LEIGHTON

Technical Report

Preliminary Economic Assessment with
Mining Plan and Cost Estimate

For Skyline Gold Corporation Vancouver, BC On The Bronson Slope Property

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3 Executive Summary

Skyline Gold Corporation ("SGC") have commissioned Leighton Asia Limited ("LAL") to complete a Preliminary Assessment with a specific focus on the development of a mine plan and process plant with full project capital and operating cost estimates for their Bronson Slope deposit, which is located in North West British Columbia, Canada. This technical report has been compiled to disclose the findings of the Preliminary Assessment.

LAL is part of the Leighton Group, Australia's largest project development and contracting group, with annual revenues exceeding USD 10 billion. LAL focuses on a number of specific market segments which include Mining, Process, Civil & Infrastructure, Building, Rail, Marine, Oil & Gas, Water, Environmental services, Utilities services, Facility & Infrastructure management and Telecommunications.

A mining project development team is based in Hong Kong providing technical support to the region as well as mining and process engineering consultancy services. This team also has the ability to call upon the experience and technical and financial expertise of the Leighton Group.

The mineral resource estimate used in this Preliminary Assessment was completed by G. H. Giroux P.Eng., MASc of Giroux Consultants Ltd, Vancouver, BC an independent Qualified Person as defined by NI 43-101. This mineral resource estimate forms part of the technical report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical report can be viewed at www.sedar.com. Please refer to Section 17 of this report for more information on the mineral resource estimate.

This report is intended to be read as a whole. Sections should not be relied upon or read out of context of the report as a whole.

3.1 Property Description

The Bronson Slope property is 100% owned by SGC, who is currently engaged in exploration activity on the property. A porphyry gold-copper-silver-molybdenum deposit is hosted within the property with approximate dimensions of 1.5km long and 0.4 to 0.6km wide. The depth of the orebody is not known at this stage however the current resource has defined a minimum vertical depth in excess of 900m.

The property is located in North West British Columbia, 280km northwest of Terrace, B.C., 110km northwest of Stewart, B.C., and 80km east of Wrangell, Alaska. The property consists of BC Mineral Claim, Tenure Numbers 517750, 517754, 523932, 523348, and 523933 and 6 Crown Granted Mineral Claims totalling approximately 186.9 hectares (Figure 6-3).

3.2 History and Ownership

3.2.1 Property Exploration and Development

The earliest recorded exploration of the Bronson Slope deposit occurred during 1907 and 1920. The Iskut Mining Company completed some surface and minor underground exploration along the Bronson Creek valley including some drifting, trenching and stripping of some gold bearing veins on the Red Bluff (now part of Bronson Slope property) and Iskut claims. Since this initial discovery various exploration and project development activities have taken place. A summary of this activity is provided below:

- 1962 to 1965 - Cominco performed scale surface mapping, prospecting and shallow drilling resulting in discovery of several promising copper and molybdenum mineralization zones.

- 1987 - Soil samples were taken along contour lines containing high grades of gold and the presence of other metals of value.
- 1988 - A 1938m diamond drilling program targeted high grade precious metal concentrations similar to the nearby Snip Gold Mine and Johnny Mountain Gold Mine however only low grade concentrations of gold, copper and molybdenum were recorded.
- 1990-1991 - SGC performed detailed geological mapping, prospecting, trenching and extensive geochemical soil sampling for Placer Dome Inc who had an option on a section of the Bronson slope property.
- 1992 - A complete review was completed by Burgoyne (1992) resulting in the recognition of a large porphyry copper-gold deposit.
- 1993 to 1997 - Further exploration programs were carried out. An induced polarization survey was performed followed by an extensive advanced exploration and drilling program, which resulted in 46 diamond drill core holes being drilled totalling more than 10 000m. In conjunction with the exploration program SGC commissioned a number of pre-feasibility studies including environmental, cash flow, metallurgical, capital and operating costs, geotechnical including pit slope and tailings dam, infrastructure and access and engineering scoping studies.
- 1997 - Two mineral titles were acquired from Prime Resources Group Inc., which were crucial to the forming of one continuous block from four principal Bronson Slope claims. The highwall area of the deposit was also acquired at this time. SGC also obtained access to previously drilled core in the area. Analysis of this data along with 6 more drill holes in the highwall zone resulted in a zone of significant gold mineralization striking parallel to the Bronson Slope porphyry deposit (strike length of approximately 800m with a true thickness of 60 to 70 metres).
- 1999 - SGC completed 19 drill holes over 1,495 metres on exploring for extensions to the Snip Gold Mine shear veins. Royal Gold Inc. funded this program.
- 2006 - Recompilation of drilling data was done followed by a core drilling program, which resulted in a further four holes for 562m.
- 2007 - 11 NQ diameter holes totalling 3936 metres were drilled in order to increase mineral resource confidence and also to develop additional resource.
- 2008 – Commissioning of this PEA and other associated development studies.

3.2.2 Mineral Resource and Estimates

A number of resource estimates have been undertaken by SGC between 1994 and 2008. The estimates have been completed by C. M. Turek (1994), G. H. Giroux and G. H. Raymond (1996 - 1997), G. H. Raymond (1997), A. A. Burgoyne (2006) and A. A. Burgoyne and G. H. Giroux (2007-2008). Refer to A. A. Burgoyne and G. H. Giroux (2008) for more details. A summary of the 2008 resource estimate has been included in Table 19-2 and Table 19-3 in Section 19.

3.3 Climate, Physiography, Infrastructure and Local Resources

The Bronson Slope property is located in the Iskut River Basin, approx 80km inland from the mouth of the Stikine River with climate conditions influenced by both the interior zone and the northwest coastal zone.

The terrain across the property is rugged with a range in elevation of approximately 900m with the mid-point of the mineralized zone outcropping approximately 400 metres above the potential mill site that has been proposed. Below the tree line the terrain is moderate to steep. Valleys are densely vegetated and peaks are barren, which is characteristic of their alpine nature.

Infrastructure from the adjacent Snip Gold Mine and the Johnny Mountain Gold Mine is all but gone however a seasonal camp with a capacity of approximately 20 to 30 people is located alongside the Iskut River adjacent to the Bronson airstrip. This airstrip is suitable to serve C-130 Hercules or similar aircraft. A core storage facility and some basic maintenance and general storage facilities are located alongside the Bronson airstrip. These facilities are utilised during the field exploration season which occurs from late May to early November.

The existing 40 km. of road access to site is comprised first of a 35km Forest Service Road leading from Bob Quinn Lake to the Eskay Creek Gold-Silver Mine turn-off and is under Road Use Permit to Barrick Gold. This 35 km. segment is followed by a segment of approximately 5 km. long operated under a License of Occupation by AltaGas for its Forrest Kerr hydroelectric project. A summary of site access and power infrastructure is found in Section 3.8. Proposed site layout of mining and site infrastructure and processing infrastructure is summarised in Section 7 and 18 respectively.

Allowances for the construction of water bores, genset pumps, pipelines (laid and buried), standpipes and storage facilities are included in the infrastructure costing within Section 25.8. Water storage tanks will be established at a higher elevation. Chlorination will also be provided for the potable water.

A sewage treatment plant is included in the mine infrastructure. Non-process waste water from some of the site facilities, such as the camp and offices, will be treated in this plant.

Up until the end of 2007 the British Columbia rate of unemployment was reducing from around 9% in 1998 to approximately 4%, which is defined as the point where the labour market is in balance: not facing any pressures either from a lack of workers or from excess supply of people looking for work. However due to a down turn in metal prices since the start of 2008 the rate of unemployment is increasing and there is uncertainty as to how long this trend may continue.

In the mining industry there continues to be a general lack of experienced personnel. Mining salaries and benefits remain high, reflecting the current demand for (and shortage of) skilled personnel. This may further pressure salaries and conditions offered by employers in the mining industry to ensure mines can attract suitably qualified personnel.

3.4 Geology, Mineralisation and Mineral Resource

The Bronson Slope Property is characterized as a large gold-copper-silver-molybdenum porphyry hydrothermal system consisting of a number of mineralized zones including the red bluff porphyry intrusive, the quartz magnetite replacement and stockworks, the lower sediments, the upper sediments and the hanging wall sediments.

The mineralized zones of potentially economic interest are briefly summarized in Figure 3-1 below.

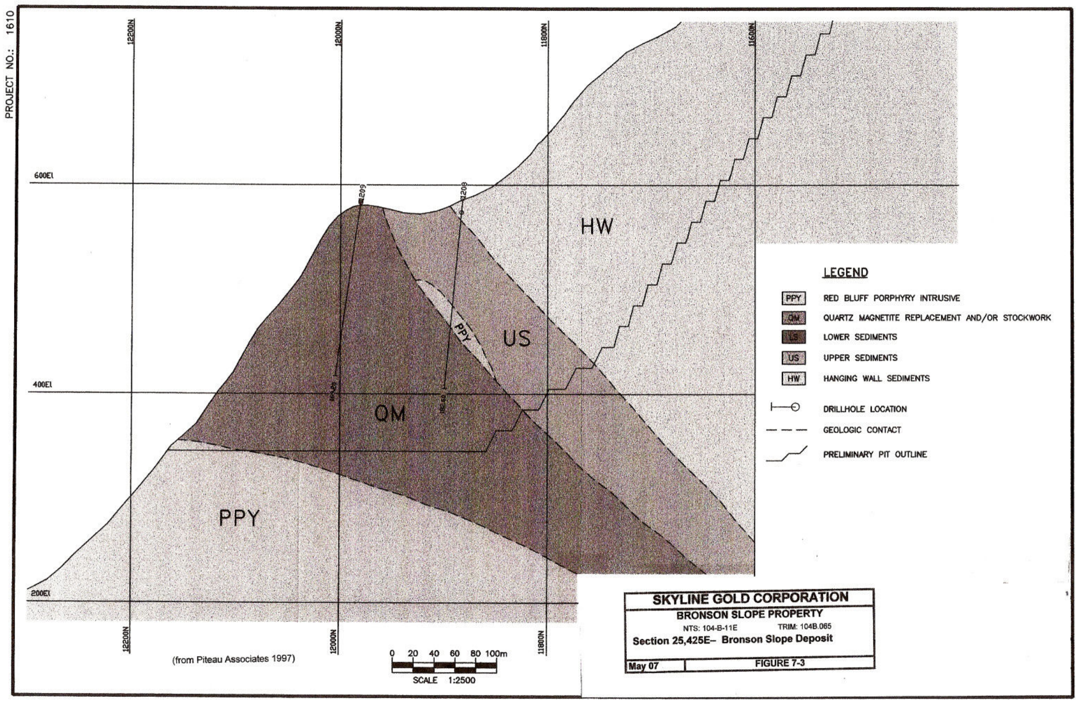


Figure 3-1: Mineralisation Section View of the Bronson Slope Deposit

Copper is present in the form of Chalcopyrite with minor Digenite, Covellite, Chalcocite, Bornite, Malachite, native copper and Cuprite. Gold occurs microscopically on Chalcopyrite, Pyrite and Quartz grain boundaries as native gold and electrum. Silver is present in the form of Argentite, Tetrahedrite, Galena, electrum and native silver and molybdenum is present as Molybdenite.

A current mineral resource estimate that meets CIMM resource standards and classifications has been completed and presented in the Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical report can be viewed at www.sedar.com. The resource estimate is based on kriging and block modelling.

The resource has been calculated based on two cases with differing metal prices. The case differences have been included in Table 3-1 below:

Table 3-1: Metal Prices Used for Resource Estimate

Metal	Case 1	Case 2
Cu	$1.50/lb	$2.00/lb
Au	$525/t.oz	$650/t.oz
Ag	$8/t.oz	$10/t.oz
Mo	$10/lb	$12/lb

*nb - all figures are in USD

Case 1 metal prices are based on the figures used in the prior 2007 Technical Report. The resource estimate for this case has been completed to provide a direct comparison between the 2007 resource estimate and the 2008 resource estimate. Case 2 metal prices are considered to be potentially more realistic and therefore a second resource estimate has been provided based on Case 2 metal prices.

The metal prices were used along with block based metallurgical recoveries to determine individual block values. The mineral resources presented in the tables below were then determined based on a cut off of USD 9.00 per tonne net recoverable value for Case 1 and Case 2 respectively.

Table 3-2: Mineral Resource Estimate based on Case 1 Metal Prices

Case 1 – Bronson Slope Resource Estimate (Cutoff USD 9/t NRV)					
Category	Metric Tonnes	Au g/t	Ag g/t	Cu %	Mo %
Measured	58,700,000	0.50	2.45	0.18	0.0058
Indicated	80,800,000	0.36	2.38	0.15	0.0094
Inferred	30,200,000	0.34	1.89	0.15	0.0070
Total Measured + Indicated	**139,500,000**	**0.42**	**2.41**	**0.17**	**0.0079**

Table 3-3: Mineral Resource Estimate based on Case 2 Metal Prices

Case 2 – Bronson Slope Resource Estimate (Cutoff USD 9/t NRV)					
Category	Metric Tonnes	Au g/t	Ag g/t	Cu %	Mo %
Measured	74,800,000	0.45	2.31	0.17	0.0059
Indicated	150,300,000	0.31	2.17	0.13	0.0087
Inferred	91,600,000	0.27	1.76	0.13	0.0080
Total Measured + Indicated	**225,100,000**	**0.36**	**2.22**	**0.14**	**0.0077**

3.5 Metallurgical Testing

A metallurgical study on the Bronson Slope samples was completed by Process Research Associates Ltd, Vancouver, BC in 1997. The objective of the metallurgical test program was to develop a preliminary process flowsheet and process design criteria for the Bronson Slope Project. Four different main mineralization types; upper sediment, upper sediment oxidized, porphyry and quartz magnetite were tested for metallurgy variability in the various mineralization zones. Another three composites categorized as “Average”, “Starter pit” and “High Grade” were prepared for the program. The test program was conducted primarily on the average composites which comprised blending of the four main mineralization types of the Bronson Slope deposit. The test program investigated the grind ability, copper and gold mineral recovery by batch and locked cycle flotation, magnetite recovery and molybdenum recovery. Thickening and product characteristic tests were also conducted on both copper concentrate and tailing samples of the locked cycle flotation.

The tests revealed that copper occurs predominately as chalcopyrite in a mixed iron oxide and silicate host. Pyrite is the other major gangue component in the mineralization types. Grind-recovery tests indicated that at a grind of 80% passing 108um, 82 to 88% of copper was recovered by flotation. A gravity concentration is recommended in the process to recover the coarse gold, which is not recovered by flotation. A combined (gravity and flotation) gold recovery of 84% is achieved for the average sample. Mill feed hardness in terms of Bond work index varied between 11.9 and 13.3kWh/t. These Bond work index figures were determined over the various mineralization types.

The flotation testing consisted of a series of batch flotation and cleaner flotation tests to evaluate the primary grind and recovery parameters, conditioning and reagent schemes for the various mineralization types. Preliminarily locked cycle tests were performed on the best conditions that were obtained from the batch flotation tests. Gold recovery correlates with the iron content, which indicates that the gold is loosely associated with the pyrite minerals.

A preliminary proposed flowsheet includes conventional crushing, grinding, rougher and scavenger flotation, regrind of rougher-scavenger bulk concentrate, three stage cleaner flotation using PAX as the primary collector and MIBC as the frother. Both copper concentrate and tailings are delivered to the dewatering facility to recover the water back to the processing plant for reuse. The makeup water is reclaimed from the tailings storage facility and from other site water collection systems.

The copper bulk concentrate was relatively clean, with minimum penalty elements identified.

Magnetite potential of the QM zone has yet to be fully determined. The metallurgical report prepared by Process Research Associates Ltd. "Metallurgical Study on the Bronson Slope Samples" dated July 1997, indicated an average head grade for a composite sample taken from the Bronson Slope QM (Quartz Magnetite) zone of 7.48% iron. Further research by BC Mining Research Ltd. indicate in a letter Progress Report dated September 18, 2008, and supported by discussion with the reports author Dr. Bern Klein, Ph.D., shows that an estimated 7% of a composite test sample from the Bronson Slope QM zone could be recovered as a high quality magnetite.

The progress report identified that the price paid by a North American consumer of magnetite for dense media separation/refining of coal was USD 211 per tonne. If magnetite recovery equivalent to 7% of mass of the QM zone resource is achieved, then based on a market price of USD 210 per tonne for high quality magnetite suitable for dense media separation, magnetite in the QM zone could potentially yield an economic benefit to the project. Further study of the magnetite potential including development of a magnetite resource for the Bronson Slope deposit is recommended.

3.6 Mine Plan

Mining Method Selection

During studies conducted by LAL in 2007 a number of mining methods were reviewed based on their cost effectiveness. These mining methods have been separated into two sections, in-pit transport and transport to stockpile (out of pit). For in-pit transportation, the following options and their associated direct costs were assessed:

- Load, haul and dump by loaders only (LHD)
- Conventional load and haul
- Mobile In-pit conveying system (grasshopper conveyors)

The cheapest of these options was to utilise in-pit conveying. This method has a low labour requirement (significant cost in Canada) however is more complex to schedule. A variation of this method has been adopted for the in pit material transport. Consideration has been made for the inflexibility of the conveying system when developing a LOM production schedule for the project. Further more detailed studies are also recommended to identify the impact of conveyor moves on mine productivity.

Methods of transporting to stockpile that were assessed include:

- Truck haulage
- Ore pass - a single ore pass, or dual ore passes
- High Angle Conveyor (HAC) regenerative system and non-regenerative system
- Dozer push

The HAC option was selected for this project as it provides a relatively low cost, relatively low risk option for mill feed and waste delivery from the mine area to the base of the Bronson

Slope. The HAC also generates an estimated 0.7Megawatts of power using its regenerative braking system.

Conventional loader and truck rehandle was selected as the method for transporting waste from the stockpile area at the base of the Bronson slope to the waste storage facility in the Triangle Lake area. Over relatively short but variable hauls the loader and truck load and haul option provides the most flexibility and is relatively cost efficient at current oil prices. Further study of the relationship between diesel oil prices and the efficiency of use of haul trucks in preference to a conveyor for waste rock transport to storage is required.

All material below the USD9.00/t NRV cut-off will be taken to the Triangle Lake storage area. An opportunity exists to segregate this storage area further into below cut-off NRV values. This will allow selective rehandling and processing of the higher value material at a later date if it is deemed economical to do so. It is recommended that further study of this below cut-off grade rehandling concept be conducted in the next phase of evaluation.

Available Hours and Utilization

As a conservative estimate, a general 85% mining mechanical availability has been used to account for maintenance, support, installation, etc. For critical high performance equipment higher availability estimates have been used given the more extensive and well planned preventative maintenance and support programs (maximum is 90%).

Consideration has been made for delays that will affect production over and above the equipment planned and corrective maintenance allowances. These delays have been summarized in Table 3-4. During some of these delays some opportune maintenance may be performed which will reduce the impact of planned maintenance on equipment operating hours. The assumptions surrounding this concept have also been included in Table 3-4. The estimate is likely to be conservative however due to the relatively variable operating climate in the area it was decided to use conservative values for this Preliminary Assessment and recommend further data collection and analysis is completed prior to preparation of a full project feasibility study.

Table 3-4: Summary of Planned Delays and Maximum Utilisation

Delay	Days	Opportune Maintenance	Net Days
Weather	28	3	25
Cleanup	3	0	3
Work Stoppages	0.5	0	.5
Misc	5	3	2
TOTAL	**36.5**	**6**	**30.5**
Utilisation	**90%**		**91.6%**

The physical and financial inputs and drivers used for the pit optimisation are provided in Table 3-5. Please note that all dollar figures provided in this table are in USD Currency.

Table 3-5: Initial Bronson Slope Mine Optimisation Parameters (USD Currency)

Parameter Description	Value & Comments	
Long term metal prices		
Cu ($/lb)	2	Supplied by SGC
Au ($/oz)	650	
Ag ($/oz)	12	
Mo ($/lb)	12	
Capital	$175,000,000	
Residual	10%	Supplied by SGC
Discount rate	8%+/-	
Geotechnical design parameters		
Average - All domains	55 degrees -->	Adapted from Piteau Report
Mining parameters		
Bench Height (m)	10	
Mining cost - Base ($/t)	$1.50/t	
Adjustment for depth ($/t/bench)	not considered	
Mining dilution (%)	5%	
Mining recovery (%)	95%	
Concentrator parameters		
Expected Mill Throughput (t/annum)	5.5Mtpa	
Variable processing cost ($/t milled)	$6.00/t	
Concentrator Recoveries		
Au	84%	
Ag	61%	
Cu	87%	
Mo	46%	
Smelting and refining		
Smelting recovery(s) (%)	97%	
Refining recovery(s) (%)	100%	
Smelting cost ($/t concentrate)	$85/t	
Refining cost(s)	$0.075/lb Cu $6.00/oz Gold, $0.4/oz Silver	
Concentrate Transport costs ($/t)	$50/t of concentrate	
Concentrate moisture content (%)	8%	
Marketing cost(s)	Nil	
Sales cost(s)	Nil	
Sales commission(s) (%)	Nil	
General and Administration Overhead costs		
Admin and Overhead unit cost ($/t milled) –	$1.00/t milled	

At the time of the optimisation a processing rate of 5.5Mtpa was considered. This has since been revised to 5.098Mtpa based on the process flow sheet and also on the expected availability of the concentrator. The effect of this small change in processing rate on the resulting optimised pit and schedule is considered to be insignificant for this study, however further revisions of the pit optimisation are recommended for the next phase of the project.

Representative mining, milling, general and administration costs and also capital costs were selected prior to completion of the study for use in the optimisation. These have since been reviewed through a more detailed cost estimating process.

The pit optimisation completed for this Preliminary Assessment used NPV as the initial key performance measure and resulted in the optimum final pit selection summarized in Table 3-6. IRR, payback period and mine life were secondary measures.

Table 3-6: Optimum Pit Sequence Summary – Initial Whittle Model

Movement	Tonne
Mill feed	87,342,491
Waste (reject)	2,265,513
Waste (other)	66,848,726
Total	156,456,730
Life (year)	17.9

Detailed Pit Design

Geotechnical guidance was taken from the report titled “Preliminary Geotechnical Assessments and Slope Design Studies for the Bronson Open Pit” completed by Piteau Associates in March 1997. Based on this review a representative overall pit wall angle of 50 degrees has been used for the pit design. This report has identified a number of geotechnical domains surrounding the final pit. The following pit design constraints have been recommended:

- Bench Height – 20m
- Batter Angle – 75^{o}
- Berm Width – 11.2 to 11.4m
- Overall Wall Angle – 50^{o}

However it is important to note that this geotechnical study was completed more than 10 years ago and further exploration drilling and core logging has taken place. Further review of the geotechnical conditions within and surrounding the revised pit limits is essential to ensure that the most accurate modelling of geotechnical risks is completed prior to commissioning of the project.

Figure 3-2 to Figure 3-3 show the detailed pit designs individually. A series of sections through these pits and the resource model have been provided in Section 25.

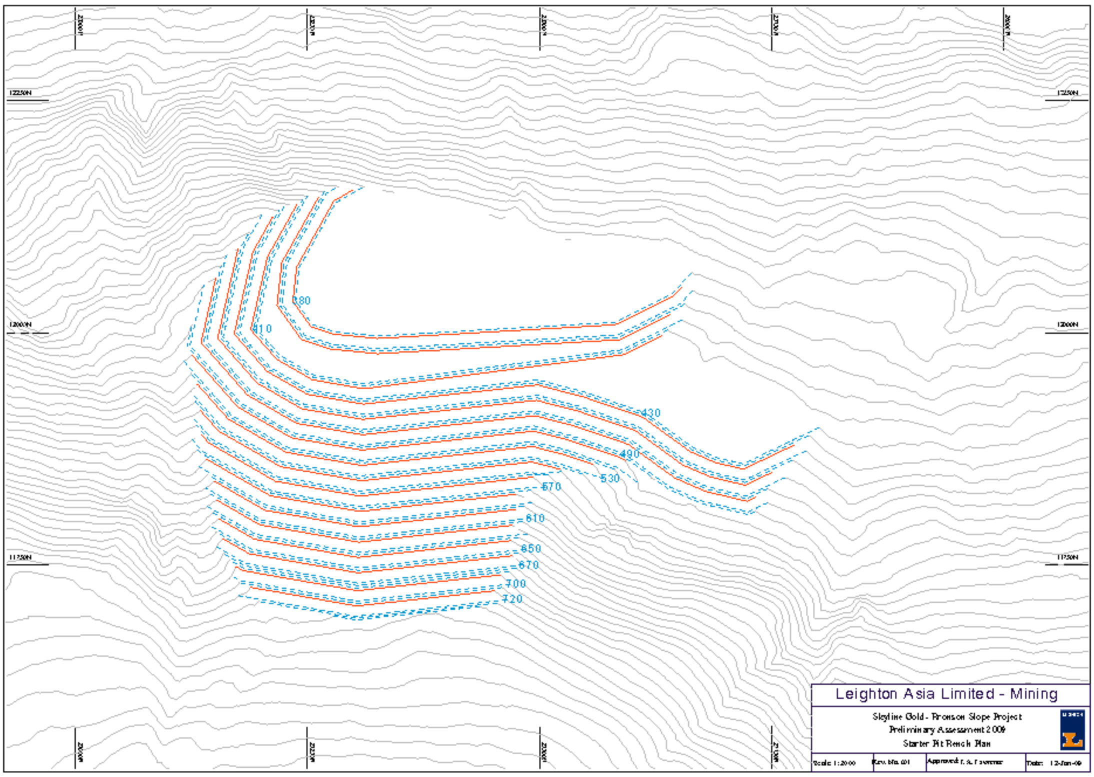


Figure 3-2: Initial Pit Bench Plan

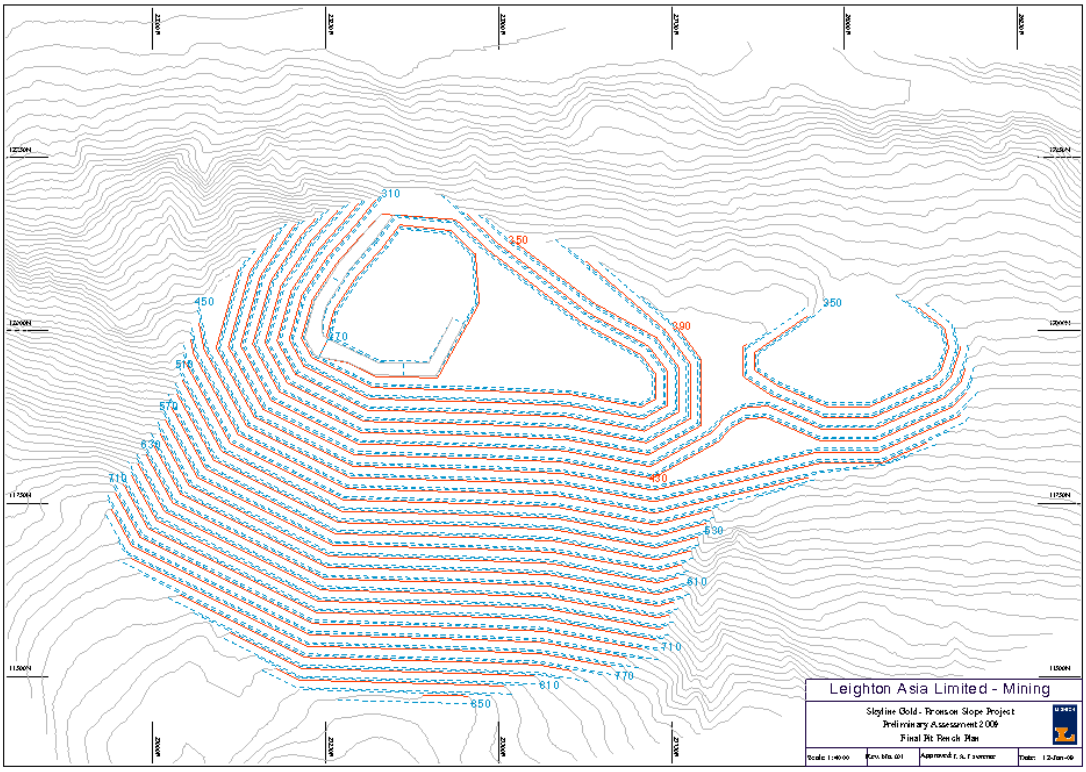


Figure 3-3: Final Detailed Pit Plan

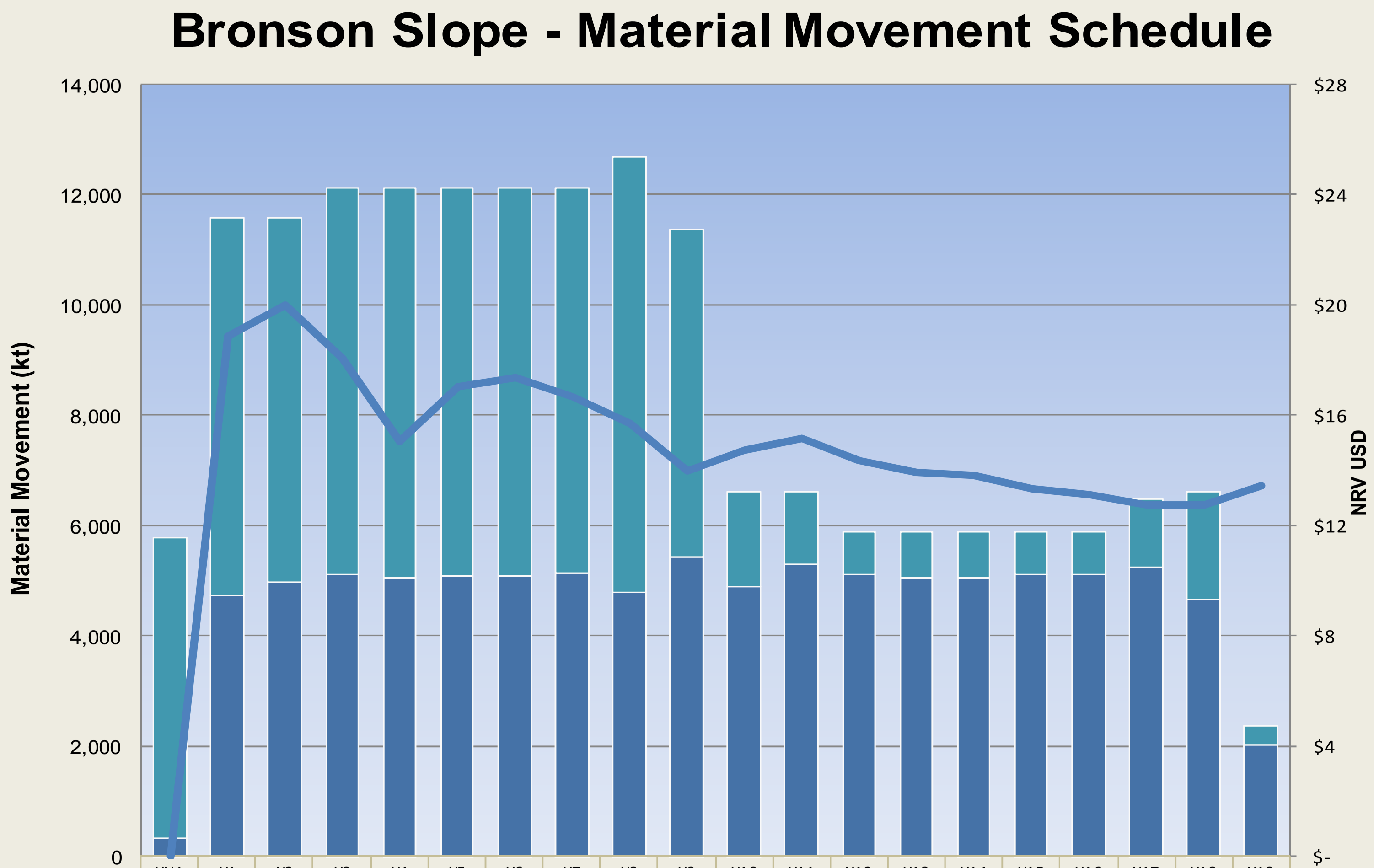
The full details of the bench grades and tonnages are included in Appendix 1. The sum of both of these phases (total LOM mill feed tonnage and grade) is included in Table 3-7.

Table 3-7: LOM Mill Feed Tonnage and Grade

Phase I&II (Total) - Tonnes and Grades ($9/t NRV Cut-off)					
Category	**Metric Tonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Mo %**
Measured	62,079,392	0.174	0.477	2.277	0.005
Indicated	31,390,522	0.120	0.393	2.497	0.007
Inferred	473,839	0.046	0.417	3.892	0.003
Total Measured + Indicated	**93,469,914**	**0.155**	**0.446**	**2.34**	**0.0058**
Total Waste (incl Inferred)	78,524,707	**Strip Ratio:**		**0.83**	
Total Mill Feed and Waste	**171,994,621**				

LOM Production Schedule

The schedule allowed for the progressive sequencing of material movement by phased pit design and bench and assumed that for each bench the material types were mined in weighted equal portions until the bench was completely mined out before progressing to the next bench. The full detailed production schedule is presented in Table 25-8. A graphical representation of the LOM schedule has been provided in Figure 3-4 below. A table containing the metal production by year has been provided in Table 3-8.



	YN1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
Waste (kt)	5,464	6,836	6,611	7,010	7,064	7,038	7,053	7,003	7,881	5,926	1,726	1,319	779	846	850	778	793	1,236	1,951	328
Bronson Ore (kt)	330	4,752	4,977	5,130	5,075	5,102	5,087	5,137	4,810	5,445	4,896	5,303	5,126	5,058	5,054	5,126	5,111	5,248	4,671	2,043
NRV (USD/t ore)	$-	$18.86	$20.00	$18.05	$15.07	$17.04	$17.37	$16.69	$15.68	$14.00	$14.72	$15.19	$14.34	$13.92	$13.82	$13.33	$13.10	$12.73	$12.73	$13.42

Figure 3-4: Material Movement Schedule

Table 3-8: Metal Production Schedule

Variable	Unit	Y -2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10
Mill Feed	000's t	0	0	4,779	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098
Copper Grade	%	0.00%	0.00%	0.21%	0.22%	0.16%	0.09%	0.15%	0.17%	0.17%	0.16%	0.17%	0.16%
Copper Recovery	%	0.00%	0.00%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%
Recovered Copper (86.6% Recovery)	000's lbs	0	0	19,171	21,122	15,941	8,410	14,870	16,953	16,558	15,787	16,139	15,911
Gold Grade	g/t	0.000	0.000	0.525	0.567	0.570	0.564	0.539	0.517	0.491	0.456	0.366	0.402
Recovered Gold (85.4% Recovery)	000's oz	0.0	0.0	68.8	79.4	79.7	79.0	75.4	72.3	68.7	63.8	51.3	56.2
Silver Grade	g/t	0.00	0.00	2.46	2.65	2.77	3.02	2.77	2.57	2.45	2.37	2.05	2.46
Recovered Silver	000's oz	0.0	0.0	240.5	276.8	288.9	315.6	289.4	267.9	255.5	247.9	214.5	256.5
CONCENTRATE GRADE													
Copper	%	0.0%	0.0%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%
Gold	g/t	0.00	0.00	62.04	64.95	86.43	162.28	87.65	73.74	71.75	69.83	54.91	61.07
Silver	g/t	0.00	0.00	216.77	226.45	313.21	648.50	336.29	273.02	266.63	271.38	229.68	278.55
Concentrate Moisture Content	%	0.0%	0.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%
WET CONCENTRATE TONNAGE	wmt	0	0	37,268	41,062	30,989	16,349	28,906	32,957	32,188	30,689	31,373	30,931

Variable	Unit	Y 11	Y 12	Y 13	Y 14	Y 15	Y 16	Y 17	Y 18	Y 19	AVG	TOTAL
Mill Feed	000's t	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	2,035	4,920	93,480
Copper Grade	%	0.17%	0.15%	0.15%	0.14%	0.14%	0.14%	0.13%	0.14%	0.15%	0.156%	
Copper Recovery	%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	86.60%	
Recovered Copper (86.6% Recovery)	000's lbs	16,227	14,879	14,351	14,046	13,599	13,216	12,637	13,172	5,973	14,682	278,962
Gold Grade	g/t	0.422	0.409	0.401	0.399	0.382	0.378	0.370	0.359	0.360	0.449	
Recovered Gold (85.4% Recovery)	000's oz	59.0	57.2	56.1	55.9	53.5	52.9	51.8	50.2	20.1	61.9	1,151
Silver Grade	g/t	2.35	2.09	1.91	2.06	2.12	2.13	2.14	2.17	2.03	2.356	
Recovered Silver	000's oz	245.5	218.4	199.1	215.1	221.4	222.3	223.6	226.4	84.8	242.4	4,510
CONCENTRATE GRADE												
Copper	%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	
Gold	g/t	62.83	66.48	67.52	68.73	67.95	69.12	70.84	65.88	58.20	73.81	
Silver	g/t	261.45	253.59	239.73	264.68	281.31	290.65	305.76	296.96	245.27	249.20	
Concentrate Moisture Content	%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	
WET CONCENTRATE TONNAGE	wmt	31,545	28,925	27,899	27,305	26,436	25,691	24,567	25,606	11,612	29,067	542,298

The schedule demonstrates that the initial pit is mined exclusively until near the end of year 1, when the stripping of the waste and mining of the high wall area of the final pit takes place. Between year 3 and 6 mill feed is encountered in the highwall zone of the second (final) pit and is fed through the process. This mill feed material has a low copper grade but is high in gold grade. Further study needs to be conducted to determine whether the gold recoveries of the highwall zone are achievable with very low copper grades. An alternate plan may be required to stockpile the highwall zone material and progressively blend it into the mill feed to reduce the impact of the reduced copper grades.

Between year 6 and 8 the majority of the mill feed is taken from the starter pit and further waste stripping is conducted from the highwall zone of the final pit. At the point when the base of the starter pit is reached the final pit becomes mill feed bound (year 9) and the production fleet is reduced to one excavator, crusher combination. From year 10 onwards, a single production fleet operates out of the final pit mining bench by bench until the base of the pit is reached in year 19.

The optimum production schedule requires a high strip ratio early in the mine life. This is somewhat contradictory to conventional phased pit mining techniques utilised by other mining projects around the world. The high strip ratio requirements are a result of the narrow shape of the pit and the orebody and also the higher grade of gold in the mill feed that is present in the high wall zone. Stripping of the final pit needs to start early in the mine life (towards the end of year 1) in order to ensure consistent mill feed of 5mtpa at the later stages in the mine schedule. Also due to the high proportional value of gold as compared to copper for this optimisation model, the higher grade gold mill feed found in the highwall zone is targeted early in the mine life and cash flow resulting in better financial performance for the project.

3.7 Processing

A conceptual multi metals flotation concentrator plant based on crushing, grinding, gravity, flotation, thickening and filtration is proposed for the Bronson Slope deposit to produce gravity gold and copper gold bulk concentrate that will be transported off site for smelting and refining. A 15,000 tonnes per day (tpd) mill feed processing rate and a concentrator utilisation of 93% will produce a total of 29,000 tonne per annum (tpa) of copper concentrate containing both gold and silver. A process flow sheet design has been provided in Figure 18-3 in Section 18.

The concentrator plant will consist of a SAG mill with a pebble crusher. The SAG mill discharge is combined with the ball mill discharge and gravity tail and fed to the ball mill cyclones. Overflow of the cyclones will be fed to the copper rougher flotation and the underflow of the cyclones is recycled to the ball mill for further liberation. A portion of the cyclones underflow is diverted to the gravity separator for coarse gold recovery.

Copper concentrate recovered from the rougher and scavenger flotation is fed to a regrind mill for further size reduction prior to the cleaner stages. Copper concentrate will be thickened with the aid of flocculants in a conventional thickener and filtered to reasonable moisture for dispatch off site.

The rougher and scavenger tails are disposed to the main tailing facility located at a valley southwest of the plant. At the end of operations the tailings will be covered with an impervious cover and flooded with water to prevent the oxidation of residual sulphidic material. Surplus water derived from the tailings and from the runoff is pumped to the plant as a source of process water to the plant or will be discharged into Bronson Creek near the plant after complying with the environmental standards set out in the environmental management plan.

3.8 Access and Power

At the time of the site inspection in July 2007, the project was accessible using fixed wing or helicopter air services via a 1780 metre long gravel airstrip which is located adjacent to the confluence of Bronson Creek and the Iskut River. SGC has a 1780m x 220m license of occupation for this airstrip. Typical fly-in origin points include Wrangell, Bob Quinn airstrip, Smithers, and Terrace and the site is accessible from Vancouver by turboprop aircraft. There is a network of basic access roads on and around the property that receive some maintenance. An old access road traverses from the Bronson airstrip around the south of the Snip Gold Mine / Bronson Slope Deposit up to the old Johnny Mountain Gold Mine site. This road requires upgrading before it can be used by vehicles to access the top of the Bronson Slope property.

An all weather mine access road runs from Bob Quinn on the Highway 37 (Stewart-Cassiar Highway) to the Barrick Gold Corporation owned Eskay Creek Gold Mine. A connecting development access road has been constructed to the Forest Kerr Hydropower construction site. The Bronson Slope property is approximately 30 km east of this access road along the Iskut River. Forsite Consultants Ltd were commissioned in 2006 to provide a conceptual access road design and location supported by a construction schedule and a preliminary cost estimate. This proposal is for a permanent mine access road for the Bronson Slope Property if the project was to go into construction and operation. It is divided into 3 sections: from Forest Kerr to Bug Lake, from Bug Lake to Bronson airstrip and then to Bronson Creek Crossing. More details can be found in Section 7.3.

It is assumed that power will be supplied from a proposed Northern Transmission Line main grid line located at Bob Quinn Lake (approximately 60km from site). An opportunity exists for a direct connection to the BC Hydro grid near the proposed Forrest Kerr hydro power station, which is much closer to site (approximately 25km from site). Other alternative electricity generation and supply options are also being evaluated including self-generation of power using hydro assets for which SGC have submitted hydro generation license applications and for which SGC has received Notice of Sufficiency of Application from the BC government. Power will be supplied using a 138kV transmission line, which will run partially within the Access Road right of way. Up to 20MW of power will be provided from the Forest Kerr run-of-river hydroelectric power station, located 25km west of the Bronson Slope property. SGC is further investigating the potential provision of power using two of its hydro license applications (Snippaker Creek and Bronson Creek) to generate 25 MW of power using water storage facilities.

3.9 Capital and Operating Costs

As part of the preliminary assessment a cost estimate was completed to identify the capital and operating costs expected for the Bronson Slope Project based on the selected mining and processing methods and schedules. It is important to note there are no allowances for currency exchange fluctuation and price escalation in the estimated capital and operating costs.

This cost estimation has been completed by utilizing a combination of techniques which have been summarized in this section. Further detail is provided in Section 25.

3.9.1 General Site Infrastructure Capital Estimate

The following table provides a summary of the expected buildings, services and infrastructure capital costs for the project.

Table 3-9: Final Estimated Infrastructure Capital Costs (excl. tax considerations)

Item Description	Estimated allowance/cost (CAD 000's) 2008
Off-site Infrastructure	$24,594
Site Development	$5,495
Infrastructure - Utilities	$3,054
Infrastructure - Buildings and Facilities	$9,662
Total Direct Costs	**$42,805**
Indirect Costs	$12,318
Total Capex for Site Infrastructure	**$55,123**

3.9.2 General and Administration Operating Cost Estimate

General and Administration costs have allowed for a number of items including administration staff, general consumables, minor support related equipment and maintenance costs, fees and insurance and other minor non production specific cost items. The total allowance for the LOM general and administration costs equates to CAD 90.1 Million (CAD 0.55/t mined, CAD 0.98/t milled).

3.9.3 Processing Capital Cost Estimate

The estimated capital cost for the concentrator plant discussed in Section 18 is based on the following:

- Preliminary flow-sheet and equipment requirements.
- Equipment costs from vendors' estimation, suppliers' quotation and Western Infomine's estimator cost data (2007, 2008).
- Escalation of the 1997 Feasibility study costs to present value using Western Infomine's capital cost index data.
- Mechanical installation costs for equipment are calculated by multiplying the labour cost by the number of labour hours required.
- Other processing plant installation costs are either adjusted by an allowance factor or using an escalated capital cost index.
- Comparison to cost models of similar operations.

Table 3-10: Summary of Processing Direct Capital Costs

Direct Costs	Total (CAD 000's)
AREA 13 – Primary Crushing and Ore Stockpile*	$6,402
AREA 16 – Grinding	$43,014
AREA 17 – Copper Flotation	$9,730
AREA 18 – Copper Concentrate Dewatering	$2,181
AREA 20 – Reagent Systems	$1,014
AREA 28 – Process Utilities	$3,618
AREA 31 – Tailings	$21,064
Total Direct Costs	**$87,023**

*Mobile crusher included in Mining capex

The estimate proportions of labour, mechanical, structural and electrical of the capital cost are summarized in Table 3-11.

Table 3-11: Processing Capital Costs by Discipline

Direct Costs	CAD 000's
Labour	$ 8,994
Mechanical	$43,027
Structural	$32,619
Electrical	$2,384
Total	**$87,023**

The indirect costs are presented in Table 3-12.

Table 3-12: Summary of Indirect Costs

Indirect Costs	CAD 000's
Construction indirect (3% of direct cost)	$2,611
EPCM Engineering services (10% of direct cost)	$8,702
First fill inventory (5% of direct cost)	$4,351
Total Indirect Cost	**$15,664**
Contingency (15% of direct cost)	**$13,053**
Total Costs (Direct + Indirect + Contingency)	**$115,740**

3.9.4 Concentrator Plant Operation Cost Estimate

The estimated plant operating costs are based on the following:

- Manpower, standard labour rate and salary packages compiled by Western Infomine's statistic data.
- Mill consumables and cost are derived from typical consumption rates in the industry.
- Reagent consumables are determined from the preliminary test work indicating the typical reagent consumption that can be expected.
- Power costs are determined by multiplying equipment power requirement by the commercial power cost supplied in British Columbia.

A summary of the costs are shown in Table 3-13 below.

Table 3-13: Summary of Process Plant Operating Costs

No.	Description	CAD/tonne of mill feed
1	Process Labour	$1.10
2	Power cost	$1.53
3	Consumable & Maintenance	$2.52
4	Surface Equipment	$0.08
Total unit direct process operating cost		**CAD 5.23/t of mill feed**
USD Process Operating Cost		**USD 4.45/t of mill feed**

3.9.5 Mining Capital Cost Estimate

The estimates for mining capital were prepared for new equipment supplied on a turn-key basis and include for the anticipated cost of sea and land transportation of each item of equipment, import duties and associated port charges, and erection and commissioning costs for the Bronson Slope site location. VAT is specifically excluded from the pricing.

All equipment costs are in CAD and primarily based on 2nd half 2008 pricing with exchange rates of USD 0.85 and AUD 1.15 to CAD applied. No escalation of costs is estimated or included.

Replacement life for equipment is based on LAL's extensive experience operating a wide range of mining equipment. The hourly operating costs calculated for these equipment items are based on operating the equipment for its full expected life (average Whole of Life operating costs).

Table 3-14 summarises the estimate of overall equipment capex requirements for the life of mine. Initial and sustaining/replacement capital estimates are provided.

