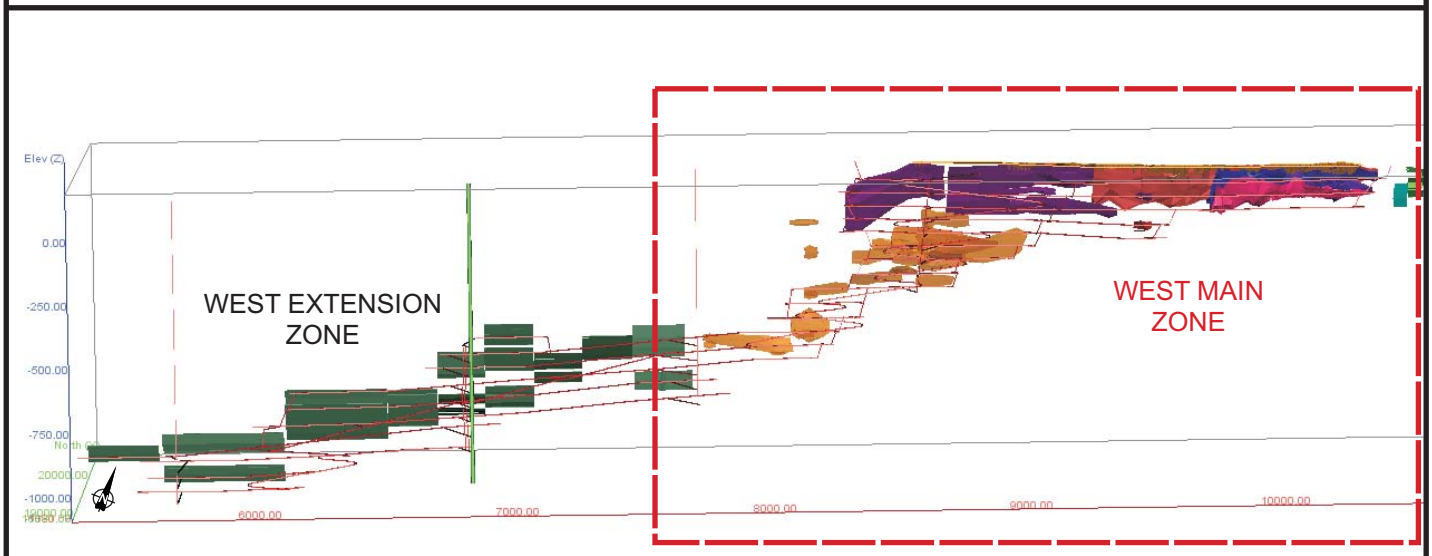


Figure 16-5

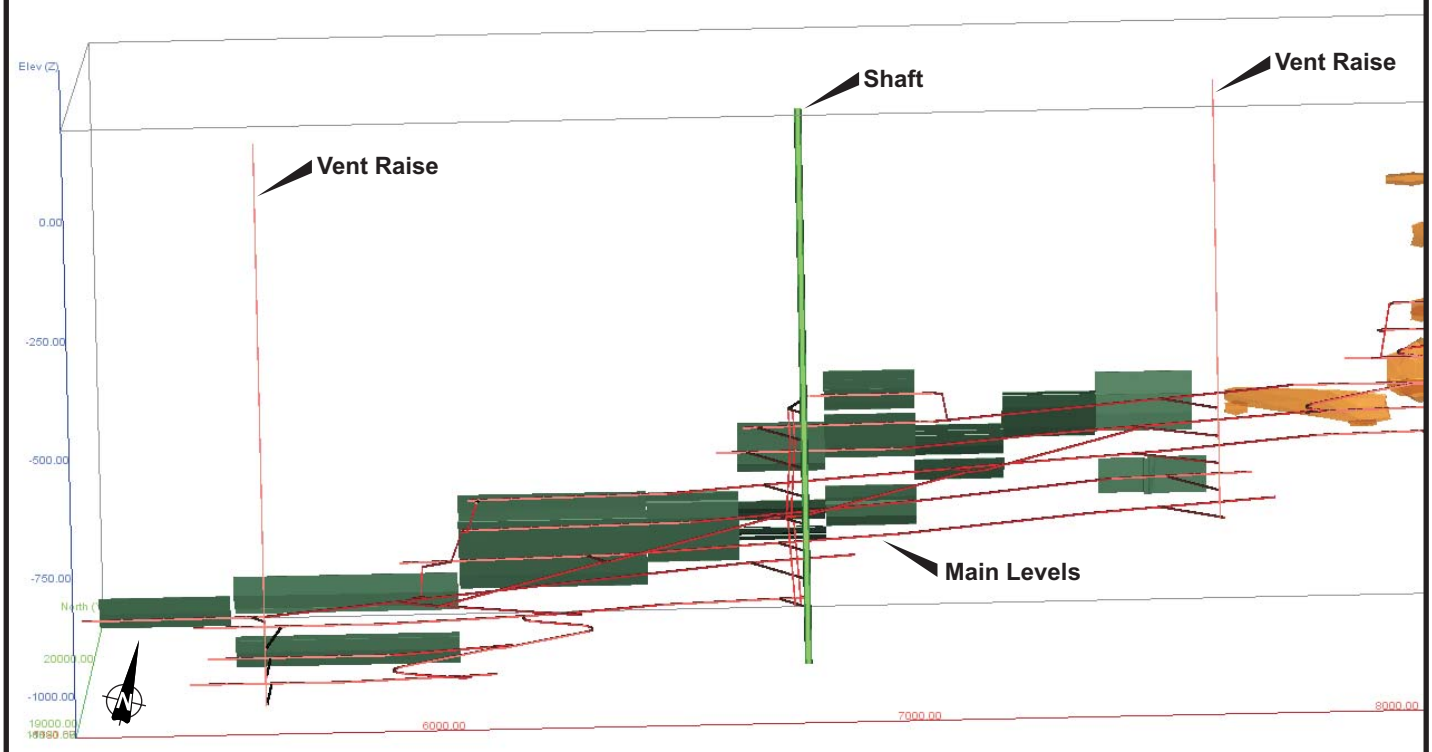
Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

West Main Zone Orebody Isometric

November 2011



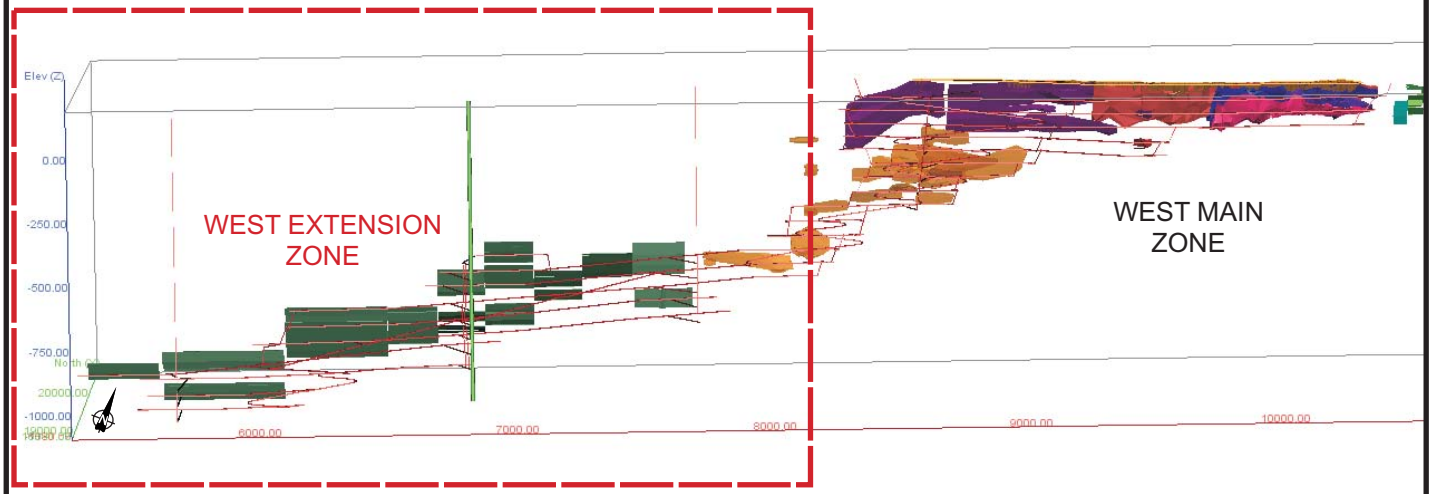


Figure 16-6

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

West Extension Orebody Isometric

November 2011

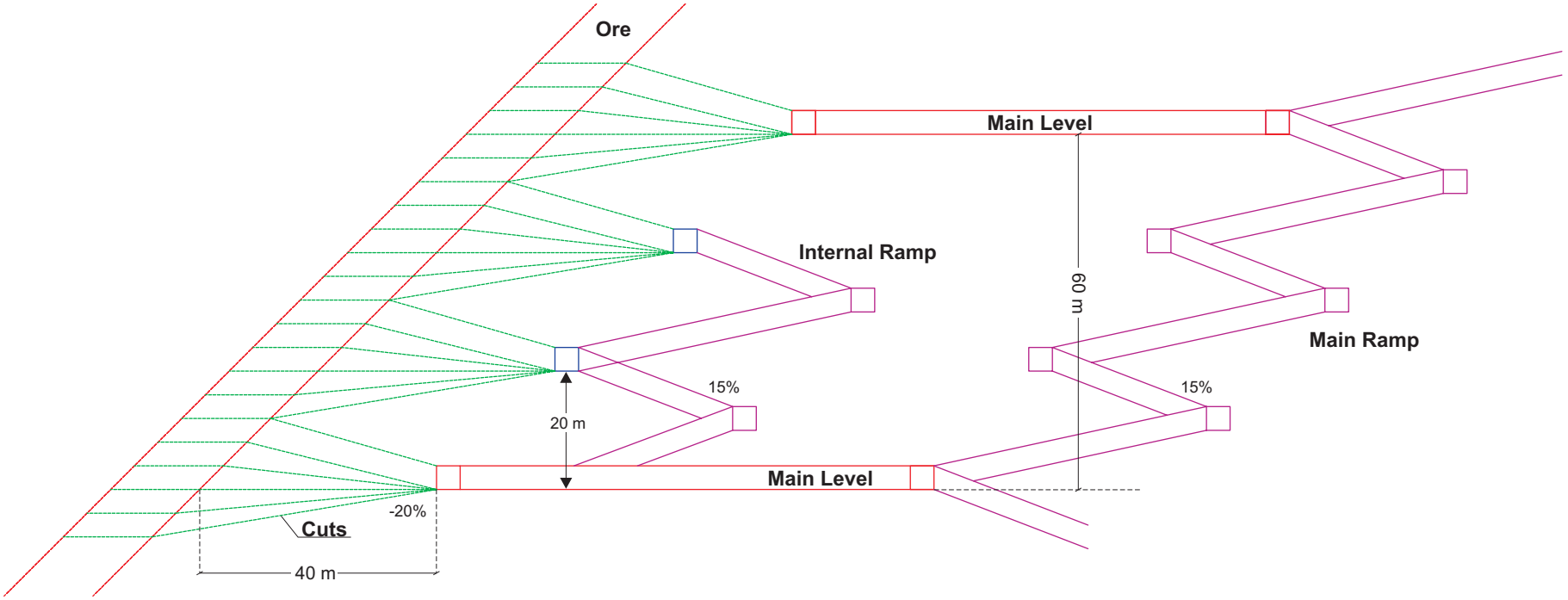


Figure 16-7

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Mine Development Schematic

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Each 60 m stoping block will be mined from the bottom cut up to the top. Initial cuts at the bottom of each block will be prepared with cemented fill, rock bolts, and wire mesh, so that the block below can be mined right to that cut, without leaving a sill pillar.

For an area with an average width of five metres, the stope cycle for a 15,000 t, four-metre-high by 200-m-long cut is estimated to be:

- Drill, blast, rockbolt and extract swell – 25 days
- Muck out after breaking – 10 days
- Backfill – 14 days
- Slash back of cross cut for next lift – 5 days
- Total of 54 days per stope cycle, or an average of 280 tpd.

The production rate for a given cut will vary with the local lens width, however, using the five metre width case as an average, 21 active cuts would be required. Allowing for delays, approximately 25 active stoping cuts would be required at any one time, to provide the planned 6,000 tpd of mill feed.

To provide enough faces for steady production, it will be necessary to operate on a number of cuts in several stoping blocks at the same time.

DILUTION

Dilution will be incurred due to:

- Irregularities along the hanging wall and footwall in both the strike and dip directions.
- Overbreak in the mining process.
- Mining the footwall side of the cuts at a steeper angle than the ore contact.
- Mining of waste bands between adjacent ore lenses, where they are too narrow to leave in place.
- Mucking of waste fill material, which forms the mucking floor.

RPA has applied 15% dilution as an initial estimate, made up of the above mentioned factors. This amount of dilution is typical for mechanized cut and fill mining operations with good geological and engineering control on each unit of ore advance, as well as

equipment matched to the size of the ore exposure on the drilling face. The drill hole spacing for much of the deposit is currently too widely spaced for detailed mine planning, however, as infill drilling is completed, available information should allow the estimate of dilution to be refined.

In addition to economic criteria (minimum NSR value), lens boundaries were defined as containing 50% or greater sulphides (i.e. massive sulphides). In terms of metal content, the lens boundaries are not sharp, and dilution is not expected to be completely barren. An allowance of 10% of average resource grade was used for dilution grades.

MINING EXTRACTION

Cut and fill mining will provide high extraction of mineralized areas down to the minimum width of four metres. More significant losses will be incurred in selection of which lenses can be mined, in those places where the lenses are stacked closely together. Initial planning was based on complete extraction along strike, including crown pillar recovery.

RPA applied an extraction factor of 95% to resource totals, to allow for the above considerations.

PRODUCTION QUANTITIES

With the mining of the open pit as outlined above, the balance of the Mineral Resources available for underground mining is shown in Table 16-7. Mineral Resources below the open pit bottom were included in the production quantities.

TABLE 16-7 UNDERGROUND PRODUCTION QUANTITIES
Starfield Resources Inc. – Ferguson Lake

	Tonnes (000)	Ni (%)	Cu (%)	Co (%)	Pt (gpt)	Pd (gpt)
Tonnage	35,306	0.66	1.06	0.08	0.27	1.65
Dilution	5,028	0.07	0.10	0.01	-	-
Subtotal	40,545	0.62	0.98	0.07	-	-
Extraction	95%					
Total	**38,545**	**0.62**	**0.98**	**0.07**	**-**	**-**

Note: Pt and Pd not considered as recoverable.

GRADE CONTROL

Grade control will be provided by geologists who will visit and inspect the work places after every round and provide guidance for mining. The geologists will also map the back and faces, for use in planning the next cut. Sampling of the face will be undertaken on a round by round basis.

Survey control lines will be painted on the walls to maintain an even level four metre lift. Similarly, fill lines will be painted on the walls to provide a reference when filling the stope and also a reference for muckers when the final stope mucking is underway. Mucking control in the final cleanup of each lift will be reviewed by geologists and supervised by the shift boss.

UNDERGROUND MINING EQUIPMENT

For the underground mines, two-boom jumbos, 30-tonne to 40-tonne trucks, and five to seven cubic metre LHDs are recommended (Table 16-8). The equipment will provide sufficient units for flexibility in completing the work while still large enough to be productive. With the current operational scenario, the mine equipment fleet will be shared approximately equally between the Main West Zone and West Extension.

A minimum mining width of four metres has been considered for the cut and fill stopes, and the haulage ways will be approximately five metres by five metres to provide ease of access and ease of ventilation.

TABLE 16-8 UNDERGROUND MINE EQUIPMENT FLEET

Starfield Resources Inc – Ferguson Lake

Number	Type	Model	Use
7	LHD	6 m^3	Production/development and fill
10	Truck	40 tonne	Production/development and fill
9	Jumbo	2-boom	Production and development
10	Rock bolt	Mechanized scissor bolter	Production and development
3	LHD	2 - 3 yd LHD	Service jobs
6	Utility	Scissor lift	Service and construction
	Utility	ANFO loader	Production and development
8	Utility	Tractors	Services/supervision/technical
4	Utility	Boom truck	Services, maintenance and construction

The larger LHDs will carry out ore, and waste development mucking and tramming, and backfill placement duties. The 40-tonne truck is recommended because the unit is compatible with the chosen LHD and is more productive than alternative 25-tonne to 30-tonne units, while requiring little additional operating space.

The jumbos are two-boom units that will be used in production and development. The units would be similar to one another to reduce the range of spare parts needed on site. The mechanized scissor bolters are suitable for one man operation. All drilling units would be equipped with their own compressor for drilling as well as dust control (dust collectors for dry drilling, or brine pumping systems).

In addition to the mobile equipment, the purchase of a raise drill unit is included for back fill raises, ventilation raises and service raises.

CRUSHING

A crusher and grinding mills will process ore at the concentrator site to generate slurry for transportation to the process site via slurry pipeline. Underground crushing has not been included in the current operating scenario, as the muck is anticipated to be fine, coming entirely from drill jumbo rounds. Grizzlies and rockbreakers have been included to handle occasional oversize pieces.

ELECTRICAL LOADS

Mine electrical power will be distributed by electrical cables to a series of small transformers to provide power for mobile equipment, fans, and other services. Initially, the mine power will be fed down the main ramp (Main West Zone) and shaft (West Extension), however, the distribution system may be modified as raises are developed and the main electrical feed can go down one or more service raises to shorten distribution lines.

Mine electrical power loads were estimated by item. The average consumption is estimated to be approximately 10 MW, the bulk of which is in surface and underground ventilation fans.

Compressed air will be supplied from onboard compressors on units that need it for drilling and from separate portable compressors for tasks such as explosives loading.

Mine shops serving the Main West Zone will be located on surface as part of a central shop, and those serving the West Extension will be located underground.

A two way radio system will be installed to provide communications around the mine. A telephone system will also be installed for communications to surface.

MINE VENTILATION

The Main West Zone will be ventilated using surface mounted exhaust fans located on raises near the mine development. The main ramp will act as an intake airway, and ventilation raises connected to the extents of the main levels will provide exhaust routes. Regulators at the raise connections will act to control the airflow. The mine design also calls for one additional intake raise system to be developed in parallel with the ramp to act as a secondary egress as the Main West Zone is developed.

The West Extension will be ventilated with the shaft as the main intake. There will be two, three metre diameter raise bore exhaust raises, both developed following shaft sinking. Intake airflow will be controlled with regulators on each level, and the raises will act as exhaust routes. Stope ventilation for both zones will be achieved with auxiliary fans and ventilation ducting to the faces.

RPA's proposed underground development incorporates fresh and return air raise systems to surface, which are connected to most of the levels. While providing ventilation, the raises can also serve as waste passes to support the underground backfill system. As such, depending on where mining is taking place at a given time, these raises can serve two purposes and provide additional flexibility to the operation.

DEWATERING

Preliminary indications from drilling imply that permafrost conditions extend approximately 300 m below surface. Within this plan, it has been assumed that the Main West Zone portion of the mine will be entirely within permafrost, and will be maintained in a frozen state. No groundwater sources are expected, and the mine is assumed to remain almost entirely dry, with the exception of decant water from tailings backfill.

Allowances for conventional sump and pumping systems have been included for both the Main West and West Extension zones.

Additional hydrological investigation is required to confirm groundwater assumptions.

BACKFILL

Rock fill will be available from the waste generated in the open pit, from underground development waste, and from surface quarrying of till and rock. The concentrator will also provide tailings that will be used for backfill purposes. The surface waste would be delivered to mining areas by boreholes from surface or by return trucking following delivery of ore to the surface ore pad. On an ongoing basis, there would be a requirement to fill a void of approximately 2,000 m^3/day. The open pit waste would provide material for approximately six years of backfilling (with no allowance for mine development waste used as fill).

17 RECOVERY METHODS

Recovery methods for the Project have been developed by Michael J. Dry, Ph. D. of Arithmetek, J. P. Evans, P. Eng., of ACSI, and Ernest Burga, P. Eng. of ACSI.

SUMMARY

Since 2001, metallurgical testwork has been carried out to determine the potential for efficient metallurgical extraction of base and precious metals from the mineralized material within the Main West Zone and West Extension. The hydrometallurgical process proposed for the previous PEA was developed by Dr. Bryn Harris and Mr. Carl White. Given the site location, the process was designed to minimize environmental effects, be highly energy efficient, and provide high base metal recoveries. Products included LME-grade nickel, copper, and cobalt metal, hematite, sulphuric acid, gangue material containing platinum and palladium, and power. Since then, the process has evolved but is still the same in the overall context of minimizing the environmental impact and retaining the same product suite for the current PEA.

A simplified process diagram showing the main processing units is shown in Figure 17-1. The process is based on the following unit operations:

- The run of mine (ROM) ore will be crushed and ground to a relatively coarse grind (P_{90} of 100 mesh, or 150 µm) at the mine site.
- Ground ore will be subjected to flotation to recover essentially all the sulphide minerals and precious metals into a bulk concentrate, while rejecting the bulk of the gangue minerals.
- The bulk sulphide concentrate will be slurried in water, transported by pipeline to the process site, and dewatered by filtration.
- Primary leaching of the ground ore will be carried out with hydrochloric acid (HCl) at a temperature of approximately 115°C, producing a ferrous chloride ($FeCl_2$) solution, hydrogen sulphide (H_2S) off gas, and a residue containing the base and precious metals.
- The ferrous chloride ($FeCl_2$) solution will be concentrated by evaporation at atmospheric pressure, then oxidized to initially precipitate one-third of the iron as hematite, and generate a ferric chloride ($FeCl_3$) solution that will be further treated to recover HCl for recycle to the primary leach, and to precipitate the iron as hematite.

- The hydrogen sulphide (H_2S) off-gas from the primary leach will be cooled to condense out most of the water and any residual HCl, prior to burning in air to generate high-strength sulphur dioxide (SO_2), which will be sent to an acid plant to produce 93% sulphuric acid. This will enable the recovery of a substantial amount of the intrinsic energy in the ore as high pressure steam, for use both directly in the process and also for electric power generation.

- Most of the solution from the primary leach will be combined with solutions from the next three steps and concentrated by boiling off water at atmospheric pressure, then oxidized with oxygen at ambient pressure to convert one third of the ferrous chloride to hematite and the balance to ferric chloride.

- The solids and some of the solution from the primary leach will be treated in a secondary leach with oxygen to dissolve the base metal sulphides and convert most of the iron to goethite, a small part remaining as dissolved ferric chloride. The sulphur in the incoming sulphide minerals will be converted to elemental sulphur. The slurry from the secondary leach will be filtered and the filter cake washed with water. The wash filtrate will be sent to the preceding step.

- The main filtrate from the secondary leach will be acidified with recycled hydrochloric acid, cooled and sent to a solvent extraction stage in which the ferric chloride and most of the free hydrochloric acid will be extracted into the organic phase, leaving the divalent metals in the raffinate. The organic phase will be stripped with water and the resulting solution of ferric chloride and hydrochloric acid will be recycled to the evaporation step just ahead of the oxidation step.

- The washed filter cake from the secondary leach will be digested in recycled strong hydrochloric acid to re-dissolve the goethite, producing a concentrated solution of ferric chloride and leaving the precious metals in a final residue of gangue material and sulphur. The resulting solution will be sent to the above evaporation and oxidation step.

- The concentrated and oxidized solutions from the primary leach, the secondary leach, the goethite re-dissolution step and the loaded strip solution from the solvent extraction of ferric chloride will be heated further, causing the ferric chloride to decompose to solid hematite and gaseous hydrochloric acid and steam (hydrolysis). The hydrochloric acid and steam will be recycled to the various steps using acid.

- The slurry from the hydrolysis step will be filtered hot in a specialized filter and the filtrate returned to either the oxidation step or the beginning of the hydrolysis step to control the solids content of the slurry in the hydrolysis step.

- The filter cake from the hydrolysis step will be quenched and diluted with recycled solutions from downstream in the process. The diluted slurry will be

thickened, filtered and washed with water. The washed filter cake will leave the circuit as by-product hematite.

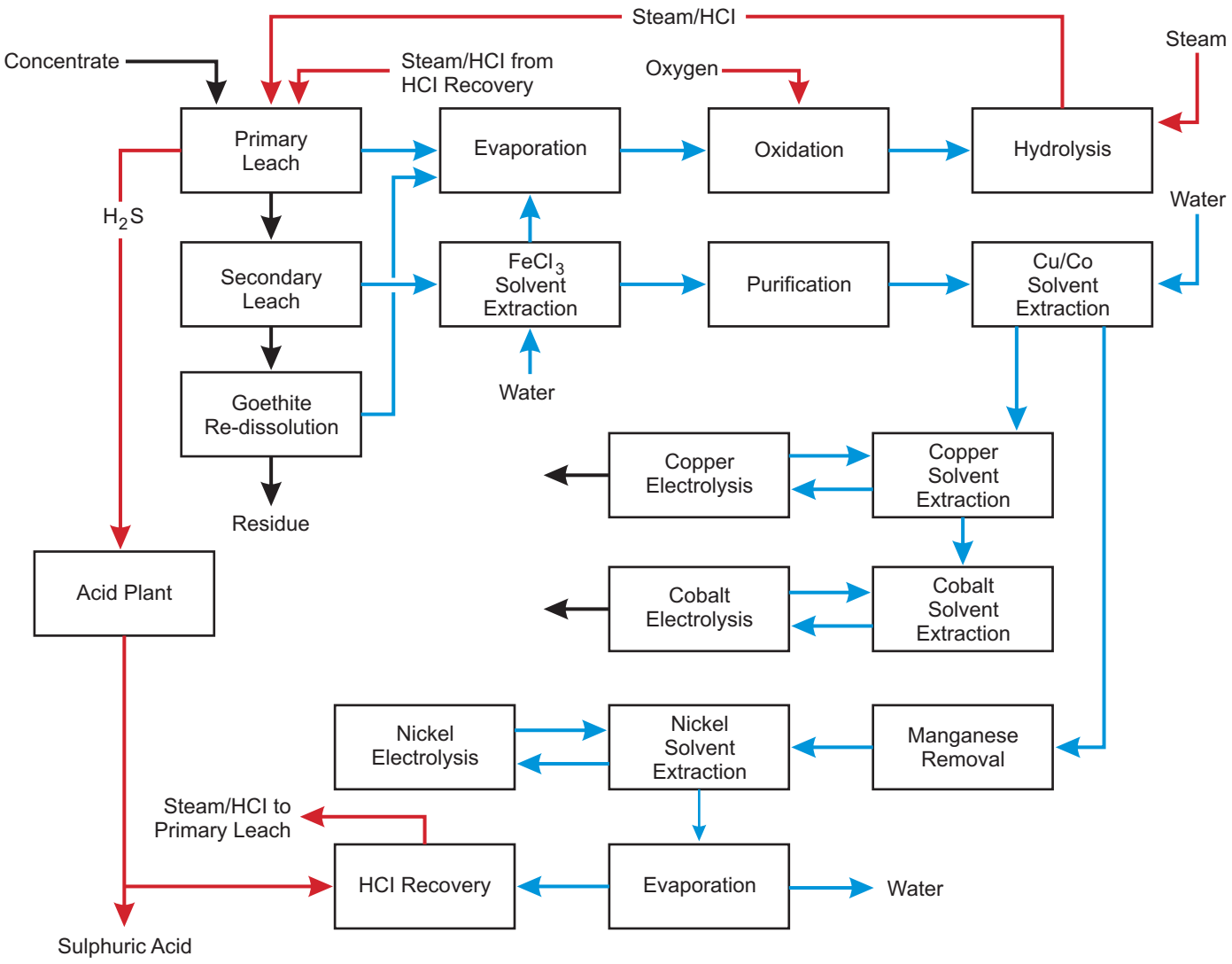
- Zinc will be removed from the raffinate ex the solvent extraction of ferric chloride, using ion exchange. The loaded resin will be stripped with water and the resultant zinc strip solution is combined with the raffinate from the downstream recovery of nickel and evaporated at reduced pressure to recover water for re-use in the circuit. The remaining concentrated solution of mainly magnesium and zinc chloride will be sent to a downstream section in which the chloride will be recovered as hydrochloric acid for recycle.

- Copper and cobalt recovery from the secondary leach solution will occur through anionic solvent extraction (SX). A second stage of solvent extraction will then separate the copper from the cobalt, and each metal will be extracted into a sulphate solution using SX and electrowon from the sulphate medium to generate LME grade metal.

- Electrolysis will be used to remove manganese as manganese dioxide from the raffinate after the anionic SX of cobalt and copper. The manganese dioxide will be discarded and the resulting solution neutralized with lime. The nickel will be recovered by SX into a sulphate solution, and electrowon from the sulphate solution to produce LME grade metal.

- The raffinate from the nickel SX step will be combined with the strip solution from the zinc ion exchange step and concentrated by evaporation at reduced pressure to recover process water.

- The remaining solution will be mixed with concentrated sulphuric acid from the acid plant and then boiled in a stripper column to remove the chloride as gaseous hydrogen chloride in steam. The steam and hydrogen chloride will be recycled and the resulting solution of mainly magnesium sulphate will be discarded.



Legend:
- *Solid*
- *Liquid*
- *Vapor*

Figure 17-1

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada
Simplified Process Diagram

November 2011

Source: ACSI, 2011.

PROCESS DESIGN SUMMARY

In addition to addressing process development, ACSI has produced an updated capital and operating cost estimate with an accuracy of +/- 40% for the Project corresponding to a scoping level study. A detailed basis of estimate for process capital and operating costs is provided in Appendix 1, and are summarized in Section 21.

The process facilities evaluated in ASCI's work include: mine site concentrator, sulphide concentrate pipeline to transfer concentrate, and the hydrometallurgical processing facilities using the HCl based technology.

The concentrator is configured to receive run of mine ore. The ore will be crushed, ground, and treated by flotation producing a bulk sulphide concentrate for transfer to the Arviat processing facility. The concentrator will produce tailings that will require disposal at the Project site, with a portion being disposed of as underground backfill.

The process design assumes a 285 km pipeline will deliver concentrate to Arviat where it will be filtered ahead of the process plant. The process plant will treat concentrate to recover valuable metals as well as producing energy by using steam from an acid plant. Excess power will be distributed for use at the concentrator site, potentially making the whole operation energy self-sufficient.

The mining rate considered in the design is 6,000 tpd of ore from which the concentrator will produce approximately 3,300 tpd of bulk sulphide concentrate. Each year, the plant will produce the following products (approximate quantities, rounded up):

- 12,000 tonnes of LME-grade nickel cathode
- 19,000 tonnes of LME-grade copper cathode
- 1,300 tonnes of LME-grade cobalt cathode

Each year, the plant will also produce approximately 850,000 tonnes of H_2SO_4 (net of reagent use), 735,000 tonnes of hematite, and 234,000 tonnes of leach residue containing PGE metals.

Process capital costs were prepared by ASCI and have been reviewed by RPA. They have been estimated to a level consistent with a scoping study, and at an accuracy of 40%. In summary, process capital costs consist of the following:

- Total equipment cost of US$658 million
- Installed capital cost of US$788 million, excluding indirect costs
- Total direct capital cost of US$1,034 million

Process operating costs have been estimated by ASCI and reviewed by RPA. Unit processing costs are estimated at US$25.31 per tonne milled.

In the process plant, the primary leach produces H_2S. It is proposed to burn the gas in air and convert the resultant SO_2 to H_2SO_4 in a Haldor Topsoe acid plant. This type of plant is more expensive than other types of acid plants, but maximizes the amount of energy recovered from the exothermic reactions. In order to increase the power production, sulphur will be recovered from the leach residue and will be burnt to SO_2 in the acid plant. The PEA assumes that acid will be stored, shipped, and sold. The acid will be stored for seven months and shipped during the 15 week summer shipping season. Some of the acid will be used in the HCl regeneration step.

The high-pressure steam generated in the acid plant will be expanded through turbines to generate power and the resulting lower pressure steam will be used in the evaporation, oxidation, and hydrolysis stages to maximize the efficient use of recovered energy.

Arithmetek produced the mass/energy balance upon which the equipment sizing is based. SGS Lakefield performed the experimental work upon which the mass/energy balance is based. ACSI used the balances and the data from SGS to size equipment and generate capital and operating cost estimates. Each piece of equipment has been sized and evaluated for suitability to the application. This has been necessary given of the novelty of the process and because of the aggressive nature of the liquids, solids, and gases.

Additional test work should be carried out to confirm operating parameters. The equipment to be used in the test plant should be made of material pre-selected for use in

the main plant. The results from the tests should be used in continuing engineering studies and to solve materials of construction concerns.

The preliminary plant layout outlines the proposed process plant configuration including all process stages and supporting plants. The total area required for the process facilities is approximately 170 m x 380 m and the acid storage facility with 27 tanks will require additional area of approximately 190 m x 520 m.

PROCESS DEVELOPMENT

Given that the Project has limited existing infrastructure, the conceptual design for the treatment plant has been prepared to be energy efficient to offset the high cost of power generation using fossil fuels. Starfield, SGS, and Arithmetic have developed a chloride-based hydrometallurgical process to treat the concentrate. Hydrochloric acid (HCl) reacts with pyrrhotite minerals to release H_2S which can be burnt with air to exothermically produce sulphuric acid and energy.

Soluble iron is oxidized with oxygen to form ferric chloride, which is hydrolyzed to form volatile HCl and stable hematite, which will be stored at the treatment plant site. The base metal sulphides are dissolved in a secondary, oxidizing leach.

Base metals will be recovered from the pregnant solution by solvent extraction (SX) and electrowinning (EW) to produce high purity metals to minimize shipping costs.

A low-grade, precious metal concentrate will be produced as a leach residue since these metals are insoluble. The removal of sulphur from this residue by flotation is proposed. The recovered sulphur will be burnt in the acid plant.

On-site energy generation from process gas waste heat boilers is critical to the viability of the Project. Importing fossils fuels is not only very expensive, but also generates a considerable volume of carbon dioxide, which would be emitted to the atmosphere.

ACSI has been employed to assist RPA and Starfield with engineering and costing of the processing facilities.

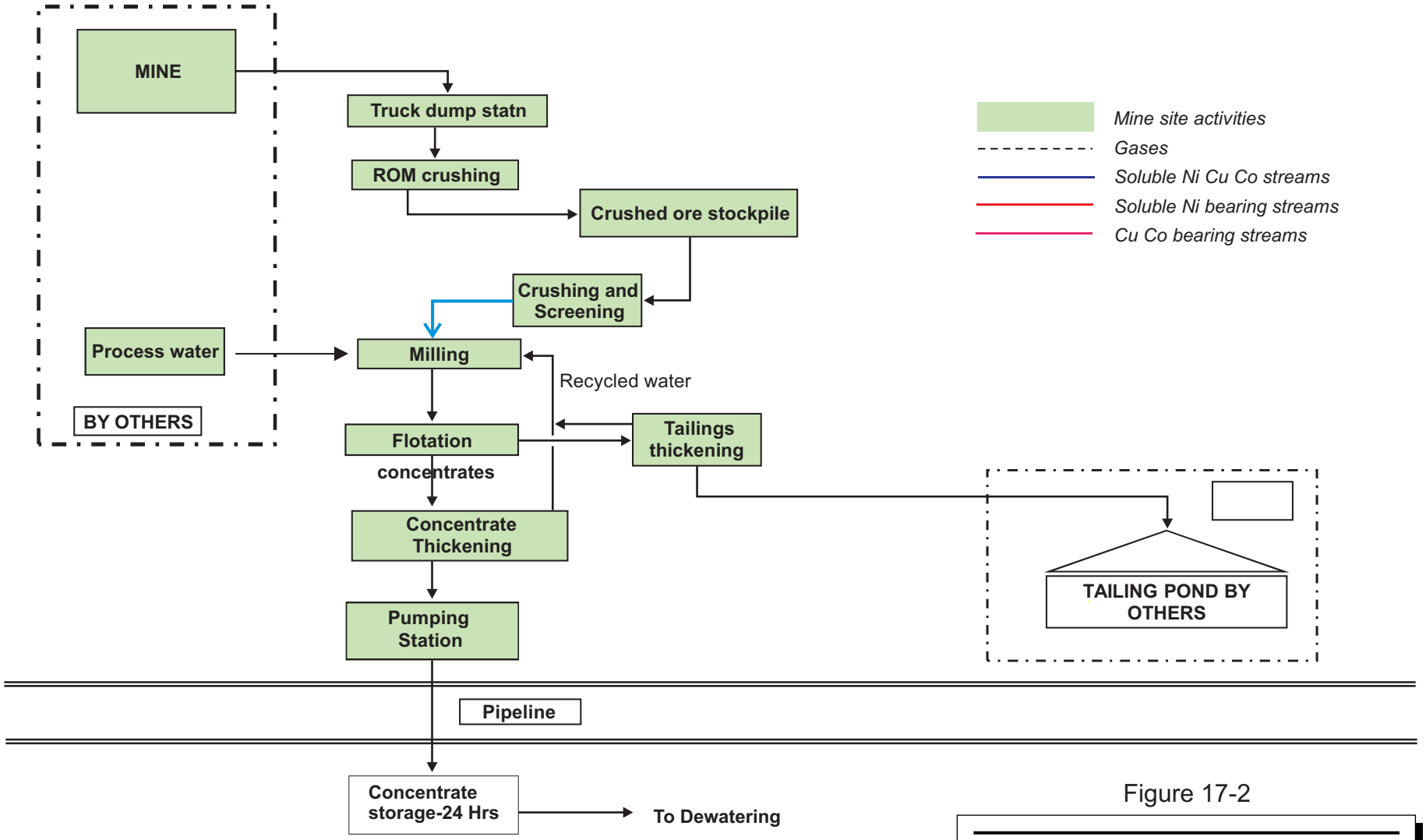


Figure 17-2

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Process Block Flow Diagram Concentrate

November 2011

Source: ACSI, 2011.

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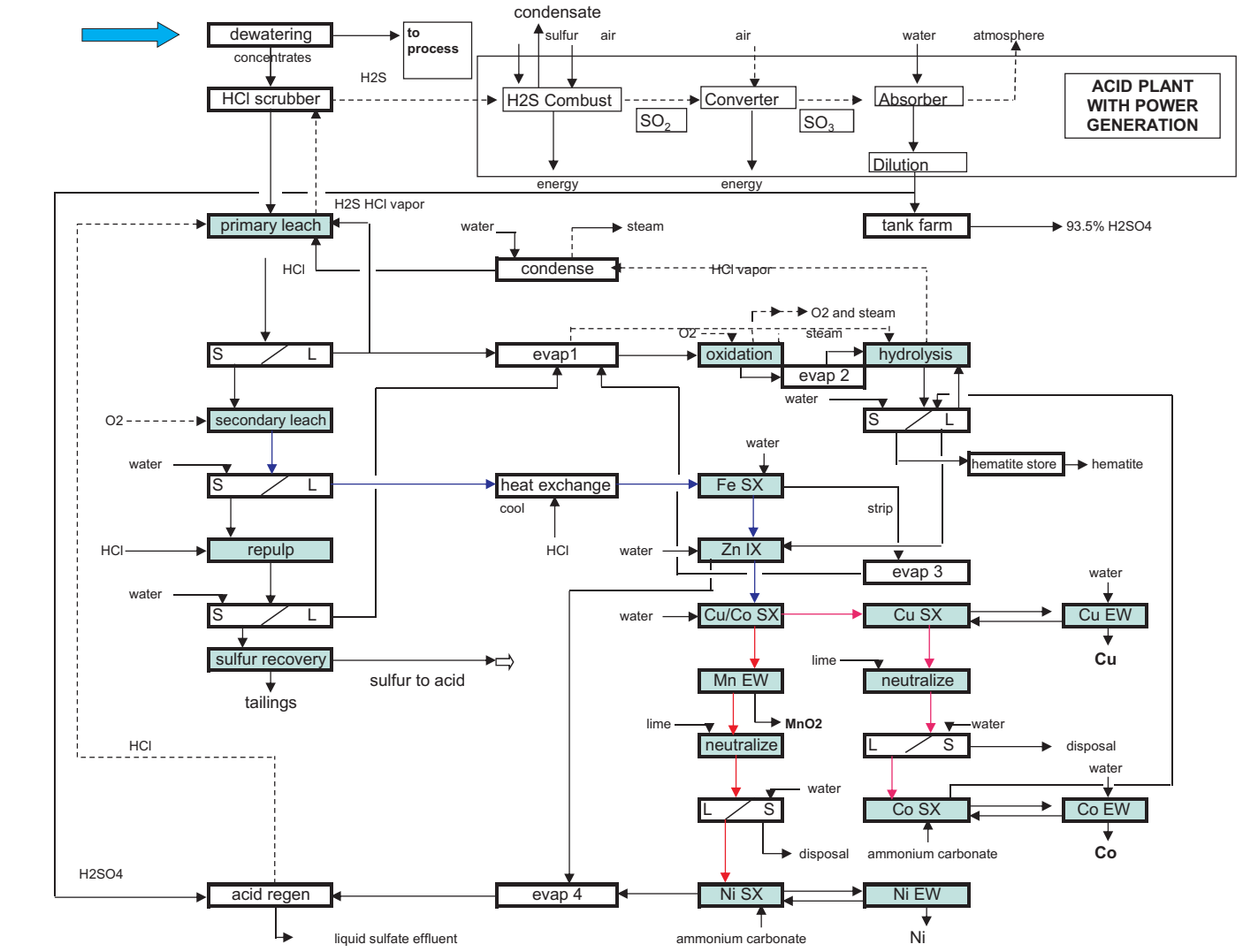

Figure 17-3

Gases
Soluble Ni Cu Co streams
Soluble Ni bearing streams
Cu Co bearing streams

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Block Flow Diagram
Hydrometallurgical Process Plant

November 2011

Source: ACSI, 2011.

ORE COMPOSITION AND PRODUCTION

Ore will be mined, crushed, and wet ground with water from Ferguson Lake, and subjected to flotation. Concentrates will be pumped via a slurry pipeline to the dewatering section of the process plant. Mill tailings will be stored in a tailings management facility (TMF) at the mine site as well as underground.

Average, LOMP feed grades are expected to be 0.61% Ni, 0.98% Cu, 0.07% Co, 36.32% Fe, and 19.98% S.

PROCESS APPLICATION

The process uses leaching at high total chloride concentration to convert the sulphide minerals to dissolved chlorides, hydrogen sulphide, and elemental sulphur. The resulting solutions are concentrated by evaporation, oxidized and heated to convert the dissolved ferrous chloride to solid hematite (Fe_2O_3) and gaseous hydrogen chloride (HCl). The HCl is recycled and the Fe_2O_3 leaves as a by-product or a benign waste, depending on market conditions for iron ores.

The main source of energy in the process is high pressure steam generated in the acid plant, where the H_2S and elemental sulphur generated in the leach steps is converted to sulphuric acid. The high pressure steam is expanded to 28 Bar to generate power, and the 28 Bar steam is condensed to provide the energy needed in the hydrolysis section. Residual 28 Bar steam is expanded to 10 Bar to raise more power and the 10 Bar steam is used to drive evaporation in the process. Energy recovery in the process is achieved by condensing recycled HCl/steam and raising pure steam at atmospheric pressure in heat exchangers. The residual 10 Bar steam is expanded to atmospheric pressure and the various high pressure condensates are flashed to atmospheric pressure. Some of the resulting atmospheric pressure steam is used to drive the lower-temperature evaporation steps in the process and the balance is used to generate more power in a condensing turbine.

The solution from the secondary leach is purified by solvent extraction of residual ferric chloride, then by ion exchange to remove zinc chloride. The purified solution is subjected to further solvent extraction in stages to separate out the individual base

metals. The individual base metals are recovered as pure metal products by electrolysis.

PROCESS DESCRIPTION

FILTRATION PLANT

There are three concentrate storage tanks, each with a storage capacity of eight hours. The filters are tower type for semi-continuous concentrate filtration.

PRIMARY LEACH STAGE (FIGURE 17-4)

Slurried concentrate is transported from the mine site by pipeline into the filtration plant. The filter cake is slurried and transported into the primary leach using ferrous chloride solution recycled from the primary leach thickener. Leaching takes place in four, cascading reactors. Slurry from the last reactor is thickened and filtered. Some of the thickener overflow is recycled to slurry the concentrate filter cake and the remainder goes to the oxidation stage. The filter cake goes to the secondary leach stage. Residence time in each reactor is about 30 minutes.

Pyrrhotite reacts with HCl to form $FeCl_2$ and H_2S. The H_2S is transferred to the acid plant to produce sulphuric acid.

The reactions between air and H_2S, SO_2 and O_2 and SO_3 and H_2O are exothermic and a significant amount of energy is recovered in waste heat boilers in the acid plant.

Base metals and precious metals do not react appreciably with HCl under reducing conditions.

The primary leach will be heated by recycled concentrated HCl and recycled brine. Boiler feed water in heat exchangers condenses most of the recycled HCl gas. Sufficient gaseous HCl remains to close the heat balance around the primary leach.

The H_2S/H_2O gases leaving the primary leach carry with them some HCl gas. This is removed in a scrubber using the incoming concentrate.

Most of the H_2O vapor in the off gasses is removed by cooling and condensation. Materials of construction for this equipment need more evaluation.

Care will be needed in designing the primary leach stage because H_2S is very toxic and mixtures of H_2S and air are explosive between 4% and 46% H_2S.

The main chemical reaction for this stage is:

$$Fe_7S_8+14HCl=7FeCl_2+7H_2S+S$$

OXIDATION STAGE (FIGURE 17-5)

The solution from primary leach is concentrated and heated in an evaporation stage. $FeCl_2$ in the liquid from evaporation is oxidized to $FeCl_3$ and hematite using oxygen in 3 cascading reactors. The reactions are exothermic and the temperature is held to 120°C by evaporation of boiler feed water. One third of the iron precipitates as hematite. Each reactor has a residence time of about 37 minutes.

The main chemical reaction for this stage is:

$$12FeCl_2+3O_2=2Fe_2O_3+8FeCl_3$$

The slurry proceeds to hydrolysis.

A second evaporation stage further concentrates and heats the slurry. High pressure steam from the acid plant waste heat boiler is passed through heating coils to evaporate water needed for the hydrolysis stage. Condensate is returned to the waste heat boiler.

HYDROLYSIS STAGE (FIGURE 17-6)

The temperature of the slurry is raised to 180°C by condensing 28 Bar steam in coils.

Hematite is precipitated and concentrated HCl vaporizes in three reactors. The vapor is condensed by boiler feed water in a heat exchanger, generating atmospheric pressure steam that is used to evaporate water elsewhere in the process. Condensed concentrated HCl goes to primary leach where it reacts with pyrrhotite.

The main chemical reaction for this stage is:

$$2FeCl_3+3H_2O=Fe_2O_3+6HCl$$

A high temperature filter separates the hematite from the liquid. The filtrate is recycled to hydrolysis or oxidation. The liquid is more a fused salt containing water than an aqueous salt solution. The filter cake is re-pulped with the raffinate from cobalt SX. The slurry is filtered and the hematite is stored prior to shipment. The filtrate is returned to the evaporation step ahead of oxidation.

The materials of construction for the reactors, heat exchangers and hot filter need to be further investigated.

SECOND STAGE LEACH (FIGURE 17-7)

Primary leach underflow is treated with oxygen in a low pressure autoclave to oxidize ferrous to ferric, to dissolve the base metals and to precipitate most of the soluble iron as goethite. A residence time of five hours in the autoclave is assumed. Precipitation of goethite minimizes the amount of $FeCl_3$ going to Fe SX and allows the soluble base metals to proceed to base metal recovery without going through hydrolysis.

The main chemical reactions in this stage are as follows:

$$CuFeS_2+4FeCl_3=CuCl_2+5FeCl_2+2S$$
$$NiS+2FeCl_3=NiCl_2+2FeCl_2+S$$
$$CoS+2FeCl_3=CoCl_2+2FeCl_2+S$$
$$FeCl_3+2H_2O=FeOOH+3HCl$$

Slurry leaving the secondary leach is thickened and filtered on a belt filter. The filter cake is washed with condensate. The filter cake is repulped with recycled concentrated HCl to dissolve the goethite. The filtrate goes to the evaporation step ahead of oxidation. The filter cake is re-pulped and elemental sulphur is recovered by flotation. The leach residue contains the PGEs and will be stored until a treatment process is available.

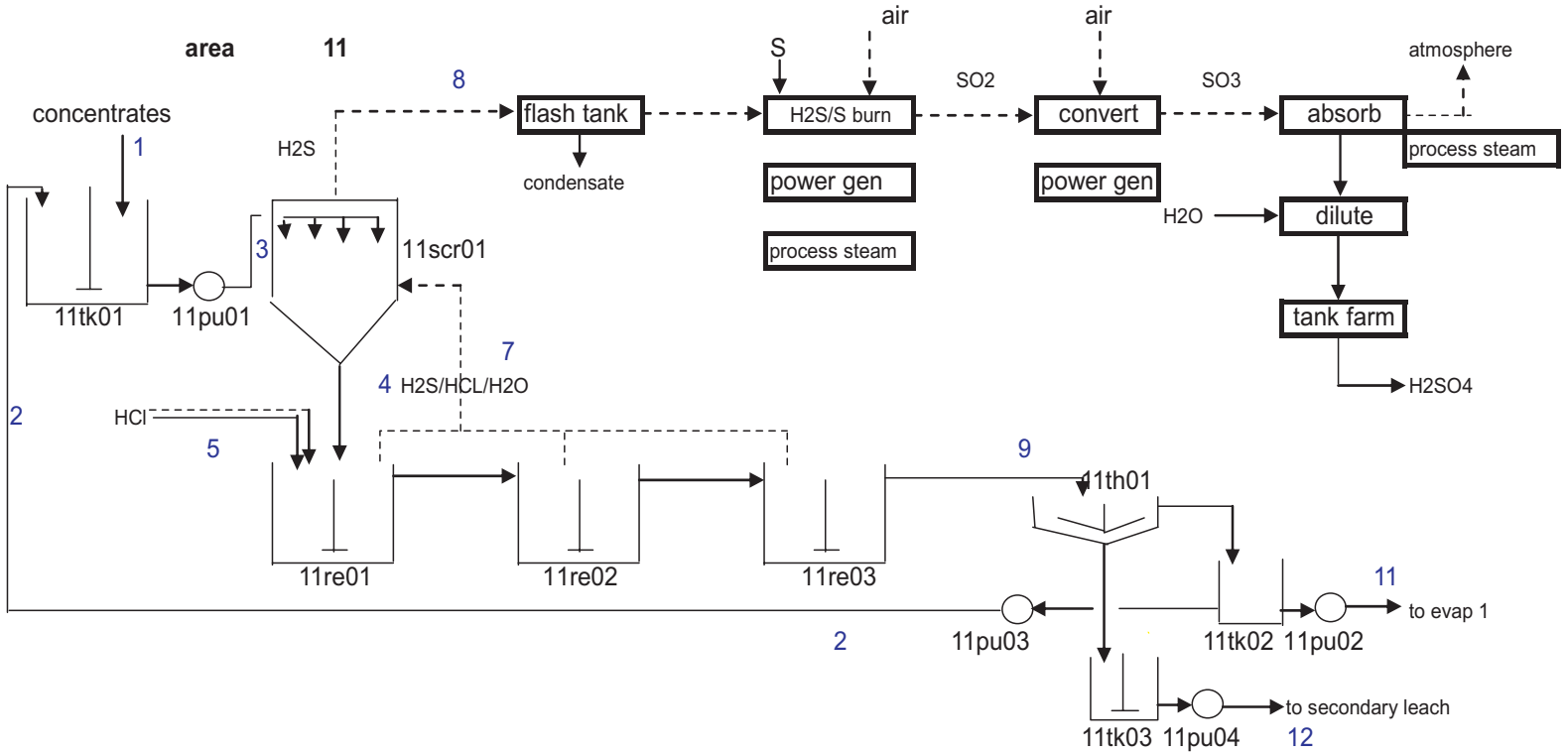

Figure 17-4

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Primary Leach PFD

November 2011

Source: ACSI, 2011.

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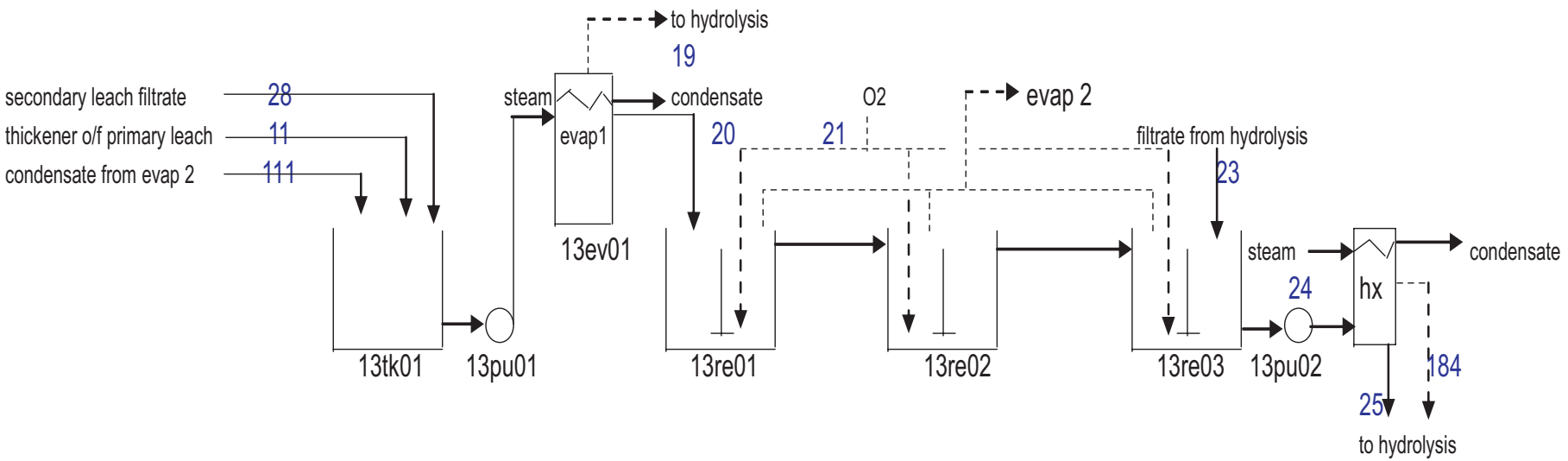

Figure 17-5

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Oxidation Stage PFD

November 2011

Source: ACSI, 2011.

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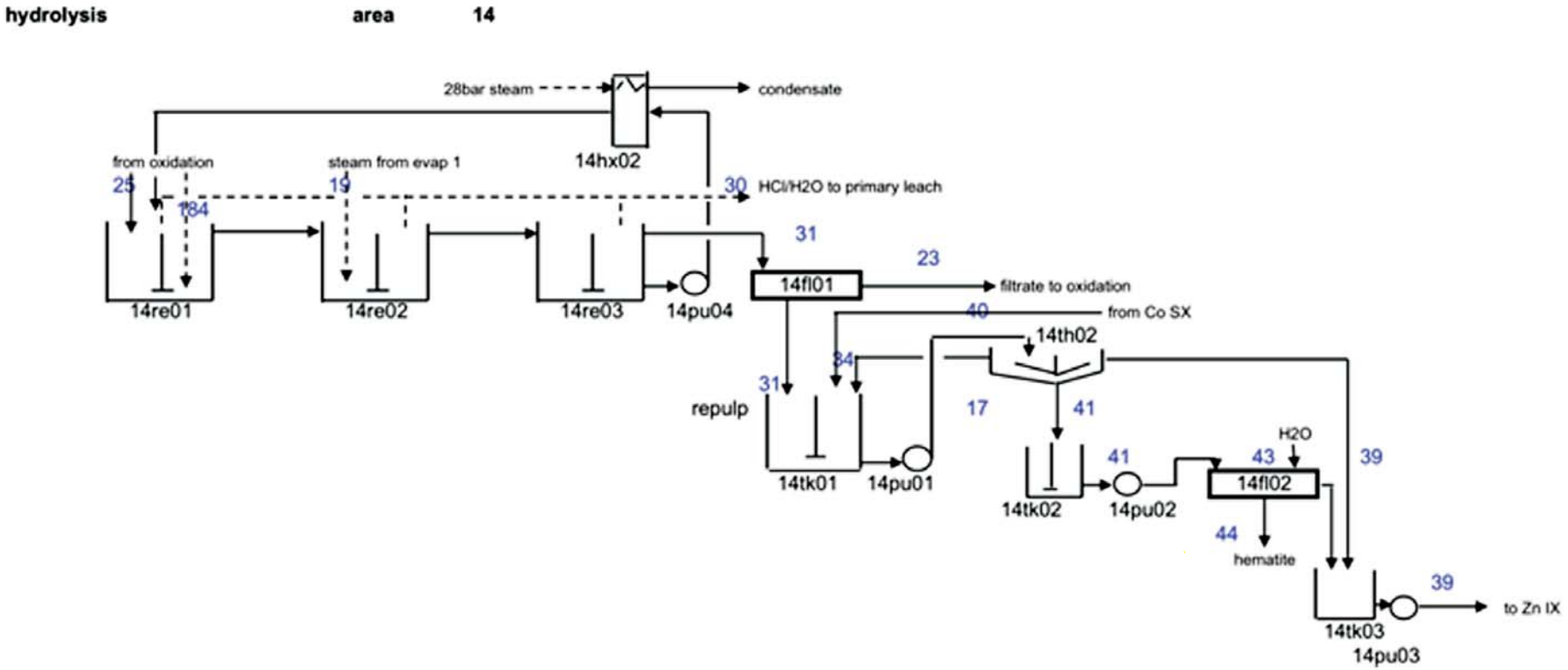

Figure 17-6

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Hydrolysis PFD

November 2011

Source: ACSI, 2011.

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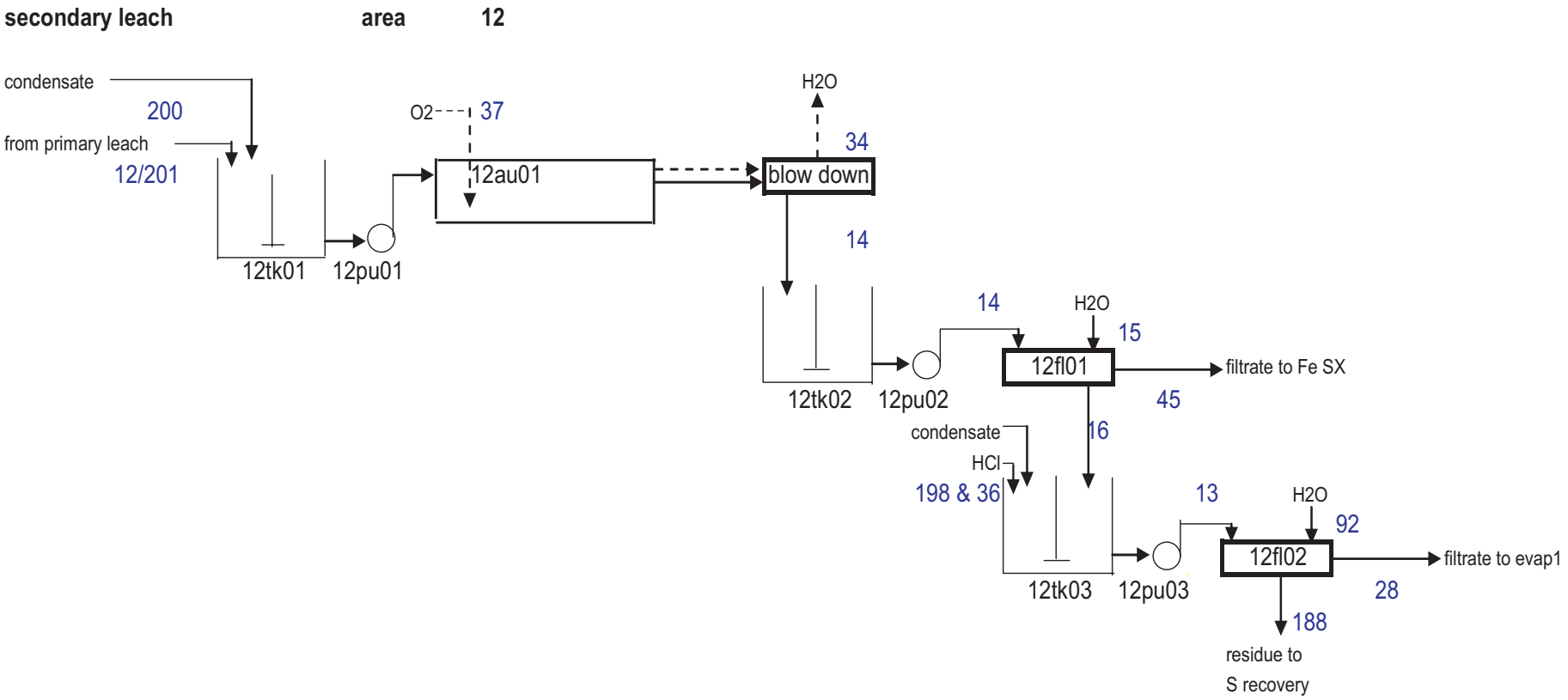

Figure 17-7

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Secondary Leach PFD

November 2011

Source: ACSI, 2011.

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ANION SOLVENT EXTRACTION STAGE

The raffinate from $FeCl_3$ SX goes to Zn IX. The resin is stripped with water and the eluate is sent to HCL recovery. The raffinate from Zn IX goes to anionic Cu/Co. The copper and cobalt are present as anions (e.g.$CuCl_2^{2-}$) and are extracted leaving the nickel in the raffinate.

The loaded Cu/Co solvent is stripped with water and the resulting loaded strip solution is fed to the copper SX stage.

The raffinate from the anionic SX step proceeds to MnO_2 EW.

COPPER SOLVENT EXTRACTION STAGE

The eluate from the anionic SX is contacted with a copper specific organic solvent. The loaded organic phase is stripped with sulphuric acid in the spent electrolyte from the subsequent electrowinning step.

The loaded strip solution from the cationic solvent containing $CuSO_4$ proceeds to EW where copper cathodes are produced in 37 EW cells. H_2SO_4 is regenerated.

$$2CuSO_4+2H_2O=2Cu+2H_2SO_4+O_2$$

COBALT SOLVENT EXTRACTION STAGE

Raffinate from the copper SX step goes to Co SX. Cobalt is loaded onto a cobalt specific organic solvent. The loaded solvent is stripped with sulphuric acid in spent electrolyte from the subsequent cobalt EW step.

The loaded strip solution from the cobalt SX step contains $CoSO_4$ and proceeds to EW where cobalt cathodes are produced in eight EW cells. H_2SO_4 is regenerated.

$$2CoSO_4+2H_2O=2Co+2H_2SO_4+O_2$$

The raffinate from Co SX is used to quench and dilute the hot hematite filter cake from the hydrolysis section.

NICKEL SOLVENT EXTRACTION STAGE

Nickel is present in the solution leaving secondary leach as a cation. It passes through the anion SX mixer settlers without loading. It is recovered from the solution leaving MnO_2 EW by a nickel specific cationic organic solvent. The loaded organic phase is stripped with sulphuric acid in the spent electrolyte from the subsequent nickel EW step. The nickel solution proceeds to EW where nickel cathodes are produced in 42 EW cells. H_2SO_4 is regenerated.

$$2NiSO_4+2H_2O=2Ni+2H_2SO_4+O_2$$

The raffinate from Ni SX goes to evaporation and then to HCl recovery. Concentrated H_2SO_4 is mixed with the raffinate and HCl vapor is released. Soluble metal sulphates such as $MgSO_4$ and Na_2SO_4 are formed. They become a liquid effluent.

COPPER ELECTROWINNING STAGE

Eluate from Cu SX goes to 37 EW cells each containing 85 – 1 m by 1 m stainless steel cathodes. The current density will be 300 amps/m^2. Cathodes will be stripped in an automatic stripping machine.

NICKEL ELECTROWINNING STAGE

Eluate from Ni SX goes to 42 EW cells each containing 50 – 1 m by 1 m stainless steel cathodes. Cathodes will be stripped in an automatic stripping machine. The anodes will have cloth diaphragms to control conditions around the cathodes.

COBALT ELECTROWINNING STAGE

Eluate from Co SX goes to eight EW cells each containing three - 1 m by 1 m stainless steel cathodes. Cathodes will be stripped in an automatic stripping machine. The anodes will have cloth diaphragms to control conditions around the cathodes.

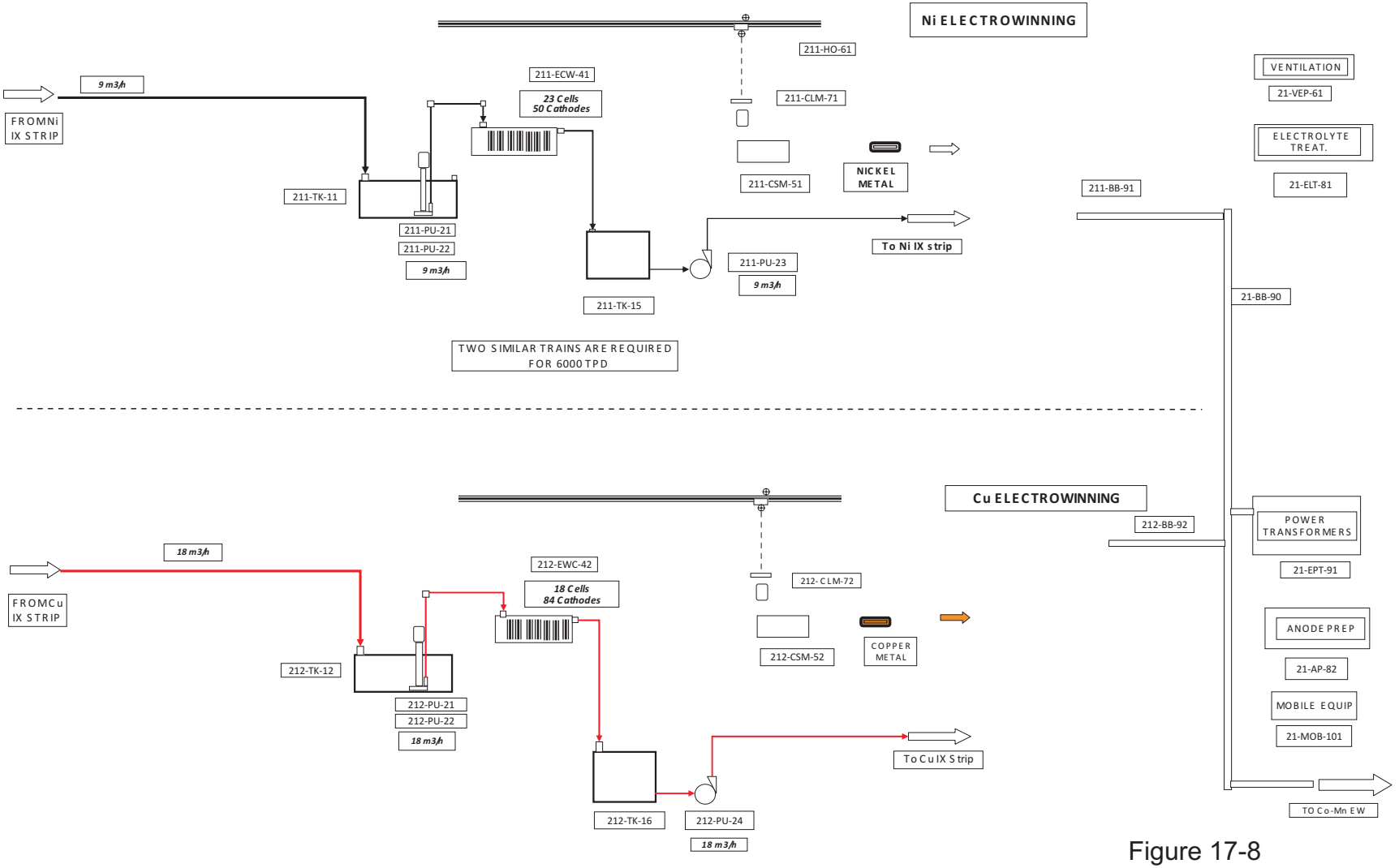

Figure 17-8

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Copper EW PFD

Source: ACSI, 2011.

www.rpacan.com

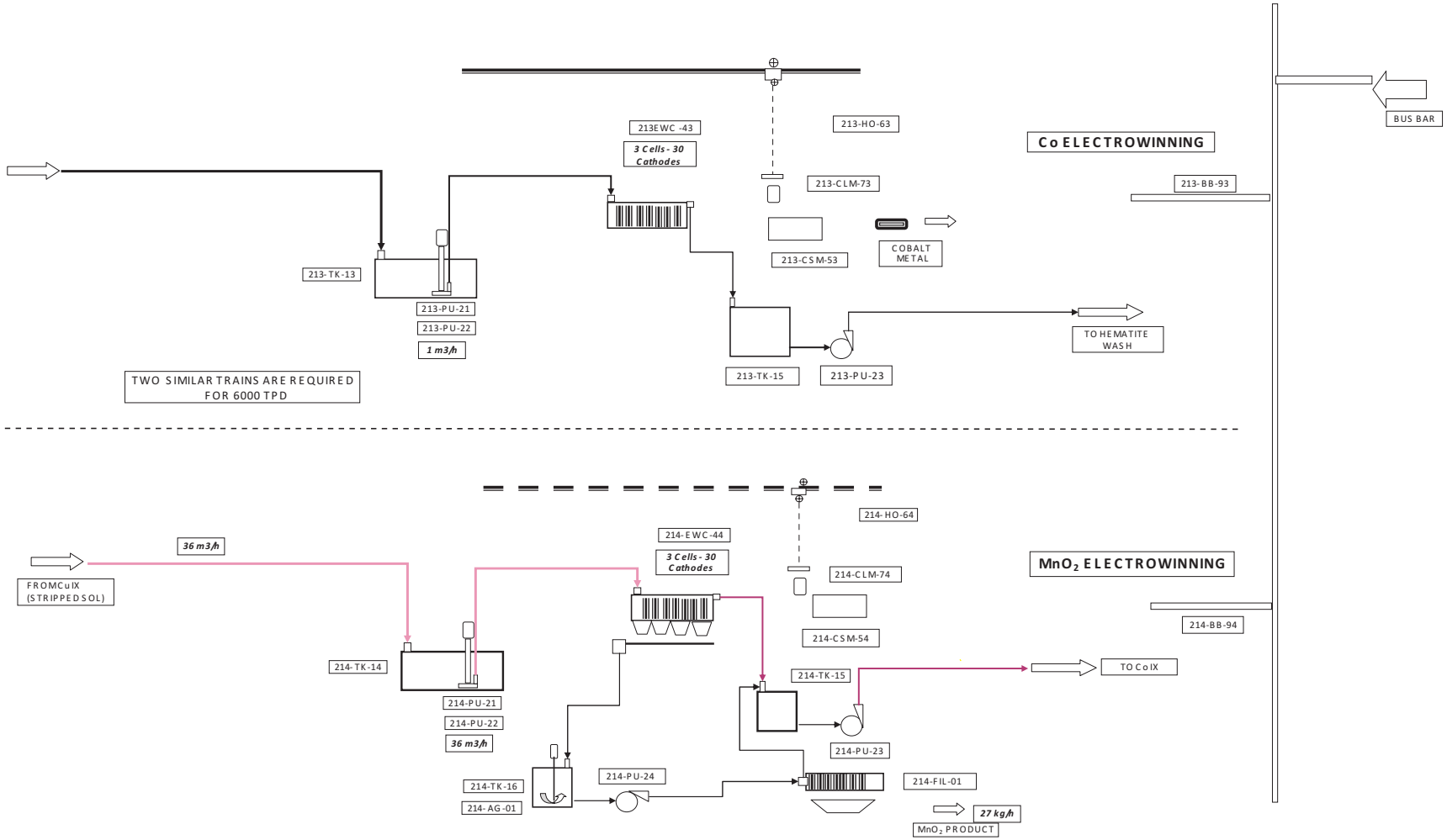

Figure 17-9

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Co and MnO_2 Electrolysis PFD

November 2011

Source: ACSI, 2011.

www.rpacan.com

MANGANESE OXIDE ELECTROWINNING STAGE

Raffinate from anionic Cu/Co SX goes to MnO_2 electrolysis in eight cells. MnO_2 is deposited at the anodes. The MnO_2 falls off the anodes and is pumped from the cells to a pressure filter. The filter cake will be sold or stored at site. The solution from MnO_2 EW is neutralized with lime and then is sent to Ni EW.

IRON S/X STAGE

Solution from secondary leach goes to ferric chloride solvent extraction. The raffinate goes to zinc IX. The solvent is stripped with water and the eluate is sent to hydrolysis.

ZINC I/X STAGE

Zinc is removed from iron SX raffinate by a zinc specific resin in IX columns. The raffinate goes to Cu-Co anionic SX. The resin is stripped with water and the eluate goes to acid regeneration.

ENERGY USE OPTIMIZATION

Energy management is of prime importance because of the high cost of generating power at site. The oxidation of H_2S, SO_2 and absorption of SO_3 are very exothermic. Most of the high pressure steam produced in the acid plant will be used to generate power. Some 28 bar steam from the acid plant is needed for the hydrolysis stage. Some low pressure steam from the acid plant is used for the evaporation stages. Figure 17-10 depicts the energy balance for the process design.

FIGURE 17-10 PROCESS ENERGY BALANCE

Energy balance

				MW
turbine 1				12.2
turbine 2				3.1
turbine 3				9.4
turbine 4				27.9
total				52.5
power to condensate pumps				0.8
power to vacuum pumps				0.0
power to acid plant				9.5
power for mining				6.0
power for crushing grinding and flotation				7.4
power for oxygen plant				3.2
power for electro winning				10.6
power for pumping agitation etc				5.3
total				42.8
net				9.6

HYDROGEN SULPHIDE TREATMENT

The H_2S from the primary leach, and the sulphur from secondary leach are burnt to SO_2. using air. The reactions are exothermic. A waste heat boiler will cool the gases and generate the 80 Bar steam, which will be expanded to 28 Bar in a turbine to recover power for generating electricity.

Approximately 1.1 Mt of acid will be produced each year, with a shipping season of approximately 15 weeks. The HCl regeneration section of the process will consume approximately 260,000 tpa of acid, leaving some 900,000 tpa of excess acid for shipment and sale to world markets. Approximately 500,000 tonnes of acid will be stored during the non-shipping season. The acid will be stored in a battery of mild steel tanks each with a diameter of 45 m diameter and 9 m high. The tank farm requires an area of 250 m x 520 m. The acid plant will produce 93.5% acid. This acid strength has the lowest freezing point.

MATERIALS OF CONSTRUCTION

INDUSTRIAL PLANT REFERENCE

Because of the high chloride background, established industry materials of construction will be used. The Magnola plant in Montreal ran with similar high magnesium chloride

solution compositions. The following sections present the summarized material selection approach used in this study.

ORE SLURRY EQUIPMENT

The ore slurry equipment inside the plant will be carbon steel, rubber-lined with Hypalon type rubber. Agitator will be also rubber lined with this type of equipment.

PRIMARY LEACH EQUIPMENT

The reactor walls will be constructed of 12 mm mild steel, 12 mm natural gum rubber, 25 mm furan tiles, and 75 mm low S high tensile graphite brick. Agitator wet parts will be made of Supper Duplex Stainless or Carpenter C lined.

SECONDARY LEACH, EVAPORATION, OXIDATION AND HYDROLYSIS EQUIPMENT

The reactor walls will be constructed of 12 mm mild steel, 12 mm natural gum rubber, 25 mm furan tiles, and 75 mm fused alumina. Agitators will be made of titanium. Slurry will be pumped through external heat exchangers. Thickeners will be constructed of Carbon steel, rubber-lined and ceramic liners. High-temperature Philippe belt filters, similar to those at Magnola, will be used.

ION EXCHANGE EQUIPMENT

FRP columns will be used.

ELECTROWINNING EQUIPMENT

Standard materials of construction for sulphate-based electrowinning systems will be used.

ALTERNATE MATERIALS OF CONSTRUCTION

Alternate materials of construction that can be considered for future evaluation include:

<u>Reactor, Secondary Vessels</u>

- PVDF lined FRP
- Glass lined CS

- Teflon Lined SS
- Hypalon lined SS
- Supper Duplex Alloy
- FRP (Derakene/Hetron types)

Thickeners

- Hypalon lined SS
- PVDF lined FRP

Slurry Pumps / Solution Pumps

- Hypalon lined SS
- Teflon lined
- Hastelloy C

Slurry Piping

- Rubber lined CS for low temperature
- PVDF lined CS

Solution Piping

- Rubber lined
- FRP
- CPVC for low temperature
- PTFE Lined CS/304L Stainless for temperatures up to 230 degrees C

Miscellaneous Process Equipment

- Hypalon lined
- Teflon lined
- Hastelloy C

PROCESS FACILITIES

ASPEN MODEL EQUIPMENT SIZING

The process model of the plant produced by Arithmetek was used to size equipment and help to prepare the equipment list. Equipment sizing is tied into the process model.

Figure 17-11 depicts the scope of the process facilities. Figures 17-12 and 17-13 depict the conceptual design of the concentrator and the process plant. The process plant layout includes the tank farm adjacent to the process plant.

EQUIPMENT LIST

The equipment list included in the cost estimate is presented in Appendix 1. The list includes preliminary sizing defined in the mass balance and the necessary items to complete the process facilities.

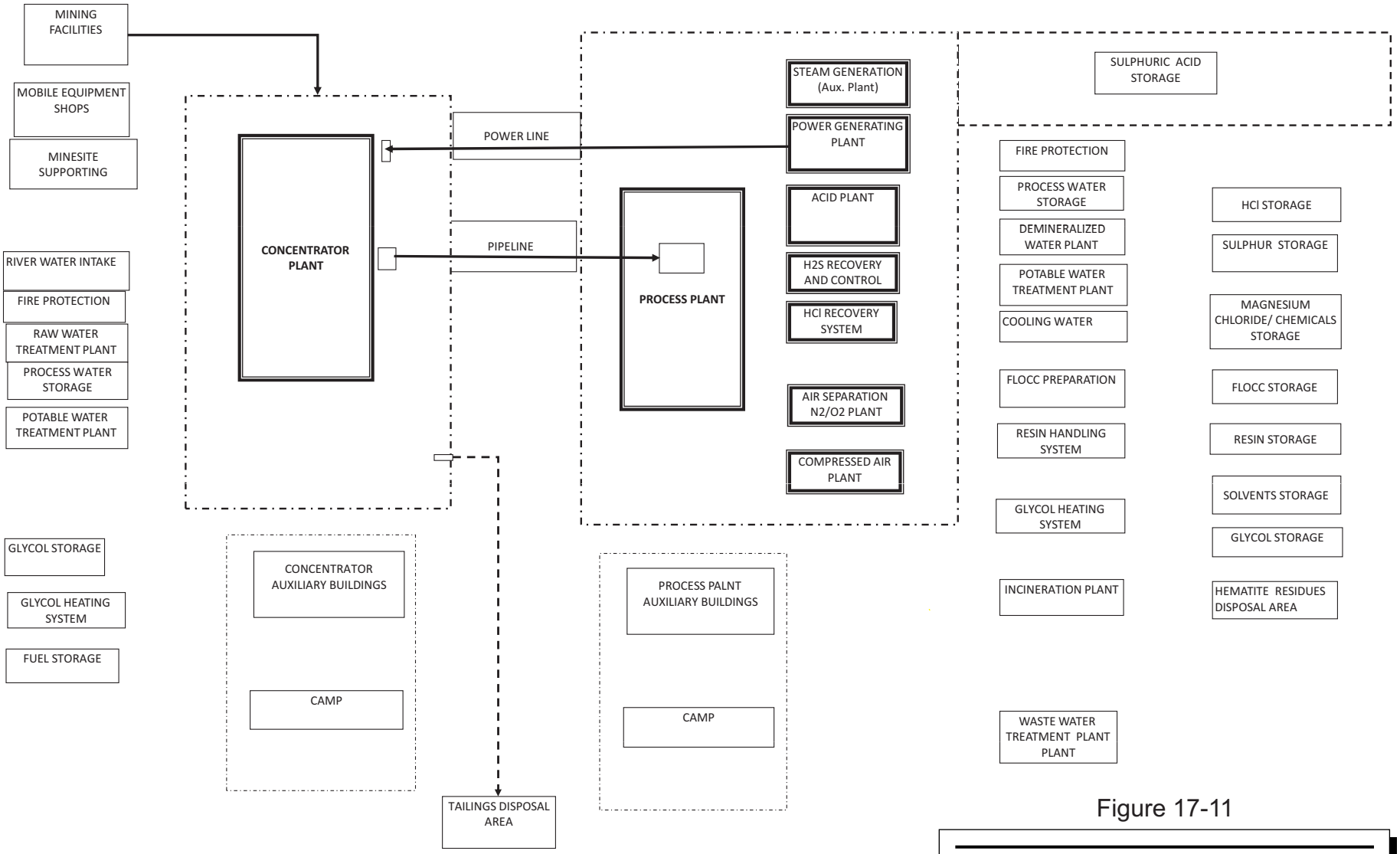

Figure 17-11

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Scope Process Facilities

November 2011

Source: ACSI, 2011.

www.rpacan.com

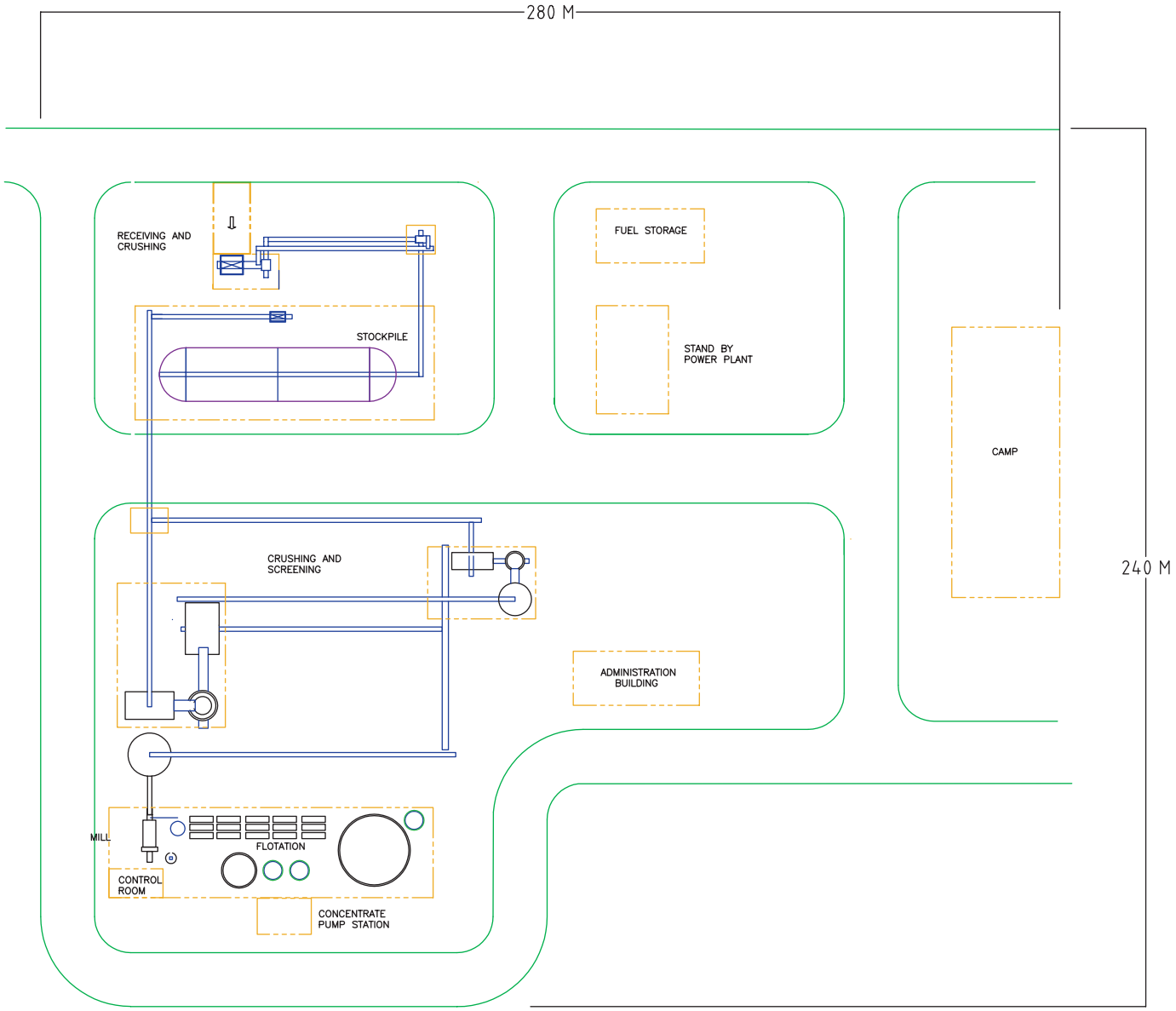

Figure 17-12

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Conceptual Concentrate Layout

November 2011

Source: ACSI, 2011.

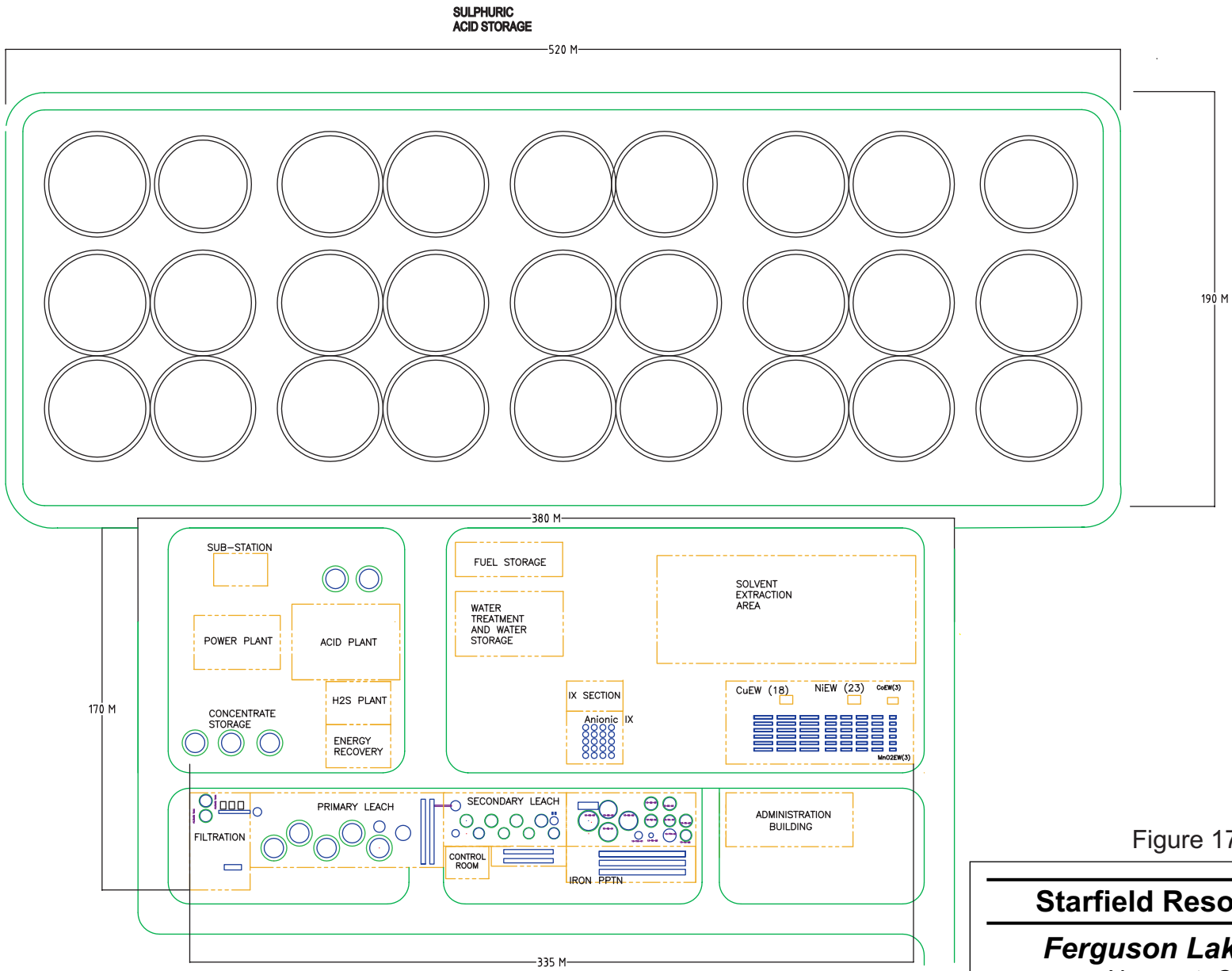

Figure 17-13

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Conceptual
Process Plant Layout

November 2011

Source: ACSI, 2011.

www.rpacan.com

18 PROJECT INFRASTRUCTURE

A new camp facility was constructed by Starfield on the western shore of Ferguson Lake in 2007. The camp is mostly a stick-built facility that incorporates some older modular sleeping units, Weatherhaven's, and a kitchen, together with a new dining room, lounge and offices that are capable of housing 70 to 80 persons. Exploration core is stored in stacks near the new camp.

There is a range of equipment on site suitable for supporting a drilling operation and minor construction. The major pieces of equipment on site are: Cat D4 dozer, Cat BL320 excavator, Bombardier Snow Cat, one Bobcat skid steer, two diamond drills (owned by Major Drilling), two pickup trucks equipped with all terrain tracks, one Cat 950F loader, one cat 140G motor grader, numerous ATV's, and numerous snow machines. There are also a number of sleds for hauling materials and/or sheds. There are three diesel generators for camp power at the new camp.

The present airstrip, which is located southwest of the new camp, was constructed in 2008. The airstrip is approximately 830 m long by 30 m wide. It is constructed of a rock base that was quarried on site and is covered by compacted gravel obtained from a nearby esker.

The existing network of site roads is limited. There are some ATV trails connecting the airstrip to the old camp and a short road from the new camp towards the new airstrip and connected to a fuel storage area.

Although the new landing strip, quarry, and water pump located on the east side of Ferguson Lake are connected by a network of site roads, all diamond drilling and drill support is undertaken by helicopter transport after the snow cover has melted.

Water for camp operations is obtained from the eastern side of Ferguson Lake and pumped to the camp via a pipeline. Water used for diamond drilling is obtained from lakes and/or streams proximal to the respective drill site.

For the purposes of this study, RPA has assumed that all fresh water needs will continue to be obtained from Ferguson Lake.

FERGUSON LAKE SITE – MATERIALS TRANSPORTATION

In Nunavut, there is no all weather road access between communities in the Territory or from Southern Canada. The Ferguson Lake site is 250 km southwest of the settlement of Rankin Inlet, which is located on the shore of Hudson Bay. To date, heavy equipment and site facilities have been brought to the site by Cat trains, over the frozen ground and lakes in winter only, via permitted right of way. The current operational scenario does not include a winter or an all-season road between the mine site and processing site at Arviat, and material transportation to the Ferguson Lake site will continue by Cat train.

Poor reliability of the Cat train units currently available in Rankin Inlet has caused problems in the past – there was no Cat train re-supply in the winter of 2006–2007. An owner-operated fleet of reliable units will be required to develop and support the Project.

Air transport of critical or lightweight items will be possible, however, no cost allowance for that has been included in the PEA, beyond regular flights for transport of personnel.

In RPA's opinion, material handling has the potential to affect Project costs and schedules, due to the isolated location of the Ferguson Lake site. Detailed logistical analysis of transportation requirements and alternatives is recommended for future studies.

FERGUSON LAKE SITE – PROJECT INFRASTRUCTURE REQUIREMENTS

The Ferguson Lake site requires infrastructure to support open pit and underground mining operations, crushing and concentrator facilities, and slurry preparation and pumping facilities. Site roads, camp facilities, power, mobile equipment, and buildings are discussed below, while process components are discussed in Section 17.

The footprint for the concentrator area will be approximately 280 m by 240 m. This footprint includes allowances for camp facilities, back-up power plant and fuel storage, ore receiving, primary crushing and crushed stockpile, secondary crushing and screening, flotation equipment, control room, concentrate pump station, and administration building.

A general site plan, with proposed new facilities, is shown in Figure 18-1.

SITE ACCESS ROADS

For the planned operation, a network of roads will be required to connect the main service facilities such as the airstrip, water supply, portal, shaft, concentrator, administration buildings, and accommodation areas. The road alignments should be planned to minimize disturbance to the tundra and identified sensitive archaeological sites. For this reason, RPA recommends that the majority of the facilities be placed in close proximity.

CAMP

Additional camp facilities will be required for the construction and subsequent operations. The new camp is suitable for exploration, early stage property development activity, and will continue to be used during construction and operations. Additional camp facilities should be located immediately adjacent to the mine and concentrator areas, for easy access to worksites, maintenance shops, and the equipment laydown area.

The Ferguson Lake operations camp will need room for approximately 250 people and will include sleeping rooms, a kitchen and dining room and recreational facilities.

POWER

Power will be generated at the Arviat process site, and transmitted by powerline to the Ferguson Lake site. The current process mass/energy balance indicates that 23 MW of power will be available to sustain mining, concentrating, and slurry pumping activities. A 44-69 kV class powerline will follow the slurry pipeline route between the process and mine sites.

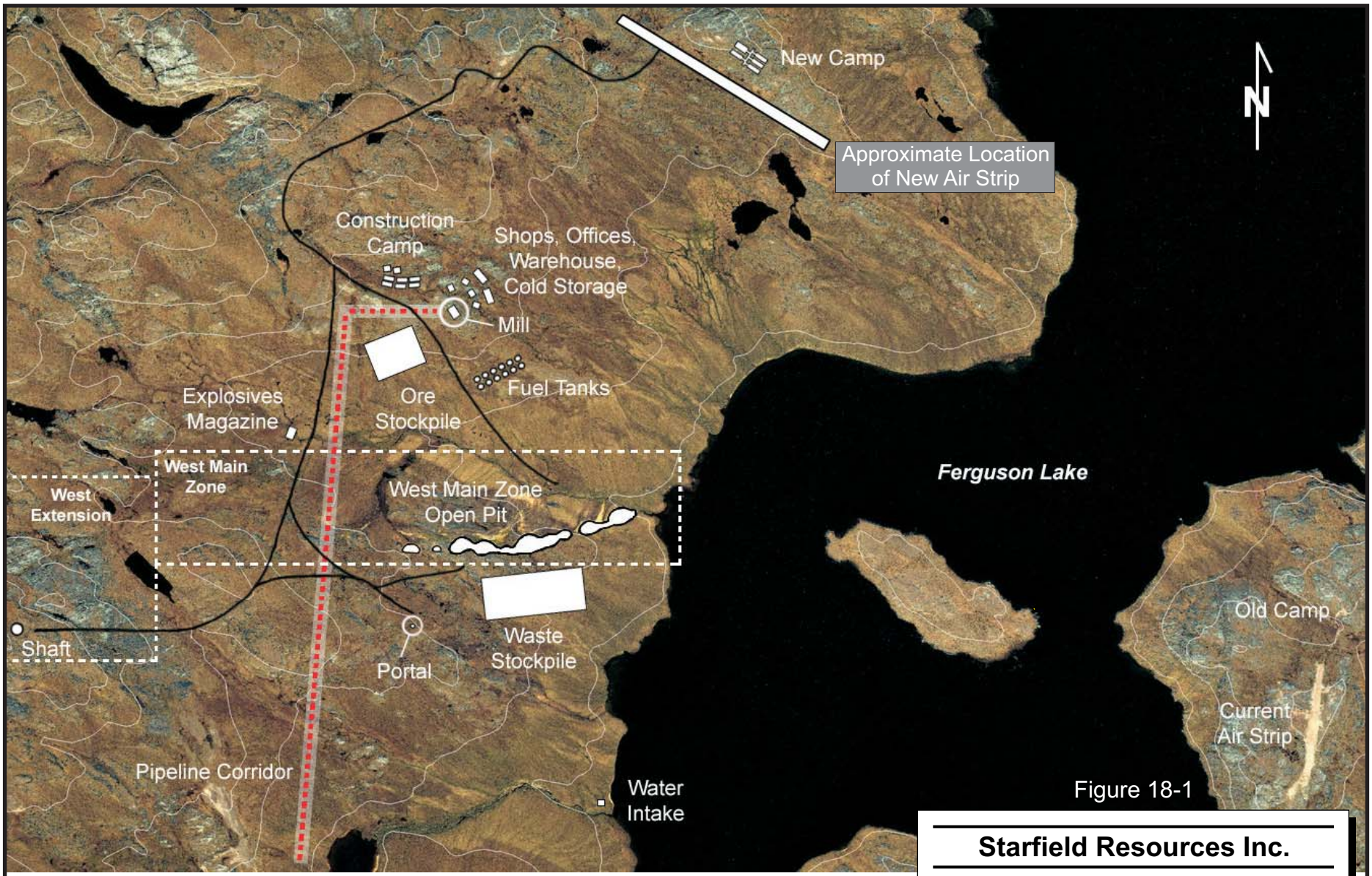

Figure 18-1

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Ferguson Lake Site Plan

November 2011

www.rpacan.com

RPA notes that process power generation is sensitive to the amount of sulphides in the process feed, and that analysis of available energy assumes steady-state operation of the process. Current process simulations indicate that approximately 11 MW of surplus power will be available, after all Project power needs have been satisfied. Depending on realized head grades, the amount of surplus power has the potential to rise to as much as 22 MW if an Organic Rankine cycle is included in the process plant design.

Back-up power at the Ferguson Lake site will be provided by diesel generators, with sufficient capacity to operate both mine and concentrator. The diesel generators will be equipped with appropriate monitoring and control instrumentation to provide reliable operation and monitoring of the units. Power factor management will be included in the power supply system. Fuel storage facilities will supply operating needs for mobile equipment, and maintain a reserve for power generation requirements.

SURFACE MOBILE EQUIPMENT

A surface mobile equipment fleet will be required to support the operations at Ferguson Lake. A preliminary list of the major items required is shown in Table 18-1.

TABLE 18-1 SURFACE MOBILE EQUIPMENT FLEET
Starfield Resources Inc – Ferguson Lake

Number	Type	Comments
2	4 x 4 Pickup	Management, technical, services
1	1 t Flat Deck	Surface materials haulage
1	3 t Flat Deck	Surface materials, includes Hiab crane
1	Excavator	Pit, waste rock, site prep, roads
1	Tractor Backhoe	Site roads
1	Grader	Pit, site roads
1	Cat 950 F Loader	Pit, site prep, roads
4	Snow Cat	Materials transportation
1	Cranes	Construction
1	Dozer D-6	Site prep, pit, waste rock
2	966 Front End Loader	Waste rock, backfill
1	Concrete Truck	Construction
1	Forklifts	Materials handling
1	Boom Truck	Materials handling
3	25 t Truck	Pit, shaft ore haulage
1	Diesel-Electric-Hydraulic Blasthole Drill Rig	Pit

There will also be a significant quantity of small construction equipment and implements for the mobile equipment that will be needed for construction and operations.

SITE BUILDINGS

The operation will require offices for administration, services, mining, and concentrating activities. For ease of operation in a harsh climate, the administration offices and mine dry will be located near, or within the concentrator building footprint and will be centralized as much as possible with the camp facilities.

Surface shops for equipment maintenance are planned to handle both the surface equipment fleet and the underground equipment maintenance (Main West Zone only). An allowance of one main underground shop per mine has been included in the Project capital requirement.

Warehouse facilities will include both heated and cold storage, with large equipment laydown areas.

A surface explosive storage facility will be isolated from other buildings and activities, and contain separate storage for explosives, blasting caps, and boosters. This facility will serve as a supply location for replenishment of underground explosive and detonator magazines.

COMMUNICATIONS

Site telephone and internet communications will be provided by satellite communications with portable satellite telephones for emergency communication. The offices and accommodation complexes will have telephone service through a common telephone system.

A two-way radio system will be installed for communication around the site.

PROCESS SITE – PROJECT INFRASTRUCTURE REQUIREMENTS

An initial, proposed location for the hydrometallurgical processing facilities is near Arviat. Final site location will be determined following a program of community consultation, and

consideration of alternatives such as Baker Lake or Rankin Inlet. With the exception of a house obtained in Rankin Inlet by Starfield on a yearly rental arrangement, there are currently no existing facilities or equipment at the proposed process site location.

The footprint for the process building will be approximately 380 m by 170 m. The building will host all of the necessary circuits required by the proposed metallurgical process, with allowances also included for a receiving terminal for the slurry, a sub-station and power plant, a control room, water treatment and water storage, fuel storage, and an administration building. Infrastructure items outside of this building footprint include the acid tank farm, port facilities, hematite/leach residue storage, and warehousing facilities. The acid storage footprint is estimated to be 520 m by 190 m based on the requirement for approximately 20 storage tanks.

A general plan for the process site is shown in Figure 18-2.

Figure 18-2

Starfield Resources Inc.

Ferguson Lake Project
Nunavut, Canada

Arviat Process Site Plan

www.rpacan.com

MATERIALS TRANSPORTATION

The process facilities will be located in close proximity to port facilities, to reduce logistical issues relating to material transportation. A deep-water inlet immediately north of Arviat is an attractive location for this purpose and should be reviewed for suitability during future studies. A fleet of surface vehicles will be used to transport supplies and materials from the port to the process site.

Supplies and materials will arrive by ship, in the ice-free season between the end of June and mid-October. Items for the Ferguson Lake site will be stored at the process site, and staged for transportation by Cat train in winter. The deep-sea-shipping route to the western shore of Hudson Bay, from major east coast Canadian ports, is northwest along the Labrador Coast, through the Labrador Sea. The route passes through Hudson Strait, between northern Quebec and Baffin Island. Vessels enter Hudson Bay as they pass west of Cape Wolstenholm, which is 110 km west of Deception Bay. Vessels can then pass to the west shore of Hudson Bay, going either south around Coats Island or north between Coats Island and Southampton Island.

Purchasing and warehousing will be managed from the mine and process sites, by a warehouse manager, buyer and warehousemen. Coordination of sea lift logistics will require consultants located in large shipping centres in the south.

Critical items may be flown into Ferguson Lake or Arviat by air. There is scheduled airline service to Rankin Inlet from Yellowknife, NT, Thompson, MB and Winnipeg, MB. Charter aircraft services are also available in Rankin Inlet.

Sulphide concentrate will be transported from the concentrator to the Arviat process site via slurry pipeline, as discussed below.

Outgoing materials include finished metal products, hematite, and acid. LME-grade metal will be transported by ship to buyers. Port facilities and shipping of acid are discussed below.

SLURRY PIPELINE

For this PEA, the slurry pipeline route is represented as a general corridor (Figure 18-3), aligned around large bodies of water. The corridor is approximately 285 km in length. Detailed study of specific alignments, number of water crossings, and alternative routes has not yet been completed. Pipeline design will be undertaken in accordance with environmental permitting requirements.

Pipeline design detail and cost estimates were originally provided by Pipeline Systems Incorporated in 2008. ASCI has updated the pipeline design and costs to reflect a smaller diameter pipeline requirement as a result of concentrating activities at Ferguson Lake. The pipeline will consist of 8" OD insulated steel pipe, laid directly on the ground wherever possible. Allowances were made for water crossings and wildlife access. There will be thickening and agitated slurry storage at Ferguson Lake, where crushed and ground ore will be converted to a slurry of 55% solids by weight. A minimum of two pump stations will be located at intervals along the pipeline, provided with two operating pumps and one standby pump, each driven by a 1,200 kW, variable-frequency drive motor. A receiving terminal with agitated slurry storage will be located at the process site. The pipeline will be controlled via a SCADA (Supervisory Control And Data Acquisition) system, operated on a data/voice telecommunications line.

Only winter road access has been included along the pipeline route. Construction and maintenance will take place during the winter, using Cat trains for transportation of heavy material and equipment. For purposes of emergency repair, Starfield will have to develop comprehensive spill management plans including procedures for access by season, which may include the use of Cat trains, all-terrain vehicles, or helicopters.

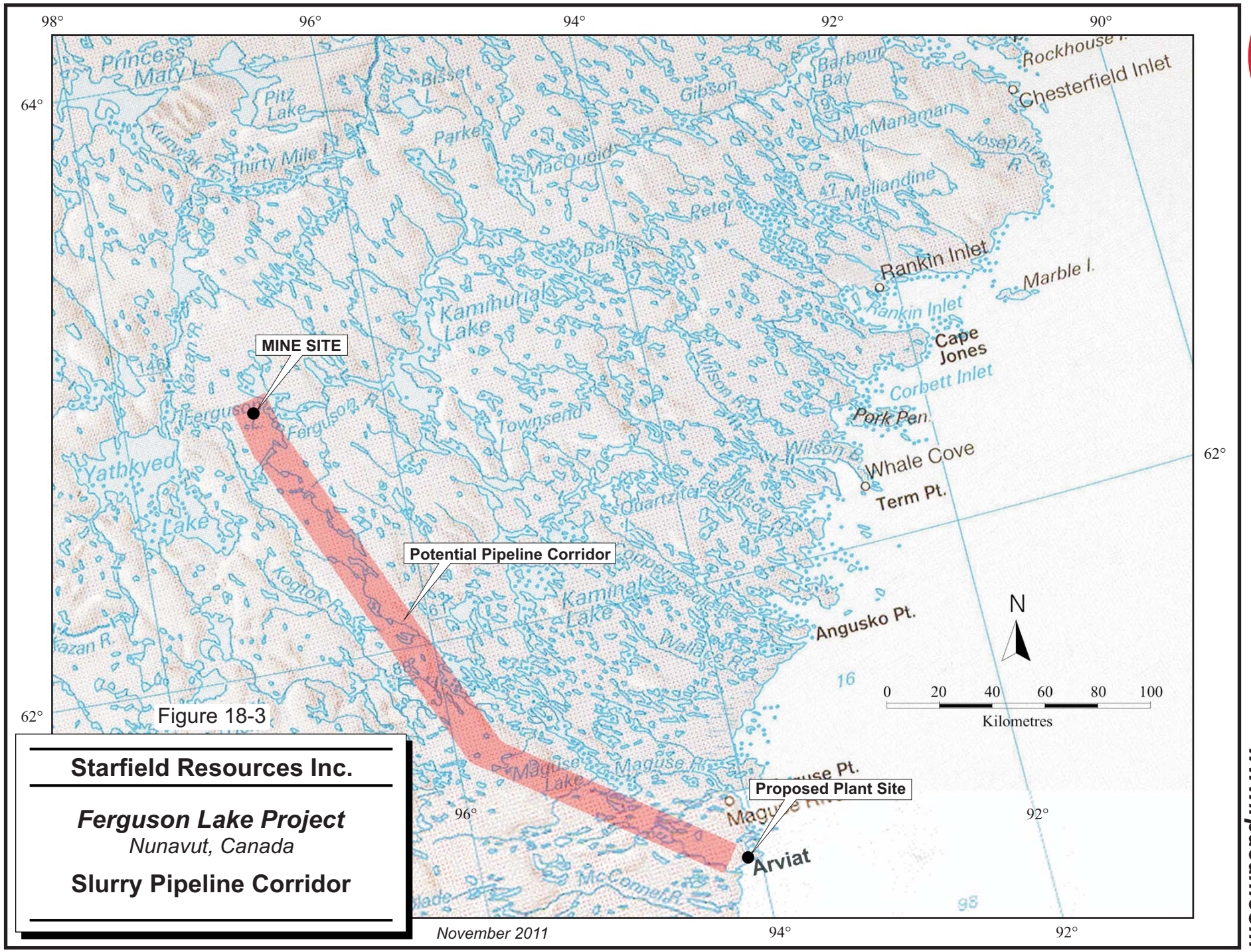

Figure 18-3

Starfield Resources Inc.

Ferguson Lake Project

Nunavut, Canada

Slurry Pipeline Corridor

November 2011

www.rpacan.com

ACID STORAGE AND TRANSPORTATION

As a by-product of the metal production process, the Project will be a net producer of approximately 850,000 tonnes of 93% pure sulphuric acid per year. It is planned that this will be shipped during the 15 week shipping season between the end of June and mid-October. Acid will be stored in a tank farm at the Arviat plant site during the winter months. The tank farm will comprise 20 tanks, with a total storage capacity of 500,000 tonnes of acid. The tanks will each be 23 m in diameter x 14 m high. They will be located in an impoundment area designed for proper containment, which will require a footprint of 520 m by 190 m. During the shipping season, the acid will be pumped to the wharf, to be loaded into ships.

The shipping requirement for acid is significant, in that the total annual volume must be moved in approximately 15 weeks. For the PEA, it has been assumed that the average distance to market will be 12,000 km, or approximately Brazil, requiring 8 x 50,000 tonnes acid capable tankers in operation during the shipping season.

LEACH RESIDUE STORAGE

The ultimate footprint for the hematite and leach residue material storage may be as much as 250 m by 250 m, based on annual LOMP quantities, and a stacking height of 10 m. Future studies are anticipated to include further processing of the leach residue to recover precious metals.

Leach residue material is anticipated to be non-reactive. Toxicity and reactivity assumptions will be tested as the Project is advanced.

PORT FACILITIES

Currently, port facilities at all potential hydrometallurgical process sites are limited. Construction of dedicated deep-water docking, handling, and storage facilities will be required.

The PEA includes allowances for construction of a dock, incorporating loading equipment appropriate for metal and acid products. As outbound transport of saleable products can only occur during the summer shipping season, allowances have been made for the required storage facilities at the process site.

Site locations for port facilities have not been studied in detail for this stage of the Project.

CAMP

Camp facilities will be developed at the process site, and will host the crews necessary during construction and operating phases of the process facilities. The camp will incorporate sleeping units, a kitchen with dining area, as well as office space. During operations, it will necessary to house approximately 120 people.

POWER

Power will be generated at the Arviat process site, and transmitted by powerline to the Ferguson Lake site. The current process mass/energy balance indicates that 23 MW of power will be available to sustain mining, concentrating, and slurry pumping activities. A 44-69 kV class powerline will follow the slurry pipeline route between the process and mine sites.

RPA notes that process power generation is sensitive to the amount of sulphides in the process feed, and that analysis of available energy assumes steady-state operation of the process. Current process simulations indicate that approximately 11 MW of surplus power will be available, after all Project power needs have been satisfied. Depending on realized head grades, the amount of surplus power has the potential to rise to as much as 23 MW if an Organic Rankine cycle is included in the process plant design.

There is potential for excess power and waste heat to be provided to the town at considerably less than current costs. Future study of the process mass/energy balance is expected to quantify this potential.

Diesel generation at both sites will provide power for construction, process start-up, and emergency back-up. The diesel generators will be equipped with appropriate monitoring and control instrumentation to provide reliable operation and monitoring of the units. Power factor management will be included in the power supply system.

SURFACE MOBILE EQUIPMENT

A surface mobile equipment fleet will be required to support operations at the Arviat process site. A preliminary list of the major items required is shown in Table 18-2.

TABLE 18-2 SURFACE MOBILE EQUIPMENT FLEET
Starfield Resources Inc – Arviat

Number	Type	Comments
2	4 x 4 Pickup	Management, technical, services
1	1 t Flat Deck	Surface materials haulage
1	Grader	Site roads
1	Crane	Construction
1	Dozer D-6	Site prep, pit, waste rock
2	966 Front End Loader	Site roads, snow clearing
1	Ambulance	
1	Fire Truck	
1	Concrete Truck	Construction
1	Forklift	Materials handling
1	Boom Truck	Materials handling
2	Dump Truck	Supplies and product handling
3	25 t Truck	Pit, shaft ore haulage
1	Skid-steer loader	Supplies and product handling

SITE BUILDINGS

As per the plant layout developed by ACSI, it is planned to have an administration building in close proximity to the process facilities. An allowance for maintenance shops is also included.

Warehouse space including cold storage, yard storage, and heated storage, will be needed at the site and at the sealift terminus for consumables, supplies, materials, metal products, and equipment.

19 MARKET STUDIES AND CONTRACTS

MARKETS

CATHODE METALS

The copper, nickel, and cobalt metal to be produced at Ferguson Lake are freely-traded products at prices that are widely known, provided that the product meets certain criteria for purity. Assuming that the metals are in pure metal form, the prospects for sale of any production are virtually assured.

SULPHURIC ACID

In Scott Wilson RPA's December 2008 Preliminary Assessment, information on the world acid market was prepared by PentaSul, a consulting and analysis group based in Houston, Texas. Historically, the world sulphuric acid market has been well supplied and prices between 1990 and 2006 ranged between approximately US$20 per tonne and US$50 per tonne. Demand for sulphuric acid is typically driven by fertilizer producers, SX/EW copper producers, and ISL uranium producers.

Beginning in early 2006, the market in the developing world went into imbalance, with acid demand significantly exceeding supply. During the period between 2006 and 2008, acid prices increased to over US$200 per tonne (US Gulf). Currently, some major South American copper producers are forecasting acid prices over the next few years in the US$110 to US$120 per tonne range, with long-term prices expected to decline toward the US$65 per tonne range. For this PEA, it has been assumed that the majority of sales would be to Latin American consumers and a long-term acid price of US$85 per tonne FOB customer has been assumed, resulting in a small netback to the Project of US$1 per tonne after accounting for transport charges.

While the acid produced from the Project is very high quality and there are reasonable prospects for its eventual sale, there are moderate to high risks in the acid sales assumptions due to the large volume being produced, the short shipping season, and the cost of transporting the acid to strong consumption markets. For context, the Project will produce approximately 850,000 tonnes of acid per year as compared to the total Canadian production in 2006 of approximately 4.8 million tonnes. The total US import

requirement in 2006 was approximately 2.5 million tonnes. Annual Latin American imports are expected to be in the order of 1.5 million tonnes to 2.0 million tonnes. Annual, global shipments of acid are in the neighbourhood of 200 million tonnes, with only a fraction of these shipments (approximately 5%) occurring via ocean vessels.

From a logistics point of view, it will be important to source customers that can store the large available volume coming from the Project in a relatively short period.

Future studies of product shipping should evaluate the possibility of domestic customers, such as uranium and fertilizer producers in the Canadian Prairies provinces, and potential supply of acid to uranium projects in Nunavut itself. In particular, a detailed trade-off exercise for the sale of sulphuric acid to Saskatchewan-based customers vs. international ocean shipments is recommended. This could review a scenario of ocean shipment to Churchill, followed by rail shipments to Saskatchewan.

RPA notes that the production of sulphuric acid is a key component of the Project's process design and acid production has important impacts on the overall power balance for the Project. Alternative flowsheet arrangements, such as conversion of acid to elemental sulphur, are expected to have significant impact on process power generation and will most likely result in the Project becoming more reliant on conventional, diesel-fired power generation.

HEMATITE

Revenue assumptions for sale of hematite are based on benchmark iron ore prices set by major producers in quarterly negotiations with steel producers. It was assumed that the filtered and dried hematite precipitate would be analogous to sinter feed (fines), rather than pellets. In early Q4 2011, the Metals Bulletin Iron Ore Index was consulted, with a finding for prices for sinter feed of US$170 per tonne delivered to China (62% Fe basis).

Since RPA's review in early Q4 2011, prices for comparable iron ore delivered to China have moderated (in the range of US$150 per tonne at end of November 2011). Given the expected iron and moisture contents of the finished hematite product, a value of US$125 per tonne has been assumed, resulting in a netback to the Project of US$55 per tonne, after shipping. RPA notes that the current process flowsheet produces hematite

product at 13% moisture content which is higher than is typically referenced in iron ore pricing and contracts, i.e., 4 to 5%.

OCEAN TRANSPORT CHARGES

RPA conducted a preliminary review of ocean shipping rates for the Arviat region, various product destinations, cargo types, handling methods, and vessel classes. Preliminary quotes for containerized shipments from Montreal, Quebec to Arviat range from approximately US$250 to US$400 per tonne, depending on ship class, lot size, and nature of the cargo. RPA notes that these rates reflect a scenario of off-loading sea containers from large ocean vessels to smaller barges as there are no deep-sea port facilities in the region. In RPA's opinion, it is likely that lower ocean shipping rates could be realized in circumstances where a long-term shipping contract is in place, and a deep-sea port has been constructed to facilitate direct transfer of cargo to large ocean vessels.

CONTRACTS

At the time of the preparation of this report there were no contracts in place for site activity, metal refining, transportation, or hedging or forward sales. The production scenario contemplated in this PEA includes operation by owner's personnel, with minor exceptions such as product shipping, camp catering, and air travel.

TAXES

The PEA has been completed on a pre-tax basis. RPA has not included any consideration for federal or provincial income taxes in this economic analysis.

ROYALTIES

Starfield's current 100%-earned interest in the property is subject to a 3% net smelter royalty (NSR) in favour of the Ferguson Lake Syndicate, on potential future mineral production, a 3% gross overriding royalty on any diamond production and a $25,000 annual advance royalty payment. Starfield has the right to purchase 1% of the NSR for

$1 million for a period of 180 days following receipt of a positive feasibility study recommending commercial production.

For purposes of the PEA, it has been assumed that the NSR purchase has been completed. A royalty of 2% NSR has been included in the cash flow.

RPA notes that at another project in the area, the final lease agreement included a net profits interest to the land owner. Such interests or royalties will be the subjects for negotiation in the production lease agreement.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

Project concepts such as site locations, product storage and handling, and process mass/energy balances are at a scoping level for this PEA, consistent with the current stage of Project advancement. Further study and engineering is expected to include environmental management planning, in order to minimize adverse environmental impacts.

The proposed process features significant advantages over conventional industry operations for similar projects. The process includes power generation from the sulphur contained in the ore, which would otherwise be generated by diesel combustion (annual requirements on the order of 100 million litres of diesel), with attendant greenhouse gas emissions. Another advantage is that process products are not very reactive, in contrast to conventional products of massive sulphide mining and processing, which are often significantly acid-producing, and require sequestration and extensive treatment.

In RPA's opinion, the Project includes some environmental management issues beyond those considered typical for northern mines, such as large-scale acid storage, handling and transportation, and concentrate transportation by slurry pipeline over long distances across the tundra. Ongoing investigation, examination of alternatives, and best-practice design with due consideration for the environment must remain a high priority as the Project is advanced, if permitting is to proceed smoothly.

PERMITTING

EXISTING PERMITS

Starfield does not yet hold any of the permits required to operate a mine. Starfield carries out its site activities under the following permits/licences and lease:

1. 2BE-FER0712 – Type B Water Licence, Nunavut Water Board (expires in July 2012). The licence allows water use for drilling, domestic purposes and related activities for mine exploration in the amount of 100 m^3/d.

2. Commercial Lease, Land Use Licence and Right of Way issued by the Kivalliq Inuit Association (KIA).

The primary area being explored on is Inuit Owned Land under the Nunavut Land Claim Agreement. As such, a negotiated Commercial Lease was entered into in 2009 with the Kivalliq Inuit Association. The purpose of the Lease agreement was for access to the land as well as environmental protection and socio-economic considerations. A separate production lease will need to be negotiated with the KIA to permit production activity.

FUTURE PERMITTING REQUIREMENTS

One of the primary permits required for the construction and operation of a mine is a Class A Water Licence from the Nunavut Water Board. The production lease from the KIA is the main land use and occupancy permission for the Project. There will be numerous additional permits and approvals required for the Project.

Following discussions with the regulatory officials, a Draft Environmental Impact Statement (DEIS) will have to be prepared and submitted in accordance with the Nunavut Impact Review Board's (NIRB) requirements for proposed mine development established by Part 5 of the Nunavut Land Claims Agreement. The NIRB issues guidelines for the preparation of Environmental Impact Statements. The objective of the DEIS and its supporting documents is to provide a detailed description of the proposed project; current physical, biological and socioeconomic conditions; potential impacts, mitigation and management strategies; and long term monitoring plans. The lead authorizing agency, NIRB, will facilitate a comprehensive review of the documents and determine whether additional information is required for the Final Environmental Impact Statement (FEIS).

The FEIS will be required to support the water licence application and for the Production Lease application with the KIA. The Nunavut Water Board and KIA will not begin reviewing the applications until they receive a Project Certificate from the NIRB.

It will be necessary to apply for a Type A water licence and a Production Lease, covering the mine site, slurry pipeline, and process site. There are issues that will trigger an environmental review under the *Canadian Environmental Assessment Act* (CEAA). This is likely to take the form of a Comprehensive Study Report although the requirement for

a Review Panel Hearing cannot be excluded. As well, at a Federal level, any disturbance or alteration of any fish habitat would involve the Department of Fisheries and Oceans (DFO), the *Fisheries Act* and the *Navigable Waters Protection Act.* Should the use of a lake be proposed for tailings storage, designation (as a tailings area) will be required by the Minister of Environment under the Metal Mining Effluent Regulations.

ENVIRONMENTAL MANAGEMENT

Starfield has developed a number of management plans as part of an Environmental Action Plan, to assist with management and operational decisions during the exploration phase. These plans will need to be expanded upon as the Project moves toward development. These management plans include:

- Spill Response
- Emergency Response
- Abandonment and Restoration
- Wildlife
- Protected and Sensitive Areas
- Fuel Storage
- Water Quality
- Waste Management
- Hazardous Materials
- Species at Risk
- Environmental Awareness and Education
- Archaeological Sites

BASELINE ENVIRONMENTAL STUDIES

Baseline environmental studies were initially conducted at the Ferguson Lake site in 1999 and have been expanded upon over the years. Some of the components to date include:

- Ferguson Lake Archaeological Investigations
- Wildlife Baseline and Monitoring Programs
- Aquatic Resources Baseline and Monitoring Programs, streams and lakes

- Fisheries Baseline
- Hydrology Baseline and Monitoring Programs
- Ferguson Lake Soils and Vegetation Baseline
- Meteorology

In 2007, a more comprehensive baseline program at the Ferguson Lake site was conducted by Rescan Environmental Services Ltd. (Rescan). The scope of work included:

- Meteorology
- Hydrology
- ML/ARD
- Aquatic resources
- Fisheries studies
- Wildlife
- Archaeology
- Hydrocarbon contamination assessment of the old camp

WEATHER

There is a weather station on site, located near the camp. The data logger has been in place since August 2005.

HYDROLOGY

The main objective of the hydrologic monitoring program has been to characterize the timing and magnitude of stream flow at various locations within the Ferguson Lake area, which may be potentially affected by the development of the project. As well, this data will be used to help make management, design and development decisions in the future. The hydrology work in 2007 was the first surface water hydrology study to be undertaken in the area since water sampling was done in 1999. Monitoring was suspended in 2009.

AQUATIC RESOURCES AND FISHERIES STUDIES

In order to monitor potential changes related to the development of the Project, water, sediment quality and aquatic biology baseline studies have been conducted in 1999 and then again in 2005, 2006 and 2007 at the Ferguson Lake site. In 1999, water quality and fisheries studies were conducted. In 2007, work included baseline studies on:

- Water Quality
- Physical Limnology
- Sediment Quality
- Primary Producers
- Secondary Producers
- Fish and Fish Habitat

Studies have been conducted in streams, rivers and lakes within the anticipated mine-area receiving environment as well as consideration being given to water bodies that have the potential to be used as containment structures and streams that may require diversion.

WILDLIFE MONITORING

Wildlife baseline information for the Ferguson Lake site has been collected to document the pre-development status of wildlife communities in the area. The collection of baseline data also contributes to the identification of any species at risk. The evaluation of the information collected will allow for the development of management strategies to identify the best mitigation measures to help minimize and/or eliminate potential detrimental effects on wildlife. This data will also provide a reference point for future reclamation and management planning efforts.

The Project site is in the migration corridor of the Beverly and Qamanirjuaq Caribou Herd. A wildlife management plan for exploration and winter road activities has been prepared and further refinement of these existing plans will be required to include mine operations.

In 2007 Starfield supported a caribou monitoring program being conducted by the KIA to collect information with regard to the spring migration of the herd to the calving grounds. In addition, four dedicated local Environment Monitors were hired and trained by Rescan to assist with monitoring and data collection.

Additional wildlife monitoring program will include breeding birds, waterfowl, raptors, grizzly bear, small mammals, and carnivore/den surveys, and more aerial surveys of caribou and muskox.

RECOMMENDATION

RPA recommends that a comprehensive environmental baseline program be conducted to build upon existing baseline data for the Ferguson Lake site, and to begin collecting information for potential slurry pipeline routes, process sites, and port sites, to support the development of a Draft Environmental Impact Statement. The program halted in 2009 should be resumed and completed.

KEY PROJECT CHARACTERISTICS

The hydrometallurgical process for recovering base metals from the Ferguson Lake project is unique. Some key features of the project are reviewed below.

MINE SITE

The mine site will include an open pit, underground mine, and surface plant to crush, grind and float the ore to produce a concentrate. The residual produced from this process includes:

- 800,000 to 1,000,000 tonnes per year of flotation tailings. These tailings have minimal residual metal content and low levels of sulphur (typically 0.5%). There is a possibility that these tailings could be mildly acid producing and this will need to be confirmed by additional testing. It is estimated that approximately 35% of the annual flotation tailings will be used as backfill in the mine.

- The open pit will produce upwards of 2 Mt of waste rock, a portion of which will be acid generating. The waste rock will be segregated with acidic rock stored on a lined pad. Waste rock will be used as backfill for the mine. At closure, any residual acid rock not used as backfill would most likely be buried in the open pit.

- Excess water from the mine and tailings area will be recycled to the concentrator for use. If required, any excess will be treated prior to discharge to the environment. Given the low precipitation for the area, and the low inflow rates of mine water expected, it is unlikely that there will be a need for discharge to the environment.

- Fresh water will be required for the concentrator. The primary fresh water supply will likely be Ferguson Lake and total demands (likely less than 5,000 m^3/d) will have no material effect on the lake. Run-off from the tailings pond and mine water will be recycled for use in the concentrator.

PORT FACILITY

The Port facility will include the hydrometallurgical plant, storage facility for sulphuric acid and elemental sulphur, and storage areas for by-products from the process. These by-products include: hematite (approximately 750,000 tonnes per year), leach residue

(approximately 250,000 tonnes per year) and manganese dioxide (approximately 1,300 tonnes per year). These materials are expected to be inert and not a source of leachable metals. Allowance has been made for permanent storage of these by-products. Additional processing of the leach residue to recover precious metals is also under consideration. The plant also will discharge about 1.2 Mt per year of water containing elevated levels of sulphate. This discharge is not expected to have any material impact.

There is a significant demand for cooling water at the port facility. Total demands are approximately 1,050,000 m^3/d and it is expected this requirement will be met through the use of sea water.

KEY MANAGEMENT ISSUES

Key environmental management issues associated with the current Project concepts consist of:

- Aquatic Resources

  At the mine:
  - Storage facility for the flotation tailings (tailings may be mildly reactive). The siting of this facility may impact on aquatic habitats.
  - Storage facility for the waste rock (some of the waste rock is likely acid generating)
  - Water usage for grinding and slurrying of ore concentrate
  - Stream diversion at open pit site and tailings management facility

  Along the pipeline
  - Water crossings, spill containment and management

  At the Port Process Plant
  - Cooling water usage.
  - Marine habitat impacts (siting, operational and transportation).

- Proximity to sensitive and protected wildlife and archaeological areas
  - Slurry pipeline route relative to Caribou Protection Area.
  - Other Project sites (mine and port facilities).

- Acid storage and handling
  - Tank farm footprint.
  - Land transport, transhipment, and ocean transport.

- Storage of process products/by-products (hematite, sulphur, residue)

- Use/management of hazardous chemicals (H_2S, HCl, etc.)

Detailed mine site water balances have yet to be estimated. A fresh water source, currently assumed to be Ferguson Lake, is likely to be required to supply water for grinding and flotation of the ore and slurrying of the concentrate prior to transportation to the process site. The underground mine is anticipated to operate in two separate sections – the Main West Zone, which is expected to be largely in permafrost, should not produce much water; the West Extension, below the permafrost boundary, may be a net water producer. Both mines will produce water if tailings are used as backfill. The initial open pit will require some waterway diversion.

Slurry pipeline routing has not been considered in detail. Routing and pipeline design will include consideration of stream and waterway crossings, so that erosion and other impacts on fish habitat are minimized, in addition to land-based considerations, such as wildlife crossings, protected areas, and archaeological sites. Bio-assays of slurry toxicity to index species will be undertaken, to quantify potential impacts in the event of a spill. Spill prevention and containment plans will be developed to minimize the risk of adverse events.

Process cooling water requirements have been initially estimated at 1,050,000 m^3 per day. A preliminary investigation indicates that fresh water sources near Arviat may not be able to provide the maximum quantity of required cooling water, due to lake size and regulatory limits. As such, it is likely that sea water will be used, however, other potential sources for cooling water supply will be reviewed as the design proceeds. It is expected that required quantities will be reduced through some or all of the following:

- Further work on the process energy balance
- Use of waste heat for buildings
- Recycling of cooling water
- Use of sea water
- Use of alternate cooling methods, such as cooling towers.

Port facility locations have not been considered in detail. Further studies will involve marine habitat impact studies, with due consideration for bathymetry, fish habitat, harvest and fishing areas, baseline conditions, and protected areas. The use of established shipping routes, to the extent possible, may also minimize new environmental impacts.

The slurry pipeline route to the proposed process site, near Arviat, bypasses a DIAND-designated Caribou Protection Area. RPA notes that Rankin Inlet, an alternative process site location, may require routing the slurry pipeline through the Caribou Protection Areas. This would be expected to require a more demanding permitting process.

RPA notes that the slurry pipeline concept consists of widely-used, industry-standard technology, and recommends a review of comparable pipelines currently in operation in the Arctic (North America, Scandinavia, Russia), to determine best practices for design and operation from an environmental viewpoint.

The acid storage tank farm includes an area allowance for spill containment. Further design work may reduce the overall footprint by increasing berm heights or taking advantage of natural topography. Acid handling, including transport from the process plant to the storage tanks, from the storage tanks to the port facility, and transhipment to ocean freighter, will require consideration of the following:

- Safety and Human Health.
- Spill Response and Management plans for land, fresh water, and marine environments.
- Transportation of Dangerous Goods regulations.

Process products not considered saleable in the current scenario include leach residue containing precious metals. The residue will be stored in a dry-stacked storage facility located at the process site. Future studies will confirm the potential for economic recovery of precious metals from the leach residue. The residue storage facility may be reduced in size or eliminated as a result.

Going forward, the following environmental considerations will have to be considered for all Project site locations:

- Archaeological sites
- Sensitive wildlife habitat and protected areas
- Stream crossings
- Permafrost
- Terrain/landforms
- Ecological habitat
- Lakes, streams and wetlands

SOCIOECONOMIC CONSIDERATIONS

The Kivalliq Region, one of three administrative regions in Nunavut, had an estimated population of over 8,300 people in 2006. The population is spread among seven communities (Chesterfield Inlet, Whale Cove, Arviat, Baker Lake, Rankin Inlet, Coral Harbour and Repulse Bay), with Rankin Inlet being the largest (2,300 people) and Whale Cove the smallest (350 people). Baker Lake, with an estimated population of 1,500, is the only inland Kivalliq Region community. The primary communities of interest to the Ferguson Lake Project are Arviat, Baker Lake and Rankin Inlet.

In an economy that is predominantly historically based on government services, there were generally limited opportunities for employment in Nunavut. There are a high proportion of young employable people, and much interest for training and employment in the exploration, construction and mining sectors has been expressed by political leaders. The development of new mines and advanced exploration projects has resulted in a drop of unemployment to 16.6% in Nunavut (drop of 2% between 2010 and 2011) and with ongoing development, future hiring outside of the Kivalliq region could be necessary to fill all the available jobs. It is anticipated that Starfield will be encouraged to open an office in either Baker Lake or Rankin Inlet.

During the Project life, approximately 500 direct jobs will be created through construction (three years), and approximately 700 jobs through operations (20 years), translating into direct (wages) and indirect (taxes, support industries) economic benefits for the Kivalliq Region.

There is potential for the process plant to generate excess power, which could be sold to the local utility at considerably lower costs than is possible using diesel generation, which is the principle current method in the communities of the Kivalliq Region. This would provide both an economic (lower power costs) and environmental benefit (reduced diesel combustion).

A socio-economic baseline study was initiated in 2007 and additional study will be required to support the environmental assessment.

INUIT IMPACT AND BENEFITS AGREEMENT

Under the Nunavut Land Claims Agreement, an Inuit Impact and Benefit Agreement (IIBA) will have to be negotiated with the Kivalliq Inuit Association. Likely included in the IIBA will be cash payments, scholarships, business opportunities and commitments to hire and train residents of the Kivalliq region (Rankin Inlet, Baker Lake, Arviat, Chesterfield Inlet and Whale Cove).

The negotiation of an Inuit Impact and Benefit Agreement is not required until a commercial production decision is made. Once that decision has been made the company has eighteen months in order to negotiate and ratify an agreement with the Kivalliq Inuit Association. The Inuit communities of Rankin Inlet, Baker Lake and Arviat are deemed to be "impact communities". Benefits in terms of employment and business opportunities, when they make business sense, are provided to these communities first before other Nunavut communities. To date a number of employees have been hired from each of these communities with varied degrees of success.

A number of local businesses, Nuna M&T, First Air, Canadian North, Calm Air, Inns North, Northern Store, Co-op store, Eskimo Point Lumber, have been utilized. It is anticipated that more local businesses could be utilized once more capacity is gained by local businesses. Benefits in terms of local employment and business opportunities will continue to grow as the project proceeds from exploration to advanced exploration and then ultimately mine development. Agreements with the Kivalliq Inuit Association for a Mining Lease as well as an Inuit Impact and Benefits Agreement will likely have to be negotiated as well as a Socio Economic Agreement with the Government of Nunavut.

COMMUNITY CONSULTATION

A Community Consultation Plan involving meetings with the general public, community and business leaders, Community Land and Resources Committees, elders' councils, schools, political and regulatory agencies as well as Non-Government Organizations has been implemented, and will continue throughout the life of the mine. Periodic tours of the Project site will take place for local politicians and local business people.

A policy for donations and sponsorships for individuals and organizations in the Kivalliq Region has been established.

Traditional Knowledge studies have begun with the Ferguson Lake Native Group in Baker Lake and will be extended to all communities in the Kivalliq Region. These studies include a Constraints Map to guide the development of the Project and visits to the site.

COMMUNITY RELATIONS

The Project has engaged consultants to assist in the community relations work. The main task is to manage the expectations of the public. There will not be jobs for everyone, and firm targets for local hiring or for use of local goods and service providers can lead to poor practices that do not provide real opportunities for the local residents. Starfield strives to maintain a balance between business services and opportunities and employment opportunities between the main communities in the Kivalliq Region. In addition, Starfield encourages contractors to utilize local businesses and services where it makes good business sense.

Starfield will need to make it a priority to keep the communities informed of project advancements or setbacks and to create constructive dialogue between all parties. Project elements will have to be planned based on community input.

SOCIOECONOMIC RECOMMENDATIONS

RPA recommends that Starfield formalize arrangements with contractors to maximize employment and business opportunities for those within the Kivalliq Region. As well, Starfield should begin discussions with Nunavut Arctic College regarding co-operative opportunities with students as the project advances.

SITE RECLAMATION

A site reclamation plan will be required as part of the design and permit application. In this area the general expectation would be that all facilities would be removed from the site and that surface disturbance would be modified to minimize the impact upon wildlife and other land users. As part of the Project design the area of disturbance will be minimized and as much as possible there will be ongoing rehabilitation work concurrent with operations.

Facilities that cannot be used in the future will be demolished and removed from site for salvage to the extent possible. The conceptual closure plan for the residual facilities is as follows:

Mine Site

- Cover and reclamation of the flotation tailings area
- Disposal and burial of acidic waste rock in the pit
- Capping of the underground mine
- Disposal burial of inert construction rubble in the open pit or in an approved surface landfill.
- Allow the open pit to flood. Provide safety berms as required.
- Reclaim all disturbed areas

Pipeline

- Remove all pipelines and spill ponds
- Reclaim disturbed areas

Port Site

- Cover and reclamation of the residue storage pile and any residual hematite by-products
- Reclaim disturbed areas

Site monitoring of conditions and water quality would continue for a period of years after the cessation of operations and removal of facilities.

A reclamation bond will be required to provide funding for the closure plan. In general the bond will be expected to provide full coverage for a government run reclamation program. It may be possible to provide some of the funds in stages during construction, however, it is forecast that the bulk of the funds will need to be in place when operations commence.

The estimated cost for mine reclamation is $35.9 million as outlined in Table 20-1. This cost will increase if tailings are reactive and require special covers.

TABLE 20-1 MINE RECLAMATION COSTS
Starfield Resources Inc. – Ferguson Lake

Item	Cost ($)
Mobilization/Demobilization	400,000
Labour	14,727,000
Equipment	2,873,000
Freight	3,510,000
Reclaim Site Roads	150,000
Reclaim Concentrate Pipeline	2,850,000
Reclaim Mine Tailings (incl. labour)	7,739,000
Reclaim Leach Residue (incl. labour)	1,018,000
Monitoring (incl. labour)	2,625,000
Total	**35,891,000**

Note: Table total may not sum due to rounding

21 CAPITAL AND OPERATING COSTS

CAPITAL COSTS

A pre-production capital cost of $1,500 million is estimated for the construction work required to bring the Project into production, including open pit pre-stripping, mining and process equipment purchase and transportation to site, construction of facilities and infrastructure at Ferguson Lake and the process site, and installation of all services required to support the Project. The estimate includes pre-production costs incurred over three years, with no allowance for escalation. Completion of underground mine development extends through the year of production, and costs for that work, totaling $113 million, have been included in the pre-production capital.

The estimate includes direct costs of equipment, materials, and labour by RPA. Estimates for direct process costs were carried out by NeoFerric (within the battery limits of the plant, as noted below). Indirect costs, estimated by RPA, include EPCM, freight and construction; temporary construction services, owner's costs, and contingencies. Costs for advancing the Project to construction stage, such as resource upgrade drilling, metallurgical testwork, process development, permitting, environmental studies, feasibility studies, and project financing, are not included.

Total Project capital costs are summarized in Table 21-1.

TABLE 21-1 CAPITAL COST SUMMARY
Starfield Resources Inc. - Ferguson Lake

Item	Pre-production ($ millions)	Ongoing ($ millions)	Total ($ millions)
Mine	84.7	66.7	151.4
Process Facilities	788.1	-	788.1
Infrastructure	94.5	-	94.5
Indirects	243.4	26.3	269.7
Contingency	176.9	19.7	196.6
Subtotal	**1,387.5**	**112.7**	**1,500.2**
Sustaining Capital	-	44.2	44.2
Reclamation	-	35.9	35.9
Total	**1,387.5**	**192.8**	**1,580.2**

Note: Totals may not add due to rounding.

PRE-PRODUCTION CAPITAL
The pre-production capital cost estimate covers costs from the time of decision, through three years of construction and into the start of production.

MINE CAPITAL
Open pit mine capital costs for pre-production include a period of eight months of open pit pre-stripping and purchase of the open pit mobile equipment fleet. Capital costs for the underground during this period include purchase of a portion of the mobile equipment fleet, main ramp and ventilation raise development in the Main West Zone, and initial development of the shaft for the West Extension. A tabulation of mine capital costs during the pre-production period is summarized in Table 21-2.

TABLE 21-2 PRE-PRODUCTION MINE CAPITAL
Starfield Resources Inc. – Ferguson Lake Project

Area	Total ($ millions)
Pre-stripping	1.2
Open Pit Mobile Fleet	2.9
Underground Mobile Fleet	27.9
Shaft	27.0
UG Development and Ventilation	25.7
Total	**84.7**

Note: Totals may not add due to rounding.

Pre-stripping costs were based on unit operating costs ($3.85 per tonne moved) described below. Mobile equipment costs were based on early 2011 quotes from manufacturers. Shaft costs in pre-production represent an allowance for ordering long lead-time items (hoist, head-gear). Mine development costs were estimated to be $6,000 per metre advance for ramping and level development.

As underground mining begins after a year of open pit mining, much of the capital cost for underground mining is deferred until after production start-up. The remaining direct capital costs for mining ($67 million) and associated indirect costs and contingencies are described below, under ongoing capital.

PROCESS CAPITAL

Process capital costs are based on estimates by ACSI, part of the NeoFerric metallurgy team, for areas within the battery limits of the process plant, and by RPA for other areas. A summary of the process capital is included in Table 21-3. The major difference from the previous study is that the concentrator is now located on the mine site and a flotation concentrate is slurried and pumped to the processing plant that will be located near Arviat.

TABLE 21-3 PRE-PRODUCTION PROCESS CAPITAL
Starfield Resources Inc. – Ferguson Lake Project

Area	Total ($ millions)
Site Prep.	2.0
Crushing & Grinding	18.2
Mill Flotation & Thickening	41.1
Backfill Plant	3.3
Concentrate Utilities	2.9
Power Plant & Sub Station	50.2
Pipeline	130.4
Leach/Oxidation/Evaporation	55.9
Hydrolysis	21.0
SX/EW	143.2
Sulphur Recovery	2.3
Reagents	6.5
Acid Regeneration	14.3
Acid Plant	140.0
Acid Storage	67.2
Oxygen Plant	25.7
Utilities	21.5
Power Generation	13.2
Buildings & Services	26.5
Tailings	3.6
Total	**788.1**

Note: Totals may not add due to rounding.

Crushing and grinding, which includes site preparation, a process building, ore receiving equipment, materials, and installation costs, was estimated by RPA, based on available metallurgical information, in-house reference data, and comparisons to similar projects.

Capital costs associated with the slurry pipeline include materials and labour for the pipeline itself, a slurry thickener, storage, pumping stations, a receiving terminal and tankage, and a SCADA/telecom system. Costs were obtained from a budget quote provided by Pipeline Systems Incorporated.

ACSI produced capital equipment cost estimates for Filtration through Auxiliary Plants areas in Table 21-3, at a stated accuracy of ± 40%, as detailed in their report (Burga, 2008). The capital costs are based on preliminary sizing of process equipment based on the flowsheet and mass balance.

The equipment costs were calculated from reference data based on the preliminary sizing. Worksheets that relate sizing with weight and costs for materials of construction were then completed to define the cost of the reactors and tanks.

In order to define costs for facilities related to the SX/IX stages and EW stages, recent budget prices were used from other installations and factored to account for size and escalation.

Normal practice for this level of study would be to calculate installation labour costs by factoring the direct equipment costs. Application of factors in this case (for Filtration through Auxiliary Plants areas in Table 21-3), however, resulted in a cost total that implied an unusually large amount of labour hours would be required to build the plant. ASCI adjusted the labour factors, based on experience in earlier PEA involvement.

The acid plant cost, including installation labour was obtained by ACSI from an acid plant supplier, Haldor Topsoe, who supplied additional information on the power to be generated in the acid plant for the present application. The acid plant cost definition includes the waste heat boiler and turbine generator equipment for a 29 MW power generation plant. Quotes were also obtained for the oxygen plant.

An estimate for an acid storage tank farm with capacity for storing eight months acid production was obtained by ACSI from a specialist.

Other capital items estimated by RPA include the dryers for the hematite precipitate, hematite product storage, assay lab, metallurgical lab, mobile equipment, metal storage, gangue material storage, and an allowance for site preparation.

INFRASTRUCTURE CAPITAL

Infrastructure capital costs include buildings, roads, mobile equipment, power, storage at the mine and process sites, a port facility, and administration. A summary of the infrastructure cost estimate is summarized in Table 21-4.

TABLE 21-4 PRE-PRODUCTION INFRASTRUCTURE CAPITAL
Starfield Resources Inc. – Ferguson Lake Project

Area	Total ($ millions)
Buildings	4.8
Roads	24.0
Mobile Equipment	7.0
Power	27.7
Ferguson Lake Site	2.7
Snow Cats	8.8
Port Facilities	18.8
Administration	0.8
Total	**94.5**

Note: Totals may not add due to rounding.

Building costs were estimated based on unit costs per square metre. Road costs were estimated at unit rates of $400,000 per km for all-season site roads, and $200,000 per km for an allowance of 25 km for preparation of winter access to the slurry line route. An allowance of $15 million was included for water crossings. Power costs include back-up diesel generators (11 units), the transmission line between sites (120 km at a rate of $100,000 per km), and fuel storage at both sites. Port facility costs were based on studies from comparable Arctic projects.

INDIRECT CAPITAL COSTS

Indirect capital costs, estimated by RPA, are summarized in Table 21-5: It should be noted that the indirect costs include those of the pre-production years plus year one.

TABLE 21-5 INDIRECT CAPITAL
Starfield Resources Inc. – Ferguson Lake

Area	Total ($ millions)
EPCM	108.6
Freight	79.2
Capital Spares	10.3
Construction Operating Costs	41.5
Owner's Costs	21.5
Other	8.5
Total	**269.7**

Note: Totals may not add due to rounding.

Costs for engineering, procurement, and construction management (EPCM) were calculated as 10% of total direct capital costs. Freight costs were estimated to be 8% of direct capital costs for relevant equipment items. Capital spares, representing consumables used during construction, were estimated to be 2% of direct capital costs. Construction operating costs include power, flights, surface infrastructure maintenance and camp operations costs for the pre-production period, calculated in the same way as for operating costs, below. Owner's costs consist of labour costs for staff and hourly personnel working on site during the pre-production period (owner's crews). Other indirect capital items include allowances for construction mobile equipment, contractor mobilization and demobilization, insurance, first fills for process chemicals, miscellaneous supplies, and vendor representatives.

CONTINGENCY

In RPA's opinion, many of the direct capital costs, particularly process equipment for some areas, were estimated at a level of detail that exceeds normal practice for a PEA. As a result, contingencies were calculated item by item, rather than applying a global percentage.

Costs based on allowances, coarse unit rates, or factors, such as mine development costs were assessed a contingency of 25%. Costs supported by vendor quotations including ancillary items were assessed a contingency of 5% to 15%. Costs for process items were considered at 15%. Total pre-production contingency of $177 million is 15% of direct and indirect pre-production capital costs.

ONGOING CAPITAL

Capital costs for completion of the underground mines, after the start of production, are tabulated in Table 21-6.

TABLE 21-6 ONGOING MINE CAPITAL
Starfield Resources Inc. - Ferguson Lake

Item	Total ($ millions)
UG mobile equipment	26.3
Shaft	13.5
UG development & ventilation	25.7
Services (incl. backfill)	1.2
Indirects	26.2
Contingency	19.7
Total	**112.6**

Note: Totals may not add due to rounding.

Mobile equipment costs were based on recent vendor quotations. Shaft costs were based on a comparable project, and on RPA internal databases. Development costs were based on unit rates of approximately $6,000 per m advance for ramps and levels, and $2,000 per m advance for raises for ventilation and backfill. Allowances for shop equipment, ventilation fans, and other equipment were included. Indirect costs include EPCM, freight, and contractor demobilization. The overall average contingency is 15.0 % of direct and indirect costs.

An annual sustaining capital cost of $2.5 million and an allowance of $35.9 million for reclamation at the end of mine life are also included in ongoing capital. Sustaining capital amounts to 3% of direct capital costs for major equipment and facilities. The reclamation costs, detailed above, under Environmental Considerations, include consideration for materials and labour, site rehabilitation and ten years of environmental monitoring.

EXCLUSIONS

The following are excluded from the capital cost estimate:

- Project financing and interest charges
- Land acquisition, leases, rights of way, and water rights
- Escalation during construction
- Permits, fees and process royalties
- Environmental impact studies
- Taxes
- Import duties and custom fees
- Cost of geotechnical investigation
- Working capital

- Sunk costs
- Pilot Plant and other testwork
- Exploration drilling
- Costs of fluctuations in currency exchanges
- Project application and approval expenses
- Future expansion
- Relocation of any facilities, if required
- Investigations required for completion of a Feasibility Study
- Feasibility Study

OPERATING COSTS

Life of Mine operating unit costs average $94.70 per tonne milled. The estimate includes mining, site services, and G&A cost estimates by RPA. Estimates for process operating costs were carried out under the direction of NeoFerric (within the battery limits of the plant).

Operating costs for the Ferguson Lake Project are summarized in Table 21-7:

TABLE 21-7 UNIT OPERATING COST SUMMARY
Starfield Resources Inc – Ferguson Lake Project

Item	Cost
Mining – Open Pit ($/t moved)	3.83
Mining – Open Pit ($/t open pit ore)	6.54
Mining – Underground ($/t ug ore)	45.36
Total Mining ($/t milled)	43.71
Processing ($/t milled)	25.31
Site Services ($/t milled)	8.26
G&A ($/t milled)	11.16
Life of Mine Plan Total	**88.43**

Note: Totals may not add due to rounding.

Open pit mining in the first two years of production reduces the LOMP average cost. Site services include surface operations and infrastructure maintenance, while G&A includes costs specific to remote operations, such as flights for personnel, and freight for supplies and consumables.

Open pit mining costs are estimated to be $3.53 per tonne moved, or $7.70 per tonne milled. Costs are inclusive of labour (owner crews), explosives, fuel, and equipment operation and maintenance.

Estimated underground mine operating costs are shown in Table 21-8.

TABLE 21-8 UNDERGROUND OPERATING COST ESTIMATE
Starfield Resources Inc. – Ferguson Lake

Item	$ per tonne milled
Staff Labour	2.87
UG Labour	19.98
Fuel & Explosives	5.30
Dewatering	0.50
Equipment Maintenance	5.24
Supplies	6.47
Backfill Preparation	5.00
Total	**45.36**

Note: Totals may not add due to rounding.

Labour costs are detailed below, under Manpower. Staff labour includes engineering, geology, and mine supervision. Material costs including the fuel and explosives were calculated based on development rates for the size of headings which was $1359 per meter of advance. Backfill preparation is an allowance to account for non-labour costs associated with sourcing and delivering waste rock fill to the underground stopes.

Estimated processing operating costs are shown in Table 21-9.

TABLE 21-9 PROCESSING OPERATING COST ESTIMATE
Starfield Resources Inc. – Ferguson Lake

Item	$ per tonne milled
Labour	10.96
Reagents	4.33
Grinding Steel	1.80
Equipment Spares	6.56
Utilities	1.66
Total	**25.31**

Note: Totals may not add due to rounding.

Processing costs have been estimated by ACSI, with input from RPA on manpower levels, wages and salaries, reagent pricing, and steel costs. Labour costs are described below, under Manpower. Reagent costs were estimated from unit costs and consumption rates for hydrochloric acid, calcium carbonate, calcium hydroxide, solvents, and flocculants. Grinding steel costs were estimated from unit costs and consumption rates. Equipment spares was estimated using an allowance of 2% of direct process equipment capital costs per year. Utilities include fuel for diesel power generation (10% utilization of seven megawatt plant) at $1.00/L, and an allowance for cooling water for the plant.

Estimated Site Services operating costs are shown in Table 21-10.

TABLE 21-10 SITE SERVICES OPERATING COST ESTIMATE
Starfield Resources Inc. – Ferguson Lake

Item	$ per tonne milled
Labour	1.62
Camp Operations	2.98
Equipment	0.36
Services	0.72
Airstrip and Road Maintenance	0.06
Pipeline Operation	2.04
Tailing Disposal Expansion	0.43
Building Installation and Maintenance	0.05
Total	**8.26**

Note: Totals may not add due to rounding.

Labour costs are described below, under Manpower. Camp operations are based on a contracted rate of $45 per person per day for food and janitorial services. Equipment costs are inclusive of the plant maintenance shop, small tools, heavy and light equipment, compressors, and any associated freight, fuel handling, and distribution costs. Costs for services include operation and maintenance of water supplies, sewage, heating systems, telecommunications, and any services relevant to the plant. Pipeline operation and maintenance costs were estimated based on a vendor quote.

Estimated G&A operating costs are shown in Table 21-11.

TABLE 21-11 G&A OPERATING COST ESTIMATE
Starfield Resources Inc. – Ferguson Lake

Item	$ per tonne milled
Labour	2.52
Air Travel	4.42
Land Freight	0.52
Sea Freight	3.28
Other	0.42
Total	**11.16**

Note: Totals may not add due to rounding.

Labour costs are described below, under Manpower. Air travel costs include small charter aircraft flying from Rankin Inlet to Ferguson Lake, and commercial flights from Rankin Inlet to southern Canadian travel hubs. Land and sea freight are allowances based on shipping tonnage and freight rates. Other costs are inclusive of items such as office supplies, insurance, property tax, consulting fees and licensing.

MANPOWER

Construction and operating crews are assumed to work two 11-hour shifts per day for periods of two to three weeks followed by two to three weeks off site. A summary of total manpower requirements for the operation is shown in Table 21-12. Non-integer quantities in the "On shift" or "On site" columns arise due when counting unique individuals, such as a General Manager, who is on site half of the time, and does not have a cross-shift.

TABLE 21-12 TOTAL MANPOWER REQUIREMENTS
Starfield Resources Inc – Ferguson Lake

Area	On shift	On site	Total
Mine	117.5	169.5	339
Plant	56.0	103	206
Surface	14.0	17.0	34
Administration	18.5	20.5	41
Total	**218**	**329**	**620**

The estimated manpower requirements by area are shown in Tables 21-13 through 21-16.

TABLE 21-13 MINE MANPOWER REQUIREMENTS
Starfield Resources Inc. – Ferguson Lake

Area	On shift	On site	Total
Supervision	7	12	24
Engineering	12.5	12.5	25
Geology	6	8	16
Production/Development	84	137	274
Total	**109.5**	**169.5**	**339**

TABLE 21-14 PLANT MANPOWER REQUIREMENTS
Starfield Resources Inc – Ferguson Lake

Area	On shift	On site	Total
Foreman	4	4	8
Metallurgist	2	2	8
Technicians	2	4	8
Assay Lab	4	4	8
Services	7	14	28
Shifters	5	10	20
Operators	31	62	124
Clerk	1	1	2
Total	**56.0**	**102**	**206**

TABLE 21-15 SURFACE MANPOWER REQUIREMENTS
Starfield Resources Inc – Ferguson Lake

Area	On shift	On site	Total
Supervisor	1	1	2
Truck Operator	3	6	12
Services	3	3	6
Maintenance	5	5	10
Equipment Operators	2	2	4
Total	**14**	**17**	**34**

TABLE 21-16 ADMINISTRATION MANPOWER REQUIREMENTS
Starfield Resources Inc – Ferguson Lake

Area	On shift	On site	Total
Management	1	1	2
Superintendents	2	2	4
Community Relations	1	1	2
Safety and Training	6	8	16
Warehouse and Purchasing	4	4	8
Port	1.5	1.5	3
Environmental Monitors	2	2	4
Receptionist	1	1	2
Total	**18.5**	**20.5**	**41**

Base salaries and wages have been estimated based on comparable projects and operations. A salary loading of 35% has been added to the base salaries, to account for benefits and scheduled overtime. Base salaries for representative positions are shown in Table 21-17:

TABLE 21-17 REPRESENTATIVE SALARIES
Starfield Resources Inc – Ferguson Lake

Position	Annual Salary ($)
General Manager	200,000
Superintendents	170,000
Planning Engineers	120,000
Electricians	100,000
Diesel Mechanics	95,000
Miner – Drill Operator	130,000
Miner – Truck Driver	90,000
Mill Operator	75,000
Labourer	60,000

22 ECONOMIC ANALYSIS

The PEA for Ferguson Lake is based on the following general assumptions:

- Mineral Resources as previously disclosed for the Main West Zone and the West Extension, modified by the addition of dilution and extraction factors.
- Mine production rate of 6,000 tonnes per day, 2.1 million tonnes per year.
- Initial mining of approximately two million tonnes from an open pit.
- Subsequent underground mining by mechanized cut and fill, in both Main West Zone and West Extension simultaneously.
- Underground access to Main West Zone and West Extension by ramp and shaft, respectively.
- Backfill using waste rock material, from underground waste development, open pit waste, surface waste sources, and tailings from the concentrator.
- Pumping of sulphide concentrate via 285 km pipeline, to a proposed process plant site, located near Arviat.
- Processing at Arviat, Nunavut, comprising a two-stage leaching followed by SX, IX to purify the solution, then SX and EW to recover the following products:
  - Nickel, copper, and cobalt as LME-grade metal
  - 93% sulphuric acid
  - Hematite
  - Gangue material
- Power generated at the Arviat plant site through burning of H_2S gas, and capture of process heat, used to power the process and, via transmission line, the mine site, concentrator, and concentrate pumping.
- Metal products and acid moved to market via bulk shipping.
- Gangue material storage at the Arviat process site plant by dry stacking.

A pre-tax Cash Flow Projection has been generated from the Life of Mine production schedule, capital and operating cost estimates, and is summarized in Table 22-1. A summary of the key criteria is provided below. The cash flow considers the Project from the construction decision and does not include allowances for permitting, pre-feasibility and feasibility studies.

ECONOMIC CRITERIA

PHYSICALS

- Mine life: 20 years
- Pre-production period: Three years (construction to start of production)
- Operations: 350 days per year
- Production rate:
  - Initial production from open pit of 2,000 tpd, 726,000 tpa, in Year 1
  - Combined production from open pit and underground of 6,000 tpd, 2,100,000 tpa, in Year 2
  - Production solely from underground of 6,000 tpa in Year 3 to end of mine life
- Total mill feed: 40.3 million tonnes
  - 0.61% Ni
  - 0.98% Cu
  - 0.07% Co
  - 36.3% Fe
  - 19.98% S
- Metallurgical recovery (two-stage leach and SX/IX/EW circuits):
  - 91% for Ni
  - 96% for Cu
  - 90% for Co
- Typical annual production of saleable products:
  - 25.6 million pounds of Ni
  - 41.6 million pounds of Cu
  - 2.8 million pounds of Co
  - 0.85 million tonnes of sulphuric acid
  - 0.74 million tonnes of hematite

REVENUE

- Sales prices: Base Case
  - US$10.00 per pound Ni
  - US$3.50 per pound Cu
  - US$20.00 per pound Co
  - US$85 per tonne acid
  - US$125 per tonne hematite
- Exchange rate: US$1.00 = C$1.00
- Payable products:
  - 100% for Ni
  - 100% for Cu
  - 100% for Co

- 100% for sulphuric acid
- 100% for hematite
- No credit for Pt, Pd

- Sales charges:
  - US$0 for Ni (saleable cathode)
  - US$0 for Cu (saleable cathode)
  - US$0 for Co (saleable cathode)
  - US$70 per tonne (metal shipping)
  - US$84 per tonne (acid shipping)
  - US$70 per tonne (hematite shipping)

- Average NSR value: $238.86 per tonne

- Net Revenue by product:
  - Nickel 51.1%
  - Copper 29.5%
  - Cobalt 11.5%
  - Acid 0.2%
  - Hematite 6.9%

COSTS

- Operating costs:
  - Total mining: $43.71 per tonne milled
  - Processing: $25.31 per tonne milled
  - Site services: $ 8.26 per tonne milled
  - G&A: $ 11.16 per tonne milled
  - Total (average): $ 88.43 per tonne milled

- Capital costs: Life of Mine
  - Pre-production $1,387.5 million
  - Ongoing UG Mine $ 112.6 million
  - Sustaining: $ 44.2 million
  - Closure: $ 35.9 million
  - Total: $1,580.2 million

- Royalties: 2.0% NSR (after buy-out of 1% NSR)

- Average unit costs (per pound of equivalent nickel, net of byproduct credits):
  - Operating: US$3.39
  - Capital: US$1.72
  - Total: US$5.11

TABLE 22-1 PRE-TAX CASH FLOW SUMMARY
Starfield Resources Inc. - Ferguson Lake

			Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
PHYSICALS	Days Mined						350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350	350
Main West Open Pit	Ore Mined	tpd					2,073	2,983																		
	To Mill	'000 t	1,770				726	1,044																		
	To Dump	'000 t	1,783			300	731	1,052																		
	Ni	%	0.51				0.51	0.51																		
	Cu	%	0.82				0.82	0.82																		
	Pd	g/t	-				0.00	0.00																		
	Pt	g/t	-				0.00	0.00																		
	Co	%	0.06				0.06	0.06																		
	Fe	%	31.98				31.98	31.98																		
	S	%	16.64				16.64	16.64																		
Main West Underground	Production	tpd					-	2,143	3,301	3,301	3,299	3,301	3,301	3,299	3,300	3,300	3,300	3,300	3,299	3,299	3,300	3,301	3,301	3,301	4,280	5,092
		tpa	22,511,582				-	749,961	1,155,344	1,155,191	1,154,639	1,155,360	1,155,447	1,154,814	1,155,124	1,154,903	1,154,869	1,155,027	1,154,823	1,154,604	1,155,054	1,155,454	1,155,463	1,155,213	1,498,139	1,782,153
	Tonnes	'000 t	22,512				-	750	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,498	1,782
	Ni	%	0.59				-	0.67	0.61	0.61	0.62	0.60	0.61	0.59	0.52	0.51	0.52	0.63	0.60	0.54	0.56	0.63	0.67	0.64	0.54	0.56
	Cu	%	0.88				-	1.02	1.18	1.10	1.03	1.07	1.00	0.82	0.68	0.66	0.63	0.77	0.69	0.72	0.80	0.99	1.13	0.89	0.73	0.86
	Pd	g/t	-																							
	Pt	g/t	-																							
	Co	%	0.07				-	0.08	0.07	0.07	0.07	0.07	0.07	0.07	0.06	0.06	0.06	0.08	0.07	0.07	0.07	0.07	0.07	0.08	0.06	0.07
	Fe	%	35.01				-	39.35	36.57	36.84	37.16	36.42	36.87	35.88	32.74	32.24	32.81	35.05	33.76	31.68	32.49	36.64	39.32	37.64	32.96	32.34
	S	%	19.17				-	21.96	20.08	20.25	20.47	19.96	20.26	19.61	17.53	17.22	17.54	19.60	18.60	17.14	17.68	20.28	21.94	20.90	17.85	17.62
West Extension Underground	Production	tpd					-	899	2,701	2,699	2,701	2,700	2,699	2,701	2,699	2,699	2,700	2,699	2,699	2,699	2,701	2,699	2,699	2,699	1,719	-
		tpa	16,033,618				-	314,707	945,303	944,798	945,210	945,050	944,698	945,374	944,629	944,508	945,106	944,530	944,541	944,545	945,312	944,616	944,503	944,595	601,594	-
	Tonnes	'000 t	16,034				-	315	945	945	945	945	945	945	945	945	945	945	945	945	945	945	945	945	602	-
	Ni	%	0.66				-	0.67	0.62	0.58	0.71	0.68	0.73	0.59	0.63	0.60	0.69	0.72	0.70	0.77	0.67	0.61	0.61	0.61	0.64	-
	Cu	%	1.14				-	1.27	1.12	1.14	1.20	1.39	1.30	0.99	1.23	1.08	1.01	1.55	1.30	1.29	1.05	0.90	0.75	0.92	0.99	-
	Pd	g/t	-																							
	Pt	g/t	-																							
	Co	%	0.08				-	0.08	0.07	0.07	0.09	0.08	0.09	0.07	0.07	0.07	0.09	0.09	0.09	0.09	0.08	0.07	0.07	0.07	0.08	-
	Fe	%	38.64				-	39.32	37.22	35.52	40.90	39.46	41.48	35.63	37.74	36.63	39.99	40.91	40.33	42.86	39.20	36.83	36.85	37.05	37.79	-
	S	%	21.48				-	21.91	20.47	19.35	23.02	22.02	23.42	19.48	20.82	20.06	22.38	23.05	22.64	24.44	21.87	20.26	20.23	20.36	20.92	-
							34%																			85%
Total Mill Feed	Production	tpd					2,073	6,025	6,002	6,000	6,000	6,001	6,000	6,001	5,999	5,998	6,000	5,999	5,998	5,998	6,001	6,000	6,000	5,999	5,999	5,092
		tpa	40,314,779				725,578	2,108,668	2,100,646	2,099,989	2,099,849	2,100,411	2,100,145	2,100,188	2,099,753	2,099,411	2,099,975	2,099,557	2,099,363	2,099,148	2,100,365	2,100,070	2,099,966	2,099,808	2,099,734	1,782,153
	Tonnes	'000 t	40,315				726	2,109	2,101	2,100	2,100	2,100	2,100	2,100	2,100	2,099	2,100	2,100	2,099	2,099	2,100	2,100	2,100	2,100	2,100	1,782
	Ni	%	0.61				0.51	0.59	0.61	0.60	0.66	0.63	0.66	0.59	0.57	0.55	0.59	0.67	0.65	0.65	0.61	0.62	0.64	0.63	0.57	0.56
	Cu	%	0.98				0.82	0.96	1.15	1.12	1.11	1.22	1.13	0.89	0.93	0.85	0.80	1.12	0.96	0.97	0.92	0.95	0.96	0.90	0.81	0.86
	Pd	g/t	-				0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Pt	g/t	-				0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Co	%	0.07				0.06	0.07	0.07	0.07	0.08	0.07	0.08	0.07	0.07	0.06	0.07	0.08	0.08	0.08	0.07	0.07	0.07	0.07	0.07	0.07
	Fe	%	36.32				31.98	35.69	36.87	36.25	38.84	37.79	38.95	35.77	34.99	34.21	36.04	37.68	36.72	36.71	35.51	36.73	38.21	37.37	34.34	32.34
	S	%	19.98				16.64	19.32	20.26	19.85	21.61	20.89	21.69	19.55	19.01	18.50	19.72	21.15	20.42	20.42	19.57	20.27	21.17	20.66	18.73	17.62
METAL RECOVERIES	Ni	%	94%			94.1%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%	94%
	Cu	%	96%			95.8%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%	96%
	Pd	%	90%			90.0%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
	Pt	%	90%			90.0%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
	Co	%	90%			89.8%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%	90%
					Payable	Deduct																				
PAYABLE METAL	Ni	'000 lbs	511,667		100%		7,643	25,864	26,615	25,999	28,816	27,635	28,932	25,632	24,730	23,964	25,848	29,208	28,091	28,113	26,533	27,148	28,075	27,434	24,827	20,560
	Cu	'000 lbs	832,206		100%		12,563	42,649	51,112	49,470	49,092	53,932	50,327	39,640	41,101	37,716	35,660	49,781	42,785	43,125	40,624	42,032	42,391	40,091	35,748	32,370
	Pd	oz	-		0%		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Pt	oz	-		0%		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Co	'000 lbs	57,462		100%		829	2,834	2,951	2,897	3,235	3,086	3,159	2,905	2,753	2,700	3,021	3,442	3,331	3,162	2,963	2,978	2,982	3,105	2,806	2,324
PRODUCTS	Nickel	'000 lbs	511,667		100%	Ni metal	7,643	25,864	26,615	25,999	28,816	27,635	28,932	25,632	24,730	23,964	25,848	29,208	28,091	28,113	26,533	27,148	28,075	27,434	24,827	20,560
	Copper	'000 lbs	832,206		100%	Cu metal	12,563	42,649	51,112	49,470	49,092	53,932	50,327	39,640	41,101	37,716	35,660	49,781	42,785	43,125	40,624	42,032	42,391	40,091	35,748	32,370
	Pd	oz	-																							
	Pt	oz	-																							
	Cobalt	'000 lbs	57,462		100%	Co metal	829	2,834	2,951	2,897	3,235	3,086	3,159	2,905	2,753	2,700	3,021	3,442	3,331	3,162	2,963	2,978	2,982	3,105	2,806	2,324
	Sulphuric Acid	tonnes	17,026,812				305,260	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	887,144	752,965
	Elemental Sulphur	tonnes	-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Hematite	tonnes	14,706,868				263,668	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	766,268	650,372
	PGE Residue	tonnes	4,677,541				83,860	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	243,713	206,852
					24-Nov-11	Oct 2011 memo																				
SALES PRICES	Exchange Rate	C$/US$			1.00	1.00																				
	Acid	US$/t	$ 85.00		$ 85.00	80.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00	$ 85.00
	Ni	US$ / lb	$ 10.00		$ 10.00	9.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00	$ 10.00
	Cu	US$ / lb	$ 3.50		$ 3.50	3.15	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50	$ 3.50
	Co	US$ / lb	$ 20.00		$ 20.00	17.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00
	Hematite	US$/t	$ 125.00		$ 125.00	100.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00	$ 125.00

			Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20
GROSS REVENUE	Acid	C$'000	$ 1,447,279				$ 25,947	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 75,407	$ 64,002
	Ni	C$'000	$ 5,116,674				$ 76,428	$ 258,637	$ 266,146	$ 259,990	$ 288,162	$ 276,354	$ 289,324	$ 256,319	$ 247,301	$ 239,639	$ 258,485	$ 292,082	$ 280,914	$ 281,132	$ 265,327	$ 271,476	$ 280,754	$ 274,344	$ 248,265	$ 205,596
	Cu	C$'000	$ 2,912,722				$ 43,969	$ 149,271	$ 178,890	$ 173,146	$ 171,821	$ 188,761	$ 176,146	$ 138,739	$ 143,853	$ 132,006	$ 124,809	$ 174,233	$ 149,749	$ 150,936	$ 142,182	$ 147,111	$ 148,367	$ 140,319	$ 125,117	$ 113,295
	Co	C$'000	$ 1,149,243				$ 16,585	$ 56,682	$ 59,012	$ 57,938	$ 64,693	$ 61,710	$ 63,179	$ 58,097	$ 55,068	$ 54,001	$ 60,415	$ 68,843	$ 66,614	$ 63,246	$ 59,268	$ 59,556	$ 59,636	$ 62,106	$ 56,113	$ 46,481
	Hematite	C$'000	$ 1,838,358				$ 32,958	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 95,784	$ 81,296
	TOTAL	**C$'000**	**$ 12,464,276**				**$ 195,888**	**$ 635,781**	**$ 675,239**	**$ 662,265**	**$ 695,867**	**$ 698,016**	**$ 699,840**	**$ 624,346**	**$ 617,413**	**$ 596,836**	**$ 614,899**	**$ 706,348**	**$ 668,468**	**$ 666,505**	**$ 637,968**	**$ 649,333**	**$ 659,949**	**$ 647,959**	**$ 600,686**	**$ 510,670**
SALES CHARGES				% Moisture																						
	Acid Shipping	C$'000	$ 1,430,252		$ 84.00	US$/t	$ 25,642	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 74,520	$ 63,249
	Metal Shipping	C$'000	$ 44,495	0%	$ 70.00	US$/t	$ 668	$ 2,265	$ 2,562	$ 2,488	$ 2,576	$ 2,688	$ 2,617	$ 2,165	$ 2,178	$ 2,044	$ 2,049	$ 2,617	$ 2,356	$ 2,362	$ 2,226	$ 2,291	$ 2,332	$ 2,243	$ 2,012	$ 1,754
	Hematite Shipping	C$'000	$ 1,163,313	13%	$ 70.00	US$/t	$ 20,856	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 60,612	$ 51,444
	TOTAL	C$'000	**$ 2,638,060**				**$ 47,166**	**$ 137,397**	**$ 137,694**	**$ 137,620**	**$ 137,708**	**$ 137,820**	**$ 137,749**	**$ 137,297**	**$ 137,310**	**$ 137,176**	**$ 137,181**	**$ 137,749**	**$ 137,488**	**$ 137,494**	**$ 137,358**	**$ 137,423**	**$ 137,464**	**$ 137,375**	**$ 137,144**	**$ 116,448**
NET SMELTER RETURN	NSR	C$'000	$ 9,826,216				$ 148,722	$ 498,384	$ 537,546	$ 524,645	$ 558,159	$ 560,197	$ 562,091	$ 487,049	$ 480,103	$ 459,660	$ 477,718	$ 568,599	$ 530,980	$ 529,011	$ 500,609	$ 511,910	$ 522,485	$ 510,584	$ 463,542	$ 394,222
	Royalty	C$'000	$ 196,524		2%		$ 2,974	$ 9,968	$ 10,751	$ 10,493	$ 11,163	$ 11,204	$ 11,242	$ 9,741	$ 9,602	$ 9,193	$ 9,554	$ 11,372	$ 10,620	$ 10,580	$ 10,012	$ 10,238	$ 10,450	$ 10,212	$ 9,271	$ 7,884
	NSR after Royalty	**C$'000**	**$ 9,629,692**				**$ 145,748**	**$ 488,416**	**$ 526,795**	**$ 514,152**	**$ 546,995**	**$ 548,993**	**$ 550,850**	**$ 477,308**	**$ 470,501**	**$ 450,467**	**$ 468,164**	**$ 557,227**	**$ 520,360**	**$ 518,431**	**$ 490,597**	**$ 501,672**	**$ 512,035**	**$ 500,373**	**$ 454,271**	**$ 386,337**
	Unit	**C$/t milled**	**$ 238.86**				**$ 201**	**$ 232**	**$ 251**	**$ 245**	**$ 260**	**$ 261**	**$ 262**	**$ 227**	**$ 224**	**$ 215**	**$ 223**	**$ 265**	**$ 248**	**$ 247**	**$ 234**	**$ 239**	**$ 244**	**$ 238**	**$ 216**	**$ 217**
OPERATING COSTS				Unit Rates																						
	Open Pit Mining	C$'000	$ 13,618		3.83	C$/t moved	$ 5,584	$ 8,034	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
	UG Mining	C$'000	$ 1,748,534		45.36	C$/t		$ 88,406	$ 113,563	$ 96,751	$ 98,538	$ 94,304	$ 95,633	$ 96,052	$ 91,074	$ 89,207	$ 89,219	$ 89,210	$ 89,206	$ 89,202	$ 89,226	$ 89,220	$ 89,218	$ 89,215	$ 89,214	$ 82,073
	Process	C$'000	$ 1,020,246		25.31	C$/t	$ 18,362	$ 53,364	$ 53,161	$ 53,144	$ 53,141	$ 53,155	$ 53,148	$ 53,149	$ 53,138	$ 53,130	$ 53,144	$ 53,134	$ 53,129	$ 53,123	$ 53,154	$ 53,146	$ 53,144	$ 53,140	$ 53,138	$ 45,101
	Site Services	C$'000	$ 333,025		8.26	C$/t	$ 5,994	$ 17,419	$ 17,353	$ 17,347	$ 17,346	$ 17,351	$ 17,349	$ 17,349	$ 17,345	$ 17,342	$ 17,347	$ 17,344	$ 17,342	$ 17,340	$ 17,350	$ 17,348	$ 17,347	$ 17,346	$ 17,345	$ 14,722
	G&A	C$'000	$ 449,765		11.16	C$/t	$ 8,095	$ 23,525	$ 23,436	$ 23,428	$ 23,427	$ 23,433	$ 23,430	$ 23,430	$ 23,426	$ 23,422	$ 23,428	$ 23,423	$ 23,421	$ 23,419	$ 23,432	$ 23,429	$ 23,428	$ 23,426	$ 23,425	$ 19,882
	TOTAL	**C$'000**	**$ 3,565,189**				**$ 38,035**	**$ 190,748**	**$ 207,513**	**$ 190,671**	**$ 192,452**	**$ 188,243**	**$ 189,560**	**$ 189,981**	**$ 184,983**	**$ 183,101**	**$ 183,138**	**$ 183,111**	**$ 183,098**	**$ 183,084**	**$ 183,163**	**$ 183,144**	**$ 183,137**	**$ 183,127**	**$ 183,122**	**$ 161,778**
		C$/t milled	**$ 88.43**				**$ 52.42**	**$ 90.46**	**$ 98.79**	**$ 90.80**	**$ 91.65**	**$ 89.62**	**$ 90.26**	**$ 90.46**	**$ 88.10**	**$ 87.22**	**$ 87.21**	**$ 87.21**	**$ 87.22**	**$ 87.22**	**$ 87.21**	**$ 87.21**	**$ 87.21**	**$ 87.21**	**$ 87.21**	**$ 90.78**
	Cash Cost (net of byproduct credits)	**US$/lb Ni Eq.**	**$ 3.39**				**$ 2.14**	**$ 3.59**	**$ 3.66**	**$ 3.41**	**$ 3.24**	**$ 3.15**	**$ 3.16**	**$ 3.66**	**$ 3.60**	**$ 3.72**	**$ 3.58**	**$ 3.01**	**$ 3.22**	**$ 3.23**	**$ 3.41**	**$ 3.34**	**$ 3.27**	**$ 3.35**	**$ 3.69**	**$ 3.85**
OPERATING CASH FLOW		**US$'000**	**$ 6,064,502**				**$ 107,713**	**$ 297,668**	**$ 319,282**	**$ 323,480**	**$ 354,544**	**$ 360,750**	**$ 361,289**	**$ 287,328**	**$ 285,518**	**$ 267,365**	**$ 285,026**	**$ 374,116**	**$ 337,262**	**$ 335,346**	**$ 307,434**	**$ 318,528**	**$ 328,898**	**$ 317,246**	**$ 271,149**	**$ 224,559**
CAPITAL COSTS	Mine	C$'000	$ 151,358	$ -	$ 24,863	$ 59,790	$ 66,705	$ -	$ -																	
	Process	C$'000	$ 788,068	$ 2,649	$ 467,732	$ 317,687	$ -	$ -	$ -																	
	Infrastructure	C$'000	$ 94,490	$ 15,758	$ 44,308	$ 34,423	$ -	$ -	$ -																	
	Indirects	C$'000	$ 269,684	$ 26,315	$ 105,259	$ 111,795	$ 26,315	$ -	$ -																	
	Contingency	C$'000	$ 196,551	$ 19,655	$ 78,620	$ 78,620	$ 19,655	$ -	$ -																	
	Reclamation	C$'000	$ 35,889																							$ 35,889
	Sustaining Capital	C$'000	$ 44,194				$	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	$ 2,455	
	TOTAL	**C$'000**	**$ 1,580,232**	**$ 64,377**	**$ 720,783**	**$ 602,316**	**$ 112,674**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 2,455**	**$ 35,889**
NET PRE-TAX CASH FLOW	**Per Year**	**C$'000**	**4,484,270**	**(64,377)**	**(720,783)**	**(602,316)**	**(4,961)**	**295,212**	**316,827**	**321,025**	**352,088**	**358,295**	**358,834**	**284,873**	**283,063**	**264,910**	**282,571**	**371,661**	**334,806**	**332,891**	**304,979**	**316,073**	**326,443**	**314,790**	**268,694**	**188,671**
	Cumulative	**C$'000**		**(64,377)**	**(785,160)**	**(1,387,476)**	**(1,392,437)**	**(1,097,224)**	**(780,398)**	**(459,372)**	**(107,284)**	**251,011**	**609,845**	**894,718**	**1,177,780**	**1,442,691**	**1,725,262**	**2,096,923**	**2,431,729**	**2,764,620**	**3,069,599**	**3,385,672**	**3,712,115**	**4,026,905**	**4,295,599**	**4,484,270**
	Payback Years	**5.3**	**5.3**				1.0	1.0	1.0	1.0	1.0	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Weeks	**276**	**276**				52	52	52	52	52	16	0	0	0	0	0	0	0	0	0	0	0	0	0	0
UNIT PRODUCTION COSTS	Operating (net of byproduct credits)	US$/lb Ni					2.14	3.59	3.66	3.41	3.24	3.15	3.16	3.66	3.60	3.72	3.58	3.01	3.22	3.23	3.41	3.34	3.27	3.35	3.69	3.85
	Life of Mine Capital	US$/lb Ni																								
	TOTAL	US$/lb Ni																								
	IRR			**16.8%**																						
NET PRESENT VALUE	**C$ Millions**	**5%**		**$1,872**																						
		8%		**$1,061**																						
		10%		**$692**																						
		15%		**$123**																						

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CASH FLOW ANALYSIS

Considering the Project on a stand-alone basis, the undiscounted pre-tax cash flow is $4,484 million over the 20 year mine life. Operating cash flow averages $303 million per year, and simple payback of pre-production capital is achieved after 5.3 years of operation.

The pre-tax Internal Rate of Return (IRR) is 16.8%. Pre-tax Net Present Value (NPV) of the project at various discount rates is as follows:

- Pre-tax NPV @ 5% $1,872 million
- Pre-tax NPV @ 8% $1,061 million
- Pre-tax NPV @ 10% $ 692 million
- Pre-tax NPV @ 15% $ 123 million

In RPA's opinion, a discount rate of 10% is appropriate for Base Case evaluation at this stage of the Project. The economic analysis contained in this report is based, in part, on Inferred Resources, and is preliminary in nature. Inferred resources are considered too geologically speculative to have mining and economic considerations applied to them and to be categorized as Mineral Reserves. There is no certainty that the mine schedules and economic forecasts on which this report is based will be realized.

CURRENT PRICE CASE

The Project evaluated at current metal prices and exchange rate (November 22, 2011) yields an undiscounted pre-tax cash flow of $3.3 billion, and IRR of 13.6%. The following sales prices and exchange rate were used:

- US$8.05 per pound Ni
- US$3.36 per pound Cu
- US$14.50 per pound Co
- US$100 per tonne Fe
- US$100 per tonne acid
- US$/C$ = 1.05

At the following discount rates, the pre-tax NPV using current prices and exchange is as follows:

- Pre-tax NPV @ 5% $1,286 million

- Pre-tax NPV @ 8% $642 million
- Pre-tax NPV @ 10% $352 million
- Pre-tax NPV @ 15% ($90) million

Current prices are lower than those projected for the LOM cash flow estimates.

SENSITIVITY ANALYSIS

Project risks can be identified in both economic and non-economic terms. Key economic risks were examined by running cash flow sensitivities to head grades, product prices, operating costs, capital costs and exchange rate. The results are shown in Figure 22-1 and Table 22-2. The cash flow was estimated over a +20% to -20% variation from the Base Case for each variable.

The Project cash flow is most sensitive to changes in the product prices, followed by exchange rate, head grade, capital cost, and operating cost.

