FEASIBILITY STUDY

Snow Lake Mine Re-activation Project

Technical Report NI 43-101

Project No AV124441

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1. EXECUTIVE SUMMARY (ALL)

Property Location and Ownership

GENIVAR was retained by Alexis Minerals Corporation (Alexis) to conduct a feasibility study (FS) of the Snow Lake Mine (SLM) property, (formerly the New Britannia Mine) based upon the updated mineral resource estimate for No. 3 Zone, and Main Mine Upper (i.e. Above 1780 Level) and review of the October 27, 2006 mineral resource estimate of the Main Mine Lower. The property is located in west-central Manitoba at 54°52'44" north latitude and 100°02'12" west longitude, within the community of Snow Lake. The town of Snow Lake is situated approximately 685 kilometres (km) north of Winnipeg, the provincial capital of Manitoba.

Alexis initially acquired over 90% of the Garson Gold Corporation ("Garson") outstanding shares on January 15th, 2010. On April 29th, 2010 Alexis acquired all remaining Garson shares by way of a plan of arrangement. Accordingly Alexis acquired the New Britannia Mine (renamed SLM) through a subsidiary of Garson located in Snow Lake, Manitoba, the subject of this feasibility study.

The SLM property is now 100% owned by Alexis through subsidiary companies. Both mineral leases ML-61 and ML-323 are subject to a 1.38% NSR payable to Bruce Dunlop Ltd (NPL).

Geology and Mineralization

The mineral zones located on the SLM property are within the Aphebian Flin Flon/ Snow Lake Greenstone Belt. One of the main structural features of the Snow Lake area is the north-northwest trending McLeod Road Thrust fault which defines the western extent of the deposit. The auriferous zone within the past producing mine lies along a curvilinear shear zone named the Howe Sound fault (also referred to as the Nor-Acme fault). The main mineralization consists of quartz-carbonate alteration zones emplaced in a simple intercalated sequence of altered felsic and mafic volcanics and pyroclastics of the Amisk Group. The host rocks are altered and include varying proportions of quartz and carbonate (mainly calcite).

Mineralization at the mine site is found at surface over a strike length of 2,000 ft or approximately 600 metres (m) and occurs in two main zones known as the Dick and Toots, respectively. At depth, the Dick Zone appears to split into a series of separate zones. Four zones occur on the 1780 Level and are, from west to east, the Toots, Dick, Ruttan and Hogg Zones. Below the 1780 Level the Toots Zone is terminated by the McLeod Road Thrust fault and only the Dick, Ruttan and Hogg Zones continue at depth. The common sulphides in the deposit are arsenopyrite, pyrrhotite and pyrite.

Arsenopyrite constitutes about two per cent of the mineralization, pyrrhotite less than one per cent and pyrite abundance is less than pyrrhotite. Trace sulphides include chalcopyrite and sphalerite. The total sulphide content averages less than five

percent. Gold is predominately associated with arsenopyrite, especially where it occurs as a mesh of fine-grained needles. When the arsenopyrite occurs as anhedral grains or massive seams, the ratio of gold to arsenopyrite is less and the corresponding grade of the ore is lower.

Exploration

The Snow Lake Mine and surrounding properties contain a number of targets and target areas that are at various stages of development and exploration. These range from resource delineation and expansion targets to conceptual targets based on geological compilation and interpretation. The proposed exploration program has two main objectives:

- To evaluate and increase the gold resources within the strategic No. 3 Zone to Boundary Zone to Main Mine corridor and to incorporate these gold resources into further preliminary assessment and feasibility level studies. Also included in the first objective is the evaluation of historical resources at the Birch Zone and Margaret Zone.

- To continue to evaluate the gold potential of the entire property through the evaluation of medium to early stage exploration targets and implementation of earlier stage exploration activities to define new drill targets.

  Infill drilling to upgrade the current inferred resources was completed by Alexis from January 2010 to June 2010.

Development and Operations

In 1925, the mineral claims that form the basis of the SLM property were staked. These claims continued to be held by the original prospector until 1938 when they were sold into a newly incorporated company named Nor-Acme Gold Mines Limited (Nor-Acme Gold). In 1941, Nor-Acme Gold was able to option the claims to the Howe Sound Exploration Company Ltd. (Howe Sound) in exchange for a mining royalty. Over the next two years, a large, low-grade gold deposit was delineated and, after a period of inactivity during the war, construction of the mine infrastructure began in August, 1945.

In 1949, the mine was opened as the Nor-Acme mine and, from 1949 to 1958, produced approximately 5,394,000 tons at an average grade of 0.142 Oz/ton for a total of 763,000 feed Oz of gold. The mine closed in 1958 and it was not until 1995 that the mine was successfully re-opened.

TVX Gold Inc. (TVX) and High River formed a joint venture on an equal basis in 1994 and in November, 1995 the mine was reopened as the New Britannia Mine. Between 1995 and January, 2005, when the mine was placed on care and maintenance, a total of 6,480,266 tons at an average grade of 0.132 Oz/ton for a total of 858,075 feed Oz of gold were produced.

The last mineral resource estimate at the New Britannia mine was conducted in December 31, 2004. Although not the subject of a NI 43-101 Technical Report, these resources were reported by both Kinross and High River in their 2004 and 2005 Annual Reports. In its 2005 Annual Report, High River stated the following "The New Britannia Mine is on care and maintenance as of January, 2005. No diluted mineral reserves were estimated for December 31, 2005 because the mineral resources were unable to be translated into an economic mine plan."

In 1995 and 1996, the No. 3 Zone was mined as feed for the New Britannia mill and during this period No. 3 Zone produced 241,574 tons at a grade of 0.126 Oz/ton gold for a total of 30,335 feed Oz. Production ceased at No. 3 Zone in May, 1996 and no development or further exploration was performed below the last mining level (450 Level). After 1996, No. 3 Zone was allowed to flood and during 1998 and 1999 the No. 3 Zone adit was sealed and reclamation was conducted intermittently on the surrounding area until it was completed in 2004.

From 1995 to 1997, various exploration programs of line cutting, ground geophysical surveys, and diamond drilling programs were conducted on other showings on ML-323. The majority of the exploration funding during this period was devoted to exploring the extent of the Birch Zone located to the northwest of the New Britannia tailings pond. In early 1996, a small bulk sample was extracted from the upper portions of the Birch Zone using open pit mining methods. The 1996 production from the Birch pit amounted to 31,353 tons grading 0.104 Oz/ton gold for a total of 3,282 feed Oz.

All surface exploration programs covering mineral lease ML-323 were curtailed in 1998 and even with the partial recovery of metal prices in 2003 and 2004, exploration never recommenced on ML-323 prior to the closure of New Britannia in early 2005.

Preliminary Assessment

Prior to the acquisition of the SLM property by Alexis, a review was performed on both the Main Mine and #3 Zone resources. The Alexis team decided to proceed with the takeover and conduct a preliminary assessment on the entire property rather than focusing on just #3 Zone. Golder Associates (Golder) completed a Preliminary Assessment (PA) in April 2010 which showed a positive return on investment in a relatively short period of time. The infrastructure in place and the average gold price notwithstanding the Bloomberg consensus pricing suggested moving the project to a full feasibility level.

Mineral Resources

The estimated mineral resources at the SLM have been revised by Alexis. For No. 3 Zone and Main Mine Upper, an updated geological model was completed based on delineation drilling of these zones. Inferred resources included in the PA were either upgraded to indicated resources for inclusion in the reserve calculation or remained in the inferred category. The following presents the updated resource estimate at a cut-off grade of 1.95 g/t Au:

- Measured Resources: 7,000 t @ 4.76 g/t Au for 1,000 Oz;
- Indicated Resources: 5,464,000 t @ 4.14 g/t Au for 727,000 Oz and
- Inferred Resources: 2,367,000 t @ 4.43 g/t Au for 336,700 Oz

Production Schedule

Alexis prepared a new mining plan based on the 2010 Measured and Indicated Resource update and a 3D representation of the block resources. Three (3) mining methods were selected: based on the geometry of and access to the mineralized zones: longhole stoping with down holes in the Main Mine Upper, longhole stoping with upholes in No 3 Zone, and Mechanized Room and Pillar Mining in the Main Mine Lower.

Both the No 3 Zone and Main Mine Upper will provide the initial production following a 12 month pre-production period. The Main Mine Lower will be developed in time to continue to provide 2000 t per day to the mill when the No 3 Zone is completed.

The mining recovery varies from 85% to 95%, and this was estimated using historical mining records. The dilution was evaluated for each mining method and 36% dilution was used for longhole stoping in Main Mine Upper, 25% for longhole stoping in No 3 Zone and 25% for room and pillar stoping in Main Mine Lower. The thickness of the crown pillar was evaluated using existing geotechnical information and previous modelling. Based on projected costs and a Bloomberg gold price deck averaging US$ 1149/Oz, cut-off grades were evaluated for various mining methods. The mineable reserves are:

- Proven Reserves:7,000 t @ 3.81 g/t Au for 900 Oz; and
- Probable Reserves : 3,470,000 t @ 4.04 g/t Au for 451,000 Oz.

Development to access the various zones was planned using specialized software (Promine). A detailed scenario for the mine ventilation was also planned. Given the extensive capital development in place, the pre-production development is limited and will be completed during the 12-month preproduction period.

A detailed development and production schedule was prepared. Alexis plans to use contractors to refurbish the existing infrastructure and complete any specialized development. It then plans to hire its own personnel for development and mining. The mine will operate on a 7-day per week schedule, working two 10-hour shifts per day with only one day shift on the weekends. The planned production rate for the mine is 2000 t/day on 350 days/ year or 700,000 t annually. The development and production rates were taken from similar operations including the extensive historical data from the SLM from 1995 – 2005 and the Lac Herbin mine operated by Alexis.

Table.1.1: Preliminary Production Schedule

Year	Tonnes (t)	Grade (g/t)	Head Oz	Recovery %	Recovered Oz
Preprod.	46,200	4.42	6,565	91.5	6,004
1	632,369	4.07	82,753	92.6	76,620
2	737,176	4.35	103,000	92.6	95,337
3	711,280	3.91	89,357	93.7	83,746
4	716,683	3.84	88,425	94.0	83,120
5	633,656	4.01	81,661	94.0	76,761
Total	3,477,367	4.04	451,761	93.3	421,588

The surface facilities, including the headframe, mill, tailings management facility and the underground 320 Level were visited by the Qualified Persons. They concluded the infrastructure is in good condition and that the ground conditions in the upper level are also good. The mine hoist and shaft require inspection, testing and recertification. The mill was placed on care and maintenance in an orderly manner and can easily be restored to an operational condition. Some buildings have been reactivated to support the ongoing exploration activities. The main office and administration building is functional as of the time of writing. Detailed reviews of the infrastructure and price quotations were provided by contractors for costing of the refurbishment.

Pre-production requirements for the Main Mine are broken down into various working areas of the mine. This includes shaft rehabilitation, escape way rehabilitation, loading pocket rehabilitation as well as reconditioning the shaft stations and the reconditioning of main level drifts.

The 3 Zone rehabilitation project will commence in parallel with the pre-development work at the Main Mine. This strategy realizes early production at the Snow Lake Mine and has an immediate positive impact on the cash flow of the project.

Mill Processing

The process plant building is in good condition with power having been maintained to provide heat and light to the building during care and maintenance. The gears on both mills are well lubricated. All mill liners were removed at the time of shut down and the mills were jacked up off the bearings. The ball mill motor is non-operational and will require rewinding or replacement. The leach circuit, comprising 5 circular tanks

located outside the mill building are in good condition, although some of the lower agitators are worn or damaged and will require repair. One of the tanks contains previous material that will have to be cleaned out. Equipment in the CIP and gold recovery circuit is in good condition.

The mill infrastructure was surveyed and reviewed and capital estimates of CAN$ 2,565,000 were made for refitting of the mill, and operating costs of CAN$ 12.20/t were estimated. No provision for arsenic pre- or post-treatment, or cyanide post treatment was allowed.

New metallurgical test work has been conducted for this Feasibility Study. This test work has confirmed the historical recoveries in the Main Mine of 94% and determined that the No 3 Zone ore recoveries will be lower at 91%. The history of the present mill design, key design criteria from study reports and mill operating reports provided by Alexis Minerals are appropriate to produce the desired recoveries when operating.

The Tailings Management Area (TMA) at the SLM is located in the former Birch Lake basin. The original design for the TMA was prepared by DNE Knight Piesold (1995). The general TMA layout is a relatively long and narrow facility, oriented southeast to northwest. The initial construction of the tailings dams were undertaken in 1995, and the deposition of tailings into the TMA began in November 1995. Additional dam construction was undertaken in 1996, and the most recent dam construction involved raising of the Repap Road Saddle Dam crest by 2 m in 2002. Annual dam inspections have been made, and the dams are in good condition. The most recent inspection was carried out by Golder in 2008. The TMA was reviewed by Knight Piesold and it is determined that with minimal dam raises, the current TMA is of sufficient capacity.

Environmental

The Province of Manitoba requires an Environment Act (EA) License to operate a mine. This document provides the licensee with the approval to undertake the development and operation of a mine and any associated activities as outlined in the License. The License includes general and specific terms under which operation of the mine may occur to ensure that “the environment is maintained in such a manner as to sustain a high quality of life, including social and economic development recreation and leisure for present and future Manitobans”. License holders are required to submit an annual report to Manitoba Conservation. The License, currently held by Garson, is in good standing and requires a transfer of ownership to Alexis Minerals.

Since January 2005, the mine has been in care and maintenance status. In January 2010, the CEO of Alexis made a presentation to the town regarding plans to re-open the mine. The municipal council has expressed support for re-opening of the mine, and a letter to this effect was sent to Alexis. The mine is projected to create approximately 250 jobs over a five year period. There is a housing shortage in the Town of Snow Lake and, Alexis plans to erect a mobile camp of 50 people.

Social

The mine has been operational for an extended period of time under different owners, and few Aboriginal concerns have been raised. There are no Indian Reserves (IR) located within the immediate vicinity of Snow Lake. However, four IR are located within a 100 km radius of the town. The nearest IR is Opekanowi Sakaheykun IR No. 09797 located 80 km north of Snow Lake. The three remaining IR include Napahkapihskow Sakhahigan IR No. 09852 (85 km northwest of Snow Lake), Wepuskow Ohnikahp IR No. 09851 (97 km North West of Snow Lake), and Mistiategameek Sipi IR No. 09878 (92 km north of Snow Lake). The area is currently considered disturbed industrial development land. As the project is not expected to result in disturbance to new areas, therefore any new concerns from First Nation communities are not anticipated.

Capital Costs

The cost estimates to recondition the Main Mine infrastructure and construct new mine infrastructure at the No. 3 Zone were provided by Contractors in August and September, 2010 through a competitive bid process. The following table provides the breakdown of the pre-production capital:

Table 1.2 Pre-Production Capital Cost Estimate

Pre-Production Capital Expenditures		CAN$
Pre-Prod GOLD PRODUCTION (Oz Au)		6,004
Revenue Credit		7,863,685
Pre-prod Cost Transfer to Capital		21,664,047
Net Pre-Prod Operating Capital		13,800,362
3 Zone		
3 Zone Surface		755,750
3 Zone Dewater		3,306,600
3 Zone Equipment		758,485
3 Zone Capital Development		2,128,000
Total 3 Zone		6,948,835
Main Mine		
Total Surface		793,825
Main Mine Hoist/Mine Rehab		10,336,765
Total Main Mine Equipment		1,018,964
Total Capital Development		
Total Main Mine		12,149,554
Mill		2,565,000
Tailings Dam Raise		25,000
Warehouse 1st Fill		1,600,000
Subtotal Capital		37,088,751
Contingency	10 %	3,708,875
Total Capital		**40,797,626**

Operating Cost

The operating costs were estimated using a zero based budget for the planned hourly rates and supplier quotations. These unit and fixed costs were then compared to the current costs at the Lac Herbin mine and the inflated historical costs from 2002-2004 for SLM. The ore transportation milling costs are estimated using detailed consumption data from the operating years in conjunction with the metallurgical test results. The estimates include all site costs but do not include royalties or head office administration.

The total operating cost estimate is provided in the following table:

Table.1.3: Operating Costs per Tonne of Ore

Description	Cost Per tonne Ore (CAN$)
Mining	61.29
Milling	12.36
Administration and lodging	7.75
Total	**81.41**

Economic Analysis

A base case cash flow schedule has been prepared, using a Bloomberg gold price deck as at August 31, 2010 of US$ 1, 277 /ounce for 2011, US$ 1,303/ounce for 2012, US$ 1,276/ounce for 2013, and US$ 1,051/ounce for 2014 and onward with an exchange rate of CAN$ 1.04 for US$ 1.00 for 2011 – 2013 and 1.06 onwards from 2014. The Snow Lake Mine economic analysis is summarized as follows:

- The current Life of Mine is 6 years (12 months pre-production and 6 years production).
- Total production of 415,600 Oz gold, with a mill recovery rate of 93.3%
- Estimated total average cash cost of CAN$ 671 (US$ 639) /Oz Au; and total costs of approximately CAN$ 851 (US$ 810) /Oz Au.
- Total sustaining capital of CAN$ 33.9 million.
- Operating cash costs per t are estimated to be CAN$ 81.31/t
- Total pre-tax cash flow, discounted cash flow at 7%, and Internal Rate of Return from the mine are projected to be CAN$ 141.1 million, CAN$ 101.1 million and 79%, respectively.
- Payback period is estimated to be 1.7 year (after tax) after commencement of commercial production.

The summary cash flow model is shown in Table 1.4:

Table 1.4: Snow Lake Mine Cash flow Model

		Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Tonnes	t		632,369	737,176	711,280	716,683	633,656	3,431,164
Grade	(g/t)		4.07	4.35	3.91	3.84	4.01	4.04
Recovery	**%**		92.6	92.6	93.7	94.0	94.0	93.3
Ounces	Oz		76,620	95,337	83,746	83,120	76,761	415,584
Gold Price	US$	1,277	1,303	1,276	1,051	1,051	1,051	1,149
Exchange	CAN$/US$	1.04	1.04	1.04	1.06	1.06	1.06	1.05
Revenue (Net Royalty)	'000		102,397	124,770	92,010	91,322	84,337	494,837
Op. Cost	'000		59,056	57,202	57,323	52,255	53,152	278,988
EBITDA	'000		43,340	67,569	34,689	39,067	31,185	215,849
Depreciation	'000		15,452	19,227	16,890	16,763	15,481	83,814
EBIT	'000		27,888	48,341	17,798	22,204	15,704	132,035
Taxes	'000		6,560	18,341	6,663	8,454	5,953	45,970
Earnings	'000		21,328	30,001	11,135	13,849	9,751	86,065
Capex	'000	40,798	22,639	9,341	1,399	237	300	74,714
Cash flow (Before Taxes)	'000	(40,798)	20,701	58,228	33,289	38,830	30,885	141,135
Cash flow (After taxes)	'000	(40,798)	14,141	39,887	26,626	30,376	24,932	95,165
Opex per t	CAN$/t		93.39	77.60	80.59	72.91	83.88	81.31
Opex per Oz	US$/Oz		745	577	646	593	653	639
Opex + Capex per Oz	US$/Oz		1,025	671	661	596	657	810

Note: All dollar amount are CAN$ unless otherwise specified,

Royalty of 1.38% has been netted in revenue.

Conclusions

The Feasibility Study of the Snow Lake Mine Re-activation shows a profitable, robust project with a high Internal Rate of Return. The project remains profitable at the long term gold price of US$ 1,051/Oz. Therefore, at the current price level of over US$ 1,300/Oz the project is very robust.

As a past producer, the Snow Lake Mine benefits from several advantages:

- Presence of existing surface infrastructures in good condition;
- Presence of extensive underground development;
- Excellent rock mass quality suggesting that reconditioning of the underground workings will be straightforward;
- Past underground production track record demonstrating
  1. Achievable productivity levels;
  2. Realistic dilution rates; and
  3. Realistic mining recovery rates;
- Past milling track record demonstrating
  1. Achievable milling rate;
  2. Realistic power and chemical consumption rates; and
  3. Realistic metallurgical recovery rates;
- Presence of an existing Tailings Management Area with remaining storage capacity; and
- Presence of existing environmental permits.

These advantages have two main benefits, namely:

- Much lower capital requirement than otherwise; and
- Much lower uncertainties than otherwise.

Consequently, the overall level of risk for the project is greatly reduced compared to green field projects.

The Snow Lake Mine Re-activation could be adversely affected by the prevailing manpower shortage and/or to a lesser degree the low availability of service providers (such as mining contractors). Finally, equipment availability could become problematic if too many new underground mining projects increase the demand in the near future (current economic indicators point in that direction).

The exploration potential of the property could extend the known ore zones or find new ore zones. This would extend mine life and enhance the profitability of the project.

Recommendations

The above conclusions lead to the following recommendations:

- Re-activate the Snow Lake Mine as soon as possible to take advantage of the high gold price;
- Allow enough time to hire personnel, particularly managerial and technical;
- To ensure on time execution, secure commitments from services providers and equipment suppliers as soon as practically possible;
- Initiate exploration around the known ore zones to increase mine life;
- Continue the aggressive exploration program that has been initiated by Alexis where the defined objectives include expansion of known zones of gold mineralization, testing targets strategically proximal to current mine development, and generate earlier stage targets on the property through the systematic application of modern-day exploration techniques.
- Start preliminary studies regarding additional storage capacity in the Tailings Management Area to allow enough room for additional ore found through exploration successes.
- Ensure timely lodging availability.

2. INTRODUCTION AND TERMS OF REFERENCE (AR)

2.1 Introduction

Alexis Minerals Corporation (Alexis) announced that it took control of Garson Gold Corp. (Garson) on December 18, 2009 (please see Alexis corresponding Press Release on Alexis' web site: www.alexisminerals.com) and completed acquisition of Garson on April 29, 2010. Garson is now a wholly owned subsidiary of Alexis. Garson's main asset was the New Brittania Gold Mine located in Snow Lake, Manitoba (Alexis subsequently renamed the mine Snow Lake Mine). Historical production at the Snow Lake Mine took place during two different episodes, i.e. from 1949 to 1958 and 1995 to 2005 and totalled 12.1 million t at 4.23 g/t Au for 1.44 million Oz of gold. For the period from 1997 to 2002 production at the Snow Lake Mine averaged over 100,000 Oz of gold per year. At the time of the transaction the mine had 2.61 million t of Measured and Indicated Resources at 5.7 g/t for 449,000 Oz of contained gold and 1.89 million t of Inferred Resources for 317,000 Oz of contained gold. As is often the case when prices are high Alexis is planning to reopen the Snow Lake Mine. This report presents the results of the NI 43-101 compliant Feasibility Study carried out for the Snow Lake Mine re-activation project.

2.2 Terms of reference

Alexis retained several consultants to conduct the NI 43-101 compliant Feasibility Study . GENIVAR was retained to:

- Review and validate the Mining Engineering carried out by Alexis, i.e. provide the Mining Engineering QP (Qualified Person), Mr. André Roy (AR), P. Eng.; and
- Assemble the contribution by other consultants into this Feasibility Study, they are:
    - **Resource Estimation:** Jamie Lavigne, P. Geo., VP Exploration Alexis Minerals (JL);
    - **Rock Mechanics:** Dave West, P.Eng. Rock Mechanics/Geotechnical Engineer, Independent Consulting Mining Engineer (DW);
    - **Process and Milling:** Ian Ward, P.Eng., Metalurgist, Independent Consulting Mining Engineer (IW);
    - **Tailings Disposal:** Matthew Parfitt, P.Eng., Knight Piésold Consulting (MP); and
    - **Environmental:** Mark Bednardz, Independent Environmental Consultant (MB).

The individuals named above are the designated QPs for their area of responsibility. Hence each of them signed a corresponding Certificate of Qualification; they are presented in Section 20 – Date & Signature Page. To avoid confusion and make it

easy to identify which QP authored a given section of the study, each section of the study bears the initials of the authoring QP.

To review and validate the Mining Engineering carried out by Alexis, GENIVAR reviewed and analysed data provided by Alexis, its consultants and the previous operators of the mine, and its representative conducted a site visit. This allowed GENIVAR to draw its own conclusions which in turn allowed its representative to sign the corresponding Certificate of Qualification.

2.3 Warning about units of measurement

Grades

The Snow Lake mine will use the Imperial units of measurements because it was originally built and operated with this system. Hence, for example, this report uses the foot, square foot and cubic foot as the main units of length, area and volume. However since nowadays in Canada the gold mining industry practice is to use the Metric Tonne as the unit of mass this is the unit used in this report unless specifically stated otherwise. Hence gold grades expressed in Grammes per Metric (g/t) as opposed to Grammes per Short Ton. For the benefit of the readers the following table gives the corresponding grades for a grade of 1.0000 Troy Oz per Short Ton (Oz/Ton):

- 31.10348 Gramme per Short Ton (g/Ton)
- 34.28572 Gramme per Metric Tonne (g/t)
- 1.10231 Oz per Metric Tonne (Oz/t)

Note: there are 2204.623 lb in a Metric Tonne

Densities

The waste and ore density is:

- 11.2 Cubic Feet per Short Ton (ft^3/Ton)
- 12.35 Cubic Feet per Metric Tonne (ft^3/t)
- 178.57 Lb per Cubic foot (lb/ft^3)
- 2.86 Metric Tonne per Cubic Meter (t/m^3)

Rounding

Finally please note that T are rounded to the nearest 1,000 whereas Oz are rounded to the nearest 100.

Units

cm	Centimeter
day	Day
CAN$	Canadian Dollar
US$	U.S. Dollar
ft	Foot
ft/shift	Foot per 10 Hours Work Shift
ft/man·shift	Foot per Man per 10 Hours Work
ft/day	Foot per Day
ft^3	Cubic Foot
ft^3/ton	Cubic Foot per Short Ton
ft^3/t	Cubic Foot per Metric Tonne
ft^3/min	Cubic Foot per Minute
g	Gramme
g/t	Gramme per Metric Tonne
hp	Horsepower
in	Inch
lb	Pound
kg	Kilogram
m	Meter
m^2	Square Meter

m^3	Cubic Meter
month	Month
Oz	Troy Once
Oz/Ton	Troy Ounce per Short Ton
T	Metric Tonne
t/m	Metric Tonne per Meter
t/shift	Metric Tonne per 10 Hours Work
t/man·shift	Metric Tonne per Man per 10
t/day	Metric Tonne per Day
t/month	Metric Tonne per Month
t/year	Metric Tonne per Year
ton	Short Ton
ton/day	Short Ton per Day
ton/ft	Short Ton per Foot
USG	US Gallon
USGPM	US Gallon per Minute
year	Year
yd	Yard
yd^3	Cubic Yard

3. RELIANCE ON OTHER EXPERTS (AR)

Several reports have been previously published about the Resource Estimation and/or the possible reopening of either the Main Mine and/or the No. 3 Zone. These previous reports were made available to all the authors in this NI 43-101 compliant Feasibility Study, and in fact in some instances the authors are the same. These reports are listed in Section 20 References. Thus the authors of this NI 43-101 compliant Feasibility Study have relied on previous reports when it was not possible to directly verify information. Each author has included the appropriate reference where necessary. This procedure has also been used for excerpts from previous reports where the text did not need revision to reflect the current situation.

4. PROPERTY DESCRIPTION AND LOCATION (JL)

During the first quarter of 2010, Alexis completed the takeover of Garson Gold Corp. and assumed 100% ownership of the New Britannia Mine property. Garson remains a wholly owned subsidiary of Alexis. The property status has not changed since the takeover was completed. All of the claims are in good standing and all of the lease payments are up to date. Alexis has sufficient work expenditure credits to keep the property in good standing. The description of the New Britannia Mine property is taken from the technical report by Micon International dated May 30, 2009. Alexis now publicly refers to the New Britania Mine Property as the Snow Lake Property (SLM property).

The SLM property is located in west-central Manitoba at 54°52'44" north latitude and 100°02'12" west longitude, within the community of Snow Lake. The town of Snow Lake is situated approximately 685 kilometres (km) north of Winnipeg, the provincial capital of Manitoba. Figure 4-1 is a general location map of Snow Lake and the SLM property.

The SLM property is composed of two surveyed mineral leases (ML-61 & ML-323) and forty-three unsurveyed mineral claims totalling 4,840 hectares (ha). The mineral claims are contiguous along the southern and eastern boundary of mineral lease ML-61 and a portion of the eastern boundary of mineral lease ML-323 have been historically referred to as the HudBay Option claims. The block of claims which are contiguous to the north of ML-323 have been historically referred to as the Squall Lake claims. Mineral Lease ML-61 contains the SLM mineral deposit and ML-323 contains the No. 3 Zone and Birch Zone mineral deposits.

The two mining leases have been legally surveyed and remain in good standing providing that the annual taxes are paid. The mineral leases are taxed at the rate of CAN$ 8.00/ha each year and are subject to a 1.38% NSR, based on production, payable to W. Bruce Dunlop.

The SLM assets, which include the mining leases, mineral claims and the surface facilities, are held by New Britannia Mine Ltd. (NBM Ltd.) which is 100%-owned by Garson. The following summarizes the sequence of corporate transactions and mergers leading to the current ownership and consolidation of the property:

- Prior to December 22, 2006, the mining leases and the HudBay Options claims were held by 1126774 Ontario Limited (1126774 Ontario) which was held by Kinross and High River Gold Ltd. in a joint venture, with each party holding 50%.
- On December 22, 2006, 1126774 Ontario was combined with Pegasus Mines Ltd. (Pegasus), with the merged entity retaining the name Pegasus.
- Pegasus was wholly-owned by the New Britannia Mine Joint Venture (NBM-JV) which consisted of Piper Capital Inc. (Piper) with 60%, and Garson Resources Ltd.

(Garson Resources) with 40%. Prior to the establishment of the NBM-JV, Garson Resources Ltd. held the Squall Lake claims.

- On June 28, 2007, Piper and Garson Resources merged to form Garson Gold Corp.
- On September 19, 2007, Pegasus changed its name to New Britannia Mines Ltd.

The terms of the agreement with Kinross have been described in the previous Micon Technical Report (Lewis, 2006).

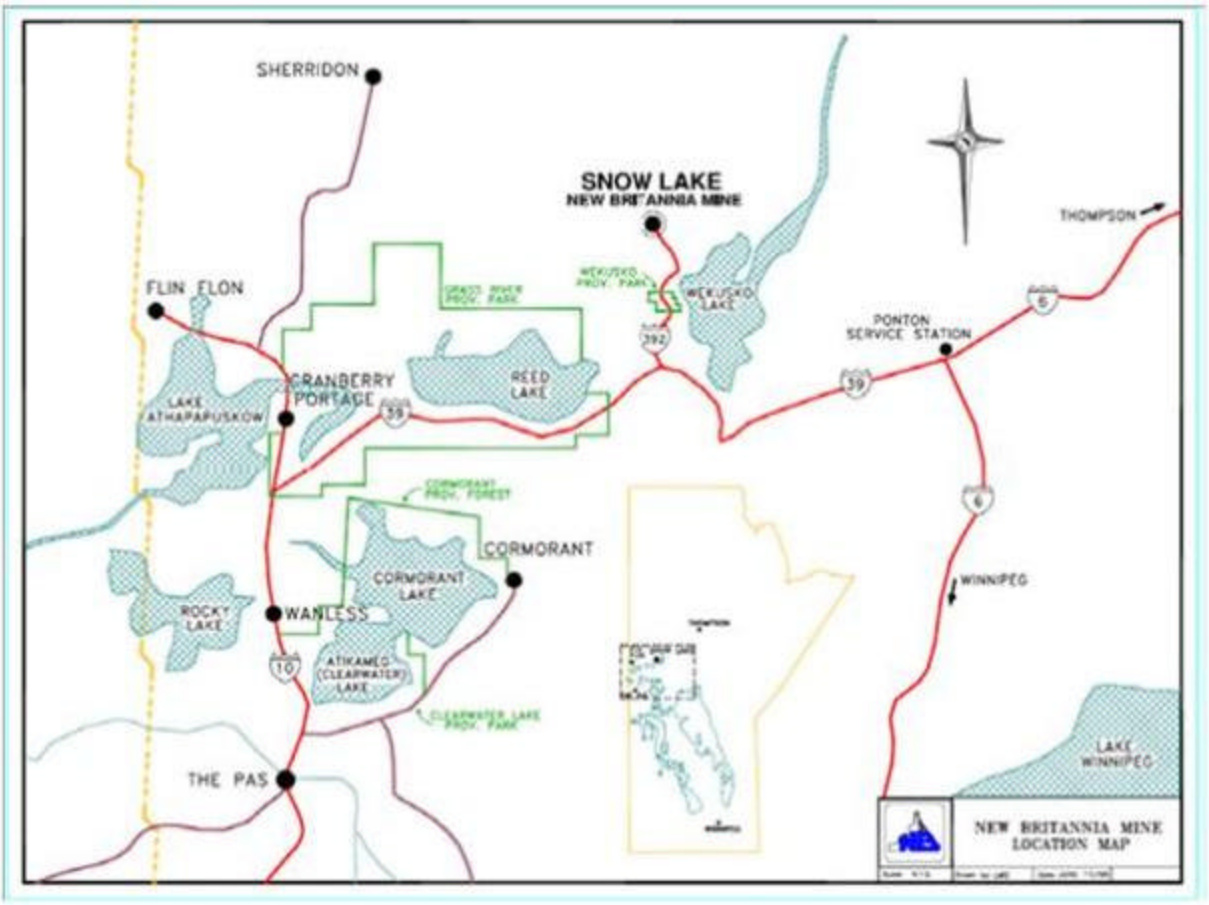

Figure 4-1: Location of the Town of Snow Lake, Manitoba and the New Britannia Mine Property.

At the time of the original transaction with Kinross, the HudBay claims were under option from Hudson Bay Minerals Inc. (HudBay). Exercise of the option by Garson required a one-time cash payment of CAN$ 400,000 and a royalty consisting of 1.5% of the gross proceeds from the sale or disposition of all metals mined and removed from the optioned claims. However, HudBay has retained the rights to the base metals on the mineral claims and should a base metal deposit be discovered and exploited on the claims, then Garson would retain a 1.5% royalty.

On December 31, 2007, the CAN$ 400,000 option payment was made and, subject to the 1.5% royalty, the HudBay claims are now 100%-owned by NBM Ltd. Prior to

conducting any production mining, the claims would have to be either incorporated into the existing mineral lease ML-61 or incorporated into a new mining lease. Micon indicated that due to the pre-existing royalty on ML-61, the claims should be incorporated into a new mining lease (Lewis, 2009). This would create a boundary which could be surveyed underground so that no conflicts between the two separate royalty holders would exist.

The Squall Lake group of claims (comprising 14 contiguous, unpatented mining claims with an area of 899 ha) is 100%-owned by Garson subject to various underlying royalties. The following is a quote from the Garson website:

"The property contains a nets profits royalty (NPI) interest payable on commencement of commercial production of 4% to a private individual, an additional 6% NPI payable to American Barrick (now Barrick), plus a 30% NPI payable to Barrick up to a maximum of $550,000. The property also contains a production royalty of $0.10 per ton, payable to a private individual, from products milled from 6 of the claims."

The mineral claims require an annual assessment work requirement of CAN$ 25.00/ha to keep them in good standing beyond their current expiry dates. In Manitoba, when the claim holder exceeds the minimum annual assessment expenditures for the mineral claims, surplus expenditures can be banked and utilized in future years to meet the annual assessment requirements. In the case of the NBM property, Garson indicates that additional credits exist on both the mineral leases and the mineral claims. The additional credits were not verified by Micon as the Manitoba government considers this as confidential information that is only available to the claim holders.

Mineral lease ML-61 (NBM) and mineral lease ML-323 are located on the NTS map sheet NE16-63K East Half. The HudBay Option mineral claims are located on NTS map sheets NE16-63K East Half, NW13-63J West Half, and SE16-63K East Half. The Squall Lake claims are located on NTS map sheet NE16-63K East Half.

Table 4-1 is a detailed list of both mineral leases that comprise the NBM site and the mineral claims comprised of the Squall Lake claims and the HudBay claims.

Figure 4-2 is a map indicating the location of the mineral leases and mineral claims.

Table 4-1: New Britannia Mine Property, Mineral Leases (ML-61 and ML-323) and Mineral Claims (former Hudson Bay Option and Squall Lake Group)

Name of Leases or Mineral Claims	Group	Claim Number	Area (ha.)	Date Staked	Expiry Date (Midnight)	Work Requirement per year (CAN$)
Mineral Leases						
ML-61		------	419	1-Apr-92	April 1, 2013	3,352.00
ML-323		------	828	28-Mar-95	March 28, 2016	6,624.00
Mineral Leases Total Area:			**1,247**		**see note 1,2**	**see note 3**
Mineral Claims						
Bud 9415	HudBay Option	CB9415	47	18-Dec-78	February 16, 2011	1,175.00
Bud 12593	HudBay Option	CB12593	155	16-Oct-81	December 15, 2011	3,875.00
Bud 12455	HudBay Option	CB12455	143	16-Oct-81	December 15, 2011	3,575.00
Bud 12456	HudBay Option	CB12456	174	16-Oct-81	December 15, 2011	4,350.00
Bud 1	HudBay Option	P6557B	21	2-Dec-64	January 31, 2011	525.00
Bud 2 Fr.	HudBay Option	P6558B	21	2-Dec-64	January 31, 2011	525.00
Bud 3	HudBay Option	P6559B	21	2-Dec-64	January 31, 2011	525.00
Bud 4	HudBay Option	P6560B	21	2-Dec-64	January 31, 2011	525.00
Bud 5	HudBay Option	P6561B	21	2-Dec-64	January 31, 2011	525.00
Bud 6Fr.	HudBay Option	P6562B	21	2-Dec-64	January 31, 2011	525.00
Bud 7	HudBay Option	P6563B	21	2-Dec-64	January 31, 2011	525.00
Bud 8	HudBay Option	P6564B	21	2-Dec-64	January 31, 2011	525.00
Bud 9	HudBay Option	P6565B	21	2-Dec-64	January 31, 2011	525.00
Bud 10Fr.	HudBay Option	P6566B	21	2-Dec-64	January 31, 2011	525.00
Bud 11	HudBay Option	P6567B	21	2-Dec-64	January 31, 2011	525.00
Bud 12	HudBay Option	P6568B	21	2-Dec-64	January 31, 2011	525.00
Bud 13	HudBay Option	P6569B	21	2-Dec-64	January 31, 2011	525.00
Bud 619	HudBay Option	P619F	206	9-Nov-92	January 8, 2011	5,150.00
Bud 618	HudBay Option	P618F	240	9-Nov-92	January 8, 2011	6,000.00
Bud 617	HudBay Option	P617F	240	9-Nov-92	January 8, 2001	6,000.00
Bud 30	HudBay Option	P3933E	21	4-May-73	July 3, 2012	525.00
Bud 54	HudBay Option	CB9419	132	15-Jan-79	March 16, 2011	3,300.00
Bud 53	HudBay Option	CB9418	87	15-Jan-79	March 16, 2011	2,175.00
	HudBay Option	CB7031	164	17-Jul-78	September 15, 2001	4,100.00
Bud 4165	HudBay Option	P4165E	172	7-Dec-84	February 5, 2011	4,300.00
	HudBay Option	CB6422	206	12-Aug-74	October 11, 2011	5,150.00
	HudBay Option	CB4737	239	4-May-73	July 3, 2011	5,975.00
	HudBay Option	CB4736	107	4-May-73	July 3, 2012	2,675.00
	HudBay Option	CB7085	88	17-Jul-78	September 15, 2011	2,200.00
CAM1	Squall Lake	CB11772	195	15-Dec-80	March 16, 2011	4,875.00
KAP1	Squall Lake	CB8827	48	10-Sep-81	November 23, 2012	1,200.00
	Squall Lake	CB7905	45	76-Dec-08	March 11, 20011	1,125.00
	Squall Lake	CB4513	119	78-Jun-6	September 5, 2011	2,975

Name of Leases or Mineral Claims	Group	Claim Number	Area (ha.)	Date Staked	Expiry Date (Midnight)	Work Requirement per year (CAN$)
	Squall Lake	CB7928	42	77-Feb-9	May 13, 2011	1,050
	Squall Lake	CB7929	43	77-Apr-6	July 8, 2011	1,075.00
CAM2	Squall Lake	W46245	17	80-Dec-15	March 16, 2011	425.00
	Squall Lake	CB6448	33	78-Jun-6	September 5, 2011	825.00
	Squall Lake	CB7176	33	78-Jun-6	September 5, 2011	825.00
	Squall Lake	CB8004	43	77-Apr-6	July 8, 2011	1,075.00
	Squall Lake	CB8005	43	77-Apr-6	July 8, 2011	1,075.00
	Squall Lake	CB4511	43	78-Jun-6	September 5, 2011	1,075.00
	Squall Lake	CB8006	43	77-Apr-6	July 8, 2011	1,075.00
	Squall Lake	CB4512	152	78-Jun-7	September 5, 2011	1,075.00
Mineral Claims Total Area:			**3,593**		**Total Claim Work Req.**	**97,076.00**
NBM Property Total Area:			**4,840**			

Note 1: Claim dispositions are in good standing for one year past the date shown.
Note 2: Includes the 60-day grace period before the final expiry of claim disposition.
Note 3: The amount per year is based on the following: 1) A cash payment of CAN$ 8.00/ha is submitted to the government each year for the mineral licenses, 2) Work requirements for the mineral claims consist of CAN$ 25.00/ha each year.
Table 4-1 provided by Alexis Minerals Corporation.

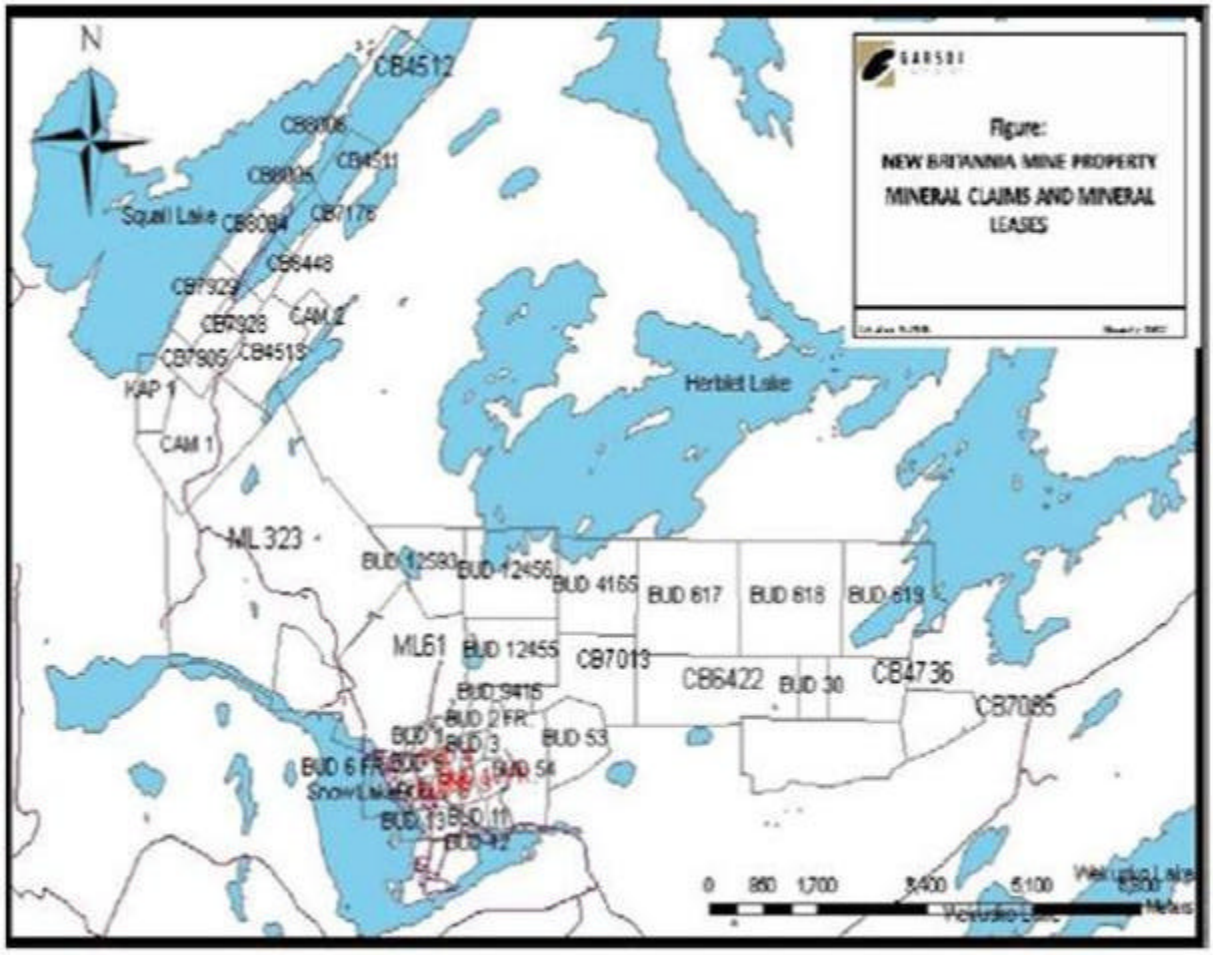

Figure 4-2: Plan of the New Britannia Mine Property Mineral Leases and Mineral Claims

Environmental reclamation on various portions of the SLM property predates the cessation of operations at the mine in early 2005. However, Kinross and High River conducted further environmental reclamation on the property since the mine was placed on care and maintenance in advance of any decision regarding either a sale of the property to a third party or final closure and full reclamation of the mine site and property. Micon indicated that the SLM property is subject to a comprehensive Closure Plan (revision 6, 2006) submitted to the Manitoba government by Kinross, in which the cost of mine closure was estimated at CAN$ 1,867,000 (Micon, 2009). This amount is currently held as a cash bond with the Manitoba government. In addition, as part of the acquisition transaction, Garson posted a cash bond of CAN$ 3,900,000, held by Kinross.

The following reclamation and closure work was conducted on the NBM property by Kinross and High River:

- The No. 3 Zone portal was capped and sealed along with the ventilation raises. Contouring and vegetation of the main areas of disturbance occurred and the site was returned to pre-mining conditions. Figure 4-3 shows the location of the No. 3 Zone portal after reclamation, with the site contoured and vegetated, and Figure 4-4 shows the former ore stockpile area after reclamation of the site.
- The main residue pile was partially capped and has been allowed to naturally re-vegetate. Although the residue pile is owned by the mine, it was created during the original mining period between 1949 and 1958 and for environmental purposes is considered an "orphan site". The residue pile is the environmental responsibility of the Manitoba Government unless it is significantly disturbed.
- Filling of the Birch Zone pit was completed at 3:1 slope and an oversize rock barriCAN$e was placed around pit walls to prevent inadvertent access to the site. The area has been vegetated and is in the process of being returned to pre-mining conditions.
- All salvageable mobile and stationary equipment was removed from the underground mine and the majority of the equipment was sold and transported off site. Any equipment left underground has had the oil and lubricants removed and the systems flushed. The equipment left underground was as follows: one 2-yard *Wagner* Scooptram, two 6-yard *Wagner* Scooptrams, eight *Toyota* service trucks, one boom truck, four scissor trucks, one fuel truck, one *JD* tractor.
- The main scrap lay-down yard was cleaned up. Contouring and vegetation of the yard, the old waste stockpile area, and the historic lay-down yard next to the main yard was completed in August, 2006.
- Federal MMER Environmental Effects Monitoring (EEM) was completed in August, 2005. The 2nd study of the Federal MMER Environmental Effects Monitoring (EEM) was completed in August of 2008 and report was completed and submitted on June of 2009. The 3rd study design report for the August 2011 EEM is due to be submitted end of February of 2011 for field work to be completed in August of 2011. The Report for the EEM is due to be submitted in June of 2012. Senes Consultants has been contracted to carry out the EEM work for the years 2010 to 2012 including all Annual Reports to be submitted in late March for those years.
- Annual treatment and discharge of approximately 148 million US gallons of tailings water to the Herblet Lake watershed was completed in August, 2006. This process is ongoing and further treatment and discharge of tailings water will be required.
- Processing plant cleanup was completed and both the tailings line and reclaim waterline have been removed.
- A waste berm was placed around the fenced-off area of the Dick Zone crown pillar to inhibit access. Figure 4-6 shows a view, looking west, along the berm with the crown pillar of the Dick Zone located on the right side of the photograph behind the fence.

Figure 4-3: No. 3 Zone Portal Site after Reclamation

Figure 4-4: No.3 Zone Former Ore Stockpile Area Reclamation

Figure 4-5: Area of the Birch Pit after Reclamation

Figure 4-6: Berm along the Fenced-off Area of the Dick Zone Crown Pillar

Environmental remedial work to be completed if the mine does not reopen includes the following:

- According to regulatory standards, the buildings and surface infrastructure will be decommissioned, dismantled and the site returned to pre-mining status. This includes removal of the surface buildings which includes the hoist house, electrical shop, exploration trailer, warehouse/dry, headframe and mill building and infrastructure.
- Four openings to surface need to be capped with concrete. Engineered designs have been completed and are located at the mine site.
- The Birch Lake tailings cover was started in October, 2005 and halted in November, 2005. The work was late getting started due to above normal wet conditions and the work was discontinued for the same reason. This work is expected to take three years once it has re-commenced. Covering of the exposed tailings is planned to impede erosion and arsenic migration. The present closure plan will leave approximately two-thirds of the northern portion of the impoundment as a water/wetland cover. The southern portion will be covered with a 0.5-metre (m) till cover and will be seeded. Upon completion of the final contouring and seeding, non-contact storm water will gravity flow to the north from the vegetated, capped surfaces and into a sediment retention basin prior to release into the Herblet Lake watershed.
- Sloping of the surface waste pile to 3:1 slope.
- Scarification and seeding of all roads and dam surfaces. All soil covered areas will be seeded to establish a permanent vegetative stand.
- Design and construction of a spillway for seasonal decant into the Herblet Lake watershed.
- Historically, an annual discharge of effluent has occurred from the tailings facility with no chemical constituents being above the mandated discharge limits. The facility has used a natural degradation for the cyanide component and a ferric sulphate addition for arsenic precipitation. The arsenic level is the main constituent of concern for the tailings closure. Personnel at NBM anticipate that three years will be required for the water discharge treatment before the facility can be closed on a standalone basis with all water discharges meeting the regulatory standards.

At the time of writing their technical report, Micon was unaware of any other outstanding environmental liabilities attached to the NBM property and was unable to comment on the quality of any remediation which may have been undertaken by the previous owners (Lewis, 2009).

Garson conducted an aggressive exploration program across the extent of the NBM property and assumed Kinross' and High River's environmental responsibilities upon signing the final agreement to purchase and upon closing of the transaction.

Garson acquired all the necessary permits for ongoing exploration work on the property. This includes permits for the operation of heavy equipment for drilling from Manitoba Conservation.

5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY (JL)

This section describes the property and is taken from the technical report by Micon International Limited, dated May 30, 2009. To the authors knowledge, there are no major changes to the infrastructure of the property or the area that would affect project development.

The SLM, (formerly NBM) property is accessible from Winnipeg, the provincial capital of Manitoba, via both paved and good quality dirt roads. Access is primarily via Provincial Highway 6 north from Winnipeg to the junction of Provincial Highway 392 at the community of Ponton. From the junction it is approximately 65 km southwest to the junction with Provincial Highway 39, and a further 33 km along highway 39 to the Town of Snow Lake. Access within the property is via all-weather private roads. The major population centres for the region are Flin Flon to the west-southwest of Snow Lake, and Thompson to the northeast. The cities of Flin Flon and Thompson, with populations of over 6,000 and 14,400 respectively (1996 census), are the supply centres for the region. Both communities also have daily air service to Winnipeg.

The SLM property is located on the edge of the Town of Snow Lake, which is the administrative, transportation, communication and supply centre for this area of northern Manitoba. Local businesses offer most goods and services required for mineral exploration and development. The town is also host to a thriving tourist industry which serves recreational hunters and fishermen. See Figure 5-1 for an aerial view, looking east, of the Snow Lake Mine, (formerly NBM) in the winter.

Figure 5-1: Aerial View of the Snow Lake Mine, (NBM), looking East

Photograph from the personal papers of William J. Lewis.

The town is also home to the HudBay Minerals Inc. (HudBay) Chisel North Mine. The Chisel North Mine is the latest in a series of base metal mines operated since 1960 by HudBay and its predecessor company Hudson Bay Mining and Smelting Company Limited (HBM&S). Consequently the area is able to furnish sufficient power, water and potential mining personnel to support a new operation. The region is also currently the focus of exploration activities for both gold and base metal mineralization by a number of companies. Micon indicated that the current land holdings are of sufficient size to support a renewal of mining extraction activities, since a fully permitted milling and tailings facility is still located on the present mineral leases (Lewis, 2009).

The topography and physiography are typical of the pre-Cambrian shield in Manitoba, consisting of low rolling hills composed of bedrock outcrops, development of thin soils, and vegetation cover that is dominated by a mixture of Poplar, Black Spruce and Balsam trees. The elevation of Snow Lake is 980 m above sea level with the relief in the area generally low and not exceeding 150 m.

Since there is no official weather station in Snow Lake the following climatic data were taken from the closest station located in Flin Flon to the west of Snow Lake on the Manitoba/Saskatchewan border. Flin Flon has two official weather stations (Flin Flon and Flin Flon A) and, for the purposes of this report, the Flin Flon weather station was used. The climate is generally cold, with the mean daily temperature being 0.6°Celsius (C). The extreme maximum temperature was recorded in 1929 at 40°C and the extreme minimum temperature was recorded in 1930 at - 46.1°C.

The average precipitation is 463.1 millimetres (mm), with 342.5 mm falling as rain and 122.8 mm falling as snow. The Snow Lake Airport operates year round, however, winter storms may temporarily close the roads or airports during the winter months.

6. HISTORY (JL)

The history of the Snow Lake area is taken from the technical report by Micon International Limited, dated May 30, 2009. The report describes the results of a preliminary assessment to re-establish production from the No.3 Zone and the re-start of the mill and surface infrastructure. Since this report, Alexis has re-assessed these activities at both the Snow Lake Mine and the No.3 Zone. This is reported in the technical report by Golder Associates entitled Snow Lake Mine NI 43-101 Technical Report, dated April 2010.

The north-western portion of Manitoba, though prospected in the 1890s, did not experience systematic exploration until after 1907. In 1914, gold was discovered approximately 20 km to the southeast of Snow Lake on the eastern shores of Wekusko Lake (also known as Herb Lake). In 1917, a shipment of ore from the Moose Horn-Ballast claims produced 108 Oz or 3.36 kilograms (kg) of gold from 28.5 tons and was treated at Trail, British Columbia. Along with the Mandy Mine on Schist Lake, which was primarily a copper mine, the Moose Horn-Ballast has the distinction of being the first gold producer in Manitoba.

In 1918, the Rex claim (Laguna mine) located approximately 2 km directly to the north of the Moose Horn-Ballast claims went into production with a 30 ton/day mill. The Laguna mine was in production intermittently until 1939 and produced 58,962 Oz (1,833.9 kg) of gold.

In 1924, the Bonter Brothers and Mr. Chris R. Parres headed to Snow Lake to work some claims the Bonters had staked previously on a gold showing called the Bounter Zone. In the fall of 1924 the Bonter Brothers left after spending the summer chasing sporadic gold assays leaving Chris Parres to prospect on his own. In October, 1924, while panning gossan material from a groundhog hole, Chris discovered fine gold; however, due to the impending freeze-up he had to wait until the following year to stake the mineral claims on his discovery.

It should be noted that the discussions related to the resource and reserve estimates conducted prior to February 1, 2001, contained in this section of the report, refer to historical estimates only. The historical resource and reserve estimates discussed in this section may have been prepared according to the accepted standards for the mining industry for the period to which they refer; however, they do not comply with the current Canadian Institute of Mining, Metallurgy and Petroleum (CIM) standards and definitions for estimating resources and reserves as required by NI 43-101 and should not be relied upon.

6.1 Nor-Acme/New Britannia Mine

On May 12, 1925, Mr. Parres recorded his claims in the town of The Pas. During the following years Chris Parres and his sons worked on the claims intermittently and were able to hold the claims by conducting the required work commitments.

In 1938, Mr. Parres placed the claims into the hands of Nor-Acme Gold Mines Ltd. (Nor-Acme Gold). This company had been incorporated on January 14, 1938 by Mr. George H. Clare who along with his partner Mr. Sam Semple had promised to raise CAN$ 50,000 to develop the gold claims. While Mr. Clare had problems raising the money, during a trip to New York he did manage to interest Mr. Frank Ebbutt of the Howe Sound Exploration Company Ltd. (Howe Sound). After visiting the property in 1940, Mr. Ebbutt returned in June, 1941 with a diamond drill to core five shallow holes on the property below the surface gold showings. Based on the assay results of these five holes Howe Sound optioned the property from Nor-Acme Gold on August 15, 1941 in exchange for a mining royalty.

In September, 1941, Howe Sound had two drills shipped to the property to begin blocking out the large low-grade mineral deposits. By February, 1942 a third drill had been added and by the spring of 1942, 50,000 ft (ft) (15,239 m) of drilling had delineated the Dick and Toots Zones. At the conclusion of the drilling program the mineral resources were estimated to be 5,000,000 tons (4,535,924 t) at an average grade of 0.150 Oz of gold per ton 4.7g/ton (grading 0.165Oz/t, 5.1g/t) confined to a depth of 1,000 ft below surface.

A period of inactivity ensued during the war years and construction of the mine infrastructure did not begin until August, 1945. The construction of the mine included the sinking of a five-compartment shaft and construction of a milling plant capable of treating 2,000 tons/day (tons/d) (1818 t/day). The company also worked closely with the officers of the Manitoba Department of Mines and Natural Resources to design a town site for the mine. As a result of this collaboration, the Snow Lake town site was the first mining town to be developed under a new section of the Local Government Districts Act, passed by the Manitoba Legislature in 1945 to provide for the orderly development of mining areas and eliminate the old style “tin town”.

In March, 1949, the mine was opened as the Nor-Acme mine (Nor-Acme) and the first bar of gold bullion was poured on April 22, 1949. Official production began on June 1, 1949 and ceased in July, 1958. Between June, 1949 and July, 1958, Nor-Acme produced approximately 5,394,000 tons (4,893,354 t) of ore and 763,000 feed Oz of gold at an average grade of 0.142 Oz/ton (4.4g/t) or 0.157Oz/t, (4.9g/t). The Nor-Acme mill recovered approximately 610,000 Oz of gold during this production period.

From 1949 to 1958, mining occurred between the surface and the 1530 Level, with the 1780 Level developed as an exploration level prior to the mine’s closure in 1958. Ownership of the mine reverted to Nor-Acme Gold Mines Ltd. on September 1, 1959 under the terms of the 1943 royalty-lease agreement with Howe Sound.

Between 1958 and the early 1990s, various companies worked on the property with the intention to bring the mine back into production; however, all these attempts were unsuccessful. Nor-Acme Gold continued to hold onto the claims during these years once the original mining lease reverted to mineral claims.

In 1962, Falconbridge conducted feasibility tests on the arsenical concentrates (residue pile). The tests indicated that the 227,000 t stockpile averaged 9.60 g/t (0.28 Oz/ton) gold.

In 1980, Hudson Bay Exploration and Development Company Ltd. (Hudson Bay or HBED) obtained a two year option to acquire a 75% interest in the Nor-Acme Mine (NBM) property. In 1981, Hudson Bay drilled 3,880 m of core to test the downward extension of the deposit and, based on the results, estimated an "indicated reserve" of 2,510,889 t grading 5.49 g/t (0.16 Oz/ton) gold (Richardson, 1996). Hudson Bay exercised its option on the property by paying a non-deductible instalment of CAN$ 100,000 in July, 1982 but dropped the option in May, 1986.

In 1987, High River Resources Ltd. (High River) optioned the Nor-Acme (NBM) property and in 1988, dewatered the shaft and workings and conducted a CAN$ 4.5 million underground exploration program. In December, 1988, High River merged with Nor-Acme Gold to form High River Gold Mines Ltd. (High River).

In 1988, High River invited third party participation in the project and on December 6, 1988 concluded an agreement whereby INCO Gold, a unit of INCO Limited, could earn a 50% stake in the property by placing the mine into production. INCO Gold agreed to provide CAN$ 6.5 million in flow-through financing to continue the underground exploration program and provide a positive feasibility study by June, 1991.

INCO Gold completed the exploration in 1988 and in September announced a "geological reserve" of 4,207,400 tons grading 0.19 Oz/ton 5.9g/tn (3,816,889 t, 0.209 Oz/t, 6.5 g/t) gold at a 0.1 Oz/ton 3.1 g/ton (0.110 Oz/t, 3.4 g/t) cut off grade. In late November, INCO Gold provided the technical portion of a pre-feasibility study jointly prepared with Wright Engineers Ltd. (Wright) which included a mine plan, mill and process design. Although technically feasible the study also indicated that higher gold prices were required to justify the high capital costs. The mine was kept dry and on care and maintenance pending an increase in the price of gold but the market continued to deteriorate and in August, 1991 the mine was allowed to flood. During this time INCO Gold and Consolidated TVX Mining Corporation merged to become TVX Gold Inc (TVX).

In September, 1992, mineral lease ML-61, covering the old Nor-Acme mine, was issued to High River. In September, 1993, High River entered into an agreement to acquire certain properties adjacent to the Nor-Acme (NBM) property which contained the No. 3 Zone and the Birch Zone mineral deposits.

TVX and High River formed a joint venture in 1994 to reopen the mine. Ownership in the joint venture was on an equal basis with each partner owning 50% of 1126774 Ontario, the numbered company which held the property. Although the joint venture was structured on an equal basis, TVX became the operator of the mine. In late 1994, the rehabilitation of the mine commenced after Bharti Engineering Associates Inc. (Bharti) published the feasibility study in April.

Ore from the secondary mining areas (No. 3 Zone and Birch pit) located on mineral lease ML-323 was used to bring the mine and mill into production prior to the completion of the rehabilitation and construction at the main shaft.

The New Britannia mill poured the first gold bullion in November, 1995 which was initially obtained from the production at No. 3 Zone. It was when production commenced in 1995 that the name of the Nor-Acme Mine was changed to the New Britannia Mine in tribute to the huge Britannia copper-silver-gold mine that Howe Sound once operated at Britannia Beach in British Columbia. The Britannia mine was once considered the sister mine to the Nor-Acme because they used the same mining methods in the 1940s and 1950s.

Production ceased at No. 3 Zone in May, 1996 and production from the Birch pit was limited to the spring and summer of 1996. Full production from the main shaft was achieved in August, 1996.

In 1995, High River and Hudson Bay Mining and Smelting Company Ltd. (HBM&S) through its subsidiary Hudson Bay entered into an option agreement covering certain mineral claims that were contiguous to the mineral leases and Sky mineral claims held by the joint venture. The HBM&S claims cover a portion of the eastern workings of the original Nor-Acme Mine and the eastern extent of the Howe Sound fault. These claims were added to the property held by 1126774 Ontario. In June, 1997, this option agreement was amended and restated to better accommodate some of the concerns the joint venture parties had with the original agreement.

In December, 1998, the size of mineral lease ML-61 was increased with the addition of the adjoining Sky claims which were incorporated into ML-61 because the north-northeast plunge of the main Dick and Ruttan Zones situated the Ruttan Zone on the claims at the 2506 Level.

In 1999, TVX and Normandy Mining Limited (Normandy) formed a strategic arrangement in which the operating management of the NBM was transferred to TVX Normandy Americas (Canada) Holdings Inc. (TVX Normandy) from TVX. At this point ownership of the mine, through the shares of 1126774 Ontario, became TVX Normandy 50% (TVX Gold 25% and Normandy 25%), and High River Gold 50%.

In January, 2002, Newmont Mining Corporation successfully purchased Normandy Mining Limited after a bidding war with AngloGold of South Africa and as a result of

the purchase, TVX Normandy was changed to TVX Newmont Americas (Canada) Holdings Inc. (TVX Newmont).

In June, 2002, TVX, Kinross Gold Corp. (Kinross) and Echo Bay Mines Ltd. (Echo Bay) announced plans to merge the three companies into a single gold producer. Also included in this merger was the sale by Newmont of its 50% portion of TVX Newmont to TVX. Completion of the three-way merger occurred on January 31, 2003 and Kinross became the operator of the mine.

In August, 2002, mineral claim Bud 4165 (P4165E) was transferred from Hudson Bay to 1126774 Ontario and in February, 2004, mineral claims CB6422, CB4737, CB4736 and CB7085 were transferred from Hudson Bay to 1126774 Ontario in exchange for the return of Bud 52 (CB9417). When these claims were added to the pre-existing claims covered by the Hudson Bay option agreement, the total number of claims covered by the option agreement rose to 29 claims.

During 2003, the mine struggled with lower production tonnage and increased dilution in the Dick and Ruttan Zones below the 3000 Level and, in early 2004, the decision was taken to close the mine for economic reasons.

In June, 2004, development mining was concluded at NBM with the completion of the 3680 Level. The original mine closure plan called for the completion of production mining by the end of October, 2004 and the start of reclamation activities on the property. However, a sharp increase in the overall grade as a result of lower dilution beginning in June led to a re-evaluation of the mine's closure. This turnaround was directly attributed to the better than expected grades, ore thickness and strike lengths for the western portion of the last two mining levels (3630 and 3680) in the Dick Zone, along with significantly less dilution (hangingwall failure) in these stopes. As a result of this turnaround in profitability and the lack of good drilling to define the western end of the Dick Zone, the joint venture partners approved funds to both conduct an exploration program on the 3630 Level and re-commence ramp and sill development below the 3680 Level at the end of August, 2004.

Production mining ceased at the end of September, 2004 and the mill was put on care and maintenance. The mineralized material generated from the reactivation of the sill development was stockpiled on surface.

The fall 2004 diamond drilling program was designed to identify both the trend and potential economic viability of the northwest trending mineralization at the western extent of the Dick Zone below the 3680 Level. The drill program was conducted from the 3630 Level exploration drill drift, which is located in the hangingwall of the 3630 Level. During the drilling program a total of 9,302.7 ft (2,835.33 m) of BQ wireline diamond drilling, covering the area between the 3680 and 4100 Levels, was completed. Of the total drilling, 7,710.7 ft (2,350.11 m) was conducted on Mineral Lease ML-61 and 1,592 ft (485.22 m) was conducted on the Bud 12593 (CB12593) mineral claim.

The results from the fall 2004 drilling program indicated that the mineralization encountered during development on the 3630 and 3680 Levels constituted a blowout (swell) in the Dick Zone, and that the zone narrows considerably between the 3680 and 3900 Levels. Another potential blowout in the mineralization was identified occurring below the 3900 Level, but determining the extent of the mineralization below this level required further drilling and this was not covered by the program. A summary of the results for the fall 2004 drilling program (Dick Zone) is contained in Table 6-1.

The drill program also confirmed that the dip of the mineralization, below the 3680 Level, flattened to between 20° and 30° and the shallow dip appeared to coincide with a general flattening of the McLeod Road Thrust fault. While the results from the drilling program indicated that the mineralization of the Dick Zone exhibited good continuity, the overall width and grade of the mineralization was not significant enough to allow it to be converted from a mineral resource to a mineable reserve. As a, all development was suspended at NBM and initiation of mine closure proceedings were commenced on January 27, 2005.

Table 6-1: Summary the Results of the Fall 2004 Drilling Program from the 3630 Level (Dick Zone)

Diamond Drill		Diamond Drill Hole Location							Mineral Intersections (Imperial Units)					Mineral Intersections (Metric Units)					Mining Level Intersected
Hole Number	Core Size	Mineral Lease/Claim	Zone	Drill Station (Easting)	Azimuth (°)	Dip (°)	Length (feet)	Length (metres)	From	To	Core Length (feet)	True Width (feet)	In-Situ Grade (oz/ton)	From	To	Core Length (metres)	True Width (metres)	In-Situ Grade (g/t)	
1st Drill Station																			
3630-01	BQ	Mineral Lease ML-61	Dick	37604.2	232.6	-17	477.4	145.5	282.4	303.0	20.6	13.1	0.112	86.1	92.4	6.3	3.99	3.84	3713
3630-02	BQ	Mineral Lease ML-61	Dick	37603.6	239.8	-16	507.0	154.5	331.9	336.7	4.8	2.3	0.232	101.2	102.6	1.5	0.70	7.95	3716
3630-03	BQ	Mineral Lease ML-61	Dick	37617.0	189	-24	394.0	120.1	292.5	296.1	3.6	3.3	0.451	89.2	90.3	1.1	1.01	15.46	3747
3630-04	BQ	Mineral Lease ML-61	Dick	37620.4	165	-25	487.4	148.6	242.9	304.5	61.6	54.8	0.187	74.0	92.8	18.8	16.70	6.41	3734
3630-05	BQ	Mineral Lease ML-61	Dick	37603.4	247.8	-20	397.9	121.3	327.5	334.7	7.2	5.0	0.141	99.8	102.0	2.2	1.52	4.83	3728
3630-06	BQ	Mineral Lease ML-61	Dick	37605.5	247.5	-45	387.5	118.1	244.5	251.7	7.2	6.0	0.105	74.5	76.7	2.2	1.83	3.60	3796
3630-07	BQ	Mineral Lease ML-61	Dick	37607.0	248.9	-76	397.0	121.0	210.0	231.0	21.0	21.0	0.132	64.0	70.4	6.4	6.40	4.53	3834
3630-08	BQ	Mineral Lease ML-61	Dick	37626.4	069.3	-77	557.0	169.8	274.0	287.5	13.5	11.5	0.111	83.5	87.6	4.1	3.51	3.81	3895
3630-09	BQ	Mineral Lease ML-61	Dick	37625.6	069.8	-60	508.0	154.8	322.2	358.0	35.8	26.0	0.191	98.2	109.1	10.9	7.92	6.55	3916
3630-10	BQ	Mineral Lease ML-61	Dick	37627.8	068.7	-45	608.0	185.3	488.7	507.0	18.3	9.0	0.149	149.0	154.5	5.6	2.74	5.11	3973
3630-11	BQ	Mineral Lease ML-61	Dick	37628.5	067.4	-39	736.6	224.5	598.5	646.6	48.1	17.0	0.175	182.4	197.1	14.7	5.18	6.00	4015
3630-12	BQ	Mineral Lease ML-61	Dick	37612.4	217.3	-25	397.0	121.0	250.2	253.9	3.7	3.1	0.239	76.3	77.4	1.1	0.94	8.19	3729
2nd Drill Station																			
3630-13	BQ	Mineral Lease ML-61	Dick	37506.1	248.2	-21	417.0	127.1	324.0	332.7	8.7	7.0	0.170	98.8	101.4	2.7	2.13	5.83	3733
3630-14	BQ	Mineral Lease ML-61	Dick	37510.4	248.1	-82	388.0	118.3	243.5	248.8	5.3	5.1	0.044	74.2	75.8	1.6	1.55	1.51	3858
3630-15	BQ	Mineral Lease ML-61	Dick	37520.4	066.8	-58	568.0	173.1	458.0	475.0	17.0	10.0	0.255	139.6	144.8	5.2	3.05	8.74	4014
3630-16	BQ	Mineral Lease ML-61	Dick	37520.9	066.8	-48	817.0	249.0	637.0	662.5	25.5	10.0	0.221	194.2	201.9	7.8	3.05	7.58	4103
3630-17	BQ	Mineral Lease ML-61	Dick	37507.3	227	-22	448.0	136.6	335.3	345.4	10.1	7.8	0.121	102.2	105.3	3.1	2.38	4.15	3750
3630-18	BQ	Mineral Lease ML-61	Dick	37511.9	205	-55	367.0	111.9	262.4	272.0	9.6	9.0	0.136	80.0	82.9	2.9	2.74	4.66	3839
3630-19	BQ	Mineral Lease ML-61	Dick	37507.8	248.2	-48	347.6	105.9	250.6	253.3	2.7	2.7	0.028	76.4	77.2	0.8	0.82	0.96	3803
3630-20	BQ	Mineral Lease ML-61	Dick	37519.3	067.0	-75	397.0	121.0	298.9	310.4	11.5	9.0	0.110	91.1	94.6	3.5	2.74	3.77	3912
3rd Drill Station																			
3630-21	BQ	Mineral Lease ML-61	Dick	37433.8	248.2	-23	379.0	115.5	346.3	347.7	1.4	1.0	0.104	105.6	106.0	0.4	0.30	3.57	3747
3630-22	BQ	Mineral Lease ML-61	Dick	37436.8	249	-57	308.0	93.9	245.9	251.8	5.9	5.9	0.211	75.0	76.7	1.8	1.80	7.23	3820
3630-23	BQ	Hudson Bay option	Dick	37459.8	67.2	-85	397.0	121.0	281.6	283.6	2.0	2.0	0.207	85.8	86.4	0.6	0.61	7.10	3893
3630-24	BQ	Hudson Bay option	Dick	37457.8	67.2	-61	498.0	151.8	429.5	430.9	1.4	1.0	0.204	130.9	131.3	0.4	0.30	6.99	3989
3630-25	BQ	Hudson Bay option	Dick	37461.0	66.2	-50	697.0	212.4	570.6	579.3	8.7	4.0	0.226	173.9	176.6	2.7	1.22	7.75	4063

Table provided by Garson Gold Corp.

As part of the site cleanup the stockpile generated during the fall of 2004 was processed by the mill between June 11, 2005 and July 4, 2005. During this period 22,714 t grading 0.102 g/t were processed and 2,113 Oz of gold were recovered. Activities related to care and maintenance and site cleanup in preparation for a possible full closure and reclamation of the site were conducted from January, 2005 until its purchase by Garson and Piper.

Between 1996 and 2004, definition drilling and exploration drilling was conducted at NBM using both mine personnel and a contractor. A bazooka drill was used by the mine to conduct definition drilling on the development sills to assist the geology department in defining the final ore contours prior to outlining the longhole drilling for each mining level. Contract drilling was used for both definition drilling and exploration drilling. Table 6-2 summarizes the total yearly definition and exploration footage conducted by the contractor at the New Britannia mine between 1996 and 2004.

Table 6-2: Summary of Definition and Exploration Drilling Amounts from 1996 to 2004

Year	Total Yearly Definition + Exploration Drilling	
	Feet	Metres
1996	190,134	57,953
1997	235,217	71,694
1998	212,122	64,655
1999	148,223	45,178
2000	177,746	54,177
2001	163,701	49,896
2002	319,996	97,534
2003	190,356	58,020
2004	26,423	8,054
Total	**1,663,918**	**507,162**
Table provided by Garson Gold Corp.		

Underground exploration drilling was curtailed at NBM in October, 2003 due to the uneconomic results obtained below the 4000 Level on both the Dick and Ruttan Zones. Definition drilling during 2004 was reduced to only the amount required to produce the geological longhole interpretation for each extraction sill.

In late 2004, a small exploration drilling program was approved to follow up on the better grades, widths and strike lengths encountered during the development mining on the 3630 and 3680 extraction levels at the western extent of the Dick Zone. The program did not confirm any continuity at depth and as a result the resources on the western extent of the Dick Zone were not translated into reserves.

The mineralization found at surface occurs in two main zones known as the Toots and Dick. At depth, between the 1030 and 1280 Levels, the Dick Zone appears to split into two separate zones (Dick and Ruttan) with four zones occurring on the 1780 Level.

The zones which occur on the 1780 Level are, from west to east, the Toots, Dick, Ruttan and Hogg Zones.

On the 1780 Level the Ruttan and Hogg Zones are separated by an area of weak mineralization approximately 100 ft (30.5 m) long. However, on the 2010 Level the mineralization is continuous between the two zones. Below the 2010 Level, the Ruttan and Hogg Zones merge into one zone down to the 2300 Level where, the two zones were combined into one zone (Ruttan) for interpretation and mining purposes.

Below the 1780 Level, the Toots Zone is terminated by the McLeod Road Thrust fault and only the Dick, Ruttan and Hogg Zones continue to depth.

Due to the orientation of the mineralization, a series of four grids were used at the Nor-Acme/NBM in order to create true sections of the different zones. The four grids are the Toots, Dick (Main Mine and North-West), and Ruttan.

The Toots Grid was a northeast to southwest trending grid used for the upper portions of the mine above the 1030 Level and the Toots crown pillar. Below the 1030 Level this grid is no longer used and the Lower Toots Zone (1280 Level to 1780 Level) uses the Dick (Main Mine) Grid, as it provided a more accurate interpretation.

The Dick Grid is also called the Main Mine Grid and is oriented north to south. However, below the 3480 Level at the western end of the Dick Zone, close to the McLeod Road Thrust fault, the mineralization changes strike direction from an east-west orientation to a northwest orientation and starts to parallel the fault. The direction change occurs to a lesser extent in the Dick Zone between the 3000 Level and 3480 Level, as well as in the Toots Zone between the 1530 Level and 1780 Level. However, only below the 3480 Level does the northwest strike length of the mineralization become extensive enough to require the creation of a new grid.

In response to the strike change of the mineralization, a second Dick Grid was created in the summer of 2004. The grid lines for this new grid are oriented at 67° east of north and allow for the northwest trending portion of the Dick Zone to be drilled and interpreted approximately perpendicular to its strike direction. The eastern portion of the Dick Zone continued to use the north-south orientated Dick Zone (Main Mine) grid for drilling layouts and sectional interpretation.

The Ruttan Grid is a northwest to southeast trending grid with the gridlines oriented 40° east of north. Between the 2456 and 3000 Levels, the Ruttan Zone starts to undertake a 30° change in strike direction. Between the 2456 Level and 2986 Level, the directional change created the illusion that the zone was flattening as the directional change caused the main portion of the Ruttan Zone to trend in an east-west strike orientation (Dick Grid) atthe 2986 Level. Therefore, for a short period of time both the old Ruttan Grid and the Dick Grid were used for interpretation. The Ruttan Grid was used from the start of the zone up to Ruttan Grid Line 10250E; after this point the Dick Grid was used for the remainder of the zone. This practice was

discontinued for the main mineralized lens below the 2986 Level, as the Ruttan Grid is obsolete for this lens below this level. The exception to this practice was the Ruttan Footwall Lens, which remained in the original orientation of the Ruttan Zone.

The Ruttan Grid is also used for the Hogg Zone interpretation. At the present time this only applies to any mineral inventory in the Upper Hogg Zone above the 1780 Level as the Hogg Zone has been mined out between the 1780 and 2300 Levels. Below the 2010 Level the Hogg and Ruttan Zones mineralization constituted a single zone and for the purpose of estimating the mineral resources, the Hogg Zone was combined with the Ruttan Zone below the 2300 Level.

NBM achieved a number of major accomplishments during its short production period from 1994 and 2004. These accomplishments are:

- 1996, Provincial Mine Rescue Champions
- 1996, One Year No Lost Time Accidents
- 1997, Provincial Mine Rescue Champions
- 1997, John T. Ryan Award – Western Canada Regional Winners
- 1997, Two Years No Lost Time Accidents
- 1997, One Million Manhours, No Lost Time Accidents (August)
- 1998, John T. Ryan Award – Canadian Metal Mines Winner
- 1998, Provincial Mine Rescue Technician Award.
- 1998, Three Years No Lost Time Accidents
- 1999, John T. Ryan Award – Western Canada Regional Winners
- 1999, Provincial Mine Rescue Competition Runner-up.
- 1999, Two Million Manhours, No Lost Time Accidents (June)
- 1999, Worked 1,386 Days No Lost Time (3.8 years)
- 2000, OHS Award of Excellence – Large Workplace
- 2002, Provincial Mine Rescue Technician Award
- 2002, John T. Ryan Award – Western Canada Regional Winners
- 2003, Provincial Mine Rescue Champions
- 2003, John T. Ryan Award – Canadian Metal Mines Winner
- 2004, One Year No Lost Time Accidents
- 2004, Provincial Mine Rescue Technician Award

Figure 6-1 is a longitudinal projection of NBM showing the present extent of the known ore zones, projected property boundaries, and mined-out areas as of January 27, 2005.