Table 3-14 Summary LOM Mining Capital Requirements

Initial and Sustaining Mining Capital Cost Items		
Initial Items	Unit Price (CAD 000's)	Total (CAD 000's)
Pre-Production Mining Costs	$ 18,090	$ 18,090
Conveying System (2300tph)	$ 11,958	$ 11,958
Rehandle Loader	$ 1,694	$ 3,388
Mobile Crusher (1200tph)	$ 1,684	$ 3,368
90t Class Rear Dump	$ 1,506	$ 6,024
Hyd Excavator 6.7m3	$ 1,365	$ 2,730
Drill DTH 203mm	$ 971	$ 1,942
Track Dozer 15' Blade	$ 682	$ 1,364
Watercart 20kL	$ 494	$ 494
Grader 12' Blade	$ 341	$ 341
IT Tool Carrier	$ 265	$ 265
25T Exc/R.breaker	$ 175	$ 175
Roller	$ 94	$ 94
SubTotal - Initial Items		**$ 50,233**
Replacement Items	Unit Price (CAD 000's)	Total (CAD 000's)
Conveying System (2300tph)	$ 11,958	$ -
Rehandle Loader	$ 1,694	$ 3,388
Mobile Crusher (1200tph)	$ 1,684	$ 1,684
90t Class Rear Dump	$ 1,506	$ 3,012
Hyd Excavator 6.7m3	$ 1,365	$ 4,095
Drill DTH 203mm	$ 971	$ 971
Track Dozer 15' Blade	$ 682	$ 682
Watercart 20kL	$ 494	$ -
Grader 12' Blade	$ 341	$ -
25T Exc/R.breaker	$ 175	$ 350
Roller	$ 94	$ -
SubTotal - Replacement Items		**$ 14,182**
TOTAL MINE CAPITAL COSTS (CAD 000's)		**$ 64,415**

It is important to note that for this operation the mining costs include the crushing of the mill feed and the waste product, which is a requirement to transport the material from the mine face using the conveying system. In most conventional operations the cost of the crushing facilities would be included in the process plant capital costs.

3.9.6 Mining Operating Cost Estimate

All equipment costs are in CAD and primarily based on 2nd half 2008 pricing with exchange rates of USD$0.85 and AUD$1.15 to CAD applied. No escalation of costs is estimated or included. Pricing of all costs in this study also excludes consideration for taxation.

A summary of the equipment fleet chosen and numbers required for mine production plan years -2 to 19 are listed in Table 3-15.

Table 3-15 Key Mine Equipment Schedule Requirements by Period

Description	Model	Yr -1	Yr 1 to 8	Year 10 to 19
100t Excavator	Komatsu PC1250	1	2	1
Mobile Crusher	Nordberg LT140	1	2	1
High Angle Conveyor	ICSI	1	1	1
20T Rockbreaker	Cat 330	1	1	1
Grader 12' blade	Cat 12M	1	1	1
Water Cart 20 kL	TBA	1	1	1
Loader 8.6m^3	Cat 990H	1	2	1
203mm DTH Drill	TBA	1	2	1
Track Dozer 15' blade	Cat D8	1	2	1
90t Class RD Truck	Cat 777F	2	4	2
Ancillary Loader IT	Cat 966 IT	1	1	1
Roller	TBA	1	1	1

Table 3-16 provides a summary of the total and unit mining costs expected for the life of the Bronson Slope Project.

Table 3-16: Mine Operating Cost Summary

Mine Operating Costs	Unit	Total CAD 000's
Drilling	**(CAD,000)**	**$24,198**
	(CAD/tonne mined)	**$0.146**
	(CAD/tonne milled)	**$0.259**
Blasting	**(CAD,000)**	**$56,273**
	(CAD/tonne mined)	**$0.339**
	(CAD/tonne milled)	**$0.602**
Loading & Conveying	**(CAD,000)**	**$103,929**
	(CAD/tonne mined)	**$0.625**
	(CAD/tonne milled)	**$1.112**
Rehandle Load and Haul	**(CAD,000)**	**$68,479**
	(CAD/tonne mined)	**$0.412**
	(CAD/tonne milled)	**$0.733**
Ancillary - Roads and Dumps	**(CAD,000)**	**$53,359**
	(CAD/tonne mined)	**$0.321**
	(CAD/tonne milled)	
Total Direct Costs	**(CAD,000)**	**$292,836**
	(CAD/tonne mined)	**$1.77**
	(CAD/tonne milled)	**$3.14**

3.9.7 Project Capital Cost Summary

A summary of the Project Capital Costs has been included in Table 3-17 below.

Table 3-17: LOM Project Capital Cost Summary

Project Capital Cost Summary	Unit	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Processing Direct Costs	CAD 000's	$ 43,512	$ 43,512	$ 400	$400	$400	$400	$400	$ 400	$ 400	$400	$ 400
Infrastructure Directs	CAD 000's	$ 21,403	$ 21,403	$ 100	$100	$100	$100	$100	$ 100	$ 100	$100	$ 100
Mine Pre-production	CAD 000's	$ 878	$ 16,690	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mining Equipment	CAD 000's	$ 8,960	$ 13,440	$ 9,408	$ -	$ -	$ -	$ -	$3,234	$6,071	$ -	$1,684
Reclamation	CAD 000's			$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Recovery	CAD 000's			$ -	$ -	$ -	$ -	$ -	$ (323)	$ (607)	$ -	$ (168)
Total Directs	**CAD 000's**	**$ 74,753**	**$ 95,044**	**$ 9,908**	**$500**	**$500**	**$500**	**$500**	**$3,411**	**$5,964**	**$500**	**$2,016**
Mill Indirects	CAD 000's	$ 7,832	$ 7,832	$ -								
Infrastructure indirects	CAD 000's	$ 6,159	$ 6,159	$ -								
Total Indirects	**CAD 000's**	**$ 13,991**	**$ 13,991**	**$ -**								
Directs + Indirects	CAD 000's	$ 88,744	$109,035	$ -								
Contingency ==> (15%)	CAD 000's	$ 13,312	$ 16,355	$ -								
Working Capital				$14,126								
LOM Project Capital Schedule	**CAD 000's**	**$102,055**	**$125,391**	**$24,034**	**$500**	**$500**	**$500**	**$500**	**$3,411**	**$5,964**	**$500**	**$2,016**

Project Capital Cost Summary	Unit	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Total
Processing Direct Costs	CAD 000's	$400	$ 400	$400	$400	$ 400	$400	$400	$400	$ 400	$ -	$ 94,223
Infrastructure Directs	CAD 000's	$100	$ 100	$100	$100	$ 100	$100	$100	$100	$ 100	$ -	$ 44,605
Mine Pre-production	CAD 000's	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 17,569
Mining Equipment	CAD 000's	$ -	$1,653	$ -	$ -	$1,540	$ -	$ -	$ -	$ -	$ -	$ 45,990
Reclamation	CAD 000's	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$5,000	$ 5,000	$ 10,000
Capital Recovery	CAD 000's	$ -	$ (165)	$ -	$ -	$ (154)	$ -	$ -	$ -	$ -	$(15,561)	$ (16,980)
Total Directs	**CAD 000's**	**$500**	**$1,988**	**$500**	**$500**	**$1,886**	**$500**	**$500**	**$500**	**$5,500**	**$(10,561)**	**$ 195,407**
Mill Indirects	CAD 000's											$ 15,664
Infrastructure indirects	CAD 000's											$ 12,318
Total Indirects	**CAD 000's**											
Directs + Indirects	CAD 000's											
Contingency ==> (15%)	CAD 000's											$ 29,667
Working Capital												$ 14,126
LOM Project Capital Schedule	**CAD 000's**	**$500**	**$1,988**	**$500**	**$500**	**$1,886**	**$500**	**$500**	**$500**	**$5,500**	**$(10,561)**	**$ 267,183**

3.9.8 Project Operating Cost Summary

A summary of the site wide operating costs has been included in the following Table 3-18. More detail of the project operating cost estimate has been provided in Section 25.8.

Table 3-18: LOM Project Operating Cost Summary

Cost Category	Unit	LOM
Direct Mining Costs	(CAD,000)	$292,836
	(CAD/tonne Mined)	$1.77
	(CAD/tonne Milled)	$3.14
Overheads and Administration	(CAD,000)	$91,652
	(CAD/tonne Mined)	$0.55
	(CAD/tonne Milled)	$0.98
Processing Costs	(CAD,000)	$488,901
	(CAD/tonne Milled)	$5.23
Total Site Operating Costs	(CAD,000)	$873,388
	(CAD/tonne Milled)	$9.34
	(USD/tonne Milled)	$7.94

3.10 Project Economics

Metal prices of USD 700/t.oz for gold, USD 2.00/lb for Copper and USD 15/t.oz for Silver were used for the cash flow analysis. An exchange rate of USD 0.85: CAD 1.00 has been used.

A summary of the financial performance of the project can be seen in Table 3-19 below. A more detailed project cash flow has been presented in Figures 25-27 to 25-29 and further in Appendix 3.

Table 3-19: Financial Performance Measure Summary

Financial Performance Summary				
			Gross	**Net Tax**
		Project IRR	**11.0%**	**10.0%**
	Disc Rate	**Unit**		
Project NPV-->	10.0%	million CAD	14.0	0.0
	7.5%	**million CAD**	**59.3**	**38.3**
	5.0%	million CAD	123.5	92.0
	0.0%	million CAD	351.1	279.4
Payback		Years	8.2	
MineLife		Years	18.4	

The pro-rata cash costs and net gold cash cost after co-product and by-product credits have been calculated within the financial model and have been provided in Table 3-20.

Table 3-20: Project Cash Cost Summary

	Gold (USD/t.oz)	Copper (USD/lb)	Silver (USD/t.oz)
Pro Rata Cash Cost	$ 428.46	$ 1.24	$ 9.28
Net Cash Cost after credits	$ 231.86		

A sensitivity analysis has also been completed on the pre tax cashflow to demonstrate the sensitivity of the financial performance of the project to major inputs such as metal prices, capital costs, operating costs, exchange rate and mill feed grade. A summary of the results have been included in Table 3-21and Figure 3-5 below.

The NPV and IRR are most sensitive to the USD:CAD exchange rate, followed by gold price, gold grade, operating costs, copper price, copper grade and finally least sensitive to changes in the Initial capital costs.

Table 3-21: Cashflow Sensitivity Summary

Pre-tax IRR Value Gross Sensitivity					
Variant	**-20%**	**-10%**	**0%**	**+10%**	**+20%**
Cu Price	7.5%	9.3%	**11.0%**	12.6%	14.2%
Au Price	5.6%	8.4%		13.5%	15.9%
US/C Exch	20.7%	15.4%		7.0%	3.4%
OpCost	15.9%	13.5%		8.4%	5.7%
Initial Capex	14.4%	12.6%		9.6%	8.5%
Cu Grade	8.2%	9.6%		12.3%	13.6%
Au Grade	5.7%	8.4%		13.5%	15.8%
After Tax IRR Value Net Sensitivity					
Variant	**-20%**	**-10%**	**0%**	**+10%**	**+20%**
Cu Price	6.9%	8.5%	**10.0%**	11.5%	12.9%
Au Price	5.2%	7.6%		12.2%	14.4%
US/C Exch	18.6%	13.9%		6.5%	3.3%
OpCost	14.4%	12.2%		7.7%	5.3%
Initial Capex	13.5%	11.6%		8.6%	7.4%
Cu Grade	7.5%	8.7%		11.2%	12.4%
Au Grade	5.3%	7.7%		12.2%	14.3%

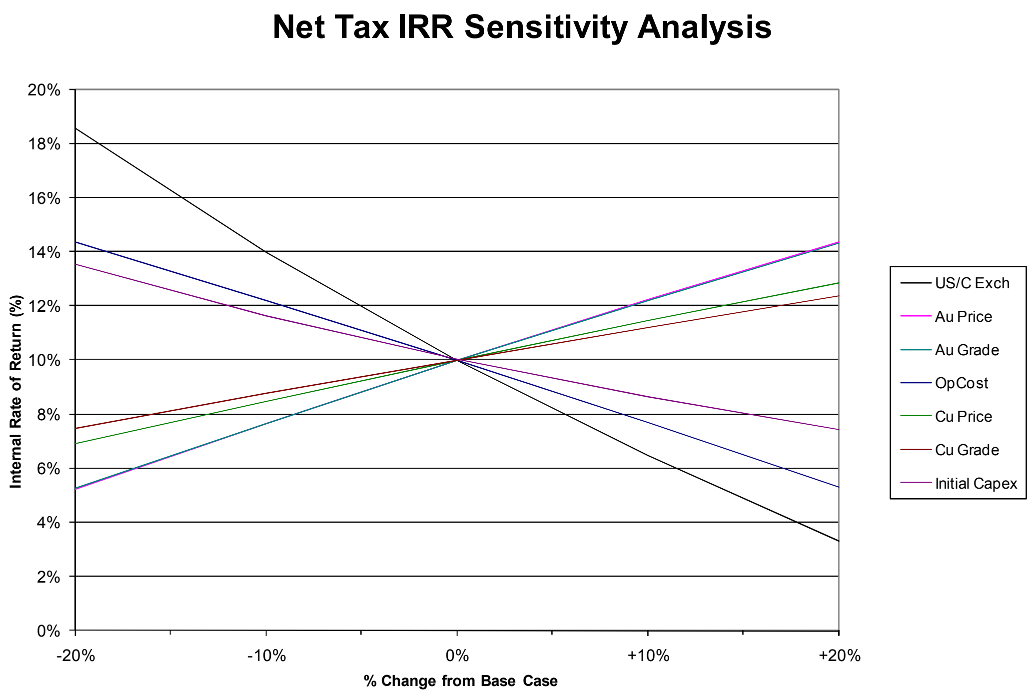


Figure 3-5: Net IRR Sensitivity

3.11 Post Preliminary Assessment Whittle Optimisation

Results from the detailed mine plan and cost estimate presented in this assessment identified that the original inputs used in the Whittle model to generate the most optimum pit were not completely representative of the costs and characteristics identified when the final mining and processing method was selected for the project. Table 3-22 shows a comparison of the detailed design, cost and revenue estimates for the project versus those used in the original optimisation.

Table 3-22: Comparison of Optimisation Inputs (All Currency in USD)

Parameter Description	Original Value	Revised Value	Difference %
Long term metal prices			
Cu (USD/lb)	2	2	0%
Au (USD/oz)	650	700	7.6%
Ag (USD/oz)	15	15	0%
Mo (USD/lb)	12	0	-
Capital (USD)	$175,000,000	$200,000,000	14%
Residual	10%	10%	0%
Discount rate	8%	8%	0%
Geotechnical design parameters			
Average - All domains	55 degrees	50 degrees	-9%
Mining parameters			
Bench height (m)	10	10	
Mining cost - waste ($/t)	$1.50/t mined	$1.65/t mined	10%
Mining cost - mill feed ($/t)	$1.50/t mined	$1.40/t mined	-7%
Mining dilution (%)	5%	5%	0%
Mining recovery (%)	95%	95%	0%
Concentrator parameters			
Expected mill throughput (t/annum)	5.5Mtpa	5.1Mtpa	-7%
Variable processing cost (USD/t milled)	$6.00/t	$4.30/t	-28%
Concentrator Recoveries			
Au	84%	84%	0%
Ag	61%	61%	0%
Cu	87%	87%	0%
Mo	46%	0%	-
Smelting and refining			
Transport + downstream proc. costs	$0/t Milled	$1.35/t Milled	-
Fixed overheads			
Admin and overhead unit cost (USD/t milled)	$1.00/t milled	$0.98/t milled	0%

The figures highlighted in red have been calculated from the more detailed study results contained within this Preliminary Assessment. LAL has completed a revised optimisation utilising these adjusted Case B revenue and cost inputs.

The Case B optimisation parameters were updated in the Whittle optimisation model and the pit optimisation process was re-run. The results are presented as follows.

Case B Whittle Results for PEA Scenario

In an attempt to present a comparable Whittle model result to the most optimal Case B scenario (based on maximising a Case A project return (IRR)) a pit selection and schedule scenario have been generated in Whittle using the Case B parameters to calculate the financial performance of this scenario. A scenario was generated selecting matching initial and final pits to those designed for this PEA and running a schedule in Whittle based on the Milawa balanced option and limiting mining to 12 mtpa (as per the PEA schedule). The IRR calculated by the Whittle model for Case B1 (see Table 3-23 below) was 11.8%, which is considered similar to the IRR demonstrated after the detailed cash flow for the Preliminary Assessment.

3.11.1 Case B Whittle Optimisation

The following scenarios were reviewed as part of the optimization:

- **Case B1: No mining Limit** – 1 to 4 pushbacks were reviewed
- **Case B2: 8mtpa mining limit** - Scenarios involving 2 to 4 pushbacks, various initial and final pits and various schedule types (e.g. maximising NPV or balancing resources) were reviewed.
- **Case B3: 12mtpa mining limit** - This scenario was completed to check the accuracy of the NPV and IRR calculations compared to what was determined through the more detailed pit design and cost estimation process. It involved selection of a production schedule representative of the pit shells selected in the original optimization. In addition to this original scenario comparison, 1 and 2 pushback scenarios were evaluated using different pit shells and with various schedules.

A summary of the results of this optimisation has been provided in Table 3-23.

Table 3-23: Case B Whittle Optimisation Summary

Case	Scenario	Mill feed Tonnes	Waste Tonnes	NPV	IRR	Mine Life	Pay back
B1	Preliminary Assessment Equivalent Pit Shells and Schedule using Case B	92Mt	90Mt	USD 55.7M	11.80%	20.8	6.23
B2	Case B Parameters, 8Mtpa mining limit	44.1Mt	27.1Mt	USD 58.9M	14.22%	10.5	5.2
B3	Case B Parameters, 12Mtpa mining limit	46.9Mt	30.5Mt	USD 81.9M	18.00%	9.77	4.14

Throughout the Case B optimisation process it was identified that the maximum return (IRR) operating scenario required mining at a total (mill feed and waste) production rate of 12mtpa throughout the first years of the mine life to reduce the stripping requirements at a later stage in the project and ensure continuous mill feed. If a 12Mtpa mining rate is not maintained in the early life of the project a continuous mill feed of 5mtpa will not be achievable.

The most optimum starter pit contained approximately 13.7Mt of mill feed with a final pit containing 33.2Mt for a total of 46.9Mt of mill feed at a stripping ratio of 0.65.

LAL recommends an update of this PEA utilizing the maximum IRR scenario including a re-design of the initial and final pits, re-run the production schedule, and revise the equipment selection, capital and operating cost estimates, revise the project cash flow model and re-submit the Preliminary Assessment report inclusive of these adjustments prior to commencement of the next phase of engineering studies.

3.12 Conclusions and Recommendations

The author has completed a Preliminary Assessment for the Bronson Slope property based on a wide variety of data, observations and previous technical reports.

Reference is made to previous independent Technical Reports on the Bronson Slope Deposit filed on SEDAR that establishes a mineral resource estimate for the Bronson Slope Project. The Technical reports are as follows:

"Technical Report for Skyline Gold Corporation on the Bronson Slope Property North-western British Columbia, Canada", dated June 1, 2006, authored by A. A. Burgoyne, P.Eng, M.Sc, from Burgoyne Geological Inc., an independent Qualified Person as defined by NI 43-101. This Technical report was posted to SEDAR on June 21, 2006.

"Technical Report Mineral Resource Estimate – Bronson Slope Deposit for Skyline Gold Corporation Vancouver, BC on The Bronson Slope Property North-western British Columbia, Canada", dated May 10, 2007, authored by G. H Giroux, and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., both independent Qualified Persons as defined by NI 43-101. This Technical report was posted to SEDAR on May 29, 2007.

"Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical report can be viewed at www.sedar.com. The mineral resource estimate that forms the basis for this preliminary assessment is the one presented in this report using Case 2 metal prices. Please refer to Section 19 for more details

These technical reports have provided a technical review of the Bronson Slope property including a detailed review and evaluation of the historical resource estimations for the Bronson Slope Au-Cu-Ag-Mo deposit. The preparation of these technical reports included certain due diligence procedures. The authors of these reports concluded that the technical fieldwork, and office data compilation, including historical resource estimation procedures, diamond core drilling, analyses, and reporting of data, completed by SGC, is of good quality and meets good practice industry standards.

Completion of this Preliminary Assessment, using a Gold price of $700/t.oz, Copper price of $2/lb and a Silver price of $15/t.oz and optimising the mine plan for maximum NPV, has shown an IRR of 10.0% after tax is achievable based on an economic mine life extending 19 years. Further opportunity exists to enhance the project return by:

1. Reducing the mined resource as set out in Section 3.11 above;
2. Economic recovery of Molybdenum and/or Magnetite contained in the mineralization through further processing.

The following is a summary of the recommendations for the Project:

1. After completion of the post Preliminary Assessment Pit and conceptual LOM schedule optimisation using more detailed cost and revenue inputs it was identified that a smaller final pit may result in a higher IRR. LAL recommends that a Preliminary Assessment Update (PAU) is conducted based on the results identified in the Case B3 Whittle optimisation. A budget estimate for completion of this PAU has been provided in Table 3-24.
2. Study of the highwall gold zone recoveries with low copper grades
3. Development of a 43-101 compliant magnetite resource for the QM zone and further investigation of the magnetite and molybdenum recovery circuits impact on project economics
4. Complete a trade-off study of the economics of the conveyor system vs. an ore pass system. Detailed scheduling of the selected ore and waste delivery method will also be required
5. Complete a study of crusher sizing and also consider electrical vs. diesel crusher economics
6. Complete a study to identify the cost savings potential of using used mining and processing equipment now available to market

7. Complete a study on selective rehandling and processing of below cut-off grade material from the waste storage area at the end of mine life for further processing and economic benefit
8. Complete further studies comparing grinding size versus recovery to identify the optimum grind size.
9. Further develop self-generation hydro projects as the basis of power supply allowing Feasibility Study undertaking
10. Complete Acid Rock Drainage testing on composite samples from within the pit limits to identify the acid producing potential of the various rock types within the pit. Quantities of ARD material should be determined so that appropriate waste storage management can be considered for the project.
11. Complete further geotechnical study of the highwall slope including higher pit wall angle and potential to improve project economics
12. Complete a revised Tailings Storage Facility design and cost estimate.

Table 3-24: Estimate for Preliminary Assessment Update

Item	Days		
	Senior Mining Engineer	Mining Estimator	Mining Engineer
Complete pit designs from whittle shells	3	0	0
Complete mine schedule	1	0	3
Complete basic mine site layout plan	1	0	1
Update infrastructure estimate to reflect Dennis review	1	0	0
Update mining and processing cost estimates	0	1	1
Financial modeling and sensitivity analysis	0	0.5	2
Compile technical report	2	0	2
Project management and review	2.5	0	1
Total time in days	**10.5**	**1.5**	**10**
Rate (USD /day)	$1,450	$1,450	$1,150
Total price for the job by resource (USD)	$15,225	$2,175	$11,500
Total Estimated Job Cost (USD)			**$ 28,900**

Also as part of the updated Preliminary Assessment a gap analysis should be conducted to identify what further studies and investigations are required prior to the completion of a feasibility study. These studies should be presented within the recommendations of the PAU.

More detailed interpretations and conclusions for the Bronson Slope property have been provided in Sections within the body of this report. It is the author's intention the report will be read in full to ensure full comprehension of all relevant interpretations and conclusions.

4 Introduction

Leighton Asia Limited was engaged in July 2007 by Skyline Gold Corporation (SGC) to conduct a Preliminary Assessment for their Bronson Slope property, which is located in North West British Columbia, Canada. The Preliminary Assessment is intended to investigate conceptual plans and costing for the development of infrastructure, and the mining and processing of the resource considered to be economical. This Technical Report has been compiled to disclose the findings of the Preliminary Assessment. The Technical Report is preliminary in nature, and **it includes references to inferred mineral resources** as defined by CIMM resource standards and classifications. However Inferred resources have not been given any economic value in the financial model and performance calculations later detailed in Item 25.9. Inferred resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Measured or Indicated Resources or as Mineral Reserves.

Reference is made to previous independent Technical Reports on the Bronson Slope Deposit filed on SEDAR that establishes a mineral resource estimate for the Bronson Slope Project. The Technical reports are as follows:

“Technical Report for Skyline Gold Corporation on the Bronson Slope Property North-western British Columbia, Canada”, dated June 1, 2006, authored by A. A. Burgoyne, P.Eng, M.Sc, from Burgoyne Geological Inc., an independent Qualified Person as defined by NI 43-101. This Technical report was posted to SEDAR on June 21, 2006.

“Technical Report Mineral Resource Estimate – Bronson Slope Deposit for Skyline Gold Corporation Vancouver, BC on The Bronson Slope Property North-western British Columbia, Canada”, dated May 10, 2007, authored by G. H Giroux, and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., both independent Qualified Persons as defined by NI 43-101. This Technical report was posted to SEDAR on May 29, 2007.

“Mineral Resource Estimate – Bronson Slope Deposit”, dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical report can be viewed at www.sedar.com.

Please refer to Item 21 for a comprehensive listing of reports and other written sources, sections, maps, charts and other diagrams utilised to complete the Preliminary Assessment for the Bronson Slope property.

A personal inspection of the property was conducted between the 21st and the 22nd of July, 2007. The inspection was completed by Mr. Julien Lawrence, Senior Mining Engineer for Leighton Asia Limited and Mr. Victor Seen, Process Engineer for Leighton Asia Limited. The personal inspection was hosted by Mr. Sandy Martin, member of the Board of Directors for SGC. The inspection involved travelling to site via Terrace and Bob Quinn, inspecting the current drilling locations and program, reviewing some drill core that was immediately available on site, carrying out basic air surveys of the access route from Forest Kerr to the Bronson airstrip, reviewing possible mill site locations, tailings pond locations, mine access roads and waste storage potential and discussing the current exploration program with the Bronson slope site supervisor. The inspection was completed over a period of approximately 24 hours.

Further discussions were held with Mr. Sandy Martin, Mr, Cliff Grandison, Mr. Jeff Smulders members of the board of directors for SGC and Mr David Jensen, P. Eng., President SGC. Discussions regarding previous technical reports and geology and mineralogy were held with Mr. A. A. Burgoyne of Burgoyne Geological Inc.. Discussions regarding metallurgical testing were held with Mr. Bern Klein from the University of British Columbia. A review of this Technical

Report has also been completed by a qualified P. Eng engaged by SGC with significant experience in the Canadian mining industry and in particular in BC.

The comprehensive consultations held with SGC personnel and their representatives based both in Vancouver and on site and discussions with external consultants associated with the Bronson Slope property along with other sources identified in Item 21 form the basis for the Preliminary Assessment.

5 Reliance on Other Experts

A Preliminary Assessment is “preliminary” in nature. It is a study that includes an economic analysis of the potential viability of mineral resources taken at an early stage of the project prior to the completion of a preliminary feasibility study. The study is used to help company directors and investors identify whether the project is economically robust enough to support a pre-feasibility or feasibility level study. This Preliminary Assessment is based on **Measured and Indicated Resources only** however inferred resources have also been identified and included in the mineral resource summary provided in Item 19.

The following section has been taken from the CIM Definition Standards on Mineral Resources and Mineral Reserves prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council, November 14, 2004.

An ‘Inferred Mineral Resource’ is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

Due to the uncertainty which may attach to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

A Pre-Feasibility or Feasibility level study uses more comprehensive information and is a more accurate measure of a projects economic viability than a Preliminary Assessment.

All costs included in this report are in Canadian dollars herein, unless otherwise stated. Costs have been identified from a number of different sources included in, but not limited to Item 23.

Due to risk factors including, but not limited to, metal prices, permitting, metallurgical recoveries, mineral resources, and capital and operating costs, there can be no certainty that any of the assumptions contained in this preliminary assessment will be realised or that the economic results projected herein will be achieved.

Information on the items set out in Items 6 to 9 of this Report can be found in the previous independent Technical Report titled “Technical Report Mineral Resource Estimate – Bronson Slope Deposit for Skyline Gold Corporation Vancouver, BC on The Bronson Slope Property North-Western British Columbia, Canada”, dated May 10, 2007, authored by G. H Giroux, and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., both independent Qualified Persons as defined by NI 43-101. This Technical Report was posted to SEDAR on May 29, 2007 (www.sedar.com).

Information on the items set out in Item 4 and Item 10 to 15 “Mineral Resource Estimate – Bronson Slope Deposit”, dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical Report can be viewed at www.sedar.com.

To LAL’s knowledge, there has not been any material change in the information since that date unless otherwise stated in the appropriate Item of this report.

Marketing information such as concentrate land and sea transport costs and port and QA charges associated with marine surveying and cargo sampling have been obtained from the report titled "Bronson Slope Project – Revised Marketing Costs (Transportation) authored by J Arthur Ganshorn (P. Eng, Ret.) and dated March 12, 2007. This report is based on a series of quotes from marine and transport service providers.

Site access development information has been obtained from the report titled "Skyline Gold Bronson Creek Access Trail and Bridges, Temporary Access Trail Proposal and Cost Estimate" prepared by Tim Dunne (P. Eng) and Michael Foster (P. Eng) of Forsite Consultants Ltd and dated December 15, 2006. An updated cost estimate for the road construction has been provided by Tahltan - Turcon LP in May 2008.

Guidance on all aspects of the tailings dam design and cost estimate have been obtained from the report titled "Bronson Slope Mine Conceptual Design of Tailings Facility", dated January 1997, prepared by Piteau Engineering Ltd for International Skyline Gold Corporation. A detailed review of this information has not been completed but is recommended prior to completion of the project feasibility study.

Guidance on the pit wall geotechnical design constraints has been obtained from the report titled "Bronson Slope Project Preliminary Geotechnical Assessments and Slope Design Studies for the Bronson Slope Open Pit", dated March 1997, prepared by Piteau Associates Vancouver.

The results and opinions expressed in this report are based on LAL's field observations and the geological and technical data listed in the Appendices. While LAL has carefully reviewed all of the information provided by SGC, and their consultants, and believes the information to be reliable, LAL has not conducted an independent in-depth investigation to verify its accuracy and completeness.

6 Property Description and Location

The following section has been extracted from the previous Technical Report titled “Mineral Resource Estimate – Bronson Slope Deposit”, dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical Report can be viewed at www.sedar.com.

6.1 Bronson Slope Mineral Claims & Crown Grants

The Property is located in northwestern British Columbia. It is centred on 131°05’ West Longitude and 56°40’ North Latitude on National Topographic Series map sheet 104B 11/E (also BC Trim Map104B 065). The Property is 110 km northwest of Stewart, B.C., 280 km northwest of Terrace, B.C., 80 km east of Wrangell, Alaska and 70 km west of Bob Quinn airstrip on the Stewart-Cassiar Highway. A mine access road leads from Bob Quinn 40 km down the south side of Iskut River to within 30 km of Bronson Slope where it turns south to the Eskay Creek gold-silver mine of Barrick Gold. Note Figure 6-1 and Figure 6-2.

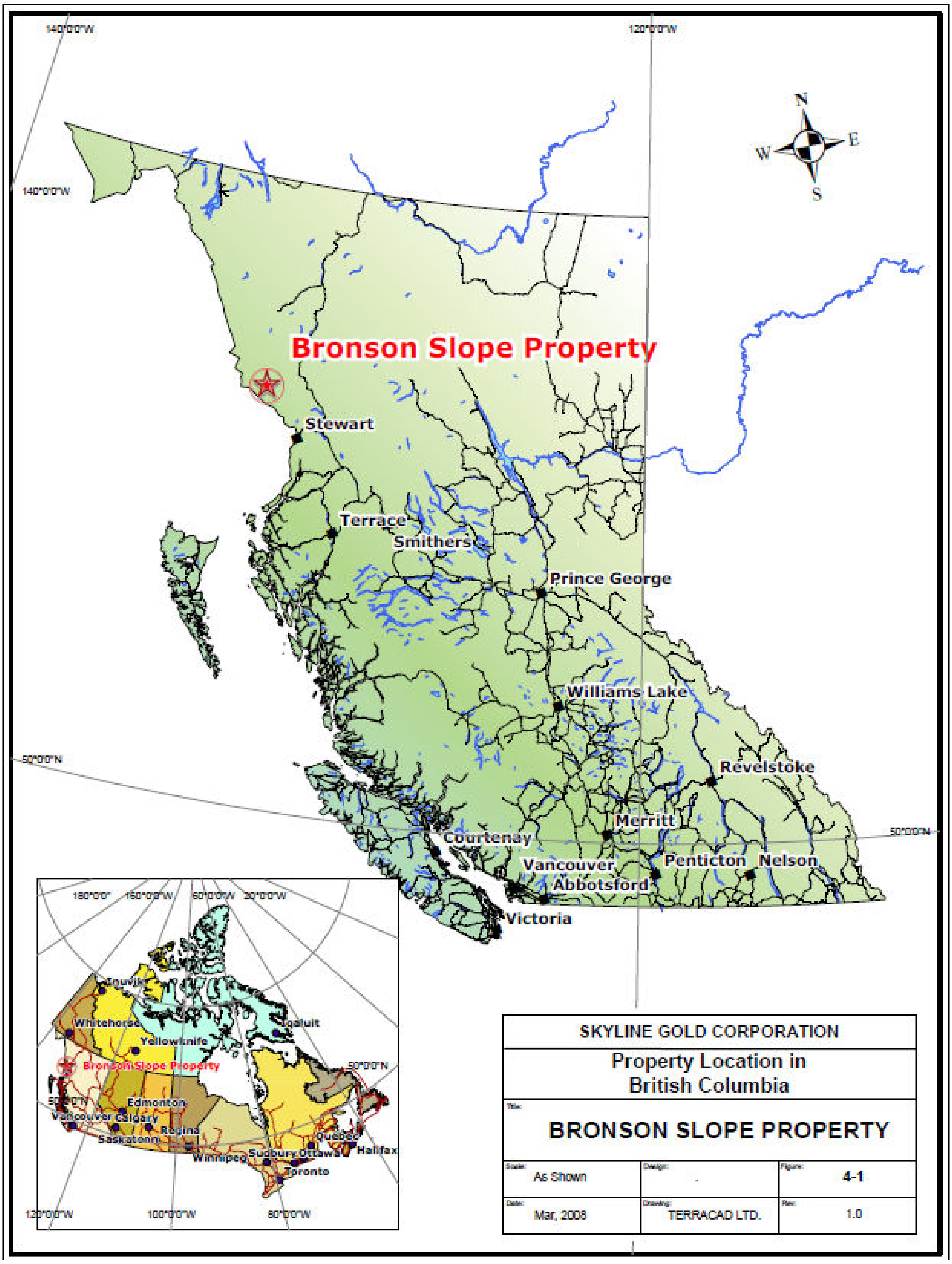


Figure 6-1: Bronson Slope Location

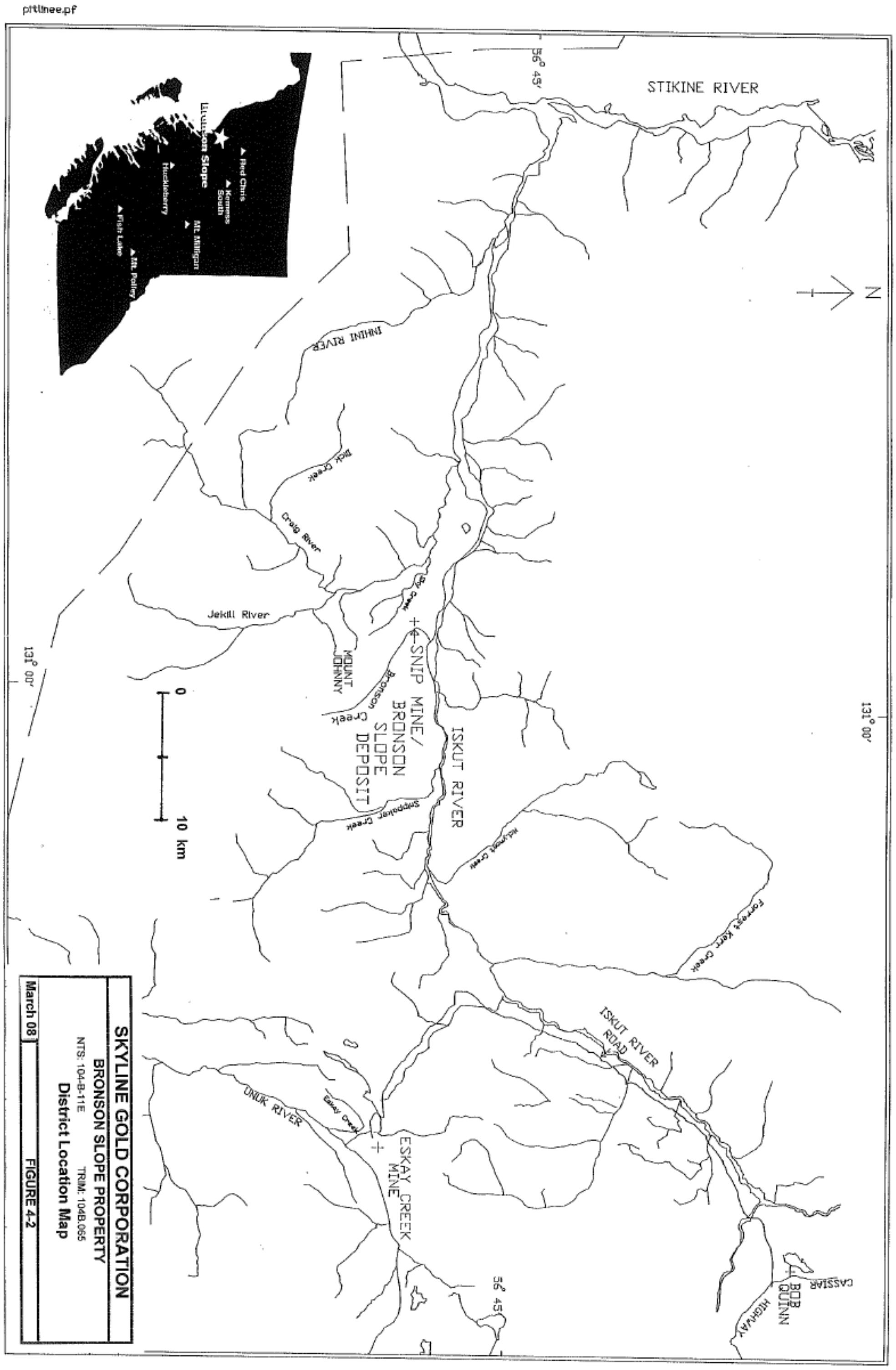


Figure 6-2: Bronson Slope Location

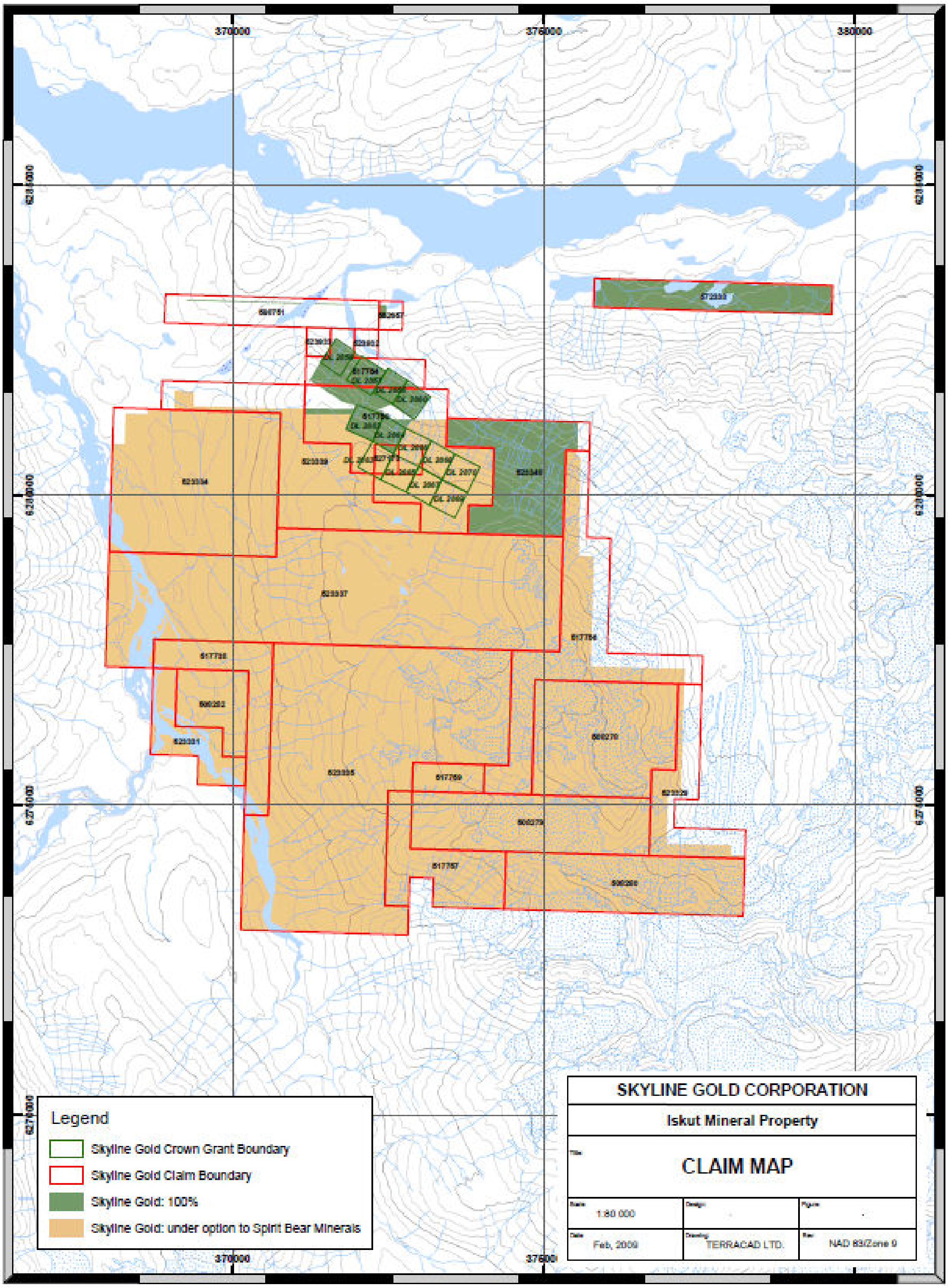


Figure 6-3: Property Boundaries

The property consists of BC Mineral Claim Tenures 517750, 517754, 523932, and 523933, and 6 Crown Granted Mineral Claims, totalling approximately 186.9 hectares located in the Liard Mining Division, owned 100% by SGC. The claims are located in NTS 104B 11/E. The Crown Granted claims portion of the Property has been legally surveyed. The known Bronson Slope Au-Cu-Ag-Mo deposit with respect to property boundaries is presented in Figure 6-3.

The Bronson Slope Property is located on the north side of the large 4400 hectare Iskut Property (Richards 2005) owned by SGC. SGC in April 2006 (SGC 2006) closed a "farm-out" of their Iskut River Property to Spirit Bear Minerals Ltd.

The Bronson Slope property mineral claim tenure and Crown Granted Mineral Claims names along with claim numbers; expiry date and size are set out in Table 6-1. All of the mineral tenures have been staked and registered with MTO (Mineral Titles Online) for the province of BC. These are electronic claims based on coordinates for the cells in UTM NAD 83 format.

Table 6-1: Bronson Slope Property Mineral Tenure

Claim Name	Tenure Number	Expiry Date
Bronson 2	517754	December 31, 2010
Bronson	517750	December 31, 2015
Katyadd	523932	December 31, 2010
Cgadd	523933	December 31, 2010
Snip1	523348	March 1, 2010
Crown Grants	**Lot Number**	**Taxes Due Date**
Red Bluff	2857	July 2, 2009
Homestake	2858	July 2, 2009
Red Bird	2859	July 2, 2009
Mermaid	2860	July 2, 2009
El Oro	2862	July 2, 2009
Golden Pheasant	2864	July 2, 2009

All proposed exploration work in the Province of British Columbia must receive prior approval by issuance of a work permit by the Ministry of Energy, Mines, Petroleum and Resources (EMPR). Such approval is routinely given and will be obtained with no difficulty in the areas to be explored subject to normal reclamation and environmental guidelines. The Mines Permit MX-1-707, approval number SMI 06-0100129-0707 was issued by the EMPR to complete mineral exploration in 2007. Under the terms of the agreement the Mines Permit, SGC is responsible for all remediation and reclamation work resulting from the 2007 drilling program where trees were cut in order to construct drill pads. It is the writers understanding that certain reclamation work will be undertaken in 2008 to meet the terms of the Mine Permit. A budget has been provided in Item 20.

In 1996 SGC entered into an agreement with Prime Resources Group Inc. to acquire Prime's claims immediately between the SGC Crown Grants Red Bird, Red Bluff, and Homestake on the north and El Oro on the south. This transaction was achieved in two stages – a claim swap for the Kathleen Fraction between Red Bird and Red Bluff Crown Grants and a purchase agreement for the Highwall claims located south of the Red Bird, Red bluff and Homestake Crown Grants. In return SGC (Yeager 2006) granted a 3.5% Net Smelter Return payable to Cominco/Prime from any production obtained on the Highwall claims only. This NSR interest is purchasable by

SGC for $500,000 (Yeager 2006). Mineral Tenure 517754 now covers the Highwall and Kathleen claims.

Cost of holding title to ground held by mineral cell claims for the first three years after registration is $4.00/hectare of exploration work plus a $0.40/hectare fee; in subsequent years the cost is $8.00 per hectare plus a fee. Crown granted mineral claims are assessed for taxes on May 1 of every year with notices sent to registered owners in May and taxes due July 2. The 2005 tax assessment rate was $1.25 per hectare. The taxes are due by July 2, 2008. All proposed exploration work in the Province of British Columbia must receive prior approval by issuance of a work permit by the EMPR. Such approval is routinely given and will be obtained with no difficulty in the areas to be explored subject to normal reclamation and environmental guidelines.

6.2 Environmental Issues

The authors are not aware of any environmental issues or liabilities that affect the property and has been informed by SGC that they are not aware of any environmental problems. At the present time, the infrastructure development at the Bronson Slope property is limited to that adjacent, but not on the Property, airstrip and buildings at the Bronson airstrip, the Johnny Mountain airstrip and a network of tote roads. There are several wooden drilling platforms on the side of Bronson Creek Valley some of which may be used as helicopter landing pads. Previous camps were located off of the Property.

There are rusty coloured seeps in the Bronson Creek valley, which are no doubt emanating from iron sulphide mineralization in the Bronson Slope deposit and these seeps are natural in origin.

7 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Reference is made to “Mineral Resource Estimate – Bronson Slope Deposit”, dated May 10, 2007 and posted to SEDAR on May 29, 2007. This report was prepared by A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc. and G. H. Giroux P.Eng., MASc. from Giroux Consultants Ltd..

7.1 Climate and Operating Season

Climate in the area is typical for this portion of British Columbia – cool summers and cold winters.

The nearest weather monitoring station was located at Bronson Creek and was in operation until 1999. Data recorded from 1994 to 1998 shows the annual precipitation ranged between 2100 and 1300 mm. Approximately 30% of all precipitation fell as snow. Precipitation levels were highest in September and October and lowest in May through August.

Mean daily temperatures were highest in July and August reaching approximately 16 ^{o}C, and lowest in January falling to -15 ^{o}C. The highest temperature recorded on site over the 5 year period was 31 ^{o}C and the lowest temperature recorded was -32 ^{o}C. A summary of the climate data recorded at the Bronson creek location is included below in Table 7-1.

Table 7-1: Bronson Creek Climate Data Summary

Year	Total Precipitation (mm)	Total Snow (mm eq.)	Mean T (°C)	Extreme Min T (°C)	Extreme Max T (°C)
1998	1300.5	212.7	4.7	-26	31
1997	1572.7	298.8	5.2	-32	28
1996	1378.1	446.6	3.2	-33	29
1995	1286.9	401.8	4.9	-31	29
1994	2110.5	799.8	4.5	-23	30

7.2 Land Availability

A potential mill site has been identified as being located on the disturbed land of the former Snip Gold Mine mill site, with surface lease currently held by Barrick Gold. If this surface tenure is not available alternative, less optimum, locations for the process plant are at the southern end of the airstrip as the full 1700m runway will not be required once road access is established, or alternatively adjacent to the TSF in between Cell A and B.

Figure 7-1 shows the proposed site layout for the Bronson Slope Project at the end of the Mine life.