FIGURE 22-1 SENSITIVITY ANALYSIS

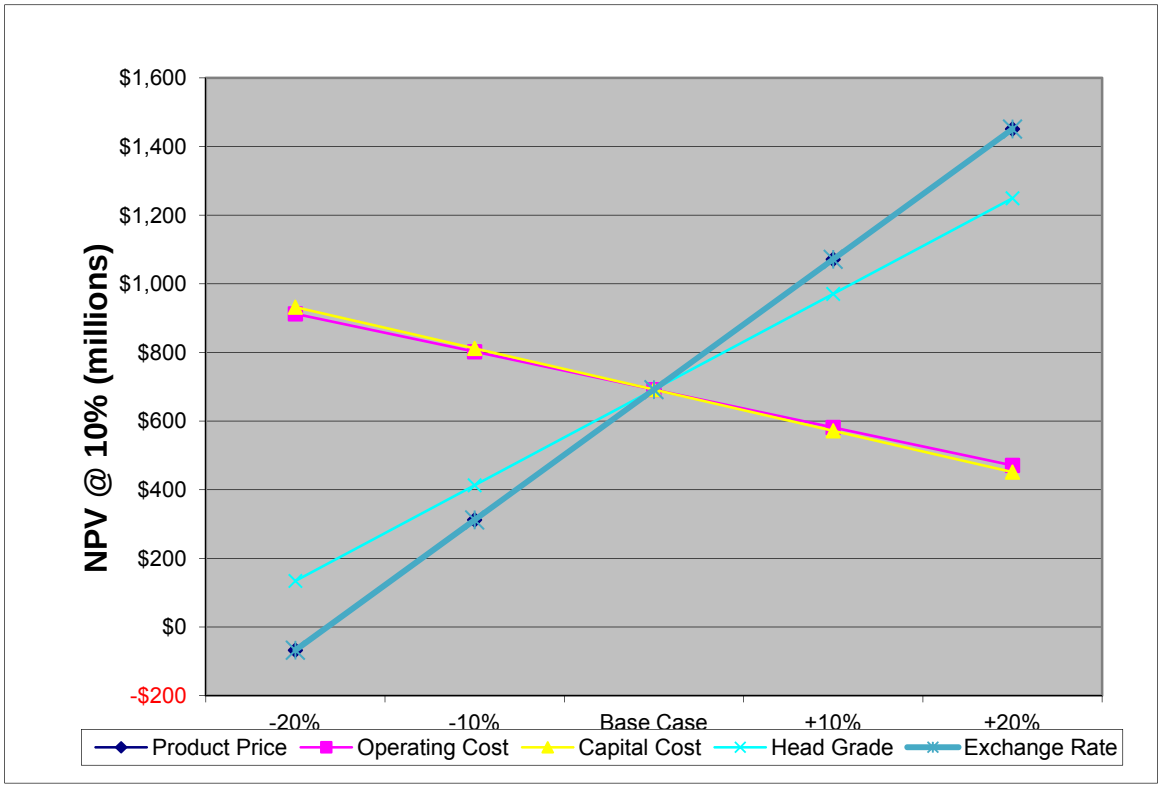


TABLE 22-2 SENSITIVITY ANALYSES
STARFIELD RESOURCES INC – FERGUSON LAKE

Parameter Variables	Units	-20%	-10%	Base	+10%	+20%
Product Price	US$/lb Ni	8.0	9.0	10.00	11.0	12.0
Exchange Rate	US$/C$	0.8	0.9	1.00	1.10	1.20
Head Grade	% Ni	0.49	0.55	0.61	0.67	0.73
Operating Cost	$/t	70.75	79.59	88.43	97.28	106.12
Capital Cost	$ millions	1,264	1,422	1,580	1,738	1,896

Pre-Tax NPV @ 10%	Units	-20%	-10%	Base	+10%	+20%
Product Price	$ millions	-67	312	692	1,072	1,451
Exchange Rate	$ millions	-67	312	692	1,072	1,451
Head Grade	$ millions	134	413	692	971	1,249
Operating Cost	$ millions	913	802	692	581	471
Capital Cost	$ millions	932	811	692	572	452

RPA notes that while Table 22-2 shows variations to product price in relation to nickel only, sensitivity factors were applied to all payable products.

PRICE SENSITIVITY UPDATE

Due to recent volatility seen in the market, the effects of wider ranges of product prices were examined. The price ranges and corresponding discounted (10%) pre-tax cash flows are shown in Table 22-3 and Figure 22-2 below. Unlike the previous table, product prices were varied one at a time in this analysis.

TABLE 22-3 PRODUCT PRICE SENSITIVITY UPDATE
Starfield Resources Inc – Ferguson Lake

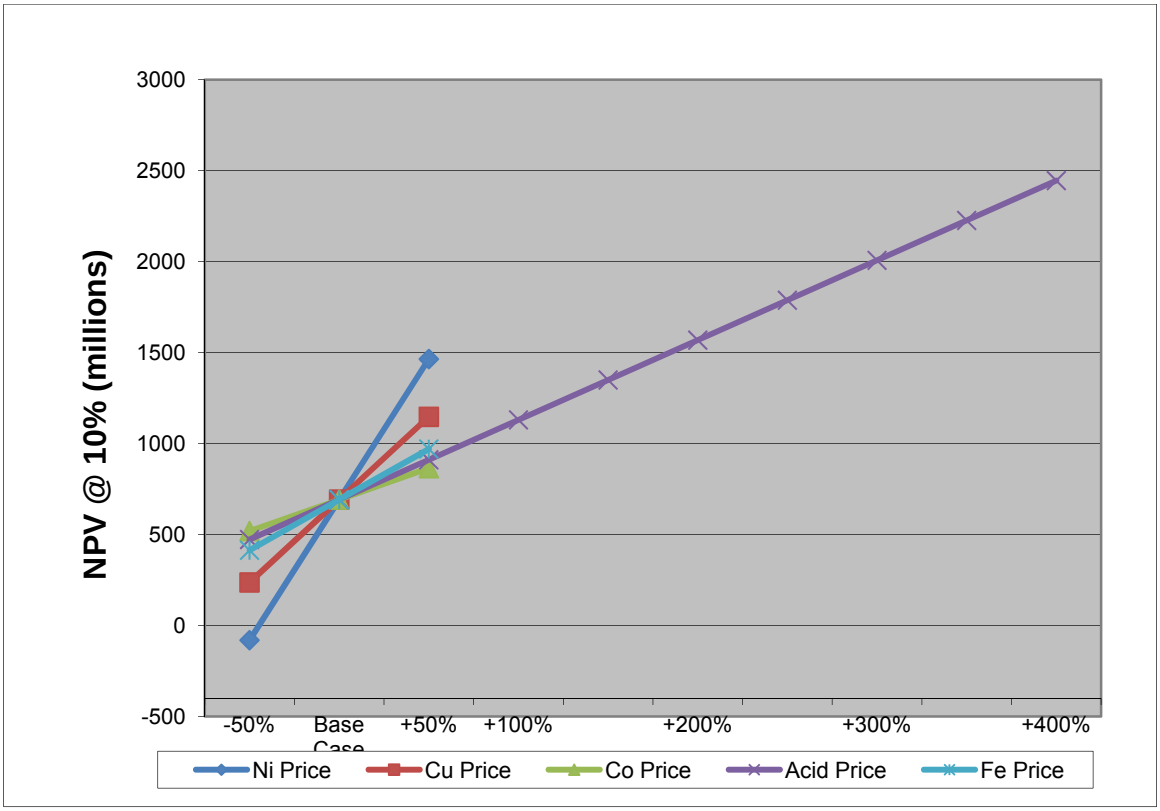
Parameter	Units					
Acid Price	US$/t	42.5*	85	127.5	170	255
Variation	%	-50%	Base	+50%	+200%	+300%
NPV@10%	$ millions	473	692	911	1,130	1,569
Ni Price	US$/lb Ni		5.00	10.00	15.00	20.00
Variation	%		-50%	Base	+150%	+200%
NPV@10%	$ millions		-81	692	1,464	2,237
Cu Price	US$/lb Cu		1.75	3.50	5.25	7.00
Variation	%		-50%	Base	+150%	+200%
NPV@10%	$ millions		237	692	1,147	1,602
Co Price	US$/lb Co		10.00	20.00	30.0	40.0
Variation	%		-50%	Base	+150%	+200%
NPV@10%	$ millions		519	692	865	1,038
Fe Price	US$/t	-	63**	125	188	250
Variation	%	-	-50%	Base	+150%	+200%
NPV@10%	$ millions	-	413	692	970	1,249

Notes:

* The shipping cost for acid is estimated at 84 US$/t. It is assumed that if the Acid Price falls below the transport cost it will not be sold.

** The shipping cost for Iron (Hematite) is estimated at 70 US$/t. It is assumed that if the Fe Price falls below the transport cost it will not be sold.

FIGURE 22-2 PRODUCT PRICE SENSITIVITY ANALYSIS



All payable products in Figure 22-2 are shown within recent price ranges for that particular product in relation to Base Case prices.

The Project cash flow is most sensitive to changes in the product price for nickel followed by all other payables, each of which have a similar effect on the Project cash flow within ±50% of Base Case prices. The price of acid is the most difficult to determine as the market is very sensitive to the annual demand from client and the global market is not large compared to other products. The table and chart above provide an idea of the sensitivity over the price ranges shown.

23 ADJACENT PROPERTIES

There are no adjacent properties of material interest at the present time.

RPA notes that uranium exploration in the area, such as at AREVA Resources Canada Inc.'s Kiggavik property (located approximately 80 km west of Baker Lake) indicates potential for local acid consumption in the future. Should this be the case, there is potential for a positive effect on the economics of the Ferguson Lake Project, due to reductions in shipping costs for the acid product.

24 OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

RPA reviewed the Project at the level of a PEA and concludes that under the Base Case assumptions, the Project has potential economic viability. In RPA's opinion, Starfield should continue work to advance the Project, by gathering information and undertaking studies in advance of completing Pre-feasibility Study followed by a Feasibility Study.

Separate comments on the various aspects of the Project follow.

RESOURCES

Mineral Resources estimated for the Project include an Indicated Resource of 15.8 million tonnes grading 0.65% Ni, 0.99% Cu, and 0.07% Co and an Inferred Resource of 30.2 million tonnes grading 0.67% Ni, 1.00% Cu, and 0.05% Co. Mineral Reserves have not been estimated.

Approximately 57% of the Mineral Resources included in the PEA are Inferred Resources from the Main West and West Extension zones. Infill drilling will be required to upgrade some portion of these resources to the Measured or Indicated categories.

Exploration work to date has focused on massive sulphide portions of the deposit, with early work sampling only for nickel and copper. Resource estimates for the East Zone lack cobalt and precious metal grade estimates, as a high proportion of the drilling in the zone was completed in the early programs. The East Zone was not included in the mine schedule for this PEA.

Recent drilling programs have tested low-sulphide, high- PGE potential, largely located below the massive sulphide lenses, with results that indicate the presence of potentially economic quantities of PGEs. The PGEs are not considered in the present Project economics.

MINING

Mechanized cut and fill mining is proposed to provide the best balance between productivity, required to achieve the production rate of 6,000 tpd, and selectivity, required to minimize dilution in the shallow-dipping mineralized lenses.

The significant horizontal extent of the deposit and the relatively narrow lenses will require a large quantity of level and internal ramp development to provide sufficient cut and fill faces for a continuous 6,000 tpd operation. In RPA's opinion, the proposed production rate is in the upper range of achievable rates, given the deposit geometry as presently understood.

The underground design parameters in this report are preliminary and further information is required for additional study and evaluation. Some of the key areas to be considered are:

- Data on geothermal gradient and rock temperatures
- Information on stability of rock when permafrost thaws
- Groundwater conditions (if any)
- Geotechnical information to assess stability of openings

There is potential to extend the mine life in future studies, by inclusion of the East Zone or other mineralized areas on the property.

PROCESSING AND TESTWORK

The recommended process will consist of:

- A concentrator producing a bulk sulphide concentrate at the mine site
- A concentrate slurry pipeline between the mine and the hydrometallurgical plant
- A hydrometallurgical plant located in Arviat

Testwork that was carried out from 2001 to 2008 by Kingston Process Metallurgy, SGS Lakefield, and NeoFerric has generated a flowsheet on which to assess the economic viability of the Project at the scoping level. Additional testwork was completed in 2009 at McGill University and in 2010/2011 at SGS Lakefield to assess oxidation and hydrolysis steps. This testwork indicated that continuous oxidation of ferrous chloride solution was successful at 120°C and produced a precipitate consisting of 97% hematite. Continuous hydrolysis of the ferric chloride solution was successful in producing a precipitate with 69% iron and 0.9% chloride and hydrochloric acid at 36% strength.

During 2011, a flotation test program was carried out to assess if the Ferguson Lake ore was amenable to upgrading by means of flotation prior to further treatment at the

hydrometallurgical process plant. A series of bench scale flotation tests established that it was possible to recover essentially all the sulphide minerals into a bulk sulphide concentrate while also rejecting about 80% of the gangue minerals.

The major unit operations (primary and secondary leaching, oxidation and hydrolysis) have been successfully carried out in continuous mini-pilot plant operation at SGS.

The process oxidation and hydrolysis steps are novel, and are not commercially proven unit operations. Fundamental measurements of heat capacity, thermal conductivity, viscosity, and other properties are required in order to determine the required heat transfer coefficients to size heat transfer surfaces, or preferably, direct measurements of heat transfer at a larger scale than has been completed thus far.

The oxidation step appears to be controlled by the rate at which oxygen is transferred from the gas phase to the bulk liquid phase. This implies that agitation is extremely important, and further work is required to properly scale up the oxidation step.

The process has the potential to generate stable, non-toxic tailings, and most importantly, may recover a sufficient quantity of the intrinsic energy in the ore to make it self-sustaining in energy usage. This means that there may be no need to import fuel oil for generating power on-site, which would otherwise be necessary in such a remote location. The energy recovered is potentially sufficient to operate both the hydrometallurgical process plant, as well as the mine, concentrator, and concentrate pipeline pump stations.

There are a number of uncertainties in the process flowsheet where it has been necessary to make assumptions. The most important of these are the iron and sulphur contents of the ore, and its accompanying mineralogy, and secondly, the lack of fundamental data in the scientific literature to enable, in particular, a rigorous energy balance to be carried out.

No formal account has been taken of the precious metals content of the ore, since no process for their recovery has yet been properly evaluated, however, it is clear that their

recovery is mandated, even as a relatively low grade concentrate, since they will comprise a significant revenue stream.

There is also potential to process low-sulphide and/or base metal, high-PGE material, not currently included in Mineral Resources. Low-sulphur-content material will not provide the energy advantages of the massive sulphide material, however, there may be value in considering a process unit that can recover PGEs from both low-sulphide and/or base metal material and from the secondary leach residue.

The process risk assessment has been updated to reflect new work since the previous risk exercise, and has been reviewed by Arithmetek and RPA. Results of this assessment show that the majority of the process has low associated risks. Medium-risk items include the oxidation and hydrolysis components of the proposed process as these areas have not yet been scaled up to commercial plant size, however, these risks can likely be mitigated through further testwork. The iron hydrolysis and HCl recovery circuit is a technique originally developed by PORI Inc. in the 1970s, and piloted by SMS Siemag for four years since 2002.

ACID

The Project will produce a large volume of high quality sulphuric acid that will be shipped to market over a short shipping season. In order to achieve the forecast revenue from acid, it will be important for the Project to develop a long term customer base that can handle and store the large volume of product. It will also be important to identify a source of the large-volume specialty ships required to transport the acid from the process site to market during the short shipping season window. The PEA includes acid storage facilities with a capacity of nine months per year of acid production.

ENVIRONMENT

The proposed process features significant advantages over conventional industry operations for similar projects. The process includes potential power generation from the sulphur contained in the ore, which would otherwise be generated by diesel combustion (annual requirements in the order of 100 million litres of diesel), with attendant greenhouse gas emissions.

The Project includes some environmental management issues beyond those considered typical for northern mines, such as large-scale acid storage, handling and transportation, and concentrate transportation by slurry pipeline. Ongoing investigation, examination of alternatives, and best-practice design with due consideration for the environment must remain a high priority as the Project is advanced, if permitting is to proceed smoothly.

MARKETING

The prices for certain products and particularly the sulphuric acid are difficult to assess over a long term. The acid price is specific to individual client-supplier agreements that can vary significantly from region to region. RPA has assumed an acid price based on available information, however, the actual price realized may vary significantly. Equally, acid transportation costs are difficult to determine and the cost was assumed based on recent prices that may also vary considerably over the Project life.

26 RECOMMENDATIONS

RPA makes the following recommendations to proceed with the assessment of the potential development of the Project.

1. Complete sampling necessary to evaluate cobalt, platinum, palladium, iron, and sulphur grades for the East Zone. This will require additional diamond drilling.

2. Conduct a sampling program to evaluate iron and sulphur grades, and mineralogy for the Main West Zone and the West Extension. A program of iron and sulphur analysis was completed for part of the Main West Zone but not for the West Extension Zone. Determine expected variations and iron and sulphur process feed grades.

3. Carry out infill drilling to upgrade Inferred Resources to Indicated Resources, and to generate material for metallurgical testing. The infill drilling has only been partially achieved but no material has been generated for additional metallurgical testing. Additional drilling is recommended in the West Zone of the property, as follows:
    a. Selected holes in the eastern part of the West Zone for a total of 2,000 m.
    b. Selected holes in the western half of the West Zone to reduce drill spacing to 30 m and to provide the basis for upgrading resources in this area to indicated. Approximately 5,000 m is estimated.

4. Carry out evaluation of other zones of mineralization on the property, including drilling on the 119 Extension Zone, to follow up previous results as follows:
    a. Two holes on section, up dip and down dip, of hole FL11-432
    b. A series of holes to test for zone development between holes FL11-432 and FL04-174
    c. Two holes on section, up dip and down dip, of hole FL04-200
    d. A series of holes to test for a potential zone southwest of hole FL04-200

5. Evaluate Mineral Resources for the East Zone and the West Extension using block modelling, once additional drill and sample results are available.

6. Carry out interpretation of the low-sulphide, high-PGE mineralization and relationship with adjacent massive sulphide intercepts Drill test the interpretation and if warranted complete updated resource estimate.

7. Complete outcrop stripping, or trenching, or other development to facilitate exposure of mineralization and confirm/evaluate the geometry and continuity of massive sulphide lenses. This could be completed in conjunction with advanced Project development study.

8. Gather geotechnical information and permafrost characteristics, for use in underground mine design.

9. Complete a trade-off evaluation of shaft hoisting vs. ramp haulage for transportation of ore to surface, to determine the optimum split between the two options.

10. Evaluate mining of low-sulphide, high-PGE mineralization, generally located below the massive sulphide lenses, as an addition to the current mining scenario.

11. Evaluate processing scenarios for PGE recovery from gangue material, and from low-sulphide, high-PGE material, as a separate circuit within the current process.

12. Generate fundamental data (water and hydrochloric acid vapour pressure, water and chloride activity, boiling point elevation, solubility data) for more accurate process design.

13. Carry out primary and secondary leach tests at a higher temperature and/or salt concentration consistent with the output of the mass/energy balance to ensure that it is possible to do so.

14. Carry out testing for heat transfer to determine fundamental measurements of heat capacity, thermal conductivity, viscosity, etc., to determine heat transfer coefficients required to size the heat transfer surfaces or, preferably, direct measurements of heat transfer.

15. Carry out testing for materials of construction, particularly for heat transfer surfaces as the fluids are aggressive and titanium or tantalum will be required.

16. Complete additional work for the oxidation step, as agitation will be extremely important to ensure this step is properly scaled up.

17. Audit the mass/energy balance to ensure the most efficient use of combustion gas energy.

18. Test methods for concentrating the platinum and palladium content of the secondary leach residue. Include possibility of concurrent processing of low-sulphide, high-PGE feed.

19. Run a pilot plant program, integrating all process units into a continuous operation.

20. Undertake a detailed marketing study to identify specific customers for the sulphuric acid product from the Project.

21. Carry out a transport and logistics study to identify carriers capable of meeting acid shipping capacity requirements.

22. Evaluate logistics for movement of materials to site for construction and operation.

23. Continue baseline environmental programs.

24. Review comparable Arctic pipelines, to assess best practices for design and operation.

25. Continue to investigate alternative process site locations and slurry pipeline routes.

26. Continue discussions for an impact benefit agreement (IBA) with local communities in the Project area.

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28 DATE AND SIGNATURE PAGE

This report titled "Preliminary Economic Assessment of the Ferguson Lake Project, Nunavut, Canada" and dated November 30, 2011, was prepared and signed by the following authors:

(Signed & Sealed) "*Graham G. Clow*"

Dated at Toronto, ON
November 30, 2011

Graham G. Clow, P.Eng.
Principal Mining Engineer

(Signed & Sealed) "*Normand Lecuyer*"

Dated at Toronto, ON
November 30, 2011

Normand Lecuyer, P.Eng.
Principal Mining Engineer

(Signed & Sealed) "*Daniel J. Rolph*"

Dated at Toronto, ON
November 30, 2011

Daniel J. Rolph, P.Eng.
Senior Mining Engineer

(Signed & Sealed) "*James G. Lavigne*"

Dated at Toronto, ON
November 30, 2011

James G. Lavigne, P.Geo.
Associate Geologist

(Signed & Sealed) "*Holger Krutzelmann*"

Dated at Toronto, ON
November 30, 2011

Holger Krutzelmann, P.Eng.
Principal Metallurgist

29 CERTIFICATE OF QUALIFIED PERSON

GRAHAM G. CLOW

I, Graham G. Clow, P.Eng., as an author of this report entitled “Preliminary Economic Assessment of the Ferguson Lake Project, Nunavut, Canada”, prepared for Starfield Resources Inc., and dated November 30, 2011, do hereby certify that:

1. I am Principal Mining Engineer with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave., Toronto, ON, M5J 2H7.

2. I am a graduate of Queen’s University, Kingston, Ontario, Canada in 1972 with a Bachelor of Science degree in Geological Engineering and in 1974 with a Bachelor of Science degree in Mining Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Reg.# 8750507). I have worked as a mining engineer for a total of 35 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a consultant on numerous mining operations and projects around the world for due diligence and regulatory requirements
   - Senior Engineer to Mine Manager at seven Canadian mines and projects
   - Senior person in charge of the construction of two mines in Canada
   - Senior VP Operations in charge of five mining operations, including two in Latin America
   - President of a gold mining company with one mine in Canada
   - President of a gold mining company with one mine in Mexico

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Ferguson Lake Project.

6. I am responsible for overall preparation of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have prepared previous Technical Reports on the Ferguson Lake Project.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of November, 2011

(Signed & Sealed) "*Graham G. Clow*"

Graham G. Clow, P. Eng.

NORMAND L. LECUYER

I, Normand L. Lecuyer, P.Eng., as an author of this report entitled “Preliminary Economic Assessment of the Ferguson Lake Project, Nunavut, Canada”, prepared for Starfield Resources Inc., and dated November 30, 2011, do hereby certify that:

1. I am Principal Mining Engineer with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.

2. I am a graduate of Queen’s University, Kingston, Canada, in 1976 with a B.Sc. (Hons.) degree in Mining Engineering.

3. I am registered as a Professional Engineer in the provinces of Ontario (Reg.#26055251) and Québec (Reg.# 34914). I have worked as a mining engineer for a total of 35 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a consultant on numerous exploration and mining projects around the world for due diligence and regulatory requirements.
   - Vice-President Operations for a number of mining companies.
   - Mine Manager at an underground gold mine in Northern Ontario, Canada.
   - Manager of Mining/Technical Services at a number of base-metal mines in Canada and North Africa.
   - Vice-President Engineering at two gold operations in the Abitibi area of Quebec, Canada.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Project.

6. I am responsible for the preparation of Sections 15 to 19, 21 (process capital and operating costs), 22, and 24 and contributed to Sections 1, 2, 25, and 26 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of November, 2011

(Signed & Sealed) "*Normand Lecuyer*"

Normand L. Lecuyer, P.Eng.

DANIEL J. ROLPH

I, Daniel J. Rolph, P.Eng., as an author of this report entitled “Preliminary Economic Assessment of the Ferguson Lake Project, Nunavut, Canada”, prepared for Starfield Resources Inc., and dated November 30, 2011, do hereby certify that:

1. I am Senior Mining Engineer with Roscoe Postle Associates Inc. of Suite 501, 55 University Avenue, Toronto, ON, M5J 2H7.

2. I am a graduate of the University of Toronto, Toronto, Ontario, Canada in 2000 with a Bachelor of Applied Science and Engineering.

3. I am registered as a Professional Engineer in the Province of Ontario (Member # 100049692). I have worked as a licensed Mining Engineer for a total of five years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and reporting as a consultant on mining operations and projects around the world for due diligence and regulatory requirements.
   - Progressively senior engineering and operations positions with mining consultants and mine operators, with responsibilities including:
     - underground mine planning, design, and scheduling
     - underground and open pit operations supervision and contractor management
     - preparation and management of third party mining agreements
     - business planning activities including the preparation of capital and operating budgets
     - design and operation of underground cut-and-fill mining operations
   - geomechanical numerical stress modelling

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Project.

6. I am responsible for the preparation of Sections 18 and 19 and collaborated with my co-author on Sections 16, 21 (except process capital and operating costs), 22, 25, and 26 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 30th day of November, 2011

(Signed & Sealed) "*Daniel J. Rolph*"

Daniel J. Rolph, P.Eng.

JAMES G. LAVIGNE

I, James G. Lavigne, M.Sc., P.Geo., as an author of this report entitled "Preliminary Economic Assessment of the Ferguson Lake Project, Nunavut, Canada", prepared for Starfield Resources Inc., and dated November 30, 2011, do hereby certify that:

1. I am an Associate Geologist with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.

2. I am a graduate of Memorial University of Newfoundland in 1986 with a B.Sc. degree in geology and the University of Ottawa in 1991 with a M.Sc. degree in geology.

3. I am registered as a Professional Geoscientist in the Province of Ontario (Reg. # 1895). I have worked as a geologist for a total of 25 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Design and implementation of gold exploration programs in Canada and in Mexico including advanced exploration and resource delineation drill programs
   - Completion of numerous Mineral Resource Estimates for gold mineralization
   - Preparation of NI 43-101 Technical Reports

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I visited the Ferguson Lake Property on two occasions during the period of September to November 2006.

6. I am responsible for Sections 4 to 12, 14, and 23 and contributed to Sections 1, 25, and 26 of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have prepared previous Mineral Resource estimates and a Technical Report for the Ferguson Lake Project.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 30th day of November, 2011

(Signed & Sealed) "*James G. Lavigne*"

James G. Lavigne, M.Sc., P.Geo.

HOLGER KRUTZELMANN

I, Holger Krutzelmann, P. Eng., as an author of this report entitled “Preliminary Economic Assessment of the Ferguson Lake Project, Nunavut, Canada”, prepared for Starfield Resources Inc., and dated November 30, 2011, do hereby certify that:

1. I am Vice President, Metallurgy & Environment with Roscoe Postle Associates Inc. of Suite 501, 55 University Ave Toronto, ON, M5J 2H7.

2. I am a graduate of Queen’s University, Kingston, Ontario, Canada in 1978 with a B.Sc. degree in Mining Engineering (Mineral Processing).

3. I am registered as a Professional Engineer with Professional Engineers Ontario (Reg.# 90455304). I have worked in the mineral processing field, in operating, metallurgical, managerial; and engineering functions, for a total of 33 years since my graduation. My relevant experience for the purpose of the Technical Report is:

    - Reviews and reports as a metallurgical consultant on a number of mining operations and projects for due diligence and financial monitoring requirements
    - Senior Metallurgist/Project Manager on numerous gold and base metal studies for a leading Canadian engineering company.
    - Management and operational experience at several Canadian and U.S. milling operations treating various metals, including copper, zinc, gold and silver.

4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

5. I have not visited the Project.

6. I am responsible for the preparation of Sections 13 and 20 and parts of Section 21 (process capital and operating costs) of the Technical Report.

7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

8. I have had no prior involvement with the property that is the subject of the Technical Report.

9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report/Section Nos. I am responsible for in the Technical Report contains/contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 30th day of November, 2011

(Signed & Sealed) "*Holger Krutzelmann*"

Holger Krutzelmann, P.Eng.

30 APPENDIX 1

FERGUSON LAKE MINING LEASES AND MINERAL CLAIMS

TABLE 30-1 FERGUSON LAKE MINING LEASES AND MINERAL CLAIMS
Starfield Resources Inc. – Ferguson Lake

DISP'N #	DISP'N NAME	NTS1	NTS2	NTS3	NTS4	AC	REC DATE	ANNIV DATE *	REG'D HOLDER	USABLE $CREDIT	ANN RENTALS	NOTES
4922	4922	065I15				2,534.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$2,534.00	Ferg1(F47820)
4923	4923	065I15				2,695.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$2,695.00	Ferg2(F47821)
4924	4924	065I15				1,527.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$1,527.00	Ferg3(F47822)
4925	4925	065I14	065I15			2,640.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$2,640.00	Ferg4(F47831)
4926	4926	065I14	065I15			2,425.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$2,425.00	Ferg5(F47832)
4927	4927	065I14				1,732.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$1,732.00	Ferg6(F47833)
4928	4928	065I15				2,616.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$2,616.00	Ferg7(F47854)
4929	4929	065I15				2,401.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$2,401.00	Ferg10(F47857)
4930	4930	065I15				2,592.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$2,592.00	Ferg11(F47858)
4931	4931	065I15				2,773.00	07-Sep-07	07-Sep-11	Starfield Resources Inc.	$0.00	$2,773.00	Ferg12(F47859)
10 mining leases						23,935.00						
F47855	FERG8	065I15				2,582.50	25-Jun-99	25-Jun-09	Starfield Resources Inc.	$27,636.64		Rpt filed 31Mar09
F85380	KAM001	065I15				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85381	KAM002	065I15				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85382	KAM003	065I15				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85383	KAM004	065I15	065I16			2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85384	KAM005	065I15	065I16			2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85385	KAM006	065I15	065I16			2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85386	KAM007	065I15	065I16			2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85387	KAM008	065I15	065I16			2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85388	KAM009	065I15	065I16			2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85389	KAM010	065I15	065I16			2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85390	KAM011	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85391	KAM012	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85392	KAM013	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85393	KAM014	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85394	KAM015	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85395	KAM016	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85396	KAM017	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85397	KAM018	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85398	KAM019	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85399	KAM020	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85400	KAM021	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85401	KAM022	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85402	KAM023	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85403	KAM024	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85404	KAM025	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85405	KAM026	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85406	KAM027	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85407	KAM028	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85408	KAM029	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85409	KAM030	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85410	KAM031	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85411	KAM032	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85412	KAM033	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85413	KAM034	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09

DISP'N #	DISP'N NAME	NTS1	NTS2	NTS3	NTS4	AC	REC DATE	ANNIV DATE *	REG'D HOLDER	USABLE $CREDIT	ANN RENTALS	NOTES
F85414	KAM035	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85415	KAM036	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85416	KAM037	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85417	KAM038	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85418	KAM039	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85419	KAM040	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85420	KAM041	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85421	KAM042	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85422	KAM043	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85423	KAM044	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85424	KAM045	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85425	KAM046	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85426	KAM047	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85427	KAM048	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85428	KAM049	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85429	KAM050	065I16				2,582.50	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85436	KAM057	055L13	065I16			1,033.00	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85437	KAM058	065I16				1,033.00	18-Jan-05	18-Jan-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; L/N 18Feb09; Rpt filed 31Mar09
F85580	NUB001	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85581	NUB002	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85582	NUB003	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85583	NUB004	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85584	NUB005	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85585	NUB006	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85586	NUB007	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85587	NUB008	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85588	NUB009	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85589	NUB010	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85590	NUB011	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85591	NUB012	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85592	NUB013	065I13				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85593	NUB014	065I13	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85594	NUB015	065I13	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85595	NUB016	065I13	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85596	NUB017	065I13	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85597	NUB018	065I13	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85598	NUB019	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85599	NUB020	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85600	NUB021	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85601	NUB022	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85602	NUB023	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85603	NUB024	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85604	NUB025	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85605	NUB026	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85606	NUB027	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85607	NUB028	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85608	NUB029	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85609	NUB030	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85610	NUB031	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85611	NUB032	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09

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F85612	NUB033	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85613	NUB034	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85614	NUB035	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85615	NUB036	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85617	NUB038	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85618	NUB039	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85619	NUB040	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85620	NUB041	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85621	NUB042	065I14				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85622	NUB043	065I14	065I15			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85623	NUB044	065I14	065I15			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85624	NUB045	065I14	065I15			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85625	NUB046	065I14	065I15			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85626	NUB047	065I14	065I15			826.40	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85628	NUB049	065I14	065I15			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85629	NUB050	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85630	NUB051	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85631	NUB052	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85632	NUB053	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85633	NUB054	065I15				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85634	NUB055	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85636	NUB057	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85637	NUB058	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85638	NUB059	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85639	NUB060	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85640	NUB061	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85641	NUB062	065I15				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85642	NUB063	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85644	NUB065	065I15				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85645	NUB066	065I15				1,291.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85646	NUB067	065I15				1,291.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85647	NUB068	065I15				1,291.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85648	NUB069	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85649	NUB070	065I15				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85650	NUB071	065I15				1,549.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85652	NUB073	065I15				1,239.60	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85653	NUB074	065I15				2,324.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85654	NUB075	065I15				2,324.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85655	NUB076	065I15				2,324.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85656	NUB077	065I15				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85657	NUB078	065I15				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85658	NUB079	065I15				1,291.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85659	NUB080	065I15				1,291.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85660	NUB081	065I15				1,291.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85661	NUB082	065I15				1,291.25	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85662	NUB083	065I15				1,033.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85664	NUB085	065I13				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85665	NUB086	065I12	065I13			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85666	NUB087	065I12				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85667	NUB088	065I12				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09

DISP'N #	DISP'N NAME	NTS1	NTS2	NTS3	NTS4	AC	REC DATE	ANNIV DATE *	REG'D HOLDER	USABLE $CREDIT	ANN RENTALS	NOTES
F85668	NUB089	065I12				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85669	NUB090	065I12				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85670	NUB091	065I12				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85671	NUB092	065I12				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85672	NUB093	065I13	065I14			2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85673	NUB094	065I11	065I12	065I13	065I14	2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85674	NUB095	065I11	065I12			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85675	NUB096	065I11	065I12			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85676	NUB097	065I11	065I12			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85677	NUB098	065I11	065I12			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85678	NUB099	065I11	065I12			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85679	NUB100	065I11	065I12			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85680	NUB101	065I14				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85681	NUB102	065I11	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85682	NUB103	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85683	NUB104	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85684	NUB105	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85685	NUB106	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85686	NUB107	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85687	NUB108	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85688	NUB109	065I14				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85689	NUB110	065I11	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85690	NUB111	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85691	NUB112	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85692	NUB113	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85693	NUB114	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85694	NUB115	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85695	NUB116	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85696	NUB117	065I14				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85697	NUB118	065I11	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85698	NUB119	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85699	NUB120	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85700	NUB121	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85701	NUB122	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85702	NUB123	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85703	NUB124	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85704	NUB125	065I14				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85705	NUB126	065I11	065I14			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85706	NUB127	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85707	NUB128	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85708	NUB129	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85709	NUB130	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85710	NUB131	065I11				2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85711	NUB132	065I11				1,549.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85712	NUB133	065I14				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85713	NUB134	065I10	065I11	065I14	065I15	2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85714	NUB135	065I10	065I11			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85715	NUB136	065I10	065I11			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85716	NUB137	065I10	065I11			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85717	NUB138	065I10	065I11			2,582.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09

DISP'N #	DISP'N NAME	NTS1	NTS2	NTS3	NTS4	AC	REC DATE	ANNIV DATE *	REG'D HOLDER	USABLE $CREDIT	ANN RENTALS	NOTES
F85718	NUB139	065I11				1,549.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85719	NUB140	065I15				2,066.00	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85720	NUB141	065I10	065I15			1,549.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85721	NUB142	065I10				1,549.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85722	NUB143	065I10				1,549.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85723	NUB144	065I10				1,549.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85724	NUB145	065I10				1,549.50	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
F85725	NUB146	065I15				1,239.60	19-Apr-05	19-Apr-09	Starfield Resources Inc.	$0.00		S81 approv 31Oct07; Rpt filed 31Mar09
K10664	IAM001	065I15				1,569.00	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10665	IAM002	065I15				1,569.00	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10666	IAM003	065I15				1,569.00	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10667	IAM004	065I15				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10668	IAM005	065I15				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10669	IAM006	065I14	065I15			2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10670	IAM007	065I14	065I15			2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10671	IAM008	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10672	IAM009	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10673	IAM010	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10674	IAM011	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10675	IAM012	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10676	IAM013	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10677	IAM014	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10678	IAM015	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K10679	IAM016	065I14				2,582.50	06-Feb-08	06-Feb-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85896	YLK001	065I10	065I11			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85902	YLK002	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85903	YLK003	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85905	YLK004	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85906	YKL005	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85907	YKL006	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85908	YLK007	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85909	YLK008	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85910	YLK009	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85911	YLK010	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85912	YLK011	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85913	YLK012	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85914	YLK013	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85915	YLK014	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85916	YLK015	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85917	YLK016	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85918	YLK017	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85919	YLK018	065I10				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85920	YLK019	065I15	065I16			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85926	HPC001	065I14	065I15			258.26	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85927	HPC002	065I14	065I15			258.26	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85928	HPC003	065I15				258.26	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85929	HPC004	065I15				258.26	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85933	YLK020	065I15	065I16			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85934	YLK021	065I16				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85935	YLK022	065I16				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09

DISP'N #	DISP'N NAME	NTS1	NTS2	NTS3	NTS4	AC	REC DATE	ANNIV DATE *	REG'D HOLDER	USABLE $CREDIT	ANN RENTALS	NOTES
F85936	YLK023	065I16				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91591	HPC005	065I15				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91592	HPC006	065I10	065I15			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91593	HPC007	065I10	065I15			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91594	HPC008	065I10	065I15			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91595	HPC009	065I15				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91596	HPC010	065I15				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91597	HPC011	065I15				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91598	HPC012	065I15	065I16			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91599	HPC013	065I15	065I16			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91634	YAT004	065I13				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91635	YAT005	065I13				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91636	YAT006	065I13				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91637	YAT007	065I13				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91638	YAT008	065I13	065I11			2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91639	YAT009	065I14				2,501.94	26-Jun-08	26-Jun-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85897	FL001	065I15				1,530.00	08-Sep-08	08-Sep-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F91640	FL002	065I15				1,952.00	08-Sep-08	08-Sep-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85921	YAT001	065I13				2,501.94	20-Nov-08	20-Nov-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85922	YAT002	065I13				2,501.94	20-Nov-08	20-Nov-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
F85923	YAT003	065I13				2,501.94	20-Nov-08	20-Nov-10	Starfield Resources Inc.	$0.00		Rpt filed 31Mar09
K00878	YATH025	065I13	065I14			2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K00879	YATH026	065I13	065I14			2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K00880	YATH030	065I13				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K00881	YATH031	065I13				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K00882	YATH032	065I13				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K00883	YATH033	065I13				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K00884	YATH034	065I13				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K00885	YATH035	065I13				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K12474	YATH012	065I14				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K12475	YATH013	065I14				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K12476	YATH014	065I14				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K12477	YATH018	065I14				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K12478	YATH019	065I14				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K12479	YATH020	065I14				2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
K12480	YATH024	065I13	065I14			2,582.50	30-Nov-09	30-Nov-11	Starfield Resources Inc.	$0.00		
F85924	TY001	065I14				2,501.94	16-Nov-10	16-Nov-12	Starfield Resources Inc.	$0.00		
F85925	TY002	065I14				2,501.94	16-Nov-10	16-Nov-12	Starfield Resources Inc.	$0.00		
273 Mineral Claims						653,655.56						
10 Mining Leases & 273 Mineral Claims						677,590.56						

* All of the claims, with the exception of those with 2011 and 2012 anniversary dates are essentially “on hold” by the NU government pending their review of the recently submitted assessment reports and work filing documentation.

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31 APPENDIX 2

SGS MINERALS SERVICES METALLURGICAL TESTWORK – 2011

An Investigation into

TESTING OF OXIDATION AND HYDROLYSIS OF IRON CHLORIDE SALTS

prepared for

STARFIELD RESOURCES

Project 12323-001 Final Report
April 13, 2011

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Executive Summary

Continuous oxidation of a ferrous chloride solution was successful at 120ºC, producing a precipitate with 97% hematite.

Increasing the reactor temperature above 120ºC caused solids built-up in the reactor walls and decreasing the reactor temperature below 120ºC decreased the oxidation efficiency.

Both pitch-blade and rushton-turbine impellers were tried successfully; using the same conditions the rushton-turbine led to slightly higher oxidation efficiency, but it could not be tested in the long-run test due to a corrosion problem.

Continuous hydrolysis of a ferric chloride solution was successful, producing a precipitate with 69% iron and 0.9% chloride, and hydrochloric acid at 22%.

Water addition is a key factor in hydrolysis performance affecting solids composition, hydrolysis extension, acid production and acid strength. Water addition should be based on whether acid strength is more important than the amount of acid produced, or vice versa.

An increment in hydrolysis temperature increased hydrolysis extension, amount of acid produced and even acid strength. However, it also increased slurry viscosity and "line plugging" problems increased exponentially. Increasing temperature above 185ºC would make hydrolysis very "challenging" from the operational point of view.

Introduction

SGS Minerals Services in Lakefield, Ontario, Canada was asked to perform a series of tests to prove the concept of ferrous iron oxidation and ferric iron hydrolysis as part of a novel chloride based proposed leaching and metal recovery flowsheet for treating the Ferguson Lake ore. The current study (Phase 1 Roadmap) was completely based on synthetic chemicals.

Throughout the program, results were discussed with Mr. Michael Moran of Starfield Resources and Dr. Mike Dry consultant to Starfield. This document describes the testwork carried out under SGS Project Number 12323-001.

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Testwork Summary

1. Introduction

Starfield Resources is developing a novel circuit for the recovery of nickel, copper, cobalt, iron oxide, sulphuric acid and precious metals from the Ferguson Lake deposit in Nunavut, Canada. In this deposit, pentlandite is too finely disseminated in pyrrhotite for the production of a smelter-grade nickel concentrate at an acceptable nickel recovery, so established technology is not applicable and a novel chloride-based circuit is under development. The enabling aspect of this circuit is the hydrolysis of ferric chloride to hematite, releasing gaseous hydrochloric acid that is recycled. Conventional steps are used for separating and recovering the valuable metals from the chloride solution generated. A proposed flowsheet is included in Figure 1.

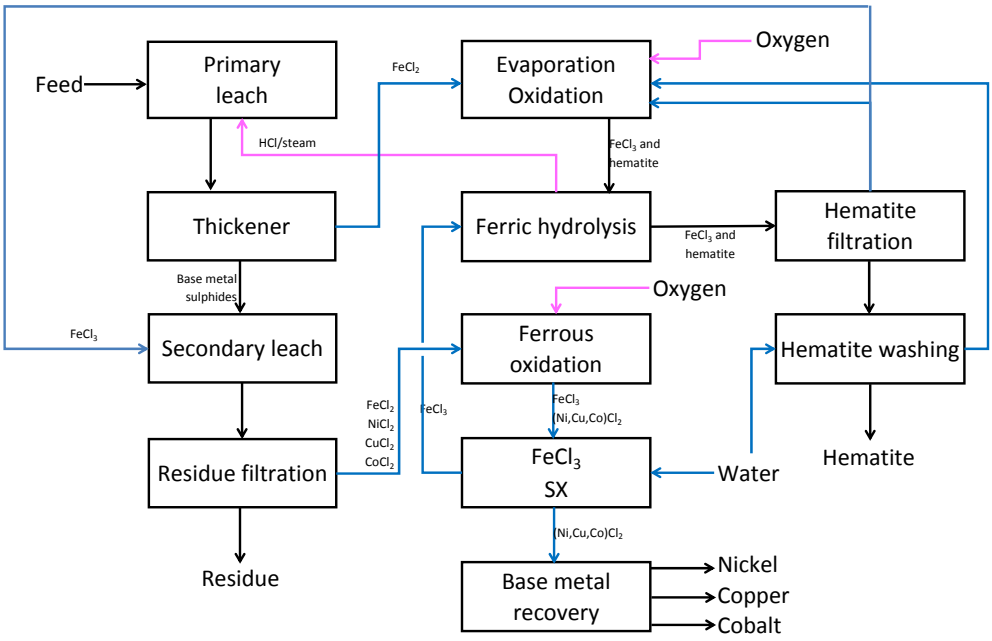


Figure 1 Preliminary Conceptual Overall Flowsheet

1.1. Roadmap

A multi phase roadmap was developed that described the various phases of the test program. It consisted of:

1. Phase 1 – bench top study of oxidation and hydrolysis steps using synthetic chemicals
2. Phase 2 – flotation of sulphide ore
3. Phase 3 – mini pilot plant – primary ore leaching

4. Phase 4 – bench top study of oxidation and hydrolysis steps using liquor produced in primary ore leach pilot plant
5. Phase 5 – mini pilot plant – secondary ore leaching + oxidation and hydrolysis
6. Phase 6 – bench scale downstream unit processes (iron polish, copper, nickel and cobalt recovery)
7. Phase 7 – select mini pilot plants of downstream unit processes

This report includes the work carried out under phase 1 of the roadmap program and provides proof of concept testing of the oxidation and hydrolysis stages of the flowsheet. Since real primary ore leach liquors were not yet available at this stage of the program, the proof of concept testing was carried out using synthetic feed liquors prepared by using synthetic chloride salts. The compositions of these liquors were based on the model output provided by Dr Mike Dry – consultant to Starfield Resources.

2. Test Procedures

A series of tests were carried out to prove the concept of the oxidation and hydrolysis of iron chloride salts. All the tests were carried out using a synthetic solution which simulated a primary ore leach discharge solution. Full test data sheets including detailed test instructions are available in Appendix A.

2.1. Oxidation Ttests

A highly concentrated ferrous chloride solution (see Table 1 for a typical feed solution composition) which also contained ferric chloride and base metal chlorides was subjected to oxidation using oxygen as reagent, in order to convert ferrous chloride to iron (III) oxide (hematite) and ferric chloride.

$$6FeCl_2 + 1.5O_2 \rightarrow 4FeCl_3 + Fe_2O_3 \qquad \textbf{[1]}$$

A photograph of the setup is included in Figure 2. Pre-heater and boiler temperature were controlled by an electrical heat mantle while the reactor temperature was controlled by passing hot oil from an oil bath through an inner jacket. Feed solution and bath solution were synthetic solutions and they were made up from solid reagents and melted in their respective vessels at the start of each test; each composition was derived from ASPEN modelling results provided to SGS by Starfield or its consultants.

Table 1 – Oxidation Tests - Feed Solution Composition

Cations	mg/L
Fe(II)	199024
Fe(III)	25424
Cu	9344
Ni	6726
Co	590
Mg	5280

The standard procedure included the following steps:

- The synthetic liquors (feed liquor, boiler bath and oxidation bath) were made up and heated up.
- Once they reached the required temperature feed was pumped out from the pre-heater into the boiler.
- The condensate from the boiler was drawn off and the solution overflowed to the reactor.
- At the point when the boiler solution began to overflow oxygen was added to the reactor. Oxygen sparging was carried out by the use of a U-shaped tube located under the impeller. Sparger consisted in a 1.2mm-diameter TEFLON tubing which was located inside the U-shaped glass tube.
- The condensate from the reactor was either drawn off or put back to the reactor.
- The discharge from the reactor was collected and diluted in water or dilute HCl to avoid crystallization of the dissolved salts.
- Subsequently, the discharge pulp was filtered and samples from the solution and the solids were submitted for assays. A summary of the tests conditions is shown in Table 2.

Prior to the start of the oxidation test series a bubble test was done to test different types of mixing regimes. Type of impeller, mixer speeds (rpm) and locations to determine the best conditions for gas dispersion were test parameters investigated.

Test 1 was carried out at 140ºC in the boiler and 150ºC in the reactor, using the pitch-blade impeller. These conditions were set based on a previous work done by McGill University. For this test the sparger was made from glass. Both boiler and reactor condensates were drawn off from their respective vessels. Discharge from the reactor was dilute in 10% HCl to dissolve any base metal chlorides trapped in the hematite. Test 2 was a boiler-only test and was carried out to determine the presence of solids -either precipitate or crystals- in the boiler at 120 ºC. Test 3 then was done at 120ºC in both vessels boiler and reactor. The condensate from the reactor was put back into the reactor. Due to the vibration caused by the mixing the glass sparger broke down after Test 1, a new sparger made from tantalum was used in the rest of the tests. Test 4 was performed using the same parameters as McGill university team did in order to corroborate their results and to prove the liability of SGS set-up. Test 5 was performed at 120ºC in the boiler and 130ºC in the reactor. The condensates from the boiler and the reactor were drawn off. Test 6 used the same conditions as Test 3 except the temperature was 110ºC in boiler and reactor. Test 7 was run at the same conditions as test 3 but this time a rushton-turbine impeller was used instead of the pitch-blade impeller used in the rest of the tests.

Figure 2 - Oxidation Test Experimental Set-up

Table 2 – Oxidation Tests – Tests Conditions

Condition		Test 1	Test 3	Test 4	Test 5	Test 6	Test 7
Target feed	mL/min	4.0	4.0	2.9	4.0	4.0	4.0
Boiler temperature	^{O}C	140	120	110	120	110	120
Reactor temperature	^{O}C	150	120	140	130	110	120
Impeller type	-	rushton turbine	pitch-blade	pitch-blade	pitch-blade	pitch-blade	rushton turbine
Reactor cond	-	Drawn off	Back to reactor	Drawn off	Drawn off	Back to reactor	Back to reactor

2.1.1. Long-run oxidation test (PP1)

Based on the results of the previous tests a “long-run” test was carried out. This test was planned for 80 hours in order to maintain the steady-state as long as possible. Boiler and reactor temperatures operated at 120ºC the entire test. Feed solution composition and oxygen-feed ratio was kept the same throughout the test. Discharge samples were taken every hour, solution samples were submitted individually and solid samples were submitted as one sample at the end of each condition.

The tested conditions were the following: Feed flowrate was set at 4mL/min, the reactor condensate was put back into the reactor and the mixer speed was tested at 750, 1000 and 1250rpm; then the mixer speed was set at 1250 rpm and the flowrate was increased to 6 and 8mL/min. Finally the feed flowrate was kept at 8mL/min with 1250 rpm but this time the reactor condensate was drawn off. Each condition lasted 12 hours, except the last one which ran for 17 hours.

Table 3 – Long-run Oxidation Test – Conditions

Condition		**Cond 1**	**Cond 2**	**Cond 3**	**Cond 4**	**Cond 5**	**Cond 6**
Start		Apr-27 8:30	Apr-27 20:30	Apr-28 8:30	Apr-28 20:50	Apr-29 9:30	Apr-29 21:30
Finish		Apr-27 20:30	Apr-28 8:30	Apr-28 20:50	Apr-29 9:30	Apr-29 21:30	Apr-30 14:30
Target feed	mL/min	4	4	4	6	8	8
Reactor rpm	rpm	1000 rpm	750 rpm	1250 rpm	1250 rpm	1250 rpm	1250 rpm
Reactor condensate		Back to reactor	Back to reactor	Back to reactor	Back to reactor	Back to reactor	Drawn off

2.2. Hydrolysis Tests

A highly concentrated ferric chloride solution which also contained ferrous chloride and base metal chlorides was subjected to hydrolysis using water as reagent, in order to convert the ferric chloride to iron (III) oxide (hematite) and recover hydrochloric acid to return to primary ore leaching.

$$2FeCl_3 + 3H_2O \rightarrow Fe_2O_3 + 6HCl \quad \textbf{[1]}$$

A photograph of the setup is shown in Figure 3. Feed solution and bath solution were synthetic solutions and they were made up from solid reagents and melted in their respective vessels at the start of each test; each composition was predicted from ASPEN modelling work provided to SGS by Starfield Resources. A summary of the tests conditions is shown in **Error! Reference source not found.**.

The standard procedure included the following steps:

- The three liquors were made up and heated up the day before the test and they were kept at temperature overnight, around 110ºC for the boiler and 140ºC for the reactor.
- The day of the test they were further heated up to target test levels and once they reached the required temperature feed was pumped out from the pre-heater to the boiler.
- The condensate from the boiler was drawn off and the solution overflowed to the reactor.
- At the point when the boiler solution began to overflow water was added to the reactor.
- The discharge from the reactor was collected and diluted with water to avoid crystallization of the salts. It was subsequently filtered and samples from the solution and the solids were submitted for assays.
- The condensate from the reactor and the boiler were collected individually and titrated for free acid (g/L of HCl). At the end of the reactor condenser a scrubber system, 2 scrubbers in series filled up with water, was set up to capture any remaining hydrochloric vapours which did not

condensate in the condenser. The scrubbers' solutions were titrated for free acid at the end of the test.

- Vacuum was set up at the end of the scrubber to help to condensate all the hydrochloric acid produced.

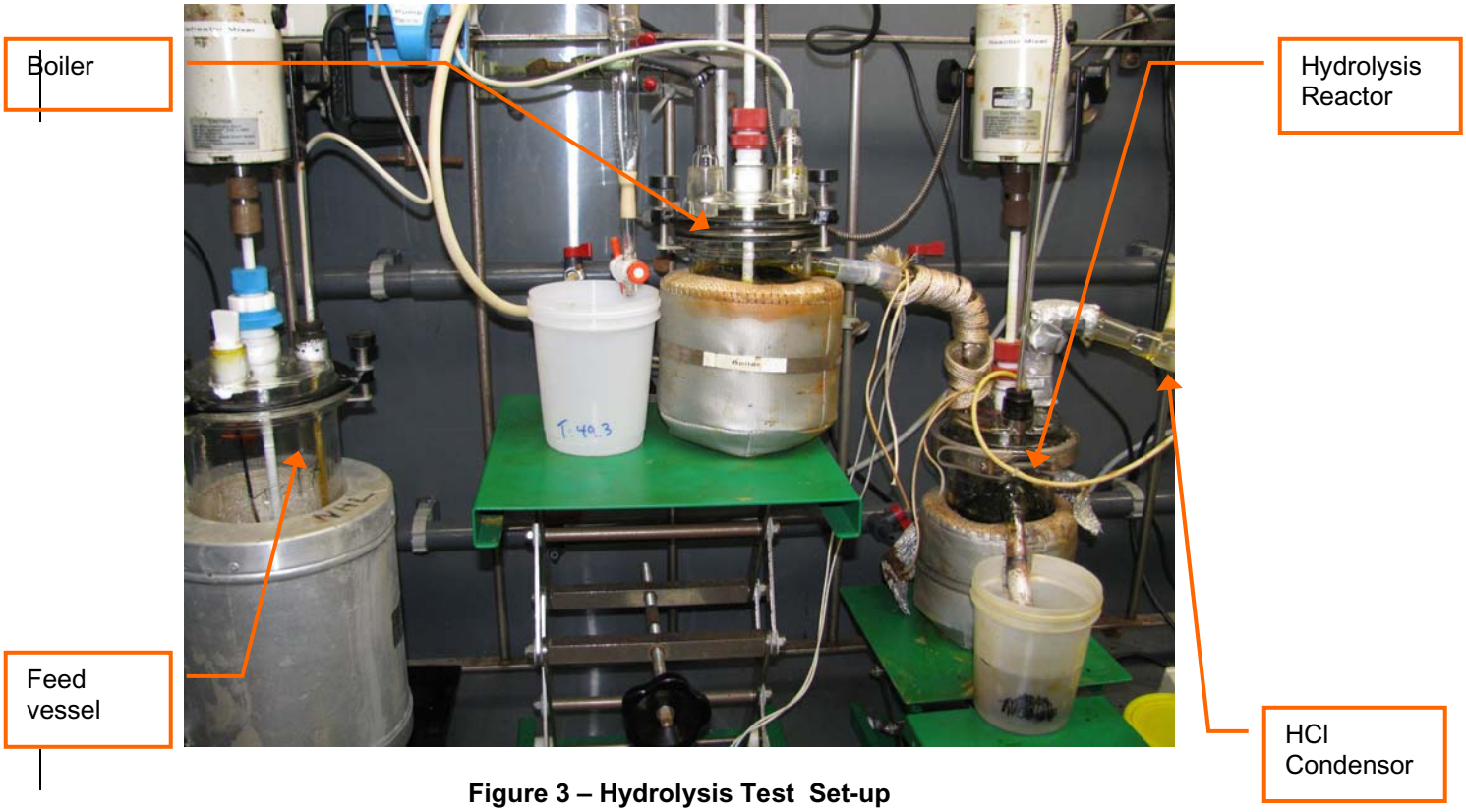


Figure 3 – Hydrolysis Test Set-up

Table 4 – Hydrolysis Tests – Tests Conditions

Condition		Test 8	Test 9	Test 10	Test 11	Test 12	Test 13	Test 14	Test 15	Test 16	Test 17	Test 18	Test 19
Date		06-May-10	14-May-10	19-May-10	01-Jun-10	08-Jun-10	15-Jun-10	17-Jun-10	24-Jun-10	29-Jun-10	05-Jul-10	13-Jul-10	15-Jul-10
Feed Composition, g/L	Fe(II)	2.7	2.7	2.7	2.7	2.7	-	-	-	2.7	2.7	2.7	38.8
	Fe(III)	213	213	213	213	213	213	213	213	213	213	213	177
	Cu	12.6	12.6	12.6	12.6	12.6	12.6	-	-	10.0	10.0	10.0	10.0
	Ni	9.1	9.1	9.1	9.1	9.1	9.1	9.1	-	-	9.1	9.1	9.1
	Co	0.8	0.8	0.8	0.8	0.8	0.8	0.8	-	-	0.8	0.8	0.8
	Mg	7.2	7.2	7.2	7.2	7.2	7.2	7.2	-	-	7.2	7.2	7.2
	H2O	699	699	699	699	699	705	732	782	755	705	705	727
Starting Hydrolysis Bath Composition, g/L	Fe(II)	12.7	12.7	12.7	12.7	12.7	-	-	-	-	4.1	4.1	59.8
	Fe(III)	414	414	414	414	414	414	414	414	335	328	328	272
	Cu	64.1	64.1	64.1	64.1	64.1	64.1	-	-	-	15.4	15.4	15.4
	Ni	46.1	46.1	46.1	46.1	46.1	46.1	46.1	-	-	14.0	14.0	14.0
	Co	4.1	4.1	4.1	4.1	4.1	4.1	4.1	-	-	1.3	1.3	1.3
	Mg	36.3	36.3	36.3	36.3	36.3	36.3	36.3	-	-	11.1	11.1	11.1
Target feed	mL/min	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Boiler temperature	°C	150	170	Variable	140	140	140	140	140	140	140	140	140
Reactor temperature	°C	180	180	180	180	180	180	180	180	180	180	190	180
Seeding	-	No	No	No	Yes	Yes	Yes	Yes	Yes	Yes	Yes	Yes	Yes
	g				98	95	93	83	88	88	92	87	85
Boiler condensate	-	Drawn off	Back to reactor	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off	Drawn off

Test 8 (first hydrolysis test) was carried out at 150ºC in the boiler and 180ºC in the reactor; reactor temperature was set based on a previous work done by McGill University, and the boiler temperature was set to a point between the feed temperature (80ºC) and the reactor temperature (180ºC). Feed flowrate was set to 2 mL/min and water was added at a rate of 0.8 mL/min; these values came from a previous modelling done by Starfields Resources. Every hour, both boiler and reactor condensates were drawn off from their respective vessels and titrated for free acid. Also, discharge from the reactor was collected and diluted in water to dissolve any base metal chlorides trapped in the precipitate. Test 9 was run at 170ºC in the boiler and 180ºC in the reactor in order to make boiler overflow solution more concentrated, this way less water was going to the reactor with the objective of making a more concentrated hydrochloric acid. Since the boiler condensate acid strength was quite high in Test 9 (130 g/L HCl), Test 10 was designed to determine a relationship between operating temperature and boiler condensate acid (HCl) strength. The test began at 120ºC in the boiler and the temperature was increased 15ºC every 2 hours. In this test boiler condensate was sampled and titrated for free acid every 30 minutes. Reactor temperature was kept at 180ºC and water addition was reduced by half (0.4 mL/min) aiming for a stronger hydrochloric acid.

Hematite produced during the oxidation tests was added as seed in Test 11 in order to induce hematite formation over undesired iron-oxygen-chloride compounds. Based on Test 10 results boiler temperature in Test 11 was set at 140ºC; reactor temperature and water flowrate were the same as Test 10.

Assuming the amount of water in the boiler solution was enough to hydrolyse all the iron in the solution Test 12 was planned to run without water addition once again aiming to a stronger hydrochloric acid. The rest of the conditions were the same as Test 11 (i.e. seeding, boiler temperature 140ºC and reactor temperature 180ºC).

High chloride content discovered in precipitates from hydrolysis tests (tests 8 – 12) three new tests were planned. This time the focus was on the effect of the other species in the ferric. Therefore changes were made in the composition of the feed solution and also in the boiler and reactor solutions. Test 13 was performed taking ferrous out of the solutions, Test 14 was run taking ferrous and also copper out of the solutions and Test 15 used a feed solution which only contained ferric chloride (no ferrous or base metals). The rest of the conditions remained the same as Test 12.

Test 16 was carried out using the same conditions reported for Neoferric in their work in order to reproduce their results.

After noticing that the final concentration in the reactor was much lower than at the beginning, and almost the same in all the tests in which all base metals and ferrous was used, Test 17 bath composition was changed trying to reproduce from the beginning the final concentration and therefore to simulate steady state. Copper was the one that suffer the bigger change going from 60 to 15 g/L. Water addition was set

at 0.4mL/min. Test 18 was performed using the same conditions as Test 17 but this time temperature in the reactor was controlled at 190C, looking for the effect of the temperature in the hydrolysis.

Test 19 was performed using a feed solution which had the same ferric/ferrous ratio than the oxidation reactor discharge. The rest of the conditions remained the same as Test 17.

2.2.1. Long-run Hydrolysis Test (PP2)

Based on the results of the previous tests a "long-run" test was designed. This test was planned for 80 hours in order to maintain the steady-state as long as possible. Boiler temperature was set to 140ºC the entire test. Feed solution composition was set as Test 19's feed (simulating oxidation reactor discharge) and was kept the same throughout the test, except during the final condition in which ferric chloride anhydrous salts were used instead of ferric chloride hydrated salts due to insufficient stock of the latter salt, but keeping the same ferric concentration in the feed solution. Discharge samples were taken every hour, solution samples were submitted individually and solid samples were submitted as one composite sample at the end of each condition. Condensate from the boiler was collected every hour, every 8 hours a composite sample was titrated for free acid. Condensate from the reactor was collected every hour and titrated for free acid, after each condition all samples were mixed up and a composite sample was submitted for iron.

The following conditions were tested:

- Feed flowrate was set at 2 mL/min;
- Water flowrate was set at 0.5mL/min, 1.0mL/min and 0.75mL/min for condition 1, 2 and 3 respectively.
- The feed was then increased to 4 mL/min and water was run at 1.0 mL/min.
- For the first 4 conditions the reactor temperature was kept at 180ºC. In condition 5 feed and water flowrate were kept the same as condition 4 while reactor temperature was increased to 185ºC.
- For the rest of the test (condition 6) feed flowrate was increased to 6mL/min and water flowrate was set at 1.5mL/min, reactor temperature remained at 185ºC.

2.2.2. Hematite Acid Washing

Although XRD analysis confirmed solids produced in hydrolysis were mostly hematite (97% of total solids) chloride content was still reasonably high (around 1%). Based on that, and the suspicion that chloride was not part of the crystal structure but crystallized chloride salts, a series of tests were planning in order to "wash out" the chloride from the solids. These tests consisted in repulping the solids from hydrolysis in a solution of hydrochloric acid varying acid concentration and temperature.

3. Test Results

3.1. Oxidation Test Results

Before the start of the oxidation test a bubble test was carried out to determine the most suitable type of impeller, rpm and location to achieve optimum gas dispersion. Tests were conducted using pitch-blade (Figure 4) and rushton-turbine type impellers (Figure 5) at 600 rpm, 1000 rpm and 1250 rpm using a U-shaped sparger tube support fitted with micro bore tubing. Tests were performed at room temperature in water to be able to observe the results with 250 mL/min of air. Tests showed that having the sparger tip 2.5 cm from the impeller bottom worked best for bubble induction into the rushton-turbine impeller and the same for the pitch-blade type. Visual tests showed little difference in gas dispersion between the two impellers although it seemed that the pitch blade impeller was slightly better (i.e. bubbles seemed more even throughout whole volume of the reactor). The impellor mixing speed (rpm) was indeed the main factor for making smaller sized bubbles; the higher the mixing speed, the better the bubbles were dispersed throughout the reactor regardless impeller design. It was concluded that either rushton-turbine or pitch-blade impellers would be suitable for actual testing; also impeller rpm would need to be tested further during actual testing conditions.

In addition, a commissioning test was carried out which consisted in heating up the solutions to the required temperatures in order to determine whether or not the heating mantles as well as the oil bath could give the calorific capacity needed for the following tests. All the required temperatures were achieved, the heating mantles achieved the right temperature faster than the oil bath did, but the oil bath had less variation (i.e. it could control the temperature in a narrower range). This test was done using the rushton-turbine impeller. This mixer consisted of a mild steel shaft with a Teflon layer to avoid corrosion; the Teflon layer was “glued” to the Teflon rushton impeller. After the test was done the mixer presented some mild signs of corrosion mostly caused because the glue did not hold on the impeller and the solution got in between the shaft and the Teflon coating. For this reason in all the tests but Test 7a Teflon-made pith-blade impeller was used.

Figure 4 – Pitch Blade Impeller Bubble Test

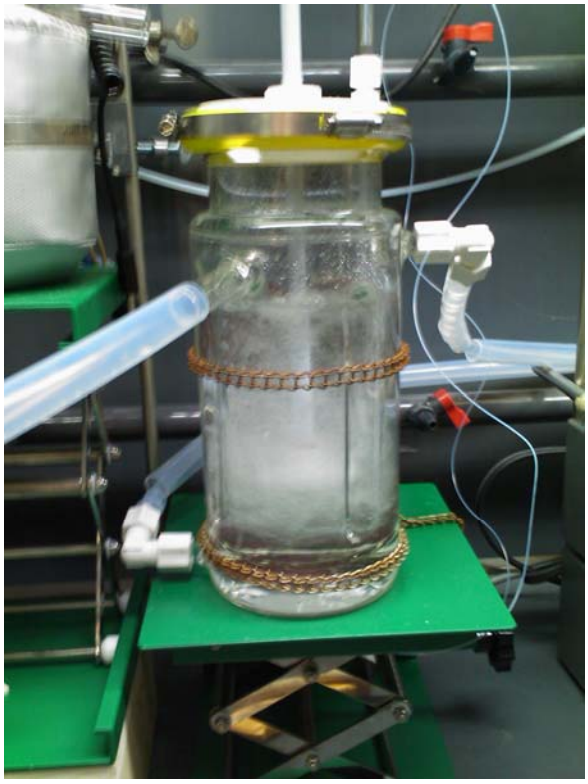


Figure 5 – Rushton Turbine Impeller Bubble Test

A summary of the main results from the oxidation tests is shown in Table 5.

Test 1 was carried out for 6 hours at 140ºC in the boiler and 150ºC in the reactor using a pitch-blade impeller, boiler and reactor condensates were drawn off from their respective vessels. 85.1% of the ferrous iron was oxidized, with an oxygen utilization of 9.3%. When the oxygen was added a foam-like thick layer was formed at the surface and the bubbles looked quite big. It seemed that the viscosity of the solution did not allow a good oxygen-solution mixing. Also, some crystallized solids were observed in the reactor walls which plugged the reactor overflow outlet making the discharge flowrate smaller than it was supposed to be. The amount of solids in the discharge was too small to submit for assays. At the end of the test pre-heater and boiler were visually inspected and some “orange” solids were found at the bottom of both vessels, as well as some crystallized solids in the boiler. The solids from the boiler were diluted in water and a solution of 10% hydrochloric acid, separately. In both cases all the solids were dissolved pointing out they were indeed crystallized salts. The solids from the reactor were diluted in a solution of 10% hydrochloric acid and most of them got dissolved, too. This brought 2 changes for the next tests: Nitrogen was added to the pre-heater to prevent any pre-oxidation occurring before the reactor, and the reactor discharge was diluted in water (previously adjusted to pH 2) in order to dissolve any crystallized salts as well as keeping the oxidation-formed solids intact. At the end of the test, the sparger was taking off the reactor and the outer layer was cracked (the sparger consisted in two concentric glass tubes). For that reason a new sparger was built, this time it was made from tantalum due to the highly corrosive properties of the solution and high operating temperatures.

Test 2 was performed to test the effect of the nitrogen blanket in the pre-heater as well as the effect of the boiler temperature in the formation of solids in these two vessels. The boiler was run at 120ºC and at the end of the test there were no solids in any of the two vessels.

Based on Test 2 results Test 3 was performed at 120 ºC in the boiler and the reactor and it ran for 8 hours. Nitrogen was added in to the pre-heater. This time the boiler condensate was drawn off but the reactor condensate was put back into the reactor. 85.9% of the ferrous ions were oxidized with an oxygen utilization of 9.3%. Iron content in the precipitate was 67.3% and it also had 2.4% chloride. This chloride content could be related to a low washing efficiency rather than as part of the solids crystal. There was no foaming effect this time and the bubbles were smaller and more dispersed suggesting a good oxygen-solution mixing. The oxygen sparger was plugged a few times during the tests; but it was typically fixed with negligible “lost” time. At the end of the test the boiler and the pre-heater were inspected and no solids were found either at the bottom or at the walls of any of the vessels. The reactor contents were filtered first using a Millipore filter paper which was digested by the liquor. After that, a Whatman 541 filter paper was used with success.

Table 5 – Oxidation Tests

Condition		Test 1	Test 3	Test 4	Test 5	Test 6	Test 7
run time	hours	6.1	8.0	5.1	9.0	10.0	10.1
avg feed flow	g/min	6.0	5.5	4.8	6.2	5.4	6.8
	mL/min	4.0	3.8	3.4	4.3	3.9	4.7
Retention time	hours	11	12	12	9	6	7
Turn-overs	-	0.6	0.7	0.4	1.0	1.7	1.4
feed	g	2190	2640	1535	3337	3266	4072
product	g pulp	307	1438	749	1642	2996	2159
	g/min pulp	0.8	3.0	2.4	3.0	5.0	3.6
	g solids	0	162	127	174	172	232
Feed	% Fe(II)	78.6	73.8	97.0	82.4	84.0	80.2
Reactor - start	% Fe(II)	6.1	6.2	28.5	11.6	8.2	12.8
Discharge	% Fe(II)	8.6	12.6	22.7	18.6	28.6	17.8
Reactor - end	% Fe(II)	11.8	7.8	20.0	24.0	19.3	8.4
% Fe(II) Oxidised	%	**85.1**	**85.9**	**60.4**	**71.0**	**60.4**	**90.5**
Oxidation rate	g/min Fe(II)	**0.65**	**0.62**	**0.46**	**0.57**	**0.42**	**0.80**
Total Fe in discharge	g/kg	196	193	341	213	155	188
Fe Accountability	%	57	107	273	96	121	74
O_2 addition	mL/min	322	354	1250	361	353	369
O_2 utilization	%	9.3	8.0	1.7	7.2	5.4	9.9
Solids	%Fe	N/A	67.3	62.7	69.6	64.7	68.3
	%Cl	N/A	2.4	3.7	1.7	3.4	2.1
	D_{80}, μm					12.57	9.52

The idea behind Test 4 was to run a test using the same conditions used by the McGill University team (Test ID: RT68) in order to prove the effect (if any) of the SGS setup. Based on that this test was ran for 5.33 hours at 110ºC in the boiler and 140ºC in the reactor and the initial composition was changed, too (See Appendix A for test details). In this test 60.4% of ferrous iron was oxidized with 1.4% of Oxygen utilization. The precipitate contained 62.7% iron and 3.7% chloride. These results reproduced the McGill University test RT68 results, thus it proved that the SGS set-up did not influence the test results.

Test 5 was run at 120ºC in the boiler and 130ºC in the reactor. The condensates from the boiler and the reactor were drawn off. Although this test was planned to run for a longer period of time it just ran for 9 hours due to the oxygen sparger got plugged and it was necessary to stop the test to unplug it. Only 71% of ferrous ions were oxidized in this test, quite below the 85% achieved in Test 3. Also, some crystallized solids were stuck to the reactor walls and the reactor overflow outlet was plugged a few times during the test. The precipitate contained 69.6% iron and 1.7% chloride.

Based on Test 3 and Test 5 results it seemed that going above 120ºC in the reactor caused some operational problems such as plugging and solids build-up in the reactor walls. Therefore Test 6 was performed at 110ºC in the boiler and reactor to determine the oxidation efficiency and solution behaviour

below 120ºC. This test was run for 10 hours and the condensate from the reactor was put back into the reactor. Only 60.4% of ferrous iron was oxidized forming a precipitate with 64.7% iron and 3.4% chloride. Every time a reactor discharge was filtered the cake cracked even before it was completely dry, this did not happen in any of the other tests.

Test 7 was performed under the same conditions of Test 3 but this time a rushton-turbine impeller was used. 90% of ferrous ions were oxidized with 9.9% of oxygen utilization. Precipitate had 68.3% iron and 2.1% chloride. The solids from this test were kept wet to use them as seed for the long-run test. A sub-sample from the solids was submitted for X-Ray Diffraction (XRD). Some results are shown in Table 6. It was confirmed that chloride was not part of the crystal structure and its content in the precipitate was related to a low washing efficiency.

After the test the impeller was severely corroded. It seemed the glue did not hold on the impeller and the solution got in between the shaft and the Teflon coating corroding the mild-steel shaft where it is joined to the impeller.

Table 6 – Semi-Quantitative X-ray Diffraction Results

Mineral	**Test 7 PPT-Reactor-Final** (wt %)
Hematite	97.3
Magnetite	1.7
Siderite	0.8
Muscovite	0.2
TOTAL	100.0

3.1.1. Long-run Oxidation Test Results (PP1)

A summary of the main results from the different conditions in the long run test is shown in Table 7. Details of the mass balance calculations and operational data (i.e. Logsheets and assays) are included in Appendix B.

The test was run for 78 hours. Sparger plugging was an issue throughout the entire test. Even when the oxygen line was changed for one with a bigger diameter to increase the oxygen flowrate, the oxygen line and the tantalum tube got plugged several times. Sometimes it was necessary to stop the feed in order to unplug the tantalum tube because it took several minutes to unplug it. Twice it was necessary to inject concentrated hydrochloric acid through the tantalum tube to dissolve the crystallized salts inside the tube. The feed hose got plugged twice and it was solved without any loss time. There were no other operational issues.

At the end of the test the three vessels were inspected and no visible damage to any of the vessels was observed. Also, there were no solids in the reactor walls or any crystallized solids in the pre-heater or in the boiler.

Because of the low flowrates the retention time was around 10 hours for Cond. 1, Cond. 2 and Con. 3 which allowed for just one turn-over for each of these conditions. For the rest of the conditions the retention time decreased due to an increment in the flowrate and 2 and 3 turn-over were achieved.

The precipitates produced throughout the test contained an average of 66.8% iron and 2.3% chloride, without any remarkable difference between each condition's solids. A particle size distribution analysis was made to each condition's solids averaging a d_{80} of 17.8 microns. PSD assays are shown in Appendix G.

Table 7 – Long-run Oxidation Test

Condition		**Cond 1**	**Cond 2**	**Cond 3**	**Cond 4**	**Cond 5**	**Cond 6**
Run time	hours	12.00	12.00	12.33	12.67	12.00	17.00
Avg feed flow	g/min	5.7	5.5	5.5	8.0	10.9	10.4
	mL/min	3.9	4.0	3.7	5.5	7.6	7.2
Retention time	hours	9.8	8.9	10.0	6.7	4.3	5.9
Turn-overs	-	1.2	1.3	1.2	1.9	2.8	2.9
Feed	g	4088	3885	3886	5912	7289	10497
Product	g pulp	2533	2281	2450	4051	5002	6535
	g/min pulp	3.5	3.2	3.3	5.3	6.9	6.4
	g solids	122	147	190	302	336	468
Feed	% Fe(II)	91.1%	84.9%	92.1%	92.1%	93.1%	90.5%
Reactor - start	% Fe(II)	7.4%	18.8%	36.7%	9.7%	26.3%	31.3%
Discharge	% Fe(II)	22.2%	31.6%	21.4%	25.1%	36.5%	22.7%
Reactor - end	% Fe(II)	18.8%	36.7%	9.7%	26.3%	31.3%	15.7%
% Fe(II) Oxidised	%	**73.8%**	**62.1%**	**101.0%**	**68.1%**	**55.1%**	**86.5%**
Oxidation rate	g/min Fe(II)	0.55	0.44	0.68	0.68	0.60	1.19
Total Fe in discharge	g/kg	180	175	174	184	173	180
Fe Accountability	%	97%	91%	100%	110%	126%	92%
O_2 addition	mL/min	349	350	379	546	726	734
O_2 utilization	%	7.2%	5.8%	8.2%	6%	4%	7%
Solids	%Fe	67.1%	66.6%	67.1%	66.3%	67.0%	66.6%
	%Cl	2.0%	2.0%	2.2%	2.6%	2.7%	2.6%
	D_{80}, μm	13.14	17.43	19.52	19.72	18.65	18.52

Conditions 1, 2 and 3 were run at the same flowrate and 1000, 750 and 1250 rpm mixing speeds respectively. The ferrous oxidized under each condition were around 73%, 62% and 100% with an oxygen utilization of 7.2%, 5.8% and 8.2% respectively. This trend confirmed what the bubbles test showed, the higher the rpm the better the oxygen dispersion and therefore this leads to improved oxidation.

In Condition 4 and 5 the mixer speed was maintained at 1250 rpm and the feed flowrate was increased 50% and 100% with respect to Cond. 3, i.e. 6 mL/min and 8 mL/min respectively. The oxidation efficiency decreased to 68.1% and 55.1% respectively. Condition 4 had the same oxidation rate (0.68 g Fe(II)/min) as Condition 3 which could indicate that the reactor was working "at full capacity" in Condition 3 and 0.68 g Fe(II)/min is the maximum oxidation rate achievable at those conditions.

Condition 6 was performed with the same parameters as Cond. 5 except the reactor condensate was drawn off. This brought a remarkable improvement to the Fe(II) oxidation which increased from 55.1 % in Condition 5 to 86.5 % in Condition 6.

3.2. Hydrolysis Tests

A summary of hydrolysis tests results is shown in Table 8.

Test 8 was performed for 5 hours running the reactor at 180ºC and the boiler temperature was set at 150ºC, but was increased to 180ºC after 3 hours. Hydrochloric acid was produced with an average strength of 15.2% and a maximum of 20.2%. Because of the overflow line from the boiler to the reactor did not have a down-comer both vessels, boiler and reactor vapour phases acted as one; that means condensates flowrates were varying throughout the test going both ways, either to the reactor or to the boiler. That could be the reason why the hydrochloric acid strength was lower than expected. Due to the low feed flowrate retention times were bigger and steady-state operation was hardly achieved, which is why the recoveries were calculated based on iron contents of the solids instead of liquor analysis. This test hydrolysed 39% of the ferric ions based on the iron content in the solids. Iron in the solids was 52.5%, really low when compared with the solids produced in the oxidation tests which contained around 67% iron. Chloride content was quite high, around 16.7%. This chloride level is unacceptable because i) it takes value off of the hematite, and ii) it is translated into less hydrochloric acid production. The high chloride content suggested the formation of some kind of iron-oxygen-chloride compounds instead of hematite.

Test 9 was carried out for 9 hours. This time a down-comer was installed from the boiler to the reactor to isolate the vapour face in each vessel. Boiler temperature was set at 180ºC but the boiler overflow started to clog the line. In order to avoid that boiler temperature was decreased to 170ºC and a heating tape was rolled over the overflow line. Reactor temperature was 180ºC throughout the test. Due to the high temperature in the boiler the free acid in the boiler condensate was around 116g/L suggesting that some hydrolysis was occurring in the boiler, this was confirmed at the end of the test due to the amount of solids founded in the boiler. For the last two hours water addition was stopped and instead boiler condensate was added to the reactor. However, there was no effect in the hydrochloric acid strength produced in the reactor due to this change, which remained around 19% the entire test. Based on the amount of solids

and its iron content 27% of the ferric ions were hydrolysed, producing solids with 49.7% iron and 17.7% chloride.

Test 10 was run to investigate the relation between boiler temperature and acid strength in the boiler condensate. The test started with the boiler at 120ºC, and temperature was gradually increased up to 180ºC. Samples from the boiler condensate were titrated for free acid. Figure 6 shows that free acid increased exponentially when the temperature was increased. It also shows that to avoid hydrolysis in the boiler temperature should be kept below 140ºC. Solids assays showed 49% iron, accounting for 31% of total ferric, and 12.8% chloride. Also, hydrochloric acid was produced in the reactor with an average strength around 24%. Because the high chloride content in the solids XRD assays were carried out on each tests solids starting with Test 10 to find out if the chloride was part of the solids structure or crystallized salts. A summary of the XRD assays is shown as part of Table 8. Only 12% of the solids were actually hematite the rest were iron-oxygen-chloride compounds, mostly Akaganeite and Iron oxide chloride.

Two changes were introduced in Test 11 in order to improve the hydrolysis performance. Hematite produced in the oxidation tests was used as seed to promote hematite formation instead of Fe-O-Cl compounds'. Also water flowrate was decreased by half aiming for a stronger acid. Based on Test 10 boiler temperature was set at 140ºC, which kept the free acid in the boiler below 3.5 g/L (around 0.3%). Only 19% of ferric ions were hydrolysed, producing solids with 50% iron and 10% chloride. Although the chloride content decreased respect to the previous tests it could be related to the hematite added as seed (less than 2% Cl) rather than to a better hydrolysis performance. XRD assay showed 70% of the solids were Fe-O-Cl compounds. Acid strength averaged 26% with values as high as 28%.

Test 12 was run without water addition, once again aiming to produce a stronger acid. This change was made based on the hypothesis that the water coming from the boiler was enough to carry out the hydrolysis. As usual the solutions were made up the day before to the test and let them at temperature overnight. In the morning the boiler solution level was below the temperature probe and the solution was overheated. For this reason some HCl (40.4 g/L) was released in the boiler condensate at the beginning of the test, but after the first hour the free acid in the boiler condensate was around 2 g/L (<0.3%) and remained the same for the rest of the test. Also, some red-like solids were found in the boiler, not enough to submit for assays though. Reactor condensate had an acid strength averaged 27%, not a big change from Test 11, though it reached 30% as maximum value. Solids contained 62% iron, accounting for 74% of total ferric ions, and 7.6% chloride showing an improvement regards previous tests but still above the target. XRD analysis showed 72% of the solids were hematite, but still 28% were Fe-O-Cl compounds.

Table 8 – Hydrolysis Tests

Condition		Test 8	Test 9	Test 10	Test 11	Test 12	Test 13	Test 14	Test 15	Test 16	Test 17	Test 18	Test 19
Date		06-May-10	14-May-10	19-May-10	01-Jun-10	08-Jun-10	15-Jun-10	17-Jun-10	24-Jun-10	29-Jun-10	05-Jul-10	13-Jul-10	15-Jul-10
run time	hours	5.0	9.0	8.0	8.0	8.0	8.0	8.0	8.1	8.0	8.0	8.0	6.0
average feed flowrate	g/min	3.9	3.6	3.6	3.4	3.2	3.0	2.9	3.2	3.1	3.7	3.6	4.2
	mL/min	2.6	2.4	2.4	2.3	2.2	2.0	2.0	2.2	2.1	2.4	2.4	2.8
average water addition	mL/min	0.80	0.83	0.38	0.38	Nill	Nill	Nill	Nill	0.38	0.39	0.39	0.39
total water addition	mole	13.3	24.7	10.0	10.2	0.0	0.0	0.0	0.0	10.2	10.4	10.3	7.8
Retention time	hours	16	12	14	7	10	12	12	15	37	12	34	17
Turn-overs	-	0.3	0.8	0.6	1.1	0.8	0.6	0.7	0.5	0.2	0.7	0.2	0.4
feed	g	1110	1992	1691	1653	1536	1449	1395	1526	1494	1773	1734	1494
Feed	% Fe(III)	98.5	98.7	98.4	98.3	96.4	96.4	96.4	100.0	98.6	98.6	98.8	83.3
	g Fe(III)	174.3	294.6	259.7	247.3	221.2	216.4	202.9	220.7	228.1	276.2	280.4	186.9
	mole Fe(III)	3.1	5.3	4.6	4.4	4.0	3.9	3.6	4.0	4.1	4.9	5.0	3.3
	mole H2O	30.4	50.7	44.0	43.2	40.7	37.8	39.1	46.5	42.5	46.1	45.6	40.2
product	g H2O boiler cond	198	517	788	358	396	345	334	445	422	391	373	397
	mole H2O boiler cond	11	29	44	20	22	19	19	25	23	22	21	22
	g reactor pulp	414	968	744	1523	991	824	847	698	306	865	351	453
	g/min pulp	1.4	1.8	1.6	3.2	2.1							
	g solids (incl seed)	128	161	163	225	365	138	136	206	282	238	416	169
	g solids produced	128	161	163	127	270	46	53	118	194	145	329	84
	mole Fe solids	1.2	1.4	1.4	1.1	3.0	0.5	0.6	1.4	2.4	1.7	4.0	1.0
	g/kg H2O	294	265	364	398	314	309	338	334	334	334	334	334
	g HCl	124	83	36	46	45	36	47	126	229	185	309	103
	mole HCl	3.4	2.3	1.0	1.3	1.2	1.0	1.3	3.5	6.3	5.1	8.5	2.8
	mole H2O w HCl	38.5	19.9	6.3	7.3	6.8	5.9	6.9	13.9	26.8	22.7	27.7	16.6
Reactor - start	% Fe(III)	96.4	96.7	96.6	98.8	116.2	99.3	100.0	100.0	100.0	98.8	98.8	82.0
Discharge	% Fe(III)	96.1	97.0	97.0	98.6	97.4	99.0	99.7	100.0	97.8	97.7	97.4	80.8
Reactor - end	% Fe(III)	96.7	96.8	97.1	97.7	99.8	99.2	99.4	100.0	98.2	98.1	97.0	78.1
% Fe(III) hydrolyzed	%	**79.4**	**68.6**	**72.7**	**8.5**	**31.6**	**30.8**	**33.2**	**36.1**	**62.7**	**41.7**	**86.5**	**58.4**
Fe Accountability	%	59	59	58	109	145	82	88	99	98	94	95	80
HCl average strength	%	15.2	18.9	24.3	26.2	26.9	25.6	27.3	33.5	32.2	31.1	38.3	25.6
HCl maximum strength	%	20.9	19.9	25.8	28.0	30.6	27.0	31.5	38.1	37.6	34.3	39.3	28.4
Solids	%Fe	52.5	49.7	49.0	49.8	62.0	56.1	67.0	65.8	68.2	65.8	67.6	68.4
	%Cu	0.023	0.043	0.072	0.03	0.03	0.05	0.02	0.00	0.003	0.02	0.02	0.02
	%Cl	16.5	17.7	12.8	9.9	7.6	10.9	1.3	1.2	1.2	1.7	1.3	1.0
	% hematite			12.3	21.6	72.2	35.5	98.0	96.4	N/A	N/A	N/A	N/A
	% akaganeite			44.0	48.9	10.8	53.0						
	% iron oxide chloride			28.0	21.4	15.3	7.8		1.1				
	Rokuhnite			13.0					1.9				
	D_{80}, µm	44.6	35.8	40.2	22.6	48.7	26.7	15.6	17.6	24.1	17.7	18.1	15.6

From Test 13 to test 15 changes were made in the feed solution to investigate how the presence of ferrous and base metal chlorides affects the hydrolysis process. Hematite was used as seed in these tests and there was no water addition. Test 13 was performed taking ferrous chloride out of the feed, boiler and reactor solutions. Only 7% of ferric iron was hydrolysed in this test and the hydrochloric acid produced had an average strength of 25.6%. Solids contained 56% iron and 10.9% chloride, and only 35.5% of the solids were determined by XRD to be hematite.

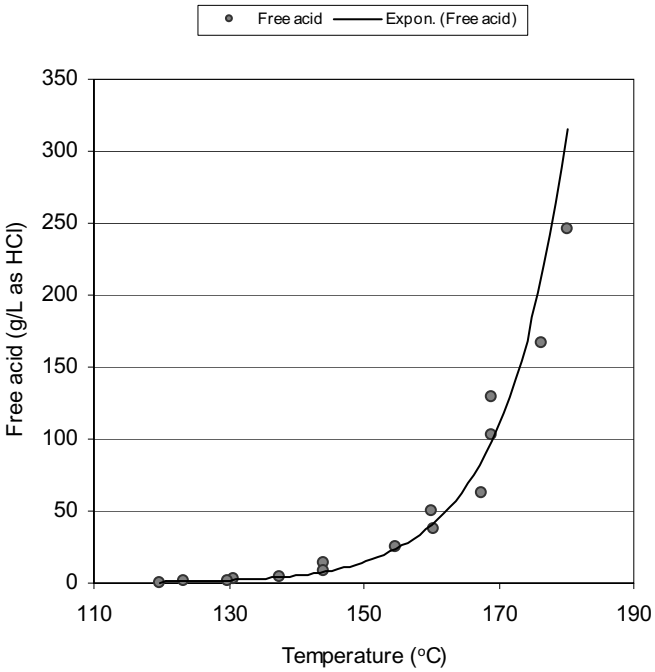


Figure 6 - Boiler Temperature vs. Condensate Acid Strength

Test 14 was run taking ferrous iron and copper chloride out of the feed and bath solutions. It was noticed that in all the previous tests acid production in the reactor (i.e. reactor condensate) started after 2 hours, some tests after 3 hours. It seemed like the bath solution was oversaturated and the water added in the first 2 hours was used to saturate the solution, and only after that the water/acid started to boil. For this reason more water was added in the reactor solution make-up and reactor condensate was collected from the beginning, averaging 27.3% throughout the test. Solids contained 67.3% iron and just 1.3% chloride, and XRD analysis showed that 98% of the solids was hematite, and only 0.5% consisted of chloride compounds. This is suggesting that chloride could be a consequence of low washing efficiency and not part of the precipitate.

Test 15 was performed using a solution of ferric chloride, i.e. no ferrous iron or base metal chlorides were added. Seed was used and no water was added. This produced a reactor condensate with an average acid strength of 33.5%. Also, for most of the test the condensate was fuming which could indicate an even higher acid strength. 35% of ferric ions were precipitated, producing solids with 65.8% iron and 1.2% chloride. Hematite represented the bulk of the solids based on XRD analysis. From this point onwards XRD was only used if the solids contained more than 2% chloride.

Test 13, Test 14 and Test 15 results showed that copper presence appeared to have an effect on hydrolysis performance. That is why Test 16 was run to replicate an early work done by NeoFerric, in which copper was not added in the initial reactor solution, but it was part of the feed solution in a lower grade that the previous tests, though. Water was added at a flowrate of 0.4 mL/min and hematite was

used as seed. This test produced solids with 68.2% iron and 1.2% chloride, precipitating 59% of ferric ions. Acid strength in the reactor condensate averaged 32.2% and some fumes were observed in some samples which had more the 35% hydrochloric acid. This test showed that high copper concentration (and no copper presence) could be the cause of low hydrolysis performance (refer to **Error! Reference source not found.** for the different copper concentration used in each test). Also this test helped to realize the different roles water plays in this process:

1. Reagent in the reaction with Fe(III) to form hematite and acid;
2. Diluent with acid (note that hydrochloric acid at 35% contains 65% water);
3. Diluent in the discharge pulp (300 g/kg pulp).

Based on a preliminary water balance it appeared that in Test 12, 13, 14 and 15 water addition was not sufficient contrary to what it was assumed at that time. The effect of water on the extension of hydrolysis reaction and on the acid strength were studied in the long-run hydrolysis test.

Test 17 used all the components (ferrous and base metals) again and bath composition was adjusted to an average of previous results (realizing that the original bath target composition was too concentrated). Seed was once again used and water addition was set at 0.4 mL/min, as in Test 16. This test precipitated 34% of ferric ions and produced solids with 68.0% iron and 1.7% chloride, suggesting hematite formation, Acid strength averaged 31.1%; fuming effect was less visible while keeping acid strength still high.

Test 18 was run to investigate the effect of higher temperature in the hydrolysis. Feed and reactor composition, seeding and water addition were the same as Test 17. Reactor temperature was set at 190ºC, but it turned reactor solution too viscous and produced some foaming which made it hard for the solution to overflow, for the first 3 hours discharge was small to the point that only one sample was taken in those first 3 hours. Because of this, temperature was decreased to 185ºC, increasing the discharge without any changes in acid production. There were few solids in the discharge, 95% of the solids remained inside the reactor, which could become a big problem in a continuous operation (i.e. longer running time). This test produced the strongest acid, averaging 38% HCl (w/w), but fuming was a real problem for collecting the acid and during titration, since acid fuming represents an acid loss. In this test, 80% of the ferric iron was precipitated, which was the optimum level observed thus far. This test led to a solids composition of 68% iron and 1.3% chloride.

For Test 19 feed composition was changed once again, this time to more accurately simulate the discharge from the oxidation stage. Base metals composition remained the same as Test 17, but the ferrous percentage was increased from <5% to 17% based on the oxidation tests performance. Seeding and water addition were the same as Test 17 and reactor temperature was 180ºC. Acid strength decreased to 25%, but there was no fuming and acid strength remained sufficiently high for primary ore

leaching, therefore this was still considered an acceptable outcome. Solids produced in this test contained 68% iron and 0.9% chloride and the ferric precipitation efficiency was calculated to be 31%.

3.2.1. Long-run Hydrolysis Test (PP2)

Based on the results of the bench scale scoping tests an extended hydrolysis campaign was carried out, consisting of a runtime of 76 hours as well as 4 hours of pre-conditioning. The boiler temperature was set to 140ºC the entire test. Feed solution composition was set as Test 19's feed (high ferrous – 17% Fe(II)) and this was kept constant throughout the test. During the last condition hydrated ferric chloride ran out and instead anhydrous ferric chloride (keeping ferric concentration constant) was used in the final feed batch. Feed flowrate, water addition and reactor temperature were varied throughout the campaign to see the effect on the hydrolysis. A summary of the long-run hydrolysis test is shown in Table 9. Details of the mass balance calculations and operational data (i.e. Logsheets and assays) are included in Appendix D.

Due to the viscosity of the slurry several times in the test the reactor overflow got plugged and some solids were building up in the reactor walls close to the baffles, especially when the reactor temperature was increased to 185ºC. It was necessary to "clean" the solids from the overflow discharge almost every hour for the last 2 conditions. At the beginning of the test, the boiler downcomer was plugged but was unplugged in 5 minutes and a heating tape was installed; it did not plug again for the rest of the test. In the last condition temperature control in the reactor stopped working for 10 minutes without any visible cause. It was reset and it did not fail again. In that lapse reactor temperature dropped to 172ºC but it was increased again once the temperature control was reset.

This test processed around 22 kg of feed, 4 litres of water were added to the reactor, producing 8.44 kg of hydrochloric acid, averaging 22%, and 1.28 kg of solids with 68.9% iron and 0.9% chloride.

Boiler parameters did not change thru the test and the operation was very stable. Condensate from the boiler was collected every hour, every 8 hours samples were mixed up and a composite sample was titrated for free acid. Figure 7 shows temperature trends, as well as feed and boiler condensate flowrates. Condensate flowrate was very steady, just increasing when feed flowrate was increased. Figure 8 shows condensate hourly amounts and free acid. Free acid values were between 1.5 g/L and 6 g/L (below 0.6%). No solids were found at the end of the test which confirmed hydrolysis did not take place in the boiler.

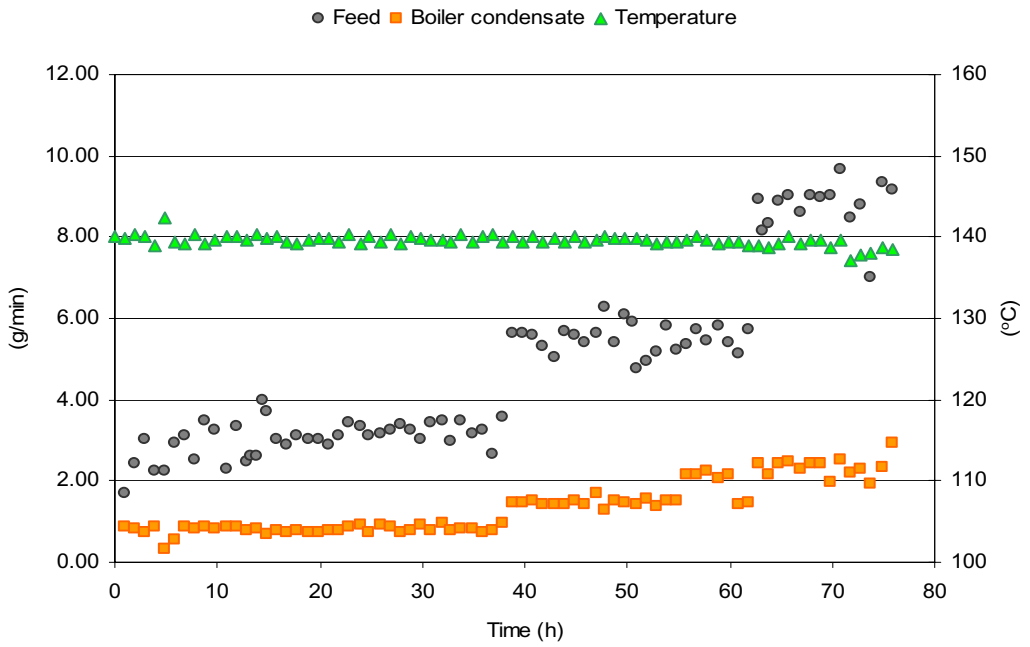


Figure 7 - Boiler Temperature and Flowrates

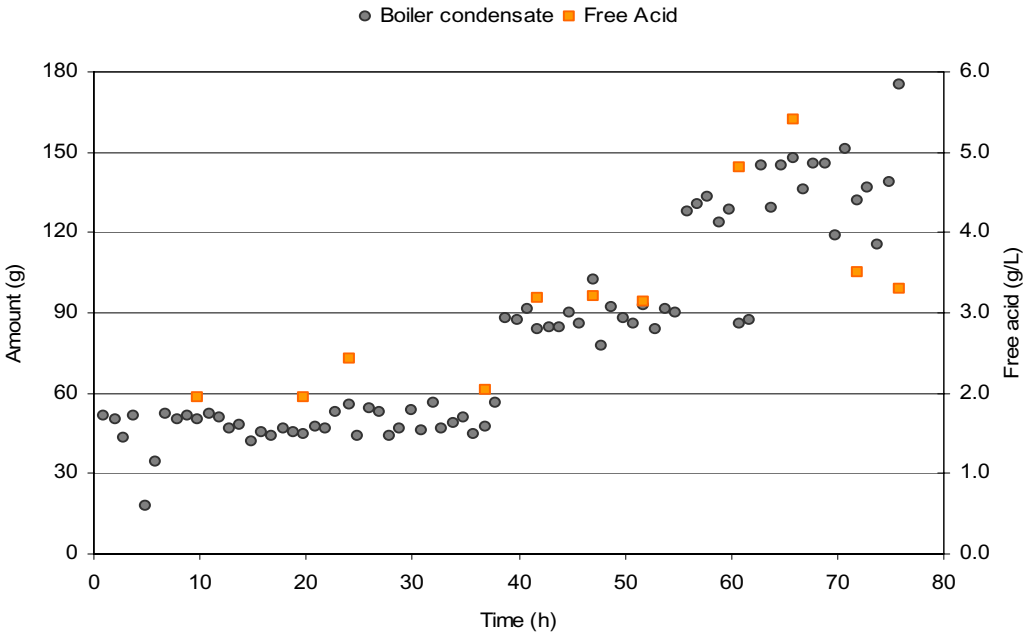


Figure 8 - Boiler Condensate and Free Acid

Different operating conditions were tested in the reactor to investigate the effect of water addition, temperature and feed flowrate in the hydrolysis performance. Figure 9 shows the main reactor parameters' trends. Since it was not possible to determine the exact amount of solids inside the reactor

for every condition, hydrolysis performance was estimated based on acid production (each 2 moles of ferric iron hydrolysed leads to the production of 1 mole of hematite and 6 moles of HCl).

Table 9 – Long-run Hydrolysis Test – Main Parameters

Condition	Run time	Avg Feed Flows		Water	T in reactor	Reactor condensate		pure HCl
	hr	g/min	mL/min	mL/min	°C	g/min	avg %HCl	mg/min
1	13.3	2.6	1.8	0.48	180	1.35	22.9%	256
2	12.6	2.9	2.0	0.89	180	1.57	21.3%	335
3	11.9	3.2	2.2	0.74	180	1.50	20.5%	307
4	12.7	5.6	3.7	0.82	180	1.69	20.6%	350
5	11.3	5.4	3.6	1.03	185	2.26	24.8%	564
6	14.0	8.5	5.7	1.40	183	2.98	22.4%	679

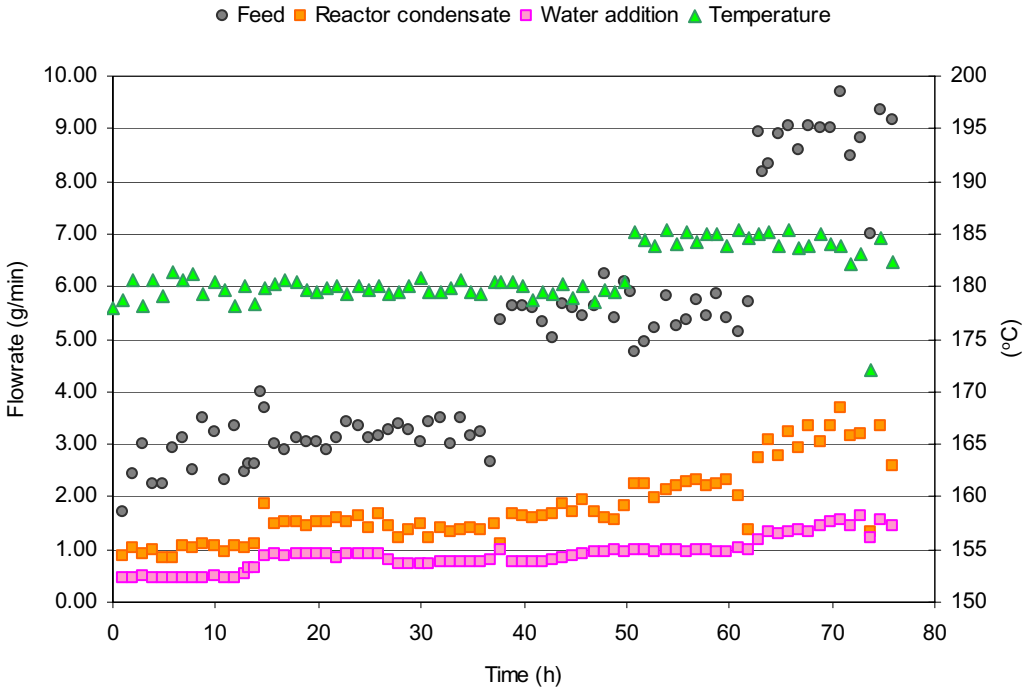


Figure 9 - Reactor Operating Parameters

At the beginning of the test feed flowrate was 2mL/min and reactor temperature was controlled at 180ºC. Water addition was the only parameter which changed for the first three conditions, varying from 0.45 mL/min, 0.90 mL/min and 0.75 mL/min for Condition 1, 2 and 3, respectively. During Condition 1, acid strength was decreasing without any visible reason. As shown in Figure 10 when water addition was increased (keeping the rest of the conditions the same) reactor condensate production increased, too. Red lines in the graphs are meant to visually separate one condition from another one. Although acid strength decreased with the increment of water addition the amount of HCl (100%) increased which confirmed the role of water as reagent, i.e. more reagents more products. Nonetheless, it will be necessary to define whether acid strength or acid production is the most important factor to the entire process, and set the water addition accordingly.

In Condition 4, feed flowrate was increased to 4 mL/min and the feed/water ratio was kept the same as Condition 1, therefore water addition was set at 0.90 mL/min, though the actual average flowrate was 0.82mL/min. As shown in Figure 11 when feed flowrate was increased acid production increased, too. It is of interest to note that although acid production did increase it did not increase in the same proportion as the feed flowrate and water increases, which could indicate that residence time was a factor in the hydrolysis performance.

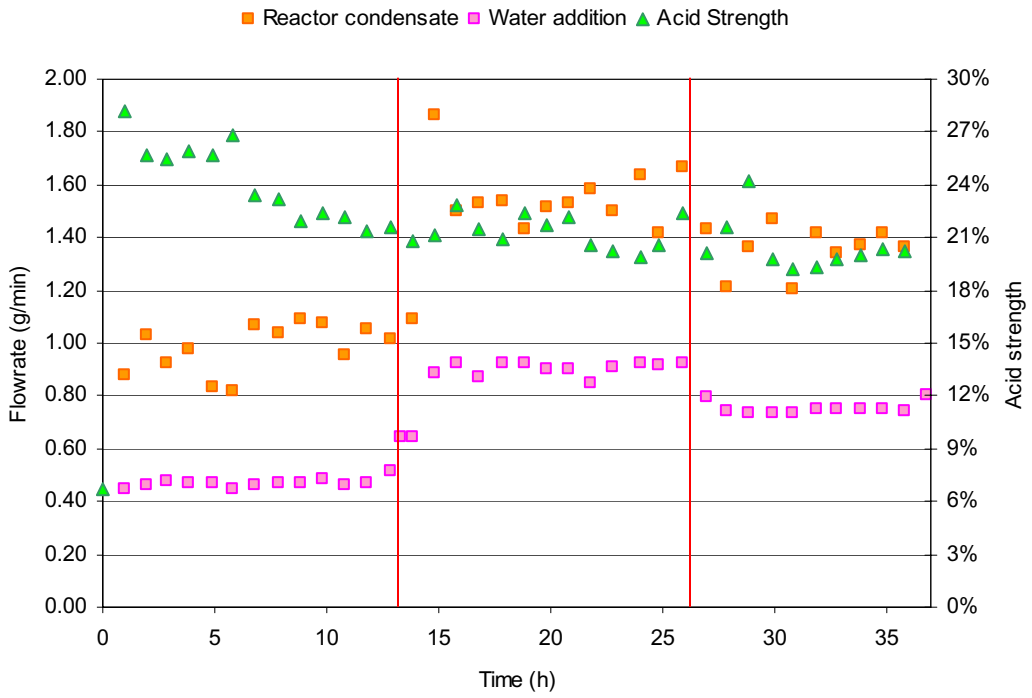


Figure 10 - Cond. 1, Cond. 2 and Cond. 3 Comparison

In Condition 5, feed flowrate and water addition were kept the same as Cond. 4 and reactor temperature was increased to 185ºC. Target water addition for both Cond. 4 and Cond. 5 was 0.90mL/min but actual water flowrate were 0.82 mL/min for Cond. 4 and 1.03 mL/min for Cond. 5. Figure 12 shows that increasing the reactor temperature from 180ºC to 185ºC resulted in increased reactor condensate production rate (from 1.69 g/min to 2.26 g/min) while keeping the feed flowrate constant. Figure 13 shows also an increase in acid strength (20.6% to 24.8%) in addition to an increase in HCl (100%) production (0.35 g/min to 0.56 g/min). Nonetheless, slurry in the reactor got thicker and “line plugging” problems increased exponentially. Some foaming-like solids built up near the baffles inside the reactor decreasing (at least visually) mixing efficiency.

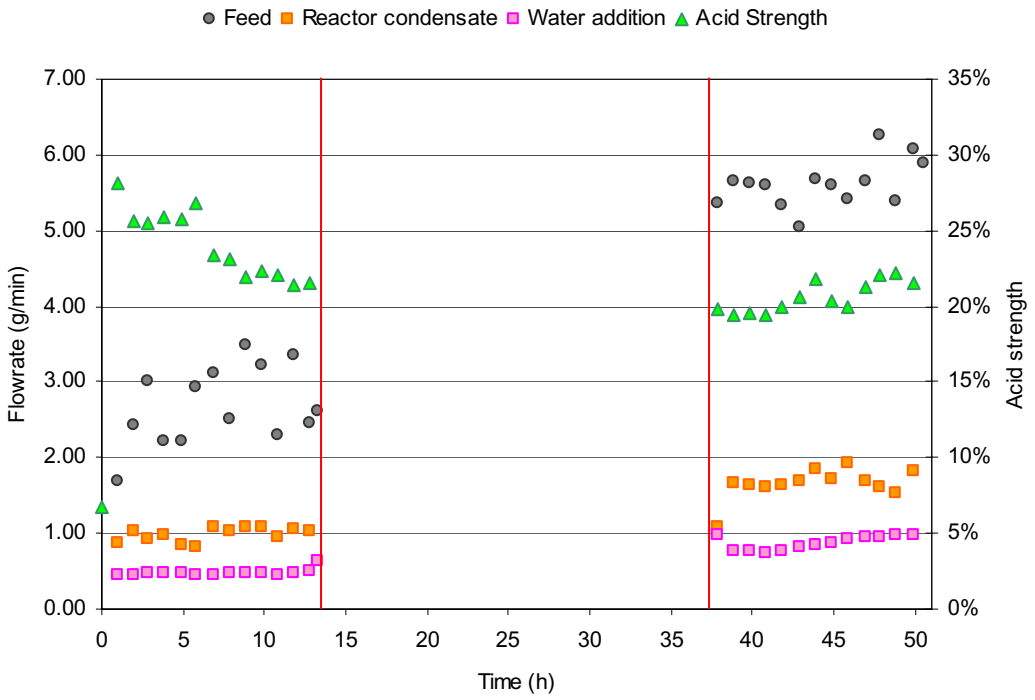


Figure 11 - Cond. 1 and Cond. 4 Comparison

During Condition 6, the feed flowrate was increased up to 6 mL/min and the feed/water ratio was kept the same as Cond. 5, therefore water addition was set at 1.40 mL/min while keeping reactor temperature at 185ºC. Figure 14 shows the main parameters from Cond. 5 and Cond. 6. As shown in Figure 15 acid production rate increased when the feed flowrate was increased while keeping acid strength fairly constant. During Cond. 6 temperature control in the reactor stopped working for 10 minutes without any visible cause. It was reset and it did not fail again. In that lapse reactor temperature dropped to 172ºC. This could explain the decline in acid strength and production rate in the final hours of the test.

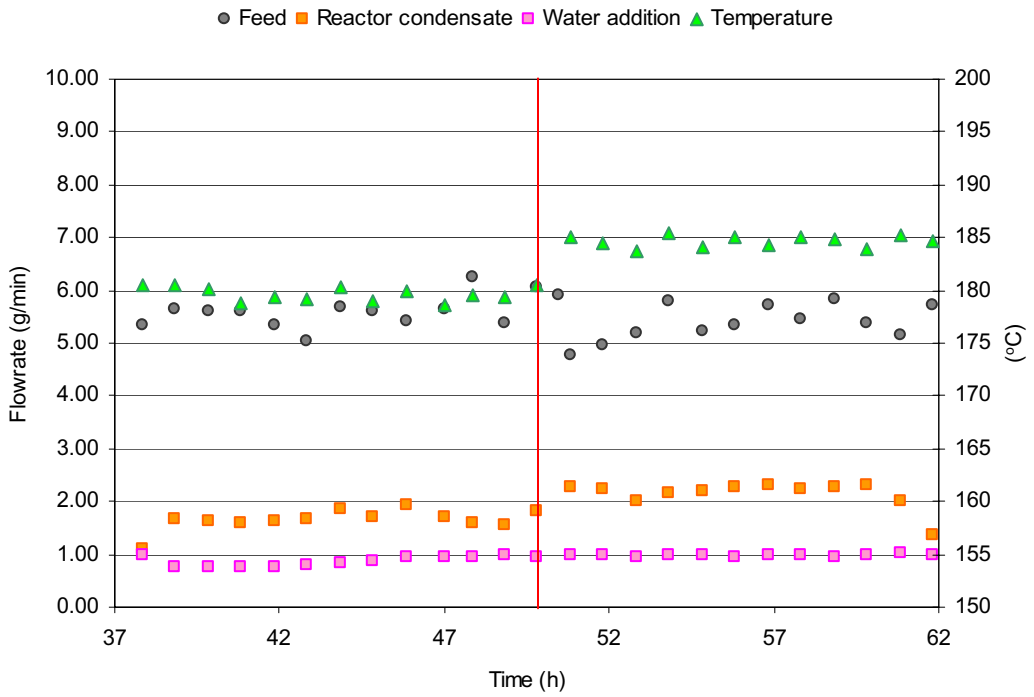


Figure 12 - Cond. 4 and Cond. 5 Comparison (I)

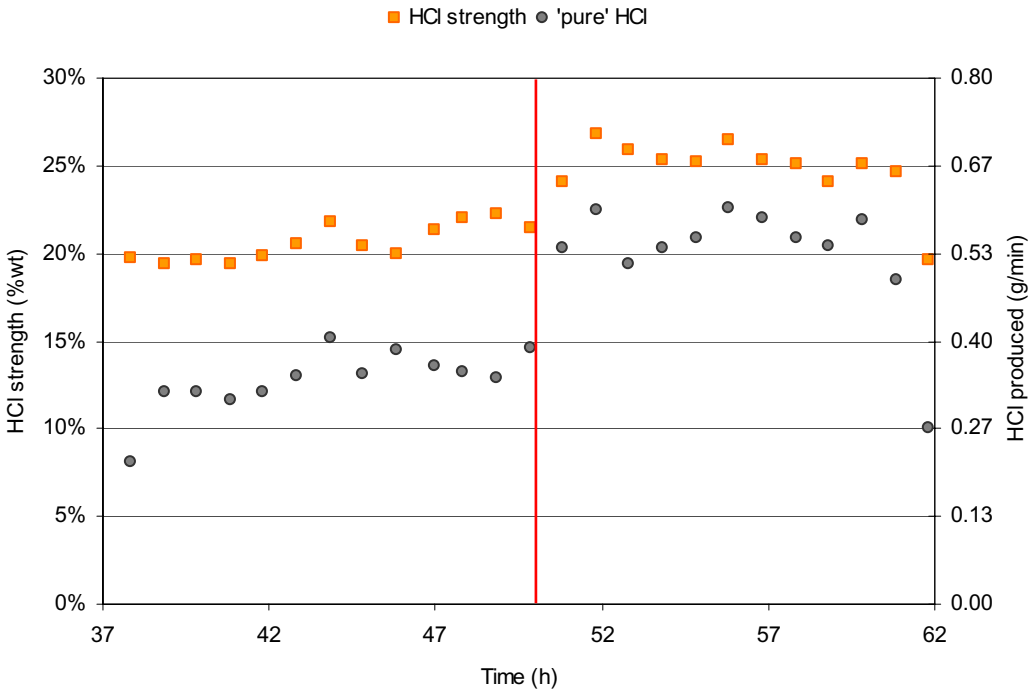


Figure 13 - Cond. 4 and Cond. 5 Comparison (II)

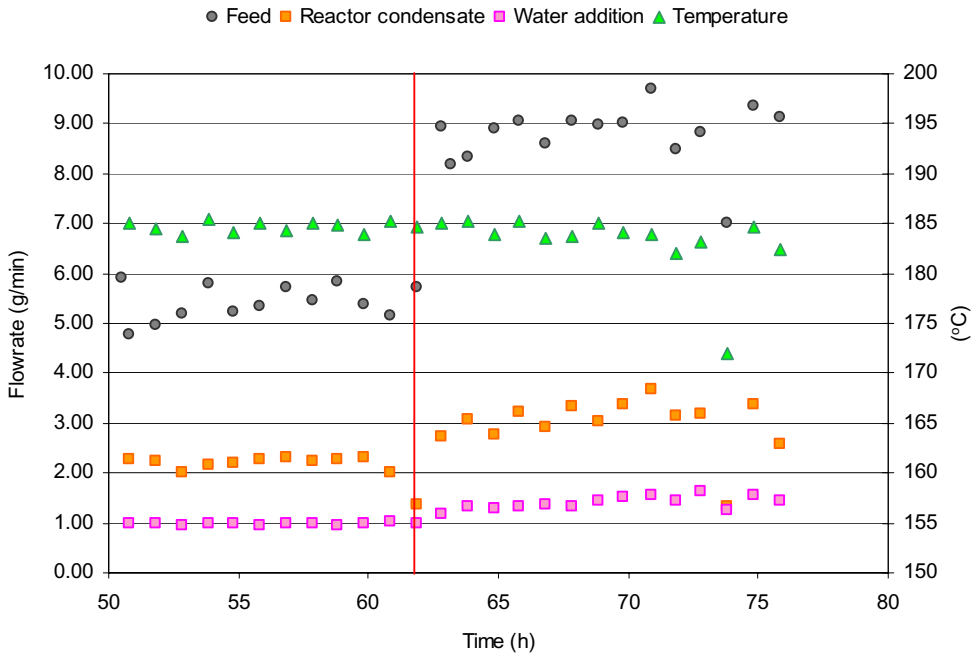


Figure 14 - Cond. 5 and Cond. 6 Comparison (I)

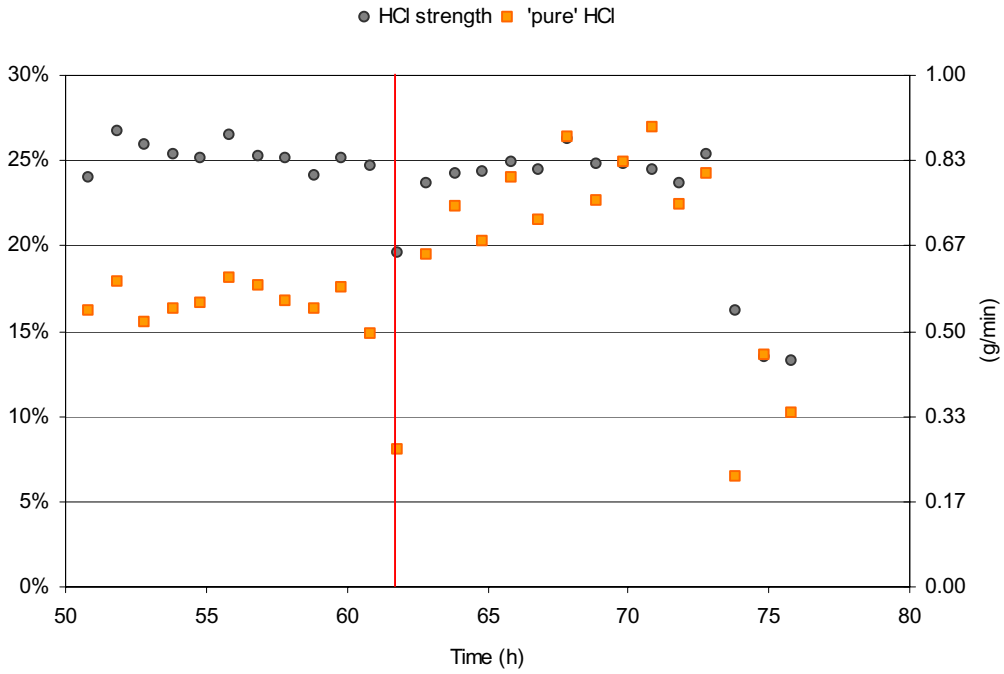


Figure 15 - Cond. 5 and Cond. 6 Comparison (II)

3.2.2. Hematite Acid Washing Results

A summary of hematite washing tests is shown in Table 10. Test details are included in Appendix F. Solids were taken from the hydrolysis campaign, specifically from Cond. 4, Cond. 5 and Final solids because they had the highest chloride content. CW1, CW2 and CW3 were run at the same temperature (50ºC) and using different hydrochloric acid solutions while CW4 use the same solution as CW2 but the temperature was increased to 90ºC. Neither temperature nor acid made significant differences regarding chloride content in the solids after the tests, as is shown in the table below. It seems that chloride content reached a "plateau" at around 0.6% in which is not possible to continuing removing chloride from the solids, at least by this washing method.

Table 10 – Hematite Washing Tests – Main Parameters and Results

Test ID	Initial Solids			HCl g/L	Temp °C	Final Solution Fe (mg/L)	Final Solids Cl (%)	Raw extraction
	Source	H_2O (%)	Cl (%)					
CW1	Final Reactor Solids	11.1	0.90	12	55.0	884	0.67	25%
CW2	Cond 4 Solids	8.9	1.14	56	52.7	16100	0.62	46%
CW3	Condition 5 Solids	12.5	0.90	106	56.2	40400	0.74	18%
CW4	Cond 4 Solids	8.9	1.14	56	94.1	23000	0.67	42%
Average		10.4	1.0				0.67	33%

Conclusions and Recommendations

1. Oxidation

- Continuous oxidation of a ferrous chloride solution was successful at 120ºC, producing a precipitate consisting of 97% hematite.

- Increasing the reactor temperature above 120ºC caused solids to built-up on the reactor walls and decreasing the reactor temperature below 120ºC decreased the oxidation efficiency.

- Both pitch-blade and rushton-turbine impellers were tested successfully; under otherwise similar conditions the rushton-turbine led to slightly higher oxidation efficiency, but it could not be tested in the long-run test due to a corrosion problem. A custom Teflon-only rushton-turbine impeller was designed and constructed, which is recommended to be used in further tests.

- Although adequate oxygen dispersion was accomplished, the sparging operation was problematic due to continuous plugging of the sparger. For future lab tests or scale-up a new sparger should be designed. Oxygen utilization should be optimized as well by testing lower oxygen addition rates.

2. Hydrolysis

- Continuous hydrolysis of a ferric chloride solution was successful, producing a precipitate with 69% iron and 0.9% chloride, and hydrochloric acid at 22%.

- Water addition is a key factor in hydrolysis performance affecting solids composition, hydrolysis extension, acid production and acid strength. Increasing the amount of water increased the extension of the hydrolysis and the amount of acid produced while sacrificing in acid strength. The exact water addition should be based on an evaluation of acid strength versus the extent of hydrolysis (or amount of hematite and acid produced).

- Increasing hydrolysis temperature led to an increased extent of hydrolysis, amount of acid produced and acid strength. However, it also increased slurry viscosity and “line plugging” problems increased significantly. Increasing temperature above 185ºC would make hydrolysis “challenging” from an operational point of view.

- Acid washing of the hematite solids was able to reduce the chloride content from 1% to 0.6%. It was not possible to decrease the chloride content below 0.6%. A more specific study of the hematite structure is recommended.

Appendix A - Oxidation Scoping Tests

Test details

Project: 12323-001 **Date:** 24-Mar-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 1** *Oxidation Tests*
Purpose: **To oxidize ferrous iron to ferric iron at maximum oxygen utilization**

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors
pressurized oxygen
10% HCl
hot solutions and hot surfaces

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required
standard PPE (safety glasses, lab coat, gloves)
Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $12FeCl_2 + 3O_2 = 2Fe_2O_3 + 8FeCl_3$

Conditions:

Preheater:
Temperature: 80 ℃
Reflux Condensation: yes (back into preheater)
prefill with: "feed to boil liquor"
volume: 3000 mL

Boiler:
Temperature: tbd estimated 120 to 140C
feed flow: 4.03 mL/min or: **6.2** g/min
sg: 1.55 g/mL
prefill with: "boiler bath liquor"
dynamic volume: 800 mL
Reflux Condensation: no (use "H" type fitting to draw off condensate at target flow)
target condensate flow: 1.18 mL/min *cell H47 PE model*

O2 utilizati Reactor:
Temperature: 150 ℃
feed flow: as is
prefill with: "oxidation bath liquor"
dynamic volume: 1200 mL
Reflux Condensation: no (use "H" type fitting to draw off condensate at target flow)
target condensate flow: 1.06 mL/min *cell L47 PE model*
O2 flow: 0.37 L/min *cell E11 PE model*
estimated dischareg liquid flow: 2.2 mL/min *cell N47 PE model* 132 mL/hour

Test Duration: 10 hr
required feed: 2418 mL
estimated boiler condensate: 708 mL
estimated oxidation condensate: 636 mL
effluent volume: 1320 mL
required O2: 222 L

Sampling Requirements & Instructions:

Partial Filtrates: Fe, Fe(II)
Final filtrate: Fe, Fe(II), Cu, Co, Zn, Mn, Mg
Precipitate: Fe, Cu, Co, Zn, Mn, Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and oxidation bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Synthetic Makeup File"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to TBD (120-140C), oxidation bath to 150C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel. Boiler temperature setpoint shuold be adjusted to such level that there is a continuous reflux overflow of condensate back into the boiler while at the same time drawing off the target amount of boiler condensate using the boiler condensate pump.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler and oxidation vessel overflow.
5. Collect boiler and oxidation condensate at target flowrates in separate containers. Record its volume collected as function of time.
6. Collect reactor discharge pulp in 10% HCl. The following dilution is proposed:
   a Collect 1 hours worth of discharge into 500.0 mL of 10% HCl of known weight.
   b Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate.
   c At end of test, combine all test products and allow solids to settle.
   c Identify samples as Ox1-PLS1 or Ox1-ppt1 to note PLS and precipitate collected in time period 1
7. Continue for total test duration of 8 to 10 hours

Project: 12323-001
Client: Starfield Resources

Date: 24-Mar-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 1
Purpose: To oxidize ferrous iron to ferric iron at maximum oxygen utilization
Health Safety:

Main Hazards: hot oil:
concentrated brine liquors

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Boiler RPM:
Oxidation Reactor RPM: 975

Test Data: Please record data every 15 min

Time		Preheater		Feed	Boiler			Oxidation Reactor							Oxidation Discharge					Observations
						Condensate		Oil	Bath	Vapour	O2	Condensate			Disch.	Weight	HCl+cup	HCl+cup	ORP	(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	Flow mL/min	Vol. mL	Temp ℃	Temp ℃	Temp ℃	flow mL/min	Flow mL/min	Vol. mL	pH	Set #	cup (tare g	(gross) g	+ discharge g	mV	
9:00	0.00	82.4	12043.5	0.00	126.9		206	160.0	143.5	103.3	383		207	2.02	1	88.71	592.27	669.72		09:00 Feed to the boiler started
10:07	1.12	79.9	11791.4	4.85	135.8	1.84	301	160.0	148.4	100.7	348	0.65	241							09:15 Boiler O/F started flowing to the Reactor
11:04	2.07	80.2	11606.8	3.93	139.3	1.09	353	160.0	151.8	100.6	348	0.74	276							09:17 O_2 addition was started
12:00	3.00	83.8	11186.3	7.51	139.1	1.59	442	160.0	148.9	102.6	348	0.02	277	2.09	2	87.1	592.62	676.49		09:18 Reactor O/F started flowing to the discharge collector
13:05	4.08	84.0	10806.7	5.84	139.3	1.13	515	160.0	145.0	104.4	190	0.60	316	2.01						09:30/09:45 Feed pump was plugged. Hose changed.
14:02	5.03	84.1	10409.8	6.96	139.5	1.42	596	160.0	144.7	107.3	348	1.06	376	1.89	3	88.7	610.84	756.92		10:45/10:55 Feed pump was plugged. Hose was changed,
15:05	6.08		9974.0	6.92		1.69	702				348	1.39	464							this time a Tygon #14 hose was used.
																				12:00 Discharge collector #2 start.
																				13:45 Discharge collector #3 start.
																				15:05 Shut down.
																				After shutting down the unit, it was detected that the reactor
																				OF line was plugged. This means that the discharge
																				amounts would have been bigger.
O2 utilization:																				
Totals/Avg:																				

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 1**
To oxidize ferrous iron to ferric iron at maximum oxygen utilization

Sampling Data

Sample Name		Sample Composition		Diluted Weight		Volume	pH at	SG	Wet	Dry	%H2O	Colour		Sample Analysis (Diluted Sample)						Calculated sample concentration								
type	set #	Diluent	Sample	pulp, g	PLS, g	PLS, mL	room T	g/mL	res, g	res, g		PLS	RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	% Fe(II)/Fe	Fe/Cu
Disch	1	503.56	77.45	581.01	566.6	510	1.02	1.112	Not enough solids			dark yellow/ brown		54000	4560	5000	500	5200	5500	355	30	325	33	3	34	36	8.4	10.8
Disch	2	505.52	83.87	589.39	575.0	508	0.79	1.133	Not enough solids			dark yellow/ brown		34000	3410	3500	370	3600	3300	206	21	185	21	2	22	20	10.0	9.7
Disch	3	522.14	146.08	668.22	653.3	552	0.66	1.183	Not enough solids			dark yellow/ brown		28000	2040	3100	240	2700	2700	106	8	98	12	1	10	10	7.3	9.0
Pre-heater		83.97	30.45	114.42	114.4	101	1.18	1.135	Not enough solids			green		49000	38500	5500	380	3800	2900	162	127	35	18	1	13	10	78.6	8.9
Boiler		80.79	28.38	109.17	109.2	96	1.03	1.133	Not enough solids			green		61000	50100	5800	530	5200	2800	207	170	37	20	2	18	10	82.1	10.5
Reactor		101.62	48.95	150.57	150.6	130	0.52	1.157	Not enough solids			dark yellow/ brown		87000	10300	12000	850	8600	8700	231	27	204	32	2	23	23	11.8	7.3
Pre-heater	hot sg							1.487																				
Boiler	hot sg							1.571																				
Reactor	hot sg							1.743																				
Boiler solids - HCl		290.00	36.33	326.33	242.3	214		1.131						45000	40900	1400	330	3200	680	265	241	24	8	2	19	4	90.9	32.1
O2 utilization:		199.72	27.56	227.28	225.2	211		1.068						34000	30400	1100	270	2500	530	260	233	28	8	2	19	4	89.4	30.9
Reactor solids - HCl		209.15	5.36	214.51	210.1	199		1.058						2900	960	5600	27	250	240	107	36	72	207	1	9	9	33.1	0.5

Final Filtration:

Diameter of filtration paper:	mm	Clarity of filtrate:	
type of paper (Whatman ##):		Colour of filtrate:	
Filtration T:	C	Clarity of wash:	
Filtration time:	hours	Colour of wash:	
Washing time:	day	Colour of residue:	
Volume of wash:	mL		
Cake thickness:	cm		

Other Notes / Observations

Comments	decreasing g/kg concentrations in reactor product samples
	FeCl2 crystallisation in boiler
	CuCl2 or CuCl crystallisation in reactor

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 1** **Re: Balance**

To oxidize ferrous iron to ferric iron at maximum oxygen utilization

				Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg		
run time:	6.08	hours										
avg feed flow:	6.00	g/min	feed comp:	162	127	35	18	1	13	10	g/kg	
feed:	2190	g	units in feed:	355	279	76	40	3	28	21	g	
product:	307	g pulp	units in product liquor:	60	5	55	6	1	6	6	g	
	0	g solids	solids comp:	0%			0.00%	0.00%	0.00%	0.00%	%	
			units in solids:	0	0	0	0.00	0.00	0.00	0.00	g	
Reactor START:	1.2	L	units in reactor - start:	343	20.8	322.3	51.3	3.5	36.7	29.1	g	(from makeup file)
Reactor END	2.1	kg	reactor comp:	231	27	204	32	2	23	23	g/kg	
			units in reactor - end:	484	57	427	67	5	48	48	g	
DELTA Reactor:			(accumulation)	141	36	104	15	1	11	19	g	
Total IN:				355	279	76	40	3	28	21	g	
Total OUT:				201	42	159	21	2	17	25	g	
Accountability:			(out/IN)	57	15	209	54	64	62	120	%	
Ferrous Oxidized:					238						g	
Ferrous Oxidized:					4						mole	
% Ferrous Oxidised:					**85**						%	
Oxidation rate:					0.65						g/min	

O2 flow	322	mL/min
	117	L
	20	psig
	21	C
pV=nRT	11.5	mole

Reaction **12FeCl2 + 3O2 = 2Fe2O3 + 8FeCl3**

Oxidizes: 1 mole O2
4 mole Fe

O2 utilization: **9** **%**

Project: 12323-001 **Date:** 26-Mar-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricady

Test ID: **Boil 1** *Oxidation Tests*
Purpose: **To operate boiler at 120C. Observe presence of crystals**

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors
~~pressurized oxygen~~
10% HCl
hot solutions and hot surfaces

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required
standard PPE (safety glasses, lab coat, gloves)
Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: ~~**12FeCl2 + 3O2 = 2Fe2O3 + 8FeCl3**~~

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"feed to boil liquor"	
volume:	2500	mL

Boiler:

Temperature:	120			
feed flow:	4.03	mL/min or:	**6.2**	g/min
sg:	1.55	g/mL		
prefill with:	"boiler bath liquor"			
dynamic volume:	800	mL		
Reflux Condensation:	no	(use "H" type fitting to draw off condensate at target flow)		
target condensate flow:	1.18	mL/min	*cell H47 PE model*	

Test Duration:	6	hr
required feed:	1450.8	mL
estimated boiler condensate:	424.8	mL
estimated oxidation condensate:	381.6	mL
effluent volume:	792	mL
required O2:	133.2	L

Sampling Requirements & Instructions:

Partial Filtrates: Fe, Fe(II)
Final filtrate: Fe, Fe(II), Cu, Co, Zn, Mn, Mg
Precipitate: Fe, Cu, Co, Zn, Mn, Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and oxidation bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Synthetic Makeup File"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Collect boiler condensate at target flowrates in separate containers. Record its volume collected as function of time.
6. Collect reactor discharge pulp in 10% HCl. The following dilution is proposed:
   a Collect 1 hours worth of discharge into 500.0 mL of 10% HCl of known weight.
   b Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate.
   c At end of test, combine all test products and allow solids to settle.
   c Identify samples as "test"-PLS1 or Ox1-ppt1 to note PLS and precipitate collected in time period 1
7. Continue for total test duration of 6 hours

Project: 12323-001
Client: Starfield Resources

Date: 26-Mar-10
Technologist: Hugh Given
Ernesto Bourricady

Test ID: Boil 1 *Oxidation Tests*
Purpose: To operate boiler at 120C. Observe presence of crystals

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Test Data: Please record data every 15 min

Time		Preheater		Feed	Boiler			Oxidation Reactor								Oxidation Discharge					Observations
						Condensate		Oil	Bath	Vapour		O2	Condensate			Discharge	Weight	HCl+cup	HCl+cup	Final	
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	Flow mL/min	Vol. mL	Temp ℃	Temp ℃	Temp ℃	RPM	flow mL/min	Flow mL/min	Vol. mL	pH	Set #	cup (tare) g	(gross) g	+ discharge g	Weight g	
9:13	0.00	82.9	10233.8	0.00	119.1	0.00	34									1	86.39	546.89	739.74	192.85	09:13 Feed to the boiler started
10:13	1.00	83.9	9896.2	5.63	119.0	0.79	81									2	88.48	528.73	789.06	260.33	09:34 Boiler O/F started flowing to the Reactor
11:13	2.00	82.2	9440.0	7.60	119.0	0.97	140									3	88.04	518.74	789.46	270.72	Discharge collector was changed every hour
12:13	3.00	81.8	9146.8	4.89	119.0	1.30	218									4	88.44	531.04	672.8	141.76	12:54 Suht down
12:54	3.68		8907.5	5.84		1.05	261														
Totals/Avg:																					

Sampling Data

Sample Name	time	Weight pulp, g	Weight PLS, g	Volume PLS, mL	emf at room T	pH at room T	SG g/mL				Wet res, g		Dry res, g	Colour PLS/RES.	Sample composition g		g
Discharge collector 1		653.4	640.8	566	534.0	0.9	1.132			NES	2.8	0.9	NES	Green	HCl:	460.5	sample: 192.9
Discharge collector 2		700.6	692.9	587	529.0	0.8	1.180			NES	3.8	1.2	NES	Green	HCl:	440.3	sample: 260.3
Discharge collector 3		701.4	693.6	585	531.8	0.8	1.186			NES	3.5	1.2	NES	Green	HCl:	430.7	sample: 270.7
Discharge collector 4		584.4	581.6	523	520.0	1.1	1.111			NES	2.9	1.0	NES	Green	HCl:	442.6	sample: 141.8
Pre-heater + HCl (pH 2)		107.3	106.7	97			1.096				not enough solids				HCl:	79.0	sample: 28.4
Boiler + HCl (pH 2)		113.8	104.1	95			1.097				not enough solids				HCl:	79.0	sample: 34.9
Boiler + HCl (100g/L)		125.2	115.8	98			1.179				not enough solids				HCl:	87.3	sample: 37.9
Pre-heater, no dilution											3.7						
Boiler, hot SG							1.520										
pre-heater, hot SG							1.436										
HCl (pH 2)							1.000										
HCl (100g/L)							1.045										
Boiler condensate					**521**	**2.15**	1.000										

Project: 12323-001 **Date:** 30-Mar-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 3** *Oxidation Tests*
Purpose: **To run a continuous oxidation test at 120C**

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors
pressurized oxygen
10% HCl
hot solutions and hot surfaces

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required
standard PPE (safety glasses, lab coat, gloves)
Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $12FeCl_2 + 3O_2 = 2Fe_2O_3 + 8FeCl_3$

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"feed to boil liquor - test 3"	
volume:	3500	mL
Nitrogen blanket:	75	mL/min N2

Test Duration:	10	hr
required feed:	2418	mL
estimated boiler condensate:		mL
estimated oxidation condensate:		mL
effluent volume:	1800	mL
required O2:	216	L

Boiler:

Temperature:	**120**			
feed flow:	4.03	mL/min or:	**6.2**	g/min
sg:	1.55	g/mL		
prefill with:	700 mL of "boiler bath liquor"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Oxidation Reactor:

Temperature:	**120**	℃
feed flow:	as is	
prefill with:	"oxidation bath liquor - test 3"	
dynamic volume:	1200	mL
Reflux Condensation:	**Yes**	
target condensate flow:	N/A	mL/min
O2 flow:	0.36	L/min
estimated dischareg liquid flow:	3	mL/min

RPM: 1000
location impellor: 1 cm above sparger
impellor: pitch blade (axial)

Sampling Requirements & Instructions:

Partial Filtrates: Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate: Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate: Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and oxidation bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Synthetic Makeup File - test 3"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Collect boiler condensate at target flowrates in separate containers. Record its volume collected as function of time.
6. Collect reactor discharge pulp in pH 2 HCl. The following dilution is proposed:
   a Collect 0.5 hours worth of discharge into 250.0 mL of pH 2 HCl of known weight.
   b Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
   c Identify samples as "test"-PLS1 or Ox1-ppt1 to note PLS and precipitate collected in time period 1
7. Continue for total test duration of 10 hours

Project: 12323-001
Client: Starfield Resources

Date: 30-Mar-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 3
Purpose: To operate boiler at 120C. Observe presence of crystals
Health Safety:
Main Hazards: hot oil:
concentrated brine liquors
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Boiler RPM:
Oxidation Reactor RPM: 975

Test Data: Please record data every 15 min 88.54 49.94

Time		Preheater		Feed	Boiler			Oxidation Reactor							Oxidation Discharge					Observations
						Condensate		Oil	Bath	Vapour	O2	Condensate			Disch.	Weight	HCl+cup	HCl+cup	ORP	(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	Flow mL/min	Vol. mL	Temp ℃	Temp ℃	Temp ℃	flow mL/min	Flow mL/min	Vol. mL	pH	Set #	cup (tare) g	(gross) g	+ discharge g	mV	
8:22	-0.63	81.5	9100.2		119.6		134	150.0	115.2	100.1			107		1	88.5	336.45	409.1	634	Oxidation bath started to boil @ 111°C
9:00	0.00	83.1	8905.4	5.13	119.8	4.42	302	136.0	119.6	100.7	348	1.57	167		2	48.69	298.02	333.16	652	Boiler bath started to boil @ 115.8 °C
10:01	1.02	81.1	8561.8	5.63	119.7	1.89	417	130.0	122.5	99.6	190	0.00	167		3	48.1	298.81	431.16	659	Feeding to boiler started @ 08:22
11:00	2.00	79.5	8236.0	5.52	119.9	1.89	529	130.0	118.5	97.6	348	0.00	167		4	48.65	298.31	414.35	666	Boiler O/F strarted to flow towards the reactor
12:00	3.00	84.0	7894.5	5.69	120.0	1.88	641	133.0	119.1	98.3	383	0.00	167		5	48.1	296.17	409.25	667	started @ 09:00
13:00	4.00	83.4	7569.7	5.41	119.8	1.82	751	133.0	119.5	97.7	383	0.00	167		6	48.65	279.81	390.93	664	Oxigen addition started @ 09:00
14:11	5.18	83.7	7174.7	5.56	119.6	2.05	896	133.0	119.9	97.5	383	0.00	167		7	48.56	303.21	376.83	653	Reactor O/F started to flow before feeding was
15:00	6.00	83.0	6906.8	5.47	119.9	1.95	992	133.0	120.2	97.3	383	0.00	167		8	49.21	293.72	387.31	652	started (next time solution make-up volume
16:00	7.00	80.5	6595.4	5.19	120.1	1.77	1098	133.0	120.5	97.3	383	0.00	167		9	48.1	296.06	398.89	664	should be smaller, around 100mL smaller)
16:59	7.98	79.0	6241.5	6.00	119.5	1.84	1207	133.0	119.5	94.8	383	0.00	167	2.68	10	48.65	296.51	385.2	662	10:00 O_2 tube seemed to be plugged. O_2 flowrate
															11	48.1	295.01	391.44	661	decreased to 190mL/min
															12	48.65	296.89	382.42	661	10:32 O_2 tube got plugged. It was changed. O_2
															13	48.55	277.38	365.54	663	flowrate back to set value.
															14	49.23	308.24	422.7	668	Unit shut down @ 16:59
															15	48.1	295.74	371.86	664	No solids were observed inside either the boiler
															16	49.18	294.99	333.35	648	or pre-heater @ the end of the test.
																				Reactor content was filtered @ the end of the test,
																				hot slurry "ate" the millipore paper, slurry was
																				re-filtered using filter paper Whatman #541.
Totals/Avg:																				

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 3**

Sampling Data

Sample Name		Sample Composition		Diluted Weight		Volume	pH at	SG	Wet	Dry	%H2O	Colour		Sample Analysis (Diluted Sample)						Calculated sample concentration								
type	set #	Diluent	Sample	pulp, g	PLS, g	PLS, mL	room T	g/mL	res, g	res, g		PLS	RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	% Fe(II)/Fe	Fe/Cu
Disch	1	247.95	72.65	320.60	310.2	284	1.02	1.093				yellow/brown	brown	38000	3570	2200	160	1500	1200	148	14	135	9	1	6	5	9.4	17.3
Disch	2	249.33	35.14	284.47	279.8	264	1.26	1.060				red/brown	brown	26000	2900	1400	100	1000	810	195	22	174	11	1	8	6	11.2	18.6
Disch	3	250.71	132.35	383.06	375.7	315	0.70	1.192				drak brown	orange/brown	83000	11900	4500	310	3200	2500	198	28	169	11	1	8	6	14.3	18.4
Disch	4	249.66	116.04	365.70	359.2	306	0.79	1.174	6.4	2.7	57.7	drak brown	orange/brown	74000	11100	4100	290	2900	2300	195	29	166	11	1	8	6	15.0	18.0
Disch	5	248.07	113.08	361.15	349.9	299	0.73	1.170				drak brown	orange/brown	73000	10200	3900	290	2800	2200	193	27	166	10	1	7	6	14.0	18.7
Disch	6	231.16	111.12	342.28	335.4	285	0.66	1.176	7.1	3.3	53.3	drak brown	orange/brown	78000	10400	4100	300	2900	2300	200	27	174	11	1	7	6	13.3	19.0
Disch	7	254.65	73.62	328.27	321.2	288	0.92	1.117				drak brown	red/brown	50000	6860	2700	200	1900	1600	195	27	169	11	1	7	6	13.7	18.5
Disch	8	244.51	93.59	338.10	330.3	288	0.77	1.146	5.5	3.1	44.1	drak brown	red/brown	63000	8260	3500	240	2400	1900	194	25	169	11	1	7	6	13.1	18.0
Disch	9	247.96	102.83	350.79	342.1	296	0.71	1.157				drak brown	red/brown	68000	8710	3700	260	2600	2100	196	25	170	11	1	7	6	12.8	18.4
Disch	10	247.86	88.69	336.55	324.5	285	0.77	1.140	7.5	4.4	41.3	drak brown	red/brown	59000	7420	3300	230	2300	1800	189	24	166	11	1	7	6	12.6	17.9
Disch	11	246.91	96.43	343.34	337.8	294	0.71	1.148				drak brown	red/brown	60000	7690	3400	240	2400	1900	183	23	160	10	1	7	6	12.8	17.6
Disch	12	248.24	85.53	333.77	325.3	286	0.77	1.137	7.0	4.8	31.9	drak brown	red/brown	60000	7160	3200	230	2300	1800	201	24	177	11	1	8	6	11.9	18.8
Disch	13	228.83	88.16	316.99	310.8	271	0.72	1.148				drak brown	red/brown	70000	7340	3500	240	2500	1900	215	23	192	11	1	8	6	10.5	20.0
Disch	14	259.01	114.46	373.47	363.9	312	0.61	1.165		6.3		drak brown	red/brown	70000	8080	3700	270	2600	2100	191	22	169	10	1	7	6	11.5	18.9
Disch	15	247.64	76.12	323.76	317.2	282	0.82	1.124				drak brown	red/brown	50000	5420	2900	210	2100	1700	185	20	165	11	1	8	6	10.8	17.2
Disch	16	245.81	38.36	284.17	277.2	260	1.03	1.068		3.7		drak brown	red/brown	30000	2840	1600	110	1100	880	203	19	184	11	1	7	6	9.5	18.8
Pre-heater	17	77.87	33.20	111.07	109.2	98	1.18	1.113				green		60000	44300	2300	170	1700	1300	177	131	46	7	1	5	4	73.8	26.1
Boiler	18	84.91	35.80	120.71	118.5	104	1.03	1.140				green		70000	56600	2800	210	2000	1600	203	164	39	8	1	6	5	80.9	25.0
Reactor	19	85.71	35.02	120.73	114.5	100	0.52	1.150				dark brown		70000	5430	3700	260	2600	2000	199	15	184	11	1	7	6	7.8	18.9
														Cl %	Fe %	Cu %	Co %	Ni %	Mg %									
Disc 1-4									6.4	2.7	57.7	unwashed solids		16.1	50.7	0.51	0.042	0.45	0.37									
Disc 5-6									7.1	3.3	53.3	unwashed solids		13.3	56.7	0.51	0.036	0.37	0.28									
Disc 7-8									5.5	3.1	44.1	unwashed solids		9	60.7	0.29	0.016	0.21	0.15									
Disc 9-10									7.5	4.4	41.3	unwashed solids		8.1	61.2	0.28	0.015	0.2	0.15									
Disc 11-12									7.0	4.8	31.9	unwashed solids		6.6	63.3	0.18	0.012	0.13	0.093									
Disc 13-14										6.3		unwashed solids		6.5	64.1	0.19	0.014	0.14	0.096									
Disc 15-16										3.7		unwashed solids		4.9	64.5	0.11	0.01	0.087	0.056									
Reactor Solids									192.6	133.2	30.9	washed solids		2.4	67.3	0.029	0.01	0.016	0.008									
Pre-heater	hot sg							1.432																				
Boiler	hot sg							1.529																				
Reactor	hot sg							1.645																				
Reactor	solids								192.6	133.2	30.9																	
Boiler Condensate																												
Reactor Condensate							2.68							54	20	4	< 0.3	3	2									

Final Filtration:

Diameter of filtration paper:	110	mm	Clarity of filtrate:	opaque
type of paper (Whatman ##):	541		Colour of filtrate:	Black
Filtration T:	105	C	Clarity of wash:	
Filtration time:	0.1	hours	Colour of wash:	
Washing time:		day	Colour of residue:	dk red (almost purple)
Volume of wash:		mL		
Cake thickness:	0.6	cm		

Other Notes / Observations

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 3** **Re: Balance**

				Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg		
run time:	7.98	hours										
avg feed flow:	5.51	g/min	feed comp:	177	131	46	7	1	5	4	g/kg	
feed:	2640	g	units in feed:	468	345	122	18	1	13	10	g	
product:	1438	g pulp	units in product liquor:	277	35	242	15	1	11	8	g	
	161.5	g solids	solids comp:	63%			0.05%	0.01%	0.03%	0.02%	%	
			units in solids:	102	0	0	0.08	0.02	0.05	0.03	g	
Reactor START:	1.2	L	units in reactor - start:	273.5	16.9	256.6	15.9	1.1	11.4	9	g	(from makeup file)
Reactor END	2.0	kg	reactor comp:	199	15	184	11	1	7	6	g/kg	
			units in reactor - end:	393	30	363	21	1	15	11	g	
DELTA Reactor:			(accumulation)	120	14	106	5	0	3	2	g	
Total IN:				468	345	122	18	1	13	10	g	
Total OUT:				499	49	348	20	1	14	11	g	
Accountability:			(out/IN)	107	14	285	112	109	105	106	%	
Ferrous Oxidized:					297						g	
Ferrous Oxidized:					5						mole	
% Ferrous Oxidised:					**86**						%	
Oxidation rate:					0.62						g/min	
O2 flow	354	mL/min										
	169	L										
	20	psig										
	21	C										
pV=nRT	16.6	mole										

Reaction **12FeCl2 + 3O2 = 2Fe2O3 + 8FeCl3**

Oxidizes: 1 mole O2 / 4 mole Fe

O2 utilization: **8** **%**

Project: 12323-001 **Date:** 05-Apr-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 4** *Oxidation Tests*
Purpose: **To simulate McGill Oxidation test RT68**

Health Safety:

Main Hazards:
- hot oil: 4 mL/min
- concentrated brine liquors 199 g/L Fe
- pressurized oxygen 0.2653 g/min ppt
- pH 2 HCl 0.396 g/min hematite
- hot solutions and hot surfaces 23.761

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: **12FeCl2 + 3O2 = 2Fe2O3 + 8FeCl3**

Conditions:

Preheater:

				0	0	0	Duration:	10	hr
Temperature:	80	℃					required feed:	1710	mL
Reflux Condensation:	yes	(back into preheater)					estimated boiler condensate:		mL
prefill with:	"feed to boil liquor - test 4"						estimated oxidation condensate:		mL
volume:	3500	mL					effluent volume:	0	mL
Nitrogen blanket:	75	mL/min N2					required O2:	900	L

Boiler:

Temperature:	110			
feed flow:	2.85	mL/min or:	**4.4**	g/min
sg:	1.55	g/mL		
prefill with:	"boil bath - test 4"			
dynamic volume:	700	mL		
Reflux Condensation:	no	(use "H" type fitting to draw off condensate at target flow)		
target condensate flow:	as req.	mL/min		

Oxidation Reactor:

Temperature:	140	℃	RPM:	1000
feed flow:	as is		location impellor:	mm of bottom
prefill with:	"oxidation bath liquor - test 4"		impellor:	pitch blade (axial)
dynamic volume:	1200	mL	location sparger:	mm below impellor
Reflux Condensation:	**no**	all to go back into reactor		
target condensate flow:	as req.	mL/min		
O2 flow:	**1.5**	L/min		
estimated dischareg liquid flow:	0	mL/min		

Sampling Requirements & Instructions:

- Partial Filtrates: Fe, Fe(II), Cu, Co, Ni, Mg
- Final filtrate: Fe, Fe(II), Cu, Co, Ni, Mg
- Precipitate: Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and oxidation bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Synthetic Makeup File - test 4"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 110C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel. Heat up oxidation reactor bath to 150C - measure drawn off condensate to get to this point
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows. Turn on oxygen flow at target flow once boiler feeds into reactor
5. Collect boiler condensate (if any) in separate containers. Record its volume collected as function of time.
6. Collect reactor discharge pulp (if any) in pH 2 HCl. The following dilution is proposed:
   a. Collect 1 hours worth of discharge into 400.0 mL of pH 2 HCl of known weight.
   b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
   c. Identify samples as "test"-PLS1 or Ox1-ppt1 to note PLS and precipitate collected in time period 1
7. Continue for total test duration (meaning feed flow and oxygen flow) of 5 hours and 20 min

Project: 12323-001
Client: Starfield Resources

Date: 05-Apr-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: **Test 4**
Purpose: **To operate boiler at 120C. Observe presence of crystals**

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Boiler RPM: 150
Oxidation Reactor RPM: 1000
Impellor: 45 mm off bottom type: pitch
sparger: 10 mm below impellor type: U type

Test Data: Please record data every 15 min 88.54 49.94

Time		Preheater		Feed	Boiler			Oxidation Reactor							Oxidation Discharge					Observations
						Condensate		Oil	Bath	Vapour	O2		Condensate		Disch.	Weight	HCl+cup	HCl+cup	ORP	(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	Flow mL/min	Vol. mL	Temp ℃	Temp ℃	Temp ℃	flow mL/min	Flow mL/min	Vol. mL	pH	Set #	cup (tare g	(gross) g	+ discharge g	mV	
				4.45																
11:00	0.00	81.9	9654.4		106.8	0.00	0	160.0	143.5	104.8	1500		1386							
12:00	1.00	82.5	9356.5	4.96	106.7	0.00	0	160.0	142.3	98.6	1500	1.48	1475		1	87.66	478.31	576.05	625	
13:00	2.00	79.7	9117.5	3.98	105.6	0.00	0	160.0	0.0	0.0	0	1.38	1557		2	88.51	465.04	641.98	645	
14:00	3.00	80.0	8815.0	5.04	106.4	0.00	0	160.0	135.7	97.6	1500	1.74	1662		3	88.04	501.72	696.52	647	Oxygen flow decreased to 1200mL/min pulled
15:00	4.00	79.5	8522.6	4.87	105.8	0.00	0	160.1	135.7	98.3	1500	2.00	1782	2.05	4	89.11	486.62	610.71	634	cleaned operation took less than 5 min
16:20	5.33	81.8	8112.9	5.13	105.2	0.00	0	160.0	136.9	102.6	1500	1.79	1924		5	50.17	454.64	609.8	632	shutdown
Totals/Avg:																				

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 4**

Sampling Data

Sample Name type	set #	Sample Composition Diluent	Sample	Diluted Weight pulp, g	PLS, g	Volume PLS, mL	**pH at room T**	SG g/mL	Wet res, g	Dry res, g	%H2O	Colour PLS	RES
disch	1	390.65	97.74	488	478	535	0.85	1.119				lt brown	brown/red
disch	2	376.53	176.94	553	538	648	0.61	1.206	11.8	7.5	36.6	brown	brown/red
disch	3	413.68	194.80	608	590	710	0.67	1.204	11.3	7.3	35.0	brown	brown/red
disch	4	397.51	124.09	522	510	582	0.73	1.141	8.1	4.9	38.8	brown	brown/red
disch	5	404.47	155.16	560	543	635	0.58	1.170	13.6	8.6	36.8	brown	brown/red
Preheater		78.77	28.42	107	104	114	1.26	1.090	n/a			green	brown/red
Boiler		82.81	30.29	113	110	117	1.25	1.064	n/a			green	brown/red
Reactor		79.25	32.05	111	106	124	0.38	1.171	n/a			brown	purple/red
Reactor solids									155.5	98.3	36.8		purple/red
HOT SG's									0.0	0.0	0.0		
Preheater								1.406					
Boiler								1.519					
Reactor								1.756					
Disc 1-2 Solids													
Disc 3 Solids													
Disc 4 Solids													
Disc 5 Solids													
Reactor Solids													
Reactor Solids													

Sample Analysis (Diluted Sample)

Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L
55000	13900	1700	10	21	42
97000	23400	3400	17	37	79
100000	22600	3600	17	39	70
71000	14900	2600	13	31	42
80000	16800	3200	15	37	45
50000	20900	1900	10	30	2
35000	28900	1400	7	22	2
87000	17400	3500	16	40	50

Calculated sample concentration

Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg mg/L	Fe(II)/Fe	Fe/Cu
301	76	225	9	0	0	0	25.3	32.4
356	86	270	12	0	0	0	24.1	28.5
365	82	282	13	0	0	0	22.6	27.8
333	70	263	12	0	0	0	21.0	27.3
328	69	259	13	0	0	0	21.0	25.0
200	83	116	8	0	0	0	41.8	26.3
135	112	24	5	0	0	0	82.6	25.0
338	68	270	14	0	0	0	20.0	24.9

Fe %	Cu %	Co %	Ni %	Mg %	Cl%
53.2	0.3	<0.01	0.012	0.009	17.000
55.10	0.22	<0.01	0.011	0.007	15.400
65.70	0.19	<0.01		0.006	4.300
64.80	0.47	<0.01	0.014	<0.005	4.330
63.6	0.3	<0.01		0.009	3.670

Final Filtration:

Diameter of filtration paper:	mm	Clarity of filtrate:	
type of paper (Whatman ##):		Colour of filtrate:	
Filtration T:	C	Clarity of wash:	
Filtration time:	hours	Colour of wash:	
Washing time:	day	Colour of residue:	
Volume of wash:	mL		
Cake thickness:	cm		

Other Notes / Observations

all samples millipored except final reactor used whatman 541

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 4** **Re: Balance**

				Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg		
run time:	5.33	hours										
avg feed flow:	4.80	g/min	feed comp:	162	157	5	7	0	0	0	g/kg	
feed:	1535	g	units in feed:	249	241	8	10	0	0	0	g	
product:	749	g pulp	units in product liquor:	255	58	197	9	0	0	0	g	
	126.6	g solids	solids comp:	63%			0.29%	0%	0%	0%	%	
			units in solids:	79	0	0	0.36	0	0	0	g	
Reactor START:	1.2	L	units in reactor - start:	367	105	263	11	0	0	0	g	(from makeup file)
Reactor END	2.1	kg	reactor comp:	338	68	270	14	0	0	0	g/kg	
			units in reactor - end:	711	142	569	29	0	0	0	g	
DELTA Reactor:			(accumulation)	344	38	306	17	0	0	0	g	
Total IN:				249	241	8	10	0	0	0	g	
Total OUT:				679	96	504	27	0	0	1	g	
Accountability:			(out/IN)	273	40	6678	260	0	0	0	%	
Ferrous Oxidized:					146						g	
Ferrous Oxidized:					3						mole	
% Ferrous Oxidised:					**60**						%	
Oxidation rate:					0.46						g/min	
O2 flow	1250	mL/min										
	400	L										
	20	psig										
	21	C										
pV=nRT	39.1	mole										
Reaction			**12FeCl2 + 3O2 = 2Fe2O3 + 8FeCl3**									
			1 mole O2									
		Oxidizes:	4 mole Fe									
O2 utilisation:			**1.7**	**%**								

Project: 12323-001 **Date:** 06-Apr-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 5** *Oxidation Tests*
Purpose: **To run a continuous oxidation test at 130C**

Health Safety:

Main Hazards:
- hot oil:
- concentrated brine liquors
- pressurized oxygen
- 10% HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: **$12FeCl_2 + 3O_2 = 2Fe_2O_3 + 8FeCl_3$**

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"feed to boil liquor - test 3"	
volume:	3500	mL
Nitrogen blanket:	75	mL/min N2

Test Duration: 15 hr

required feed:	3627	mL
estimated boiler condensate:		mL
estimated oxidation condensate:		mL
effluent volume:	2700	mL
required O2:	324	L

Boiler:

Temperature:	**120**	℃		
feed flow:	4.03	mL/min or:	**5.8**	g/min
sg:	1.43	g/mL		
prefill with:	700 mL of "boiler bath liquor"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate at target flow)		
target condensate flow:	as req.	mL/min		

Oxidation Reactor:

Temperature:	**130**	℃	RPM:	1000
feed flow:	as is			
prefill with:	"oxidation bath liquor - test 3"			
dynamic volume:	1100	mL		
Reflux Condensation:	**No**			
target condensate flow:	N/A	mL/min		
O2 flow:	0.36	L/min		
estimated dischareg liquid flow:	3	mL/min		

Sampling Requirements & Instructions:

- Partial Filtrates: Fe, Fe(II), Cu, Co, Ni, Mg
- Final filtrate: Fe, Fe(II), Cu, Co, Ni, Mg
- Precipitate: Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and oxidation bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Synthetic Makeup File - test 3"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 130C, reactor to 130C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel.
3. Once target T has been reached, take a profile sample of each vessel. Feed to boiler can be started once all vessels have reached target temperature and start samples have been taken. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Collect boiler condensate at target flowrates in separate containers. Record its volume collected as function of time.
6. Collect reactor discharge pulp in pH 2 HCl. The following dilution is proposed:
   a. Collect 1 hours worth of discharge into 500.0 mL of pH 2 HCl of known weight.
   b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any) and displacement wash solids with 100 mL pH 2 HCl. Solids will be submitted separately or combined (dependent on availability)
   c. Identify samples as "test"-PLS1 or Ox1-ppt1 to note PLS and precipitate collected in time period 1
7. Continue for total test duration of 15 hours
8. Recover solids from reactor at end of test. Repulp wash and determine wet and dry weight.

Project: 12323-001
Client: Starfield Resources

Date: 06-Apr-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 5

Purpose: To operate boiler at 120C. Observe presence of crystals

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Boiler RPM:
Oxidation Reactor RPM:

Sparger Location:
Impellor Location:

Test Data:

Time		Preheater		Feed	Boiler			Oxidation Reactor							Oxidation Discharge					Observations
						Condensate						Condensate								
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	Flow mL/min	Vol. mL	Oil Temp ℃	Bath Temp ℃	Vapour Temp ℃	O2 flow mL/min	Flow mL/min	Vol. mL	pH	Disch. Set #	Weight cup (tare g	HCl+cup (gross) g	HCl+cup + discharge g	ORP mV	(continue on other page if requried)
8:20	-2.17	79.9	10589.8		119.6	0.00	255	140.0	130.9	103.1	365	0.00	55	2.09						Boiler O/F started to flow @ 9:11
9:20	-1.17	81.0	10220.9	6.15	119.8	0.96	312	137.0	128.3	96.4	365	0.00	55							Reactor discharge started to flow @10.38
10:30	0.00	79.6	9782.5	6.26	119.9	1.50	417	147.0	126.0	98.6	365	0.00	55							Some solids started to build up in the reactor
11:30	1.00	80.9	9392.1	6.51	119.8	1.97	535	140.0	127.9	103.0	365	0.42	81		1	49.97	502.66	614.52	644	discharge tube
12:30	2.00	82.2	9031.1	6.02	120.1	1.88	648	145.0	128.7	103.7	365	0.55	113	2.07	2	49.36	516.37	698.82	652	19:30 Air sparge plugged
13:30	3.00	83.1	8661.7	6.16	120.0	2.46	796	144.0	130.7	102.9	365	0.00	113	2.1	3	49.97	483.11	684.03	647	20:00 Feed stopped. In order to unplugged the air
14:52	4.37	81.8	8231.7	5.24	120.0	2.01	960	144.5	133.2	104.7	365	0.06	119		4	49.39	493.75	692.55	643	sparge it was neccesary to take the reactor lid off
15:30	5.00	80.2	7962.0	7.10	119.5	2.48	1055	144.5	130.1	99.6	365	0.13	123	2.05	5	49.97	499.5	683.03	642	21:40 Air sparge unplugged.
16:34	6.07	80.2	7579.1	5.98	119.6	2.08	1188	144.6	130.6	99.7	365	0.08	128		6	49.53	501.98	610.98	643	21:40 Since everything was stopped and we
17:36	7.10	79.1	7228.4	5.66	119.6	2.28	1329	144.5	130.1	99.5	348	0.09	134	2.04	7	49.97	518.6	728.43	649	were running out of time, the unit was shut down.
18:30	8.00	83.9	6908.2	5.93	119.8	1.94	1434	144.5	130.2	99.7	348	0.10	139		8	49.43	532.58	721.01	643	At the end of the test "orange" solids were
19:31	9.02	88.1	6491.8	6.83	119.8	2.32	1576	144.5	132.3	100.8	348	0.09	145	2.02	9	49.96	525.55	782.71	642	observed in both the pre-heater and the boiler.
Shut down			6275.6				1625						164							There were solids in the reactor walls, they look
																				like cristalized solids, no hematite.
Totals/Avg:																				

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 5**

Sampling Data

Sample Name		Sample Composition		Diluted Weight		Volume	pH at	SG	Wet	Dry	Colour		Sample Analysis (Diluted Sample)						Calculated sample concentration								
type	set #	Diluent	Sample	pulp, g	PLS, g	PLS, mL	room T	g/mL	res, g	res, g	PLS	RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	% Fe(II)/Fe	Fe/Cu
Pre-heater	start - 8:30	91.27	31.87	123.14	121.5	111	**0.20**	1.095			yel/brown		46000	37800	1900	130	1400	1100	160	132	29	7	0.5	5	4	82.2	24.2
Boiler	start - 8:30	83.11	37.83	120.94	119.12	104	**0.30**	1.148			yel/brown		73000	60200	3000	200	2100	1700	200	165	35	8	0.5	6	5	82.5	24.3
Reactor	start - 8:30	89.25	28.03	117.28	114.75	101	**0.29**	1.136			yel/brown		57000	6600	3400	230	2300	1900	205	24	182	12	0.8	8	7	11.6	16.8
Pre-heater	mid - 15:00	86.92	26.33	113.3	111.9	103	0.32	1.084			yel/brown		40000	34300	1800	110	1200	950	157	134	22	7	0.4	5	4	85.8	22.2
Boiler	mid - 15:00	82.42	34.69	117.1	117.1	103	0.33	1.139			yel/brown		69000	57100	2800	180	2000	1600	205	169	35	8	0.5	6	5	82.8	24.6
Reactor	mid - 15:00	78.07	22.69	100.8	97.6	86	0.33	1.132			yel/brown		55000	11700	3200	210	2200	1800	209	44	165	12	0.8	8	7	21.3	17.2
Pre-heater	end - 21:30	84.04	38.82	122.86	113.9	102	0.34	1.120					59000	46800	2400	170	1700	1400	155	123	32	6	0.4	4	4	79.3	24.6
Boiler	end - 21:30	84.28	32.21	116.49	111.4	99	0.62	1.129					61000	50100	2800	180	1900	1600	187	153	33	9	0.6	6	5	82.1	21.8
Reactor	end - 21:30	85.07	37.67	122.74	121.5	103	0.23	1.183					81000	19400	4500	250	3000	2500	221	53	168	12	0.7	8	7	24.0	18.0
Disch	1	452.69	111.86	564.55	553.8	499	0.32	1.109			yel/brown		45000	5600	2600	170	1800	1500	201	25	176	11.6	0.8	8.0	6.7	12.4	17.3
Disch	2	467.01	182.45	649.46	638.4	549	0.28	1.163			yel/brown		68000	9700	4000	260	2800	2200	205	29	175	12.0	0.8	8.4	6.6	14.3	17.0
Disch	3	433.14	200.92	634.06	622.4	524	0.29	1.187			yel/brown		80000	15000	4800	310	3300	2700	209	39	170	12.5	0.8	8.6	7.0	18.8	16.7
Disch	4	444.36	198.80	643.16	632.0	532	0.21	1.187	15.6	12.1	(1 - 4)	brown	81000	15500	4800	300	3200	2600	217	42	175	12.9	0.8	8.6	7.0	19.1	16.9
Disch	5	449.53	183.53	633.06	625.2	536	0.23	1.166			yel/brown		74000	15600	4300	260	2800	2300	216	46	171	12.6	0.8	8.2	6.7	21.1	17.2
Disch	6	452.45	109.00	561.45	550.8	497	0.40	1.109			yel/brown		47000	7500	2400	150	1800	1500	214	34	180	10.9	0.7	8.2	6.8	16.0	19.6
Disch	7	468.63	209.83	678.46	672.4	568	0.10	1.184			yel/brown		83000	13800	4700	270	3100	2600	225	37	187	12.7	0.7	8.4	7.0	16.6	17.7
Disch	8	483.15	188.43	671.58	665.6	573	0.12	1.161			yel/brown		70000	16300	4500	290	2800	2100	213	50	163	13.7	0.9	8.5	6.4	23.3	15.6
Disch	9	475.59	257.16	732.75	709.9	589	-0.04	1.206	30.6	24.6	(5 - 9)	brown	93000	19600	5600	340	3600	2800	213	45	168	12.8	0.8	8.2	6.4	21.1	16.6
													Cl %	Fe %	Cu %	Co %	Ni %	Mg %									
Disc 1-4									15.6	12.1			2.59	68.5	0.02	<0.01	0.03	<0.005	8								
Disc 5-9									30.6	24.6			2.37	70.4	0.02	<0.01	0.02	<0.005	17								
									167.8	137.4			1.74	69.6	0.02	<0.01	0.01	<0.005	96								
Pre-heater	hot sg							1.447																			
Boiler	hot sg							1.583																			
Reactor	hot sg							1.721																			
Reactor	solids								167.8	137.4		brown															
Boiler Condensate																											
Reactor Condensate																											

Final Filtration:

Diameter of filtration paper:	mm	Clarity of filtrate:	
type of paper (Whatman ##):		Colour of filtrate:	
Filtration T:	C	Clarity of wash:	
Filtration time:	hours	Colour of wash:	
Washing time:	day	Colour of residue:	
Volume of wash:	mL		
Cake thickness:	cm		

Other Notes / Observations

solids were clay like and seemed very packed on filter unlike previous tests.

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 5** **Re: Balance**

				Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg		
run time:	9.02	hours										
avg feed flow:	6.17	g/min	feed comp:	157	130	28	7	0.4	5	4	g/kg	
feed:	3337	g	units in feed:	524	432	92	22	1.5	16	12	g	
product:	1642	g pulp	units in product liquor:	350	65	285	21	1.3	14	11	g	
	174.07	g solids	solids comp:	69.6%			0.018%	0.010%	0.013%	0.005%	%	
			units in solids:	121	0	0	0.03	0.02	0.02	0.01	g	
Reactor START:	1.2	L	units in reactor - start:	424	49	375	25	2	17	14	g	(from profile sampling)
Reactor END	2.1	kg	reactor comp:	221	53	168	12	1	8	7	g/kg	
			units in reactor - end:	456	109	347	25	1	17	14	g	
DELTA Reactor:			(accumulation)	32	60	-28	0	0	0	0	g	
Total IN:				524	432	92	22	1	16	12	g	
Total OUT:				503	125	257	21	1	14	11	g	
Accountability:			(out/IN)	96	29	278	93	67	87	88	%	
Ferrous Oxidized:					307						g	
Ferrous Oxidized:					5						mole	
% Ferrous Oxidised:					**71**						%	
Oxidation rate:					0.57						g/min	
O2 flow	361	mL/min										
	195	L										
	20	psig										
	21	C										
pV=nRT	19.1	mole										

Reaction

12FeCl2 + 3O2 = 2Fe2O3 + 8FeCl3

1 mole O2

4 mole Fe

7 %

Project: 12323-001 **Date:** 13-Apr-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 6** *Oxidation Tests*
Purpose: **To run a continuous oxidation test at 110C**

Health Safety:

Main Hazards:
- hot oil:
- concentrated brine liquors
- pressurized oxygen
- 10% HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: **$12FeCl_2 + 3O_2 = 2Fe_2O_3 + 8FeCl_3$**

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"feed to boil liquor - test 3"	
volume:	3500	mL
Nitrogen blanket:	75	mL/min N2

Test Duration: 10 hr

required feed:	2418	mL
estimated boiler condensate:		mL
estimated oxidation condensate:		mL
effluent volume:	1800	mL
required O2:	216	L

Boiler:

Temperature:	**110**			
feed flow:	4.03	mL/min or:	**6.2**	g/min
sg:	1.55	g/mL		
prefill with:	700 mL of "boiler bath liquor"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Oxidation Reactor:

Temperature:	**110**	℃
feed flow:	as is	
prefill with:	"oxidation bath liquor - test 3"	
dynamic volume:	1200	mL
Reflux Condensation:	**Yes**	
target condensate flow:	N/A	mL/min
O2 flow:	0.36	L/min
estimated dischareg liquid flow:	3	mL/min

RPM: 1000
location impellor: 1 cm above sparger
impellor: pitch blade (axial)

Sampling Requirements & Instructions:

Partial Filtrates: Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate: Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate: Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and oxidation bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Synthetic Makeup File - test 3"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Collect boiler condensate at target flowrates in separate containers. Record its volume collected as function of time.
6. Collect reactor discharge pulp in pH 2 HCl. The following dilution is proposed:
   a. Collect 0.5 hours worth of discharge into 250.0 mL of pH 2 HCl of known weight.
   b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
   c. Identify samples as "test"-PLS1 or Ox1-ppt1 to note PLS and precipitate collected in time period 1
7. Continue for total test duration of 10 hours
8. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash and determine wet and dry weight.

Project: 12323-001
Client: Starfield Resources

Date: 13-Apr-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 6

Purpose: To operate boiler at 120C. Observe presence of crystals

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Boiler RPM:
Oxidation Reactor RPM: 975
O2 line pressure: 20 psig

Test Data:

Time		Preheater		Feed	Boiler			Oxidation Reactor							Oxidation Discharge					Observations
						Condensate		Oil	Bath	Vapour	O2	Condensate			Disch.	Weight	HCl+cup	HCl+cup	ORP	(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	Flow mL/min	Vol. mL	Temp ℃	Temp ℃	Temp ℃	flow mL/min	Flow mL/min	Vol. mL	pH	Set #	cup (tare g	(gross) g	+ discharge g	mV	
7:45	0.00	76.1	9472.0		112.4	0.00	0	120.0	122.0	97.6	365	0.00	0		0	49.95	446.10	823.50	692.8	Feed was started @ 7:45
8:46	1.02	81.1	9180.7	4.78	109.7	0.00	0	122.0	108.8	96.1	365	0.00	0		1	49.91	445.81	663.65	626.0	O_2 addition was started @ 7:50
9:45	2.00	83.0	8966.8	3.63	109.5	0.00	0	124.0	110.3	96.1	365	0.00	0		2	50.00	450.61	620.55	652.3	09:10 Feed tube clogged. Tube was changed.
10:45	3.00	83.1	8540.4	7.11	110.5	0.61	36	123.0	110.9	95.5	365	0.00	0		3	49.99	455.32	863.30	645.1	09:20 O2 flowrate decreased. The end of the tube
11:45	4.00	80.1	8300.8	3.99	111.1	0.52	68	123.0	111.0	95.9	365	0.00	0		4	50.44	438.03	714.26	631.8	was cut and kept for ?analysis?
12:46	5.02	82.4	7936.2	5.98	110.5	0.62	105	123.0	110.8	95.8	331	0.00	0		5	49.97	450.91	696.89	629.5	11:40 Feed tube clogged. Tube was changed.
13:45	6.00	83.5	7553.9	6.48	111.6	0.68	145	123.0	110.9	95.7	348	0.00	0		6	49.91	454.42	720.95		14:00 Condensate from reactor started to overflow,
14:45	7.00	81.1	7267.4	4.77	109.1	0.72	189	123.0	110.6	95.5	348	0.00	0		7	49.95	432.30	691.37	621.8	next test should start with the "h" tube full.
15:45	8.00	83.1	6930.8	5.61	109.1	0.72	232	123.1	110.3	95.4	348	0.00	0		8	49.93	453.09	734.99	628.9	In the "end" profile, some orange solids appeared in
16:48	9.05	83.5	6563.8	5.83	111.3	0.80	282	123.0	110.2	95.6	348	0.00	0		9	49.91	463.52	758.96	629.1	the pre-heater and boiler samples.
17:45	10.00	85.8	6206.6	6.27	110.5	0.81	328	123.0	110.0	95.6	348	0.00	0		10	49.93	453.15	650.60	617.9	Filter cakes start craking even before they were
			3265.4																	completely dry.
Totals/Avg:																				

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 6**

Sampling Data

Sample Name		Sample Composition		Diluted Weight		Volume	pH at	SG	Wet	Dry	%H2O	Colour		Sample Analysis (Diluted Sample)						Calculated sample concentration								
type	set #	Diluent	Sample	pulp, g	PLS, g	PLS, mL	room T	g/mL	res, g	res, g		PLS	RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	% Fe(II)/Fe	Fe/Cu
Pre-heater	start - 7:40	87.38	30.93	118.31	109.3	100	1.24	1.098				green		45000	37800	2100	130	1400	1100	145	122	23	7	0	5	4	84.0	21.4
Boiler	start - 7:40	87.82	34.27	122.09	120.9	106	1.17	1.146				green		69000	56600	3000	190	2100	1600	212	174	38	9	1	6	5	82.0	23.0
Reactor	start - 7:40	91.02	24.65	115.67	113.2	104	1.16	1.086				brown		33000	2700	2000	130	1400	1100	140	11	128	8	1	6	5	8.2	16.5
Pre-heater	mid - 12:30	88.04	26.99	115.03	112.9	104	1.41	1.086				green		40000	33700	1700	120	1200	950	154	130	24	7	0	5	4	84.3	23.5
Boiler	mid - 12:30	83.36	28.97	112.33	110.8	101	1.50	1.098				green		45000	37400	2000	130	1400	1100	157	130	26	7	0	5	4	83.1	22.5
Reactor	mid - 12:30	81.81	37.55	119.36	116.6	102	1.16	1.148				brown		58000	18800	3200	210	2300	1800	157	51	106	9	1	6	5	32.4	18.1
Pre-heater	end - 17:10	85.87	25.92	111.79	108.3	100	1.40	1.085				green		38000	33100	1700	110	1200	850	146	127	19	7	0	5	3	87.1	22.4
Boiler	end - 17:10	84.72	28.91	113.63	111.3	102	1.48	1.094				green		44000	36100	1900	130	1400	1000	155	127	28	7	0	5	4	82.0	23.2
Reactor	end - 17:10	83.87	28.56	112.43	104.4	94	1.16	1.108				brown		44000	8500	2400	160	1700	1300	145	28	117	8	1	6	4	19.3	18.3
Disch	0	396.15	377.40	773.55	760.7	628	1.01	1.211				brown		85000	5500	5400	330	3600	2700	141	9	132	9	1	6	4	6.5	15.7
Disch	1	395.90	217.84	613.74	604.5	525	1.06	1.151				brown		59000	7100	3600	230	2500	1900	142	17	125	9	1	6	5	12.0	16.4
Disch	2	400.61	169.94	570.55	563.2	495	1.13	1.137				brown		52000	8300	3000	200	2100	1700	152	24	127	9	1	6	5	16.0	17.3
Disch	3	405.33	407.98	813.31	788.9	633	0.89	1.247				brown		110000	26500	6300	370	4000	3100	171	41	129	10	1	6	5	24.1	17.5
Disch	4	387.59	276.23	663.82	641.1	536	1.06	1.195				brown		82000	26100	4700	290	3200	2400	159	51	109	9	1	6	5	31.8	17.4
Disch	5	400.94	245.98	646.92	623.1	530	0.98	1.175				brown		74000	23700	4000	260	2800	2200	160	51	108	9	1	6	5	32.0	18.5
Disch	6	404.51	266.53	671.04	652.6	560	1.03	1.166				brown		74000	35200	3800	240	2700	2000	155	74	81	8	1	6	4	47.6	19.5
Disch	7	382.35	259.07	641.42	625.0	535	1.08	1.169				brown		76000	35500	3800	240	2700	2000	157	73	84	8	0	6	4	46.7	20.0
Disch	8	403.16	281.90	685.06	660.8	558	0.90	1.185				brown		79000	25000	4300	290	3100	2400	156	49	107	9	1	6	5	31.6	18.4
Disch	9	413.61	295.44	709.05	685.1	578	0.92	1.185				brown		80000	25800	4200	280	3000	2300	157	50	106	8	1	6	5	32.3	19.0
Disch	10	403.22	197.45	600.67	588.3	517	1.09	1.138				brown		57000	20900	3000	210	2200	1700	149	55	95	8	1	6	4	36.7	19.0
														Cl %	Fe %	Cu %	Co %	Ni %	Mg %									
Disc 0-2									5.5	2.0	73%			5.2	59.8	0.012	<0.01	<0.002	<0.005	1.20								
Disc 3-4									33.4	8.3	80%			6.9	58.6	0.014	<0.01	<0.002	<0.005	4.88								
Disc 5-6									33.3	9.0	79%			5.7	59.7	0.019	<0.01	0.005	<0.005	5.38								
Disc 7-8									36.1	11.1	76%			4.6	61.7	0.013	<0.01	<0.002	<0.005	6.86								
Disc 9-10									28.5	19.7	59%			3.8	64.1	0.012	<0.01	0.002	<0.005	12.60								
Reactor Solids									221.0	122.1	64%			3.4	64.7	0.019	<0.01	0.005	0.010	78.97								
Pre-heater	hot sg							1.401																				
Boiler	hot sg							1.417																				
Reactor	hot sg							1.525																				
Boiler Condensate																												
Reactor Cor	20						2.68																					

Final Filtration:

Diameter of filtration paper:		mm	Clarity of filtrate:
type of paper (Whatman ##):			Colour of filtrate:
Filtration T:		C	Clarity of wash:
Filtration time:		hours	Colour of wash:
Washing time:		day	Colour of residue:
Volume of wash:		mL	
Cake thickness:		cm	

Other Notes / Observations

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 6** **Re: Balance**

				Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg		
run time:	10.00	hours										
avg feed flow:	5.44	g/min	feed comp:	148	126	22	7	0	5	3	g/kg	
feed:	3266	g	units in feed:	485	412	72	22	1	15	11	g	
product:	2996	g pulp	units in product liquor:	465	133	332	26	2	18	14	g	
	172.17	g solids	solids comp:	64%			0.017%	0.010%	0.004%	0.009%	%	
			units in solids:	110	0	0	0.03	0.02	0.01	0.01	g	
Reactor START:	1.2	L	units in reactor - start:	255	21	235	15	1	11	9	g	(from makeup file)
Reactor END	1.8	kg	reactor comp:	145	28	117	8	1	6	4	g/kg	
			units in reactor - end:	265	51	214	14	1	10	8	g	
DELTA Reactor:			(accumulation)	10	30	-20	-1	0	-1	-1	g	
Total IN:				485	412	72	22	1	15	11	g	
Total OUT:				585	163	311	25	2	17	13	g	
Accountability:			(out/IN)	121	40	432	116	115	116	114	%	
Ferrous Oxidized:					249						g	
Ferrous Oxidized:					4						mole	
% Ferrous Oxidised:					**60**						%	
Oxidation rate:					0.42						g/min	
O2 flow	353	mL/min										
	212	L										
	20	psig										
	21	C										
pV=nRT	20.7	mole										

Reaction **12FeCl2 + 3O2 = 2Fe2O3 + 8FeCl3**

1 mole O2

Oxidizes: 4 mole Fe

O2 utilisation: **5** **%**

Project: 12323-001 **Date:** 15-Apr-10

Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy
Charlotte Forstner

Test ID: **Test 7** *Oxidation Tests*

Purpose: To run a continuous oxidation test at 120C with rushton turbine impellor installed

Health Safety:

Main Hazards:
- hot oil:
- concentrated brine liquors
- pressurized oxygen
- 10% HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: **$12FeCl_2 + 3O_2 = 2Fe_2O_3 + 8FeCl_3$**

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"feed to boil liquor - test 3"	
volume:	3500	mL
Nitrogen blanket:	75	mL/min N2

Boiler:

Temperature:	**120**			
feed flow:	4.03	mL/min or:	**6.2**	g/min
sg:	1.55	g/mL		
prefill with:	700 mL of "boiler bath liquor"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Oxidation Reactor:

Temperature:	**120**	℃
feed flow:	as is	
prefill with:	"oxidation bath liquor - test 3"	
dynamic volume:	1200	mL
Reflux Condensation:	**Yes**	
target condensate flow:	N/A	mL/min
O2 flow:	0.36	L/min
estimated dischareg liquid flow:	3	mL/min

Test Duration:	10	hr
required feed:	2418	mL
estimated boiler condensate:		mL
estimated oxidation condensate:		mL
effluent volume:	1800	mL
required O2:	216	L

RPM: **1200**
location impellor: 1 cm above sparger
impellor: **rushton turbine (radial**

Sampling Requirements & Instructions:

- Partial Filtrates: Fe, Fe(II), Cu, Co, Ni, Mg
- Final filtrate: Fe, Fe(II), Cu, Co, Ni, Mg
- Precipitate: Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and oxidation bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Synthetic Makeup File - test 3"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Collect boiler condensate at target flowrates in separate containers. Record its volume collected as function of time.
6. Collect reactor discharge pulp in pH 2 HCl. The following dilution is proposed:
   a. Collect 1 hours worth of discharge into 500.0 mL of pH 2 HCl of known weight.
   b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
   c. Identify samples as "test"-PLS1 or Ox1-ppt1 to note PLS and precipitate collected in time period 1
7. Continue for total test duration of 10 hours
8. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash and determine wet and dry weight.

