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TECHNICAL REPORT

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REVISED TECHNICAL REPORT

Simberi Gold Project, Simberi Island, Papua New Guinea

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3.0 SUMMARY

3.1 Scope

This revised technical report is required to document the mineral resource estimates, reserve estimates, and processing at the Simberi Gold Project for Allied Gold Limited. It has been prepared in response to the September 10, 2010, announcement of an increase in the oxide and sulphide resources at the Pigiput, Pigibo and Samat deposits on Simberi Island.

3.2 Property Description

The Simberi Gold Project is located on Simberi Island in the Tabar Islands Group. The Tabar Islands are situated in the New Ireland Province of Papua New Guinea (PNG) at approximately latitude 2.5° South and longitude 152° East. The four sparsely inhabited islands of the Tabar Group are located 130 km east of the capital city of New Ireland Province, Kavieng. The Simberi Gold Project is located within Mining Lease ML136 which covers the eastern half of Simberi Island.

3.3 Ownership

Simberi Gold Company Ltd ("Simberi Gold"), a fully owned subsidiary of Allied Gold Pty Ltd ("Allied") operates the Simberi Gold Mine on Simberi Island.

3.4 Geology

Simberi Island is the oldest and northernmost island of the Tabar Island group. It forms part of a silica-poor, potassium-rich alkaline volcanic Pliocene-Pleistocene island arc to the immediate north of New Ireland, PNG.

The currently known gold prospects on Simberi Island are located in the eastern half of the island, within the central volcanic core and are contained within a sub-cropping epithermal alteration system extending 4 km north-south and 2 km east-west. The host rocks for the mineralisation comprise altered andesitic lava flows or intrusives (porphyries), volcanoclastics and tuffs.

3.5 Mineralisation

Gold mineralisation, however, does not appear to be closely associated with any particular lithology. Where recognised, the main primary control of gold mineralisation is steeply dipping fracture systems, in places associated with milled breccia dykes (diatremes). Particularly high grades are associated with diatreme-country rock contact zones. Gold mineralisation is generally associated with sulphides or iron oxides occurring within all variety of hydraulic fractures, such as simple fracture infills, single vein coatings and crackle brecciation in the more competent andesite units and broad disseminations in the naturally porous volcanoclastic rocks.

In the Oxidised zone, the gold is predominantly associated with iron oxides after sulphides, with higher grades occurring with rare vuggy and chalcedonic quartz.

The Sulphide zone mineralisation includes refractory gold hosted by pyrite or marcasite and scarcer arsenopyrite at depth. Trace element analyses indicate the pyrite is arsenian.

3.6 Exploration

The Tabar Island Group has been extensively explored by a number of operators since the discovery of *in situ* gold in 1981. Exploration in ML 136 is focussed around the identified deposits of Sorowar, Pigiput, Pigibo, Botlu, Pigicow, Bekou and Samat. In addition to these, there are a number of smaller prospects yet to be properly defined. Exploration uses a combination of channel sampling, reverse circulation and diamond core drilling.

In addition to ML 136, Allied also hold Exploration Lease 609 covering most of the rest of the Tabar Island Group. EL 609 is also being actively explored.

3.7 Resource

Simberi Resource (0.5 g/t Au cut off – depleted to eom October 2010)

Deposit	Material	Measured			Indicated			Inferred		
		Mt	Au g/t	koz	Mt	Au g/t	koz	Mt	Au g/t	koz
Bekou	Oxide				0.04	1.74	2	0.06	1.14	2
	Transition				0.01	1.17	0	0.05	1.16	2
	Sulphide				0.02	1.93	2	0.92	1.38	41
					0.07	**1.76**	**4**	**1.02**	**1.36**	**45**
Bekou Total					**0.07**	**1.76**	**4**	**1.02**	**1.36**	**45**
Botlu	Oxide	1.22	1.14	44	0.45	1.23	18	0.31	1.16	11
	Sulphide							1.45	1.81	84
		1.22	**1.14**	**44**	**0.45**	**1.23**	**18**	**1.76**	**1.70**	**96**
Botlu Total		**1.22**	**1.14**	**44**	**0.45**	**1.23**	**18**	**1.76**	**1.70**	**96**
Pigibo	Oxide				2.96	1.11	106	0.60	0.89	17
	Transition				2.19	1.19	84	0.51	0.92	15
	Sulphide				3.86	1.11	137	6.22	0.94	188
					9.00	**1.13**	**327**	**7.33**	**0.93**	**220**
Pigibo Total					**9.00**	**1.13**	**327**	**7.33**	**0.93**	**220**
Pigicow	Oxide				0.15	1.65	8	0.29	1.30	12
	Transition							0.11	1.29	4
	Sulphide							2.00	1.26	81
					0.15	**1.65**	**8**	**2.39**	**1.26**	**97**
Pigicow Total					**0.15**	**1.65**	**8**	**2.39**	**1.26**	**97**
Pigiput	Oxide	3.02	0.87	85	4.60	0.92	137	2.01	0.79	51
	Transition				1.95	0.89	56	0.77	0.83	21
	Sulphide				32.56	1.51	1,583	32.25	1.00	1,042
		3.02	**0.87**	**85**	**39.12**	**1.41**	**1,775**	**35.03**	**0.99**	**1,114**
Pigiput Total		**3.02**	**0.87**	**85**	**39.12**	**1.41**	**1,775**	**35.03**	**0.99**	**1,114**
Samat East	Oxide							0.40	1.13	14
	Transition							0.08	0.78	2
	Sulphide							3.50	0.78	88
								3.98	**0.82**	**105**
Samat East Total								**3.98**	**0.82**	**105**
Samat north A	Oxide				0.16	0.75	4	0.11	0.84	3
	Transition				0.04	1.29	2	0.01	1.26	0
	Sulphide				0.36	0.81	9	1.08	0.86	30
					0.56	**0.83**	**15**	**1.20**	**0.86**	**33**
Samat North A					**0.56**	**0.83**	**15**	**1.20**	**0.86**	**33**

Deposit	Material	Measured			Indicated			Inferred		
Total										
Samat North B	Oxide				0.12	0.86	3	0.12	0.75	3
	Transition				0.05	2.86	5	0.02	0.78	1
	Sulphide				1.90	1.22	74	1.05	0.73	25
					2.06	**1.24**	**82**	**1.19**	**0.73**	**28**
Samat North B Total					**2.06**	**1.24**	**82**	**1.19**	**0.73**	**28**
Samat South A	Oxide				0.02	2.11	1	0.16	1.28	7
	Transition				0.01	0.98	0	0.01	0.77	0
	Sulphide				0.02	1.01	1	1.73	0.97	54
					0.05	**1.53**	**2**	**1.90**	**0.99**	**61**
Samat South A Total					**0.05**	**1.53**	**2**	**1.90**	**0.99**	**61**
Samat South B	Oxide				0.05	2.94	5	0.17	1.51	8
	Transition				0.05	2.03	3	0.02	0.99	1
	Sulphide				1.70	1.76	96	3.36	1.07	115
					1.80	**1.80**	**104**	**3.54**	**1.09**	**124**
Samat South B Total					**1.80**	**1.80**	**104**	**3.54**	**1.09**	**124**
Sorowar	Oxide	5.81	1.30	243	8.56	1.08	298	2.40	1.09	84
	Transition	0.54	1.17	20	1.46	1.14	53	0.29	0.83	8
	Sulphide	1.30	0.93	39	6.93	0.92	205	19.03	0.90	549
		7.64	**1.23**	**302**	**16.95**	**1.02**	**556**	**21.72**	**0.92**	**641**
Sorowar Total		**7.64**	**1.23**	**302**	**16.95**	**1.02**	**556**	**21.72**	**0.92**	**641**
Sorowar South	Oxide							0.68	0.82	18
	Transition							0.28	0.68	6
	Sulphide							5.39	0.66	114
								6.35	**0.67**	**138**
Sorowar South Total								**6.35**	**0.67**	**138**
Grand Total		**11.88**	**1.13**	**431**	**70.21**	**1.28**	**2,892**	**87.42**	**0.96**	**2,701**

Rounding may cause numerical discrepancies

3.8 Reserves

Simberi Mineral Reserves and Resources within Design Pits (October 2010).

Deposit	Tonnage (Mt)	Au (g/t)
Sorowar		
Proven	7.17	1.21
Probable	6.66	1.22
Total Mineral Reserves	**13.83**	**1.22**
Inferred Resource	0.49	0.91
Pigiput		
Proven	4.07	0.74
Probable	19.53	1.83
Total Mineral Reserves	**23.60**	**1.64**
Inferred Resource	3.15	1.18
Pigibo		
Proven	0.00	0.00
Probable	5.51	1.14
Total Mineral Reserves	**5.51**	**1.14**
Inferred Resource	0.60	0.77
Samat North A		
Proven	0.00	0.00
Probable	0.13	0.73
Total Mineral Reserves	**0.13**	**0.73**
Inferred Resource	0.02	0.64
Samat North B		
Proven	0.00	0.00
Probable	0.50	2.03
Total Mineral Reserves	**0.50**	**2.03**
Inferred Resource	0.00	0.68
Samat South		
Proven	0.00	0.00
Probable	0.81	2.41
Total Mineral Reserves	**0.81**	**2.41**
Inferred Resource	0.02	1.68
Botlu South		
Proven	0.74	1.35
Probable	0.12	1.61
Total Mineral Reserves	**0.86**	**1.39**
Inferred Resource	0.02	1.40
Total All Pits		
Proven	11.97	1.06
Probable	33.26	1.61

Total Mineral Reserves	**45.24**	**1.46**
Inferred Resource	4.29	1.09

3.9 Mining

The current mine plan consists of the open pit mining of oxide deposits. The Samat oxide pits have been depleted. Ore is currently being extracted from the Sorowar pit.

The Simberi mining operations is a conventional load and haul operation using a mixed fleet of owner and contractor equipment (Table 3-1). Ore from the Sorowar pit is transported to the processing plant by an aerial conveying system. Currently just over 100,000 t of ore per month is being mined

Table 3-1: Simberi Gold Company (SGC) Mining Fleet

Machine	Number
Cat 740 articulated truck[(1)]	8
Cat 725 articulated truck	2
Cat 330 excavator	2
Komatsu PC450 LC-7 excavator	2
Cat D10N dozer	1
Cat D9N dozer	1
Cat D6R dozer	3
Cat 140 grader	1
Cat CS-563E roller	1
Cat IT38G[(2)]	1
Cat 980G FEL[(2)]	2

3.10 Processing

This report is based on the proposed expansion of the oxide processing plant to 3.5 Mtpa and development of a 1.5 Mtpa sulphide processing facility.

The current oxide plant is treating greater than 2 Mtpa at a recovery of greater than 92% gold.

Expansion of the oxide plant is based on providing increased comminution capacity including a SAG mill, and increased leaching and carbon adsorption capacity. Capital cost is estimated at A$32 M for the oxide expansion project.

Metallurgical testwork has aimed at developing a viable flotation, roast, leach flowsheet for the sulphide ore. A preliminary process design has been developed. Preliminary engineering at PFS level has resulted in a capital cost estimate of $188 M for the sulphide plant, and a further $7.6 M for infrastructure associated with the use of Heavy Fuel Oil (HFO) for power supply.

Operating cost for the expanded oxide plant is estimated at $9.25/t ore milled. This is based on the current operating cost adjusted for increased throughput. Operating cost for the sulphide plant has been estimated from first principles at $23.29/t.

NPV of the combined project has been calculated at $334 M. This includes the contribution of the existing oxide project. Cash operating costs are $678/oz.

All dollars are in Australian Dollars (A$) unless otherwise denoted.

3.11 Author's Conclusions

3.11.1 Resources

- The Simberi resource is a robust and proven gold deposit.
- The current resource models show a positive reconciliation in the mined pits. This issue should be investigated with the objective of optimising production and potentially increasing the mine life.
- The historical resources at Botlu is accepted as Inferred based on the Author's validations. The resource models should be re-estimated using the current data set and understanding of the deposits' geology.
- The Sulphide resource at Sorowar needs to be drilled and assessed.
- Beyond the Simberi mining lease, the exploration ground held by Allied over the Island group is all very prospective.

3.11.2 Reserves

- The reserves at Simberi, being based on the resource are also proven and robust.
- Improvements to the reserve position come directly from improvements and increases in the resource.
- The development of some of the satellite deposits will add to the reserve base.
- Inclusion of more Sulphide material has the potential to significantly increase reserves.

3.11.3 Processing

- The current oxide processing plant is operating well and achieving good oxide recoveries.
- The infrastructure that is in place at Simberi Gold is adequate for the 2 Mtpa operation, and with some upgrading including the wharf facility, will be adequate for the 3.5 Mtpa oxide operation and 1.5 Mtpa sulphide operation.
- BatteryLimits is of the opinion that the oxide plant expansion modifications and additions as proposed by GRES will be effective in increasing oxide plant throughput to 3.5 Mtpa.
- The capital cost estimate of $32 M for the oxide plant expansion to 3.5 Mtpa is reasonable for the modifications and additions proposed.
- The capital estimate of $188 M for the 1.5 Mtpa sulphide plant design and construction as estimated by GRES is reasonable.
- The gold grades to be fed to the oxide process plant are low at 1.1 g/t average over the mine life, and maintaining low overhead and operating costs will be critical to project viability.
- The combined 3.5 Mtpa oxide project and 1.5 Mtpa sulphide project is commercially viable with NPV of $334 M (at 10% discount factor) pre-tax, using US$1,000/oz gold price and A$:US$ exchange rate of 0.8. The NPV is sensitive to gold price.

3.12 Recommendations

- The remaining historical resources should be re-estimated incorporating the current geological understanding of the island, recent drilling and other geological data and any updated topographical information. Outsourced this is estimated to cost AUD$50 K.

- The positive mining reconciliation should be investigated. This could be done in-house or outsourced at an estimated cost of AUD$30 K.
- A critical review of current mining operations should be undertaken and any revisions applied to the optimisation, pit design and scheduling. This could be done in-house or outsourced at an estimated cost of AUD$40 K.
- Optimisation of the resource and reserve calculations, final pit designs and scheduling should be revisited on a regular basis. The pit optimisations need to be updated to reflect more recent increasing trend in the gold price.
- Exploration drilling to investigate the nature of the sulphide mineralisation beneath the current Sorowar pit design should proceed as a matter of priority. There appears to be significant potential to increase the sulphide resource and reserve base at Simberi, and thus improve the overall economics of the project, through improving the understanding of the sulphide mineralisation in the Sorowar area.
- Expansion of the oxide plant to 3.5 Mtpa should proceed. Estimated cost is $32 M.
- Allied Gold should consider a Bankable Feasibility Study, in conjunction with additional drilling, that will provide greater confidence to the combined oxide and sulphide project. The cost of a Bankable Study is estimated at $6-$10 M. The total cost of the combined oxide and sulphide project is estimated at $278 M including the $32 M oxide plant expansion.
- Allied should closely manage its operating costs to keep its costs within budget estimates.

4.0 INTRODUCTION AND TERMS OF REFERENCE

Golder Associates Pty Ltd (Golder) has been retained by Allied Gold Limited (Allied) to prepare an independent technical report on Allied's Simberi Gold Project in Papua New Guinea (PNG). This report is prepared to conform to National Instrument 43-101 and Form 43-101F1. The technical report is required to document the mineral resource estimates, mineral reserve estimates, metallurgy and processing at the Simberi Gold Project.

The Simberi Gold Project is part of Allied's PNG holdings and is located on Simberi Island approximately 500 km north-east of the PNG mainland. The prospect has been systematically explored by a number of parties since 1981 and has been wholly controlled by Allied since 2005.

John Battista, Associate, Mining Engineer, with Golder Associates, visited Simberi Island between 28 October 2008 and 31 October 2008.

Phil Hearse, Managing Director, with Battery Limits Pty Ltd, visited Simberi Island between 17 and 19 February 2009, and 7 and 12 June 2009.

Table 4-1: Glossary of Terms

AC	Air Core
Accuracy	The ability to obtain the correct result
ALS	ALS Laboratory Group, ALS Chemex is the groups Mineral Division
ASX	Australian Stock Exchange
BFS	Bankable Feasibility Study
BHP	Originally Broken Hill Proprietary Limited, now BHP Billiton
Blank	Sample without metal content to check possible contamination during assaying (e.g. crushed glass)
Cut off	Grade above which mineralised material is considered to be ore.
CRA	Originally Conzinc Rio Tinto Australia now CRA
DataShed	Database Management software – www.datashed.com.au
DD	Diamond Drill
DEC	Department of Environment and Conservation (PNG)
DTM	Digital terrain model – Electronic computer model of topography
Duplicate	Sample that has been split from another to check the field sampling or laboratory's precision
EOM	End Of Month
HQ	Diamond core 63.5 mm
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
IP	Induced Polarisation – geophysical exploration technique
JORC	Australasian Joint Ore Reserves Committee
Kriging	Grade estimation technique incorporating variability by distance
Leapfrog	3D drill data analysis and viewing software
MRA	Mineral Resources Authority
ML	Mining Lease
Multiple Indicator Kriging	Estimation of grades into block model using probabilistic grade estimation techniques incorporating variability by distance
NATA	National Association of Testing Authorities, Australia
NQ	Diamond core 47.6 mm
Ordinary Kriging	Estimation of grades into block model using a grade estimation technique incorporating variability by distance
Ore	Mineralised material that can be economically mined
PNG	Papua New Guinea
ppb	Parts Per Billion
ppm	Parts Per Million – 10,000 ppm = 1%
PQ	Diamond core 85.0 mm
Precision	The ability to obtain the same result each time
RAB	Reverse Air Blast
RC	Reverse Circulation
SEM	Scanning Electron Microscope
Standard Sample	Specially prepared sample whose metal grade is very accurately known and certified
Strip Ratio	Ratio of waste that needs to be mined to obtain a unit of ore expressed as tonnes of waste to tonnes of ore.

AC	Air Core
Tailings	The reject material from the processing plant
TIG	Tabar Island Grid
TSX	Toronto Stock Exchange
Variogram	Mathematical and graphical way of representing variation of data as a function of separation distance
Vulcan	Computer program by Maptek that is used to carry out resource estimation and mine planning – www.vulcan3D.com.
XRD	X-Ray Diffraction analytical technique.

5.0 RELIANCE ON OTHER EXPERTS

This report has been compiled by Golder Associates Pty Ltd (Golder) with contributions from Battery Limits Pty Ltd (Battery Limits) for Allied Gold Limited (Allied). The information, interpretations, conclusions, opinions, and recommendations contained herein are based upon:

- Information available to Golder and Battery Limits at the time of preparation of this report
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and opinions supplied by Allied and other third party sources are listed as references.

6.0 PROPERTY DESCRIPTION AND LOCATION

6.1 Area and Location

Allied Gold's Simberi Gold Project is located on Simberi Island in the Tabar Islands Group. The Tabar Islands are situated in the New Ireland Province of Papua New Guinea (PNG) at approximately latitude 2.5° South and longitude 152° East. The four sparsely inhabited islands of the Tabar Group are located 130 km east of the capital city of New Ireland Province, Kavieng, and 80 km north-west of the Lihir Island site of the world class Ladolan gold deposit. The southern-most island in the group, Tabar Island, lies approximately 30 km north of mainland New Ireland. Simberi Island is the northern-most island of the Tabar Group and measures approximately 10 km east-west and 8 km north-south.

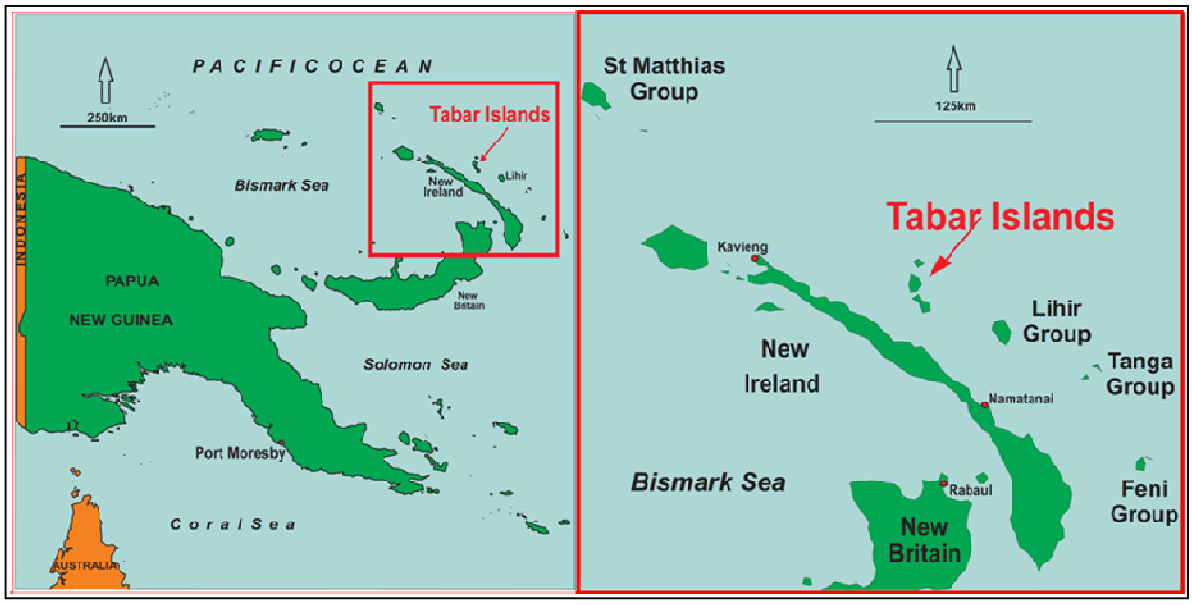

Figure 6-1: Figure 6 1: Simberi Gold Project – Location

6.2 Title

The Simberi Gold Project is operated by The Simberi Gold Company, a fully owned subsidiary of Nord Pacific Limited which is, in turn, a fully owned subsidiary of Allied Gold Limited.

The Simberi Gold Project comprises:

- an open-pit mining operation with an associated gold processing plant, located within the 2,560 ha Mining Lease ML136 on the eastern side of Simberi Island, and
- a larger 69 graticular sub-block/233 km^2 Exploration Licence, EL609, covering the remainder of Simberi Island and most of the adjacent Tatau and Big Tabar Islands to the south.

Mining Leases (MLs) in PNG are issued by the Mineral Resources Authority (MRA) on behalf of the National Government. The ML conditions address surface rights such as lost land, trees, vegetation and surface water with compensation due to the lease owner. Alluvial gold rights belong to the citizens of PNG and the landowner.

The lease was conferred on 3 December 1996 for a term of 12 years. On 4 May 2007 the MRA granted The Simberi Gold Company an extension on ML 136 for a term of ten years commencing 3 December 2008 and ending 2 December 2018.

The Simberi Gold Project oxide processing plant has an annual throughput rate of 2 Mtpa with a feasibility study planned to look at increasing this to 3.5 Mtpa.

Exploration for gold within the ML136 is focused on the replacement of oxide resources, hosted by volcanic rocks and investigation of underlying sulphide mineralisation.

The lease is subject to the following conditions:

- The Lessee must follow an approved proposal. A proposal has been submitted and approved.
- The Lessee must comply with the Mining (Safety) Act.

- The Lessee must comply with conditions imposed by the Department of Environment and Conservation and by conditions set out by the Bureau of Water Resources.
- The Lessee must provide the Department of Mining and Petroleum, now the MRA, with six monthly reports on exploration and monthly production figures.
- The Lessee may not use the land without the consent of the State for other purposes than for which the Lease was granted.
- The Lessee must not interfere with the cultural use of the land without permission and will accommodate traditional uses subject to efficient and safe mining practices.
- The Lessee shall compensate the owners of private land which is located within the Mining Lease to a level at least within accordance of the Act.
- The Lessee shall provide the Department of Mining and Petroleum with a Closure Plan and Schedule at least one year before shutdown.
- The Lessee shall stockpile all topsoil, where practical, to be used for re-vegetation purposes.
- The Lessee shall submit the open pit mine plan to the Chief Inspector of Mines six weeks before start of mining operations.
- The Lessee shall submit to the Chief Inspector of Mines all mine plant plans and details, for the mine construction phase and later.

The conditions of the Approved Mining Proposal were:

- That gold production should have commenced by 31 December 1998. This date was extended to 31 December 2006.
- The Lessee shall provide an alternative water supply to any village whose normal water supply is impacted by the development. This condition has been addressed by the installation of rainwater tanks, piped spring water and bores installed with hand pumps in a number of the affected areas.
- The Lessee shall maintain all drainage channels from the mining areas to minimise flood impacts on village areas.

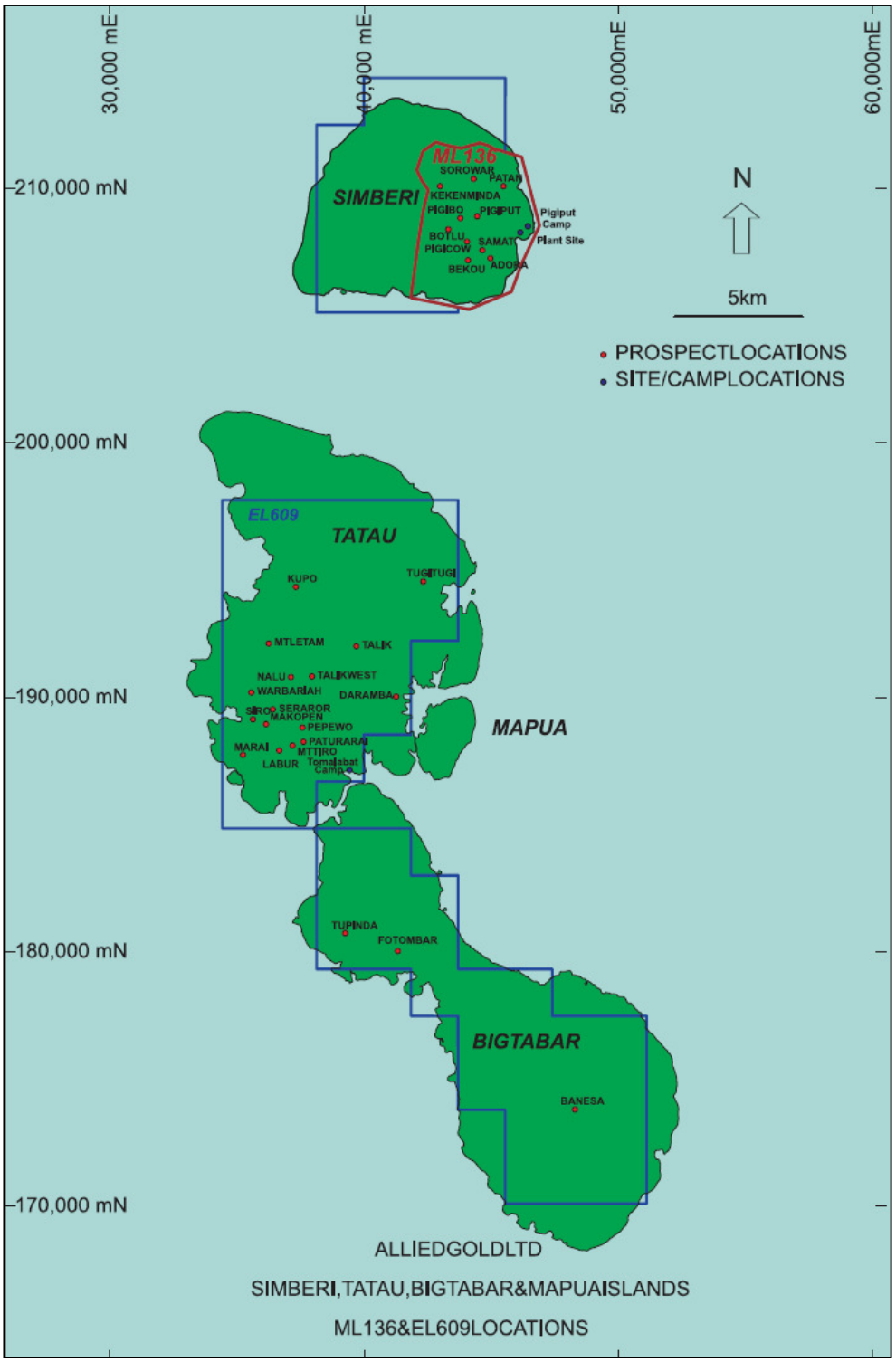

Figure 6-2: Simberi Gold Project – Mining Licences with Principal Prospects and Deposits, shown with local metric grid coordinates (TIG)

6.3 Property Boundaries

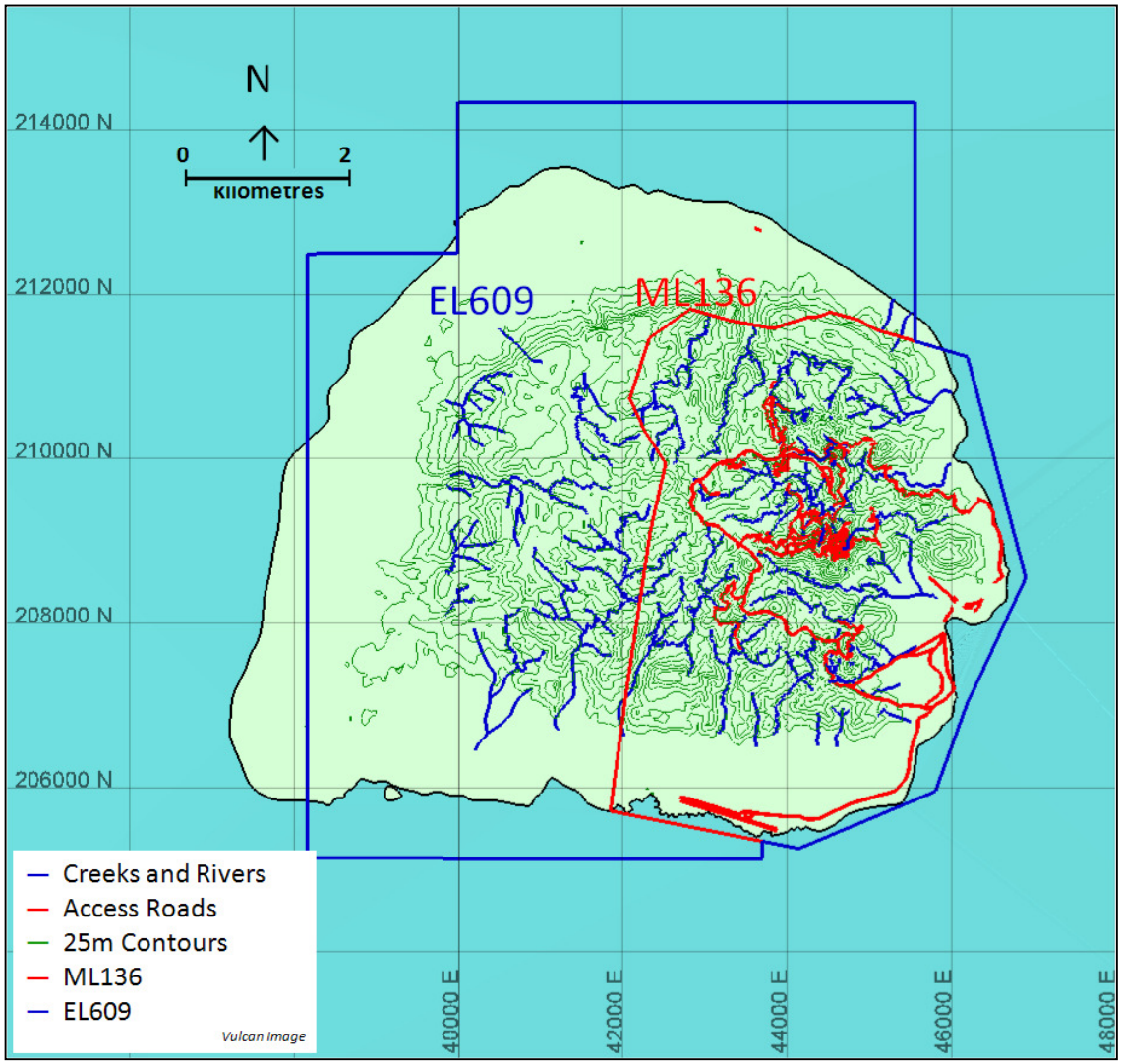

Figure 6-3: Simberi Island

The bounding points of ML136 were surveyed and pegged by Palanga Surveyors (Rabaul) in consultation and agreement of landowners. The limits of EL609 coincide with the boundaries of selected blocks within the MRA's pre-defined graticule system.

Table 6-1: ML136 – Survey

Direction	Latitude	Longitude	TIG_East_m	TIG_North_m
a line starting from	2° 39' 41.5771"S	151° 58' 59.5071"E	41838.1	205690.2
then to	2° 38' 30.9722"S	151° 59' 08.8972"E	42128.3	207859.0
then to	2° 37' 50.7484"S	151° 59' 15.2380"E	42324.1	209094.6
then to	2° 37' 23.6744"S	151° 59' 21.5694"E	42519.8	209926.1
then to	2° 37' 11.3034"S	151° 59' 13.2708"E	42263.4	210306.1
then to	2° 36' 58.2258"S	151° 59' 07.2152"E	42076.4	210707.8
then to	2° 36' 34.0965"S	151° 59' 15.0260"E	42317.6	211449.0
then to	2° 36' 22.5467"S	151° 59' 31.2811"E	42819.8	211803.7
then to	2° 36' 26.9386"S	151° 59' 46.6216"E	43293.7	211668.8
then to	2° 36' 30.2907"S	152° 00' 03.9012"E	43827.5	211565.9
then to	2° 36' 26.2656"S	152° 00' 16.6492"E	44221.3	211689.5
then to	2° 36' 25.5196"S	152° 00' 21.0398"E	44356.8	211712.4
then to	2° 36' 24.4487"S	152° 00' 24.6617"E	44468.7	211745.3
then to	2° 36' 23.4575"S	152° 00' 26.4053"E	44522.6	211775.7
then to	2° 36' 26.7222"S	152° 00' 37.7219"E	44872.2	211675.4
then to	2° 36' 30.6035"S	152° 00' 44.5937"E	45085.9	211556.2
then to	2° 36' 41.4078"S	152° 01' 20.6950"E	46199.7	211224.3
then to	2° 38' 09.3200"S	152° 01' 43.3186"E	46898.5	208524.0
then to	2° 38' 57.9380"S	152° 01' 20.6165"E	46197.0	207030.5
then to	2° 39' 34.1136"S	152° 01' 07.5247"E	45792.6	205919.3
then to	2° 39' 56.3371"S	152° 00' 13.8167"E	44133.6	205236.8
then to	2° 39' 41.5771"S	151° 58' 59.5071"E	41838.1	205690.2

There is no requirement for boundary markers on the ground in respect of exploration licences, however, concrete survey markers have been installed. Location of boundaries when required is done using Global Positioning Equipment (GPS). In applying for an ML or SML all lease boundaries must be clearly marked out in accordance with the Mining Act.

6.4 Location of Mineralisation and Mine Workings

All identified mineralisation and mine workings on Simberi Island are located within ML 136 on the eastern side of the island (Figure 6-3).

6.5 Royalties and Encumbrances

A Memorandum of Agreement dated 21 November 1996 details the relationship between the National Government, Provincial Government, Simberi Landowners Association, Tabar Community Government and Simberi Gold Company. Normally the holder of a Mining Lease must pay the PNG government a 2% royalty on the free-on-board (FOB) value of the product if exported without smelting or refining. However, under the Memorandum of Agreement (MOA) all of the royalty is being returned to the landowners whereby 60% goes to Simberi Island, 15% each goes to Tabar and Tatau islanders and 10% to the Central New Ireland Local Level Government. The New Ireland Provincial Government has the right under the MOA to review the royalty distribution if the gold production exceeds 100,000 ounces in any one or more years. A 0.25% mining levis based on gross revenue is paid to the MRA .

6.6 Environmental Liabilities

Historically, there has been no large scale mining and the previous alluvial workings have had no significant impact. There are no pre-existing environmental liabilities.

Prior to construction Simberi must implement an Environmental Management and Monitoring Program (EMMP). Nord submitted a draft EMMP in May 1999 and an amendment to the EMMP (now known as the EMP) addressing changes to the previous scope, prepared by Nord's consultants, was submitted in July 2003.

A baseline environmental survey was undertaken in June 2003 and two further supplementary environmental baseline studies were completed in March 2004 and December 2004. During the environmental baseline studies, a network of monitoring stations was established to support the ongoing collection of data.

The 2005 feasibility study addresses the environmental impacts associated with:

- open cut mining operations and haul roads
- ore processing operations
- pipeline and power line corridors
- deep sea tailings placement, and
- project infrastructure.

The mitigating engineering measures incorporated into the designs are also described.

The project has the following objectives with respect to the environment:

6.6.1 Waste Dumps

Elimination of all waste dumps through adoption of bulk mining techniques at the Samat, Botlu and Pigiput deposits. Waste generation at the Sorowar and Pigibo deposits will be placed within the mining voids. The recent reserve increase has also increased waste tonnages and this additional waste will be stored in land sited waste repositories.

6.6.2 Open Pits

Progressive rehabilitation of the pits to minimise soil erosion and complete re-vegetation.

6.6.3 Pipelines

Provision of engineering safeguards in the design of pipelines.

6.6.4 Access/Haul Roads

Minimisation of soil erosion and dust generation.

6.6.5 Process Plant

Minimise dust generation from crusher and conveyors, containment of chemicals and reagents, comply with air quality and water quality regulations.

6.6.6 Stormwater

Stormwater management of open pits, mine infrastructure and processing plant, minimisation of soil erosion, collection of runoff for use in processing in preference to other supplies, installation of silt curtains at mouth of streams most likely to be affected by the project where practical installation and operation can be obtained.

6.6.7 Deep Sea Tailings

Provision of a proven and accepted tailings disposal method, conservative design parameters and compliance with environmental permit conditions.

A letter on the 24 February 2004 from the Acting Secretary for the Department of Environment & Conservation confirms that the Environmental Plan Approval for the Simberi Oxide Gold Project that was issued on 30 December 1996 under the Environmental Planning Act 1978 is valid and deemed to be an Environmental (Waste Discharge) Permit for the purposes of the Environment Act 2000.

It also notes that the Department of Environment and Conservation is processing the amendment application which was submitted due to the changes in the mine plan and engineering concept.

A further update to the EMP was prepared and submitted in conjunction with the Optimised Feasibility Study and is under consideration by the Department of Environment and Conservation.

6.7 Required Permits

To commence mining in Papua New Guinea the operator needs a granted Mining Lease and an Environmental Permit.

Allied hold water extraction permit WE-L3 (15), issued and commencing 30 December 1996 and expiring 31 December 2053, allowing them to extract water from sources within ML 136 for mining purposes.

Allied also hold Mining Waste Discharge permit WD-L3(36), issued and commencing 30 December 1996 and expiring 31 December 2053.

As part of this process Allied was required to submit a waste management plan. This plan covers the management and disposal of domestic and industrial wastes.

6.8 Surface Rights

From the PNG Mining Act (1992) the granting of ML 136 to Allied confers the following rights:

1) A Mining Lease authorises the holder, in accordance with the Mining (Safety) Act (Chapter l95A) and any conditions to which the mining lease is subject, to;

   a) enter and occupy the land over which the mining lease was granted for the Purpose of mining the minerals on that land and carry on such operations and undertake such works as may be necessary or expedient for that purpose, and

   b) construct a treatment plant on that land and treat any mineral derived from mining operations, whether on that land or elsewhere, and construct any other facilities required for treatment including waste dumps and tailings dams, and

   c) take and remove rock, earth, soil and minerals from the land, with or without treatment, and

   d) take and divert water situated on or flowing through such land and use it for any purpose necessary for his mining or treatment operations subject to and in accordance with the Water Resources Act (Chapter 205), and

   e) do all other things necessary or expedient for the undertaking of mining or treatment operations on that land.

2) Subject to this Act, the holder of a mining lease:

   a) is entitled to the exclusive occupancy for mining and mining purposes of the land in respect of which the mining lease was granted, and

   b) owns all minerals lawfully mined from that land.

The granting of EL 609 to Allied confers the following rights:

1) An exploration licence authorises the holder, in accordance with any conditions to which it may be subject, to:

   a) enter and occupy the land which comprises the exploration licence for the purpose of carrying out exploration for minerals on that land

   b) subject to Section 162, extract, remove and dispose of such quantity of rock earth, soil or minerals as may be permitted by the approved program

   c) take and divert water situated on or flowing through such land and use it for any purpose necessary for his exploration activities subject to and in accordance with the provisions of the Water Resources Act (Chapter 205), and

   d) do all other things necessary or expedient for the undertaking of exploration on the land.

2) The holder of an exploration licence is entitled to the exclusive occupancy for exploration purposes of the land in respect of which the exploration licence was granted.

7.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

7.1 Access

Site personnel work rotating rosters and, as Simberi is not serviced by regular passenger services, charter flights are used to transport company personnel from various locations in PNG, including Port Moresby, Rabaul and Kavieng. A company-built airstrip, located at the southern end of Simberi Island on Company owned land called Pikung Plantation, can accommodate aircraft up to the size of a Bombardier Dash 8 (18.6t, 56 seat twin turboprop).

The company also operates a loading barge, the Lady Geraldine, ferrying bulk supplies to the island on a regular schedule from the mainland towns of Lae in Morobe Province and Rabaul in the New Britain Province. The Lady Geraldine's capacity is supplemented, as required, by privately owned barges chartered from Lae and Rabaul.

The wharf, processing plant, offices and camp accommodation are located at Pigiput Bay on Simberi Island. Material is brought to the process plant by either 40 t articulated dump trucks along the haul road or a suspended aerial conveying system. An island ring road provides access to all parts of the island and the many hamlets inhabited by the islanders along the coast.

7.2 Physiography

The Tabar Group islands are fringed by coral reefs and, with the exception of low-lying and sandy Mapua Island, have rugged and hilly interiors typically up to 300 m in elevation on Simberi Island and up to 600 m on Big Tabar Island. Flat coastal areas may extend up to several kilometres inland.

Simberi Island, where the know gold deposits occur, is approximately 9 km in diameter and comprises a central rugged volcanic core of Pliocene age which rises to over 300 m in height. The volcanic core is flanked by Pliocene to Pleistocene raised limestone reefs. The reefs are surrounded by Quaternary deposits including lower level raised coral reefs, alluvium and present day fringing coral reefs.

The central part of Simberi Island is deeply dissected, a result of the prevailing high rainfall tropical climate, giving rise to sharp ridges and incised valleys. Features such as volcanic craters or caldera structures cannot be pinpointed with any degree of certainty.

The raised limestone reefs form a broken ring of bluffs around the outermost flanks of the central volcanic terrain. These limestone bluffs are topographic highs that rise to elevations of between 180 m to 210 m. The more recent Quaternary reef complexes and alluvial sediments form a flat lying coastal strip.

The vegetation of the Tabar Group of islands varies greatly. Coconut plantations grow notably on Simberi Island and village gardens are tended along the low-lying coastal fringe. Secondary and tertiary forest growth generally covers the hinterland and dense primary rainforest is more restricted to the interior.

7.3 Climate

The Tabar Group is located approximately 2.5° south of the equator, making for a generally humid and wet environment. The islands, however, are not affected by tropical cyclones, being well north of the Southern Cyclone Belt that extends from 10°S to 20°S. Rainfall shows no real seasonal pattern and is distributed throughout the year with a not so distinct wetter period from October through to April. Mean annual rainfall is about 3,000 mm, with monthly totals typically varying from about 30 mm to 600 mm. This range is consistent with data collected at Lihir Island, 80 km to the south-east. There is a no well defined relationship between elevation however rainfall can vary from the coast to the interior. Close to sea level, the winds are generally light, with monthly wind speed usually less than 5 knots. Two wind seasons occur during the year, and vary in duration from year to year. Generally, prevailing winds are from the south-east from May to October, while from November to April, they are mainly from the north-west.

7.4 People and Infrastructure

At the 1990 census, the population of Tabar Island as of the 1990 Census was 660 with 1,415 people living on the Tatau-Mapua Islands. A census of Simberi, in the early part of the 21st century, showed 1,093 inhabitants. The Tabar people speak a common language, Mandara, and share the same cultural rites and beliefs. Many also speak English and Pidgin, the preferred languages for business dealings with Allied. Approximately 280 local people are employed directly by the mining operation making up just over 57% of the total workforce. The remainder of the workforce is made up Australian expatriates (4%) and non-local Papua New Guineans (39%). In addition there are approximately 158 contractor personnel engaged made up in similar proportions of expatriates, non-local and local Papua New Guineans as the mine breakup.

The people are grouped under clans and sub-clans with intermarriage generally forbidden within the same clan. The clans in the Tabar Group are matrilineal, in which the children of a married couple belong to the mother's clan and look to their mother's people, (particularly her brothers) for ultimate support and guidance. Under a matrilineal inheritance system, maternal uncles pass on land rights to their sisters' children. Land can also be "purchased" in the traditional sense with "shell-money" or "cash-money".

The Tabar Islands are accessible by sea and air and therefore have encountered a great deal of external economic, political, social and cultural influence since the turn of the century. Outside influences have included missionaries and planters as well as German, Japanese, Australian and PNG government administrations.

The coastal strips fringing the islands support the villages' needs. The villagers subsist mainly on food grown in their gardens, with sweet potato being the most popular crop, supplemented by tapioca, banana, taro, coconut and yam. Food gardens are now extending further inland and on cleared slopes of mountains as the coastal strips become increasingly crowded with homes and established plantings of trees such as coconut, betel nuts, traditional nut trees, breadfruit, mango and other fruit trees. Pigs play an extremely important part in the Tabar-peoples' culture as a medium for exchange, gifts, compensation-settlements and for ceremonial feasts.

With the exception of several small portions of land purchased by locals and two plantation leases held by the Allied, most of Simberi Island is held as communal land, owned by various clans. The mineral deposits relevant to this report occur on this communal land. Compensation agreements have been negotiated with each clan within the proposed mining area for loss and disruption caused by mining activities. Title to the

deposits and mineralisation rests with the National Government, which then grants rights to development through the issue of mining leases.

The two long term plantation leases, referred to as Pigiput and Pikung Plantations, are located within ML 136 on Simberi Island (Figure 6-3). Both leases are fully controlled by Allied and valid for 99 years from the grant date (Pigiput, 13th June 1985 and Pikung, 1st January 1938). This land is not subject to communal claims from landowners.

The wharf, processing plant, offices and camp accommodation are all located within the 203 ha Pigiput Plantation Lease and the project's airstrip is located fully within the limits of the 140 ha Pikung Plantation lease.

8.0 HISTORY

There is evidence of very limited alluvial gold workings, probably dating from the 1920s, to the north of Sorowar in Matanabol Creek and at Tugi Tugi on Tatau Island.

Prior to systematic exploration being applied in 1981, earlier exploration consisted of reconnaissance surveys searching for copper by Conzinc Rio Tinto and Broken Hill Propriety Ltd.

Nord Pacific Ltd ("Nord") acquired an exploration licence over the Tabar Islands Group in 1981 at which time modern reconnaissance exploration on Simberi Island indentified gold mineralisation. In 1982 a joint venture was established between Kennecott Exploration (Kennecott), Nord Resources and Niugini Mining Ltd, with Kennecott as the operator. The Tabar Joint Venture, Kennecott Explorations (Australia) Ltd (61.6%), Nord (30%) and Niugini Mining Ltd (8.4%), was formed in 1982 to explore for gold mineralisation in Prospecting Authority PA 609-1, on the Tabar Island Group.

Reconnaissance geochemical exploration and geological mapping by Kennecott identified anomalous gold over a large area of eastern Simberi Island. Within this area, there were several prospects with evidence of higher grade and coherent mineralisation. These prospects were systematically aircore and diamond drilled between 1983 and 1990, initially testing both oxidised and sulphidic gold mineralisation, but later concentrating on the oxide material with a view to testing the viability of a low-cost open-cut mining operation.

In 1993, Nord re-acquired the interests of both Kennecott and Niugini Mining. Subsequently Nord continued exploration and in 1996 commissioned a feasibility study. The study demonstrated favourable project economics and culminated in the grant of a Mining Lease (ML 136) to Nord Pacific in December 1996. This Study included a detailed review of deposit geology and mineralisation, data acquisition methods, data validation and Mineral Resource estimation methodology. Resource estimates of the Sorowar, Pigiput, Samat South, Samat North and Samat East deposits were reported. The Project became uneconomic with the decline in the gold price in early 1997.

In 1997, an extensive drilling program tested extensions to Sorowar and two additional deposits, Botlu and Pigibo. In 1998, the oxide Resources at Sorowar were re-estimated using the larger drill database and new estimates were made for the Botlu and Pigibo deposits (see Section 19.1). The program was successful in almost doubling known resources but a desktop update of the feasibility study carried out in 1999 indicated that the low gold price still precluded development.

In 2002, Nord Pacific entered into two joint ventures with PGM Ventures Corporation (PGM), a Canadian public company. The first venture was known as the Simberi Mining Joint Venture (SMJV) and covered development of the gold resources within ML 136. The second was Tabar Exploration Joint Venture and covered exploration within Exploration Licence 609 over the remaining areas in the Tabar Group.

The Simberi Oxide Gold Project is wholly contained within ML 136, which was issued in December 1996 to Simberi Gold Company Limited, a wholly owned PNG subsidiary of Nord Pacific Limited. In January 2003, PGM announced an agreement to vend its interest in the Simberi Project into another Canadian public

company, Alive International Inc. This transaction resulted in Alive International becoming a 60% owned subsidiary of PGM.

On 15th November 2004, Allied Gold Limited entered into an agreement with Simberi Gold Corporation ("SGC") to acquire its interests in the Tabar Islands Gold Project. Under the terms of the Agreement, Allied Gold moved to an 87.5% interest (ML136) in the Simberi Mining Joint Venture, which was joint venture between Nord Pacific and SGC, and a 100% interest in EL609 which covers the remainder of the Tabar Island Group. In October 2005, agreement was reached with SGC for Allied Gold to purchase the remaining 12.5% free carried interest in the Simberi Oxide Gold Project. Accordingly, Allied Gold now controls a 100% interest in ML136.

Allied also currently holds title to the 233 km^2 Exploration Licence EL609 comprising three blocks A, B and C, illustrated in Figure 6-3. EL609 Block A covers the remainder of Simberi Island not included in ML136. Blocks B and C cover the islands of Tatar and Tabar respectively. Under the PNG Mining Act (1992), an Exploration Licence may be granted for up to two years and is renewable, subject to the holder reporting on and satisfying prescribed expenditure requirements on approved exploration activities. Documentation for the renewal of Exploration EL609 is currently under review by the PNG Mining Advisory Board.

EL609 Blocks B and C were subject to a Letter of Intent, signed with Barrick (PNG) Exploration Ltd in March 2008, through till 28 February 2010 during which time Barrick (PNG) managed all exploration activity. Allied negotiated a deal allowing Barrick (PNG)'s withdrawal from the terms of the Letter of Intent and has since reassumed management of exploration in the entire area of permit EL609. Allied made an immediate $AUD2.5M payment with a further $A3M to be made in July 2010 to Barrick (PNG) as either cash or Allied shares. Barrick agreed to place its holding of Allied in escrow until 2012.

Table 8-1: History of Exploration Activity on ML136

Year	Activity
1930s	Small-scale alluvial workings in Matanabol, Darum & Waturu Creeks north of Sorowar Figure 6-3.
1960s	CRA (formerly Conzinc Rio Tinto of Australia Limited) undertook reconnaissance for porphyry copper and bauxite.
1969	To 1974, regional mapping by the Bureau of Mineral Resources
1973	BHP reconnaissance for porphyry copper
1981	Nord Pacific Ltd granted PA451
1982	Joint venture formed: Kennecott Explorations Australia Ltd (61.6%), Nord (30.0%), Niugini Mining Limited (8.4%). PA451 converted to PA609
1993	Nord re-acquired all Kennecott and Niugini Mining interests in PA451, after the latter elected to focus on Lihir. PA609 converted to EL609
1996	Nord commissioned feasibility study. ML136 excised from EL609 and granted in December
1997	Drilling campaign almost doubled resources
1998	Project put in care and maintenance due to falling gold price
2001	Simberi Mining Joint Venture between Nord Pacific and PGM Ventures
2003	Lycopodium Pty Ltd updated the 1996 feasibility study
2004	Allied Gold Ltd commenced acquisition of interests in ML136 and EL609
2005	Allied Gold Ltd acquired 100% interest in ML136 and EL609. Optimised Feasibility Study completed by Lycopodium Engineering
2006	Allied Gold Limited started mine & mill construction.
2008	Mine production began and mill commissioned.
Mar 2008	Blocks B and C of EL609 joint ventured to Barrick
Jul 2009	Simberi Oxide Plant production achieved total 100,000oz gold

Year	Activity
Jul 2009	Simberi Reserves exceed 1Moz (pre-mining Proven & Probable 23.9Mt @ 1.36g/t Au for 1.04Moz gold)
Feb 2010	Allied Gold assumes full control of EL609 after negotiating Barrack (PNG) withdrawal
Mar 2010	Simberi pre-mining MEAS + IND resources total 3.2Moz gold and INF resources total 2.5Moz (78.4Mt @ 1.27 g/t Au for 3.20 Moz of gold, with additional Inferred 78.2Mt @ 0.99 g/t Au for 2.49 Moz of gold)
Sep 2010	Completion of Sulphide Project pre-feasibility study. Reserve increased to 1.31 Moz to 2.15 Moz

9.0 GEOLOGICAL SETTING

9.1 Regional Geology

Simberi Island is the oldest and northernmost island of the Tabar Island group. It forms part of a silica-poor, potassium-rich alkaline volcanic Pliocene-Pleistocene island arc to the immediate north of New Ireland, PNG. This island arc includes the Lihir, Feni and Tanga Island groups that lie to the south-east of the Tabar Group.

The island arc was distorted by a complex underlying subduction system causing the islands to form along north-south tension gashes (Figure 9-1). It is inferred that these deep-seated structures have controlled the location of porphyry intrusives derived from re-melting of the oceanic crust following reversal of subduction coupled with epithermal alteration and associated mineralisation .

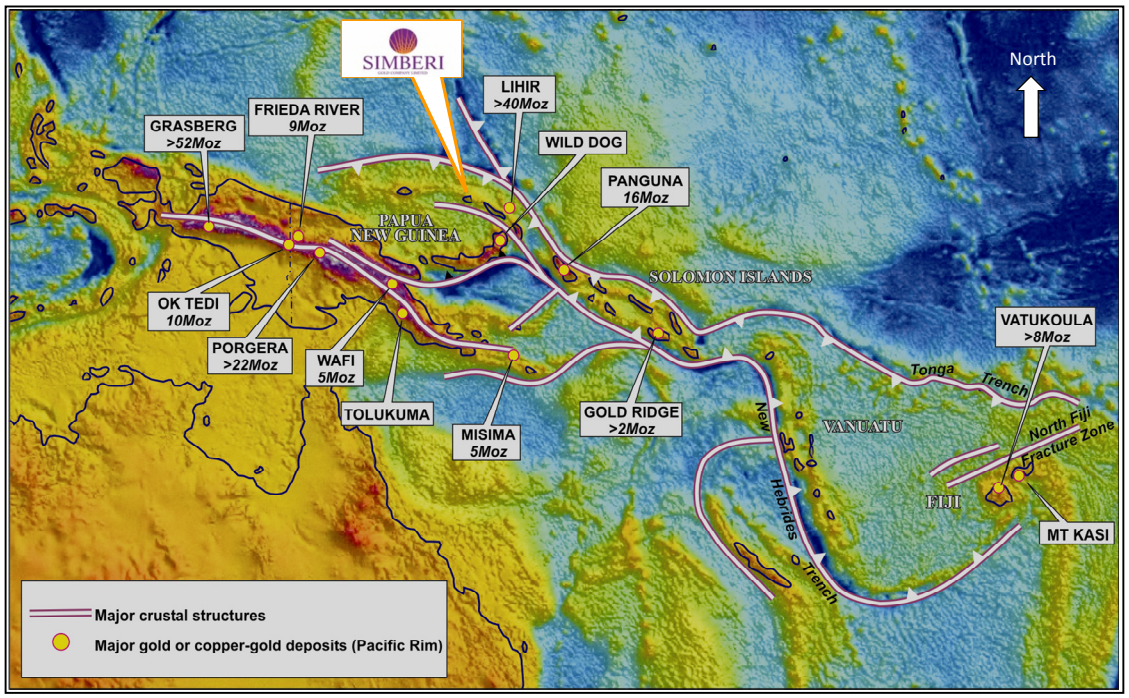

Figure 9-1: Simberi – Regional Tectonic map and Gold Mines

Simberi Island itself is approximately 9 km in diameter and is the smallest of the three islands forming the Tabar Island Group. The rugged central part of the island contains volcanic and intrusive rocks in an area about 6 km in diameter, rising to over 300 m elevation. The volcanic edifice is partially encircled by raised bioclastic reefal limestone.

The interpreted geological history of the island is that of a volcano-intrusive centre in an ocean island arc setting in which the alteration and epithermal gold mineralisation, driven by porphyritic igneous bodies intruded below, was emplaced in the waning stages of volcanism.

Results from a high definition airborne magnetic and spectrometer geophysical survey, dipole-dipole Induced Polarisation geophysical surveys carried out in early 2005 and a helicopter borne high definition time – doimain electromagnetic (Heli-TEM) survey undertaken in December 2008 support this interpretation. The geophysics indicates the likely presence of highly resistive porphyry intrusives at depth. Conductive and chargeable clay-pyrite alteration zones lie above and trend away from the interpreted intrusives. Gold mineralisation is often associated with chargeable zones on the margins of intrusives.

9.2 Prospect Geology

The currently known gold prospects on Simberi Island are located in the eastern half of the island, within the central volcanic core and are contained within a sub-cropping epithermal alteration system and structural corridor extending 4 km north-south and 2 km east-west. The host rocks for the mineralisation comprise altered alkaline lava flows or intrusives (porphyries), volcanoclastics and tuffs. Some local fine-grained bedded tuffs with soft-sediment structures may represent lake sediments deposited in maar-like depressions or craters (Figure 9-2). Gold mineralisation, however, does not appear to be closely associated with any particular lithology.

The epithermal system is characterised by widespread surface gold anomalism, a distinctive airborne radiometric and magnetic geophysical signature and ground IP geophysical chargeability and resistivity anomalies. The identified (4 km by 2 km) epithermal alteration system appears coincident with a broad anomalous >0.2 g/t gold surface geochemical halo. There is no thermal spring activity, though remnants of such activity exist.

Subsequent drilling of higher-grade anomalies within this halo has mostly focused on the definition of seven shallow oxide resources and, potentially open pit mining Reserves. Evaluation of deeper sulphide mineralisation has concentrated primarily on the Pigiput and Pigibo deposits, with lesser amounts of deeper drilling being undertaken on the Samat, Botlu and Sorowar deposits. All the known oxide deposits, except Samat East, are immediately underlain or proximal to sulphide gold mineralisation.

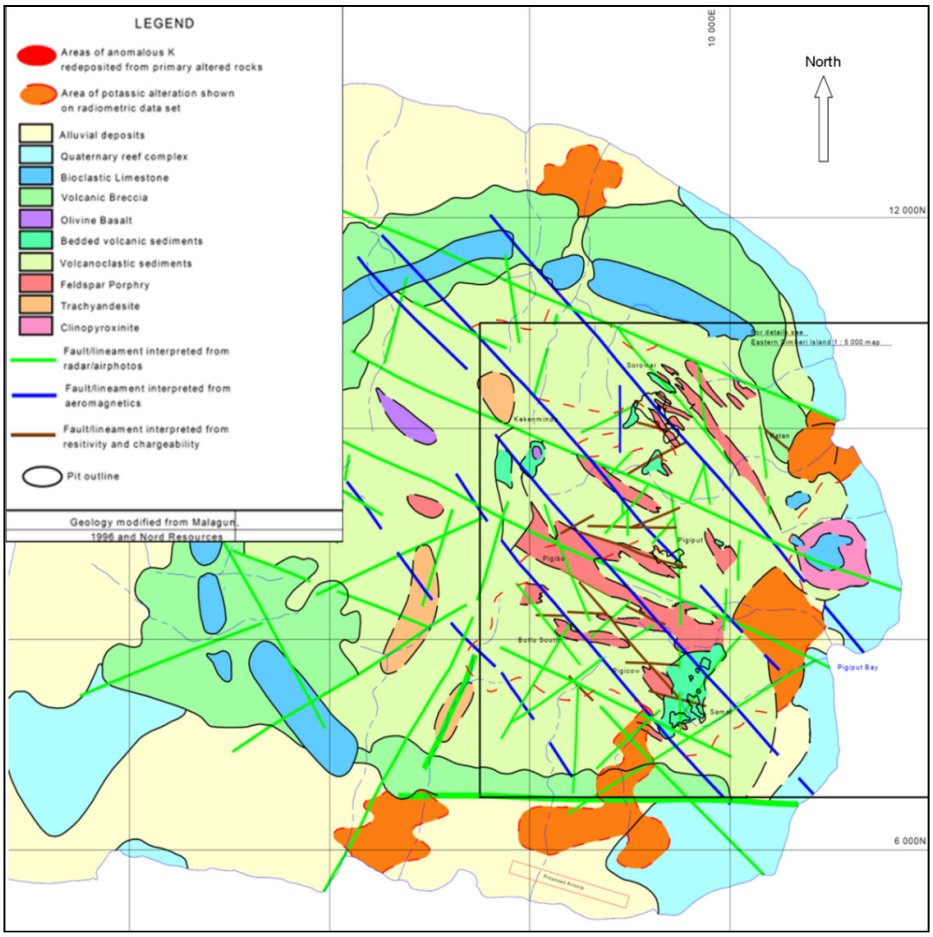

Figure 9-2: Simberi Island Geology (Simberi Island Grid)

9.3 Interpreted Geological History

The following interpreted sequence of geological events has provided a framework to guide resource estimation and further exploration.

Research suggests typical island-arc volcanoes (10-100 km^3 in volume) have typical life spans of 200,000-600,000 years indicating a likely timeframe for events 1 to 5 in Table 9-1. Events 6 to 9 occurred once the volcano became dormant. Deep weathering and erosion under a tropical weathering regime is presently active.

Table 9-1: Interpreted Geological History

Late Pliocene (3Ma) to Present	9	Extensional block faulting, producing horsts and grabens
	8	Development of latersol on the sediments and deep oxidation of some of the mineralisation
	7	Development of local and partial cover of clayey sediments, possibly at the same time as the formation of the now-raised limestone reefs
	6	Weathering and oxide development, affecting the upper part of the mineralisation, with oxidation preferentially extending to some depth along major structural zones
	5	Introduction of gold mineralisation into some of the zones of hydraulic fracturing, with some spreading into adjacent clastics where structural pathways are available
	4	Further alteration (clay, feldspar, magnetite destruction) by porphyry-sourced magmatic fluids, in some cases causing otherwise clayey and altered diatremes to become competent and providing a favourable lithology for subsequent fracturing and mineralisation emplacement
	3	Emplacement of multiple diatremes by phreatomagmatic explosive processes along zones of weakness above the intrusive(s)
	2	Porphyry intrusion(s) at depth along the same zone, moving up into the volcanic pile, with associated propylitic alteration
	1	Formation of a pile of crystalline and clastic volcanics sourced from volcanic activity along a major zone of crustal weakness

10.0 DEPOSIT TYPES

10.1 Simberi

The Simberi Gold Project plans to exploit seven separate prospects on the eastern portion of the island. Sorowar in the north is by far the largest Oxide gold resource (Figure 10-1). Samat North, South and East lie to the south and while relatively small are also relatively high grade. Pigiput, Pigibo and Botlu South lie between the Sorowar and Samat areas and have Oxide gold resources of intermediate tonnage but at a grade similar to Sorowar. Pigiput has the largest Sulphide gold resource. All prospects lie within 2-3 km of each other. The project area also includes other less well defined prospects and anomalies.

The main Oxide deposits are Sorowar, Pigiput, Samat South, North and East, Botlu and Pigibo. The base of oxidation varies between a few metres to greater than 50 m vertical depth and generally favours higher elevations, from 100 m elevation at Samat to 280 m at Sorowar.

The grade of the mineralisation is related to the natural porosity and degree of fracturing of the host rocks, greatest in the vicinity of steep and moderately dipping feeder structures interpreted to have been the pathways for both alteration and mineralising fluids. Gold bearing Sulphide (unoxidised) mineralisation has been identified at depth beneath Oxide mineralisation at Sorowar, Pigiput and Pigibo, beneath oxide mineralisation and near-surface at Botlu and immediately beneath oxide and transition mineralisation at Samat South and Samat North. Primary gold occurs within sulphide minerals, dominantly arsenean-pyrite, and remains very fine grained when released by oxidation.

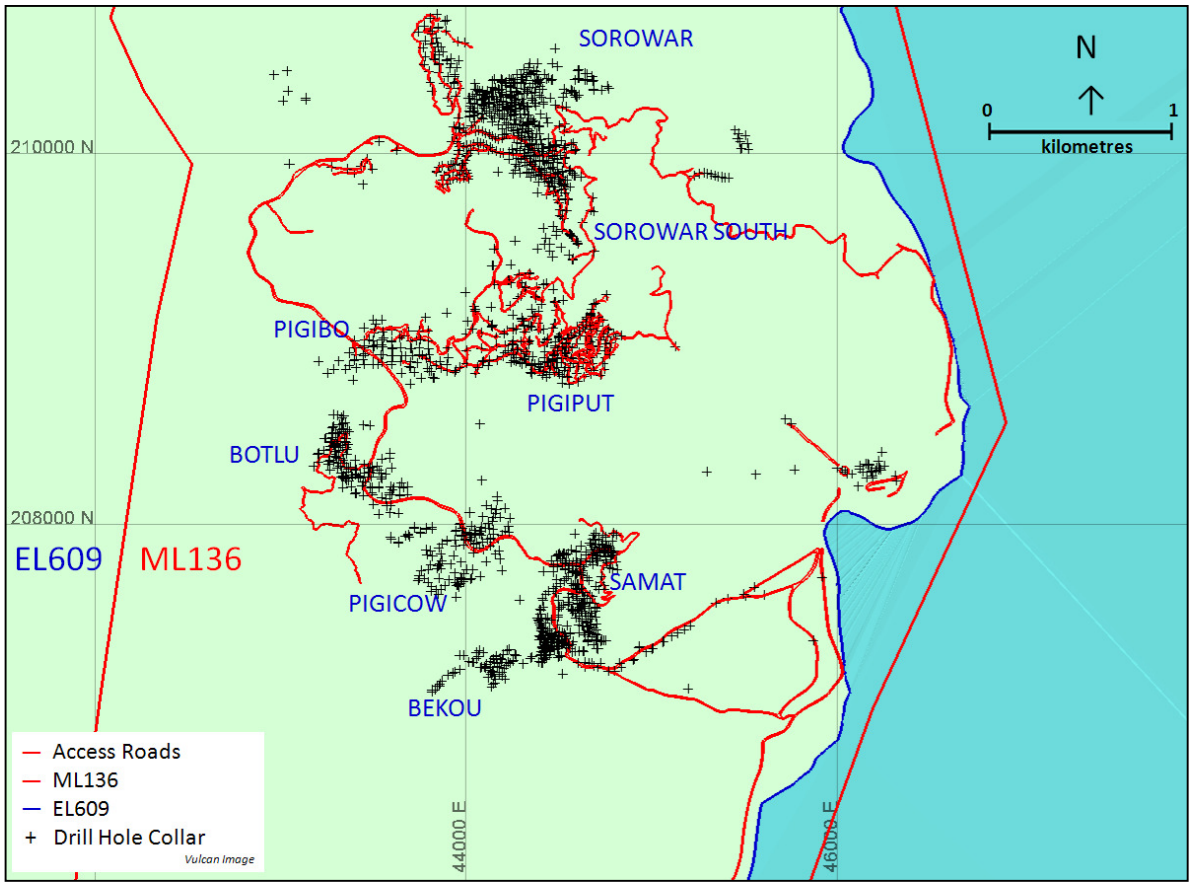

Figure 10-1: Simberi – Mineralised deposit locations

10.1.1 Sorowar

At Sorowar, the largest known deposit on Simberi, the oxide gold mineralisation is associated with an alternating sequence of NW (315°) trending porphyry and volcanoclastic tuff interbeds or intercalations with lesser volcanic diatreme bodies or slices that dip at moderate angles (25° to 45°) to the SW. The diatreme bodies appear structurally controlled, exploiting a series of more recent E-W and NE-SW trending structures.

The main part of the deposit is located at the intersection of a major E-W and NE-SW structures with the most favourable lithological zones being diatreme and brecciated formations. Higher grade zones have an E-W trend.