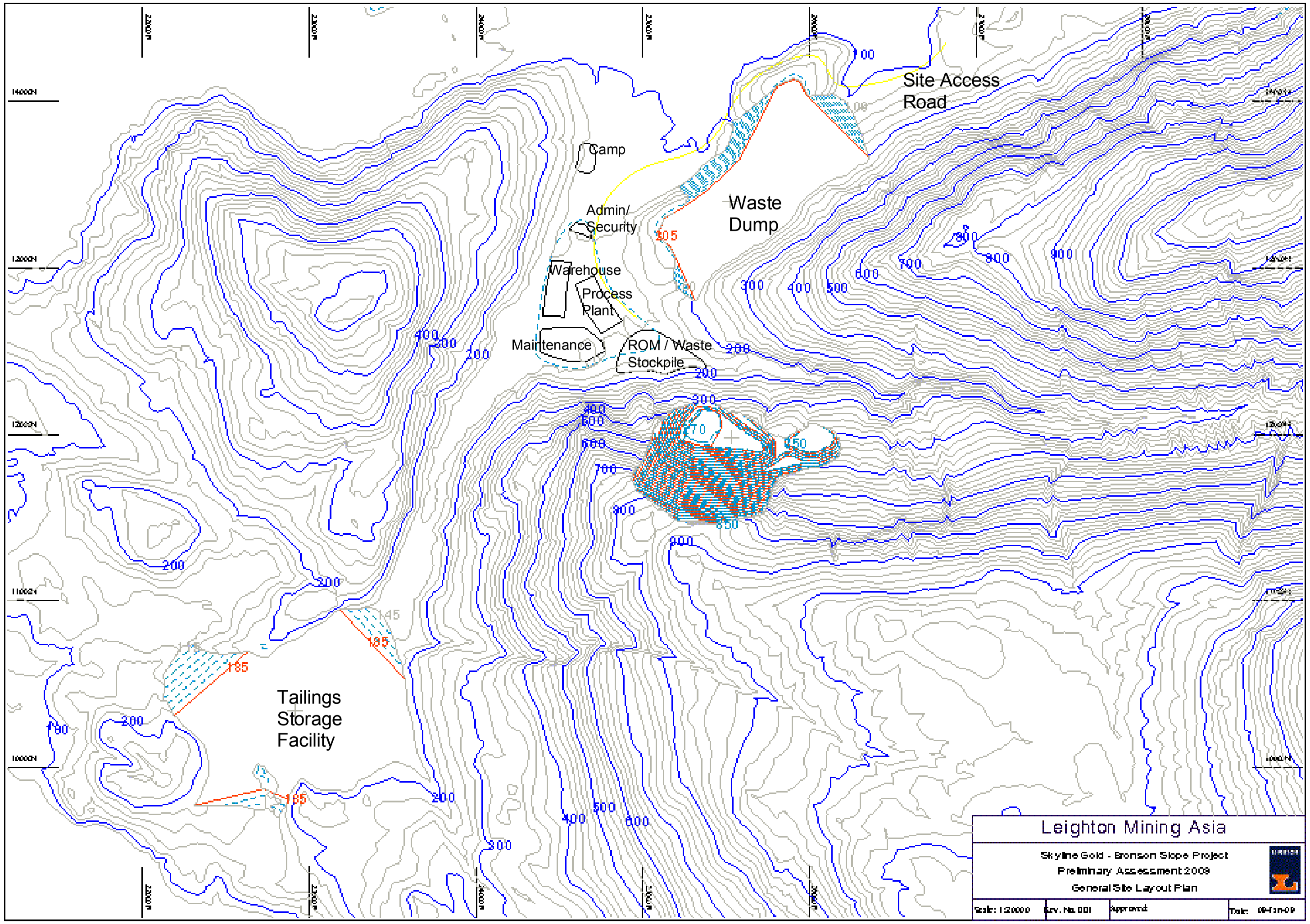


Figure 7-1: Site Layout Plan

7.3 Site Access and Transport Infrastructure

The Bronson Slope property is 110 km northwest of Stewart, B.C and 70 km west of Bob Quinn airstrip on the Stewart-Cassiar Highway. The existing 40 km of road access to site is comprised first of a Forest Service Road leading from Bob Quinn Lake to the Eskay Creek gold-silver mine turn-off and is under Road Use Permit to Barrick Gold. This 35 km segment is followed by a segment of approximately 5 km. long operated under a License of Occupation by AltaGas for its Forrest Kerr hydroelectric project. A mutual road sharing agreement will be required between SGC and Alta gas to maintain the road. The proposed access road location is shown in Figure 7-2. The road is a single lane “forest industry style” gravel road, shown in Figure 7-3.

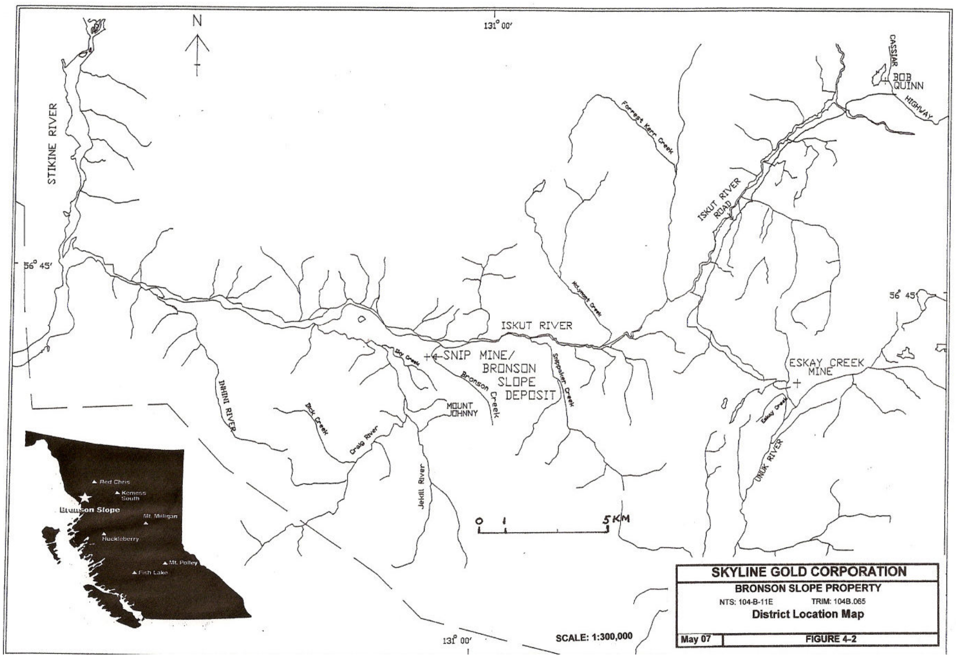


Figure 7-2: Site Location With Respect to the Iskut River

Figure 7-3: Forest Industry Style Road

From Bob Quinn, the route towards Port Stewart uses Highway 37 and 37A for 142 km and 67 km respectively. This majority of the route consists of paved/hard surfaces with only a small gravel section after the town of Bell II. There are no restrictions on hours for travelling the public road/highway and large trucks use the highway 24 hours a day. Road conditions vary depending on the weather and the time of year. Even in good weather some sections of the highway will have potholes and broken seal coat. Snow can occur at any time of the year (although not typically during the summer months) causing slippery conditions and poor visibility. Current information on road works and road conditions can be found on the Ministry of Transport for British Columbia website (http://www.drivebc.ca/).

A trip from Port Stewart to the Bronson Slope property is summarised in Table 7-2.

Table 7-2: Summary of Road Conditions from Port Stewart to Bronson Slope

From	To	Road	Length (km)	Surface
Port Stewart	Meziadin Junction	Highway 37A	67	Paved
Meziadin Junction	Bob Quinn	Highway 37	142	15% Gravel, 85% Paved
Bob Quinn	Eskay Creek mine road intersection	Iskut River Rd	40	Gravel
Eskay Creek mine road intersection	Bronson Slope mine	Proposed access rd	32	Gravel
		Total	**284**	**188km Paved 96km Gravel**
		Round Trip (x2)	**568**	**376km Paved 192km Gravel**

The transportation of non bulk commodity items (such as supplies, parts and equipment) to and from suppliers and the mine site will vary depending on the route, distance and other special requirements. Fuel will typically be delivered from Terrace or Smithers, BC, supplies from southern BC and lime from the rail head at Topley Landing.

7.3.1 Proposed Mine Access Road from Forest Kerr to Bronson Slope Property

The following is based on the report titled "Pre-Feasibility Study on Access Road Location and Cost Estimate" prepared by Forsite (Original Oct 2006 and cost estimate revised in 2008). This proposal is for a permanent mine access road for the Bronson Slope Property if the project was to go into construction and operation.

The proposed permanent road access to the project can be divided into two main sections. The first section is the relatively flat road on old lava beds. The second section starts at the end of the lava flats east of Bug Lake and continues to the Bronson Creek crossing and the airstrip.

7.3.1.1 Section 1 – Forest Kerr to Bug Lake

This section consists of approximately 24 km of road and four proposed bridge crossings. It starts at the west end of the existing built Forest Kerr power project access road and continues west to the end of the lava flats below the Bug Lake area. The proposed road includes a number of bridges over tributaries to the Iskut River. The smaller tributaries are called Jennifer Creek, Seth Creek, and Snippaker Creek. The road will be a single lane permanent "forest industry style" with road and bridge structures similar to those used on the Eskay Creek Mine access road with road grades generally expected to range from -5% to +5% with short sections over 10%. There is one section of more difficult road construction proposed where the road crosses through exposed bedrock but with moderate side slopes between the Seth and Jennifer Creek crossings. Drilling and blasting will be required for at least 50% of all cut volumes. This latter section is approximately 400m in length.

7.3.1.2 Section 2 – Bug Lake to Bronson Air Strip

The second section of proposed road starts at the end of the lava flats east of Bug Lake and continues to the Bronson Creek crossing and the airstrip. It is characterized by steeper side slopes and rock that will require drilling and blasting. It includes bridge crossings over Bronson

Creek and tributaries of the Bug, Middle, and Triangle Lakes. The proposal for this section includes four bridges; one approximately 65m crossing over Bronson Creek and three 15m crossings over Bug and Middle Lake tributaries. The maximum road grades are not expected to exceed 16%. Most areas of the route were found to be at between 6 and 10% grade.

From Bug Lake to Bronson Airstrip, the road is dominated by old volcanic origin bedrock. Some road sections on the lower slopes of Snippaker Mountain (south-east of Bug Lake and South of Triangle Lake) have soils that are a silty sand/gravel and imported gravel surfacing will be necessary. It is estimated that roughly 1/4 of the road will require imported surfacing. There are two possible options to pass by Bug Lake. One route has been proposed south of Bug Lake and another North. The route south would have a lower construction cost than the road location to the north but would require avalanche hazard management measures, possibly including winter road closures. As such, road maintenance costs would be much higher for the route south of Bug Lake. In addition to the higher road maintenance costs the planned position of the waste dump eliminates this route as an option for access to the Bronson Slope property.

The complete section 2 route can be broken into three general ground types.

- Gentle: Gentle side slopes with weathered rock and soils present.
- Moderate: Dominantly bedrock but with moderate side slopes. Drilling and blasting will be required for at least 50% of all cut volumes.
- Difficult: Extensive bedrock areas with little or no soils. In some cases, the bedrock may be in the form of irregularly shaped rolling gullies and ridges. Almost all material will need to be drilled and blasted prior to road construction.

7.3.1.3 Bronson Creek Crossing

The Bronson Creek is consistently wide and would require structures over 90m for a crossing. The location at the top of the alluvial fan at the Iskut River will require a 2 span structure with an approximate overall length of 60m. The bridge would be a 2 span structure on piles - 1 set of piles would be set in a gravel bar near mid-stream. A mid-winter or late summer installation would work best as stream flows are generally at lower levels during those periods.

The crossing is located on deep fluvial deposits. On the airstrip side (west), there is an existing gabion wall that helps to keep the river from eroding the airstrip. A more detailed estimate for the cost of this structure is required once the detailed site survey and general arrangement drawings are completed.

7.3.1.4 Access Road Construction and Cost Estimate

A number of cost estimates have been completed by Forsite Engineering and Geoscience and Tahltan – Tercon LP. The most recent and the one used for this Preliminary Assessment was completed for SGC in May 2008. The total construction cost is approximately $7.576M. The cost estimate includes supervision, survey, mechanical repair, fuel, travel, camp, office, safety and project support and is based on the contracting group working a 3 weeks on – 1 week off rotation. The expected time to complete the construction of the road excluding the bridges is 6 months. Details of the cost estimate are included in Table 7-3 below:

Table 7-3 - May 2008 Access Road Construction Cost Estimate

TAHLTAN - TERCON LP Skyline Gold - Bronson Creek Access Road 32Kms - Budget Forsite Engineering & Geoscience					
Item No.	Budget Unit Prices	QTY	UOM	Price per Unit CAD	Extended Price CAD
1	Clear & Grubbing				
	Clear, Grub and Dispose	64	ha	$3,500	$224,000
2	Stripping				
	Doze, Pile but no trimming	75,000	m3	$3	$240,000
3	Excavation to Embankment				
	Load, Haul Place including Rock	200,000	m3	$8	$1,600,000
	90,000 m3 - doze to place				
	90,000 m3 - load and haul				
	20,000 m3 - rock excavation				
4	Road Surfacing				
	25mm Crush @ .150 depth				
	75mm Crush @ .150 depth	28,800	m3	$21	$604,800
	SGSB Select @ .300 depth	31,200	m3	$19	$592,800
	Includes maximum 10 km haul distance	67,200	m3	$15	$1,008,000
5	Pull-out Construction				
	200m x 8m Turnouts, 2 per Km	12,800	Im	$10	$128,000
6	Drainage Applicances				
	600 mm CMP	1,400	Im	$150	$210,000
	800 mm CMP	150	Im	$275	$41,250
	1000 mm CMP	70	Im	$375	$26,250
	1200 mm CMP	50	Im	$480	$24,000
	1600 mm CMP	40	Im	$670	$26,800
	2200 mm CMP	20	Im	$1,420	$28,400
	2400 mm CMP	20	Im	$1,650	$33,000
7	Open Bottom Arches				
	1.8m - 3.0m	60	Im	$2,715	$162,900
8	Culvert Bedding and Backfill				
	Processed 25mm bedding and backfill	14,000	m3	$20	$280,000
9	Bridges				
	12 m Bridge - 1 each	12	Im	$10,000	$120,000
	15 m Bridge - 4 each	60	Im	$10,500	$630,000
	24 m Bridge - 1 each	24	Im	$11,000	$264,000
	35 m Bridge - 1 each	35	Im	$9,000	$315,000
	42 m Bridge - 1 each	42	Im	$9,500	$399,000
	65 m Bridge - 1 each	65	Im	$9,500	$617,500
	(4.8 m Width)				
	Budget Price Total				$7,575,700

7.3.1.5 Access Road Considerations and Recommendations

Depending on the vehicle type travelling on the road and the frequency of travel, the "forest industry style" design will have to consider certain factors to ensure safe and effective operation.

- Road alignment (both vertical and horizontal) – the ability of the vehicle operator to see ahead a distance equal to or greater than the stopping distance required, grade and brake relationships, maximum and sustained grades (vertical alignment), superelevation and rate/runout (horizontal alignment), curve and width design and areas where there is a combination of horizontal and vertical alignment factors.

- Construction materials – Selection of adequate sub-base and surface material weight for the estimated frequency of traffic
- Proper lane widths – sufficient room for manoeuvring for all planned equipment
- Cross slope – balance between adequate drainage (slope) and driver steerability (level)
- Drainage provisions – ditch configuration and location, ditch capacity and protection and culvert location, type, size, placement and inlet-outlet requirements
- Traffic control – Adequate signage for speed, stop, curves/intersections, culvert crossings, limited access, traffic control and safety access indicators.
- Road and vehicle maintenance – cost analysis and adequate maintenance planning
- Runaway vehicle safety provisions – Conventional vehicle arresting or impact attenuation devices to stop runaways including the design of conventional parallel berms, collision berms and escape lanes (entrance, deceleration and stopping requirements)
- Erosion and avalanche safety – routine maintenance and checks, planned seasonal closures and adequate safety measures (such as signage, drainage, protection berms, etc.)

Costs associated with road construction to remedy safety hazards can be considerable and a beneficial cost to profit ratio must be maintained. It is important to ensure that cost efficiency take into account less tangible aspects such as operator safety and proper equipment utilization. Time and resources spent early in the project in long term planning can determine current and future use of the road, and in turn, result in capital savings in road reconstruction and design during the LOM.

7.3.2 Transport of Bulk Commodities Study

The following information has been obtained from the Revised Marketing Costs (Transportation) submitted on March 12 2007 by J. Arthur Ganshorn (P. Eng. Ret). The transportation costs for copper concentrate are summarized as follows:

7.3.2.1 Copper Concentrate Transportation

Based on 30,000 WMT/yr, 40 WMT per truck load (B-Trailer) and exporting 3,000 to 5,000 WMT per shipment (100WMT lots), the trucking costs are CAD40.00/t to Port Stewart.

The shipping charges and sea freight costs are summarised in Table 7-4 and Table 7-5.

Table 7-4: Ship Loading and Marine Services Charges

	3000WMT/shipment	5000WMT/shipment
Storage and Ship loading costs	CAD 15.00/WMT	CAD 15.00/WMT
Marine Services	CAD 4.24/WMT	CAD 3.49/WMT
Total Canadian Cost	**CAD 59.24**	**CAD 58.49**

Table 7-5: Ocean Freight from Port Stewart

	3000WMT/shipment	5000WMT/shipment
Korea	USD 70-75/WMT	USD 55-60/WMT
Japan	USD 70-75/WMT	USD 55-60/WMT

7.4 Power

Confirmation of the line voltage, capacity and substations will be developed as part of further more detailed studies. However this study assumes that power will be supplied by BC Hydro, a provincial crown corporation reporting to the minister of energy and mines and regulated by the British Columbian Utilities Commission delivering electricity throughout British Columbia. Power will be supplied under their industrial tariff. The tariff is estimated to be $0.055 / kWh.

It is assumed that power will be supplied from a proposed Northern Transmission Line main grid line located at Bob Quinn Lake (approximately 60km from site). An opportunity exists for a direct connection to the BC Hydro grid near the proposed Forrest Kerr hydro power station, which is much closer to site (approximately 25km from site). Other alternative electricity generation and supply options are also being evaluated including self-generation of power using hydro assets for which SGC have submitted hydro generation license applications and for which SGC has received Notice of Sufficiency of Application from the BC government. Power will be supplied using a 138kV transmission line, which will run partially within the Access Road right of way. Up to 20MW of power will be provided from the Forest Kerr run-of-river hydroelectric power station, located 25km west of the Bronson Slope property.

Initially the BC Government had plans to extend the Northwest transmission line from Terrace to Bob Quinn. This was going to be completed in two stages with the first completed in 2009 providing 138kV and the second stage completed in 2011 increasing the line voltage to 287kV. However a delay has been announced by the EMPR due to the Galore Creek Project ceasing construction. In November 2008, the BC government announced that the environmental assessment process for the NTL was being restarted. SGC are currently investigating alternative sources of power supply. The options include obtaining power from Alaska where there is adequate capacity and relatively low infrastructure requirements or local generation of electricity from coal, coal gas or hydro.

In addition to these options SGC has announced completion of the first stage of three water license applications on 7 water catchments. Preliminary indications are that hydro electric electricity, if augmented by water storage on Snippaker and Bronson Creek, will supply 25 MW or power required by the Bronson Slope Project.

7.5 Water

7.5.1 Site Water Supply

Allowances for the construction of water bores, genset pumps, pipelines (laid and buried), standpipes and storage facilities such as sumps, turkey's nests or above ground tanks are included in infrastructure costing within Item 25. The tanks will be established at a higher elevation. Chlorination will also be provided for the potable water.

It will be necessary to calculate rates of water consumption before designing pumping and piping requirements. Based on previous experience it is estimated a mine worker will consume approximately 0.19m^3 of potable water per day. Potable water treatment units will comprise of 1-micron and 10-micron cartridge filters, UV disinfection unit, a hypochlorite addition systems, raw water tank, small mix tank, metering pumps and booster pumps. The design process will minimize the requirement of freshwater and maximize the recycling of water. Flow metres will be installed to monitor fresh water consumption.

7.5.2 Sewage/Waste Water Treatment

A sewage treatment plant is included in the mine infrastructure cost estimate. Non-process waste water from some of the site facilities, such as the camp and offices, will be treated in this plant. The treated water will be tested for compliance to water quality parameters before being released to the environment. Sludge material produced by the sewage treatment plant will be stored in an engineered facility within the mine concession.

7.6 Mining Personnel

7.6.1 Labour Market in British Columbia

The labour force statistics for February 2006 indicated that BC was approaching the 4% natural rate of unemployment which is defined as the point where the labour market is in balance: not facing any pressures either from a lack of workers or from excess supply of people looking for work. Recent statistics have shown an increase in the unemployment rate in BC, mostly due to the current financial crisis. Long term average unemployment rates in BC range between 4 and 8%.

In the mining industry there is a general lack of skilled personnel. Direct employment in BC's mining industry totalled 7,345 during 2006, compared to 7,071 in 2005. Salary and benefits totalled $734 million in 2006, an increase from $661 million reported in 2005 which reflects the increase in the average salary and benefits per employee from $93,600 in 2005 to $99,900 in 2006. Mining salaries and benefits remain high, reflecting the current demand for (and shortage of) skilled personnel.

Provided in Table 7-6 below is data on the current labour costs and benefits in a BC mine site similar to the proposed Bronson Slope project. The data is based on a case study copper-molybdenum surface mine with 7,000,000 tonnes of mill feed mined per year. The mine during operation consists of 241 employees - 46 managerial/administrative, 168 hourly and 27 contractor employees.

Table 7-6: Hourly Wage Base for Mining and Maintenance Personnel

Job Classification	Hourly Wage Base CAD	Job Classification	Hourly Wage Base CAD
Mine Department		**Maintenance Department**	
Shovel Operator	29.85	Journeyman Electrician	31.11
Blaster	28.50	Journeyman Heavy Duty Mechanic	31.11
Driller	28.50	Journeyman Millwright	31.11
Equipment Operator	27.27	Journeyman Pipe fitter	31.11
Haul Truck Driver	26.10	Journeyman Welder	31.11
		Journeyman Gas Mechanic	31.11
Mill Department		Journeyman Plumber	31.11
Mill Operator 1	29.85	Crane Operator	30.20
Mill Operator 2	28.50	Apprentice 4	28.50
Mill Operator 3	26.10	Warehouseman	27.17
Serviceman	26.10	Apprentice 3	26.10
Mill Operator 4	24.98	Serviceman	26.10
Sample Bucker	24.98	Apprentice 2	23.90
Mill Operator 5	21.88	Apprentice 1	21.88
Labourer	20.05	Labourer	20.05

Nb. Wages increased 3.0% in the 12 months to January 2008. Rates provided by Informine

For the purposes of this study a total on cost inclusive of all accommodation, transport, benefits and burdens for each employee is considered to be 42% of their base salary (based on 2 weeks on 2 weeks off rotating roster). This is based on market research that has been conducted as part of this Preliminary Assessment.

7.6.2 Availability of Labour and Training

As at the end of 2007, there are metal, coal and industrial mineral operations widely distributed over the BC region. Mining has fuelled economic development in BC and is well developed in this region. Thus, the training and labour requirements for the Bronson Slope project can be sourced from within the region. The current shortage of skilled personnel in the industry however may bring significant escalation of labour costs in the near future.

7.7 Tailings Storage

7.7.1 Tailings Storage Location and Capacity Requirements

An initial conceptual design of the tailings storage facility was completed by PITEAU Engineering Ltd in 1997 which was published in the report titled “Bronson Slope Mine Conceptual Design of Tailings Facility” dated January 1997.

The proposed tailing storage facility is located at the Sky Creek valley, which is approximately 2.5km over relatively flat ground southwest of the plant site. The tailings area and the water diversion structure diagram are illustrated in Figure 7-4.

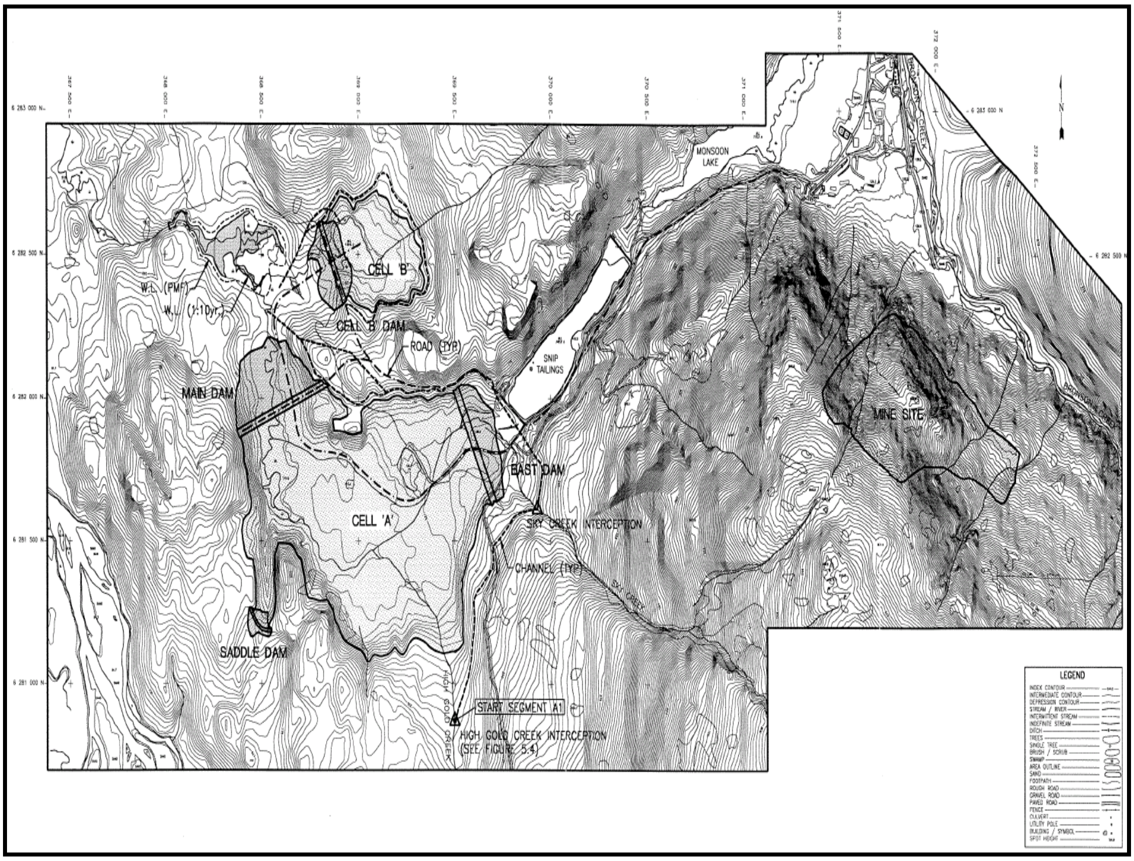


Figure 7-4: Proposed Tailing Storage Facility Area

The process results in generating two types of tailings referred to as “Rougher tailings” and “Cleaner tailings”. The main valley design Cell A has an area of one square km, located south west of the Snip tailings pond. Rougher tailings can be discharged to Cell A, which contains three embankments; Main, East and Saddle. The Rougher tailings contain only trace amounts of pyrite and chalcopyrite, are non-acid generating and constitutes 95% of the waste solids.

Cell B is a small 130 m x 275 m depression (called Boundary Lake) located north of main valley and is proposed as the site for storing the cleaner tailings. Cleaner tailings have acid generating potential caused by high content of pyrite and other sulphide material. The two tailings are kept separately. It is anticipated that water reclaimed from both tailings facilities will be suitable for the plant.

Approximately 51.7Mm^3 of tailings is expected to be produced during the life of mine from the process plant at a mill throughput of 15,000tpd. A single line is proposed for each of the rougher and cleaner tailings with associated water recycle line for each.

7.7.2 Original Tailings Dam Design Concept

Dam design for the main dam includes a starter dike. It consists of compacted impervious clay surrounded by compacted random fill. The starter dike is raised to an elevation of 132m above sea level to allow sufficient storage for two years of operation. Disposal of waste rock to Cell A during the first few years of the operation may be desirable. After 2 years the dike will be raised by tailings sand captured in cells. The dam was originally designed to a final elevation of 165m (above sea level) with raises by this method, leaving a downstream slope of 4:1. Seepage from the drainage system is collected downstream of the dam, in a lined collection cell. The dam height will be maintained to contain a 1 in 200 year flood event. In the event of a 1 in 200 year flood occurrence, both tailing and recycle lines can be used to move water from the tailing pond. Figure 7-5 shows a cross section of the Main Dam.

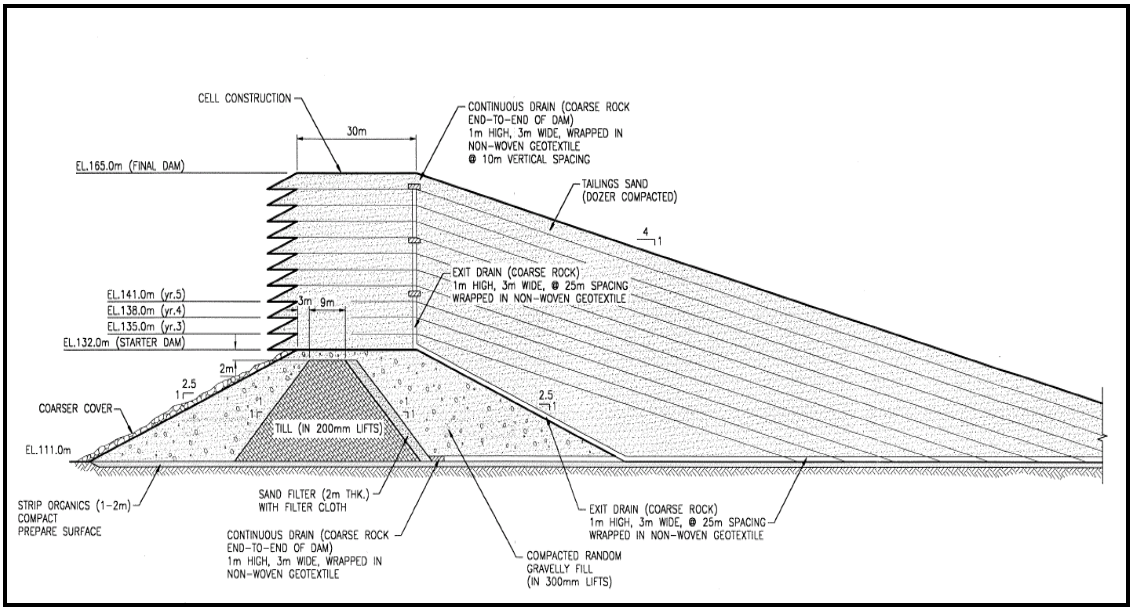


Figure 7-5: A Cross Section of the Main Dam (Piteau 1997)

The East Dam embankment is located west of the existing Snip Mine tailings pond. The location of this embankment should be reconsidered if surface tenure on the Snip Tailings can be obtained from Barrick Gold, since a more suitable location for the dam would be further east where the valley narrows. Additional storage capacity can also be achieved by moving this embankment further to the east. A retaining dike will not be required later as the elevation of the ground surface at the East Dam embankment is 20m higher than the Main Dam embankment. The eastern dam embankment will be constructed using tailings sand via the cell construction methods. Seepage from the dam will be collected in a lined cell and returned to the reservoir. The height of the East Dam embankment will be maintained at the same level as the Main Dam.

Saddle dam construction will not be required in the initial years. It is constructed of rock fill. The dam will be constructed in stages to act as an emergency spillway. Advance exploration will be required to appraise the availability of rock fill at this location. A rip rap protective face is required because the face of this dam will not be protected by beached tailings.

Cell B dam will be constructed from mined (waste) material and built before the Main Dam starter dike. Boundary Lake will be drained and weak material at the bottom of the lake will be removed. An impervious liner will then be placed on the floor of the reservoir to contain the acid generating cleaner tailings. The construction will be divided into 2 stages; the first stage will be constructed to an elevation of 133m and the second stage to an elevation of 145m. Seepage from the dam is collected in a lined cell and pumped back into Cell B.

A monitoring program shall be established during the early stages of the tailings facility to determine the tailings behaviour for the final design of the tailings management facility. Secondly, it can monitor safe performance of the fluid retaining structures while also confirming satisfactory performance of reclamation activities.

Surface water management will be designed to minimize the volume of water runoff entering the tailings impoundments and to preserve the normal flow regime of Sky Creek downstream of the impoundments. Natural runoff is diverted around the tailings ponds. Interception facilities capture runoff and route it to the lower reaches of Sky Creek. Most of the flow of the diverted runoff is in open ditches but a large pipeline is used to convey the water across the valley. The interception facility has runoff capacities up to a 1:100 year flood event. Larger floods of up to 1:200 year event are directed into the pond which has the capacity to store them.

Storage for the fine grained waste stream is provided in a small valley north of the main facility. An earth filled dam provides a small reservoir that is lined to prevent leakage. The fine grained stream is delivered to this site. Surplus water is recovered and returned to the plant. At the end of operation, the tailings will be covered with an impervious cover. A high water level will also be maintained in the deposit to limit oxygen access to a potential acid generating sulphidic material.

The development relies on observing and learning from operations during the start-up years. Observed behaviour will be used to confirm assumptions about tailings behaviour and to finalize the design. However before commencing construction, assumptions will have to be confirmed by

1. Validation of the design adequacy given present day standards and guidelines,
2. Data from pilot operations,
3. Exploration to confirm site condition and the quantity and quality of construction materials,
4. Information gained from observing material behaviour during the early years of the operation.

Site survey and permit approvals for water discharges to Bronson creek or Sky creek shall be sought while the design crest elevation should also consider the tailings pond water and material balances for dry, normal and wet-year scenarios. The facility has been designed with reclamation in mind. Reclamation procedures will be developed as operations proceed.

7.7.3 Adjustments Required for Capacity

The tailings facility design for the Bronson slope has been based on the 1997 mine plan. This mine plan considered a mill process rate of 12ktpd over a mine life of 14 years. The revised mine plan based on the most recent resource and optimisation is based on 15ktpd and a 20 year mine life. The total tailings storage requirement is estimated at 80.7Mt. A conceptual extension to the previously proposed tailings facility has been considered as part of this assessment to ensure sufficient capacity to store the complete tailings produced over the 20 year mine life. It is estimated that the east and main embankments will need to be raised a further 20m to a final elevation of 185m (above sea level) to allow sufficient capacity. It is important to note that the revised designs are conceptual only and have been completed on a conceptual basis only to ensure sufficient storage capacity is available for the tailings that will be produced from the mine plan. Further work must be conducted by a qualified tailings storage design specialist to ensure the proposed adjustments to the 1997 design by Piteau Engineering Ltd are in accordance with and in line with current environmental and dam stability and management requirements.

7.8 Mine Waste Disposal

Mine Waste will be transported from the mine via the High Angle Conveyor on the Eastern side of the pit. The waste will be stacked on a stockpile on the northern side of the ROM pad (see Figure 7-1) using the mobile stacker. At this point the waste will be rehandled using a wheel loader and a fleet of 90t class rear dump haul trucks and dumped at the waste storage facility located in the Triangle Lake area. “Triangle Lake” is a filled 300m by 350m depression to the east of the Bronson airstrip (Refer to Figure 7-1)   The majority of the mine waste is mined in the first 10 years of the mine life. A conceptual waste dump design has been provided in Figure 7-6.

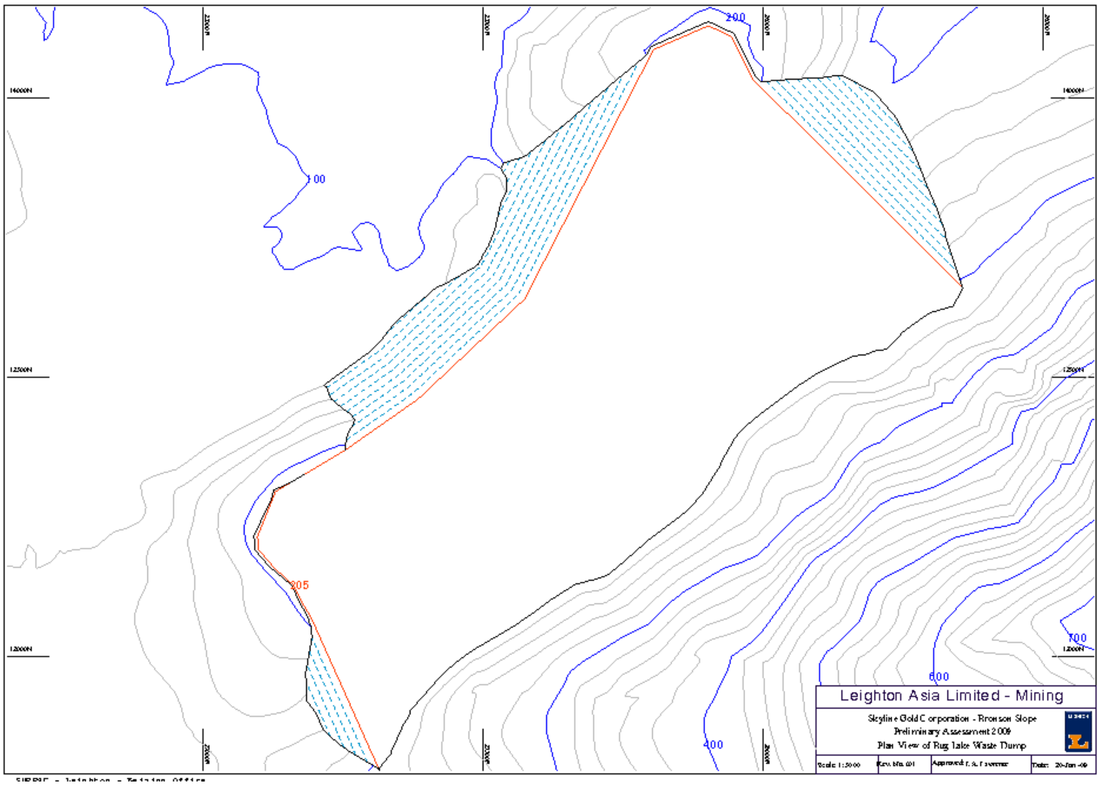


Figure 7-6: Conceptual Waste Dump

The waste storage facility will also be segregated into below cut-off grade mineralised rock (selected based on a marginal NRV cutoff and stockpiled as low grade for potential future processing) and waste (with no potential economic value). It is anticipated the marginal mineralised rock portion of the waste dump will allow for potential acid mine runoff. Further testing is required to determine the quantities of Potentially Acid Forming (PAF) waste that will be generated by the Bronson Project. A rehabilitation plan for the waste dump will be required with consideration for the PAF portion of the waste dump including the marginal low grade stockpile in the case that it is not considered economic to rehandle. Based on a NRV marginal grade cutoff of USD 7.00/t approximately 40% of the waste storage facility may be considered for rehandle at a later date. No economic consideration for below cut-of low grade rehandle has been given for this Preliminary Assessment. Further studies are required to determine if there is any economic benefit to this below cut-off grade material.

7.9 Process Plant Site

An economic concentrator plant site is situated at the south east of the property. It is located along the access road and it is around 250m south east of the Snip Mine airstrip. The plant site is bounded by Bronson creek to the east and Johnny Mountain to the south. The concentrator plant location is shown in Figure 7-7. The surface tenure at this site is currently held by Barrick Gold Corp.

Figure 7-7: Potential Concentrator Plant Location

Initial preparation of the site will include clearing and grubbing of the site for the coarse mill feed stock piling area, the concentrator plant, warehouse and power sub-station areas. Excavation and backfill of the areas for coarse mill feed stockpile and the plant is required in the early stages. Access roads to the stockpile and plant area will be outlined and developed. The warehouse, step down sub-station and an administration office will be constructed to provide support and electricity for the construction activities. Civil construction work on the floor foundation for the grinding area, flotation area and dewatering area will be carried out. Once the foundation is set, the milling and flotation building will commence construction. All heavy, large process equipment will be erected to allocated venues, assembly of the pipe work and electrical work will then be installed. Installation of pipe work and electric lines will be connected to all the machinery, lightings, utilities and other electrical accessories.

In the event that this site cannot be obtained for the process plant an alternative site on the southern end of the airstrip, which is 500 metres beyond the Snip mill site, may be considered. A portion of the airstrip for which SGC holds a license of occupation would need to be reclaimed for this purpose, however, given installation of the access road to Bronson Slope, the full length of the 1750m runway would not be required for regional aircraft use.

8 History

The following Item 8 – History has been extracted from Section 6 – History of the previous Technical Report titled “Mineral Resource Estimate – Bronson Slope Deposit”, dated May 10, 2007 and posted to SEDAR (www.sedar.com) on May 29, 2007. This report was prepared by A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc. and G. H. Giroux P.Eng., MASc. from Giroux Consultants Ltd..

8.1 Summary

SGC personnel have worked on the Property since 1988 and it was during a 1992 review of all exploration and drilling data by Burgoyne (1992) that the alteration and then defined mineralization indicated the potential for a large low-grade porphyry copper-gold deposit. In 1993 SGC performed Induced Polarization and Chargeability surveys as noted by Burgoyne (1993a) and a limited drilling program of 872 metres over 7 drill holes on two separate cross-sections of the deposit. This program was successful and is recognized in partially defining the Bronson Slope porphyry copper- gold deposit. A total of 15,276 metres of drilling over 81 diamond drill core holes were drilled in 1965, 1984, 1988, 1993 through 1997, and 2006. This drilling has defined the current resource that is detailed in **Item 19**. Also during the period of 1995 to 1997, extensive pre-feasibility engineering and scoping studies were completed.

8.2 Chronology

The major exploration activities on the Bronson Slope Property occurred between 1993 and 1997; however, exploration started much earlier and a summary review of each of these years' activities is given below. References include Yeager (1994), Yeager (1998b), and Yeager (2003).

1907-1920 - The earliest recorded work on the deposit was by the Iskut Mining Company who completed, between 1907 and 1920, surface and minor underground exploration of a number of base and precious metal prospects on the southwest slope of Bronson Creek valley. In the period 1911 to 1920 the Iskut Mining Company reported drifting, trenching and stripping a number of gold bearing veins on the Red Bluff and Iskut claims.

1962-1965 -The next phase of work for which accurate records were available was done during the period 1962 to 1965 (Parsons, 1965) during which time Cominco Ltd. had an option to develop the ground. Both regional and property scale surface mapping and prospecting were performed. This culminated, in 1965, with a packsack drill program comprising seven holes for a total of 337 metres of drilling. This program discovered several areas of promising copper and molybdenum mineralization, however the relatively low copper grade and gold prices prevailing at the time prohibited realization of the potential of the deposit.

1987-1988 - During the construction, in 1987, of the Johnny Mountain mine facilities by SGC Explorations Ltd., several contour lines were soil sampled in the vicinity of the Red Bluff as a preliminary step to performing a comprehensive exploration program to rediscover the object of the early 1900's prospecting and claim staking activity. The soil samples contained, among other metals, extremely high gold values. In 1988, following initial grid soil sampling and prospecting, a total of 1938 metres of diamond drilling was performed in five areas of the Bronson Slope, defined by anomalous gold concentrations in rock and soil samples and by base metal sulphide mineralization. The object of the drilling was to locate high-grade concentrations of precious metals similar to the nearby Stonehouse (Johnny Mountain gold deposit) and Twin Zone deposits (Snip Gold Mine) and therefore it was directed at mineralized cross structures. Again, promising low-grade concentrations of gold, copper and molybdenum were found but the values encountered were insufficiently high to interest the company in continuing the program.

1990-1991 – SGC completed exploration programs on behalf of Placer Dome Inc. in 1990 and 1991 who had an option on a block of the SGC ground including where the current Bronson Slope deposit is located. This work consisted of detailed geological mapping, prospecting, trenching, and extensive geochemical soil sampling for precious and base metals. Placer was exploring for gold-vein mineralization contained within a southeasterly extension of the then producing Snip Gold Mine owned by Cominco Ltd. In excess of $ 1 million dollars was spent.

1992 – A complete review of the Bronson Slope data was made by Burgoyne (1992) and on the basis of this evaluation, the recognition of a potential large porphyry copper-gold deposit was recognized and appropriate exploration recommendations, including diamond drilling, were made; these were subsequently followed out in 1993 through 1997.

1993-1997 - SGC performed a limited program of Induced Polarization surveys on the Bronson Slope copper-gold porphyry system in 1993. This was followed by an extensive program of advanced exploration and drilling (10,215 metres over 46 diamond drill core holes) between 1993 and 1997. **Note Item 13 – Drilling** for details. All drilling and exploration on the deposit ended in 1997.

Also during 1996 and 1997 SGC completed a substantial amount of engineering scoping, environmental, cash flow, metallurgical, capital and operating costs, geotechnical, infrastructure and access, and other pre-feasibility studies on the Bronson Slope deposit.

In late July 1997 the Company was able to announce the acquisition of two key mineral titles from Prime Resources Group Inc., which helped to enhance the Bronson Slope project. Two properties. The Kathleen fraction, and High Wall both of which are adjacent to SGC's Bronson Slope. The Kathleen fraction allowed SGC to consolidate its four principal Bronson Slope claims into one continuous block as indicated in Figure 6-3.

Upon acquisition of the High Wall area (of the Bronson Slope deposit) from Prime Resources Group, SGC also obtained access to previously drilled core completed in this area. SGC's 1997 program included the surveying of 7 historic core holes, re-logging of the drill holes, core splitting, and geochemical analyses of un-sampled porphyry mineralization. A six hole drill program conducted on the High Wall zone in 1997 defined a zone of gold mineralization with a strike length of 800 metres parallel to both the Bronson Slope porphyry deposit and to the Snip shear zone vein deposit. The zone contains disseminated gold mineralization grading in the 0.5 g/t to 0.6 g/t range over a true thickness of 60 -70 metres; this has substantial exploration tonnage potential.

1999 - In 1999 SGC completed an underground drifting program of 200.4 metres and 19 drill holes over 1494.5 metres on exploring extensions to the Snip Gold Mine shear veins; this program was funded by Royal Gold Inc. These drill holes are not included in the Bronson Slope database.

It is estimated that in the order of $3.5 million 2006 dollars was spent in the period of 1988 through 1997 on drilling, geology, and other exploration surveys. It is estimated that an equivalent amount was also spent on development and engineering studies from 1996 and 1997 (Yeager 2006).

2006 - During 2006 an office recompilation of drilling data was done followed in September and October by a four hole 561.6 metre HQ diameter core drilling program. The total 2006 exploration expenditures were $1.45 million.

8.3 Historical Mineral Resource Estimates

8.3.1 Base Case Historical Resource Estimate

The base case historical mineral resource estimate for the Bronson Slope deposit is that completed by Giroux (1996b) and is detailed in SGC's 43-101 Technical Report dated June 2006. The Burgoyne (2006) reports detail the rationale and reasons for the definition of this historical resource estimate. Here Giroux used a block model and ordinary kriging to determine the resource. The base case estimate, at US $1.00 equivalent to C$1.33, a US $6 NSR (Net Smelter Return) cut off, after using US $ 385 / ounce for gold, US $5.25 / ounce for silver, and US $1.10 / pound for copper and metal recoveries, smelter payments, refining charges, treatment charges and transportation is given below in Table 8-1:

Table 8-1: Bronson Slope Historical Resource - Base Case

Category	Tonnes	Au g/t	Ag g/t	Cu %
Measured	2,280,000	0.574	2.59	0.210
Indicated	65,000,000	0.527	2.46	0.195
Total Measured + Indicated	**67,280,000**	**0.528**	**2.46**	**0.196**
Inferred	24,300,000	0.454	2.23	0.199

The Giroux (1996b) historical resource base case, detailed in Burgoyne (2006), is the second of four historical estimates done by Giroux.

8.3.2 Background

Several resource estimates were undertaken by SGC in the period of 1994 through 1997. Some of the initial estimates were done mainly to identify zones of mineralization for future drilling and define tonnage ranges for future engineering studies. C.M. Turek undertook the in-house SGC resource estimates, at this time, using the PC-EXPLORE software of Gemcom Services in Vancouver. The second group of resource estimates were completed over a plus one year period (April 1996 to July 1997) using outside consultants G.H. Giroux, P. Eng. and G.F. Raymond, P.Eng. At the time of the consultant's estimates and later, SGC also engaged Mr. W. Martin, a SGC employee, to undertake combined resource /mine plan estimates for modelling and economic analyses using SURPAC software and Whittle optimization pit plan.