Project: 12323-001
Client: Starfield Resources

Date: 15-Apr-10
Technologist: Hugh Given
Ernesto Bourricaudy
Charlotte Forstner

Test ID: **Test 7**

Purpose: **To operate boiler at 120C. Observe presence of crystals**

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Boiler RPM:
Oxidation Reactor RPM: **1200**
O2 line pressure: psig

Impellor: **Rushton Turbine**

Test Data:

Time		Preheater		Feed	Boiler			Oxidation Reactor							Oxidation Discharge					Observations
						Condensate		Oil	Bath	Vapour	O2	Condensate			Disch.	Weight	HCl+cup	HCl+cup	ORP	(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	Flow mL/min	Vol. mL	Temp ℃	Temp ℃	Temp ℃	flow mL/min	Flow mL/min	Vol. mL	pH	Set #	cup (tare g	(gross) g	+ discharge g	mV	
9:30	0.00	79.9	9502.7		119.8	0.00	167	140.0	119.5	101.7	365	0.00	108							Feed started @ 09:30
10:30	1.00	79.8	9086.6	6.94	119.9	2.77	333	137.5	120.9	101.1	365	0.00	108		1	49.23	482.45	832.40	671	16:30 Solids looked different from this point, a dark
11:30	2.00	81.2	8680.2	6.77	119.0	1.56	427	137.0	120.1	100.8	365	0.00	108		2	49.89	437.85	653.57	656	brown color, also the cake did not crack and it was
12:30	3.00	79.6	8282.4	6.63	119.7	2.70	589	137.0	121.0	102.8	365	0.00	108		3	49.71	472.29	649.87	651	more "compact".
13:30	4.00	80.6	7702.2	9.67	120.0	2.10	715	136.0	120.9	103.5	365	0.00	108		4	49.89	443.90	663.98	661	In both, "mid" and "end" profile, some orange solids
14:41	5.18	79.7	7415.3	4.04	119.9	2.00	857	136.0	122.4	98.0	365	0.00	108		5	49.71	432.92	659.90	652	appeared in the pre-heater and boiler samples.
15:30	6.00	83.3	7108.0	6.27	119.9	1.92	951	135.0	122.2	97.9	383	0.00	108		6	49.89	436.01	566.33	652	19:33 Shut down.
16:30	7.00	83.7	6727.9	6.34	120.1	2.01	1071	133.0	121.6	98.0	383	0.00	108		7	49.71	447.06	666.10	655	After the test, the impeller was in really bad shape
17:28	7.97	79.4	6370.9	6.16	119.6	1.79	1175	130.0	121.3	99.4	365	0.00	108		8	49.89	417.02	741.40	680	it seems that the cement did not work very well and
18:30	9.00	84.1	5995.4	6.06	119.7	1.88	1292	132.0	120.4	98.1	365	0.00	108		9	49.30	463.99	646.01	661	the shaft, right where it's joined to the impeller was
19:33	10.05	85.8	5449.5	8.67	119.9	1.61	1393	132.0	121.3	96.5	348	0.00	108		10	49.93	438.79	551.70	642	completely corroded.
																				Final solids reactor were kept wet. A sub-sample
																				was dried and submitted for analyses.
Totals/Avg:																				

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 7**

Sampling Data

Sample Name type	Sample Name set #	Sample Composition Diluent	Sample Composition Sample	Diluted Weight pulp, g	Diluted Weight PLS, g	Volume PLS, mL	**pH at room T**	SG g/mL	Wet res, g	Dry res, g	%H2O	Colour PLS	Colour RES	Sample Analysis (Diluted Sample) Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl	Calculated sample concentration Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	% Fe(II)/Fe	Fe/Cu
Pre-heater	start - 09:30	88.00	31.94	119.94	118.0	108	1.09	1.098						48000	38500	2000	130	1400	1100		162	130	32	7	0	5	4	80.2	24.0
Boiler	start - 09:30	80.42	36.81	117.23	115.3	100	1.01	1.152						76000	59800	3100	200	2200	1700		207	163	44	8	1	6	5	78.7	24.5
Reactor	start - 09:30	77.19	29.91	107.10	105.3	92	0.78	1.150						65000	8300	3500	230	2400	1900		199	25	174	11	1	7	6	12.8	18.6
Pre-heater	mid - 14:30	83.25	39.11	122.36	118.7	106	1.07	1.123						62000	48300	2500	160	1800	1400		168	131	37	7	0	5	4	77.9	24.8
Boiler	mid - 14:30	83.48	41.74	125.22	119.2	102	1.05	1.163						80000	64700	3300	220	2400	1900		196	159	38	8	1	6	5	80.9	24.2
Reactor	mid - 14:30	79.69	20.48	100.17	96.5	87	0.80	1.105						42000	4700	2500	160	1700	1400		179	20	159	11	1	7	6	11.2	16.8
Pre-heater	end - 18:45	83.48	32.08	115.56	112.9	102	1.09	1.105						56000	41300	2100	140	1500	1200		178	132	47	7	0	5	4	73.8	26.7
Boiler	end - 18:45	82.65	34.52	117.17	115.1	101	1.08	1.141						72000	56700	2900	190	2100	1600		210	166	45	8	1	6	5	78.8	24.8
Reactor	end - 18:45	86.46	31.79	118.25	111.9	99	0.65	1.135						55000	4600	3300	220	2300	1800		171	14	156	10	1	7	6	8.4	16.7
Disch	1	433.22	349.95	783.17	775.0	623	0.78	1.243						100000	21700	5900	390	4100	2900		178	39	140	11	1	7	5	21.7	16.9
Disch	2	387.96	215.72	603.68	596.6	501	0.59	1.191						82000	20800	4600	310	3200	2200		190	48	142	11	1	7	5	25.4	17.8
Disch	3	422.58	177.58	600.16	586.0	506	0.66	1.158						67000	11900	3900	250	2600	1900		191	34	157	11	1	7	5	17.8	17.2
Disch	4	394.01	220.08	614.09	603.8	504	0.46	1.199						83000	13000	5100	320	3400	2400		190	30	160	12	1	8	5	15.7	16.3
Disch	5	383.21	226.98	610.19	703.0	584	0.53	1.204						81000	16100	4900	320	3400	2500		208	41	167	13	1	9	6	19.9	16.5
Disch	6	386.12	130.32	516.44	510.1	455	0.68	1.121						50000	9600	2600	180	1900	1400		175	34	141	9	1	7	5	19.2	19.2
Disch	7	397.35	219.04	616.39	604.9	506	0.32	1.195						82000	15900	4900	300	3200	2200		189	37	153	11	1	7	5	19.4	16.7
Disch	8	367.13	324.38	691.51	656.3	516	-0.14	1.272						120000	12600	7200	440	5000	3300		191	20	171	11	1	8	5	10.5	16.7
Disch	9	414.69	182.02	596.71	581.2	501	0.49	1.160						66000	8700	3700	240	2500	1800		182	24	158	10	1	7	5	13.2	17.8
Disch	10	388.86	112.91	501.77	490.0	439	0.80	1.115						47000	7300	2600	180	1900	1400		183	28	155	10	1	7	5	15.5	18.1
														Fe %	Fe(II) %	Cu %	Co %	Ni %	Mg %	Cl %								
Disc 0-2									7.6	3.4	55.1		Red	61.7	1.13	0.03	<0.01	0.003	<0.005	4.4								
Disc 3-4									14.6	8.6	41.3		Dk Red	65.9	0.18	0.024	<0.01	0.003	<0.005	3.2								
Disc 5-6									14.0	9.8	29.9		Dk Red	64.9	0.23	0.022	<0.01	0.002	<0.005	2.6								
Disc 7-8									32.2	24.4	24.4		Blk/Red	66.9	0.21	0.018	<0.01	0.003	<0.005	2.5								
Disc 9-10									17.8	13.6	23.8		Blk/Red	70	0.18	0.017	<0.01	0.002	<0.005	2.2								
Reactor Solids									221.8	172.6	22.2		Blk/Red	68.3	0.18	0.017	<0.01	0.002	<0.005	2.1								
Pre-heater	hot sg							1.446																				
Boiler	hot sg							1.565																				
Reactor	hot sg							1.620																				
Reactor	solids																											
Boiler Condensate																												
Reactor Condensate																												

Final Filtration:

Diameter of filtration paper:		mm	Clarity of filtrate:
type of paper (Whatman ##):			Colour of filtrate:
Filtration T:		C	Clarity of wash:
Filtration time:		hours	Colour of wash:
Washing time:		day	Colour of residue:
Volume of wash:		mL	
Cake thickness:		cm	

Other Notes / Observations

Project: 12323-001
Client: Starfield Resources

Test ID: **Test 7** **Re: Balance**

				Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg		
run time:	10.05	hours										
avg feed flow:	6.75	g/min	feed comp:	169	131	39	7	0	5	4	g/kg	
feed:	4072	g	units in feed:	689	532	157	27	2	19	15	g	
product:	2159	g pulp	units in product liquor:	406	72	334	24	2	16	12	g	
	232	g solids	solids comp:	68%			0.018%	0.010%	0.002%	0.005%	%	
			units in solids:	158	0	0	0	0	0	0	g	
Reactor START:	1.2	L	units in reactor - start:	387	49	337	21	1	14	11	g	(from makeup file)
Reactor END	1.9	kg	reactor comp:	171	14	156	10	1	7	6	g/kg	
			units in reactor - end:	331	28	304	20	1	14	11	g	
DELTA Reactor:			(accumulation)	-55	-22	-34	-1	0	0	0	g	
Total IN:				689	532	157	27	2	19	15	g	
Total OUT:				509	51	300	23	2	16	11	g	
Accountability:			(out/IN)	74	10	191	83	85	82	72	%	
Ferrous Oxidized:					481						g	
Ferrous Oxidized:					9						mole	
% Ferrous Oxidised:					**90**						%	
Oxidation rate:					0.80						g/min	
O2 flow	369	mL/min										
	223	L										
	20	psig										
	21	C										
pV=nRT	21.8	mole										

Reaction **12FeCl2 + 3O2 = 2Fe2O3 + 8FeCl3**

1 mole O2

Oxidizes: 4 mole Fe

O2 utilisation: **10** **%**

Appendix B - Oxidation Long Run
Test details

Project: 12323-01
Client: Starfield Resources

Test ID: PP1
Purpose: To operate boiler at 120C. Observe presence of crystals

Impellor: **Pitch blade**

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Boiler RPM:
O2 line pressure: 20 psig

Time		Preheater		Feed		Boiler			Oxidation Reactor								Observations
					target		Condensate		Oil	Bath	Vapour		O2	Condensate			(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Flow mL/min	Temp ℃	Flow g/min	Vol. g	Temp ℃	Temp ℃	Temp ℃	RPM	flow mL/min	Flow g/min	Vol. g	pH	
Apr-27 8:30	0.00	82.5	0.0	0.00	4	119.6	0.00	37	130.0	115.4	99.0	1002	365	0.00	164	4.5	161.77g of seed material added
Apr-27 9:40	1.17	79.7	380.2	5.43	4	119.9	1.49	142	137.0	120.0	101.5	1000	365	0.29	184	4.3	O2 flow irregular pulled tube and cut end ~1min
Apr-27 10:30	2.00	81.5	663.5	5.67	4	119.8	2.07	245	137.0	119.9	101.4	1000	365	0.00	184		
Apr-27 11:30	3.00	82.1	1005.0	5.69	4	119.8	2.10	371	137.0	120.3	101.0	1000	365	0.00	184		
Apr-27 12:30	4.00	80.6	1361.5	5.94	4	119.8	1.78	478	137.0	120.6	100.5	1000	365	0.00	184		
Apr-27 13:30	5.00	82.7	1670.0	5.14	4	119.8	2.09	603	136.0	121.0	100.4	1000	365	0.00	184		
Apr-27 14:30	6.00	83.5	2018.5	5.81	4	120.0	1.98	722	134.0	120.9	99.3	1000	365	0.00	184		
Apr-27 15:30	7.00	79.6	2351.5	5.55	4	119.9	1.82	831	134.0	120.9	99.8	1000	234	0.00	184		unplugged air line @15:30, 16:00
Apr-27 16:30	8.00	82.6	2701.2	5.83	4	119.1	1.54	924	134.0	119.9	99.7	1000	280	0.00	184		16:30 unplugged air line, 16:15 Boiler condensate pump was off 45min, back on.
Apr-27 17:30	9.00	82.6	3017.7	5.28	4	119.7	1.54	1016	134.0	120.1	99.3	1000	366	0.00	184		
Apr-27 18:30	10.00	82.1	3375.5	6.51	4	119.8	1.83	1126	134.0	120.5	98.4	1000	366	0.00	184		17:50 Preheater Weight=6391.0g, 17:55 Temp.= 82.1oC, Preheater Weight=10635.5g
Apr-27 19:30	11.00	82.7	3699.7	5.40	4	119.8	1.79	1233	134.0	120.6	97.8	1000	366	0.00	184		18:45 unplugged air, 19:10
Apr-27 20:30	12.00	79.4	4087.9	6.47	4	119.7	1.91	1348	134.0	121.1	98.8	750	366	0.00	184		Condition 2 started -> reactor RPM d750
Apr-27 21:30	13.00	82.4	4429.5	5.69	4	120.1	1.62	1445	134.0	120.7	99.3	750	366	0.00	184		22:17 unplugged air
Apr-27 22:30	14.00	79.5	4824.4	6.58	4	119.8	1.77	1552	134.0	120.5	99.1	750	366	0.00	184		
Apr-27 23:30	15.00	82.4	5055.3	3.85	4	119.7	1.87	1664	134.1	120.9	100.2	750	348	0.00	184		O2 sparger clogged. It was unplugged immediately
Apr-28 0:30	16.00	81.8	5402.4	5.79	4	119.5	1.77	1770	134.0	121.6	99.7	750	331	0.00	184		
Apr-28 1:31	17.02	82.1	5721.4	5.23	4	120.1	1.89	1885	133.0	121.0	100.0	750	348	0.00	184		O2 sparger clogged. It was unplugged immediately
Apr-28 2:30	18.00	80.5	6036.7	5.34	4	119.9	1.61	1980	133.0	120.7	99.0	750	348	0.00	184		
Apr-28 3:30	19.00	82.7	6397.3	6.01	4	119.8	1.80	2088	133.0	120.5	99.2	750	331	0.00	184		O2 sparger clogged. It was unplugged immediately
Apr-28 4:32	20.03	84.3	6693.3	4.77	4	118.2	1.45	2178	133.0	120.4	100.3	750	331	0.00	184		
Apr-28 5:30	21.00	81.4	7011.7	5.49	4	120.0	1.57	2269	133.0	119.7	99.6	750	348	0.00	184		
Apr-28 6:33	22.05	83.1	7270.8	4.11	4	119.1	1.50	2363	134.0	121.0	99.9	750	348	0.00	184		
Apr-28 7:30	23.00	83.5	7663.3	7.55	4	120.0	1.96	2475	134.0	121.1	99.3	750	365	0.00	184		feed swap; very full
Apr-28 8:30	24.00	81.3	7972.4	6.18	4	120.2	1.93	2590	134.0	121.3	101.2	1250	365	0.00	184		O2 sparger blocked ~10 min tantalum tume was caked as well
Apr-28 9:30	25.00	80.2	8254.9	4.71	4	120.0	1.29	2668	134.0	121.0	101.2	1250	365	0.00	184		
Apr-28 10:30	26.00	80.0	8575.6	5.35	4	119.9	2.34	2808	134.0	121.9	100.5	1250	365	0.00	184		
Apr-28 11:30	27.00	81.4	8800.4	3.75	4	119.3	0.96	2866	134.0	120.4	100.6	1250	365	0.00	184		
Apr-28 12:30	28.00	79.5	9235.8	7.26	4	119.7	1.27	2942	134.0	119.8	100.3	1250	365	0.00	184		
Apr-28 13:30	29.00	79.5	9575.3	5.66	4	119.5	1.27	3018	134.0	119.3	99.7	1250	365	0.00	184		
Apr-28 14:30	30.00	82.5	9793.9	3.64	4	119.4	2.26	3154	134.0	121.4	99.6	1250	365	0.00	184		

Project: 12323-01
Client: Starfield Resources

Test ID: PP1
Purpose: To operate boiler at 120C. Observe presence of crystals

Impellor: **Pitch blade**

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors

Boiler RPM:
O2 line pressure: 20 psig

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Time		Preheater		Feed		Boiler			Oxidation Reactor								Observations
					target		Condensate		Oil	Bath	Vapour		O2	Condensate			(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Flow mL/min	Temp ℃	Flow g/min	Vol. g	Temp ℃	Temp ℃	Temp ℃	RPM	flow mL/min	Flow g/min	Vol. g	pH	
Apr-28 15:30	31.00	79.4	10161.4	6.13	4	119.6	1.72	3257	134.0	121.2	100.2	1240	365	0.00	184		discharge samples are slower to filter than last afternoon shift (all shift)
Apr-28 16:30	32.00	80.3	10560.4	6.65	4	119.8	1.83	3367	134.1	121.1	100.4	1249	366	0.00	184		
Apr-28 17:30	33.00	82.4	10844.4	4.73	4	119.9	1.75	3472	134.0	121.2	100.8	1249	366	0.00	184		bubbles in residue for discharge sample
Apr-28 18:30	34.00	83.5	11191.4	5.78	4	120.0	1.80	3580	133.8	121.1	101.0	1252	366	0.00	184		bubbles in residue for discharge sample
Apr-28 19:30	35.00	81.5	11365.2	3.16	4	119.9	1.76	3685	133.5	121.2	100.9	1252	365	0.00	184		feed swap; very full
Apr-28 20:30	36.00	81.6	11679.7	5.24	4	120.0	0.69	3727	133.5	121.0	99.9	1250	365	0.00	184		
Apr-28 20:50	36.33	82.4	11857.9	8.91	6	119.9	3.67	3800	133.5	120.8	97.9	1247	547	0.00	184		condition 4 start
Apr-28 21:30	37.00	82.3	12213.7	8.90	6	119.8	2.45	3898	133.5	120.1	98.0	1255	547	0.00	184		21:00 unplugged air
Apr-28 22:30	38.00	79.9	12695.7	8.03	6	119.6	2.37	4041	133.5	120.0	98.2	1254	547	0.00	184		
Apr-28 23:30	39.00	81.6	13144.4	7.48	6	119.7	2.39	4184	135.0	119.0	98.8	1250	547	0.00	184		
Apr-29 0:24	39.90	82.9	13496.2	6.51	6	118.2	1.20	4249	135.0	118.2	99.2	1250	436	0.00	184		Air line clogged
Apr-29 1:27	40.95	80.3	13978.7	10.05	6	119.7	2.93	4433	140.0	119.2	100.3	1250	436	0.00	184		Air line clogged. Feed was stopped around 15 min because it took a while to
Apr-29 2:30	42.00	82.8	14571.5	9.41	6	119.9	2.65	4600	140.0	120.0	99.0	1250	564	0.00	184		unplug the air line.
Apr-29 3:30	43.00	84.6	15020.1	7.48	6	119.7	2.83	4770	140.0	120.9	99.2	1250	547	0.00	184		
Apr-29 4:30	44.00	79.4	15472.7	7.54	6	119.8	2.98	4948	140.0	121.0	99.0	1250	547	0.00	184		
Apr-29 5:30	45.00	84.1	15828.9	6.48	6	120.0	2.72	5111	136.0	121.7	98.6	1250	564	0.00	184		feed swap; very full
Apr-29 6:30	46.00	80.4	16258.3	7.16	6	119.8	2.87	5283	136.0	121.0	98.7	1250	547	0.00	184		
Apr-29 7:30	47.00	80.1	16676.2	6.97	6	119.7	2.50	5433	136.0	120.5	98.8	1250	547	0.00	184		
Apr-29 8:30	48.00	80.0	17200.0	8.73	6	119.6	2.73	5597	136.0	120.2	98.8	1259	547	0.00	184		
Apr-29 9:30	49.00	82.9	17769.9	9.50	8	119.2	2.21	5730	136.0	119.6	98.5	1250	727	0.00	184		Feed rate up to 8 mL/min, oxygen up to 630 mL/min
Apr-29 10:30	50.00	79.5	18498.2	12.14	8	119.7	2.21	5862	137.0	117.0	98.0	1250	727	0.00	184		
Apr-29 11:30	51.00	80.8	19165.2	16.68	8	119.6	2.80	6030	140.0	118.8	98.7	1250	710	0.00	184		Circuit shutdown for 20 minutes to clear plugged oxygen line
Apr-29 12:45	52.25	79.3	19727.0	7.49	8	119.4	3.15	6267	140.0	114.8	98.6	1250	727	0.00	184		
Apr-29 13:40	53.17	83.9	20347.2	12.40	8	119.7	3.53	6461	140.0	115.7	99.1	1250	727	0.00	184		feed swap; very full
Apr-29 14:40	54.17	80.6	20945.0	9.96	8	117.9	2.61	6618	140.0	118.3	101.3	1250	727	0.00	184		
Apr-29 15:30	55.00	82.6	21593.4	12.97	8	119.0	3.08	6772	143.5	117.6	99.5	1245	727	0.00	184		
Apr-29 16:35	56.08	79.4	22322.2	11.21	8	119.4	3.99	7032	145.0	117.9	100.1	1253	727	0.00	184		
Apr-29 17:38	57.13	78.0	22601.2	4.43	8	111.3	1.98	7157	132.5	116.0	93.6	1250	727	0.00	184		
Apr-29 18:31	58.02	79.4	23161.3	10.57	8	119.5	2.84	7307	145.0	120.6	100.0	1250	727	0.00	184		PO2 info; reactor condesate accum. test = 2.5 mL /55 s, 5 mL / 122 s
Apr-29 19:30	59.00	81.8	23817.2	11.12	8	119.9	3.81	7532	143.1	120.9	99.8	1250	727	0.75	228		20:00 unplugged oxygen line
Apr-29 20:30	60.00	80.1	24362.4	9.91	8	116.7	0.92	7587	143.0	120.1	99.2	1250	727	0.00	228		swapped feed, cleared reactor discharge port (plugged almost solid since last sample)

Project: 12323-01
Client: Starfield Resources

Test ID: PP1
Purpose: To operate boiler at 120C. Observe presence of crystals

Impellor: **Pitch blade**

Health Safety:

Main Hazards: hot oil:
concentrated brine liquors

Boiler RPM:
O2 line pressure: 20 psig

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Time		Preheater		Feed		Boiler			Oxidation Reactor								Observations
					target		Condensate		Oil	Bath	Vapour		O2	Condensate			(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Flow mL/min	Temp ℃	Flow g/min	Vol. g	Temp ℃	Temp ℃	Temp ℃	RPM	flow mL/min	Flow g/min	Vol. g	pH	
Apr-29 21:30	61.00	82.3	25058.4	11.60	8	118.1	3.96	7825	143.4	119.0	98.8	1250	727	0.00	228		condition 6 start; reactor condensate pump on
Apr-29 22:30	62.00	80.3	25713.4	10.92	8	119.9	3.56	8038	132.1	120.2	98.3	1250	727	2.24	363	4.3	
Apr-29 23:30	63.00	83.4	26449.5	12.27	8	119.8	2.89	8212	131.0	119.4	97.8	1250	745	1.02	424	2.8	
Apr-30 0:30	64.00	80.0	27109.7	11.00	8	119.9	2.84	8382	133.0	119.5	98.3	1250	727	0.97	482	2.6	
Apr-30 1:30	65.00	81.8	27762.0	10.87	8	120.1	2.92	8557	133.0	120.2	98.0	1250	745	1.10	548	2.7	
Apr-30 2:30	66.00	84.4	28373.5	10.19	8	119.9	2.75	8722	133.0	120.6	97.9	1250	727	1.03	610	2.7	
Apr-30 3:30	67.00	81.2	28999.7	11.39	8	119.9	2.64	8881	133.0	120.9	98.8	1250	727	0.90	664	2.7	feed swap; very full
Apr-30 4:30	68.00	82.7	29671.5	11.20	8	119.8	2.80	9049	131.0	119.7	100.4	1250	745	0.89	718	2.6	
Apr-30 5:26	68.93	80.1	30302.1	11.26	8	119.8	2.86	9209	133.0	119.5	99.0	1250	655	0.95	771	2.6	
Apr-30 6:30	70.00	81.9	30995.6	10.84	8	119.6	2.68	9381	135.0	119.0	101.2	1250	727	0.78	821	3.0	
Apr-30 7:30	71.00	83.0	31608.3	10.21	8	119.6	2.89	9554	135.0	121.1	100.4	1280	745	1.27	897	2.6	
Apr-30 8:30	72.00	84.1	32236.5	11.42	8	119.6	2.84	9725	1354.0	121.6	100.4	1277	745	1.28	974	2.6	feed swap; very full
Apr-30 9:30	73.00	82.0	32824.5	9.80	8	120.0	2.91	9899	135.0	122.1	100.7	1280	745	1.24	1048	2.8	
Apr-30 10:30	74.00	80.0	33405.5	9.68	8	119.6	0.96	9957	134.0	120.1	101.2	1268	745	0.62	1085	2.5	
Apr-30 11:30	75.00	82.3	34017.4	10.20	8	118.9	3.56	10170	134.0	120.0	100.5	1280	745	1.62	1183	2.6	
Apr-30 12:30	76.00	83.1	34643.0	10.43	8	118.8	2.96	10348	134.0	123.9	99.0	1280	745	1.24	1257	2.7	feed pump clogged
Apr-30 13:30	77.00	80.8	35212.4	9.49	8	119.7	2.41	10492	134.0	122.8	100.0	1280	745	0.73	1301	2.6	
Apr-30 14:30	78.00	83.5	35555.3	5.72	8	119.4	2.08	10617	134.0	123.1	97.5	1280	745	0.71	1343	2.6	SHUTDWON
Totals/Avg:																	

Project: 12323-01
Client: Starfield Resources

Test ID: PP1

Sampling Data

Sample Name			Weight	HCl+cup	HCl+cup	pulp	Sample Composition		Diluted Weight		Volume			Wet	Dry		Colour		Sample Analysis (Diluted Sample)						Calculated sample concentration							
Time	Elapsed time hr	set #	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	pH at room T	SG g/mL	res g	res g	%H2O	PLS PLS	RES RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	% Fe(II)/Fe	Fe/Cu
Apr-27 9:40	1.17	1	49.5	427.8	632.9	688	378.34	205.14	583.48	558.9	484	0.70	1.154				brown	dk brown	64000	6200	3600	250	2500	2000	151	15	136	8	1	6	5	9.7	17.8
Apr-27 10:30	2.00	2	50.0	446.1	716.2	675	396.06	270.09	666.15	631.4	517	0.43	1.222				brown	dk brown	95000	16600	5200	390	3700	2900	182	32	150	10	1	7	6	17.5	18.3
Apr-27 11:30	3.00	3	49.5	438.7	611.4	638	389.19	172.75	561.94	586.4	490	0.57	1.196				brown	dk brown	86000	19900	4700	360	3300	2600	244	56	188	13	1	9	7	23.1	18.3
Apr-27 12:30	4.00	4	50.0	441.8	630.5	646	391.78	188.68	580.46	558.0	475	0.64	1.174				brown	dk brown	61000	18300	3300	260	2300	1900	154	46	108	8	1	6	5	30.0	18.5
Apr-27 13:30	5.00	5	49.5	424.8	610.7	643	375.30	185.91	561.21	538.7	457	0.68	1.180				brown	dk brown	77000	18800	4100	330	2900	2300	189	46	143	10	1	7	6	24.4	18.8
Apr-27 14:30	6.00	6	49.5	433.9	629.1	649	384.44	195.15	579.59	558.0	472	0.66	1.183				brown	dk brown	78000	18500	4100	330	2900	2300	189	45	144	10	1	7	6	23.7	19.0
Apr-27 15:30	7.00	7	49.5	433.9	663.6	654	384.44	229.66	614.10	584.4	484	0.57	1.208				brown	dk brown	88000	20700	4700	380	3400	2600	185	44	142	10	1	7	5	23.5	18.7
Apr-27 16:30	8.00	8	50.0	449.1	733.2	657	399.08	284.15	683.23	651.4	529	0.46	1.231				brown	dk brown	98000	24400	5500	410	3700	2900	183	45	137	10	1	7	5	24.9	17.8
Apr-27 17:30	9.00	9	49.5	476.2	650.0	644	426.77	173.72	600.49	580.7	506	0.81	1.148				brown	dk brown	62000	15700	3300	260	2300	1900	181	46	135	10	1	7	6	25.3	18.8
Apr-27 18:30	10.00	10	50.0	451.8	638.7	649	401.75	186.94	588.69	565.4	484	0.69	1.167				brown	dk brown	68000	15900	3600	290	2600	2000	176	41	135	9	1	7	5	23.4	18.9
Apr-27 19:30	11.00	11	49.5	426.4	637.6	656	376.94	211.21	588.15	565.4	475	0.59	1.191				brown	dk brown	81000	16800	4300	340	3100	2400	182	38	144	10	1	7	5	20.7	18.8
Apr-27 20:30	12.00	12	50.0	452.6	682.3	662	402.61	229.65	632.26	599.2	500	0.53	1.198	149.7	121.9	19%	brown	dk brown	73000	15500	4000	320	2900	2300	159	34	125	9	1	6	5	21.2	18.3
Apr-27 21:30	13.00	13	49.5	440.4	470.1	626	390.89	29.76	420.65	414.4	402	1.33	1.031				yellow	black	11700	3000	690	48	480	390	158	41	117	9	1	6	5	25.6	17.0
Apr-27 22:30	14.00	14	50.0	452.2	661.0	653	402.20	208.77	610.97	583.5	493	0.50	1.184				brown	dk brown	73600	17000	4100	310	2900	2300	174	40	134	10	1	7	5	23.1	18.0
Apr-27 23:30	15.00	15	49.5	455.3	661.0	654	405.79	205.73	611.52	585.1	495	0.69	1.182				brown	dk brown	74800	18300	4200	300	3000	2300	180	44	136	10	1	7	6	24.5	17.8
Apr-28 0:30	16.00	16	50.0	456.3	730.4	648	406.33	274.06	680.39	649.9	534	0.53	1.218				brown	dk brown	88200	29000	5000	360	3600	2700	172	56	115	10	1	7	5	32.9	17.6
Apr-28 1:31	17.02	17	49.5	466.0	636.9	642	416.45	170.94	587.39	574.8	498	0.74	1.155				brown	dk brown	60400	15100	3400	230	2500	2000	176	44	132	10	1	7	6	25.0	17.8
Apr-28 2:30	18.00	18	50.0	443.1	659.5	637	393.05	216.37	609.42	581.4	491	0.61	1.183				brown	dk brown	77500	28400	4000	320	2900	2200	176	65	112	9	1	7	5	36.6	19.4
Apr-28 3:30	19.00	19	49.5	468.3	678.2	649	418.83	209.84	628.67	607.5	516	0.71	1.177				brown	dk brown	71300	21800	4000	280	2800	2200	175	54	122	10	1	7	5	30.6	17.8
Apr-28 4:32	20.03	20	50.0	466.6	830.0	653	416.61	363.38	779.99	750.5	593	0.45	1.265				brown	dk brown	108000	35600	6200	410	4300	3300	176	58	118	10	1	7	5	33.0	17.4
Apr-28 5:30	21.00	21	49.5	452.8	604.5	635	403.28	151.75	555.03	542.0	478	0.87	1.133				brown	dk brown	55700	18800	3000	210	2100	1700	176	59	116	9	1	7	5	33.8	18.6
Apr-28 6:33	22.05	22	50.0	457.6	610.7	629	407.55	153.13	560.68	551.3	486	0.74	1.135				brown	dk brown	54600	21100	3000	210	2100	1700	173	67	106	10	1	7	5	38.6	18.2
Apr-28 7:30	23.00	23	49.5	471.0	562.9	630.7	421.50	91.87	513.37	498.9	458	1.23	1.089				brown	dk brown	35700	12700	2100	140	1400	1100	178	63	115	10	1	7	5	35.6	17.0
Apr-28 8:30	24.00	24	50.0	432.3	638.0	637	382.28	205.72	588.00	565.5	478	0.53	1.182	177.8	147.1	17%	brown	dk brown	74000	27700	4100	270	2900	2200	172	64	108	10	1	7	5	37.4	18.0
Apr-28 9:30	25.00	25	49.5	464.3	614.6	591.2	414.79	150.39	565.18	558.5	488	1.22	1.144				lt brown	dk brown	58400	15700	3200	200	2300	1900	190	51	139	10	1	7	6	26.9	18.3
Apr-28 10:30	26.00	26	50.0	469.3	653.3	580.3	419.31	183.99	603.30	553.3	463	0.89	1.196				brown	dk brown	79200	23900	4400	320	3200	2400	199	60	139	11	1	8	6	30.2	18.0
Apr-28 11:30	27.00	27	49.5	490.9	779.3	598	441.45	288.40	729.85	694.6	572	0.72	1.215				brown	dk brown	91200	18600	4900	340	3600	2700	181	37	144	10	1	7	5	20.4	18.6
Apr-28 12:30	28.00	28	50.0	458.7	775.0	585	408.67	316.30	724.97	694.2	566	0.69	1.227				brown	dk brown	92000	26400	5200	350	3800	2900	165	47	117	9	1	7	5	28.7	17.7
Apr-28 13:30	29.00	29	49.5	463.5	693.2	590.8	414.01	229.71	643.72	621.1	524	0.93	1.185				lt brown	dk brown	72600	15200	4300	290	3100	2300	166	35	131	10	1	7	5	20.9	16.9
Apr-28 14:30	30.00	30	50.0	464.3	581.1	522	414.34	116.77	531.11	526.1	486	1.16	1.083				green	dk brown	37100	24000	1700	110	1200	940	154	100	54	7	0	5	4	64.7	21.8
Apr-28 15:30	31.00	31	49.5	446.7	626.7	601	397.19	180.07	577.26	554.5	479	0.85	1.158				brown	dk brown	61300	9300	3600	250	2600	2000	163	25	138	10	1	7	5	15.2	17.0
Apr-28 16:30	32.00	32	50.0	468.5	687.7	607	418.47	219.23	637.70	601.0	508	0.74	1.183				brown	dk brown	71500	9600	4200	290	3000	2300	166	22	143	10	1	7	5	13.4	17.0
Apr-28 17:30	33.00	33	49.5	436.6	640.3	610	387.13	203.66	590.79	559.0	475	0.69	1.177				brown	dk brown	67700	8900	4200	280	3000	2300	158	21	137	10	1	7	5	13.1	16.1
Apr-28 18:30	34.00	34	50.0	451.4	679.5	606	401.43	228.05	629.48	597.2	501	0.73	1.191				brown	dk brown	74900	10700	4500	300	3200	2500	165	24	141	10	1	7	5	14.3	16.6
Apr-28 19:30	35.00	35	49.5	451.2	635.8	607	401.76	184.55	586.31	555.6	477	0.79	1.166				brown	dk brown / red to	63900	8200	3900	260	2800	2200	165	21	144	10	1	7	6	12.8	16.4
Apr-28 20:30	36.00	36	50.0	452.2	600.8	599	402.15	148.69	550.84	583.0	490	0.72	1.191	216.1	190.3	12%	brown	dk brown	73800	9200	4300	300	3100	2400	243	30	213	14	1	10	8	12.5	17.2
Apr-28 21:30	37.00	37	49.5	453.7	705.8	606	404.25	252.08	656.33	617.5	511	0.59	1.208				brown	dk brown	92100	13200	5100	330	3500	2700	187	27	160	10	1	7	5	14.3	18.1
Apr-28 22:30	38.00	38	50.0	426.9	759.1	589	376.85	332.26	709.11	675.7	542	0.46	1.247				brown	dk brown	111000	27700	6200	390	4100	3100	181	45	136	10	1	7	5	25.0	17.9
Apr-28 23:30	39.00	39	49.5	458.0	842.8	606	408.55	384.79	793.34	745.5	586	0.38	1.273				brown	dk brown	124000	20300	7300	440	4700	3500	189	31	158	11	1	7	5	16.4	17.0
Apr-29 0:24	39.90	40	50.0	463.1	955.8	610	413.10	492.70	905.80	837.2	640	0.34	1.308				brown	dk brown	141000	27400	7400	490	5200	3900	183	36	148	10	1	7	5	19.4	19.1
Apr-29 1:27	40.95	41	49.5	459.4	730.7	578	409.86	271.33	681.19	653.0	545	0.65	1.199				brown	dk brown	89400	28100	4700	300	3200	2500	179	56	123	9	1	6	5	31.4	19.0
Apr-29 2:30	42.00	42	50.1	454.0	593.1	563	403.87	139.14	543.01	524.5	466	1.00	1.125				brown	dk brown	53500	16500	2800	190	2000	1600	179	55	124	9	1	7	5	30.8	19.1
Apr-29 3:30	43.00	43	49.6	464.0	788.6	581	414.43	324.52	738.95	693.0	558	0.49	1.242				brown	dk brown	112000	34800	5800	380	4000	3100	193	60	133	10	1	7	5	31.1	19.3
Apr-29 4:30	44.00	44	50.1	454.4	770.1	581	404.26	315.66	719.92	678.4	544	0.51	1.246				brown	dk brown	112000	34000	5800	380	4000	3100	193	59	135	10	1	7	5	30.4	19.3
Apr-29 5:30	45.00	45	49.5	460.5	668.6	578	410.97	208.13	619.10	590.3	500	0.71	1.181				brown	dk brown	74000	22800	3800	260	2700	2200	178	55	123	9	1	6	5	30.8	19.5
Apr-29 6:30	46.00	46	50.2	462.3	808.4	591	412.13	346.10	758.23	706.3	561	0.48	1.259				brown	dk brown	105000	28100	5900	400	4200	3300	170	46	125	10	1	7	5	26.8	17.8
Apr-29 7:30	47.00	47	49.6	479.8	834.8	593	430.20	355.00	785.20	733.3	583	0.43	1.258				brown	dk brown	105000	27800	6000	410	4300	3400	172	46	127	10	1	7	6	26.5	17.5
Apr-29 8:30	48.00	48	50.2	457.0	730.7	586	406.78	273.79	680.57	698.8	557	0.33	1.254	359.3	301.8	16%	brown	dk brown	107000	27600	5700	380	4100	3200	218	56	162	12	1	8	7	25.8	18.8
Apr-29 9:30	49.00	49	49.6	460.9	816.4	593	411.34	355.46	766.80	718.6	570	0.65	1.261				brown	dk brown	109000	29000	5900	400	4200	3300	175	46	128	9	1	7	5	26.6	18.5

Project: 12323-01
Client: Starfield Resources

Test ID: PP1

Sampling Data

Sample Name			Weight	HCl+cup	HCl+cup	pulp	Sample Composition		Diluted Weight		Volume							Colour		Sample Analysis (Diluted Sample)						Calculated sample concentration								
Time	Elapsed time hr	set #	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	pH at room T	SG g/mL	Wet res g	Dry res g	%H2O		PLS PLS	RES RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	% Fe(II)/Fe	Fe/Cu
Apr-29 10:30	50.00	50	50.2	459.2	839.7	594	408.99	380.54	789.53	740.7	584	0.56	1.269					brown	dk brown	111000	32900	6200	420	4400	3500	170	50	120	10	1	7	5	29.6	17.9
Apr-29 11:30	51.00	51	49.6	452.5	900.4	592	402.86	447.94	850.80	794.9	610	0.28	1.304					brown	dk brown	126000	38700	7000	470	5000	3900	171	53	119	10	1	7	5	30.7	18.0
Apr-29 12:45	52.25	52	50.2	451.3	1114.5	592	401.10	663.23	1064.33	1001.6	755	0.36	1.326					brown	dk brown	145000	57600	7600	520	5400	4300	165	66	100	9	1	6	5	39.7	19.1
Apr-29 13:40	53.17	53	49.6	475.1	882.9	582	425.45	407.85	833.30	793.0	631	0.55	1.257					brown	dk brown	113000	43700	5700	390	4100	3200	175	68	107	9	1	6	5	38.7	19.8
Apr-29 14:40	54.17	54	50.2	426.4	838.2	587	376.22	411.81	788.03	744.7	581	0.59	1.282					brown	dk brown	123000	48600	6500	430	4600	3600	173	69	105	9	1	6	5	39.5	18.9
Apr-29 15:30	55.00	55	49.6	447.7	870.5	565	398.07	422.83	820.90	777.6	604	0.36	1.287					brown	dk brown	122000	51700	6500	430	4600	3600	174	74	100	9	1	7	5	42.4	18.8
Apr-29 16:35	56.08	56	50.0	457.8	928.2	582	407.76	470.44	878.20	825.4	632	0.44	1.305					brown	dk brown	132000	50800	6900	470	4900	3800	177	68	109	9	1	7	5	38.5	19.1
Apr-29 17:38	57.13	57	49.6	442.5	567.1	567	392.94	124.54	517.48	506.3	452	1.16	1.119					brown	red-brown	47400	17200	2700	180	1900	1500	172	62	110	10	1	7	5	36.3	17.6
Apr-29 18:31	58.02	58	50.0	463.5	780.2	593	413.54	316.70	730.24	690.4	556	0.75	1.242					brown	s dk brown slow	99200	32600	6200	380	4200	3300	174	57	117	11	1	7	6	32.9	16.0
Apr-29 19:30	59.00	59	49.6	456.9	817.2	594	407.28	360.32	767.60	741.8	585	0.64	1.269					brown	dk brown	112000	35800	6600	370	4200	3500	182	58	124	11	1	7	6	32.0	17.0
Apr-29 20:30	60.00	60	50.0	439.8	996.5	593	389.80	556.70	946.50	887.5	652	0.19	1.362	404.6	335.9	17%		brown	dk brown	154000	57300	8900	560	6200	4500	180	67	113	10	1	7	5	37.2	17.3
Apr-29 21:30	61.00	61	49.5	428.6	867.4	589	379.14	438.80	817.94	764.6	585	0.33	1.308					brown	dk brown	127000	46500	7500	480	5100	4100	169	62	107	10	1	7	5	36.6	16.9
Apr-29 22:30	62.00	62	50.0	458.9	558.7	585	408.86	99.80	508.66	499.4	454	1.30	1.100					brown	dk brown/red to	38900	11400	2200	150	1600	1200	177	52	125	10	1	7	5	29.3	17.7
Apr-29 23:30	63.00	63	49.5	450.4	880.9	602	400.91	430.53	831.44	786.6	601	0.22	1.309					brown	dk brown	134000	33100	7900	480	5500	4300	187	46	141	11	1	8	6	24.7	17.0
Apr-30 0:30	64.00	64	50.0	449.9	913.1	604	399.90	463.20	863.10	817.3	619	0.15	1.321					brown	dk brown	136000	32300	8100	500	5600	4100	182	43	139	11	1	7	5	23.8	16.8
Apr-30 1:30	65.00	65	49.6	455.2	833.3	604	405.57	378.10	783.67	746.6	584	0.22	1.278					brown	dk brown	118000	27600	6600	430	4700	3700	182	43	140	10	1	7	6	23.4	17.9
Apr-30 2:30	66.00	66	50.1	447.1	849.9	606	396.98	402.83	799.81	758.8	586	0.16	1.294					brown	dk brown	126000	28300	7100	470	5000	3900	183	41	142	10	1	7	6	22.5	17.7
Apr-30 3:30	67.00	67	49.5	467.1	829.6	604	417.61	362.46	780.07	741.7	587	0.24	1.264					brown	dk brown	111000	27900	6100	400	4400	3400	180	45	135	10	1	7	6	25.1	18.2
Apr-30 4:30	68.00	68	50.1	459.4	928.0	607	409.34	468.58	877.92	825.4	627	0.05	1.316					brown	dk brown	154000	28100	7000	460	5500	3800	206	38	169	9	1	7	5	18.2	22.0
Apr-30 5:26	68.93	69	49.7	483.4	935.4	609	433.72	451.97	885.69	825.8	639	0.18	1.293					brown	dk brown	123300	24000	7700	470	5300	3900	174	34	140	11	1	7	6	19.5	16.0
Apr-30 6:30	70.00	70	50.0	459.2	1004.2	601	409.20	544.96	954.16	884.2	667	-0.06	1.325					brown	dk brown	142600	33200	8400	550	5900	4600	175	41	134	10	1	7	6	23.3	17.0
Apr-30 7:30	71.00	71	49.6	459.9	779.3	604	410.37	319.35	729.72	685.5	548	-0.20	1.251					brown	dk brown	102200	24900	6200	410	4300	3400	175	43	133	11	1	7	6	24.4	16.5
Apr-30 8:30	72.00	72	50.0	466.9	853.7	613	416.94	386.76	803.70	752.5	586	-0.20	1.283					brown	dk brown	115800	23900	7600	480	5300	4000	176	36	139	12	1	8	6	20.6	15.2
Apr-30 9:30	73.00	73	49.5	456.0	869.9	610	406.55	413.89	820.44	764.8	589	-0.35	1.298					brown	dk brown	129000	26300	7700	500	5300	4100	184	37	146	11	1	8	6	20.4	16.8
Apr-30 10:30	74.00	74	50.0	435.1	1101.7	621	385.13	666.57	1051.70	961.3	695	-0.54	1.384					brown	dk brown	176900	43900	9600	640	6700	5200	184	46	139	10	1	7	5	24.8	18.4
Apr-30 11:30	75.00	75	49.5	462.6	782.4	598	413.16	319.78	732.94	690.8	559	-0.10	1.236					brown	dk brown	100800	27200	5900	380	4100	3100	176	48	129	10	1	7	5	27.0	17.1
Apr-30 12:30	76.00	76	50.0	446.8	694.2	593	396.76	247.40	644.16	613.5	508	0.25	1.208					brown	dk brown	82200	22900	5200	330	3600	2800	169	47	122	11	1	7	6	27.9	15.8
Apr-30 13:30	77.00	77	49.5	452.4	788.1	619	402.91	335.71	738.62	680.4	540	-0.27	1.261					brown	dk brown	103400	18100	6800	420	4700	3600	166	29	137	11	1	8	6	17.5	15.2
Apr-30 14:30	78.00	78	50.0	455.3	698.4	603	405.33	243.05	648.38	613.2	506	0.37	1.213	560.7	468.4	16%		brown	dk brown	81300	16400	5200	330	3600	2800	169	34	135	11	1	7	6	20.2	15.6
Solids assays																				%Fe	%Cu	%Co	%Ni	%Mg	%Cl							
Cond 1	(1-12)													149.7	121.9	19%				67.1	0.015	<0.01	0.012	<0.005	2.0									
Cond 2	(13-24)													177.8	147.1	17%				66.6	0.015	<0.01	0.014	<0.005	2.0									
Cond 3	(25-36)													216.1	190.3	12%				67.1	0.020	<0.01	0.010	<0.005	2.2									
Cond 4	(37-48)													359.3	301.8	16%				66.3	0.020	<0.01	0.016	0.006	2.6									
Cond 5	(49-60)													404.6	335.9	17%				67.0	0.016	<0.01	0.011	<0.005	2.7									
Cond 6	(60-78)													560.7	468.4	16%				66.6	0.022	<0.01	0.014	<0.005	2.6									
Final solids														268.2	230.9	14%				66.0	0.160	<0.01	0.038	0.017	3.5									
Reactor OF solids																				65.5	0.030	<0.01	0.021	0.014	2.8									

Project: 12323-001
Client: Starfield Resources

Test ID: PP1

Sampling Data

Time	Sample Name		Weight	HCl+cup	HCl+cup	Hot	Sample Composition		Diluted Weight		Volume						Colour		Sample Analysis (Diluted Sample)						Calculated sample concentration								
	Name	set #	cup (tare)	(gross)	+ discharge	SG	Diluent	Sample	pulp	PLS	PLS	pH at	SG	Wet res	Dry res	%H2O	PLS	RES	Fe	Fe(II)	Cu	Co	Ni	Mg	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	%	
			g	g	g	g/mL	g	g	g	g	mL	room T	g/mL	g	g		PLS	RES	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	g/kg	g/kg	g/kg	g/kg	g/kg	g/kg	g/kg	Fe(II)/Fe	Fe/Cu
Apr-27 8:30	Feed	Cond 1	18.1	102.2	137.4	1.480	84.08	35.29	119.37	117.5	106	1.08	1.112			0.0	green		48300	44000	2200	150	1600	1200	145	132	13	7	0	5	4	91.1	22.0
Apr-27 8:30	Boiler	Cond 1	18.1	102.3	140.1	1.590	84.20	37.79	121.99	120.1	104	1.09	1.158			0.0	green		67600	60900	3200	320	2300	1800	185	167	18	9	1	6	5	90.1	21.1
Apr-27 8:30	Reactor	Cond 1	18.1	103.4	135.9	1.640	85.26	32.54	117.80	112.7	99	0.78	1.133			0.0	brown		49900	3700	3100	210	2200	1700	153	11	141	9	1	7	5	7.4	16.1
Apr-27 20:30	Feed	Cond 2	18.9	95.1	122.7	1.400	76.13	27.60	103.73	101.9	93	1.19	1.101			0.0	green		46300	39300	2000	140	1400	1100	155	132	23	7	0	5	4	84.9	23.2
Apr-27 20:30	Boiler	Cond 2	19.0	101.3	131.2	1.550	82.37	29.90	112.27	110.2	98	1.22	1.126			0.0	green		57500	49300	2300	170	1700	1300	188	161	27	8	1	6	4	85.7	25.0
Apr-27 20:30	Reactor	Cond 2	19.1	102.7	135.9	1.620	83.55	33.20	116.75	111.0	96	0.74	1.151			0.0	brown	brown	58000	10900	3400	260	2500	1900	168	32	137	10	1	7	6	18.8	17.1
Apr-28 8:30	Feed	Cond 3	18.1	103.7	136.0	1.460	85.63	32.29	117.92	115.6	105	1.24	1.102			0.0	green		43000	39600	2000	130	1400	1100	140	129	11	6	0	5	4	92.1	21.5
Apr-28 8:30	Boiler	Cond 3	18.2	102.5	139.7	1.570	84.35	37.18	121.53	119.9	105	1.17	1.146			0.0	green		68400	57700	2900	200	2100	1600	193	162	30	8	1	6	5	84.4	23.6
Apr-28 8:30	Reactor	Cond 3	18.6	104.4	145.2	1.640	85.76	40.76	126.52	121.0	103	0.71	1.170			0.0	brown		70600	25900	3800	250	2700	2100	179	66	113	10	1	7	5	36.7	18.6
Apr-28 20:30	Reactor	Cond 4	18.6	103.4	139.6	1.687	84.86	36.20	121.06	112.6	98	0.83	1.153			0.0	brown	brown	63000	6100	3800	240	2600	2000	170	16	153	10	1	7	5	9.7	16.6
Apr-29 9:30	Feed	Cond 5	18.5	103.5	140.1	1.440	85.00	36.54	121.54	119.8	109	0.72	1.094			0.0	green		38900	36200	1800	110	1200	970	117	108	8	5	0	4	3	93.1	21.6
Apr-29 9:30	Boiler	Cond 5	18.5	94.4	131.1	1.580	75.91	36.73	112.64	110.7	96	0.88	1.156			0.0			72200	61000	3100	210	2200	1700	188	159	29	8	1	6	4	84.5	23.3
Apr-29 9:30	Reactor	Cond 5	18.6	102.2	135.3	1.650	83.59	33.12	116.71	111.6	97	1.31	1.146			0.0			57700	15200	3200	220	2300	1800	170	45	125	9	1	7	5	26.3	18.0
Apr-29 21:30	Reactor	Cond 6	18.7	95.6	136.6	1.630	76.96	40.94	117.90	129.9	109	0.71	1.188			0.0	brown	brown	77700	24300	4200	280	3000	2300	208	65	143	11	1	8	6	31.3	18.5
Apr-30 14:30	Feed	End	18.4	100.0	129.7	1.460	81.51	29.74	111.25	109.6	100	1.00	1.098			0.0	green		44100	39900	2000	140	1400	1100	148	134	14	7	0	5	4	90.5	22.1
Apr-30 14:30	Boiler	End	18.7	103.0	139.8	1.580	84.32	36.79	121.11	118.9	104	0.59	1.141			0.0	green		65800	57600	3000	200	2100	1700	186	163	23	8	1	6	5	87.5	21.9
Apr-30 14:30	Reactor	End	18.7	99.7	138.6	1.720	81.05	38.82	119.87	111.0	95	0.24	1.164			0.0	brown		64400	10100	4000	270	2800	2300	158	25	133	10	1	7	6	15.7	16.1
							0.00	0.00	0.00		-					0.0																	

Project: 12323-001
Client: Starfield Resources

Test ID: PP1

Condition:	**Cond 1**
initial:	Apr-27 8:30
final:	Apr-27 20:30
Feed:	4 mL/min
Reactor rpm:	1000 rpm
Reactor condensate:	Back to reactor

run time:	12.00	hours
avg feed flow:	5.73	g/min
	3.87	mL/min
Reactor retention time:	9.80	hours
Turnovers:	1.2	
feed:	4088	g
product:	2533	g pulp
	122	g solids

Preheater temp	81.7	℃
Boiler temp	119.7	℃
Reactor temp	120.1	℃
Vapours temp	99.8	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg
	grams							
Feed	4088	591	539	53	27	2	20	15
Reactor - start	1968	300	22	278	19	1	13	10
Discharge	2533	457	102	355	25	2	17	14
Solids	122	82			0.02	0.01	0.01	0.01
Reactor - end	1944	327	62	266	19	1	14	11
Condensates	1532							
(in-out)	47	25	398	-290	2	0	1	0
out/in	99%	97%	29%	188%	96%	110%	96%	98%

Ferrous Oxidized:	398	g
Ferrous Oxidized:	7	mol
% Ferrous Oxidised:	**74%**	

O_2 consumption	348.58	mL/min
	250.98	L
	20	psig
	21	C
	25	mole
O_2 utilization:	7%	

Project: 12323-001
Client: Starfield Resources

Test ID: PP1

Condition:	**Cond 2**
initial:	Apr-27 20:30
final:	Apr-28 8:30
Feed:	4 mL/min
Reactor rpm:	750 rpm
Reactor condensate:	Back to reactor

run time:	12.00	hours
avg feed flow:	5.55	g/min
	3.96	mL/min
Reactor retention time:	8.94	hours
Turnovers:	1.3	
feed:	3885	g
product:	2281	g pulp
	147	g solids

Preheater temp	82.1	℃
Boiler temp	119.7	℃
Reactor temp	120.8	℃
Vapours temp	99.7	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg
	grams							
Feed	3885	603	512	91	26	2	18	14
Reactor - start	1944	327	62	266	19	1	14	11
Discharge	2281	399	126	273	22	2	16	12
Solids	147	98			0.02	0.01	0.02	0.01
Reactor - end	1968	353	129	223	19	1	13	10
Condensates	1243							
(in-out)	337	81	318	-139	4	0	3	2
out/in	94%	91%	45%	139%	91%	86%	90%	91%

Ferrous Oxidized:	318	g
Ferrous Oxidized:	6	mol
% Ferrous Oxidised:	**62%**	

O_2 consumption	349.54	mL/min
	251.67	L
	20	psig
	21	C
	25	mole
O_2 utilization:	6%	

Project: 12323-001
Client: Starfield Resources

Test ID: PP1

Condition:	**Cond 3**
initial:	Apr-28 8:30
final:	Apr-28 20:50
Feed:	4 mL/min
Reactor rpm:	1250 rpm
Reactor condensate:	Back to reactor

run time:	12.33	hours
avg feed flow:	5.46	g/min
	3.74	mL/min
Reactor retention time:	10.00	hours
Turnovers:	1.2	
feed:	3886	g
product:	2450	g pulp
	190	g solids

Preheater temp	81.1	℃
Boiler temp	119.8	℃
Reactor temp	120.9	℃
Vapours temp	100.2	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg
	grams							
Feed	3886	543	500	43	25	2	18	14
Reactor - start	1968	353	129	223	19	1	13	10
Discharge	2450	427	91	336	24	2	18	14
Solids	190	128			0.04	0.02	0.02	0.01
Reactor - end	2024	344	33	311	21	1	14	11
Condensates	1210							
(in-out)	170	-4	505	-381	-1	0	-1	0
out/in	97%	100%	20%	243%	102%	104%	102%	101%

Ferrous Oxidized:	505	g
Ferrous Oxidized:	9	mol
% Ferrous Oxidised:	**101%**	

O_2 consumption	379.17	mL/min
	280.59	L
	20	psig
	21	C
	27	mole
O_2 utilization:	8%	

Project: 12323-001
Client: Starfield Resources

Test ID: PP1

Condition:	**Cond 4**
initial:	Apr-28 20:50
final:	Apr-29 9:30
Feed:	6 mL/min
Reactor rpm:	1250 rpm
Reactor condensate:	Back to reactor

run time:	12.67	hours
avg feed flow:	8.02	g/min
	5.49	mL/min
Reactor retention time:	6.74	hours
Turnovers:	1.9	
feed:	5912	g
product:	4051	g pulp
	302	g solids

Preheater temp	81.6	℃
Boiler temp	119.6	℃
Reactor temp	120.1	℃
Vapours temp	98.9	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg
	grams							
Feed	5912	826	761	65	38	2	27	21
Reactor - start	2024	344	33	311	21	1	14	11
Discharge	4051	746	187	559	40	3	28	22
Solids	302	200			0.06	0.03	0.05	0.02
Reactor - end	1980	336	89	248	19	1	13	10
Condensates	1930							
(in-out)	-24	-112	518	-430	0	0	0	0
out/in	100%	110%	35%	214%	100%	104%	101%	101%

Ferrous Oxidized:	518	g
Ferrous Oxidized:	9	mol
% Ferrous Oxidised:	**68%**	

O_2 consumption	546.38	mL/min
	415.25	L
	20	psig
	21	C
	41	mole
O_2 utilization:	6%	

Project: 12323-001
Client: Starfield Resources

Test ID: PP1

Condition:	**Cond 5**
initial:	Apr-29 9:30
final:	Apr-29 21:30
Feed:	8 mL/min
Reactor rpm:	1250 rpm
Reactor condensate:	Back to reactor

run time:	12.00	hours
avg feed flow:	10.87	g/min
	7.55	mL/min
Reactor retention time:	4.31	hours
Turnovers:	2.8	
feed:	7289	g
product:	5002	g pulp
	336	g solids

Preheater temp	80.6	℃
Boiler temp	118.4	℃
Reactor temp	118.1	℃
Vapours temp	98.9	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg
	grams							
Feed	7289	850	791	59	39	2	26	21
Reactor - start	1980	336	89	248	19	1	13	10
Discharge	5002	868	317	551	48	3	34	26
Solids	336	225			0.05	0.03	0.04	0.02
Reactor - end	1956	406	127	279	22	1	16	12
Condensates	2140							
(in-out)	171	-313	435	-523	-12	-1	-10	-7
out/in	98%	126%	50%	271%	121%	126%	124%	121%

Ferrous Oxidized:	435	g
Ferrous Oxidized:	8	mol
% Ferrous Oxidised:	**55%**	

O_2 consumption	725.58	mL/min
	522.42	L
	20	psig
	21	C
	51	mole
O_2 utilization:	4%	

Project: 12323-001
Client: Starfield Resources

Test ID: PP1

Condition:	**Cond 6**
initial:	Apr-29 21:30
final:	Apr-30 14:30
Feed:	8 mL/min
Reactor rpm:	1250 rpm
Reactor condensate:	Drawn off

run time:	17.00	hours
avg feed flow:	10.40	g/min
	7.23	mL/min
Reactor retention time:	5.89	hours
Turnovers:	2.9	
feed:	10497	g
product:	6535	g pulp
	468	g solids

Preheater temp	82.0	℃
Boiler temp	119.7	℃
Reactor temp	120.8	℃
Vapours temp	99.4	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg
	grams							
Feed	10497	1553	1405	148	70	5	49	39
Reactor - start	1956	406	127	279	22	1	16	12
Discharge	6535	1179	268	911	69	4	48	37
Solids	468	312			0.10	0.05	0.07	0.02
Reactor - end	1956	309	49	261	19	1	13	11
Condensates	3907							
(in-out)	55	159	1216	-745	4	1	3	3
out/in	100%	92%	21%	274%	95%	90%	95%	94%

Ferrous Oxidized:	1216	g
Ferrous Oxidized:	22	mol
% Ferrous Oxidised:	**87%**	

O_2 consumption	734.41	mL/min
	749.10	L
	20	psig
	21	C
	73	mole
O_2 utilization:	7%	

Project: 12323-001
Client: Starfield Resources

Test ID: PP1

Condition:	**Overall**
initial:	Apr-27 8:30
final:	Apr-30 14:30
Feed:	8 mL/min
Reactor rpm:	1250 rpm
Reactor condensate:	Drawn off

run time:	78.00	hours
avg feed flow:	7.82	g/min
	5.40	mL/min
Reactor retention time:	6.94	hours
Turnovers:	11.2	
feed:	35555	g
product:	22852	g pulp
	1796	g solids

Preheater temp	81.5	℃
Boiler temp	119.5	℃
Reactor temp	120.2	℃
Vapours temp	99.5	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg
	grams							
Feed	35555	5007	4513	495	227	15	159	124
Reactor - start	1956	298	22	276	19	1	13	10
Discharge	22852	4076	1091	2984	229	15	161	124
Solids	1796	1197			0.67	0.18	0.29	0.12
Reactor - end	1956	309	49	261	19	1	13	11
Condensates	11760							
(in-out)	944	-276	3395	-2474	-3	0	-3	-1
out/in	97%	105%	25%	421%	101%	102%	101%	101%

Ferrous Oxidized:	3395	g
Ferrous Oxidized:	61	mol
% Ferrous Oxidised:	**75%**	

O_2 consumption	526.40	mL/min
	2464	L
	20	psig
	21	C
	241	mole
O_2 utilization:	6%	

Appendix C - Hydrolysis Scoping Tests
Test details

Project: 12323-001 **Date:** 06-May-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 8** *Hydrolysis Tests*
Purpose: **To run a continuous hydrolysis test at 180C with pitch-blade impellor installed**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: **$2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$**

Conditions:

Preheater:

Temperature:	80	℃			
Reflux Condensation:	yes	(back into preheater)			
prefill with:	"Feed Solution (feed to hydrolysis) - Test 8"				
volume:	3000	mL			
Nitrogen blanket:	TBD	mL/min N2			

Test Duration: 8 hr
required feed: 960 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Boiler:

Temperature:	150	℃		
feed flow:	2	mL/min or:	2.8	g/min
sg:	1.4	g/mL	**0.168**	kg/h
prefill with:	"Boiler - Test 8"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	℃
feed flow:	as is	
prefill with:	"Hydrolysis bath - Test 8"	
dynamic volume:	700	mL
Water flow:	0.80	mL/min
Reflux Condensation:	**No**	
target condensate flow:	TBD	mL/min
estimated dischareg liquid flow:	TBD	mL/min

RPM: **600**
location impellor: 1
impellor: **pitch blade**

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 160C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its volume collected as function of time.
7. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
8. Collect reactor discharge pulp in pH 2 HCl. The following dilution is proposed:
   a. Collect 1 hours worth of discharge into 400.0 mL of pH 2 HCl of known weight.
   b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
   c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
9. Continue for total test duration of **8** hours
10. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash and determine wet and dry weight.

Project: 12323-001 **Date:** 06-May-10

Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 8** *Hydrolysis Tests*

Purpose: **Hydrolysis Scoping Test**

Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:	
Boiler RPM:	450
Hydrolysis Reactor RPM:	600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor						Observations
						Condensate			Bath	H2O		Condensate			(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Vol. mL	Temp ℃	Vol. mL	Flow mL/min	Vol. g	FAT g/L HCl	Density g/mL	
		80		2.00	150				180	1.05					
8:50	-1.67	83.2	10693.6	0.00	139.6				182.9	771	0.00				8:50 Feeding started
10:30	0.00	80.5	10208.0	4.86	150.1			51.4	180.3	771	0.00	113.8	104	1.045	10:30 Boiler began overflowing
11:30	1.00	82.3	9956.2	4.20	150.1	43.50	1.019	183.3	178.1	723	0.81	6.2	180	1.081	10:30 Started water addition
12:30	2.00	79.9	9764.2	3.20	150.0			9.85	180.8	671	0.86	127.6	144	1.065	11:30 Turned vaccum on
13:33	3.05	83.7	9525.2	3.79	156.1			10.31	178.8	623	0.77	175.4	154	1.069	13:15 Boiler temp increased to 180C
14:31	4.02	83.1	9333.8	3.30	179.5			12.77	181.0	576	0.80	249.6	164	1.074	15:30 Shutdown
15:30	5.00	80.8	9097.8	4.00	180.3	136.10	1.062	14.37	181.7	531	0.76	148.6	231	1.105	Sample HCl-2 was too little to do ORP and density
									Scrubber 1			800	8.1		**Conclusion:**
									Scrubber 2			950	0.2		Boiler and reactor condenate could not be
															separated. Both condensate streams
															were combined either as "boiler
															condensate" or as "reactor condensate"
															This can be remedied by installing a
															reactor downcomer below reactor pulp
															level.
Totals/Avg:															

Project: 12323-001
Client: Starfield Resources

Test ID: Test 8

Sampling Data

Sample Name		Weight	H2O+cup	HCl+cup	pulp	Sample Composition		Diluted Weight		Volume						Colour		Sample Analysis (Diluted Sample)								Calculated sample concentration								
Time	set #	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	pH at room T	SG g/mL	Wet res g	Dry res g	%H2O	PLS	RES	Fe mg/L	Fe(II) mg/L	Fe(III) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
10:30	Reactor initial	49.4	506.6	724.4		457.25	217.75	675	665.9	534	0.16	1.247	3.7	2.7	26%	blk/brown	red	83400	3010		15000	830	10000	8200	207300	205	7	197	37	2.0	25	20	508	204
11:30	Disch 1	50.2	463.6	610.5	685	413.41	146.86	560	545.7	459	0.19	1.190				green	red	57100	2270		17000	570	6600	5300	155600	178	7.1	171	53	1.8	21	17	486	244
12:30	Disch 2	49.4	451.8	545.1	678	402.44	93.29	496	481.7	427	0.51	1.129				green	red	38700	1530		11000	380	4000	3200	105700	177	7.0	170	50	1.7	18	15	483	255
13:33	Disch 3	50.2	461.2	503.4	663	411.02	42.15	453	442.6	418	0.92	1.058				green	red	17500	680		3800	170	1800	1400	47480	174	6.7	167	38	1.7	18	14	471	284
14:31	Disch 4	49.4	487.7	560.5	682	438.32	72.82	511	495.5	457	0.72	1.085				brown	red	27700	1020		3300	250	2600	2100	70660	174	6.4	167	21	1.6	16	13	443	331
15:30	Disch 5	50.2	463.5	522.0	664	413.26	58.53	472	455.6	426	0.70	1.070	20.8	11.9	43%	green	red	23100	840		2500	200	2100	1700	58560	168	6.1	162	18	1.5	15	12	426	359
10:30	HCl - 0				611				113.8	109		1.045						105								0.10								
11:30	HCl - 1								6.2	6		1.081						182								0.17								
12:30	HCl - 2				580				127.6	120		1.065						159								0.15								
13:33	HCl - 3				585				175.4	164		1.069						160								0.15								
14:31	HCl - 4				595				249.6	232		1.074						175								0.16								
15:30	HCl - 5				569				148.6	134		1.105						329								0.30								
11:30	Boiler 1								183.3	180		1.019																						
15:30	Boiler 2								14.4	14		1.062																						
15:30	Feed	18.5	96.3	129.3		77.88	32.97	110.85	109.5	97	0.66	1.134						54400	790		3000	210	2100	1800	55630	159	2	157	9	1	6	5	163	657
15:30	Boiler	18.6	105.6	145.0		87.00	39.40	126.40	118.6	99	0.23	1.202						87700	1640		4600	370	3800	3100	87030	220	4	215	12	1	10	8	218	533
15:30	Reactor	18.6	97.7	135.3		79.07	37.66	116.73	99.8	83	0.72	1.205						80700	2670		8300	660	7000	5500	87140	178	6	172	18	1	15	12	192	584
	Feed	hot SG										1.492																						
	Boiler	hot SG										1.748																						
	Reactor	hot SG										1.893																						
Solids assays																		%Fe	%Cu		%Co	%Ni	%Mg	%Cl										
	Disch 1-5												20.8	11.9	43%			52.2	0.023		<0.01	<0.01	0.12	18.54											
	Final Solids												270.5	116.4	57%			52.5	0.029		<0.01	<0.01	0.12	16.24											

Project: 12323-001
Client: Starfield Resources

Test ID: Test 8

initial: 10:30
final: 15:30
Feed: 2mL/min
Reactor rpm: 600
Reactor condensate: Drawn off

run time:	5.00	hours
avg feed flow:	3.89	g/min
	2.61	mL/min
average water addition	0.80	mL/min
average product flow	1.38	g/min
average product flow	0.73	mL/min
Reactor retention time:	16	hours
Test Turnovers:	0.3	
feed:	1110	g
product:	414	g pulp
	128	g solids

Preheater temp	81.7	℃
Boiler temp	161.0	℃
Reactor temp	180.1	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1110	177	3	174	10	1	7	6	181
Reactor - start	1325	271	10	261	49	3	32	27	674
H2O feed to reactor	240								
Discharge	414	73	3	70	17	1	8	6	194
Solids	128	67			0.04	0.01	0.01	0.15	21.11
Reactor - end	1325	235	8	227	24	2	20	16	254
Boiler Condensate	198								9
Reactor condensate	821	0.1							121
Scrubbers									6
Accumulation	0	-36	-2	-34	-25	-1	-12	-11	-420
IN:	1350	177	3	174	10	1	7	6	181
OUT:	1432	104	1	36	-8	0	-5	-4	-69
Accountability, OUT/IN	106	59	31	21	-81	-10	-66	-75	-38
Metal Hydrolysed				138	18	1	11	10	
% Metal Hydrolysed				79	181	110	166	175	
				39%					

Average HCl strength	15.2%
Maximum HCl strength	20.9%
mol of Fe hydrolysed	2.478
mol of Hematite	1.206
mol of HCl produced	3.848

Project: 12323-001 **Date:** 14-May-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 9** *Hydrolysis Tests*
Purpose: **To determine the effect of a more concentrated solution going into the reactor by operating the boiler @ 170°C.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: **$2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$**

Conditions:

Preheater:

Temperature:	80	°C		Test Duration:	8	hr
Reflux Condensation:	yes	(back into preheater)		required feed:	960	mL
prefill with:	"Feed Solution (feed to hydrolysis) - Test 8"			estimated boiler condensate:		mL
volume:	3000	mL		estimated reactor condensate:		mL
Nitrogen blanket:	TBD	mL/min N2		effluent volume:	TBD	mL

Boiler:

Temperature:	170	°C		
feed flow:	2	mL/min or:	2.8	g/min
sg:	1.4	g/mL	**0.168**	kg/h
prefill with:	"Boiler - Test 8"			
dynamic volume:	700	mL		
Reflux Condensation:	**Yes**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	°C	RPM:	**600**
feed flow:	as is		location impellor:	1
prefill with:	"Hydrolysis bath - Test 8"		impellor:	**pitch blade**
dynamic volume:	700	mL		
Water flow:	0.83	mL/min		
Reflux Condensation:	**No**			
target condensate flow:	TBD	mL/min		
estimated dischareg liquid flow:	TBD	mL/min		

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 160C. Prefill boiler condensor to point that "h" fitting overflows back into boiler vessel.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its volume collected as function of time.
7. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
8. Collect reactor discharge pulp in water. The following dilution is proposed:
   a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
   b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
   c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
9. Continue for total test duration of **8** hours
10. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weig

Project: 12323-001
Client: Starfield Resources

Date: 14-May-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 9 *Hydrolysis Tests*
Purpose: Hydrolysis Scoping Test
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:	
Boiler RPM:	450
Hydrolysis Reactor RPM:	600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Vol. g	Temp ℃	Vol. g	Flow g/min	Vol. g	ORP mV	FAT g/L HCl	Density g/mL	
6:30	-0.50	85.0	10591.5	0.00	184.8			0.0	185.9	0	0.00	0.0				06:30 Feed was started
7:00	0.00	82.1	10495.8	3.19	178.6			0.0	183.8	0	0.00	0.0				07:00 Boiler started to overflow. Water addition
8:03	1.05	80.0	10333.9	3.77	185.0			0.0	180.0	31	0.71	0.0				began at this point.
9:00	2.00	83.2	10163.0	3.00	169.3	106.04	1.048	13.0	183.2	66	0.61	0.0				07:08/07:20 Boiler O/F was plugged. Feed was
10:00	3.00	81.7	9928.9	3.90	170.1	99.00	1.045	73.2	180.5	117	0.86	40.0	565	214	1.095	turned off during this time.
11:00	4.00	80.1	9742.8	3.10	169.9	105.70	1.048	71.9	178.8	167	0.83	68.0	548	198	1.088	07:55/08:03 Boiler O/F was plugged. Feed was
12:00	5.00	83.5	9558.6	3.07	169.7	112.46	1.052	71.5	180.7	218	0.86	75.9	551	194	1.086	turned off during this time. Boiler temperature was
13:00	6.00	82.3	9359.9	3.31	170.4	120.16	1.053	72.0	179.9	270	0.86	68.9	551	197	1.083	decreased to 170C to avoid O/F plugging.
14:00	7.00	79.6	9082.0	4.63	170.5	124.17	1.054	74.5	180.8	56	0.93	70.3	547	207	1.088	13:00 Water addition was stopped. Instead boiler
15:00	8.00	83.6	8882.8	3.32	170.3	128.84	1.056	71.2	180.4	108	0.88	56.8	553	210	1.089	condensate was added to the reactor at the same
16:00	9.00	81.4	8599.6	4.72	170.4	130.52	1.057	69.9	180.3	162	0.89	63.6	552	218	1.093	flowrate water addition was.
																16:00 Shut down
									Scrubber 1			1007		18	1.008	
									Scrubber 2			950		0.1	0.999	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 9

Sampling Data

Sample Name		Weight	H2O+cup	HCl+cup	pulp	Sample Composition		Diluted Weight		Volume					Colour		Sample Analysis (Diluted Sample)							Calculated sample concentration								
Time	set #	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	FAT g/L HCl	SG g/mL	Wet res g	Dry res g	%H2O	PLS PLS	RES RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
	Reactor initial	50.2	512.5	1274.4	692	462.30	761.93	1224	1162.9	718	1.00	1.620						243000	7940	19000	2000	21000	22000	554100	229	7	221	18	2	20	21	522	189
	Disch 1	49.2	465.3	636.2	738	416.05	170.91	587	578.7	464	171	1.247						83100	2640	17000	810	10000	7800	204700	226	7	218	46	2	27	21	556	122
	Disch 2	49.2	487.7	762.3	726	438.56	274.60	713	686.8	519	220	1.322						111700	3200	20000	1200	14000	10000	269700	211	6	205	38	2	26	19	510	193
	Disch 3	49.2	460.5	669.3	719	411.25	208.83	620	599.0	480	169	1.249						87800	2450	14000	930	11000	8200	208500	202	6	196	32	2	25	19	479	241
	Disch 4	49.2	455.0	487.6	682	405.80	32.60	438	433.5	415	30	1.044						14000	440	2400	140	1600	1200	35730	178	6	173	31	2	20	15	455	299
	Disch 5	49.2	465.1	493.6	678	415.90	28.46	444	440.7	425	27	1.037						11100	380	2000	140	1400	1100	30160	166	6	160	30	2	21	16	450	315
	Disch 6	49.2	455.3	483.3	678	406.12	27.98	434	430.8	415	28	1.038						11200	400	1900	130	1400	1000	29970	166	6	160	28	2	21	15	445	324
	Disch 7	49.2	471.7	549.1	693	422.43	77.47	500	487.7	444	67	1.098						31600	1040	4600	380	3900	3000	79040	181	6	175	26	2	22	17	453	298
	Disch 8	49.2	429.3	525.6	697	380.09	96.32	476	460.0	407	89	1.131						42400	1400	6400	500	5800	4300	107200	179	6	173	27	2	24	18	453	297
	Disch 9	49.2	429.1	479.7	681	379.92	50.57	430	419.8	391	50	1.073	32.2	17.7	45.0			22800	760	3600	270	2900	2100	59400	176	6	171	28	2	22	16	460	295
	Disch 1-9 wash								445.3	451		1.012	52.1					9000		610	36	400	290		9			1	0	0	0	#DIV/0!	
	Reactor Wash								1622.7	1644		1.013						3400		680	29	320	250		3			1	0	0	0	#DIV/0!	
	HCl-1								40.0	44		1.095						1010															
	HCl-2								68.0	74		1.088						199															
	HCl-3								75.9	82		1.086						196															
	HCl-4								68.9	75		1.083						202															
	HCl-5								70.3	76		1.088						205															
	HCl-6								56.8	62		1.089						215															
	HCl-7								63.6	70		1.093						224															
	Feed	18.6	99.9	133.8	730	81.24	33.94	115.18	114.0	101		1.130						50400	660	3300	230	2300	1900	109300	150	2	148	10	1	7	6	325	502
	Boiler	18.7	96.3	138.1	724	77.59	41.86	119.45	105.6	86		1.221						84200	640	7000	610	7100	5300	107500	174	1	173	14	1	15	11	222	563
	Reactor	18.5	98.3	139.9	720	79.80	41.61	121.41	100.3	82		1.225						77200	2450	12000	860	9900	6900	189800	152	5	147	24	2	19	14	374	416
	Feed	hot SG										1.488																					
	Boiler	hot SG										1.783																					
	Reactor	hot SG										1.838																					
Solids assays																		%Fe	%Cu	%Co	%Ni	%Mg	%Cl									
	Disch 1-9												32.2	17.7	45.0%			49.6	0.043	<0.010	0.055	0.055	31.0										
	Boiler												119.9	52.7	56.0%			49.3	0.012	0.011	0.030	0.060	13.8										
	Reactor												177.6	90.9	48.8%			49.9	0.016	0.010	0.060	0.030	17.3										

Project: 12323-001
Client: Starfield Resources

Test ID: Test 9

initial: 7:00
final: 16:00
Feed: 2mL/min
Reactor rpm: 600
Reactor condensate: Drawn off

run time:	9.00	hours
avg feed flow:	3.60	g/min
	2.42	mL/min
average water addition	0.83	mL/min
average product flow	1.79	g/min
average product flow	0.97	mL/min
Reactor retention time:	12	hours
Test Turnovers:	0.8	
feed:	1992	g
product:	968	g pulp
	161	g solids

Preheater temp	82.0	℃
Boiler temp	173.5	℃
Reactor temp	181.3	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1992	299	4	295	20	1	14	11	647
Reactor - start	1287	295	10	285	23	2	25	27	672
H2O feed to reactor	432								
Discharge	968	194	6	188	34	2	24	18	477
Solids	161	80			0.01	0.01	0.03	0.04	28.47
Reactor - end	1287	196	6	189	30	2	25	17	481
Boiler Condensate	517								56
Reactor condensate	443	0.1							81
Scrubbers									18
Accumulation	0	-99	-3	-96	7	0	0	-9	-191
IN:	2424	299	4	295	20	1	14	11	647
OUT:	1928	175	2	93	41	2	24	9	469
Accountability, OUT/IN	80	59	63	31	210	136	175	79	72
Metal Hydrolysed				202	-22	0	-10	2	
% Metal Hydrolysed				69	-110	-36	-75	21	
				27%					

Average HCl strength	18.9%
Maximum HCl strength	19.9%
mol of Fe hydrolysed	3.618
mol of Fe (as hematite)	1.435
mol of HCl produced	4.361

Project: 12323-001 **Date:** 19-May-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 10** *Hydrolysis Tests*
Purpose: **To investigate the relationship between boiler operating temperature and free acid in the boiler condensate.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"Feed Solution (feed to hydrolysis) - Test 8"	
volume:	3000	mL
Nitrogen blanket:	TBD	mL/min N2

Boiler:

Temperature:	Variable	℃		
feed flow:	2	mL/min or:	2.8	g/min
sg:	1.4	g/mL	**0.168**	kg/h
prefill with:	"Boiler - Test 8"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Test Duration:	8	hr
required feed:	960	mL
estimated boiler condensate:		mL
estimated reactor condensate:		mL
effluent volume:	TBD	mL

Hydrolysis Reactor:

Temperature:	**180**	℃
feed flow:	as is	
prefill with:	"Hydrolysis bath - Test 8"	
dynamic volume:	700	mL
Water flow:	1.05	mL/min
Reflux Condensation:	**yes**	
target condensate flow:	TBD	mL/min
estimated dischareg liquid flow:	TBD	mL/min

RPM:	**600**
location impellor:	1
impellor:	**pitch blade**

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001 **Date:** 19-May-10

Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 10** *Hydrolysis Tests*

Purpose: Hydrolysis Scoping Test

Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:	
Boiler RPM:	150
Hydrolysis Reactor RPM:	600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
7:00	-1.42	82.1	10638.4		118.8	1.53	1.000	105.95	179.5	0		0.00				07:00 Feed was started
8:25	0.00	80.9	10279.9	4.22	121.0				181.5	0		0.00				08:25 Boiler started to overflow. Water addition
8:55	0.50					0.39	0.999	6.55		11	0.37					began at this point. Solution was boiling in the downcomer.
9:25	1.00	80.2	10064.7	3.59	119.7	0.24	1.000	2.97	185.5	21	0.34	0.00				09:25 Temperature in the boiler was increased up to 130℃
9:55	1.50	81.2	9983.0	2.72	123.4	1.44	0.997	62.43	178.3	33	0.40					10:55 Temperature in the boiler was increased up to 145℃
10:25	2.00	83.4	9890.6	3.08	130.7	2.74	0.998	90.09	180.4	44	0.37	18.57	530	259	1.117	At 160℃ boiler condensate started "gaining" color.
10:55	2.50	79.6	9778.3	3.74	129.9	1.45	0.999	13.77	180.6	55	0.38					12:25 Temperature in the boiler was increased up to 160℃
11:25	3.00	81.8	9674.8	3.45	137.5	4.86	0.999	89.11	180.2	67	0.38	16.98	482	264	1.116	At 170℃ some crystals apeeared in boiler walls.
11:55	3.50	83.5	9570.5	3.48	144.1	13.70	1.004	62.10	180.3	78	0.38					13:55 Temperature in the boiler was increased up to 170℃
12:25	4.00	79.4	9461.5	3.63	144.1	8.92	1.003	24.82	179.8	91	0.43	18.48	486	282	1.122	15:25 Temperature in the boiler was increased up to 180℃
12:55	4.50	81.5	9358.9	3.42	154.8	25.28	1.009	76.42	180.6	101	0.34					16:25 Shut down
13:25	5.00	83.4	9267.0	3.06	160.2	50.19	1.022	52.43	180.6	113	0.37	16.89	498	279	1.123	
13:55	5.50	82.7	9145.5	4.05	160.4	37.61	1.017	28.54	180.9	124	0.39					Exaust condensate flowrate (vaccuum) was 15mL/min
14:25	6.00	81.2	9049.6	3.20	167.5	62.74	1.027	61.33	181.2	135	0.37	26.12	500.0	265	1.117	
14:55	6.50	82.1	8940.6	3.63	168.9	102.70	1.047	43.10	179.6	147	0.40					
15:25	7.00	83.7	8823.0	3.92	169.1	128.80	1.058	40.50	179.9	159	0.40	28.67	594	266	1.114	
15:55	7.50	80.6	8736.4	2.89	176.3	166.44	1.073	67.66	180.1	169	0.34					
16:25	8.00	83.0	8589.2	4.91	180.3	245.62	1.106	65.71	177.3	180	0.36	23.45	491	291	1.126	
									Scrubber 1			981.90		17	1.007	
									Scrubber 2			967.20		4	1	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 10

Sampling Data

Sample Name		Weight	H2O+cup	HCl+cup	pulp	Sample Composition		Diluted Weight		Volume					Colour		Sample Analysis (Diluted Sample)							Calculated sample concentration								
Time	set #	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	PLS PLS	RES RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
8:25	Reactor initial	49.8	459.1	725.1	650	409.33	265.99	675	660	480	-0.26	1.375						130910	4400	21000	1300	16000	13000	341100	236	8	228	38	2	29	23	615	56
9:25	Disch 1	49.4	483.3	568.6	704	433.96	85.24	519	507.0	453	0.54	1.118						41050	1290	6800	410	4900	3800	101500	218	7	212	36	2	26	20	540	157
10:25	Disch 2	50.0	499.1	836.6	714	449.09	337.55	787	757.7	564	-0.38	1.343						123860	3650	20000	1200	14000	11000	313800	207	6	201	33	2	23	18	524	191
11:25	Disch 3	49.4	436.8	506.6	635	387.39	69.88	457	451.7	421	0.47	1.074						23140	684	4000	230	2500	1900	72200	139	4	135	24	1	15	11	435	374
12:25	Disch 4	50.0	461.1	513.2	621	411.13	52.10	463	456.4	442	0.66	1.032						8810	274	1500	89	1000	760	34410	75	2	72	13	1	8	6	292	605
13:25	Disch 5	49.4	449.3	504.0	634	399.90	54.73	455	445.8	423	0.47	1.053						15630	466	2500	150	1700	1300	52680	121	4	117	19	1	13	10	408	428
14:25	Disch 6	50.0	431.4	479.7	627	381.43	48.26	430	419.5	398	0.66	1.054						16500	508	2700	170	1800	1400	51310	136	4	132	22	1	15	12	423	391
15:25	Disch 7	49.4	439.3	489.2	620	389.89	49.90	440	428.4	406	0.81	1.056						17110	531	2800	170	1900	1500	51230	139	4	135	23	1	15	12	416	393
16:25	Disch 8	50.0	454.7	501.1	623	404.69	46.46	451	441.3	420	0.77	1.050						15720	468	2400	150	1700	1300	47920	142	4	138	22	1	15	12	434	374
	HCl-1									17								1660															
	HCl-2									15								278															
	HCl-3									16								265															
	HCl-4									15								270															
	HCl-5									23								299															
	HCl-6									26								276															
	HCl-7									21								317															
16:00	Feed	18.5	105.6	138.3		87.13	32.64	119.77	119.6	107		1.120						47700	745	3100	210	2300	1800	104000	156	2	154	10	1	8	6	340	480
16:00	Boiler	18.6	93.8	135.3		75.14	41.59	116.73	111.4	89		1.247						101000	1910	5100	480	5400	4300	233000	217	4	213	11	1	12	9	501	250
16:00	Reactor	18.5	93.5	131.0		75.00	37.42	112.42	102.4	84		1.222						85800	2520	13000	750	9400	7200	212600	192	6	186	29	2	21	16	476	264
	Feed	hot SG										1.506																					
	Boiler	hot SG										1.728																					
	Reactor	hot SG										1.849																					
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl									
	Disch 1-8												26.3	16.5	37.5%			40.7	22.3	0.072	0.01	0.052	0.023	13.3									
	Reactor												147.1	84.6	42.5%			48.8	1.9	0.012	<0.01	0.043	<0.005	22.1									
	Boiler												105.7	62.1	41.2%			51.4	0.18	0.03	<0.01	0.018	<0.005	21.0									

Project: 12323-001
Client: Starfield Resources

Test ID: Test 10

initial: 8:25
final: 16:25
Feed: 2mL/min
Reactor rpm: 600
Reactor condensate: Drawn off

run time:	8.00	hours
avg feed flow:	3.56	g/min
	2.37	mL/min
average water addition	0.38	mL/min
average product flow	1.55	g/min
average product flow	0.84	mL/min
Reactor retention time:	14	hours
Test Turnovers:	0.6	
feed:	1691	g
product:	744	g pulp
	163	g solids

Preheater temp	81.8	℃
Boiler temp	147.5	℃
Reactor temp	180.4	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1691	264	4	260	17	1	13	10	575
Reactor - start	1294	305	10	295	49	3	37	30	796
H2O feed to reactor	180								
Discharge	744	129	4	125	21	1	15	11	352
Solids	163	80			0.02	0.01	0.04	0.01	20.91
Reactor - end	1294	249	7	241	38	2	27	21	616
Boiler Condensate	788								44
Reactor condensate	149	0.1							35
Scrubbers									19
Accumulation	0	-57	-3	-54	-11	-1	-10	-9	-180
IN:	1871	264	4	260	17	1	13	10	575
OUT:	1681	152	1	71	10	0	4	2	273
Accountability, OUT/IN	90	58	22	27	56	35	35	19	47
Metal Hydrolysed				189	7	1	8	8	
% Metal Hydrolysed				73	44	65	65	81	
				31%					

Average HCl strength	24.3%
Maximum HCl strength	25.8%
mol of Fe hydrolysed	3.378
mol of Fe (as hematite)	1.431
mol of HCl produced	2.791

Project: 12323-001 **Date:** 01-Jun-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 11** *Hydrolysis Tests*
Purpose: **To determine the effect of hematite seeding in the formation of hematite and/or iron oxy chloride compunds.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: **$2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$**

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"Feed Solution (feed to hydrolysis) - Test 8"	
volume:	3000	mL
Nitrogen blanket:	TBD	mL/min N2

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	2.8	g/min
sg:	1.4	g/mL	**0.168**	kg/h
prefill with:	"Boiler - Test 8"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Test Duration: 8 hr
required feed: 960 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Hydrolysis Reactor:

Temperature:	**180**	℃
feed flow:	as is	
prefill with:	"Hydrolysis bath - Test 8"	
dynamic volume:	700	mL
Water flow:	0.40	mL/min
Reflux Condensation:	**No**	
target condensate flow:	TBD	mL/min
estimated dischareg liquid flow:	TBD	mL/min

RPM: **600**
location impellor: 1
impellor: **pitch blade**

Sampling Requirements & Instructions:

Partial Filtrates: Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate: Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate: Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 01-Jun-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 11 *Hydrolysis Tests*
Purpose: **Hydrolysis Scoping Test**
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:	
Boiler RPM:	150
Hydrolysis Reactor RPM:	600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
																Around 98grams (dry equivalent) of hematite produced in the
8:00	0.00	83.3	10171.9		137.4	3.00	1.000	288.56	189.2	0		0				oxidation test were added to the reactor as seed
9:00	1.00	82.2	9965.8	3.44	140.0	3.42	1.000	42.07	179.2	24	0.39	0				Seeding made the slurry really thick (pudding-like), so the
10:00	2.00	79.9	9751.2	3.58	139.1	3.68	1.001	46.90	180.7	49	0.42	21	428	288	1.127	reactor rpm was set to 1000 rpm to achieve a better mixing
11:00	3.00	82.8	9558.5	3.21	140.4	3.65	1.001	45.75	179.3	70	0.35	26	462	293	1.128	Feed was started at 8:00, as well as water addition.
12:00	4.00	83.1	9356.5	3.37	139.9	3.47	1.001	45.10	179.0	93	0.39	30	410	290	1.127	Vacuum was unstable the whole test, this could cause
13:00	5.00	83.8	9154.2	3.37	139.8	3.49	1.000	44.17	178.3	120	0.45	32	406	299	1.131	some HCl stayed in the reactor.
14:00	6.00	82.6	8948.7	3.43	140.3	3.39	1.001	45.08	179.0	141	0.34	15	402	290	1.124	16:00 Shut down
15:00	7.00	83.3	8753.5	3.25	139.8	3.52	1.001	43.64	178.9	162	0.36	30	410	292	1.127	
16:00	8.00	81.4	8518.8	3.91	139.5	3.48	1.000	45.61	179.1	184	0.37	23	450	319	1.137	
									Scrubber 1			965.75		5.7	1.002	
									Scrubber 2			975.65		0.6	0.999	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 11

Sampling Data

Sample Name		Weight	H2O+cup	HCl+cup	pulp	Sample Composition		Diluted Weight		Volume						Colour		Sample Analysis (Diluted Sample)							Calculated sample concentration								
Time	set #	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	PLS	RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
8:00	Reactor initial	49.2	458.0	615.6	626	408.74	157.60	566	549	474	0.32	1.160				brown	red	58300	671	5200	410	4900	4800	133200	175	2	173	16	1	15	14	400	379
9:00	Disch 1	49.9	482.9	620.6	656	433.03	137.66	571	549.3	477	0.41	1.152				brown	red	51600	678	6500	400	5200	4600	124300	179	2	176	23	1	18	16	431	333
10:00	Disch 2	49.2	431.5	733.2	651	382.22	301.77	684	628.7	478	-0.21	1.316				brown	red	125300	1380	14000	890	11000	9600	271100	198	2	196	22	1	17	15	429	316
11:00	Disch 3	49.9	440.9	598.8	625	390.96	157.88	549	533.6	454	0.21	1.175				brown	red	63600	868	7400	490	5800	4600	143900	183	2	180	21	1	17	13	414	351
12:00	Disch 4	49.2	425.8	970.6	671	376.54	544.82	921	746.1	505	-0.49	1.476				brown	red	175000	2240	25000	1500	18000	14000	414900	162	2	160	23	1	17	13	385	398
13:00	Disch 5	49.9	430.2	491.4	601	380.24	61.26	442	435.7	412	0.42	1.058				green	red	18100	265	2800	170	1700	1200	52730	122	2	120	19	1	11	8	354	484
14:00	Disch 6	49.2	414.6	520.7	594	365.32	106.15	471	457.1	410	0.23	1.114				green	red	37800	600	5600	320	3300	2300	97400	146	2	144	22	1	13	9	377	433
15:00	Disch 7	49.9	441.7	556.5	594	391.80	114.80	507	483.7	433	0.24	1.117				green	red	39500	712	6100	360	3700	2600	97340	149	3	146	23	1	14	10	367	436
16:00	Disch 8	49.2	438.4	537.5	582	389.13	99.14	488	469.1	426	0.25	1.102				green	red	35200	944	5100	290	3000	2200	85960	151	4	147	22	1	13	9	369	434
	HCl-1								21.2	19		1.127						535															
	HCl-2								26.4	23		1.128						391															
	HCl-3								29.5	26		1.127						465															
	HCl-4								31.6	28		1.131						465															
	HCl-5								14.9	13		1.124						413															
	HCl-6								30.2	27		1.127						408															
	HCl-7								23.3	20		1.137						444															
16:00	Feed	18.5	97.3	132.9	112.93	78.87	35.58	114.45	112.9	99		1.143						54800	935	3500	230	2300	1800	115800	152	3	150	10	1	6	5	322	505
16:00	Boiler	18.6	101.0	139.0	119.23	82.31	38.08	120.39	119.2	100		1.192						77400	1300	5100	330	3300	2700	162400	203	3	200	13	1	9	7	427	340
16:00	Reactor	18.5	98.5	139.7	100.23	79.96	41.25	121.21	100.2	82		1.217						78500	1790	12000	760	7800	5800	181000	157	4	153	24	2	16	12	361	429
	Feed	hot SG										1.486																					
	Boiler	hot SG										1.575																					
	Reactor	hot SG										1.938																					
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl									
	Disch 1-8								1762.0			1.008	126.3	74.7	40.9%		orange	55.7	0.01	0.030	<0.01	0.011	<0.005	7.6									
	Reactor								3945.1			1.013	242.3	150.2	38.0%		red	46.9	0.78	0.030	0.02	0.300	0.006	11.0									
	Boiler												0.0																				

Project: 12323-001
Client: Starfield Resources

Test ID: Test 11

initial: 8:00
final: 16:00
Feed: 2mL/min
Reactor rpm: 1000
Reactor condensate: Drawn off

run time:	8.00	hours
avg feed flow:	3.44	g/min
	2.32	mL/min
average water addition	0.38	mL/min
average product flow	3.17	g/min
average product flow	1.64	mL/min
Reactor retention time:	7	hours
Test Turnovers:	1.1	
feed:	1653	g
product:	1523	g pulp
	225	g solids

Preheater temp	82.5	℃
Boiler temp	139.6	℃
Reactor temp	180.3	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
	grams								
Feed	1653	252	4	247	16	1	11	8	532
Seed	98	65			0	0	0	0	3
Reactor - start	1357	238	3	235	21	2	20	20	543
H2O feed to reactor	184								
Discharge	1523	257	4	253	34	2	24	19	604
Solids	225	112			0.07	0.04	0.46	0.01	22.17
Reactor - end	1357	213	5	208	33	2	21	16	490
Boiler Condensate	358								2
Reactor condensate	177	0.1							45
Scrubbers									6
Accumulation	0	-25	2	-27	11	0	1	-4	-53
IN:	1936	317	4	247	16	1	11	8	534
OUT:	2059	344	6	226	45	3	26	16	621
Accountability, OUT/IN	106	109	133	91	282	235	246	189	116
Metal Hydrolysed				21	-29	-1	-15	-7	
% Metal Hydrolysed				9	-182	-135	-146	-89	
				19%					

Average HCl strength	26.2%
Maximum HCl strength	28.0%
mol of Fe hydrolysed	0.378
mol of Fe (precipitate)	0.834
mol of HCl produced	1.494

Project: 12323-001 **Date:** 08-Jun-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 12** *Hydrolysis Tests*
Purpose: **To determine the effect of water addition in hydrochloric acid strength.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"Feed Solution (feed to hydrolysis) - Test 8"	
volume:	3000	mL
Nitrogen blanket:	TBD	mL/min N2

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	2.8	g/min
sg:	1.4	g/mL	**0.168**	kg/h
prefill with:	"Boiler - Test 8"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Test Duration: 8 hr
required feed: 960 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Hydrolysis Reactor:

Temperature:	**180**	℃
feed flow:	as is	
prefill with:	"Hydrolysis bath - Test 8"	
dynamic volume:	700	mL
Water flow:	Nill	mL/min
Reflux Condensation:	**No**	
target condensate flow:	TBD	mL/min
estimated dischareg liquid flow:	TBD	mL/min

RPM: **600**
location impellor: 1
impellor: **pitch blade**

Sampling Requirements & Instructions:

Partial Filtrates: Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate: Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate: Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 08-Jun-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 12 *Hydrolysis Tests*
Purpose: **Hydrolysis Scoping Test**
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
																95.4 grams (dry weight) were added to the reactor as seed.
7:30	0.00	82.6	9791.3		130.0	40.44	1.017	337.15	178.0	0		0.0				As usual the solutions were made up the day before to the
8:30	1.00	80.0	9625.1	2.77	140.3	4.03	1.000	110.02	178.4	0	0.00	0.0				test and let them at temperature overnight. In the morning
9:30	2.00	83.6	9424.0	3.35	140.0	2.81	1.000	41.20	180.5	0	0.00	3.8		274	1.12	the boiler solution level was below the temperature probe and
10:30	3.00	82.3	9233.9	3.17	139.1	2.01	1.000	38.06	178.7	0	0.00	16.2	500	294	1.13	was overheated. Because of this, some HCl (40.44g/L) was
11:30	4.00	79.4	9038.5	3.26	139.9	2.18	1.000	43.54	180.6	0	0.00	27.9	530	298	1.132	released in the boiler condensate and some red-like solids
12:30	5.00	84.0	8860.6	2.96	140.1	2.34	1.000	40.10	180.6	0	0.00	26.7	773	298	1.134	were found in the boiler.
13:30	6.00	82.6	8657.8	3.38	140.0	2.32	1.000	41.72	180.3	0	0.00	25.7	855	300	1.136	Feed was added at 7:30, boiler solution began overflowing
14:30	7.00	79.8	8470.5	3.12	140.3	2.55	0.999	42.68	178.4	0	0.00	24.6	873	318	1.14	at the same time.
15:30	8.00	84.1	8254.9	3.59	140.2	2.48	1.000	38.34	180.4	0	0.00	38.6	847	353	1.153	reactor condensate @9:30 was not enough to measure ORP
																Because of the condenser geometry some liquid built up at
																the beginning of the test, that's why the first condensate
																amount is smaller than the rest, as well as the last one is
																bigger than the rest because we recovered the liquid
																inside the condenser at the end of the test.
																15:30 Shut down.
															1	
															1	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 12

Sampling Data

Sample Name		Weight	H2O+cup	HCl+cup	pulp	Sample Composition		Diluted Weight		Volume					Colour		Sample Analysis (Diluted Sample)							Calculated sample concentration								
Time	set #	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	PLS	RES	Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O
7:30	Reactor initial	50.0	457.8	673.7	629	407.76	215.93	624	586	464	0.31	1.261						89800	3100	13000	870	10000	7900	218400	193	7	186	28	2	22	17	470	269
8:30	Disch 1	49.3	438.0	735.6	684	388.75	297.54	686	639.3	479	0.24	1.335						116000	2050	20000	1100	13000	10000	290600	187	3	183	32	2	21	16	468	275
9:30	Disch 2	50.0	419.2	496.7	662	369.14	77.53	447	482.2	410	-0.06	1.176						61500	1550	9500	560	6200	4600	149800	325	8	317	50	3	33	24	792	228
10:30	Disch 3	49.3	440.2	565.0	648	390.91	124.84	516	495.3	433	0.25	1.143						48000	1540	7200	420	4900	3700	120500	167	5	161	25	1	17	13	418	359
11:30	Disch 4	50.0	447.4	553.7	651	397.36	106.29	504	482.8	425	0.53	1.136						47400	1850	6400	400	4400	3300	113900	190	7	182	26	2	18	13	455	297
12:30	Disch 5	49.3	429.2	556.5	677	379.87	127.35	507	483.2	415	0.41	1.164						56500	1300	7400	470	5200	4200	138800	184	4	180	24	2	17	14	452	307
13:30	Disch 6	50.0	453.7	547.1	670	403.67	93.38	497	474.0	423	0.71	1.120						42500	1660	5100	340	3600	3000	100300	193	8	185	23	2	16	14	455	298
14:30	Disch 7	49.3	440.1	553.5	667	390.80	113.40	504	476.8	418	0.43	1.142						50800	750	6200	400	4400	3500	121300	187	3	184	23	1	16	13	447	313
15:30	Disch 8	50.0	427.8	478.6	645	377.81	50.80	429	413.3	387	0.97	1.068						22600	780	2700	170	1900	1400	56700	172	6	166	21	1	14	11	432	349
9:30	HCl-1								3.8	3.4		1.120						670														
10:30	HCl-2								16.2	14.4		1.130						449														
11:30	HCl-3								27.9	24.6		1.132						351														
12:30	HCl-4								26.7	23.6		1.134						369														
13:30	HCl-5								25.7	22.6		1.136						383														
14:30	HCl-6								24.6	21.6		1.140						388														
15:30	HCl-7								38.6	33.4		1.153						440														
	Feed	12.3	96.4	129.7		84.12	33.33	117.45	116.1	103		1.127						48300	1740	3000	210	2100	1700	107200	149	5	144	9	1	6	5	331	498
	Boiler	12.3	87.0	124.6		74.70	37.52	112.22	108.5	90		1.200						83000	2540	5600	370	3800	2800	177700	200	6	194	13	1	9	7	428	342
15:30	Reactor	12.4	96.3	137.7		83.89	41.35	125.24	114.2	93		1.226						87100	210	8900	620	6700	5100	196900	196	0	196	20	1	15	11	444	312
	Feed	hot SG										1.466																				
	Boiler	hot SG										1.640																				
	Reactor	hot SG										1.844																				
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl								
	Disch 1-8								1322.4			1.006	119.8	63.5	47.0%			59.9		0.034	0.034	0.024	0.006	6.0								
	Reactor												340.7	290.8	14.7%			62.9		0.068	0.032	0.043	0.150	8.2								
	GF solids													11.0				50.9		0.016	0.029	0.039	0.006	23.6								

Project: 12323-001
Client: Starfield Resources

Test ID: Test 12

initial: 7:30
final: 15:30
Feed: 2mL/min
Reactor rpm: 1000
Reactor condensate: Drawn off

run time:	8.00	hours
avg feed flow:	3.20	g/min
	2.18	mL/min
average water addition	Nill	mL/min
average product flow	2.06	g/min
average product flow	1.12	mL/min
Reactor retention time:	10	hours
Test Turnovers:	0.8	
feed:	1536	g
product:	991	g pulp
	365	g solids

Preheater temp	82.0	℃
Boiler temp	138.9	℃
Reactor temp	179.5	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1536	229	8	221	14	1	10	8	509
Seed	95	64			0	0	0	0	2
Reactor - start	1291	249	9	290	45	3	32	25	730
H2O feed to reactor	0								
Discharge	991	193	5	188	28	2	19	15	474
Solids	365	227			0.22	0.11	0.14	0.44	27.71
Reactor - end	1291	253	1	253	26	2	19	15	573
Boiler Condensate	396								1
Reactor condensate	163	0.1							44
Scrubbers									0
Accumulation	0	4	-8	-37	-19	-1	-13	-11	-157
IN:	1632	293	8	221	14	1	10	8	512
OUT:	1550	424	-3	151	9	1	6	5	389
Accountability, OUT/IN	95	145	-40	68	66	70	64	56	76
Metal Hydrolysed				70	5	0	4	4	
% Metal Hydrolysed				32	34	30	36	44	
				74%					

Average HCl strength	26.9%
Maximum HCl strength	30.6%
mol of Fe hydrolysed	1.251
mol of Fe (as hematite)	2.919
mol of HCl produced	1.632

Project: 12323-001 **Date:** 15-Jun-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 13** *Hydrolysis Tests*
Purpose: **To determine the effect of a ferrous-free feed solution in the hydrolysis.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"Feed Solution (feed to hydrolysis) - Test 13"	
volume:	3000	mL
Nitrogen blanket:	TBD	mL/min N2

Test Duration: 8 hr
required feed: 960 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	3.2	g/min
sg:	1.6	g/mL	**0.192**	kg/h
prefill with:	"Boiler - Test 8"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	℃
feed flow:	as is	
prefill with:	"Hydrolysis bath - Test 13"	
dynamic volume:	700	mL
Water flow:	Nill	mL/min
Reflux Condensation:	**No**	
target condensate flow:	TBD	mL/min
estimated dischareg liquid flow:	TBD	mL/min

RPM: **600**
location impellor: 1
impellor: **pitch blade**

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Nill - Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 15-Jun-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 13 *Hydrolysis Tests*
Purpose: Hydrolysis Scoping Test
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
																92.6 grams of hematite were added as seed
7:50	0.00	80.2	10232.7		134.9	4.45	1.000	211.50	181.2							07:50 Feed Started. Boiler O/F started to flow.
8:50	1.00	82.0	10052.8	3.00	139.7	3.06	1.000	49.48	180.9							09:50 The amount of condensate HCl was too small. It was
9:51	2.02	82.6	9855.6	3.23	139.2	5.01	1.001	59.41	180.3							kept in the condenser and collected at 10:50
10:50	3.00	80.9	9680.5	2.97	139.9	3.36	1.001	37.14	180.6			14.9	883	295.0	1.128	At the end of the test the condenser was emptied, that's why
11:50	4.00	82.0	9500.2	3.00	139.0	3.36	1.000	39.71	179.6			17.3	952	273.1	1.120	the last reading is bigger than the rest.
12:50	5.00	80.9	9314.4	3.10	138.9	3.43	1.000	41.59	178.9			24.4	983	278.9	1.123	15:51 Shutdown
13:51	6.02	79.9	9128.2	3.05	140.2	3.06	1.000	39.16	179.2			25.0	976	283.6	1.126	
14:51	7.02	83.7	8954.3	2.90	139.6	3.36	1.000	37.20	179.8			23.4	993	293.0	1.130	
15:51	8.02	81.3	8784.0	2.84	140.3	3.29	1.000	41.14	180.7			36.0	975	306.2	1.134	
									Scrubber 1			978.60		0.9	0.999	
									Scrubber 2			1006.30		0.3	0.999	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 13

Sampling Data

Sample Name Time	set #	Weight cup (tare) g	H2O+cup (gross) g	HCl+cup + discharge g	pulp ORP mV	Sample Composition Diluent g	Sample g	Diluted Weight pulp g	PLS g	Volume PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	Colour PLS	RES	Sample Analysis (Diluted Sample) Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Calculated sample concentration Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
7:30	Reactor initial	48.7	406.4	585.7	710	357.79	179.26	537	512	408		1.254				yellow/brown	red	84700	580	13000	890	10000	7700	217600	193	1	191	30	2	23	18	495	240
8:50	Disch 1	48.1	422.9	562.2	726	374.82	139.31	514	495.4	417	0.42	1.189				green/brown	red	61600	580	9900	630	7000	5300	158900	184	2	183	30	2	21	16	475	272
9:51	Disch 2	48.7	423.6	504.6	701	374.95	81.00	456	438.8	396	0.40	1.107				green/brown	red	35800	360	5300	340	3800	2800	92700	175	2	173	26	2	19	14	454	311
10:50	Disch 3	48.1	433.0	544.6	713	384.96	111.60	497	482.0	423	0.36	1.140				green/brown	red	48100	450	6700	400	4500	3700	122500	182	2	181	25	2	17	14	464	296
11:50	Disch 4	48.7	412.5	530.4	708	363.84	117.90	482	466.0	405	0.54	1.151				green/brown	red	51300	460	7600	430	4700	4100	132200	176	2	175	26	1	16	14	454	312
12:50	Disch 5	48.1	430.0	539.5	726	381.89	109.52	491	474.6	416	0.41	1.141				green/brown	red	47400	500	6200	400	4400	3500	121400	180	2	178	24	2	17	13	461	304
13:51	Disch 6	48.7	409.4	514.1	717	360.72	104.70	465	448.0	392	0.56	1.143				green/brown	red	48800	460	6000	390	4300	3300	123300	183	2	181	22	1	16	12	462	303
14:51	Disch 7	48.1	453.1	558.4	716	404.99	105.29	510	489.7	434	0.67	1.128				yellow/brown	red	43600	460	5500	350	3800	3100	109300	180	2	178	23	1	16	13	451	317
15:51	Disch 8	48.7	401.6	456.7	704	352.96	55.07	408	376.5	349	1.04	1.080				green/brown	red	26700	330	3100	210	2300	1700	67400	169	2	167	20	1	15	11	427	358
10:50	HCl-1								14.9	13.2		1.128						646															
11:50	HCl-2								17.3	15.4		1.120						327															
12:50	HCl-3								24.4	21.7		1.123						377															
13:51	HCl-4								25.0	22.2		1.126						360															
14:51	HCl-5								23.4	20.7		1.130						387															
12:50	HCl-6								36.0	31.7		1.134						405															
Reactor solids wash									1831.8	1739.6		1.053						14500	330	8000	96	1100	900	41000									
15:30	Feed	18.5	108.3	141.4		89.85	33.02	122.87	121.6	109		1.119						45600	220	2800	190	2000	1500	99800	150	1	149	9	1	7	5	328	500
15:30	Boiler	18.4	99.1	135.4		80.70	36.26	116.96	115.8	97		1.196						77500	290	4400	310	3200	2400	166500	207	1	206	12	1	9	6	445	321
15:30	Reactor	18.2	101.5	143.1		83.25	41.64	124.89	117.3	95		1.237						85800	690	9500	650	6900	5300	202500	195	2	194	22	1	16	12	461	293
15:30	Feed	hot SG										1.498																					
15:30	Boiler	hot SG										1.608																					
15:30	Reactor	hot SG										1.817																					
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl									
	Disch 1-8												53.8	42.7	20.6%			58.6		0.045	0.029	0.017		6.0									
	Reactor												174.5	95.7	45.1%			55.0		0.041	0.028	0.026		13.1									

Project: 12323-001
Client: Starfield Resources

Test ID: Test 13

initial: 7:50
final: 15:51
Feed: 2mL/min
Reactor rpm: 1000
Reactor condensate: Drawn off

run time:	8.02	hours
avg feed flow:	3.01	g/min
	2.01	mL/min
average water addition	Nill	mL/min
average product flow	1.71	g/min
average product flow	0.94	mL/min
Reactor retention time:	12	hours
Test Turnovers:	0.6	
feed:	1449	g
product:	824	g pulp
	138	g solids

Preheater temp	81.5	℃
Boiler temp	139.1	℃
Reactor temp	180.1	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1449	217	1	216	13	1	10	7	476
Seed	93	62			0	0	0	0	2
Reactor - start	1272	245	2	243	38	3	29	22	630
H2O feed to reactor	0								
Discharge	824	148	1	147	21	1	14	11	378
Solids	138	78			0.06	0.04	0.03	0.00	15.10
Reactor - end	1272	248	2	246	28	2	20	15	586
Boiler Condensate	345								2
Reactor condensate	141	0.1							35
Scrubbers									1
Accumulation	0	3	0	3	-10	-1	-9	-7	-43
IN:	1541	279	1	216	13	1	10	7	478
OUT:	1310	229	2	150	10	1	5	4	387
Accountability, OUT/IN	85	82	170	69	78	69	55	60	81
Metal Hydrolysed				67	3	0	4	3	
% Metal Hydrolysed				31	22	31	45	40	
				7%					

Average HCl strength	25.6%
Maximum HCl strength	27.0%
mol of Fe hydrolysed	1.195
mol of Fe (as hematite)	0.287
mol of HCl produced	1.054

Project: 12323-001 **Date:** 17-Jun-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricady

Test ID: **Test 14** *Hydrolysis Tests*
Purpose: **To determine the effect of a ferrous-and-copper-free feed solution in the hydrolysis.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: **$2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$**

Conditions:

Preheater:

Temperature:	80	℃			Test Duration:	8	hr
Reflux Condensation:	yes	(back into preheater)			required feed:	960	mL
prefill with:	"Feed Solution (feed to hydrolysis) - Test 14"				estimated boiler condensate:		mL
volume:	3000	mL			estimated reactor condensate:		mL
Nitrogen blanket:	TBD	mL/min N2			effluent volume:	TBD	mL

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	3.2	g/min
sg:	1.6	g/mL	**0.192**	kg/h
prefill with:	"Boiler - Test 14"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	℃	RPM:	**600**
feed flow:	as is		location impellor:	1
prefill with:	"Hydrolysis bath - Test 14"		impellor:	**pitch blade**
dynamic volume:	700	mL		
Water flow:	Nill	mL/min		
Reflux Condensation:	**No**			
target condensate flow:	TBD	mL/min		
estimated dischareg liquid flow:	TBD	mL/min		

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Nill - Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 17-Jun-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 14 *Hydrolysis Tests*
Purpose: **Hydrolysis Scoping Test**
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:	
Boiler RPM:	150
Hydrolysis Reactor RPM:	600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
																82.9 grams of hematite were added as seed.
7:20	0.00	83.0	10097.6		138.6				180.4			67.9		149	1.068	07:20 Feed was started. Boiler O/F started to flow.
8:20	1.00	82.8	9947.4	2.50	139.0	4.89	1.001	58.05	180.1			10.0	960	303	1.137	This time HCl was produced since the beginning, mostly
9:20	2.00	80.9	9772.6	2.91	139.5	2.63	1.000	40.72	180.3			17.2	955	326	1.143	because more water was added to the reactor in the making
10:20	3.00	82.8	9591.2	3.02	140.2	2.34	1.000	42.86	180.3			17.1	981	312	1.139	up.
11:20	4.00	83.0	9412.8	2.97	140.3	5.11	1.001	40.37	180.7			14.6	975	324	1.141	
12:20	5.00	80.1	9236.5	2.94	139.2	3.03	1.000	27.64	179.2			19.7	966	336	1.147	
13:20	6.00	83.8	9065.6	2.85	140.2	3.62	1.000	47.15	178.8			16.0	942	335	1.145	
14:20	7.00	81.1	8869.6	3.27	139.3	2.41	1.000	36.67	180.2			11.8	942	346	1.152	
15:20	8.00	83.6	8702.8	2.78	140.2	2.11	1.000	40.98	180.6			22.5	935	366	1.160	
									Scrubber 1			841.38		0.4	0.999	
									Scrubber 2			703.71		0.1	0.999	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 14

Sampling Data

Sample Name Time	set #	Weight cup (tare) g	H2O+cup (gross) g	HCl+cup + discharge g	pulp ORP mV	Sample Composition Diluent g	Sample g	Diluted Weight pulp g	PLS g	Volume PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	Colour PLS	RES	Sample Analysis (Diluted Sample) Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Calculated sample concentration Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
	Reactor initial					0.00	0.00	0		#DIV/0!															#######	#######	#######	#######	#######	#######	#######	#######	
8:20	Disch 1	48.7	430.1	570.0	724	381.41	139.96	521	503.5	432	0.46	1.165						59200	180		590	7300	5600	146300	183	1	182	0	2	23	17	452	324
9:20	Disch 2	48.4	420.1	529.5	723	371.69	109.46	481	466.1	411	0.46	1.135						44200	130		490	5600	4200	117400	166	0	165	0	2	21	16	440	355
10:20	Disch 3	48.7	409.6	529.8	727	360.95	120.24	481	460.9	400	0.54	1.153						58400	190		620	6200	4700	134000	194	1	194	0	2	21	16	445	322
11:20	Disch 4	48.4	415.4	519.2	722	367.00	103.78	471	455.4	403	0.54	1.130						49900	120		460	4900	3900	115100	194	0	193	0	2	19	15	447	323
12:20	Disch 5	48.7	425.1	529.9	706	376.43	104.77	481	465.6	412	0.62	1.131						48300	140		420	5000	3800	114600	190	1	189	0	2	20	15	450	324
13:20	Disch 6	48.4	421.5	532.1	714	373.15	110.55	484	463.5	408	0.52	1.135						50300	180		460	4700	3700	119500	186	1	185	0	2	17	14	441	340
14:20	Disch 7	48.7	407.5	516.0	709	358.87	108.52	467	452.3	399	0.62	1.135						50900	200		400	4900	3700	121200	187	1	186	0	1	18	14	445	335
15:20	Disch 8	48.4	415.4	465.1	708	366.98	49.76	417	412.2	390	0.85	1.058						20700	130		160	1800	1400	54600	162	1	161	0	1	14	11	428	384
	HCl-initial								67.9	63.6		1.068						74															
	HCl-1								10.0	8.8		1.137						228															
	HCl-2								17.2	15.0		1.143						319															
	HCl-3								17.1	15.0		1.139						343															
	HCl-4								14.6	12.8		1.141						394															
	HCl-5								19.7	17.2		1.147						373															
	HCl-6								16.0	13.9		1.145						365															
	HCl-7								11.8	10.3		1.152						390															
	HCl-8								22.5	19.4		1.160						409															
Reactor solids wash									2211.0	2142.4		1.032						11500	160		98	1100	1000	26900									
15:30	Feed	18.5	104.2	136.5		85.65	32.33	117.98	115.8	104		1.117						45600	260		210	2300	1900	98800	146	1	145	0	1	7	6	317	523
15:30	Boiler	18.5	95.6	112.2		77.08	16.55	93.63	111.7	93		1.197						83300	150		330	4000	2900	172400	470	1	469	0	2	23	16	972	-483
15:30	Reactor	18.7	103.1	141.5		84.32	38.48	122.80	117.5	97		1.205						83400	530		710	7700	6000	178500	211	1	210	0	2	20	15	452	300
15:30	Feed	hot SG										1.449																					
15:30	Boiler	hot SG										1.629																					
15:30	Reactor	hot SG										1.835																					
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl									
	Disch 1-8												64.8	51.2	21.0%			67.1	0.04	0.036	<0.010	0.015	<0.005	1.1									
	Reactor												102.6	84.7	17.5%			67.0	0.01	<0.010	<0.010	0.011	0.006	1.4									

Project: 12323-001
Client: Starfield Resources

Test ID: Test 14

initial: 7:20
final: 15:20
Feed: 2mL/min
Reactor rpm: 1000
Reactor condensate: Drawn off

run time:	8.00	hours
avg feed flow:	2.91	g/min
	2.01	mL/min
average water addition	Nill	mL/min
average product flow	1.76	g/min
average product flow	0.96	mL/min
Reactor retention time:	12	hours
Test Turnovers:	0.7	
feed:	1395	g
product:	847	g pulp
	136	g solids

Preheater temp	82.3	℃
Boiler temp	139.6	℃
Reactor temp	180.1	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1395	204	1	203	0	1	10	9	442
Seed	83	55			0	0	0	0	2
Reactor - start	1285	290	0	290	0	3	32	25	668
H2O feed to reactor	0								
Discharge	847	156	1	155	0	1	17	13	377
Solids	136	91			0.03	0.01	0.02	0.01	1.75
Reactor - end	1285	271	2	270	0	2	25	20	581
Boiler Condensate	334								1
Reactor condensate	197	0.0							46
Scrubbers									0
Accumulation	0	-18	2	-20	0	-1	-7	-6	-87
IN:	1478	259	1	203	0	1	10	9	444
OUT:	1378	229	2	135	0	1	9	7	338
Accountability, OUT/IN	93	88	193	67	147	95	90	80	76
Metal Hydrolysed				67	0	0	1	2	
% Metal Hydrolysed				33	-47	5	10	20	
				18%					

Average HCl strength	27.3%
Maximum HCl strength	31.5%
mol of Fe hydrolysed	1.206
mol of Fe (precipitate)	0.642
mol of HCl produced	1.533

Project: 12323-001 **Date:** 23-Jun-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 15** *Hydrolysis Tests*
Purpose: **To determine the effect of a ferric-only feed solution in the hydrolysis.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$

Conditions:

Preheater:

Temperature:	80	℃				
Reflux Condensation:	yes	(back into preheater)				
prefill with:	"Feed Solution (feed to hydrolysis) - Test 15"					
volume:	3000	mL				
Nitrogen blanket:	TBD	mL/min N2				

Test Duration: 8 hr
required feed: 960 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	3.2	g/min
sg:	1.6	g/mL	**0.192**	kg/h
prefill with:	"Boiler - Test 15"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	℃	RPM:	**600**
feed flow:	as is		location impellor:	1
prefill with:	"Hydrolysis bath - Test 15"		impellor:	**pitch blade**
dynamic volume:	700	mL		
Water flow:	Nill	mL/min		
Reflux Condensation:	**No**			
target condensate flow:	TBD	mL/min		
estimated dischareg liquid flow:	TBD	mL/min		

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Nill - Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 23-Jun-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 15 *Hydrolysis Tests*
Purpose: Hydrolysis Scoping Test
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:	
Boiler RPM:	150
Hydrolysis Reactor RPM:	600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
																81.6g of hematite were added as seed.
7:45	0.00	82.7	9090.1		138.9	2.70	0.999	382.39	180.4			210.5	1039	146.2	1.066	HCl condensate has a lot of fumes, it's too concentrated.
8:45	1.00	81.4	8900.7	3.16	139.1	6.42	1.002	67.36	180.0			26.7	876	451.9	1.187	
9:45	2.00	81.9	8712.7	3.13	139.6	4.02	1.001	54.40	179.4			40.5	890	425.2	1.182	
10:45	3.00	81.6	8529.1	3.06	140.2	3.54	1.000	53.38	179.9			30.9	890	421.9	1.184	
11:45	4.00	83.1	8343.9	3.09	139.7	3.58	1.000	52.76	179.3			32.9	895	386.3	1.186	
12:45	5.00	81.1	8150.5	3.22	140.3	3.78	1.000	54.03	180.9			33.7	888	432.9	1.183	
13:45	6.00	79.3	7956.3	3.24	140.1	3.42	1.000	51.36	179.7			32.4	890	427.3	1.183	
14:45	7.00	81.4	7760.9	3.26	139.3	3.42	1.000	50.23	179.6			32.4	892	425.1	1.181	
15:48	8.05	83.7	7564.1	3.12	140.2	3.55	1.001	61.45	181.7			44.0	913	415.0	1.182	
									Scrubber 1			1027.85		13.4	1.005	
									Scrubber 2			994.70		0.6	0.999	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 15

Sampling Data

Sample Name Time	set #	Weight cup (tare) g	H2O+cup (gross) g	HCl+cup + discharge g	pulp ORP mV	Sample Composition Diluent g	Sample g	Diluted Weight pulp g	PLS g	Volume PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	Colour PLS	RES	Sample Analysis (Diluted Sample) Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Calculated sample concentration Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
	Reactor initial					0.00	0.00	0		#DIV/0!															#######	#######	#######	#######	#######	#######	#######	#######	
	Disch 1	48.1	422.3	490.6	689	374.14	68.31	442	425.0	391	0.52	1.087						36600						78800	209	0	209	0	0	0	0	451	340
	Disch 2	48.7	439.1	522.9	691	390.48	83.80	474	455.4	412	0.30	1.104						46000						92500	226	0	226	0	0	0	0	455	319
	Disch 3	48.1	410.6	491.6	695	362.46	80.98	443	424.6	383	0.32	1.109						44700						96800	211	0	211	0	0	0	0	458	331
	Disch 4	48.7	421.3	503.7	693	372.68	82.40	455	436.3	394	0.36	1.106						44700						95000	214	0	214	0	0	0	0	455	331
	Disch 5	48.1	429.4	518.1	692	381.30	88.71	470	448.2	404	0.33	1.111						46100						99300	210	0	210	0	0	0	0	452	339
	Disch 6	48.7	415.6	506.9	691	366.97	91.23	458	435.5	390	0.23	1.117						50400						106000	215	0	215	0	0	0	0	453	332
	Disch 7	48.1	406.9	503.4	689	358.81	96.49	455	433.0	385	0.20	1.125						51800						113400	207	0	207	0	0	0	0	452	341
	Disch 8	48.7	414.1	479.5	688	365.43	65.42	431	413.1	381	0.37	1.085						34500						78400	201	0	201	0	0	0	0	456	343
	HCl-initial								210.5	197.5		1.066						120															
	HCl-1								26.7	22.5		1.187						80															
	HCl-2								40.5	34.3		1.182						98															
	HCl-3								30.9	26.1		1.184						105															
	HCl-4								32.9	27.7		1.186						104															
	HCl-5								33.7	28.5		1.183						102															
	HCl-6								32.4	27.4		1.183						99															
	HCl-7								32.4	27.4		1.181						99															
	HCl-8								44.0	37.2		1.182						98															
Reactor solids wash																																
	Feed	18.6	81.2	112.1		62.58	30.92	93.50	92.2	81		1.132						54900						111000	145	0	145	0	0	0	0	292	563
	Boiler	18.5	95.9	129.5		77.39	33.66	111.05	109.3	92		1.186						82100						163000	225	0	225	0	0	0	0	446	329
	Reactor	18.7	89.0	129.6		70.28	40.54	110.82	103.0	83		1.242						108000						216600	221	0	221	0	0	0	0	443	336
	Feed	hot SG										1.424																					
	Boiler	hot SG										1.627																					
	Reactor	hot SG										1.883																					
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl									
	Disch 1-8												91.9	76.0	17.3%			67.1	0.03					1.0									
	Reactor												150.5	130.0	13.6%			65.1	0.04					1.3									

Project: 12323-001
Client: Starfield Resources

Test ID: Test 15

initial: 7:45
final: 15:48
Feed: 2mL/min
Reactor rpm: 1000
Reactor condensate: Drawn off

run time:	8.05	hours
avg feed flow:	3.16	g/min
	2.22	mL/min
average water addition	Nill	mL/min
average product flow	1.44	g/min
average product flow	0.77	mL/min
Reactor retention time:	15	hours
Test Turnovers:	0.5	
feed:	1526	g
product:	698	g pulp
	206	g solids

Preheater temp	81.8	℃
Boiler temp	139.7	℃
Reactor temp	180.1	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1526	221	0	221	0	0	0	0	446
Seed	88	59			0	0	0	0	2
Reactor - start	1318	290	0	290	0	0	0	0	552
H2O feed to reactor	0								
Discharge	698	139	0	139	0	0	0	0	298
Solids	206	136			0.00	0.00	0.00	0.00	2.49
Reactor - end	1318	291	0	291	0	0	0	0	584
Boiler Condensate	445								2
Reactor condensate	484	0.0							123
Scrubbers									14
Accumulation	0	2	0	2	0	0	0	0	33
IN:	1614	280	0	221	0	0	0	0	449
OUT:	1627	277	0	141	0	0	0	0	458
Accountability, OUT/IN	101	99	#DIV/0!	64	0	0	0	0	102
Metal Hydrolysed				80	0	0	0	0	
% Metal Hydrolysed				36	100	100	100	100	
				35%					

Average HCl strength	33.5%
Maximum HCl strength	38.1%
mol of Fe hydrolysed	1.427
mol of Fe (as hematite)	1.371
mol of HCl produced	3.484

Project: 12323-001 **Date:** 29-Jun-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 16** *Hydrolysis Tests*
Purpose: **To reproduce Neoferric results using SGS setup.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$

Conditions:

Preheater:

Temperature:	80	℃			
Reflux Condensation:	yes	(back into preheater)			
prefill with:	"Feed Solution (feed to hydrolysis) - Test 16"				
volume:	3000	mL			
Nitrogen blanket:	TBD	mL/min N2			

Test Duration: 8 hr
required feed: 960 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	3.2	g/min
sg:	1.6	g/mL	**0.192**	kg/h
prefill with:	"Boiler - Test 16"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	℃	RPM:	**600**
feed flow:	as is		location impellor:	1
prefill with:	"Hydrolysis bath - Test 16"		impellor:	**pitch blade**
dynamic volume:	700	mL		
Water flow:	Nill	mL/min		
Reflux Condensation:	**No**			
target condensate flow:	TBD	mL/min		
estimated dischareg liquid flow:	TBD	mL/min		

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Nill - Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 29-Jun-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: **Test 16** *Hydrolysis Tests*
Purpose: **Hydrolysis Scoping Test**
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:	
Boiler RPM:	150
Hydrolysis Reactor RPM:	600

Impellor: **Pitch-blade**

Test Data:

Time		**Preheater**		**Feed**	**Boiler**				**Hydrolysis Reactor**							**Observations**
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
8:05	0.00	80.8	9323.8		130.0	1.11	0.999	237.89	181.1	0.0		245.2	1059	125.6	1.056	
9:05	1.00	82.4	9130.2	3.23	140.1	4.27	1.000	103.79	179.7	22.2	0.37	64.0	863	449.3	1.194	
10:05	2.00	82.4	8957.6	2.88	139.2	3.19	1.000	46.58	179.8	45.0	0.38	73.6	887	414.2	1.182	
11:07	3.03	80.1	8758.7	3.21	139.6	3.38	1.001	47.20	180.1	69.1	0.39	74.2	895	405.5	1.182	
12:05	4.00	82.6	8572.4	3.21	139.7	3.00	1.001	44.60	180.3	92.4	0.40	69.3	885	394.5	1.181	
13:06	5.02	82.9	8387.8	3.03	139.2	2.83	1.000	46.40	180.8	115.4	0.38	78.0	870	407.8	1.180	
14:06	6.02	79.8	8190.3	3.29	140.2	2.99	1.000	47.18	178.8	138.8	0.39	75.7	881	407.7	1.177	
15:05	7.00	82.5	8009.4	3.07	140.2	2.72	1.000	45.97	180.1	161.2	0.38	62.2	883	406.1	1.178	
16:05	8.00	82.4	7830.3	2.99	138.5	2.63	1.001	40.39	178.8	183.9	0.38	84.3	879	364.6	1.162	
									Scrubber			1018.97		18.0	1.005	
												941.37		0.3	0.999	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 16

Sampling Data

Sample Name Time	set #	Weight cup (tare) g	H2O+cup (gross) g	HCl+cup + discharge g	pulp ORP mV	Sample Composition Diluent g	Sample g	Diluted Weight pulp g	PLS g	Volume PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	Colour PLS	RES	Sample Analysis (Diluted Sample) Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Calculated sample concentration Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
	Reactor initial					0.00	0.00	0		#DIV/0!																							
10:05	Disch 1	48.1	431.4	470.0	656	383.32	38.62	422	410.8	394	1.02	1.042						14900	234	85				42800	152	2	150	1	0	0	0	437	410
11:07	Disch 2	48.7	428.2	465.1	673	379.53	36.85	416	404.6	388	0.97	1.043						16700	345	208				40500	176	4	172	2	0	0	0	426	396
12:05	Disch 3	48.1	442.9	472.2	690	394.77	29.34	424	413.0	399	0.99	1.035						13200	294	232				33100	180	4	176	3	0	0	0	450	367
13:06	Disch 4	48.7	437.6	470.6	676	388.87	33.08	422	412.3	397	0.84	1.039						15000	340	346				36100	180	4	176	4	0	0	0	433	383
14:06	Disch 5	48.1	472.4	506.5	677	424.28	34.15	458	447.7	432	0.85	1.037						13900	365	471				33500	176	5	171	6	0	0	0	423	395
15:05	Disch 6	48.7	420.2	458.0	659	371.48	37.81	409	398.9	384	0.80	1.040						14900	417	590				41100	151	4	147	6	0	0	0	417	426
16:05	Disch 7	48.1	465.0	513.1	672	416.87	48.14	465	453.4	426	0.79	1.064						25400	646	1080				57000	225	6	219	10	0	0	0	505	261
	Disch 8					0.00	0.00	0		0		1.000																					
	HCl-initial								245.2	232.2		1.056						119															
	HCl-1								64.0	53.6		1.194						884															
	HCl-2								73.6	62.2		1.182						927															
	HCl-3								74.2	62.7		1.182						762															
	HCl-4								69.3	58.7		1.181						798															
	HCl-5								78.0	66.1		1.180						738															
	HCl-6								75.7	64.3		1.177						725															
	HCl-7								62.2	52.8		1.178						730															
	HCl-8								84.3	72.6		1.162						688															
Reactor solids wash																																	
	Feed	18.7	97.4	131.0		78.71	33.56	112.27	110.8	99		1.123						52700	756	2440				103800	155	2	153	7	0	0	0	305	533
	Boiler	18.8	98.9	136.5		80.18	37.59	117.77	116.5	97		1.195						87100	1010	4050				170700	226	3	223	11	0	0	0	443	321
	Reactor	18.5	83.6	131.2		65.12	47.58	112.70	96.0	76		1.262						121000	2170	4170				239100	193	3	190	7	0	0	0	382	418
	Feed	hot SG										1.472																					
	Boiler	hot SG										1.632																					
	Reactor	hot SG										1.998																					
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl									
	Disch 1-8												41.7	36.6	12.4%			66.6	0.04	0.023				0.9									
	Reactor												279.0	245.8	11.9%			68.4	0.03					1.2									

Project: 12323-001
Client: Starfield Resources

Test ID: Test 16

initial: 8:05
final: 16:05
Feed: 2mL/min
Reactor rpm: 1000
Reactor condensate: Drawn off

run time:	8.00	hours
avg feed flow:	3.11	g/min
	2.11	mL/min
average water addition	0.38	mL/min
average product flow	0.64	g/min
average product flow	0.32	mL/min
Reactor retention time:	37	hours
Test Turnovers:	0.2	
feed:	1494	g
product:	306	g pulp
	282	g solids

Preheater temp	81.8	℃
Boiler temp	138.5	℃
Reactor temp	179.9	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1494	231	3	228	11	0	0	0	456
Seed	88	59			0	0	0	0	2
Reactor - start	1399	235	0	235	0	0	0	0	447
H2O feed to reactor	184								
Discharge	306	55	1	54	2	0	0	0	133
Solids	282	192			0.01	0.00	0.00	0.00	3.34
Reactor - end	1399	270	5	266	9	0	0	0	534
Boiler Condensate	422								1
Reactor condensate	826	0.3							222
Scrubbers									18
Accumulation	0	36	5	31	9	0	0	0	88
IN:	1765	290	3	228	11	0	0	0	458
OUT:	1554	284	6	85	11	0	0	0	448
Accountability, OUT/IN	88	98	183	37	101	0	0	0	98
Metal Hydrolysed				143	0	0	0	0	
% Metal Hydrolysed				63	-1	100	100	100	
				59%					

Average HCl strength	32.2%
Maximum HCl strength	37.6%
mol of Fe hydrolysed	2.560
mol of Fe (as hematite)	2.391
mol of HCl produced	6.819

Project: 12323-001 **Date:** 05-Jul-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 17** *Hydrolysis Tests*
Purpose: **To determine the effect of low copper feed solution in the hydrolysis**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$

Conditions:

Preheater:

Temperature:	80	℃			Test Duration:	8	hr
Reflux Condensation:	yes	(back into preheater)			required feed:	960	mL
prefill with:	"Feed Solution (feed to hydrolysis) - Test 17"				estimated boiler condensate:		mL
volume:	3000	mL			estimated reactor condensate:		mL
Nitrogen blanket:	TBD	mL/min N2			effluent volume:	TBD	mL

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	3.2	g/min
sg:	1.6	g/mL	**0.192**	kg/h
prefill with:	"Boiler - Test 17"			
dynamic volume:	No	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	℃	RPM:	**600**
feed flow:	as is		location impellor:	1
prefill with:	"Hydrolysis bath - Test 10"		impellor:	**pitch blade**
dynamic volume:	700	mL		
Water flow:	0.40	mL/min		
Reflux Condensation:	**No**			
target condensate flow:	TBD	mL/min		
estimated dischareg liquid flow:	TBD	mL/min		

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Nill - Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 05-Jul-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 17 *Hydrolysis Tests*
Purpose: Hydrolysis Scoping Test
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
7:53	0.00	80.1	9353.2		140.9	3.21	1.000	206.13	179.1	0.0		281.3	1050	82.6	1.038	
8:53	1.00	80.3	9131.2	3.70	139.0	2.85	1.000	45.95	180.5	24.2	0.40	56.1	864	402.7	1.173	
9:53	2.00	82.8	8917.8	3.56	140.3	3.28	1.000	51.12	179.2	46.6	0.37	63.1	902	367.3	1.161	
10:53	3.00	82.4	8690.5	3.79	139.5	2.60	1.000	47.40	180.6	70.1	0.39	66.7	913	359.4	1.156	
11:53	4.00	79.8	8470.9	3.66	139.7	2.85	1.000	50.63	179.8	93.8	0.39	64.6	914	358.3	1.153	
12:53	5.00	83.0	8252.4	3.64	138.9	2.87	1.001	46.75	180.6	117.1	0.39	69.0	930	345.2	1.150	
13:53	6.00	82.1	8033.4	3.65	139.8	2.74	1.000	52.02	179.8	140.3	0.39	59.5	914	337.2	1.148	
14:53	7.00	79.6	7809.0	3.74	140.3	2.57	1.000	46.60	179.3	163.9	0.39	69.5	910	346.0	1.151	
15:55	8.03	83.0	7580.6	3.68	140.2	2.81	1.000	50.24	180.2	187.1	0.37	75.1	914	355.1	1.153	
									Scrubber 1			938.70		2.9	1.001	
									Scrubber 2			1007.73		0.3	1.000	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 17

Sampling Data

Sample Name Time	set #	Weight cup (tare) g	H2O+cup (gross) g	HCl+cup + discharge g	pulp ORP mV	Sample Composition Diluent g	Sample g	Diluted Weight pulp g	PLS g	Volume PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	Colour PLS	RES	Sample Analysis (Diluted Sample) Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Calculated sample concentration Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
	Reactor initial					0.00	0.00	0		#DIV/0!															######	######	######	######	######	######	######	######	
8:53	Disch 1	48.1	428.5	462.2	677	380.44	33.65	414	402.7	385	1.13	1.045						16000	350	1200	71	730	580	37500	183	4	179	14	1	8	7	429	358
9:53	Disch 2	48.7	411.4	508.1	682	362.68	96.73	459	442.0	394	0.74	1.123						45900	1000	3000	230	2300	1900	107400	187	4	183	12	1	9	8	437	346
10:53	Disch 3	48.1	459.0	570.9	683	410.91	111.88	523	498.0	442	0.73	1.127						46700	1100	2900	220	2200	1800	108800	184	4	180	11	1	9	7	430	358
11:53	Disch 4	48.7	444.9	560.5	687	396.18	115.67	512	490.4	430	0.75	1.141						50800	1200	3300	250	2500	2100	120100	189	4	184	12	1	9	8	446	335
12:53	Disch 5	48.1	433.6	577.8	694	385.47	144.24	530	499.4	428	0.47	1.167						59800	1400	4000	320	3200	2600	145300	177	4	173	12	1	9	8	431	362
13:53	Disch 6	48.7	429.0	538.2	690	380.34	109.21	490	467.6	412	0.59	1.135						48900	1200	3600	270	2600	2200	116800	184	5	180	14	1	10	8	441	342
14:53	Disch 7	48.1	446.8	560.6	686	398.72	113.75	512	490.1	432	0.61	1.134						47800	1200	3900	280	2700	2200	115600	182	5	177	15	1	10	8	439	345
15:55	Disch 8	48.7	437.0	534.2	664	388.33	97.21	486	466.8	418	0.67	1.117						41800	1100	3100	230	2100	1900	102400	180	5	175	13	1	9	8	440	349
	HCl-initial								281.3	271.0		1.038						53														
	HCl-1								56.1	47.9		1.173						611														
	HCl-2								63.1	54.3		1.161						553														
	HCl-3								66.7	57.7		1.156						550														
	HCl-4								64.6	56.0		1.153						542														
	HCl-5								69.0	60.0		1.150						540														
	HCl-6								59.5	51.8		1.148						713														
	HCl-7								69.5	60.4		1.151						517														
	HCl-8								75.1	65.2		1.153						508														
Reactor solids wash																																
15:00	Feed	18.7	98.9	131.3		80.21	32.42	112.63	111.4	99		1.130						52000	740	2200	210	2000	1600	108100	158	2	156	7	1	6	5	329	495
15:00	Boiler	18.4	92.8	127.1		74.34	34.31	108.65	107.3	90		1.198						77500	980	3600	340	3200	2700	171600	202	3	200	9	1	8	7	448	324
15:00	Reactor	18.6	99.9	142.7		81.25	42.81	124.06	114.8	94		1.222						89100	1670	4300	480	4800	3900	197000	196	4	192	9	1	11	9	432	343
	Feed	hot SG										1.504																				
	Boiler	hot SG										1.586																				
	Reactor	hot SG										1.881																				
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl								
	Disch 1-8												103.6	87.0	16.0%			65.6	0.13	0.017	<0.01	<0.01	<0.005	2.1								
	Reactor												175.9	150.5	14.4%			65.9	0.1	0.023	<0.01	<0.01	<0.005	1.5								

Project: 12323-001
Client: Starfield Resources

Test ID: Test 17

initial: 7:53
final: 15:55
Feed: 2mL/min
Reactor rpm: 600
Reactor condensate: Drawn off

run time:	8.03	hours
avg feed flow:	3.68	g/min
	2.44	mL/min
average water addition	0.39	mL/min
average product flow	1.79	g/min
average product flow	0.95	mL/min
Reactor retention time:	12	hours
Test Turnovers:	0.7	
feed:	1773	g
product:	865	g pulp
	238	g solids

Preheater temp	81	℃
Boiler temp	140	℃
Reactor temp	180	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1773	280	4	276	12	1	11	9	583
Seed	92	62			0	0	0	0	2
Reactor - start	1317	233	3	230	11	1	10	8	489
H2O feed to reactor	187								
Discharge	865	141	3	138	10	1	7	6	335
Solids	238	156			0.05	0.02	0.02	0.01	4.09
Reactor - end	1317	258	5	253	12	1	14	11	569
Boiler Condensate	391								1
Reactor condensate	805	0.2							180
Scrubbers									3
Accumulation	0	25	2	23	2	0	4	3	81
IN:	2052	342	4	276	12	1	11	9	585
OUT:	2061	323	5	161	11	1	11	10	600
Accountability, OUT/IN	100	94	131	58	95	110	105	110	103
Metal Hydrolysed				115	1	0	-1	-1	
% Metal Hydrolysed				42	5	-10	-5	-10	
				34%					

Average HCl strength	31.1%
Maximum HCl strength	34.3%
mol of Fe hydrolysed	2.061
mol of Fe (as hematite)	1.694
mol of HCl produced	5.176

Project: 12323-001 **Date:** 13-Jul-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricaudy

Test ID: **Test 18** *Hydrolysis Tests*
Purpose: **To determine the effect of temperature in the hydrolysis.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"Feed Solution (feed to hydrolysis) - Test 18"	
volume:	3000	mL
Nitrogen blanket:	TBD	mL/min N2

Test Duration: 8 hr
required feed: 960 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	3.2	g/min
sg:	1.6	g/mL	**0.192**	kg/h
prefill with:	"Boiler - Test 18"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**190**	℃
feed flow:	as is	
prefill with:	"Hydrolysis bath - Test 18"	
dynamic volume:	700	mL
Water flow:	0.40	mL/min
Reflux Condensation:	**No**	
target condensate flow:	TBD	mL/min
estimated dischareg liquid flow:	TBD	mL/min

RPM: **600**
location impellor: 1
impellor: **pitch blade**

Sampling Requirements & Instructions:

Partial Filtrates: Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate: Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate: Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Nill - Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 13-Jul-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 18 *Hydrolysis Tests*
Purpose: Hydrolysis Scoping Test
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
																87.2g of hematite was added as seed.
7:25	0.00	82.9	9308.0		140.0	4.21	1.001	232.58	189.3	0.0		321.8	1037	280.5	1.120	HCl condensatehad a lot of fumes.
8:25	1.00	80.3	9083.4	3.74	140.3	4.28	1.001	50.60	189.3	23.5	0.39	77.3	860	446.6	1.199	From 9:25 to 12:25 the discharge was too smal to take a
9:25	2.00	83.3	8868.5	3.58	139.8	2.29	1.000	44.09	189.8	46.5	0.38	80.8	860	471.1	1.199	sample from it.
10:25	3.00	81.6	8637.2	3.85	139.9	1.93	1.000	48.61	189.0	69.9	0.39	71.2	798	450.6	1.197	Due to some foaming effect at the top of the reactor and the
11:25	4.00	82.3	8421.6	3.59	139.7	2.37	1.000	48.00	189.9	93.2	0.39	70.2	775	470.3	1.199	small amount of discharge produced, reactor temperature
12:25	5.00	82.1	8215.9	3.43	140.3	2.19	1.000	45.36	189.4	116.5	0.39	67.3	719	466.4	1.198	was decreased to 185C to make the solution less viscous
13:26	6.02	80.1	7992.8	3.66	140.0	2.26	1.000	44.72	184.9	130.5	0.23	70.1	713	461.6	1.190	There was not too much solids in the discharge, 95% of the
14:26	7.02	82.6	7778.4	3.57	139.5	2.01	1.000	45.07	185.5	163.3	0.55	77.4	690	449.0	1.194	solids remained inside the reactor untill the end of the test
15:25	8.00	82.4	7574.3	3.46	140.6	2.09	1.000	46.62	184.8	185.9	0.38	83.7	379	447.7	1.192	
									Scrubber 1			1016.30		86.2	1.041	
									Scrubber 2			979.70		0.5	1.000	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 18

Sampling Data

Sample Name Time	set #	Weight cup (tare) g	H2O+cup (gross) g	HCl+cup + discharge g	pulp ORP mV	Sample Composition Diluent g	Sample g	Diluted Weight pulp g	PLS g	Volume PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	Colour PLS	RES	Sample Analysis (Diluted Sample) Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Calculated sample concentration Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
	Reactor initial					0.00	0.00	0		#DIV/0!															#######	#######	#######	#######	#######	#######	#######	#######	
8:25	Disch 1	48.6	451.9	490.3	1143	403.28	38.34	442	432.1	422	0.91	1.025						2200	5	160	12	100	77	47200	24	0	24	2	0	1	1	519	453
9:25	Disch 2	48.1	455.2	488.1	660	407.09	32.86	440	432.1	422	0.87	1.024						3400	27	250	23	200	150	40200	44	0	43	3	0	3	2	516	432
13:25	Disch 3	48.6	410.5	559.4	562	361.90	148.86	511	485.9	437	0.61	1.113						25000	590	2100	200	1700	1400	161900	73	2	72	6	1	5	4	475	436
14:25	Disch 4	48.1	452.7	520.8	663	404.56	68.13	473	455.9	420	0.90	1.085						25000	760	9600	240	2000	1700	72300	154	5	149	59	1	12	10	446	316
15:25	Disch 5	48.6	431.0	447.6	632	382.40	16.58	399	394.2	390	1.26	1.012						2300	64	280	17	160	120	17800	54	2	53	7	0	4	3	418	514
	Disch 6					0.00	0.00	0		0		1.000																					
	Disch 7					0.00	0.00	0		0		1.000																					
	Disch 8					0.00	0.00	0		0		1.000																					
	HCl-initial								321.8	287.3		1.120						95															
	HCl-1								77.3	64.5		1.199						863															
	HCl-2								80.8	67.3		1.199						928															
	HCl-3								71.2	59.4		1.197						834															
	HCl-4								70.2	58.5		1.199						795															
	HCl-5								67.3	56.1		1.198						786															
	HCl-6								70.1	58.9		1.190						601															
	HCl-7								77.4	64.8		1.194						473															
	HCl-8								83.7	70.2		1.192						515															
Reactor solids wash									1746.5	1508.2		1.158						55000	2070	12000	530	4600	4100	133200									
15:00	Feed	18.0	92.2	127.6		74.22	35.39	109.61	108.2	95		1.140						61000	710	2800	250	2500	2000	117400	164	2	162	8	1	7	5	315	501
15:00	Boiler	18.3	98.0	136.2		79.74	38.20	117.94	116.7	98		1.195						86000	1120	3800	370	3500	2900	174200	220	3	217	10	1	9	7	445	308
15:00	Reactor	18.4	99.1	145.2		80.68	46.14	126.82	108.9	91		1.199						82000	2490	5100	760	7000	5600	178900	161	5	157	10	1	14	11	352	450
	Feed	hot SG										1.487																					
	Boiler	hot SG										1.587																					
	Reactor	hot SG										2.156																					
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl									
	Disch 1-8												28.4	22.6	20.3%			68.4	0.03	0.023	0.01	0.01	0.005	1.0									
	Reactor												425.4	393.3	7.6%			67.6	0.01	0.016	0.01	0.01	0.005	1.3									

Project: 12323-001
Client: Starfield Resources

Test ID: Test 18

initial: 7:25
final: 15:25
Feed: 2mL/min
Reactor rpm: 600
Reactor condensate: Drawn off

run time:	8.00	hours
avg feed flow:	3.61	g/min
	2.43	mL/min
average water addition	0.39	mL/min
average product flow	0.73	g/min
average product flow	0.34	mL/min
Reactor retention time:	34	hours
Test Turnovers:	0.2	
feed:	1734	g
product:	351	g pulp
	416	g solids

Preheater temp	82	℃
Boiler temp	140	℃
Reactor temp	188	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1734	284	3	280	13	1	12	9	546
Seed	87	58			0	0	0	0	3
Reactor - start	1509	233	3	230	11	1	10	8	489
H2O feed to reactor	186								
Discharge	351	32	1	31	6	0	2	2	161
Solids	416	281			0.07	0.04	0.04	0.02	5.48
Reactor - end	1509	244	7	236	15	2	21	17	532
Boiler Condensate	373								1
Reactor condensate	920	0.3							301
Scrubbers									82
Accumulation	0	11	4	7	4	1	11	9	43
IN:	2007	341	3	280	13	1	12	9	549
OUT:	1644	325	5	38	10	2	13	11	511
Accountability, OUT/IN	82	95	162	13	77	144	115	117	93
Metal Hydrolysed				243	3	-1	-2	-2	
% Metal Hydrolysed				87	23	-44	-15	-17	
				80%					

Average HCl strength	38.3%
Maximum HCl strength	39.3%
mol of Fe hydrolysed	4.343
mol of Fe (as hematite)	4.007
mol of HCl produced	10.830

Project: 12323-001 **Date:** 15-Jul-10
Client: Starfield Resources **Technologist:** Hugh Given
Ernesto Bourricady

Test ID: **Test 19** *Hydrolysis Tests*
Purpose: **To determine the effect of ferrous in the hydrolysis**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCL vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts

Chemical Equation: $2FeCl_3 + 3H_2O = Fe_2O_3 + 6HCl$

Conditions:

Preheater:

Temperature:	80	℃			
Reflux Condensation:	yes	(back into preheater)			
prefill with:	"Feed Solution (feed to hydrolysis) - Test 19"				
volume:	3000	mL			
Nitrogen blanket:	TBD	mL/min N2			

Test Duration: 8 hr
required feed: 960 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	3.2	g/min
sg:	1.6	g/mL	**0.192**	kg/h
prefill with:	"Boiler - Test 19"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	℃	RPM:	**600**
feed flow:	as is		location impellor:	1
prefill with:	"Hydrolysis bath - Test 19"		impellor:	**pitch blade**
dynamic volume:	700	mL		
Water flow:	0.40	mL/min		
Reflux Condensation:	**No**			
target condensate flow:	TBD	mL/min		
estimated dischareg liquid flow:	TBD	mL/min		

Sampling Requirements & Instructions:

Partial Filtrates:	Fe, Fe(II), Cu, Co, Ni, Mg
Final filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg
Precipitate:	Fe, Cu, Co, Ni Mg, Cl

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 120C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Nill - Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted separately or combined (dependednt on availability)
    c. Identify samples as "test"-PLS1 or "test"-ppt1 to note PLS and precipitate collected in time period 1
11. Continue for total test duration of **8** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Date: 15-Jul-10
Technologist: Hugh Given
Ernesto Bourricaudy

Test ID: Test 19 *Hydrolysis Tests*
Purpose: Hydrolysis Scoping Test
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600

Impellor: **Pitch-blade**

Test Data:

Time		Preheater		Feed	Boiler				Hydrolysis Reactor							Observations
						Condensate			Bath	H_2O		Condensate				(continue on other page if requried)
(24 h)	(h) elapsed	Temp ℃	Weight g	Flow g/min	Temp ℃	FAT g/L HCl	Density g/mL	Weight g	Temp ℃	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	
																84.99 g of hematite as seed was added
9:10	0.00	82.5	9330.4		138.9	1.71	0.999	352.60	180.9	0.0		222.2	739	63.9	1.030	Not enough discharge after 1 hour, the level in the reactor
10:10	1.00	79.9	9095.7	3.91	138.8	3.82	1.001	70.30	179.4	23.7	0.40	59.1	427	324.4	1.143	was lower than usual.
11:10	2.00	82.6	8812.8	4.72	139.3	2.67	1.000	69.00	180.8	47.7	0.40	61.3	420	288.7	1.127	At the beginning the solids were red-ish and the moisture
12:10	3.00	79.8	8588.3	3.74	140.1	2.81	1.000	64.70	180.8	71.3	0.39	56.6	426	299.9	1.132	content was high (visual determination), but at the end the
13:10	4.00	83.1	8356.6	3.86	139.6	2.83	1.000	59.40	181.1	94.7	0.39	57.2	433	259.3	1.115	solids were darker and less humid (like the tests before).
14:10	5.00	80.1	8103.7	4.22	139.6	2.99	1.001	61.70	180.5	118.2	0.39	55.5	478	248.4	1.112	
15:10	6.00	82.1	7836.2	4.46	139.4	2.88	1.001	71.50	180.8	141.3	0.39	69.4	442	259.9	1.115	
							1.000								1.000	
							1.000								1.000	
									Scrubber 1			1017.80		1.1	1.000	
									Scrubber 2			960.10		0.1	1.000	
Totals/Avg:																

Project: 12323-001
Client: Starfield Resources

Test ID: Test 19

Sampling Data

Sample Name Time	set #	Weight cup (tare) g	H2O+cup (gross) g	HCl+cup + discharge g	pulp ORP mV	Sample Composition Diluent g	Sample g	Diluted Weight pulp g	PLS g	Volume PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H2O	Colour PLS	RES	Sample Analysis (Diluted Sample) Fe mg/L	Fe(II) mg/L	Cu mg/L	Co mg/L	Ni mg/L	Mg mg/L	Cl mg/L	Calculated sample concentration Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Cl g/kg	H2O g/kg
	Reactor initial					0.00	0.00	0		#DIV/0!															######	######	######	######	######	######	######	######	
11:10	Disch 1	48.7	448.9	490.1	586	400.20	41.20	441	432.3	414	0.63	1.045						16000	2860	1100	90	770	590	39900	161	29	132	11	1	8	6	401	413
12:10	Disch 2	48.1	475.5	525.4	607	427.40	49.90	477	465.9	439	0.77	1.062						24000	4410	1500	140	1200	940	50300	211	39	172	13	1	11	8	442	314
13:10	Disch 3	48.7	461.4	593.9	608	412.70	132.50	545	519.7	449	0.35	1.158						62000	12160	4200	380	3300	2600	130300	210	41	169	14	1	11	9	441	313
14:10	Disch 4	48.1	420.1	496.1	610	372.00	76.00	448	435.4	393	0.61	1.109						43000	7750	2200	250	2200	1800	88300	222	40	182	11	1	11	9	456	288
15:10	Disch 5	48.7	447.7	561.9	630	399.00	114.20	513	489.9	430	0.54	1.140						56000	10920	4300	330	2900	2300	112900	211	41	170	16	1	11	9	425	327
	Disch 6					0.00	0.00	0		0		1.000																				
	Disch 7					0.00	0.00	0		0		1.000																				
	Disch 8					0.00	0.00	0		0		1.000																				
	HCl-initial								222.2	215.7		1.030																				
	HCl-1								59.1	51.7		1.143																				
	HCl-2								61.3	54.4		1.127																				
	HCl-3								56.6	50.0		1.132																				
	HCl-4								57.2	51.3		1.115																				
	HCl-5								55.5	49.9		1.112																				
	HCl-6								69.4	62.2		1.115																				
	HCl-7								0.0	0.0		1.000																				
	HCl-8								0.0	0.0		1.000																				
Reactor solids wash									1774.6	1600.2		1.109						42000	9170	5600	290	2400	1800	88300									
	Feed	18.3	99.6	129.0		81.30	29.40	110.70	109.4	98		1.115						45000	7520	2200	210	2000	1600	91900	150	25	125	7	1	7	5	307	523
	Boiler	18.4	108.0	142.5		89.60	34.50	124.10	122.1	104		1.179						73000	12050	3400	350	3200	2500	144900	219	36	183	10	1	10	8	435	318
	Reactor	18.1	108.2	147.7		90.10	39.50	129.60	121.9	101		1.210						75000	16410	4800	500	4400	3400	171000	191	42	149	12	1	11	9	436	339
	Feed	hot SG										1.502																				
	Boiler	hot SG										1.586																				
	Reactor	hot SG										1.845																				
Solids assays																		%Fe	%Fe(II)	%Cu	%Co	%Ni	%Mg	%Cl								
	Disch 1-8												49.6	40.8	17.7%			67.5	0.03	0.024	0.01	0.01	<0.005	1.0								
	Reactor												158.1	128.4	18.8%			68.7	0.03	0.018	0.01	0.01	<0.005	1.0								

Project: 12323-001
Client: Starfield Resources

Test ID: Test 19

initial: 9:10
final: 15:10
Feed: 2mL/min
Reactor rpm: 600
Reactor condensate: Drawn off

run time:	6.00	hours
avg feed flow:	4.15	g/min
	2.76	mL/min
average water addition	0.39	mL/min
average product flow	1.26	g/min
average product flow	0.68	mL/min
Reactor retention time:	17	hours
Test Turnovers:	0.4	
feed:	1494	g
product:	453	g pulp
	169	g solids

Preheater temp	81	℃
Boiler temp	139	℃
Reactor temp	181	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Cl
					grams				
Feed	1494	224	37	187	11	1	10	8	458
Seed	85	57			0	0	0	0	2
Reactor - start	1107	233	42	191	11	1	10	8	464
H2O feed to reactor	141								
Discharge	453	93	18	76	6	1	5	4	197
Solids	169	116			0.03	0.02	0.02	0.01	1.73
Reactor - end	1292	247	54	193	16	2	14	11	563
Boiler Condensate	397								1
Reactor condensate	581	0.0							100
Scrubbers									1
Accumulation	185	14	12	2	5	1	5	3	99
IN:	1720	281	37	187	11	1	10	8	461
OUT:	1616	224	30	78	11	1	10	7	400
Accountability, OUT/IN	94	80	80	42	102	126	96	91	87
Metal Hydrolysed				109	0	0	0	1	
% Metal Hydrolysed				58	-2	-26	4	9	
				31%					

Average HCl strength	25.6%
Maximum HCl strength	28.4%
mol of Fe hydrolysed	1.953
mol of Fe (as hematite)	1.053
mol of HCl produced	2.896

Appendix D - Hydrolysis Long Run
Test details

Project: 12323-001
Client: Starfield Resources

Date: Aug 3rd, 2010
Technologist: Hugh Given (night)
Ernesto Bourricaudy (day)
Andrew Ferguson (afternoon)

Test ID: PP2 *Hydrolysis Tests*
Purpose: **To test different conditions in a hydrolysis long run.**

Health Safety:

Main Hazards:
- concentrated brine liquors
- HCl vapour
- concentrated HCl
- hot solutions and hot surfaces

PPE and Precautions:
- operate test with sash closed
- if work access is requried to the test area (open sash), a face shield is required
- standard PPE (safety glasses, lab coat, gloves)
- Use of heat resistant gloves when handling "hot" equipment or parts
- review MSDS of HCl

Chemical Equation: **2FeCl3 + 3H2O = Fe2O3 + 6HCl**

Conditions:

Preheater:

Temperature:	80	℃
Reflux Condensation:	yes	(back into preheater)
prefill with:	"Feed Solution (feed to hydrolysis) - Test PP2"	
volume:	3000	mL
Nitrogen blanket:	TBD	mL/min N2

Test Duration: 80 hr
required feed: 9600 mL
estimated boiler condensate: mL
estimated reactor condensate: mL
effluent volume: TBD mL

Boiler:

Temperature:	140	℃		
feed flow:	2	mL/min or:	3.2	g/min
sg:	1.6	g/mL	**0.192**	kg/h
prefill with:	"Boiler - Test PP2"			
dynamic volume:	700	mL		
Reflux Condensation:	**No**	(use "H" type fitting to draw off condensate)		
target condensate flow:	as req.	mL/min		

Hydrolysis Reactor:

Temperature:	**180**	℃
feed flow:	as is	
prefill with:	"Hydrolysis bath - Test PP2"	
dynamic volume:	700	mL
Water flow:	**variable**	mL/min
Reflux Condensation:	**No - drawn off (concentrated HCl)**	
target condensate flow:	TBD	mL/min
estimated dischareg liquid flow:	TBD	mL/min

RPM: **600**
location impellor: 1
impellor: **pitch blade**

Sampling Requirements & Instructions:

Reactor D/C Filtrate:	Fe, Fe(II), Cu, Co, Ni, Mg	every hour
Reactor D/C PPT:	Fe, Cl	combine last 8 hours of each conditions
HCl Condensate:	(Fe - see note)	take every hour - weigh and titrate. Determine after campaign which samples to submit
Final Precipitate:	Fe, Cu, Co, Ni Mg, Cl	at end of run

Instructions - Prefill:

1. Fill preheater, boiler and hydrolysis bath with pre-weighed chemicals and water. Dissolve all solids. Beware with addition of anhydrous salts - use faceshield. Data on synthetic composition is included in "Starfield solution Makeup instructions - hydrolysis"
2. Once all salts are added, commence heat up: preheat to 80C, boiler to 140C.
3. Feed to boiler can be started once all vessels have reached target temperature. Turn on feed pump. Record weight in preheater and continue to monitor the mass rate. Make adjustments as required.
4. Record when boiler vessel overflows.
5. Start adding the water to the reactor.
6. Collect boiler condensate at target flowrates in separate containers. Record its weight, FAT and SG.
7. After 2 hours increase the boiler temperature 10C and noted any changes in the boiler condensate (FAT)
8. Repeat step 7 until boiler condensate reach 1mol/L of HCl (36.5g/L).
9. Collect 1 hours worth of reactor condensate.
   a. Identify samples as "test"-HCl1 to note HCl collected in time period 1
   b. Titrate for HCl.
   c. Set aside in fumehood. Samples may be submitted for Fe after completion fo the campaign
10. Collect reactor discharge pulp in water. The following dilution is proposed:
    a. Collect 1 hours worth of discharge into 400.0 mL of water of known weight.
    b. Weigh diluted sample. Allow solids to settle - submit sample of decant filtrate for analysis. Determine sg of filtrate. Filter solids (if any). Solids will be submitted every condition.
    c. Identify samples as "test"-PLSx or "test"-ppty-x to note PLS and precipitate collected in time period x
11. Continue for total test duration of **80** hours
12. Recover solids from reactor at end of test. Take a sample for Malvern. Repulp wash with 10g/L HCl solution and determine wet and dry weight.
13. Submit a sample of the wash liquor.