In October, 2006 a NI 43-101 Technical Report on the NBM property was completed for Garson Resources, Pegasus and Piper by Messrs. Lewis, P. Geo. and Gowans, P. Eng., of Micon. Thereafter, Garson Resources and Piper acquired 100%-interest of the NBM property in December, 2006, and commenced detailed drilling of the existing mineral resources at the No. 3 Zone at the end of February, 2007.

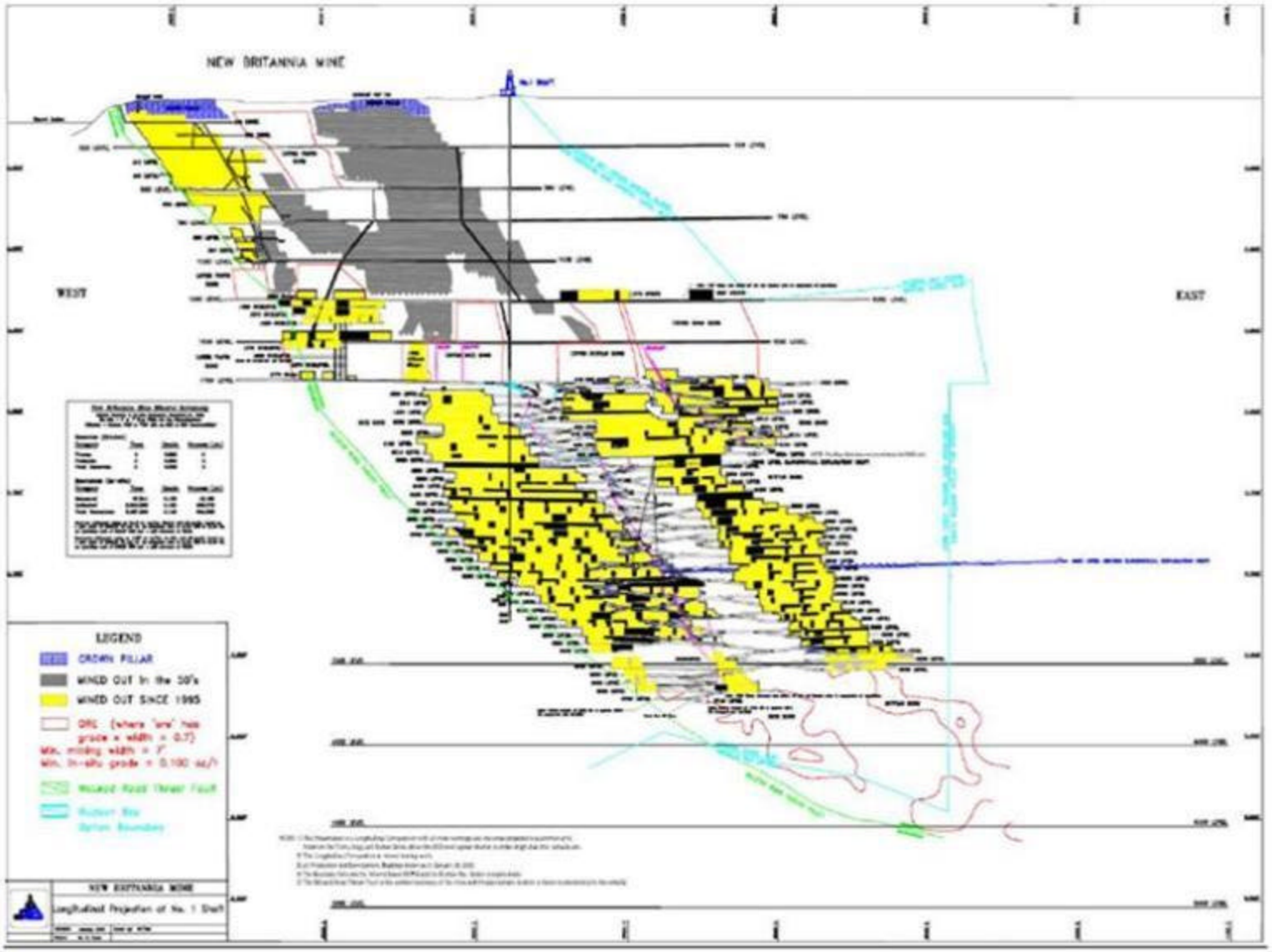

Figure 6-1: Longitudinal Projection of New Britannia Mine

Currently, Garson (formed through the amalgamation of Piper and Garson Resources in June, 2007) owns 100% of the SLM property.

6.2 No. 3 zone and birch zone (mineral lease ml-323)

The following historical exploration and production activities have been performed on the Mineral Lease ML-323. However, for the purposes of this report, all the resources and reserves estimated prior to February 1, 2001 should be considered as historical estimates as defined by NI 43-101 regulations and should not be relied upon.

Early records indicate that the property was explored for gold as early as the 1920s in the area of No. 3 Zone and that by 1944 gold mineralization had been located in the area of the Birch Zone.

During the period from 1942 to 1945, the area was explored by Northern Canada Mines and Pioneer Gold Mines. In early 1946, Koona Lake Mines Limited was formed to take over the exploration and development on most of the Snow group of claims which covered the area of No. 3 Zone. It was during this period that mapping, trenching and diamond drilling resulted in the discovery of the No. 3 Zone. No. 3 Zone is named such because it was the third showing on the Snow group of claims. No results of the drilling appear to have survived in the literature.

From the late 1940s to 1964 there is not much information published on this area and a dormant phase of interest occurs.

In 1964, Mr. J. McNevin recorded the November 1 mineral claim covering the area. The claim was optioned to the Canadian Nickel Company Limited in 1965 and it performed a ground magnetometer survey. The claim was cancelled in 1970.

In 1971, W. Bruce Dunlop Limited (Dunlop) staked mineral claim CB3881 and performed exploration on the mineral claim for several years. Granges Exploration Aktiebolag (Granges) optioned this claim in 1980 to 1981 and conducted line cutting, soil sampling an electromagnetic survey, and diamond drilling for base metals.

The property was transferred to Darius Gold Mine Incorporated in 1982 and a year later it was transferred to Gold Fields Canadian Mining Limited (Gold Fields) which carried out line cutting, VLF-EM, IP, and magnetic geophysical surveys, a humus geochemical survey and diamond drilling. Gold Fields drilling tested the No. 3 Zone to a depth of about 900 ft (274.3 m), but the results of the drilling have not survived and there are only passing references to this program.

In 1985 and 1986 Hudson Bay optioned the property and drilled approximately 3,406 m. After the drilling program was completed, Hudson Bay estimated that the “drill indicated reserves” for No. 3 Zone were 200,000 t grading 15.09 g/t gold (Richardson, 1996). Hudson Bay dropped the option on the property after one year.

In 1986, Silver Hart Mines Limited (Silver Hart) acquired the property under an agreement with Dunlop. Silver Hart created Snow Lake Mines Ltd. (Snow Lake Mines) as a company to explore and develop the property and diamond drilling was conducted during both the 1987 and 1988 exploration seasons. In 1987, Snow Lake Mines announced "reserves" of 617,000 t grading 9.6 g/t Au for No. 3 Zone between the 122 m and 335 m Levels (Fedikow, 1989 and Bamburak, 1990). Table 6-3 is a summary of a portion of the drilling that occurred on No. 3 Zone between 1940 and 1988.

Table 6-3: Summary – Portion of Drilling on No. 3 Zone from 1940 to 1988

Company	Drill Holes	Footage, ft (metres, m)
Northern Canada Mines	49	4,987 ft (1,520 m)
Granges	7	920 ft (280 m)
Goldfields	107	61,728 ft (18,814 m)
Hudson Bay	28	10,346 ft (3,153 m)
Snow Lake Mines	124	71,490 ft (21,790 m
Total	**315**	**149,471 ft (45,559 m)**

Table provided by Garson Gold Corp.

In 1993, High River optioned the claims covering the No. 3 Zone and Birch Zone from Dunlop. Between 1993 and 1994, High River conducted drilling programs with the intention of proving up an economically viable deposit above the 500-ft Level, in the upper portion of the No. 3 Zone. In April 1994, Bharti noted in the Snow Lake feasibility study that No. 3 Zone had a "geological reserve" of 215,000 tons grading 0.295 Oz/ton 9.2g/ton (195,045 t grading 0.325 Oz/t, 10.1 g/t) gold down to the 400 Level and that drill holes between the 400 Level and 1150 Level indicated the potential for another 343,000 tons grading 0.28 Oz/ton 8.7 g/ton (311,164 t grading 0.309 Oz/t, 9.6 g/t) gold.

When TVX and High River formed the joint venture in 1994 to reopen the Nor-Acme Mine, the option on the claims was assigned to 1126774 Ontario as part of the joint venture.

In April, 1995, the mineral claims were converted into mineral lease ML-323. However, development and testing of the No. 3 Zone deposit had commenced with the first blast for the ramp in June, 1994. Access to the No. 3 Zone was via a portal and ramp with all material hauled to surface stockpiles and then trucked to the New Britannia mill. Mining accessed the ore along strike from the ramp and used mechanized undercut longhole retreat methods.

In 1995 and 1996, the No. 3 Zone was mined as feed for the New Britannia mill. During this period, No. 3 Zone produced 241,574 tons (219,152 t) at a grade of 0.126 Oz/ton 3.9 g/ton (0.139 Oz/t, 4.3 g/t) gold for a total of 30,335 feed Oz. Production ceased at No. 3 Zone in May, 1996 and no development or further

exploration was performed below the last mining level (450 Level). After 1996, No. 3 Zone was allowed to flood and during 1998 and 1999 the No. 3 Zone adit was sealed and reclamation was done intermittently on the surrounding area until it was completed in 2004.

From 1995 to 1997, various exploration programs of line-cutting, ground geophysical surveys, and diamond drilling were done on other showings of ML-323. The majority of the exploration funding during this period was devoted to exploring the extent of the Birch Zone located to the northwest of the New Britannia tailings pond.

In early 1996, a small bulk sample was extracted from the upper portions of the Birch Zone using open pit mining methods. The 1996 production from the Birch Zone pit amounted to 31,353 tons (28,443) grading 0.104 Oz/ton 3.2 g/ton (grading at 0.115 Oz/t, 3.6 g/t) gold for a total of 3,282 feed Oz.

In late 1997, it was determined from the diamond drilling programs that the Birch Zone would not be economically viable due to problems with the continuity of the mineralization. As a consequence, all surface exploration programs covering mineral lease ML-323 were curtailed in 1998 and exploration never recommenced on ML-323 prior to the closure of NBM in early 2005.

Exploration activities on mineral lease ML-323 were revived in early 2007 following the acquisition of 100% interest in the NBM property by Garson Resources and Piper in December, 2006. A program of diamond drilling that focused on No. 3 Zone and the surrounding area was commenced in February, 2007.

The aims of the drilling program were threefold, viz. (i) to confirm the known resources as detailed in the October, 2006 Technical Report, (ii) to upgrade the Inferred Resources to the Measured or Indicated categories, and (iii) to expand the resource base. As of end of April, 2008, a total of 29,000 m of drilling had been completed by Garson. The resources based on this drilling were discussed in a report entitled “NI 43-101 Technical Report, Audit of the No. 3 Zone Mineral Resource Estimate on the New Britannia Mine Property, Snow Lake, Manitoba”, dated July 20, 2008.

Between May and October, 2008, Garson continued to drill and explore the SLM property and completed a further 6,634 m in 39 drill holes on the Main Mine (Hogg Zone), Boundary and Kim Zones. A total of 1,403 m was conducted on the Main Mine (Hogg Zone), 2,100 m on the Boundary Zone and 3,131 m on the Kim Zone.

6.3 Hudson bay mineral claims

The following historical exploration work has been conducted on various portions of the mineral claims which were part of the Hudson Bay option until Garson exercised the option.

Between 1948 and 1958, limited exploration was conducted to the east of the Nor-Acme shaft (New Britannia shaft) from both underground and surface by Howe Sound and a portion of the old track drifts on the 320, 780 and 1280 Levels are now situated on the Bud 1 (P6557B) and Bud 2 Fr. (6558B) mineral claims, which are part of the Hudson Bay option.

From 1968 to 1991, Hudson Bay and other companies performed geophysical and geochemical surveys and drilled numerous holes on the Bud claims. Hudson Bay was primarily looking for base metal mineralization while the other companies were interested in the Bounter Zone.

Geosearch Consultants Ltd. (1979, electromagnetic survey), Northwest Geophysics Ltd. (1982, magnetometer survey) and Brad Koop Exploration Services Inc. (1989, Phoenix VLF-2 GSM-18 MaxMin 11+ survey) all conducted surveys for Hudson Bay on the Hudson Bay mineral claims.

In 1995, High River and Hudson Bay Mining and Smelting Company Ltd. (HBM&S), through its subsidiary Hudson Bay, entered into an option agreement covering certain mineral claims that were contiguous to the mineral leases and Sky claims held by the joint venture. The HBM&S claims lie along the Sky mineral claims southern and eastern boundaries and covered a portion of the eastern workings of the original Nor-Acme Mine and the eastern extent of the Howe Sound fault. In June, 1997, this option agreement was amended and restated to better accommodate some of the concerns the joint venture parties.

After the option agreement with Hudson Bay was signed, TVX and High River completed a line-cutting program and geophysical program over a portion of optioned claims and diamond drilling was conducted on the Bounter Zone. In 1996, compilation of the previous work on the Hudson Bay option was completed and in 1997, Jasper Exploration completed a line-cutting program and a geophysical (Mag./VLF) survey over the new grid. A general prospecting program was also completed during the summer of 1997.

Between 1998 and 2000, exploration programs were curtailed on the Hudson Bay option due to the low gold prices and the option agreement was renegotiated to reflect the reduction in exploration work during this period.

In 2000, underground exploration drilling was performed on the Bud 12455 (CB12455) mineral claim from the first 3000 Level hangingwall exploration drift located within the

NBM. A report for this work was filed with the Mines Branch of the Manitoba Ministry of Industry, Trade and Mines for assessment credits totalling CAN$ 200,601.41. This work went towards fulfilling the obligations contained in the Hudson Bay option by the joint venture partners TVX and High River.

Limited surface exploration on the Hudson Bay option was re-commenced in 2002 with a drilling program focused on identifying any mineralized zones located to the east of the shaft. The last exploration drilling in this area dated from the original Nor-Acme production period which had identified areas of mineralization that were uneconomic during the 1950s. Between January and March, 2002, 6,216.6 ft (1,894.73 m) of surface drilling was conducted on the Bud 1 mineral claim. The surface drilling program was approved for inclusion within the Manitoba Mineral Exploration Assistance Program (MEAP) in January, 2002, and assigned the identification number 011101-01. A report recording the details of this work was filed with the Mines Branch for assessment credits totalling CAN$ 160,396.62.

At the same time as the surface drilling program an underground diamond drill drift was being excavated in the hangingwall of the Dick and Ruttan mineralized Zones on mineral claim Bud 12455 (CB12455) on the 3000 Level in the NBM. The drill drift was an extension of the existing diamond drill drifts and a report was filed with the Mines Branch for assessment credits totalling CAN$ 92,415.44.

Between November 1, 2002 and March 31, 2003, an underground drilling program was performed on the Bud 12455, Bud 12456 and Bud 12593 mineral claims and on mineral lease ML-61. This project was approved for inclusion in the fall 2002 MEAP in January, 2003 and was assigned the identification number 021101-30. A report detailing the results of this program was filed with the Mines Branch for assessment credits totalling CAN$ 367,751.23.

From the beginning of January to May 31, 2003, the company continued to establish the 3000 Level diamond drill drift on the Bud 12455 (CB12455) and Bud 12456 (CB12456) mineral claims and filed a report with the Mines Branch for assessment credits totalling CAN$ 446,557.22.

Between April 1, 2003 and October 31, 2003, underground drilling totalling 40,488 ft (12,340.14 m) of BQ and NQ wireline drilling was conducted on the Bud 12455 (CB12455) and Bud 12456 (CB12456) mineral claims. MEAP approval for this project was obtained in June, 2003 and assigned the identification number 030401-25. Upon completion of the project, a two volume report was prepared on the project and recorded with the Mines Branch for assessment credits totalling CAN$ 648,734.36. This was the last major underground drilling program conducted on the Bud mineral claims and excerpts from the recommendations and conclusions are quoted below.

> "The drilling program conducted between April and October, 2003 on the Bud mineral claims was laid out to investigate both the trend and potential economic viability of the mineralization below the 3800 Level

and to test the potential of a parallel mineralized lens located in the hangingwall of the Ruttan Zone on the 3000 Level. The drilling program was comprised of 23 diamond drill holes totalling 40,488 ft (12,340.14 m)."

"All of the diamond drill holes, with the exception of two holes (3000- 1289 & 3000-1501) intersected varying degrees of mineralization. Drill hole 3000-1289 in the Ruttan Zone did not intersect any mineralization and drill hole 3000-1501 intersected its target (the parallel hangingwall lens) but no visible mineralization was present."

"Of the nineteen diamond drill holes that were drilled to investigate the Ruttan Zone below the 3800 Level, eighteen intersected mineralization that varied in grade from a low of 0.031 Oz/ton gold over a calculated true width of 2.1 ft (0.6 m) in hole 3000-1375 to a high of 0.270 Oz/ton gold over a calculated true width of 1.6 ft (0.5 m) in hole 3000-1262. Four holes (3000-1361, 3000-1371, 3000-1372, and 3000-1380) intersected significant intervals of mineralization, ranging in grade from 0.126 Oz/ton gold over a calculated true width of 13.8 ft (4.2 m) in hole 3000-1361 to 0.260 Oz/ ton gold over a calculated true width of 16.0 ft (4.9 m) in hole 3000- 1372."

"The results from the eighteen drill holes indicate, that the mineralization of the Ruttan Zone does continue to dip and plunge onto the Bud mineral claims below the 3800 Level and east of the present mine workings. However, due to the variability in grade and width of the Ruttan Zone encountered within the drill holes the zone is uneconomic to mine at the present time."

"The four diamond drill holes which intersected the parallel hangingwall lens in the Ruttan Zone varied in grade from a low of 0.001 Oz/ton gold over a calculated true width of 10.3 ft (3.1 m) in hole 3000-1501 to a high of 0.208 Oz/ton gold over a calculated true width of 2.2 ft (0.7 m) in hole 3000-1300. The weighted average for all four of the parallel hangingwall lens drill holes is 0.053 Oz/ton gold over a calculated true width of 4.6 ft (1.4 m)."

"The four holes drilled into the hangingwall of the Ruttan Zone indicated that the mineralization within the potential hangingwall lens was discontinuous and uneconomic to mine."

"Due to the uneconomic nature of the Ruttan Zone below the 3800 Level and the failure to prove continuity to the mineralization within the hangingwall lens at the 3000 Level, it is recommended that no further work be conducted on the Bud mineral claims from the 3000 Level exploration drift"

A summary of the results for the 2003 underground drilling program on the Hudson Bay option is contained in Table 6-4.

On December 31, 2007, Garson exercised the option on the Hudson Bay mineral claims by making the CAN$ 400,000 option payment. The mineral claims are now 100%-owned by Garson, subject to a royalty consisting of 1.5% of the gross proceeds from the sale or disposition of all metals mined and removed from the claims. Prior to any production mining, the claims would have to be either incorporated into the existing mineral lease ML-61 or incorporated into a new mining lease.

Garson completed a detailed aeromagnetic survey of the greater region in 2007. On the basis of this survey, a number of exploration targets have been defined which remain to be tested.

6.4 Squall lake mineral claims

The only gold deposit which is near the NBM property and not on the property itself is the Squall Lake deposit located approximately 4 km northwest of the community of Snow Lake. This property adjoins mineral lease ML-323 along the northwestern boundary of the lease and consists of 14 contiguous mineral claims totalling 899 ha. Garson owns 100% of the Squall Lake property and the property has been the subject of an NI 43-101 report entitled “Report on the Squall Lake Property, The Pas Mining Division, Snow Lake, Manitoba” by D. Beilhartz, dated April, 2006 and posted on the System for Electronic Document Analysis and Retrieval (SEDAR) website.

Table 6-4: Summary of the results for the Underground Drilling Program from the 3000 Level (Hudson Bay Mineral Claims)

Diamond Drill		Diamond Drill Hole Location							Mineral Intersections (Imperial Units)					Mineral Intersections (Metric Units)					Mining Level Intersected[4]
Hole Number	Core Size	Mineral Lease/Claim[1]	Zone[2]	Drill Station (Easting)	Azimuth (°)	Dip (°)	Length (feet)	Length (metres)	From	To	Core Length (feet)	True Width (feet)	In-Situ Grade (oz/ton)	From	To	Core Length (metres)	True Width (metres)	In-Situ Grade (g/t)[3]	
3000-1243	BQ	Bud 12455 (CB12455)	Rattan	39201	169	-67	1,012.4	308.6	946.8	949.8	3.0	3.0	0.070	288.6	289.5	0.9	0.9	2.40	3861
3000-1289	NQ	Bud 12455 (CB12455)	Rattan	39427	172	-58	1,251.6	381.5	No Intersection									------	------
3000-1290	NQ	Bud 12455 (CB12455)	Rattan	39428	166	-71	1,208.9	368.5	1,037.6	1,040.3	2.7	2.7	0.039	316.3	317.1	0.8	0.8	1.34	3952
3000-1291	NQ	Bud 12455 (CB12455)	Rattan	39431	168	-84	1,356.6	413.5	1,100.0	1,104.2	4.2	4.2	0.044	335.3	336.6	1.3	1.3	1.51	4034
3000-1297	BQ	Bud 12455 (CB12455)	Rattan	39201.5	060	-79	1,357.6	413.8	1,129.5	1,133.8	4.3	3.8	0.193	344.3	345.6	1.3	1.2	6.62	4063
3000-1298	BQ	Bud 12455 (CB12455)	Rattan	39200.5	169	-80	1,062.3	323.8	987.5	993.8	6.3	6.3	0.161	301.0	302.9	1.9	1.9	5.59	3917
3000-1299	BQ	Bud 12455 (CB12455)	Rattan	39201	176	-5	1,356.3	413.4	1,207.9	1,212.9	5.0	3.0	0.108	368.2	369.7	1.5	0.9	3.70	3900
3000-1300	BQ	Bud 12455 (CB12455)	Rattan	39424.5	183	1	1,651.2	503.3	1,240.8	1,243.9	3.1	2.2	0.208	378.2	379.1	0.9	0.7	7.13	2872
3000-1358	NQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39426	013	-75	1,671.5	509.5	1,331.2	1,336.2	5.0	4.3	0.162	405.7	407.3	1.5	1.3	5.55	4229
3000-1361	BQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39202	021	-74	1,721.1	524.6	1,283.7	1,306.2	22.5	13.8	0.126	391.3	398.1	6.9	4.2	4.32	4292
3000-1362	BQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39202	015	-67	1,908.1	581.6	1,441.9	1,444.2	2.3	1.0	0.213	439.5	440.2	0.7	0.3	7.30	4380
3000-1363	BQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39204	010	-60	2,074.6	632.3	1,969.9	1,975.6	5.7	1.5	0.058	600.4	602.2	1.7	0.4	1.99	4610
3000-1366	NQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39428	006	-61	2,243.5	683.8	1,757.6	1,763.8	6.2	2.5	0.134	535.7	537.6	1.9	0.8	4.59	4505
3000-1370	BQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39201	010	-56	2,301.5	701.5	2,152.0	2,156.5	4.5	1.0	0.158	655.9	657.3	1.4	0.3	5.42	4708
3000-1371	BQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39202	023	-81	1,308.0	398.7	1,155.0	1,169.5	14.5	9.4	0.229	352.0	356.5	4.4	2.9	7.85	4092
3000-1372	NQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39428	004	-57	2,144.0	653.5	1,957.5	1,981.0	23.5	16.0	0.260	596.6	603.8	7.2	4.9	8.91	4610
3000-1373	BQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39200.5	010	-62	1,908.0	581.6	1,708.2	1,711.5	3.3	1.2	0.142	520.7	521.7	1.0	0.4	4.87	4450
3000-1375	NQ	Bud 12456 (CB12456)	Rattan	39801	006	-59	2,580.8	786.6	1,961.2	1,966.6	5.4	2.1	0.031	597.8	599.4	1.6	0.6	1.06	4620
3000-1376	BQ	Bud 12455 (CB12455); Bud 12456 (CB12456)	Rattan	39200.5	010	-68	1,603.3	488.7	1,445.9	1,449.0	3.1	1.6	0.270	440.7	441.7	0.9	0.5	9.26	4292
3000-1380	NQ	Bud 12456 (CB12456)	Rattan	39800.5	006	-52	2,813.0	863.5	2,274.1	2,336.7	62.6	30.0	0.146	693.1	712.2	19.1	9.1	5.01	4760
3000-1391	NQ	Bud 12456 (CB12456)	Rattan	39800.5	006	-52	2,636.1	803.5	2,481.9	2,496.7	14.8	6.0	0.036	756.5	761.0	4.5	1.8	1.23	4842
3000-1501	BQ	Bud 12455 (CB12455)	Rattan	39425	179	-1	1,603.7	488.8	1,445.9	1,465.9	20.0	10.3	0.001	440.7	446.8	6.1	3.1	0.03	2829
3000-1502	BQ	Bud 12455 (CB12455)	Rattan	39425	179	-8	1,459.2	444.8	1,265.2	1,270.0	4.8	3.0	0.063	385.6	387.1	1.5	0.9	2.16	3042

Table provided by Garson Gold Corp.

Note 1: Hole Trace indicates on which claim or mineral lease the hole was collared and coverage of the hole trace.
i.e. Bud 12455 (CB 12455) indicates that the hole was collared and is entirely contained within the mineral claim. The designation Bud 12455, Bud12456, ML-61 indicates that the hole was either angled to intersect the mineralization on another claim or drill hole deviation caused the hole to wander onto the mineral lease.

Note 2: Mineral Zone indicates which mineralized zone (Dick or Rattan) believed to have been encountered by the drill hole.

Note 3: The Conversion used for "ounces per short ton to grams per tonne" is:

Ounces to Grams = 31.1035
Short tons to Metric tons = 0.9072
Oz/ton to g/t = 31.1035/0.9072 = 34.2857

Note 4: Mining Level Intersected, this column indicates the potential mining level where the mineralization was intersected, in imperial units of measure.

Note 5: Any discrepancy between the total drilled footage contained in this table and the total footage claimed for assessment and MEAP purposes is because some holes were in progress prior to April 1, 2003 and only the footage drilled after April 1, 2003 is eligible for financial assistance under the terms of the April to October, 2003 offering of the Manitoba Exploration Assistance Program (MEAP). Any drill footage completed prior to April 1, 2003 would have been claimed under the previous MEAP offering.

While the Squall Lake property is located adjacent to the NBM property, the information regarding the Squall Lake deposit is not necessarily indicative of the mineralization found on the NBM property. Micon has not independently verified any of the information on the Squall Lake property.

The first mineral claims over the Squall Lake deposit were recorded by A. Peberdy in 1924 and he held the claims until 1926. Between 1926 and 1944, numerous parties held the claims but no substantial work was recorded during this period. In 1944, Wekusko Consolidated Limited (Wekusko) staked and optioned numerous claims in the area. Wekusko carried out a 8,309 m drilling program and received assays ranging from 3.09 g/t gold to 14.4 g/t gold over 0.76 m to 2.75 m. In 1945 Squall Lake Gold Mines Limited acquired the claims and conducted an additional 7,672 m of diamond drilling. Further trenching and sampling were performed in 1948. In 1948 leases were issued but very little work appears to have been done on the property prior to 1977 when the leases lapsed.

In 1977, Stan Major staked part of the property and it was subsequently transferred to Eldorado Exploration Ltd., Corporate Oil and Gas Ltd., W. Bruce Dunlop Ltd. (NPL) and Camflo Mines Ltd. (Camflo). In 1978, Hudson Bay staked the area previously known as the Moon/Gertie Zone.

Magnetometer, electromagnetic, VLF-EM and induced polarization surveys as well as mapping and diamond drilling were carried out. In 1984, Camflo announced “drill indicated reserves” of 1,088,621 t grading 6.85 g/t gold (Fedikow, 1989). In 1984, Camflo amalgamated with Barrick Resources Corporation (Barrick). Also in 1984, Barrick and Zenco Resources Inc. (Zenco) undertook a further 5,000 m of diamond drilling and outlined near surface mineralization totalling 680,000 t grading 3.43 g/t gold. Zenco acquired 100% interest in the property in 1986 and conducted 3,658 m of diamond drilling in 1987. In 1987, Zenco estimated that the property contained an “inferred” mineral resource of 156,000 t grading 5.04 g/t gold and 112,000 t grading 4.34 g/t gold in the Lower and Upper Silicified Zones (Beilhartz, 2006). Zenco changed its name to Solidor Resources Inc. (Solidor) in 1988 and later changed its name again in 1999 to Sheffield Resources Inc. (Sheffield).

In 1988, Graham Gold Mining Corp. (Graham) acquired an interest in the property and in 1989 conducted a mini-bulk sample on the Margaret South occurrence. Graham estimated an inferred mineral resource of 70,000 t grading 4.73 g/t in the Upper Silicified Zone and 210,000 t grading 5.35 g/t in the Lower Silicified Zone. Further exploration programs were conducted between 1989 and 1990 which consisted of surface stripping at the F1, Margaret, Margaret South, Margaret Extension and K7 zones and 11 diamond drill holes totalling 1,067 m. Graham was unable to fulfill its option obligations and the property was returned to Solidor.

A private corporation, Tri-Energy Inc. (Tri-Energy) acquired 100% of the Squall Lake property from Sheffield in 1999.

In 2002, Mighty Beaut Minerals Inc. (Mighty Beaut) obtained an option to earn a 100%-interest in the Squall Lake property from Tri-Energy. Mighty Beaut commissioned an NI 43-101 report from OreQuest Consultants Ltd. (OreQuest) which was written by George Cavey. According to the author of the OreQuest report "The various diamond drilling programs at the Margaret Extension area have outlined an Inferred Mineral Resource which stands at 100,000 t grading 4.85 g/t gold in the Upper Silicified Zone and 337,000 t grading 5.40 g/t gold in the Lower Silicified Zone. The historic resource figures generated by others have been reviewed and redefined to conform to the CIM approved standards as required in NI 43-101." Mighty Beaut was unable to complete its obligations as required by the option agreement and the property was returned to Tri-Energy.

In 2003, Coniagas Resources Ltd. (Coniagas) optioned the property, re-established the former grid and conducted a 19 hole drilling program totalling 1,447.5 m. The property was returned to Tri-Energy when Coniagas was unable to meet further obligations under the option agreement.

Garson Resources was created from the amalgamation of Tri-Energy and its wholly-owned subsidiary, Garson Resources, effective November 8, 2005 under amalgamation number BC0739911.

On June 28, 2007, Piper and Garson Resources merged to form Garson.

6.5 HISTORICAL PRODUCTION

6.5.1 Nor-Acme/New Britannia Mine Production

The mineralized zones which comprise the main deposit on the NBM property have witnessed two periods of production during the last 60 years. The original period of production lasted from 1949 to 1958 when the mine was closed for economic reasons. This period of production saw not only the inception of mining on this deposit but the establishment of a new mining community in Manitoba, Snow Lake. Between 1949 and 1958, the Nor-Acme Mine produced a total of 5,393,970 tons (4.893.327 t) at an average grade of 0.142 Oz/ton 3.2 g/ton (0.115 Oz/t, 3.6 g/t) for a total of 763,254 Oz of mill feed. Of the total feed Oz the mill recovered, 610,458 Oz of gold.

The Nor-Acme Mine used blasthole mining with nearly 100% extraction with no backfilling. From 1949 to early 1951, all blasthole drilling was done with diamond drills using AX coring bits. In early 1951, the mine discontinued the use of diamond drills and began using percussion drills with sectional rods and tungsten carbide insert bits. During this period of mining the Toots and Dick Zones were mined from the Dick Zone crown pillar to the 1530 Level. An exploration drift on the 1780 Level was developed prior to the mine closing in 1958 when the Ruttan and Hogg Zones were discovered but not developed.

The second period of mining began in 1994 with the commencement of dewatering in May, 1994, and levelling and construction at the mill and mine site. The official opening of the mine occurred on November 14, 1995 with the name changed to the New Britannia Mine.

Between 1995 and 2005, a total of 6,480,266 tons (435,690 t) were mined at an average grade of 0.132 Oz/ton 4.1 g/ton (0.146 Oz/t, 4.5 g/t) for a total of 858,075 feed Oz. During the production period from 1995 to 2005, the mill recovered a total of 794,492 Oz of gold.

Ramp access was available below he 1780 and the ore was mined using a mechanized undercut longhole retreat method. This method was applied to the three main mineralized zones below the 1780 Level namely the Dick, Ruttan and Hogg Zones. A modified version of undercut longhole retreat mining using jacklegs and captive equipment was applied between the 1280 and 1780 Levels in what was referred to as the Lower Toots Zone. In the Upper Toots Zone mining was conducted by conventional captive mining using a sublevel retreat method. Excavation of the mineralization occurred on the development levels by following the footwall of the deposit along strike to its economic limits. Figures 6-2 and 6-3 illustrade illustrate the undercut longhole retreat and sublevel retreat mining methods used at NBM in both longitudinal and sectional views.

In February, 2000, the Nor-Acme/NBM reached a milestone in gold production when the one millionth Oz, excluding production from satellite deposits, was poured.

During the last year of production (2004) the New Britannia mill throughput was 431,086 feed tons (391,075 t) grading 0.118 Oz/ton 3.7 g/ton (0.130 Oz/t, 4.0 g/t) gold, yielding 50,727 feed Oz Au. In 2004, a total of 49,909 Oz of gold was recovered which includes the Oz of gold recovered during the mill circuit cleanout in October. The annual underground mine production for 2004 was 446,335 tons(446,335 t) grading 0.118 Oz/ton 3.7 g/ton (0.130 Oz/t, 4.0 g/t) gold which totalled 52,549 feed Oz of gold.

Any yearly variation in tonnage between the mill production and the underground production was accounted for by a broken tonnage inventory within the mine. This involved the tonnage contained within the longhole stopes, ore passes, both the coarse and fine ore bins, and the occasional surface stockpile.

The New Britannia mill ceased production at the end of September, 2004. During the remaining portion of 2004, while the mine was undertaking an exploration program and reduced production, a surface stockpile consisting of approximately 22,000 tons (19,958t) was created.

In late January, 2005, the decision to completely close the mine and put the surface facilities on care and maintenance was taken by the joint venture partners.

As part of the site cleanup the stockpile generated during the fall of 2004 was processed by the mill between June 11 and July 4, 2005 with 22,714 tons (20,606 t) grading 0.102 Oz/ton 3.2 g/ton (0.112 Oz/t, 3.5 g/t) gold processed and 2,113 Oz of gold recovered.

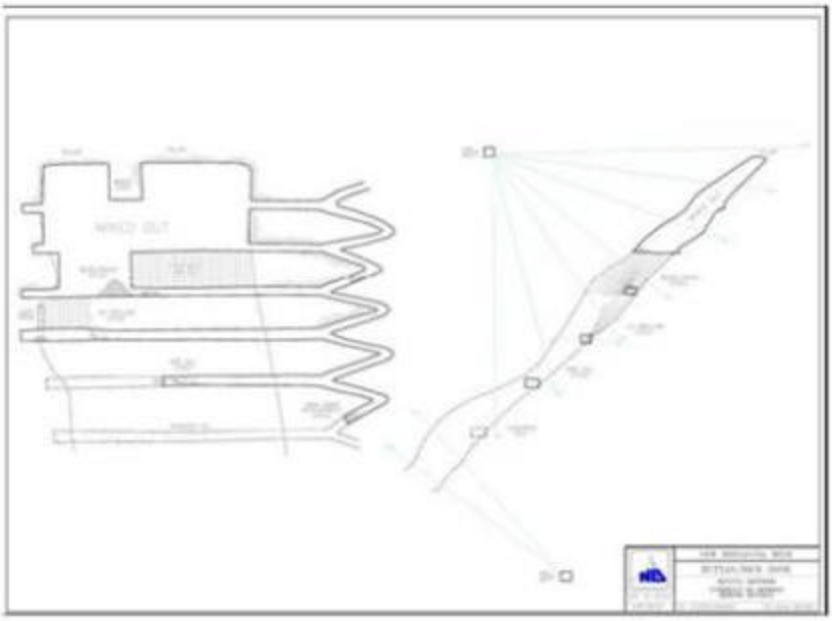

Figure 6-2: SubLevel Retreat Longhole Mining Method used by the New Britannia Mine

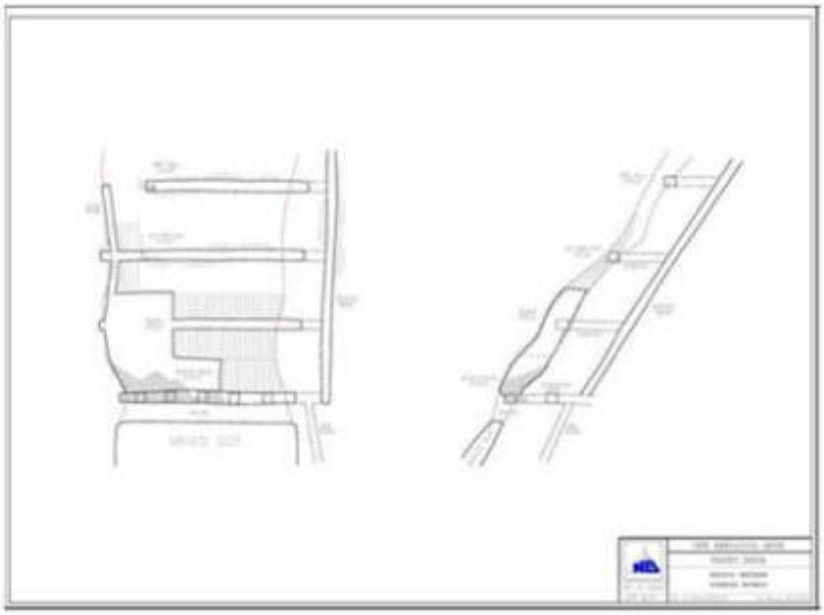

Figure 6-3: SubLevel Retreat Longhole Mining Method used by the New Britannia Mine

Table 6-5 summarizes the production statistics for the Nor-Acme/New Britannia Mine and does not include material from satellite deposits.

Table 6-5: Summary of the Nor-Acme/New Britannia Mine Production

Mine	Operating Years	Mill Feed Tons (dry)	Mill Head Grade (Oz/ton)	Mill Feed Oz	Recovered Oz
Nor-Acme	1949	362,454	0.143	51,832	37,671.9
	1950	664,153	0.139	92,317	75,188.2
	1951	702,092	0.125	87,763	69,810.0
	1952	540,000	0.154	83,160	65,891.5
	1953	522,673	0.158	82,582	67,377.4
	1954	540,030	0.149	80,465	67,661.3
	1955	611,732	0.134	81,972	65,849.8
	1956	567,692	0.145	82,315	63,979.9
	1957	587,559	0.138	81,083	63,816.6
	1958	295,585	0.135	39,765	33,211.4
Nor-Acme Total		5,393,970	0.142	763,254	610,458
New Britannia	1995	22,952	0.130	2,977	2,437.4
	1996	457,026	0.129	59,13	52,570.9
	1997	758,555	0.134	101,753	91,416.0
	1998	808,792	0.132	107,111	97,603.0
	1999	866,974	0.127	109,864	100,911.0
	2000	819,955	0.139	113,675	105,512.0
	2001	795,545	0.154	122,117	114,528.0
	2002	827,983	0.138	114,303	107,490.0
	2003	668,684	0.111	74,047	70,002.0
	2004	431,086	0.118	50,727	49,909.0
	2005	22,714	0.102	2,318	2,113
New Britannia Total		6,480,266	0.132	858,075	794,492
Total Production		11,874,236	0.137	1,621,329	1,404,950

6.5.2 No. 3 Zone and Birch Zone Production

The No, 3 Zone ramp was started In June, 1994. Access to the No. 3 Zone was via a portal and ramp with all material hauled to surface stockpiles and then trucked to the New Britannia mill. The mining accessing the ore along strike and used mechanized undercut longhole retreat methods.

In April, 1995, the mineral claims were converted into mineral lease ML-323.

The ore from the No. 3 Zone was used to comission the New Britannia mill and the first gold bullion was poured in November, 1995. During 1995 and 1996, the No. 3 Zone produced 241,574 tons (219,152 t) at grade of 0.126 Oz/ton 3.2 g/ton (0.112 Oz/t, 3.5 g/t) gold for a total of 30,335 Oz. Production ceased at the No. 3 Zone in May, 1996 and no development or further exploration was done below the 450 Level. After 1996, the No. 3 Zone was allowed to flood and during 1998 and 1999 the portal was sealed and reclamation was performed intermittently until it was completed in 2004.

In early 1996, a small bulk sample was extracted from the upper portions of the Birch Zone using open pit mining methods. The 1996 production from the Birch pit amounted to 31,353 tons (28,443 t) grading 0.104 Oz/ton 3.2 g/ton (0.115 Oz/t, 3.6 g/t) gold for a total of 3,282 Oz.

In the summer and fall 2004 the Birch pit was filled in at a 3:1 slope and an oversize rock barricade was placed around the pit walls to prevent inadvertent access. The area has been revegetated and reclamation has been completed.

Table 6-6 summarizes the production statistics for both the No. 3 Zone and the Birch pit during 1995 and 1996. Figure 6-4 shows a longitudinal projection of No. 3 Zone showing the mined out area and the inferred resources remaining below the final level of mining.

Table 6-6: Summary of the No. 3 Zone and Birch Zone Production

Mine	Operating Years	Mill Feed Tons (dry)	Mill Head Grade (Oz/ton)	Mill Feed Oz	Recovered Oz
No. 3 Zone1	1995	116,681	0.126	14,702	12,056
	1996	124,893	0.125	15,633	13,226
No. 3 Zone Total		**241,574**	**0.126**	**30,335**	**25,281**
Birch Zone2	1996	31,353	0.104	3,282	2,777
Birch Zone Total		**31,353**	**0.104**	**3,282**	**2,777**
Total Production		**272,927**	**0.123**	**33,617**	**28,058**
Note 1: Production ceased at No. 3 Zone in May, 1996					
Note 2: Production at the Birch Zone was limited to the spring and summer of 1996					
Table provided by Garson Gold Corp					

6.5.3 Squall Lake Production

There has been no known mineral production from the Squall Lake property.

6.5.4 Total Historic Production

Total historic production for the Nor-Acme/NBM since 1949 (including satellite deposits) has totalled 12,147,163 tons (11,019,721 t) at a mill feed grade of 0.136 Oz/ton 4.2g/ton (0.150Oz/t, 4.7g/t) of gold, for a total yield of 1,654,946 feed Oz of gold. Between 1995 and 2005, 1,048 doré bars of gold were produced with an average weight of 784.88 Oz. Table 6-7 contains the historical production statistics for the Nor-Acme/NBM.

6-4: Longitudinal Projection of No. 3 Zone

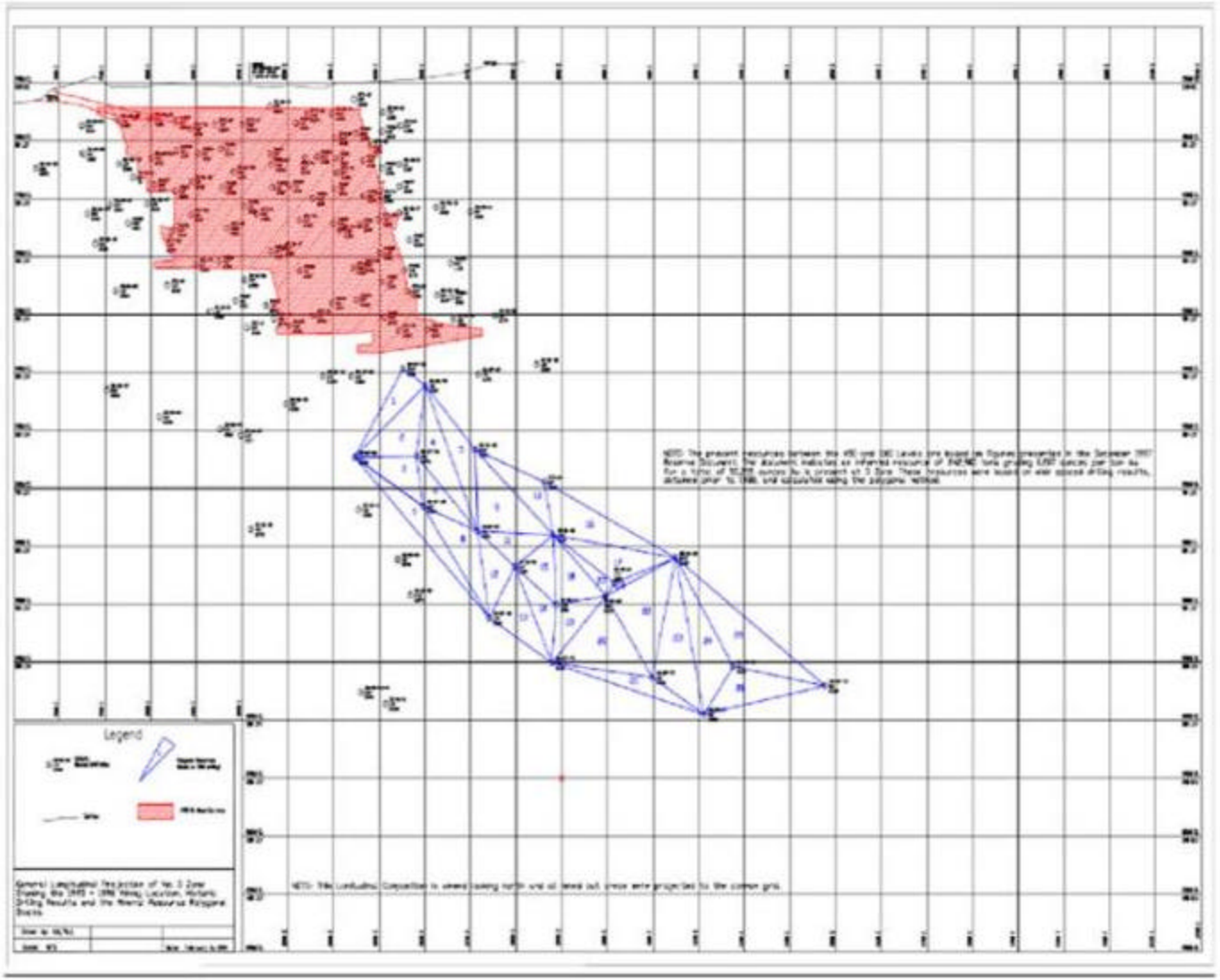

Table 6-7: Summary of Nor-Acme/New Britannia Mine Production including the Satellite Deposits (No. 3 Zone and Birch Zone)

Mining Area	Operating Years	Mill Feed Tons (dry)	Mill Head Grade (Oz/ton)	Mill Feed Oz	Recovered Oz
Nor-Acme mine	1949	5,393,970	0.142	763,254	
	1958				610,458
New Britannia mine	1995	6,480,226	0.132	858,075	
	2005				794,492
No. 3 Zone	1995-1996	241,574	0.126	30,335	25,281
Birch Zone	1996	31,353	0.104	3,282	2,777
Total Production		**12,147,163**	**0.136**	**1,654,946**	**1,433,008**
Table provided by Garson Gold Corp.					

Figure 6-5 shows a photograph of one of a gold pours at the New Britannia Mine during the last production phase from 1995 to 2004.

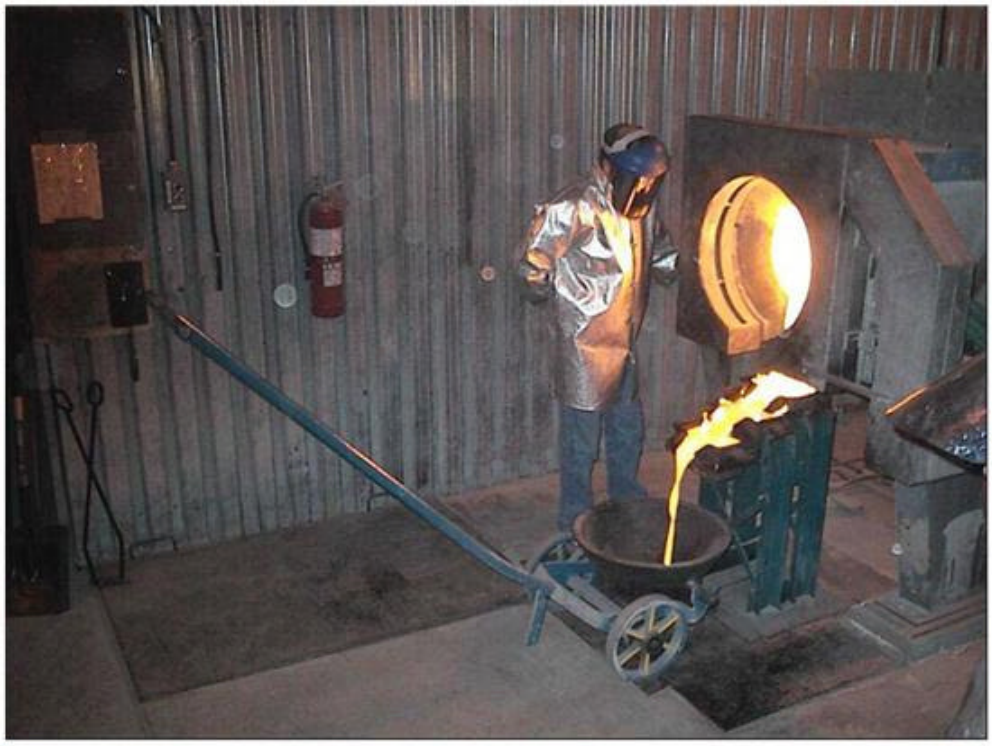

Figure 6-5: Gold Pour at the New Britannia Mine

6.6 Previous resource and reserve estimates

6.6.1 New Britannia Mine

The resource and reserve estimates discussed in this section were conducted according to the accepted resource classification definitions for the periods to which they refer. Prior to the resource and reserve estimate of December 31, 1997 the resource estimates did not comply with the CIM standards and definitions for estimating resources and reserves as required by NI 43-101. However, for the purposes of this report all resources and reserves estimated prior to February 1, 2001 should be considered as historical estimates as defined by NI-43-101 regulations and should not be relied upon.

The categories of the December 31, 1997 resource and reserve estimate were standardized in the TVX corporate format and were based on the CIM categories published in February, 1996. The CIM guidelines continued to be the basis for the resource and reserve categories until the mine was placed on care and maintenance in late January, 2005.

The resources at the NBM were last estimated as of December 31, 2004. Although these resources have not been the subject of a NI 43-101 Technical Report, they have been reported in the Kinross and High River Annual Reports for both 2004 and

2005. In its 2005 Annual Report, High River stated the stated the following "The New Britannia Mine is on care and maintenance as of January 2005. No diluted mineral reserves were estimated for December 31, 2005 because the mineral resources were unable to be translated into an economic mine plan."

Micon reviewed the December 31, 2004 resource estimate conducted by Kinross for Garson and Piper and, for the reasons stated in Section 17 of the October, 2006 Technical Report, Micon considered that the 2004 resource estimate was compiled in accordance of the NI 43-101 standards and remained relevant. It was reasonable for Garson to rely on the estimate as the basis to justify a program of compilation work and further exploration on the project. Between December, 1995 and December, 2004, the parameters and assumptions used in the resource and reserve estimations, as well as the estimations themselves were the subject of yearly internal "ore" reserve or mineral inventory reports. Copies are located at the NBM office in Snow Lake.

Table 6-8 summarizes the reserve estimates by year for the Nor-Acme/NBM during both production periods and notes the parameters on which the estimates were based. All reserves estimated prior to February, 2001 should be considered as historical estimates as defined by NI 43-101.

Table 6-8: Summary of the Nor-Acme/New Britannia In-situ Reserves and Diluted Reserve Estimates 1949 to 1958 and 1994 to 2005

Mine	Year	In-Situ Reserve Estimate			Diluted Reserve Estimate		
		Tons	Grade	Ounces	Tons	Grade	Ounces
Nor-Acme1	1949	--------	--------	--------	2,639,662	0.151	397,269
	1950	--------	--------	--------	2,603,862	0.152	395,266
	1951	--------	--------	--------	2,317,494	0.151	349,942
	1952	2,099,990	0.184	386,219	2,309,985	0.167	386,229
	1953	2,245,706	0.185	415,576	2,245,706	0.167	373,910
	1954	2,255,950	0.174	393,008	2,255,950	0.157	353,507
	1955	2,025,444	0.167	337,283	2,025,444	0.150	303,817
	1956	1,391,700	0.162	225,375	1,405,200	0.145	203,319
	1957	--------	--------	--------	873,797	0.143	124,953
	1958	--------	--------	--------	--------	--------	--------
	1994	4,286,036	0.189	811,053	4,436,047	0.165	729,947
	1995	4,197,001	0.186	781,670	4,630,506	0.157	727,868
	1996	3,366,386	0.176	592,644	4,157,486	0.135	530,013
	1997	3,605,098	0.177	638,646	4,452,296	0.136	606,716
	1998	3,159,249	0.188	593,221	3,971,245	0.145	576,439
New Britannia	1999	3,134,277	0.196	614,036	3,890,097	0.152	590,078
	2000	2,427,188	0.204	495,942	3,074,085	0.155	477,680
	2001	2,031,658	0.183	371,709	2,524,279	0.142	357,247
	2002	1,775,013	0.171	303,459	2,389,841	0.132	314,862
	2003	376,338	0.194	73,172	441,013	0.147	64,609
	2004	0	0.000	0	0	0.000	0
	2005	0	0.000	0	0	0.000	0

Note 1: 1949 - 1958 Data taken from the General Manager's Year-End Reports. The tonnage and dilution factors appear to have been floating figures.

Note 2: 1994 Resource figures used a five foot true mining width and 0.100 Oz/ton gold cut off for in-situ resources. Diluted reserves used a 15% dilution factor grading 0.000 Oz/ton gold with a 90% mining recoverability.

Note 3: 1995 Resource figures used a minimum seven foot horizontal mining width and 0.100 Oz/ton gold cut off for in-situ resources. Diluted reserves used a variable dilution factor (avg. 23%) grading 0.036 Oz/ton gold with a 90% mining recoverability.

Note 4: 1996 & 1997 Resource figures used a minimum seven foot in-situ true mining width and 0.100 Oz/ton gold cut off for in-situ resources (0.077 Oz/ton gold diluted cut off). Diluted reserves used 30% dilution factor grading 0.000 Oz/ton gold with a 95% mining recoverability.

Note 5: 1998 Resource figures used a minimum seven foot in-situ true mining width and diluted cut off grade of 0.094 Oz/ton gold. Diluted reserves used a variable dilution factor based on the addition of blocks of graded material, comprised of 2 ft of the footwall & 3 ft of the hangingwall added to the in-situ blocks, with a 90% mining recoverability.

Note 6: 1999 Resource figures used a minimum seven foot in-situ true mining width and diluted cut off grade of 0.080 Oz/ton gold. Diluted reserves were derived using the same method used in calculating the 1998 Reserves.

Note 7: 2000 Resource figures used a minimum seven foot in-situ true mining width and diluted cut off grade of 0.099 Oz/ton gold. Diluted reserves were derived using the same method used in calculating the 1998 & 1999 Reserves.

Note 8: 2001 Resource figures used a minimum seven foot in-situ true mining width and diluted cut off grade of 0.095 Oz/ton gold. Diluted reserves used a variable dilution factor based on the addition of blocks of graded material, comprised of 5 ft of hangingwall material added to the in-situ blocks, with a 90% mining recoverability.
Note 9: 2002 Resource figures used a minimum five or seven foot in-situ true mining width and diluted cut off grade of 0.096 Oz/t on gold. Diluted reserves used a variable dilution factor based on the addition of blocks of graded material, comprised of 5 ft or 7 ft of hangingwall material added to the in-situ blocks, with a 90% mining recoverability. The addition of 5 ft of hangingwall material is used for the Dick and Ruttan Zones below the 3000 Level and the addition of 7 ft of hangingwall material is used for the zones above the 1780 Level.
Note 10: 2003 Resource figures used a minimum five or seven foot in-situ true mining width and diluted cut off grade of 0.121 Oz/t on gold. Diluted reserves used a variable dilution factor based on the addition of blocks of graded material, comprised of 5 ft or 7 ft of hangingwall material added to the in-situ blocks, with an 80% or 90% mining recoverability. The addition of 7 ft of hangingwall material is used for the Dick and Ruttan Zones below the 3000 Level and the addition of 5 ft of hangingwall material is used for the zones above the 1780 Level.
Note 11: 2004 No figures are shown for the December 31, 2004 reserve estimates because the resource estimates as outlined were uneconomic. The mine entered its closure phase on January 27, 2005 but this was changed to a care and maintenance phase when interest was shown by third parties to acquire the mine and assets.

6.6.2 No. 3 Zone

Prior to Garson's work at the No. 3 Zone, the remaining inferred in-situ resource was estimated to be 242,900 tons (220,355 t) grading 0.207 Oz/ton 6.4 g/ton (0.228 Oz/t, 7.1 g/t) gold. The remaining inferred resources occurred both in the crown pillar and below the 450 Level (between the 500 and 1100 Levels). The resources between the 500 and 1100 Levels were estimated using the polygonal method and were originally reported in the December, 1995 ore reserve document for the NBM. No exploration was conducted at the No. 3 Zone after its closure in 1996.

The historical December, 1995 resource estimate for the No. 3 Zone has been superseded by Garson's 2008 resource estimate that was published in a Micon Technical Report, dated July 20, 2008.

6.6.3 Birch Zone

The exploration department estimated an inferred in-situ resource for the Birch Zone in 1997, using the polygonal method. These resources were not re-estimated by NBM. The 1997 inferred resource estimate totalled 290,071 tons (263,148 t) grading 0.183 Oz/ton 6.3 g/ton (0.202Oz/t, 6.3 g/t) of gold for the Birch area and 337,323 tons (306,014 t) grading 0.083 Oz/ ton 2.6 g/ton (0.091 Oz/t, 2.8 g/t) gold for the Birch Deep area.

To avoid any confusion arising from the separation of the Birch Zone resource estimations into the Birch and Birch Deep areas, the 1999 NBM ore reserve document combined the two areas under the heading "Birch Zone". The two areas were combined as any future study would have to account for the economic viability of all the Birch Zone.

Prior to the closure of the NBM in early 2005, no exploration programs had been conducted at the Birch Zone since December, 1997.

The historical 1997 Birch Zone estimate has not been superseded by a subsequent resource estimate. Garson should not rely on the historical Birch Zone resource until the historical estimate has been verified.

7. GEOLOGICAL SETTING (JL)

The description of the geological setting is taken directly from the technical report by Micon International Limited, dated May 30, 2009. Alexis is currently in the process of updating a description of the regional setting and geological history of the property.

The mineral deposits in the Snow Lake area are all located within the Aphebian Flin Flon-Snow Lake Greenstone Belt, which is situated in the south-eastern portion of the (exposed) Churchill Province of the Canadian Shield. The dimensions of the greenstone belt are approximately 250 km east-west by 50 km north-south, and it extends from 30 km east of Snow Lake, Manitoba to 50 km west of Flin Flon, Manitoba into Saskatchewan.

7.1 Regional geology

Historically the supracrustal rocks of the Flin Flon-Snow Lake Belt have been divided into the volcanic and derived sedimentary rocks of the Amisk Group and the unconformably overlying continental sedimentary rocks of the Missi Group (Richardson, 1996). Numerous felsic-mafic dykes, stocks and sills and granitic batholiths intrude the Flin Flon-Snow Lake Greenstone Belt. Figure 7-1 is a regional geology map of the Snow Lake area.

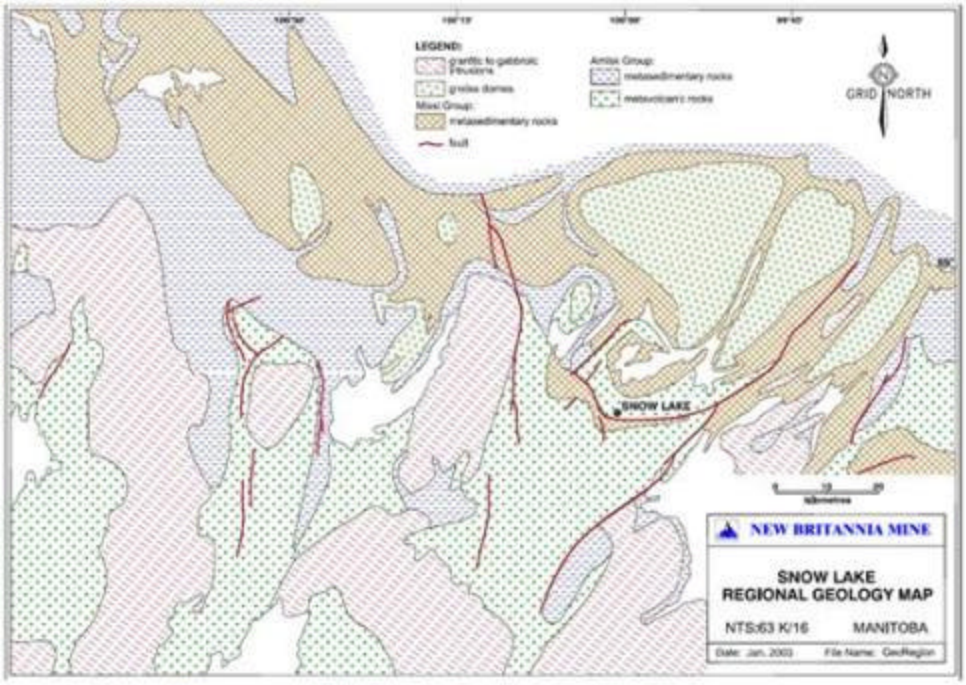

Figure 7-1: Regional Geological Map of the Snow Lake Area

7.1.1 Amisk Group

The Amisk Group in the File Lake-Snow Lake area comprises mafic to felsic volcanic rocks with intercalated volcanogenic sedimentary units that become dominant near the top of the succession. At Snow Lake, felsic volcanic rocks are as abundant as mafic volcanic rocks within the Amisk Group (Walford and Franklin, 1982). Approximately 1,000 m of Amisk Group volcanogenic sandstone/mudstone turbidites overly the volcanic rocks. The turbidites and substantial volume of felsic rocks, especially in the immediate vicinity of Snow lake, distinguish the Amisk Group in this area from that in the Flin Flon region.

Chemically, the Amisk Group volcanic rocks exhibit both tholeiitic and calc-alkaline affinities. The prevalence of pillowed lavas, mafic to felsic volcanism, submarine volcanoclastics and turbidite deposits and tholeiitic chemical affinity displayed by at least the altered mafic volcanic rocks suggest deposition in an island-arc tectonic environment.

7.1.2 Missi Group

The Missi Group is the youngest supracrustal succession in the Flin Flon Greenstone Belt. At Snow Lake, Froese and Moore (1980) describe the rocks of the Missi succession as a monotonous sequence of metamorphosed lithic arenites. In the File Lake and Snow Lake areas, the Missi Group differs from its type localities in the Flin Flon region by the relative absence of conglomerate layers and apparently conformable deposition over turbiditic greywacke units rather than subaerially weathered volcanic rocks.

Volcanic rocks that include welded tuffs have been identified in the Missi Group east of Wekusko Lake in the Snow Lake area (Gordon and Gail, 1982). The volcanic rocks are predominantly fragmental felsic units; however, massive intermediate flows and related breccias occur locally. The lack of pillow structures and presence of welded tuffs have led Shanks and Bailes (1977) and Gordon and Gail (1982) to infer a subaerial environment of deposition.

7.1.3 Intrusive Rocks

Intrusive rocks of various ages and compositions occur throughout the Flin Flon-Snow Lake Belt. The earliest intrusions are recognized as synvolcanic mafic to felsic sills and dykes, and granitoid, commonly porphyritic plutons that are restricted to the Amisk Group and related to Amisk volcanism. Numerous strongly differentiated gabbroic sills, some possibly syn-volcanic, occur throughout the belt and have been documented in the Flin Flon, File Lake and Snow Lake areas.

Stauffer (1974) identified a heterogeneous granitoid complex unconformably overlain by Missi Group strata near Flin Flon and has used this to indicate that a plutonic event occurred after the cessation of Amisk volcanism and prior to deposition of the Missi Group. In addition, the widespread occurrence of granitic clasts in the Missi suggests that plutonism, uplift and erosion took place prior to sedimentation of the Missi Group. Later granitic-granodioritic plutons of up to batholithic dimensions intrude and in places, segment the belt. These intrusions are considered to be largely syntectonic as their margins are broadly concordant with stratigraphy of the adjacent greenstone belt and contain the same foliation as the supracrustal rocks (Price, 1977). Late granitic and pegmatitic intrusions cross-cut all earlier rocks in the Flin Flon-Snow Lake Greenstone Belt.

7.1.4 Metamorphism

Regional metamorphism began after deposition of the Missi Group. Grades of metamorphism in the Snow Lake area are generally higher than those documented in the rest of the belt. Froese and Moore (1980) recognized four metamorphic zones in this area that range from a lower amphibolite zone north to an upper amphibolite zone.

7.1.5 Structure

In the File Lake-Snow Lake area three periods of folding have been recognized, all of which are considered to be post-Missi (Froese and Moore, 1980). Early isoclinal folds have been refolded about northeasterly trending open folds (e.g. The Threehouse Syncline at Snow Lake). Deformation associated with minor folds that postdate the second phase of folding in the Snow Lake area was observed in the immediate vicinity of gneissic domes on the north margin of the belt and is considered to be related to increased tightening of second phase folds (Froese and Moore, 1980). The McLeod Lake thrust fault (Russell, 1957), a major structural break in the Snow lake area, is interpreted to represent an early nappe structure associated with the early isoclinal folding (Froese and Moore, 1980).

7.2 Property geology

The mineralized zones and deposits on the NBM property are located within the Aphebian Flin Flon/Snow Lake Greenstone Belt, which is an assemblage of polydeformed volcanosedimentary supracrustal sequences intruded by pre- and syn-tectonic ultramafic and mafic intrusions and syn- to post-tectonic granitoid.

The geological sequences near the NBM consist of a succession of intercalated mafic and felsic volcanic and pyroclastic rocks (Amisk Group) intruded by gabbroic bodies and unconformably overlain by arkosic sediments (Missi Group). The metamorphic assemblages are characteristic of the low to mid amphibolite facies.

One of the main structural features of the Snow Lake area is the north-northwest trending McLeod Road Thrust fault. The McLeod Road Thrust fault is the structural break that occurs between the metavolcanic (Amisk Group) and metasedimentary (Missi Group) assemblages and defines the western extent of the deposits at the NBM.

7.2.1 Nor-Acme/New Britannia Mine

The auriferous zones within the mine lie along a curvilinear shear (fault) zone named the Howe Sound fault. In some literature this fault is also known as the Nor-Acme fault. One or more “slip planes” accompanied by a variety of altered mylonitic zones mark the location of the fault. The mylonitic zones (quartz-carbonate-mica) are predominantly less than 1 ft thick but can thicken locally to between 10 ft and 20 ft (3 m to 6 m). The gold bearing quartz carbonate rocks are almost always situated next to or astride the fault.

The main mineralization consists of quartz-carbonate alteration zones emplaced in a simple intercalated sequence of altered felsic and mafic volcanics and pyroclastics of the Amisk Group. The host rocks are altered and include varying proportions of quartz and carbonate (mainly calcite). The contacts of the mineralized zones are often gradational; however, there have been sharp shear and fault contacts noted within some portions of the deposit and a biotite alteration halo occurs within a few feet of the mineralized zone.

The main controls with respect to the location of the mineralization within the deposit are contacts between rocks of differing competency, flexures within the Howe Sound fault, and changes in the dip of the fault plane. The dip of the fault plane averages 45° but can vary from 25° to 80° locally.

7.2.2 No. 3 Zone

The gold bearing horizons in the No. 3 Zone consist of silicified, quartz rich, shear zones containing arsenopyrite plus gold and occasionally free gold. The host rocks include brittle basalts and tuffs with varying amounts of quartz and carbonate.

The geology of the mineralized zone observed during mining on the 65 Level noted that the primary mineralization on this level was located in a quartz vein which varied in true width from 0 ft to 5 ft (0 m to 1.5 m). Banded tuffs with biotite, quartz and arsenopyrite occur within 1 ft (0.3 m) and therefore the mineralization is not confined to the quartz vein and can occur in some cases up to several feet (metres) from it. The main mineralized zone strikes 095° to 105° in azimuth, dip varies from 40° to 45° to the north and the plunge is approximately 40° at an azimuth of 060°. The surrounding country rock consists of mafic tuffs with alteration zones of carbonate, biotite, chlorite and clay minerals.

In the western end of the 65 Level a fault zone with up to 2 inches of gouge forms the hangingwall of the quartz vein. A mylonitic zone up to 2 ft (0.6 m) was noted at the far end of the level on the footwall side. Garnet porphyroblasts are noted in hangingwall rocks adjacent to the mineralized zone.

To the east on the 65 Level, similar geology prevails with a fault occurring on the footwall of the quartz vein. Numerous shears and joints are noted in both drifts with occasional vugs. Shears run subparallel to the quartz vein. Foliation can vary wildly, implying intense deformational events.

Joint sets also occur with some cathedraling of the back in the western end of the 65 Level, with blocky ground occasionally encountered in the development.

7.2.3 Birch Zone

The Birch Zone has been interpreted as being hosted in a silicified shear zone which cuts an assortment of rock types, including graphitic argillite, siltstone, chert, mafic volcanic rocks (flow, fragmental, and tuff units) and gabbroic to dioritic sediments (Taylor. 1989). The zone dips at 45° and plunges at approximately 15°.

Figure 7-2 is a geological map of the NBM property. The mineral leases, Hudson Bay option, satellite deposits, various mineralized occurrences, and geology are all indicated on this plan.

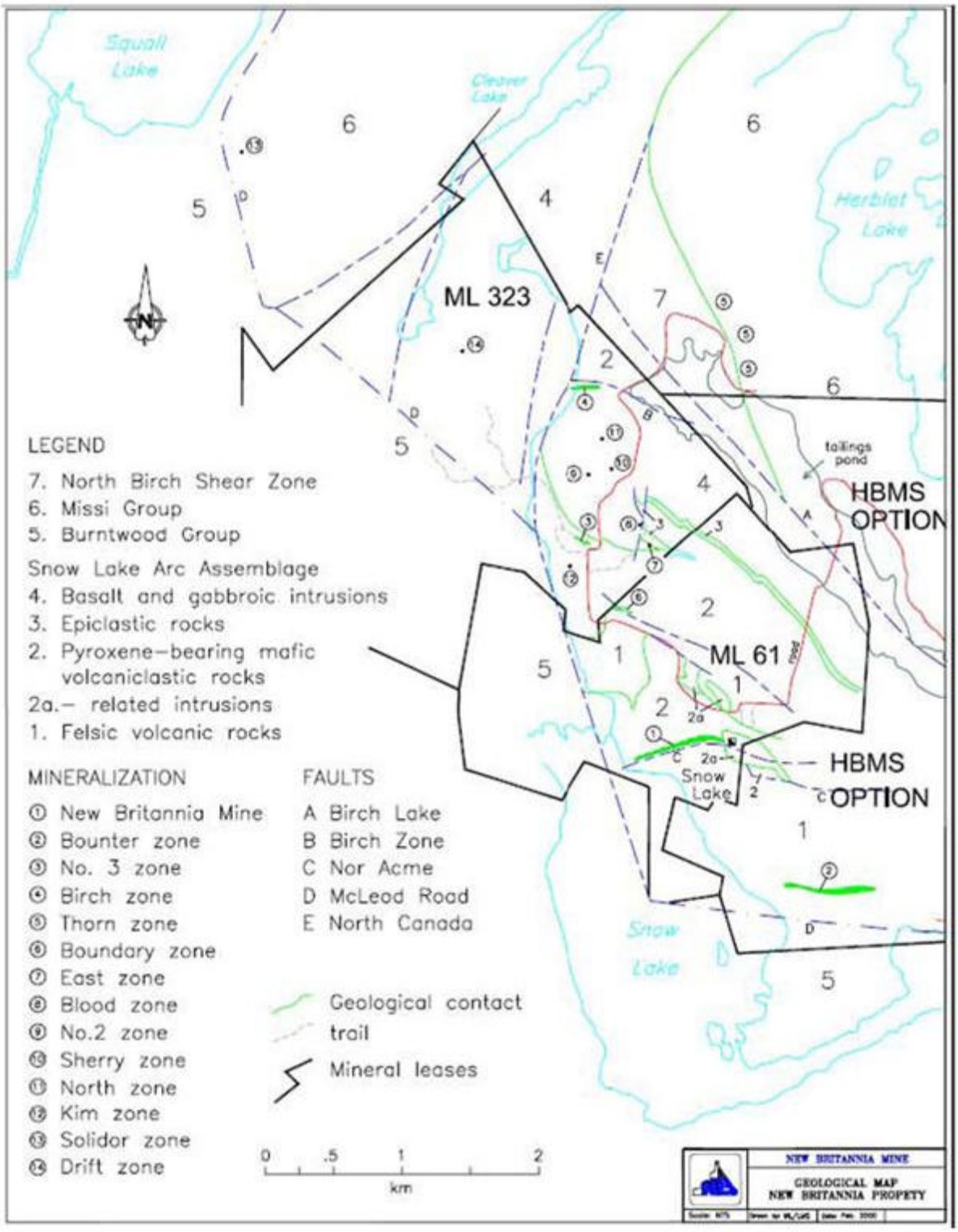

Figure 7-2: Local Geological Setting of the New Britannia Mine

Figure provided by Garson Gold Corp

8. DEPOSIT TYPES (JL)

The description of the deposit types is taken directly from the technical report by Micon International Limited, dated May 30, 2009.

The main mineral deposits (NBM, No. 3 Zone and Birch Zone) located on the NBM property are considered to belong to the quartz-carbonate vein gold subtype of lode gold deposits. This subtype of gold deposits consists of simple to complex quartz-carbonate vein systems associated with brittle-ductile rock behaviour, corresponding to intermediate depths within the crust, and compressive tectonic settings. Deposits of this type have been commonly referred to as mesothermal gold quartz vein deposits, but they encompass both mesothermal and hypothermal classes as initially defined by Lindgren (1933).

At the regional scale, quartz-carbonate vein gold deposits occur in two contrasting geological environments: deformed clastic sedimentary terranes and deformed volcano-plutonic terranes containing diverse volcanic assemblages of island-arc and oceanic affinities. Despite lithological and structural differences, these two types of environment share the following characteristics: greenschist to locally lower amphibolite metamorphic facies, brittle-ductile nature of deformation, and geological structures recording compressional to transpressional tectonic settings.

Quartz-carbonate vein gold deposits in these environments tend to occur in clusters, or districts, and they are by far more abundant in volcano-plutonic terranes than in clastic sedimentary terranes. Both types of environments are present in a number of districts, in which they are separated by major fault zones. However, in such cases auriferous quartzcarbonate veins preferentially occur in the volcano-plutonic domains.

9. MINERALIZATION (JL)

Alexis is currently in the process of updating description of the mineralized zones on the property as exploration and delineation drilling continues. This includes further establishing and refining the geological controls on mineralization. In conjunction with regional and property scale activities, this approach has the potential to enhance the success of future exploration activities. The following description of the mineralization is taken directly from the technical report by Micon International Limited, May 30, 2009.

9.1 The New Britannia Mine Zones

Mineralization is found at surface over a strike length of 2,000 ft (609.6 m) and occurs in two main zones known as the Toots and Dick, respectively. At depth, between the 1030 Level and the 1280 Level the Dick Zone appears to split into two separate zones (Dick and Ruttan) with four zones occurring on the 1780 Level. The zones which occur on the 1780 Level are, from west to east, the Toots, Dick, Ruttan and Hogg Zones.

Below the 1780 Level, the Toots Zone is terminated by the McLeod Road Thrust fault and only the Dick, Ruttan and Hogg Zones continue to depth.

On the 1780 Level, the Ruttan and Hogg Zones are separated by an area of weak mineralization approximately 100 ft (30.5 m) long; however, by the time the zones reach the 2010 Level, the mineralization is continuous between the two zones. Below the 2010 Level the Ruttan and Hogg Zones continue as one zone down to the 2300 Level, where due to the decreasing strike length of the mineralization the two zones were combined by the geological staff of the NBM into one zone (Ruttan) for interpretation and mining purposes.

The zones of mineralization have a predominantly east-west strike direction, dip 45° north and plunge approximately 030° to the north-northeast following the trend of the Howe Sound fault with the mineralized zones pinching and swelling (boudinage) in both plan and sectional views.

On the western extent of both the Toots (1780 Level) and Dick (below the 3000 Level) Zones, where the mineralization is in close proximity to the McLeod Road Thrust fault, there is a slight orientation to the northwest with the orientation becoming more prominent below the 3200 Level. The orientation to the northwest in both cases appears to follow not only the trend of the Howe Sound fault but also the trend of the thrust fault closely and may be potentially related to remobilization of the mineralization either during the original emplacement of the thrust fault or during a period of re-activation of the thrust fault.

Development on the western portion of the 3630 and 3680 Levels within the Dick Zone appeared to indicate that the northwest orientation of the mineralization was increasing in both strike length and true thickness at depth. The in-situ grade of the mineralization had also increased significantly and appeared to be consistently higher throughout the northwest trending portion of the Dick Zone than in other areas (zones) of the mine. A drill program conducted from the 3630 exploration drift in the fall of 2004 covered the northwest trending portion of the Dick Zone. The drilling results indicated that the mineralization encountered on the 3630 and 3680 Levels constituted blowouts (swells) in the zone, as the zone narrowed considerably between the 3680 and 3900 Levels. The diamond drilling identified another potential blowout in the mineralization below the 3900 Level; however, determining the full extent of the mineralization below the 3900 Level will require further drilling.

On the western extent of the Dick Zone, the northwest trending mineralization below the 3680 Level flattens to between 20° and 30°, which generally coincides with the flattening of the McLeod Road Thrust fault in this area. The drilling results from the 3630 exploration level indicate that the mineralization continues to trend to the northwest onto the Bud 12593 (CB 12593) mineral claim, although it is only a couple of feet wide (true width) for the most part and uneconomic at the present time.

The mineralization in the Ruttan Zone continues both below the 3600 Level and to the east onto the Hudson Bay option mineral claims, but the 2003 underground diamond drilling results indicate that the mineralization becomes more erratic at depth and it is uneconomic at the present time.

The common sulphides in the New Britannia deposit are arsenopyrite, pyrrhotite and pyrite. Arsenopyrite constitutes about two per cent of the mineralization, pyrrhotite less than one per cent and pyrite less than one-quarter of one per cent. Trace sulphides include chalcopyrite and sphalerite with the total sulphide content averaging less than five percent.