Table 8-2 below**,** illustrates the historical *base case estimate* plus several other historical estimates that were completed. These historical estimates are discussed in detail by Burgoyne (2006). The other historical estimate that is relevant and valid is that of Raymond (1997) where he uses essentially the same block modelling criteria and parameters and metal prices as that done by Giroux, to estimate 63.4 million tonnes grading 0.55 g/t gold, 2.59 g/t silver and 0.197 % copper in the measured and indicated category. No inferred resource was estimated although Raymond states there is an un-estimated inferred resource component. The measured and indicated resource and metal grades are quite similar to that of the base case Giroux study.

In addition to the above resource, SGC, in late 1997, completed preliminary estimations as to the size and grade of the High Wall Gold Zone, which is located on the south side of the deposit, within the High Wall area of a potential open pit. There was no formal independent historical resource report and the resource estimations done by SGC were not 43-101 or CIMM compliant and are not relevant on this zone. However, drilling indicated an exploration potential in the range of 12 to 15 million tonnes grading 0.5 to 0.6 g/t gold.

Table 8-2: Bronson Slope Historical Resource Estimates

Study*	Date US $ 6 NSR Cut Off	Method	Category**	Tonnes (millions)	Au g/t	Ag g/t	Cu %	Mo ppm	NSR
Giroux 40 ddh	April 30 96	Kriging 100m	Ind	54.7	0.557	2.38	0.186		8.89
			Inf	20.7	0.473	1.84	0.169		7.69
		Kriging 250 m	Ind	53	0.557	2.37	0.186		
			Inf	84.5	0.455	1.8	0.166		
Giroux 47 ddh	**Oct 8 96 Base Case**	**Kriging 100 m**	**Meas + Ind**	**67.3**	**0.528**	**2.37**	**0.196**		**8.72**
			Inf	**24.3**	**0.454**	**2.23**	**0.199**		**7.95**
		Kriging 250 m	Meas + Ind	67.3	0.529	2.37	0.196		8.72
			Inf	103	0.459	2.34	0.182		7.77
Giroux 56 ddh	Dec 16 96	Kriging100 m	Meas + Ind	74.5	0.559	2.65	0.198		9.1
			Inf						
		Kriging250 m	Meas + Ind	78.4	0.638	2.74	0.194		9.87
			Inf	103.6	0.718	2.87	0.175		10.45
Giroux 63 ddh	May 1 97	Kriging100m	Meas + Ind	85.9	0.59	3.05	0.163		8.91
			Inf	41.1	0.629	3.62	0.116		8.66
		Kriging 250 m	Meas + Ind	90.6	0.646	3.07	0.159		9.47
			Inf	179.7	0.67	3.35	0.123		9.2
Raymond	**July 15 97**	**Kriging**	**Meas + Ind**	**63.4**	**0.55**	**2.59**	**0.197**	**65**	**8.97**
62 ddh		Polygon	Meas + Ind	55.4	0.652	3.27	0.225	75	10.53

* ddh = diamond drill hole   * * Meas = Measured,   Ind = Indicated

The independent resource estimates given in Table 8-2 are all based on a specific gravity of 2.65. The resource portion of the High Wall (HW) is not taken into account in the Giroux (1996b) and Raymond (1997) studies.

8.3.3 Raymond 1997 Study

Raymond objectives (Raymond 1997) were to review deposit modelling on previously completed resource estimates; to review the problem with repeatability of higher grade gold assays; and to recommend a drill spacing for feasibility mineral reserve estimates. The database consisted of 4284 samples with assays (typically 3 m samples) from 12,549 m of drilling in 62 drill holes. In assessing potential gold assay problems he used the strong correlation between gold and copper. Raymond's concern, at the time, was to define mineral resources of the measured and indicated categories that were drilled close enough for that required for production mine design and scheduling and consequently on completion of a pre-feasibility (or feasibility) study converted to a mineral reserve. Consequently the approach was one of constraint with respect to the geologic model, the assays and resource estimation. The resource part of the study was concerned with the measured and indicated categories; the inferred resource component was not estimated.

Raymond made a series of recommendations to firm up the Bronson Slope database and for future drilling. All of these recommendations, where applicable, were completed by SGC in 1997 and in 2006 (Burgoyne 2006) and/or have been carried out in this report.

9 Geological Setting

The following Item 9 – Geological Setting has been extracted from Section 7 – Geological Setting within the previous Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated May 10, 2007 and posted to SEDAR (www.sedar.com) on May 29, 2007. This report was prepared by A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc. and G. H. Giroux P.Eng., MASc. from Giroux Consultants Ltd..

9.1 Regional Geology

The Iskut River region is within the Intermontane Belt on the western margin of the Stikine Terrane. Three distinct stratigraphic elements are recognised in the western portion of the area (Anderson, 1989): (i) Upper Paleozoic schists, argillites, coralline limestone and volcanic rocks of the Stikine Assemblage, (ii) Triassic Stuhini Group volcanic and sedimentary arc related strata, and (iii) Lower to Middle Jurassic Hazelton Group volcanic and sedimentary arc related strata.

Intrusive rocks in the Iskut River region comprise five plutonic suites. The Stikine plutonic suite comprises Late Triassic calc-alkaline intrusions, which are coeval with Stuhini Group strata. The Copper Mountain, Texas Creek and Three Sisters plutonic suites are variable in composition but are roughly coeval and co-spatial with Hazelton Group volcanic strata. Tertiary elements of the Coast Plutonic Complex are represented by predominantly granodiorite to monzonite Eocene intrusions of the Hyder plutonic suite, exposed 12 kilometres south of the Bronson Slope deposit (Alldrick et al., 1990).

The age, mineralogy and texture of the Red Bluff porphyry stock (associated with the Bronson Slope deposit), suggest that it belongs to the metallogenetically important Early Jurassic Texas Creek plutonic suite (Alldrick et al, 1990). Plutons of this suite are widespread in the Stewart, Iskut River region and range in age from 196 to 185 million years (Anderson, 1993; MacDonald et al., 1992). Figure 9-1 illustrates Regional Geology taken from Rhys (1995b).

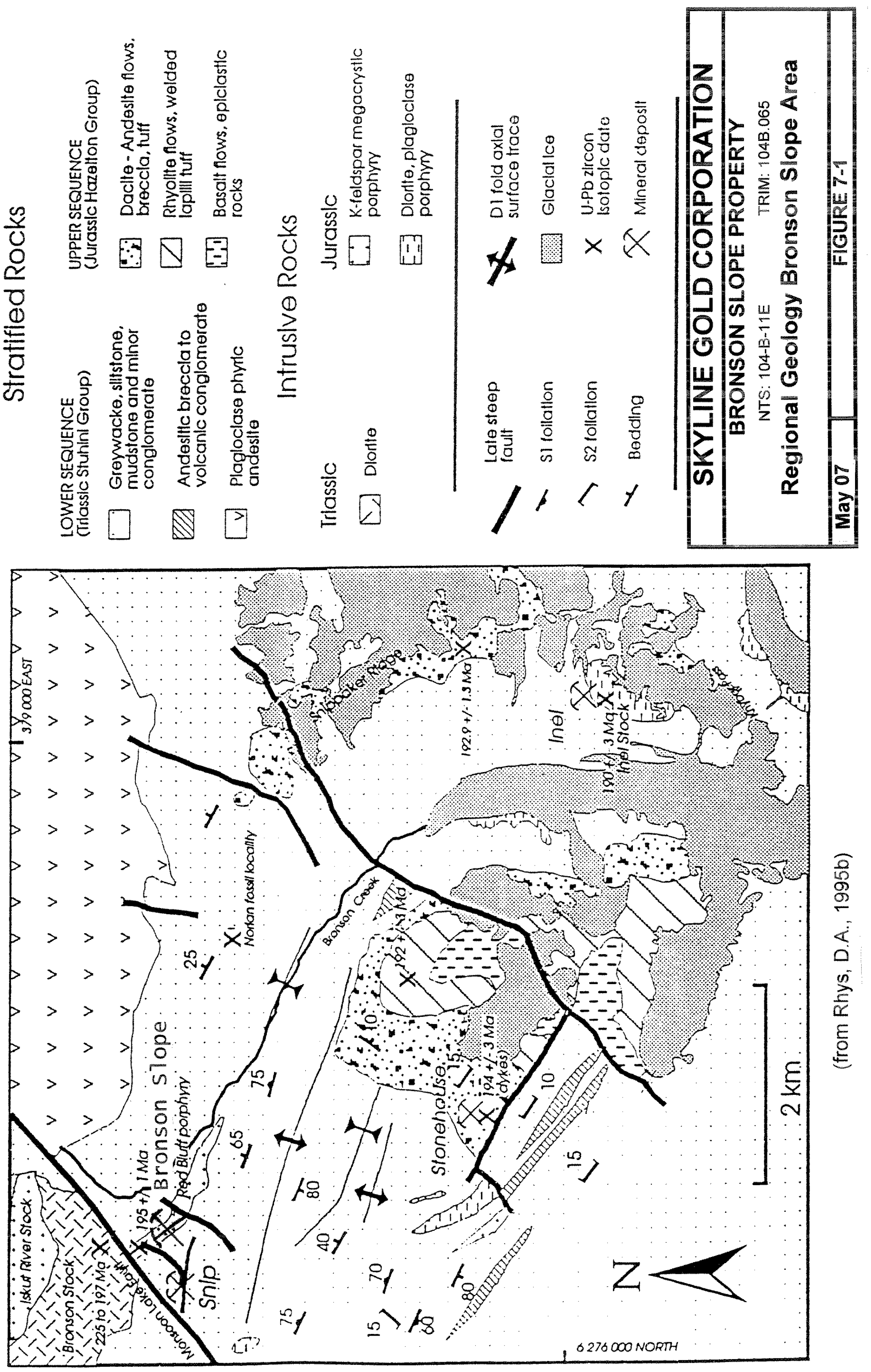


Figure 9-1: Regional Geology

9.2 Property Geology

The description on property geology is taken largely from Rhys (1995a), Rhys (1995 b), Yeager (1998b) and Yeager (2003) and the surface geology, illustrated in Figure 9-2**,** is taken from Piteau Associates (1997). Geological sections 25425E and 25,700E, illustrated in Figure 9-3 **and** Figure 9-4 are also taken from Piteau (1997).

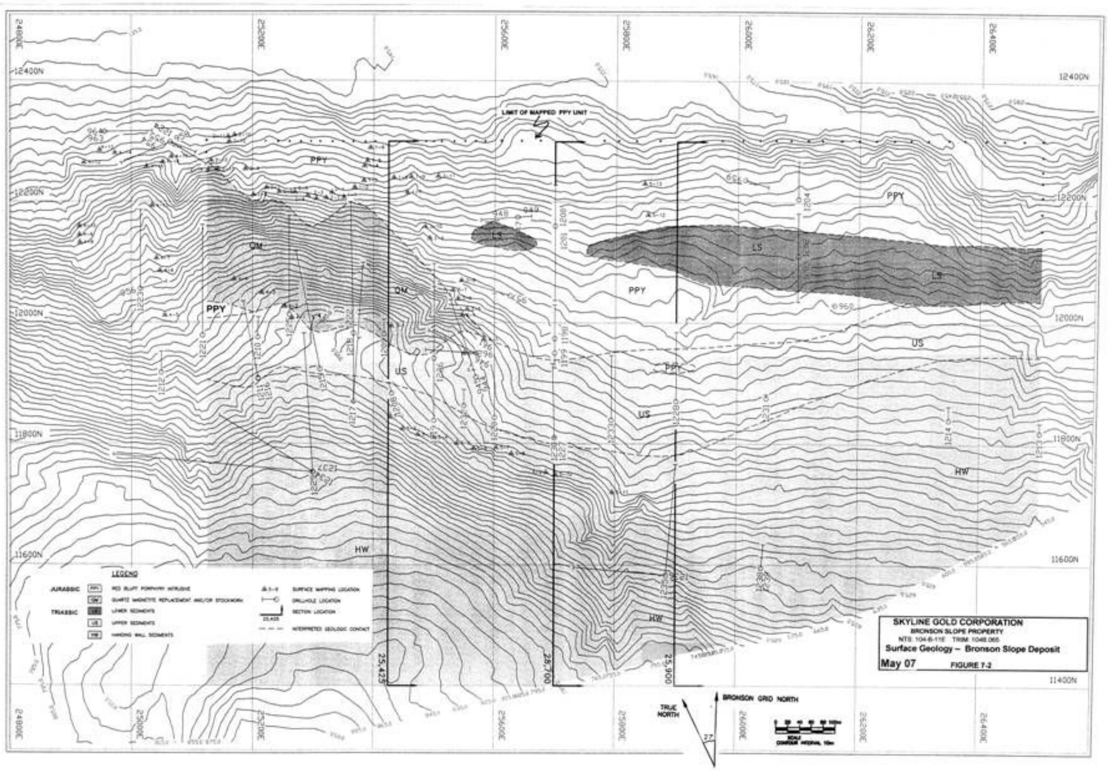


Figure 9-2: Surface Geology (Piteau, 1997)

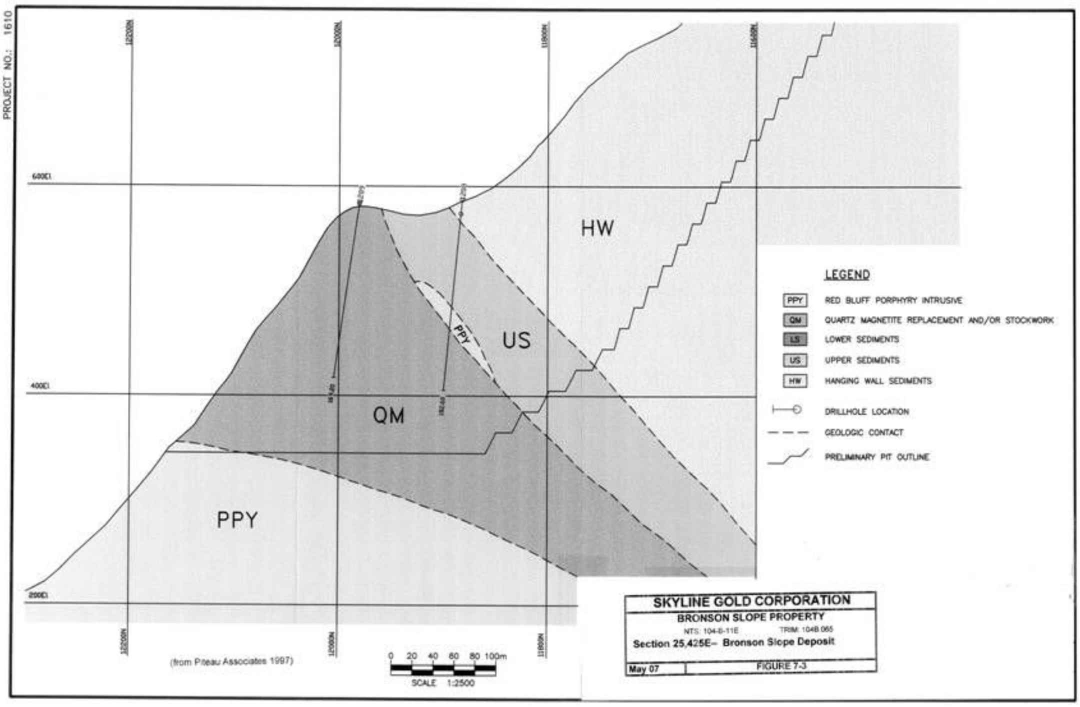


Figure 9-3: Deposit Section 25425E (Piteau, 1997)

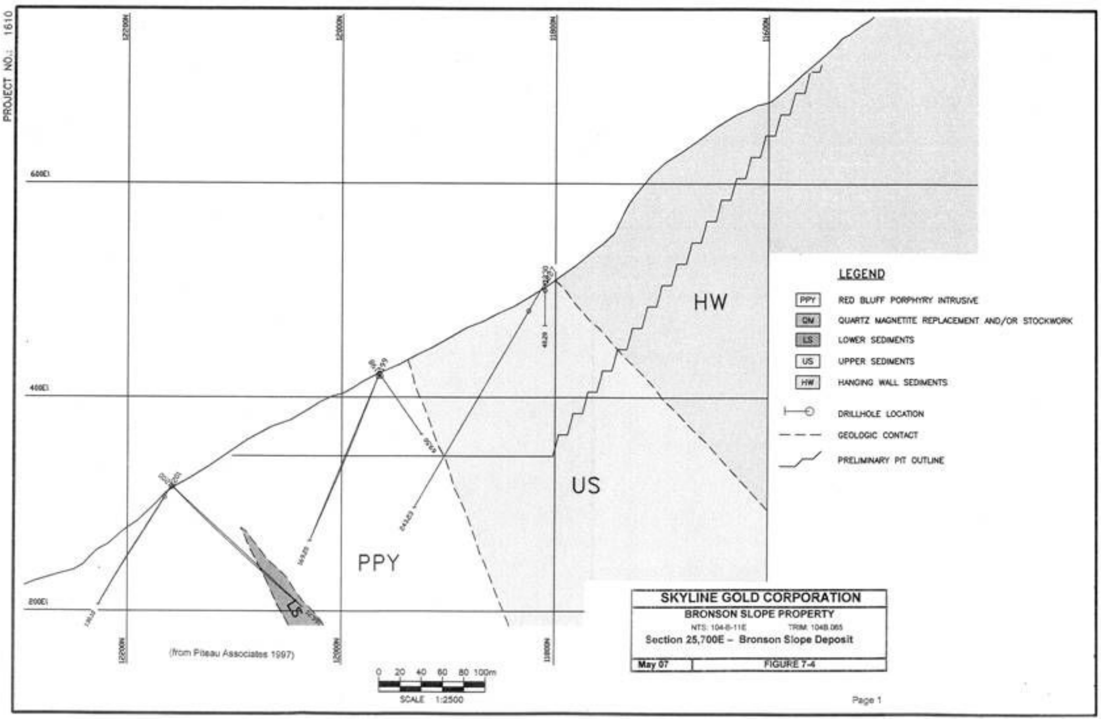


Figure 9-4: Deposit Section 25700E (Piteau, 1997)

A folded sequence of turbiditic feldspathic greywackes with subordinate inter-bedded siltstones, mudstones, volcanic conglomerate and rare, carbonate lenses is intruded by the Red Bluff porphyry. The greywackes are massive to crudely bedded. Individual graded beds may have sharp, scoured basal contacts and may contain siltstone or mudstone rip up clasts. The sequence is weakly to moderately metamorphosed (lower greenschist facies). Alteration ranges from weak to strong in the vicinity of mineral prospects. Pebble to cobble sized clasts of fine-grained and porphyritic mafic to felsic volcanic rocks are present in coarser beds, and coupled with the common presence of angular to sub rounded plagioclase grains in greywacke units, imply a proximal volcanic source. These rocks are probably lateral equivalents of Stuhini Group strata exposed on Snippaker Ridge 4 km southeast of Bronson Slope, which contain Upper Triassic fossils.

Early Jurassic felsic to intermediate volcanoclastic, pyroclastic and flow rocks that probably belong to the Lower Hazelton Group are exposed on Johnny Mountain. They are flat-lying to moderately tilted and unconformably overlie the greywacke sequence noted above. The sequences are separated by a flat lying to gently dipping regional unconformity exposed approximately one kilometer to the northeast of the Johnny Mountain Gold mine.

The Bronson stock is a heterogeneous, medium-grained equigrangular plagioclase + clinopyroxene +/- amphibole phyric diorite. The stock lies north of the former producing Snip Gold Mine. A poorly constrained Late Triassic U-Pb zircon age date of between 197Ma and 225 Ma was obtained from a K feldspar + plagioclase phyric monzodiorite phase of this unit (Macdonald et al, 1992). Several small stocks, sills and dikes of unknown age and intermediate to mafic composition intrude the Bronson stock. Lamprophyre dykes of probable Jurassic age have been mapped at numerous locations on the property and in addition lower Jurassic feldspar porphyry dykes and Tertiary intrusive stocks have been noted. Basalt dykes, possibly correlative with Recent volcanism, have also been observed

The lower sequence is intruded by the Red Bluff porphyry stock (Bronson Slope deposit), a hydrothermally altered, potassium feldspar megacrystic, plagioclase porphyritic intrusion of probable granodioritic composition. The stock is approximately 2.0 kilometres long, up to 0.3 kilometres wide and trends southeast along the southwest side of the Bronson Creek valley. Contacts of the stock with country rocks are not well defined, but where observed in drill core or underground workings are either faulted or intrusive. The southwest and northeast contacts appear to be southwesterly dipping. Screens of altered greywacke up to 40 m wide are common throughout the intrusion. The age of the Red Bluff intrusive is Lower Jurassic.

The Red Bluff porphyry is a hydrothermally altered K-feldspar megacrystic, plagioclase porphyritic intrusion of probable quartz diorite to quartz monzonite composition. Subhedral tabular pink K-feldspar phenocrysts generally range in length from 2 mm to 20 mm. They usually comprise from less than 1% to 5% of the modal mineralogy. The matrix to the K-feldspar megacrysts consists of medium-grained porphyry containing phenocrysts of albitic plagioclase, altered amphibole and quartz. The plagioclase is usually completely altered to aggregates of sericite+/- quartz +/- K-feldspar. Mafic phenocrysts, probably original hornblende from grain shapes, are commonly altered to magnetite, hematite, pyrite, biotite, and chlorite. Equant, clear to smoky sub rounded quartz phenocrysts, 0.2 mm to 1.5 mm in diameter, comprise less that 1% to 4%. In areas of moderate to intense alteration original quartz is difficult to identify. Accessory minerals include apatite, zircon and titanite. The fine-grained matrix to the phenocrysts forms between 35% and 70% of the rock volume.

Mineralization and alteration in and adjacent to the Red Bluff porphyry system are detailed in **Item 11** and summarized below:

- Quartz-magnetite-hematite veins are the earliest phase of veining in the Red Bluff porphyry system. They form an intense stock work that is spatially related to the Red bluff porphyry.

- The quartz-Fe-oxide stock work and altered sediments on its southwest margin are overprinted by quartz-pyrite+/-chalcopyrite veins/alterations and pyrite + chalcopyrite veinlets that are associated with the highest gold and copper grades. Where quartz-pyrite assemblages overprint and sulphidize the quart-Fe-oxide stock work there is a net loss of iron from the system. Veins are discrete, with sharp boundaries outside the stock work in greywacke, but have indistinct alteration boundaries with quartz-Fe-oxide veins within the stock work.
- The overall sequence from intense early Fe-oxide veining to less intense Quartz-pyrite-chalcopyrite veins and finally to pyrite and carbonate stringers corresponds with a progressive decrease in the total amount and intensity of veining though time.
- A 25 to 50 metre wide zone known as the transition zone of K-feldspar + Fe oxide alteration in greywacke occurs along the western upper periphery of the quartz-magnetite-hematite stock work and separates stock work from biotitic greywacke to the west. Calcite veinlets, common in the biotitic greywacke, become predominantly quartz veinlets in the transition zone.

Biotite lamprophyre dikes, un-deformed and unaltered, intrude northeast-trending faults in the Red Bluff cliff area. They are confined immediately adjacent or within fault zones.

9.3 Structure

The Triassic strata on Johnny Mountain are folded into an anticlinal structure defined by tight, locally overturned, northwest-trending regional and parasitic folds. An adjacent syncline follows the Bronson Creek valley along strike from the Red Bluff porphyry. The folds are associated with a moderate to northeast-dipping axial planar phyllitic flattening fabric (S1). All of the structures, and the entire Triassic-Jurassic sequence were subject to a later deformation resulting in shallowly dipping to sub-horizontal foliation (S2). Abundant shallow-dipping extension veins cut the fabrics on Johnny Mountain. Moderate to steep northwest-dipping and southwest-dipping fault sets cut all other lithologies and structures in the area.

To date, with the exception of the Red Bluff porphyry system, other mineral prospects on the property appear to be in veins or silicified shear zones. Most of the mineralized prospects conform to the following three shear directions:

- northwest dipping shears ($060^{O}/70^{O}$ NW) – e.g., Stonehouse Gold Deposit, Johnny Mountain
- southwest dipping shears ($120^{O}/45^{O}$ SW) - e.g., former mined Snip Gold Deposit, and,
- northeast dipping shears ($130^{O}/45^{O}$ NE).

In the case of the Snip shear direction, which trends onto Bronson Slope, the shearing may be related to regional folds that vary in intensity from small open fold belts to anticline-syncline pairs that can result locally in overturned bedding. The axial plane cleavage developed in these folds has created weakness in the rock and these zones of weakness have created conditions favorable for shearing in a northwest-southeast direction. The adjacent Snip veins appear to be emplaced in a shear zone that has developed in the axial plane cleavage of an anticline inferred from SGC mapping of the sedimentary rocks further south along the Bronson Creek valley. The Red Bluff porphyry may be emplaced parallel to the axial plane cleavage of the corresponding syncline lying just to the northeast of the Snip anticline.

10 Deposit Types

The following Item 10 – Deposit Types has been extracted from Section 8 – Deposit Types within the previous Technical Report titled “Mineral Resource Estimate – Bronson Slope Deposit”, dated May 10, 2007 and posted to SEDAR (www.sedar.com) on May 29, 2007. This report was prepared by A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc. and G. H. Giroux P.Eng., MASc. from Giroux Consultants Ltd..

The Bronson Slope copper-gold-silver-molybdenum mineralization is considered to be a porphyry copper-gold deposit type. Porphyry deposits (Kirkham, R.V. and Sinclair, W.D., 1996) are large, low to medium-grade deposits in which hypogene ore minerals are primarily structurally controlled and which are spatially and genetically related to epizonal and mezonal, felsic to intermediate porphyritic intrusions. The large size and structural control (e.g., veins, vein stock works, fractures, crackled zones, and breccia pipes) are of fundamental importance and serve to separate porphyry deposits from genetically-related (e.g., some skarns, high-temperature mantos, breccias pipes, etc.) and unrelated deposit types. Orientations of mineralized structures appear to be related to local stress environments around the top of the pluton or can reflect regional stress conditions.

Supergene minerals may be developed in enriched zones in porphyry deposits by weathering of primary sulphides.

The Bronson Slope deposit is considered to be a porphyry copper-gold subtype. This style of mineralization, many of which, but not all, are commonly associated with alkaline intrusive rocks. Bronson Slope is an exception in that it is associated with a plagioclase-clinopyroxene diorite or granodiorite intrusion. This subtype is defined if the gold content is greater than 0.4 g/t gold. If the content exceeds 0.8 g/t gold, the subtype can be identified as a porphyry gold deposit.

In British Columbia porphyry copper-gold deposits are commonly associated with Triassic and Lower Jurassic silica saturated intrusions, formed in an island-arc setting, but possibly during periods of extension.

11 Mineralization

The following Item 11 – Mineralization has been extracted from Section 9 – Mineralization within the previous Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated May 10, 2007 and posted to SEDAR (www.sedar.com) on May 29, 2007. This report was prepared by A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc. and G. H. Giroux P.Eng., MASc. from Giroux Consultants Ltd..

This discussion of mineralization is taken mostly from Rhys (1995a).

On the southwest side of the Red Bluff porphyry foliated sedimentary greywacke rocks contain calcite+/-quartz +/-pyrite +/- chlorite veinlets and stringers are either parallel to foliation or folded by the foliation. Sericitic shear zones are developed locally and are parallel to the surrounding pervasive foliation. On the southwest side of the Red bluff porphyry within 25 to 50 metres of the quartz magnetite-hematite stock work that defines the core of the system, foliation in the sediments generally disappears, magnetite appears as disseminations in veinlets and in quartz veins. Here veins become quartz-dominant with sparse calcite and intense K-feldspar alteration is widespread. The rock is commonly pale to dark green, mottled with disseminated blebs of magnetite + hematite. Quartz-magnetite-hematite veins, generally 0.3 to 2 cm wide, increase in density and thickness gradually down hole as the quartz-Fe-oxide stock work is approached. This area of distinctive alteration is termed the transition zone.

The Red Bluff porphyry hydrothermal system is dominated by an intense quartz-magnetite-hematite stock work that trends northwest along the northern slope of Johnny Mountain and the south side of Bronson Creek valley. The stock work overprints and is intimately associated with the Red Bluff porphyry intrusion. Margins of the stock work are usually discrete. Over intervals of a few metres vein abundance increases from 10-25% of the total rock outside the stock work to greater than 60% within it. The veins form an intense stock work that usually contains less than 20% interstitial rock. Drill intersections of 20 to +100 metres long are composed entirely of intersecting to sheeted sets of quartz-magnetite-hematite veins. Individual veins range from 0.5 to 10 cm in thickness. Vein to core axis angles are highly variable.

The quartz-magnetite-hematite stock work is overprinted by quartz + pyrite + chalcopyrite +/- carbonate veins and by carbonate and pyrite veins. The textures suggest that much of the quartz-pyrite may be an in situ alteration of the quartz-Fe-oxide assemblage. The total sulphide content in the quartz-pyrite assemblage is around 5%. The quartz-pyrite assemblage comprises less than 10% of the older quartz-magnetite-hematite veins.

Pyrite + chalcopyrite +/- carbonate veinlets and veins frequently cut, but are intimately associated with the quartz-pyrite veins and alteration. They commonly have consistent core to axis angles suggesting they are sheeted.

The quartz-pyrite veins/alteration are locally brecciated. Breccias have variable contacts with the surrounding quartz veins that vary from gradational to sharp. A late set of quartz veins, possibly Tertiary in age, cuts all of the rock types and veins. These veins are flat to shallow southeast dipping, lenticular in shape and commonly occur in en echelon arrays. In drill core they are difficult to distinguish from veins in the Red Bluff porphyry system.

Gold and copper grades reflect the distribution of the different veins and alteration types. Areas of quartz-magnetite-hematite veining with sparse or no pyrite-chalcopyrite or quartz-pyrite overprinting typically grade less than 600 ppm copper, and less than 0.2g/t gold (Rhys 1995a). Higher copper and gold grades occur in quartz-pyrite-chalcopyrite veins and alteration and in areas of abundant pyrite-chalcopyrite veining both inside the quartz-Fe-oxide stock work and in adjacent greywacke; here grades can vary from greater than 600 ppm to 5000 ppm copper and greater than 0.2g/t gold to 10 g/tonne gold (Rhys 1995a).

The Red Bluff potassium feldspar porphyry is defined by an intense gossan and cliff zone. This in turn is surrounded by an intense phyllitic zone comprising quartz, sericite, and pyrite. To the southeast along the south side of Bronson Creek Valley this alteration grades into a propylitic zone of quartz, biotite, pyrite and chlorite contained within sandstone/siltstone/wacke sedimentary and dacitic volcanic units.

12 Exploration

The following Item 12 – Exploration has been extracted from Section 10 – Exploration of the previous Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical Report can be viewed at www.sedar.com.

Pre 1987 exploration conducted prior to SGC is summarized in Burgoyne and Giroux (2007**)**. The earliest recorded work on the deposit was by the Iskut Mining Company who completed, between 1907 and 1920, surface and minor underground exploration of a number of base and precious metal prospects on the southwest slope of Bronson Creek valley. In the period 1911 to 1920 the Iskut Mining Company reported drifting, trenching and stripping a number of gold bearing veins on the Red Bluff and Iskut claims of the Property.

The next phase of work for which accurate records are available was done during the period 1962 to 1965 during which time Cominco Ltd. had an option to develop the ground. Both regional and property scale surface mapping and prospecting were performed. This culminated, in 1965, with a packsack drill program comprising seven holes for a total of 337 metres of drilling. This program discovered several areas of promising copper and molybdenum mineralization, however the low copper grades and low gold prices prevailing at the time prohibited realization of the potential of the Property.

Exploration expenditures carried out by SGC on the Bronson Slope Property have been extensive but have not been quantified, as much of these expenditures have been included with those on the adjoining Iskut Property including the Johnny Mountain gold mine. Although no quantitative numbers are available, considering the amount of drilling and other ground surveys, it is estimated that the equivalent of $3.5 million of 2006 dollars has been spent (Yeager, 2006). Prior to 2006, the exploration programs by SGC occurred over an 11-year period from 1987 through 1997 with most of the exploration consisting of diamond core drilling from 1993 through 1997. In 2006 (November 1, 2005 through October 31, 2006) an HQ diameter drilling program and other studies cost $ 1.4 million. In 2007 (November 1, 2006 to October 31, 2007) an NQ diameter drilling program and mine development studies cost $ 3.7 million. The drilling is detailed in **Item 13.**

During the construction, in 1987, of the Johnny Mountain mine facilities by SGC several contour lines were soil sampled in the vicinity of the Red Bluff (on the Red Bluff crown granted claim which is in part underlain by intrusive quartz porphyry) as a preliminary step to performing a comprehensive exploration program to rediscover the object of the early 1900's prospecting and claim staking activity. The soil samples contained, among other metals, extremely high gold values. In 1988, following initial grid soil sampling and prospecting, a total of 1938 metres of diamond drilling was performed in five areas of the Bronson Slope, defined by anomalous gold concentrations in rock and soil samples and by base metal sulphide mineralization. The object of the drilling was to locate high-grade concentrations of precious metals similar to the nearby Stonehouse (Johnny Mountain gold deposit) and Twin Zone deposits (Snip Gold Mine) and therefore it was directed at mineralized cross structures. Again, promising low-grade concentrations of gold, copper and molybdenum were found but the values encountered were insufficiently high to interest the company in continuing the program.

SGC completed exploration programs on behalf of Placer Dome Inc. in 1990 and 1991 who had an option on a block of the SGC ground including where the current Bronson Slope deposit is located. This work consisted of 1:2500 scale geological mapping, prospecting, trenching, extensive geochemical soil sampling for precious and base metals. Geochemical and geological survey lines were oriented grid north (025° 12' azimuth) and were at about 100-metre spacing. This mapping work was instrumental in defining a geological favourable or "anomalous area" of alteration and anomalous soil geochemistry covering a southeast strike through the Bronson Slope Property parallel to the Bronson Creek Valley. The "anomalous area" is found over the complete strike length of the property, which is about 1800 metres. The Red Bluff potassium feldspar porphyry is defined by an intense gossan and cliff zone. This in turn is surrounded by an intense phyllitic zone comprising quartz, sericite, and pyrite. To the southeast along the south side of Bronson Creek Valley this alteration grades into a propylitic zone of quartz, biotite, pyrite and chlorite contained within sandstone/siltstone/wacke sedimentary and dacitic volcanic units. The "anomalous area" is for the most part underlain by a strong, well defined, and continuous in-situ gold anomaly. The anomaly threshold is considered to be 91 parts per billion gold, however, a majority of the anomaly is characterized by + 250 ppb values. Coinciding copper and zinc in-situ soil anomalies occur intermittently. A strong 1.2 kilometre long copper soil anomaly is coincident to the gold anomaly at its western edge. Most of the copper values are in excess of 400 parts per million (ppm). Placer was exploring for gold-vein mineralization contained within a south easterly extension of the then producing Snip Gold Mine owned by Cominco Ltd.; consequently they did not recognize or consider the porphyry copper-gold potential. In excess of $ 1 million dollars was funded by Placer for this 1990 and 1991 exploration although part of it was spent on the adjoining Iskut Property owned by SGC.

A complete review of the Bronson Slope data was made by Burgoyne (1992) and on the basis of this evaluation, the recognition of a potential large porphyry copper-gold deposit was recognized and appropriate exploration recommendations, including diamond drilling were made; these recommendations were subsequently followed out in 1993 through 1997. SGC performed a limited program of Induced Polarization surveys on the Bronson Slope copper-gold porphyry system in 1993. These surveys were done by Scott Geophysics and covered most of the trend of the now Bronson Slope deposit and included 12 cross lines that varied from 330 to 700 metres in length. Most of the exploration work completed after the geophysical survey of 1993 was directed to core drilling in 1993, 1994, 1995, 1996, and 1997. This core drilling included nine separate drilling programs by SGC (1994 and 1996 were subject to two separate programs each). All exploration and drilling ended on the deposit in 1997. **Note Table 13-1 and Item 13.** Upon acquisition of the High Wall area (of the Bronson Slope deposit) from Prime Resources Group, SGC also obtained access to previously drilled core completed in this area. SGC's 1997 program included the surveying of 7 historic Cominco/Prime core holes from 1986 and 1994 totalling 2332 metres, re-logging of the drill holes, core splitting, and geochemical analyses of un-sampled porphyry mineralization. Also during the 1993-1997 period, but mostly in 1997, extensive pre-feasibility, engineering, and scoping studies were completed; some of this work was done in 1998 and is detailed in **Item 20.** In 1999 SGC completed an underground drifting program of 200.4 metres and 19 drill holes over 1494.5 metres on exploring for extensions to the Snip Gold Mine shear veins. Royal Gold Inc funded this program.

During September and October 2006 SGC completed 561.6 metres over 4 HQ diameter holes within the Red Bluff Zone, a higher grader part of the Bronson Slope deposit. This drilling was done, in part, to compare the HQ core diameter results to those of previous NQ diameter holes in this particular area.

During July through October 2007 SGC completed 3936 metres over 11 NQ diameter holes within the Bronson Slope deposit. This drilling was done, in part, to develop additional resource and to increase mineral resource confidence by up grading inferred and indicated categories in certain parts of the deposit to measured and indicated, respectively.

In summary the placement of grids, surveying, collection of the soil samples, the extensive geological mapping, the location of the drill holes, the drill hole orientations, the analyses, and the collection and analyses of core samples appears to be to good industry standards.

13 Drilling

The following Item 13 – Drilling has been extracted from Section 11 – Drilling within the previous Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical Report can be viewed at www.sedar.com.

This drilling has defined the Bronson Slope porphyry gold-copper-silver-molybdenum system in the order of 1.5 km long and 0.4 to 0.6 km wide and an additional gold-pyrite zone known as the High Wall or Snip Extension located on the south side of the deposit. The plan of drill hole locations is illustrated on Figure 13-1 and the distribution of the Bronson Slope porphyry style gold-copper-silver-molybdenum deposit and the High Wall Gold Zone are illustrated on Figure 6-3. The High Wall Gold Zone is about 800 metres in length, 60-70 metres wide, and is located on the south side of the Bronson slope deposit.

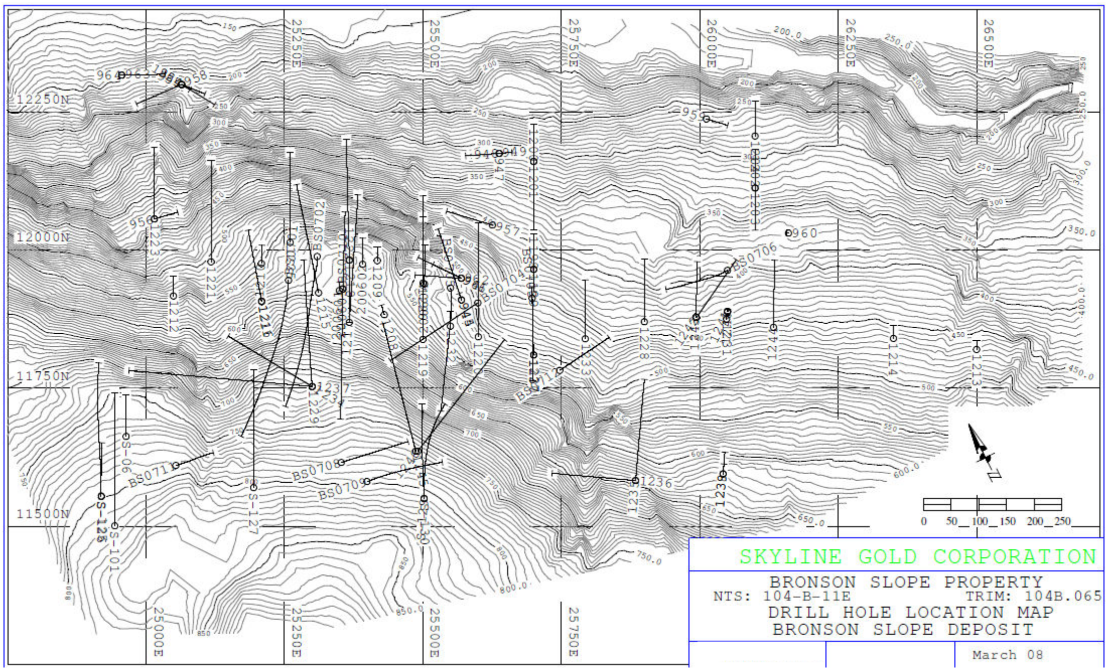


Figure 13-1: Drill Hole Locations

All drilling to date has been by wire line diamond core drilling. Drilling on the Bronson Slope Deposit in 1965, 1986,1988 and 1993 through 1997, and 2006 and 2007 involved a total of 19,320 metres over 92 core drill holes. Drilling by SGC in 1988 and 1993 through 1997 involved a total of 12,153 metres over 63 core drill holes. Drilling in 2006 involved a total of 562 metres over 4 holes. Drilling in 2007 involved a total of 3936 metres over 11 holes.

Drilling done in 1986 and 1994 by Cominco and Prime Resources, with respect to exploration on the adjacent Snip Gold Mine, was acquired by SGC in 1997 – this drilling, in the High Wall of the Bronson Slope deposit, was evaluated in 1997 and included the surveying of 7 historic core holes, re-logging of the drill holes, core splitting, and geochemical analyses of un-sampled porphyry mineralization.

The summary of diamond core drilling is given in Table 13-1.

Table 13-1: Summary of Diamond Drilling - Bronson Slope Desposit

Period	Company	Drilling Contractor	Core Size*	Hole Numbers	Holes	Metres
1965	Cominco	Cominco	Pack sack	1073 to 1080	7	337
1986	Cominco		BQ	S 6	1	108
1994	Prime Resources	Olympic Drilling	BQ	S101, S125-127, S129, S130	6	2224
1988	SGC	Falcon Drilling	BQ tw	944 to 949, 954 to 964	17	1,938
1993	SGC	Boisvenu Drilling	BQ tw	1198 to 1204	7	872
1994	SGC	Olympic Drilling	BQ tw	1208 to 1216	9	1,550
1995	SGC	Olympic Drilling	BQ tw	1217 to 1223	7	2,429
1996	SGC	Britton Brothers	BQ tw	1224 to 1239	16	3,529
1997	SGC	Britton Brothers	NQ**	1240 to 1246	7	1,835
2006	SGC	Phil's Drilling & Boart Longyear	HQ	#BS0601 to BS0604	4	562
2007	SGC	Blackhawk Drilling	NQ	BS 0701 to BS 0706, BS 0708 to BS 0712	11	3936
	* tw = thin wall	** One HQ hole		**Totals**	**92**	**19,320**

Holes also reported as 200601 to 200604

Diamond drill hole data including hole number, depth, northing, easting, elevation, azimuth and dip are given in Table 13-2. Figure 13-1 should be referred to for exact drill hole location.

The surface drilling by SGC consisted of drill holes that were completed over the Bronson Slope deposit. Drill holes varied from 28 m to 452.9 m and were BQ size diameter for the 1993 through 1996 campaigns and NQ size in the 1997 drilling campaign. The 2006 drilling program consisted of HQ diameter size. The Cominco 1965 drilling was by packsack and therefore less than 1.3 cm diameter. The SGC drills were transported to the drill site location by helicopter. The drilling contractors are given in Table 13-1. All drill hole collars were transit surveyed – down the hole acid etch dip deviation surveys were completed on most core holes (from 50 to 125 metre intervals) generally on holes greater than 100 metres. No down hole surveys were done on the 1965 Cominco holes

The drilling was completed over approximately 1400 metres of strike length and 600 to 700 metres across trend on drill lines perpendicular to the assumed strike of the deposit. The stratigraphic trend is 115 degrees and many of the drill lines were perpendicular at 025 degrees azimuth. The mineralization is in the form of stock works that dip in the order of 45 to 60 degrees to the south. Many of the earlier 1988 drill holes were drilled oblique to the trend. These drill hole sections were nominally at 100 m spacing over defined mineralization although this varied in parts of the grid and was lesser and greater in certain parts of the deposit. Much of the drilling, as indicated above, was positioned to intersect the mineralization perpendicular to the trend and to its probable dip. Weighted drill core recovery for drill holes in the Bronson Slope deposit is in the order of 95% to 99% (Yeager, 2006).

It is significant that the Bronson Slope deposit is open to the east and at depth.

Appendix 3 illustrates the wire line surface diamond drill mineralized intercepts. Porphyry mineralization is contained in highly potassic altered intrusive and sedimentary country rock within stock works of quartz-magnetite-hematite and pyrite veined mineralization-hosting gold-copper-silver-molybdenum.

The lengths of the mineralized intersections in Appendix 3 have not been corrected for true thickness due to the porphyry nature of the deposit. During calculation of the resource in Item 17 the spatial location (in three dimensions) of the mineralization is constrained by the outline, strike and dip, and form of the different rock units from which a block model was constructed. The 1965 Cominco drill hole results were not used in the resource estimation although assay results are given in Appendix 3.

Drill core from the SGC 1994 through 1997, 2006, 2007 programs, and the Cominco/Prime 1986 and 1994 programs are contained in core racks at the Bronson Airstrip. Drill core from the 1988 and 1993 campaigns is no longer available having been lost on collapse of drill core sheds due to snow load in the winter of 2000-2001. Also, remaining drill core sample rejects and drill sample pulps were disposed of by SGC subsequent to 2000.

The following yearly summaries of drilling were taken, in large part, from Yeager (1997b) and Moore (1997b).

Cominco Program (1965)

Cominco Ltd. performed the first recorded diamond drilling on the property in 1965 as part of the work requirements of an option agreement referred to as the “Tuksi - Jodi Joint Venture”. Porphyry copper mineralization was the primary target of this exploration program, although other deposit types were also targeted. The drilling was performed using a packsack drill. Eight holes were attempted (65-1 to 65-8) but only seven drill logs are recorded; hole number 65-6 may not have been drilled due to a severe windstorm blowing most of the equipment off the set up. A total of 337 metres were drilled. The drill hole collar locations have been confirmed by mapping or estimated from the Cominco assessment report and the assay information entered into the SGC database. The holes have been renumbered for the purpose of computer entry as follows:

Renumbering of 1965 Drill Holes

Cominco No.	65-1	65-2	65-3	65-4	65-5	65-7	65-8
SGC No.	1073	1074	1076	1077	1078	1079	1080

Samples from this program were shipped to the Cominco smelter at Trail, B.C. where they were assayed for copper and in the case of one hole, 1080 (65-8), for molybdenum. Spot assays were performed for gold. A number of samples were reported as lost in shipping. The core from this program is presumed to have been completely used for sampling or otherwise disposed of; certainly none is known to exist today.

1988 SGC Program

Skyline Explorations Ltd. performed the next drill program in 1988. The program was designed to test gold bearing shears, faults, veins and soils that had been detected by ground exploration work during 1987 and early 1988. Most of the gold targets were auriferous cross structures cutting the main porphyry containing gold grades potentially mineable by underground methods. Seventeen drill holes were completed and numbered RB-1 to RB-17. A total of 1938 metres were drilled. The holes have been renumbered for the purpose of computer entry as follows:

Renumbering of 1988 Drill Holes

1988 No.	RB-1	RB-2	RB-3	RB-4	RB-5	RB-6	RB-7	RB-8	RB-9
Present No.	944	945	946	947	948	949	954	955	956
1988 No.	RB-10	RB-11	RB-12	RB-13	RB-14	RB-15	RB-16	RB-17	
Present No.	957	958	959	960	961	962	963	964	

Samples from this program were split at the Red Bluff field camp and prepared and assayed at SGC's mill assay lab at the Johnny Mountain Gold Mine. A number of samples were lost in processing. The split core was stored at the Red Bluff field camp then later moved to the core storage building at Johnny Mountain.

A detailed description of the quality control issues and certain problems with this 1988 drill core program is discussed and detailed in Item 16.1. Essentially all of the 1988 core was re-assayed during 1994 and 1995 and these new values have been used in the block model resource estimate.