Project: 12323-001
Client: Starfield Resources

Start Date: Aug 3rd, 2010
Technologist: Hugh Given (night)
Ernesto Bourricaudy (day)
Andrew Ferguson (afternoon)

Test ID: PP2 *Hydrolysis Tests*
Purpose: **Hydrolysis Long run**
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
hot pulps (180C)

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600
Impellor: **Pitch-blade**

Test Data:

Date-Time			Preheater			Feed	Boiler						Hydrolysis Reactor								Observations
									Condensate				Bath		DI H_2O		Condensate				(hard copy - continue on other page if requried)
(24 h)	(h) elapsed	Condition	Temp ℃	Weight g	Hot s.g. g/mL	Flow g/min	Temp ℃	Hot s.g. g/mL	FAT g/L HCl	Density g/mL	Weight g	g/min	Temp ℃	Hot s.g. g/mL	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	(e-copy - write down comment on appropriate row - link event to time. If more than 1 row is required, adjust row height and format as wrap text)
			1/hr	1/hr	1/4hr		1/hr	1/4hr	1/hr	1/hr	1/hr		1/hr	1/4hr	1/hr	1/hr	1/hr	1/hr	1/hr	1/hr	
8/3/10 10:40	0	1	81.9	9048.0			140.0				378.80		178.0		693.6		314.0	608	69.4	1.033	90g of hematite were added as seed. (Test 18, dry)
8/3/10 11:40	1		78.0	8946.2		1.70	139.9				51.20	0.85	178.8		666.6	0.45	52.8	654	321.8	1.142	Some foaming inside the reactor
8/3/10 12:40	2		82.4	8801.1		2.42	140.3				50.40	0.84	180.6		638.9	0.46	62.0	647	290.0	1.131	
8/3/10 13:30	3		82.3	8650.1		3.02	140.0				43.40	0.72	178.1		615.1	0.48	55.4	648	287.8	1.131	
8/3/10 14:30	4		80.9	8516.4		2.23	138.9				51.20	0.85	80.0		586.8	0.47	58.8	642	292.8	1.130	
8/3/10 15:33	5		82.9	8376.0		2.23	142.4				17.90	0.30	179.0		557.0	0.47	50.0	645	290.3	1.130	14:45/15:15 Boiler downcomer got plugged. It was necessary to stop the feed to solve the problem.
8/3/10 16:30	6		81.2	8208.8		2.93	139.4				34.20	0.57	181.4		531.5	0.45	49.1	652	304.8	1.135	
8/3/10 17:30	7		80.5	8022.2		3.11	139.1				51.90	0.87	180.6	1.882	504	0.46	64.20	652	262.1	1.118	
8/3/10 18:30	8		81.2	7871.4		2.51	140.2				50.10	0.84	181.2		476	0.47	62.40	653	258.0	1.115	
8/3/10 19:30	9		80.9	7700.8		3.50	139.1				51.20	0.85	179.2		448	0.47	65.30	651	244.0	1.113	
8/3/10 20:30	10		81.5	7506.8		3.23	139.7		1.95	1.001	50.30	0.84	180.4		418	0.49	64.50	656.0	248.5	1.112	
8/3/10 21:30	11		82.3	7368.4		2.31	140.0				52.10	0.87	179.6		391	0.46	57.30	650.0	245.1	1.108	
8/3/10 22:30	12		82.1	7167.2		3.35	140.1				51.10	0.85	178.1		363	0.47	63.20	650	236.7	1.106	
8/3/10 23:30	13		79.3	7019.1		2.47	139.6				46.60	0.78	180.0	1.906	332	0.52	61.10	648	239.2	1.108	
8/4/10 0:00	13.3	2		*6940*		2.62									*312*	0.64					new H20 flow rate
8/4/10 0:30	14		83.0	6861.7		2.62	140.2				48.20	0.80	178.4		293	0.64	65.50	644	229.1	1.103	No disch sample keep till next hour
8/4/10 1:08	14			6668.1		4.00															
8/4/10 1:08	14			10916.3																	new feed
8/4/10 1:30	15		79.3	10834.8		3.70	139.8				41.90	0.70	179.8		240	0.89	111.90	646	234.2	1.106	
8/4/10 1:30															885						`DI fill up
8/4/10 2:30	16		81.1	10654.3		3.01	140.1				45.60	0.76	180.2		830	0.92	90.00	649	255.0	1.114	
8/4/10 3:30	17		78.9	10481.3		2.88	139.4				44.00	0.73	180.6		777	0.87	91.60	646	237.2	1.107	
8/4/10 4:30	18		79.7	10295.1		3.10	139.1				46.60	0.78	180.4		722	0.92	92.30	650	230.8	1.104	
8/4/10 5:30	19		80.1	10113.5		3.03	139.7				45.30	0.76	179.6		667	0.93	85.90	657	248.7	1.113	
8/4/10 6:30	20		80.7	9931.8		3.03	139.9		1.94	1.001	44.80	0.75	179.5		613	0.90	91.00	658	239.9	1.108	no disch sample
8/4/10 7:30	21		81.6	9759.3		2.88	139.8				47.20	0.79	179.8	2.044	558	0.91	92.00	652	245.8	1.110	the temperature probe in the reactor was full of solids, no letting the slurry to overflow. It was "shaken up" and the slurry started to overflow again.
8/4/10 8:30	22		80.6	9571.5		3.13	139.5				46.90	0.78	180.1		508	0.85	94.90	650	227.2	1.102	
8/4/10 9:30	23		78.9	9365.6		3.43	140.2				52.90	0.88	179.2		453	0.91	90.20	650	222.9	1.099	

Project: 12323-001
Client: Starfield Resources

Start Date: Aug 3rd, 2010
Technologist: Hugh Given (night)
Ernesto Bourricaudy (day)
Andrew Ferguson (afternoon)

Test ID: PP2 *Hydrolysis Tests*
Purpose: **Hydrolysis Long run**
Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
hot pulps (180C)

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600
Impellor: **Pitch-blade**

Test Data:

Date-Time			Preheater			Feed	Boiler						Hydrolysis Reactor								Observations
									Condensate				Bath		DI H_2O		Condensate				(hard copy - continue on other page if requried)
(24 h)	(h) elapsed	Condition	Temp ℃	Weight g	Hot s.g. g/mL	Flow g/min	Temp ℃	Hot s.g. g/mL	FAT g/L HCl	Density g/mL	Weight g	g/min	Temp ℃	Hot s.g. g/mL	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	(e-copy - write down comment on appropriate row - link event to time. If more than 1 row is required, adjust row height and format as wrap text)
8/4/10 10:40	24		81.7	9130.4		3.36	139.2		2.42	1.001	55.40	0.92	180.1		388	0.93	98.00	650	218.9	1.099	
8/4/10 11:30	25		84.0	8975.1		3.11	140.0				44.30	0.74	179.7		343	0.92	84.80	653	226.2	1.101	
8/4/10 12:35	26	3	82.1	8769.3		3.17	139.3				54.50	0.91	180.1		283	0.92	99.80	654	248.0	1.110	Condition change to 0.75mL/min
8/4/10 13:37	27		81.5	8566.9		3.26	140.3				52.80	0.88	179.3		234	0.79	86.10	647	221.0	1.100	
8/4/10 14:31	28		82.7	8383.5		3.40	139.2				44.00	0.73	179.5		194	0.74	72.80	646	238.3	1.106	
8/4/10 14:32	28														727						Filled up the DI water
8/4/10 15:30	29		81.4	8190.3		3.27	140.1				46.50	0.78	180.1		685	0.74	81.80	647	266.2	1.102	
8/4/10 16:35	30		82.3	7993.2		3.03	139.9				53.90	0.90	180.8		637	0.73	88.20	403	217.5	1.100	cleared solids built up in the reactor which produced a large discharge into the next sample.
8/4/10 17:30	31		82.8	7805.0		3.42	139.6				46.00	0.77	179.4		596	0.74	72.30	410	210.0	1.095	level in Reactor dropped a lot when solids cleared
8/4/10 18:35	32		79.1	7578.0		3.49	139.6				56.60	0.94	179.5		548	0.75	84.90	383	212.2	1.096	re-calibrate Probes due to weird ORP values
8/4/10 19:30	33		83.2	7413.0		3.00	139.5				46.90	0.78	179.9		506	0.75	80.30	386	216.9	1.097	
8/4/10 20:30	34			7202.8		3.50	140.2				48.80	0.81	180.6		462	0.75	82.40	374	220.5	1.100	
8/4/10 21:30	35		82.7	7014.5		3.14	139.5				50.80	0.85	179.5		417	0.75	84.80	355	224.0	1.100	
8/4/10 22:30	36		79.0	6819.5		3.25	140.1				44.60	0.74	179.3		372	0.74	81.60	379	222.1	1.100	
8/4/10 23:30	37			6660.1		2.66									324	0.80					Feed change more DI as well
8/4/10 23:30	37	4	80.2	10909.5			140.3		2.03	1.000	47.20	0.79	180.4	1.950	855		89.40	374	218.6	1.102	
8/5/10 0:30	38		81.8	10695.2		3.57	139.3				56.60	0.94	180.4		816	0.65	65.70	402	216.9	1.097	
8/5/10 1:30	39		80.0	10356.7		5.64	140.0				87.80	1.46	180.5		770	0.76	99.90	373	213.2	1.097	
8/5/10 2:30	40		79.9	10019.4		5.62	139.4				87.40	1.46	180.1		724	0.77	98.50	414	214.8	1.097	
8/5/10 3:30	41		80.2	9683.5		5.60	140.1				91.60	1.53	178.7		679	0.75	96.10	421	213.2	1.097	
8/5/10 4:30	42		82.6	9363.9		5.33	139.4		3.18	1.001	83.90	1.40	179.4		633	0.78	97.60	426	218.5	1.099	
8/5/10 5:30	43		82.1	9061.9		5.03	139.9				84.30	1.41	179.2		584	0.81	100.80	400	226.5	1.102	
8/5/10 6:30	44		80.4	8721.9		5.67	139.4				84.20	1.40	180.3		533	0.85	111.30	378	240.0	1.103	
8/5/10 7:30	45		79.2	8385.9		5.60	140.0				89.80	1.50	178.9		480	0.88	102.60	405	224.6	1.101	
8/5/10 8:30	46		82.8	8060.5		5.42	139.5				85.90	1.43	180.0		424	0.93	115.80	406	219.7	1.100	
8/5/10 9:40	47		80.2	7665.7		5.64	139.6		3.20	1.002	102.30	1.71	178.6	1.994	357	0.95	101.80	398	235.2	1.105	Some foaming is formed inside the reactor.
8/5/10 10:30	48		79.0	7353.1		6.25	140.0				77.70	1.30	179.6		310	0.94	96.50	395	243.9	1.109	
8/5/10 11:30	49		82.9	7029.5		5.39	139.9				91.80	1.53	179.4		252	0.98	92.50	395	246.2	1.109	
8/5/10 11:30	49														736						DI water filled up.
8/5/10 12:30	50		79.5	6664.9		6.08	139.8				87.70	1.46	180.4		678	0.96	109.00	441	238.0	1.107	

Project: 12323-001
Client: Starfield Resources

Start Date: Aug 3rd, 2010
Technologist: Hugh Given (night)
Ernesto Bourricaudy (day)
Andrew Ferguson (afternoon)

Test ID: PP2 *Hydrolysis Tests*
Purpose: **Hydrolysis Long run**

Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
hot pulps (180C)

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600
Impellor: **Pitch-blade**

Test Data:

Date-Time			Preheater			Feed	Boiler						Hydrolysis Reactor								Observations
									Condensate				Bath		DI H_2O		Condensate				(hard copy - continue on other page if requried)
(24 h)	(h) elapsed	Condition	Temp ℃	Weight g	Hot s.g. g/mL	Flow g/min	Temp ℃	Hot s.g. g/mL	FAT g/L HCl	Density g/mL	Weight g	g/min	Temp ℃	Hot s.g. g/mL	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	(e-copy - write down comment on appropriate row - link event to time. If more than 1 row is required, adjust row height and format as wrap text)
8/5/10 13:10	50	5		6428.9		5.90															13:00 Chaged condition. Temp increased up to 185C
8/5/10 13:13	51			10500.1																	
8/5/10 13:30	51		80.1	10419.1		4.76	139.9				85.80	1.43	185.1		620	0.97	135.30	428	268.9	1.120	
8/5/10 14:30	52		82.1	10122.5		4.94	139.7		3.14	1.001	92.90	1.55	184.5		562	0.97	134.10	405	303.9	1.136	
8/5/10 15:30	53		81.8	9810.6		5.20	139.2				83.50	1.39	183.8		505	0.94	119.50	382	293.0	1.130	cleared solids from O/F
8/5/10 16:30	54		79.9	9462.0		5.81	139.3				91.70	1.53	185.4		446	0.98	128.70	380	285.3	1.127	cleared solids from O/F
8/5/10 17:30	55		82.4	9147.6		5.24	139.3				90	1.50	184.0		387	0.98	132.40	378	283.7	1.127	cleared solids from O/F
8/5/10 18:30	56		82.9	8826.5		5.35	139.6				128	2.13	185.1		331	0.95	136.40	460	298.0	1.125	cleared solids from O/F
8/5/10 19:30	57		81.2	8483.1		5.72	140.1				130	2.17	184.3		272	0.98	139.50	485	284.5	1.126	cleared solids from O/F. ORP Broke New Re-calibrated
8/5/10 20:30	58		79.4	8155.8		5.46	139.6				133	2.22	185.0		213	0.98	133.10	492	283.0	1.125	cleared solids from O/F
8/5/10 21:30	59		83.5	7805.6		5.84	139.2				124	2.06	184.9		156	0.95	135.30	500	270.1	1.120	cleared solids from O/F
8/5/10 22:30	60		81.0	7482.0		5.39	139.5				128	2.14	183.9		98	0.97	139.40	411	281.5	1.120	cleared solids from O/F
8/5/10 22:30	60														572						
8/5/10 23:30	61		83.9	7173.0		5.15	139.3		4.82	1.001	86	1.43	185.3	1.980	510	1.02	120.30	381	276.6	1.122	
8/6/10 0:30	62	6	81.6	6830.1		5.72	139.0				87	1.45	184.6		451	0.98	81.80	502	215.4	1.098	new targets 6mL feed 1.5 mL DI
8/6/10 1:30	63		84.6	6294.0		8.94	138.9				145	2.42	185.0		380	1.19	164.40	393	264.8	1.118	
8/6/10 1:50	63			6130.6		8.17															new feed
8/6/10 1:50	63			10242.8																	
8/6/10 2:30	64		80.7	9909.5		8.33	138.8				129	2.16	185.2		301	1.32	183.70	379	271.6	1.119	
8/6/10 3:30	65		80.0	9376.0		8.89	139.2				145	2.41	183.9		224	1.28	166.70	380	273.3	1.121	
8/6/10 3:30	65														851						DI top up
8/6/10 4:30	66		82.5	8833.3		9.05	140.0		5.41	1.001	148	2.46	185.3		772	1.32	193.40	409	278.9	1.122	
8/6/10 5:30	67		80.4	8318.0		8.59	139.2				136	2.27	183.6		690	1.36	175.80	383	274.3	1.122	
8/6/10 6:30	68		82.9	7775.3		9.05	139.6				146	2.43	183.8		610	1.34	200.70	369	295.6	1.126	
8/6/10 7:32	69		81.5	7217.7		8.99	139.6				146	2.43	185.0		521	1.44	182.30	404	279.2	1.125	
8/6/10 7:59	69			10837.6																	Feed was changed
8/6/10 8:30	70		79.5	10558.5		9.00	138.6				119	1.99	184.1		434	1.51	201.30	423	278.3	1.125	
8/6/10 9:32	71		80.1	9958.0		9.69	139.6				151	2.52	183.9		338	1.55	220.60	383	275.0	1.125	
8/6/10 10:30	72		79.3	9466.0		8.48	137.1		3.51	1.102	132	2.20	182.1		255	1.43	189.70	433	264.5	1.119	
8/6/10 11:29	73		82.5	8946.3		8.81	137.9				137	2.28	183.1		158	1.64	191.70	448	284.6	1.124	

Project: 12323-001
Client: Starfield Resources

Start Date: Aug 3rd, 2010
Technologist: Hugh Given (night)
Ernesto Bourricaudy (day)
Andrew Ferguson (afternoon)

Test ID: PP2 *Hydrolysis Tests*
Purpose: Hydrolysis Long run

Health Safety:

Main Hazards: concentrated brine liquors
concentrated HCl
hot pulps (180C)

PPE and Precautions: operate test with sash closed
if work access is requried to the test area (open sash), a face shield is required

Feed RPM:
Boiler RPM: 150
Hydrolysis Reactor RPM: 600
Impellor: **Pitch-blade**

Test Data:

Date-Time			Preheater			Feed	Boiler						Hydrolysis Reactor								Observations
									Condensate				Bath		DI H_2O		Condensate				(hard copy - continue on other page if requried)
(24 h)	(h) elapsed	Condition	Temp ℃	Weight g	Hot s.g. g/mL	Flow g/min	Temp ℃	Hot s.g. g/mL	FAT g/L HCl	Density g/mL	Weight g	g/min	Temp ℃	Hot s.g. g/mL	Vol. g	Flow g/min	Weight g	ORP mV	FAT g/L HCl	Density g/mL	(e-copy - write down comment on appropriate row - link event to time. If more than 1 row is required, adjust row height and format as wrap text)
8/6/10 11:29	73														630						DI water was filled up.
8/6/10 12:30	74		81.0	8518.9		7.01	138.1				115	1.92	172.0		555	1.23	80.20	456	175.3	1.080	Temp control stopped working for around 10 minutes no apparent cause. Temp in the reactor drooped down to 172C.
8/6/10 13:30	75		82.7	7957.0		9.37	138.6				139	2.31	184.6		462	1.56	201.60	498	143.0	1.065	
8/6/10 14:30	76		81.8	7408.3	1.500	9.15	138.5	1.6	3.30	1.001	175	2.92	182.4		375	1.44	154.40	486	140.7	1.065	
8/6/10 15:30	77					---										---					
						---							Scrubber 1			---	1039.86		23.4	1.011	
						---							Scrubber 2			---	983.36		0.6	1.000	
Totals/Avg:																					

Project: 12323-001
Client: Starfield Resources

Test ID: PP2

Discharge Sampling Data

Sample Name			Weight	H_2O+cup	H_2O + cup	pulp	Sample Composition		Diluted Weight		Volume			Wet	Dry	%H_2O	Filtration	Colour		Observations	Calculated Filtrate Concentration							Solids Composition					
date -time	Sample ID	Condition	cup (tare)	(gross)	+ discharge	ORP	Diluent	Sample	pulp	PLS	PLS	pH	SG	res	res		time	PLS	RES		Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	Fe	Cu	Co	Ni	Mg	Cl
			g	g	g	mV	g	g	g	g	mL		g/mL	g	g		min				g/kg	g/kg	g/kg	g/kg	g/kg	g/kg	g/kg	%	%	%	%	%	%
8/3/10 12:40	1	1	48.1	416.5	441.7	609	368.40	25.20	394	383.0	371	0.81	1.033			---					184	22	162	8	1	8	7						
8/3/10 13:40	2		48.7	441.8	474.9	602	393.10	33.10	426	418.9	400	1.03	1.048			---					228	27	201	8	1	11	9						
8/3/10 14:30	3		48.1	432.1	524.7	605	384.00	92.60	477	457.9	409	0.61	1.120			---					215	40	175	16	1	11	8						
8/3/10 15:30	4		48.7	410.4	435.0	602	361.70	24.60	386	379.2	364	1.13	1.041			---					246	26	220	10	1	11	10						
8/3/10 16:30	5		48.1	453.3	477.8	596	405.20	24.50	430	424.5	410	1.18	1.035			---					233	17	216	6	1	9	9						
8/3/10 17:30	6		48.7	424.6	482.6	606	375.90	58.00	434	418.1	389	0.83	1.076			---					206	45	162	19	1	11	7						
8/3/10 18:30	7		48.1	452.6	541.9	612	404.50	89.30	494	474.9	427	0.56	1.112			---					218	39	179	14	1	12	9						
8/3/10 19:30	8		48.7	426.5	446.9	605	377.80	20.40	398	390.6	378	1.18	1.035			---					278	41	237	13	1	13	11						
8/3/10 20:30	9		48.1	409.3	563.3	612	361.20	154.00	515	488.4	408	0.33	1.198			---					216	40	176	13	1	12	9						
8/3/10 21:30	10		48.7	426.4	541.7	608	377.70	115.30	493	472.1	411	1	1.150			---					221	39	182	13	1	12	9						
8/3/10 22:30	11		48.1	401.0			352.90	---	---		---					---					---	---	---	---	---	---	---						
8/3/10 23:30	12		48.7	413.8	551.9	607	365.10	138.10	503	481.1	406.3	0.52	1.184	100.1	89.3	11%		br	dk rd	C1 disch 1-12 solids	235	38	196	12	1	14	10	66.9	0.02				0.94
8/4/10 1:30	13	2	48.1	432.9	476.1	609	384.80	43.20	428.00	415.7	5.2	1.07	#####			---		gr			2	0	2	0	0	0	0						
8/4/10 2:30	14		48.7	431.4	480.7	611	382.70	49.30	432.00	422.8	399.2	0.74	1.059			---		gr			164	35	130	22	1	11	8						
8/4/10 3:30	15		48.1	467.0	563.0	607	418.90	96.00	514.90	497.4	444.1	0.80	1.120			---		br			212	41	171	11	1	14	11						
8/4/10 4:30	16		48.7	420.4	480.0	614	371.70	59.60	431.30	421.0	388.0	0.95	1.085			---		br			209	46	163	10	1	14	10						
8/4/10 5:30	17		48.1	423.4	462.5	604	375.30	39.10	414.40	410.4	395.8	0.81	1.037			---		gr			134	29	104	10	1	9	6						
8/4/10 7:30	18		48.7	392.4	575.2	604	343.70	182.80	526.50	494.7	402.5	-0.01	1.229			---					216	40	176	12	1	13	11						
8/4/10 8:30	19		48.1	436.3	458.5	612	388.20	22.20	410.40	404.0	392.2	1.15	1.030			---					186	39	146	11	1	13	9						
8/4/10 9:30	20		48.7	426.4	522.9	606	377.70	96.50	474.20	455.5	407.4	0.81	1.118			---					188	42	146	12	1	14	10						
8/4/10 10:40	21		48.1	434.9	484.8	614	386.80	49.90	436.70	425.8	399.8	0.91	1.065			---					203	43	160	25	1	13	10						
8/4/10 11:30	22		48.7	419.4	442.2	613	370.70	22.80	393.50	389.8	377	1.14	1.033			---					207	39	168	10	1	11	10						
8/4/10 12:30	23		48.1	452.1	489.3	610	404.00	37.20	441.20	433.1	415	0.89	1.044	70.2	67.6	4%					181	31	150	7	1	10	9	66.3	0.034				0.51
8/4/10 13:30	24	3	48.7	405.1	445.1	614	356.40	40.00	396.40	388.3	368	1.06	1.056			---					196	39	156	9	1	11	10						
8/4/10 14:30	25		48.1	442.1	570.0	609	394.00	127.90	521.90	500.6	433	0.54	1.155			---					211	47	164	12	1	13	11						
8/4/10 15:30	26		48.7	448.5	473.2	598	399.80	24.70	424.50	415.9	401	1.38	1.036			---		orange	dk brown		227	35	193	8	1	12	12						
8/4/10 16:30	27		48.1	446.2	567.5	575	400.40	119.00	519.40	495.1	430	0.70	1.151			---		dk brown	dk brown		219	49	170	11	1	14	12						
8/4/10 17:30	28		48.7	457.1	716.6	593	397.50	270.40	667.90	614.5	482	0.25	1.274			---		dk brown	dk brown		201	45	155	12	1	14	10						
8/4/10 18:30	29		48.1	436.3	510.4	563	409.00	53.30	462.30	445.9	406	1.00	1.097			---		brown	dk brown		295	64	230	17	2	19	14						
8/4/10 19:30	30		48.7	423.5	512.1	565	387.60	75.80	463.40	443.5	398	0.90	1.113			---		lt brown	dk brown		232	55	177	14	1	16	12						
8/4/10 20:30	31		48.1	420.1	465.5	560	375.40	42.00	417.40	405.8	380	1.12	1.068			---		amber	dk brown		245	51	194	12	1	15	13						
8/4/10 21:30	32		48.7	410.4	477.8	550	371.40	57.70	429.10	414.6	378	0.82	1.097			---		amber	dk brown		250	54	196	13	1	16	13						
8/4/10 22:30	33		48.1	408.4	490.1	544	362.30	79.70	442.00	425.0	381	0.79	1.115			---			dk brown		217	49	167	12	1	14	11						
8/4/10 23:30	34		48.7	448.4	571.8	585	399.70	123.40	523.10	500.2	436	0.69	1.146	134.5	126.8	6%		brown	blk	45.6g to oven t=18.2	216	43	174	10	1	13	11	67.9					0.95
8/5/10 0:30	35	4	48.1	432.3	480.4	564	384.20	48.10	432.30	423.3	397	1.16	1.065			---		brown	blk	cond 3 solids retained wet= 135.4g	207	42	165	12	1	14	11						
8/5/10 1:30	36		48.7	416.9	639.5	561	368.20	222.60	590.80	549.9	439	0.50	1.253			---		brown	blk		221	50	172	13	1	14	10						
8/5/10 2:30	37		48.1	410.9	601.6	549	362.80	190.70	553.50	523.6	417	0.57	1.257			---		brown	blk		220	47	173	12	1	14	10						
8/5/10 3:30	38		48.7	425.1	509.3	556	376.40	84.20	460.60	451.0	403	0.91	1.118			---		brown	blk		231	48	183	12	1	13	11						
8/5/10 4:30	39		48.1	439.2	550.7	566	391.10	111.50	502.60	487.5	426	0.76	1.144			---		brown	blk		228	47	181	14	1	13	11						
8/5/10 5:30	40		48.7	417.3	583.5	568	368.60	166.20	534.80	511.4	424	0.56	1.205			---		brown	blk		223	46	177	15	1	13	10						
8/5/10 6:30	41		48.1	438.6	719.7	556	390.50	281.10	671.60	624.1	482	0.22	1.294			---		brown	blk		220	47	173	13	1	13	10						
8/5/10 7:30	42		48.7	429.9	758.1	598	381.20	328.20	709.40	661.2	490	-0.24	1.350			---					212	49	163	14	1	13	10						
8/5/10 8:30	43		48.1	427.2	776.7	583	379.10	349.50	728.60	684.2	500	-0.04	1.369			---					220	48	171	14	1	13	10						
8/5/10 9:30	44		48.7	444.1	584.7	588	395.40	140.60	536.00	516.9	443	0.48	1.166			---					199	50	149	16	1	12	9						

Project: 12323-001
Client: Starfield Resources

Test ID: PP2

Discharge Sampling Data

Sample Name			Weight	H_2O+cup	H_2O + cup	pulp	Sample Composition		Diluted Weight		Volume			Wet	Dry	%H_2O	Filtration	Colour		Observations	Calculated Filtrate Concentration							Solids Composition					
date -time	Sample ID	Condition	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	pH	SG g/mL	res g	res g		time min	PLS	RES		Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Fe %	Cu %	Co %	Ni %	Mg %	Cl %
8/5/10 10:30	45		48.1	470.5	562.0	580	422.40	91.50	513.90	504.7	455	0.86	1.110			---					212	44	168	12	1	12	10						
8/5/10 11:30	46		48.7	413.5	449.5	531	364.80	36.00	400.80	397.1	388	0.64	1.023			---					64	24	40	8	0	4	2						
8/5/10 12:30	47		48.1	431.3	552.2	548	383.20	120.90	504.10	480.6	421	0.65	1.142			---					189	41	148	11	1	11	10						
8/5/10 13:30	48			445.0	499.5	593	---	54.50	---	443.2	414	1.07	1.071	232.4	211.8	8.9%				cond 4 solids put inot oven 21:45	203	43	160	9	1	11	10	68.50					1.14
8/5/10 14:30	49	5	48.1	415.1	625.3	576	367.00	210.20	577.20	537.9	435	0.12	1.237			---				retained is 173.8g wet solids of cond 4	200	47	153	12	1	11	9						
8/5/10 15:30	50		48.7	419.7	544.7	576	371.00	125.00	496.00	472.6	407	0.62	1.160			---					203	48	155	12	1	12	10						
8/5/10 16:30	51		48.1	454.0	598.8	560	405.90	144.80	550.70	522.1	445	0.64	1.173			---					216	49	167	12	1	13	11						
8/5/10 17:30	52		48.7	430.3	557.3	577	381.60	127.00	508.60	486.5	418	0.56	1.165			---					220	51	169	12	1	13	11						
8/5/10 18:30	53		48.1	397.4	543.1	588	349.30	145.70	495.00	462.9	385	0.54	1.202			---					220	52	167	12	1	13	11						
8/5/10 19:30	54		48.7	450.3	670.7	589	401.60	220.40	622.00	576.1	462	0.36	1.246			---					213	52	161	13	1	13	10						
8/5/10 20:30	55		48.1	446.2	567.0	618	398.10	120.80	518.90	493.4	429	0.69	1.150			---					211	52	159	12	1	13	11						
8/5/10 21:30	56		48.7	438.5	525.1	573	389.80	86.60	476.40	461.9	413	0.88	1.119			---					222	51	170	12	1	13	11						
8/5/10 22:30	57		48.1	405.4	569.0	568	357.30	163.60	520.90	484.7	397	0.46	1.221			---					221	55	165	13	1	13	10						
8/5/10 23:30	58		48.7	406.2	626.1	562	357.50	219.90	577.40	541.3	429	0.45	1.262			---					212	50	162	13	1	12	10						
8/6/10 0:30	59		48.1	420.9	697.6	568	372.80	276.70	649.50	606.9	475	0.36	1.278	255.4	223.5	12.5%		br	blk	42.4g wet to oven T= 12.5	204	45	159	12	1	11	10	68.40					0.90
8/6/10 1:30	60	6	48.7	426.9	567.9	580	378.20	141.00	519.20	477.0	407	0.65	1.173			---					204	51	153	15	1	12	9						
8/6/10 2:30	61		48.1	431.7	804.8	565	383.60	373.10	756.70	683.0	508	0.11	1.344			---					227	52	175	14	0	4	3						
8/6/10 3:30	62		48.7	413.2	650.5	539	364.50	237.30	601.80	562.2	439	0.36	1.281			---					222	51	171	14	1	8	6						
8/6/10 4:30	63		48.1	413.3	745.1	589	365.20	331.80	697.00	633.7	473	0.28	1.341			---					216	53	163	14	0	1	1						
8/6/10 5:30	64		48.7	410.2	669.7	597	361.50	259.50	621.00	568.5	442	0.41	1.286			---					217	53	164	14	0	2	2						
8/6/10 6:30	65		48.1	424.6	649.5	553	376.50	224.90	601.40	516.9	410	0.41	1.262			---					203	48	155	12	0	6	4						
8/6/10 7:30	66		48.7	434.3	684.9	581	385.60	250.60	636.20	584.4	461	0.27	1.269			---					200	50	150	11	0	4	3						
8/6/10 8:30	67		48.1	422.9	658.2	584	374.80	235.30	610.10	563.2	443	0.23	1.272			---					204	52	152	12	0	2	1						
8/6/10 9:30	68		48.7	427.7	748.3	588	379.00	320.60	699.60	633.9	474	-0.06	1.337			---					211	51	159	13	1	9	7						
8/6/10 10:30	69		48.1	425.2	740.3	584	377.10	315.10	692.20	630.9	489	-0.06	1.291			---					209	53	157	13	0	1	1						
8/6/10 11:30	70		48.7	425.5	490.8	575	376.80	65.30	442.10	439.3	405	0.70	1.085			---					209	47	162	10	2	19	15						
8/6/10 12:30	71		48.1	414.6	955.8	608	366.50	541.20	907.70	814.7	554	-0.60	1.471			---					228	54	174	16	1	14	10						
8/6/10 13:30	72		48.7	430.4	733.0	580	381.70	302.60	684.30	637.8	493	-0.06	1.293			---					204	47	157	12	1	12	9						
8/6/10 14:30	73		48.1	445.4	648.5	573	397.30	203.10	600.40	569.5	468	0.40	1.217	545.0	385.6	29%				remainder of solids 528.8g wet	204	48	156	14	1	12	9	70.60					0.67
							---	---	---		---			267.4	237.6	11.1%				final reactor solids 246.2g wet	---	---	---	---	---	---	---	69.20					0.90
							---	---	---		---					---				scrubber 1 net wt. 1039.86g SG 1.0114 FAT=23.39	---	---	---	---	---	---	---						
							---	---	---		---					---				scrubber 2 net wt. 983.36 SG 0.9999 FAT=0.57	---	---	---	---	---	---	---						
							---	---	---		---					---					---	---	---	---	---	---	---						

Project: 12323-001
Client: Starfield Resources

Test ID: PP2

Profile Sampling Data

Sample Name		Weight	H_2O+cup	HCl+cup	pulp	Sample Composition		Diluted Weight		Volume					Filtration	Colour		Observations	Calculated Filtrate Concentration							Solids Composition					
date -time	Sample ID	cup (tare) g	(gross) g	+ discharge g	ORP mV	Diluent g	Sample g	pulp g	PLS g	PLS mL	pH	SG g/mL	Wet res g	Dry res g	%H_2O	time min	PLS	RES		Fe g/kg	Fe(II) g/kg	Fe(III) g/kg	Cu g/kg	Co g/kg	Ni g/kg	Mg g/kg	Fe %	Cu %	Co %	Ni %	Mg %	Cl %
8/3/10 7:00						---	---	---		---					---					---	---	---	---	---	---	---						
8/3/10 8:00						---	---	---		---					---					---	---	---	---	---	---	---						
8/4/10 1:00	Reactor C1	18.1	98.1	140.2	596	80.00	42.10	122.10	114.3	92.9	0.26	1.230	5.6	5.1	9%			brown		213	40	173	13	1	13	10	68.0	0.034				1.19
8/4/10 12:00	Reactor C2	18.2	107.0	130.3		88.80	23.30	112.10	105.4	93		1.129	Not enough		---					197	45	152	12	1	14	11						
8/4/10 23:45	Reactor C3	18.3	100.5	153.1	563	82.20	52.60	134.80	123.7	98	0.34	1.262	9.5	8.6	9%		br	blk		209	48	162	13	1	14	10	67.3					1.15
8/5/10 12:45	Reactor C4	18.2	97.4	136.7		79.20	39.30	118.50	107.3	88	0.49	1.221	4.7	4.1	13%					200	46	153	14	1	12	9	66.7					2.27
8/6/10 0:00	Reactor C5	18.2	97.0	150.1	540	78.80	53.08	131.88	117.8	94	-0.10	1.252	9.7	8.5	12%	<1 min	br	blk	T=12.28	198	51	147	13	1	14	9	67.7					1.26
8/6/10 13:00	Feed C6	18.4	116.7	148.8		98.30	32.10	130.40	128.9	117		1.105			---					152	28	123	8	1	7	5						
8/6/10 13:00	Boiler C6	18.5	112.2	150.4		93.70	38.20	131.90	128.2	109		1.178			---					210	39	171	10	1	9	8						
8/6/10 13:00	Reactor C6	18.1	108.7	151.4		90.60	42.70	133.30	136.0	113		1.200			---					205	55	150	16	1	13	10						
																				150	25	125	7	1	7	5						

Project: 12323-001
Client: Starfield Resources

Test ID: PP2

initial: 08/03/2010 10:40
final: 08/06/2010 14:30
Feed: variable
Reactor rpm: 600
Reactor condensate: Drawn off

run time:	75.83	hours
avg feed flow:	4.88	g/min
	3.25	mL/min
average water addition	0.89	mL/min
average product flow	2.33	g/min
average product flow	1.19	mL/min
Reactor retention time:	10	hours
Test Turnovers:	7.7	
feed:	22201	g
product:	10608	g pulp
	1369	g solids

Preheater temp	81	℃
Boiler temp	140	℃
Reactor temp	180	℃

Balance

Stream	Weight	Fe	Fe(II)	Fe(III)	Cu	Co	Ni	Mg	
					grams				
Feed	22201	3364	629	2735	171	14	148	121	
Seed	90	61			0	0	0	0	
Reactor - start	1317	233	42	191	11	1	10	8	
H2O feed to reactor	4066								
Discharge	10608	2252	506	1745	138	10	113	88	
Solids	1369	943							
Reactor - end	1386	284	76	209	22	2	18	14	
Boiler Condensate	6471								
Reactor condensate	8503	1.5							
Scrubbers									
Accumulation	69	52	34	18	11	1	8	6	
IN:	26357	3425	629	2735	171	14	148	121	
OUT:	25651	3248	540	1763	149	11	122	94	
Accountability, OUT/IN	97	95	86	64	87	75	82	78	
Metal Hydrolysed				972	22	4	26	27	
% Metal Hydrolysed				36	13	25	18	22	
				32%					

Average HCl strength	22.1%
Maximum HCl strength	28.2%
mol of Fe hydrolysed	17.404
mol of Fe (as hematite)	15.797
mol of HCl produced	51.751

Appendix E - XRD assays
Oxidation and Hydrolysis samples

Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resources*
Project Number/ LIMS No.	*12323-001/MI4508-APR10*
Reporting Date:	*May 19, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
2) Summary of Mineral Asemblages
3) Semi-Quantitative XRD Results
4) Chemical Balance(s)
5) XRD Pattern(s)

Anita Coppaway
Mineralogical Technologist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

Interpretations do not reflect the presence of non-crystalline and/or amorphous compounds. Mineral proportions are based on relative peak heights and may be strongly influenced by crystallinity, structural group or preferred orientations. Interpretations and relative proportions should be accompanied by supporting petrographic and geochemical data (Whole Rock Analysis, Inductively Coupled Plasma - Optical Emission Spectroscopy, etc.).

Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Test 7 PPT-Reactor-Final	hematite	-	-	*magnetite, *siderite, *mica

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Hematite	Fe_2O_3
Magnetite	Fe_3O_4
Muscovite	$KAl_2(AlSi_3O_{10})(OH)_2$
Siderite	$FeCO_3$

Semi-Quantitative X-ray Diffraction Results

Mineral	Test 7 PPT-Reactor-Final (wt %)
Hematite	97.3
Magnetite	1.7
Siderite	0.8
Muscovite	0.2
TOTAL	100.0

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

Chemical Balance

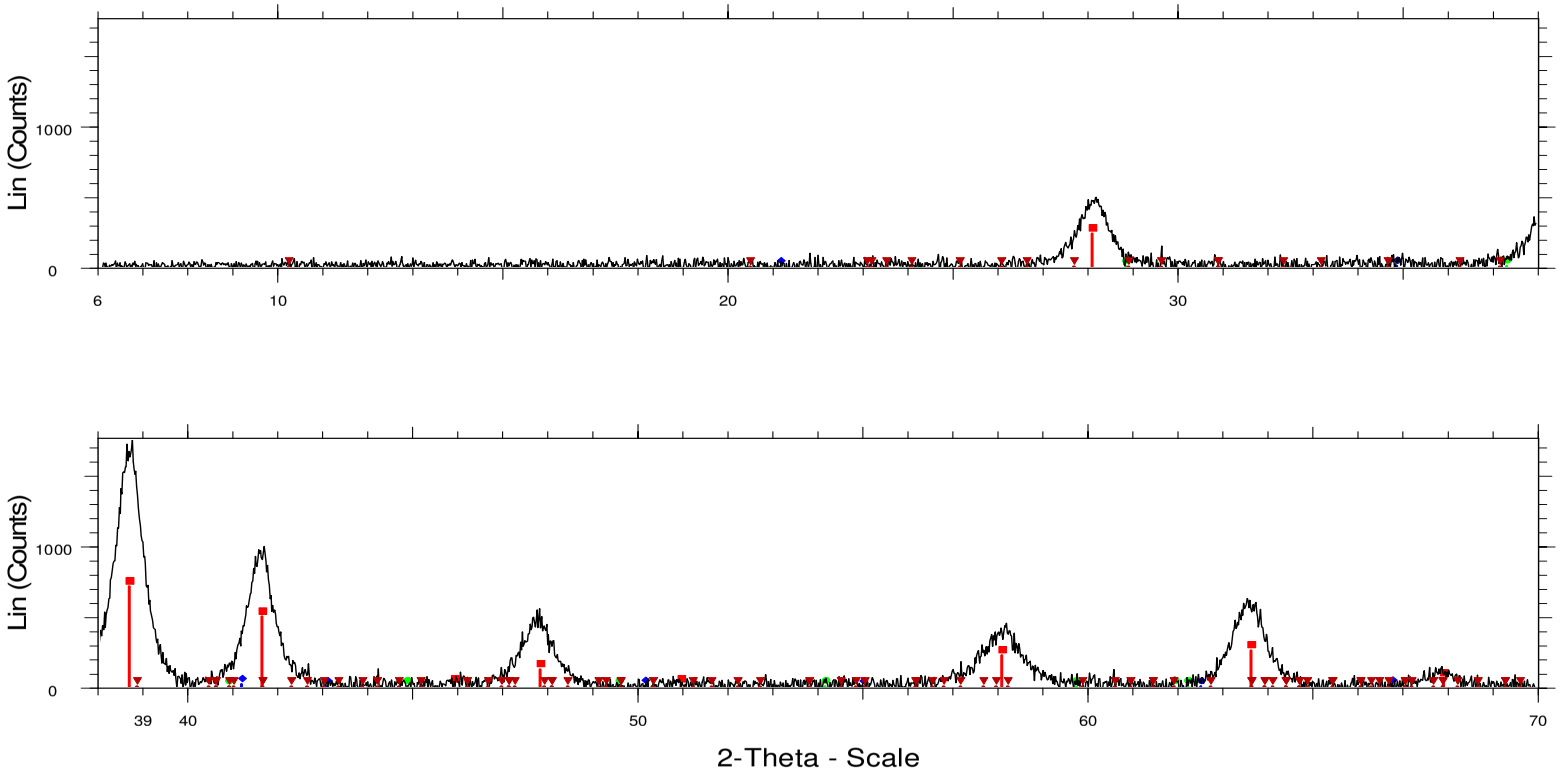
Test 7 PPT-Reactor-Final

Name	Assay[1]	SQD[2]	Delta	Status
Iron	67.4	69.7	-2.30	Both
Oxygen	29.1	30.1	-1.07	Both
Chlorine	2.10	-	2.10	XRF
Vanadium	0.04	-	0.04	XRF
Silicon	0.04	0.05	0.00	Both
Calcium	0.04	-	0.04	XRF
Potassium	0.03	0.02	0.01	Both
Aluminum	0.03	0.04	-0.02	Both
Copper	0.02	-	0.02	XRF
Chromium	0.01	-	0.01	XRF
Carbon	-	0.08	-0.08	SQD
Hydrogen	-	0.00	0.00	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

Test 7 PPT-Reactor_final



Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resources*
Project Number/ LIMS No.	*12323-001/MI4505-MAY10*
Reporting Date:	*June 3, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
2) Summary of Mineral Asemblages
3) Semi-Quantitative XRD Results
4) Chemical Balance(s)
5) XRD Pattern(s)

Anita Coppaway
Mineralogical Technologist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

Interpretations do not reflect the presence of non-crystalline and/or amorphous compounds. Mineral proportions are based on relative peak heights and may be strongly influenced by crystallinity, structural group or preferred orientations. Interpretations and relative proportions should be accompanied by supporting petrographic and geochemical data (Whole Rock Analysis, Inductively Coupled Plasma - Optical Emission Spectroscopy, etc.).

Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Test 10 Reactor	akaganeite	iron oxide chloride, rokuhnite, hematite	-	*nickel chloride hydrate, *chlorite

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Akaganeite	$Fe_{7.6}Ni_{0.4}O_{6.4}(OH)_{9.7}Cl_{1.3}$
Chlorite	$(Fe,(Mg,Mn)_5,Al)(Si_3Al)O_{10}(OH)_8$
Hematite	Fe_2O_3
Iron Oxide Chloride	FeOCl
Nickel Chloride Hydrate	$NiCl_2(H_2O)_6$
Rokuhnite	$FeCl_2{\cdot}(H_2O)$

Semi-Quantitative X-ray Diffraction Results

Mineral	**Test 10 Reactor** (wt %)
Iron Oxide Chloride	28.0
Akaganeite	44.4
Rokuhnite	13.0
Hematite	12.3
Nickel Chloride Hydrate	0.3
Clinochlore	2.0
TOTAL	100.0

Chemical Balance

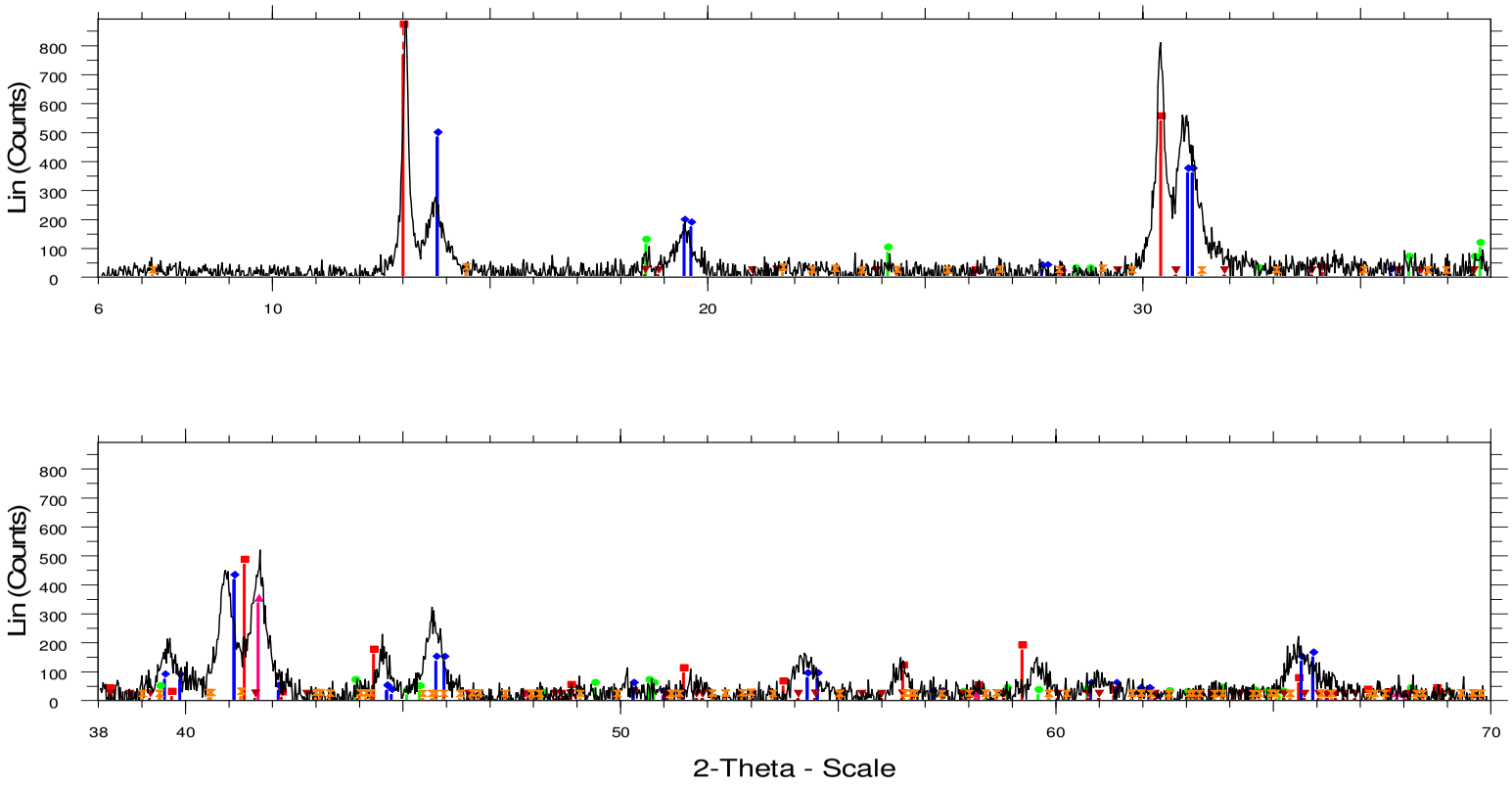
Test 10 Reactor

Name	Assay[1]	SQD[2]	Delta	Status
Iron	48.8	53.9	-5.13	Both
Chlorine	22.1	17.7	4.41	Both
Nickel	0.04	0.07	-0.03	Both
Copper	0.01	-	0.01	XRF
Hydrogen	-	0.84	-0.84	SQD
Oxygen	-	26.6	-26.6	SQD
Magnesium	-	0.40	-0.40	SQD
Aluminum	-	0.15	-0.15	SQD
Silicon	-	0.30	-0.30	SQD
Chromium	-	0.02	-0.02	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

Test 10 Reactor



Test 10 Reactor - File: May4505-1.raw
Operations: Background 1.000,1.000 | Import
01-072-0619 (C) - Iron Oxide Chloride - FeOCl
00-042-1315 (I) - Akaganeite-M - Fe8(O,OH)16Cl1.3
00-025-1040 (*) - Rokuhnite, syn - FeCl2·2H2O
01-073-0603 (C) - Hematite, syn - Fe2O3
01-074-1191 (C) - Nickel Chloride Hydrate - NiCl2(H2O)6
01-079-0761 (C) - Clinochlore - (Mg4.715Al.694Fe.269Fe.109Cr.128Ni.011)(Si3.056Al.944)O10(OH)8

Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resources*
Project Number/ LIMS No.	*12323-001/MI4500-JUN10*
Reporting Date:	*June 28, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
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4) Chemical Balance(s)
5) XRD Pattern(s)

Anita Coppaway
Mineralogical Technologist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

Interpretations do not reflect the presence of non-crystalline and/or amorphous compounds. Mineral proportions are based on relative peak heights and may be strongly influenced by crystallinity, structural group or preferred orientations. Interpretations and relative proportions should be accompanied by supporting petrographic and geochemical data (Whole Rock Analysis, Inductively Coupled Plasma - Optical Emission Spectroscopy, etc.).

Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Test 11 Reactor Solids	akaganeite	iron oxide chloride, hematite	iron manganese oxide chloride	*nickel chloride hydrate, *fluorite

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Akaganeite	$Fe_{7.6}Ni_{0.4}O_{6.4}(OH)_{9.7}Cl_{1.3}$
Fluorite	CaF_2
Hematite	Fe_2O_3
Iron Manganese Oxide Chloride	$FeMn_7O_{10}Cl_3$
Iron Oxide Chloride	FeOCl
Nickel Chloride Hydrate	$NiCl_2(H_2O)_6$

Semi-Quantitative X-ray Diffraction Results

Mineral	**Test 11 Reactor Solids** (wt %)
Akaganeite	48.9
Iron Oxide Chloride	21.4
Hematite	21.6
Iron Manganese Oxide Chloride	4.3
Nickel Chloride Hydrate	1.9
Fluorite	1.9
TOTAL	100.0

Chemical Balance

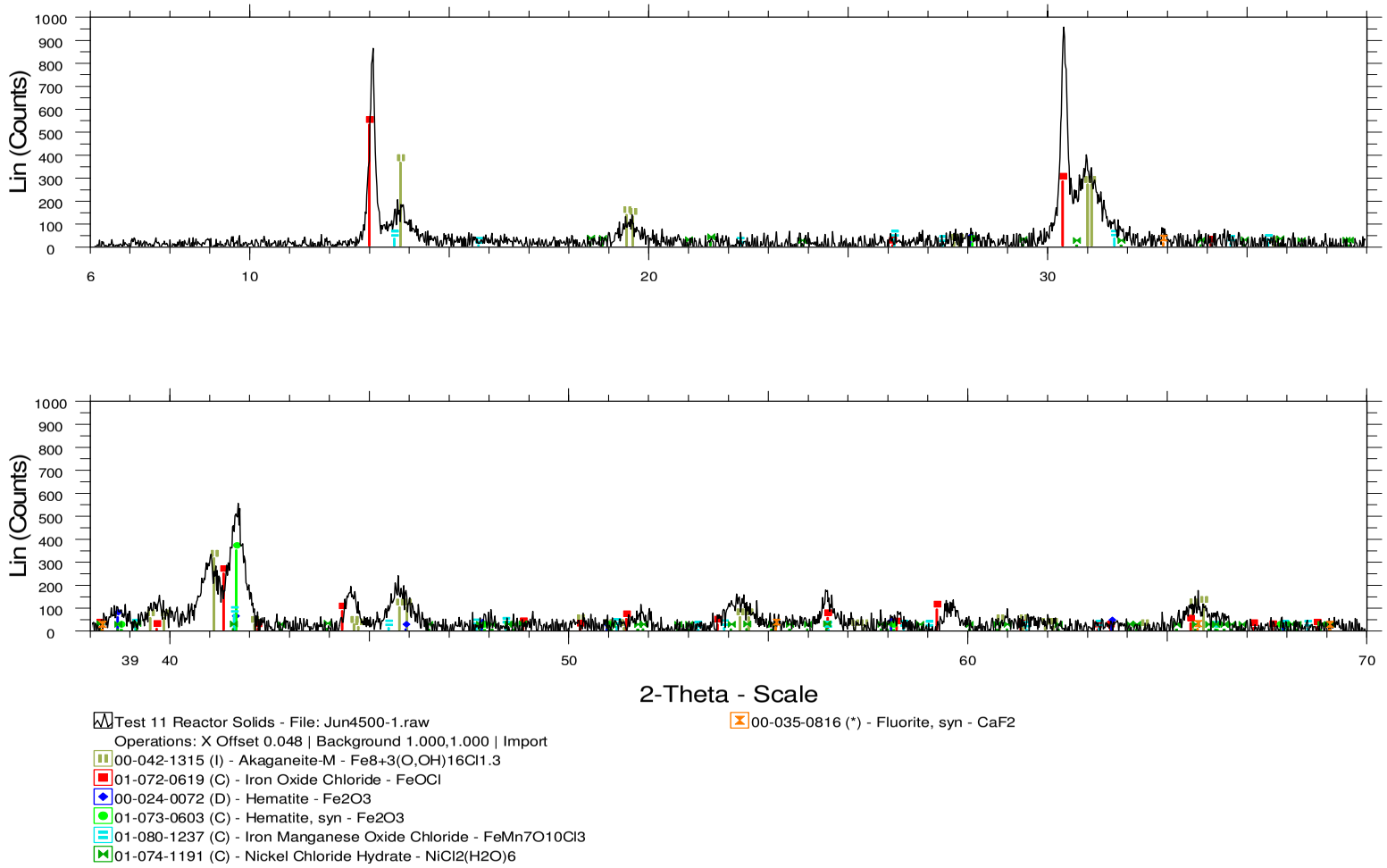
Test 11 Reactor Solids

Name	Assay[1]	SQD[2]	Delta	Status
Iron	46.9	55.4	-8.54	Both
Chlorine	11.0	11.3	-0.29	Both
Nickel	0.30	0.47	-0.17	Both
Copper	0.03	-	0.03	XRF
Hydrogen	-	0.62	-0.62	SQD
Oxygen	-	28.0	-28.0	SQD
Fluorine	-	0.94	-0.94	SQD
Calcium	-	0.99	-0.99	SQD
Manganese	-	2.34	-2.34	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

Test 11 Reactor Solids



Test 11 Reactor Solids - File: Jun4500-1.raw
Operations: X Offset 0.048 | Background 1.000,1.000 | Import
00-042-1315 (I) - Akaganeite-M - Fe8+3(O,OH)16Cl1.3
01-072-0619 (C) - Iron Oxide Chloride - FeOCl
00-024-0072 (D) - Hematite - Fe2O3
01-073-0603 (C) - Hematite, syn - Fe2O3
01-080-1237 (C) - Iron Manganese Oxide Chloride - FeMn7O10Cl3
01-074-1191 (C) - Nickel Chloride Hydrate - NiCl2(H2O)6
00-035-0816 (*) - Fluorite, syn - CaF2

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resources*
Project Number/ LIMS No.	*12323-001/MI4504-JUN10*
Reporting Date:	*June 28, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

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Anita Coppaway
Mineralogical Technologist

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Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

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Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Test 12 Reactor Solids	hematite	iron oxide chloride, akaganeite	-	*iron nickel chloride hydroxide hydrate, *ilmenite

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Akaganeite	$Fe_{7.6}Ni_{0.4}O_{6.4}(OH)_{9.7}Cl_{1.3}$
Hematite	Fe_2O_3
Ilmenite	$FeTiO_3$
Iron Nickel Chloride Hydroxide Hydrate	$Fe_{0.67}Ni_{1.33}Cl(OH)_4{\cdot}xH_2O$
Iron Oxide Chloride	FeOCl

Semi-Quantitative X-ray Diffraction Results

Mineral	**Test 12 Reactor Solids** (wt %)
Hematite	72.2
Iron Oxide Chloride	15.3
Akaganeite	10.8
Iron Nickel Chloride Hydroxide Hydrate	0.4
Ilmenite	1.3
TOTAL	100.0

Chemical Balance

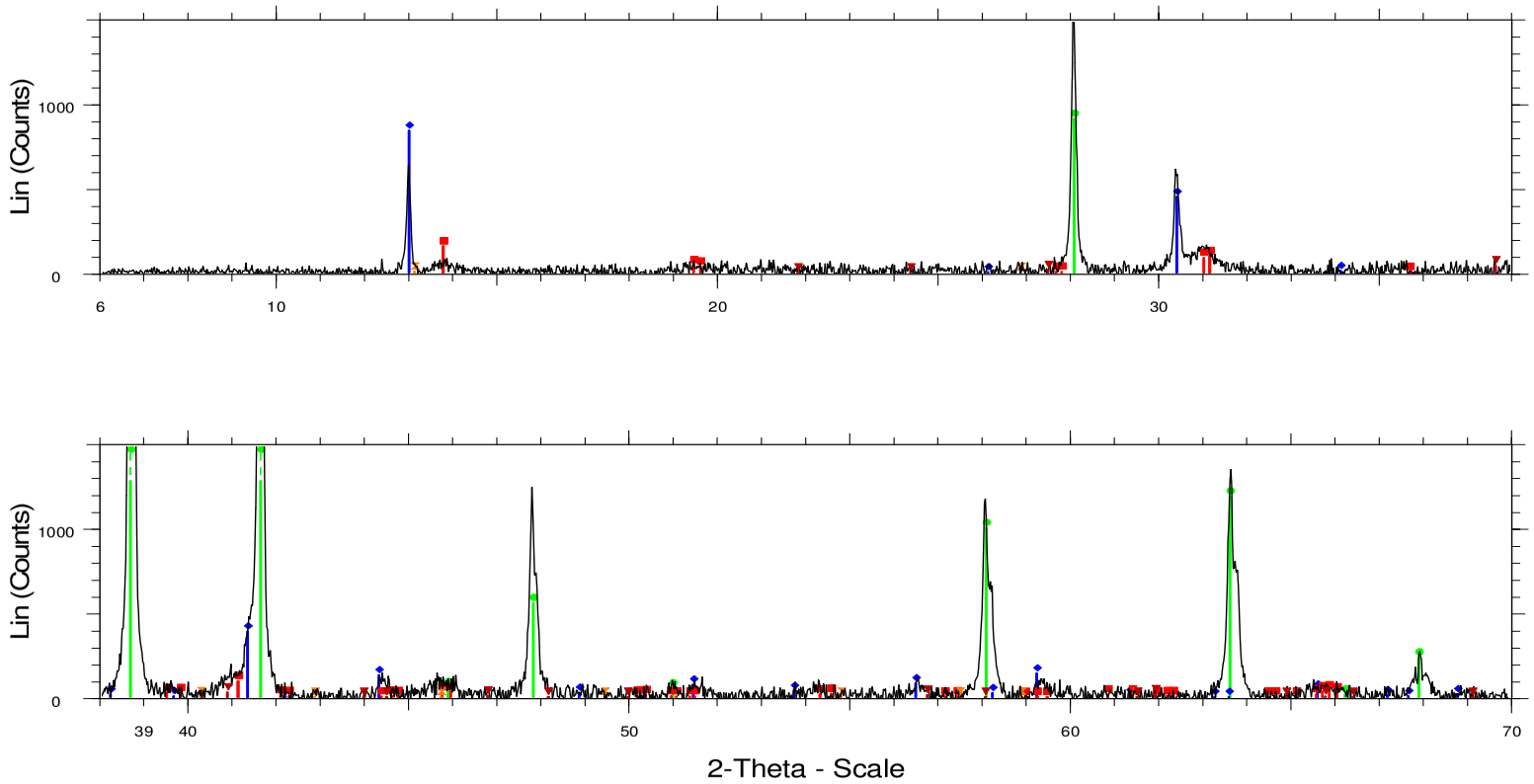
Test 12 Reactor Solids

Name	Assay[1]	SQD[2]	Delta	Status
Iron	62.9	65.4	-2.48	Both
Chlorine	8.22	5.81	2.41	Both
Magnesium	0.15	-	0.15	XRF
Copper	0.07	-	0.07	XRF
Nickel	0.04	0.15	-0.10	Both
Hydrogen	-	0.12	-0.12	SQD
Oxygen	-	28.1	-28.1	SQD
Titanium	-	0.40	-0.40	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

Test 12 Reactor Solids



Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resouces*
Project Number/ LIMS No.	*12323-001/MI4507-JUN10*
Reporting Date:	*June 28, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
2) Summary of Mineral Asemblages
3) Semi-Quantitative XRD Results
4) Chemical Balance(s)
5) XRD Pattern(s)

Anita Coppaway
Mineralogical Technologist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

Interpretations do not reflect the presence of non-crystalline and/or amorphous compounds. Mineral proportions are based on relative peak heights and may be strongly influenced by crystallinity, structural group or preferred orientations. Interpretations and relative proportions should be accompanied by supporting petrographic and geochemical data (Whole Rock Analysis, Inductively Coupled Plasma - Optical Emission Spectroscopy, etc.).

Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Test 13 Reactor Solids	akaganeite, hematite	-	iron oxide chloride	*rokuhnite, *iron vanadium oxide, *nickel chloride hydrate

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Akaganeite	$Fe_{7.6}Ni_{0.4}O_{6.4}(OH)_{9.7}Cl_{1.3}$
Hematite	Fe_2O_3
Iron Oxide Chloride	FeOCl
Iron Vanadium Oxide	$FeVO_4/(Fe_2O_3{\cdot}V_2O_5)$
Nickel Chloride Hydrate	$NiCl_2(H_2O)_2$
Rokuhnite	$FeCl_2{\cdot}2H_2O$

Semi-Quantitative X-ray Diffraction Results

Mineral	**Test 13 Reactor Solids** (wt %)
Akaganeite-M	53.0
Hematite	35.5
Iron Oxide Chloride	7.8
Rokuhnite, syn	2.0
Iron Vanadium Oxide	1.2
Nickel Chloride Hydrate	0.5
TOTAL	100.0

Chemical Balance

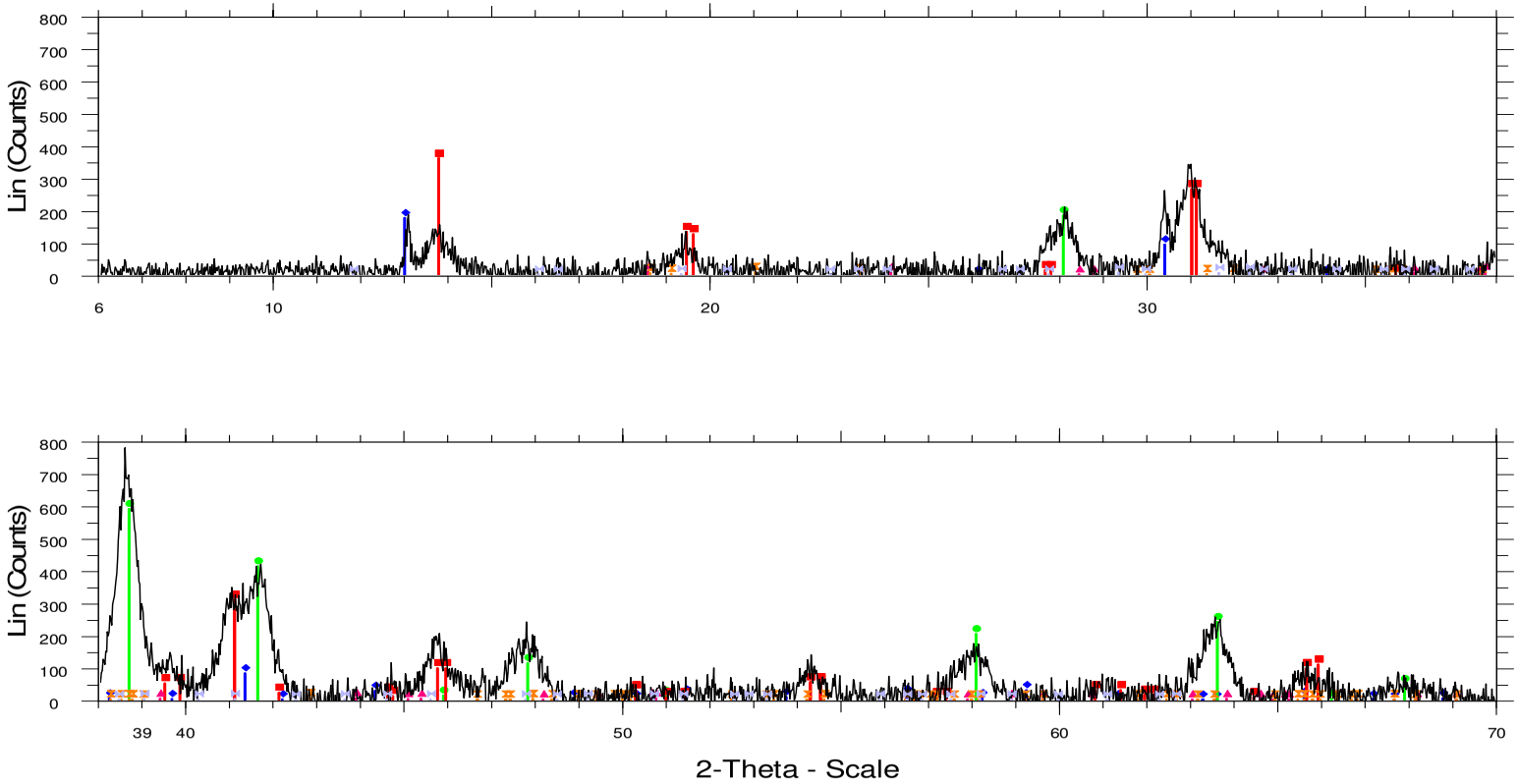
Test 13 Reactor Solids

Name	Assay[1]	SQD[2]	Delta	Status
Iron	52.1	61.2	-9.13	Both
Oxygen	22.8	30.7	-7.90	Both
Chlorine	13.1	6.89	6.21	Both
Vanadium	0.21	0.37	-0.16	Both
Silicon	0.15	-	0.15	XRF
Titanium	0.05	-	0.05	XRF
Copper	0.04	-	0.04	XRF
Nickel	0.03	0.18	-0.15	Both
Aluminum	0.03	-	0.03	XRF
Chromium	0.01	-	0.01	XRF
Phosphorus	0.01	-	0.01	XRF
Hydrogen	-	0.63	-0.63	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

Test 13 Reactor Solids



Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resources*
Project Number/ LIMS No.	*12323-001/MI4509-JUN10*
Reporting Date:	*July 16, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

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2) Summary of Mineral Asemblages
3) Semi-Quantitative XRD Results
4) Chemical Balance(s)
5) XRD Pattern(s)

Anita Coppaway
Mineralogical Technologist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

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Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Test 14 Reactor Solids	hematite	-	-	*magnetite, *iron nickel chloride hydrate, *iron vanadium oxide

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Hematite	Fe_2O_3
Iron Nickel Chloride Hydrate	$Ni_{0.5}Fe_{0.5}Cl_2{\cdot}4H_2O$
Iron Vanadium Oxide	$FeVO_4/(Fe_2O_3{\cdot}V_2O_5)$
Magnetite	Fe_3O_4

Semi-Quantitative X-ray Diffraction Results

Mineral	**Test 14 Reactor Solids** (wt %)
Hematite	98.0
Magnetite, titanian	0.5
Iron Vanadium Oxide	1.0
Iron Nickel Chloride Hydrate	0.5
TOTAL	100.0

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

Chemical Balance

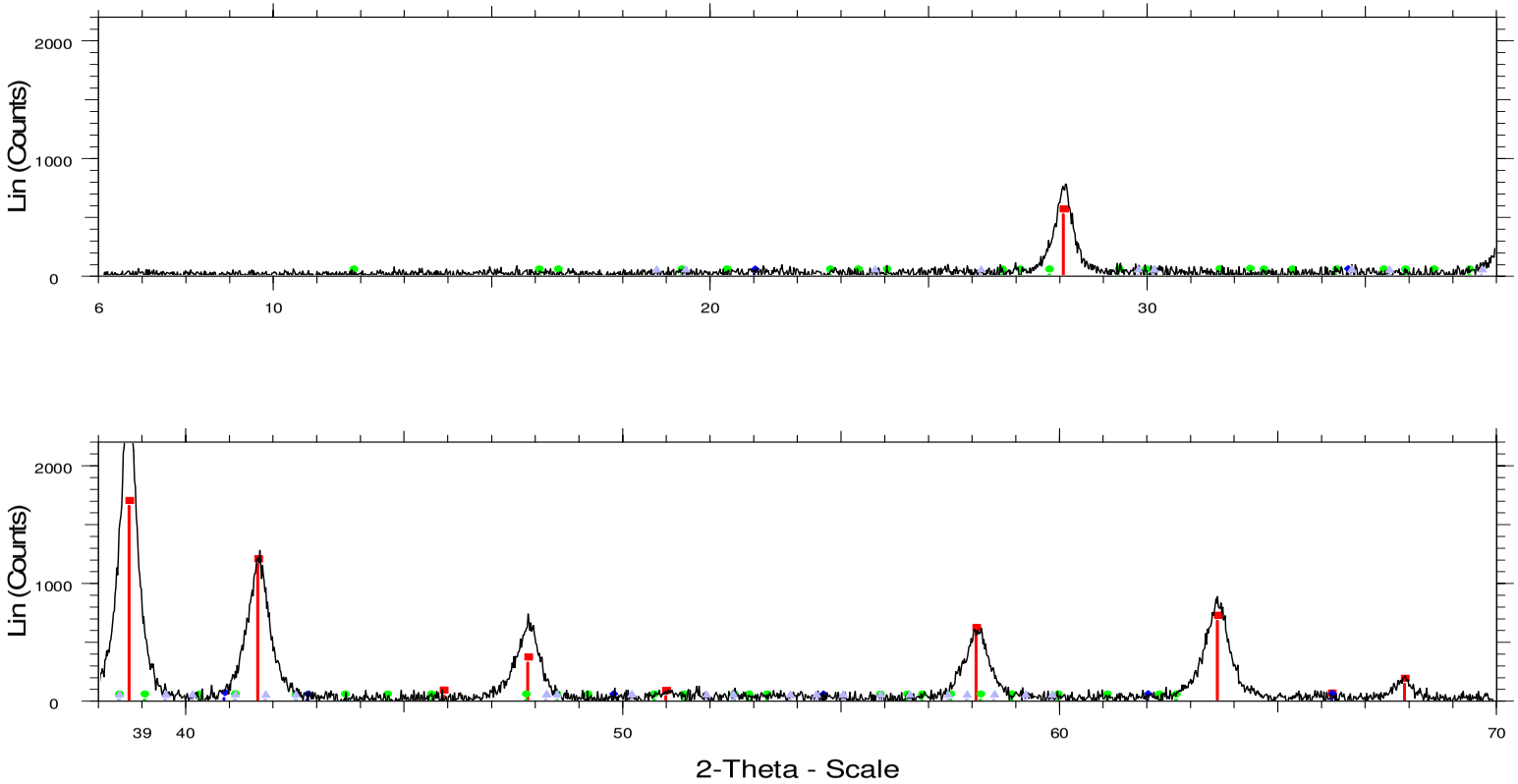
Test 14 Reactor Solids

Name	Assay[1]	SQD[2]	Delta	Status
Iron	66.4	69.2	-2.82	Both
Oxygen	28.9	30.1	-1.23	Both
Chlorine	1.40	0.19	1.21	Both
Vanadium	0.31	0.29	0.02	Both
Titanium	0.07	0.08	-0.01	Both
Silicon	0.05	-	0.05	XRF
Manganese	0.05	-	0.05	XRF
Chromium	0.01	-	0.01	XRF
Nickel	0.01	0.08	-0.07	Both
Hydrogen	-	0.02	-0.02	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

Test 14 Reactor Solids



Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resources*
Project Number/ LIMS No.	*12323-001/MI4512-JUN10*
Reporting Date:	*July 15, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
2) Summary of Mineral Asemblages
3) Semi-Quantitative XRD Results
4) Chemical Balance(s)
5) XRD Pattern(s)

Anita Coppaway
Mineralogical Technologist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

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Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Test 15 Reactor Solids	hematite	-	-	*rohuhnite, *iron vanadium oxide, *iron chloride hydrate

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Hematite	Fe_2O_3
Iron Chloride Hydrate	$FeCl_2 \cdot 4H_2O$
Iron Vanadium Oxide	$FeVO_4/(Fe_2O_3 \cdot V_2O_5)$
Rokuhnite	$FeCl_2 \cdot 2H_2O$

Semi-Quantitative X-ray Diffraction Results

Mineral	**Test 15 Reactor Solids** (wt %)
Hematite	96.4
Rokuhnite	1.9
Iron Chloride Hydrate	1.1
Iron Vanadium Oxide	0.6
TOTAL	100.0

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

Chemical Balance

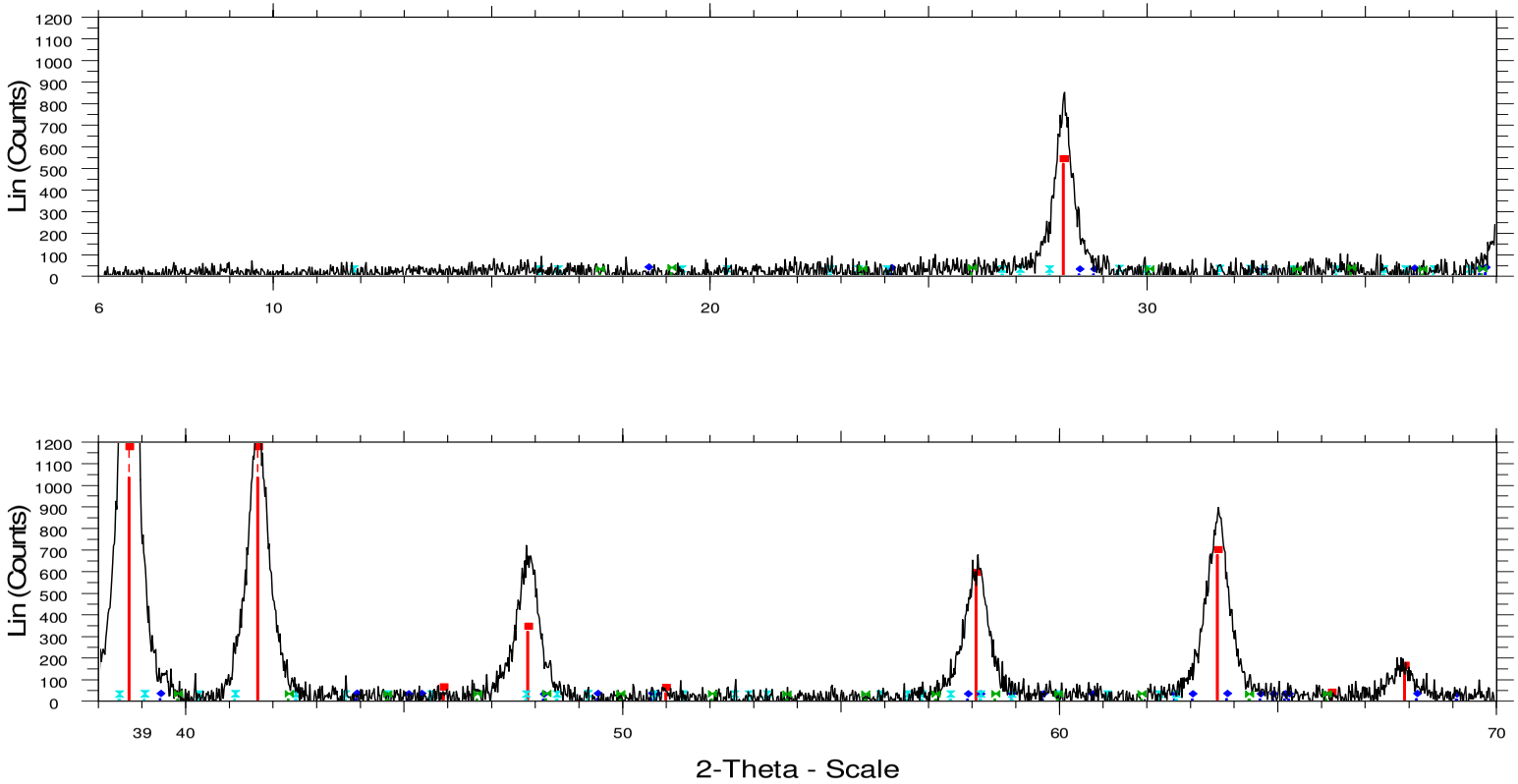
Test 15 Reactor Solids

Name	Assay[1]	SQD[2]	Delta	Status
Iron	65.7	68.6	-2.85	Both
Oxygen	28.6	29.9	-1.32	Both
Chlorine	1.32	1.21	0.11	Both
Vanadium	0.22	0.18	0.04	Both
Silicon	0.07	-	0.07	XRF
Titanium	0.04	-	0.04	XRF
Magnesium	0.03	-	0.03	XRF
Aluminum	0.03	-	0.03	XRF
Phosphorus	0.02	-	0.02	XRF
Chromium	0.01	-	0.01	XRF
Hydrogen	-	0.09	0.09	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

Test 15 Reactor Solids



Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resources*
Project Number/ LIMS No.	*12323-001/MI4503-JUL10*
Reporting Date:	*August 16, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
2) Summary of Mineral Asemblages
3) Semi-Quantitative XRD Results
4) Chemical Balance(s)
5) XRD Pattern(s)

Anita Coppaway
Mineralogical Technologist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

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Interpretations do not reflect the presence of non-crystalline and/or amorphous compounds. Mineral proportions are based on relative peak heights and may be strongly influenced by crystallinity, structural group or preferred orientations. Interpretations and relative proportions should be accompanied by supporting petrographic and geochemical data (Whole Rock Analysis, Inductively Coupled Plasma - Optical Emission Spectroscopy, etc.).

Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
Test 17 Reactor Solids	hematite		rokuhnite, siderite, bernalite	*quartz, *karelianite

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Bernalite	$Fe(OH)_3$
Hematite	Fe_2O_3
Hydroniumjarosite	$(H_3O)Fe(SO_4)_2(OH)_6$
Karelianite	V_2O_3
Quartz	SiO_2
Rokuhnite	$FeCl_2 \cdot (H_2O)$
Siderite	$FeCO_3$

Semi-Quantitative X-ray Diffraction Results

Mineral	**Test 17 Reactor Solids** (wt %)
Hematite	88.2
Rokuhnite	4.9
Siderite	3.2
Bernalite	2.8
Quartz	0.5
Karelianite	0.4
TOTAL	100.0

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

Chemical Balance

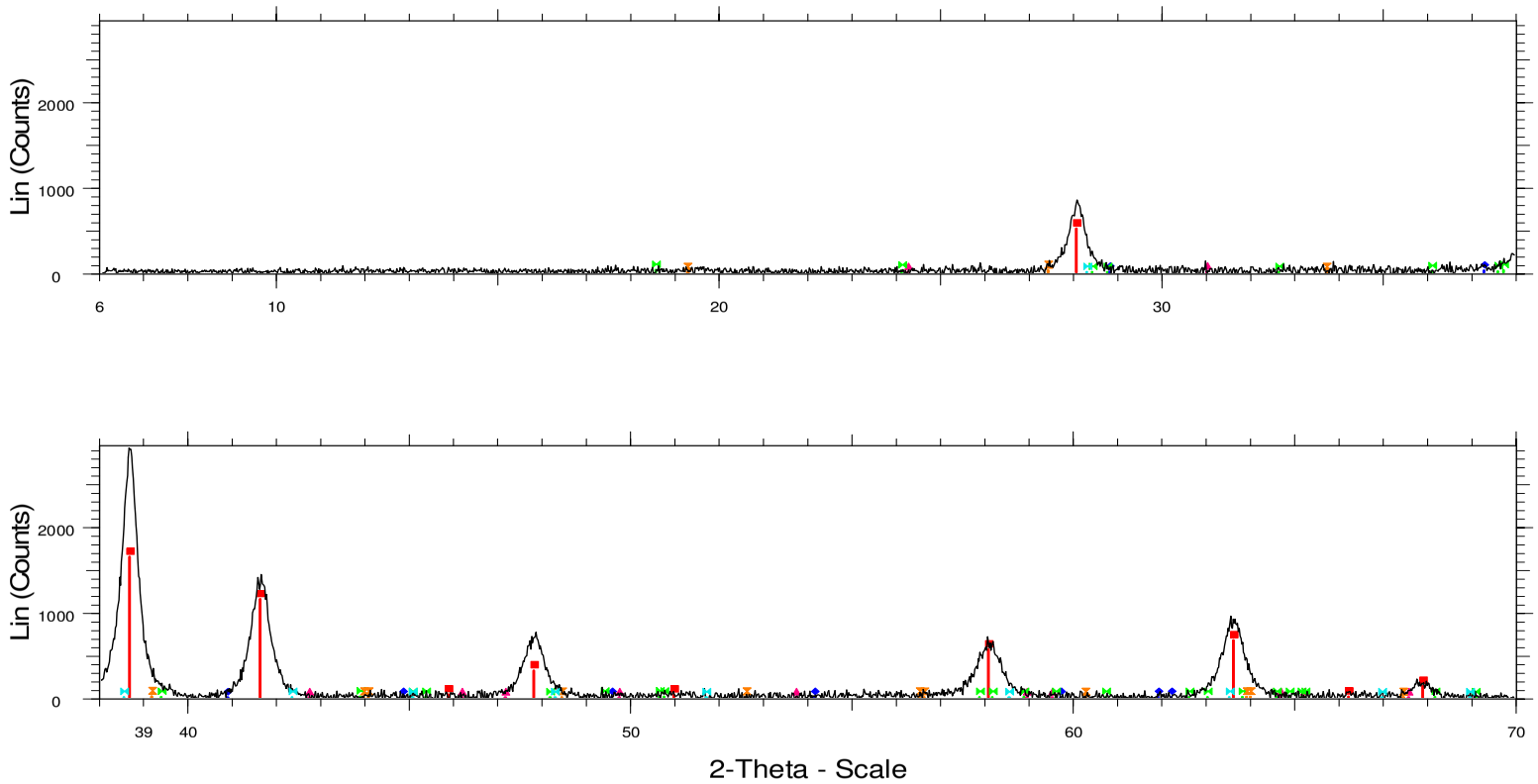
Test 17 Reactor Solids

Name	Assay[1]	SQD[2]	Delta	Status
Iron	66.0	66.4	-0.33	Both
Oxygen	28.6	30.5	-1.86	Both
Chlorine	1.50	2.12	-0.63	Both
Vanadium	0.20	0.24	-0.04	Both
Silicon	0.07	0.22	-0.15	Both
Titanium	0.04	-	0.04	XRF
Copper	0.02	-	0.02	XRF
Hydrogen	-	0.21	-0.21	SQD
Carbon	-	0.34	-0.34	SQD

1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

Test 17 Reactor Solids



Test 17 Reactor Solids - File: Jul4503-1.raw
Operations: Background 1.000,1.000 | Import
01-072-0469 (C) - Hematite - Fe2O3
00-029-0696 (*) - Siderite - FeCO3
01-079-1910 (C) - Quartz - SiO2
00-025-1040 (*) - Rokuhnite, syn - FeCl2·2H2O
01-081-2022 (C) - Bernalite - Fe(OH)3(H2O)0.25
00-034-0187 (*) - Karelianite, syn - V2O3

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

Semi-Quantitative X-Ray Diffraction

Report Prepared for:	*Starfield Resources*
Project Number/ LIMS No.	*12323-001/MI4505-AUG10*
Reporting Date:	*August 25, 2010*

Instrument:	BRUKER AXS D8 Advance Diffractometer
Test Conditions:	Co radiation, 40 kV, 35 mA Regular Scanning: Step: 0.02°, Step time:0.2s, 2θ range: 3-70°
Interpretations:	PDF2/PDF4 powder diffraction databases issued by the International Center for Diffraction Data (ICDD). DiffracPlus Eva software.
Detection Limit:	0.5-2%. Strongly dependent on crystallinity.

Contents:

1) Method Summary
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5) XRD Pattern(s)

Anita Coppaway
Mineralogical Technologist

Huyun Zhou, Ph.D.
Senior Mineralogist

Method Summary

Mineral Identification and Interpretation:

Mineral identification and interpretation involve matching the diffraction pattern of an unknown material to patterns of single-phase reference materials. The reference patterns are compiled by the Joint Committee on Powder Diffraction Standards - International Center for Diffraction Data (JCPDS-ICDD) database and released on software as Powder Diffraction Files (PDF).

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Semi-Quantitative Analysis:

The Semi-Quantitative analysis (RIR method) is performed based on each mineral's relative peak heights and of their respective I/Icor values, which are available from the PDF database. Mineral abundances for the bulk sample (in weight %) are generated by Bruker-EVA Software. These data are reconciled with a bulk chemistry (e.g. whole rock analysis including SiO_2, Al_2O_3, Na_2O, K_2O, CaO, MgO, Fe_2O_3, Cr_2O_3, MnO, TiO_2, P_2O_5, V_2O_5 or other chemical data). A chemical balance table shows the difference between the assay results and elemental concentrations determined by XRD.

Summary of Semi-Quantitative X-ray Diffraction Results

Crystalline Mineral Assemblage (relative proportions based on peak height)

Sample	Major (>30% Wt)	Moderate (10% -30% Wt)	Minor (2% -10% Wt)	Trace (<2% Wt)
(1) PP2-Cond 4 Solids	hematite	-	-	*rokuhnite, *ilmenite
(2) PP2-Cond 5 Solids	hematite	-	-	*rokuhnite, *ilmenite

** tentative identification due to low concentrations, diffraction line overlap or poor crystallinity*

Mineral	Composition
Hematite	Fe_2O_3
Ilmenite	$FeTiO_3$
Rokuhnite	$FeCl_2 \cdot (H_2O)$

Semi-Quantitative X-ray Diffraction Results

Mineral	**(1) PP2-Cond 4 Solids** (wt %)	**(2) PP2-Cond 5 Solids** (wt %)
Hematite	97.5	97.5
Rokuhnite	2.0	2.0
Ilmenite	0.5	0.5
Bernalite	-	-
TOTAL	100.0	100.0

The Qualitative XRD method (METH # 8-8-1) used by SGS Minerals Services, P.O. Box 4300, 185 Concession Street, Lakefield, Ontario, Canada K0L 2H0.
Tel: (705) 652-2000 Fax: (705) 652-6365 Mini-method available upon request.

Chemical Balance

(1) PP2-Cond 4 Solids

Name	Assay[1]	SQD[2]	Delta	Status
Iron	67.0	69.1	-2.06	Both
Chlorine	1.00	0.88	0.12	Both
Hydrogen	-	0.05	-0.05	SQD
Oxygen	-	29.9	-29.9	SQD
Titanium	-	0.15	-0.15	SQD

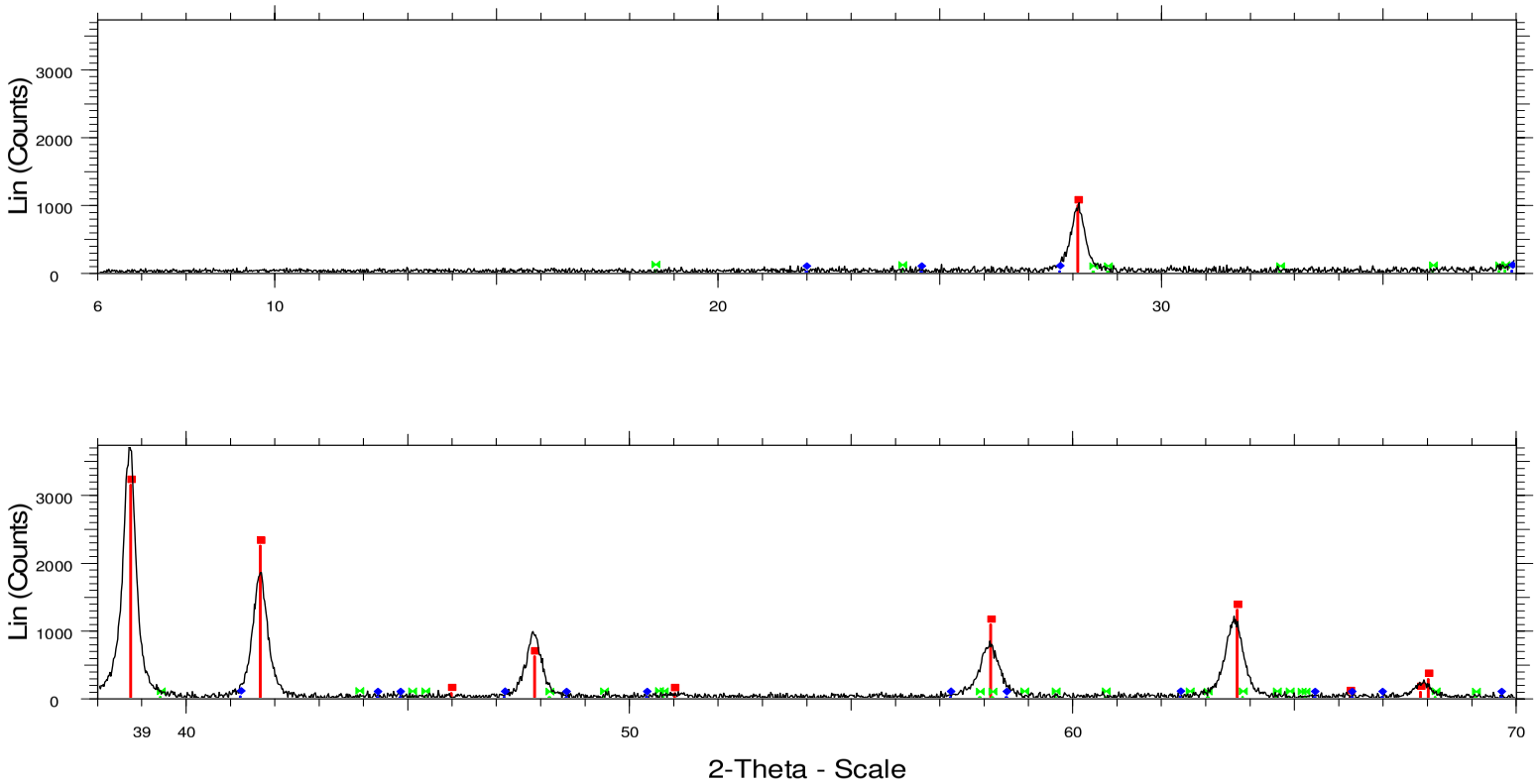
(2) PP2-Cond 5 Solids

Name	Assay[1]	SQD[2]	Delta	Status
Iron	67.0	69.1	-2.07	Both
Chlorine	1.00	0.87	0.13	Both
Hydrogen	-	0.05	-0.05	SQD
Oxygen	-	29.9	-29.9	SQD
Titanium	-	0.15	-0.15	SQD

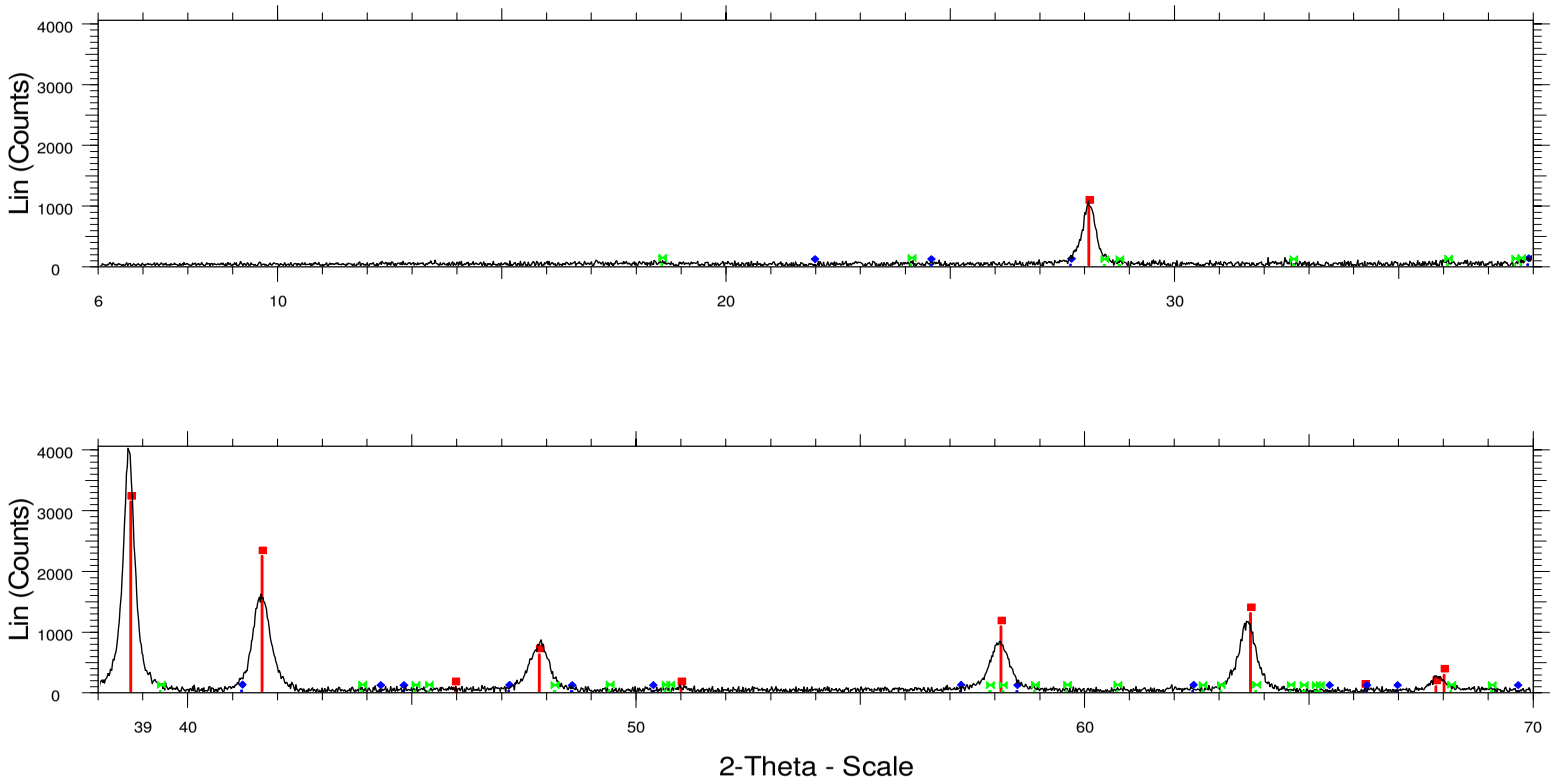
1. Values measured by chemical assay.

2. Values calculated based on mineral/compound formulas and quantites identified by semi-quantitative XRD.

PP2-Cond 4 Solids



PP2-Cond 5 Solids



Appendix F - Hematite acid washing

Tests details

Project: 12323-001 **Date:** 24-Aug
Client: Starfield Resources (Ferguson Lake) **Technologist(s):** L.Smith

Test: **CW-1**

Purpose: **To determine the effect of Hot HCl washing on a hematite sample with ~1% chloride.**

Sample: Final Reactor Solids 11 %H2O **Grind:** as received

Procedure:

1. A volume of HCl solution (10 g/L), as calculated was placed in a 1L reactor with the hematite sample. The reactor was covered with a lid and heated to 50℃ with a heating mantle.
2. The solution was mixed with a mechanical mixer at speed of 250 RPM
3. At the end of the test the pulp was filtered.
4. The PLS was collected and a sample submitted for analysis.
5. The filtercake was repulped 2X, first with 450 mL DI water then with 300 mL with DI water.
6. The filtercake was then displacement washed 3X 150 mL with DI water.
7. The wash solutions were combined and volume determined.
8. The washed filtercake was dried, weighed and submitted for analysis.

Conditions:

Targets

Feed Weight (g):	75.0
HCl solution (mL):	300
HCl strenght (g/L):	10
Leach Time (min):	120
Temperature (℃):	50

Actual

Wet Feed Weight (g):	84.3
HCl solution (mL):	300
HCl solution (g):	305
HCl strenght (g/L):	11.5

Leach Data:

Actual Time	Elapsed Time(min)	Temp (℃)	Notes
9:45			Start mix start heat
9:55	0	51.9	At temperature, start reaction time
10:10	15	61.5	
10:25	30	59.3	
10:55	60	49.2	
11:55	120	53.3	End test

Results:

Filtration (fast, slow):	fast	
Final Pulp wt:	373.05	*g*
Calculated Filtrate Volume:	298.2	*mL*
Collected Filtrate Wt.:	278.4	*g*
Filtrate SG:	1.007	
Filtrate Volume:	276.5	*mL*
Filtrate pH	1.19	
Filtrate ORP	510	*mV*
Filtrate Colour:	Light yellow, clear	
Wash Soln. Wt.:	1209.98	*g*
Wash Soln. Volume:	1205	*mL*
Wash Colour	colourless, clear	
Dry wt solids, residue:	72.7	*g*
Residue colour	eggplant	

Filtration Details

Filtration Time	0.5	min
Washing Time	6.5	min
Paper Type	3	
Paper Size	125	*mm*
Cake Thickeness	50	*mm*

Tare wt =	20.59	g
Wet wt =	107.6	g
Dry wt =	93.32	g
% Moisture =	16.4	%

Samples for assay:

Filtrate:	Fe
Solids:	Cl

Notes:

Project: 12323-001 **Date:** 24-Aug
Client: Starfield Resources (Ferguson Lake) **Technologist(s):** L.Smith

Test: **CW-2**

Purpose: **To determine the effect of Hot HCl washing on a hematite sample with ~1% chloride.**

Sample: Condition 4 Solids 9 %H2O **Grind:** as received

Procedure:

1. A volume of HCl solution (10 g/L), as calculated was placed in a 1L reactor with the hematite sample. The reactor was covered with a lid and heated to 50℃ with a heating mantle.
2. The solution was mixed with a mechanical mixer at speed of 250 RPM
3. At the end of the test the pulp was filtered.
4. The PLS was collected and a sample submitted for analysis.
5. The filtercake was repulped 2X, first with 450 mL DI water then with 300 mL with DI water.
6. The filtercake was then displacement washed 3X 150 mL with DI water.
7. The wash solutions were combined and volume determined.
8. The washed filtercake was dried, weighed and submitted for analysis.

Conditions:

Targets

Feed Weight (g):	75.0
HCl solution (mL):	300
HCl strenght (g/L):	50
Leach Time (min):	120
Temperature (℃):	50

Actual

Wet Feed Weight (g):	82.4
HCl solution (mL):	300
HCl solution (g):	306
HCl strenght (g/L):	55.8

Leach Data:

Actual Time	Elapsed Time(min)	Temp (℃)	Notes
7:00			Start mix start heat
7:15	0	49.6	At temperature, start reaction time
7:30	15	58.4	
7:45	30	53.7	
8:15	60	50.7	
9:15	120	51.3	End test

Results:

Filtration (fast, slow):	fast	
Final Pulp wt:	374.64	*g*
Calculated Filtrate Volume:	293.3	*mL*
Collected Filtrate Wt.:	272.6	*g*
Filtrate SG:	1.1	
Filtrate Volume:	259.6	*mL*
Filtrate pH	0.85	
Filtrate ORP	575	*mV*
Filtrate Colour:	yellow, clear	
Wash Soln. Wt.:	1134.73	*g*
Wash Soln. Volume:	1130	*mL*
Wash Colour	yellow/orange, clear	
Dry wt solids, residue:	66.7	*g*
Residue colour	Red/Brown	

Filtration Details

Filtration Time	2 min	
Washing Time	9 min	
Paper Type	3	
Paper Size	125	*mm*
Cake Thickeness	50	*mm*

Tare wt =	20.92	g
Wet wt =	99.6	g
Dry wt =	87.59	g
% Moisture =	15.2	%

Samples for assay:

Filtrate:	Fe
Solids:	Cl

Notes:

Project: 12323-001 **Date:** 24-Aug
Client: Starfield Resources (Ferguson Lake) **Technologist(s):** L.Smith

Test: **CW-3**

Purpose: **To determine the effect of Hot HCl washing on a hematite sample with ~1% chloride.**

Sample: Condition 5 Solids 13 %H2O *Solids not homogenized* **Grind:** as received

Procedure:

1. A volume of HCl solution (10 g/L), as calculated was placed in a 1L reactor with the hematite sample. The reactor was covered with a lid and heated to 50℃ with a heating mantle.
2. The solution was mixed with a mechanical mixer at speed of 250 RPM
3. At the end of the test the pulp was filtered.
4. The PLS was collected and a sample submitted for analysis.
5. The filtercake was repulped 2X, first with 450 mL DI water then with 300 mL with DI water.
6. The filtercake was then displacement washed 3X 150 mL with DI water.
7. The wash solutions were combined and volume determined.
8. The washed filtercake was dried, weighed and submitted for analysis.

Conditions:

Targets

Feed Weight (g):	75.0
HCl solution (mL):	300
HCl strenght (g/L):	100
Leach Time (min):	120
Temperature (℃):	50

Actual

Wet Feed Weight (g):	86.8
HCl solution (mL):	300
HCl solution (g):	311
HCl strenght (g/L):	105.7

Leach Data:

Actual Time	Elapsed Time(min)	Temp (℃)	Notes
10:20			Start mix start heat
10:30	0	57.4	At temperature, start reaction time
10:45	15	63.6	
11:00	30	58.7	
11:30	60	49.4	
12:30	120	51.8	End test

Results:

Filtration (fast, slow):	fast	
Final Pulp wt:	378.73	*g*
Calculated Filtrate Volume:	289.8	*mL*
Collected Filtrate Wt.:	299.4	*g*
Filtrate SG:	1.11	
Filtrate Volume:	269.2	*mL*
Filtrate pH	0.82	
Filtrate ORP	575	*mV*
Filtrate Colour:	amber, clear	
Wash Soln. Wt.:	1261.88	*g*
Wash Soln. Volume:	1255	*mL*
Wash Colour	orange, opaque	
Dry wt solids, residue:	56.4	*g*
Residue colour	Red/black	

Filtration Details

Filtration Time	1.5 min	
Washing Time	11 min	
Paper Type	3	
Paper Size	120	*mm*
Cake Thickness	50	*mm*

Tare wt =	19.36	g
Wet wt =	91.1	g
Dry wt =	75.79	g
% Moisture =	21.4	%

Samples for assay:

Filtrate:	Fe
Solids:	Cl

Notes:

Project: 12323-001 **Date:** 24-Aug
Client: Starfield Resources (Ferguson Lake) **Technologist(s):** L.Smith

Test: **CW-4**

Purpose: **To determine the effect of Hot HCl washing on a hematite sample with ~1% chloride.**

Sample: Condition 4 Solids 9 %H2O **Grind:** as received

Procedure:

1. A volume of HCl solution (10 g/L), as calculated was placed in a 1L reactor with the hematite sample.
   The reactor was covered with a lid and equipped with a condenser, then heated to 95℃ with a heating mantle .
2. The solution was mixed with a mechanical mixer at speed of 250 RPM
3. At the end of the test the pulp was filtered.
4. The PLS was collected and a sample submitted for analysis.
5. The filtercake was repulped 2X, first with 450 mL DI water then with 300 mL with DI water.
6. The filtercake was then displacement washed 3X 150 mL with DI water.
7. The wash solutions were combined and volume determined.
8. The washed filtercake was dried, weighed and submitted for analysis.

Conditions:

Targets

Feed Weight (g):	75.0
HCl solution (mL):	300
HCl strenght (g/L):	50
Leach Time (min):	120
Temperature (℃):	95

Actual

Wet Feed Weight (g):	82.6
HCl solution (mL):	300
HCl solution (g):	306
HCl strenght (g/L):	55.8

Leach Data:

Actual Time	**Elapsed Time(min)**	**Temp** (℃)	**Notes**
7:15			Start mix start heat
7:45	0	91.5	At temperature, start reaction time
8:00	15	94.4	
8:15	30	94.7	
8:45	60	95.1	
9:45	120	94.9	End test

Results:

Filtration (fast, slow):	fast	
Final Pulp wt:	360.51	*g*
Calculated Filtrate Volume:	282.0	*mL*
Collected Filtrate Wt.:	274.6	*g*
Filtrate SG:	1.062	
Filtrate Volume:	258.6	*mL*
Filtrate pH	0.68	
Filtrate ORP, mV	590	*mV*
Filtrate Colour:	Light yellow, clear	
Wash Soln. Wt.:	1237.13	*g*
Wash Soln. Volume:	1225	*mL*
Wash Colour	orange, clear	*mL*
Dry wt solids, residue:	61.1	*g*
Residue colour	Red/Brown	

Filtration Details

Filtration Time	1.5	min
Washing Time	17.5	min
Paper Type	3	
Paper Size	125	*mm*
Cake Thickeness	50	*mm*

Tare wt =	21.09	g
Wet wt =	97.2	g
Dry wt =	82.14	g
% Moisture =	19.8	%

Samples for assay:

Filtrate:	Fe
Solids:	Cl

Notes:

Appendix G - PSD analysis

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 7 Final solids - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
Ernesto

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Friday, April 16, 2010 10:13:50 AM

Analysed:
Friday, April 16, 2010 10:13:51 AM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
0.749 %

Sensitivity:
Enhanced

Obscuration:
5.55 %

Result Emulation:
Off

Concentration:
0.0051 %Vol

Specific Surface Area:
0.89 m²/g

Span :
1.026

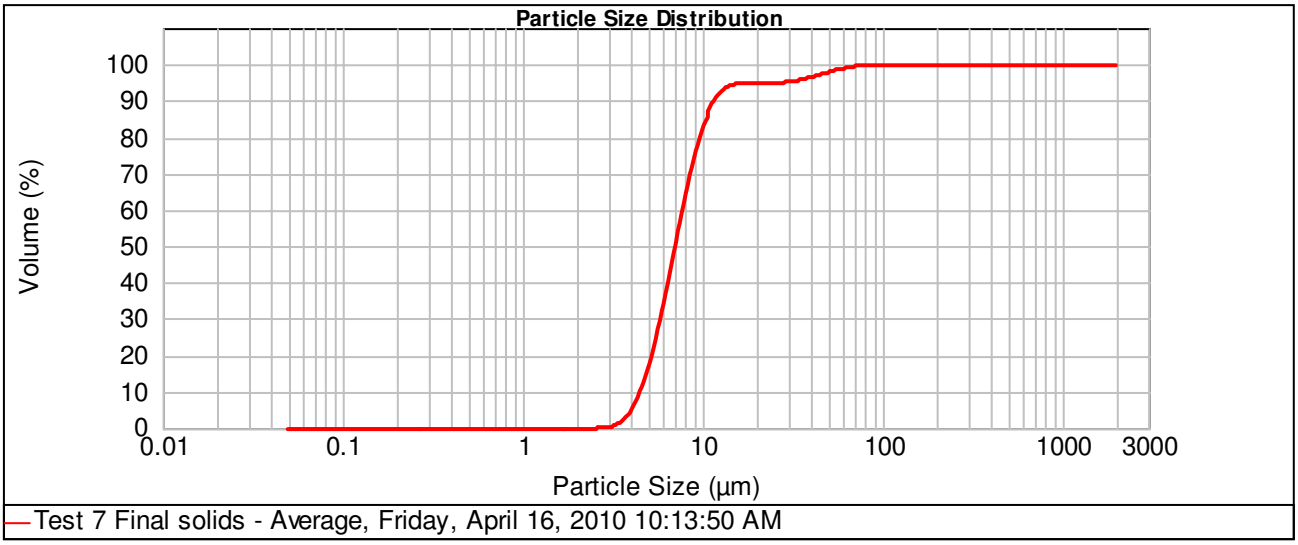
Surface Weighted Mean D[3,2]:
6.739 um

Uniformity:
0.554

Vol. Weighted Mean D[4,3]:
9.252 um

Result units:
Volume

d(0.1): 4.468 um **d(0.5):** 6.952 um **D(0.80) : 9.52 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	89.60	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	93.15	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	94.56	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	94.98	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	95.04	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	95.04	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	95.06	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.12	30.200	95.28	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.88	34.674	95.82	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	3.33	39.811	96.59	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	8.77	45.709	97.46	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	17.30	52.481	98.30	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	29.73	60.256	99.00	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	44.37	69.183	99.50	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	59.45	79.433	99.82	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	72.87	91.201	99.96	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	83.07	104.713	100.00	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 6 Final solids - Average

SOP Name:
Hematite (red light only)

Measured:
Wednesday, April 14, 2010 11:49:25 AM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Wednesday, April 14, 2010 11:49:26 AM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
5.52 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.805 %

Result Emulation:
Off

Concentration:
0.0056 %Vol

Span :
1.616

Uniformity:
0.668

Result units:
Volume

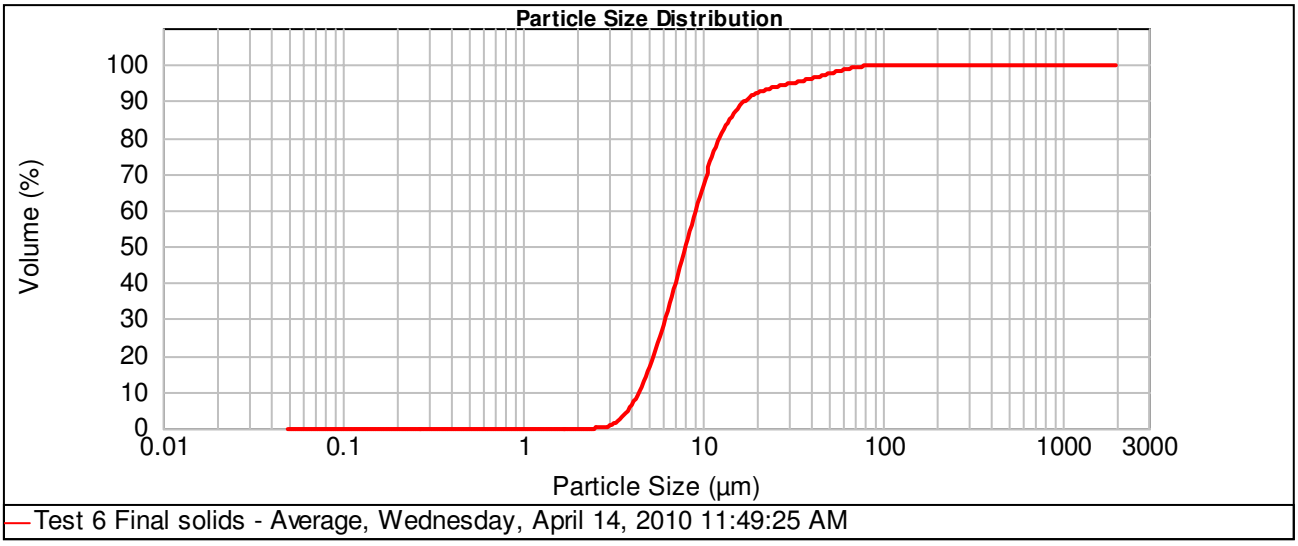
Specific Surface Area:
0.807 m²/g

Surface Weighted Mean D[3,2]:
7.434 um

Vol. Weighted Mean D[4,3]:
11.035 um

---

d(0.1): 4.423 um **d(0.5):** 7.963 um **D(0.80) : 12.57 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	75.40	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	82.07	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	86.88	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	90.09	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	92.08	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	93.27	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	94.05	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.26	30.200	94.70	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	1.62	34.674	95.38	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	4.58	39.811	96.15	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	9.43	45.709	96.97	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	16.31	52.481	97.80	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	25.05	60.256	98.55	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	35.23	69.183	99.17	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	46.14	79.433	99.62	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	56.97	91.201	99.91	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	66.93	104.713	99.99	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP1 - Cond 1 solids - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
CF

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Tuesday, April 27, 2010 10:55:47 PM

Analysed:
Tuesday, April 27, 2010 10:55:48 PM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
1.145 %

Sensitivity:
Enhanced

Obscuration:
6.08 %

Result Emulation:
Off

Concentration:
0.0075 %Vol

Specific Surface Area:
0.668 m²/g

Span :
0.979

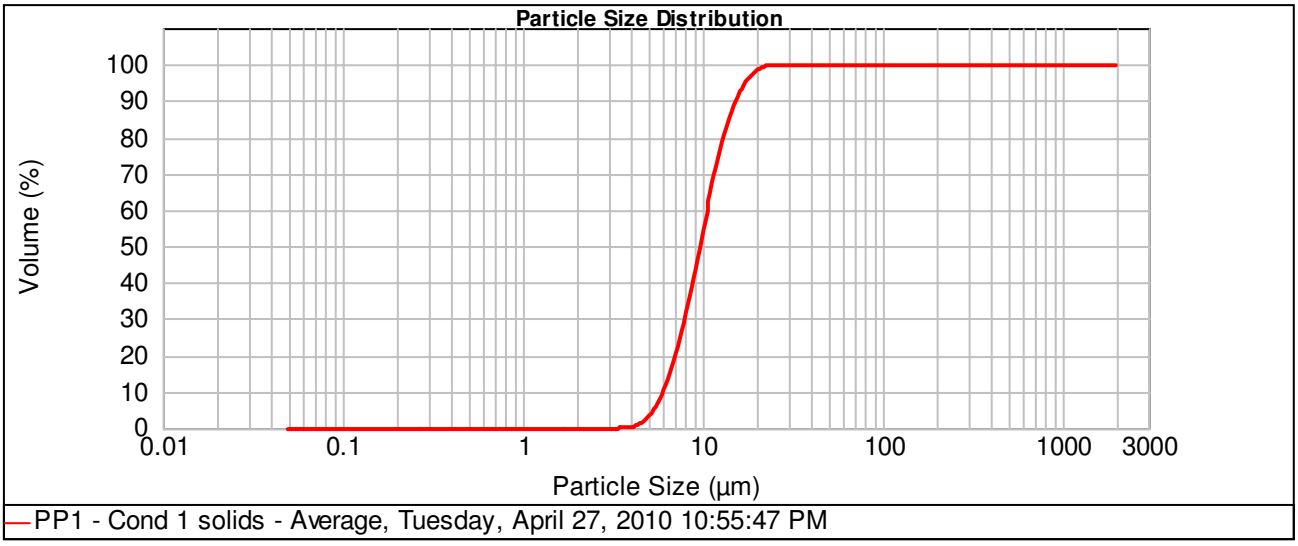
Surface Weighted Mean D[3,2]:
8.983 um

Uniformity:
0.305

Vol. Weighted Mean D[4,3]:
10.214 um

Result units:
Volume

d(0.1): 5.971 um **d(0.5):** 9.580 um **D(0.80) : 13.14 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	68.34	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	80.26	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	89.25	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	95.17	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	98.38	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	99.73	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	99.99	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	100.00	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.00	34.674	100.00	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	0.07	39.811	100.00	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	0.90	45.709	100.00	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	3.35	52.481	100.00	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	8.20	60.256	100.00	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	16.04	69.183	100.00	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	26.88	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	40.13	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	54.48	104.713	100.00	1096.478	100.00		

Operator notes: *Average of 2 measurements from MG2010.mea*

MALVERN INSTRUMENTS

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP1 - Cond 2 solids - Average

SOP Name:
Hematite (red light only)

Measured:
Wednesday, April 28, 2010 10:03:13 AM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Wednesday, April 28, 2010 10:03:14 AM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
5.73 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.685 %

Result Emulation:
Off

Concentration:
0.0098 %Vol

Span :
0.890

Uniformity:
0.278

Result units:
Volume

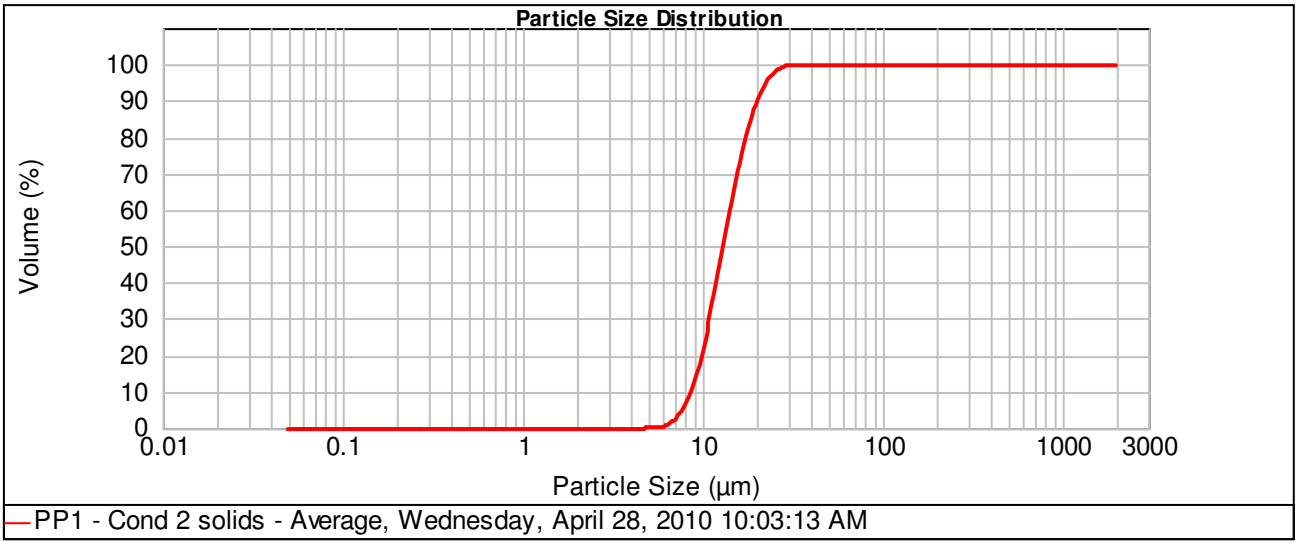
Specific Surface Area:
0.485 m²/g

Surface Weighted Mean D[3,2]:
12.377 um

Vol. Weighted Mean D[4,3]:
13.796 um

---

d(0.1): 8.487 um **d(0.5):** 13.067 um **D(0.80) : 17.43 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	35.42	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	51.01	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	66.47	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	79.76	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	89.55	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	95.65	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	98.70	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	99.82	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.00	34.674	100.00	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	0.00	39.811	100.00	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	0.00	45.709	100.00	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	0.01	52.481	100.00	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	0.21	60.256	100.00	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	1.54	69.183	100.00	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	4.93	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	11.50	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	21.77	104.713	100.00	1096.478	100.00		

Operator notes: *Average of 2 measurements from Arnie Jan 2010.mea*

Result Analysis Report

Sample Name:
PP1 - Cond 3 solids - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
Ernesto

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Tuesday, May 11, 2010 2:02:10 PM

Analysed:
Tuesday, May 11, 2010 2:02:11 PM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
1.485 %

Sensitivity:
Enhanced

Obscuration:
3.97 %

Result Emulation:
Off

Concentration:
0.0041 %Vol

Specific Surface Area:
0.774 m²/g

Span :
1.705

Surface Weighted Mean D[3,2]:
7.747 um

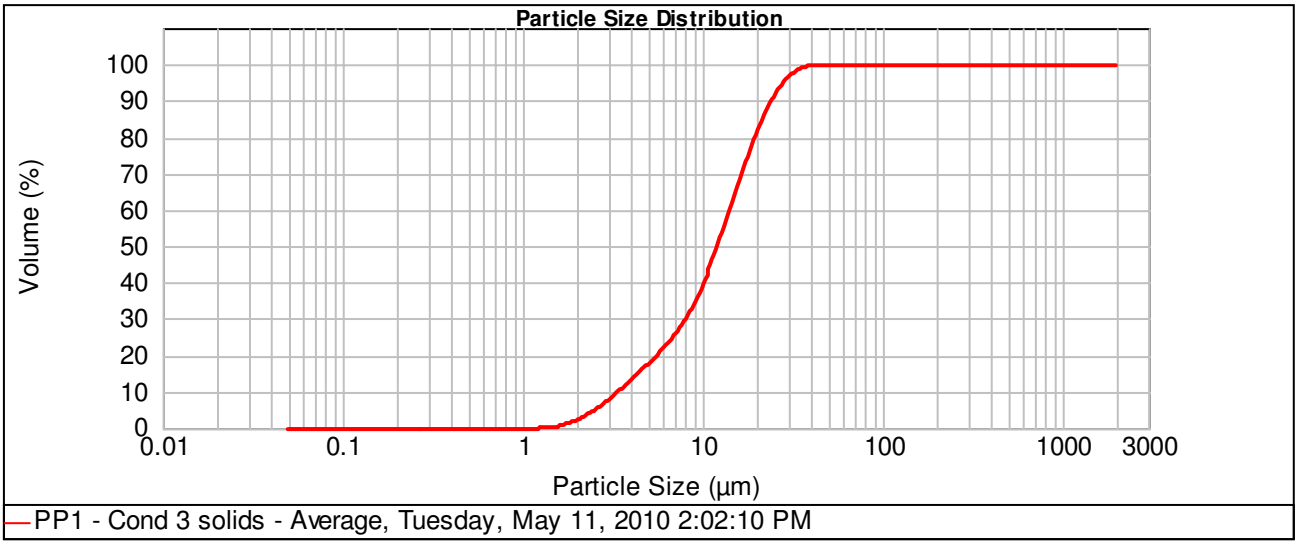
Uniformity:
0.524

Vol. Weighted Mean D[4,3]:
13.075 um

Result units:
Volume

---

d(0.1): 3.382 um **d(0.5):** 12.088 um **D(0.80) : 19.52 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	46.99	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.04	13.183	55.33	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.26	15.136	64.24	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.85	17.378	73.09	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	1.80	19.953	81.21	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	3.18	22.909	88.06	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	4.98	26.303	93.30	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	7.14	30.200	96.85	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	9.60	34.674	98.91	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	12.26	39.811	99.82	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	15.04	45.709	100.00	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	17.92	52.481	100.00	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	20.98	60.256	100.00	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	24.41	69.183	100.00	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	28.48	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	33.48	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	39.63	104.713	100.00	1096.478	100.00		

Operator notes: *Average of 2 measurements from MG 2010.mea*

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP1 - Cond 4 solids - Average

SOP Name:
Hematite (red light only)

Measured:
Tuesday, May 11, 2010 8:56:36 AM

Sample Source & type:
Factory = 12323-001

Measured by:
lr_hydro1

Analysed:
Friday, April 08, 2011 10:13:34 AM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Pyrrothite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
1.560

Absorption:
0.05

Size range:
0.020 to 2000.000 um

Obscuration:
3.92 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.889 %

Result Emulation:
Off

Concentration:
0.0042 %Vol

Span :
2.387

Uniformity:
0.762

Result units:
Volume

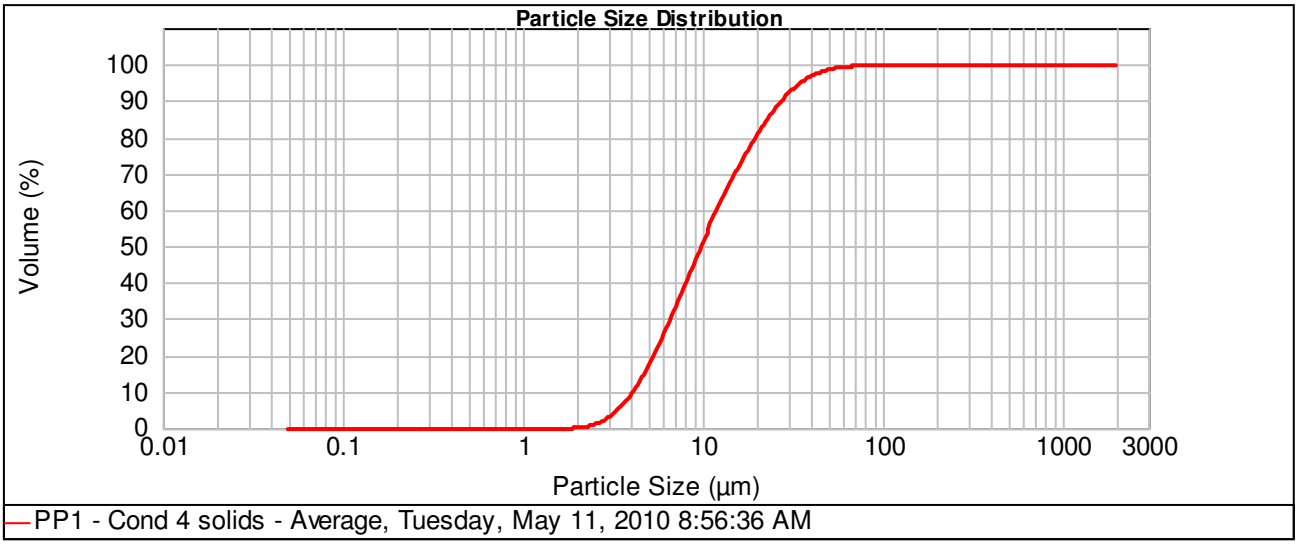
Specific Surface Area:
0.747 m²/g

Surface Weighted Mean D[3,2]:
8.036 um

Vol. Weighted Mean D[4,3]:
13.410 um

---

d(0.1): 4.090 um **d(0.5):** 9.728 um **D(0.80) : 19.72 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	57.89	120.226	99.94	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	64.07	138.038	99.98	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	69.89	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	75.35	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	80.42	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.26	22.909	85.01	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.96	26.303	89.02	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	2.34	30.200	92.34	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	4.58	34.674	94.94	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	7.84	39.811	96.83	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	12.16	45.709	98.11	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	17.48	52.481	98.90	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	23.63	60.256	99.36	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	30.38	69.183	99.60	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	37.42	79.433	99.75	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	44.48	91.201	99.83	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	51.35	104.713	99.89	1096.478	100.00		

Operator notes: *Average of 2 measurements from MG 2010.mea*

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP1 - Cond 5 solids - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
HG

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Monday, May 03, 2010 9:41:46 AM

Analysed:
Monday, May 03, 2010 9:41:47 AM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
0.763 %

Sensitivity:
Enhanced

Obscuration:
5.67 %

Result Emulation:
Off

Concentration:
0.0067 %Vol

Specific Surface Area:
0.689 m²/g

Span :
1.709

Surface Weighted Mean D[3,2]:
8.705 um

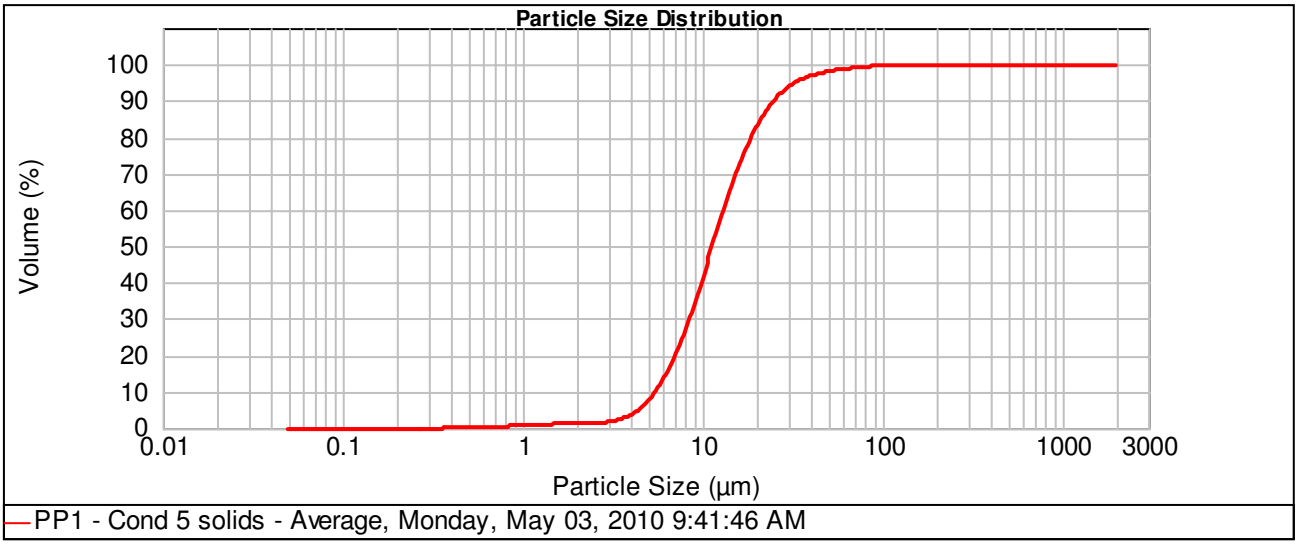
Uniformity:
0.603

Vol. Weighted Mean D[4,3]:
14.265 um

Result units:
Volume

---

d(0.1): 5.410 um **d(0.5):** 11.296 um **D(0.80) : 18.65 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.85	11.482	51.12	120.226	99.84	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.98	13.183	60.41	138.038	99.88	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	1.09	15.136	69.04	158.489	99.91	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	1.18	17.378	76.63	181.970	99.94	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	1.24	19.953	82.91	208.930	99.97	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	1.29	22.909	87.84	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	1.34	26.303	91.49	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	1.57	30.200	94.07	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	2.10	34.674	95.82	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	3.15	39.811	96.99	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.04	4.365	4.99	45.709	97.78	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.10	5.012	7.87	52.481	98.35	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.18	5.754	12.02	60.256	98.78	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.29	6.607	17.57	69.183	99.13	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.41	7.586	24.51	79.433	99.41	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.55	8.710	32.67	91.201	99.62	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.70	10.000	41.69	104.713	99.77	1096.478	100.00		

Operator notes: *Average of 2 measurements from PP1 Reactor final.mea*

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP1 - Cond 6 solids - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
Hugh

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Monday, May 03, 2010 9:25:03 AM

Analysed:
Monday, May 03, 2010 9:25:04 AM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
2.410 %

Sensitivity:
Enhanced

Obscuration:
5.84 %

Result Emulation:
Off

Concentration:
0.0061 %Vol

Span :
2.175

Uniformity:
0.712

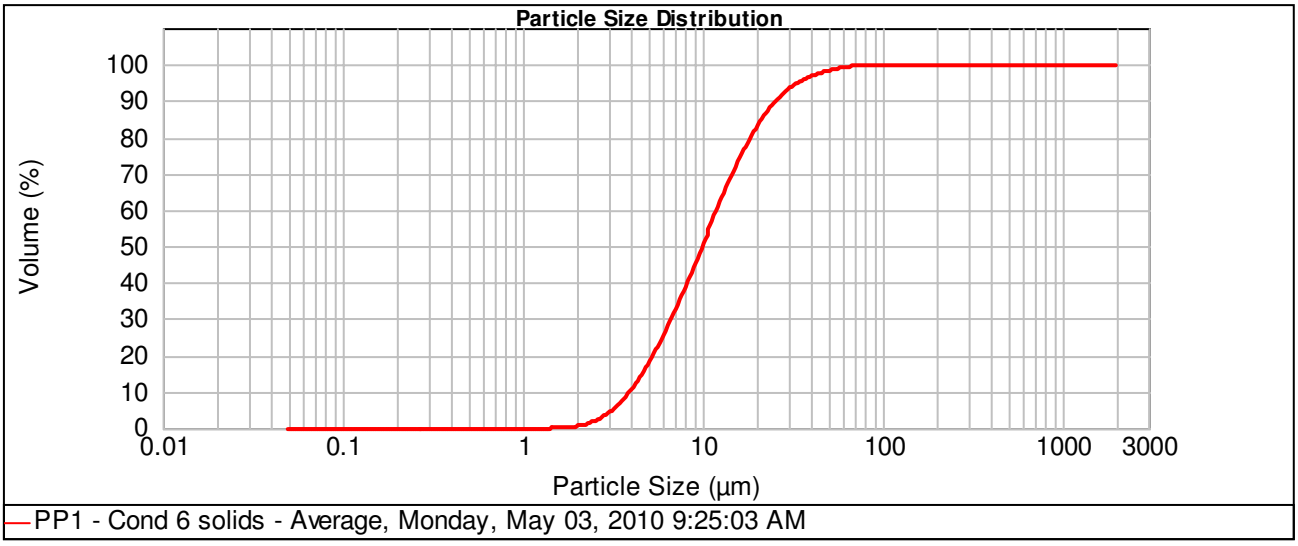
Result units:
Volume

Specific Surface Area:
0.769 m²/g

Surface Weighted Mean D[3,2]:
7.799 um

Vol. Weighted Mean D[4,3]:
13.000 um

d(0.1): 3.920 um **d(0.5):** 9.874 um **D(0.80) : 18.52 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	57.78	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	64.74	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.01	15.136	71.36	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.07	17.378	77.44	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.34	19.953	82.78	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.91	22.909	87.25	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	1.94	26.303	90.81	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	3.59	30.200	93.51	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	5.97	34.674	95.47	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	9.18	39.811	96.86	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	13.24	45.709	97.87	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	18.12	52.481	98.63	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	23.72	60.256	99.23	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	29.92	69.183	99.68	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	36.58	79.433	99.97	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	43.54	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	50.66	104.713	100.00	1096.478	100.00		

Operator notes: *Average of 2 measurements from Arnie Jan 2010.mea*

MALVERN INSTRUMENTS

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 8 Final solids - Average

SOP Name:
Hematite (red light only)

Measured:
Thursday, May 06, 2010 4:08:50 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Thursday, May 06, 2010 4:08:52 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
7.24 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
1.493 %

Result Emulation:
Off

Concentration:
0.0110 %Vol

Span :
2.323

Uniformity:
1.05

Result units:
Volume

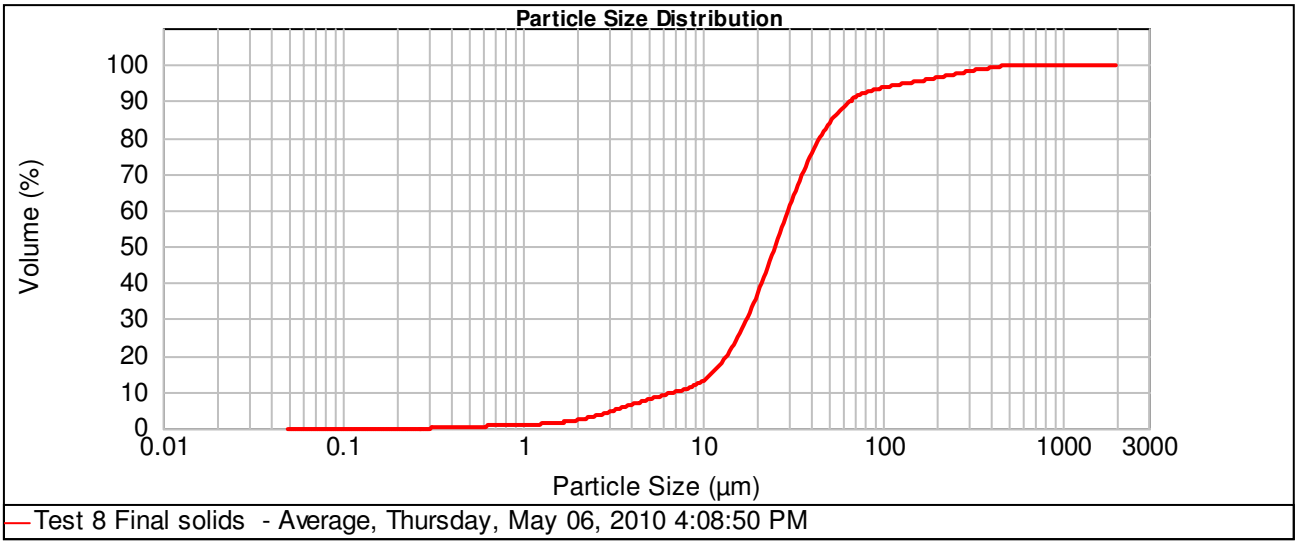
Specific Surface Area:
0.505 m²/g

Surface Weighted Mean D[3,2]:
11.883 um

Vol. Weighted Mean D[4,3]:
41.173 um

---

d(0.1): 7.075 um **d(0.5):** 25.248 um **D(0.80) : 44.58 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.96	11.482	15.42	120.226	94.37	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	1.11	13.183	18.77	138.038	94.89	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	1.31	15.136	23.35	158.489	95.44	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	1.61	17.378	29.21	181.970	96.01	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	2.02	19.953	36.24	208.930	96.62	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	2.57	22.909	44.14	239.883	97.22	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	3.26	26.303	52.49	275.423	97.80	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	4.09	30.200	60.77	316.228	98.35	3311.311	100.00
0.030	0.00	0.316	0.01	3.311	5.02	34.674	68.51	363.078	98.85	3801.894	100.00
0.035	0.00	0.363	0.10	3.802	6.00	39.811	75.31	416.869	99.30	4365.158	100.00
0.040	0.00	0.417	0.21	4.365	6.99	45.709	80.93	478.630	99.68	5011.872	100.00
0.046	0.00	0.479	0.32	5.012	7.92	52.481	85.30	549.541	99.91	5754.399	100.00
0.052	0.00	0.550	0.44	5.754	8.78	60.256	88.51	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.55	6.607	9.59	69.183	90.72	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.66	7.586	10.45	79.433	92.19	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.76	8.710	11.54	91.201	93.15	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.85	10.000	13.09	104.713	93.83	1096.478	100.00		

Operator notes:

MASTERSIZER 2000 200

Result Analysis Report

Sample Name:
Test 9 - hydrolysis - reactor solids -

SOP Name:
Hematite (red light only)

Measured:
Thursday, May 13, 2010 3:36:52 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Thursday, May 13, 2010 3:36:53 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
5.37 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
3.913 %

Result Emulation:
Off

Concentration:
0.0072 %Vol

Span :
1.593

Uniformity:
0.467

Result units:
Volume

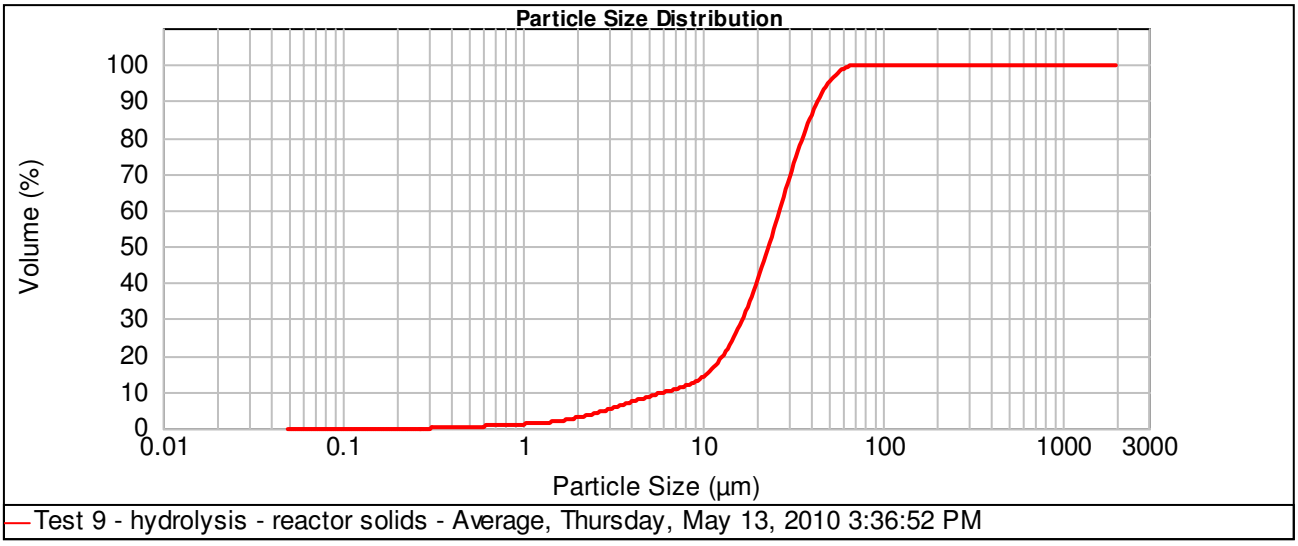
Specific Surface Area:
0.563 m²/g

Surface Weighted Mean D[3,2]:
10.666 um

Vol. Weighted Mean D[4,3]:
24.548 um

---

d(0.1): 6.213 um **d(0.5):** 23.251 um **D(0.80) : 35.79 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	1.19	11.482	16.63	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	1.41	13.183	20.24	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	1.71	15.136	25.26	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	2.09	17.378	31.78	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	2.60	19.953	39.78	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	3.22	22.909	48.96	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	3.98	26.303	58.84	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	4.84	30.200	68.77	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.01	3.311	5.79	34.674	78.04	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.09	3.802	6.77	39.811	86.03	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.20	4.365	7.74	45.709	92.27	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.32	5.012	8.67	52.481	96.59	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.45	5.754	9.53	60.256	99.05	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.58	6.607	10.38	69.183	99.96	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.72	7.586	11.29	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.86	8.710	12.46	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	1.01	10.000	14.13	104.713	100.00	1096.478	100.00		

Operator notes:

MASTERSIZER 2000 201

Result Analysis Report

Sample Name:
Test 10 Final solids - Average

SOP Name:
Hematite (red light only)

Measured:
Friday, May 21, 2010 2:18:43 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Friday, May 21, 2010 2:18:44 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
3.74 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
1.584 %

Result Emulation:
Off

Concentration:
0.0081 %Vol

Span :
1.542

Uniformity:
0.518

Result units:
Volume

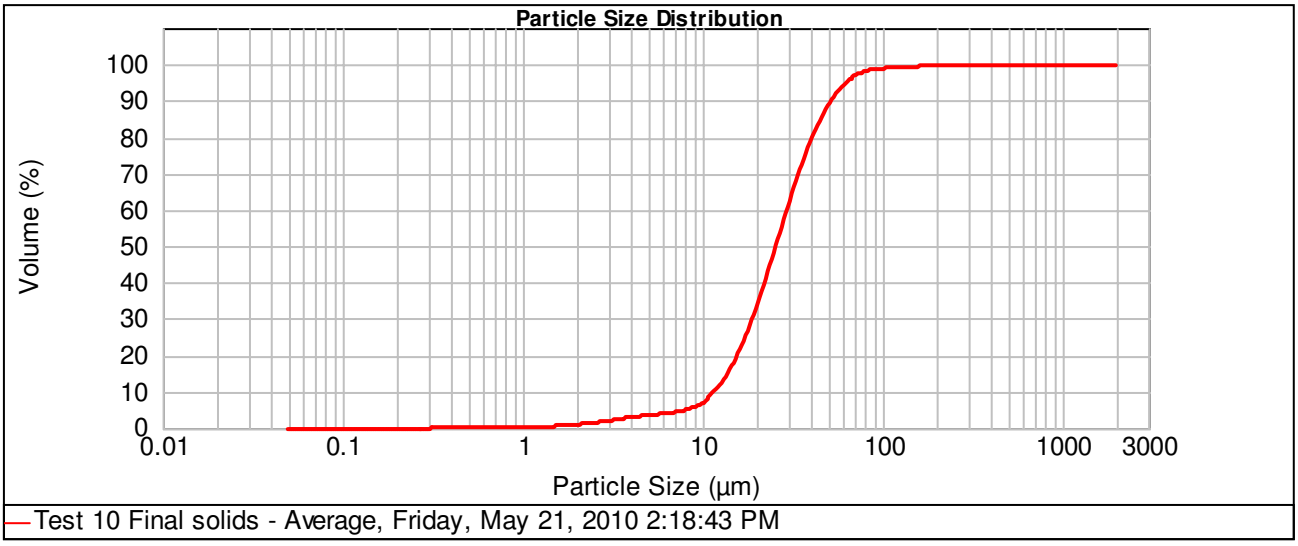
Specific Surface Area:
0.375 m²/g

Surface Weighted Mean D[3,2]:
16.017 um

Vol. Weighted Mean D[4,3]:
29.536 um

---

d(0.1): 11.668 um **d(0.5):** 25.391 um **D(0.80) : 40.23 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.40	11.482	9.64	120.226	99.17	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.44	13.183	13.38	138.038	99.34	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.53	15.136	18.61	158.489	99.56	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.67	17.378	25.36	181.970	99.80	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.87	19.953	33.51	208.930	99.97	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	1.14	22.909	42.73	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	1.50	26.303	52.52	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	1.91	30.200	62.29	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.01	3.311	2.36	34.674	71.43	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.05	3.802	2.81	39.811	79.45	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.10	4.365	3.23	45.709	86.02	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.16	5.012	3.59	52.481	91.03	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.21	5.754	3.91	60.256	94.54	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.26	6.607	4.25	69.183	96.78	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.30	7.586	4.75	79.433	98.06	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.34	8.710	5.62	91.201	98.70	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.37	10.000	7.14	104.713	99.00	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 11 Final solids - Average

SOP Name:
Hematite (red light only)

Measured:
Wednesday, June 02, 2010 10:56:49 AM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Wednesday, June 02, 2010 10:56:51 AM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
8.60 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.332 %

Result Emulation:
Off

Concentration:
0.0088 %Vol

Span :
1.949

Uniformity:
2.16

Result units:
Volume

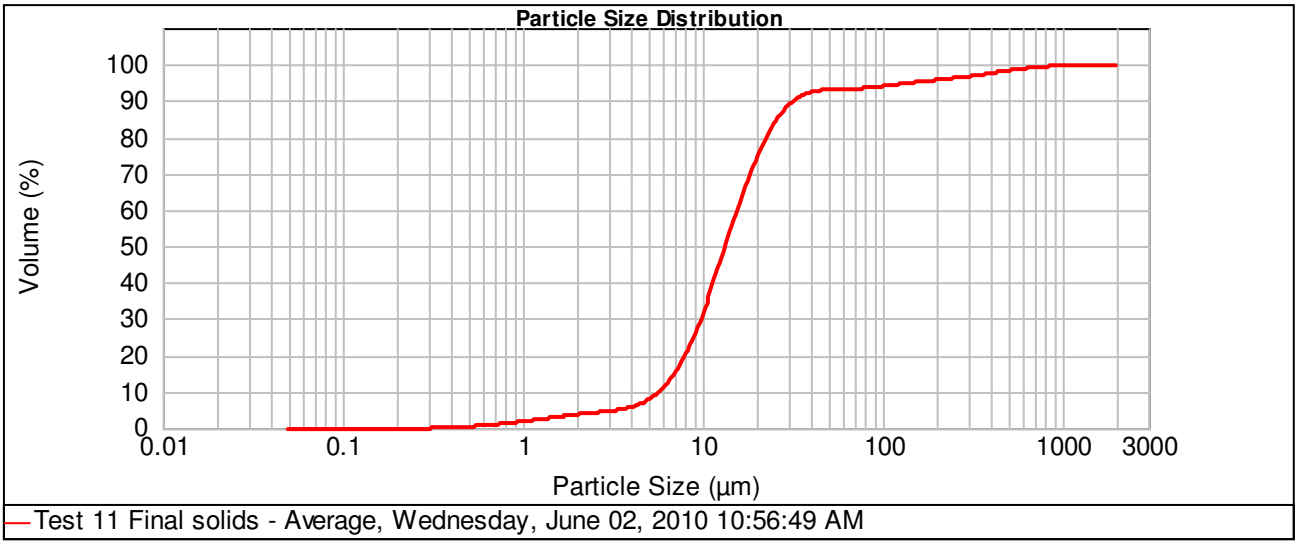
Specific Surface Area:
0.755 m²/g

Surface Weighted Mean D[3,2]:
7.942 um

Vol. Weighted Mean D[4,3]:
37.345 um

---

d(0.1): 5.676 um **d(0.5):** 13.412 um **D(0.80) : 22.59 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	2.11	11.482	39.95	120.226	94.54	1258.925	99.95
0.011	0.00	0.120	0.00	1.259	2.49	13.183	48.87	138.038	94.92	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	2.89	15.136	57.90	158.489	95.27	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	3.27	17.378	66.50	181.970	95.58	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	3.63	19.953	74.18	208.930	95.88	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	3.97	22.909	80.59	239.883	96.19	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	4.27	26.303	85.53	275.423	96.53	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	4.57	30.200	89.02	316.228	96.90	3311.311	100.00
0.030	0.00	0.316	0.01	3.311	4.94	34.674	91.25	363.078	97.32	3801.894	100.00
0.035	0.00	0.363	0.08	3.802	5.50	39.811	92.47	416.869	97.76	4365.158	100.00
0.040	0.00	0.417	0.19	4.365	6.43	45.709	93.03	478.630	98.20	5011.872	100.00
0.046	0.00	0.479	0.35	5.012	7.93	52.481	93.21	549.541	98.62	5754.399	100.00
0.052	0.00	0.550	0.55	5.754	10.27	60.256	93.27	630.957	98.99	6606.934	100.00
0.060	0.00	0.631	0.80	6.607	13.69	69.183	93.35	724.436	99.30	7585.776	100.00
0.069	0.00	0.724	1.08	7.586	18.37	79.433	93.52	831.764	99.54	8709.636	100.00
0.079	0.00	0.832	1.39	8.710	24.38	91.201	93.80	954.993	99.73	10000.000	100.00
0.091	0.00	0.955	1.74	10.000	31.65	104.713	94.16	1096.478	99.86		

Operator notes:

MASTERSIZER 2000 203

Result Analysis Report

Sample Name:
Test 12 Final solids - Average

SOP Name:
Hematite (red light only)

Measured:
Wednesday, June 09, 2010 9:32:42 AM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Wednesday, June 09, 2010 9:32:43 AM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
3.19 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
1.043 %

Result Emulation:
Off

Concentration:
0.0078 %Vol

Span :
1.677

Uniformity:
0.534

Result units:
Volume

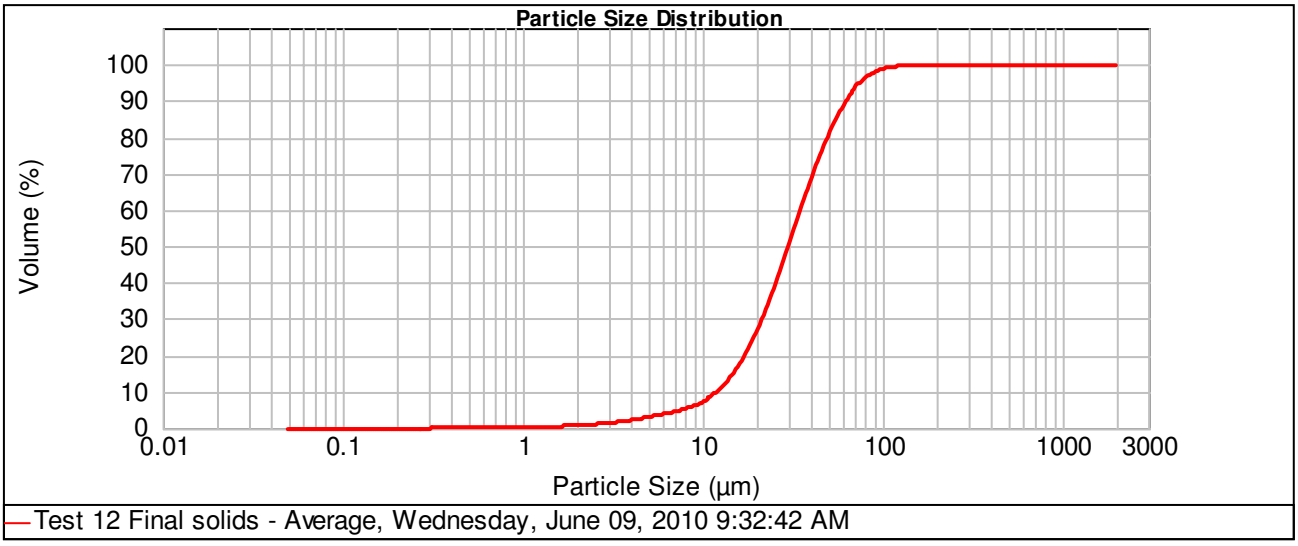
Specific Surface Area:
0.334 m²/g

Surface Weighted Mean D[3,2]:
17.941 um

Vol. Weighted Mean D[4,3]:
34.304 um

---

d(0.1): 12.001 um **d(0.5):** 29.787 um **D(0.80) : 48.74 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.41	11.482	9.24	120.226	99.55	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.44	13.183	11.91	138.038	99.75	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.49	15.136	15.57	158.489	99.81	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.56	17.378	20.41	181.970	99.85	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.67	19.953	26.49	208.930	99.89	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.82	22.909	33.77	239.883	99.94	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	1.03	26.303	42.02	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	1.31	30.200	50.90	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	1.66	34.674	59.96	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.04	3.802	2.08	39.811	68.69	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.08	4.365	2.55	45.709	76.66	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.13	5.012	3.08	52.481	83.52	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.18	5.754	3.66	60.256	89.05	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.24	6.607	4.30	69.183	93.23	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.29	7.586	5.06	79.433	96.15	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.34	8.710	6.03	91.201	98.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.38	10.000	7.35	104.713	99.05	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 13 Final solids - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
Ernesto

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Monday, June 21, 2010 9:33:10 AM

Analysed:
Monday, June 21, 2010 9:33:11 AM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
0.997 %

Sensitivity:
Enhanced

Obscuration:
3.74 %

Result Emulation:
Off

Concentration:
0.0041 %Vol

Span :
1.692

Uniformity:
0.514

Result units:
Volume

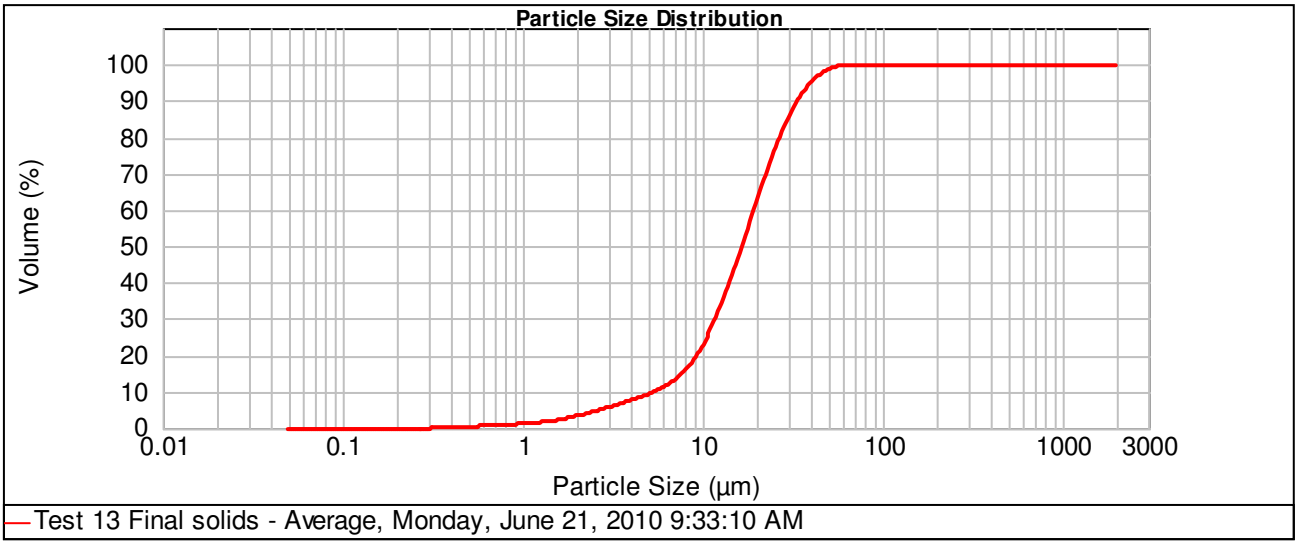
Specific Surface Area:
0.687 m²/g

Surface Weighted Mean D[3,2]:
8.729 um

Vol. Weighted Mean D[4,3]:
18.323 um

---

d(0.1): 5.304 um **d(0.5):** 16.572 um **D(0.80) : 26.69 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	1.38	11.482	28.85	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	1.66	13.183	35.96	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	2.04	15.136	44.18	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	2.51	17.378	53.13	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	3.11	19.953	62.32	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	3.81	22.909	71.18	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	4.61	26.303	79.21	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	5.49	30.200	86.02	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.01	3.311	6.41	34.674	91.37	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.11	3.802	7.36	39.811	95.25	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.23	4.365	8.36	45.709	97.78	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.37	5.012	9.48	52.481	99.22	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.51	5.754	10.84	60.256	99.86	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.66	6.607	12.63	69.183	99.99	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.81	7.586	15.08	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.97	8.710	18.47	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	1.15	10.000	23.00	104.713	100.00	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 14 Final solids - Average

SOP Name:
Hematite (red light only)

Measured:
Monday, June 21, 2010 9:44:45 AM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Monday, June 21, 2010 9:44:47 AM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
3.03 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.752 %

Result Emulation:
Off

Concentration:
0.0037 %Vol

Span :
1.506

Uniformity:
0.544

Result units:
Volume

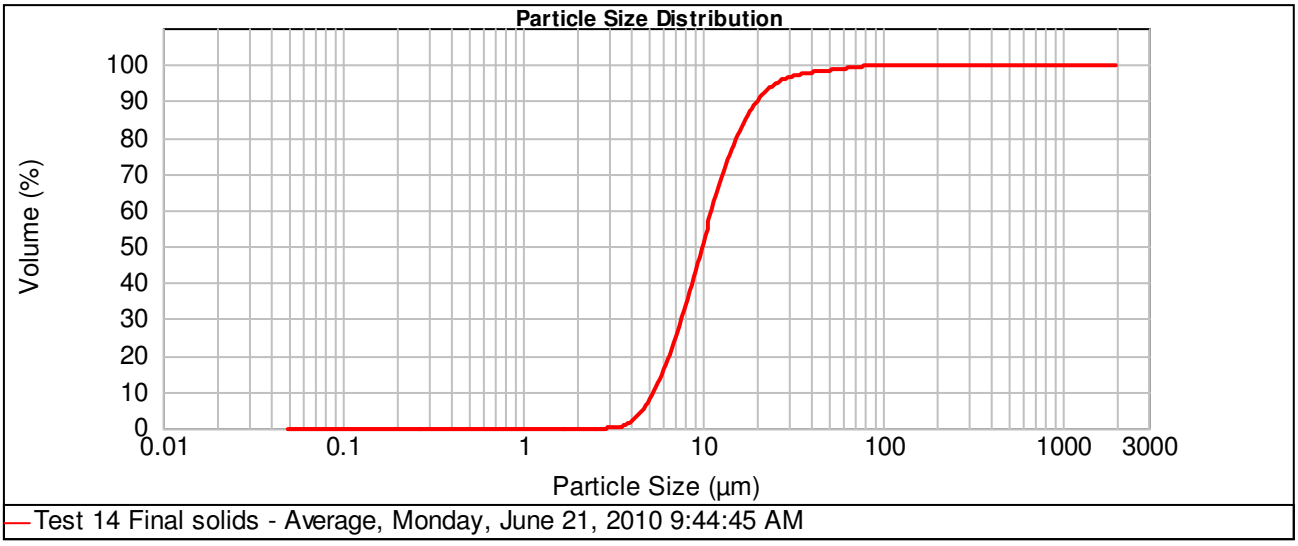
Specific Surface Area:
0.666 m²/g

Surface Weighted Mean D[3,2]:
9.009 um

Vol. Weighted Mean D[4,3]:
12.308 um

---

d(0.1): 5.327 um **d(0.5):** 9.864 um **D(0.80) : 15.59 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	61.26	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	70.56	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	78.50	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	84.89	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	89.68	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	93.04	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	95.23	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	96.56	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.17	34.674	97.35	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	1.21	39.811	97.84	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	3.60	45.709	98.21	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	7.64	52.481	98.55	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	13.51	60.256	98.90	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	21.17	69.183	99.26	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	30.34	79.433	99.58	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	40.51	91.201	99.83	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	51.04	104.713	99.96	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 15 Final solids - Average

SOP Name:
Hematite (red light only)

Measured:
Thursday, June 24, 2010 2:08:53 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Thursday, June 24, 2010 2:08:54 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
3.12 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
1.284 %

Result Emulation:
Off

Concentration:
0.0051 %Vol

Span :
0.978

Uniformity:
0.304

Result units:
Volume

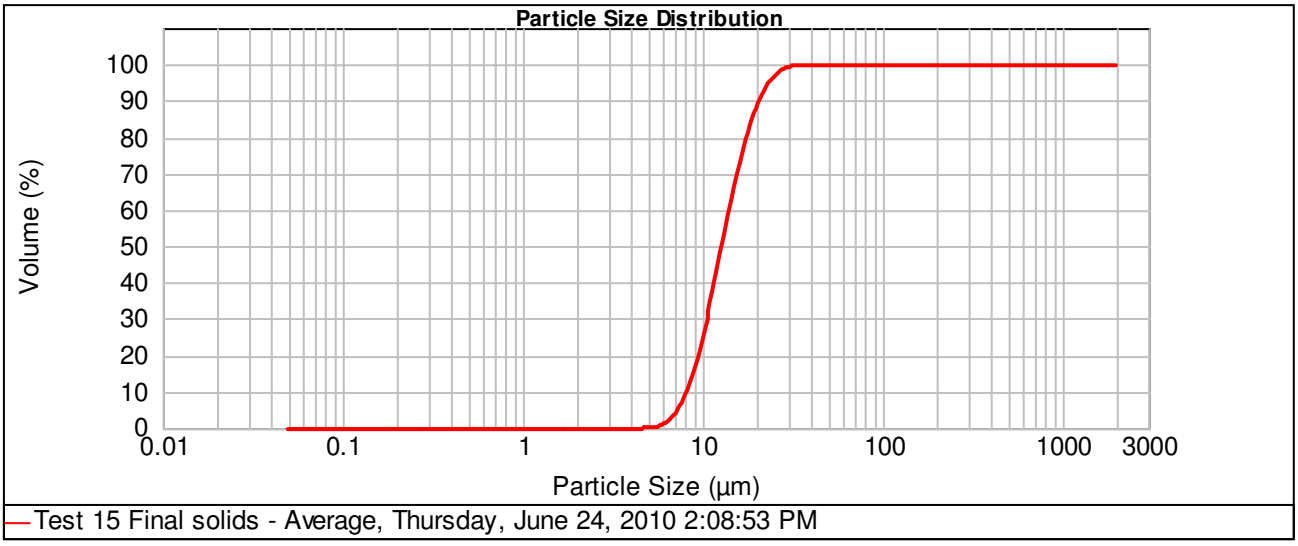
Specific Surface Area:
0.498 m²/g

Surface Weighted Mean D[3,2]:
12.051 um

Vol. Weighted Mean D[4,3]:
13.693 um

---

d(0.1): 8.043 um **d(0.5):** 12.807 um **D(0.80) : 17.57 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	38.60	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	53.04	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	67.06	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	79.17	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	88.39	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	94.53	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	97.96	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	99.51	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.00	34.674	99.94	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	0.00	39.811	100.00	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	0.00	45.709	100.00	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	0.04	52.481	100.00	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	0.65	60.256	100.00	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	2.75	69.183	100.00	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	7.20	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	14.72	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	25.38	104.713	100.00	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 16 Final solids - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
Ernesto

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Friday, July 09, 2010 2:49:08 PM

Analysed:
Friday, July 09, 2010 2:49:09 PM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
1.669 %

Sensitivity:
Enhanced

Obscuration:
3.14 %

Result Emulation:
Off

Concentration:
0.0070 %Vol

Specific Surface Area:
0.37 m²/g

Span :
4.109

Surface Weighted Mean D[3,2]:
16.229 um

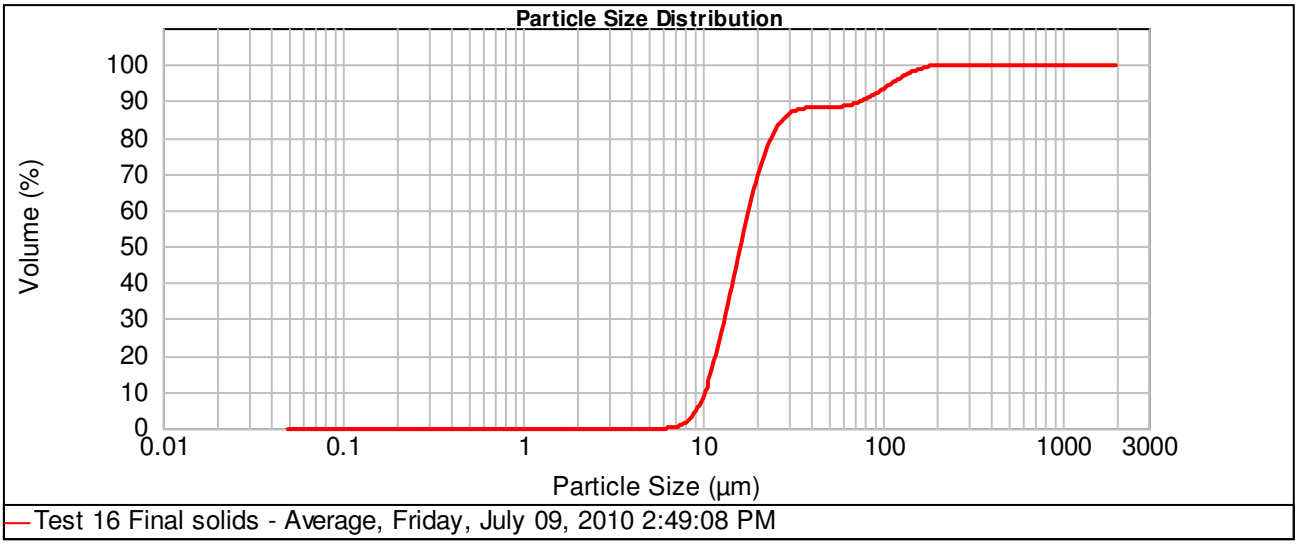
Uniformity:
0.928

Vol. Weighted Mean D[4,3]:
27.545 um

Result units:
Volume

---

d(0.1): 10.236 um **d(0.5):** 16.299 um **D(0.80) : 24.09 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	17.34	120.226	95.72	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	28.95	138.038	97.41	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	42.53	158.489	98.72	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	56.27	181.970	99.58	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	68.33	208.930	99.96	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	77.49	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	83.39	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	86.53	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.00	34.674	87.82	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	0.00	39.811	88.18	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	0.00	45.709	88.26	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	0.00	52.481	88.35	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	0.00	60.256	88.62	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	0.05	69.183	89.17	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	0.95	79.433	90.28	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	3.54	91.201	91.92	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	8.79	104.713	93.82	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 17 Final solids - Average

SOP Name:
Hematite (red light only)

Measured:
Friday, July 09, 2010 2:59:49 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Friday, July 09, 2010 2:59:50 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
3.39 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
2.753 %

Result Emulation:
Off

Concentration:
0.0056 %Vol

Span :
1.074

Uniformity:
0.615

Result units:
Volume

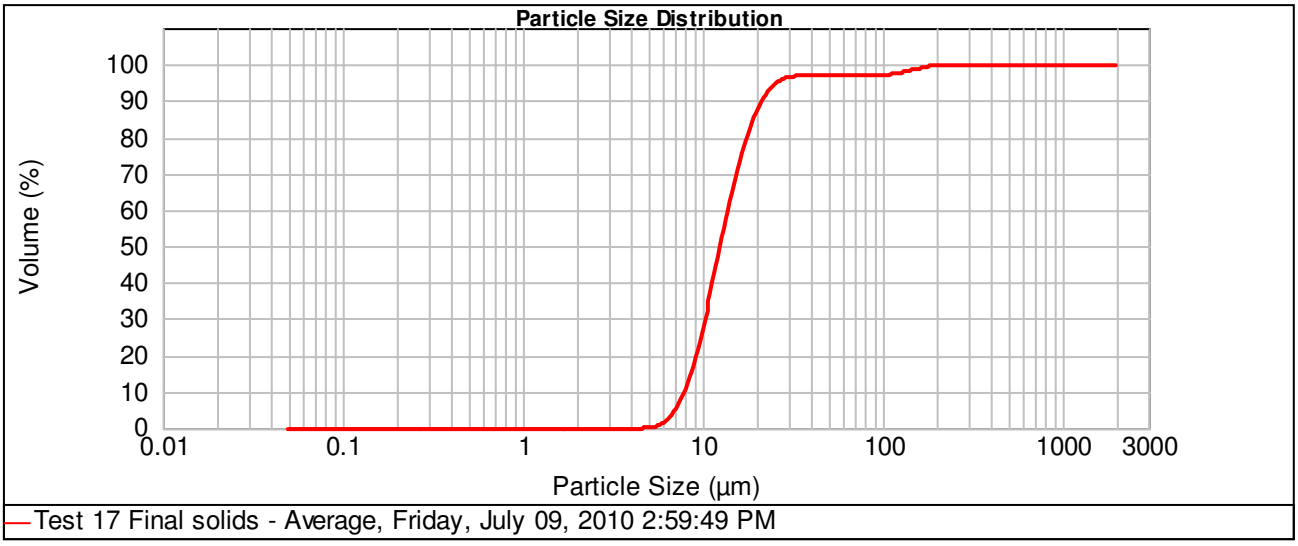
Specific Surface Area:
0.5 m²/g

Surface Weighted Mean D[3,2]:
12.005 um

Vol. Weighted Mean D[4,3]:
17.337 um

---

d(0.1): 7.825 um **d(0.5):** 12.571 um **D(0.80) : 17.65 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	40.83	120.226	97.63	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	54.79	138.038	98.23	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	67.93	158.489	98.93	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	78.95	181.970	99.62	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	87.11	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	92.40	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	95.30	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	96.56	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.00	34.674	96.91	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	0.00	39.811	96.91	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	0.00	45.709	96.91	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	0.06	52.481	96.91	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	0.87	60.256	96.91	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	3.40	69.183	96.91	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	8.44	79.433	96.91	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	16.56	91.201	96.98	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	27.61	104.713	97.21	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
Test 18 - Average

SOP Name:
Hematite (red light only)

Measured:
Monday, September 13, 2010 12:59:50 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Monday, September 13, 2010 12:59:51 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
5.96 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.489 %

Result Emulation:
Off

Concentration:
0.0103 %Vol

Span :
1.000

Uniformity:
0.403

Result units:
Volume

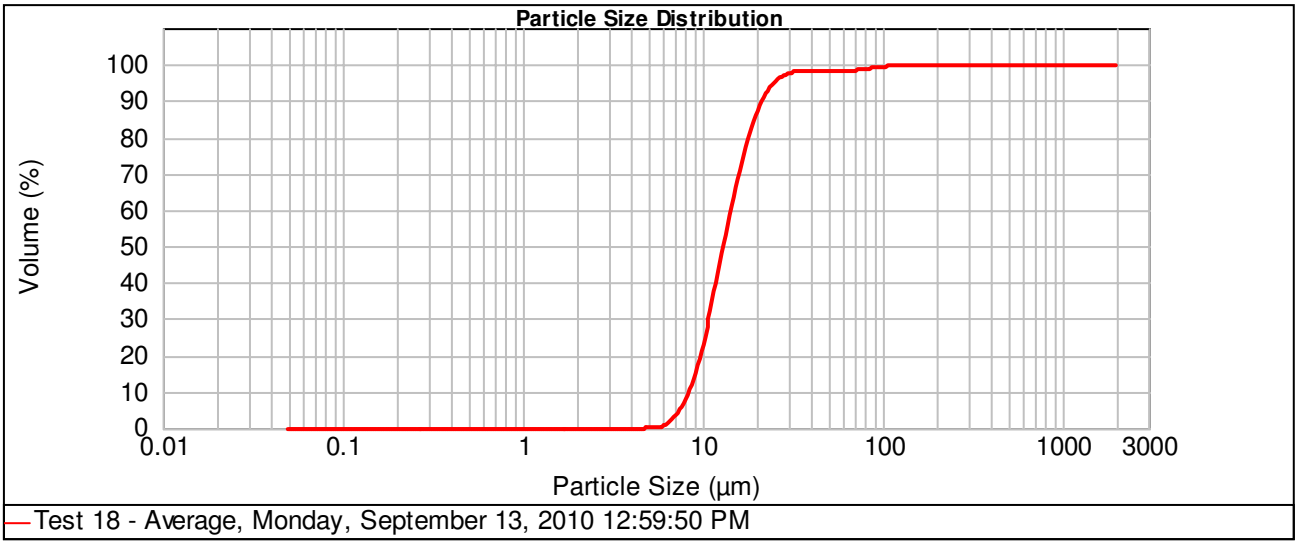
Specific Surface Area:
0.481 m²/g

Surface Weighted Mean D[3,2]:
12.472 um

Vol. Weighted Mean D[4,3]:
15.320 um

---

d(0.1): 8.289 um **d(0.5):** 13.115 um **D(0.80) : 18.06 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	36.08	120.226	99.81	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	50.55	138.038	99.95	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	64.75	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	77.07	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	86.45	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	92.66	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	96.11	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	97.64	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.00	34.674	98.10	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	0.00	39.811	98.15	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	0.00	45.709	98.15	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	0.03	52.481	98.15	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	0.34	60.256	98.15	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	2.05	69.183	98.35	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	5.92	79.433	98.74	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	12.87	91.201	99.16	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	23.08	104.713	99.54	1096.478	100.00		

Operator notes:

MASTERSIZER 2000 210

Result Analysis Report

Sample Name:
Test 19 - Average

SOP Name:
Hematite (red light only)

Measured:
Monday, September 13, 2010 1:10:30 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Monday, September 13, 2010 1:10:31 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
5.89 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.660 %

Result Emulation:
Off

Concentration:
0.0069 %Vol

Span :
1.816

Uniformity:
0.618

Result units:
Volume

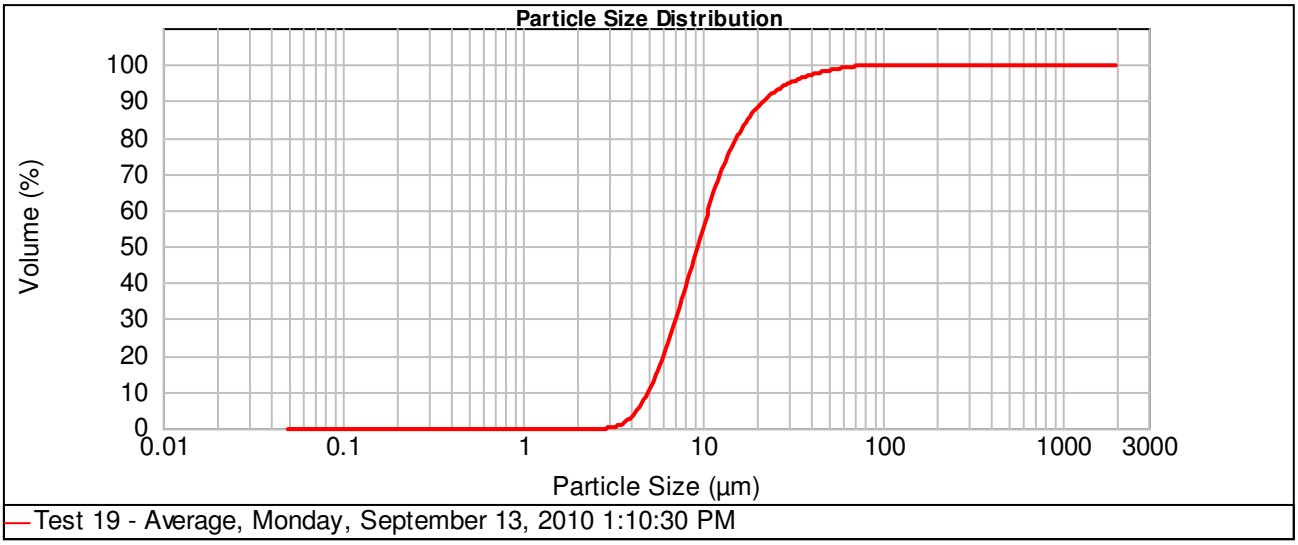
Specific Surface Area:
0.701 m²/g

Surface Weighted Mean D[3,2]:
8.558 um

Vol. Weighted Mean D[4,3]:
12.258 um

---

d(0.1): 4.967 um **d(0.5):** 9.282 um **D(0.80) : 15.59 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	64.32	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	72.23	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	78.76	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	83.96	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	87.95	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	90.95	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	93.19	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	94.88	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.40	34.674	96.16	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	2.08	39.811	97.17	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	5.32	45.709	97.97	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	10.39	52.481	98.61	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	17.28	60.256	99.11	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	25.76	69.183	99.49	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	35.35	79.433	99.76	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	45.40	91.201	99.92	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	55.25	104.713	99.99	1096.478	100.00		

Operator notes:

MASTERSIZER 2000 211

Result Analysis Report

Sample Name:
PP2 - Cond 1 - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
Ernesto

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Thursday, August 05, 2010 9:04:44 AM

Analysed:
Thursday, August 05, 2010 9:04:45 AM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
1.300 %

Sensitivity:
Enhanced

Obscuration:
2.41 %

Result Emulation:
Off

Concentration:
0.0049 %Vol

Specific Surface Area:
0.407 m²/g

Span :
0.833

Surface Weighted Mean D[3,2]:
14.733 um

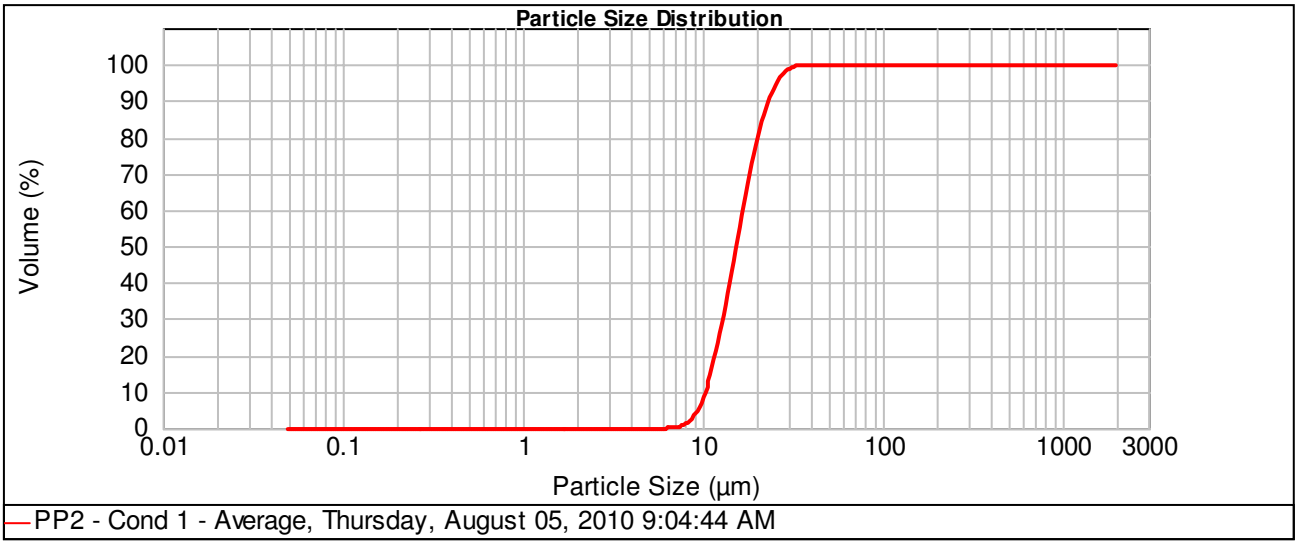
Uniformity:
0.262

Vol. Weighted Mean D[4,3]:
16.225 um

Result units:
Volume

---

d(0.1): 10.306 um **d(0.5):** 15.463 um **D(0.80) : 20.27 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	17.69	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	31.13	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	47.38	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	64.05	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	78.57	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	89.27	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	95.75	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	98.85	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.00	34.674	99.86	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	0.00	39.811	100.00	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	0.00	45.709	100.00	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	0.00	52.481	100.00	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	0.00	60.256	100.00	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	0.03	69.183	100.00	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	0.68	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	3.00	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	8.31	104.713	100.00	1096.478	100.00		

Operator notes:

MALVERN INSTRUMENTS

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP2 - Cond2 - Average

SOP Name:
Hematite (red light only)

Measured:
Thursday, August 05, 2010 9:16:50 AM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Thursday, August 05, 2010 9:16:51 AM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
3.06 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
2.408 %

Result Emulation:
Off

Concentration:
0.0084 %Vol

Span :
0.875

Uniformity:
0.274

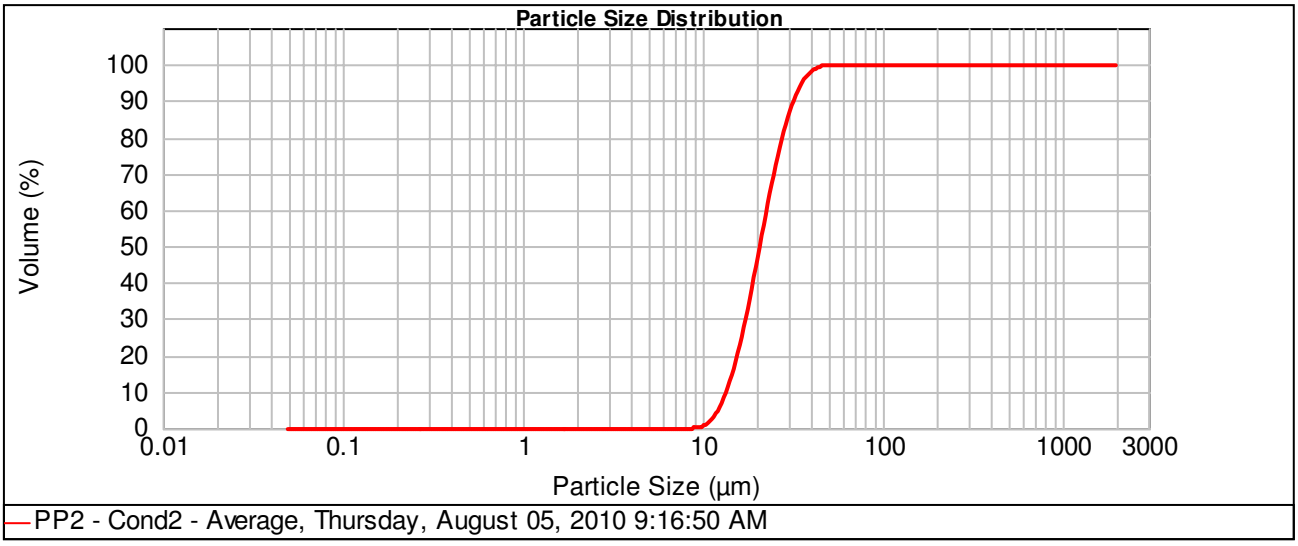
Result units:
Volume

Specific Surface Area:
0.304 m²/g

Surface Weighted Mean D[3,2]:
19.750 um

Vol. Weighted Mean D[4,3]:
21.921 um

d(0.1): 13.622 um **d(0.5):** 20.784 um **D(0.80) : 27.64 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	2.95	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	8.21	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	17.23	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	29.94	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	45.26	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	61.19	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.00	26.303	75.57	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.00	30.200	86.77	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	0.00	34.674	94.15	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	0.00	39.811	98.12	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	0.00	45.709	99.69	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	0.00	52.481	100.00	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	0.00	60.256	100.00	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	0.00	69.183	100.00	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	0.00	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	0.01	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	0.60	104.713	100.00	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP2 Condition 3 - Average

SOP Name:
Hematite (red light only)

Measured:
Monday, September 13, 2010 12:22:03 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Monday, September 13, 2010 12:22:04 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
6.49 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
1.078 %

Result Emulation:
Off

Concentration:
0.0100 %Vol

Span :
1.584

Uniformity:
0.454

Result units:
Volume

Specific Surface Area:
0.526 m²/g

Surface Weighted Mean D[3,2]:
11.403 um

Vol. Weighted Mean D[4,3]:
21.762 um

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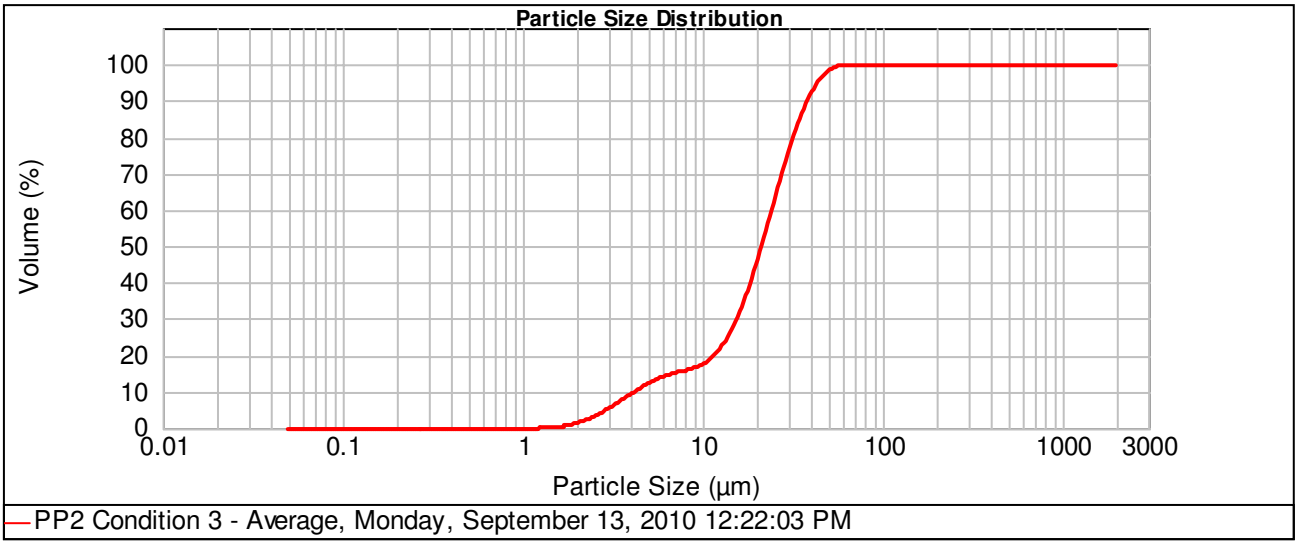
d(0.1): 4.167 um **d(0.5):** 21.224 um **D(0.80) : 31.64 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	19.74	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.02	13.183	23.31	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.15	15.136	28.75	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.49	17.378	36.19	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	1.09	19.953	45.43	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	2.01	22.909	55.88	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	3.28	26.303	66.68	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	4.86	30.200	76.86	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	6.71	34.674	85.58	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	8.69	39.811	92.26	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	10.64	45.709	96.72	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	12.41	52.481	99.15	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	13.84	60.256	99.98	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	14.90	69.183	100.00	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	15.69	79.433	100.00	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	16.46	91.201	100.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	17.63	104.713	100.00	1096.478	100.00		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP2 Condition 4 - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
Ernesto

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Monday, September 13, 2010 12:38:53 PM

Analysed:
Monday, September 13, 2010 12:38:54 PM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
0.432 %

Sensitivity:
Enhanced

Obscuration:
5.80 %

Result Emulation:
Off

Concentration:
0.0057 %Vol

Span :
3.497

Uniformity:
3.37

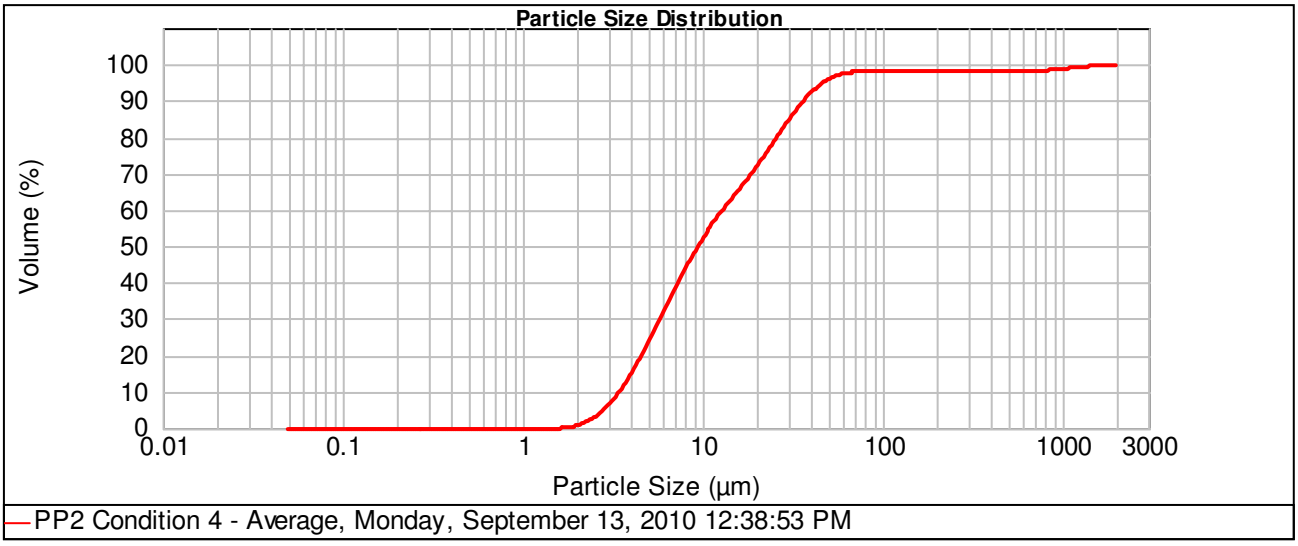
Result units:
Volume

Specific Surface Area:
0.813 m²/g

Surface Weighted Mean D[3,2]:
7.376 um

Vol. Weighted Mean D[4,3]:
36.730 um

d(0.1): 3.441 um **d(0.5):** 9.318 um **D(0.80) : 25.75 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	56.60	120.226	97.97	1258.925	99.37
0.011	0.00	0.120	0.00	1.259	0.00	13.183	60.45	138.038	97.97	1445.440	99.64
0.013	0.00	0.138	0.00	1.445	0.00	15.136	64.15	158.489	97.97	1659.587	99.85
0.015	0.00	0.158	0.00	1.660	0.01	17.378	67.93	181.970	97.97	1905.461	99.97
0.017	0.00	0.182	0.00	1.905	0.43	19.953	71.95	208.930	97.97	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	1.36	22.909	76.25	239.883	97.97	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	3.00	26.303	80.69	275.423	97.97	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	5.49	30.200	85.04	316.228	97.97	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	8.89	34.674	89.01	363.078	97.97	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	13.21	39.811	92.34	416.869	97.97	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	18.35	45.709	94.87	478.630	97.97	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	24.12	52.481	96.58	549.541	97.99	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	30.22	60.256	97.55	630.957	98.06	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	36.35	69.183	97.94	724.436	98.21	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	42.22	79.433	97.97	831.764	98.44	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	47.59	91.201	97.97	954.993	98.73	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	52.37	104.713	97.97	1096.478	99.05		

Operator notes:

MASTERSIZER 2000

Result Analysis Report

Sample Name:
PP2 Condition 5 - Average

Sample Source & type:
NV = 12323-001

Sample bulk lot ref:
Ernesto

SOP Name:
Hematite (red light only)

Measured by:
lr_hydro1

Result Source:
Averaged

Measured:
Monday, September 13, 2010 12:49:12 PM

Analysed:
Monday, September 13, 2010 12:49:14 PM

Particle Name:
Haematite

Particle RI:
2.940

Dispersant Name:
Water

Accessory Name:
Hydro 2000G (A)

Absorption:
0.1

Dispersant RI:
1.330

Analysis model:
General purpose

Size range:
0.020 to 2000.000 um

Weighted Residual:
0.503 %

Sensitivity:
Enhanced

Obscuration:
6.63 %

Result Emulation:
Off

Concentration:
0.0074 %Vol

Specific Surface Area:
0.728 m²/g

Span :
3.465

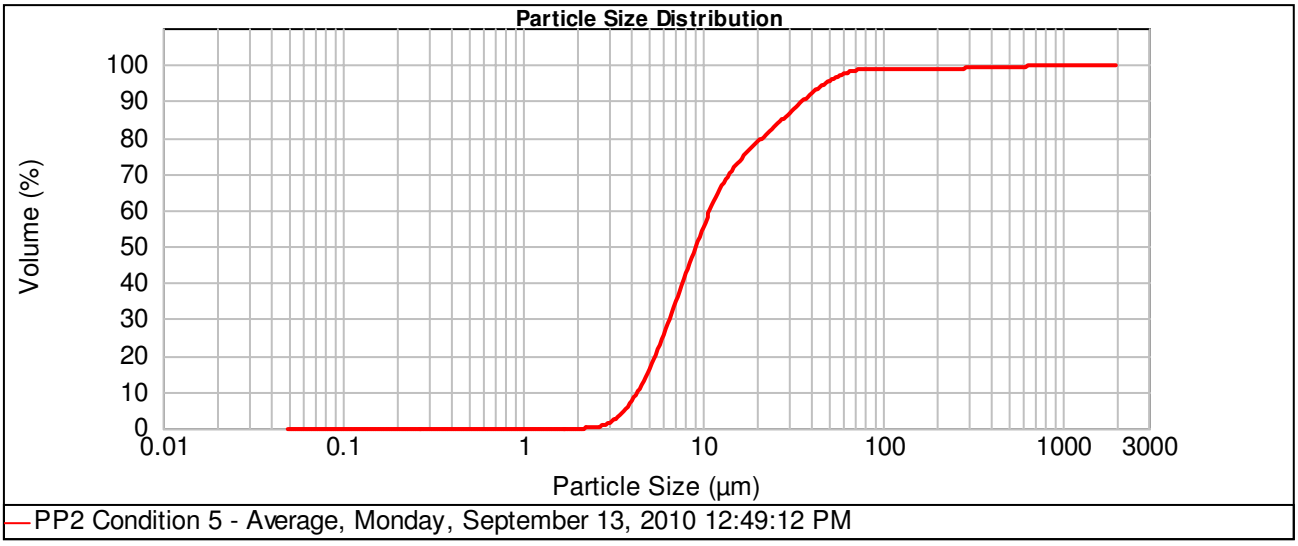
Surface Weighted Mean D[3,2]:
8.238 um

Uniformity:
1.6

Vol. Weighted Mean D[4,3]:
20.484 um

Result units:
Volume

d(0.1): 4.344 um **d(0.5):** 9.073 um **D(0.80) : 21.61 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	0.00	11.482	61.92	120.226	99.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	0.00	13.183	67.43	138.038	99.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	0.00	15.136	71.83	158.489	99.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	0.00	17.378	75.39	181.970	99.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	0.00	19.953	78.39	208.930	99.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	0.00	22.909	81.15	239.883	99.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	0.15	26.303	83.88	275.423	99.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	0.97	30.200	86.64	316.228	99.03	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	2.76	34.674	89.39	363.078	99.08	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	5.78	39.811	91.99	416.869	99.16	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	10.18	45.709	94.26	478.630	99.26	5011.872	100.00
0.046	0.00	0.479	0.00	5.012	16.00	52.481	96.10	549.541	99.40	5754.399	100.00
0.052	0.00	0.550	0.00	5.754	23.06	60.256	97.44	630.957	99.55	6606.934	100.00
0.060	0.00	0.631	0.00	6.607	31.03	69.183	98.30	724.436	99.70	7585.776	100.00
0.069	0.00	0.724	0.00	7.586	39.41	79.433	98.77	831.764	99.84	8709.636	100.00
0.079	0.00	0.832	0.00	8.710	47.66	91.201	98.97	954.993	99.94	10000.000	100.00
0.091	0.00	0.955	0.00	10.000	55.29	104.713	99.00	1096.478	100.00		

Operator notes:

Result Analysis Report

Sample Name:
PP2 Condition 6 - Average

SOP Name:
Hematite (red light only)

Measured:
Tuesday, September 14, 2010 12:38:10 PM

Sample Source & type:
NV = 12323-001

Measured by:
lr_hydro1

Analysed:
Tuesday, September 14, 2010 12:38:11 PM

Sample bulk lot ref:
Ernesto

Result Source:
Averaged

Particle Name:
Haematite

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Enhanced

Particle RI:
2.940

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
6.13 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.756 %

Result Emulation:
Off

Concentration:
0.0044 %Vol

Span :
1.824

Uniformity:
0.731

Result units:
Volume

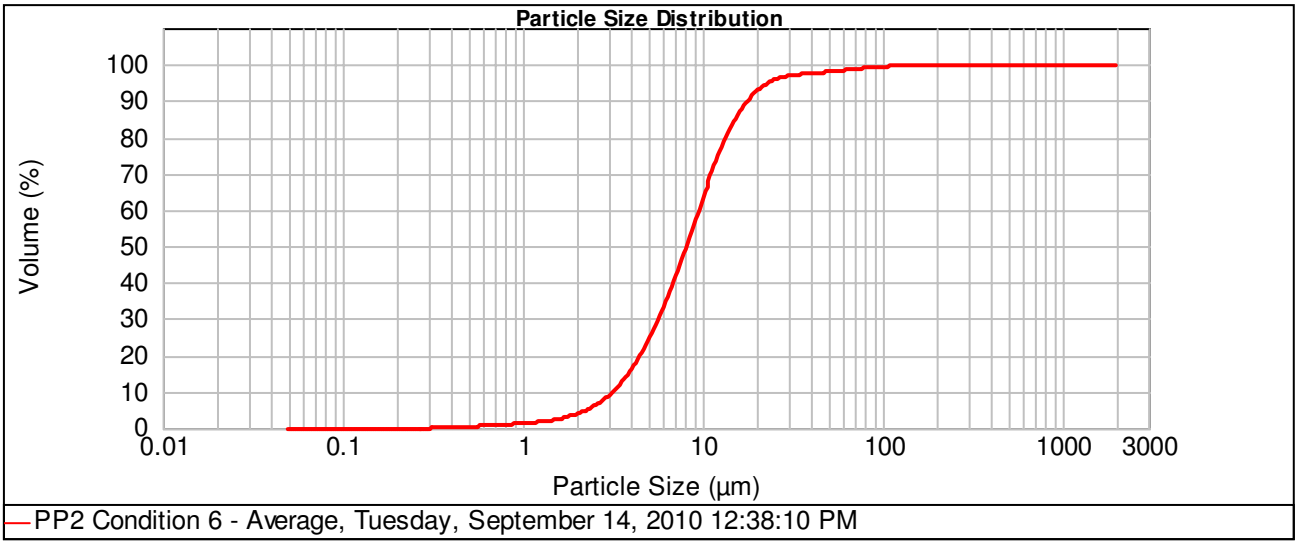
Specific Surface Area:
1.07 m²/g

Surface Weighted Mean D[3,2]:
5.593 um

Vol. Weighted Mean D[4,3]:
10.808 um

---

d(0.1): 3.166 um **d(0.5):** 8.034 um **D(0.80) : 13.57 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	1.46	11.482	71.48	120.226	99.67	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	1.77	13.183	78.64	138.038	99.82	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	2.17	15.136	84.63	158.489	99.92	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	2.74	17.378	89.29	181.970	99.97	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	3.52	19.953	92.65	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	4.62	22.909	94.87	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	6.15	26.303	96.20	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	8.22	30.200	96.94	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.01	3.311	10.98	34.674	97.35	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.10	3.802	14.54	39.811	97.61	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.21	4.365	19.04	45.709	97.85	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.35	5.012	24.53	52.481	98.11	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.50	5.754	31.01	60.256	98.41	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.66	6.607	38.40	69.183	98.71	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.83	7.586	46.50	79.433	99.01	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	1.02	8.710	54.98	91.201	99.27	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	1.22	10.000	63.46	104.713	99.49	1096.478	100.00		

Operator notes:

An Investigation into

THE FLOTATION PROPERTIES OF A COMPOSITE FROM THE FERGUSON LAKE DEPOSIT

prepared for

STARFIELD RESOUCRES INC

Project 12323-002 – Final Report
January 17, 2011

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Executive Summary

Starfield Resources Inc commissioned an extensive programme of experimental work on a novel metallurgical process for their Ferguson Lake project. A flotation test program was carried out on a composite sample from the Ferguson Lake Ni-Cu-PGM deposit to determine whether or not the gangue minerals can be rejected into a tailings product without significant losses of the valuable elements (base and precious metals).