Irregularities in the depth of oxidation over the Sorowar deposit indicate interplay between lithologies and structures. A NW trending steep dipping surface, interpreted as a fault, marks the western boundary of the main body of oxide mineralisation and coincides with a zone of deep oxidation. It also marks the western edge of a prominent ridge and the eastern side of a shallow valley that contains a superficial clay cover.

Finer grained tuffaceous sediments, with local bedding and soft sediment deformation structures, outcrop on a ridge coinciding with the north-west margin of the pit design. These are interpreted to be either part of tuff-ring sediments ejected around diatreme emplacement centres or maar deposits.

Most of the mineralisation exposed at surface by mining is in fractured porphyry or volcanoclastic/diatreme material, with gold grade being related to the intensity of fracturing and rock porosity. The higher gold grades, those over 3.0g/t, are often found in intensely fractured (crackle brecciated) porphyries adjacent to a steeply dipping fractures interpreted to have been a fluid flow pathways. Narrow vertical structures and

groups of structures in the overlying sediments that can carry gold up to 16.0 g/t, suggest fluids or gases rose upwards through a thick slurry of tuffaceous sand and clay.

Sorowar mineralisation occupies both a shallow valley and ridges to the east and north. The depth of oxidation is greater in the valley than on the ridges, particularly the ridge to the east.

There are several different phases of volcanoclastics and breccias, either mono- or hetreo-lithic, with varying clast sizes varying from sand-sized to several tens of metres. While the tenor of gold mineralisation is more closely related to fracture intensity than lithology the naturally porous volcanic tuffs and breccias host significant disseminated mineralisation.

The main part of the Sorowar mineralised zone is situated at the intersection of an interpreted E-W structural zone with a favourable lithological sequence. Originally, three separate pods of mineralisation were identified outside the original (Oct 2006) planned pit, to the northwest, south-east and east. Subsequent infill drilling, in 2007 and 2008, linked these bodies to the main area of mineralisation. The current pit design (May 2009) has a pronounced elongation of the Oxide pit in a NW-SE direction, consistent with the orientation of the main NW trending fractures (Figure 10-2).

The base of oxidation at Sorowar was re-interpreted in 2008, using three-dimensional surface modelling (wireframing). As in earlier models, broad zones of deep oxidation were identified, though, in detail, significant irregularities in the depth of oxidation indicate interaction between intersecting lithologies and structures. In some places, the model was made to conform to distinct linear deeper-oxidation zones (usually east-west).

There are two oxidised zone forms at Sorowar:

- an upper-oxide zone that matches the topographic profile and rarely exceeds 30m depth from surface, and
- a sheet-like zone that penetrates to a depth of ~100 m below surface and results from the deeper weathering of structural fractures.

Detached sulphide ‘pods’ in the sheet-like zone, that occur where the rock is less permeable, whether due to the original lithology or its alteration, to allow meteoric fluids to penetrate, are common.

The Sulphide gold mineralisation at Sorowar is less drilled, due to the past focus of solely drilling for shallow Oxide gold. To date, intercepts in the sulphide zone are narrow and generally of lower grade than those in the oxide zone.

Mining of the Sorowar deposit commenced in May 2008.

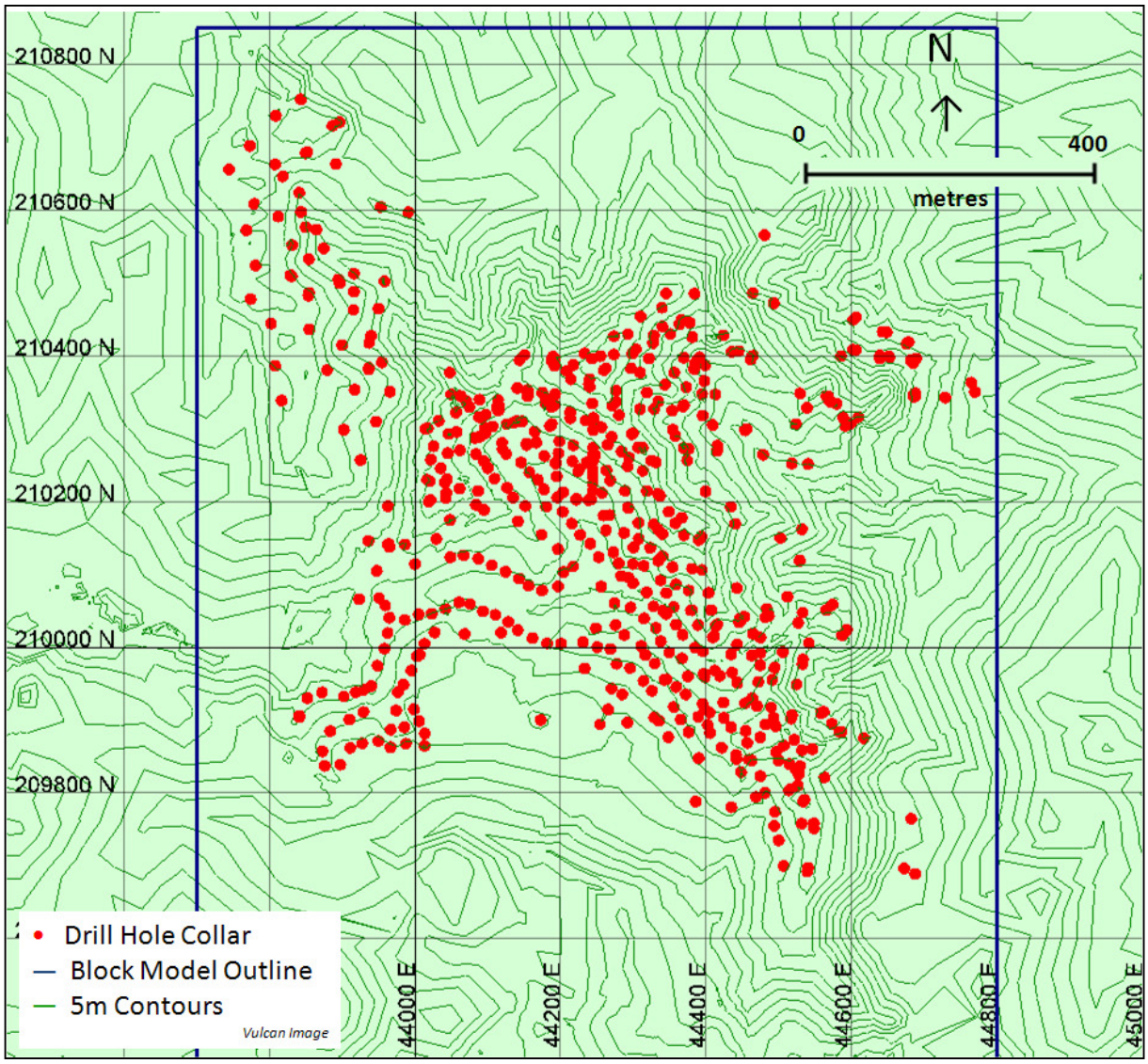

Figure 10-2: Sorowar Deposit – Drill Collar Locations and Topographic Contours

10.1.2 Samat

At Samat the pattern of the thin layers of mineralisation is suggestive of mineralisation sources along discrete structural zones, with spreading by supergene processes and soil creep at surface, modified by local erosion.

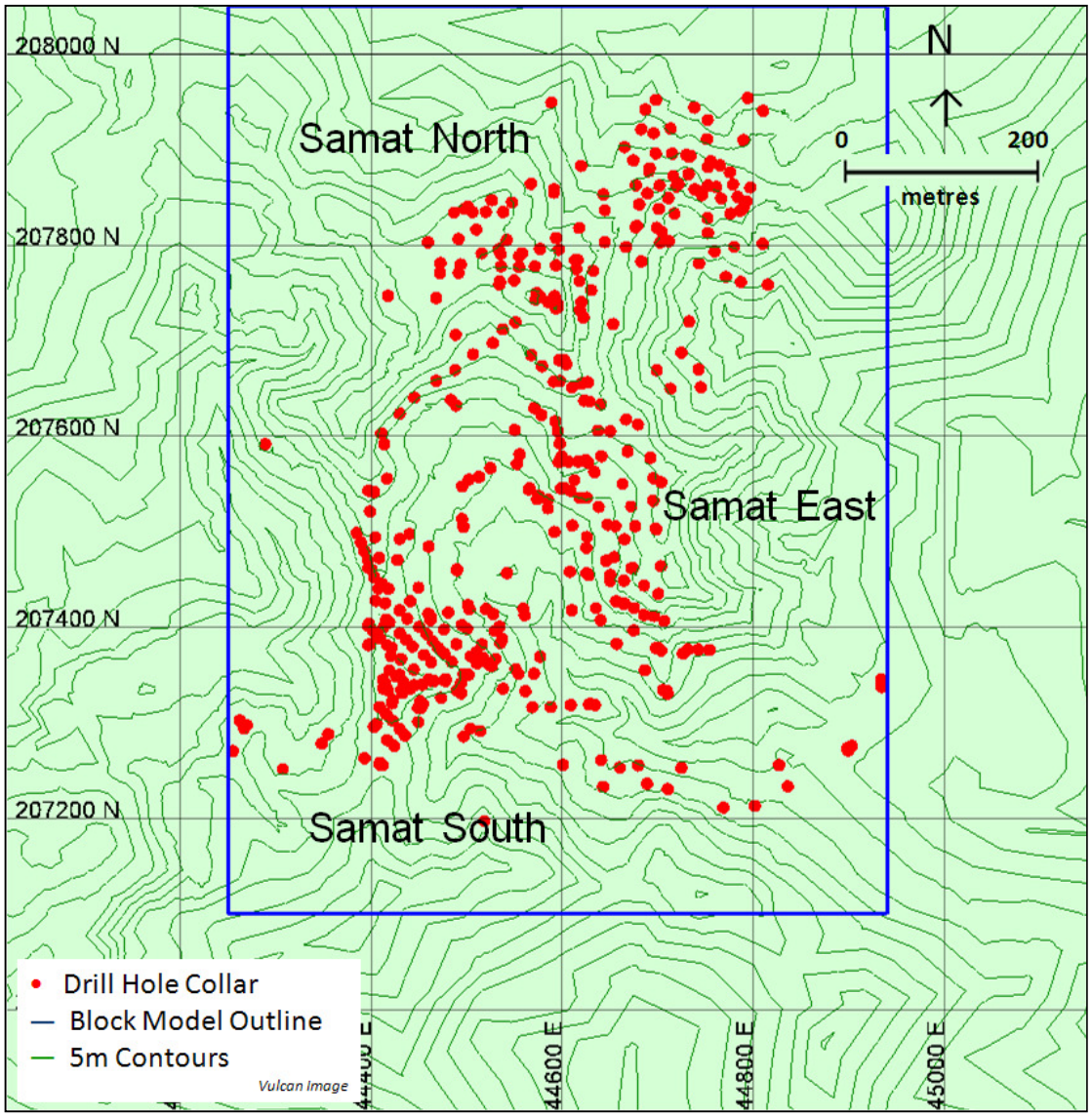

Figure 10-3: Samat – Drill hole locations and pit design outlines

At Samat South, volcanoclastics and diatremes are closely associated with the mineralisation. An interpreted 085° trending fault to the south marks a boundary between hydrothermally altered and unaltered rocks. Within the mineralised zone itself, structures are not sufficiently well defined to provide limits that can be used in resource estimation.

At Samat North, differences between several twinned drill holes indicate structural complexity. The northern boundary of the mineralisation appears to lie along a NW-SE trending fault. A NE-SW trending fracture zone may pass through the mineralised zone and intersect the northern end of the Samat East structure.

At Samat East, a thin layer of oxide mineralisation extends in a NNE direction, on the western slopes of a prominent ridge.

10.1.3 Botlu

Oxidised material at Botlu is generally thin, up to 10 m deep, however at one location, in the north of the prospect area, oxidation extends to 80 m from surface over an area approximately 100 m by 150 m.

Oxide gold grades tend to be higher near surface decrease with depth. In the south-east of the prospect moderate to high-grade gold in un-oxidised material occurs from near surface to 100 m depth.

Gold mineralisation is associated with highly fractured and altered porphyritic andesite and volcanoclastic rocks, occasionally with minor quartz veining and silicification.

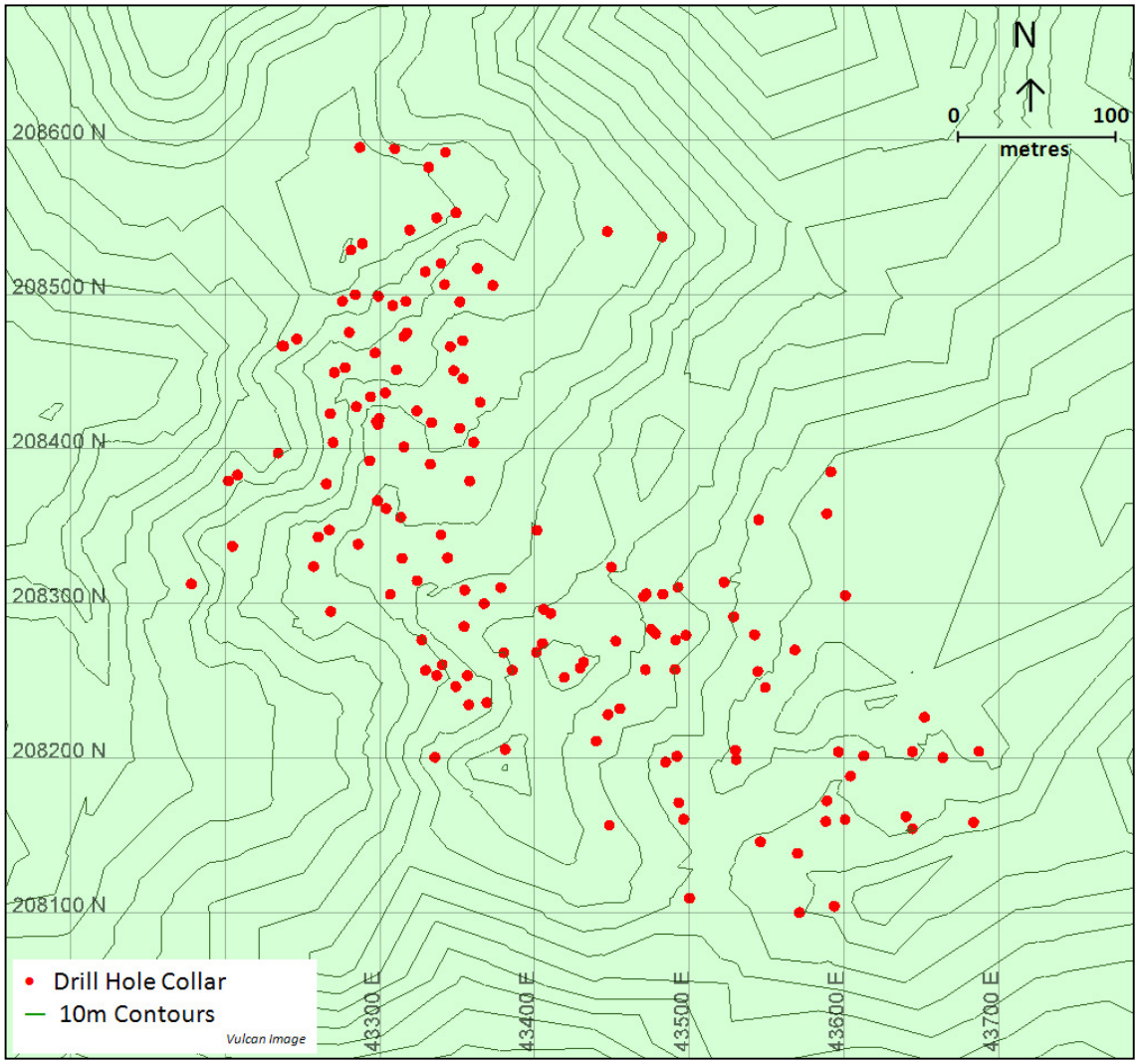

Figure 10-4: Botlu – Drill hole collar locations and proposed pit outline

10.1.4 Pigiput

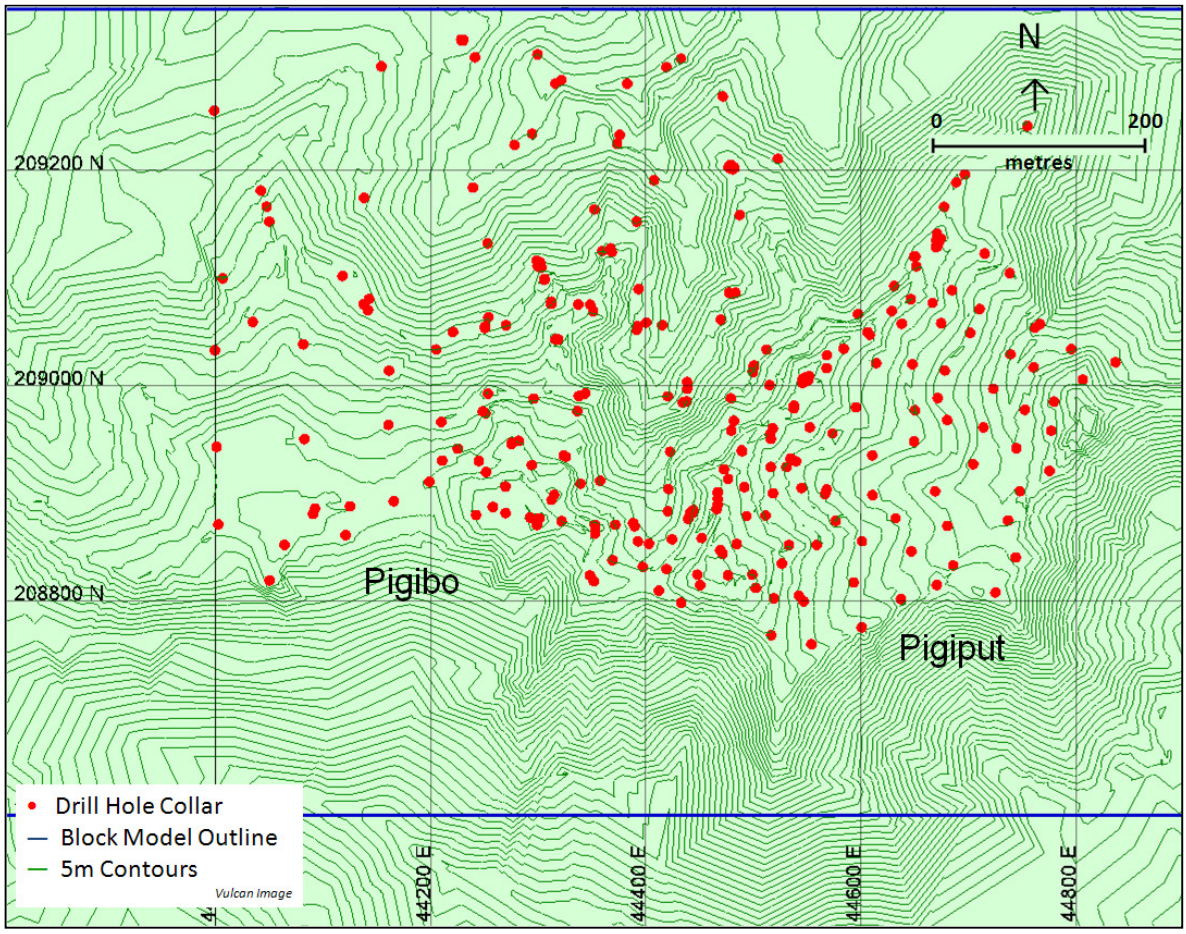

Figure 10-5: Pigiput drill hole collar location plan with proposed pit outline

At Pigiput, the near-surface oxide mineralisation is closely associated with an east-west trending ridge. The deeper sulphide mineralisation is interpreted to be related to alteration of the shallow dipping stratigraphy best developed around steeper dipping dominantly NW, N and E trending fractures. The dominant lithologies intersected near-surface in drill holes, are crudely bedded, coarse to fine grained tuffs, with occasional volcanic breccia. Deeper drill holes have intersected fine-grained crystalline rocks that may be either intrusive or extrusive feldspar porphyries. Petrographic analysis reveals a typical latite andesite composition.

The latite andesite porphyries have potential to act as aquitards, allowing alteration veining and replacement to develop in underlying more permeable units around structures that cross cut lithlogical boundaries. The porosity of tuff units is inherently significantly greater than the interbedded andesite units allowing more widespread alteration to occur, leading to total replacement in some cases. Both permeability and porosity in the andesite units is provided by crackle brecciation, best developed at andsite/tuff contacts, particularly around cross-cutting structures.

The main geological units mapped at surface are moderate to intensely altered and interbedded tuffs with alternating coarse to fine beds dipping at shallow angles (5° to 20°) to the south and east.

Structural mapping has highlighted three main trends of fracturing:

1) NW striking, moderate NE dipping

2) N-striking and steeply dipping, and

3) E-W striking, moderate to shallow N dipping.

Higher grade gold mineralisation is mainly associated with silica-carbonate-pyrite and clay-pyrite alteration haloes particularly well developed around the N-striking, steeply dipping structures and in the hanging wall of the moderate to shallow dipping NW trending structures.

The higher-grade Oxide mineralisation occurs in a near-surface layer on the ridge crests, suggesting some supergene enrichment. Oxide mineralisation can extend to over 100 m depth in the mineralised zone. Mineralisation is contiguous into the transitional and sulphide zones.

Sulphide mineralisation at the Pigiput deposit was the main focus of Allied's 2009 exploration programme. Core holes drilled tested the down dip extension of the Pigiput Prospect gold mineralisation defining a broad zone of NW dipping mineralisation that contains structurally related higher grades within lower grade disseminated haloes in both tuffs and brecciated porphyritic andesites.

Gold mineralisation is associated with broad zones of variably intense carbonate and quartz-carbonate replacement and fracture fill alteration. This differs from the clay-sulphide dominant alteration seen in the upper levels of the Pigiput Prospect and also the Samat deposits 1 km south.

At Pigiput, gold mineralisation (>=0.5g/t Au) is associated with pyrite and geochemically elevated arsenic (average 855.6 ppm in 5,682 drill hole sample analyses) and copper values (average 121.4 ppm). While relatively low average base metals levels of other base metal elements (molybdenum 52.0 ppm, lead 163.8 ppm and zinc 271.8 ppm) are noticeably higher than those at the neighbouring Pigibo Deposit.

Base metal contents vary with depth and sulphur content, the latter generally linked to degree of weathering. In the upper 100 m of holes drilled at Pigiput, where sulphur values average 1.0% (typical of Oxide zone), arsenic contents are relatively high (average 1200 ppm) while base metals are low (molybdenum 11.8 ppm, lead 13.0 ppm and zinc 76.5 ppm). This contrasts significantly with samples below 100 m downhole, where sulphur averages 4.0% (typical of Sulphide zone) arsenic contents are noticeably lower (average 541.7 ppm) while base metals are higher (molybdenum 80.7 ppm, lead 271.4ppm and zinc 405.3 ppm). Copper values are similar in both zones, averaging 126.0 ppm to 100 m downhole and 134.5 ppm below 100 m, suggesting the difference in base metal contents is genetic and not simply caused by weathering.

10.1.5 Pigibo

At Pigibo, gold mineralisation (>=0.5g/t Au) is associated with pyrite and geochemically elevated arsenic (average 712.8 ppm in 1670 drill hole sample analyses) and copper values (average 165.8 ppm) and otherwise low average base metals levels (molybdenum 5.3 ppm, lead 10.3 ppm and zinc 90.3 ppm) similar to the upper parts of the Pigiput Deposit. Host rocks are predominantly, porphyritic intermediate intrusive, crackle brecciated where mineralised, with lesser tuffs.

The mineralised zones are interpreted to strike east-west and dip moderately to steeply northwards. These structures run through host rocks (predominantly andesitic in nature) and have a pervasive lower grade halo around them. The 2009/2010 drill programme tested mineralisation over 600 m in length. Drill holes are oriented -60° south to intersect the mineralisation.

The oxide zone is on average 30-40 m deep although in localised areas it can go deeper. Bottle roll tests on the RC drill chips showed good cyanide leachability of the gold in the oxide zone, consistent with other deposits at Simberi. Poor recoveries were achieved in transition and sulphide materials.

Recent RC drilling in at the Pigibo Prospect, improved section-to-section continuity of the main zone of gold mineralisation (Figure 10-5). The mineralisation follows the crest and southern flank of a hill and is modelled around a moderately north dipping surface, consistent form section-to-section. Two steep NNW trending

flexures are interpreted to cross the surface, causing right-lateral off-sets of the mineralised zones and a drop in grade over short strike lengths

The limits of the Pigibo gold mineralisation are yet to be fully defined, being still open at depth on most sections and along strike to the east.

10.1.6 Pigicow

At Pigicow, narrow intrusives of intermediate composite commonly cross cut coarse grained volcanoclastics and andesite/latite flows. In nearly all cases, they are steep to sub-vertical and trend WNW. The contacts are usually gouged and there is little evidence of wall rock alteration. Such contacts may have been activated repeatedly during the volcanic evolution of Simberi and, likely, subsequently.

Diatreme breccias, exhibiting milled clasts and a rock flour, or clay, matrix also trend WNW. Adjacent bodies of intrusive and breccia are common. The diatremes represent a late event, as they often contain clasts of the intrusive lithologies.

Gold mineralisation is rather diluted in oxide, and the protolith fabric is largely degraded yet the primary occurrence is not totally obliterated. Higher grade channel samples may be correlated with porous breccias, for the most part and in such cases, the rocks are usually heavily stained by both hematite and goethite, after pyrite. Lesser jarosite is also present as a supergene accessory in zones of higher gold grade.

IP chargeability profiles define a conductive body that likely represents disseminated primary sulphides (pyrite ± arsenopyrite). This body can be traced as WNW trending throughout Samat (five 2005 IP sections) and within Pigicow (one 1997 IP section). Gold grades are attributable to the margins of such a body and this correlation is evident within many drill core/chargeability comparisons, whereby high chargeability represents a clay-pyrite halo (or core), with gold mineralisation focused along the margins. Strong fracturing of margins of such bodies produces a low resistivity.

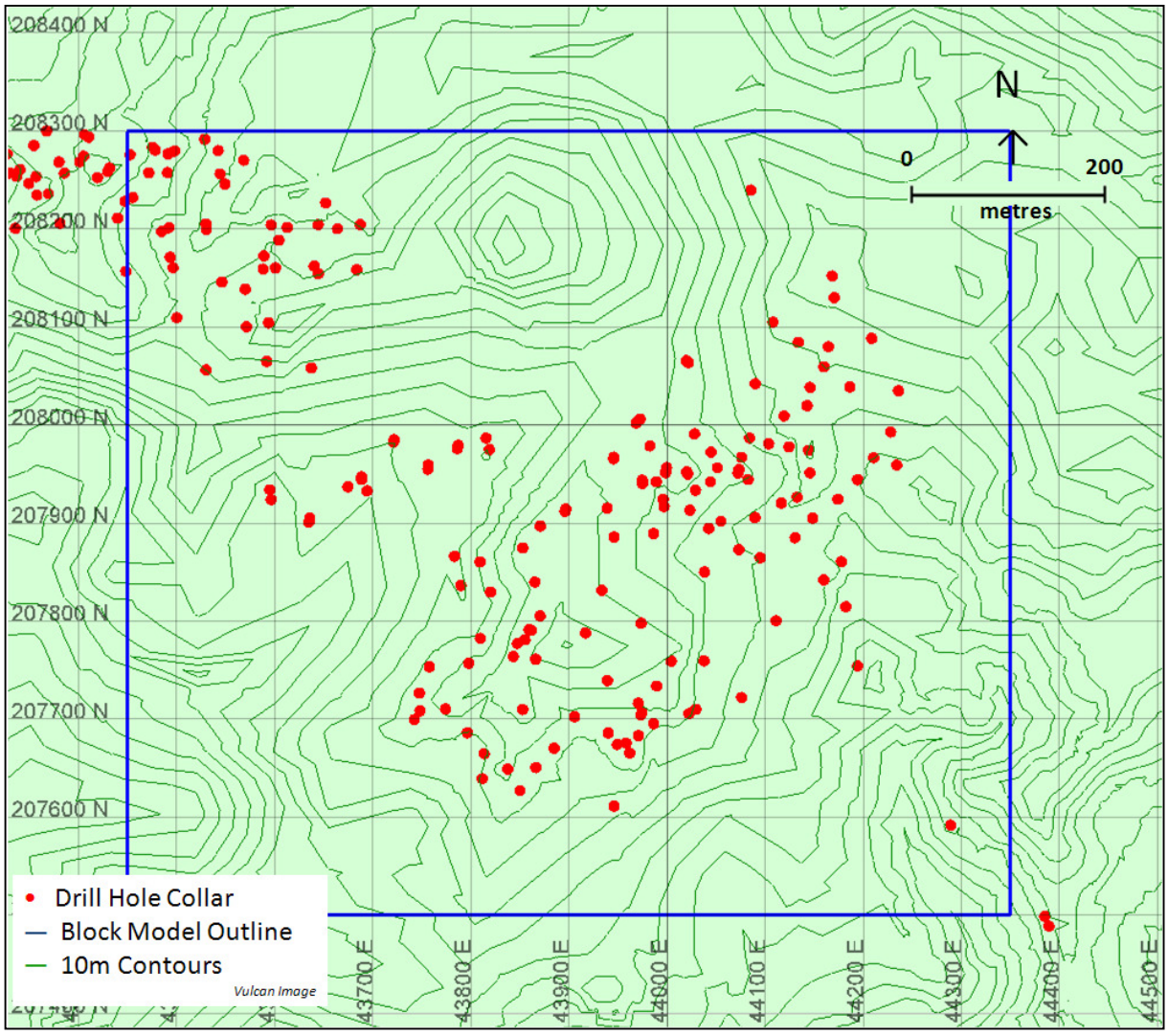

Figure 10-6: Pigicow – drill hole locations

10.1.7 Bekou

Mapping of the Bekou tracks in late 1996 revealed the presence of numerous diatreme breccias containing abundant disseminated pyrite with jarosite staining and pervasive clay alteration. These pipes are largely subvertical trending in WNW direction although flat lying and N-S breccias have been observed. These are invariably bounded by strong fracturing with associated hematite and goethite in boxworks after sulphides. These breccias are relatively devoid of Au mineralisation but their margins are generally well mineralised over some distance.

Field mapping and correlation with gold grades in channel samples at Bekou reveals that the majority of gold mineralisation may be attributed to both fine and large scale structures. Mineralisation is very intermittent and probably represents limited zones of palaeo-dilation within the WNW trending bodies. The broad scale trend of the prospect scale ore bodies is WNW (reflecting the regional trend), with haloes of higher grades to the diatremes commonly highly oblique to this, representing early tension structures which are present throughout the Bismarck region. Such zones are characterised by abundant hematite-goethite replacement of sulphides in structures.

Both diatreme breccias and intrusives have exploited the steep WNW trending structures. Au grades in such diatreme breccias are variable, suggesting that they formed prior to the main mineralisation event. Where they are mineralised, it is usually along the margins or within fracture sets. These diatremes are readily recognisable on the surface by their distinctive blue-grey smectite-pyrite alteration and below the surface by their remarkable high chargeability values (generally 30-40 mV/V).

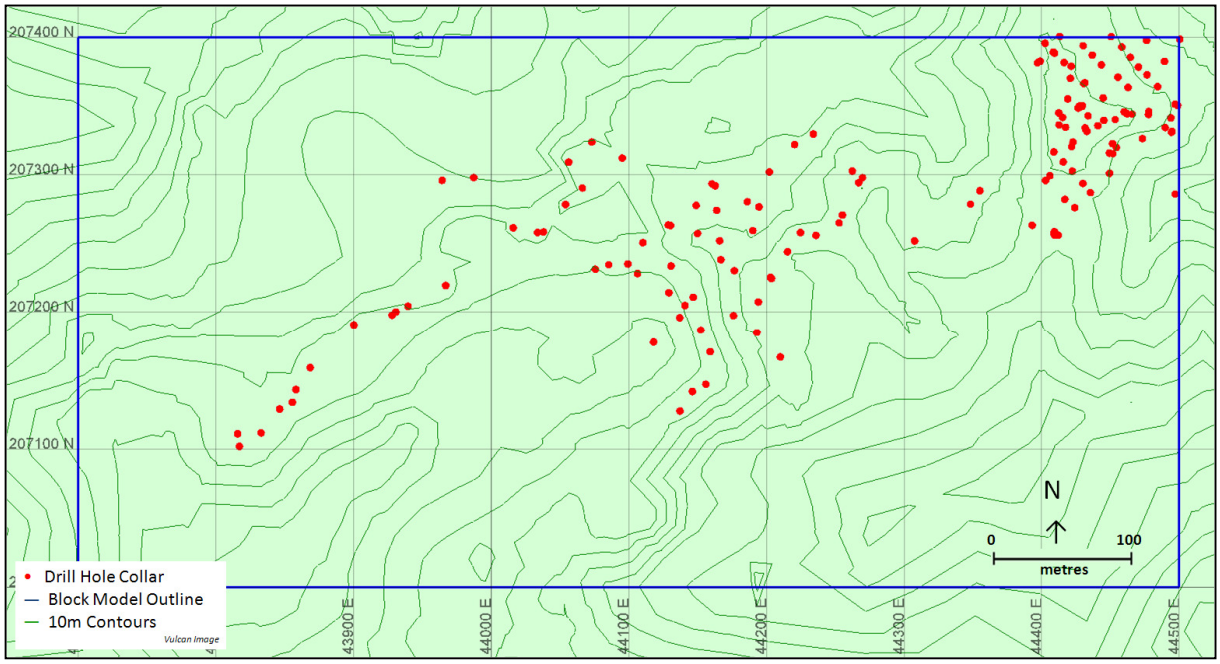

Figure 10-7: Bekou – drill hole locations

11.0 MINERALISATION

Where recognised, the main primary control of gold mineralisation is steeply dipping fracture systems, in places associated with milled breccia dykes (diatremes). Particularly high grades are associated with diatreme-country rock contact zones. Gold mineralisation is generally associated with sulphides or iron oxides occurring within all variety of hydraulic fractures, such as simple fracture infills, single vein coatings and crackle brecciation. This fracturing can occur in any rock type competent enough to fail under stress, particularly porphyry (either intrusive or extrusive) but also altered diatreme breccias and tuff deposits. Higher gold grades tend to be associated with higher sulphide/iron oxide content and greater degrees of fracturing. Alteration intensity and areal extent can be correlated with gold mineralisation.

In the oxidised zone, the gold is predominantly associated with iron oxides after sulphides, with higher grades being associated with rare vuggy and chalcedonic quartz.

The gold in the oxide zone occurs as 'free' gold as verified by high gold recoveries during cyanide leach. Gold in mineralised primary rocks includes refractory gold hosted by pyrite.

The particle size of the gold in the oxide zone is very fine (the bulk of it is less than 50 microns) and limited testwork has shown that the gold in primary rocks occurs as solid solution within the fresh pyrite and or marcasite and occasional arsenopyrite crystal lattice. Gold sometimes also occurs with carbonates and base metal sulphides at depth, with or without silver. The most notable silver – gold occurrence is in NW Sorowar.

Fluid inclusion studies of quartz from Sorowar identified a low temperature of homogenisation (207°C). This, along with the fine-grained nature of the quartz and inter-layered clay alteration, is believed indicative of cooler near-surface epithermal levels of a hydrothermal system

12.0 EXPLORATION

12.1 Surveys and Investigations

Gold mineralisation was discovered by modern reconnaissance exploration on Simberi in 1981, though there is evidence of very limited alluvial gold workings, probably dating from the 1920s, to the north of Sorowar.

Reconnaissance geochemical exploration and geological mapping by Kennecott identified a large area of eastern Simberi as being anomalous in gold, within which there were several prospects with evidence of higher grade and coherent mineralisation. These prospects were systematically aircore and diamond drilled between 1983 and 1990, initially testing both oxidised and sulphidic gold mineralisation. Later drilling programs focused on oxide material only.

A helicopter borne magnetic and radiometric survey carried out in 1987 was reinterpreted in 1995 to produce composite radiometric and reduced-to-pole magnetic images. The radiometric data highlights the potassic alteration associated with the known mineralisation. The aeromagnetic data shows the known mineralisation is associated with zones of subdued magnetic response, interpreted as zones of alteration-related magnetite destruction (Figure 12-1).

The aeromagnetic data shows the known mineralisation is associated with zones of interpreted magnetite destruction. These features are evident as zones of subdued magnetic response in contrast to the highly variable magnetic response seen over most of the western half of the island.

Several dipole-dipole IP surveys, pole-dipole surveys, and gradient arrays IP surveys have been carried out on Simberi Island. An image of the dipole-dipole and pole dipole surveys is shown in Figure 12-1.

In conjunction with Barrick's exploration of the Tabar and Tatau Islands, an AEROTEM electromagnetic survey was flown over the leases in December 2008/January 2009. Approximately 530 line km of data was acquired over Simberi Island.

Barrick geophysicists interpreted the main features, shown in Figure 12-2 to be:

- conductive ring due to seawater
- central resistive zone that hosts the known mineralisation
- moderately conductive ring, likely to be a stratigraphic unit, and
- weakly conductive cover, possibly volcanic ash, that is considered prospective and under-explored, (see also Figure 12-3 that overlays the drill hole coverage).

Subsequent systematic exploration programs using aircore, RC and diamond holes by Nord, PGM Ventures and Allied Gold together with surface geochemistry and IP geophysical surveys have continued to increase resources on Simberi.

Ongoing drilling by Allied resulted in increased resources and reserves at Sorowar, Pigiput/Pigibo and Samat. This drilling includes RC and diamond core holes drilled to verify older holes completed by Kennecott and Nord in respect to mineralisation orientations, grade and metallurgy.

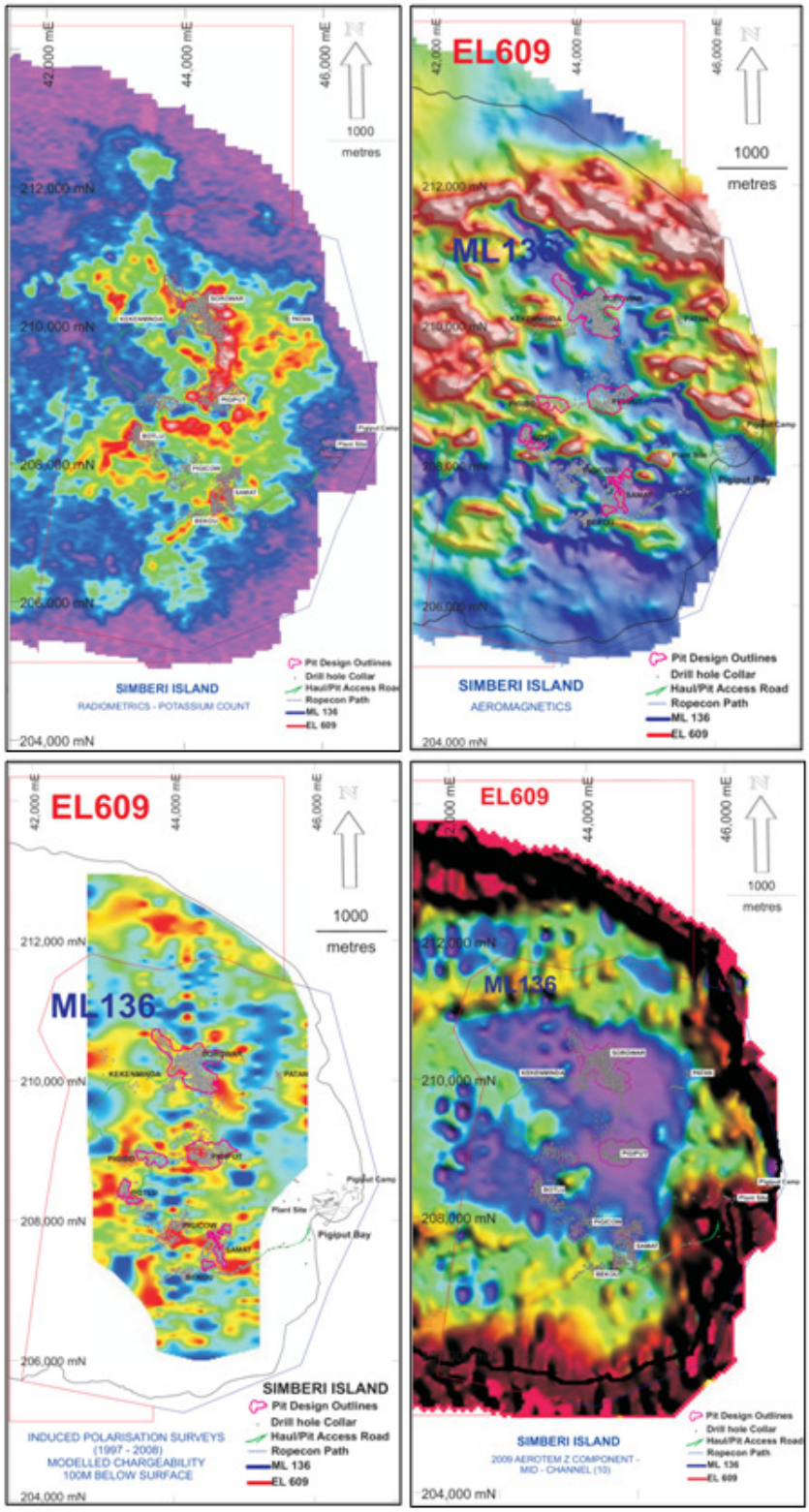

Figure 12-1: Simberi Is – ML136 – Comparative Images of Geophysics Data

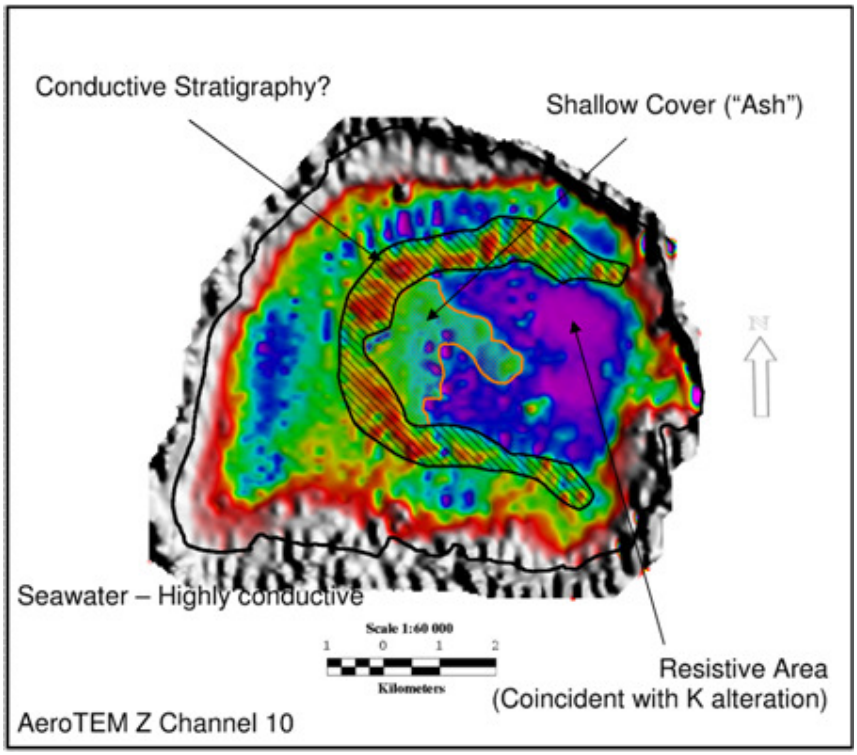

Figure 12-2: AEROTEM Survey – Interpretation (Barrick, 2009)

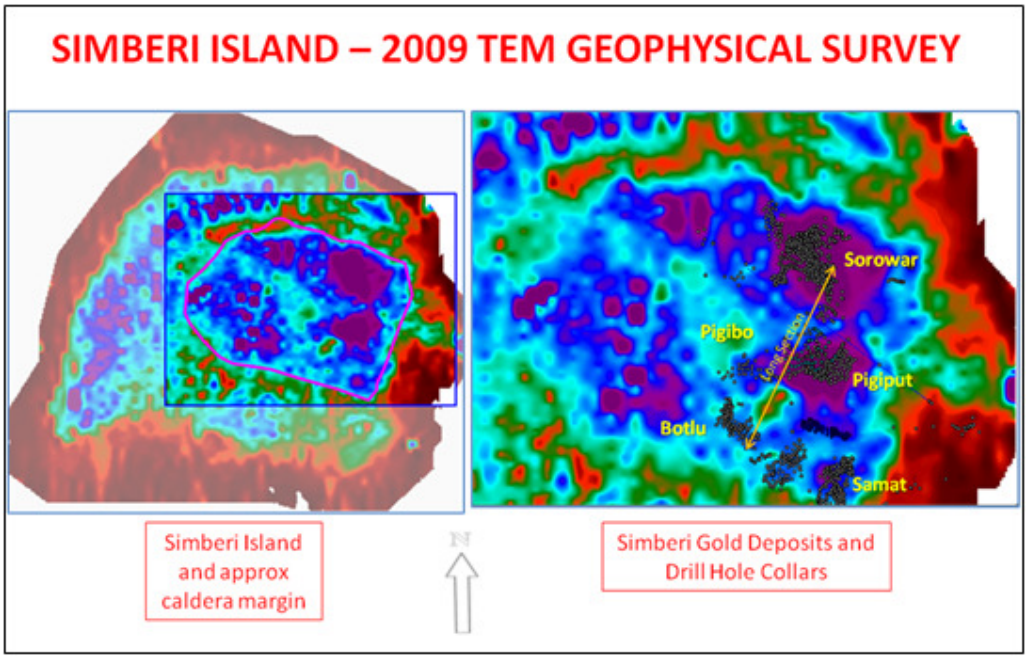

Figure 12-3: AEROTEM 100 m Resistivity with Simberi Drill hole Collars

13.0 DRILLING

Simberi has been explored and developed using channel sampling, Reverse Circulation Drilling and Diamond Drilling. A summary of the drilling history is provided below:

Table 13-1: Adora – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1995	Nord	RC	4	113.00
1997	Nord	RC	6	384.00
2006	Allied	RC	6	268.00
2007	Allied	DDH	4	579.42
		RC	16	884.00
2008	Allied	DDH	1	179.50
Total Adora			**37**	**2,367.92**

Table 13-2: Bekou – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1997	Nord	DDH	3	508.05
		RC	20	1,259.00
2006	Allied	RC	48	1,122.00
Total Bekou			**71**	**2,889.05**

Table 13-3: Botlu – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1986	Kennecott	DDH	6	1,587.10
1987	Kennecott	AC	30	2,662.50
		DDH	3	666.15
1988	Kennecott	DDH	1	218.00
1989	Kennecott	AC	10	1,141.00
		DDH	1	185.70
1996	Nord	MET	2	184.20
		RC	8	302.00
1997	Nord	DDH	3	371.70
		RC	83	4,010.00
1998	Nord	RC	3	450.00
2003	Allied	MET	2	66.00
2010	Allied	RC	2	240.00
		DD	9	2,148.80
Total Botlu			**152**	**11,844.35**

Table 13-4: Kekenminda – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1998	Nord	RC	6	621.00
Total Kekenminda			**6**	**621.00**

Table 13-5: Patan – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1989	Kennecott	AC	10	569.00
2007	Allied	RC	11	848.00
Total Patan			**21**	**1,417.00**

Table 13-6: Pigibo – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1986	**Kennecott**	**DDH**	**1**	**302.00**
1987	Kennecott	AC	12	1,094.00
1989	Kennecott	AC	2	289.00
1997	Nord	DDH	2	266.10
		RC	31	3,372.00
1998	Nord	DDH	3	878.10
		RC	1	100.00
2005	Allied	RC	4	188.00
2008	Allied	RC	11	1,309.00
2009	Allied	DDH	18	2,376.80
		RC	23	2,753.00
		DD	2	429.60
2010	Allied	RC	31	2,055.00
		DD	6	1,228.20
Total			**147**	**16640.80**

Table 13-7: Pigicow – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1987	Kennecott	AC	10	972.00
1997	Nord	DDH	1	201.00
		RC	21	1,399.00
1998	Nord	DDH	1	100.00
		RC	1	100.00
2004	Allied	RC	1	30.00
2006	Allied	DDH	6	832.00
		RC	100	3,312.00
			141	6946.00

Table 13-8: Pigiput – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1984	Kennecott	DDH	2	526.80
1985	Kennecott	DDH	13	3,052.35
1988	Kennecott	AC	51	6,604.50
		DDH	8	1,855.35
		MET	4	274.00
1989	Kennecott	AC	20	2,605.50
		DDH	10	2,287.60
1996	Nord	RC	15	930.00
1997	Nord	DDH	12	2,830.50
		RC	20	2,015.10
2005	Allied	DDH	7	1,614.90
		RC	7	790.00
2006	Allied	DDH	2	326.10
		RC	4	444.00
2008	Allied	DDH	9	1,933.74
		RC	66	7,283.00
2009	Allied	DDH	47	12,673.20
		MET	5	1,083.10
		RC	4	722.00
		DD	7	2,047.50
2010	Allied	RC	2	328.00
		DD	14	3,211.10
Total Pigiput			**329**	**55,438.34**

Table 13-9: Plant – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1995	Nord	GEOTECH	18	275.30
		RC	6	226.00
2006	Allied	DDH	4	494.00
		RC	1	56.00
2010	Allied	DD	14	271.00
Plant Total			**43**	**1,322.30**

Table 13-10: Samat – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1985	Kennecott	DDH	2	332.80
1987	Kennecott	AC	27	1,905.00
1988	Kennecott	AC	115	8,276.00
		DDH	3	560.20
		MET	6	147.30
1989	Kennecott	AC	68	7,534.50
		DDH	8	1,081.91
		MET	1	38.70
1996	Nord	MET	4	119.40
		RC	95	2,337.00
1997	Nord	DDH	3	673.40
		MET	2	110.00
2006	Allied	DDH	3	659.50
		RC	22	1,030.40
2007	Allied	DDH	2	412.84
		RC	10	406.00
2008	Allied	DDH	1	76.00
		RC	42	1,598.00
2010	Allied	RC	12	1,358.10
		DD	3	577.00
			429	29,234.05

Table 13-11: Sorowar – Drilling History

Year	Company	Hole Type	Number of Holes	Metres
1985	Kennecott	DDH	3	732.60
1986	Kennecott	DDH	3	562.25
1987	Kennecott	AC	61	6,252.50
		DDH	2	435.10
1988	Kennecott	AC	16	1,992.50
		DDH	7	1,583.91
1989	Kennecott	AC	16	1,829.00
1995	Nord	DDH	4	278.40
		MET	6	480.80
		RC	25	1,808.60
1996	Nord	DDH	3	307.40
		MET	2	184.00
		RC	59	3,984.00
1997	Nord	RC	28	2,830.00
2004	Allied	RC	74	6,259.00
2005	Allied	DDH	8	1,330.00
		MET	3	240.10
		RC	120	10,391.20
2006	Allied	DDH	7	1,583.70
		RC	33	3,553.00
2007	Allied	DDH	4	292.17
2008	Allied	MET	5	938.42
		RC	173	14,670.00
2009	Allied	RC	1	98.00
2010	Allied	DD	33	6,828.20
Sorowar Total			**696**	**69,444.85**

Table 13-12: All Deposits – Total Drilling

Deposit	Number of Holes	Total Metres
Adora	37	2,407.92
Bekou	71	2,889.05
Botlu	163	14,233.15
Kekenminda	6	621.00
Patan	21	1,417.00
Pigibo	147	16,640.80
Pigicow	141	6,946.00
Pigiput	329	55,438.34
Plant	43	1,322.30
Samat	429	29,234.05
Sorowar	696	69,444.85
Grand Total	2083	200,594.46

In addition to drilling, various operators have undertaken surface channel sampling. A total of 14,804 samples, each representing approximately 5 m of channel, were collected and analysed and the results stored in the Allied database. The samples have an average grade of 0.52 g/t Au. Some of the channel sampling were used in the estimation of resources done by Minstat Pty Ltd (Botlu and Samat). No channel samples were used in the estimation of Sorowar, Pigiput, Pigibo, Pigicow or Bekou.

To give perspective to the extent and variety of drill sampling undertaken, Table 13-13 to Table 13-18 provide a breakdown of samples generated by prospect, period and drilling method for each of the prospects.

Table 13-13: Drilling Samples by Period and Drilling Method – Sorowar Prospect

Prospect	Year	Company	Drill Company	Hole Type	Samples
SOROWAR	1985	Kennecott	PNG Drillers and Wallis Drilling	DDH	733
	1986	Kennecott	PNG Drillers and Wallis Drilling	DDH	563
	1987	Kennecott	PNG Drillers and Wallis Drilling	AC	3507
				DDH	434
	1988	Kennecott	PNG Drillers and Wallis Drilling	AC	1935
				DDH	1477
	1989	Kennecott	PNG Drillers and Wallis Drilling	AC	1703
	1995	Nord	PNG Drillers	DDH	335
				MET	212
				RC	1802
	1996	Nord	PNG Drillers	DDH	133
				MET	180
				RC	3943
	1997	Nord	Pacific Drilling Services	RC	2794
	2004	Allied	Zenex	RC	6254
	2005	Allied	Zenex	DDH	731
				RC	10391

Prospect	Year	Company	Drill Company	Hole Type	Samples
	2006	Allied	Zenex	DDH	1123
				RC	3553
	2007	Allied	Allied	DDH	276
	2008	Allied	Allied	MET	989
				RC	14607
	2009	Allied	Allied	RC	86
	2010	Allied	Allied	DD	31
				Total	**57792**

Table 13-14: Drilling Samples by Period and Drilling Method – Pigiput Prospect

Prospect	Year	Company	Drill Company	Hole Type	Samples
PIGIPUT	1984	Kennecott	PNG Drillers and Wallis Drilling	DDH	526
	1985	Kennecott	PNG Drillers and Wallis Drilling	DDH	3047
	1988	Kennecott	PNG Drillers and Wallis Drilling	AC	6187
				DDH	1368
				MET	24
	1989	Kennecott	PNG Drillers and Wallis Drilling	AC	2466
				DDH	1352
	1996	Nord	PNG Drillers	RC	899
	1997	Nord	Pacific Drilling Services	DDH	2432
				RC	1985
	2005	Allied	Zenex	DDH	825
				RC	791
	2006	Allied	Zenex	DDH	115
				RC	444
	2008	Allied	Allied	DDH	1528
				RC	7286
	2009	Allied	Allied	DDH	1317
			Capital	DDH	151
				Met	0
				RC	714
	2010		Allied	RC	3
			Allied	DDH	20
				Total	**42373**

Table 13-15: Drilling Samples by Period and Drilling Method – Samat Prospect

Prospect	Year	Company	Drill Company	Hole Type	Samples
SAMAT	1985	Kennecott	PNG Drillers and Wallis Drilling	DDH	333
	1987	Kennecott	PNG Drillers and Wallis Drilling	AC	958
	1988	Kennecott	PNG Drillers and Wallis Drilling	AC	8246
				DDH	557
				MET	19
	1989	Kennecott	PNG Drillers and Wallis Drilling	AC	6683
				DDH	550
				MET	9
	1996	Nord	PNG Drillers	MET	19
				RC	2325
	1997	Nord	Pacific Drilling Services	DDH	730
	2006	Allied	Allied	RC	59
			Zenex	DDH	334
				RC	970
	2007	Allied	Allied	DDH	427
				RC	377
	2008	Allied	Allied	DDH	82
				RC	1605
	2010	Allied	Allied	RC	12
		Allied	Allied	DDH	3
				Total	**24298**

Table 13-16: Drilling Samples by Period and Drilling Method – Botlu Prospect

Prospect	Year	Company	Drill Company	Hole Type	Samples
BOTLU	1986	Kennecott	PNG Drillers and Wallis Drilling	DDH	1579
	1987	Kennecott	PNG Drillers and Wallis Drilling	AC	1464
				DDH	661
	1988	Kennecott	PNG Drillers and Wallis Drilling	DDH	218
	1989	Kennecott	PNG Drillers and Wallis Drilling	AC	1037
				DDH	123
	1996	Nord	PNG Drillers	MET	179
				RC	294
	1997	Nord	Pacific Drilling Services	DDH	315
				RC	3020
			PNG Drillers	DDH	25
				RC	891
	1998	Nord	Pacific Drilling Services	RC	447

	2010	Allied	Allied	RC	2
				DDH	10
				Total	**10265**

Table 13-17: Drilling Samples by Period and Drilling Method – Pigibo Prospect

Prospect	Year	Company	Drill Company	Hole Type	Samples
PIGIBO	1986	Kennecott	PNG Drillers and Wallis Drilling	DDH	302
	1987	Kennecott	PNG Drillers and Wallis Drilling	AC	551
	1989	Kennecott	PNG Drillers and Wallis Drilling	AC	279
	1997	Nord	Pacific Drilling Services	DDH	266
				RC	3067
			PNG Drillers	RC	60
	1998	Nord	Pacific Drilling Services	DDH	398
				RC	98
	2005	Allied	Zenex	RC	187
	2008	Allied	Allied	RC	1309
	2009	Allied	Allied	DDH	2405
				RC	2742
				Total	**11664**

Table 13-18: Drilling Samples by Period and Drilling Method – Other Prospects

Prospect	Year	Company	Drill Company	Hole Type	Samples
ADORA	1995	Nord	PNG Drillers	RC	110
	1997	Nord	Pacific Drilling Services	RC	377
	2006	Allied	Zenex	RC	268
	2007	Allied	Allied	DDH	540
				RC	908
	2008	Allied	Allied	DDH	188
				Total	**2391**
BEKOU	1997	Nord	Pacific Drilling Services	DDH	507
				RC	1233
	2006	Allied	Zenex	RC	1120
				Total	**2860**
KEKENMINDA	1998	Nord	Pacific Drilling Services	RC	613
				Total	**613**
PATAN	1989	Kennecott	PNG Drillers and Wallis Drilling	AC	557
	2007	Allied	Allied	RC	847
				Total	**1404**
PIGICOW	1987	Kennecott	PNG Drillers and Wallis Drilling	AC	489
	1997	Nord	Pacific Drilling Services	DDH	201
				RC	1353

Prospect	Year	Company	Drill Company	Hole Type	Samples
	1998	Nord	Pacific Drilling Services	DDH	50
				RC	99
	2004	Allied	Zenex	RC	28
	2006	Allied	Zenex	DDH	604
				RC	3308
				Total	**6132**
PLANT	1995	Nord	PNG Drillers	RC	111
	2006	Allied	Zenex	DDH	438
				RC	56
				Total	**605**
OTHER				**Total**	**14005**

13.1 Surveying

13.1.1 Collar Surveys

All hole collars were surveyed (by EDM) during Kennecott's initial exploration program and selectively (51 holes) validated by Nord in mid-1994. The collars of all of the holes drilled by Nord in 1994-97 were surveyed by EDM. During the period 1994-96 detailed topographic surveys were carried out by CPS Palanga, a group of surveyors based in Port Moresby, to yield 2 m contours in the Samat area and 5 m contours in the Sorowar and Pigiput areas. CPS Palanga produced topographic maps showing 10 m contours based on its surveying and field observations, probably the best resolution that could be expected from this survey point density.

During 1996-97 additional surveying located all of the latest drill collars, numerous 5 m channel samples (usually every fifth endpoint), the edges of track formations and many control points. All these data were used to construct topographic surfaces to constrain Nord's oxide Resource models. In 1995, a topographic map over the whole of Simberi Island was prepared using photogrammetry.

All 2004/2005, Sorowar drill hole collars were surveyed by APS surveyors using traditional EDM instruments. Before 2004, drill hole collars and control points were surveyed by CPS Palanga using the same method. An audit by McMullen Nolan and Partners Surveyors Ltd, using two dual frequency GPS units, determined that the Simberi survey had a very high accuracy, with drill holes and control points is accurate to within a few centimetres.

The McMullen Nolan and Partners survey was undertaken to investigate operational problems related to GPS usage and to create a plane coordinate system for the Tabar Islands (Table 13-19) based on UTM WGS84 (IRTF2003).

Table 13-19: Tabar Islands Grid (TIG) Parameters

Element	Value
Ellipsoid	GRS80 (or WGS84)
Projection	Transverse Mercator
Latitude of Origin	0°
Central Meridian	151°58'
False Easting	40,000
False Northing	500,000
Scale Factor	1.00002985
Zone Width	15'
Datum	ITRF2000 (Epoch 12.7.2005)

The conversion formula between the original Simberi Grid and TIG is:

- TIG East = (Sim East * 1.0000184742) + (Sim North * 0.0000723594) + 34993.496.
- TIG North = (Sim North * 1.0000184742) – (Sim East * 0.0000723594) + 104.755 + 200000.

The original Tatau and Tabar grid (the "Tatau Grid") is directly related to the Simberi grid by the addition of constants to the easting and northing. The most extreme difference is for the southernmost hole, hole BA01 at Banesa, where the difference between WGS84 and TIG 31 m east and 14 m north.

The high quality ground topographic surveys over each of the prospect areas remain valid. Between the prospects and over the rest of the island, there is now a topographic digital elevation model (DEM) derived from the airborne geophysical survey. This is an improvement over the previous photogrammetric DEM though, where no ground truthing is done, still represents tree canopy elevations.

Since 2007, CPS Palanga has routinely surveyed major roads, tracks and collars used for drilling and access to mining areas. The topographic surfaces created in these areas are continually updated with the new data, increasing their overall accuracy.

Since late 2007, an additional QC step was introduced to record all collars with a GPS used to cross check the values delivered by the surveying company. This ensures that hole identifiers have not been transposed. The collars are also checked using modelling software against tracks and geographic markers.

13.1.2 Downhole Surveys

Downhole surveys are restricted to some of the diamond drill holes. Nord used a downhole Eastman camera to survey holes at Sorowar (11), Samat South (4), Pigiput (10) and Botlu (3). The lack of downhole surveys for the bulk of the drill holes should have limited effect on the determination of the relatively shallow oxide resources.

Since 2006, Allied has surveyed the bulk of diamond core holes using a Ranger 1100 multi-shot downhole camera. Surveys are typically once the hole is drilled, generally at 30 m intervals from 15 m downhole.

In some instances, surveys were not be taken due to hole collapse or un-retrievable drill strings in the ground. In 2009, the surveying technique was modified to include a reading at 15 m below surface, with this 15 m shot is used as an estimation value for the surface dip and azimuth. Previously, the practice was to use the rig mast line up orientation as the top of hole survey.

Current drill contractors, Capital Drilling, record downhole surveys with an Easy Track multi-shot camera, on a run in basis with a shot taken every 30 m.

All downhole survey data is reviewed and outlier readings are invalidated.

13.2 Drill hole Database

A technical review of the geology, data collection and resource estimation relating to the Pigiput, Samat South, Samat North and Samat East gold deposits was made during June and early July 1996 by Behre Dolbear Associates, who concluded that the work done and the approach was satisfactory. The geology, data collection and resource estimation relating to the Sorowar, Botlu and Pigibo gold deposits have not been externally reviewed

Drilling in 2004 and 2005 by Allied Gold was subject to significant external review. Golder Associates visited the site in April 2004 and reviewed data collection procedures. The laboratory procedures on Simberi Island were reviewed by ALS Chemex in October 2004 and found to be satisfactory.

In early 2009, the historic data was transferred into a Maxwell's Datashed model. Subsequently generated data is now entered directly into the Datashed SQL database directly, according to the flow chart illustrated in Figure 13-1 and Figure 13-2. The flow chart is a formalised adaptation of the practice followed by Allied since 2007.

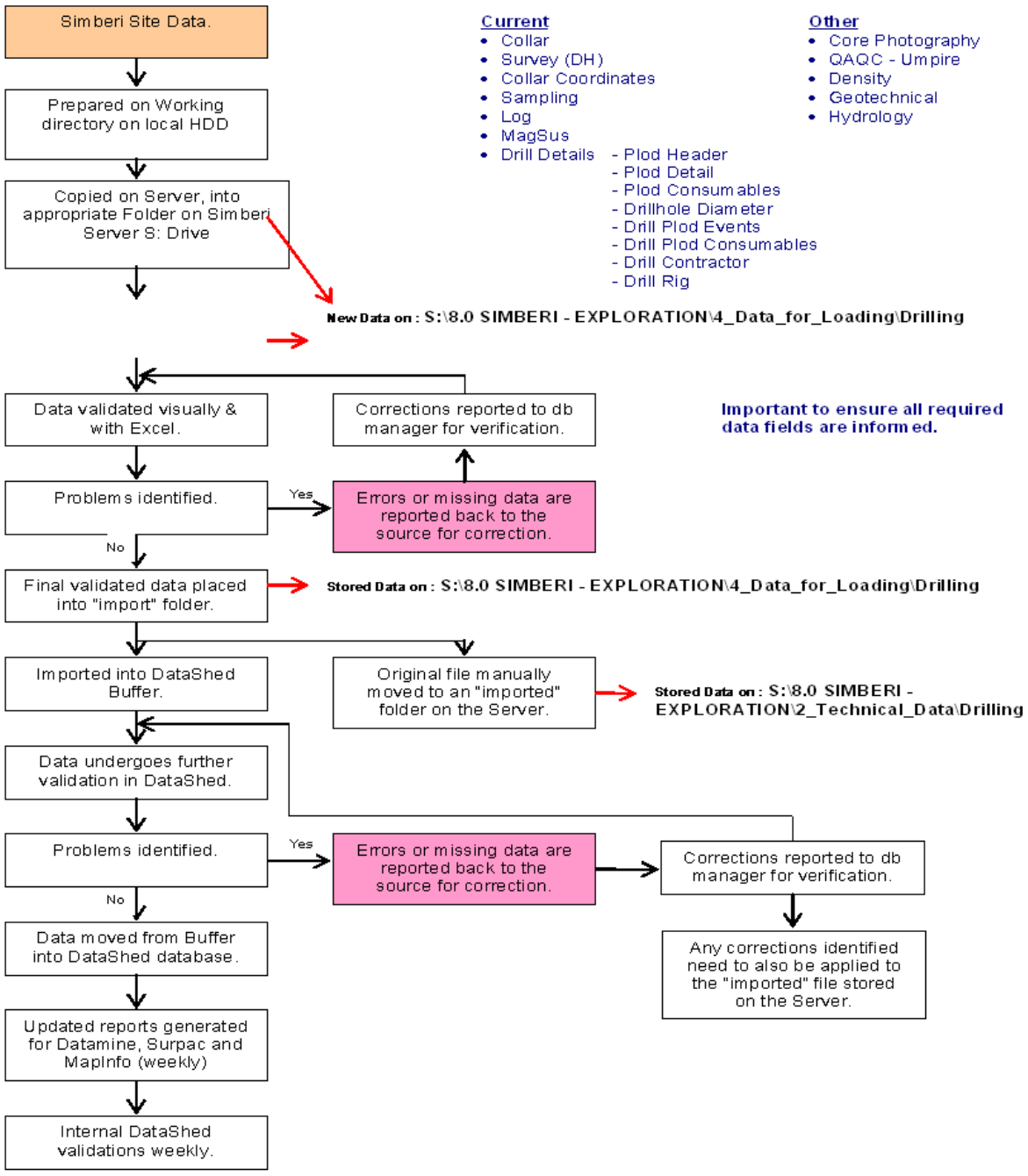

Figure 13-1: Data Flow – Site Data

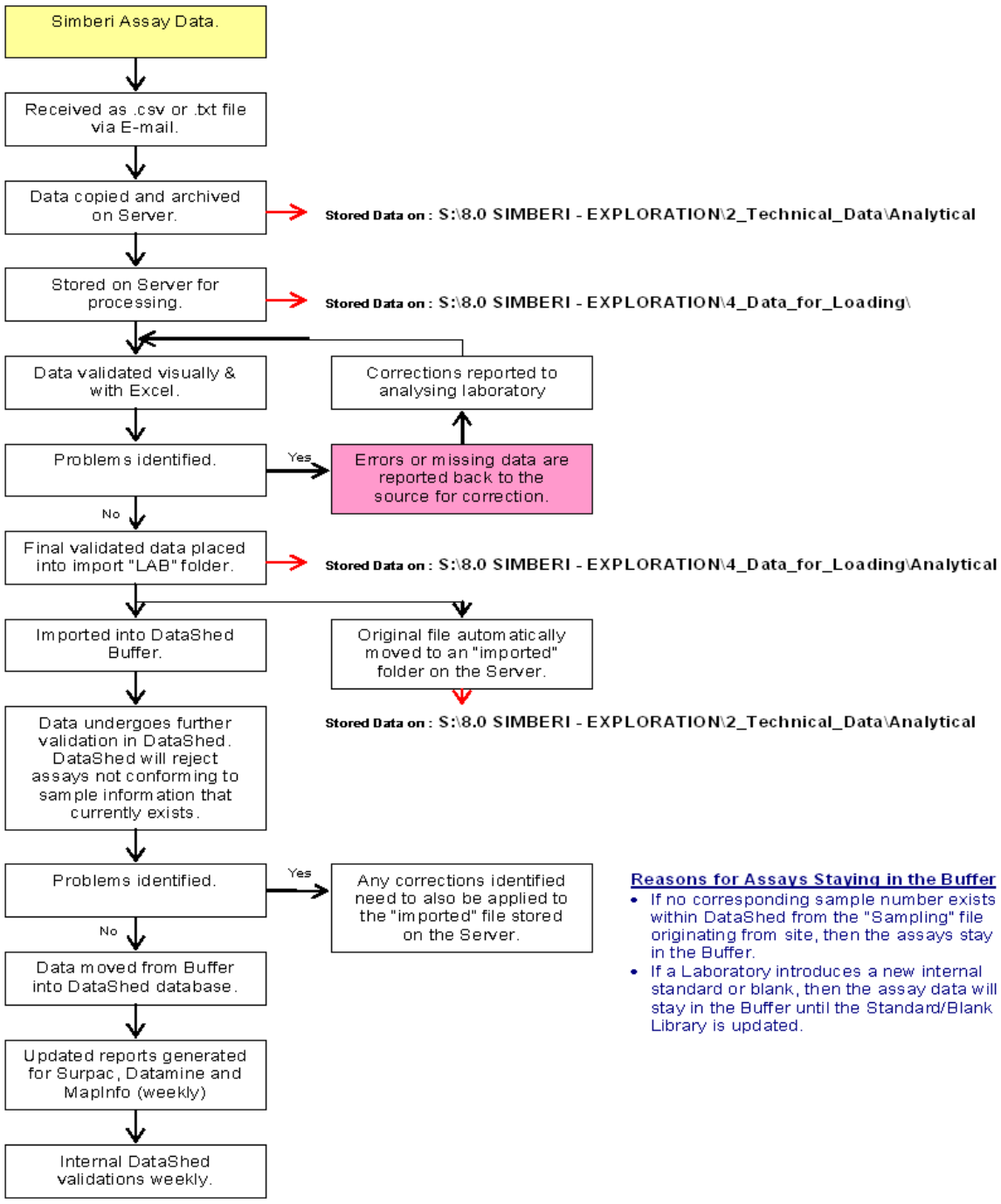

Figure 13-2: Data Flow – Assays

14.0 SAMPLING METHOD AND QUALITY CONTROL MEASURES

14.1 Kennecott Sampling

During the early part of the aircore drilling program (up to approximately RC320, February-May 1989), each 1 m sample was collected from a cyclone in a calico bag. The sample was dried and jaw crushed to less than 7 mm and a 1.5 kg riffle split sub-sample dispatched for assay. Approximately 100 g of chips from each sample were sieved and retained in transparent plastic boxes. The remainder of each sample, approximately 7 kg, was stored on site but are now not accessible because the calico or plastic bags containing them have disintegrated.