1993 SGC Program

After several corporate reorganizations and the suspension of operations at the Johnny Mountain Gold Mine, exploration attention focused on the large tonnage, low-grade gold potential of the Bronson Slope deposit. An Induced Polarisation survey outlined a low resistivity zone and two fences of holes (1198 to 1204) were drilled across the trend of the zone to determine metal zoning within it. The program comprised 872 metres of drilling in 7 drill holes.

Core from this program was split and sampled at the Johnny Mountain core shed where the split core was stored. The samples were sent to Chemex Labs Ltd. in North Vancouver, B.C. for preparation and analysis. The Chemex assay values for the 1993 drilling have been used in the block model estimate of the deposit.

During the 1995 re-sampling of the 1988 core, hole number 1198 was also re-split as a further variability check. The quartered samples were assayed at Rossbacher Labs. The 1993 Chemex assays were used in the block model on the assumption that half split core would be more representative than quarter split core.

1994 SGC Program

The two phases of the 1994 program comprised 1550 metres of drilling in 9 drill holes (1208 to 1216) that were designed to define the probable strike (phase 1) and depth (phase 2) extent of the deposit. Split core samples were prepared and assayed at Rossbacher Labs. The remaining core was stored at the exploration camp belonging to Pamicon Developments Ltd. located at the northwest end of the Bronson Creek airstrip. Hole 944 (1988) was re-sampled in 1994.

1995 SGC Program

The 1995 program comprised 2429 metres of drilling in 7 drill holes (1217 to 1223) that were designed to define the probable depth extent of the deposit and to increase confidence in the predicted resource grades. Split core samples were prepared and assayed at Rossbacher Labs.

The remaining core was stored at the exploration camp belonging to Pamicon Developments Ltd. located at the northwest end of the Bronson Creek airstrip.

Holes 944 to 949 and 954 to 964 (1988) were re-sampled and analysed in 1995.

1996 SGC Program

The 1996 program comprised 3529 metres of drilling in 16 drill holes (1224 to 1239) drilled in two phases. Drill holes 1224 to 1234 were designed to: (i) fill in the drill hole pattern to the minimum spacing required to allow the resource block model to be filled using a 75 metre data search, and (ii) further explore the easterly extent of a trend of higher grade gold and copper values detected in earlier drilling. Holes 1235 to 1239 were drilled at the request of Prime Explorations Group Inc. as part of the requirements of an agreement whereby the portion of Prime ground necessary for the purpose of mining the Bronson Slope deposit could be acquired by SGC. The holes were designed to test for gold bearing structures capable of being mined from underground. If any such structures were found, Prime would have the right to extract ore from them, if feasible, by underground methods before the ground could be acquired by SGC. Split core samples were prepared and assayed at Rossbacher Labs. The remaining core was stored at the Johnny Mountain mine site.

1997 SGC Program

The 1997 program comprised 1835.2 metres of drilling in 7 drill holes (1240 to 1246). Drill holes 1240 to 1244 were designed to further explore the easterly extent of a trend of higher grade gold, copper and molybdenum values detected in earlier drilling. Split core samples were prepared and assayed at Rossbacher Labs. The remaining core was stored at the Pamicon Developments Ltd. exploration camp at Bronson Creek airstrip.

Drill hole 1231 and 1232, drilled in the 1996 phase II program contained anomalously high gold values. All of 1231 and the first part of 1232 were re-sampled in 1997 to determine the repeatability of the 1996 results. It was apparent that gold contamination of the 1996 samples had occurred. The holes were re-sampled and re-assayed.

Upon acquisition (Moore, 1997b) of the High Wall area (of the Bronson Slope deposit) from Prime Resources Group, SGC also obtained access to previously drilled core completed in this area. This 1997 program included the surveying of 7 historic core holes from 1986 and 1994 totalling 2332 metres, re-logging of the drill holes, core splitting, and geochemical analyses of un-sampled porphyry mineralization. As a result of the evaluation of this drill core, SGC identified a high-grade gold intersection, which was on strike with the Snip deposit's Twin Zone; this intersection contained 2.0 metres of 15.7 grams per tonne and is probably an extension of the Snip 412 Zone vein. In addition, evaluation of the earlier Prime Resources drilling, defined the High Wall Gold Zone.

1999 SGC Program

In 1999 SGC completed an underground drifting program of 200.4 metres and 19 drill holes over 1494.5 metres on exploring for extensions to the Snip Gold Mine shear veins. Royal Gold Inc funded this program. The drilling was not directed toward Bronson Slope deposit porphyry style mineralization but in defining extensions to the Snip Gold Mine shear veins. Drilling results were disappointing and SGC recommended that no further work be done in this respect.

These drilling results are not tabulated in Tables 11-1 and 11-2 as they are of no relevance and importance in defining the Bronson Slope Deposit. These results are not part of the database for Bronson Slope Deposit and consequently are not reported in Appendix 3.

2006 SGC Program

During September and October 2006 SGC completed 561.6 metres over 4 HQ diameter holes within the Red Bluff Zone, a higher grade part of the Bronson slope deposit. This drilling was done, in part, to compare the HQ core diameter results to those of previous NQ diameter holes in this particular area.

BS0601, which only reached a depth of 30 metres, was re drilled by hole BS0603 from essentially the same drill site. BS0603, drilled to a depth of 270 metres is an in fill hole between holes 1215 and 1218 drilled in 1994 and 1995. Similar copper and gold grades were obtained. Hole BS0602, drilled to a depth of 138.6 metres is an in fill hole between holes 1209 and 1222 again drilled in 1994 and 1995. Note assay values in Appendix 3.

BS0604, drilled to a depth of 122 metres, is a near twin to hole 1226 drilled in 1996. In hole 1226 the top 117.5 metres weight averages 0.42 g/t gold and 0.208 % copper versus the top 117 metres in hole BS0604 weight averages 0.397 g/t gold and 0.208 % copper. The differences in grades in the holes can be explained by normal variance.

2007 SGC Program

During July through October 2007 SGC completed 3936.2 metres over 11 NQ diameter holes using Blackhawk Drilling of Smithers, BC. These drill holes were done within the Red Bluff Zone, a higher grade part of the Bronson Slope deposit; on the east side of Bronson Slope deposit; and in the High Wall Zone. The azimuth orientation of the three High Wall drill holes was directed grid northeast as opposed to a more favourable grid north direction. The drill hole assay results are tabulated in Appendix 3, located on Figure 13-1 and with drill hole data given in Table 13-2.

Table 13-2: Bronson Slope Diamond Drill Hole Data

Hole	Total Depth (metres)	Easting** (metres)	Northing** (metres)	Elevation (metres)	Azimuth** (degrees)	Dip (degrees)
944	206.3	25569.6	11909.7	513.9	342	-50
945	48.0	25569.6	11909.7	513.9	342	-75
946	121.6	25569.3	11948.9	507.0	296	-45
947	78.9	25637.8	12174.5	315.0	270	-90
948	66.8	25637.8	12174.5	315.0	086	-70
949	123.4	25637.8	12174.5	315.0	266	-60
954	100.0	25064.3	12299.1	222.0	122	-46
955	91.1	25064.3	12299.1	222.0	112	-60
956	88.1	25015.0	12056.0	425.0	074	-60
957	118.6	25625.6	12044.9	403.0	287	-45
958	127.1	25064.3	12299.1	222.0	247	-45
959	108.8	26012.0	12237.0	261.0	107	-68
960	185.6	26160.2	12030.2	373.0	270	-90
961	133.2	25569.3	11948.9	507.0	296	-65
962	117.3	25569.3	11948.9	507.0	274	-46
963	115.5	24956.0	12316.0	196.0	270	-90
964	109.4	24956.0	12316.0	196.0	089	-44
1198	69.5	25700.0	11965.0	422.0	180	-55
1199	169.2	25700.0	11965.0	422.0	360	-69
1200	166.7	25700.0	12160.0	317.0	180	-45

Table 13- 2: Bronson Slope Diamond Drill Hole Data (cont'd)

1201	130.1	25700.0	12160.0	317.0	360	-59
1202	108.8	26100.0	12112.0	322.0	180	-45
1203	120.1	26100.0	12112.0	322.0	360	-55
1204	105.8	26100.0	12206.0	282.0	360	-55
1208	182.6	25430.3	11882.7	587.3	343	-84
1209	169.2	25417.6	11980.7	584.7	360	-82
1210	124.1	25208.4	11975.5	551.8	002	-74
1211	165.5	25208.6	11907.7	601.3	351	-82
1212	135.3	25049.4	11916.8	546.0	360	-75
1213	84.7	26500.0	11820.0	466.0	360	-79
1214	89.9	26350.0	11840.0	465.0	360	-75
1215	342.4	25311.6	11922.6	597.4	348	-54
1216	256.7	25208.8	11906.3	601.5	349	-60
1217	423.4	25367.0	11869.1	603.5	004	-59
1218	408.1	25367.0	11981.7	597.9	359	-56
1219	452.9	25500.0	11838.2	572.5	0	-63
1220	369.1	25600.8	11842.9	513.0	0	-55
1221	312.1	25117.2	11978.5	506.5	0	-54
1222	199.0	25367.0	11981.7	597.9	180	-54
1223	264.0	25015.0	12056.0	425.0	0	-60
1224	27.0	25064.3	12299.1	222.0	122	-46
1225	299.9	25260.2	12013.5	566.9	0	-57
1226	275.4	25500.0	11940.0	543.1	0	-55
1227	243.2	25700.0	11810.0	507.4	358	-60
1228	200.0	25900.0	11870.0	432.1	0	-55
1229	450.2	25299.7	11752.7	714.1	356	-59
1230	40.2	25700.0	11810.0	507.4	0	-90
1231	46.0	26048.5	11875.1	438.0	0	-82
1232	219.6	25550.0	11862.0	531.8	0	-83
1233	219.5	25793.0	11839.0	465.0	0	-60
1234	327.7	25300.0	11753.0	714.1	301	-54
1235	402.0	25884.0	11583.0	642.0	005	-59
1236	236.8	25884.0	11583.0	642.0	275	-47
1237	446.5	25300.0	11753.0	714.1	275	-45
1238	61.0	26042.0	11595.0	625.0	005	-50
1239	36.0	26042.0	11595.0	625.0	005	-60
1240	28.0	26050.4	11888.9	437.0	002	-82
1241	150.3	26050.0	11888.6	437.0	025	-89
1242	199.6	25994.1	11877.6	430.4	035	-49
1243	169.5	25993.0	11878.8	432.0	002	-54
1244	249.1	26133.3	11860.1	457.1	001	-59
1245	493.0	25487.0	11636.1	764.0	346	-61
1246	545.7	25490.9	11636.2	765.3	037	-61
S-06	107.6	24964.1	11662.6	775.0	0	-45
S-101	314.8	24943.5	11501.1	811.9	0	-45
S-125	404.0	24919.0	11555.2	798.9	0	-47

Table 13- 2: Bronson Slope Diamond Drill Hole Data (cont'd)

S-126	425.3	24919.0	11554.0	798.9	0	-75
S-127	461.9	25194.2	11570.1	804.7	0	-67
S-129	382.6	25502.3	11549.8	804.0	358	-60
S-130	233.2	25502.1	11550.7	804.2	357	-44
*BS0601	31.0	25348.9	11926.4	594.2	3.5	-60
*BS0602	138.6	25391.1	11974.0	598.6	0.5	-70
*BS0603	270.0	25355.7	11929.8	590.4	1.5	-59
*BS0604	122.0	25502.7	11938.7	545.5	0.5	-55
BS0701	466.3	25257.4	11945.1	585.7	212.6	-53.5
BS0702	393.2	25309.0	11988.1	591.2	201.1	-46.5
BS0703	295.7	25697.4	11912.9	457.8	197.1	-62.9
BS0704	360.6	25599.3	11904.4	497.6	263.0	-58.3
BS0705	429.8	25549.8	11931.2	524.3	195.4	59.4
BS0706	274.6	26050.1	11962.9	400.0	268.9	-64.5
BS0708	360.6	25352.5	11615.7	783.9	99.0	-69.5
BS0709	390.9	25399.1	11581.1	793.3	102.3	-63.5
BS0710	365.8	25355.0	11929.2	590.5	207.4	-50.0
BS0711	300.0	25053.8	11610.4	795.4	87.9	-76.0
BS0712	298.7	25747.1	11782.2	518.2	82.7	-69.0

* Holes also reported as 200601 to 200604

** The Northing, Easting, and Azimuth values are based on Grid North which is 025° 12' 22"

14 Sampling Method and Approach

The following **Item 14 – Sampling Method and Approach** has been extracted from **Section 12 – Sampling Method and Approach** within the previous Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical Report can be viewed at www.sedar.com.

Bronson Slope sampling data includes surface core drilling results from SGC in the period of 1988 and 1993 through 1997, 2006 and 2007, and historic drilling by Cominco and Prime Resources in 1986 and 1994. All drill data, excluding the Cominco 1965 drilling, was used to define the current resource estimate completed in early 2008.

Diamond Drilling 2007 and 2006

SGC core diamond drilling in 2007, all of NQ core size, was completed by Blackhaw Drilling of Smithers, BC. The 2006 drilling, all of HQ core size was completed by Phils Drilling and Boart Longyear. The core, for both years, was moved by the drilling contractor via helicopter to the core logging facility at Bronson Creek airstrip where a team consisting of a SGC geologist and technicians logged, including RQD data, and photographed the drill core in detail. It was subsequently marked, split, sampled, bagged, and packed. Technicians split the drill core with a Longyear diamond drill core splitter. The sampling interval averaged 3 metres continuous intersections, which were bagged, labeled and secured, placed in sacks, and then forwarded, in 2006, by aircraft to Acme Laboratories in Vancouver, BC. In 2007 the core was sent to Bob Quinn on the Cassiar Stewart Highway by aircraft and thence by truck (Bandstra Transportation) to the Acme laboratories preparation laboratory in Smithers, BC. The analyses and assays were gold, copper, silver and molybdenum. The Core Handling Procedure (Delong 2006a) was developed prior to the drilling and included a detailed protocol on laying out core, geotechnical logging, sample layout including standard and blank sample insertions, and core logging procedures on descriptive terminology for alteration and lithology, type of structures, mineralization, veins and styles/types, and storage of core. The core, at all time, was under direct supervision of SGC personnel and kept in a secure and locked core logging building.

Diamond Drilling 1988 & 1993 - 1997

SGC diamond drill core from surface, mostly of BQ core size, (1988, 1993 through 1996) and some NQ core size (1997) was completed by Falcon, Boisenvu, Olympic, and JT Thomas Drilling over this period of time. The drill core in 1988 was moved from the respective diamond drill setup by helicopter to the Red Bluff exploration camp where it was logged and split. In 1993 the drill core was taken from the drill sites to the main Johnny Mountain mine site. In the period of 1994 through 1997 the core was moved by the drilling contractor via helicopter to an exploration campsite on the north end of the Bronson Creek airstrip where a team of SGC geologists logged, including RQD data, and photographed the drill core in detail. It was subsequently marked, split, sampled, bagged, and packed. Technicians split the drill core with a Longyear diamond drill core splitter. The sampling interval varied from 1.5 to 4-metre, and averaged 3 metre range, continuous intersections, which were bagged, labeled and secured, placed in sacks, and then forwarded by aircraft to the Rossbacher Laboratories in Burnaby, BC, and where applicable, to Chemex Labs in North Vancouver, BC. The analyses and assays were predominantly gold, copper, silver and molybdenum, and in certain cases, other metals were completed. The above information is given by Yeager (2006).

In 1997 the former Cominco and Prime Resources drill core, acquired by SGC, was re-sampled (quartered where previously assayed) and un-assayed portions were split and assayed at Chemex Labs Ltd.

The surface drilling, logging, and sampling procedures were essentially constant over the continuous five-year drilling period.

15 Sample Preparation, Analyses & Security

The following **Item 15 – Sample Preparation, Analyses & Security** has been extracted from **Section 13 – Sample Preparation, Analyses & Security** within the previous Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical Report can be viewed at www.sedar.com.

15.1 Sample Preparation & Analyses

2007 & 2006

The diamond drill core (2006 and 2007) and sample pulps derived from drill core and the respective standards and duplicates were delivered to Acme Analytical Laboratories Ltd. in Vancouver, BC. The laboratory staff would then assume the chain of command of the samples. Acme Laboratories is an ISO 9001:2000 accredited company and uses accepted and good quality analytical technology and protocol with respect to current industry standards. The samples were recorded, dried, crushed, split with the split portion being ground or pulverized. Standard sample procedure during this period, was pulverization of split core so that 1 kg is crushed to 70% passing 10 mesh with a 250 gram split pulverized to 95% passing 150 mesh sieve size screen. The samples were geochemically analysed for gold (parts per billion), silver and molybdenum (parts per million) and copper (percent).

Copper was analysed in percent by Group 7AR (aqua regia) (HCL-HNO3-H2O) method using 1 gram of sample pulp, diluted to100ml and analysed by ICP-ES. A detection limit of 0.001% (10 ppm) with high precision was achieved.

Gold was analyzed using Group 3B where a 50 gram pulp is used for a lead-collection fire-assay fusion for total sample decomposition followed by digestion of the Ag-dore bead and ICP-ES. A detection limit of 2 ppb is achieved. As part of the check assay protocol in 2006, 12 sample pulps were reanalyzed by the above procedure. In addition, 11 of these above 12 samples were re assayed by a total gold analyses which included the metallic gold, from the remaining rejects. Here large sample weights ranging from 214 to 646 grams, but mostly in the +500 gram range were used. The native gold was separated out and its quantity determined and then the remaining gold was determined and integrated to give a total gold assay. In 2007 59 duplicate samples were used as checks. The scope and magnitude of the re analyses and use of standards and duplicates is detailed in **Item 16.**

Silver and molybdenum were analyzed by the Group 1D method where a 0.25 gram sample split is digested by aqua regia (hydrochloric and nitric acids) and taken to dryness. The residue is dissolved in Hydrochloric acid and analysed by ICP-MS where detection limits of 0.5 ppm silver and 2 ppm molybdenum are achieved.

Quality control measures are discussed in **Item 16**; only the analytical laboratory sampling protocol preparation and analyses procedures are given here. The diamond drill core and other rock samples from the 1993 through 1997 programs were mostly delivered to Rossbacher Laboratories in Burnaby, BC and to Chemex Labs in North Vancouver BC (for duplicate analyses and re split drill core). The Cominco-Prime Group holes from 1986 and 1994 were analysed at Chemex Labs. Quality control analytical work and re assaying of all of the 1988 drill core (which were initially done at the Johnny Mountain laboratory of Skyline Gold) were sent to both Rossbacher Laboratories and Chemex Labs and re-assayed. The laboratory staff would then assume the chain of command of the samples. The laboratories, at this time, were considered to have been using good quality analytical technology and protocol with respect to current industry standards. The samples were recorded, dried, crushed, split with the split portion being ground or pulverized. Standard sample procedure during this period was pulverization of split core (or one quarter core on subsequent re-sampling and analyses) so that, it is believed, at least 85% would pass a –200-mesh sieve size screen. The samples were geochemically analysed for gold (parts per billion), copper, silver and molybdenum (parts per million).

The assay analytical methodology utilized for standard drill core and rock samples is not detailed in the Rossbacher Laboratories reports other than the samples were acid digested and analysed by atomic absorption methods. The standard sample pulp size for analyses for the Chemex samples was 10 grams for gold and 1 gram for silver, copper and molybdenum. The samples pulp sizes for analyses used by Rossbacher Laboratories is reportedly the same to that of Chemex Labs. The Chemex gold analyses were normally by fire assay followed by acid digestion and atomic absorption analysis.

The re-assaying of higher-grade samples sent to Rossbacher Labs Ltd. (Yeager 1997) were routinely analysed for gold, copper, silver and molybdenum. The method for analysis for gold is fire assay extraction using a 20-gram pulp sample then gold determination by atomic absorption spectroscopy. Copper, silver and molybdenum were determined by traditional atomic absorption spectroscopy techniques. Check assays were routinely performed on higher-grade samples. The standard protocol that has been established for check assaying higher-grade samples is as follows:

All samples assayed as containing greater than 3.0 grams gold per tonne were reassayed using a one-assay ton fire assay followed by gravimetric finish. If lab personnel detect high variability in internal lab checks, sections of core assaying greater than 1.0 gram per tonne may have been subjected to re-assay after discussion with SGC personnel.

All samples analyzed as containing greater than 0.6 % copper were reassayed using classic wet chemical analysis.

Samples exhibiting unusual variability in gold grade, as determined by the comparison of check assay with original assay, were assayed a third time using the metallic screen method of averaging the effect of the coarse gold particles.

The author is of the opinion that the techniques and analytical methods used by the SGC external labs, at the time, were “state of the art” and were effective in determining accurately the amounts of gold, copper, silver, and molybdenum in the mineralized drill core.

The Certificates of Analyses from Chemex Labs expressed the gold content in grams per tonne and in PPM (parts per million). The Rossbacher Laboratory expressed gold in PPB (parts per billion) and copper, silver and molybdenum in percent. The 2006 and 2007 assay results from Acme expressed gold in ppb (parts per billion, copper in percent and silver and molybdenum in parts per million. In this Technical Report the format of the metal values used by the writer is the same as those used by SGC and their consultants in technical and engineering report on the Bronson Slope Property. In the reporting of the mineral resource, gold and silver are reported in grams per tonne, copper as a percentage and molybdenum in parts per million. In **Appendix 3**, Bronson Slope Composite Mineralized Intercepts in drill core, all metal values are expressed in parts per million.

15.2 Site Security and Chain of Custody

During the 2007 and 2006 drilling programs the site security was documented and a protocol was developed as part of the Quality Assessment and Quality Control (QA/QC). A sampling/chain of custody was adhered to as outlined in Delong (2006b). The boxes of sealed core were delivered by helicopter to the SGC Logging Facility directly from the drill sites by helicopter under the supervision of the drilling contractor. The lids for the boxes of core were removed carefully in the core facility where it was photographed; the boxes were labelled with aluminium tags showing hole number, box number and to/from measurements. The core was logged and split and sampled in the logging facility. At night when no employees were present, the core was placed in a locked cupboard. After sampling the split core, it is placed in poly bags with the appropriate sample tags. The individual sample bags are sealed with a numbered locking security (NLS) zap strap tie. This NLS tie number is recorded. The samples were then placed in sealed boxes and sent by aircraft and ground transportation to Acme in Vancouver or Smithers, BC.

The pre 2006 site security at the Johnny Mountain and Bronson airstrip exploration camps is not documented; it is assumed that they followed normal mining company security standard of the time, which was strict. Normally drill core and bagged core samples were kept in a secure room or place. The chain of custody for the samples would be from the exploration personnel at the camps to commercial transport personnel and finally to the laboratory personnel in the respective analytical laboratories as documented in **Item 14**.

16 Data Verification

The following **Item 16 – Data Verification** has been extracted from **Section 14 – Data Verification** within the previous Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical Report can be viewed at www.sedar.com.

16.1 Quality Control and Quality Assurance Program 2007 Program

In the order of approximately 1312 core samples were sent for analyses for gold, copper, molybdenum and silver and the QA/QC program of Delong (2006b) was used. SGC instituted a QA/QC program where varying contents of gold, copper and gold, and molybdenum for rock standard pulps, totalling 65 samples, were inserted into the sample chain at different times that were analysed for the four elements named above. All of these samples were analysed at ACME laboratories. In addition SGC completed additional duplicate analyses ("re assays) of pulps for gold, copper, molybdenum and silver of 59 samples; these samples were sent to a second laboratory, ALS Chemex Labs. The primary metals of QA/QC concern is gold and copper because of their economic significance in the Bronson slope deposit. Of secondary interest is molybdenum. Silver contents are relatively low and of minor economic importance. The same internal standards, used for gold and copper, were also used to compare laboratory repeatability and precision for molybdenum and silver.

In addition ACME Labs completed their own QA/QC program on the submitted samples that included approximately 98 repeat analyses of the drill core from the pulps. ACME also inserted a series of five standards including the G-1 blank for all metals, BLK for gold, the OxD57 for gold, the DS7 for Mo and Ag, and the R-3 or R3A for Cu. In total about 430 separate metal analyses was subject to internal ACME standards.

16.1.1 Inserted Standards

SGC used a blank standard for gold and five separate standards for gold and copper that were inserted as rock pulps into the sample chain. These standards although developed primarily for copper and gold, can also be used for molybdenum and silver. A seventh standard, used for molybdenum, was also inserted into the sample chain. CDN Resource laboratories in Delta, BC supplied the rock standards. They include:

- CDN BL-3 of less than 0.01 g/t gold (<10 ppb);
- CG-8 - 0.105 +/- 0.008% Cu and 0.080 +/- 0.012 g/t Au (80 +/-12 ppb);
- CDN CGS - 11 0.683 +/- 0.026% Cu and 0.73 +/-0.068 g/t Au (730 +/- 68 ppb);
- CGS-12 - 0.26 +/- 0.015% Cu and 0.29 +/- 0.04 g/t Au (290 +/- 40 ppb);
- CDN CGS - 13 0.329 +/- 0.018% Cu and 1.01 +/-0.11 g/t Au (1010 +/- 110 ppb)
- CDN CGS - 16 0.112 +/- 0.005% Cu an 0.14+/-0.046 g/t Au (112 +/- 46 ppb)
- CDN-MoS1 - 0.065 Mo +/- 0.008% (650 +/- 80 ppm)

The results of the SGC standards are given in Table 16-1**.**

Table 16-1: Au_Cu_Mo_Ag Results For Inserted Standards

Sample	Au ppb	Cu %	Mo ppm	Ag ppm
CDN -BL-3 (Blank) <0.01 g/t Au (10 ppb)				
430087	<2	0.003	5	<.3
430290	<2	0.003	5	<.3
430386	<2	0.004	5	<.3
430489	<2	0.004	5	<.3
430778	2	0.003	5	<.3
430875	<2	0.003	5	<.3
431185	3	0.004	4	<.3
431383	6	0.003	4	<0.3
696692	4	0.004	4	<0.3
Average		**0.003**	**5**	
CDN CGS - 8 0.105 +/- 0.008% Cu & 0.080 +/-0.012 g/t Au (12 ppb)				
430067	85	0.103	6	0.3
430348	66	0.107	6	0.3
430366	70	0.107	5	0.3
430426	47	0.106	5	<.3
430449	71	0.105	6	0.5
430471	82	0.108	6	<.3
430738	81	0.104	6	<.3
430839	79	0.108	6	<.3
431122	93	0.108	6	0.3
431164	85	0.102	6	0.3
431227	72	0.108	6	<.3
431305	82	0.109	6	<.3
696750	132	0.109	14	0.8
Average	**80**	**0.106**	**6.5**	
CDN CGS - 11 0.683 +/- 0.026% Cu & 0.73 +/-0.068 g/t Au (68 ppb)				
430268	801	0.668	8	2.4
430407	816	0.702	8	2.1
430759	719	0.676	7	1.8
430800	788	0.69	7	2
Average	**781**	**0.684**	**7.5**	**2.1**
CDN CGS - 12 0.265 +/- 0.015% Cu & 0.29 +/-0.04 g/t Au (40 ppb)				
430309	248	0.278	204	3.3
430719	403	0.26	200	2.9
430818	266	0.258	190	3.1
430858	420	0.265	202	2.9
430896	281	0.263	209	3.1

Table 16-1: Au_Cu_Mo_Ag Results For Inserted Standards (cont'd)

430915	298	0.26	216	3.1
Sample	**Au ppb**	**Cu %**	**Mo ppm**	**Ag ppm**
430952	211	0.265	211	3
431008	160	0.261	166	2.8
431069	165	0.27	194	2.7
431205	302	0.265	206	3.1
Average	**274**	**0.263**	**201**	**3.0**
CDN CGS - 13 0.329 +/- 0.018% Cu & 1.01 +/-0.11 g/t Au (110 ppb)				
430971	562	0.316	216	3.4
430989	542	0.327	224	3.4
431029	583	0.319	210	3.3
431142	1164	0.328	219	3.5
Average	**713**	**0.323**	**217**	**3.4**
CDN CGS - 16 0.112 +/- 0.005% Cu & 0.14+/-0.046 g/t Au (46 ppb)				
431050	224	0.117	15	1
431100	138	0.113	16	1
431245	147	0.110	16	0.4
431265	133	0.113	15	0.8
431284	174	0.114	16	1
431326	141	0.110	14	1.1
431345	272	0.103	13	1.1
431364	137	0.105	12	1.1
431407	151	0.109	14	1.1
431426	147	0.101	13	1
431445	197	0.112	15	0.8
431466	165	0.101	15	0.8
431490	136	0.106	14	0.8
696510	154	0.105	13	1.7
696550	86	0.104	13	0.8
696572	85	0.104	14	1.5
696592	112	0.106	13	1.3
696612	141	0.110	14	0.7
696652	165	0.111	14	0.8
696682	131	0.107	15	0.7
696712	109	0.111	16	0.6
696731	87	0.110	15	0.8
696770	117	0.114	13	0.6
Average	**146**	**0.109**	**14**	**0.9**
CDN-MoS1 0.065% Mo (650 ppm) +/- 0.008% (80 ppm)				
696632	6	0.011	666	<0.3

The results for each standard and the respective metal are discussed below:

- The CDN BL-3 standard for gold should and are less than 10 ppb; the values range from <3 to 6 ppb. The copper values average from 0.003 to 0.004% and average 0.004%. The molybdenum values range fro 4-5 ppm and average 5 ppm. The silver values are all less than 0.3 ppm.
- The CGS-8 standard ranges from 47 to 132 ppb Au and averages 80 ppb (0.080 g/t), exactly the recommended value. The Cu content ranges from 0.103 to 0.108% and averages 0.106%, very close to the recommended value of 0.105%. The molybdenum values range from 7 to 8 ppm and average 7.5 ppm. The silver values range from 1.8 to 2.4 ppm and average 2.1 ppm.
- The CGS-11 standard ranges from 719 to 801 ppb and averages 781 ppb (0.781 g/t), somewhat above the recommended value of 683 +/- 68 ppb, The Cu content ranges from 0.676 to 0.702% and averages 0.684%, very close to the recommended value of 0.683%. The molybdenum values are mostly 5 or 6 ppm with one at 14 ppm; the average is 6.5 ppm. The silver values are mostly <3 or 0.3 ppm with one value at 0.5 and a second at 0.8 ppm.
- The CGS-12 standard ranges from 160 to 420 ppb and averages 274 ppb (0.274 g/t), well within the accepted range for this standard. The Cu content ranges from 0.316 to 0.328% and averages 0.323%, very close to the recommended value of 0.329%. The molybdenum values range from 210 to 219 ppm and average is 217 ppm. The silver values range from 3.3 to 3.5 ppm and average 3.4 ppm.
- The CGS-13 standard ranges from 542 to 1164 ppb and averages 713 ppb (0.713 g/t), well below the recommended value of 1010 ppb. It appears that there can be significant variance on the higher value gold standards. The Cu content ranges from 0.258 to 0.278% and averages 0.263%, very close to the recommended value of 0.265%. The molybdenum values range from 166 to 216 ppm and average is 201 ppm. The silver values range from 2.7 to 3.3 ppm and average 3.0 ppm.
- The CGS-16 standard ranges from 87 to 272 ppb and averages 146 ppb (0.146 g/t), near the recommended value of 140 ppb (0.14 g/t). The Cu content ranges from 0.101 to 0.117% and averages 0.109%, very close to the recommended value of 0.112%. The molybdenum values range from 13 to 16 ppm and average is 14 ppm. The silver values range from 0.6 to 1.5 ppm and average 0.9 ppm.
- The MoS1 standard for one value gives 666 ppm compared to the recommended value of 650 +/- 80 ppm.

16.1.2 Duplicate Samples

A total of 59 core sample pulps, originally assayed by ACME, were forwarded to ALS Chemex for reanalyses. The analytical methods and sample weights used are believed to be similar or identical. The duplicates samples analysed by ACME are discussed for each of the four respective metals, gold, copper, molybdenum, and silver below and the results are illustrated in **Table 14-2** and **Figures 14-1 to 14-3**.

For gold duplicate sample results there is obviously some natural variance as given in Table 16-2**;** this is expected at gold contents at the concentrations analysed. Figure 16-1 illustrates the plot of gold in the original samples versus that in the duplicates. It is concluded that the duplicates generally show good repeatability and good correlation to the original samples.

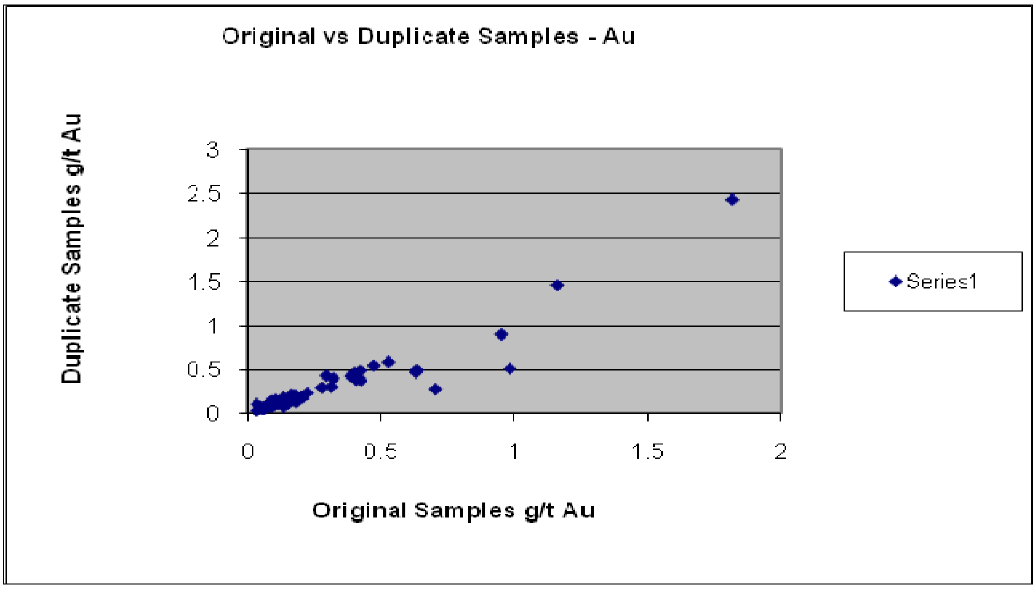


Figure 16-1: Original vs Duplicate Samples – Au

Table 16-2: Au-Cu-Mo-Ag Results - Original vs. Duplicate Samples

Sample	Orig. Au g/t	Dup. Au g/t	Orig. Cu %	Dup. Cu %	Orig. Mo ppm	Dup. Mo ppm	Orig. Ag ppm	Dup. Ag ppm
430058	0.192	0.201	0.143	0.137	253	261	0.8	1.4
430076	0.108	0.132	0.123	0.126	236	283	1.1	1.2
430099	0.133	0.124	0.108	0.128	95	111	1	1.1
430257	0.078	0.087	0.058	0.069	96	101	0.6	0.8
430280	0.124	0.112	0.052	0.060	94	100	1	0.9
430300	0.074	0.135	0.084	0.099	148	160	1.8	1.6
430321	0.075	0.062	0.064	0.067	109	88	1.1	1.4
430339	0.035	0.035	0.038	0.043	77	108	0.5	0.5
430356	0.043	0.037	0.055	0.006	1	14	<0.2	0.9
430377	0.468	0.400	0.271	0.252	87	54	1.1	1.2
430387	0.499	0.633	0.238	0.210	6	5	1.1	1
430397	0.549	0.474	0.268	0.216	24	22	1	1.1
430418	2.432	1.815	0.152	0.146	4	4	14.1	16.4
430438	0.491	0.422	0.205	0.202	147	157	0.9	0.9
430459	0.428	0.389	0.093	0.098	44	32	2.4	2.6
430479	0.112	0.111	0.066	0.067	112	173	1.2	1
430497	0.187	0.203	0.060	0.066	69	71	1.4	0.7
430710	1.461	1.16	0.650	0.684	175	157	3.3	3
430731	0.402	0.321	0.188	0.208	146	169	1.1	1.1
430748	0.393	0.409	0.246	0.249	114	109	1.6	1.3
430769	0.149	0.139	0.147	0.148	58	81	1.1	1
430788	0.065	0.072	0.096	0.116	44	49	0.9	0.8
430811	0.902	0.950	0.517	0.568	45	28	7.8	7
430828	0.590	0.527	0.303	0.291	26	27	2.9	3.1

Table 16-2: Au-Cu-Mo-Ag Results - Original vs. Duplicate Samples (cont'd)

430850	0.381	0.424	0.150	0.158	147	170	2.5	2
430866	0.165	0.126	0.100	0.074	50	39	0.6	1.1
430885	0.215	0.165	0.094	0.098	14	19	12.1	16.8
430906	0.148	0.089	0.057	0.048	47	69	1.9	2.5
430925	0.052	0.054	0.050	0.048	40	70	1.7	1.7
430941	0.384	0.407	0.160	0.164	60	59	1	0.5
430961	0.280	0.705	0.224	0.458	43	37	6.9	2.2
430979	0.473	0.630	0.268	0.270	57	63	1.4	1.1
430999	0.121	0.127	0.066	0.068	271	303	1.5	1.5
431018	0.099	0.099	0.109	0.116	62	63	0.7	0.6
431038	0.044	0.040	0.058	0.057	43	46	0.4	0.4
431062	0.106	0.096	0.159	0.156	50	30	1.3	1.5
431079	0.131	0.115	0.105	0.101	81	69	0.8	1.1
431354	0.433	0.295	0.293	0.033	2	1	2.9	2.9
431375	0.190	0.135	0.020	0.021	2	<1	0.4	<0.3
431395	0.513	0.983	0.041	0.045	6	3	3.9	3.4
431417	0.236	0.224	0.028	0.027	3	4	0.5	0.6
431434	0.137	0.111	0.004	0.004	2	5	0.2	<0.3
431455	0.138	0.134	0.018	0.018	2	5	1.2	1.3
431479	0.138	0.182	0.003	0.004	2	2	0.6	0.3
431497	0.088	0.091	0.005	0.006	1	<1	0.2	<0.3
696524	0.111	0.074	0.007	0.008	1	<1	0.8	0.8
696542	0.117	0.153	0.011	0.012	2	1	0.5	0.3
696561	0.128	0.145	0.014	0.016	1	<1	1.5	1.2
696581	0.297	0.28	0.037	0.037	8	4	1.4	1.7
696620	0.207	0.178	0.132	0.150	60	92	1.9	2.1
696640	0.132	0.123	0.094	0.092	114	140	3.6	3.6
696660	0.055	0.065	0.058	0.068	93	124	1.5	1
696680	0.087	0.051	0.061	0.075	390	546	1.9	1.4
696689	0.112	0.036	0.120	0.124	70	71	1.9	1.9
696702	0.162	0.104	0.077	0.072	161	176	0.9	0.9
696722	0.128	0.118	0.145	0.126	30	45	1	1
696741	0.17	0.13	0.121	0.118	52	55	1.8	2
696762	0.051	0.06	0.059	0.072	18	59	1.8	1.3
696781	0.306	0.312	0.201	0.216	142	165	3.2	3.7

For copper duplicate sample results there is obviously some natural variance as given in Table 16-2. Figure 16-2 illustrates the plot of copper in the original samples versus that in the duplicates.

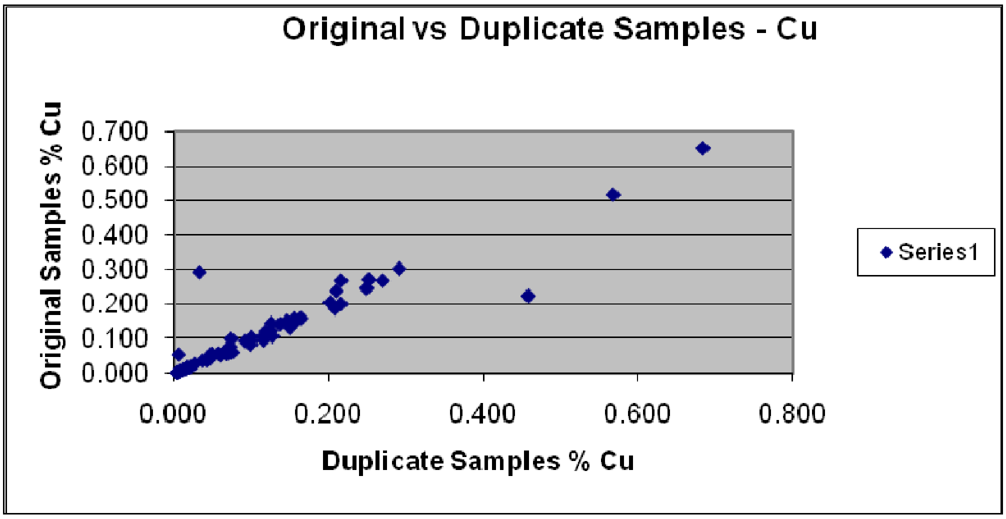


Figure 16-2: Original Vs Duplicate - Cu

It is concluded that the duplicates generally show good repeatability and good correlation to the original samples. One sample, 430356, appears to be out an order of magnitude, 0.055% Cu in the original versus 0.006% in the duplicate; an analytical transcription error is suspected for either the original or the duplicate and this sampled should be re assayed.

For molybdenum duplicate sample results there is obviously some natural variance as given in Table 16-2. Figure 16-3 illustrates the plot of molybdenum in the original samples versus that in the duplicates. It is concluded that the duplicates generally show good repeatability and good correlation to the original samples.

For silver duplicate sample results there is obviously some natural variance as given in Table 16-2. Figure 16-4illustrates the plot of silver in the original samples versus that in the duplicates. It is concluded that the duplicates generally show good repeatability and good correlation to the original samples although the variance is more than for the other metal analysed. There is variance at the lower values close to the detection limit for silver and this is considered to be an analytical issue. Only values to 4 ppm are illustrated on Figure 16-4**.**

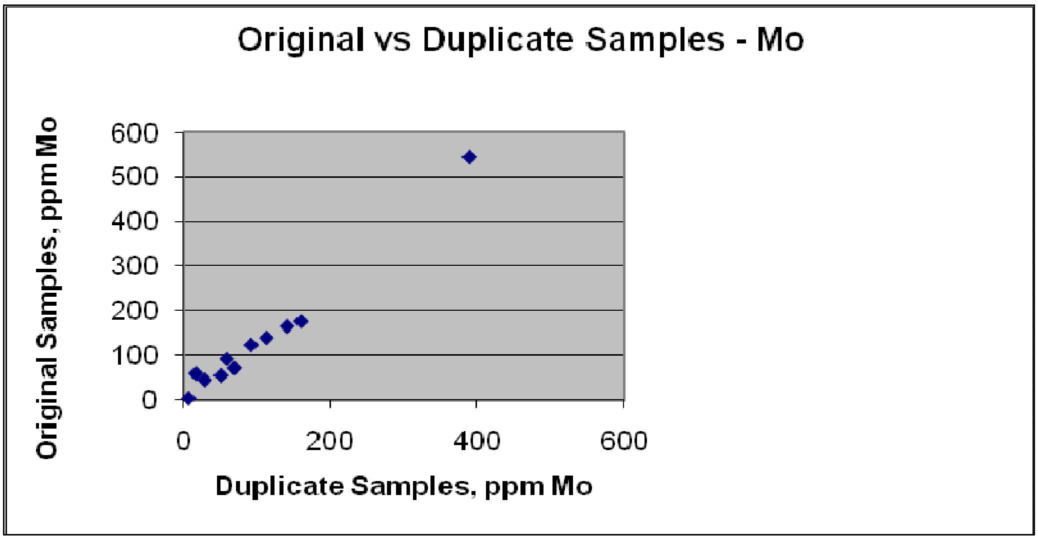


Figure 16-3: Original vs Duplicate - Mo

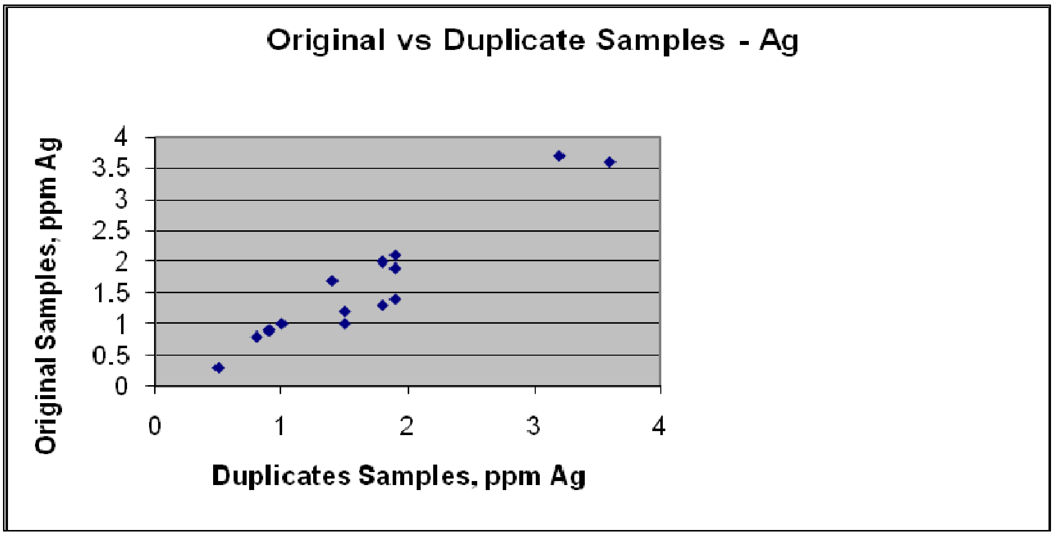


Figure 16-4: Original Vs Duplicate Samples - Ag

16.1.3 ACME Check Analysis For Cu-Au-Mo-Ag Analyses

ACME Labs completed approximately 98 repeat analyses from both the pulps and the rejects for each of the four metals. These samples are similar to duplicates except the original sample results were known to the lab. Again, on review of the metal values for these check or repeat analysis, there is good repeatability and generally low variance. Generally copper, molybdenum and silver variance is low whereas the gold variance can be higher on certain higher value samples that exceed 500 ppb; this is thought to be a natural variance.

16.1.4 ACME Internal Standards

ACME inserted a series of five standards including the G-1 for gold, copper, molybdenum and silver, the BLK for gold, the OxD57 for gold, the DS57 for molybdenum and silver, the R-3 or R3A for copper. In total about 430 separate metal standard analyses were completed as part of the internal ACME QA/QC. The ACME internal QA/QC appears to be extremely thorough and comprehensive. The standards as discussed below are fully documented by ACME.

The G-1 blank is <2 ppb Au, <0.001% Cu, <1 ppm Mo, and <0.3 ppm Ag. The BLK standard is <2 ppb Au. The OXD57 standard is 413 +/- 5 ppb Au at the 95% confidence level. The R-3a standard is 0.811 +/- 0.019 % copper. The DS7 standard for Mo and Ag is 20.92 +/- 1.69 ppm and 0.9 +/- 0.1 ppm, respectively. Variability for these internal Acme standards was extremely low and mostly within the accepted ranges published by ACME.

16.2 Quality Control and Quality Assurance Program 2006 Program

The results of the 2006 QA/QC program are summarized below and discussed in substantial detail in Burgoyne and Giroux (2007) that is found on <www.sedar.com>.

In the order of 165 core samples were sent for analyses for gold, copper, molybdenum and silver and the QA/QC program of Delong (2006b) was used.

SGC instituted a QA/QC program where varying contents of copper and gold for rock standard pulps, totalling 13 samples, were inserted into the sample chain at different times that were analysed for the four elements named above. . SGC used three separate standards that were inserted as rock pulps into the sample chain. These standards were developed primarily for copper and gold but can also be used for molybdenum and silver. CDN Resource laboratories in Delta, BC supplied the rock standards. They included a CG-8 where the recommended copper content is 0.105 +/- 0.008% and the gold content is 0.080 +/- 0.012 g/t, a CGS-12 where the copper content is 0.26 +/- 0.015% and the gold content is 0.29 +/- 0.04 g/t. The molybdenum and silver content of CG-8 is approximately 5 and <0.3 ppm, respectively; CG-12 is approximately 200 and 3 ppm for molybdenum and silver, respectively. The blank standards were less than 2 ppb gold, approximately equal or less than 0.003% copper, and equal or less than 5 ppm molybdenum and <0.4 ppm silver.