A total of 11 rougher flotation tests and a single cleaner flotation test were carried out to establish suitable flotation conditions. Under optimized conditions 99.7-99.9% of the Cu units and 98-99% of the Ni units were recovered into the rougher flotation concentrate at 80% mass recovery. Further, more than 97% of Pt and Pd as well as approximately 97% of the Au were recovered into the rougher concentrate. At the same time, approximately 75-80% of the gangue minerals were rejected into the tailings stream.

A primary grind size target of K_{80} = 110-120 microns and a simple reagent suite consisting of PAX and MIBC combined with a natural pH of approximately 7.5 produced the best results. The use of the gangue depressant sodium silicate did not improve the flotation selectivity. While the Ni and S losses were originally slightly higher using sea water due to a lower mass recovery into the rougher concentrate, a more aggressive froth removal matched the original results using fresh water.

A single batch cleaner test rejected another 4% of the gangue minerals, albeit at increased Cu and Ni losses of 0.13% and 0.44%, respectively. Unless a maximum rejection of gangue minerals is critical, the rougher concentrate should be considered a final product for the downstream hydrometallurgical process.

A set of 10 kg bulk flotation tests was carried out using a total of 250 kg of the Ferguson Lake composite sample to produce sufficient concentrate for experimental work on the downstream hydrometallurgical process. The flotation concentrate and tailings from the individual tests were combined and sub-sampled for chemical analysis. The mass balance of this bulk flotation test is presented in Table 1 and reveals results similar to the ones that were achieved in the batch tests.

Table 1: Bulk Flotation Mass Balance

Product	Weight	Assay (%)			Distribution (%)		
	%	Cu	Ni	S	Cu	Ni	S
Rougher Concentrate	84	1.70	1.03	35.5	99.6	97.8	99.1
Rougher Tails	16	0.04	0.12	1.71	0.4	2.2	0.9
Feed (calculated)	100	1.43	0.86	29.2	100	100	100
Feed (calculated)		1.48	0.87	29.0			

High metal losses to the magnetic fraction of a magnetic separation test on the rougher concentrate rendered this process unattractive. However, magnetic separation on the rougher tailings yielded more encouraging results and possibly a magnetite by-product could be generated with further optimization tests.

It is recommended to evaluate the stability of the proposed flotation flowsheet on a number of variability composites. These composites should be selected from various areas and depths of the deposit and take into account any differences in lithology and visible grain size distribution differences.

A mineralogical characterization of the flotation tails would identify the nature of metal losses. These findings could then be used to carry out an optimization program to further decrease metal losses.

Introduction

Starfield Resources Inc (Starfield) is developing a novel hydrometallurgical flowsheet for their Ferguson Lake massive sulphide deposit in the Nunavut Territory to recover copper, nickel, cobalt, PGMs and hematite. A multi-phase program was proposed by the SGS hydrometallurgical group to develop this flowsheet with the aid of laboratory, mini pilot plant, and integrated pilot plant testing. One of the phases included laboratory scale flotation tests to examine the potential of rejecting silicate minerals prior to leaching of the sulphides. The objective was to maximize sulphide recovery while minimizing the recovery of silicates.

This report documents the findings of the flotation component only, and the results of the hydrometallurgical program are presented in separate reports. The flotation test program commenced in June 2010 and was completed over the course of the following 3 months. Results were forwarded to Mr. Mike Moran, Dr. Mike Dry (client representative) and the SGS hydrometallurgical group as they became available.

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Testwork Summary

1. Objectives

The primary objective of the test program was to determine whether or not the Ferguson Lake mineralization is amenable to upgrading by means of flotation prior to further treatment in a hydrometallurgical process. The strategy of the flotation stage was to recover the sulphides with the associated metal values into a flotation concentrate and to reject the silicates into a tailings stream. The gangue minerals that do not dissolve in the hydrometallurgical process report to the leach residue with the PGMs and lower the PGM grade of the residue. The gangue minerals that do dissolve in the leach increase the acid requirement in the leach.

If flotation proved to be a viable process, a secondary objective was the generation of a bulk concentrate for the hydrometallurgical flowsheet development work.

2. Sample Preparation

The sample that was used for the flotation test program was provided by the hydrometallurgical group at SGS. The material was stage-crushed to minus 10 mesh and rotary split into 50 x 2 kg test charges and 20 x 20 kg test charges. Further, a representative head sample was collected and submitted for chemical analysis. The test charges were stored in a freezer to minimize the risk of oxidation.

Details on the sample origin and receipt at the SGS Lakefield site are included in the report of the hydrometallurgical group.

3. Head Analysis

3.1. Master Composite

A representative sub-sample of the Master composite was submitted for chemical analysis, which included individual assays for primary elements of interest and an ICP scan. The results are presented in Table 2. In addition to 1.48% Cu and 0.87% Ni, the composite also contained a significant concentration of PGMs totalling a combined grade of 4.5 g/t Pd, Pd, Ag, and Au.

Table 2: Head Analysis of Master Composite

Sample	Assays (%)					Assays (g/t)			
	Cu	Ni	S	S^{2-}	SiO_2	Ag	Pt	Pd	Au
Master Composite	1.48	0.87	29.0	29.0	3.7	2.0	0.20	2.28	0.08

Sample	Assays (g/t)									
	Al	As	Ba	Be	Bi	Ca	Cd	Co	Cr	Fe
Master Composite	5,400	< 30	25	< 0.1	< 20	5,000	< 2	870	29	56,800
Sample	**K**	**Li**	**Mg**	**Mn**	**Mo**	**Na**	**P**	**Pb**	**Sb**	**Se**
Master Composite	710	< 50	3,800	490	< 5	1,100	< 200	< 200	< 10	< 30
Sample	**Sn**	**Sr**	**Ti**	**Tl**	**U**	**V**	**Y**	**Zn**		
Master Composite	< 20	5.9	1,000	< 30	< 200	260	1.6	86		

3.2. Site Water

Since the commercial process is expected to use sea water as process water, the majority of the flotation tests were carried out with artificial sea water or site water that was collected at the project site. A sample of the site water was submitted for chemical analysis and the results are presented in Table 3. For comparison purposes, the typical chemical composition of sea water at 3.5% salinity is provided in the same Table. [1] The differences in detection limits are related to the fact that the sea water analysis results were published results, while the site water analysis was carried out as part of this test program.

[1] *http://www.seafriends.org.nz/oceano/seawater.htm#salinity, December 27, 2010*

Table 3: Chemical Composition of Sea Water from Project Site

Sample ID	Cl mg/L	F mg/L	NO_3 as N mg/L	NO_2 as N mg/L	SO_4 mg/L	$NH_3 + NH_4$ as N mg/L	Thiosalts as S_2O_3 mg/L
Site Water	14,700	1	< 1.25	< 1.5	2,407	0.9	< 10
Sea Water	19,400	13	-	-	2,701	-	-

Sample ID	Assays (mg/L)							
	Ag	Al	As	Ba	Be	Bi	Ca	Cd
Site Water	< 0.08	< 0.2	< 3	< 0.007	< 0.002	< 1	370	< 0.09
Sea Water	0.00028	0.001	0.0026	0.021	0.0000006	0.00002	411	0.00011
Sample ID	**Co**	**Cr**	**Cu**	**Fe**	**K**	**Li**	**Mg**	**Mn**
Site Water	< 0.3	< 0.1	< 0.1	< 0.2	340	< 2	1,100	< 0.04
Sea Water	0.00039	0.0002	0.009	0.0034	392	0.17	1,290	0.0004
Sample ID	**Mo**	**Na**	**Ni**	**P**	**Pb**	**Sb**	**Se**	**Sn**
Site Water	< 0.6	10,000	< 0.6	< 5	< 2	< 1	< 3	< 2
Sea Water	0.01	10,800	0.0066	0.088	0.0003	0.00033	0.0009	0.00081
Sample ID	**Sr**	**Ti**	**Tl**	**U**	**V**	**W**	**Y**	**Zn**
Site Water	6.4	< 0.02	< 3	< 2	< 0.2	< 2	< 0.02	< 0.7
Sea Water	8.1	0.001	-	0.0033	0.0019	< 0.000001	0.000013	0.005

4. Flotation

4.1. Rougher Flotation Tests

A total of eleven rougher flotation tests were carried out on the Ferguson Lake composite using 2 kg flotation test charges. A summary of the pertinent test conditions and results of these tests is presented in Table 4. Detailed mass balances including sizing data is provided in Appendix A. Note that the assays and distribution of chalcopyrite, pentlandite, pyrrhotite, and non-sulphide gangue minerals are based on calculations using typical chemical compositions of those minerals. Since no electron microprobe analysis was carried out on the Ferguson Lake samples to determine the actual chemical composition of those minerals, these calculations have to be treated as estimates.

Table 4: Summary of Rougher Flotation Results (F1 to F8, F10 to F12)

Test No	Products	Assays, %, g/t								% Distribution						
		Wt	Cu	Ni	S	Cp	Pn	Po	Gn	Cu	Ni	S	Cp	Pn	Po	Gn
	Rougher Conc 1	13.0	5.55	1.77	36.2	16.1	4.18	76.8	2.92	48.3	26.4	16.1	48.3	32.6	14.0	1.7
F1	Rougher Conc 1-2	55.4	2.58	1.15	36.4	7.46	2.29	87.0	3.24	95.2	73.0	68.8	95.2	75.9	67.2	8.1
P_{80}=117 microns	Rougher Conc 1-3	74.0	1.99	1.09	36.2	5.77	2.09	88.1	4.08	98.3	92.0	91.3	98.3	92.6	90.8	13.6
60g/t PAX	Rougher Conc1-4	80.6	1.84	1.06	35.9	5.34	2.03	87.9	4.76	99.1	98.1	98.7	99.1	97.9	98.7	17.3
Natural pH	Rougher Conc 1-5	81.4	1.83	1.06	35.8	5.29	2.02	87.6	5.13	99.2	98.7	99.3	99.2	98.4	99.4	18.8
	Rougher Tail	18.6	0.07	0.06	1.04	0.19	0.15	2.43	97.2	0.8	1.3	0.7	0.8	1.6	0.6	81.2
	Head (calc)	100.0	1.50	0.87	29.3	4.34	1.67	71.7	22.2	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
	Rougher Conc 1	5.85	8.24	2.48	35.2	23.9	6.3	65.1	4.65	33.2	16.9	7.0	33.2	22.8	5.3	1.2
F2	Rougher Conc 1-2	39.9	3.37	1.27	36.1	9.76	2.67	83.8	3.78	92.7	59.1	49.2	92.7	65.4	46.6	6.7
P_{80}=100 microns	Rougher Conc 1-3	62.8	2.25	1.13	36.2	6.51	2.24	87.3	3.93	97.4	83.2	77.7	97.4	86.6	76.4	11.0
60g/t PAX	Rougher Conc1-4	75.8	1.89	1.07	36.0	5.49	2.05	87.8	4.67	99.0	94.6	93.1	99.0	95.5	92.8	15.8
Natural pH	Rougher Conc 1-5	80.6	1.79	1.04	35.7	5.20	1.98	87.4	5.45	99.7	98.2	98.2	99.7	98.3	98.1	19.6
	Rougher Tail	19.4	0.03	0.08	2.75	0.07	0.14	7.05	92.7	0.3	1.8	1.8	0.3	1.7	1.9	80.4
	Head	100.0	1.45	0.86	29.3	4.20	1.63	71.8	22.4	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
	Rougher Conc 1	7.26	8.75	2.40	35.1	25.4	6.13	63.7	4.82	42.8	20.0	8.6	42.9	26.9	6.3	1.7
F3	Rougher Conc 1-2	32.4	4.21	1.44	36.3	8.41	2.32	86.8	2.45	91.9	53.4	39.7	92.1	62.2	36.1	4.7
P_{80}=86 microns	Rougher Conc 1-3	65.2	2.23	1.14	36.3	0.81	1.33	93.6	4.23	98.1	84.9	79.7	98.3	88.6	78.0	11.3
60g/t PAX	Rougher Conc1-4	77.1	1.91	1.08	36.0	0.43	1.10	89.2	9.30	99.3	95.0	93.5	99.5	96.4	92.5	16.7
Natural pH	Rougher Conc 1-5	80.8	1.83	1.06	35.8	0.32	0.89	80.0	18.8	99.6	97.7	97.4	99.8	98.4	96.6	20.0
MagSep of Ro Tails	Rougher Tail - Mags	13.0	0.02	0.10	5.06	0.06	0.14	13.1	86.7	0.2	1.5	2.2	0.2	1.1	2.4	53.0
	Rougher Tail - Non Mags	6.2	0.06	0.11	1.95	0.17	0.26	4.75	94.8	0.2	0.8	0.4	0.2	1.0	0.4	27.4
	Head	100.0	1.48	0.87	29.7	4.30	1.65	73.3	20.8	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
	Rougher Conc 1	10.2	4.56	1.63	35.8	13.2	3.76	78.8	4.25	30.6	18.8	12.6	30.6	22.4	11.3	1.9
F4	Rougher Conc 1-2	34.9	3.99	1.31	35.9	11.6	2.82	81.3	4.32	91.9	52.1	43.3	91.9	57.7	40.2	6.5
P_{80}~ 117 microns	Rougher Conc 1-3	52.9	2.78	1.23	35.9	8.07	2.53	84.9	4.52	97.2	73.7	65.7	97.2	78.5	63.6	10.2
60g/t PAX	Rougher Conc1-4	73.9	2.04	1.11	35.6	5.91	2.18	86.4	5.49	99.4	93.1	91.0	99.4	94.3	90.4	17.4
90g/t Na_2SiO_4	Rougher Conc 1-5	79.9	1.89	1.08	35.2	5.49	2.09	85.7	6.71	99.8	97.7	97.1	99.8	98.1	97.0	23.0
	Rougher Tail	20.1	0.01	0.10	4.12	0.03	0.16	10.7	89.1	0.2	2.3	2.9	0.2	1.9	3.0	77.0
	Head	100.0	1.51	0.88	28.9	4.39	1.70	70.6	23.3	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
	Rougher Conc 1	12.3	4.46	1.56	35.6	12.9	3.56	78.7	4.83	36.8	22.1	15.0	36.8	26.2	13.6	2.6
F5	Rougher Conc 1-2	39.3	3.39	1.17	35.9	9.83	2.40	83.3	4.50	89.9	53.4	48.7	89.9	56.9	46.2	7.6
K80 ~ 117 microns	Rougher Conc 1-3	64.5	2.23	1.08	36.2	6.46	2.08	87.5	3.96	96.8	80.5	80.5	96.8	80.9	79.6	11.0
60g/t PAX	Rougher Conc1-4	80.3	1.84	1.06	35.8	5.34	2.03	87.5	5.09	99.6	98.5	99.1	99.6	98.2	99.1	17.6
pH=6 (H_2SO_4)	Rougher Conc 1-5	81.3	1.82	1.05	35.5	5.28	2.00	86.9	5.82	99.6	98.5	99.6	99.6	98.0	99.6	20.4
	Rougher Tail	18.7	0.03	0.07	0.64	0.10	0.17	1.44	98.3	0.4	1.5	0.4	0.4	2.0	0.4	79.6
	Head	100.0	1.49	0.87	29.0	4.31	1.66	70.9	23.1	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							

Continued

Test No	Products	Assays, %, g/t								% Distribution						
		Wt	Cu	Ni	S	Cp	Pn	Po	Gn	Cu	Ni	S	Cp	Pn	Po	Gn
	Rougher Conc 1	17.4	3.35	1.29	37.2	9.71	2.70	86.6	0.98	38.9	26.1	21.7	38.9	29.0	20.6	0.8
F6	Rougher Conc 1-2	62.4	2.26	1.06	37.1	6.55	2.00	89.9	1.51	94.2	77.0	77.7	94.2	77.2	76.8	4.5
P_{80}=174 microns	Rougher Conc 1-3	70.9	2.07	1.06	37.0	5.99	2.00	90.2	1.79	97.8	87.5	88.0	97.8	87.5	87.5	6.1
60g/t PAX	Rougher Conc1-4	76.3	1.95	1.06	36.9	5.64	2.00	90.2	2.18	99.1	94.4	94.3	99.1	94.6	94.1	7.9
	Rougher Conc 1-5	79.2	1.88	1.06	36.7	5.46	1.99	89.9	2.69	99.6	97.6	97.5	99.6	97.7	97.4	10.2
	Rougher Tail	20.8	0.03	0.10	3.60	0.09	0.18	9.23	90.5	0.4	2.4	2.5	0.1	0.2	9.2	90.5
	Head	100.0	1.50	0.86	29.8	4.34	1.62	73.1	20.9	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
	Rougher Conc 1	0.58	4.48	2.46	32.7	13.0	6.24	68.7	12.1	1.7	1.7	0.6	1.7	2.2	0.6	0.3
F7	Rougher Conc 1-2	32.3	3.04	1.21	36.3	8.80	2.49	85.3	3.36	64.7	45.4	39.5	64.7	49.3	37.9	5.1
K80 ~ 117 microns	Rougher Conc 1-3	74.0	1.99	1.05	36.7	5.77	1.97	89.4	2.83	97.1	90.0	91.4	97.1	89.5	91.2	9.8
60g/t PAX	Rougher Conc1-4	81.2	1.85	1.04	36.3	5.37	1.96	89.0	3.67	99.3	98.3	99.5	99.3	97.7	99.6	13.9
pH 6 (HCl)	Rougher Conc 1-5	81.9	1.84	1.04	36.1	5.34	1.96	88.4	4.32	99.6	99.0	99.7	99.6	98.5	99.8	16.5
	Rougher Tail	18.1	0.03	0.05	0.45	0.09	0.13	0.99	98.8	0.4	1.0	0.3	0.4	1.5	0.2	83.5
	Head	100.0	1.52	0.86	29.7	4.39	1.63	72.6	21.4	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
	Rougher Conc 1	14.6	8.00	1.49	36.2	23.2	3.45	70.9	2.45	77.0	24.8	17.7	77.0	30.1	14.1	1.7
F8	Rougher Conc 1-2	40.6	3.54	1.27	36.7	10.3	2.66	84.9	2.23	95.1	59.0	50.0	95.1	64.9	47.2	4.3
K80 ~ 117 microns	Rougher Conc 1-3	61.7	2.41	1.17	36.7	6.98	2.34	88.2	2.45	98.3	82.6	76.0	98.3	86.6	74.6	7.2
60g/t PAX	Rougher Conc1-4	75.5	1.99	1.10	36.5	5.77	2.12	89.0	3.12	99.4	94.7	92.5	99.4	96.1	92.0	11.2
Sea water	Rougher Conc 1-5	79.3	1.90	1.08	36.4	5.51	2.07	88.8	3.62	99.7	97.6	96.6	99.7	98.2	96.4	13.7
	Rougher Tail	20.7	0.02	0.10	4.83	0.06	0.14	12.5	87.3	0.3	2.4	3.4	0.3	1.8	3.6	86.3
	Head	100.0	1.51	0.88	29.8	4.38	1.67	73.0	21.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
F10	Ro Conc Mags	70.0	0.77	1.00	36.4	2.23	1.80	92.2	3.80	36.0	82.1	86.7	36.0	78.2	89.7	12.0
K80 ~ 117 microns	Ro Conc Non Mags	7.07	13.4	1.74	33.7	38.8	4.41	49.1	7.67	63.3	14.4	8.1	63.3	19.3	4.8	2.5
60g/t PAX	Rougher Tail	22.9	0.04	0.13	6.67	0.12	0.17	17.3	82.4	0.6	3.5	5.2	0.6	2.5	5.5	85.5
Sea water	Head (calc.)	100.0	1.50	0.85	29.4	4.34	1.61	72.0	22.1	100.0	100.0	100.0	100.0	100.0	100.0	100.0
MagSep of Ro Conc	(direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
	Rougher Conc 1	20.4	3.89	1.42	36.3	11.3	3.12	82.4	3.17	55.3	31.1	25.5	55.3	34.3	23.7	2.8
F11	Rougher Conc 1-2	68.2	2.07	1.17	35.9	5.99	2.36	86.8	4.80	98.2	86.0	84.2	98.2	87.1	83.4	14.2
K80 ~ 117 microns	Rougher Conc 1-3	76.5	1.87	1.16	35.8	5.41	2.32	87.1	5.13	99.4	95.3	94.3	99.4	95.9	93.9	17.1
60g/t PAX	Rougher Conc1-4	78.5	1.82	1.15	35.7	5.28	2.30	87.0	5.39	99.6	97.1	96.4	99.6	97.5	96.2	18.4
Sea water	Rougher Conc 1-5	80.3	1.78	1.14	35.5	5.17	2.27	86.8	5.78	99.7	98.4	98.2	99.7	98.5	98.1	20.2
As F8 & longer	Rougher Tail	19.7	0.02	0.08	2.63	0.06	0.14	6.75	93.1	0.3	1.6	1.8	0.3	1.5	1.9	79.8
flotation time	*Head*	100.0	1.44	0.93	29.0	4.16	1.85	71.0	23.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							
	Rougher Conc 1	20.0	4.56	1.31	35.0	13.2	2.86	77.4	6.48	62.0	28.9	24.2	62.0	32.1	21.9	5.5
F11	Rougher Conc 1-2	70.0	2.07	1.13	35.4	5.99	2.25	85.8	6.00	98.4	87.5	85.9	98.4	88.6	85.1	17.9
K80 ~ 117 microns	Rougher Conc 1-3	76.6	1.91	1.12	35.3	5.53	2.22	85.7	6.50	99.5	94.8	93.5	99.5	95.6	93.2	21.2
60g/t PAX	Rougher Conc1-4	79.1	1.85	1.11	35.1	5.37	2.20	85.6	6.86	99.7	97.1	96.2	99.7	97.6	95.9	23.1
Sea water	Rougher Conc 1-5	80.6	1.82	1.10	35.0	5.27	2.17	85.3	7.29	99.8	98.2	97.6	99.8	98.5	97.5	25.0
As F11 & increased	Rougher Tail	19.4	0.02	0.09	3.51	0.05	0.14	9.07	90.7	0.2	1.8	2.4	0.2	1.5	2.5	75.0
collector	*Head*	100.0	1.47	0.90	28.9	4.26	1.78	70.5	23.5	100.0	100.0	100.0	100.0	100.0	100.0	100.0
	Head (direct)		1.48	0.87	29.0	4.29	1.68	70.9	23.1							

Four tests evaluated the impact of the primary grind size on the rougher recovery. Four grind sizes ranging from P_{80} = 86 microns to P_{80} =174 microns were evaluated. The mass recovery versus the Ni and Cu recovery curves for the four tests are plotted in Figure 1 and Figure 3, respectively. Since maximizing the Cu and Ni recovery was a primary objective, Figure 2 and Figure 4 focus on the end points of the rougher flotation for Ni and Cu, respectively. The four tests recovered between 97.5% and 99% of the sulphides and approximately 20% of the gangue minerals into the rougher concentrate. The losses of Ni of 1.3 to 2.4% were slightly higher compared to the Cu losses, which remained below 1% in all four tests. At the finest and coarsest grind sizes, the Ni losses were higher compared to the intermediate grind sizes and they reached their minimum of 1.30% at P_{80}=117 microns. Although the differences were only marginal, the higher Ni losses at the finer and coarser grind sizes may have been the result of slimes and insufficient liberation, respectively. The Cu flotation response and recovery was unaffected within the evaluated grind size range. For illustration purposes, the final Cu and Ni recovery has been plotted against the primary grind size in Figure 5. Based on these results, a grind size target of P_{80} = 110-120 microns was selected for all other flotation tests.

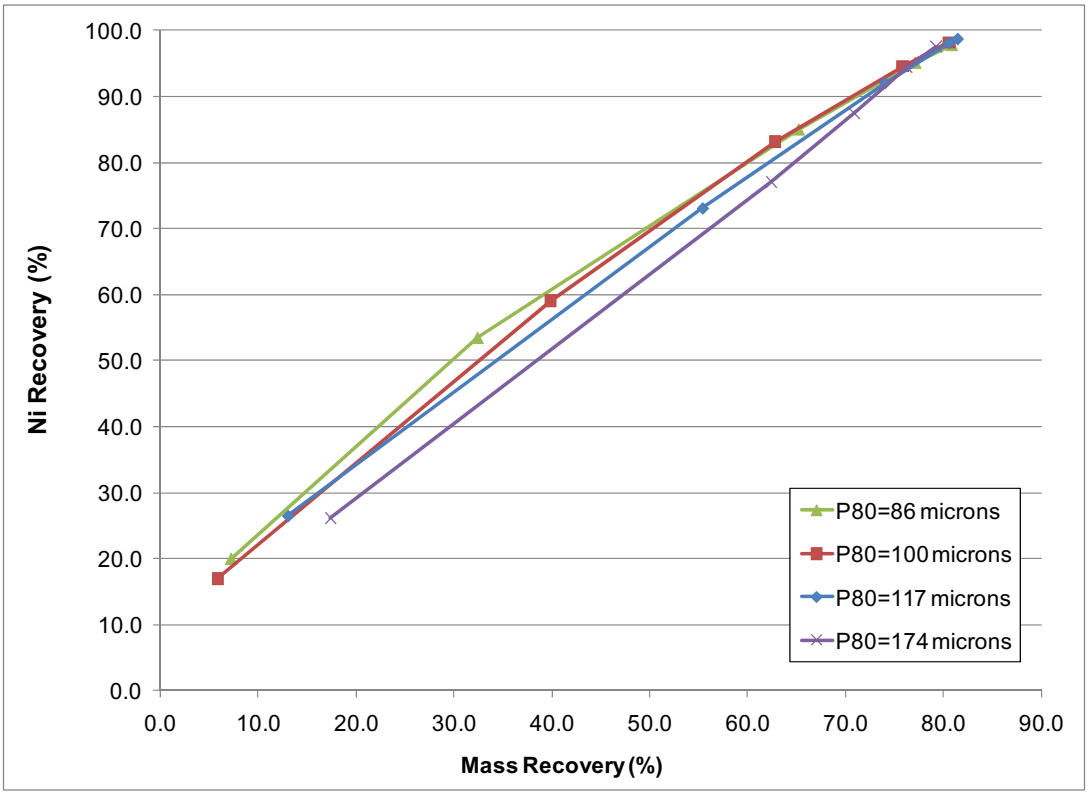


Figure 1: Impact of Primary Grind Size of Ni Flotation Performance (F1-F3, F6)

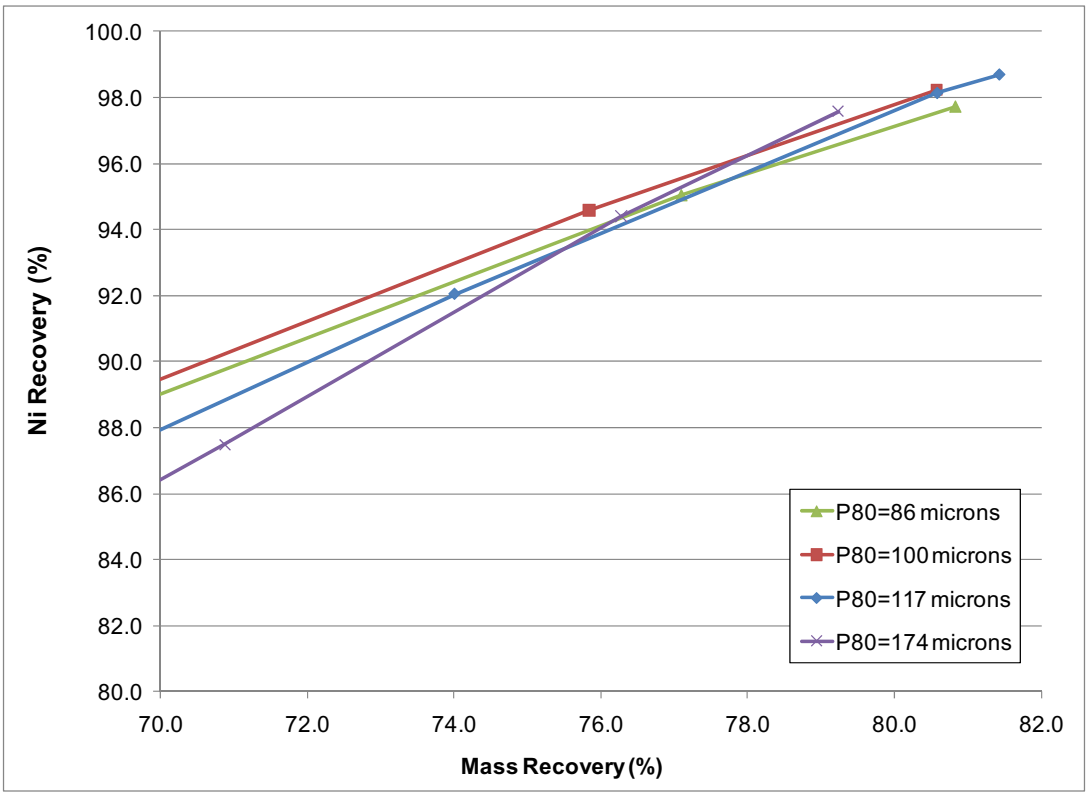


Figure 2: Impact of Primary Grind Size on Ni Flotation Performance (Zoom)

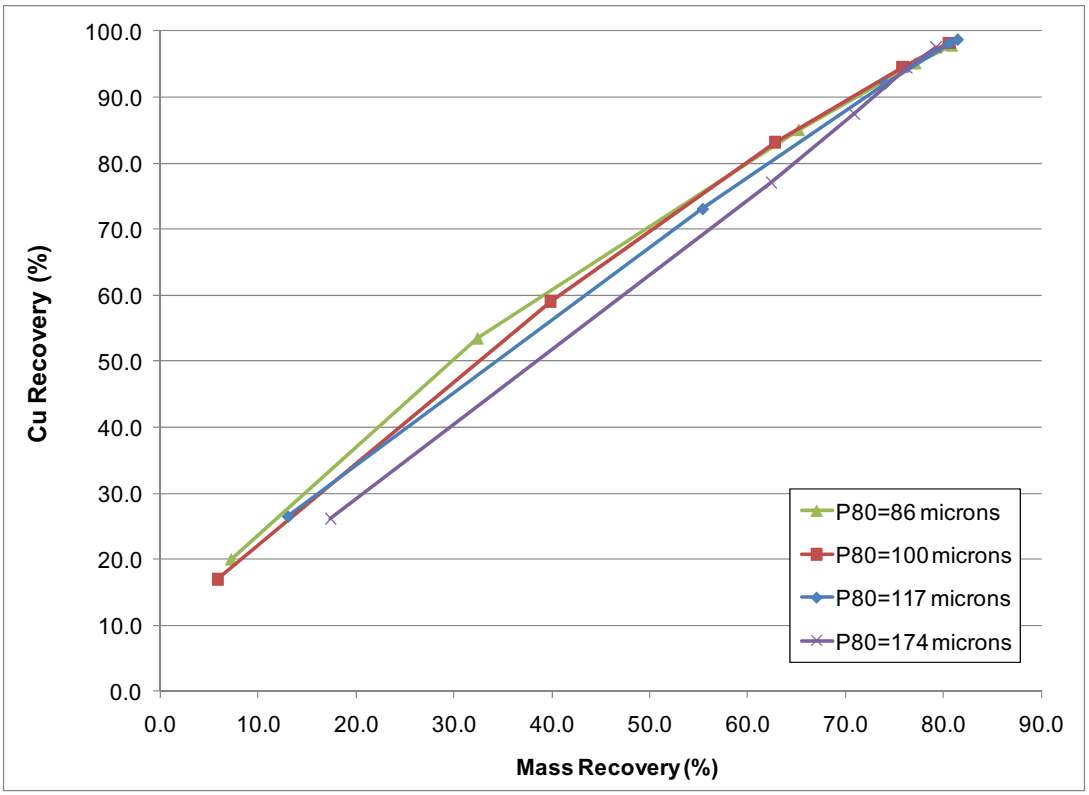


Figure 3: Impact of Primary Grind Size of Cu Flotation Performance (F1-F3, F6)

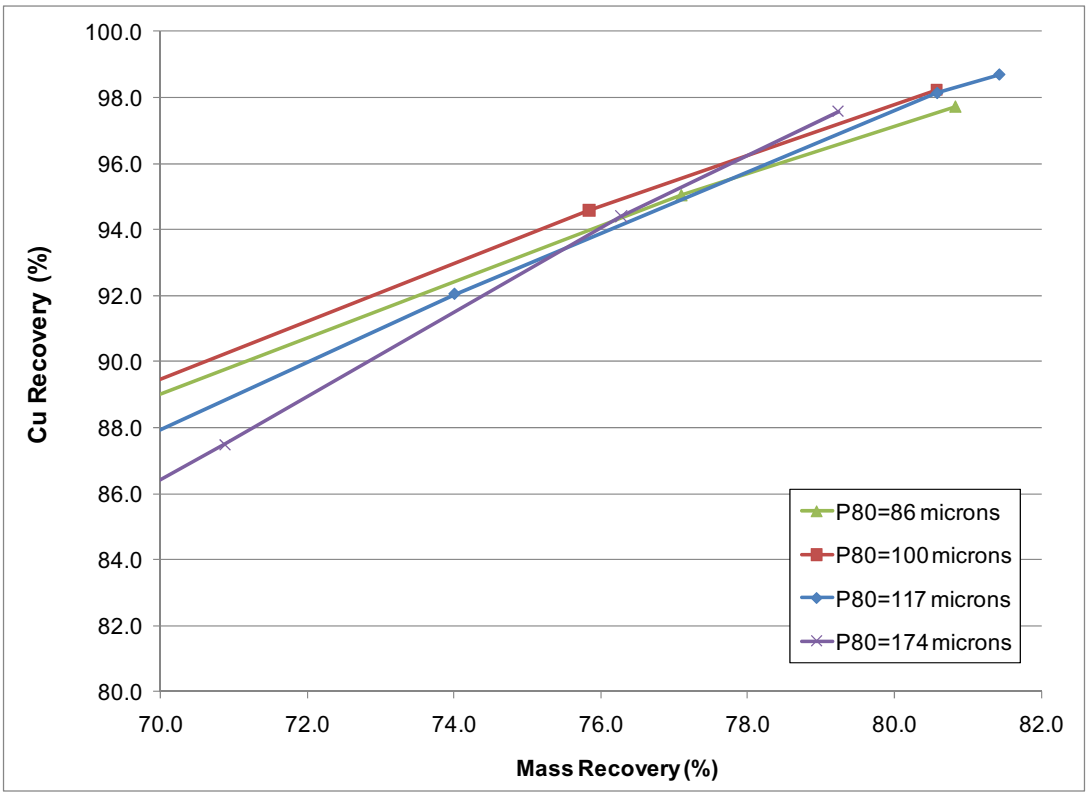


Figure 4: Impact of Primary Grind Size on Cu Flotation Performance (Zoom)

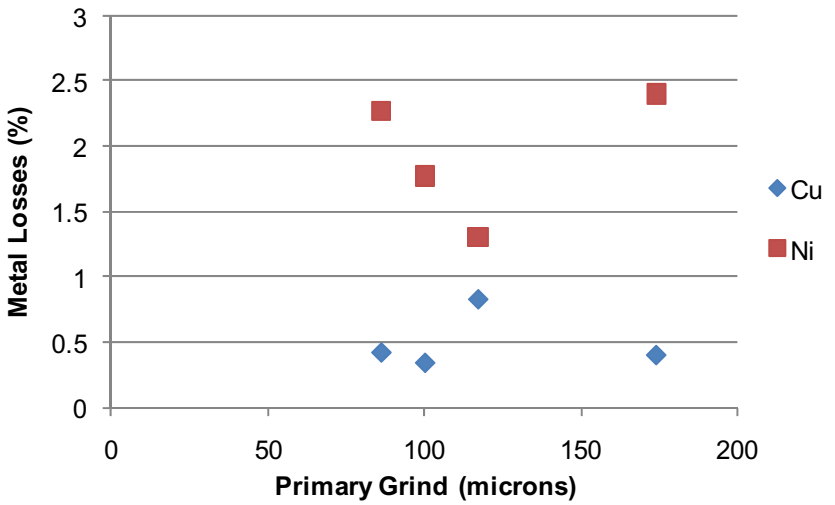


Figure 5: Impact of the Primary Grind Size on Cu and Ni Losses

The effect of the pH on the flotation response of the Ferguson Lake composite was evaluated in two tests using sulphuric acid (H_2SO_4) and hydrochloric acid (HCl). In both tests, the pH was lowered to 6.0 from the natural pH of approximately 7.5 in an effort to promote the flotation of sulphide minerals. The initial rougher flotation kinetics for Cu and Ni were superior for the test at a natural pH, followed by the sulphuric acid test and the hydrochloric acid test. However, these differences were significantly reduced after 5

minutes of flotation and were non-existent after 15 minutes. The pertinent test results are presented for Ni and Cu in Figure 6 and Figure 7, respectively. The graphs only depict the end points of the rougher flotation to magnify the differences between the two tests and the baseline test F1 at natural pH. While the curves reveal that test F7 with HCl produced marginal higher Cu and Ni recoveries compared to the other two tests, the differences are too small to be statistically significant. Based on these results, the choice of the circuit pH would be driven primarily by the operability of the circuit rather than the metallurgy. Hence, a decision was made to continue the flotation program at a natural pH since it would simplify the processing with lower costs in a commercial operation.

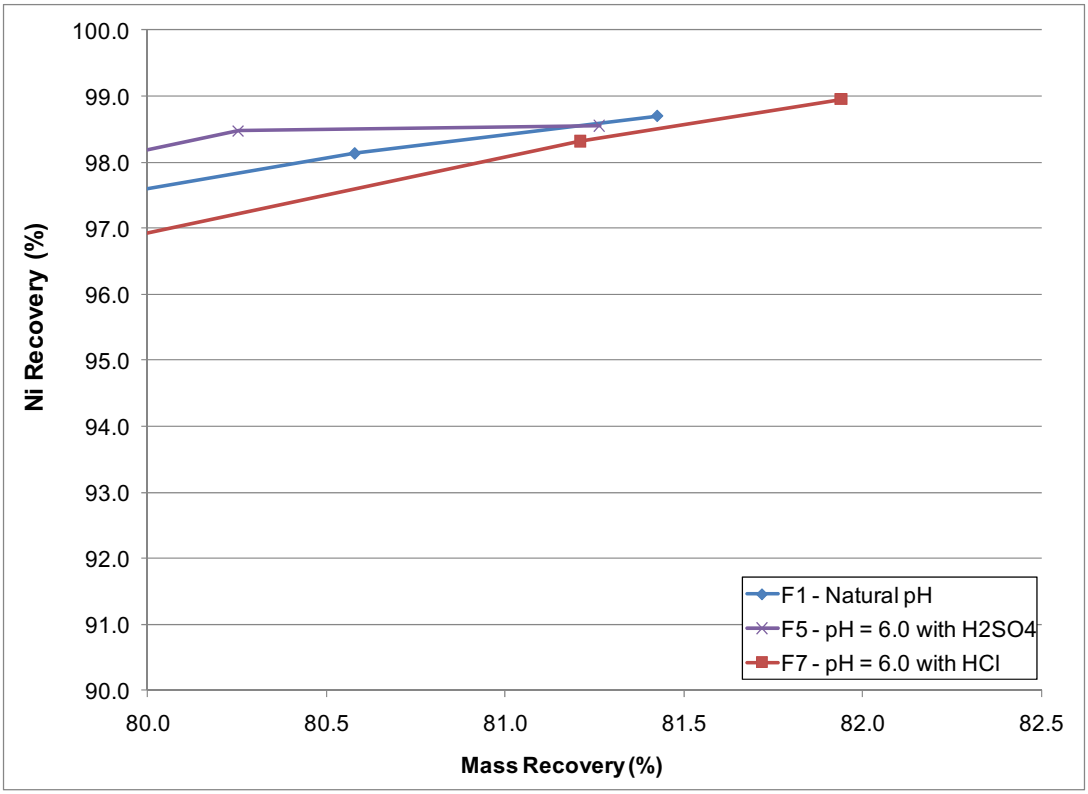


Figure 6: Effect of pH on the Ni Flotation Performance (F1, F5, and F7)

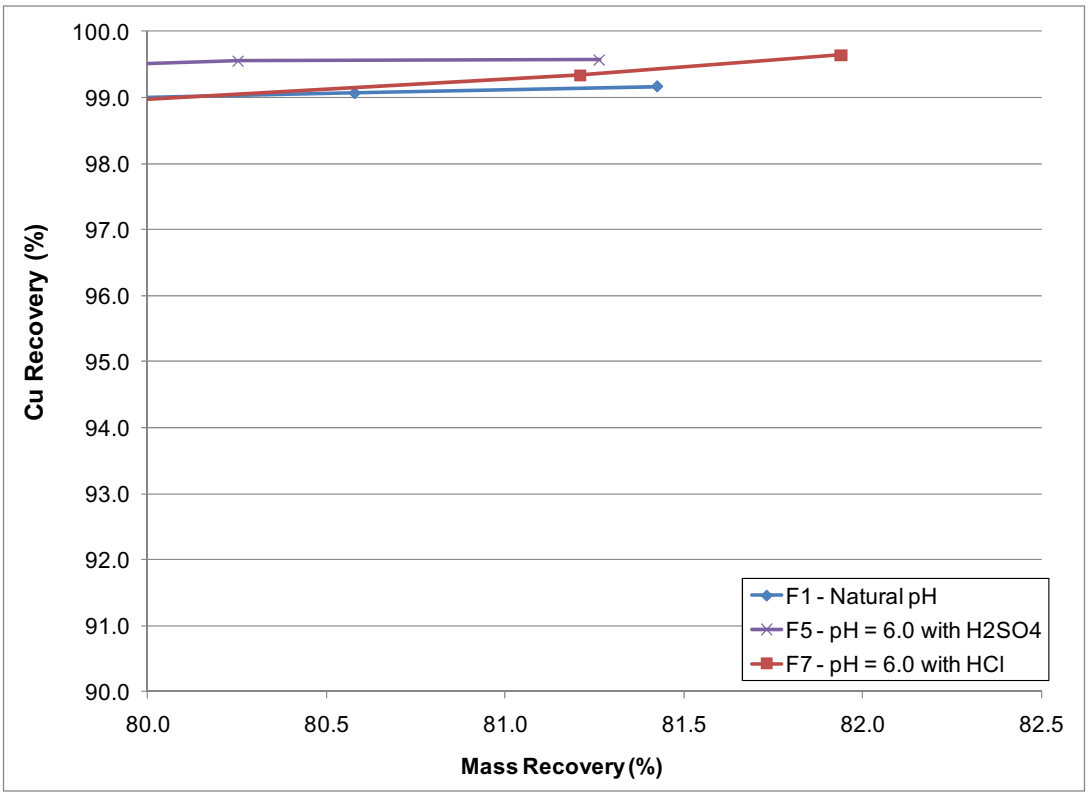


Figure 7: Effect of pH on the Cu Flotation Performance (F1, F5, and F7)

The remaining rougher flotation tests evaluated the impact of sodium silicate as a gangue depressant, artificial sea water as process water, a 50% longer flotation time, and a 50% higher collector dosage. The artificial sea water was generated by mixing 30 g of Tropic Marin salt mixture in 1 L of tap water. Tropic Marin sea salt is manufactured from pharmaceutically pure salts and is based upon the most recent scientific analysis of the tropical oceans. The results are illustrated in Figure 8 and Figure 9 for Ni and Cu respectively. The differences in the flotation performance of Cu and Ni for the four tests were only marginal and could be contributed to test-to-test variance and analytical precisions.

In conclusion, only the variation of the primary grind size produced measurable differences in the flotation response of the Ferguson Lake composite, albeit they were only small in the evaluated grind size range. A reduced flotation pH, addition of a gangue depressant as well as increased flotation and collector dosages did not yield superior flotation results. However, since a longer flotation time did not result in an increased recovery of gangue minerals, a longer flotation time was chosen for the optimized flotation conditions to ensure a consistently high recovery of Cu and Ni for different feed composites. Approximately 99% of the Cu and Ni were recovered into a flotation concentrate that contained 80% of the original flotation feed weight.

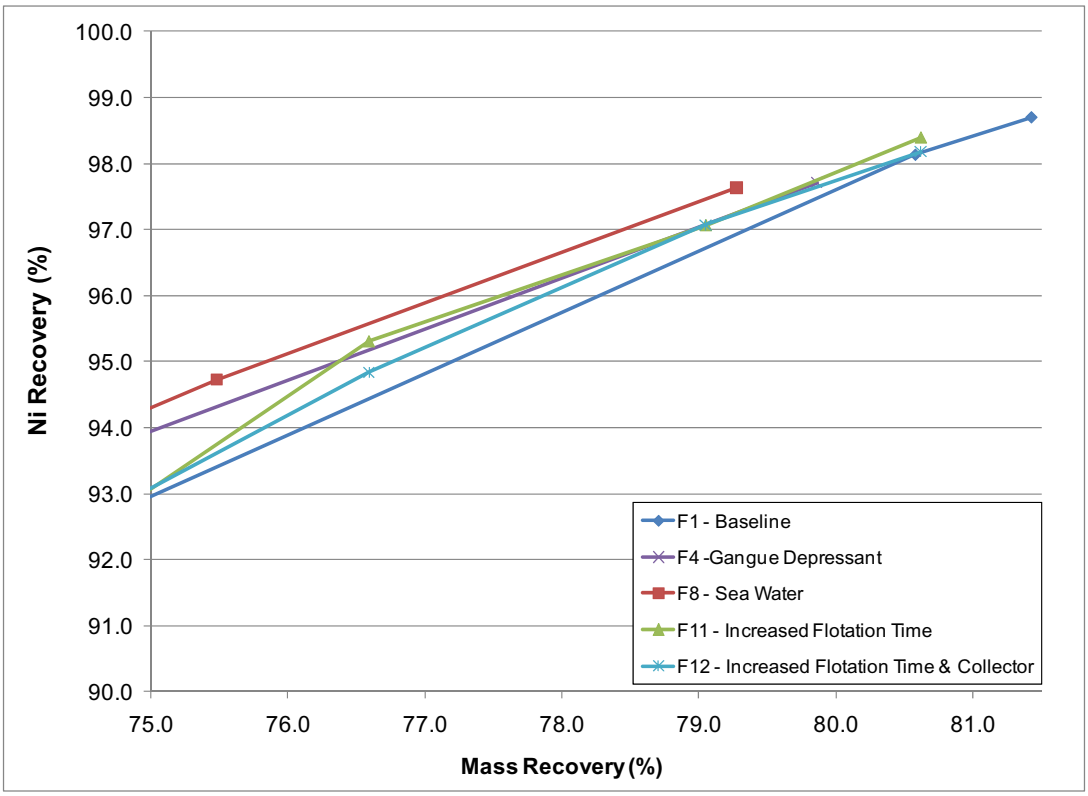


Figure 8: Effect of Selected Flotation Variable on the Ni Flotation Performance (F4, F8, F11, F12)

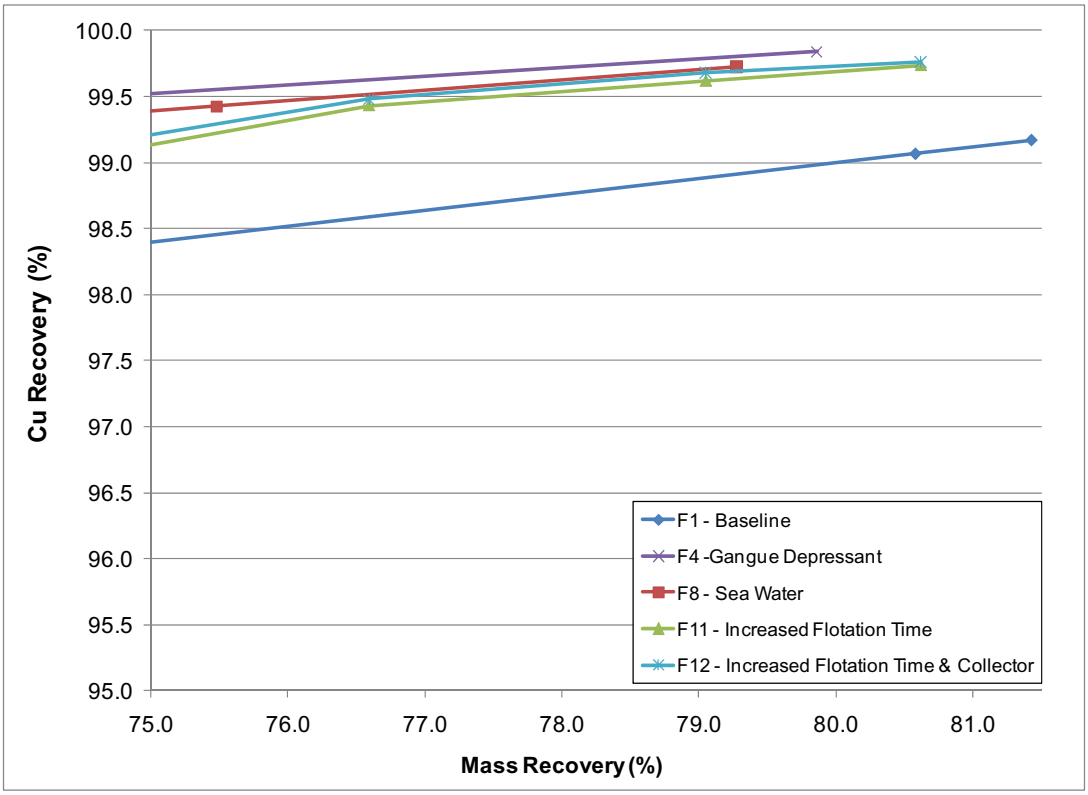


Figure 9: Effect of Selected Flotation Variable on the Cu Flotation Performance (F4, F8, F11, F12)

4.2. Cleaner Flotation Test

A single batch cleaner test F9 was carried out. The rougher flotation conditions were identical to test F8. The rougher concentrate was then subjected to a 1st cleaner with four incremental concentrates. The stage-recovery of the sulphide and gangue minerals into the combined 1st cleaner concentrate is illustrated in Figure 10. Between 99.4% and 99.9% of the sulphide minerals that reported to the rougher concentrate reported to the combined 1st cleaner concentrate. Since the stage-recovery of the gangue minerals was only 83.5%, the 1st cleaner was successful to further reject undesired gangue minerals from the flotation concentrate. The overall recovery of Cu and Ni into the 1st cleaner concentrate was 99.7% and 97.7%, respectively, while the Cu and Ni losses to the 1st cleaner tailings yielded 0.1% and 0.5%, respectively. Almost 80% of the gangue minerals were rejected into the rougher and 1st cleaner tails.

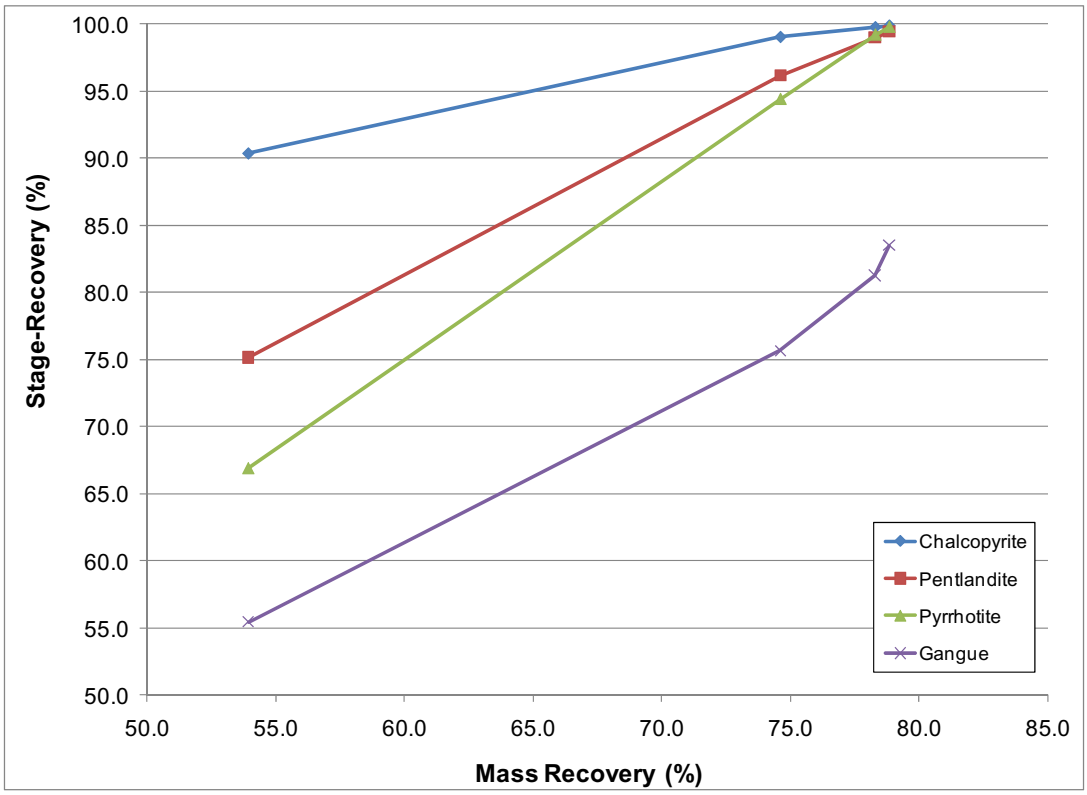


Figure 10: 1st Cleaner Mass versus Stage Recovery of Sulphide and NSG Minerals

4.3. Tailings Analysis

A sample of the F8 rougher tails was submitted for whole rock analysis (WRA), Pt, Pd, and Au analysis to determine the chemical composition of the sample. The results of the whole rock analysis are shown in Table 5.

Table 5: Whole Rock Analysis of F8 Rougher Tails

Product	Assays (%)						
	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O
F-8 Ro Tail	14.5	3.81	73.4	2.52	2.04	0.58	0.29
Product	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5	LOI	Sum
F-8 Ro Tail	0.59	0.03	0.26	0.01	0.19	0.79	99

5. Magnetic Separation

5.1. Magnetic Separation of Rougher Concentrate

Test F10 evaluated suitability of magnetic separation on the rougher concentrate. The mass balance for the magnetic separation component is presented in Table 6. Although the non-magnetic fraction contained significantly higher concentrations of all pay-metals, the losses to the magnetic fraction were very high and ranged between 36.0% for Cu and 82.1% for Ni. Hence, magnetic separation does not constitute a viable upgrading method for the rougher concentrate.

Table 6: Magnetic Separation of F10 Rougher Concentrate

Product	Weight	Assays, %, g/t						% Distribution					
	%	Cu	Ni	S	Au	Pt	Pd	Cu	Ni	S	Au	Pt	Pd
Ro Conc Mags	70.0	0.77	1.00	36.4	0.07	0.17	1.99	36.0	82.1	86.7	37.5	42.2	58.4
Ro Conc Non Mags	7.1	13.4	1.74	33.7	1.09	2.18	13.2	63.3	14.4	8.1	59.0	54.6	39.2
Rougher Tail	22.9	0.04	0.13	6.67	0.02	0.04	0.25	0.6	3.5	5.2	3.5	3.2	2.4
Head (calc.)	100.0	1.50	0.85	29.4	0.13	0.28	2.38	100.0	100.0	100.0	100.0	100.0	100.0
(direct)		1.48	0.87	29.0	0.02	0.20	2.28						

The results for F10 were also used to calculate the PGM recovery into the rougher concentrate. The results are presented in Table 7 and reveal that the recovery ranged between 96.5% for Au and 97.6% for Pd.

Table 7: PGM Recovery to Flotation Concentrate

	Mass	Au	Pt	Pd
	%	g/t	g/t	g/t
Flotation Concentrate	77.1	0.16	0.36	3.02
Feed	100	0.13	0.29	2.39
Recovery		96.5	96.8	97.6

The two magnetic separation products and the flotation rougher tails were submitted for a whole rock analysis to quantify the deportment of the major elements that are present in the gangue minerals. The results of this analysis are presented in Table 8.

Table 8: Whole Rock Analysis of Magnetics, Non-Magnetics, and Rougher Tails

Product	Assays (%)						
	SiO_2	Al_2O_3	Fe_2O_3	MgO	CaO	Na_2O	K_2O
Ro Conc Mags	0.56	0.17	88.3	0.07	< 0.05	-	< 0.05
Ro Conc Non-Mags	5.49	2.11	57.6	0.88	0.60	-	0.16
Ro Tails	14.0	3.77	72.5	2.39	1.91	0.54	0.29
Product	**TiO_2**	**P_2O_5**	**MnO**	**Cr_2O_3**	**V_2O_5**	**LOI**	**Sum**
Ro Conc Mags	0.03	-	0.01	0.01	< 0.015	-	-
Ro Conc Non-Mags	0.25	-	0.08	< 0.012	< 0.015	-	-
Ro Tails	0.54	0.03	0.23	0.01	0.16	1.51	97.9

5.2. Magnetic Separation of Rougher Tails

In order to determine whether or not a separate magnetite by-product could be generated, the tailings from the rougher flotation tests F1 and F4 were subjected to a series of Davis Tube tests at grind sizes of P_{80} = 134, 65, 27, and 19 microns. In order to generate the finer products, the flotation tailings were stage-pulverized to the desired target size.

Davis Tube tests are carried out to determining the magnetic content in ore samples. The Davis Tube equipment consists of an electromagnet, which can generate a magnetic field intensity of up to 4,000 Gauss, a glass separation tube and a motor driven agitation mechanism. The tube is positioned between the poles of the magnet. During the operation of a small electric motor moves the glass tube forward and backward while rotating it simultaneously. Any magnetic particles present in the sample are collected in the zone of magnetism. Eventually all non-magnetics are flushed from the tube. A clean concentrate of magnetics remains in the tube and is manually removed at the end of the test.

The results are presented in Table 9. The three coarsest samples yielded encouraging results with very high Fe_3O_4 concentrations and SiO_2 grades of less than 1% SiO_2, which is generally considered a target value for a magnetite concentrate. However, the sulphur grades of approximately 2% were elevated. Hence, further optimization of the conditions would be required to produce a clean magnetite concentrate.

Satmagan tests that were carried out on the magnetic fractions of the Davis Tube tests yielded concentrations between 64.9% and 70.1% magnetic Fe. The highest value was achieved for the unpulverized sample and then gradually decreased at finer sizes. The magnetic Fe content of the unpulverized sample was close to the theoretic maximum of 72.4% Fe of pure magnetite, which further confirmed the encouraging initial results.

Table 9: Davis Tube Testing of F1 and F2 Rougher Tails

DT 1: F1 and F2 Rougher Tail unpulverised

Product	Weight		Assay, %														
	g	%	S	SiO_2	Al_2O_3	Fe_3O_4	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5	LOI	Sum
Mags	7.08	69.1	1.79	0.73	0.42	96.7	0.12	0.06	0.03	0.02	0.25	< 0.01	0.13	0.02	0.28	-2.3	98.3
Non Mags	3.16	30.9	0.75	43.9	11.2	23	7.54	6.17	1.51	0.82	1.15	0.10	0.51	0.03	0.04	2.7	99.4
Rougher Tail (Feed)	10.2	100.0	1.47	14.1	3.75	74.0	2.41	1.95	0.49	0.27	0.53	-	0.25	0.02	0.21	-0.8	98.6
	Weight		Distribution, %														
	%		S	SiO_2	Al_2O_3	Fe_3O_4	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5		
Mags	69.1		84.2	3.6	7.8	90.4	3.4	2.1	4.3	5.2	32.8	-	36.4	59.9	94.0		
Non Mags	30.9		15.8	96.4	92.2	9.6	96.6	97.9	95.7	94.8	67.2	-	63.6	40.1	6.0		

DT 2: F1 and F2 Rougher Tail pulverised to -200 mesh

Product	Weight		Assay, %														
	g	%	S	SiO_2	Al_2O_3	Fe_3O_4	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5	LOI	Sum
Mags	6.99	74.2	2.28	0.53	0.32	97.6	0.10	0.04	< 0.01	< 0.01	0.20	< 0.01	0.12	0.03	0.28	-2.1	99.4
Non Mags	2.43	25.8	1.14	43.5	11.1	24.0	7.46	6.10	1.48	0.71	0.97	0.08	0.48	0.03	0.04	2.8	99.9
Rougher Tail (Feed)	9.4	100.0	1.99	11.6	3.10	78.6	2.00	1.60	-	-	0.40	-	0.21	0.03	0.22	-0.8	99.5
	Weight		Distribution, %														
	%		S	SiO_2	Al_2O_3	Fe_3O_4	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5		
Mags	74.2		85.2	3.4	7.7	92.1	3.7	1.9	-	-	37.2	-	41.8	74.2	95.3		
Non Mags	25.8		14.8	96.6	92.3	7.9	96.3	98.1	-	-	62.8	-	58.2	25.8	4.7		

DT 3: F1 and F2 Rougher Tail pulverised to -325 mesh

Product	Weight		Assay, %														
	g	%	S	SiO_2	Al_2O_3	Fe_3O_4	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5	LOI	Sum
Mags	6.86	79.4	1.76	0.95	0.49	95.8	0.19	0.10	0.02	< 0.01	0.18	< 0.01	0.13	0.03	0.27	-2.1	97.8
Non Mags	1.78	20.6	1.33	42.1	10.5	25.3	6.93	6.09	1.42	0.55	1.42	0.10	0.54	0.03	0.05	3.6	99.9
Rougher Tail (Feed)	8.6	100.0	1.67	9.43	2.55	81.3	1.58	1.33	0.31	-	0.44	-	0.21	0.03	0.22	-0.9	98.3
	Weight		Distribution, %														
	%		S	SiO_2	Al_2O_3	Fe_3O_4	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5		
Mags	79.4		83.6	8.0	15.2	93.6	9.6	6.0	5.1	-	32.8	-	48.1	79.4	95.4		
Non Mags	20.6		16.4	92.0	84.8	6.4	90.4	94.0	94.9	-	67.2	-	51.9	20.6	4.6		

DT 4: F1 and F2 Rougher Tail pulverised to -500 mesh

Product	Weight		Assay, %														
	g	%	S	SiO_2	Al_2O_3	Fe_3O_4	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5	LOI	Sum
Mags	7.35	85.5	1.83	2.13	0.82	93.4	0.38	0.23	0.05	0.05	0.21	< 0.01	0.14	0.04	0.26	-1.8	97.7
Non Mags	1.25	14.5	0.99	42.3	10.3	25.6	6.72	6.10	1.46	0.47	1.48	0.10	0.54	0.05	0.05	3.4	99.6
Rougher Tail (Feed)	8.6	100.0	1.71	7.97	2.20	83.5	1.30	1.08	0.25	0.11	0.39	-	0.20	0.04	0.23	-1.1	98.0
	Weight		Distribution, %														
	%		S	SiO_2	Al_2O_3	Fe_3O_4	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	Cr_2O_3	V_2O_5		
Mags	85.5		91.6	22.8	31.9	95.5	25.0	18.1	16.8	38.5	45.5	-	60.4	82.5	96.8		
Non Mags	14.5		8.4	77.2	68.1	4.5	75.0	81.9	83.2	61.5	54.5	-	39.6	17.5	3.2		

6. Bulk Flotation Tests

In preparation for the bulk concentrate production, test B-1 was a rougher kinetics test using a 10 kg test charge in a 28 L flotation cell. The chosen test conditions were identical to test F11 with the required adjustment of the flotation time by a factor of two to scale up from a 4 L to a 28 L flotation cell. Further, real sea water from the project site was used in the test. The Cu and Ni recovery versus the mass pull into the rougher concentrate for test B-1 and the corresponding 2-kg rougher test F-11 is plotted in Figure 11. Complete mass balances are included in Appendix C. The individual Cu and Ni recovery versus mass recovery data points fell on the same overall curves for Cu and Ni. However, the flotation rate for the 10 kg test was noticeably slower. Hence, the flotation time for the 10 kg test was increased from 44 minutes to 51 minutes. This adjustment (B2) produced a Ni recovery that was in line with the results of test F11 as evidenced in Figure 12. These modified conditions were then used for the production of the bulk concentrate for downstream hydrometallurgical testing, which comprised a total of twenty-five 10 kg batch flotation tests. This bulk test was designated B3.

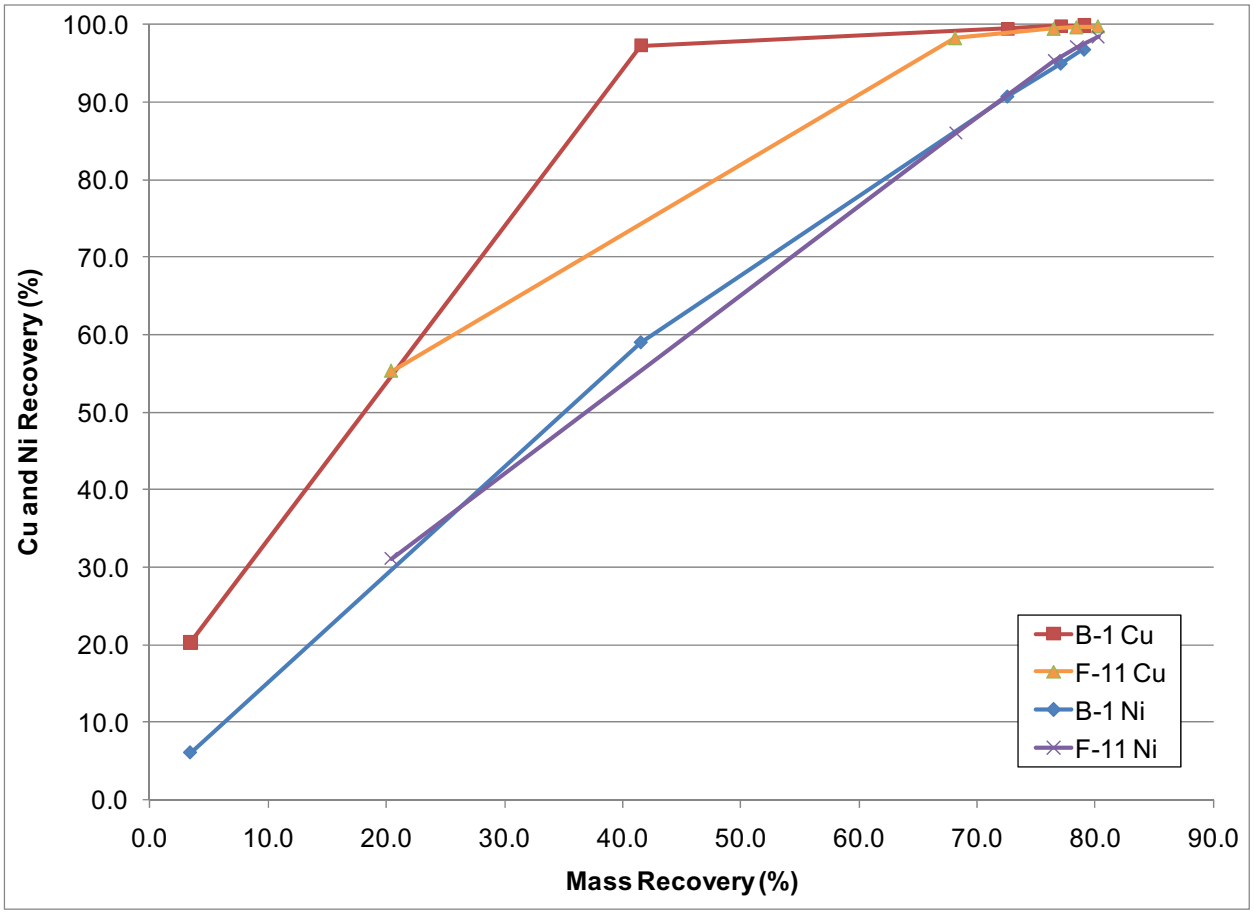


Figure 11: Cu and Ni Recovery versus Mass Recovery into the Rougher Concentrate

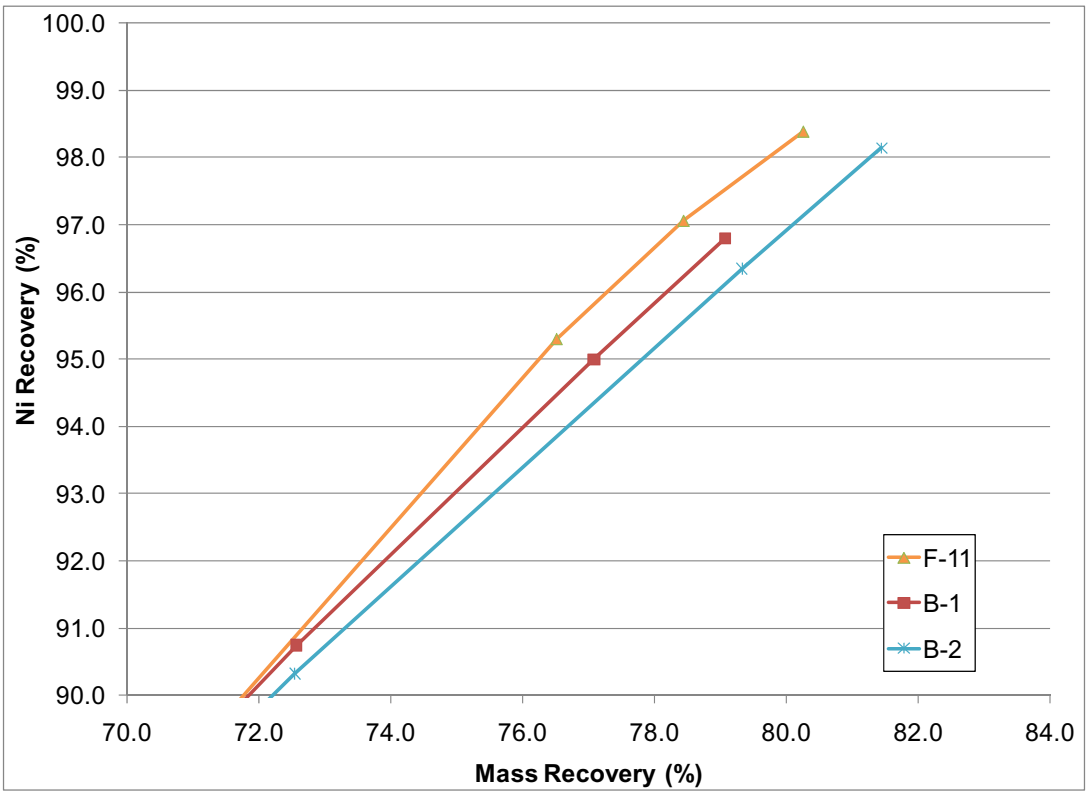


Figure 12: Mass Recovery versus Ni Recovery (F11, B1, B2)

The bulk concentrates and the rougher tails from the twenty-five batch tests were combined at the end of the bulk flotation testing. The combined rougher concentrate was dewatered using a large pilot-scale pan filter, while the combined rougher tails was kept as a pulp. Both products were sub-sampled and submitted for chemical analysis. Note that three cycles were not included in the bulk product as the material was forwarded to the hydrometallurgical group for preliminary testing before the entire bulk flotation test was completed. The mass balance for the bulk flotation test is presented in Table 10 and the results were similar to those of test F11.

Table 10: Bulk Flotation Mass Balance (B3)

Product	Weight		Assays, %, g/t			% Distribution		
	kg	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc	184.7	84.0	1.70	1.03	35.5	99.6	97.8	99.1
Rougher Tail	35.2	16.0	0.04	0.12	1.71	0.4	2.2	0.9
Head (calc.)	219.9	100.0	1.43	0.88	30.1	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

7. Conclusions and Recommendations

A flotation test program was carried out on a composite sample from the Ferguson Lake deposit to determine whether or not the gangue minerals can be rejected into a tailings product without significant loss of the valuable elements. Under optimized conditions 99.7-99.9% of the Cu units and 98-99% of the

Ni units were recovered into the rougher flotation concentrate at 80% mass recovery. Further, between 96.5% and 97.6% of Pt, Pd, and Au were recovered into the rougher concentrate. At the same time, approximately 75-80% of the gangue minerals were rejected into the tailings stream.

A primary grind size target of K_{80} = 110-120 microns was chosen and a simple reagent suite consisting of staged addition of 60 g/t PAX and 15 g/t MIBC, combined with flotation at a natural pH of approximately 7.5 produced the best results. An aeration time of 1 minute was applied prior to flotation to increase the pulp potential from -180 to -200 mV to approximately zero. Considering the short aeration time, a separate aeration vessel will likely not be needed in a commercial plant as the transfer from the pump sump to the flotation cell may be sufficient to increase the pulp potential. Further, the pulp potential would rapidly increase in the first rougher flotation cell. The use of the gangue depressant sodium silicate failed to improve the flotation selectivity. While the Ni and S losses were originally slightly higher using sea water due to a lower mass recovery into the rougher concentrate, a more aggressive froth removal matched the original results.

The cleaner test rejected another 4% of the gangue minerals, albeit an increased Cu and Ni losses of 0.13% and 0.44%, respectively. Unless a maximum rejection of gangue minerals is critical, the rougher concentrate should be considered a final feed for the downstream hydrometallurgical process.

Magnetic separation on the rougher concentrate resulted in increased metal grades in the non-magnetic fraction, but the high metal losses of up to 82% rendered this process unattractive. However, magnetic separation on the rougher tailings yielded more encouraging results and possibly a magnetite by-product could be generated. The addition of a magnetic separation circuit would be relatively inexpensive and the extraction of a saleable magnetite by-product would also reduce the amount of final gangue that has to be impounded. Approximately 75% of the flotation tailings reported to a magnetite concentrate in the Davis Tube tests, thus reducing the tailings mass for disposal from 20% to only 5% of the feed rate. While the results are preliminary in nature, the generation of a magnetite by-product should be further investigated in future test programs.

Increasing the flotation time by 50% in test F11 produced a slightly higher Ni recovery of 98.4% compared to 97.6% in test F8. While this increase may be attributed to a normal test-to-test variance, the longer flotation time benefits the recovery of slow-floating fine sulphide minerals. Hence, this longer flotation time is recommended as it will represent a safety margin on a commercial scale, especially since the flotation selectivity against silicates was not negatively affected by the longer flotation time.

It is recommended to evaluate the stability of the proposed flotation flowsheet on a number of variability composites. These composites should be selected from various areas and depths of the deposit and take into account any differences in lithology and visible grain size distribution differences.

A mineralogical characterization of the flotation tails would identify the nature of metal losses. These findings could then be used to carry out an optimization program to further decrease metal losses.

Appendix A – Batch Flotation Test Mass Balances

Test No.: F-1 **Project No.**: 12323-002 **Operator**: KS **Date**: June 21/10

Purpose: Incremental rougher flotation of the Ferguson Lake Comp

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill.#4 K_{80} = 117µm

Conditions:

	Reagents, g/t		Time, minutes				
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	5		20			7.5	-150
Bulk Circuit							
Aerate					1	7.0	-60.0
Bulk Rougher 1	5	10		1	1	7.0	-60.0
Bulk Rougher 2	10			1	2		
Bulk Rougher 3	10			1	2		
Bulk Rougher 4	10	5		1	4		
Bulk Rougher 5	20	5		1	6		
Total	60	20	20	5	16		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Metallurgical Balance

Product	**Weight**		**Assays, %, g/t**			**% Distribution**		
	g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	260.3	13.0	5.55	1.77	36.2	48.3	26.4	16.1
Rougher Conc 2	846.0	42.4	1.66	0.96	36.5	46.9	46.6	52.7
Rougher Conc 3	372.0	18.6	0.25	0.89	35.4	3.1	19.0	22.5
Rougher Conc 4	131.2	6.57	0.18	0.81	33.2	0.8	6.1	7.4
Rougher Conc 5	16.9	0.85	0.18	0.58	22.7	0.1	0.6	0.7
Rougher Tail	371.0	18.6	0.07	0.06	1.04	0.8	1.3	0.7
Head (calc.)	1997.4	100.0	1.50	0.87	29.3	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	**Weight**	**Assays, %, g/t**			**% Distribution**		
	%	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	13.0	5.55	1.77	36.2	48.3	26.4	16.1
Rougher Conc 1-2	55.4	2.58	1.15	36.4	95.2	73.0	68.8
Rougher Conc 1-3	74.0	1.99	1.09	36.2	98.3	92.0	91.3
Rougher Conc 1-4	80.6	1.84	1.06	35.9	99.1	98.1	98.7
Rougher Conc 1-5	81.4	1.83	1.06	35.8	99.2	98.7	99.3

Mineral Balance

Product	**Weight**		**Assays, %**				**% Distribution**			
	g	**%**	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	260.3	13.0	16.1	4.18	76.8	2.92	48.3	32.6	14.0	1.7
Rougher Conc 2	846.0	42.4	4.81	1.71	90.1	3.34	46.9	43.4	53.2	6.4
Rougher Conc 3	372.0	18.6	0.72	1.50	91.2	6.59	3.1	16.7	23.7	5.5
Rougher Conc 4	131.2	6.57	0.52	1.33	85.7	12.4	0.8	5.2	7.8	3.7
Rougher Conc 5	16.9	0.85	0.52	0.99	58.4	40.1	0.1	0.5	0.7	1.5
Rougher Tail	371.0	18.6	0.19	0.15	2.43	97.2	0.8	1.6	0.6	81.2
Head (calc.)	1997.4	100.0	4.34	1.67	71.7	22.2	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

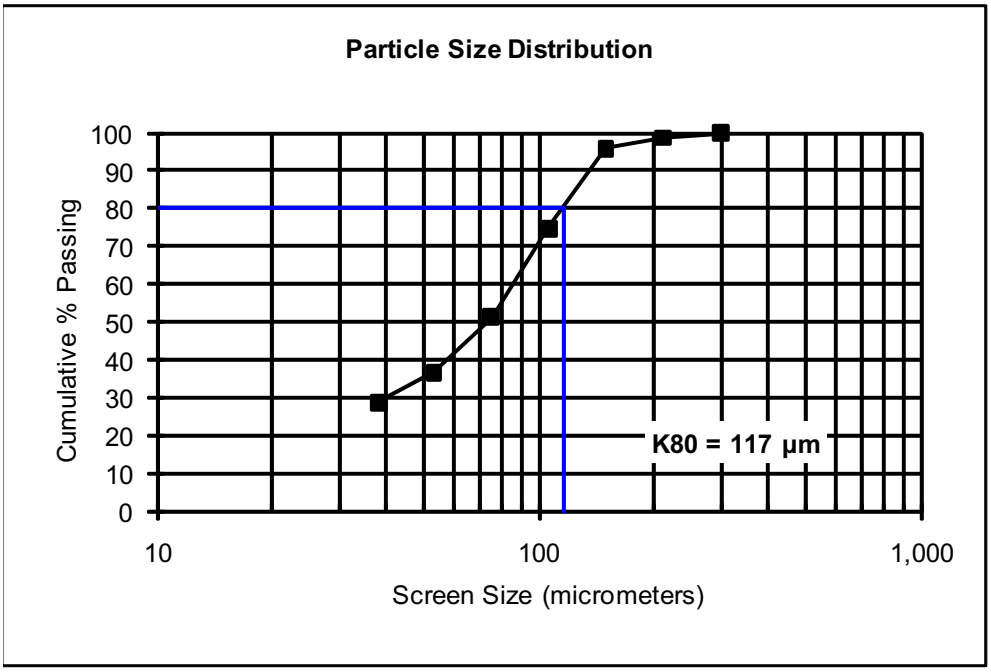
Product	**Weight**	**Assays, %**				**% Distribution**			
	%	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	13.0	16.1	4.18	76.8	2.92	48.3	32.6	14.0	1.7
Rougher Conc 1-2	55.4	7.46	2.29	87.0	3.24	95.2	75.9	67.2	8.1
Rougher Conc 1-3	74.0	5.77	2.09	88.1	4.08	98.3	92.6	90.8	13.6
Rougher Conc1-4	80.6	5.34	2.03	87.9	4.76	99.1	97.9	98.7	17.3
Rougher Conc 1-5	81.4	5.29	2.02	87.6	5.13	99.2	98.4	99.4	18.8

SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **Comb Prod** Test No.: **F1**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
48	300	0.2	0.1	0.1	99.9
65	212	2.0	1.1	1.3	98.7
100	150	5.0	2.9	4.1	95.9
150	106	37.2	21.3	25.4	74.6
200	75	41.0	23.4	48.8	51.2
270	53	25.6	14.6	63.4	36.6
400	38	13.7	7.8	71.3	28.7
Pan	-38	50.3	28.7	100.0	0.0
Total	**-**	**175.0**	100.0	**-**	**-**
K80	**117**				



Test No.: F-2 **Project No.**: 12323-002 **Operator**: KS **Date**: June 21/10

Purpose: Incremental rougher flotation of the Ferguson Lake Comp

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 25 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill.#4 K_{80} = 100µm

Conditions:

	Reagents, g/t		Time, minutes				
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	5		25			7.5	-140
Bulk Circuit							
Areate				1		7.3	-50.0
Bulk Rougher 1	5	10		1	1	7.3	-50.0
Bulk Rougher 2	10			1	2		
Bulk Rougher 3	10			1	2		
Bulk Rougher 4	10	5		1	4		
Bulk Rougher 5	20	5		1	6		
Total	60	20	25	6	15		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Metallurgical Balance

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	113.1	5.85	8.24	2.48	35.2	33.2	16.9	7.0
Rougher Conc 2	658.4	34.0	2.53	1.06	36.3	59.4	42.1	42.2
Rougher Conc 3	443.7	22.9	0.30	0.90	36.3	4.7	24.1	28.4
Rougher Conc 4	251.8	13.0	0.18	0.75	34.8	1.6	11.4	15.5
Rougher Conc 5	91.5	4.73	0.19	0.66	31.1	0.6	3.6	5.0
Rougher Tail	375.8	19.4	0.03	0.08	2.75	0.3	1.8	1.8
Head (calc.)	1934.3	100.0	1.45	0.86	29.3	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	Weight	Assays, %, g/t			% Distribution		
	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	5.85	8.24	2.48	35.2	33.2	16.9	7.0
Rougher Conc 1-2	39.9	3.37	1.27	36.1	92.7	59.1	49.2
Rougher Conc 1-3	62.8	2.25	1.13	36.2	97.4	83.2	77.7
Rougher Conc 1-4	75.8	1.89	1.07	36.0	99.0	94.6	93.1
Rougher Conc 1-5	80.6	1.79	1.04	35.7	99.7	98.2	98.2

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	113.1	5.85	23.9	6.34	65.1	4.65	33.2	22.8	5.3	1.2
Rougher Conc 2	658.4	34.0	7.33	2.03	87.0	3.63	59.4	42.6	41.3	5.5
Rougher Conc 3	443.7	22.9	0.87	1.50	93.4	4.21	4.7	21.2	29.9	4.3
Rougher Conc 4	251.8	13.0	0.52	1.11	90.1	8.23	1.6	8.9	16.3	4.8
Rougher Conc 5	91.5	4.73	0.55	0.97	80.5	18.0	0.6	2.8	5.3	3.8
Rougher Tail	375.8	19.4	0.07	0.14	7.05	92.7	0.3	1.7	1.9	80.4
Head (calc.)	1934.3	100.0	4.20	1.63	71.8	22.4	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

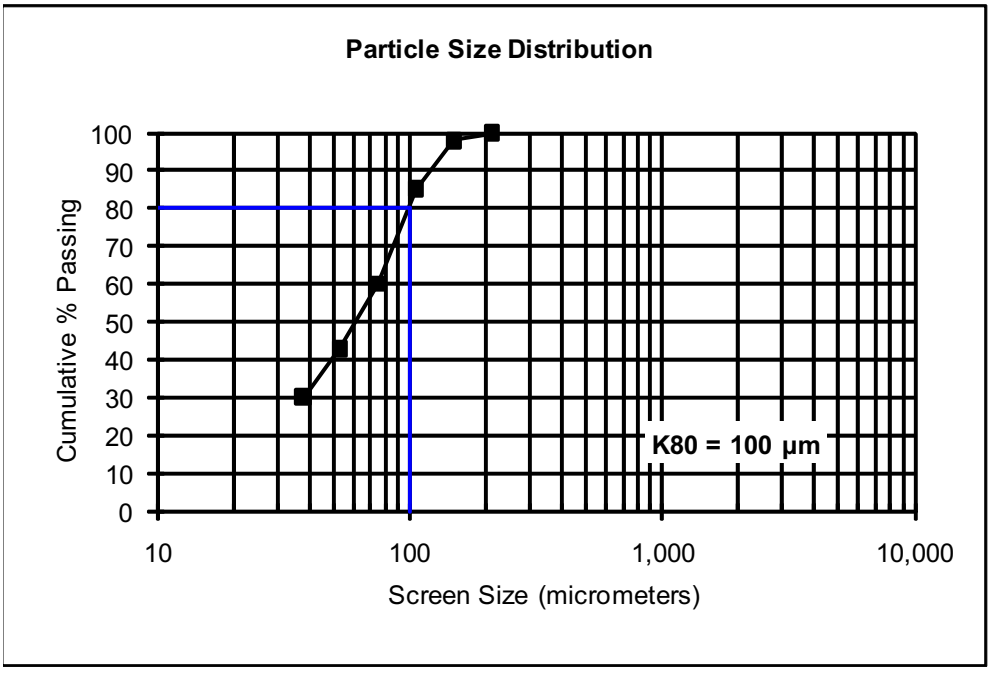
Product	Weight	Assays, %				% Distribution			
	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	5.85	23.9	6.34	65.1	4.65	33.2	22.8	5.3	1.2
Rougher Conc 1-2	39.9	9.76	2.67	83.8	3.78	92.7	65.4	46.6	6.7
Rougher Conc 1-3	62.8	6.51	2.24	87.3	3.93	97.4	86.6	76.4	11.0
Rougher Conc1-4	75.8	5.49	2.05	87.8	4.67	99.0	95.5	92.8	15.8
Rougher Conc 1-5	80.6	5.20	1.98	87.4	5.45	99.7	98.3	98.1	19.6

SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **Comb Prod** Test No.: **F2**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.3	0.2	0.2	99.8
100	150	3.4	1.9	2.1	97.9
150	106	22.3	12.7	14.9	85.1
200	75	43.8	25.0	39.9	60.1
270	53	30.0	17.1	57.0	43.0
400	38	22.7	13.0	70.0	30.0
Pan	-38	52.5	30.0	100.0	0.0
Total	**-**	**175.0**	100.0	**-**	**-**
K80	**100**				



Test No.: F-3 **Project No.**: 12323-002 **Operator**: KS **Date**: June 21/10

Purpose: Incremental rougher flotation of the Ferguson Lake Comp plus magsep of rougher tails

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 30 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill.#4 K_{80} = 86µm

Conditions:

	Reagents, g/t		Time, minutes				
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	5		30			7.8	-180
Bulk Circuit							
Aerate				1		7.0	-50.0
Bulk Rougher 1	5	10		1	1	7.0	-50.0
Bulk Rougher 2	10			1	2		
Bulk Rougher 3	10			1	2		
Bulk Rougher 4	10	5		1	4		
Bulk Rougher 5	20	5		1	6		
Rougher Tail							
Mag Sep					1		
Total	60	20	30	6	16		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Metallurgical Balance

Product	**Weight**		**Assays, %, g/t**			**% Distribution**		
	g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	140.6	7.26	8.75	2.40	35.1	42.8	20.0	8.6
Rougher Conc 2	487.2	25.2	2.90	1.16	36.7	49.1	33.4	31.1
Rougher Conc 3	634.9	32.8	0.28	0.84	36.3	6.2	31.6	40.1
Rougher Conc 4	230.6	11.9	0.15	0.74	34.4	1.2	10.1	13.8
Rougher Conc 5	72.2	3.73	0.11	0.63	30.8	0.3	2.7	3.9
Rougher Tail Mags	252.2	13.0	0.02	0.10	5.06	0.2	1.5	2.2
Rougher Tail N-Mags	119.2	6.2	0.06	0.11	1.95	0.2	0.8	0.4
Head (calc.)	1936.9	100.0	1.48	0.87	29.7	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	**Weight**	**Assays, %, g/t**			**% Distribution**		
	%	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	7.26	8.75	2.40	35.1	42.8	20.0	8.6
Rougher Conc 1-2	32.4	4.21	1.44	36.3	91.9	53.4	39.7
Rougher Conc 1-3	65.2	2.23	1.14	36.3	98.1	84.9	79.7
Rougher Conc 1-4	77.1	1.91	1.08	36.0	99.3	95.0	93.5
Rougher Conc 1-5	80.8	1.83	1.06	35.8	99.6	97.7	97.4
Rougher Tail	19.2	0.03	0.10	4.06	0.4	2.3	2.6

Mineral Balance

Product	**Weight**		**Assays, %**				**% Distribution**			
	g	**%**	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	140.6	7.26	25.4	6.13	63.7	4.82	42.8	26.8	6.4	1.6
Rougher Conc 2	487.2	25.2	8.41	2.32	86.8	2.45	49.1	35.1	30.0	2.9
Rougher Conc 3	634.9	32.8	0.81	1.33	93.6	4.23	6.2	26.2	42.2	6.5
Rougher Conc 4	230.6	11.9	0.43	1.10	89.2	9.30	1.2	7.8	14.6	5.2
Rougher Conc 5	72.2	3.73	0.32	0.89	80.0	18.8	0.3	2.0	4.1	3.3
Rougher Tail Mags	252.2	13.0	0.06	0.14	13.1	86.7	0.2	1.1	2.4	53.0
Rougher Tail N-Mags	119.2	6.15	0.17	0.26	4.75	94.8	0.2	1.0	0.4	27.4
Head (calc.)	1936.9	100.0	4.30	1.66	72.8	21.3	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

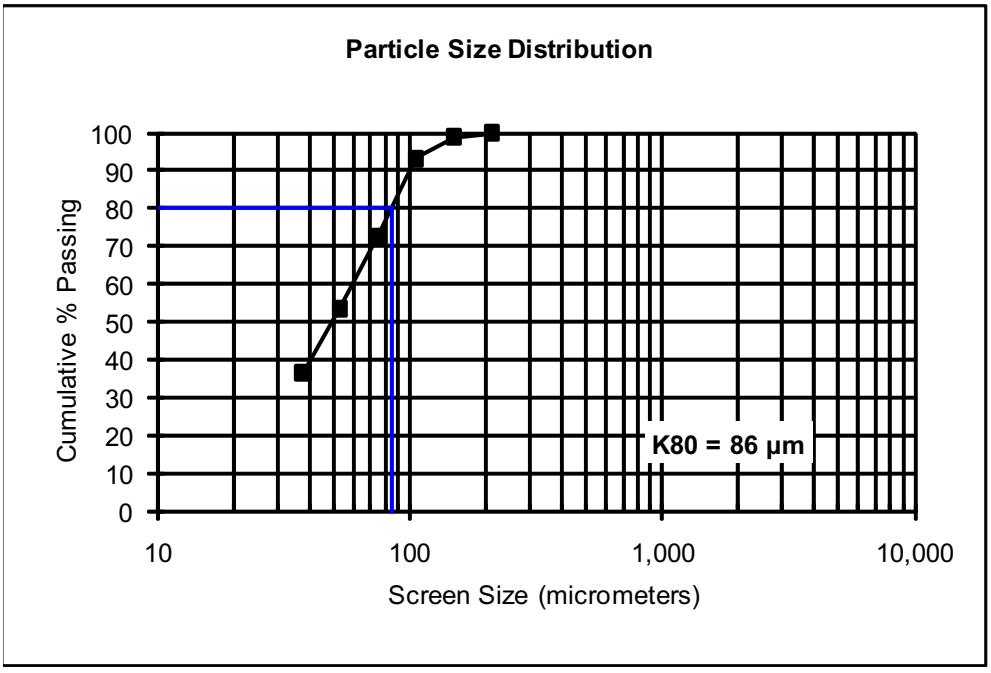
Product	**Weight**	**Assays, %**				**% Distribution**			
	%	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	7.26	25.4	6.13	63.7	4.82	42.8	26.8	6.4	1.6
Rougher Conc 1-2	32.4	12.2	3.17	81.6	2.98	91.9	61.9	36.4	4.5
Rougher Conc 1-3	65.2	6.48	2.25	87.7	3.61	98.1	88.1	78.6	11.1
Rougher Conc1-4	77.1	5.54	2.07	87.9	4.49	99.3	96.0	93.1	16.3
Rougher Conc 1-5	80.8	5.30	2.01	87.5	5.15	99.6	98.0	97.2	19.6
Ro Tail	19.2	0.09	0.18	10.45	89.3	0.4	2.0	2.8	80.4

SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **Comb Prod** Test No.: **F3**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	0.2	0.1	0.1	99.9
100	150	1.8	1.0	1.1	98.9
150	106	9.9	5.7	6.8	93.2
200	75	36.2	20.7	27.5	72.5
270	53	33.3	19.0	46.5	53.5
400	38	29.5	16.9	63.4	36.6
Pan	-38	64.1	36.6	100.0	0.0
Total	**-**	**175.0**	100.0	**-**	**-**
K80	**86**				



Test No.: F-4 **Project No.**: 12323-002 **Operator**: KS **Date**: June 30/10

Purpose: Incremental rougher flotation with gangue depressant of the Ferguson Lake Comp

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg old laboratory mild steel rod mill #4. K_{80} = ~ 117 µm

Conditions:

	Reagents added, g/t			Time, minutes				
Stage	Na_2SiO_4	PAX	MIBC	Grind	Cond.	Froth	pH	Eh
Primary Grind		5		20			7.4	-200
Bulk Circuit								
Aerate					1		7.2	-50.0
Condition	50						7.2	-50.0
Bulk Rougher 1		5	10		1	1		
Bulk Rougher 2	10	10			1	2		
Bulk Rougher 3	10	10			1	2		
Bulk Rougher 4	10	10	5		1	4		
Bulk Rougher 5	10	20	5		1	6		
Total	90	60	20	20	6	15		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Metallurgical Balance

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	203.5	10.2	4.56	1.63	35.8	30.6	18.8	12.6
Rougher Conc 2	495.6	24.7	3.75	1.18	35.9	61.3	33.2	30.7
Rougher Conc 3	359.9	18.0	0.45	1.06	36.0	5.3	21.7	22.4
Rougher Conc 4	420.1	21.0	0.16	0.81	34.9	2.2	19.3	25.3
Rougher Conc 5	120.2	6.00	0.10	0.68	29.7	0.4	4.6	6.2
Rougher Tail	403.3	20.1	0.01	0.10	4.12	0.2	2.3	2.9
Head (calc.)	2002.6	100.0	1.51	0.88	28.9	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	Weight	Assays, %, g/t			% Distribution		
	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	10.2	4.56	1.63	35.8	30.6	18.8	12.6
Rougher Conc 1-2	34.9	3.99	1.31	35.9	91.9	52.1	43.3
Rougher Conc 1-3	52.9	2.78	1.23	35.9	97.2	73.7	65.7
Rougher Conc 1-4	73.9	2.04	1.11	35.6	99.4	93.1	91.0
Rougher Conc 1-5	79.9	1.89	1.08	35.2	99.8	97.7	97.1

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	203.5	10.2	13.2	3.76	78.8	4.25	30.6	22.4	11.3	1.9
Rougher Conc 2	495.6	24.7	10.9	2.43	82.4	4.35	61.3	35.3	28.9	4.6
Rougher Conc 3	359.9	18.0	1.30	1.98	91.8	4.90	5.3	20.9	23.4	3.8
Rougher Conc 4	420.1	21.0	0.46	1.28	90.3	7.95	2.2	15.8	26.8	7.2
Rougher Conc 5	120.2	6.00	0.29	1.06	77.0	21.7	0.4	3.7	6.5	5.6
Rougher Tail	403.3	20.1	0.03	0.16	10.7	89.1	0.2	1.9	3.0	77.0
Head (calc.)	2002.6	100.0	4.39	1.70	70.6	23.3	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

Product	Weight	Assays, %				% Distribution			
	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	10.2	13.2	3.76	78.8	4.25	30.6	22.4	11.3	1.9
Rougher Conc 1-2	34.9	11.6	2.82	81.3	4.32	91.9	57.7	40.2	6.5
Rougher Conc 1-3	52.9	8.07	2.53	84.9	4.52	97.2	78.5	63.6	10.2
Rougher Conc1-4	73.9	5.91	2.18	86.4	5.49	99.4	94.3	90.4	17.4
Rougher Conc 1-5	79.9	5.49	2.09	85.7	6.71	99.8	98.1	97.0	23.0

Test No.: F-5 **Project No.**: 12323-002 **Operator**: KS **Date**: June 30/10

Purpose: Incremental rougher flotation at low pH of the Ferguson Lake Comp

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill.#4 K_{80} ~117µm

Conditions:

	Reagents added, g/t			Time, minutes				
Stage	H_2SO_4	PAX	MIBC	Grind	Cond.	Froth	pH	Eh
Primary Grind		5		20			7.5	-280
Bulk Circuit								
Aerate					1		7.3	-50.0
pH Adjust	250						6.0	-80.0
Bulk Rougher 1		5	10		1	1	6.0	-50.0
Bulk Rougher 2		10			1	2	6.0	-40.0
Bulk Rougher 3		10			1	2	6.0	10.0
Bulk Rougher 4		10	5		1	4	6.0	20.0
Bulk Rougher 5		20	5		1	6	6.0	40.0
Total	250	60	20	20	6	15		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Metallurgical Balance

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	245.2	12.3	4.46	1.56	35.6	36.8	22.1	15.0
Rougher Conc 2	542.1	27.1	2.91	1.00	36.0	53.1	31.3	33.6
Rougher Conc 3	503.4	25.2	0.41	0.93	36.7	6.9	27.0	31.8
Rougher Conc 4	315.1	15.7	0.26	0.99	34.2	2.8	18.0	18.6
Rougher Conc 5	20.2	1.01	0.02	0.07	13.5	0.0	0.1	0.5
Rougher Tail	374.9	18.7	0.03	0.07	0.64	0.4	1.5	0.4
Head (calc.)	2000.9	100.0	1.49	0.87	29.0	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	Weight	Assays, %, g/t			% Distribution		
	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	12.3	4.46	1.56	35.6	36.8	22.1	15.0
Rougher Conc 1-2	39.3	3.39	1.17	35.9	89.9	53.4	48.7
Rougher Conc 1-3	64.5	2.23	1.08	36.2	96.8	80.5	80.5
Rougher Conc 1-4	80.3	1.84	1.06	35.8	99.6	98.5	99.1
Rougher Conc 1-5	81.3	1.82	1.05	35.5	99.6	98.5	99.6

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	245.2	12.25	12.9	3.56	78.7	4.83	36.8	26.2	13.6	2.6
Rougher Conc 2	542.1	27.1	8.43	1.88	85.3	4.35	53.1	30.7	32.6	5.1
Rougher Conc 3	503.4	25.2	1.19	1.58	94.1	3.12	6.9	23.9	33.4	3.4
Rougher Conc 4	315.1	15.7	0.75	1.83	87.7	9.70	2.8	17.3	19.5	6.6
Rougher Conc 5	20.2	1.01	0.07	-0.22	35.7	64.5	0.0	-0.1	0.5	2.8
Rougher Tail	374.9	18.7	0.10	0.17	1.44	98.3	0.4	2.0	0.4	79.6
Head (calc.)	2000.9	100.0	4.31	1.66	70.9	23.1	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

Product	Weight	Assays, %				% Distribution			
	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	12.3	12.9	3.56	78.7	4.83	36.8	26.2	13.6	2.6
Rougher Conc 1-2	39.3	9.83	2.40	83.3	4.50	89.9	56.9	46.2	7.6
Rougher Conc 1-3	64.5	6.46	2.08	87.5	3.96	96.8	80.9	79.6	11.0
Rougher Conc1-4	80.3	5.34	2.03	87.5	5.09	99.6	98.2	99.1	17.6
Rougher Conc 1-5	81.3	5.28	2.00	86.9	5.82	99.6	98.0	99.6	20.4

Test No.: F-6 **Project No.**: 12323-002 **Operator**: KS **Date**: July 5/10

Purpose: Incremental rougher flotation at a coase grind of the Ferguson Lake Comp

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 15 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill #4. K_{80} = 174 µm

Conditions:

	Regents, g/t		Time, minutes				
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	5		15			7.4	-200
Bulk Circuit							
Aerate				1		6.8	-50.0
Bulk Rougher 1	5	10		1	1	6.8	-50.0
Bulk Rougher 2	10			1	2		
Bulk Rougher 3	10			1	2		
Bulk Rougher 4	10	5		1	4		
Bulk Rougher 5	20	5		1	6		
Total	60	20	15	6	15		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Metallurgical Balance

Product	**Weight**		**Assays, %, g/t**			**% Distribution**		
	g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	346	17.4	3.35	1.29	37.2	38.9	26.1	21.7
Rougher Conc 2	896.4	45.0	1.84	0.97	37.1	55.3	50.9	56.0
Rougher Conc 3	168.3	8.46	0.64	1.06	36.4	3.6	10.4	10.3
Rougher Conc 4	107.5	5.40	0.37	1.10	35.1	1.3	6.9	6.4
Rougher Conc 5	58.9	2.96	0.23	0.92	31.8	0.5	3.2	3.2
Rougher Tail	413.3	20.8	0.03	0.10	3.6	0.4	2.4	2.5
Head (calc.)	1990.4	100.0	1.50	0.86	29.8	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	**Weight**	**Assays, %, g/t**			**% Distribution**		
	%	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	17.4	3.35	1.29	37.2	38.9	26.1	21.7
Rougher Conc 1-2	62.4	2.26	1.06	37.1	94.2	77.0	77.7
Rougher Conc 1-3	70.9	2.07	1.06	37.0	97.8	87.5	88.0
Rougher Conc 1-4	76.3	1.95	1.06	36.9	99.1	94.4	94.3
Rougher Conc 1-5	79.2	1.88	1.06	36.7	99.6	97.6	97.5

Mineral Balance

Product	**Weight**		**Assays, %**				**% Distribution**			
	g	**%**	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	346.0	17.4	9.71	2.70	86.6	0.98	38.9	29.0	20.6	0.8
Rougher Conc 2	896.4	45.0	5.33	1.73	91.2	1.72	55.3	48.2	56.2	3.7
Rougher Conc 3	168.3	8.46	1.86	1.97	92.4	3.80	3.6	10.3	10.7	1.5
Rougher Conc 4	107.5	5.40	1.07	2.12	89.5	7.27	1.3	7.1	6.6	1.9
Rougher Conc 5	58.9	2.96	0.67	1.70	81.6	16.0	0.5	3.1	3.3	2.3
Rougher Tail	413.3	20.8	0.09	0.18	9.2	90.5	0.4	2.3	2.6	89.8
Head (calc.)	1990.4	100.0	4.34	1.62	73.1	20.9	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

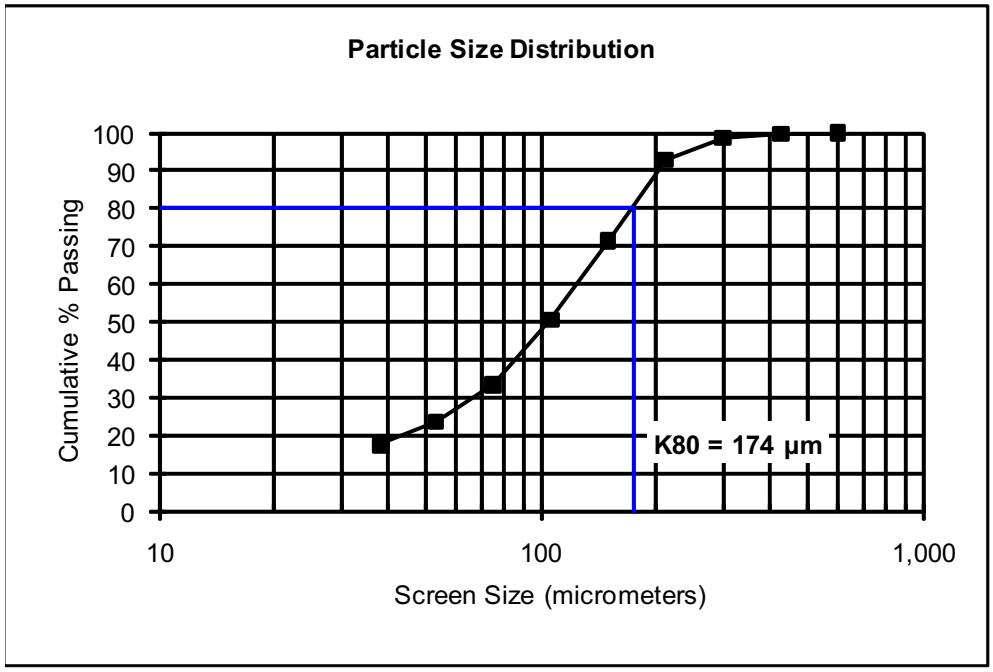
Product	**Weight**	**Assays, %**				**% Distribution**			
	%	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	17.4	9.71	2.70	86.6	0.98	38.9	29.0	20.6	0.8
Rougher Conc 1-2	62.4	6.55	2.00	89.9	1.51	94.2	77.2	76.8	4.5
Rougher Conc 1-3	70.9	5.99	2.00	90.2	1.79	97.8	87.5	87.5	6.1
Rougher Conc1-4	76.3	5.64	2.00	90.2	2.18	99.1	94.6	94.1	7.9
Rougher Conc 1-5	79.2	5.46	1.99	89.9	2.69	99.6	97.7	97.4	10.2

SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **Comb Prod** Test No.: **F6**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
28	600	0.1	0.1	0.1	99.9
35	425	0.2	0.2	0.3	99.7
48	300	1.0	1.0	1.3	98.7
65	212	5.8	5.8	7.1	92.9
100	150	21.6	21.6	28.7	71.3
150	106	20.8	20.8	49.5	50.5
200	75	17.0	17.0	66.5	33.5
270	53	9.9	9.9	76.4	23.6
400	38	6.0	6.0	82.4	17.6
Pan	-38	17.6	17.6	100.0	0.0
Total	**-**	**100.0**	100.0	**-**	**-**
K80	**174**				



Test No.: F-7 **Project No.**: 12323-002 **Operator**: KS **Date**: July 5/10

Purpose: Incremental rougher flotation with HCl as the pH modifier of the Ferguson Lake Comp

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill #4. K_{80} = ~ 117 µm

Conditions:

	Reagents added, g/t			Time, minutes				
Stage	**HCl**	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind		5		20			7.5	-200
Bulk Circuit								
Aerate					1		7.0	-50.0
pH Adjust	750						6.0	-100.0
Bulk Rougher 1		5	10		1	1	6.0	-100.0
Bulk Rougher 2		10			1	2	5.8	-100.0
Bulk Rougher 3		10			1	2	5.5	0.0
Bulk Rougher 4		10	5		1	4	5.2	10.0
Bulk Rougher 5		20	5		1	6	5.5	10.0
Total	750	60	20	20	6	15		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Metallurgical Balance

Product	**Weight**		**Assays, %, g/t**			**% Distribution**		
	g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	11.7	0.58	4.48	2.46	32.7	1.7	1.7	0.6
Rougher Conc 2	633.8	31.7	3.01	1.19	36.4	62.9	43.8	38.9
Rougher Conc 3	835.0	41.7	1.18	0.92	36.9	32.5	44.6	51.9
Rougher Conc 4	144.1	7.20	0.46	0.99	33.2	2.2	8.3	8.1
Rougher Conc 5	14.6	0.73	0.64	0.75	8.76	0.3	0.6	0.2
Rougher Tail	361.3	18.1	0.03	0.05	0.45	0.4	1.0	0.3
Head (calc.)	2000.5	100.0	1.52	0.86	29.7	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	**Weight**	**Assays, %, g/t**			**% Distribution**		
	%	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	0.58	4.48	2.46	32.7	1.7	1.7	0.6
Rougher Conc 1-2	32.3	3.04	1.21	36.3	64.7	45.4	39.5
Rougher Conc 1-3	74.0	1.99	1.05	36.7	97.1	90.0	91.4
Rougher Conc 1-4	81.2	1.85	1.04	36.3	99.3	98.3	99.5
Rougher Conc 1-5	81.9	1.84	1.04	36.1	99.6	99.0	99.7

Mineral Balance

Product	**Weight**		**Assays, %**				**% Distribution**			
	g	**%**	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	11.7	0.58	13.0	6.24	68.7	12.1	1.7	2.2	0.6	0.3
Rougher Conc 2	633.8	31.7	8.72	2.42	85.7	3.20	62.9	47.0	37.4	4.7
Rougher Conc 3	835.0	41.7	3.42	1.57	92.6	2.42	32.5	40.2	53.2	4.7
Rougher Conc 4	144.1	7.20	1.33	1.86	84.5	12.3	2.2	8.2	8.4	4.1
Rougher Conc 5	14.6	0.73	1.86	1.92	19.7	76.5	0.3	0.9	0.2	2.6
Rougher Tail	361.3	18.1	0.09	0.13	0.99	98.8	0.4	1.5	0.2	83.5
Head (calc.)	2000.5	100.0	4.39	1.63	72.6	21.4	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

Product	**Weight**	**Assays, %**				**% Distribution**			
	%	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	0.58	13.0	6.24	68.7	12.1	1.7	2.2	0.6	0.3
Rougher Conc 1-2	32.3	8.80	2.49	85.3	3.36	64.7	49.3	37.9	5.1
Rougher Conc 1-3	74.0	5.77	1.97	89.4	2.83	97.1	89.5	91.2	9.8
Rougher Conc1-4	81.2	5.37	1.96	89.0	3.67	99.3	97.7	99.6	13.9
Rougher Conc 1-5	81.9	5.34	1.96	88.4	4.32	99.6	98.5	99.8	16.5

Test No.: F-8 **Project No.**: 12323-002 **Operator**: KS **Date**: July 5/10

Purpose: Incremental rougher flotation using sea salt water as flotation water of the Ferguson Lake Comp

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill #4 K_{80} ~117 µm

Conditions:

	Reagents, g/t		Time, minutes				
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	5		20			7.8	-200
Bulk Circuit							
Aerate				1		6.8	50.0
Bulk Rougher 1	5			1	1	6.5	40.0
Bulk Rougher 2	10			1	2	6.5	20.0
Bulk Rougher 3	10	5		1	2	6.5	20.0
Bulk Rougher 4	10	5		1	4	6.8	0.0
Bulk Rougher 5	20	5		1	6	7.2	-50.0
Total	60	15	20	6	15		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Tropic Marine Salt used at 0.03kg/l

Metallurgical Balance

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	290.0	14.6	8.00	1.49	36.2	77.0	24.8	17.7
Rougher Conc 2	520.0	26.1	1.05	1.15	37.0	18.1	34.3	32.4
Rougher Conc 3	419.6	21.1	0.23	0.98	36.8	3.2	23.6	26.0
Rougher Conc 4	274.6	13.8	0.12	0.77	35.6	1.1	12.1	16.4
Rougher Conc 5	75.7	3.80	0.12	0.67	32.8	0.3	2.9	4.2
Rougher Tail	413	20.7	0.02	0.10	4.83	0.3	2.4	3.4
Head (calc.)	1992.9	100.0	1.51	0.88	29.8	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	Weight	Assays, %, g/t			% Distribution		
	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	14.6	8.00	1.49	36.2	77.0	24.8	17.7
Rougher Conc 1-2	40.6	3.54	1.27	36.7	95.1	59.0	50.0
Rougher Conc 1-3	61.7	2.41	1.17	36.7	98.3	82.6	76.0
Rougher Conc 1-4	75.5	1.99	1.10	36.5	99.4	94.7	92.5
Rougher Conc 1-5	79.3	1.90	1.08	36.4	99.7	97.6	96.6

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	290.0	14.6	23.2	3.45	70.9	2.45	77.0	30.1	14.1	1.7
Rougher Conc 2	520.0	26.1	3.04	2.23	92.6	2.10	18.1	34.9	33.1	2.6
Rougher Conc 3	419.6	21.1	0.67	1.72	94.7	2.88	3.2	21.7	27.3	2.9
Rougher Conc 4	274.6	13.8	0.35	1.14	92.4	6.14	1.1	9.5	17.4	4.0
Rougher Conc 5	75.7	3.80	0.35	0.94	85.2	13.5	0.3	2.1	4.4	2.5
Rougher Tail	413.0	20.7	0.06	0.14	12.5	87.3	0.3	1.8	3.6	86.3
Head (calc.)	1992.9	100.0	4.38	1.67	73.0	21.0	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

Product	Weight	Assays, %				% Distribution			
	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	14.6	23.2	3.45	70.9	2.45	77.0	30.1	14.1	1.7
Rougher Conc 1-2	40.6	10.3	2.66	84.9	2.23	95.1	64.9	47.2	4.3
Rougher Conc 1-3	61.7	6.98	2.34	88.2	2.45	98.3	86.6	74.6	7.2
Rougher Conc1-4	75.5	5.77	2.12	89.0	3.12	99.4	96.1	92.0	11.2
Rougher Conc 1-5	79.3	5.51	2.07	88.8	3.62	99.7	98.2	96.4	13.7

Test No.: F-9 **Project No.**: 12323-002 **Operator**: KS **Date**: July15 /10

Purpose: Cleaning the rougher concentrate with sea water using F8 rougher conditions

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill #4. K_{80} = ~117µm

Conditions:

	Reagents added, g/t		Time, minutes				
Stage	PAX	MIBC	Grind	Cond.	Froth	pH	Eh
Primary Grind	5		20			7.5	-180
Bulk Circuit							
Aerate				1		6.8	-60.0
Bulk Rougher 1	5	10		1	1	6.8	-60.0
Bulk Rougher 2	10			1	2		
Bulk Rougher 3	10			1	2		
Bulk Rougher 4	10	5		1	4		
Bulk Rougher 5	20	5		1	6		
Rougher Concentrate 1-5							
Bulk Cleaner 1	10			1	1	6.9	-80.0
Bulk Cleaner 2	10			1	2		
Bulk Cleaner 3				1	3		
Bulk Cleaner 4	10	5		1	4		
Total	90	25	20	10	25		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Tropic Marine Salt used at 0.03kg/l

Metallurgical Balance

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Cleaner Conc 1	1081	53.9	2.36	1.13	35.3	90.2	71.1	66.9
Cleaner Conc 2	414.1	20.7	0.59	0.94	35.6	8.6	22.7	25.8
Cleaner Conc 3	73.8	3.68	0.28	0.79	34.4	0.7	3.4	4.4
Cleaner Conc 4	10.9	0.54	0.29	0.75	28.3	0.1	0.5	0.5
Cleaner Tail	23.6	1.18	0.15	0.32	5.54	0.1	0.4	0.2
Rougher Tail	400.5	20.0	0.02	0.08	2.96	0.2	1.9	2.1
Head (calc.)	2003.9	100.0	1.41	0.86	28.5	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Cleaner Conc 1		53.9	2.36	1.13	35.3	90.2	71.1	66.9
Cleaner Conc 1-2		74.6	1.87	1.08	35.4	98.8	93.8	92.7
Cleaner Conc 1-3		78.3	1.79	1.06	35.3	99.6	97.2	97.2
Cleaner Conc 1-4		78.8	1.78	1.06	35.3	99.7	97.7	97.7
Rougher Conc		80.0	1.76	1.05	34.9	99.8	98.1	97.9

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Cleaner Conc 1	1081	53.9	6.84	2.26	84.6	6.27	90.2	73.9	65.5	13.8
Cleaner Conc 2	414.1	20.7	1.71	1.65	90.7	5.96	8.6	20.6	26.9	5.0
Cleaner Conc 3	73.8	3.68	0.81	1.24	88.7	9.25	0.7	2.8	4.7	1.4
Cleaner Conc 4	10.9	0.54	0.84	1.31	72.6	25.3	0.1	0.4	0.6	0.6
Cleaner Tail	23.6	1.18	0.43	0.76	13.5	85.3	0.1	0.5	0.2	4.1
Rougher Tail	400.5	20.0	0.04	0.14	7.6	92.2	0.2	1.7	2.2	75.1
Head (calc.)	2003.9	100.0	4.09	1.65	69.7	24.5	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

Cleaner Conc 1		53.9	6.84	2.26	84.6	6.27	90.2	73.9	65.5	13.8
Cleaner Conc 1-2		74.6	5.42	2.09	86.3	6.18	98.8	94.5	92.3	18.8
Cleaner Conc 1-3		78.3	5.20	2.05	86.4	6.33	99.6	97.3	97.0	20.2
Cleaner Conc 1-4		78.8	5.17	2.05	86.3	6.46	99.7	97.7	97.6	20.8
Rougher Conc		80.0	5.10	2.03	85.3	7.62	99.8	98.3	97.8	24.9

Test No.: F-10 **Project No.**: 12323-002 **Operator**: KS **Date**: July 5/10

Purpose: Incremental rougher flotation using sea salt water as flotation water of the Ferguson Lake Comp
To make a rougher concentrate for magnetic separation upgrading

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill #4 K_{80} ~ 117 µm

Conditions:

	Reagents, g/t		Time, minutes				
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	5		20			7.7	-180
Bulk Circuit							
Aerate				1		7.7	20.0
Bulk Rougher 1	5			1	1	7.6	20.0
Bulk Rougher 2	10			1	2		
Bulk Rougher 3	10	5		1	2		
Bulk Rougher 4	10	5		1	4		
Bulk Rougher 5	20	5		1	6		
Bulk Rougher Conc to Mag Sep'n							
Mag Sep					5		
Rare Earth hand magnet							
Total	60	15	20	6	20		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Tropic Marine Salt used at 0.03kg/l

Metallurgical Balance

Product	Weight		Assays, %, g/t						% Distribution					
	g	%	Cu	Ni	S	Au	Pt	Pd	Cu	Ni	S	Au	Pt	Pd
Ro Conc Mags	1401.1	70.0	0.77	1.00	36.4	0.07	0.17	1.99	36.0	82.1	86.7	37.5	42.2	58.4
Ro Conc Non Mags	141.5	7.1	13.4	1.74	33.7	1.09	2.18	13.2	63.3	14.4	8.1	59.0	54.6	39.2
Rougher Tail	458.2	22.9	0.04	0.13	6.67	0.02	0.04	0.25	0.6	3.5	5.2	3.5	3.2	2.4
Head (calc.)	2000.8	100.0	1.50	0.85	29.4	0.13	0.28	2.38	100.0	100.0	100.0	100.0	100.0	100.0
(direct)			1.48	0.87	29.0	0.02	0.20	2.28						

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Ro Conc Mags	1401.1	70.0	2.2	1.80	92.2	3.80	36.0	78.2	89.7	12.0
Ro Conc Non Mags	141.5	7.1	38.84	4.41	49.1	7.67	63.3	19.3	4.8	2.5
Rougher Tail	458.2	22.9	0.12	0.17	17.3	82.4	0.6	2.5	5.5	85.5
Head (calc.)	2000.8	100.0	4.34	1.61	72.0	22.1	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Test No.: F-11 **Project No.**: 12323-002 **Operator**: KS **Date**: August 16, 2010

Purpose: Repeat of test F8 with longer flotation time of 22 minutes instead of 15 minutes in an effort to increase the Ni and Cu recovery

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill #4 K_{80} ~ 117 µm

Conditions:

	Reagents, g/t		Time, minutes				
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	5		20			7.9	-350
Bulk Circuit							
Aerate				1		7.1	-100.0
Bulk Rougher 1	5			1	1	7.1	-80.0
Bulk Rougher 2	10			1	4		
Bulk Rougher 3	10	5		1	4		
Bulk Rougher 4	10	5		1	5		
Bulk Rougher 5	20	5		1	8		
Total	60	15	20	6	22		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Tropic Marine Salt used at 0.03kg/l

Metallurgical Balance

Product	**Weight**		**Assays, %, g/t**			**% Distribution**		
	g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	410.9	20.4	3.89	1.42	36.3	55.3	31.1	25.5
Rougher Conc 2	962.4	47.8	1.29	1.07	35.7	42.9	54.9	58.7
Rougher Conc 3	168.2	8.35	0.21	1.04	34.9	1.2	9.3	10.0
Rougher Conc 4	38.8	1.93	0.14	0.85	32.0	0.2	1.8	2.1
Rougher Conc 5	36.6	1.82	0.10	0.68	29.3	0.1	1.3	1.8
Rougher Tail	397.5	19.7	0.019	0.08	2.63	0.3	1.6	1.8
Head (calc.)	2014.4	100.0	1.44	0.93	29.0	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	**Weight**	**Assays, %, g/t**			**% Distribution**		
	%	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	20.4	3.89	1.42	36.3	55.3	31.1	25.5
Rougher Conc 1-2	68.2	2.07	1.17	35.9	98.2	86.0	84.2
Rougher Conc 1-3	76.5	1.87	1.16	35.8	99.4	95.3	94.3
Rougher Conc 1-4	78.5	1.82	1.15	35.7	99.6	97.1	96.4
Rougher Conc 1-5	80.3	1.78	1.14	35.5	99.7	98.4	98.2

Mineral Balance

Product	**Weight**		**Assays, %**				**% Distribution**			
	g	**%**	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	410.9	20.4	11.3	3.12	82.4	3.17	55.3	34.3	23.7	2.8
Rougher Conc 2	962.4	47.8	3.74	2.04	88.7	5.49	42.9	52.8	59.7	11.4
Rougher Conc 3	168.2	8.35	0.61	1.95	89.6	7.85	1.2	8.8	10.5	2.9
Rougher Conc 4	38.8	1.93	0.41	1.49	82.5	15.6	0.2	1.5	2.2	1.3
Rougher Conc 5	36.6	1.82	0.28	1.08	75.9	22.7	0.1	1.1	1.9	1.8
Rougher Tail	397.5	19.7	0.06	0.14	6.75	93.1	0.3	1.5	1.9	79.8
Head (calc.)	2014.4	100.0	4.16	1.85	71.0	23.0	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

Product	**Weight**	**Assays, %**				**% Distribution**			
	%	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	20.4	11.3	3.12	82.4	3.17	55.3	34.3	23.7	2.8
Rougher Conc 1-2	68.2	5.99	2.36	86.8	4.80	98.2	87.1	83.4	14.2
Rougher Conc 1-3	76.5	5.41	2.32	87.1	5.13	99.4	95.9	93.9	17.1
Rougher Conc1-4	78.5	5.28	2.30	87.0	5.39	99.6	97.5	96.2	18.4
Rougher Conc 1-5	80.3	5.17	2.27	86.8	5.78	99.7	98.5	98.1	20.2

Test No.: F-12 **Project No.**: 12323-002 **Operator**: KS **Date**: August 19, 2010

Purpose: Repeat of test F11 with increased collector dosage in an effort to increase Ni and Cu recovery (90 g/t instead of 60 g/t)

Procedure: As outlined below.

Feed: 2 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 20 minutes / 2 kg @ 65% solids in a 2 kg laboratory mild steel rod mill #4 K_{80} ~ 117 µm

Conditions:

	Reagents, g/t		Time, minutes				
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**	**pH**	**Eh**
Primary Grind	5		20			7.8	-320
Bulk Circuit							
Aerate				1		6.9	-120.0
Bulk Rougher 1	5			1	1	6.9	-90.0
Bulk Rougher 2	15			1	4		
Bulk Rougher 3	15	5		1	4		
Bulk Rougher 4	20	5		1	5		
Bulk Rougher 5	30	5		1	8		
Total	90	15	20	6	22		

Stage	Bulk Rougher
Flotation Cell	500g-D1
Speed: rpm	1500

Tropic Marine Salt used at 0.03kg/l

Metallurgical Balance

Product	**Weight**		**Assays, %, g/t**			**% Distribution**		
	g	**%**	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	399.7	20.0	4.56	1.31	35.0	62.0	28.9	24.2
Rougher Conc 2	1000.0	50.0	1.07	1.06	35.6	36.4	58.6	61.6
Rougher Conc 3	132.5	6.62	0.25	1.00	33.4	1.1	7.3	7.7
Rougher Conc 4	49.2	2.46	0.12	0.82	31.0	0.2	2.2	2.6
Rougher Conc 5	31.4	1.57	0.08	0.64	27.1	0.1	1.1	1.5
Rougher Tail	387.7	19.4	0.018	0.09	3.51	0.2	1.8	2.4
Head (calc.)	2000.5	100.0	1.47	0.90	28.9	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	**Weight**	**Assays, %, g/t**			**% Distribution**		
	%	**Cu**	**Ni**	**S**	**Cu**	**Ni**	**S**
Rougher Conc 1	20.0	4.56	1.31	35.0	62.0	28.9	24.2
Rougher Conc 1-2	70.0	2.07	1.13	35.4	98.4	87.5	85.9
Rougher Conc 1-3	76.6	1.91	1.12	35.3	99.5	94.8	93.5
Rougher Conc 1-4	79.1	1.85	1.11	35.1	99.7	97.1	96.2
Rougher Conc 1-5	80.6	1.82	1.10	35.0	99.8	98.2	97.6

Mineral Balance

Product	**Weight**		**Assays, %**				**% Distribution**			
	g	**%**	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	399.7	20.0	13.2	2.86	77.4	6.48	62.0	32.1	21.9	5.5
Rougher Conc 2	1000.0	50.0	3.10	2.01	89.1	5.81	36.4	56.5	63.2	12.4
Rougher Conc 3	132.5	6.62	0.72	1.88	85.6	11.8	1.1	7.0	8.0	3.3
Rougher Conc 4	49.2	2.46	0.35	1.43	80.0	18.2	0.2	2.0	2.8	1.9
Rougher Conc 5	31.4	1.57	0.22	1.03	70.2	28.5	0.1	0.9	1.6	1.9
Rougher Tail	387.7	19.4	0.05	0.14	9.07	90.7	0.2	1.5	2.5	75.0
Head (calc.)	2000.5	100.0	4.26	1.78	70.5	23.5	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

Product	**Weight**	**Assays, %**				**% Distribution**			
	%	**Cp**	**Pn**	**Po**	**Gangue**	**Cp**	**Pn**	**Po**	**Gangue**
Rougher Conc 1	20.0	13.2	2.86	77.4	6.48	62.0	32.1	21.9	5.5
Rougher Conc 1-2	70.0	5.99	2.25	85.8	6.00	98.4	88.6	85.1	17.9
Rougher Conc 1-3	76.6	5.53	2.22	85.7	6.50	99.5	95.6	93.2	21.2
Rougher Conc1-4	79.1	5.37	2.20	85.6	6.86	99.7	97.6	95.9	23.1
Rougher Conc 1-5	80.6	5.27	2.17	85.3	7.29	99.8	98.5	97.5	25.0

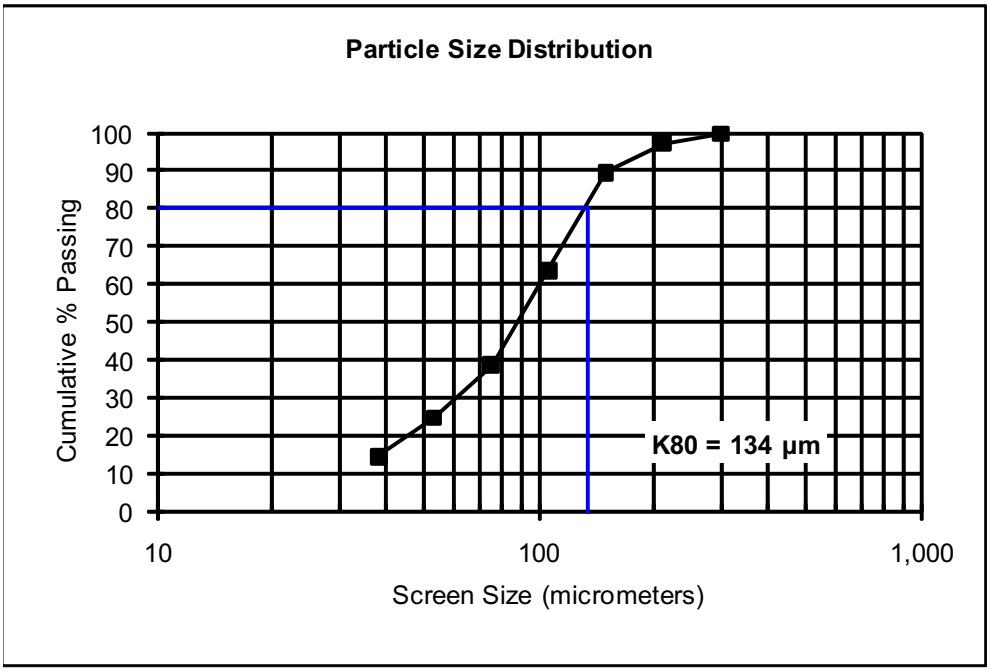
Appendix B – Sizing of Davis Tube Samples

SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **DT1** Test No.: **AS IS**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
48	300	0.3	0.3	0.3	99.7
65	212	2.8	2.5	2.7	97.3
100	150	8.8	7.7	10.4	89.6
150	106	29.6	25.9	36.4	63.6
200	75	28.7	25.2	61.5	38.5
270	53	15.8	13.8	75.4	24.6
400	38	11.6	10.2	85.5	14.5
Pan	-38	16.5	14.5	100.0	0.0
Total	**-**	**114.1**	100.0	**-**	**-**
K80	**134**				

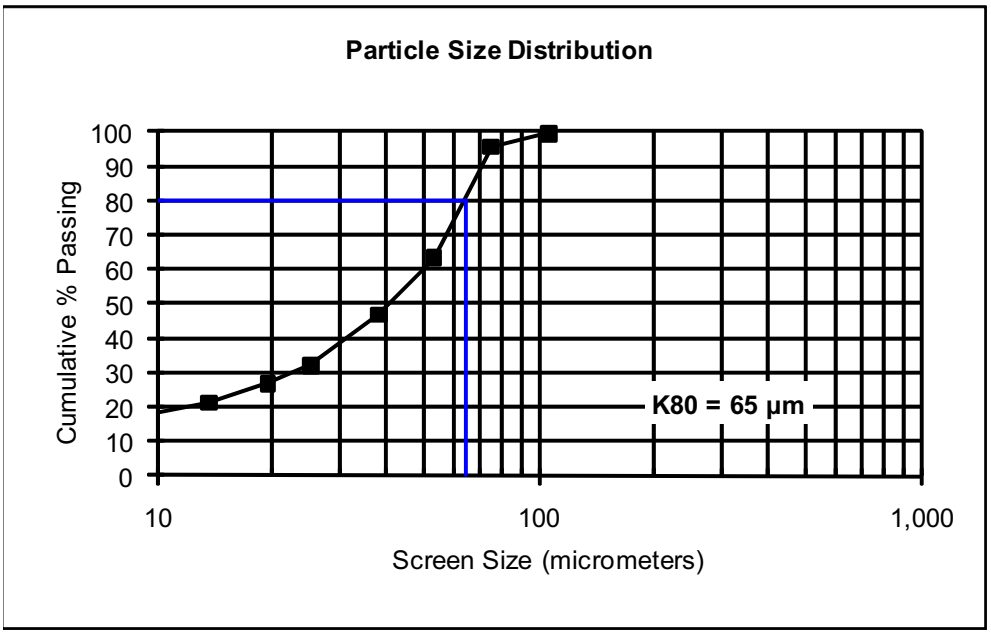


SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **DT2** Test No.: **Minus 200m**

Dry Solids S.G.=		**4.17**	Water Temperature =		**26.00 Cº**
Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
150	106	0.3	0.5	0.5	99.5
200	75	2.4	3.9	4.4	95.6
270	53	20.2	32.4	36.8	63.2
400	38	10.1	16.3	53.1	46.9
	25	9.2	14.7	67.8	32.2
	20	3.3	5.3	73.1	26.9
	14	3.5	5.5	78.7	21.3
	9	2.2	3.5	82.2	17.8
	7	1.3	2.1	84.2	15.8
	-7	9.8	15.8	100.0	0.0
Total	**-**	**62.2**	100.0	**-**	**-**
K80	**65**				

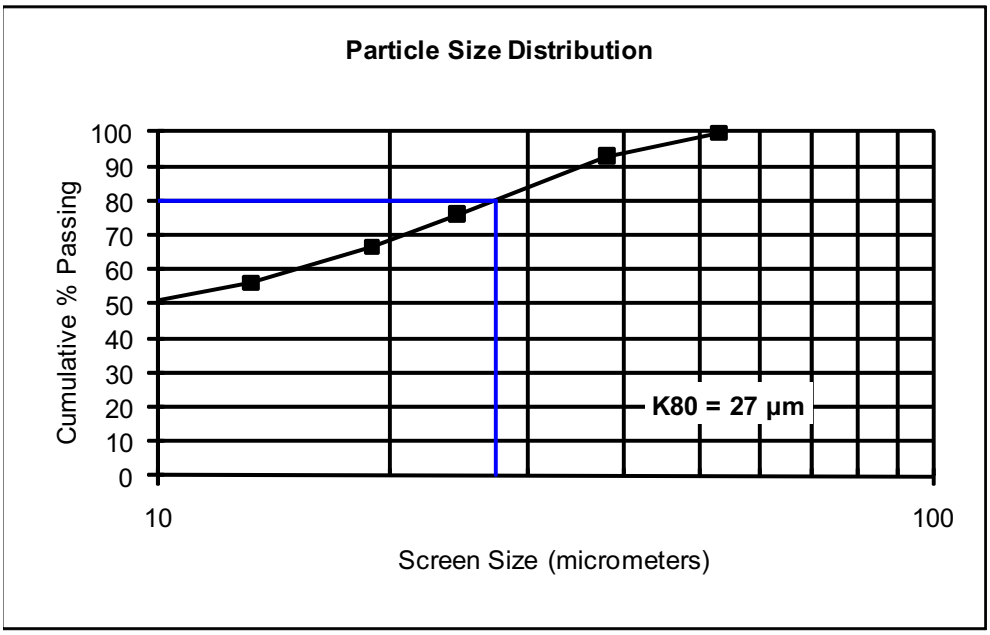


SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **DT3** Test No.: **Minus 325m**

Dry Solids S.G.=		**4.40**	Water Temperature =		**26.00 C°**
Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
270	53	0.1	0.2	0.2	99.8
400	38	4.2	6.9	7.1	92.9
	24	10.3	16.9	24.0	76.0
	19	5.7	9.4	33.4	66.6
	13	6.4	10.6	44.0	56.0
	9	4.3	7.0	51.0	49.0
	7	3.9	6.3	57.3	42.7
	-7	25.9	42.7	100.0	0.0
Total	**-**	**60.8**	100.0	**-**	**-**
K80	**27**				

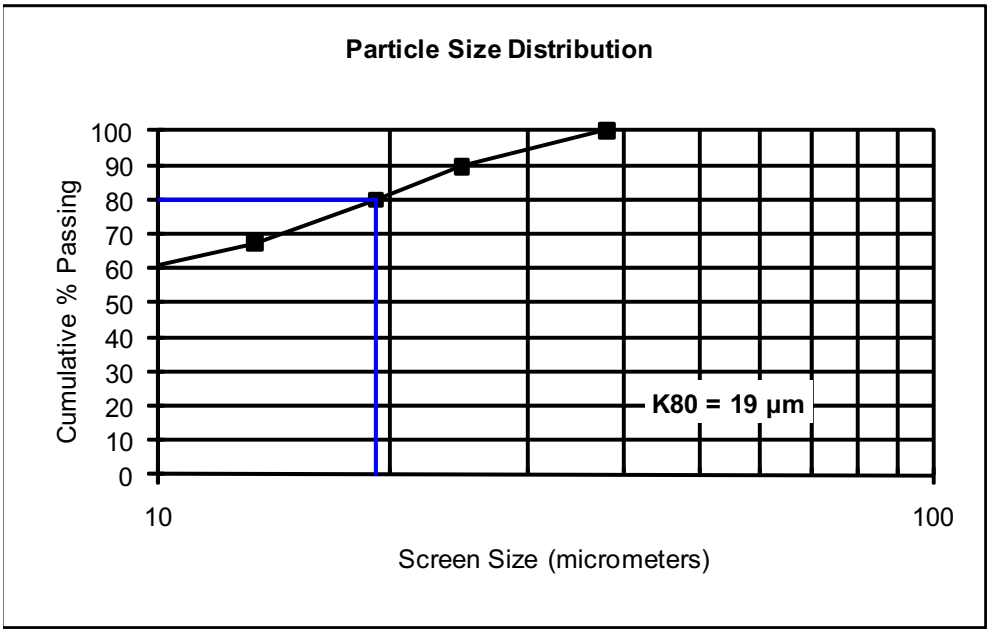


SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **DT4** Test No.: **Minus 500m**

Dry Solids S.G.=		**4.31**	Water Temperature =		**26.00 Cº**
Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
400	38	0.0	0.0	0.0	100.0
	25	6.1	10.3	10.3	89.7
	19	5.8	9.7	20.1	79.9
	13	7.4	12.5	32.6	67.4
	9	4.9	8.3	40.9	59.1
	7	4.5	7.7	48.5	51.5
	-7	30.5	51.5	100.0	0.0
Total	**-**	**59.2**	100.0	**-**	**-**
K80	**19**				



Appendix C – Bulk Flotation Test Mass Balances

Test No.: B-1 **Project No.**: 12323-002 **Operator**: KS **Date**: August 30, 2010

Purpose: Repeat of test F11 using a 10kg test charge in prepration for bulk flotation

Procedure: As outlined below.

Feed: 10 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 34 minutes / 10 kg @ 65% solids in a 10 kg laboratory rod mill K_{80} =116 µm

Conditions:

Stage	Reagents added, g/t		Time, minutes			pH	Eh
	PAX	MIBC	Grind	Cond.	Froth		
Primary Grind	5		34			8.1	-180
Bulk Circuit							
Aerate				1		8.0	0.0
Bulk Rougher 1	5			1	2	8.0	0.0
Bulk Rougher 2	10			1	8		
Bulk Rougher 3	10	5		1	8		
Bulk Rougher 4	10	5		1	10		
Bulk Rougher 5	20	5		1	16		
Total	60	15			44		

Stage	Bulk Rougher
Flotation Cell	28 L Cell
Speed: rpm	1500

Tropic Marine Salt used at 0.03kg/l

Metallurgical Balance

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	336	3.4	8.37	1.51	34.5	20.3	6.1	4.1
Rougher Conc 2	3,760	38.1	2.84	1.18	35.6	77.0	52.9	47.2
Rougher Conc 3	3,057	31.0	0.10	0.87	35.0	2.2	31.7	37.7
Rougher Conc 4	445	4.5	0.10	0.80	33.2	0.3	4.2	5.2
Rougher Conc 5	196	1.99	0.06	0.77	31.2	0.1	1.8	2.2
Rougher Tail	2,062	20.9	0.01	0.13	5.05	0.1	3.2	3.7
Head (calc.)	9,856	100.0	1.41	0.85	28.8	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Combined Products

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	336	3.4	8.37	1.51	34.5	20.3	6.1	4.1
Rougher Conc 1-2	4,096	41.6	3.29	1.21	35.5	97.3	59.0	51.3
Rougher Conc 1-3	7,153	72.6	1.93	1.06	35.3	99.4	90.7	89.0
Rougher Conc 1-4	7,598	77.1	1.82	1.05	35.2	99.8	95.0	94.2
Rougher Conc 1-5	7,794	79.1	1.78	1.04	35.1	99.9	96.8	96.3

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	336	3.4	24.3	3.57	65.3	6.83	20.3	7.5	3.2	1.0
Rougher Conc 2	3,760	38.1	8.23	2.41	84.0	5.35	77.0	56.7	45.4	8.6
Rougher Conc 3	3,057	31.0	0.28	1.45	90.6	7.68	2.2	27.7	39.8	10.1
Rougher Conc 4	445	4.5	0.29	1.30	86.0	12.4	0.3	3.6	5.5	2.4
Rougher Conc 5	196	1.99	0.17	1.28	80.8	17.7	0.1	1.6	2.3	1.5
Rougher Tail	2,062	20.9	0.03	0.22	13.1	86.7	0.1	2.9	3.9	76.5
Head (calc.)	9,856	100.0	4.08	1.62	70.6	23.7	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

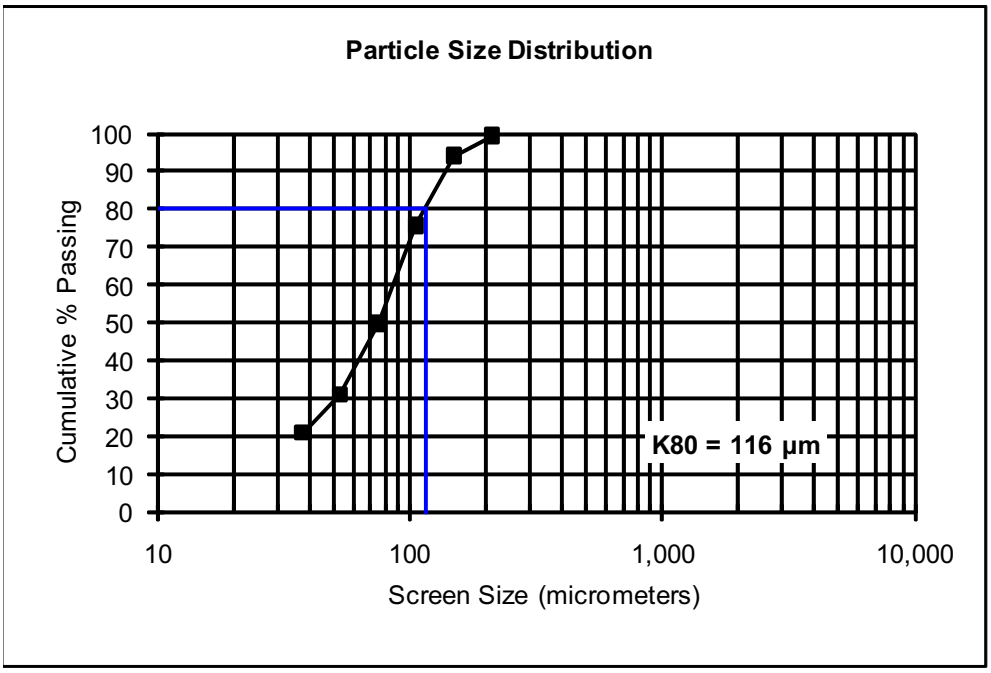
Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	336	3.4	24.3	3.57	65.3	6.83	20.3	7.5	3.2	1.0
Rougher Conc 1-2	4,096	41.6	9.55	2.51	82.5	5.47	97.3	64.2	48.5	9.6
Rougher Conc 1-3	7,153	72.6	5.59	2.06	85.9	6.42	99.4	91.9	88.3	19.6
Rougher Conc1-4	7,598	77.1	5.28	2.01	85.9	6.77	99.8	95.6	93.8	22.0
Rougher Conc 1-5	7,794	79.1	5.15	1.99	85.8	7.04	99.9	97.1	96.1	23.5

SGS Minerals Services
Size Distribution Analysis

Project No.
12323-002

Sample: **Ro TI** Test No.: **B1**

Size		Weight	% Retained		% Passing
Mesh	µm	grams	Individual	Cumulative	Cumulative
65	212	1.0	0.7	0.7	99.3
100	150	7.8	5.2	5.9	94.1
150	106	27.6	18.4	24.3	75.7
200	75	39.2	26.1	50.4	49.6
270	53	28.0	18.7	69.1	30.9
400	38	15.1	10.1	79.1	20.9
Pan	-38	31.3	20.9	100.0	0.0
Total	**-**	**150.0**	100.0	**-**	**-**
K80	**116**				



Test No.: B-2 **Project No.**: 12323-002 **Operator**: KS **Date**: September 9, 2010

Purpose: Repeat Test of B1 with a longer flotation time to recover additional Ni units

Procedure: As outlined below.

Feed: 10 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 34 minutes / 10 kg @ 65% solids in a 10 kg laboratory rod mill K_{80} =~116 µm

Conditions:

Stage	Reagents added, g/t		Time, minutes			pH	Eh
	PAX	MIBC	Grind	Cond.	Froth		
Primary Grind	5		34				
Bulk Circuit							
Aerate				1			
Bulk Rougher 1	5			1	3		
Bulk Rougher 2	10			1	9		
Bulk Rougher 3	10	5		1	9		
Bulk Rougher 4	10	5		1	12		
Bulk Rougher 5	20	5		1	18		
Total	60	15			51		

Stage	Bulk Rougher
Flotation Cell	28 L Cell
Speed: rpm	1500

Tropic Marine Salt used at 0.03kg/l

Metallurgical Balance

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	525	5.3	16.3	1.27	35.4	58.3	7.6	6.3
Rougher Conc 2	3,335	33.4	1.71	1.31	36.3	38.9	49.9	40.9
Rougher Conc 3	3,378	33.9	0.07	0.85	36.5	1.6	32.8	41.6
Rougher Conc 4	676	6.8	0.12	0.78	33.6	0.6	6.0	7.7
Rougher Conc 5	212	2.1	0.09	0.74	29.8	0.1	1.8	2.1
Rougher Tail	1,850	18.5	0.04	0.09	2.27	0.5	1.9	1.4
Head (calc.)	9,976	100.0	1.47	0.88	29.7	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Estimated value

Combined Products

Product	Weight		Assays, %, g/t			% Distribution		
	g	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc 1	525	5.3	16.3	1.27	35.4	58.3	7.6	6.3
Rougher Conc 1-2	3,860	38.7	3.69	1.30	36.2	97.2	57.5	47.2
Rougher Conc 1-3	7,237	72.6	2.00	1.09	36.3	98.8	90.3	88.8
Rougher Conc 1-4	7,914	79.3	1.84	1.07	36.1	99.4	96.3	96.4
Rougher Conc 1-5	8,126	81.5	1.80	1.06	35.9	99.5	98.1	98.6

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	525	5.3	47.2	3.09	47.0	2.71	58.3	9.7	3.4	0.7
Rougher Conc 2	3,335	33.4	4.96	2.73	88.6	3.72	38.9	54.5	40.7	5.8
Rougher Conc 3	3,378	33.9	0.20	1.35	94.7	3.75	1.6	27.2	44.1	6.0
Rougher Conc 4	676	6.8	0.35	1.23	87.0	11.4	0.6	5.0	8.1	3.6
Rougher Conc 5	212	2.13	0.27	1.23	77.1	21.4	0.1	1.6	2.3	2.1
Rougher Tail	1,850	18.5	0.12	0.19	5.7	94.0	0.5	2.1	1.5	81.8
Head (calc.)	9,976	100.0	4.26	1.68	72.7	21.3	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Combined Products

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc 1	525	5.3	47.2	3.09	47.0	2.71	58.3	9.7	3.4	0.7
Rougher Conc 1-2	3,860	38.7	10.71	2.78	82.9	3.58	97.2	64.2	44.1	6.5
Rougher Conc 1-3	7,237	72.6	5.81	2.11	88.4	3.66	98.8	91.4	88.2	12.5
Rougher Conc1-4	7,914	79.3	5.34	2.04	88.3	4.32	99.4	96.4	96.3	16.1
Rougher Conc 1-5	8,126	81.5	5.21	2.01	88.0	4.77	99.5	97.9	98.5	18.2

Test No.: B-3 **Project No.**: 12323-002 **Operator**: KS & Adam **Date**: September, 2010

Purpose: Repeat Test of B1 with a longer flotation time to recover additional Ni units

Procedure: As outlined below.

Feed: 10 kg of minus 10 mesh Ferguson Lake Comp sample

Grind: 34 minutes / 10 kg @ 65% solids in a 10 kg laboratory rod mill K_{80} =~116 µm

Conditions:

	Reagents added, g/t		Time, minutes		
Stage	**PAX**	**MIBC**	**Grind**	**Cond.**	**Froth**
Primary Grind	5		34		
Bulk Circuit					
Aerate				1	
Bulk Rougher 1	5			1	3
Bulk Rougher 2	10			1	9
Bulk Rougher 3	10	5		1	9
Bulk Rougher 4	10	5		1	12
Bulk Rougher 5	20	5		1	18
Total	60	15			51

Stage	Bulk Rougher
Flotation Cell	28 L Cell
Speed: rpm	1500

Sea water (from mine site) used for grinding
Process water used for make up water during flotation
Cycle B & C - sent for Assay
Cycle P - sent for solid liquid separation
Conc from each cycle combined filtered and sampled at end of test
Tail from each cycle combined and sampled at end of test

Metallurgical Balance

Product	Weight		Assays, %, g/t			% Distribution		
	kg	%	Cu	Ni	S	Cu	Ni	S
Rougher Conc	184.7	84.0	1.70	1.03	35.5	99.6	97.8	99.1
Rougher Tail	35.2	16.0	0.04	0.12	1.71	0.4	2.2	0.9
Head (calc.)	219.9	100.0	1.43	0.88	30.1	100.0	100.0	100.0
(direct)			1.48	0.87	29.0			

Mineral Balance

Product	Weight		Assays, %				% Distribution			
	g	%	Cp	Pn	Po	Gangue	Cp	Pn	Po	Gangue
Rougher Conc	184.7	84.0	4.9	1.95	87.2	5.93	99.6	97.2	99.1	24.60
Rougher Tail	35.2	16.0	0.12	0.30	4.1	95.5	0.45	2.81	0.90	75.4
Head (calc.)	219.9	100.0	4.16	1.68	73.9	20.3	100.0	100.0	100.0	100.0
(direct)			4.29	1.68	70.9	23.1				

Adam - Cycle A, B, D to Z
Kevin - Cycle C -assay (not included in Bulk Conc to Hydro)
Cycle B - assay on hold (not included in Bulk Conc to Hydro)
Sea water used for grinding sent for assay

An Investigation into

PRIMARY ORE LEACHING OF FERGUSON LAKE FLOTATION CONCENTRATE

prepared for

STARFIELD RESOURCES

Project 12323-003 – Final Report
May 13, 2011

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Executive Summary

A proof of concept roadmap study is underway investigating the Starfield hydrometallurgical flowsheet, as shown in Figure 1. The roadmap consists of various phases. The objective of this phase of the program was to produce Ferguson Lake Primary Ore Leach PLS as feed to primary oxidation and hydrolysis.

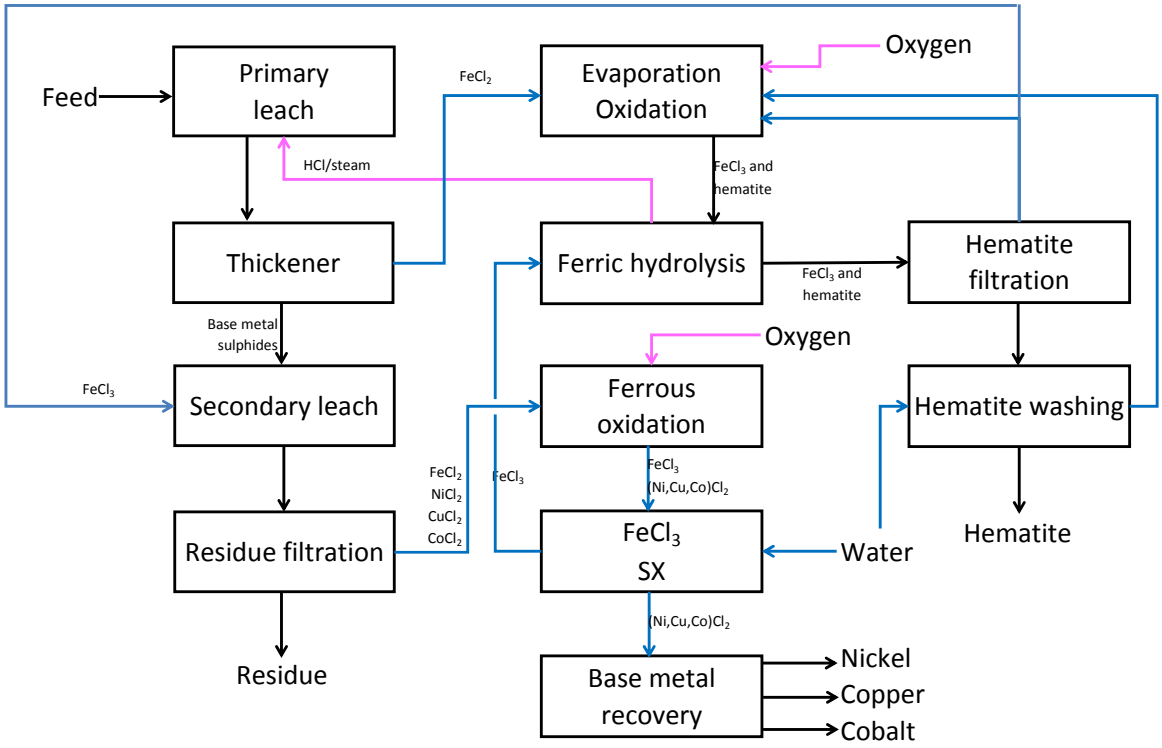

Figure 1 Preliminary Conceptual Overall Flowsheet

Primary ore leaching was successfully run in a mini pilot plant configuration, producing the necessary liquor to move forward in oxidation and hydrolysis testing with real Ferguson Lake ore leached liquor.

Acid consumption was optimized in a series of bench tests and confirmed in the pilot plant run. Acid dosage around 600kg of hydrochloric acid per tonne of solids was sufficient to obtain 73% iron extraction and 63% sulphide extractions while minimizing base metal extraction (6% Ni and 2% Co). Acid dosage in excess of 600 kg/t could lead to base metals leaching.

The scrubbing media was also optimized. Hydrogen sulphide was continuously scrubbed during the pilot plant run using 2 counter-current columns filled with a solution of 100 g/L of sodium hydroxide. This

system provided for reliable S balancing, while at the same time solving a challenging health and safety issue.

Profile sampling showed that iron and sulphide extraction was virtually complete after 1 hour but some base metals began to be leached after that time.

A PGM survey was done with all the streams in this process and it showed that PGM remained in the leach residue.

Liquid-solid separation testing was carried out on the flotation concentrate (feed to the primary ore leach circuit) and a sample from the leach residue. The main results were:

- Flotation concentrate

  Static settling-thickening tests were performed using the selected flocculation reagent. The optimized sample settled well when the feed was diluted (auto-diluted) from the "as received" solids density of 31.5% wt/wt to 22% wt/wt paired with the addition of 9 g/t of medium-charge anionic flocculant CIBA Magnafloc 155; these conditions yielded the following thickener sizing data: 0.015 m^2/t/d thickener underflow unit area (TUFUA), 0.003 m^2/t/d thickener hydraulic unit area (THUA) and 1027 m^3/m^2/day initial settling rate (ISR); the solids density of the resulting underflow was 79% wt/wt and the supernatant was clear after 60 minutes of elapsed time.

  A vane sensor was used to determine the peak shear stress of the sample at the highest naturally settled solids density (post shearing). The test was performed at 86.9% wt/wt solids and resulted in a peak shear stress of 448 Pa. The shear stress value would generally be considered at the high end of commercial pumping range, therefore additional attention should be considered for this product during the pump design/sizing.

  The thickened underflow sample was subjected to laboratory vacuum filtration testing whereby an example of the predicted throughput achievable by a belt filter ranged from 8141 kg/m^2/h through 280 kg/m^2/h, at 10% to 6% wt/wt residual moisture, respectively, at a cake thickness of 10 mm. The disc or drum filter example of predicted throughput (under balanced filter operating conditions) varied from 6564 kg/m^2/h through 3282 kg/m^2/h for cake thickness ranging from 10 through 20 mm, respectively corresponding to higher residual cake moisture content values (under balanced filter operating conditions) ranging from 10.0% through 9.1% wt/wt, respectively.

- Leach residue results

  Static settling-thickening tests were performed using the selected flocculation reagent. The optimized sample settled well when the feed was tested at the "as received" solids density of 3% wt/wt paired with the addition of 48 g/t of medium-charge anionic flocculant CIBA Magnafloc 155; these conditions yielded the following thickener sizing data: 0.07 m^2/t/d thickener underflow unit area (TUFUA), 0.03 m^2/t/d thickener hydraulic unit area (THUA) and 960 m^3/m^2/day initial settling rate (ISR); the solids density of the resulting underflow was 27% wt/wt and the supernatant was slightly cloudy after 60 minutes of elapsed time. In another test the underflow density was 39% but at a higher thickener unit.

  A vane sensor was used for shear stress determination. The Leach Residue sample (pH = <1) was tested as a flocculated sample with Ciba Magnafloc 155 at a dosage of 48 g/t. The vane sensor was used to determine the peak shear stress of the sample at the highest naturally settled solids

density. The test was performed at 43.4% wt/wt solids and resulted in a peak shear stress of 17 Pa. This value generally represents slurry that would be suitable for pumping.

The thickened underflow sample was subjected to laboratory vacuum filtration testing at a feed density of ~31% wt/wt solids; this diluted "thickened" solids density was required in order to be able to load the vacuum filter. The feed pulp dilution was required because the initially prepared thickener underflow (at target solids density~45-48% wt.) could not be handled. Chemically induced rheopexy was suspected, yet not proven as the underlying cause for this flow behaviour.

Introduction

Starfield Resources is developing a novel circuit for the recovery of nickel, copper, cobalt, iron oxide, sulphuric acid and precious metals from the Ferguson Lake deposit in Nunavut, Canada. In this deposit, pentlandite is too finely disseminated in pyrrhotite for the production of a smelter-grade nickel concentrate at an acceptable nickel recovery, so established technology is not applicable and a novel chloride-based circuit is under development. The enabling aspect of this circuit is the hydrolysis of ferric chloride to hematite, releasing gaseous hydrochloric acid that is recycled. Conventional steps are used for separating and recovering the valuable metals from the chloride solution generated. A proposed flowsheet is included in Figure 1.

SGS Minerals Services in Lakefield, Ontario, Canada was contracted to conduct a roadmap study on Starfield hydromet process. A multi phase roadmap was developed that describes the various phases of the test program. It consisted of:

1. Phase 1 – bench top study of oxidation and hydrolysis steps using synthetic chemicals
2. Phase 2 – flotation of sulphide ore
3. Phase 3 – mini pilot plant – primary ore leaching
4. Phase 4 – bench top study of oxidation and hydrolysis steps using liquor produced in primary ore leach pilot plant
5. Phase 5 – mini pilot plant – secondary ore leaching + oxidation and hydrolysis
6. Phase 6 – bench scale downstream unit processes (iron polish, copper, nickel and cobalt recovery)
7. Phase 7 – select mini pilot plants of downstream unit processes

The objective of the current program was to produce leach liquor (PLS) from real Ferguson Lake ore by running a continuous mini pilot plant of the primary ore leach process. Feed to this circuit consisted of flotation concentrate as produced and reported under SGS Project # 12323-002. The work covered in this report was designed to produce:

• PLS for confirmatory oxidation and hydrolysis testing;

• Primary ore leach residue for:

- Liquid-solid separation studies (included in this report);
- Secondary ore leaching (this work is being carried out under SGS project # 12323-005).

Throughout the program, results were discussed with Mr. Michael Moran of Starfield Resources and Dr. Mike Dry, consultant to Starfield. This document describes the testwork carried out under SGS Project Number 12323-003.

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Testwork Summary

The objective of this work was to produce real PLS (leach solution from actual Ferguson Lake ore) to demonstrate the viability of oxidation and hydrolysis processes using actual PLS. In addition, primary leach residue will be used for testing the Secondary Ore Leaching process.

1. Pre Pilot Bench Test Program

1.1. Test Conditions

Ferguson Lake ore or flotation concentrate was subjected to batch hydrochloric acid leaching in which iron in the form of pyrrhotite is leached producing ferrous chloride and hydrogen sulphide (see the equation below). A series of bench tests were carried out in order to optimize hydrochloric acid consumption.

$$Fe_7S_8 + 14HCl_{(ia)} = 7FeCl_{2(ia)} + 7H_2S_{(g)} + S \quad \textbf{[1]}$$

Since hydrogen sulphide is used as an energy source its accountability becomes relevant for the whole process; hence the scrubbing of hydrogen sulphide was also optimized. Different scrubbing reagents (copper chloride, copper sulphate and sodium hydroxide) were evaluated and sodium hydroxide was found to work best. This was an experimental necessity – in commercial operation the hydrogen sulphide would not be scrubbed into sodium hydroxide, it would be burned to sulphur dioxide and that would be converted into concentrated sulphuric acid.

Each test was carried out in a 3L glass stirred-tank reactor at 105°C. Hydrochloric acid was added continuously for a period of approximately one hour. Nitrogen gas was added to the reactor vapour phase to displace any oxygen/air which could cause oxidation. A vapour condenser was connected to the reactor and a line from the top of the condenser was directed to the scrubbing system. The scrubbing system consisted of 3 in-series 4L vessels (see Figure 2). The exhausted-gas line was connected to a vacuum pump to "pull" the gas throughout the scrubbing system. The 3rd vessel was always filled with a solution of 100 g/L of sodium hydroxide while the first two were tested with different solutions. Based on the equations below hydrogen sulphide reacted with the scrubbing media producing either a solid or aqueous sulphur compound which was analyzed for sulphur in order to calculate how much hydrogen sulphide was produced.

$$2H_2S_{(g)} + 2CuCl_{2(ia)} = Cu_2S + S + 4HCl_{(ia)} \quad \textbf{[2]}$$

$$2H_2S_{(g)} + 2CuSO_{4(ia)} = Cu_2S + S + 2H_2SO_4 \quad \textbf{[3]}$$

$$H_2S_{(g)} + NaOH_{(ia)} = NaSH_{(ia)} + H_2O \quad \textbf{[4]}$$

Figure 2 - Scrubbing system for the bench tests

Table 1 shows the different parameters tested in each test. From L1 to L5 Ferguson Lake ore was used as feed while L6 used a flotation concentrate instead of ore. In L1, L2 and L3 different scrubbing reagents were tested and from that point on sodium hydroxide was used in the rest of the tests. Also, a continuous scrubbing test was run to simulate the scrubbing of an off gas coming from a continuous leaching.

Table 1 - Bench tests conditions

Test ID	Feed	HCl dose kg/t	Scrubbing media
L1	Ferguson Lake ore	558	$CuCl_2$
L2	Ferguson Lake ore	415	$CuSO_4$
L3	Ferguson Lake ore	650	NaOH
L4	Ferguson Lake ore	600	NaOH
L5	Ferguson Lake ore	700	NaOH
L6	Float Concentrate	650	NaOH

1.2. Test Results

A summary of the bench test results is shown in Table 2. Tests details are shown in Appendix A. Each test was run for around 3 to 4 hours, depending how fast the acid was added. Kinetic samples (solids and solution) were taken as soon as all the acid was added and one hour after that. Each test was stopped 2 hours after the acid addition was completed. A synthetic solution simulating recycling liquor coming from Primary Ore Leaching was made up for every test, keeping the same composition throughout.

Table 2 - Bench tests results

Test ID	HCl dose	Extraction - %wt (Cu-tie method)							Free Acid
	kg/t	wieght loss	Fe	Ni	Co	Mg	S	S=	g/L
L1	558	54.7	77	3	1	12	54	61	
L2	415	45.6	62	4	0	27	29	44	0
L3	649	69.2	87	0	0	25	59	77	1
L4	599	67.5	83	2	0	23	66	68	1
L5	699	71.9	92	14	0	29	55	85	8
L6	650	78.2	89	11	20	54	74	83	1

Test L1 was run based on a modelling done by Dr. Mike Dry (consultant to Starfield Resources). Scrubbers 1 and 2 were filled up with a solution of 100 g/L of Cu (as copper chloride), scrubber 3 was filled up with a solution of 100 g/L of sodium hydroxide. A sample from the Ferguson Lake ore was mixed up with a synthetic solution made up for this test. There was no visible sign that a reaction was taking place, no change in pH or ORP was observed. Temperature then was increased to 100°C and was kept to 100°C during acid addition. Acid was added in a 1.8 hour period and the dosage used was 558 kg HCl/t ore. The temperature was subsequently increased to 105°C. Once the temperature reached 105°C it was observed that black-like solids were formed in scrubber 1 suggesting that hydrogen sulphide was being formed in the reactor. The test was run for 2 hours after the acid addition was finished, Subsequently, heating was stopped, the reactor contents were filtered and a sample from the PLS and well washed solids were analyzed. A sample from each scrubber was also subjected to S analysis. This test achieved a 54.7% weight loss with 77% Fe extraction, 61% sulphide extraction and relatively low base metals extractions. Some foaming was observed inside the reactor when the temperature reached 105°C. With respect to the scrubbing, some froth-like solids were observed in scrubbers 1 and 2 making it difficult to operate in a continuous-run process. Although the amount of copper chloride in scrubber 1 was theoretically enough to “scrub” all the hydrogen sulphide produced in the leach some sulphur was detected in the third scrubber indicating that the first two scrubbers were not working efficiently.

Test L2 was planned for an acid dosage of 650 kg/t and used a solution of copper sulphate in scrubbers 1 and 2, while keeping scrubber 3 filled with a solution of sodium hydroxide. In this test the Ferguson Lake ore sample and the synthetic solution were heated up to 105°C before acid addition. This time the reaction was instant, as soon as acid was added to the reactor, solids appeared in scrubber 1. The foaming layer started to increase in scrubber 1 and it was necessary to take scrubber 1 off line and connect the off-gas line straight to scrubber 2. Around 10 minutes later the foaming layer started to rise in scrubber 2 and it was taken off-line, too. At this point the acid addition was stopped and the off-gas line

was connected to scrubber 3. The actual acid dosage was 415kg/t, producing a weight loss of 45.6% with 62% Fe extraction and 44% sulphide extraction.

In test L3 a solution of 100 g/L of sodium hydroxide was used in the 3 scrubbers. Since the reaction between hydrogen sulphide and sodium hydroxide does not produce any solids it was a much easier operation. Assay results from scrubbers 2 and 3 solutions showed that all the hydrogen sulphide was captured in scrubber 1. Once again acid dosage was planned for 650 kg/t and it was added once the slurry was at 105°C. Foaming inside the reactor was higher than in the previous tests; it appears that foaming is related to the acid addition rate, higher acid addition rates leading to increased foaming. Scrubber 1 solution turned darker as the test was carried out. Weight loss for this test was 69.2% with 87% Fe extraction and 77% sulphide extraction.

Since there was no problem in the hydrogen sulphide scrubbing using sodium hydroxide in all three scrubbers, this strategy was repeated in test L4 and L5. Acid dosage was set to 600 kg/t and 700 kg/t for test L4 and L5, respectively. Foaming was present in both tests, with the same trend that in test L3, faster the acid addition bigger the foaming. Regarding chemical performance, test 4 had a weight loss of 67.5% with 83% Fe extraction and 68% sulphide extraction, while test L5 had 71.9% weight loss and 92% and 85% extraction for Fe and sulphide, respectively.

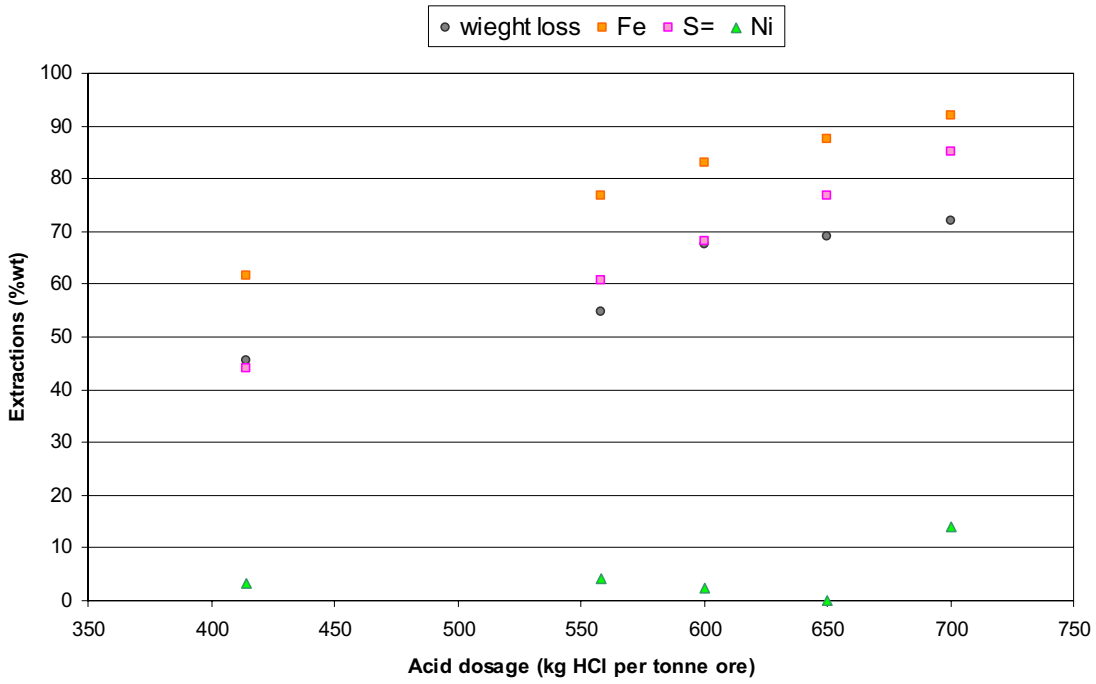

Figure 3 - Bench tests L1-L5 extractions

A Ferguson Lake ore sample was subjected to a flotation process (see also SGS Progress Report 12323-002) and the froth concentrate was used as feed for test L6. As shown in Figure 3 extractions are directly proportional to acid addition, and when acid dosage exceeded 650 kg/t some nickel was leached, which is not a positive result since base metals are supposed to be leached in the Secondary Ore Leaching stage and not at this point. Based on this finding, acid dosage for Test L6 was set at 650 kg/t as it was in test L3. Foaming continued to develop inside the reactor. Fe extractions behaved as expected, 89% vs. 87% in test L3, but the weight loss was higher in test L6, 78.2% vs. 69.2%, which is logical since L6 used flotation concentrate instead of ore. Base metals extractions were relatively high, which should be followed carefully in the Pilot Plant.

After all bench tests were completed, the acid addition was optimized and scrubber media was chosen, a continuous scrubbing test was carried out. Based on test L6 sulphide extraction and the target solids flowrate for the pilot plant, a hydrogen sulphide production was estimated as well as the stoichiometric (plus 50% excess) sodium hydroxide flowrate required to scrub the H_2S. The same scrubbing system used in the bench tests was used, but this time fresh sodium hydroxide solution was pumped into the scrubber 3. Scrubber 3 solution was then pumped into scrubber 2 and so on. Spent caustic solution from scrubber 1 was discharged into a pail. A gas cylinder was used as hydrogen sulphide source and it was connected to scrubber 1. A flask filled with a copper chloride solution was attached to scrubber 3 gas outlet to check if any hydrogen sulphide was leaving the scrubbing system; this flask was connected to the main vacuum system to “pull” the gas throughout the scrubbing system. The test ran for about 30 minutes. It was very difficult to control the pressure balance in the system, due to the pressure in the gas cylinder and the vacuum at the end of the circuit. Based on that the vacuum was close and the gas was “pushed” throughout the circuit by the gas cylinder. At the end of the test there was no sign of hydrogen sulphide in the copper chloride solution which indicated all the hydrogen sulphide was scrubbed in the scrubbing system.

2. Pilot plant

Prior to this pilot plant Ferguson Lake ore was subjected to a flotation process to produce concentrate which was used as feed for the Primary Ore Leaching (POL). This work was carried out under SGS project # 12323-002. Only flotation concentrate was used in the PL leach campaigns. Summarised data is included in the sections below and detailed data sheets are included in Appendix B.

2.1. Circuit Set-up

2.1.1. Leaching Circuit

A schematic of the Leaching flowsheet is shown in Figure 4 and a photograph of circuit is included in Figure 5. Flotation concentrate was mixed with Primary Ore Leaching PLS which was recycled after the

filter. A synthetic solution was used to simulate the Primary Ore Leaching PLS at the start-up. The feed system consisted of 2 tanks; one tank was feeding to the circuit while the next feed batch was being made up in the other feeding tank. The slurry was pumped from the bottom of the feed tank to a pre-heater tank located atop the circuit, in which the pulp was heated up to 105°C, then overflowed by gravity to the first reactor (R1).

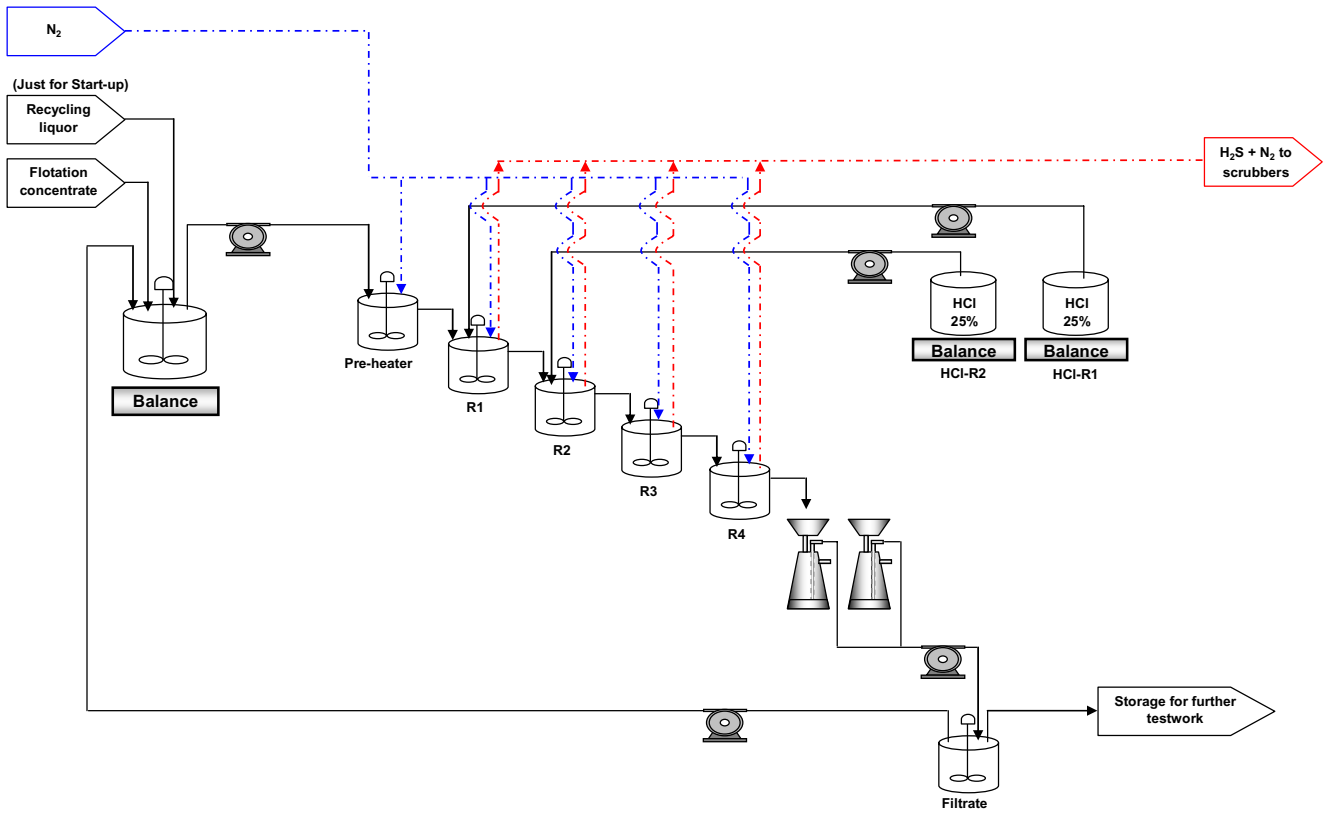

Figure 4 - Leaching flowsheet

The leaching circuit consisted of 4 stirred-tank reactors; R1, R2, R3, R4 with a total 2 hours residence time. Each reactor as well as the pre-heater was made from glass with an active volume of 1.5 L and an internal diameter of 97 mm, equipped with 4 baffles (13mm wide and 1/3 of the reactor height, located at the bottom of the tank) as well as agitator with a 55mm diameter pitch-blade type impeller (the impeller and the shaft were made of TEFLON. A vapour condenser was connected at the top of each reactor and a line from the top of the condensers was directed to the scrubbing system. Temperature in each reactor was controlled by an electrical heating mantle. Nitrogen was added to each reactor, the pre-heater and the feed tanks in order to avoid any oxidation. Due to the foaming that was observed in the bench tests, hydrochloric acid addition was split in two and added to R1 and R2 tanks.

The slurry was transferred through the Leaching circuit by gravity to a Buchner funnel type filter. Filtrate was recycled back to the circuit and/or stored for further testwork. The filter cakes were washed and stored wet for further testwork.

Figure 5 - Leaching circuit (photograph)

2.1.2. Scrubbing Circuit

A schematic of the scrubbing operation is shown in Figure 6 and a photograph of the circuit is included in Figure 7. A mix of hydrogen sulphide, nitrogen and probably some oxygen and water was pulled out of the leaching circuit and directed to the scrubbing circuit by a large (B/T) peristaltic pump. The scrubbing circuit consisted of 2 counter-current columns filled with a solution of 100 g/L sodium hydroxide. Each column was made of clear PVC with an active volume of 14 L and a diameter of 102 mm. Gas coming from the leaching circuit was pumped to the bottom of Column 2, it went all the way to the top and pushed to the bottom of Column 1. Clean gas left Column 1 through the top of the column and was directed to the main plant scrubber. A back-up system was designed in case of failure of the B/T pump (this is not shown in the flow diagram) and it consisted of using the main plant vacuum system to “pull” off gas from the top of Column 1 while by-passing the B/T pump (see the valves around the B/T pump in Figure 6). A solution

of 100 g/L sodium hydroxide was pumped counter-current to the gas, i.e. it was pumped to Column 1, then from Column 1 to Column 2 and finally to a drum for disposal.

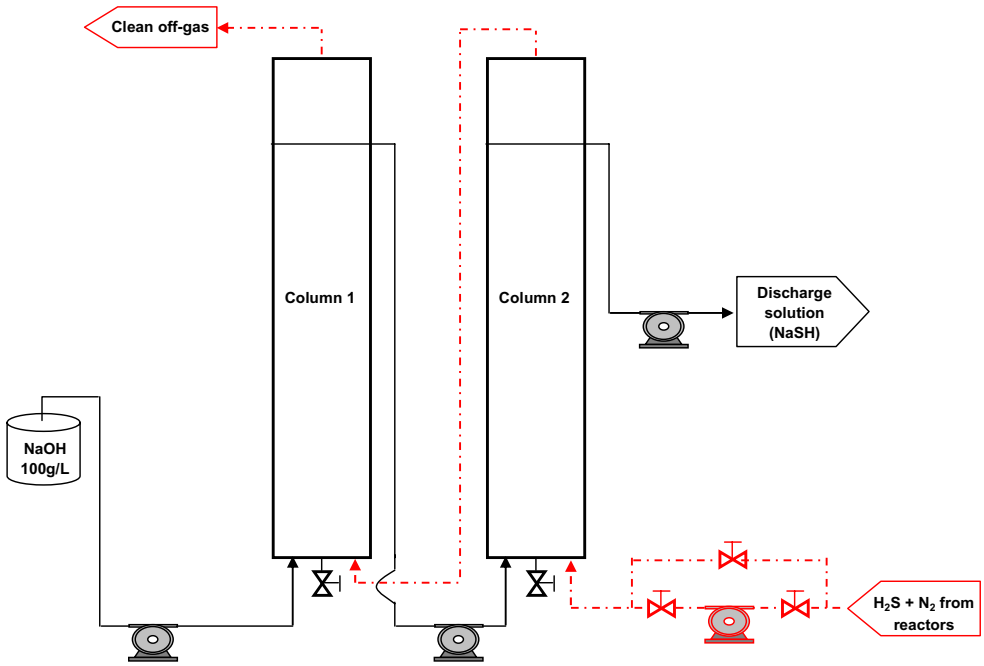

Figure 6 - Scrubbing flowsheet

Figure 7 - Scrubbing circuit (photograph)

2.2. Feed Streams

The circuit was run in two campaigns, PP1 and PP2. Each campaign was run for 1 week and there were 2 weeks of down time between the campaigns to allow detailed data analysis.