At the laboratory, the 1.5 kg jaw-crushed sample was split to 250 g, disc pulverised to -80 mesh, further split to a 50 g sub-sample and finely pulverised for assay (Figure 14-1).

Lack of correlation between duplicate and original sample assays obtained from a preliminary check assay program led Kennecott to believe that the sampling procedure was deficient. To address this issue, the sample preparation procedure was revised (Figure 14-2). The original sample was then dried, jaw-crushed, hammer-milled to a -80 mesh then a 250 g split was taken and sent to the laboratory.

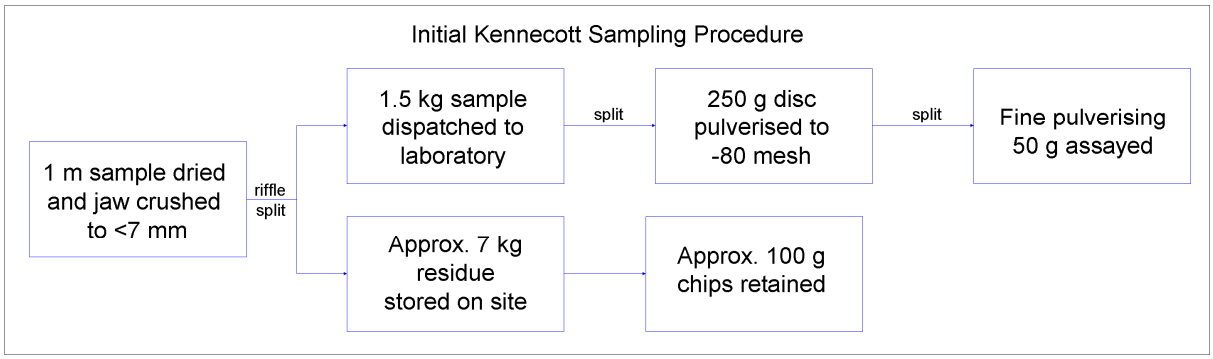

Figure 14-1: Kennecott – Initial Aircore and RC Sample Preparation Chart

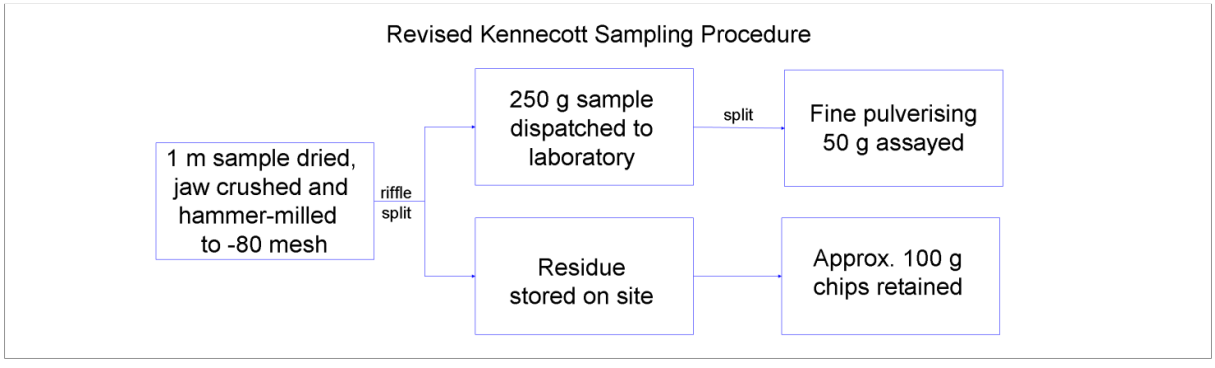

Figure 14-2: Kennecott – Revised Aircore and RC Sample Preparation Chart

The 1 m diamond drill core samples were cut in half using a diamond saw, dried, jaw crushed, hammer-milled to -30 mesh then a 200-250 g sub-sample was pulverised to -80 mesh (Figure 14-3). The pulverised sample was dispatched to the laboratory where a 50 g split was finely pulverised for assay.

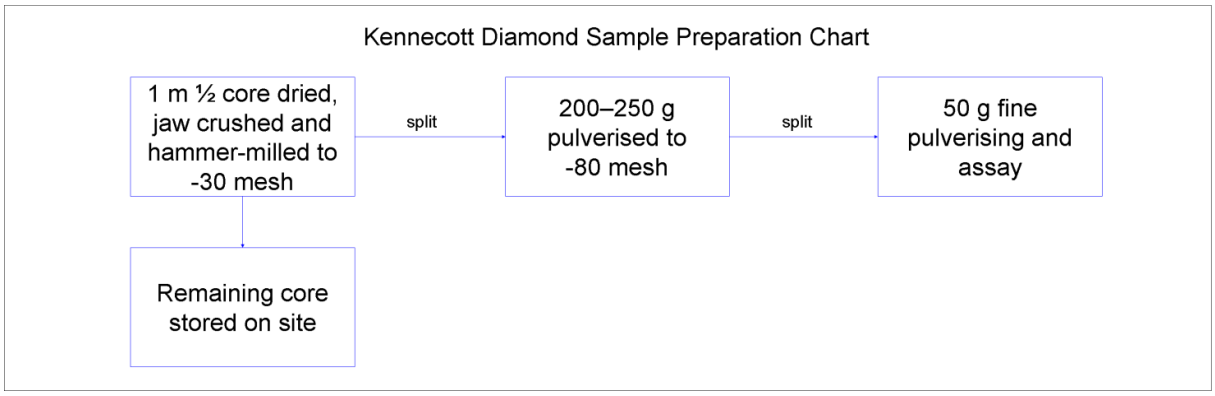

Figure 14-3: Kennecott – Diamond Core Sample Preparation Chart

14.2 Nord Sampling

Nord sampled aircore, RC and diamond holes every 1 m. Approximately 100 g of every RC sample (chips) were washed, dried and retained for reference. Aircore and RC samples weigh around 7-9 kg (depending on bit diameter). They were collected in polyweave bags direct from a cyclone. Every aircore and RC sample was dried in a wood-fired oven and hammer-milled at a Nord sample preparation facility, located on Simberi Island, to approximately -30 mesh (Figure 14-4). The sample preparation facility was supervised by contract personnel from Astrolabe Pty Ltd, an analytical laboratory in Madang. A 1 kg subsample was riffle split for dispatch for assay and the remainder stored.

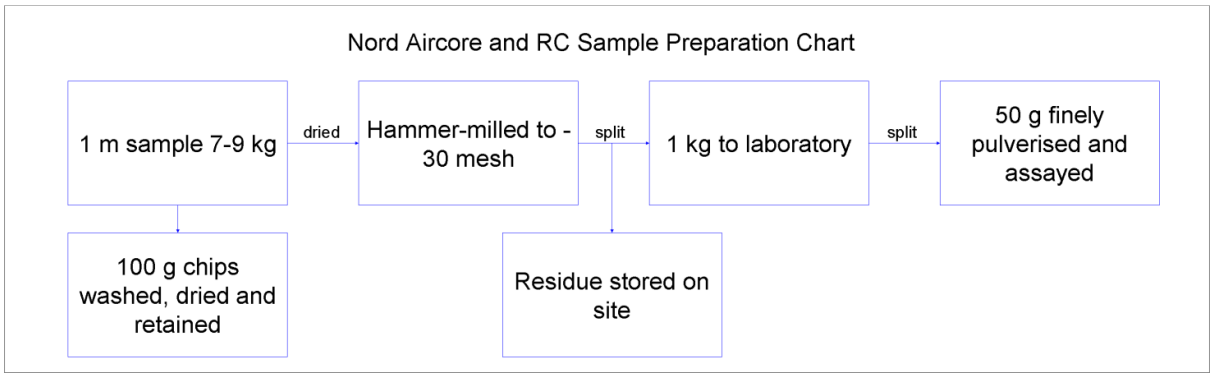

Figure 14-4: Nord – Aircore and RC Sample Preparation Chart

Diamond core was photographed, logged and cut in half using a diamond saw. One half was dried, jaw-crushed, hammer-milled and reduced to a 1 kg sub-sample using a riffle splitter. The sub-samples were dispatched to Astrolabe (Madang, PNG) for final preparation and assay up until September 1996. At the laboratory the 1 kg sub-samples were dried, pulverised and a 50 g sub-sample was fire assayed for gold using an AAS finish (Figure 14-5). The laboratory maintained its own quality control with two Gannet gold standards and blanks. After September 1996, the samples were dispatched to Australian Laboratory Services (ALS) in Townsville, Queensland, for preparation and assay using the same method. Pulp rejects from the ALS Laboratory are in secure storage.

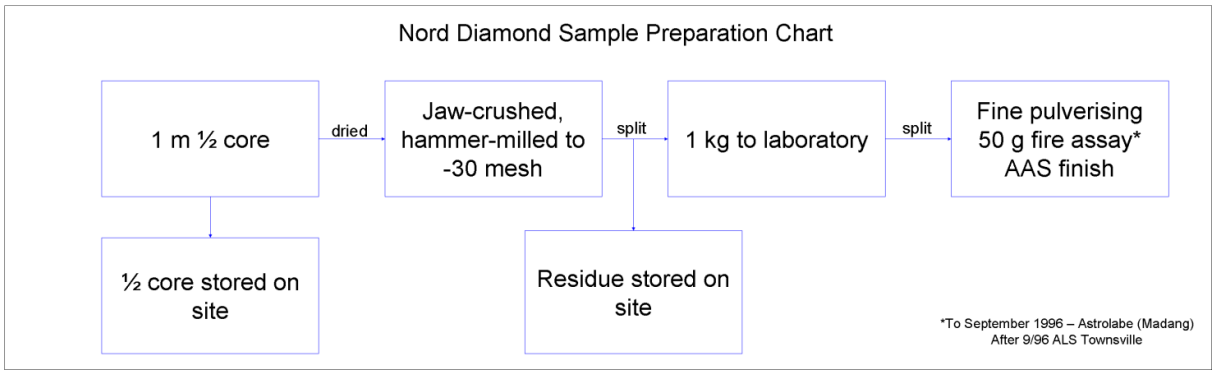

Figure 14-5: Nord – Diamond Sample Preparation Chart

14.3 Allied Sampling

14.3.1 RC Sample Processing

Whole RC samples were collected at 1 m intervals. Approximately 100 g of chips from each RC sample were washed, dried and retained for reference. The remainder of the samples were dried on galvanised core trays in wood fired copra driers. Each sample was then jaw-crushed, hammer-milled to -80 mesh and reduced to two approximate 1 kg sub-samples using a riffle splitter. One 1 kg sample was hammer-milled to -30 mesh and the other "reject" split was archived on site for a minimum of 3 months after assays were returned (Figure 14-6).

The 1 kg crushed samples were dispatched to ALS and finely pulverised. A 50 g sub-sample was fire assayed for base metal analysis and the remainder for stored at their facility in Garbutt, Queensland. The Simberi processing equipment was flushed with glass before each hole was processed.

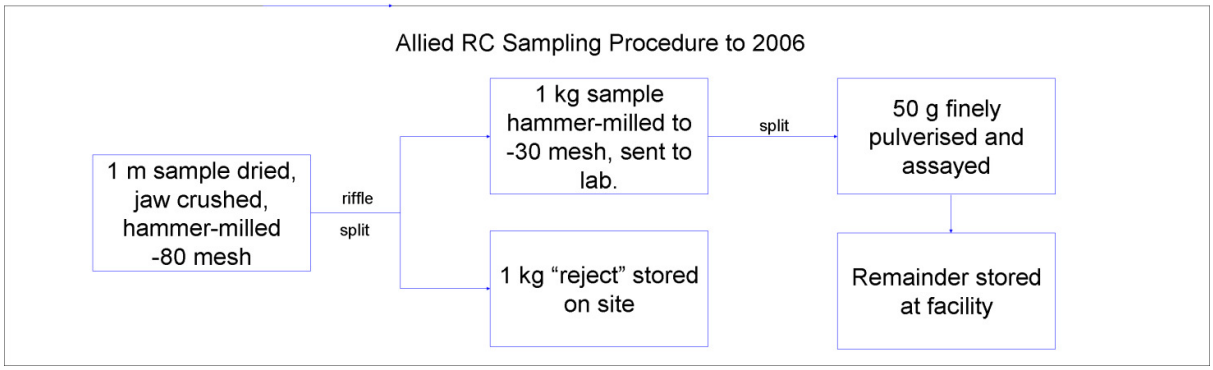

Figure 14-6: Allied – RC Sample Preparation Chart

In mid-2008, a new core shed and sample preparation facility was constructed with upgraded security and new sample processing equipment to reduce processing risks (e.g. contamination, theft, etc).

This allowed a change to the RC sampling and preparation procedures, as described below:

RC Field Technicians:

1. Collect whole samples from cyclone in large polyweave bags. Bags are to be marked with hole identification (Id) and depth-from and depth-to information (in metres).
2. Weigh entire sample (procedure started later 2008), record in notebook along with condition of sample (Wet/Dry/Moist).
3. Place sub-samples in calico bags with the following information recorded on the bag: date collected, the hole Id, prospect, depth-from, depth-to and whether the sample is a "reject" or duplicate.
   - 3.1 For dry/damp samples, reduce volume using riffle splitter to approximately 500 g for processing and approximately 500 g for a "reject" archive sample.
   - 3.2 Spear wet samples to obtain two 800 g sub-samples, one for archive and one for processing (quantity may change depending on the capacity of the drying oven).
   - 3.3 Collect riffle split field duplicate samples.
   - 3.4 Collect samples for reference chip trays, by washing out the fines using water and an appropriate sieve, and record the relevant data on the chip tray (hole Id, prospect, depth-from, depth-to).
4. Send sub-samples, grouped by hole Id., to sample preparation technician in a logical order and notify a geologist of their readiness.

Sample Preparation Technician:

1) Place sub-samples in the electric ovens until dry, recording the date the sample arrived and where from, and actively update the ledger and white board of all sample movements during preparation.
2) When drying, samples can be turned and broken up if high clay content causes significant clumping.
3) Receive samples Id and QC insertion sheets from geologist, locate blanks and required CRM samples. Field duplicates and Blanks must be processed in the sequential order as provided on the sampling sheet.
4) Sample pulverising:
   a) Clean bowls and pucks at the beginning of the run using the provided material (typically clean glass). Process samples in order and as a single batch per hole. Bowls are to be blown out with compressed air and a brush. If caking has occurred then bowls are to be cleaned with crushed glass.
   b) RC samples of up to 600 g are to be milled in the LM2 to obtain a 90% pass through 75 microns for dispatch to the Laboratory. Running time is approximately minimum 3.5 minutes. If the sample feels gritty after an initial run, it is pulverised again.
      i) Three samples are to be collected directly from the bowl (QC samples plus an extra).
      ii) Approximately 100 g samples are dispatched in 3 × 4 wire tie craft bags. To ensure samples are easily locatable and sorted, groups of 10 samples are placed in a larger poly bags with sample number ranges recorded on them.
      iii) The 150-200 g archive pulp samples are stored in 5 × 9 wire tie craft bags for permanent storage. All samples are to be bagged into the large poly bags then into polyweave bags.
      iv) To collect pulp duplicates – resample from the bowl as indicated on the sampling list provided by the geologist.
      v) Collect the Inter-laboratory check samples at the rate of (2-3%) and record in the ticket book the parent Id. Inter-laboratory check samples are stored separately ready for dispatch.

vi) One teaspoon per sample is collected within a batch and placed into a polyethylene bag (mixing samples together) this is for sizing fraction determinations where the mixed sample is weighed and the wet sample is sieved, the coarse fraction is weighed and the grid efficiency is determined. The generated fractions are included for analysis, at the end of the batch to determine whether grinding and sample homogenisation are adequate.

5) Samples ready for dispatch:

a) Once in a logical order with sequential sample Ids, the samples will be dispatched as a single hole batch to avoid risks associated with sample swaps. Notify the geologist of hole Ids, sample number ranges and that the batch will be dispatched. When the required documents are received, place a copy in the box with the samples and a copy on the box, clearly addressed to the appropriate laboratory(s).

6) The preparation white board and ledger are updated accordingly.

7) The sampling sheet is returned to the geologist with any variations (e.g. if an alternate CRM was used or if a variation in sample Id assignment per sampling interval occurred).

The RC sample flow chart used since mid-2008 is presented in Figure 14-7.

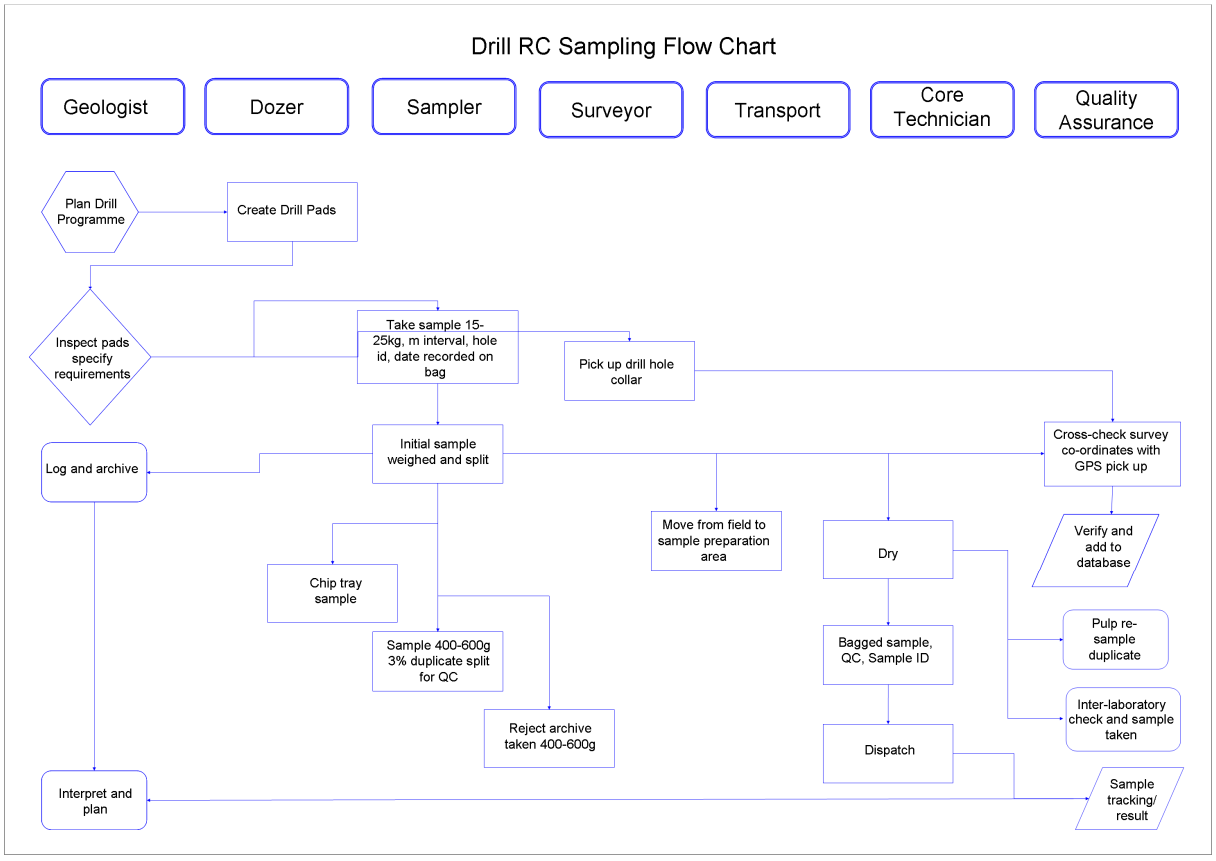

Figure 14-7: Allied – mid-2008 RC Sample Flow Chart

14.3.2 Core Sample Processing

Before mid-2008, drill core samples were processed in a similar way to the RC samples. Core was sampled on 1 m intervals, cut in half using diamond saws and dried in wood fired copra driers on galvanised core trays. One half of each sample was stored on site in the secured core shed, the other half of each sample was crushed with a jaw crusher and split to two approximately 1 kg. One 1 kg sample was hammer-milled to -30 mesh and the "reject" sample was archived in the core shed for a minimum period of 3 months after assays were returned.

The Simberi processing equipment was flushed with glass before each hole was processed. The 1 kg samples were dispatched to ALS Townsville for pulverising and a 50 g sub-sample was fire assayed for base metal analysis (Figure 14-8). ALS forwarded the remainder of the 1 kg samples for storage at their facility in Garbutt, Queensland.

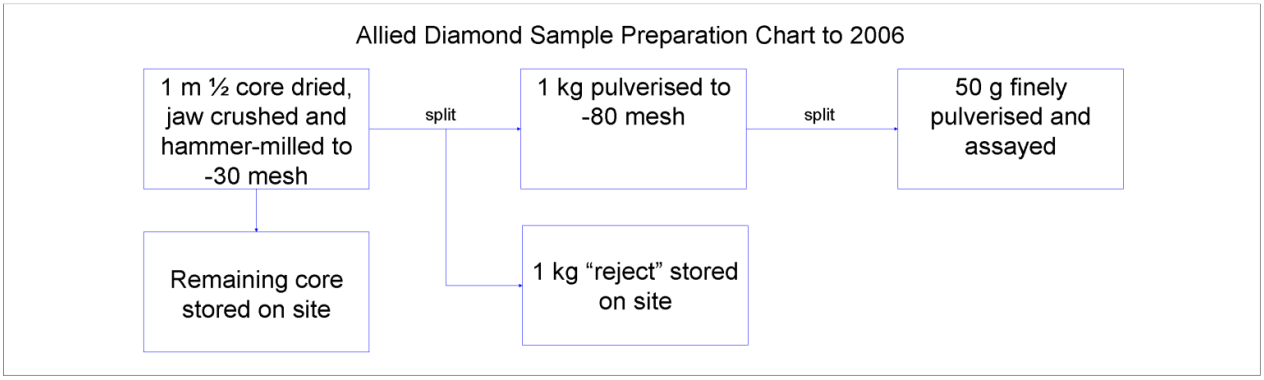

Figure 14-8: Allied – Diamond Core Sample Preparation Chart – to 2006

In early-2008, an Almonte automatic core saw was commissioned with the interest of reducing core handling and the advantage of closing core cradles to preserve the integrity of the core during cutting. Also, the security of the logging, preparation area and storage facility areas were upgraded with extra fencing, and repairs to damaged doors and windows. Furthermore, core trays were changed from galvanised metal to plastic, to remove the problem of tray deterioration due to sulphides reacting with the metal.

The inauguration of the new logging and sample preparation facility in mid 2008 allowed a change to the core drill sample methodology as described below:

Core Yard Technicians:

Note: Sampling of diamond core can only occur once the logging and technical operations have occurred.

1) Cutting core, using the Almonte automatic core saw, if the sample is not suitable for cutting then it is sampled by splitting with another appropriate method. Notes are made on the sampling list if variations or discrepancies have been identified and amended, after consultation with the responsible geologist (samples should be no smaller that 30 cm and no longer than 1.2 m). Samples are based on geology but have a maximum length of 1.2 m with the target being 1 m.

2) Following the sampling list, half core is put in calico bags ready for drying, bags are marked with the hole id, depth-from and depth-to information and date.

Sample Preparation Technicians:

Primary duties of the core yard technician include performing bulk density determinations, putting trays in sequential order, photographing of core and weighing the core.

After the geologist has provided the cutting and sampling list and has marked the core for cutting, the following procedure applies:

1. Samples are transported to sample prep in their groups and kept in a logical order where possible, and geologist are notified of readiness.
2. Preparation technician receives samples places in electric ovens until dry, the preparation ledger is updated recording the date arrived and location of samples in preparation, the information is updated with all sample movements.
3. Core is dried in the electric oven at 106°C, the Blank samples are added to the process cue.
4. Once dry the samples are crushed the using the Terminator Jaw Crusher, cleaning the jaws with compressed air in between samples.
5. Sub-samples of approximately 500 g for processing are generated using a riffle splitter, duplicate splits of crush are also taken in this step, the residual jaw crush is archived for a minimum of 3 months after assay results are returned and are only disposed of upon the Chief Geologist's consent.
6. Sample pulverising:
    - 6.1 Bowls and pucks are cleaned at the beginning of the run utilising the provided material (typically clean glass). Processing occurs with samples in order and as a single batch per hole.
    - 6.2 Approximately 600 g is processed in the LM2. Typical running time is 3.5 minutes. If the sample feels gritty after the run, mill the sample again.
    - 6.3 Typically 3 samples are collected directly from the bowl (more samples are required to include the QC samples):
        - 6.3.1 For pulp sample for analysis (approximately 100 g), place them in the 3 × 4 wire tie craft bags and package them so that they can be easily sorted and audited.
        - 6.3.2 An archive pulp sample (approximately 200 g) is collected and placed in the 5 × 9 wire tie craft bags for permanent storage. All samples are placed into the large polyethylene bags then into a polyweave bag and stored in the core shed.
        - 6.3.3 Pulp duplicate samples are collected as indicated on the sampling list as provided by the geologist.
        - 6.3.4 Inter-laboratory check samples are taken at the rate of 2-3%. The parent id is recorded in the ticket book and samples are stored separately and prepared for dispatch.
        - 6.3.5 One teaspoon of each sample within a batch is taken and placed into a polyethylene bag for mixing. This is for grind efficiency determinations where the generated fractions are to be included for analysis at the end of the batch.
7. Samples are dispatched in a clear and logical order with sequential sample ids and as a single hole per batch.
8. The preparation lab whiteboard and ledger are updated with all movements of the samples through the facility.
9. The sampling sheet is returned to the geologist, with any variations to the sampling being noted (e.g. if an alternate CRM was used, or if a variation in sample id assignment occurred).

The drill core sample flow chart used since mid-2008 is presented in Figure 14-9.

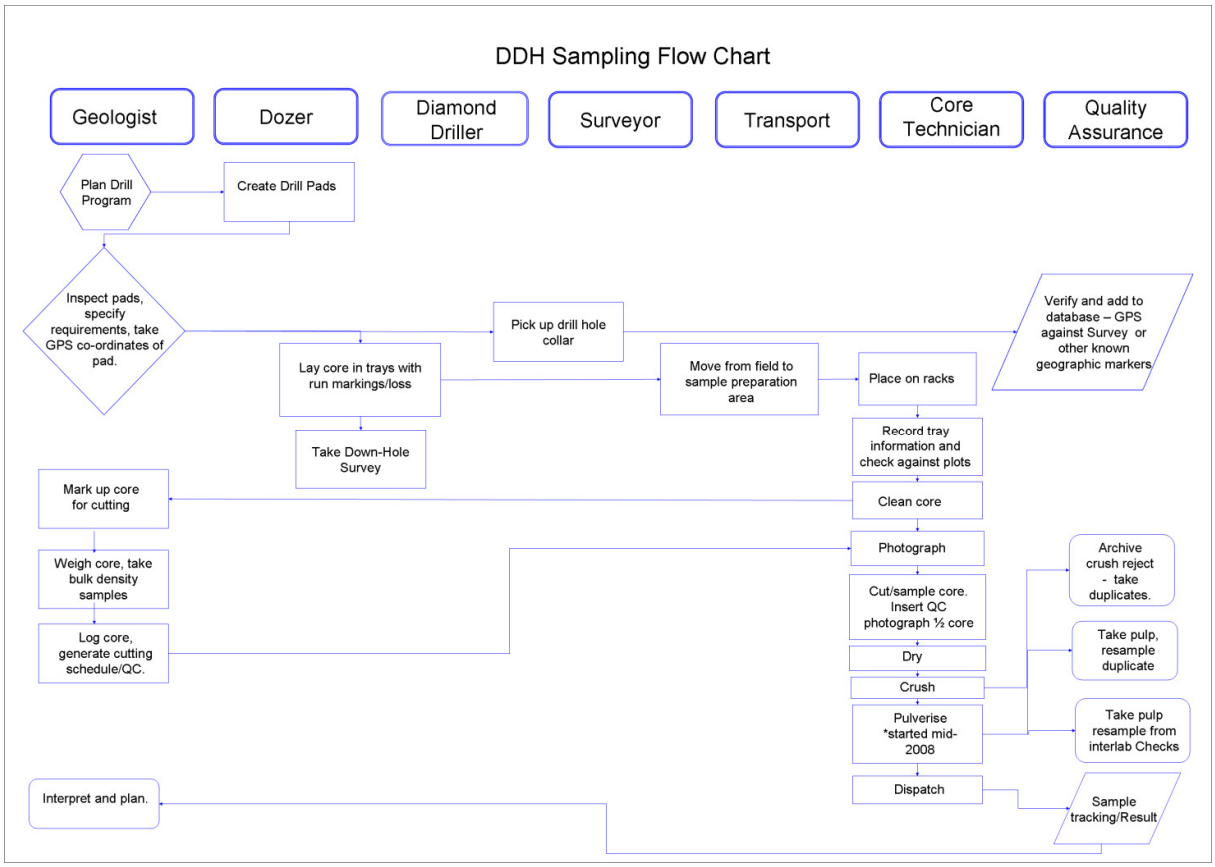

Figure 14-9: Allied – Post mid-2008 Diamond Drill Hole Sample Flow Chart

14.3.3 Procedural Variations

In late-2007, 11 drill holes (918 samples) were analysed by Kalassay Laboratories in Western Australia. In the first quarter of 2008 26 RC holes and 4 diamond holes (2,161 samples) were sent to Quantum Laboratories in Western Australia. Both laboratories performed Fire Assay on a 50 g charge. The use of both laboratories was discontinued due to slow turnaround times.

14.4 Bulk Densities

A bulk density of 1.7 t/m^3 was used in resource estimates prior to 1994 for Sorowar, Pigiput and Samat deposits. This was based on laboratory tests done by Kennecott comprising 179 diamond hole samples from Samat South (41), Pigiput (69) and Sorowar (69). Kennecott commented that the samples that were tested were generally more competent, so there may have been some selection bias in the results.

In late 1994 Nord carried out some dry bulk density determinations of near-surface material exposed in bulldozer cuts crossing the prospects. This was achieved by forcing a steel box of known volume into the material and weighing and drying the excavated material. These tests yielded average values of only 1.0 t/m^3.

Bulk densities of oxide material measured in drill core samples by Nord ranged from 1.0 at surface to 1.9 at 55 m depth. There was no relationship between grade and bulk density.

For the Sorowar resource estimate reported in Lycopodium, 2003 a total of 525 bulk density determinations were made from nine PQ diamond holes (SO15 – SO19, SO22 – SO24 and SO27). These holes were drilled within the 1995 conceptual pit, between 36 m and 115 m deep and for metallurgical or grade

characterisation purposes. These determinations showed that there was an increase in dry bulk density and a decrease in water content at depth.

Forty bulk density measurements were taken from one diamond hole (SO28) at Sorowar for the 2005 FS update. Results were consistent with the existing data.

Since 2006, Allied have carried out bulk density measurements on Sorowar, Pigicow and Pigiput deposits (Table 14-1), bringing the number of measurements taken to over4,800. From 2008, bulk density measurements were routinely taken from full core before cutting and sampling.

Since 2009, a program of determining density by tray has been used to validate the point density measurements. Nearly 5,000 tray measurements have been made for core from Pigiput and Pigibo.

Table 14-1: Bulk Density Measurements – since Inception

Material	Prospect	Number of samples	Average Dry Density (tm^{-3})
OXIDE	Botlu	144	1.84
	Pigiput	589	1.65
	Pigicow	21	1.61
	Samat	50	1.42
	Sorowar	464	1.67
	Pigibo	89	1.78
OXIDE Total		**1357**	**1.67**
TRANSITION	Botlu	2	1.73
	Pigiput	264	1.92
	Pigicow	4	1.65
	Samat	32	1.57
	Sorowar	26	1.85
	Pigibo	72	1.94
TRANSITION Total		**400**	**1.89**
SULPHIDE	Botlu	201	2.01
	Pigiput	3568	2.13
	Pigicow	50	1.86
	Samat	218	2.05
	Sorowar	125	2.03
	Pigibo	342	2.12
SULPHIDE Total		**4504**	**2.12**
Grand Total		**6261**	**2.01**

14.5 Bulk Density Determination Method

A detailed description of the bulk density determination method used is given in "Physical and Metallurgical Characterisation of Diamond Holes SO15-SO19, Sorowar Oxide Deposit" by Tony Macfarlane, June 1995. In summary, the method was as follows.

A total of 525 bulk density determinations were made from diamond core from nine PQ diamond holes (SO15-SO19, SO22-24 and SO27) drilled to between 36 m and 115 m depth for metallurgical, geological or grade characterisation, within the 1995 Sorowar conceptual pit. Diamond hole SO19 had an average dry

bulk density of 1.69 t/m^3. Nord found an increase in dry bulk density and a decrease in water content with depth. They found no relationship between mineralisation (gold and arsenic values) and depth.

Additional dry bulk density determinations were made for diamond holes SO15-SO18, SO22-24 and SO27. Nord found small differences between the depth and bulk density relationship in holes in the centre and the north-east sector of the deposit. Near the centre of the 1995 conceptual pit dry bulk densities increased from approximately 1.0 t/m^3 at surface to about 1.8-1.9 t/m^3 at 40-55 m depth, after which the dry bulk density remains constant. This was located within an interpreted zone of deep oxidation affecting both porphyry and volcanoclastic/diatreme lithologies.

In the north-east, SO17 has a dry bulk density of approximately 0.8 t/m^3, near surface, increasing to 1.6 t/m^3 at 20 m depth then remaining constant. SO18 has a constant dry bulk density, of approximately 1.6 t/m^3, between 4 m and 35 m depth. These holes were collared at higher elevations than the others and intersected tuffs and diatreme breccias rather than porphyry.

It is clear that there is a progressive increase in dry bulk density with depth. Nord chose to model the BD using a spherical variogram to estimate between 1.0 t/m^3 at surface and 1.9 t/m^3 at and beyond 55 m depth. The spherical model parameters were nugget=1.0, sill=0.9 and range=55.

14.5.1 Allied – Since 2008

All bulk density determinations have been performed using the water displacement method. Samples are approximately 20 cm long and taken on average of 1 sample per tray. After sample weights are recorded wet, the samples are wrapped in cling-film and placed into a cylinder of water, with the volume of displaced water being recorded. The samples are then dried overnight in an electric oven at 106°C. Once dry the weight is again recorded. Density and moisture content is then calculated. Outliers are invalidated and new readings of weights and volumes are performed when possible.

In 2009, the density data interpretation changed from using depth in hole to a calculate sample depth below the topographic surface to give an increase accuracy and account for variations derived from angled drilling. The impact of the change in method for the Pigiput data set was minimal, as most of the density samples were derived from vertical drill holes.

Density measurements are also determined for each core tray, measured before the core is cut. The trays are weighed on an electronic balance and the volume estimated by multiplying the recovered length by the average area for the particular core size. The method provides a means of validating the point density data and is particularly effective in the sulphide zone.

15.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

15.1 Sample Preparation and Analyses

Sample preparation and analyses for the operators that have explored Simberi are detailed in Section 14.0.

15.2 Sampling Security

The chain of custody for sample handling and transportation is such that an Allied employee is with the samples at all time until they are locked up in a secure facility. This is common to both RC chip samples and diamond drill core. No samples are left unattended unless locked up. RC drilling is on day shift only while core drilling is done on double shift. The drill rig is visited by an Exploration Department staff member at least once a day.

The sample storage facility/core shed was examined by John Batista, Golder Associates in 2008.

The author considers the sample handling and management provides adequate security to minimise the risk of sample contamination or tampering.

16.0 DATA VERIFICATION

16.1 Kennecott

Between 1987 and 1989 Kennecott carried out a limited assay check program that identified some sample preparation procedure problems and an assay bias in results reported by Pilbara Laboratories in Lae. The bias was identified by standard samples returning values fluctuating between 10% above and below the recommended values, with the degree of bias varying month by month.

Kennecott re-assayed 1,062 Lihir drill core samples at Fox Laboratories, Sydney and Comlabs, Perth, during 1989, and calculated that there was a Pilbara Laboratories bias of +3.9%. It was then proposed that a correction factor of 0.9615 to be applied to the original Pilbara Laboratory assays. In September 1989, consultant Harry Parker evaluated the bias issue and proposed a factor of 0.959 be applied.

Between October 1989 and March 1990 Fox Laboratories in Sydney re-assayed a total of 3,208 Simberi aircore samples and 187 standards, previously assayed by Pilbara Laboratories. The RC samples selected for re-assay were nominally above 2.0 g/t Au and included RC samples that had previously been only jaw-crushed on site before taking an assay split. However, not all of the original samples which had assayed above 2.0 g/t Au were available. For the re-assay program, new samples for assay were prepared by hammer-milling the remaining jaw-crushed bulk rejects to -30 mesh (0.6 mm) and a 250 g split taken for dispatch to the laboratory. If the bulk rejects had already been hammer-milled to -30 mesh, a new 250 g split was taken.

Kennecott evaluated the results of the re-assay program in March 1990 for its oxide resource calculations and determined a Pilbara bias of +8.4% for oxide zone samples. Only the oxide zone samples in the drill holes required for the resource estimates were corrected using the factor. In 1992 a more exhaustive evaluation was carried out, dividing the data into oxide, transition and sulphide and grade classes between 0.0-3.3 g/t Au, 3.3-8.3 g/t Au, 8.3-14.4 g/t Au and above 14.4 g/t Au. Overall Pilbara bias figures were determined of +6.1% for oxide, +10.3% for transition and +9.2% for sulphide.

16.1.1 Nord

Every 20th 1 m drill interval (5% of the samples) was re-sampled twice and the additional (duplicate) 1 kg sample submitted for assay under a different sample number. Nord concluded that the majority of the duplicate pairs agreed well.

Nord also prepared a range of internal standard samples by taking about 32 kg (dry weight) of oxide material from bulldozed benches where 5 m channel sample gold grades were available. Each standard was dried, hammer-milled and split into 32 × 1 kg sub-samples.

One internal standard was submitted for assay with every 50 drill samples. Nord considered agreement between the assays was acceptable.

16.1.2 Allied

Allied's sample preparation and analytical control procedures include the use of blanks to monitor contamination, duplicates to test splitting and milling efficiency and standards to monitor analytical accuracy and precision

From the Feasibility Update in 2003 to 2005, Allied's QC sample insertion rate remained the same, with one standard or control sample every 50th sample and duplicates every 20th sample.

During this period, Au assays for 288 standards showed precision well within the deemed acceptable limits of mean (+/- two standard deviations). Au assays for 574 duplicates, representing 4.2% of the samples assayed show good agreement with a correlation coefficient of 0.994. In addition, Au assays for 570 samples submitted to a second laboratory, as a cross-laboratory check, also showed good agreement, with a correlation coefficient of 0.996 (Hastings, 2005). Blanks were collected from beach sands some distance from the Pigiput Bay industrial area and expected to contain no trace of any of the elements tested.

Initially five locally prepared standards were in use at Simberi. These standard samples (179312, 179313, 179314, 179315 and 179316) were collected from the proposed Sorowar pit area. All the standard samples were processed once and separate from all other samples. Each standard sample was then subdivided into a total of 64 splits, identified as Sample_id/1 to Sample_id/64. The sub-sample standards were packed in large numbered polyethylene bags and stored in the on-site preparation laboratory. The sample preparation crew were instructed to use these samples randomly from the various bags.

From late-2006, QC sample insertion was increased to 10%. Phase 5 (from 2008) saw the switch from in-house standards to commercially produced (Geostats Pty Ltd) CRM materials with a spread of gold grades and sulphur contents (Table 16-1). The blank material was changed from carbonate beach sand to crushed local barren intrusive to better approximate the volcanic matrix of the drill hole samples.

Table 16-1: Allied – Commercially prepared CRM Standards used 2008 to April 2009

Provided CRM details			Deemed Acceptable Bounds (defined by standard deviations)			
Product Code	Mean	SDev	Mean +1SDev	Mean -1SDev	Mean +2SDev	Mean -2SDev
G901-7	1.52	0.06	1.58	1.46	1.64	1.40
G907-6	7.25	0.29	7.53	6.96	7.82	6.68
G306-1	0.41	0.03	0.43	0.38	0.46	0.36
G306-2	1.05	0.07	1.12	0.98	1.19	0.90
G905-7	3.91	0.14	4.05	3.76	4.19	3.62
G01	0.02	0.02	0.04	0.00	0.06	-0.02
G306-4	21.57	0.78	22.35	20.79	23.13	20.01
G306-6	48.53	2.29	50.82	46.24	53.11	43.95
G906-2	2.46	0.11	2.57	2.35	2.68	2.24
G302-10	0.18	0.03	0.21	0.15	0.24	0.12

Sampling practice and QC protocol.

Between drill holes, sample preparation equipment is cleaned with crushed glass and compressed air. Between samples, the same equipment is cleaned with compressed air and a brush. If caking of sample on to components occurs, the equipment is cleaned with a glass run.

1) RC samples split at the rig site to 400 g, samples, and a rig split archive kept for 3 months after drilling. The samples are placed in oven at 106 degrees Celsius, and dried overnight, then crushed with the terminator jaw crusher, to a nominal <2 mm. The whole sample is pulverised in the LM2 to a nominal 85% pass 75 microns. Two samples are taken; 1 ~+50 g for dispatch, and a +100 g pulp archive. Currently 5% of the sample pulps are sub sampled to 50 g FAA for inter lab checks.

2) Diamond drill samples, are cut in half or split with knife if soft oxides. Samples are dried and crushed, the crush is split using a riffle splitter to 400 g with the split rejects archived indefinitely. The whole 400 g sample is pulverised in the LM2 to an 85% pass 75 microns. Two samples are taken; 1 ~+50 g for dispatch, and a +100 g pulp archive. Currently 5% of the samples are sub sampled to 50 g for inter lab check sampling.

With the commissioning of the on-site dedicated EXLAB Sample Analysis Facility in late Sep 2009, the pulp sample size was doubled to allow for confirmation analysis of mineralised zones by fire assay at a commercial laboratory. Also, to improve the reliability of the aqua regia digest used by EXLAB, pulverization time was increased to produce >95% pass 75 microns.

The “Grey Hill” blank is introduced into the sample stream in the form of jaw crushed -2 mm fraction. This fraction is riffle split down to 400 gram samples and stored in calico bags. Blanks are taken to the field, placed with the RC samples to come down for drying and pulverising. A second blank source has been identified and is now in use, it is composed of oxide ash, and extensive testing identified the material as consistently barren.

Certified Reference material is supplied by Geostats, a commercial producer of geochemical standards and a ten different CRM’s of varying grade and composition are used.

Duplicates are sampled in two forms:

- Field Duplicate (FD), a second riffle split of the drill cuttings or crushed core testing the first sample division stage, and
- Pulp Duplicate (PD), a second scoop of pulverised material from the LM2 bowl testing the second and final sample division stage.

16.1.2.1 Statistics of the Standards

A statistical analysis of the standards was performed by subscribing to the Horrats method of calculating a residual standard deviation (RSD) and comparing it with a predicted RSD. The RSD prescribes to the reproducibility of the standards assay values.

The Au has in all instances come in better than the predicted value (Table 16-2). The predicted value is 8 and the normal predicted is 16, hence the gold assay values for the standard are no-less than excellent.

After clustering the standard samples, analysed in 2008 and until Apr 2009, into seven populations, reported values were found to be rarely more than two standard deviations from the population average. The outlier, a “standard” used with samples from hole RC1362, does not fit into any of the populations. Gold contents of all samples from the batch eliminated the possibility of a sample swap and it appears likely the standard belonged for either Pop 2 or Pop 7 with either a spuriously reported gold or copper content.

Table 16-2: Allied – Summary of Reported Values Site Prepared Standards

Gold in Standards: RSD and Averages

Population/Group	Average Au (g/t)	Standard Deviation	RSD
POP 1	2.77	0.12	4.20
179312	2.76	0.11	4.14
POP 2	2.25	0.14	6.09
179313	2.25	0.13	5.68
POP 3	2.26	0.13	5.68
179314	2.27	0.13	5.62
POP 4	1.49	0.07	4.77
179315	1.47	0.07	4.98
POP 5	1.03	0.05	4.60
179316	1.03	0.04	3.68
POP 6	3.25	0.17	5.19
POP 7	3.69	0.17	4.75

As two different analytical laboratories and methods were used from late 2009, the CRM data was split accordingly, as its statistical analysis based upon laboratory.

CRM-type insertion errors have decreased since the Apr 2009 QC report with implementation of training and documentation procedures. A total of 6 out of 265 sent to ALS were recorded incorrectly. Their true identity could be identified from their reported base metals and Au contents.

CRM insertion errors in batches sent to EXLAB cannot be independently identified as EXLAB has no multi-element analysis equipment. Six likely CRM insertion record errors were identified in the 137 CRM samples dispatched. EXLAB batches that made up the commissioning data set are excluded, as a full pulp duplicate split was analysed by FAA at ALS Townsville as part of the EXLAB commissioning process

Table 16-3: Allied – Commercially prepared CRM Standards used since Apr 2009

Certified mean values for CRMs

Standard ID	Fire assay		Aqua-regia	
	Mean (Au)	Stdev	Mean (Au)	Stdev
G01	**0.02**	0.02	**0.02**	0.02
G302-10	**0.18**	0.03	**0.16**	0.02
G306-1	**0.41**	0.03	**0.41**	0.05
G306-2	**1.05**	0.07	**1.05**	0.09
G901-7	**1.52**	0.06	**1.53**	0.11
G905-7	**3.91**	0.14	**3.88**	0.24
G906-2	**2.46**	0.11	**2.4**	0.14
G907-6	**7.25**	0.29	**7.3**	0.32

The populations of the ALS analysed CRM's are graphically illustrated in Figure 16-1.

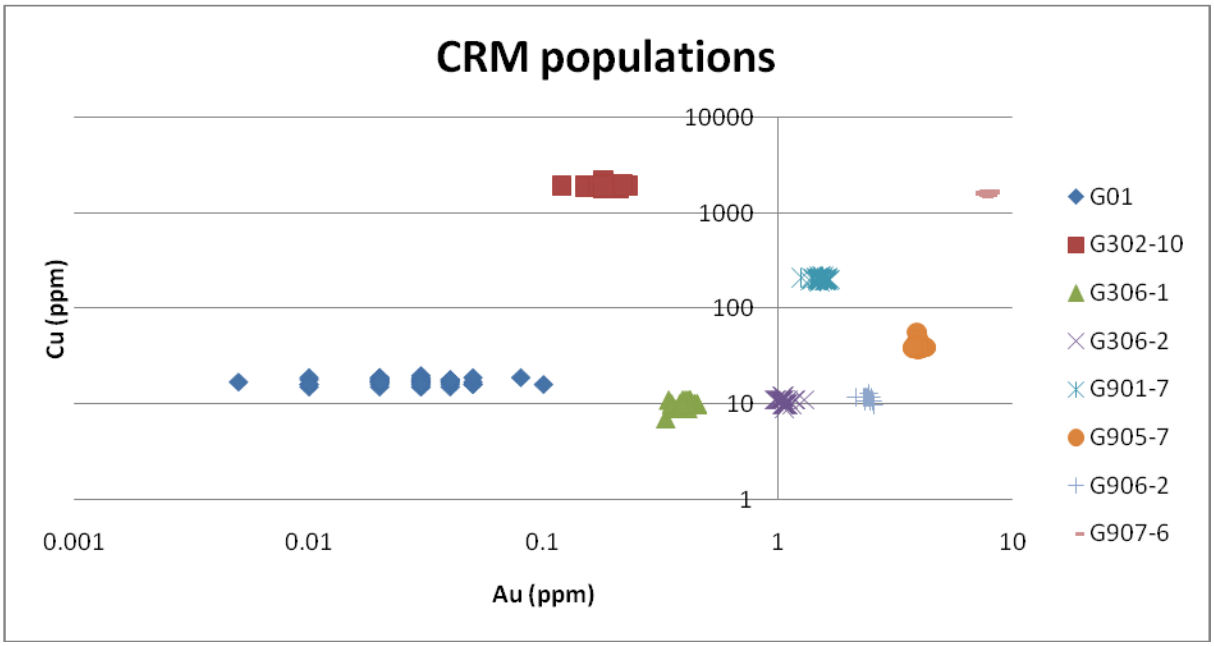

Figure 16-1: CRM population defined by Au vs. Cu values

The reference material has a similar very limited scatter to that found in the Apr 2009 QC Study.

Table 16-4: CRM Analyses – Apr to Dec 2009 – ALS_TSV Fire Assay

ALS_TSV – Fire Assay Results

REF SAMPLE	COUNT QC	AVE Au	Max Au	Min Au	Count> 2 STD DEV	% > 2 STD DEV	Count> 3 STD DEV	% > 3 STD DEV
CRM_G01	46	0.03	0.1	0.005	2	4.35	2	4.35
CRM_G302-10	29	0.18	0.23	0.12	0	0.00	0	0.00
CRM_G306-1	39	0.41	0.45	0.33	3	7.69	0	0.00
CRM_G306-2	31	1.07	1.3	0.97	1	3.23	1	3.23
CRM_G901-7	38	1.51	1.66	1.25	4	10.53	1	2.63
CRM_G905-7	28	3.98	4.25	3.77	3	10.71	0	0.00
CRM_G906-2	13	2.41	2.55	2.15	2	15.38	0	0.00
CRM_G907-6	41	7.40	7.98	7.01	1	2.44	0	0.00
TOTAL	265				16	6.04	4	1.51

Approximately 94% of certified reference material (CRM) sample results from ALS are within 2 standard deviations of the certified values. As noted in the previous QC report (April 2009), CRM_901-7 and CRM_G905-7, have higher variances than the other CRMs.

Table 16-5: CRM Analyses – Apr to Dec 2009 – EXLAB Aqua Regia digest/AAS finish

EXLAB Aqua-regia Results

REF SAMPLE	COUNT QC	AVE Au	Max Au	Min Au	Count> 2 STD DEV	% > 2 STD DEV	Count> 3 STD DEV	% > 3 STD DEV
CRM_G01	12	0.01	0.01	0.01	0	0.00	0	0.00
CRM_G302-10	12	0.13	0.18	0.01	2	16.67	2	16.67
CRM_G306-1	17	0.43	1.03	0.31	0	0.00	0	0.00
CRM_G306-2	11	1.03	1.1	0.92	0	0.00	0	0.00
CRM_G901-7	16	1.46	1.6	1.26	0	0.00	0	0.00
CRM_G905-7	16	3.63	4.09	0.16	1	6.25	1	6.25
CRM_G906-2	1	0.37	0.37	0.37	1	100.00	1	100.00
CRM_G907-6	10	6.97	7.57	6.47	0	0.00	0	0.00
TOTAL	95				4	4.21	4	4.21

NOTE G906-2 is a single CRM of its type in this data set. The outlier value is considered either a faulty analysis or a sample swap.

Excluding CRMs analysed during the commissioning period, approximately 96% of the CRM results determined by EXLAB are within +-2 STDEV of the certified values.

16.1.2.2 Duplicates

The results of duplicate pairs analysed in 2008 were plotted on a scattergram and the correlation-coefficient calculated. Gold returned a value of 97.5% correlation in the duplicate splits (Figure 16-2).

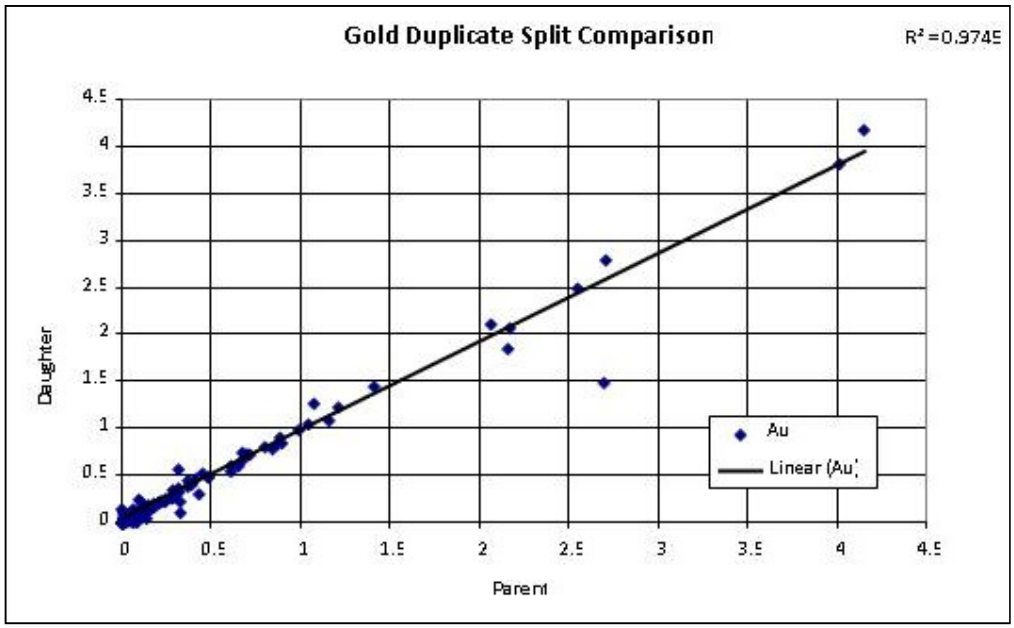

Figure 16-2: Allied – Au in Pulp Duplicates Scattergram

From April to December 2009, a total of 341 sample pairs were analysed (excluding samples analysed by EXLAB and subsequently dispatched for confirmation fire assay). The data has been split according to Laboratory and Duplicate sample type groups for statistical analysis. Predictably the "Field" Duplicates (FD) demonstrate a higher variance compared to a pulp duplicates; and the Aqua-regia method data produced by EXLAB has a higher variance than the ALS Fire Assay data.

A summary tabulation of the HARD (Half Absolute Relative Difference) averages for duplicates with Au values greater than ten times detection, and defined into populations based on sample type, and analytical method (ALS = FAA, EXLAB = AR).

Where sample pair results were both less than ten times detection, they were treated as outliers and removed from the statistical analyses.

Table 16-6: Duplicates Analyses – Apr to Dec 2009 – EXLAB Aqua Regia digest/ALS Fire assays

Lab and Duplicate type	Full Num Pairs	Count > 10X D.L.	% HARD Ave > 10X D.L.
EXLAB PD	78	46	6.9
EXLAB FD	82	51	10.2
ALS FD	96	78	7.6
ALS PD	85	67	5.8
Total	**341**	**242**	

PD = pulp resample

FD = chips riffle split duplicate

D.L. = Detection Limit

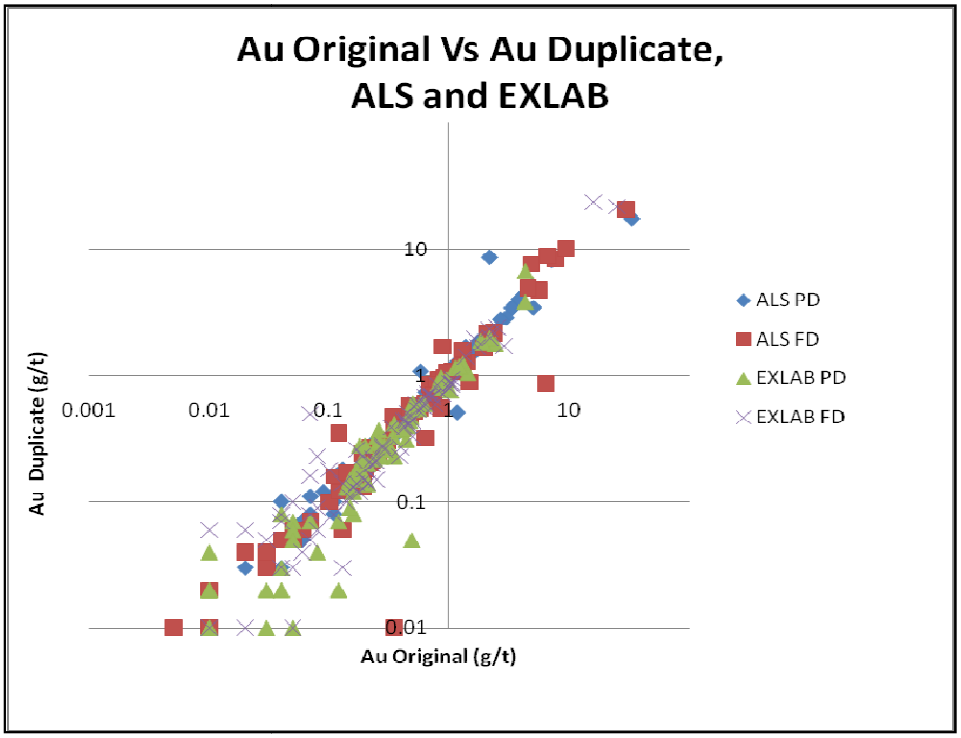

Figure 16-3 Duplicates Analyses – Apr to Dec 2009 – EXLAB Aqua Regia digest/ALS Fire assays

Duplicates data indicates sampling consistency with acceptable reproducibility as demonstrated by the 4 different statistical analyses of the data, with the majority of the sample scatter being located in the less than 0.1 g/t region of Au concentration and at higher grades (+6 g/t Au).

Overall, the duplicate sample results demonstrate a remarkable reproducibility for both duplicate types and analytical methods.

An average HARD of EXLAB field duplicate pairs of 10.2% is somewhat elevated by the presence of two hig grade sample pairs with large differences between the original and duplicate values. These large differences were attributed to insufficient grinding for the aqua-regia digest method. Subsequently grinding time was increased to achieve a nominal 95% pass 75 microns.

The average HARD of 6.9% for Pulp Duplicates analysed by EXLAB indicates slightly better reproducibility than found in Field Duplicates.

The average HARD of 7.6% for Field Duplicates analysed by ALS indicates the initial reduction split is satisfactory and the sample is representivity though some scatter is evident at the higher grades. An obvious outlier was considered to be a sample swap.

An average HARD of 5.8% for Pulp Duplicates analysed by ALS indicates the pulps are homogenous and can be considered representative. The -1.6% average HRD suggests a slight bias as does the Q-Q plot however this is due to 3 assays with large variance as seen in the Au vs. Au plot.

16.1.2.3 Blanks

Due to the poor initial selection of blank material, the blanks analysis data could not be used to accurately determine the degree of contamination in sample preparation or at the ALS laboratories with any certainty. Seven blanks returned gold values greater than 0.05 g/t. As a result, the use of beach sand as a blank was abandoned and a new local source, Grey Hill (TIG coordinates 43650 mE:208100 mN) was selected.

A total of 429 Grey Hill Blank samples were submitted with samples from Pigiput-Pigibo.

98.6% of the blanks returned Au values less than ten times the detection limit (of the respective analytical methods) thus providing confidence that minimal sample contamination has occurred.

Six blanks returned values more ten times detection Au, representing 1.4% of the samples. Investigation showed three of the six were the result of sample swap, and another the result of contamination.

The spurious high blank values are from various holes and the different sample processing dates eliminate a systematic contamination issue in sample preparation. It does however show a minor QC sample handling problem. Sample preparation workers periodically undergo refresher training to minimise the instances of QC sample swapping.

Table 16-7: Blank samples analysed by EXLAB and ALS, Apr – Dec 2009.

LAB	NUM BLANKS	NUM > D.L.	Num > 10X D.L.	% > 10X D.L.
ALS	278	37	5	1.80
EXLAB	151	13	1	0.66
Total	**429**	**50**	**6**	**1.40**

D.L. = Detection Limit

16.1.3 Round Robin Inter-laboratory Checks

In 2009, round robin checks were included as a routine part of Allied's QAQC program, with 2-3% of samples selected from each hole.

In March 2009, an inter-laboratory round robin check was done on selected samples from 11 holes (Table 16-8), to check Analytical Laboratory Services (ALS) Townsville, the laboratory used for routine analyses. Samples were dispatched to Genalysis (GLS) and SGS Laboratories Townsville and analysed using 50 g fire assay with AAS finish. Control samples, standards, blanks and duplicates were included at a rate of 1 in 20.

Table 16-8: Allied – Holes used for the March 2009 round robin checks

Hole_id	ALS Lab_Batch_ID
SDH012	TV08157095
RC1759	TV08165120
RC1758	TV08164449
RC1731	TV08139939
RC1744	TV08149837
RC1724	TV08136556
SDH011	TV08132961
RC1718	TV08132964
RC1736	TV08144662
RC1748	TV08152753
SDH015	TV08180185

A second set of Round Robin checks was completed in Jan 2010. The study used 286 samples from 53 holes selected, in compliance with the inter-laboratory check QC protocol, between 13 November 2008 and 22 September 2009. With a small number of exceptions, the results of the check assays received closely match the original results from ALS.

The Round Robin check protocol followed at the Exploration Sample Preparation Facility at Simberi calls for a second scoop sample to be taken from the pulverizing bowl. This sample set aside for analysis, by the

same fire assay method, at a different laboratory from the original (ALS Townsville), in this case Genalysis Laboratory Service laboratory in Perth. The pulp duplicate sampling frequency is approximately one sample of every 30 original sample.

The majority (245) of the total 286 round robin checks came from 49 holes drilled at Pigiput (29 holes/168 samples) and Pibigo (20 holes/77 samples). The remainder, 4 holes/41 samples, were from holes drilled at the Sorowar Prospect.

A small low bias is evident when results returned from Genalysis (GLS) are compared with the ALS data. On average the GLS results are 0.016 g/t Au lower, an insignificant difference when the detection limit for the 50 g fire assay method at both laboratories is 0.01 g/t Au.

Table 16-9: Jan 2010 Round Robin Check Programme – Summary Statistics

Intra Batch DUPs	ALS_FAAu_orig	GLS FAA Au	Mean	Diff	HRD (%)	HARD (%)
Mean	**0.85**	**0.84**	**0.85**	**-0.01**	**-4.6**	**10.4**
Max	30.1	32.0	31.0	2.22	88.7	88.7
Min	0.01	0.01	0.01	-1.34	-7.7	0.0
Kurtosis	100.5	111.0				
Skewness	8.66	9.16				
Median	0.24	0.22	0.23	-0.01	-2.13	3.59
Std Deviation	2.26	2.35	2.30	0.25	19.18	16.73
CV	5.26					3.21
No. Pairs	286					
Pearson CC	0.99					
F-test	0.55					

Graphical representations for statistical analysis are presented below. An minor positive bias is evident in the Q-Q plot and HRD histogram.

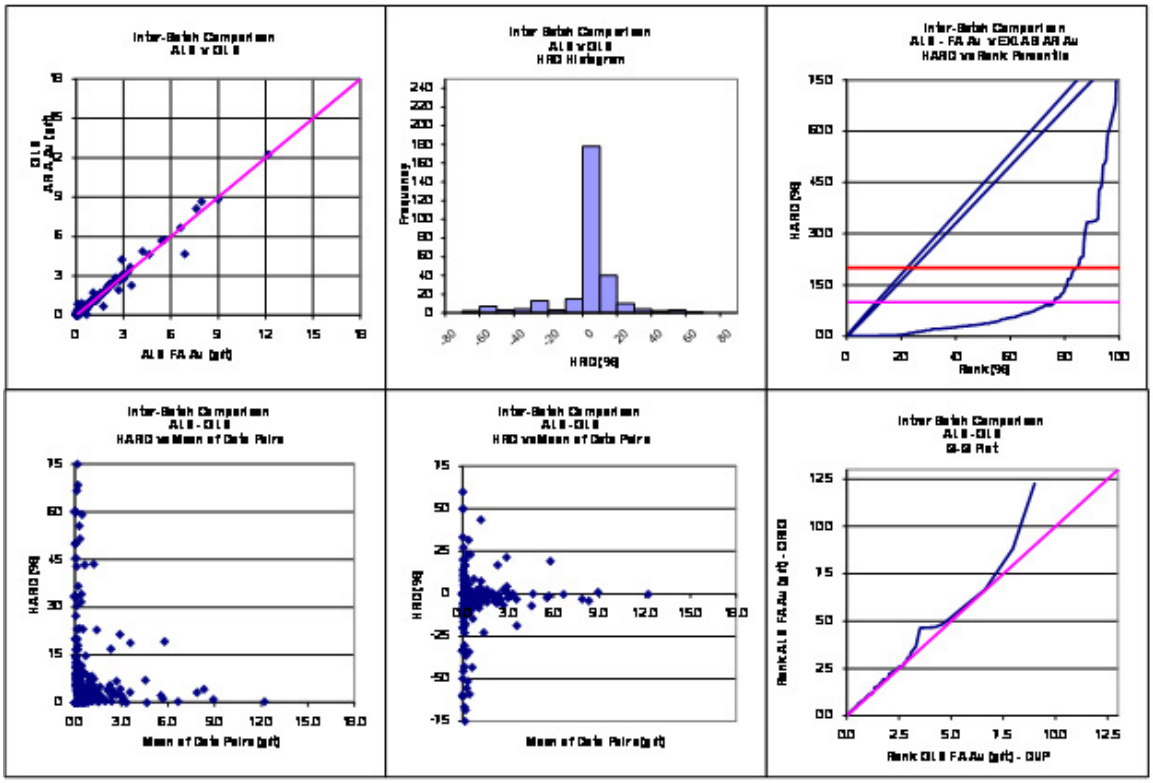

Figure 16-4: Jan 2010 Round Robin Check Programme – Graphical Illustrations of Summary Statistics

Excluding outliers, an overall correlation coefficient of 98.9% is achieved (Figure 16-5).