In addition SGC completed and additional reanalyses (“re assays) of pulps for gold and copper of 12 samples that were identified as anomalously low or high from plotting the copper content versus the gold content. Those samples have abnormally high gold to copper ratios or alternatively had abnormally high copper to gold ratios. Also, 11 out of these 12 samples were reanalysed, from the rejects, for gold only by the “metallics assay” method. The same standards used for gold and copper was used to compare laboratory repeatability and precision for molybdenum and silver.

Acme Analytical laboratories also completed a reanalysis (RE) of the sample pulps or reanalysis (REE) of the sample rejects of 13 samples for all metals. Acme also inserted standards, including blanks, into their sample chain to measure repeatability and precision for all metals.

16.2.1 Copper Analyses

The assay results for 13 SGC copper standards inserted into the sample chain over the drilling program show good repeatability, low variability, and good correlation to the accepted standard values.

SGC also completed 10 check assays from the sample pulps and along with the Acme check assays that included re assays of 7 assays of the sample pulps and 6 assays of splits from the sample rejects.

The original SGC copper assay compared to the re assay values illustrate low variability, good correlation and good repeatability. This same trend is also seen in the Acme check samples.

Acme also inserted their R-2a standard of 0.562 +/- 0.0016 % copper, their SF-3 standard (0.771 +/- 0.022 %) and their G-1 blank for a total of 16 times. Variability for these internal Acme standards was extremely low and within the accepted ranges. The G-1 blank standard was used to check for contamination between crushing and pulverizing of the samples. The R-2a standard (1 sample) assayed 0.564% Cu, the SF-3 (8 samples) standard varied from 0.756 to 0.786% and the G-1 standard (7 samples) varied from <0.001 to 0.001% copper. All of the samples are within the accepted two standard deviations.

16.2.2 Gold Analyses

The assay results for 13 SGC gold standards inserted into the sample chain over the drilling program show good repeatability, low variability, and good correlation to the accepted standard values.

SGC also completed 12 check assays from the sample pulps. Acme check assays included re assays of 7 assays of the sample pulps and 6 assays of splits from the sample rejects. SGC additionally had 11 of the original 12 samples re assayed for gold using the total gold method including the metallics component of the samples. Here the rejects were used with sample weights varying from 214 to 635 grams for gold analyses. The 12 check assays done by SGC and the 13 check assays done by Acme were done on 50-gram sample weights and analysed by fire assay fusion and ICP-ES. This is the protocol use for all gold analyses completed on drill core.

When a comparison of the original SGC samples is made to the corresponding “Re Assay” value checks, there is generally limited variance and good correlation of values.

Acme also inserted their OxF41 internal standard of 0.815 +/-0.011 g/t gold for a total of 9 times. Variability for these internal Acme standards was extremely low and within the accepted range. The G-1 blank standard was used to check for contamination between crushing and pulverizing of the samples. The OxF41 standard (9 sample) varied from 0.794 to 0.818 g/t gold and the G-1 standard (4 samples) varied from <0.002 to 0.005 g/t gold, again within the accepted range. All of the sample values are within the accepted two standard deviations.

16.2.3 Molybdenum & Silver Analyses

The analytical results for molybdenum for the 13 SGC standards showed consistent repeatability and very low variance attesting to good assay techniques.

SGC did not complete any molybdenum and silver check assays; however, Acme completed check assays including 7 re assays of the sample pulps and 6 re assays of splits from the sample rejects. The results generally define low variance and good repeatability.

Acme also inserted their DS7 internal standard of 20.9 +/- 1.7 ppm molybdenum and 0.9 +/- 0.1 ppm silver for a total of 9 times. Variability for these internal Acme standards was extremely low and within the accepted range. The G-1 blank standard was used to check for contamination between crushing and pulverizing of the samples. The DS7 standard (9 samples) varied from 20 to 21 ppm molybdenum, and 0.6 to 1.1 ppm silver. The G-1 standard (8 samples) varied from <1 to 1 ppm molybdenum, and <0.3 to 0.3 ppm silver, again within the accepted range. All of the sample values are within the accepted two standard deviations.

16.3 Quality Control and Quality Assurance Pre 2006 Programs

The results of the pre 2006 QA/QC are summarized below and discussed in substantial detail in Burgoyne and Giroux (2007) which is found on <www.sedar.com>.

There is no record that any regular QC/QA or quality control/quality assurance program, as is common today, was in place with respect to sampling and subsequent assaying and analyses. The period in which the exploration drilling and sampling was done at Bronson Slope was before formal QA/QC became established in the mineral exploration industry and the widespread use of duplicates, blanks and internal standards. It does appear, however, that there was substantial re-sampling and re-analyses of drill core and some inter laboratory checks. The analytical laboratories used by SGC, Chemex Labs Ltd. and Rossbacher Laboratories Ltd., of course, maintained a series of internal standards and checks.

The drill core assay quality assurance program through to the end of 1997 consisted of: (i) re-assay of high grade pulps as previously described in Item 13, (ii) the re-splitting and re-sampling programs previously described in Item 13 and (iii) monitoring of the comparison between average drill core assay grades and grades from metallurgical testing of drill core composites as reported in Item 16. While this has been an adequate program to ensure that the average metal values of the deposit are adequately represented, it has done little to provide a detailed check of the reliability of individual assays.

Raymond (1997) in his geostatistical analysis of the Bronson Slope assay data and resource estimate suggested a program comprising: (i) plotting copper versus gold scatter plots for each hole as the assay results are received to identify anomalous gold values and (ii) instituting a cross lab assay check whereby ten percent of all samples, both high and low gold grade, are assayed at another lab. Giroux (1997) suggested a similar practice with the additional check of resubmitting the pulps to the original lab after random renumbering. It had been decided in 1997 to initiate the program, using five percent of the sample base, and then evaluate the need to incur the additional expense of testing the full ten percent. The ratios of gold to copper grades were also calculated for all drilled intervals and anomalous intervals were reviewed and if necessary, re assayed.

Company geologists decided in 1997, in addition to the above measures, to include with all core shipments, core samples with known low gold grades as blank check standards. The core for this program was split from drill hole 901 drilled in 1989. The hole was drilled for stratigraphic purposes in un-mineralized Jurassic meta sedimentary rocks. The object of this measure is to detect cases of gold contamination of samples.

It was decided in 1994 that the assay lab quality control problems that had been evident at the Johnny Mountain lab in 1988 could have been too great to make it possible to receive accurate gold assays in the < 1.0-gram/tonne range required for the Bronson Slope deposit. Accordingly, in 1994, hole 944 (RB-1) was re-split (quartered) and prepared and assayed at Rossbacher Labs Ltd. in Burnaby, B.C. The results of this re-sampling test indicated the need to re-assay all of the 1988 drilling. The remainder of the 1988 holes were re-sampled in 1995 and the samples of the quartered core were again assayed at Rossbacher Labs. The core remaining from the re-sampling of hole 944 in 1994 was re-sampled and assayed again in 1995. The assay values used in the present block model resource estimates for the 1988 drilling are from the 1994 re-assay of hole 944 (on the assumption that half split core would be more representative than quarter split core) and predominantly from the 1995 re-assay of the remainder of the holes.

Drill hole 1231 and 1232, drilled in the 1996 phase II program, contained anomalously high gold values. All of 1231 and the first part of 1232 were re-sampled in 1997 to determine the repeatability of the 1996 results. It was apparent that gold contamination of the 1996 samples had occurred. The holes were re-sampled and re assayed.

Burgoyne and Giroux (2007) should be reviewed for details on:

- Results of Re-Sampling Drill Core from 1988, 1993, and 1994 drilling periods.
- Re-Assaying of Higher Grade Samples
- High Grade Contamination of Specific Sample Batches and how this was dealt with and corrected in 1994, 1995, 1996 and 1997.

16.4 Recommendations for Future Work and Drilling

- Gold/copper ratios should continue to be plotted for all drill holes. Intervals with anomalously high or low ratios should be re assayed. Any unresolved discrepancies in the comparison of re-assay results to originals should be resolved by assay of the lab crusher rejects.
- All samples in excess of 2 g/t gold selected for gold re-assay should be assayed by the “metallics assay” method that is screened for metallics then both fractions assayed by classical fire assay.
- In future, all core-sampling programs should continue with the QA/QC program initiated in 2006 and improved upon in 2007.

16.5 Technical Review by Authors

The 2007 and 2006 drilling programs contained a good QA/QC protocol that has established good repeatability and relatively low variance for the sample chains. This QA/QC program has demonstrated no laboratory contamination and good accuracy.

Although no formal QA/QC program was present in the pre 2006 drilling programs at Bronson Slope, a significant amount of gold reanalyses at Rossbacher Laboratories was completed commencing in 1994 and 1995 when all of the 18 holes completed in 1988 (holes 944 to 949, and 954 to 964) that were initially analysed at the SGC Johnny Mountain Laboratory. Also, Rossbacher Laboratories reanalyzed hole 1198, initially analyzed by Chemex in 1993, in 1995. Three further holes from the 1994 program including holes 1211, 1212, and 1216, initially analysed by Rossbacher Laboratories, were subsequently analyzed by Chemex Labs in 1997. These latter four holes served to help evaluate the inter laboratory comparison accuracy for gold analyses. It is concluded that, with this reanalyses of four drill holes, the Rossbacher laboratories analytical accuracy appears to be satisfactory, although in two of the four holes gold values are somewhat higher than the Chemex labs and the reasons for this are discussed in Burgoyne and Giroux (2007).

There is no reason to doubt the quality or veracity of these data. All of the exploration work conducted on the Bronson Slope property from 1988 through 2007 was performed by competent, professionally qualified persons.

The writer (Burgoyne) did not collect any samples for analyses during the course of the recent field examination. Enough drilling and sampling has been done in the period of 1988 through 2007 to provide a reasonable assessment of average grades and, in the view of the writer, the collection of a few surface samples for analyses would not provide any meaningful results.

Due diligence studies by the writers include those completed during the review of the data on this property during December 2007 and January through March 2008, the October 2007 site visit, the site visits by one of the writers (Burgoyne) in June, September, 2006 and October 2007, the early historical review of data in 1992 and 1993 (Burgoyne 1992, Burgoyne, 1993a and Burgoyne 1993b), and examination of 1988 drill core in August 1993. This evaluation work in 2006 and 2007 is summarized as:

- Property site visits including review of geology, mineralization and site setting.
- An examination of drill core at the Bronson Airstrip from the 1988 and 1994 through 1997, and 2006 and 2007 programs.
- The location of drill hole locations and old mine workings.
- A detailed review of a large database of technical reports and many maps and sections dealing with the property.
- A review of the geologic model with respect to controls on mineralization at the Bronson Slope deposit.
- Auditing and checking of calculations leading to Mineral Resource estimates, a review of the drill hole and assay database and resource methodology parameters, and evaluation of mineralized cross-sections.
- Detailed review of the QA/QC procedures.
- A detailed review of all mine development studies undertaken in 2007 and 2008.

17 Adjacent Properties

The following Item 17 – Adjacent Properties has been extracted from Section 15 – Adjacent Properties within the previous Technical Report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical Report can be viewed at www.sedar.com.

There are nearly four hundred mineral occurrences in the Iskut River are of NTS 104B. Only those major deposits that are within several kilometres of Bronson Slope and/or where production is recorded are described here.

17.1 Iskut Deposit

Newcastle Minerals Ltd. owns the Snip North property, located 2.5 km northwest of Bronson Slope deposit. . The limited drilling and preliminary geological modeling to date has defined the Iskut gold-copper-molybdenum deposit. Based on eight 2007 and 2006 drill holes and four historical drill holes the following definition on the geometry and grade of the deposit was reported in the February 22, 2008 Newcastle Minerals Press Release and is detailed in Burgoyne (2008). Using dimensions of 500 and 600 metres in strike length, a width of 225 metres and a depth of 175 metres along with a specific gravity of 2.90 yields a potential quantity of 57.1 to 68.5 million tonnes. The grade varies from 0.3 to 0.6g/t gold, 0.09 to 0.17% copper and 0.003 to 0.023% molybdenum. ***This estimate of quantity and grade is conceptual in nature and there has been insufficient exploration and drilling to define a mineral resource and that it is uncertain if further exploration will result in the target being delineated as a mineral resource***

17.2 Eskay Creek Deposit

At the famous Eskay Creek (Minfile 104B 008), owned by Barrick, production is currently in progress. To 2002 the mine has produced 68,500 kg of gold and 3,100,000 kg of silver from a precious metal volcanogenic-type deposit. The 21-zone mineralization of the Eskay Creek Mine is unusual and the most important of over 30 distinct mineralized zones at this mine, which lies 40 km east of the Property. Eskay Creek is Canada's highest grade gold mine and world's fifth largest silver producer. Most of the ore lies within stratiform lenses of precious metal rich sulphides and sulfosalts overlying rhyolite domes in a volcanogenic massive sulphide setting. High-grade footwall veins were the focus of exploration for 50 years leading up to the discovery of the main zone. Production and reserves total 4.0 m ounces gold and 153 million ounces silver at grades of 1.4 oz/T Au and 63 oz/T Ag. These reserves and resources may not be NI 43-101 compliant.

17.3 Snip Deposit

The adjacent Snip Mine (Minfile 104B 250), located within 500m of the north boundary of the Bronson Slope property was operated by Cominco Limited, and Prime Resources Group and Homestake Canada Inc. From 1991 to 1999, the Snip Mine produced 32,093 kilograms of gold, 12,183 kilograms of silver, and 249,000 kilograms of copper from about 1,267,642 million tonnes of ore. The Twin vein zone is a 0.5 to 15 meter wide sheared quartz-carbonate-sulphide vein that cuts through a massively bedded feldspathic greywacke-siltstone sequence. The mineralization occupies a 120° structure with dips varying from 30 to 90 degrees southwest. A post-mineralization dyke divides the vein into two parts for most of its length. The dip length of the deposit is about 500m and has been traced over a strike length of 1000m.

17.4 Johnny Mountain

The closed Johnny Mountain Gold Mine (Stonehouse gold deposit - Minfile 104B 107) of Skyline Gold, located 4.5 km south-southeast of Bronson Slope, is optioned out to Spirit Bear Minerals Ltd. Recorded production from 1987-1993 totals 2815.4 kilograms of gold from 227,247 tonnes. This is a structurally disrupted mesothermal gold-bearing quartz vein deposit. Mineralization includes pyrite, chalcopyrite with some sphalerite, galena and minor pyrrhotite within a number of sub parallel sulphide-K-feldspar-quartz veins and stock work systems occurring along a series of northeast-trending structures in close proximity to plagioclase porphyry dykes.

The writer is unable to verify the above information, except that on Iskut Deposit, and the information is not necessarily indicative of the mineralization on the Bronson Slope property.

18 Mineral Processing and Metallurgical Testing

18.1 Metallurgical Testing

18.1.1 Introduction

SGC has performed a series of metallurgical studies on Bronson Slope drill core samples from 1994 to 1997 as part of engineering scoping and process flow sheet development studies.

- In 1994 Lakefield Research was commissioned by SGC to conduct a preliminary metallurgical testing of the Bronson Slope ore. The purpose of the test was to determine recoverability of copper gold minerals using a conventional flotation method.
- In January 1995 SGC commissioned Process Research Associates (PRA, Vancouver, BC) to conduct additional metallurgical test work to further define the expected metallurgical results.
- In 1996 further metallurgical testing was commissioned by PRA and Beattie Consulting Ltd. The program was designed to assess the preliminary ore characterisation, copper and molybdenum flotation and acid base accounting test work.
- In 1997 PRA was retained by SGC to undertake an expanded metallurgical test work program. The objective was to obtain design criteria as part of a feasibility study.

From then onwards no further metallurgical work has been carried out until recently in 2007 some testing has been conducted on some drill core samples of high wall material, which hasn't been tested before.

The report entitled "Metallurgical Study on the Bronson Slope Samples" by Process Research Associates (PRA), 1997 forms the basis for metallurgical comments within this report.

18.1.2 Metallurgy Summary

Seven drill core composite samples of four main mineralization types have been prepared by PRA for the metallurgical program in 1997 with the objective of obtaining design criteria for a feasibility study. The test composite samples were categorized as Upper Sediment, Upper Sediment Oxidized, Porphyry, Quartz Magnetite, Average, Starter Pit and High Grade. The flowsheet development was primarily conducted based on the "Average" composite sample.

Table 18-1 and Table 18-2 summarize the 1997 metallurgical head grades and a comparison of metallurgical assaying grades, the later information is extracted from the technical report entitled "Mineral resource estimated – Bronson Slope deposit, 2007, Burgoyne geological Inc."

Table 18-1: Head Assay of Composites

Composites		Average	US	USO	PPY	QM	SP	HG
		Average Blend	Upper Sediment	Upper Sediment Oxidized	Porphyry	Quartz Magnetite	Starter Pit	High Grade
Au	g/t	0.472	0.446	0.776	0.369	0.518	0.517	0.724
Ag	g/t	2.44	2.42	3.18	2.66	2.79	2.74	3.72
Cu	%	0.192	0.206	0.252	0.133	0.181	0.227	0.358
Mo	%	0.007	0.009	0.014	0.008	0.006	0.007	0.009
Fe	%	6.43	4.76	4.03	5.66	7.48	7.06	7.11

Table 18-2: Comparison of Assaying Grades of Metallurgical Test Samples

Metallurgical Assaying									
Composite		BC	US	USO	PPY	QM	SP	HG	Average of all composite
Number of test		28	3	2	2	4	2	2	
Average metallurgical calculated head grade	Au; g/t	0.472	0.446	0.776	0.369	0.518	0.517	0.724	0.546
	Ag; g/t	2.44	2.42	3.18	2.66	2.79	2.74	3.72	2.85
	Cu; %	0.192	0.206	0.252	0.133	0.181	0.227	0.358	0.221
	Mo; %	0.007	0.009	0.014	0.008	0.006	0.007	0.009	0.009

Core Sample Assaying									
Composite		BC	US	USO	PPY	QM	SP	HG	Average of all composite
Number of Core samples		1448	462	195	199	817	115	145	
Unweighted average assay grade	Au; g/t	0.50	0.49	0.90	0.40	0.54	0.58	0.66	0.581
	Ag; g/t	2.60	2.70	3.30	2.50	2.60	2.90	3.10	2.81
	Cu; %	0.180	0.190	0.230	0.140	0.180	0.190	0.240	0.193
	Mo; %	0.005	0.008	0.013	0.006	0.003	0.004	0.007	0.007

Metallurgical Assaying									
Composite		BC	US	USO	PPY	QM	SP	HG	Average of all composite
Number of Core samples		1448	462	195	199	817	115	145	
Weighted average assay grade	Au; g/t	0.470	0.470	0.990	0.400	0.490	0.540	0.630	0.570
	Ag; g/t	2.30	2.20	3.30	2.20	2.40	2.30	3.10	2.54
	Cu; %	0.160	0.180	0.230	0.130	0.160	0.170	0.230	0.180
	Mo; %	0.004	0.007	0.013	0.006	0.003	0.003	0.007	0.006

(BC = Bulk composite 350m pit, 1996. US = Upper sediment rock, 1997. USO = Oxidised upper sediment rock, 1997. PPY = Porphyry, 1997. SP = Starter Pit, 1997, HG = High grade, 1997)

Some coarse gold effect was observed in the average composite sample since gold grade of this sample varied from 0.37g/t to 0.86g/t. Gravity recovered gold of 25.5% with a gold content of 23.8g/t gold was recovered using a gravity Knelson concentrator. Other composite samples were also tested. The gold recovery varies from 18.7% for the upper sediment to 38% for the quartz magnetite. The study showed that pre-concentration with a gravity separator should be included in the process to recover the coarse gold that will not be recovered by the flotation process.

The Bond mill work index of the composites ranged from11.5 kWh/t to 13.3 kWh/tonne. The specific gravity ranged from 2.72t/m^3 to 2.83t/m^3.

The projected copper and gold recoveries of the bulk copper flotation are as follows:

- Average composite – 84% Au, 87% Cu, 61% Ag, 46% Mo at 27% copper concentrate.
- Upper Sediment – 82% Au, 89% Cu, 68% Ag, 58% Mo at 24% copper concentrate.
- Upper Sediment Oxidised – 88% Au, 82% Cu, 50% Ag, 52% Mo at 22.8% copper concentrate.
- Porphyry – 83% Au, 83% Cu, 67% Ag, 53% Mo at 20% copper concentrate.
- Quartz Magnetite – 88% Au, 87% Cu, 66% Ag, 33% Mo at 19% copper concentrate.
- Starter Pit – 87% Au, 88% Cu, 66% Ag, 43% Mo at 24% copper concentrate.

- High Grade – 86% Au, 90% Cu, 68% Ag, 53% Mo at 22% copper concentrate.

The recovery of gold is a combined gravity and flotation recovery.

The locked cycle flotation of the average composite showed that 86.8% recovery at 27% copper grade can be produced in a rougher copper bulk concentrate. The grind size for the rougher flotation was established at 80% passing 108 micron. The test results and mineralogical analyses indicated that at a grind size of 80% passing 30 micron will be sufficient for the copper and molybdenum separation as at this size the mineral liberation was practically complete.

The flowsheet developed in the study comprised a Knelson concentrator, rougher scavenger copper flotation, regrind, cleaner copper flotation, molybdenum rougher flotation, regrind and molybdenum cleaner flotation.

The magnetite recovery circuit is comprised of rougher, regrind and cleaner magnetite separators.

The reagents regime adopted in the study has been listed in Table 18-3.

Table 18-3: Projected Reagents Regime for Flotation

Reagent	g/t	Point of Addition
Potassium Amyl Xanthate (PAX)	8	Bulk Rougher 1 and 2
Aerofloat 208	8	Bulk Rougher 1 and 2; Bulk Regrind Mill
Aero 5100	2	Bulk Cleaner-Scavenger
Lime	60	Bulk Regrind Mill and Cleaners 1, 2 and 3
Sodium Cyanide	14	Bulk Regrind Mill and Cleaners 1, 2 and 3
MIBC	32	Bulk Rougher 1 and 2; Bulk Cleaner 1, 2 and 3
Sodium hydrosulphide ($NaHS.xH_2O$)	120	Mo Rougher, Scavenger and Cleaner
Fuel Oil	3	Mo Rougher, Scavenger and Cleaner
Dowfroth 250	1	Mo Rougher, Scavenger and Cleaner
Nitrogen gas (N_2)	n/a	Mo Rougher, Scavenger and Cleaner

(n/a = not available)

The rougher magnetic separation test reported a weight recovery of 7.2% with a concentrate grade of 53.7% Fe. Regrinding the rougher magnetic concentrate to 98% minus 37 micron with an additional cleaning stage will lower the product yield to 4.6% but will enhance the magnetic content to 98.3% in the cleaner magnetite concentrate, with a specific gravity of 5.08 and iron grade of 71.4% Fe.

Cyanidation was not feasible for the copper bulk concentrate as high sodium cyanide consumption is required for the high copper content in the concentrate. Low recovery was obtained in the cleaner tails sample.

The key observations from the 1997 metallurgical test work specific to the average composite are as follows:

- A marketable copper concentrate can be produced from the Bronson Slope mineralization with a conventional rougher-scavenger flotation and followed by regrinding and three cleaning stages.
- A primary grind size of 80% passing 108 micron is sufficient for the copper rougher scavenger recovery. For the cleaning stages a regrind is required for an effective separation.

- For copper and molybdenum separation a regrind size of 80% passing 30 micron is sufficient for effective copper molybdenum separation because at this size full liberation will be achieved.
- Gravity concentration is required to recover the coarse gold to minimise the gold lost that is not recovered from the flotation.
- Saleable magnetite concentrate can be produced but further test work is required for reproducibility and validation of testing results.
- Marketable molybdenum concentrate can possibly be produced but further test work is required for verification.
- Bronson Slope mineralizations contain potentially acid generating materials, which will require further study and planning to manage this environmentally sensitive issue.

18.1.3 Metallurgical Test Work Details

18.1.3.1 Preliminary Mineralization Characteristic

Seven composite samples were prepared for the program. The samples are categorized as follows;

- Upper Sediment (US)
- Upper Sediment Oxidised (USO)
- Porphyry (PPY)
- Quartz Magnetite (QM)
- Average (Which was comprised of the above four materials)
- Starter Pit material
- High Grade material.

The above composites were prepared by PRA using a standard sampling preparation method. Various tests were conducted on the seven composites. A representative matrix of the testing is listed in Table 18-4 below. Mineralogy analyses indicated that the principal copper mineral in the materials was chalcopyrite. Pyrite was the other major sulphide mineral occurring with the copper sulphide, iron oxides, sphalerite, molybdenite and silicates gangue material were the remaining constituents. The composite head assays of the metallurgical work are listed in the Table 18-5.

Table 18-4: Test Program Matrix

Tests	Composite Sample						
	Average	US	USO	PPY	QM	SP	HG
Head Assay	x	x	x	X	x	x	x
Specific Gravity	x	x	x	X	x		
Ball Mill Grindability	x	x	x	X	x		
Batch Flotation	x	x	x	X	x	x	x
Locked Cycle Flotation	x	x	x	X	x	x	x
Magnetic Separation	x						
Cyanidation	x						
Thickener Sizing	x						
Acid base Accounting	x						

Table 18-5: Characteristic of Bronson Slope Composites

Composites		Average	US	USO	PPY	QM	SP	HG
		Average Blend	Upper Sediment	Upper Sediment Oxidized	Porphyry	Quartz Magnetite	Starter Pit	High Grade
Au	g/t	0.472	0.446	0.776	0.369	0.518	0.517	0.724
Ag	g/t	2.44	2.42	3.18	2.66	2.79	2.74	3.72
Cu	%	0.192	0.206	0.252	0.133	0.181	0.227	0.358
Mo	%	0.007	0.009	0.014	0.008	0.006	0.007	0.009
Fe	%	6.43	4.76	4.03	5.66	7.48	7.06	7.11

The specific gravity and the mineralization hardness, in terms of Ball mill work index for each composite, are shown in Table 18-6. The bond work index ranged from 11.5 to 13.3 kWh/t. Low variation in the mineralization hardness was found in the mineralization types.

Table 18-6: Mineralization SG and Bond Work Index

Composite	Specific Gravity	Bond Mill Grindability (kWh/tonne)
Average	2.82	13.3
US	2.80	12.9
USO	2.72	11.5
PPY	2.78	11.9
QM	2.83	12.9

18.1.3.2 Gravity Gold Recovery and Batch Open Circuit Flotation

Batch rougher and cleaner flotation tests have been performed at grind sizes of p80 = 136, 107 and 85 microns to determine the relationship of grind size on metals recoveries and concentrate grade. The gravity separation was included to recover the coarse gold that is present in the composites, by passing ground slurry through a Knelson concentrator and hand panning the heavy products. The metallurgical gravity gold and bulk copper flotation recoveries are summarized in Table 18-7 and Table 18-8.

Table 18-7: Gravity Gold Recoveries

Average Composite	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.46	25.5	6.5				23.8	32.6				
Cu-Mo Bulk Conc.	0.57	58.5	60.9	86.8	46.5	2.9	43.6	245	27	0.557	31.3	31.6
Rougher Tail	94.72	8.6	12.9	8.1	45.9	81	0.04	0.314	0.02	0.003	5.33	120
Clnr-Scav Tail	4.24	7.4	19.8	5	7.6	16.1	0.74	10.8	0.21	0.012	23.6	31.6
Final Tail	98.97	16	32.7	13.2	53.5	97.1	0.07	0.762	0.02	0.004	6.11	

Upper Sediment	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.23	18.7	3.5				35.3	34.9				
Cu-Mo Bulk Conc.	0.74	63.5	68.5	88.8	58.4	4.5	36.9	211	23.7	0.625	27	21.3
Rougher Tail	93.17	9	11.8	7	37.1	66.9	0.04	0.292	0.02	0.003	3.2	126
Clnr-Scav Tail	5.85	8.8	16.2	4.2	4.5	28.6	0.65	6.4	0.14	0.006	21.7	21.3
Final Tail	99.03	17.8	28	11.2	41.6	95.5	0.08	0.65	0.02	0.003	4.29	

Upper Sediment Oxidized	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.22	33.7	5.3				109.1	72.7				
Cu-Mo Bulk Conc.	0.91	54.6	50.4	81.6	52.3	7.8	43.4	169	22.8	0.766	31.4	19.7
Rougher Tail	92.78	7.3	25.2	12.5	41.6	57.4	0.06	0.829	0.03	0.006	2.28	140
Clnr-Scav Tail	6.08	4.5	19.1	5.9	6.2	34.8	0.54	9.59	0.25	0.014	21.1	19.7
Final Tail	98.86	11.8	44.2	18.4	47.7	92.2	0.09	1.367	0.05	0.006	3.44	

Porphyry	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.34	27.2	5.8				27.8	42.6				
Cu-Mo Bulk Conc.	0.55	55.7	66.7	83.4	53.2	2.9	34.8	300	19.7	0.726	28.8	11
Rougher Tail	94.51	8.2	5.1	10	37.8	78.3	0.03	0.132	0.01	0.003	4.46	116
Clnr-Scav Tail	4.6	8.9	22.4	6.6	9	18.8	0.66	12	0.19	0.015	22	11
Final Tail	99.12	17.1	27.5	16.6	46.8	97.1	0.06	0.683	0.02	0.004	5.28	

Table 18-7: Gravity Gold Recoveries (Cont'd)

Quartz Magnetite	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.21	38	3.8				96	51.8				
Cu-Mo Bulk Conc.	0.82	49.6	66.3	87.1	33.5	4.5	32.5	234	19.4	0.227	35.7	18
Rougher Tail	95.83	7	12.3	7.9	54.7	82.4	0.04	0.371	0.02	0.003	5.58	107
Clnr-Scav Tail	3.14	5.5	17.6	5.1	11.9	13.1	0.93	16.2	0.29	0.021	27	18
Final Tail	98.97	12.4	29.9	12.9	66.5	95.5	0.07	0.874	0.02	0.004	6.26	

Starter Pit	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.23	32.8	3.7				84.2	43.3				
Cu-Mo Bulk Conc.	0.79	53.8	65.5	88.3	43.1	3.7	39.6	211	24.3	0.35	32.6	19
Rougher Tail	94.14	7.1	14.2	6.9	46.3	78	0.04	0.404	0.02	0.003	5.71	101
Clnr-Scav Tail	4.84	6.4	16.6	4.8	10.6	18.3	0.77	9.19	0.22	0.014	26	101
Final Tail	98.98	13.4	30.8	11.7	56.9	96.3	0.08	0.834	0.03	0.004	6.7	

High Grade	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.21	28.1	3				92.3	52.6				
Cu-Mo Bulk Conc.	1.39	57.7	68.1	90.4	52.8	7.5	28.3	176	21.9	0.333	34.1	19.9
Rougher Tail	93.07	6.9	12.5	5.1	37.3	73.5	0.05	0.483	0.02	0.004	4.98	102
Clnr-Scav Tail	5.34	7.4	16.4	4.5	9.9	19	0.94	11	0.28	0.016	22.4	19.9
Final Tail	98.41	14.3	28.9	9.6	47.2	92.5	0.1	1.054	0.03	0.004	5.93	

Table 18-8: Bulk Copper Flotation Recoveries

Average Composite	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.46	25.5	6.5				23.8	32.6				
Cu-Mo Bulk Conc.	0.57	58.5	60.9	86.8	46.5	2.9	43.6	245	27.0	0.557	31.3	31.6
Rougher Tail	94.72	8.6	12.9	8.1	45.9	81	0.04	0.314	0.02	0.003	5.33	120
Clnr-Scav Tail	4.24	7.4	19.8	5.0	7.6	16.1	0.74	10.8	0.21	0.012	23.6	31.6
Final Tail	98.97	16.0	32.7	13.2	53.5	97.1	0.07	0.762	0.02	0.004	6.11	

Upper Sediment	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.23	18.7	3.5				35.3	34.9				
Cu-Mo Bulk Conc.	0.74	63.5	68.5	88.8	58.4	4.5	36.9	211	23.7	0.625	27.0	21.3
Rougher Tail	93.17	9.0	11.8	7.0	37.1	66.9	0.04	0.292	0.02	0.003	3.20	126
Clnr-Scav Tail	5.85	8.8	16.2	4.2	4.5	28.6	0.65	6.4	0.14	0.006	21.7	21.3
Final Tail	99.03	17.8	28	11.2	41.6	95.5	0.08	0.65	0.02	0.003	4.29	

Upper Sediment Oxidized	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.22	33.7	5.3				109.1	72.7				
Cu-Mo Bulk Conc.	0.91	54.6	50.4	81.6	52.3	7.8	43.4	169	22.8	0.766	31.4	19.7
Rougher Tail	92.78	7.3	25.2	12.5	41.6	57.4	0.06	0.829	0.03	0.006	2.28	140
Clnr-Scav Tail	6.08	4.5	19.1	5.9	6.2	34.8	0.54	9.59	0.25	0.014	21.1	19.7
Final Tail	98.86	11.8	44.2	18.4	47.7	92.2	0.09	1.367	0.05	0.006	3.44	

Porphyry	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.34	27.2	5.8				27.8	42.6				
Cu-Mo Bulk Conc.	0.55	55.7	66.7	83.4	53.2	2.9	34.8	300	19.7	0.726	28.8	11.0
Rougher Tail	94.51	8.2	5.1	10.0	37.8	78.3	0.03	0.132	0.01	0.003	4.46	116
Clnr-Scav Tail	4.6	8.9	22.4	6.6	9	18.8	0.66	12.0	0.19	0.015	22.0	11.0
Final Tail	99.12	17.1	27.5	16.6	46.8	97.1	0.06	0.683	0.02	0.004	5.28	

Table 18-8: Bulk Copper Flotation Recoveries (cont'd)

Quartz Magnetite	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.21	38.0	3.8				96.0	51.8				
Cu-Mo Bulk Conc.	0.82	49.6	66.3	87.1	33.5	4.5	32.5	234	19.4	0.227	35.7	18.0
Rougher Tail	95.83	7.0	12.3	7.9	54.7	82.4	0.04	0.371	0.02	0.003	5.58	107
Clnr-Scav Tail	3.14	5.5	17.6	5.1	11.9	13.1	0.93	16.2	0.29	0.021	27.0	18.0
Final Tail	98.97	12.4	29.9	12.9	66.5	95.5	0.07	0.874	0.02	0.004	6.26	

Starter Pit	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.23	32.8	3.7				84.2	43.3				
Cu-Mo Bulk Conc.	0.79	53.8	65.5	88.3	43.1	3.7	39.6	211	24.3	0.35	32.6	19.0
Rougher Tail	94.14	7.1	14.2	6.9	46.3	78.0	0.04	0.404	0.02	0.003	5.71	101
Clnr-Scav Tail	4.84	6.4	16.6	4.8	10.6	18.3	0.77	9.19	0.22	0.014	26.0	101
Final Tail	98.98	13.4	30.8	11.7	56.9	96.3	0.08	0.834	0.03	0.004	6.7	

High Grade	Weight Recovery	Metal Recovery (%)					Grades					Particle size (p80)
Stream	%	Au	Ag	Cu	Mo	Fe	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Fe (%)	um
Gravity Conc.	0.21	28.1	3.0				92.3	52.6				
Cu-Mo Bulk Conc.	1.39	57.7	68.1	90.4	52.8	7.5	28.3	176	21.9	0.333	34.1	19.9
Rougher Tail	93.07	6.9	12.5	5.1	37.3	73.5	0.05	0.483	0.02	0.004	4.98	102
Clnr-Scav Tail	5.34	7.4	16.4	4.5	9.9	19.0	0.94	11.0	0.28	0.016	22.4	19.9
Final Tail	98.41	14.3	28.9	9.6	47.2	92.5	0.10	1.054	0.03	0.004	5.93	

The gravity and flotation tests showed that a combined gold recovery (gravity and flotation) of 80 to 88% can be achieved. Copper recovery of 81 to 90% was obtained in the rougher flotation tests. The results suggested that gravity concentration will recover a portion of gold that is not recoverable by flotation.

The grind effect on copper and gold recoveries is shown in Figure 18-1 and Figure 18-2. A finer grind did not improve copper recovery but has increased the final copper concentrate grade. The gravity separation indicated that gold recovery is likely to be greater at a coarser grind but this may be caused by the higher head grade.

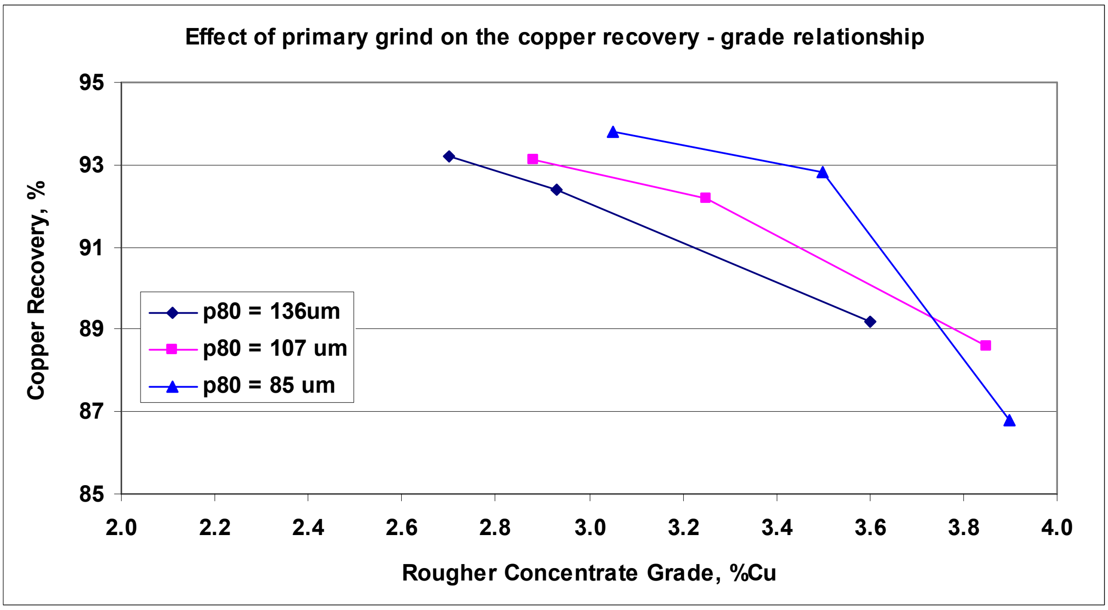


Figure 18-1: Effect of Primary Grind on the Copper Recovery and Grade

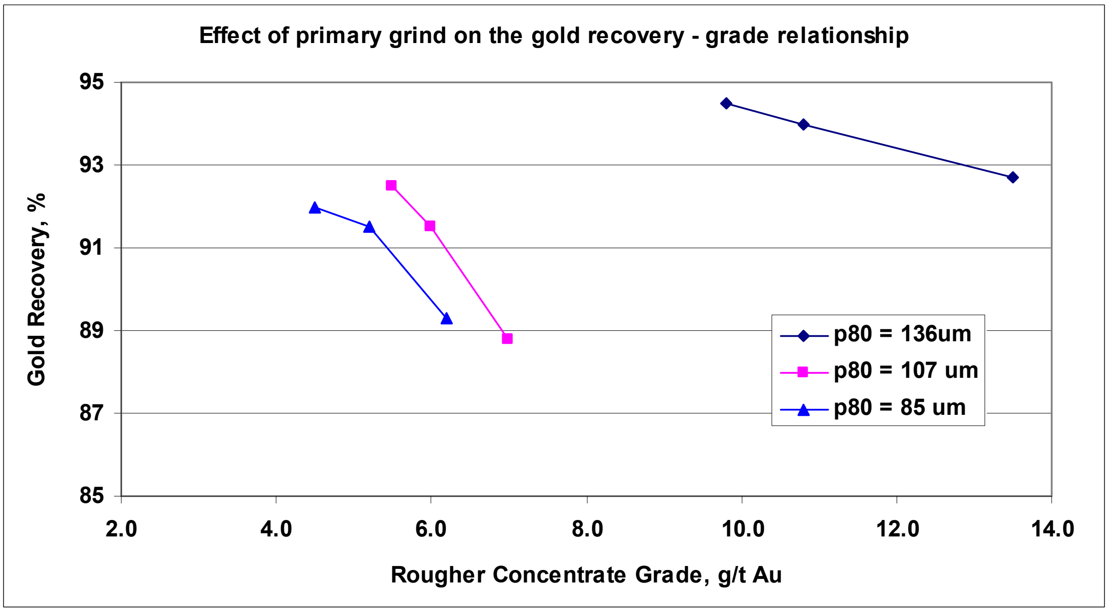


Figure 18-2: Effect of Primary Grind on the Gold Recovery and Grade

18.1.3.3 Effect of Reagents on the Rougher and Scavenger Flotation at p80 = 136um

Reagents screening tests were conducted on the rougher and scavenger flotation of 2kg and 20kg batches to investigate the effect of various collectors on the copper and gold recovery. The collectors investigated were potassium amyl xanthate (PAX), sodium isopropyl xanthate (SIPX), Aero 3302, Aero 3477 and Aerofloat 208.

Some variations on metal recoveries were obtained but this could be caused by variation in the head grade. The results of these tests are summarized in Table 18-9 and Table 18-10.

Table 18-9: Reagents Screening Flotation at p80= 136um

Conditions	Weight %	Grade				Recovery			
		Cu	Mo	Au	Ag	Cu	Mo	Au	Ag
		%	%	g/t	g/t	%	%	%	%
Rougher: A3302/PAX	6.1	2.94	0.064	10.7	28.4	92.3	57.0	82.9	82.2
Scavenger: AF208/PAX	6.7	2.67	0.060	9.65	26.2	93.2	59.1	83.5	84.5
Head grade		0.193	0.007	0.779	2.09				
Tail Assay		0.014	0.003	0.046	0.30				
Rougher: A3302/PAX	6.1	3.05	0.067	5.48	36.6	91.5	57.7	75.1	86.1
Scavenger: AF208/PAX	7.1	2.66	0.061	4.77	32.3	92.7	60.7	76.0	88.2
Head grade		0.203	0.007	0.443	2.58				
Tail Assay		0.016	0.003	0.046	0.30				
Rougher: A3302/PAX	5.7	3.05	0.056	5.27	36.2	92.1	51.9	74.4	85.5
Scavenger: AF208/PAX	6.5	2.72	0.052	4.69	32.6	93.1	54.5	75.2	87.4
Head grade		0.189	0.006	0.403	2.41				
Tail Assay		0.014	0.003	0.046	0.30				
Rougher: A3302/PAX	5.8	3.11	0.074	5.15	34.9	91.5	59.6	73.0	84.7
Scavenger: AF208/PAX	6.5	2.80	0.068	4.64	31.9	92.4	61.1	73.7	86.8
Head grade		0.197	0.007	0.408	2.38				
Tail Assay		0.016	0.003	0.046	0.30				
Rougher: A3302/PAX	7.7	2.43	0.061	4.60	26.5	92.8	62.0	79.9	85.5
Scavenger: AF208/PAX	8.6	2.18	0.056	4.11	24.1	93.6	63.6	80.3	87.5
Head grade		0.201	0.008	0.442	2.37				
Tail Assay		0.014	0.003	0.034	0.30				

Table 18-10: Rougher Flotation of 20 kg (Average Composite)

Conditions	Weight	Grade				Recovery			
		Cu	Mo	Au	Ag	Cu	Mo	Au	Ag
	%	%	%	g/t	g/t	%	%	%	%
Grind to P80 = 125um	5.8	2.81	0.067	6.4	32.7	86.9	57.9	74.1	79.4
A3302/PAX to rougher;									
AF208/PAX to scavenger									
Conditions	**Weight**	**Grade**				**Recovery**			
		Cu	**Mo**	**Au**	**Ag**	**Cu**	**Mo**	**Au**	**Ag**
	%	**%**	**%**	**g/t**	**g/t**	**%**	**%**	**%**	**%**
Grind to P80 = 112um	7.9	2.32	0.059	3.9	28.2	92.6	62.7	67.8	88.3
A3302/PAX to rougher;									
AF208/PAX to scavenger									
Conditions	**Weight**	**Grade**				**Recovery**			
		Cu	**Mo**	**Au**	**Ag**	**Cu**	**Mo**	**Au**	**Ag**
	%	**%**	**%**	**g/t**	**g/t**	**%**	**%**	**%**	**%**
Grind to P80 = 105um	6.1	2.88	0.068	5.25	39.8	90.2	65.6	67.9	86.3
A3302/PAX to rougher;									
no scavenger									

Established standard flotation conditions were then used for the average composite to evaluate the gold and copper recoveries for the Bronson Slope deposits. The test results are listed in Table 18-11. Copper recoveries ranged from 88.7 to 92.8% and molybdenum recoveries ranged from 40.9 to 58.2%. Gold recoveries in the rougher concentrates depended on the preceding gravity concentration recovery and thus varied from 57.1 to 85.1%. However the total gold recovery was maintained at 88.7 to 90.9%.

Table 18-11: Rougher Flotation with PAX and AF208 at Grind p80 = 136um

Gravity Concentrate			Rougher flotation concentrate									
Weight	Recovery, %		Weight	Grade				Recovery				Total
	Au	Ag		Cu	Mo	Au	Ag	Cu	Mo	Au	Ag	Au Rec
%			%	%	%	g/t	g/t	%	%	%	%	%
0.1	32.9	6.0	5.6	3.15	0.071	5.44	32.6	88.7	51.7	57.1	81.5	90.0
0.1	24.1	1.9	5.3	3.33	0.061	6.04	34.7	89.5	40.9	66.8	85.1	90.9
0.3	19.1	2.3	5.3	3.08	0.080	5.18	33.8	89.6	52.8	69.8	84.3	88.9
0.1	23.9	3.8	5.5	3.11	0.061	5.72	40.4	91.6	50.3	66.9	85.3	90.8
-	-	-	5.6	3.04	0.076	5.82	42.4	91.5	58.2	85.1	87.4	85.1
0.2	11.8	8.9	5.3	3.23	n/a	6.12	35.2	90.9	n/a	76.9	86.6	88.7
0.3	24.2	3.4	5.7	2.98	n/a	4.79	25.5	92.8	n/a	64.6	90.7	88.8

* without gravity pre-concentration

n/a = not assayed

18.1.3.4 Cleaner Flotation Tests

Reagents screening tests and grind effect on metal recoveries had also been conducted in the cleaner flotation. A summary of the results are listed in Table 18-12 and Table 18-3. Higher copper concentrate grade can be achieved with a finer grind size but with a lower metal recovery. A positive relationship of gold and iron has inferred that the gold is associated with the pyrite minerals.

Table 18-12: Reagents Screening on Cleaner Flotation at a Regrind Size p80 = 18.5um

Cleaner stage	Collector	pH	NaCN g/t	Grade			Recovery			
				Cu	Mo	Fe	Cu	Mo	Au	Ag
				%	%	%	%	%	%	%
1	A3302/A3477	10.0	10	7.16	0.175		85.8	56.0	61.1	74.0
2		10.5	5	12.70	0.293		81.8	50.5	56.0	64.1
3		11.0	5	17.60	0.360	30.2	76.6	42.0	51.1	55.5
1	A3477	10.0	10	5.34	0.130		89.1	57.5	64.5	78.5
2		10.5	5	12.50	0.277		83.3	49.0	57.7	64.6
3		11.0	5	18.40	0.355	30.4	80.4	41.3	53.9	57.8
1	A3477	10.5	10	6.08	0.154		88.5	57.7	65.2	78.5
2		10.5	5	14.90	0.336		83.2	48.3	59.1	65.1
3		10.5	10	19.60	0.424	32.0	79.1	43.9	55.7	57.2
1	A3302	11.0	10	7.97	0.190		87.7	55.2	61.5	73.2
2		11.0	5	17.50	0.353		82.2	43.8	55.3	60.9
3		11.0	5	22.00	0.405	29.7	77.3	37.6	51.2	52.9

Note: Collectors (4g/t) and NaCN (10g/t) in stage 1 were added in the regrind mill.