2.2.1. Flotation Concentrate

Flotation concentrate was divided into 11 charges. Each charge was used to make up a feed batch; a sample from each charge was submitted to the lab. Flotation concentrate solids compositions are shown in Table 3 and Table 4.

Table 3 - PP1 Concentrate Charges Assays (%wt)

Charges	Fe	Ni	Cu	Co	Mg	S	S=
Charge 1	55.9	1.01	1.72	0.10	0.070	33.6	33.3
Charge 2	56.2	1.02	1.68	0.10	0.088	33.2	33.2
Charge 3	56.0	1.02	1.70	0.10	0.083	33.4	33.4
Charge 4	56.7	1.04	1.74	0.10	0.080	33.9	33.4
Charge 5	56.0	1.03	1.74	0.10	0.075	34.3	33.0
Average	56.2	1.02	1.72	0.10	0.079	33.7	33.3
Min	55.9	1.01	1.68	0.10	0.070	33.2	33.0
Max	56.7	1.04	1.74	0.10	0.088	34.3	33.4
Rel.STD	1%	1%	2%	0%	9%	1%	1%

Table 4 - PP2 Concentrate Charges Assays (%wt)

Charges	Fe	Ni	Cu	Co	Mg	S	S=
Charge 6	56.4	1.02	1.67	0.10	0.070	34.6	33.7
Charge 7	56.6	1.04	1.64	0.10	0.070	34.8	34.5
Charge 8	58.0	1.06	1.72	0.11	0.079	36.6	34.4
Charge 9	56.6	1.03	1.72	0.10	0.078	36.0	33.2
Charge 10	54.7	1.03	1.81	0.10	0.081	36.2	32.3
Charge 11	55.4	1.01	1.67	0.10	0.073	35.9	33.8
Average	56.3	1.03	1.71	0.10	0.075	35.7	33.7
Min	54.7	1.01	1.64	0.10	0.070	34.6	32.3
Max	58.0	1.06	1.81	0.11	0.081	36.6	34.5
Rel.STD	2%	2%	4%	4%	6%	2%	2%

2.2.2. Slurry

Besides a sample from each concentrate charge, the feed pulp was characterised by 12 hour composite samples taken just before the pre-heater tank. The feed liquor and solids compositions are displayed in Table 5 and Table 6 for PP1 and Table 7 and Table 8 for PP2, respectively.

Table 5 - PP1 Feed Solution Assays (mg/L)

Date	Fe	Fe(II)	Ni	Cu	Co	Mg
13-Oct AM	198000	191870	102	102	<5	1820
13-Oct PM	188700	187300	121	243	<5	1370
14-Oct AM	212000	210570	116	229	6	1322
14-Oct PM	212300	208000	153	246	8	1198
15-Oct AM	223800	212600	68	177	<5	899
15-Oct PM	229635	203800	117	208	7	845
16-Oct AM	227200	213100	179	254	9	802
Average	213091	203891	122	208	6	1179
Min	188700	187300	68	102	<5	802
Max	229635	213100	179	254	9	1820
Rel.STD	7%	5%	29%	26%	25%	31%

Table 6 - PP1 Feed Solids Assays (%wt)

Date	Fe	Ni	Cu	Co	Mg	S	S=
13-Oct AM	55.4	0.97	1.73	0.10	0.075	33.6	31.8
13-Oct PM	54.8	1.04	1.67	0.11	0.056	36.4	33.7
14-Oct AM	57.8	1.02	1.65	0.10	0.074	35.6	31.9
14-Oct PM	56.6	1.02	1.66	0.10	0.067	34.2	30.6
15-Oct AM	55.8	1.06	1.69	0.10	0.072	35.1	30.7
15-Oct PM	58.4	1.07	1.72	0.11	0.014	35.2	34.3
16-Oct AM	56.4	1.06	1.83	0.11	0.063	35.0	34.9
Average	56.5	1.03	1.71	0.10	0.060	35.0	32.6
Min	54.8	0.97	1.65	0.10	0.014	33.6	30.6
Max	58.4	1.07	1.83	0.11	0.075	36.4	34.9
Rel.STD	2%	3%	4%	5%	36%	3%	5%

Table 7 - PP2 Feed Solution Assays (mg/L)

Date	Fe	Fe(II)	Ni	Cu	Co	Mg
01-Nov PM	239600	217800	310	89	8	494
02-Nov AM	234100	234100	445	132	12	435
02-Nov PM	239900	217400	656	281	21	474
03-Nov AM	236500	213700	694	332	22	481
03-Nov PM	236200	215700	597	449	23	397
04-Nov AM	230900	214300	472	363	15	354
04-Nov PM	232400	198800	406	344	12	314
05-Nov AM	214400	199600	374	258	10	270
05-Nov PM	218100	203260	436	382	14	295
Average	231344	212740	488	292	15	390
Min	214400	198800	310	89	8	270
Max	239900	234100	694	449	23	494
Rel.STD	4%	5%	27%	40%	36%	22%

Table 8 - PP2 Feed Solids Assays (%wt)

Date	Fe	Ni	Cu	Co	Mg	S	S=
01-Nov PM	57.0	1.07	1.81	0.11	0.065	35.8	32.5
02-Nov AM	57.9	1.04	1.67	0.10	0.064	36.5	33.4
02-Nov PM	57.8	1.06	1.75	0.10	0.053	37.5	34.5
03-Nov AM	56.4	1.11	1.80	0.11	0.067	37.3	34.1
03-Nov PM	56.6	1.07	1.85	0.11	0.063	37.5	35.2
04-Nov AM	56.1	1.02	1.70	0.10	0.070	37.1	34.8
04-Nov PM	56.7	1.01	1.71	0.10	0.069	36.0	32.8
05-Nov AM	55.8	1.07	2.02	0.11	0.085	36.9	33.5
05-Nov PM	56.2	1.02	1.76	0.10	0.072	36.9	32.4
Average	56.7	1.05	1.79	0.10	0.068	36.8	33.7
Min	55.8	1.01	1.67	0.10	0.053	35.8	32.4
Max	57.9	1.11	2.02	0.11	0.085	37.5	35.2
Rel.STD	1%	3%	6%	5%	13%	2%	3%

2.3. Start-up Set points

Start-up operating parameters are displayed in Table 9.

Table 9 - Start-up Parameters

Parameters	Feed mL/min	Temperature °C	N_2 mL/min	HCl(25%) g/min	NaOH mL/min
Feed Tank	20	-	300	-	-
Pre-heater	-	105	100	-	-
R1	-	105	100	13	-
R2	-	105	100	13	-
R3	-	105	100	-	-
R4	-	105	100	-	-
Scrubber	-	-	-	-	38

2.4. Circuit Operation

The circuit started up on October 12th at 11:00 and shut down on October 16th at 02:00 for two weeks, and then started up again on November 1st at 09:00 and shut down on November 6th at 00:30, for a total of 198.5 operating hours. The circuit ran well mechanically. The main operational issue was mixing. Due to the high specific gravity of the solids good mixing was not always achieved which caused some variability in the feed pulp density. There was some plugging in the overflow line from R1 to R2 but it happened just once. The feed line got plugged a few times probably because of the small feed flowrate and high slurry SG. Filtrate lines also got plugged once; temperature in the line dropped causing some crystallization and plugging but it did not happen again after the lines were insulated. On the scrubbing circuit the hose in B/T pump broke twice, each time the back-up system was aligned and the hose was replaced for a new one. During both PP1 and PP2 campaigns samples from the filtrate and filter cakes were taken for liquid/solid separation testing.

2.4.1. Flowrates and Densities

Table 10 and Table 11 presents 12 hourly average flow rates and pulp densities (in g/L and %solids) for the relevant streams used in the mass balance and efficiency calculations. All flow rates and pulp densities (in g/L) are operator measured values. Feed % solids were calculated at the time each feed batch was made up. Wet cakes (leach residue) were collected in 12 hour periods and not as a continuous stream.

Table 10 - PP1 Average flowrates and densities

	Feed				Acid addition		Filtrate	
	Availability	Slurry	Solids Density		R1	R2	Flowrate	Density
Date	%	mL/min	g/mL	%wt	g/min (25%HCl)		mL/min	g/mL
13-Oct AM	100%	20	1.673	28.0%	11	11	48	1.390
13-Oct PM	100%	20	1.474	28.0%	12	12	30	1.422
14-Oct AM	100%	20	1.665	28.0%	17	0	29	1.426
14-Oct PM	100%	21	1.712	28.0%	26	0	40	1.425
15-Oct AM	96%	21	1.757	28.0%	22	0	30	1.438
15-Oct PM	94%	21	1.726	28.0%	21	0	33	1.441
16-Oct AM	100%	21	1.757	28.0%	26	0	32	1.432

Table 11 - PP2 Average flowrates and densities

	Feed				Acid addition		Filtrate	
	Availability	Slurry	Solids Density		R1	R2	Flowrate	Density
Date	%	mL/min	g/mL	%wt	g/min (25%HCl)		mL/min	g/mL
01-Nov PM	94%	20	1.707	28.0%	23	0	37	1.433
02-Nov AM	100%	20	1.736	27.9%	23	0	43	1.448
02-Nov PM	97%	21	1.443	27.4%	23	0	28	1.407
03-Nov AM	97%	21	1.725	27.5%	24	0	39	1.410
03-Nov PM	92%	23	1.410	28.0%	23	0	41	1.385
04-Nov AM	100%	22	1.694	28.0%	29	0	40	1.387
04-Nov PM	100%	20	1.721	28.0%	29	0	44	1.384
05-Nov AM	100%	20	1.648	27.8%	28	0	42	1.367
05-Nov PM	97%	20	1.712	27.7%	27	0	37	1.382

2.4.2. Temperatures

Temperature control was constant throughout both campaigns, as is shown in Table 12 and Table 13. Temperature in R1 started decreasing without any apparent reason, the heating mantle was ON all the time but temperature did not go back to 105°C. In the other tanks average temperatures were within 1 degree Celsius difference from target.

Table 12 - PP1 Average temperatures

	Temperature (°C)				
Date	PH	R1	R2	R3	R4
13-Oct AM	104.6	104.7	104.8	104.9	104.5
13-Oct PM	104.8	104.1	105.5	105.3	105.5
14-Oct AM	104.4	104.1	105.2	105.5	105.7
14-Oct PM	105.3	102.7	105.0	105.3	106.0
15-Oct AM	104.5	103.2	105.0	105.0	105.7
15-Oct PM	104.8	102.0	104.9	105.2	105.3
16-Oct AM	104.9	104.1	105.2	105.3	105.5

Table 13 - PP2 Average temperatures

	Temperature (°C)				
Date	PH	R1	R2	R3	R4
01-Nov PM	104.2	103.8	105.1	105.4	104.9
02-Nov AM	104.7	103.6	105.3	102.8	104.8
02-Nov PM	106.8	104.3	105.0	105.4	105.0
03-Nov AM	106.6	101.4	105.0	105.2	105.0
03-Nov PM	106.9	102.9	105.0	105.4	104.5
04-Nov AM	105.4	100.0	104.9	105.3	104.9
04-Nov PM	105.8	100.1	104.8	104.3	104.6
05-Nov AM	106.4	101.1	104.4	105.1	104.5
05-Nov PM	106.4	101.1	105.3	104.8	104.6

2.5. Chemical Performance

2.5.1. Acid Addition

Based on the foaming observations made in the bench test program, the, acid addition was split in the first 2 reactors, R1 and R2, to decrease the foaming effect. However, during the first week (PP1) it appeared that foaming was not excessive and the acid was from thereon only added into the first reactor. Acid addition was kept that way in the second week (PP2).

Also, based on the results from the bench program acid addition was set at 650 kg of hydrochloric acid per tonne. Due to some high free acid values in the filtrate the acid addition was then controlled to give a free acid value of less than 5 g/L at the end of the circuit.

The average acid addition was 570 kg/t and 609 kg/t for PP1 and PP2, respectively. Daily values are shown in Table 14 and Table 15. Due to crystallization in the filtrate and its high iron concentration (see Table 19 for PP2 filtrate composition) it was decided to "dilute" the filtrate. This was done by adding water to the acid. Table 15 shows average acid strength in each period.

Table 14 - PP1 Average acid consumption

	Ore	HCl @ 25%wt (g/min)			Dosage
Date	g/min	R1	R2	Total	kg/t ore
13-Oct AM	10.0	11	11	22	540
13-Oct PM	8.0	12	12	24	741
14-Oct AM	10.0	17	0	17	421
14-Oct PM	10.5	26	0	26	612
15-Oct AM	9.9	22	0	22	563
15-Oct PM	10.8	21	0	21	485
16-Oct AM	10.4	26	0	26	627
AVERAGE	9.9	19	3	22	570
MIN	8.0	11	0	17	421
MAX	10.8	26	12	26	741
RELSTD	9%	33%	171%	14%	18%

Table 15 - PP2 Average acid consumption

	Ore	HCl (g/min)				Dosage
Date	g/min	R1	R2	Total	%wt	kg/t ore
01-Nov PM	10.6	23	0	23	25.6%	549
02-Nov AM	9.3	23	0	23	25.6%	625
02-Nov PM	7.2	23	0	23	23.7%	764
03-Nov AM	10.4	24	0	24	23.5%	537
03-Nov PM	9.4	23	0	23	23.5%	578
04-Nov AM	11.1	29	0	29	21.9%	569
04-Nov PM	10.6	29	0	29	21.7%	595
05-Nov AM	9.5	28	0	28	22.7%	679
05-Nov PM	11.0	27	0	27	23.6%	583
AVERAGE	9.9	25	0	25	23.5%	609
MIN	7.2	23	0	23	21.7%	537
MAX	11.1	29	0	29	25.6%	764
RELSTD	12%	11%	0%	11%	6%	12%

2.5.2. Metal Extractions

Overall leach extractions in the leaching circuit were calculated for both campaigns are listed in Table 16.These extractions are derived on a mass balance basis for the entire campaign and calculated as:

$$Extraction = \left(1 - \frac{Me\ Units\ in\ Residue}{Me\ Units\ in\ PLS + Me\ Units\ in\ Residue}\right) \times 100\% \quad \textbf{[5]}$$

Table 16 - Metal extractions for PP1 and PP2

Campaign	weight loss	Fe	Ni	Cu	Co	Mg	S
PP1	70%	73%	6%	0%	2%	*50%*	63%
PP2	76%	83%	12%	1%	5%	38%	69%

Weight loss and Fe extraction data were in line with expectations derived from the bench tests. Higher weight loss and Fe extraction in PP2 could be related to a higher acid addition in that campaign. Copper extraction was almost nil in both campaigns (1% in PP2). Cobalt and Nickel extractions were relatively low. This is a positive result meaning that most of the base metals remained in the residue and can be extracted in the Secondary Ore Leaching as it is planned in the process flowsheet. Sulphur extraction was high and it only reflects the sulphur converted to hydrogen sulphide, without taking into account the sulphide converted to elemental sulphur.

Extractions were also calculated for the 12-hour compositing periods by a copper tie method as follows:

$$Cu-tieExtraction = 1 - \frac{Me_{res}}{Me_{feed}} \times \frac{Cu_{feed}}{Cu_{res}} \quad \textbf{[6]}$$

Cu-tie based extractions are plotted in Figure 8 and Figure 9 for PP1 and PP2, respectively. The average extraction determined by this method was 6.7% for Ni and 3.4% for Co during PP1, and 12.9% for Ni and 2.3% for Co during PP2.

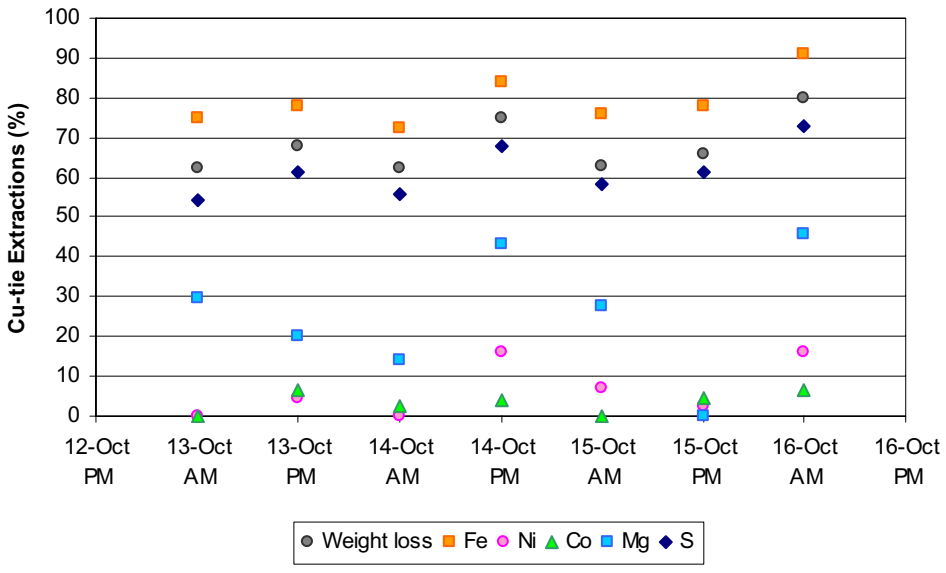

Figure 8 - PP1 Cu-tie extractions

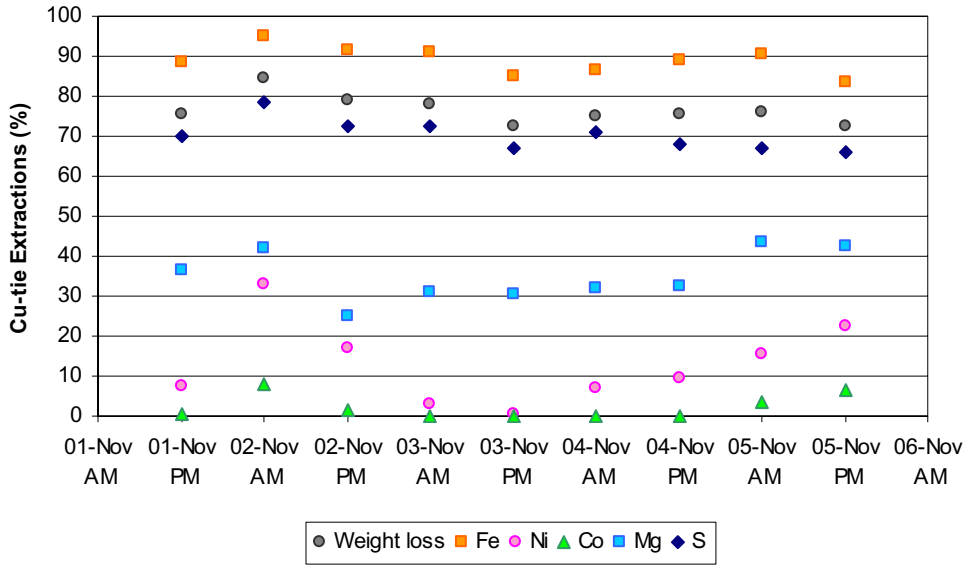

Figure 9 - PP2 Cu-tie extractions

2.5.3. Trends

Leaching discharge was characterised by 12-hour composite samples. Solution samples were taken after the filtration unit, and the filter cake was collected every 12 hours, washed and analyzed. Filtrate and residue 12-hour average assays are shown in Table 17 and Table 18 for PP1 and Table 19 and Table 20 for PP2, respectively.

Table 17 - PP1 Filtrate assays (mg/L)

Date	Fe	Fe(II)	Ni	Cu	Co	Mg
13-Oct AM	197800	192410	29	62	<5	1050
13-Oct PM	213800	209900	35	5	<5	890
14-Oct AM	219800	219800	<5	<5	<5	787
14-Oct PM	230500	230500	298	6	7	573
15-Oct AM	236000	226400	119	5	<5	550
15-Oct PM	248550	207400	<5	<5	<5	495
16-Oct AM	224500	214200	327	16	13	442
Average	224421	214373	117	15	6	684
Min	197800	192410	5	5	5	442
Max	248550	230500	327	62	13	1050
Rel.STD	7%	6%	119%	143%	47%	33%

Table 18 - PP1 Residue (filter cake) composition (%wt)

Date	Fe	Ni	Cu	Co	Mg	S	S=
13-Oct AM	36.6	3.01	4.58	0.28	0.14	40.5	35.8
13-Oct PM	37.4	3.10	5.21	0.32	0.14	44.0	42.6
14-Oct AM	42.9	2.82	4.40	0.26	0.17	41.9	34.6
14-Oct PM	35.9	3.38	6.56	0.38	0.15	43.8	33.2
15-Oct AM	36.0	2.64	4.53	0.28	0.14	39.3	28.7
15-Oct PM	37.9	3.07	5.07	0.31	0.11	40.3	36.0
16-Oct AM	25.8	4.41	9.09	0.51	0.17	47.6	24.7
Average	36.1	3.20	5.63	0.33	0.146	42.5	33.7
Min	25.8	2.64	4.40	0.26	0.110	39.3	24.7
Max	42.9	4.41	9.09	0.51	0.170	47.6	42.6
Rel.STD	14%	18%	30%	26%	14%	7%	17%

Table 19 - PP2 Filtrate assays (mg/L)

Date	Fe	Fe(II)	Ni	Cu	Co	Mg
01-Nov PM	237400	217800	484	<5	8	370
02-Nov AM	217800	214420	945	187	34	319
02-Nov PM	211300	203400	510	178	19	291
03-Nov AM	217500	209500	164	9	<5	282
03-Nov PM	203800	197500	230	<5	<5	210
04-Nov AM	207000	199600	216	<5	<5	211
04-Nov PM	206800	195800	255	7	<5	184
05-Nov AM	188200	182000	429	59	16	189
05-Nov PM	200500	197670	588	76	25	206
Average	210033	201966	425	59	14	251
Min	188200	182000	164	5	5	184
Max	237400	217800	945	187	34	370
Rel.STD	7%	5%	58%	127%	79%	26%

Table 20 - PP2 Residue (filter cake) composition (%wt)

Date	Fe	Ni	Cu	Co	Mg	S	S=
01-Nov PM	26.9	4.07	7.43	0.45	0.17	43.9	28.5
02-Nov AM	19.5	4.49	10.80	0.59	0.24	50.8	27.4
02-Nov PM	23.7	4.20	8.37	0.47	0.19	49.5	23.6
03-Nov AM	23.5	4.90	8.20	0.50	0.21	46.4	29.1
03-Nov PM	30.9	3.87	6.74	0.41	0.16	45.3	28.5
04-Nov AM	30.2	3.78	6.77	0.41	0.19	43.0	24.3
04-Nov PM	25.2	3.72	6.96	0.41	0.19	47.1	30.1
05-Nov AM	21.6	3.76	8.39	0.44	0.20	50.8	29.5
05-Nov PM	33.7	2.87	6.40	0.34	0.15	45.8	25.1
Average	26.1	3.96	7.78	0.45	0.19	47.0	27.3
Min	19.5	2.87	6.40	0.34	0.15	43.0	23.6
Max	33.7	4.90	10.80	0.59	0.24	50.8	30.1
Rel.STD	18%	14%	17%	16%	14%	6%	9%

Magnesium tenors in the filtrate were decreasing throughout the campaign which suggested that the Mg composition in the synthetic liquor used in the start-up was not accurate. At the end of PP2 Mg tenors were around 200 mg/L pointing out that should have been the composition for the start-up liquor. Due to a higher weight loss in PP2 base metals content in the residue are also higher, going from 5.63% Cu in PP1 to 7.78% Cu in PP2.

Since the filtrate was (partially) sent back to the feed it was expected that both feed and filtrate solutions would have the same Cu tenors, but as it shown in Figure 10 there is a noticeable difference between copper in the filtrate and copper in the feed solution. Since the feed batches were made up right after one

of the feed tanks was empty, concentrate and filtrate were mixing for around 10 to 12 hours before feeding to the circuit. Based on that, some Cu leaching might have occurred in the feeding tanks, increasing the Cu in solution, and then Cu in solution precipitated when it came into contact with the hydrogen sulphide produced in the leaching circuit (cementation reaction), decreasing the Cu in the filtrate.

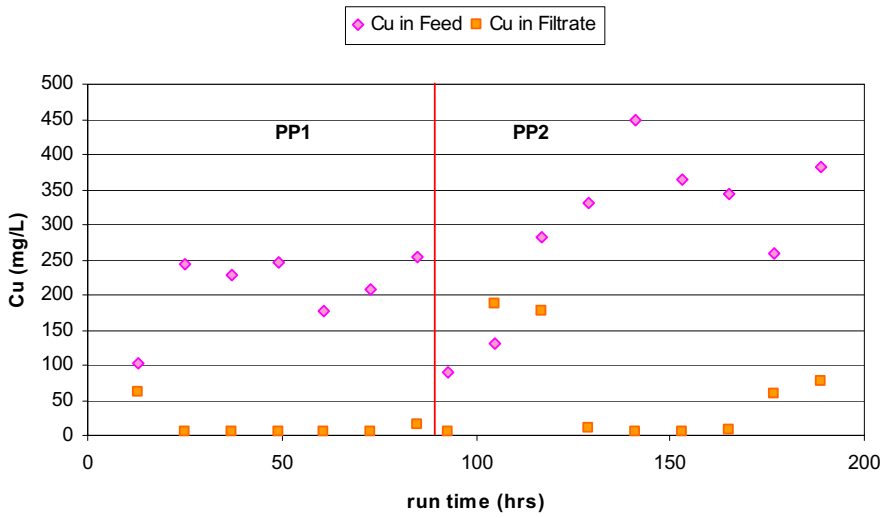

Figure 10 - Cu tenors in the feed and filtrate solutions

2.5.4. Profile Sampling

Profile samples were taken from each tank once a day during the 2 campaigns. For PP1, after R2 (around 1 hour residence time) most of the solid compositions remain almost the same as it is shown in Figure 11. For PP2 (Figure 12) not only did they remain the same after PP2, but the nickel, copper and cobalt levels in the solids started to decreased after R2, which could indicate some of the base metals were leached after R2.

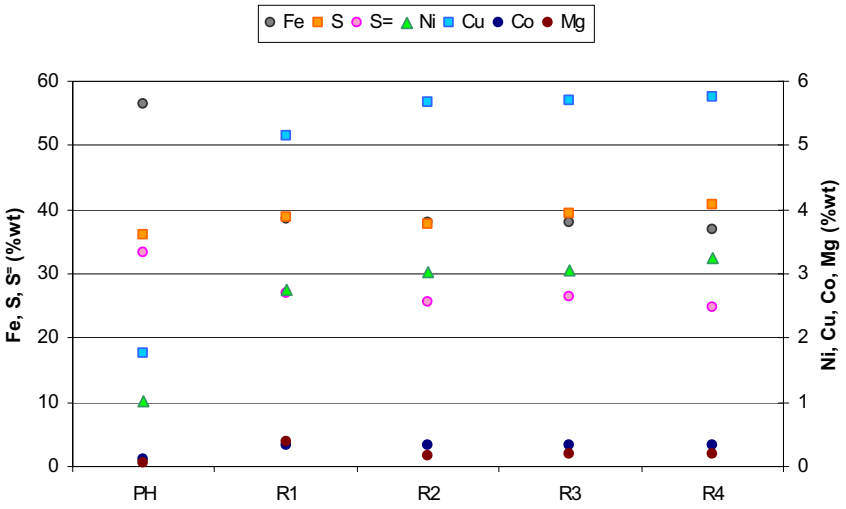

Figure 11 - PP1 average profile solids composition

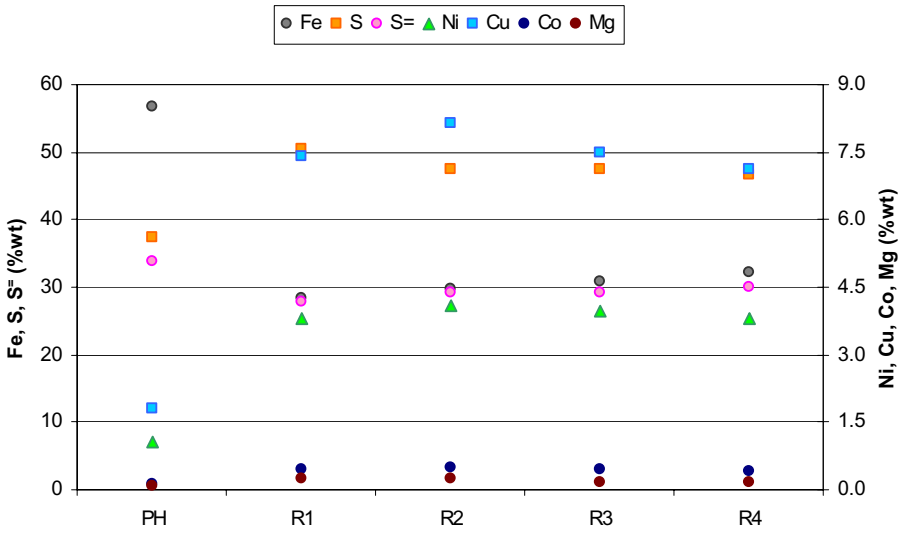

Figure 12 - PP2 average profile solids composition

2.5.5. Scrubbing Operation

Overall extraction efficiencies in the scrubbing circuit were calculated for both campaigns and are listed in Table 21. These extractions are derived from a mass balance for the entire campaign and calculated as:

$$Efficiency = \frac{S_{SCRUBBING-LIQUOR}}{S_{OFF-GAS}} \times 100\% \quad \textbf{[7]}$$

Table 21 - Scrubbing efficiency

Campaign	*S* in off-gas (kg)	*S* in scrubbing liquor (kg)	Efficiency
PP1	10.94	10.70	98%
PP2	14.66	12.01	82%

Scrubbing efficiency for PP1 was 98%, but in PP2 decreased to 82%. There is no apparent reason for the difference between campaigns. Although there were some irregularities in PP2 operation, twice the hose in the B/T pump broke and the back up system was put on line, it does not justify the difference between PP1 and PP2 scrubbing efficiencies.

2.5.6. PGM Survey

Samples from Ferguson Lake ore, flotation concentrate, flotation tails and primary ore leach residue were analyzed for PGM. As it is shown in Table 22 all the PGM remained with the flotation concentrate after the flotation process as well as with the primary ore leach residue after the leaching. This is a very positive result meaning that PGM could be extracted downstream as it was planned.

Table 22 - PGM survey

Sample	Au*	Ag	Pt	Pd	Ru	Rh	Ir
Ferguson Lake ore	0.08	2.00	0.20	2.28	0.06	0.02	<0.02
Flotation Concentrate	0.20	2.10	0.20	3.17	0.04	0.04	<0.02
Flotation Tails	0.03	<0.5	0.05	0.2	0.04	0.04	<0.02
Primary ore leach liquor	<0.01	1.63	<0.01	<0.01	-	-	-
Primary ore leach residue	0.25	7.80	1.12	9.84	0.05	0.11	<0.02

*mg/L for liquor, g/t for solids

3. Liquid-solid Separation Testing

A sample from the flotation concentrate used as feed for the PL pilot plant running was subjected to liquid-solid separation testing. Also, a sample from the Primary Ore Leach residue was collected during the pilot plant and submitted to liquid-solid separation testing. The testwork program is summarized in Table 23 indicating the test items performed on each sample. Summarised test results are included in the sections below and detailed data are included in Appendix C.

Table 23 - Liquid-solid separation testwork program summary

Sample identification	Testing Performed
B-3 concentrate	Flocculant Selection, Static Settling, Vacuum Filtration, Pressure Filtration, Rheology
Leach Residue	Flocculant Selection, Static Settling, Vacuum Filtration, Rheology

3.1. Procedures

3.1.1. Sample Receipt and Description

Particle size determinations were performed on representative aliquots retrieved from the individual samples. The concentrate and leach residue determinations were performed on a Malvern Mastersizer

2000 laser particle sizer. The resulting k_{80} for each sample is summarized in Table 24. The details of the individual laser particle size determinations are included in Appendix C.

Table 24 - Sample Characterization

Sample	k_{80} (microns)
B-3 concentrate	49.46
Leach Residue	78.38

Pulp from each sample received was decanted and combined to collect sufficient process water and pulp for the subsequent rheology and/or liquid-solid separation tests. A sample aliquot was retrieved from each of the remaining pails and filtered; the wet cake produced during the filtration was analyzed for solid content in a Halogen Moisture Analyzer HR83 (Mettler-Toledo), and in a conventional oven for comparison purposes. One of the dried samples was submitted for specific gravity (SG) determination using a nitrogen-purged “Model 1305 Micromeritics” multi-volume pycnometer.

3.1.2. Static Settling-Thickening Tests

Static settling-thickening tests involved three distinct stages:

- Flocculant screening, selection and confirmation;
- Static settling-thickening tests-Thickener feed solids optimization;
- Static settling-thickening tests-Flocculant dosage optimization.

3.1.2.1. Reagent Screening and Selection

Commercially available flocculation reagents were tested (“screened”) by performing small scale settling tests. Freshly prepared solutions of polyelectrolytes and polyacrylamides, containing 0.25 g/L active substance were used for testing/screening. Variable amounts of the reagent solutions tested were added to aliquots of slurry samples at a 100 mL constant volume. Settling behaviour of the resulting slurries was examined in terms of relative effectiveness of the reagents with respect to particle aggregation, floc formation, resulting structure characteristics and supernatant clarity. The main purpose of the flocculant screening tests was to determine a generic type of reagent or combination that is likely to be the most suitable for the separation process. A second important objective was to find a common, widely available and inexpensive reagent for all samples tested.

3.1.2.2. Static Settling Tests

Standard Kynch settling tests (with rakes) were performed; the sample was tested at variable feed slurry solids densities and at variable reagent additions as selected based on the reagent screening results. The test data produced consisted of interface height and feed slurry density profile under a given flocculant

dosage. Thickener sizing was performed using the Talmadge and Fitch method. The thickener unit area and hydraulic capacity were calculated.

3.1.3. Vacuum Filtration Tests

For both samples vacuum filtration testing was performed on simulated underflow samples produced by static settling-thickening testing. The supernatant was decanted from the thickened pulp and only the thickened "underflow" was used as filter test feed. The testwork was performed using a filter cloth selected through preparatory scoping tests.

Standard vacuum filter ("leaf-pour-on") testing was performed using a 113 mm diameter "Pannevis Model" leaf filter. The B-3 Concentrate tests were performed with varying slurry volumes to yield a range of filter cake thicknesses from ~8 mm to ~25 mm, and the Leach residue tests were performed with varying slurry volumes to yield a range of filter cake thicknesses from ~5 mm to ~15 mm. These cake thicknesses were determined through preparatory scoping tests. Drying times in both cases were varied to obtain a range of cake moistures. Test data produced included vacuum level during form time and dry time, form time, drying time, observations regarding cracking or separation from the filter wall, filtrate volume, cake thickness, wet cake weight and dry cake weight. It should be noted that due to limited concentrate sample availability, only partial settling and filtration testing could be performed, therefore, the results reported above may not be optimised.

Based on the above data, the following parameters were calculated: cake moisture, feed solids density, unit weight and dry time factor. Relevant graphical dependencies were established. Filter output for a given cake thickness and moisture content was calculated. Test details are included in Appendix C.

3.1.4. Pressure Filtration Testwork

The B-3 Concentrate sample aliquot was subjected to pressure filtration testwork. The pressure filtration tests involved three distinct stages:

- Filter cloth selection;
- Standard pressure filtration testing;
- Data reduction.

Standard pressure filter testing was carried out. Up to 3 filter tests were performed with varying slurry volumes to yield filter cake thickness of 37 mm, as allowed by the available filter chambers. The maximum feed pressure achievable in the test unit is 100 psig (~ 115 psi). The test apparatus consists of a "Shriver Unit 3" filterpress (Eimco) and necessary ancillaries: feed tanks, pumps, valves, pressure gage, etc. The details of the tests are included in Appendix C.

3.1.5. Rheology Sample Preparation and Characterization

The rheology test samples consisted of B-3 Concentrate and Leach residue underflow samples at solids densities exceeding their respective compression points determined during static settling-thickening testing. Representative test-sample aliquots were retrieved from the unflocculated samples and submitted to particle size distribution ("PSD") determination using a Malvern laser particle sizer "Model Mastersizer 2000".

After the retrieval of the PSD samples, the homogenized main test samples were flocculated at the "optimum" dosage as determined during the settling-thickening testing. Representative test-sample aliquots were retrieved from the homogenised and flocculated samples and subjected to rheology measurements as described below.

Duplicate aliquot slurry samples were retrieved from each homogeneous slurry sub-sample under sufficient manual mixing to allow for complete homogeneity, while gentle enough to preserve particle integrity. These duplicate aliquots were used for the simultaneous characterisation of the "measurement-ready" sample and for the rheological measurements themselves.

Several slurry sub-samples were produced, in decreasing order of their solids density, and subjected to rheological measurements in order to allow for their comparative characterisation in terms of rheological behaviour. The characterisation of the "measurement-ready" samples consisted of pulp density and solids density determinations. The pulp density was determined by weighing exactly measured volumes of de-aerated slurry samples at ~20-25ºC; the determinations were performed in duplicate. Solids density was determined by drying to constant weight at 100ºC; the determinations were performed in duplicate. The specific gravity of the dry solids was determined using a nitrogen-purged "Model 1305 Micromeritics" multi-volume pycnometer.

The actual specific gravity of the (ASG) of the solid particles within the slurry was calculated based on the measured slurry density. The results allowed for the quantification of the relationship between a) ASG versus SG (specific gravity of dried solids) and b) slurry density versus solid content (also known as solids density).

The Starfields leach res and conc. used the Vane determination, which for each pulp sample at varying thickened densities, simply provides the maximum (peak) shear stress at a constant rotation of $0.01s^{-1}$ for 5 minutes. These samples were tested at there maximum thickened underflow solids densities.

3.2. Results

3.2.1. B-3 Concentrate Results

A Particle size determination was performed using a Malvern Mastersizer 2000 laser particle sizer on a representative aliquot retrieved from the sample, the resulting particle size d80 was 49 micron.

Flocculant screening and selection was performed using commercially available flocculation reagents by performing small scale (100 mL) settling tests. The tested slurry sample required the use of a medium-charge density anionic flocculant – a relatively common reagent called Magnafloc 155, manufactured by CIBA. The sample settled very fast with a clear supernatant using a reagent addition of 5 g/t dry solids.

2000-mL static settling-thickening tests were performed using the selected flocculation reagent. The optimized sample settled well when the feed was diluted (auto-diluted) from the "as received" solids density of 31.5% wt/wt to 22% wt/wt paired with the addition of 9 g/t of medium-charge anionic flocculant CIBA Magnafloc 155; these conditions yielded the following thickener sizing data: 0.015 m^2/t/d thickener underflow unit area (TUFUA), 0.003 m^2/t/d thickener hydraulic unit area (THUA) and 1027 m^3/m^2/day initial settling rate (ISR); the solids density of the resulting underflow was 79% wt/wt and the supernatant was clear after 60 minutes of elapsed time. The results converged to suggest that the sample displayed good settling and thickening properties.

A rheological characterization of the underflow (at 79% solids) was attempted using CCRV (Concentric Cylinder Rotational Viscometry), but was not successful because the sample began to separate further as it was sheared early in the test sequence and the CCRV sensor was restricted from reaching the measurement position in the sample. Therefore, a vane sensor was used to determine the peak shear stress of the sample at the highest naturally settled solids density (post shearing). The test was performed at 86.9% wt/wt solids and resulted in a peak shear stress of 448 Pa. The shear stress value would generally be considered at the high end of commercial pumping range, therefore additional attention should be considered for this product during the pump design/sizing.

The thickened underflow sample was subjected to laboratory vacuum filtration testing whereby an example of the predicted throughput achievable by a belt filter ranged from 8141 kg/m^2/h through 280 kg/m^2/h, at 10% to 6% wt/wt residual moisture, respectively, at a cake thickness of 10 mm. The disc or drum filter example of predicted throughput (under balanced filter operating conditions) varied from 6564 kg/m^2/h through 3282 kg/m^2/h for cake thickness ranging from 10 through 20 mm, respectively corresponding to higher residual cake moisture content values (under balanced filter operating conditions) ranging from 10.0% through 9.1% wt/wt, respectively. Please note that the operating conditions and filter outputs that are reported are examples based on the data produced during testing and do not incorporate any safety or scale up factor. Additional design outputs, that are more suited to an individual process, can

be calculated using the vacuum filtration graphs and resulting relationship values that are included in this reporting package.

The thickened underflow sample was also subjected to laboratory pressure filtration testing. One test was performed that resulted in a throughput of 444 kg/m^2/h at 13.8% wt/wt residual moisture at a cake thickness of 37 mm. Settling solids in the diaphragms of the pump resulted in pressure loss and eventual plugging during the 6 minute test duration. Multiple attempts were made to avoid plugging but were not successful; therefore, the tests were aborted. The result reported for pressure filtration should be dismissed on that basis.

3.2.2. Leach Residue Results

A Particle size determination was performed using a Malvern Mastersizer 2000 laser particle sizer on a representative aliquot retrieved from the sample, the resulting particle size d_{80} was 78 micron.

Flocculant screening and selection was performed using commercially available flocculation reagents by performing small scale (100 mL) settling tests. The tested slurry sample required the use of a medium-charge density anionic flocculant – a relatively common reagent called Magnafloc 155, manufactured by CIBA. The sample settled fast with a slightly cloudy supernatant using a reagent addition of 47 g/t dry solids.

2000-mL static settling-thickening tests were performed using the selected flocculation reagent. The optimized sample settled well when the feed was tested at the "as received" solids density of 3% wt/wt paired with the addition of 48 g/t of medium-charge anionic flocculant CIBA Magnafloc 155; these conditions yielded the following thickener sizing data: 0.07 m^2/t/d thickener underflow unit area (TUFUA), 0.03 m^2/t/d thickener hydraulic unit area (THUA) and 960 m^3/m^2/day initial settling rate (ISR); the solids density of the resulting underflow was 27% wt/wt and the supernatant was slightly cloudy after 60 minutes of elapsed time. In other tests underflow density was 39% but at a higher thickener unit area. If a denser underflow is the goal this could be achieved, sacrificing thickener unit area and overflow clarity. The results converged to suggest that the sample displayed good settling and thickening properties.

The bench testing indicated that significant increases in the flocculant dosage would result in better supernatant clarity, however, underflow solids density decreased significantly due to over-flocculation. It is recommended that the use of an overflow clarifier be investigated if supernatant clarity is critical to the operation.

It should be noted that the static settling results and subsequent design criteria do not incorporate any safety or scale up factor. Depending on factors such as throughput, cost, risk tolerance, etc., the equipment suppliers may require additional data, including continuous thickener testing, in order to establish the final design criteria.

The CCRV (Concentric Cylinder Rotational Viscometry) system could not be used due to the corrosive nature of the sample, making it incompatible with the material of construction of this system. Therefore, a vane sensor was used for shear stress determination. The Leach Residue sample (pH = <1) was tested as a flocculated sample with Ciba Magnafloc 155 at a dosage of 48 g/t. The vane sensor was used to determine the peak shear stress of the sample at the highest naturally settled solids density. The test was performed at 43.4% wt/wt solids and resulted in a peak shear stress of 17 Pa. This value generally represents slurry that would be suitable for pumping.

The thickened underflow sample was subjected to laboratory vacuum filtration testing at a feed density of ~31% wt/wt solids; this diluted "thickened" solids density was required in order to be able to load the vacuum filter. The feed pulp dilution was required because the initially prepared thickener underflow (at target solids density~45-48% wt.) could not be handled. Chemically induced rheopexy (shear-thickening) was suspected, yet not proven as the underlying cause for this flow behaviour. An example of the predicted throughput achievable by a belt filter ranged from 567 kg/m^2/h through 127 kg/m^2/h, at 46% to 44% wt/wt residual moisture, respectively, at a cake thickness of 5 mm. The disc or drum filter example of predicted throughput (under balanced filter operating conditions) varied from 193 kg/m^2/h through 64 kg/m^2/h for cake thickness ranging from 5 through 15 mm, respectively corresponding to similar residual cake moisture content values (under balanced filter operating conditions) ranging from 44.5% through 43.9% wt/wt, respectively. Significant cracking occurred in the cakes during the drying cycle of the filtration tests, which may render cake washing as an unsuitable method of liquor displacement. A filter vendor could have suitable equipment design remedies available.

Based on the results of the tests and the visual observations noted during the testing, it was determined that this sample was chemically unstable; therefore, the results produced herein may be conservative in nature, particularly with respect to the filtration test. Additionally, the lack of rheology data in conjunction with the apparent flow-adverse behaviour of the underflow is of concern. Since the amenability of the commercially produced underflow for continuous processing cannot be fully ascertained from the work done, it would be appropriate to raise a "process-risk" caution flag at this point in time.

It is recommended that the above unit operations are tested on a Pilot scale in order to produce real-time results on a stable discharge pulp.

Conclusions and Recommendations

- Primary ore leaching was successfully run in a mini pilot plant configuration, producing the necessary liquor to move forward in oxidation and hydrolysis testing with real Ferguson Lake ore leached liquor.

- Acid consumption was optimized in a series of bench tests and confirmed in the pilot plant runs. Acid dosage around 600kg of hydrochloric acid per tonne of solids was sufficient to obtain 73% iron extraction and 63% sulphide extractions while minimizing base metal extraction (6% Ni and 2% Co). Acid dosage in excess of 600 kg/t could lead to base metals leaching.

- Scrubbing media was also optimized. Hydrogen sulphide was continuously scrubbed during the pilot plant run using 2 counter-current columns filled with a solution of 100 g/L of sodium hydroxide. This would not be needed commercially, though, as the hydrogen sulphide would be converted to concentrated sulphuric acid in a conventional acid plant.

- A more detailed study should be carried out to determine the optimum residence time for this circuit. Profile sampling showed that after 1 hour iron and sulphide extractions do not have a considerable increment while at the same time there were indications that base metals began to be leach after 1 hour.

- A PGM survey done on all the streams in this process showed that PGM remained with the primary leach residue and no losses to the PL liquor were incurred.

- Liquid-solid separation testing was carried out on a sample form the flotation concentrate and a sample from the leach residue. The main results were:

  - Flotation concentrate

    The tested slurry sample required the use of a medium-charge density anionic flocculant – a relatively common reagent called Magnafloc 155, manufactured by CIBA.

    Static settling-thickening tests were performed using the selected flocculant CIBA Magnafloc 155 yielded the following thickener sizing data: 0.015 m^2/t/d thickener underflow unit area (TUFUA), 0.003 m^2/t/d thickener hydraulic unit area (THUA) and 1027 m^3/m^2/day initial settling rate (ISR); the solids density of the resulting underflow was 79% wt/wt and the supernatant was clear after 60 minutes of elapsed time.

    A vane sensor was used to determine the peak shear stress of the sample at the highest naturally settled solids density (post shearing). The test was performed at 86.9% wt/wt solids and resulted in a peak shear stress of 448 Pa. The shear stress value would generally be considered at the high end of commercial pumping range, therefore additional attention should be considered for this product during the pump design/sizing..

    The thickened underflow sample was subjected to laboratory vacuum filtration testing whereby an example of the predicted throughput achievable by a belt filter ranged from 8141 kg/m^2/h through

280 kg/m^2/h, at 10% to 6% wt/wt residual moisture, respectively, at a cake thickness of 10 mm. The disc or drum filter example of predicted throughput (under balanced filter operating conditions) varied from 6564 kg/m^2/h through 3282 kg/m^2/h for cake thickness ranging from 10 through 20 mm, respectively corresponding to higher residual cake moisture content values (under balanced filter operating conditions) ranging from 10.0% through 9.1% wt/wt, respectively.

Based on the high throughput results of the thickened-feed vacuum filtration tests, it is recommended that further vacuum filtration tests are performed using the direct discharge (unthickened flotation discharge) pulp as feed to the vacuum filter; this would determine if it is necessary to have a thickener included as a part of the concentrate L/S separation process.

- Leach residue results

The tested slurry sample required the use of a medium-charge density anionic flocculant – a relatively common reagent called Magnafloc 155, manufactured by CIBA.

Static settling-thickening tests were performed using the selected flocculation reagent. The optimized sample settled well when the feed was tested at the "as received" solids density of 3% wt/wt paired with the addition of 48 g/t of medium-charge anionic flocculant CIBA Magnafloc 155; these conditions yielded the following thickener sizing data: 0.07 m^2/t/d thickener underflow unit area (TUFUA), 0.03 m^2/t/d thickener hydraulic unit area (THUA) and 960 m^3/m^2/day initial settling rate (ISR); the solids density of the resulting underflow was 27% wt/wt and the supernatant was slightly cloudy after 60 minutes of elapsed time.

A vane sensor was used for shear stress determination. The Leach Residue sample (pH = <1) was tested as a flocculated sample with Ciba Magnafloc 155 at a dosage of 48 g/t. The vane sensor was used to determine the peak shear stress of the sample at the highest naturally settled solids density. The test was performed at 43.4% wt/wt solids and resulted in a peak shear stress of 17 Pa. This value generally represents a slurry that would be suitable for pumping.

The thickened underflow sample was subjected to laboratory vacuum filtration testing at a feed density of ~31% wt/wt solids; this diluted “thickened” solids density was required in order to be able to load the vacuum filter. The feed pulp dilution was required because the initially prepared thickener underflow (at target solids density~45-48% wt.) could not be handled. Chemically induced rheopexy was suspected, yet not proven as the underlying cause for this flow behaviour

It is recommended that the above unit operations are tested on a Pilot scale in order to produce real-time results on a stable discharge pulp.

Appendix A – Bench Test Data Sheets

Appendix A - Bench test details

Project No.: 12323-003 **Project Technologist:** Hugh Given
Client: Starfield Resources **Date of Test:** 9-Jul-11
Test No.: L1
Objective: 4-hr leach test, 23% solids from Ferguson Lake Core with 552 kg/t HCl - 25% HCl - 105C

Conditions Overview:

Leaching

Reaction: $Fe_7S_8 + 14HCl \rightarrow 7Fe^{2+} + 7H2S + S + 14Cl^-$

Temperature:	**105**	℃			Target Grind: k80 = 50 micron
Time:	**4**	hr	(sample at 1,2 and 4 hr)		Actual Grind: **92** 20to400 wet screen
Solids density:	**28%**	solids (before acid addition)			Grind Info: **30 min at 65% in 2kg rod mill (#4) pulped in sea water**
Feed:	**275**	g solids (dry)	sg:	4.6	
Feed cake moisture:	**8.8%**	H_2O			
Wet cake to add:	**302**	g solids (wet)			
Calculated pulp weight:	982	g pulp (before acid)			
Recycle Liquor to add:	**681**	g liquor	sg:	1.5	
HCl dosage:	552	kg/t			
Acid strength:	25%	w/w			
HCl (25%) to add:	**607.2**	g HCl (25%)	sg:	1.13	Calculated volume: 1280 mL
N_2 Blanket Flow:	**~200**	mL/min	(gas blanket - sparging is not required)		

H2S Scrubbing — Condensor: yes - reflux into reactor

Reaction: **2CuCl2 + 2H2S = Cu2S + S + 4HCl**

Scrubbers:	**3**	at:	2.5	L per scrubber
Scrubber 1, 2 composition:	**100**	g/L Cu		
Scrubber 3 composition:	**100**	g/L NaOH		

Sampling Requirements:

	h	solids	liquids
	1h	partial	partial
(just before LOW COMP addition)	2h	partial	partial
	final (4h)	full	full
	final wash		Fe
	scrubber 1	Cu, S	
	scrubber 2	Cu, S	
	scrubber 3		S

Liquor Suite: HCl, Fe, Fe(II), Cu, Ni, Co, Mg
Solids Suite: S=, Fe, Cu, Ni, Co, Mg

H&S **This test produces H2S. Perform test in fumehood with plant scrubber operational. Review MSDS for H2S**
Hot acidic pulps. Review MSDS for HCl and FeCl2
Concentrated acid. The test uses 25% HCl. Use appropriate precautions when handling this and test product pulps
Use face shield, post signs warning of H2S risk in area

Procedure:

1. Review MSDS for concentrated hydrochloric acid (HCl), ferrous chloride ($FeCl_2$) and hydrogen sulphide gas (H_2S)
2. Set up 3 L glass leach vessel with vapour condenser and mixer. Install three scrubbers with spargers, in series connected to vacuum and the top of the condensor). Fill the first two scrubbers with ~100 g/L Cu (as CuCl2) and the third one with ~100 g/L NaOH. Make provisions for acid addition onder the impeller. Provide a N2 blanket on top of the leach liquor. Note: No metal exposed surfaces (prefered use of glass, PP and teflon materials) .
3. Prepare Recycle Liquor by mixing the correct amounts of $FeCl_2.4H_2O$, $MgCl_2.6H_2O$, H_2O and HCl into the leach vessel. Rfefer to separate makeup instruction sheet.
4. Obtain target amount of wet weight feed and add to recycle liquor. Note observations (pH, ORP, T and evolution of H_2S if any)
5. Commence heat up to 90C and start adding 25% HCl acid (use pump or burette). Add acid slowly (target addition of 30 to 60 min - depending on development of heat and foaming). Record observations.
9. Collect ~75 mL aliquot samples at 1, 2 and 4 hr. Filter, submit filtrate, wash 5x with DIW (V=Vfiltrate). and dry solids for chem. analysis.
12. When total time has elapsed stop heating & mixer. Obtain final pulp weight.
14. HOT vacuum filter final leach slurry. Measure density and weight of PLS. Collect sample of PLS to submit to lab. Investgate if PLS dilutions are requried to prevent crystallization.
15. Wash & displace 5X with hot DI water of same volume as PLS. Combine wash solutions. Measure density and weight of wash solution. Collect sample and submit to lab.
16. Obtain wet weight of filtercake. Dry in oven to constant weight. Record final weight of dried product. Submit dried solids for lab analysis.
19. Weigh scrubber solutions, filter solids, determine dry weight, wash extensively with DI water, submit solids for Cu and S.

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Hugh Given
Date of Test: 9-Jul-11

Leach Test Data:

Time		*Reactor*				*Reagents / Feed*				*Comments*
(24 h)	(h) elapsed	Temp ℃	FAT	pH	ORP	ore pulp g	Recycle liquor g	HCl 25% g		
9:00		24.7				302	681	0		Nothing happened when the ore and the
9:50		91.1		2.47	222			0.0		recycled liquor were mixed.
10:00		99.9		0.86	183			97.8		Acid wass added from 9:50 to 11:40,
10:10		101.7		0.89	-31			144.5		temperature was kept below 100C in this
10:20		100.0		-0.06	-40			266.7		time interval.
10:52		98.2		-0.22	-41			371.5		pH probe was broken.
11:23		100.1		-0.09	-41			507.8		
11:40		99.2		-0.31	-41			615.1		
12:40		105.4			-41					
13:40		103.6			-42					
Totals/Avg:		101.0		0.18	-12	302		615	0.0	

dry (g): **0**
Final Pulp weight: **g**
Calculated Acid Addition: **558** **kg/t**

Sampling INFO

		Weight		Volume	**emf at**	**pH at**	SG	Calc PLS	FAT	Wet	Dry	Colour PLS/RES.
Sample #	time	pulp, g	PLS, g	PLS, mL	**room T**	**room T**	g/mL	Vol, mL	g/L HCl	res, g	res, g	Filtration
1	1 h	111.5	99.8	72	283	-0.24	1.377			10.70		
2	2 h	123.7	99.1	72	352	0.26	1.371			14.10		
3	4 h	1229.2	979.1	707	348	0.02	1.385			131.50	115.6	
4	wash		742.9	677			1.098	677				
5	S1	3165.2	2763.5	2358	391	0.38	1.172		17	336.60	83.7	
6	S2	2553.7	2501.1	2070	424	0.75	1.208		4	31.67	9.4	
7	S3	2756.4	2756.4	2501	-708	12.35	1.102	2501				

Final Filtration:

Diameter of filtration paper:	150 mm	Clarity of filtrate:	clear
type of paper (Whatman ##):	3	Colour of filtrate:	green
Filtration time:	0.08 hrs	Clarity of wash:	clear
Washing time:	0.08 hrs	Colour of wash:	very light green
Volume of wash:	677 mL	Colour of residue:	black with metallic specks
Cake thickness:	cm		

Other Notes / Observations

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Hugh Given
Date of Test: 9-Jul-11

Metallurgical Balance

Product	Amount	g/L	Assay (mg/L, *g/t*)							
min	(ml, g)	FA	Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake	275		56.80		1.480	0.870	0.087	0.38	29.0	29.0
Rec. Liquor	469		218492	218492				2003		
1hr PLS	72	0	201000	201000	1.8	<4	<8	1000		
2hr PLS	72	0	196000	196000	0.1	<4	<8	1000		
4hr PLS	707	0	204500	204500	0.4	<4	<8	1100		
wash	677	0	49600							
1hr RES										
2hr RES										
4hr RES	116		29.3		3.27	1.86	0.19	0.74	29.6	25.2
S1 solids	84	17			45.3				30.2	
S2 solids	9	4			22.6				49.3	
S3 liquor	2501								610	
Metal Units, mg			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake			156200	0	4070	2393	239	1045	79750	79750
Rec. Liquor			102557	102557	0	0	0	940	0	0
Total Units IN:			258757	102557	4070	2393	239	1985	79750	79750
1hr PLS			14568	14568	0	0	1	72	0	0
2hr PLS			14167	14167	0	0	1	72	0	0
4hr PLS			144567	144567	0	3	6	778	0	0
wash			33559	0	0	0	0	0	0	0
1hr RES			0	0	0	0	0	0	0	0
2hr RES			0	0	0	0	0	0	0	0
4hr RES			33870	0	3780	2150	220	855	34217	29131
S1 solids			0	0	37934	0	0	0	25289	0
S2 solids			0	0	2122	0	0	0	4629	0
S3 liquor			0	0	0	0	0	0	1526	0
Total Units OUT:			240732	173303	3780	2154	226	1778	65661	29131
Distribution			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
1hr PLS			43.0	#DIV/0!	0.0	0.0	0.3	8.5	0.0	0.0
2hr PLS			41.8	#DIV/0!	0.0	0.0	0.3	8.4	0.0	0.0
4hr PLS			426.8	#DIV/0!	0.0	0.1	2.6	90.9	0.0	0.0
wash			99.1	#DIV/0!	0.0	0.0	0.0	0.0	0.0	0.0
1hr RES			0.0	#DIV/0!	0.0	0.0	0.0	0.0	0.0	0.0
2hr RES			0.0	#DIV/0!	0.0	0.0	0.0	0.0	0.0	0.0
4hr RES			100.0	#DIV/0!	100.0	100.0	100.0	100.0	100.0	100.0
S1 solids			0.0	#DIV/0!	1003.5	0.0	0.0	0.0	73.9	0.0
S2 solids			0.0	#DIV/0!	56.1	0.0	0.0	0.0	13.5	0.0
S3 liquor			0.0	#DIV/0!	0.0	0.0	0.0	0.0	4.5	0.0
SUM:			710.8	#DIV/0!	1159.7	100.2	103.1	207.8	191.9	100.0
Extraction - Cu-tie										
4 hr			77			3	1	12	54	61
Head (cal)[1]			50.25	25.73	1.37	0.78	0.08	0.30	23.88	10.59
Head (direct)			56.80	0.00	1.48	0.87	0.09	0.38	29.00	29.00
Accountability (OUT/IN), %			93.0	169.0	92.9	90.0	94.6	89.6	82.3	36.5

weight loss
58.0%
55%

[1]Calculated Head includes both solid and liquid feed streams.

Project No.: 12323-003 **Project Technologist:** Ernesto
Client: Starfield Resources **Date of Test:** 20-Jul-10
Test No.: L2
Objective: 4-hr leach test, 23% solids from Ferguson Lake Core with 650 kg/t HCl - 25% HCl - 105C

Conditions Overview:

Leaching

Reaction: **$Fe_7S_8 + 14HCl \rightarrow 7Fe^{2+} + 7H2S + S + 14Cl^-$**

Temperature:	**105**	℃			Target Grind: k80 = 50 micron
Time:	**4**	hr	(sample at 1,2 and 4 hr)		Actual Grind: **92** 20to400 wet screen
Solids density:	**28%**	solids (before acid addition)			Grind Info: **30 min at 65% in 2kg rod mill (#4) pulped in sea water**
Feed:	**275**	g solids (dry)	sg:	4.6	
Feed cake moisture:	**8.8%**	H_2O			
Wet cake to add:	**302**	g solids (wet)			
Calculated pulp weight:	982	g pulp (before acid)			
Recycle Liquor to add:	**681**	g liquor	sg:	1.5	
HCl dosage:	415	kg/t			
Acid strength:	25%	w/w			
HCl (25%) to add:	**456.7**	g HCl (25%)	sg:	1.13	Calculated volume: 1147 mL
N_2 Blanket Flow:	**~200**	mL/min	(gas blanket - sparging is not required)		

H2S Scrubbing — Condensor: yes - reflux into reactor

Reaction: **CuSO4 + H2S = CuS + H2SO4**

Scrubbers:	**3**	at:	2.5	L per scrubber
Scrubber 1 composition:	**50**	g/L Cu		
Scrubber 2 composition:	**100**	g/L Cu		
Scrubber 3 composition:	**100**	g/L NaOH		

Sampling Requirements:

	h	solids	liquids
	1h	partial	partial
(just before LOW COMP addition)	2h	partial	partial
	final (4h)	full	full
	final wash		Fe
	scrubber 1	Cu, S	
	scrubber 2	Cu, S	
	scrubber 3		S

Liquor Suite: HCl, Fe, Fe(II), Cu, Ni, Co, Mg
Solids Suite: S=, Fe, Cu, Ni, Co, Mg

H&S **This test produces H2S. Perform test in fumehood with plant scrubber operational. Review MSDS for H2S**
Hot acidic pulps. Review MSDS for HCl and FeCl2
Concentrated acid. The test uses 25% HCl. Use appropriate precautions when handling this and test product pulps
Use face shield, post signs warning of H2S risk in area

Procedure:

1. Review MSDS for concentrated hydrochloric acid (HCl), ferrous chloride ($FeCl_2$) and hydrogen sulphide gas (H_2S)
2. Set up 3 L glass leach vessel with vapour condenser and mixer. Install three scrubbers with spargers, in series connected to vacuum and the top of the condensor). Fill the first two scrubbers with ~100 g/L Cu (as CuCl2) and the third one with ~100 g/L NaOH. Make provisions for acid addition onder the impeller. Provide a N2 blanket on top of the leach liquor. Note: No metal exposed surfaces (prefered use of glass, PP and teflon materials) .
3. Prepare Recycle Liquor by mixing the correct amounts of $FeCl_2.4H_2O$, $MgCl_2.6H_2O$, H_2O and HCl into the leach vessel. Rfefer to separate makeup instruction sheet.
4. Obtain target amount of wet weight feed and add to recycle liquor. Note observations (pH, ORP, T and evolution of H_2S if any)
5. Commence heat up to 90C and start adding 25% HCl acid (use pump or burette). Add acid slowly (target addition of 30 to 60 min - depending on development of heat and foaming). Record observations.
9. Collect ~75 mL aliquot samples at 1, 2 and 4 hr. Filter, submit filtrate, wash 5x with DIW (V=Vfiltrate). and dry solids for chem. analysis.
12. When total time has elapsed stop heating & mixer. Obtain final pulp weight.
14. HOT vacuum filter final leach slurry. Measure density and weight of PLS. Collect sample of PLS to submit to lab. Investgate if PLS dilutions are requried to prevent crystallization.
15. Wash & displace 5X with hot DI water of same volume as PLS. Combine wash solutions. Measure density and weight of wash solution. Collect sample and submit to lab.
16. Obtain wet weight of filtercake. Dry in oven to constant weight. Record final weight of dried product. Submit dried solids for lab analysis.
19. Weigh scrubber solutions, filter solids, determine dry weight, wash extensively with DI water, submit solids for Cu and S.

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto
Date of Test: 20-Jul-10

Leach Test Data:

Time		*Reactor*				*Reagents / Feed*				*Comments*
(24 h)	(h) elapsed	Temp ℃	FAT	pH	ORP	ore pulp g	Recycle liquor g	HCl 25% g		
7:15		23.3		1.09	427	302	681	0		Started acid addition at 7:40
7:40		105.2			36			0.0		Scrubber 1 turned black immediately
7:45		106.3			182			65.3		A lot of foaming in scrubber 1
7:52		103.4			186			95.0		Scrubber 2 with a lot of foaming, too
8:00		106.6			204			138.0		8:35 Foaming was too high in scrubber 1,
8:10		106.7			214			187.1		it was taken out of the circuit
8:42		105.7			66			335.6		8:47 Foaming was too high in scrubber 2,
9:40		104.8			23			388.5		it was taken out of the circuit, acid addition
10:40		106.2			16			456.7		was stopped.
										pH probe was broken
										Condensate collected was really small and
										the white solids got dissolved at the end
Totals/Avg:		105.7		#DIV/0!	127	302		457	0.0	

dry (g): **0**
Final Pulp weight: **g**
Calculated Acid Addition: **415** **kg/t**

Sampling INFO

Sample #	time	Weight pulp, g	Weight PLS, g	Volume PLS, mL	**emf at room T**	**pH at room T**	SG g/mL	Calc PLS Vol, mL	FAT g/L HCl	Wet res, g	Dry res, g	Colour PLS/RES. Filtration
1	1 h			0			1.000					
2	2 h			0			1.000					
3	4 h	1283.3	1091.7	770			1.418		0	191.62	151.8	
4	wash		1273.4	1257			1.013					
5	S1	2745.2	2629.4	2360			1.114		7	72.19	17.4	
6	S2	2812.0	2723.3	2256			1.207		5	54.29	10.8	
7	S3	2896.0	2896.0	2645			1.095					

Final Filtration:

Diameter of filtration paper:	150	mm	Clarity of filtrate:	clear
type of paper (Whatman ##):	3		Colour of filtrate:	green
Filtration time:	0.08	hrs	Clarity of wash:	clear
Washing time:	0.08	hrs	Colour of wash:	very light green
Volume of wash:	1257	mL	Colour of residue:	black with metallic specks
Cake thickness:		cm		

Other Notes / Observations

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto
Date of Test: 20-Jul-10

Metallurgical Balance

Product	Amount	g/L	Assay (mg/L, *g/t*)							
min	(ml, g)	FA	Fe	Fe(II)	Cu	Ni	Co	Mg	S	S=
Fer Lake	275		56.80		1.480	0.870	0.087	0.38	29.0	29.0
Rec. Liquor	469		218492	218492				2003		
1hr PLS	0	0								
2hr PLS	0	0								
4hr PLS	770	0	220000	220000	0.4	<3	<8	1500		
wash	1257	0	6660							
1hr RES										
2hr RES										
4hr RES	152		40.1		2.72	1.53	0.16	0.51	37.8	29.78
S1 solids	17				66.1				29.5	
S2 solids	11				59.7				32.9	
S3 liquor	2645								3400	
Metal Units, mg			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S=
Fer Lake			156200	0	4070	2393	239	1045	79750	79750
Rec. Liquor			102557	102557	0	0	0	940	0	0
Total Units IN:			258757	102557	4070	2393	239	1985	79750	79750
1hr PLS			0	0	0	0	0	0	0	0
2hr PLS			0	0	0	0	0	0	0	0
4hr PLS			169375	169375	0	2	6	1155	0	0
wash			8372	0	0	0	0	0	0	0
1hr RES			0	0	0	0	0	0	0	0
2hr RES			0	0	0	0	0	0	0	0
4hr RES			60886	0	4130	2323	243	774	57394	45217
S1 solids			0	0	11515	0	0	0	5139	0
S2 solids			0	0	6436	0	0	0	3547	0
S3 liquor			0	0	0	0	0	0	8992	0
Total Units OUT:			238633	169375	4130	2325	249	1929	75071	45217
Distribution			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S=
1hr PLS			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
2hr PLS			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
4hr PLS			71.0	100.0	0.0	0.1	2.5	59.9	0.0	0.0
wash			3.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0
1hr RES			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
2hr RES			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
4hr RES			25.5	0.0	100.0	99.9	97.5	40.1	76.5	100.0
S1 solids			0.0	0.0	278.8	0.0	0.0	0.0	6.8	0.0
S2 solids			0.0	0.0	155.8	0.0	0.0	0.0	4.7	0.0
S3 liquor			0.0	0.0	0.0	0.0	0.0	0.0	12.0	0.0
	SUM:		100.0	100.0	534.6	100.0	100.0	100.0	100.0	100.0
Extraction - Cu-tie										
4 hr			62			4	0	27	29	44
Head (cal)[1]			49.48	24.30	1.50	0.85	0.09	0.36	27.30	16.44
Head (direct)			56.80	0.00	1.48	0.87	0.09	0.38	29.00	29.00
Accountability (OUT/IN), %			92.2	165.2	101.5	97.2	104.1	97.2	94.1	56.7

weight loss
44.8%
46%

[1]Calculated Head includes both solid and liquid feed streams.

Project No.: 12323-003 **Project Technologist:** Ernesto
Client: Starfield Resources **Date of Test:** 21-Jul-10
Test No.: L3
Objective: 4-hr leach test, 23% solids from Ferguson Lake Core with 650 kg/t HCl - 25% HCl - 105C

Conditions Overview:

Leaching

Reaction: $Fe_7S_8 + 14HCl \rightarrow 7Fe^{2+} + 7H2S + S + 14Cl^-$

Temperature:	**105**	℃			Target Grind: k80 = 50 micron
Time:	**4**	hr	(sample at 1,2 and 4 hr)		Actual Grind: **92** 20to400 wet screen
Solids density:	**28%**	solids (before acid addition)			Grind Info: **30 min at 65% in 2kg rod mill (#4) pulped in sea water**
Feed:	**275**	g solids (dry)	sg:	4.6	
Feed cake moisture:	**8.8%**	H_2O			
Wet cake to add:	**302**	g solids (wet)			
Calculated pulp weight:	982	g pulp (before acid)			
Recycle Liquor to add:	**681**	g liquor	sg:	1.5	
HCl dosage:	650	kg/t			
Acid strength:	25%	w/w			
HCl (25%) to add:	**715.0**	g HCl (25%)	sg:	1.13	Calculated volume: 1376 mL
N_2 Blanket Flow:	**~200**	mL/min	(gas blanket - sparging is not required)		

H2S Scrubbing

Condensor: yes - reflux into reactor

Reaction: **2CuCl2 + 2H2S = Cu2S + S + 4HCl**

Scrubbers:	**3**	at: 2.5	L per scrubber
Scrubber 1 composition:	**50**	g/L Cu	
Scrubber 2 composition:	**100**	g/L Cu	
Scrubber 3 composition:	**100**	g/L NaOH	

Sampling Requirements:

	h	solids	liquids
	1h	partial	partial
(just before LOW COMP addition)	2h	partial	partial
	final (4h)	full	full
	final wash		Fe
	scrubber 1	Cu, S	
	scrubber 2	Cu, S	
	scrubber 3		S

Liquor Suite: HCl, Fe, Fe(II), Cu, Ni, Co, Mg
Solids Suite: S=, Fe, Cu, Ni, Co, Mg

H&S **This test produces H2S. Perform test in fumehood with plant scrubber operational. Review MSDS for H2S**
Hot acidic pulps. Review MSDS for HCl and FeCl2
Concentrated acid. The test uses 25% HCl. Use appropriate precautions when handling this and test product pulps
Use face shield, post signs warning of H2S risk in area

Procedure:

1. Review MSDS for concentrated hydrochloric acid (HCl), ferrous chloride ($FeCl_2$) and hydrogen sulphide gas (H_2S)
2. Set up 3 L glass leach vessel with vapour condenser and mixer. Install three scrubbers with spargers, in series connected to vacuum and the top of the condensor). Fill the first two scrubbers with ~100 g/L Cu (as CuCl2) and the third one with ~100 g/L NaOH. Make provisions for acid addition onder the impeller. Provide a N2 blanket on top of the leach liquor. Note: No metal exposed surfaces (prefered use of glass, PP and teflon materials) .
3. Prepare Recycle Liquor by mixing the correct amounts of $FeCl_2.4H_2O$, $MgCl_2.6H_2O$, H_2O and HCl into the leach vessel. Rfefer to separate makeup instruction sheet.
4. Obtain target amount of wet weight feed and add to recycle liquor. Note observations (pH, ORP, T and evolution of H_2S if any)
5. Commence heat up to 90C and start adding 25% HCl acid (use pump or burette). Add acid slowly (target addition of 30 to 60 min - depending on development of heat and foaming). Record observations.
9. Collect ~75 mL aliquot samples at 1, 2 and 4 hr. Filter, submit filtrate, wash 5x with DIW (V=Vfiltrate). and dry solids for chem. analysis.
12. When total time has elapsed stop heating & mixer. Obtain final pulp weight.
14. HOT vacuum filter final leach slurry. Measure density and weight of PLS. Collect sample of PLS to submit to lab. Investgate if PLS dilutions are requried to prevent crystallization.
15. Wash & displace 5X with hot DI water of same volume as PLS. Combine wash solutions. Measure density and weight of wash solution. Collect sample and submit to lab.
16. Obtain wet weight of filtercake. Dry in oven to constant weight. Record final weight of dried product. Submit dried solids for lab analysis.
19. Weigh scrubber solutions, filter solids, determine dry weight, wash extensively with DI water, submit solids for Cu and S.

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto
Date of Test: 21-Jul-10

Leach Test Data:

Time		*Reactor*				*Reagents / Feed*				*Comments*
(24 h)	(h) elapsed	Temp ℃	FAT	pH	ORP	ore pulp g	Recycle liquor g	HCl 25% g		
8:20		24.0				302	681	0.0		No ORP/pH reading today
8:50		105.0						0.0		09:00/09:10 Test was stopped because the
8:55		105.5						100.5		main vacuum pump was broken
9:00		106.0						200.5		Some foaming inside the reactor, this have
9:15		105.0						211.7		to be taken into account for design purposes
9:20		105.7						578.3		
10:00		105.8						715.0		
10:10		106.0						715.0		
11:10		105.6						715.0		
12:10		105.4						715.0		
Totals/Avg:		105.6		#DIV/0!	#DIV/0!	302		715	0.0	

dry (g): **0**
Final Pulp weight: **g**
Calculated Acid Addition: **649** **kg/t**

Sampling INFO

Sample #	time	Weight pulp, g	Weight PLS, g	Volume PLS, mL	**emf at room T**	**pH at room T**	SG g/mL	Calc PLS Vol, mL	FAT g/L HCl	Wet res, g	Dry res, g	Colour PLS/RES. Filtration
1	h	86.6	75.6	54			1.405		3.79	6.85	3.6	
2	1 h	94.4	80.8	58			1.403		1.56	8.44	4.3	
3	4 h	1386.3	1243.5	881			1.411		1.45	109.80	74.0	35.85
4	wash		1596.6	1584			1.008		0.00			44
5	S1		2746.2	2506			1.096					
6	S2		2717.9	2466			1.102					
7	S3		2710.6	2460			1.102					

Final Filtration:

Diameter of filtration paper:	150 mm	Clarity of filtrate:	clear
type of paper (Whatman ##):	3	Colour of filtrate:	green
Filtration time:	0.08 hrs	Clarity of wash:	clear
Washing time:	0.08 hrs	Colour of wash:	very light green
Volume of wash:	1584 mL	Colour of residue:	black with metallic specks
Cake thickness:	cm		

Other Notes / Observations

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto
Date of Test: 21-Jul-10

Metallurgical Balance

Product	Amount	g/L	Assay (mg/L, *g/t*)							
min	(ml, g)	FA	Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake	275		56.80		1.480	0.870	0.087	0.38	29.0	29.0
Rec. Liquor	469		218492	218492				2003		
0.1hr PLS	54	4	209000	209000	<0.1	10	<1	1300		
1hr PLS	58	2	208000	208000	<0.1	<2	<1	1300		
4hr PLS	881	1	215000	215000	<0.1	<2	<1	1300		
wash	1584	0	3790							
0.1hr RES	3.6		24.4		4.03	2.57	0.25	0.95		
1hr RES	4.3		24		4.55	2.85	0.29	0.88		
4hr RES	74		23.2		4.80	2.86	0.29	0.93	38.7	21.7
S1 liquor	2506								14020	
S2 liquor	2466								<10	
S3 liquor	2460								<10	
Metal Units, mg			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake			156200	0	4070	2393	239	1045	79750	79750
Rec. Liquor			102557	102557	0	0	0	940	0	0
Total Units IN:			258757	102557	4070	2393	239	1985	79750	79750
0.1hr PLS			11246	11246	0	1	0	70	0	0
1hr PLS			11979	11979	0	0	0	75	0	0
4hr PLS			189477	189477	0	2	0	1146	0	0
wash			6003	0	0	0	0	0	0	0
0.1hr RES			878	0	145	93	9	34	0	0
1hr RES			1030	0	195	122	12	38	0	0
4hr RES			17157	0	3550	2115	214	688	28619	16047
S1 liquor			0	0	0	0	0	0	35128	0
S2 liquor			0	0	0	0	0	0	25	0
S3 liquor			0	0	0	0	0	0	25	0
Total Units OUT:			237770	212702	3890	2332	236	2050	63797	16047
Distribution			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
0.1hr PLS			4.7	5.3	0.0	0.0	0.0	3.4	0.0	0.0
1hr PLS			5.0	5.6	0.0	0.0	0.0	3.7	0.0	0.0
4hr PLS			79.7	89.1	0.0	0.1	0.2	55.9	0.0	0.0
wash			2.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0.1hr RES			0.4	0.0	3.7	4.0	3.8	1.7	0.0	0.0
1hr RES			0.4	0.0	5.0	5.2	5.3	1.8	0.0	0.0
4hr RES			7.2	0.0	91.3	90.7	90.7	33.5	44.9	100.0
S1 liquor			0.0	0.0	0.0	0.0	0.0	0.0	55.1	0.0
S2 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S3 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
SUM:			100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Extraction - Cu-tie										
0 hr			84			-8.5	-5.5	8.2		
1 hr			86			-6.6	-8.4	24.7		
4 hr			87			-1	-3	25	59	77
Head (cal)[1]			49.17	40.05	1.41	0.85	0.09	0.40	23.20	5.84
Head (direct)			56.80	0.00	1.48	0.87	0.09	0.38	29.00	29.00
Accountability (OUT/IN), %			91.9	207.4	95.6	97.5	98.8	103.3	80.0	20.1

weight loss
70.2%
69%

[1]Calculated Head includes both solid and liquid feed streams.

Project No.: 12323-003 **Project Technologist:** Ernesto
Client: Starfield Resources **Date of Test:** 23-Jul-10
Test No.: L4
Objective: 4-hr leach test, 23% solids from Ferguson Lake Core with 600 kg/t HCl - 25% HCl - 105C

Conditions Overview:

Leaching

Reaction: **$Fe_7S_8 + 14HCl \rightarrow 7Fe^{2+} + 7H2S + S + 14Cl^-$**

Temperature:	**105**	℃			Target Grind: k80 = 50 micron
Time:	**4**	hr	(sample at 1,2 and 4 hr)		Actual Grind: **92** 20to400 wet screen
Solids density:	**28%**	solids (before acid addition)			Grind Info: **30 min at 65% in 2kg rod mill (#4) pulped in sea water**
Feed:	**275**	g solids (dry)	sg:	4.6	
Feed cake moisture:	**8.8%**	H_2O			
Wet cake to add:	**302**	g solids (wet)			
Calculated pulp weight:	982	g pulp (before acid)			
Recycle Liquor to add:	**681**	g liquor	sg:	1.5	
HCl dosage:	600	kg/t			
Acid strength:	25%	w/w			
HCl (25%) to add:	**660.0**	g HCl (25%)	sg:	1.13	Calculated volume: 1327 mL
N_2 Blanket Flow:	**~200**	mL/min	(gas blanket - sparging is not required)		

H2S Scrubbing — Condensor: yes - reflux into reactor

Reaction: **2CuCl2 + 2H2S = Cu2S + S + 4HCl**

Scrubbers:	**3**	at:	2.5	L per scrubber
Scrubber 1 composition:	**50**	g/L Cu		
Scrubber 2 composition:	**100**	g/L Cu		
Scrubber 3 composition:	**100**	g/L NaOH		

Sampling Requirements:

	h	solids	liquids
	1h	partial	partial
(just before LOW COMP addition)	2h	partial	partial
	final (4h)	full	full
	final wash		Fe
	scrubber 1	Cu, S	
	scrubber 2	Cu, S	
	scrubber 3		S

Liquor Suite: HCl, Fe, Fe(II), Cu, Ni, Co, Mg
Solids Suite: S=, Fe, Cu, Ni, Co, Mg

H&S **This test produces H2S. Perform test in fumehood with plant scrubber operational. Review MSDS for H2S**
Hot acidic pulps. Review MSDS for HCl and FeCl2
Concentrated acid. The test uses 25% HCl. Use appropriate precautions when handling this and test product pulps
Use face shield, post signs warning of H2S risk in area

Procedure:

1. Review MSDS for concentrated hydrochloric acid (HCl), ferrous chloride ($FeCl_2$) and hydrogen sulphide gas (H_2S)
2. Set up 3 L glass leach vessel with vapour condenser and mixer. Install three scrubbers with spargers, in series connected to vacuum and the top of the condensor). Fill the first two scrubbers with ~100 g/L Cu (as CuCl2) and the third one with ~100 g/L NaOH. Make provisions for acid addition onder the impeller. Provide a N2 blanket on top of the leach liquor. Note: No metal exposed surfaces (prefered use of glass, PP and teflon materials) .
3. Prepare Recycle Liquor by mixing the correct amounts of $FeCl_2.4H_2O$, $MgCl_2.6H_2O$, H_2O and HCl into the leach vessel. Rfefer to separate makeup instruction sheet.
4. Obtain target amount of wet weight feed and add to recycle liquor. Note observations (pH, ORP, T and evolution of H_2S if any)
5. Commence heat up to 90C and start adding 25% HCl acid (use pump or burette). Add acid slowly (target addition of 30 to 60 min - depending on development of heat and foaming). Record observations.
9. Collect ~75 mL aliquot samples at 1, 2 and 4 hr. Filter, submit filtrate, wash 5x with DIW (V=Vfiltrate). and dry solids for chem. analysis.
12. When total time has elapsed stop heating & mixer. Obtain final pulp weight.
14. HOT vacuum filter final leach slurry. Measure density and weight of PLS. Collect sample of PLS to submit to lab. Investgate if PLS dilutions are requried to prevent crystallization.
15. Wash & displace 5X with hot DI water of same volume as PLS. Combine wash solutions. Measure density and weight of wash solution. Collect sample and submit to lab.
16. Obtain wet weight of filtercake. Dry in oven to constant weight. Record final weight of dried product. Submit dried solids for lab analysis.
19. Weigh scrubber solutions, filter solids, determine dry weight, wash extensively with DI water, submit solids for Cu and S.

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto
Date of Test: 23-Jul-10

Leach Test Data:

Time		*Reactor*				*Reagents / Feed*				*Comments*
(24 h)	(h) elapsed	Temp ℃	FAT	pH	ORP	ore pulp g	Recycle liquor g	HCl 25% g		
8:15		24.0				302	681	0.0		A lot of foaming was inside the reactor. It seems its related with the acid addition flowrate (i.e faster the addition bigger the foaming)
8:30	0.00	105.6						0.0		
8:35	0.08	105.1						70.2		
8:40	0.17	106.7						116.6		
8:47	0.28	103.9						179.3		
9:03	0.55	106.9						325.4		
9:15	0.75	105.0						431.1		
9:33	1.05	105.3						595.4		
9:40	1.17	105.8						660.0		
10:40	2.17	106.1						660.0		
11:40	3.17	104.8						660.0		
Totals/Avg:		105.5		#DIV/0!	#DIV/0!	302		660	0.0	

dry (g): **0**
Final Pulp weight: **g**
Calculated Acid Addition: **599** **kg/t**

Sampling INFO

Sample #	time	Weight pulp, g	Weight PLS, g	Volume PLS, mL	**emf at room T**	**pH at room T**	SG g/mL	Calc PLS Vol, mL	FAT g/L HCl	Wet res, g	Dry res, g	Colour PLS/RES. Filtration
1	h	90.6	81.9	59			1.390		2.21	5.2	4.1	
2	1 h	102.2	91.9	66			1.395		0.79	8.5	5.2	
3	4 h	1356.8	1188.9	849			1.400		0.56	114.1	80.6	
4	wash			0			1.000					
5	S1		2699.5	2463			1.096					
6	S2		2639.8	2395			1.102					
7	S3		2711.0	2462			1.101					

Final Filtration:

Diameter of filtration paper:	150	mm	Clarity of filtrate:	clear
type of paper (Whatman ##):	3		Colour of filtrate:	green
Filtration time:	0.08	hrs	Clarity of wash:	clear
Washing time:	0.08	hrs	Colour of wash:	very light green
Volume of wash:	0	mL	Colour of residue:	black with metallic specks
Cake thickness:		cm		

Other Notes / Observations

Project No.: 12323-003 **Project Technologist:** Ernesto
Client: Starfield Resources **Date of Test:** 23-Jul-10

Metallurgical Balance

Product	Amount	g/L	Assay (mg/L, *g/t*)							
min	(ml, g)	FA	Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake	275		56.8		1.480	0.870	0.087	0.38	29.0	29.0
Rec. Liquor	469		218492	218492				2003		
0hr PLS	59	2	204600	204600	<0.1	13.0	<0.8	1100		
1hr PLS	66	1	206200	206200	<0.1	<1.0	<0.8	1200		
4hr PLS	849	1	209800	209800	1.0	<1.0	<0.8	1200		
wash	0	0								
0hr RES	4.1		34.4		3.33					
1hr RES	5.2		32.1		4.45					
4hr RES	81		29.8		4.55	2.61	0.27	0.9	30.0	28.4
S1 liquor	2463								38387	
S2 liquor	2395								14	
S3 liquor	2462								<5	
Metal Units, mg			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake			156200	0	4070	2393	239	1045	79750	79750
Rec. Liquor			102557	102557	0	0	0	940	0	0
Total Units IN:			258757	102557	4070	2393	239	1985	79750	79750
0hr PLS			12060	12060	0	1	0	65	0	0
1hr PLS			13583	13583	0	0	0	79	0	0
4hr PLS			178158	178158	1	1	1	1019	0	0
wash			0	0	0	0	0	0	0	0
0hr RES			1393	0	135	0	0	0	0	0
1hr RES			1660	0	230	0	0	0	0	0
4hr RES			24033	0	3670	2105	218	726	24195	22904
S1 liquor			0	0	0	0	0	0	94549	0
S2 liquor			0	0	0	0	0	0	34	0
S3 liquor			0	0	0	0	0	0	12	0
Total Units OUT:			230886	203800	4035	2107	219	1889	118789	22904
Distribution			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
0hr PLS			5.2	5.9	0.0	0.0	0.0	3.4	0.0	0.0
1hr PLS			5.9	6.7	0.0	0.0	0.0	4.2	0.0	0.0
4hr PLS			77.2	87.4	0.0	0.0	0.3	54.0	0.0	0.0
wash			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0hr RES			0.6	0.0	3.3	0.0	0.0	0.0	0.0	0.0
1hr RES			0.7	0.0	5.7	0.0	0.0	0.0	0.0	0.0
4hr RES			10.4	0.0	90.9	99.9	99.6	38.4	20.4	100.0
S1 liquor			0.0	0.0	0.0	0.0	0.0	0.0	79.6	0.0
S2 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S3 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
SUM:			100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Extraction - Cu-tie										
0 hr			73							
1 hr			81							
4 hr			83			2	-1	23	66	68
Head (cal)[1]			46.67	36.82	1.47	0.77	0.08	0.35	43.20	8.33
Head (direct)			56.80	0.00	1.48	0.87	0.09	0.38	29.00	29.00
Accountability (OUT/IN), %			89.2	198.7	99.1	88.1	91.3	95.2	149.0	28.7

weight loss
67.3%
67.5%

[1]Calculated Head includes both solid and liquid feed streams.

Project No.: 12323-003 **Project Technologist:** Ernesto
Client: Starfield Resources **Date of Test:** 27-Jul-10
Test No.: L5
Objective: 4-hr leach test, 23% solids from Ferguson Lake Core with 552 kg/t HCl - 25% HCl - 105C

Conditions Overview:

Leaching

Reaction: $Fe_7S_8 + 14HCl \rightarrow 7Fe^{2+} + 7H2S + S + 14Cl^-$

Temperature:	**105**	℃			Target Grind: k80 = 50 micron
Time:	**4**	hr	(sample at 1,2 and 4 hr)		Actual Grind: **92** 20to400 wet screen
Solids density:	**28%**	solids (before acid addition)			Grind Info: **30 min at 65% in 2kg rod mill (#4) pulped in sea water**
Feed:	**275**	g solids (dry)	sg:	4.6	
Feed cake moisture:	**8.8%**	H_2O			
Wet cake to add:	**302**	g solids (wet)			
Calculated pulp weight:	982	g pulp (before acid)			
Recycle Liquor to add:	**681**	g liquor	sg:	1.5	
HCl dosage:	700	kg/t			
Acid strength:	25%	w/w			
HCl (25%) to add:	**770.0**	g HCl (25%)	sg:	1.13	Calculated volume: 1424 mL
N_2 Blanket Flow:	**~200**	mL/min	(gas blanket - sparging is not required)		

H2S Scrubbing — Condensor: yes - reflux into reactor

Reaction: **2CuCl2 + 2H2S = Cu2S + S + 4HCl**

Scrubbers:	**3**	at:	2.5	L per scrubber
Scrubber 1 composition:	**50**	g/L Cu		
Scrubber 2 composition:	**100**	g/L Cu		
Scrubber 3 composition:	**100**	g/L NaOH		

Sampling Requirements:

	h	solids	liquids
	1h	partial	partial
(just before LOW COMP addition)	2h	partial	partial
	final (4h)	full	full
	final wash		Fe
	scrubber 1	Cu, S	
	scrubber 2	Cu, S	
	scrubber 3		S

Liquor Suite: HCl, Fe, Fe(II), Cu, Ni, Co, Mg
Solids Suite: S=, Fe, Cu, Ni, Co, Mg

H&S **This test produces H2S. Perform test in fumehood with plant scrubber operational. Review MSDS for H2S**
Hot acidic pulps. Review MSDS for HCl and FeCl2
Concentrated acid. The test uses 25% HCl. Use appropriate precautions when handling this and test product pulps
Use face shield, post signs warning of H2S risk in area

Procedure:

1. Review MSDS for concentrated hydrochloric acid (HCl), ferrous chloride ($FeCl_2$) and hydrogen sulphide gas (H_2S)
2. Set up 3 L glass leach vessel with vapour condenser and mixer. Install three scrubbers with spargers, in series connected to vacuum and the top of the condensor). Fill the first two scrubbers with ~100 g/L Cu (as CuCl2) and the third one with ~100 g/L NaOH. Make provisions for acid addition onder the impeller. Provide a N2 blanket on top of the leach liquor. Note: No metal exposed surfaces (prefered use of glass, PP and teflon materials) .
3. Prepare Recycle Liquor by mixing the correct amounts of $FeCl_2.4H_2O$, $MgCl_2.6H_2O$, H_2O and HCl into the leach vessel. Rfefer to separate makeup instruction sheet.
4. Obtain target amount of wet weight feed and add to recycle liquor. Note observations (pH, ORP, T and evolution of H_2S if any)
5. Commence heat up to 90C and start adding 25% HCl acid (use pump or burette). Add acid slowly (target addition of 30 to 60 min - depending on development of heat and foaming). Record observations.
9. Collect ~75 mL aliquot samples at 1, 2 and 4 hr. Filter, submit filtrate, wash 5x with DIW (V=Vfiltrate). and dry solids for chem. analysis.
12. When total time has elapsed stop heating & mixer. Obtain final pulp weight.
14. HOT vacuum filter final leach slurry. Measure density and weight of PLS. Collect sample of PLS to submit to lab. Investgate if PLS dilutions are requried to prevent crystallization.
15. Wash & displace 5X with hot DI water of same volume as PLS. Combine wash solutions. Measure density and weight of wash solution. Collect sample and submit to lab.
16. Obtain wet weight of filtercake. Dry in oven to constant weight. Record final weight of dried product. Submit dried solids for lab analysis.
19. Weigh scrubber solutions, filter solids, determine dry weight, wash extensively with DI water, submit solids for Cu and S.

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto
Date of Test: 27-Jul-10

Leach Test Data: L-5

Time		*Reactor*				*Reagents / Feed*				*Comments*
(24 h)	(h) elapsed	Temp ℃	FAT	pH	ORP	ore pulp g	Recycle liquor g	HCl 25% g		
7:40		25.0				302	681	0.0		Acid addition took an hour. Lots of foaming
8:14	0.00	105.1						0.0		inside the reactor, same trend as previous
8:19	0.08	106.7						60.4		tests, faster the addition bigger the foaming
8:30	0.27	106.8						148.8		
8:47	0.55	105.0						367.7		
9:00	0.77	105.6						510.9		
9:15	1.02	105.1						770.0		
10:15	2.02	104.1						770.0		
11:15	3.02	103.9						770.0		
Totals/Avg:		105.3		#DIV/0!	#DIV/0!	302		770	0.0	

dry (g): **0**
Final Pulp weight: **g**
Calculated Acid Addition: **699** **kg/t**

Sampling INFO

Sample #	time	Weight pulp, g	Weight PLS, g	Volume PLS, mL	**emf at room T**	**pH at room T**	SG g/mL	Calc PLS Vol, mL	FAT g/L HCl	Wet res, g	Dry res, g	Colour PLS/RES. Filtration
1	h	132.5	117.9	85	249	-0.42	1.387		14.29	7.3	3.7	
2	1 h	109.6	91.0	65	256	-0.38	1.397		9.48	9.7	4.3	
3	4 h	1397.9	1262.1	900	275	-0.34	1.403		7.96	104.5	68.6	
4	wash		848.2	835			1.016					
5	S1		2627.1	2399			1.095					
6	S2		2608.8	2369			1.101					
7	S3		2669.1	2420			1.103					

Final Filtration:

Diameter of filtration paper:	150 mm	Clarity of filtrate:	clear
type of paper (Whatman ##):	3	Colour of filtrate:	green
Filtration time:	0.08 hrs	Clarity of wash:	clear
Washing time:	0.08 hrs	Colour of wash:	very light green
Volume of wash:	0 mL	Colour of residue:	black with metallic specks
Cake thickness:	cm		

Other Notes / Observations

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto
Date of Test: 27-Jul-10

Metallurgical Balance

Product	Amount	g/L	Assay (mg/L, *g/t*)							
min	(ml, g)	FA	Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake	275		56.80		1.480	0.870	0.087	0.38	29.0	29.0
Rec. Liquor	469		218492	218492				2003		
0hr PLS	85	14.3	205200	205200	2	230	<10	1200		
1hr PLS	65	9.5	199100	199100	<1	280	11	1300		
4hr PLS	900	8.0	201600	201600	<1	300	12	1300		
wash	835	0.0	7250							
0hr RES	3.7		18.2		4.56					
1hr RES	4.3		17.2		3.62					
4hr RES	69		16.3		5.27	2.66	0.31	0.96	46.8	15.5
S1 liquor	2399									
S2 liquor	2369								<5	
S3 liquor	2420								<5	
Metal Units, mg			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake			156200	0	4070	2393	239	1045	79750	79750
Rec. Liquor			102557	102557	0	0	0	940	0	0
Total Units IN:			258757	102557	4070	2393	239	1985	79750	79750
0hr PLS			17443	17443	0	20	1	102	0	0
1hr PLS			12974	12974	0	18	1	85	0	0
4hr PLS			181354	181354	1	270	11	1169	0	0
wash			6053	0	0	0	0	0	0	0
0hr RES			673	0	169	0	0	0	0	0
1hr RES			731	0	154	0	0	0	0	0
4hr RES			11182	0	3615	1825	213	659	32106	10633
S1 liquor			0	0	0	0	0	0	0	0
S2 liquor			0	0	0	0	0	0	12	0
S3 liquor			0	0	0	0	0	0	12	0
Total Units OUT:			230409	211770	3939	2132	225	2015	32130	10633
Distribution			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
0hr PLS			7.6	8.2	0.0	0.9	0.4	5.1	0.0	0.0
1hr PLS			5.6	6.1	0.0	0.9	0.3	4.2	0.0	0.0
4hr PLS			78.7	85.6	0.0	12.7	4.8	58.0	0.0	0.0
wash			2.6	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0hr RES			0.3	0.0	4.3	0.0	0.0	0.0	0.0	0.0
1hr RES			0.3	0.0	3.9	0.0	0.0	0.0	0.0	0.0
4hr RES			4.9	0.0	91.8	85.6	94.5	32.7	99.9	100.0
S1 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S2 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S3 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
SUM:			100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Extraction - Cu-tie										
0 hr			90							
1 hr			88							
4 hr			92			14	0	29	55	85
Head (cal)[1]			46.5	39.7	1.4	0.8	0.1	0.4		
Head (direct)			56.8	0.0	1.5	0.9	0.1	0.4	29.0	29.0
Accountability (OUT/IN), %			89.0	206.5	96.8	89.1	94.1	101.5	40.3	13.3

weight loss
72.2%
71.9%

[1]Calculated Head includes both solid and liquid feed streams.

Project No.: 12323-003 **Project Technologist:** Ernesto, Charlotte
Client: Starfield Resources **Date of Test:** 15-Sep-10
Test No.: L6
Objective: 2-hr leach test, 28% solids from flotation froth from Ferguson Lake Core with 650 kg/t HCl - 25% HCl - 105C

Conditions Overview:

Leaching

Reaction: $Fe_7S_8 + 14HCl \rightarrow 7Fe^{2+} + 7H2S + S + 14Cl^-$

Temperature:	**105**	℃			Target Grind: k80 = 50 micron
Time:	**2**	hr	(sample at 1,2 and 4 hr)		Actual Grind: 20to400 wet screen
Solids density:	**28%**	solids (before acid addition)			Grind Info: **30 min at 65% in 2kg rod mill (#4) pulped in sea water**
Feed:	**275**	g solids (dry)	sg:	4.6	
Feed cake moisture:	**0.0%**	H_2O			
Wet cake to add:	**275**	g solids (wet)			
Calculated pulp weight:	982	g pulp (before acid)			
Recycle Liquor to add:	**707**	g liquor	sg:	1.5	
HCl dosage:	650	kg/t			
Acid strength:	25%	w/w			
HCl (25%) to add:	**715.0**	g HCl (25%)	sg:	1.13	Calculated volume: 1402 mL
N_2 Blanket Flow:	**~200**	mL/min	(gas blanket - sparging is not required)		

H2S Scrubbing — Condensor: yes - reflux into reactor

Reaction: **2CuCl2 + 2H2S = Cu2S + S + 4HCl**

Scrubbers:	**3**	at:	2.5	L per scrubber
Scrubber 1 composition:	**100**	g/L NaOH		
Scrubber 2 composition:	**100**	g/L NaOH		
Scrubber 3 composition:	**100**	g/L NaOH		

Sampling Requirements:

	h	solids	liquids
	1h	partial	partial
(just before LOW COMP addition)	2h	partial	partial
	final (4h)	full	full
	final wash		Fe
	scrubber 1		S
	scrubber 2		S
	scrubber 3		S

Liquor Suite: HCl, Fe, Fe(II), Cu, Ni, Co, Mg
Solids Suite: S=, Fe, Cu, Ni, Co, Mg

H&S **This test produces H2S. Perform test in fumehood with plant scrubber operational. Review MSDS for H2S**
Hot acidic pulps. Review MSDS for HCl and FeCl2
Concentrated acid. The test uses 25% HCl. Use appropriate precautions when handling this and test product pulps
Use face shield, post signs warning of H2S risk in area

Procedure:

1. Review MSDS for concentrated hydrochloric acid (HCl), ferrous chloride ($FeCl_2$) and hydrogen sulphide gas (H_2S)
2. Set up 3 L glass leach vessel with vapour condenser and mixer. Install three scrubbers with spargers, in series connected to vacuum and the top of the condensor). Fill the 3 scrubbers with ~100 g/L NaOH. Make provisions for acid addition under the impeller. Provide a N2 blanket on top of the leach liquor. Note: No metal exposed surfaces (prefered use of glass, PP and teflon materials) .
3. Prepare Recycle Liquor by mixing the correct amounts of $FeCl_2.4H_2O$, $MgCl_2.6H_2O$, H_2O into the leach vessel. Refer to separate makeup instruction sheet.
4. Obtain target amount of wet weight feed and add to recycle liquor. Note observations (pH, ORP, T and evolution of H_2S if any)
5. Commence heat up to 105C and start adding 25% HCl acid (use pump or burette). Add acid slowly (target addition of 30 to 60 min - depending on development of heat and foaming). Record observations.
9. Collect ~75 mL aliquot samples when acid addition is finished, after 1 hour and after 2 hours. Filter, titrate filtrate for free acid and submit to analitical lab, wash 5x with DIW (V=Vfiltrate) and dry solids for chemical analysis.
12. When total time has elapsed stop heating & mixer. Obtain final pulp weight.
14. HOT vacuum filter final leach slurry. Measure density and weight of PLS. Collect sample of PLS to submit to lab. Investgate if PLS dilutions are requried to prevent crystallization.
15. Wash & displace 5X with hot DI water of same volume as PLS. Combine wash solutions. Measure density and weight of wash solution. Collect sample and submit to lab.
16. Obtain wet weight of filtercake. Dry in oven to constant weight. Record final weight of dried product. Submit dried solids for lab analysis.
19. Weigh scrubber solutions, filter solids, determine dry weight, wash extensively with DI water, submit solids for Cu and S.

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto, Charlotte
Date of Test: 15-Sep-10

Leach Test Data: L-5

Time		*Reactor*				*Reagents / Feed*				*Comments*
(24 h)	(h) elapsed	Temp ℃	FAT	pH	ORP	ore pulp g	Recycle liquor g	HCl 25% g		
9:00	0.00	22.0				275.00	707			Heating to 105 C
9:55	0.92	103.4						0.0		Adding acid
10:25	1.42	104.6						230.0		
11:00	2.00	104.3						715.0		Finished acid addition. Zero hour sample.
12:00	3.00	105.4								1 hour sample
13:00	4.00	105.0								Heat off
13:30	4.50	90.0								Agitator off. Final sample collected.
Totals/Avg:		102.1		#DIV/0!	#DIV/0!	275	707	715	0.0	

dry (g): **275**
Final Pulp weight: **1358.0 g**
Calculated Acid Addition: **650 kg/t**

Sampling INFO

Sample #	time	Weight pulp, g	Weight PLS, g	Volume PLS, mL	**emf at room T**	**pH at room T**	SG g/mL	Calc PLS Vol, mL	FAT g/L HCl	Wet res, g	Dry res, g	Colour PLS/RES. Filtration
1	h	90.9	82.8	59	235	-0.79	1.409	58.8	5.57	4.6	2.7	green/black
2	1 h	101.3	91.9	65	281	-0.87	1.419	64.8	1.59	6.1	3.5	green/black
3	2 h	1358.0	1216.6	854	270	-0.81	1.425	853.8	0.94	88.9	59.3	green/black
4	wash	1124.0	1124.0	1104	316	3.14	1.018	1104.1				very light green
5	S1	122.9	122.9	112	-695	12.44	1.097	112.0				grey
6	S2	101.6	101.6	92	-277	12.53	1.106	91.8				clear
7	S3	103.0	103.0	93	-208	12.53	1.107	93.1				clear

Final Filtration:

Diameter of filtration paper:	150	mm
type of paper (Whatman ##):	3	
Filtration time:	0.08	hrs
Washing time:	0.08	hrs
Volume of wash:	1104	mL
Cake thickness:	1.0	cm

Clarity of filtrate:	clear
Colour of filtrate:	green
Clarity of wash:	clear
Colour of wash:	very light green
Colour of residue:	black with metallic specks

Other Notes / Observations

Maintained 53% flow of nitrogen to blanket (corresponds to 100 mL/min AIR).
RPM 300 while heating to 105 C, then increased to 360 for remainder of test (helped with foaming).
Foaming was observed more at the start of acid addition than near the end.
At 10:10, 105 C reached, foaming started; scrubber #1 turned yellow, then orange then grey.
Ran agitator for 30 minutes after 2 hr time was reached to dissipate heat.

Project No.: 12323-003
Client: Starfield Resources

Project Technologist: Ernesto, Charlotte
Date of Test: 15-Sep-10

Metallurgical Balance

Product	Amount	g/L	Assay (mg/L, *g/t*)							
min	(ml, g)	FA	Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake	275		56.80		1.7	1.05	0.120	0.09	34.5	34.5
Rec. Liquor	469		218500					2003		
0hr PLS	59	5.6	232000	207500	1.38	15.9	<0.05	953		
1hr PLS	65	1.6	242000	217900	1.08	0.9	<0.05	981		
2hr PLS	854	0.9	242000	229900	1.16	2.7	<0.05	992		
wash	1104	0.0	10200							
0hr RES	2.7		33.8		7.02	4.53	0.44	0.15	30.6	
1hr RES	3.5		29.4		7.80	4.66	0.46	0.16	36.8	
2hr RES	59		27.6		7.92	4.30	0.44	0.18	41.4	27.4
S1 liquor	112								25000	
S2 liquor	92								<5	
S3 liquor	93								<5	
Metal Units, mg			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
Fer Lake			156200	0	4758	2888	330	234	94875	94875
Rec. Liquor			102561	0	0	0	0	940	0	0
Total Units IN:			258761	0	4758	2888	330	1174	94875	94875
0hr PLS			13634	12195	0	1	0	56	0	0
1hr PLS			15671	14110	0	0	0	64	0	0
2hr PLS			206623	196292	1	2	0	847	0	0
wash			11262	0	0	0	0	0	0	0
0hr RES			913	0	190	122	12	4	826	0
1hr RES			1029	0	273	163	16	6	1288	0
2hr RES			16358	0	4694	2548	261	107	24536	16239
S1 liquor			0	0	0	0	0	0	2801	0
S2 liquor			0	0	0	0	0	0	0	0
S3 liquor			0	0	0	0	0	0	0	0
Total Units OUT:			265489	222597	5158	2837	289	1083	29452	16239
Distribution			Fe	Fe(II)	Cu	Ni	Co	Mg	S	S
0hr PLS			5.1	5.5	0.0	0.0	0.0	5.2	0.0	0.0
1hr PLS			5.9	6.3	0.0	0.0	0.0	5.9	0.0	0.0
2hr PLS			77.8	88.2	0.0	0.1	0.0	78.2	0.0	0.0
wash			4.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0
0hr RES			0.3	0.0	3.7	4.3	4.1	0.4	2.8	0.0
1hr RES			0.4	0.0	5.3	5.7	5.6	0.5	4.4	0.0
2hr RES			6.2	0.0	91.0	89.8	90.3	9.9	83.3	100.0
S1 liquor			0.0	0.0	0.0	0.0	0.0	0.0	9.5	0.0
S2 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
S3 liquor			0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
SUM:			100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Extraction - Cu-tie										weight loss
0 hr			85	#DIV/0!	0.0	-6.3	9.6	56.5	78.1	100.0	76.2%
1 hr			89	#DIV/0!	0.0	1.6	15.0	58.3	76.3	100.0	78.2%
4 hr			89	#DIV/0!	0	11	20	54	74	83	
Head (cal)[1]			59.2	80.9	1.9	1.0	0.1	0.1		
Head (direct)			56.8	0.0	1.7	1.1	0.1	0.1	34.5	34.5
Accountability (OUT/IN), %			102.6	#DIV/0!	108.4	98.3	87.5	92.2	31.0	17.1

[1]Calculated Head includes both solid and liquid feed streams.

Appendix B – Pilot Plant Data Sheets

Appendix B - Pilot plant data

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

PP1 LOGSHEET

		Feed								Pre-heater		R1				R2				R3		R4		Filtration			Scrubb
Date	Time	Feed tank weight kg	calc flow g/min	R1 Feed Flowrate OFF min	Feed Flowrate mL/min	Feed Weight g	Feed Volume mL	Feed Density g/mL	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Acid weight kg	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Acid weight g	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Filtrate flowrate mL/min	Drum Volume Gallons	Filter cake wet weight g	Fresh Caustic mL/min
TARGET					**20**				**100**	**105**	**100**	**105**	**100**		**13.0**	**105**	**100**		**13.0**	**105**	**100**	**105**	**100**				**38**
12-Oct	0:00							-							-				-								
12-Oct	0:30							-							-				-								
12-Oct	1:00							-							-				-								
12-Oct	1:30							-							-				-								
12-Oct	2:00							-							-				-								
12-Oct	2:30							-							-				-								
12-Oct	3:00							-							-				-								
12-Oct	3:30							-							-				-								
12-Oct	4:00							-							-				-								
12-Oct	4:30							-							-				-								
12-Oct	5:00							-							-				-								
12-Oct	5:30							-							-				-								
12-Oct	6:00							-							-				-								
12-Oct	6:30							-							-				-								
12-Oct	7:00							-							-				-								
12-Oct	7:30							-							-				-								
12-Oct	8:00							-							-				-								
12-Oct	8:30							-							-				-								
12-Oct	9:00							-							-				-								
12-Oct	9:30							-							-				-								
12-Oct	10:00							-							-				-								
12-Oct	10:30							-							-				-								
12-Oct	11:00	57.3			20			-	300	103.0	100	105.4	100	9.68	-	105.7	100	10.43	-	104.2	100	104.2	100				
12-Oct	11:30	56.1	-					-	300	104.4	100	106.2	100	9.51	5.6	104.0	100	10.43	-	105.4	100	106.0	100				38
12-Oct	12:00							-							-				-								
12-Oct	12:30	54.4	27.8		22	47.4	29	1.634	300	105.9	100	103.8	100	8.88	10.6	105.2	100	10.06	6.2	104.8	100	104.6	100	40			38
12-Oct	13:00							-							-				-								
12-Oct	13:30	52.3	34.8		22	39.0	22	1.773	300	106.0	100	105.9	100	8.18	11.7	105.5	100	9.28	13.0	104.2	100	105.6	100	47			
12-Oct	14:00							-							-				-								
12-Oct	14:30	50.2	35.7		22	34.0	22	1.545	300	105.4	100	104.0	100	7.58	9.9	105.6	100	8.61	11.2	104.8	100	104.9	100	0			40
12-Oct	15:00							-							-				-								
12-Oct	15:30	46.9	54.8		20	33.7	20	1.685	200	103.6	100	105.4	100	6.84	12.4	104.5	100	8.61	-	104.3	105	100.0	100	50			
12-Oct	16:00							-							-				-								
12-Oct	16:30	44.8	34.7		20	65.0	39	1.667	200	104.6	100	103.8	100	6.22	10.3	104.7	100	8.61	-	105.5	106	100.0	100	50			38
12-Oct	17:00							-							-				-								
12-Oct	17:30	43.9	16.2		20	68.6	40	1.715	200	103.5	100	103.7	100	5.49	12.1	103.2	100	8.47	2.3	104.5	100	105.7	100	100			
12-Oct	18:00							-							-				-								
12-Oct	18:30	41.4	41.2		20	34.7	20	1.735	200	103.9	100	105.7	100	4.81	11.4	104.9	100	7.72	12.5	104.7	100	104.8	100	25			39
12-Oct	19:00							-							-				-								
12-Oct	19:30	39.3	34.2		20	34.9	20	1.745	300	103.8	100	104.8	100	4.11	11.7	104.8	100	6.95	12.8	105.8	100	105.2	100	20			
12-Oct	20:00							-							-				-								
12-Oct	20:30	37.2	36.3		20	33.0	20	1.650	300	105.5	100	104.4	100	3.42	11.4	103.5	100	6.19	12.7	105.6	100	105.9	100	104			40
12-Oct	21:00							-							-				-								
12-Oct	21:30	34.5	43.8		20	31.5	20	1.575	300	103.4	100	104.0	100	2.71	11.9	104.7	100	5.41	13.0	105.0	100	105.0	100	50	0.50		
12-Oct	22:00							-							-				-								
12-Oct	22:30	32.4	36.0		20	65.4	39	1.677	300	104.6	100	105.9	100	2.57	2.3	104.9	100	4.63	13.0	104.2	100	106.1	100	40	0.50		39
12-Oct	23:00							-							-				-								
12-Oct	23:30	30.4	33.5		20			-	300	104.6	150	104.5	150	11.92	-	105.6	150	3.78	14.2	105.1	150	106.0	150		0.50		39
13-Oct	0:00							-							-				-								
13-Oct	0:30	56.7	-		20	65.7	40	1.643	300	105.0	150	104.5	150	11.30	10.3	105.5	150	10.65	-	106.0	150	105.3	150	27	0.50		39
13-Oct	1:00							-							-				-								
13-Oct	1:30	54.7	33.0		20			-	300	104.3	150	104.0	150	10.59	11.8	105.6	150	9.86	13.2	103.9	150	105.2	150	21	0.00		39
13-Oct	2:00							-							-				-								
13-Oct	2:30	52.7	33.5		20	60.4	40	1.510	300d200	104.5	150d100	105.0	150d100	9.87	12.1	105.1	150d100	9.07	13.2	103.9	150d100	104.0	150d100	33	0.50		38
13-Oct	3:00							-							-				-								
13-Oct	3:30	50.9	29.8		20			-	200	104.3	100	105.3	100	9.17	11.6	105.1	100	8.31	12.7	104.3	100	105.7	100	18	0.00		38
13-Oct	4:00							-							-				-								
13-Oct	4:30	48.9	33.0		19	57.0	38	1.500	200	103.8	100	104.3	100	8.47	11.7	105.2	100	7.54	12.8	105.0	100	106.3	100	52	0.00		38
13-Oct	5:00							-							-				-								
13-Oct	5:30	47.2	29.3		20			-	200	104.5	100	103.7	100	7.76	11.8	106.2	100	7.01	8.8	106.1	100	105.8	100	31	0.00		38
13-Oct	6:00							-							-				-								
13-Oct	6:30	45.4	29.2		20	55.2	40	1.380	200	105.1	100	103.5	100	7.06	11.7	106.1	100	7.01	-	105.3	100	104.3	100	36	0.00		38
13-Oct	7:00							-							-				-								
13-Oct	7:30	43.7	28.5		18	26.7	18	1.483	300	104.1	100	104.8	100	6.34	12.1	106.2	100	7.01	-	106.3	100	104.2	100	34			37
13-Oct	8:00							-							-				-								
13-Oct	8:30	42.0	27.8		21	27.9	21	1.329	300	104.9	100	103.3	100	5.62	11.9	103.9	100	7.01	-	105.8	100	106.6	100	34			37
13-Oct	9:00							-							-				-								
13-Oct	9:30	40.8	20.0		22			-	300	104.6	100	103.8	100	4.92	11.7	104.8	100	7.01	-	106.3	100	106.5	100	34			39
13-Oct	10:00							-							-				-								
13-Oct	10:30	39.4	24.2		20			-	300	106.1	100	103.6	100	4.21	11.8	105.7	100	7.01	-	104.3	100	106.4	100	24			39
13-Oct	11:00							-							-				-								
13-Oct	11:30	37.8	26.3		20			-	300	105.8	100	103.8	100	3.56	11.0	106.1	100	7.01	-	106.2	100	105.1	100	21			38
13-Oct	12:00							-							-				-								
13-Oct	12:30	36.6	20.8		18			-	300	105.9	100	105.3	100	11.53	-	105.5	100	7.01	-	106.0	100	106.3	100	22			38
13-Oct	13:00							-							-				-								
13-Oct	13:30	34.9	28.3		18			-	300	103.7	100	103.5	100	10.63	15.0	105.3	100	7.01	-	104.5	100	106.0	100	20			40
13-Oct	14:00							-							-				-								
13-Oct	14:30	33.1	30.0		22	34.0	22	1.545	300	103.7	100	103.5	100	9.66	16.2	106.3	100	7.01	-	105.1	100	106.3	100	20			40

		Feed								Pre-heater		R1				R2				R3		R4		Filtration			Scrubb
Date	Time	Feed tank weight kg	calc flow g/min	R1 Feed Flowrate OFF min	Feed Flowrate mL/min	Feed Weight g	Feed Volume mL	Feed Density g/mL	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Acid weight kg	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Acid weight g	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Filtrate flowrate mL/min	Drum Volume Gallons	Filter cake wet weight g	Fresh Caustic mL/min
TARGET					**20**				**100**	**105**	**100**	**105**	**100**		**13.0**	**105**	**100**		**13.0**	**105**	**100**	**105**	**100**				**38**
13-Oct	15:00							-							-				-								
13-Oct	15:30	30.6	40.3		21	68.1	41	1.661	300	104.8	100	103.1	100	8.76	15.0	106.0	100	7.01	-	106.2	100	106.1	100	27	1.00		39
13-Oct	16:00							-							-				-								
13-Oct	16:30	28.8	31.2		20	67.2	39	1.723	300	105.6	100	103.5	100	7.85	15.3	105.2	100	7.01	-	105.8	100	105.1	100	25	1.50		40
13-Oct	17:00							-							-				-								
13-Oct	17:30	26.3	41.8		22	63.0	44	1.432	300	104.1	100	104.3	100	6.95	15.0	104.0	100	7.01	-	105.7	100	105.7	100	50	1.50		40
13-Oct	18:00							-							-				-								
13-Oct	18:30	23.6	44.8		20	69.4	40	1.735	300	104.6	100	103.8	100	6.00	15.7	106.5	100	7.01	-	105.5	100	106.6	100	18	2.00		39
13-Oct	19:00							-							-				-								
13-Oct	19:30	71.2	-		21	70.1	41	1.710	300	103.5	100	105.2	100	4.88	18.7	105.2	100	7.01	-	104.5	100	105.3	100	10	2.00		39
13-Oct	20:00							-							-				-								
13-Oct	20:30	68.8	39.3		20	68.8	40	1.720	300	103.8	100	103.3	100	3.87	17.0	103.8	100	7.01	-	105.5	100	104.6	100	15	2.00		40
13-Oct	21:00							-							-				-								
13-Oct	21:30	66.5	38.7		20	68.0	40	1.700	300	105.9	100	103.3	100	10.27	-	104.0	100	7.01	-	106.1	100	106.5	100	85			40
13-Oct	22:00							-							-				-								
13-Oct	22:30	63.8	45.2		20	68.5	40	1.713	300	103.1	100	104.6	100	9.04	20.5	104.0	100	7.01	-	106.5	100	105.7	100				40
13-Oct	23:00							-							-				-								
13-Oct	23:30	61.9	31.7		20	68.4	40	1.710	300	104.4	100	105.2	100	7.84	19.9	106.5	100	7.02	-	104.2	100	104.5	100	25	1.00		39
14-Oct	0:00							-							-				-								
14-Oct	0:30	59.4	42.6		20	68.3	40	1.708	300	103.9	100	104.2	100	6.46	23.0	106.4	100	7.02	-	105.0	100	105.3	100	11	1.50		38
14-Oct	1:00							-							-				-								
14-Oct	1:30	57.2	35.9		20	68.8	40	1.720	300	106.1	100	103.1	100	5.04	23.7	104.9	100	7.02	-	106.3	100	106.7	100	30	1.50		
14-Oct	2:00							-							-				-								
14-Oct	2:30	54.6	42.8		20	68.7	40	1.718	300	106.8	100	104.3	100	3.59	24.2	104.9	100	7.02	-	104.0	100	106.2	100	94	1.00		38
14-Oct	3:00							-							-				-								
14-Oct	3:30	52.6	34.5		20	67.1	39	1.721	300	105.4	100	102.5	100	11.86	-	104.1	100	7.02	-	105.8	100	106.6	100	60	1.00		
14-Oct	4:00							-							-				-								
14-Oct	4:30	50.2	40.2		19	65.8	38	1.732	300	104.3	100	101.9	100	10.23	27.2	104.2	100	7.02	-	105.2	100	106.8	100	47	1.50		33
14-Oct	5:00							-							-				-								
14-Oct	5:30	47.8	38.5		20	67.1	40	1.678	300	105.8	100	99.3	100	8.41	30.4	103.8	100	7.02	-	105.5	100	106.2	100	32	1.00		
14-Oct	6:00							-							-				-								
14-Oct	6:30	45.4	41.5		20	67.7	40	1.693	300	105.7	100	102.7	100	6.52	31.5	104.6	100	7.02	-	105.3	100	105.7	100	42	1.00		30
14-Oct	7:00							-							-				-								
14-Oct	7:30	42.8	43.3		23	39.4	23	1.713	300	105.5	100	103.3	100	4.64	31.2	104.3	100	7.03	-	106.0	100	106.6	100	38	3.50		
14-Oct	8:00							-							-				-								
14-Oct	8:30	40.4	40.0		20	31.8	20	1.590	300	104.8	100	98.1	100	9.85	-	104.8	100	7.03	-	104.8	100	104.9	100	10	3.50		47
14-Oct	9:00							-							-				-								
14-Oct	9:30	38.7	27.8		22	38.0	22	1.727	300	105.9	100	104.5	100	8.67	19.7	106.1	100	7.03	-	104.9	100	106.5	100	34	3.00		
14-Oct	10:00							-							-				-								
14-Oct	10:30	36.7	33.0					-	300	104.2	100	105.4	100	7.25	23.5	106.1	100	7.03	-	106.5	100	105.8	100	38	5.00		44
14-Oct	11:00							-							-				-								
14-Oct	11:30	34.9	30.0		22	40.3	22	1.832	300	105.7	100	103.2	100	5.89	22.7	105.8	100	7.03	-	104.2	100	104.2	100	40	5.50		
14-Oct	12:00							-							-				-								
14-Oct	12:30	33.1	29.7		22	33.7	21	1.644	300	103.9	100	104.9	100	4.66	20.6	103.9	100	7.03	-	106.2	100	105.8	100	34	6.00		44
14-Oct	13:00							-							-				-								
14-Oct	13:30	68.9	-	30				-	300	107.3	100	103.6	100	3.24	23.6	105.4	100	7.03	-	103.8	100	104.5	100		6.00		
14-Oct	14:00							-							-				-								
14-Oct	14:30	68.2	10.8		25	41.1	25	1.644	300	104.5	100	103.2	100	10.52	-	104.3	100	7.03	-	104.0	100	107.8	100	45	6.00		44
14-Oct	15:00							-							-				-								
14-Oct	15:30	65.6	42.8		24	83.3	47	1.772	300	103.5	100	102.3	100	9.21	21.7	104.6	100	7.03	-	106.2	100	105.7	100	70	7.50		
14-Oct	16:00							-							-				-								
14-Oct	16:30	63.4	37.3		20	70.5	39	1.808	300	105.2	100	103.9	100	7.99	20.4	106.1	100	7.03	-	104.3	100	104.4	100	25	7.50		45
14-Oct	17:00							-							-				-								
14-Oct	17:30	61.4	34.2		20	70.7	40	1.768	300	103.6	100	104.4	100	6.69	21.6	105.8	100	7.03	-	104.4	100	105.0	100	28	7.50		47
14-Oct	18:00							-							-				-								
14-Oct	18:30	59.2	35.3		20	70.9	40	1.773	300	105.8	100	103.3	100	5.35	22.3	106.4	100	7.03	-	106.6	100	106.5	100	28	7.50		46
14-Oct	19:00							-							-				-								
14-Oct	19:30	57.0	37.2		20	69.8	39	1.790	300	103.5	100	103.1	100	3.94	23.6	105.6	100	7.03	-	104.0	100	106.6	100	15	7.50		
14-Oct	20:00							-							-				-								
14-Oct	20:30	54.6	40.0		20	108.7	60	1.812	300	103.6	100	103.1	100	10.21	-	103.8	100	7.03	-	105.7	100	106.5	100	20	7.50		94
14-Oct	21:00							-							-				-								
14-Oct	21:30	52.3	38.0		20	71.6	40	1.790	300	104.3	100	103.8	100	8.78	23.8	104.6	100	7.03	-	106.2	100	106.6	100	30	9.00		96
14-Oct	22:00							-							-				-								
14-Oct	22:30	50.4	32.8		20	70.7	40	1.768	300	103.4	100	98.8	100	7.55	20.4	104.2	100	7.03	-	104.1	100	104.5	100	25	9.50		
14-Oct	23:00							-							-				-								
14-Oct	23:30	48.3	34.2					-	300	105.0	100	103.7	100	6.08	24.4	105.6	100	7.03	-	104.3	100	104.9	100	10	8.50		
15-Oct	0:00							-							-				-								
15-Oct	0:30	47.5	13.3	30	19	65.6	37	1.773	300	104.0	100	104.1	100	5.65	7.2	106.0	100	7.03	-	105.9	100	104.6	100		8.50		95
15-Oct	1:00							-							-				-								
15-Oct	1:30	45.1	39.8		21	72.2	41	1.761	300	104.5	100	101.8	100	4.90	12.5	103.9	100	7.03	-	104.3	100	106.1	100	32	8.50		
15-Oct	2:00							-							-				-								
15-Oct	2:30	42.8	39.2		21	73.2	41	1.785	300	106.3	100	102.8	100	3.41	24.9	104.0	100	7.03	-	105.7	100	105.9	100	20	8.50		93
15-Oct	3:00							-							-				-								
15-Oct	3:30	40.8	32.3		20	69.7	39	1.787	300	106.5	100	102.5	100	8.79	-	104.2	100	7.03	-	106.4	100	106.2	100	35	8.50		92
15-Oct	4:00							-							-				-								
15-Oct	4:30	38.2	43.3		20	72.7	40	1.818	300	105.8	100	101.9	100	7.23	25.9	105.2	100	7.03	-	104.4	100	106.3	100	28	8.50		92
15-Oct	5:00							-							-				-								
15-Oct	5:30	35.0	53.3		20	70.2	40	1.755	300	104.3	100	101.7	100	5.66	26.2	105.7	100	7.03	-	106.5	100	105.1	100	34	9.00		92

		Feed								Pre-heater		R1				R2				R3		R4		Filtration			Scrubb
Date	Time	Feed tank weight kg	calc flow g/min	R1 Feed Flowrate OFF min	Feed Flowrate mL/min	Feed Weight g	Feed Volume mL	Feed Density g/mL	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Acid weight kg	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Acid weight g	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Filtrate flowrate mL/min	Drum Volume Gallons	Filter cake wet weight g	Fresh Caustic mL/min
TARGET					**20**				**100**	**105**	**100**	**105**	**100**		**13.0**	**105**	**100**		**13.0**	**105**	**100**	**105**	**100**				**38**
15-Oct	6:00							-							-				-								
15-Oct	6:30	33.8	19.7		20	72.3	40	1.808	300	104.7	100	101.3	100	4.11	25.9	104.5	100	7.04	-	105.4	100	104.6	100	42	9.00		92
15-Oct	7:00							-							-				-								
15-Oct	7:30	31.4	41.3		20	33.2	20	1.660	300	103.5	100	101.2	100	11.91	-	104.1	100	7.04	-	104.7	100	104.5	100	41	9.00		
15-Oct	8:00							-							-				-								
15-Oct	8:30		-	15	23	51.7	37	1.397	300	105.7	100	103.7	100	11.30	10.1	105.6	100	7.04	-	103.9	100	104.1	100	37	10.00		90
15-Oct	9:00							-							-				-								
15-Oct	9:30	72.7	-		23	39.5	23	1.717	300	103.9	100	100.1	100	9.95	22.6	104.6	100	7.04	-	104.6	100	104.9	100	33	11.50		
15-Oct	10:00							-							-				-								
15-Oct	10:30	70.5	36.2		23	39.6	23	1.722	300	104.6	100	100.7	100	8.30	27.5	105.2	100	7.04	-	104.1	100	105.8	100	27	12.50		100
15-Oct	11:00							-							-				-								
15-Oct	11:30	68.0	41.8		21	35.5	21	1.732	300	103.3	100	102.0	100	6.74	25.8	105.5	100	7.04	-	106.2	100	105.6	100	32	13.00		
15-Oct	12:00							-							-				-								
15-Oct	12:30	66.6	23.5		21	37.9	21	1.805	300	105.2	100	103.4	100	5.22	25.4	105.0	100	7.04	-	105.2	100	104.2	100	40	13.50		95
15-Oct	13:00							-							-				-								
15-Oct	13:30	64.3	38.8		21	36.9	21	1.757	300	105.6	100	103.1	100	3.72	25.0	105.7	100	7.04	-	104.2	100	106.0	100	35	14.50		
15-Oct	14:00							-							-				-								100
15-Oct	14:30	62.2	34.2		23	43.0	23	1.870	300	103.4	100	102.7	100	10.30	-	105.9	100	7.04	-	106.3	100	104.8	100	35	15.50		
15-Oct	15:00							-							-				-								
15-Oct	15:30	60.2	33.3		20	71.5	40	1.788	300	104.6	100	103.9	100	8.56	29.1	104.1	100	7.04	-	106.1	100	104.9	100	20	16.00		55
15-Oct	16:00							-							-				-								
15-Oct	16:30	58.1	35.0		20	71.2	40	1.780	300	104.8	100	104.5	100	7.00	25.9	103.9	100	7.04	-	105.6	100	106.3	100	28	16.50		54
15-Oct	17:00							-							-				-								
15-Oct	17:30	55.6	42.0		20	67.2	39	1.723	300	106.0	100	103.5	100	5.44	26.0	105.1	100	7.04	-	106.3	100	105.0	100	18	17.00		
15-Oct	18:00							-							-				-								
15-Oct	18:30	53.4	37.3		19	62.7	38	1.650	300	104.7	100	103.2	100	3.88	26.1	105.4	100	7.04	-	104.9	100	106.5	100	30	17.50		54
15-Oct	19:00							-							-				-								
15-Oct	19:30	51.3	34.0		21	70.2	41	1.712	300	105.7	100	104.6	100	2.33	25.7	106.2	100	7.04	-	104.1	100	104.9	100	42	18.00		
15-Oct	20:00							-							-				-								
15-Oct	20:30	49.4	31.5		20	65.7	40	1.643	300	105.3	100	105.0	100	8.93	-	105.7	100	7.04	-	105.4	100	104.5	100	22	18.50		56
15-Oct	21:00							-							-				-								
15-Oct	21:30	47.3	35.2		20	67.5	40	1.688	300	105.6	100	103.8	100	7.37	25.9	104.4	100	7.04	-	105.5	100	105.5	100	30	18.50		
15-Oct	22:00							-							-				-								
15-Oct	22:30	45.2	35.8		20	66.8	39	1.713	300	104.8	100	104.8	100	5.82	25.8	105.0	100	7.04	-	106.2	100	106.7	100	28	19.50		54
15-Oct	23:00							-							-				-								
15-Oct	23:30	42.1	51.0		21	75.1	41	1.832	300	104.2	100	104.7	100	4.26	26.1	106.1	100	7.04	-	104.2	100	106.6	100	37	20.50		53
16-Oct	0:00							-							-				-								
16-Oct	0:30	40.0	35.8		21	74.0	41	1.805	300	105.7	100	105.6	100	2.68	26.3	105.0	100	7.04	-	106.1	100	105.2	100	38	22.00		53
16-Oct	1:00							-							-				-								
16-Oct	1:30	36.9	50.3		26	95.6	52	1.838	300	103.3	100	104.7	100	5.61	-	104.9	100		-	104.9	100	105.5	100	39	21.50		54
16-Oct	2:00	35.7	40.0					-	300	104.9	100	103.8	100	4.84	26.0	105.0	100		-	104.2	100	106.3	100				54
16-Oct	2:30		-					-							-				-								
16-Oct	3:00							-							-				-								
16-Oct	3:30		-					-							-				-								
16-Oct	4:00							-							-				-								
16-Oct	4:30		-					-							-				-								
16-Oct	5:00							-							-				-								
16-Oct	5:30		-					-							-				-								
16-Oct	6:00							-							-				-								
16-Oct	6:30		-					-							-				-								
16-Oct	7:00							-							-				-								
16-Oct	7:30		-					-							-				-								
16-Oct	8:00							-							-				-								
16-Oct	8:30		-					-							-				-								
16-Oct	9:00							-							-				-								
16-Oct	9:30							-							-				-								

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

PP2 LOGSHEET

Date	Time	ing Discharge NaSH mL/min	FAT final filtrate g/L HCl	Comments
TARGET			<5	
12-Oct	0:00			
12-Oct	0:30			
12-Oct	1:00			
12-Oct	1:30			
12-Oct	2:00			
12-Oct	2:30			
12-Oct	3:00			
12-Oct	3:30			
12-Oct	4:00			
12-Oct	4:30			
12-Oct	5:00			
12-Oct	5:30			
12-Oct	6:00			
12-Oct	6:30			
12-Oct	7:00			
12-Oct	7:30			
12-Oct	8:00			
12-Oct	8:30			
12-Oct	9:00			
12-Oct	9:30			
12-Oct	10:00			
12-Oct	10:30			
12-Oct	11:00			Start feeding to the circuit
12-Oct	11:30			11:20 Start acid addition to Reactor 1. 11:50 Scrubber 2 solution changed color. Start to add Fresh caustic.
12-Oct	12:00			Start acid addition in Reactor 2
12-Oct	12:30			
12-Oct	13:00		31.43	
12-Oct	13:30			Slurry "ate" the filter paper #3. Paper filter was changed to Glass fiber filter paper. Some solids went to recycling because of that.
12-Oct	14:00			Solids in the preheater were at the bottom. When rpm were increased a lot of solids went to Reactor 1 and provoque a huge reaction. Acid was turned off.
12-Oct	14:30			Acid in reactor 1 was turned on again
12-Oct	15:00			
12-Oct	15:30			FEED TANK MASS is with mixer turned off.
12-Oct	16:00		80.90	
12-Oct	16:30			FEED flowrate increased a "hair"
12-Oct	17:00		13.27	17:20 Acid Pump #2 turned ON.
12-Oct	17:30			
12-Oct	18:00			
12-Oct	18:30			
12-Oct	19:00			
12-Oct	19:30			
12-Oct	20:00			20:20 switched to using Filter No.1
12-Oct	20:30			
12-Oct	21:00		8.41	
12-Oct	21:30		16.55	
12-Oct	22:00			
12-Oct	22:30			
12-Oct	23:00			
12-Oct	23:30			acid pail 1 changed at 23:20 (initial 2.522 kg, final 12.062 kg)
13-Oct	0:00		30.00	acid pail 2 changed at 00:00 (initial 3.50 kg, final 10.90 kg)
13-Oct	0:30			switched to feed batch 2; started filling tank 1 with filtrate (stop at 57.46 kg)
13-Oct	1:00		5.61	
13-Oct	1:30	35u38		
13-Oct	2:00		10.57	
13-Oct	2:30	37u42		
13-Oct	3:00			
13-Oct	3:30	40d37		
13-Oct	4:00			
13-Oct	4:30	37	18.50	
13-Oct	5:00		20.40	at 5:10 shut off acid to Reactor 2 to bring down FAT in filtrate to 5 g/L
13-Oct	5:30	37	23.80	
13-Oct	6:00			
13-Oct	6:30	38	38.20	
13-Oct	7:00			
13-Oct	7:30			
13-Oct	8:00			
13-Oct	8:30			
13-Oct	9:00		4.43	
13-Oct	9:30			
13-Oct	10:00			
13-Oct	10:30			
13-Oct	11:00		0.06	
13-Oct	11:30			Filtrate to Drum.
13-Oct	12:00			acid pail 1 changed at 11:50 (initial 11.982 kg, final 3.142 kg)
13-Oct	12:30			12:00 Acid addition increased to 15g/min
13-Oct	13:00		0.10	
13-Oct	13:30			
13-Oct	14:00			
13-Oct	14:30			

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

PP2 LOGSHEET

Date	Time	ing Discharge NaSH mL/min	FAT final filtrate g/L HCl	Comments
TARGET			<5	
13-Oct	15:00		17.02	
13-Oct	15:30			Feed flowrate decreased from 43mL/min to 41mL/min
13-Oct	16:00			
13-Oct	16:30			16:35 turned feed rate up a hair.
13-Oct	17:00		0.11	Started NEW filter No. 2
13-Oct	17:30			
13-Oct	18:00			NEW feed started. OLD FEED final mass=25.13 (25.01 w/o mixer running)
13-Oct	18:30			NEW Feed starting mass= 73.99 @18:00
13-Oct	19:00			10:05 switched to new filter no. 1
13-Oct	19:30			
13-Oct	20:00			Drum (gal) totalizer does not appear to be working.
13-Oct	20:30			8:35:00 PM Changed to new acid for R1. Final (old) mass= 3.831, NEW mass=11.330
13-Oct	21:00			Increased acid to R1(from ~18 up to 21)
13-Oct	21:30			FIL to drum is surging.
13-Oct	22:00			
13-Oct	22:30			
13-Oct	23:00			
13-Oct	23:30	39	0.00	23:45 Increased R1 Acid Addition from 21 g/min to 24 g/min
14-Oct	0:00			
14-Oct	0:30	38		
14-Oct	1:00			
14-Oct	1:30	39	0.00	
14-Oct	2:00			
14-Oct	2:30	38	0.00	02:30 Increased R1 Acid Addition from 24 g/min to 27 g/min
14-Oct	3:00			2:55:00 AM changed out acid pail to R1 - end 3,00 kg start at 12.868 kg
14-Oct	3:30		0.00	
14-Oct	4:00			
14-Oct	4:30	41	0.00	04:30 Increased R1 Acid Addition from 27 g/min to 30 g/min
14-Oct	5:00			5:10 Added solids to Batch 4 Feed, mixing
14-Oct	5:30		0.00	
14-Oct	6:00			
14-Oct	6:30	38		
14-Oct	7:00			
14-Oct	7:30		10.65	
14-Oct	8:00			Acid pail 1 changed. Finished with 3.114, start with 10.167
14-Oct	8:30			
14-Oct	9:00		16.99	
14-Oct	9:30			
14-Oct	10:00			
14-Oct	10:30			
14-Oct	11:00		10.46	
14-Oct	11:30			
14-Oct	12:00			
14-Oct	12:30			
14-Oct	13:00		27.60	
14-Oct	13:30			
14-Oct	14:00			Acid pail 1 changed. Finished with 3.138, start with 10.791
14-Oct	14:30			
14-Oct	15:00			15:13 Tubing in FEED pump split. Acid stopped; Feed line repaired, Acid & Feed back on by 15:17
14-Oct	15:30			
14-Oct	16:00			
14-Oct	16:30			
14-Oct	17:00			17:25 Switched to Filter No. 2 for discharge.
14-Oct	17:30			
14-Oct	18:00		0.00	Increased Acid flow to R1 to ~23g/m
14-Oct	18:30			
14-Oct	19:00			19:50 Started adding FIL to the drum again.
14-Oct	19:30			19:45 Changed back to Filter No. 1. 19:55 Increased FRESH NaOH IN flowrate to ~95mL/min
14-Oct	20:00		0.00	20:01 Swapped out HCl pail. (OLD Final mass= 3.206) NEW HCl Initial Pail mass = 10.855
14-Oct	20:30			
14-Oct	21:00		0.00	Increased acid to Feed to 25gpm
14-Oct	21:30			
14-Oct	22:00			22:05 H2S alarm went off; R1 O/F plugged.
14-Oct	22:30			22:07 Feed and Acid off. 22:17 Fixed plugged O/F, Feed and Acid back on.
14-Oct	23:00			
14-Oct	23:30		0.00	23:49 to 00:19 Feed and Acid off to unplug PH to R1 O/F
15-Oct	0:00		0.00	00:50 swapped fresh and spent caustic drums
15-Oct	0:30	102	0.00	01:00 Acid to R1 was left off accidentaly - back on NOW
15-Oct	1:00		0.00	
15-Oct	1:30		0.00	
15-Oct	2:00			
15-Oct	2:30		0.00	2:35 Changed Acid -End 3.246 kg , Start 10.005 kg
15-Oct	3:00			
15-Oct	3:30		0.00	3:55 Increased Acid addition to R1 to 26 g/min
15-Oct	4:00			
15-Oct	4:30		0.00	
15-Oct	5:00		0.00	
15-Oct	5:30		0.00	Something has changed with the drum level indicator.

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

PP2 LOGSHEET

Date	Time	ing Discharge NaSH mL/min	FAT final filtrate g/L HCl	Comments
TARGET			**<5**	
15-Oct	6:00			
15-Oct	6:30		0.00	
15-Oct	7:00			Acid pail changed. Finished:2.885 Start: 12.214
15-Oct	7:30		1.53	
15-Oct	8:00			Feed tank changed. Finished 30.6, start @74.66
15-Oct	8:30			
15-Oct	9:00		2.63	
15-Oct	9:30			
15-Oct	10:00			
15-Oct	10:30			
15-Oct	11:00		1.75	
15-Oct	11:30		1.68	
15-Oct	12:00			
15-Oct	12:30			
15-Oct	13:00		1.97	
15-Oct	13:30			
15-Oct	14:00			Acid pail changed. Finished:2.991 Start: 11.013
15-Oct	14:30			
15-Oct	15:00			15:17 FRESH IN Caustic flowrate increased to 55mL/min.
15-Oct	15:30			
15-Oct	16:00		1.64	
15-Oct	16:30			
15-Oct	17:00		3.14	
15-Oct	17:30			Switched to Filter No. 2.
15-Oct	18:00			
15-Oct	18:30			Turned up feed a hair
15-Oct	19:00			
15-Oct	19:30			Acid pail to R1 changed to full one. (NEW Pail Initial mass = 10.463) Feed flowrate was 17.5; increased to20.5
15-Oct	20:00			
15-Oct	20:30			20:43 impeller keeps falling out of chuck on Feed tank; fixed again. Lost feed for a period because the the tube was no longer submerged.
15-Oct	21:00		29.00	
15-Oct	21:30			
15-Oct	22:00			
15-Oct	22:30			
15-Oct	23:00			
15-Oct	23:30		16.00	
16-Oct	0:00		9.04	Videos taken of the operation.
16-Oct	0:30		8.17	At 00:37 acid pail to R1 changed. (start = 2.5241 kg, end = 6.992 kg)
16-Oct	1:00		6.56	Final profile taken
16-Oct	1:30		4.38	
16-Oct	2:00			SHUTDOWN. Acid, feed and heat off.
16-Oct	2:30			Caustic pumps off.
16-Oct	3:00			
16-Oct	3:30			
16-Oct	4:00			B/T pump off. Power off.
16-Oct	4:30			
16-Oct	5:00			Pulp weights (g)
16-Oct	5:30			PH = 2413.3 Collected in a pail for disposal
16-Oct	6:00			R1 = 1556.8 Filtered. Solids not washed. Solids and filtrate kept.
16-Oct	6:30			R2 = 1890.2
16-Oct	7:00			R3 = 1848.5
16-Oct	7:30			R4 = 1916.8
16-Oct	8:00			R2-R4 pulps combined and filtered as one. Filtrate into Filtrate Drum.
16-Oct	8:30			R2-R4 solids washed, sampled and kept. Wash sampled and disposed of.
16-Oct	9:00			Crystals collected from overflow between the R3 and R4.
16-Oct	9:30			

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

PP1 FEED BATCHES

Feed Batch No.	Charge number	Concentrate wet weight kg	Concentrate % H2O %wt	Concentrate dry weight kg	Filtrate target kg	Filtrate actual kg	Pulp weight kg	Pulp %solids %	Start feeding at dd-mmm hh:mm	Comments
For example		**10.3**	**3.2%**	**10.0**	**25.31**	**25.31**	**35.61**	**28.0%**		
1	1	12.6	5.2%	12.0	30.08	30.09	42.70	27.99%	12-Oct 11:00	
2	2	12.6	6.0%	11.9	29.72	29.72	42.33	28.00%	13-Oct 0:20	
3	3	12.6	6.6%	11.8	29.46	29.42	42.04	28.03%	13-Oct 18:30	
4	4	12.6	6.1%	11.9	29.71	29.74	42.36	27.98%	14-Oct 14:00	
5	5	12.6	6.7%	11.8	29.46	29.50	42.12	27.97%	15-Oct 8:00	
6				-	-		0.00	-		
7				-	-		0.00	-		
8				-	-		0.00	-		
9				-	-		0.00	-		
10				-	-		0.00	-		
11				-	-		0.00	-		
12				-	-		0.00	-		
13				-	-		0.00	-		
14				-	-		0.00	-		
15				-	-		0.00	-		
16				-	-		0.00	-		
17				-	-		0.00	-		
18				-	-		0.00	-		
19				-	-		0.00	-		
20				-	-		0.00	-		

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

PP1 SHIFT LEADER COMMENTS

Date / Time	Shift Leader	Observation Comment
12-Oct 18:00	D. Johnson	After the problems just as the shift changed, we monitored the free acid and kept the acid addition to R2 at zero until it dropped. In hindsight this was a mistake since we simply left un-reacted ore in all the reactors. The free acid eventually dropped to below 20 and we restarted the acid to R@. There was some foaming and a small emission of H2S (0.3 on the monitor). The free acid readings on all of the profile samples taken at 18:00 were zero. I am certain that there will be a lot of un-reacted ore in the solids and the filtrates will be disappointing. You may wish to schedule another profile rather than wait an entire day.
12-Oct 18:00	D. Johnson	The H2S monitor read 1.2 ppm after sampling. There were no unpleasant blasts of gas. The plug on R1 was very tight. Mechanical assistance was required.
12-Oct 20:20	D. Johnson	The filter ceased filtering. The entire filtration system wants to die.The buchners won't sit flat. The tubing doesn't like working against a dead-headed system and gives up the ghost after a few hours. We flipped to the other filter and will assess the situation.
12-Oct 21:10	D. Johnson	Tested the batch of filtrate to see if any H2S evolution took place when the ore was added. None detected with the portable detector or the shift leader's nose. No gas evolution or increase in temperature. We sampled for FAT before and after. Went ahead and mixed batch 2 of feed. The free acid dropped from 29.5 to 16.6 in the test. I suggest measuring the free acid again just before starting to use the tank.
13-Oct 01:00	C.Forstner	Acid to Reactor 1 pail ran empty at shift change. Reactor 1 was acid starved for ~30 minutes while filling pail. Filtrate FAT appeared to drop from 30 to 8 g/L from 00:00 to 00:30. Likely explanation is that filtrate pumps were drawing filtrate from both funnels which contained older slurry (~1 -2 L) prior to composite at midnight. After midnight, filtrate collected was representative of realtime slurry leaving circuit.Switched feed batch at 00:20. Running well. Kind of smelly, but we nightshifters are tough.
13-Oct 05:00	C.Forstner	Filtrate FAT has risen gradually since start of batch 2 feed. Shut off acid pump to reactor 2 to compensate for high acid in feed slurry.
13-Oct 08:00	Ernesto	Filtrate lines were putting down (less hight) to facilitate the filtration. From time to time move the hoses inside the filtrate pumps to avoid the hoses get "flat".
13-Oct 12:00	Ernesto	Filtrate free acid @ 11:00 was 0.06g/L. Acid addition was increased to 15g/min in reactor 1, still no acid in reactor 2. A little bit more of foaming activity but nothing to worry about it so far.
13-Oct 12:00	Ernesto	Every time funnel is change for the first 10-20 minutes some solids pass through the filter. At the end we'll need to filter the filtrate drum. We changed both filtrate pumps hose.
13-Oct 12:00	Ernesto	All the filtrate needed for the feed batch is in. Filtrate was directed to the drum. Now we need to start reading the volume of the drum. Pushed the button on the meter until the light goes on and then take the reading.
		When the feed tank is "close to be empty" add the solids in the other tank. This is because we suspect some leaching may be happening in the feed tank, so avoid to mix the solids with the recycle liquor before it is needed.
13-Oct 13:00	Ernesto	B/T pump was increased to 6L/min. Every time we take readings, as soon as you open any reactor H2S "comes out" of the reactor. This could be an indication there is no much vacuum inside the reactors.
13-Oct 16:00	D. Johnson	We had a lot of trouble making sense of the FAT measurements. Changing the aliquot size changed completely the reading. Doubling the volume of oxalate changed the reading. We now believe that using larger aliquot sizes requires significantly more oxalate than we are using. We have submitted two samples to analytical for FAT to compare with the same samples run by us. Lims number is Oct02622. For the moment we are using only 2 ml aliquots and 25 mls of oxalate.
13-Oct 18:00	D. Johnson	Changed feed to batch 3. The solution density seemed to be slowly dropping over time in the last batch. There are at least two theories why this could be so. Regardless it is affecting our acid/ore ratio. This batch has started out with 1.735. We have started taking the volume over two minutes to improve the precision.
13-Oct 18:30	D. Johnson	Increased acid to aprox. 18 gpm. Will measure FAT of filtrate after two hours and adjust accordingly. All acid to go into R1.
13-Oct 20:30	D. Johnson	The level detector in the filtrate drum doesn't seem to be functioning. It claims to have 1.5 gallons, but the weight of the drum suggests otherwise.
13-Oct 20:30	D. Johnson	FAT of filtrate effectively zero. Increased the acid to R1 to 21gpm at 21:00
13-Oct 21:00	D. Johnson	Excitement. We overfilled an acid pail because the acid pump was left unattended. Acid got onto the floor of the fume hood. As we were cleaning up, one of the receptacles on the front face of the fume hood caught fire. We extinguished the fire and tried to find a breaker to kill the power to the hood. None of the boxes or switches are properly labeled. Had to call in an electrician. Mike Cannon came, disconnected the hood and took apart the receptacle. They are ground fault connected but a short across the live side of the plugs can still cause a fire. The receptacles are NOT waterproof. The breaker in the main panel did not seem to have tripped.
13-Oct 23:00	C.Forstner	Cross-over instructions are to continue increasing the acid addition to reactor 1 until acid is detected in filtrate. Make up batch 4 of feed. Keep an eye on tubing for filtrate pumps for wear. FAT using 2 mL aliquot. Stay awake. Be cool.
13-Oct 23:45	C.Forstner	No acid in filtrate. Increased acid addition rate to reactor 1 to 24 gpm.
14-Oct 01:00	C.Forstner	Neoprene vent lines from reactors show grey discoloration. Not sure why.
14-Oct 02:30	C.Forstner	No acid in filtrate. Increased acid addition rate to reactor 1 to 27 gpm.

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

PP1 SHIFT LEADER COMMENTS

Date / Time	Shift Leader	Observation Comment
14-Oct 04:30	C.Forstner	No acid in filtrate. Increased acid addition rate to reactor 1 to 30 gpm.
14-Oct 08:00	Ernesto	Scrubber 1 looked like is changing color. Fresh caustic flowrate increased to 47g/min
14-Oct 08:15	Ernesto	Filtrate free acid @7:30 was 10.92g/L. Acid addition decreased to 26g/min
14-Oct 12:00	Ernesto	When taking the scrubber sample @12:00 the level of H2S on the main detector was up to 2.6ppm. Let's find a way to take this sample without increasing the H2S level in the room!!
14-Oct 13:30	Ernesto	Excitement!! Feed tubing got plugged. Feed and acid were stopped for around 30 minutes. The whole tube was changed. Also the downcomer from the pre-heater to reactor 1 seemed plugged, but now is OK, keep an eye on this. Check from time to time the feedwell in the feed tank, it also seemed plugged.
14-Oct 19:00	D. Johnson	The clean column in the scrubber is getting progressively darker. I blew some of the effluent gas past the portable monitor and it went off scale. I have increased the NaOH addition to the scrubber to 96 ml/min. However I suspect that the lead column is completely saturated and the clean column nearly saturated. This will mean poor scrubbing efficiencies.
14-Oct 19:50	D. Johnson	The H2S concentration over the main scrubber solution is 2 ppm.
14-Oct 20:10	D. Johnson	Sample RES 10140000 (a fairly massive sample) had dried after 3.5 hours and started fuming in the oven. We removed it. Then smaller samples in the oven caught fire. The filter paper was burning. I have turned the oven off and we are airing out the room. We are not amused.
14-Oct 21:00	D. Johnson	I again blew some scrubber exhaust gas past the portable detector. It still went off scale but returned to "normal" much more quickly. The H2S above the main scrubber solution is now below detection.
14-Oct 22:00	D. Johnson	Increased acid to R1 to 25 gpm after measuring free acid at zero in filtrate.
14-Oct 22:05	D. Johnson	Portable alarm sounded and the large monitor showed 0.9. R1 was foaming up into the condenser. The outlet from R1 to R2 was plugged. Turned acid and feed off at 22:07 and tried a variety of "tools" to poke down into the vessel then into the overflow tube. Not a very easy task, but Aurele eventually got it freed up and the acid and feed were restarted at 22:17. The next generation of this pilot plant should be built with cleaning out the overflows as a design criteria.
14-Oct 23:00	C.Forstner	Cross-over instructions from Dave: Sample the NHL scrubber at 1AM. Keep an eye on feed line and reactor overflows for plugging. At midnight or 1AM consider adjusting the acid flowrate to R1 by 1 g/min depending on FAT. Continue filling feed make-up tank with filtrate to make a full last batch of feed. Wait until dayshift to add solids to feed make-up tank. Do not use paper in the oven. Observations: Scrubber 1 is lightening up in colour even within last 30 minutes. Steady state should be reached in a few more hours.
14-Oct 01:00	C.Forstner	We pulled apart the connector between the pre-heater and reactor 1 to clear a plug. The feed was off for 30 minutes and the acid was off for an hour (forgot to turn it back on…oops). We swapped out the caustic drums (fresh and spent) and at 1:30 we sampled the NHL scrubber. No H2S was detected in the NHL scrubber. The circuit should be steady again at 3:30.
14-Oct 04:00	C.Forstner	Increased acid to R1 to 26 gpm after measuring free acid at zero in filtrate.
15-Oct 08:30	Ernesto	Feed line got plugged; the entire line was changed once again. It happened when feed tank was changed. It seems every time the feed pump is stopped somehow the feed line get cold and get clogged. Acid was stopped for 15-20 minutes, it could have been running for a while, not sure how long though. Free acid increased up to 2.63g/L.
15-Oct 14:00	Ernesto	What was left in Feed tank # 2 was pumped out to a pail. Pail weight: 14.164kg (gross) Tare:1.140kg
15-Oct 15:15	D. Johnson	Passed scrubber exhaust over the portable H2S sensor and got zero H2S. Lowered the caustic feed rate to 55ml/min
15-Oct 18:10	D. Johnson	The impellor fell off in the feed mix tank.
15-Oct 21:55	D. Johnson	The feed to the preheater plugged earlier. The free acid in the discharge is now elevated. I think that this is temporary so I am not adjusting the acid flow. Charlotte can take another reading in an hour or so and if it is still high, lower the acid.
15-Oct 23:30	C. Forstner	Filtrate free acid is 16 gpL – still high but down from 21:00 reading of 29 gpL. Not changing acid flowrate.

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

PP1 MASS BALANCE (g/min)

		Feed								Solid assays							Solution assays					
		Availability	Slurry		Solids Density		Ore	Solution		Fe	Ni	Cu	Co	Mg	S	S=	Fe	Fe(II)	Ni	Cu	Co	Mg
Shift Starting	Shift Ending	%	mL/min	g/min	g/mL	%wt	g/min	mL/min	SG	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
12-Oct 12:00	13-Oct 00:00	100%	20	36	1.673	27.99%	10.0	19	1.388	55.4	0.97	1.73	0.10	0.075	33.6	31.8	198000	191870	102	102	<5	1820
13-Oct 00:00	13-Oct 12:00	100%	20	29	1.474	28.00%	8.0	15	1.390	54.8	1.04	1.67	0.11	0.056	36.4	33.7	188700	187300	121	243	<5	1370
13-Oct 12:00	14-Oct 00:00	100%	20	36	1.665	28.02%	10.0	18	1.412	57.8	1.02	1.65	0.10	0.074	35.6	31.9	212000	210570	116	229	6	1322
14-Oct 00:00	14-Oct 12:00	100%	21	38	1.712	28.03%	10.5	19	1.423	56.6	1.02	1.66	0.10	0.067	34.2	30.6	212300	208000	153	246	8	1198
14-Oct 12:00	15-Oct 00:00	96%	21	35	1.757	27.98%	9.9	18	1.427	55.8	1.06	1.69	0.10	0.072	35.1	30.7	223800	212600	68	177	<5	899
15-Oct 00:00	15-Oct 12:00	94%	21	38	1.726	27.98%	10.8	19	1.425	58.4	1.07	1.72	0.11	0.014	35.2	34.3	229635	203800	117	208	7	845
15-Oct 12:00	16-Oct 02:00	100%	21	37	1.757	27.97%	10.4	19	1.442	56.4	1.06	1.83	0.11	0.063	35.0	34.9	227200	213100	179	254	9	802

		Acid			
		R1	R2	Total	Dosage
Shift Starting	Shift Ending	g/min	g/min	g/min	kg/t ore
12-Oct 12:00	13-Oct 00:00	11	11	22	540
13-Oct 00:00	13-Oct 12:00	12	12	24	741
13-Oct 12:00	14-Oct 00:00	17	0	17	421
14-Oct 00:00	14-Oct 12:00	26	0	26	612
14-Oct 12:00	15-Oct 00:00	22	0	22	563
15-Oct 00:00	15-Oct 12:00	21	0	21	485
15-Oct 12:00	16-Oct 02:00	26	0	26	627
AVERAGE		19	3	22	570
MIN		11	0	17	421
MAX		26	12	26	741
RELSTD		33%	171%	14%	18%

		Discharge								Solid assays							Solution assays						Wash water assays					
		Filtrate			Residue			Wash		Fe	Ni	Cu	Co	Mg	S	S=	Fe	Fe(II)	Ni	Cu	Co	Mg	Fe	Fe(II)	Ni	Cu	Co	Mg
Shift Starting	Shift Ending	mL/min	SG	g/min	g (wet)	% moisture	g (dry)	g	SG	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
12-Oct 12:00	13-Oct 00:00	48	1.390	66	1673	41.5%	979	5386	1.040	36.6	3.01	4.58	0.28	0.140	40.5	35.8	197800	192410	29	62	<5	1050	18900	17800	26	<5	<5	83
13-Oct 00:00	13-Oct 12:00	30	1.422	43	2395	29.4%	1692	5410	1.046	37.4	3.10	5.21	0.32	0.140	44.0	42.6	213800	209900	35	<5	<5	890	24500	20300	39	28	<5	13
13-Oct 12:00	14-Oct 00:00	29	1.426	41	3048	31.8%	2077	5460	1.065	42.9	2.82	4.40	0.26	0.170	41.9	34.6	219800	219800	<5	<5	<5	787	30900	30900	10	<5	<5	114
14-Oct 00:00	14-Oct 12:00	40	1.425	57	3650	31.4%	2503	5250	1.097	35.9	3.38	6.56	0.38	0.150	43.8	33.2	230500	230500	298	6	7	573	48000	43500	126	<5	<5	126
14-Oct 12:00	15-Oct 00:00	30	1.438	43	2682	31.0%	1850	5094	1.056	36.0	2.64	4.53	0.28	0.140	39.3	28.7	236000	226400	119	<5	<5	550	28100	24400	82	<5	<5	65
15-Oct 00:00	15-Oct 12:00	33	1.441	47	3858	23.9%	2936	5109	1.105	37.9	3.07	5.07	0.31	0.110	40.3	36.0	248550	207400	<5	<5	<5	495	53400	49100	10	<5	<5	110
15-Oct 12:00	16-Oct 02:00	32	1.432	45	3302	52.9%	1556	5752	1.090	25.8	4.41	9.09	0.51	0.170	47.6	24.7	224500	214200	327	16	13	442	45900	43600	174	<5	<5	85

		Off gas	
		S	H_2S
Shift Starting	Shift Ending	g/min	g/min
12-Oct 12:00	13-Oct 00:00	2.8	3.0
13-Oct 00:00	13-Oct 12:00	1.9	2.0
13-Oct 12:00	14-Oct 00:00	2.3	2.5
14-Oct 00:00	14-Oct 12:00	2.1	2.2
14-Oct 12:00	15-Oct 00:00	2.5	2.6
15-Oct 00:00	15-Oct 12:00	2.1	2.3
15-Oct 12:00	16-Oct 02:00	2.8	2.9

S (g/min) in the off gas was calculated as the difference between Sulphide (g/min) in the feed and total S (g/min) in the residue.