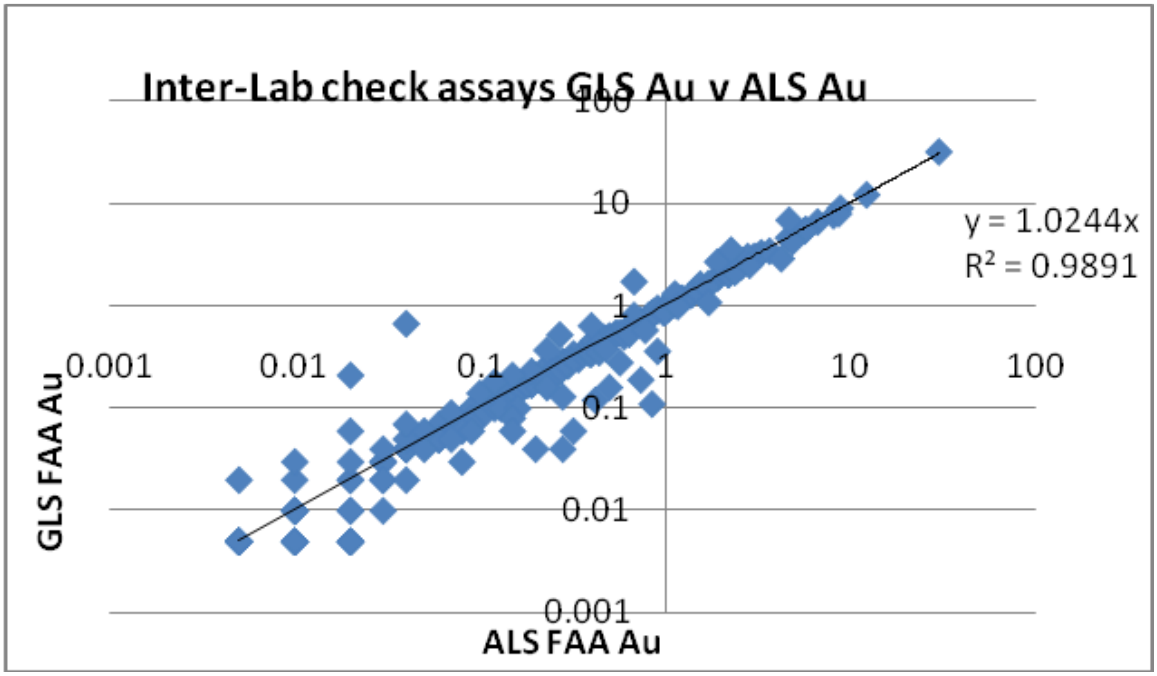

Figure 16-5: Inter Lab assay correlation

Excluding sample pairs where both assays are less than 10 times the detection limit, twenty three check samples displayed relatively larger variance (>15% HARD), this represents 8% of the total dataset. Of these samples, ten include at least one assay value greater than 0.5g/t Au.

The results are considered to demonstrate the validity of the ALS fire assay gold data albeit with a slight positive bias, particularly in assays less than 1 g/t Au.

16.1.3.1 Analysis of Inserted Standards

The Certified Reference Material passed QC requirements returning with in 2 standard deviations of certified Au grade. Also no anomalous values were obtained for the blank material.

CRM values and results from SGS and GLS laboratories are provided in Table 16-10:

Table 16-10: CRM values and results from SGS and GLS Laboratories

Control Sample Type	Recomm Au ppm	SGS Townsville		Genalysis Perth		QC analysis
		Samp id	Reported Au ppm	Samp id	Reported Au ppm	
blank		209257	X	209357	0.01	PASS
blank		209258	0.02	209358	0.01	PASS
blank		209265	0.02	209365	0.01	PASS
blank		209279	X	209379	X	PASS
blank		209280	X	209380	X	PASS
blank		209287	X	209387	0.03	PASS
blank		209318	X	209418	0.01	PASS
CRM_G306-1	0.41	209259	0.39	209359	0.38	PASS
CRM_G901_7	1.52			209229	1.51	PASS
CRM_G302-10	0.18	209273	0.14	209373	0.18	PASS
CRM_G906-2	2.46	209266	2.42	209366	2.33	PASS
CRM_G901-7	1.52	209256	1.43	209356	1.5	PASS
CRM_G901-7	1.52	209293	1.41	209393	1.54	PASS
CRM_G907-6	7.25	209286	7.14	209386	7.27	PASS
CRM_G907-6	7.25	209300	7.65	209400	7.28	PASS
CRM_G01	0.02	209272	0.04	209372	0.04	PASS

16.1.3.2 Analysis of Pulp Duplicate Results

Figure 16-6, Figure 16-7 and Figure 16-8 below illustrate the correlation of Au values of samples submitted in the round robin check and the original Au values reported by ALS Townsville, the laboratory used for routine drill hole sample analyses. While samples near detection limits show some spread, correlation coefficients of greater than 97% were calculated, demonstrating the accuracy of the original ALS results were acceptable.

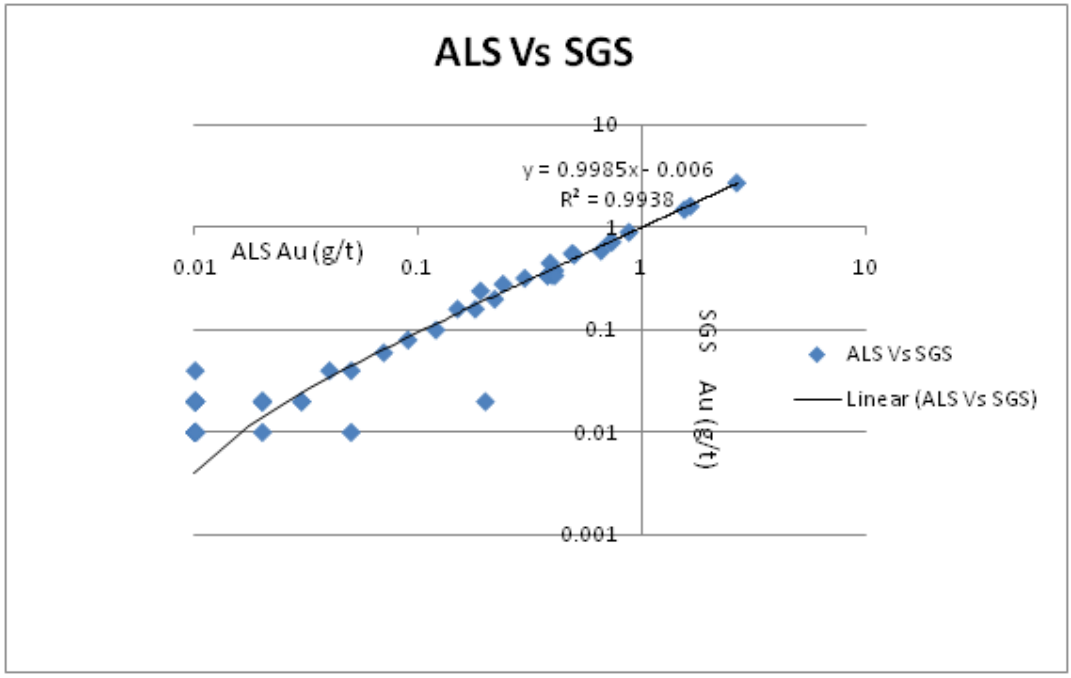

Figure 16-6: Allied – Round Robin Check – March Qtr 2009 – Gold by FAA – ALS vs. SGS Townsville

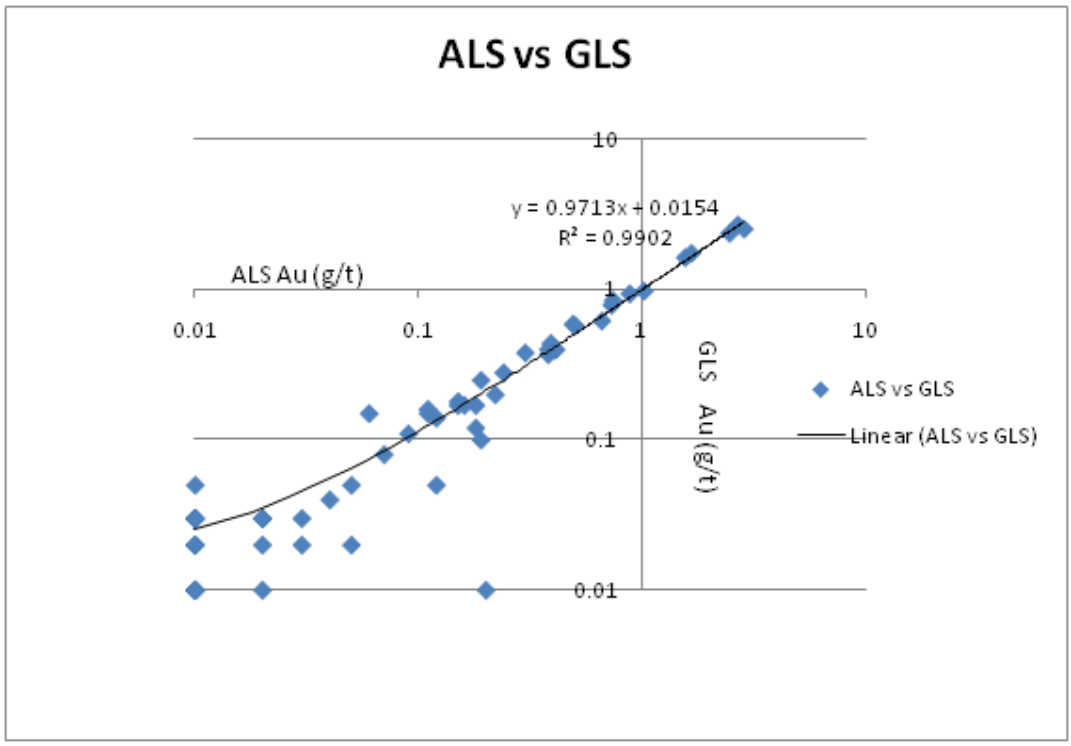

Figure 16-7: Allied – Round Robin Check – March Qtr 2009 – Gold by FAA – ALS vs. Genalysis

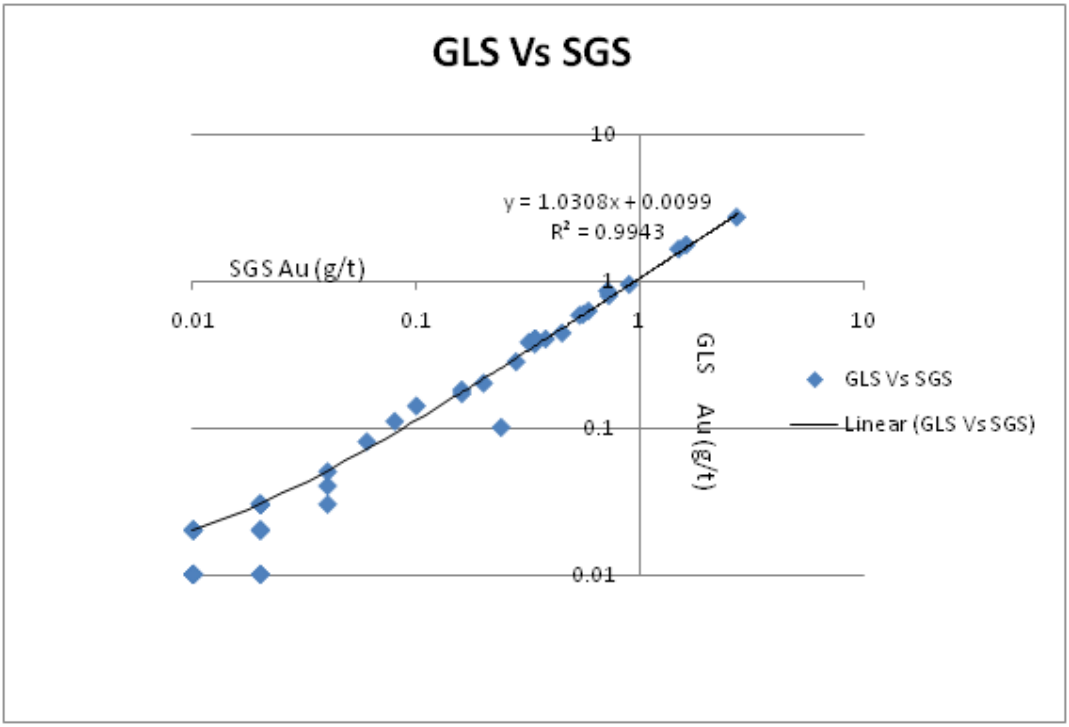

Figure 16-8: Round Robin Check – March Qtr 2009 – Gold by FAA – Genalysis vs. SGS Townsville

16.1.3.3 Diamond Twins of RC Holes

A limited number of RC holes were twinned by diamond core holes in late-2008. An analysis of the results, indicated that while the comparison was good, poor recovery issues can affect the reliability of both RC and core sample assays. The reliability of drill hole data needs to be assessed on a hole-by-hole basis.

16.2 Author's Verification

16.2.1 Site Visit

The Authors representative, John Batista, Associate with Golder, visited Simberi between 28 October 2008 and 31 October 2008.

The drilling and sampling operations were observed and sample storage facilities visited. The mining operation and plant facilities were also inspected.

16.2.2 Database Validation

For the deposits modelled by Golder Associates routine internal database validation checks are performed on the drill hole data sets prior to analysis, modelling and estimation. These checks include:

- Cross table checks (holes in collar but not in assay, etc).
- Collar depth against final assay and geology depth.
- Overlapping intervals or gaps in the assay and geology tables.
- Duplicate hole names and duplicate coordinates.
- Coordinate values of zero.

- Integer coordinate values (indicate of a lack of detailed survey data).
- Extreme variations (>= 10°) in drill hole azimuth or dip between consecutive downhole survey records

Any anomalies or errors noted were brought to the attention and resolved. In all instances the anomalies or errors noted were minor and the Author considers that they would have made no material difference to the resource if left unresolved.

Maxwell Geoservices were commissioned to audit the full Allied Simberi database for the purpose of general data integrity and compliancy.

The key areas identified as requiring attention were:

- Identify and update missing coordinates for 2 drill holes.
- Source and update missing:
  - collar survey dates and methods
  - drill dates
  - sample dates, types and methods
  - downhole survey methods and dates
  - Local Grid establishment data, and
  - Implement procedure change to ensure that this data is collected for all future drilling programs.
- Review codes with no descriptions.
- Review BDL result treatment.
- Review Pb values > 100%.
- Review un-ranked assay data.
- Update Lab Element mapping.
- Update Lab Method Priority:
  - DIL.
- Review sample and drilling metadata collection.

These recommendations will improve the overall data quality and in some local instances the data integrity but in the Author considers that this would make no material difference to the resource.

17.0 ADJACENT PROPERTIES

There are no adjacent properties under not under the control of Allied. Allied controls the exploration lease over the west half of Simberi, and covering most of the other islands in the group. A summary of these prospects is included in Section 20.0.

18.0 MINERAL PROCESSING AND METALLURGICAL TESTING

Section 18.0 has been prepared by Battery Limits Pty Ltd under the supervision of Phil hearse, Managing Director

A Scoping Study, based on historical testwork, and a Prefeasibility Study, based on significant new metallurgical testing on the proposed sulphide plant has been conducted as part of this Revised Technical Report. Test work included sulphide ore comminution, flotation, direct cyanidation, ultra-fine grinding-cyanidation, and roasting – cyanidation. The most appropriate option for the sulphide plant was found to be roasting – cyanidation. The testwork performed is discussed in this section, and an assessment of this information and projected performance of the process plant is discussed in Section 25.0 of this report.

18.1 Historical Testwork

Historical testwork had shown that the sulphide ore was refractory, and not amenable to gold and silver recovery by conventional cyanidation. However, it was amenable to conventional flotation and pyritic refractory ore treatment by pressure oxidation. Pressure oxidation was found to result in high gold recovery on both the ore and the flotation concentrate. No testwork was undertaken to evaluate other conventional refractory treatment routes such as roasting or biological oxidation (BIOX®).

Samples used in the historical sulphide testwork came from the Botlu, Samat, and Pigiput deposits. Cyanidation testwork demonstrated that cyanide extractable gold generally ranged from 13% to 45%. Mineralogical investigations indicated very little visible gold. As the main sulphide mineral was crystalline pyrite, most of the gold was assumed to be ultra-fine or solid solution gold in pyrite.

18.2 Simberi Sulphide Scoping Study

Allied Gold completed a Scoping Study for the Simberi Sulphide Project in 2009. The scoping study was mostly based on results from the historical testwork, as well as benchmarking of other projects.

Key metallurgical and processing outcomes from the Scoping Study included:

- The Simberi sulphide ore is refractory and requires sulphide oxidation to release the gold for cyanidation.
- Production of a flotation concentrate and export is potentially the most economic option for exploitation of the sulphide ore. However, a market will need be secured for the concentrate product.
- Biological oxidation using BIOX® is not viable because of high power consumption and high power costs.
- Roasting with off-gas scrubbing is a potentially economic and low risk option.

Metallurgical recommendations from the Scoping Study included:

- Conduct further testwork to determine comminution characteristics and flotation parameters
- Undertake a preliminary roasting metallurgical testwork program on flotation concentrate
- Conduct metallurgical testwork to confirm that UFG (ultra-fine grinding) is not a viable option for the project.

A program of testwork was then commenced as part of a Prefeasibility Study for the Project.

18.3 PFS Testwork

Testwork commenced on samples of Sorowar sulphide ore, and was then followed by a more detailed program of work with Pigiput samples. Pigiput accounts for over 90% of the gold in the Simberi sulphide resource.

18.3.1 Sample Selection

18.3.1.1 Sorowar Deposit

Four distinct alteration zones for the Sorowar deposit were noted by geological personnel, namely: clay and siliceous located in the upper and middle alteration zones; and carbonate and chlorite in the lower alteration zones. For testwork purposes two composite samples were prepared. Composite 1 consisted of clay and silicate material from the upper and middle alteration zones and Composite 2 comprised carbonate and chlorite material from the lower alteration zone.

Core samples were selected in 3 m intervals, which represented approximately half the anticipated mining bench height. Composite 1 consisted of eight intervals that ranged from depths of 63.5 m to 153 m, with gold grades from 0.90 g/t to 9.35 g/t and sulphur grades from 2.8% to 4.2%. Composite 2 consisted of four intervals that ranged from depths of 153 m to 192 m, with gold grades from 0.80 g/t to 1.0 g/t and sulphur grades from 3.3% to 4.7%.

18.3.1.2 Pigiput Deposit

Metallurgical drill holes used to generate samples for testwork were twinned (Holes ID SDH 022 to SDH 027) from previous drill hole collars (Holes ID P017, P033, P016, P021 and P020 respectively). Two composites were prepared to represent the Porphyry and Tuff lithologies. The sample intervals used in the Porphyry and Tuff composites is summarised in Table 18-1.

Table 18-1: Pigiput Sample Intervals Composites

Hole ID	Sample Interval (m)		Lithology Description		Average Geological Grade of Intervals		
	From	To			Au g/t	As ppm	S%
Porphyry							
SDH 022	110	141	Porphyry		2.64	538	2.8
SDH 024	151	173	Porphyry		4.84	1184	3.6
SDH 026	139	145	Porphyry		1.51	202	4.7
SDH 027	62	150	Porphyry		6.21	379	3.7
Tuff							
SDH 024	132	217	Tuff		2.94	608	4.2
SDH 026	98	123	Tuff		3.77	802	5.5
SDH 027	60	62	Tuff		2.04	1184	6.4

18.3.2 Head Assays

18.3.2.1 Sorowar

The gold content of Composite 2 was significantly lower than Composite 1 although the sulphide component of both composites was similar at 3.9%. Both composites contained high carbonate levels, which can affect processing performance with some refractory technologies such as pressure and biological oxidation but is unlikely to have a significant effect on the roasting option.

Head analyses for the two Sorowar composites are provided in Table 18-2.

Table 18-2: Sorowar Composite Head Analyses

Element		Composite 1	Composite 2
Au	ppm	2.79	0.97
Ag	ppm	2.10	1.90
As	ppm	498	250
S^{2-}	%	3.9	3.9
CO_3^{2-}	%	11.8	5.5

18.3.2.2 Pigiput

The head assays for the two ore types are similar. Repeat gold assays on samples showed only small variation suggesting a low presence of coarse free gold.

Head analyses for the two Pigiput composites are provided in Table 18-3

Table 18-3: Pigiput Ore Composite Head Analyses

Element		Porphyry	Tuff
Au (mean)	ppm	3.40	2.71
Ag	ppm	4.2	6.1
As	ppm	548	645
S^{2-}	%	3.55	4.47
CO_3^{2-}	%	0.36	1.27

18.3.3 Comminution

Comminution testwork was carried out to characterise the grinding energy requirements of the ore samples.

18.3.3.1 Sorowar

The results of the Bond rod and ball mill work indices suggest that the composite samples have moderate grinding requirements for primary grinding, and that there are no significant differences between the two alteration samples. The abrasion indices for two composites indicate that both samples have low abrasiveness and that grinding media and liner consumption would be low.

The ratio of RWI to BWI is frequently used as an indication of amenability to SAG milling and potential for critical size build-up. The values of this ratio for both composites infer that SAG milling is likely to be applicable for this material.

The results of the testwork for the two composites are presented in Table 18-4.

Table 18-4: Sorowar Comminution Testwork Results

Bond Parameter		Composite 1	Composite 2
Abrasion Index	A_i	0.082	0.027
Rod Mill Work Index	kWh/t	14.7	13.7
Ball Mill Work Index	kWh/t	13.9	14.7
Rod:Ball Wi Ratio	RWI:BWI	1.06	0.93

18.3.3.2 Pigiput

The results of the bond rod and ball mill work indices suggest that the composite samples have moderate grinding requirements for primary grinding. The Ai result indicates that the Porphyry ore is abrasive and is likely to result in significant grinding media and liner consumption.

Results of the testwork for the two composites are presented in Table 18-5.

Table 18-5: Pigiput Comminution Testwork Results

Bond Parameter		Porphyry	Tuff
Abrasion Index	Ai	0.121	0.066
Rod Mill Work Index	kWh/t	16.1	N/A**
Ball Mill Work Index	kWh/t	17.8	16.2
Rod:Ball Wi Ratio	RWI:BWI	0.90	N/A

** Sample result invalid due to incorrect crushing procedure

18.3.4 Cyanidation Leaching

Cyanidation leaching testwork was undertaken on both Sorowar and Pigiput composites. Tests included direct cyanidation and carbon in leach (CIL) testing. Tests were undertaken with one hour pre-aeration and oxygen addition. The tests were performed in agitated vessels with an initial cyanide concentration of 0.1% (w/v) and subsequent additions to maintain 0.05% (w/v).

18.3.4.1 Sorowar

Composite 2 proved to be slower leaching and significantly more refractory than Composite 1. CIL leaching appeared to increase the rate of gold extraction for Composite 1 compared to direct leaching. Both samples can be considered refractory to conventional cyanidation processing.

The test results at different leach times are summarised in Table 18-6.

Table 18-6: Sorowar Cyanide Leaching Results

Test		Composite 1		Composite 2	
		24 h	48 h	24 h	48 h
Direct Leaching	Au Ext %	30.3	41.6	7.4	7.4
CIL Leaching	Au Ext %	39.6	47.8	7.3	7.3

18.3.4.2 Pigiput

Only direct leaching was performed on Pigiput composites. For the Porphyry composite there was a significant increase in leach extraction between 24 hours and 48 hours, indicating that a portion of slow leaching gold. It is likely that gold extraction would continue increase slightly with longer residence times. In contrast, the Tuff composite showed no significant increase in leach extraction between 8 hours and 48 hours residence, indicating that the leachable gold present is fast leaching. Overall leach extractions are low at less than 40%. Both samples can be considered refractory to conventional cyanidation processing.

The test results at different leach times are shown in Table 18-7.

Table 18-7: Pigiput Cyanide Direct Leaching Results

Test		Porphyry			Tuff		
		8 h	24 h	48 h	8 h	24 h	48 h
Direct Leaching	Au Ext %	28	33	39	25	26	27

18.3.5 Rougher Flotation Testwork

18.3.5.1 Grind Size Optimisation

A series of flotation tests were undertaken, initially on Sorowar composites, to establish a relationship between grind size and gold recovery for the primary rougher flotation. Tests were performed at varying P_{80} grind sizes of 150 µm, 125 µm, 106 µm and 75 µm, using identical reagents and conditions for each test. The reagent scheme used in the four tests was $CuSO_4$ as an activator, a combination of the dithiophosphate reagent (DTP) and potassium amyl xanthate (PAX) as collectors, and methyl isobutyl carbinol (MIBC) as a frother.

For the Sorowar composites, the grind optimisation testwork showed that flotation performance improved as the grind size decreased from 150 µm to 106 µm on both composites. No further improvement was achieved with finer grinding. The flotation test using PAX alone at the optimum 106 µm grind showed improved flotation performance over the equivalent test with PAX and DTP. The Pigiput composites behaved similarly to the Sorowar samples. Based on the results from the flotation tests, a 106 µm grind and PAX collector was recommended for primary rougher flotation for both Sorowar and Pigiput ores.

18.3.5.2 Flotation Reagent Optimisation

For all composites of both ore types, flotation tests were performed at the optimum 106 µm grind size with varying reagent schemes. The primary rougher optimisation testwork was carried out varying the quantity of $CuSO_4$ activator added, decreasing flotation density and increasing both the flotation time and collector addition.

The test results indicated that there were only minor improvements in flotation performance with increased activator and collector additions. However, increasing flotation time from 22 minutes to 32 minutes for Sorowar composites improved gold recovery and was recommended for all subsequent primary rougher floats. The improvement in performance was considered to be due to the presence of a very slow floating pyritic gold component.

18.3.5.3 Cleaner Flotation Testwork

Initial rougher flotation testwork was carried out on both Sorowar composites at the optimised conditions. Rougher scavenger concentrate was reground before being cleaned at natural pH to produce cleaner concentrate. Cleaner flotation testwork was then carried out on both composites at the optimised conditions.

The tests showed that improved concentrate grade could be achieved at an optimum regrind size of 53 µm. Gold recovery to concentrate significantly improved and the concentrate mass pull was reduced by about 50% with regrinding for both concentrates. Similar results were achieved with Pigiput composites.

18.3.5.4 Bulk Flotation

For the Pigiput samples, a bulk roughing and cleaning flotation test was performed on both the Porphyry and Tuff ore composites. The tests were conducted at 30% solids using a feed mass of 50 kg to the mill. Tests were conducted as follows, using optimum conditions which were determined in previous testing:

- Grind size of P80 = 106 µm.
- Natural pH in roughing and cleaning.
- Addition of 120 g/t $CuSO_4$ activator prior to roughing flotation.
- Staged additions of PAX collector during roughing and scavenging and staged addition of MIBC frother.
- Total of four minutes in roughing flotation and 70 minutes in scavenging flotation.
- Two stages of cleaning flotation.

The results showed that for Porphyry ore, a maximum 93% gold recovery could be achieved into a 14% mass concentrate at a gold grade of 22 g/t. For Tuff, 92% gold recovery was obtained into a 20% mass concentrate containing 12.5 g/t. Most of the sulphide sulphur (92% to 94%) and arsenic (81% to 87%) also reported to these concentrates, resulting in sulphur and arsenic grades of 20% to 23% and 0.27% to 0.31% respectively.

18.3.6 Leaching of Ultra Fine Ground Concentrate

The cyanidation leach response of Ultra-Fine Ground (UFG) flotation concentrate was determined for both composites for Sorowar ore only. The test results were compared to leaching of whole ore. Gold dissolution increased significantly from 42% to 69% for Composite 1 but less for Composite 2 (7.4% to 15%) for UFG samples (estimated size of 5 µm) compared with direct leaching of ore at 106 µm. Although improvements were noted, the gold dissolution after UFG was considered too low for this technology to be considered further as a possible treatment route for Sorowar ore. It was expected that Pigiput ore would behave similarly.

18.3.7 Roast-Leach Testwork

Roast-leach tests were undertaken on Pigiput concentrate samples prepared by bulk flotation. Roasting tests were conducted on the "as received" flotation concentrate (P_{80} =106 µm), and also on concentrate ground to P_{80} sizes of 63, 38 and 25µm. Roasted samples were subjected to 24 and 96 hour cyanidation leach tests to determine the amount of recoverable gold. Roasting in a rotating Midrex furnace produced a better roast-leach performance than roasting in a static muffle furnace with rabbling, though neither technique provided a good simulation of a fluid bed roaster.

The results of the outlined testwork are summarised in Table 18-8.

Table 18-8 Calcine Grind Optimisation

Parameter	63 µm Regrind	38 µm Regrind	25 µm Regrind	Midrex Test No Regrind
Porphyry				
24 Hour Leach Extraction (%)	89.5	90.6	90.1	88.9
96 Hour Leach Extraction (%)	90.4	90.6	91.1	90.1
Tuff				
24 Hour Leach Extraction (%)	82.5	83.6	86.5	87.0
96 Hour Leach Extraction (%)	84.0	86.0	87.3	88.1

The Porphyry concentrate showed negligible increases in gold extraction with decreasing grind size, but the Tuff material showed greater an increase of 4% (24 hour leach) for a change in regrind size from 63 µm down to 25 µm. The Midrex roasting technique, without any calcine regrind step, resulted in similar or better extractions to that of a 25 µm regrind with the muffle roast.

The significant increase in recovery for Tuff material using both the Midrex roasting technique, and UFG was considered unusual, and suggests full piloting will be required in the bankable feasibility study to optimise the roasting parameters and downstream calcine treatment steps to maximize design gold recovery.

18.3.8 Concentrate Thickening, Filtration and Physical Testing

18.3.8.1 Thickening Testing

The concentrate samples were tested for thickening and filtration performance by Outotec technology. The concentrate samples were raked only with bottom rakes to assist in dewatering. Flocculant screening tests showed that supernatant clarity was achieved with all tested flocculants after an addition of 30 g/t. Magnafloc M 5250 was selected as it resulted in the best settling performance.

The results from the continuous test runs are summarised in Table 18-9.

Table 18-9: Concentrate Thickening Tests

Sample	Diluted Feed			Floc		Underflow		Overflow
	Flux	Liquor Rate	Solids Density	Type	Dose	Solids Density	Yield Stress	Clarity
	t/m^2h	m/h	%	g/t		%	Pa	mg/L
Porphyry	0.25	1.30	17.0	M 5250	30	61.3	152	<50
Tuff	0.25	1.30	17.0	M 5250	20	67.3	149	<50

18.3.8.2 Filtration Testing

Samples of both Porphyry and Tuff concentrates were tested by Larox to evaluate the filtration characteristics of each concentrate. The objective of the testwork was to determine optimum cake thickness, cake moisture and filtration rate capacity. The testwork was conducted in a mini Larox 100 test unit. The target moisture for the testwork was <10%.

Both concentrate samples were readily dewatered in the laboratory filter press with a 25 mm chamber and the filter cakes produced were found to be solid and easily handled. Key data from the tests are presented in Table 18-10.

Table 18-10: Concentrate Thickening Tests

Parameter		Porphyry	Tuff
Residual Cake Moisture	% w/w	12.0	9.9
Cake Thickness	mm	19.5	23
Specific Filtration Rate (solids)	$kgDSm^2h$	84.3	133
Specific Filtration Rate (liquid)	L/m^2h	30	39.7
Slurry Sizing Measurement	P_{80} µm	69	72

18.3.8.3 Physical Characteristics of Concentrate

Samples of both Porphyry and Tuff concentrate were despatched to ATSIS in Newcastle for physical testing. The main objective of the testwork was to characterise the concentrate to ensure safe exportation of the material from site, if required. ATSIS use international and national test methods, which meet the needs of concentrate exporters and ensure that the materials can be safely exported. The parameters measured and results are presented in Table 18-11.

Table 18-11: ATSIS Physical Test Results

Parameter		Porphyry	Tuff
Bulk Density (as received)	wt/m^3	2.182	2.075
(calculated)	dt/m^3	1.813	1.708
Stowage Factor	m^3/t	0.458	0.482
Angle of Repose	deg	43.4	42.9
Transportable Moisture Limit	%	12.2	11.9
Crossing Point Temperature	℃	>350	>350

The TML values are within expectations for sulphide concentrates, and can be obtained by standard filtration techniques.

The Crossing Point Temperature is a measure of the pyrophoric nature of concentrates. The "Crossing Point" is defined as the point at which the temperature of the sample exceeds the ambient temperature of the oven, caused by the heat from the oxidation of the sample. Temperatures over 250°C are considered stable for storage and shipping. The two samples analysed have a high Crossing Point and therefore represent a negligible risk for spontaneous combustion in transport.

In summary, both the Porphyry and Tuff concentrates settled readily with low suspended solids. Both settled concentrate samples were readily dewatered in a pressure filter to reasonable moisture contents, and the filter cakes produced were found to be solid and easily handled, with transportable moisture limits (TML) and crossing points suitable for safe export.

19.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

19.1 Mineral Resource

Section 19.1 has been prepared by Golder Associates under the supervision of Stephen Godfrey, Associate, Senior Resource Geologist.

19.1.1 Introduction

The Simberi resources have been classified in accordance with the guidelines of the Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC, 2004).

The resource classification was based on data quality, data density, confidence in the geological interpretation and confidence in the estimation.

The resource has been classified as Inferred, Indicated and Measured. The JORC classification is comparable with the CIM definitions for the same categories as presented in Table 19-1.

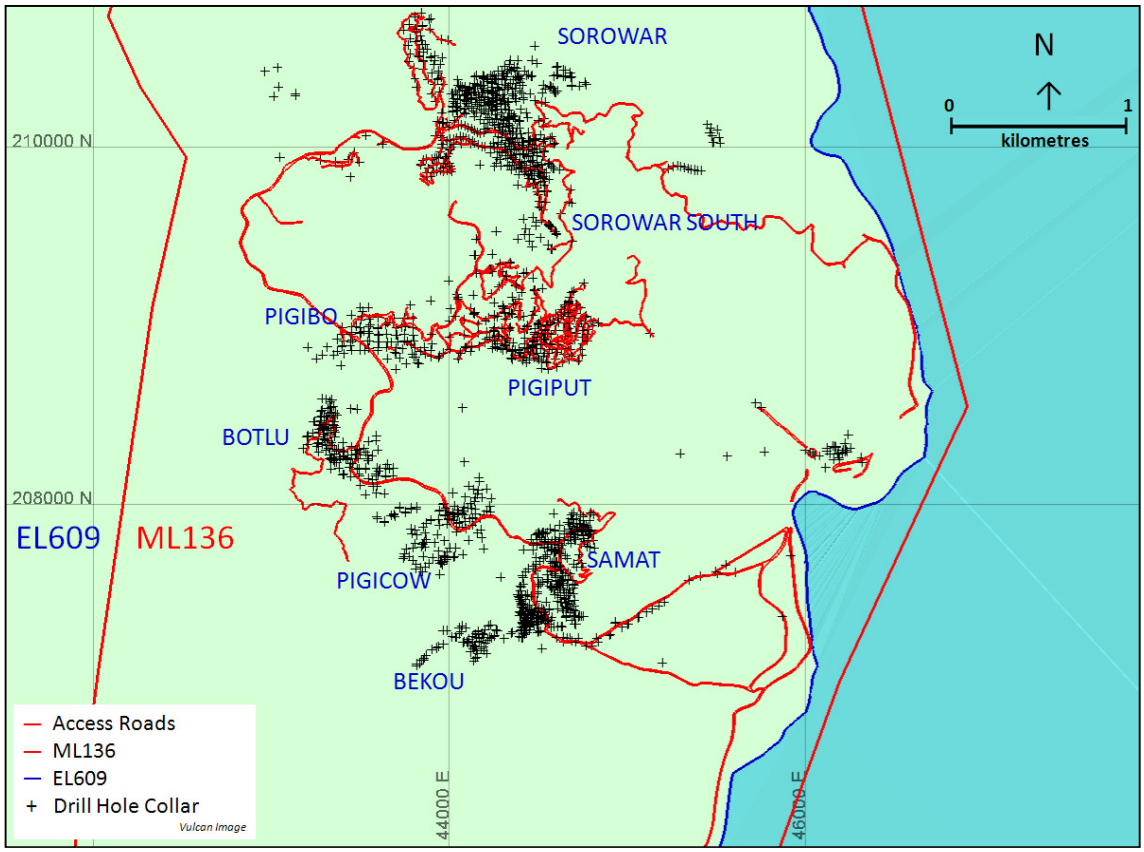

Figure 19-1: Simberi resource locations

Table 19-1: Comparison of JORC and CIM classification

JORC	CIM
An '**Inferred Mineral Resource**' is that part of a Mineral Resource for which tonnage, grade and mineral content can be estimated with a low level of confidence. It is Inferred from geological evidence and assumed but not verified geological and/or grade continuity. It is based on information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes which may be limited or of uncertain quality and reliability.	An '**Inferred Mineral Resource**' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.
An '**Indicated Mineral Resource**' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a reasonable level of confidence. It is based on exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are too widely or inappropriately spaced to confirm geological and/or grade continuity but are spaced closely enough for continuity to be assumed.	An '**Indicated Mineral Resource**' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
A '**Measured Mineral Resource**' is that part of a Mineral Resource for which tonnage, densities, shape, physical characteristics, grade and mineral content can be estimated with a high level of confidence. It is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes. The locations are spaced closely enough to confirm geological and grade continuity.	A '**Measured Mineral Resource**' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

The Simberi resources have been estimated by Minstat Pty Ltd and Golder Associates Pty Ltd. The Sorowar, Pigiput, Pigibo, Bekou and Pigicow resources have been estimated by, or estimated under the direct supervision of, the Author.

The Botlu resource is an historical estimate undertaken by M. Binns of Minstat Pty Ltd. The Author has reviewed these models and satisfactorily undertaken independent validations of the estimations and believes they are a still relevant and reliable.

Table 19-2 details the Simberi resources depleted to the end of October 2010.

Table 19-2: Simberi Mineral Resources (0.5 g/t Au cut off – depleted to eom October 2010)

Deposit	Material	Measured			Indicated			Inferred		
		Mt	Au g/t	koz	Mt	Au g/t	koz	Mt	Au g/t	koz
Bekou	Oxide				0.04	1.74	2	0.06	1.14	2
	Transition				0.01	1.17	0	0.05	1.16	2
	Sulphide				0.02	1.93	2	0.92	1.38	41
					0.07	**1.76**	**4**	**1.02**	**1.36**	**45**
Bekou Total					**0.07**	**1.76**	**4**	**1.02**	**1.36**	**45**
Botlu	Oxide	1.22	1.14	44	0.45	1.23	18	0.31	1.16	11
	Sulphide							1.45	1.81	84
		1.22	**1.14**	**44**	**0.45**	**1.23**	**18**	**1.76**	**1.70**	**96**
Botlu Total		**1.22**	**1.14**	**44**	**0.45**	**1.23**	**18**	**1.76**	**1.70**	**96**
Pigibo	Oxide				2.96	1.11	106	0.60	0.89	17
	Transition				2.19	1.19	84	0.51	0.92	15
	Sulphide				3.86	1.11	137	6.22	0.94	188
					9.00	**1.13**	**327**	**7.33**	**0.93**	**220**
Pigibo Total					**9.00**	**1.13**	**327**	**7.33**	**0.93**	**220**
Pigicow	Oxide				0.15	1.65	8	0.29	1.30	12
	Transition							0.11	1.29	4
	Sulphide							2.00	1.26	81
					0.15	**1.65**	**8**	**2.39**	**1.26**	**97**
Pigicow Total					**0.15**	**1.65**	**8**	**2.39**	**1.26**	**97**
Pigiput	Oxide	3.02	0.87	85	4.60	0.92	137	2.01	0.79	51
	Transition				1.95	0.89	56	0.77	0.83	21
	Sulphide				32.56	1.51	1,583	32.25	1.00	1,042
		3.02	**0.87**	**85**	**39.12**	**1.41**	**1,775**	**35.03**	**0.99**	**1,114**
Pigiput Total		**3.02**	**0.87**	**85**	**39.12**	**1.41**	**1,775**	**35.03**	**0.99**	**1,114**
Samat East	Oxide							0.40	1.13	14
	Transition							0.08	0.78	2
	Sulphide							3.50	0.78	88
								3.98	**0.82**	**105**
Samat East Total								**3.98**	**0.82**	**105**
Samat north A	Oxide				0.16	0.75	4	0.11	0.84	3
	Transition				0.04	1.29	2	0.01	1.26	0
	Sulphide				0.36	0.81	9	1.08	0.86	30
					0.56	**0.83**	**15**	**1.20**	**0.86**	**33**
Samat North A Total					**0.56**	**0.83**	**15**	**1.20**	**0.86**	**33**
Samat North B	Oxide				0.12	0.86	3	0.12	0.75	3
	Transition				0.05	2.86	5	0.02	0.78	1
	Sulphide				1.90	1.22	74	1.05	0.73	25
					2.06	**1.24**	**82**	**1.19**	**0.73**	**28**
Samat North B					**2.06**	**1.24**	**82**	**1.19**	**0.73**	**28**

Deposit	Material	Measured			Indicated			Inferred		
Total										
Samat South A	Oxide				0.02	2.11	1	0.16	1.28	7
	Transition				0.01	0.98	0	0.01	0.77	0
	Sulphide				0.02	1.01	1	1.73	0.97	54
					0.05	**1.53**	**2**	**1.90**	**0.99**	**61**
Samat South A Total					**0.05**	**1.53**	**2**	**1.90**	**0.99**	**61**
Samat South B	Oxide				0.05	2.94	5	0.17	1.51	8
	Transition				0.05	2.03	3	0.02	0.99	1
	Sulphide				1.70	1.76	96	3.36	1.07	115
					1.80	**1.80**	**104**	**3.54**	**1.09**	**124**
Samat South B Total					**1.80**	**1.80**	**104**	**3.54**	**1.09**	**124**
Sorowar	Oxide	5.81	1.30	243	8.56	1.08	298	2.40	1.09	84
	Transition	0.54	1.17	20	1.46	1.14	53	0.29	0.83	8
	Sulphide	1.30	0.93	39	6.93	0.92	205	19.03	0.90	549
		7.64	**1.23**	**302**	**16.95**	**1.02**	**556**	**21.72**	**0.92**	**641**
Sorowar Total		**7.64**	**1.23**	**302**	**16.95**	**1.02**	**556**	**21.72**	**0.92**	**641**
Sorowar South	Oxide							0.68	0.82	18
	Transition							0.28	0.68	6
	Sulphide							5.39	0.66	114
								6.35	**0.67**	**138**
Sorowar South Total								**6.35**	**0.67**	**138**
Grand Total		**11.88**	**1.13**	**431**	**70.21**	**1.28**	**2,892**	**87.42**	**0.96**	**2,701**

Rounding may cause numerical discrepancies

19.1.2 Sorowar Resource Estimation

19.1.2.1 Database

The Sorowar resource model is based on the geological database as at 31 October 2008. The Sorowar dataset comprised 581 RC holes for a total of 51,625 m and 57 Diamond Holes for a total of 9701 m. The RC holes were 5.25 inch diameter. The Diamond holes were predominantly NQ with 10 HQ and 1 PQ hole.

The Sorowar deposit as modelled in 2008 has had the Sorowar South domain excised from the block model. Sorowar South was re-estimated in 2010 in conjunction with the Pigiput and Pigibo block models and is reported separately.

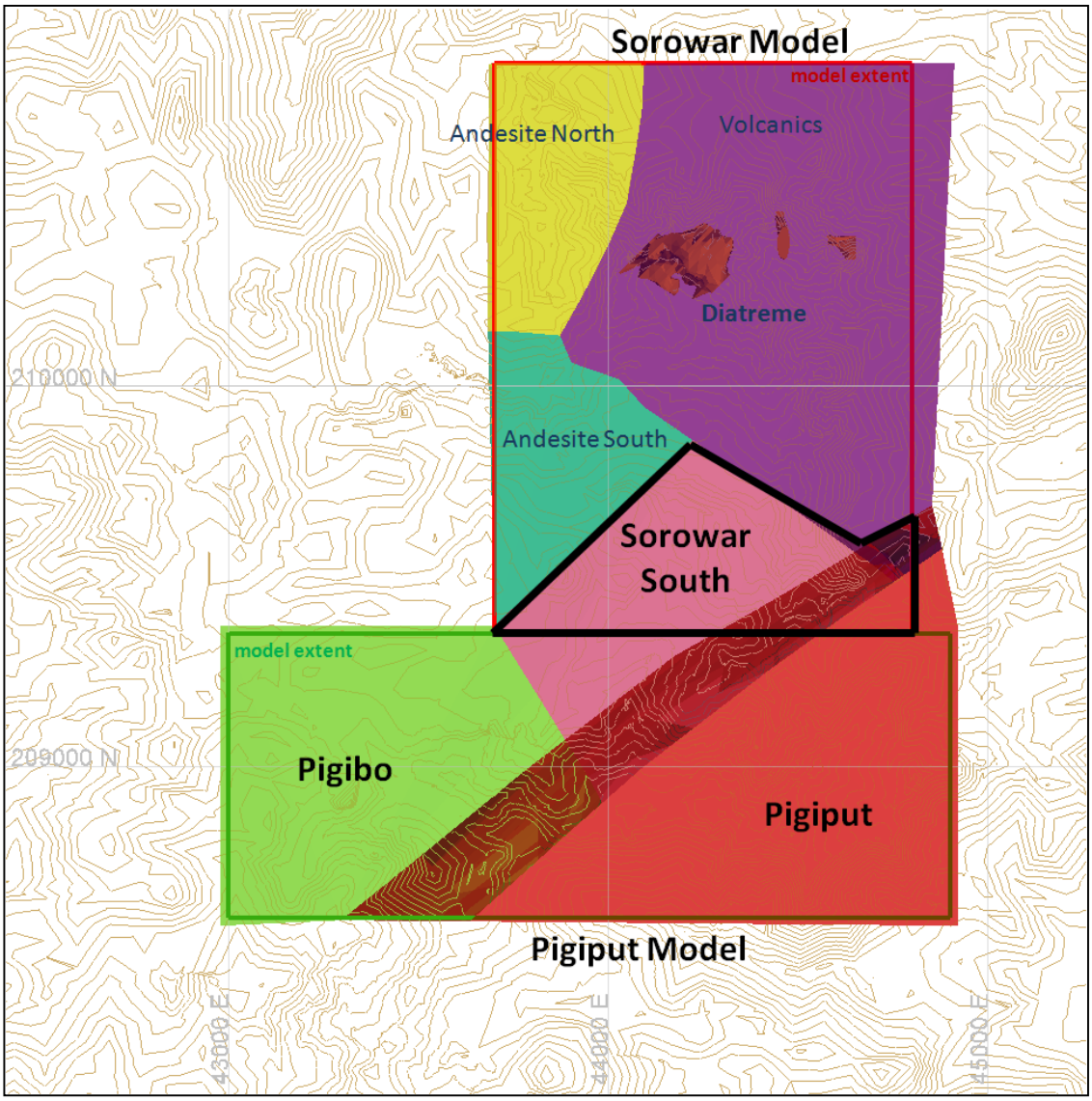

Figure 19-2: Sorowar – Deposit Geology and drill collars

19.1.2.2 Exploratory Data Analysis

19.1.2.2.1 Assays

Preliminary data analysis was done on the raw assay data. Display of the data highlighting logged geology and grades Indicated trends in the data. In general Ag grades were observed to be higher in the northern part of the deposit coincident with a change in geology from predominantly volcanics to andesite. The geology was digitised and the drill hole data flagged for analysis.

Cumulative probability plots for Au and Ag clearly show how the different geological units host different populations of Au and Ag. The diatreme (domain 4) hosts the highest grade gold mineralisation and the northern andesite (domain 1) hosts the highest Ag grades. As a result of this analysis the geology was added into the estimation as an additional domain.

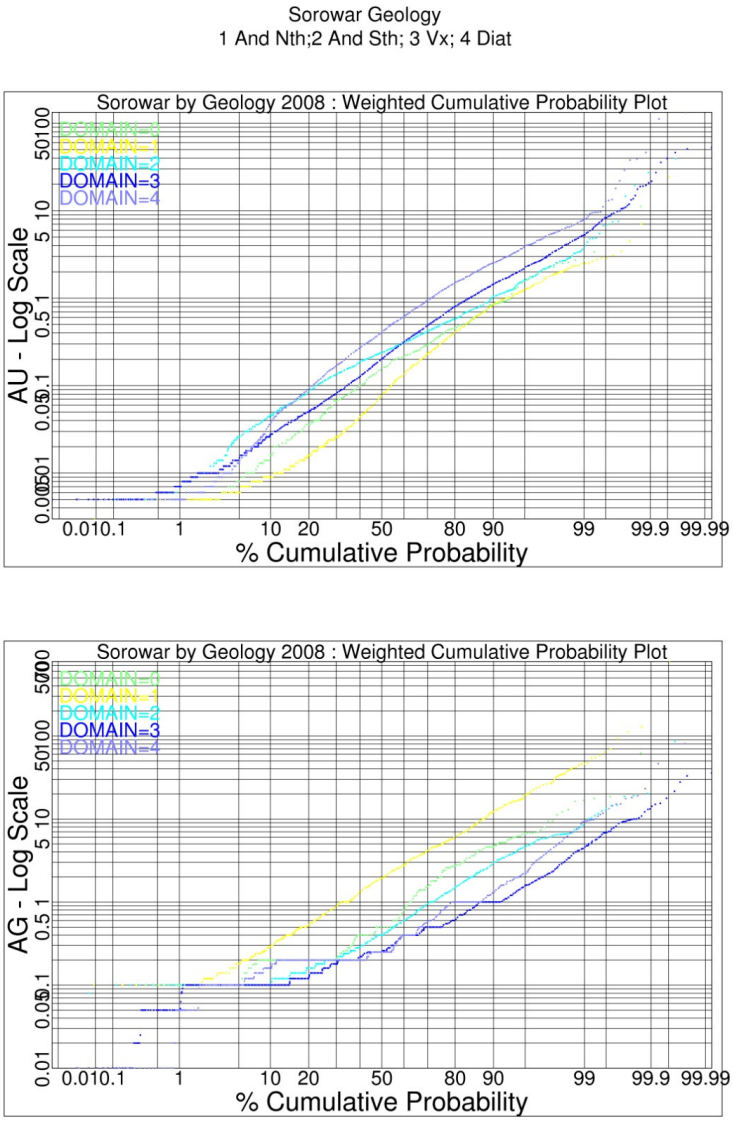

Figure 19-3: Sorowar – Cumulative probability plots – Au and Ag by geological unit

19.1.2.2.2 Composites

For detailed statistical analysis, spatial analysis and the estimation the drill hole data was composited to 5 m.

19.1.2.2.3 Geological Interpretation

Geological modelling of the deposit was undertaken by Stephen Godfrey from Golder Associates Pty Ltd ("Golder") in consultation with Carmel Grant (Resource Geologist) and Ross Hastings (Development Manager) from Allied. Solid modelling of the geology, construction of the block model and grade estimation was undertaken by Stephen Godfrey. The wireframe geological model was constructed using Maptek's Vulcan software.

19.1.2.2.4 Spatial Analysis

Directional variography was undertaken to determine the directions of maximum continuity of mineralisation and provide input parameters for the kriging process. Au showed maximum continuity towards approximately 130° (400°) direction with ranges of the order of 60 to 90 m for all domains. Ag show a similar direction of maximum continuity and range with the andesites modelling slightly longer ranges than the volcanics.

The semi-major variographic direction was modelled down dip to the south west with ranges of 60 to 80 m.

19.1.2.2.5 Domaining

The composited drill hole data was flagged by the weathering domains representing Oxide, Transition and Sulphide material types. Additionally, pods of 'floating' transition and sulphide material were modelled within the Oxide domain. The 2008 modelling also defined Volcanic, Andesitic and Diatreme lithological domains over the deposit. The Andesitic domain was further subdivided into North and South to define an area of significantly higher Silver grades.

19.1.2.2.6 Resource Block Model

The resource block model was constructed aligned to the Tabar Island Grid. Figure 19-2 illustrates the position of the model and Table 19-3 details its dimension and block sizes. Sub-celling was used to help the model honour the geological boundaries.

Table 19-3: Sorowar – Model dimensions

Sorowar		X	Y	Z
Model	Min	43790	209600	-80
	Max	44700	210650	300
Parent Cell	Min	10	10	10
	Max	10	10	10
Subcell	Min	5	5	5
	Max	10	10	10

Domains were defined in the model based in the modelled geology units and weathering surfaces.

Table 19-4: Sorowar – Domain coding

	Volcanics	Andesite South	Andesite North	Diatreme
Oxide	1001	2001	3001	4001
Transition	1002	2002	3002	4002
Fresh	1003	2003	3003	4003

The topographic data as of 15 September 2008 was provided by Allied in Surpac string and dtm format and imported into Vulcan.

19.1.2.2.7 Interpolation Plan

Grade estimations for Gold (Au) and Silver (Ag) were done using Ordinary Kriging. High grade restraining was applied to the Au and Ag estimates. Where composites have a grade greater than the values specified in Table 19-5 the influence of that composite was restricted to one block model cell. A grade restricting envelope was applied to the model to limit the extrapolation of grades beyond the area drilled.

Table 19-5: Sorowar – High Grade thresholds

Au	Oxide	Transition	Fresh
Volcanics	25	10	9
Andesite South	25	10	9
Andesite North	25	10	9
Diatreme	25	10	9
Ag			
Volcanics	20	20	20
Andesite South			20
Andesite North	65	30	40
Diatreme	30	10	10

19.1.2.2.8 Bulk Densities

Dry Bulk Density was assigned using a spherical function developed by Allied Gold. The function has a nugget of 1.0 t/m^3 and a sill of 0.9 t/m^3 at a range 55 m, i.e. at surface the bulk density is 1.0 and at 55 m below surface bulk density is 1.9. This was calculated with the following formulae:

$$\gamma(h) = c_0 + c\left[\frac{3}{2}\frac{h}{a} - \frac{1}{2}\frac{h^3}{a^3}\right] \text{ for h} \le \text{a}$$

and

$$\gamma(h) = c_0 + c \text{ for h} > \text{a}$$

Table 19-6 explains the variables used above.

Table 19-6: Sorowar – Details of density variables

Variable	Name	Description	Value
c_o = 1.0,	Nugget	Minimum SG – at surface	1
c = 0.9,	Sill differential	Maximum SG	0.9
a = 55	Range	Depth at which maximum SG reached	55
h	Lag	Depth below surface	variable

The bulk density was calculated based on the depth of the block below the topographic surface.

19.1.2.2.9 Mineral Resource Classification

The resource has been classified based on the quality and density of the drilling data used in the modelling and estimation. Golder's review of the analytical and survey data for the resource has shown it to be of a consistent quality and the resource has been classified Measured, Indicated or Inferred based principally on drill spacing. The classification was applied to the model using wireframe solids for each classification category.

Table 19-7 list the Sorowar gold resource depleted to the end of October 2010 mining surface.

Table 19-7: Sorowar – Mineral Resource – Depleted to October 2010

Class	Mt	Au g/t	Au Oz
Measured	7.6	1.23	302
Indicated	17.0	1.02	556
Indicated + Measured	24.6	1.08	858
Inferred	21.7	0.92	641

19.1.2.2.10 Block Model Validation

Statistical and visual assessment of the block model were undertaken to assess successful application of the various estimation passes and to ensure that as far as the data allowed, all blocks within ore domains were estimated and the model estimates considered acceptable.

Visual Assessment of Grade Estimates

Onscreen validations between samples and blocks were completed on the model. The onscreen validation process involved comparing block estimates and composites grades in cross-section and in plan.

Mean Grade Reproduction

The global estimated mean domain grade against 5 m composite data was checked as a means to validate reproduction of the mean grade of the data. Global statistical comparisons of the mean grade are influenced by varying drill hole density, data sharing between domains and high-grade treatment. These issues can sometimes distort the comparison. In particular, peripheral blocks representing extrapolated estimates can create apparent anomalies in the grade conformance. Differences observed in the global average grades may not be evident on swath comparisons, which is an advantage of using swath plots to assess conformance.

Overall, reproduction of the mean grade at Sorowar is acceptable. Some apparent under and over-estimation of the domain average in the block model for some domains is likely to be an artefact of the issues described above.

Swath Plot Validations

Swath plots are used to assess the block model estimates for global bias; the estimates should have a close relationship to the drill hole composite data used for estimation.

The process involved averaging both the blocks and samples in panels of 40 m (easting) by 40 m (northing) by 10 m (RL). The relationship between model and sample panel averages was assessed in the form of scatter plots and Q-Q plots. Conformance of the model and sample average grades was assessed in the form of easting, northing and RL swaths of the panel averages. As an example the Swath plots for Au in the Oxide domain are presented in Figure 19-4.

Generally good agreement is observed between the data and block model au mean grade for easting, northing and RL slices. QQ and scatter plots for the averaged sample data vs. block model results show deviation from the 45° line, with overstatement of low grades and understatement of high grades by the block model. This is a natural expected behaviour of moving from sample size data to a much bigger volume as represented by Kriged models.

The swath plot validation process shows that the block model estimates follow the trend of the 5 m composite grades across the deposit. From the perspective of conformance of the average model grade to the input data, Golder considers the model to be a satisfactory representation of the drill hole data used.

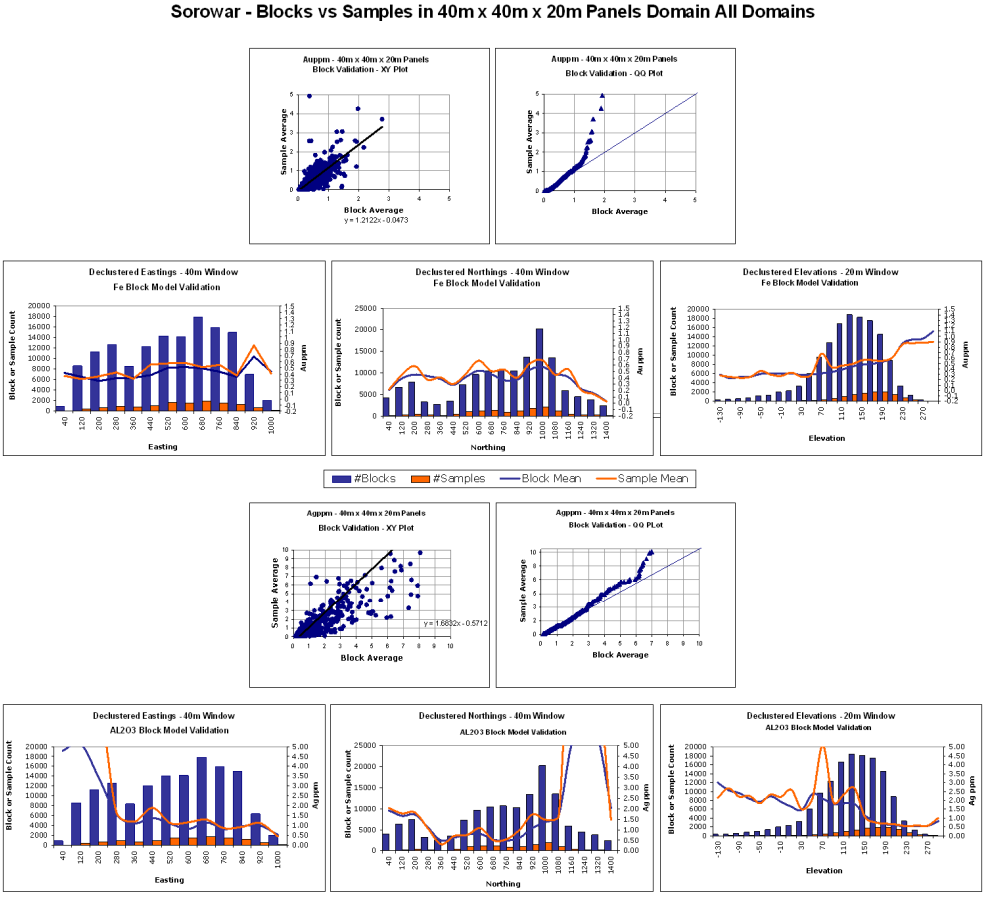

Figure 19-4: Sorowar – Swath plot validations Au Ag

19.1.3 Pigiput/Pigibo/Sorowar South Resource Estimation

The Pigiput and Pigibo deposits have been developed to the extent where they have merged and can be estimated as one model. The two deposits are still distinct within this model and are separated by a geological boundary. Additionally the northern part of the model has been recognised as being continuous with the southern part of the Sorowar area. Consequently this zone called Sorowar South was added to the Pigiput and Pigibo models and an additional model was built to estimated that part of the Sorowar South extending into the original Sorowar model. The original Sorowar model was depleted for the blocks estimated in Sorowar South.

19.2 Geological Modelling

The geological models and resource estimation are based on an extract of the Datashed SQL database was supplied to Golder in the form of *.csv* files containing drill hole collar, assay, lithology, oxide, RQD, survey and specific gravity information. The extract contains data to 4 June 2010.

Sectional interpretation of the oxide and sulphide horizons were completed by Allied in May 2010. The interpretation strings were imported from Surpac in to Vulcan, validated and used to create surfaces for use in the construction of the geological block model.

The oxide interpretation occurs at the base of the oxidation zone. The interpretation of the sulphide boundaries occurs on north south sections based on the weathering profile in the lithological logs and interpreted at the base of the transition zone.

Mineralisation estimated into the block model was controlled vertically by the bounding weathering wireframes interpreted for the oxide, transition and sulphide domains

Structural domain boundaries were supplied to delimit Pigiput from Pigibo. These structural boundaries resulted in a third domain being created which extends north of the Pigiput/Pigibo model boundary into Sorowar. This domain is identified hereafter as Sorowar South (Figure 19-5). Previously Pigibo and Pigiput were demarcated geographically by the 44,000 m Easting.

Interpreted wireframe models were supplied by Allied to define the Tuff unit mostly within Pigiput. The remaining geology defaulted to Andesite (Figure 19-6).

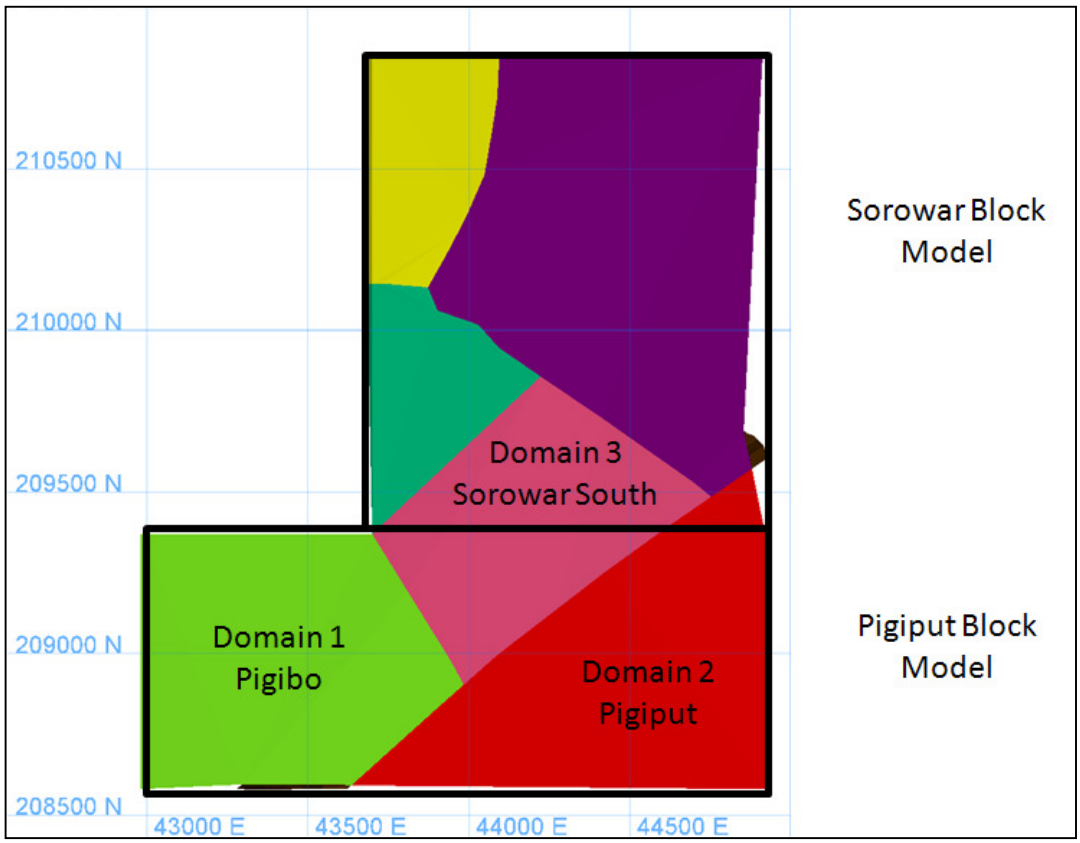

Figure 19-5: Pigiput/Pigibo domains

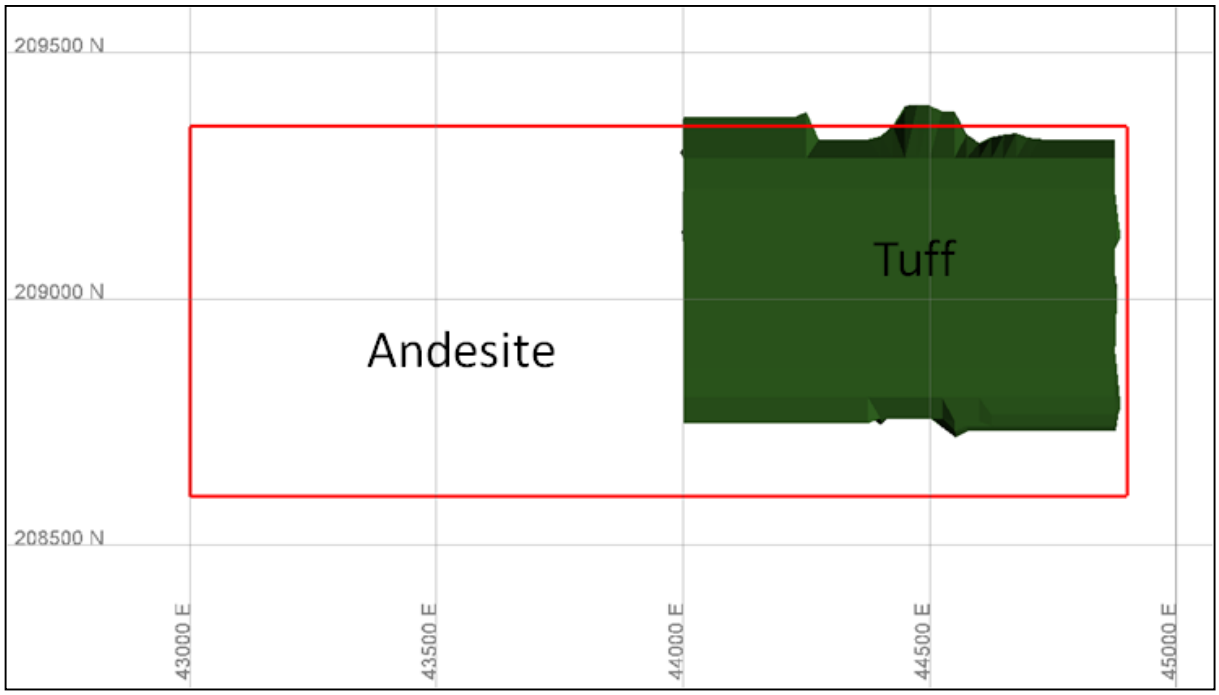

Figure 19-6: Lithological domains

Table 19-8 summarises the relationship between the model codes and the relevant triangulations used to construct the .block models.

Table 19-8: Summary of Model Codes in Relation to the Wireframes

Model Variable	Description	Wireframe	Code
Geology	Tuff	pig_tuff_1006	10
	Andesite	default	20
Domain	Pigibo	pigibo_1006	1
	Pigiput	pigiput_1006	2
	Sorowar South	central_1006	3
Weathering	oxide (base)	pig_oxide_1006	1
	transition	n/a	2
	sulphide (top)	pig_sulphide_1006	3

19.3 Geological Block Model

19.3.1 Block Model Parameters

Two block models were constructed using Vulcan 3D software. One covering the Pigiput/Pigibo domains and the southern part of Sorowar South and the other a duplicate of the Sorowar model but with only the Sorowar South and Pigiput parts of the estimated. The models are hereafter referred to as Pigiput and Sorowar South.

A 10 m by 10 m by 10 m parent cell was chosen for the model based on the drill sample spacing. A sub-block size of 5 m by 5 m by 5 m was used to improve the volumetric accuracy of the model in and

around the wireframe boundaries. The dimensions for the block models along with the respective block sizes are provided in Table 19-9 and Table 19-10

Table 19-9: Block Model Dimensions – Pigiput/Pigibo

Orientation	Minimum	Maximum	Length	Parent Block	Sub-Block
	(m)	(m)	(m)	(m)	(m)
X – Easting	43000	44900	1900	10	5
Y – Northing	208600	209350	750	10	5
Z – RL	-250	300	550	10	5

Table 19-10: Block Model Dimensions – Sorowar South

Orientation	Minimum	Maximum	Length	Parent Block	Sub-Block
	(m)	(m)	(m)	(m)	(m)
X – Easting	43700	44800	1100	10	5
Y – Northing	209350	210850	1500	10	5
Z – RL	-250	300	550	10	5

19.3.2 Model Domain Codes

Variables were assigned to each block and set to a default value. Each block variable was then flagged by wireframe surfaces or solids and assigned a value according to the block's position relative to the wireframe. In the case of multiple, overlapping wireframes, the priority assigned to the block dictated the final block variable value. Wireframes with the highest priority took precedence over wireframes with lower priorities for a particular block variable. Where wireframes have the same priority, the sequential order determines the flagging code – blocks are flagged by the first wireframe that encompasses them. The block model coding schema used is detailed in Table 19-11. Blocks were flagged in the domain code order (1–3; default of -99 used).