Table 18-13: Cleaning Flotation with Varying Reagents and Regrind Size

Regrind time, min	Size P80, um	Cleaner stage	pH	NaCN g/t	Grade			Recovery			
					Cu	Mo	Fe	Cu	Mo	Au	Ag
					%	%	%	%	%	%	%
45	99% passing 37um	1	10.0	10	14.7	0.293	28.8	81.8	41.8	47.0	64.4
		2	10.0	5	25.4	0.488	29.5	77.8	38.3	44.4	56.9
		3	10.5	5	29.8	0.563	29.3	70.8	34.3	39.1	48.2
30	21	1	10.0	10	17.7	0.297	31.5	80.9	33.7	54.6	64.7
		2	10.0	0	26.5	0.420	30.8	76.2	30.0	50.5	56.3
		3	10.0	0	28.0	0.420	30.5	75.0	27.9	49.8	54.3
20	23	1	10.0	10	14.2	0.341	30.9	82.7	44.8	58.1	65.2
		2	10.0	0	22.2	0.499	30.7	79.2	40.3	53.6	56.0
		3	10.0	0	25.2	0.500	30.0	77.2	34.6	51.6	51.8
10	53	1	10.0	10	8.8	0.150	33.3	87.2	41.7	57.7	71.4
		2	10.0	0	11.8	0.190	36.4	84.8	38.2	53.9	68.1
		3	10.0	0	13.2	0.205	37.0	83.7	36.5	52.6	64.8

18.1.3.5 Batch Flotation Variability

The six individual composites US, USO, PPY, QM, SP and HG were tested using the standard flotation conditions to assess the variability of mineralization type for the Bronson Slope deposit. The variable flotation showed that copper recovery ranged from 71 to 82.4%. The concentrate copper grades varied from 23 to 26% copper. This is shown in Table 18-14.

The starter pit, quartz magnetite and high grade composites required more sodium cyanide to improve the cleaner concentrate grade. This could be attributed to higher iron content of these samples.

Table 18-14: Batch Flotation Variability Tests

Composite	Product	Weight	Grade				Recovery			
			Cu	Mo	Au	Ag	Cu	Mo	Au	Ag
		%	%	%	g/t	g/t	%	%	%	%
Average	Gravity Conc.	0.3			24.6	15.8			19.1	2.3
	Cleaner Conc.	0.6	25.2	0.500	36.3	197	77.2	34.6	51.6	51.8
US	Gravity Conc.	0.5			21.3	32.2			21.3	6.2
	Cleaner Conc.	0.6	26.4	0.642	39.8	196	80.6	50.7	55.6	52.3
USO	Gravity Conc.	0.5			53.5	73.1			32.7	11.6
	Cleaner Conc.	0.7	23.2	0.723	43.8	174	71.0	37.7	40.7	42.1
PPY	Gravity Conc.	0.4			28.8	26.7			28.3	3.9
	Cleaner Conc.	0.4	25.6	0.948	44	236	66.9	41.5	40.3	31.9
QM	Gravity Conc.	0.4			52.5	25.9			36.7	3.9
	Cleaner Conc.	0.7	19.2	0.228	30.3	174	80.5	29.9	40.8	50.5
QM *	Gravity Conc.	0.5			22.6	26.7			23.9	4.9
	Cleaner Conc.	0.5	25.6	0.323	40.2	242	77.8	27.5	44.6	46.2
SP	Gravity Conc.	0.2			21.1	20.8			8.3	1.4
	Cleaner Conc.	0.7	24.8	0.330	33.6	182	75.7	30.7	54.9	49.7
HG	Gravity Conc.	0.2			95.4	42.7			23.8	2.2
	Cleaner Conc.	1.2	26.2	0.361	31.5	167	82.4	48.9	49.7	54.6

* One extra stage of cleaning with 2 g/t NaCN.

18.1.3.6 Locked Cycle Flotation Tests

The locked cycle flotation tests were carried out on average, upper sediment, upper sediment oxidized, porphyry and quartz magnetite composites to determine achievable metal recovery and copper concentrate grade. The locked cycle tests were conducted with two cleaning stages and three cleaning stages. The test results of average composite showed that a copper grade of 27% was obtainable with a metal recovery of 86%. Combined gold recoveries ranged from 80.9 to 85.2%. This is shown in Table 18-15.

The variability cleaning flotation tests in Table 18-16 and Table 18-17 indicate that with an additional cleaning stage, higher copper concentrate grade can be achieved with a minimal loss in metal recovery.

Table 18-15: Locked Cycle Flotation Tests of Average Composite

Test	Conc. Grade	Recovery; %		Gold Grade; g/t	
	%Cu	Cu	Au *	Head	Tails
F18	12.8	83.2	80.9	0.604	0.117
F19	19.1	88.3	85.2	0.586	0.088
F25	27.1	86.9	81.6	0.444	0.082
F26	27.0	86.8	84.0	0.443	0.069
F27	20.5	88.1	81.2	0.439	0.084

Table 18-16: Locked Flotation Variability Test with Two Cleaning Stages

Composite	Product	Weight	Grade				Recovery			
			Cu	Mo	Au	Ag	Cu	Mo	Au	Ag
		%	%	%	g/t	g/t	%	%	%	%
Upper Sediment	Gravity Concentrate	0.2			35.3	34.9			18.7	3.5
	Cu bulk Cleaner Concentrate	0.7	23.7	0.625	36.9	211	88.8	58.4	63.5	68.5
	Final tailings	99.1	0.022	0.003	0.078	0.65	11.2	41.6	17.8	28.0
	Flotation Feed	100.0	**0.199**	**0.008**	**0.432**	**2.30**				
Upper Sediment Oxidized	Gravity Concentrate	0.2			109	72.7			33.7	5.3
	Cu bulk Cleaner Concentrate	0.9	22.8	0.766	43.4	169	81.6	52.3	54.6	50.4
	Final tailings	98.9	0.048	0.006	0.087	1.367	18.4	47.7	11.8	44.2
	Flotation Feed	100.0	**0.256**	**0.013**	**0.726**	**3.06**				
Porphyry	Gravity Concentrate	0.3			27.8	42.6			27.2	5.8
	Cu bulk Cleaner Concentrate	0.5	19.7	0.726	34.8	300	83.4	53.2	55.7	66.7
	Final tailings	99.2	0.022	0.004	0.059	0.683	16.6	46.8	17.1	27.5
	Flotation Feed	100.0	**0.130**	**0.007**	**0.343**	**2.47**				
Quartz Magnetite	Gravity Concentrate	0.2			96	51.8			38.0	3.8
	Cu bulk Cleaner Concentrate	0.8	19.4	0.227	32.5	234	87.1	33.5	49.6	66.3
	Final tailings	99	0.024	0.004	0.067	0.874	12.9	66.5	12.4	29.9
	Flotation Feed	100.0	**0.183**	**0.006**	**0.537**	**2.89**				

Table 18-17: Locked Flotation Variability Test with Three Cleaning Stages

Composite	Product	Weight	Grade				Recovery			
			Cu	Mo	Au	Ag	Cu	Mo	Au	Ag
		%	%	%	g/t	g/t	%	%	%	%
Upper Sediment	Gravity Concentrate	0.1			49.7	34.5			16.1	2.1
	Cu bulk Cleaner Concentrate	0.6	27.4	0.68	40.7	206	87.6	47.5	61.7	59.0
	Final tailings	99.3	0.025	0.005	0.095	0.877	12.4	52.5	22.3	38.8
	Flotation Feed	100.0	**0.201**	**0.009**	**0.424**	**2.24**				
Upper Sediment Oxidized	Gravity Concentrate	0.2			73.5	42.3			27.7	3.2
	Cu bulk Cleaner Concentrate	0.6	27.1	0.359	44.9	267	86.8	31.2	54.9	65.2
	Final tailings	99.2	0.024	0.005	0.084	0.767	13.2	68.8	17.3	31.7
	Flotation Feed	100.0	**0.183**	**0.007**	**0.479**	**2.40**				
Starter Pit	Gravity Concentrate	0.2			84.2	43.3			32.8	3.7
	Cu bulk Cleaner Concentrate	0.8	24.3	0.35	39.6	221	88.3	43.1	53.8	65.5
	Final tailings	99	0.026	0.004	0.079	0.834	11.7	56.9	13.4	30.8
	Flotation Feed	100.0	**0.218**	**0.006**	**0.583**	**2.68**				
High Grade	Gravity Concentrate	0.2			92.3	52.6			28.1	3.0
	Cu bulk Cleaner Concentrate	1.4	21.9	0.333	28.3	176	90.4	52.8	57.7	68.1
	Final tailings	98.4	0.033	0.004	0.099	1.054	9.6	47.2	14.3	28.9
	Flotation Feed	100.0	**0.335**	**0.009**	**0.680**	**3.59**				

18.1.3.7 Multi-elements Analysis

Multi-element ICP analyses for trace elements and assays of penalty elements of copper concentrate were performed on selected locked cycle tests. The analysis showed low levels of deleterious elements were found in the copper bulk concentrate. The analysis results are listed in Table 18-18 and Table 18-19.

Table 18-18: Multi-elements ICP Analysis of Flotation Concentrate and Head Samples

Element		Average composite		SP composite		HG composite	
Symbol	unit	Head	Conc.	Head	Conc.	Head	Conc.
Al	ppm	5352	400	6308	298	6209	123
Sb	ppm	7	73	< 5	33	10	< 5
As	ppm	< 5	44	< 5	14	< 5	< 5
Ba	ppm	33	17	50	20	41	9
Bi	ppm	< 2	65	< 2	< 2	< 2	< 2
Cd	ppm	0.3	73	< 0.1	28.3	< 0.1	2.2
Ca	ppm	6440	3917	6641	9464	5871	463
Cr	ppm	329	34	292	31	447	15
Co	ppm	15	56	15	50	17	52
Cu	ppm	1947	27%	2662	24%	4352	11%
Fe	%	5	29%	7	30%	7.4	18%
La	ppm	< 2	< 2	< 2	< 2	< 2	< 2
Pb	ppm	21	972%	6	431%	6	100
Mg	ppm	6715	387	6962	800	6096	106
Mn	ppm	607	229%	662	456%	692	66
Hg	ppm	< 3	< 3	< 3	< 3	< 3	< 3
Mo	ppm	56	302	63	211	95	23
Ni	ppm	187	62	154	56	244	52
P	ppm	437	8360	448	< 100	516	3113
K	ppm	4969	< 100	5516	< 100	5251	< 100
Sc	ppm	2	2	1	2	< 1	< 1
Ag	ppm	2	84	2.7	200	3.6	85.1
Na	ppm	165	129	152	< 100	152	< 100
Sr	ppm	21	14	22	29	22	2
Tl	ppm	< 10	< 10	< 10	< 10	< 10	< 10
Ti	ppm	477	< 100	475	< 100	477	< 100
W	ppm	< 5	49	< 5	46	< 5	26
V	ppm	38	9	51	10	52	4
Zn	ppm	150	11993	109	5101	93	700
Zr	ppm	1	5	< 1	5	< 1	2

* Samples digested by multi-acids

Table 18-19: Penalty Chemical Analysis of Copper Concentrates

Description		Average	Upper Sediment	Upper Sediment Oxidized	Porphyry	Quartz Magnetite	Starter Pit	High Grade
Major Elements	Au (g/t)	43.6	36.9	43.4	34.8	32.5	39.6	28.3
	Ag (g/t)	245	211	169	300	234	211	176
	Cu (%)	27.0	23.7	22.8	19.7	19.4	24.3	21.9
	Mo (%)	0.557	0.625	0.766	0.726	0.227	0.350	0.333
	Fe (%)	31.3	27.0	31.4	28.8	35.7	32.6	34.1
Penalty Elements	As (ppm)	< 5	< 5	< 5	n/a*	< 5	14	< 5
	Sb (ppm)	133	78	47	n/a*	24	33	< 5
	Pb (%)	0.11	0.24	0.04	n/a*	0.05	0.04	0.01
	Zn (%)	1.02	1.5	1.63	n/a*	0.42	0.51	0.07
	Hg (ppm)	1.5	1.9	0.7	n/a*	n/a*	< 3	< 3
	Bi (ppm)	60	37	< 2	n/a*	75	< 2	< 2
	Cl (%)	n/a*	0.12	0.11	n/a*	n/a*	n/a*	n/a*
	F (%)	n/a*	n/a*	n/a*	n/a*	n/a*	n/a*	n/a*
	Al_2O_3 (%)	0.48	0.48	0.37	n/a*	0.11	n/a*	n/a*
	Se (ppm)	133	215	108	n/a*	n/a*	n/a*	n/a*
	Te (ppm)	281	287	257	n/a*	n/a*	n/a*	n/a*

(n/a* Not assayed)

18.1.3.8 Cyanidation

Cyanide leach tests were conducted on rougher concentrate and cleaner tail samples of the average composite samples to determine amenable of gold and silver to cyanide leach. The tests were carried out at a pH of 10.5, 1.0 g/L of sodium cyanide addition for 24 hours leach duration. The results showed that a poor recovery of gold and high consumption of cyanide. This eliminates cyanidation as a viable process option. High extraction of silver was observed in cleaner tails. SEM-EDX analysis of the cleaner tail samples indicated that a large proportion of silver bearing minerals were occurred on the rim of pyrite particles. Results are summarised in Table 18-20.

Table 18-20: Cyanidation Results

Sample	Sample Grade			NaCN consumption	Extraction		
	Au	Ag	Cu		Au	Ag	Cu
	g/t	g/t	%	kg/t	%	%	%
F9B Cleaner tail 1	0.41	6.2	0.15	2.69	64.9	80.2	23.8
F10 Cleaner tail 1	1.20	8.5	0.30	3.64	63.6	68.0	22.5
F15 rougher concentrate	6.12	35.2	3.23	7.03	62.4	32.9	8.7
F16 rougher concentrate	4.79	25.5	2.98	5.42	61.1	23.9	10.0

18.1.3.9 Copper - Molybdenum Flotation

Scoping copper molybdenum flotation was conducted to produce saleable copper concentrate and molybdenum concentrate, bulk copper cleaner concentrate were utilized for the tests. The concentrate was thickened to about 60% solids, conditioned with fuel oil and sodium hydrosulphide until the pulp potential was below -460 mV (Ag/AgCl electrode) then the pulp was diluted to about 30% solids and float with nitrogen gas. The recovered molybdenum rougher concentrate was cleaned twice with fuel oil and sodium hydrosulphide in the same pulp potential conditions as in the rougher float.

A small quantity and low grade of molybdenum product was produced in the rougher and cleaner test. Insufficient quantity of sample was available for the test. It precluded the production of a final cleaned molybdenum concentrate. Therefore no chemical analysis is available. The results are listed in Table 18-21.

Table 18-21: Copper Molybdenum Flotation

Product	Weight	Grade, %		Recovery, %	
	%	Cu	Mo	Cu	Mo
Mo cleaner conc.	1.0	6.38	15.5	0.3	42.2
Mo rougher tail (Cu Conc.)	82.6	23.4	0.031	86.9	7.0
Feed (Cu bulk conc.)		22.2	0.364		
Mo cleaner conc.	1.8	3.37	23.7	0.1	33.7
Mo rougher tail (Cu Conc.)	65.7	24.4	0.010	70.7	2.3
Feed (Cu bulk conc.)		22.7	0.289		

18.1.3.10 Magnetite Recovery

Magnetic recovery test was carried out on the rougher tails. A rougher and a cleaner magnetite concentrate were produced by passing the copper rougher tails through a Sala laboratory magnetic separator. Regrind is required on the cleaner product to improve the magnetite concentrate grade. Table 18-22, Table 18-23 and Table 18-24 shows the results of the magnetic separation test.

Table 18-22: Magnetic Separation Tests

No regrind		
Rougher concentrate	weight % wrt feed	7.2
Cleaner concentrate	weight % wrt feed	6.5
Cleaner concentrate	%Fe	53.7
Cleaner concentrate	Specific gravity	4.13
With regrind		
Weight recovery, cleaner to re-cleaner, %		76.4
Re-cleaned concentrate, Fe%		71.4
Re-cleaned concentrate, specific gravity		5.08
Re-cleaned concentrate, magnetic content		98.3

Table 18-23: Cleaner Magnetic Concentrate Sizing

Size, Mesh	Weight, %	Cumulative Wt. %	Fe assay, %	Fe distribution, %
100	10.0	89.9	28.4	5.3
150	20.5	69.4	40.4	15.4
200	18.9	50.6	52.4	18.3
325	21.3	29.3	60.1	23.8
-325	29.3		67.9	37.1
Total	100.0		53.7	

Table 18-24: Chemical Analysis of the Cleaned Magnetite Concentrate

Element	Fe	Cu	P	Ti	Ni	Mo	S
%	71.4	0.02	< 0.01	0.06	0.20	0.20	0.03

18.1.3.11 Concentrate and Tailings Thickening

Cleaner concentrate, cleaner scavenger tail and final rougher tail were submitted to Eimco process equipment for settling tests. The tests were conducted by the Eimco personnel to determine the flocculent requirement and for thickener sizing. It is recommended that a feed dilution (e.g. E-Duk self dilution system) shall be installed. The results of the test work are summarized in the Table 18-25.

Table 18-25: Concentrate and Tailing Thickening

Sample	Percol 351 dosage g/t	Unit thickener area m2/tpd
F 17 Cleaner concentrate	0	0.43
	50	0.39
	75	0.21
	100	0.20
F 17 Cleaner scavenger tail	30	0.65
	60	0.57
	172	0.37
F17 Final rougher tail	10	0.11
	20	0.07
	30	0.05

18.1.3.12 Concentrate Filtration

The cleaner copper bulk flotation concentrate was sent to Larox, Inc for pressure filtration tests. The tests indicated that cake moisture of 7.02% at a filtration rate of 669 kg/m^2h could be achieved. This is shown in Table 18-26.

Table 18-26: Pressure Filtration Test Results

Filtration	Unit	
Sizing p95	um	45.0
Feed density	%	59.5
Cycle time		
feeding	min	1.0
Pressing	min	0.5
Air Blow	min	1.0
Discharge	min	4.0
Total	min	6.5
Cake Thickness	mm	30.0
Cake Moisture	%	7.9

18.1.3.13 Acid Base Accounting Tests

Acid base accounting tests were conducted on rougher tails and cleaner-scavenger tails from the locked cycle tests. The rougher tails reported an average neutralization potential (NP) of 22.6 kg $CaCO_3$ equivalent per tonne, but the cleaner-scavenger tails reported an average of -526.1 kg $CaCO_3$ equivalent per tonne. This indicates that the cleaner-scavenger tail has a high acid generating potential, presumably due to the high pyrite content. Even when both the rougher tails and the cleaner scavenger tails are combined the net NP is -0.55 kg $CaCO_3$ equivalent per tonne. This indicates that the combined final tailings have a slight acid generating potential.

18.2 Processing

Based on the metallurgical test work, a conventional gravity and flotation concentrator processing plant can be proposed for the Bronson Slope ores. The concentrator plant will consist of SAG mill, Ball mill, Knelson gravity concentrator, Copper flotation, Regrind mill, and Dewatering facilities. At 15,000 tpd milling rate, it is expected to produce 27,000tpa of bulk copper concentrate. It contains of copper, gold and silver. Figure 18-3 illustrates a schematic flow sheet for the processing plant.

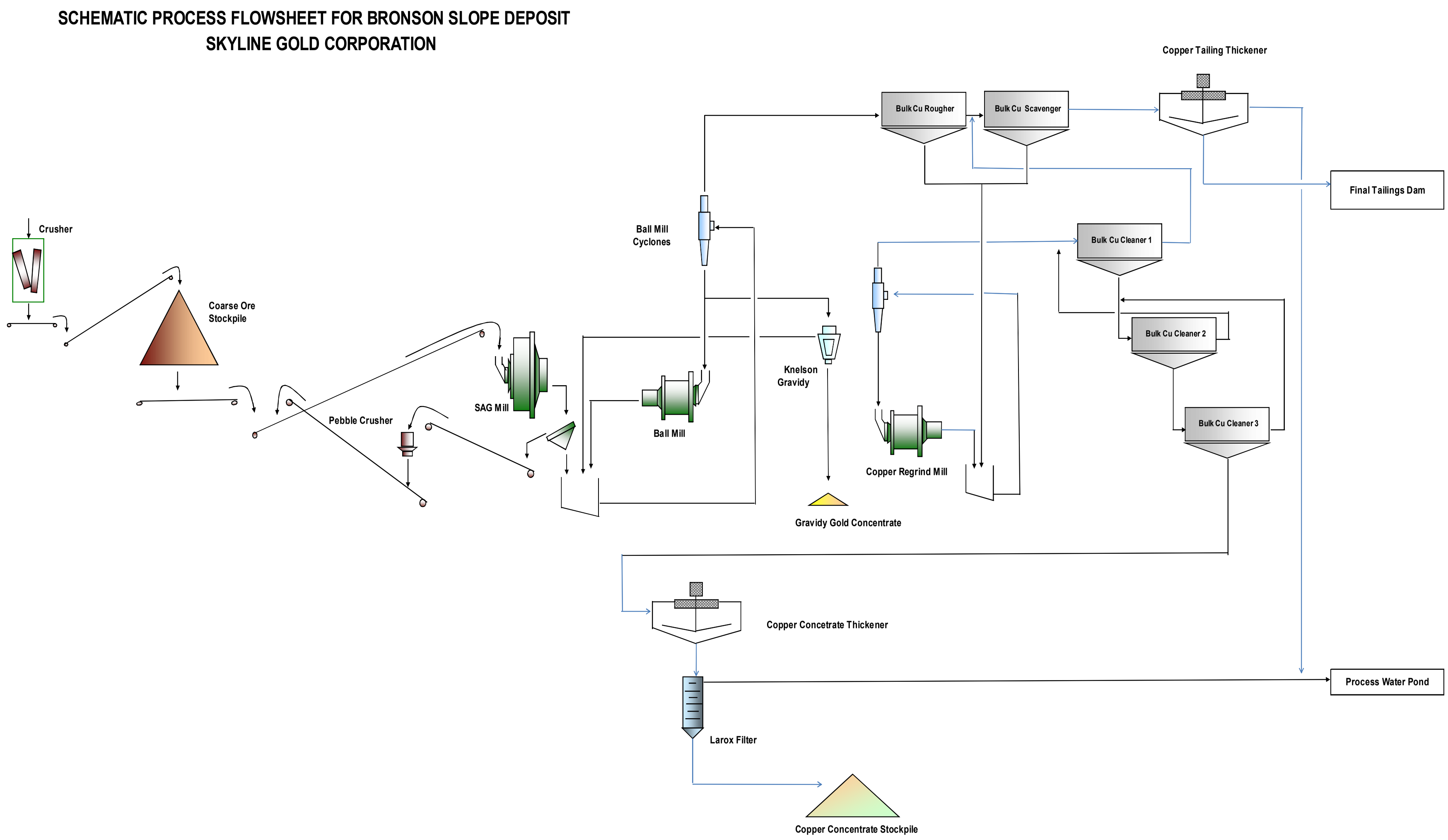


Figure 18-3: Proposed Process Flowsheet

The processing plant is comprised of the following areas:

- Crushing and ore reclaim
- Milling
- Copper flotation
- Copper concentrate dewatering
- Process water
- Compressed air
- Reagents handling

18.2.1.1 Crushing and Ore Reclaim

Crushing will be undertaken in the pit using mobile crushers. The run of mine (ROM) material is required to be crushed to 250mm to be transported effectively using the mobile and fixed conveying systems planned for the project. For maximum SAG mill productivity the ore material from the pit will be crushed to -150mm. The mill feed is delivered to the base of the orebody where it will either be discharged using a stacker conveyor onto the ROM stockpile or directly onto the Coarse Ore Stockpile. Further information on the crushing and mill feed and waste transport is included in Item 25.

A potential site for the concentrator is near the Bronson creek, adjacent to the previous location of the Snip mill facilities at the base of the Bronson Slope deposit.

The coarse mill feed is reclaimed from the stockpile with two (42” diameter by 15’long) apron feeders. The apron feeders will feed onto a single SAG mill feed conveyor. Dust collector facilities will be installed to collect dust and fine particles around the crusher, conveyor and mill feed transfer point to minimize dust in the area.

18.2.1.2 Milling

The mill feed is conveyed to a 32’ diameter by 14’ (9.75m diameter by 4.27m) grate discharge SAG mill, driven by a 5,600 kW synchronous motor. The SAG mill discharge slurry is screened by a trommel screen and a vibrator deck screen with opening apertures of 10mm, the oversize pebbles is returned to the SAG mill via a pebble crusher. The SAG mill screen undersize discharges to a common hopper of the SAG mill and Ball mill, then it is mixed with the ball mill discharge slurry and gravity tails to form a cyclone feed. Lime is added to the SAG mill to control the slurry pH for the flotation feed.

The secondary mill is a 19’diameter by 32’ (5.79m diameter by 9.75m) ball mill which operates in a closed circuit and is driven by a 5,600kw synchronous motor. The combined slurry of the SAG mill, ball mill and gravity tail is pumped to a cluster of cyclones (8 x 26”; 5 in operation) for separation. The overflow (fines) material is sent to the rougher-scavenger copper flotation. The underflow (coarse) material is circulated back into the ball mill. A portion of the cyclones underflow will be diverted to the Knelson concentrator for coarse gold recovery. The Knelson concentrate is discharged to a shaking table (1.83m by 4.57m) on a batch basis for further upgrade. The gravity and the tabling tails will re-circulate back to the ball mill.

18.2.1.3 Copper Flotation

Cyclone overflow slurry is fed to a bank of 8 x 50m3 capacity rougher and scavenger flotation cells. The rougher and scavenger concentrate is fed to a regrind mill for additional grinding before it is delivered to the cleaning circuits. Three cleaner stages are required for the copper concentrate production. Tailing from the cleaner circuit is returned to the scavenger flotation cells and then it is disposed to the tailings storage facility via a conventional thickener process where water is recycled and returned to the processing plant.

18.2.1.4 Copper Concentrate Dewatering

An Eimco conventional thickener will be installed for the copper concentrate dewatering. The settling process is enhanced by addition of flocculent agents. The overflow water from the thickener will be recycled to flotation as launder dilution water. The underflow slurry of 60% solids will be directed to one of the two concentrate holding tanks, which can hold a production of 24 hours at nominal head grade and tonnage.

A further water reduction is carried out by a filtration plant. Filtered concentrate product with a moisture content of 8% will be produced in a continuous Larox pressure filter. The plant is designed for an average of 12 hours operation per day.

A load truck scale will be installed for proper recording of the concentrate load trucks.

18.2.1.5 Process Water

Process water will be supplied from the recycled water from the tailings and concentrate thickeners overflows. The water is collected in the process water pond. The makeup process water will be reclaimed from the tailings dam. Process water will mainly be used for the dilution of mill feeds and as a transportation media for the concentrate.

Raw water will be used for fire fighting, cooling, gland sealing and other applications. The water will be obtained from boreholes. If the reclaimed water from the tailings is suitable for this purpose this can be used for raw water.

18.2.1.6 Compressed Air

Air compressors will be installed in the concentrator plant. They will provide compressed air for pulsating of the dust collector, instrument air and for pneumatic tool operation around the plant. Low pressure air will also be provided by air blowers to the flotation cells to generate froth for the flotation process.

18.2.1.7 Reagents Handling

The reagents will be delivered to the site by drums, containers or bulk bags. The reagents will be stored in a chemicals storage warehouse. A minimum of four weeks reagents stock is recommended to be kept on site for operations. A mixing and storage facility for reagents will be installed where reagents will be prepared daily.

19 Mineral Resource and Mineral Reserve Estimates

The Bronson Slope Property hosts a porphyry gold-copper-silver-molybdenum deposit. According to G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., from Burgoyne Geological Inc., the quality of the SGC exploration work is considered to be of good quality and meets industry standards. A current mineral resource has been estimated for the Bronson Slope deposit that meets CIMM resource standards and classifications.

The resource estimate, based on block modelling and kriging is considered reliable and relevant. The resource estimate is presented based on two separate cases of metal prices. These are presented in Table 19-1 below:

Table 19-1: Metal Prices Used for 2008 Resource Estimate

Metal	Case 1	Case 2
Cu	$1.50/lb	$2.00/lb
Au	$525/t.oz	$650/t.oz
Ag	$8/t.oz	$10/t.oz
Mo	$10/lb	$12/lb

*nb - all figures are in USD

Case 1 metal prices are based on the values used in the 2007 resource estimate. The resource estimate for this case has been completed to provide a direct comparison between the 2007 resource estimate and the 2008 resource estimate. Case 2 metal prices are considered to be potentially more realistic and therefore a second resource estimate has been provided based on these revised metal prices.

The metal prices were used along with block based metallurgical recoveries to determine individual block values. The mineral resources presented in the tables below were then determined based on a cut off of USD 9.00 per tonne net recoverable value for Case 1 and Case 2 respectively.

Table 19-2: Case 1 2008 Bronson Slope Resource Estimate

Case 1 – Bronson Slope Resource Estimate (Cutoff USD 9/t NRV)					
Category	**Metric Tonnes**	**Au g/t**	**Cu %**	**Ag g/t**	**Mo %**
Measured	58,700,000	0.50	2.45	0.18	0.0058
Indicated	80,800,000	0.36	2.38	0.15	0.0094
Inferred	30,200,000	0.34	1.89	0.15	0.0070
Total Measured + Indicated	**139,500,000**	**0.42**	**2.41**	**0.17**	**0.0079**

Table 19-3: Case 2 2008 Bronson Slope Resource Estimate

Case 2 – Bronson Slope Resource Estimate (Cutoff USD 9/t NRV)					
Category	**Metric Tonnes**	**Au g/t**	**Cu %**	**Ag g/t**	**Mo %**
Measured	74,800,000	0.45	2.31	0.17	0.0059
Indicated	150,300,000	0.31	2.17	0.13	0.0087
Inferred	91,600,000	0.27	1.76	0.13	0.0080
Total Measured + Indicated	**225,100,000**	**0.36**	**2.22**	**0.14**	**0.0077**

For further details of on mineral resource calculations refer to "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., and posted on SEDAR (www.sedar.com).

20 Other Relevant Data and Information

Previously SGC have completed a number of studies and evaluation reports in order to identify the engineering requirements for the project and to evaluate potential cash flow, metallurgical recoveries and process design, anticipated capital and operating costs, geotechnical design, infrastructure and access requirements, and other pre-feasibility studies on the Bronson Slope deposit between 1995 and 1997. Item 21 details the studies and reports that were referred to for this study. Table 20-1 also details a brief description of what each of the reports includes.

Table 20-1: Previous Studies and Evaluation Reports

TITLE (ALPHABETIC ORDER)	AUTHOR	DATE	COMMENTS
Bronson Slope Project – Appendix to Marketing Report	J. Arthur Ganshorn (Marketing Consultant)	May 1, 1997	Includes fax and memos of quotes for delivering products produced from the mine to the market including trucking costs from 3 different suppliers, rail transport option costs, port weighing sampling and assaying costs, and ocean freight rates from Port of Stewart.
Bronson Slope Project – Marketing report 1997	J. Arthur Ganshorn (Marketing Consultant)	May 1, 1997	Identify marketing and smelter costs and forecast average values for the cost for the life of mine, producing net smelter returns.
Bronson Slope ~ Penalty Element Assays~1997	Bern Klein (Process Research Associates)	June 17, 1997	Facsimile containing results of analyses on the concentrates for penalty element. Tests done by Process Research Associates Ltd for International Skyline Gold Corp.
Acid Base Accounting (ABA) Test Results	Bern Klein (Process Research Associates)	April 29, 1997	ABA tests and multi-element ICP analyses on rock samples from the Bronson Slope deposit. Results include Net Neutralization Potential (Net NP) and average NP, average AP and sulphide sulphur content.
Cambria Gordon EA Application & Temporary Access Road	Forsite – Forest Management Specialist	December 15, 2006	Contains report for temporary access trail proposal. Using previous pre-feasibility studies, this report outlines the possible construction schedule, cost estimate and maintenance for a temporary exploration trail from Forrest Kerr Access Road to Bronson Airstrip.
Capital Cost Estimates	Michael Moore	December 8, 2007	Capital cost spreadsheet and outline revision that includes 2 scenarios – with and without a Moly circuit.
Final Project Report Specifications - VI	Bronson Slope Project Committee	June 18, 1996	Environmental Assessment for the proposed project – a detailed impact assessment submission) seeking project approval certificate under the environmental assessment act. Includes project specifications, environmental assessment process, status of the review of the mine proposal and specifications that ISGC is required to prepare.
Final Project Report Specifications - VII	Bronson Slope Project Committee	June 18, 1996	Appendices A to E
Final Project Report Specs - VIII - Part 1	Bronson Slope Project Committee	June 18, 1996	Appendices 1-5
Final Project Report Specifications - VIII - Part 2	Richard Weir (Habitat Geologist)	April 11, 1996	Includes predicted preliminary steel discharge summary and a memorandum stating the deficiencies for ISGC's application for the project approval certificate.
Long Term Water Treatment Options	Hallam Knight Piesold Ltd	July 1997	Proposal for preparation of long term water treatment options for the Bronson slope project, including ARD Geochemistry, preliminary design of passive water treatment system and preparation of the report.
Metallurgical Report - Part 1	Qi Liu (Senior Metallurgist) Process Research Associates	July 18, 1997	Metallurgical study on the Bronson Slope samples including types of tests carried out and results and discussion. Average mineral grade and recoverable rates included.
Metallurgical Report - Part 2	Qi Liu (Senior Metallurgist) Process Research Associates	July 18, 1997	Test reports and Appendices for the above report.
Mine Plan	Christopher Turek (P.Eng), David Yeager (P.Geo)	September 22, 1995	Mine plan that includes location/present access, site facilities, geology, pit access, mining, waste management plan (inc. water balance), power supplies, manpower requirements and project alternatives.
Operating Cost Estimates	David Yeager	April 12, 2000	Memorandum and revised report that compiles sensitivity analyses of operating costs and metal prices on Bronson Slope project and review various operating costs of estimates and similar Canadian projects.
Prelim Feasibility Study - Mill Facilities	Rescan Engineering Limited	March 27, 1995	Preliminary Feasibility Study on Mill Facilities – estimating the capital and operating costs for a 12,000 tonne per day flotation mill including ore stockpile and concentrate storage facilities.
Revised Marketing Costs - Transportation	J. Arthur Ganshorn (Marketing Consultant)	March 12, 2007	Transportation costs updated from 1997.
Rock Waste Volume Estimates	R.C. Dick (Geotechnical Engineering Consultant)	28 April, 1995	Calculation of rock waste volume from topographic maps for waste disposal purposes.

Title (alphabetic order)	Author	Date	Comments
Seismic Refraction Investigation	Agra Earth and Environmental Limited (M.R. Torney, R.A. Hillman)	May, 1997	Seismic refraction survey – to determine thickness and overburden materials and the depths to bedrock for two proposed storage dams for mine tailings. Includes survey method, analysis and geophysical results.
Sky Creek Tailings Area	R.C. Dick (Geotechnical Engineering Consultant)	11 August, 1995	Reviewed estimated capacity for Sky Creek tailings area based on two sets of new information received at the time.
Conceptual Design of Tailings Facility - Piteau Engineering	Piteau Engineering	January, 1997	Conceptual study of tailings facility and the related costs for waste produced from the 12,000 t/day process plant.
Copper Concentrate Report [of Mkg Rpt]~1997	J. Arthur Ganshorn (Marketing Consultant)	May 1, 1997	Part of the “Bronson Slope Project –Marketing report 1997”, identifies the marketing/smelter terms and evaluates copper (Au+Ag) concentrate and forecasts the average values for LOM – used to calculate the “Net Smelter Return”.
Engineering Report by Robert Dick	R.C. Dick (Geotechnical Engineering Consultant)	1997	Summary of discussions (1996 and 1997) regarding proposed tailings impoundment.
Molybdenum Report [of Mkg Rpt]~1997	J. Arthur Ganshorn (Marketing Consultant)	May 1, 1997	Evaluate the average value of molybdenum (FOB mine gate) over LOM.
Nordberg Face Crushing System	Nordberg	June, 1996	Includes correspondence, typical mining plans, equipment specifications and equipment brochures for a proposed face crushing system at Bronson Slope.
Preliminary Geotechnical Assessments and Slope Design Studies	Alan Stewart (Piteau Associates Engineering Ltd.)	May 8, 1997	Memo and preliminary geotechnical report that summarizes data collection, analyses and assessments conducted by Piteau Associates Engineering Ltd.
TOC of Conceptual Design of Tailings Facility of Bronson Slope Mine	Piteau Associates Engineering Ltd.	January, 1997	Contents page for Bronson Slope Mine Conceptual Design of Tailings Facility
TOC of Nordberg Face Crushing System Report	Nordberg		Contents page for “Nordberg Face Crushing System” report
TOC of Preliminary Geotechnical Assessments and Slope Design Studies for the Bronson Open Pit	Piteau Associates Engineering Ltd.	May 8, 1997	Contents page for “Preliminary Geotechnical Assessments and Slope Design Studies for the Bronson Open Pit” report
TOC of VI Mine Design for the Bronson Slope Project	Fourth Year Class, Department of Mining and Mineral Process Engineering, University of British Columbia.	December, 1996	Executive Summary and contents page for the Stage 2 feasibility analysis of the copper-gold porphyry deposit.
TOC of VII Mill Design for the Bronson Slope Project	As above	April, 1997	Contents page for the Mill Design report
TOC of VIII Financial Analysis for the Bronson Slope Project	As above	April 1997	Contents page for the Financial Analysis report
VI Mine Design for the Bronson Slope Project	As above	December, 1996	Volume One of the Mine Design report (feasibility analysis) containing background information, geology, site layout and infrastructure, mining (method and development, equipment), materials handling, mine services requirements, staffing and administration, scheduling, environmental impacts, economic/social impacts, cultural and heritage impacts and CEAA requirements.
VII Mill Design for the Bronson Slope Project	As above	April 1997	Volume two – continuation of the above report, including mineral processing testworks, plant design and environmental protection. Also include Tables and Appendices.
VIII Financial Analysis for the Bronson Slope Project	As above	April 1997	Volume three – continuation of the above report – financial analyses including commodity forecasting, smelter contracts, capital cost estimations, operating costs, taxation and analysis of economics.

The content of these reports is likely to be out of date and require review. The work completed as part of this Preliminary Assessment has reviewed some of these reports and reference has been made to them where appropriate. It would not be appropriate to rely on any of these reports without professional review to ensure that the engineering practices and methods contained in the reports are still current and suitable for the Bronson Slope Project. In particular the cost estimation that has been completed in the past will have diminished relevance to today's costs other than to provide an itemized guideline for future cost estimates based on more current quotations of capital and operating costs.

20.1 Cost Estimation Accuracy

This report provides a revised capital and operating cost estimate for the project based on more up to date quotes for major plant items, and revised unit cost estimates for labour, materials, minor plant and consumables. The cost estimate provided is accurate to ±30%. Also included in the cost estimate is an allowance for contingency of between 10 and 15% depending on the cost item. More detail is provided in Item 25.

20.2 Waste and Tailings Storage Facility

The waste storage facility is based on a conceptual design with no review of environmental, geotechnical or land ownership constraints. Alternative waste storage areas may be east of the tailings dam, within the tailings dam (given enough capacity) and/or upstream within the Bronson creek area. LAL recommends a study be completed to determine the viability of placing waste in the Triangle lake area or within these alternative areas.

The original tailings storage facility has been designed to the 165m RL to provide adequate capacity for the mine plan generated in 1997. In order to demonstrate capacity for the current mine plan the tailings embankments were extended to the 185mRL. The author recommends further review of this extension and the design concepts for the tailings facility to ensure adequacy of the design with consideration for current environmental and geotechnical standards.

20.3 Post Preliminary Assessment Pit Optimisation

It was recognised after the detailed mine plan and cost estimate presented in this assessment was completed that the original inputs used to generate the most optimum pit were not completely representative of the final mining and processing method selected for the project. Table 20-2 shows a comparison of the detailed design, cost and revenue estimates for the project versus those used in the original optimisation.

Table 20-2: Comparison of Optimisation Inputs (All Currency in USD)

Parameter Description	Original Value	Revised Value	Difference %
Long term metal prices			
Cu (USD/lb)	$2	$2	0%
Au (USD/oz)	$650	$700	7.6%
Ag (USD/oz)	$15	$15	0%
Mo (USD/lb)	$12	$0	-
Capital (USD)	$175,000,000	$200,000,000	14%
Residual	10%	10%	0%
Discount rate	8%	8%	0%
Geotechnical design parameters			
Average - All domains	55 degrees	50 degrees	-9%
Mining parameters			
Bench height (m)	10	10	
Mining cost - waste ($/t)	$1.50/t mined	$1.65/t mined	10%
Mining cost - mill feed ($/t)	$1.50/t mined	$1.40/t mined	-7%
Mining dilution (%)	5%	5%	0%
Mining recovery (%)	95%	95%	0%
Concentrator parameters			
Expected mill throughput (t/annum)	5.5Mtpa	5.1Mtpa	-7%
Variable processing cost ($/t milled)	$6.00/t	$4.30/t	-28%
Concentrator Recoveries			
Au	84%	84%	0%
Ag	61%	61%	0%
Cu	87%	87%	0%
Mo	46%	0%	-
Smelting and refining			
Transport + downstream proc. costs	$0/t Milled	$1.35/t Milled	-
Fixed overheads			
Admin and overhead unit cost ($/t milled) –	$1.00/t milled	$1.00/t milled	0%

The figures highlighted in red have been calculated throughout the more detailed studies contained within this Preliminary Assessment. LAL has completed a revised optimisation utilising these adjusted Case B revenue and cost inputs.

The mining cost was taken from the detailed cost estimation completed as part of this Preliminary Assessment. A cost differential between mining mill feed and waste was also built into the Case B optimisation based on hauling waste material to the Triangle Lake area utilising a preliminary haul profile measured from the natural topography.

The adjustment to milling costs is significant. This change is mostly associated with the removal of the magnetite separation and molybdenum concentration circuits within the process. The mill capital and operating costs have been adjusted to reflect this. In addition to this major adjustment the milling operating costs from Case A were revised to include further allowance for maintenance costs (increased opex costs by CAD 0.52/t) as well as other minor adjustments. The offsite transport and processing costs have been taken from the financial model calculations completed for the Preliminary Assessment base case and levied across each milled tonne. The capital costs have been based on Preliminary Assessment base case capital costs estimation. Sustaining capital has also been built into the Case B optimisation where this was not previously considered. All costs previously calculated in CAD have been converted to USD using an exchange rate of 0.85 USD to 1 CAD. The pit wall angle was reduced to 50 degrees to better reflect the acceptable final wall design parameters indicated by geotechnical studies completed previously for the project.

20.3.1 Case B Whittle Results for PEA Scenario

As the original optimisation was based on different drivers (metal prices and capital and operating costs) the results are not really comparable to those achieved with Case B.

In an attempt to present a comparable Whittle result to the most optimal Case B scenario (based on maximising project return) a pit selection and schedule scenario have been generated in Whittle using the Case B parameters to calculate the financial performance of this scenario. The selected schedule is presented in the following Figure 20-1.

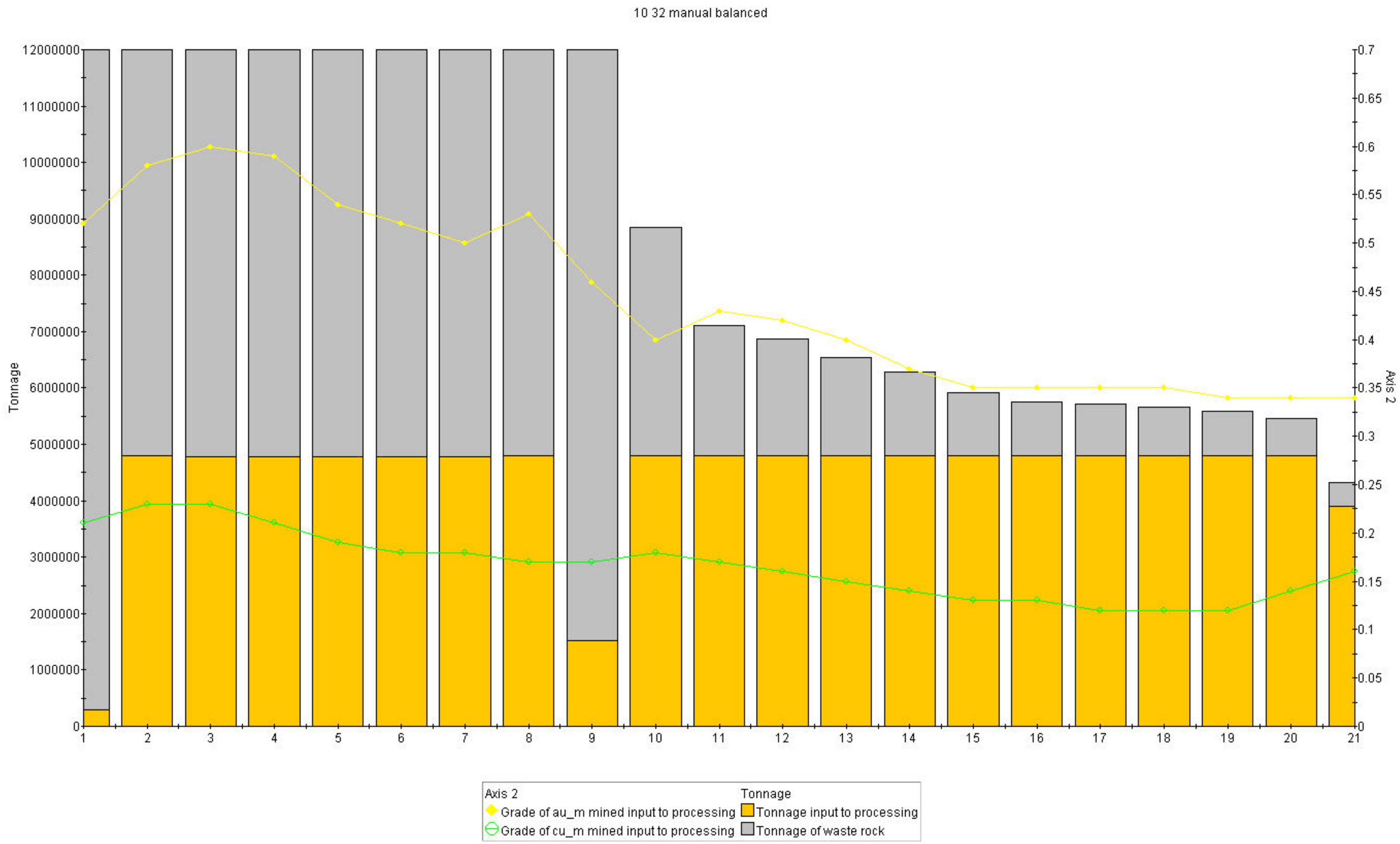


Figure 20-1: Comparable PEA Pit and Schedule Presented in Whittle

The schedule above was generated selecting matching initial and final pits to those designed for the PEA and running a schedule in Whittle based on the Milawa balanced option. It is not exactly the same as the one generated using detailed bench tonnes and grades however it is reasonable approximation given project limitations. The performance as calculated by Whittle for this scenario is shown in Table 20-3.

Table 20-3: Comparable Preliminary Assessment Pit and Schedule Performance

Movement	Tonne	Product	Input grade
Mill feed	92,040,417	Au (g/t)	0.44
Waste (reject)	3,377,218	Ag (g/t)	2.30
Waste (other)	86,649,905	Cu (%)	0.16
Total	182,067,540		
Measures	Specified	Measures	Specified
Strip Ratio	0.98	Payback (year)	6.23
NPV ($)	55,762,428	**IRR%**	**11.8**
Life (year)	20.81		

As the pit dimensions have been driven by different inputs (such as gold price) it was difficult to select an equivalent final pit. A similar mill feed production rate was used for this comparison however it is important to note the stripping ratio for this comparison sequence is higher than what was produced during the first phase of the PEA. Ultimately the NPV and IRR shown in the comparison scenario above is similar to that calculated through detailed pit design, scheduling, cost estimation and cashflow modelling for the project.