		TOTAL IN - g/min								Solids - g/min							Solution - g/min				
Shift Starting	Shift Ending	mass	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
12-Oct 12:00	13-Oct 00:00	57.4	9.2	0.1	0.2	0.01	0.04	3.4	3.2	5.5	0.1	0.2	0.010	0.008	3.4	3.2	3.7	0.002	0.002	0.0001	0.034
13-Oct 00:00	13-Oct 12:00	52.4	7.2	0.1	0.1	0.01	0.02	2.9	2.7	4.4	0.1	0.1	0.009	0.004	2.9	2.7	2.8	0.002	0.004	0.0001	0.020
13-Oct 12:00	14-Oct 00:00	52.5	9.6	0.1	0.2	0.01	0.03	3.6	3.2	5.8	0.1	0.2	0.010	0.007	3.6	3.2	3.9	0.002	0.004	0.0001	0.024
14-Oct 00:00	14-Oct 12:00	63.2	10.0	0.1	0.2	0.01	0.03	3.6	3.2	6.0	0.1	0.2	0.011	0.007	3.6	3.2	4.0	0.003	0.005	0.0002	0.023
14-Oct 12:00	15-Oct 00:00	57.6	9.5	0.1	0.2	0.01	0.02	3.5	3.0	5.5	0.1	0.2	0.010	0.007	3.5	3.0	4.0	0.001	0.003	0.0001	0.016
15-Oct 00:00	15-Oct 12:00	59.3	10.7	0.1	0.2	0.0	0.02	3.8	3.7	6.3	0.1	0.2	0.012	0.002	3.8	3.7	4.5	0.002	0.004	0.0001	0.016
15-Oct 12:00	16-Oct 02:00	63.3	10.1	0.1	0.2	0.0	0.02	3.6	3.6	5.9	0.1	0.2	0.011	0.007	3.6	3.6	4.2	0.003	0.005	0.0002	0.015

		TOTAL OUT - g/min								Solids - g/min							Solution - g/min					Wash water - g/min				
Shift Starting	Shift Ending	mass	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg
12-Oct 12:00	13-Oct 00:00	71.8	10.1	0.0	0.1	0.004	0.05	3.4	0.5	0.5	0.0	0.1	0.0	0.0	0.6	0.5	9.5	0.001	0.003	0.0002	0.050	0.1	0.0	0.000	0.0000	0.001
13-Oct 00:00	13-Oct 12:00	48.6	7.6	0.1	0.1	0.008	0.03	2.9	1.0	0.9	0.1	0.1	0.0	0.0	1.0	1.0	6.5	0.001	0.000	0.0002	0.027	0.2	0.000	0.000	0.0000	0.000
13-Oct 12:00	14-Oct 00:00	47.8	7.8	0.1	0.1	0.008	0.03	3.6	1.0	1.2	0.1	0.1	0.0	0.0	1.2	1.0	6.3	0.000	0.000	0.0001	0.023	0.2	0.000	0.000	0.0000	0.001
14-Oct 00:00	14-Oct 12:00	63.8	10.7	0.1	0.2	0.014	0.03	3.6	1.2	1.2	0.1	0.2	0.0	0.0	1.5	1.2	9.1	0.012	0.000	0.0003	0.023	0.3	0.001	0.000	0.0000	0.001
14-Oct 12:00	15-Oct 00:00	49.5	8.2	0.1	0.1	0.007	0.02	3.5	0.7	0.9	0.1	0.1	0.0	0.0	1.0	0.7	7.1	0.004	0.000	0.0002	0.017	0.2	0.001	0.000	0.0000	0.000
15-Oct 00:00	15-Oct 12:00	54.9	10.0	0.1	0.2	0.013	0.02	3.8	1.5	1.5	0.1	0.2	0.0	0.0	1.6	1.5	8.2	0.000	0.000	0.0002	0.016	0.3	0.000	0.000	0.0000	0.001
15-Oct 12:00	16-Oct 02:00	52.0	7.8	0.1	0.2	0.010	0.02	3.6	0.5	0.5	0.1	0.2	0.0	0.0	0.9	0.5	7.1	0.010	0.001	0.0004	0.014	0.3	0.001	0.000	0.0000	0.001

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

PP1 MASS BALANCE (g/min)

		OUT/IN - %							
Shift Starting	Shift Ending	mass	Fe	Ni	Cu	Co	Mg	S	S=
12-Oct 12:00	13-Oct 00:00	125%	110%	43%	37%	40%	128%	100%	15%
13-Oct 00:00	13-Oct 12:00	93%	105%	87%	89%	87%	123%	100%	37%
13-Oct 12:00	14-Oct 00:00	91%	81%	78%	75%	76%	90%	100%	31%
14-Oct 00:00	14-Oct 12:00	101%	107%	118%	127%	127%	97%	100%	36%
14-Oct 12:00	15-Oct 00:00	86%	86%	68%	68%	74%	89%	100%	24%
15-Oct 00:00	15-Oct 12:00	93%	94%	107%	110%	107%	120%	100%	40%
15-Oct 12:00	16-Oct 02:00	82%	78%	82%	87%	85%	82%	100%	13%
AVERAGE		96%	94%	83%	85%	85%	104%	100%	28%
MIN		82%	78%	43%	37%	40%	82%	100%	13%
MAX		125%	110%	118%	127%	127%	128%	100%	40%
RELSTD		15%	14%	30%	34%	32%	18%	0%	39%

		EXTRACTION - %							
Shift Starting	Shift Ending	Weight loss	Fe	Ni	Cu	Co	Mg	S	S=
12-Oct 12:00	13-Oct 00:00	85%	91%	58%	64%	62%	75%	84%	85%
13-Oct 00:00	13-Oct 12:00	63%	80%	13%	8%	15%	27%	65%	63%
13-Oct 12:00	14-Oct 00:00	69%	79%	20%	23%	25%	34%	66%	69%
14-Oct 00:00	14-Oct 12:00	64%	79%	0%	0%	0%	26%	58%	64%
14-Oct 12:00	15-Oct 00:00	76%	83%	35%	30%	27%	49%	71%	76%
15-Oct 00:00	15-Oct 12:00	60%	75%	0%	0%	0%	0%	57%	60%
15-Oct 12:00	16-Oct 02:00	87%	92%	26%	12%	17%	52%	76%	87%
AVERAGE		72%	83%	22%	20%	21%	37%	68%	72%
MIN		60%	75%	0%	0%	0%	0%	57%	60%
MAX		87%	92%	58%	64%	62%	75%	84%	87%
RELSTD		15%	8%	95%	115%	101%	64%	14%	15%

Extraction=1-Me in Residue / Me in Feed

		EXTRACTION - %							
Shift Starting	Shift Ending	Weight loss	Fe	Ni	Cu	Co	Mg	S	S=
12-Oct 12:00	13-Oct 00:00	85%	92%	0%	2%	5%	90%	84%	0%
13-Oct 00:00	13-Oct 12:00	63%	82%	0%	0%	1%	68%	65%	0%
13-Oct 12:00	14-Oct 00:00	69%	69%	0%	0%	1%	0%	66%	0%
14-Oct 00:00	14-Oct 12:00	64%	81%	8%	0%	1%	14%	58%	0%
14-Oct 12:00	15-Oct 00:00	76%	78%	4%	0%	1%	20%	71%	0%
15-Oct 00:00	15-Oct 12:00	60%	72%	0%	0%	0%	11%	57%	0%
15-Oct 12:00	16-Oct 02:00	87%	87%	9%	0%	3%	0%	76%	0%
AVERAGE		72%	80%	3%	0%	2%	29%	68%	0%
MIN		60%	69%	0%	0%	0%	0%	57%	0%
MAX		87%	92%	9%	2%	5%	90%	84%	0%
RELSTD		15%	10%	134%	265%	77%	123%	14%	0%

Extraction= 1-Me in residue / Me Total out

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

PP1 OVERALL MASS BALANCE

STREAMS IN

Feed				Solids assays							Solids - kg						
Batch	wet	moisture	dry	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	S	S=
No.	kg	%wt	kg	%	%	%	%	%	%	%	kg	kg	kg	kg	kg	kg	kg
1	12.61	5.2%	11.95	55.9	1.01	1.72	0.10	0.070	33.6	33.3	6.68	0.12	0.21	0.01	0.01	4.02	3.98
2	12.61	6.0%	11.85	56.2	1.02	1.68	0.10	0.088	33.2	33.2	6.66	0.12	0.20	0.01	0.01	3.94	3.94
3	12.62	6.6%	11.78	56.0	1.02	1.70	0.10	0.083	33.4	33.4	6.60	0.12	0.20	0.01	0.01	3.94	3.94
4	12.62	6.1%	11.85	56.7	1.04	1.74	0.10	0.080	33.9	33.4	6.72	0.12	0.21	0.01	0.01	4.02	3.96
5	12.62	6.7%	11.78	56.0	1.03	1.74	0.10	0.075	34.3	33.0	6.60	0.12	0.21	0.01	0.01	4.04	3.89
TOTAL	63.08		59.23								33.26	0.61	1.02	0.06	0.05	19.95	19.70

Synthetic solution				Solution assays					Solution - kg				
Pail	weight	SG	volume	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg
No.	kg		L	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg
1	20.00	1.445	13.84	212863	0	0	0	1995	2.95	0.00	0.00	0.00	0.03
2	20.00	1.445	13.84	212863	0	0	0	1995	2.95	0.00	0.00	0.00	0.03
3	3.86	1.445	2.67	212863	0	0	0	1995	0.57	0.00	0.00	0.00	0.01
TOTAL	43.86		30.35						6.46	0.00	0.00	0.00	0.06

Acid			
Drum	weight	SG	volume
	kg	-	L
PP1	103.56	1.125	92.05

Fresh Caustic			
Drum	weight	SG	volume
	kg	-	L
PP1	311	1.098	283.24

STREAMS OUT

Filter Cake				Solid assays							Solution assays					Solids - kg							Solution - kg				
Batch	wet	moisture	dry	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
No.	kg	%wt	kg	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg
1	1.67	41.5%	0.98	36.6	3.01	4.58	0.28	0.140	40.5	35.8	197800	29	62	<5	1050	0.36	0.03	0.04	0.00	0.00	0.40	0.35	0.10	0.00	0.00	0.00	0.00
2	2.39	29.4%	1.69	37.4	3.10	5.21	0.32	0.140	44.0	42.6	213800	35	<5	<5	890	0.63	0.05	0.09	0.01	0.00	0.74	0.72	0.11	0.00	0.00	0.00	0.00
3	3.05	31.8%	2.08	42.9	2.82	4.40	0.26	0.170	41.9	34.6	219800	<5	<5	<5	787	0.89	0.06	0.09	0.01	0.00	0.87	0.72	0.15	0.00	0.00	0.00	0.00
4	3.65	31.4%	2.50	35.9	3.38	6.56	0.38	0.150	43.8	33.2	230500	298	6	7	573	0.90	0.08	0.16	0.01	0.00	1.10	0.83	0.19	0.00	0.00	0.00	0.00
5	2.68	31.0%	1.85	36.0	2.64	4.53	0.28	0.140	39.3	28.7	236000	119	<5	<5	550	0.67	0.05	0.08	0.01	0.00	0.73	0.53	0.14	0.00	0.00	0.00	0.00
6	3.86	23.9%	2.94	37.9	3.07	5.07	0.31	0.110	40.3	36.0	248550	<5	<5	<5	495	1.11	0.09	0.15	0.01	0.00	1.18	1.06	0.16	0.00	0.00	0.00	0.00
7	3.30	52.9%	1.56	25.8	4.41	9.09	0.51	0.170	47.6	24.7	224500	327	16	13	442	0.40	0.07	0.14	0.01	0.00	0.74	0.38	0.27	0.00	0.00	0.00	0.00
TOTAL	20.61		13.59													4.96	0.43	0.76	0.05	0.02	5.76	4.59	1.11	0.00	0.00	0.00	0.00

Filtrate				Solution assays					Solution - kg				
Drum	weight	SG	volume	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg
	kg	-	L	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg
PP1	117	1.430	82	226700	337	17	11	534	18.55	0.03	0.00	0.00	0.04

Wash water				Solution assays					Solution - kg				
Drum	weight	SG	volume	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg
	kg	-	L	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg
1	5.39	1.040	5.18	18900	26	<5	<5	83	0.10	0.00	0.00	0.00	0.00
2	5.41	1.046	5.17	24500	39	28	<5	13	0.13	0.00	0.00	0.00	0.00
3	5.46	1.065	5.13	30900	10	<5	<5	114	0.16	0.00	0.00	0.00	0.00
4	5.25	1.097	4.79	48000	126	<5	<5	126	0.23	0.00	0.00	0.00	0.00
5	5.09	1.056	4.82	28100	82	<5	<5	65	0.14	0.00	0.00	0.00	0.00
6	5.11	1.105	4.62	53400	10	<5	<5	110	0.25	0.00	0.00	0.00	0.00
7	5.75	1.090	5.28	45900	174	<5	<5	85	0.24	0.00	0.00	0.00	0.00
TOTAL	37.46		34.99						1.24	0.00	0.00	0.00	0.00

H_2S stream		
Campaign	S	H_2S
	kg	kg
PP1	10.94	11.63

Spent Caustic					
Drum	weight	SG	volume	S	S
	kg	-	L	mg/L	kg
Drum 1	175	1.033	169.36	40333	6.83
Drum 2	123	1.077	113.79	29000	3.30
TOTAL	297.50		283.15		10.13

FINAL INVENTORY

Feed							Solid assays							Solution assays					Solids - kg							Solution - kg				
Feed	Slurry	Solids		Solution			Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
Tank	kg	%wt	kg	kg	SG	L	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg
1	13.02	27.98%	3.64	9.38	1.427	6.6	55.8	1.06	1.69	0.10	0.072	35.1	30.7	223800	68	177	<5	899	2.03	0.04	0.06	0.00	0.00	1.28	1.12	1.47	0.00	0.00	0.00	0.01
2	13.54	27.97%	3.79	9.75	1.442	6.8	56.4	1.06	1.83	0.11	0.063	35.0	34.9	227200	179	254	9	802	2.14	0.04	0.07	0.00	0.00	1.33	1.32	1.54	0.00	0.00	0.00	0.01
TOTAL	26.56		7.43																4.17	0.08	0.13	0.01	0.01	2.60	2.44	3.01	0.00	0.00	0.00	0.01

Reactors							Solid assays							Solution assays					Solids - kg							Solution - kg				
Reactors	Slurry	Solids		Solution			Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
ID	kg	%wt	kg	kg	SG	L	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg
PH	2.41	28.6%	0.69	1.72	1.461	1.2	56.0	1.00	1.72	0.10	0.070	35.4	33.8	231200	487	<5	12	466	0.39	0.01	0.01	0.00	0.00	0.24	0.23	0.27	0.00	0.00	0.00	0.00
R1	1.56	1.8%	0.03	1.53	1.438	1.1	25.8	3.76	6.39	0.44	0.590	46.4	19.8	225100	351	<5	5	477	0.01	0.00	0.00	0.00	0.00	0.01	0.01	0.24	0.00	0.00	0.00	0.00
R2	1.89	4.7%	0.09	1.80	1.441	1.2	38.5	2.72	4.97	0.31	0.110	45.0	25.4	230000	273	<5	<5	459	0.03	0.00	0.00	0.00	0.00	0.04	0.02	0.29	0.00	0.00	0.00	0.00
R3	1.85	4.4%	0.08	1.77	1.438	1.2	34.4	3.26	6.03	0.37	0.130	46.0	24.1	220300	318	<5	<5	442	0.03	0.00	0.00	0.00	0.00	0.04	0.02	0.27	0.00	0.00	0.00	0.00
R4	1.92	3.3%	0.06	1.85	1.418	1.3	28.4	3.73	6.93	0.42	0.160	50.7	24.9	232800	250	<5	10	852	0.02	0.00	0.00	0.00	0.00	0.03	0.02	0.30	0.00	0.00	0.00	0.00
TOTAL	9.63		0.95	8.67		6.03													0.47	0.02	0.03	0.00	0.00	0.37	0.30	1.37	0.00	0.00	0.00	0.00

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

PP1 OVERALL MASS BALANCE

Scrubbers					
Scrubber	weight	SG	volume	S	S
	kg	-	L	mg/L	kg
Scrubber 1	14	1.103	13.06	1000	0.01
Scrubber 2	17	1.091	15.53	35000	0.54
TOTAL	31.34		28.59		0.56

SAMPLING

Leaching								Solid assays							Solution assays					Solids - kg							Solution - kg				
Sample		Slurry	Solids		Solution			Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
ID		kg	%wt	kg	kg	SG	L	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg
PLS	10121300	0.07	-	-	0.07	1.366	0.05	-	-	-	-	-	-	-	183550	54	11	<5	1636	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	10121700	0.07	-	-	0.07	1.412	0.05	-	-	-	-	-	-	-	212400	143	58	<5	1255	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
R4 DC	10122100	0.11	3.7%	0.00	0.10	1.410	0.07	36.6	-	4.61	-	-	-	40.3	210000	14	<5	<5	900	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10130100	0.11	-	-	0.11	1.418	0.07	-	-	-	-	-	-	-	218100	<5	<5	<5	1060	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10130500	0.09	-	-	0.09	1.416	0.06	-	-	-	-	-	-	-	218700	83	<5	<5	760	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
R4 DC	10130900	0.07	5.4%	0.00	0.07	1.410	0.05	38.1	-	5.18	-	-	-	39.8	212400	<5	<5	<5	840	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	10131300	0.09	-	-	0.09	1.417	0.06	-	-	-	-	-	-	-	223000	<5	<5	<5	854	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	10131700	0.04	-	-	0.04	1.421	0.03	-	-	-	-	-	-	-	228100	<5	<5	<5	838	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
R4 DC	10132100	0.13	5.8%	0.01	0.12	1.425	0.08	41.1	-	4.19	-	-	-	39.2	217900	<5	<5	<5	810	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10140100	0.10	-	-	0.10	1.435	0.07	-	-	-	-	-	-	-	230600	<5	<5	<5	680	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10140500	0.10	-	-	0.10	1.432	0.07	-	-	-	-	-	-	-	225400	<5	<5	<5	580	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	10140900	0.13	3.8%	0.01	0.13	1.408	0.09	24.7	-	9.04	-	-	-	28.4	212500	622	<5	15	580	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10141300	0.09	-	-	0.09	1.400	0.06	-	-	-	-	-	-	-	212400	694	90	25	598	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	10141700	0.07	-	-	0.07	1.401	0.05	-	-	-	-	-	-	-	234000	21	50	<5	648	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
R4 DC	10142100	0.13	4.7%	0.01	0.12	1.410	0.09	39.0	-	3.81	-	-	-	40.3	235700	<5	<5	<5	475	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10150100	0.10	-	-	0.10	1.444	0.07	-	-	-	-	-	-	-	241800	<5	<5	<5	490	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10150500	0.10	-	-	0.10	1.441	0.07	-	-	-	-	-	-	-	237100	<5	<5	<5	484	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	10150900	0.13	5.5%	0.01	0.12	1.431	0.08	35.8	-	6.03	-	-	-	33.0	233700	<5	<5	<5	470	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10151300	0.14	-	-	0.14	1.428	0.10	-	-	-	-	-	-	-	227210	6	<5	<5	438	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10151700	0.11	-	-	0.11	1.431	0.08	-	-	-	-	-	-	-	226300	53	<5	<5	437	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	10152100	0.15	3.0%	0.00	0.14	1.400	0.10	19.8	-	12.80	-	-	-	25.0	208800	711	24	31	420	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	10160100	0.12	-	-	0.12	1.428	0.09	-	-	-	-	-	-	-	232600	604	6	15	450	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	10121800	0.17	37.8%	0.06	0.10	1.366	0.08	56.7	0.97	1.47	0.10	0.050	35.6	33.5	192400	34	<5	<5	1800	0.04	0.00	0.00	0.00	0.00	0.02	0.02	0.01	0.00	0.00	0.00	0.00
R1	10121800	0.14	29.4%	0.04	0.10	1.415	0.07	51.6	1.45	2.51	0.15	0.072	38.4	38.1	211000	<5	<5	<5	1280	0.02	0.00	0.00	0.00	0.00	0.02	0.02	0.01	0.00	0.00	0.00	0.00
R2	10121800	0.13	12.1%	0.02	0.11	1.426	0.08	41.4	2.21	3.60	0.22	0.100	36.3	33.8	222700	<5	<5	<5	1020	0.01	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
R3	10121800	0.13	4.0%	0.01	0.12	1.401	0.09	44.9	2.14	3.25	0.20	0.140	38.1	36.2	204900	<5	<5	<5	1070	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	10121800	0.11	3.5%	0.00	0.11	1.401	0.08	43.1	2.50	4.25	0.24	0.170	38.0	34.7	212100	<5	<5	<5	1170	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	10131800	0.14	16.7%	0.02	0.12	1.407	0.08	55.6	1.09	1.97	0.11	0.087	37.5	35.2	215000	155	<5	<5	1408	0.01	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
R1	10131800	0.17	42.0%	0.07	0.10	1.437	0.07	49.5	2.23	4.88	0.26	0.110	38.8	33.8	228200	<5	<5	<5	928	0.04	0.00	0.00	0.00	0.00	0.03	0.02	0.02	0.00	0.00	0.00	0.00
R2	10131800	0.13	6.7%	0.01	0.12	1.430	0.09	42.2	2.71	4.44	0.27	0.160	41.9	39.5	229500	<5	<5	<5	920	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	10131800	0.13	6.6%	0.01	0.13	1.420	0.09	43.1	2.78	4.89	0.28	0.150	40.1	39.7	222500	<5	<5	<5	877	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	10131800	0.13	6.6%	0.01	0.12	1.414	0.09	42.4	2.73	4.19	0.26	0.160	40.0	39.3	218400	<5	<5	<5	840	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	10141800	0.16	27.0%	0.04	0.11	1.394	0.08	57.3	1.03	1.73	0.10	0.062	35.6	32.1	210800	70	<5	<5	807	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
R1	10141800	0.16	17.0%	0.03	0.13	1.435	0.09	46.6	2.77	4.86	0.28	0.088	27.9	23.5	235400	<5	<5	<5	486	0.01	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
R2	10141800	0.13	2.1%	0.00	0.13	1.442	0.09	47.2	3.42	4.77	0.32	0.220	12.3	8.3	242000	<5	<5	<5	529	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	10141800	0.13	2.7%	0.00	0.13	1.436	0.09	46.1	2.90	4.09	0.28	0.190	16.7	9.8	241800	<5	<5	<5	541	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	10141800	0.12	3.7%	0.00	0.12	1.430	0.08	48.6	2.69	3.78	0.25	0.180	21.3	6.1	233200	<5	<5	<5	558	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	10151800	0.15	23.3%	0.03	0.11	1.420	0.08	56.2	1.03	1.92	0.11	0.073	35.9	32.5	212100	168	<5	6	753	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
R1	10151800	0.15	0.8%	0.00	0.15	1.397	0.10	19.1	3.56	7.07	0.47	1.060	42.7	19.4	198500	644	<5	29	395	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R2	10151800	0.13	2.4%	0.00	0.13	1.420	0.09	21.2	4.09	10.50	0.53	0.300	53.0	21.3	217600	658	<5	24	426	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	10151800	0.13	2.9%	0.00	0.13	1.424	0.09	21.6	4.21	10.20	0.55	0.290	55.3	22.2	219900	564	<5	15	438	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	10151800	0.12	3.0%	0.00	0.12	1.414	0.09	21.5	4.55	9.57	0.55	0.240	53.0	19.5	213600	424	<5	8	454	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	10130000	0.17	21.8%	0.04	0.13	1.388	0.10	55.4	0.97	1.73	0.10	0.075	33.6	31.8	198000	102	102	<5	1820	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
PLS	10130000	0.15	-	-	0.15	1.390	0.11	-	-	-	-	-	-	-	197800	29	62	<5	1050	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	10131200	0.13	25.9%	0.03	0.10	1.390	0.07	54.8	1.04	1.67	0.11	0.056	36.4	33.7	188700	121	243	<5	1370	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.01	0.00	0.00	0.00	0.00
PLS	10131200	0.14	-	-	0.14	1.422	0.10	-	-	-	-	-	-	-	213800	35	<5	<5	890	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	10140000	0.18	21.3%	0.04	0.14	1.412	0.10	57.8	1.02	1.65	0.10	0.074	35.6	31.9	212000	116	229	6	1322	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
PLS	10140000	0.15	-	-	0.15	1.426	0.10	-	-	-	-	-	-	-	219800	<5	<5	<5	787	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	10141200	0.18	25.5%	0.05	0.14	1.423	0.10	56.6	1.02	1.66	0.10	0.067	34.2	30.6	212300	153	246	8	1198	0.03	0.00	0.00	0.00	0.00	0.02	0.01	0.02	0.00	0.00	0.00	0.00
PLS	10141200	0.16	-	-	0.16	1.425	0.12	-	-	-	-	-	-	-	230500	298	6	7	573	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.03	0.00	0.00	0.00	0.00
Fd	10150000	0.17	23.5%	0.04	0.13	1.427	0.09	55.8	1.06	1.69	0.10	0.072	35.1	30.7	223800	68	177	<5	899	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
PLS	10150000	0.18	-	-	0.18	1.438	0.12	-	-	-	-	-	-	-	236000	119	<5	<5	550	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.03	0.00	0.00	0.00	0.00
Fd	10151200	0.19	31.0%	0.06	0.13	1.425	0.09	58.4	1.07	1.72	0.11	-	35.2	34.3	229635	117	208	7	845	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
PLS	10151200	0.15	-	-	0.15	1.441	0.10	-	-	-	-	-	-	-	248550	<5	<5	<5	495	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.03	0.00	0.00	0.00	0.00
Fd	10160000	0.19	29.3%	0.06	0.14	1.442	0.09	56.4	1.06	1.83	0.11	0.063	35.0	34.9	227200	179	254	9	802	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
PLS	10160000	0.16	-	-	0.16	1.432	0.11	-	-	-	-	-	-	-	224500	327	16	13	442	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.03	0.00	0.00	0.00	0.00
FD	SHUTDOWN	0.13	22.2%	0.03	0.10	1.447	0.07	56.4	0.98	1.80	0.10	0.084	33.6	33.5	226100	205	314	11	839	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
PH	SHUTDOWN	0.14	28.6%	0.04	0.10	1.461	0.07	56.0	1.00	1.72	0.10	0.070	35.4	33.8	231200	487	<5	12	466	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
R1	SHUTDOWN	0.10	1.8%	0.00	0.10	1.438	0.07	25.8	3.76	6.39	0.44	0.590	46.4	19.8	225100	351	<5	5	477	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R2	SHUTDOWN	0.11	4.7%	0.01	0.11	1.441	0.07	38.5	2.72	4.97	0.31	0.110	45.0	25.4	230000	273	<5	<5	459	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	SHUTDOWN	0.12	4.4%	0.01	0.11	1.438	0.08	34.4	3.26	6.03	0.37	0.130	46.0	24.1	220300	318	<5	<5	442	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	SHUTDOWN	0.11	3.3%	0.00	0.11	1.418	0.07	28.4	3.73	6.93	0.42	0.160	50.7	24.9	232800	250	<5	10	852	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
TOTAL		8.02		0.81																0.42	0.01	0.02	0.00	0.00	0.28	0.26	1.12	0.00	0.00	0.00	0.00

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP1

Scrubbing						
Sample ID		Solution			S	S
		kg	SG	L	mg/L	kg
SL	10130000	0.09	1.169	0.08	42000	0.00
SL	10131200	0.07	1.138	0.06	41000	0.00
SL	10140000	0.06	1.077	0.05	38000	0.00
SL	10141200	0.05	1.043	0.05	32000	0.00
SL	10150000	0.05	1.023	0.05	16000	0.00
SL	10151200	0.08	1.090	0.08	27000	0.00
SL	10160000	0.11	1.086	0.10	35000	0.00
TOTAL		0.50				0.02

LEACH

	kg	Fe	Ni	Cu	Co	Mg	S	S=
TOTAL IN	183.93	32.54	0.53	0.88	0.05	0.09	17.34	17.26
TOTAL OUT	166.88	28.01	0.49	0.81	0.05	0.08	17.34	5.15
OUT/IN	91%	86%	93%	92%	96%	82%	100%	30%
EXTRACTION	**70%**	**80%**	**13%**	**8%**	**6%**	**50%**	**63%**	**70%**
EXTRACTION	**70%**	**73%**	**6%**	**0%**	**2%**	**-45%**	**63%**	

Extraction=1-Me in Residue / Me in Feed

Extraction= 1- Me in Residue / Me total out

SCRUBBING

	kg	S
TOTAL IN	322.63	10.94
TOTAL OUT	329.35	10.70
OUT/IN	102%	**98%**

		Feed										Pre-heater		R1							R2				R3		
Date	Time	Feed tank weight kg	calc flow pulp g/min	solids density %wt	calc flow ore g/min	R1 Feed Flowrate OFF min	Feed Flowrate mL/min	Feed Weight g	Feed Volume mL	Feed Density g/mL	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Acid weight kg	Acid Strength % (w/w)	calc flow 25% HCl g/min	calc flow 100% HCl g/min	HCl dose kg/t	Temp °C	N_2 flowrate mL/min	Acid weight g	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Temp °C
TARGET					**10**						**100**		**100**					**13.0**		**650.0**		**100**		**13.0**	**105**	**100**	**105**
1-Nov										-								-									
1-Nov	0:30									-								-						-			
1-Nov	1:00									-								-						-			
1-Nov	1:30									-								-						-			
1-Nov	2:00									-								-						-			
1-Nov	2:30									-								-						-			
1-Nov	3:00									-								-						-			
1-Nov	3:30									-								-						-			
1-Nov	4:00									-								-						-			
1-Nov	4:30									-								-						-			
1-Nov	5:00									-								-						-			
1-Nov	5:30									-								-						-			
1-Nov	6:00									-								-						-			
1-Nov	6:30									-								-						-			
1-Nov	7:00		-							-								-						-			
1-Nov	7:30		-							-								-						-			
1-Nov	8:00									-								-						-			
1-Nov	8:30									-								-						-			
1-Nov	9:00	55.9								-	300	71.6	150	99.8	150	21.64		-			107.0	150		-	105.8	150	106.0
1-Nov	9:30	55.8								-	300	104.7	150	105.4	150	21.58	25.6		0.0		104.0	150		-	103.9	150	103.5
1-Nov	10:00									-								-						-			
1-Nov	10:30	53.7					20	69.2	40	1.730	300	105.1	150	104.8	150	20.52	25.6	17.7	4.5	-	106.0	150		-	106.1	150	105.6
1-Nov	11:00									-								-						-			
1-Nov	11:30	51.6	34.8	28.0%	9.8		20			-	300	104.5	150	105.1	150	19.43	25.6	18.2	4.7	477	106.1	150		-	105.1	150	104.0
1-Nov	12:00									-								-						-			
1-Nov	12:30	49.6	33.0	28.0%	9.2		20	68.2	41	1.663	300	104.6	150	105.0	150	18.47	25.6	16.0	4.1	443	106.3	150		-	104.3	150	105.0
1-Nov	13:00									-								-						-			
1-Nov	13:30	47.3	37.8	28.0%	10.6		20	66.8	40	1.670	300	105.5	150	104.4	150	17.22	25.6	20.8	5.3	504	105.6	150		-	106.5	150	105.0
1-Nov	14:00									-								-						-			
1-Nov	14:30	45.1	36.7	28.0%	10.3		19	66.6	38	1.753	300	104.2	150	103.5	150	15.84	25.6	23.0	5.9	574	106.0	150		-	104.0	150	104.2
1-Nov	15:00									-								-						-			
1-Nov	15:30	43.1	34.3	28.0%	9.6		19			-	300	103.1	150	103.3	150	14.30	25.6	25.7	6.6	684	104.1	150		-	106.1	150	105.6
1-Nov	16:00									-								-						-			
1-Nov	16:30	41.1	32.7	28.0%	9.1		21	37.5	21	1.786	300	102.4	150	103.3	150	12.88	25.6	23.7	6.1	662	105.6	150		-	106.3	150	105.5
1-Nov	17:00									-								-						-			
1-Nov	17:30	38.2	48.2	28.0%	13.5		21			-	300	103.6	100	103.0	100	11.10	25.6	29.7	7.6	563	105.3	150		-	104.2	150	104.0
1-Nov	18:00									-								-						-			
1-Nov	18:30	35.9	38.8	28.0%	10.9		22	36.5	22	1.698	300	105.7	150	103.7	150	9.55	25.6	25.8	6.6	608	106.3	150		-	104.4	150	104.6
1-Nov	19:00									-								-						-			
1-Nov	19:30	33.8	35.3	28.0%	9.9		21			-	300	104.3	150	102.5	150	8.16	25.6	23.2	5.9	600	106.1	150		-	106.0	150	104.3
1-Nov	20:00									-								-						-			
1-Nov	20:30	31.3	41.8	28.0%	11.7		24	37.7	24	1.571	300	103.0	150	102.5	150	6.49	25.6	27.8	7.1	608	104.1	150		-	106.3	150	103.6
1-Nov	21:00									-								-						-			
1-Nov	21:30	57.8	-	27.9%	-	20	20			-	300	103.8	150	104.1	150	5.56	25.6	15.5	4.0	-	104.1	150		-	104.6	150	105.9
1-Nov	22:00									-								-						-			
1-Nov	22:30	56.3	24.8	27.9%	6.9	20	21	36.8	21	1.752	300	104.9	150	106.0	150	20.17	25.6	-	-	-	104.1	150		-	106.0	150	105.1
1-Nov	23:00									-								-						-			
1-Nov	23:30	54.5	29.0	27.9%	8.1		18	63.4	36	1.761	300	104.8	150	104.2	150	19.08	25.6	18.2	4.7	574	104.0	150		-	106.0	150	105.5
2-Nov	0:00									-								-						-			
2-Nov	0:30		-	27.9%	-					-							25.6	-	-	-				-			
2-Nov	1:00									-								-						-			
2-Nov	1:30	52.2	-	27.9%	-		20	68.8	40	1.720	300	106.3	150	105.1	150	17.28	25.6	-	-	-	106.0	150		-	105.4	150	105.3
2-Nov	2:00									-								-						-			
2-Nov	2:30	49.9	38.8	27.9%	10.8		19	69.0	38	1.816	300	105.7	150	102.8	150	15.80	25.6	24.7	6.3	582	105.9	150		-	104.0	150	103.7
2-Nov	3:00									-								-						-			
2-Nov	3:30	47.5	39.2	27.9%	10.9		20	35.8	20	1.790	300	103.3	150	103.0	150	14.32	25.6	24.7	6.3	577	106.1	150		-	104.9	150	104.9
2-Nov	4:00									-								-						-			
2-Nov	4:30	45.3	37.2	27.9%	10.4		19	67.8	38	1.784	300	105.6	150	103.2	150	12.83	25.6	24.8	6.4	612	105.9	150		-	105.3	150	103.9
2-Nov	5:00									-								-						-			
2-Nov	5:30	43.2	34.3	27.9%	9.6		20	68.0	39	1.744	300	103.6	150	104.1	150	11.48	25.6	22.5	5.8	601	104.6	150		-	105.2	150	103.7
2-Nov	6:00									-								-						-			
2-Nov	6:30	41.1	35.8	27.9%	10.0		19	62.7	37	1.695	300	104.6	150	104.2	150	10.13	25.6	22.5	5.8	575	106.0	150		-	104.5	150	105.6
2-Nov	7:00									-								-						-			
2-Nov	7:30	39.0	35.3	27.9%	9.9		17	58.5	34	1.721	300	105.2	150	104.5	150	8.76	25.6	22.8	5.8	592	104.0	150		-	106.2	150	104.8
2-Nov	8:00									-								-						-			
2-Nov	8:30	37.0	32.0	27.9%	8.9		18	30.8	18	1.711	300	104.7	150	103.3	150	7.42	25.6	22.3	5.7	640	105.7	150		-	102.1	150	103.8
2-Nov	9:00									-								-						-			
2-Nov	9:30	35.1	31.8	27.9%	8.9		27	91.4	54	1.693	300	105.3	150	100.6	150	6.19	25.6	20.5	5.2	590	105.9	150		-	83.6	150	104.7
2-Nov	10:00									-								-						-			
2-Nov	10:30	32.7	40.7	27.9%	11.4		20	67.7	40	1.693	300	103.3	150	103.4	150	5.06	25.6	18.8	4.8	424	103.9	150		-	104.1	150	106.2
2-Nov	11:00									-								-						-			
2-Nov	11:30	32.3	6.5	27.9%	1.8					-		104.0	150	105.7	150	21.09	25.6	-	-	-	104.8	150		-	106.0	150	106.2
2-Nov	12:00									-								-						-			
2-Nov	12:30		-		-					-							25.6	-	-	-				-			
2-Nov	13:00									-								-						-			
2-Nov	13:30	69.6	-	27.4%	-		21	67.6	41	1.649	300	109.4	150	106.6	150	19.41	23.5	-	-	-	105.6	150		-	105.9	150	103.6
2-Nov	14:00									-								-						-			
2-Nov	14:30	68.1	26.0	27.4%	7.1		20	63.9	39	1.638	300	109.2	150	106.3	150	18.89	23.5	8.7	2.0	286	104.1	150		-	104.9	150	106.1

		Feed										Pre-heater		R1							R2				R3		
Date	Time	Feed tank weight kg	calc flow pulp g/min	solids density %wt	calc flow ore g/min	R1 Feed Flowrate OFF min	Feed Flowrate mL/min	Feed Weight g	Feed Volume mL	Feed Density g/mL	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Acid weight kg	Acid Strength % (w/w)	calc flow 25% HCl g/min	calc flow 100% HCl g/min	HCl dose kg/t	Temp °C	N_2 flowrate mL/min	Acid weight g	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Temp °C
TARGET					**10**						**100**		**100**					**13.0**		**650.0**		**100**		**13.0**	**105**	**100**	**105**
2-Nov	15:00									-								-						-			
2-Nov	15:30		-	27.4%	-					-							23.5	-	-	-				-			
2-Nov	16:00									-								-						-			
2-Nov	16:30	64.2	-	27.4%	-		21	29.7	21	1.414	300	105.1	150	104.5	150	19.21	23.5	-	-	-	104.9	150		-	106.5	150	106.0
2-Nov	17:00									-								-						-			
2-Nov	17:30	64.2	-	27.4%	-		21			-	300	106.7	150	103.8	150	17.74	23.5	24.5	5.8	-	103.9	150		-	105.7	150	105.3
2-Nov	18:00									-								-						-			
2-Nov	18:30	62.4	29.0	27.4%	7.9		20	28.6	20	1.430	300	105.0	150	104.3	150	16.08	23.5	27.7	6.5	819	105.8	150		-	104.7	150	103.7
2-Nov	19:00									-								-						-			
2-Nov	19:30	60.7	28.8	27.4%	7.9		19			-	300	106.4	150	102.8	150	14.50	23.5	26.3	6.2	784	103.8	150		-	106.1	150	105.5
2-Nov	20:00									-								-						-			
2-Nov	20:30	59.1	26.0	27.4%	7.1		20	26.6	20	1.330	300	106.6	150	102.8	150	12.85	23.5	27.5	6.5	909	106.0	150		-	105.9	150	106.0
2-Nov	21:00									-								-						-			
2-Nov	21:30	57.4	28.7	27.4%	7.8	10	19			-	300	105.7	150	102.3	150	11.16	23.5	28.2	6.6	844	103.8	150		-	105.5	150	104.0
2-Nov	22:00									-								-						-			
2-Nov	22:30	56.1	21.0	27.4%	5.7		19	26.9	19	1.416	300	107.0	150	103.5	150	9.86	23.5	21.7	5.1	886	105.8	150		-	104.2	150	105.7
2-Nov	23:00									-								-						-			
2-Nov	23:30	55.1	17.3	27.4%	4.7	15				-	300	106.7	150	105.9	150	8.64	23.5	20.3	4.8	1008	106.0	150		-	104.1	150	103.8
3-Nov	0:00						29	71.1	58	1.226								-						-			
3-Nov	0:30	53.9	20.5	27.4%	5.6	25	29	89.9	58	1.550	300	104.1	150	100.6	150	7.65	23.5	16.5	3.9	691	104.8	150		-	104.6	150	106.2
3-Nov	1:00									-								-						-			
3-Nov	1:30	50.6	55.3	27.4%	15.1		28d19	48.3	28	1.725	300	104.3	150	101.3	150	5.97	23.5	28.0	6.6	435	106.1	150		-	105.9	150	104.2
3-Nov	2:00									-								-						-			
3-Nov	2:30	48.4	36.5	27.4%	10.0		17u20	58.9	34	1.732	300	106.1	150	102.7	150	20.46	23.5	-	-	-	105.3	150		-	105.0	150	104.6
3-Nov	3:00									-								-						-			
3-Nov	3:30	46.0	38.8	27.4%	10.6		20	70.0	39	1.795	300	104.8	150	100.1	150	19.16	23.5	21.7	5.1	479	105.9	150		-	106.1	150	104.0
3-Nov	4:00									-								-						-			
3-Nov	4:30	43.9	35.0	27.4%	9.6		20	35.1	20	1.755	300	106.7	150	101.4	150	17.56	23.5	26.7	6.3	654	104.8	150		-	106.3	150	106.0
3-Nov	5:00									-								-						-			
3-Nov	5:30	41.5	40.3	27.4%	11.0		20	70.6	40	1.765	300	104.3	150	101.0	150	15.93	23.5	27.2	6.4	579	103.8	150		-	104.1	150	105.7
3-Nov	6:00									-								-						-			
3-Nov	6:30	39.4	36.0	27.4%	9.8		18	62.1	35	1.774	300	106.5	150	101.0	150	14.20	23.5	28.8	6.8	688	105.3	150		-	104.5	150	104.0
3-Nov	7:00									-								-						-			
3-Nov	7:30	37.3	34.5	27.4%	9.4		20	70.2	39	1.800	300	108.7	150	100.3	150	12.65	23.5	25.8	6.1	643	104.0	150		-	104.2	150	105.8
3-Nov	8:00									-								-						-			
3-Nov	8:30	35.1	36.0	27.4%	9.8		19	67.1	37	1.814	300	108.9	150	101.0	150	11.06	23.5	26.5	6.2	632	103.7	150		-	106.1	150	105.5
3-Nov	9:00									-								-						-			
3-Nov	9:30	33.2	31.3	27.4%	8.6					-	300	109.2	150	106.7	150	9.86	23.5	20.0	4.7	548	104.8	150		-	106.3	150	104.3
3-Nov	10:00									-								-						-			
3-Nov	10:30	75.5	-	28.0%	-		32d20	55.5	32	1.734	300	105.4	150	99.4	150	8.99	23.5	14.5	3.4	-	105.7	150		-	103.9	150	104.5
3-Nov	11:00									-								-						-			
3-Nov	11:30	73.2	38.5	28.0%	10.8		20	61.1	40	1.528	300	109.7	150	101.7	150	7.41	23.5	26.3	6.2	574	106.1	150		-	105.4	150	105.6
3-Nov	12:00									-								-						-			
3-Nov	12:30	71.2	32.2	28.0%	9.0		20	59.0	40	1.475	300	108.2	150	102.1	150	5.82	23.5	26.5	6.2	691	104.2	150		-	106.0	150	105.0
3-Nov	13:00									-								-						-			
3-Nov	13:30	69.3	32.2	28.0%	9.0		21	63.6	42	1.514	300	109.3	150	104.8	150	4.19	23.50	27.2	6.4	709	106.1	150		-	106.2	150	103.1
3-Nov	14:00									-								-						-			
3-Nov	14:30	66.8	41.7	28.0%	11.7		23			-	300	104.9	150	107.6	150	18.73	23.50	-	-	-	106.0	150		-	106.2	150	103.6
3-Nov	15:00									-								-	-	-				-			
3-Nov	15:30	64.2	43.3	28.0%	12.1	20	20	27.2	20	1.360	300	107.5	150	100.3	150	17.63	23.50	18.3	4.3	355.1	103.9	150		-	105.7	150	105.7
3-Nov	16:00									-								-	-	-				-			
3-Nov	16:30	63.6	9.3	28.0%	2.6	10	23	33.0	23	1.435	300	107.9	150		150	17.08	23.50	9.2	2.2	824.3	103.7	150		-	105.7	150	105.7
3-Nov	17:00									-								-	-	-				-			
3-Nov	17:30	61.6	34.3	28.0%	9.6		23	32.8	23	1.426	300	108.2	150	103.2	150	15.22	23.50	31.0	7.3	757.8	105.5	150		-	106.0	150	103.7
3-Nov	18:00									-								-	-	-				-			
3-Nov	18:30	59.5	34.3	28.0%	9.6		23	32.6	23	1.417	300	107.4	150	105.0	150	13.85	23.50	22.8	5.4	558.2	105.1	150		-	106.1	150	105.2
3-Nov	19:00									-								-	-	-				-			
3-Nov	19:30	57.7	30.7	28.0%	8.6		22	32.1	22	1.459	300	105.7	150	104.8	150	12.24	23.50	26.8	6.3	734.4	105.0	150		-	105.2	150	105.1
3-Nov	20:00									-								-	-	-				-			
3-Nov	20:30	55.6	34.2	28.0%	9.6		24	33.4	24	1.392	300	108.8	150	100.5	150	10.96	23.50	21.3	5.0	524.1	106.3	150		-	105.0	150	104.3
3-Nov	21:00									-								-	-	-				-			
3-Nov	21:30	54.4	21.0	28.0%	5.9	25	23	32.2	23	1.400	300	104.8	150	100.8	150	9.96	23.50	16.7	3.9	666.1	104.2	150		-	104.1	150	104.2
3-Nov	22:00									-								-	-	-				-			
3-Nov	22:30	52.5	31.5	28.0%	8.8		24	31.9	24	1.329	300	106.2	150	100.6	150	8.38	23.50	26.3	6.2	701.7	105.6	150		-	104.5	150	104.1
3-Nov	23:00									-								-	-	-				-			
3-Nov	23:30	50.9	27.0	28.0%	7.6		26	66.7	51	1.308	300	103.7	150	102.0	150	6.72	23.50	27.7	6.5	860.0	104.6	150		-	103.9	150	104.6
4-Nov	0:00									-								-	-	-				-			
4-Nov	0:30	48.5	40.2	28.0%	11.2		25	67.2	49	1.371	300	106.1	150	100.5	150	5.10	23.50	27.0	6.3	564.2	103.9	150		-	105.3	150	102.6
4-Nov	1:00									-								-	-	-				-			
4-Nov	1:30	46.1	38.8	28.0%	10.9		22	72.4	44	1.645	300	106.3	150	101.3	150	17.38	21.70	-	-	-	105.7	150		-	106.2	150	103.8
4-Nov	2:00									-								-	-	-				-			
4-Nov	2:30	43.7	40.2	28.0%	11.2		21	71.2	41	1.737	300	105.1	150	100.5	150	15.77	21.70	26.8	5.8	517.8	104.3	150		-	106.0	150	105.3
4-Nov	3:00									-								-	-	-				-			
4-Nov	3:30	41.2	41.2	28.0%	11.5		22	74.4	43	1.730	300	105.4	150	101.2	150	14.04	21.70	28.8	6.3	542.8	105.5	150		-	104.7	150	104.8
4-Nov	4:00									-								-	-	-				-			
4-Nov	4:30	38.9	38.8	28.0%	10.9		22	78.2	44	1.777	300	106.3	150	100.0	150	12.72	21.70	22.0	4.8	439.1	103.9	150		-	104.1	150	105.6
4-Nov	5:00									-								-	-	-				-			
4-Nov	5:30	36.5	39.8	28.0%	11.2		20	64.7	39	1.659	300	105.5	150	99.4	150	11.08	21.70	27.3	5.9	531.8	105.4	150		-	105.2	150	105.1

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

PP2 LOGSHEET

		Feed										Pre-heater		R1							R2				R3		
Date	Time	Feed tank weight kg	calc flow pulp g/min	solids density %wt	calc flow ore g/min	R1 Feed Flowrate OFF min	Feed Flowrate mL/min	Feed Weight g	Feed Volume mL	Feed Density g/mL	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Temp °C	N_2 flowrate mL/min	Acid weight kg	Acid Strength % (w/w)	calc flow 25% HCl g/min	calc flow 100% HCl g/min	HCl dose kg/t	Temp °C	N_2 flowrate mL/min	Acid weight g	calc flow 25% HCl g/min	Temp °C	N_2 flowrate mL/min	Temp °C
TARGET					**10**						**100**		**100**					**13.0**		**650.0**		**100**		**13.0**	**105**	**100**	**105**
4-Nov	6:00									-								-	-	-				-			
4-Nov	6:30	75.1	-	28.0%	-		30	104.5	59	1.771	300	103.7	150	101.1	150	9.36	21.70	28.7	6.2	-	105.6	150		-	106.2	150	104.0
4-Nov	7:00									-								-	-	-				-			
4-Nov	7:30	72.9	37.0	28.0%	10.4		19	65.8	37	1.778	300	104.3	150	99.5	150	7.56	21.70	30.0	6.5	628.4	105.5	150		-	104.3	150	105.3
4-Nov	8:00									-								-	-	-				-			
4-Nov	8:30	70.4	42.3	28.0%	11.9		22	75.7	44	1.720	300	105.6	150	98.6	150	5.80	21.70	29.3	6.4	537.1	104.2	150		-	104.9	150	105.7
4-Nov	9:00									-								-	-	-				-			
4-Nov	9:30	68.0	39.2	28.0%	11.0		23	77.0	45	1.711	300	104.7	150	98.7	150	4.04	21.70	29.3	6.4	580.5	105.8	150		-	105.2	150	105.7
4-Nov	10:00									-								-	-	-				-			
4-Nov	10:30	65.8	37.5	28.0%	10.5		23	78.8	46	1.732	300	106.3	150	99.1	150	2.39	21.70	27.5	6.0	568.4	104.2	150		-	106.2	150	105.6
4-Nov	11:00									-								-	-	-				-			
4-Nov	11:30		-		-					-								39.8	0.0	-				-			
4-Nov	12:00									-								-	-	-				-			
4-Nov	12:30	61.0	39.3	28.0%	11.0		20	67.3	39	1.726	300	105.1	150	98.9	150	13.78	21.70	-	-	-	104.8	150		-	104.5	150	104.2
4-Nov	13:00									-								-	-	-				-			
4-Nov	13:30	59.2	31.5	28.0%	8.8		14	48.9	28	1.746	300	104.7	150	100.2	150	12.02	21.70	29.3	6.4	721.8	104.7	150		-	104.2	150	103.5
4-Nov	14:00									-								-	-	-				-			
4-Nov	14:30	57.3	31.0	28.0%	8.7		16	54.9	32	1.716	300	104.2	150	100.9	150	10.27	21.70	29.2	6.3	729.2	104.6	150		-	104.1	150	104.7
4-Nov	15:00									-								-	-	-				-			
4-Nov	15:30	55.1	37.0	28.0%	10.4		23	40.7	23	1.770	300	105.3	150	99.1	150	8.75	21.70	25.3	5.5	530.7	105.6	150		-	103.8	150	105.4
4-Nov	16:00									-								-	-	-				-			
4-Nov	16:30	52.6	41.8	28.0%	11.7		22	37.9	22	1.723	300	106.1	150	99.6	150	6.95	21.70	30.0	6.5	555.8	104.1	150		-	103.9	150	104.9
4-Nov	17:00									-								-	-	-				-			
4-Nov	17:30	50.1	41.8	28.0%	11.7		22	35.2	22	1.600	300	105.6	150	99.5	150	5.22	21.70	28.8	6.3	534.2	104.1	150		-	103.7	150	105.8
4-Nov	18:00									-								-	-	-				-			
4-Nov	18:30	47.6	40.5	28.0%	11.3		22	37.9	22	1.723	300	106.1	150	99.1	150	3.43	21.70	29.8	6.5	570.9	105.5	150		-	104.5	150	103.5
4-Nov	19:00									-								-	-	-				-			
4-Nov	19:30	45.2	40.8	28.0%	11.4		19	32.5	19	1.711	300	108.0	150	99.9	150	18.74	21.70	-	-	-	105.9	150		-	105.7	150	105.2
4-Nov	20:00									-								-	-	-				-			
4-Nov	20:30	42.7	40.8	28.0%	11.4		22	37.0	22	1.721	300	106.3	150	101.1	150	16.91	21.70	30.5	6.6	578.9	105.7	150		-	103.9	150	104.0
4-Nov	21:00									-								-	-	-				-			
4-Nov	21:30	40.2	42.8	28.0%	12.0		19	33.8	19	1.779	300	106.4	150	101.2	150	14.95	21.70	32.7	7.1	591.1	104.7	150		-	104.8	150	103.7
4-Nov	22:00									-								-	-	-				-			
4-Nov	22:30	38.3	31.5	28.0%	8.8		21	35.3	21	1.681	300	105.5	150	101.1	150	13.44	21.70	25.2	5.5	619.2	104.0	150		-	104.0	150	104.1
4-Nov	23:00									-								-	-	-				-			
4-Nov	23:30	36.1	35.8	28.0%	10.0		19	65.0	37	1.757	300	106.7	150	100.7	150	11.64	21.70	30.0	6.5	648.9	104.1	150		-	105.0	150	105.6
5-Nov	0:00									-								-	-	-				-			
5-Nov	0:30	33.8	37.8	28.0%	10.6		20	70.8	40	1.770	300	107.5	150	100.6	150	9.91	21.70	28.8	6.3	590.7	103.9	150		-	105.8	150	105.1
5-Nov	1:00									-								-	-	-				-			
5-Nov	1:30	31.0	46.7	28.0%	13.1		23	78.4	45	1.742	300	105.6	150	101.0	150	8.14	21.70	29.5	6.4	490.0	104.4	150		-	105.8	150	103.3
5-Nov	2:00									-								-	-	-				-			
5-Nov	2:30	76.0	-	27.7%	-		24	78.6	47	1.672	300	106.2	150	100.9	150	6.34	21.70	30.0	6.5	-	105.7	150		-	105.7	150	104.0
5-Nov	3:00									-								-	-	-				-			
5-Nov	3:30	74.0	33.5	27.7%	9.3		19	63.5	38	1.671	300	105.4	150	101.6	150	4.56	21.70	29.7	6.4	692.7	104.5	150		-	104.7	150	103.6
5-Nov	4:00									-								-	-	-				-			
5-Nov	4:30	72.1	32.3	27.7%	9.0		18	59.3	36	1.647	300	105.3	150	100.3	150	19.54	23.23	-	-	-	104.4	150		-	104.1	150	103.3
5-Nov	5:00									-								-	-	-				-			
5-Nov	5:30	70.1	32.7	27.7%	9.1		21	68.3	42	1.626	300	106.9	150	101.3	150	17.78	23.23	29.3	6.8	752.0	104.8	150		-	105.2	150	104.2
5-Nov	6:00									-								-	-	-				-			
5-Nov	6:30	68.2	32.3	27.7%	9.0		20	64.4	40	1.610	300	105.3	150	99.6	150	16.10	23.21	28.0	6.5	724.6	104.3	150		-	105.2	150	104.0
5-Nov	7:00									-								-	-	-				-			
5-Nov	7:30	65.8	38.7	27.7%	10.7		19	59.8	38	1.574	300	106.4	150	99.7	150	14.40	23.21	28.3	6.6	613.2	103.8	150		-	105.6	150	105.0
5-Nov	8:00									-								-	-	-				-			
5-Nov	8:30	64.0	30.8	27.7%	8.6		18	55.4	35	1.583	300	107.3	150	101.2	150	12.74	23.21	27.7	6.4	750.4	103.8	150		-	106.0	150	105.4
5-Nov	9:00									-								-	-	-				-			
5-Nov	9:30	62.1	32.3	27.7%	9.0		19	59.9	37	1.619	300	105.3	150	102.5	150	11.07	23.21	27.9	6.5	720.8	103.8	150		-	104.0	150	105.3
5-Nov	10:00									-								-	-	-				-			
5-Nov	10:30	60.4	27.0	27.7%	7.5		18	57.1	35	1.631	300	107.2	150	102.3	150	9.41	23.21	27.7	6.4	857.4	103.8	150		-	104.9	150	105.3
5-Nov	11:00									-								-	-	-				-			
5-Nov	11:30	58.5	32.0	27.7%	8.9		19	60.1	37	1.624	300	107.9	150	102.2	150	7.88	23.21	25.5	5.9	666.8	105.8	150		-	104.2	150	105.4
5-Nov	12:00									-								-	-	-				-			
5-Nov	12:30	56.5	33.8	27.7%	9.4		19	64.0	39	1.662	300	106.1	150	102.3	150	6.19	23.21	28.2	6.5	696.6	105.7	150		-	105.6	150	104.8
5-Nov	13:00									-								-	-	-				-			
5-Nov	13:30	54.5	32.5	27.7%	9.0		19	60.6	37	1.638	300	105.4	150	101.7	150	4.52	23.21	27.8	6.5	716.6	105.6	150		-	106.3	150	104.7
5-Nov	14:00									-								-	-	-				-			
5-Nov	14:30	52.4	35.8	27.7%	9.9		20	65.9	39	1.690	300	107.7	150	102.2	150	2.85	23.21	27.8	6.5	650.0	105.2	150		-	104.1	150	103.9
5-Nov	15:00									-								-	-	-				-			
5-Nov	15:30	50.1	38.3	27.7%	10.6		19	31.8	19	1.674	300	105.3	150	101.8	150	19.68	23.77	-	-	-	103.8	150		-	103.8	150	104.3
5-Nov	16:00									-								-	-	-				-			
5-Nov	16:30	48.1	33.8	27.7%	9.4		19	31.9	19	1.679	300	107.3	150	101.5	150	17.90	23.77	29.7	7.1	751.3	105.6	150		-	104.0	150	105.2
5-Nov	17:00									-								-	-	-				-			
5-Nov	17:30	45.9	35.5	27.7%	9.8		18	30.1	18	1.672	300	107.3	150	100.5	150	16.04	23.77	31.0	7.4	748.2	104.0	150		-	105.6	150	104.2
5-Nov	18:00									-								-	-	-				-			
5-Nov	18:30	46.3	-	27.7%	-		19	31.4	19	1.653	300	107.2	150	100.5	150	14.47	23.77	26.2	6.2	-	105.9	150		-	105.6	150	105.4
5-Nov	19:00									-								-	-	-				-			
5-Nov	19:30	44.3	33.7	27.7%	9.3		20	33.0	20	1.692	300	107.1	150	100.8	150	12.85	23.77	27.0	6.4	687.2	105.7	150		-	104.9	150	104.7
5-Nov	20:00					20				-								-	-	-				-			

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

PP2 LOGSHEET

		Feed										Pre-heater		R1							R2				R3		
Date	Time	Feed tank weight kg	calc flow pulp g/min	solids density %wt	calc flow ore g/min	R1 Feed Flowrate OFF min	Feed Flowrate mL/min	Feed Weight g	Feed Volume mL	Feed Density g/mL	N_2 flowrate mL/min	Temp ℃	N_2 flowrate mL/min	Temp ℃	N_2 flowrate mL/min	Acid weight kg	Acid Strength % (w/w)	calc flow 25% HCl g/min	calc flow 100% HCl g/min	HCl dose kg/t	Temp ℃	N_2 flowrate mL/min	Acid weight g	calc flow 25% HCl g/min	Temp ℃	N_2 flowrate mL/min	Temp ℃
TARGET					**10**						**100**		**100**					**13.0**		**650.0**		**100**		**13.0**	**105**	**100**	**105**
5-Nov	20:30	42.4	30.8	27.7%	8.6		23	40.8	23	1.774	300	101.7	150	99.0	150	11.69	23.77	19.3	4.6	537.3	106.1	150		-	104.1	150	105.7
5-Nov	21:00									-								-	-	-				-			
5-Nov	21:30	39.9	41.8	27.7%	11.6		22	39.1	22	1.777	300	105.8	150	100.7	150	10.12	23.77	26.2	6.2	536.0	105.0	150		-	104.6	150	103.0
5-Nov	22:00									-								-	-	-				-			
5-Nov	22:30	35.1	81.3	27.7%	22.6		23	40.4	23	1.757	300	107.7	150	101.0	150	8.41	23.77	28.5	6.8	300.3	104.7	150		-	104.6	150	104.8
5-Nov	23:00									-								-	-	-				-			
5-Nov	23:30	33.6	24.2	27.7%	6.7	2	16	60.0	32	1.875	300	108.1	150	101.0	150	6.86	23.77	25.8	6.1	916.0	105.9	150		-	104.1	150	104.3
6-Nov	0:00									-								-	-	-				-			
6-Nov	0:30	31.1	42.2	27.7%	11.7		25	93.7	49	1.912	300	108.3	150	101.4	150	5.15	23.77	28.5	6.8	579.1	104.3	150		-	105.9	150	104.0
6-Nov	1:00									-								-	-	-				-			
6-Nov	1:30		-		-					-								85.8	0.0	-				-			
6-Nov	2:00									-								-	-	-				-			
6-Nov	2:30		-		-					-								-	-	-				-			
6-Nov	3:00									-								-	-	-				-			
6-Nov	3:30		-		-					-								-	-	-				-			
6-Nov	4:00									-								-	-	-				-			
6-Nov	4:30		-		-					-								-	-	-				-			
6-Nov	5:00									-								-	-	-				-			
6-Nov	5:30		-		-					-								-	-	-				-			
6-Nov	6:00									-								-	-	-				-			
6-Nov	6:30		-		-					-								-	-	-				-			
6-Nov	7:00									-								-	-	-				-			

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

PP2 LOGSHEET

		R4		Filtration			Scrubbing		FAT	
Date	Time	pulp density g/mL	N_2 flowrate mL/min	Filtrate flowrate mL/min	Drum Volume Gallons	Filter cake wet weight g	Fresh Caustic mL/min	Discharge NaSH mL/min	final filtrate g/L HCl	Comments
TARGET		hot	100				38			
1-Nov										
1-Nov	0:30									
1-Nov	1:00									
1-Nov	1:30									
1-Nov	2:00									
1-Nov	2:30									
1-Nov	3:00									
1-Nov	3:30									
1-Nov	4:00									
1-Nov	4:30									
1-Nov	5:00									
1-Nov	5:30									
1-Nov	6:00									
1-Nov	6:30									
1-Nov	7:00									
1-Nov	7:30									
1-Nov	8:00									
1-Nov	8:30									
1-Nov	9:00		150							Started feed at 9 am but, had to shut down due to plug
1-Nov	9:30		150							Re-started feed at 9:30 am
1-Nov	10:00									
1-Nov	10:30		150	22			96			
1-Nov	11:00									
1-Nov	11:30		150							
1-Nov	12:00									
1-Nov	12:30		150	33			95			12:32 increased acid flowrate to about 21 g/min
1-Nov	13:00									
1-Nov	13:30		150	50			95			14:02 increased acid flow rate to 26 g/min
1-Nov	14:00									
1-Nov	14:30		150	94			95			
1-Nov	15:00									
1-Nov	15:30		150	21	0.00		98			
1-Nov	16:00									
1-Nov	16:30		150	24	0.00		95			
1-Nov	17:00									
1-Nov	17:30		150	23	0.00					
1-Nov	18:00									
1-Nov	18:30		150	21	0.00		95			18:45 Fresh caustic flowrate was decreased to 65mL/min
1-Nov	19:00									Filtrate free acid was stable on 10g/L. Acid addition was decreased to 23g/L
1-Nov	19:30		150	18	0.00					
1-Nov	20:00									
1-Nov	20:30		150	35	0.00		66			
1-Nov	21:00									
1-Nov	21:30		150	45	0.00					acid pail change finish 5.56 start 56.70
1-Nov	22:00									Scrubber 2 was getting colorless. Fresh caustic flowrate was decreased to 50mL/min
1-Nov	22:30		150	36	0.00		52			Feed line keeps cloging.
1-Nov	23:00									
1-Nov	23:30		150	47	0.00				22	increased feed flowrate.
2-Nov	0:00								50	
2-Nov	0:30									Circuit down. Glass in R1. NO READINGS
2-Nov	1:00									
2-Nov	1:30		150	65			50		43	
2-Nov	2:00								43	Filtrate RUSH sample taken at 02:00 not 01:00
2-Nov	2:30		150							
2-Nov	3:00									
2-Nov	3:30		150	48						03:40 FIL line keeps plugging with crystals and blowing apart.
2-Nov	4:00						52			Noticed no foaming in R1 for shift so far.
2-Nov	4:30		150						26	04:20 FA are still high in filtrate though kg acid/tonne ore seems right (670-700); acid reduced from 23 to 21 g/min.
2-Nov	5:00									Checked concentration of acid in feed pail (289.6 g HCl/L/1.13=25.57% w/w HCl)
2-Nov	5:30		150	45			54		23	
2-Nov	6:00									
2-Nov	6:30		150				52			
2-Nov	7:00									
2-Nov	7:30		150							
2-Nov	8:00									
2-Nov	8:30		150	28			50			
2-Nov	9:00									9:10 shut acid off to clean feed line, 9:13 feed back on, 9:15 acid on
2-Nov	9:30		150							9:49 decreased acid addition to ~17 g/min
2-Nov	10:00									
2-Nov	10:30		150	27			49			10:59 switched acid pails end pail wt 4.53 kg new pail wt 21.53 kg
2-Nov	11:00									
2-Nov	11:30		150							11:25 feed line plugged - acid off
2-Nov	12:00									11:40 feed and acid going again, - 11:45 feed tank wt (Batch #3) = 73.15 kg
2-Nov	12:30									12:22 acid flow decreased
2-Nov	13:00									13:08 to 13:10 acid and feed off
2-Nov	13:30		150							13:20 to 13:46 feed, acid and filtration pump off as filtration lines were wrapped with line heater
2-Nov	14:00									
2-Nov	14:30		150	19			50			

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Client: Starfield Resources
Campaign: PP2

PP2 LOGSHEET

		R4		Filtration			Scrubbing		FAT	
Date	Time	pulp density g/mL	N_2 flowrate mL/min	Filtrate flowrate mL/min	Drum Volume Gallons	Filter cake wet weight g	Fresh Caustic mL/min	Discharge NaSH mL/min	final filtrate g/L HCl	Comments
TARGET		hot	100				38			
2-Nov	15:00									
2-Nov	15:30									feed line clogged restart 14:10
2-Nov	16:00									
2-Nov	16:30		150	44	0.00		43			acid was changed to 26g/min
2-Nov	17:00									
2-Nov	17:30		150	35	0.00					
2-Nov	18:00									
2-Nov	18:30		150	27	0.00		50			
2-Nov	19:00									
2-Nov	19:30		150	18	0.00					
2-Nov	20:00									
2-Nov	20:30		150	44	0.00		50			
2-Nov	21:00									
2-Nov	21:30		150	33	0.00					some solution from scrubber 2 went into scrubber 1 so now scrubber 1 is light yellow
2-Nov	22:00									
2-Nov	22:30		150	36	0.00		50			
2-Nov	23:00									
2-Nov	23:30		150	0	0.00					Feed line problems from 23:15 to 23:55; adjusted flow rate up, replaced tubing and line.
3-Nov	0:00									23:15 Feed and Acid off while fixing feed line
3-Nov	0:30		150	30	0.00		52		57	
3-Nov	1:00								56	
3-Nov	1:30		150	46	0.00					Acid and Feed mass' taken at 01:35. Scrubber 2 darkening due to higher ore rate; Scbr 1 still off-colour from previous shift upset; Caustic rate wasincreased from 52 to 64 mL/min.
3-Nov	2:00									Changed acid pail. Final pail mass =5.26kg; New pail mass =21.23kg
3-Nov	2:30		150	38	0.00		64		6	Free acid is on spec again!!!
3-Nov	3:00									03:15 Acid pump was left off by accident for 20mins.
3-Nov	3:30		150	44	0.00				3	
3-Nov	4:00									
3-Nov	4:30		150	32	0.00		66		3	
3-Nov	5:00								2	
3-Nov	5:30		150	43	0.00				6	
3-Nov	6:00									
3-Nov	6:30		150	46	1.50		65		4	Acid and Feed mass' taken at 06:30
3-Nov	7:00									
3-Nov	7:30		150	33	2.50		64			
3-Nov	8:00									
3-Nov	8:30		150	53	3.50					8:34 feed plugged -feed & acid off, 8:38 feed on, 8:41 acid on
3-Nov	9:00									9:20 feed plugged -feed & acid off, 9:24 feed on, 9:25 acid on
3-Nov	9:30		150				60			9:28 feed plugged -feed & acid off, 9:30 feed on, 9:31 acid on
3-Nov	10:00									9:34 feed plugged -feed & acid off, 9:38 feed on, 9:39 acid on
3-Nov	10:30		150	30	4.00		60			9:58 feed plugged -feed & acid off, 10:00 feed on, 10:01 acid on
3-Nov	11:00									10:20 start new feed batch initial wt. = 76.28 kg, old batch final wt = 32.42 kg
3-Nov	11:30		150	30	3.50		59			
3-Nov	12:00									
3-Nov	12:30		150	76	4.50		60			
3-Nov	13:00									
3-Nov	13:30		150	17	4.50		60			13:37 changed out acid old wt = 4.03 kg new wt = 19.34 kg
3-Nov	14:00									13:38 BT pump not working properly increased speed to get bubbles
3-Nov	14:30		150							14:00 shut down due to no O/F from Pre-heater to #1, but once mixers were off the (froth)_ slurry started to pass through leaving PH with a lot of froth before it fell
3-Nov	15:00									
3-Nov	15:30		150							
3-Nov	16:00									
3-Nov	16:30		150	39	3.50		64			
3-Nov	17:00									
3-Nov	17:30		150	36	3.50					
3-Nov	18:00									
3-Nov	18:30	1.366	150	26	3.50		62			
3-Nov	19:00									
3-Nov	19:30		150	60	3.50					
3-Nov	20:00									
3-Nov	20:30		150	32	3.50		60			
3-Nov	21:00									
3-Nov	21:30		150	38	3.50					stopped to clean down commer between preheater and R1
3-Nov	22:00									
3-Nov	22:30		150	42	3.50		60			
3-Nov	23:00									
3-Nov	23:30		150	44	3.50					23:40 turned up feed to 27mL/min.
4-Nov	0:00									
4-Nov	0:30		150	50	3.50					FRESH Caustic IN being tweaked at this time after changing BT pump tubing in scrubber
4-Nov	1:00									01:20 Changed to new acid pail. OLDFinal mass=3.79kg, NEWInitial=17.60kg @21.7%
4-Nov	1:30		150	36	3.50		60		4	Feed pulp density is back on target.
4-Nov	2:00									
4-Nov	2:30		150	40	3.50					
4-Nov	3:00									H2S alarm due to split B/T pump tubing. See Engineers Notes. Acid and fee off for 10 minutes.
4-Nov	3:30		150	48	3.00					Scale weights taken at 3:40.
4-Nov	4:00									
4-Nov	4:30		150	36	3.00		60			
4-Nov	5:00									
4-Nov	5:30		150	42	3.50					

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Client: Starfield Resources
Campaign: PP2

PP2 LOGSHEET

		R4		Filtration			Scrubbing		FAT	
Date	Time	pulp density g/mL	N_2 flowrate mL/min	Filtrate flowrate mL/min	Drum Volume Gallons	Filter cake wet weight g	Fresh Caustic mL/min	Discharge NaSH mL/min	final filtrate g/L HCl	Comments
TARGET		hot	100				38			
4-Nov	6:00									06:15 started new feed batch (OLD mass=34.95kg, NEW mass=76.0kg)
4-Nov	6:30		150	48	3.50		62			
4-Nov	7:00									
4-Nov	7:30		150	11	4.00		60			
4-Nov	8:00									
4-Nov	8:30		150	40	4.00		60			
4-Nov	9:00									
4-Nov	9:30		150	48	4.50		60			
4-Nov	10:00									10:30 switched acid pails: old wt= 2.39 kg ; new wt= 17.27 kg.
4-Nov	10:30		150	38	4.00					10:40 Reduced feed rate to 20 ml/min
4-Nov	11:00									
4-Nov	11:30									BT pump tubing split
4-Nov	12:00									
4-Nov	12:30		150	77	4.00		60			12:45 reduced feed rate to 17 mls/min
4-Nov	13:00									
4-Nov	13:30		150	22	4.00		60			
4-Nov	14:00									
4-Nov	14:30		150	16	4.50		60			14:40 changed feed rate to 17.5 mls/min
4-Nov	15:00									
4-Nov	15:30		150							Feed was increased to 22mL/min
4-Nov	16:00									
4-Nov	16:30		150		7.50		61			
4-Nov	17:00									
4-Nov	17:30		150	47	7.50					
4-Nov	18:00									
4-Nov	18:30		150		8.00		61			Acid pail changed. Finish 3.43kg Start 20.46kg
4-Nov	19:00									
4-Nov	19:30		150		8.50					
4-Nov	20:00									
4-Nov	20:30		150	60	9.00		61			
4-Nov	21:00									
4-Nov	21:30		150		11.50					
4-Nov	22:00									
4-Nov	22:30		150		12.00		60			
4-Nov	23:00									
4-Nov	23:30		150		13.50					Changed out Spent Caustic Drum
5-Nov	0:00									
5-Nov	0:30		150	43	14.50		64			00:40 increased Caustic frate from 64 to 80mL/min as per NV.
5-Nov	1:00								9	Calculated that a new drum of NaOH will be needed before midnight tomorrow.
5-Nov	1:30		150	50	14.50					
5-Nov	2:00									Feed tank change OLD 30.0 kg NEW 77.14 kg
5-Nov	2:30		150	63	15.00		80		2	Feed flowrate decreased slightly
5-Nov	3:00									
5-Nov	3:30		150	41	16.50					03:55 Switch to new acid pail. FINAL Mass: 3.86kg, NEW Mass: 20.61kg
5-Nov	4:00									
5-Nov	4:30		150	40	17.00		80		1	
5-Nov	5:00								4	
5-Nov	5:30		150	44	17.50				6	05:30 Reduced Acid flowrate to 28mL/min.
5-Nov	6:00									
5-Nov	6:30		150	45	18.00		81			
5-Nov	7:00									
5-Nov	7:30		150	43	18.50		77			
5-Nov	8:00									
5-Nov	8:30		150	32	19.50		76			
5-Nov	9:00									
5-Nov	9:30		150	33	19.50		76			
5-Nov	10:00									
5-Nov	10:30		150	35	20.00		76			10:38 feed/acid off to replace MF tubing in feed pump
5-Nov	11:00									10:42 feed/acid back on
5-Nov	11:30		150	37	21.00		76			
5-Nov	12:00									
5-Nov	12:30		150	35	22.00		76			
5-Nov	13:00									
5-Nov	13:30		150	38	23.00		76			
5-Nov	14:00									14:04 REDUCED FRESH CAUSTIC FLOW TO 62 mL/min
5-Nov	14:30		150	33	24.50		62			14:33 old acid wt = 2.77 kg, new acid wt = 21.17 kg
5-Nov	15:00									
5-Nov	15:30		150		24.50					
5-Nov	16:00									
5-Nov	16:30		150		25.50		63			
5-Nov	17:00									
5-Nov	17:30		150		25.50					
5-Nov	18:00									
5-Nov	18:30		150		25.50		64			Some slurry was pumped out from feed tank 1 to feed tank 2.
5-Nov	19:00									
5-Nov	19:30		150		26.50					
5-Nov	20:00									

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PP2 LOGSHEET

		R4		Filtration			Scrubbing		FAT	
Date	Time	pulp density g/mL	N_2 flowrate mL/min	Filtrate flowrate mL/min	Drum Volume Gallons	Filter cake wet weight g	Fresh Caustic mL/min	Discharge NaSH mL/min	final filtrate g/L HCl	Comments
TARGET		**hot**	**100**				**38**			
5-Nov	20:30		150		27.00		63			stopped for 20 min to unclogg R1 inlet scale caught in glass fitting
5-Nov	21:00									
5-Nov	21:30		150		27.00					
5-Nov	22:00									
5-Nov	22:30		150		28.00		64			Feed flowrate jumped to 80mL/min turned back to 23mL/min
5-Nov	23:00									23:00 Stopped feed/acid for 2mins. To lower feed tank agitator.
5-Nov	23:30		150	43	29.50				0	Feed flowrate increased. PLS FA is low; likely caused by sudden feed increase at 22:30.
6-Nov	0:00									
6-Nov	0:30		150		29.50					Feed and Acid numbers are for 00:33.
6-Nov	1:00									SHUTDOWN at 00:33. Feed/Acid/Heat OFF.
6-Nov	1:30									
6-Nov	2:00									2:10 Reactors emptied, scrubber OFF, N2 OFF, power OFF to circuit
6-Nov	2:30									Noticed the N2 was not getting to R2 and R4 (partial) due to green crystals blocking inlet.
6-Nov	3:00									**Reactor contents**
6-Nov	3:30									PH combined with feed tank
6-Nov	4:00									PH slurry weight = 2674 g
6-Nov	4:30									R1 to R4 filtered as one. Solids repulped and washed. Stored as wet cake.
6-Nov	5:00									Filtrate from R1 to R4 stored.
6-Nov	5:30									R1 slurry weight = 1542.1 g
6-Nov	6:00									R2 slurry weight = 1880.2 g
6-Nov	6:30									R3 slurry weight = 1811.5 g
6-Nov	7:00									R4 slurry weight = 1815.8 g

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Client: Starfield Resources
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PP2 FEED BATCHES

Feed Batch No.	Charge number	Concentrate wet weight kg	Concentrate % H2O %wt	Concentrate dry weight kg	Filtrate target kg	Filtrate actual kg	Pulp weight kg	Pulp %solids %	Start feeding at dd-mmm hh:mm	Comments
For example		**10.3**	**3.2%**	**10.0**	**25.31**	**25.31**	**35.61**	**28.0%**		
1	6	12.6	6.05%	11.9	29.77	29.77	42.41	28.0%	01-Nov 9:30	
2	7	12.7	6.61%	11.8	29.55	29.65	42.30	27.9%	01-Nov 20:45	
3	8	12.0	5.52%	11.4	28.53	29.48	41.50	27.4%	02-Nov 11:40	
4	9	12.0	5.30%	11.4	28.69	28.69	40.73	28.0%	03-Nov 10:20	
5	10	12.1	5.69%	11.4	28.57	28.57	40.63	28.0%	04-Nov 6:15	
6	11	12.0	6.00%	11.3	28.29	28.67	40.67	27.7%	05-Nov 1:55	
7				-	-		0.00	-		
8				-	-		0.00	-		
9				-	-		0.00	-		
10				-	-		0.00	-		
11				-	-		0.00	-		
12				-	-		0.00	-		
13				-	-		0.00	-		
14				-	-		0.00	-		
15				-	-		0.00	-		
16				-	-		0.00	-		
17				-	-		0.00	-		
18				-	-		0.00	-		
19				-	-		0.00	-		
20				-	-		0.00	-		

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Client: Starfield Resources
Campaign: PP2

PP2 SHIFT LEADER COMMENTS

Date / Time	Shift Leader	Observation Comment
1 Nov. 10:30	D. Johnson	Painless start up. The feed line plugged and two impellor shafts slipped out of their chucks' embrace. The scrubber gas "pump" is set at 8.5 rather than the target because I thought that there was a stronger than normal H2S smell.
1 Nov. 12:00	D. Johnson	We took and submitted grabs rather than the composites called for on the schedule. The next composite will start at 15:00.
1 Nov. 14:00	D. Johnson	Acid feed is now at 26.1 g/min.
1 Nov. 15:00	Ernesto	Nitrogen cylinder was changed.
1 Nov. 18:45	Ernesto	Fresh caustic flowrate was decreased to 65mL/min. 95 seemed excessive though I don't have any assays to confirm it.
1 Nov. 19:00	Ernesto	Filtrate free acid was stable on 10g/L. Acid addition was decreased to 23g/L
1 Nov. 20:00	Ernesto	When the filtrate line to the feed tank was closed the line to the drum blew out because it had some crystallized solids in it (we didn't see them when we closed the line). Before closing any line make sure you check whether the other line is plugged or not.
1 Nov. 21:00	Ernesto	Having problems with the feed line for a while. Replace the tubing and turn the head upside down to prevent plugging.
1 Nov. 22:00	Ernesto	Scrubber 2 was getting colorless. Fresh caustic flowrate was decreased to 50mL/min. the scrubber assays should have been done @ 5pm, it's 10pm and it's not there. Niels please take care of this in the morning.
1 Nov. 22:30	Ernesto	I am really "ashamed" to say this but we were filling the acid pail and it overflowed. There was no damage, all the acid was inside the container and we pumped it back. Since there was no damage and no acid went on the floor I didn't fill any incident report.
1 Nov. 23:30	C. Forstner	At shift change, moved position of feed line in tank so that it pulls from long downcomer. Filtrate FA at 23:30 is 22 g/L. This might be due to feed plugging issues on previous shift. Checked FA of R2 and found it to be 50 g/L.
2 Nov. 1:00	C.Forstner	While attempting to check FA of R1, we broke the glass pipette. Shut down feeds and heat to PH and R1 from 0:20 to 0:50 to extract broken glass section (clean break – no small bits of glass swirling around in the reactor thankfully). Heat and agitation back on to PH and R1 at 0:50. Feeds on at 1:05 once at temperature.
		While down, a filtrate line burst due to crystallization near end of line (in reactor). Composite PLS sample contained substantial amounts of crystals (kept).
2 Nov. 6:00	C.Forstner	I have had some pretty rough night shifts before; clearing CCD discharges, scooping out goo from SX settlers gone to pudding.
		This night shift was worthy of the Halloween season. My next years costume is decided; I shall dress as "Crystallization".
		We battled with the feed line plugging occasionally. It seemed to come in waves as plugged slurry lines tend to do. We checked it every 15 minutes or so just to make sure it was flowing and that seemed to help. The feed density seemed on spec all shift.
		The free acid jumped from afternoon shift to our first reading (40 g/L acid from ~10 g/L). As well the filtrate was crystallizing at the end of the filtrate line. It burst several times. I have disconnected it at the tee so there is nothing to burst apart in the line we are drawing from. However it will still plug. The filtrate grabs would crystallize before we could get them titrated for acid. At 4 AM we started diluting the sample with half DI water and a few mL of the 25% acid solution. The dilution information for these samples is located on a second tab in the free acid worksheet file. We also took and kept a sample to be undiluted. We found the free acid values to be lower on these samples (20 g/L). Keep this up until Niels instructs us differently. We were unsure how you wanted this data entered on the sample logsheet. Ernesto?
		I have been trying to figure out why we are suddenly having crystallization issues. There was a feed batch change just prior to the night shift. I wonder if the issue is the feed. The consumption of acid/ore looks OK. I noticed all through my shift that there was no obvious foam band in R1. I distinctly recall the band in PP1.
		The filtrate looks somewhat yellow to me, though mostly green. Niels thought this might mean ferric; which would mean we were oxidizing. The ORP is higher than at the end of PP1 which supports the suggestion of oxidation though I do not know the threshold. The nitrogen bottle was changed on afternoon shift and has given us no issues. I turned down the scrubber pump rate by half a litre per minute. Keep an eye on the ORP and see if they settle out around 320 mV (end of PP1).
		The acid has been 21 g/min since 4ish.
2 Nov. 700	D. Johnson	Reduced the exhaust to the scrubber on the assumption that we are drawing air into the circuit and producing sulphites. Observed that the scrubber solutions are not yellow even though the caustic flow has been turned down.
2 Nov. 7:30	D. Johnson	Filtrate line blew up just after the pump. Filtrate sprayed everywhere. Nobody hurt. Installed plastic sheeting to cut down spray. Andrew and Chris installing a heating line to try to keep the filtrate warm.
2 Nov. 9:50	D. Johnson	Turned down the acid.
		Submitted a scrubber sample using the paperwork for 12:00
		Submitted the solids that Tom/Charlotte collected for a rush ICP scan.
2 Nov. 11:30	D. Johnson	Increased temperature in PH and R1 earlier. Foam began to appear when T=105.1 in R1.

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Client: Starfield Resources
Campaign: PP2

PP2 SHIFT LEADER COMMENTS

Date / Time	Shift Leader	Observation Comment
2 Nov. 11:40	D. Johnson	Popped another line. Chris got sprayed. Will shut down and trace and insulate the second filtrate line. We were down 26 mins. No foaming when we restarted.
2 Nov. 12:38	D. Johnson	Diluted existing acid as per calculation, added 1.52 kg of DI water.
2 Nov. 15:30	Ernesto	Still fighting with the feed line. Feed was down for around 30min. New feedwell was made up.
2 Nov. 15:30	Ernesto	Acid addition was increased to 26g/min. it should be kept at this value for around 8 hours no matter the filtrate free acid. Some foaming started to show up.
2 Nov. 19:30	Ernesto	Filtrated turned yellow-ish. This happened at the same time I changed filters, so it could be that the pump is sucking air and it's slightly oxidizing the ferrous.
2 Nov. 20:30	Ernesto	Feed pulp density has been decreasing gradually. It seems the pulp has some air in it because the pulp density is smaller than the filtrate density.
3 Nov. 00:00	C. Forstner	Feed line plugged at 23:00. Feed and acid off for 45 minutes to replace line and attain target solids flowrate. Targeting 10 g ore per minute as per NV instructions.
		Feed pulp density is very low and foamy. Pulp density from grab of feed tank near surface is 1220 g/L (low). Pushed feed line deeper into tank such that some of it extends beyond the bottom of the downcomer.
		PLS is yellow-green with no crystallization. FA is high (55 g/L HCl)
3 Nov. 01:30	C. Forstner	Colour in scrubber 2 is deepening and scrubber 1 is still off-colour; increased caustic rate from 52 to 64 mL/min. Change acid pail at 02:00 (diluted acid).
		Feed density is back to 1.7 g/mL. Had a bump in solids to circuit due to density increase between readings.
3 Nov. 02:30	C. Forstner	FA in PLS is 6 g/L. seeing the effect of the increased ore rate. PLS is clear and green again.
3 Nov. 03:15	C. Forstner	Acid pump was left off for ~20 minutes accidently. Noticing a lightening up of scrubber 1.
3 Nov. 06:30	C. Forstner	Circuit is happy. Filtration is behaving (5 hours on one funnel). We have had no issues with feed line plugging since foamy feed at beginning of the shift.
		The 0500 PLS grab sample was on spec FA-wise. Look out for the assay. Note to next shift: When you make up the next batch of feed, take a sample of the tank 1) before solid addition, 2) immediately after solids addition and 3) an hour after solids addition. This is at the request of NV. Continue adjusting the pulp feed rate hourly to meet the 10 g/min ore target. Continue diluting the acid with water to 23.5%.
3 Nov. 07:00	D. Johnson	R1 was at 100.2C so increased the preheat set point to 110C. No foam in T1 but some in T2.
		The R1 temp stayed at 100.2 even as the PH temp increased from 105 to 110. The R1 heater was on continuously. Hmmmn.
3 Nov. 09:30	D. Johnson	Two feed blockages since the start of shift.
3 Nov. 14:00	D. Johnson	BT Pump stopped. Chris restarted it and it works again. This is annoying. We have increased the speed to 9.2.
3 Nov. 14:30	D. Johnson	We had several cms of stable foam in the pre-heat. The feed tank also had a stable foam layer on the top (SG=1.25) and the SG at the bottom of the tank was 1.55. We reduced the mixer speed in the mix tank. Control in future will have to be by mass out of the feed tank rather than volumetric pump flows.
3 Nov. 15:30	Ernesto	Feed line got plugged several times, a new feedwell was made.
3 Nov. 18:30	Ernesto	Slurry from the preheater to R1 is going by "batches", no reason why, this is making the foaming in R1 also intermittent and some foaming is developing inside R2. We cleaned the overflow line from PH to R1 but it continued doing the same. In one of those "foam explosion" we put the H2S detector near R1 impeller and it was crazy, we increased the B/T pump flowrate but it didn't have any effect on that.
3 Nov. 20:00	Ernesto	Keep fighting the foaming, I am starting to think that the problem is inside the preheater, it's like is really "frothty" and it doesn't flow until the weight is enough to push it through the overflow (This is me thinking aloud, you can say your theory, too)
3 Nov. 21:00	Ernesto	Preheater overflow line was plugged. Stop feed, acid, heat and mixing for like 25 min. overflow fitting was taken out and cleaned out, now it's crystal clear (or almost).
4 Nov. 00:00	C. Forstner	Positioned slurry line in feed tank so that ~1 inch of it extends out of bottom of downcomer. Confirmed Dave's findings that density is greater in bottom of tank than at slurry surface. Feed density is 1.3 (up from 1.2 at shift change). No foaming issues in PH or R1 yet.
		Scrubber tubing is walking due to flattening; changed tubing.
4 Nov. 03:30	C. Forstner	H2S alarm sounded 21 pmm due to split tubing on B/T pump. Switched to vacuum to clear room of gas. Feed and acid off from 3:15 to 3:25. Changed tubing.
4 Nov. 08:00	D.Johnson	Foaming no longer a problem.
		The tubing in the BT pump has not walked since we started.
		Acid density is not what we thought yesterday (1.130 but rather 1.096).
4 Nov. 10:30	D.Johnson	BT pump hose split again. Luckily it was noticed right away and the H2S level in the enclosure only got to 1.8 ppm. Ran for an hour on the vacuum system while we decided whether to once again replace the hose. The hose is stained a chocolate brown inside. It had split at the side opposite the drive at the line of maximum bending. Decided to replace the hose but tighten it up less than the maximum recommended.
4 Nov. 10:30	D.Johnson	There is solution in the scrubber inlet line. Is this the source of our problems? Is it condensate saturated with H2S or solution that made it up through the condenser? (R1 looks like solution and solids made it up the condenser).
4 Nov. 15:30	Ernesto	Feed was increased to 22mL/min. R1 temperature dropped to 99C and it hasn't increased since then.
4 Nov. 18:45	Ernesto	Stop acid addition for like 3 min and R1 temperature increased to 103.6C. As soon as the acid was turned back on temperature started to decrease again. It seems that the mantle does not have enough power to keep the temperature over 105C (most accurate would be: the mantle is losing power). Put some isolation in R1 to see what happens.

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Client: Starfield Resources
Campaign: PP2

PP2 SHIFT LEADER COMMENTS

Date / Time	Shift Leader	Observation Comment
4 Nov. 21:00	Ernesto	R1 heating mantle has been ON for at least 2 days, I believe this could be dangerous so I changed the set point to 101.6C, it wasn't getting higher than that and now it can be OFF for a few minutes now and then.
5 Nov. 04:00	C. Forstner	Turned up the caustic pump from 64 mL/min to 80 mL/min as per NV in anticipation of reaching saturation.
		If we run at this flowrate for the remainder of the campaign, we will run out of caustic by midnight tomorrow.
		We started feeding batch 11 of the feed at 2AM. If this batch lasts 18 hours like batch 10, we will be able to run until 8PM. We have not started filling up the next feed make-up tank with filtrate.
5 Nov. 05:30	C. Forstner	Changed acid pail at 4:00. Started noticing FA in PLS increasing slightly at 5:30 (6 g/L from 1). At 5:30, dropped acid flowrate from 30 to 28 g/min to accommodate change in acid concentration of new pail (21.7 up to 23.23 g/L HCl).
5 Nov. 10:30	D. Johnson	Having trouble keeping the feed flow stable. Replaced the tubing.
5 Nov. 18:00	Ernesto	Some slurry was pumped out from feed tank 1 to feed tank 2 and the rest to a pail. Pail weight: 10.31kg (net)
5 Nov. 20:00	Ernesto	Preheater overflow line was getting plugged. Stop feed, acid, heat and mixing for like 20 min. overflow fitting was taken out and cleaned out, now it's crystal clear (or almost).

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Client: Starfield Resources
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PP2 MASS BALANCE (g/min)

		Feed								Solid assays							Solution assays					
		Availability	Slurry		Solids Density		Ore	Solution		Fe	Ni	Cu	Co	Mg	S	S=	Fe	Fe(II)	Ni	Cu	Co	Mg
Shift Starting	Shift Ending	%	mL/min	g/min	g/mL	%wt	g/min	mL/min	SG	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
01-Nov 12:00	02-Nov 00:00	94%	20	38	1.707	27.98%	10.6	19	1.431	57.0	1.07	1.81	0.11	0.065	35.8	32.5	239600	217800	310	89	8	494
02-Nov 00:00	02-Nov 12:00	100%	20	33	1.736	27.94%	9.3	17	1.446	57.9	1.04	1.67	0.10	0.064	36.5	33.4	234100	234100	445	132	12	435
02-Nov 12:00	03-Nov 00:00	97%	21	26	1.443	27.36%	7.2	13	1.448	57.8	1.06	1.75	0.10	0.053	37.5	34.5	239900	217400	656	281	21	474
03-Nov 00:00	03-Nov 12:00	97%	21	38	1.725	27.47%	10.4	19	1.447	56.4	1.11	1.80	0.11	0.067	37.3	34.1	236500	213700	694	332	22	481
03-Nov 12:00	04-Nov 00:00	92%	23	34	1.410	28.00%	9.4	17	1.439	56.6	1.07	1.85	0.11	0.063	37.5	35.2	236200	215700	597	449	23	397
04-Nov 00:00	04-Nov 12:00	100%	22	39	1.694	28.00%	11.1	20	1.435	56.1	1.02	1.70	0.10	0.070	37.1	34.8	230900	214300	472	363	15	354
04-Nov 12:00	05-Nov 00:00	100%	20	38	1.721	28.00%	10.6	19	1.435	56.7	1.01	1.71	0.10	0.069	36.0	32.8	232400	198800	406	344	12	314
05-Nov 00:00	05-Nov 12:00	100%	20	34	1.648	27.78%	9.5	18	1.402	55.8	1.07	2.02	0.11	0.085	36.9	33.5	214400	199600	374	258	10	270
05-Nov 12:00	06-Nov 00:00	97%	20	40	1.712	27.74%	11.0	20	1.412	56.2	1.02	1.76	0.10	0.072	36.9	32.4	218100	203260	436	382	14	295

		Acid				
		R1	R2	Total	Strength	Dosage
Shift Starting	Shift Ending	g/min	g/min	g/min	%wt	kg/t ore
01-Nov 12:00	02-Nov 00:00	23	0	23	25.6%	549
02-Nov 00:00	02-Nov 12:00	23	0	23	25.6%	625
02-Nov 12:00	03-Nov 00:00	23	0	23	23.7%	764
03-Nov 00:00	03-Nov 12:00	24	0	24	23.5%	537
03-Nov 12:00	04-Nov 00:00	23	0	23	23.5%	578
04-Nov 00:00	04-Nov 12:00	29	0	29	21.9%	569
04-Nov 12:00	05-Nov 00:00	29	0	29	21.7%	595
05-Nov 00:00	05-Nov 12:00	28	0	28	22.7%	679
05-Nov 12:00	06-Nov 00:00	27	0	27	23.6%	583
AVERAGE		25	0	25	23.5%	609
MIN		23	0	23	21.7%	537
MAX		29	0	29	25.6%	764
RELSTD		11%	-	11%	6%	12%

		Discharge								Solid assays							Solution assays					
		Filtrate			Residue			Wash		Fe	Ni	Cu	Co	Mg	S	S=	Fe	Fe(II)	Ni	Cu	Co	Mg
Shift Starting	Shift Ending	mL/min	SG	g/min	g (wet)	% moisture	g (dry)	g	SG	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
01-Nov 12:00	02-Nov 00:00	37	1.433	53	2778	47.0%	1472	7287	1.071	26.9	4.07	7.43	0.45	0.17	43.9	28.5	237400	217800	484	5	8	370
02-Nov 00:00	02-Nov 12:00	43	1.448	62	1592	47.2%	841	5224	1.054	19.5	4.49	10.80	0.59	0.24	50.8	27.4	217800	214420	945	187	34	319
02-Nov 12:00	03-Nov 00:00	28	1.407	40	1530	46.7%	816	5475	1.040	23.7	4.20	8.37	0.47	0.19	49.5	23.6	211300	203400	510	178	19	291
03-Nov 00:00	03-Nov 12:00	39	1.410	54	2198	49.5%	1109	5279	1.066	23.5	4.90	8.20	0.50	0.21	46.4	29.1	217500	209500	164	9	<5	282
03-Nov 12:00	04-Nov 00:00	41	1.385	57	2448	40.0%	1469	1392	1.037	30.9	3.87	6.74	0.41	0.16	45.3	28.5	203800	197500	230	5	<5	210
04-Nov 00:00	04-Nov 12:00	40	1.387	55	2937	43.1%	1672	5398	1.080	30.2	3.78	6.77	0.41	0.19	43.0	24.3	207000	199600	216	5	<5	211
04-Nov 12:00	05-Nov 00:00	44	1.384	61	2885	42.9%	1646	7158	1.041	25.2	3.72	6.96	0.41	0.19	47.1	30.1	206800	195800	255	7	<5	184
05-Nov 00:00	05-Nov 12:00	42	1.367	58	2491	47.5%	1307	5202	1.069	21.6	3.76	8.39	0.44	0.20	50.8	29.5	188200	182000	429	59	16	189
05-Nov 12:00	06-Nov 00:00	37	1.382	51	2810	45.0%	1545	1606	1.026	33.7	2.87	6.40	0.34	0.15	45.8	25.1	200500	197670	588	76	25	206

		Off gas	
		S	H2S
Shift Starting	Shift Ending	g/min	g/min
01-Nov 12:00	02-Nov 00:00	2.9	3.1
02-Nov 00:00	02-Nov 12:00	2.8	3.0
02-Nov 12:00	03-Nov 00:00	2.1	2.3
03-Nov 00:00	03-Nov 12:00	3.2	3.4
03-Nov 12:00	04-Nov 00:00	2.6	2.8
04-Nov 00:00	04-Nov 12:00	3.1	3.3
04-Nov 12:00	05-Nov 00:00	2.7	2.9
05-Nov 00:00	05-Nov 12:00	2.6	2.7
05-Nov 12:00	06-Nov 00:00	3.1	3.3

S (g/min) in the off gas was calculated as the difference between S (g/min) in the feed and total S (g/min) in the residue.

		TOTAL IN - g/min								Solids - g/min							Solution - g/min				
Shift Starting	Shift Ending	mass	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
01-Nov 12:00	02-Nov 00:00	60.4	10.6	0.12	0.19	0.01	0.02	3.8	3.4	6.0	0.1	0.2	0.012	0.007	3.8	3.4	4.6	0.006	0.002	0.0002	0.009
02-Nov 00:00	02-Nov 12:00	55.8	9.2	0.10	0.16	0.01	0.01	3.4	3.1	5.4	0.1	0.2	0.009	0.006	3.4	3.1	3.9	0.007	0.002	0.0002	0.007
02-Nov 12:00	03-Nov 00:00	49.3	7.3	0.08	0.13	0.01	0.01	2.7	2.5	4.1	0.1	0.1	0.007	0.004	2.7	2.5	3.1	0.009	0.004	0.0003	0.006
03-Nov 00:00	03-Nov 12:00	61.8	10.4	0.13	0.19	0.01	0.02	3.9	3.6	5.9	0.1	0.2	0.011	0.007	3.9	3.6	4.5	0.013	0.006	0.0004	0.009
03-Nov 12:00	04-Nov 00:00	56.6	9.3	0.11	0.18	0.01	0.01	3.5	3.3	5.3	0.1	0.2	0.010	0.006	3.5	3.3	4.0	0.010	0.008	0.0004	0.007
04-Nov 00:00	04-Nov 12:00	68.3	10.8	0.12	0.20	0.01	0.01	4.1	3.8	6.2	0.1	0.2	0.011	0.008	4.1	3.8	4.6	0.009	0.007	0.0003	0.007
04-Nov 12:00	05-Nov 00:00	67.0	10.4	0.11	0.19	0.01	0.01	3.8	3.5	6.0	0.1	0.2	0.011	0.007	3.8	3.5	4.4	0.008	0.007	0.0002	0.006
05-Nov 00:00	05-Nov 12:00	62.6	9.1	0.11	0.20	0.01	0.01	3.5	3.2	5.3	0.1	0.2	0.010	0.008	3.5	3.2	3.8	0.007	0.005	0.0002	0.005
05-Nov 12:00	06-Nov 00:00	66.6	10.6	0.12	0.20	0.01	0.01	4.0	3.6	6.2	0.1	0.2	0.011	0.008	4.0	3.6	4.4	0.009	0.008	0.0003	0.006

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

PP2 MASS BALANCE (g/min)

		TOTAL OUT - g/min								Solids - g/min							Solution - g/min				
Shift Starting	Shift Ending	mass	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
01-Nov 12:00	02-Nov 00:00	60.3	9.7	0.10	0.15	0.01	0.02	3.8	0.6	0.5	0.1	0.2	0.0	0.0	0.9	0.6	9.1	0.019	0.000	0.000	0.014
02-Nov 00:00	02-Nov 12:00	66.7	9.6	0.09	0.13	0.01	0.02	3.4	0.3	0.2	0.1	0.1	0.0	0.0	0.6	0.3	9.4	0.041	0.008	0.001	0.014
02-Nov 12:00	03-Nov 00:00	44.4	6.4	0.06	0.10	0.01	0.01	2.7	0.3	0.3	0.0	0.1	0.0	0.0	0.6	0.3	6.2	0.015	0.005	0.001	0.008
03-Nov 00:00	03-Nov 12:00	60.9	9.0	0.08	0.13	0.01	0.01	3.9	0.4	0.4	0.1	0.1	0.0	0.0	0.7	0.4	8.6	0.007	0.000	0.000	0.011
03-Nov 12:00	04-Nov 00:00	63.0	9.2	0.09	0.14	0.01	0.01	3.5	0.6	0.6	0.1	0.1	0.0	0.0	0.9	0.6	8.6	0.010	0.000	0.000	0.009
04-Nov 00:00	04-Nov 12:00	62.4	9.2	0.10	0.16	0.01	0.01	4.1	0.6	0.7	0.1	0.2	0.0	0.0	1.0	0.6	8.5	0.009	0.000	0.000	0.009
04-Nov 12:00	05-Nov 00:00	68.4	10.0	0.10	0.16	0.01	0.01	3.8	0.7	0.6	0.1	0.2	0.0	0.0	1.1	0.7	9.4	0.012	0.000	0.000	0.008
05-Nov 00:00	05-Nov 12:00	63.9	8.6	0.09	0.15	0.01	0.01	3.5	0.5	0.4	0.1	0.2	0.0	0.0	0.9	0.5	8.2	0.019	0.003	0.001	0.008
05-Nov 12:00	06-Nov 00:00	58.6	8.4	0.08	0.14	0.01	0.01	4.0	0.5	0.7	0.1	0.1	0.0	0.0	1.0	0.5	7.7	0.023	0.003	0.001	0.008

		OUT/IN - %							
Shift Starting	Shift Ending	mass	Fe	Ni	Cu	Co	Mg	S	S=
01-Nov 12:00	02-Nov 00:00	100%	92%	86%	79%	81%	109%	100%	17%
02-Nov 00:00	02-Nov 12:00	120%	104%	90%	86%	89%	126%	100%	10%
02-Nov 12:00	03-Nov 00:00	90%	88%	74%	78%	79%	106%	100%	11%
03-Nov 00:00	03-Nov 12:00	99%	87%	64%	65%	67%	90%	100%	13%
03-Nov 12:00	04-Nov 00:00	111%	99%	80%	76%	80%	96%	100%	18%
04-Nov 00:00	04-Nov 12:00	91%	85%	79%	81%	86%	88%	100%	15%
04-Nov 12:00	05-Nov 00:00	102%	96%	84%	85%	89%	96%	100%	20%
05-Nov 00:00	05-Nov 12:00	102%	94%	80%	79%	82%	92%	100%	17%
05-Nov 12:00	06-Nov 00:00	88%	80%	70%	70%	73%	80%	100%	15%
AVERAGE		100%	92%	79%	77%	81%	98%	100%	15%
MIN		88%	80%	64%	65%	67%	80%	100%	10%
MAX		120%	104%	90%	86%	89%	126%	100%	20%
RELSTD		10%	8%	10%	8%	9%	14%	0%	21%

		EXTRACTION - %							
Shift Starting	Shift Ending	Weight loss	Fe	Ni	Cu	Co	Mg	S	S=
01-Nov 12:00	02-Nov 00:00	83%	91%	26%	21%	21%	49%	76%	83%
02-Nov 00:00	02-Nov 12:00	90%	96%	46%	18%	25%	53%	82%	90%
02-Nov 12:00	03-Nov 00:00	89%	94%	37%	24%	26%	43%	79%	89%
03-Nov 00:00	03-Nov 12:00	87%	94%	35%	33%	33%	54%	82%	87%
03-Nov 12:00	04-Nov 00:00	82%	88%	21%	21%	19%	45%	74%	82%
04-Nov 00:00	04-Nov 12:00	85%	89%	22%	16%	14%	43%	76%	85%
04-Nov 12:00	05-Nov 00:00	80%	90%	21%	12%	12%	41%	72%	80%
05-Nov 00:00	05-Nov 12:00	83%	93%	33%	21%	24%	55%	74%	83%
05-Nov 12:00	06-Nov 00:00	85%	88%	45%	29%	33%	59%	76%	85%
AVERAGE		85%	91%	32%	22%	23%	49%	77%	85%
MIN		80%	88%	21%	12%	12%	41%	72%	80%
MAX		90%	96%	46%	33%	33%	59%	82%	90%
RELSTD		4%	3%	31%	29%	33%	13%	5%	4%

Extraction=1-Me in Residue / Me in Feed

		EXTRACTION - %						
Shift Starting	Shift Ending	Weight loss	Fe	Ni	Cu	Co	Mg	S
01-Nov 12:00	02-Nov 00:00	83%	89%	13%	0%	2%	58%	76%
02-Nov 00:00	02-Nov 12:00	90%	96%	39%	4%	16%	70%	82%
02-Nov 12:00	03-Nov 00:00	89%	92%	12%	2%	5%	51%	79%
03-Nov 00:00	03-Nov 12:00	87%	92%	0%	0%	0%	39%	82%
03-Nov 12:00	04-Nov 00:00	82%	88%	0%	0%	0%	40%	74%
04-Nov 00:00	04-Nov 12:00	85%	85%	0%	0%	0%	27%	76%
04-Nov 12:00	05-Nov 00:00	80%	90%	4%	0%	0%	36%	72%
05-Nov 00:00	05-Nov 12:00	83%	92%	15%	0%	6%	49%	74%
05-Nov 12:00	06-Nov 00:00	85%	82%	18%	0%	9%	38%	76%
AVERAGE		85%	89%	11%	1%	4%	45%	77%
MIN		80%	82%	0%	0%	0%	27%	72%
MAX		90%	96%	39%	4%	16%	70%	82%
RELSTD		4%	5%	111%	226%	131%	29%	5%

Extraction=1-Me in Residue / Me TOTAL OUT

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

PP2 OVERALL MASS BALANCE

STREAMS IN

Feed				Solids assays							Solids - kg						
Batch	wet	moisture	dry	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	S	S=
No.	kg	%wt	kg	%	%	%	%	%	%	%	kg	kg	kg	kg	kg	kg	kg
6	12.64	6.0%	11.87	56.4	1.02	1.67	0.10	0.070	34.6	33.7	6.70	0.12	0.20	0.01	0.01	4.11	4.00
7	12.65	6.6%	11.82	56.6	1.04	1.64	0.10	0.070	34.8	34.5	6.69	0.12	0.19	0.01	0.01	4.11	4.08
8	12.02	5.5%	11.35	58.0	1.06	1.72	0.11	0.079	36.6	34.4	6.58	0.12	0.20	0.01	0.01	4.16	3.91
9	12.04	5.3%	11.40	56.6	1.03	1.72	0.10	0.078	36.0	33.2	6.46	0.12	0.20	0.01	0.01	4.11	3.79
10	12.06	5.7%	11.38	54.7	1.03	1.81	0.10	0.081	36.2	32.3	6.22	0.12	0.21	0.01	0.01	4.12	3.67
11	12.00	6.0%	11.28	55.4	1.01	1.67	0.10	0.073	35.9	33.8	6.25	0.11	0.19	0.01	0.01	4.05	3.81
TOTAL	73.42		69.11								38.90	0.71	1.18	0.07	0.05	24.65	23.26

PP1 Filtrate				Solution assays					Solution - kg				
Drum	weight	SG	volume	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg
No.	kg	-	L	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg
PP1	37.80	1.430	26.43	226700	337	17	11	534	5.99	0.01	0.00	0.00	0.01

Acid			
Drum	weight	SG	volume
	kg	-	L
PP2	162	1.125	144.31

Fresh Caustic			
Drum	weight	SG	volume
	kg	-	L
PP2	471	1.098	429

STREAMS OUT

Filter Cake				Solid assays							Solution assays					Solids - kg							Solution - kg				
Batch	wet	moisture	dry	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
No.	kg	%wt	kg	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg
02-Nov AM	2.78	47.0%	1.47	26.9	4.07	7.43	0.45	0.170	43.9	28.5	237400	484	<5	8	370	0.40	0.06	0.11	0.01	0.00	0.65	0.42	0.22	0.00	0.00	0.00	0.00
02-Nov PM	1.59	47.2%	0.84	19.5	4.49	10.80	0.59	0.240	50.8	27.4	217800	945	187	34	319	0.16	0.04	0.09	0.00	0.00	0.43	0.23	0.11	0.00	0.00	0.00	0.00
03-Nov AM	1.53	46.7%	0.82	23.7	4.20	8.37	0.47	0.190	49.5	23.6	211300	510	178	19	291	0.19	0.03	0.07	0.00	0.00	0.40	0.19	0.11	0.00	0.00	0.00	0.00
03-Nov PM	2.20	49.5%	1.11	23.5	4.90	8.20	0.50	0.210	46.4	29.1	217500	164	9	<5	282	0.26	0.05	0.09	0.01	0.00	0.51	0.32	0.17	0.00	0.00	0.00	0.00
04-Nov AM	2.45	40.0%	1.47	30.9	3.87	6.74	0.41	0.160	45.3	28.5	203800	230	<5	<5	210	0.45	0.06	0.10	0.01	0.00	0.67	0.42	0.14	0.00	0.00	0.00	0.00
04-Nov PM	2.94	43.1%	1.67	30.2	3.78	6.77	0.41	0.190	43.0	24.3	207000	216	<5	<5	211	0.50	0.06	0.11	0.01	0.00	0.72	0.41	0.19	0.00	0.00	0.00	0.00
05-Nov AM	2.88	42.9%	1.65	25.2	3.72	6.96	0.41	0.190	47.1	30.1	206800	255	7	<5	184	0.41	0.06	0.11	0.01	0.00	0.78	0.50	0.19	0.00	0.00	0.00	0.00
05-Nov PM	2.49	47.5%	1.31	21.6	3.76	8.39	0.44	0.200	50.8	29.5	188200	429	59	16	189	0.28	0.05	0.11	0.01	0.00	0.66	0.39	0.16	0.00	0.00	0.00	0.00
06-Nov AM	2.81	45.0%	1.55	33.7	2.87	6.40	0.34	0.150	45.8	25.1	200500	588	76	25	206	0.52	0.04	0.10	0.01	0.00	0.71	0.39	0.18	0.00	0.00	0.00	0.00
SHUTDOWN	0.29	26.4%	0.21	43.3	2.18	3.55	0.21	0.140	39.6	32.5	215200	<5	<5	<5	201	0.09	0.00	0.01	0.00	0.00	0.08	0.07	0.01	0.00	0.00	0.00	0.00
TOTAL	21.96		12.09													3.28	0.47	0.90	0.05	0.02	5.61	3.33	1.48	0.00	0.00	0.00	0.00

Filtrate				Solution assays					Solution - kg				
PP2	weight	SG	volume	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg
	kg	-	L	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg
Drum	142	1.368	104	192400	617	70	26	197	19.94	0.06	0.01	0.00	0.02
Pail to SL	24	1.387	18	207000	216	<5	<5	211	3.63	0.00	0.00	0.00	0.00
TOTAL	166		121						23.57	0.07	0.01	0.00	0.02

H_2S stream		
Campaign	S	H_2S
	kg	kg
PP1	14.66	15.58

Spent Caustic					
Drum	weight	SG	volume	S	S
	kg	-	L	mg/L	kg
Drum 5548	143	1.077	133.11	26000	3.46
Drum 6165	118	1.084	108.81	25000	2.72
Drum 6501	197	1.069	184.42	30000	5.53
TOTAL	458.45		426.35		11.71

FINAL INVENTORY

Feed							Solid assays							Solution assays					Solids - kg							Solution - kg				
Feed	Slurry	Solids		Solution			Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
Tank	kg	%wt	kg	kg	SG	L	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg
2	13.09	19.87%	2.60	10.49	1.402	7.5	55.8	1.07	2.02	0.11	0.085	36.9	33.5	214400	374	258	10	270	1.45	0.03	0.05	0.00	0.00	0.96	0.87	1.60	0.00	0.00	0.00	0.00
1	17.21	39.61%	6.82	10.39	1.419	7.3	57.5	1.00	1.39	0.10	0.043	37.6	33.8	222400	431	399	14	300	3.92	0.07	0.09	0.01	0.00	2.56	2.30	1.63	0.00	0.00	0.00	0.00
TOTAL	30.30		9.42																5.37	0.10	0.15	0.01	0.01	3.52	3.18	3.23	0.01	0.00	0.00	0.00

Reactors							Solid assays							Solution assays					Solids - kg							Solution - kg				
Reactors	Slurry	Solids		Solution			Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
ID	kg	%wt	kg	kg	SG	L	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg
PH	2.67	33.4%	0.89	1.78	1.430	1.2	56.3	0.98	1.56	0.10	0.065	37.3	33.2	233400	489	<5	14	311	0.50	0.01	0.01	0.00	0.00	0.33	0.30	0.29	0.00	0.00	0.00	0.00
R1	1.54	5.1%	0.08	1.46	1.408	1.0	42.9	2.73	4.64	0.29	0.140	39.8	26.3	216300	263	<5	<5	197	0.03	0.00	0.00	0.00	0.00	0.03	0.02	0.22	0.00	0.00	0.00	0.00
R2	1.88	4.2%	0.08	1.80	1.396	1.3	39.6	3.69	6.08	0.37	0.210	38.4	27.5	209000	92	<5	<5	186	0.03	0.00	0.00	0.00	0.00	0.03	0.02	0.27	0.00	0.00	0.00	0.00
R3	1.81	5.3%	0.10	1.71	1.397	1.2	40.4	3.04	4.90	0.29	0.170	39.7	32.3	212700	<5	<5	<5	190	0.04	0.00	0.00	0.00	0.00	0.04	0.03	0.26	0.00	0.00	0.00	0.00
R4	1.82	7.4%	0.13	1.68	1.405	1.2	42.5	2.36	3.83	0.23	0.150	41.4	33.8	215200	<5	<5	<5	201	0.06	0.00	0.01	0.00	0.00	0.06	0.05	0.26	0.00	0.00	0.00	0.00
TOTAL	9.72		1.28	8.44		6.00													0.66	0.02	0.03	0.00	0.00	0.49	0.42	1.30	0.00	0.00	0.00	0.00

Scrubbers					
Scrubber	weight	SG	volume	S	S
	kg	-	L	mg/L	kg
Scrubber 1	13	1.093	12.01	5	0.00
Scrubber 2	14	1.086	12.70	21000	0.27
TOTAL	26.92		24.71		0.27

SAMPLING

Leaching								Solid assays							Solution assays					Solids - kg							Solution - kg				
Sample		Slurry	Solids		Solution			Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg	Fe	Ni	Cu	Co	Mg	S	S=	Fe	Ni	Cu	Co	Mg
ID		kg	%wt	kg	kg	SG	L	%	%	%	%	%	%	%	mg/L	mg/L	mg/L	mg/L	mg/L	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg	kg
PLS	11011300	0.12	-	-	0.12	1.456	0.08	-	-	-	-	-	-	-	230700	<5	<5	<5	428	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11011700	0.12	-	-	0.12	1.441	0.08	-	-	-	-	-	-	-	223800	577	21	10	367	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	11012100	0.16	3.1%	0.01	0.16	1.452	0.11	23.6	-	9.49	-	-	25.6	25.6	244300	440	<5	<5	366	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.03	0.00	0.00	0.00	0.00
PLS	11020200	0.08	-	-	0.08	1.407	0.06	-	-	-	-	-	-	-	208000	885	207	33	328	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	11020500	0.00	-	-	0.00	1.417	0.00	-	-	-	-	-	-	-	123700	460	69	19	152	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
R4 DC	11020900	0.10	1.3%	0.00	0.10	1.239	0.08	18.5	-	10.70	-	-	28.2	28.2	121600	432	10	13	142	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	11021300	0.09	-	-	0.09	1.424	0.06	-	-	-	-	-	-	-	192500	885	233	34	312	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	11021700	0.12	-	-	0.12	1.457	0.08	-	-	-	-	-	-	-	243900	23	40	<5	321	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	11022100	0.14	1.7%	0.00	0.14	1.374	0.10	19.3	-	12.00	-	-	21.5	21.5	181000	845	107	34	239	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11030100	0.11	-	-	0.11	1.353	0.08	-	-	-	-	-	-	-	175100	876	184	33	238	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	11030500	0.14	-	-	0.14	1.411	0.10	-	-	-	-	-	-	-	225300	92	13	<5	280	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	11030900	0.10	2.5%	0.00	0.10	1.397	0.07	24.9	-	8.16	-	-	21.4	21.4	215000	181	<5	<5	270	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11031300	0.13	-	-	0.13	1.399	0.10	-	-	-	-	-	-	-	219100	491	6	5	238	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11031700	0.09	-	-	0.09	1.397	0.06	-	-	-	-	-	-	-	217400	<5	<5	<5	237	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
R4 DC	11032100	0.14	3.8%	0.01	0.14	1.379	0.10	32.2	-	7.77	-	-	30.3	30.3	213900	367	<5	<5	225	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11040100	0.10	-	-	0.10	1.380	0.07	-	-	-	-	-	-	-	201100	582	<5	7	202	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	11040500	0.11	-	-	0.11	1.388	0.08	-	-	-	-	-	-	-	208600	118	<5	<5	220	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	11040900	0.11	2.4%	0.00	0.11	1.369	0.08	28.8	-	8.48	-	-	25.6	25.6	196100	474	<5	<5	201	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11041300	0.11	-	-	0.11	1.383	0.08	-	-	-	-	-	-	-	208100	32	<5	<5	185	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11041700	0.13	-	-	0.13	1.375	0.09	-	-	-	-	-	-	-	203300	99	9	<5	194	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	11042100	0.12	2.4%	0.00	0.11	1.398	0.08	21.0	-	10.00	-	-	23.0	23.0	208400	574	<5	13	203	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11050100	0.10	-	-	0.10	1.398	0.07	-	-	-	-	-	-	-	213900	768	10	17	194	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11050500	0.12	-	-	0.12	1.385	0.09	-	-	-	-	-	-	-	208400	26	<5	<5	200	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	11050900	0.15	1.8%	0.00	0.15	1.346	0.11	18.4	-	11.10	-	-	23.8	23.8	175800	682	74	26	188	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11051300	0.12	-	-	0.12	1.337	0.09	-	-	-	-	-	-	-	167700	639	109	27	183	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
PLS	11051700	0.15	-	-	0.15	1.337	0.11	-	-	-	-	-	-	-	187000	760	53	29	191	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4 DC	11052100	0.15	2.7%	0.00	0.14	1.373	0.10	19.0	-	10.90	-	-	24.7	24.7	188600	774	12	28	197	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PLS	11060100	0.11	-	-	0.11	1.416	0.08	-	-	-	-	-	-	-	217700	<5	<5	<5	205	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Pre-solids	11031000	0.17	11.2%	0.02	0.15	1.429	0.10	56.6	1.07	1.62	0.11	0.043	39.2	35.6	235000	568	145	18	372	0.01	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
Solids	11031000	0.20	27.8%	0.05	0.14	1.426	0.10	56.9	1.06	1.74	0.11	0.054	37.5	35.2	223200	569	149	18	387	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
30 min	11031030	0.19	27.5%	0.05	0.14	1.429	0.10	57.0	1.06	1.76	0.11	0.056	37.7	33.7	228300	571	166	19	402	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
R1	11011800	0.10	2.1%	0.00	0.10	1.398	0.07	20.7	4.88	10.80	0.62	0.250	53.5	25.1	201600	677	23	18	313	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
R2	11011800	0.12	2.7%	0.00	0.12	1.442	0.08	23.3	4.66	9.69	0.57	0.240	48.0	25.1	238900	597	21	9	332	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	11011800	0.11	2.5%	0.00	0.11	1.439	0.08	22.4	4.76	9.47	0.58	0.220	50.8	25.6	245800	551	20	<5	336	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	11011800	0.14	2.7%	0.00	0.14	1.437	0.09	21.8	4.78	9.73	0.58	0.210	51.9	26.1	238600	598	17	7	342	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	11011800	0.15	26.9%	0.04	0.11	1.420	0.08	57.0	1.03	1.65	0.11	0.065	36.7	34.3	234900	273	17	6	580	0.02	0.00	0.00	0.00	0.00	0.02	0.01	0.02	0.00	0.00	0.00	0.00
R1	11021800	0.11	1.2%	0.00	0.11	1.357	0.08	19.0	4.71	8.61	0.59	0.230	58.0	27.1	179400	732	21	29	199	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.00	0.00	0.00	0.00
R2	11021800	0.14	1.9%	0.00	0.14	1.392	0.10	22.2	4.87	10.00	0.58	0.220	51.6	38.2	203800	672	6	22	241	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	11021800	0.13	2.5%	0.00	0.12	1.401	0.09	27.7	4.87	9.16	0.53	0.200	48.8	34.9	208300	407	<5	7	266	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	11021800	0.12	3.1%	0.00	0.12	1.414	0.09	32.5	4.73	7.86	0.46	0.140	45.8	37.1	214200	154	<5	<5	450	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	11021800	0.15	22.3%	0.03	0.11	1.443	0.08	57.3	1.09	1.87	0.11	0.059	37.8	34.7	250400	602	<5	18	288	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
R1	11031800	0.13	0.9%	0.00	0.13	1.335	0.09	24.3	4.44	6.78	0.51	0.390	51.0	33.8	188700	669	<5	26	186	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R2	11031800	0.14	3.1%	0.00	0.13	1.379	0.10	29.2	4.95	9.09	0.55	0.320	46.3	29.3	201100	450	<5	7	224	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	11031800	0.12	3.9%	0.00	0.12	1.383	0.09	37.0	3.94	6.43	0.38	0.150	44.3	29.3	213900	38	<5	<5	224	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	11031800	0.12	3.8%	0.00	0.12	1.391	0.09	39.2	3.49	5.41	0.34	0.140	42.6	32.2	216200	<5	<5	<5	229	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	11031800	0.19	23.8%	0.05	0.15	1.418	0.10	57.2	1.09	1.86	0.11	0.067	38.2	33.3	243000	538	6	17	391	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.03	0.00	0.00	0.00	0.00
R1	11041800	0.12	3.7%	0.00	0.11	1.356	0.08	37.8	2.72	4.83	0.29	0.140	45.5	30.3	189800	327	<5	<5	178	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R2	11041800	0.13	5.6%	0.01	0.13	1.364	0.09	41.2	2.75	4.81	0.28	0.120	43.3	29.5	187200	102	<5	<5	178	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	11041800	0.13	4.9%	0.01	0.12	1.363	0.09	38.9	3.04	4.79	0.30	0.130	42.5	27.4	196500	6	<5	<5	187	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	11041800	0.12	4.5%	0.01	0.12	1.362	0.08	38.4	3.21	4.86	0.31	0.140	42.0	27.0	190500	<5	<5	<5	189	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	11041800	0.16	25.6%	0.04	0.12	1.440	0.08	56.1	1.02	1.80	0.10	0.065	37.3	34.1	236300	423	<5	11	330	0.02	0.00	0.00	0.00	0.00	0.02	0.01	0.02	0.00	0.00	0.00	0.00
R1	11051800	0.12	2.2%	0.00	0.11	1.358	0.08	42.9	2.73	4.64	0.29	0.140	39.8	26.3	185200	718	<5	28	158	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R2	11051800	0.13	2.0%	0.00	0.13	1.370	0.09	39.6	3.69	6.08	0.37	0.210	38.4	27.5	194100	759	9	31	177	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	11051800	0.12	2.1%	0.00	0.12	1.364	0.09	40.4	3.04	4.90	0.29	0.170	39.7	32.3	188100	750	14	30	182	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	11051800	0.12	2.0%	0.00	0.11	1.365	0.08	42.5	2.36	3.83	0.23	0.150	41.4	33.8	187200	736	21	29	184	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	11051800	0.16	25.1%	0.04	0.12	1.423	0.09	56.3	0.98	1.56	0.10	0.065	37.3	33.2	234900	460	<5	15	295	0.02	0.00	0.00	0.00	0.00	0.02	0.01	0.02	0.00	0.00	0.00	0.00
Fd	11011200	0.12	25.4%	0.03	0.09	1.422	0.06	57.1	1.02	1.66	0.12	0.074	35.2	33.1	221300	82	8	268	552	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.01	0.00	0.00	0.00	0.00
PLS	11011200	0.10	-	-	0.10	1.456	0.07	-	-	-	-	-	-	-	228100	7	<5	<5	489	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	11020000	0.21	25.5%	0.05	0.16	1.431	0.11	57.0	1.07	1.81	0.11	0.065	35.8	32.5	239600	310	89	8	494	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.03	0.00	0.00	0.00	0.00
PLS	11020000	0.18	-	-	0.18	1.433	0.12	-	-	-	-	-	-	-	237400	484	<5	8	370	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.03	0.00	0.00	0.00	0.00
Fd	11021200	0.19	29.8%	0.06	0.13	1.446	0.09	57.9	1.04	1.67	0.10	0.064	36.5	33.4	234100	445	132	12	435	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
PLS	11021200	0.16	-	-	0.16	1.448	0.11	-	-	-	-	-	-	-	217800	945	187	34	319	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	11030000	0.13	24.2%	0.03	0.10	1.448	0.07	57.8	1.06	1.75	0.10	0.053	37.5	34.5	239900	656	281	21	474	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
PLS	11030000	0.15	-	-	0.15	1.407	0.10	-	-	-	-	-	-	-	211300	510	178	19	291	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	11031200	0.18	25.6%	0.05	0.13	1.447	0.09	56.4	1.11	1.80	0.11	0.067	37.3	34.1	236500	694	332	22	481	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
PLS	11031200	0.16	-	-	0.16	1.410	0.11	-	-	-	-	-	-	-	217500	164	9	<5	282	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	11040000	0.15	29.3%	0.04	0.10	1.439	0.07	56.6	1.07	1.85	0.11	0.063	37.5	35.2	236200	597	449	23	397	0.02	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
PLS	11040000	0.15	-	-	0.15	1.385	0.11	-	-	-	-	-	-	-	203800	230	<5	<5	210	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	11041200	0.18	22.8%	0.04	0.14	1.435	0.10	56.1	1.02	1.70	0.10	0.070	37.1	34.8	230900	472	363	15	354	0.02	0.00	0.00	0.00	0.00	0.02	0.01	0.02	0.00	0.00	0.00	0.00
PLS	11041200	0.16	-	-	0.16	1.387	0.11	-	-	-	-	-	-	-	207000	216	<5	<5	211	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	11050000	0.18	24.3%	0.04	0.14	1.435	0.09	56.7	1.01	1.71	0.10	0.069	36.0	33.8	232400	406	344	12	314	0.02	0.00	0.00	0.00	0.00	0.02	0.01	0.02	0.00	0.00	0.00	0.00
PLS	11050000	0.14	-	-	0.14	1.384	0.10	-	-	-	-	-	-	-	206800	255	7	<5	184	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	11051200	0.17	19.9%	0.03	0.14	1.402	0.10	55.8	1.07	2.02	0.11	0.085	36.9	33.5	214400	374	258	10	270	0.02	0.00	0.00	0.00	0.00	0.01	0.01	0.02	0.00	0.00	0.00	0.00
PLS	11051200	0.16	-	-	0.16	1.367	0.12	-	-	-	-	-	-	-	188200	429	59	16	189	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
Fd	11060000	0.18	28.7%	0.05	0.13	1.412	0.09	56.2	1.02	1.76	0.10	0.072	36.9	32.4	218100	436	382	14	295	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
PLS	11060000	0.16	-	-	0.16	1.382	0.12	-	-	-	-	-	-	-	200500	588	76	25	206	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
FD	SHUTDOWN	0.15	39.6%	0.06	0.09	1.419	0.07	57.5	1.00	1.39	0.10	0.043	37.6	33.8	222400	399	14	431	300	0.04	0.00	0.00	0.00	0.00	0.02	0.02	0.01	0.00	0.00	0.00	0.00
R1	SHUTDOWN	0.12	5.1%	0.01	0.11	1.408	0.08	42.9	2.73	4.64	0.29	0.140	39.8	26.3	216300	<5	<5	263	197	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R2	SHUTDOWN	0.12	4.2%	0.00	0.11	1.396	0.08	39.6	3.89	6.08	0.37	0.210	38.4	27.5	209000	<5	<5	92	186	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R3	SHUTDOWN	0.12	5.3%	0.01	0.12	1.397	0.08	40.4	3.04	4.90	0.29	0.170	39.7	32.3	212700	<5	<5	<5	190	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
R4	SHUTDOWN	0.14	7.4%	0.01	0.13	1.405	0.09	42.5	2.36	3.83	0.23	0.150	41.4	33.8	215200	<5	<5	<5	201	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.02	0.00	0.00	0.00	0.00
PH	SHUTDOWN	0.15	33.4%	0.05	0.10	1.430	0.07	56.3	0.98	1.56	0.10	0.065	37.3	33.2	233400	<5	14	489	311	0.03	0.00	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.00
TOTAL		11.07		1.00																0.54	0.01	0.02	0.00	0.00	0.37	0.33	1.52	0.00	0.00	0.00	0.00

Project No.: 12323-003
Client: Starfield Resources
Campaign: PP2

Scrubbing						
Sample ID		Solution			S	S
		kg	SG	L	mg/L	kg
SL	11011200	0.09	1.172	0.07	14000	0.00
SL	11020000	0.10	1.054	0.10	26000	0.00
SL	11021200	0.09	1.045	0.09	30000	0.00
SL	11030000	0.09	1.087	0.08	26000	0.00
SL	11031200	0.08	1.093	0.07	27000	0.00
SL	11040000	0.11	1.075	0.10	30000	0.00
SL	11041200	0.11	1.074	0.11	36000	0.00
SL	11050000	0.07	1.080	0.07	35000	0.00
SL	11051200	0.11	1.087	0.11	23000	0.00
SL	11060000	0.10	1.082	0.10	27500	0.00
TOTAL		0.96				0.02

LEACH

	kg	Fe	Ni	Cu	Co	Mg	S	S=
TOTAL IN	243.27	36.29	0.62	1.03	0.06	0.06	21.13	20.08
TOTAL OUT	224.46	32.36	0.57	0.97	0.06	0.05	21.13	4.08
OUT/IN	92%	89%	92%	94%	97%	94%	100%	20%
EXTRACTION	**76%**	**87%**	**19%**	**7%**	**8%**	**48%**	**69%**	**80%**
EXTRACTION	**76%**	**83%**	**12%**	**1%**	**5%**	**38%**	**69%**	

Extraction=1-$\frac{\text{Me in Residue}}{\text{Me in Feed}}$

Extraction= $\frac{\text{Me in Liquor}}{\text{Me TOTAL OUT}}$

SCRUBBING

	kg	S
TOTAL IN	486.98	14.66
TOTAL OUT	486.33	12.01
OUT/IN	100%	**82%**

Appendix C – LSR Data Sheets

Appendix C - Liquid-Solid Separation data

Result Analysis Report

Sample Name:
B-3 - Average

SOP Name:
default

Measured:
Tuesday, October 19, 2010 4:52:07 PM

Sample Source & type:
Factory = 12323-003

Measured by:
lr_hydro1

Analysed:
Tuesday, October 19, 2010 4:52:08 PM

Sample bulk lot ref:
Starfields

Result Source:
Averaged

Particle Name:
Default

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Normal

Particle RI:
1.520

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
10.25 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
3.779 %

Result Emulation:
Off

Concentration:
0.0129 %Vol

Span :
2.704

Uniformity:
0.834

Result units:
Volume

Specific Surface Area:
0.715 m²/g

Surface Weighted Mean D[3,2]:
8.387 um

Vol. Weighted Mean D[4,3]:
29.281 um

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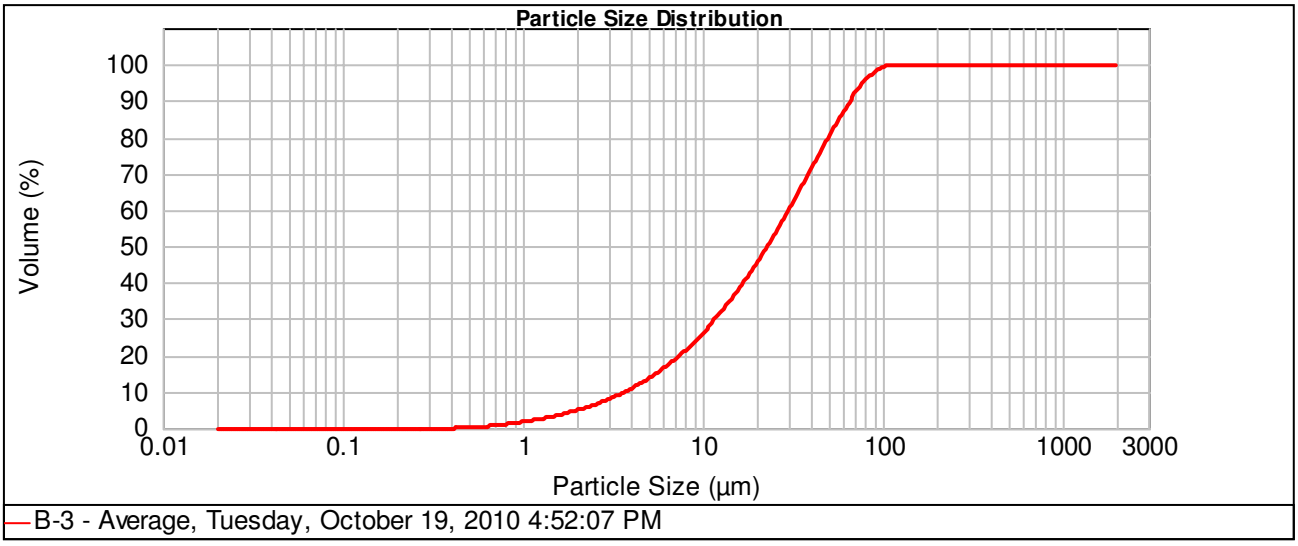
d(0.1): 3.701 um **d(0.5):** 22.836 um **D(0.80) : 49.46 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	2.06	11.482	29.42	120.226	100.00	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	2.58	13.183	32.95	138.038	100.00	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	3.17	15.136	36.78	158.489	100.00	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	3.83	17.378	40.92	181.970	100.00	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	4.59	19.953	45.36	208.930	100.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	5.46	22.909	50.11	239.883	100.00	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	6.45	26.303	55.14	275.423	100.00	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	7.57	30.200	60.40	316.228	100.00	3311.311	100.00
0.030	0.00	0.316	0.00	3.311	8.85	34.674	65.84	363.078	100.00	3801.894	100.00
0.035	0.00	0.363	0.00	3.802	10.29	39.811	71.39	416.869	100.00	4365.158	100.00
0.040	0.00	0.417	0.00	4.365	11.92	45.709	76.91	478.630	100.00	5011.872	100.00
0.046	0.00	0.479	0.09	5.012	13.74	52.481	82.27	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.25	5.754	15.77	60.256	87.27	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	0.50	6.607	18.02	69.183	91.70	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	0.80	7.586	20.49	79.433	95.35	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	1.17	8.710	23.21	91.201	98.00	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	1.59	10.000	26.18	104.713	99.67	1096.478	100.00		

Operator notes:

Reference: Starfields Resources **Date: October 18, 2010**
SGS Project No.: 12323-003 **Operator(s): ZJ**

Details of Flocculant Selection Tests

Objective: Flocculant selection

Slurry Sample ID	B-3 Bulk Rougher Concentrate
Slurry Sample Description	Copper-Nickel-Cobalt Concentrate
Safety Note, MSDS, etc	
Slurry pH	6.6
Pulp Density, kg/m^3	1078
Solids Density, %	10.0
Actual Specific Gravity, kg/m^3	n.d.
Dried Specific Gravity, <->	4.46
Solids K_{80}, microns	49
Solution pH	6.5
Solution Density, kg/m^3	1000
Temperature, ºC	20
Other, i.e. Yield Stress, Pa	n.d.
Other, if applicable, indicate	n.d.
Flocculant stock, g/L	0.25
Volume of Test Sample, mL	100
Weight of Sample, g	108
Wt. of Solids from Sample, g	10.78

Standard Operating Procedure was applied - the results are summarized below:

Floc. Sol'n Add'n, mL		0.0	0.1	0.2	0.5	1.0	2.0	5.0		
Product	Type									
None	n/a	PP3(F)(6.5)								
CIBA Magnafloc 10	AFL		VF2(VF)(6.5)	VF1(VF)(6.5)	F1(VF)(7.5)					
CIBA Magnafloc 155	AFM		F2(VF)(6.5)	F1(VF)(6.5)	F1(VF)(7)					
CIBA Magnafloc 156	AFH			VF1(F)(6.5)						
CIBA Magnafloc 351	NF		VF2(VF)(6.5)	F1(VF)(6.5)						
CIBA Zetag 7623	CFL		VF2(VF)(6.5)	VF1(VF)(6.5)						
Dosage, g/metric t Dry Solids			2	5	12	23	46	116	0	0
Dosage, g/ metric t Slurry			0.2	0.5	1.2	2.3	4.6	11.6	0.0	0.0
Dosage, g/ m^3 pulp			0.3	0.5	1.3	2.5	5.0	12.5	0.0	0.0

Legend to Product Notation - Charge Level
First Digit: A, N, C - anionic, non-ionic and cationic products, respectively.
Second Digit: F, C, K, D - flocculant, coagulant, filter aid, dispersant, rsp'ly.
VL, L, M, H, VH - from very low to very high charge density.
Note: indicate when not polyacrylic product(s) is/are used

Legend to structure notation	
PP - pin point	M - medium
VF - very fine	MH - medium to heavy
F - fine	H, VH - heavy, very heavy, respectively
Liquid Clarity Notation: 1 to 3 indicates decreasing O/F clarity (1=clear)	

Legend to settling rate notation	
(VS) - very slow	(F) - fast
(S) - slow	(VF) - very fast
(M) - moderate	(NS) - No Settling

Legend to compaction notation
Indicates 24 h mudline reading in mL, eg. (12) - 12 mL

Conclusions/ selected products for further testing

Product	Dosage, g/metric t Dry Solids
CIBA Magnafloc 155	5

Note:

Reference: Starfields **Date: Oct 19, 2010**
SGS Project Number: 12323-003 **Operator: ZJ**

Summary of Static Settling Tests

Stage 1: Solid Densities Optimisation Tests:

Settling Test Number	1	2	3	4
Sample Identification	**B-3 (Bulk Rougher Concentrate)**			
Raked, Yes/No	yes			
Particle Size, d_{80}, microns	49			
Pulp pH	6.6			
Pulp Temperature, ℃	21			
Solution Density, g/L	1000			
Undiluted Pulp Solid Content, % wt.	31.5 ("as received")			
Diluted Feed Solid Content, % wt.	6	11	17	**23**
CIBA Magnafloc 155, g/t dry	5	4	4	4
U/F Solids Density, % wt.	71	81	79	81
Thickener Underflow Unit Area, m^2/t/day	0.015	0.011	0.012	0.011
Thickener Hydraulic Unit Area, m^2/t/day	0.010	0.006	0.005	0.003
Initial Settling Rate, m^3/m^2/day	1629	1299	995	1016
Supernatant Clarity, 10 minutes	Cloudy	S.C.	S.C.	S.C.
Supernatant Clarity, 60 minutes	Cloudy	S.C.	S.C.	S.C.
Supernatant Clarity, Final	S.C.	S.C.	S.C.	S.C.

All values calculated without safety factor.

Legend: S.C. = Slightly Cloudy

23 Indicates feed solids density selected for flocc optimisation tests (Stage 2).

Stage 2: Flocculant Dosage Optimisation Tests:

Settling Test Number	**5**	6	7	
Sample Identification	**B-3 (Bulk Rougher Concentrate)**			
Raked, Yes/No	yes			
Particle Size, d_{80}, microns	49			
Pulp pH	6.6			
Pulp Temperature, ℃	21			
Solution Density, g/L	1000			
Undiluted Pulp Solid Content, % wt.	31.5 ("as received")			
Diluted Feed Solid Content, % wt.	**22**	22	22	
CIBA Magnafloc 155, g/t dry	**9**	14	19	
U/F Solids Density, % wt.	**79**	79	79	
Thickener Underflow Unit Area, m^2/t/day	**0.015**	0.015	0.015	
Thickener Hydraulic Unit Area, m^2/t/day	**0.003**	0.003	0.003	
Initial Settling Rate, m^3/m^2/day	**1027**	1276	1286	
Supernatant Clarity, 10 minutes	**S.C.**	Clear	Clear	
Supernatant Clarity, 60 minutes	**Clear**	Clear	Clear	
Supernatant Clarity, Final	**Clear**	Clear	Clear	

All values calculated without safety factor.

Legend: S.C. = Slightly Cloudy

5 Indicates test that produced the best overall results (as tested).

Reference: Starfields **Date: October 29, 2010**
SGS Project No.: 12323-003 **Operator: M.G.**

Vacuum Filtration Summary

Sample: B-3 Concentrate
Pulp pH: 6.6
PSD, d80, microns: 49
Filter Cloth: Polypropylene, Supplier # 05-800-SK012, Permeability: 7 cfm, 12 micron, Warp/Weft: Mono/multi, Weave: Double Layer Twill, Weight: 13.4 oz/yd
Vacuum: average, Form = 21"Hg, Dry = 21"Hg
Filter Aid: CIBA Magnafloc 155 at 10 g/t

	Feed Solids % Wt.	Cake Thickness mm	Cake Moisture % wt.	Vacuum Filter Cycle Times Form minutes	Vacuum Filter Cycle Times Dry minutes	Output Dry kg/m^2/hr	Filtrate Clarity
Belt Filter	80.4	10	6.0	0.08	5.80	283	Clear
		10	8.0	0.08	0.68	2206	
		10	10.0	0.08	0.13	8141	
		20	6.0	0.30	11.60	280	
		20	8.0	0.30	1.36	2004	
	▼	20	10.0	0.30	0.26	5934	▼

	Feed Solids % Wt.	Cake Thickness mm	Cake Moisture % wt.	Vacuum Filter Cycle Times Form minutes	Vacuum Filter Cycle Times Dry minutes	Output Dry kg/m^2/hr	Filtrate Clarity
[1]Disc or Drum Filter	80.4	10	10.0	0.08	0.13	6564	Clear
		12	9.8	0.11	0.18	5470	
		14	9.6	0.15	0.25	4689	
		16	9.4	0.19	0.32	4103	
		18	9.3	0.25	0.41	3647	
	▼	20	9.1	0.30	0.51	3282	▼

Notes:
No scale-up or safety factors have been applied.
[1] Disc or Drum cycle times and resulting cake moisture/cake output are based on operation under balanced filter conditions (i.e. no form or dry time capacity limitations).
The operating conditions and filter outputs that are reported in this table are examples based on the data produced during testing.
Additional design outputs, that are more suited to an individual process, can be calculated using the vacuum filtration relationships/graphs that are included in this reporting package.

SGS - Mineral Services

VACUUM FILTRATION SIZING - Belt Filter

Company: Starfields Resources
Sample: B-3 (Bulk Rougher Concentrate)
Project No. 12323-003 Test: 1-6

Filter cloth: **Polypropylene, Supplier # 05-800-SK012, Permeability: 7 cfm, 12 micron, Warp/Weft: Mono/multi, Weave: Double Layer Twill, Weight: 13.4 oz/yd**

			Filter Capacity for Various Conditions					
Filter Feed Conditions:								
Feed Density:	%Solids	80.4	80.4	80.4	80.4	80.4	80.4
pH:		6.6	6.6	6.6	6.6	6.6	6.6
Feed Temp:	^{0}C	21.0	21.0	21.0	21.0	21.0	21.0
Filter Aid:		CIBA Magnafloc 155, 10 g/t					
Filter Operating Conditions:							
Vacuum:	"Hg	21	21	21	21	21	21
Cake Specification:							
Cake Thickness:	mm	**10**	**10**	**10**	**20**	**20**	**20**
Cake Moisture:	%water	6.00	8.00	10.00	6.00	8.00	10.00
Filter Test Results:							
Form Time:	min	0.08	0.08	0.08	0.30	0.30	0.30
Dry Time (Calc):	min	5.80	0.68	0.13	11.60	1.36	0.26
Dry Cake Weight:	kg/m^2	27.7	27.7	27.7	55.5	55.5	55.5
Dry Bulk Density:	kg/m^3	2774	2774	2774	2774	2774	2774
Wet Bulk Density:	kg/m^3	2951	3016	3083	2951	3016	3083
Filter Conditions:							
Form Time	% of Total	1.3	10.1	37.2	2.6	18.3	54.2
Dry Time	% of Total	98.7	89.9	62.8	97.4	81.7	45.8
Discharge Time	% of Total						
Vacuum Filter Capacity							
Filter Output Rate:	**dry kg/m^2/hr**	**283**	**2206**	**8141**	**280**	**2004**	**5934**
Capacity Limiting		**-**	**-**	**-**	**-**	**-**	**-**

F=Form Time D=Dry Time -=Balanced

Note: *No scale-up or safety factors have been applied.*

SGS - Mineral Services

VACUUM FILTRATION SIZING - Disc or Drum Filter

Company: Starfields Resources
Sample: B-3 (Bulk Rougher Concentrate)
Project No. 12323-003 Test: 1-6

Filter cloth: **Polypropylene, Supplier # 05-800-SK012, Permeability: 7 cfm, 12 micron, Warp/Weft: Mono/multi, Weave: Double Layer Twill, Weight: 13.4 oz/yd**

			Filter Capacity for Various Conditions					
Filter Feed Conditions:								
Feed Density:	%Solids	80.4	80.4	80.4	80.4	80.4	80.4
pH:		6.6	6.6	6.6	6.6	6.6	6.6
Feed Temp:	^{0}C	21.0	21.0	21.0	21.0	21.0	21.0
Filter Aid:		CIBA Magnafloc 155, 10 g/t					
Filter Operating Conditions:							
Vacuum:	"Hg	21	21	21	21	21	21
Cake Specification:							
Cake Thickness:	mm	**10**	**12**	**14**	**16**	**18**	**20**
Cake Moisture:	%water	**10.0**	**9.8**	**9.6**	**9.4**	**9.3**	**9.1**
Filter Test Results:							
Form Time:	min	0.08	0.11	0.15	0.19	0.25	0.30
Dry Time (Calc):	min	0.13	0.18	0.25	0.32	0.41	0.51
Dry Cake Weight:	kg/m^2	27.7	33.3	38.8	44.4	49.9	55.5
Dry Bulk Density:	kg/m^3	2774	2774	2774	2774	2774	2774
Wet Bulk Density:	kg/m^3	3083	3075	3068	3062	3057	3053
Filter Conditions:							
Form Time	% of Total	30	30	30	30	30	30
Dry Time	% of Total	50	50	50	50	50	50
Discharge Time	% of Total	20	20	20	20	20	20
Vacuum Filter Capacity							
Filter Output Rate:	**dry kg/m^2/hr**	**6564**	**5470**	**4689**	**4103**	**3647**	**3282**
Capacity Limiting		**-**	**-**	**-**	**-**	**-**	**-**

F=Form Time D=Dry Time -=Balanced

Note: *No scale-up or safety factors have been applied.*

SGS - Miner

VACUUM FILTRATION TEST DATA

Company:	**Starfields Resources**	Table No:	
Sample:	B-3 (Bulk Rougher Concentrate)	Test No:	1-6
Project No.	12323-003	Date:	22-Oct-10
Filter cloth:	Polypropylene, Supplier # 05-800-SK012, Permeability: 7 cfm, 12 micron, Warp/Weft: Mono/multi, Weave: Double Layer Twill, Weight: 13.4 oz/yd	By:	ZJ
Filter Aid:	CIBA Magnafloc 155, 10 g/t		

pH:	6.6		Feed Density:	80.4	% Solids (wt)	
Feed Temp:	21	°C	Filter diameter:	113	mm	
S.G. Solids:	4.46	g/cm^3	Filter Area:	0.108	ft^2	0.010 m^2
S.G Liquid:	1.00	g/cm^3				

Input Measurements

Test No.	Approx. Sample	Vacuum ("Hg)		Time (Min)		Cake		Volume (mL)	Cake Thickness	Gross Weights (g)			Filtrate Solids	
	Vol.,mL	Form	Dry	Form	Dry	Cracks	Wall Sep.	Filtrate	(mm)	Wet	Dry	Tare	Visual	mg/L
1	350	21	21	0.23	0.03	No	No	56	25	839.4	735.8	47.8	Clear	
2	250	20	20	0.17	0.05	No	No	41	17	581.0	511.8	35.8	Clear	#VALUE!
3	150	20	20	0.10	0.03	No	No	27	10	361.7	319.1	35.5	Clear	#VALUE!
4	150	20	20	0.08	0.17	No	No	31	8	298.4	273.3	35.9	Clear	#VALUE!
5	150	21	21	0.10	0.50	No	No	50	11	354.0	329.3	35.8	Clear	#VALUE!
6	120	21	21	0.08	1.10	No	No	44	8	291.5	273.19	37.29	Clear	#VALUE!

Calculated Values

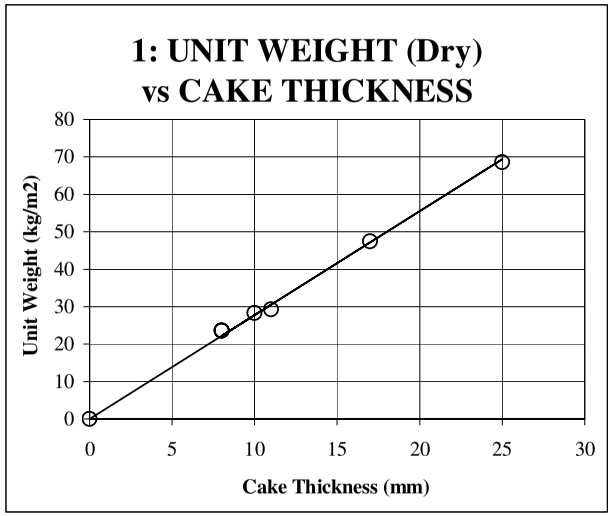
Test No.	Net Weights (g)		Cake Moisture	Feed Solids	Unit Weight "W"	Dry Time Factor
	Wet	Dry	%	%	kg/m^2	$min.m^2/kg$
1	791.6	688.0	13.1	81.2	68.6	0.00
2	545.1	475.9	12.7	81.2	47.5	0.00
3	326.1	283.5	13.1	80.3	28.3	0.00
4	262.5	237.4	9.6	80.9	23.7	0.01
5	318.2	293.5	7.8	79.7	29.3	0.02
6	254.2	235.9	7.2	79.1	23.5	0.05

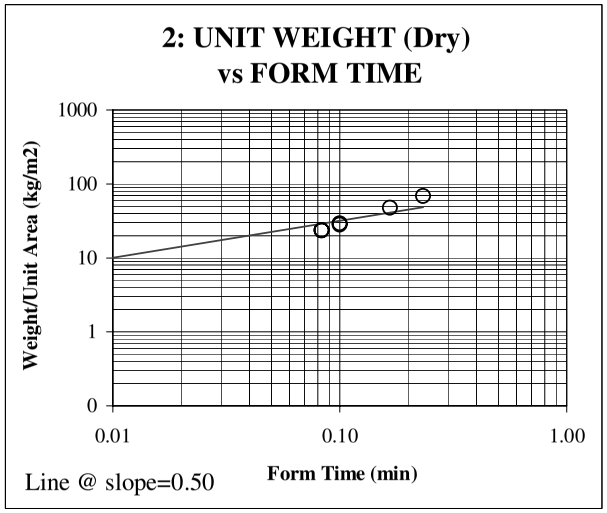
Filtrate Solids	
mg	%
#VALUE!	#VALUE!
#VALUE!	#VALUE!
#VALUE!	#VALUE!
#VALUE!	#VALUE!
#VALUE!	#VALUE!

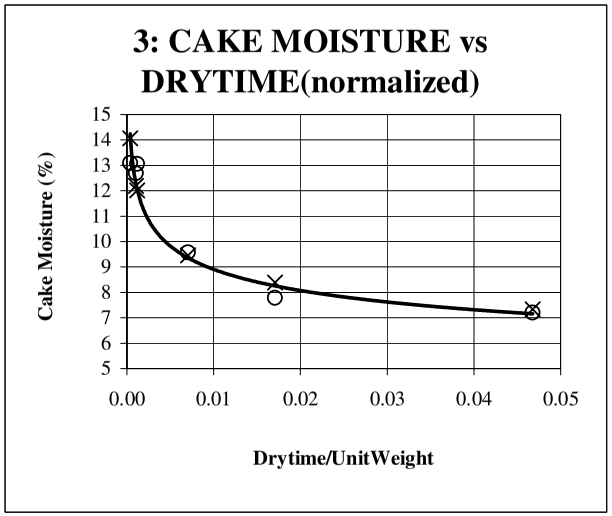
Notes:

VACUUM FILTRATION TEST DATA

Company:	**Starfields Resources**	Figure:	0
Project:	B-3 (Bulk Rougher Concentrate)	Test No:	1-6
Filter cloth:	Polypropylene, Supplier # 05-800-SK012, Permeability: 7 cfm, 12 micron, Warp/Weft: Mono/multi, Weave: Double Layer Twill, Weight: 13.4 oz/yd	Date:	22-Oct-10







Relationships:

Unit Weight =	2.774		* Cake Thickness
Form Time =	0.233	*	(Unit Weight / 48.6)^2
Moisture =	4.864	*	(Drytime/Unit Weight) ^ -0.134
DryTime/UnitWeight =			(Moisture / 4.864) ^ (1/-0.134)

Client / Project: Starfields | **Technologist(s): MG**
SGS Reference No.: 12323-003 | **Date: October 27, 2010**

Pressure Filtration
Calculation Sheet and Test Results Summary

Feed/ sample ID: B-3 Concentrate

Equipment: SHRIVER UNIT 3 by EIMCO
ASG or SG, g/cm3: 4.46
Filter aid: CIBA Magnafloc 155, 10 g/t
Filter Cloth: Pannevis: PP 2676, Polypropylene, Permeability: 12 cfm, Porosity: 20 micron, Warp/Weft: Mono/multi, Weave: Double Layer Twill, Weight: 14.2 oz/yd, Max. Width: 98 inches
Pulp pH: 6.6
Pulp Temperature, ºC: 21
Particle Size, d80, microns: 49

Test ID	**Test 1**
	m
Filter diameter, D	0.0725
Actual cake thickness, H	0.037
	m^2
Filter area, A = (PI/4)*D^2	0.0083
	L
Actual cake volume, V=A*H	0.153
	kg
Cake wet weight, CW	0.425
Cake dry weight, CD	0.366
	% wt.
Wet cake solids, S%	86.2
	kg/L
Wet cake density, WD = CW/V	2.780
Dry cake density, DD=WD*CS/100	2.398
	L
Filtrate volume, V	0.089
	L/m^2
Filter loading, L= (Volume/Area)	29.3
	h
Feeding time, T	0.10

Filter Throughput versus Conditions

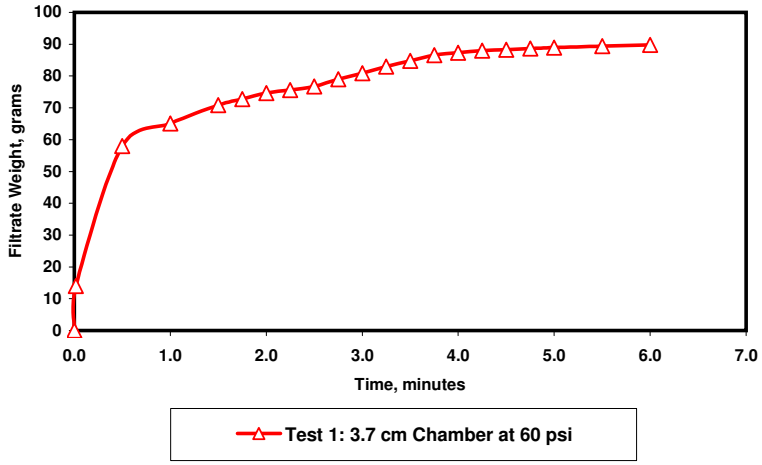
	psi
Feed Pressure	60
	cm
Cake thickness	3.7
	% wt.
Feed solids,	75
Final wet cake moisture	14
	kg/m^2h
Dry solids capacity, KS=CD/(A*T)	**444**
	L/m^2h
Filter capacity, KL=L/T	293

Client / Project: Starfields
SGS Reference No.: 12323-003

Technologist(s): MG
Date: October 27, 2010

Feed/ sample ID: B-3 Concentrate

Filtrate Weight versus Filtering Time



Filtration Notes:

General Notes:
-The pulp was targeted to be 80% solids, but was diluted to 75% to facilitate mixing and pumping.

Test 1 Notes:
-Settling solids in the diaphragms of the pump resulted in pressure loss and eventual plugging during the 6 minute test duration.
-Multiple attempts were made to avoid plugging but were not successful. Tests aborted.
-Some fines initially in the filtrate, then clear filtrate. The cake partially adhered to the cloth. Wet, cohesive cake.

Project No. : 12323-003
Reference: Starfields
Testing Instrument: Haake RS75
Sensor: FL 22 (Vane)

Date: October 29, 2010
Operator(s): M.G.

Rheology - Results Summary

Sample: Thickened B-3 Concentrate

	Test Code	Test Method	Size, d_{80} microns	[1] ASG g/cm^3	Sample [2] SG g/cm^3	[3]a ASG/SG	Temp ºC	Solids %	[4] Dens. g/L	Shear Stress [5]Peak, Pa	Notes \ Observations
	T2	Vane	49	4.42	4.46	0.99	21.0	86.9	3053	448	See General Observations

General Observations:

This sample was not amenable to CCRV (Concentric Cylinder Rotational Viscometry) due to inability of the sensor to reach the measurement position in the test cup at high pulp solid content.
Therefore, a vane sensor was used for determination. The sample was tested at a solid content (86.9%) that remained as a stable pulp (i.e. no longer settled over the duration of the test).

Notes:

The B-3 Concentrate sample was tested as a flocculated sample with Ciba Magnafloc 155 at a dosage of 10 g/t (vs. dry solids).
The values are based on data produced by the unsheared slurry sample.
Shear Stress value was produced by subjecting the slurry sample to a constant rotation at 0.10 s^{-1} for 5 minutes.

[1] ASG - Actual specific gravity of dry solids, calculated from the measured pulp density ([4] Dens.)

[2] SG - Dried solids specific gravity, measured with the pycnometer

[3] α - Correction Factor, i.e. solids density ratio

[4] Pulp density in g/L

[5] Maximum (peak) initial shear stress achieved.

Result Analysis Report

Sample Name:
Leach Residue - Average

SOP Name:
default

Measured:
Monday, November 15, 2010 2:08:10 PM

Sample Source & type:
Factory = 12323-003

Measured by:
lr_hydro1

Analysed:
Monday, November 15, 2010 2:08:12 PM

Sample bulk lot ref:
Starfields

Result Source:
Averaged

Particle Name:
Default

Accessory Name:
Hydro 2000G (A)

Analysis model:
General purpose

Sensitivity:
Normal

Particle RI:
1.520

Absorption:
0.1

Size range:
0.020 to 2000.000 um

Obscuration:
13.89 %

Dispersant Name:
Water

Dispersant RI:
1.330

Weighted Residual:
0.544 %

Result Emulation:
Off

Concentration:
0.0177 %Vol

Span :
4.427

Uniformity:
1.45

Result units:
Volume

Specific Surface Area:
0.776 m²/g

Surface Weighted Mean D[3,2]:
7.734 um

Vol. Weighted Mean D[4,3]:
46.575 um

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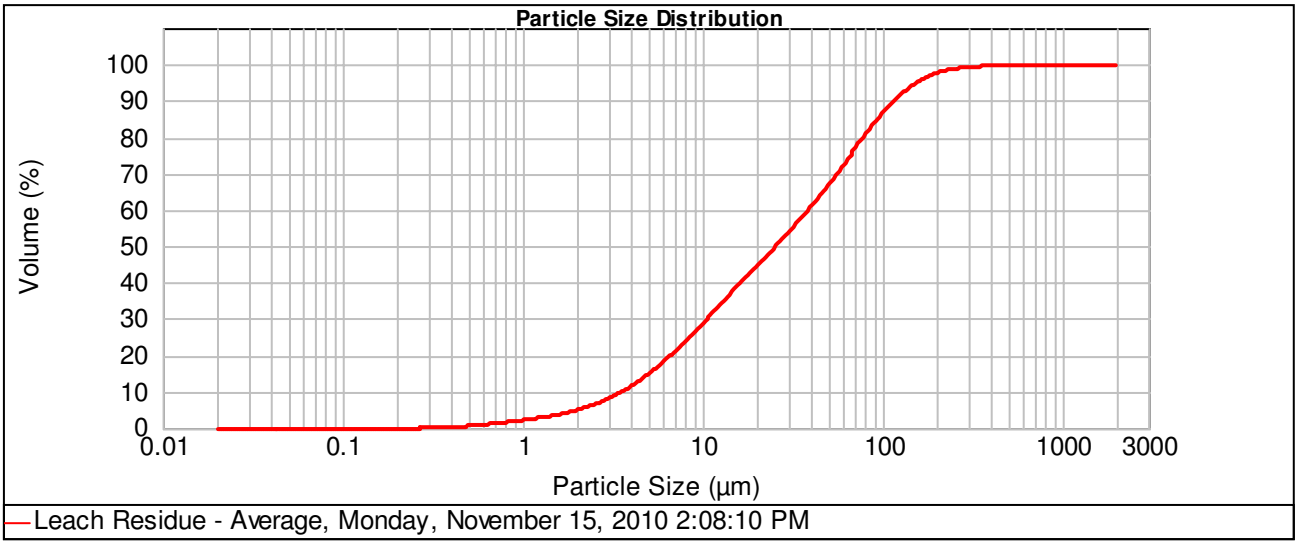
d(0.1): 3.548 um **d(0.5):** 25.363 um **D(0.80) : 78.38 µm**



Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %	Size (µm)	Vol Under %
0.010	0.00	0.105	0.00	1.096	2.50	11.482	32.01	120.226	90.79	1258.925	100.00
0.011	0.00	0.120	0.00	1.259	2.93	13.183	35.11	138.038	93.39	1445.440	100.00
0.013	0.00	0.138	0.00	1.445	3.42	15.136	38.23	158.489	95.44	1659.587	100.00
0.015	0.00	0.158	0.00	1.660	4.00	17.378	41.34	181.970	96.96	1905.461	100.00
0.017	0.00	0.182	0.00	1.905	4.69	19.953	44.48	208.930	98.00	2187.762	100.00
0.020	0.00	0.209	0.00	2.188	5.52	22.909	47.64	239.883	98.69	2511.886	100.00
0.023	0.00	0.240	0.00	2.512	6.53	26.303	50.85	275.423	99.12	2884.032	100.00
0.026	0.00	0.275	0.00	2.884	7.75	30.200	54.15	316.228	99.40	3311.311	100.00
0.030	0.00	0.316	0.05	3.311	9.19	34.674	57.55	363.078	99.62	3801.894	100.00
0.035	0.00	0.363	0.15	3.802	10.88	39.811	61.07	416.869	99.81	4365.158	100.00
0.040	0.00	0.417	0.30	4.365	12.82	45.709	64.74	478.630	99.95	5011.872	100.00
0.046	0.00	0.479	0.51	5.012	15.02	52.481	68.54	549.541	100.00	5754.399	100.00
0.052	0.00	0.550	0.77	5.754	17.46	60.256	72.46	630.957	100.00	6606.934	100.00
0.060	0.00	0.631	1.07	6.607	20.11	69.183	76.43	724.436	100.00	7585.776	100.00
0.069	0.00	0.724	1.39	7.586	22.93	79.433	80.38	831.764	100.00	8709.636	100.00
0.079	0.00	0.832	1.74	8.710	25.88	91.201	84.17	954.993	100.00	10000.000	100.00
0.091	0.00	0.955	2.11	10.000	28.92	104.713	87.69	1096.478	100.00		

Operator notes:

Reference: Starfields Resources **Date: November 5, 2010**
SGS Project No.: 12323-003 **Operator(s): ZJ**

Details of Flocculant Selection Tests

Objective: Flocculant selection

Slurry Sample ID	Leach Residue
Slurry Sample Description	
Safety Note, MSDS, etc	Highly Acidic
Slurry pH	<1
Pulp Density, kg/m^3	1425
Solids Density, %	3.7
Actual Specific Gravity, kg/m^3	n.d.
Dried Specific Gravity, <->	4.23
Solids K_{80}, microns	78
Solution pH	<1
Solution Density, kg/m^3	1390
Temperature, ºC	50
Other, i.e. Yield Stress, Pa	n.d.
Other, if applicable, indicate	n.d.
Flocculant stock, g/L	0.25
Volume of Test Sample, mL	100
Weight of Sample, g	143
Wt. of Solids from Sample, g	5.27

Standard Operating Procedure was applied - the results are summarized below:

Floc. Sol'n Add'n, mL		0.0	1.0	2.0	4.0					
Product	Type									
None	n/a	M3(S)(15)								
CIBA Magnafloc 10	AFL		MH1.5(F)(15)	H1.5(VF)(16)	H1.5(VF)(16)					
CIBA Magnafloc 155	AFM		**MH1.5(F)(15)**		H1.5(VF)(16)					
CIBA Magnafloc 156	AFH		MH1.5(S)(15)							
CIBA Magnafloc 919	AFH		MH1.5(S)(15)							
CIBA Magnafloc 351	NF		MH1.5(F)(15)		H1.5(VF)(16)					
CIBA Zetag 7623	CFL		MH1.5(F)(15)							
CIBA Magnafloc 368	CCVH		M3(S)(15)							
CIBA Magnafloc 155+368			MH1.5(F)(16)							
Dosage, g/metric t Dry Solids			**47**	95	190	0	0	0	0	0
Dosage, g/ metric t Slurry			1.8	3.5	7.0	0.0	0.0	0.0	0.0	0.0
Dosage, g/ m^3 pulp			2.5	5.0	10.0	0.0	0.0	0.0	0.0	0.0

Legend to Product Notation - Charge Level
First Digit: A, N, C - anionic, non-ionic and cationic products, respectively.
Second Digit: F, C, K, D - flocculant, coagulant, filter aid, dispersant, rsp'ly.
VL, L, M, H, VH - from very low to very high charge density.
Note: indicate when not polyacrylic product(s) is/are used

Legend to structure notation	
PP - pin point	M - medium
VF - very fine	MH - medium to heavy
F - fine	H, VH - heavy, very heavy, respectively
Liquid Clarity Notation: 1 to 3 indicates decreasing O/F clarity (1=clear)	

Legend to settling rate notation	
(VS) - very slow	(F) - fast
(S) - slow	(VF) - very fast
(M) - moderate	(NS) - No Settling

Legend to compaction notation
Indicates 24 h mudline reading in mL, eg. (12) - 12 mL

Conclusions/ selected products for further testing

Product	Dosage, g/metric t Dry Solids
CIBA Magnafloc 155	47

Note:
Supernatant clear at 47 g/t after ~60 min (except for Mag 368)

Reference: Starfields **Date: Nov 8, 2010**
SGS Project Number: 12323-003 **Operator: ZJ**

Summary of Static Settling Tests

Stage 1: Solid Densities Optimisation Tests:

Settling Test Number	8	**9**	10
Sample Identification		**Leach Residue**	
Raked, Yes/No		yes	
Particle Size, d_{80}, microns		78	
Pulp pH		<1	
Pulp Temperature, ℃		50	
Solution Density, g/L		1390	
Undiluted Pulp Solid Content, % wt.		3 ("as received")	
Diluted Feed Solid Content, % wt.	1.4	**2.9**	4.8
CIBA Magnafloc 155, g/t dry	48	**48**	43
U/F Solids Density, % wt.	23	**27**	39
Thickener Underflow Unit Area, m^2/t/day	0.08	**0.07**	0.11
Thickener Hydraulic Unit Area, m^2/t/day	0.06	**0.03**	0.03
Initial Settling Rate, m^3/m^2/day	1140	**960**	552
Supernatant Clarity, 10 minutes	Cloudy	**Cloudy**	S.C.
Supernatant Clarity, 60 minutes	Cloudy	**S.C.**	S.C.
Supernatant Clarity, Final	Clear	**Clear**	Clear

All values calculated without safety factor.

Legend: S.C. = Slightly Cloudy

2.9 Indicates feed solids density selected for flocc optimisation tests (Stage 2).

Stage 2: Flocculant Dosage Optimisation Tests:

Settling Test Number	11	12	13
Sample Identification		**Leach Residue**	
Raked, Yes/No		yes	
Particle Size, d_{80}, microns		78	
Pulp pH		<1	
Pulp Temperature, ℃		50	
Solution Density, g/L		1390	
Undiluted Pulp Solid Content, % wt.		3 ("as received")	
Diluted Feed Solid Content, % wt.	2.9	3.0	3.0
CIBA Magnafloc 155, g/t dry	95	189	329
U/F Solids Density, % wt.	26	26	30
Thickener Underflow Unit Area, m^2/t/day	0.07	0.06	0.06
Thickener Hydraulic Unit Area, m^2/t/day	0.03	0.02	0.02
Initial Settling Rate, m^3/m^2/day	1099	1343	1427
Supernatant Clarity, 10 minutes	Cloudy	Cloudy	S.C.
Supernatant Clarity, 60 minutes	S.C.	S.C.	Clear
Supernatant Clarity, Final	Clear	Clear	Clear

All values calculated without safety factor.

Legend: S.C. = Slightly Cloudy

9 Indicates test that produced the best overall results (as tested).

Reference: Starfields **Date: November 12, 2010**
SGS Project No.: 12323-003 **Operator: M.G.**

Vacuum Filtration Summary

Sample: Leach Residue
Pulp pH: <1
PSD, d80, microns: 78
Filter Cloth: Polypropylene, Supplier # 05-8000-K050, Permeability: 52 cfm, Porosity: 50 micron, Warp/Weft: Mono/Mono, Weave: Double Layer Twill, Weight: 13.0 oz/yd
Vacuum: average, Form = 22"Hg, Dry = 17"Hg
Filter Aid: CIBA Magnafloc 155 at 48 g/t

	Feed Solids % Wt.	[2] Cake Thickness mm	Cake Moisture % wt.	Vacuum Filter Cycle Times Form minutes	Vacuum Filter Cycle Times Dry minutes	Output Dry kg/m^2/hr	Filtrate Clarity
Belt Filter	30.9	5	44.0	0.56	2.26	127	Clear
		5	45.0	0.56	0.40	373	
		5	46.0	0.56	0.07	567	
		15	44.0	5.00	6.79	91	
		15	45.0	5.00	1.20	173	
	▼	15	46.0	5.00	0.22	205	▼

	Feed Solids % Wt.	Cake Thickness mm	Cake Moisture % wt.	Vacuum Filter Cycle Times Form minutes	Vacuum Filter Cycle Times Dry minutes	Output Dry kg/m^2/hr	Filtrate Clarity
[1]Disc or Drum Filter	30.9	5	44.5	0.56	0.93	193	Clear
		7	44.3	1.09	1.81	138	
		9	44.2	1.80	3.00	107	
		11	44.1	2.69	4.48	88	
		13	44.0	3.75	6.25	74	
	▼	15	43.9	5.00	8.33	64	▼

Notes:
No scale-up or safety factors have been applied.
[1] Disc or Drum cycle times and resulting cake moisture/cake output are based on operation under balanced filter conditions (i.e. no form or dry time capacity limitations).
The operating conditions and filter outputs that are reported in this table are examples based on the data produced during testing.
Additional design outputs, that are more suited to an individual process, can be calculated using the vacuum filtration relationships/graphs that are included in this reporting package.
[2] Significant cake cracking occurred at these cake thickness during dry cycle.

SGS - Mineral Services

VACUUM FILTRATION SIZING - Belt Filter

Company: Starfields Resources
Sample: Leach Residue
Project No. 12323-003 Test: 1-4

Filter cloth: **Polypropylene, Supplier # 05-8000-K050, Permeability: 52 cfm, Porosity: 50 micron, Warp/Weft: Mono/Mono, Weave: Double Layer Twill, Weight: 13.0 oz/yd**

		Filter Capacity for Various Conditions					
Filter Feed Conditions:							
Feed Density:	%Solids	30.9	30.9	30.9	30.9	30.9	30.9
pH:		n.d.	n.d.	n.d.	n.d.	n.d.	n.d.
Feed Temp:	^{0}C	50.0	50.0	50.0	50.0	50.0	50.0
Filter Aid:		CIBA Magnafloc 155, 48 g/t					
Filter Operating Conditions:							
Vacuum:	"Hg	22	22	22	22	22	22
Cake Specification:							
Cake Thickness:	mm	**5**	**5**	**5**	**15**	**15**	**15**
Cake Moisture:	%water	44.00	45.00	46.00	44.00	45.00	46.00
Filter Test Results:							
Form Time:	min	0.56	0.56	0.56	5.00	5.00	5.00
Dry Time (Calc):	min	2.26	0.40	0.07	6.79	1.20	0.22
Dry Cake Weight:	kg/m^2	5.9	5.9	5.9	17.8	17.8	17.8
Dry Bulk Density:	kg/m^3	1189	1189	1189	1189	1189	1189
Wet Bulk Density:	kg/m^3	2124	2162	2202	2124	2162	2202
Filter Conditions:							
Form Time	% of Total	19.7	58.1	88.3	42.4	80.6	95.8
Dry Time	% of Total	80.3	41.9	11.7	57.6	19.4	4.2
Discharge Time	% of Total						
Vacuum Filter Capacity							
Filter Output Rate:	**dry kg/m^2/hr**	**127**	**373**	**567**	**91**	**173**	**205**
Capacity Limiting		**-**	**-**	**-**	**-**	**-**	**-**

F=Form Time D=Dry Time -=Balanced

Note: *No scale-up or safety factors have been applied.*

SGS - Mineral Services

VACUUM FILTRATION SIZING - Disc or Drum Filter

Company: Starfields Resources
Sample: Leach Residue
Project No. 12323-003 Test: 1-4

Filter cloth: **Polypropylene, Supplier # 05-8000-K050, Permeability: 52 cfm, Porosity: 50 micron, Warp/Weft: Mono/Mono, Weave: Double Layer Twill, Weight: 13.0 oz/yd**

			Filter Capacity for Various Conditions					
Filter Feed Conditions:								
Feed Density:	%Solids	30.9	30.9	30.9	30.9	30.9	30.9
pH:		n.d.	n.d.	n.d.	n.d.	n.d.	n.d.
Feed Temp:	^{0}C	50.0	50.0	50.0	50.0	50.0	50.0
Filter Aid:		CIBA Magnafloc 155, 48 g/t					
Filter Operating Conditions:							
Vacuum:	"Hg	22	22	22	22	22	22
Cake Specification:							
Cake Thickness:	mm	**5**	**7**	**9**	**11**	**13**	**15**
Cake Moisture:	%water	**44.5**	**44.3**	**44.2**	**44.1**	**44.0**	**43.9**
Filter Test Results:							
Form Time:	min	0.56	1.09	1.80	2.69	3.75	5.00
Dry Time (Calc):	min	0.93	1.81	3.00	4.48	6.25	8.33
Dry Cake Weight:	kg/m^2	5.9	8.3	10.7	13.1	15.5	17.8
Dry Bulk Density:	kg/m^3	1189	1189	1189	1189	1189	1189
Wet Bulk Density:	kg/m^3	2143	2136	2130	2126	2122	2119
Filter Conditions:							
Form Time	% of Total	30	30	30	30	30	30
Dry Time	% of Total	50	50	50	50	50	50
Discharge Time	% of Total	20	20	20	20	20	20
Vacuum Filter Capacity							
Filter Output Rate:	**dry kg/m^2/hr**	**193**	**138**	**107**	**88**	**74**	**64**
Capacity Limiting		**-**	**-**	**-**	**-**	**-**	**-**
F=Form Time D=Dry Time -=Balanced							

Note: *No scale-up or safety factors have been applied.*

SGS - Miner

VACUUM FILTRATION TEST DATA

Company:	**Starfields Resources**					Table No:	
Sample:	Leach Residue					Test No:	1-4
Project No.	12323-003					Date:	12-Nov-10
Filter cloth:	Polypropylene, Supplier # 05-8000-K050, Permeability: 52 cfm, Porosity: 50 micron, Warp/Weft: Mono/Mono, Weave: Double Layer Twill, Weight: 13.0 oz/yd					By:	MG/ZJ
Filter Aid:	CIBA Magnafloc 155, 48 g/t						
pH:	n.d.		Feed Density:	40.0	% Solids (wt)		
Feed Temp:	50	°C	Filter diameter:	113	mm		
S.G. Solids:	4.23	g/cm^3	Filter Area:	0.108	ft^2	0.010	m^2
S.G Liquid:	1.39	g/cm^3					

Input Measurements

Test No.	Approx. Sample	Vacuum ("Hg)		Time (Min)		Cake		Volume (mL)	Cake Thickness	Gross Weights (g)			Filtrate Solids	
	Vol.,mL	Form	Dry	Form	Dry	Cracks	Wall Sep.	Filtrate	(mm)	Wet	Dry	Tare	Visual	mg/L
1	500	23	23	6.75	0.67	No	No	208	18	382.0	207.9		Clear	
2	300	22	14	4.10	0.82	Minor	Minor	152	13	279.0	153.9		Clear	#VALUE!
3	250	22	14	3.10	3.10	Major	Minor	132	10	229.4	127.1		Clear	#VALUE!
4	150	22	18	1.13	4.25	Major	No	77	5	126.5	71.4		Clear	#VALUE!

Calculated Values

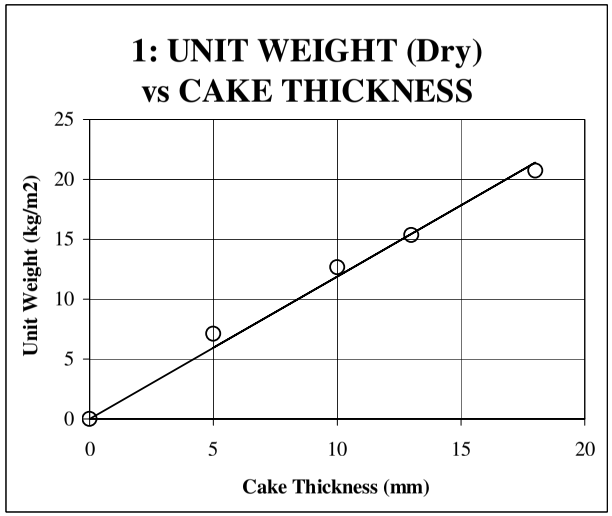
Test No.	Net Weights (g)		Cake Moisture	Feed Solids	Unit Weight "W"	Dry Time Factor
	Wet	Dry	%	%	kg/m^2	min.m^2/kg
1	382.0	207.9	45.6	31.0	20.7	0.03
2	279.0	153.9	44.8	31.4	15.3	0.05
3	229.4	127.1	44.6	30.8	12.7	0.24
4	126.5	71.4	43.6	30.6	7.1	0.60

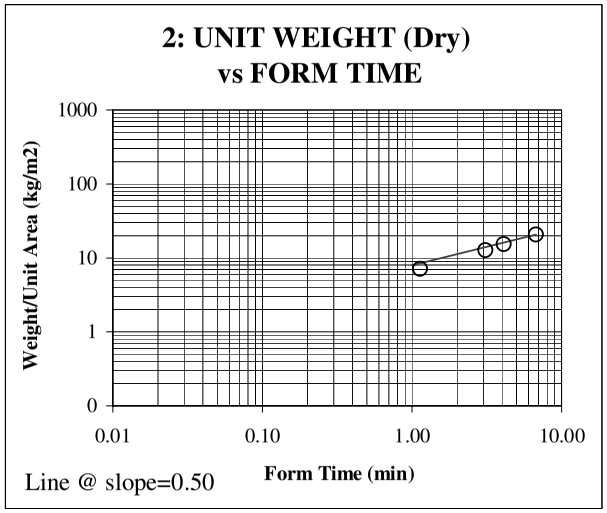
Filtrate Solids	
mg	%
#VALUE!	#VALUE!
#VALUE!	#VALUE!
#VALUE!	#VALUE!

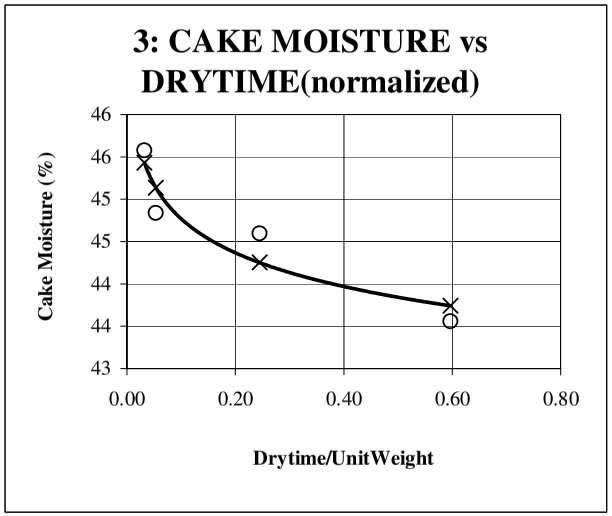
Notes:

VACUUM FILTRATION TEST DATA

Company:	**Starfields Resources**	Figure:	0
Project:	Leach Residue	Test No:	1-4
Filter cloth:	Polypropylene, Supplier # 05-8000-K050, Permeability: 52 cfm, Porosity: 50 micron, Warp/Weft: Mono/Mono, Weave: Double Layer Twill, Weight: 13.0 oz/yd	Date:	12-Nov-10







Relationships:

Unit Weight =	1.189		* Cake Thickness
Form Time =	6.750	*	(Unit Weight / 20.73)^2
Moisture =	43.453	*	(Drytime/Unit Weight) ^ -0.013
DryTime/UnitWeight =			(Moisture / 43.453) ^ (1/-0.013)

Project No. : 12323-003
Reference: Starfields
Testing Instrument: Haake RS75
Sensor: FL 22 (Vane)

Date: November 11, 2010
Operator(s): M.G.

Rheology - Results Summary

Sample: Thickened Leach Residue

	Test Code	Test Method	Size, d_{80} microns	[1] ASG g/cm^3	Sample [2] SG g/cm^3	[3] a ASG/SG	Temp ºC	Solids %	[4] Dens. g/L	Shear Stress [5] Peak, Pa	Notes \ Observations
	T3	Vane	78	4.18	4.23	0.99	50	43.4	1957	17	See Notes Below
	T4	↓	↓	4.19	↓	0.99	↓	40.3	1902	7	See Notes Below

Notes:

The CCRV (Concentric Cylinder Rotational Viscometry) system could not be used due to the corrosive nature of the sample, making it incompatible with the material of construction of this system. Therefore, a vane sensor was used for determination.

The Leach Residue sample (pH = <1) was tested as a flocculated sample with Ciba Magnafloc 155 at a dosage of 48 g/t.

The values are based on data produced by the unsheared slurry sample.

Shear Stress value was produced by subjecting the slurry sample to a constant rotation at 0.10 s^{-1} for 5 minutes.

[1] ASG - Actual specific gravity of dry solids, calculated from the measured pulp density ([4] Dens.)

[2] SG - Dried solids specific gravity, measured with the pycnometer

[3] α - Correction Factor, i.e. solids density ratio

[4] Pulp density in g/L

[5] Maximum (peak) initial shear stress achieved.

32 APPENDIX 3

PROCESS DEVELOPMENT AND CAPEX/OPEX UPDATE FOR PROCESS FACILITIES

CAPEX AND OPEX UPDATE FOR STARFIELD RESOURCES

Ferguson Lake Project

PROCESS DEVELOPMENT

AND

CAPEX /OPEX FOR PROCESS FACILITIES

FOR

STARFIELD METALLURGICAL RECOVERY PLANT

REPORT: A11-115-SR-SS101 REV0

Prepared by: P. Evans

Revised By: E. Burga

Andeburg Consulting Services Inc (ACSI)

November 2011

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EXECUTIVE SUMMARY

ACSI has produced an updated capital and operating cost estimate with an accuracy of +/- 40% for the Starfield Ferguson Lake project corresponding to scoping study level.

The process facilities evaluated in this work include: The concentrator: the pipeline to transfer concentrate and the processing facilities using the HCl based technology.

The concentrator is configured to receive run of mine ore. The ore will be crushed, ground, treated by flotation producing concentrate for transfer to the processing facility and tailings for disposal. A 270 km pipeline will deliver concentrate to Rankin Inlet where it will be filtered ahead of the process plant. The process plant will treat concentrate to recover valuable metals as well as producing energy by using steam from an acid plant. Excess power will be distributed for use at the concentrator site making the whole operation energy self-sufficient.

The mining rate will be 6,000 tpd of ore from which the concentrator will produce 3,311 tpd of bulk sulphide concentrate. The plant will produce:

- Nickel: 11,279 tpa,
- Copper: 18,363 tpa
- Cobalt: 1,296 tpa

The plant will also generate:

- H_2SO_4: 856,142 tpa (net),
- Hematite 744,785 tpa, and
- Leach residue 241,357 tpa containing all the PGEs.

The CAPEX has been completed at the level of scoping study and estimated at an accuracy of 40%. The following are the main results:

- The equipment cost is estimated at CAD $ 697,828.200,
- The installed capital cost is estimated at CAD $ 834,835,100 excluding indirect costs, and
- The total investment for the processing facilities have been estimated at CAD $ 1,173,144,100.

The CAPEX includes the concentrator, the concentrate pipeline and the process plant with auxiliaries including a concentrate sulphuric acid tank farm. Facilities that are not included are: Mine development, Tailings disposal at the concentrator, Hematite disposal at the process plant and ship loading facilities for concentrate acid and final product.

The operating cost is estimated to be CAD $ 40,533,230 per year, which is equivalent to 59.32c/lb of combined Ni+Cu+Co product exclusive of any byproduct credit for platinum group metals, sulfuric acid or excess power.

In the process plant, the primary leach produces H_2S. It is proposed to burn the gas in air and convert the resultant SO_2 to H_2SO_4 in a Haldor Topsoe acid plant. This type of plant is a little more expensive in capital cost than other types of acid plants, but maximizes the amount of energy recovered from the exothermic reactions. In order to increase the power production, sulfur will be recovered from the leach residue and will be burnt to SO_2 in the acid plant. The study assumes that the acid can be stored, shipped and sold. The acid will be stored for 7 months and shipped during the 5-month summer shipping season. Some of the acid will be used in the HCl regeneration step.

Most of the high-pressure steam generated in the acid plant will be used to generate power. Some of the steam generated in the acid plant will be used in the evaporation, oxidation and hydrolysis stages to maximize the efficient use of recovered energy.

Arithmetek Inc. (Arithmetek) produced the mass/energy balance upon which the equipment sizing is based. SGS Minerals Services, Lakefield (SGS) performed the experimental work upon which the mass/energy balance is based. ACSI used the balances and the data from SGS to size equipment and generate CAPEX and OPEX estimates. Each piece of equipment has been sized and evaluated for suitability to the application. This has been necessary because of the novelty of the process and because of the aggressive nature of the liquids, solids and gases.

Additional test work should be carried out to confirm operating parameters. The equipment to be used in the test plant should be made of material pre-selected for use in the main plant. The results from the tests should be used in continuing engineering studies and to solve materials of construction concerns.

The preliminary plant layout outlines the proposed process plant configuration including all process stages and supporting plants. The total area required for the process facilities is 170 m x 380 m and the acid storage facility with 35 tanks will require additional area of 350 m x 520 m.

The process plant will likely be located at Rankin Inlet. In terms of the processing plant, the site selection will be related to the availability of port facilities to ship sulphuric acid.

SECTION A

2.0 PROCESS DEVELOPMENT

2.1 Introduction

Starfield Resources owns the Ferguson Lake massive sulfide ore-body located near Rankin Inlet in Nunavut, containing base metals and precious metals. The ore body has been found to be amenable to concentration by flotation. A 6,000 tpd concentrator will be installed in the mine site. The concentrate slurry will be pumped to Rankin Inlet using a 270 km pipeline. Tailings will be stored at the mine site. Water from Ferguson Lake will be used to slurry the concentrate. The slurry will be filtered at the processing site and excess water will be used in the process.

There is no infrastructure at site so the treatment plant needs to be energy efficient to offset the high cost of bringing in fossil fuels. Starfield, SGS and Arithmetic have developed a chloride-based hydrometallurgical process to treat the concentrate. HCl reacts with pyrrhotite to release H_2S which can be burnt with air to exothermically produce sulfuric acid and energy.

Soluble iron is oxidized with oxygen to form ferric chloride, which is hydrolyzed to form volatile HCl and stable hematite which will be stored at the treatment plant site. The base metal sulphides are dissolved in a secondary, oxidizing leach.

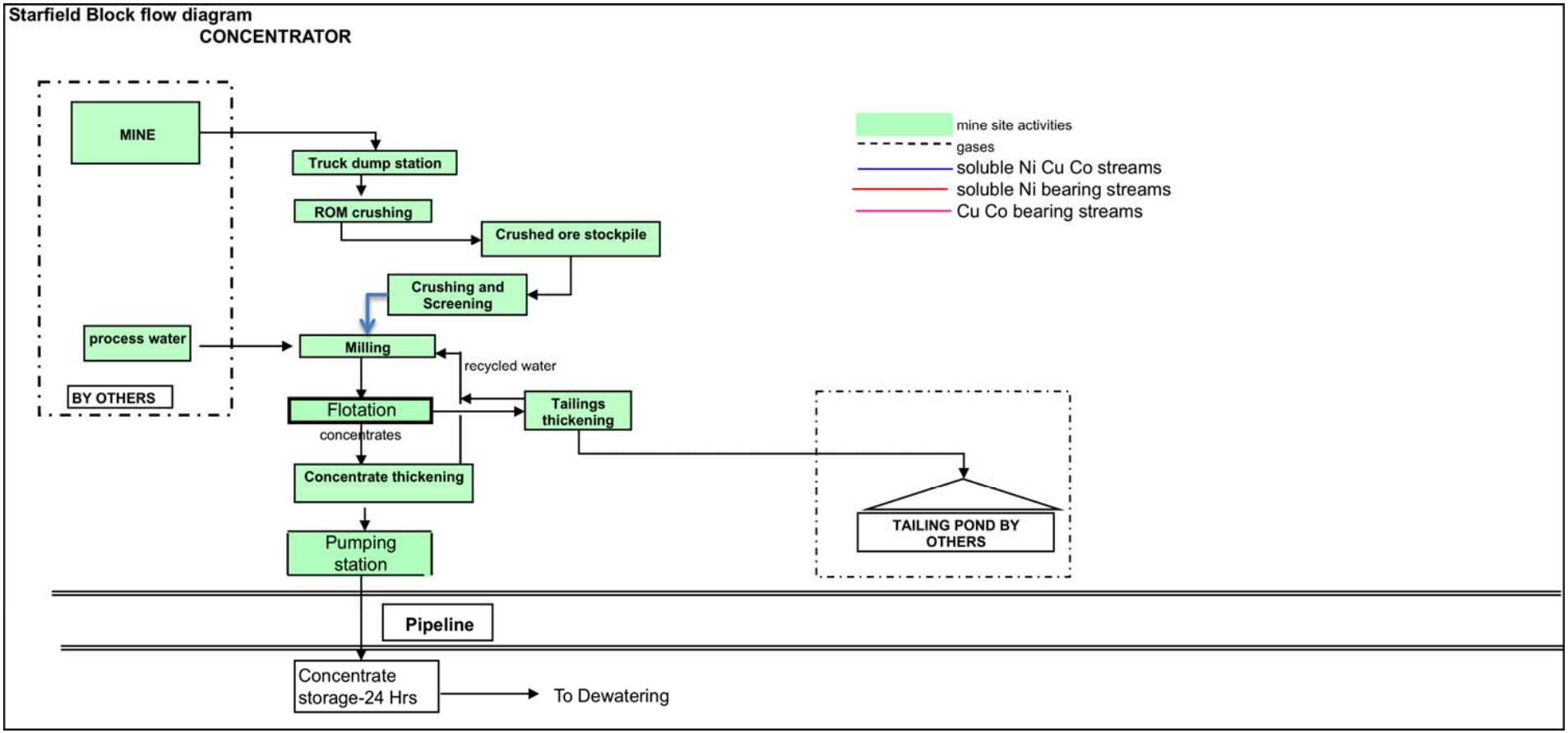
Base metals will be recovered from the pregnant solution by liquid ion exchange and electro-winning to produce high purity metals to minimize shipping costs.

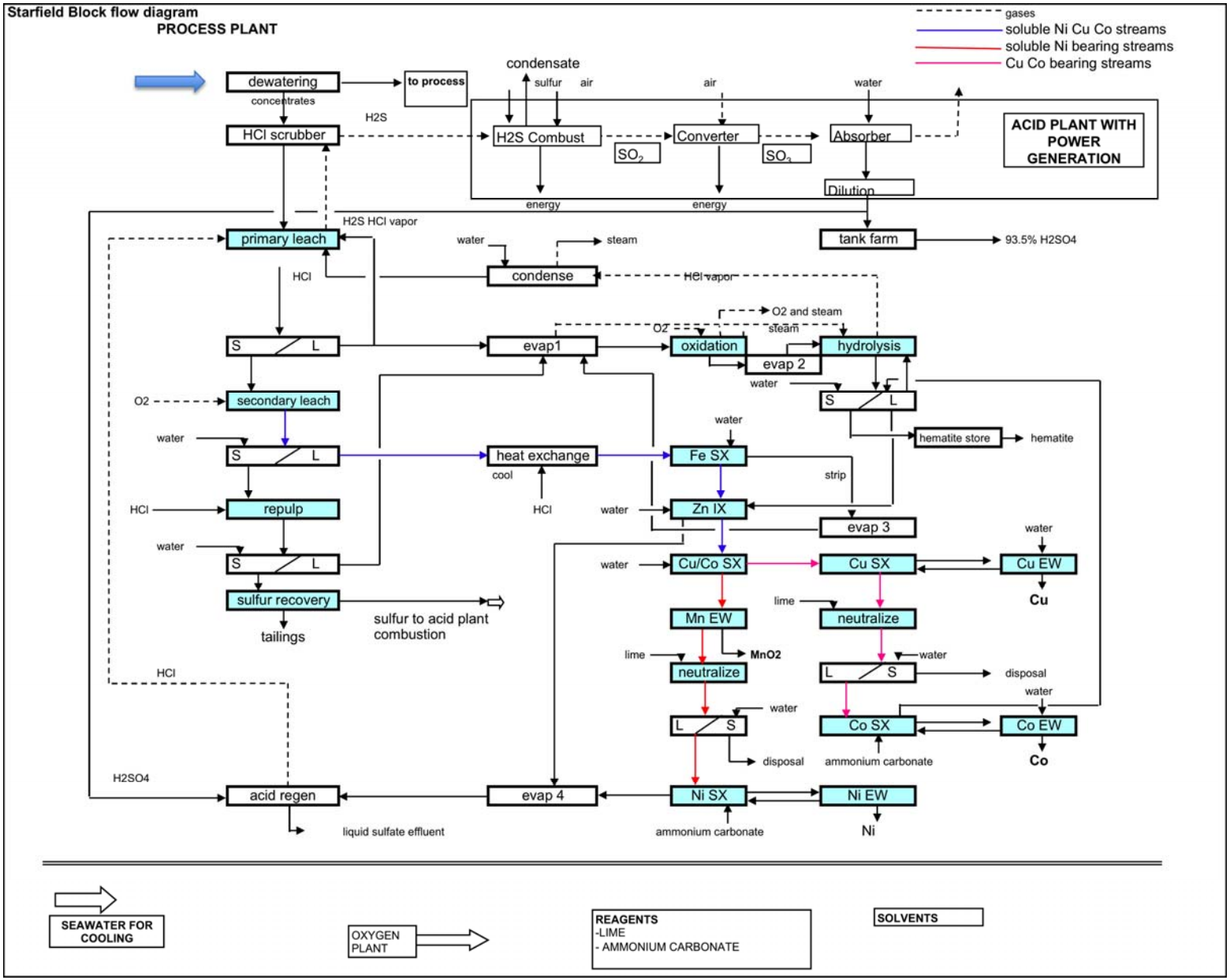
A low grade precious metal concentrate will be produced as a leach residue since these metals are insoluble. The removal of sulfur from this residue by flotation is proposed. The recovered sulfur will be burnt in the acid plant.

On-site energy generation from process gas waste heat boilers is critical to the viability of the project. Importing fossils fuels is not only very expensive but also generates a considerable volume of carbon dioxide, which would be emitted to atmosphere.

RPA has been employed to provide overall engineering and costing for the mine and process plant. ACSI has been employed to assist with engineering and costing of the process plant.

2.2 Process Block Flow Diagram




Starfield Block flow diagram
PROCESS PLANT
gases
soluble Ni Cu Co streams
soluble Ni bearing streams
Cu Co bearing streams
dewatering
concentrates
to process
condensate
sulfur
air
air
water
H2S
HCl scrubber
H2S Combust
Converter
Absorber
ACID PLANT WITH POWER GENERATION
SO2
SO2
Dilution
energy
energy
H2S HCl vapor
primary leach
water
steam
condense
tank farm
93.5% H2SO4
HCl
HCl vapor
O2 and steam
O2
steam
S
L
evap1
oxidation
evap 2
hydrolysis
water
O2
secondary leach
S
L
water
hematite store
hematite
water
S
L
heat exchange
cool
HCl
Fe SX
strip
HCl
repulp
water
Zn IX
evap 3
water
water
S
L
water
Cu/Co SX
Cu SX
Cu EW
Cu
sulfur recovery
tailings
sulfur to acid plant combustion
Mn EW
lime
neutralize
MnO2
HCl
lime
neutralize
water
L
S
disposal
water
L
S
water
Co SX
Co EW
Co
disposal
ammonium carbonate
H2SO4
acid regen
evap 4
Ni SX
Ni EW
liquid sulfate effluent
ammonium carbonate
Ni
SEAWATER FOR COOLING
OXYGEN PLANT
REAGENTS
-LIME
- AMMONIUM CARBONATE
SOLVENTS

2.3 Ore Composition and Production

Ore will be mined; crushed, wet ground with water from Ferguson Lake, subject to flotation and the concentrates will be pumped via a slurry pipeline to the dewatering section of the processing plant. The mill tailings will be stored at the mine site. Some of the tailings will be used for back fill at the mine.

Valuable base metals in the ore include nickel, copper and cobalt. Precious metals include platinum and palladium.

6,000 tpd ore will be treated containing:

- 0.59% Ni
- 0.91% Cu
- 0.07% Co

The following product production is forecast from 6,000 tpd ore:

- Ni: 11,279 tpa
- Cu: 18,363 tpa
- Co: 1,296 tpa
- H_2SO_4: 856,142 tpa (net)
- Hematite: 744,785 tpa
- Leach residue: 241,357 tpa

2.4 Process Application

The process uses leaching at high total chloride concentration to convert the sulphide minerals to dissolved chlorides, hydrogen sulphide and elemental sulphur. The resulting solutions are concentrated by evaporation, oxidized and heated to convert the dissolved ferrous chloride to solid hematite (Fe_2O_3) and gaseous hydrogen chloride (HCl). The HCl is recycled and the Fe_2O_3 leaves as a by-product or a benign waste, depending on market conditions for iron ores.

The main source of energy in the process is high pressure steam generated in the acid plant, where the H_2S and elemental sulphur generated in the leach steps is converted to sulphuric acid. The high pressure steam is expanded to 28 Bar to generate power, and the 28 Bar steam is condensed to provide the energy needed in the hydrolysis section. Residual 28 Bar steam is expanded to 10 Bar to raise more power and the 10 Bar steam is used to drive evaporation in the process. Energy recovery in the process is achieved by condensing recycled HCl/steam and raising pure steam at atmospheric pressure in heat exchangers. The residual 10 Bar steam is expanded to atmospheric pressure and the various high pressure condensates are flashed to atmospheric pressure. Some of the resulting atmospheric pressure steam is used to drive the lower-temperature evaporation

steps in the process and the balance is used to generate more power in a condensing turbine.

The solution remaining after the removal of ferrous iron as HCl and Fe_2O_3 is purified by solvent extraction of residual ferric chloride, then by ion exchange to remove zinc chloride. The purified solution is subjected to further solvent extraction in stages to separate out the individual base metals. The individual base metals are recovered as pure metal products by electrolysis.

3.0 Process Description

3.1 Filtration Plant

There are three concentrate storage tanks, each with an storage capacity of eight hours. The filters are tower type for semi-continuous concentrate filtration.

3.2 Primary Leach Stage

3.2.1 Primary Leach Overview

Slurried concentrate is transported from the mine site by pipeline into the filtration plant. The filter cake is slurried and transported into the primary leach using ferrous chloride solution recycled from the primary leach thickener. Leaching takes place in 4 cascading reactors. Slurry from the last reactor is thickened and filtered. Some of the thickener overflow is recycled to slurry the concentrate filter cake and the remainder goes to the oxidation stage. The filter cake goes to the secondary leach stage. Residence time in each reactor is about 30 minutes.

Pyrrhotite reacts with HCl to form $FeCl_2$ and H_2S. The H_2S is transferred to the acid plant to produce sulfuric acid.

The reactions between air and H_2S, SO_2 and O_2 and SO_3 and H_2O are exothermic and a significant amount of energy is recovered in waste heat boilers in the acid plant.

Base metals and precious metals do not react appreciably with HCl under reducing conditions.

The primary leach will be heated by recycled concentrated HCl and recycled brine. Boiler feed water in heat exchangers condenses most of the recycled HCl gas. Sufficient gaseous HCl remains to close the heat balance around the primary leach.

The H_2S/H_2O gases leaving the primary leach carry with them some HCl gas. This is removed in a scrubber using the incoming concentrate.