Table 19-11: Block Model Domain Codes

Variable	Description	Default	Coding
ox	Weathering Code	-99	1 ox, 2 tr, 3 fr
geology	Geology Code	20	10 tuff 20 andesite
estdom	Estimation Domain	-99	(ox × 1000)+(geology)
mined	Mined Flag	0	0 = *in situ* 1 = mined
domain	Domain Code	-99	

Table 19-12: Pigiput coding schema

Wireframe	Variable	Code	Priority
simtopo_may2010b.00t	ox	-99	4
pig_oxide_1006.00t	ox	1	2
pig_sulphide_1006.00t	ox	2	1
pig_sulphide_1006.00t	ox	3	3
pigiput_may10_eom.00t	mined	1	1
simtopo_may2010b.00t	mined	-99	4
pbo_tuff_1006.00t	geology	10	1
pig_tuff_1006.00t	geology	10	1
pigibo_1006.00t	domain	1	1
pigiput_1006.00t	domain	2	2
central_1006.00t	domain	3	3
pig_fw_1006.00t	domain	-99	10
simtopo_may2010b.00t	domain	-99	10

Table 19-13: Sorowar South coding schema

Wireframe	Variable	Code	Priority
simtopo_may2010c.00t	ox	-99	4
transition2.00t	ox	1	2
sulphide2.00t	ox	2	1
sulphide2.00t	ox	3	3
simtopo_may2010c.00t	ox	-99	14
pig_oxide_1006.00t	ox	1	12
pig_sulphide_1006.00t	ox	2	11
pig_sulphide_1006.00t	ox	3	13
pigiput_may10_eom.00t	mined	1	1
simtopo_may2010c.00t	mined	-99	4
pbo_tuff_1006.00t	geology	10	1
pig_tuff_1006.00t	geology	10	1
pigibo_1006.00t	domain	1	1
pigiput_1006.00t	domain	2	2
central_1006.00t	domain	3	3
pig_fw_1006.00t	domain	-99	10
simtopo_may2010c.00t	domain	-99	10

19.4 Statistical Analysis

19.4.1 Data Extraction and Processing

Initial data selection was from the Vulcan database *sim02062010.tab.isis.* Eleven assay variables were available: Au, Ag, As, Cu, Fe, Mn, Mo, Pb, S, Sb, Zn. The Pigiput/Pigibo analysis and resource estimation was completed for Au, Ag and S only.

19.4.2 Sample Flagging

Prior to compositing, the raw sample intervals in the geological database *sim02062010.tab.isis* were flagged to the geological interpretation wireframes with the appropriate domain codes for weathering (oxide transition or sulphide), geology (tuff or andesite) and structural domain (Pigiput, Pigibo or Sorowar South). The database flagging used the same wireframes and code values as those in the block model construction.

Flagging directly into the Vulcan database is based on sample interval centroids, and the downhole depth from and to intervals are preserved.

19.4.3 Compositing

95% of the samples in the database are 1 m in length. A composite length of 5 m was generated on the Pigiput data breaking the compositing at the domain boundary. A 5 m composite length was used for estimation, to minimise differences in support for all samples and moderating the sample variance whilst retaining sufficient numbers of samples for grade estimation. The 5 m length was considered a compromise between smoothing of the original variability and maintaining sufficient resolution for the geological wireframes. The composite length matches the block height in the resource model and is equivalent to the selective mining unit being used.

The parameter file used for compositing was *sim5m.cm1*.

- *sim5m.map* (main composite file output) – 5 m downhole composites.

This mapfile was used in the exploratory data analysis and estimation of Au, Ag (g/t) and S (%) grades in the resource.

19.5 Exploratory Data Analysis

19.5.1 Univariate Statistics

Univariate statistics based on the clustered sample configuration were generated from raw composites and the 5 m composited data, as shown in Table 19-14 and Table 19-15 for Au, Ag and S. Statistics are reported by domain codes and geology. Due to the regular nature of the drilling grid, spatial declustering was considered unnecessary. The statistics were calculated on length-weighted data.

Table 19-14: Pigiput – Univariate Statistics

	Domain	Lithology	Weathering	Obs	Minimum	Maximum	Mean	Variance	CV	Median
AU	1001	Tuff	oxide	2407	0.00	20.62	0.70	1.42	1.70	0.37
AU	1002	Tuff	transition	734	0.00	15.24	0.43	0.86	2.16	0.17
AU	1003	Tuff	sulphide	2102	0.00	146.10	0.74	20.74	6.17	0.20
AU	2001	Andesite	oxide	105	0.01	5.14	0.62	0.74	1.40	0.29
AU	2002	Andesite	transition	257	0.01	4.13	0.37	0.42	1.74	0.12
AU	2003	Andesite	sulphide	3534	0.01	36.19	0.79	4.26	2.61	0.28
AU	Global			9149	0.00	146.10	0.71	6.88	3.68	0.26
AG	1001	Tuff	oxide	2297	0.01	10.98	0.24	0.11	1.38	0.20
AG	1002	Tuff	transition	660	0.01	5.16	0.27	0.11	1.21	0.20
AG	1003	Tuff	sulphide	1859	0.01	146.20	0.91	16.35	4.42	0.26
AG	2001	Andesite	oxide	103	0.10	0.76	0.21	0.02	0.59	0.20
AG	2002	Andesite	transition	252	0.01	1.20	0.25	0.05	0.93	0.18
AG	2003	Andesite	sulphide	3058	0.01	91.65	2.62	29.25	2.06	0.90
AG	Global			8237	0.01	146.20	1.28	15.66	3.10	0.22
S	1001	Tuff	oxide	2443	0.00	5.65	0.14	0.21	3.33	0.00
S	1002	Tuff	transition	743	0.00	6.32	0.77	1.75	1.71	0.00
S	1003	Tuff	sulphide	2122	0.00	11.61	1.63	4.39	1.29	0.00
S	2001	Andesite	oxide	105	0.00	4.68	0.20	0.28	2.65	0.03
S	2002	Andesite	transition	260	0.00	3.04	0.60	0.68	1.39	0.12
S	2003	Andesite	sulphide	3648	0.00	9.28	1.88	3.88	1.05	1.61
S	Global			10604	0.00	11.61	0.95	3.41	1.94	0.00

Table 19-15: Pigibo – Univariate Statistics

Variable	Domain	Lithology	Weathering	Obs	Minimum	Maximum	Mean	Variance	CV	Median
AU	2001	Andesite	oxide	747	0.01	26.66	0.82	2.23	1.82	0.40
AU	2002	Andesite	transition	412	0.00	7.70	0.55	1.03	1.85	0.17
AU	2003	Andesite	sulphide	1017	0.01	14.59	0.51	1.50	2.40	0.13
AU	Global	Andesite		2178	0.00	26.66	0.62	1.68	2.08	0.21
AG	2001	Andesite	oxide	704	0.04	20.02	0.73	1.81	1.84	0.50
AG	2002	Andesite	transition	346	0.02	39.60	1.26	7.93	2.23	0.50
AG	2003	Andesite	sulphide	832	0.05	35.80	1.20	4.52	1.78	0.50
AG	Global	Andesite		1884	0.02	39.60	1.03	4.18	1.98	0.50
S	2001	Andesite	oxide	749	0.00	5.83	0.19	0.34	3.13	0.00
S	2002	Andesite	transition	413	0.00	8.29	1.03	2.98	1.68	0.03
S	2003	Andesite	sulphide	1102	0.00	14.87	1.43	5.15	1.58	0.00
S	Global	Andesite		2300	-1.00	14.87	0.92	3.46	2.03	0.00

19.5.2 Population Statistics

Cumulative probability plots were generated for all domains. Comparison of the complete oxide mineralisation to the complete transitional and sulphide mineralisation (Figure 19-7) supports the separate domaining of the mineralisation types. The transitional domain has a limited number of observations for each of the variables so due to the support of a similar mean and median to the sulphide observations, the composites were grouped with the composites bearing a sulphide code for the purpose of grade estimation.

The log probability plots support the univariate statistical observation of a difference in populations when separated by geological domain.

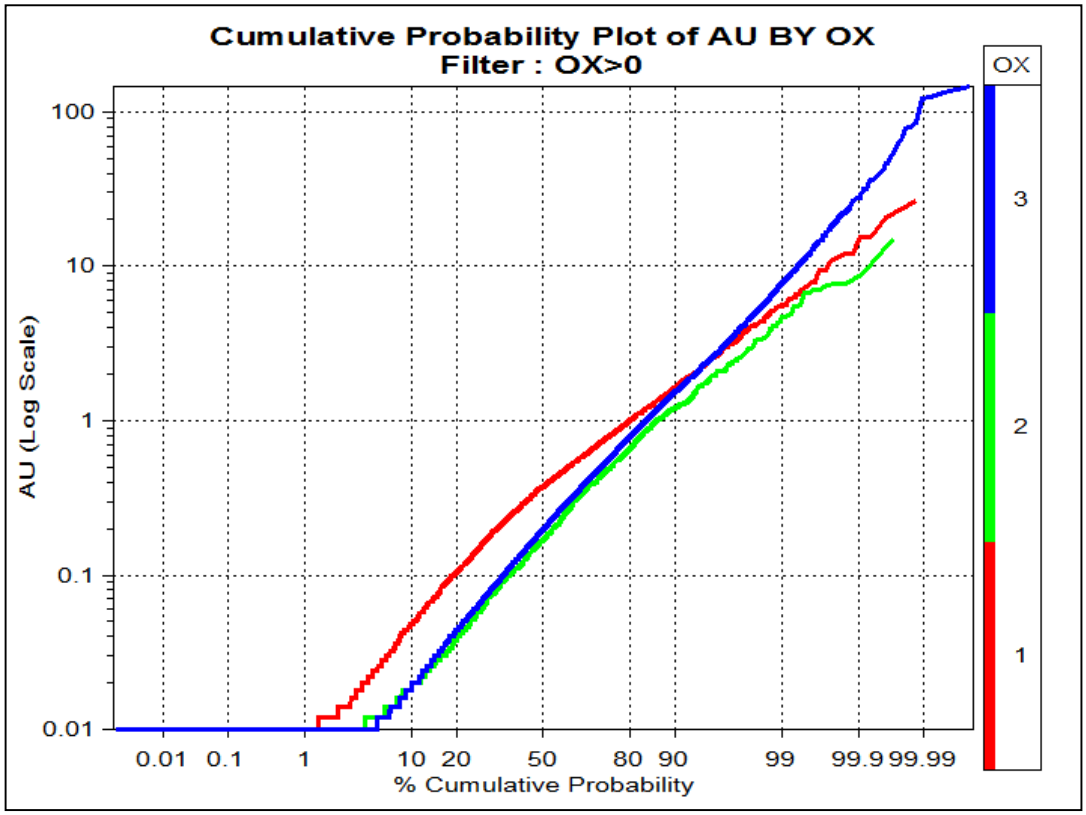

Figure 19-7: Log probability plot of Au plotted for oxide (red), transitional (green) and sulphide (blue) comparisons

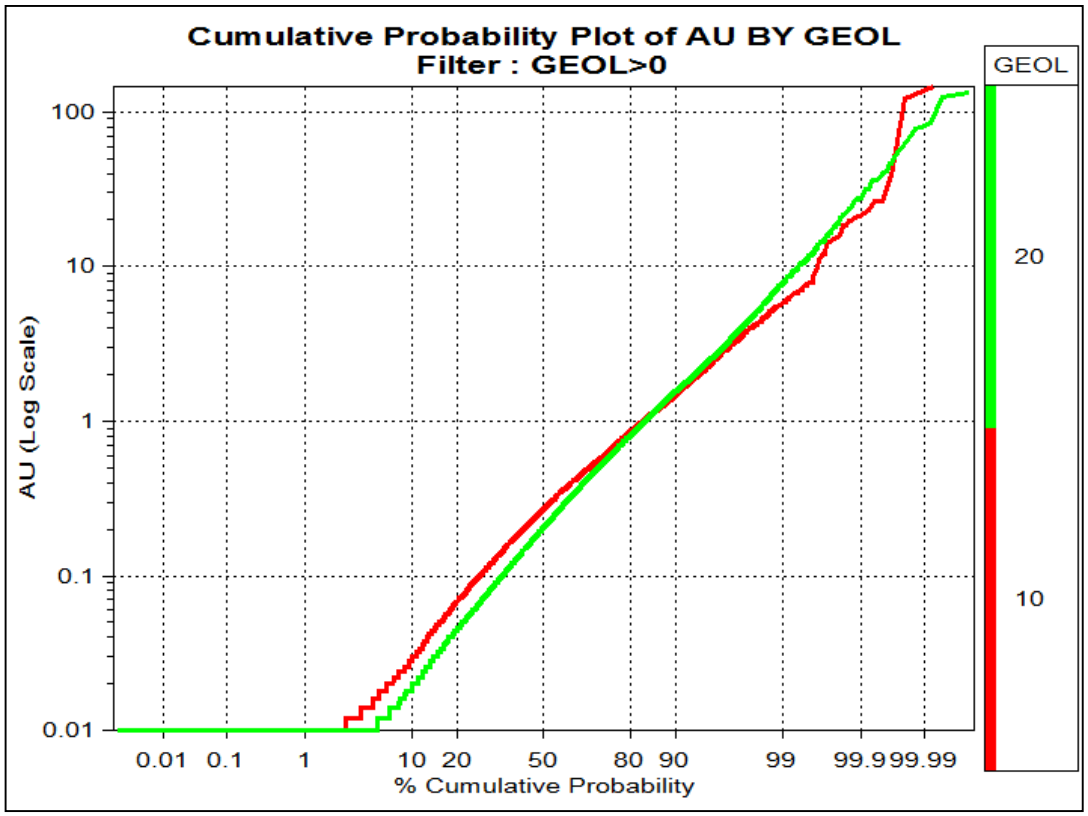

Figure 19-8: Log probability comparing Au with Geology. Geol = 2 is Tuff; Geol = 1 is Andesite

19.6 Treatment of High-Grade Samples during Estimation

Cumulative log-probability plots were examined for each estimation domain to assess for population outliers (anomalous values) that may adversely impact on grade estimation. These values were identified from the probability plots by observing for significant points of inflexion, which represent a change in variance, or checking for a tail of very high value(s) that depart from the overall trend of the data. Final thresholds were selected from the total oxide and total transitional/sulphide plots for Pigiput.

High-grade outlier values were managed during the grade estimation phase by spatial restraining. No high grade cutting was applied to Au, Ag or S in the oxide domain.

The high grade thresholds applied to the models are detailed under the kriging plan.

19.7 Variographic Analysis

19.7.1 Introduction

The objectives of the variography were to establish the directions of major grade continuity for grade variables Au, Ag and S and to provide variogram model parameters for use in geostatistical grade interpolation.

Variography was examined by oxide and transitional/sulphide mineralisation type. Experimental variograms were generated from the 5 m composite data.

Directional variography requires search tolerances to be used for calculation of variograms, to address the fact that the drill hole samples are not perfectly aligned in a given direction in 3D space, and are not equally spaced along that direction. This requires the use of angular and distance tolerances. The tolerances used

for directional variogram calculation are provided in Table 19-16 between the angular and distance tolerances with respect to the direction in which the variogram is required to be calculated.

An overview of the variography procedure used is as follows:

- Variograms were generated with a 20 m lag interval, in 10° increments horizontally and 10° increments vertically to provide directional coverage. The vertical bandwidth was selected to be as constrained as possible to avoid vertical smoothing through the profile when calculating variograms.
- Correlograms were used for spatial continuity analysis as these generally produced the clearest variogram structure in the domains examined.
- The nugget variances were modelled from downhole variograms based on a 5 m lag, reflecting the downhole composite spacing. The downhole variogram model provided the most accurate interpretation of the nugget effect due to the closer sample spacing.
- Directions of continuity were selected from variogram maps. A east-north-east to west-south-west trend (70°) was identified as the primary direction of mineralisation in the transitional/sulphide material for grade variable Au which is consistent with the regional trend. The primary direction of continuity of the oxide material was downhole (plunge 33 -> 000). This was partially masked by the north-east to south-west distribution of lodes. The primary direction of mineralisation for grade variable Ag was a north-east to south-west trend (50°) in both oxide and transition/sulphide domains.
- Directional variograms were generally poorly structured for S. Omni-directional variography was modelled for S in the oxide and transitional/sulphide to obtain kriging parameters for the resource estimation.

Table 19-16: Tolerances used for directional variogram calculation

Parameter	Value
Start Azimuth (**ang**)	0°
End Azimuth	180°
Step Azimuth	10°
Start Plunge	-90°
End Plunge	90°
Step Plunge	5°
Horizontal Angle Tolerance (**atol**)	20°
Vertical Angle Tolerance (**vtol**)	20°
Horizontal Distance Bandwidth (**bandw**)	10 m
Vertical Distance Bandwidth	10 m
Lag Distance (**xlag**)	20 m
Lag Tolerance (**xltol**)	10 m

19.7.2 Variography Interpretation and Modelling

Variography was carried out for each variable (Au, Ag and S) within the two variographic groups (oxide and transitional/sulphide) separately for the Pigiput, Pigibo and Sorowar South Domains. The approach was to interpret and model the orthogonal orientations which reflect the interpreted major, semi-major and minor axes of continuity. The principal directions were modelled with two-structure spherical models.

The sub-horizontal nature of the mineralisation resulted in correlograms modelled along a horizontal plane, with the major axis orientated in the general strike orientation of the mineralisation.

19.8 Grade Interpolation Strategy

Grade estimation of the Pigiput deposit was carried out using the geostatistical method of Ordinary Kriging (OK) using Vulcan. All variables were interpolated by OK. The OK method used estimation parameters defined by the variography. The estimation was performed by oxide and transitional/sulphide zone according to the respective wireframes. Au, Ag and S grades were estimated by OK within all domains.

The geological block model used a parent block size of 10 by 10 by 10 m for blocking purposes with sub-blocks set as 5 by 5 by 5 m. Given the nominal average drilling density of 20 by 40 m, the parent block size is reasonable to ensure local estimation quality and reduce conditional bias.

Un-estimated blocks were assigned default grade values. These default values were the mean estimated block grade by domain, for each variable. The default values estimation pass was assigned to 4.

The interpolated Vulcan block models were named *pp_062010.bmf (Pigiput and Pigibo) and ss_062010.bmf (Sorowar South).*

19.9 Kriging Plan

The OK estimation was run in a three-pass kriging plan, the latter passes using progressively larger search neighbourhoods to enable the estimation of blocks un-estimated on the first pass.

Further explanation and details are as follows:

- Sample weighting determined by variogram model parameters (kriging approach) for all variables.
- Block discretisation was set to 5 (X) by 5 (Y) by 2 (Z) to estimate block grades of 10 m by 10 m by 10 m parent blocks, using nominal 5 m composites. Sub-cells of 5 m by 5 m by 5 m received the parent cell estimate.
- A three-pass kriging plan was used; Pass 1 search of -40 by 40 by 10 m; Pass 2 search of 8 by 80 by 20 m and a third of 120 by 12 by 20 m. Search dimensions were selected based on drilling density and configuration. The Z direction was limited to restrict grade from estimating far beyond the last sample.
- A minimum of two composites using an octant based search where a maximum of 2 samples per octant were allowed equating to a maximum of 16 composites per estimate. Pass 2 used a minimum of four and two composites and pass 3 one and one respectively.
- Any remaining un-estimated blocks were assigned the average value from the kriged estimates and an estimation pass value of 4.
- Spatial restraining was used to limit the influence of outlier high grades for all estimations. A 5.1 by 5.1 by 5.1 m search was used for the restraint effectively limiting the outlier high grades to one block in the model. Table 19-19 details the high grades restraint values used.

A summary of parameters is provided in Table 19-17 and Table 19-18.

Table 19-17: Pigiput deposit Kriging Plan Parameters

Parameter	Values
Discretisation (X/Y/Z)	5/5/2
Search type	Octant search
Minimum No. samples (Pass 1/2/3)	4/4/1
Maximum No. samples	32 (4 per octant)
Minimum Octants	2/2/1
min samp per oct	1

Table 19-18: Search Parameters

Domain	Orientation	Ellipsoid radii (m)								
	Azimuth/Plunge	pass1			pass2			pass3		
	Dip (R1/R2/R3)	X	Y	Z	X	Y	Z	X	Y	Z
Andesite (Au/Ag/S)										
oxide	90/0/40	40	40	10	80	80	20	120	120	20
Trans/Fresh		40	40	10	80	80	20	120	120	20
Tuff (Au/Ag/S)										
oxide	90/0/40	40	40	10	80	80	20	120	120	20
Trans/Fresh		40	40	10	80	80	20	120	120	20

19.10 Treatment of High-Grade Samples During Estimation

Grades estimates are usually strongly influenced by outlier values. This can result in an over-estimation of the global grades. To prevent this, high-grade treatment was applied to the Pigiput model using spatial restraining above grade thresholds.

The high grade thresholds were initially chosen according to discontinuities in the cumulative probability plots for the respective estimation domains, and iteratively adjusted based on visual and statistical validation of the model against the drill hole data. The final thresholds used are summarised in Table 19-19.

Table 19-19: Summary of high grade thresholds

Geology	Domain	Threshold Au	Threshold Ag	Threshold S
Andesite	oxide	2	6	999
	Trans/Fresh	6	6	8
Tuff	oxide	7	6	4
	Trans/Fresh	8	20	12

19.11 Density Assignment

Dry Bulk Density values were applied to the model based on depth below surface. Independent regression equations were developed for the oxide, transition and sulphide zones in all domains based on the available core density measurements stored in the Allied drill holes database.

Table 19-20: Dry Bulk Density Calculation

Material	Calculation
oxide	1.40 + (depth/250*1.06)
transition	1.65 + (depth/100*0.456)
sulphide	2.00 + (depth/350*0.500)

19.11.1 Validation of Grade Estimates

Statistical and visual assessment of the block model were undertaken to assess successful application of the various estimation passes and to ensure that as far as the data allowed, all blocks within ore domains were estimated and the model estimates considered acceptable.

19.11.2 Visual Assessment of Grade Estimates

Onscreen validations between samples and blocks were completed on the model. The onscreen validation process involved comparing block estimates and composite grades in cross-section and in plan.

19.11.3 Statistical Assessment of Grade Estimates

Statistical assessment of the grade estimates included mean grade reproduction checks and swath plots. The grade comparisons are influenced by the method of high-grade treatment applied in the estimation and data sharing between domains, which make representative comparisons more difficult. Composites and blocks were compared within the oxide and sulphide domains for each model.

19.11.4 Mean Grade Reproduction

The global estimated mean domain grade against 5 m composite data was checked as a means to validate reproduction of the mean grade of the data. This comparison is shown for all estimation domains in Table 19-21. Global statistical comparisons of the mean grade are influenced by varying drill hole density, data sharing between domains and high-grade treatment. These issues can sometimes distort the comparison. In particular, peripheral blocks representing extrapolated estimates can create apparent anomalies in the grade conformance. Differences observed in the global average grades may not be evident on swath comparisons, which is an advantage of using swath plots to assess conformance.

Overall, reproduction of the data average is acceptable. Some apparent under and over-estimation of the domain average in the block model for some domains is likely to be an artefact of the issues described above. The swath validations do not show the same discrepancies and are probably a more representative indication of the average grade conformance.

Table 19-21: Composite vs. Block Model Grades

Domain	Variable	1 m Drill Hole Composites				Ordinary Kriging			OK/DH(%)[1]
		No.	Mean	Decl. Mean	Var.	No.	Mean	Var.	
Pigibo	*AU*	2232	0.66	0.63	1.66	51559.00	0.47	0.26	74.92
Pigiput	*AU*	9279	0.76	0.70	7.02	130129.00	0.65	0.63	92.86
Sor.Sth	*AU*	1455	0.39	0.36	1.77	29514.00	0.16	0.02	44.01
Pigibo	*AG*	1802	1.48	1.44	10.40	46172.00	1.38	1.47	95.83
Pigiput	*AG*	8011	1.66	1.56	29.54	129045.00	2.04	11.81	130.98
Sor.Sth	*AG*	1234	1.35	1.31	3.39	28246.00	1.17	0.92	89.02
Pigibo	*SUL*	1036	2.34	2.39	6.41	37963.00	3.03	4.42	126.74
Pigiput	*SUL*	4520	2.36	2.23	3.42	122942.00	2.60	2.21	116.43
Sor.Sth	*SUL*	891	3.12	3.22	2.52	15499.00	3.58	1.32	111.12

19.11.4.1 Swath Plot Validations

Swath plots are used to assess the block model estimates for global bias; the estimates should have a close relationship to the drill hole composite data used for estimation.

The process involved averaging both the blocks and samples in panels of 20 m (easting) by 20 m (northing) by 20 m (RL). The relationship between model and sample panel averages was assessed in the form of scatter plots and Q-Q plots. Conformance of the model and sample average grades was assessed in the form of easting, northing and RL swaths of the panel averages.

Generally good agreement is observed between the data and block model Au mean grade for easting, northing and RL slices. QQ and scatter plots for the averaged sample data vs. block model results show deviation from the 45° line, with overstatement of low grades and understatement of high grades by the block model. This is a natural expected behaviour of moving from sample size data to a much bigger volume as represented by Kriged models.

Overall, the swath plot validation process shows that the block model estimates follow the trend of the 5 m composite grades across the deposit. From the perspective of conformance of the average model grade to the input data, Golder considers the model to be a satisfactory representation of the drill hole data used.

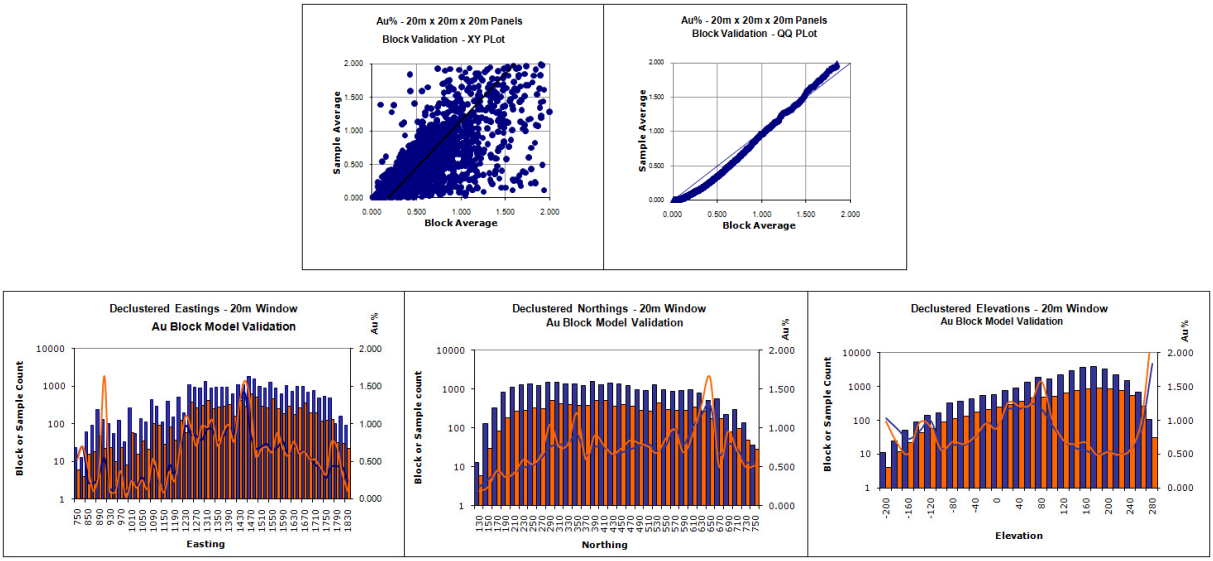

Figure 19-9: Scatter Plot, QQ plot and swath plots of Au in Pigiput

19.12 Mineral Resource

19.12.1 Resource Classification

Resources were classified in compliance with the Australasian Code for the Reporting of Identified Mineral Resources and Ore Reserves (JORC, 2004).

The Au estimation has been classified as Measured, Indicated and Inferred based on data quality and density. The Measured Resource is confined to the oxide material at Pigiput. The Indicated Resource has a nominal data density of 50 m by 50 m and the Inferred Resource 100 m by 100 m.

Material south of the interpreted mineralisation footwall (*pig_fw.00t*) and outside the Inferred Resource boundary is not classified. Due to the under sampling of the historical drilling, Ag and S are classified as Inferred.

19.12.2 Mineral Resource Statement

The 2010 Resource Statement for Pigiput, Pigibo and Sorowar South deals with the Gold Mineral Resource for these deposits located on Simberi Island of the Tabar Island Group of Papua New Guinea.

Allied Gold Limited (Allied) commissioned Golder Associates Pty Ltd (Golder) to combine and update the Pigiput and Pigibo deposit resource models on Simberi Island in June 2010. The combined model is based on the geological database as at 31 May 2010. The database includes RC and Diamond drill holes and some Diamond Core 'tails'. Geological modelling of the deposit was undertaken by Carmel Grant (Resource Geologist) and Phil Davies (Chief Geologist) from Allied. Construction of the block model and grade estimation was undertaken by Stephen Godfrey (Associate, Principal Resource Geologist) and Marnie

Mathews (Senior Resource Geologist) and reviewed by Richard Gaze (Principal). For this update, the oxide, transition and sulphide resource was modelled and estimated for gold (Au), silver (Ag) and sulphur (S).

The Pigiput-Pigibo model has been based on the structural/geological controls interpreted by Allied. The Sorowar South domain extends into the existing Sorowar model.

Drill sample data was composited to five metres and flagged using domain, geology, oxide, transition and sulphide wireframes. The flagged composites were used to estimate their corresponding geological/weathering domains in the block model.

Grade estimations were done using Ordinary Kriging. High grade restraining was applied to restrict the influence of high grade composites.

Dry Bulk Density values were applied to the model based on depth below surface. Independent regression equations were developed for the oxide, transition and sulphide zones in Pigibo and Pigiput.

The Au estimation has been classified as Measured, Indicated and Inferred based on data quality and density. The Measured Resource is confined to the oxide material at Pigiput. The Indicated Resource has a nominal data density of 50 m by 50 m and the Inferred Resource 100 m by 100 m. Material south of the interpreted mineralisation footwall and outside the Inferred Resource boundary is not classified. Due to the under sampling of the historical drilling, Ag and S are classified as Inferred.

The Pigiput/Pigibo/Sorowar South resource stated here is an *in situ* resource and has not been depleted for any mining activity.

The tables below summarise the Mineral Resource by classification, geology and material type at a cut-off of 0.5 g/t Au.

Table 19-22: Pigiput Model (PP_062010.bmf) – Measured and Indicated resource (0.5 g/t Au cut off)

Domain	Geology	Material	kt	Au g/t	k oz
Pigibo	Andesite	oxide	7	0.74	0.2
		transition	189	0.84	5
		sulphide	1097	0.94	33
	Andesite Total		1294	0.92	38
	Tuff	oxide	2952	1.11	106
		transition	1996	1.23	79
		sulphide	2759	1.17	104
	Tuff Total		7707	1.16	289
Pigibo Total			9001	1.13	327
Pigiput	Andesite	oxide	520	0.81	14
		transition	594	0.86	16
		sulphide	22190	1.59	1134
	Andesite Total		23304	1.55	1164
	Tuff	oxide	7415	0.93	222
		transition	1356	0.91	39
		sulphide	10373	1.34	448
	Tuff Total		19144	1.15	710
Pigiput Total			42448	1.37	1874
Grand Total			51449	1.33	2201

Table 19-23: Pigiput Model (PP_062010.bmf) – Inferred resource (0.5 g/t Au cut off)

Domain	Geology	Material	kt	Au g/t	k oz
Pigibo	Andesite	oxide	94	0.87	3
		transition	454	0.94	14
		sulphide	5737	0.92	169
	Andesite Total		6285	0.92	185
	Tuff	oxide	508	0.89	15
		transition	54	0.84	1
		sulphide	484	1.22	19
	Tuff Total		1047	1.04	35
Pigibo Total			7332	0.93	220
Pigiput	Andesite	oxide	278	0.7	6
		transition	88	0.74	2
		sulphide	23564	1.02	774
	Andesite Total		23929	1.02	782
	Tuff	oxide	1731	0.81	45
		transition	685	0.84	18
		sulphide	5896	0.99	187
	Tuff Total		8311	0.94	250
Pigiput Total			32240	1	1033
Sorowar South	Andesite	oxide		-	-
		transition		-	-
		sulphide	891	0.6	17
	Andesite Total		891	0.6	17
	Tuff	oxide		-	-
		transition		-	-
		sulphide	64	0.55	1
	Tuff Total		64	0.55	1
Sorowar South Total			955	0.6	18
Grand Total			40527	0.98	1271

Table 19-24: Sorowar South Model (SS_062010.bmf) – Inferred resource (0.5 g/t Au cut off)

Domain	Geology	Material	kt	Au g/t	k oz
Pigiput	Andesite	oxide		-	-
		transition		-	-
		sulphide	2791	0.9	81
	Andesite Total		2791	0.9	81
	Tuff	oxide		-	-
		transition		-	-
		sulphide		-	-
	Tuff Total			-	-
Pigiput Total			2791	0.9	81
Sorowar South	Andesite	oxide	682	0.82	18
		transition	283	0.68	6
		sulphide	4433	0.67	95
	Andesite Total		5398	0.69	120
	Tuff	oxide		-	-
		transition		-	-
		sulphide		-	-
	Tuff Total			-	-
Sorowar South Total			5398	0.69	120
Grand Total			8189	0.76	201

Notes:
The Sorowar South model contains only Inferred resource.
The Sorowar South domain of the Sorowar South model replaces 12.2 Mt at 0.86 g/t Au in the Sorowar resource (based on 0.5 g/t Au cut off in model *sor_0902.bmf*).

Table 19-25: Pigiput Model (PP_062010.bmf) – Inferred Ag and S resource (0.5 g/t Au cut off)

Domain	Geology	Material	kt	Ag ppm	S ppm
Pigibo	Andesite	oxide	102	0.98	1.15
		transition	643	1.42	2.80
		sulphide	6835	1.91	3.54
	Andesite Total		7579	1.86	3.45
	Tuff	oxide	3460	0.80	0.33
		transition	2050	1.96	1.98
		sulphide	3243	2.69	3.67
	Tuff Total		8754	1.77	1.95
Pigibo Total			16333	1.81	2.65
Pigiput	Andesite	oxide	798	0.30	0.18
		transition	682	0.47	0.78
		sulphide	45753	5.21	3.56
	Andesite Total		47233	5.06	3.47
	Tuff	oxide	9145	0.56	0.46
		transition	2040	0.68	2.49
		sulphide	16269	1.92	3.66
	Tuff Total		27455	1.37	2.50
Pigiput Total			74688	3.71	3.11
Sorowar South	Andesite	oxide		-	-
		transition		-	-
		sulphide	891	2.77	3.93
	Andesite Total		891	2.77	3.93
	Tuff	oxide		-	-
		transition		-	-
		sulphide	64	1.35	3.58
	Tuff Total		64	1.35	3.58
Sorowar South Total			955	2.67	3.91
Grand Total			91976	3.36	3.04

Table 19-26: Sorowar South Model (SS_062010.bmf) – Inferred Ag and S resource (0.5 g/t Au cut off)

Domain	Geology	Material	kt	Ag ppm	S ppm
Pigiput	Andesite	oxide		-	-
		transition		-	-
		sulphide	2791.1	4.98	3.59
	Andesite Total		2791.1	4.98	3.59
	Tuff	oxide		-	-
		transition		-	-
		sulphide		-	-
	Tuff Total			-	-
Pigiput Total			2791.1	4.98	3.59
Sorowar South	Andesite	oxide	682.1	0.36	1.31
		transition	282.7	0.76	2.62
		sulphide	4432.8	2.67	3.64
	Andesite Total		5397.7	2.28	3.29
	Tuff	oxide		-	-
		transition		-	-
		sulphide		-	-
	Tuff Total			-	-
Sorowar South Total			5397.7	2.28	3.29
Grand Total			8188.7	3.2	3.4

19.12.3 Pigicow and Bekou Resource Estimation

19.12.3.1 Databases

The Pigicow and Bekou resources are both based on drill hole databases as at 23 October 2006.

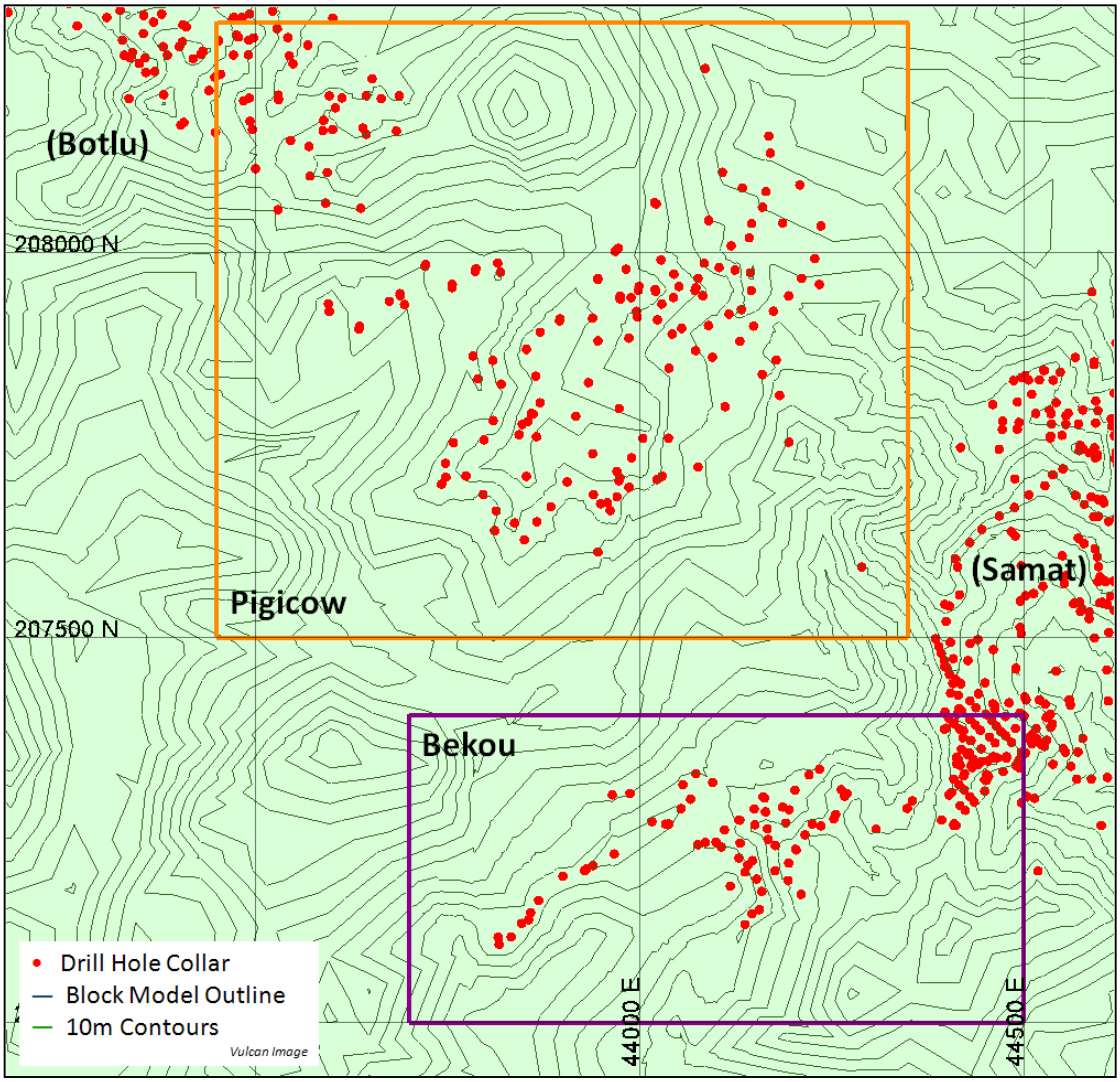

Figure 19-10: Pigicow and Bekou – Resource model areas and drill hole locations

Table 19-27: Pigicow and Bekou – Drill Holes

Pigicow	Reverse Circulation	Diamond	RC/Diamond Tail	Total
Number of holes	123	n/a	2	125
Metres	5175	n/a	450.4	5625.4
Bekou	**Reverse Circulation**	**Diamond**	**RC/Diamond Tail**	**Total**
Number of holes	68	3	n/a	71
Metres	2381	508.05	n/a	2889.05

Table 19-28: Pigicow and Bekou – Azimuths of Angled holes

Azimuth	Number of holes
350° - 0°	19
30° - 50°	12
180°	18
200° - 230°	32
270º	2
Total	***83***

19.12.3.2 Exploratory Data Analysis

19.12.3.2.1 Composites

93% of the samples in the database are 1 m in length. After analysis a 3 m composite size was used for estimation, to minimise differences in support for all samples and moderating the sample variance whilst retaining sufficient numbers of samples for grade estimation. The 3 m length was also considered a compromise between smoothing of the original variability and maintaining sufficient resolution for the geological wireframes. The same 3 m composite length was used for the Bekou drilling.

19.12.3.2.1.1 Statistical Analysis

The Bekou and Pigicow composites were summarised statically. Figure 19-11 illustrates the sample populations and shows the difference in grades between the oxide and sulphide domains which supports treating them separately during the interpolation of the block models. The high grade outliers for both deposits can be seen on the graphs also.

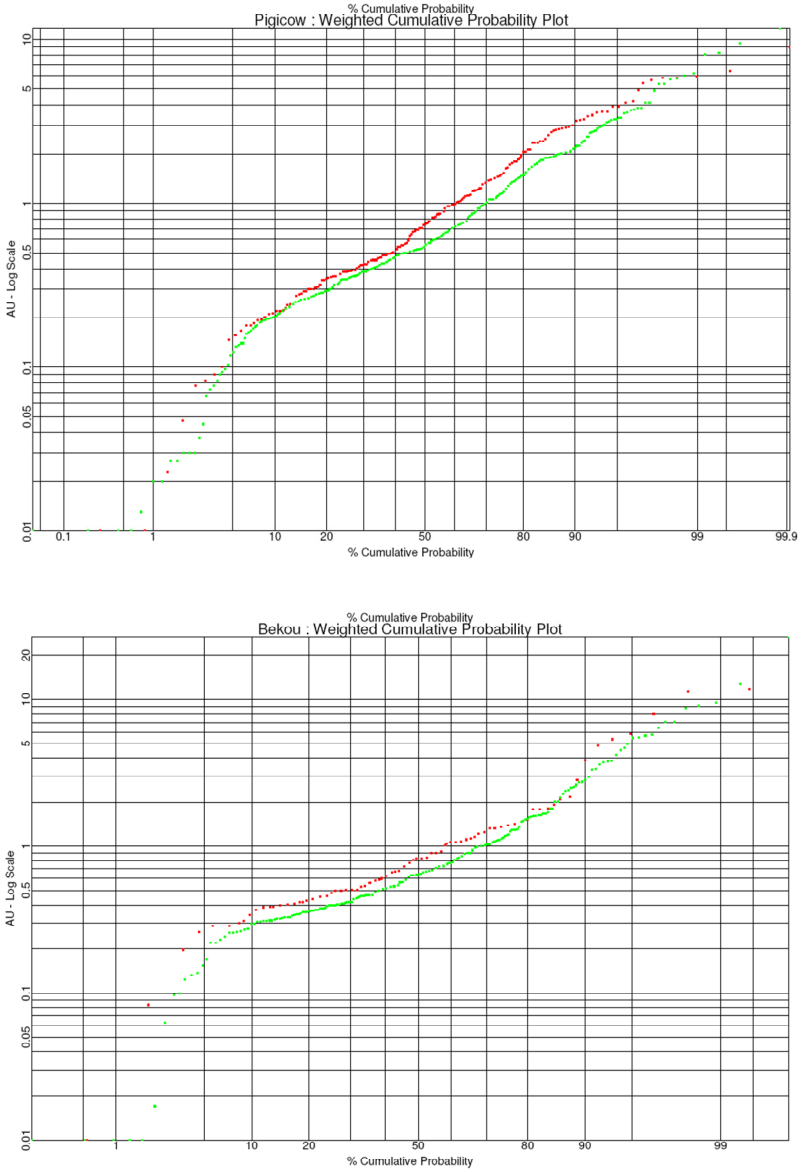

Figure 19-11: Bekou and Pigicow Oxide Sulphide Comparison

19.12.3.3 Assays

Table 19-29 details the assays available in the database. Au is available for all composites Ag and As for most. For the Bekou and Pigicow resource only Au was estimated

Table 19-29: Bekou and Pigicow – Number of Assays per analyte

	Au	Ag	As	Cu	Fe	Mn	Pb	S	Sb	Zn
Assays	7894	7503	7501	4766	3964	4506	4766	4330	4672	4576
Percentage	99.9%	95.0%	94.9%	60.3%	50.2%	57.0%	60.3%	54.8%	59.1%	57.9%

19.12.3.4 Geological Interpretation

Mineralisation domains were interpreted and wireframed by Allied. Mineralisation follows the regional south-east to north-west trends and to a lesser extent the south-west to north-east trend. Pigicow and Bekou have been modelled as a series of subvertical sulphide bodies with an overlying sub-horizontal oxide component. Figure 19-12 illustrates the position of the Pigicow and Bekou mineralisation relative to the drilling at Pigicow, Bekou and the adjacent Samat.

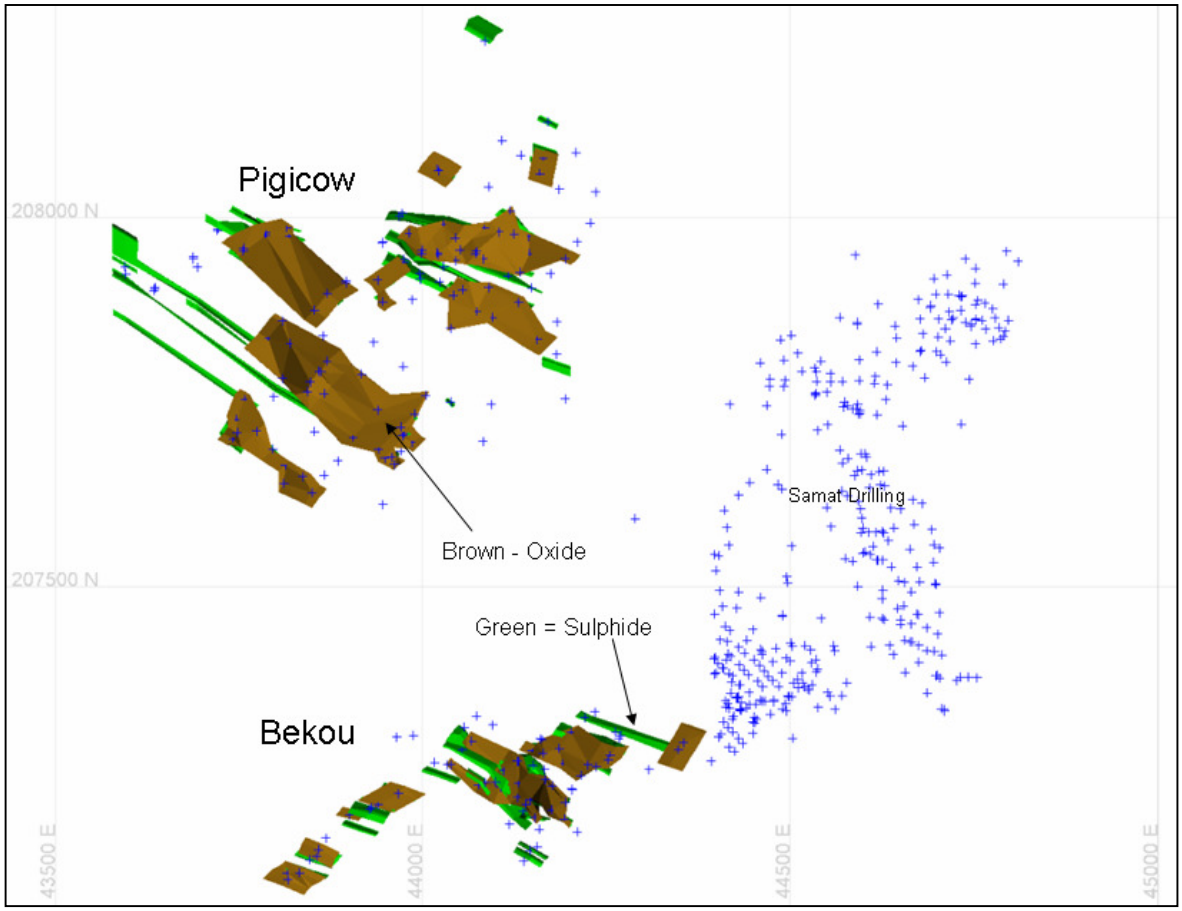

Figure 19-12: Pigicow and Bekou Mineralisation (blue crosses mark drill hole collar locations)

The Bekou and Pigicow mineralisation was interpreted and modelled in 3D by Allied Gold. For model construction and estimation Allied provided Golder with the mineralisation wireframe models summarised in Table 19-30 . Wireframe surfaces were also supplied for the base of oxide and top of sulphide. The topography wireframe was based on survey in November 2006.

Table 19-30: Bekou and Pigicow – mineralisation triangulations

Deposit	Sulphide	Oxide
Bekou	14	8
Pigicow	38	8

The wireframe models were validated by Golder prior to incorporation into the resource models.

19.12.3.5 Spatial Analysis

Directional variography was undertaken to determine the directions of maximum continuity of mineralisation and provide input parameters for the kriging process. Poor directional variograms were produced for both Bekou and Pigicow. This is due to the dispersed nature of the mineralisation and drill density over the areas. Omni directional variograms were modelled to provide parameter for the estimation kriging.

19.12.3.6 Domaining

During the block model construction and estimation the individual pods of oxide and sulphide mineralisation were treated as separate domains. Only samples within a pod of ore were used in the estimation of that pod.

19.12.3.7 Resource Block Model

A geological block models for Pigicow and Bekou were constructed in Vulcan. The models were aligned to the local grid with an origin at 0 m north, 0 m East and an RL of 0 m.

Offsets from the origin and block sizes are detailed in Table 19-31

Table 19-31: Model Dimensions

Model/Direction	Offset Start	Offset End	Model Extent (m)	Parent Cell Size (m)	Sub-Cell Size (m)
Bekou					
X	43700	44500	800	10	1
Y	207000	207400	400	10	1
Z	-60	170	230	5	1
Pigicow					
X	43450	44350	900	10	1
Y	207500	208300	800	10	1
Z	-160	230	390	5	1

19.12.3.8 Interpolation Plan

The OK estimation was run in a three-pass kriging plan, the second and third passes using progressively larger search neighbourhoods to enable the estimation of blocks un-estimated on the first pass. A summary of parameters is provided in Table 19-32.

Further explanation and details are as follows:

- Sample weighting determined by variogram model parameters (kriging approach) for all variables, using unfolding to reference surfaces for each domain.
- Block discretisation was set to 3 (X) by 3 (Y) by 2 (Z) to estimate block grades of 10 m by 10 m by 5 m parent size, using nominal 3 m composites. Sub-cells received the parent cell estimate.
- A three-pass kriging plan was used; Pass 1 search – 40 by 40 by 20 m; Pass 2 search -80 by 80 by 16 m; Pass 3 search 160 by 160 by 80 m. Search dimensions were selected based on drilling density and configuration.
- A minimum of 6 composites in total and a maximum of 4 composites per octant was used which equates to a maximum of 32 composites per estimate. Passes 2 and 3 used a minimum of 4 and 2 composites per octant respectively.

- The effect of high-grade composites above designated thresholds was limited by applying spatial restraining of values within a search of 5.1 by 5.1 by 7.5 m.

Table 19-32: Pigicow and Bekou Kriging Plan Parameters

Estimation Method Search Radius	OK
X radius (pass 1/2/3)	40/80/160
Y radius (pass 1/2/3)	40/80/160
Z radius (pass 1/2/3)	20/40/80
Discretisation (x/y/z)	3/3/2
Search type	Octant
Minimum No. composites (pass 1/2/3)	6/4/2
Maximum No. composites per octant (pass 1/2/3)	4/4/4
Minimum No. composites per octant	1
Maximum No. composites per hole	5
Minimum No. of octants (pass 1/2/3)	4/4/1

High-grade outlier composites were managed in the Pigicow and Bekou models using spatial restraining above grade thresholds.

The high grade thresholds were initially chosen according to discontinuities in the cumulative probability plots for the respective estimation domains, and iteratively adjusted based on visual and statistical validation of the model against the drill hole data. The final thresholds used are summarised in Table 19-33.

Table 19-33: Summary of High-Grade Thresholds

Model	Domains	Threshold
Bekou	Oxide	10
Bekou	Sulphide	10
Pigicow	Oxide	6
Pigicow	Sulphide	10

19.12.3.9 Bulk Densities

Dry Bulk Density was assigned using a spherical function developed by Allied Gold. The function has a nugget of 1.3 t/m^3 and a sill differential of 0.45 t/m^3 at a range 35 m, i.e. at surface the bulk density is 1.3 and at 35 m below surface bulk density is 1.75. This was calculated with the following formulae:

$$\gamma(h) = c_0 + c\left[\frac{3}{2}\frac{h}{a} - \frac{1}{2}\frac{h^3}{a^3}\right] \text{ for h} \leq \text{a}$$

and

$$\gamma(h) = c_0 + c \text{ for h} > \text{a}$$

Table 19-34 explains the variables used above.

Table 19-34: Pigicow and Bekou Details of Variables

Variable	Name	Description	Value
c_o = 1.3,	Nugget	Minimum SG – at surface	1
c = 0.45,	Sill differential	c + c_o = Maximum SG	0.45
a = 35	Range	Depth at which maximum SG reached	35
h	Lag	Depth below surface	variable

19.12.3.10 Mineral Resource Classification

Resources were classified in compliance with the Australasian Code for the Reporting of Identified Mineral Resources and Ore Reserves (JORC, 2004).

The resource estimates have been classified based on data density, data quality, confidence in the geological interpretation and confidence in the estimation. The resources for Pigicow and Bekou are predominantly Inferred with small areas of closely drilled oxide material being classified as Indicated.

19.12.3.11 Mineral Resource Tabulation

Table 19-35: Pigicow – Mineral Resource – 0.5 g/t Au cut off

	Indicated			Inferred		
Zone	**Kt**	**Au**	**Oz**	**Kt**	**Au**	**Oz**
Oxide	150	1.65	7,957	289	1.30	12,079
Transition			0	105	1.29	4,355
Sulphide			0	1,999	1.26	80,979
Total	150	1.65	7,957	2393	1.27	97,413

Table 19-36: Bekou – Mineral Resource – 0.5 g/t Au cut off

	Indicated			Inferred		
Zone	**kt**	**g/t Au**	**Oz**	**kt**	**Au**	**Oz**
Oxide	42	1.74	2,350	58	1.14	2,126
Transition	6	1.17	226	48	1.16	1,790
Sulphide	24	1.93	1,489	916	1.39	40,936
Total	72	1.76	4,064	1022	1.37	44,852

19.12.3.12 Block Model Validation

Statistical and visual assessment of the block model were undertaken to assess successful application of the various estimation passes and to ensure that as far as the data allowed, all blocks within ore domains were estimated and the model estimates considered acceptable.

Visual Assessment of Grade Estimates

Onscreen validations between samples and blocks were completed on the model. The onscreen validation process involved comparing block estimates and composites grades in cross-section and in plan.

Mean Grade Reproduction

For the Pigicow and Bekou models the overall reproduction of the mean grade is acceptable. Some apparent under and over-estimation of the domain average in the block model for some domains is likely to

be an artefact of the issues described above. The swath validations do not show the same discrepancies and are probably a more representative indication of the average grade conformance.

Swath Plot Validations

Swath plots are used to assess the block model estimates for global bias; the estimates should have a close relationship to the drill hole composite data used for estimation.

The process involved averaging both the blocks and samples in panels of 40 m (easting) by 40 m (northing) by 10 m (RL). The relationship between model and sample panel averages was assessed in the form of scatter plots and Q-Q plots. Conformance of the model and sample average grades was assessed in the form of easting, northing and RL swaths of the panel averages.

Generally good agreement is observed between the data and block model au mean grade for easting, northing and RL slices. QQ and scatter plots for the averaged sample data vs. block model results show deviation from the 45° line, with overstatement of low grades and understatement of high grades by the block model. This is a natural expected behaviour of moving from sample size data to a much bigger volume as represented by Kriged models.

Overall, the swath plot validation process shows that the block model estimates follow the trend of the 3 m composite grades across the deposit. From the perspective of conformance of the average model grade to the input data, Golder considers the model to be a satisfactory representation of the drill hole data used (Figure 19-13 and Figure 19-14).

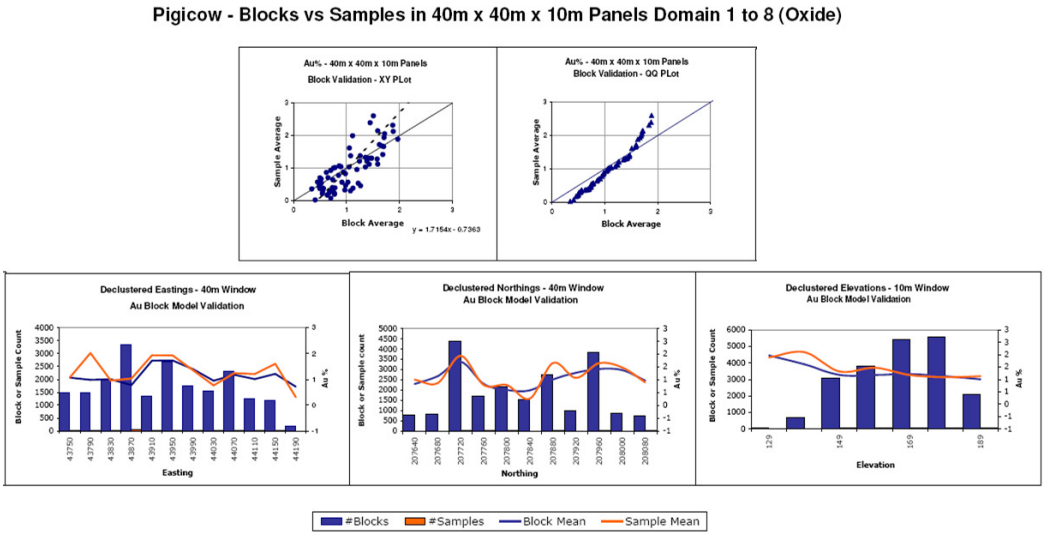

Figure 19-13: Pigibo – Block Model validation oxide swath plots

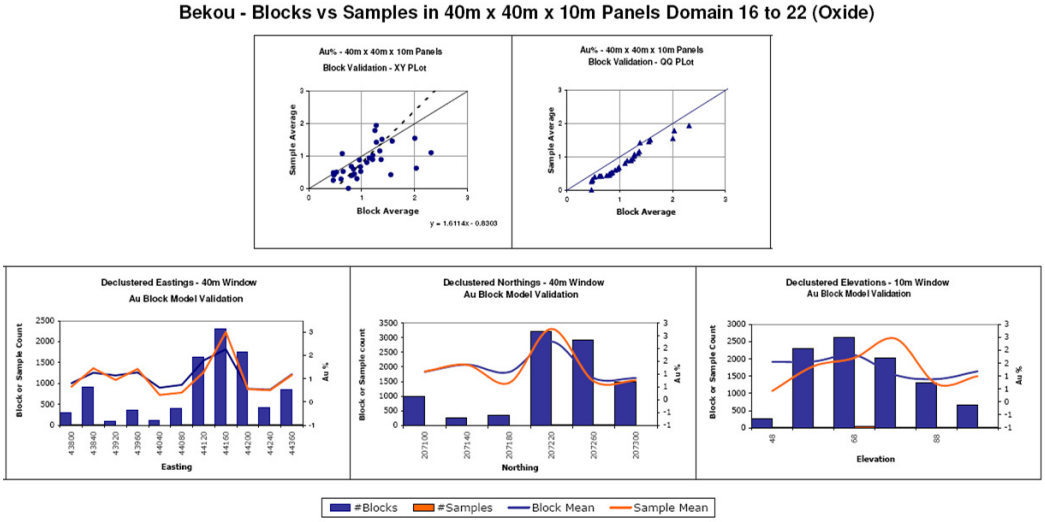

Figure 19-14: Bekou – Block Model validation oxide swath plots

19.12.4 Botlu Oxide Resource Estimation

The following modelling description is adapted from the M. Binns report on his modelling of the deposit.

The Botlu oxide resource block model was estimated using multiple indicator kriging between the topographic surface and an interpreted base-of-oxide surface. Most resource drilling was completed by 1996, but in 1997, several holes were drilled at the northern end of the oxide deposit, targeting IP geophysical anomalies.

In summary, there is a thin (up to 10 m) blanket of oxidised material over most of Botlu. In one location in the north of the prospect, oxidation extends down to 80 m depth in an area of about 100 m by 150 m. Surface 5 m channel samples return grades up to 12 g/t gold, and define a horseshoe shaped area of gold mineralisation. Drilling in general intersected higher oxide gold grades near surface, decreasing in tenor with depth. In the south-east of the prospect, however, moderate to high-grade gold in unoxidised material occurs from near surface to 100 m depth.

As in the other East Simberi prospects, at Botlu the oxide gold mineralisation is associated with highly fractured and altered porphyry and volcanoclastic lithologies. However, unlike the other prospects, quartz veining and silicification are common. Higher gold grades are associated with the silicification. The depth of oxidation is variable, from a few metres thick in parts of the south and south-east of the prospect, to about 80 m thick in the north.

Thicker oxide development occupies an area of about 100 m by 150 m, and appears to have reasonably well defined boundaries to its east and north-west. The east boundary coincides with a shallow valley, a tributary to Botlu Creek, and the north-west boundary coincides with Botlu Creek itself. Both boundaries may reflect structural control, and the deeper oxidation may have taken place in fractured ground between the structures. The area of deep oxidation also coincides with a ridge, probably due to the presence of silica. About 90 surface 5 m channel samples over this immediate area average 1.9 g/t gold. Gold grades (in channel samples) progressively decrease to the south-west, and the host lithologies become finer grained, and ultimately become crudely bedded tuffs.

Drilling within the deeply oxidised area shows that, on average, gold grades decrease with depth, though grade variability downhole is consistent with intersection of a series of steeply dipping mineralised zones.

In the areas of shallow oxide development, in the south and south-east of the prospect, both surface channel sampling and drilling indicate that there are several small areas of moderate grade oxide gold mineralisation,

immediately underlain by moderate grade sulphides. Past drilling by Kennecott intersected high-grade gold zones to about 100 m depth beneath this area. Additional drilling is required to better define and test the sulphide resources.

As in the other prospects, the main primary control of gold mineralisation is likely to be steeply dipping fracture systems, in places associated with milled breccia dykes (diatremes). Higher grades are associated with quartz veining and silicification, and in some places with sulphide veining. Iron oxides on fractures, and crackle brecciation are common.

The main information sources for interpretation of the oxide zone include:

- Prospect topography, partly surveyed by CPS in 1996, with additional compass and chain data, and 1997 CPS drill hole and channel sample location data.
- Gold assays from drill holes (Kennecott and Nord).
- Gold assays from Nord 5 m channel samples.
- Gold assays from Kennecott surface sampling (not used in block estimates).
- Oxide, transition and sulphide coding downhole.

Interpretation of mineralisation continuity on cross-sections and plan for the purpose of resource estimation concluded that the mineralisation was too complex at small scales for any higher-grade mineralised structures to be identified.

Consequently the oxide material was defined as the volume confined by the topography, the interpreted base-of-oxide boundary, and a peripheral boundary based on either a local average grade cut-off or lack of data. For the purpose of this resource estimate, the definition of the interpreted structurally controlled margin on the east of the deeper oxide zone was improved by making the interpreted oxidised zone thinner to the east of the margin.

Short and isolated (usually 1-2 m) transition or sulphide intersections were included within the oxide interpretation to avoid unnecessary complexity, but normally the base of the oxide zone was placed above the first appearance (downhole) of transition or sulphide material. In places, there were oxide intersections below the oxide boundary, probably in more deeply weathered structures, but these were not included in the oxide resource model.

19.12.4.1 Surveying

There has been no specific surveying campaign at Botlu to establish a topographic model. However, a substantial amount of survey data does exist.

All of the Kennecott hole collars were surveyed (by EDM) during Kennecott's initial exploration programme, and most were confirmed by re-surveying. CPS Palanga surveyed every fifth Nord 5 m channel sample endpoint, and the northing and easting of all the other channel sample midpoints were determined by fitting compass/chain surveys of the sampling to the CPS survey. CPS Palanga produced a map showing 10 m contours based on their surveying and field observations, which is probably the best resolution that can be expected from their survey point density. Also, CPS Palanga produced surveyed road and track formation lines, with 3-D coordinates. All these data, along with other point data (e.g. control points) were used to form a topographic DTM.

19.12.4.2 The Database

The total Botlu drill hole assay database consisted of 135 aircore or aircore/RC drill holes (total 8,565 m), with an additional 186 m of diamond tail, and 15 diamond drill holes (total 3,027 m). The drill hole traverse spacing was nominally 25-30 m, but very irregular due to the steep topography. The resource was estimated on the Simberi grid.

The base of oxidation was interpreted on cross-sections 20 m apart (8010N to 8490N), and digitised on-screen, snapping onto drill holes where possible. A digital terrain model (DTM) of the base of oxidation surface was formed, as was one of the surface topography. The two surfaces were linked to form a solid model that enclosed all the potentially mineralised oxidised material. This model was used to identify drill assay data inside it, for use in variography and block grade estimation.

Table 19-37 and Table 19-38 compare 2 m composite data from drill holes only, and from both drill holes and 5 m surface channel samples, all within or on the upper surface of the interpreted oxide model. There is a greater proportion of higher grades in the channel sample data than in the drill data, but at this prospect, there is a strong tendency for higher grades to occur near surface. There is also a clustering of surface data in higher-grade sections of the deposit, so these raw average grades will be biased.

Table 19-37: Botlu – 2 m composite statistics – drill holes

Topcut Avg >0.5	Avg	Median	Min	Max	SD	CV	
No cut	1.02	0.66	0.00	19.30	1.40	1.38	1.47
Topcut to 7.9 g/t (99%)	0.99	0.66	0.00	7.90	1.16	1.18	1.43
Topcut to 5.1 g/t (98%)	0.95	0.66	0.00	5.10	0.97	1.02	1.36

Table 19-38: Botlu – 5 m channel sample statistics

Topcut Avg >0.5	Avg	Median	Min	Max	SD	CV	
No cut	1.10	0.68	0.00	19.30	1.53	1.39	1.64
Topcut to 8.0 g/t (99%)	1.06	0.68	0.00	8.00	1.29	1.21	1.58
Topcut to 5.3 g/t (98%)	1.02	0.68	0.00	5.30	1.10	1.08	1.52

The full drill hole plus channel sample data set was used as input for block grade estimation, on the basis that the two data sets are statistically similar. However, only the drill hole composites were used to determine grade continuity.

Figure 19-15 illustrates the Botlu sample population distribution for the Oxide horizon.

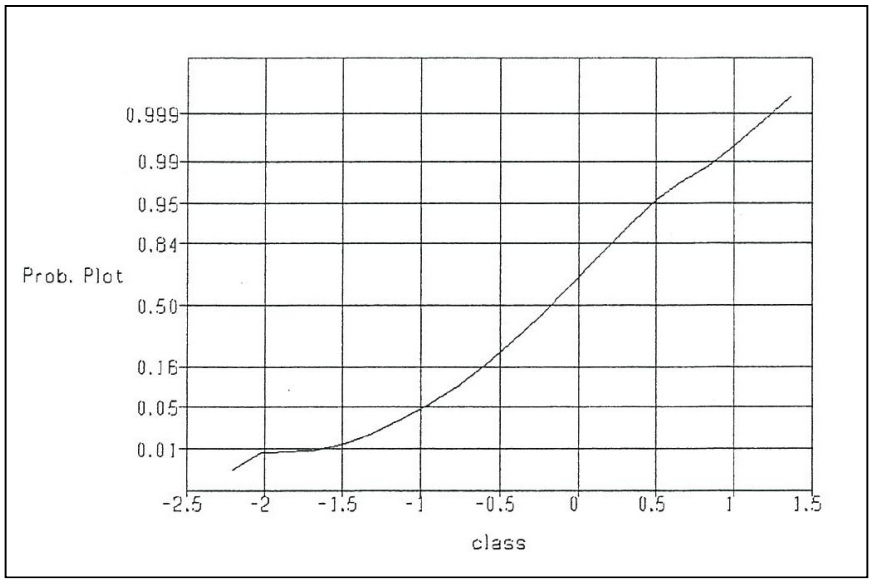

Figure 19-15: Botlu – Oxide Au cumulative frequency plot

19.12.4.3 Bulk Densities

In December 1996, Nord drilled two diamond holes (BO012, BO013) at Botlu, both in the area of deep oxide development. In March 1997, Nord drilled another three diamond holes (BO014-16) in areas of both oxide and sulphide mineralisation. A total of 144 oxide and 201 sulphide bulk density determinations were made. As in the other prospects, there is an increase in oxide dry BD with depth, though there is more scatter. The results showed that bulk densities were generally higher than had been determined in the other prospects, probably due to silica content. In BO012, there were some lower BDs (around 1.2-1.5) in the vicinity of core loss, probably in a fracture zone. In BO013, there were some BDs around 2.0 associated with a near-surface sulphidic zone with oxide above and below it. Both these occurrences were discounted when estimating the relationship between oxide bulk density and depth below surface. Figure BO-4 shows the relationships between depth and bulk density for both oxide and sulphide samples.