Gold is predominately associated with arsenopyrite, especially where the arsenopyrite occurs as a mesh of fine-grained needles. However, when the arsenopyrite occurs as anhedral grains or massive seams, the ratio of gold to arsenopyrite is less and the corresponding grade of the ore is lower. The affinity of gold for arsenopyrite appears to be structural with the gold occurring as minute lenticular masses and veinlets in openings on arsenopyrite crystal boundaries, and within fractures in the arsenopyrite. The gold also occurs as minute particles in quartz and calcite with this generally the case when the gold is visible macroscopically, a very rare occurrence.

9.2 No. 3 zone

The gold bearing zone is 2 m to 11 m thick with a strike length of 200 m. Numerous faults occur within the mineralized zone as well as a quartz vein stockwork that is developed at a low angle to the fault direction. The gold zone is bounded in the hangingwall by a 2 centimetre (cm) to 60 cm wide fault with mylonitic texture.

The fault is characterized by a mineralogy dominated by carbonate, quartz and epidote. The hangingwall fault, subsidiary faults and sub-parallel gold zone are curvilinear with the strike of the mineralized zone varying from 285° at its northwest extremity to 080° at its eastern end.

Examinations of the surface exposures of the occurrence indicate that the strike of the foliation is asymmetrical about the veins. The Z-shape of the foliation indicates sinistral movement within a brittle-ductile shear zone. Harrison (1949) and Galley et al. (1986) propose fluid emplacement in a schistose zone that developed at or near the contact between the mafic heterolithic fragmental rocks, which host the deposit, and the massive pillowed basalt for the formation of the No. 3 Zone deposit. Alteration of the mafic heterolithic fragmental rocks is reflected by the presence of biotite, chlorite and garnet.

The mineralization in the gold zone is comprised of up to 10% medium to fine grained arsenopyrite, from 0% to 5% pyrite, traces of chalcopyrite and up to 5% tourmaline. The highest gold values are correlated to fine grained felted masses of arsenopyrite that commonly occur along the margins of quartz veins (Galley et al., 1986).

9.3 Birch zone

The mineralization and alteration are very similar to No. 3 Zone. A large shear zone (fault) cuts through the main portion of the mineral occurrence and several parallel and sub parallel shears also contribute to the complexity of the deposit. The gold is generally associated with arsenopyrite but some free gold has also been noted on rare occasions. Drilling by both Snow Lake Mines in the late 1980s and by TVX and High River in the 1990s indicated that the rock types and assays correlated poorly between drill holes at depth.

10. EXPLORATION (JL)

A description of the historical exploration programs completed on the property during and prior to the last phase of mining (1995 – 2005) on the is contained in Item 6 of this report. An outline of the recent exploration completed on the property, since 2007, is contained below which comes from the technical report by Micon International Limited, dated May 30, 2009 and the technical report by Golder Associates dated April, 2010. In January 2010, Alexis initiated exploration on the property which includes evaluation and execution of the exploration program previously proposed and presented in the technical report by Golder Associates dated April, 2010.

Since Alexis acquired the Snow Lake property in January 2010 one phase of diamond drilling has been completed. Between January 14, 2010 and June 30, 2010 a total of 17,962 metres of diamond drilling was completed. This drilling added data in areas with the potential to provide mineral resources to the mining plan. Exploration targets have also been tested on the Snow Lake property. A second phase of diamond drilling is currently underway to find and define additional resources to extend the mine life. Compilation work of historical exploration information continues and a program to acquire additional land in the Snow Lake area is underway.

10.1 Airborne magnetic survey

Garson contracted Goldak Airborne Surveys Ltd. (Goldak) to complete a high density airborne magnetic survey over the entire property. The survey was completed between May 23 and May 26, 2008. The following description of survey equipment and specifications is taken from the survey report (Goldak, 2007):

> "Aircraft equipment operated included four cesium vapour, digitally compensated magnetometers, a GPS real-time and post-corrected differential positioning system, a flight path recovery camera, VHS titling and recording system, as well as radar and barometric altimeters. All data was recorded digitally in GEDAS binary file format."
>
> "Reference ground equipment included a GEM Systems GSM-19W Overhauser magnetometer and a Novatel 12 channel GPS base station which was set up at the base of operations for differential post flight corrections."
>
> "Three flights (one calibration and two production) were required to complete the survey block. A total of 1,413 line km of high resolution magnetic gradiometer data were collected, processed and plotted. Flin Flon, MB was used as the base of operations for the survey."
>
> "The traverse lines were flown on a spacing of 100 m with a control line separation of 1,000 m. Nominal terrain clearance was specified at 80 m above ground."

Complete and detailed survey specifications are contained in the survey report (Goldak, 2007). Goldak provided to Garson a digital data set and three processed images for interpretive purposes and integration into the property wide GIS compilation.

Detailed high quality magnetic surveys are common in exploration programs both as a direct method of detection (e.g. detection of magnetic massive sulphide bodies) and, as in the case of the NBM property, as an indirect method where the magnetic survey is used to support geological and structural interpretation. The results of the Goldak surveys have been integrated into the property wide GIS compilation.

10.2 Geochemistry surveys

10.2.1 Mobile Metals Ion (MMI-M) Survey

A MMI-M exploration soil geochemical survey was completed on three target areas during the 2007 exploration program. A total 943 inorganic soil samples were collected and analyzed for a suite of 46 elements.

The MMI-M survey was designed in conjunction with consulting geologist/geochemist, Dr. Mark Fedikow. Dr. Fedikow compiled the geochemical data and completed a survey report including the documentation of a number of multi-element anomalies (Mount Morgan Resources Ltd., 2007).

10.2.2 Soil Gas Hydrocarbon (SGH) Geochemistry Survey

A SGH soil survey was completed on the NBM property during the summer of 2008 on a single target area. The initial SGH survey was designed to repeat part of the 2007 MMI survey. A total of 360 soil samples were collected and analyzed for 162 hydrocarbon compounds in the C5-C17 carbon series range. The survey resulted in the recognition of two halo anomalies (Figure 10-1), the larger of which is coincident with a previous MMI survey anomaly.

10.3 Data compilation

The property has been the subject of various exploration programs since the initial discovery of gold in the area. A major component of the current exploration effort by Garson is the compilation of the historical exploration data and integration with current exploration data and results from the various federal and provincial geological survey initiatives. The compilation utilizes both GIS and 3-D modelling software (Gemcom), and forms the basis for ongoing exploration and interpretation.

10.4 Exploration programs since the 2008 resource estimate

The primary exploration work since the 2008 resource estimate for the No. 3 Zone was completed has involved further diamond drilling. Between May and October, 2008, a total of 6,634 m in 39 drill holes were completed on a number of targets. The drilling included 2,100 m on the Boundary Zone, 3,131 m on the Kim Zone and 1,403 m on the Hogg Zone which is part of the Main Mine area. The interpretation of the drilling results is ongoing.

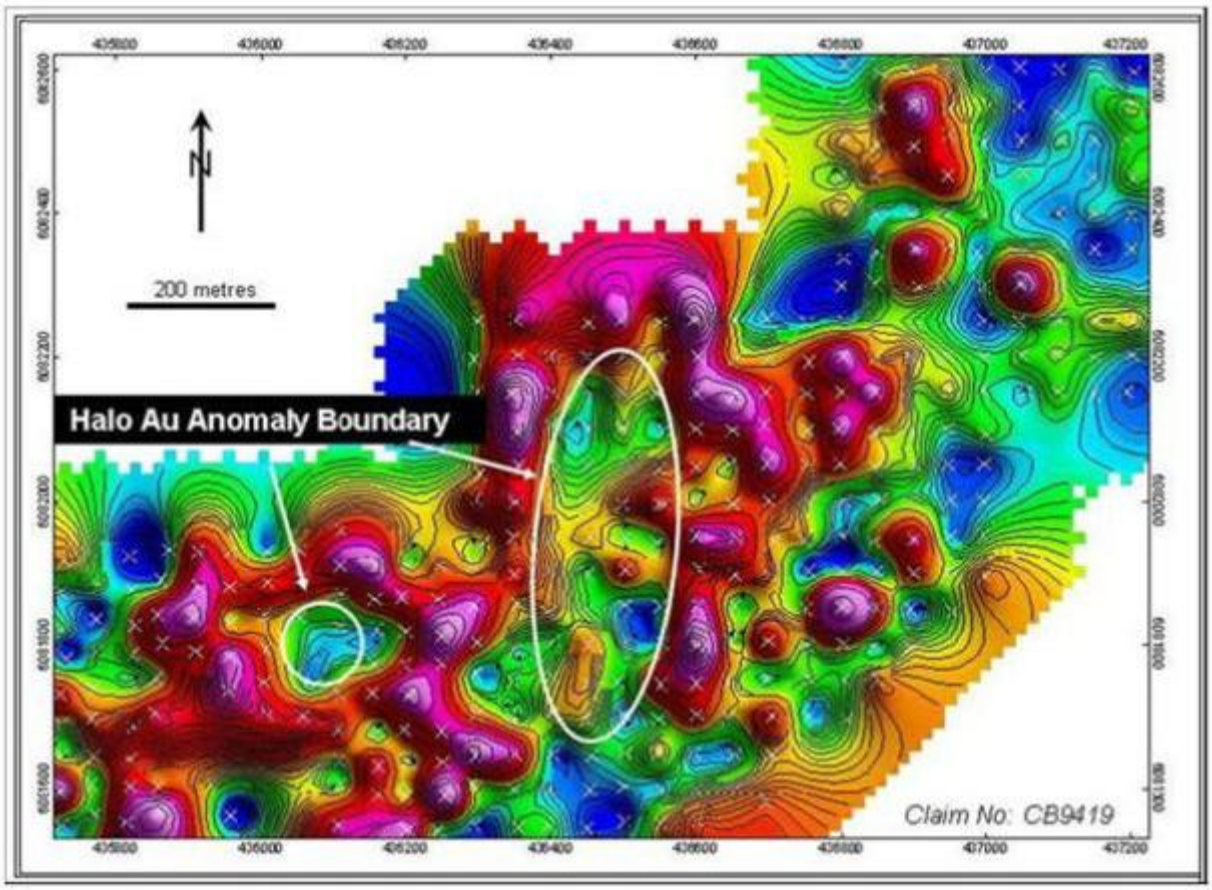

Figure 10-1: Colour Contour Map Illustrating SGH Geochemical Anomaly

10.5 Exploration targets and future exploration

The NBM property contains a number of targets that are at various stages of exploration. These range from resource delineation and expansion targets to conceptual targets based on geological compilation and interpretation. The classification scheme of exploration opportunities on the New Britannia property is summarized as follows:

1) Development and Resource Expansion Stage Opportunities. Includes opportunities that are the subject of an advanced study, i.e. No. 3 Zone and, as in the case of No. 3 Zone, are open for resource expansion. The NBM in part falls into this category as the resource evaluation requires the application of capital and operating cost models.
2) Resource Evaluation Stage Opportunities. This includes the NBM, Birch and Margaret Zones all of which have historical resource estimates and require further evaluation including geological modelling and/or resource modelling.
3) Advanced Stage Exploration Targets. This includes those zones that have sufficient drilling and have been advanced to the stage where the geological interpretation and resource estimation is in progress. This includes the Boundary and Bounter Zones.
4) Medium Stage Exploration Targets. This class of targets is based on the occurrence of surface showings and drill intercepts but with insufficient data or knowledge to base a resource potential. This includes the Caper Zone and the No. 3 Hangingwall Zones.
5) Early Stage Exploration Targets. This includes targets that are based on early stage exploration data and the interpretation of compiled geological, geochemical and geophysical exploration data.
6) Property Development Projects. This includes proposed work including compilation and surveys planned to generate early stage exploration targets.

The exploration targets for the NBM property are summarized in Tables 10.1a and 10.1b. The target areas are shown in Figure 10-2.

10.6 Exploration program

As discussed previously, the NBM property contains a number of exploration targets at various stages of advancement (Table 10-2). Garson has proposed an ongoing exploration program with two main objectives:

1) The first objective is to evaluate and increase the gold resources within the strategic No. 3 Zone to the Boundary Zone and the Main mine corridor and to incorporate these gold resources into future assessments. The evaluation of historical resources at the Birch Zone and Margaret Zone is included in this objective.
2) The second objective is to continue to evaluate the gold potential of the entire property through medium to early stage exploration targets and new drill targets.

Infill drilling to upgrade the current resource classifications is not included in the recommended exploration program. The infill drilling will be contingent upon the identification and inclusion of further resources into the preliminary assessment. A seismic survey that has been shownin Table 10-2, has not included as it is under evaluation. The proposed exploration program is summarized in Table 10-2. Phase 1

activities are weighted towards desk top and field exploration, whereas Phase 2 includes a larger component of drilling. Although significant drilling is anticipated in Phase 2, this will be contingent upon the results of Phase 1.

Table 10-1 Summary and Status of Garson's Exploration Targets

Development and Resource Expansion Stage			
Deposit/Zone/Target/Area	**Status**	**Opportunity**	**Recommended Program**
No.3 Zone Indicated Resource	updated resource completed and audited/reported include approx. 85,000 Oz @ 6.72 g/t Au to 1300 ft vertical; drill spacing somewhat irregular with locally wider spacing	infill drilling and upgrade resource consistent with feasibility level evaluation	Drilling: 7 holes (8800 ft, 2680m) at selected infill positions in resource
No.3 Zone Inferred Resource	updated resource completed and audited; approx. 54,000 Oz @ 9.19 g/t Au from 1300 ft to 1600 ft vertical; PEA completed	upgrade resource from inferred to indicated status	Drilling: 7 holes (13,000 ft, 4000m) within limits of inferred resource
No.3 Zone Resource Expansion	No.3 Zone resource open to depth	expand the No.3 Zone resource down plunge to inferred status;	Drilling: 6 hole (15,000 ft, 4570m) to expand No.3 Zone 600 ft vertical
Resource Evaluation Stage			
Deposit/Zone/Target/Area	**Status**	**Opportunity**	**Recommended Program**
Main Mine	historical resource: 364,000 Oz @ 5.11 g/t Au (Measured and Indicated) and 45,000 Oz @ 4.59 g/t (Inferred); 4 holes drilled by GG in Hogg Zone	evaluation leading to potential conversion of Main Mine resources, interpretation potentially leading to resource extension, quantification of geological controls of mineralization applicable to property wide targeting	completion of database compilation and validation, completion of geological interpretation, re-estimation of resources and extensional targeting; budget for current staffing to completion Drilling: Prioritize the Upper Hogg/Ruttan Zone. Drilling contingent upon economic model to include infill drilling and potential up dip resource expansion drilling.
Birch Zone	historical resource: 81,000 Oz @ 4.42 g/t Au (inferred), 2 holes drilled by GG confirmed historical intercepts, drill hole data base compiled and validated, interpretation initiated but currently inactive	evaluation leading to resource conversion, potentially open pit as well underground opportunity to be evaluated, geological interpretation leading to	Desktop: geological interpretation and geological model leading to 1) updated resource estimate and 2) targeting and exploration program design for expansion of resources and discovery in the "Birch" environment.

		targeting immediate environment for extension to new mineralization and new zones	Drilling: If, where, and footage contingent on outcome of desktop study. Use 5000 ft for budget contingency
Margaret Zone	historical resource: 75,000 Oz @ 5.90 g/t Au (inferred); mineralization appears to be stratbound lode type gold; drill data has been compiled otherwise no systematic work completed by GG	appears to be gently dipping and possibly favourable for pitable resource; appears to be "stratabound" lode mineralization and therefore may have "apparent" exploration potential	Desktop: geological interpretation and geological model leading to 1) updated resource estimate and 2) targeting and exploration program design for expansion of resources. Budget 1 geo 3 months
			Drilling: If, where, and footage contingent on outcome of desktop study. Use 5000 ft for budget contingency
Advanced Exploration Stage			
Deposit/Zone/Target/Area	**Status**	**Opportunity**	**Recommended Program**
Boundary Zone	significant historical drilling completed, significant drilling completed by GG, no historical resource estimates completed, preliminary geological interpretation and model of mineralized lodes completed by GG, revised geological model and resource estimate by GG in progress	potential for rhyolite hosted mineralization to support open pitable resource, significant intersections comparable to No.3 Zone represent potential for UG resource	Desktop: Geological model and resource model to be completed and pit potential to be evaluated by pit optimization routine, 1 month in-house geologist budgeted to complete geo model and resource and budget for consultant for pit evaluation
			Drilling: current geological interpretation includes recommended drill program to follow up down plunge of high grade intersections includes 16 holes for total of 21,400 ft (6522m)
Bounter Zone	significant historical drilling completed, compilation and preliminary interpretation completed by GG - very wide, very steeply dipping, zones of relatively low grade mineralization with sporadic internal high grade intercepts	width and orientation **may** lend itself to bulk underground mining method, geological environment may be similar to Main Mine and continued interpretation may provide	Desktop: Validation of geological interpretation and if warranted completion of resource model and evaluation of reserve conversion potential. Integrate with property data for targeting. One month in-house geologist budgeted to complete geo

		targeting vector	model and resource and budget for consultant for pit evaluation
Medium Exploration Stage			
Deposit/Zone/Target/Area	**Status**	**Opportunity**	**Recommended Program**
Caper Zone	historical drilling intercepted locally high grade mineralization, GG program of compilation, field work, and evaluation completed and includes recommendations for continued exploration	medium stage exploration opportunity	Drilling: Review of Caper Zone report and recommendations. Pending review, report recommends several drill programs based on distinct structural targets. For preliminary budget use 5000 ft (1500m) in a phase 1 program is proposed
No.3 Zone Hangingwall	GG drilling intersected numerous potentially ore grade intercepts in the hangingwall of the No.3 Zone, a number of historical surface showings similarly occur between No.3 Zone and Birch Zone; this area and area encompassing the Boundary to Birch corridor was prioritized for field work, interpretation, targeting, and drilling last year but was not successfully completed	medium stage exploration opportunity; potential to develop resources between Birch and No.3 Zone through interpretation and development of drill targets	Desktop: Continued and completion of detailed compilation of historical geological/geophysical data including integration of No.3 Zone drilling and hangingwall intercepts, budget 1 geologist for 2 months Field: Pending outcome of desktop compilation, field work to include outcrop/showing stripping and detailed mapping to constrain range and orientation of showings/intercepts. 1 geologist and 2 technicians budgeted here for 2 months Drilling: Pending outcome of desktop and field work drilling to evaluate tonnage potential of selected showings or area. Use 20 - 300 ft holes for contingency
Early Exploration Stage			

Deposit/Zone/Target/Area	Status	Opportunity	Recommended Program
Birch Zone Structure	target area based on magnetic/structural interp. in the area of Birch Zone and to the west, some drilling in the area is gold mineralized, preliminary field investigation in fall 08 indicates area poorly exposed and wet	early stage exploration opportunity	Field: Although indications are limited outcrop the area may benefit from prospecting following up structures/magnetic lineaments, possibly geochem and ground geophysics
			Drilling: Short hole program to follow up structural interpretation and historical mineralized intercepts. Pending field program and formal program design and layout budget 10 - 100 metre holes
Geochem Follow up	MMI soil geochemistry survey completed in the summer of 2007 and SGH survey completed during the summer of 2008 both resulted in anomalies for follow-up, MMI field follow-up was inconclusive, SGH survey has not been followed-up. In one instance MMI and SGH anomalies are coincident in Caper area	early stage exploration opportunity	Field: Geology and prospecting follow up of Au and As anomalies
			Drilling: Proposed 5 short holes (100m each) to test blind geochem anomalies, further drilling contingent on field work
Eastern Target Area	The Eastern Target area consists includes MMI geochem As and Au anomalies, potentially favourable structure as derived from magnetics,	early stage exploration opportunity	Field: Prospecting and geological mapping

Property Exploration and Evaluation			
Deposit/Zone/Target/Area	**Status**	**Opportunity**	**Recommended Program**
Prospecting	not completed by GG and not known to have been completed by TVX/Kinross, certainly completed by earlier operators	property evaluation and early stage exploration opportunity	Field: Prospecting team consisting of prospector and assistant (student?) for 4 month season (June-September) to complete directed prospecting which includes evaluating and potentially expanding historical showings and "directed" prospecting to follow up geological/structural interpretations
IP Compilation	a number of IP surveys completed by previous companies but have not been "pulled together", geo-registered, and interpreted	property evaluation and early stage exploration opportunity	Desktop: Compilation and interpretation of archived IP surveys by geophysicist/IP Specialist and integration into property GIS
Seismic Survey	conceptual; seismic surveys completed successfully in Flin Flon area ref: PDAC 2009 Technical Session	property evaluation and early stage exploration opportunity	complete 3D seismic survey pending discussion/confirmation of application of method to property specific problems in the context of physical rock properties; potentially image thrust fault geometry including "flexures"

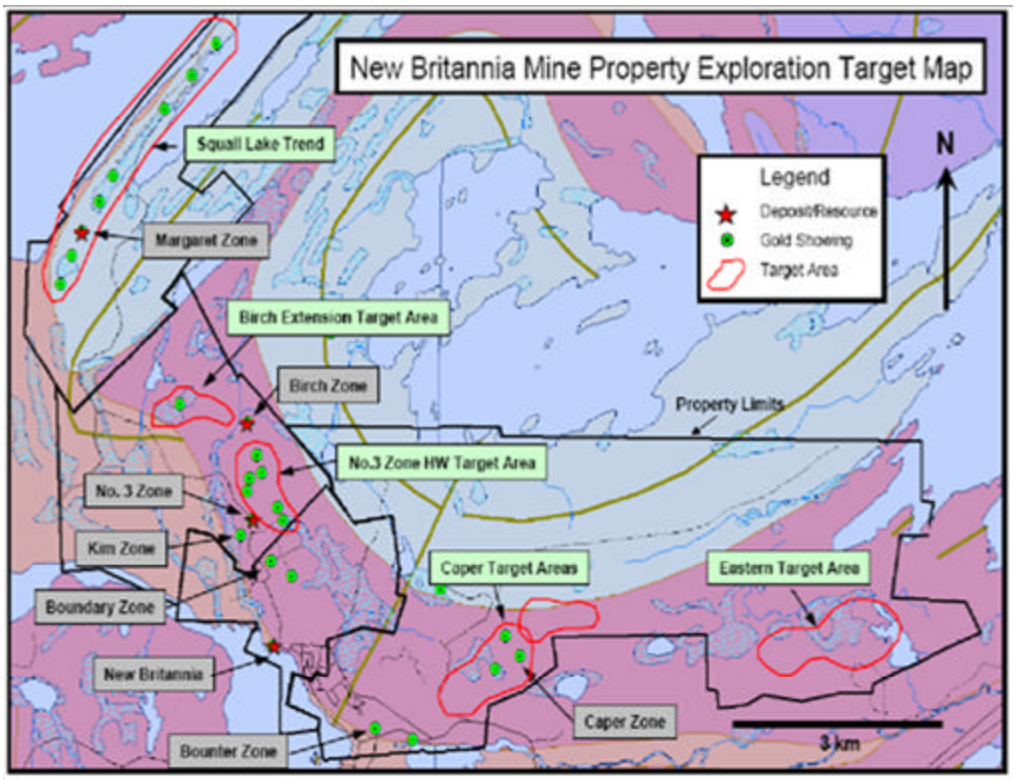

Figure 10-2: Map Showing Garson's Major Exploration Targets within the NBM Property

Table 10-2: Summary of Garson's Proposed Exploration Program

Exploration Objective 1: Resource Expansion and Resource Evaluation		
Deposit/Zone/Target/Area	**Recommended Program - Phase 1**	**Recommended Program - Phase 2**
No.3 Zone	Drilling: 6 hole (15,000 ft, 4570m) to expand No.3 Zone 600 ft vertical	Further work to upgrade resources contingent upon overall project
Main Mine	completion of database compilation and validation, completion of geological interpretation, re-estimation of resources and extensional targeting; budget for current staffing to completion	Drilling: Prioritize the Upper Hogg/Ruttan Zone. Drilling contingent upon economic model to include infill drilling and potential up dip resource expansion drilling. Use 10,000 ft for budget contingency
Birch Zone	Desktop: geological interpretation and geological model leading to 1) updated resource estimate and 2) targeting and exploration program design for expansion of resources and discovery in the "Birch" environment.	Drilling: If, where, and footage contingent on outcome of desktop study. Use 5,000 ft for budget contingency
Margaret Zone	Desktop: geological interpretation and geological model leading to 1) updated resource estimate and 2) targeting and exploration program design for expansion of resources. Budget 1 geo 3 months	Drilling: If, where, and footage contingent on outcome of desktop study. Use 5,000 ft for budget contingency
Boundary Zone	Desktop: Geological model and resource model to be completed and pit potential to be evaluated by pit optimization routine, 1 month in-house geologist budgeted to complete geo model and resource and budget for consultant for pit evaluation	Drilling: current geological interpretation includes recommended drill program to follow up down plunge of high grade intersections includes 16 holes for total of 20,000 ft
Bounter Zone	Desktop: Validation of geological interpretation and if warranted completion of resource model and evaluation of reserve conversion potential. Integrate with property data for targeting. One month in-house geologist budgeted to complete geo model and resource and budget for consultant for pit evaluation	Drilling: Contingent upon the results of phase 1. Use 2,000 ft for preliminary test and budget

Exploration Objective 2: Medium Stage to Early Stage Target Evaluation and Exploration		
Deposit/Zone/Target/Area	**Recommended Program - Phase 1**	**Recommended Program - Phase 2**
Caper Zone	Drilling: Review of Caper Zone report and recommendations. Pending review, report recommends several drill programs based on distinct structural targets. For preliminary budget use 5000 ft (1500m) in a phase 1 program is proposed	Drilling: Contingent upon the results of phase 1. Use 2,000 ft for preliminary test and budget
No.3 Zone Hangingwall	Desktop: Continued and completion of detailed compilation of historical geological/geophysical data including integration of No.3 Zone drilling and hangingwall intercepts, budget 1 geologist for 2 months	Field: Pending outcome of desktop compilation, field work to include outcrop/showing stripping and detailed mapping to constrain range and orientation of showings/intercepts. 1 geologist and 2 technicians budgeted here for 2 months Drilling: Pending outcome of desktop and field work drilling to evaluate tonnage potential of selected showings or area. Use 20 - 300 ft holes for contingency
Birch Zone Structure	Field: Although indications are limited outcrop the area may benefit from prospecting following up structures/magnetic lineaments, possibly geochem and ground geophysics	Drilling: Short hole program to follow up structural interpretation and historical mineralized intercepts. Pending field program and formal program design and layout budget 10 - 300 ft holes
Geochem Follow up	Field: Geology and prospecting follow up of Au and As anomalies	Drilling: Proposed 5 short holes (300 ft each) to test blind geochem anomalies, further drilling contingent on field work
Eastern Target Area	Field: Prospecting and geological mapping	Drilling: Contingent upon the results of phase 1. Use 2000 ft for preliminary test and budget
Prospecting	Field: Prospecting team consisting of prospector and assistant (student?) for 4 month season (June-September) to complete directed prospecting which includes evaluating and potentially expanding historical showings and "directed" prospecting to follow up geological/structural interpretations	
IP Compilation	Desktop: Compilation and interpretation of archived IP surveys by geophysicist/IP Specialist and integration into property GIS	

10.7 Exploration Completed by Alexis and Exploration Plans

Since Alexis acquired the Snow Lake property through its acquisition of Garson in January 2010, one phase of diamond drilling has been completed. Between January 14, 2010 and June 30, 2010 a total of 17,962 m of diamond drilling was completed. This drilling added data in areas with potential to provide mineral resources to the mining plan. Exploration targets have also been tested on the Snow Lake property. This drilling including summary of significant intercepts is presented in Item 11 of this report. A second phase of diamond drilling is currently underway to find and define additional resources to extend the mine life. Compilation work of historical exploration information continues and a program to acquire additional land in the Snow Lake area is underway. The table of targets presented above (Table 10.1) and the exploration program (Table 10.2), remain the framework for continued exploration and evaluation by Alexis.

11. DRILLING (JL)

Details of the drilling on the property during and preceding the last phase of mining are contained in Item 6 of this report. The drilling completed after 2007 is described below and includes sections from the technical reports by Micon International Limited dated May 5, 2009, and Golder Associates dated April, 2010.

The following photograph shows one of the Boyles drills on No. 3 Zone during the Micon site visit.

Figure 11-1: Boyles 37 Drill Rig Set-up at the No. 3 Zone, NBM Property

The No.3 Zone has been the primary focus for diamond drilling in Garson's exploration program. More recently, the scope of the drill program has been extended to include the Kim Zone and the Boundary Zone. As an initial test, 2 holes have been completed at the Birch Zone. The number of holes and total metres drilled by zone are described in Table 11-1.

Table 11-1: Summary of the Diamond Drilling Completed by Garson at the NBM Property Prior to May, 2008

Zone	Number of Drill Holes	Total Footage	Total Metres	Number of Samples
No.3 Zone	62	95,281	29,042	2,946
Birch Zone	2	2,742	836	126
Boundary Zone	14	13,493	4,113	573
Kim Zone	7	5,674	1,730	442

11.1 Diamond drilling methods

All drilling completed by Garson on the property has been NQ diamond core drilling. The drilling has been completed using two drill rigs: 1 Boyles 37 and 1 Longyear 38. Dig-It Exploration, based in Alida, Saskatchewan has completed most of the drilling. Barkor Drilling based in Snow Lake completed the two holes that were drilled at the Main Mine.

The locations of all of the drill holes are surveyed and tied into the established grid systems on the property. The survey includes the location of the hole as well as the controls for the azimuth of the hole (front sights and back sights). The final set up of the drill (azimuth and dip) is checked and confirmed by Garson geologists prior to the commencement of drilling.

Down hole surveys are completed using a Fordia Flexit Smart-tool System. As a test of alignment, the first test is taken approximately 10 m down the hole. Subsequent tests are taken at a rate of one test every 50 to 60 m. Additional down hole surveys are completed as required.

Orientated core has been performed on some of the holes drilled at the No. 3 Zone and at Boundary Zone using the Ball Mark Core Orientation System. The orientated core will be logged for the structural orientation of the veins and the rock fabric. The structural measurements are made using a goniometer. Figure 11-2 shows a photograph of the goniometer used in the No. 3 Zone drilling program.

Geotechnical logging for the No. 3 Zone includes an estimate of the core recovery for the total hole in addition to determining the Rock Quality designation (RQD) for the interval of core 100 ft above the mineralized zone. Core recovery approaches 100% and no major issues have been recognized from the rock quality determinations.

Core is delivered daily from the drill rigs to the core shack by the drill contractor. The core is in the custody or control of the drill contractor or Garson personnel at all times.

Figure 11-2: Goniometer used by Garson for Structural Measurements

11.2 Diamond drilling results prior to May, 2008

The following section contains an outline of Garson's drilling programs and the results on the various zones.

11.2.1 No. 3 Zone Drilling

In the most recent campaign, 48 holes have been drilled at No. 3 Zone for a total footage of 74,630 ft (22,747 m). Drill hole collar locations and final lengths are contained in Table 11-2. The layout of No. 3 Zone drill holes is shown on Figure 11-3. The significant intersections are contained in Table 3.

The drilling at No. 3 Zone has confirmed the presence of mineralization in the historical inferred resource and extended the mineralization to a depth of approximately 1,600 vertical ft. The mineralization intersected is consistent with that described in the previous Technical Reports and in Section 9 of this Technical Report. The in-fill and extension drilling has resulted in the interpretation of both a Main Zone of mineralization and a Footwall Zone which both strike approximately 110° (grid east-west) and dip 40° to 45° to the north. Both zones plunge to the north-northeast. A representative section and long sections of the Main and Footwall Zones, are contained in Section 17 of this report.

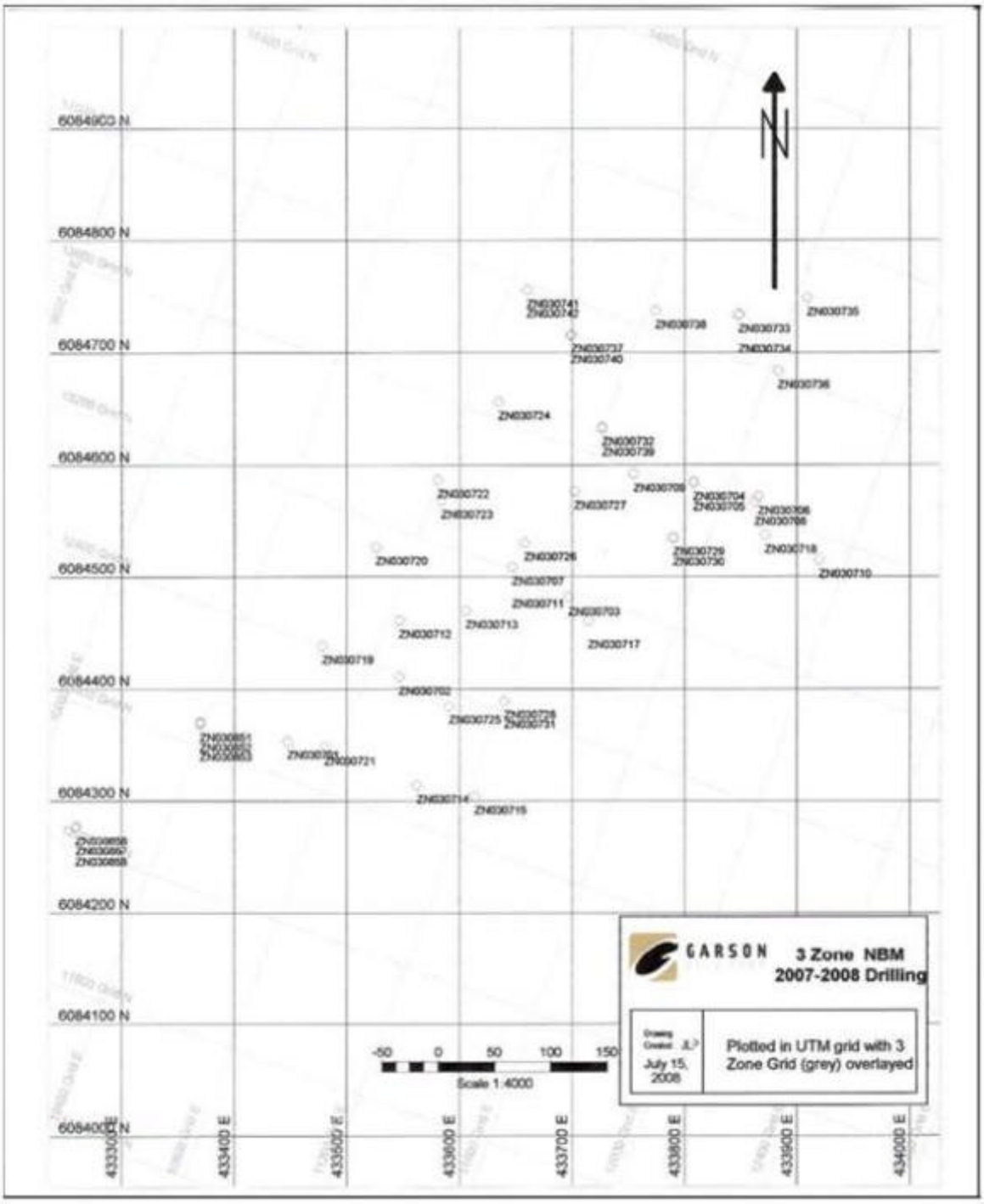

Figure 11-3: Drill Hole Collar Locations for the No. 3 Zone

Table 11-2: Summary of the No. 3 Zone Drill Hole Collar Locations and Lengths

Drill Hole Number	Easting	Northing	Azimuth (°)	Dip (°)	Length (ft)	Length (m)
ZN030701	10648.0	12501.0	175	-67	961.0	292.91
ZN030702	10888.5	12793.1	173	-68	1,099.0	334.98
ZN030703	11272.5	13180.0	176	-69	1,685.9	513.86
ZN030704	11501.3	13619.5	169	-58	1,718.7	523.86
ZN030705	11501.3	13620.5	169	-68	1,827.5	557.02
ZN030706	11692.3	13645.0	174	-67	1,915.5	583.84
ZN030707	11088.4	13206.3	174	-67	1,483.0	452.02
ZN030708	11688.5	13629.0	172	-53	1,719.2	524.01
ZN030709	11327.3	13582.9	174	-65	1,975.2	602.04
ZN030710	11923.0	13532.4	172	-66	1,926.0	587.04
ZN030711	11088.4	13207.3	174	-77	1,581.0	481.89
ZN030712	10833.0	12950.0	178	-69	1,388.0	423.06
ZN030713	11005.3	13042.3	174	-66	1,384.0	421.84
ZN030714	11047.4	12512.0	175	-63	1,030.2	314.00
ZN030715	11215.5	12539.1	166	-68	1,512.4	460.98
ZN030717	11349.8	13136.8	172	-66	1,483.0	452.02
ZN030718	11750.1	13546.5	170	-64	1,876.6	571.99
ZN030719	10645.4	12802.5	175	-65	1,122.0	341.99
ZN030720	10695.5	13126.5	174	-64	1,384.6	422.03
ZN030721	10754.1	12524.1	171	-65	981.0	299.01
ZN030722	10797.7	13372.3	171	-65	1,512.5	461.01
ZN030723	10827.5	13320.0	172	-65	1,374.7	419.01
ZN030724	10885.5	13648.8	170	-68	1,689.7	515.02
ZN030725	11055.9	12761.5	171	-70	1,227.0	373.99
ZN030726	11097.7	13285.6	166	-70	1,561.7	476.01
ZN030727	11183.6	13476.4	172	-68	1,660.1	506.00
ZN030728	11200.7	12829.7	174	-68	1,335.3	407.00
ZN030729	11500.0	13448.0	176	-55	1,679.8	512.00
ZN030730	11500.0	13447.0	171	-68	1,729.0	527.00
ZN030731	11200.7	12830.7	174	-62	1,325.9	404.13
ZN030732	11195.3	13680.2	169	-70	2,073.5	632.00
ZN030733A	11457.6	14124.2	177	-56	75.5	23.01
ZN030733	11457.6	14124.2	177	-58	2,260.5	689.00
ZN030734	11457.6	14127.2	175	-67	2,122.7	647.00
ZN030735	11620.5	14011.0	174	-60	2,132.5	649.99
ZN030736	11620.5	14012.0	171	-68	2,171.9	662.00
ZN030737	11017.5	13904.6	173	-65	1,984.9	605.00
ZN030738	11223.7	14053.9	173	-68	2,203.0	671.47
ZN030739	11195.3	13679.2	173	-63	1,994.8	608.02
ZN030740	11017.5	13903.6	176	-59	2,289.4	697.81
ZN030741	10852.0	13983.3	176	-54	1,781.5	543.00

Drill Hole Number	Easting	Northing	Azimuth (°)	Dip (°)	Length (ft)	Length (m)
ZN030742	10852.0	13984.3	173	-68	1,866.8	569.00
ZN030851	10389.6	12464.6	172	-50	1,069.1	325.86
ZN030852	10389.6	12465.6	172	-60	1,382.5	421.39
ZN030853	10389.6	12466.6	172	-68	1,060.3	323.18
ZN030856	10153.7	12055.9	158.7	-69	1,157.7	352.87
ZN030857	10153.7	12056.9	180	-52	734.9	224.00
ZN030858	10153.7	12054.9	188	-71	1,118.8	341.01
Total:					**74,630**	**22,747**

Table 11-3: Summary of Drill Hole Intersections for the No. 3 Zone

DrillHole	Mineral Intersections (Imperial Units)				Mineral Intersections (Metric Units)			
			Core	In-Situ			Core	In-Situ
Number	From	To	Length (ft)	Grade (Oz/ton)	From	To	Length (metres)	Grade (g/t)
ZN030701	818.7	834.4	15.7	0.07	249.54	254.33	4.79	2.25
Includes	832.9	834.4	1.5	0.27	253.87	254.33	0.46	9.16
ZN030701	44.0	897.7	3.3	0.11	13.41	273.62	1.01	3.65
ZN030702	970.0	986.2	16.2	0.04	295.66	300.59	4.94	1.47
ZN030702	1,059.7	1,063.5	3.8	0.42	323.00	324.15	1.16	14.44
ZN030702	1,079.4	1,092.3	12.9	0.05	329.00	332.93	3.93	1.86
ZN030703	1,370.0	1,371.7	1.7	0.05	417.58	418.09	0.52	1.71
ZN030704	1,572.7	1,592.4	19.7	0.12	479.36	485.36	6.00	4.11
Includes	1,586.0	1,591.0	5.0	0.23	483.41	484.94	1.52	7.89
ZN030705	1,692.8	1,697.6	4.8	0.21	515.97	517.43	1.46	7.20
ZN030706	1,722.4	1,729.7	7.3	0.10	524.99	527.21	2.23	3.43
ZN030707	325.9	334.0	8.1	0.15	99.33	101.80	2.47	5.14
Includes	325.9	330.2	4.3	0.22	99.33	100.64	1.31	7.54
ZN030707	1,228.9	1,235.4	6.5	0.31	374.57	376.55	1.98	10.63
ZN030708	256.6	278.7	22.1	0.09	78.21	84.95	6.74	3.09
Includes	256.6	272.3	15.7	0.12	78.21	83.00	4.79	4.11
ZN030709	163.3	178.1	14.8	0.14	49.77	54.28	4.51	4.80
Includes	170.3	178.1	7.8	0.21	51.91	54.28	2.38	7.20
ZN030709	1,571.2	1,572.4	1.2	0.31	478.90	479.27	0.37	10.63
ZN030709	1,611.0	1,615.1	4.1	0.14	491.03	492.28	1.25	4.80
ZN030710	494.4	496.9	2.5	0.09	150.69	151.46	0.76	3.09
ZN030711	659.2	664.5	5.3	0.12	200.92	202.54	1.62	4.11
Includes	659.2	661.8	2.6	0.18	200.92	201.72	0.79	6.17
ZN030711	1,308.5	1,315.1	6.6	0.24	398.83	400.84	2.01	8.23
Includes	1,308.5	1,312.1	3.6	0.40	398.83	399.93	1.10	13.71
ZN030712	1,124.9	1,128.0	3.1	0.32	342.87	343.81	0.94	10.97
ZN030712	1,256.5	1,259.3	2.8	0.18	382.98	383.83	0.85	6.17
ZN030712	1,277.6	1,279.2	1.6	0.16	389.41	389.90	0.49	5.49
ZN030713	1,136.1	1,142.7	6.6	0.34	346.28	348.29	2.01	11.66
ZN030713	1,244.4	1,253.6	9.2	0.32	379.29	382.10	2.80	10.97
ZN030714	921.4	922.9	1.5	0.07	280.84	281.30	0.46	2.40
ZN030715	1,280.4	1,296.3	15.9	0.11	390.27	395.11	4.85	3.77
Includes	1,287.1	1,294.0	6.9	0.20	392.31	394.41	2.10	6.86
ZN030717	1,339.0	1,364.8	25.8	0.08	408.13	415.99	7.86	2.74
Includes	1,357.0	1,364.8	7.8	0.16	413.61	415.99	2.38	5.49
ZN030718	1,474.0	1,475.2	1.2	0.11	449.28	449.64	0.37	3.77
ZN030718	1,579.6	1,584.9	5.3	0.06	481.46	483.08	1.62	2.06
ZN030719	1,047.7	1,048.5	0.8	0.53	319.34	319.58	0.24	18.17

DrillHole	Mineral Intersections (Imperial Units)				Mineral Intersections (Metric Units)			
			Core	In-Situ			Core	In-Situ
Number	From	To	Length (ft)	Grade (Oz/ton)	From	To	Length (metres)	Grade (g/t)
ZN030720	1,199.0	1,203.7	4.7	0.15	365.46	366.89	1.43	5.14
ZN030721	811.5	823.5	12.0	0.09	247.35	251.00	3.66	3.09
Includes	817.6	821.0	3.4	0.24	249.20	250.24	1.04	8.23
ZN030721	866.7	872.0	5.3	0.09	264.17	265.79	1.62	3.09
Includes	868.2	872.0	3.8	0.13	264.63	265.79	1.16	4.46
ZN030722	189.5	193.6	4.1	0.12	57.76	59.01	1.25	4.11
ZN030722	1,350.5	1,351.8	1.3	0.29	411.63	412.03	0.40	9.94
ZN030723	954.4	958.7	4.3	0.11	290.90	292.21	1.31	3.77
Includes	955.0	956.8	1.8	0.21	291.08	291.63	0.55	7.20
ZN030723	1,329.9	1,330.8	0.9	0.07	405.35	405.63	0.27	2.40
ZN030724	1,538.0	1,545.9	7.9	0.30	468.78	471.19	2.41	10.29
Includes	1,538.0	1,542.9	4.9	0.48	468.78	470.28	1.49	16.46
ZN030725	971.5	979.9	8.4	0.23	296.11	298.67	2.56	7.89
ZN030726	670.9	685.7	14.8	0.54	204.49	209.00	4.51	18.51
ZN030726	1,309.6	1,319.8	10.2	0.17	399.17	402.28	3.11	5.83
Includes	1,313.6	1,319.8	6.2	0.23	400.39	402.28	1.89	7.89
ZN030726	1,424.3	1,432.0	7.7	0.12	434.13	436.47	2.35	4.11
ZN030726	1,459.2	1,462.5	3.3	0.19	444.76	445.77	1.01	6.51
Includes	1,459.2	1,460.3	1.1	0.46	444.76	445.10	0.34	15.77
ZN030727	588.6	603.0	14.4	0.02	179.41	183.79	4.39	0.69
ZN030727	727.2	729.0	1.8	0.10	221.65	222.20	0.55	3.43
ZN030727	1,390.4	1,404.1	13.7	0.34	423.79	427.97	4.18	11.66
Includes	1,393.0	1,403.1	10.1	0.45	424.59	427.66	3.08	15.43
ZN030727	1,499.9	1,510.3	10.4	0.08	457.17	460.34	3.17	2.74
Includes	1,505.2	1,505.8	0.6	1.08	458.78	458.97	0.18	37.03
ZN030728	1,114.1	1,122.1	8.0	0.13	339.58	342.02	2.44	4.46
Includes	1,120.0	1,122.1	2.1	0.39	341.38	342.02	0.64	13.37
ZN030729	no significant assays							
ZN030730	1,544.3	1,544.8	0.5	0.27	470.70	470.86	0.15	9.26
ZN030731	1,074.3	1,078.4	4.1	0.05	327.45	328.70	1.25	1.71
ZN030731	1,121.2	1,122.5	1.3	0.09	341.74	342.14	0.40	3.09
ZN030732	1,553.9	1,558.0	4.1	0.45	473.63	474.88	1.25	15.43
ZN030733	1,848.0	1,852.8	4.8	0.22	563.27	564.73	1.46	7.54
Includes	1,848.0	1,849.5	1.5	0.62	563.27	563.73	0.46	21.26
ZN030734	1,781.5	1,788.1	6.6	0.21	543.00	545.01	2.01	7.20
ZN030734	1,867.2	1,870.6	3.4	0.34	569.12	570.16	1.04	11.66
ZN030735	1,822.6	1,823.9	1.3	0.13	555.53	555.92	0.40	4.46
ZN030736	174.9	176.9	2.0	0.12	53.31	53.92	0.61	4.11
ZN030736	186.3	187.6	1.3	0.17	56.78	57.18	0.40	5.83

DrillHole	Mineral Intersections (Imperial Units)				Mineral Intersections (Metric Units)			
			Core	In-Situ			Core	In-Situ
Number	From	To	Length (ft)	Grade (Oz/ton)	From	To	Length (metres)	Grade (g/t)
ZN030736	237.2	238.8	1.6	0.19	72.30	72.79	0.49	6.51
ZN030736	1,907.7	1,908.8	1.1	0.13	581.47	581.80	0.34	4.46
ZN030737	1,668.6	1,675.1	6.5	0.16	508.59	510.57	1.98	5.49
Includes	1,668.6	1,672.3	3.7	0.27	508.59	509.72	1.13	9.26
ZN030738	1,243.1	1,251.4	8.3	0.07	378.90	381.43	2.53	2.40
Includes	1,246.7	1,249.2	2.5	0.13	379.99	380.76	0.76	4.46
ZN030738	1,758.5	1,766.6	8.1	0.40	535.99	538.46	2.47	13.71
ZN030739	1,512.5	1,520.7	8.2	0.17	461.01	463.51	2.50	5.83
Includes	1,512.5	1,515.4	2.9	0.40	461.01	461.89	0.88	13.71
ZN030739	1,620.2	1,622.1	1.9	0.24	493.84	494.42	0.58	8.23
ZN030740	811.6	812.8	1.2	0.18	247.38	247.74	0.37	6.17
ZN030740	1,096.1	1,104.0	7.9	0.22	334.09	336.50	2.41	7.54
ZN030740	1,307.4	1,315.6	8.2	0.11	398.50	400.99	2.50	3.77
Includes	1,310.8	1,315.6	4.8	0.14	399.53	400.99	1.46	4.80
ZN030740	1,330.5	1,335.5	5.0	0.11	405.54	407.06	1.52	3.77
ZN030740	1,657.7	1,661.7	4.0	0.16	505.27	506.49	1.22	5.49
ZN030740	1,778.2	1,781.0	2.8	0.37	542.00	542.85	0.85	12.69
ZN030741	1,687.8	1,690.1	2.3	0.09	514.44	515.14	0.70	3.09
ZN030742	1,095.6	1,100.8	5.2	0.16	333.94	335.52	1.58	5.49
Includes	1,095.6	1,097.7	2.1	0.13	333.94	334.58	0.64	4.46
ZN030742	1,745.4	1,751.6	6.2	0.15	532.00	533.89	1.89	5.14
Includes	1,745.4	1,746.1	0.7	1.15	532.00	532.21	0.21	39.43
ZN030851	788.0	788.9	0.9	0.61	240.18	240.46	0.27	20.91
ZN030851	819.1	830.4	11.3	0.19	249.66	253.11	3.44	6.51
Includes	819.1	823.5	4.4	0.40	249.66	251.00	1.34	13.71
ZN030852	802.7	814.8	12.1	0.14	244.66	248.35	3.69	4.80
Includes	811.4	814.8	3.4	0.45	247.31	248.35	1.04	15.43
ZN030852	986.0	988.3	2.3	0.15	300.53	301.23	0.70	5.14
ZN030853	827.6	831.4	3.8	0.24	252.25	253.41	1.16	8.23
ZN030856	565.4	566.7	1.3	0.19	172.33	172.73	0.40	6.51
ZN030856	651.0	656.2	5.2	0.08	198.42	200.01	1.58	2.74
ZN030857	522.5	524.7	2.2	0.10	159.26	159.93	0.67	3.43
ZN030857	631.9	634.4	2.5	0.12	192.60	193.37	0.76	4.11
ZN030857	685.7	687.4	1.7	0.17	209.00	209.52	0.52	5.83
ZN030857	725.0	726.4	1.4	0.10	220.98	221.41	0.43	3.43
ZN030858	no significant assays							

Table provided by Garson Gold Corp.

11.2.2 Kim Zone Drilling

The Kim Zone is located approximately 200 ft to 500 ft south of the No. 3 Zone. Previous drilling, although limited, had intersected significant mineralization at the Kim Zone. The current drilling has been designed to follow-up the positive historical results and to test along the strike for a similar controlling structure orientated at an azimuth of 110°. The location and total footage of the holes drilled at Kim Zone are summarized in Table 11-4 and the significant intercepts are summarized in Table 11-5. Figure11-4 illustrates the drill collar locations for the Kim Zone.

Geological interpretation of the Kim Zone and integration of the drilling into the regional geological picture and the relationship with the Boundary and No. 3 Zone are in progress.

11.2.3 Boundary Zone Drilling

The Boundary Zone is located approximately 1,000 ft to 1,500 ft south of the No. 3 Zone approximately 500 ft to 1,200 ft south of the Kim Zone. Significant previous drilling at the Boundary Zone has resulted in the identification of three near-surface zones of low grade mineralization hosted by felsic volcanic rocks. The drilling was designed to test the potential for a larger tonnage in the Boundary Zone and has resulted in the intersection of significant gold mineralization hosted by mafic volcaniclastic rocks between the Boundary felsic volcanic rocks and the Kim Zone. The location and total footage of the holes drilled at Boundary Zone are summarized in Table 11-4 and significant intercepts are summarized in Table 11-5. Figure 11-4 illustrates the drill collar locations for the Boundary Zone.

Geological interpretation of the Boundary Zone and integration of the drilling into the regional geological picture and relationship with the Kim and No. 3 Zones are in progress.

11.2.4 Birch Zone Drilling

The Birch Zone is located approximately 1,500 ft north of the No. 3 Zone. The Birch Zone has had significant historical drilling and contains a historical inferred resource. Two holes were drilled in the Birch Zone by Garson. The location of the Birch holes and significant assays are contained in Table 11-4 and Table 11-5 respectively. Figure 11-5 illustrates the drill collar locations for the Birch Zone.

Further drilling at Birch Zone will be based on the results of an ongoing desk top evaluation of the historical drilling and inferred resource.

Table 11-4: Boundary (BDY), Birch (B) and Kim (Kim) Zones Drill Hole Collars and Lengths

Drill Hole Number	Zone	Easting	Northing	Azimuth (°)	Dip (°)	Length (ft)	Length (m)
BDY08-23	Boundary	34177.9	44904.9	231	-61	1,148.3	350.00
BDY08-24	Boundary	34177.9	44905.9	242	-62	1,161.4	353.99
BDY08-25	Boundary	34177.9	44903.9	219	-58	1,246.7	379.99
BDY08-27	Boundary	33896.1	44954.1	228	-67	1,158.1	352.99
BDY08-28	Boundary	33895.1	44953.1	228	-45	814.0	248.11
BDY08-29	Boundary	33897.1	44955.1	228	-86	1,276.2	388.99
BDY08-30	Boundary	33879.0	45064.9	226	-67	925.2	282.00
BDY08-31	Boundary	33887.8	45064.9	226	-45	882.6	269.02
BDY08-32	Boundary	33888.0	45064.9	226	-86	1,128.6	344.00
BDY08-36	Boundary	33707.0	45196.0	208	-69	902.2	274.99
BDY08-37	Boundary	33706.0	45197.0	229	-67	876.0	267.00
BDY08-38	Boundary	33705.0	45198.0	246	-66	876.0	267.00
BDY08-39	Boundary	34663.3	45478.0	241	-67	1,473.7	449.18
BDY08-40	Boundary	34663.3	45479.0	241	-54	1,098.1	334.70
KIM08-01	Kim Zone	10599.7	11190.0	220	-45	459.3	139.99
KIM08-02	Kim Zone	10599.7	11189.0	220	-75	783.3	238.75
KIM08-03	Kim Zone	10799.9	11179.0	193	-45	617.1	188.09
KIM08-04	Kim Zone	10799.9	11180.0	193	-70	715.2	217.99
KIM08-05	Kim Zone	10998.2	11129.0	193	-45	951.4	289.99
KIM08-06	Kim Zone	10998.2	11130.0	193	-75	960.3	292.70
KIM08-07	Kim Zone	11335.2	11070.4	193	-45	1,187.7	362.01
B07-35	Birch	10093.0	16044.8	173	-59	1,601.0	487.98
B07-36	Birch	10093.0	16044.8	173	-69	1,140.8	347.72

Table 11-5: Boundary (BDY), Birch (B) and Kim (Kim) Zones Drill Hole Intersections

Drill Hole	Mineral Intersections (Imperial Units)				Mineral Intersections (Metric Units)			
			Core	In-Situ			Core	In-Situ
Number	From	To	Length	Grade	From	To	Length	Grade
			(ft)	(Oz/ton)			(metres)	(g/t)
BDY08-23	599.0	602.4	3.4	0.13	182.58	183.61	1.04	4.46
BDY08-23	621.6	635.0	13.4	0.06	189.46	193.55	4.08	2.06
BDY08-24	634.7	639.5	4.8	0.10	193.46	194.92	1.46	3.43
BDY08-24	804.3	808.8	4.5	0.08	245.15	246.52	1.37	2.74
BDY08-24	817.4	820.5	3.1	0.11	249.14	250.09	0.94	3.77
BDY08-25	600.8	603.0	2.2	0.11	183.12	183.79	0.67	3.77
BDY08-25	639.9	659.3	19.4	0.10	195.04	200.95	5.91	3.43
Includes	639.9	644.1	2.8 4.2	2.9 0.27	195.04	196.32	1.2.12 1.28	.13 9.26
BDY08-27	5 454.6	16 464.1	.2.17 9.5	2.18 0.46	138.56	141.46	1.2.21 2.90	22 15.77
BDY08-27	4 507.0	25 508.5	.2.26 1.5	2.27 0.17	154.53	154.99	1.2.30 0.46	.31 5.83
BDY08-28	3 473.9	34 484.4	.35 10.5	2.36 0.36	144.44	147.65	1.2.39 3.20	40 12.34
BDY08-28	2 623.5	43 640.5	.44 17.0	2.45 0.06	190.04	195.22	1.2.48 5.18	.49 2.06
BDY08-29	1 527.7	52 530.6	2.53 2.9	2.54 0.07	160.84	161.73	1.2.57 0.88	.58 2.40
BDY08-29	0 668.7	61 670.6	2.62 1.9	2.63 0.11	203.82	204.40	1.2.66 0.58	.67 3.77
BDY08-29	9 722.3	70 725.1	2.71 2.8	2.72 0.11	220.16	221.01	1.2.75 0.85	.76 3.77
BDY08-29	8 759.1	79 760.0	2.80 0.9	2.81 0.14	231.37	231.65	1.2.84 0.27	.85 4.80
BDY08-29	7 899.1	88 903.3	2.89 4.2	2.90 0.13	274.05	275.33	1.2.93 1.28	.94 4.46
BDY08-29	6 980.3	97 981.3	2.98 1.0	2.99 0.16	00 298.80	1 299.10	1.2.102 0.30	.103 5.49
BDY08-30	05 570.8	106 574.3	2.107 3.5	2.108 0.25	09 173.98	0 175.05	1.2.111 1.07	.112 8.57
BDY08-30	14 814.0	115 821.1	2.116 7.1	2.117 0.08	.8 248.11	9 250.27	1.2.120 2.16	.121 2.74
BDY08-31	23 525.4	124 528.6	2.125 3.2	2.126 0.14	27 160.14	8 161.12	1.2.129 0.98	.130 4.80
BDY08-32	32 647.4	133 671.1	.134 23.7	2.135 0.22	36 197.33	7 204.55	1.2.138 7.22	.139 7.54
includes	41 647.4	142 658.4	.143 11.0	2.144 0.44	45 197.33	6 200.68	1.2.147 3.35	148 15.09
BDY08-32	50 819.5	151 821.9	2.152 2.4	2.153 0.37	54 249.78	5 250.52	1.2.156 0.73	157 12.69
BDY08-32	59 871.8	160 880.7	2.161 8.9	2.162 0.10	63 265.72	4 268.44	1.2.165 2.71	.166 3.43
BDY08-36	no significant assays							
BDY08-37	no significant assays							
BDY08-40	1037.3	1057.1	19.8	0.12	316.17	322.20	6.04	4.11
includes	1040.9	1048.0	7.1	0.19	317.27	319.43	2.16	6.51
BDY08-40	1355.1	1368.7	13.6	0.06	413.03	417.18	4.15	2.06
KIM08-01	180.5	182.0	1.5	0.60	55.02	55.47	0.46	20.57
KIM08-01	294.7	309.6	14.9	0.21	89.82	94.37	4.54	7.20
KIM08-02	352.8	354.2	1.4	0.26	107.53	107.96	0.43	8.91
KIM08-03	176.3	178.7	2.4	0.10	53.74	54.47	0.73	3.43
KIM08-04	196.1	198.3	2.2	0.44	59.77	60.44	0.67	15.09
KIM08-05	no significant assays							
KIM08-06	44.8	45.9	1.1	0.12	13.66	13.99	0.34	4.11
KIM08-07	no significant assays							
B07-35	940.4	971.9	31.5	0.08	286.63	296.24	9.60	2.72

Drill Hole	Mineral Intersections (Imperial Units)				Mineral Intersections (Metric Units)			
			Core	In-Situ			Core	In-Situ
includes	940.4	948.3	7.9	0.10	286.63	289.04	2.40	3.50
And	969	971.9	2.9	0.31	295.35	296.24	0.90	10.63
B07-35	1177.8	1180.3	1	0.38	358.99	359.76	0.30	12.92
B07-36	579.7	582.4	2.7	0.17	176.69	177.52	0.80	5.83
B07-36	708.7	711.6	2.9	0.11	216.01	216.90	0.90	3.81
B07-36	772.3	775.6	3.3	0.10	235.40	236.40	1.00	3.43

Table provided by Garson Gold Corp.

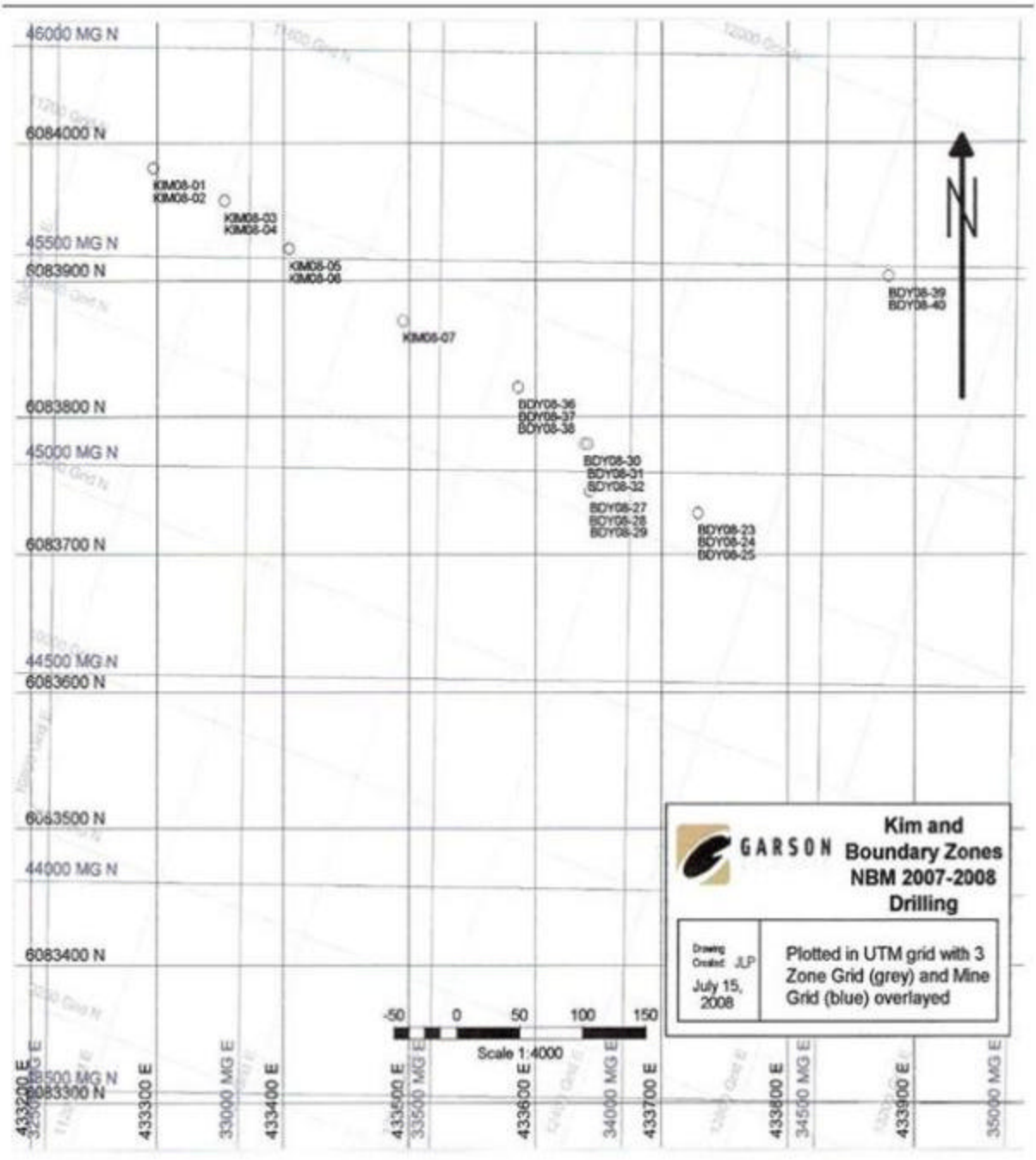

Figure 11-4: Drill Hole Collar Locations for the Kim and Boundary Zones

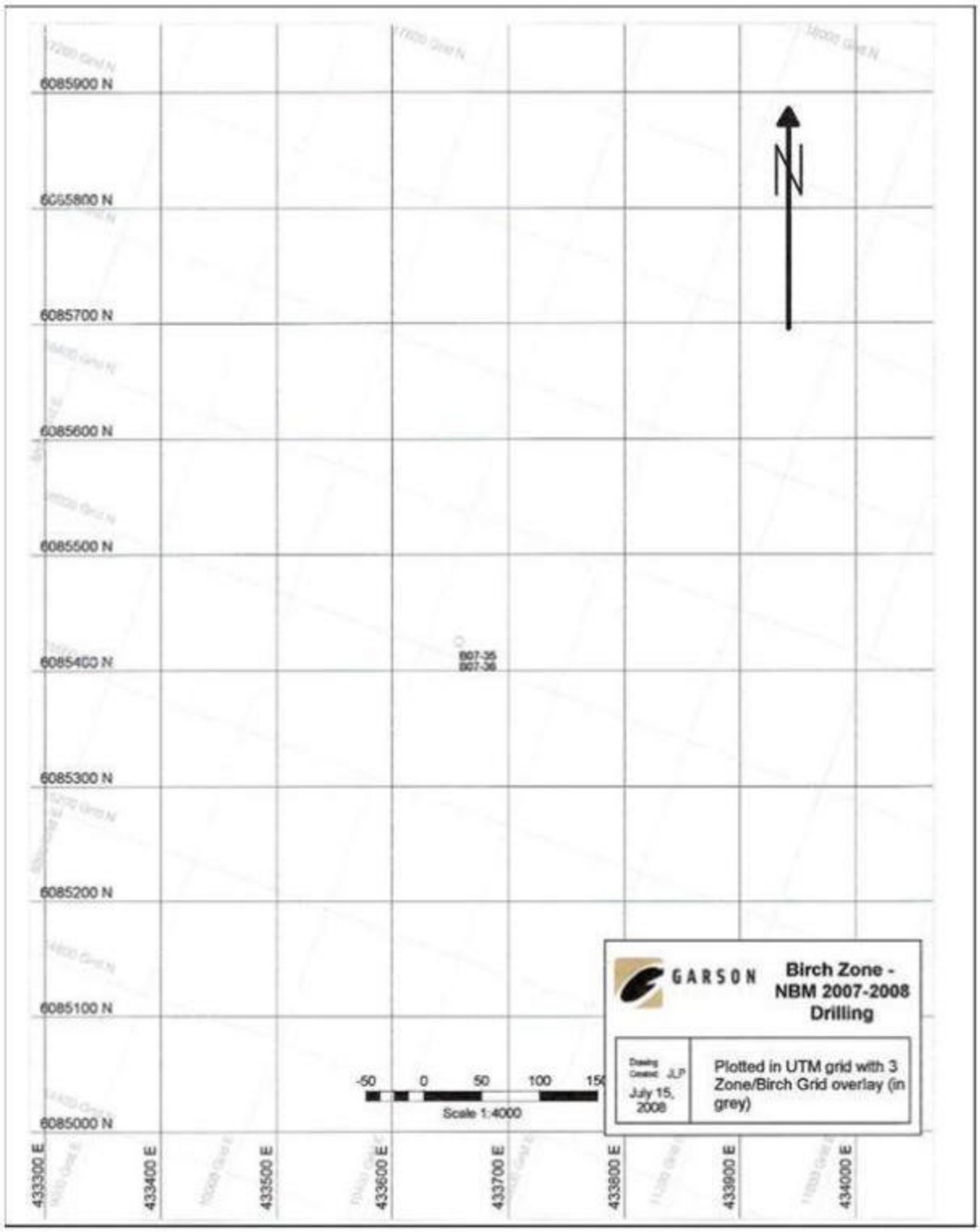

Figure 11-5: Drill Hole Collar Locations for the Birch Zone

11.3 Diamond Drilling Results for May to October, 2008

This section will discuss the drilling programs conducted by Garson on a number of mineralized zones between May and October, 2008. These results were not discussed in the July, 2008 Technical Report. The preliminary assessment for the No. 3 Zone is not impacted by the results of the May to October, 2008 drilling program.

The May to October, 2008 drilling focused on further defining the Kim and Boundary Zones as well as the Hogg Zone in the Main Mine. The total number of drill holes, the metres drilled and the number of samples taken for each zone are summarized in Table 11-6.

Table 11-6: Summary of the Diamond Drilling Completed from May to October, 2008

Zone	Number of Drill Holes	Total Footage	Total Metres	Number of Samples
Boundary	24	6,889	2,100	1,575
Kim	11	10,272	3,131	975
Hogg (Main Mine)	4	4,603	1,403	777
Total	**39**	**21,764**	**6,634**	**3,327**
Table provided by Garson Gold Corp.				

The results of the drilling programs are discussed below.

11.3.1 Boundary Zone Drilling

The initial drilling by Garson was designed to test down plunge of the earlier drilling and intercepted excellent mineralization in the mafic volcanic rocks. Through the course of Garson's drilling, it was recognized that there could be a low grade/large tonnage mineral potential within the felsic rocks. The most recent drilling has targeted the area immediately below the overburden in this area. The drill holes have intersected significant widths of potentially open pitable grade mineralization.

A total of 24 drill holes were completed on the Boundary Zone between May and October, 2008. The total footage contained in the 24 drill holes was 6,889 ft (2,100 m). A total of 1,575 samples were taken during the Boundary Zone drilling program.

The location and total footage of the holes drilled at the Boundary Zone are summarized in Table 11-7 and significant intercepts are summarized in Table 11-8. Figure 11-6 illustrates the drill collar locations for the most recent drilling at Boundary Zone.

Table 11-7: Summary of the Boundary Zone Drill Hole Collars and Locations from May to October, 2008

Drill Hole Number	Easting	Northing	Azimuth (_)	Dip (_)	Length (ft)	Length (m)
BDY08-57	433619.8	6083607.5	238	-50	626.6	191.0
BDY08-58	433615.8	6083468.1	235	-48	95.1	29.0
BDY08-59	433628.4	6083479.3	235	-77	213.0	64.9
BDY08-60	433610.6	6083501.8	235	-50	154.4	47.1
BDY08-61	433555.8	6083504.2	235	-51	134.5	41.0
BDY08-62	433563.1	6083509.8	235	-87	547.9	167.0
BDY08-63	433610.4	6083537.0	235	-51	308.4	94.0
BDY08-64	433549.1	6083520.7	243	-49	488.9	149.0
BDY08-65	433580.8	6083561.2	235	-45	547.9	167.0
BDY08-66	433485.5	6083534.6	235	-50	301.8	92.0
BDY08-67	433456.1	6083542.2	235	-50	295.3	90.0
BDY08-68	433411.0	6083547.3	235	-47	173.9	53.0
BDY08-69	433447.7	6083579.0	235	-55	183.7	56.0
BDY08-70	433394.3	6083575.8	235	-65	193.6	59.0
BDY08-71	433444.5	6083614.9	235	-65	497.7	151.7
BDY08-72	433498.2	6083634.5	250	-60	331.4	101.0
BDY08-73	433485.3	6083493.7	235	-50	351.1	107.0
BDY08-74	433419.4	6083486.1	235	-50	164.0	50.0
BDY08-75	433413.8	6083520.0	235	-55	124.7	38.0
BDY08-76	433611.0	6083467.5	55	-55	154.2	47.0
BDY08-77	433559.7	6083510.8	55	-63	370.7	113.0
BDY08-79	433429.1	6083500.0	55	-65	223.1	68.0
BDY08-80	433461.5	6083506.1	10	-62	154.2	47.0
BDY08-81	433426.0	6083561.1	55	-55	252.6	77.0

Table 11-8: Summary of the Boundary Zone Drill Hole Intersections from May to October, 2008

Drill Hole Number	Mineral Intersections							
	Imperial Measurements (ft)			Gold Assay (Oz/ton)	Metric Measurements (metres)			Gold Assay (g/t)
	From	Core Length			To	Core Length		
BDY08-57	258.6	294.3	35.7	0.08	78.8	89.7	10.9	2.76
Includes	285.5	294.3	8.8	0.10	87.0	89.7	2.7	3.49
BDY08-57	402.5	412.0	9.5	0.06	122.7	125.6	2.9	2.09
BDY08-57	513.0	517.2	4.2	0.10	156.4	157.6	1.3	3.27
BDY08-57	552.0	568.7	16.7	0.07	168.3	173.3	5.1	2.52
Includes	554.0	560.5	6.5	0.13	168.9	170.8	2.0	4.56
BDY08-58	No significant assays							
BDY08-59	No significant assays							
BDY08-60	62.2	67.9	5.7	0.03	19.0	20.7	1.7	1.02
BDY08-61	No significant assays							
BDY08-62	110.3	117.3	7.0	0.07	33.6	35.8	2.1	2.45
BDY08-62	253.5	259.0	5.5	0.07	77.3	78.9	1.7	2.44
BDY08-62	431.6	434.2	2.6	0.08	131.6	132.4	0.8	2.80
BDY08-62	459.3	464.6	5.3	0.06	140.0	141.6	1.6	2.05
BDY08-63	118.9	123.0	4.1	0.08	36.2	37.5	1.3	2.59
BDY08-63	150.7	164.0	13.3	0.04	45.9	50.0	4.1	1.39
Includes	155.0	159.8	4.8	0.08	47.2	48.7	1.5	2.76
BDY08-63	170.4	190.0	19.6	0.06	51.9	57.9	6.0	1.91
Includes	181.0	190.0	9.0	0.08	55.2	57.9	2.7	2.89
BDY08-64	208.8	212.8	4.0	0.08	63.6	64.9	1.2	2.71
BDY08-64	289.0	290.1	1.1	0.11	88.1	88.4	0.3	3.84
BDY08-64	299.1	307.5	8.4	0.08	91.2	93.7	2.6	2.82
BDY08-64	313.8	321.1	7.3	0.07	95.7	97.9	2.2	2.28
BDY08-64	374.6	378.2	3.6	0.14	114.2	115.3	1.1	4.83
BDY08-65	104.7	107.5	2.8	0.05	31.9	32.8	0.9	1.66
BDY08-65	190.0	199.0	9.0	0.06	57.9	60.7	2.8	1.94
BDY08-65	421.4	424.2	2.8	0.04	128.4	129.3	0.9	1.20
BDY08-65	445.2	454.5	9.3	0.05	135.7	138.5	2.8	1.54
Includes	450.6	454.5	3.9	0.07	137.3	138.5	1.2	2.31
BDY08-66	60.5	62.6	2.1	0.10	18.4	19.1	0.6	3.27
BDY08-66	104.7	106.7	2.0	0.16	31.9	32.5	0.6	5.38
BDY08-66	117.8	129.8	12.0	0.06	35.9	39.6	3.7	2.17
BDY08-66	178.6	209.5	30.9	0.06	54.4	63.9	9.4	2.17
Includes	192.7	209.5	16.8	0.07	58.7	63.9	5.1	2.50
BDY08-67	23.3	26.1	2.8	0.06	7.1	8.0	0.9	2.14
BDY08-67	80.4	83.7	3.3	0.09	24.5	25.5	1.0	3.03
BDY08-67	110.0	151.0	41.0	0.03	33.5	46.0	12.5	1.08

Drill Hole Number	Mineral Intersections							
	Imperial Measurements (ft)			Gold Assay (Oz/ton)	Metric Measurements (metres)			Gold Assay (g/t)
	From	Core Length			To	Core Length		
Includes	110.0	119.8	9.8	0.07	33.5	36.5	3.0	2.45
BDY08-68	76.8	89.3	12.5	0.06	23.4	27.2	3.8	2.03
BDY08-69	90.6	119.0	28.4	0.04	27.6	36.3	8.7	1.48
Includes	90.6	95.3	4.7	0.09	27.6	29.1	1.4	2.97
Includes	109.7	119.0	9.3	0.07	33.4	36.3	2.8	2.32
BDY08-69	125.6	135.3	9.7	0.07	38.3	41.2	3.0	2.31
BDY08-70	84.3	90.2	5.9	0.04	25.7	27.5	1.8	1.37
BDY08-71	125.0	126.4	1.4	0.15	38.1	38.5	0.4	5.14
BDY08-71	150.0	151.2	1.2	0.12	45.7	46.1	0.4	4.08
BDY08-71	209.3	210.5	1.2	0.08	63.8	64.2	0.4	2.71
BDY08-71	232.7	236.0	3.3	0.04	70.9	71.9	1.0	1.45
BDY08-72	261.6	262.8	1.2	0.18	79.7	80.1	0.4	6.15
BDY08-72	290.7	297.7	7.0	0.05	88.6	90.7	2.1	1.72
BDY08-73	167.8	179.2	11.4	0.04	51.2	54.6	3.5	1.37
Includes	167.8	171.4	3.6	0.06	51.2	52.2	1.1	1.93
BDY08-74	57.7	62.0	4.3	0.03	17.6	18.9	1.3	0.87
BDY08-75	No significant assays							
BDY08-76	No significant assays							
BDY08-77	201.4	361.4	160.0	0.07	61.4	110.2	48.8	2.43
BDY08-77	70.6	104.5	33.9	0.06	21.5	31.9	10.3	2.09
Includes	70.6	82.4	11.8	0.10	21.5	25.1	3.6	3.44
BDY08-77	146.2	189.0	42.8	0.07	44.6	57.6	13.1	2.46
Includes	148.2	159.7	11.5	0.10	45.2	48.7	3.5	3.56
Includes	170.9	187.0	16.1	0.11	52.1	57.0	4.9	3.70
BDY08-77	201.4	251.7	50.3	0.08	61.4	76.7	15.3	2.68
Includes	201.4	220.2	18.8	0.12	61.4	67.1	5.7	4.09
Includes	231.9	313.9	82.0	0.12	70.7	73.2	2.5	4.11
BDY08-77	262.5	356.6	94.1	0.08	80.0	108.7	28.7	2.65
Includes	297.0	307.4	10.4	0.17	90.5	93.7	3.2	5.83
Includes	320.6	361.4	40.8	0.10	97.7	110.2	12.4	3.46
BDY08-79	26.2	111.9	85.7	0.06	8.0	34.1	26.1	2.23
Includes	64.0	111.7	47.7	0.10	19.5	34.1	14.6	3.36
BDY08-80	No significant assays							
BDY08-81	105.0	231.4	126.4	0.06	32.0	70.6	38.6	2.18
Includes	105.0	122.8	17.8	0.10	32.0	37.5	5.5	3.40
Includes	192.3	231.5	39.2	0.08	58.6	70.6	12.0	2.69

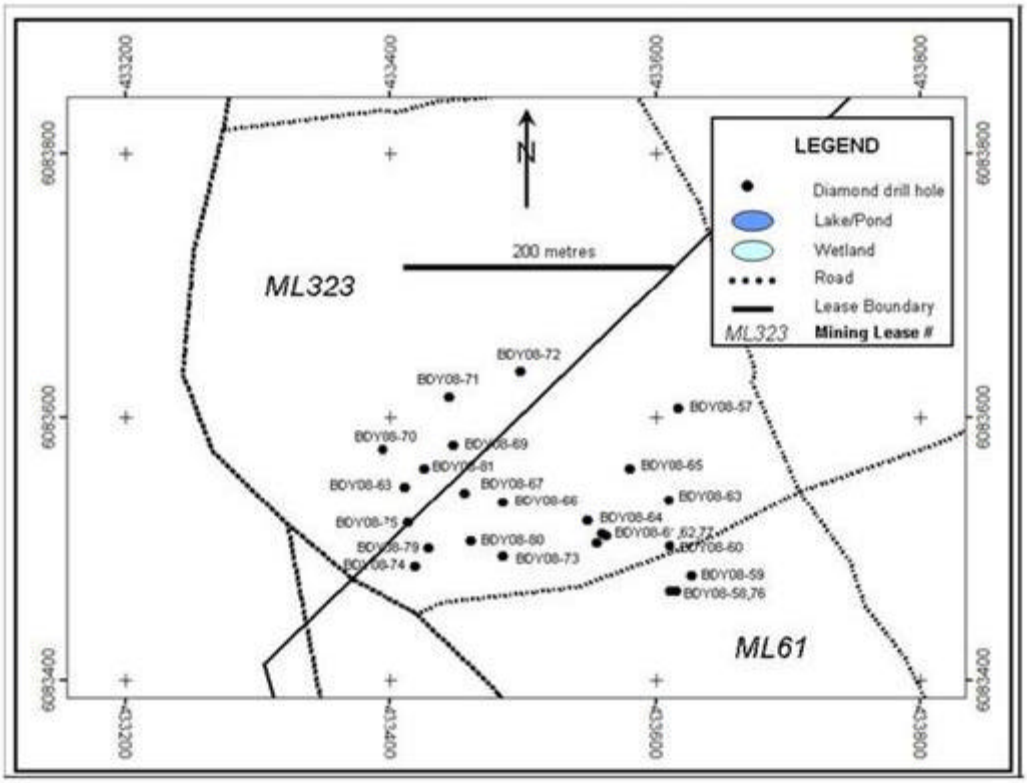

Figure 11-6: Drill Hole Collar Locations for the Boundary Zone

11.3.2 Kim Zone Drilling

Limited previous drilling has intersected significant mineralization at the Kim Zone. Garson's drilling program prior to May, 2008 was designed to follow-up the positive results obtained by the previous programs and to test along the strike for a controlling structure oriented at an azimuth of 110° similar to the No. 3 Zone. Garson's recent drilling program from May to October, 2008 was planned to test the potential plunge of mineralization between two previously drilled holes (ZN0307-15 and KIM08-01) containing significant mineral intersections.