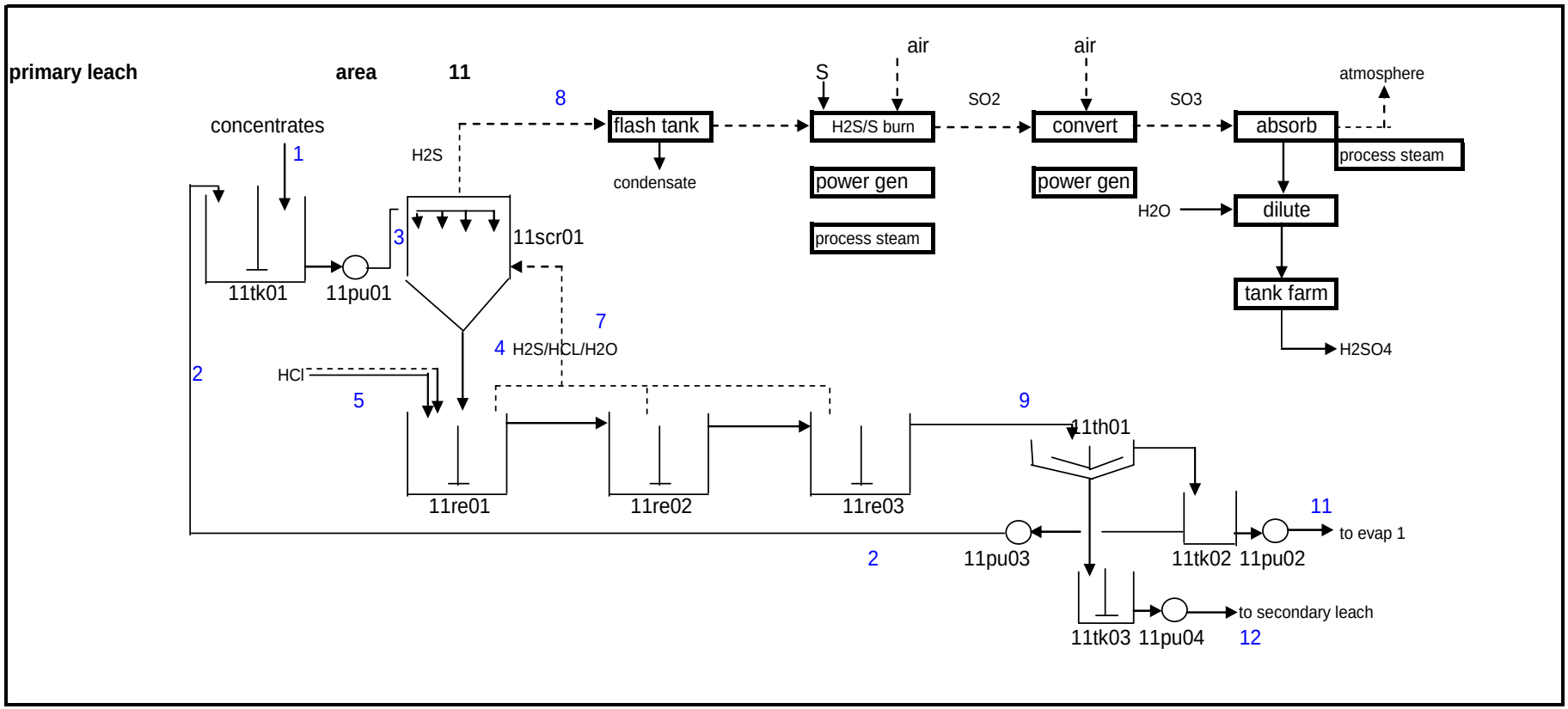
Most of the H_2O vapor in the off gasses is removed by cooling and condensation. Materials of construction for this equipment need more evaluation.

Care will be needed in designing the primary leach stage because H_2S is very toxic and mixtures of H_2S and air are explosive between 4% and 46% H_2S.

The main chemical reaction for this stage is:

$$Fe_7S_8+14HCl=7FeCl_2+7H_2S+S$$

3.2.2 Primary Leach PFD



3.3 Oxidation Stage

3.3.1 Oxidation Stage Overview

The solution from primary leach is concentrated and heated in an evaporation stage. $FeCl_2$ in the liquid from evaporation is oxidized to $FeCl_3$ and hematite using oxygen in 3 cascading reactors. The reactions are exothermic and the temperature is held to 120°C by evaporation of boiler feed water. One third of the iron precipitates as hematite. Each reactor has a residence time of about 37 minutes.
The main chemical reaction for this stage is:

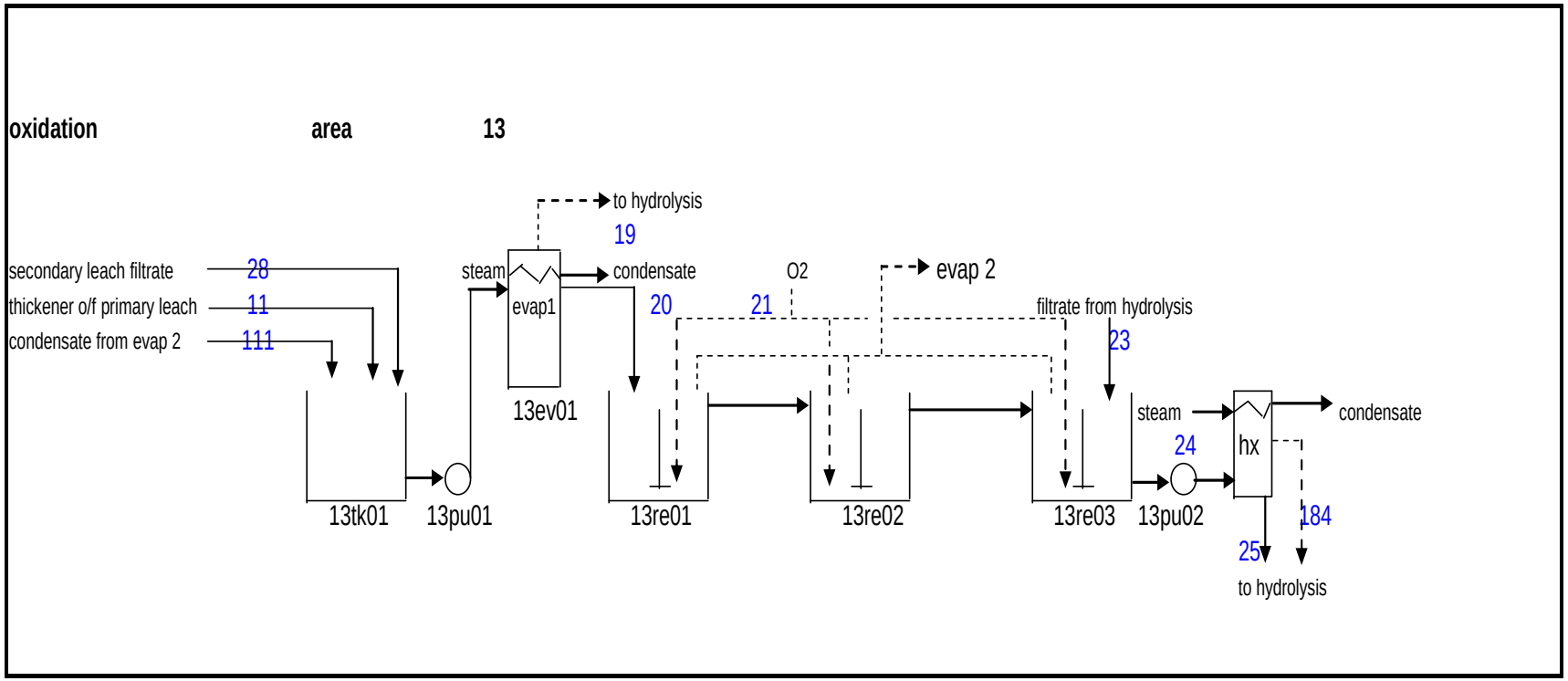
$$12FeCl_2+3O_2=2Fe_2O_3+8FeCl_3$$

The slurry proceeds to hydrolysis.

A second evaporation stage further concentrates and heats the slurry. High pressure steam from the acid plant waste heat boiler is passed through heating coils to evaporate water needed for the hydrolysis stage. Condensate is returned to the waste heat boiler.

HCl vapor and water vapor pass through the heat exchanger are condensed and sent to primary leach.

3.3.2 Oxidation Stage PFD



3.4 Hydrolysis Stage

3.4.1 Hydrolysis Stage Overview

The temperature of the slurry is raised to 180°C by using waste heat boiler steam in coils.

Hematite is precipitated and concentrated HCl vaporizes in 3 reactors. The vapor is condensed by boiler feed water in a heat exchanger. The cooling water is vaporized in the heat exchanger and the resultant water vapor forms steam which is sent to the waste heat boilers. Condensed concentrated HCl goes to primary leach where it reacts with pyrrhotite.

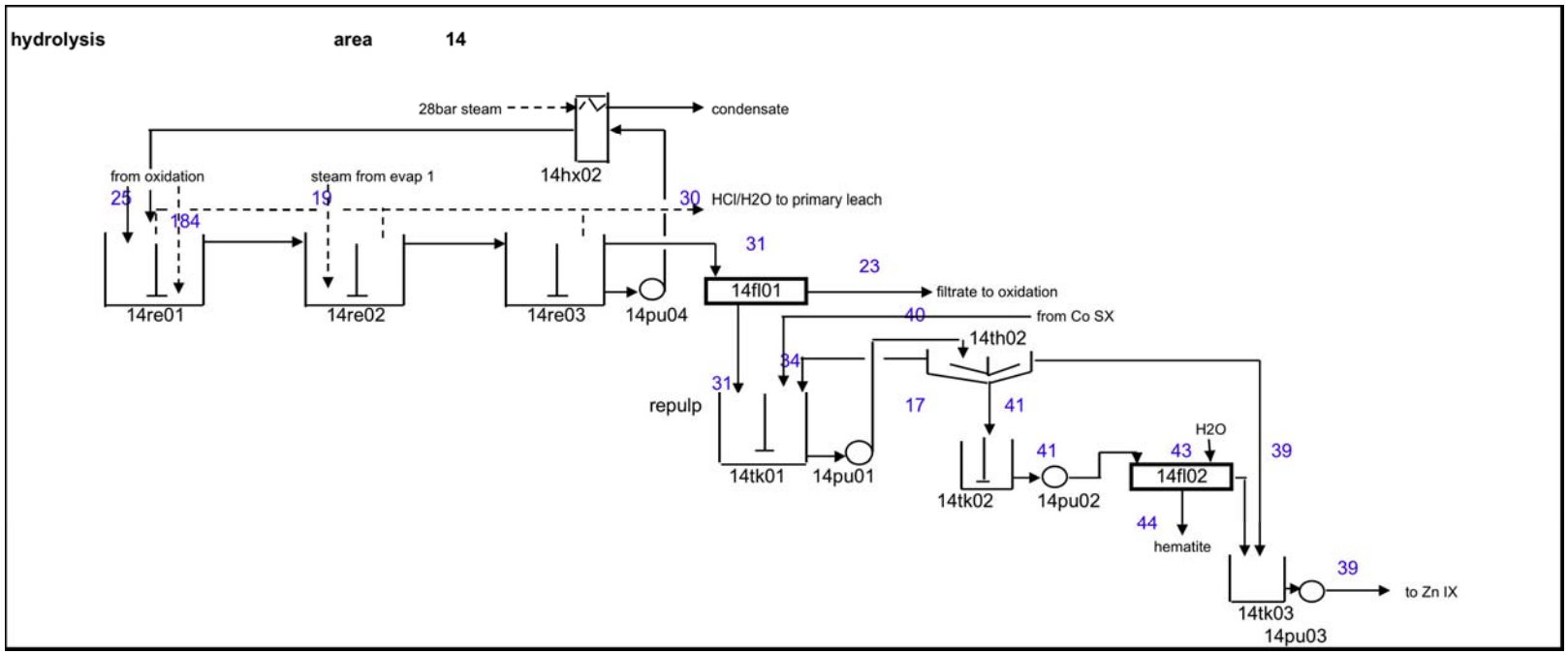
The main chemical reaction for this stage is:

$$2FeCl_3+3H_2O=Fe_2O_3+6HCl$$

A high temperature filter separates the hematite from the liquid. The filtrate is recycled to hydrolysis or oxidation. The liquid is now a fused salt. The filter cake is re-pulped with the raffinate from cobalt SX. The slurry is filtered and the hematite is stored prior to shipment. The filtrate goes to Zn IX.

The materials of construction for the reactors, heat exchangers and hot filter need to be further investigated.

3.4.2 Hydrolysis PFD



3.3 Secondary Leach Stage

3.3.1 Secondary Leach Stage Overview

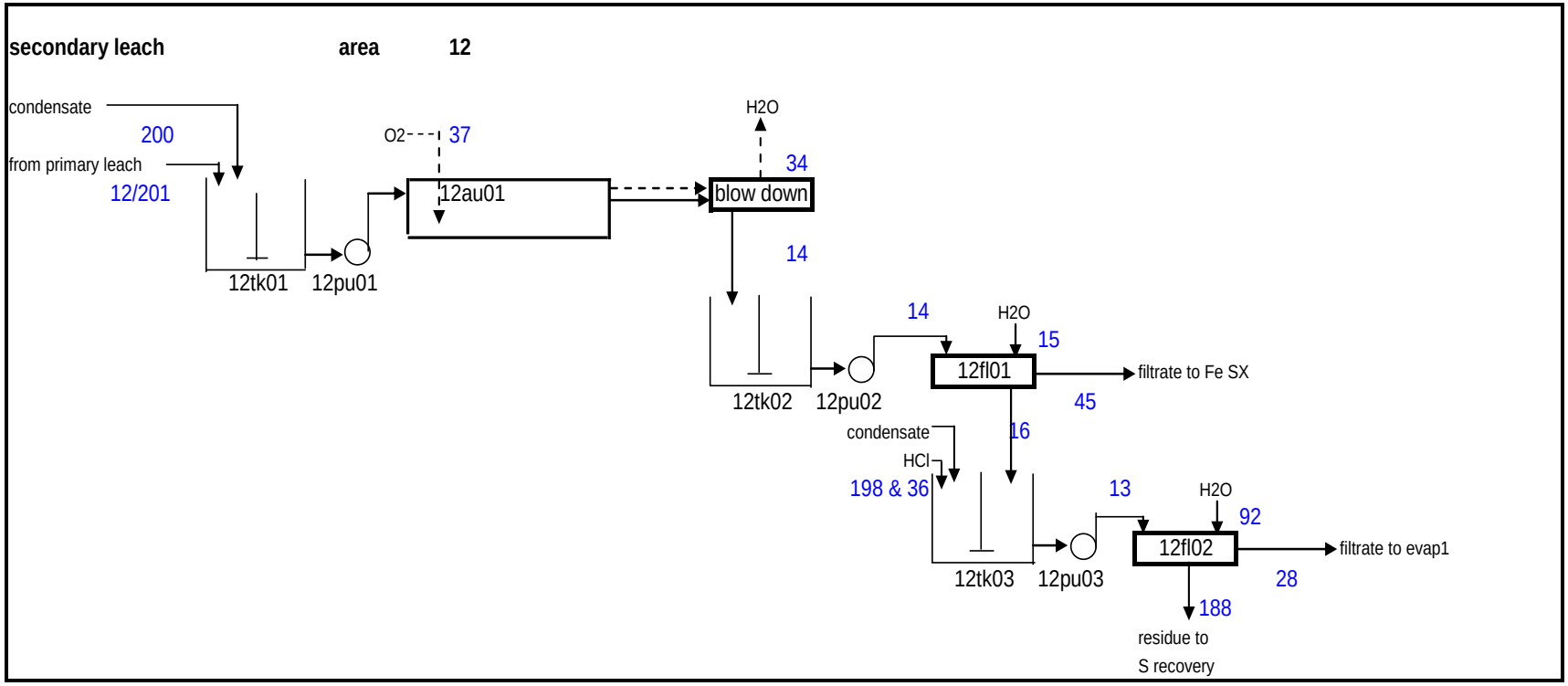
Primary leach underflow is treated with oxygen in a low pressure autoclave to oxidize ferrous to ferric, to dissolve the base metals and to precipitate most of the soluble iron as goethite. A residence time of 5 hours in the autoclave is required. Precipitation of goethite minimizes the amount of $FeCl_3$ going to Fe SX and allows the soluble base metals to proceed to base metal recovery without going through hydrolysis.

The main chemical reactions in this stage are as follows:

$$CuFeS_2+4FeCl_3=CuCl_2+5FeCl_2+2S$$
$$NiS+2FeCl_3=NiCl_2+2FeCl_2+S$$
$$CoS+2FeCl_3=CoCl_2+2FeCl_2+S$$
$$FeCl_3+2H_2O=FeOOH+3HCl$$

Slurry leaving the secondary leach is thickened and filtered on a belt filter. The filter cake is washed with condensate. The filter cake is repulped with recycled concentrated HCl to dissolve the goethite. The filtrate goes to the evaporation step ahead of oxidation. The filter cake is re-pulped and elemental sulfur is recovered by flotation. The leach residue contains the PGMs and will be stored until a treatment process is available.

3.3.2 Secondary leach PFD



3.4 Anion Solvent Extraction Stage

3.4.1 Anionic SX Overview

The raffinate from $FeCl_3$ SX goes to Zn IX and the raffinate from Zn IX goes to nickel SX. The resin is stripped with water and the eluate is sent to HCl recovery.

Most of the metals in solution in a high chloride media are present as anions (e.g.$CuCl_2^{2-}$).

Solution from Zn SX is by contact with an organic solvent that loads Cu, Co, as anions.

The loaded solvent is stripped with water and the resulting loaded strip solution is fed to the Copper Solvent Extraction.

The raffinate from the anionic solvent extraction step proceeds to MnO_2 electro-winning.

3.4.2 Copper SX Overview

The eluate from the anionic SX is contacted with a copper specific organic solvent. The loaded organic phase is stripped with sulphuric acid in the spent electrolyte from the subsequent electrowinning step.

The loaded strip solution from the cationic solvent containing $CuSO_4$ proceeds to electrowinning (EW) where copper cathodes are produced in 37 EW cells. H_2SO_4 is regenerated.

$$2CuSO_4+2H_2O=2Cu+2H_2SO_4+O_2$$

3.5 The raffinate from the copper SX step goes to the cobalt SX step in the cobalt SX-EW loop.Cobalt Solvent Extraction Stage

3.5.1 Cobalt SX Stage Overview

Raffinate from the copper SX step goes to Co SX. Cobalt is loaded onto a cobalt specific organic solvent. The loaded solvent is stripped with sulphuric acid in spent electrolyte from the subsequent cobalt electrowinning step.

The loaded strip solution from the cobalt SX step contains $CoSO_4$ and proceeds to electrowinning (EW) where cobalt cathodes are produced in 8 EW cells. H_2SO_4 is regenerated.

$$2CoSO_4+2H_2O=2Co+2H_2SO_4+O_2$$

The raffinate from Co SX is used to quench and dilute the hot hematite filter cake from the hydrolysis section.

3.6 Nickel Solvent Extraction Stage

3.6.1 Nickel SX Stage Overview

Nickel is present in the solution leaving secondary leach as a cation. It passes through the anion SX mixer settlers without loading. It is recovered from the solution leaving MnO_2 EW by a nickel specific cationic organic solvent. The loaded organic phase is stripped with sulphuric acid in the spent electrolyte from the subsequent nickel electro-winning step.

The proceeds to electro-winning (EW) where nickel cathodes are produced in 42 EW cells. H_2SO_4 is regenerated.

$$2NiSO_4+2H_2O=2Ni+2H_2SO_4+O_2$$

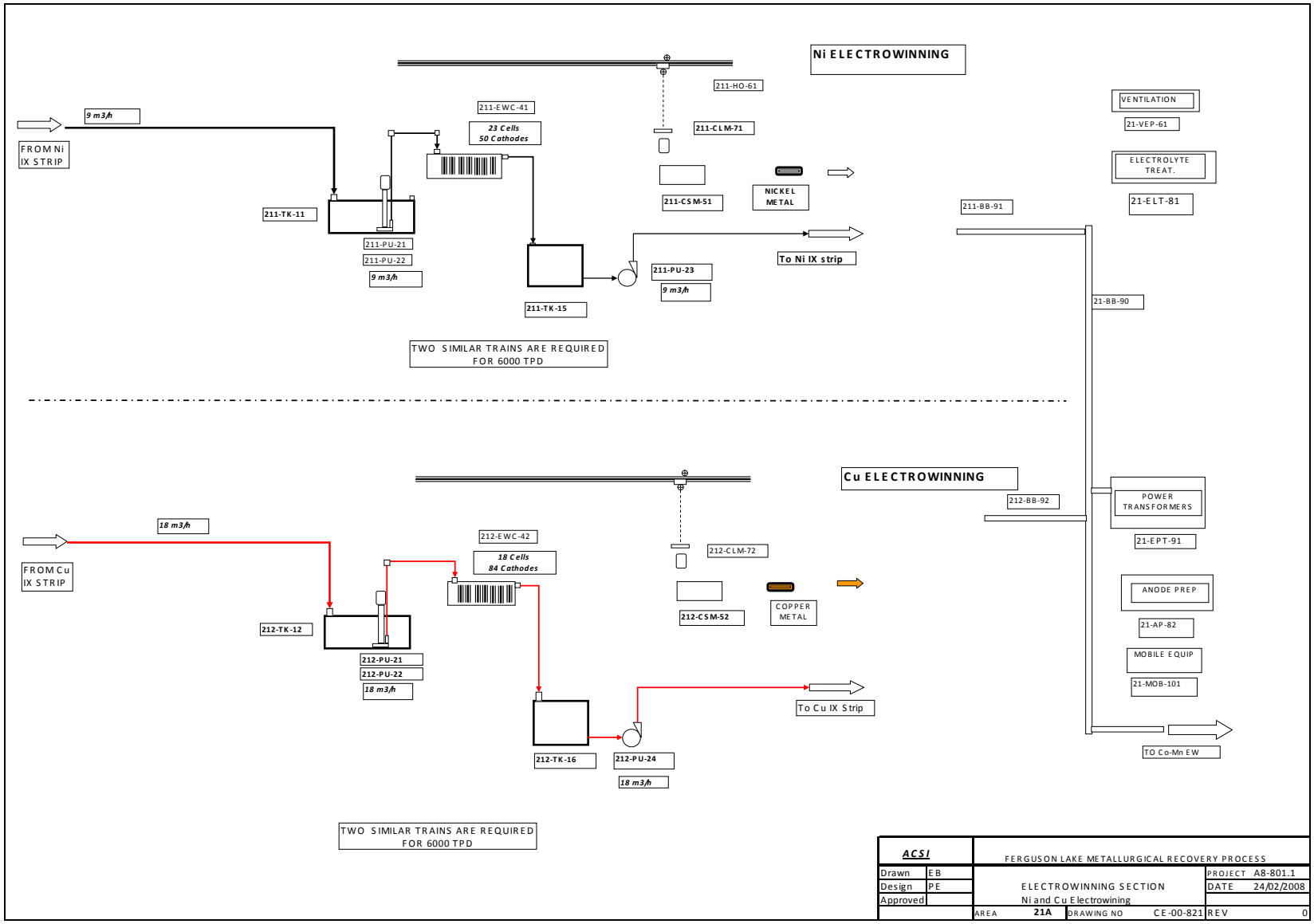
The raffinate from Ni SX goes to evaporation and then to HCl recovery. Concentrated H_2SO_4 is mixed with the raffinate and HCl vapor is released. Soluble metal sulfates such as $MgSO_4$ and Na_2SO_4 are formed. They become a liquid effluent.

3.7 Copper Electro-winning Stage

3.7.1 Copper EW Overview

Eluate from Cu SX goes to 37 electro-winning cells each containing 85 1m by 1m stainless steel cathodes. The current density will be 300 amps/m^2. Cathodes will be stripped in an automatic stripping machine.

3.7.2 Copper EW PFD



3.8 Nickel Electro-winning Stage

3.8.1 Nickel EW Overview

Eluate from Ni SX goes to 42 electro-winning cells each containing 50 1m by 1m stainless steel cathodes. Cathodes will be stripped in an automatic stripping machine. The anodes will have cloth diaphragms to control conditions around the cathodes.

3.8.2 Nickel EW

Nickel EW is shown with the Cu EW PFD

3.9 Cobalt Electro-winning Stage

3.9.1 Cobalt EW Stage Overview

Eluate from Co SX goes to 8 electro-winning cells each containing 3 1m by 1m stainless steel cathodes. Cathodes will be stripped in an automatic stripping machine. The anodes will have cloth diaphragms to control conditions around the cathodes.

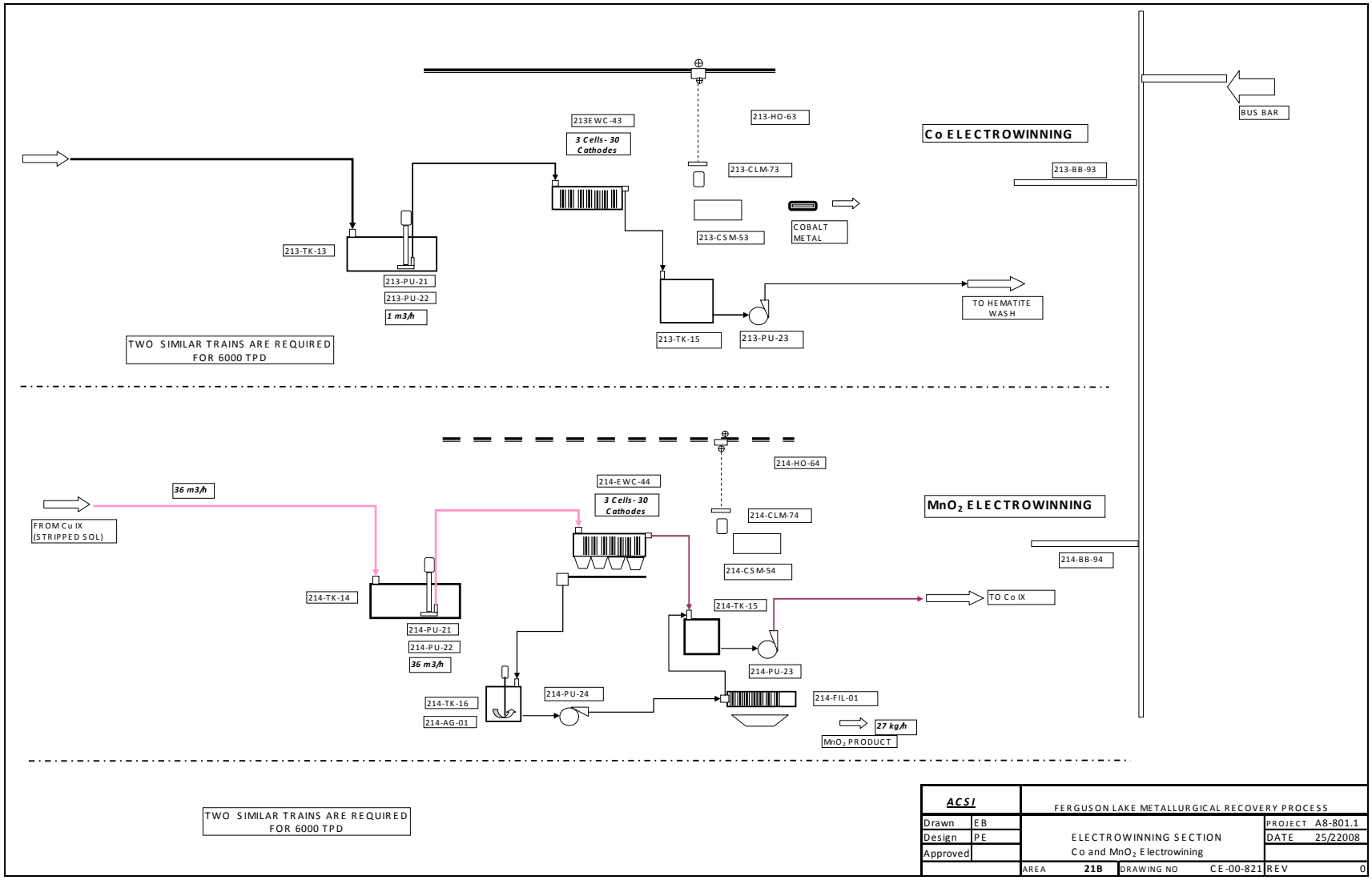
3.9.2 Cobalt EW PFD

Cobalt EW PFD Shown with the MnO_2 EW PFD

3.10 MnO_2 Electro-winning Stage

3.10.1 MnO_2 EW Overview

3.10.2 Co and MnO_2 electrolysis PFD



3.10.3 MnO2 Electro-winning

Raffinate from anionic Cu/Co SX goes to MnO_2 electrolysis in 8 cells. MnO_2 is deposited at the anodes. The MnO_2 falls off the anodes and is pumped from the cells to a pressure filter. The filter cake will be sold or stored at site.

The solution from MnO2 EW is neutralized with lime and then is sent to Ni EW

3.11 Iron SX Stage

Solution from secondary leach goes to ferric chloride solvent extraction. The raffinate goes to zinc IX. The solvent is stripped with water and the eluate is sent to hydrolysis.

3.12 Zinc IX Stage

Zinc is removed from iron SX raffinate by a zinc specific resin in IX columns. The raffinate goes to Cu-Co anionic SX. The resin is stripped with water and the eluate goes to acid regeneration.

3.13 Energy Use Optimization

Energy management is of prime importance because of the high cost of generating power at site. The oxidation of H_2S, SO_2 and absorption of SO_3 are very exothermic. Most of the high pressure steam produced in the acid plant will be used to generate power. Some 28bar steam from the acid plant is needed for the hydrolysis stage. Some low pressure steam from the acid plant is used for the evaporation stages.

Energy balance

				MW
turbine 1				12.2
turbine 2				3.1
turbine 3				9.4
turbine 4				27.9
total				52.5
power to condensate pumps				0.8
power to vacuum pumps				0.0
power to acid plant				9.5
power for mining				6.0
power for crushing grinding and flotation				7.4
power for oxygen plant				3.2
power for electro winning				10.6
power for pumping agitation etc				5.3
total				42.8
net				9.6

3.14 H_2S Treatment

3.14.1 Sulfuric acid production

The H_2S from the primary leach and the sulfur from secondary leach are burnt to SO_2. using air. The reactions are exothermic. A waste heat boiler will cool the gases and generate The 80 Bar steam will be expanded to 28 Bar in a turbine to recover power for generating electricity.

.

The reactions are:

$$2H_2S+3O_2=2SO_2+2H_2O$$
$$2SO_2+O_2=2SO_3$$
$$SO_3+H_2O=H_2SO_4$$

This study assumes that sulfuric acid will be made and sold.

1.1 million tons of acid will be produced each year and there will be a five-month shipping period. The HCl regeneration section of the process will consume 276,834 tpa , leaving 856,141 tpa for shipping.
500,000 tonnes of acid will be stored during the non-shipping season. The acid will be stored in a battery of mild steel tanks each with a diameter of 45m diameter and 9 m high. The tank farm requires an area of 250 m x 520 m. The acid plant will produce 93.5% acid. This acid strength has the lowest freezing point.

4.0 Materials of Construction

4.1 Industrial Plant Reference

Because of the high chloride background, established industry materials of construction will be used.

The Magnola plant in Montreal ran with similar high magnesium chloride solution compositions.

The next sections present the summarized material selection approach used in this study.

4.2 Ore slurry equipment

The ore slurry equipment inside the plant will be Carbon steel-rubber lined with Hypalon type rubber. Agitator will be also rubber lined with this type of equipment.

4.3 Primary leach equipment

The reactor walls will be constructed of:

- 12 mm mild steel
- 12 mm natural gum rubber
- 25 mm furan tiles
- 75 mm low S high tensile graphite brick

Agitators wet parts will be made of Supper Duplex Stainless or Carpenter C lined

4.4 Secondary leach, evaporation, oxidation and hydrolysis Equipment

The reactor walls will be constructed of:

- 12 mm mild steel
- 12 mm natural gum rubber
- 25 mm furan tiles
- 75 mm fused alumina

Agitators will be made of titanium Slurry will be pumped through external heat exchangers

Thickeners and belt filters

Thickeners will be constructed of Carbon steel, rubber lined and ceramic liner.

High temperature Philippe belt filters similar to those at Magnola will be used.

4.5 Ion exchange Equipment

FRP columns will be used.

4.6 Electro-winning Equipment

Standard materials of construction for sulfate based electro-winning systems will be used.

4.7 Alternate Materials of Construction

Alternate materials of construction that can be considered for future evaluation include:

Reactor, Secondary vessels:

- PVDF lined FRP
- Glass lined CS
- Teflon Lined SS
- Hypalon lined SS
- Supper Duplex Alloy
- FRP (Derakene/Hetron types)

Thickeners

- Hypalon lined SS
- PVDF lined FRP

Slurry pumps/Solution Pumps

- Hypalon lined SS
- Teflon lined
- Hastelloy C

Slurry Piping

- Rubber lined CS for low temperature
- PVDF lined CS

Solution Piping

- Rubber lined
- FRP
- CPVC for low temperature
- PTFE Lined CS/304L Stainless for temp up to 230

Miscellaneous process equipment

- Hypalon lined
- Teflon lined
- Hastelloy C

SECTION B:

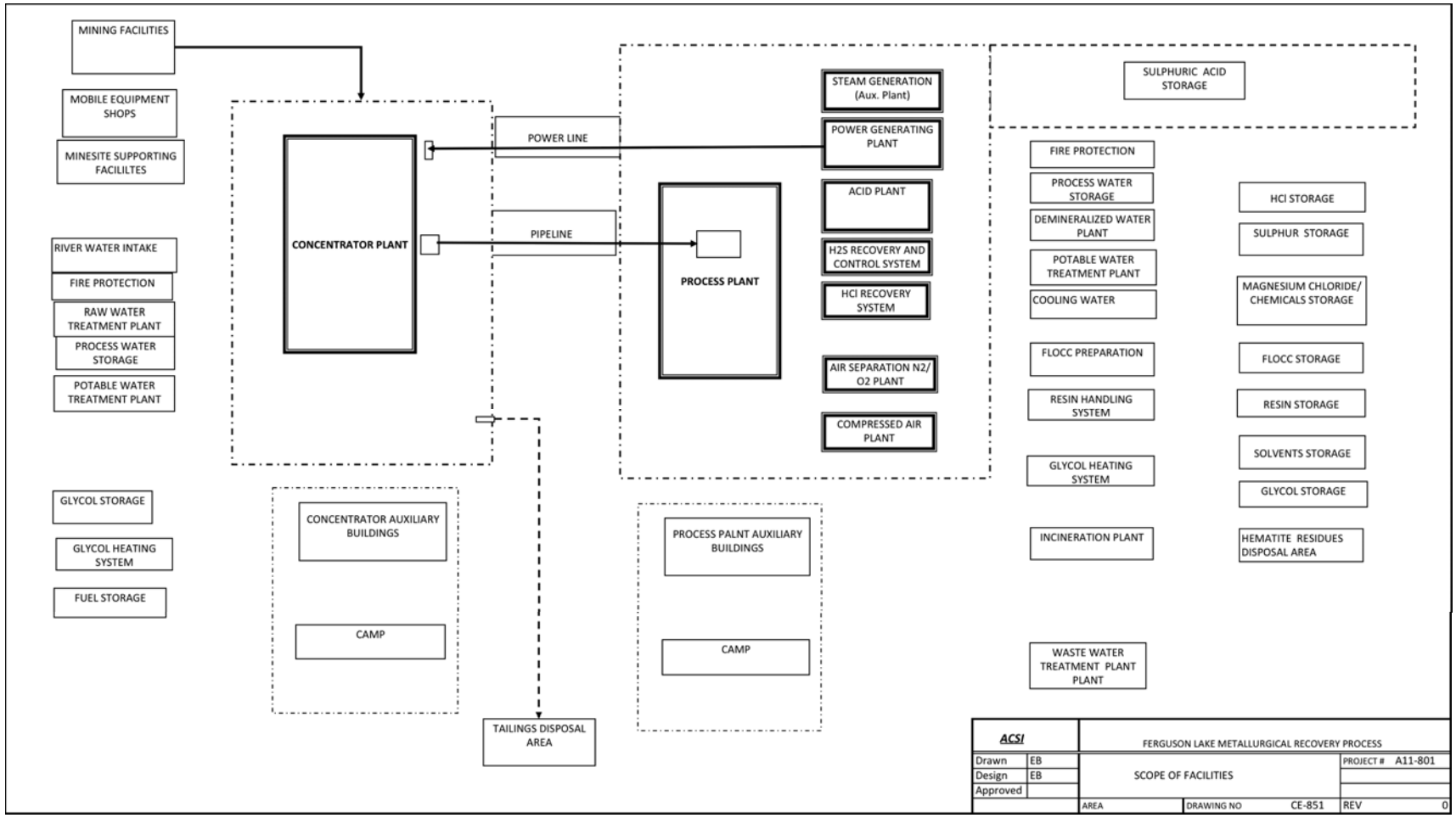
5.0 Process Facilities

5.1 Aspen Model and Equipment Sizing

The process model of the plant produced by Arithmetek was used to size equipment and help to prepare the equipment list.

Equipment sizing is tied into the process model.

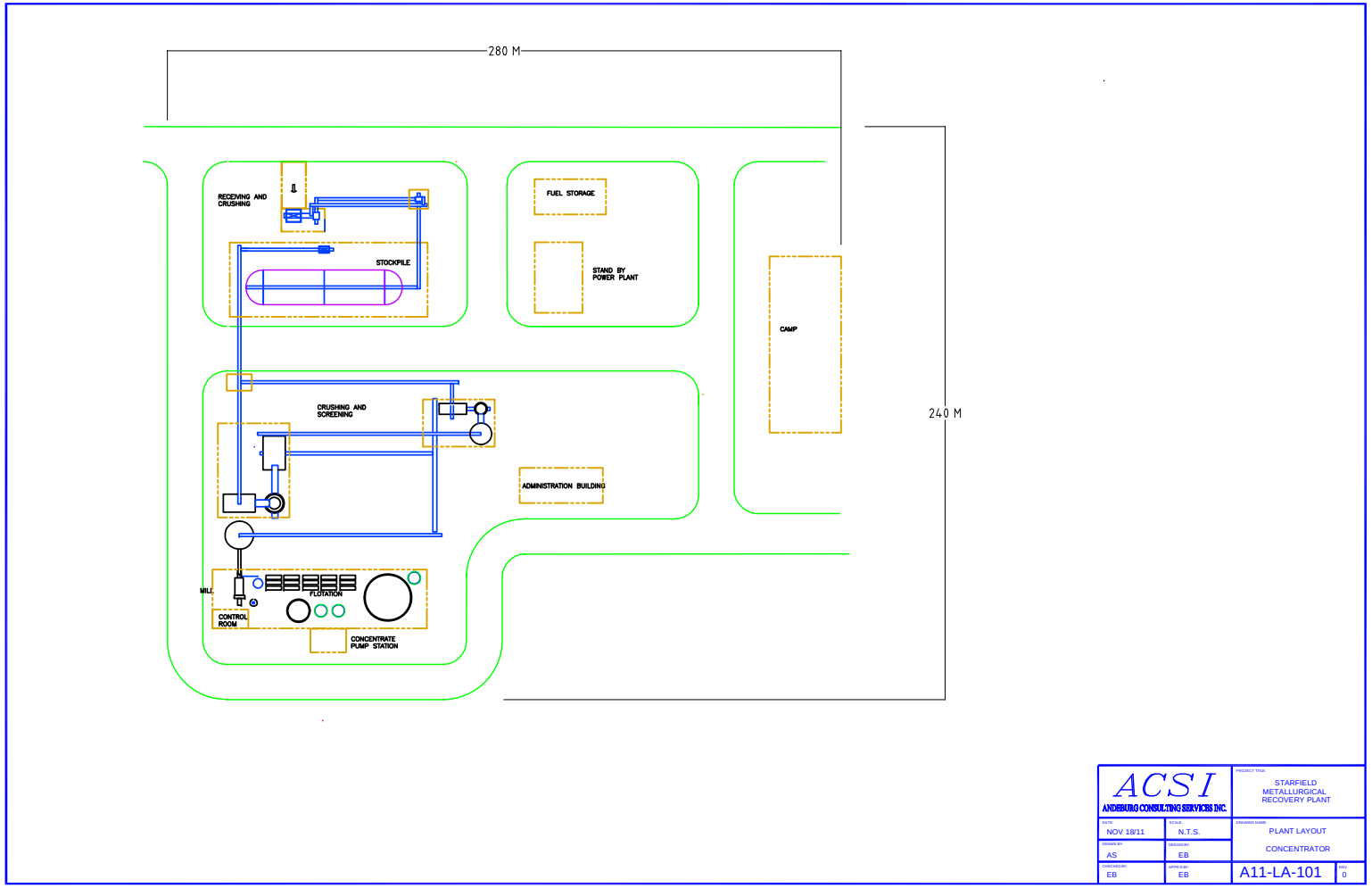
5.2 Process Facilities


MINING FACILITIES
MOBILE EQUIPMENT SHOPS
MINESITE SUPPORTING FACILILTES
RIVER WATER INTAKE
FIRE PROTECTION
RAW WATER TREATMENT PLANT
PROCESS WATER STORAGE
POTABLE WATER TREATMENT PLANT
GLYCOL STORAGE
GLYCOL HEATING SYSTEM
FUEL STORAGE
CONCENTRATOR PLANT
POWER LINE
PIPELINE
PROCESS PLANT
CONCENTRATOR AUXILIARY BUILDINGS
CAMP
PROCESS PALNT AUXILIARY BUILDINGS
CAMP
TAILINGS DISPOSAL AREA
STEAM GENERATION (Aux. Plant)
POWER GENERATING PLANT
ACID PLANT
H2S RECOVERY AND CONTROL SYSTEM
HCl RECOVERY SYSTEM
AIR SEPARATION N2/ O2 PLANT
COMPRESSED AIR PLANT
SULPHURIC ACID STORAGE
FIRE PROTECTION
PROCESS WATER STORAGE
DEMINERALIZED WATER PLANT
POTABLE WATER TREATMENT PLANT
COOLING WATER
FLOCC PREPARATION
RESIN HANDLING SYSTEM
GLYCOL HEATING SYSTEM
INCINERATION PLANT
WASTE WATER TREATMENT PLANT PLANT
HCl STORAGE
SULPHUR STORAGE
MAGNESIUM CHLORIDE/ CHEMICALS STORAGE
FLOCC STORAGE
RESIN STORAGE
SOLVENTS STORAGE
GLYCOL STORAGE
HEMATITE RESIDUES DISPOSAL AREA
ACSI
FERGUSON LAKE METALLURGICAL RECOVERY PROCESS
Drawn EB
Design EB
Approved
SCOPE OF FACILITIES
PROJECT # A11-801
AREA
DRAWING NO CE-851
REV 0

5.3 Plant Layout

The following pages present a conceptual design for the concentrator and the process Plant.

The process plant layout includes the tank farm adjacent to the process plant.


280 M
240 M
RECEIVING AND CRUSHING
STOCKPILE
FUEL STORAGE
STAND BY POWER PLANT
CAMP
CRUSHING AND SCREENING
ADMINISTRATION BUILDING
MILL
FLOTATION
CONTROL ROOM
CONCENTRATE PUMP STATION
ACSI
ANDEBURG CONSULTING SERVICES INC.
STARFIELD METALLURGICAL RECOVERY PLANT
NOV 18/11
N.T.S.
PLANT LAYOUT
AS
EB
CONCENTRATOR
EB
EB
A11-LA-101
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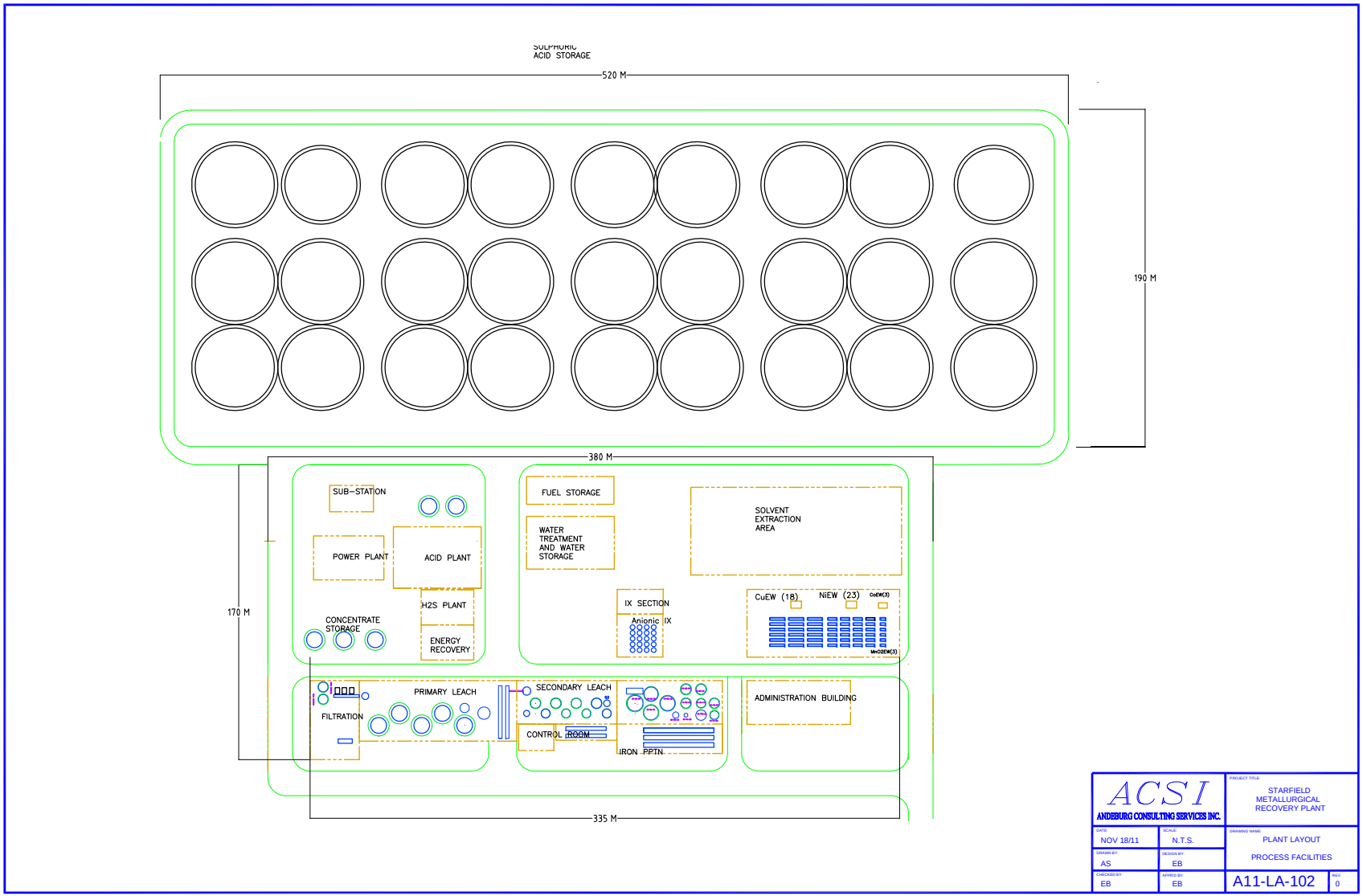
SULPHURIC
ACID STORAGE
520 M
190 M
380 M
170 M
335 M
SUB-STATION
POWER PLANT
ACID PLANT
H2S PLANT
CONCENTRATE
STORAGE
ENERGY
RECOVERY
FUEL STORAGE
WATER
TREATMENT
AND WATER
STORAGE
SOLVENT
EXTRACTION
AREA
IX SECTION
Anionic IX
CuEW (18)
NiEW (23)
CoEW(3)
MnO2EW(3)
PRIMARY LEACH
SECONDARY LEACH
FILTRATION
CONTROL ROOM
IRON PPTN
ADMINISTRATION BUILDING
ACSI
ANDEBURG CONSULTING SERVICES INC.
PROJECT TITLE:
STARFIELD
METALLURGICAL
RECOVERY PLANT
DATE:
NOV 18/11
SCALE:
N.T.S.
DRAWING NAME:
PLANT LAYOUT
PROCESS FACILITIES
DRAWN BY:
AS
DESIGN BY:
EB
CHECKED BY:
EB
APPRD BY:
EB
A11-LA-102
REV.
0

5.4 Equipment List

The equipment list included in the cost estimate is presented in Appendix 2. The list includes preliminary sizing defined in the mass balance and the necessary items to complete the process facilities.

6.0 CAPEX

6.1 Capital Cost Estimate Approach

The estimate presents the costs of the main process equipment identified in the process flow diagram and the factored cost for installation of equipment. A factor of 20% to cover labour cost has been used in all sections of the plant.

The equipment cost was taken from reference data and calculated prices. A preliminary sizing has been completed to satisfy for the process requirements. Worksheets that relate sizing with weight and cost of material of construction were completed to define cost of reactors and tanks.

In order to define costs for facilities related to the IX, Solvent Extraction and EW stages, budget prices were used from other installations and factored to account for size and escalation.

The acid plant cost was obtained from an acid plant supplier. Additional information was provided on the power to be generated in the acid plant including turbine cost for the present application.

Preliminary quotes were also obtained for the oxygen plant.

Price of concentrator equipment was obtained from file reference. Pipeline cost was defined by factoring previous work considering a reduced flow and deleting return water line.

Using Carbon steel as fabrication material and defining weight of steel to be used calculated acid storage tank price.

6.2 Indirect Cost

The indirect cost was obtained using the following factors:

EPCM: 10% of the installed cost
Owner Cost: 3% of the Installed cost
Spare Parts: 2% of the Equipment cost
Transportation: 8% of the Equipment cost
First Fill: 1 % of the Equipment cost

6.3 Cost Estimate Summary

The Summary for the overall project and the summary for each of main section of the plant are as follows:

STARFIELD RESOURCES: FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE SUMMARY SHEET
6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

1.0 DIRECT COST

AREA	DESCRIPTION	MATERIAL	LABOUR	TOTAL
P1	CONCENTRATOR	$97,362,000	$19,472,000	$116,834,000
P2	PIPELINE	$114,201,500	$22,840,300	$137,041,800
P3	PROCESS PLANT	$486,264,700	$94,694,600	$580,959,300
	SUBTOTAL DIRECT COST:	$697,828,200	$137,006,900	$834,835,100

2.0 INDIRECT COST

ENGINEERING EPCM (%):	10%		$83,484,000
OWNER COST	3%		$25,045,000
SPARE PARTS	2%		$13,957,000
TRANSPORTATION	8%		$55,826,000
CHEMICALS INITIAL LOAD	1%		$6,978,000
SUBTOTAL INDIRECT COST:			$185,290,000

3.0 PROJECT COST

TOTAL COST ESTIMATE			$1,020,125,100
CONTINGENCY (%):	15%		$153,019,000
TOTAL CAPITAL INVESTMENT :			$1,173,144,100

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET.RECOVERY PROCESS HCl BASED PROCESS	Page:	1
		Rev.:	2
Prepared By: E. Burga	Reviewed By: P. Evans	Date:	Nov 23,2011

STARFIELD RESOURCES: FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE SUMMARY SHEET
6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

1.0 DIRECT COST

AREA	DESCRIPTION	MATERIAL	LABOUR	TOTAL
100	ROM HANDLING AND STORAGE	$5,253,000	$1,050,600	$6,303,600
150	ORE CRUSHING AND CLASSIFICATION	$9,894,000	$1,978,800	$11,872,800
200	MILL - FLOTATION AND THICKENING	$35,166,200	$7,033,200	$42,199,400
300	BACKFILL PLANT	$2,745,500	$549,100	$3,294,600
400	REAGENTS	$586,500	$117,300	$703,800
500	CONC UTILITIES AND SERVICES	$1,849,600	$369,900	$2,219,500
600	STAND BY POWER PLANT	$14,366,700	$2,873,300	$17,240,000
700	MAIN SUBSTATION	$27,500,000	$5,500,000	$33,000,000
	SUBTOTAL DIRECT COST:	$97,362,000	$19,472,000	$116,834,000

2.0 INDIRECT COST

ENGINEERING EPCM (%):	10%	$11,683,000
OWNER COST	3.0%	$3,505,000
SPARE PARTS	2%	$1,947,200
TRANSPORTATION	8%	$7,789,000
CHEMICALS INITIAL LOAD	1%	$974,000
SUBTOTAL INDIRECT COST:		$25,898,200

3.0 PROJECT COST

TOTAL COST ESTIMATE		$142,732,200
CONTINGENCY (%):	15%	$21,410,000
TOTAL CAPITAL INVESTMENT :		$164,142,200

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Prepared By: E. Burga	Reviewed By: P. Evans	Date:	Nov 23,2011

STARFIELD RESOURCES: FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE SUMMARY SHEET
6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

1.0 DIRECT COST

AREA	DESCRIPTION	MATERIAL	LABOUR	TOTAL
910	CONCENTRATE PUMP STATION	$60,189,600	$12,037,900	$72,227,500
920	PIPELINE	$40,745,100	$8,149,000	$48,894,100
010	CON RECEIVING AND FILTRATION	$13,266,800	$2,653,400	$15,920,200
	SUBTOTAL DIRECT COST:	$114,201,500	$22,840,300	$137,041,800

2.0 INDIRECT COST

ENGINEERING (%):	10%		$13,704,180
OWNER COST	3.0%		$4,111,254
SPARE PARTS	2%		$2,284,030
TRANSPORTATION	8%		$9,136,120
CHEMICALS INITIAL LOAD	1%		$1,142,000
SUBTOTAL INDIRECT COST:			$30,377,584

3.0 PROJECT COST

TOTAL COST ESTIMATE			$167,419,384
CONTINGENCY (%):	15%		$25,113,000
TOTAL CAPITAL INVESTMENT :			$192,532,384

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STARFIELD RESOURCES: FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE SUMMARY SHEET
6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

1.0 DIRECT COST

AREA	DESCRIPTION	MATERIAL	LABOUR	TOTAL
011	PRIMARY LEACH	$22,021,000	$4,404,000	$26,425,000
012	SECONDARY LEACH	$18,521,000	$3,704,200	$22,225,200
013	OXIDATION - EVAPORATION	$6,023,000	$1,205,000	$7,228,000
014	HYDROLYSIS	$34,724,000	$6,944,900	$41,668,900
015	Fe SX	$4,224,000	$821,000	$5,045,000
016	Zn IX RECOVERY PACKAGE	$19,479,400	$2,106,300	$21,585,700
017	Cu/Co SOLVENT EXTRACTION	$11,923,000	$2,385,000	$14,308,000
018	Cu EW	$32,626,500	$6,370,300	$38,996,800
019	Co SXEW	$11,114,900	$2,220,000	$13,334,900
020	Ni SXEW	$33,857,000	$6,563,900	$40,420,900
021	Mn EW	$8,183,000	$1,336,500	$9,519,500
022	SULFUR RECOVERY	$1,799,400	$479,800	$2,279,200
023	REAGENTS	$5,590,000	$918,000	$6,508,000
024	ACID REGENERATION AND EVAPORATION	$2,875,000	$575,000	$3,450,000
025	EVAPORATION 2&3	$9,055,500	$1,811,100	$10,866,600
1100	ACID PLANT WITH POWER GENERATION SET	$116,630,000	$23,326,000	$139,956,000
1050	ACID STORAGE	$75,221,500	$15,044,300	$90,265,800
1200	OXYGEN PLANT	$21,450,000	$4,290,000	$25,740,000
1300	SITE SUPPORTING PLANTS	$1,870,000	$374,000	$2,244,000
1400	UTILITIES-	$16,005,000	$3,201,000	$19,206,000
1500	POWER GENERATION PROCESS RECOVERY	$11,000,000	$2,200,000	$13,200,000
1600	BUILDINGS AND SITE SERVICES	$22,071,500	$4,414,300	$26,485,800
	SUBTOTAL DIRECT COST:	$486,264,700	$94,694,600	$580,959,300

2.0 INDIRECT COST

ENGINEERING (%):	10%	$58,095,930
OWNER COST	3.0%	$17,428,779
SPARE PARTS	2%	$9,725,294
TRANSPORTATION	8%	$38,901,176
CHEMICALS INITIAL LOAD	1%	$4,863,000
SUBTOTAL INDIRECT COST:		$129,014,179

3.0 PROJECT COST

TOTAL COST ESTIMATE		$709,973,479
CONTINGENCY (%):	15%	$106,496,000
TOTAL CAPITAL INVESTMENT :		$816,469,479

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Page:	IV
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	PROCESS PLANT		
Prepared By: E. Burga	Reviewed By: P. Evans	Date:	Nov 23,2011

Notes:

1 Ore Receiving and crushing is not in the process plant scope

2 Acid plant package to include a turbine for power generation

7.0 OPEX

7.1 Operating Cost Development

7.2.1 Utilities Consumption

STARFIELD FERGUSSON LAKE PROJECT
SUMPTION SUMMARY

By E. Burga
Date Nov 23-11
Rev 2

AREA	Description	Installed HP	Sub totals	Motor Effic	Utilization	Kw	Spare	CONSIDER	SELECTION
				0.95	0.8	0.74	25%		
TOR									
100	ROM HANDLING AND STORAGE	315							
150	ORE CRUSHING AND CLASSIFICATION	960							
200	MILL - FLOTATION AND THICKENING	12540							
300	BACKFILL PLANT	395							
400	REAGENTS	55							
500	CONC UTILITIES AND SERVICES	1825							
600	STAND BY POWER PLANT	10							
700	MAIN SUBSTATION	0							
	TOTAL CONCENTRATOR		16,100.00	16,950.00	13560	10030	2510	12540	12.5 MW

AREA	Description	Installed HP	Sub totals	Motor Effic	Utilization	Kw	Spare	CONSIDER	SELECTION
				0.95	0.8	0.74	25%		
910	CONCENTRATE PUMP STATION	2000							
920	PIPELINE	0							
10	CON RECEIVING AND FILTRATION	440							
	TOTAL PIPELINE		2,440.00	2,570.00	2184.5	1620	410	2030	2.5 MW

POWER FOR CONCENTRATE SITE 14570 15 MW

NOTE:
Power supply to Concentrator site should include the Mine power supply

ANT

AREA	Description	Installed HP	Sub totals	Motor Effic	Utilization	Kw	Spare	CONSIDER	SELECTION
				0.95	0.8	0.74	25%		
11	PRIMARY LEACH	340							
12	SECONDARY LEACH	385							
13	OXIDATION - EVAPORATION	450							
14	HYDROLYSIS	515							
15	Fe SX	243							
16	Zn IX RECOVERY PACKAGE	300							
17	Cu/Co SOLVENT EXTRACTION	662							
18	Cu EW	6145							
19	Co SXEW	1545							
20	Ni SXEW	4982							
21	Mn EW	1500							
22	SULFUR RECOVERY	204							
23	REAGENTS	315							
24	ACID REGENERATION AND EVAPORATION	300							
25	EVAPORATION 2&3	100							
1100	ACID PLANT WITH POWER GENERATION SET	300							
1050	ACID STORAGE	260							
1200	OXYGEN PLANT	250							
1300	SITE SUPPORTING PLANTS	300							
1400	UTILITIES-	700							
1500	POWER GENERATION PROCESS RECOVERY	50							
1600	BUILDINGS AND SITE SERVICES	150							
	TOPTAL PROCESS PLANT		19,996.00	21,050.00	16840	12460	3120	15580	16 MW

TOTAL REQUIRED 31MW

7.2 Plant Operating Cost OPEX

ACSI									
STARFIELD FERGUSON LAKE PROJECT							Date	11-11-23	
							Rev	02	
UPDATED OPERATING COST							By	P Evans	
ORE PROCESSING PLANT - CONCENTRATE PIPELINE - PROCESS PLANT							Revised by:	E. Burga	
A.- OPERATING DATA									
Concentrator		**6000**	tpd ore	Process Plant	**3311**	tpd concentrate			
Operating days per year		**350**	Days						
B.0 OPEX DEFINITON								**Daily cost**	**Yearly cost**
1.0 Chemicals and raw materials									
			tpa		**tpd**	**$/t**			
Flotation chemicals concentrator			40.00		0.1	$7,610		$870	$304,400
Ball mill balls			1,890.00		5.4	$1,500		$8,100	$2,835,000
Hydrochloric acid 35%			7,457.73		21.3	$120		$2,557	$894,928
Sulfuric acid 93%			276,833.59		791.0	$1		$791	$276,834
Oxygen			96,198.61		274.9	$1		$275	$96,199
Resin and solvent replacement			20.00		0.1	$10,000		$571	$200,000
ammonium carbonate			17,331.51		49.5	$200		$9,904	$3,466,302
Quick lime			19,216.89		54.9	$100		$5,491	$1,921,689
Floculants hydromet			40.00		0.1	$7,610		$870	$304,400
					Sub total1.0 Chemicals and raw materials			$20,458	$7,160,352
2.0 Consumables									
Spare parts % of equipment costs		2		$ 697,828,000.00				$39,876	
Packaging product						$20		$20	
Fuel mobile equipment						$100		$100	
Office supplies						$6		$6	
Plant supplies						$300		$300	
				Sub total 2.0 Consumables				$40,302	$14,105,660
3.0 Labour									
	# of employees	Shifts	Total empoyees	Social charges %	%	Yearly salaty		$/day	
3.1 Operations									
3.3.1 Management									
Management	1	1	1	135	100	$200,000		$740	$270,000
Superintendents	6	1	6	135	100	$170,000		$3,773	$1,377,000
Engineers	2	3	6	135	100	$120,000		$2,663	$972,000
Technicians	2	3	6	135	100	$75,000		$1,664	$607,500
Clerical	3	1	3	135	100	$56,000		$621	$226,800
3.1.2 Labour									
Crush grind float pump	3	3	9	135	100	$75,000		$2,497	$911,250
Filtration	1.5	3	4.5	135	100	$75,000		$1,248	$455,625
Leach 1&2	1.5	3	4.5	135	100	$75,000		$1,248	$455,625
Oxidation hydrolysis HCl regen	2	3	6	135	100	$75,000		$1,664	$607,500
IX SX	3	3	9	135	100	$75,000		$2,497	$911,250
EW	5	3	15	135	100	$75,000		$4,161	$1,518,750
Acid plant	2	3	6	135	100	$75,000		$1,664	$607,500
Oxygen plant	1	3	3	135	100	$75,000		$832	$303,750
Materials handling	1	3	3	135	100	$75,000		$832	$303,750
Additional process labour	12	3	36	135	100	$50,000		$6,658	$2,430,000
Laboratory	8	1	8	135	100	$65,000		$1,923	$702,000
Sub total operations labour			126			Sub total operations Cost		$34,686	$12,660,300
3.2 Pipeline									
Supervisor	1	**2**	2	135	100	$75,000		$555	$202,500
Labour	2	**3**	6	135	100	$50,000		$1,110	$405,000
Sub total Pipeline Labour			8			Sub total Pipeline Cost		$1,664	$607,500
3.3 Maintenance									
Supertendents	3	1	3	135	100	$155,000		$1,720	$627,750
Supervisors	1	3	3	135	100	$100,000		$1,110	$405,000
Mechanical	4	3	12	135	100	$90,000		$3,995	$1,458,000
Instrument/electrical	3	3	9	135	100	$100,000		$3,329	$1,215,000
Non-skilled	5	3	15	135	100	$50,000		$2,774	$1,012,500
Sub total maintenance Labour			42			Sub total maintenance cost		$12,927	$4,718,250
Total number operators			**168**						
				Sub total 3.0 Labour cost				**$49,277**	**$17,986,050**
4.0 Utilities									
Power 7MW power plant			Kwh	-	$/Kwh	$0		$0	$0
Fuel for 7MW power plant(10% of time)			l/h diesel	1,279.02	$/h	$1,279		$3,070	$1,074,377
Boiler feed water			m3/day	535.23	$/m3	$2		$1,070	$374,660
Cooling water			m3/day	326,448.00	$/m3	$0		$1,632	$571,284
				Sub total 4.0 Utilities				**$5,772**	**$2,020,321**
C.- SUMMARY									
				Total daily operating cost				**$115,809**	
				Total annual operating cost					**$40,533,230**

END OF SECTION !

CAPEX AND OPEX UPDATE FOR STARFIELD RESOURCES

Ferguson Lake Project

PROCESS DEVELOPMENT

AND

CAPEX /OPEX FOR PROCESS FACILITIES

FOR

STARFIELD METALLURGICAL RECOVERY PLANT

REPORT: A11-115-SR-SS101 REV0

Prepared by: P. Evans

Revised By: E. Burga

Andeburg Consulting Services Inc (ACSI)

November 23 2011

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APPENDIX 1 MASS BALANCE

The mass/energy balance for the process was calculated by Arithmetek Inc and is presented in a separate report.

ACSI used the mass/nergy balance to identify the equipment and evaluate the equipment requirements.

APPENDIX 2: EQUIPMENT LIST

The equipment list is included in the CAPEX files.

APPENDIX 3: Capital Cost Estimate

STARFIELD RESOURCES

FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE

6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

STARFIELD RESOURCES: FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE SUMMARY SHEET
6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

1.0 DIRECT COST

AREA	DESCRIPTION	MATERIAL	LABOUR	TOTAL
P1	CONCENTRATOR	$97,362,000	$19,472,000	$116,834,000
P2	PIPELINE	$114,201,500	$22,840,300	$137,041,800
P3	PROCESS PLANT	$486,264,700	$94,694,600	$580,959,300
	SUBTOTAL DIRECT COST:	$697,828,200	$137,006,900	$834,835,100

2.0 INDIRECT COST

ENGINEERING EPCM (%):	10%		$83,484,000
OWNER COST	3%		$25,045,000
SPARE PARTS	2%		$13,957,000
TRANSPORTATION	8%		$55,826,000
CHEMICALS INITIAL LOAD	1%		$6,978,000
SUBTOTAL INDIRECT COST:			$185,290,000

3.0 PROJECT COST

TOTAL COST ESTIMATE			$1,020,125,100
CONTINGENCY (%):	15%		$153,019,000
TOTAL CAPITAL INVESTMENT :			$1,173,144,100

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STARFIELD RESOURCES: FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE SUMMARY SHEET
6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

1.0 DIRECT COST

AREA	DESCRIPTION	MATERIAL	LABOUR	TOTAL
100	ROM HANDLING AND STORAGE	$5,253,000	$1,050,600	$6,303,600
150	ORE CRUSHING AND CLASSIFICATION	$9,894,000	$1,978,800	$11,872,800
200	MILL - FLOTATION AND THICKENING	$35,166,200	$7,033,200	$42,199,400
300	BACKFILL PLANT	$2,745,500	$549,100	$3,294,600
400	REAGENTS	$586,500	$117,300	$703,800
500	CONC UTILITIES AND SERVICES	$1,849,600	$369,900	$2,219,500
600	STAND BY POWER PLANT	$14,366,700	$2,873,300	$17,240,000
700	MAIN SUBSTATION	$27,500,000	$5,500,000	$33,000,000
	SUBTOTAL DIRECT COST:	$97,362,000	$19,472,000	$116,834,000

2.0 INDIRECT COST

ENGINEERING EPCM (%):	10%	$11,683,000
OWNER COST	3.0%	$3,505,000
SPARE PARTS	2%	$1,947,200
TRANSPORTATION	8%	$7,789,000
CHEMICALS INITIAL LOAD	1%	$974,000
SUBTOTAL INDIRECT COST:		$25,898,200

3.0 PROJECT COST

TOTAL COST ESTIMATE		$142,732,200
CONTINGENCY (%):	15%	$21,410,000
TOTAL CAPITAL INVESTMENT :		$164,142,200

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AREA ROM HANDLING AND STORAGE

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$3,090,000	$618,000	$3,708,000
002	PIPING	20%	$618,000	$123,600	$741,600
003	CIVIL/STRUCTURAL	30%	$927,000	$185,400	$1,112,400
003	ELECTRICAL/INSTRUMENTATION	20%	$618,000	$123,600	$741,600
		Labour Alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$5,253,000	105060000%	$6,303,600

INDIRECT COST

	3%		
ENGINEERING (%):	2%		$126,072
OWNER COST	3%		$189,108
SPARE PARTS	2%		$105,060
TRANSPORTATION	8%		$420,240
CHEMICALS INITIAL LOAD	1%		$53,000
INDIRECT COST SUB-TOTAL			$893,480

AREA COST

TOTAL COST ESTIMATE			$7,197,080
	15%		$1,080,000
TOTAL CAPITAL INVESTMENT :			$8,277,080

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DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item Nº.	Description			MATERIAL Qty.	U.O.M.	Unit Cost	(%)	(%)	(%)	*Total*			LABOUR Factor (%)	Total	Total
100-HO-01	Dump Hopper			1		$200,000				$200,000			20%	$40,000	$240,000
100-RB-01	Rock Breaker		20	1		$75,000				$75,000			20%	$15,000	$90,000
100-CP-01	Pan feeder	1000 tph 42"x 7	30	1		$210,000				$210,000			20%	$42,000	$252,000
100-CR-01	Primary Crusher	1000 tph 34x44	100	1		$1,500,000				$1,500,000			20%	$300,000	$1,800,000
100-CP-02	Transfer pan conveyor	1000 tph 42"x 7	30	1		$210,000				$210,000			20%	$42,000	$252,000
100-BC-01	Belt conveyor	1200 tph 42"-60 m	40	1		$240,000				$240,000			20%	$48,000	$288,000
100-BC-02	Belt conveyor	1200 tph 42"-20 m	15	1		$80,000				$80,000			20%	$16,000	$96,000
100-BC-03	Belt conveyor	1200 tph 42"-40 m	20	1		$160,000				$160,000			20%	$32,000	$192,000
100-RF-01A/	Reclaim feeder	525 tph 42"-10m	10	2		$100,000				$200,000			20%	$40,000	$240,000
100-BC-04	Belt conveyor	525 tph 30"-40 m	20	1		$140,000				$140,000			20%	$28,000	$168,000
100-DC-01A/	Dust collector		20	2		$25,000				$50,000			20%	$10,000	$60,000
100-PU-01	Sump pump		10	1		$25,000				$25,000			20%	$5,000	$30,000
		TOTAL:								$3,090,000	0			$618,000	0

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AREA ORE CRUSHING AND CLASSIFICATION

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL			$5,820,000	$1,164,000	$6,984,000
002	PIPING	20%		$1,164,000	$232,800	$1,396,800
003	CIVIL/STRUCTURAL	30%		$1,746,000	$349,200	$2,095,200
003	ELECTRICAL/INSTRUMENTATION	20%		$1,164,000	$232,800	$1,396,800
		Labour Alloc factor	20%			
	DIRECT COST SUB-TOTAL:			$9,894,000	$1,978,800	$11,872,800

INDIRECT COST

ENGINEERING (%):	10%	$1,187,280
OWNER COST	3%	$356,184
SPARE PARTS	2%	$197,880
TRANSPORTATION	8%	$791,520
CHEMICALS INITIAL LOAD	1%	$99,000
INDIRECT COST SUB-TOTAL		$2,631,864
TOTAL COST ESTIMATE		$14,504,664
	15%	$2,176,000
TOTAL CAPITAL INVESTMENT :		$16,680,664

ANDEBURG CONSULTING SERVICES INC.

TITLE: STARFIELD RESOURCES
FERGUSON LAKE MET. RECOVERY PROCESS
DEPT.: ORE CRUSHING AND CLASSIFICATION

Prepared By: E. Burga | Reviewed By: P. Evans | Appv'd. (Eng.):

Area 150
Page: 1 of 2
Rev.: 1 of 2
Date: Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item N°.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	(%	(%	(%	*Total*	LABOUR		Factor (%)	Total	Total
150-CR-02	Secondary crusher	HP 400		300	1		$700,000				$700,000			20%	$140,000	$840,000
150-CR-03	Tertiary Crusher	MP 1000		200	1		$2,200,000				$2,200,000			20%	$440,000	$2,640,000
150-SCR-01	Secondary screen	8x24		25	1		$180,000				$180,000			20%	$36,000	$216,000
150-SCR-02	Tertiary screen	8x24		25	1		$180,000				$180,000			20%	$36,000	$216,000
	Tertiary crusher feed silo	150 m3			1		$30,000				$30,000			20%	$6,000	$36,000
	Fine Ore Silo	500	m3		2		$250,000				$500,000			20%	$100,000	$600,000
														20%		
	36" Conveying system	525	tph	100	1		$300,000				$300,000			20%	$60,000	$360,000
	30"Conveying system	400	tph	150	1		$1,200,000				$1,200,000			20%	$240,000	$1,440,000
	24" Conveying system	250	tph	100	1		$350,000				$350,000			20%	$70,000	$420,000
														20%		
	belt weightf eeder	250	tph	15	1		$60,000				$60,000			20%	$12,000	$72,000
														20%		
	Vib feeder	250	tpf	15	1		$45,000				$45,000			20%	$9,000	$54,000
150-DC-01A/B	Dust collector			20	2		$25,000				$50,000			20%	$10,000	$60,000
150-PU-01	Sump pump			10	1		$25,000				$25,000			20%	$5,000	$30,000
			TOTAL:								$5,820,000	0			$1,164,000	$6,984,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	150
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: ORE CRUSHING AND CLASSIFICATION	Rev.:	2 of 2
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA MILL - FLOTATION AND THICKENING

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL			$20,686,000	$4,137,200	$24,823,200
002	PIPING	20%		$4,137,200	$827,440	$4,964,640
003	CIVIL/STRUCTURAL	30%		$6,205,800	$1,241,160	$7,446,960
003	ELECTRICAL/INSTRUMENTATION	20%		$4,137,200	$827,440	$4,964,640
		Labour Alloc factor	20%			
	DIRECT COST SUB-TOTAL:			$35,166,200	$7,033,200	$42,199,400

INDIRECT COST

ENGINEERING (%):	10%		$4,219,940
OWNER COST	3%		$1,265,982
SPARE PARTS	2%		$703,324
TRANSPORTATION	8%		$2,813,296
CHEMICALS INITIAL LOAD	1%		$352,000
INDIRECT COST SUB-TOTAL			$9,354,542
		TOTAL COST ESTIMATE	$51,553,942
	15%		$7,733,000
		TOTAL CAPITAL INVESTMENT :	$59,286,942

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET. RECOVERY PROCESS DEPT.: MILL - FLOTATION AND THICKENING	Area 200 Page: 1 of 2 Rev.: 001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date: Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item N°.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	(%)	(%)	(%)	*Total*			LABOUR Factor (%)	Total	Total
														20%		
200-BM-01	Ball Mill	19x36	6.25	8446	1		$17,000,000				$17,000,000			20%	$3,400,000	$20,400,000
200-HC-01	Hydrocyclone pack	4x30			1		$320,000				$320,000			20%	$64,000	$384,000
200-Pu-01A	Cyclone Feed Pump	14x12	40 Hp	80	2		$150,000				$300,000			20%	$60,000	$360,000
200-TH-01	Concentrate Thickener system	6.50		10	1		$400,000				$400,000			20%	$80,000	$480,000
200-TH-02	Tailing thickener system	010		10	1		$600,000				$600,000			20%	$120,000	$720,000
200-TK-02	Conditioner Tank	100	m3		1		$107,000				$107,000			20%	$21,400	$128,400
200-AG-01	Conditioner Tank Agitator			30	1		$25,000				$25,000			20%	$5,000	$30,000
200-FC-01	Rougher Flot Cell			30	1		$600,000				$600,000			20%	$120,000	$720,000
200-TK-03	Tailing thickener U/F tank	100	m3		1		$88,000				$88,000			20%	$17,600	$105,600
200-AG-03	Tailing thickener U/F tank Agitator			20	1		$50,000				$50,000			20%	$10,000	$60,000
200-PU-03	Thickener U/F Pump			10	1		$40,000				$40,000			20%	$8,000	$48,000
200-PU-13	Tailings pumps	2800	m3/h	3000	2		$250,000				$500,000			20%	$100,000	$600,000
200-TK-04	Thickener O/F tank	1200	m3		1		$416,000				$416,000			20%	$83,200	$499,200
200-PU-04	Thickener O/F Pump	6000	m3/h	900	2		$120,000				$240,000			20%	$48,000	$288,000
														20%		
				TOTAL:							$20,686,000	0			$4,137,200	0

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	200
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: MILL - FLOTATION AND THICKENING	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA BACKFILL PLANT

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL			$1,615,000	$323,000	$1,938,000
002	PIPING	20%		$323,000	$64,600	$387,600
003	CIVIL/STRUCTURAL	30%		$484,500	$96,900	$581,400
003	ELECTRICAL/INSTRUMENTATION	20%		$323,000	$64,600	$387,600
		Labour Alloc factor	20%			
	DIRECT COST SUB-TOTAL:			$2,745,500	$549,100	$3,294,600

INDIRECT COST

ENGINEERING (%):	10%	$329,460
OWNER COST	3%	$98,838
SPARE PARTS	2%	$54,910
TRANSPORTATION	8%	$219,640
CHEMICALS INITIAL LOAD	1%	$27,000
INDIRECT COST SUB-TOTAL		$729,848
TOTAL COST ESTIMATE		$4,024,448
	15%	$604,000
TOTAL CAPITAL INVESTMENT :		$4,628,448

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET. RECOVERY PROCESS	Area 300 Page: 1 of 2
	DEPT.: BACKFILL PLANT	Rev.: 001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date: Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item N°.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	%	%	%	*Total*			LABOUR Factor (%)	Total	Total
300-PKG-01	Backfill Plant Allowance											20%		
												20%		
	Mix Tank		1		$125,000				$125,000			20%	$25,000	$150,000
	Mix tank Agitator	75	1		$90,000				$90,000			20%	$18,000	$108,000
	Pumps	300	2		$500,000				$1,000,000			20%	$200,000	$1,200,000
	Additive feed system	20	1		$200,000				$200,000			20%	$40,000	$240,000
	Storage tanks		1		$200,000				$200,000			20%	$40,000	$240,000
	TOTAL:								$1,615,000	0			$323,000	0

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	300
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: BACKFILL PLANT	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA REAGENTS

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL			$345,000	$69,000	$414,000
002	PIPING	20%		$69,000	$13,800	$82,800
003	CIVIL/STRUCTURAL	30%		$103,500	$20,700	$124,200
003	ELECTRICAL/INSTRUMENTATION	20%		$69,000	$13,800	$82,800
		Labour Alloc factor	20%			
	DIRECT COST SUB-TOTAL:			$586,500	$117,300	$703,800

INDIRECT COST

ENGINEERING (%):	10%	$70,380
OWNER COST	3%	$21,114
SPARE PARTS	2%	$11,730
TRANSPORTATION	8%	$46,920
CHEMICALS INITIAL LOAD	1%	$6,000
INDIRECT COST SUB-TOTAL		$156,144
TOTAL COST ESTIMATE		$859,944
	15%	$129,000
TOTAL CAPITAL INVESTMENT :		$988,944

ANDEBURG CONSULTING SERVICES INC.

TITLE: STARFIELD RESOURCES
FERGUSON LAKE MET. RECOVERY PROCESS
DEPT.: REAGENTS

Area 400
Page: 1 of 2
Rev.: 001
Date: Nov 19,2011

Prepared By: E. Burga | Reviewed By: P. Evans | Appv'd. (Eng.):

DETAILED PROJECT COST ESTIMATE (MECHANICAL)																	
120					MATERIAL								LABOUR				
Item							Unit							Factor			
N°.	Description				Qty.	U.O.M.	Cost	%	%	%	*Total*			(%)	Total	Total	
400-BI-01	Lime Silo	080	tons		1		$90,000				$90,000			20%	$18,000	$108,000	
400-PC-01	Pneumatic conveyiong system			25	1		$75,000				$75,000			20%	$15,000	$90,000	
400-SKR-01	LimeSlaker			20	1		$150,000				$150,000			20%	$30,000	$180,000	
	Lime Slurry Storage tank	020	m3		1		$15,000				$15,000			20%	$3,000	$18,000	
	Agitator			10	1		$15,000				$15,000			20%	$3,000	$18,000	
				TOTAL:							$345,000	0			$69,000	$414,000	
				ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD RESOURCES									Area	400	
						FERGUSON LAKE MET. RECOVERY PROCESS									Page:	2 of 2	
					DEPT.:	REAGENTS									Rev.:	001	
		Prepared By:		E. Burga					Appv'd. (Eng):						Date:	Nov 19,2011	

AREA CONC UTILITIES AND SERVICES

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$1,088,000	$217,600	$1,305,600
002	PIPING	20%	$217,600	$43,520	$261,120
003	CIVIL/STRUCTURAL	30%	$326,400	$65,280	$391,680
003	ELECTRICAL/INSTRUMENTATION	20%	$217,600	$43,520	$261,120
		20%			
	DIRECT COST SUB-TOTAL:		$1,849,600	$369,900	$2,219,500

INDIRECT COST

ENGINEERING (%):	10%	$221,950
OWNER COST	3%	$66,585
SPARE PARTS	2%	$36,992
TRANSPORTATION	8%	$147,968
CHEMICALS INITIAL LOAD	1%	$18,000
INDIRECT COST SUB-TOTAL		$491,495
TOTAL COST ESTIMATE		$2,710,995
	15%	$407,000
TOTAL CAPITAL INVESTMENT :		$3,117,995

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET. RECOVERY PROCESS	Area	500
		Page:	1 of 2
	DEPT.: CONC UTILITIES AND SERVICES	Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)													
120		MATERIAL							LABOUR				
Item N°.	Description	Qty.	U.O.M.	Unit Cost	(%)	(%)	(%)	*Total*			Factor (%)	Total	Total
500-PV-01/02	Air Receivers	2	ea	$40,000				$80,000			20%	$16,000	$96,000
500-AC-01/02	Air compressors 002 75 hp 150	2	ea	$125,000				$250,000			20%	$50,000	$300,000
500-TK-01	Process Water Tank 1000 m3	1		$268,000				$268,000			20%	$53,600	$321,600
500-PU-01/02	Process Water distribution Pump 7500 m3/h 1600	2		$120,000				$240,000			20%	$48,000	$288,000
500-PKG-01	Fire Protecction System 75	1		$250,000				$250,000			20%	$50,000	$300,000
	TOTAL:							$1,088,000	0			$217,600	0

ANDEBURG CONSULTING SERVICES INC.
TITLE: STARFIELD RESOURCES
FERGUSON LAKE MET. RECOVERY PROCESS
DEPT.: CONC UTILITIES AND SERVICES
Area: 500
Page: 2 of 2
Rev.: 001
Date: Nov 19,2011
Prepared By: E. Burga
Appv'd. (Eng):

AREA STAND BY POWER PLANT

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$10,642,000	$2,128,400	$12,770,400
002	PIPING	10%	$1,064,200	$212,840	$1,277,040
003	CIVIL/STRUCTURAL	10%	$1,064,200	$212,840	$1,277,040
003	ELECTRICAL/INSTRUMENTATION	15%	$1,596,300	$319,260	$1,915,560
		20%			
	DIRECT COST SUB-TOTAL:		$14,366,700	287330000%	$17,240,000

INDIRECT COST

ENGINEERING (%):	10%		$1,724,000
OWNER COST	3%		$517,200
SPARE PARTS	2%		$287,334
TRANSPORTATION	8%		$1,149,336
CHEMICALS INITIAL LOAD	1%		$144,000
INDIRECT COST SUB-TOTAL			$3,821,870
TOTAL COST ESTIMATE			$21,061,870
	15%		$3,159,000
TOTAL CAPITAL INVESTMENT :			$24,220,870

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET. RECOVERY PROCESS	Area	600
		Page:	1 of 2
	DEPT.: STAND BY POWER PLANT	Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item N°.	Description			Qty.	U.O.M.	MATERIAL Unit Cost	%	%	%	*Total*			LABOUR Factor (%)	Total	Total
													20%		
600-PW-01	Stand By Power Plant												20%		
	5 MW Diesel			1		$10,000,000				$10,000,000			20%	$2,000,000	$12,000,000
													20%		
	Diesel Storage tanks	1000	m3	2		$306,000				$612,000			20%	$122,400	$734,400
	Feed Pumps		10	2		$15,000				$30,000			20%	$6,000	$36,000
	TOTAL:									$10,642,000	0			$2,128,400	0

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	600
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: STAND BY POWER PLANT	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA MAIN SUBSTATION

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$22,000,000	$4,400,000	$26,400,000
002	PIPING	0%	$0	$0	$0
003	CIVIL/STRUCTURAL	15%	$3,300,000	$660,000	$3,960,000
003	ELECTRICAL/INSTRUMENTATION	10%	$2,200,000	$440,000	$2,640,000
		20%			
	DIRECT COST SUB-TOTAL:		$27,500,000	$5,500,000	$33,000,000

INDIRECT COST

ENGINEERING (%):	10%		$3,300,000
OWNER COST	3%		$990,000
SPARE PARTS	2%		$550,000
TRANSPORTATION	8%		$2,200,000
CHEMICALS INITIAL LOAD	1%		$275,000
INDIRECT COST SUB-TOTAL			$7,315,000
		TOTAL COST ESTIMATE	$40,315,000
	15%		$6,047,000
		TOTAL CAPITAL INVESTMENT :	$46,362,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET. RECOVERY PROCESS	Area	700
	DEPT.: MAIN SUBSTATION	Page:	1 of 2
		Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item N°.	Description	Qty.	U.O.M.	MATERIAL Unit Cost	(%)	(%)	(%)	*Total*			LABOUR Factor (%)	Total	Total
700-PKG1	20 MW Sub station	1		$5,000,000				$5,000,000			20%	$1,000,000	$6,000,000
	HV Power Supply 270 KM	1		$17,000,000				$17,000,000			20%	$3,400,000	$20,400,000
	TOTAL:							$22,000,000	0			$4,400,000.00	$26,400,000.00

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	700
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: MAIN SUBSTATION	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

STARFIELD RESOURCES: FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE SUMMARY SHEET
6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

1.0 DIRECT COST

AREA	DESCRIPTION	MATERIAL	LABOUR	TOTAL
910	CONCENTRATE PUMP STATION	$60,189,600	$12,037,900	$72,227,500
920	PIPELINE	$40,745,100	$8,149,000	$48,894,100
010	CON RECEIVING AND FILTRATION	$13,266,800	$2,653,400	$15,920,200
	SUBTOTAL DIRECT COST:	$114,201,500	$22,840,300	$137,041,800

2.0 INDIRECT COST

ENGINEERING (%):	10%		$13,704,180
OWNER COST	3.0%		$4,111,254
SPARE PARTS	2%		$2,284,030
TRANSPORTATION	8%		$9,136,120
CHEMICALS INITIAL LOAD	1%		$1,142,000
SUBTOTAL INDIRECT COST:			$30,377,584

3.0 PROJECT COST

TOTAL COST ESTIMATE			$167,419,384
CONTINGENCY (%):	15%		$25,113,000
TOTAL CAPITAL INVESTMENT :			$192,532,384

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES		
	FERGUSON LAKE MET.RECOVERY PROCESS	Page:	III
	PIPELINE SUMMARY	Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans	Date:	Nov 19,2011

AREA CONCENTRATE PUMP STATION

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL			$56,252,000	$11,250,400	$67,502,400
002	PIPING	3%		$1,687,560	$337,512	$2,025,072
003	CIVIL/STRUCTURAL	2%		$1,125,040	$225,008	$1,350,048
003	ELECTRICAL/INSTRUMENTATION	2%		$1,125,040	$225,008	$1,350,048
			20%			
	DIRECT COST SUB-TOTAL:			$60,189,600	$12,037,900	$72,227,600

INDIRECT COST

ENGINEERING (%):	10%	$7,222,760
OWNER COST	3%	$2,166,828
SPARE PARTS	2%	$1,203,792
TRANSPORTATION	8%	$4,815,168
CHEMICALS INITIAL LOAD	1%	$602,000
INDIRECT COST SUB-TOTAL		$16,010,548
TOTAL COST ESTIMATE		$88,238,148
	15%	$13,236,000
TOTAL CAPITAL INVESTMENT :		$101,474,148

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET. RECOVERY PROCESS DEPT.: CONCENTRATE PUMP STATION	Area 910 Page: 1 of 2 Rev.: 001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date: Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item N°.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	(%)	(%)	(%)	*Total*			LABOUR Factor (%)	Total	Total
												20%		
910-PKG-01	Pump Station Package	2000	1	lot	$56,252,000				$56,252,000			20%	$11,250,400	$67,502,400
	TOTAL:								$56,252,000	0			$11,250,400	0

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	910
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: CONCENTRATE PUMP STATION	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA PIPELINE

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	Pipeline			$37,041,000	$7,408,200	$44,449,200
003	CIVIL/STRUCTURAL	5%		$1,852,050	$370,410	$2,222,460
003	ELECTRICAL/INSTRUMENTATION	5%	20%	$1,852,050	$370,410	$2,222,460
		DIRECT COST SUB-TOTAL:		$40,745,100	$8,149,000.00	$48,894,100

INDIRECT COST

ENGINEERING (%):	10%	$4,889,410
OWNER COST	3%	$1,466,823
SPARE PARTS	2%	$814,902
TRANSPORTATION	8%	$3,259,608
CHEMICALS INITIAL LOAD	1%	$407,000
INDIRECT COST SUB-TOTAL		$10,837,743
TOTAL COST ESTIMATE		$59,731,843
	15%	$8,960,000
TOTAL CAPITAL INVESTMENT :		$68,691,843

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET. RECOVERY PROCESS	Area 920 Page: 1 of 2
	DEPT.: PIPELINE	Rev.: 001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date: Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item N°.	Description	MATERIAL Qty.	U.O.M.	Unit Cost	%	%	%	*Total*			LABOUR Factor (%)	Total	Total
920-PKG-01	8" Pipeline	1	lot	$37,041,000				$37,041,000			20% 20%	$7,408,200	$44,449,200
	TOTAL:							$37,041,000	0			$7,408,200	0

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	920
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: PIPELINE	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA CON RECEIVING AND FILTRATION

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$7,804,000	$1,560,800	$9,364,800
002	PIPING	20%	$1,560,800	$312,160	$1,872,960
003	CIVIL/STRUCTURAL	30%	$2,341,200	$468,240	$2,809,440
003	ELECTRICAL/INSTRUMENTATION	20%	$1,560,800	$312,160	$1,872,960
		20%			
	DIRECT COST SUB-TOTAL:		$13,266,800	$2,653,400.00	$15,920,200

INDIRECT COST

ENGINEERING (%):	10%	$1,592,020
OWNER COST	3%	$477,606
SPARE PARTS	2%	$265,336
TRANSPORTATION	8%	$1,061,344
CHEMICALS INITIAL LOAD	1%	$133,000
INDIRECT COST SUB-TOTAL		$3,529,306
TOTAL COST ESTIMATE		$19,449,506
	15%	$2,917,000
TOTAL CAPITAL INVESTMENT :		$22,366,506

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET. RECOVERY PROCESS DEPT.: CON RECEIVING AND FILTRATION	Area 10 Page: 1 of 2 Rev.: 001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date: Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

120 Item N°.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	%	%	%	*Total*	LABOUR		Factor (%)	Total	Total
10tK-AG-01	Slurry tank Agitators			90	3		$120,000				$360,000			20%	$72,000	$432,000
10-TK-101/	Slurry storage tanks	890	m3		3		$537,000				$1,611,000			20%	$322,200	$1,933,200
10-FP-101	Filters			150	3		$1,709,000				$5,127,000			20%	$1,025,400	$6,152,400
	Cake transfer conveyor	130 ft-L;150 TPH		15	1		$156,000				$156,000			20%	$31,200	$187,200
10-AC-111a	Air compressors			150	2		$120,000				$240,000			20%	$48,000	$288,000
														20%		
														20%		
														20%		
														20%		
10-TK-113	Cloth wash water tank	075	m3		1		$68,000				$68,000			20%	$13,600	$81,600
10-PU-113	Cloth wash water pump				1		$15,000				$15,000			20%	$3,000	$18,000
10-TK-114	Flush water tank	027	m3		1		$33,000				$33,000			20%	$6,600	$39,600
10-PU-113	Flush water pump			20	1	EA	$10,000				$10,000			20%	$2,000	$12,000
														20%		
10-TK-135	Filtrate Tank	180	m3		1	EA	$114,000				$114,000			20%	$22,800	$136,800
10-PU-135	Filtrate Transfer pumps			15	2	ea	$35,000				$70,000			20%	$14,000	$84,000
					1											
	TOTAL:										$7,804,000	0			$1,560,800	0

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	10
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: CON RECEIVING AND FILTRATION	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

STARFIELD RESOURCES: FERGUSON LAKE PROJECT

PROJECT COST ESTIMATE SUMMARY SHEET
6,000TPD METALLURGICAL METAL SULPHIDES RECOVERY PLANT

1.0 DIRECT COST

AREA	DESCRIPTION	MATERIAL	LABOUR	TOTAL
011	PRIMARY LEACH	$22,021,000	$4,404,000	$26,425,000
012	SECONDARY LEACH	$18,521,000	$3,704,200	$22,225,200
013	OXIDATION - EVAPORATION	$6,023,000	$1,205,000	$7,228,000
014	HYDROLYSIS	$34,724,000	$6,944,900	$41,668,900
015	Fe SX	$4,224,000	$821,000	$5,045,000
016	Zn IX RECOVERY PACKAGE	$19,479,400	$2,106,300	$21,585,700
017	Cu/Co SOLVENT EXTRACTION	$11,923,000	$2,385,000	$14,308,000
018	Cu EW	$32,626,500	$6,370,300	$38,996,800
019	Co SXEW	$11,114,900	$2,220,000	$13,334,900
020	Ni SXEW	$33,857,000	$6,563,900	$40,420,900
021	Mn EW	$8,183,000	$1,336,500	$9,519,500
022	SULFUR RECOVERY	$1,799,400	$479,800	$2,279,200
023	REAGENTS	$5,590,000	$918,000	$6,508,000
024	ACID REGENERATION AND EVAPORATION	$2,875,000	$575,000	$3,450,000
025	EVAPORATION 2&3	$9,055,500	$1,811,100	$10,866,600
1100	ACID PLANT WITH POWER GENERATION SET	$116,630,000	$23,326,000	$139,956,000
1050	ACID STORAGE	$75,221,500	$15,044,300	$90,265,800
1200	OXYGEN PLANT	$21,450,000	$4,290,000	$25,740,000
1300	SITE SUPPORTING PLANTS	$1,870,000	$374,000	$2,244,000
1400	UTILITIES-	$16,005,000	$3,201,000	$19,206,000
1500	POWER GENERATION PROCESS RECOVERY	$11,000,000	$2,200,000	$13,200,000
1600	BUILDINGS AND SITE SERVICES	$22,071,500	$4,414,300	$26,485,800
	SUBTOTAL DIRECT COST:	$486,264,700	$94,694,600	$580,959,300

2.0 INDIRECT COST

ENGINEERING (%):	10%	$58,095,930
OWNER COST	3.0%	$17,428,779
SPARE PARTS	2%	$9,725,294
TRANSPORTATION	8%	$38,901,176
CHEMICALS INITIAL LOAD	1%	$4,863,000
SUBTOTAL INDIRECT COST:		$129,014,179

3.0 PROJECT COST

TOTAL COST ESTIMATE		$709,973,479
CONTINGENCY (%):	15%	$106,496,000
TOTAL CAPITAL INVESTMENT :		$816,469,479

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES		
	FERGUSON LAKE MET.RECOVERY PROCESS	Page:	IV
	PROCESS PLANT	Rev.:	002
Prepared By: E. Burga	Reviewed By: P. Evans	Date:	Nov 23,2011

Notes:
1 Ore Receiving and crushing is not in the process plant scope
2 Acid plant package to include a turbine for power generation

AREA PRIMARY LEACH

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL 1		$9,424,000	$1,884,800	$11,308,800
002	PROCESS AND MECHANICAL 2		$3,922,000	$784,400	$4,706,400
003	PIPING	20%	$2,669,200	$533,840	$3,203,040
004	CIVIL/STRUCTURAL	20%	$2,669,200	$533,840	$3,203,040
005	ELECTRICAL/INSTRUMENTATION	25%	$3,336,500	$667,300	$4,003,800
	Labour alloc factor	20%			
	DIRECT COST SUB-TOTAL:		$22,020,900	$4,404,200	$26,425,100

INDIRECT COST

ENGINEERING (%):	10%	$2,642,510
OWNER COST	3%	$792,753
SPARE PARTS	3%	$660,627
TRANSPORTATION	2%	$440,418
CHEMICALS INITIAL LOAD	1%	$220,000
INDIRECT COST SUB-TOTAL		$4,756,308
TOTAL COST ESTIMATE		$31,181,408
CONTINGENCY (%):	15%	$4,677,000
TOTAL CAPITAL INVESTMENT :		$35,858,408

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE METALLURGICAL RECOVERY PROCESS DEPT.: PRIMARY LEACH	Area	11
		Page:	1 of 3
		Rev.:	
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL PAGE 1)

Item N°.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
11-WC-31	FEED SOLID LIQUID CONTACTOR			20	1	EA	$250,000	$250,000	20%	$50,000	$300,000
11-PU-15 A/B	RECYCLE BRINE TRANSFER PUMP			30	2	EA	$90,000	$180,000	20%	$36,000	$216,000
11-TK-01	FEED SLURRYING TANK	044	M3		1	EA	$90,000	$90,000	20%	$18,000	$108,000
11-AG-11	FEED SLURRING TANK AGITATOR				2	EA	$150,000	$300,000	20%	$60,000	$360,000
11-PU-01 A/B	PRIMARY LEACH FEED PUMP	235	m3/h	60	2	EA	$125,000	$250,000	20%	$50,000	$300,000
11-PU-11	Scrubber Discharge Pump	235	m3/h	60	2	EA	$125,000	$250,000	20%	$50,000	$300,000
11-RE-01	PRIMARY LEACH REACTOR # 1	280	m3		1	EA	$728,000	$728,000	20%	$145,600	$873,600
11-RE-02	PRIMARY LEACH REACTOR # 2	280	m3		1	EA	$728,000	$728,000	20%	$145,600	$873,600
11-RE-03	PRIMARY LEACH REACTOR # 3	280	m3		1	EA	$728,000	$728,000	20%	$145,600	$873,600
11-RE-04	PRIMARY LEACH REACTOR # 4	280	m3		1	EA	$728,000	$728,000	20%	$145,600	$873,600
11-AG-01	PRIM. LEACH REACTOR #1 AGITATOR			100	1	EA	$300,000	$300,000	20%	$60,000	$360,000
11-AG-02	PRIM. LEACH REACTOR #2 AGITATOR			100	1	EA	$300,000	$300,000	20%	$60,000	$360,000
11-AG-03	PRIM. LEACH REACTOR #3 AGITATOR			100	1	EA	$300,000	$300,000	20%	$60,000	$360,000
11-AG-04	PRIM. LEACH REACTOR #4 AGITATOR			100	1	EA	$300,000	$300,000	20%	$60,000	$360,000
									20%		
11-SP-01	PRIM. LEACH REACTOR # 1GAS SPARGER				2	EA	$150,000	$300,000	20%	$60,000	$360,000
11-SP-01	PRIM. LEACH REACTOR # 2 GAS SPARGER				2	EA	$150,000	$300,000	20%	$60,000	$360,000
11-SP-01	PRIM. LEACH REACTOR #3 GAS SPARGER				2	EA	$150,000	$300,000	20%	$60,000	$360,000
11-SP-01	PRIM. LEACH REACTOR # 4 GAS SPARGER				2	EA	$150,000	$300,000	20%	$60,000	$360,000
					0				20%		
11-TH-01	THICKENER THANK/MECHANISM	7.6 M Dia		10	1	EA	$750,000	$750,000	20%	$150,000	$900,000
11-TK-14	U/F SURGE TANK				1	EA	$90,000	$90,000	20%	$18,000	$108,000
11-TK-14	U/F SURGE TANK AGITATOR			10	1		$85,000	$85,000	20%	$17,000	$102,000
	SCRUBBER TANK				1	EA	$344,000	$344,000	20%	$68,800	$412,800
11-ft-01	FLASH TANK	2.9dx5.8			1	ea	$257,000	$257,000	20%	$51,400	$308,400
11-FT-02	Flash Tank for Air to sulphur burning				3		$422,000	$1,266,000	20%	$253,200	$1,519,200
				TOTAL:				$9,424,000	0	$1,884,800	$11,308,800

ANDEBURG CONSULTING SERVICES INC.

TITLE: STARFIELD RESOURCES
FERGUSON LAKE METALLURGICAL RECOVERY PROCESS
DEPT.: PRIMARY LEACH

Prepared By: E. Burga

Appv'd. (Eng):

Area	11
Page:	2 of 3
Rev.:	0
Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL -PAGE 2)

Item N°.	Description				MATERIAL Qty.	U.O.M	Unit Cost	*Total*	LABOUR Factor	Total	Total
11-PU-03 A/B	THIKENER O/F RECYCLE PUMP	215	m3/h	40	2		$75,000	$150,000	20%	$30,000	$180,000
11-TKM-01	PRIM. LEACH THICKENER MECHANISM				1		$600,000	$600,000	20%	$120,000	$720,000
11-TTK-01	PRIM. LEACH THICKENER TANK				1		$400,000	$400,000	20%	$80,000	$480,000
11-PU-05 A/B	PRIM. LEACH THICKENER U/F TRANSFER PUMP				2		$100,000	$200,000	20%	$40,000	$240,000
11-TK-02	PRIM. LEACH THICKENER O/F TANK				1		$271,000	$271,000	20%	$54,200	$325,200
11-PU-04 A/B	PRIM. LEACH THICKENER O/F TRANSFER PUMP				2		$60,000	$120,000	20%	$24,000	$144,000
11-FA-21	HCl ID BLOWER			200	1		$400,000	$400,000	20%	$80,000	$480,000
11-FA-22	PRIM LEACH EXHAUST FAN			100	1		$300,000	$300,000	20%	$60,000	$360,000
11-FLM-01	FLOCCULANT MET PACKAGE				1		$75,000	$75,000	20%	$15,000	$90,000
11-TK-03	THICKENER U/F SURGE TANK				1		$100,000	$100,000	20%	$20,000	$120,000
11-HX-01	HCl Condenser	Allow			1		$219,000	$219,000	20%	$43,800	$262,800
11-HX-02	HCl Condenser	Allow			1		$847,000	$847,000	20%	$169,400	$1,016,400
									20%		
									20%		
									20%		
									20%		
									20%		
									20%		
									20%		
									20%		
									20%		
									20%		
									20%		
									20%		
11-HO-01	Cake Feed Hopper	080	m3		1		$150,000	$150,000	20%	$30,000	$180,000
11-BC-03	CAKE TRANSFER BELT CONVEYOR				1		$90,000	$90,000	20%	$18,000	$108,000
				TOTAL:				$3,922,000	0	$784,400	$4,706,400

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	11
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	3 of 3
	DEPT.: PRIMARY LEACH	Rev.:	000
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA SECONDARY LEACH

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$10,583,400	$2,116,700	$12,700,100
002	PIPING	20%	$2,116,680	$423,300	$2,539,980
003	CIVIL/STRUCTURAL	30%	$3,175,020	$635,000	$3,810,020
004	ELECTRICAL/INSTRUMENTATION	25%	$2,645,850	$529,200	$3,175,050
	Labour alloc factor	20%			
	DIRECT COST SUB-TOTAL:		$18,521,000	$3,704,200	$22,225,200

INDIRECT COST

ENGINEERING (%):	10%	$2,222,520
OWNER COST	3%	$666,756
SPARE PARTS	2%	$370,420
TRANSPORTATION	8%	$1,481,680
CHEMICALS INITIAL LOAD	1%	$185,000
INDIRECT COST SUB-TOTAL		$4,926,376
TOTAL COST ESTIMATE		$27,151,576
CONTINGENCY (%):	15%	$4,073,000
TOTAL CAPITAL INVESTMENT :		

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	12
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	1 of 2
	DEPT.: SECONDARY LEACH	Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description				MATERIAL Qty.	I.O.M	Unit Cost	(%)	(%)	(%)	*Total*			LABOUR Factor (%)	Total	Total
12-tk-01	Feed tank	018	m3		1		$82,400				$82,400			20%	$16,500	$98,900
12-AG-021a	Feed tank Agitator			15	1		$25,000				$25,000			20%	$5,000	$30,000
														20%		
12-RE-01a	Pressure Leach Reactor				1		$6,000,000				$6,000,000			20%	$1,200,000	$7,200,000
12-AG-01a/	Reactor Agitator	75 hp each		300	4		$400,000				$1,600,000			20%	$320,000	$1,920,000
	Flash tank				1		$500,000				$500,000			20%	$100,000	$600,000
12-TK-02	Discharge Tank	036	m3		1		$153,000				$153,000			20%	$30,600	$183,600
12-AG-02	Discharge tank agitator			20	1		$75,000				$75,000			20%	$15,000	$90,000
12-PU-02	Filter feed pump	034	m3/hr	10	2		$45,000				$90,000			20%	$18,000	$108,000
12-BF-01	Belt Filter				1		$1,012,000				$1,012,000			20%	$202,400	$1,214,400
12-tk-03	Repulper tank	053	m3		1		$196,000				$196,000			20%	$39,200	$235,200
12-pu-03	Transfer pump	050	m3/hr	15	2		$40,000				$80,000			20%	$16,000	$96,000
12-SB-21a/	Gas Scrubber package			25	1		$150,000				$150,000			20%	$30,000	$180,000
12-DT-01	Vent system i				1		$120,000				$120,000			20%	$24,000	$144,000
12-FA-21a/	Gas Blower				2		$250,000				$500,000			20%	$100,000	$600,000
																0
			TOTAL:								$10,583,400	0			$2,116,700	$12,700,100

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES		Area 12
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS		Page: 2 of 2
	DEPT.: SECONDARY LEACH		Rev.: 001
Prepared By: E. Burga		Appv'd. (Eng):	Date: Nov 19,2011

AREA OXIDATION - EVAPORATION

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$4,015,000	$803,000	$4,818,000
002	PIPING	20%	$803,000	$160,600	$963,600
003	CIVIL/STRUCTURAL	15%	$602,250	$120,450	$722,700
004	ELECTRICAL/INSTRUMENTATION	15%	$602,250	$120,450	$722,700
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$6,022,500	$1,204,500	$7,227,000

INDIRECT COST

ENGINEERING (%):	10%		$722,700
OWNER COST	3%		$216,810
SPARE PARTS	2%		$120,450
TRANSPORTATION	8%		$481,800
CHEMICALS INITIAL LOAD	1%		$60,000
INDIRECT COST SUB-TOTAL			$1,601,760
TOTAL COST ESTIMATE			$8,828,760
CONTINGENCY (%):	15%		$1,324,000
TOTAL CAPITAL INVESTMENT :			$10,152,760

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	13
	FERGUSON LAKE MET RECOVERY PROCESS	Page:	1 of 2
	DEPT.: OXIDATION - EVAPORATION	Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item Nº.	Description					Qty.	U.O.M.	MATERIAL Unit Cost	MATERIAL *Total*		LABOUR Factor (%)	LABOUR Total	Total
13-RE-01	First Oxid Reactor	6.55mdx11m h				1		$757,000	$757,000		20%	$151,400	$908,400
13-RE-02	Second Oxid Reactor	6.55mdx11m h				1		$757,000	$757,000		20%	$151,400	$908,400
13-RE-03	Third Oxid Reactor	6.55mdx11m h				1		$757,000	$757,000		20%	$151,400	$908,400
13-AG-01	First Oxid Reactor Agitator				100	1		$130,000	$130,000		20%	$26,000	$156,000
13-AG-02	Second Oxid Reactor Agitator				100	1		$130,000	$130,000		20%	$26,000	$156,000
13-AG-03	Third Oxid Reactor Agitator				100	1		$130,000	$130,000		20%	$26,000	$156,000
13-TK-01	Feed Tank	2.8m dx 3.6 m H				1		$123,000	$123,000		20%	$24,600	$147,600
											20%		
											20%		
											20%		
13-TK-01	FeedTank	4.7 d x6.1 m h				1		$123,000	$123,000		20%	$24,600	$147,600
											20%		
13-PU-01A/B	Reactor Feed Pumps		207	m3/h	40	2		$15,000	$30,000		20%	$6,000	$36,000
13-PU-02A/B	Oxidation Discharge pump		430	m3/h	100	2		$15,000	$30,000		20%	$6,000	$36,000
	Heat Exchanger evap					1		$240,000	$240,000		20%	$48,000	$288,000
	Heat Exchanger condenser					1		$265,000	$265,000		20%	$53,000	$318,000
13-VC-01	Exhaust Blower				10	2		$80,000	$160,000		20%	$32,000	$192,000
13-FT-01	Vessel for Oxidation flash					1		$383,000	$383,000		20%	$76,600	$459,600
	TOTAL:								$4,015,000	0		$803,000	$4,818,000

ANDEBURG CONSULTING SERVICES INC.

TITLE: STARFIELD RESOURCES
FERGUSON LAKE MET RECOVERY PROCESS
DEPT.: OXIDATION - EVAPORATION

Area: 13
Page: 2 of 2
Rev.: 001
Date: Nov 19,2011

Prepared By: E. Burga
Appv'd. (Eng):

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
15.tk-01	Stage Feed tak	36	m3		1		$59,000	$59,000	20%	$11,800	$70,800
15-pu-01	Sol Feed pump	36	m3/h	1.5	2		$15,000	$30,000	20%	$6,000	$36,000
15-exc-01	Extraction cell	3.mWx 10.6mL			4		$55,800	$223,200	20%	$44,640	$267,840
15-pm-01	Cell pump and mixer assembly		015	60	4		$75,000	$300,000	20%	$60,000	$360,000
15-tk-1570	Organic feed tranfer tank	126	m3		1		$127,000	$127,000	20%	$25,400	$152,400
15-pu-1570	Organic transfer pump	126	m3/h	15	2		$30,000	$60,000	20%	$12,000	$72,000
15-tk-02	Organic feed tank	144	m3		1		$132,000	$132,000	20%	$26,400	$158,400
15-pu-02	Organic feed pump	126	m3/h	15	2		$30,000	$60,000	20%	$12,000	$72,000
15-tk-1571	Organic transfer tank	126	m3		1		$127,000	$127,000	20%	$25,400	$152,400
15-pu-1571	Organic transfer pump	126	m3/h	15	2		$30,000	$60,000	20%	$12,000	$72,000
15-stc-01	Stripping cell	4.3m Wx17.1 m L			2		$82,200	$164,400	20%	$32,880	$197,280
15-pm-01	Cell pump and mixer assembly		010	20	2		$75,000	$150,000	20%	$30,000	$180,000
15-tk-1553	Strip sol feed tank	031	m3		1		$56,000	$56,000	20%	$11,200	$67,200
15-PU-1553	Strip solution feed pump	031	m3/h	4	2		$15,000	$30,000	20%	$6,000	$36,000
15-TK-1581	Stripped sol surge tank	031	m3		1		$56,000	$56,000	20%	$11,200	$67,200
15-pu-1581	Stripped sol transfer pump	031	m3/h	4	2		$15,000	$30,000	20%	$6,000	$36,000
15-tk-03	Treated Solution Transfer Tank	036	m3		1		$59,000	$59,000	20%	$11,800	$70,800
15-pu-03	Treated Solution Transfer Pump	036	m3/h	20	2		$15,000	$30,000	20%	$6,000	$36,000
15-WC-03	Wash Cells	3.7mWx 12.9 mL			4		$62,200	$248,800	20%	$49,760	$298,560
15-PM-03	Wash cell pump-mixer assembly	031	m3/h	4	2		$75,000	$150,000	20%	$30,000	$180,000
15-TK-1591	Wash sol feed tank	031	m3		1		$56,000	$56,000	20%	$11,200	$67,200
15-pu-1591	Wash sol feed pump	031	m3/h	50	2		$15,000	$30,000	20%	$6,000	$36,000
15-TK-1592	Wash sol surge tank	031	m3		1		$56,000	$56,000	20%	$11,200	$67,200
15-pu-1592	Wash sol transfer pump	031	m3/h	5	2		$15,000	$30,000	20%	$6,000	$36,000
15-TK-1581	Washed Sol surge tank	031	m3	10	1		$56,000	$56,000	20%	$11,200	$67,200
15-pu-1581	Washed sol transfer pump	031	m3/h	4	2		$15,000	$30,000	20%	$6,000	$36,000
15-SU-1502	Sump 2	099	m3		1		$43,200	$43,200	20%	$8,640	$51,840
15-SU1503	Emergency sump	126	m3		1		$69,900	$69,900	20%	$13,980	$83,880
	Repulper system agitator				1		$60,000		0%		
15-su-1504	Sump 4	031	m3		1		$22,300	$22,300	20%	$4,460	$26,760
15-su-1505	Vent system i	019	m3		1		$120,000	$120,000	0%	$0	$120,000
15-PKG-01	Solvent treatment package			15	1		$150,000	$150,000	20%	$30,000	$180,000
				TOTAL:				$2,815,800	0	$539,200	$3,355,000

ANDEBURG CONSULTING SERVICES INC. TITLE: STARFIELD RESOURCES
FERGUSON LAKE MET.RECOVERY PROCESS
DEPT.: Fe SX

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Date: Nov 19,2011

Prepared By: E. Burga
Appv'd. (Eng):

AREA Zn IX RECOVERY PACKAGE

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL			$7,801,000	$1,560,200	$9,361,200
002	PIPING	15%		$1,170,150	$234,030	$1,404,180
003	CIVIL/STRUCTURAL	10%		$780,100	$156,020	$936,120
004	ELECTRICAL/INSTRUMENTATION	10%		$780,100	$156,020	$936,120
		Labour alloc factor	20%			
		RESIN LOAD		$8,948,000		$8,948,000
	DIRECT COST SUB-TOTAL:			$19,479,400	$2,106,300	$21,585,600

INDIRECT COST

ENGINEERING (%):	10%	$2,158,560
OWNER COST	3%	$647,568
SPARE PARTS	2%	$389,588
TRANSPORTATION	8%	$1,558,352
CHEMICALS INITIAL LOAD	1%	$195,000
INDIRECT COST SUB-TOTAL		$4,949,068

TOTAL COST ESTIMATE		$26,534,668
CONTINGENCY (%):	15%	$3,980,000
TOTAL CAPITAL INVESTMENT :		$30,514,668

ANDEBURG CONSULTING SERVICES INC.	TITL STARFIELD RESOURCES FERGUSON LAKE MET.RECOVERY PROCESS	Area	16
	DEPT Zn IX RECOVERY PACKAGE	Page:	
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Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	1 of 2

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	(%)	(%)	(%)	*Total*			LABOUR Factor (%)	Total	Total
16-PKG-01	Zn IX PACKAGE	543	m3/h	300	1		$7,801,000				$7,801,000			20%	$1,560,200	$9,361,200
	RESIN				1	LOT	$8,948,000									$8,948,000
	TOTAL:										$7,801,000	0			$1,560,200	$9,361,200

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES	Area	16
	FERGUSON LAKE MET.RECOVERY PROCESS	Page:	2 of 2
	DEPT.: Zn IX RECOVERY PACKAGE	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	2 of 2

AREA Cu/Co SOLVENT EXTRACTION

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$7,948,700	$1,553,700	$9,502,400
002	PIPING	20%	$1,589,740	$310,740	$1,900,480
003	CIVIL/STRUCTURAL	15%	$1,192,305	$238,461	$1,430,766
004	ELECTRICAL/INSTRUMENTATION	15%	$1,192,305	$238,461	$1,430,766
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$11,923,100	$2,341,400	$14,264,400

INDIRECT COST

ENGINEERING (%):	10%		$1,426,440
OWNER COST	2%		$285,288
SPARE PARTS	2%		$238,462
TRANSPORTATION	8%		$953,848
CHEMICALS INITIAL LOAD	1%		$119,000
INDIRECT COST SUB-TOTAL			$3,023,038

TOTAL COST ESTIMATE			$17,287,438
CONTINGENCY (%):	15%		$2,593,000
TOTAL CAPITAL INVESTMENT :			$19,880,438

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES FERGUSON LAKE MET.RECOVERY PROCESS	Area	17
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	DEPT.: Cu/Co SOLVENT EXTRACTION	Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description				MATERIAL Qty.	U.O.M	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
17.tk-01	Stage Feed tak	328	m3		1		$216,000	$216,000	20%	$43,200	$259,200
17-pu-01	Sol Feed pump	547	m3/h	80	2		$60,000	$120,000	20%	$24,000	$144,000
17-exc-01	Extraction cell	9.7mWx 32.75mL			4		$292,400	$1,169,600	20%	$233,920	$1,403,520
17-pm-01	Cell pump and mixer assembly		025	100	4		$180,000	$720,000	20%	$144,000	$864,000
17-tk-1570	Organic disc tranfer tank	328	m3		1		$216,000	$216,000	20%	$43,200	$259,200
17-pu-1570	Organic transfer pump	547	m3/h	50	2		$60,000	$120,000	20%	$24,000	$144,000
17-tk-02	Organic feed tank	328	m3		1		$216,000	$216,000	20%	$43,200	$259,200
17-pu-02	Organic feed pump	547	m3/h	50	2		$60,000	$120,000	20%	$24,000	$144,000
17-tk-1571	Organic transfer tank	287	m3		1		$200,000	$200,000	20%	$40,000	$240,000
17-pu-1571	Organic transfer pump	547	m3/h	50	2		$60,000	$120,000	20%	$24,000	$144,000
17-stc-01	Scrub cell	.7x36.3 m L			2		$285,800	$571,600	20%	$114,320	$685,920
17-pm-01	Cell pump and mixer assembly		025	50	2		$180,000	$360,000	20%	$72,000	$432,000
17-tk-1553	Scrubsol feed tank	287	m3		1		$200,000	$200,000	20%	$40,000	$240,000
17-PU-1553	Scrub solution feed pump	547	m3/h	50	2		$60,000	$120,000	20%	$24,000	$144,000
17-TK-1581	Scrub sol surge tank	287	m3		1		$200,000	$200,000	20%	$40,000	$240,000
17-pu-1581	Scrub sol transfer pump	547	m3/h	50	2		$60,000	$120,000	20%	$24,000	$144,000
17-tk-03	Treated Solution Transfer Tank	287	m3		1		$200,000	$200,000	20%	$40,000	$240,000
17-pu-03	Treated Solution Transfer Pump	547	m3/h	50	2		$60,000	$120,000	20%	$24,000	$144,000
17-WC-03	Strip Cells	7.3mWx 25.4 mL			4		$184,800	$739,200	20%	$147,840	$887,040
17-PM-03	Strip cell pump-mixer assembly	656	m3/h	80	4		$180,000	$720,000	20%	$144,000	$864,000
17-TK-1591	Strip sol feed tank	144	m3		1		$132,000	$132,000	20%	$26,400	$158,400
17-pu-1591	Strip sol feed pump	110	m3/h	4	2		$40,000	$80,000	20%	$16,000	$96,000
17-TK-1592	Strip sol surge tank	144	m3		1		$132,000	$132,000	20%	$26,400	$158,400
17-pu-1592	Strip	110	m3/h	4	2		$40,000	$80,000	20%	$16,000	$96,000
17-TK-1581	Stripped Sol surge tank	144	m3		1		$132,000	$132,000	20%	$26,400	$158,400
17-pu-1581	Washed sol transfer pump	110	m3/h	4	2		$118,100	$236,200	20%	$47,240	$283,440
17-SU-1502	Sump 2	099	m3		1		$118,100	$118,100	20%	$23,620	$141,720
17-SU1503	Repulper system agitator	126	m3		1		$60,000	$60,000	0%	$0	$60,000
17-su-1504	Sump 4	031	m3		1		$40,000	$40,000	20%	$8,000	$48,000
17-su-1505	Vent system i	019	m3		1		$120,000	$120,000	0%	$0	$120,000
17-PKG-01	Solvent treatment package			40	1		$250,000	$250,000	20%	$50,000	$300,000
			TOTAL:					$7,948,700	0	$1,553,700	$9,502,400

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD RESOURCES		Area 17
	FERGUSON LAKE MET.RECOVERY PROCESS		Page: 2 of 2
	DEPT.: Cu/Co SOLVENT EXTRACTION		Rev.: 001
Prepared By: E. Burga		Appv'd. (Eng):	Date: Nov 19,2011

AREA Cu EW

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL	SXEW PACKAGE		$21,751,000	$4,195,200	$25,946,200
002	CIVIL/STRUCTURAL	30%		$6,525,300	$1,305,060	$7,830,360
003	ELECTRICAL/INSTRUMENTATION	20%		$4,350,200	$870,040	$5,220,240
			Labour alloc factor 20%			
		DIRECT COST SUB-TOTAL:		$32,626,500	$6,370,300	$38,996,800

INDIRECT COST

ENGINEERING EPCM (%):	10%		$3,899,680
OWNER COST	3%		$1,169,904
SPARE PARTS	2%		$652,530
TRANSPORTATION	8%		$2,610,120
CHEMICALS INITIAL LOAD	1%		$326,000
INDIRECT COST SUB-TOTAL		TOTAL:	$8,658,234

TOTAL COST ESTIMATE		$47,655,034
CONTINGENCY (%):	15%	$7,148,000
TOTAL CAPITAL INVESTMENT :		$54,803,034

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD		Area	18
		FERGUSON LAKE MET. RECOVERY PROCESS		Page:	
	DEPT.:	Cu EW		Rev.:	001
Prepared By: E. Burga	Reviewed By:	P. Evans	Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item Nº.	Description			MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
18-PKG-01	Cu SX PACKAGE	146 m3/h	250	1		$3,598,000	$3,598,000	20%	$719,600	$4,317,600
18-PKG-02	Cu EW PACKAGE		5895	1		$13,253,000	$13,253,000	20%	$2,650,600	$15,903,600
	37 cells- 3135 Cathodes									
18.-BB-92	Power Bar package			1	lot	$800,000	$800,000	20%	$160,000	$960,000
18-PWT-91	Power Transformer			1	lot	$1,000,000	$1,000,000	20%	$200,000	$1,200,000
18-HO-61	Overhead Crane			1	lot	$600,000	$600,000	10%	$60,000	$660,000
18-CSM-51	Stripping machine			1	lot	$800,000	$800,000	10%	$80,000	$880,000
18-CLM-72	Cathode Handling Device			1	ea	$75,000	$75,000		$0	$75,000
18-fl-02	Polish filter package			1	ea	$125,000	$125,000	20%	$25,000	$150,000
18-PKG-03	Neutralization circuit package			1	ea	$1,500,000	$1,500,000	20%	$300,000	$1,800,000
	TOTAL:						$21,751,000	0	$4,195,200	$25,946,200

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	18
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
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Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA Co SXEW

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$7,409,900	$1,479,000	$8,888,900
002	CIVIL/STRUCTURAL	30%	$2,222,970	$444,594	$2,667,564
003	ELECTRICAL/INSTRUMENTATION	20%	$1,481,980	$296,396	$1,778,376
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$11,114,900	$2,220,000	$13,334,800

INDIRECT COST

ENGINEERING EPCM (%):	10%		$1,333,480
OWNER COST	3%		$400,044
SPARE PARTS	3%		$333,447
TRANSPORTATION	8%		$889,192
CHEMICALS INITIAL LOAD	1%		$111,000
INDIRECT COST SUB-TOTAL		TOTAL:	$3,067,163

TOTAL COST ESTIMATE			$16,401,963
CONTINGENCY (%):	15%		$2,460,000
TOTAL CAPITAL INVESTMENT :			$18,861,963

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD		Area	19
		FERGUSON LAKE METALLURGICAL RECOVERY PROCESS		Page:	1 of 2
	DEPT.:	Co SXEW		Rev.:	001
Prepared By: E. Burga	Reviewed By:	P. Evans	Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description		Qty.	U.O.M.	MATERIAL Unit Cost	*Total*	LABOUR	Factor (%)	Total	Total
20.1-PKG-01	Co SX Package	161 m3/h 200	1		$3,816,000	$3,816,000		20%	$763,200	$4,579,200
21.1-PKG-02	Co EW Package 8 cells with 239 cathodes	1345	1		$3,108,900	$3,108,900		20%	$621,800	$3,730,700
21.3-BB-92	Power Bar package		1	lot	$150,000	$150,000		20%	$30,000	$180,000
213--CSM-51	Stripping machine		1	lot	$120,000	$120,000		20%	$24,000	$144,000
213-HO-61	Overhead Crane		1	lot	$200,000	$200,000		20%	$40,000	$240,000
213-CLM-71	Catode Handling Device		1		$15,000	$15,000			$0	$15,000
	TOTAL:					$7,409,900	0		$1,479,000	$8,888,900

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	19
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
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Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA Ni SXEW

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$22,571,400	$4,306,800	$26,878,200
002	CIVIL/STRUCTURAL	30%	$6,771,420	$1,354,284	$8,125,704
003	ELECTRICAL/INSTRUMENTATION	20%	$4,514,280	$902,856	$5,417,136
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$33,857,100	$6,563,900	$40,421,000

INDIRECT COST

ENGINEERING EPCM (%):	10%	$4,042,100
OWNER COST	3%	$1,212,630
SPARE PARTS	2%	$677,142
TRANSPORTATION	8%	$2,708,568
CHEMICALS INITIAL LOAD	1%	$339,000
INDIRECT COST SUB-TOTAL	TOTAL:	$8,979,440

TOTAL COST ESTIMATE		$49,400,440
CONTINGENCY (%):	15%	$7,410,000
TOTAL CAPITAL INVESTMENT :		$56,810,440

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	20
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Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item Nº.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
20.1 PKG-01	Ni SX PACKAGE	449 m3/ 500	1		$7,061,000	$7,061,000	20%	$1,412,200	$8,473,200
21.1 PKG-01	Ni EW PACKAGE	4457	1		$11,155,400	$11,155,400	20%	$2,231,100	$13,386,500
	42 cells -2086 cathodes								
21.1-BB-92	Power Bar package		1	lot	$800,000	$800,000	20%	$160,000	$960,000
21-PWT-91	Power Transformer		1	lot	$1,000,000	$1,000,000		$0	$1,000,000
21-VEP-61	Ventilation System		1	lot	$200,000	$200,000	20%	$40,000	$240,000
21-ELT-81	Electrolyte Treatment		1	lot	$400,000	$400,000	20%	$80,000	$480,000
21-AP-82	Anode Preparation package		1	lot	$300,000	$300,000	20%	$60,000	$360,000
211—CSM-51	Stripping machine	10	1	lot	$800,000	$800,000	20%	$160,000	$960,000
211-HO-61	Overhead Crane	10	1	lot	$600,000	$600,000	20%	$120,000	$720,000
211-CLM-71	Catode Handling Device		1	e	$75,000	$75,000	10%	$7,500	$82,500
20-fl-02	Polish filter package	005	1	ea	$180,000	$180,000	20%	$36,000	$216,000
	TOTAL:					$22,571,400	0	$4,306,800	$26,878,200

ANDEBURG CONSULTING SERVICES INC.

TITLE: STARFIELD
FERGUSON LAKE MET. RECOVERY PROCESS
DEPT.: Ni SXEW

Prepared By: E. Burga

Appv'd. (Eng):

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AREA Mn EW

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$5,455,000	$791,000	$6,246,000
002	CIVIL/STRUCTURAL	30%	$1,636,500	$327,300	$1,963,800
003	ELECTRICAL/INSTRUMENTATION	20%	$1,091,000	$218,200	$1,309,200
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$8,182,500	$1,336,500	$9,519,000

INDIRECT COST

ENGINEERING EPCM (%):	10%		$951,900
OWNER COST	3%		$285,570
SPARE PARTS	2%		$163,650
TRANSPORTATION	8%		$654,600
CHEMICALS INITIAL LOAD	2%		$164,000
INDIRECT COST SUB-TOTAL		TOTAL:	$2,219,720

TOTAL COST ESTIMATE		$11,738,720
CONTINGENCY (%):	15%	$1,761,000
TOTAL CAPITAL INVESTMENT :		$13,499,720

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Area	21
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			Rev.:	001
Prepared By: E. Burga	Reviewed By:	P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR	Factor (%)	Total	Total
21.4-PKG-01	Mn EW Package 17 cell with 525 cathodes	1350	1	lot	$3,305,000	$3,305,000		20%	$661,000	$3,966,000
21.4-BB-92	Power Bar package		1	lot	$150,000	$150,000		20%	$30,000	$180,000
21.4-BB-91	Power Transformer		1	lot	$500,000	$500,000		20%	$100,000	$600,000
18-PKG-03	Neutralization circuit package	150	1	ea	$1,500,000	$1,500,000			$0	$1,500,000
	TOTAL:					$5,455,000	0		$791,000	$6,246,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	21
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	1 of 2	2 of 2
	DEPT.: Mn EW	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	2 of 2	Nov 19,2011

AREA SULFUR RECOVERY

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$1,499,500	$299,900	$1,799,400
002	CIVIL/STRUCTURAL	10%	$149,950	$89,970	$239,920
003	ELECTRICAL/INSTRUMENTATION	10%	$149,950	$89,970	$239,920
		Labour alloc factor 20%			
		DIRECT COST SUB-TOTAL:	$1,799,400	$479,800	$2,279,200

INDIRECT COST

ENGINEERING EPCM (%):	10%	$227,920
OWNER COST	3%	$68,376
SPARE PARTS	2%	$35,988
TRANSPORTATION	8%	$143,952
CHEMICALS INITIAL LOAD	2%	$35,988
INDIRECT COST SUB-TOTAL	TOTAL:	$512,224

TOTAL COST ESTIMATE		$2,791,424
CONTINGENCY (%):	15%	$419,000
TOTAL CAPITAL INVESTMENT :		$3,210,424

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Area	22
		Page:	1 of 2
	DEPT.: SULFUR RECOVERY	Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
									20%		
22-TK-01	Feed Tank	2.8 m dx3.6 m h	22 m3		1		$38,200	$38,200	20%	$7,640	$45,840
22-AG-01	Feed Tank Agitator			15	1		$60,000	$60,000	20%	$12,000	$72,000
22-PU-01	Flotation Feed Pump		63 m3/h	10	2		$35,000	$70,000	20%	$14,000	$84,000
22-FLT-01	Rough Flotation cell	1.77 mdx2.25 mh		25	1		$170,000	$170,000	20%	$34,000	$204,000
22-BWL-01	Flotation Blower			20	1		$40,000	$40,000	20%	$8,000	$48,000
22-TK-02	Thick O/F tank	2.75mdx4 mh	18 m3		1		$41,000	$41,000	20%	$8,200	$49,200
22-TK-03	Thick Feed tank	2.75mdx4 mh	18 m3		1		$41,000	$41,000	20%	$8,200	$49,200
22-TK-04	Flot Cell disch tank	2.2mdx3m H	10 m3		1		$29,300	$29,300	20%	$5,860	$35,160
22-TK-05	Tailing Pump Tank	3.35mdx4.5mH	32 m3		2		$60,000	$120,000	20%	$24,000	$144,000
22-AG-03	Thick Feed tank Agitator			10	1		$35,000	$35,000	20%	$7,000	$42,000
22-AG-04	Flot Cell disch tank Agitator			10	1		$35,000	$35,000	20%	$7,000	$42,000
22-AG-05	Tailing Pump Tank Agitator			30	2		$45,000	$90,000	20%	$18,000	$108,000
22-TH-01	Thickener Tank and Mechanism		3m D	4	1		$200,000	$200,000	20%	$40,000	$240,000
22-PU-02	Thickener feed pump		55 m3/h	10	2		$30,000	$60,000	20%	$12,000	$72,000
22-PU-03	Thickener U/f Pumps		32 m3/h	10	2		$20,000	$40,000	20%	$8,000	$48,000
22-PU-04	Thickener O/F Recirc. Pumps		60 m3/h	10	2		$25,000	$50,000	20%	$10,000	$60,000
22-PU-05	Tailing Transfer Pumps		35 m3/h	30	2		$40,000	$80,000	20%	$16,000	$96,000
22-FL-01	Filter		8 m2	20	1		$300,000	$300,000	20%	$60,000	$360,000
				TOTAL:				$1,499,500	0	$299,900	$1,799,400

ANDEBURG CONSULTING SERVICES INC.

TITLE: STARFIELD
FERGUSON LAKE METALLURGICAL RECOVERY PROCESS
DEPT.: SULFUR RECOVERY

Area: 22
Page: 2 of 2
Rev.: 001

Prepared By: E. Burga

Appv'd. (Eng):

Date: Nov 19,2011

AREA REAGENTS

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$4,300,000	$660,000	$4,960,000
002	CIVIL/STRUCTURAL	15%	$645,000	$129,000	$774,000
003	ELECTRICAL/INSTRUMENTATION	15%	$645,000	$129,000	$774,000
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$5,590,000	$918,000	$6,508,000

INDIRECT COST

ENGINEERING EPCM (%):	10%		$650,800
OWNER COST	3%		$195,240
SPARE PARTS	3%		$167,700
TRANSPORTATION	8%		$447,200
CHEMICALS INITIAL LOAD	1%		$55,900
INDIRECT COST SUB-TOTAL	TOTAL:		$1,516,840
TOTAL COST ESTIMATE			$1,516,840
CONTINGENCY (%):	15%		$228,000
TOTAL CAPITAL INVESTMENT :			$1,744,840

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Area	23
	DEPT.: REAGENTS	Page:	1 of 2
		Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
									20%		
22-TK-01	Feed Tank	2.8 m dx3.6 m h	22 m3		1		$38,200	$38,200	20%	$7,640	$45,840
22-AG-01	Feed Tank Agitator			15	1		$60,000	$60,000	20%	$12,000	$72,000
22-PU-01	Flotation Feed Pump		63 m3/h	10	2		$35,000	$70,000	20%	$14,000	$84,000
22-FLT-01	Rough Flotation cell	1.77 mdx2.25 mh		25	1		$170,000	$170,000	20%	$34,000	$204,000
22-BWL-01	Flotation Blower			20	1		$40,000	$40,000	20%	$8,000	$48,000
22-TK-02	Thick O/F tank	2.75mdx4 mh	18 m3		1		$41,000	$41,000	20%	$8,200	$49,200
22-TK-03	Thick Feed tank	2.75mdx4 mh	18 m3		1		$41,000	$41,000	20%	$8,200	$49,200
22-TK-04	Flot Cell disch tank	2.2mdx3m H	10 m3		1		$29,300	$29,300	20%	$5,860	$35,160
22-TK-05	Tailing Pump Tank	3.35mdx4.5mH	32 m3		2		$60,000	$120,000	20%	$24,000	$144,000
22-AG-03	Thick Feed tank Agitator			10	1		$35,000	$35,000	20%	$7,000	$42,000
22-AG-04	Flot Cell disch tank Agitator			10	1		$35,000	$35,000	20%	$7,000	$42,000
22-AG-05	Tailing Pump Tank Agitator			30	2		$45,000	$90,000	20%	$18,000	$108,000
22-TH-01	Thickener Tank and Mechanism		3m D	4	1		$200,000	$200,000	20%	$40,000	$240,000
22-PU-02	Thickener feed pump		55 m3/h	10	2		$30,000	$60,000	20%	$12,000	$72,000
22-PU-03	Thickener U/f Pumps		32 m3/h	10	2		$20,000	$40,000	20%	$8,000	$48,000
22-PU-04	Thickener O/F Recirc. Pumps		60 m3/h	10	2		$25,000	$50,000	20%	$10,000	$60,000
22-PU-05	Tailing Transfer Pumps		35 m3/h	30	2		$40,000	$80,000	20%	$16,000	$96,000
22-FL-01	Filter		8 m2	20	1		$300,000	$300,000	20%	$60,000	$360,000
				TOTAL:				$1,499,500	0	$299,900	$1,799,400

ANDEBURG CONSULTING SERVICES INC.

TITLE: STARFIELD
FERGUSON LAKE METALLURGICAL RECOVERY PROCESS
DEPT.: SULFUR RECOVERY

Area: 22
Page: 2 of 2
Rev.: 001
Date: Nov 19,2011

Prepared By: E. Burga

Appv'd. (Eng):

AREA REAGENTS

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$4,300,000	$660,000	$4,960,000
002	CIVIL/STRUCTURAL	15%	$645,000	$129,000	$774,000
003	ELECTRICAL/INSTRUMENTATION	15%	$645,000	$129,000	$774,000
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$5,590,000	$918,000	$6,508,000

INDIRECT COST

ENGINEERING EPCM (%):	10%		$650,800
OWNER COST	3%		$195,240
SPARE PARTS	3%		$167,700
TRANSPORTATION	8%		$447,200
CHEMICALS INITIAL LOAD	1%		$55,900
INDIRECT COST SUB-TOTAL		TOTAL:	$1,516,840

TOTAL COST ESTIMATE			$1,516,840
CONTINGENCY (%):	15%		$228,000
TOTAL CAPITAL INVESTMENT :			$1,744,840

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD		Area	23
		FERGUSON LAKE METALLURGICAL RECOVERY PROCESS		Page:	1 of 2
	DEPT.:	REAGENTS		Rev.:	001
Prepared By: E. Burga	Reviewed By:	P. Evans	Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item Nº.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
							20%		
23-PKG-01	Floculant storage and preparation	15	1		$150,000	$150,000	20%	$30,000	$180,000
23-PKG-02	Lime Receiving/storage and handling	50	1		$1,500,000	$1,500,000	20%	$300,000	$1,800,000
23-PKG-03	Lime Preparation Package	40	1		$500,000	$500,000	20%	$100,000	$600,000
23-PKG-04	Ammonium Carbonate Rece/stor and handlg	50	1		$1,000,000	$1,000,000		$0	$1,000,000
23-PKG-05	Ammonium Carbonate Preparation Package	40	1		$400,000	$400,000	20%	$80,000	$480,000
23-PKG-06	Conc HCl Receiving and Prep	30	1		$100,000	$100,000	20%	$20,000	$120,000
23-PKG-07	Conc H2SO4 Receiving and Prep	30	1		$150,000	$150,000	20%	$30,000	$180,000
							20%		
23-PKG-08	Solvent Storage package	60	1		$500,000	$500,000	20%	$100,000	$600,000
	TOTAL:					$4,300,000	0	$660,000	$4,960,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	23
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
	DEPT.: REAGENTS	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA ACID REGENERATION AND EVAPORATION

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$2,500,000	$500,000	$3,000,000
002	CIVIL/STRUCTURAL	10%	$250,000	$50,000	$300,000
003	ELECTRICAL/INSTRUMENTATION	5%	$125,000	$25,000	$150,000
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$2,875,000	$575,000	$3,450,000

INDIRECT COST

ENGINEERING EPCM (%):	10%		$345,000
OWNER COST	3%		$103,500
SPARE PARTS	3%		$86,250
TRANSPORTATION	8%		$230,000
CHEMICALS INITIAL LOAD	1%		$28,750
INDIRECT COST SUB-TOTAL		TOTAL:	$793,500

TOTAL COST ESTIMATE		$793,500
CONTINGENCY (%):	15%	$119,000
TOTAL CAPITAL INVESTMENT :		$912,500

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD		Area	24
		FERGUSON LAKE METALLURGICAL RECOVERY PROCESS		Page:	1 of 2
	DEPT.:	ACID REGENERATION AND EVAPORATION		Rev.:	002
Prepared By: E. Burga	Reviewed By:	P. Evans	Appv'd. (Eng.):	Date:	Nov 23,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
24-PKG-01	Acid regeneration Package (allowance)	300	1		$2,500,000	$2,500,000	20% 20% 20% 20%	$500,000	$3,000,000
	TOTAL:					$2,500,000	0	$500,000	$3,000,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	24
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
	DEPT.: ACID REGENERATION AND EVAPORATION	Rev.:	002
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 23,2011

AREA EVAPORATION 2&3

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$6,037,000	$1,207,400	$7,244,400
002	CIVIL/STRUCTURAL	30%	$1,811,100	$362,220	$2,173,320
003	ELECTRICAL/INSTRUMENTATION	20%	$1,207,400	$241,480	$1,448,880
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$9,055,500	$1,811,100	$10,866,600

INDIRECT COST

ENGINEERING EPCM (%):	10%		$1,086,660
OWNER COST	3%		$325,998
SPARE PARTS	3%		$271,665
TRANSPORTATION	2%		$181,110
CHEMICALS INITIAL LOAD	1%		$90,555
INDIRECT COST SUB-TOTAL		TOTAL:	$1,955,988

TOTAL COST ESTIMATE		$1,955,988
CONTINGENCY (%):	15%	$293,000
TOTAL CAPITAL INVESTMENT :		$2,248,988

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD	Area	25
		FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	1 of 2
	DEPT.:	EVAPORATION 2&3	Rev.:	001
Prepared By: E. Burga	Reviewed By:	P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item Nº.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
25-ft-02	Vessel for Evap 2B	3.7mDX7.5 mH	1		$345,000	$345,000	20%	$69,000	$414,000
25-ft-03	Vessel for EVAP 2C		1		$81,000	$81,000	20%	$16,200	$97,200
25-ft-04	Vessel for EVAP 2D		2		$81,000	$162,000	20%	$32,400	$194,400
25-ft-05	Vessel for EVAP 3		3		$483,000	$1,449,000	20%	$289,800	$1,738,800
25-TB-01	Evap 3 Turbine (Allowance)	778 m3/h	1		$4,000,000	$4,000,000	20%	$800,000	$4,800,000
	TOTAL:					$6,037,000	0	$1,207,400	$7,244,400

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	25
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
	DEPT.: EVAPORATION 2&3	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA ACID PLANT WITH POWER GENERATION SET

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$109,000,000	$21,800,000	$130,800,000
002	CIVIL/STRUCTURAL	5%	$5,450,000	$1,090,000	$6,540,000
003	ELECTRICAL/INSTRUMENTATION	2%	$2,180,000	$436,000	$2,616,000
		Labour alloc factor 20%			
	DIRECT COST SUB-TOTAL:		$116,630,000	$23,326,000	$139,956,000

INDIRECT COST

ENGINEERING EPCM (%):	10%	$13,995,600
OWNER COST	3%	$4,198,680
SPARE PARTS	3%	$3,498,900
TRANSPORTATION	2%	$2,332,600
CHEMICALS INITIAL LOAD	1%	$1,166,300
INDIRECT COST SUB-TOTAL	TOTAL:	$25,192,080

TOTAL COST ESTIMATE		$165,148,080
CONTINGENCY (%):	15%	$24,772,000
TOTAL CAPITAL INVESTMENT :		$189,920,080

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	1100
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	1 of 2
	DEPT.: ACID PLANT WITH POWER GENERATION SET	Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	(%)	*Total*	LABOUR	Factor (%)	Total	Total
2100-PKG-01	Acid Plant		1		$109,000,000		$109,000,000		20%	$21,800,000	$130,800,000
	Turbine generator	included	1								
									0.00		
									0.00		
	TOTAL:						$109,000,000	0		$21,800,000	$130,800,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	1100
	FERGUSON LAKE MET. RECOVERY PROCESS	Page:	2 of 2
	DEPT.: ACID PLANT	Rev.:	2 of 2
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description		Qty.	U.O.M.	MATERIAL Unit Cost	*Total*	LABOUR Factor (%)	LABOUR Total	Total
50PCK-01	ACID STORAGE 700,000 Tons storage cap including 25% for operation support 27 tanks Dia 45 H 9.7		1		$65,290,000	$65,290,000	20%	$13,058,000	$78,348,000
50-PU-01	Pumping station	60	2		$60,000	$120,000	20%	$24,000	$144,000
	Recirculatin pumps- 10 allocated (price included in tank farm)	200					20%		
	TOTAL:					$65,410,000	0	$13,082,000	$78,492,000

ANDEBURG CONSULTING SERVICES INC.

TITLE: STARFIELD
FERGUSON LAKE METALLURGICAL RECOVERY PROCESS
DEPT.: ACID STORAGE

Area 1050
Page: 2 of 2
Rev.: 002

Prepared By: E. Burga

Appv'd. (Eng):

Date: Nov 23,2011

AREA OXYGEN PLANT

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL			$19,500,000	$3,900,000	$23,400,000
002	CIVIL/STRUCTURAL		10%	$1,950,000	$390,000	$2,340,000
003	ELECTRICAL/INSTRUMENTATION	(INCLUDED IN PACKAGE)		$0	$0	$0
			Labour alloc factor			
			20%			
		DIRECT COST SUB-TOTAL:		$21,450,000	$4,290,000	$25,740,000

INDIRECT COST

ENGINEERING EPCM (%):	10%	$2,574,000
OWNER COST	3%	$772,200
SPARE PARTS	2%	$429,000
TRANSPORTATION	8%	$1,716,000
CHEMICALS INITIAL LOAD	1%	$214,500
INDIRECT COST SUB-TOTAL	TOTAL:	$5,705,700

TOTAL COST ESTIMATE		$31,445,700
CONTINGENCY (%):	15%	$4,717,000
TOTAL CAPITAL INVESTMENT :		$36,162,700

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD	Area	1200
		FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	1 of 2
	DEPT.:	OXYGEN PLANT	Rev.:	002
Prepared By: E. Burga	Reviewed By:	P. Evans Appv'd. (Eng.):	Date:	Nov 23,2011

AREA OXYGEN PLANT

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION			MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL			$19,500,000	$3,900,000	$23,400,000
002	CIVIL/STRUCTURAL		10%	$1,950,000	$390,000	$2,340,000
003	ELECTRICAL/INSTRUMENTATION	(INCLUDED IN PACKAGE)		$0	$0	$0
			Labour alloc factor			
			20%			
		DIRECT COST SUB-TOTAL:		$21,450,000	$4,290,000	$25,740,000

INDIRECT COST

ENGINEERING EPCM (%):	10%	$2,574,000
OWNER COST	3%	$772,200
SPARE PARTS	2%	$429,000
TRANSPORTATION	8%	$1,716,000
CHEMICALS INITIAL LOAD	1%	$214,500
INDIRECT COST SUB-TOTAL	TOTAL:	$5,705,700

TOTAL COST ESTIMATE		$31,445,700
CONTINGENCY (%):	15%	$4,717,000
TOTAL CAPITAL INVESTMENT :		$36,162,700

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD		Area	1200
		FERGUSON LAKE METALLURGICAL RECOVERY PROCESS		Page:	1 of 2
	DEPT.:	OXYGEN PLANT		Rev.:	001
Prepared By: E. Burga	Reviewed By:	P. Evans	Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item Nº.	Description				MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
1515	Oxygen Plant	214	tpd	250	1	lot	$19,500,000	$19,500,000	20%	$3,900,000	$23,400,000
				TOTAL:				$19,500,000	0	$3,900,000	$23,400,000.00

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	1200
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
	DEPT.: OXYGEN PLANT	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA SITE SUPPORTING PLANTS

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$1,700,000	$340,000	$2,040,000
002	CIVIL/STRUCTURAL	10%	$170,000	$34,000	$204,000
003	ELECTRICAL/INSTRUMENTATION	(INCLUDED IN PACKAGE)	$0	$0	$0
		LABOUR ALLOC 20%			
	DIRECT COST SUB-TOTAL:		$1,870,000	$374,000	$2,244,000

INDIRECT COST

ENGINEERING EPCM (%):	10%	$224,400
OWNER COST	3%	$67,320
SPARE PARTS	2%	$37,400
TRANSPORTATION	8%	$149,600
CHEMICALS INITIAL LOAD	1%	$18,700
INDIRECT COST SUB-TOTAL	TOTAL:	$497,420

TOTAL COST ESTIMATE		$2,741,420
CONTINGENCY (%):	15%	$411,000
TOTAL CAPITAL INVESTMENT :		$3,152,420

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD FERGUSON LAKE METALLURGICAL RECOVERY PROCESS DEPT.: SITE SUPPORTING PLANTS	Area 1300 Page: 1 of 2 Rev.: 001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date: Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
1301-PKG-01	Potable Water	50	1	lot	$400,000	$400,000	20%	$80,000	$480,000
1301-PKG-03	Sanitary WWTP	75	1	lot	$300,000	$300,000	20%	$60,000	$360,000
1301-PKG-03	Incinirator	50	1	lot	$250,000	$250,000	20%	$50,000	$300,000
1300-PKG-04	Plant heating system	125	1	lot	$750,000	$750,000	20%	$150,000	$900,000
	TOTAL:					$1,700,000	0	$340,000	2040000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	1300
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
	DEPT.: SITE SUPPORTING PLANTS	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA UTILITIES-

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$14,550,000	$2,910,000	$17,460,000
002	CIVIL/STRUCTURAL	5%	$727,500	$145,500	$873,000
003	ELECTRICAL/INSTRUMENTATION	5%	$727,500	$145,500	$873,000
		LABOUR ALLOCAC 20%			
	DIRECT COST SUB-TOTAL:		$16,005,000	$3,201,000	$19,206,000

INDIRECT COST

ENGINEERING EPCM (%):	10%	$1,920,600
OWNER COST	3%	$576,180
SPARE PARTS	3%	$480,150
TRANSPORTATION	2%	$320,100
CHEMICALS INITIAL LOAD	1%	$160,050
INDIRECT COST SUB-TOTAL	TOTAL:	$3,457,080

TOTAL COST ESTIMATE		$22,663,080
CONTINGENCY (%):	15%	$3,399,000
TOTAL CAPITAL INVESTMENT :		$26,062,080

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Area	1400
	DEPT.: UTILITIES-	Page:	1 of 2
		Rev.:	001
Prepared By: E. Burga	Reviewed By: P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
1400-PKG-01	Boiler (Allow)	50	1		$3,000,000	$3,000,000	20%	$600,000	$3,600,000
1400-PKG-02	Air compressors plant air (allow)	250	1		$600,000	$600,000	20%	$120,000	$720,000
1400-PKG-03	Cooling water system	300	1		$750,000	$750,000	20%	$150,000	$900,000
1400-PKG-04	Fire Protection System	75	1		$200,000	$200,000	20%	$40,000	$240,000
1400-PKG-05	Emergency Generating Plant	25	1		$10,000,000	$10,000,000	20%	$2,000,000	$12,000,000
	TOTAL:					$14,550,000	0	$2,910,000	$17,460,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	1400
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
	DEPT.: UTILITIES-	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA POWER GENERATION PROCESS RECOVERY

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$10,000,000	$2,000,000	$12,000,000
002	CIVIL/STRUCTURAL	5%	$500,000	$100,000	$600,000
003	ELECTRICAL/INSTRUMENTATION	5%	$500,000	$100,000	$600,000
		LABOUR ALLOC 20%			
	DIRECT COST SUB-TOTAL:		$11,000,000	$2,200,000	$13,200,000

INDIRECT COST

ENGINEERING EPCM (%):	10%		$1,320,000
OWNER COST	3%		$396,000
SPARE PARTS	3%		$330,000
TRANSPORTATION	2%		$220,000
CHEMICALS INITIAL LOAD	1%		$110,000
INDIRECT COST SUB-TOTAL	TOTAL:		$2,376,000

TOTAL COST ESTIMATE		$15,576,000
CONTINGENCY (%):	15%	$2,336,000
TOTAL CAPITAL INVESTMENT :		$17,912,000

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Area	1500
	DEPT.:	POWER GENERATION PROCESS RECOVERY	Page:	1 of 2
			Rev.:	001
Prepared By: E. Burga	Reviewed By:	P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description	MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
1520-PG-01	Heat and energy Recovery System (as allowance for additional components)	1		$10,000,000	$10,000,000	20% 20% 20%	$2,000,000	$12,000,000
TOTAL:					$10,000,000	0	$2,000,000	$12,000,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	1500
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
	DEPT.: POWER GENERATION PROCESS RECOVERY	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

AREA BUILDINGS AND SITE SERVICES

PROJECT COST ESTIMATE SUMMARY SHEET

ESCALATION ALLOWED (%): 0

COMMENTS

DIRECT COST

ITEM	DESCRIPTION		MATERIAL	LABOUR	TOTAL
001	PROCESS AND MECHANICAL		$20,065,000	$4,013,000	$24,078,000
002	CIVIL/STRUCTURAL	5%	$1,003,250	$200,650	$1,203,900
003	ELECTRICAL/INSTRUMENTATION	5%	$1,003,250	$200,650	$1,203,900
		LABOUR ALLOC 20%			
	DIRECT COST SUB-TOTAL:		$22,071,500	$4,414,300	$26,485,800

INDIRECT COST

ENGINEERING EPCM (%):	10%	$2,648,580
OWNER COST	3%	$794,574
SPARE PARTS	3%	$662,145
TRANSPORTATION	8%	$1,765,720
CHEMICALS INITIAL LOAD	1%	$220,715
INDIRECT COST SUB-TOTAL	TOTAL:	$6,091,734

TOTAL COST ESTIMATE		$32,577,534
CONTINGENCY (%):	15%	$4,887,000
TOTAL CAPITAL INVESTMENT :		$37,464,534

ANDEBURG CONSULTING SERVICES INC.	TITLE:	STARFIELD FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Area	1600
	DEPT.:	BUILDINGS AND SITE SERVICES	Page:	1 of 2
			Rev.:	001
Prepared By: E. Burga	Reviewed By:	P. Evans Appv'd. (Eng.):	Date:	Nov 19,2011

DETAILED PROJECT COST ESTIMATE (MECHANICAL)

Item N°.	Description		MATERIAL Qty.	U.O.M.	Unit Cost	*Total*	LABOUR Factor (%)	Total	Total
							20%		
1600	Allowances for buildings		1				20%		
							20%		
	Administration building	30x50	1500	m2	$900	$1,350,000	20%	$270,000	$1,620,000
	Process Building	120x80	9600	m2	$700	$6,720,000	20%	$1,344,000	$8,064,000
	Laboratory	30x40	1200	m2	$900	$1,080,000	20%	$216,000	$1,296,000
	Warehouse	50x80	4000	m2	$600	$2,400,000	20%	$480,000	$2,880,000
	Storage	60X120	7200	m2	$400	$2,880,000	20%	$576,000	$3,456,000
	Concentrator Building	60X80	4800	m2	$700	$3,360,000	20%	$672,000	$4,032,000
	Communication Package		1		$75,000	$75,000	20%	$15,000	$90,000
	Site Lighting		1		$200,000	$200,000	20%	$40,000	$240,000
	In plant roads 4 km		4		$500,000	$2,000,000	20%	$400,000	$2,400,000
	TOTAL:					$20,065,000	0	$4,013,000	$24,078,000

ANDEBURG CONSULTING SERVICES INC.	TITLE: STARFIELD	Area	1600
	FERGUSON LAKE METALLURGICAL RECOVERY PROCESS	Page:	2 of 2
	DEPT.: BUILDINGS AND SITE SERVICES	Rev.:	001
Prepared By: E. Burga	Appv'd. (Eng):	Date:	Nov 19,2011

APPENDIX 4 Cost Estimate Back-up

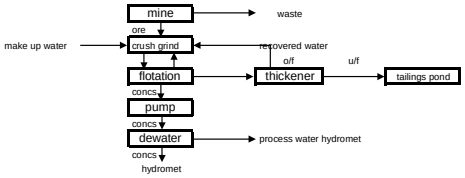
Starfield Equipment Sizing

STARFIELD EQUIPMENT SIZING

6000 tpd plant — By: Nov 19-2011 P.E — REF — Mb

mine

		tpd
waste		1000
ore		6000
water		600



crush grind

	tpd
ore	6000
total water	10000
water to flotation	6000
recyled water from grind	6000
recycled water from tails	0
make up water	3400

this is the amount of fresh water needed at the mine site

flotation

	tpd
concentrates	3311
water with concs	10000
tails solids	2689
water with tails	1793
concentrates	3311
water with concs	2207
recycled water	6000

tailings pond

	tpd
tails solids	2689
water with tails	1793
recycled water	0

concentrate pumping

	tpd
concentrates	3311
water with concs	2207
	5518.333333

concentrate dewatering

	tpd
concentrates	3311
water with concs	523
water to process	1684

this is less than process water consumption(2705tpd)

5 solids SG
1.7 solution SG

vessels

area 10 concentrate dewatering

equipment	flow	flow	flow	retention	active	total				Hbatch	H freeboard	Hheel	*Diameter*	*Htank*	
number	m3/h solids	m3/h liquid	m3/h slurry	time mins	capacity m3	capacity m3	Batch Ratio	Freboard % V							
10tk01	12.9	92.0	105	480	839	890	1.2	0.06		11.54	0.69	0.01	***9.62***	***12.25***	
10tk02	12.9	92.0	105	480	839	890	1.2	0.06		11.54	0.69	0.01	***9.62***	***12.25***	
10tk03	12.9	92.0	105	480	839	890	1.2	0.06		11.54	0.69	0.01	***9.62***	***12.25***	

area 11 primary leach

equipment	flow	flow	flow	retention	active	total			stream #	Hbatch	H freeboard	Hheel	*Diameter*	*Htank*	
number	m3/h solids	m3/h liquid	m3/h slurry	time mins	capacity m3	capacity m3	Batch Ratio	Freboard % V							temp c
11tk01	12.9	235	248	10	41	44	1.2	0.06	3	4.23	0.25	0.01	***3.53***	***4.50***	104
11re01	9.3	390	400	30	200	280	1.2	0.4	9	7.16	2.86	0	***5.96***	***10.02***	110
11re02	9.3	390	400	30	200	280	1.2	0.4	9	7.16	2.86	0	***5.96***	***10.02***	
11re03	9.3	390	400	30	200	280	1.2	0.4	9	7.16	2.86	0	***5.96***	***10.02***	
11re04	9.3	390	400	30	200	280	1.2	0.4	9	7.16	2.86	0	***5.96***	***10.02***	
11tk02		381	381	30	190	202	1.2	0.06	2 & 11	7.04	0.42	0.01	***5.87***	***7.47***	110
11tk03	9.3	10	19	30	10	14	1.2	0.06	12	2.59	0.16	1.01	***2.16***	***3.76***	110

area 12 secondary leach

equipment	flow	flow	flow	retention	active	total	Batch Ratio	Freboard % V	stream #	Hbatch	H freeboard	Hheel	*Diameter*	*Htank*	
number	m3/h solids	m3/h liquid	m3/h slurry	time mins	capacity m3	capacity m3									
12tk01	9.3	24	34	30	17	18	1.2	0.06	200+201	3.13	0.19	0.01	***2.61***	***3.33***	70
	60	% full	4	ratio length to diameter							r cubed	radius		*length*	
12au01	10.5	23	34	300	170	283			14	283	11	2.2	***4.48***	***17.93***	120
12tk02	10.5	23	34	60	34	36	1.2	0.06	14	3.96	0.24	0.01	***3.30***	***4.21***	
12tk03	9.0	41	50	60	50	53	1.2	0.06	13	4.51	0.27	0.01	***3.76***	***4.79***	90

area 13 oxidation 54769800 g/h Fe++ oxidized 68 g/h/litre — 805 m3 — 796 *m3*

equipment	flow	flow	flow	retention	active	total	Batch Ratio	Freboard % V	stream #	Hbatch	H freeboard	Hheel	*Diameter*	*Htank*	
number	m3/h solids	m3/h liquid	m3/h slurry	time mins	capacity m3	capacity m3									
13tk01		207	207.4	30	103.7	110.1	1.2	0.06	24	5.75	0.35	0.01	***4.79***	***6.11***	120
13re01	4.3	426	430.0	37	265.2	371.3	1.2	0.4	24	7.86	3.15	0	***6.55***	***11.01***	
13re02	4.3	426	430.0	37	265.2	371.3	1.2	0.4	24	7.86	3.15	0	***6.55***	***11.01***	
13re03	4.3	426	430.0	37	265.2	371.3	1.2	0.4	24	7.86	3.15	0	***6.55***	***11.01***	

area 14 hydrolysis 0.9 litres for 3.0 litres oxidation — 242 m3 — 244 *m3*

equipment	flow	flow	flow	retention	active	total	Batch Ratio	Freboard % V	stream #	Hbatch	H freeboard	Hheel	*Diameter*	*Htank*	
number	m3/h solids	m3/h liquid	m3/h slurry	time mins	capacity m3	capacity m3									
14re01	16.8	227	243.6	20	81.2	113.9	1.2	0.4	31	5.30	2.12	0.01	***4.42***	***7.43***	180
14re02	16.8	227	243.6	20	81.2	113.9	1.2	0.4	31	5.30	2.12	0.01	***4.42***	***7.43***	
14re03	16.8	227	243.6	20	81.2	129.2	1.2	0.4	31	5.30	2.12	1.01	***4.42***	***8.43***	
14tk01	16.8	187	203.5	30	101.7	108.0	1.2	0.06	17	5.71	0.34	0.01	***4.76***	***6.07***	90
14tk02	16.8	80	96.8	60	96.8	102.8	1.2	0.06	41	5.62	0.34	0.01	***4.68***	***5.97***	72

STARFIELD EQUIPMENT SIZING

6000 tpd plant — By: — Nov 19-2011 — REF — Mb — P.E

	area	22	S recovery										
equipment	flow	flow	flow	retention	active	total	Batch Ratio	Freboard % V		Hbatch	H freeboard	Hheel	Diameter	Htank
number	m3/h solids	m3/h liquid	m3/h slurry	time mins	capacity m3	capacity m3								
22tk01	9.0	54	62.8	20	20.9	22.3	1.2	0.06		3.37	0.20	0.01	***2.81***	***3.59***
22tk02		54	53.9	20	18.0	19.1	1.2	0.06		3.20	0.19	0.01	***2.67***	***3.41***
22flt01	9.0	54	62.8	5	5.2	5.6	1.2	0.06		2.13	0.13	0.01	***1.77***	***2.26***

	area	23	reagents										
equipment	flow	flow	flow	retention	active	total	Batch Ratio	Freboard % V		Hbatch	H freeboard	Hheel	Diameter	Htank
number	m3/h solids	m3/h liquid	m3/h slurry	time mins	capacity m3	capacity m3								
23tk01	1.0	15	15.9	24	6.4	6.8	1.2	0.06	lime	2.27	0.14	0.01	***1.89***	***2.41***
23tk02		26	25.8	24	10.3	11.0	1.2	0.06	amm carb	2.66	0.16	0.01	***2.22***	***2.83***
23tk03		0.26	0.3	24	0.1	0.1	1.2	0.06	HCl	0.57	0.03	0.01	***0.48***	***0.62***
23tk04		18	18.0	24	7.2	7.7	1.2	0.06	H2SO4	2.36	0.14	0.01	***1.97***	***2.52***

thickeners — temperature c

	concentrate thickening before pumping							
equipment		flow	area	200				
number		m3/d	m3/m2/d	scale up	m2	d2	d m	
200th01		10000	1027	0.5	19.5	24.8	5.0	
			flow					
			tpd solids	m2/t/d	m2	d2	d m	
			3311	0.01	33.1	42.2	6.5	
	tailings thickening x flotation							
equipment		flow	area	200				
number		m3/d	m3/m2/d	scale up	m2	d2	d m	
200th02		6000	1000	0.5	12.0	15.3	3.9	
			flow					
			tpd solids	m2/t/d	m2	d2	d m	
			2689	0.01	26.9	34.2	5.9	
	primary leach							
equipment		flow						stream #
number		m3/d	m3/m2/d	scale up	m2	d2	d m	
11th01		9592	960	0.5	20.0	25.4	5.0	10 110
			flow					
			tpd solids	m2/t/d	m2	d2	d m	
			915	0.05	45.8	58.2	7.6	
	hydrolysis							
equipment		flow						17 90
number		m3/d	m3/m2/d	scale up	m2	d2	d m	
14th01		4883	1000	0.5	9.8	12.4	3.5	
			flow					
			tpd solids	m2/t/d	m2	d2	d m	
			2128	0.02	42.6	54.2	7.4	
	S recovery							
equipment		flow						
number		m3/d	m3/m2/d	scale up	m2	d2	d m	
22th01		1293	1000	0.5	2.6	3.3	1.8	
			flow					
			tpd solids	m2/t/d	m2	d2	d m	
			494	0.02	9.9	12.6	3.5	

pumps

	area	300	concentrate pumping			
equipment	flow	flow	flow		stream #	temperature c
number	m3/h solids	m3/h liquid	m3/h slurry			10
300pu01	13	92	105			
	area	10	concentrate dewatering			
equipment	flow	flow	flow			
number	m3/h solids	m3/h liquid	m3/h slurry			
10pu01	13	92	105			10
	area	11	primary leach			
equipment	flow	flow	flow			
number	m3/h solids	m3/h liquid	m3/h slurry			
11pu01	13	235	248		3	104
11pu02		165	165		11	110
11pu03		215	215		2	110
11pu04	9	10	19		12	110
	area	12	secondary leach			
equipment	flow	flow	flow			
number	m3/h solids	m3/h liquid	m3/h slurry			
12pu01	9	24	34		200&201	70
12pu02	11	23	34		14	120
12pu03	9	41	50		13	90
	area	13	oxidation			
equipment	flow	flow	flow			
number	m3/h solids	m3/h liquid	m3/h slurry			
13pu01		207	207		11&28&111	90
13pu02	4	426	430		24	120
	area	14	hydrolysis			
equipment	flow	flow	flow			
number	m3/h solids	m3/h liquid	m3/h slurry			
14pu01	17	187	203		17	90
14pu02	17	175	192		41	72
14pu03		419	419		39	63
	area	14	hydrolysis			
equipment	flow	flow	flow			
number	m3/h solids	m3/h liquid	m3/h slurry			
14pu01			0			
14pu02			0			
14pu03			0			

# of pumps and stations ?

STARFIELD EQUIPMENT SIZING

6000 tpd plant — By: Nov 19-2011 P.E — REF — Mb

	area	22	S recovery						
equipment	flow	flow	flow						
number	m3/h solids	m3/h liquid	m3/h slurry						
22pu01	9	54	63						
22pu02	8	47	55						
22pu03	8	24	32						
22pu04		23	23						
	area	23	reagents						
equipment	flow	flow	flow						
number	m3/h solids	m3/h liquid	m3/h slurry						
23pu01	3	45	48	lime					
23pu02		26	26	ammonium carbonate					
23pu03		0.00	0	conc HCl					
23pu04		18	18	H2SO4					

filters

10fl/01	area	10	concentrate dewatering			
	flow		leach			
	kg/h solids			kg/m2/hsolids	m2	
	137942			3000	46	
12fl/01	area	12	secondary			
	flow		leach			
	kg/h solids			kg/m2/hsolids	m2	
	36676			1624	23	temperature 120c
12fl/02	area	12	secondary			
	flow		leach			
	kg/h solids			kg/m2/hsolids	m2	
	27429			1624	17	temperature 90c
14fl/01	area	14	hydrolysis			
	flow					
	kg/h solids			kg/m2/hsolids	m2	
	88665			1000	89	high temperature filter 180c
14fl02	area	14	hydrolysis			
	flow					
	kg/h solids			kg/m2/hsolids	m2	
	88665			2000	44	temperature 72c
18fl02	area	18	Cu SX			polishing filter
	flow					
	kg/h solids	m3/h liquid				temperature 59c
	20	164				
20fl02	area	20	Ni SX			polishing filter
	flow					
	kg/h solids	m3/h liquid				temperature 40c
	3	499				
12fl03	area	12	tails sulfur recovery			
	flow		leach			
	kg/h solids			kg/m2/hsolids	m2	
	28872			1624	18	
12fl04	area	12	sulfur recovery			
	flow		leach			
	kg/h solids			kg/m2/hsolids	m2	
	7943			1000	8	
iX/SX						stream #
	area	15	FeCl3			45
	flow					
	m3h	36				
	area	16	Zn iX			50
	flow					
	m3h	543				
	area	17	anionic SX			50
	flow					
	m3h	543				
	area	18	Cu SX			53
	flow					
	m3h	146				
	area	19	Co SX			65
	flow					
	m3h	161				
	area	20	Ni SX			80
	flow					
	m3h	449				

EW									stream #	
	area	18A	Cu EW							
	flow			2.2	volts	4.6	MW			

E

STARFIELD EQUIPMENT SIZING

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	m3h									
	152								56	
	kg/h Cu	gms/amp/day	amps	a/m2	m2	m2/cathode	cathodes	cathodes/cell		cells
	2186	27.89	######	300	6270.49	2	3135	84	59	37
	area	20A	Ni EW							
	flow			2.5	volts	3.5	MW			
	m3h									
	68								83	
	kg/h Ni	gms/amp/day	amps	a/m2	m2	m2/cathode	cathodes	cathodes/cell		cells
	1343	25.74	######	300	4172.54	2	2086	50	86	42
	area	19A	Co EW							
	flow			2.5	volts	0.4	MW			
	m3h									
	4								68	
	kg/h Co	gms/amp/day	amps	a/m2	m2	m2/cathode	cathodes	cathodes/cell		cells
	154	25.85	143265	300	477.552	2	239	30	71	8
	area	21	MnO2 EW							
	flow			3	volts	1.0	MW			
	m3h									
	541								52	
	kg/h Mn	gms/amp/day	amps	a/m2	m2	m2/cathode	cathodes	cathodes/cell		cells
	158	12.04	314947	300	1049.822	2	525	30	75	17

heat exchangers

						stream #	temp c				hx areas
	area	11	HCl condenser								
equipment	flow	flow	flow								
number		tph BFW	tph water vapor					Mcal/h	Mcal/h difference		
11hx01		300		m3/h	in	113	60	10492			
		146	154	263034	out	96	100	101118.5	90627		
	m3h gas	tph HCl/H2O	%v HCl	%v H2O							
	389964	240	38	62	in	94	164				
	76830	240			out	5	102				

						stream #	temp c				
	area	11	HCl condenser								
equipment	flow	flow	flow								
number		tph BFW	tph water vapor					Mcal/h	Mcal/h difference		
11hx02		146	154	m3/h	in	96	100	101064			
		117	182	311632	out	93	100	115620.5	14556		
	m3h gas	tph HCl/H2O	%v HCl	%v H2O							
	389964	47	38	62	in	95	164				
	24564	47			out	183	105				

						stream #	temp c				
	area	14	hydrolysis								
equipment	flow	flow	flow								
number	m3h gas	tph liquid	tph solid	tph gas				Mcal/h	Mcal/h difference		
14hx01		671	23		in	25	171	98382			
	148479			73		184	171	42736.95			
	1322			0.7		19	120	436		MW	
		22				26	102	1717			
heat of reaction								-8000			
	426037			248	out	30	180	103927.8	41568		48
		453	89			31	180	72913			

						stream #	temp c				
	area	13	evaporation 1								
		flow	flow								
25hx01		tph liquid	tph water vapor					Mcal/h	Mcal/h difference		
		277			in	11	110	26776			
		52				28	89				
		1.4				111	43				
			0.7		out	19	120	31709	4934		6
		329				20	120				

						stream #	temp c				
25hx02	area	25	evaporation 2a								
		flow	flow								
		tph liquid	tph water vapor					Mcal/h	Mcal/h difference		
		0.2			in	125	30	131			
		2.9				194	70				
		2.3			out	123	79	593	463		1
			0.8			193	60				

						stream #	temp c				
	area	25	evaporation 2b								
		flow	flow								
		tph liquid	tph water vapor					Mcal/h	Mcal/h difference		
25hx03		2.3			in	124	43	40			
		0.0				192					
			0.9		out	97	43	534	494		1
		1.4				110	43				

						stream #	temp c				
	area	25	evaporation 3								
		flow	flow								
25hx04		tph liquid	tph water vapor					Mcal/h	Mcal/h difference		
		262.9			in	98	60	9200			

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			112.9		out	99	50	69909	60709	71	
		150.0				102	50				

	area	24	HCl regeneration			stream #	temp c					
		flow	flow									
24hx01		tph liquid	tph water vapor					Mcal/h	Mcal/h difference			
		150.0			in	104	60	11461				mcal/h
		33.0				107	25				heat of dilution	6212
				tph HCl vapor								
			15.6	23	out	108	87	22526	11065	13		
		144.6				109	101					

turbines

	area	11		H2S turbine			stream #	temp c
	flow	flow			pressure			
11tb01	tph H2S	tph water vapor		m3/h	bar abs			
	35.1	0.8		26930	1	in	133	30
	35.1	0.8		16830	2	out	134	106

	area	1100		air for H2S combustion			stream #	temp c
	flow	flow	tph H2S		pressure			
1100tb01	air tph	tph water vapor		m3/h	bar abs			
	298.7	2.6	0.10	246283	1	in	132	12
	298.7	2.6	0.10	160357	2	out	131	98

	area	25		evap 3			stream #	temp c
	flow	flow	tph H2S		pressure			
25tb01	air tph	tph water vapor		m3/h	bar abs			
				778	0.1	in	103	30
				147	1	out	122	299

flash tanks

	area	11		H2S			stream #	temp c					
	flow	flow			pressure				retention	volume	height		
11ft01	tph liquid	tph gas		m3/h gas	bar abs				time gas	flash	diameter	diameter	height
		69.0		91504	1	in	8	107		tank	ratio	m	m
									minutes	m3			
		35.9		26930	1	out	133	30	0.083333	37	1	3.6	3.6
	33.1				1		128	30					

	area	11		air for H2S burn			stream #	temp c					
	flow	flow			pressure				retention	volume	height		
11ft02	tph liquid	tph gas		m3/h gas	bar abs				time gas	flash	diameter	diameter	height
	33.1				1	in	128	30		tank	ratio	m	m
		296		254171	1		130	25	minutes	m3			
	30.4				1	out	129	25	0.083333	342	2	6.0	12.0
		299		246283	1		132	12					

	flash	diameter	diameter
	tank	ratio	m
N tanks 3	m3		
	114	3	3.6

	area	25		evap 2A			stream #	temp c					
	flow	flow			pressure				retention	volume	height		
25ft01	tph liquid	tph gas		m3/h gas	bar abs				time gas	flash	diameter	diameter	height
	0.8				0.2	in	126	48		tank	ratio	m	m
									minutes	m3			
	0.8				0.2	out	191	30	0.083333	0	2	0.0	0.0

ignore

	area	25		evap 2B			stream #	temp c					
	flow	flow			pressure				retention	volume	height		
25ft02	tph liquid	tph gas		m3/h gas	bar abs				time gas	flash	diameter	diameter	height
		0.9		58600	0.02	in	97	43		tank	ratio	m	m
									minutes	m3			
						out	29		0.083333	81	2	3.7	7.5
	0.9				0.02		105	23					

	area	25		evap 2C			stream #	temp c					
	flow	flow			pressure				retention	volume	height		
25ft03	tph liquid	tph gas		m3/h gas	bar abs				time gas	flash	diameter	diameter	height
		1.5		1864	1	in	22	120		tank	ratio	m	m
		0.4		98.6	5		38	120	minutes	m3			
	149.6				1		115	25					
	150.0				1	out	116	27	0.083333	2	2	1.0	2.1
		1.5		1223	1		117	27					

	area	25		evap 2D			stream #	temp c					
	flow	flow			pressure				retention	volume	height		
25ft04	tph liquid	tph gas		m3/h gas	bar abs				time gas	flash	diameter	diameter	height
		1.5		1223	1	in	117	27		tank	ratio	m	m
									minutes	m3			
		1.5		1179	1	out	118	20	0.083333	2	2	1.0	2.0
	0.01				1		119	20					

	area	13		oxidation			stream #	temp c

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	flow	flow			pressure				retention	volume	height							
13ft01	tph liquid	tph gas		m3/h gas	bar abs				time gas	flash	diameter	diameter	height			flash	diameter	diameter
	766.7				1	in	24	120		tank	ratio	m	m			tank	ratio	m
									minutes	m3				N tanks	2	m3		
	693.8				1	out	25	171	0.083333	206	2	5.1	10.2			103	2	4.0
		73		148479	1		184	171										

	area	25		evap 3			stream #	temp c										
	flow	flow			pressure				retention	volume	height							
25ft05	tph liquid	tph gas		m3/h gas	bar abs				time gas	flash	diameter	diameter	height			flash	diameter	diameter
		112.9		1681310	0.1	in	99	50		tank	ratio	m	m			tank	ratio	m
									minutes	m3				N tanks	16	m3		
	112.8				0.1	out	100	30	0.083333	2335	2	11.4	22.8			146	3	4.0
		0.1		778	0.1		103	30										

scrubbers

	area	11	primary leach off gas			stream #	temp c
equipment	flow	flow	flow				
number	tph solids	tph liquid	m3/h slurry				
11scr01	138	382	520		in	3	104
	m3h gas	%v H2S	%v HCl	%v H2O			
	95926	47	2	50	in	7	110
	91504	51	0	49	out	8	107
	0.083333	minutes retention time for gas					
	127	m3 scrubber capacity					
	2.5	height to diameter ratio					
	4	m diameter					
	10	m height					

acid plant — area 1100

stream #		133	134	143
		inlet turbine	intoH2Sburn	outlet acid plant
Nm3/h		24263	24263	221744
%v H2S		95.3	95.3	
%v H2O		4.3	4.3	
%v SO2				
%v O2		0.003	0.003	1.3
%v N2		0.007	0.007	97.4
%v CO2				
temp c		30	106	30
pressure bar a		1	2	1

H2SO4 to HCl regen

	146	146	107	107	tpa to tank farm
	t/h	tpa	t/h	tpa	and shipment
93% h2SO4	135	1132975	33	276833.6	856142

stream #		135
S to H2S burner		
tph		7.9

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H2SO4 tank farm area 1150

tpa H2SO4 to and from tank farm		856142		
months/annum shipping		5	SG	m3
tonnes H2SO4 stored		499415.94	1.83	272904.9
tonnes/month H2SO4 shipped		171228.32		

oxygen plant area 1200

			stream #
tpd to oxidation		214.3	21
pressure bar absolute		1	
tpd to secondary leach		70.6	37
pressure bar absolute		5	
tpd total		284.9	

Energy balance

		MW	
turbine 1		12.2	
turbine 2		3.1	
turbine 3		9.4	
turbine 4		27.9	
total		52.5	
power to condensate pumps		0.8	0.8
power to vacuum pumps		0.0	0.0
power to acid plant		9.5	9.5
power for mining		6.0	11.0
power for crushing grinding and flotation		7.4	12.5
Powe for Pipeline			2.5
Power for Process Plant			16.0
power for oxygen plant		3.2	3.2
power for electro winning		10.6	
power for pumping agitation etc		5.3	
total		42.8	55.5
net		9.6	

STARFIELD
ACID STORAGE AT PLANT

BY: E. Burga
Date Nov 3/11

REFERENCES	Production		850,000	tpa			
Reference for acid storage tank dimensions from industrial applications							
	Maximum diameter		50	m			
	High between		8-10	m			
DESIRED STORAGE							
Provision for storage - months		7	495,833.33				
Provision for plant operating safety		25.00%	123,958.33				
			619,791.67				
Design Storage Cap	Agreed		700,000.00	tons			
	sg		1.83				
			382,513.66	m3	From Equip sizing		
	Total Capacity		382,513.66		Corrected in Tonnes	700000	tonnes
D			45	m			
1 H		0.215	9.675				
	DeducT from H		0.6				
	Effective H		9.075				
NUMBER OF TANKS							
	Volume per tank		14,433.16	m3			
					26.50		
	Require for design capacity				27.00		
	TOTAL NUMBER OF TANKS			Enter	***27.00***		
				Actual storage cap		713,142.63	tonns

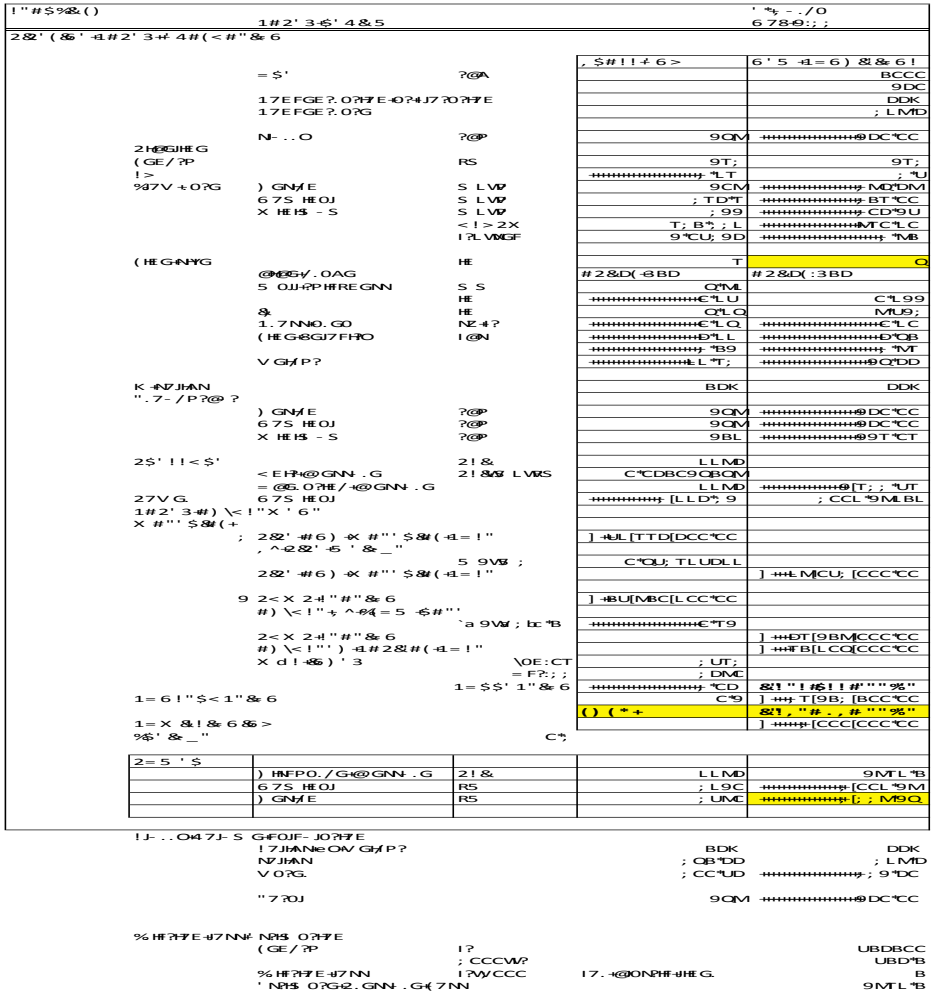
WEIGHT PER TANK

Plate thick	Weight of plate		
	lb/ft2		kg/m2
6.4	10.48		51.28
	4		
25.4	41.92	lb/ft2	205.10
19			153.42

1 ft	0.3048	m
1 sqm	10.76	sq ft

TANK WEIGHT DEF

		sq m		Plate thick	Weight/sqm	Total weight Kg
Cylinder Area	A1	1,367.77		25.40	153.42	209,847
Bottom Area	A2	73.52		25.4	205.10	15,079
Cover	A3	73.52		19	153.42	11,279
	Micell W as per of total		10%			23,620



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