The variation of dry bulk densities with depth was modelled by the spherical function used for representing variograms. Using this, the dry BD is 1.25 at surface, and 1.9 at 35 m depth and below, with a progressive variation according to the spherical function in between. The spherical model parameters are nugget = 1.25, sill = 0.65, range = 35.

19.12.4.4 Variography

For the full indicator estimates, nine cut-offs were selected which divided the drill hole 2 m composites data into class intervals that became smaller with higher grades. The lowest grade class represented 15.8% of the data and the highest class represented 3.4% of the data.

Table 19-39 details the cut-offs used for the grade classes are listed below, along with class statistics.

Table 19-39: Botlu – Grade class cut offs and statistics

Class	From % Cu	To % Cu	Average % Cu	Median % Cu	% of Data
1	0.00	0.23	0.13	0.14	15.8
2	0.23	0.35	0.30	0.30	11.3
3	0.35	0.48	0.42	0.42	9.8
4	0.48	0.62	0.55	0.55	10.6
5	0.62	0.78	0.70	0.70	10.8
6	0.78	0.99	0.88	0.86	10.2
7	0.99	1.32	1.16	1.14	10.8
8	1.32	2.02	1.62	1.57	10.3
9	2.02	3.45	2.55	2.42	7.0
10	3.45	19.3	6.62	5.25	3.4

This table shows that for all classes except the highest, the class average is almost the same as the class median. This means that the average for these classes is robust, with no outliers. For the highest class, however, there is a significant difference between the average and the median, a consequence of the presence of a few exceptionally high-grade composites. Median values were used for the two highest classes in this estimate.

A series of global and directional variograms established that there was a preferred directional continuity (maximum range) at 160°, with a minimum at 70°, for the median indicator variogram. This was supported by normal and log variograms. The same preferred directional continuity was also present at higher-grade cut-offs. At lower cut-offs, the mineralisation became isotropic.

The data used for the variograms was derived from the oxide zone, including the south-east where there is some geological evidence for a north-north-east orientation of sulphide mineralisation beneath the thin oxide zone. However, exclusion of the data in this area had no effect on the variograms, probably because the zone is thin and the relative amount of data is small. Furthermore, variograms of the 5 m channel samples, which test only the near-surface oxide material, showed isotropic continuity supporting some gold remobilisation and homogenisation near surface.

For the purpose of block estimation, continuity of gold grade in 2 m downhole oxide composites was established using indicator variograms at the 9 cut-offs in Table 19-40.

Table 19-40: Botlu – Indicator variogram cut offs and models

Cut-off g/t Au	Co	C1	R1(DH) m	R1(160) m	R1(70) m	C2	R2(DH) m	R2(160) m	R2(70) m
0.23	0.32	0.18	9.0	11.0	11.0	0.50	70	140	140
0.35	0.32	0.26	10.0	11.0	11.0	0.42	70	100	100
0.48	0.32	0.30	10.0	14.0	10.0	0.38	55	80	80
0.62	0.32	0.32	12.0	14.0	8.0	0.36	35	70	35
0.78	0.32	0.32	10.0	15.0	7.0	0.36	33	50	23
0.99	0.32	0.34	8.0	14.0	6.0	0.34	30	47	20
1.32	0.32	0.35	7.5	12.0	6.0	0.33	20	45	20
2.02	0.32	0.35	5.0	10.0	6.0	0.33	17	44	20
3.45	0.32	0.35	4.5	8.0	6.0	0.33	15	40	20

Key:
Co – Nugget variance; C – Variance of spherical model structure;
R(160), R(70) – Horizontal ranges; R(DH) – Downhole range

19.12.4.5 Block Model Estimation

The block estimation was done in Simberi grid coordinates using a Surpac block model. The model extent was 8,170 m to 8,700 m east, 7,940 m to 8,510 m north, and 1,050 m to 1,250 m RL. Block size was 10 m by 10 by 5 m, sub-blocked to 5 m by 5 m by 2.5 m against topographic and base-of-oxide surfaces. This block size is consistent with the drill hole density, and the type of mineralisation being modelled.

2 m downhole oxide drill composite data, and 5 m surface channel samples, were used.

Bulk density was estimated for each block on the basis of depth below surface, using diamond drill core data.

19.12.4.6 Classification

Classification into Measured, Indicated and Inferred Resources was done using proximity to source data, as Indicated by the average number of samples inside the search ellipsoid during a separate block estimation run for this purpose. Solid models enclosing a range of average sample numbers were made, and visually compared to drill density. Two models were selected to contain Measured and Indicated Resources, and edited to delete isolated and non-coherent segments. The average number of samples used for Measured Resources was 45, and 23 for Indicated Resources. Blocks estimated from fewer than three composites were not included in the model. As expected, Measured blocks were located in well-drilled, coherent and central portions of the deposit, and Inferred blocks were located around its margins.

19.12.4.7 Block Estimate Results

At a cut-off of 0.5 g/t gold the model contains 1.67 million tonnes at 1.16 g/t gold, with an average bulk density of 1.59.

19.12.4.8 Block Estimate Validation

Average grades of the 2 m composite and 5 m channel sample source data, and of blocks in the oxide block model, were compared in 10 m flitches. Figure 19-16 shows the results graphically. The average grades of the data are those remaining after a 10 g/t Au top cut (which removes a few outlier values). The grades of the blocks (10 m by 10 m by 5 m) result from indicator kriging for which the grade of the top class was reduced from the average 6.62 g/t to the median 5.25 g/t. Taking these factors into account, the agreement between the source data and the block grades is quite good for that part of the deposit with good data density. In areas of less data, the block model effectively averages the composite data. Note that there is a double peak in all the data on the graph. The RL range 1,120 m to 1,190 m relates to deeper oxide

resource, while the range 1,190 m to 1,230 m corresponds to the thinner oxides in the south-east of the prospect.

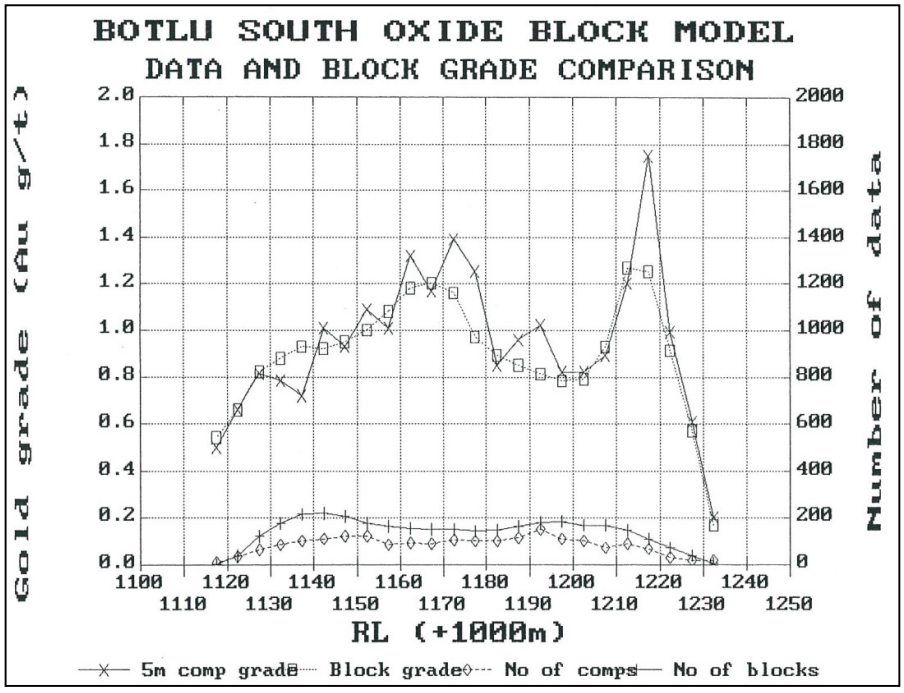

Figure 19-16: Botlu – Block model historical validation

19.12.5 Botlu Sulphide Resource Estimation

Resource tonnage inside nominal 1.0 g/t Au envelope. Average grade of 10 ×10 × 5 m blocks estimated by ID from data inside envelope, cut at 98th percentile.

A block grade model, inside a sulphide envelope interpreted from cross-sections, was estimated by inverse distance squared (ID) interpolation. Search parameters were 35 m isotropic. The block size used was 10 m North by 10 m East by 5 m RL, sub-blocked to 5 by 5 by 2.5 m against topographic, oxidation and grade boundaries. 2 m downhole drill composite data were used.

Bulk density was taken to be 1.9 t/m^3 (constant).

The grade data are approximately log normally distributed. Two data sets were used, cut to the 99th and 98th percentiles.

Table 19-41 details the 2 m Sulphide composite statistics for the 671 composite samples.

Table 19-41: Botlu – 2-m Sulphide composite statistics.

Topcut	Mean	Median	Min	Max	SD	CV
No cut	2.22	1.18	0	115.5	5.64	2.54
Topcut to 22g/t	2	1.18	0	22	2.7	1.35
Topcut to 9g/t	1.86	1.18	0	9	1.89	1.02

Table 19-42: Botlu – Sulphide Block model statistics

Grade range	Tonnes	Grade (uncut)	Grade (cut 22)	Grade (cut 9)	BD
0.0-1.0 g/t	307,206	0.71	0.71	0.71	1.9
>1.0 g/t	1,203,769	2.77	2.29	2.02	1.9
TOTAL	1,510,975	2.35	1.97	1.76	1.9

19.12.5.1 Authors Validation

The author supervised the validation of the historical Botlu model. A statistical and visual assessment of the block model were undertaken to assess the successful application of the estimation and to ensure that as far as the data allowed, all blocks within ore domains were estimated and the model estimates were considered acceptable.

The model was imported into the mining software package Vulcan and examined against the raw drilling data where an acceptable match was observed. Statistically, the mean grade of the data was preserved in the oxide and sulphide models. Figure 19-17and Figure 19-18 show swath plots across the deposit illustrating a good correlation between the composite data and the block models across the deposit.

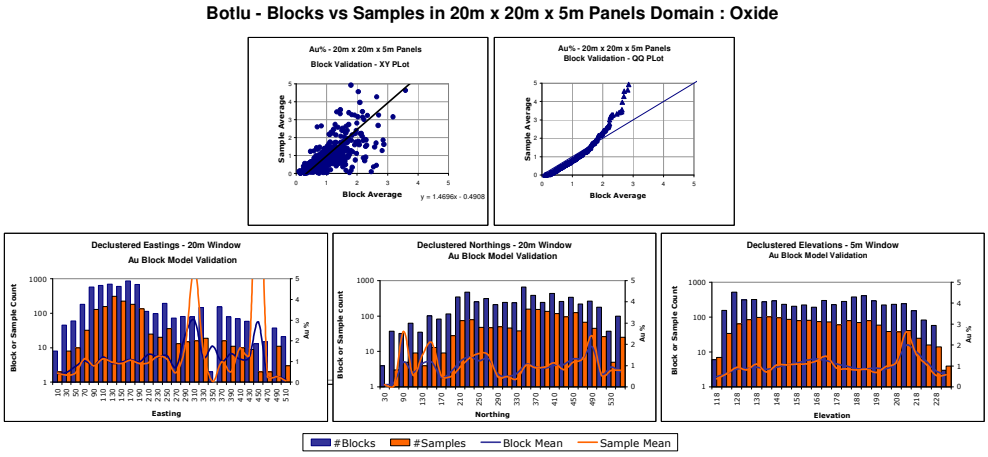

Figure 19-17: Botlu – Oxide – Authors validation

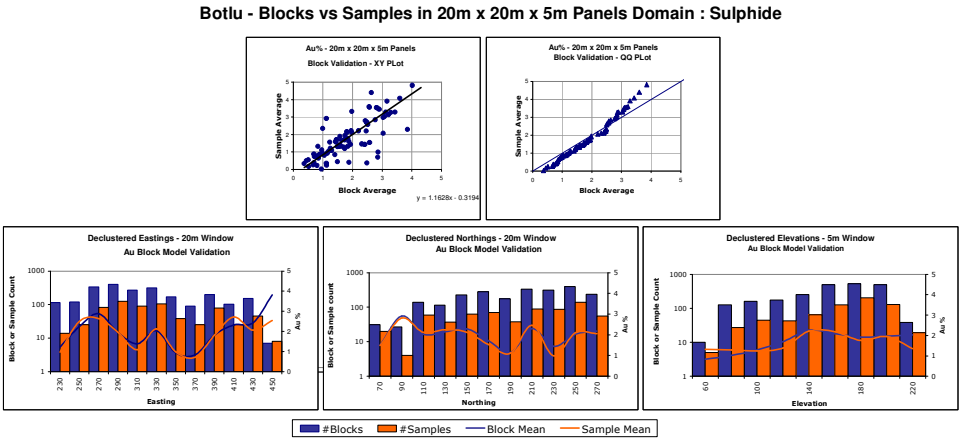

Figure 19-18: Botlu – Sulphide – Authors validation

19.12.6 Samat Resource Estimation

19.12.7 Geological Interpretation

Pre-2010

Previously at Samat definitive controls on mineralisation were not identified. The pattern of the thin layers of mineralisation is suggestive of mineralisation sources along discrete structural zones, with spreading by supergene processes and soil creep at surface, modified by local erosion.

At Samat South, volcanoclastics and diatremes are closely associated with the mineralisation. An interpreted 085° trending fault to the south marks a boundary between hydrothermally altered and unaltered rocks. Within the mineralised zone itself, structures are not sufficiently well defined to provide limits that can be used in resource estimation.

At Samat North, differences between several twinned drill holes indicate structural complexity. The northern boundary of the mineralisation appears to lie along a NW-SE trending fault. A NE-SW trending fracture zone may pass through the mineralised zone and intersect the northern end of the Samat East structure.

At Samat East, a thin layer of oxide mineralisation extends in a NNE direction, on the western slopes of a prominent ridge.

2010

Allied supplied Golder with a geological interpretation consisting of wireframe surfaces defining the weathering profiles and the structural controls on the mineralisation at Samat (Figure 19-19). Golder used the structural wireframe surfaces to construct wireframe solids defining the geological domains (Figure 19-20), which were used to flag the drill hole database and construct the geological block model. Allied also supplied Golder with the latest pit surveys.

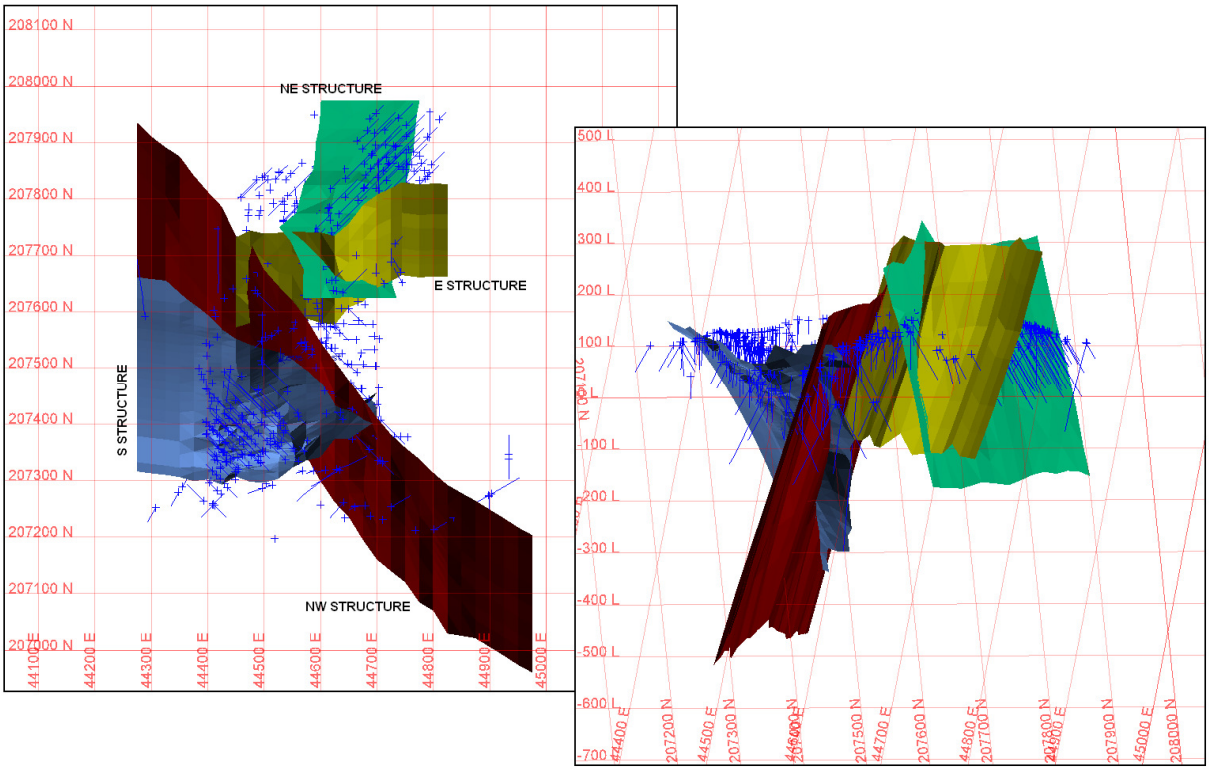

Figure 19-19: Samat structural wireframe surfaces

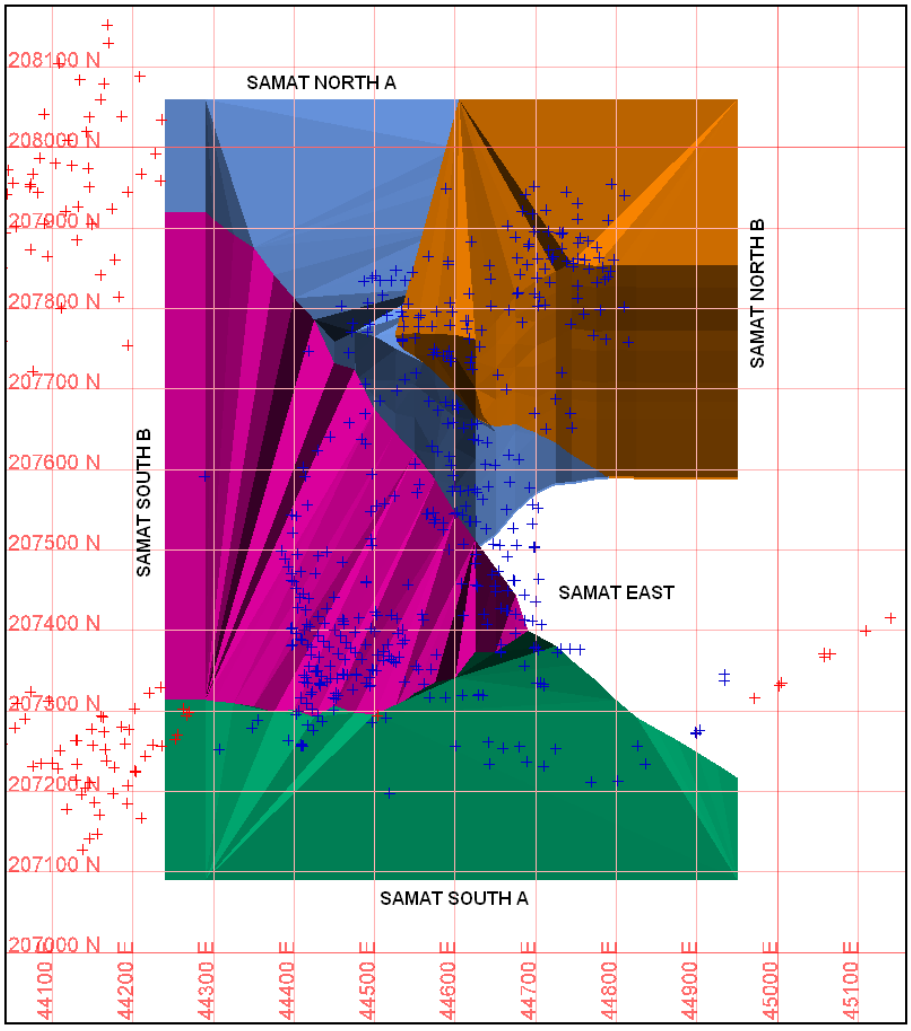

Figure 19-20: Samat geological domains, completed by Golder

19.13 Geological Block Model

A geological block model for Samat (*samat1007.bmf*) was created in Vulcan software.

19.13.1 Block Model Parameters

The parent block size of 10 by 10 by 10 m (X, Y and Z) reflects the average drill spacing in the deposit, which is predominantly 10 to 20 m between and on section. The minimum sub-blocking was defined as 5 by 5 by 5 m. The summary of block model parameters is provided in Table 19-43. The block model origins and extensions are shown in Figure 19-21.

Table 19-43: Block model origin and extensions

Parameter	Easting	Northing	RL
Parent Block Size (m)	10	10	10
Sub Cell Block Size (m)	5	5	5
Origin Minimum (m)	44250	207100	-250
Origin Maximum (m)	44940	208050	300
Extension (m)	690	950	550
Number of Cells:	69	95	55

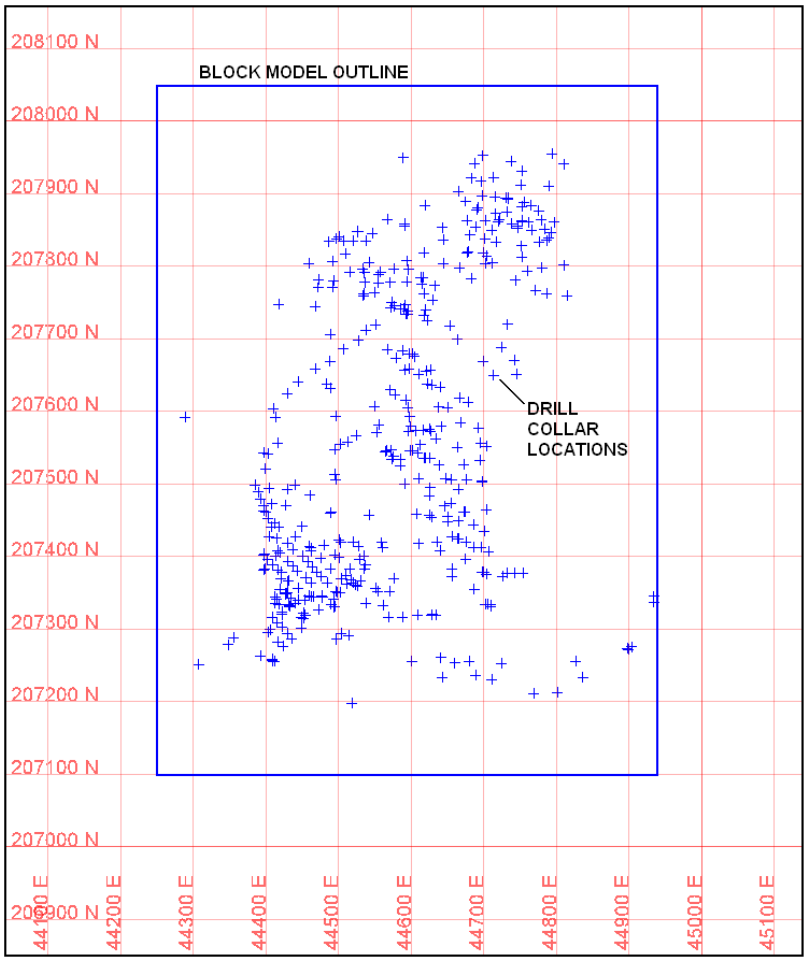

Figure 19-21: Plan view showing the Samat model area (blue polygon) as well as the drill hole collar locations

19.13.2 Block Model Variables and Coding

Variables with default values were assigned to each block. Table 19-44 summarises the variables created for the geological block model. Each block geological variable was then coded by the wireframe surfaces and solids and assigned a value according to the block's position relative to the wireframe (Table 19-45). In the case of multiple wireframes, the sequential order in which the flagging occurred dictated the final block variable value.

Table 19-44: Block model variables

Variable Name	Default	Type	Description
OX	-99	Integer	Weathering Codes: 1 = oxide 2 = transition 3 = sulphide
ESTDOM	-99	Integer	Estimation Domains: 1000s = Samat S A 2000s = Samat S B 3000s = Samat N A 4000s = Samat N B 5000s = Samat E
MINED	0	Integer	Mined Code: 0 = *in situ* 1 = mined
DOMAIN	5000	Integer	Geological Domains: 1000 = Samat S A 2000 = Samat S B 3000 = Samat N A 4000 = Samat N B 5000 = Samat E

Table 19-45: Wireframes, flagging values and priorities applied to the geological block model

Variable	Priority	Projection	Projection Direction	Value	Wireframe	Wireframe Description
OX	1	none	Z	1	10_sim_topo.00t	Topographic surface
OX	2	none	Z	2	10_samat_base_oxide.00t	Base of complete oxidation surface
OX	3	none	Z	3	10_samat_top_sulphide.00t	Top of sulphide (base of transition) surface
OX	4	partial	Z	-99	10_sim_topo.00t	Topographic surface
MINED	1	partial	Z	1	10_samat_north_east_pits.00t	Combined North A Pit and East Pit
MINED	2	partial	Z	1	10_samat_south_pit.00t	South Pit
MINED	3	partial	Z	1	10_samat_northb_pit.00t	North B Pit
MINED	4	partial	Z	-99	10_sim_topo.00t	Topographic surface
DOMAIN	1	none	Z	1000	10_samat_south_A.00t	Samat South A solid
DOMAIN	2	none	Z	2000	10_samat_south_B.00t	Samat South B solid
DOMAIN	3	none	Z	3000	10_samat_north_A.00t	Samat North A solid
DOMAIN	4	none	Z	4000	10_samat_north_B.00t	Samat North B solid
DOMAIN	5	partial	Z	-99	10_sim_topo.00t	Topographic surface

Estimation domains (ESTDOM) were calculated based on combinations of geological domain codes and oxidation codes. These are summarised in Table 19-46.

Table 19-46: Samat Estimation Domains

ESTDOM	Description
1001	Samat South A Oxidised
1002	Samat South A transition
1003	Samat South A sulphide
2001	Samat South B Oxidised
2002	Samat South B transition
2003	Samat South B sulphide
3001	Samat North A Oxidised
3002	Samat North A transition
3003	Samat North A sulphide
4001	Samat North B Oxidised
4002	Samat North B transition
4003	Samat North B sulphide
5001	Samat East Oxidised
5002	Samat East transition
5003	Samat East sulphide

19.14 Exploratory Data Analysis

19.14.1 Data Preparation and Compositing

Raw samples in the validated Vulcan database *sim16072010.tig.isis* were flagged using the geological interpretation for DOMAIN and OX based on the centroid position of the raw sampling interval. The flagging of raw data was carried out using a process identical to that used to assign the variables to the geological block model. Therefore, the variable names, variable values and definitions and the wireframes names used are all identical. ESTDOM codes were calculated the same as for the block model.

The nominal raw sampling interval in the Samat database is 1 m (around 90% of the Samat drill holes). A relatively small proportion (less than 10%) of intervals in the database range from 0.06 to 19.8 m in length. These predominantly reflect waste or low-grade regions of the deposit.

Data was composited to 5 m lengths. The 5 m length was selected to reflect minimum sub-blocking height of 5 m. Compositing reduces the impact of high outlier values by reducing spikes 9 the variance) in the data.

The compositing was broken on changes in the DOMAIN code, retaining majority codes for OX and ESTDOM.

Gaps in the downhole sequence were treated as missing during compositing with no default values assigned to represent the missing sample intervals. Assay intervals that were not sampled for Au were excluded from compositing. The standard filtering routine (in Vulcan) was used to exclude from composite creation any null values representing non-sampled intervals for other variables. This routine excludes sample values >-9999 ≤0.001.

19.14.2 Univariate Statistics

A summary of the global statistics broken down by DOMAIN is presented in Table 19-47. Statistics broken down by ESTDOM is summarised in Table 19-48. All statistics are weighted by length.

Table 19-47: Statistics by DOMAIN

DOMAIN	No.Obs.	Minimum	Maximum	Mean	Variance	Coeff.Var	Median
Au							
1000	718	0.01	21.48	0.27	0.97	3.68	0.05
2000	1877	0.01	67.64	1.73	15.01	2.24	0.43
3000	633	0.01	21.75	0.43	1.35	2.72	0.19
4000	1352	0.01	123.25	1.46	30.75	3.81	0.33
5000	1124	0.01	23.99	0.52	1.30	2.18	0.20
Global	***5704***	***0.01***	***123.25***	***1.11***	***13.22***	***3.29***	***0.23***
Ag							
1000	604	0.01	35.20	0.72	2.42	2.15	1.00
2000	1600	0.05	65.00	1.80	15.00	2.15	1.00
3000	538	0.05	90.00	1.02	15.76	3.88	1.00
4000	1184	0.10	146.00	2.52	89.46	3.75	1.00
5000	994	0.02	230.00	2.25	91.74	4.26	1.00
Global	***4920***	***0.01***	***230.00***	***1.85***	***47.35***	***3.72***	***1.00***
S							
1000	211	0.01	8.22	2.87	4.69	0.75	2.69
2000	197	0.01	9.54	2.83	4.85	0.78	3.09
3000	92	0.01	6.48	1.28	3.09	1.37	0.33
4000	181	0.01	7.19	2.64	2.76	0.63	2.54
5000	253	0.01	9.72	2.35	5.30	0.98	1.93
Global	***934***	***0.01***	***9.72***	***2.51***	***4.55***	***0.85***	***2.39***

Table 19-48: Statistics by ESTDOM

ESTDOM	No.Obs.	Minimum	Maximum	Mean	Variance	Coeff.Var	Median
Au							
1001	75	0.01	15.89	0.94	4.53	2.28	0.21
1002	25	0.01	1.48	0.27	0.18	1.58	0.05
1003	618	0.01	21.48	0.19	0.50	3.80	0.05
2001	293	0.01	67.64	3.59	42.61	1.82	1.41
2002	115	0.01	27.05	2.21	18.16	1.93	0.45
2003	1469	0.01	54.32	1.31	8.19	2.18	0.35
3001	179	0.01	21.75	0.69	2.96	2.48	0.31
3002	38	0.01	14.49	0.81	5.28	2.84	0.28
3003	416	0.01	6.03	0.27	0.19	1.58	0.16
4001	241	0.02	21.08	2.07	11.79	1.66	0.52
4002	79	0.02	123.25	5.86	334.44	3.12	0.47
4003	1032	0.01	54.66	0.96	9.28	3.17	0.29
5001	434	0.01	23.99	0.68	2.53	2.34	0.23

ESTDOM	No.Obs.	Minimum	Maximum	Mean	Variance	Coeff.Var	Median
5002	94	0.01	3.10	0.31	0.26	1.64	0.07
5003	596	0.01	6.83	0.44	0.48	1.59	0.19
Global	***5704***	***0.01***	***123.25***	***1.11***	***13.22***	***3.29***	***0.23***
Ag							
1001	66	0.05	35.20	1.40	19.98	3.19	1.00
1002	23	0.01	1.00	0.48	0.16	0.83	0.20
1003	515	0.05	4.20	0.65	0.21	0.71	1.00
2001	252	0.05	16.04	1.24	3.70	1.56	1.00
2002	95	0.05	14.00	1.27	3.55	1.49	1.00
2003	1253	0.05	65.00	1.96	18.09	2.17	1.00
3001	160	0.05	5.37	0.64	0.54	1.16	0.33
3002	32	0.12	3.56	1.00	0.61	0.78	1.00
3003	346	0.05	90.00	1.21	24.35	4.08	1.00
4001	208	0.10	31.00	1.79	11.70	1.91	1.00
4002	71	0.10	145.00	8.72	689.35	3.01	1.00
4003	905	0.10	146.00	2.20	56.16	3.42	1.00
5001	404	0.02	90.00	1.38	27.41	3.79	0.50
5002	86	0.08	70.00	3.03	96.11	3.24	1.00
5003	504	0.04	230.00	2.84	144.06	4.23	1.00
Global	***4920***	***0.01***	***230.00***	***1.85***	***47.35***	***3.72***	***1.00***
S							
1001	31	0.03	3.22	0.63	0.76	1.39	0.20
1002	15	0.01	3.89	1.97	1.31	0.58	1.63
1003	165	0.06	8.22	3.34	4.45	0.63	3.31
2001	44	0.01	4.50	0.57	0.76	1.53	0.20
2002	13	0.01	2.40	0.46	0.46	1.47	0.08
2003	140	0.01	9.54	3.81	3.43	0.49	3.89
3001	50	0.01	4.70	0.38	0.50	1.86	0.21
3002	7	0.01	6.48	1.14	4.80	1.92	0.20
3003	35	0.19	6.04	2.67	3.47	0.70	2.50
4001	41	0.01	4.04	0.89	0.96	1.10	0.42
4002	15	0.31	3.78	1.85	0.88	0.51	1.92
4003	125	0.38	7.19	3.33	2.02	0.43	3.12
5001	71	0.01	8.68	0.30	1.20	3.65	0.05
5002	34	0.01	7.41	0.76	2.17	1.93	0.30
5003	148	0.25	9.72	3.82	3.23	0.47	3.86
Global	***934***	***0.01***	***9.72***	***2.51***	***4.55***	***0.85***	***2.39***

Golder also implemented data declustering on the drill holes. The declustered data was mainly used in validations of the grade estimation process against the drill hole data, to achieve more representative comparisons with the grade estimates.

The declustering weights were calculated via a cell declustering technique which uses the moving window method. The window size was set to 40 by 40 by 5 m orientated at an azimuth of 340° with a -90° dip, to match the overall mineralisation trend.

19.14.3 Grade Distribution Analysis

The distributions of Au, Ag and S across the DOMAIN and ESTDOM domains were analysed using cumulative log-probability plots. This analysis was carried out to assess distribution behaviour across the domains and also to identify outliers and thresholds for potential spatial restraining of high-grades during estimation.

Figure 19-22 displays cumulative log-probability plots of the Au distribution within each geological domain (DOMAIN) as overlays, for all weathering profiles combined. The distributions indicate clear differences in the grade distributions between the domains.

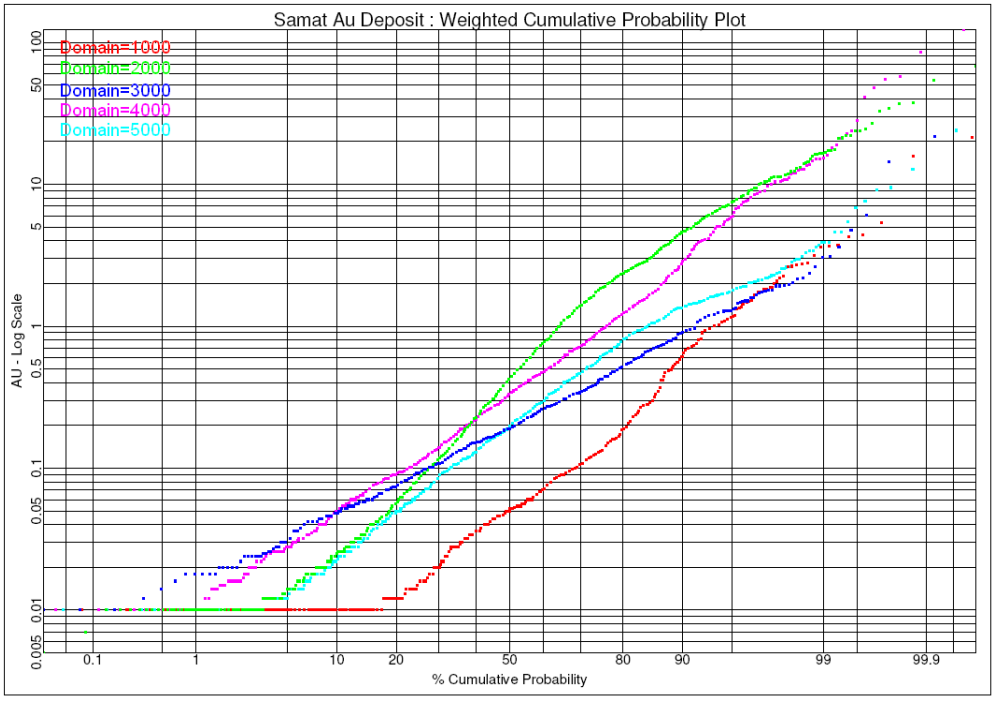

Figure 19-22: Log-probability plot of the Au distribution by DOMAIN

Grade distributions by weathering profiles were also compared to assess for any global differences between *in situ* oxide, transition and sulphide material. Figure 19-23 to Figure 19-27 show examples of log-probability plots for Au in each geological domain with the *in situ* oxide, transition and sulphide distributions compared as overlays. The distributions also highlight clear differences in grade between the weathering profiles in each geological domain.

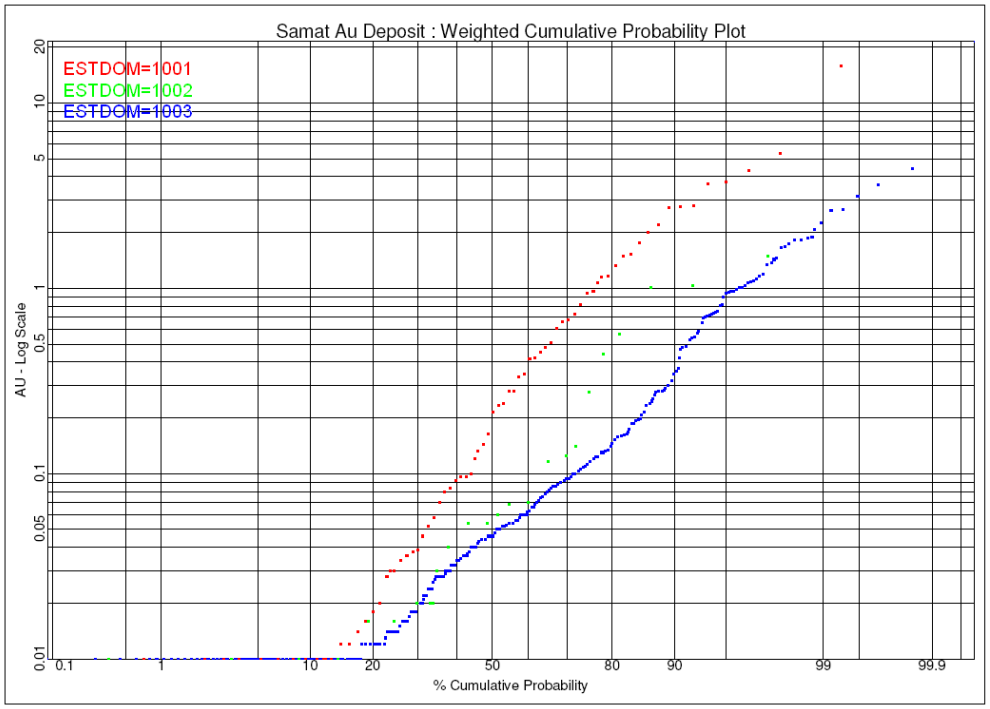

Figure 19-23: Cumulative probability plots for Samat South A by ESTDOM

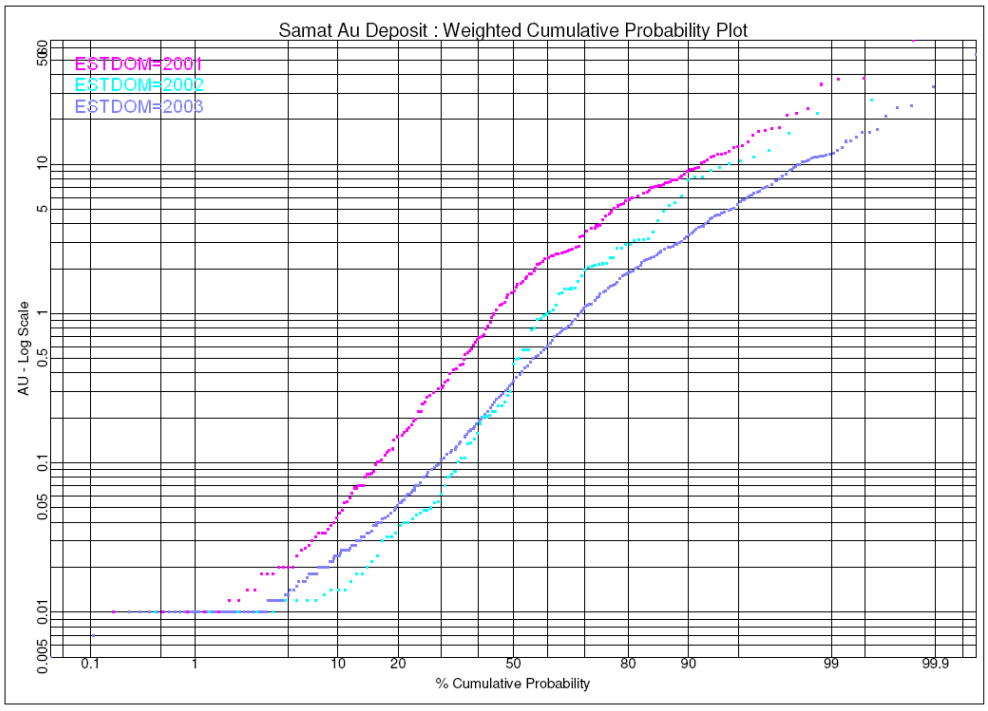

Figure 19-24: Cumulative probability plots for Samat South B by ESTDOM

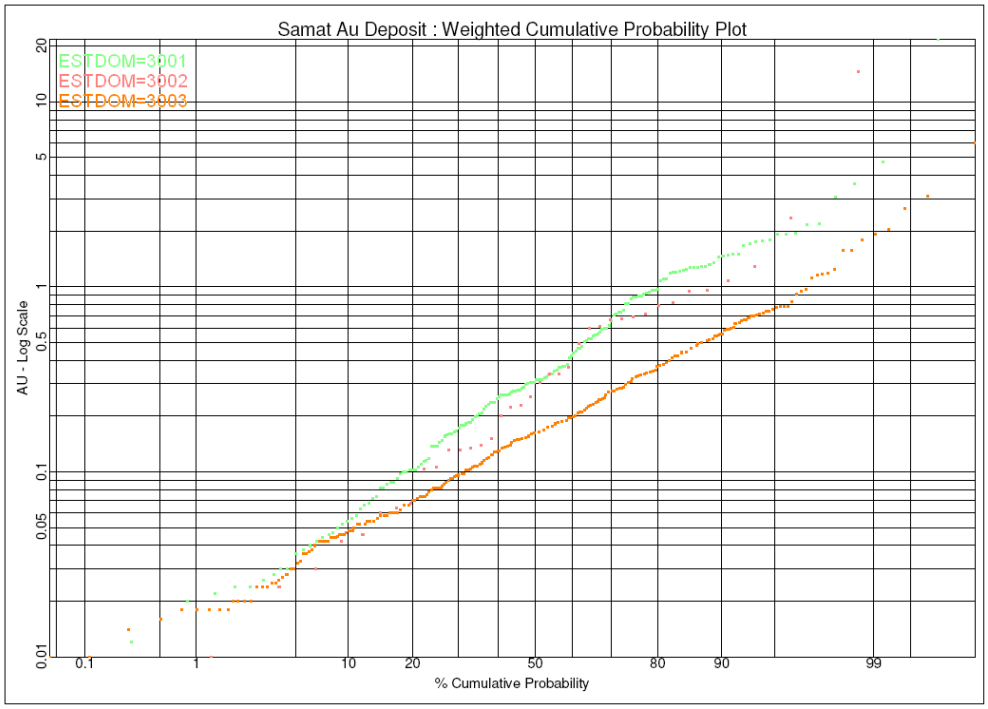

Figure 19-25: Cumulative probability plots for Samat North A by ESTDOM

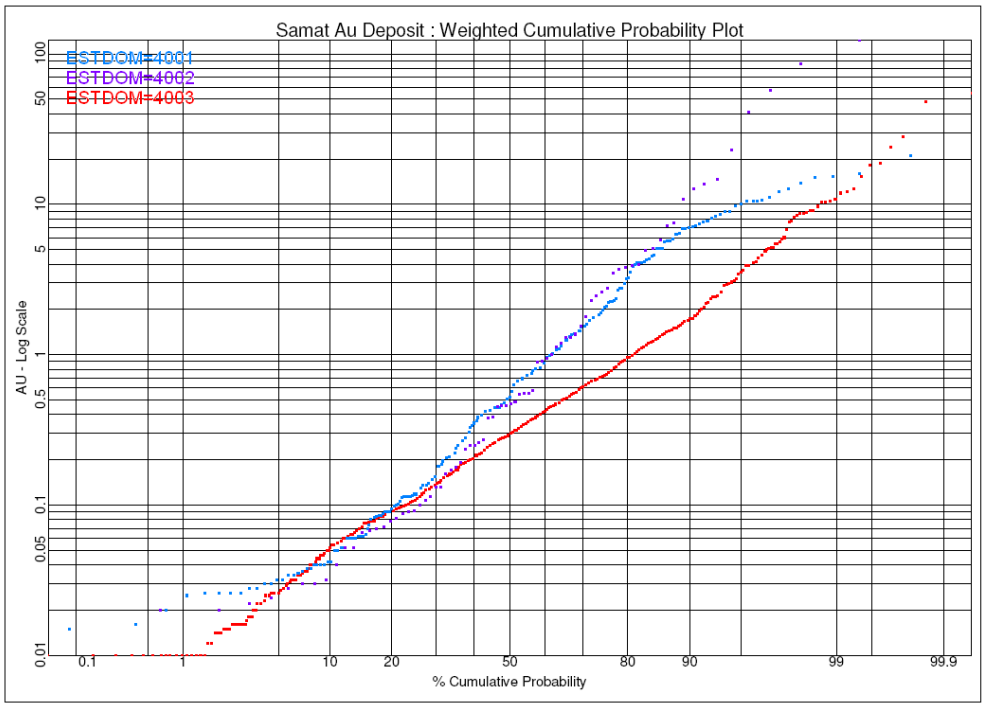

Figure 19-26: Cumulative probability plots for Samat North B by ESTDOM

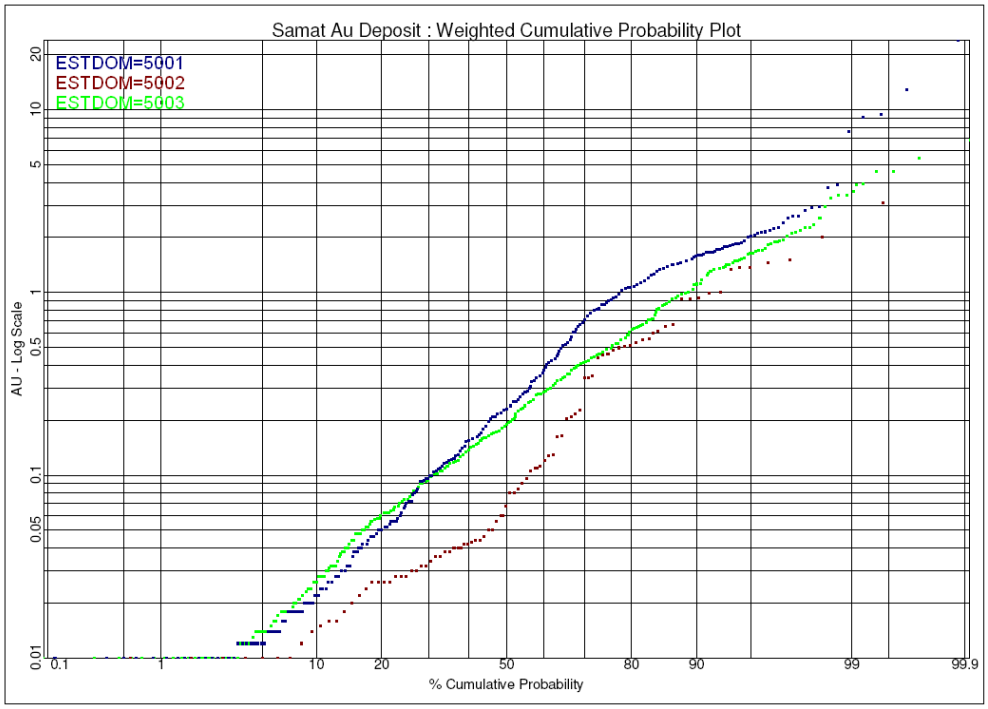

Figure 19-27: Cumulative probability plots for Samat East by ESTDOM

This process was repeated for Ag and S. Ag has similar differences in grade distributions across the domains as Au, however, the probability plots for S in sulphide material (Figure 19-28) shows that the grade distributions in all geological domain are similar.

Based on the limited number of samples of S in transition material and on the comparison of S in sulphide material (above), the decision was made to combined S for all transition and sulphide material irrespective of geological domain or weathering profile.

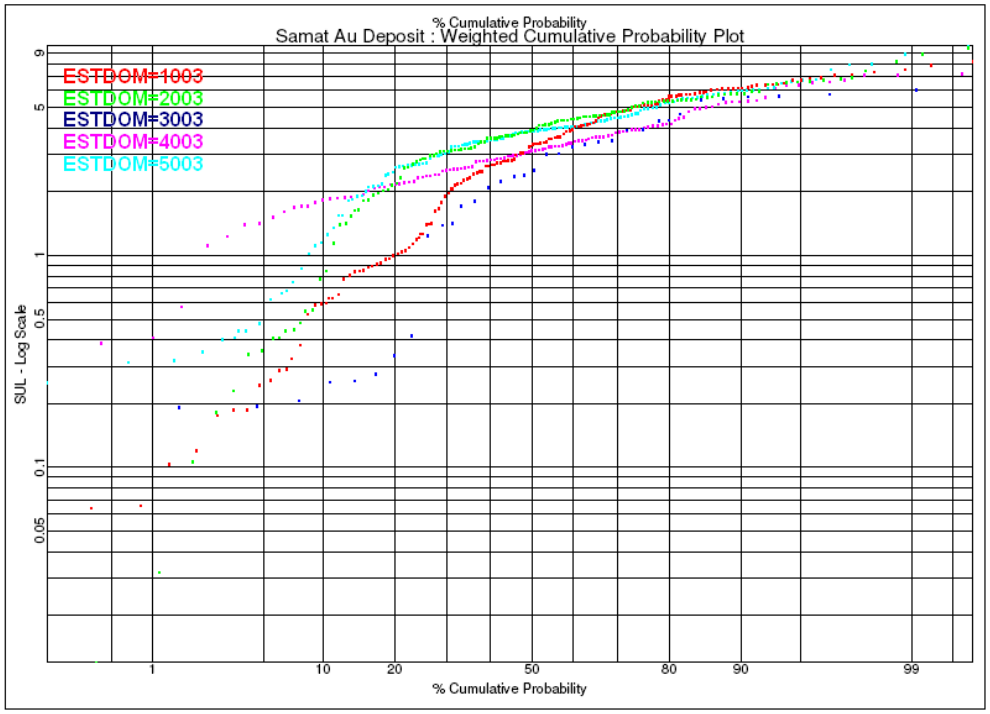

Figure 19-28: Cumulative probability plots for S in sulphide material by DOMAIN (ESTDOMs ending in 3)

19.15 Variographic Analysis

19.15.1 Variography Objectives and Approach

Variography was undertaken to model the spatial variability of the Au, Ag and S mineralisation. The objectives of the variography were to:

- establish the directions of major grade continuity for each domain
- quantify spatial continuity (variability, anisotropy and overall continuity), and
- provide variogram model parameters for use in geostatistical grade interpolation.

19.15.2 Variography Domains

Variography focused on the ESTDOM combinations are summarised in Table 19-49. Each variography domain was treated as a global unit, to derive global variogram models with good short and long-scale variogram definition.

Table 19-49: Variography domains

ESTDOM Combination	Description
1002/1003 (Au and Ag)	Samat South A transition and sulphide material
2002/2003 (Au and Ag)	Samat South B transition and sulphide material
3002/3003 (Au and Ag)	Samat North A transition and sulphide material
4002/4003 (Au and Ag)	Samat North B transition and sulphide material
5002/5003 (Au and Ag)	Samat East transition and sulphide material
OX = 1 (Au, Ag and S)	All oxide material
OX = 2/3 (S)	All transition and sulphide material for S

19.15.3 Summary of Variogram Models

Table 19-50 shows the orientations of mineralisation continuity that were identified and modelled.

The "separation" is the angle between the major and semi-major vectors. The "rotation" applies to the semi-major axis; it is the angle required to rotate the search ellipsoid around the major axis vector to align the ellipsoid with the interpreted resultant plane for grade interpolation.

Table 19-50: Summary of Variogram Continuity Orientations

Domain	Major Axis	Semi-Major Axis	Resultant Plane	Major & Semi-Major	Rotation Angles
	Plunge/Direction	Plunge/Direction	Dip/Dip Direction	Separation Angle	R1/R2/R3
1002/1003 (Au and Ag)	0 → 120	70 → 030	30 → 060	90	0/120/30
2002/2003 (Au and Ag)	0 → 100	60 → 010	10 → 060	90	0/100/10
3002/3003 (Au and Ag)	0 → 135	70 → 220	20 → 070	90	0/130/20
4002/4003 (Au and Ag)	0 → 080	20 → 170	70 → 020	90	0/080/70
5002/5003 (Au and Ag)	0 → 130	70 → 220	20 → 070	90	0/130/20
OX = 1 (Au, Ag and S)	0 → 150	0 → 240	0 → 000	90	0/150/0
OX = 2/3 (S)	0 → 160	0 → 250	0 → 000	90	0/160/0

KEY: R1/R2/R3 Search orientations (conventional left hand rule)
R1 = azimuth rotation clockwise from north
R2 = plunge along R1 direction (+ve = up, -ve = down)
R3 = dip rotation around R1-R2 axis (+ve = anticlockwise, -ve = clockwise)

As an example, Figure 19-29 illustrates the Au variogram models for Samat North A transition and sulphide material (ESTDOMs 3002/3003).

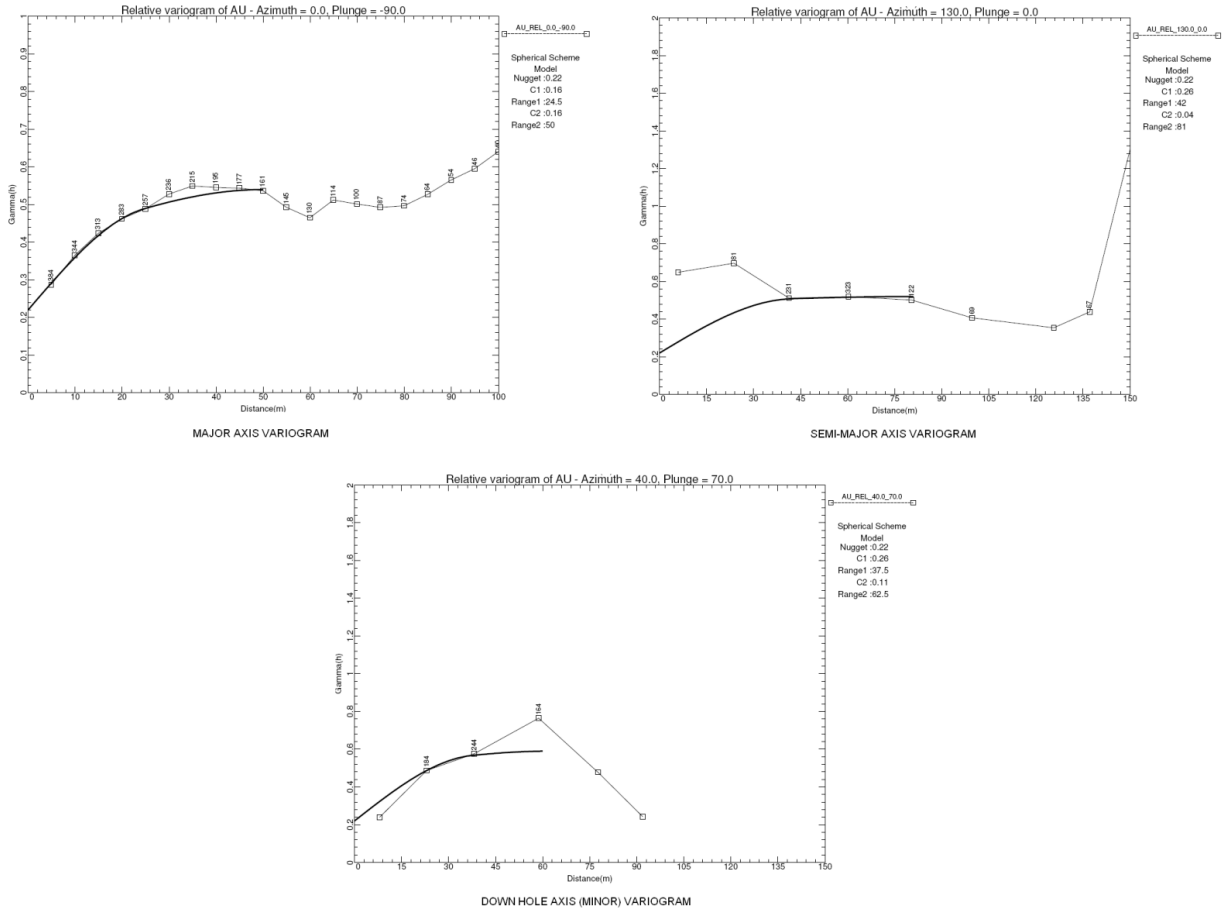

Figure 19-29: Variogram models for Samat North A transition and sulphide material (ESTDOMs 3002/3003)

19.15.4 Grade Interpolation Strategy

Grade estimation of the Samat deposit was carried out using the geostatistical method of Ordinary Kriging (OK) using Vulcan. All variables were interpolated by OK. The OK method used estimation parameters defined by the variography. The estimation was performed by oxide and transitional/sulphide zone according to the respective wireframes. Au, Ag and S grades were estimated by OK within all domains.

The geological block model used a parent block size of 10 by 10 by 10 m for blocking purposes with sub-blocks set as 5 by 5 by 5 m. For the Samat drilling density the parent block size is reasonable to ensure local estimation quality and reduce conditional bias (Figure 19-21).

Un-estimated blocks were assigned default grade values. These default values were the mean domain grade from the estimated blocks. Default values pass was identified as pass 4.

The interpolated Vulcan block model was named *samat1007_OK.bmf.*

19.15.5 Kriging Plan

The OK estimation was run in a three-pass kriging plan, the latter passes using progressively larger search neighbourhoods to enable the estimation of blocks un-estimated on the first pass.

Further explanation and details are as follows:

- Sample weighting determined by variogram model parameters (kriging approach) for all variables.
- Block discretisation was set to 5 (X) by 5 (Y) by 2 (Z) to estimate block grades of 10 m by 10 m by 10 m parent blocks, using nominal 5 m composites. Sub-cells of 5 m by 5 m by 5 m received the parent cell estimate.
- A three-pass kriging plan was used; Pass 1 search of -40 by 40 by 10 m; Pass 2 search of 8 by 80 by 20 m and a third of 120 by 12 by 20 m. Search dimensions were selected based on drilling density and configuration. The Z direction was limited to restrict grade from estimating far beyond the last sample.
- A minimum of two composites using an octant based search where a maximum of 2 samples per octant were allowed equating to a maximum of 16 composites per estimate. Pass 2 used a minimum of four and two composites and pass 3 one and one respectively.
- Any remaining un-estimated blocks were assigned the average value from the kriged estimates and an estimation pass value of 4.
- Spatial restraining was used to limit the influence of outlier high grades for all estimations. A 5.1 by 5.1 by 5.1 m search was used for the restraint effectively limiting the outlier high grades to one block in the model. Table 19-53 details the high grades restraint values used.

Table 19-51, Table 19-52 and Table 19-53 detail the kriging parameters used in the Samat estimation.

Table 19-51: Search radius and ellipsoid orientation by pass according to estimation domain

Domain	Orientation	Ellipsoid radii (m)								
	Azimuth/Plunge Dip (R1/R2/R3)	pass1			pass2			pass3		
		X	Y	Z	X	Y	Z	X	Y	Z
1002/1003 (Au and Ag)	0/120/30	40	40	10	80	80	20	120	120	20
2002/2003 (Au and Ag)	0/100/10	40	40	10	80	80	20	120	120	20
3002/3003 (Au and Ag)	0/130/20	40	40	10	80	80	20	120	120	20
4002/4003 (Au and Ag)	0/080/70	40	40	10	80	80	20	120	120	20
5002/5003 (Au and Ag)	0/130/20	40	40	10	80	80	20	120	120	20
oxide (Au, Ag and S)	0/150/0	40	40	10	80	80	20	120	120	20
S (transition/sulphide)	0/160/0	40	40	10	80	80	20	120	120	20

KEY: R1/R2/R3 Search orientations (conventional left hand rule)
R1 = azimuth rotation clockwise from north
R2 = plunge along R1 direction (+ve = up, -ve = down)
R3 = dip rotation around R1-R2 axis (+ve = anticlockwise, -ve = clockwise)

Table 19-52: Samat Kriging Plan Parameters

Parameter	Values
Discretisation (X/Y/Z)	5/5/2
Search type	Octant search
Minimum No. samples (Pass 1/2/3)	4/4/1
Maximum No. samples	32 (4 per octant)

Table 19-53: Samat high grade restraining thresholds by estimation domain

Domain	Threshold Au	Threshold Ag	Threshold S
1002/1003 (Au and Ag)	999	999	-
2002/2003 (Au and Ag)	13	25	-
3002/3003 (Au and Ag)	999	999	-
4002/4003 (Au and Ag)	12	30	-
5002/5003 (Au and Ag)	999	20	-
oxide (Au, Ag and S)	18	17	999
S (transition/sulphide)	-	-	999

999 = no high grade threshold

19.16 Validation of Grade Estimates

An on-screen validation between samples and blocks was completed on the model. The estimated block model was also validated for conformance of the block average grades against the drill hole data using swath plots and reproduction of the declustered point support (5 m composite) data distribution in the kriged model

19.16.1 Visual Assessment of Grade Estimates

The on-screen validation process involved comparing block average grade estimates and composite grades in cross-section and in plan view for each modelled domain. The inspection showed a good correlation between the estimated blocks and the drill hole data.

19.16.2 Global Statistical Validation

The global estimated mean domain grade against 5 m composite data was checked as a means to validate reproduction of the mean grade of the data. This comparison is shown for all estimation domains in Table 19-54. Global statistical comparisons of the mean grade are influenced by varying drill hole density, data sharing between domains and high-grade treatment. These issues can sometimes distort the comparison. In particular, peripheral blocks representing extrapolated estimates can create apparent anomalies in the grade conformance. Differences observed in the global average grades may not be evident on swath comparisons, which is an advantage of using swath plots to assess conformance.

Overall, reproduction of the data average is acceptable to poor in places. Some apparent under and over-estimation of the domain average in the block model for some domains is likely to be an artefact of the issues described above. The swath validations do not show the same discrepancies and are probably a more representative indication of the average grade conformance.

Table 19-54: Au – Composite Grades vs. Block Model

Domain	5 m Composites			Block Model			Average	Variance
	No. Obs	Average	Variance	No. Obs	Average	Variance	% Difference	ADJ Ratio
1000	643	0.19	0.49	24489	0.21	0.14	109.6%	0.286
2000	1584	1.38	8.99	17102	0.50	0.56	36.4%	0.062
3000	454	0.32	0.65	14602	0.25	0.08	79.6%	0.119
4000	1111	1.32	34.81	13850	0.31	0.35	23.2%	0.010
5000	690	0.42	0.45	24090	0.26	0.07	61.3%	0.152

19.16.3 Swath Plot Validation

Swath plots are used to assess the block model estimates for global bias; the estimates should have a close relationship to the drill hole composite data used for estimation. The plots are useful for assessing average grade conformance, and also to detect any obvious interpolation issues. The relationship between model and sample panel averages was assessed in the form of scatter plot and Q-Q plot. This allows some assessment of the smoothing effect of the interpolation to be made. General swath plots were generated for lodes and supergene mineralisation.

The process involved averaging both the blocks and samples in panels of 20 m by 20 m by 20 m RL. Conformance of the model and sample average grades was assessed in the form of easting, northing and RL swaths of the panel averages.

Figure 19-30 illustrates the scatter and Q-Q plots as well as the swath plots along easting, northing and vertical direction for Au in the oxide material at Samat.

Generally good agreement was observed between the drill hole data and block model grades, indicating that in general the model has reproduced the local average grades of the drilling data with no obvious global bias. The block model grade appears lower/higher than the drill hole grade in places which is likely a function of the declustering effect of the kriging, and the high grade restraining used.

The Q-Q plot of block model and composite panel averages indicates the smoothing effect of the interpolation, which in this case seems to affect the high-grade end of the distribution where the block model understates the Au grades. This reflects the effect of outliers which were treated for estimation purposes. This smoothing effect is an expected behaviour of moving from sample size data to a bigger volume as represented by kriged models.

The scatter plot shows deviation from the 45° line, which is normal. Generally, the graphs show good conformance (see angular coefficient).

Overall, the swath plot validation process shows that the block model grade estimates show acceptable conformance with the drill hole data.

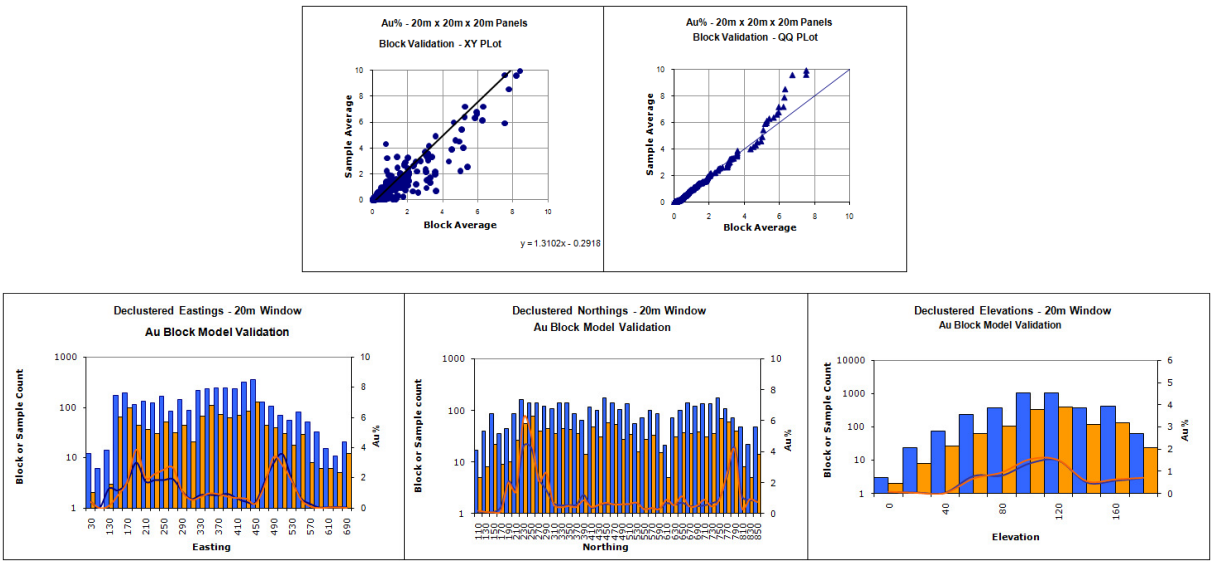

Figure 19-30: Swath Plot validation for oxide Au – Samat

19.17 Bulk Density

Dry Bulk Density was assigned using a spherical function of depth below surface developed by Allied Gold. The function was developed from empirical data in the form of core density measurements across the deposit. The function has a nugget of 1.3502 g/cc and a sill differential of 0.46 g/cc at a range 45 m below topographic surface and was applied to the oxide and transition domains only, i.e. at surface the bulk density is 1.3502 and at 45 m below surface bulk density is 1.81. This was calculated with the following formulae:

$$\gamma(h) = c_0 + c\left[\frac{3}{2}\frac{h}{a} - \frac{1}{2}\frac{h^3}{a^3}\right] \text{ for h} \leq \text{a}$$

and

$$\gamma(h) = c_0 + c \text{ for h} > \text{a}$$

Table 19-55 explains the variables used above.