A total of 11 drill holes were completed on the Kim Zone between May and October, 2008. The total footage contained in the 11 drill holes was 10,272 ft (3,131 m). A total of 975 samples were taken during the Kim Zone drilling program.

The location and total footage of the holes drilled at Kim Zone are summarized in Table 11-9 and Figure 11-7 significant intercepts are summarized in Table 11-10.

Table 11-9: Summary of the Kim Zone Drill Hole Collars and Locations from May to October, 2008

Drill Hole Number	Easting	Northing	Azimuth (_)	Dip (_)	Length (ft)	Length (m)
KIM08-08	433275.9	6084057.4	213	-68	626.6	191.0
KIM08-09	433336.6	6083985.1	204	-74	636.5	194.0
KIM08-10	433368.1	6084130.7	204	-53	872.7	266.0
KIM08-11	433405.3	6084104.9	198	-53	921.9	281.0
KIM08-12	433405.3	6084104.9	198	-75	902.2	275.0
KIM08-13	433474.8	6084056.2	208	-60	882.6	269.0
KIM08-14	433478.1	6084116.8	199	-68	905.5	276.0
KIM08-15	433484.7	6084164.3	198	-77	1138.5	347.0
KIM08-16	433544.6	6084132.7	199	-62	1030.2	314.0
KIM08-17	433544.6	6084132.7	199	-78	1276.3	389.0
KIM08-18	433596.5	6084098.7	199	-80	1079.4	329.0

Table 11-10: Summary of the Kim Zone Drill Hole Intersections from May to October, 2008

Drill Hole Number	Mineral Intersections							
	Imperial Measurements (ft)			Gold Assay (Oz/ton)	Metric Measurements (metres)			Gold Assay (g/t)
	From	Core Length			To	Core Length		
KIM08-08	396.8	398.9	2.1	0.12	121.0	121.6	0.6	4.15
KIM08-09	101.3	102.0	0.7	0.12	30.9	31.1	0.2	4.15
KIM08-09	262.7	266.9	4.2	0.11	80.1	81.4	1.3	3.60
KIM08-09	366.3	368.6	2.3	0.09	111.7	112.4	0.7	3.25
KIM08-09	412.7	415.2	2.5	0.27	125.8	126.6	0.8	9.29
KIM08-09	573.0	575.8	2.8	0.12	174.7	175.5	0.9	4.08
KIM08-10	627.2	628.7	1.5	0.07	191.2	191.6	0.5	2.24
KIM08-10	796.0	807.2	11.2	0.06	242.6	246.0	3.4	1.89
Includes	796.0	800.7	4.7	0.09	242.6	244.1	1.4	3.18
KIM08-11	800.8	809.8	9.0	0.06	244.1	246.8	2.7	2.13
KIM08-12	354.0	357.8	3.8	0.13	107.9	109.1	1.2	4.37
KIM08-13	774.1	775.1	1.0	0.17	236.0	236.3	0.3	5.83
KIM08-14	596.8	599.5	2.7	0.11	181.9	182.7	0.8	3.91
KIM08-15	229.1	233.7	4.6	0.06	69.8	71.2	1.4	2.07
KIM08-15	501.4	502.6	1.2	0.23	152.8	153.2	0.4	7.88
KIM08-15	1001.7	1003.4	1.7	0.26	305.3	305.8	0.5	8.78
KIM08-16	No significant assays							
KIM08-17	157.5	159.1	1.6	0.08	48.0	48.5	0.5	2.74
KIM08-17	762.4	763.8	1.4	0.22	232.4	232.8	0.4	7.44
KIM08-17	935.2	938.0	2.8	0.14	285.1	285.9	0.9	4.77
KIM08-17	1015.0	1016.5	1.5	0.09	309.4	309.8	0.4	2.91
KIM08-18	794.9	803.4	8.5	0.06	242.3	244.9	2.6	2.06

11.3.3 Main Mine – Hogg Zone

The Hogg Zone is located on the eastern side of the NBM. Mining was conducted below the 1780 Level during the 1995 to 2005 production phase with a limited amount of production from the 1280 Level prior to the closure of the mine in 2005. The current drill holes were planned for the preliminary interpretation of the up dip potential at the Upper Hogg Zone adjacent to the development in the Main Mine.

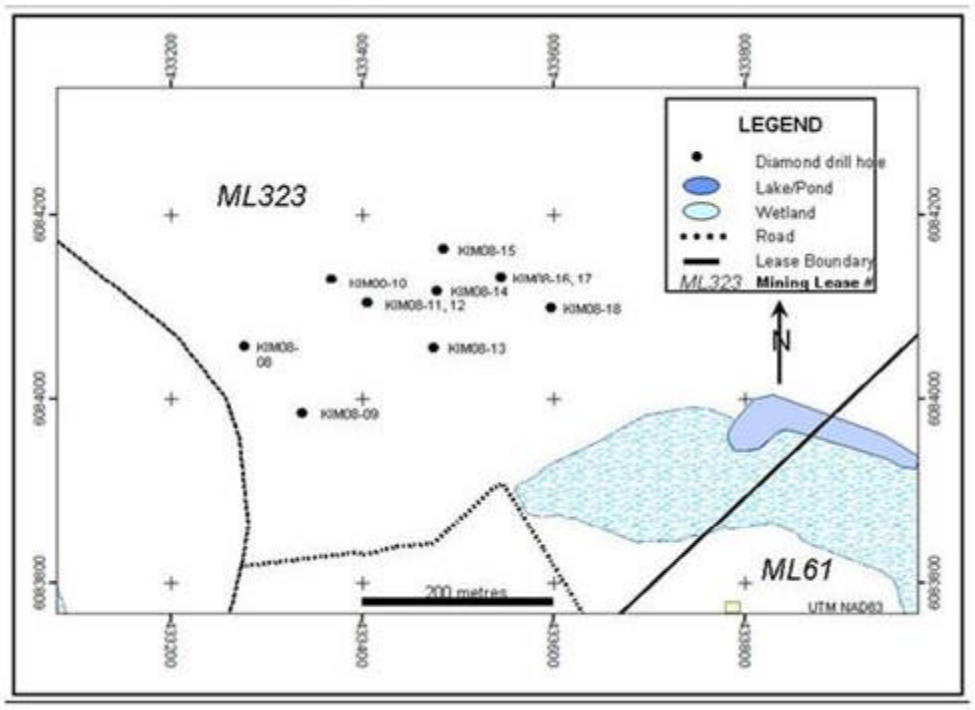

Figure 11-7: Drill Hole Collar Locations for the Kim Zone

A total of 4 drill holes were completed on the Hogg Zone between May and October, 2008. The total footage contained in the 4 drill holes was 4,603 ft (1,403 m). A total of 777 samples were taken during the Hogg Zone drilling program. Further drilling at the Main Mine will be based on the results of an ongoing desk top evaluation of the historical drilling to determine the potential for resource expansion and exploration targets.

The location of the Hogg Zone drill holes and significant assays are contained in Table 11-11 and Table 11-12, respectively. Figure 11-8 illustrates the drill collar locations for the Hogg Zone at the Main Mine.

Table 11-11: Summary of the Hogg Zone Drill Hole Collars and Locations from May to October, 2008

Drill Hole Number	Easting	Northing	Azimuth (_)	Dip (_)	Length (ft)	Length (m)
HB08-39	434551.4	6082816.9	212.3	-56	1099.1	335.0
HB08-40	434551.2	6082816.9	211.3	-79	1197.5	365.0
HB08-41	434557.4	6082749.4	213.8	-71	902.2	275.0
HB08-42	434664.7	6082855.2	209.8	-71	1404.2	428.0
Table provided by Garson Gold Corp.						

Table 11-12: Summary of the Hogg Zone Drill Hole Intersections from May to October, 2008

Drill Hole Number	Mineral Intersections							
	Imperial Measurements (ft)			Gold Assay (Oz/ton)	Metric Measurements (metres)			Gold Assay (g/t)
	From	Core Length			To	Core Length		
HB08-39	982.6	983.9	1.3	0.17	299.5	299.9	0.4	5.93
HB08-39	1039.0	1045.9	6.9	0.04	316.7	318.8	2.1	1.54
HB08-40	914.7	921.4	6.7	0.05	278.8	280.8	2.0	1.62
HB08-40	937.7	939.0	1.3	0.10	285.8	286.2	0.4	3.46
HB08-41	410.4	412.1	1.7	0.15	125.1	125.6	0.5	4.97
HB08-41	570.9	571.9	1.0	0.16	174.0	174.3	0.3	5.42
HB08-41	637.5	639.3	1.8	0.23	194.3	194.9	0.6	7.75
HB08-41	734.9	740.0	5.1	0.08	224.0	225.6	1.6	2.61
Includes	734.9	736.2	1.3	0.20	224.0	224.4	0.4	6.82
HB08-41	869.4	871.8	2.4	0.12	265.0	265.7	0.7	4.18
HB08-42	1121.1	1142.5	21.4	0.15	341.7	348.2	6.5	5.25
HB08-42	1286.7	1288.4	1.7	0.11	392.2	392.7	0.5	3.87
HB08-42	1363.5	1364.7	1.2	0.37	415.6	416.0	0.4	12.65

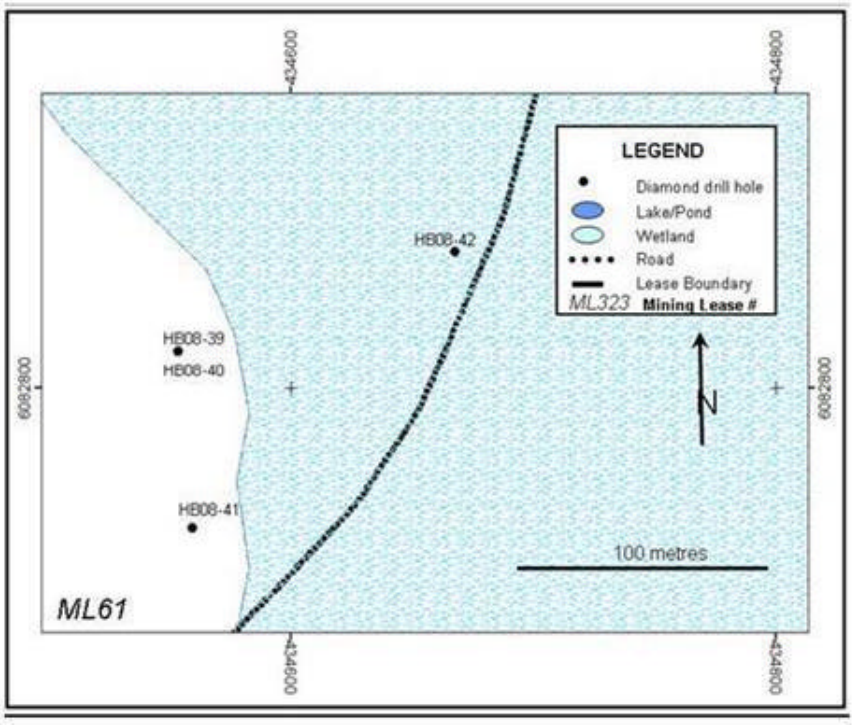

Figure 11-8: Drill Hole Collar Locations for the Hogg Zone

11.4 Diamond drilling results for the main mine and the no.3 zone october 2008 to december 2009

Very limited diamond drilling was completed over the subject time period. Holes completed that have not been included in Micon, May 2008 report include 5 holes drilled and No.3 Zone and 7 holes drilled at Snow Lake Main Mine.

11.4.1 No.3 Zone Drilling and Results

Five holes have been completed at the No.3 Zone since completion of the resource estimate and the Preliminary Assessment reported in Micon, May 2008. The locations and orientations of the holes are shown in Table 11-13 and the significant intercepts are reported in Table 11-14.

The locations and results of the holes are illustrated on the long sections for the No.3 Main Zone and the No.3 Footwall Zone in Table 11-13 and Figures 11-9 and 11-10 respectively. In the No.3 Main Zone, the intercepts are consistent with previous diamond drill results and the previously defined limits of mineralization.

The 5 holes intersected comparatively erratic mineralization in the No.3 Footwall Zone. ZN0308-59 which is located in the mineralized domain did not contain any significant assays in the footwall but did contain the altered structure. By contrast, the adjacent hole intersected 25.4 ft @ .098 opt. Of the holes outside of the mineralized domain, ZN0308-62 intersected 3.4 ft @ .096 opt while the other 2 holes did not intersect any significant mineralization.

Table 11-13: Location data for No.3 Zone drillholes

Hole Number	Easting	Northing	Azimuth (°)	Dip (°)	Length (ft)	Length (metres)
ZN030859	10580.5	12333.3	180	-64	1305.8	398.01
ZN030860	10665.1	12375.0	180	-66	1423.9	434.00
ZN030861	10675.4	12813.6	180	-66	1473.1	449.00
ZN030862	10868.5	12369.2	160	-63	1335.3	407.00
ZN030863	11164.6	12042.3	166	-76	1246.7	379.99
Total:					6,785	2,068

Table 11-14: Significant intercepts for No. 3 Zone drilling

	Mineral Intersections (Imperial Units)				Mineral Intersections (Metric Units)			
Hole Number	From	To	Core Length	In-Situ Grade (Oz/ton)	From	To	Core Length	In-Situ Grade (g/ton)
ZN030859	188.8	193.6	4.8	0.22	57.55	59.01	1.46	7.46
ZN030859	707.7	711.9	4.2	0.15	215.71	216.99	1.28	5.11
ZN030859	911.3	911.9	0.6	0.28	277.76	277.95	0.18	9.60
ZN030859	1029.4	1030.4	1.0	0.46	313.76	314.07	0.30	15.77
ZN030860	296.7	298.2	1.5	0.12	90.43	90.89	0.46	3.98
ZN030860	722.7	724.9	2.2	0.63	220.28	220.95	0.67	21.47
ZN030860	729.7	731.3	1.6	0.14	222.41	222.90	0.49	4.77
ZN030860	742.4	767.8	25.4	0.10	226.28	234.03	7.74	3.36
Includes	745.4	747.5	2.1	0.22	227.20	227.84	0.64	7.65
Includes	755.7	764.0	8.3	0.17	230.34	232.87	2.53	5.90
ZN030860	843.5	846.5	3.0	0.10	257.10	258.01	0.91	3.43
ZN030861	565.5	567.6	2.1	0.20	172.36	173.00	0.64	6.86
ZN030861	603.1	607.5	4.4	0.11	183.82	185.17	1.34	3.83
ZN030861	1055.8	1060.5	4.7	0.11	321.81	323.24	1.43	3.80
ZN030862	776.3	777.3	1.0	0.26	236.62	236.92	0.30	9.05
ZN030862	850.5	853.9	3.4	0.10	259.23	260.27	1.04	3.31
ZN030862	1198.5	1199.5	1.0	0.14	365.30	365.61	0.30	4.70
ZN030863	1080.3	1081.3	1.0	0.06	329.28	329.58	0.30	2.02

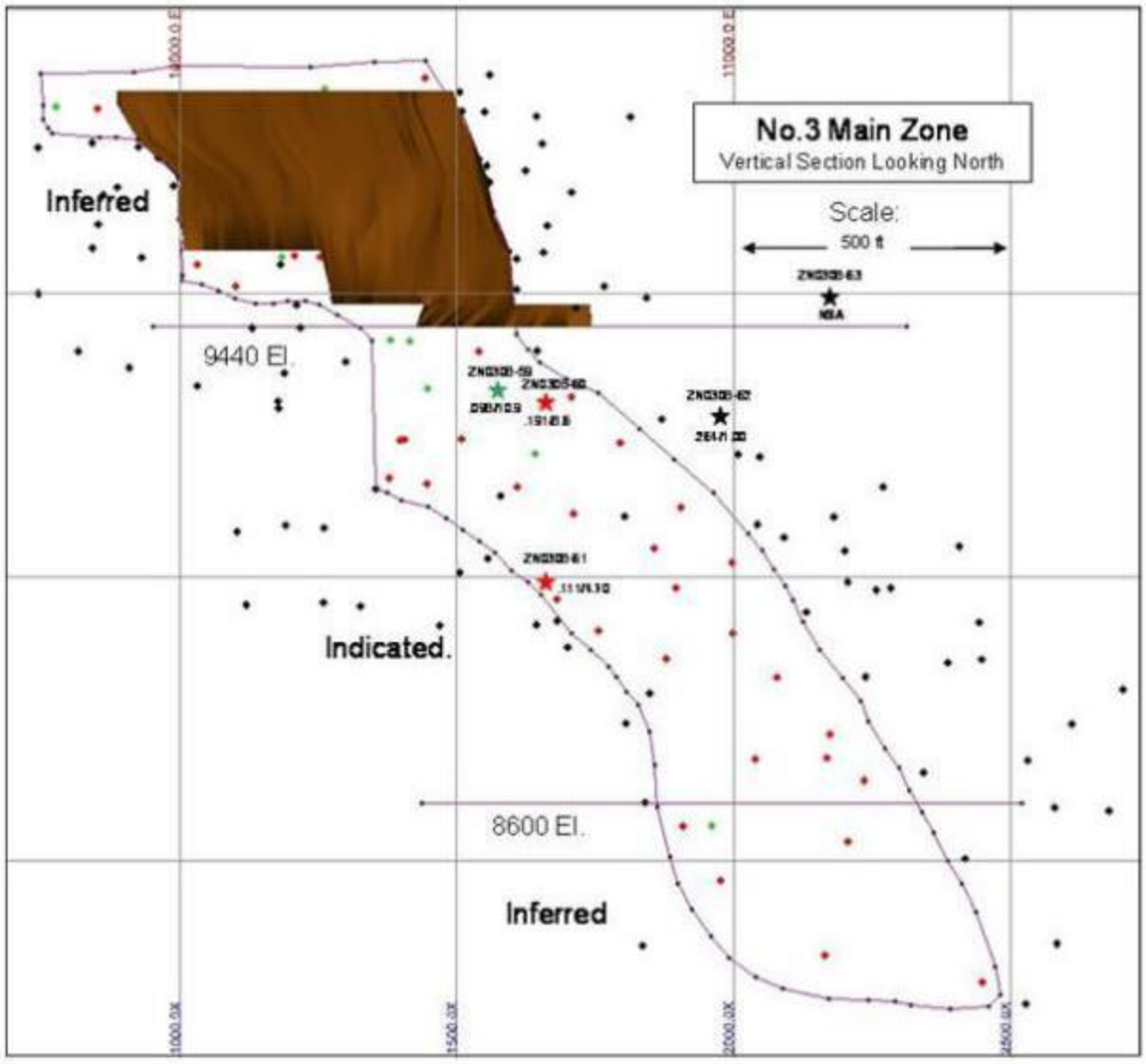

Figure 11-9: Long section with ddh pierce points and the outline of the mineralized domain of the No.3 Main Zone

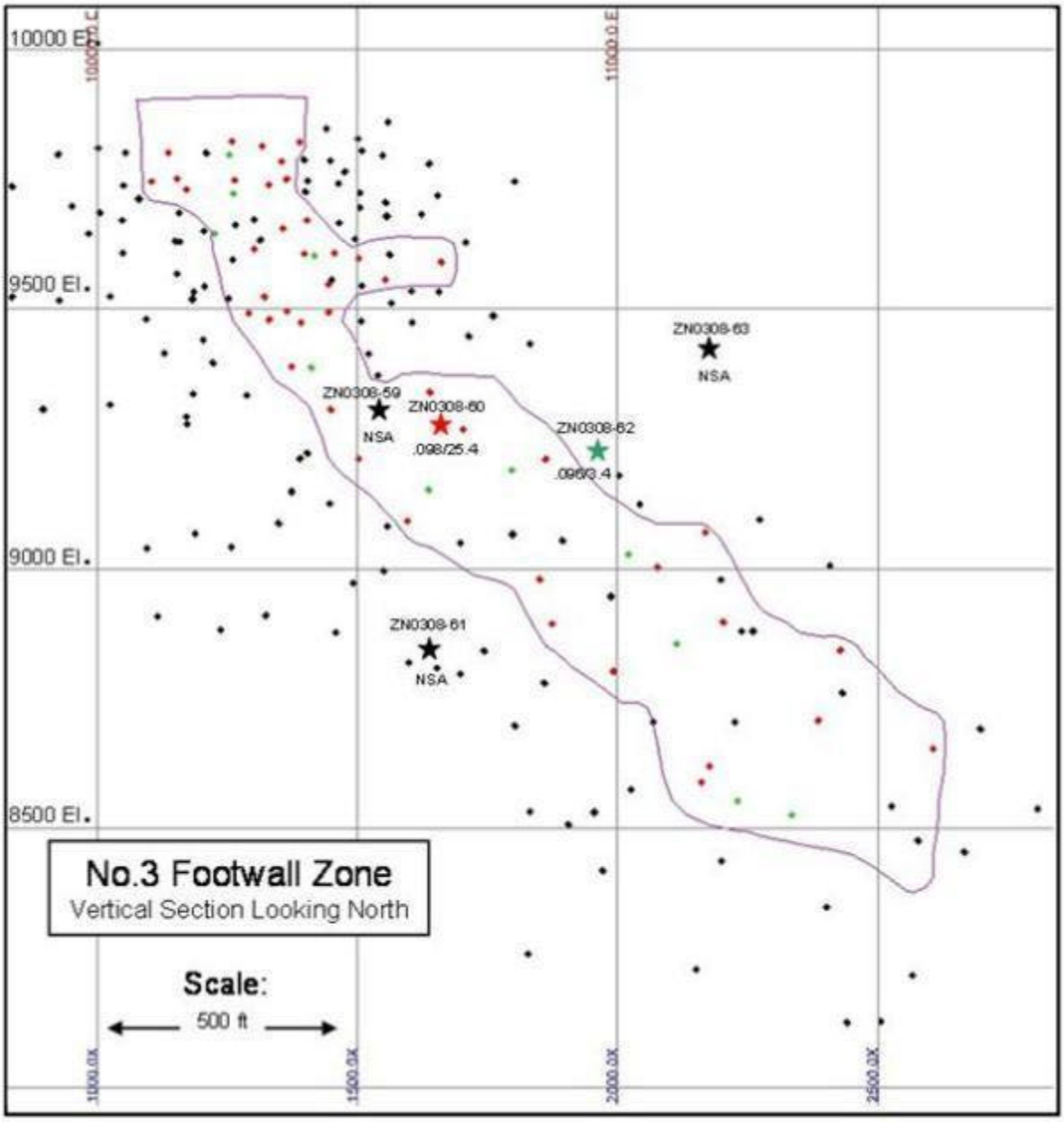

Figure 11-10: Long section with ddh pierce points and the outline of the mineralized domain of the No.3 Footwall Zone

11.4.2 Main Mine Drilling

The holes drilled in 2008, (HB08-39 to HB08-42), have been incorporated into the geological interpretation and resource estimation that is reported in Section 17. The locations and significant assays of these holes are reported in Table 11-15 and Table 11-16 respectively. The results of the 3 holes drilled but not incorporated into the resource model are as follows:

HB09-43: Intersected high grade but narrow mineralization and is coincident with a high grade part of the resource model.

HB09-44: Is located outside of the domain of the resource model and intersected sub-cut off grade mineralization at the zone horizon

HB09-45: Intersected potentially ore grade, .12 opt over 9.9 ft and is coincident with the model of similar width and blocks of grade from .16 to .18 opt.

Table 11-15: Location Data for SLM Drillholes

Drill Hole Number	Easting	Northing	Azimuth	Dip	Length (ft)	Length (m)
HB08-39	434551.4	6082816.9	212.3	-56	1099.1	335.0
HB08-40	434551.2	6082816.9	211.3	-79	1197.5	365.0
HB08-41	434557.4	6082749.4	213.8	-71	902.2	275.0
HB08-42	434664.7	6082855.2	209.8	-71	1404.2	428.0
HB09-43	434573.6	6082750.3	213.2	-70	1022.0	311.5
HB09-44	434713.6	6082698.7	219.2	-70	1049.8	320.0
HB09-45	434369.4	6082813.0	209.2	-75	925.2	282.0

Table 11-16: SLM Significant Drill Intersections

Drill Hole Number	Mineral Intersections (Imperial Units)				Mineral Intersections (Metric Units)			
	From (ft)	To (ft)	Core Length (ft)	In-Situ Grade (Oz/ton)	From (m)	To (m)	Core Length (m)	In-Situ Grade (g/t)
HB08-39	982.6	983.9	1.3	0.17	299.5	299.9	0.4	5.93
HB08-39	1039.0	1045.9	6.9	0.04	316.7	318.8	2.1	1.54
HB08-40	914.7	921.4	6.7	0.05	278.8	280.8	2.0	1.62
HB08-40	937.7	939.0	1.3	0.10	285.8	286.2	0.4	3.46
HB08-41	410.4	412.1	1.7	0.15	125.1	125.6	0.5	4.97
HB08-41	570.9	571.9	1.0	0.16	174.0	174.3	0.3	5.42
HB08-41	637.5	639.3	1.8	0.23	194.3	194.9	0.6	7.75
HB08-41	734.9	740.0	5.1	0.08	224.0	225.6	1.6	2.61
Includes	734.9	736.2	1.3	0.20	224.0	224.4	0.4	6.82
HB08-41	869.4	871.8	2.4	0.12	265.0	265.7	0.7	4.18
HB08-42	1121.1	1142.5	21.4	0.15	341.7	348.2	6.5	5.25
HB08-42	1286.7	1288.4	1.7	0.11	392.2	392.7	0.5	3.87
HB08-42	1363.5	1364.7	1.2	0.37	415.6	416.0	0.4	12.65
HB09-43	765.8	768.5	2.7	0.16	233.4	234.2	0.8	5.49
HB09-43	790.9	792.7	1.8	0.68	241.1	241.6	0.5	23.45
HB09-44	132.9	135.7	2.8	0.08	40.5	41.4	0.9	2.57
HB09-44	743.3	747.6	4.3	0.06	226.6	227.9	1.3	1.94
HB09-45	748.0	757.9	9.9	0.12	228.0	231.0	3.0	4.05

11.5 Diamond drilling results for the boundary zone, birch west, snow lake main mine and no.3 zone areas december 2009 to july 2010

11.5.1 Boundary Zone Drilling and Results

Nineteen holes were completed in the Boundary Zone area to delineate the near surface gold mineralization and test the extent of the mineralization at depth. A total of 4,564 metres of drilling was completed. The collar location data is shown in table 11.17, significant intersections in table 11.18 and a plan of drill hole traces is shown in Figure 11.11. Drill results from shallow holes indicate potential for a low grade open pit resource. Deeper drill results show that typical Boundary style mineralization extends to at least 500 metres.

Table 11-17 Collar Data for Boundary Holes

Hole Number	Easting	Northing	Azimuth (°)	Dip (°)	Length (ft)	Length (m)
BDY09-82	433492.7	6083476.4	197.2	-62	488.9	149.0
BDY09-83	433534.7	6083536.2	234.2	-45	420.8	128.3
BDY09-84	433535.7	6083537.2	237.2	-67	498.9	152.1
BDY09-85	433406.7	6083621.1	233.2	-61	105.1	32.0
BDY09-86	433391.0	6083647.4	232.2	-46	232.9	71.0
BDY09-87	433392.2	6083648.2	232.2	-71	242.8	74.0
BDY09-88	433428.6	6083673.8	231.2	-61	318.2	97.0
BDY09-89	433389.3	6083683.8	232.2	-46	252.4	76.9
BDY09-90	433389.9	6083684.4	232.2	-71	252.6	77.0
BDY10-91	433814.7	6083824.1	225.2	-63	1273.4	388.1
BDY10-92	433809.9	6083824.1	224.2	-83	1538.7	469.0
BDY10-93	433849.6	6083774.4	224.2	-63	1112.5	339.1
BDY10-94	433850.2	6083775.0	223.2	-83	1483.5	452.2
BDY10-95	433908.6	6084026.3	223.2	-78	1707.7	520.5
BDY10-96	433998.7	6083931.0	220.2	-69	1817.9	554.1
BDY10-97	433943.9	6083975.3	231.2	-69	1719.2	524.0
BDY10-98	433772.1	6083824.4	242.2	-67	1266.6	386.1
BDY10-99	433943.5	6083974.8	231.2	-55	98.4	30.0
BDY10-100	433944.4	6083975.6	231.2	-90	147.9	45.1

Table 11-18 Significant Intersections for Boundary Holes

Hole Number	Mineral Intersections (Imperial Units)				Minerals Intersections (Metric Units)			
	From (ft)	To (ft)	Core Length (ft)	In-Situ Grade (Oz/ton)	From (m)	To (m)	Core Length (m)	In-Situ Grade (g/t)
BDY09-82	42.0	48.0	6.0	0.06	12.8	14.6	1.8	2.23
BDY09-83	54.2	62.3	8.1	0.06	16.5	19.0	2.5	2.02
includes	59.5	62.3	2.8	0.15	18.1	19.0	0.9	5.14
BDY09-83	336.5	345.0	8.5	0.07	102.6	105.2	2.6	2.35
BDY09-84	72.7	75.9	3.2	0.16	22.2	23.1	1.0	5.31
BDY09-84	326.4	330.8	4.4	0.11	99.5	100.8	1.3	3.70
includes	326.4	328.7	2.3	0.19	99.5	100.2	0.7	6.38
BDY09-85	No Significant Assays							
BDY09-86	No Significant Assays							
BDY09-87	112.5	113.4	0.9	0.18	34.3	34.6	0.3	6.31
BDY09-88	125.0	128.3	3.3	0.18	38.1	39.1	1.0	6.17
BDY09-89	No Significant Assays							
BDY09-90	No Significant Assays							
BDY10-91	1225.0	1226.0	1.0	0.11	373.4	373.7	0.3	3.74
BDY10-92	1239.1	1243.4	4.3	0.16	377.7	379.0	1.3	5.35
includes	1240.9	1243.4	2.5	0.20	378.2	379.0	0.8	6.86
BDY10-93	No Significant Assays							
BDY10-94	1245.1	1247.2	2.1	0.10	379.5	380.1	0.6	3.46
BDY10-95	1350.0	1359.6	9.6	0.07	411.5	414.4	2.9	2.54
includes	1356.0	1359.6	3.6	0.11	413.3	414.4	1.1	3.81
BDY10-96	1547.4	1547.7	0.3	0.15	471.6	471.7	0.1	5.28
BDY10-97	78.5	82.3	3.8	0.10	23.9	25.1	1.2	3.45
BDY10-97	1312.4	1313.4	1.0	0.12	400.0	400.3	0.3	4.08
BDY10-98	1011.9	1016.1	4.2	0.09	308.4	309.7	1.3	3.02
includes	1015.1	1016.1	1.0	0.25	309.4	309.7	0.3	8.57
BDY10-98	1126.8	1131.9	5.1	0.10	343.4	345.0	1.6	3.43
includes	1126.8	1127.3	0.5	0.44	343.4	343.6	0.2	14.91
BDY10-99	No Significant Assays							
BDY10-100	106.9	109.1	2.2	0.17	32.6	33.3	0.7	5.76

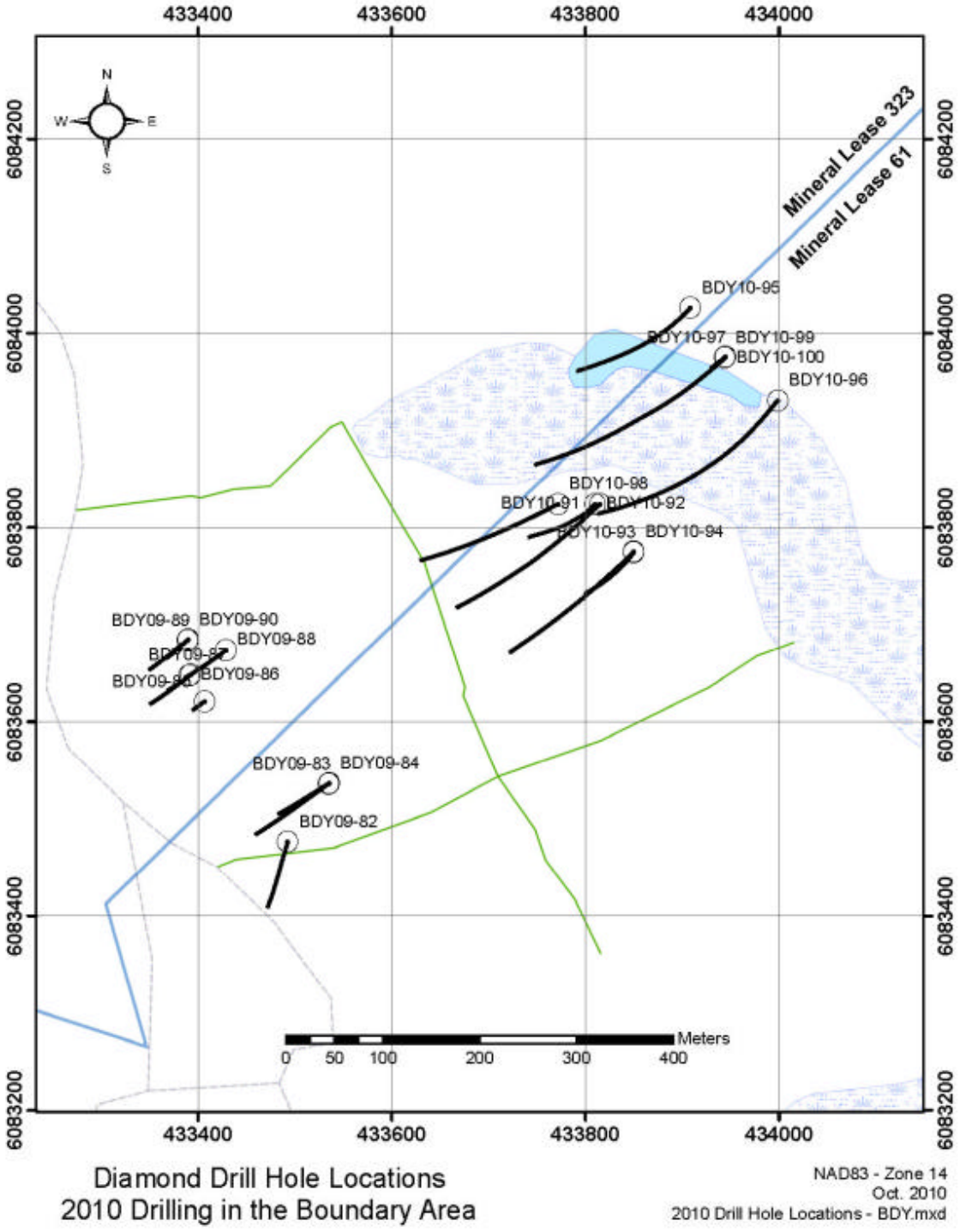

Figure 11-11: Diamond drill hole locations 2010 drilling in the Boundary area

11.5.2 Birch West Drilling and Results

Four holes were completed in the Birch West area to test the theory that the Birch Deposit mineralization would continue to the west and towards the McLeod Road Thrust Fault. A total of 1,046 metres of drilling was completed. The collar location data is shown in Table 11.19, significant intersections in Table 11.20 and a plan map of drill hole traces is shown in Figure 11-12. No significant gold mineralization was intersected although scattered arsenopyrite mineralization was seen near the collar of hole BZW10-04.

Table 11-19 Collar Data for Birch West Holes

Hole Number	Easting	Northing	Azimuth (°)	Dip (°)	Length (ft)	Length (m)
BZW10-01	432120.0	6085625.0	180	-50	1000.7	305.0
BZW10-02	432015.0	6085340.0	180	-50	803.8	245.0
BZW10-03	432242.0	6085171.0	180	-50	744.8	227.0
BZW10-04	432540.0	608509.0	180	-50	885.0	269.7

Table 11-20 Significant Intersection for Birch West Holes

Hole Number	Mineral Intersections (Imperial Units)				Minerals Intersections (Metric Units)			
	From (ft)	To (ft)	Core Length (ft)	In-Situ Grade (Oz/ton)	From (m)	To (m)	Core Length (m)	In-Situ Grade (g/t)
BZW10-01	No Significant Assays							
BZW10-02	No Significant Assays							
BZW10-03	No Significant Assays							
BZW10-04	No Significant Assays							

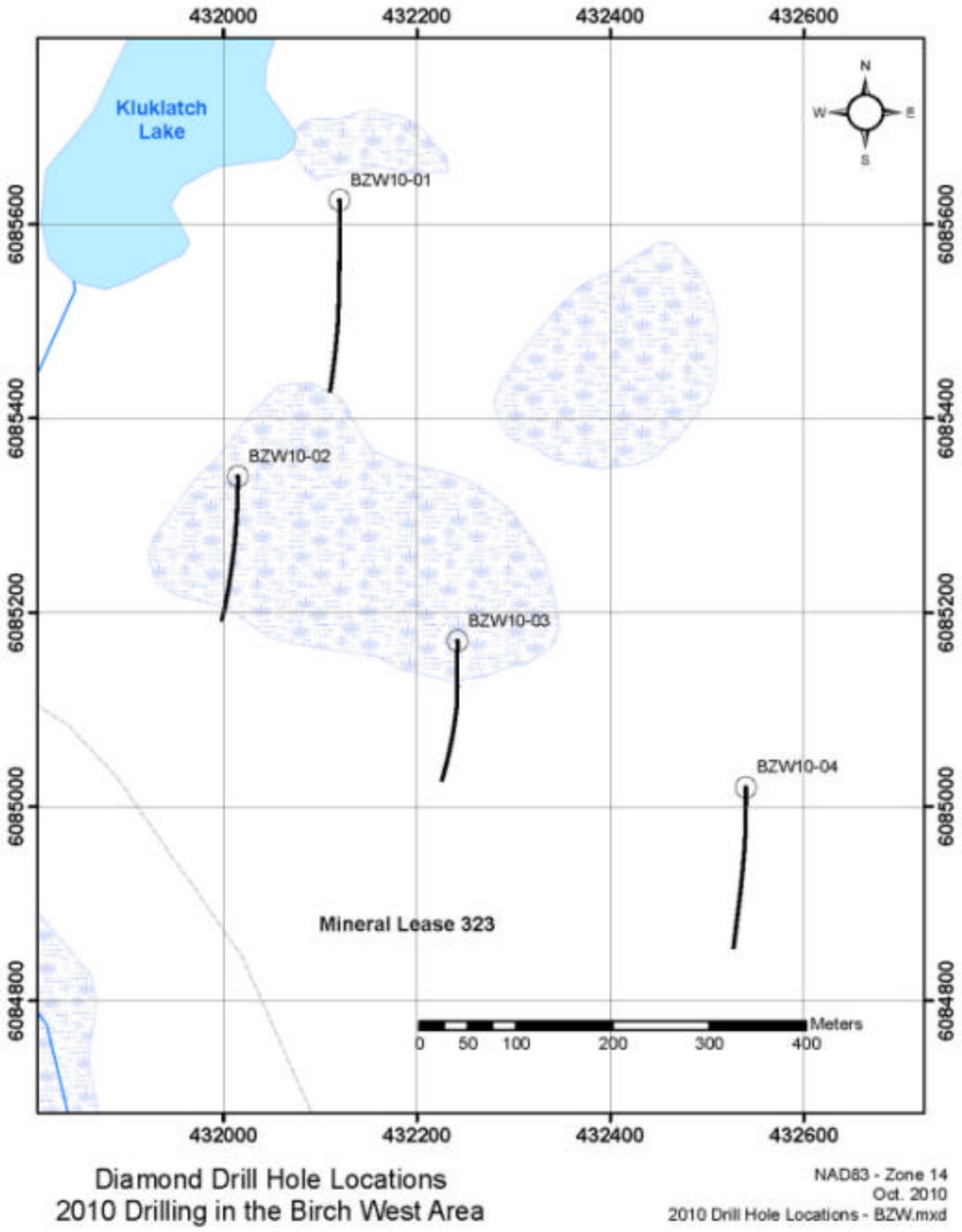

Figure 11-12: Diamond drill hole locations 2010 drilling in the birch west area

11.5.3 Snow Lake Main Mine Drilling and Results

Twenty-four holes were completed in the Snow Lake Main Mine area to test and define gold mineralization east of the Main Mine and up-dip from the Ruttan and Hogg Zones. A total of 8,413 metres of drilling was completed. The collar location data is shown in Table 11-21, significant intersections in Table 11-22 and a plan map of drill hole traces is shown in Figure 11-13. The results outlined several small zones of mineralization that may be mineable using underground methods.

Table 11-21 Collar Data for Snow Lake Main Mine Holes

Hole Number	Easting	Northing	Azimuth (°)	Dip (°)	Length (ft)	Length (m)
HB10-46	434826.4	6082576.3	219.2	-50	705.4	215.0
HB10-47	434826.4	6082576.3	219.2	-70	745.3	227.2
HB10-48	434805.5	6082614.5	219.2	-50	689.6	210.2
HB10-49	434725.7	6082606.7	219.2	-48	689.0	210.0
HB10-50	434726.2	6082607.2	220	-70	695.2	211.9
HB10-51	434848.3	6082538.3	219.2	-50	695.5	212.0
HB10-52	434848.9	6082539.0	219.2	-70	695.5	212.0
HB10-53	434874.6	6082496.4	220	-45	800.4	244.0
HB10-54	434875.2	6082497.2	219.2	-70	839.8	256.0
HB10-55	434667.6	6082858.2	219	-74	1446.9	441.0
HB10-56	434667.6	6082858.2	219	-59	1476.4	450.0
HB10-57	434667.6	6082858.2	208	-58	1285.9	391.9
HB10-58	434667.6	6082858.2	189	-62	1305.8	398.0
HB10-59	434667.6	6082858.2	202	-76	1591.4	485.1
HB10-60	434558.0	6082868.3	211	-61	1305.7	398.0
HB10-61	434558.0	6082868.3	226	-62	1276.4	389.0
HB10-62	434367.3	6082820.4	200	-68	1049.9	320.0
HB10-63	434356.1	6082715.6	221	-75	577.4	176.0
HB10-64	434391.0	6082681.0	208	-73	725.2	221.0
HB10-65	434500.9	6082736.2	217	-65	1020.3	311.0
HB10-66	434474.2	6082681.8	213	-68	951.4	290.0
HB10-67	434571.2	6082747.2	204	-61	1196.0	364.5
HB10-68	435052.8	6082644.9	220	-66	1050.0	320.0
HB10-69	434570.1	6082410.2	250	-50	1733.3	528.3

Table 11-22 Significant Intersections for Snow Lake Main Mine Holes

Hole Number	Mineral Intersections (Imperial Units)				Minerals Intersections (Metric Units)			
	From (ft)	To (ft)	Core Length (ft)	In-Situ Grade (Oz/ton)	From (m)	To (m)	Core Length (m)	In-Situ Grade (g/t)
HB10-46	491.3	516.5	25.2	0.16	149.7	157.4	7.7	5.41
includes	493.8	508.1	14.3	0.25	150.5	154.9	4.4	8.50
HB10-47	No Significant Assays							
HB10-48	529.0	548.0	19.0	0.10	161.2	167.0	5.8	3.35
HB10-49	436.7	443.0	6.3	0.19	133.1	135.0	1.9	6.58
HB10-50	No Significant Assays							
HB10-51	No Significant Assays							
HB10-52	421.6	442.0	20.4	0.07	128.5	134.7	6.2	2.46
includes	421.6	428.6	7.0	0.10	128.5	130.6	2.1	3.57
HB10-53	No Significant Assays							
HB10-54	No Significant Assays							
HB10-55	1286.0	1293.2	7.2	0.73	392.0	394.2	2.2	25.05
includes	1286.0	1291.0	5.0	1.01	392.0	393.5	1.5	34.79
HB10-56	1058.0	1085.0	27.0	0.14	322.5	330.7	8.2	4.75
includes	1074.4	1085.0	10.6	0.22	327.5	330.7	3.2	7.63
HB10-56	1306.0	1308.3	2.3	0.26	398.1	398.8	0.7	8.74
HB10-57	No Significant Assays							
HB10-58	449.4	484.2	34.8	0.03	137.0	147.6	10.6	0.97
HB10-58	1069.4	1079.1	9.7	0.13	326.0	328.9	3.0	4.62
HB10-59	1163.3	1181.0	17.7	0.06	354.6	360.0	5.4	1.98
HB10-59	1278.9	1305.7	26.8	0.35	389.8	398.0	8.2	11.92
includes	1284.4	1296.4	12.0	0.69	391.5	395.1	3.7	23.67
includes	1285.9	1291.3	5.4	1.32	391.9	393.6	1.6	45.32
HB10-59	1341.9	1344.8	2.9	1.20	409.0	409.9	0.9	41.04
HB10-60	940.0	953.4	13.4	0.09	286.5	290.6	4.1	3.19
HB10-60	965.6	968.2	2.6	0.27	294.3	295.1	0.8	9.39
HB10-61	967.9	976.4	8.5	0.08	295.0	297.6	2.6	2.65
HB10-62	708.5	709.9	1.4	0.11	216.0	216.4	0.4	3.91

HB10-63	No Significant Assays							
HB10-64	605.9	609.7	3.8	0.22	184.7	185.8	1.2	7.69
HB10-64	707.1	712.9	5.8	0.12	215.5	217.3	1.8	4.26
HB10-65	640.6	641.3	0.7	0.13	195.3	195.5	0.2	4.49
HB10-66	No Significant Assays							
HB10-67	224.0	225.1	1.1	0.26	68.3	68.6	0.3	9.05
HB10-68	No Significant Assays							
HB10-69	124.3	125.9	1.6	0.67	37.9	38.4	0.5	22.94

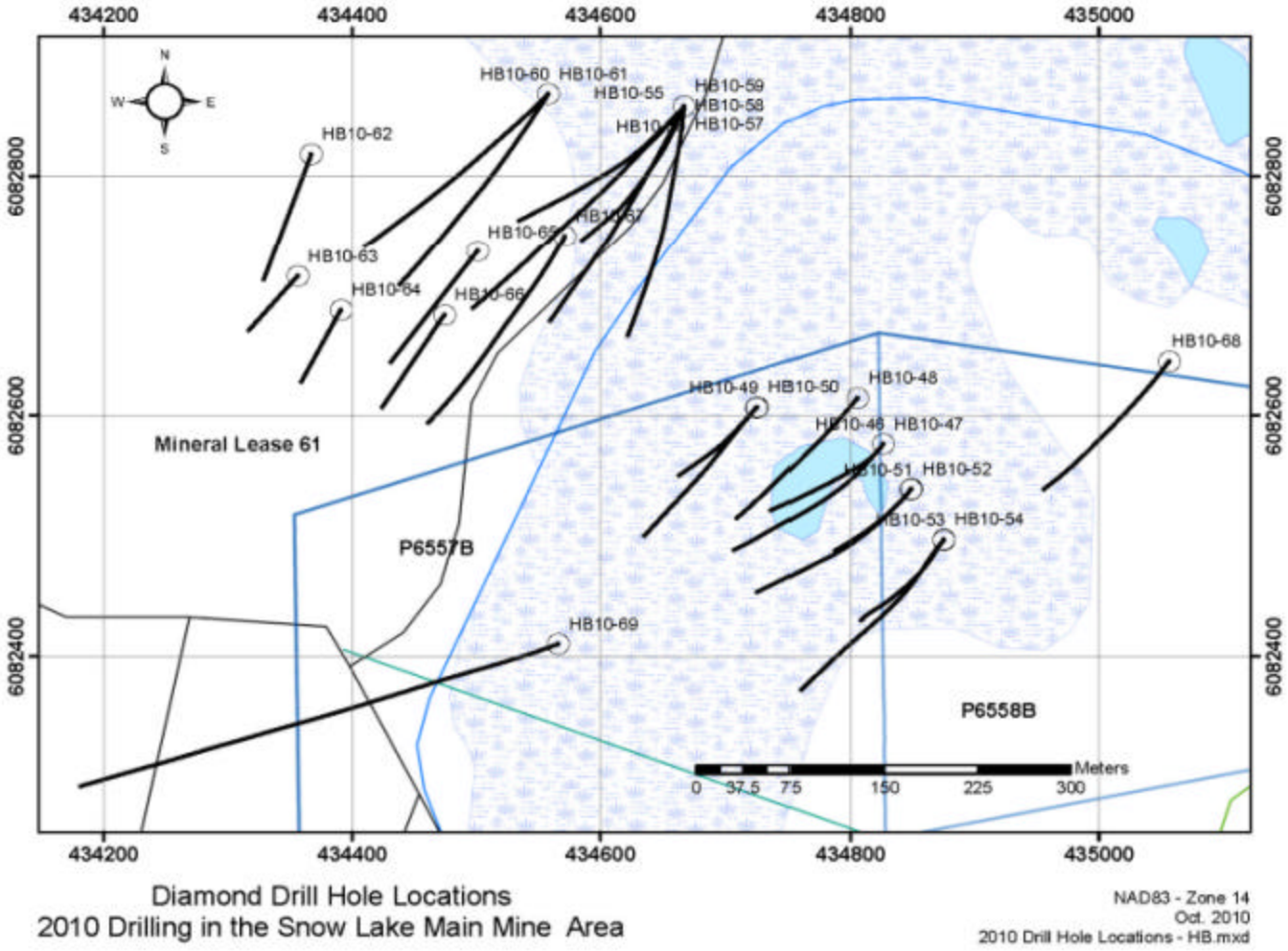

Figure 11-13: Diamond drill hole locations 2010 drilling in the Snow Lake Main Mine Area

11.5.4 No. 3 Zone Drilling and Results

Twelve holes were completed in the No. 3 Zone to test the lower extent of the No. 3 Zone Main Lens. A total of 5,759 metres of drilling was completed. The collar location data is shown in Table 11.23, significant intersections in Table 11.24 and a plan map of drill hole traces is shown in Figure 11.14. The drilling was successful providing data for a resource in the lower portion of the No. 3 Zone Main Lens. Figure 11.15 is a vertical long section showing the locations of the intersections with the Main No. 3 Zone lens.

Table 11-23 Collar Data for No. 3 Zone Holes

Hole Number	Easting	Northing	Azimuth (°)	Dip (°)	Length (ft)	Length (m)
ZN031064	433727.3	6084794.9	160	-70	2053.8	626.0
ZN031065	433727.6	6084794.1	171	-54	2024.3	617.0
ZN031066	433745.7	6084736.7	179	-63	1906.2	581.0
ZN031067	433745.3	6084737.0	172	-63	2014.4	614.0
ZN031068	433773.6	6084660.5	176	-63	1759.6	536.3
ZN031069	433773.7	6084661.3	163	-76	853.0	260.0
ZN031070	433844.8	6084744.0	180	-62	2064.6	629.3
ZN031071	433844.2	6084744.3	178	-58	2162.1	659.0
ZN031072	433844.4	6084744.6	178	-68	656.2	200.0
ZN031073	433848.6	6084792.5	170	-62	85.3	26.0
ZN031074	433848.9	6084792.2	170	-62	2801.9	854.0
ZN031075	434044.2	6084751.4	227	-55	597.1	182.0

Table 11-24 Significant Intersections for No. 3 Zone Holes

Hole Number	Mineral Intersections (Imperial Units)				Minerals Intersections (Metric Units)			
	From (ft)	To (ft)	Core Length (ft)	In-Situ Grade (Oz/ton)	From (m)	To (m)	Core Length (m)	In-Situ Grade (g/t)
ZN03-10-64	1230.0	1232.0	2.0	0.12	374.9	375.5	0.6	3.94
ZN03-10-64	1871.8	1889.3	17.5	0.07	570.5	575.9	5.3	2.24
includes	1888.3	1889.3	1.0	0.66	575.6	575.9	0.3	22.63
ZN03-10-65	949.5	955.2	5.7	0.15	289.4	291.1	1.7	5.28
ZN03-10-65	1794.8	1813.6	18.8	0.14	547.1	552.8	5.7	4.76
includes	1800.3	1813.3	13.0	0.18	548.7	552.7	4.0	6.04
includes	1800.3	1808.2	7.9	0.21	548.7	551.1	2.4	7.31
ZN03-10-66	1225.2	1234.3	9.1	0.14	373.4	376.2	2.8	4.89
includes	1226.3	1232.7	6.4	0.18	373.8	375.7	2.0	6.33
ZN03-10-66	1731.2	1746.0	14.8	0.21	527.7	532.2	4.5	7.35
ZN03-10-67	1006.1	1008.0	1.9	0.21	306.7	307.2	0.6	7.26
ZN03-10-67	1305.0	1306.5	1.5	0.23	397.8	398.2	0.5	7.75
ZN03-10-67	1775.9	1782.9	7.0	0.07	541.3	543.4	2.1	2.27
ZN03-10-68	845.4	857.0	11.6	0.12	257.7	261.2	3.5	4.27
includes	845.4	852.0	6.6	0.19	257.7	259.7	2.0	6.53
ZN03-10-68	907.8	921.9	14.1	0.08	276.7	281.0	4.3	2.65
ZN03-10-68	1608.1	1613.3	5.2	0.10	490.1	491.7	1.6	3.38
ZN03-10-69	No Significant Assays							
ZN03-10-70	503.5	508.1	4.6	0.14	153.5	154.9	1.4	4.97
ZN03-10-70	1777.5	1783.8	6.3	0.07	541.8	543.7	1.9	2.43
ZN03-10-71	114.5	115.1	0.6	19.33	34.9	35.1	0.2	662.85
ZN03-10-71	1763.0	1771.6	8.6	0.30	537.4	540.0	2.6	10.12
ZN03-10-72	No Significant Assays							
ZN03-10-73	No Significant Assays							
ZN03-10-74	1455.8	1459.1	3.3	0.20	443.7	444.7	1.0	6.78
ZN03-10-74	1830.4	1832.9	2.5	0.17	557.9	558.7	0.8	5.93
ZN03-10-74	1941.6	1943.9	2.3	0.45	591.8	592.5	0.7	15.53
ZN03-10-74	2070.2	2076.7	6.5	0.21	631.0	633.0	2.0	7.14
ZN03-10-75	531.4	532.3	0.9	0.08	162.0	162.2	0.3	2.78

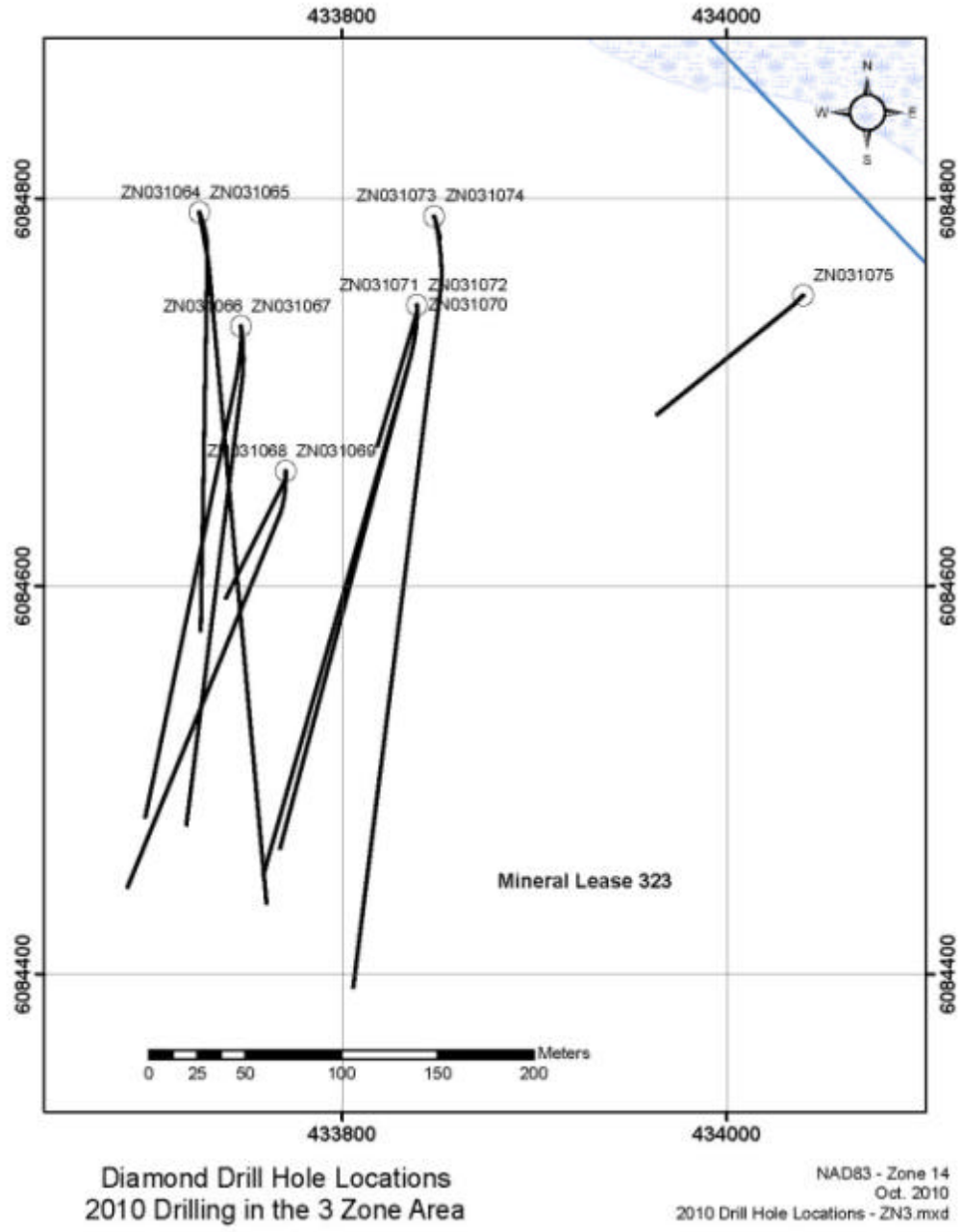

Figure 11-14: 2010 Diamond drill hole locations in No 3 Zone Area

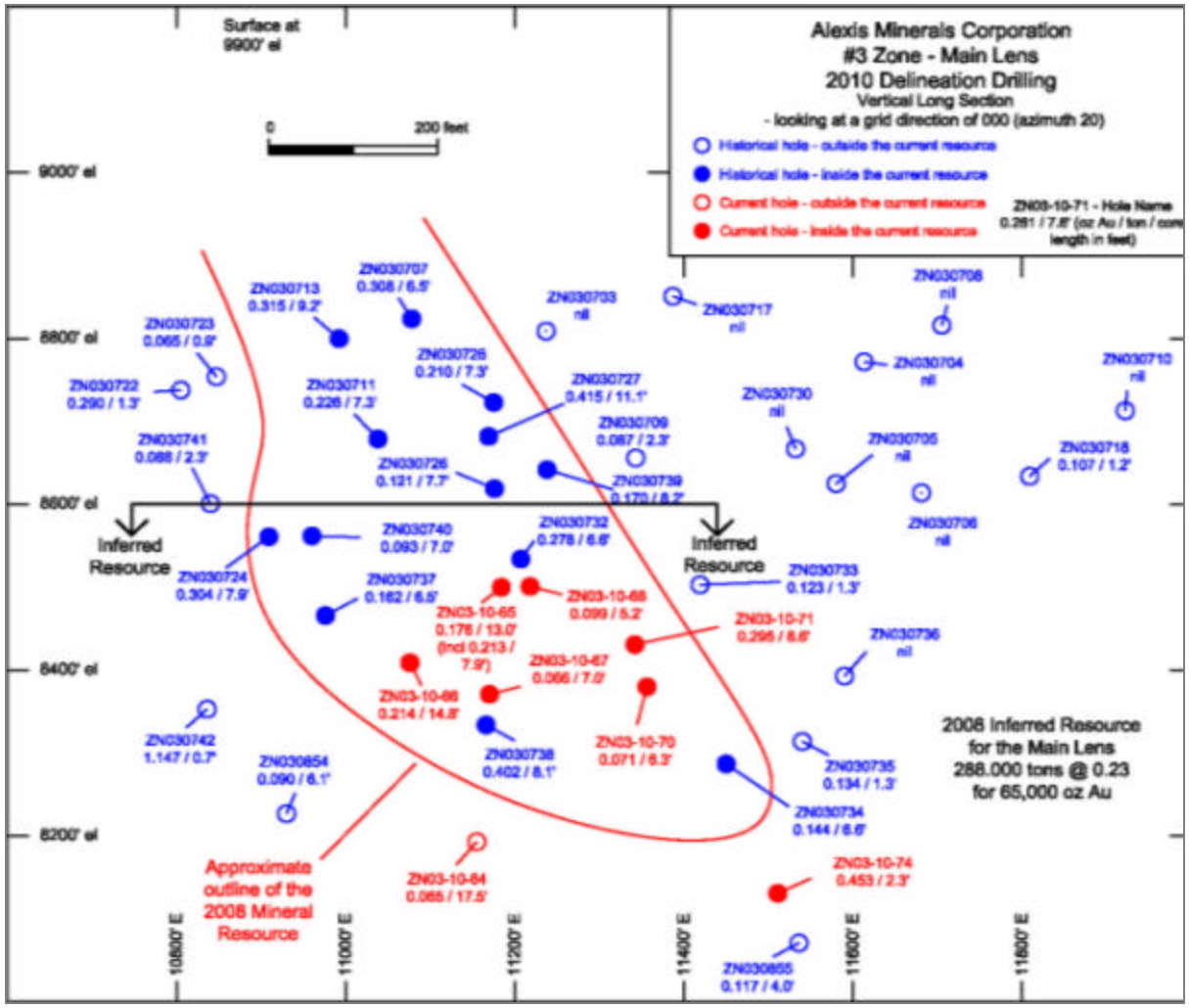

Figure 11-15: # 3 Zone – Main lens 2010 delineation drilling – vertical long section

12. SAMPLING METHOD AND APPROACH (JL)

The details of the sampling methods used by NBM during the latest production phase (1995 to 2005) have been recorded in various reports within the mine's archive. However, details of the sampling methods conducted by earlier operators prior to the 1995 to 2005 production phase have been lost. The sampling method and approach used during the latest operational phase at the NBM described below is directly from . the technical report by Micon International Limited, dated October 27, 2006.

The details of the sampling methods and approach used by in exploration drilling by Garson on the most recent phase of drilling are described later in this chapter and is from the technical report by Micon International Limited, dated May 30, 2009. The method and approach of sampling used by Garson has been retained by Alexis and is consistent with industry best practices and is appropriate for the deposit type being evaluated.

12.1 New Britannia Mine Sampling Method and Approach

The following summary of the historical sampling method and approach was taken from various reports on the exploration definition and drilling programs as well as underground sampling of the development faces. The reports were prepared by the NBM geology staff for Kinross and High River.

12.1.1 Exploration Drifting, Drilling and Definition Drilling

Between 1995 and 2003, NBM excavated a number of exploration drill drifts on the 1780 and 3000 Levels. Drill stations were established every 50 ft to 100 ft (15.24 m to 30.48 m) along the exploration drifts.

Two drill drifts on the 3000 Level were located 1,000 ft and 1,500 ft (304.79 m and 457.19 m) in the hangingwall of the deposit. Drill stations were established every 100 ft (30.48 m) in these drifts.

Exploration drifts in the hangingwall of the 1810, 2300 and 3630 Levels were driven using mechanical two boom jumbos. The spacing of the drill stations varied between 50 ft and 300 ft apart depending on the quantity of drilling that was planned for each level.

The initial exploration drilling was conducted along the interpreted trend of the mineralization on 200 ft (60.5 m) centres along strike and 300 ft (91.5 m) centres down dip. Once the initial drilling results indicated the continuity of the mineralization, then definition drilling was performed on 100 ft (30.5 m) centres along the strike and dip of the deposit. Final definition drilling of the deposit was done on 50 ft (15.2 m) centres prior to the initiation of the mining sequence.

Once the individual mining sills (levels) had been excavated further definition drilling was conducted on 25 ft (7.6 m) centres to define any footwall lenses and to establish the location of the hangingwall prior to initiating the longhole drill layouts.

During the life of the NBM, Major-Midwest Drilling, a Manitoba subsidiary of Major Drilling, undertook the exploration and definition drilling at first with AW conventional drilling equipment and later with BQ and NQ wireline equipment. In addition to the larger drills, Major-Midwest had a small drill on site to perform the definition drilling using AW equipment. However, a NBM Bazooka drill using imperial EW equipment was the primary drill for the definition drilling in the extraction sills.

Core recovery was generally excellent with between 96% and 100% for all the holes drilled on the NBM property.

Tropari down-hole surveying of the diamond drill holes was done by the drillers three or four times per hole for the purpose of maintaining accurate drill plots on the mine sections. The tests were reviewed by geological staff at the mine and were retaken if the readings appeared to be incorrect. The drillers were trained in the Tropari surveyng by the geology staff at the mine and this was periodically reviewed to ensure the testing procedures were adhered to. Down-hole surveying was done on all drill holes over 100 ft (30.5 m) in length.

The core was collected by the diamond drillers and placed in wooden core boxes containing the equivalent to 20 ft (6.1 m) runs. Once the core box was full a second core box was placed on top of the first box to act as a lid for the first box and the outside of the boxes were marked with the hole number, core box number and the to and from footage of core contained in the box prior to being wired shut. At the end of each shift the boxes were brought out to the shaft station for transportation to surface. Once the core boxes reached surface a forklift either brought them directly into the core shack or placed them in the laydown area in front of the core shack to await their turn to be placed in the core shack.

All diamond drill core was logged at the mine site, generally by one or two geologists and a technician. The technician would help to arrange the core, split and sample the core according to the geologists' instructions and ensure the general cleanliness of the core shack. The core was logged onto a report sheet along with all the sampling information. The written log was then entered into the mine's database using a logging program written for the geology department by Gemcom Software International Inc.

12.1.2 Chip Sampling of Drift Faces and Bazooka Drilling Prior to Longhole Interpretation

Development of the extraction sills (levels) followed the footwall of the each mineralized zone within the deposit to facilitate efficient longhole drilling along the footwall.

The mine geologist will map, take chip samples, and "mark" each development face before the next round is drilled. The chip sampling and marking the face allow the geology department to closely follow the footwall of the mineralization and to minimize the dilution.

Most of the development faces are chip sampled unless adverse ground conditions prohibit access to the face. If sampling of the face is missed no further development rounds may be blasted until the geologist can sample the next face. Chip sampling is generally limited to the face and a 4-ft sample along each wall. The sample length is dictated by the width of the rock types encountered as no sample can cross the boundary between two lithological units.

Chip samples are collected by the geologist and either taken from the back of the drift if the muck pile had not been removed or taken 4.5 ft to 5 ft above the floor of the drift if the blast muck has been removed. In either case the development face is washed and mapped prior to sampling. All mapping and sampling information is written onto pro-forma sheets where the true dimensions of the face are recorded to scale.

The geologist is responsible for bringing the samples to the preparation area of the mill laboratory and recording the number of each sample into a record book.

In addition to chip sampling the development faces, definition diamond drilling is performed on 25 ft to 50 ft (7.6 m to 15.2 m) centres along the strike length of the level. The interval and the length of the holes is determined by the geologist responsible for the interpretation of the zone. This drilling is conducted to identify the location of the hangingwall prior to issuing a longhole interpretation to the engineering department. In some cases, a separate footwall or hangingwall lens may be present and the definition drilling will determine if this is economically mineable or not.

The results of the diamond drilling and the face chip samples form the assay template for the level. The assay templates assist in the creation of the geological "ore solid" between the levels. Once the geological "ore solid" is created it is given to the engineering department to plan the longhole-drilling layout.

The engineering department surveys the collars of all the diamond drill holes. Only drill holes that are longer than 100 ft (30.5 m) in length are Tropari surveyed.

After an extraction level has been completed, the back and walls are washed and the geological mapping is finalized. The geological mapping helps to identify the relationships between the major structural features, rock types and the mineralization. The geological mapping is also used to identify areas where extra ground support is required.

12.1.3 Longhole Sludge Sampling and Haul Truck Sampling

A number of other sampling methods and procedures were initiated at NBM between 1995 and 2004. The two most important procedures were the longhole sludge sampling and sampling the ore between the stope and the ore pass or crusher.

The longhole sludge sampling was initiated to determine the approximate grade of the stope being drilled and to minimize the internal dilution. This method was usually applied to up hole drilling. The method can involve significant contamination if the procedures are not followed. The sample from the drill hole is collected in a 45 gallon drum for the equivalent length of two drill rods or 8 ft (2.4 m). At the end of each drill run the drum is replaced and the cuttings are allowed to settle.

Once the cuttings have settled the excess water is poured from the drum and the cuttings are placed in a sample bag and tagged. The sample tag has two parts, the smaller portion containing the sample number and the larger portion showing the details of the drill hole. The sample tags are delivered to the Shift Boss who in turn submits them to the geology department.The sample drums are thoroughly washed before re-using to ensure no contamination. This sampling is useful to outline areas of weak mineralization between the levels which could be considered for pillars.These samples are not used for resource estimation or for reconciliation due to the potential for contamination.

Truck or scoop bucket samples are taken when the ore is moved to the ore passes and the crusher. These samples provide information on the grade of the material sent from each stope to the mill. The sampling frequency is approximately one sample for every 42 to 52 tons or one sample for every second truck of ore sent to the mill. If a scoop is mucking directly to an ore pass or to the crusher, then a sample is taken every 6 buckets. The truck and scoop samples are used to compile the daily production tonnage and grade from each development heading in ore or longhole stope. This figure is used to reconcile the underground production to the mill production on a monthly basis. Each sample is collected in a plastic bag along with a sample tag. The sample tag is a two part tag, with the smaller portion containing a sample number. The larger portion of the sample tag describes the details of the sample location. The sample tags are delivered to the Shift Boss at the end of the shift who in turn submitsthem to the geology department.

All samples arebrought to the surface at the end of each shift and brought to the assay laboratory preparation room by either a geologist or the core shack technician.

Micon (October, 2006) reviewed the reports containing information on the sampling methods and approaches and concluded that NBM conformed to the industry standards and theses were adequate to ensure a representative determination for the type of gold mineralization identified on the NBM property.

12.2 Garson sampling method and approach

All of the sampling completed by Garson athat contributed to the updated No. 3 Zone resource estimate, and the results from the Boundary, Kim and Birch Zones, has been from diamond drill core. The following section summarizes the sampling method and approach used by Garson:

Samples are selected by the logging geologist using the following criteria:

- As a guide, samples are to be no more than 3 ft in length and no less than 6 inches in length.
- Sample selection and distinction is based on lithology, mineralogy, and texture.
- Samples do not cross clear lithological boundaries. At the No. 3 Zone, most of the ore is hosted by mafic volcanoclastic rocks and therefore samples are of similar protolith composition.
- Samples are comprised of mineralogically similar core. A primary control or indicator of mineralization is the presence of arsenopyrite. The distribution of fine grained acicular arsenopyrite appears to be important and is distinguished from coarse grained and more equant arsenopyrite. Biotite alteration is also important and forms a sample limit criterion.
- Samples are comprised of texturally similar core. This includes the distinction of veins and alteration textures at No. 3 Zone and the occurrence of fault and sheared rocks characterized by intense foliation, mylonite and breccia.

Samples encompass all mineralized and potentially mineralized zones and all mineralized zones have “zero grade” “shoulders”. All logs and assays are inspected for proper and complete sampling that includes proper sample shoulders on the zones.

Core recovery and ground conditions at No. 3 Zone and in the other zones are very good and there are no drilling factors that are known that could materially impact the reliability of the sampling.

Drilling at No. 3 Zone has been designed in an attempt to have intersection lengths as close as possible to true width. In most cases, true width is > 90% of the intersection length although in some holes the zones have been intersected at more oblique angles. Sample and composite lengths and statistics are described and discussed more fully in Section 17.

13. SAMPLE PREPARATION, ANALYSES AND SECURITY (JL)

The description of the sample preparation, analysis and security at NBM is taken directly from the technical report by Micon International Limited, dated May 30, 2009. The updated resource estimates for the No.3 Zone and the Snow Lake Mine in this study are based on sampling preparation, analyses, and security proptocols that are described below. Alexis continues to use the same accredited commercial assay laboratory and has assumed the same sample preprartion, analyses, and security protocols as established by Garson. These are all consistent with the industry best practices.

13.1 New Britannia Mine Sampling Preparation, Analysis and Security

The following summary regarding the historical sample preparation, analysis and security procedures at NBM is taken from various internal reports prepared by the geological staff and the assay laboratory personnel.

Underground sampling was conducted by the geological and underground departments at the NBM property. For diamond drilling and chip sampling, the samples were collected and prepared for shipment to the on-site assay laboratory by the geology staff. In the case of underground samples from sludge sampling, longhole drilling and muck sampling from the trucks and scoops, these were sampled by the operators of the equipment using procedures issued by the geological department.

Sampling of the drill core was generally limited to the zone of mineralization and a 5 ft to 10 ft (1.5 m to 3 m) waste section on either side. The maximum sample length was no more than 5 ft with no minimum length established. Upon the identification of coarse gold within the sample, the samples were assayed for metallics in addition to undergoing the regular fire assay. Any core samples assaying higher than 0.500 Oz/ton gold were routinely assayed for metallics. However, during many of the exploration programs 0.250 Oz/ton gold was used as the cut off for the routine assay of metallics. Appendix L of the December 31, 2002, mineral inventory document for NBM contains a copy of the assay laboratory quality assurance/quality control (QA/QC) protocols and analytical procedures.

Intervals of whole core were assayed and any intervals not sampled during the logging process were generally discarded. However, occasionally, either split core representing the ore zone and 10 ft to 20 ft (3.05 m to 6.10 m) on either side or an entire drill hole was retained for reference purposes. The drill core retained for reference purposes is stockpiled behind the mill in a designated area.

All core samples were sent to the assay laboratory located at the New Britannia mill for gold analysis. The assay laboratory obtained certificates of laboratory proficiency in gold analysis for the years of 2001, 2002 and 2003 from the Canadian Certified Preference Materials Project, which is a branch of CanMet. No certificate was obtained for 2004 due to the impending closure of the mine and it was originally believed the assay laboratory would be closed during the second testing period in

November. Copies of the assay proficiency certificates are located in Appendix K of the mineral inventory document, dated December 31, 2004.

A minimum of ten crushed core reject samples per month were chosen at random for independent analysis, with the core rejects sent out every three months to the independent laboratory. If needed, additional core samples could be sent out for testing. For the exploration programs, the exploration department would request that 10% to 15% of the core rejects were sent out for sampling and in some cases the core would be sent to an outside laboratory to be assayed. If the core was sent to an outside laboratory, the rejects and pulps would be returned to NBM for storage.

All of the crushed rejects and pulps were retained on site for one to two years prior to being discarded. The crushed rejects and pulps were placed in 5-gallon white plastic pails with the date and sample numbers marked on the outside of the pail. Once the pails were full they were placed on a pallet and stored in a lay-down area behind the mill.

Most of the development faces on each level were chip-sampled unless adverse ground conditions prohibited access to the fac. If the face sampling was missed, no further development rounds could be blasted until the geologist sampled the next face. Chip sampling was generally limited to the face and a 4-ft sample along each wall, with a maximum sample length of no more than 5 ft and no minimum length. Sample length was dictated by the width of the rock types encountered as no sample should cross the boundary between two different lithological units.

Chip samples were conducted by the geologist and were either taken from the back of the drift if the muck pile had not been removed, or the sample was taken 4.5 ft to 5 ft above the floor of the drift, if the blast muck had been removed. In both cases the development face was washed and mapped prior to sampling. All mapping and sampling information was written onto pro-forma sheets prepared for sampling where the true dimensions of the face were recorded to scale.

The geologist secured each sample and, upon returning to surface, was responsible for bringing the samples to the preparation area of the mill laboratory and recording the number of each type of sample into a record book.

Chip samples assaying higher than 0.500 Oz/ton gold were routinely assayed for metallics and, in areas of the deposit that were suspected to be higher grade, the automatic assaying was reduced to 0.250 Oz/ton gold at the request of the geological department.

Acme Analytical Laboratories Ltd. (Acme Analytical) in Vancouver B.C. conducted most of the independent assaying for the mine because it was among the first laboratories to become an ISO 9002 accredited company. However, during the 3630 Level exploration program in 2004, TSL Laboratories (TSL) in Saskatoon, Saskatchewan, was used once it had become an accredited company and the turnaround time was shorter.

The analytical procedures are outlined in an internal report put together and continually updated by Mr. Richard Jones, the Chief Assayer at NBM, and sections of the report are contained in the following paragraphs.

The following preparation methods were described in the NBM analytical procedures manual for the assay laboratory:

- All samples were dried either by placing the pans on a hot plate or in one of the three ovens located at the assay laboratory.
- Crushing would reduce the sample to ~2 millimetres (mm) in size and the sample was placed in the crusher in small batches to ensure proper crushing and prevent rock from being jammed or hung up inside the crusher.
- The whole core sample was crushed first by using the jaw crusher and then by using the rhino crusher while the entire chip or truck sample was crushed using the rhino crusher.
- The crushed core, chip or truck sample was dumped into the middle of a 24-slot riffle, one half is removed and split again. The half from the original split is then split. The pan containing material from both the original splits is then reduced to a 250 g +/- 10% assay sample.
- Once splitting was completed, the sample was placed into a 250 ml grinding bowl and a small amount of methyl hydrate added to remove moisture. The sample was pulverized for a predetermined time (sludges: 30 seconds, drill core, chips, trucks, and mill samples: 60 seconds). The pulverizer, mat, and bench top are cleaned with compressed air between samples and additional cleaning with silica sand was required between high grade and low grade samples.
- Drill core and mill samples were also matted after being pulverized. This consists of rolling the pulp to each of the four corners a minimum of 10 to 20 times (one roll being one rotation to the four corners).
- Drill core and chips had the sample ID tag stapled to the bag while sludge and truck samples were placed in pulp cans with ID tags attached to clips.
- For the drill core assays, an aliquot of 30 g was taken—chip samples: 20 g; truck and sludge samples: 5g. In the case of sampling for metallics, assaying was conducted on the whole sample for the plus fraction and a 30 g cut of the minus fraction.

Drill core and chip samples were assayed using the fire assay/atomic absorption finish (FA/AA) method of analysis and the truck and sludge samples were assayed using the Methyl Isobutyl Ketone/atomic absorption finish (MIBK/AA) method.

Standard, blank and duplicate assay samples were added to each run of 21 samples for drill core, to a maximum of 24 samples for chip samples and to each run of 27 samples for both sludge and truck samples. These were in addition to the blind duplicates and blanks added randomly to the assaying process by the geology department.

All samples were assayed for gold and, on rare occasions, samples were assayed for silver or base metals. Requests for assaying to be conducted for elements other than gold were usually restricted to exploration samples from surrounding properties.

The QA/QC protocol followed by the NBM assay laboratory is quoted below:

- "**Control Charts**: Control values for in-house standard reference material are plotted daily to maintain statistical control and to identify any blunders, bias, or trends that may lead to out of control situations."
- "**Duplicate Assays**: Mill samples are charted on a daily basis to determine if the %RSD (relative standard deviation) or the precision of analysis is within limits (~5%)."
- "**Calibration Verification**: All atomic absorption calibrations are checked with certified reference solutions (1 ppm and 5 ppm) and is charted on a daily basis."
- "**Method Validation**: On a weekly basis in-house controls and dayshift cyclone samples (for a one week period) are re-assayed with a fire assay/gravimetric finish (FA/GRAV) and the aqua regia/MIBK method. These values will be compared to the original fire assay/atomic absorption measurement to assess method accuracy and validity."
- "**Outside Laboratory Checks**: On a 3-month basis, 30 core rejects and selected mill products are sent to ACME Analytical, Vancouver."
- "**Proficiency Testing (Round Robin)**: Bi-yearly participation in conjunction with CANMET, Ottawa."
- "**Sample Preparation Assessment**: Random samples selected monthly for proper sample preparation (representative and repeatable). Grind tests are performed on a weekly basis. Split core samples (blind to the lab) are selected randomly by geology to assess preparation variance and possible contamination/human error."
- "**Staff Training, Preventative Maintenance, and Procedure Audits**: These are ongoing as a commitment to quality, and are an integral part of the QA/QC process."
- "**Contamination Assessment**: Through the use of blanks and re-assayed samples, this is assessed daily."
- "**CANMET'S**: Certified reference material is assayed on a monthly basis – charted and variance determined."

Micon (October, 2006) reviewed the internal reports containing information on the sample collection, sample preparation, security, and analytical procedures that were followed at NBM during period of production from 1995 to 2005. It concluded that the procedures followed conform to industry standards currently in effect and that the procedures were adequate to ensure a representative determination of the gold content of any intervals of mineralization or alteration that were observed in the samples assayed at the NBM site.

13.2 Garson sample preparation, analyses, and security

Sample intervals are marked on the core by the geologist. As a QA procedure, a convention is followed where the samples are marked in red grease pencil including lines to mark sample limits and arrows to indicate the sample limit lines. The sample number is written in red grease pencil at the end of the sample interval. The sample interval is recorded in the sample booklet and the corresponding sample tags (2) are placed under the core.

The samples are cut into symmetrical halves using a saw equipped with a diamond blade. One-half of the core is placed in a sample bag with one of the two sample tags, and the other half of the core is replaced in the core tray with the second sample tag stapled to the core box.

The core trays containing the samples are archived in a secure location at the NBM site.

All of the sampling on the project was completed by Garson employees.

Garson ensured sample validity/representativeness by making sure that the core was split into symmetrical halves before being sampled under close supervision by a Qualified Person (QP).

Figure 13-1 depicts half split core of which the other half has been sampled. The sample collection was similarly under close supervision by a QP. The sample security was ensured by storage, transportation and delivery under lock and key (Figure 13-2).

Sample analyses including determination of gold and arsenic on all samples and specific gravity and multi-element geochemistry on selected samples was completed at TSL Laboratories Inc. (TSL) in Saskatoon, Saskatchewan. TSL was established in 1981, is ISO/IEC 17025 accredited, and participates in the proficiency testing program sponsored by the Canadian Certified Reference Materials Project. TSL is certified by the Standards Council of Canada for certain tests that include gold analyses utilizing instrumental or gravimetric finish.

At TSL the samples are dried and crushed and a 250 g riffle split of the crushed sample is pulverized to 95% passing 150 mesh (106 microns). Gold concentration is determined on all samples from a 30 g split of the pulverized sample by fire assay with atomic absorption (AA) finish. All samples with a gold concentration greater than 1,000 ppb (1 gram/t) are re-assayed using fire assay with a gravimetric finish. Specific gravity and ICP-MS multi-element analyses (aqua regia digestion) were completed on selected samples as requested by Garson project geologists.

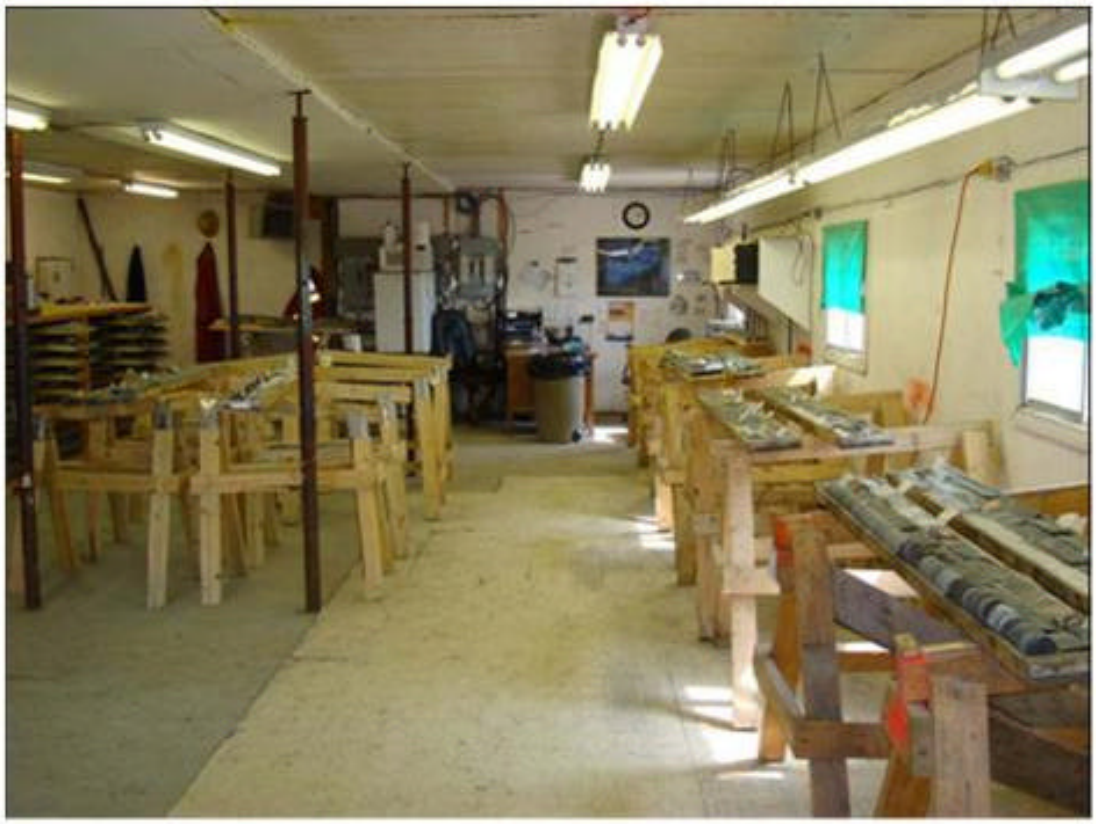

Figure 13-1: Core Shack at the NBM Showing the Half Split Core on the Table after Sampling

Figure 13-2: Locked Trailer Containing Garson's Drill Core Samples Awaiting Transportation to the TSL Laboratory

Analytical data are received from TSL first in electronic form via email which is followed by the delivery of the signed assay certificates. The electronic assay data file was checked against the assay certificate before being merged with the logged sample intervals using Gemcom software. The following quality control procedures were in place to ensure database integrity:

- Gold values were evaluated with logged sample descriptions.
- Gold values and sample descriptions were compared with the archived half core.
- Final assay certificates were compared with the electronic files for consistency.
- Garson used standard reference samples (standards) and check assays for analytical quality assurance.
- The standards used were selected to include ranges of gold grades and are tabulated below (Table 13-1).

Table 13-1: List of Standards used by Garson

Standard Name	Value (gold)
OREAS 50Pb	727 ppb
OREAS 17Pb	2.56 ppm
OREAS 10Pb	7.15 ppm
Table provided by Garson Gold Corp.	

Check assay work included re-assay of selected samples at an independent testing laboratory and at TSL using the following protocol:

Garson requested that TSL return the coarse rejects of selected samples to the Snow Lake office. These samples were repackaged and shipped to ALS Chemex Ltd. in Thunder Bay, Ontario, and these duplicate samples constitute the check samples for the exploration program. A 250 g riffle split of the coarse reject was pulverized to 85% < 70 microns. Gold concentration was determined on all samples from a 30 g split of the pulverized sample by ICP-AES. All samples with a gold concentration greater than 1,000 ppb (1 g/t) were re-assayed using fire assay with a gravimetric finish.

Garson requested that TSL return the coarse rejects of selected samples to the Snow Lake office. These samples were placed in new sample bags and were given new sample numbers and were re-submitted to TSL for gold determination as per the routine sample preparation and analyses used on Garson samples at TSL. These re-submitted samples act as Garson's duplicate samples.

As part of the process to establish the nugget effect (short range variance) for the No. 3 Zone and to serve as a form of quality assurance, a number of samples were quartered and submitted to TSL for routine gold analyses.

The analytical quality assurance data were evaluated on an 'as-received' basis. As well, the standard data are evaluated for detection of any trends over time. All issues that have been identified have been resolved and the assay data received from TSL are deemed to be of industry standard quality.

Micon reviewed the sampling program with Garson and with the exception of not submitting blank samples into the sample stream, Micon indicated that Garson's sampling program and protocols were sufficient to be used in a mineral resource estimate. Micon recommended that Garson add blank samples to its sampling program.

14. DATA VERIFICATION (JL)

The updated resource estimate for Snow Lake Mine and the No.3 Zone is based on the historical sampling and assay data as described in the previous section. Verification of the historical data has been completed and is described below, which is taken directly from the technical report by Micon International Limited, dated May 30, 2009. Alexis and its predecessor Garson, have completed a complete data verification of the historical Snow Lake Mine drill hole database for all diamond drill holes above the 1780 Level. The procedures and report on this work were reviewed for the current resource estimation. The QA/QC program and results for the drilling completed in 2010 by Alexis which is included in the current resource estimation work was reviewed with Alexis geological staff in Snow Lake. An independent check of assay certificates against database entries was also completed. It is concluded that the drill hole database is reliable and can form the basis for resource estimation.

Micon's first site visit was conducted by Richard Gowans, P. Eng on October 12, 2006. Mr. Gowans reviewed the current state of the facilities (mill, headframe, maintenance buildings, etc.) and completed an audit of the physical surface assets located on the NBM property. Micon's second site visit was conducted by William J. (Bill) Lewis, BSc., P.Geo and Charley Murahwi, MSc., P.Geo. from April 28 to May 2, 2008. During the second site visit, the following validation tasks were completed:

- review of the state of geological/mineralization knowledge
- review of the drilling quality assurance/control (QA/QC) procedures
- review of survey techniques for collar positions and down-hole surveys
- review of the core handling and sampling procedures
- review of the sample security procedures
- review of the assay QA/QC procedures
- review of the SG/bulk density determination procedures
- review of database integrity and storage procedures

A comprehensive review of the geology/mineralization wireframes was conducted at Micon's offices in Toronto on May 20, 2008, followed by a thorough review and audit of the resource estimate at the same venue during the period from June 2 to 4, 2008.

14.1 Current State of the New Britannia Mine Property

14.1.1 Main Mine Workings

Since the cessation of mining and gold processing activities at NBM, the facilities have been placed under care and maintenance. The mine facilities comprise an underground mine, headframe, hoist room, compressor house, administration office, electrical substation, surface maintenance shop, dry and warehouse, primary and secondary crushing buildings, process plant and associated support offices and laboratories, cold storage facility and a tailings storage facility.
Access to the underground workings is via a five-compartment shaft with a cage for men and equipment and separate ore and waste skipping facilities. While the older workings above the 1780 Level are accessible from the shaft, an internal ramp system begins on the 1780 Level to access the lower workings, excavated since 1995. In addition, the shaft can be accessed from the ramp on the 3000 Level via the 3000 Level haulage drift. The ramp extends to below the 3746 Level.

Access to the underground workings was closed off in March, 2005, and the mine has been allowed to flood since that time. It is estimated that the water inflow to the mine is 60 gal/minute and that it will take approximately 27 years for the mine to completely flood (D. Armsworthy, personnel communication, 2006).

Most of the mining equipment has been sold and removed from site although the hoist, hoisting control system and associated auxiliary equipment is complete and maintained in good operating condition. The main ventilation and heating system is also complete. Access could be re-established fairly quickly as all the facilities required to access the underground workings remain in place.

The crusher buildings, mill buildings, process equipment and associated electrical equipment are complete and in excellent condition. Only minor work would be required to re-commission the crushing and metallurgical processing facilities.

All the surface buildings have been well maintained since the stoppage of mine production, and the administration office is being used on a daily basis by the care and maintenance crew.

The tailings storage facility is being maintained and site environmental monitoring is continuing.

14.1.2 No. 3 Zone Workings

The No. 3 Zone workings were not inspected as the portal was sealed by NBM prior to closure of the main workings to the southeast. Prior to the closure of the No. 3 Zone workings, the portal was making water and the workings were completely flooded.

However, it is believed that the ramp to the underground workings can be easily rehabilitated. Due to the limited extent and shallow nature of the previously mined area, the capital input for re-entry would be much less than the input required to re-access the main NBM workings. The mined out area on No. 3 Zone is shown if Figure 6-4.

14.2 Validation of procedures and database integrity

14.2.1 State of Geological/Mineralization Knowledge

Following its review, Micon was satisfied that a sufficient amount of geological work had been undertaken by Garson to gain a thorough understanding of the geological and mineralization characteristics of the No. 3 Zone deposit (Lewis, 2009).

14.2.2 Drilling QA/QC and Survey

The drilling procedures as observed by Micon (Lewis, 2009) were standard for the industry, where the drill crew chief ensures quality drilling output by minimizing core losses, determining and correctly labelling the advanced metreage under the oversight of the drill geologist. Drill hole azimuths, inclinations, collar and down-hole surveys are the responsibility of the site geologist. Measures are in place to ensure that that all long (i.e. > 60 m) holes have downhole surveys at 50 m to 60 m intervals.

14.2.3 Core Handling, Sampling & Security Procedures

Micon (Lewis, 2009) was satisfied that the handling, logging, sampling and security procedures as practiced by Garson were consistent with acceptable practices in the mining industry. Locked containers and trailers in which the samples were transported were witnessed (Figure 13-2). However, the capacity of the core storage facilities will soon be inadequate and it was suggested that Garson would need to address this issue while the quantities of core are still manageable. Micon recommended that photographs of core for the entire lengths of drill holes be taken and then portions of the drill holes with mineralization intercepts incorporating both hangingwall and footwall sections are stored in a secure compound area (Lewis, 2009).

14.2.4 Assay QA/QC and Density/SG Data

In addition to using an accredited laboratory, Garson's assay quality assurance and control measures involved the use of certified reference material (standards), repeats and duplicate analyses, and check analyses using a different laboratory. As an additional precaution, Micon recommended that blanks be incorporated as well (Lewis, 2009). Micon's analysis of the duplicate samples and standards that were being used shows that the resource data is reliable for resource estimation.

The accuracy of assays from the early exploration drill holes has been of concern. There has not been twinning of older holes with new ones but several new holes that have been drilled close to the older ones have yielded comparatively similar assay results. Consequently, any concern on the accuracy of assays from the early exploration drill holes has been adequately addressed.

Density determinations were conducted using acceptable water immersion methods on the basis of which a tonnage factor of 11.21 ft^3/ton has been determined.

14.2.5 Electronic Database Validation

To verify completeness and reliability of electronic database files, Micon carried out random checks for errors in assay and logging/survey entries on 7 and 11 drill holes, respectively. In every one of these cases, no errors were found (Lewis, 2009). Consequently, it can be reasonably assumed that the database is relatively free of errors.

14.3 Conclusions on resource database

Based on the technical review of data collection and quality control procedures, Micon was satisfied that both the integrity and quality of the mineral resource database met the industry norms. Micon determined that the resource database formed a good basis upon which Garson could conduct a resource estimate (Lewis, 2009).

14.4 Review of the 2008 resource estimate for the no. 3 zone deposit

The independent Qualified Person responsible for the preparation of portions of this report, review of the 2008 resource estimate and the opinion on the propriety of the proposed exploration program is William J. Lewis, B.Sc., P.Geo. Prior to joining Micon in April, 2005, as a Senior Geologist, Mr. Lewis was employed at NBM as the Senior/Chief Geologist from 1997 to the closure and placement of the mine on care and maintenance in January, 2005. Garson requested Mr. Lewis to conduct the review of the No. 3 Zone resource because of his intimate knowledge concerning the former mining operations and his knowledge of the surrounding area.

Micon's review and audit of the 2008 resource included a review of the geological model, mineralization wireframes, domain statistical validation and visual trend slice validation. Available historical data and technical reports were also examined. At the time of the report, Micon was satisfied that its review of the material presented by Garson represented as complete a review of the NBM property as was possible based on the material presented and to the best of Micon's knowledge, information and belief, the Technical Report contained all scientific and technical information that is required to be disclosed and not make the report misleading.

15. ADJACENT PROPERTIES (JL)

The details of the adjacent properties are described in Section 15 and are taken directly from the technical report by Micon International Limited, May 30, 2009. To the authors knowledge, there has been no known change in property ownership in the immediately adjacent properties since the writing of the technical report by Micon International Limited, May 30, 2009 with the exception of the discovery and delineation of the Lalor polymetallic sulphide deposit which is described below and was included in the Golder Associates technical report dated April 2010.

The Nor-Acme/NBM has been the only major gold mine found within the Snow Lake district although a number of gold properties with limited production are known within the surrounding area. The majority of the past producing gold mines in the Snow Lake area are located on the eastern shore of Wekusko Lake, where the mineralization was comprised of gold, pyrite and arsenopyrite located within narrow quartz veins situated in shear zones. The majority of these producers saw intermittent small scale production prior to 1939 and due to their distance from Snow Lake do not directly affect the interpretation, evaluation of the mineralization, or anomalies found on the NBM property.

The Snow Lake area is more famous for the number of past producing lead, zinc and copper mines operated by HBM&S within the immediate vicinity of Snow Lake. Tables 15.1 and 15.2 show the known mines and mineral deposits surrounding Snow Lake. The majority of the information for these tables was obtained from the Manitoba Government report by Bamburak published in 1990. The other sources used for compiling this section were Fedikow, 1989 and Beilhartz, 2006. Resource and reserve estimates discussed in this section use the terminology utilized at the time of the estimate and are not to current CIM approved standards required by NI 43-101 regulations and should not be relied upon.

The only gold deposit which is near the NBM property and not on the property itself is the Squall Lake deposit located approximately 4 km northwest of the community of Snow Lake. This property adjoins mineral lease ML-323 along the northwestern boundary of the lease and consists of 14 contiguous mineral claims totalling 899 ha. Garson owns 100% of the Squall Lake property and the property has been the subject of an NI 43-101 report entitled “Report on the Squall Lake Property, The Pas Mining Division, Snow Lake, Manitoba” by D. Beilhartz, cited April 2006 and posted on the System for Electronic Document Analysis and Retrieval (SEDAR) website.

The geology, nature of the mineralization, number of known mineral showings, and historical production for both gold and base metals over the last century , continue to make the Snow Lake area a good place for further exploration. Micon, considers the Snow Lake area to be prospective in the search for new zones of gold mineralization and the NBM property holds good potential.

Table 15-1: List of Adjacent and Nearby Mining Operations

Name of Mine	Location NTS	Commodity	Recorded Production (For Years Recorded)	Years in Production
Anderson Lake (HBM&S)	63J/13 SW	Copper, Zinc	2,193,546 t, 3.41% Cu, 0.1% Zn (1970 to 1984)	1970 to 1988
Ballast-Moosehorn	63J/13 SW	Gold	3.4 kg Au (1917), 1.6 kg Au (1918), 1.1 kg Au 0.3 kg Ag (1931)	1917, 1918, 1931
Birch zone (New Britannia)	63K/16 NE	Gold	31,353 tons, 3,282 oz Au (1996)	1996
Chisel Lake (HBM&S)	63K/16 SE	Lead, Zinc, Copper	5,061,059 t, 0.60% Cu, 10.94 % Zn (1960 to 1984)	1960 to 1988
Chisel Lake North (HBM&S)	63K/16 SE	Lead, Zinc, Copper	No data available	2000 to Present
Ferro	63J/13 SE	Gold	16.3 kg Au, 0.9 kg Ag (1932)	1932 with limited production in later years
Ghost Lake/Lost Lake (HBM&S)	63K/16 SE	Lead, Zinc, Copper	478,277 t, 1.34% Cu, 8.87% Zn (1972 to 1984)	1972 to 1988
Laguna (Rex)	63J/13 SW	Gold	42.8 kg Au (1918), 10,1 kg Au (1920 to 1921), 9,193 t 171 kg Au, 21.0 kg Ag (1924 to 1925), 101,130 t 1,643.6 kg Au, 254.7 kg Ag (1936 to 1939)	1918, 1920 to 1921, 1924 to 1925, 1936 t0 1939
Nor-Acme/New Britannia	63K/16 NE	Gold	5,393,970 tons, 763,254 oz Au (1949 to 1958), 6,480,266 tons, 858,075 Au (1995 to 2005)	1949 to 1958, 1995 to 2005
No. 3 Zone	63K/16 NE	Gold	241,574 tons 30,335 oz Au (1995 to 1996)	1995 to 1996
Osborne Lake (HBM&S)	63J/13 NE	Copper, Zinc	2,852,007 t, 3.14% Cu, 1.52% Zn (1968 to 1983)	1968 to 1983
Photo Lake (HBM&S)	63K/16 SE	Lead, Zinc, Copper	No data available	1994 to 1998
Rod No 1 (Little Stall Lake) (HBM&S)	63J/13 SW	Copper, Zinc	22,675 t, 5.00% Cu, 4.5% Zn	1962 to 1964
Rod No 2 (HBM&S)	63J/13 SW	Copper, Zinc	42,621 t, 6.39 % Cu, 2.2% Zn	Mar 1984
Spruce Point (HBM&S)	63K/9 SW	Copper, Zinc	1,364,000 t, 2.36% Cu, 2.8% Zn, 2.0 g/t Au, 25.0 g/t Ag (1982 to 1988)	1982 to mid 1990s
Stall Lake (HBM&S)	63J/13 SW	Copper, Zinc	4,158,096 t, 4.29% Cu, 0.57% Zn, (1964 to 1984)	1982 to mid 1990s

Table compiled from Manitoba Government publications.

Table 15-2: Summary of Mineral Deposits Surrounding Snow Lake, Manitoba

Name of Deposit	Location NTS	Commodity	Estimated Historical Resources	Source for Resource Information
Apex	63J/13 SW	Gold	360,000 t at 2.4 g/t gold	George Cross Newsletter, Dec. 8 1983
Bomber	63K/16 SE	Copper, Zinc	549 m x 1.8 m x 244 m, at 0.10 g.t Au, 8.57 g/t Ag, 0.04% Cu, 1.0% Zn	Albert Koffman c 1960, MB Unpub. Info file
Farewell Lake	63K/08 NE	Copper	257,000 t at 2.03% Cu to 190 m plus lower grade to 370 m	Manitoba Mineral 1980-81 Annual Report
Ferguson	63J/13 NW	Gold	5,400 t at 51.43 g/t Au on the dumps	Northern Miner, March 29, 1951
Joannie	63K/16 SE	Copper	500,000 t at 1.28% Cu	Gail et al (1980, p. 31, 113)
Linda-McKayseff	63J/13 SW	Copper, Zinc	Linda 2 zone 11.8 Mt of massive pyritic sulphide averaging 0.3% Cu, 0.8% Zn, 10.29 g/t Ag and 0.86 g/t Au	Northern Miner, December 17, 1987
McCarrerty	63J/13 SE	Gold	Drill indicated 18,000 t at 6.82 g/t Au across 1.37 m for 137 m. Resampling in 1981 gave an average of 10.29 g/t across 1.14 m.	Northern Miner, April 11, 1935, March 12, 1981
Norris Lake	63K/15 NE	Copper, Zinc, Gold	Drill indicated 227,000 t at 0.55 g/t Au, 20.78 g/t Ag, 2.51 % Cu, 4.82% Zn over an average width of 2 m to a depth of 366 m.	Bar resources Limited, FS 250/81 V.S.E.
Pot Lake	63K/16 SE	Copper, Zinc	102,000 t at 1.43% Cu, 4.5% Zn, 18.86 g/t Ag, 3.77 g/t Au	MB Assessment, File 90238
Rail Lake	63K/10 NE	Copper, Zinc	295,000 t at 3.0% Cu, 0.7% Zn to 305 m	HBM&S 1970 Annual Report
Rainbow Group	63J/13 SE	Gold	18 t of ore from a pit treated at the Ferro mill produced 404 g of Au (1932), 27 t produced 198 g Au (1933)	Rainbow Lease Ledger, MB Recording Office
Ram	63J/13 SW	Copper, Zinc	Copper-zinc mineralization	Frose & Moore (1980, p. 7)
Reed Lake	63K/10 NE	Copper	1.36 Mt at 2.09% Cu (diluted 10%) to 549 m and open at depth	HBM&S 1974 Annual Report
Squall Lake	63K/16 NW	Gold	Margaret zone – drill indicated 680,000 t at 3.43 g/t Au (1985)	Northern Miner, June 13, 1988

Table compiled from Manitoba Government publications.

16. MINERAL PROCESSING AND METALLURGICAL TESTING (IW)

No mineral processing or metallurgical test work has been completed for the preliminary assessment and none was completed by the previous owners, Garson Gold Corp. However, historical metallurgical test work on material from the NBM has been conducted by various organizations in addition to the internal studies conducted while the mine was in production between 1995 and 2005. Description of the mineral processing and prior metallurgical testing are taken directly from the technical report by Micon International Limited, May 30, 2009. A metallurgical test program was conducted in 2010 using representative core samples from the planned production areas of the Main and # zones. These tests confirmed the attainable gold recovery estimated from prior operations and testing.

16.1 New Britannia Mine Processing Data

The existing New Britannia mill has a capacity of approximately 860,000 short tons (780,179 t) per annum. Gold recovery is by cyanide leach (carbon-in-pulp) and annual mill recoveries of 92% to 94% were typically achieved during its operating life.

During production (1995 to 2005), the New Britannia mill typically operated seven days a week and 365 days a year, except for periods of scheduled maintenance. Between 1995 and 2005, the New Britannia mill processed 6,753,193 tons (6,126,394 t) of ore at an average head grade of 0.132 Oz/ton 4.1g/ton for a total mill feed of 891,692 Oz of gold. During this same production period the mill recovered 822,550 Oz of gold for an overall recovery of 92.2%. These figures include production from the NBM and the satellite deposits (No. 3 Zone and Birch pit).

The common sulphide minerals in the deposit are arsenopyrite, pyrrhotite and pyrite. Gold is predominately associated with the arsenopyrite with the affinity of gold for arsenopyrite appearing to be structural as the gold occurs as minute lenticular masses and veinlets in openings on arsenopyrite crystal boundaries, and within fractures in the arsenopyrite. The gold also occurs as minute particles in quartz and calcite with this generally the case when the gold is visible macroscopically, a very rare occurrence.

16.1.1 Characteristics of the Mineralization at the New Britannia Mine

Although associated with the arsenopyrite, the gold is not considered to be refractory. The fine grained nature of the gold in the arsenopyrite necessitates grinds of a minimum of 80% passing 74 microns. Finer grinding does improve recoveries and when operating, the plant targeted a grind of 89% passing 74 microns.

The specific gravity of the ore fluctuates between 2.7 and 2.9 g/cm^3 and the Bond Work Index for the main mineralized zones within the mine is about 10.5 kWh/short ton.

A pre-aeration with lime and lead nitrate prior to cyanidation was used in previous operations to passivate the sulphide mineralization and to optimize leach kinetics and gold recovery.

Some gold occurs in solid solution in arsenopyrite and to a lesser degree in pyrite and chalcopyrite. Typically, 60% to 85% of the un-dissolved gold in the leach residues was in the minus 400 mesh fraction, which also contained 3 to 8 times more arsenic than the plus 400 mesh fraction.

Table 16-1 tabulates the typical “ore” composition of the NBM.

Table 16-1: Typical Mineralogical Composition of the New Britannia Mine

Chemical Analysis		**Mineralogical Analysis**	
Au	4.63 g/t	**Major Phase:**	
Ag	0.63 g/t	Quartz and albite	55.0%
Si	22.6%	Clinochlore and biotite	32.0%
Fe	5.6%		
Al	5.4%	**Minor Phase:**	
Ca	5.9 %	Ankerite	6.2 %
S	1.29%	Arsenopyrite	3.0 %
Mg	2.7%	Pyrite	2.7 %
Cu	<0.03%	Ilmenite and iron oxide	1.0 %
Zn	<0.06%		
Pb	<0.22%	**Traces:**	
Ni	<0.01%	Pyrrhotite, chalcopyrite and sphalerite	
As	1.2%		
Sb	0.0007%		
Te	0.001%		

In the previous mill operating period, cyanide leaching after moderately fine grinding has yielded plant gold recovery in the order of 94% for the Main Zone, which was the only feed for most of the period.. The limited historical production from 3 Zone indicated lower recovery, averaging 82% in 1995 during the period of initial mill operations. It is likely that this low recovery was a result of inadequate grinding and typical plant tune up difficulties.

For the feasibility study metallurgical study, new diamond drill core samples were selected from each of the Main and 3 Zone deposits to represent the material to be mined in future. Two composites were prepared with weighted (according to core interval length) amounts of the individual core samples, and targeting a gold content similar to the deposit average. Several available core samples from 3 Zone were excluded since they showed above average gold content. These composites were subjected to metallurgical tests then combined for subsequent testing. Core sample

locations are shown on the following sections through the two deposits, Figures 16-1 and 16-2.

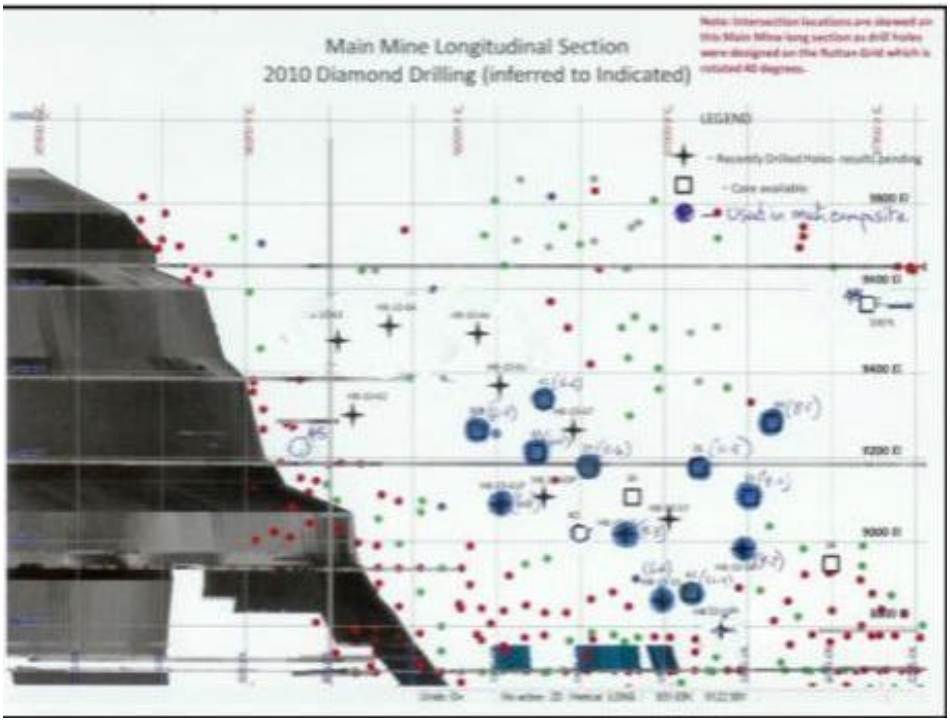

Figure 16-1 : Main Zone sample selection

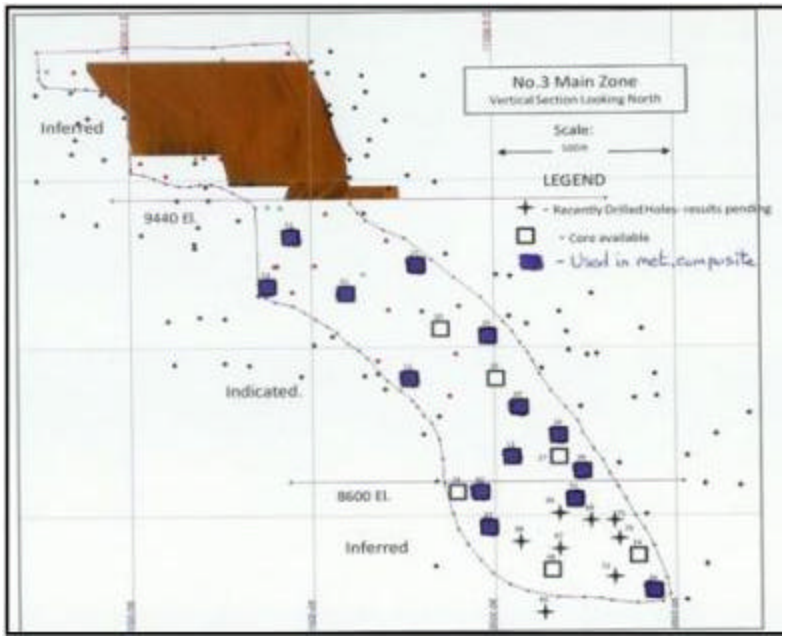

Figure 16-2 : 3 Zone sample selection

Metallurgical testing was conducted by the PRA Metallurgical Division of Inspectorate America Corporation at the laboratory in Richmond B.C. during July and August 2010.

Each composite was submitted for a Bond ball mill work index test based on a 75 micron closing screen, and results were (metric) 12.6 for Main Zone and 17.0 for 3 Zone. The higher hardness of the 3 zone material is attributed to a significantly higher quartz content of this zone.

A summary of the grinding and leaching determination tests is shown in Table 16.2.

Table 16-2 : Metallurgical testing summary

Test	Sample type	Grind p80	Leach hours	CN level g/L	Feed Au g/t	Extraction Au %	Consumption kg/t CN	Consumption kg/t Lime	Notes
1	Main	131	48	0.5	3.80	90.2	1.08	0.76	
2	Main	69	48	0.5	3.75	94.0	1.13	0.85	Scoping Main zone
3	Main	56	48	0.5	3.76	94.5	0.77	0.80	
4	3 zone	87	48	0.5	5.65	84.3	0.81	0.36	
5	3 zone	68	48	0.5	6.09	90.3	0.81	0.36	Scoping 3 zone
6	3 zone	53	48	0.5	6.35	91.0	0.84	0.44	
7	Main	72	36	0.5	3.82	93.2	0.51	0.82	confirm main
8	3 zone	64	36	0.5	6.79	89.1	0.61	0.63	confirm 3Z
9	M/3Z	57	48	0.25	4.36	92.0	0.54	1.13	M/3Z at 64:36
10	M/3Z	53	48	0.4	4.92	91.1	0.73	0.90	M/3Z at 64:36
11	M/3Z	58	48	0.4	4.00	92.9	0.69	0.66	M/3Z at 64:36, 10kg test

Testing with 3 Zone showed a higher sensitivity to grind size and leach time than for Main Zone. Optimum extraction for Main Zone was attained after 36 hours and a grind K80 of close to 69 microns. For 3 Zone, the optimum extraction was attained at a grind of 53 microns and 48 hours leaching, and showed that additional time may be beneficial. Subsequent tests with a 64:36 blend, to represent planned processing in initial operations, confirmed these values, summarized in Figures 16.3 and 16.4.

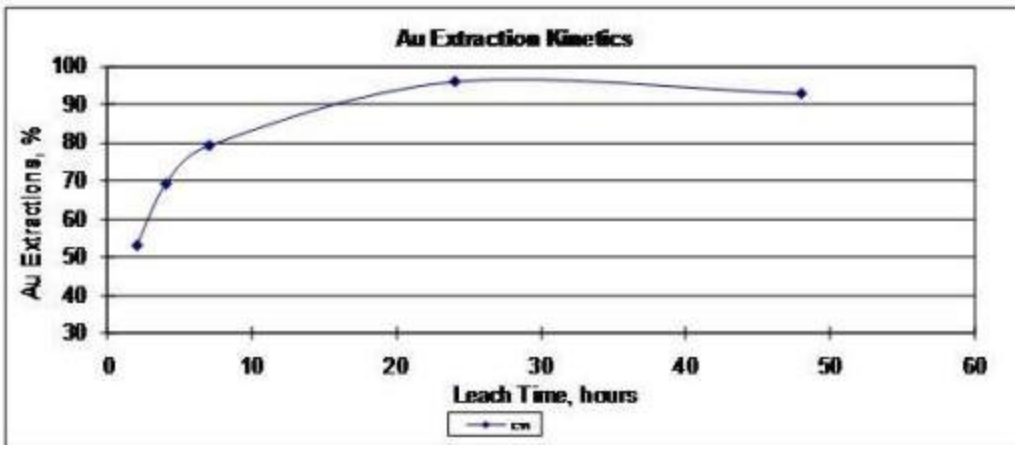

Figure 16-3 : Gold extraction from composite blend, test 11

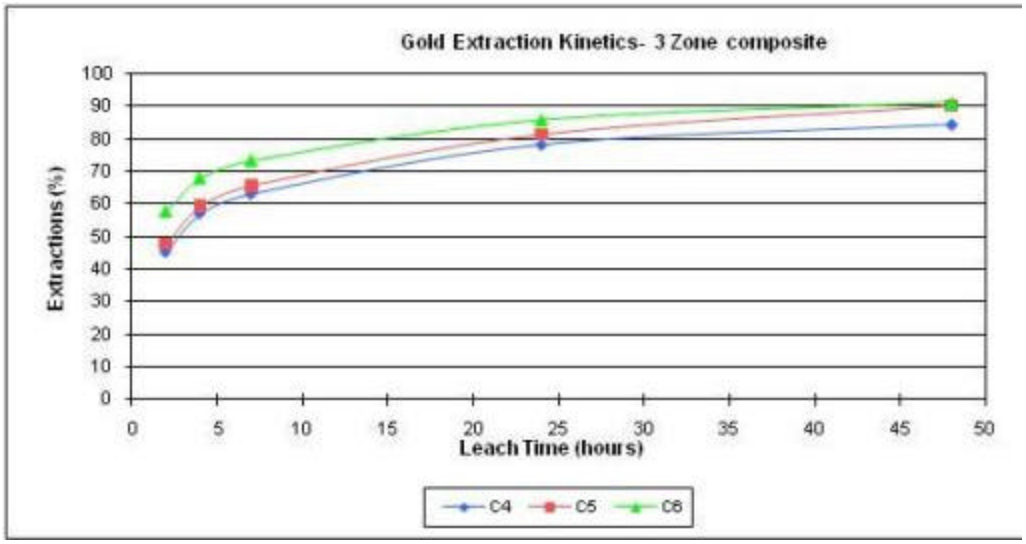

Figure 16-4 : gold extraction of 3 Zone, at three grind sizes

These tests confirmed the response of the ore types which had been demonstrated in the previous plant operations from 1995-2004. The plant traditionally ground the ore, in later years, to a size of approximately 70 microns which appears suitable for the Main Zone production, although the available equipment could have maintained a finer grind.

Based on these testing results a predicted gold extraction in the mill is 94.0% for Main Zone and 91.0% for 3 Zone, when ground to 65 microns for Main Zone and 53 microns for 3 Zone. Silver extraction is predicted at 70%. When processing a blend of main and 3 Zone material at the planned 64:36 proportion, an extraction of 92.9% is predicted. Indicated lime consumption levels are similar to the historic plant levels, and cyanide consumption in these tests, as expected, were somewhat higher than in the plant due to intermittent control.

The final test 11 was conducted on a large 10kg sample to allow testing of cyanide destruction, using the Inco SO2/air process, and arsenic reduction with ferric sulphate addition. Cyanide destruction was successful, reducing the plant tailings slurry to a level well below 1 mg/L (CN WAD) in three hours. Aging tests and arsenic removal tests on the final effluent are continuing. A sample of tailings residue was tested for Acid-Base Accountability levels and showed the residue to be a highly positive net acid consumer due to the low sulphide content and high alkaline rock content.

17. MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 Summary of Previous Resource Estimates on the Snow Lake Property

Mineral resources on the Snow Lake Mine property are located at the Main Mine, No.3 Zone, Birch Zone, and Margaret Zone (Table 17-1). Resources at the Main Mine, No.3 Zone, and Birch Zone were originally documented in the Micon technical report dated October 2006 and resources at the Margaret Upper and Lower Zones are documented in the technical report by Beilhartz dated April 2006. The mineral resources for the No.3 Zone were updated and documented in the Micon technical report dated July, 2008. The mineral resources for the Main Mine were updated and documented in the technical report on the preliminary assessment of the Snow Lake Mine dated April, 2010. These resource estimates are summarized in Table 17-1 and compiled here for historical reference only.

Table 17-1: Summary of previous resource estimates on the Snow Lake Property.

Zone	Resource Category	Tons	Grade (opt)	Ounces Au
NBM Main Mine[1]	Measured	7,000	0.139	1,000
NBM Main Mine[1]	Indicated	3,090,000	0.145	449,000
No.3 Zone - Main[2]	Indicated	434,000	0.20	85,000
Total Snow Lake Mine Property Measured and indicated		3,531,000	0.152	535,000
NBM Main Mine[1]	Inferred	625,418	0.136	85,129
No.3 Zone - Main[2]	Inferred	288,000	0.230	65,000
No.3 Zone - Footwall[2]	Inferred	346,000	0.150	51,000
Birch Zone[3]	Inferred	627,000	0.129	81,000
Squall Margaret Upper[4]	Inferred	110,000	0.142	15,600
Squall Margaret Lower[4]	Inferred	371,000	0.158	58,500
Total Snow Lake Mine Property Inferred Resources		2,367,418	0.152	356,229

1) NI 43-101 technical report on the Snow Lake Mine by Eric Nimton, P.Eng, Andrew Bamber, PhD, Patrick Sevigny, Ing, and Jamie Lavigne P.Geo., Golder Associates, April 2010

2) NI 43-101 Technical report - Preliminary Assessment of the No. 3 zone on the New Britannia Mine Property, Snow Lake Manitoba by William J. Lewis, P. Geo, Charley Murahwi, M. Sc, P. Geo and Christopher Jacobs, CEng, Micon International Ltd, October 2006

3) NI 43-101 Technical report on the New Britannia Mine Property and Review of the Mineral Resource Estimate, Snow Lake Manitoba by William J. Lewis, P. Geo and Richard Gowans, P. Eng., Micon International Ltd, October 2006

4) NI 43-101 Technical report on the Squall Lake Property, The Pas Mining Division Snow Lake Manitoba completed by D. Bellhartz, P. Geo, April 2006

17.2 Recent exploration and evaluation work and current scope (JL)

Since the previous resource estimates and associated technical reports (Table 17-1), Alexis has completed diamond drilling at the No. 3 Zone and Main Mine. In addition, Alexis has initiated a feasibility study for the Snow Lake Mine. The scope and mandate of the current work is to update and review the resources on the Snow Lake Mine property to support the feasibility study. Specifically this includes:

1) Update the Main Mine resource estimate above the 1780 Level (Main Mine Upper) to incorporate the results of drilling completed since the most recent estimate;
2) Update the No.3 Zone resource estimate to incorporate the results of drilling completed since the most recent estimate;
3) Evaluate and re-estimate resources above the 1780 Level in the Main Mine and at No.3 Zone using a revised cut-off grade, and;
4) Complete an audit of the Main Mine resources below the 1780 Level (Main Mine Lower) that were reported in the Micon 2006 technical report (Micon, 2006). These resources will be incorporated into the feasibility study.

17.3 Estimation methodology, personnel, and software (JL)

The estimation of gold resources at the Main Mine and at the No.3 Zone have been completed using a conventional approach of sectional interpretation and wireframing, statistical and geostatistical evaluation, and block modeling (outlined in Table 17-2)

Table 17-2: Summary of resource estimation methodology

Procedure	Description
Geological Modeling	Sectional interpretation of geology and mineralization; verification of interpretation in plan, creation of geological surfaces and wireframes; includes application of cut off grade and minimum width parameters; creation of surface and overburden wireframe models; compilation and creation of excavation models based on historical mining
Statistics and Geostatistics	Statistical analyses of assay data; evaluation of assay populations within and related to geological and mineralization solids; evaluation of assay data for top cut value; calculation and evaluation of composites for grade interpolation; variography
Block Modeling:	Creation of block model; grade estimation of blocks (interpolation) using appropriate method; estimation of block density; estimation of percent of block within (and outside of) geological or mineralogical domains (wireframes); estimation of tonnage; tabulation of tonnage and grade
Classification	Evaluation of the block model; extraction of mined out volume and otherwise non-resource blocks; classification of resource blocks based on geological, data density, and

	statistical/geostatistical criteria
Validation	Various procedures and tests to ensure model validity

The resource estimates have been completed by, and under the supervision of, Jamie Lavigne, P.Geo and Qualified Person as defined by National Instrument 43-101. Assistance with geological modeling and wireframe creation has been provided by Riverbend Geological Services Inc., based in Beausejour, MB and by Alexis personnel based in Snow Lake. Statistical analysis and variography was completed by SRK Consulting Engineers and Scientists ("SRK"). Alexis personnel have completed internal QA/QC reports as well as the referenced data base validation reports.

Geological modeling, wireframe creation, and block modeling, as well as the numerous procedures and routines related to block modeling, grade estimation, and resource tabulation have been completed using Gemcom Software (Gems 6.2.4). Statistical and geostatistical work by SRK was completed using Geostatistical Software Library ("GSLib") and Datamine Software.

17.4 Economic parameters and cut-off grade (JL)

The previous estimates have utilized a cut off grade interpreted over a minimum true width. The 2006 estimate was based on a cut off grade of 0.097 Oz per ton gold over true widths which varied from 5 ft to 7 ft depending on the location in the mine. The 2008 Micon estimate for the No.3 Zone and Golder 2010 estimate for the Main Mine upper used a cut off grade of 0.08 Oz per ton over a minimum true width of 6 ft. The current resources have been estimated over a 6 ft minimum true width and a cut off grade of 0.057 ounce per ton. The economic parameters and basis for the cut-off grade used for the current estimates is laid out in Table 17-3.

Table 17-3: Summary of economic parameters and cut off grade (Source: Alexis)

Parameter	Units	Value
Estimated Mining Cost	CAN$/t	46.00
Estimated Milling Cost	CAN$/t	14.00
Estimated G&A	CAN$/t	14.00
Total Operating Cost	CAN$/t	74.00
Assumed Gold Price	US$/Tr.Oz	1,200
Assumed Exchange Rate	CAN$/US$	1.05
Gold Recovery	%	93
Cut Off Grade	g/t	1.96
Cut Off Grade	Oz/ton	0.057

17.5 Estimation of mineral resources Main Mine Upper (JL)

17.5.1 Database and Database Validation

The complete drill hole data base for the Snow Lake Mine consists of 10,075 holes that include surface holes, and underground exploration and delineation holes. The number of holes and total footage is summarized by date and company in Table 17-4. The holes drilled in 2009 and 2010 (28 total) have been incorporated into the current interpretation and resource estimation.

Table 17-4: Summary of Snow Lake Mine upper diamond drilling

Year	Company	Number of Drill Holes	Total Length of Drilling Program	
			feet	metres
2010	Alexis Minerals Coproration	25	25,548	7,787
2009	Garson Gold Corp.	3	2,997	913
2007-2008	Garson Gold Corp.	10	11,869	3,618
2003-2004	Kinross Gold Corp. & High River Gold Mines Ltd.	1,127	222,725	67,886
1994-2002	TVX Gold Inc. & High River Gold Mines Ltd.	7,107	1,623,202	494,750
1988-1989	High River Gold Mines Ltd.	88	32,181	9,809
1988-1989	INCO Gold Co.	86	58,206	17,741
1948-1956	Howe Sound Exploration Co. Ltd & Britannia Mining & Smelting Co. Ltd.	1,629	304,352	92,766
Total		**10,075**	**2,281,080**	**695,271**

The assay database consists of a total of 131,433 samples with Au analyses for a total length of assayed core of 459,759 ft. The number of samples and footage analyzed are summarized in Table 17-5. Included are a total of 1934 samples (4245 total core footage) from the 2009 and 2010 drilling (Table 17-5).

Table 17-5: Summary of Snow Lake Mine assays

Year	Company	Number of Samples	Total Length of Sampling	
			feet	metres
2010	Alexis Minerals Corporation	1706	3701	1,128
2009	Garson Gold Corp.	228	544	166
2007-2008	Garson Gold Corp.	1,093	2,356	718
2003-2004	Kinross Gold Corp. & High River Gold Mines Ltd.	12,542	43,543	13,272
1994-2002	TVX Gold Inc. & High River Gold Mines Ltd.	87,770	285,204	86,930
1988-1989	High River Gold Mines Ltd.	2,519	6,923	2,110
1988-1989	INCO Gold Co.	4,597	12,867	3,922
1948-1956	Howe Sound Exploration Co. Ltd & Britannia Mining & Smelting Co. Ltd.	22,912	108,866	33,182
Total		**131,433**	**459,759**	**140,134**

The database for the Snow Lake Mine originated with the previous operator of the mine, Kinross Gold Corp. Alexis, and the predecessor to Alexis, Garson Gold Corp. ("Garson"), have completed a comprehensive data base validation for all holes in the Main Mine Upper. This work is documented in Alexis internal reports that are held in the Snow Lake office and has been described in the April 2010 technical report (Golders, 2010). During the course of mining during the period 1995 to 2005, an on-site assay laboratory was used for delineation and exploration drilling with quality control checks sent to an external, accredited, assay laboratory. The on-site laboratory, methods and associated QA/QC have been described in the Micon (2006) technical report.

For the present purposes, the QA/QC data for the 2009 and 2010 drilling was reviewed with Alexis geologists in Snow Lake and an independent check of the data base was supervised by the author.

It is concluded that the Main Mine Upper drill hole data base is appropriate for use as the basis of resource estimation.

17.5.2 Gemcom Database

The database used in Gemcom was extracted from the Snow Lake Mine data base and consists of the following tables and information:

1) Header – Drill hole collar coordinates in the local Main Mine grid system.
2) Survey – Down-hole survey data including dip and grid azimuth.
3) Assay – Sample intervals with assayed gold values.
4) Lithology – Rock type intervals re-coded by Garson Gold and Alexis geologists.
5) Structures – Core Axis Measurements recorded during logging.

17.5.3 Gemcom Solids of Previous Mining and Development.

Three dimensional wireframes representing historical mining and development have been used in the assessment of the Main Mine Upper resources. The wireframes are from two sources:

1) Wirefarmes representing mined out volumes and development from the later phase of mining, 1995 to 2005, which are in located in archived Kinross Gold Corp. Gemcom projects, and,

2) Wireframes of the first phase of mining and development (1948 to 1958) created by Garson, in AutoCAD and Gemcom, from archived geology and engineering plans.

Some checks of the Gemcom wireframes were completed by the author and their validity was discussed with the mine personnel and it is concluded that they are a suitable representation of mined shapes and positions for the purposes of resource estimation.

A long section of the mined out stopes and development is shown in Figure 17-1.

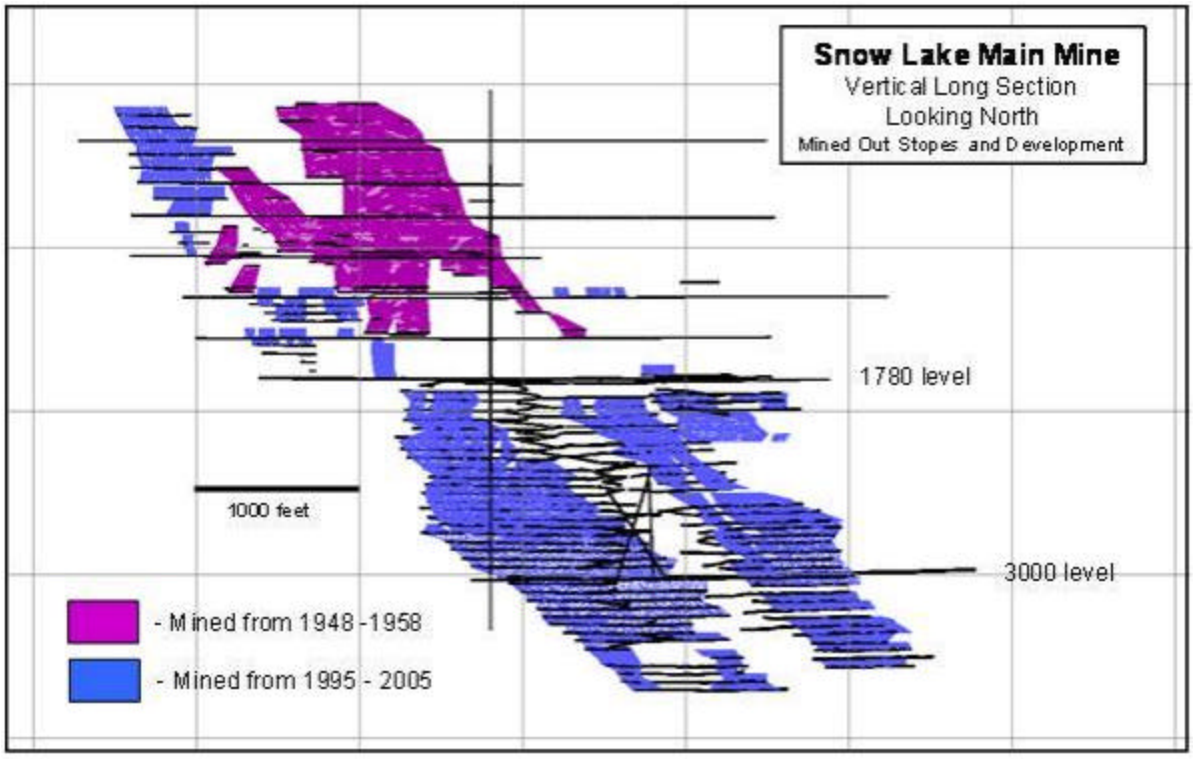

Figure 17-1: Vertical long section looking north of he Snow Lake Mine with development and mined stopes

17.5.4 Geological Model

Based on historical description of the Main Mine Upper as well as verbal description from mine geologists who worked underground during the 1995 – 2005 phase of mining as well as limited first hand examination of drill core and dump samples, mineralization is the lode gold class of deposit and appears dominantly to be of the pervasive replacement variety with lesser vein, sensu-stricto, type mineralization. Inspection of the grade profiles indicates relatively sharp grade contacts in both the hangingwall and footwall. Following the methodology used for resource estimation during the 1995 – 2005 phase of mining and that used for the most recent update, the gold lode has been interpreted on drill sections applying a cut-off grade of 0.057 Oz per ton and a minimum true width of 6 ft. An example of a sectional interpretation is contained in Figure 17-2

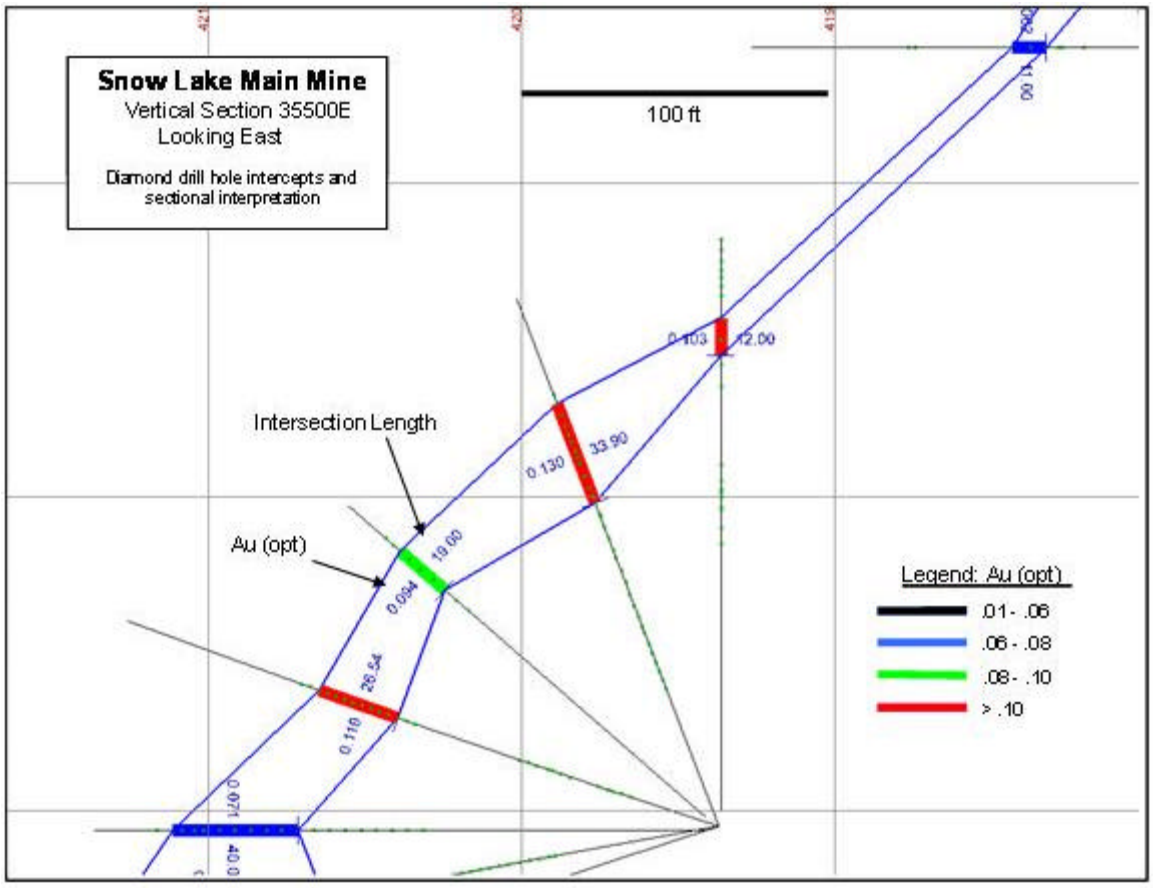

Figure 17-2: Snow Lake Mine cross section looking east of diamond drill hole intercepts interpreted at a 0.057 cut off grade

The sectional interpretation was completed across the entire strike of the zone of mineralization from surface to below the 1780 Level. The interpretation incorporates, and is completely consistent with, the previously mined stopes. From the sectional interpretation, wireframes were completed which forms the basis for statistical and geostatistical analyses and resource modeling.

The mineralization, (Figures 17-3 and 17-4), forms a curviplanar shape with a North East plunging axis and distinct orientation domains to the east and west of the plunging axis. The zone to the east of the axis and approximately east of the shaft in the long section view, (Figure 17-3), is referred to as the Upper Hogg Ruttan Zone ("UHR"). The zone west of the fold axis is referred to as the Upper Dick Toots Zone ("UDT"). Drill hole intercepts within the wireframes are also shown on the long section in Figure 17-4. Given the distinction between the UDT and the UHR, the statistics and geostatistics have been completed on each domain separately.

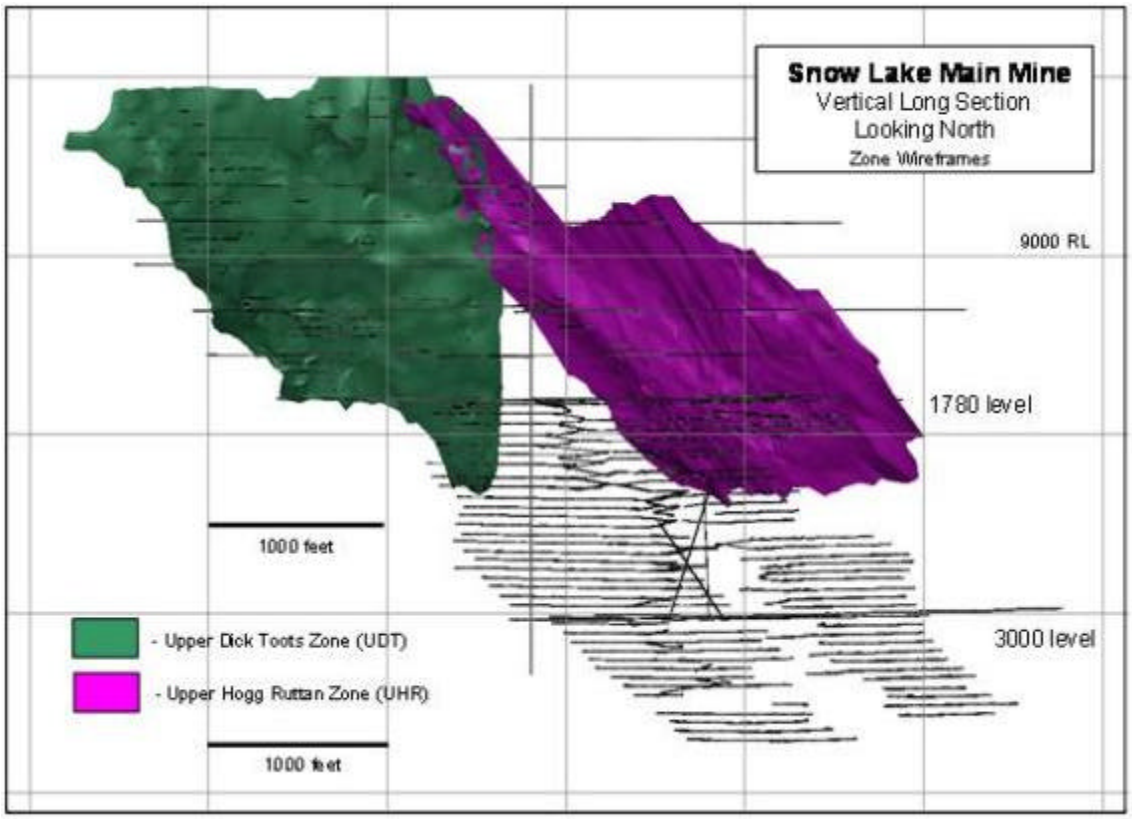

Figure 17-3: Vertical long section looking north of the Snow Lake Mine with development and modeled zones (wireframes).Figure 17.3. Vertical long section looking north of the Snow Lake Mine with development and modeled zones (wireframes).

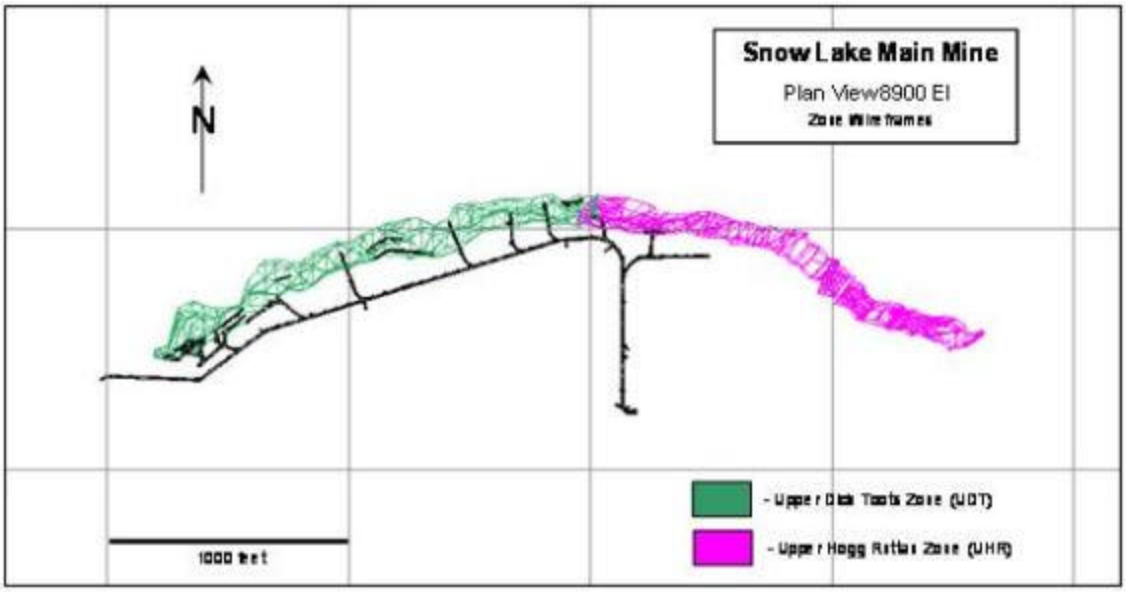

Figure 17-4: Plan view at elevation 8900 of the Snow Lake Mine with development and modeled zones (wireframes).

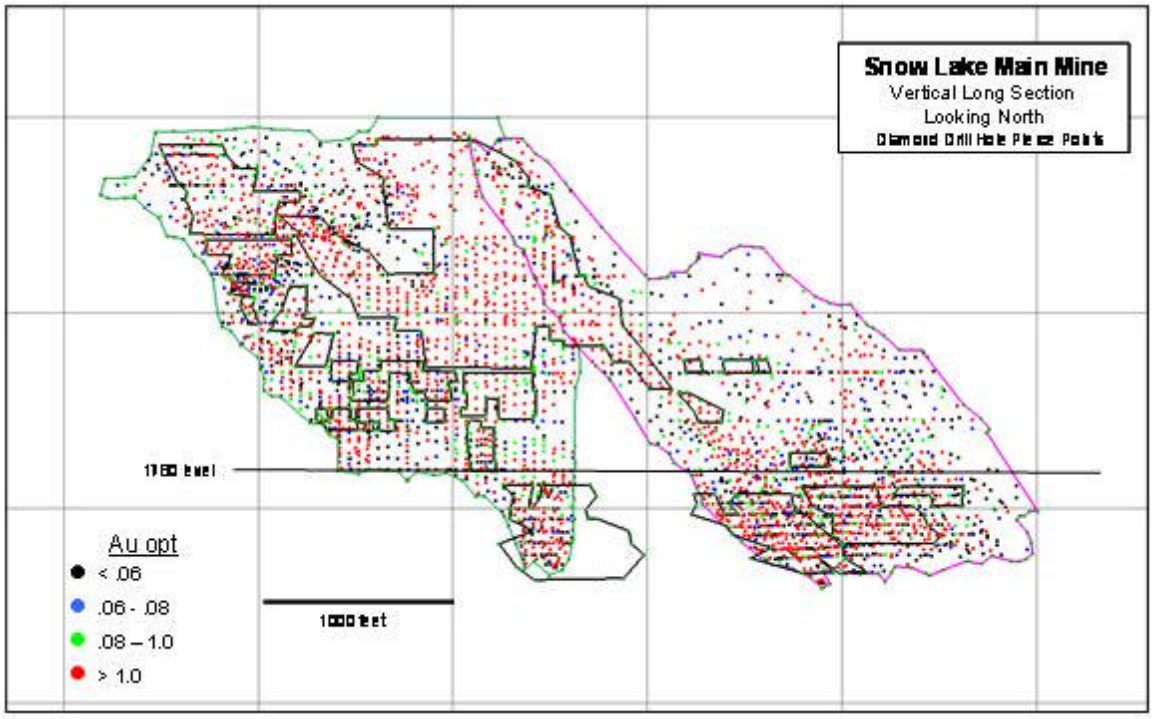

Figure 17-5: Diamond drill hole intercepts with wireframes and mined out areas

17.5.5 Statistics and Geostatistics

17.5.5.1 Compositing

Samples composited to standard length and utilized for grade interpolation are those samples within the limits of the wireframe. The dominant sample length is 5 ft and accordingly was chosen as an appropriate composite length. Samples were composited from the from the top of the hole in a down hole direction. This process results in the last composite in an interval being less than 5 ft. Last composite intervals greater than 2.5 ft have been used as is and those composites less than 2.5 ft have been added to the previous composite and the length weighted average gold grade calculated. The summary statistics of the composites are contained in Table 17-6.

Table 17-6: Statistics of the composites, Main Mine Upper

Statistic	UDT		UHR	
	Au (opt)	AuCut (opt)	Au (opt)	AuCut (opt)
n	12040	12040	7123	7123
mean	0.131	0.130	0.123	0.122
std dev	0.121	0.115	0.139	0.107
CV	0.922	0.882	1.129	0.882
min	0.001	0.001	0.001	0.001
Q1	0.059	0.059	0.050	0.050
median	0.104	0.104	0.100	0.100
Q3	0.175	0.175	0.164	0.164
max	2.040	0.820	7.224	0.700

17.5.5.2 Grade Capping Evaluation

SRK evaluated the composite data for both the UDH and UHR zones using histograms and probability plots. SRK found clear breaks in the composite grade distributions at 0.70 Oz per ton for the UHR domain and 0.82 Oz per ton for the UDH domain (Table 17-7) in both cases representing greater than 99.7% of the data.

Table 17-7: Summary of grade capping, Snow Lake Mine

Domain	Grade Capping Value (opt)	Number Capped/Total Number in Domain	Percent Equivalent
UHR	0.70	21/7125	99.70%
UDT	0.82	30/12040	99.75%

17.5.5.3 *Variography*

Variography was completed by SRK on each of the domains using the capped composite values. SRK examined 1) traditional variograms, 2) traditional correlograms, 3) normal scores variograms and 4) normal scores correleograms. SRK determined clear and well defined directions of continuity (anisotropy) (Figure 17-6). SRK completed variogram modeling primarily on the traditional variograms which is summarized in Table 17-8.

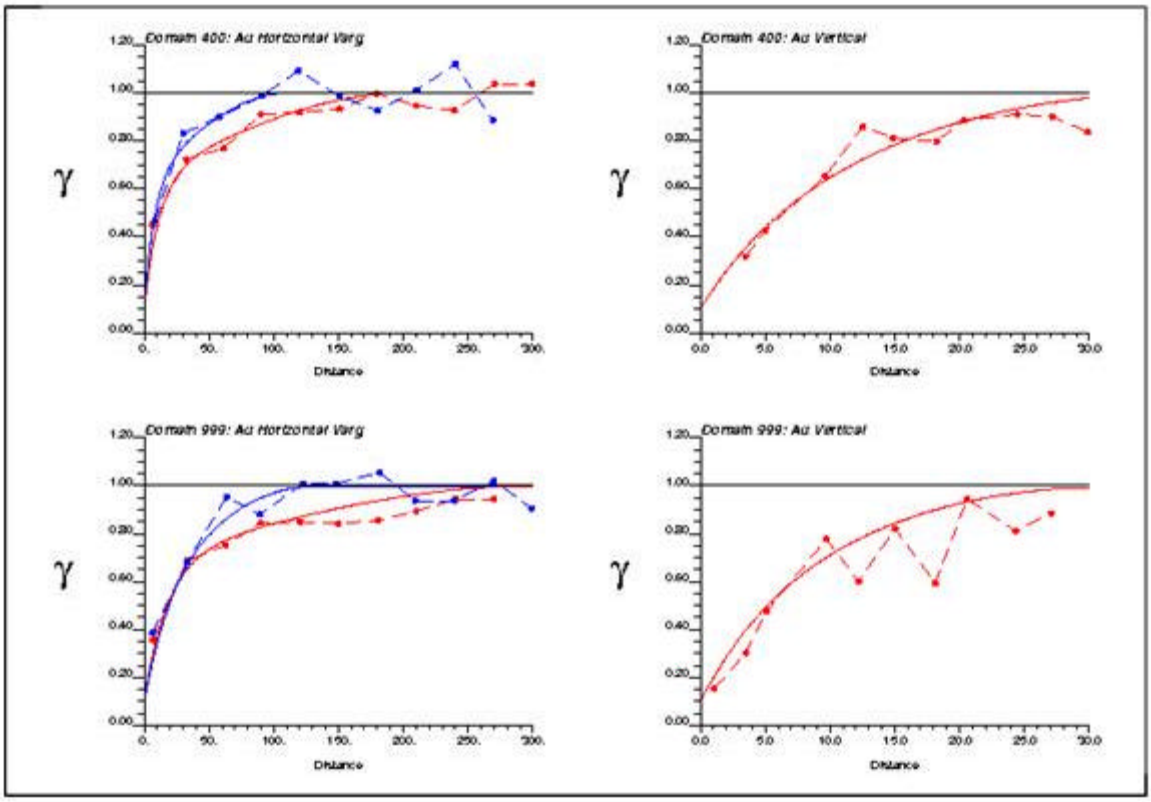

Figure 17-6: Modeled variograms for Gold Grade in UHR Zone (top row) and UDT Zone (bottom row): 'Horizontal' variograms display planar continuity where red corresponds to the down dip direction and blue is perpendicular to the down dip and in the plane; 'Vertical' corresponds to perpendicular to vein direction. *Source: SRK Consulting report to QP*

Table 17-8: Variogram model specifications, Snow Lake Mine

DOMAIN	GSLIB Rotation			GEMS (azm, dip, azm)			Variogram Model						
	ANG1	ANG2	ANG3	azm1	dip	azm2	Nugget	IST	TYPE	CC	RX	RY	RZ
UHR	45	-45	-10	45	-45	-52	0.1	1	EXPONENTIAL	0.55	35	25	23
	45	-45	-10	45	-45	-52		2	SPHERICAL	0.35	200	110	35
UDT	45	-35	-50	45	-35	-79	0.1	1	EXPONENTIAL	0.6	55	70	20
	45	-35	-50	45	-35	-79		2	SPHERICAL	0.3	300	130	30

Notes:
RX corresponds to direction specified by ANG1 (azm) and ANG2 (dip).
RY corresponds to direction perpendicular to dip and across the vein.
RZ corresponds to direction of the width of the vein.

17.5.6 Block Model Parameters

A block model was established in Gemcom with its geometry and limits defined to support modeling both the UHR and the UDT domains. The block size is 15ft x 15ft x 15ft with an origin at 33,500E, 40650N, and a maximum elevation of 10,000RL. The model extends 4875 ft east, 2700 ft north, and 2400 ft vertical. The block size is appropriate given the nominal drill spacing in the Main Mine Upper.

17.5.7 Grade Estimation

Utilizing the well developed variogram models, ordinary kriging was used as the primary grade interpolation method. For comparative and validation purposes, inverse distance to the power 2 (ID2) and Nearest Neighbor (NN) models were also estimated. The variogram model parameters for both the UHR and the UDT Zones given in Table 17.9 were used in the Gemcom kriging profile. Interpolation parameters are summarized in Table 17-9.

Table 17-9: Interpolation parameters for the OK models for the UHR and UDT Zones

Zone	Pass	Search Ellipse Orientation			# composites		
		Az	Dip	Az	Min/Block	Max/Block	Max/Hole
UHR	1	45	-45	-52	4	8	3
UHR	2	45	-45	-52	4	8	3
UDT	1	45	-35	-79	4	8	3

Notes: The range for the first pass of the UHR Zone is 200 feet in the directio of the principal Az, 110 feet along strike, and 35 feet across the zone

The range for he first pass at the UDH Zone is 300 feet in the direction of the principal Az, 130 feet alon strike and 30 feet across the zone

The range for the second pass for the UHR Zone is 2x the first pass range

17.5.8 Tonnage Factor and Tonnage

A tonnage factor of 11.2 ft^3 per ton (0.0893 tons/ft^3) was previously used for resource estimation at the Snow Lake Mine. This value was initially established through specific gravity measurements during exploration drilling and subsequently verified by testing at Lakefield Research during 1997 and in 2001 (Micon, 2006). This value has been used as the tonnage factor for the current resource estimation.

Based on user inputs, Gemcom calculates the fraction (percent) of a given block within the mineralization wireframe. The tonnage of mineralization in a given block is determined by:

block volume * percent within mineralization wireframe * 0.0893

17.5.9 Validation

The results of the modeling process have been evaluated through:

1) A visual review of the results
2) Block model and wireframe volume comparisons
3) Comparison with other in interpolation methods
4) Comparison of the mined out and not mined out parts of the model
5) Swath plots

17.5.9.1 *Visual Inspection*

Detailed visual inspection of the model has been completed in cross section and in long section. The proper coding of blocks by zone including those coded above the bedrock surface, below the 1780 Level, mined out, or in the area of the McLeod Road Thrust Fault was confirmed. In the UDT Zone, those blocks that are tabulated in the UHR zone where the wireframes overlap have been eliminated. The distribution of block grades was compared relative to composite point data and relative to both composite and raw assay data plotted on drill hole traces to ensure the proper representation of the underlying data in the model. Block model long sections of each the UHR and UDT zones were created with a single block across (horizontal) the lode with weighted average grade. The distribution of block grade in long section is consistent with the distribution of intercept grades and outlines grade domains. This is especially apparent within the UHR zone consistent with the distribution of low grade and high grade domains (Figures 17-7 and 17-8).

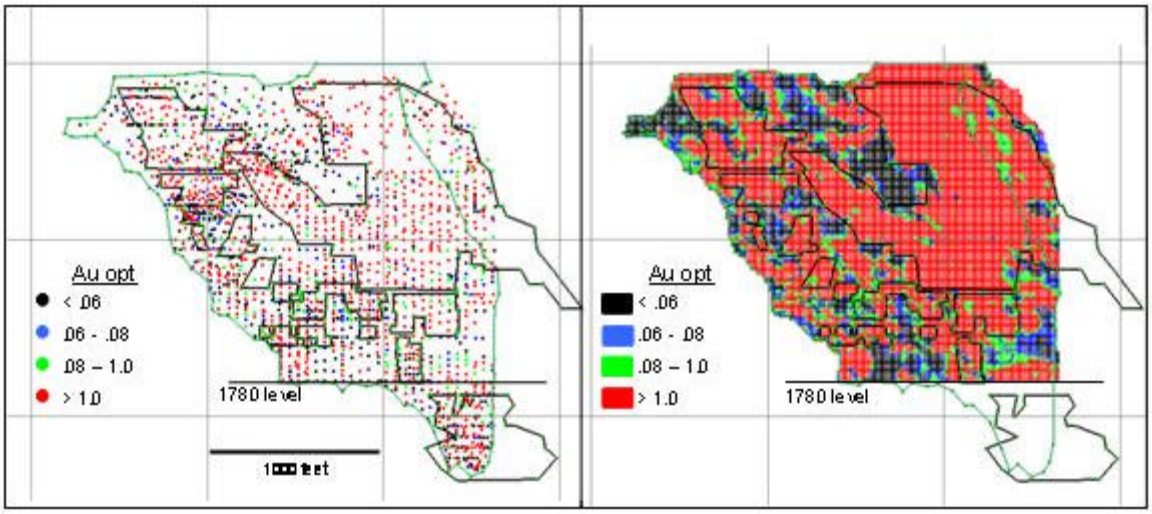

Figure 17-7: Long section of the UDT Zone. Right hand side is composite intersection grade and left side is block model coded by block grade

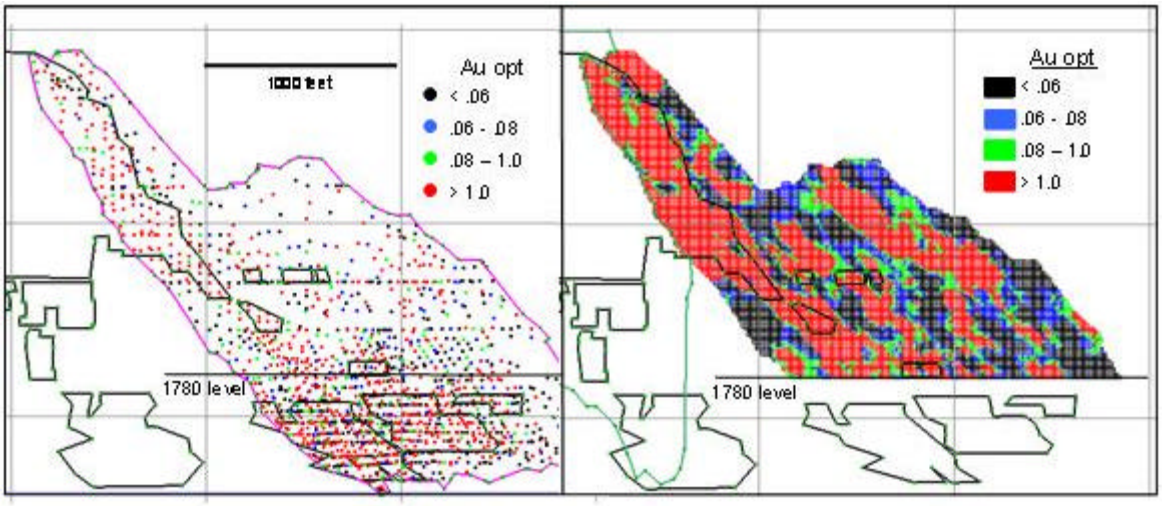

Figure 17-8: Long section of the UHR Zone. Right hand side is composite intersection grade and left side is block model coded by block grade

17.5.9.2 *Block model and wireframe volume comparison*

The total block model volumes for the UHR and UDT Zones was compared with the corresponding wireframes. The results show a slight difference that is within acceptable limits (Table 17-10).