Table 19-55: Details of Variables used in the spherical bulk density model

Variable	Name	Description	Value
c_o	Nugget	Minimum SG – at surface	1.3502
c	Sill differential	c + c_o = Maximum SG	0.46
a	Range	Depth at which maximum SG reached	45
h	Lag	Depth below surface	variable

For the sulphide domain a regression against depth was applied:

$$\text{Dry Bulk Density}_{(\text{sulphide})} = (0.0024 * \text{depth below surface}) + 1.81$$

The application of this regression at depth should be critically reviewed in future models as it will be overestimating density at depth.

19.18 Resource Classification

Resources were classified in compliance with the Australasian Code for the Reporting of Identified Mineral Resources and Ore Reserves (JORC, 2004).

The Au estimation has been classified as Indicated and Inferred based on data quality and density. The Indicated Resource has a nominal data density of 20 m by 10 m and the Inferred Resource 30-40 m by 20 m. Due to the under sampling of the historical drilling, Ag and S are classified as Inferred.

19.19 Resource Statement

The tables below summarise the depleted Mineral Resources by classification and material type at a 0.5 g/t Au cut-off grade (model *samat1007_OK.bmf*).

The model has been depleted to the current mining surfaces:

- 10_samat_east_pit.00t
- 10_samat_north_east_pits.00t
- 10_samat_northb_pit.00t
- 10_samat_south_pit.00t
- adroa_june2008.00t.

Depletion used the above mining surfaces to flag the *in situ* blocks as mined allowing reporting of *in situ*, depleted and mined tonnes and grade from the model.

Table 19-56: Samat Model – Depleted Mineral Resources for Au, 0.5 g/t Au cut-off By Domain

Domain	Classification	Material	KTonnes	Au g/t	Au KOz
Samat East	**Inferred**	oxide	402.1	1.13	14.6
		sulphide	3499.7	0.78	88.3
		transition	82.7	0.78	2.1
	Inferred Total		**3984.4**	**0.82**	**104.9**
Samat North A	**Indicated**	oxide	163.6	0.75	4.0
		sulphide	358.9	0.81	9.4
		transition	43.3	1.29	1.8
	Indicated Total		**565.8**	**0.83**	**15.1**
	Inferred	oxide	115.1	0.85	3.1
		sulphide	1080.6	0.86	29.7
		transition	10.6	1.26	0.4
	Inferred Total		**1206.3**	**0.86**	**33.3**
Samat North B	**Indicated**	oxide	117.1	0.87	3.3
		sulphide	1900.2	1.22	74.5
		transition	51.9	2.86	4.8
	Indicated Total		**2069.2**	**1.24**	**82.5**
	Inferred	oxide	119.1	0.75	2.9
		sulphide	1048.3	0.73	24.6
		transition	20.8	0.78	0.5
	Inferred Total		**1188.3**	**0.73**	**28.0**

Domain	Classification	Material	KTonnes	Au g/t	Au KOz
Samat South A	**Indicated**	oxide	21.6	2.11	1.5
		sulphide	18.9	1.01	0.6
		transition	5.0	0.98	0.2
	Indicated Total		**45.6**	**1.53**	**2.2**
	Inferred	oxide	161.1	1.28	6.6
		sulphide	1730.8	0.97	53.9
		transition	8.5	0.77	0.2
	Inferred Total		**1900.4**	**0.99**	**60.7**
Samat South B	**Indicated**	oxide	50.2	2.94	4.7
		sulphide	1696.1	1.76	96.1
		transition	51.2	2.03	3.3
	Indicated Total		**1797.5**	**1.80**	**104.2**
	Inferred	oxide	165.9	1.51	8.1
		sulphide	3356.3	1.07	115.4
		transition	21.8	0.99	0.7
	Inferred Total		**3544.0**	**1.09**	**124.2**

Table 19-57: Samat Model – Depleted Mineral Resources for Au, 0.5 g/t Au cut-off by Classification

Classification	Material	KTonnes	Au g/t	Au KOz
Indicated	oxide	352.5	1.18	13.4
	sulphide	3974.1	1.41	180.6
	transition	151.5	2.07	10.1
Indicated Total		**4478.1**	**1.42**	**204.1**
Inferred	oxide	963.3	1.14	35.3
	sulphide	10715.7	0.91	311.9
	transition	144.5	0.85	3.9
Inferred Total		**11823.4**	**0.92**	**351.1**

Table 19-58: Samat Model – Depleted Mineral Resources for Au, 0.5 g/t Au cut-off by Material type

Material	Classification	KTonnes	Au g/t	Au KOz
oxide	Indicated	352.5	1.18	13.4
	Inferred	963.3	1.14	35.3
sulphide	Indicated	3974.1	1.41	180.6
	Inferred	10715.7	0.91	311.9
transition	Indicated	151.5	2.07	10.1
	Inferred	144.5	0.85	3.9

Table 19-59: Samat Model – Depleted Inferred Mineral Resources for Ag and S, 0.5 g/t Au cut-off By Domain

Domain	Material	Tonnes	Ag g/t	Ag Oz	S ppm*
Samat East	oxide	402.1	1.47	19.0	0.48
	sulphide	3462.5	2.93	325.9	3.36
	transition	82.7	2.59	6.9	3.07
Samat North A	oxide	278.7	0.67	6.0	0.43
	sulphide	1435.3	2.11	97.2	2.82
	transition	53.9	1.11	1.9	1.27
Samat North B	oxide	236.2	1.42	10.8	0.77
	sulphide	2948.6	2.29	217.2	2.86
	transition	72.7	3.67	8.6	2.61
Samat South A	oxide	182.7	1.59	9.3	0.74
	sulphide	1682.3	0.37	20.2	3.60
	transition	13.6	0.59	0.3	4.46
Samat South B	oxide	216.1	1.76	12.2	0.53
	sulphide	5051.8	2.55	414.5	3.10
	transition	73.1	1.62	3.8	3.32

Table 19-60: Samat Model – Depleted Inferred Mineral Resources for Ag and S, 0.5 g/t Au cut-off by Material type

Material	KTonnes	Ag g/t	Ag KOz	S ppm*
oxide	1315.8	1.35	57.3	0.57
sulphide	14580.5	2.29	1075.0	3.16
transition	296.0	2.25	21.5	2.71

* S has not been estimated into all blocks due to the inconsistent assay suites for samples.

19.20 Mineral Reserves

Section 19.20 has been prepared by Golder Associates under the supervision of John Battista, Principal Mining Engineer.

John Battista of Golder, Perth, Western Australia is responsible for the development of the Mineral Reserves.

- Mineral Reserves for the Botlu South deposit are based on historical reserves from a 2005 Optimised Feasibility Study, depleted by mining up to the end of May 2009. No further mining occurred at Botlu South between May 2009 and end of October 2010.
- Mineral Reserves for the Sorowar deposit are based on a 2009 Technical Report, with reserves depleted by mining up to end of October 2010.
- Mineral Reserves for the Pigiput, Pigibo and Samat deposits are based on a 2010 Technical Report, with reserves depleted by mining up to the end of October 2010.

All Technical Reports were completed by Golder.

The original Mineral Reserve estimates are prepared in accordance with JORC standards. The JORC Proven and Probable Ore Reserves categories have been directly transferred to Proven and Probable Mineral Reserves as prescribed by National Instrument 43-101.

The pit designs from which the Mineral Reserves estimates are derived were based on Lerchs-Grossmann pit optimisations using various gold prices, costs and other technical parameters, as outlined in Table 19-61.

Table 19-61: Pit Optimisation Parameters

Technical Study	2005	2010	2009	2009
Deposit	Botlu South	Samat	Sorowar	Pigiput/Pigibo
Mining Method	Selective	Selective	Selective	Selective
Gold Price (USD/oz)	350	1000	750	1000
Waste Mining Cost (AUD/tonne)	3.98	Ox – 4.00 Tr – 4.00 Pri – 5.00	Oxide – 4.00 Trans – 4.00 Primary – 5.00	Ox – 4.00 Tr – 4.00 Pri – 5.00
Ore Mining Cost (AUD/tonne)	3.98	Ox – 4.00 Tr – 4.00 Pri – 5.00	Oxide – 4.00 Trans – 4.00 Primary – 5.00	Ox – 4.00 Tr – 4.00 Pri – 5.00
Processing Cost (AUD/tonne)	10.36	Ox – 12.96 Tr – 12.96 Pri – 35.65	Oxide – 14.00 Trans – 14.00 Primary – 14.00	Ox – 12.96 Tr – 12.96 Pri – 35.65
Au Recovery (%)	Oxide – 92.2	Ox – 89 Tr – 71.8 Pri – 81	Oxide – 86.5 Tr – 71.8 Pri – 65.6	Ox – 89 Tr – 71.8 Pri – 81
Slope Angles	45	47	47	Oxide – 35 Pri – 47
Royalty (%)	2.0	2.0	2.0	2.0
Discount Rate (10%)	10	10	10	10

There is some allowance for dilution inherent within the block modelling process for all of the models and no additional dilution or mining recovery factors used in the optimisation process.

Golder has used Whittle Four-X™ optimisation and Maptek Vulcan™ software with mine production scheduling and cost estimation completed using a Microsoft Excel spreadsheet.

The mining methodology is a conventional “selective mining” or “selective mode” approach, where material above a nominated cut-off grade is fed to the mill and material below the nominated cut-off grade is sent to waste dumps.

Table 19-62: Bulk Mining Pits – Theoretical Au Cut-off Grades

Deposit	Oxide (g/t Au)	Transitional (g/t Au)	Primary (g/t Au)
Samat East	0.69	2.05	n/a
Samat South	0.71	2.05	n/a
Samat North	0.67	2.05	n/a
Botlu South	0.91		n/a

For the selective mode optimisations, the cut-off grades used are as summarised in Table 19-63.

Table 19-63: Selective Mining Pits – Theoretical Au Cut-off Grades

Deposit	Oxide (g/t Au)	Transitional (g/t Au)	Primary (g/t Au)
Sorowar	0.51	0.62	0.68
Pigiput	0.37	0.46	1.12
Pigibo	0.37	0.46	1.12
Samat North A	0.37	0.46	1.12
Samat North B	0.37	0.46	1.12
Samat South	0.37	0.46	1.12
Botlu South	0.91		n/a

The cut-off grade for Primary material at Pigiput. Pigibo and Samat assumes that a new sulphide processing circuit will be installed on Simberi Island in future. The technical work on this processing path is currently at the PFS stage and is sufficiently advanced to permit the inclusion of the Pigiput, Pigibo and Samat sulphide material in Mineral Reserves Estimates.

The pit optimisation runs on which the Mineral Reserves are based did not consider material in the Inferred resource category and hence the pit limits are not driven by any Inferred resources. However, the pit designs resulting from these optimisations do include some Inferred class material and this is reported separately from the Mineral Reserves.

Table 19-64 shows the Proven and Probable Mineral Reserves and the Inferred Resources contained within the various designed pits.

Table 19-65 shows the Mineral Reserves and the Inferred Resources within the pit designs, categorised by Material Type (Oxide, Transitional and Primary).

Table 19-64: Mineral Reserves and Resources within Design Pits

Deposit	Tonnage (Mt)	Au (g/t)
Sorowar		
Proven	7.17	1.21
Probable	6.66	1.22
Total Mineral Reserves	**13.83**	**1.22**
Inferred Resource	0.49	0.91
Pigiput		
Proven	4.07	0.74
Probable	19.53	1.83
Total Mineral Reserves	**23.60**	**1.64**
Inferred Resource	3.15	1.18
Pigibo		
Proven	0.00	0.00
Probable	5.51	1.14
Total Mineral Reserves	**5.51**	**1.14**
Inferred Resource	0.60	0.77
Samat North A		
Proven	0.00	0.00

Deposit	Tonnage (Mt)	Au (g/t)
Probable	0.13	0.73
Total Mineral Reserves	**0.13**	**0.73**
Inferred Resource	0.02	0.64
Samat North B		
Proven	0.00	0.00
Probable	0.50	2.03
Total Mineral Reserves	**0.50**	**2.03**
Inferred Resource	0.00	0.68
Samat South		
Proven	0.00	0.00
Probable	0.81	2.41
Total Mineral Reserves	**0.81**	**2.41**
Inferred Resource	0.02	1.68
Botlu South		
Proven	0.74	1.35
Probable	0.12	1.61
Total Mineral Reserves	**0.86**	**1.39**
Inferred Resource	0.02	1.40
Total All Pits		
Proven	11.97	1.06
Probable	33.26	1.61
Total Mineral Reserves	**45.24**	**1.46**
Inferred Resource	4.29	1.09

Table 19-65: Mineral Reserve and Inferred Resource by Material Type

Classification	Oxide		Transition		Sulphide		Total Ore	
	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)	Tonnes (Mt)	Au (g/t)
Sorowar								
Proven	6.23	1.23	0.55	1.08	0.38	1.10	7.17	1.21
Probable	5.40	1.18	0.52	1.29	0.74	1.49	6.66	1.22
Total Mineral Reserves	**11.64**	**1.21**	**1.07**	**1.18**	**1.12**	**1.36**	**13.83**	**1.22**
Inferred Resource	0.43	0.84	0.00	0.82	0.06	1.39	0.49	0.91
Pigiput								
Proven	4.07	0.74					4.07	0.74
Probable	5.35	0.82	1.67	0.86	12.51	2.39	19.53	1.83
Total Mineral Reserves	**9.42**	**0.79**	**1.67**	**0.86**	**12.51**	**2.39**	**23.60**	**1.64**
Inferred Resource	1.79	0.62	0.34	0.71	1.02	2.33	3.15	1.18
Pigibo								
Proven								

Classification	Oxide		Transition		Sulphide		Total Ore	
Probable	3.43	1.00	1.65	1.21	0.43	2.04	5.51	1.14
Total Mineral Reserves	**3.43**	**1.00**	**1.65**	**1.21**	**0.43**	**2.04**	**5.51**	**1.14**
Inferred Resource	0.58	0.77	0.02	0.63			0.60	0.77
Samat North A								
Proven								
Probable	0.12	0.69	0.00	1.80	0.00	1.60	0.13	0.73
Total Mineral Reserves	**0.12**	**0.69**	**0.00**	**1.80**	**0.00**	**1.60**	**0.13**	**0.73**
Inferred Resource	0.02	0.64	0.00	0.00	0.00	0.00	0.02	0.64
Samat North B								
Proven								
Probable	0.12	0.79	0.05	3.61	0.33	2.28	0.50	2.03
Total Mineral Reserves	**0.12**	**0.79**	**0.05**	**3.61**	**0.33**	**2.28**	**0.50**	**2.03**
Inferred Resource	0.00	0.67	0.00	0.98	0.00	0.00	0.00	0.68
Samat South								
Proven								
Probable	0.08	2.39	0.05	2.16	0.68	2.43	0.81	2.41
Total Mineral Reserves	**0.08**	**2.39**	**0.05**	**2.16**	**0.68**	**2.43**	**0.81**	**2.41**
Inferred Resource	0.01	1.76	0.00	0.49	0.00	1.62	0.02	1.68
Botlu South								
Proven	0.74	1.35					0.74	1.35
Probable	0.12	1.61					0.12	1.61
Total Mineral Reserves	**0.86**	**1.39**					**0.86**	**1.39**
Inferred Resource	0.02	1.40					0.02	1.40
Total All Pits								
Proven	11.04	1.06	0.55	1.08	0.38	1.10	11.97	1.06
Probable	14.63	1.01	3.95	1.11	14.68	2.34	33.26	1.61
Total Mineral Reserves	**25.67**	**1.03**	**4.50**	**1.11**	**15.06**	**2.31**	**45.24**	**1.46**
Inferred Resource	2.85	0.69	0.36	0.71	1.08	2.27	4.29	1.09

The stated Mineral Reserves are included in the Mineral Resources quoted in 19.1.

As part of the mine planning process, the effect of changes in variables such as price and metallurgical recovery were examined. There are no particular factors other than those covered in this report that would have a significant impact upon these estimates.

The categorisation of Proven and Probable Reserves in this study has used the accepted guidelines defined within the Canadian NI 43-101.

The 4.29 Mt of Total Inferred Resource in 19.1 is not included in the Total Mineral Reserve Estimate.

20.0 OTHER RELEVANT DATA AND INFORMATION

20.1 Regional Exploration

There are no adjacent properties on Simberi Island (Figure 20-1) or the other Tabar Islands however there are other exploration stage prospects within EL609.

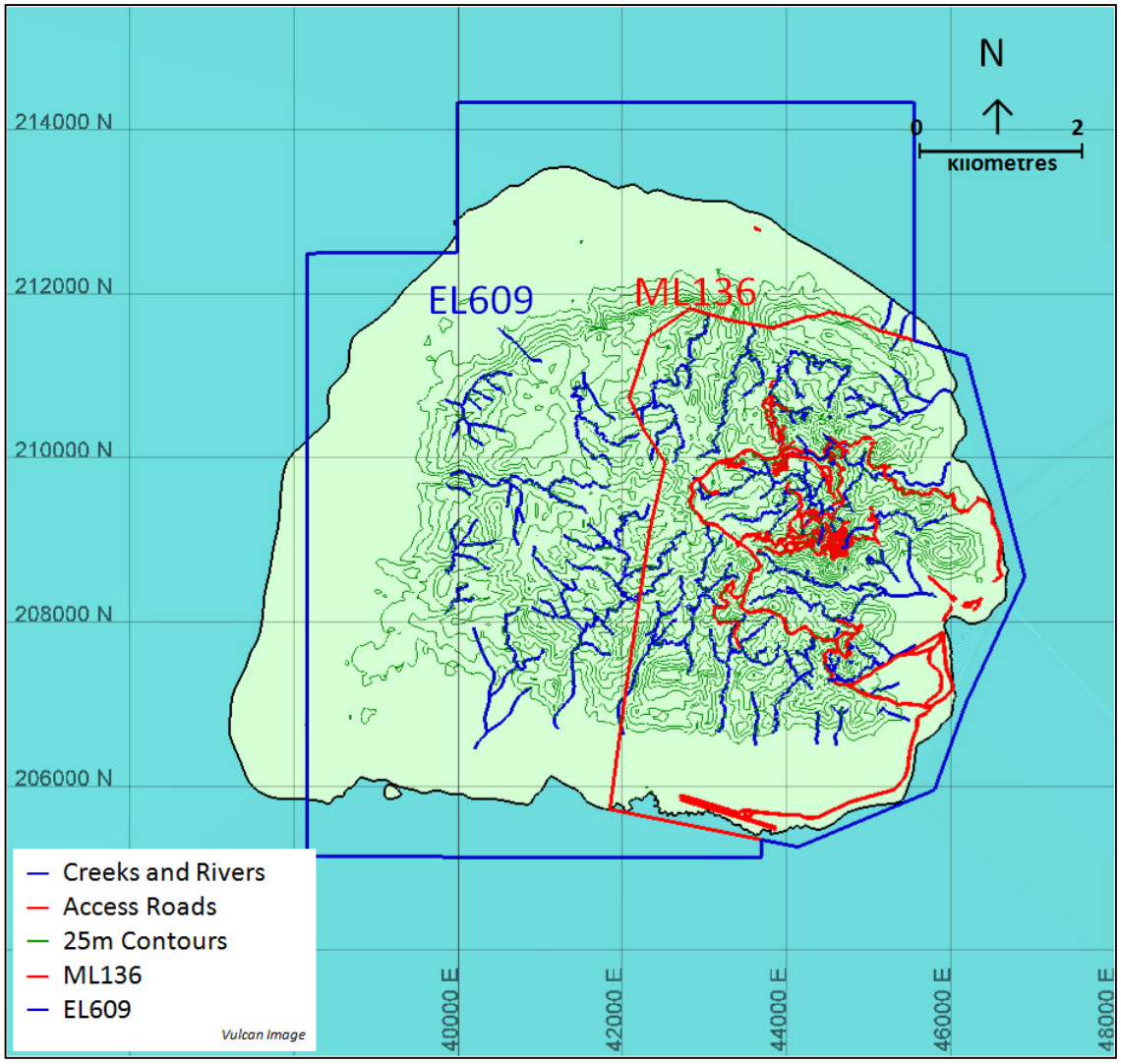

Figure 20-1 Simberi Island

20.1.1 Simberi Island

20.1.1.1 Patan

The Patan deposit is covered by a 250 m by 50 m gold-in-soil anomaly defined by a 0.2 g/t Au contour. Drilling by Kennecott in 1990 intersected gold mineralisation in two out of ten holes but did not adequately test the area. An IP gradient array geophysical survey was completed in 1997 but identified no anomaly. Allied drilled 11 RC holes in 2007 with poor results.

20.1.1.2 Adora

Drilling at Adora in 1997 intersected anomalous oxide mineralisation in two holes. During 2007 and 2008 Allied drilled five diamond holes and 16 RC holes and encountered no significant mineralisation.

20.1.1.3 Kekenminda

Drilling at Kekenminda to the west of Sorowar was targeting a radiometric (potassium count) anomaly by Nord. Drill holes intersected weakly mineralised zones with best intersection of 3 m at 2.13 g/t from RC870 received, from 6 RC of holes totalling 621 m drilled.

20.1.2 Tatau Island

20.1.2.1 Daramba

Daramba is a copper prospect located on the east coast of Tatau Island (refer to Figure 20-1). Diamond core drilling of a 1.1 km long by 600 m wide north-south trending copper-in-soil anomaly was completed in June 1996 with poor results. An IP gradient array geophysical survey completed in June 1997 identified a 1.2 km long by 200 m wide north-south striking chargeability anomaly which coincides with the greater than 300 ppm Cu soil anomaly.

20.1.2.2 Tugi Tugi

Gold was initially discovered at Tugi Tugi on the east coast of Tatau Island in the 1930s. Several small hard rock and alluvial workings occur within the prospect. Soil sampling and trenching by the Tabar Joint Venture (TJV) and Nord identified a broad gold anomaly defined by a 0.2 g/t Au contour. Narrow zones of gold mineralisation were intersected by the 32 holes that have been drilled by Kennecott and Nord.

20.1.2.3 Talik and Talik West

In late-1995, a soil auger program first identified the Talik and Talik West prospects in central Tatau Island. Bulldozer benching at these prospects has defined copper and gold anomalies. An IP gradient array geophysical survey was completed over the Talik prospect in July 1997 and identified a broad, strong chargeability anomaly that strikes north-east for 350 m.

20.1.2.4 Kupo

A copper-gold anomaly was originally identified by TJV near Kupo, on the west coast of Tatau Island. Nord confirmed this anomaly with samples taken from Pakinapote Creek and other westward flowing creeks draining the central and north-west areas of Tatau. Detailed rock chip sampling, hand dug benching, limited bulldozer benching and a Wacker drilling program was carried out at the Kupo prospect by Nord in 1997.

20.1.2.5 West Tatau Prospects

The TJV identified numerous gold prospects in the south-west corner of Tatau. These prospects had strong surface gold geochemical anomalies identified by soil sampling and benching, and several of them were drilled. Drilling results were erratic with many zones of anomalous gold intersected. Limited IP surveys were also completed over specific prospect areas. Nord has not undertaken any exploration work over this area.

20.1.3 Tabar Island (also see Barrick)

20.1.3.1 Banesa

Exploration at the Banesa copper-gold prospect, in the south of Tabar Islands, by the TJV comprised rock chip channel sampling, hand dug benching, auger sampling and one diamond core hole.

In November 1997 Nord sampled a shallow auger grid over the prospect, and confirmed a copper anomaly (>500 ppm Cu) covering a 450 m by 200 m area centred on Mt Fotombar, partially coincident with a 250 m long by 100 m wide gold anomaly (>0.10 g/t Au).

20.1.3.2 Tupinda

Some soil and rock chip sampling was carried out by Kennecott together with an airborne geophysical survey. Nord carried out benching aimed at opening up areas of identified gold anomalism.

20.1.3.3 Fotombar

All exploration work to date was carried out by Kennecott. Reconnaissance work and soil and chip sampling identified a 170 m by 70 m zone of anomalous gold and arsenic defined by the 1.0 g/t Au contour and coincident 100 ppm As anomaly. Bulldozer benching and drilling intersected zones of mineralisation.

Table 20-1: Drilling on Tabar/Tatau

Island	Prospect	ID	Year	Company	Drill Company	Hole Type	Total
Tabar	BANESA	BAN	1996	Nord	PNG Drillers	DD	379
			2009	Barrick	Capital	DD	1,409
		DAR	1996	Nord	PNG Drillers	DD	849
	TUPINDA	TUP	2008	Barrick	Capital	DD	701
			2009	Barrick	Capital	DD	1,130
	FOTOMBAR	FOT	1989	Kennecott	PNG Drillers and Wallis Drilling	DD	63
						RC	182
			1990	Kennecott	PNG Drillers and Wallis Drilling	DD	333
						RC	129
						Total RC	**129**
						Total DD	**5,175**
Tatau	DURAMBA	DAR	1996	Nord	PNG Drillers	DD	849
		LAB	1989	Kennecott	PNG Drillers and Wallis Drilling	RC	961
		LET	1989	Kennecott	PNG Drillers and Wallis Drilling	RC	530
			1990	Kennecott	PNG Drillers and Wallis Drilling	DD	235
						RC	34
	MAOKOEN	MAK	1990	Kennecott	PNG Drillers and Wallis Drilling	DD	214
	NALU	NAL	1989	Kennecott	PNG Drillers and Wallis Drilling	RC	922
	PEPEWO	PEP	1989	Kennecott	PNG Drillers and Wallis Drilling	RC	1,085
			1990	Kennecott	PNG Drillers and Wallis Drilling	DD	76
						RC	109
	SERARO	SER	1989	Kennecott	PNG Drillers and Wallis Drilling	RC	1,508
	SIRO	SIR	1989	Kennecott	PNG Drillers and Wallis Drilling	RC	617
	TALIK	TAL	2006	Allied	Zenex	DD	1,190
						RC	465
	MT TIRO	TIR	1989	Kennecott	PNG Drillers and Wallis Drilling	RC	1,409
			1989	Kennecott	PNG Drillers and Wallis Drilling	DD	271
	TUGI TUGI	TUG	1989	Kennecott	PNG Drillers and Wallis Drilling	RC	1,114
			1996	Nord	PNG Drillers	DD	535
						RC	246
	WARBARIAH	WAR	1990	Kennecott	PNG Drillers and Wallis Drilling	DD	107
						RC	49
						Total RC	**9049**

Island	Prospect	ID	Year	Company	Drill Company	Hole Type	Total
						Total DD	3477
Tabar/Tatau						Total RC	9360
						Total DD	7492
						Grand Total	16,852

20.1.4 Kennecott

Table 20-2: Kennecott RC Drilling Results – Pepewo, Tatau Island

Hole	TIG North	TIG East	RL (m)	Dip/Azi°	From (m)	To (m)	Intercept (m)	Au Grade (g/t)
RC1016	**188728.8**	**37747.8**	**167.4**	**-60/045**				
Pepewo					48	67	19	9.43
				inc.	***53***	***67***	***14***	***12.32***
				and	***53***	***60***	***7***	***22.99***
				and	***54***	***59***	***5***	***29.78***
					69	96	27	1.25
				inc.	***71***	***73***	***2***	***3.01***
				and	***80***	***87***	***7***	***2.11***
				and	***89***	***92***	***3***	***1.54***
					103	105	2	1.15
RC1029	**188744.1**	**37743.0**	**157.9**	**-60/045**				
Pepewo					14	30	16	5.65
				inc.	***14***	***27***	***13***	***6.41***
				and	***15***	***18***	***3***	***4.44***
				and	***15***	***16***	***1***	***6.50***
					20	27	7	9.58
				and	***21***	***27***	***6***	***10.5***
				and	***23***	***24***	***1***	***33.0***
					47	72	25	1.38
				inc.	***47***	***57***	***10***	***2.08***
				and	***50***	***52***	***2***	***2.77***
				and	***54***	***56***	***2***	***2.87***
RC1098	***188702.3***	***37774.7***		**-90/007**				
Pepewo					34	50	16	2.84
				inc.	***34***	***44***	***10***	***4.02***
				and	***38***	***44***	***6***	***5.67***
				and	***38***	***40***	***2***	***8.73***

Table 20-3: Kennecott RC Drilling Results – Seraro, Tatau Island

Hole	TIG North	TIG East	RL (m)	Dip/Azi°	From (m)	To (m)	Intercept (m)	Au Grade (g/t)
RC1027	**189326.5**	**36428.0**	**186.8**	**-60/225**				
Seraro					22	33	11	1.53
				inc.	***22***	***27***	***5***	***2.60***
				and	***22***	***24***	***2***	***4.74***
				and	***23***	***24***	***1***	***5.05***
					46	52	6	37.1
				inc.	***47***	***50***	***3***	***57.7***
					56	60	4	1.26
					112	114	2	0.88
					118	128	10	0.86
RC1020	**189429.6**	**36308.9**	**139.6**	**-60/225**				
Seraro					0	4	4	11.9
				inc.	***1***	***4***	***3***	***15.6***
				and	***1***	***3***	***2***	***20.3***
					130	136	6	0.93
RC1043	**189312.5**	**36261.7**	**100.0**	**-60/225**				
Seraro					0	2	2	0.62
					50	52	2	7.57
					50	51	1	14.0
					55	59	4	1.97
					75	84	9	2.68
				inc.	***75***	***79***	***4***	***4.92***
				and	***75***	***78***	***3***	***5.94***
				and	***75***	***76***	***1***	***12.7***
				inc.	***81***	***83***	***2***	***1.30***
					93	95	2	1.81
RC1050	**189322.5**	**36437.7**	**187.0**	**-60/315**				
Seraro					75	78	3	0.77
					95	101	6	0.80
					104	109	5	8.28
				inc.	***104***	***108***	***4***	***9.60***
				and	***105***	***107***	***2***	***11.0***

Table 20-4: Kennecott RC Drilling Results – Mt Tiro, Tatau Island

Hole	TIG North	TIG East	RL (m)	Dip/Azi°	From (m)	To m)	Intercept (m)	Au Grade (g/t)
RC1037	**188417.3**	**37270.0**	**280.7**	**-60/045**				
Mt Tiro					0	4	4	2.23
				inc.	***0***	***3***	***3***	***2.75***

Hole	TIG North	TIG East	RL (m)	Dip/Azi°	From (m)	To m)	Intercept (m)	Au Grade (g/t)
				and	***1***	***3***	***2***	***3.38***
					47	53	6	2.26
				inc.	***47***	***52***	***5***	***2.58***
				and	***50***	***51***	***1***	***5.67***
					86	100	14	3.23
				inc.	***88***	***98***	***10***	***4.14***
				and	***88***	***92***	***4***	***5.28***
				and	***96***	***98***	***2***	***5.34***
					104	124	20	3.08
				inc.	***110***	***120***	***10***	***5.71***
				and	***116***	***120***	***4***	***11.4***
				and	***118***	***120***	***2***	***14.2***
					128	140	12	0.61
					143	150	7	1.37
				inc.	***147***	***150***	***3***	***2.31***
DDHT1	**188413.9**	**37233.5**	**292.5**	**-90/007**				
Mt Tiro					0	16	16	2.26
				inc.	***1***	***6***	***5***	***1.68***
				and	***9***	***15***	***6***	***3.94***
				and	***10***	***11***	***1***	***8.06***
					41	47	6	0.66
					50	52	2	0.75
					66	70	4	0.57
					86	89	3	3.20
				inc.	***87***	***88***	***1***	***8.07***
					109	111	2	1.15
					116	122	6	2.57
				inc.	***117***	***122***	***5***	***2.91***
				and	***121***	***122***	***1***	***10.2***
					125	131	6	0.96
				inc.	***126***	***130***	***4***	***1.09***
					134	147	13	2.06
				inc.	***134***	***136***	***2***	***1.40***
				inc.	***138***	***139***	***1***	***5.63***
				inc.	***141***	***145***	***4***	***3.69***
					172	174	2	1.28
					176	183	7	1.09
				inc.	***179***	***181***	***2***	***2.31***
					193	199	6	0.77

20.1.5 Barrick

Under the terms of a letter of intent, effective from May 2008 to Feb 2010, Barrick (PNG) explored within those blocks of Allied Gold's exploration licence EL609 covering the adjacent Tatau (EL609 Block B) and Big Tabar (EL609 Block C) Islands. Barricks activities included:

- Completion of 4 diamond core holes totalling 1,927 m at the Tupinda Prospect, Big Tabar
- Completion of 6 diamond core holes totalling 2,625 m at the Banesa Prospect, Big Tabar, and
- Reconnaissance geological mapping and rock-chip sampling of the alkaline intrusive complex on central Tatau Island.

20.1.5.1 Banesa Prospect

A six hole diamond drilling program at Banesa was completed in May 2009. Holes BND2, and BND4-7 intersected zones of Cu/Au alkaline porphyry style mineralisation. Significant intersections using a 0.5 g/t Au equivalent cut off and maximum of 2 m internal dilution are summarised in Table 20-5.

Logged Cu/Au mineralisation in hole BND2 (best intercept 22 m at 1.58 g/t Au and 0.32% Cu from 54 m) is associated with a potassic altered micromonzonite porphyry between 20 and 211 m.

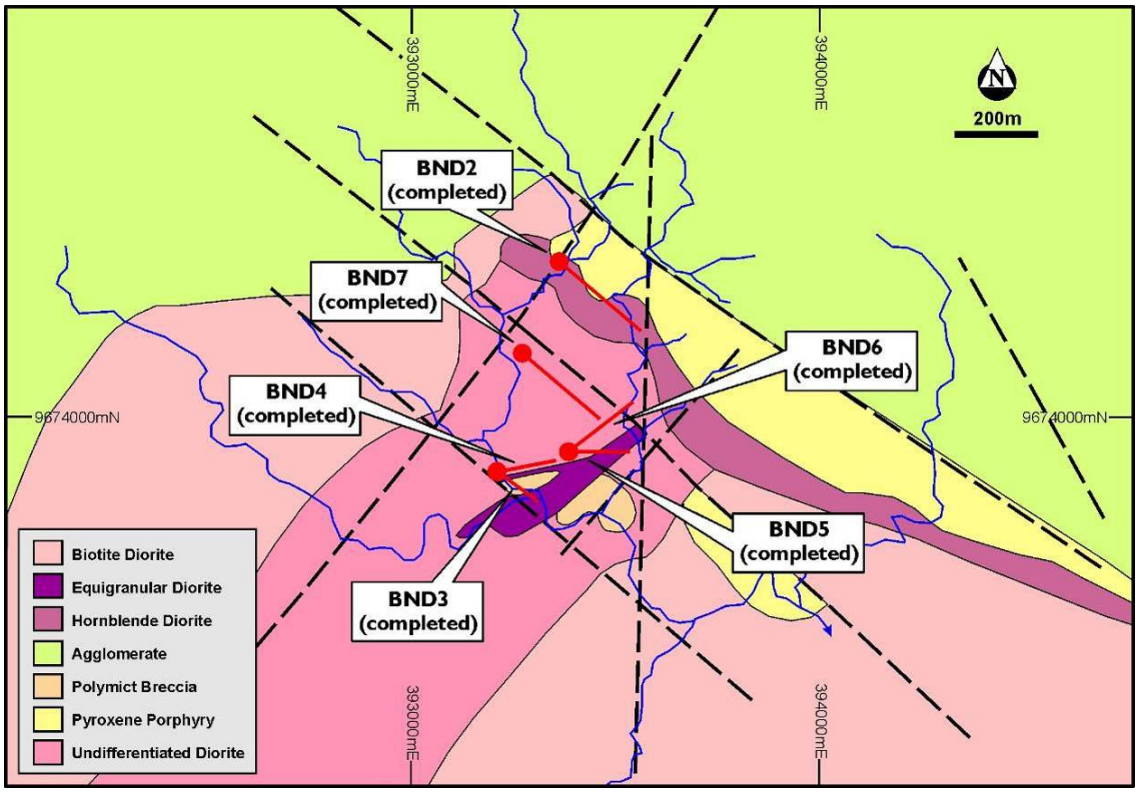

Figure 20-2 Plan view geology and location map of Banesa target area on Tabar Island

Hole BND5 (best intercept 69 m at 0.83 g/t Au and 1.13% Cu from 50 m) intersected a brecciated feldspar-hornblende monzodiorite that, from 99 to 140 m, hosts sheeted quartz veinlets (up to 10% of rock volume) with phyllic (sericite-silica) alteration over-printing earlier potassic alteration.

Hole BND6 (best intercept 77 m at 0.46 g/t Au and 0.95% Cu from 62 m) commenced in phyllic altered polymictic breccia before entering a porphyritic monzodiorite between 74 m and 118 m. The monzodiorite

shows phyllic overprinting potassic alteration is locally brecciated and contains a sheeted quartz vein zone between 74 m to 95 m, similar to that intersected in BND5. The hole re-enters altered polymictic breccia to 158 m with 1% disseminated pyrite. A contact between an upper pyroxene-porphyritic monzodiorite and a lower plagioclase-biotite-horneblende monzodiorite occurs at 235 m (9.8 m at 0.69 g/t Au from 235.2 m).

From 237 m to 435 m, hole BND7 (collared 200 m SW of BND2, best intercept 18 m at 1.38 g/t Au and 0.36% Cu from 398 m) intersected medium grained plagioclase-pyroxene monzodiorite. Copper mineralisation, as fracture controlled chalcopyrite with minor bornite, appears at 240 m and continues sporadically to the end of hole.

Barrick are assessing whether coherent zones of potassic alteration are likely along strike of or below the Banesa drilling.

Table 20-5: Banesa drill hole geochemistry *(Note BND1 is a Kennecott 1989 drill hole)*

Hole ID	Utm North	Utm East	Utm RL	Dip/Azim	From (m)	To (m)	Intercept (m)	Au Grade (g/t)	Cu Grade (%)
BND2	393355	9674385	147	-60°/129	0.0	521.0			
					54.0	76.0	22.0	1.58	0.32
				incl	***62.0***	***63.6***	***1.6***	***5.08***	***0.83***
					100.0	112.0	12.0	0.87	0.33
					134.0	142.0	8.0	1.75	0.30
					146.0	150.0	4.0	1.18	0.50
					180.2	184.1	3.9	1.01	0.29
					203.6	207.6	4.0	0.90	0.37
BND3	393208	9673872	112	-60°/125	0.0	164.3			
					No significant intercepts				
BND4	393208	9673871	112	-60°/080	0.0	485.6			
					210.0	215.2	5.2	0.94	0.37
					227.0	245.0	18.0	1.29	0.56
					368.0	372.3	4.3	0.79	0.24
					458.0	464.0	6.0	0.52	0.11
BND5	393376	9673921	207	-60°/090	0.0	483.6			
				***	50.0	119.0	69.0	0.83	1.13
				incl	***61.0***	***83.0***	***22.0***	***1.05***	***1.22***
				incl	***63.0***	***67.0***	***4.0***	***2.74***	***1.75***
				and	***103.0***	***119.0***	***16.0***	***1.59***	***1.54***
					121.7	125.0	3.3	0.90	0.09
					127.6	138.4	10.8	1.20	0.14
BND6	393376	9673922	207	-60°/050	0.0	535.4			
					40.0	44.0	4.0	0.63	0.11
				***	62.0	139.0	77.0	0.46	0.95
				incl	***88.0***	***116.8***	***28.8***	***0.91***	***1.31***
					235.2	245.0	9.8	0.69	0.06
BND7	393246	9674196	146	-60°/135	0.0	435.1			
					130.0	133.0	3.0	0.67	
					356.0	363.6	7.6	1.33	0.26

Hole ID	Utm North	Utm East	Utm RL	Dip/Azim	From (m)	To (m)	Intercept (m)	Au Grade (g/t)	Cu Grade (%)
					398.0	416.0	18.0	1.38	0.36

NOTE: Except where indicated by ***, downhole intercepts are determined using an Au cut-off of 0.5 g/t Au and minimum 2 m length. Such intercepts may include material below cut-off but no more than 2 sequential metres of such material and except where the average drops below the cut-off. Selvage is only included where its average grade exceeds 0.5g/t. Using the same criteria for included sub-grade, supplementary cut-offs of 2.5 g/t, 5.0 g/t and 10 g/t are used to highlight higher grade zones and spikes. Single assays intervals are reported only where >5.0 g/t and >=1 m downhole. The limit of detection for gold is <0.005 g/t. A sample, with a reported below detection grade, is assigned a grade of half the detection limit, in this case 0.0025 g/t. Intercept grades are calculated using sample grades weighted by sampled length divided by interval length. This results in any included core loss being assigned zero grade. Cu grade is its weighted average over the same intervals as those defined by the Au intercepts. The Cu detection limit is 1 ppm and is determined by Aqua Regia digest of a 0.5g charge followed by ICP AES analysis. Where indicated (***), the intercept is defined by using a Cu cut-off of 0.5% Cu and minimum 2 m length.

20.1.5.2 Tupinda

Assay results received for four holes drilled at the Tupinda Prospect showed weak Cu-Mo-Au mineralisation in TPD1, TPD3 and TPD4 (Figure 20-3 and Table 20-6).

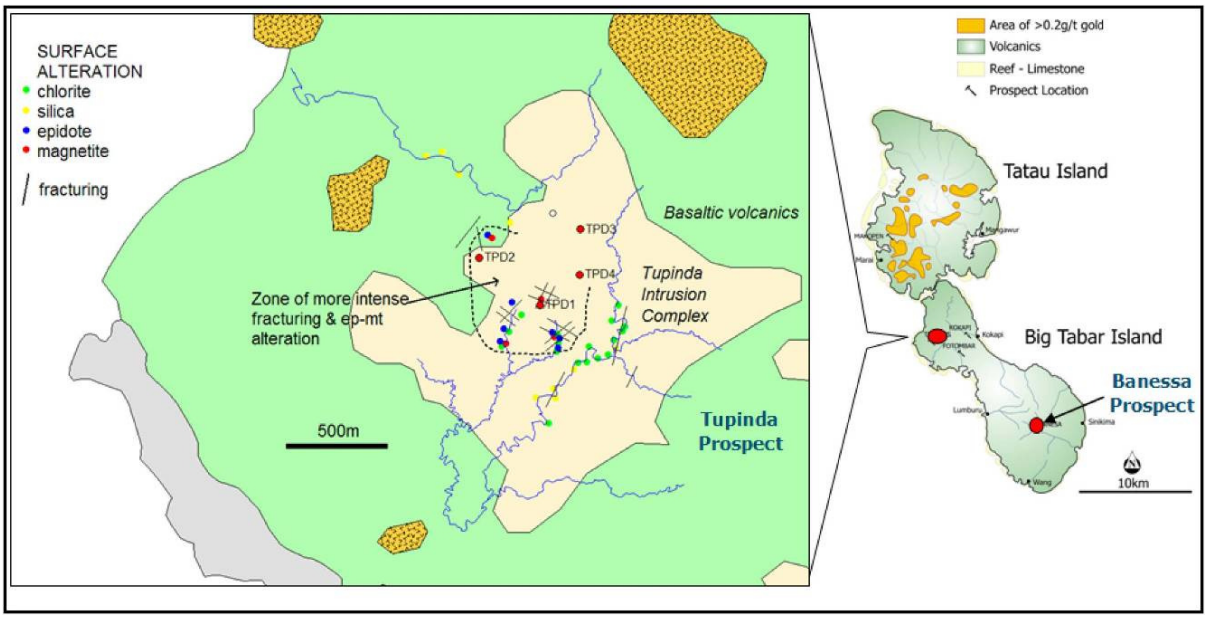

Figure 20-3: Tupinda project alteration and drilling

Table 20-6: Tupinda drill hole geochemistry

Hole ID	Utm North	Utm East	Utm RL	Dip/Azim	From (m)	To (m)	Intercept (m)	Au Grade (g/t)	Cu Grade (%)
TPD001	384670	9681262	72	-88°/332	0.0	491.9			
				***	229.0	236.0	7.0	0.14	0.42
TPD002	384344	9681516	63	-90°/170	0.0	279.4			
					no significant intersections				
TPD003	384859	9681664	162	-89°/226	0.0	500.0			
					356.0	359.5	3.5	1.17	0.19
TPD004	384868	9681383	141	-90°/105	0.0	655.6			
				***	297.1	299.0	1.9	0.25	0.38

NOTE: Except where indicated by ***, downhole intercepts are determined using an Au cut-off of 0.5 g/t Au and minimum 2 m length. Such intercepts may include material below cut-off but no more than 2 sequential metres of such material and except where the average drops below the cut-off. Selvage is only included where its average grade exceeds 0.5 g/t. Using the same criteria for included sub-grade, supplementary cut-offs of 2.5 g/t, 5.0 g/t and 10 g/t are used to highlight higher grade zones and spikes. Single assays intervals are reported only where >5.0 g/t and >=1 m downhole. The limit of detection for gold is <0.005 g/t. A sample, with a reported below detection grade, is assigned a grade of half the detection limit, in this case 0.0025 g/t. Intercept grades are calculated using sample grades weighted by sampled length divided by interval length. This results in any included core loss being assigned zero grade. Cu grade is its weighted average over the same intervals as those defined by the Au intercepts. The Cu detection limit is 1 ppm and is determined by Aqua Regia digest of a 0.5g charge followed by ICP AES analysis. Where indicated (***), the intercept is defined by using a Cu cut-off of 0.5% Cu and minimum 2 m length.

20.1.5.3 Tatau Island

First pass mapping of Tatau Island (70% complete) identified a broad area of medium grained equigranular monzodiorite, underlying most of the Cu-Mo anomalous area defined by soil and surface rock chip sampling. Zones of propylitic and phyllic alteration are mapped out within the monzodiorite that is also intruded by N and NE-striking feldspar porphyry and mafic dykes.

A previously identified strong Cu-Au surface rock chip sampling anomaly was traced to an outcrop of a 10 m wide feldspar porphyry dyke in the upper Talik drainage. The dyke shows strong silica-sericite alteration with quartz-chalcopyrite-magnetite veinlets. Other feldspar porphyry dykes in the Talik area show variable intensities of chlorite-carbonate alteration with rare chalcopyrite-galena-pyrite fracture fill. Within the area mapped, neither convincing alteration zonation, quartz stockwork veining or porphyry style mineralisation was encountered.

20.2 Infrastructure

20.2.1 Overview

This section is included to show that much of the infrastructure required for the Simberi combined oxide/sulphide project is currently in place as part of the Simberi oxide operations. The oxide processing plant (centre right), the aerial rope conveyor (top right), warehouse and exploration laboratory (centre front), wharf and Company vessel “Lady Geraldine” (left) are shown in Figure 20.4.

Figure 20.4 shows an overview of the infrastructure and process plant already existing at the coast.

Figure 20.4: Coastal Infrastructure Simberi Gold

20.2.2 Power

20.2.2.1 Historical

The existing power station is operated by Aggreko under contract number 183-EC-02. The power station is diesel fired and rated to a maximum demand of 6 MW. The power plant normally operates at a load of between 3 MW and 3.5 MW.

Up to 10% of the total demand is provided by the aerial rope conveyor, which generates power while moving the ore down from Sorowar over a vertical fall of about 230 m.

Power is distributed from the power station at 11 000 V to the main transformers adjacent to the motor control centres within the process plant, the Sorowar mining area and via two overhead powerlines to the warehouse, wharf and the accommodation village.

The individual generator sets are Aggreko NHC20/KTA50G3 units having a peak rating of 1250 kVA at 50 Hz and a continuous load rating of 850 kW. The units operate coupled to dedicated step-up transformers (415 V to 11 kV) that feed the main power station distribution and control switchboard. The output terminals of the distribution and control switchboard constitute the metering and energy transfer point under the contract.

The basis of the BOO Contract with Aggreko is comprised of three pricing components:

- Base facilities charges that also include the generation charges for the first 500 kW/d of load.

- Generation charges made on the basis of metered kWh for all load above 500 kW/d.
- Maintenance consumables.

Allied provides diesel fuel, freight for equipment, consumables required for power plant operation and accommodation and messing for the Aggreko power plant attendants.

The average power load for the existing Simberi project is estimated at 3070 kW. Power output from the power plant was 3250 kW. Allied Gold has assessed the diesel consumption for 2009, which amounted to A$0.9864/kWh. Cost of power based on usage of 1800 MWh/month is A$0.37/kWh.

For the expanded oxide case of 3.5 Mtpa, cost is estimated at $0.34/kWh, based on average load of 4090 kW and maximum demand of 6000 kW under the same type of contract.

HFO is under consideration as an alternative fuel source for power supply for the combined oxide expansion and sulphide project.

20.2.2.2 Combined Oxide/Sulphide Expansion Project

Power demand, which is based on the estimated motor load and plant utilisation, of 8 MW is required for the sulphide plant, with a total power demand of 14 MW for the combined oxide and sulphide plant. Installed power will be 25 MW for the sulphide and the proposed oxide plant expansion. The installed power does not account for energy recovered from the roaster, and is calculated based on maximum load and 100% plant utilisation for all plant equipment.

A study has been completed by GRES, “Simberi Gold Project – HFO and Power Options Study”, that investigates power supply for the project, particularly HFO. Recent political development in PNG has removed the opposition by the government for the use of HFO in mining facilities within PNG. It is proposed to utilise HFO for power generation and this choice has predominately been driven by the large power requirement for the sulphide plant and subsequent substantial operating cost for this entity. Lihir Gold Mine currently operates its power station using HFO and subsequently the vessel that delivers the fuel to Lihir can also be used for Simberi, allowing the potential delivery to site via a submersible pipeline.

HFO costs are based on capital cost for additional wharfage and pylons, as well as tankage of $7.6 M. Operating cost is based on a BOO response received by GRES from D&G Energy for a HFO power station contract supply that calculated to A$0.2235/kWh. Although this was based on a lower power supply for 8 MW, it is considered a reasonable estimate to use for power costs for the PFS.

The roaster design incorporates waste heat recovery from the roaster off-gas of approximately 3 MW. This is based on the assumption that the roaster operates at 700°C and the thermal to electrical energy ratio is 3:1, resulting in 45% of the energy from the roaster bed and 55% by heat recovery from the exhaust gas. The waste heat recovery system consists of a waste heat boiler, a steam drum and bed coils, which allows efficient use of the numerous and critical steam/roaster interfaces, from a mechanical, process control and operational perspective. The use of coils in the steam system eliminates the need for a gas cooler and reduces the volume of water in the system, consequently reducing the roaster freeboard diameter, duct sizes, Electrostatic Precipitator (ESP), scrubber vessel sizes and ID Fan.

Figure 20.5: shows the current power station at Simberi.

20.2.3 Water

Potable water for the camp and the main treatment plant is obtained from a spring and from a water bore. Water is filtered through a sand and UV filtration system. Rainwater is harvested from rooftops of some of the main buildings and stored in poly tanks with bottled potable water supplied for drinking.

Raw water supply for the main treatment plant was initially sourced from the ocean via a pump station located on the western side of the wharf. In 2009, process water has been sourced from surface water that has ponded in a pit that was formed during construction to win coronus. This pond is located about 500 m south-west of the process plant and the water is pumped to the process raw water pond and is augmented with sea water when required. Adjacent to this, lies a smaller pond with fresh water which is used as process water.

Simberi Gold holds extraction licenses for creek water (Monun and Darum Creeks) and groundwater (Pigiput Plantation) and Monun Creek catchment although currently none of these sources are being used. Additional water will be provided from a silt collection pond downstream of the pits and in later years from pit dewatering pumps.

Additional water for the expanded 3 5 Mtpa case and the sulphide plant will be mainly sea water.

20.2.4 Buildings

Process plant buildings, located at 10 m RL, include the site administration office, mill operations offices and training rooms, site assay laboratory, plant maintenance workshop and stores, and site security buildings. Simberi Island is located in a Category 2 earthquake zone and the engineering design of the process plant and associated buildings and infrastructure has taken this into account.

No additional buildings are required for the expanded oxide or sulphide case.

20.2.5 Camp Accommodation

The accommodation centre has current capacity of 208 beds, which includes a mix of single rooms with ensuites or shared adjoining bathrooms, single room dormitories with central bathroom and bunk houses with common ablutions. New single room blocks with bathroom ensuites (15 rooms) are currently being installed in the new accommodation centre and an additional five single room blocks are being installed in the old exploration camp (50 beds), which already has shared and single rooms with a common ablutions block (32 beds).

The accommodation centre has a kitchen and dining area and wet mess, which was upgraded in Q3 2009. A laundry facility caters for washing residents' clothes, towels and linen and catering for the camp is contracted. Sewage and waste water from the camp is pumped into the tailings discharge line for subsea disposal at a depth of 115 m below sea level.

Additional camp capacity will include a 100 man construction camp to facilitate the construction crew for the sulphide project, and upon completion of the construction phase, 50 beds will be removed leaving 50 additional beds for the sulphide plant operations team.

Figure 20.6 shows the Simberi Camp with the operation in the background.

Figure 20.6: Simberi Camp

20.2.6 Airstrip and Wharf

20.2.6.1 Airstrip

Simberi Gold has constructed a 1400 m long class "Y" airstrip within the Company's Pikung Plantation that is located on the south coast of Simberi Island approximately 3 km from the process plant site (Figure 20.7). This facility can accommodate aircraft up to Dash 8 size and citation jets.

It is planned to upgrade the airstrip for night operation to allow 24 hour medivac access. Non-local workforce is rostered on a fly in fly out (FIFO) operation that currently has two rosters of 16/12 (days on/days off) and

28/14. PNG nationals are flown back to their points of hire and expatriates are flown back to Australia to their points of hire. The aircraft (Dash 8) is under a charter contract with PNG carrier Airlines of PNG, and services ports of Kavieng in New Ireland Province, Rabaul in East New Britain, and Port Moresby.

Figure 20.7: Simberi Airstrip

20.2.6.2 Wharf

A 20 m long concrete wharf constructed by Simberi Gold is located on the northern side of Pigiput Bay (Figure 20.8). It is proposed to install dolphins off the wharf, which will allow one vessel to berth onshore while the other vessel is tied up. There is an option to construct the dolphins using the coronus limestone reserves on the island. This will also facilitate the off-loading of HFO, which will be delivered from the supply vessel to site via a submersible pipeline, while also increasing the capacity of the Lady G LCT vessel to transport other reagents and consumables.

Ships with up to 6.5 m draft can be received and offloaded at the wharf. A 110 t crawler crane capable of unloading 25 t containers at 12 m reach operates at the wharf. The Company owned landing barge, MV Lady Geraldine, services the operation with regular sailings to Lae (three return trips per month) and Rabaul (one to two trips as required). The vessel has a capacity of 450 t of cargo and can hold 500 kL of diesel, and has greater capacity for transportation of other reagents and consumables required for the plant, given that HFO fuel will be delivered via a submersible pipe. A barge ramp is located at the head of Pigiput Bay and used to discharge roll on roll off cargo.

Figure 20.8: Simberi Wharf

20.2.6.3 Roads and Communications

The existing perimeter road (26 km) at Simberi was upgraded during construction and is used to transport local employees to and from their villages to work, using man-hauls that are either owned by the company or contracted from local clan business groups. All weather internal mine access roads have been constructed as well as a 1500 m long haul road from the Samat pits to the process plant and a 500 m long haul road from the Sorowar pit to the Ropecon tip head.

Site communications including voice and data are provided through a VSat system operating on a band width of 1500 kbyte down and 280 kbyte up. A PNG Telikom VSat provides six incoming (five phones and one fax) lines. Phone extensions link the camp, treatment plant and infrastructure facilities and allow for two payphones with one being located in the camp and the other for public use. A PNG Telikom mobile phone tower has been installed on Simberi and provides mobile service to the island.

A new haul road will need to be constructed from the Pigiput Mine down to the coast for transportation of ore and waste material.

21.0 INTERPRETATION AND CONCLUSIONS

21.1 Resources

The Simberi resource is a robust and proven gold deposit.

The current resource models show a positive reconciliation in the mined pits. This issue should be investigated with the objective of optimising production and potentially increasing the mine life.

The historical resources at Botlu, Pigibo are accepted as Inferred based on the Author's validations. The resource models should be re-estimated using the current data set and understanding of the deposits' geology.

The Sulphide resource appears to have potential and should be included in all future deposit assessments

Beyond the Simberi mining lease the exploration ground held by Allied over the Island group is all very prospective.

21.2 Reserves

The reserves at Simberi, being based on the resource are also proven and robust. Improvements to the reserve position come directly from improvements and increases in the resource. The development of some of the satellite deposits will add to the reserve base. Inclusion of more Sulphide material has the potential to significantly increase reserves.

21.3 Processing

- The current oxide processing plant is operating well and achieving good oxide recoveries.
- The infrastructure that is in place at Simberi Gold is adequate for the 2 Mtpa operation, and with limited upgrading including the wharf facility, will be adequate for the 3.5 Mtpa oxide plus the 1.5 Mtpa sulphide operation.
- The estimated capital cost for the oxide plant expansion to 3.5 Mtpa by GRES of $32 M is reasonable for the modifications and additions proposed.
- The estimated capital cost for the sulphide plant by GRES of $188 M is reasonable for the plant proposed.
- Operating costs for the expanded oxide plant of $9.25/t are based on operating costs for the current six month operating period with a reduction associated with fixed costs for the expanded plant.
- Operating costs for the sulphide plant of $23.29/t have been calculated based on first principles and the use of HFO. This cost has been benchmarked against a comparable operation, and is considered reasonable.
- The gold grades which will be fed to the oxide process plant average 1.01 g/t. Sulphide gold grades average 2.41 g/t. Maintaining low operating costs will be critical to project viability.
- The 3.5 Mtpa oxide case in conjunction with the 1.5 Mtpa sulphide case is commercially viable with pre-tax NPV of $334 M at 10% discount factor (this includes the current 2 Mtpa operation). Cash cost is $678/oz.

22.0 RECOMMENDATIONS

- The remaining historical resources should be re-estimated incorporating the current geological understanding of the island, recent drilling and other geological data and any updated topographical information. Outsourced this is estimated to cost AUD$50 K.
- The positive mining reconciliation should be investigated. This could be done in-house or outsourced at an estimated cost of AUD$30 K.
- A critical review of current mining operations should be undertaken and any revisions applied to the optimisation, pit design and scheduling. This could be done in-house or outsourced at an estimated cost of AUD$40 K.
- Optimisation of the resource and reserve calculations, final pit designs and scheduling should be revisited on a regular basis. The pit optimisations need to be updated to reflect more recent increasing trend in the gold price.
- Exploration drilling to investigate the nature of the sulphide mineralisation beneath the current Sorowar pit design should proceed as a matter of priority. There appears to be significant potential to increase the sulphide resource and reserve base at Simberi, and thus improve the overall economics of the project, through improving the understanding of the sulphide mineralisation in the Sorowar area.
- Expansion of the oxide plant to 3.5 Mtpa should proceed. Estimated cost is $32 M.
- Allied Gold should consider a Bankable Feasibility Study, in conjunction with additional drilling, that will provide greater confidence to the combined oxide and sulphide project. The cost of a Bankable Study is estimated at $6-$10 M. The total cost of the combined oxide and sulphide project is estimated at $278 M including the $32 M oxide plant expansion.
- Allied should closely manage its operating costs to keep its costs within budget estimates.

23.0 REFERENCES

Documents referenced and reviewed during the compilation of the Technical Report

Author	Date	Title
Allied Gold (a)	2003	A Brief Report on Simberi Mining Area Association Project Development Issues and Simberi Village Survey, R. Hastings 30 September 2003
Allied Gold (b)	2009	Project to Date Met Accounting Spreadsheet
Allied Gold (c)	2009	SIM Mine Input Model – Final Budget 17 July 2009
Allied Gold (d)	2009	Battery08_09 Spreadsheet Operating Cost Data
Allied Gold (e)	2009	Email Frank TerraNova 2 Sep 2009 with Hedging Information
Allied Gold (f)	2009	Emails Ross Hastings Simberi NI43-101 Infrastructure 25 Aug 2009
Allied Gold (g)	2009	Application for Extension of Term of EL609, 2 Februaury 2009
Allied Gold (h)	2009	EL609 Graticule Map
Behre Dolbear Australia	1996	Independent Technical Review of the Geology, Data Collection and Resource Estimation for The Simberi Gold Project – PNG Nord Resources (Pacific) Pty Ltd, 12 July 1996
Coffey (a)	2008	Compliance_Monitoring Report (2008), Coffey Natural Systems Pty Ltd, October 2008
Coffey (b)		3 Mtpa Expansion Environmental Assessment Proposal
DEC (a)	2005	Approval of Environmental Management Program for Dimberi Gold Mine, Department of Environment & Conservation, 26th January, 2005
DEC (b)	2006	Grant of Environmental Permits, 17 July 2006
Dept of Attorney General	1996	Memorandum of Agreement relating to the Simberi Mining Project between The Independent State of Papua New Guinea and the New Ireland Interim Provincial Government and Simberi Gold Company Pty Ltd and Simberi Landowners Association and Tabar Community Government, 21/11/1996
Deputy Registrar of Companies	1996	Certificate of Incorporation – Simberi Gold Company Pty Ltd, 8 November 1996
Enesar (a)	2004	Supplementary Baseline Study, Eneasr Consulting Pty Ltd, April 2004
Enesar (b)	2005	Supplementary Baseline Study No. 2, Eneasr Consulting Pty Ltd, August 2005
Enesar (c)	2006	Waste Management Plan, Eneasr Consulting Pty Ltd, September 2006
Enesar (d)	2006	Conceptual Mine Closure Plan, Eneasr Consulting Pty Ltd, May 2006
GRES (a)		11551 – 0009 Power Study Options A
GRES (b)		11551 – 0021 3 Mtpa Study
GRES (c)		11550 – 0086 Debottlenecking Study Actions Rev E

Author	Date	Title
Investment Promotion Authority	1997	Certificate Permitting a Foreign Enterprise to Carry on Business in an Activity 0 Certificate No 1655, 6 January 1997
Lycopodium	2003	Feasibility Study Simberi Mining Joint Venture, Simberi Oxide Gold Project, August 2003. Lycopodium Pty Ltd.
MRA (a)	2007	Renewal – Mining Lease No 136, PNG MRA Office of the Registrar of Mineral Tenements, 4 May 2007
MRA (b)	2009	Notice of Application for extension of a term of a tenement for exploration licence No 609 under section 106 (A) (D) Mining Act, 1992, 23 February 2009
MRA (c)	2008	Notification of Grant of Extension of Term – EL 609, 3 June 2008
NSR (a)	2003	Simberi Oxide Gold Project, Environmental Baseline Study, December 2003, NSR Environmental Consultants Pty Ltd
NSR (b)	2008	Simberi Oxide Gold Project Environmental Plan, September 1996, NSR Environmental Consultants Pty Ltd
Oakvale Capital Limited	2009	Simberi Gold Company Hedging Position, 30/6/2009
Office of Civil Aviation	1999	Aerodrome Licence, 16 March, 1999
Pangtel	2007	PNG Radio and Telecommunications Authority – Issue of Corporate licences – Land Mobile Repeater Station, 4/07/2007
PNG	1992	Mining Act 1992 and Regulation, Independent State of Papua New Guinea.
QSA Global	2007	Certificate for Sealed Radioactive Sources No 109179-OV726, 22 May 2007
Registrar of Tenements	2009	Consent to Export Geological Samples and Specimens, 30 July 2009
Registrar of Titles (a)	1938	Land (ownership of Freeholds) ACT 1976 – Lease – Pikung Plantation
Registrar of Titles (b)	1985	Land (ownership of Freeholds) ACT 1976 – Lease – Pigiput Plantation
Simberi Gold Company Ltd	2005	Understanding on Issues for the Simberi Project Tabar Islands Group, N.I.P. PNG between Simberi Mining Joint Venture and The Landowners of Simberi Island, 6/4/2005
Sode, D.	2005	Tax Exemption Letter, Internal Revenue Commission, Port Moresby, PNG

Contract documents referenced and reviewed during the compilation of the Technical Report

Author	Date	Title
Allied Gold Limited c1	16/10/2008	Property Lease Level 1, 15 Mallon St Bowen Hills Qld 4006, 24/11/2008 – 3 yrs + 3mnths
Allied Gold Limited c2	14/09/2004	Transfer of Nord Pacific Limited stock to Allied
Simberi Gold Company Ltd c1	2007	Oxide Pit Clans Limited – Bulldozer Hire – 01/11/2007 to 31/10/2009

Author	Date	Title
Simberi Gold Company Ltd c2	2007	Sordar Clan – Rubbish Collection and Small Tipper Truck Services – 01/3/2007 to 28/2/2009
Simberi Gold Company Ltd c3	2008	Manhaul Services by Berar Clan 01/11/2008 to 31/10/2009
Simberi Gold Company Ltd c4	2008	Manhaul Services by Lavamadeis Clan 01/11/2008 to 31/10/2009
Simberi Gold Company Ltd c5	2008	Perivot Clan – Provision of Stone Pitching on Simberi Island – one year – dated 18/01/2008
Simberi Gold Company Ltd c6	2009	Airport Ground Clearing – Wuitgamgam Clan – 3 months ending 31 march 2009
Simberi Gold Company Ltd c7	2009	Various documents relating to export/import of geological core for geochemical analysis by ALS Chemex
Simberi Gold Company Ltd c8	2009	Camp Grass Clearing – Sitchium Clan – 3 months ending 31 March 2009
Simberi Gold Company Ltd c9	2009	ELTA Construction and Engineering Limited – Day Labour Services – one year ending 30 June 2010
Simberi Gold Company Ltd c10	2009	Lane Investments – Ground Clearing – 3 months ending 31 March 2009
Simberi Gold Company Ltd c11	2009	Kaparak Clan – Maintenance of Air Conditioning – 3 months ending 31 March 2009
Simberi Gold Company Ltd c12	2009	Rumarik Clan – Provision of Day Labour – one year to 28 February 2010
Simberi Gold Company Ltd c13	2009	Oxide Pit Clans Limited – Day Labour Services – one year ending 30 June 2010
Simberi Gold Company Ltd c14	2009	Oxide Pit Clans Limited – Wet Hire Trucking Services – one year ending 9 April 2010
Simberi Gold Company Ltd c15	2009	Shunammite Engineering Ltd – Wet Hire of Plant – 12 months ending April 2010
Simberi Gold Company Ltd c16	2009	Simberi Resources Development Company Limited – Labour Hire for the Light Vehicle Workshop – 12 months ending 1/3/2010
Simberi Gold Company Ltd c17	2009	JT & Sons Tyre Service – provision of tyre service on Simberi Island for one year – dated 29/11/2008
Simberi Gold Company Ltd c18	2/12/1996	Compensation Agreement for the Simberi Project Tabar Island Group, N.I.P. PNG between Simberi Gold Company Pty Ltd and The Landowners (revised march 2005 and 29/6/2005)
Simberi Gold Company Ltd c19	7/10/2005	Mine Site Construction services – Cat D6R LGP Dozer
Simberi Gold Company Ltd c20	Dec-06	Catering Contract between Simberi Gold Company Limited and Eurest (PNG) Catering and Services Limited
Simberi Gold Company Ltd c21	21/12/2006	Telikom PNG Installation and Hire telecommunications services + various documents/communications
Simberi Gold Company Ltd c22	18/04/2007	Contract for Power Generation Services between Simberi Gold Company and Aggreko Generator Rentals Pty Ltd
Simberi Gold Company Ltd c23	5/12/2007	Refining Agreement – Simberi Gold Company Limited and AGR Matthey

Author	Date	Title
Simberi Gold Company Ltd c24	18/01/2009	ADT 740 Ejector Hire – Hastings Deering (PNG) Limited and Simberi Gold Company (B1R00478)
Simberi Gold Company Ltd c25	1/03/2009	Offers by Islands Petroleum Ltd to supply petroleum products to Simberi Gold Ltd
Simberi Gold Company Ltd c26	11/06/2009	Aircraft Charter Agreement between Simberi Gold Company and Airlines of Papua New Guinea Limited
Simberi Gold Company Ltd c27	13/06/2009	Diamond Drilling Contract between Capital Drilling Limited and Simberi Gold Company Limited
Simberi Gold Company Ltd c28	17/07/2009	Special Lease and Dry Hire agreement with Mrs Linus Vunagoi and Mr Danny Maris on behalf of BIK Tabar Island for Fork Lift Hire 1/3/2009 to 30/11/2011
Simberi Gold Company Ltd c29	1/12/2009	ADT 740 Ejector Hire – Hastings Deering (PNG) Limited and Simberi Gold Company (B1R00477)

24.0 QUALIFIED PERSONS STATEMENTS

CERTIFICATE OF QUALIFIED PERSON

I, Stephen Godfrey, of Perth, Australia do hereby certify that:

- I am a Principal Resource Geologist with Golder Associates Pty Ltd., 1 Havelock Street, West Perth, Australia.
- I am a graduate of The University of New England, NSW, Australia, B.Sc.(Hons), 1982.
- I am a member in good standing of the Australian Institute of Mining and Metallurgists.
- I have practiced my profession for 26 years since graduation.
- My relevant experience with respect to the Simberi Gold Project includes 17 years resource modelling of a variety of metalliferous projects including 8 years working with greenstone gold deposits.
- I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("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 am a "qualified person" for the purposes of NI 43-101.
- I am responsible for the preparation of Sections 1-17, 19.1, 20.1, 21, 22, 23, 24, 25.3, 25.6, 25.10, 25.11 of the technical report titled "Simberi Gold Project, Simberi Island, Papua New Guinea", dated November 2010 (the "Revised Technical Report").
- I have been involved with the Simberi Gold Project since 2004, as an independent geological consultant with Golder Associates Pty Ltd.
- As of the date of this certificate, 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.
- I am independent (as defined by Section 1.4 of NI 43-101) of Allied Gold Limited
- I have read NI 43-101 and 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

Signed and dated this 18th day of November, 2010 at Perth, Australia

signed Stephen Godfrey

Signature

CERTIFICATION OF QUALIFIED PERSON

I, Phil Hearse, do hereby certify that:

- I am a graduate from University of South Australia, Australia, with a Bachelor of Applied Science degree in Primary Metallurgy in 1973 and a graduate from Hull University United Kingdom with a Master of Business Administration degree in 1994. I have continually practiced the profession of metallurgist since 1973.
- I am a Fellow of The Australasian Institute of Mining and Metallurgy.
- I am Managing Director of BatteryLimits Pty Ltd, a firm of consulting metallurgists and process and risk engineers which has been practicing in this profession since 2004. I hold office at 140 Hay St, Subiaco, WA 6008, Australia and have been employed as such since 2004. Prior to this time I was Director Advisory and Principal Consultant for the engineering company GRD Minproc Ltd for four years and before that Managing Director of the metallurgical consultancy Normet Pty Ltd for 18 years.
- I have held technical and operational roles at Broken Hill in silver lead zinc flotation, at Bougainville Copper in copper/gold flotation and heap leaching, at Queensland Nickel in nickel and cobalt hydrometallurgy and at Gove alumina in alumina processing. With 35 years mining industry experience, I have provided operational and consultancy services to many mining projects and operations.
- I have provided consultancy services to numerous gold projects.
- I have read the definition of “qualified person” as 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 fulfil the requirements to be a qualified person for the purpose of NI 43-101.
- I have prepared Sections 18, 20.2, 21, 22, 23, 24, 25.2, 25.4, 25.5, 25.7-9 of the technical report titled “Simberi Gold Project, Simberi Island, Papua New Guinea, dated November 2010 (the “Revised Technical Report”).
- I have visited the Simberi Gold site in February 2009 and most recently in June 2009.
- I have no personal knowledge, as of the date of this Certificate, of any material fact or change, which is not reflected in this report, the omission to disclose that would make this report misleading.
- Neither I, nor any affiliated entity of mine is at present, or under an agreement, arrangement or understanding expects to become, an insider, associate, affiliated entity or employee Allied Ltd, and/or any associated or affiliated entities.
- A related party to me does hold securities in Allied Gold Ltd, amounting to less than 0.01% of the company’s total securities. Allied Gold Ltd has advised that it does not consider this material in compromising my independence. As such, I am independent (as defined by Section 1.5 of NI43-101) of Allied Gold Ltd.
- I have read the NI 43-101 and Form 43-101F1, and CIM Standards on Mineral Resources and Reserves, and have prepared the technical report in compliance with NI 43-101 and Form 43-101F1.