Table 17-10: Block model and wireframe volume comparisons

	Volume (ft3)		Percent
Zone	Wireframe	Block Model	Differenc
UHR	61,808,846	61,805,732	0.005
UDH	103,779,366	103,772,661	0.006

17.5.9.3 *Comparison of other interpolation methods*

For comparison purposes, grade block models were generated using nearest neighbor and inverse distance squared interpolation methods. The nearest neighbor model is interpreted as the unbiased declustered representation of the underlying composite data. Similarity between the NN and OK estimates is interpreted as a measure of the OK model estimated being globally unbiased. The global average of the ID2 model is essentially the same as the OK and NN models. This consistancy increases the confidence in the overall estimate.

Table 17-11: Interpolation methods comparisons

Zone	Method	Volume	Density	Tonnage	Grade (Au opt)
		ft**3 x 1000	tons/ft**3	tons x 1000	Au opt
UDH	OK	103,772.612	0.089	9,256.517	0.140
UDH	NN	103,772.612	0.089	9,256.517	0.141
UDH	ID2	103,772.612	0.089	9,256.517	0.141
UHR	OK	61,805.732	0.089	5,513.071	0.117
UHR	NN	61,805.732	0.089	5,513.071	0.118
UHR	ID2	61,805.732	0.089	5,513.071	0.118

17.5.9.4 *Comparison of the mined and unmined portions*

The vast majority of historical mining from above the 1780 Level has come from the UDT zone. Although the continuity of mineralization within the UDT wireframe, i.e. within and between mined out areas, indicates that the zone is a continuous geological feature, the difference in mean composite grades inside and outside of the mined out stopes indicates distinct economic domains (Table 17-12).

Table 17-12: Summary Statistics of composites inside and outside of the mined out stopes in the UDT Zone.

Statistic	Not Mined	Mined
Mean	0.103	0.147
Median	0.081	0.120
StdDev	0.094	0.125
Var	0.009	0.016
CV	0.915	0.850

To evaluate the performance of the OK model, the average block model grade was compared with that of the NN grade model for the mined and unmined areas. The results indicate on a global scale, that the underlying data from the mined areas has not biased the results of unmined areas.

Table 17-13: Average grade from the the NN and OK block models

Domain in UDT Zone	Volume	Density	Tonnage	Grade (Au opt)	
	ft**3 x 1000	tons/ft**3	tons x 1000	NN	OK
Not Mined	**25,741.871**	**0.089**	**2,296.175**	**0.108**	**0.107**
Mined Out	57,776.031	0.089	5,153.622	0.160	0.159

17.5.9.5 *Swath Plots*

On a local scale the NN model does not provide reliable estimates of grade but on a larger scale the NN model represents an unbiased estimation of the grade distribution based on the underlying data. If the OK model is unbiased the grade trend may show local fluctuations on the swath plot but overall the trend should be similar to the NN grade distribution. Overall there is good correlation in the 200 ft section swaths and 100 ft plan swaths and good agreement with the ID2 model (Figure 17-9).

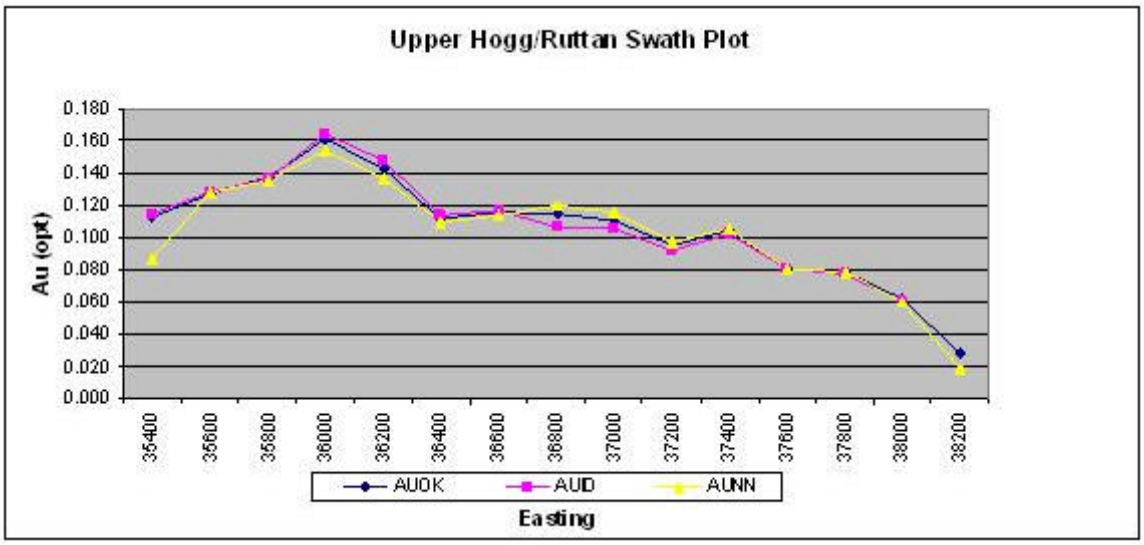

Figure 17-9: Swath plot of sections from the UHR Zone

17.5.10 Results and Classification

For the purposes of tonnage and grade tabulation, blocks were coded as either being resource or not resource. Blocks that are not resource are those that are inside the mined out stopes, below the 1780 Level elevation, above the bedrock

surface, and outside the host rock domain defined by the thrust fault. The coding of the blocks is based on the block being either 50% inside or outside a respective domain. In the area where the UDT and the UHR zones merge, only those blocks from the UHR Zone were tabulated.

Based on the previous mining, the density of drilling, the composite data statistics, and the previous resource estimates the mineralization above the 1780 Level is classified as indicated. A portion of the resource in the upper part of the UHR Zone is classified as inferred as it is not supported by sufficient drilling. The revised estimate of resource completed here that includes the integration of recent drilling and a reevaluation of mineralization at a 0.057 ounce per ton cut off grade is contained in Table 17-14.

Table 17-14: Summary of resources from the Main Mine Upper

Zone	Tonnage (tons)	Grade (Au opt)	Ounces Au
Indicated Resources			
UHR	1,834,701	0.099	180,804
UDT	2,296,175	0.107	246,792
Total Indicated	4,130,876	0.104	427,596
Inferred Resources			
UHR	491,218	0.104	51,180
Total Inferred	491,218	0.104	51,180

No blocks or partial blocks were eliminated from this resource to represent pillars adjacent to the mined out areas. The pillar requirement will be part of the geotechnical and mining component of this study.

17.6 Estimation of No. 3 Zone resources (JL)

17.6.1 Database and Database Validation

The database contains 227 diamond drill holes totaling 211,519 ft for No. 3 Zone (Table 17-15). All the holes have been drilled from surface. The last 3 holes drilled by Garson in 2008 and the 12 holes drilled by Alexis in 2010 are incorporated into the current interpretation of the mineralized zones and resource estimate. The 3 recent holes drilled by Garson were completed in the upper parts of the resource area and were targeted at delineation and expansion of the Footwall Lens. All of the holes drilled by Alexis have targeted the lower part of the No.3 Zone Main Lens with the objective of elevating the inferred resources to the status of indicated.

Table 17-15: Summary of No.3 Zone drilling

Year	Company	Number of Drill Holes	Total Length of Drilling Program	
			feet	metres
2010	Alexis Minerals Corporation	12	18,979	5,785
2007/2008	Garson Gold Corp.	63	102,065	31,110
1994	High River Gold Mines	43	17,281	5,267
1986/1987	Silver Hart Mines	35	22,811	6,953
1985	Hudson Bay Exploration	28	10,346	3,153
1981	Gold Fields Exploration Canada	46	40,037	12,203
Total		**227**	**211,519**	**64,471**

A total of 11,426 samples were taken from the No.3 Zone (Table 17-16). The data base, including all but 3 of the Garson drill holes, was audited and reported in the Micon, 2008 technical report. Approximately 20% of the entries of drill location, down hole survey and assays from the Alexis drilling was checked against original data and assay certificates and no errors were found.

Table 17-16: Summary of No.3 Zone sampling

Year	Company	Number of Samples	Total Length of Sampling	
			feet	metres
2010	Alexis Minerals Corporation	1038	2188	667
2007/2008	Garson Gold Corp.	2946	6,626	2,020
1994	High River Gold Mines	776	1,855	565
1986/1987	Silver Hart Mines	1503	3,765	1,148
1985	Hudson Bay Exploration	876	893	272
1981	Gold Fields Exploration Canada	4323	9,931	3,027
Total		**11,462**	**25,257**	**7,699**

It is concluded that the No.3 Zone drill hole data base is appropriate for the basis of resource estimation.

17.6.2 Gemcom Database

As per the Main Mine, the Gemcom data base consists of the following primary tables and information:

1) Header – Drill hole collar coordinates in the local No.3 Zone grid system.
2) Survey – Down-hole survey data including dip and grid azimuth.
3) Assay – Sample intervals with assayed gold values.
4) Lithology – Rock type intervals re-coded for project consistency.

17.6.3 Gemcom Solids of Previous Mining and Development

The No.3 Zone was mined during parts of 1995 and 1996 from a surface portal down approximately 450 ft vertical. For the purposes of geological interpretation as well removing the mined volume for the resource estimate, the mined out volume was created in Gemcom using surveyed AutoCAD level plans. This was reviewed and reported in the previous No.3 Zone resource report.

Although the shape and volume of the mined stopes and development are considered here to be suitable for the purposes of resource estimation, it is assumed that production engineering will rely on original survey plans.

17.6.4 Geological Model

Mining at the No.3 Zone was focused on a vein structure with an east west strike and a moderate, approximately 45^0 north dip. The structure, termed the Main Lens, was mined over a strike length of approximately 550 ft and a vertical extent of approximately 400 to 450 ft. Historical drilling had traced the 45^0 north dipping structure to a vertical depth of approximately 1100 ft and suggested a NE plunge to the mineralization. The modern phase of exploration at No.3 Zone, initiated in 2007 (see Table 17-15), confirmed the NE plunge of the mineralization and extended it to a vertical depth of approximately 1800 ft. The Main Lens is a very tabular zone consisting of variably foliated, altered and quartz veined mafic volcanic and volcaniclastic rocks.

During the course of mining the Main lens, a separate zone of mineralization was intersected in the footwall of the Main lens and is termed the Footwall lens. In the mined out area, the Footwall lens is sub-parallel to the Main lens. However, with depth, the Footwall lens has been modeled as a discrete unit, divergent at a low angle to the Main lens. The No.3 Zone wireframes representing the Main lens and the Footwall lens are illustrated in Figure 17-10.

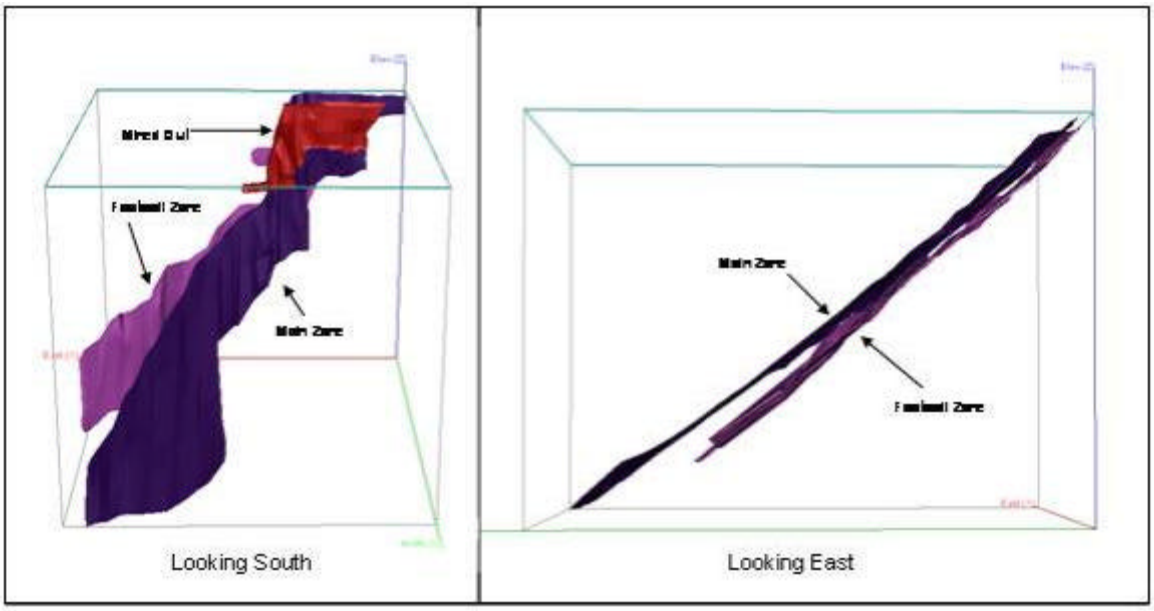

Figure 17-10: Isometric views of the No.3 Zone wireframes for the Main Lens, the Footwall Lens, and the mined out stope

The Main and Footwall Lenses are based on sectional interpretation of mineralized intercepts greater than 0.057 Oz per ton and 6 ft minimum true width. The Main Lens exhibits very good continuity down plunge with only the occasional hole below the cut off grade parameters (Figure 17-11). The Footwall Lens exhibits good continuity in the upper part of the No.3 Zone where it appears parallel to the Main Lens. The continuity becomes more sporadic down plunge on the Footwall Lens where the interpretation has been expanded to include foliated and altered, but weakly mineralized mafic volcanic rocks. This area contains intercepts below the current minimum economic criteria (Figure 17-12).

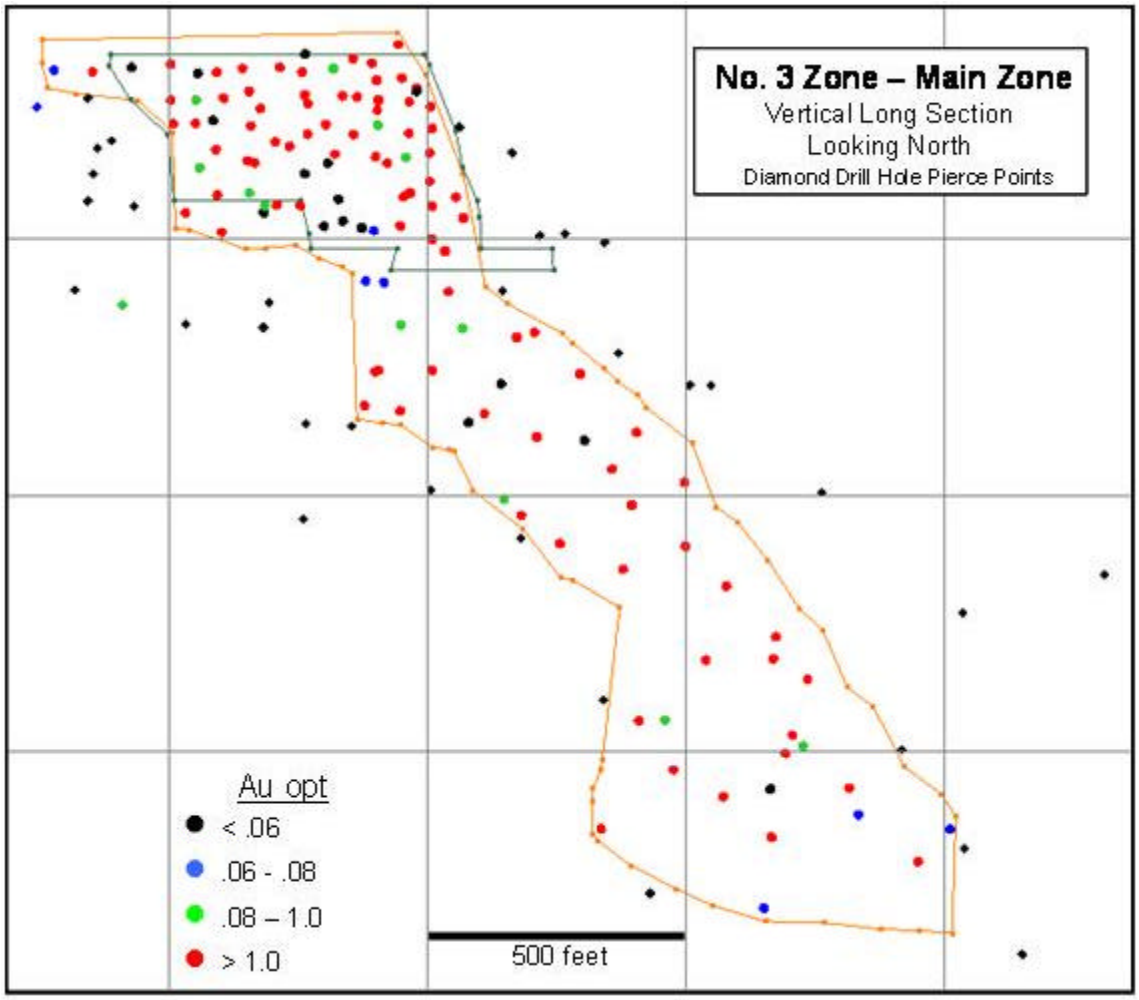

Figure 17-11: Long section of the No.3 Main Lens

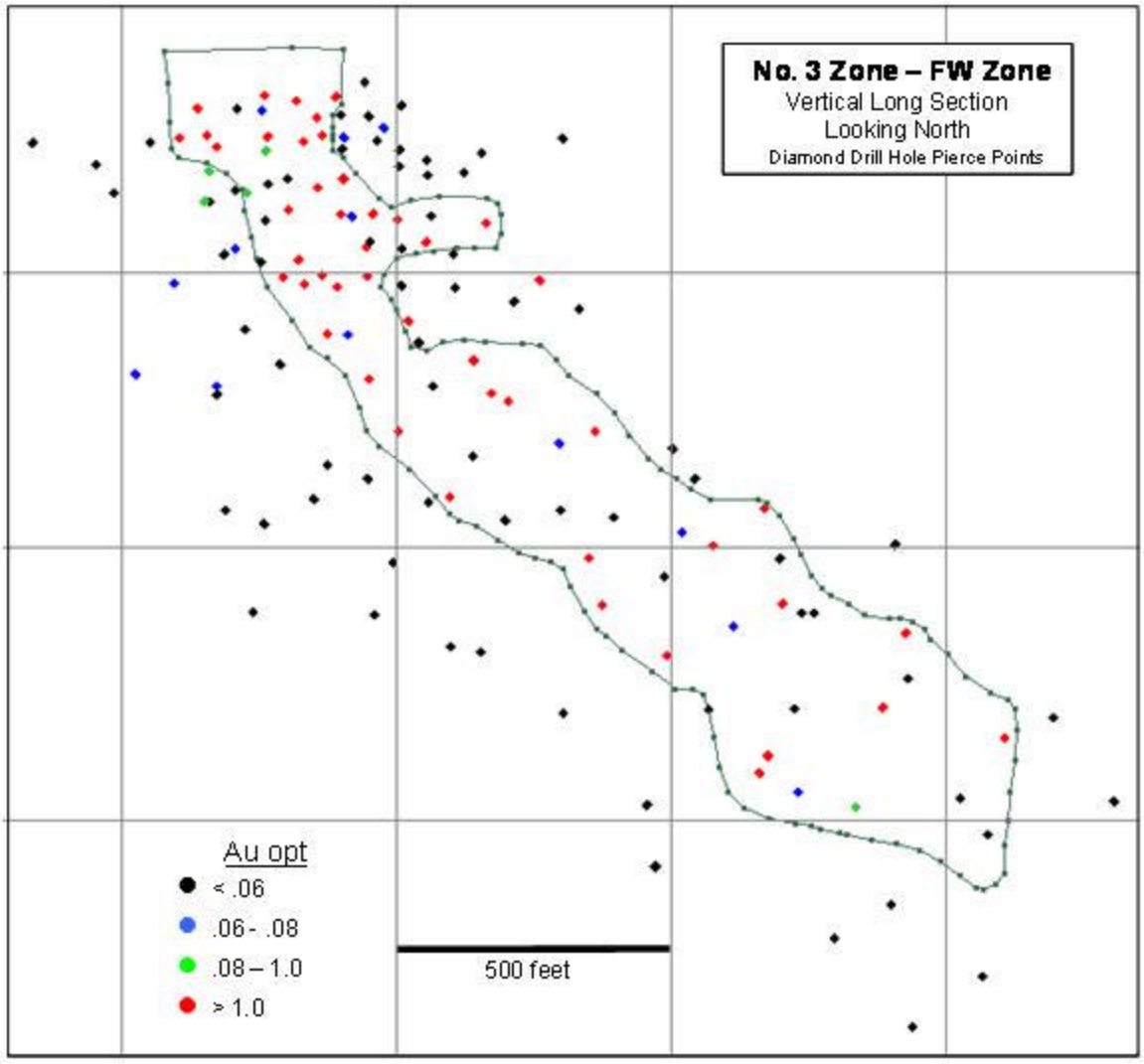

Figure 17-12: Long section of the No.3 Footwall Lens

The interpretation of the Main Lens and Footwall Lens as discrete zones could be related to their origin as fault surfaces during the D1 and D2 episodes of deformation. It is recommended that alternative explanations to establish the relationship between the Main and Footwall lenses are evaluated. This includes continued drilling and interpretation particularly below the mined out stope where there is a contact between low and higher grade mineralization.

17.6.5 Statistics and Geostatistics

17.6.5.1 *Compositing*

Samples selected within the Main Lens and the Footwall Lens wireframes were composited to a standard length for the purposes of variography and grade estimation. The raw sample lengths were evaluated and 3 ft was determined to be an appropriate sample length as 95% of the samples in the Main Lens are 3 ft or less. Compositing was completed from the top (hangngwall) of the wireframe downwards. End of interval composites less than 1.5 ft were added to the up-hole composite and the length weighted average grade calculated and assigned. End of interval composites greater than 1.5 ft were are used on an as-is basis.

17.6.5.2 *Grade Capping Evaluation*

SRK evaluated the composite data for both the Main Lens and the Footwall Lens zones using histograms and probability plots. SRK found clear breaks in the composite grade distributions at 1.15 Oz per ton for the Main Lens and 0.42 Oz per ton for the Footwall Lens (Table 17-17).

Table 17-17: Summary of grade capping, No.3 Zone

Domain	Grade Capping Value (opt)	Number Capped/Total Number in Domain	Percent Equivalent
Main Lens	1.15	4/404	99.01%
Footwall Lens	0.42	13/199	93.47%

The composite data was evaluated by the author and although a top grade of 0.45 Oz per ton may also be appropriate for the Footwall Lens, the author concurs with the choices of top cuts made by SRK (Table 17-17). Summary statistics for uncut and cut gold grades for both the Main Lens and the Footwall Lens are summarized in Table 17-18.

Table 17-18: Summary statistics of composites, No.3 Zone

Statistic	Main Zone		Footwall Zone	
	Au (opt)	AuCut (opt)	Au (opt)	AuCut (opt)
n	404	404	199	199
mean	0.200	0.197	0.221	0.134
std dev	0.212	0.201	0.755	0.130
CV	1.062	1.016	3.417	0.968
min	0.001	0.001	0.001	0.001
Q1	0.07	0.07	0.03	0.03
median	0.15	0.15	0.1	0.1
Q3	0.26	0.26	0.19	0.19
max	1.6	1.6	7.93	0.42

17.6.5.3 *Variography*

Variography on the Main Lens and the Footwall Lens was completed by SRK using the capped composite values. SRK examined 1) traditional semivariograms, 2) traditional correlograms, 3) normal scores semivariograms and 4) normal scores correleograms. SRK noted the limited quantity of composites for No.3 Zone variography and the variograms do not illustrate well defined structures. Figure 17-13 shows the modeled directional variograms for the Main Lens and the Footwall Lens which are summarized in Table 17-19.

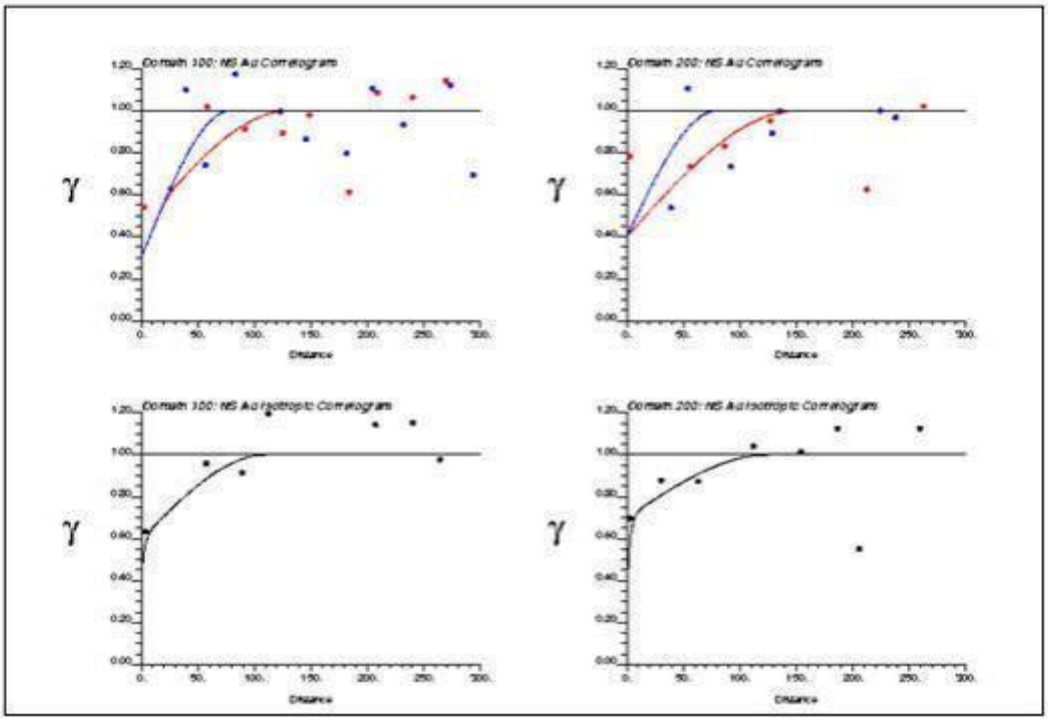

Figure 17-13: Modeled variograms from the No.3 Zone Main Lens (Domain 100) and the No.3 Zone Footwall Lens (Domains 200)

Table 17-19: Variogram model specifications, No.3 Zone

Type	Zone	CO	IST	TYPE	CC	AH_MAX	AH_MIN	A_VERT	ANG1	ANG2	ANG3
Directional	100	0.3	1	Spherical	0.15	30	80	5	30	-30	-30
			2	Spherical	0.55	130	80	5	30	-30	-30
Directional	200	0.4	1	Spherical	0.6	150	80	5	45	-30	-30
Isotropic	100	0.4	1	Exponential	0.2	8	8	5	30	-30	-30
			2	Spherical	0.4	110	110	5	30	-30	-30
Isotropic	200	0.3	1	Exponential	0.4	8	8	5	45	-30	-30
			2	Spherical	0.3	125	125	5	45	-30	-30

Notes: AH_MAX corresponds to direction specified by ANG1 (azimuth) and ANG2 (dip).
AH_MIN corresponds to direction perpendicular to dip and across the vein.
A_VERT corresponds to direction of the width of the vein.
IST indicates the structure number

17.6.6 Block Model Parameters

A block model was created in Gemcom to model the Main Lens and the Footwall Lens. Block size is 10ft x 10ft x 10ft with an origin at 9700E, 11260N, and a maximum elevation of 10,000RL (No.3 Zone grid system). The model extends 1960 ft east, 2450 ft north, and 2000 ft vertical. The block size is similar to the average width of the Main Lens.

17.6.7 Grade Estimation

Given the lack of well defined structure in the variography combined with the observation that the Main Zone is very planar feature with very good continuity and where the majority of intersections are above the minimum grade and width modeling parameters, an inverse distance to the power 2 method was used for block grade estimation. The interpolation parameters are set out in table 17-20.

Table 17-20: Interpolation parameters for block grade estimation using ID2 at the No.3 Zone

Zone	Pass	Search Ellipse Orientation			Search Type			# composites	
		Az	Dip	Az	Type	Min. # Octants	Max. Samples per Octant	Min/Block	Max/Block
Main	1	30	-30	-76	octant	3	3	2	12
Main	2	30	-30	-76	octant	3	3	2	12
Main	3	30	-30	-76	octant	2	3	2	12
Footwall	1	45	-30	-61	ellipsoidal	--	–	4	6
Footwall	2	45	-30	-61	ellipsoidal	--	–	2	6
Footwall	3	45	-30	-61	ellipsoidal	--	–	2	4

Notes: Range for Main Zone search ellipse 1 is: 130 feet in principal Az (down plunge), 80 feet across strike, and 5 feet across the zone
Range for Main Zone Search ellipses 2 and 3 are 2x and 3x range for ellipse 1
range or Footwall Zone search ellipse 1 is: 150 feet in principal Az (down plunge), 80 feet across strike, and 5 feet across the zone
Range for Footwall Zone Search ellipses 2 and 3 are 2x and 3x range for ellipse 1

17.6.8 Tonnage Factor and Tonnage Estimate

The tonnage factor used in the previous No.3 Zone resource estimate (Micon, 2008) was 0.0893 tons per cubic ft. This number is based on the specific gravity determinations of 357 samples. The specific gravity data was evaluated and reported by Micon (2008) and is considered to be reliable for the tonnage factor.

Based on user inputs, Gemcom calculates the fraction (percent) of a given block within the mineralization wireframe. The tonnage of mineralization in a given block is determined by:

block volume * per cent inside mineralization wireframe * 0.0893

17.6.9 Validation

The results of the modeling process have been evaluated through:

1) A visual review of the results;
2) Block model and wireframe volume comparisons;
3) Comparison with other in interpolation methods;
4) Comparison of the mined out and not mined out parts of the model;
5) Swath plots.

17.6.9.1 *Visual Inspection*

Detailed visual inspection of the model has been completed in cross section and in long section. The proper coding of blocks by zone and distribution within and outside of mined out volumes was confirmed. The distribution of block grades was compared relative to composite point data and relative to

both composite and raw assay data plotted on drill hole traces to ensure the proper representation of the underlying data in the model. A block model long section was created with a single row of blocks across (horizontal) the deposit with weighted average grade. The distribution of block grade in long section is consistent with the distribution of intercept grades (Figures 17-14 and 17-15).

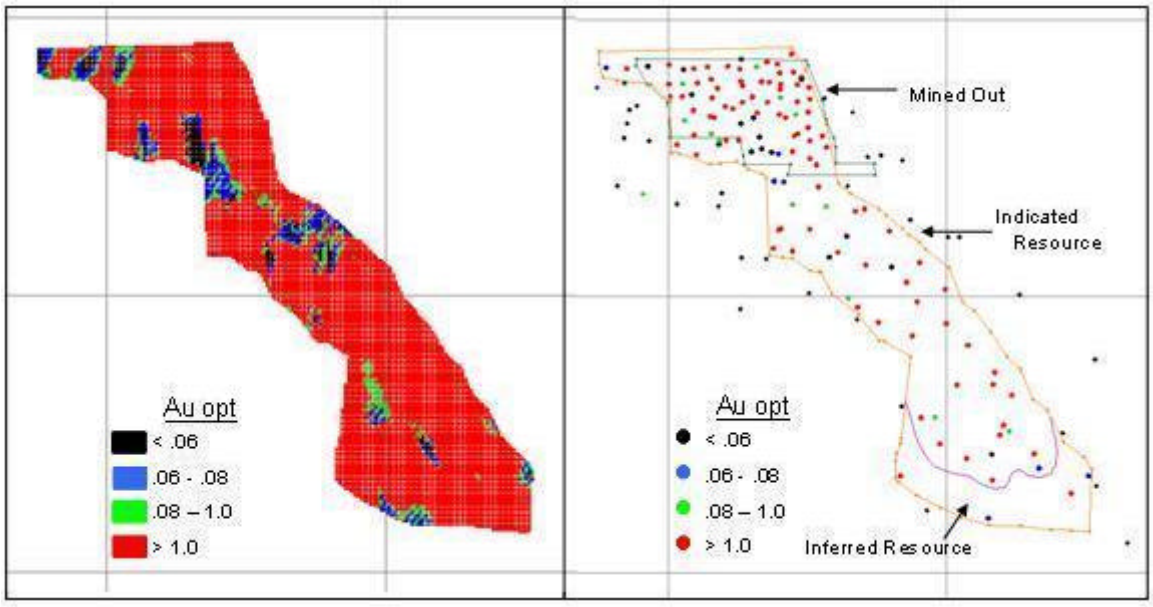

Figure 17-14: No.3 Zone block model and outline with diamond drill hole pierce points, mined out stope and indicated and inferred resources

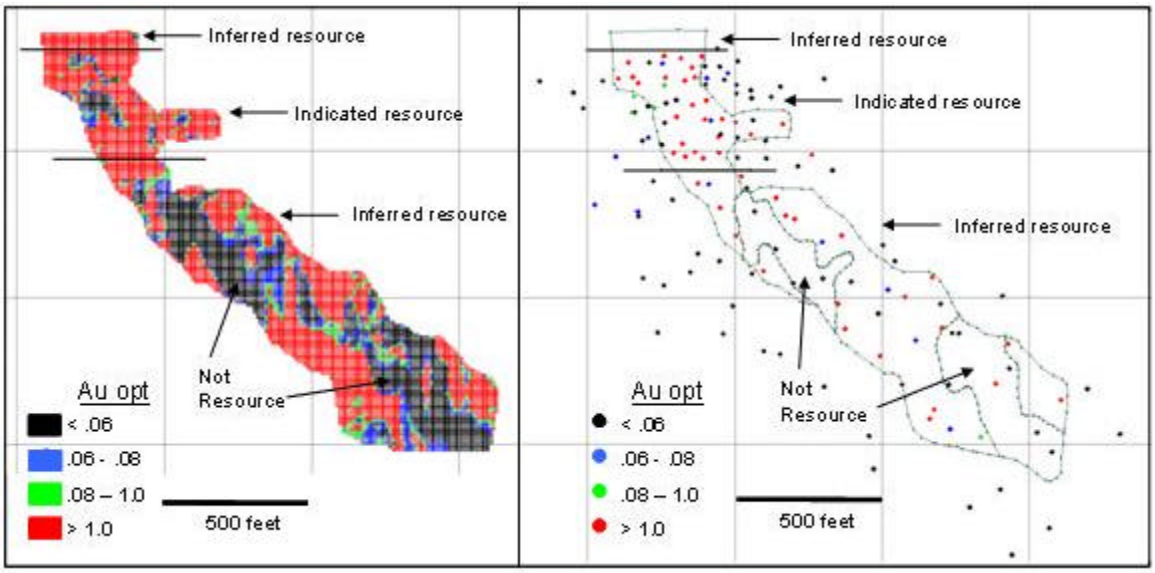

Figure 17-15: No.3 Footwall lens block model and outline with diamond drill hole intercepts and indicated and inferred resource

17.6.9.2 *Block model and wireframe volume comparison*

The total block model volume is confirmed with the volume of the underlying wireframes. The comparison of the volumes is contained in Table 17-21.

Table 17-21: Comparison of block model and wireframe volumes

	Volume (ft3)		Percent
Zone	Wireframe	Block Model	Difference
Main	11,724,884	11,724,305	0.005
Footwall	6,592,356	6,578,787	0.206

17.6.9.3 *Comparison of other interpolation methods*

For comparison purposes, grade block models were generated using nearest neighbor and ordinary kriging methods. The ordinary kriging parameters used are reported by SRK. The nearest neighbour model is interpreted as the declustered representation of the underlying composite data and similarity with the estimate is interpreted as a measure of the ID2 estimate being globally unbiased. The ordinary kriging model has a 3.6% higher average grade, this is interpreted as a reasonable replication of the global average grade (Table 17-22).

Table 17-22: Average model grades with NN, ID2, and OK methods for the No.3 Main Lens

Method	Volume (ft3)	Density	Tonnage	Grade (Au opt)
ID2	11,724.305	0.089	1,045.808	0.191
NN	11,724.305	0.089	1,045.808	0.188
OK	11,724.305	0.089	1,045.808	0.197

17.6.9.4 *Comparison of the mined un mined portions*

A number of declustering and search strategies were tested to ensure that the composite data from the mined area did not unduly affect the grade distribution of the model in the unmined resource of the Main Lens. As a test, the ID2 model is compared to the nearest neighbor model in the three categories of the Main Lens; the mined out volume, the indicated resource, and the inferred resource (Table 17-23).

Table 17-23: Average grades for the mined, indicated, and inferred resource categories in the No.3 Main Lens

Category	Volume	Density	Tonnage	Grade (Au opt)	
	ft**3 x 1000	tons/ft**3	tons x 1000	NN	ID2
Indicated Resource	7,489.152	0.089	668.032	0.186	0.188
Inferred Resource	1,711.952	0.089	152.706	0.166	0.179
Mined	2,523.201	0.089	225.070	0.207	0.210
Total	11,724.305	0.089	1,045.808	0.188	0.191

17.6.9.5 *Swath Plots*

On a local scale the NN model does not provide reliable estimates of grade but on a much larger scale it represents an unbiased estimation of the grade distribution based on the underlying data. If the ID2 model is unbiased the grade trend may show local fluctuations on the swath plot but overall the trend should be similar to the NN grade distribution. Overall there is good correlation in the 200 ft section swaths and 100 ft plan swaths for the Main Lens. In both the sectional and plan swath plots, the ID2 model tracks the NN model closer than the OK model (Figure 17-16).

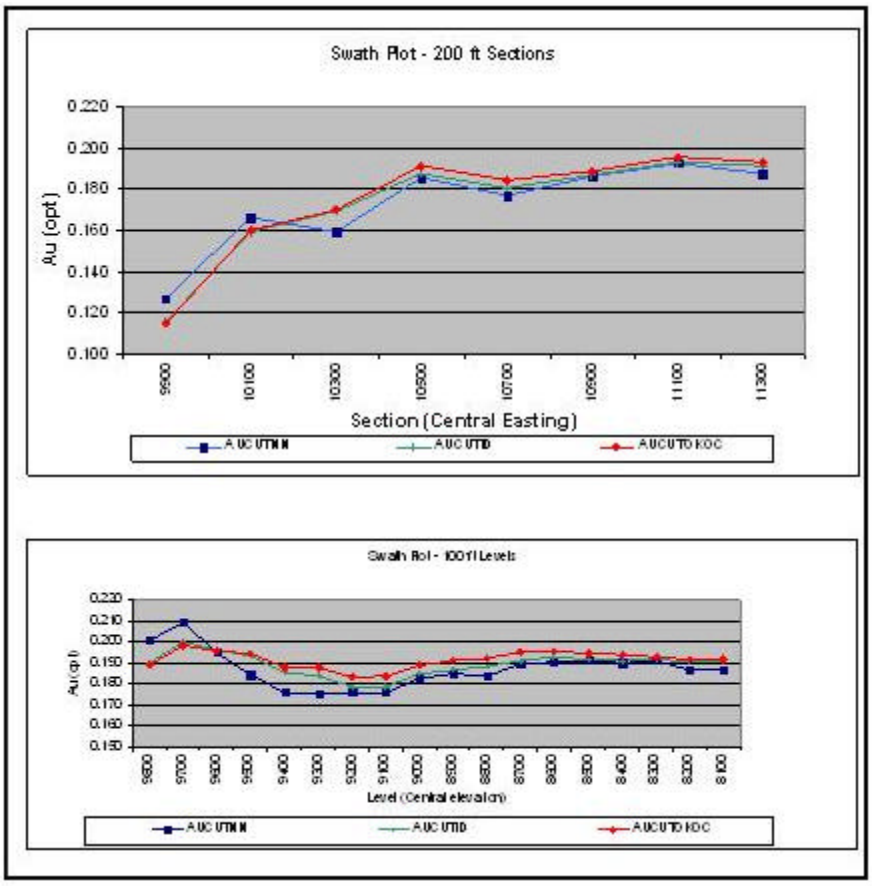

Figure 17-16: Swath plots for the No.3 Main Lens

17.6.10 Results and Classification

For the purposes of model tonnage and grade tabulations, the blocks in each of the Main Lens and the Footwall Lens were coded as either indicated resource, inferred resource, or mined. The coding of the blocks is based on the block being 50% inside or outside of the respective domain.

Blocks in the Main Lens are classified as either indicated, inferred, or mined (Figure 17.11a). The Main Lens is mostly classified as an indicated resource with the down dip extension as inferred. There are two drill holes below the inferred resource that contain the Main Lens structure with gold intercepts slightly less than the minimum grade and thickness parameters.

Blocks in the Footwall Lens are classified as either indicated, inferred, or mined (Figure 17.14). In addition, blocks forming distinct and continuous domains of low grade mineralization have been removed from the resource (Figure 17.15). The inferred resource is restricted to areas with a greater drill density and where the Footwall Lens forms a continuous domain sub-parallel to the Main Lens.

The resources of the No.3 Zone include the recent drilling and the revised economic parameters are contained in Table 17.24.

Table 17-24: No.3 Zone resources

Zone	Tonnage (tons)	Grade (Au opt)	Ounces Au
Indicated Resources			
Main	668,032	0.188	125,590
Footwall	82,412	0.166	13,680
Total Indicated	750,444	0.186	139,270
Inferred Resources			
Main	**152,706**	**0.179**	**27,334**
Footwall	**319,328**	**0.128**	**40,874**
Total Inferred	472,034	0.144	68,208

The wireframe of the mined stope was extended 15 ft into the footwall and hangingwall. Blocks in the Footwall Lens, outside of the mined stope have been included as a resource. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

17.7 Resources Main Mine Lower (JL)

Main Mine lower resources, from the Dick and the Ruttan Zones, were estimated by Kinross Gold Corp. and were audited and reported in the NI-43-101 technical report by Micon International Ltd. dated October, 2006. The methodology and results of the resource models have been verified here.

17.7.1 Methodology

The resource estimate for the Dick and Ruttan Zones followed a conventional and industry standard approach and has been reported in the technical report by Micon International Ltd. dated October, 2006. The approach and general methods of resource estimation of the Dick and Ruttan Zones are summarized as follows:

1) Sectional interpretation of mineralization zones at a cut-off grade of 0.097 Oz per ton and a minimum true width of 7 ft.
2) All assays greater than 0.50 Oz per ton were cut to 0.500 Oz per ton.
3) The sectional interpretation of the mineralization was digitized and wireframed, using Gemcom Software.
4) The assay data was composited into 3 ft intervals and basic statistics were completed on the assay and composited assay data.
5) Geostatistical analysis was completed and variogram models established.
6) A block model with dimensions 15 ft x 15 ft x 15 ft was used for grade estimation.

7) Grade interpolation was completed using OK using a number of interpolation passes.

8) Validation of the block model using graphical and statistical techniques.

9) Tonnage was established using the percent model utility in Gemcom and tabulated by rock type using the percentage of the block within the mineralized wireframe.

10) The blocks were composited, using Gemcom, into 50 ft Levels and 50 ft sections for tabulation and reporting of tons and grade.

11) Tabulation and reporting of the resources was completed at US$ 400, US$ 450, and US$ 500 gold prices corresponding to cut-off grades of 0.097 Oz per ton, 0.086 Oz per ton, and 0.077 Oz per ton.

17.7.2 Verification

The composite files and the various profiles used for the resource estimation including kriging profiles, search parameters, etc. and the tables of composite data have not been located. It appears that the geostatistical grade estimates were competed in a separate Gemcom project and the resulting block models were imported back into the respective Gemcom projects. The verification of the resource estimates is limited to:

1) Visual inspection of the block model grades and comparison with assay data plotted on diamond drill hole traces, and;

2) Tabulation of selected regions of the Dick and Ruttan models and comparison with the grade and tonnage tabulated in the Kinross resource report dated December 31, 2004.

Several cross sections from the Ruttan and the Dick EW Zones were evaluated where the block model grades were plotted with the drill hole traces displaying assay grades. In all cases, the block model grades were consistent with the assay data. The Ruttan Zone and Dick EW block models were color coded to display blocks above and below the cut off grade. The grade distribution in plan view is consistent with the block grade distributions and the mineralization outlines on long sections.

Selected regions of the block models from the Ruttan Zone and the Dick EW Zone were tabulated for both tons and grade. The tabulations were compared with the long sections and level plans from the Kinross resource block model. The tabulations in the Kinross resource are at a 0.097 ounce per ton cut-off.

The review of the reports and the two procedures verify that the Kinross resource estimate, (audited and reported in the Micon International Ltd. NI-43-101 technical report dated October 2006), is a consistent representation of the drill data. The geological model or wireframes have not been evaluated in detail. Given that the Gemcom profiles,estimation parameters and the tables of composite data have not been recovered it is recommended that Alexis should re-estimate the resource and perform a complete verification.

17.8 Updated resources for the Snow Lake Mine property (JL)

Table 17.25 contains the updated resource estimate for the Snow Lake property. This includes the updated resources estimates for the No.3 Zone and the Main Mine Upper. The tabulation for the Main Mine Lower is derived from the Micon International Ltd. NI 43-101 technical report dated October 2006 using a 0.077 Oz/ton cut off. The resource estimates for the Squall Zone, Upper and Lower Margaret, are taken from the NI-101 technical report by D. Beilhartz April 2006. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 17-25: Updated gold resource estimate for the Snow Lake property

	Resource Category	Tonnes	Grade (g/t)	Ounces Au
Main Mine Lower1	Measured	7,000	4.76	1,000
Main Mine Upper2	Indicated	3,748,000	3.57	427,600
Main Mine Lower1	Indicated	1,035,000	4.81	160,100
No. 3 Zone - Main2	Indicated	606,000	6.44	125,600
No. 3 Zone - Footwall2	Indicated	75,000	5.69	13,700
Total Property Measured and Indicated Resources		5, 471,000	4.14	728,000
Main Mine Upper2	Inferred	446,000	3.57	51,200
Main Mine Lower1	Inferred	486,000	3.98	62,200
No. 3 Zone - Main2	Inferred	139,000	6.14	27,300
No. 3 Zone - Footwall2	Inferred	290,000	4.39	40,900
Birch1	Inferred	569,000	4.42	81,000
Squall Margaret Upper3	Inferred	100,000	4.87	15,600
Squall Margaret Lower3	Inferred	337,000	5.42	58,500
Total Snow Lake Mine Property Inferred Resources		2, 367,000	4.43	336,700

1) NI43-101 Technical report on the New Britannia Mine Property and Review of the Mineral Resource Estimate, Snow Lake Manitoba By Wiliam J. Lewis, P. Geo and Richard Gowans, P. Eng., Micon International Ltd, October 2006. Cut-off grade 0,077 opt
2) NI43-101 Technical report on the Snow Lake Mine Property – Main Mine and No. 3 Zone, Snow Lake Manitoba By Jamie Lavigne, P. Geo, VP Exploration, Alexis Minerals, December 10, 2010. Cut-off grade 0,057 opt.
3) NI 43-101 Technical Report on the Squall Lake Property, The Pas Mining Division Snow Lake Manitoba. Completed for Garson Resources Ltd. Bye D. Beilhartz, P. Geo., April 2006.

17.9 Mineral reserve estimate (AR)

The following sections describe the process by which Mineral Reserves were estimated from the Mineral Resources estimates discussed in the earlier part of this Section.

The first step of the process consisted of calculating cut-off grades for the 2 mining methods assuming a gold price of US$ 1,000/Oz. The mining methods are Longhole mining in the Main Mine Upper and No. 3 Zone and Mechanised Room and Pillar in the Main Mine Lower.

Table 17-26 shows the direct mining costs for each mining method:

Table 17-26: Details of Direct Mining Costs Estimate

Mining Operation	Longhole (CAN$/t)	Mechanised Room & Pillar (CAN$/t)
Definition Drilling	4.05	4.05
Operating Development	13.80	13.80
Drilling	9.00	n / a
Blasting	2.50	n / a
Mucking	8.00	n / a
Stoping	n / a	40.00
Rock Movement	3.86	9.30
Mine Services	22.35	22.35
Capital Dev. Allocation	-2.42	-2.42
Pre-pro Transfer to Capital	-4.28	-4.28
TOTAL	**56.86**	**82.80**

Table 17-27 shows the Cut-Off grade calculations.

Table 17-27: Cut-Off Grades Calculations

	Longhole	Mechanised Room & Pillar
Mining	56.86 CAN$/t	82.80 CAN$/t
Milling	12.20 CAN$/t	12.20 CAN$/t
General & Administration	7.65 CAN$/t	7.65 CAN$/t
Total Cost	76.71 CAN$/t	102.65 CAN$/t
Gold Price	US$ 1,000/Oz	US$ 1,000/Oz
Exchange Rate	1.05 CAN$/US$	1.05 CAN$/US$
Mill Recovery	93%	93%
Royalty	1.38%	1.38%
Cut-Off Grade	**2.48 g/t**	**3.32 g/t**

17.9.1 Main Mine Upper

The Reserve Estimation process was slightly different for the East and the West areas of the Main Mine Upper. The process for the East of the Main Mine Upper will be described first.

Main Mine Upper – East Side

A mining engineer defined 21 stope boundaries on a longitudinal section of the geological resource. The stope boundaries included all of the blocks with a grade higher than 0.100 Oz/ton and a fringe of blocks with a grade between 0.080 Oz/ton to 0.100 Oz/ton (the fringe was one bloc wide). The geology department then calculated the resources, dilution, tonnage, grade and the contained gold for each stope. The dilution corresponds to 5 ft of overbreak into the hangingwall.

The resource and dilution parameters (tonnage, grade and gold content) were recalculated to account for the presence of pillars.

The tonnage of any development ore was added to the resource tonnage.

Finally the resource and dilution tonnages and gold content were combined to determine the average grade for each stope.

Main Mine Upper – West Side

In the case of the West side the mining engineer defined stope boundaries that did not include any boundary pillars. This allowed the geologist to calculate the resource and dilution tonnages, grade and contained gold for 20 blocks.

The mining engineer then calculated the parameters for any sub blocks according to the stope designs.

The tonnage of any development ore was added to the resource tonnages. The resource and dilution tonnages and gold content were combined to calculate the average grade for each stope.

Table **17-28** shows the final reserve estimate by stope with a 0.070 Oz/ton cut off.

Table 17-28: Main Mine Upper Stope Reserves

	Au Oz/t	Tons	Au Oz
0320-00345	0,092	26 213	2 408
0320-00348	0,118	62 600	7 414
0320-00358	0,165	47 483	7 832
0580-00341	0,094	22 800	2 145
0580-00349	0,099	69 782	6 939
0580-00360	0,108	7 315	788
0780-00341	0,098	17 969	1 759
1030-00342	0,116	45 580	5 287
1030-00346	0,097	47 201	4 583
1030-00363	0,126	11 434	1 436
1280-00345	0,120	129 892	15 621
1280-00349	0,096	114 989	11 082
1280-00359	0,104	39 628	4 138
1280-00362	0,159	28 614	4 546
1280-00367	0,105	103 528	10 869
1280-00370	0,079	31 785	2 507
1280-00377	0,096	160 716	15 500
1530-00347	0,089	53 304	4 736
1530-00351	0,106	23 761	2 521
1530-00359	0,111	74 063	8 248
1530-00366	0,104	107 668	11 227
1530-00371	0,085	61 202	5 209
1530-00377	0,093	81 919	7 655
1530-00380	0,073	8 265	601
1780-00348	0,128	74 890	9 569
1780-00352	0,104	49 818	5 175
1780-00358	0,098	104 047	10 222
1780-00365	0,080	29 385	2 348
1780-00370	0,101	212 812	21 496
1780-00374	0,101	69 498	6 985
1780-00376	0,087	42 397	3 691
1780-00377	0,109	42 343	4 629
Total	0,104	2 002 901	209 168

17.9.2 Main Mine Lower

In the case of the Main Mine Lower the resource blocks (50' x 50' or 50' x 100') were combined into 200' x 200' stopes and 25% dilution was added to the resource tonnages and the grades were recalculated. This procedure is different to the reserve estimate in the Main Mine Upper and is justified by the change of the mining method to Mechanised Room and Pillar.

The reserve estimate was then adjusted to account for a 15% reduction in recovery due to pillar requirements and loss in the stope floors. The final results are presented in Table 17-29.

Table 17-29: Main Mine Lower Stope Reserves

	Au Oz/t	Tons	Au Oz
BLOC D-EW-01	0,130	17 711	2 306
BLOC D-EW-02	0,158	29 996	4 727
BLOC D-EW-03	0,155	49 472	7 672
BLOC D-EW-04	0,000	0	0
BLOC D-EW-05	0,000	0	0
BLOC D-EW-06	0,127	13 107	1 664
BLOC D-EW-07	0,157	51 211	8 065
BLOC D-EW-08	0,127	56 127	7 101
BLOC D-EW-09	0,000	0	0
BLOC D-EW-10	0,000	0	0
BLOC D-EW-11	0,150	6 613	995
BLOC D-EW-12	0,134	41 183	5 523
BLOC D-EW-13	0,123	48 186	5 946
BLOC D-EW-14	0,000	0	0
BLOC D-EW-15	0,000	0	0
BLOC D-EW-16	0,172	16 801	2 884
BLOC D-EW-17	0,185	35 036	6 488
BLOC D-EW-18	0,153	26 095	4 003
BLOC D-EW-19	0,000	0	0
BLOC D-EW-20	0,102	8 849	906
BLOC D-EW-21	0,152	37 654	5 712
BLOC D-EW-22	0,121	38 343	4 649
BLOC D-EW-23	0,173	11 629	2 009
BLOC R-EW-01	0,000	0	0
BLOC R-EW-02	0,123	15 489	1 902
BLOC R-EW-03	0,172	4 083	701
BLOC R-EW-04	0,118	8 290	977
BLOC R-EW-05	0,135	12 939	1 746

BLOC R-EW-06	0,153	10 396	1 586
BLOC R-EW-07	0,149	21 958	3 272
BLOC R-EW-08	0,109	3 047	331
BLOC R-EW-09	0,145	6 948	1 010
BLOC R-EW-10	0,148	6 489	963
BLOC R-EW-11	0,139	17 083	2 379
BLOC R-EW-12	0,139	16 465	2 282
BLOC R-EW-13	0,132	3 864	511
BLOC R-EW-14	0,156	6 554	1 022
BLOC R-EW-15	0,120	7 903	947
BLOC R-EW-16	0,107	24 708	2 655
BLOC D-NW-01	0,000	0	0
BLOC D-NW-02	0,119	42 328	5 052
BLOC D-NW-03	0,125	10 742	1 339
BLOC D-NW-04	0,000	0	0
BLOC D-NW-05	0,108	39 358	4 244
BLOC D-NW-06	0,128	16 498	2 108
BLOC D-NW-07	0,215	3 564	766
BLOC D-NW-08	0,165	9 055	1 492
BLOC D-NW-09	0,000	0	0
BLOC D-NW-10	0,232	216	50
BLOC D-NW-11	0,169	7 498	1 270
BLOC D-NW-12	0,000	0	0
BLOC D-NW-13	0,122	17 093	2 086
BLOC D-NW-14	0,121	19 548	2 368
BLOC D-NW-15	0,000	0	0
Total	**0,139**	**820 127**	**113 710**

Note : The extra development corresponds to the development which passes in a block which is below the stope cut-off grade

17.9.3 No. 3 Zone

For the Main Lens and the Footwall Lens of the No. 3 Zone, the corresponding pillars tonnage was removed from indicated resource. A 25% dilution allowance was added with a grade of 0.013 Oz/ton and a 95% recovery factor was apllied. The final results is presented in Table 17-30.

Table 17-30: No 3 Zone Level by Level Reserves

Level	Au Oz/t	Model Tons	Au Oz
0065 Level	0,17	3 202	544
0130 Level	0,17	14 995	2 474
0195 Level	0,15	16 646	2 442
0260 Level	0,09	13 800	1 234
0325 Level	0,11	30 104	3 301
0390 Level	0,13	45 575	5 794
0415 Level	0,11	15 768	1 790
0425 Level	0,13	10 526	1 320
540 Level	0,09	28 918	2 697
590 Level	0,12	23 646	2 915
640 Level	0,16	29 503	4 601
690 Level	0,13	32 035	4 224
740 Level	0,12	37 102	4 300
790 Level	0,12	35 943	4 470
840 Level	0,17	32 553	5 405
890 Level	0,19	30 300	5 857
940 Level	0,19	3 034	583
990 Level	0,18	20 467	3 596
1040 Level	0,16	25 739	4 186
1090 Level	0,17	22 467	3 769
1140 Level	0,17	19 767	3 417
1190 Level	0,18	23 446	4 247
1240 Level	0,19	26 516	5 069
1290 Level	0,18	28 183	5 087
1340 Level	0,16	3 384	555
1390 Level	0,15	25 955	3 785
1440 Level	0,14	34 656	4 889
1490 Level	0,16	32 265	5 037
1540 Level	0,17	26 483	4 436
1590 Level	0,23	6 437	1 449
Total	0,148	699 418	103 475

18. OTHER RELEVANT DATA AND INFORMATION

The Snow Lake Mine Re-activation falls within the "Development Properties and Production Properties" category of National Instrument 43-101. This section of the report complies with Item 25 – Additional Requirements for Technical Report on Development Properties and Production Properties of the National Instrument 43-101 – Standards of Disclosure for Mineral Projects.

Note: all references to production rates are metric unless otherwise specified.

18.1 Snow lake mine re-activation project overview (AR)

Production areas

The Snow Lake Mine Re-activation calls for producing a nominal 2,000 t/day (700,000 t/year) of ore from two deposits that have been mined in the past, namely:

- Main Mine;
- No. 3 Zone.

The Main Mine is divided into two areas; Upper (Figure 18-1) and Lower (Figure 18-2) The Main Mine is shaft access and the No. 3 Zone (Figure 18-3) is ramp access. Main Mine Upper refers to the resources above 1780 Level (i.e. 1780 ft below surface) and Main Mine Lower refers to the resources that are below 1780 Level. Main Mine Lower resources are located below the shaft bottom that extends from the 1780 Level to the bottom of the mine, (3725 Level).

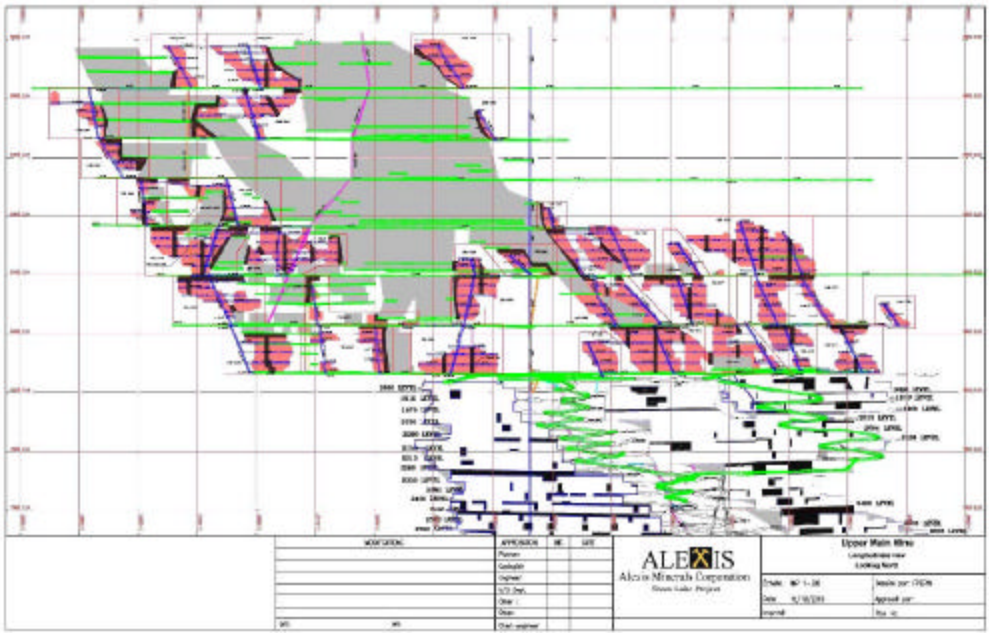

Figure 18-1: Main Mine Upper Long Section

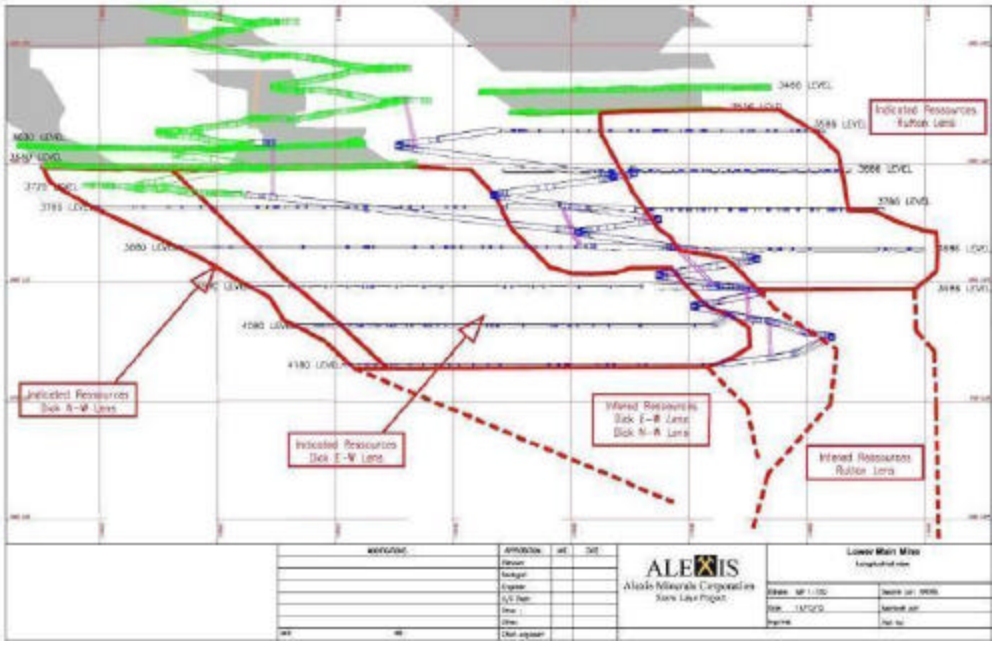

Figure 18-2: Main Mine Lower Long Section

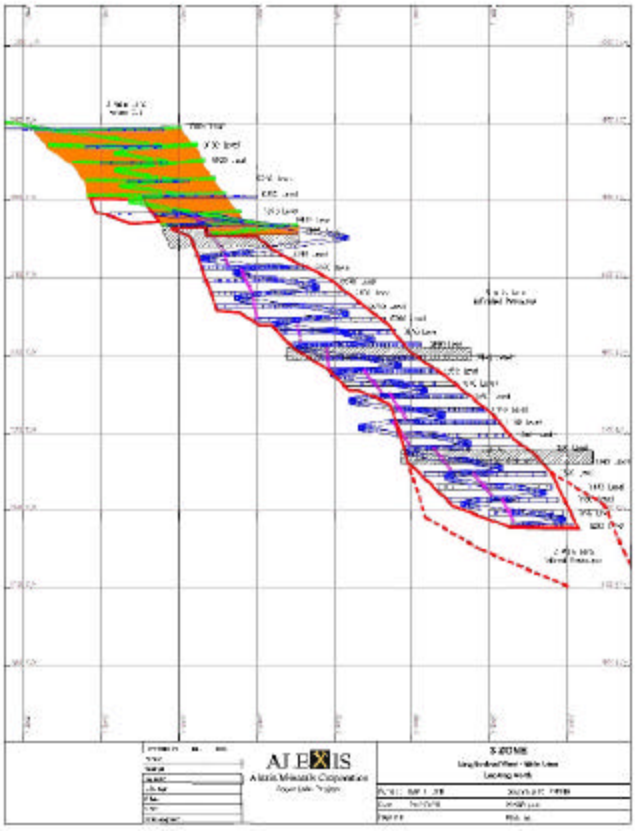

Figure 18-3: No. 3 Zone Long Section (Main Lens)

The mining blocks in the Main Mine Upper are located east of Upper Hogg-Ruttan Zones and between existing stopes in the Upper Dick-Toots Zones. Where necessary, boundary pillars will be left in place in between old and new stopes. The existing Surface Pillar will be left in place.

The Main Mine Lower resources that remained at the conclusion of mining in 2005 in the Dick East-West, Dick North-South and the Ruttan East-West Zones have been planned to be mined in this study. It should be noted that this section of the mine is located below the bottom of the shaft and will be accessed by a ramp, (Figure 18-4). It should be noted that production from the Main Mine Lower is scheduled to come on stream during year 3 to gradually replace the declining production from No. 3 Zone.

The No. 3 Zone ramp is located about 6,000 ft to the North-West of the Main Mine shaft, (Figure 18-5). The No. 3 Zone is known to extend from the surface to 1590 ft. It has been mined out to the 425 Level, (425 ft below surface). The existing surface crown pillar will be left in place. The main characteristics of each area are given in Table 18-1.

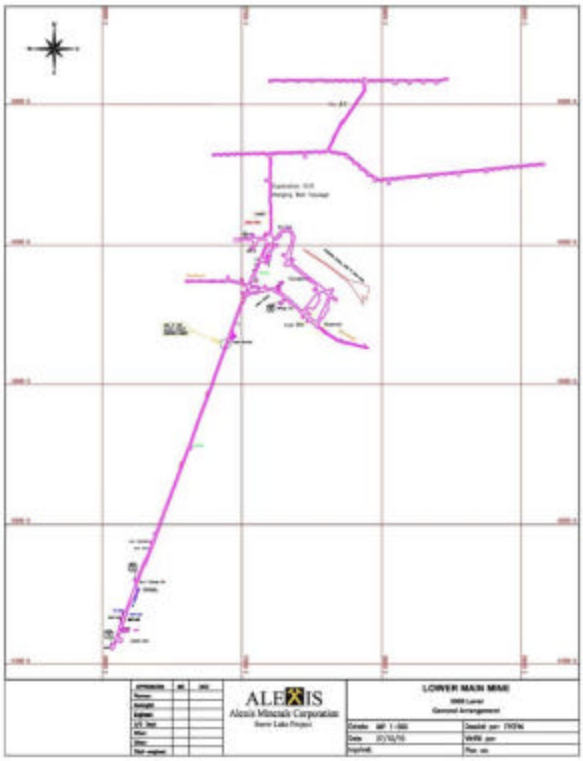

Figure 18-4: Main Mine Lower 3000 Level Plan

Table 18-1 : Main characteristics of the production areas

	MAIN MINE UPPER	MAIN MINE LOWER	No. 3 ZONE
Orientation (Strike)	West - East	West - East	West - East
Horizontal Extent	3,700 ft	2,000 ft	500 ft
Dip	45° North	30° North	45° North
Vertical Extent	1,700 ft	500 ft	1500 ft
Average Thickness	9 ft	9 ft	9 ft
Past Mining Method	Sub level small diameter longholes not backfilled	Upper small diameter blast hole retreat not backfilled	Upper small diameter blast hole retreat not backfilled
Planned Mining Method	Sub level small diameter long holes not backfilled	Mechanised Room and Pillar not backfilled	Upper small diameter blast hole retreat not backfilled
Nominal Daily Production Rate	1,250 t/day	750 t/day[1]	750 t/day[1]

Note 1: Production from the Main Mine Lower is scheduled to come on stream as production declines at No. 3 Zone to maintain the nominal production rate of 2,000 t/day. In year 5 it will increase to 1,800 t/day to compensate the dealining production rate in the Main Mine Upper.

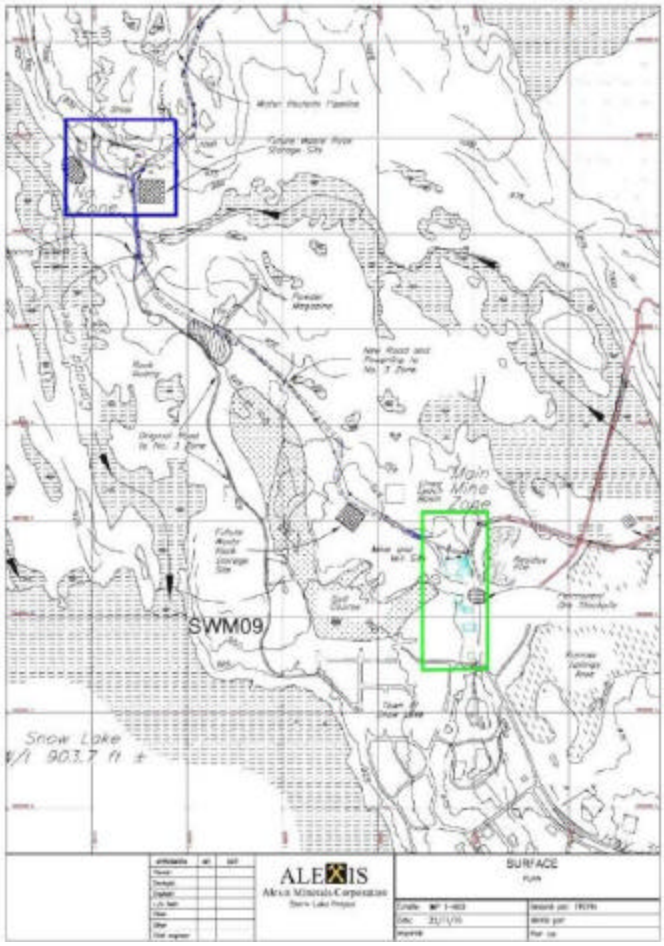

Figure 18-5: Snow Lake Mine Surface Plan

Infrastructure

The ore produced from all three production areas will be milled in the existing 2,150 t/day mill. The existing infrastructure, including but not limited to the mill, headframe, hoistroom and hoists will be refurbished as necessary to put them back in working order. The underground openings including drifts, ramps and raises (ventilation raises and/or escape ways) are not expected to require significant refurbishment. The ground conditions are known to be good in the mine and this was confirmed following an inspection of the 320 Level.

Mine Plan & Scheduling

A master development and production schedule provides details of the ore, from development and production, and waste tonnages on a month by month basis for the first 4 years of the project, i.e. year 0, 1, 2, and 3, and annually thereafter for the Main Mine Upper, the Main Mine Lower and the No. 3 Zone. The development footages for each heading are scheduled on a month by month basis (please see appendicies A, B and C). It should be noted that in order to be slightly conservative the theoretical raise and sublevel quantities were increased by 5%. This data is presented on the detailed production schedules for the Main Mine Upper, Main Mine Lower and No. 3 Zone.

Human Resources / Manpower

Work will occur on 10 hours shifts with 2 shifts per day, leaving 2 hours between shifts to ventilate any blast fumes. The regular work schedule will be on a 3 weeks cycle, termed 7 / 6 / 5 / 3, this means:

- 7 days at work followed by
- 6 days off work followed by
- 5 days at work followed by
- 3 days off work.

This schedule allows for 3 work teams identified as A, B and C. By starting the cycle on Monday, the working shifts are arranged so that 2 shifts per day are worked five days per week and only 1 day shift is worked on Friday and Saturday. This yields 12 working shifts per week, or 36 shifts per 3 weeks cycle, or an average of 52 work shifts per month. It is likely that contractor's personnel will work on other schedules.

Alexis is planning to employ Snow Lake based personnel as much as possible, however, since the available manpower in Snow Lake is limited, Alexis has planned to use remote personnel on fly in – fly out basis. To address the limited accommodation available in Snow Lake, the study allows for a 50 person camp to be built at the Snow Lake Mine site. This camp will include a kitchen and sanitary installations (duly permitted by the regulating authorities).

Project Phases

The Snow Lake Mine Re-activation will consist of three broad phases:

- Reconditioning and Construction of Mine Infrastructure;
- Refurbishment of the Mill ; and
- Development and Production.

18.2 Reconditioning and construction of mine infrastruture (AR)

The cost estimates to recondition the existing Main Mine infrastructure and construct new mine infrastructure required at the No. 3 Zone were provided by Contractors in August and September, 2010 through a competitive bid process.

The Call for Tender broke down the Scope of Work of the Reconditioning Contract into 3 areas:

- Main Mine Surface and Shaft Reconditioning;
- Main Mine Underground Facilities Reconditioning; and
- No. 3 Zone Ramp Dewatering, Reconditioning.

18.2.1 Main Mine Surface and Shaft Reconditioning

The surface infrastructure located in the Main Mine Area is shown in Figure 18-6. The infrastructure has been well maintained and will require minimal refurbishment to put them back in working order.

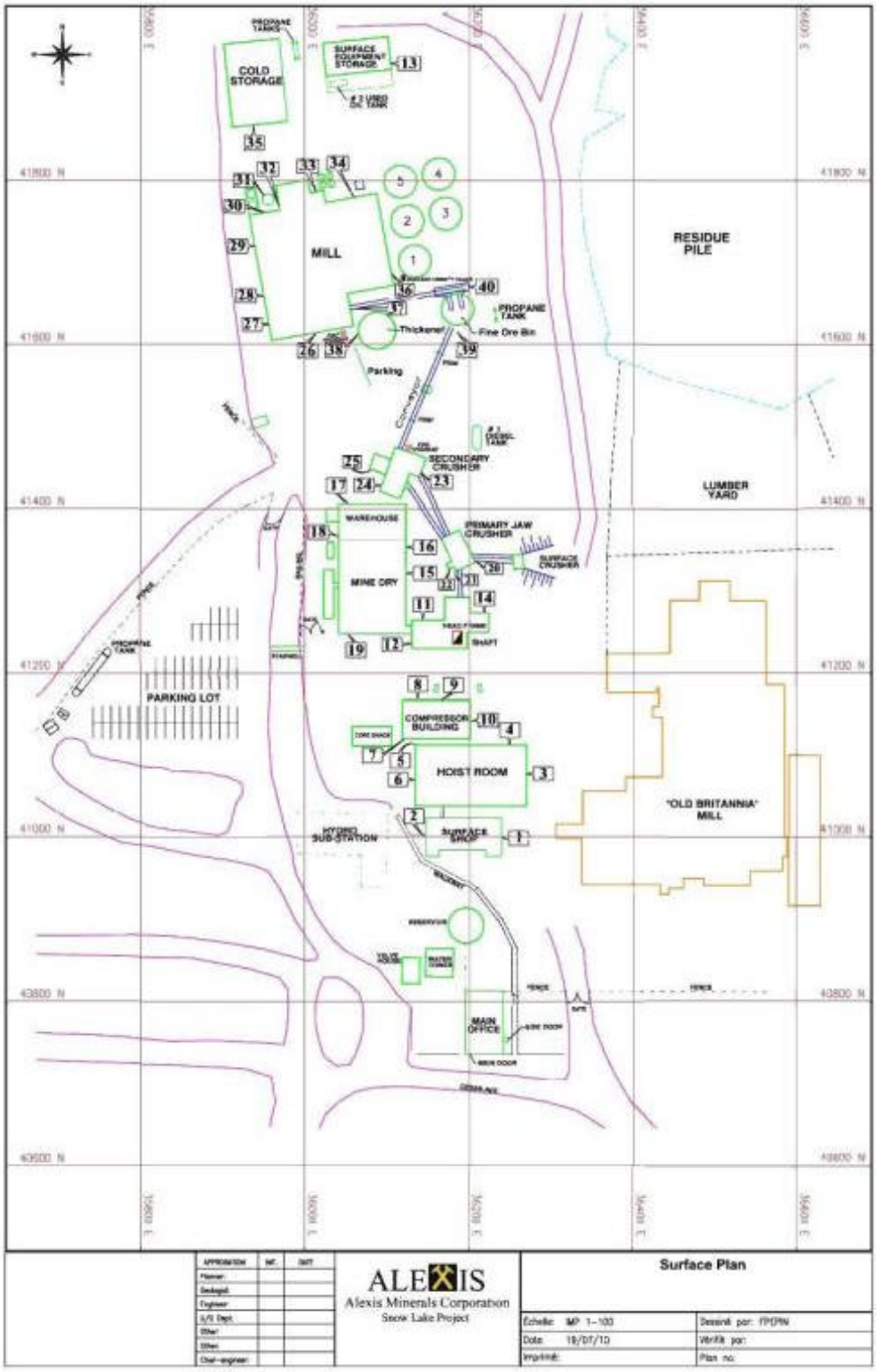

Figure 18-6: Snow Lake Mine Existing Surface Infratructures

The Main Mine reconditioning includes the headframe /collar house / ore & waste bins, hoist & hoist building, mine shaft, 9 level stations, level dumps, 2 loading pockets, and a spill pocket. The rehabilitation of the production infrastructure will be completed during the pre-production period, whereas the production infrastructure in the Main Mine Lower will be completed after the SLM is in production.

Dewatering of the workings will be required. Alexis has estimated that the current water level is 2,900 ft below surface and the workings contain 190 million USG of water. At a maximum pumping rate of 1,200 USGPM it will take over 110 days of continuous pumping to dewater the Main Mine Lower (the water recharge is believed to be about 72 USGPM). The discharged water will be pumped to the mill and ultimately to the tailings dam.

Offices

The main office building has been used by the exploration group since the mine closed in 2005. A new computer system was installed to upgrade the technology. The offices will require some cleaning, basic office furniture and computers to be functional.

Mine Dry

A mine dry for both men and women is fully equipped with baskets and lockers.

Shops

A surface shop is functional and is being used by a local contractor.

Main Electric Sub-Station

The main electrical sub-station will require cleaning and the main power line to the No. 3 Zone will require upgrading to handle the increased power requirements.

Main Fan & Air Heater

The surface air heater and propane tank will be reconditioned by the propane supplier prior to start up. The main ventilation fan is located underground and will require a new motor. The fan could be turned by hand during the underground inspection and appeared to be in good condition.

Compressor Room & Compressors

The compressors will need extensive repair and it was decided to replace them with three new high efficiency screw compressors. The piping from the compressor room to the shaft will be repaired as required.

Hoist Room & Hoist

In the hoistroom, the cable openings need to be opened and the overhead crane requires recertification. The hoists are in good condition. The electrical motors were verified and are in operating condition. Contractors have provided estimates for the cost of refurbishment. New hoist cables need to be installed.

Headframe
The steel headframe was erected in 1994 and remains in good condition and will require an engineer's certification. The sheave wheels will require some repairs.

Main Mine Shaft
The mine shaft is expected to be in good condition. The shaft piping (compressed air, process water and discharge water) is planned to be replaced. The two loading pockets are expected to be in good condition. Statements from miners that worked in the shaft confirmed that the two loading pockets had been refurbished prior to the shut down. The spill pocket below the 1890 loading pocket will have to be rebuilt.

A shaft pump will be installed after the main pumping system is operational on the 1780 Level. This will provide the maximum pumping rates to dewater the Main Mine Lower.

18.2.2 Main Mine Underground Facilities Reconditioning

The following infrastructure requires reconditioning. The costs included in the feasibility study were based on quotations provided by contractors:

- 5 trackless levels (320, 580, 780, 1030 and 1530);
- 2 track levels (1280 and 1780);
- The ramp extending from the 3000 Level to the 3746 Level;
- The escape way from surface to 3746 Level ;
- Refuge stations;
- Powder and cap magazines; and
- Other ancillary excavations (pump stations, dumps, chutes, etc;)
- Electric power;
- Compressed air and process water piping;
- Secondary ventilation; and
- A communication system.

A small amount of development work in the Main Mine Upper is required in the pre-production period:

- 837 ft of sub-levels; and
- 558 ft of raise.

18.2.3 No. 3 Zone Ramp Dewatering and Reconditioning

There is no infrastructures in the area of the No. 3 Zone portal as this site was reclaimed after operations ceased in 1996. Figure 18-7 shows the proposed general arrangement at the No. 3 Zone portal. The following temporary and permanent infrastructure will be installed:

- Electrical hook-up with transformer station;
- Office and parking area (temporary trailers);
- Temporary ventilation fan and mine air heating;
- Permanent main fan and mine air heating;
- Surface Powder Magazine; and
- Surface Cap Magazine.

The following permanent services will be installed in cut-outs in the ramp:

- Compressor Station;
- Transformer Station;
- Refuge Station/Lunch room;
- Underground Communication system;
- Underground Powder mMgazine;
- Underground Cap Magazine.

After the infrastructure is established, the No. 3 Zone underground workings will be dewatered to the 425 Level, (27.6 million USG of water). At the planned pumping rate of 400 USGPM, 48 days of continuous pumping will be required to complete the dewatering. The water will be pumped into the Tailings Disposal Area where it will be managed with the tailings water.

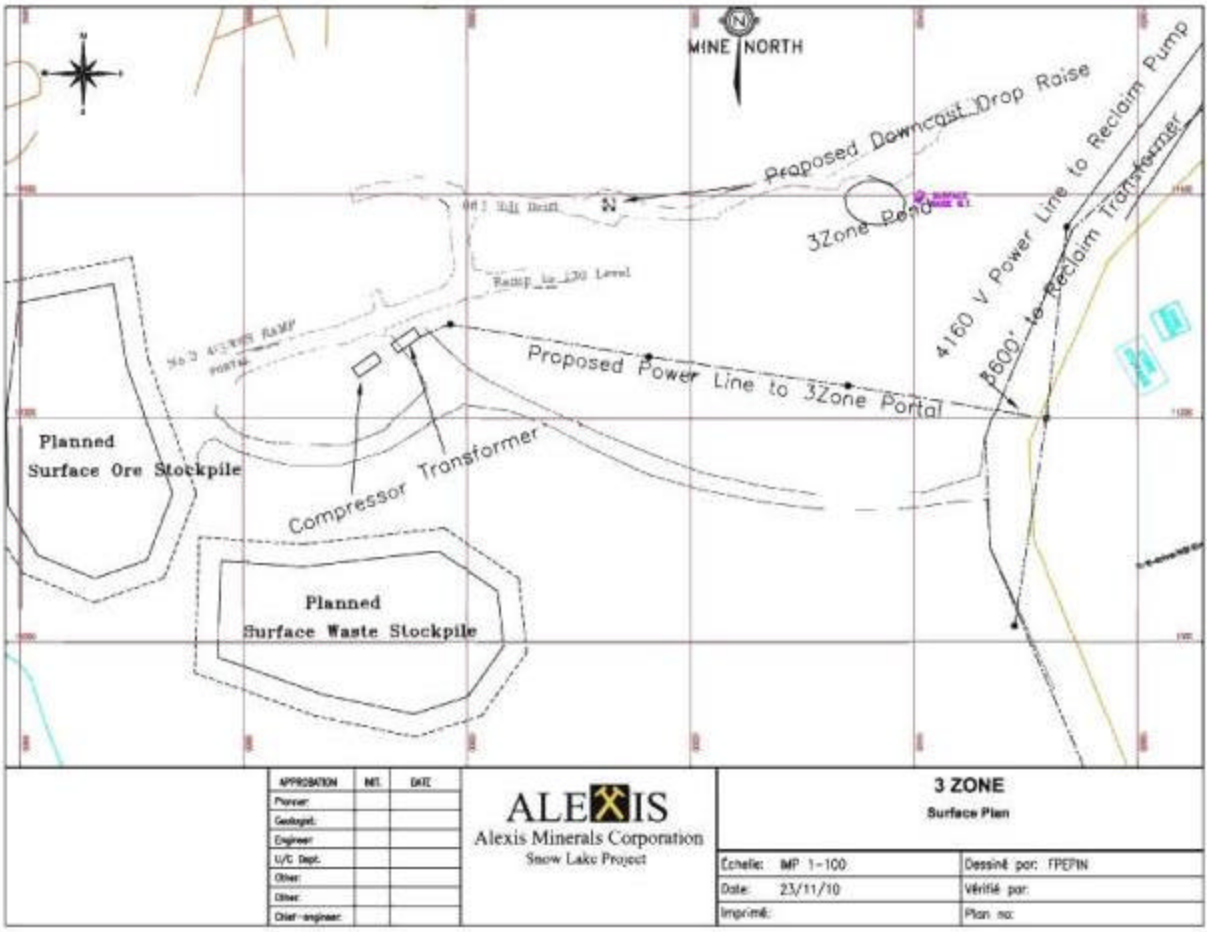

Figure 18-7: No. 3 Zone proposed surface arrangement

The surface shop at the Main Mine will be used to repair the No, 3 Zone equipment.

A 1,000 kVA transformer station is planned at the No 3 Zone portal. The power required for No 3 Zone will exceed the capacity of the current power line and an upgrade is planned.

The No 3 Zone ventilation system consists of a 200,000 CFM fan and a 16 MBtu heater. The initial dewatering will be completed using a temporary ventilation fan and heater at the portal. The main ventilation fan will be installed on a raise that will be developed into the open stope.

The dewatering system is designed to consist of a series of submersible pumps in excavated sumps.

The waste generated from the development will be hauled to the surface and dumped in a designated area near the portal, then trucked to the main waste stockpile. This

disposal will continue until mined out stopes can be backfilled with the development waste.

The ore will be hauled to the surface and dumped in a designated stockpile, then hauled to the mill.

18.3 Development and production

18.3.1 Geotechnical Assessment (DW)

The Preliminary Assessment (PA) report "Snow Lake Mine NI 43-101 Technical Report", April 2010 by Golder Associates identified that no formally collected geotechnical information such as borehole logs or underground geotechnical mapping was available for their report. The PA was based on various geotechnical reports and stability assessments that were undertaken during the operation of the mine. In their recommendations Golder Associates identified a number of issues to be analysed and investigations required for future study, these included:

- The stability analysis of the proposed surface crown pillar above the Main and Toots Zones.
- The impact of the proposed mining in the Main and Toots Zones on the stability of the crown pillar.
- The geotechnical characterization (rock quality) of No. 3 Zone.

The recommended investigations to resolve these issues included:

- A review of the historical information and knowledge of the geometry and stability of the surface crown pillar above the Main and Toots Zones.
- Review geometry and rock quality of potential ore adjacent to crown pillar
- Conduct stability assessment of current crown pillar and impact of proposed adjacent mining
- Develop recommendations for proposed mining to ensure the stability of the crown pillar is maintained
- Review historical core hole drilling and geotechnical characterization of No.3 Zone.
- Compare rock quality in No. 3 Zone with available information on rock quality in Main Mine
- Conduct stability assessments of proposed mining excavations in No. 3 Zone

A geotechnical data collection and testing program was performed to address these issues. This included the collection and interpretation of structural geology data, rock mass classification according to Barton's Q and Bieniawski's RMR systems and the point load testing of diamond drill core to derive the equivalent uniaxial compressive strength of the rock types.

A summary of the data collection and analysis is presented in a series of reports:

- Preliminary Stability Analysis of the Upper Toots Crown Pillar, Snow Lake Mine, July 2010.
- Preliminary Stability Analysis of the 3 Zone Crown Pillar, Snow Lake Mine, July 2010.
- Review of the Preliminary Stope and Pillar Dimensions for the Main and 3 Zones, Snow Lake Mine, August 2010.

The procedures, analysis, conclusions and recommendations of these reports are presented in the following sections:

18.3.1.1 *Data Collection and Analysis*

The data collection and analysis program comprised of the collection and interpretation of structural geology data, rock mass classification according to Barton's Q and Bieniawski's RMR systems and the point load testing of diamond drill core to derive the equivalent uniaxial compressive strength of the rock types. The following conclusions are made from the data collection and analysis program:

- The following structures were identified at Snow Lake Mine:
  - Foliation - $40^{o}/315^{o}$
  - Faults/Shears - $50^{o}/245^{o}$
  - Joint - $24^{o}/155^{o}$
- It was determined that no kinematically feasible structural failures are likely.
- Rock mass classifications give a conservative, or lower bound estimate of 26 for Q' and an RMR of 75. This represents ground conditions which can be described as Good. These values of Q and RMR correlate well and indicate that the data collection procedure did not contain any inherent bias.
- A difference in the rock strength rating probably reflects that a higher degree of silicification is present at No. 3 Zone than Main Zone.

- The Main Zone rock mass classification data shows slightly more variability in the RMR than 3 Zone. Overall, the data suggests that similar to slightly better ground conditions can be anticipated in the 3 Zone deposit.
- All the rock types tested display high average equivalent compressive strengths in excess of 100 MPa. The ore zone strengths are in excess of 200 MPa.
- The in situ stress values as determined by Arjang, (1996) were selected for the present study and an average horizontal to vertical stress ratio of 1.6 has been assumed.

18.3.1.2 *Crown Pillar Analysis*

The Crown Pillar stability analysis was performed using three empirical and one analytical method. The empirical methods included: the Unsupported Safe Span as developed by the Norwegian Geotechnical Institute; Carter's Crown Pillar Stability Analysis; and the Hoek and Brown Pillar Strength Assessment. An analytical calculation using the program CPILLAR was applied to determine the stability analysis in three-dimensions. In addition, a finite element analysis using the program Phase2 was applied to the 3 Zone geometry to examine the mining induced stress redistribution.

Main Zone

The following conclusions are made for the Upper Toots crown pillar:

- The crown pillar and associated abutments are expected to remain stable indefinitely.
- There are no time-dependent factors that were discovered at the property that could adversely affect the degree of stability.
- A sensitivity analysis with low strength values supports this conclusion.
- The results of Carter's empirical procedure show that the current crown pillar is stable. The ground conditions would have to deteriorate below a critical value of Q<2.0 before a potentially unstable condition is reached. This level of deterioration is not likely.
- The results of the Hoek and Brown pillar strength procedure indicates a crown pillar strength between 35 MPa to 84 MPa. The stress analysis indicates a maximum pre-mining horizontal stress of 10 MPa and a mining-induced maximum stress in the order of 20 MPa. This level of stress is significantly less than the calculated strength of the crown pillar, therefore a stress-driven failure mechanism is not likely.
- The results of the CPILLAR analysis shows the crown pillar is stable.

No. 3 Zone

The following conclusions are made for the 3 Zone crown pillar:

- The crown pillar and associated abutments are expected to remain stable indefinitely.
- There are no time-dependent factors that were discovered at the property that could adversely affect the degree of stability.
- A sensitivity analysis with low strength values supports this conclusion.
- The results of Carter's empirical procedure show that the current crown pillar is stable. The ground conditions would have to deteriorate to a critical value of Q<2.0 before a potentially unstable condition is reached. This level of deterioration is not likely.
- The results of the Hoek and Brown pillar strength procedure indicates a crown pillar strength between 51 MPa to 119 MPa. The stress analysis indicates a maximum pre-mining horizontal stress of 10 MPa and a mining-induced maximum stress in the order of 20 MPa. This level of stress is significantly less than the calculated strength of the crown pillar, therefore a stress-driven failure mechanism is not likely.
- The results of the CPILLAR analysis shows the crown pillar is stable.
- The results of the Phase2 analysis indicate that the geometry of the 3 Zone crown pillar is stable.

18.3.1.3 *Preliminary Stope Dimensions*

The analysis of the stope dimensions used the widely accepted Mathews/Potvin Stability Graph procedure. The following conclusions are made from this analysis:

- For planning purposes, the Main Zone should consider the following stope dimensions:

  Width = variable

  Height = variable

  Length = 120 ft

- In some cases the Stability Index plots across the Stable/Transition portion of the Stability Graph. This means the back and endwalls for the dimensions analysed should be stable, however artificial support may be required locally in the hangingwall to supplement the rib pillars. This aspect should be examined further at the detailed design phase.

- For planning purposes, the feasibility study for 3 Zone should consider the following stope dimensions:

  Width = 20 ft

  Height = 340 ft

  Length = 120 ft

- After stope heights of 250ft have been mined the 3 Zone hangingwall approaches the potentially unstable area on the Stability Graph. This is considered to represent an adequate level of accuracy at the feasibility study and preliminary design stage. The flexibility exists in the mining method where a reduction in the stope length can be made during the production planning stage when more site specific details will be known. This means that artificial support may be required locally in the hangingwall to supplement the rib pillars. The cost/benefit analysis of this aspect should be examined further at the detailed design phase.

18.3.1.4 *Pillar Dimensions*

The sill, rib and boundary pillar dimensions were initially determined using the procedure established by Salamon and Munro and then compared with the more recent procedures as developed by Lunder and Pakalnis.

Rib Pillars

- The rib pillar strike length (y) versus the ore width (x) can be expressed as y=0.743x + 8.212 and a minimum 20ft wide rib or boundary pillar is adequate for ore widths up to 17ft. Larger rib pillars may be required in greater ore width locations.
- The range of rib pillar width/height ratio versus average pillar load/UCS from the Salamon and Munro pillar design calculation plots within the potentially unstable zone on the Pillar Stability Chart of Lunder and Pakalnis. This indicates a factor of safety between 1.0 and 1.4 and represents Class 2 and Class 3 pillars as described by Lunder and Pakalnis. This means that some pillars may exhibit minor to moderate levels of deterioration. For the purpose of a feasibility study this preliminary design is considered acceptable. Further work will be required at the detailed design phase to select the appropriate pillar design constants and to determine an appropriate target factor of safety.
- Larger rib pillars may be required where ore widths exceed 17ft based on the equation y=0.743x + 8.212.

Sill Pillars

- A back analysis of the former 3 Zone workings using the Mathews/Potvin Stope Stability Graph technique indicates that the maximum spans have been achieved for the given ground conditions in this area and that a system of rib pillars will be required as the mining is extended below the 500 Level.
- Three sill pillars are proposed for 3 Zone at the 500, 900 and 1300 Levels. 40ft thick sill pillars are stable at the 500 and 900 Levels, a 50ft thick sill pillar is stable at the 1300 Level for target factors of safety of 1.4. It should be noted that the analysis does not take the combination of both the rib and sill pillars into account. The analysis represents a worst case, where it can be assumed that the combination of rib and sill pillars will further reduce the level of mining induced stress and improve the factor of safety. The currently proposed sill pillar design is considered to be acceptable for the purpose of a feasibility study. Further work, possibly in the form of three dimensional numerical modelling may be required at the detailed design phase.
- The proposed sill and rib pillar configuration for 3 Zone results in a theoretical recovery of 0.799.

18.3.1.5 *Main Zone Room and Pillar mining below 3000 Level*

Mechanized room & pillar mining methods are planned for the Main Zone ore body below the 3000 Level. The proposed method subdivides the ore into 200ft x 200ft extraction panels, and each panel is accessed with an internal 8 ft x 8 ft ramp driven at 17% in the ore. The room & pillar mining will use a pattern of 8 ft x 8 ft square pillars and 16ft room spans. It is planned to apply the mechanized room & pillar methods to extraction panels that extend from the 3486 Level to the 4636 Level.

An analysis of the structural geology was performed using to program Unwedge to determine if any adverse structural combinations could occur. In all cases it was concluded that a structural failure is unlikely.

The pillar sizes and room spans were validated using the confinement theory of Lunder and Pakalnis according to the depth of the extraction panels. A minimum target Factor of Safety of 1.4 is achieved for 8 ft x 8 ft pillars on 24 ft centres from the 2000 Level to the 4000 Level. For the 4500 Level, 8 ft x 8 ft pillars on 24 ft centres result in a Factor of Safety of 1.33, which is less than the minimum target. In this case either one of two approaches could be adopted: 10 ft x 10 ft pillars on 24 ft centres could be applied to the 4500 Level, or the response of 8 ft x 8 ft pillars on 24 ft centres could be monitored in a top down mining sequence to determine whether this configuration could be applied to the 4500 Level.

It is concluded that the proposed configuration using 8ft X 8ft pillars on 24ft centres is acceptable for the Feasibility Study and mine planning

purposes. Further work on the detailed design and monitoring the pillars will be required during the implementation phase.

18.3.2 Main Mine Upper (AR)

18.3.2.1 *Mining Method*

The Main Mine Upper contains proven and probable ore reserves of 1,817,0002 t grading 3.58 g/t for 209,168 contained Oz of gold. The tabular ore body has an average dip and width of 45° and 9 ft, respectively. Approximately half of this reserve lies between the mined out stopes with the remaining reserves to the east of the mined out area.

The Mining Method selected for the Main Mine Upper is Longhole Retreat Open Stoping with small diameter holes and no backfill, (Figure 18-8). This area was mined by Kinross using the same method. Minimal capital development is required to access the planned stopes, therefore a typical mining sequence is:

- Develop an access cross-cut (11.5 H x 10 ft W) and Alimak raise;
- Drive a Service raise in ore, (6 ft H x 9 ft W);
- Develop sub-levels (11.5 ft H x 10 ft W) at vertical intervals of 50 ft;
- Drill the 2½ in diameter longholes and drop raises;
- Blast the drop raise; and
- Blast the benches retreating towards the access raise.

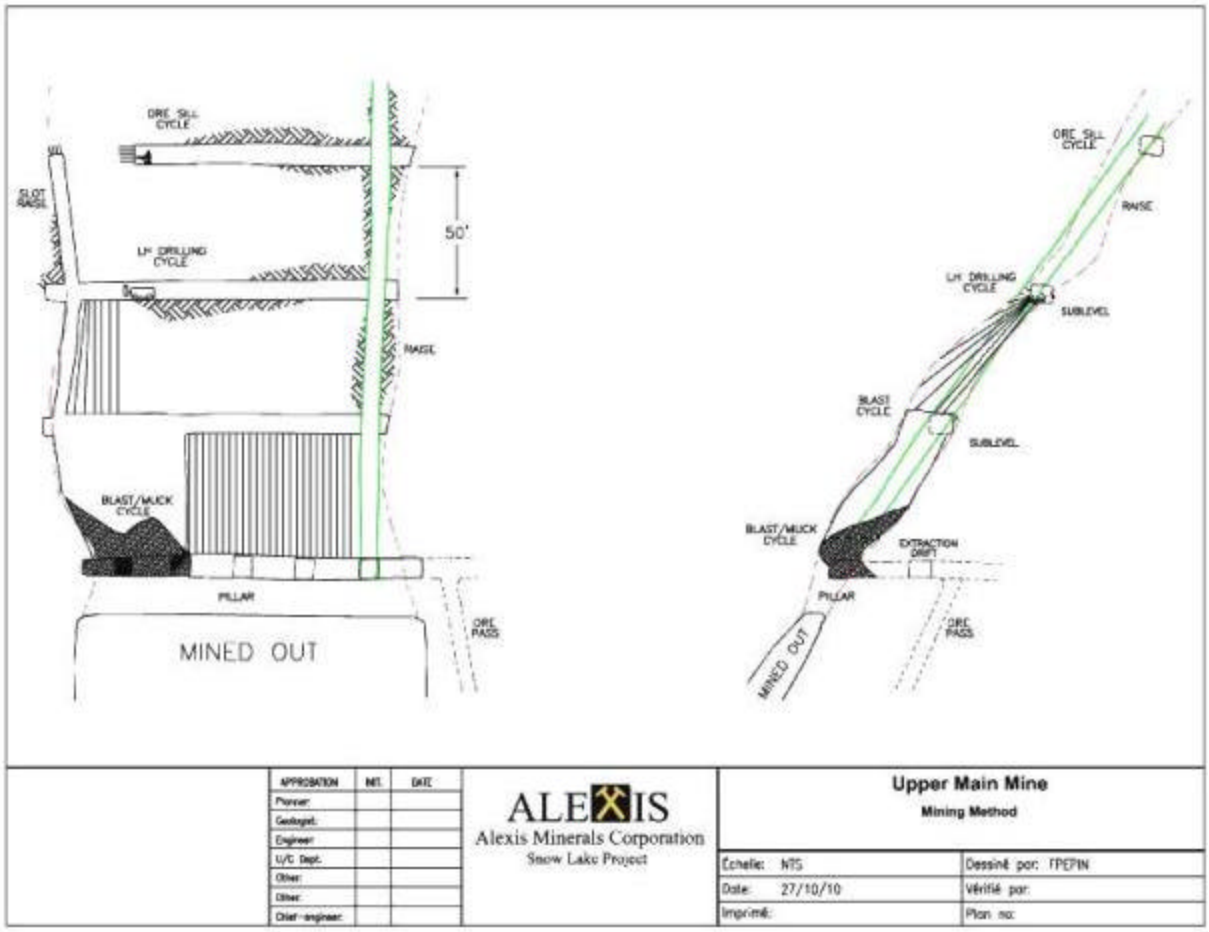

Figure 18-8: Concept of the longholeole retreat open stoping

The broken ore will be mucked out using 3.5 yd^3 remotely operated LHDs. A series of 20 ft wide rib pillars are left in place to avoid spans more than 120 ft along strike.

The Main Mine Upper contains 32 stopes, ranging from 6,636 t to 193,060 t (see Table 17-28). The planned daily production rate is 1,250 t/day or 37,500 t/month. This will require 5 to 6 stopes in the production stage.

Ventilation

Of the total 360,000 cfm of mine ventilation, 185,000 cfm is planned for the Main Mine Upper area.

A typical stope will require 10,000 to 15,000 cfm and will be delivered with secondary fans located at the top of the service raises.

Equipment

The equipment list is:

- 4 Rubber tired Long-Toms;
- 2 Slushers;
- 2 Alimak Raise Climbers;
- 5 Longhole Drills;
- 4 LHDs (3.5 yd^3);
- 4 LHDs (2 yd^3);
- 2 Scissor Lift trucks;
- 20 wagons of 5 t;
- 4 Battery Locomotives (7 T);
- 25 Jack Leg Drills;
- 25 Stoper Drills;
- 8 Tuggers

Manpower

The manpower in the Main Mine Upper will vary according to the development and production requirements. 4 -5 stopes will be in production while 3 other stopes are being developed, and this will require 23 workers on a typical work shift, as follows:

- Sub-Level development: 4 crews of 2;
- Alimak raising: 2 crews of 2;
- Longhole drillers: 3 ;
- Loading and blasting: 1 crew of 2;
- Mucking: 4 LHD operators for both development and production; and
- Hauling: 2 locomotive operators.

Longhole Drilling and Blasting

The average longhole will be 58 ft long and holes deviation will be minimized with good quality control.

Longhole rows are planned with an average burden of 2½ ft and spacing of 4½ ft. Each row will contain about 105 t and it will be necessary to blart an average of 12 rows to produce 1,250 t/day.

The historical practice has used ANFO in both development headings and longhole blasts.

Sub-Level Development Productivity
Based on 2 shifts per day, the sub-level development is scheduled for 14 ft/day or 450 ft/month.

Alimak Raising Productivity
Based on 2 shifts per day, the Alimak raising is scheduled for 5.0 ft/day or 150 ft/month.

Longhole Drilling Productivity
The scheduled drilling rate is 2,070 ft/day. A fleet of 5 longhole drills will be required assuming 82% availability.

Mucking Productivity
Mucking will use 3.5 yd3 remote LHDs. Each LHD unit will service 2 stopes.

4 LHD units will be assigned to the Upper Main Mine with 69% availability to achieve 1,250 tpd.

Hauling Productivity
Rail transportation to the shaft will occur on the 1280 and the 1780 Levels. Eight 5 t rail cars will be used in trains and transport 40 t. 32 trains will be required to achieve 1,250 t/day.

Dilution Allowance
The historical dilution allowance of 5 ft in the hangingwall at an average grade of 0,79 g/t has been included in the production schedule.

Mining Recovery
The historical recovery has been 90%. This has also been included in the reserve estimation and the production schedule.

18.3.2.2 *Development*

The development program will start by rehabilitating the 7 Levels in the Main Mine Upper:

- 320 Trackless
- 580 Trackless
- 780 Trackless
- 1030 Trackless

- 1280 with tracks
- 1530 Trackless
- 1780 with tracks

The rehabilitation program is detailed in section 18.2.2 Main Mine Underground and Facilities Reconditioning.

The Main Mine Upper Detailed Development Schedule shows the following development quantities:

Table 18-2 : Main Mine Upper Development Summary

	Alimak Raising	Sub-Levels
In Ore	7,892 ft	26,373 ft
In Waste	3,289 ft	11,211 ft
Total	11,181 ft	37,584 ft

The development requires 3 Alimak raises and 4 sub-levels to be developed simultaneously.

The development program in the Main Mine Upper will be completed within 3 years. The following table provides a summary of the ore and waste quantities.

Table 18-3 : Main Mine Upper Development Summary

	Year 0	Year 1	Year 2
Waste	420 ft	8,586 ft	5,534 ft
	2,550t	67,312 t	48,953 t
Ore	1,530 ft	18,454 ft	14,282 ft
	8,202 t	148,614 t	123,372 t
	3.74 g/t	3.67 g/t	3.36 g/t
	986 Oz	17,527 Oz	13,315 Oz

18.3.2.3 *Production*

The Main Mine Upper production will occur from years 1 to 4 as follows:

Table 18-4 : Main Mine Upper ProductionSummary

	Year 1	Year 2	Year 3	Year 4	Year 5
Ore	204,576 t	325,683 t	449,056 t	449,056 t	108,443 t
	4.00 g/t	3.62 g/t	3.56 g/t	3.45 g/t	3.45 g/t
	26,300 Oz	37,923 Oz	51,218 Oz	49,866 Oz	12,033 Oz

18.3.3 Main Mine Lower (AR)

18.3.3.1 Mining Method

General Description

The Lower Main Mine consists of the Dick East-West, Dick North-South and Ruttan East-West Zones. These zones were mined from 1995 to 2004. Remaining proven and probable reserves total 769,605 t grading 4.72 g/t for 116,689 contained Oz of gold in the Dick East-West, Dick North-South and Ruttan East-West Zones (Figure 18-2). The longhole mining method is planned, (Section 18.3.4.1).

The average dip and thickness of the Dick and Ruttan Zones are 30° and 9ft respectively. This is flatter than the Main Mine Upper. Mechanised Room and Pillar is a more suitable mining method, (Figure 18-9). A 40 ft vertical sill pillar will be left to separate the new mining horizon from the old workings.

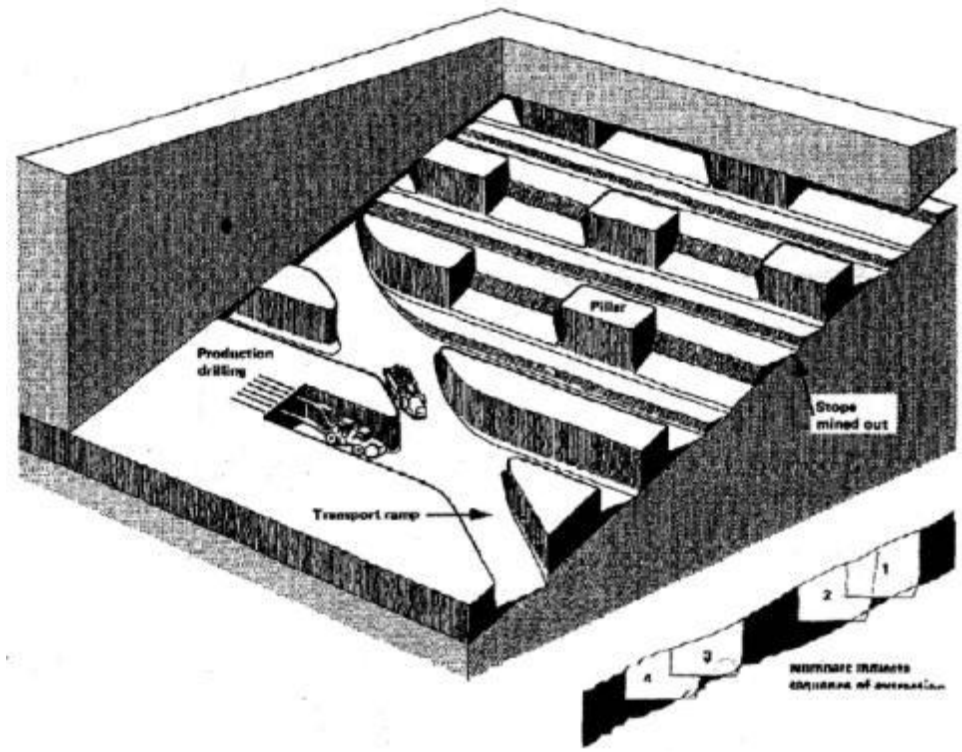

Figure 18-9: Mechanised room and pillar mining

The method involved the developmentof a series of 8ft H x 8ft W drifts in the ore body at a vertical interval of 100 ft. The resource blocks are 200 ft long (along strike).

An internal access ramp will be driven at 17% in each resource block, (Figure 18-10). Figures 18-9 and 18-12 show the method.

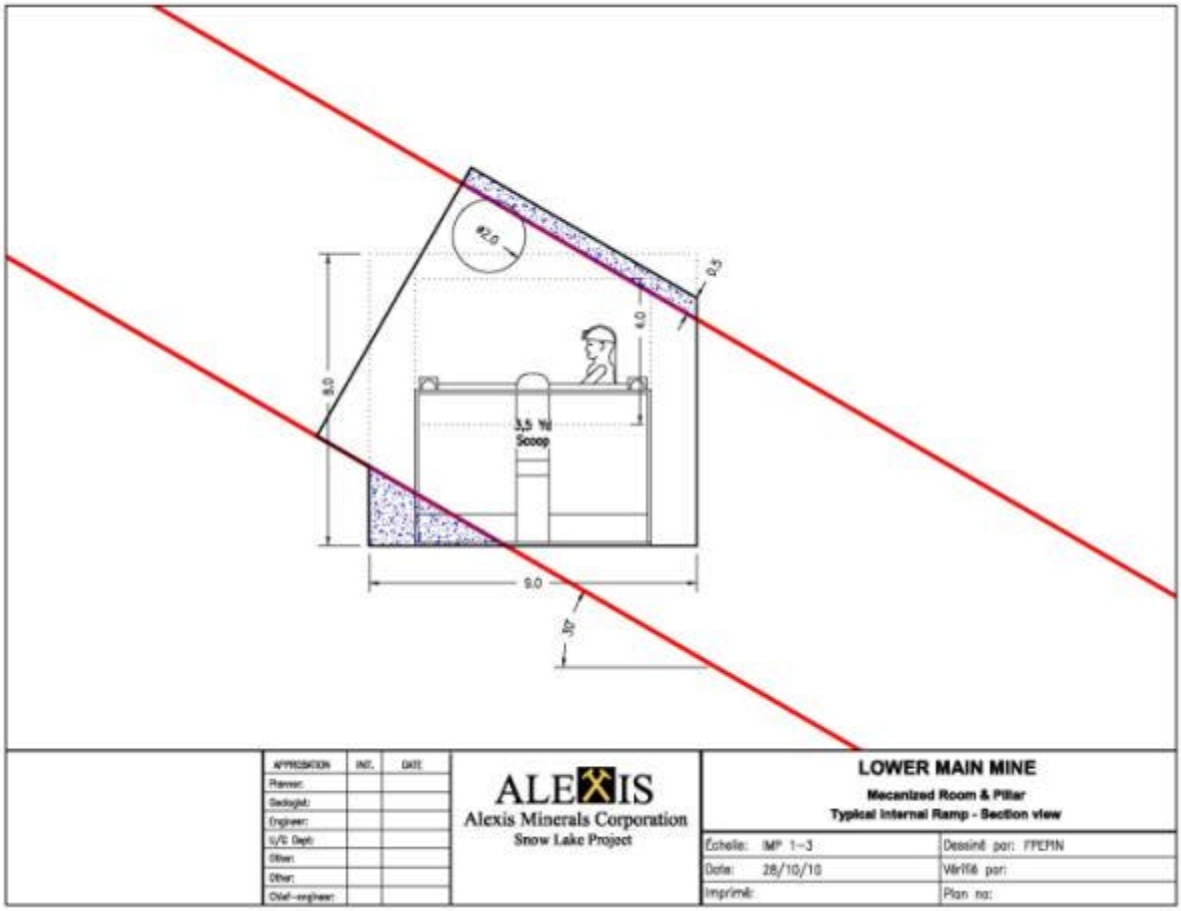

Figure 18-10: Mechanised room and pillar internal ramp section

The Main Mine Lower consists of 54 stopes, ranging in size from 196 t to 50,918 t (Table 17-29). The planned daily production for the Main Mine Lower is 750 t/day or 22,500 t/month. The average production rate of a stope is 75 t/day, therefore 11 stopes will be required in production at all times.

Development waste will be transported with 26 ton trucks and dumped into mined out stopes at a higher elevation. The broken ore will be transported with 26 ton trucks to the 2906 Level where it will be dumped into a storage bin before being hauled by locomotives on the 3000 Level to the shaft (Figure 18-13).

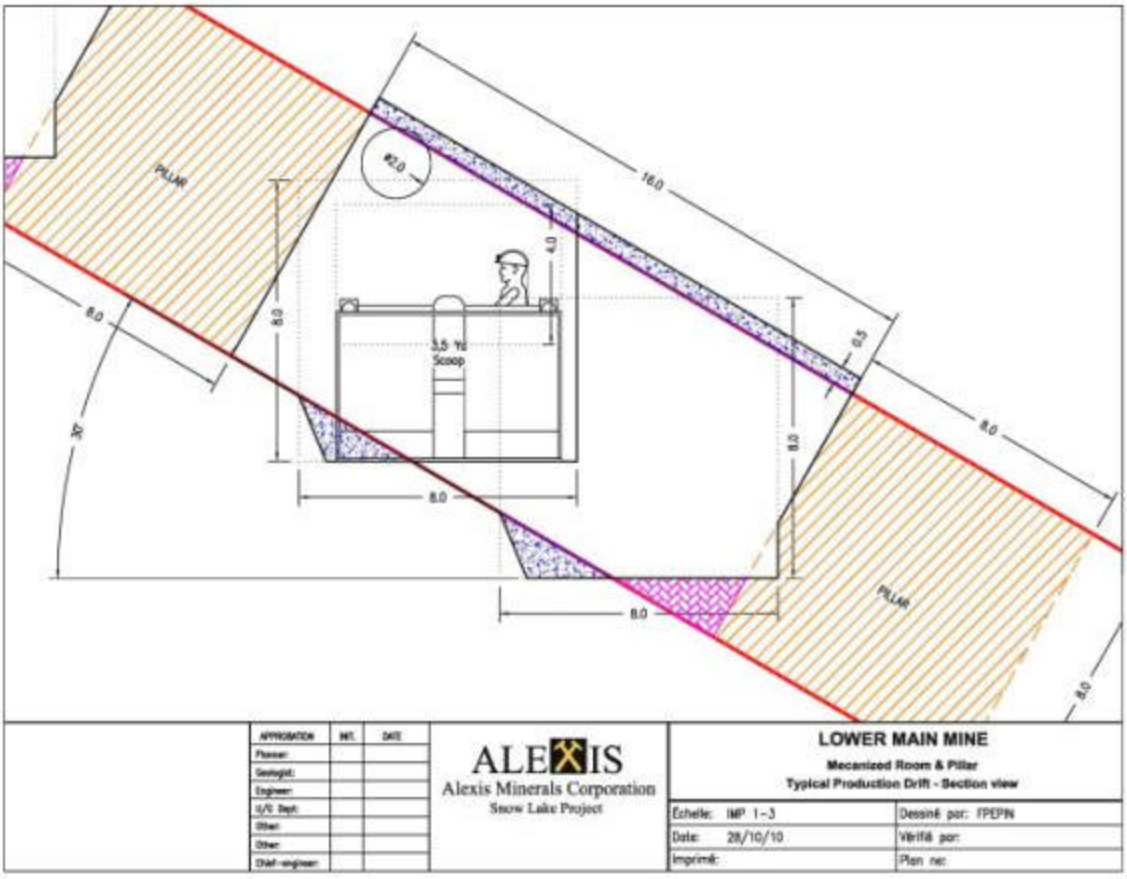

Figure 18-11: Mechanised room and pillar flat rooms cross section

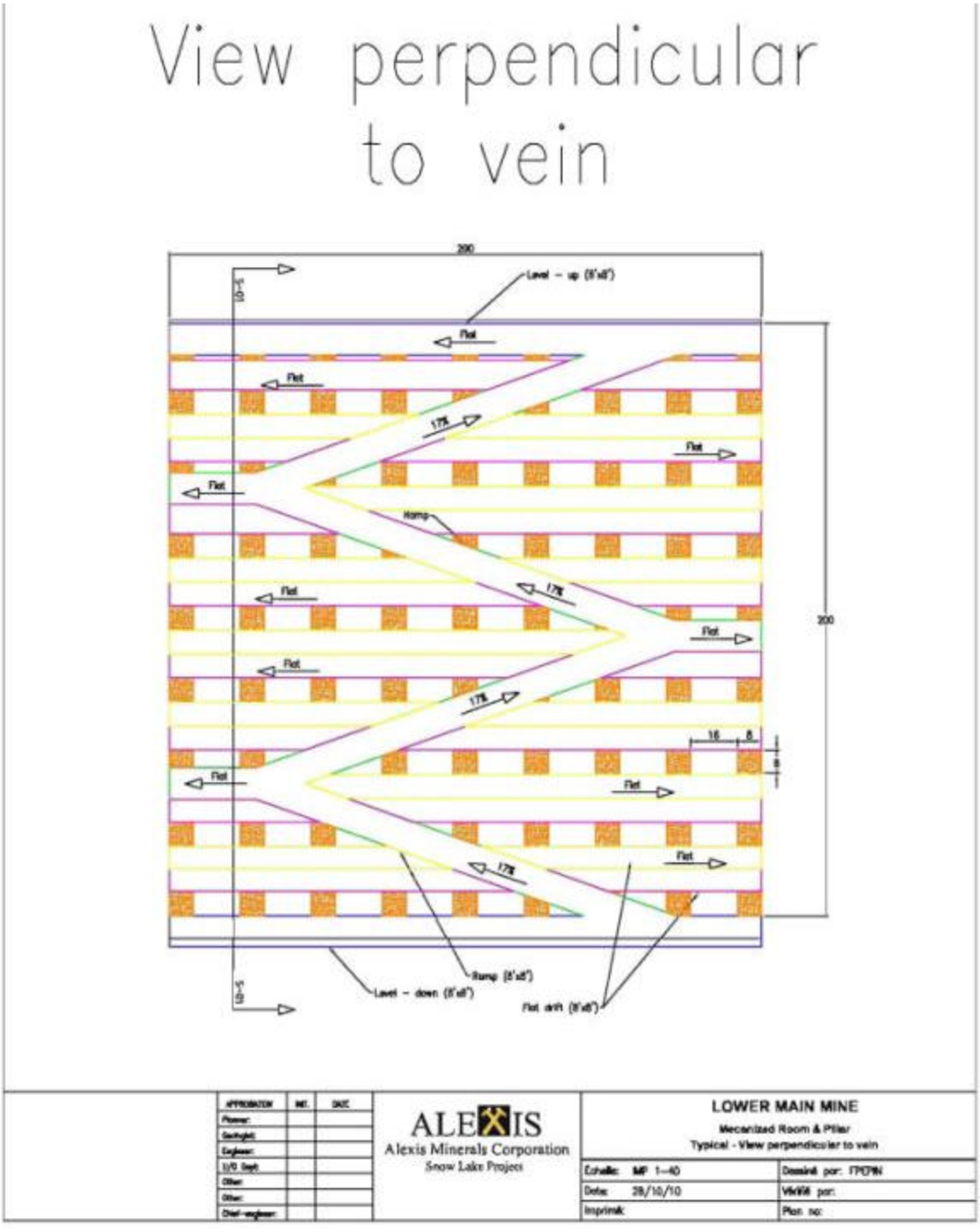

Figure 18-12: Mechanised room and pillar layout

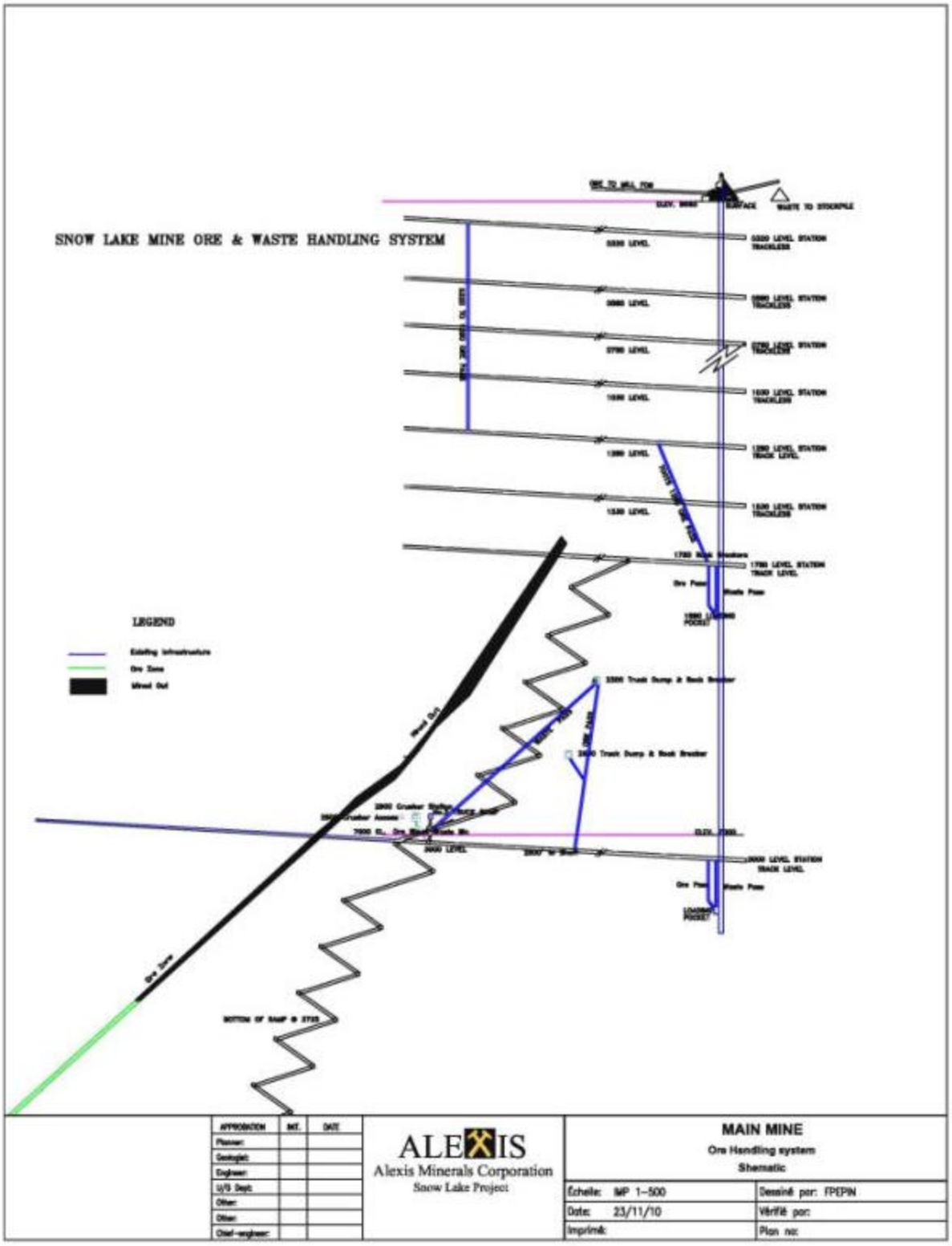

Figure 18-13: Main Mine ore handling system

Ventilation

The ventilation circuit of the Main Mine Lower consists of a series of ventilation raises and exhausts via the main ramp. The escape way is located in fresh air.

Equipment

The equipment list is:

- 3 Rubber tired Long-Toms;
- 2 Jumbos (1 boom);
- 3 LHDs (3.5 yd3);
- 2 LHDs (6 yd3);
- 1 Scissor Lift truck;
- 5 26 ton Trucks ;
- 10 Wagons of 120 cu ft;
- 1 Battery Locomotives (10 T);
- 10 Jack Leg Drills;
- 10 Stoper Drills.

Manpower

The manpower for the Main Mine Lower will vary according to the development and production requirements. 11 stopes will be in production while 5 other stopes are being developed and this will require 29 workers per shift, as follows:

- Main Ramp development: 1 crew of 5
- Sub-Level development: 1 crew of 5;
- Internal ramp development: 1 crew of 5;
- Room and Pillar mining: 1 crew of 9;
- Mucking: 1 LHD operator;
- Hauling: 3 Truck operators;
- Tramming: 1 locomotive operator.

Ramp and Sub-Level Development Productivity

A 2 man development crew using a 1boom Jumbo is scheduled for an average productivity of 7 ft/shift or 14 ft/day. The corresponding advance rate is 400 ft/month.

Room & Pillar Mining Productivity

A 2 man crew is scheduled for an average productivity of 35 t/man·shift from multiple faces.

Truck Transportation Productivity

5 trucks will be required for haulage from the lower levels. The fleet will be transferred from No 3 Zone. 1, 825 t per day will be track hauled from the Main Mine Lower Zone to the shaft.

Dilution Allowance

Mechanised Room and Pillar mining method is more selective method than the Longhole Retreat method and the dilution allowance is lower, (25%), (Figure 18-11).

Mining Recovery

The overall recovery of the room and pillar method is 85%, (Figure 18-12).

18.3.3.2 Development

The development of the Main Mine Lower will begin in Year 2, following the dewatering and reconditioning of the lower part of the mine. The development will increase significantly during Year 3 as the number of work faces will increases, (Table 18-05). The bulk of the ore and waste development will occur during Year 2 and Year 3 and will be completed in Year 4.

Table 18-5 : Main Mine Lower Development Schedule

	Year 2	Year 3	Year 4
Waste	2,105 ft	4,343 ft	284 ft
	10,914 t	22,515 t	1,471 t
Ore	3,495 ft	16,111 ft	3,569 ft
	18,116 t	83,517 t	18,502 t
	4.02 g/t	4.63 g/t	4.98 g/t
	2,339 Oz	12,438 Oz	2,961 Oz

18.3.3.3 Production

Production in Main Mine Lower is scheduled to start in Year 3 and reach a peak of 260,000 t/year in Year 4.

Table 18-6 : Main Mine Lower Production Summary

	Year 3	Year 4	Year 5
Ore	87,063 t	237,664 t	324,743 t
	4.91 g/t	4.79 g/t	4.66 g/t
	13,735 Oz	36,583 Oz	48,633 Oz

18.3.4 No. 3 Zone (AR)

18.3.4.1 Mining Method

General Description

The No. 3 Zone is a tabular ore body dipping at 45° north with an average thickness of 9 ft, similar to the mined out areas of the Main Mine Lower. A total of 219,152 t @ 4.3 g/t yielding 30,297 Oz of contained gold was mined in 1995 and 1996 between the 065 and 425 Levels. The No 3 Zone contains proven and probable ore reserves of 634,501 t grading 5.07 g/t for 103,475 contained Oz of gold extending to the 1590 Level.

The Mining Method for the No. 3 Zone is Small Diameter Longhole Uppers Retreat with no backfill, (Figure 18-14). The ore is divided in stoping blocks along strike with a maximum length of 120 ft. Mining will progress horizontally towards the the access cross-cuts.

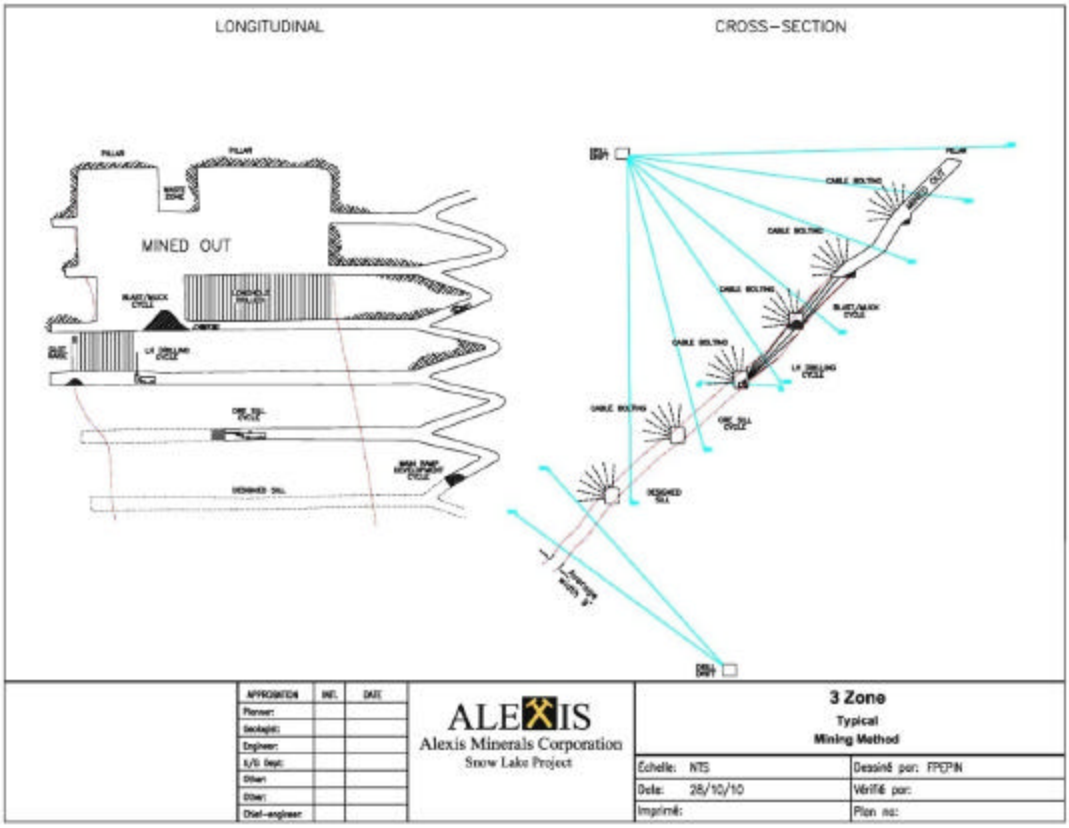

Figure 18-14: Ssmall diameter Longhole Retreat Stoping

The sequence of operation for a typical stope is:

- develop longitudinal drifts (8 ft H x 8 ft W) in the ore at a vertical interval of 50 ft;
- drill small-diameter (2.5 in diameter) longholes uppers including an inverse drop raise;
- blast the inverse drop raise; and
- Sequentially blast 3 to 4 rows of longholes.

Mucking will use 3,5 yd^3 remote LHDs.

The longhole uppers will be drilled to the floor of the next Sub-Level. The inverse drop raisewill be fired in a single blast. Blasting will continuewith groups of 3 to 4 rows of longholes retreating from the drop raise to the stope entrance, (Figure 18-15).

It is planned to leave 3 sill pillars to limit the size of the openings. Sub-Levels will be developed underneath each sill pillar to re-establish a new mining horizon.

Footwall Zone

The footwall Zone, between the 065 and 425 Levels, contains probable reserves of 83,025 t grading 4.62 g/t for 12,338 Oz of contained gold. This ore will be mined using the same method as the No. 3 Zone.

Ventilation

The No. 3 Zone ventilation circuit consists of :

- 10 ft x 10 ft air intake raise (to be excavated);
- 10 ft x 525 ft open stope down to 370 Level;
- A series of 10 ft x 10 ft raises, 1490 Level;
- Sub-Levels, through regulators;
- Ramp, as the exhaust to the surface.

Fresh air will be distributed on the levels with appropriately sized ventilation fans and flexible ducting.

Equipment

The diesel powered equipment required in the No. 3 Zone will consist of:

- Ramp and access cross-cut development
  - 1 Jumbo (2 booms)
  - 1 LHD (6 yd^3)
  - 1 Scissor Lift
  - 6 Service Trucks

- Sub-Level Development
  - 1 Jumbo (1 boom)
  - 1 LHD (3.5 yd^3)
  - 1 Scissor Lift

- Production
  - 3 Longhole Drill rigs
  - 3 Trucks (26 ton)
  - 2 LHDs (3.5 yd^3)
  - 1 LHD (6 yd^3)

Manpower

The typical crew will consist of 20 workers per shift:

- Ramp development (12 ft H x 16 ft W): 1 crew of 4;
- Sub-Level Development (8 ft H x 8 ft W): 1 crew of 3;
- Raise Development : 1 crew of 2;
- Longhole drilling and blasting: 3;
- Mucking: 4 LHD operators ; and
- Hauling: 4 Truck operators.

Longhole Drilling and Blasting

The longhole drilling pattern is identical to the Upper Main Mine and the average length of the longholes will be 58 ft.

Longhole rows are planned with an average spacing of 4½ ft and a burden between the rows of 2½ ft. An average row will contain about 105 t and it will be necessary to blast 7.1 rows to produce 750 t/day. ANFO will be used in the development headings and in the longhole blasting.

Ramp and Access Cross-cut Development Productivity

A 3 man development crew using a 2 boom Jumbo is scheduled for an advance rate of 13.5 ft/shift in a single heading. The monthly advance rate is 450 ft/month. When more than 1 face is available a fourth man will be added to the development crew and

it is expected that the advance rate will increase to 675 ft/month. This corresponds to productivities of 2.88 ft/man·shift and 3.25 ft/man·shift. The historical productivities for multiple faces were 4.5 ft/man·shift .

Sub-Level Development Productivity

A single face advance rate of 12 ft/shift (i.e. drill 12.5 ft break 12 ft) is scheduled for an advance rate of 450 ft/month. When more than 1 face is available it is expected the monthly advance rate will increase to 600 ft/month, this corresponds to productivities of 2.88 t/man·shift and 3.85 ft/man·shift which are below the historical average.

Long-Hole Drilling Productivity

The required longhole drilling rate is 412 ft/day. One longhole drill will be required.

Dilution Allowance

A 25% dilution allowance was included in the reserve estimation. This amounts to 133,579 t at an average grade of 0.45 g/t.

Mining Recovery

The No. 3 Zone Mining Recovery has been assumed to be 95%. The mining recovery has been included in the production schedule.

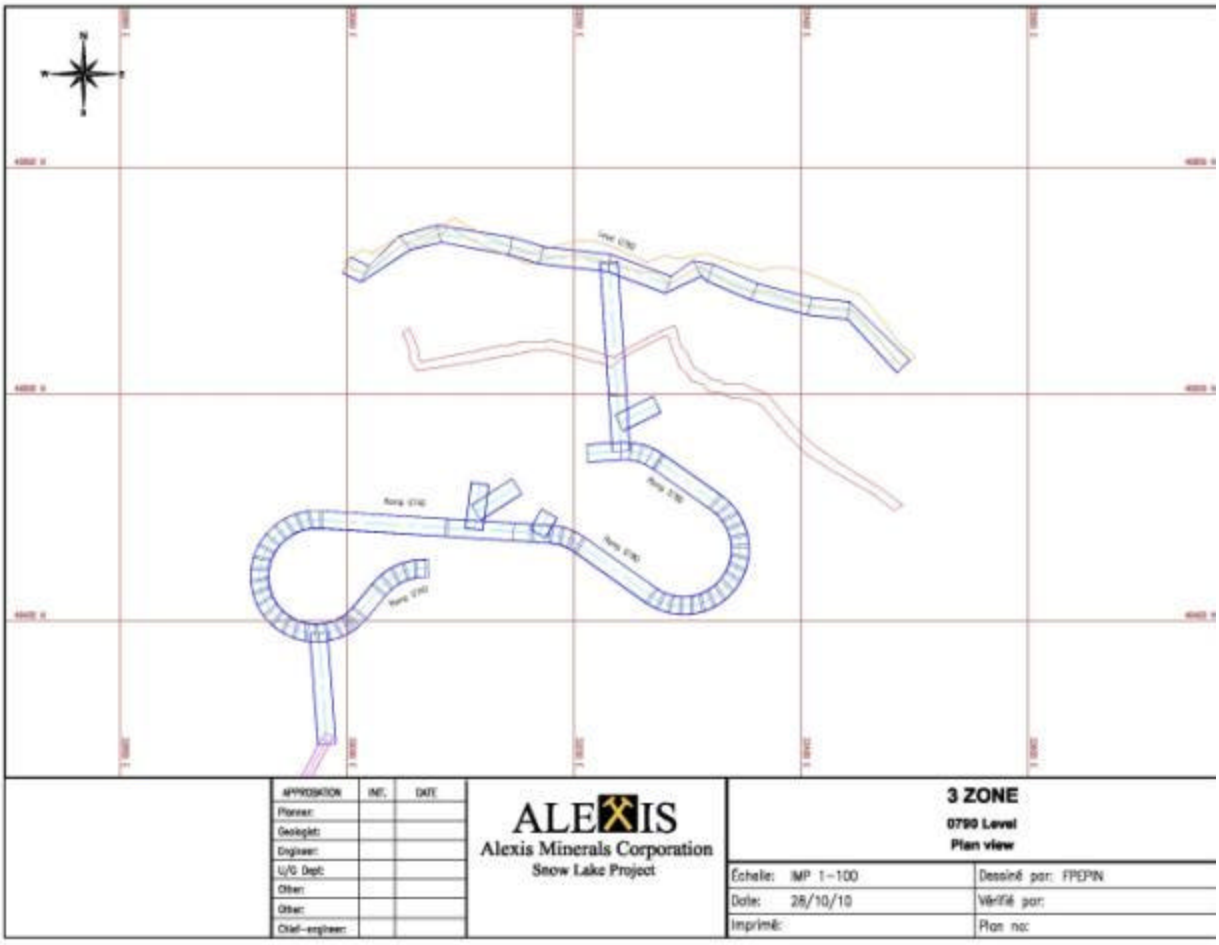

Figure 18-15: No.3 Zone 790 Level

18.3.4.2 Development programme

Table 18-7 : No. 3 Zone Development Summary

	Year 0	Year 1	Year 2
Waste	6,096 ft	10,881 ft	2,534 ft
	71,513 t	133,708 t	35,220 t
Ore	2,798 ft	6,065 ft	3,951 ft
	14,505 t	31,440 t	20,482 t
	4.45 g/t	4.74 g/t	5.06 g/t
	2,074 Oz	4,788 Oz	3,335 Oz

18.3.4.3 Production schedule

Table 18-8 : No. 3 Zone Stoping Summary

	Year 0	Year 1	Year 2	Year 3
Ore	47,670 t	223,558 t	249,517 t	46,000 t
	4.36 g/t	4.31 g/t	5.74 g/t	5.62 g/t
	6,677 Oz	30,967 Oz	46,088 Oz	8,314 Oz

18.4 Metallurgy and processing (IW)

18.4.1 Historical Mill operations

The original Snow Lake mill was newly constructed in 1994. The design capacity was 2300stpd. The processing operation comprised a conventional crushing plant, rod mill and ball mill grinding circuit, 5-stage cyanide leaching circuit, a carbon-in-pulp gold recovery circuit, carbon stripping and refinery, to produce gold bullion (Figure 18-16).

The mill operated continuously until late 2004, processing ore from the Main mine. During the period 1998-2202, the mill treated an annual average of approximately 750,000 t with a peak of 786,520 t in 1999. Gold recovery in the mill improved steadily from the initial level of 88% and attained over 94% in 2002-2003.

A review of the mill reports indicates an improvement to recovery was obtained by finer grinding (to a k80 of about 70 microns) and good control of cyanide levels in leaching obtained with a cyanide analyser. Intial operation in 1995 with 3 Zone ore showed a lower recovery at 82%, part of which may be attributed to normal start up difficulties and the known lower recovery for 3 Zone ore at the grind size that was employed.

In 1996 a cyanide destruction system was installed using the Inco SO2/air process. It operated satisfactorily until mid-1998 when it was discontinued since it was found that regulated discharge levels of cyanide in the effluent from the tailings area could be attained by natural degradation. Ferric sulphate was added to the polishing pond to ensure that arsenic content of the effluent was below permitted limits.

18.4.2 Process Description

The Snow Lake mill provides conventional processing to fine grind and leach the ore.

Ore from the mine is conveyed to 36x48 inches jaw crusher which in turn feeds a vibrating screen set at 12mm. The screen oversize feeds a 5-1/2ft cone crusher from which the product is also conveyed to feed the screen. Screen undersize is conveyed to a fine ore storage silo. Ore from other sources such as 3 Zone is delivered to an outside hopper from where it is conveyed to feed the jaw crusher.

Crushed ore is withdrawn from the silo at a controlled rate by two belt feeders, delivering ore to the rod mill feed belt. A weightometer on this belt is used to monitor the feedrate and control the feeders. The rod mill is a 11.5 ft x 15 ft long Koppers mill driven by a 1000hp synchronous motor, and employs 3.5 inches diameter rods. The slurry discharging from the rod mill is pumped together with ball mill discharge to feed a set of 15-inch diameter hydrocylones to control the grinding product size at 70 microns (K80). The cyclone underflow feeds a 14ftx 20ft long Nordberg ball mill driven by a 2000hp synchronous motor and utilizes 1.5 inches grinding balls. Lead nitrate solution is added to the grinding circuit to retard dissolution of the sulphide minerals.

The cyclone overflow is screened to remove trash and coarse particles and is pumped to a 40 ft diameter thickener together with flocculant solution. Thickener overflow solution returns to the grinding circuit as make up water together with mine water. Thickener underflow slurry at approximately 48% solids is pumped to the first of five agitated leaching tanks each 40ft dia by 40 ft high. Cyanide solution is added to the slurry in tank 1 at a controlled level to promote leaching of gold and silver. The tanks are agitated by dual prop mixers and compressed air is added to maintain oxygen saturation. At a nominal processing rate of 2075 t/d these tanks provide a total of 52 hours residence time.

Leached slurry flows to the first tank in a series of six carbon-in-pulp (CIP) contactors, comprising agitated tanks 17.5 ft dia by 17 ft high equipped with carbon retention screens where the slurry is contacted by activated carbon granules at a concentration averaging 15 g/L. The carbon adsorbs the dissolved gold and silver and is removed periodically from the first tank when its metal loading attains the desired level. The carbon in the sequential tanks is advanced periodically by pumping counter current to the slurry flow, to maintain the concentration in each tank, and fresh or recycled carbon is added regularly to the last tank.

The loaded carbon is washed on a screen to remove slurry then is pumped to a vessel for acid washing, to remove lime scale, then is pumped to the carbon stripping vessel. When the quantity in the vessel is close to 5 t of carbon the stripping process is initiated. Under pressure, strip solution comprising water at 140 degrees C and containing sodium hydroxide and cyanide, is pumped through the carbon bed in the vessel to extract (elute) the gold and silver from the carbon. An elution cycle requires approximately 8 hours. On completion of elution the carbon is washed with water, the pressure removed from the vessel, and the stripped carbon is pumped to a holding vessel. The circuit includes electrical heaters and heat exchangers to both heat the solution entering the strip vessel and to cool it upon exit.

Solution from the elution vessel flows to an electrowinning cell in the refinery where the gold and silver are precipitated from solution by electrolysis. The barren solution returns to the elution system where after heating it is recycled to the stripping vessel.

Approximately once per week the metal sludge is pumped from the electrowinning cell and recovered in a filter press, then dried, mixed with fluxes and melted in an electric

induction furnace to produce bullion and slag. Bullion is poured into molds to form gold bars for shipment.

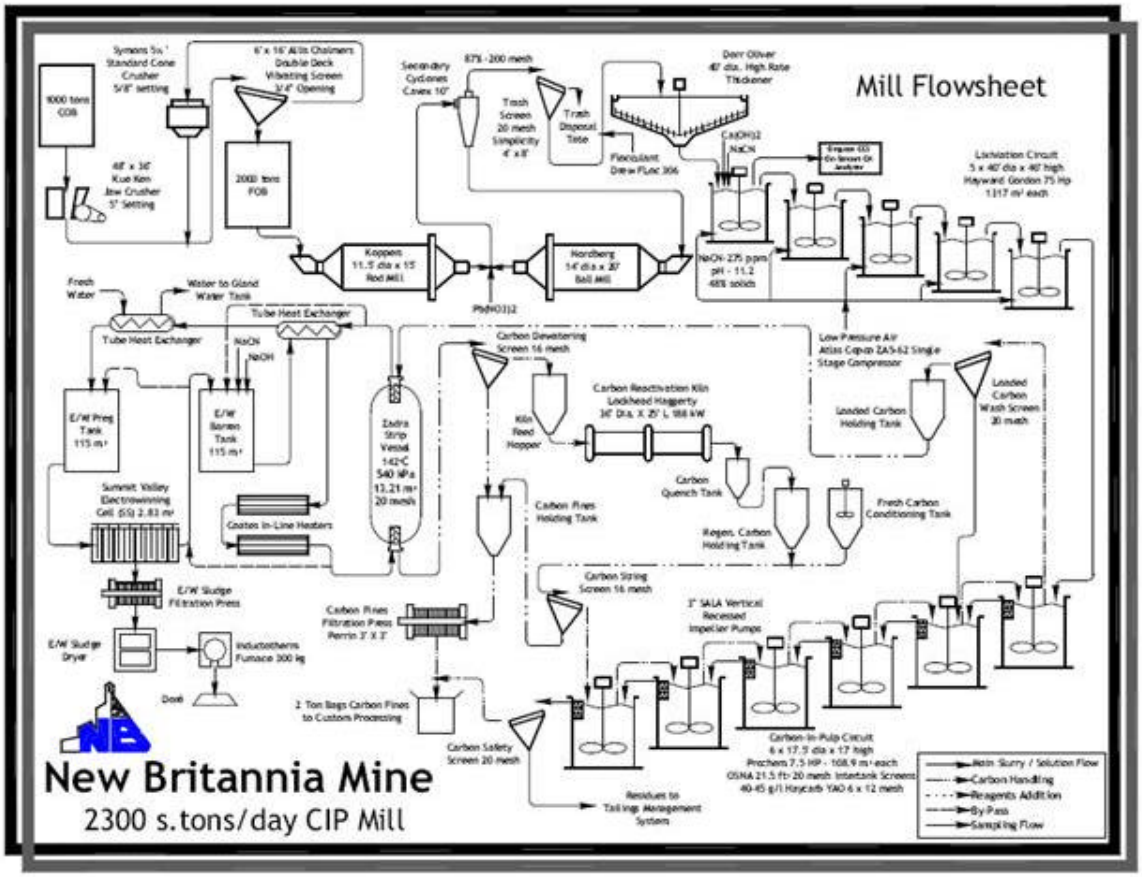

Figure 18-16: Snow Lake Mill Process Flowsheet

The carbon after elution to remove gold is screened and the adsorption activity restored by heating in a rotating, electrically heated kiln to 700 degrees C. Carbon discharging from the kiln is quenched in a tank with water then pumped to a storage vessel. Additional fresh carbon, to make up for losses of fines, is attritioned in an agitated tank and also screened then added to the storage vessel. The regenerated and fresh carbon is periodically delivered to the last CIP stage in sequence with carbon advance through the process.

Carbon fines resulting from the preparation and handling of carbon, which contain several Oz per t of gold/silver, are recovered by pumping solutions through a filter press. The fine carbon cake is saved for periodic sale to a smelter and payment for the small amount of contained gold.

Slurry tailings discharged from the last CIP tank are screened to recover any fine carbon and pumped to the tailings storage area at approximately 35% solids. A cyanide destruction system is in place, comprising a reaction tank, SO2 storage and

feeding system and compressed air are used together with copper sulphate and lime addition to reduce the cyanide content of the slurry to below 5 ppm CN Total. This system is available for use if the natural degradation employed in the tailings area is not successful in maintaining the required effluent quality.

Water for the process is supplied mainly by reclaim from the tailings area water pond and is pumped to the mill process water tank for distribution via the process water pumps. Fresh water is added to the process via slurry pump gland seals, for mixing reagents and in the carbon stripping process. Water pumped from the mine contains slimes and is used in the grinding circuit.

Reagent make up facilities include a lime silo and slaking system, a cyanide mixing (from solid sodium cyanide) and distribution system, lead nitrate and caustic soda mixing systems. Air for leaching is sullied from two dedicated low pressure compressors.

Maintenance of the milling equipment is conducted in a shop area within the mill building.

18.4.3 Plant Design

The Snow Lake mill has previously operated successfully treating both Main Zone and 3 Zone ores and the process design is appropriate for renewed treatment of these ores. No changes to the milling process are required to obtain optimum gold recovery and operating cost. The average design processing rate is 2075 t/d which has been attained easily in previous operations with the plant.

The 3 Zone mineralization is significantly harder than Main Zone and requires a relatively fine grind to yield gold recovery above 90%. This can be achieved with the existing grinding equipment operating at maximum media loads and power input. The plant will be operated to obtain a grind of 55 microns K80 with all ore blends to obtain optimum gold recovery. With the blend of 3 Zone with Main Zone ore, the power requirement will be below maximum, and with Main Zone only will be lower still.

During Year 0, as the ore supply will be restricted, the mill will start at a lower rate of 1300 t/d and be operated 3 or 4 days per week with a reduced personnel roster. Production rate will be ramped up to 2075 t/d gradually during the first four months and additional operating crews assigned as the production approaches 7 days per week.

18.4.3.1 Scope of work for Plant Modifications and Start up

The existing crushing plant and mill are in good condition without many items requiring significant repairs. During the previous production stoppage the mill and tanks were cleaned. Some residue remains from

the short milling campaign in 2005. The grinding mills have been emptied of media and liners and jacked off the bearings. The building has been heated during the shut-down period which should result in most electrical motors remaining in good condition. All of the conveyors are in good condition. Some of the instruments and control systems are obsolete and will be replaced.

Planned major items of repair include:

- Dismantling the jaw and cone crushers to inspect liners, bearings, drives and check out lubrication systems
- Repair of ball mill motor, will require dismantling and rewinding at contract shop
- Possible repair of rod mill motor and/or drive system
- Replacement of liners in both grinding mills, installation of the mills on the bearings, checking drives and lubrication systems, and charging with media
- Emptying and repair of thickener tank and rake mechanism
- Replacement of a number of leach tank agitator blades
- Replacement of two carbon retention screens
- Replacement of several CIP agitator blades
- Inspection of all pumps and replacement of worn liners and seals
- Installation of new tailings and reclaim water pipelines

As part of the refurbishing plan, all motors, drives and instruments will be checked and repaired as required but no significant costs are expected.

With the exception of the repairs to the mill motors, all of the planned work is within the capabilities of a typical mill maintenance crew and it is intended to hire this crew early, supplanted by some additional labour for cleaning, and commence the rehabilitation work approximately three months in advance of the planned startup. Some of the non-critical items can be completed during the startup period. A typical crew will comprise 4 mechanics, two electricians and 2-4 labourers predominantly for cleaning and to assist the mechanics. Supervision of the crew by the Master Mechanic and support from Purchasing for parts supply will be required.

For the complete rehabilitation exercise, the work by the maintenance crew is estimated at 4885 man-hours.

Process Equipment

The process equipment existing in the crushing plant and mill is suitable for the renewed operation of the mill with the ore types planned and at the planned processing rate of 2075 t/d. After inspection, cleaning and repair noted above, the existing equipment will be adequate to obtain the planned mill performance. No changes to the equipment are contemplated for future operations.

Power Supply

The existing power supply to the mill and the distribution switchgear within the mill are suitable for the renewed operation. The original system has continued in use since the shutdown in 2004. Inspection of the supply side power system and main transformer may be required by Manitoba Hydro due the low use since 2004. The planned rehabilitation will include inspection of all the distribution switchgear and cabling.

The total installed motor and lighting, heating load in the mill is 7506 hp or 5600kW. After allowance for spare equipment and periodic operations, the average full load draw is 4238kW. For average operating conditions an average motor draw of 80% of full load is assumed for an average draw of 3390kW.

The mill power requirement will vary slightly due to the blend of ore types in order to maximize fine grinding. The predicted power requirement for each ore blend, at the planned full processing rate of 2075 t/d, is shown below:

	3 Zone	Main	blend
kW demand	3548	3044	3226
kWh/t	41,0	35,2	37,3

18.5 Tailings disposal (mp)

18.5.1 Tailings Management Background

The Snow Lake Mine has a long history attributed to two periods of previous gold production. The initial operation occurred from 1949 to 1958 during which approximately 5 million t of ore were mined. Tailings from the initial operation were mainly discharged into a natural lake (Birch Lake basin) north of the mine although some tailings were also discharged to a low lying swampy area east of the original mill. The mine was re-opened in 1995 and operated until 2005 as the New Britannia Mine. During this second period of operations, tailings disposal was re-activated within

the Birch Lake basin. In order to do this, tailings/water impoundment structures were constructed to provide additional containment for tailings solids, supernatant and runoff water.

The design for the New Britannia Mine Tailings Disposal Facility was completed by DNE Knight Piesold in 1995 with an initial phase of construction completed in 1995 and a second phase completed in 1996. The 1995 design was based on providing capacity for approximately 5.4 million t at an estimated density of 1.4 t per cubic metre (dry) or about 4 million cubic metres. The 1995 design included the construction of 5 impoundment structures, of which two were at the southeast end of the basin where the tailings was discharged and three were constructed as water retaining dams at the northwest end of the basin where supernatant and runoff water collected. As a result, an exposed tailings beach slopes from the southeast end of the basin towards the northwest where a supernatant pond is located.

During the New Britannia Mine operations, some internal waste rock fill berms were constructed to create an internal polishing pond and prevent tailings from settling within the supernatant pond. Dam Safety Inspections were completed at least every two years, with the most recent being completed in July 2008 by Golder Associates Ltd. All impoundment structures were noted to be in generally good condition and instrumentation readings were within expected limits.

18.5.2 Current Status

A complete site inspection of the TMA and associated facilities was completed by Knight Piesold on July 29, 2010. This included an inspection of all perimeter containment dams with documentation of conditions through digital photos and handheld GPS tracking. The following comments are provided with respect to the site visit findings:

- In general all of the perimeter retaining dams were found to be in good condition with no evidence of instability or excessive settlement. Visual observations of seepage flows indicate that embankment seepage rates are not high.
- The tailings delivery and reclaim water pipelines have been removed and new pipeline systems will need to be installed for the proposed new operation.
- The tailings solids capacity within the TMA has been fully utilized in the southeast end of the basin (from Clay Berm to Birch Lake Causeway).
- Additional tailings solids capacity is available within the central part of the basin by discharging tailings slurry to the north from the Birch Lake Causeway. A second waste rock fill dyke, which is partially constructed to the north, could be utilized to prevent solids from inundating the existing supernatant pond.
- There is also potential tailing solids storage available in the west end of the basin by discharging from the Canada Creek Cut-off Embankment. This will result in the supernatant pond being reduced in area and volume. In addition, the water

reclaim intake would need to be moved to allow water to be collected and pumped back to the mill for the process.

18.5.3 Tailings Management Design Criteria

Based on information provided by Alexis, the following project development criteria have been established for the Snow Lake Mine TMA feasibility design:

- Total projected ore / tailings tonnage - 3,200,000 t (3.53 million tons).
- Anticipated mine life - 6 years.
- Production rate / mill throughput - design flow of 2,075 t/day with average of 1,500 t/day over 6 years.
- Tailings Slurry Consistency - 35% solids by weight.
- Estimated tailings settled in-situ density - 1.5 t/m^3 (94 lbs per cubic ft) - based on this density, a total of 2.13 million m3 or 75 million ft^3 of solids storage volume will be required.
- Reclaim water flow rate - 75% average with maximum up to 90% of process flow.
- Mine water flow – Initially at 2,200 m3/day (400 usgpm for 20 hours per day) for initial mine dewatering followed by expected low flows during mining operations (reported to be a dry mine).

The main update to the Design Criteria from the 1995 design involves conforming to the 2007 Canadian Dam Association (CDA) guidelines to determine the hazard consequence classification and rating for the TMA. The original criteria adopted for the 1995 design include:

- Environmental Design Storm (EDS): The TMA was designed to contain this event, which consists of the runoff from the 20 year wet precipitation event together with the 1 in 100 year, 24 hour storm in October when the annual water level is the highest. Any flows from storms greater than this event would be discharged over the emergency spillway.
- Inflow Design Flood (IDF): This event was derived from the 24 hour Probable Maximum Precipitation event with a depth of 26 inches and runoff calculations to determine a design inflow of 2,100 cubic ft per second (cfs) at the emergency spillway location.
- Maximum Design Earthquake (MDE): The 1 in 10,000 year earthquake was selected. This event has a horizontal acceleration of 0.05g and is equivalent to the Maximum Credible Earthquake (MCE).

The 2007 CDA guidelines were reviewed to determine the updated Dam Consequence Classification category for the TMA. The guidelines indicate that the TMA is in the HIGH category based on the following Table 18-9.

Table 18-9 Extract of the 2007 CDA Guidelines

Dam Consequence Classification Category	Population at Risk	Loss of Life	Incremental Losses Environmental and Cultural Values	Incremental Losses Infrastructue and Economics
HIGH	Permanent	10 or feweer	Significant loss of deterioration of important fish or wildlife habitat. Restoration or comensation in kind highly possible.	High economic losses affecting infrastructure, public transportation and commercial facilities.

Is it highly likely that population would not be at risk or that loss of life would occur in the event of dam failure. The HIGH rating is mainly due to the existence of important fish habitat (existing commercial fishery of Herblet Lake) downstream of the TMA. Based on a HIGH consequence classification category, an IDF of 1/3 between the 1 in 1,000 year storm and the PMF and a MDE of 1 in 2500 years are specified. Therefore, the existing IDF and MDE criteria will satisfy the 2007 CDA guidelines for the re-activation design of the TMA.

18.5.4 Tailings Management Design Concept

An initial options review was completed by Knight Piesold to determine the preferred option for Snow Lake Mine TMA development. The chosen option involves maximizing use of available capacity within the existing TMA and eliminates the need to raise perimeter dams by constructing an internal waste rock fill berm to generate additional solids storage capacity. A final arrangement for the chosen development scenario is shown on Figure 18-17 and includes the following development phases.