Signed and dated this 18th day of November, 2010 at Perth, Australia

signed Phil Hearse

Signature

CERTIFICATE OF QUALIFIED PERSON

I, John Battista, of Perth, Australia do hereby certify that:

- I am a Principal Mining Engineer with Golder Associates Pty Ltd., 1 Havelock Street, West Perth, Australia.
- I am a graduate of the Western Australian School of Mines, Kalgoorlie, Western Australia (a branch of Curtin University of Technology), B.Eng.(Mining).
- I am a member and Chartered Professional in good standing of the Australasian Institute of Mining and Metallurgy (AusIMM).
- I have practiced my profession for 21 years since graduation.
- My relevant experience with respect to the Simberi Gold Project includes approximately 11 years' experience in technical, mine operations and management positions in open pit gold mines, including two years as Senior Mine Planning Engineer at the Martha Hill gold mine in New Zealand, which is mining an epithermal gold deposit having similar mineralisation style to the Simberi deposits.
- I have read the definition of "qualified person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("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 am a "qualified person" for the purposes of NI 43-101.
- I am responsible for the preparation of Sections 19.2, 21, 22, 23, 24, 25.1of the technical report titled "Simberi Gold Project, Simberi Island, Papua New Guinea", dated November, 2010 (the "Revised Technical Report").
- I most recently personally inspected the Simberi Gold Project between 28th October 2008 and 31st October 2008.
- I have been involved with the Simberi Gold Project since 2004, as an independent mining consultant with Golder Associates Pty Ltd.
- As of the date of this certificate, 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.
- I am independent (as defined by Section 1.4 of NI 43-101) of Allied Gold Limited.
- I have read NI 43-101 and 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.

Signed and dated this 18th day of November, 2010 at Perth, Australia

Signed John Battista

Signature

25.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

25.1 Mining

25.1.1 Operations

The current mine plan consists of the open pit mining of seven oxide deposits as indicated on the plan below. The mining sequence was to commence at the smallest but highest grade deposits at Samat, then start mining at the largest deposit at Sorowar. Mining of Pigiput and Botlu would occur in conjunction with waste removal from Sorowar and finally the Pigibo deposit would be mined last.

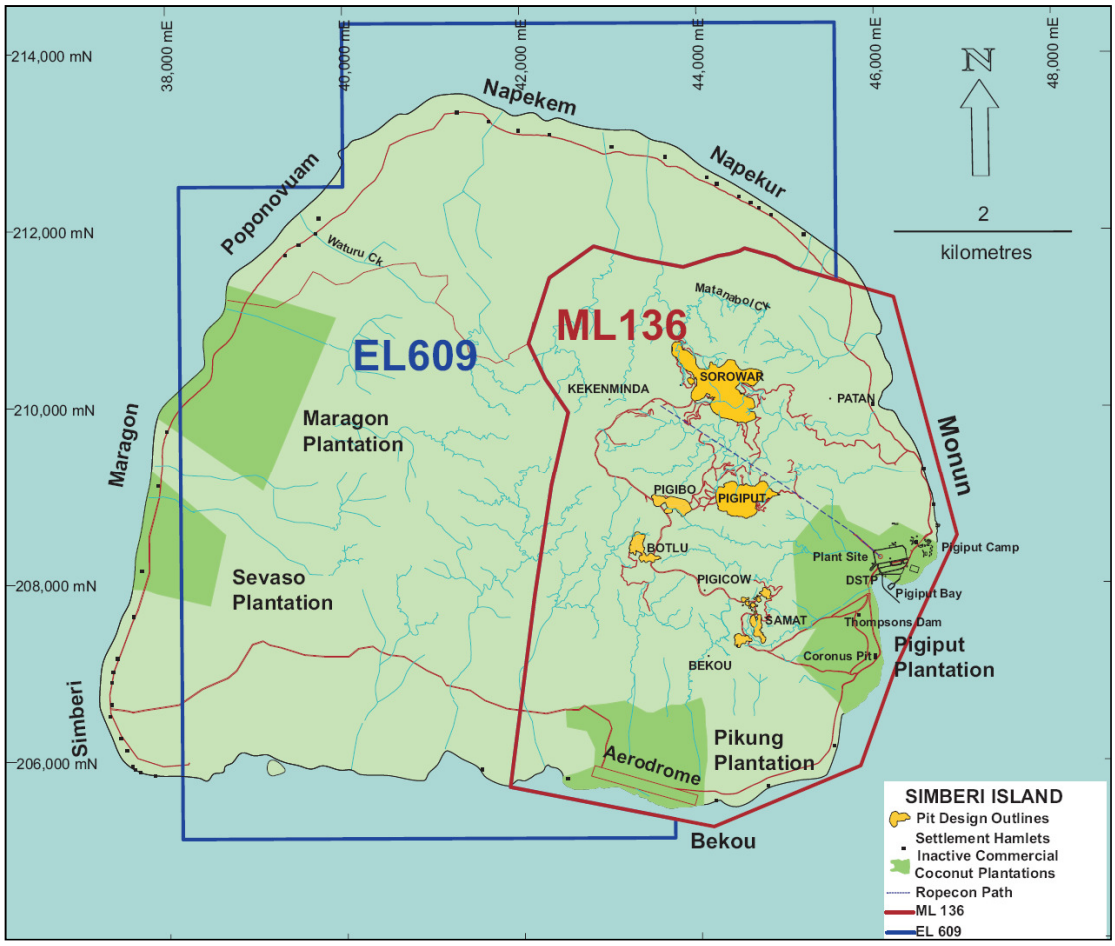

Figure 25-1: Simberi Island – Oxide pit outlines (Q2 2010)

The mineralisation in all oxide deposits excluding the deeper levels of the Sorowar deposit is disseminated to the point that there is very little internal waste. The limited waste present is still mineralised but at levels below cut-off and is contained in small pockets which would be inefficient to mine separately. For these reasons all the material extracted from the pits excluding the deeper levels of the Sorowar deposit and selected zones within the Pigibo deposit will be treated as ore. Material taken from pits with no waste generation is termed “bulk mining”, while material from pits that will produce waste is termed “selective mining”. The decision to mine some material selectively is based on practicability and economics. Any

waste generated will be contained within the mined void, thereby mitigating any potential environmental impacts.

Figure 25-2: Mining at Sorowar after rain

Figure 25-3: Pantera 1100 blast hole drill

The soft ground conditions and high rainfall combine to give a situation where maintaining large trafficable operating areas are impractical. To overcome this, the proposed mining method requires development of all weather sheeted roads onto the resource areas and then using dozers to push the soft material to the road where it will be either loaded into trucks at Samat and transported to the process plant at Pigiput Bay or similarly loaded into trucks and transported to aerial conveyor for conveying to the process plant at Pigiput Bay. Operational experience at Sorowar has resulted in a practice of ceasing mining operations during in rainfall events and feeding the process plant with ROM ore from either the larger (70,000 wet tonnes)Pigiput stockpile or the smaller (6,000 wet tonnes) Sorowar stockpile. This requirement to cease mining operation during rainfall periods has necessitated a larger than initially planned mining fleet to allow feeding the plant as well as replenishing ROM stockpiles while clear weather conditions permit.

As the larger pits at Sorowar, Pigiput and Pigibo deepen the ore although still oxidised becomes stronger to the point where ripping with a dozer is not cost effective or practical and light blasting will be carried out to break the ore up and make excavation easier. Drilling will be carried out using a top drifter blast hole drill. The nominal drill hole diameter will be 102 mm and holes will be drilled to 10.7 m with samples for grade control collected each 2.5 m and the 0.7 m sub-drill will not be sampled. Typical blast pattern design would be 4.5 m spacing and 4.0 m burden on a staggered pattern. Typical powder factor would be initially <0.4 kg/t but this would increase with pit depth to compensate for harder ore and waste. Groundwater for most part is absent although the moisture content of surface and near surface ores can be as high as 35%. It is anticipated that ANFO will make up the majority of explosive used although in wet area packaged water proof explosive will be used.

Currently grade control for ore/waste definition is carried out using a Super Rock Drill 3000 machine utilising a 102 mm reverse circulation (RC) drill string and down the hole RC hammer. The holes are normally drilled to 30 m depth with samples being collected each metre and assays being composited over 2.5 m which is the current mining height. The sample assays are subjected to a normal kriged analysis to produce grade blocks and ore/waste boundaries. Ore blocks are subdivided in to low grade and high grade material to allow ore blending to be carried out to maintain a somewhat constant mill feed head grade. Ore/waste boundaries are also pegged in the field.

Mining operations commenced at the Samat East deposit in November 2007 to provide a parcel of ore for plant commissioning. In February 2008 after plant commissioning, ore was sourced from the Samat deposits (North, South and East) and the ore was delivered to the process plant via a 1,500 m long haul road. The ore was excavated using a combination of Caterpillar 30t and Komatsu 45t hydraulic excavators. The ore was free digging and “bulk” mined and no waste was produced. Caterpillar 40t articulated trucks were used to deliver the ore directly into the dump pocket reclaimer or to the ROM stockpile at the process plant.

In May 2008 the aerial conveying system (Ropecon®) was commissioned and ore was mined from the Sorowar deposit and trucked approximately 400 m from the pit edge to the Ropecon dump pocket. The ore at Sorowar is free digging and mined by 45t hydraulic excavators and hauled to the dump pocket by 40t articulated trucks.

The dump pocket at Sorowar consists of a Stammler chain conveyor and hydraulic breaker rated at 600 wet tph. The ore is transferred from the breaker to the rope conveyor via a 49 m long sacrificial conveyor fitted with a metal detector. The rope conveyor consists of a 650 mm wide conveyor belt fitted with side skirts and has a capacity of 600 wet tph. The conveyor length is 2,498 m and falls a vertical distance of 210 m and is supported by 3 pylons placed on intervening ridges. At the bottom end of the rope conveyor ore is transferred to a conventional overland conveyor over a distance of 359 m before ore is transferred to a radial stacker that can feed directly into the process plant dump pocket or can be discharged onto the ROM which has a capacity of approximately 70,000 wet tonnes of ore. Ore on the ROM is either pushed by D6 dozer or tipped into the dump pocket by a Caterpillar 880 front end loader.

Figure 25-4: Super Rock drill 3000 undertaking grade control drilling at Sorowar

Figure 25-5: Mining at Samat South

Figure 25-6: Ropecon showing the discharge end and the overland conveyor with the process plant in the background

The Simberi mining operations is a conventional load and haul and this is reflected in the equipment schedule indicated below. Currently the fleet is a mix of owner and contractor equipment and the mining contractor is responsible for supervision of mining operations. The mining contractor is Mine Site Construction Services (MSCS), a Perth WA based company. Machine operators are a mix of non-local experienced PNG citizens and local PNG citizens that have been trained or undergoing training as machine operators. Mine technical support is provided by the Mine engineering Department that is responsible for mine geology and grade control, mine planning and scheduling, and mine survey. Work force levels are shown in the table below. Mine shifts comprise 2 × 12 hour shifts, however nominally local operators work

minimum 8 hours and up to 12 hours if required (typically happens after rainfall events). Non-local PNG operators and expatiates supervisors and staff operate on a fly in fly out (FIFO) roster of 28 days on/14 days off and 16 days on/12 days off respectively.

Mobile equipment repairs and maintenance is carried out on site through the company's maintenance department. This is supported by local caterpillar dealer (Hastings Deering) with consignment spares and mechanical labour.

Table 25-1: Simberi Gold Company (SGC) Mining Fleet

Machine	Number
Cat 740 articulated truck(1)	8
Cat 725 articulated truck	2
Cat 330 excavator	2
Komatsu PC450 LC-7 excavator	2
Cat D10N dozer	1
Cat D9N dozer	1
Cat D6R dozer	3
Cat 140 grader	1
Cat CS-563E roller	1
Cat IT38G(2)	1
Cat 980G FEL(2)	2

(1)Two trucks fitted with retractor trays

(2) Machines part of Process Department used for stockpile rehandle

Table 25-2: Simberi Mine Operations Work Force Levels (as at July 2009)

Source	Number
Expatriate	8
Non-Local PNG	44
Local PNG	67
Contractor	14
Total Mining	133

25.1.2 Reconciliation

Records for reconciliation of actual mined material versus geological model are kept by Simberi mining department staff on a monthly basis and summarised in Excel spreadsheets. This has been done in a consistent manner since start of mining on the island in February 2008.

The mining actual volume and tonnages for the reconciliation are based on both daily truck count figures and end of month survey volumes, with moisture contents, wet and dry densities based on regular sampling.

Overall, there is a positive reconciliation of actual against model in both tonnes (by an overall 6.4%) and grade (by an overall 1.9%), giving a positive reconciliation in actual contained gold mined against the models of some 8.4%.

25.2 Metallurgical Processes

25.2.1 Current Process Plant

25.2.1.1 Ore Transport and Scrubbing

Ore is mined at the Sorowar mine site and trucked to the Sorowar feeder. The Sorowar feeder and breaker delivers ore to the rope conveyor that conveys ore to the plant site on the coast. The rope conveyor has capacity to transport 600 wet t/h.

The rope conveyor discharges onto an overland conveyor and then stacking conveyor both with current carrying capacity of 512 wet t/h. The stacking conveyor discharges as feed to the Pigiput feeder and sizer.

The Pigiput feeder/reclaimer feeds the wet scrubber that is 3 m diameter by 7.5 m EGL (Effective Grinding Length). Water is added to the scrubber to slurry the ore. The unit is fitted with dual 160 kW drive motors. The capacity of the scrubber is a nominal 330 dry t/h.

Oversize from the scrubber has been fed to the ball mill but will feed directly to a pebble crushing circuit to be installed to reduce scats from the ball mill.

25.2.1.2 Grind-Leach

Scrubber undersize is pumped to a cyclone circuit. Cyclone underflow flows by gravity to a 3.4 m diameter by 5.0 m EGL ball mill. The mill drive is 850 kW and the mill rotates at 17.8 rpm (76% of the critical speed). Make-up ball addition is a mixture of 80 mm and 65 mm balls.

Cyclone overflow passes to the leach tanks. The cyclone overflow density feeding the leach circuit is operated at lower density than the original design (31%-34% solids versus 40% solids for design). A trash screen is installed on the leach feed to minimise oversize material in the carbon-in-leach (CIL) tanks that will potentially block the inter-tank screens.

The leaching and adsorption circuit at Simberi is comprised of a single leach stage followed by five CIL tanks. Recovery and adsorption efficiency is impacted by low pulp density and reduced residence time. The CIL circuit design includes only five stages of adsorption and does not deliver an acceptable soluble loss.

25.2.1.3 Elution and Carbon Regeneration

The acid-wash and elution functions are performed in a single 5 tonne capacity column. The column is butyl-lined carbon steel. The acid-wash and rinse is performed at ambient temperature. The elution process is AARL with pregnant liquor stored in a 98 m^3 tank in closed circuit with two electrowinning cells. A kiln is used to regenerate the carbon.

25.2.1.4 Tailings Disposal

Tailings are discharged using Deep Sea Tailings Placement (DSTP). The slurry is diluted with sea water to the ratio of 8:1 sea water to tailings volume. The diluted tailings overspills a central well in the dilution tank and flows by gravity from the outer annulus of the tank to the DSTP pipeline to be discharged 115 m below surface level on a steep submarine slope.

25.2.1.5 Reagents

Quicklime is currently fed dry onto the ball mill feed conveyor (CV06) from a variable speed screw conveyor from the quicklime storage bin. Coarse granular quicklime is purchased in bulk bags (arriving in sea containers) and manually handled into the storage bin by plant operations personnel.

Reagent storage volumes and dosing equipment are adequate for the operation.

25.2.2 Expanded Oxide Process Plant to 3.5 Mtpa

The expansion design concentrated on the ore handling and reclamation, grinding and classification and tailings thickening and disposal areas. Flexibility has been built into the milling circuit to handle 3.5 Mtpa throughput.

Major changes are:

- The Sorowar ore handling circuit will be modified to include a drive-over dump arrangement with the equipment upgraded to a capacity of 600 wet t/h. The Pigiput ore handling circuit will also be upgraded to a capacity of 600 wet t/h. Pigiput ore will be reclaimed by a new apron feeder discharging onto the mill feed conveyor.
- A new single stage, high aspect, 2.5 MW Semi-Autogenous Grinding (SAG) mill operating in closed circuit with the pebble crushing circuit and classification cyclones will be installed. The pebble crusher product will be recycled to the SAG mill feed conveyor, while the cyclone underflow will be returned to the SAG mill feed chute.
- The new SAG mill will be 6.7 m in diameter with an effective grinding length (EGL) of 3.2 m. This circuit is still under review.
- The leaching circuit will comprise two new 2500 m^3 mechanically agitated tanks in addition to the existing 5 × 1500 m^3 tanks already installed. Sodium cyanide is added into the leach tank distribution box.
- Oxygen gas from a new Pressure Swing Adsorption (PSA) plant will be added to the leach circuit via the tank agitator shafts. Slurry exiting the leach circuit flows by gravity to six mechanically agitated adsorption tanks. The adsorption tank interstage screens will be upgraded to 19 m^2 Alloytech screens as part of the debottlenecking to handle the increased slurry volume flow.
- For the expanded plant the daily loaded carbon required to be eluted is 6 t, while the existing elution circuit capacity is 5 t/cycle. To meet this requirement, eight to ten elution cycles per week are required. To manage this, rather than undertaking carbon acid washing and stripping in the one column, a new SAF 2205 column will be installed to separate these duties. Acid washing will be completed in the existing rubber lined column, while elution will be completed in the new SAF 2205 column.
- To meet the expanded gold productivity, the pregnant liquor storage and electrowinning facilities will be duplicated.
- Screened slurry discharging the adsorption circuit will be pumped to a new 26 m diameter high rate tailings thickener which will be located adjacent to the Deep Sea Tailings Pipeline (DSTP) dilution tank in Pigiput Bay. A new flocculant plant will also be installed to service the tailings thickener.
- The tailings are thickened to an underflow density of 53% solids and pumped to the tailings dilution tank. The pulp will then be diluted with sea water to the ratio of 8:1 sea water to tailings volume. The diluted tailings overspills a central well in the dilution tank and flows by gravity from the outer annulus of the tank to the DSTP.
- With the installation of the tailings thickener, the tailings dilution tank, DSTP and sea dilution pumps have adequate capacity for 3 Mtpa and upside capacity of 3.6 Mtpa.

25.2.3 Proposed Sulphide Process Plant

The sulphide plant is designed to treat 1.5 Mtpa (dry tonnes) of ore, with a design availability for the milling and flotation plant of 91.3%, based on 7,998 operating h/a. The plant has standby equipment in critical areas and sufficiently automated plant control to minimise the need for operator interface on a continuous basis but allow manual override and control if required.

25.2.3.1 Comminution

ROM ore will be dumped directly into the ROM bin by the haul trucks. Stockpiled ROM ore may also be reclaimed from the ROM pad by a Front-End loader (FEL). This will be fed to the 80 t capacity ROM bin. The ROM bin will be equipped with a 600 mm aperture static grizzly. Oversize material may be broken on the ROM pad area or returned to the mine.

Comminution will be via a semi-autogenous grinding (SAG) mill/ball mill/crusher configuration (SABC). Ore will be drawn from the ROM bin, via a 1200 mm × 6400 mm variable speed apron feeder into a 1070 mm × 1400 mm single toggle jaw crusher operating with a CSS of 130 mm. The primary crusher product (P_{80} = 108 mm) will discharge onto the 900 mm mill feed conveyor belt, fitted with a weightometer, which in turn will discharge into the SAG mill containing a nominal ball charge of 8-12% and a total charge of 26% of mill volume. High pressure water spray heads on all feeder discharge point, will provide dust suppression. The SAG mill (1,500 kW) will operate in open circuit and the ball mill (2,800 kW) grinding in closed circuit, with a recycle pebble crusher, to produce a P_{80} grind size of 106µm.

Eight 640 mm hydrocyclones will be used for the fine classification duty around the ball mill. The hydrocyclones are designed to operate with an overflow density of 30% w/w solids for direct feed to the flotation circuit. Coarse particles will pass through the cyclone spigots as a dense slurry and gravitate to the ball mill for further grinding.

25.2.3.2 Flotation

The ground and classified ore will be treated in a rougher/scavenger flotation circuit consisting of seven 100 m^3 tank cells in series, and a cleaner flotation circuit consisting of three banks of cleaners with a total of seven cells. The flotation feed, flotation concentrate and scavenger tails streams will be monitored by online analysis for sulphur grade. The results of these measurements will be transmitted to the SCADA system and will allow calculation of plant recovery, warnings and trend displays.

The scavenger flotation tailings will be pumped to the 12 m high-rate tails thickener. The rougher concentrate will be combined with cleaner concentrate and will be thickened in a single 12 m high-rate concentrate thickener at a flocculant dosage of 10 g/t. Thickened flotation tailings slurry at the required density will be pumped from the bottom of the thickener into the dilution hopper where it is diluted with sea-water then pumped to the roaster off-gas scrubbing circuit where it will assist in neutralising the roaster off-gases. Thickener overflow will gravitate to the process water dam.

Thickened concentrate underflow slurry will be pumped to the concentrate storage tank. The storage tank is an agitated tank designed to hold six hours storage capacity of concentrate at the average production rate. The tank will operate with a varying slurry level to allow for storage during peak production periods and smoothing out of the treatment rate for the downstream filtration equipment.

Thickened slurry will be pumped from the base of the storage tank to the concentrate filter. The concentrate filter will be a chamber-style membrane filter press fitted with 60 chambers, and equipped with hydraulic pressing, water squeeze and air blow cake drying cycles. The filter will operate with the following cycles; fill, press, squeeze, blow, discharge. The typical cycle time for the filter will be15 minutes, which allows the filter to operate four cycles per hour. Filtrate water discharging from the chamber press will be recycled back to the concentrate thickener.

Concentrate discharging from the filter will drop under gravity directly onto a variable speed conveyor below the filter, from where it can either feed a bucket-type conveyor that supplies the roaster or it can drop into a bunker with 24 hours storage capacity for later reclaim.

25.2.3.3 Concentrate Roasting

Concentrate feed will be received at the Reversible Belt Conveyor, which alternately fills the Feed Surge Bins. Variable Speed Weigh Belt Feeders, discharge and meter concentrate feed to two Slinger Feeders. The Slinger Feeders will be high-speed belt feeders designed to "throw" filter cake feed into the FluoSolids Roaster and spread the feed across the cross-sectional area of the bed surface. Preheated luidising air will be supplied to the roaster by the Fluidising Air Blower. A roaster preheat system, consisting of two Preheat Burners, Preheat Burner Air Blower, and Preheat Burner Fuel Oil Pump will be used to preheat the roaster during start-up to a temperature sufficient to allow safe introduction of sulphide concentrate feed into the roaster. During operation, the roaster will be operated with a slight negative pressure in the freeboard to ensure that gases and hot solids are not discharged into the plant area through the open feed ports.

Dust laden roaster exhaust gases will pass through two Primary Cyclones for primary dust capture. The exhaust gases from the cyclone will be cooled in the Quench Tower by evaporation of water, prior to introduction of the gases into the Electrostatic Precipitator (ESP). The temperature of gases entering the ESP will be maintained above the dew point to prevent condensation of sulphuric acid on the internal surfaces of the ESP and duct shells.

Solids discharged from the bed overflow nozzles of the Roaster, Primary Cyclone solids discharge lines, and from the ESP via the ESP Drag Conveyor and ESP Solids Double Flap Valve will be quenched with water in Quench Tanks. The combined Calcine slurries from the quench tanks (approximately 10% solids) will be collected in the Quench Slurry Launder.

Roaster exhaust gas, at the gas discharge of the ESP will be sent to the Gas Scrubbing System for removal of acid gases and residual particulates.

25.2.3.4 Gas Scrubbing

A simplified explanation of the process is that the sulphur dioxide (SO_2) gas, which dissolves in the re-circulated slurry, reacts with calcium carbonate present in the tailings stream to form calcium sulphite hemi-hydrate ($CaSO_3.½H_2O$) according to the following reaction:

$$SO_2 + CaCO_3 + ½H_2O \rightarrow CaSO_3.½H_2O + CO_2$$

The tail-gas scrubbing system will comprise a primary and secondary scrubber, each with two stages of reverse-jet nozzles which promote efficient gas/liquid contact. Hot ESP treated gas, at approximately 350°C, will enter the top of the primary scrubber inlet barrel and collide with the circulation liquor, creating a highly efficient environment for simultaneous gas quenching, particulate scrubbing and SO_2 absorption.

From the contacting zone, the gas/liquor mixture will enter a separation vessel. The liquor will drop into the sump of the vessel (reaction tank) and the gas will exit the vessel and pass into the secondary scrubber, where the process will be repeated.

The sumps of the disengagement vessels will act as reaction tanks. The tailings stream will be added to the secondary scrubber sump, overflow from the secondary into the primary scrubber sump and be discharged from the primary scrubber sump under level control. The vessel sumps will be sized to allow maximum utilisation of the calcium carbonate present in the tailings.

The treated gases will pass from the secondary scrubber and will be discharged to atmosphere via a stack.

25.2.3.5 Calcine Neutralisation and Regrind

Quenched calcine will be pumped to the neutralisation circuit where lime slurry is added to neutralise acids formed during roasting. The calcine neutralisation circuit will consist of two 250 m³ (live) leach tanks in series interconnected with launders to allow the slurry to flow by gravity through the tank train. Neutralised calcine will gravitate to the calcine thickener. The calcine thickener is a high-rate thickener designed to achieve a high underflow density. Thickened calcine slurry at the required density will be pumped from the bottom of the thickener into the calcine regrind mill. Thickener overflow gravitates through a cooler tower to the calcine water dam.

Thickened calcine will be pumped to the calcine regrind circuit. The calcine regrind circuit will incorporate a ball mill operating in closed circuit to produce P_{80} passing 30 µm product. Reground calcine will be pumped to the calcine leach.

25.2.3.6 Calcine Leaching and Gold Recovery

The reground calcined slurry, discharged from the quench slurry launder will be pumped to downstream processing for gold recovery by a conventional carbon-in-pulp (CIP) circuit consisting of four leach tanks (725 m^3) followed by six adsorption tanks (100 m^3), giving a leach residence time of 94 h at 50% w/w solids pulp density. In the adsorption tanks, an average carbon concentrations between 10 g/L and 15 g/L will be used to ensure high gold adsorption efficiencies and low solution tails.

Barren carbon returning to the CIP circuit from the oxide carbon regeneration kiln will be screened over a vibrating screen to remove fine carbon, which is discarded to the tailings hopper.

Sodium cyanide solution will be metered into leach tanks 1, 2 and 4 via a ring main system. Compressed air will be injected into the leach tanks, to provide oxygen for the cyanidation reaction.

Tailings slurry from the last CIP tank will gravitate to the carbon safety linear screen to recover any carbon which may be present in the event of damage, wear or incorrect installation of the final stage inter-stage screen. Carbon recovered on the screen will be delivered to a bulk bag for re-use. Tailings discharging from the carbon safety screen will gravitate to the tailings tank and will be pumped to the tailings mixing tank. .

Mixed tails slurry will deposited underwater via a gravity pipeline for DSTP.

The existing oxide plant AARL elution will be used for recovering gold from loaded carbon.

25.2.4 Process Production Forecasts

25.2.4.1 Historical

Table 25-3 shows the production summary for the current oxide plant for the period June to September 2010.

Table 25-3: Production Summary

	Units	June 2010 to September 2010 Actual	June 2010 to September 2010 Actual Annualised
Mill Throughput	dry tonnes	765,824	2,300,000
Au Recovery	%	92.3	92.3
Operating Cost	$/t	11.15	11.15

This demonstrates the capacity of the plant to treat oxide ore at greater than 2 Mtpa capacity and to achieve recovery of greater than 92% on oxide ores.

Operating cost for the process plant for the four-month period was $11.15/t.

Annualised G&A cost based on the four-month period was $20.1 M.

25.2.4.2 Future Oxide Capacity – Expanded Plant

In May 2009, Allied commissioned GRES to undertake an evaluation of the process engineering options to expand the process plant capacity to 3 Mtpa and to provide capital cost estimates for the expanded process facilities and construction.

Allied instructed that the process design and major equipment selection was to include a throughput allowance of 20%, giving a maximum treatment rate of 3.6 Mtpa.

This case is now referred to as the 3.5 Mtpa oxide plant expansion.

Key criteria used for financial modelling purposes are shown in Table 25-4.

Table 25-4: Basis for Financial Modelling

Criteria	Units	Expansion
Annual Throughput	dry t/a	3,500,000
Average Gold Grade	g/t Au	1.01
Gold Recovery – Oxide Ore	%	91.3
Gold Recovery – Transition Ore	%	71.0
Gold Recovery – Sulphide Ore	%	65
Overall Average Gold Recovery	%	86.1

Production profile for the oxide project is shown in Figure 25.7.

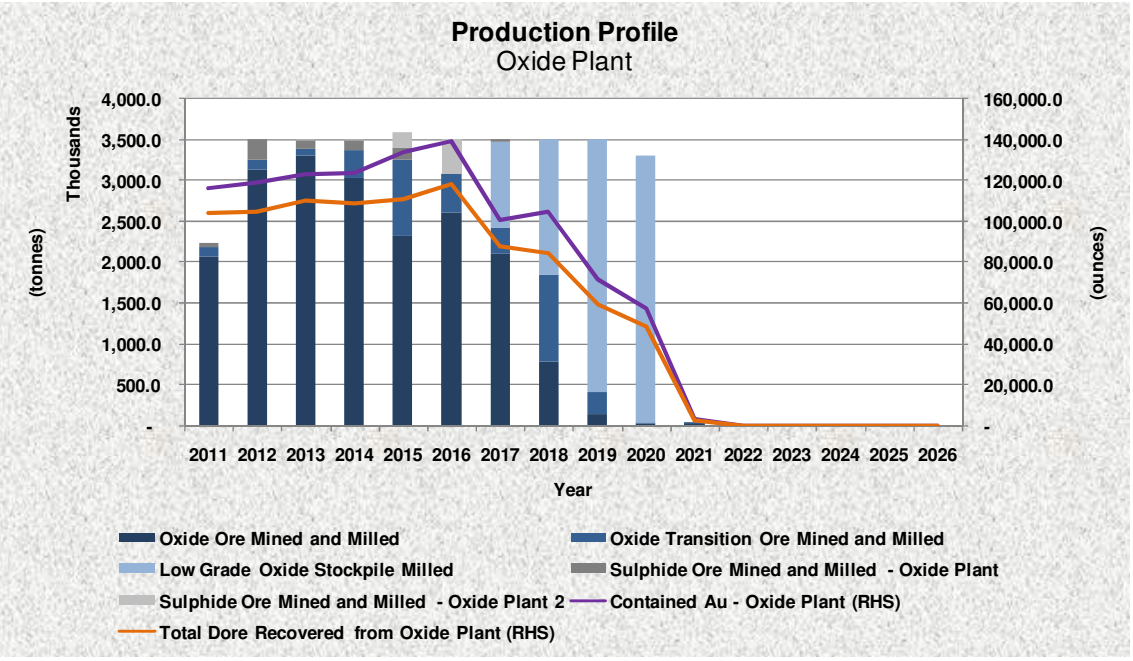

Figure 25.7: Production Profile

25.2.4.3 Sulphide Project

During 2010, BatteryLimits has proceeded with the metallurgical testwork and process design for a 1.0 Mtpa sulphide project at Simberi. The testwork, as described in Section 18.0 of this report, has focussed on flotation, roasting and CIL to produce gold dore. GRES undertook engineering and capital cost estimating for a 1.0 Mtpa and then 1.5 Mtpa sulphide processing project based on the process design and equipment selection provided by BatteryLimits.

The process description is provided in Section 25.2.1.

The sulphide project is based on established technology of flotation to produce a sulphide concentrate, and roasting of the concentrate to free the gold for conventional cyanide leach extraction and recovery.

Key criteria developed for the sulphide project at PFS stage are shown in Table 25-5.

Table 25-5: Key Criteria for Sulphide Project

Parameter	Value
Plant Throughput	1.5 Mtpa
Feed Grade Gold	2.41 g/t
Recovery	83.3%

Production profile for the sulphide project is shown in Figure 25.8.

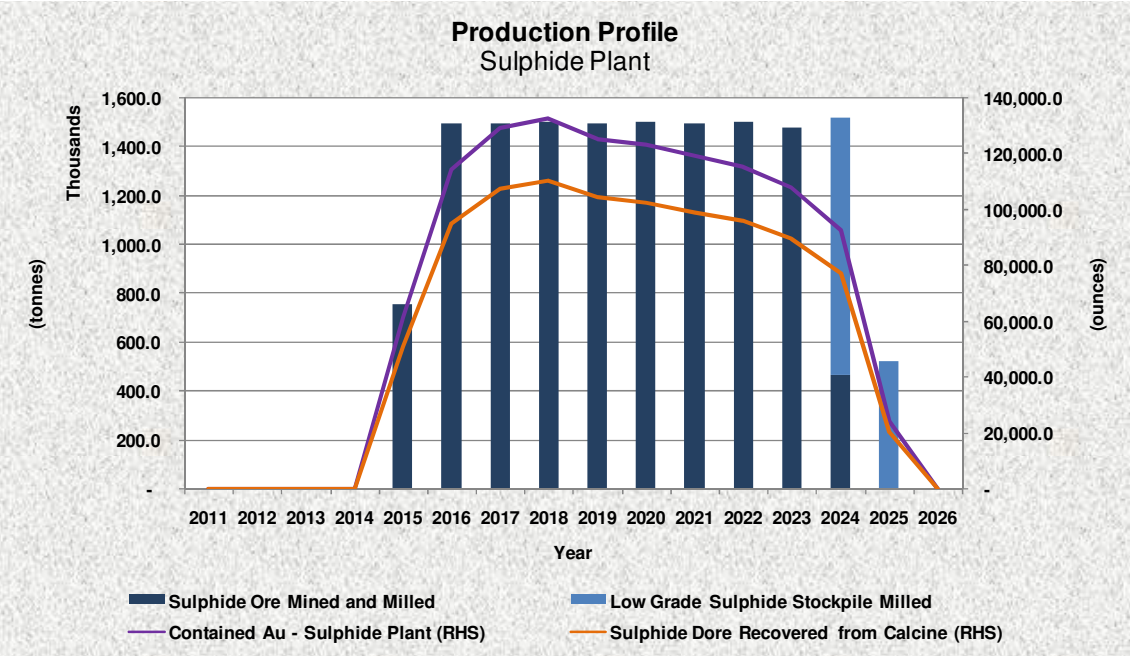

Figure 25.8: Production Profile for the Sulphide Project

25.2.4.4 Combined Oxide/Sulphide Project

Key criteria for the combined 3.5 Mtpa oxide project and 1.5 Mtpa sulphide project at PFS stage are shown in Table 25-6.

Table 25-6 Key Criteria for combined 3.5 Mtpa oxide project and 1.5 Mtpa sulphide project

Parameter	Value
Plant Throughput	5.0 Mtpa
Feed Grade Gold	1.43 g/t
Recovery	84.7%

Production profile for the sulphide project is shown in Figure 25.9.

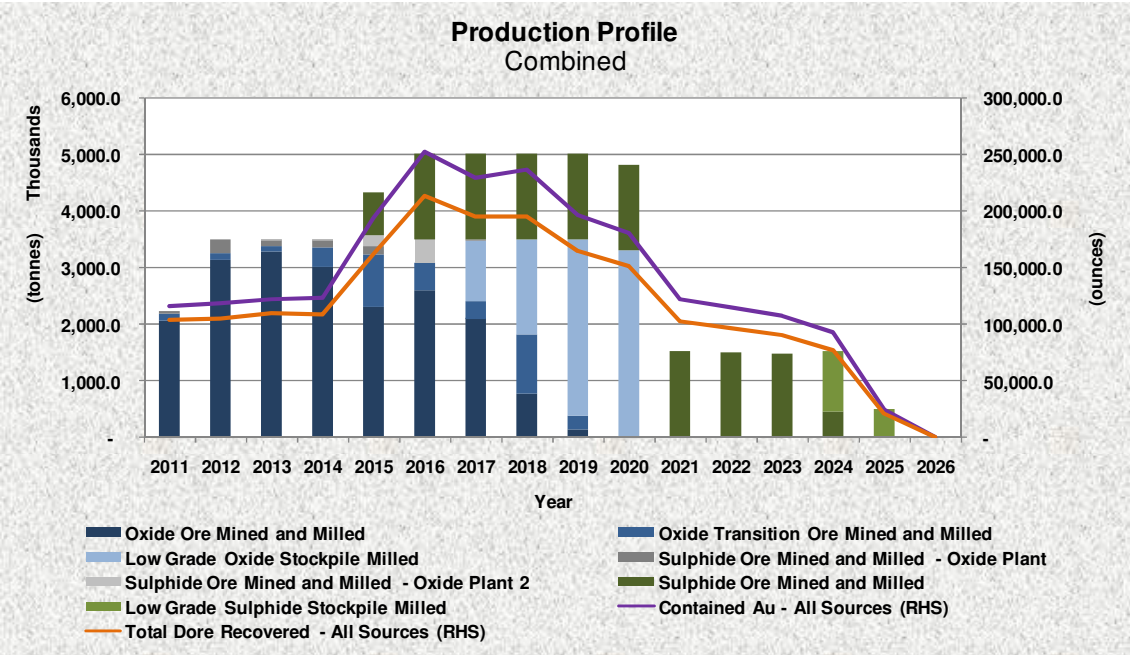

Figure 25.9: Combined Production Profile

25.3 Markets

Simberi Gold Company Limited sells its gold in the international commodities markets.

Simberi Gold does not have any long term contracts in place for the sale of gold.

25.4 Contracts

Allied Gold has had no gold hedging in place since March 2010. It has no FX hedging in place.

25.5 Environment

25.5.1 Introduction and Debottlenecked Case

Currently the Simberi Gold Mine is permitted to mine and process oxide ore at a nameplate capacity of 2 Mtpa with a nominal production capacity of 85 000 oz/a. This covers the debottlenecked case and no additional permitting is required for this case.

25.5.2 Combined Oxide and Sulphide Expansion Case

Additional permitting will be required to cover the expanded 3.5 Mtpa oxide and the 1.5Mtpa sulphide case. In February 2010, SGC sought approval from the PNG government to increase oxide ore production to a nominal rate of 3 Mtpa (to a peak throughput of 3.6 Mtpa).

Proposed relevant changes to achieve 3.5 Mtpa oxide and 1.5 Mtpa sulphide are:

- Tailing discharge to an annual rate of 6.7 Mm^3/a, and abstraction of sea water for dilution of tailing of 53 m^3/a.
- Abstraction of additional process water from the current 500 m^3/h to about 825 m^3/h, to achieve the 3.6 Mtpa plant throughput.
- Additional process equipment, including scats crushing circuit, two additional leach tanks, a SAG mill, thickener, roaster and off-gas stack.
- Utilisation of Heavy Fuel Oil (HFO) for power generation, due to an increase in power draw from 3 MW to 14 MW and additional fuel storage on site.

An increase in ore mining from 2.2 Mtpa to 3.5 Mtpa in addition to a 1.5 Mtpa sulphide plant, will result in:

- Bringing waste forward from the Sorowar deposit and will require approval for waste disposal by means of ocean disposal or alternatively by construction of a waste rock storage emplacement.
- Expansion of the Pigiput pit footprint from about 2 ha to about 8 ha.
- Upgrading the Pigiput to Sorowar mine access road to a haul road.
- Scrubbing of the roaster off gas to remove volatised As and sulphur dioxide for disposal in the tailings stream and off gas stack.

As part of the 3.5 Mtpa expansion, it is proposed to install a tailings thickener to recycle water, lime and cyanide. The thickener will also avoid the need to expand or duplicate the Deep Sea Tailings Placement (DSTP) system. The thickened tailings (at 3.5 Mtpa) will deliver approximately the same volume of thickened tailings slurry to the DSTP mix tank as is currently delivered from unthickened tailings from the existing 2 Mtpa plant. It is expected that the thickener will enable the existing pre-discharge dilution ratio of 8:1 (seawater to tailings slurry by volume) to be maintained. If so, there should be no change to the designated DSTP mixing zone. In fact, increasing the solids in the feed to the DSTP system will increase the density of combined mixture of tailing and seawater at the outfall terminus. This will cause the tailing density current to be more coherent for a greater distance down slope and less material should be lost to subsurface tailing plumes.

In June 2010, EBA Engineering Consultants (formerly known as Hay & Company Consultants) investigated the expansion of the DSTP system to include the oxide and sulphide project. The EBA report found that if the production rate is greater than 6 Mtpa, the discharged velocity in the outfall pipe will exceed 4.5 m/s, which could lead to hydraulic instability in the outfall pipe. EBA recommended that a separate DSTP system be considered in order to accommodate the additional tailings slurry if the production rate exceeds 6 Mtpa. EBA's analysis of the mix tank showed that when the production is less than 6 Mtpa, additional seawater will need to be pumped into the mix tank to prevent air ingestion in the outfall pipe.

The potential for acid rock drainage will also need investigating as part of further studies.

25.5.3 Permits

The environmental permits required for operation were discussed in Section. Conditions 28 and 29 of Environment Permit WDL3(36) require that a Waste Management Plan to be submitted to the Department of Environment and Conservation ("DEC"). A waste management plan was prepared by Enesar Consulting Pty Ltd (September 2006). The plan has been submitted to the DEC and reviewed by the Author.

The Sulphide Project will require specific approval under the Environment Act 2000. SGC will negotiate with DEC on the requirements for seeking approval for the Sulphide Project. At worst, this may require an Environmental Impact Statement (EIS) to be prepared.

25.5.4 Auditing

In 2008 Coffey Natural Systems undertook a compliance monitoring audit. This report showed compliance in most areas, with any impacts they are consistent with predictions made in the Environmental Plan submitted with the original mining lease application. Coffey recommended focusing attention on controlling erosion and sedimentation upstream in order to reduce the impacts on the nearshore fringing coral reef and on the streams and rivers around Simberi Island.

25.6 Taxes

On the 13th April 2005 the Simberi Gold Company Limited was granted a zero tax rating status with respect o GST on local purchases, service contracts or at the point of entry of import (excluding cars) (Sode, 2005).

Simberi is subject to import duties on certain items, however this is assessed on a case by case basis.

It is subject to company corporate tax at the rate of 30%, however has not as yet been in a tax paying situation.

There are no payroll taxes in PNG.

There are no material other taxes that are levied on Simberi Gold Ltd.

25.7 Capital Cost Estimates

25.7.1 Summary

Capital cost estimate summary for the combined 3.5 Mtpa oxide and 1.5 Mtpa sulphide project is shown in Figure 25.10.

Capital Expenditure Assumptions	31 Dec 2011 (A$)	31 Dec 2012 (A$)	31 Dec 2013 (A$)	31 Dec 2014 (A$)	Toal (A$)
Mining Fleet Tranche 1	$ 23,249,072	$ -	$ -	$ -	$ 23,249,072
Mining Fleet Tranche 2	$ -	$ 2,168,320	$ -	$ -	$ 2,168,320
Mining Fleet Tranche 3	$ -	$ -	$ 10,230,000	$ -	$ 10,230,000
Mining Fleet Tranche 4	$ -	$ -	$ -	$ 7,230,300	$ 7,230,300
Sulphide Plant	$ -	$ -	$ 94,142,500	$ 94,142,500	$ 188,285,000
Oxide Throughput Upgrade	$ 32,000,000	$ -	$ -	$ -	$ 32,000,000
Haul Road	$ 3,600,000	$ 3,600,000	$ -	$ -	$ 7,200,000
HFO Power	$ -	$ 7,570,000	$ -	$ -	$ 7,570,000
Total	**$ 58,849,072**	**$ 13,338,320**	**$ 104,372,500**	**$ 101,372,800**	**$ 277,932,692**

Figure 25.10: Combined Oxide and Sulphide Project Capital Cost

25.7.2 Mining Fleet

Mining fleet costs have been estimated by Golder. Haul road cost construction has been supplied by Allied Gold and verified to PFS level of accuracy by BatteryLimits based on Allied Gold's experience at Simberi Island.

25.7.3 3.5 Mtpa Expanded Oxide Processing Plant

The capital cost for the 3.5 Mtpa processing plant expansion has been developed by GRES to an accuracy of ± 20%.

The capital cost estimate for the design, supply, fabrication, transport, construction and commissioning of the process facilities was reported by GRES in October 2009 as A$30.67 million. This has been updated to A$32 M for this current cost estimate. Breakdown of the cost estimate is shown in Table 25-7.

Table 25-7: Capital Cost Estimate Breakdown

Area	Materials A$	Labour hours	Labour A$	Freight A$	Contingency A$	Total A$
Direct Costs						
Earthworks	275,000	0	0	0	34,375	309,375
Civil works	676,223	21,871	1,350,073	184	185,904	2,212,384
Buildings	0	0	0	0	0	0
Mechanical equipment	9,804,463	11,411	2,420,348	377,805	1,059,454	12,041,834
Platework	1,597,852	7,403	703,243	95,790	257,780	2,654,665
Structural steel	1,947,986	10,642	769,064	212,182	310,843	3,250,210
Piping	658,273	4,597	321,435	34,734	134,799	1,149,240
Electrical installations	1,555,417	7,848	570,671	40,000	162,457	2,328,545
Construction equipment	0	0	0	0	0	0
Total Direct Costs	**16,515,215**	**63,773**	**6,134,834**	**760,695**	**2,145,612**	**23,946,254**
Indirect Costs						
Temporary construction facilities	322,280	924	97,265	56,400	44,590	520,535
Mobilisation and demobilisation	418,200	0	0	0	41,820	460,020
Project and procurement management	74,750	10,035	1,271,675	0	102,851	1,449,276
Engineering design	0	11,170	1,503,805	0	112,785	1,616,590
Site supervision	0	15,825	1,449,375	0	108,703	1,558,078
Commissioning	28,000	3,859	520,375	0	41,828	590,203
Owners Costs	0	0	0	0	0	0
Spare Parts & First Fills	412,700	0	0	75,000	48,770	536,470
Total Indirect Costs	**1,255,930**	**41,813**	**4,842,495**	**131,400**	**501,347**	**6,731,172**
TOTAL CAPITAL ESTIMATE (±20%)	**17,771,145**	**105,586**	**10,977,329**	**892,095**	**2,646,959**	**30,677,426**

Source: Simberi Gold Project 3 Mtpa Expansion Study, October 2009

25.7.4 1.5 Mtpa Sulphide Processing Plant

The capital cost for the 1.5 Mtpa expansion has been developed by GRES to an accuracy of ± 25%.

The capital cost estimate for the design, supply, fabrication, transport, construction and commissioning of the process facilities was estimated as A$195.8 M including A$7.57 M for HFO infrastructure facilities.

An area-by-area breakdown is shown in Table 25-8.

Table 25-8: Summary of Capital Cost Estimate

	Base Cost A$	Contingency A$	Total A$
AREA 310 – SAG Feed and Pebble Crushing	6,155,010	591,991	6,747,000
AREA 330 – Grinding & Classification	14,466,633	1,474,895	15,941,527
AREA 340 – Flotation	7,051,518	708,978	7,760,497
AREA 350 – Concentrate Thickening & Filtration	4,562,388	462,045	5,024,433
AREA 360 – Fluosolids Roaster System	33,874,179	3,455,632	37,329,811
AREA 370 – Tail Gas Scrubbing	17,671,369	1,781,790	19,453,159
AREA 380 – Neutralisation, Calcine Thickener & Regrind	3,549,815	365,176	3,914,991
AREA 390 – Concentrate Leach	3,180,568	379,112	3,559,680
AREA 391 – Concentrate Adsorption	2,456,105	273,676	2,729,781
AREA 392 – Reagents	1,463,122	155,092	1,618,214
AREA 393 – Sea & Fire Water Area	1,997,045	224,024	2,221,069
AREA 401 – Process & Raw Water Area	474,408	53,983	528,391
AREA 339 – Plant Piping	11,535,841	1,153,584	12,689,425
AREA 370 – Power & Reticulation	11,124,847	1,113,296	12,238,143
AREA 400 – Tailings Thickening & Disposal	2,421,173	242,140	2,663,313
AREA 420 – Air Compressors	734,845	79,339	814,184
AREA 382 – HFO Capital Inc Wharf Piling & LCT Vessel	6,881,810	688,181	7,570,000
AREA 430 – Plant Buildings	338,714	35,366	374 080
AREA 440 – Workshop/Stores	1,385,127	150,967	1,536,094
AREA 450 – Plant Site Bulk Earthworks	4,487,630	452,963	4 940 593
AREA 640 – Accommodation Camp	1,598,663	165,446	1,764,109
AREA 460 – Fencing/Civil Trades	94,721	9,472	104,193
AREA 470 – Fuel Stores Extension	1,063,291	110,315	1,173,605
AREA 500 – Engineering	14,345,485	1 442,829	15,788,314
AREA 510 – Commissioning	1,689,901	168,990	1 858,891
AREA 550 – Initial Fills	4,199,533	419,953	4,619,487
AREA 560 – Insurance Spares	3,274,800	327,480	3,602,280
AREA 804 – Contractors Equipment	4 589,369	458,937	5,048,305
AREA 600 – Temporary Facilities	3,227,800	322,780	3,550,580
TOTAL CAPITAL ESTIMATE (±25%)	**178,050,000**	**17,805,000**	**195,855,000**

Source: GRES Simberi Gold Project 3 Mtpa Expansion Study, August 2009

25.8 Operating Cost Estimates

Operating cost estimates vary from year to year according the total material movements of ore and waste mined. The financial model considers total material movement on a year-by-year basis. Costs are based on the assumptions as set out in the following sections.

25.8.1 Mining

Mining costs are based on the assumptions outlined in Table 25-9.

Table 25-9: Mining Costs

Parameter	Cost Estimate/t Material
Oxide Ore Mined	$3.26/t
Oxide Waste Mined	$3.26/t
Sulphide Waste Mined	$3.83/t
Sulphide Waste Mined	$3.83/t

In years when total mining volumes (ore and waste) exceed 12.5 Mtpa, it is assumed that contract mining and haulage arrangements are utilised. Additional cost for contract haulage on top of the mining cost estimates in Table 25-9 of $1.70/t has been estimated, that includes cost of haulage to the coast for ore and waste.

In addition to the mining expenditure rates for the mining of ore, ore rehandling expenditure of A$1.00 per tonne of ore is assumed to be incurred at the time of processing the low grade stockpiles.

25.8.2 Expanded 3.5 Mtpa Oxide Processing Plant

BatteryLimits has reviewed operating cost of the current processing plant. Based on the four-month period June to September 2010, average operating cost is $11.15/t. BatteryLimits has applied fixed and variable cost ratios to the operating costs, and has estimated oxide plant processing operating cost at $9.25/t for the 3.5 Mtpa expanded case.

25.8.3 1.5 Mtpa Sulphide Processing Plant

Operating costs for the sulphide processing plant have been developed from first principles. Sulphide plant operating cost is estimated at $23.29/t. Operating cost assumptions are shown in Figure 25.11.

25.8.4 G&A Costs

G&A costs have been estimated at $13.65 M/a for the oxide plant and a further $3 M/a for the sulphide plant. Current G&A costs are around $20 M/a, but this is expected to reduce as the operation matures.

25.9 Financial Assessment

25.9.1 Financial Model Basis

An advanced financial model has been developed by Modus Capital for BatteryLimits for the financial evaluation. Inputs and assumptions are shown in Figure 25.11 for the NPV.

High-level Assumption Summary		Scenario 1	Scenario 2
		Base Case - 3.5 Mtpa Oxide & 1.5 Mtpa Sulphide	Additional Scenario - 3.5 Mtpa Oxide & 2.5 Mtpa Sulphide
Capital Expenditure			
Mining Fleet Tranche 1	(A$ M)	$ 23.2	$ 23.2
Mining Fleet Tranche 2	(A$ M)	$ 2.2	$ 2.2
Mining Fleet Tranche 3	(A$ M)	$ 10.2	$ 10.2
Mining Fleet Tranche 4	(A$ M)	$ 7.2	$ 7.2
Sulphide Plant	(A$ M)	$ 188.3	$ 255.8
Oxide Throughput Upgrade	(A$ M)	$ 32.0	$ 32.0
Haul Road	(A$ M)	$ 7.2	$ 7.2
HFO Power	(A$ M)	$ 7.6	$ 7.6
Total	(A$ M)	**$ 277.9**	**$ 345.5**
General Assumptions			
Exchange Rates	(A$:US$)	0.8000	0.8000
Gold Price	(US$ per oz.)	$ 1,000.00	$ 1,000.00
Gold Price	(A$ per oz.)	$ 1,250.00	$ 1,250.00
Oxide Credits	(% of Gross Oxide Rev.)	0.1700%	0.1700%
Dore RC	(A$ per oz.)	$ 0.69	$ 0.69
Taxation Rate	(%)	30.0%	30.0%
Working Capital - Revenue Days	(days)	45.0	45.0
Working Capital - Expenditure Days	(days)	45.0	45.0
Operating Expenditure			
Sulphide Ore - Mining	(A$ per dmt)	$ 3.83	$ 3.83
Oxide Ore - Mining	(A$ per dmt)	$ 3.26	$ 3.26
Sulphide Waste - Mining	(A$ per dmt)	$ 3.83	$ 3.83
Oxide Waste - Mining	(A$ per dmt)	$ 3.26	$ 3.26
Contract Mining Operating Expenditure Profit Margin	(%)	17.2%	17.2%
Contract Mining Capital Equipment Charge	(A$ per dmt)	$ 1.02	$ 0.58
Average Contract Mining Cost	(A$ per dmt)	$ 1.70	$ 1.28
Contract Mining Mobilisation Cost Rate	(A$)	$ 500,000.00	$ 500,000.00
Contract Mining De-mobilisation Cost Rate	(A$)	$ 500,000.00	$ 500,000.00
Handling Expenditure - Oxide Stockpile	(A$ per dmt)	$ 0.90	$ 0.90
Mean Power Cost - Sulphide Plant	(A$ per Kw/h)	$ 0.2325	$ 0.2325
Power Consumption - Sulphide Plant	(kw / p.a. @ 1.5 Mtpa)	61,315,868	61,315,868
Oxide Power HFO Saving	(%)	33.6%	33.6%
Oxide Processing Expenditure Power Component	(%)	40.0%	40.0%
Sulphide Process - Plant Head Count	(FTE)	71.0	71.0
Sulphide Process - Labour Cost	(A$ p.a. @ 1.5 Mtpa)	$ 2,471,385	$ 2,876,853
Sulphide Process - Maintenance Materials	(A$ p.a. @ 1.5 Mtpa)	$ 3,480,053	$ 3,480,053
Sulphide Process - Reagents, Consumables and Services	(A$ per t ore)	$ 9.5671	$ 9.5671
Sulphide Process - Miscellaneous Processing Costs	(A$ p.a. @ 1.5 Mtpa)	$ 200,000	$ 200,000
Oxide Process - Miscellaneous Processing Costs	(A$ per t ore)	$ 9.25	$ 9.25
Sulphide Dore Transport	(US$ per pa)	$ 120,000	$ 120,000
Haul Road Haulage Cost - Owned Fleet	(US$ per DMT, real)	$ 0.40	$ 0.40
Additional Contract Haulage Operating Cost	(A$ per DMT, real)	$ -	$ -
Additional Contract Haulage Mobilisation Cost	(A$)	$ -	$ -
G&A, Transport and Infrastructure - Sulphide	(A$ per pa)	$ 3,000,000	$ 3,000,000
G&A, Transport and Infrastructure - Oxide	(A$ per pa)	$ 13,650,000	$ 13,650,000
Processing Labour Fixed Cost	(%)	75.00%	75.00%
Processing Maintenance Materials Fixed Cost	(%)	50.00%	50.00%
Royalty and Mining Levy	(%)	2.25%	2.25%
Contingency	(%)	0.00%	0.00%
Material Handling Assumptions			
Existing Rope Conveyor Capacity	(tonnes pa)	3,500,000.0	3,500,000.0
New Owned and Operated Truck Fleet Capacity	(tonnes pa)	9,000,000.0	9,000,000.0
Total	(tonnes pa)	**12,500,000.0**	**12,500,000.0**

Figure 25.11: NPV Inputs and Assumptions

25.9.2 Financial Model Outputs

Key outputs are shown in Table 25-10.

Table 25-10: Financial modelling outputs

Parameter	Value
NPV at 10% discount factor (pre-Tax)	$334 M
Cash Costs	$678/oz

Note that the NPV includes the value of the current 2 Mtpa oxide operation. IRR's for the combined expanded oxide and sulphide project have not been reported because BatteryLimits considers IRR is not a valid financial performance indicator given of the contribution of the existing operation.

Figure 25.12 shows cumulative net cash flows for the combined project.

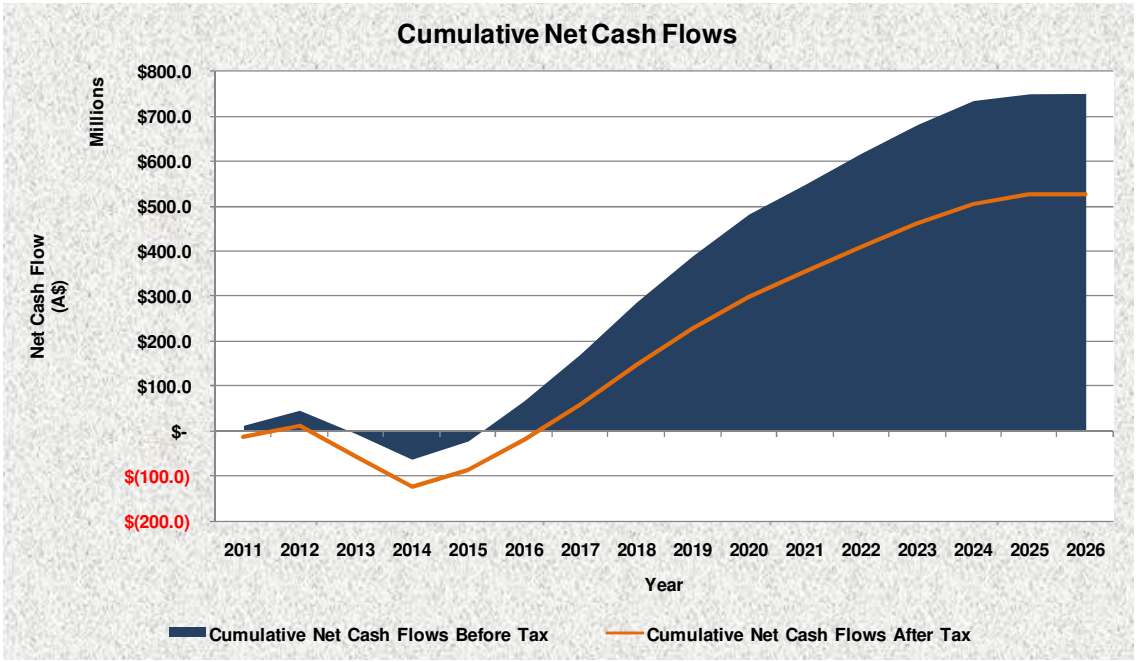

Figure 25.12: Cumulative Net Cash Flows

25.9.3 Sensitivity Analysis

The sensitivity of the financial performance of the production profile of the 3.5 Mtpa oxide and 1.5 Mtpa sulphide project to variations in key assumptions is detailed Figure 25.13 and Figure 25.14. The sensitivity of the project to variable gold price movements is shown in Figure 25.13. The sensitivity analysis indicates that the project is reasonably sensitive to variations in mining expenditure, processing expenditure (excluding power) and capital expenditure assumptions.

The analysis of the production profile's sensitivity to the US$ gold price indicates that the project is strongly leveraged to gold price movements. Due to the strong financial performance of the project, decreases in the US$ gold price of more than 20% can be sustained with the project maintaining a positive NPV @ 10% (nominal, before tax). This would indicate a favourable risk versus reward assessment of the project in terms of its exposure to the key variable of US$ gold prices. However, it must again be noted that the financial performance of the project includes the impact of cash flows from the existing oxide plant operation.

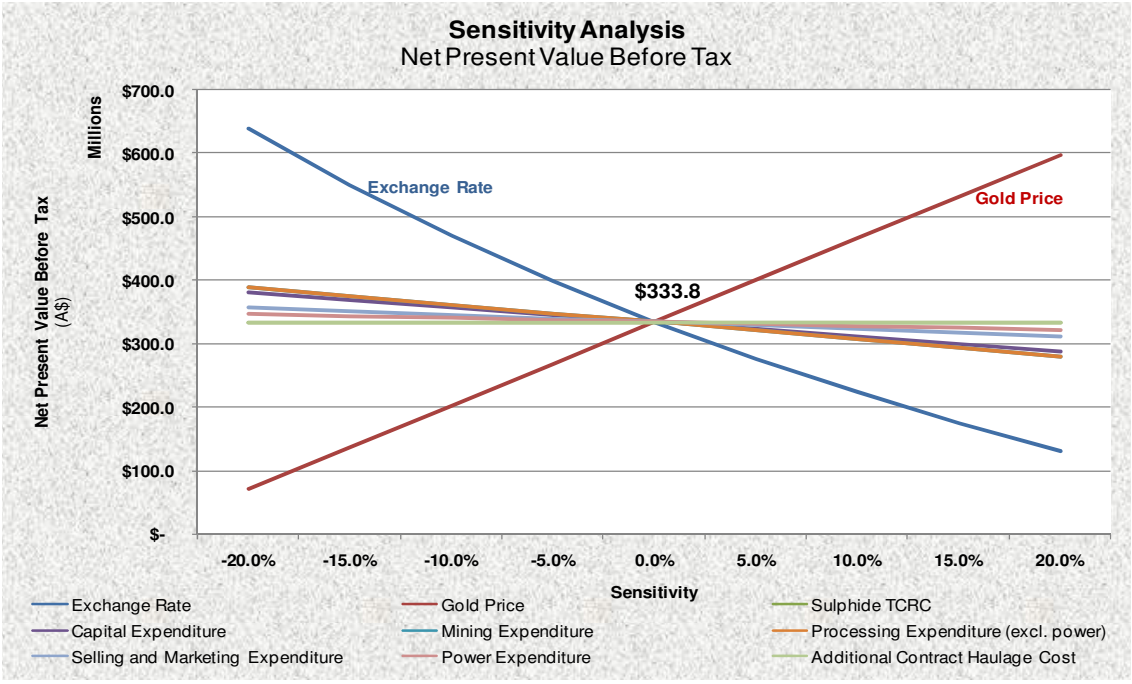

Figure 25.13: Sensitivity Analysis to variable gold price movements

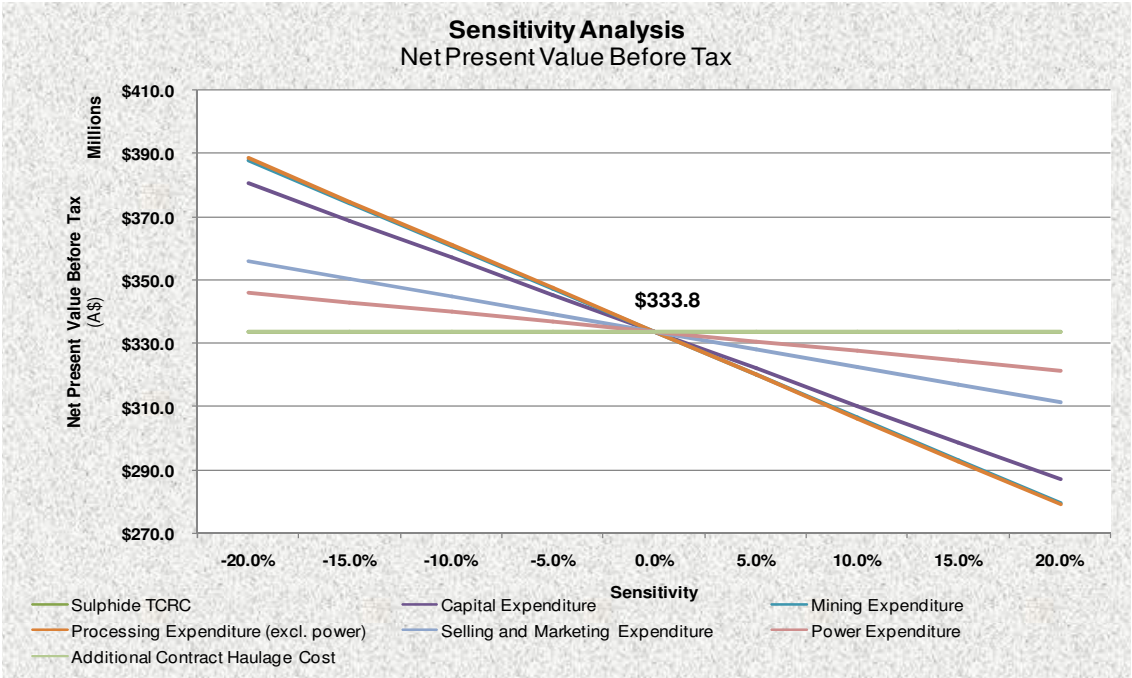

Figure 25.14: Sensitivity Analysis

25.10 Mine Life

Under the combined 3.5 Mtpa oxide and 1.5 Mtpa sulphide operating scenario, current reserves will support a 13-14 year mine life commencing 2012 as Year 1.

25.11 Payback

Payback for the combined project is estimated at 7 years from 2011 on an after tax basis.

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