1) Phase 1 - Start tailings discharge within the Central Portion of the TMA from the existing Birch Lake Causeway to the northwest. This requires no earthworks and minimizes start-up pipe work for tailings and reclaim water systems.
2) Phase 2 - Move the tailings discharge to a newly constructed Southeast waste rock fill dyke. This can be done without interrupting the reclaim water system and with minimal disruption to tailings discharge as only about 1,300 m of pipeline will require relocation. The waste rock fill dyke can be stage constructed to further offset ongoing capital costs for the TMA during the Phase 2 deposition period.
3) Phase 3 – Move tailings discharge to the West End of the TMA. This will require relocation of 2,100 m of tailings pipe and addition of 1,400 m of new tailings pipe. In addition, approximately 415 m of new perimeter access road will need to be constructed to accommodate tailings deposition pipe around a north bay within the

basin. A new reclaim intake structure will need to be constructed to the east of the existing intake tower along with 700 m of new access road, pipeline and power line to the new intake location. Deferral of these significant work items to later in operations is judged to be attractive due to the costs and larger risk of longer interruption to production. In addition, other options for extending the life of the TMA can be explored further prior to Phase 3 design and construction.

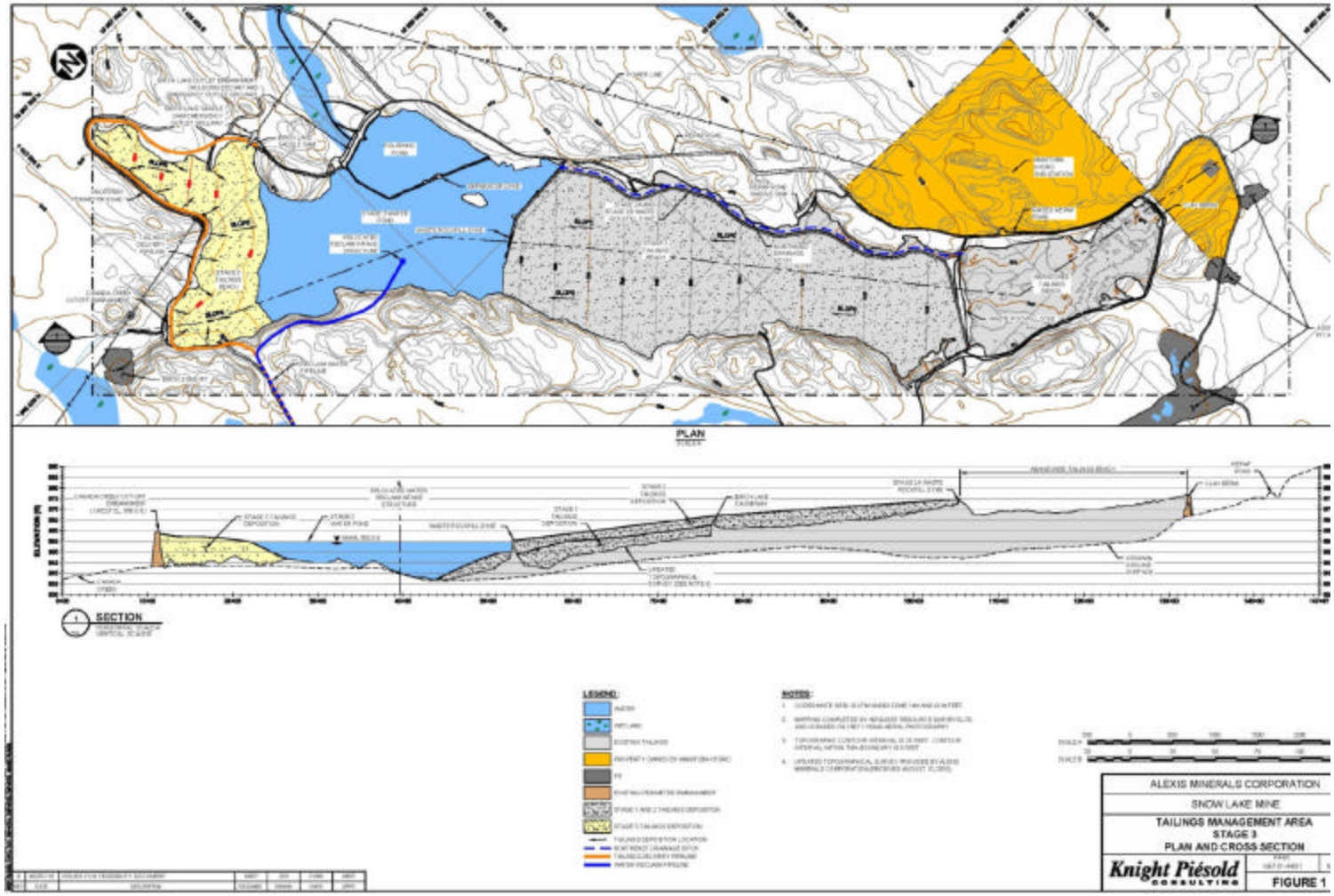

Figure 18-17: Tailings Disposal Areas Development Scenario

18.5.5 Tailings Deposition Plan

The tailings deposition plan has been based on the following general assumptions:

1) The tailings properties including grain size and beach slopes will be very similar to that generated during previous operations.

2) Recent testing on the tailings beach indicates that a in-situ dry density of 1.5 t/m^3 was achieved during past operations. This value has been used for future capacity projections.
3) The tailings solids are not acid generating.

4) Ferric Sulphate will be added at the tailings sump to precipitate arsenic within the settled tailings solids.

5) Due to the large supernatant pond size, it is proposed that cyanide will be managed by natural destruction from sunlight in the summer months. Cyanide destruction equipment will be maintained and operated if necessary to maintain acceptable cyanide levels.

Deposition of tailings into the Snow Lake Mine TMA will be conducted in a similar manner to previous operations in that the slurry will be discharged from a raised dyke to create a sloping beach that is largely exposed above the water pond. This allows the tailings to air dry and helps to maximize density. Some of the tailings will be deposited within the existing water pond, particularly during Phase 1 and Phase 2. This will reduce the size of the existing water pond, but due to its large size it is judged this will not impact achievable water quality significantly.

18.5.6 Water Management Plan

The main objectives of the water management plan for the TMA are as follows:

1) Minimize discharge of process impacted water to the environment by maximizing reclaim back to the mill process.

2) Maintain suitable water quality for discharge to environment through use of cyanide degradation and arsenic precipitation.

3) Provide suitable freeboard and emergency overflow spillway capacity to contain and/or route extreme flows from the TMA in a controlled and stable manner.

A water balance analysis was completed using average precipitation and evaporation conditions to estimate the amount of water that is anticipated to report to the TMA over the course of the mine life. The results of the water balance analysis are summarized as follows:

- In order to maximize reclaim, the TMA will generally need to be operated to maximize the pond level going into the fall or start of winter so that an adequate volume of water is available through the winter due to ice formation losses.
- The annual volume of available reclaim water is estimated to be from 76% to 88% of the process water requirements under average precipitation conditions. There may be some months, during the winter in particular, where the reclaim rate will be less than 76% of the process water requirements.
- Discharge of water from the TMA to the environment will likely be conducted in the late summer to early fall once cyanide levels are low from natural degradation.

Storm water analyses were completed to ensure that the TMA will be able to contain the EDS and pass the IDF. From a containment perspective, runoff up to the EDS will need to be routed from the southeast end of the TMA to the supernatant pond along the north side of the facility. This will require some upgrades and improvements to the channel that is left between the waste rock dyke and north basin slope. For the runoff in excess of the EDS and up to the IDF, it is proposed that a new spillway be constructed at southeast end of the TMA. All runoff from the IDF reporting to the supernatant pond will discharge from the existing Birch Lake Outlet Embankment emergency overflow spillway and the new proposed spillway.

18.6 Environmental and social impact (mb)

18.6.1 Environmental

In the Province of Manitoba, the following environmental permits are required to operate a mine:

- Environmental Act License
- Water Rights License
- Waste Disposal License
- Waste Generator License
- Sewage Disposal License

18.6.1.1 *Closure Plan*

Environmental Act License

Currently the License # is 1878-S3RR and was originally issued in October 1995 with revisions in 1996 and 1999. This License grants permission to discharge into the environment a deleterious substance or a substance which may have an adverse affect. The permit will outline the criteria which must be met in order to operate. Although the mine is on care and maintenance, the permit does still apply to the site. This

permit can be transferred to Alexis Minerals without issue as long as the operating parameters have not changed and all conditions on the permit remain the same.

Water Rights License

Currently the License # is 95-18 and was originally issued in 1995. This License grants permission to use surface and/or groundwater for industrial purposes– the regulations state that: "Any individual or corporation who wishes to use water for agricultural, municipal, industrial, irrigation or other purposes must first obtain a license. "

In speaking with the Water Resources Branch of Manitoba Conservation, the transfer of the Water Rights License to Alexis Minerals is possible without issue as long as the operating parameters have not changed and all conditions on the permit remain the same.

Waste Disposal License

Currently the License # is 5-060 and was originally issued in April 1996. This License grants permission to operate a waste disposal site at the mine. In speaking with the Environmental Officer from Manitoba Conservation, the transfer of the Waste Disposal License to Alexis Minerals is possible without issue. We were advised that the permit and some of the operating parameters may change to reflect the updates in the regulations and the required operating procedures, however these changes will not be major and should not affect the operation of the site.

Waste Generator License

Currently the License # is MB1002341 and was originally issued in September 1994. This License grants permission to generate and dispose of, through approved methods, hazardous wastes. This permit can be transferred to Alexis Minerals without issue as long as the operating parameters have not changed and all conditions on the permit remain the same.

Sewage Disposal License

Currently the License # is 01-TPD-97-31 and was originally issued in November 1997. This License grants permission to dispose of sewage from the mill and offices. This permit can be transferred to Alexis Minerals without issue as long as the operating parameters have not changed and all conditions on the permit remain the same.

Closure Plan

Under Manitoba Mine Closure Regulation 67/99, prior to a mine being granted permission to operate, a Closure Plan along with the financial assurance must be filed with the Government. During our review, we found a closure plan which has been accepted and a financial assurance

of $1.8M CDN which has been filed. The mine is presently on a Care and Maintenance mode, therefore, notification with regards to the new ownership and the change from a Care and Maintenance mode to a production mode is required.

18.6.1.2 *Federal Regulations*

Metal Mining Effluent Regulations

Federally, Environment Canada, under the MMER (Metal Mine Effluent Regulations), govern the discharge of effluent during the operation of a mine. These regulations have been in effect since 2002 and the SLM certainly fall under these regulations and the respective requirements. The requirements of the MMER are staged and the SLM have completed Cycles 1 and 2 in 2006 and 2009 and have been submitting the required annual reports up to 2009 with relatively minor issues downstream of the site which have been attributed to historical operations. The next cycle will be due in 2012.

Notification to Environment Canada with regards to mine ownership will be required as well as a part of the transfer.

18.6.1.3 *Social*

The various social impacts/requirements were outlined in the 2010 Preliminary Assessment and have not changed since that time. Public notification for the Municipal Government of Snow Lake should continue at regular intervals with the various milestones and major decisions of the project. In addition, it is recommended that an open house with the residents of the Town of Snow Lake be undertaken at some point once the decision is made to open the mine.

With regards to First Nations, we agree with the Golder Associcates Preliminary Report which stated:

"The mine has been operational for an extended period of time under different owners and few Aboriginal concerns have been raised. There are no Indian Reserves (IR) located within the immediate vicinity of Snow Lake. However, four IR's are located within a 100 km raidius of the Town. The nearest IR is Opekanowi Sakaheyn IR # 09797 located approximately 80 km north of Snow Lake. The three remaining IR include Napahkapihskow IR # 09851 (approximately 85 km northwest of Snow Lake), Wepuskow Ohnikahp IR # 09851 (approximately 97 km north west of Snow Lake), and Mistiategameek Sipi IR # 09878 (approximately 92 km north of Snow Lake). The area is currently considered disturbed industrial development land. As the project is not expected to result in disturbance to new areas, new concerns on behalf of First Nation communities are not anticipated."

18.7 Project schedule (ar)

The Snow Lake Mine re-activation project will last 6 years in total, i.e. 1 year of pre-production and 5 years of production. Figure 18-18 shows a condensed schedule for the first 2 years of the project, i.e. the year of pre-production (year 0) and the first year of production (year 1). It shows the main steps to be accomplished before production can start in the No.3 Zone and subsequently in the Main Mine. The schedule also shows when the mill refurbishment will be carried out.

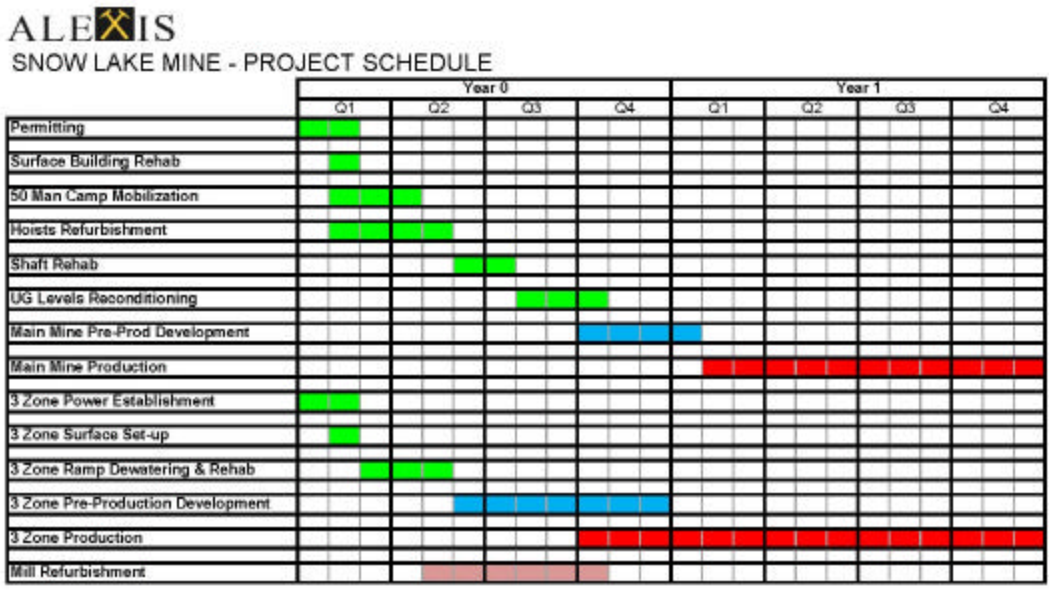

Figure 18-18: Project Schedule

In the No. 3 Zone the main steps are:

- Power establishment;
- Surface set-up;
- Ramp dewatering & reconditioning; and
- Pre-production development.

The schedule shows that in the No. 3 Zone production will start in the 4th quarter of the first year. However the Snow Lake Mine (as a whole) will only fulfill the criteria for officially being in production in the 1st quarter of the second year.

Production in the No. 3 Zone will end during the 1st quarter of year 3.

In the Main Mine the main steps before production starts are:

- Permitting;
- Building reconditioning;
- Camp set-up;
- Hoist refurbishment;
- Shaft reconditioning;
- U/G Levels reconditioning; and
- Pre-production development.

Production will start in the 2nd quarter of year 1 and will last until the 4th quarter of year 5.

The refurbishment of the mill will take 6 months. The mill will start processing ore during the 4th quarter of year 0.

18.8 Capital cost estimate

18.8.1 Mine (AR)

GENERAL

A summary of the Capital Costs is presented in Table 18-10. Net initial Capital expenditures are CAN$ 40.8 million and subsequent sustaining capital is CAN$ 33.9 million, bringing Total Capital expenditures for the duration of the project to CAN$ 74.7 million.

Table 18-9 : Capital Costs Summary

	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Pre-Production	13 800 362						13 800 362
Main Mine							
Equipment & Reconditioning	12 149 554	8 488 923	80 000			300 000	21 018 477
Development			3 360 740	638 480			3 999 220
Capital Development Allocation			2 878 967	623 601			3 502 568
Total Main Mine	12 149 554	8 488 923	6 319 707	1 262 081		300 000	28 520 265
No. 3 Zone							
Equipment & Reconditioning	4 820 835	5 131 599	525 000				10 477 434
Development	2 128 000	4 307 000	1 289 960				7 724 960
Capital Development Allocation	0	3 711 884	1 105 040				4 816 924
Total No. 3 Zone	6 948 835	13 150 483	2 920 000				23 019 318
Mill	2 565 000						2 565 000
Tailings Management	25 000		101 000	137 000	237 000		500 000
Warehouse First Fill	1 600 000	1 000 000					2 600 000
Total before contingency	37 088 751	22 639 406	9 340 707	1 399 081	237 000	300 000	71 004 945
Contingency (10%)	3 708 875						3 708 875
Total	**40 797 626**	**22 639 406**	**9 340 707**	**1 399 081**	**237 000**	**300 000**	**74 713 820**

Note: All figures in CAN$

Net pre-production expenditures amount to CAN$ 13.8 million after a credit for pre-production gold revenues of CAN$ 7.9 million (a detail of those expenses is given in the Operating Cost Summary in Section 18.7). The Capital cost to equip and recondition the Main Mine is $12.1 million and CAN$ 6.9 million for the No. 3 Zone. Details of the Main Mine and No. 3 Zone equipment and reconditioning costs are given in Tables 18-11 and 18-12 and are discussed in subsequent paragraphs. It should be noted that in order to keep the initial capital investment as low as possible Alexis has elected to rent most the equipment required during year 0 both in the Main Mine and in the No. 3 Zone. This equipment is later purchased in year 1 and 2 when production starts to generate a regular cash flow.

The Mill, Tailings Disposal area and Warehouse first fill respectively amounts to CAN$ 2.6 million, CAN$ 25,000 and CAN$ 1.6 million. Finally a contingency of CAN$ 3.7 million was included in the initial capital expenditures (no contingency was included in the sustaining capital estimates).

In year 1 of operation the sustaining capital expenditures are estimated to CAN$ 22.6 million. This is made up of three components. The first is CAN$ 8.5 million to complete equipment purchases and reconditioning in the Main Mine. The second is CAN$ 13.2 million in the No. 3 Zone to continue equipment purchase and reconditioning (CAN$ 5.1 million), development (CAN$ 4.3 million) and Capital Development Allocation (CAN$ 3.7 million). The last component is CAN$ 1,0 million to complete the warehouse first fill.

In year 2 of operation the sustaining capital expenditures drops to CAN$ 9.3 million. The Main Mine requires CAN$ 6.3 million in Development and associated Capital Development Allocation whereas CAN$ 2.9 million is spent in the No. 3 Zone, including CAN$ 0.5 million to complete the equipment purchase.

Sustaining Capital expenditures drop sharply in the third year of operation to CAN$ 1.4 million. The completion of the development program in the Main Mine requires CAN$ 1.3 million, leaving CAN$ 0.1 million for the Tailing Management Area.

The only forecasted sustaining capital expense during year 4 is CAN$ 0.2 million for the Tailing Management Area.

Again in year 5 there is only one forecasted sustaining capital expense: CAN$ 0.3 million to dismantle the camp.

MAIN MNE EQUIPMENT & RECONDITIONING

Details of the Main Mine Equipment and Reconditioning Capital costs are given in Table 18-11. The bulk of the Capital Expenditures of CAN$ 21.0 million is related to the underground mine, i.e. CAN$ 19.9 million vs CAN$ 1.1 million for the surface

Table 18-10 : Main Mine – Equipment & Reconditioning Capital Costs Break Down

	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
SURFACE							
General							
Office and Dry	36 238						36 238
Fresh Water Supply System	100 000						100 000
Compressor Building	62 587						62 587
Camp Set-Up	300 000					300 000	600 000
Total General	498 825					300 000	798 825
Fixed Equipment	295 000						295 000
TOTAL SURFACE	793 825					300 000	1 093 825
UNDERGROUND							
Hoist & Mine Reconditioning							
Hoist	1 059 381						1 059 381
Shaft	843 137	250 000					1 093 137
Loading	78 550	21 300					99 850
Levels	2 511 457	791 738					3 303 195
Ramp		886 885					886 885
Vent Circuit	73 640	223 640					297 280
Electrical Circuit	1 414 600	894 100	80 000				2 388 700
Total Hoist & Mine Reconditioning	5 980 765	3 067 663	80 000				9 128 428
Contractor Indirects	4 356 000						4 356 000
Equipment							
Fixed	400 000						400 000
Mobile	618 964	387 200					1 006 164
Equipment Buy-Out		5 034 060					5 034 060
Total Equipment	1 018 964	5 421 260					6 440 224
TOTAL UNDERGROUND	11 355 729	8 488 923	80 000				19 924 652
TOTAL MAIN MINE	**12 149 554**	**8 488 923**	**80 000**			**300 000**	**21 018 477**

Note: All figures in CAN$

Almost all the investments related to the surface infrastructure, CAN$ 0.8 million, are made in Year 0 to repair the office and dry, re-establish the fresh water distribution system, the compressor building, establish the camp and buy some fixed equipment. The only exception is CAN$ 0.3 million spent in year 5 to dismantle the camp.

The capital expenditures directly related to the reconditioning of the underground infrastructures amount to CAN$ 9.1 million almost all spent in Year 0 (CAN$ 5.9 million) and year 1 (CAN$ 3.1 million). Only CAN$ 0.8 million is spent in Year 2 to complete the reconditioning of the electrical circuit. A major cost of the reconditioning work is the Contractor indirect costs amounting to CAN$ 4.4 million, all spent in Year 0 as Alexis is planning to finish off the reconditioning work with its own personnel thereafter.

The purchase of equipment for the underground mine will cost CAN$ 6.4 million in total. Most of this expense (CAN$ 5.0 million) is delayed in Year 1 to buy-out the equipment rented during Year 0. This leaves CAN$ 1.0 million in Year 0 to buy fixed and mobile equipment and additional CAN$ 0.4 million in Year 1.

No. 3 ZONE EQUIPMENT & RECONDITIONING

Details of the No. 3 Zone Equipment and Reconditioning Capital costs are given in Table 18-12. Again the bulk of the Capital Expenditures of $10.5 million is related to the underground mine, i.e. $9.5 million vs $0.9 million for the surface.

Table 18-11 : No. 3 Zone – Equipment & Reconditioning Capital Costs Break Down

	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total
Surface Genral	755 750	188 500					944 250
Undergroud							
Dewatering & Recontioning	3 306 600						3 306 600
Equipment							
Fixed	477 698	299 349					777 047
Mobile	280 787	21 650					302 437
Buy-Out		4 622 100	525 000				5 147 100
Total Equipment	758 485	4 943 099	525 000				6 226 584
Total Underground	4 065 085	4 943 099	525 000				9 533 184
Total No. 3 Zone	**4 820 835**	**5 131 599**	**525 000**				**10 477 434**

Note: All figures in CAN$

In Year 0 the surface related expenditures will amount to CAN$ 0.8 million to recondition the ramp portal, install an electric substation, install compressors and the ventilation system. All the Year 1 surface expenditures of CAN$ 0.2 million are related to electricity.

The cost of dewatering and reconditioning the No. 3 Zone workings are estimated at CAN$ 3.3 million to be spent in Year 0. Alexis is planning to have this work performed by the same contractor that will carry out the Main Mine reconditioning.

The total cost to purchase the fixed and mobile equipment is CAN$ 6.2 million. Most of this expense is in Year 1 (CAN$ 4.9 million) and Year 2 (CAN$ 0.5 million) to buy-out the rented equipment. The direct purchase of fixed and mobile equipment is limited to CAN$ 0.8 million in year 0 and CAN$ 0.3 million in Year 1.

18.8.2 Mill (IW)

18.8.2.1 *Basis of Estimate*

The basis of the capital cost estimate for the plant rehabilitation is developed as follows:

- A detailed inspection to generate a work list for each item of equipment and service facilities such as building, offices;
- Inspection of the grinding mill motors by GE technician;
- Estimated manhours for each task based on experience with similar projects and normal Manitoba mining area productivity;
- Allowance for unproductive labour time due to awaiting instruction, collecting parts etc;
- Current mid-2010 area labour rates and 32% payroll burden, assuming direct hire workers, based on normal 40 hour work week with no overtime payments;
- Major replacement parts and materials costed with quotations from suppliers;
- Freight cost allowed at 10%;
- All taxes are excluded;
- Contingency allowance is excluded;

- Costs are expressed in current, mid-2010 CAN$;
- Estimate precision is judged to be appropriate for Feasibility study at +/- 15%

18.8.2.2 *Capital Cost Estimate*

The estimated cost to rehabilitate the existing plant comprises labour, parts and materials costs.

Labour costs are developed based on current typical area rates for direct hire employees and include company paid benefits and overhead. A crew rate has been developed including part time supervision.

Costs fo spare parts are developed from the estimate of items known to require replacement plus an allowance for expected replacement items such as a certain quantity of pump liners, equipment seals and bearings etc.

Material and contract costs include contracted work for such items as the mill motor repair and laying of pipeleines, and costs of bulk materials such as pipe, building repair material, grinding media and general material consumed in overall inspection.

A summary of the estimated capital cost follows in Table 18-13.

Table 18-12 : Estimated Plant Rehabilitation Capital cost

Area	CAN$
Labour	147,100
Parts	575,100
Materials and contracts	1,539,500
Total cost	**2,261,700**
First fill	**332,400**
Warehouse inventory	**500,000**

A contingency allowance should be added to the total to allow for missed items and unexpected delays and repairs.

18.9 Operating cost estimate

18.9.1 Mine (AR)

The Operating Cost Summary is presented on Table 18-14. Year 0 Operating Costs are CAN$ 21.7 million after a credit of $7.8 million for the gold production during this period. The net capitalised Pre-Production cost reported on the Capital Cost Summary is $13.8 million.

Table 18-13 : Operating Costs Summary

	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Total 1 - 5	Total
Main Mine								
Ore (t)	8 202	353 193	467 175	665 279	716 683	633 656	2 835 986	2 844 188
Definition Drilling	32 809	1 412 774	1 868 700	2 661 116	2 866 733	2 534 624	11 343 947	11 376 756
Operating Developemnt	1 630 500	14 723 100	11 847 450	10 364 040	2 174 760		39 109 350	40 739 850
Stoping		3 989 247	6 350 819	12 653 692	18 263 172	23 122 919	64 379 849	64 379 849
Rock Movement	11 515	410 893	629 875	2 376 707	2 855 269	5 157 516	11 430 260	11 441 775
Mine Services	4 744 384	8 848 102	9 145 020	13 477 931	13 340 813	10 660 627	55 472 493	60 216 877
Capital Development Allocation			-2 878 967	-623 601			-3 502 568	-3 502 568
Total Main Mine	6 419 208	29 384 116	26 962 897	40 909 885	39 500 747	41 475 686	178 233 331	184 652 539
Cost per t mined (CAN$/t)	782,64	83,20	57,71	61,49	55,12	65,45	62,85	64,92
No. 3 Zone								
Ore (t)	62 176	255 000	270 001	46 001			571 002	633 178
Definition Drilling	248 702	1 020 000	1 080 004	184 004			2 284 008	2 532 710
Operating Developemnt	1 868 000	2 971 000	1 779 000				4 750 000	6 618 000
Stoping	929 571	4 359 362	4 865 582	897 020			10 121 964	11 051 535
Rock Movement	1 395 950	3 227 837	2 672 990	455 410			6 356 237	7 752 187
Mine Services	4 744 384	6 816 371	5 285 308	931 939			13 033 618	17 778 002
Capital Development Allocation		-3 711 884	-1 105 040				-4 816 924	-4 816 924
Total No. 3 Zone	9 186 607	14 682 686	14 577 844	2 468 373			31 728 903	40 915 510
Cost per t mined (CAN$/t)	147,75	57,58	53,99	53,66			55,57	64,62
Milling & Processing								
Ore (t)	46 302	632 369	737 176	711 280	716 683	633 656	3 431 164	3 477 466
Gold (Oz)	6 004	76 620	95 337	83 746	83 120	76 761	415 584	421 588
Milling & Processing	1 035 911	8 485 041	9 157 157	8 518 202	8 405 726	7 866 012	42 432 138	43 468 049
Administration	3 810 000	3 810 000	3 810 000	3 810 000	3 810 000	3 810 000	19 050 000	22 860 000
Accommodation	1 212 321	2 694 048	2 694 048	1 616 429	538 810		7 543 335	8 755 656
Total	**21 664 047**	**59 055 891**	**57 201 946**	**57 322 889**	**52 255 283**	**53 151 698**	**278 987 707**	**300 651 754**
Cost per t milled (CAN$/t)	**467,89**	**93,39**	**77,60**	**80,59**	**72,91**	**83,88**	**81,31**	**86,46**
Cost per Oz (CAN$/Oz)	**3 608**	**771**	**600**	**684**	**629**	**692**	**671**	**713**

Note 1: All figures other than t, Oz, CAN$/t, CAN$/Oz are CAN$

Note 2: Year 0 operating cost less a credit of CAN$ 7,863,685 for gold produced yield a pre -production capital of CAN$ 13,800362

Note 3: A stockpile of 24,176 t is milled remaining at the end of Year 0 is milled at the beginning of Year 1

The yearly operating costs remain fairly constant, varying from CAN$ 59.1 million in year 1 to CAN$ 52.3 million in year 4. The cost per t follows the same trendand varies from a maximum of CAN$ 93.39 in year 1 to a low ofCAN$ 72.91 in year 4. The Life of Mine average cost per t is CAN$ 81.31. The cost per Oz produced is also higher in year 1 at CAN$ 771 and reaches its minimum in year 2 at CAN$600.00 due to the higher gold grades mined in that year. The Life of Mine average cost per Oz is $671

Main Mine Operating Costs are initially CAN$ 29.4 and CAN$ 27.0 million in year 1 and 2 before jumping to an average of about CAN$ 40 million in year 3, 4 and 5, when the Main Mine Lower area comes into production. The highest cost per t mined, CAN$ 83.20, occurs in year 1 whereas the lowest is in year 4 at CAN$ 55.12. The average Life of Mine cost per t in the Main Mine is CAN$ 62.85.

No. 3 Zone Operating Costs are just under CAN$ 15 million in year 1 and 2 before dropping to CAN$ 2.4 million in year 3. The cost per t is CAN$ 57.58 in year 1 and stabilises around $54 thereafter.

Milling costs vary from CAN$ 8.4 to 9.1 million per year except in year 5 where it drops to CAN$ 7.9 million due to the slightly lower tonnage milled.

Administration costs were estimated to be CAN$ 3.8 million per year and do not vary over the mine life.

Finally, accommodation costs vary from CAN$ 1.2 million in year 0 to CAN$ 2.6 million in years 1 and 2.

18.9.2 Mill (IW)

18.9.2.1 *Basis of Estimate*

The operating cost estimate for the plant is developed as follows:

- A detailed review of past operating reports from the period 1995-2004, to develop typical consumables levels, manning list, power requirements and water supply quantities;
- Metallurgical testing on composite samples of the two ore types to verify power requirement, size of grind and reagent consumptions;
- Milling rates taken from current LOM plan;
- Selection of labour rates for each position based on current rates in the area, application of 35% payroll burden, production bonus levels supplied by Alexis, and allowance for overtime and absenteeism;
- Quoted current costs for consumables either as delivered cost or with 10% added for freight;

- Power unit costs taken from Manitoba Hydro 2010 tariff;
- Current water unit cost rate provided by (to come);
- Appropriate allowances as $/t based on similar operations for maintenance supplies and operating supplies;
- Adjustment of power, grinding media and liner costs based on hardness variation of ore blend treated in each year;
- All taxes are excluded;
- Contingency is excluded;
- Costs are expressed in current, mid-2010 CAN$;
- The accuracy of the cost estimate is +/- 10% and is considered to be appropriate for the Feasibility Study.

Operating Cost Estimate

The estimated plant operating cost is developed from the planned processing of 2075 t/d and the ore blend predicted in the LOM plan. As the ore blend changes through the LOM, the costs for power, grinding media and reagents will also vary.

The previous operating period of the plant was used as a reference for power and consumable consumption levels, (recognizing that the plant was treating only Main Zone ore and at a slightly coarser grind than will be planned in the future). In the years 2002 and 2003 the plant processed 750,981 and 606,497 t respectively and achieved a gold recovery above 94%. The level of consumables in these years were used to predict the level of consumables in the LOM plan with an adjustment made for the ore blend.

Curent unit prices were obtained mid-2010 for all consumables from typical or recommended suppliers. Electrical unit costs were obtained from Manitoba Hydro using the current tariff for General Large service. Water rates were obtained from Manitoba Water Conservation Agency.

Operating and maintenance labour rates were developed from the current typical area costs and company paid benefits and costs were included. The plant will be operated by 6 staff and 35 hourly employees, as listed in Table 18-15.

Table 18-14 : Mill Operating Personnel

Position	Number
Mill superintendent	1
Metallurgist	1
Mill technician	1
Mill clerk	1
Maintenance foreman	1
chief assayer	1
mill shift boss	4
leach/cip op	4
grinding	4
reagents/tailings op	2
crusher lead	4
crusher op helper	4
labourer/trainee	2
millwright	3
welder	1
jr mech	1
electrician/instruments	2
assay technician	3
sample bucker	1
total mill staff	6
total mill hourly	35

The summary of the estimated plant operating cost by year is shown in Table 18-16.

Table 18-15 : Estimated Plant Operating Cost by Year

year	0	1	2	3	4	5	LOM
total metric tonnes milled	**46,202**	**632,369**	**737,176**	**711,280**	**716,683**	**633,656**	3,477,367
tonnes 3 zone	38,000	279,176	270,001	46,001	-	-	
tonnes main zone	8,202	353,193	467,175	665,279	716,683	633,656	
total costs SCDN							
mill labour incl OH	630,620	3,322,241	3,322,241	3,322,241	3,322,241	3,322,241	17,241,825
reagents and grinding media	215,915	2,744,743	3,151,212	2,853,114	2,834,295	2,505,944	14,305,223
crusher and mill liners	51,427	617,123	617,123	421,096	308,562	308,562	2,323,893
power	67,349	846,345	964,335	884,080	905,824	811,329	4,479,262
raw water supply	3,718	60,379	70,675	68,303	68,798	61,169	333,042
maintenance parts	34,652	474,277	552,882	533,460	537,512	475,242	2,608,025
operating supplies	9,240	126,474	147,435	142,256	143,337	126,731	695,473
Total annual cost CAN$	**1,012,921**	**8,191,582**	**8,825,904**	**8,224,549**	**8,120,569**	**7,611,218**	**41,986,744**
total costs SCDN/t milled							
mill labour incl OH	13.649	5.254	4.507	4.671	4.636	5.243	4.958
reagents and grinding media	4.673	4.340	4.275	4.011	3.955	3.955	4.114
crusher and mill liners	1.113	0.976	0.837	0.592	0.431	0.487	0.668
power	1.458	1.338	1.308	1.243	1.264	1.280	1.288
raw water supply	0.080	0.095	0.096	0.096	0.096	0.097	0.096
maintenance parts	0.750	0.750	0.750	0.750	0.750	0.750	0.750
operating supplies	0.200	0.200	0.200	0.200	0.200	0.200	0.200
Total annual cost CAN$/t	**21.924**	**12.954**	**11.973**	**11.563**	**11.331**	**12.012**	**12.074**

18.10 Economic analysis (ar)

18.10.1 Assumptions and inputs

A full economic model was developed for the Snow Lake Mine Re-activation. All the costs used are in third quarter 2010 Canadian Dollar (CAN$). No inflation was applied for future years.

A base case was developed with the anticipated revenues based on projected gold prices and exchange rates as provided by Alexis. The projected gold prices and exchange rates are the Bloomberg consensus price deck dated August 31, 2010. This practice is accepted by auditors and investment banking consultants. The gold price deck using the Bloomberg data is:

Year	Gold price ($US/Oz)	Exchange Rate (CAN$/US$)
0 - 2011	1227	1.04
1 - 2012	1303	1.04
2 - 2013	1276	1.04
3 - 2014 - Onward	1051	1.06

The gold milling recovery for the Main Mine and No 3 Zone is 94% and 91%, respectively. All the capital and operating costs as previously described are used in the base case. The economic analysis was performed with the IRR and the Net Present Value (NPV) methods. The IRR on an investment is defined as the rate of

interest earned on the unrecovered balance of an investment. An investment decision can be taken if IRR is greater than the minimum attractive rate of return for the project. The NPV method converts all cash flows of investment and revenues occurring throughout the planning horizon of a project to an equivalent single sum at the present time using a discount rate (7%). A positive NPV represents a profitable investment when the initial investment plus any financing interest are recovered. No evaluation was done to take into account the financing of the project. No tax credit for exploration expenses or new mine tax holidays were introduced to reflect the fiscal advantages offered in Manitoba.

The mine will reach commercial production when it reaches 60% of its nominal Production Rate for a period of 3 consecutive months. The nominal milling production rate is 2,000 t/day. The project schedule indicates that the mine is expected to reach that point during month13.

18.10.2 Base Case Analysis

A base case cash flow schedule has been prepared, using a Bloomberg gold price deck as at August 31, 2010. The Snow Lake Mine Re-activation economic analysis is summarized as follows:

- Total production of 415,600 Oz gold, with a mill recovery rate of 93.3%
- Estimated total average cash cost of CAN$ 671 (US$ 639) /Oz Au; and total costs of approximately CAN$ 851 (US$ 810) /Oz Au.
- Total sustaining capital of CAN$ 33.9 million.
- Operating cash costs per t are estimated to be CAN$ 81.31/t
- Total pre-tax cash flow, discounted cash flow at 7%, and Internal Rate of Return from the mine are projected to be CAN$ 141.1 million, CAN$ 101.1 million and 79%, respectively.
- Payback period is estimated to be 1.7 year (after tax) after commencement of commercial production

The summary cashflow model is depicted on Table 18-17.

Table 18-16 : Summary Cashflow Model

ALEXIS			SNOW LAKE MINE		FEASIBILITY STUDY - COST MODEL				Oct-10
						3 Zone and Mill start at same time, no 1700, Equipment rental 18 mo			
			YEAR 0	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	TOTAL
	Sensitivity								
Production Milled									
TONNAGE		(kt)	-	632,369	737,176	711,280	716,683	633,656	3,431,164
GRADE		(g/t)	-	4.07	4.35	3.91	3.84	4.01	4.04
ONCES FEED		(oz)	-	82,753	103,000	89,357	88,426	81,661	445,196
RECOVERY		(%)		92.6%	92.6%	93.7%	94.0%	94.0%	93.3%
PRODUCTION		(oz)	-	76,620	95,337	83,746	83,120	76,761	415,584
REVENUES									
GOLD PRICE		(US$/oz)	1,277	1,303	1,276	1,051	1,051	1,051	1,149
GOLD PRICE		(Cdn$/oz)	1,328	1,355	1,327	1,114	1,114	1,114	1,207
Exchange rate			1.04	1.04	1.04	1.06	1.06	1.06	1.05
REVENUES		(CDN$)	-	103,828,449	126,516,424	93,288,247	92,600,110	85,616,738	501,760,968
OUNCES TO LEGENDS			-	-	-	-	-	-	-
REDUCTION TO REVENUE				-	-	-	-	-	-
ROYALTIES		1.38%	-	(1,432,848)	(1,745,927)	(1,287,516)	(1,277,892)	(1,180,131)	(6,924,301)
NET REVENUES		(CDN$)	-	**102,395,602**	**124,770,497**	**92,010,731**	**91,322,229**	**84,436,607**	494,836,667
OPERATING COSTS									
Mining		(CDN$)	-	44,066,801	41,540,739	43,378,257	39,500,747	41,475,685	209,962,230
Milling		(CDN$)	-	8,485,041	9,157,157	8,518,202	8,405,726	7,866,012	42,432,139
Administration		(CDN$)	-	3,810,000	3,810,000	3,810,000	3,810,000	3,810,000	19,050,000
Accomodation		(CDN$)	-	2,694,048	2,694,048	1,616,429	538,810	-	7,543,333
Interest cost associated with ounces to Legends		(CDN$)	-	-	-	-	-	-	-
OPERATING COSTS		**(CDN$)**	-	**59,055,890**	**57,201,944**	**57,322,888**	**52,255,283**	**53,151,698**	278,987,702
EBITDA		**(CDN$)**	-	**43,340,713**	**67,568,553**	**34,687,843**	**39,066,946**	**31,184,910**	215,848,964
DDA		(CDN$)	-	15,452,515	19,227,343	16,889,675	16,763,292	15,480,997	83,813,822
EBT			-	27,888,197	48,341,210	17,798,168	22,303,654	15,703,913	132,035,143
INTEREST EXPENSE	coupon ent gold at cost		-	-	-	-	-	-	-
Taxes		(CDN$)	-	6,559,881	18,340,518	6,662,702	8,454,192	5,953,106	45,970,398
EARNINGS			-	21,328,316	30,000,692	11,135,467	13,849,463	9,750,807	86,064,744
Capital Expenditures		**(CDN$)**	**40,797,628**	**22,639,406**	**9,340,787**	**1,399,081**	**237,000**	**300,000**	74,713,822
		(US$)	**39,228,488**	**21,768,659**	**8,981,449**	**1,319,888**	**223,585**	**283,019**	71,805,088
OPERATING AND CAPITAL COST		**(CDN$)**	**40,797,628**	**81,695,295**	**86,542,651**	**58,721,969**	**52,492,283**	**53,451,698**	**353,701,524**
		(US$)	**39,228,488**	**78,553,168**	**83,983,319**	**55,398,084**	**49,521,821**	**50,426,130**	337,110,211
NET CASH FLOW		(CDN$)	**(40,797,628)**	**20,701,307**	**58,227,846**	**33,288,762**	**38,829,946**	**30,884,910**	141,135,143
(Before taxes and interest)		(US$)	**(39,228,488)**	**19,905,193**	**55,988,313**	**31,404,492**	**36,632,825**	**29,136,787**	133,838,152
NET CASH FLOW		(CDN$)	**(40,797,628)**	**20,701,307**	**58,227,846**	**33,288,762**	**38,829,946**	**30,884,910**	141,135,143
(Before taxes)		(US$)	**(39,228,488)**	**19,905,193**	**55,988,313**	**31,404,492**	**36,632,825**	**29,136,787**	133,838,152
NET CASH FLOW after interest		(CDN$)	**(40,797,628)**	**14,141,426**	**39,887,327**	**26,626,060**	**30,375,754**	**24,931,884**	95,164,744
(After taxes and interest)		(US$)	**(39,228,488)**	**13,597,525**	**38,353,199**	**25,118,925**	**28,656,372**	**23,528,570**	90,018,103
UNIT COSTS									
Operating Cash Costs		Cdn$/t milled	-	93.39	77.80	80.59	72.91	83.88	81.31
		Cdn$ / oz	-	771	600	684	629	692	671
		US$ / oz	-	741	577	646	593	653	639
Operating and Capital Costs		Cdn$/t milled	-	129.19	90.27	82.56	73.24	84.35	103.08
		Cdn$ / oz	-	1,066	696	701	632	696	851
		US$ / oz	-	1,025	671	661	596	657	810
Total All Cash Costs including interest and taxes		Cdn$/t milled		138.56	115.15	91.93	85.04	93.75	116.48
		Cdn$ / oz	-	1,152	890	781	733	774	962
		US$ / oz	-	1,108	856	737	692	730	915
			BEFORE TAX				**AFTER TAX**		
			NPV 7%		$101,144,937		NPV 7%		$65,366,487
			IRR		78%		IRR		55%
			PAYBACK		1.35		PAYBACK		1.67

18.10.3 Sensitivity Analysis

A variety of sensitivity analyses were carried out to assess the economics of the project if certain factors change. The factors included variations in the price of gold, the grade, currency exchange rates, operating costs and capital costs. A summary of the results of these analyses are presented in Table 18-18.

Table 18-17 : Summary of Sensivity Analysis

CASE ID	SENSITIVITY FACTOR	FEASIBILITY		
	Gold price US$/Oz	Pre-tax net cash flow (CDN$)	Pre-tax NPV (7)	IRR
20	1 379	241,83	179,32	135%
15	1 321	216,66	159,78	121%
10	1 264	191,04	139,90	108%
5	1 207	166,31	120,69	93%
	1 149,08	141,14	101,14	79%
-5	1 092	115,96	81,60	66%
-10	1 034	90,79	62,06	52%
-15	977	65,61	42,51	39%
-20	919	40,44	22,97	25%
	Exchange rate ($CDN/$US)			
20	1,26	241,83	179,32	135%
15	1,21	216,66	159,78	121%
10	1,16	191,48	140,23	107%
5	1,10	166,31	120,69	93%
	1,05	141,14	101,14	79%
-5	1,00	115,96	81,60	66%
-10	0,95	90,79	62,06	52%
-15	0,89	65,61	42,51	39%
-20	0,84	40,44	22,97	25%
	Grade (g/T)			
20	4,84	240,10	177,70	130%
15	4,64	215,36	158,56	117%
10	4,44	190,62	139,42	105%
5	4,24	165,88	120,28	92%
	4,04	141,14	101,14	79%
-5	3,83	116,39	82,01	67%
-10	3,63	91,65	62,87	54%
-15	3,43	66,91	43,73	40%
-20	3,23	42,17	24,59	26%
	Operating costs			
20	97,57	85,34	58,21	51%
15	93,51	99,29	68,94	58%
10	89,44	113,24	79,68	65%
5	85,38	127,19	90,41	72%
	81,31	141,14	101,14	79%
-5	77,24	155,08	111,88	86%
-10	73,18	169,03	122,61	93%
-15	69,11	182,98	133,35	100%
-20	65,05	196,93	144,08	107%
	Capital Costs			
20	89,66	126,19	87,75	60%
15	85,92	129,93	91,10	64%
10	82,19	133,66	94,45	69%
5	78,45	137,40	97,80	74%
	74,71	141,14	101,14	79%
-5	70,98	144,87	104,49	85%
-10	67,24	148,61	107,84	92%
-15	63,51	152,34	111,19	100%
-20	59,77	156,08	114,54	108%

The project is highly sensitive to the gold price, currency exchange and grade. Any of these factors reduce the revenue in the model directly by the percentage sensitivity applied. The sensitivity analysis increased or decreased the gold price, currency exchange and grade from + 20% to -20%. A reduction 20% resulted in an average gold price of US$ 919/Oz, exchange of 0.84 CAN$/US$ or grade of 3.23 g/t all resulting in a pre-tax cashflow, NPV(7) and IRR of CAN$ 40.4 million, CAN$ 23.0 million and 25%, respectively.

The sensitivity on the Cashflow, NPV, and IRR with varying operating costs and capital costs is not as pronounced as the revenue parameters of the gold price, exchange rate and grade. Increasing operating cost by 20% reduces the base case pre-tax cashflow, NPV and IRR to CAN$ 85.3 million, CAN$ 58.2 million and 51% respectively. Increasing capital costs by 20% reduces the base case pre-tax cashflow, NPV and IRR to CAN$ 126.2 million, CAN$ 87.8 million and 60%.

The sensitivity charts are illustrated in Figures 18-19 and 18-20.

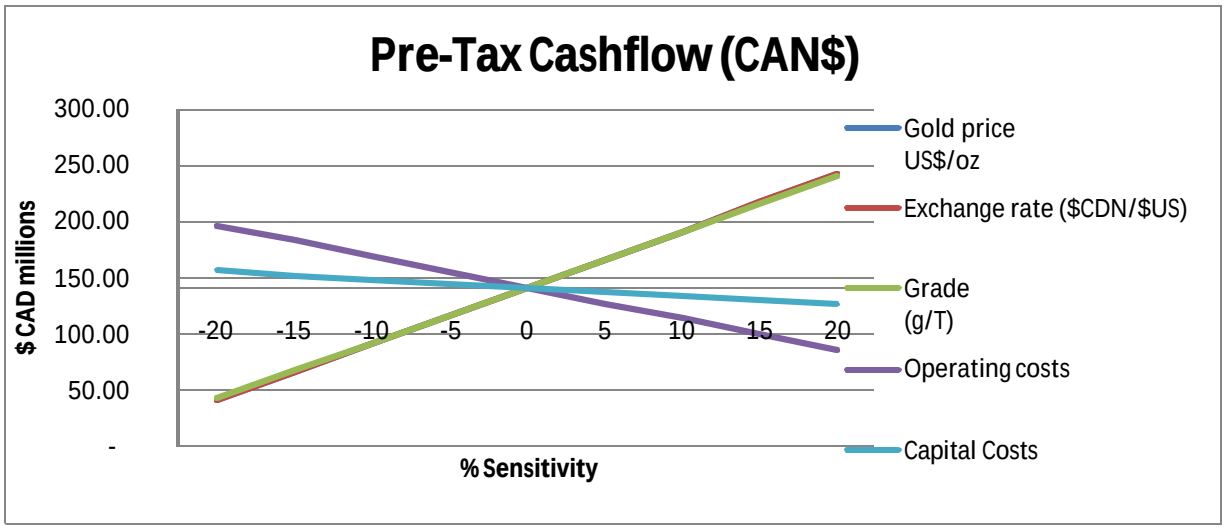

Figure 18-19: Pre-tax Cashflow sensivity

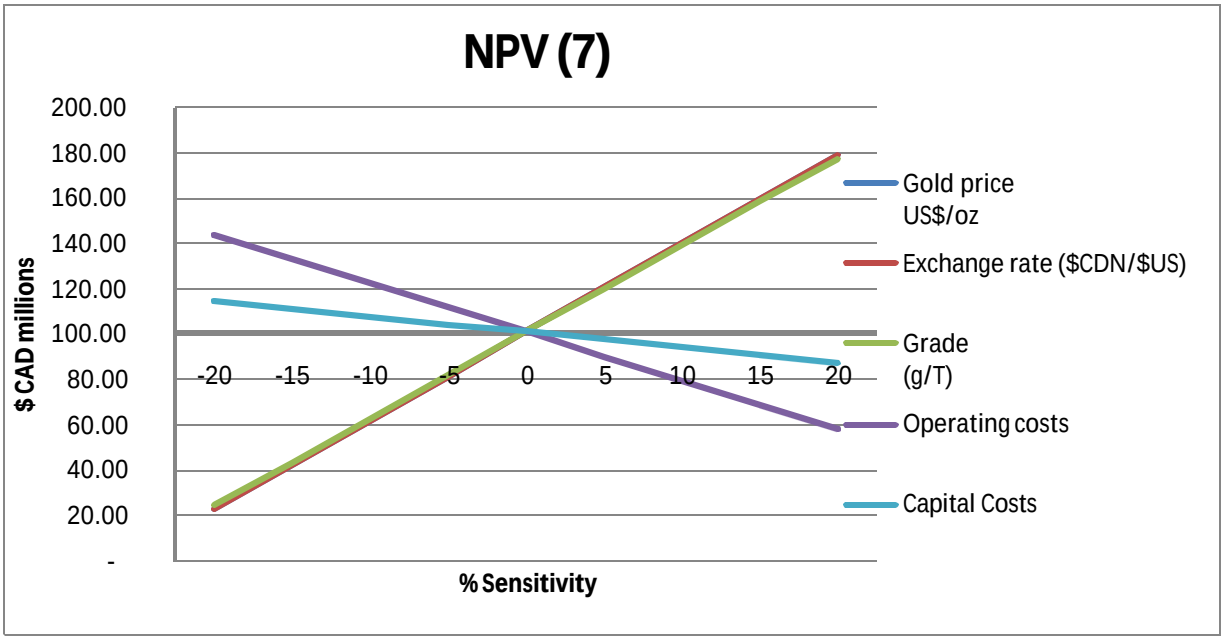

Figure 18-20: NPV sensitivity

18.11 Risk analysis (AR)

This section discusses the project Risk Factors, their likelihood of occurrence, their probable effects and their possible mitigation measures.

Financial Risk Factors

In the current world economic and geopolitical environment a declining price of gold does not appear to be likely. A declining price of gold would reduce the revenues of the project, which in turn would translate into lower profits. Forward sales or purchase of gold put options, i.e. hedging, could mitigate that risk. However a hedging policy would remove any benefit of an increasing gold price. Consequently, hedging is not seen as an attractive option in the current environment by most gold producers. It should also be noted that at the time of writing, the current gold prices are higher than the prices used in this Feasibility Study.

The second financial risk factor is an unfavourable variation in the exchange rate, i.e. the US dollar becoming weaker or the Canadian dollar becoming stronger. In the recent past the US dollar has weekened significantly compared to the Canadian dollar. In the current world economic environment a strengthening of the US dollar does not appear to be likely in the short term, however a strengthening of the Canadian dollar appears to be possible. Mitigation of this risk could be achieved by the purchase of financial instruments, furure contracts or options, on the Canadian dollar, if necessary.

Procurement Risk Factors

The difficulty to obtain materials, supplies and equipment in a timely manner is a risk that could lead to delays. The delays could result in a lower profitability due to production delays, which could in turn increase costs (e.g. higher fixed costs). Mitigation measures could include the sourcing of both new and used equipment, and allowing for flexibility in the required specifications. The appointment of a proactive procurement team will help to minimize the impact of key equipment delays.

As in any project, higher material and equipment costs are a risk factor. The impact would be lower profitability due to higher capital and operating costs. The mitigation of this risk will depend on the procurement team and a rigourous procurement process that neglects no possible appointment of a proactive procurement team.

Labour Related Risk Factors

Difficulty to find the required experienced manpower either locally, or within a larger area is a risk factor. This could result in a lower profitability due to production delays, which in turn could increase costs, (higher fixed costs). Mitigation measures include paying competitive wages and the creation of a rewarding work environment.The recruitment of external manpower is a mitigation measure and Alexis will provide a camp as the shortage of local manpower is considered likely.

The possibility that the cost of labour could be higher than estimated is a risk factor that could result into lower profitability due to higher capital and operating costs. This risk could be significant.

Lower productivities could be a risk factor. Lower productivities would result in a lower profitability due to a production delay, which in turn would increase the costs, (higher fixed costs). Mitigation measures involve the hiring of experienced personnel. This risk is present as it may be difficult to recruit experienced personnel.

Risks Related to Infrastructure, Facilities and Underground Openings

A higher level of reconditioning and rehabilitation, (surface facilities, mill and underground openings) is a risk factor thast could lead to lower profitability due to an increased capital cost and could delay production. A mitigation measure is the timely recruitment of the required manpower and to avoid delays. Based on the available information on the good ground conditions this risk is considered to be low.

The possibility of additional dewatering in the Main Mine is a risk factor. The only mitigation measure is the monitoring and measurment of the dewatering program. The dewatering plan must be flexible and include alternative options. This is not considered to be a high risk and the consequences are limited.

Risks Due to Lower Gold Production

The possibility of producing lower gold grades is a risk. This is an operational risk that is difficult to mitigate. However this risk considered to be low given the historical recoveries involved with the previous mining

There is also a risk associated with higher dilution. This risk is considered to be low given the history of the previous mining.

The risk of lower mill recoveries is also considered to be low due to the previous milling history.

Risks Due to Infrastructure Failure

A tailings dam failure could result in a production slow down or stoppage. This risk can be mitigated through regular inspections to detect any problems. This risk is considered to be very low.

The destruction of surface buildings by fire could disturb operations and stop production for a period of time. Mitigation measures consist of having adequate fire prevention procedures in place. Adequate fire insurance is essential.

19. INTERPRETATION AND CONCLUSIONS (ALL)

The Snow Lake Mine is a former producing mine that has been on care and maintenance with most of the key infrastructure already in place. It is located in a very accessible area in Manitoba with all the required services. The site is close to Flin Flon and The Pas which will facilitate the availability of personnel.

The Mineral Resource Estimate for the Snow Lake Mine has been prepared under the direct supervision of Jamie Lavigne, P. Geo. Mr. Lavigne is satisfied that the database is of suitable quality and is appropriate for the estimation of resources for the Snow Lake Mine.

The Proven and Probable Reserves, which are planned to be mined by underground longhole mining and room-and-pillar methods stand at 3,477,000 t grading 4.04 g/t Au for 451,900 Oz of contained gold.

The current mine life is 6 years with the potential for additional resources through ongoing exploration. The reserves are based on the measured and indicated resources updated by Alexis. A detailed mine plan has been prepared for the underground production. Most of the production will come from longhole and mechanized room-and-pillar mining methods.

The cost estimates have been developed to an accuracy of ± 10%. The capital expenditures are based on current quotations for equipment and surface installations. The development and mining costs are based on the actual costs, costs from similar bench mark operations and from the Lac Herbin mine currently operated by Alexis.

19.1 RECOMMENDATIONS

A large part of the technical support for mineral resource estimate, mineral reserve estimate, metallurgy, project design, operating cost estimates, capital cost estimates, environmental studies, and permitting are documented in a 2009 Technical Report by Micon and a 2010 Technical Report by Golder Associates. Capital cost estimates were updated by Alexis in September 2010. Capital and operating cost estimates were again updated by Alexis and GENIVAR in 2010. GENIVAR reviewed the September 2010 capital and operation costs and provided input and opinions as of that date. The technical basis for mineral resources and mineral reserves meet the requirements of Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) Standards on Mineral Resources and Reserves (2000) and Canadian National Instrument 43-101.

Mineralization and Alteration

The geology is well understood and is documented with appropriate geological mapping and drill hole logging. Modelling of ore controls is suitable to support the resource estimates.

Drilling

Drilling equipment and procedures conform to industry standard practices and have produced information suitable to support resource estimates. The sample recovery as documented was acceptable. Collar surveying was of suitable accuracy to ensure reliable location of drill holes relative to the mine grid and other drill holes. Down hole surveys were complete and provide confidence in the position of individual intercepts of deep mineralization. The holes that are not surveyed are dominated by delineation drilling, (less than 90 m deep).

Resource Classification

The mineral resources of the Snow Lake Project were classified as Measured, Indicated, and Inferred Mineral Resources by Alexis. Classification is based on proximity to historical mining, data density, and consideration of the classification scheme in historical estimates. Inspection of the model and drill hole data on plans and sections combined with spatial statistical work and validation results was done by Alexis.

Mineral Resources

Mineral resource estimates were done using 3-dimensional block models using Gemcon software. The capped grades were applied to composited assays. Modelling for gold grades consisted of grade interpolation by Ordinary Kriging (OK) and by Inverse Distance (ID) methods. Only capped grades were interpolated. Nearest-Neighbour (NN) grades were also interpolated for validation purposes. Bulk density values were appropriately assigned into the resource block model.

The Mineral Resources estimates comply with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves (2000), CIM Definition Standards for Mineral Resources and Mineral Reserves (December 11, 2005) and Canadian National Instrument 43-101 of the Canadian Securities Administrators (December 30, 2005).

Mineral Reserves

Mineral reserves are estimated using a cut off grade strategy. GENIVAR concurs with this approach. The Mineral Reserve estimates comply with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and

Reserves (2000), CIM Definition Standards for Mineral Resources and Mineral Reserves (December 11, 2005) and Canadian National Instrument 43-101 of the Canadian Securities Administrators (December 30, 2005).

Geotechnical

The geotechnical information and analysis is considered to be adequate to satisfy the requirements of the Feasibility Study. The Feasibility Study has analysed the rib and sill pillars as separate, stand-alone cases that were examined in two dimensions. Additional work is required to analyse the combined effect of the rib and sill pillar geometries. This type of analysis involves three dimensional modelling and will be performed at the detailed design stage. The three dimensional analysis of the rib and sill pillar combination will effectively reduce the level of mining induced stress. This in turn will result in a higher factor of safety than the two dimensional analysis. An analysis of the combined geometry in three dimensions will result in a higher level of confidence for the detailed design of the pillars. A ground control program is recommended to monitor the stability of the crown pillars on a continuous basis using advanced technology.

Environmental

The Snow Lake Mine has been under care and maintenance since January 2005. However, the Snow Lake Mine was and continues to be a fully permitted mine with all the required permits in good standing. The mine was not closed, as defined by the *Manitoba Mining Act*, in 2005, but put into a “Care and Maintenance” mode which allows for an easy transfer to an “Active or Producing” mode.

All of the required Provincial environmental permits have not expired and are transferable to Alexis Minerals. All of the required environmental baseline work is up to date.

The Manitoba Government has accepted and approved a Closure Plan for the site and there is financial assurance of CAN$ 1.8 million filed for the site.

The Snow Lake Mine have completed Cycles 1 and 2 in 2006 and 2009 for the Federal Environmental Mining Regulations; as well, the required annual reports have been submitted up to 2009. The next cycle will be due in 2012.

Tailings Management Facility

The Tailings Management Area (TMA) at the SLM is located in the former Birch Lake basin. The original design for the TMA was prepared by DNE Knight Pieshold (1995). Annual dam inspections have been undertaken, and it is reported that the dams are in good condition. The most recent inspection was carried out by Golder in 2008. The

TMA was reviewed by Knight Piesold and it is determined that with minimal dam raises, the current TMA is of sufficient capacity to hold the mine tailings.

Metallurgical Testing

The Snow Lake mill has previously operated successfully treating both Main Zone and 3 Zone ores and the process design is appropriate for renewed treatment of these ores. No changes to the milling process are required to obtain optimum gold recovery and operating cost. The average design processing rate is 2075 t/day which has been attained easily in previous operations with the plant.

New metallurgical test work has been conducted for the purposes of this Feasibility Study. The test work confirmed the historical recoveries in the Main Mine of 94% and determined that the No. 3 Zone ore recoveries will be lower at 91%. The history of the present mill design, and the previous key design criteria are appropriate to produce the desired recoveries.

There is no new technical information that materially alters the validity of any previous work. Plant design concepts are reasonable and reflect typical, state-of-the-art, design concepts and application practices.

Sampling

Logging of drill core followed the procedures suitable for recording lithology, alteration, and mineralization in a gold deposit. Geological data and interpretations are suitable to support the resource estimates. Sample collection and handling of drill core was done in accordance with industry standard practices, with procedures to limit sample losses and sampling biases. Sample preparation and assay protocols meet the industry standard practices for gold. Assay methods conform to industry standard practices.

Mineralogical Analysis

Bulk density values for ore and waste units are based on numerous core sample measurements and confirm historical bulk densities. Bulk densities are assigned by a combination of lithology and degree of mineralization. The methods conform to industry standard practices and are suitable for estimates of tonnage.

Geological, geotechnical, and analytical information were developed over a period of multiple exploration programs involving several past producers. Entry of information into databases uses a variety of techniques and procedures to check the integrity of the data entered. Assays were received electronically from the laboratories and imported directly into the drill hole database.

Mine Designs and Production Plans

Conventional underground methods are planned for Snow Lake Mine. The mine plan features a 6 years mine life. GENIVAR reviewed the economic modelling methodology and parameters applied. They are considered to be standard practice and appropriate for this deposit. An independent check by GENIVAR has confirmed the results.

Equipment selections are generally appropriate for the mine design, production rate and production schedule. Generally speaking, the work done by Alexis and Garson appears to be of high quality and appropriate for the design stage and nature of the deposit. No fatal flaws were detected. Any areas where GENIVAR had concerns with the mine plan have been addressed. GENIVAR believes the mine plan is achievable.

Operating Cost Estimates

Alexis developed operating cost estimates from information provided by similar Canadian operations, new quotations, and in-house data. GENIVAR concurs of the methodology used and the results obtained.

Capital Cost Estimates

Alexis estimated total project capital costs to be CAN$ 74.7 million, of which CAN$ 40.8 million are pre-production costs and CAN$ 33.9 million are sustaining capital costs. GENIVAR reviewed capital estimates and found them to be appropriate.

Economic Analysis

The economic analysis of the Snow Lake Project is based on a discounted cash flow analysis on a pre-tax basis, using Proven and Probable Mineral Reserves and annual production plans. Projections for annual revenues and costs are based on data developed for the mine, process plant, capital expenditures, and operating costs. Discounted cash flow analysis indicates that the project offers a positive return. Payback period is 1.7 year. Life–of-mine is 6 years. As with many projects of this type, the Snow Lake Project is most sensitive to changes in metal price and less sensitive to changes in operating cost and capital expenditures. In GENIVAR's opinion, the level of detail used in the economic analysis is appropriate for a Feasibility Study

Based on projected costs and a Bloomberg gold price deck averaging US$ 1,149, cut off grades were evaluated for various mining methods. The reserves stand at:

- Proven Reserves: 7,000 t @ 3.81 g/t Au for 900 Oz; and
- Probable Reserves : 3,470,000 t @ 4.04 g/t Au for 451,000 Oz.

The mine will operate on a 7-day per week schedule, working two 10 hour shifts per day with only one day shift on weekends. The planned production rate for the mine is a nominal 2000 t/day on 350 days/year or 700,000 t annually.

Table 19-1 : Preliminary Production Schedule

Year	Tonnes (t)	Grade (g/t)	Contained Gold (Oz)	Recovery %	Recovered Gold (Oz)
Pre-prod.	46,200	4.42	6,565	91.5	6,004
1	632,369	4.07	82,753	92.6	76,620
2	737,176	4.35	103,000	92.6	95,337
3	711,280	3.91	89,357	93.7	83,746
4	716,683	3.84	88,425	94.0	83,120
5	633,656	4.01	81,661	94.0	76,761
Total	3,477,367	4.04	451,761	93.3	421,588

A base case cash flow schedule has been prepared, using a Bloomberg gold price deck as at August 31, 2010 of US$ 1, 277 /ounce for 2011, US$ 1,303/ounce for 2012, US$ 1,276/ounce for 2013, and US$ 1,051/ounce for 2014 and onward with an exchange rate of CAN$ 1.04 for US$ 1.00 for 2011 – 2013 and 1.06 onwards from 2014. The Snow Lake Mine economic analysis is summarized as follows:

- The Life of Mine with the existing reserves is 6 years, including 1 year od pre-production.
- Total production of 415,600 Oz gold, with a mill recovery of 93.3% (421,588 Oz including pre-production).
- Estimated total average cash cost of CAN$ 672 (US$ 640) /Oz Au; and total costs of approximately CAN$ 852 (US$ 811) /Oz Au.
- Total sustaining capital of CAN$ 36.1 million.
- Operating cash costs are estimated to be CAN$ 81.41/t
- Total pre-tax cash flow, discounted cash flow at 7%, and Internal Rate of Return from the mine are projected to be CAN$ 140.7 million, CAN$ 100.8 million and 80%, respectively.
- Payback period is estimated to be 1.7 year (after tax) after commencement of commercial production

Sensitivity Analysis

GENIVAR considers that the capital and operating costs have been compiled with adequate contingencies to mitigate unforeseen over-expenditures. Combined with the ability of the project (as demonstrated by the sensitivity matrix), to withstand a further 20% increase in capital or operating costs, the project is seen as robust in this regard.

The economic model inputs are appropriate and, apart from the initial period of development, all future annual cash flows are positive.

The reserve assessment in this report comprises planning, scheduling and cost estimates for the production of 415,600 Oz gold from the mining and processing of approximately 445,200 t of ore reserve at a grade of 4.04 g/t gold (after dilution and losses).

The project has an IRR of 80% and a NPV(7) of $CAN 100.8 million. The absence of significant capital expenses for new infrastructure during the re-activation have reduced the capital cost and improved the current profitability of the project.

The project is highly sensitive to the gold price, currency exchange and grade. Any of these factors reduce the revenue in the model directly by the percentage sensitivity applied. The sensitivity analysis increased or decreased the gold price, currency exchange and grade from + 20% to -20%. A reduction 20% resulted in an average gold price of US$$ 919/Oz, exchange of 0.84 CAN$/US$ or grade of 3.23 g/t all resulting in a pre-tax cashflow, NPV(7) and IRR of CAN$ 40 million, CAN$ 22.6 million and 24%, respectively.

The sensitivity on the Cashflow, NPV, and IRR with varying operating costs and capital costs is not as pronounced as the revenue parameters of the gold price, exchange rate and grade. Increasing operating cost by 20% reduces the base case pre-tax cashflow, NPV and IRR to CAN$ 85.3 million, CAN$ 58.2 million and 51% respectively. Increasing capital costs by 20% reduces the base case pre-tax cashflow, NPV and IRR to CAN$ 126.2 million, CAN$ 87.8 million and 60%.

This report demonstrates that the Snow Lake Mine has the potential for economic extraction and warrants re-activation. In the current analysis, payback occurs after 1.7 year. Alexis is in the favourable position of having the potential to identify further economic resources.

Exploration Potential

The Snow Lake Mine and surrounding properties contain a number of targets and target areas that are at various stages of development and exploration. These range from resource delineation and expansion targets to conceptual targets based on geological compilation and interpretation. The proposed exploration program has two main objectives:

- To evaluate and increase the gold resources within the strategic No. 3 Zone to Boundary Zone to Main Mine corridor and to incorporate these gold resources into further preliminary assessment and feasibility level studies. Also included in the first objective is the evaluation of historical resources at the Birch Zone and Margaret Zone.

- To continue to evaluate the gold potential of the entire property through the evaluation of medium to early stage exploration targets and implementation of earlier stage exploration activities to define new drill targets.

Infill drilling to upgrade the current inferred resources was completed by Alexis from January 2010 to June 2010.

The preparation and completion of the Snow Lake Feasibility Study has been a cooperative effort by a number of firms and individuals. The various authors are in agreement that the completed project as met its original objectives. Based on the results of the Snow Lake Feasibility Study, the authors recommend the continued development of the Snow Lake Re-activation Project.

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- *VERBAL – AUGUST AND SEPTEMBER 2010 - MR. ERNIE ARMITT – DIRECTOR MINES ADMINISTRATION, GOVERNMENT OF MANITOBA*

- *VERBAL – JULY, AUGUST AND SEPTEMBER 2010 - MS. AUDREY ROMANCHUK – REGIONAL SUPERVISOR MANITOBA CONSERVATION*
- *VERBAL – JUNE 2010 - MR. DARRYL ARMSWORTHY – SNOW LAKE MINE*
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21. DATE AND SIGNATURE PAGE WITH CERTIFICATE AND CONSENT

This report entitled “Feasibility Study - Snow Lake Mine Re-activation Project” Technical Report 43-101 and dated December, 10, 2010, has been prepared by the following person:

Dated in Val-d’Or, Quebec December, 10, 2010

GENIVAR

Signed & Sealed by: André Roy, Eng.

CERTIFICATE of author
André Roy

As a contributing author of this report on the Snow Lake Mine, I, André Roy do hereby certify that:

1. I am Mining Engineer employed as Manager Mining and Geology Abitibi by GENIVAR, 1075 3rd Avenue East, Val-d'Or, Quebec, J9P 4N9. I carried out this assignment for:

   Alexis Minerals Corporation 65 Queen Street West, Suite 815, Toronto Ontario M5H 2M5, tel (416) 861-9505 fax (416) 861-8165.

   This certificate applies to the Technical Report titled "Feasibility Study Snow Lake Mine Re-activation Project, Technical Report NI 43-101", dated 10 December, 2010;

2. I hold the following academic qualifications:

   B.Sc.A. Mining Engineering, École Polytechnique de Montreal, Quebec, 1980,

   M.Sc.A. Mining Engineering (rock mechanics), École Polytechnique de Montreal, Quebec, 1985,

   M.B.A. John Molson School of Business, Concordia University, Montreal, Quebec, 2002.

3. I am a registered member of the Quebec Order of Engineers (membership number 34588); and I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum;

4. I have worked in the mining industry for over 24 years;

5. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the management of technical studies and numerous mine designs;

6. I visited the property which is the subject of this report during July 12-14, 2010;

7. I am responsible for the following sections of the report:

   - Parts of Section 1 - Executive Summary;
   - Section 2 – Introduction and Terms of Reference;
   - Section 3 – Reliance on Other Experts;
   - Section 17.9 – Reserve Estimate
   - Section 18.1 – Snow Lake Mine Re-Activation Project Overview;
   - Section 18.2 – Reconditioning and Construction of Mine Infrastructures;
   - Section 18.3 – Development and Production, except Section 18.3.1 Geotechnical Assessment
   - Scetion 18.7 – Project Schedule
   - Section 18.8.1 – Capital Cost Estimate – Mine
   - Section 18.9.1 – Operating Cost Estimate – Mine
   - Section 18.10 – Economic Analysis
   - Section 18.11 – Risk Analysis
   - Parts of Section 19 – Interpretations and Conclusions.

8. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;

9. I have read NI-43-101 and I consider that this report has been prepared in compliance with the

instrument;

10. 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; and

11. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 10 December, 2010

original signed and stamped certificate on file

André Roy, Eng.

CERTIFICATE of author
Jamie Lavigne

As a contributing author of this report on the Snow Lake Mine, I, Jamie Lavigne do hereby certify that:

1. I have completed the work reported here for:
2. Alexis Minerals Corporation, 65 Queen Street West, Suite 815, Toronto Ontario M5H 2M5, tel (416) 861-9505 fax (416) 861-8165.
3. This certificate applies to the Technical Report titled "Feasibility Study Snow Lake Mine Re-activation Project, Technical Report NI 43-101", dated 10 December, 2010, prepared for Alexis Minerals Corporation;
4. I hold the following academic qualifications:
5. B.Sc. Geology, Memorial University of Newfoundland, St. Johns, Newfoundland, 1986
6. M.Sc. Geology, University of Ottawa, Ottawa, Ontario, 1991
7. I am a Professional Geologist registered with the Association of Professional Geoscientists of Ontario (member number: 1895)
8. I have worked in the mining industry for 25 years;
9. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes mineral exploration, resource estimation, and the completion of a number of technical reports;
10. I have completed and supervised gold exploration work on the property from January 2007 to December 2009 as Vice President, Exploration for Garson Gold Corp. and from January 2010 to May 2010 as Vice President, Exploration for Alexis Minerals Corporation;
11. I am responsible for Items 4 to 15 and for sections 1 to 8 of Item 17 (Mineral Resource Estimates);
12. I am not independent of Alexis Minerals Corporation as I was Vice President, Exploration, Alexis Minerals Corporation at the time of preparation of the resource estimates contained in this report.
13. I have read NI-43-101 and I consider that this report has been prepared in compliance with the instrument;
14. 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; and
15. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 10 December, 2010

original signed and stamped certificate on file

Jamie Lavigne, P. Geo.

CERTIFICATE of author
David West

As a contributing author of this report on the Snow Lake Mine, I, David West do hereby certify that:

1. I am self-employed as a Geotechnical Engineer, and carried out this assignment for;
2. Alexis Minerals Corporation65 Queen Street West, Suite 815, Toronto Ontario M5H 2M5, tel (416) 861-9505 fax (416) 861-8165. This certificate applies to the technical report titled "Technical Report on the Snow Lake Mine Feasibility Study, Snow Lake, Manitoba", dated 30 September, 20106, prepared for Alexis Minerals Corporation;
3. I hold the following academic qualifications:
4. B.Sc. (Hons) Geotechnical Engineering, University of Oxford, U.K. 1976
5. M.Sc. Rock Mechanics and Excavation Engineering, University of Newcastle upon Tyne, U.K. 1977
6. I am a registered Professional Engineer of Ontario (membership number 90523853); and I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum;
7. I have worked in the mining industry for over 30 years;
8. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the management of technical studies and numerous mine designs;
9. I am responsible for sections 18.5.1 (Geotechnical Assessment) and 19.1.1.1 (Interpretations and Conclusions - Geotechnical).
10. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;
11. I have read NI-43-101 and I consider that this report has been prepared in compliance with the instrument.
12. 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.
13. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 10 September, 2010

"Dave West"

original signed and stamped certificate on file

David West P.Eng.

CERTIFICATE of author
Ian R. Ward, P.Eng

As an author of this report, I, Ian R. Ward do hereby certify that:

1. I am currently sole proprietor of Ian Ward Consulting Services, 16 Ambleside Avenue, Etobicoke, Ontario M8Z2H6, Tel 416-252-3416. I carried out this assignment for:

   Alexis Minerals Corporation 65 Queen Street West, Suite 815, Toronto Ontario M5H 2M5, tel (416) 861-9505 fax (416) 861-8165.

2. This certificate applies to the Technical Report titled "Feasibility Study Snow Lake Mine Re-activation Project, Technical Report NI 43-101", dated 10 December, 2010;

3. I hold the following academic qualifications:

   B.Sc. (Hons) Minerals Engineering, The University of Birmingham, U.K. 1968

4. I am a registered Professional Engineer of Ontario (membership number 48869010); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum;

5. I have worked as a metallurgist in the minerals industry and in consulting engineering companies for the last 42 years. My work experience includes the management of technical and feasibility studies, management of metallurgical testwork programs and the design of numerous mineral processing plants;

6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101;

7. I visited the property which is the subject of this report during June 8-10, 2010;

8. I am responsible for Sections, 16, 18.4, 18.8.2, 18.9.2 and portions of Section 1 and 19 of the report.

9. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;

10. I have no prior involvement with the property or the issuer other than providing consulting services related to the previous Feasibility Study.

11. I have read NI-43-101 and I consider that this report has been prepared in compliance with the instrument.

12. 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.

13. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated December 10, 2010

"Ian R. Ward"

original signed and stamped certificate on file

Ian R. Ward P.Eng.

CERTIFICATE of author
Matthew R. Parfitt

As a contributing author of this report on the Snow Lake Mine, I, Matthew R. Parfitt do hereby certify that:

1. I am employed as a Geological Engineer with Knight Piésold Consulting, and carried out this assignment for:
2. Alexis Minerals Corporation, 65 Queen Street West, Suite 815, Toronto Ontario M5H 2M5, tel (416) 861-9505 fax (416) 861-8165.
3. This certificate applies to the Technical Report titled "Feasibility Study Snow Lake Mine Re-activation Project, Technical Report NI 43-101", dated 10 December, 2010, prepared for Alexis Minerals Corporation;
4. I hold the following academic qualifications:
5. B.Sc. (Hons) Geological Engineering, Queen's University, Kingston ON. 1988.
6. I am a registered Professional Engineer in Ontario (membership number 35415702); and I am a registered Professional Engineer in Manitoba (membership number 8397);
7. I have worked in the mining industry for over 20 years;
8. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the management of technical studies and numerous mine water and tailings management designs;
9. I visited the property which is the subject of this report during July 28-29, 2010;
10. I am responsible for section 18.5 (Tailings Disposal);
11. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;
12. I have read NI-43-101 and I consider that this report has been prepared in compliance with the instrument;
13. 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; and
14. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 10 December, 2010

original signed and stamped certificate on file

Matthew R. Parfitt, P.Eng.

CERTIFICATE of author
Mark J. Bednarz

As a contributing author of this report on the Snow Lake Mine, I, Mark J. Bednarz do hereby certify that:

1. I am self-employed as an Environmental Geoscientist, and carried out this assignment for:
2. Alexis Minerals Corporation65 Queen Street West, Suite 815, Toronto Ontario M5H 2M5, tel (416) 861-9505 fax (416) 861-8165.
3. This certificate applies to the Technical Report titled "Feasibility Study Snow Lake Mine Re-activation Project, Technical Report NI 43-101", dated 10 December, 2010, prepared for Alexis Minerals Corporation;
4. I hold the following academic qualifications:
5. B.E.S., Environmental Studies, University of Waterloo, ON, Canada, 1991.
6. I am a registered Professional Geoscientist of Ontario (membership number 0762); and I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum and the Prospectors and Developers of Canada;
7. I have worked in the mining industry for over 18 years;
8. I am familiar with NI 43-101 and, by reason of education, experience and professional registration; I fulfill the requirements of a Qualified Person as defined in NI 43-101.
9. I visited the property which is the subject of this report between June 14-16, 2010;
10. I am responsible for sections 18.8 (Environmental Requirements), 18.9 (Social Requirements) and 19.1.1.2 (Interpretations and Conclusions - Environmental);
11. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;
12. I have read NI-43-101 and I consider that this report has been prepared in compliance with the instrument;
13. 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; and
14. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 10 December, 2010

original signed and stamped certificate on file

Mark J. Bednarz, P.Geo.

APPENDIX A

MAIN MINE UPPER PLANNING

APPENDIX B

MAIN MINE LOWER PLANNING

APPENDIX C

NO.3 ZONE PLANNING