20.3.2 Case B Whittle Optimisation Scenarios

The following scenarios were reviewed as part of the optimization:

- **No Mining Limit** – 1 to 4 pushbacks were reviewed
- **8 mtpa mining limit** - Scenarios involving 2 to 4 pushbacks, various initial and final pits and various schedule types (e.g. maximising NPV or balancing resources) were reviewed
- **12 mtpa mining limit** - This scenario was completed to check the accuracy of the NPV and IRR calculations compared to what was determined through the more detailed pit design and cost estimation process. It involved selection of a production schedule representative of the pit shells selected in the original optimization. In addition to this original scenario comparison, 1 and 2 pushback scenarios were evaluated using different pit shells and with various schedules.

8 mtpa Results

The results for the best 8Mtpa pit sequence and schedule (highest IRR) are presented in Figure 20-2 and Table 20-4.

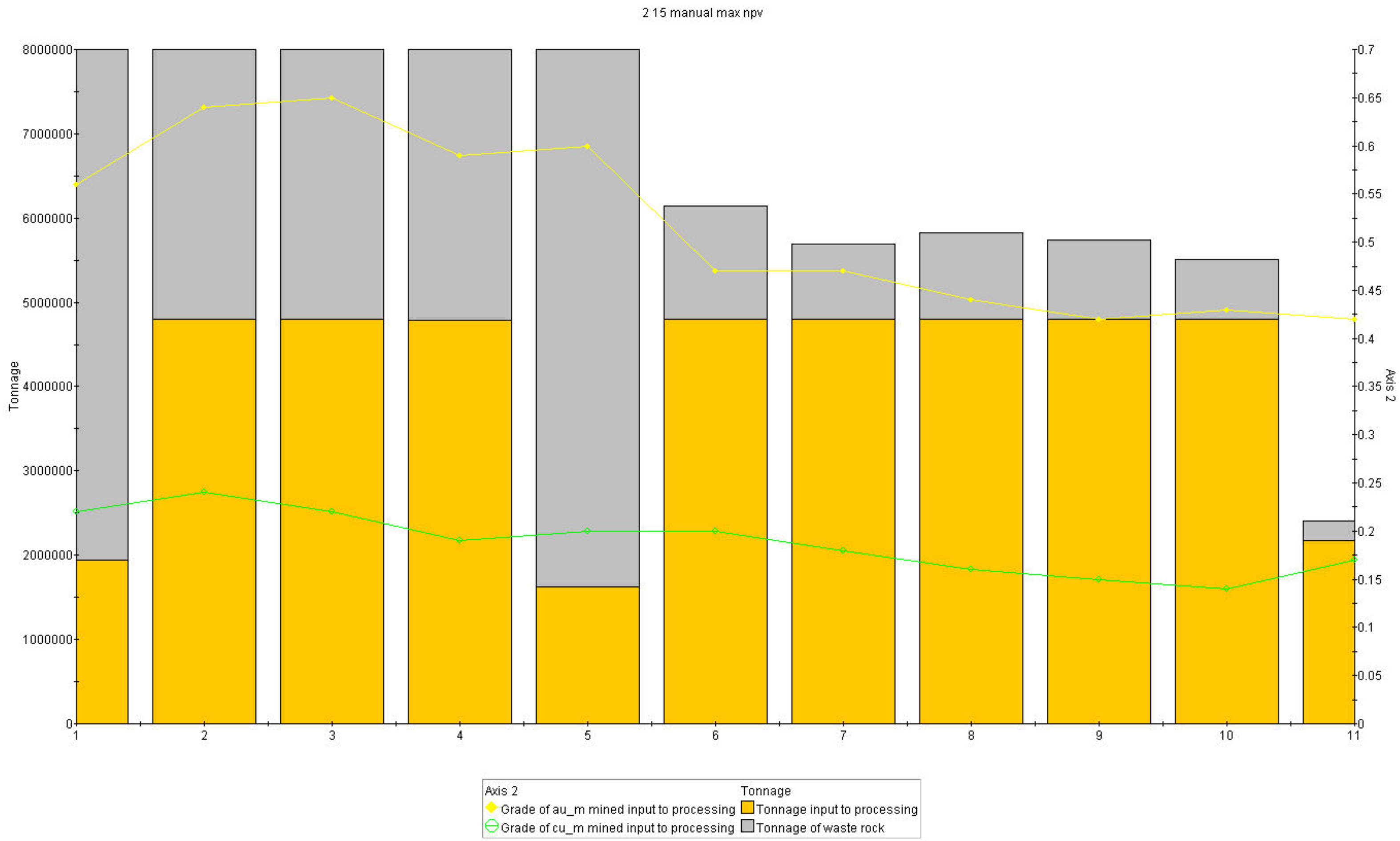


Figure 20-2: Best Case B 8mtpa Whittle Schedule

Table 20-4: Performance Summary for Best Case B 8mtpa Scenario

Movement	Tonne	Product	Input grade
Mill feed	44,116,714	Au (g/t)	0.52
Waste (reject)	443,993	Ag (g/t)	2.52
Waste (other)	26,744,673	Cu (%)	0.19
Total	71,305,380		
Measures	**Specified**	**Measures**	**Specified**
Strip Ratio	0.62	Payback (year)	5.23
NPV ($)	58,956,671	**IRR%**	**14.22**
Life (year)	10.45		

The 8Mtpa mining scenario is not optimal for the following reasons:

- Strip in advance is not quick enough presenting a problem for continuous mill feed at 5Mtpa.
- In the later years of the mine life it is optimum to mine and crush at a rate of 6Mtpa. If during the early years a capacity of 8Mtpa is required an additional 2Mtpa fleet would be required. A fleet of this capacity would be too small to work safely and effectively with 10m mining benches.

12 mtpa Results

Throughout the Case B optimisation process it was proven that the most profitable operating scenario is to mine at an all in production rate of 12mtpa throughout the first years of the mine life to reduce the stripping requirements at a later stage in the project. The most optimum starter pit contained approximately 13.7Mt of mill feed with a final pit containing 33.2Mt for a total of 46.9Mt of mill feed at a stripping ratio of 0.65. The schedule and performance as calculated by Whittle are provided in Figure 20-3 and Table 20-5 below.

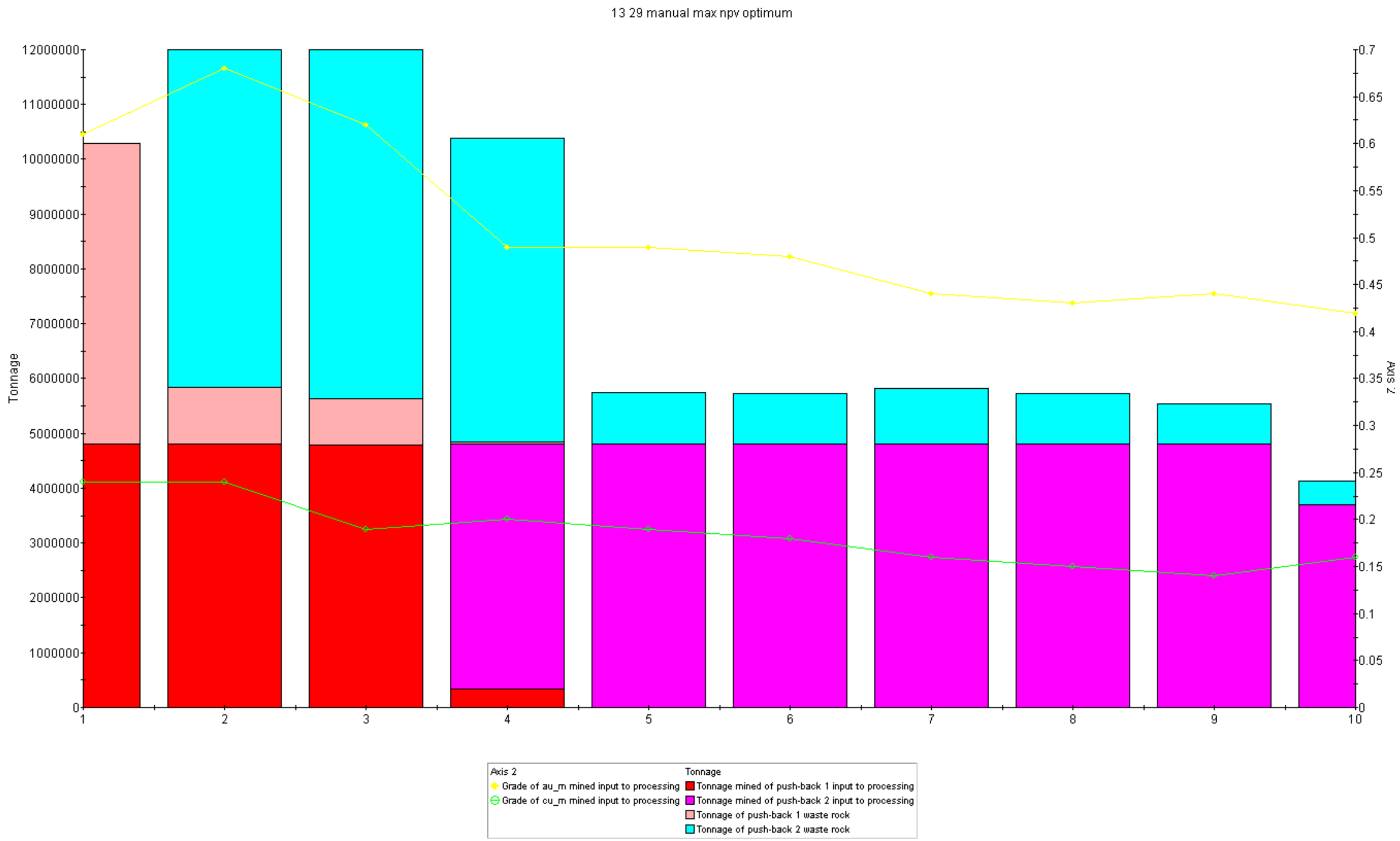


Figure 20-3: Whittle Case B Optimum Schedule

Table 20-5: Case B Optimum Pit and Schedule Sequence (12mtpa)

Movement	Tonne	Product	Input grade
Mill feed	46,896,799	Au (g/t)	0.51
Waste (reject)	535,333	Ag (g/t)	2.49
Waste (other)	29,945,008	Cu (%)	0.19
Total	77,377,140		
Measures	Specified	Measures	Specified
Strip Ratio	0.65	Payback (year)	4.14
NPV ($)	81,914,424	**IRR%**	**18.00**
Life (year)	9.77		

20.3.3 Summary of Post Preliminary Assessment Pit Optimisation

In Summary the Case B whittle runs provide an updated optimization with more accurate revenue and cost inputs. The Case B optimisation has applied a range of realistic mining rates for the project. Operationally 12mtpa makes sense as this will mean 2 equal sized inpit fleets. If 8mtpa was selected the mine would need to use an oversized non standard crusher or have a 6mtpa crusher and an additional 2mtpa crusher for a couple of years. This would introduce more complex equipment selection and application issues.

The most economic Whittle scenario from an IRR point of view is the 12mtpa scenario. Further analysis was completed to identify the most optimum initial and final pit size with consideration for IRR and payback. The optimum scenario is an initial pit of 13.7Mt of Mill feed (S/R of 0.62) and a final pit of 47Mt (S/R of 0.65 inclusive of initial pit tonnages). The resulting IRR has been calculated by Whittle as 18.0% with a payback of around 4 years.

LAL recommend proceeding with a re-design of the initial and final pits, re-run the production schedule, and revise the equipment selection, capital and operating cost estimates, revise the project cash flow model and re-submit the Preliminary Assessment report inclusive of these adjustments prior to commencement of the next phase of engineering studies.

20.4 Magnetite Recovery

Magnetite potential of the QM zone has yet to be fully determined. The metallurgical report prepared by Process Research Associates Ltd. “Metallurgical Study on the Bronson Slope Samples” dated July 1997, indicated an average head grade for a composite sample taken from the Bronson Slope QM (Quartz Magnetite) zone of 7.48% iron. Further research by BC Mining Research Ltd. indicate in a letter Progress Report dated September 18, 2008, and supported by discussion with the reports author Dr. Bern Klein, Ph.D., shows that an estimated 7% of a composite test sample from the Bronson Slope QM zone could be recovered as a high quality magnetite.

The progress report identified that the price paid by a North American consumer of magnetite for dense media separation/refining of coal was USD 211 per tonne. If magnetite recovery equivalent to 7% of mass of the QM zone resource is achieved, then based on a market price of USD 210 per tonne for high quality magnetite suitable for dense media separation, magnetite in the QM zone could potentially yield an economic benefit to the project. Further study of the magnetite potential including development of a magnetite resource for the Bronson Slope deposit is recommended.

21 Interpretation and Conclusions

The author has completed a Preliminary Assessment for the Bronson Slope property based on a wide variety of data, observations and previous technical reports.

Reference is made to previous independent Technical Reports on the Bronson Slope Deposit filed on SEDAR that establishes a mineral resource estimate for the Bronson Slope Project. The Technical reports are as follows:

"Technical Report for Skyline Gold Corporation on the Bronson Slope Property North-western British Columbia, Canada", dated June 1, 2006, authored by A. A. Burgoyne, P.Eng, M.Sc, from Burgoyne Geological Inc., an independent Qualified Person as defined by NI 43-101. This Technical report was posted to SEDAR on June 21, 2006.

"Technical Report Mineral Resource Estimate – Bronson Slope Deposit for Skyline Gold Corporation Vancouver, BC on The Bronson Slope Property North-western British Columbia, Canada", dated May 10, 2007, authored by G. H Giroux, and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., both independent Qualified Persons as defined by NI 43-101. This Technical report was posted to SEDAR on May 29, 2007.

"Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical report can be viewed at www.sedar.com. The mineral resource estimate that forms the basis for this preliminary assessment is the one presented in this report using Case 2 metal prices. Please refer to Item 19 for more details

These technical reports have provided a technical review of the Bronson Slope property including a detailed review and evaluation of the historical resource estimations for the Bronson Slope Au-Cu-Ag-Mo deposit. The preparation of these technical reports included certain due diligence procedures. The authors of these reports concluded that the technical fieldwork, and office data compilation, including historical resource estimation procedures, diamond core drilling, analyses, and reporting of data, completed by SGC, is of good quality and meets good practice industry standards.

In order to prepare capital and operating cost estimates for the mining and processing of the Bronson Slope orebody a number of budget prices and quotations were obtained from various suppliers for major items such as large mining equipment, major milling and processing components, major infrastructure development such as access roads and other consumables such as power and diesel. LAL have reviewed these budget prices and quotations against other similar projects in the past 12 months and subsequently believe they are within acceptable variance for this study.

A number of other interpretations and conclusions for the Bronson Slope property have been provided in items within the body of this report. It is the author's intention the report will be read in full to ensure full comprehension of all relevant interpretations and conclusions.

22 Recommendations

After completion of the post Preliminary Assessment Pit and conceptual LOM schedule optimisation using more detailed cost and revenue inputs it was identified that a smaller final pit may result in a higher IRR. LAL recommends that a Preliminary Assessment Update (PAU) is conducted based on the results identified in the 2nd Whittle optimisation. A budget estimate for completion of this PAU has been provided in Table 22-1.

Table 22-1: Estimate for Preliminary Assessment Update

Preliminary Assessment Update Estimate	Senior Mining Engineer	Mining Estimator	Mining Engineer
Item		**Days**	
Complete pit designs from whittle shells	3	0	0
Complete mine schedule	1	0	3
Complete basic mine site layout plan	1	0	1
Update infrastructure estimate to reflect Dennis review	1	0	0
Update mining and processing cost estimates	0	1	1
Financial modeling and sensitivity analysis	0	0.5	2
Compile technical report	2	0	2
Project Management and Review	2.5	0	1
Total time in days	**10.5**	**1.5**	**10**
Rate (USD /day)	$1,450	$1,450	$1,150
Total price for the job by resource (USD)	$15,225	$2,175	$11,500
Total Estimated Job Cost (USD)			**$ 28,900**

Also as part of the PAU a gap analysis should be conducted to identify what further studies and investigations are required prior to the completion of a feasibility study. These studies should be presented within the recommendations of the PAU.

Recommendations made throughout the body of this report include:

- Outline what studies may need to be completed as part of a more detailed feasibility study (GAP Analysis). Some areas that will need to be considered are:
    - Tailings Storage Facility Design and Evaluation
    - Waste Storage Facility Evaluation and Design
    - Detailed Pit Optimisation with consideration of the costs identified throughout the PEA process
    - Complete an update to the preliminary assessment cash flow model and sensitivity based on the revised mine plan
    - Environmental Impact Assessment and Licensing
    - Consultation with local community groups.
- **Road Access** - A proper assessment of avalanche risk is required to decide whether the main site access road should be built to the North or South of Bug Lake. This should include:
    - Time-statistical observation data,
    - Probability cost analysis of trail cleanup and

    - Costs involved in the interruption and delay of exploration

- **Slurry pipeline** – An opportunity may exist to transport Concentrate from Bronson Slope via a Slurry pipeline across the border into Alaska, significantly reducing the transport costs. A detailed scoping study evaluation needs to be conducted to identify the best approach for transport of concentrate to the market.
- **In-pit Conveyor** - Further more detailed studies are recommended to identify the impact of conveyor moves on mine productivity.
- **Crusher Selection** – Further scoping studies need to be conducted to evaluate the most appropriate rock crushing and transportation solution for the Bronson Slope Project.
- **High Angle Conveyor System viability** – A more detailed scoping study should be conducted by the suppliers of the HAC system proposed for the Bronson Slope property. Site specific operating conditions should be taken into consideration as part of this more detailed scoping study.
- **Ore pass material transport method** – It has been considered to be more risky however further more detailed studies are required to fully rule out this option as an ore and waste transport method.
- **Pit Geotechnical Constraints** - Further review of the geotechnical conditions within and surrounding the revised pit limits is essential to ensure that the most accurate modelling of geotechnical risks is completed prior to commissioning of the project.
- **Highwall Zone Gold Recoveries -** Further study needs to be conducted to determine whether the gold recoveries of the highwall zone are achievable with very low copper grades. An alternate plan may be required to stockpile the highwall zone material and progressively blend it into the mill feed to reduce the impact of the reduced copper grades.
- **Waste Rock Placement -** LAL recommends a study be completed to determine the viability of placing waste in the Triangle lake area or within alternative areas such as the TSF area or in Bronson Creek.
- **Acid Rock Drainage –** More testing is required to determine the quantities and characteristics of potentially acid forming waste rock at the Bronson Slope project.
- **Below Cut-off Grade Stockpiling -** Complete a study on selective rehandling and processing of below cut-off grade material from the waste storage area at the end of mine life for further processing and economic benefit
- **Grind Size vs. Recovery –** Further study is required to optimise the grind size with respect to metal recoveries.
- **Power Generation -** Further develop self-generation hydro projects as the basis of power supply allowing a Feasibility Study undertaking

  Magnetite Resource – A 43-101 compliant magnetite resource for the QM zone should be developed and further investigation of the magnetite and molybdenum recovery circuits impact on project economics should be completed

23 References

The following is a list of references utilised for the purposes of completing this Preliminary Economic Assessment. In addition to these referenced reports SGC have conducted a number of evaluation and engineering studies previously which have been listed in Table 20-1.

Alldrick, D.J., Britton, J.M., Maclean, M.E., Hancock, K.D., Fletcher, B.A., And Giebert, S.N. :1990. Geology and Mineral Deposits – Snippaker Area. B.C. Ministry of Energy, Mines and Petroleum Resources Open File Map 1990-16.

Anderson, R.G., 1993: A Mesozoic Stratigraphic and Plutonic Framework for North western Stikinia (Iskut River Area), north western British Columbia in Mesozoic Paleography of the Western United States, edited by G.Dunne and K. MacDougall Society of Economic paleonotologists and Mineralogists, Vol II, Pacific Section.

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Piteau 1997: Conceptual Design of Tailings Facility dated January 1997, Piteau Engineering

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Ganshorn, A. J., 1997: Molybdenum Marketing Report 1997 dated May 1, 1997, J. Arthur Ganshorn (Marketing Consultant).

Giroux, G.H., 1996b: A Geostatistical Resource Evaluation on Bronson Slope Project for International Skyline Gold Corporation dated September 30, 1996, amended October 8, 1996.

Klein, B., 2008: Metallurgical Study on High Wall Gold Zone of the Bronson Slope Deposit, prepared for Skyline Gold Corporation, BC Mining Research Limited and dated January 16, 2008

Klein, B., 1997: Bronson Slope – Penalty Element Assays, 1997, dated June 17, 1997, Bern Klein (Process Research Associates).

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Yeager, D., 2000: Operating Cost Estimates dated April 12, 2000, David Yeager.

Also see Table 20-1: Previous Studies and Evaluation Reports in Item 20 for further details of these studies.

23.1 Reference for Costing, Estimating and Miscellaneous Data

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23.2 Equipment Specifications

23.2.1 Mining

CAT 990H Specifications – see www.cat.com

CAT 777F Specifications – see www.cat.com

Komatsu PC 1250LC-8 Specifications – see www.komatsu.com

High Angle Conveyor System – see www.innovativeconveying.com

Other Caterpillar equipment – www.cat.com

23.3 List of Company Contacts and Associated Products

Consep Pty Ltd – Knelson separator

Conveyor Design Consultants – High angle conveyor

Dos Santos International – Sandwich high angle conveyor

Dyno Nobel – Explosives facilities

F.L. Smidth Minerals Ltd – Tech-Taylor valve

Innovative Conveying Systems International (ICS) – Enclosed High Angle Conveyor System

Krebs Engineers - Cyclones

Larox Inc – Larox filters

Metso Minerals – Mobile Crusher, SAG, Ball and Regrind Mills, Apron Feeders, Magnetic Separators

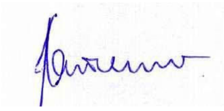
Outotec – Flotation Cells

Thermo Fisher Scientific – Sampler and on stream analyser

Finning (Canada) – Caterpillar Equipment

24 Date and Signature Page

The report titled "Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation on the Bronson Slope Property" dated 4th of March, 2009 was completed and signed by the authors defined below.



J. A. R. Lawrence, MAusIMM (#209746)

Leighton Asia Limited

4th of March, 2009

V. Seen, MAusIMM

Leighton Asia Limited

4th of March, 2009

STATEMENT OF QUALIFIED PERSON

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julien.lawrence@leightonasia.com

I, Julien A. R. Lawrence, MAUSIMM, am a Senior Mining Engineer for Leighton Asia Limited (LAL), located at 39th Floor, Sun Hung Kai Centre, 30 Harbour Road, Hong Kong.

I graduated with first class Honours from the University of Queensland with a Bachelor of Engineering degree in Mining in 2000. I am a member of the Australian Institute of Mining and Metallurgy (#209746).

Since 2000 I have practiced continually as a tertiary qualified Mining Engineer working in base metal, ferrous and precious metal mines throughout Australia and Asia.

I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101 for this report. I have mining engineering experience on several porphyry deposits including the Ernest Henry deposit in Australia and the Xietongmen deposit in P. R. China.

I have visited the Bronson Slope Property on 21ST AND 22ND of July 2007. I am responsible for preparation of all or a portion of Items 1 to5, 7 and Items 19 to 26 of the Technical Report titled “**Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation on the Bronson Slope Property**” with an effective date of 4th of March, 2009, relating to the Bronson Slope Property.

As of the date of the certificate, to the best of the qualified person’s knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

I am independent of Skyline Gold Corporation in accordance with the application of Section 1.4 of National Instrument 43-101.

I have no prior involvement with the Property that is the subject of this report.

I have read National Instrument 43-101 and Form 43-101FI, and this report has been prepared in compliance with that instrument and form.

Dated at Beijing, P.R China, this 4th day of March, 2009.

Julien A. R. Lawrence, MAUSIMM.

STATEMENT OF QUALIFIED PERSON

Victor Seen PM, BSc (Extractive Metallurgy), MAUSIMM

I, Victor Seen PM hereby certify that:

- I am an independent consulting process metallurgist/engineer employed by Leighton Asia Limited with a business address of 39/F, 30 Harbour Road, Hong Kong.
- I am a graduate of the University of Murdoch in Western Australia, Australia, with a Bachelor of Science in Extractive Metallurgy, (1993).
- I am a member of Australian Institution of Mining and Metallurgy.
- I have practiced my profession metallurgist for over 13 years since my graduation from the university.
- I have been involved in production of mineral processing, metallurgical pilot plant projects and pre-feasibility test work.
- I have read the definition of “qualified person” set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a “qualified person” for the purposes of NI 43-101.
- I have visited the Bronson Slope Property on 21st and 22nd of July 2007. I am responsible for preparation of all or a portion of Items 3.7, 3.9, 7.7, 7.9, 20.2 and 25.8 of the Technical Report titled “**Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation on the Bronson Slope Property**” with an effective date of 4th of March, 2009, relating to the Bronson Slope Property. I have not had prior involvement with the property that is the subject of the Technical Report.
- 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 this report not misleading.

Dated at Perth, Western Australia, this 4th day of March, 2009.

Victor P.M. Seen, MAUSIMM.

25 Additional Requirements

25.1 Mining Operations

25.1.1 Available Hours and Utilization

For the purposes of this study Availability can be defined as the time a machine is required and able to work divided by the calendar time during the period of interest. Utilization is the time a machine is actually used divided by the calendar time during the period of interest. Thus the two have shared components, where utilization can be calculated as a percentage of the available time per shift using the mechanical availability factor which includes normal mandatory delays (such as maintenance, support, installation, etc). The following Items aim to clarify the theory used to determine equipment availability and utilisation for the Bronson Slope Project.

25.1.1.1 Availability

In this study Availability does not measure total downtime. It measures the degree to which the maintenance personnel are able to support production by ensuring that equipment is up and running for as long as possible during a defined production shift. Availability can also be much improved by performing all planned and preventative maintenance actions off shift. High levels of availability are thus relatively easy to achieve if the machine is required to work only a portion of the total hours in a day or week. This theory of detailed maintenance planning and operating delay management has been adopted for this study of the Bronson Slope Project.

For a conservative estimate, a general 85% mechanical availability will be used to account for maintenance, support, installation, etc. For critical high performance equipment higher availability estimates have been used based on a more extensive and well planned preventative maintenance and support program (maximum is 90%).

25.1.1.2 Utilization

The total utilization factor can be affected by:

- Weather – rain and snowfall that can disrupt production
- Holidays – where equipment operation may stop
- Work stoppages – such as strike, union disagreements, etc
- Miscellaneous lost time – such as meetings, safety, training sessions, etc.

It is assumed that the operation will not stop during public holidays.

Based on the study carried out in the World Socialist Reports titled “Work Stoppages in North America”, Canada has a percentage lost time of 0.07% - 0.14% of total working time. This study was carried out from 1989 to 2003 and shows a general downward trend. To be conservative though, 0.14% will be used.

Other delays such as meetings, fortnightly safety toolboxes, training sessions, etc. are allocated 3-5 days of delay per year. The final total delays and resulting utilization are summarised in Table 25-1. It is envisaged that, with appropriate planning, 6 days can be utilised for opportune maintenance on the mining equipment reducing the burden.

Table 25-1: Summary of Delays and the Resulting Utilisation

Delay	Days	Opportune Maintenance	Nett Days
Weather	28	3	25
Cleanup	3	0	3
Work Stoppages	0.5	0	.5
Misc	5	3	2
TOTAL	**36.5**	**6**	**30.5**
Utilisation	**90%**		**91.6%**

25.1.1.3 Effective Working Hours in the Day

For every shift, it is assumed that at least 45 minutes is lost for meal breaks and more time is lost for less critical mining equipment, and another half hour is lost for shift change. This equates to 9.75 effective hours per 12 hour shift. Thus, 19.5 effective hours per day is used. The excavators will have fewer delays as an intra-shift operator rotation will be conducted to reducing the delays associated with meal breaks. An allowance of 30 minutes per shift has also been made to account for lost time due to blasting. Blasting will take place approximately 3 times per week on day shift only. Delays to production on blasting days will exceed 30 minutes however on non blasting days there will be no delay and therefore the average 30 minute delay per shift is acceptable.

25.1.1.4 Machine Working Minutes per Hour

For calculations on machine productivity, 50 working minutes per hour is adopted for an allowance for work related lost time such as short tramming, short delays due to visibility (dust), cleanups and other miscellaneous short delays.

25.1.2 Material Transport Methods

As part of the mine planning review for this Preliminary Assessment several types of material transporting methods were compared for suitability to the Bronson Slope project. The final recommendation is based on a combination of factors including capital and operating cost, suitability to conditions (geographic, terrain, climate, etc), environmental impacts, cost sensitivities to market value of commodities (such as electricity and fuel) and compliance with regulation requirements. To help simplify the review cost comparisons the material transport was broken into three sections. These sections are:

a. In pit transport

b. From the pit to the stockpile

c. From the base of the orebody to the waste dump

25.1.2.1 In Pit Transport

The average in-pit hauling distance from the mining face to the bench centroid is approximately 300m, but can reach a haul distance of up to 650m. The maximum bench length is 1300m. The following scenarios were evaluated:

- **Load, Haul and Dump by Loaders only (LHD)** - This option uses solely LHD vehicles, such as rubber tire loaders, for digging and transporting material. This option is dismissed due to the high operating cost per tonne of ore. The average tramming distance is also beyond the recommended tramming distances for wheel loaders, - which operate most efficiently over a haul distance of 10 to 160 metres (Caterpillar Handbook Ed. 26).
- **Conventional Load and Haul** - A scenario using mid sized excavators and matching haul trucks was reviewed and a cost estimate for the hauling of material from the face to either a conveyor feed point or an ore pass was completed. This resulted in a higher unit

cost due to the cost of labour and consumables such as diesel for the relatively small haul trucks.

- **In-Pit Mobile Conveying System** - Using an average distance of 300m, eight 50m (horizontally) mobile in-pit conveyors may be utilized (spares for mobility). If there are two operating diggers, 16 conveyors may be needed. There will be additional labour costs associated with managing the operation and maintenance of the mobile conveyors and also regular clean up required at transfer points in the system. An ancillary loader will be required to provide this capability.
- **Enclosed Belt Conveying System** – Innovative Conveying Systems International (ICSI) have developed a conveyor system that is capable of providing a flexible feed end that is connected to a main High Angle Conveyor (HAC) system which will transport the material down the slope to the stockpile. This system is relatively cost effective to run and also provides some flexibility in the positioning of the conveyor. It also provides for a clean transfer of material to the main system. Further details of the HAC and its associated components are included in the next section.

After consideration of the above methods and also how the system will be utilised to transport the material from the pit bench to the stockpile, the most appropriate in pit material transport system for Bronson slope is the Enclosed Belt Conveying System, which is part of the HAC selected for Bronson Slope in the next section. The operating costs for this system are low, and the mine plan can be set to accommodate the flexibility issues that may arise from this type of system.

The ICSI HAC system is relatively new and has not been utilised in a mining environment to date. This conveyor system is being utilised in other industrial applications with high angle conveying requirements. Other mining companies are currently evaluating the system for applications within Australia.

The system has also not been used in very cold climates similar to the climate that will be experienced at Bronson Slope during the winter months. Further detailed studies are required to ensure the applicability and cost estimation for this system in the application and environment described.

25.1.2.2 Transport from Pit to Stockpile

A number of methods have been considered for transporting the material from the pit to the stockpile area (mill feed or waste stockpile). These are:

a. **High Angle Conveyor (HAC)** - For the steep terrain at Bronson Slope, it is possible to use a high angle conveyor system to transport the material downhill. Downhill HAC have various designs and can use traditional belts, high angle sandwich belts, bucket conveyors, ropecon conveyors and the formed belt conveyors. A regenerative, material weight drive system is preferred as the energy generated can be fed into the mains or used for controlled braking of the conveyor.

   In relation to the HAC, if mining was to start at the highest point of Bronson Slope and move downwards, then the total distance of the conveyor required will decrease as the mine progresses. This creates an opportunity for cost savings by replacing worn conveyor parts at the bottom with the ones no longer in use from the top. The change in demand in conveyor length throughout the LOM makes a mobile multiple modular conveyors more suitable to this operation.

   LAL has contacted Innovative Conveying Systems (ICS) and technical reports show promising potential for the use of new HAC technologies for this project. An image of the ICS conveying system and discharge end has been included in Figure 25-1 below:

Figure 25-1: ICS High Angle Conveyor Solution

The ICSI High Angle Conveying (HAC) system is an enclosed conveyor that uses a patented corrugated belt to allow flexibility in both the horizontal and vertical directions. The belt is supported on custom design idler assemblies. During angle transport, the belt is enclosed, while at loading and unloading points, the belt is 'unfolded'. The system is also modular and mobile. LAL believe the most appropriate transport method for material from the pit to the stockpile area is by using the ICSI developed HAC system.

b. **Truck Haulage** - Due to the safe operating conditions required for downhill haulage (ramp gradients of 8%, and restrictive speed limits) the option of using conventional dump truck haulage from the pit to the mill feed stockpile or waste dump is very expensive. A ramp network from the top of the pit to the base would extend beyond 7km and would require significant maintenance due to being located on the side of the mountain adjacent to the Bronson Slope. A broad comparative cost estimate was completed by LAL for the conventional truck haulage. The resulting haulage only cost was CAD2.20 per tonne. This method of material transport would significantly impact on the financial viability of the project and therefore was not selected.

c. **Ore Pass** - An ore pass system feeding onto an angled (or flat) conveyor has been the basis of most previous studies done on the Bronson Slope mine. Preliminary design consisted of a twin ore pass system within the centroid of the pit which then converges at the bottom of the pit to a central underground crushing facility. A schematic taken from an earlier study has been provided in Figure 25-2 below.

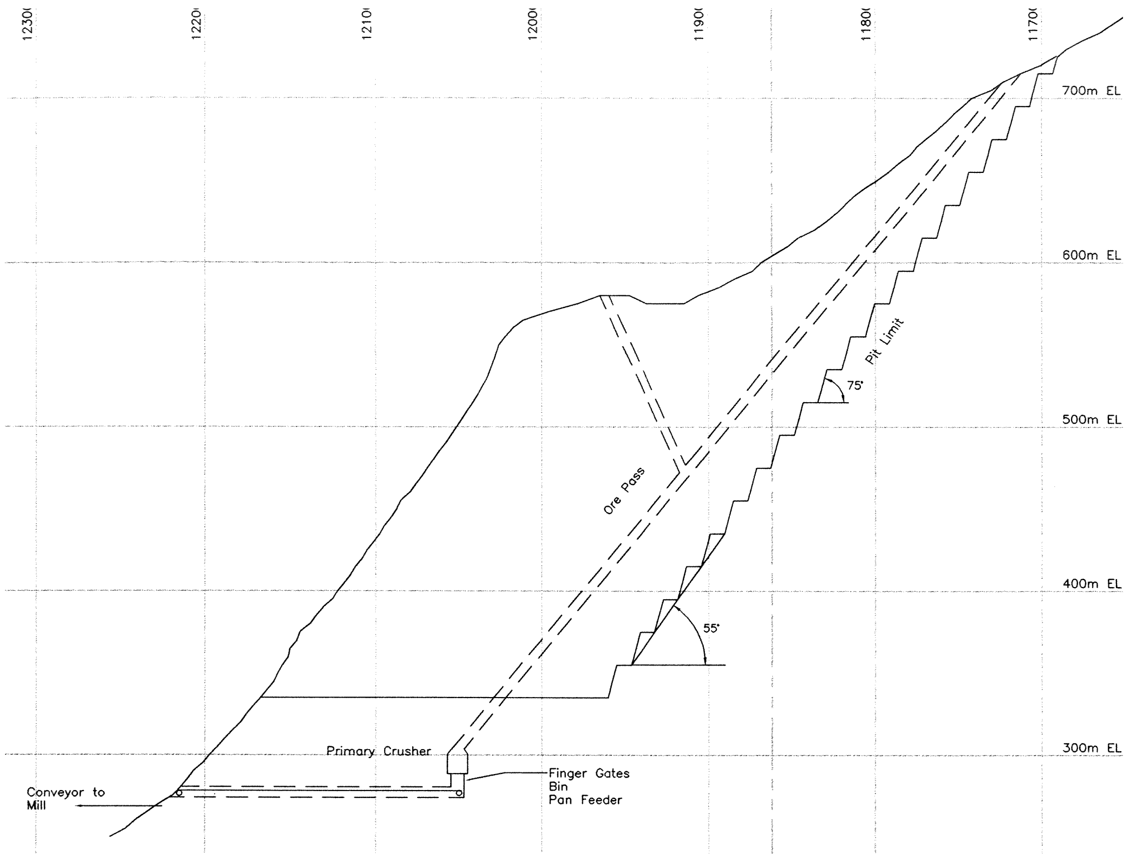


Figure 25-2: Ore Pass Concept

One of the major concerns with this design is that the second ore pass has to cover the entire cutback of the initial pit. The risk of a material hang up part way along the ore pass is significant over this length and will cut off mill feed supply to the processing facility. The ability to access a hang up to clear it, if required, would be expensive to establish and maintain. Without a secondary access the risk of cutting off mill feed supply to the concentrator is significant. This is the main reason why an ore pass type system has not been selected for the Bronson Slope project. However further more detailed studies are required to fully rule out this option as an ore and waste transport method.

d. **Dozer Push –** Consideration was given to the use of Dozers to push the material down the side of the hill to the base where it could be picked up and placed either on an mill feed stockpile or waste stockpile. A number of issues would need to be reviewed relating to safety, ore quality control and dilution and dozer capacity, amongst other things. This method has not been selected for the Bronson Slope Property due to the cost and also the complexity this material movement technique would introduce. In addition to these concerns this method is inherent with some operating and safety risk, which may be significant.

25.2 Mine Plan

25.2.1 Block Model Conversion

In order to complete the pit optimisation and evaluation the block model information provided by SGC (***Filename**: BRONSON_SLOPE_MAY2008_BLOCK_MODEL.csv*) needed to be converted into Gemcom Surpac format. A total of 33 attributes were created within the block model to cater for the existing attributes from the supplied file. Additional attributes were included to allow for appropriate optimisation attributes such as cost adjustment factors for mining and processing, and ore classification grouping attributes. Figure 25-3 is an image taken from Surpac showing blocks in the block model constrained by the final pit design and coloured by Cu grade.

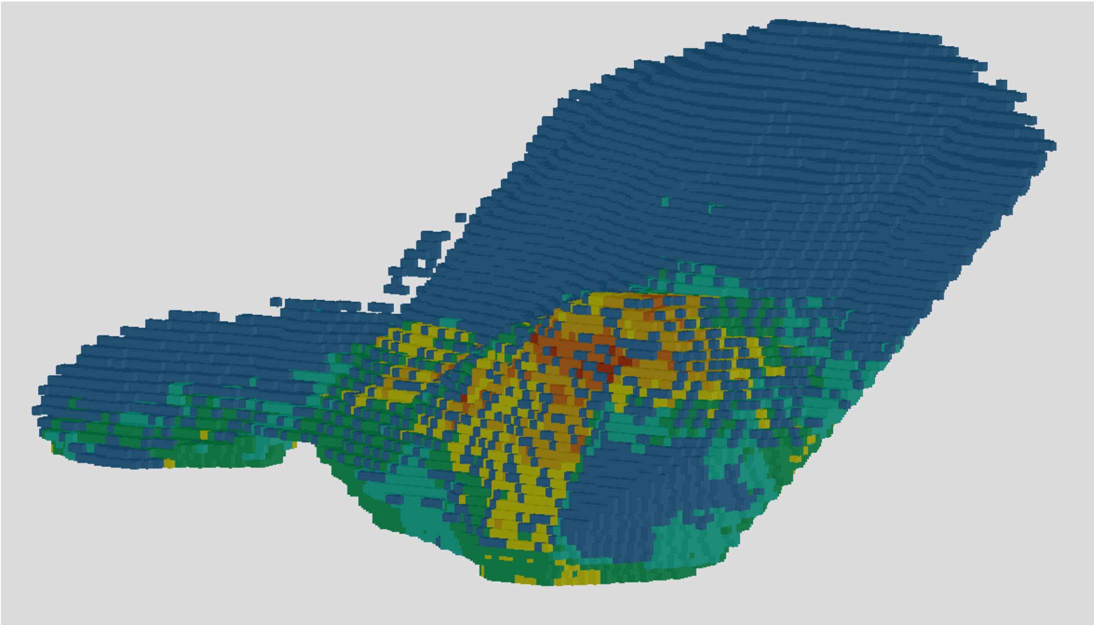


Figure 25-3: 3D Image of Final Pit Constrained Surpac Block Model

25.2.2 Resource Estimate Comparison

The resource estimate used for this study has been taken from the Technical report titled "Mineral Resource Estimate – Bronson Slope Deposit", dated April 30, 2008, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. This Technical report can be viewed at www.sedar.com. This resource estimate has been included in Item 19.

LAL performed a block model report calculation to determine how the imported block model tonnes and grades compare with this resource estimate. The Resource estimate calculated using the newly generated geology block model has been provided in Table 25-2. This table also details the difference between the new model and the supplied model (using the supplied model as a base).

Table 25-2: Surpac Resource Estimate USD 9.00 NRV cut-off for Case 2

Case 2 – Bronson Slope Resource Estimate (Cutoff USD 9/t NRV)					
Category	**Metric Tonnes**	**Au g/t**	**Ag g/t**	**Cu %**	**Mo %**
Measured	74,890,196	0.449	2.312	0.168	0.0059
Indicated	150,812,842	0.315	2.172	0.131	0.0087
Inferred	91,958,620	0.27	1.76	0.126	0.0080
Total Measured + Indicated	**225,703,038**	**0.359**	**2.22**	**0.143**	**0.0078**
Variance from Burgoyne and Giroux 2008 Resource Estimate					
Measured	0.1%	-0.2%	0.1%	-1.2%	0.0%
Indicated	0.3%	1.6%	0.1%	0.8%	0.0%
Inferred	0.4%	0.0%	0.0%	-3.1%	0.0%
Total Measured + Indicated	**0.3%**	**-0.1%**	**-0.1%**	**2.3%**	**0.9%**

It can be seen from this comparison that there is insignificant difference between the resource estimate within the block model developed by Burgoyne and Giroux (2008) and the one completed after importing into Surpac. All pit optimisation, design and reporting of grades, volumes and tonnages throughout this study have been completed using the Surpac block model and the Case 2 metal prices. The mill feed cut-off has been calculated based on a USD 9.00/t NRV.

25.2.3 Pit Optimisation and Strategic Planning

The process of pit optimisation and strategic scheduling are linked in the following sequence:

Whittle – Pit Optimisation

- Using Whittle Model run base case pit optimization
- Select cutbacks (mining areas) using optimisation functionality of software
- Run various schedules to optimise material movement within the cutbacks
- Produce pit shells and end of year face positions to review in Surpac mine design software

Surpac - Pit Design and Reserve

- Design pits with ramps and appropriate geotechnical conditions
- Develop reports from block model of pit shells and contained volumes and grades of various material types
- Build available grades and tonnage information into scheduling template

Fleet Selection and Scheduling

- Produce first pass material movement schedule to achieve optimum mining sequence (based on grade and stripping ratio)
- Determine mining fleets that can achieve the required production schedule
- Use haul profile simulations, productivity and cost analysis to determine the optimal fleet
- Re-enter parameters for selected optimal fleet (haul cycles, productivities, etc) into production schedule
- Determine theoretical production fleet numbers (trucks, excavators/shovels, drills,) required from production schedule
- Smooth out equipment numbers required
- Determine ancillary equipment required

Throughout the mining industry a number of software packages have been developed based on proven historical calculation techniques to help identify the most profitable way to develop an insitu resource. LAL have adopted a software package called Whittle to perform the pit optimisation for this mine plan and cost estimate.

The Whittle system uses the Lerchs-Grossmann algorithm developed by H. Lerchs & I. F. Grossmann. Put simply a net value of each block in a block model is calculated and then based on pit slopes and other defined mining constraints the blocks are progressively mined top, down. The resulting pit outline will be defined by the blocks that give the highest combined net return whilst adhering to all imposed constraints.

The whittle optimisation process then works to develop a set of nested pit shells based on application of scaling revenue factors. The expected revenue for each block is then multiplied by these factors and run through the LG algorithm to generate a series of pit shells representative of the revenue factors. This series of nested pit shells are then used to help identify the most effective sequence to mine the ore body and what is the ultimate pit limit.

25.2.3.1 Summary & Sequence of Pit Optimisation Process

Whittle software requires a different format block model in order to complete the pit optimization. The whittle block model is evaluated to ensure that it accurately reflects the same grades and tonnages of the client supplied and publicly available resource estimate. Next a base case operational scenario is set up to generate a series of pit shells based on the LG theory previously described. A representative graph of the base case scenario has been included in Item 25.2.3.3. This base case scenario is then evaluated through a series of iterations to identify the best financial performance for the project. This involves evaluating the most appropriate number of pushbacks (pit shells or phases), mine life, production rates and strategic scheduling of the benches within the mining phases. The most optimum sequence and schedule is then selected and the results output to other evaluation software for further analysis and pit design review.

25.2.3.2 Inputs and Drivers for the Pit Optimisation

Various pit optimisation scenarios have been run to gain a better understanding of the key drivers around mining the Bronson Slope ore body using open pit methods. The scenarios are also focused around identifying the most optimum ultimate pit and developing a high level strategic schedule representative of the best return for the project that can physically be achieved.

In order to generate these scenarios a series of physical and financial inputs and drivers are required for use in the calculation. Table 25-3 details the values of inputs used for the pit optimisation scenarios initially used for this study.

Table 25-3: Optimisation Parameters for the Bronson Slope Project

Optimization Parameters for Bronson Slope Project		
Parameter Description	**Value & Comments**	
Long term metal prices		
Cu (USD/lb)	2	Supplied by SGC
Au (USD/oz)	650	
Ag (USD/oz)	12	
Mo (USD/lb)	12	
Capital (USD)	$175,000,000	Supplied by SGC
Residual	10%	
Discount rate	8%+/-	
Geotechnical design parameters		
Average - All domains	55 degrees -->	Adapted from Piteau Report
Mining parameters		
Bench Height (m)	10	
Mining cost - Base (USD/t)	$1.50/t	
Mining dilution (%)	5%	
Mining recovery (%)	95%	
Concentrator parameters		
Expected Mill Throughput (t/annum)	5.5Mtpa	
Variable processing cost (USD/t milled)	$6.00/t	
Concentrator Recoveries		
Au	84%	
Ag	61%	
Cu	87%	
Mo	46%	
Smelting and refining		
Smelting recovery(s) (%)	97%	
Refining recovery(s) (%)	100%	
Smelting cost (USD/t concentrate)	$85/t	
Refining cost(s) (USD)	$0.075/lb Cu $6.00/oz Gold, $0.4/oz Silver	
Concentrate Transport costs (USD/t)	$50/t of concentrate	
Concentrate moisture content (%)	8%	
Marketing cost(s)	Nil	
Sales cost(s)	Nil	
Sales commission(s) (%)	Nil	
General and Administration Overhead costs		
Admin and Overhead unit cost (USD/t milled) –	$1.00/t milled	

It can be seen that at the time of the optimisation a processing rate of 5.5Mtpa was considered. This has since been revised to 5.089Mtpa based on the process flow sheet and also on the expected availability of the concentrator. The effect of this small change in processing rate on the resulting optimised pit and schedule is considered to be insignificant for this study, however further revisions of the pit optimisation are recommended for the next phase of the project. Representative mining, milling, general and administration costs and also capital costs were selected prior to completion of the study for use in the optimisation. These have since been reviewed through a more detailed cost estimating process.

25.2.3.3 Pit Optimisation Results

A number of financial performance measures are used to evaluate each scenario within the pit optimisation software. These are Net Present Value (NPV), Internal Rate of Return (IRR), Payback life, and Payback ratio.

A summary of the pit optimisation results follow however a series of graphs and chart outputs from Whittle showing more detail behind the selection of the optimum pit shell sequence and schedule for all scenarios is available on request.

Figure 25-4 is a graph that shows the mill feed and waste tonnes for each of the progressive pit shells and the indicative best and worst case NPV's for that pit shell for the base case scenario (unlimited mining production rate, 5.5mtpa milling rate).

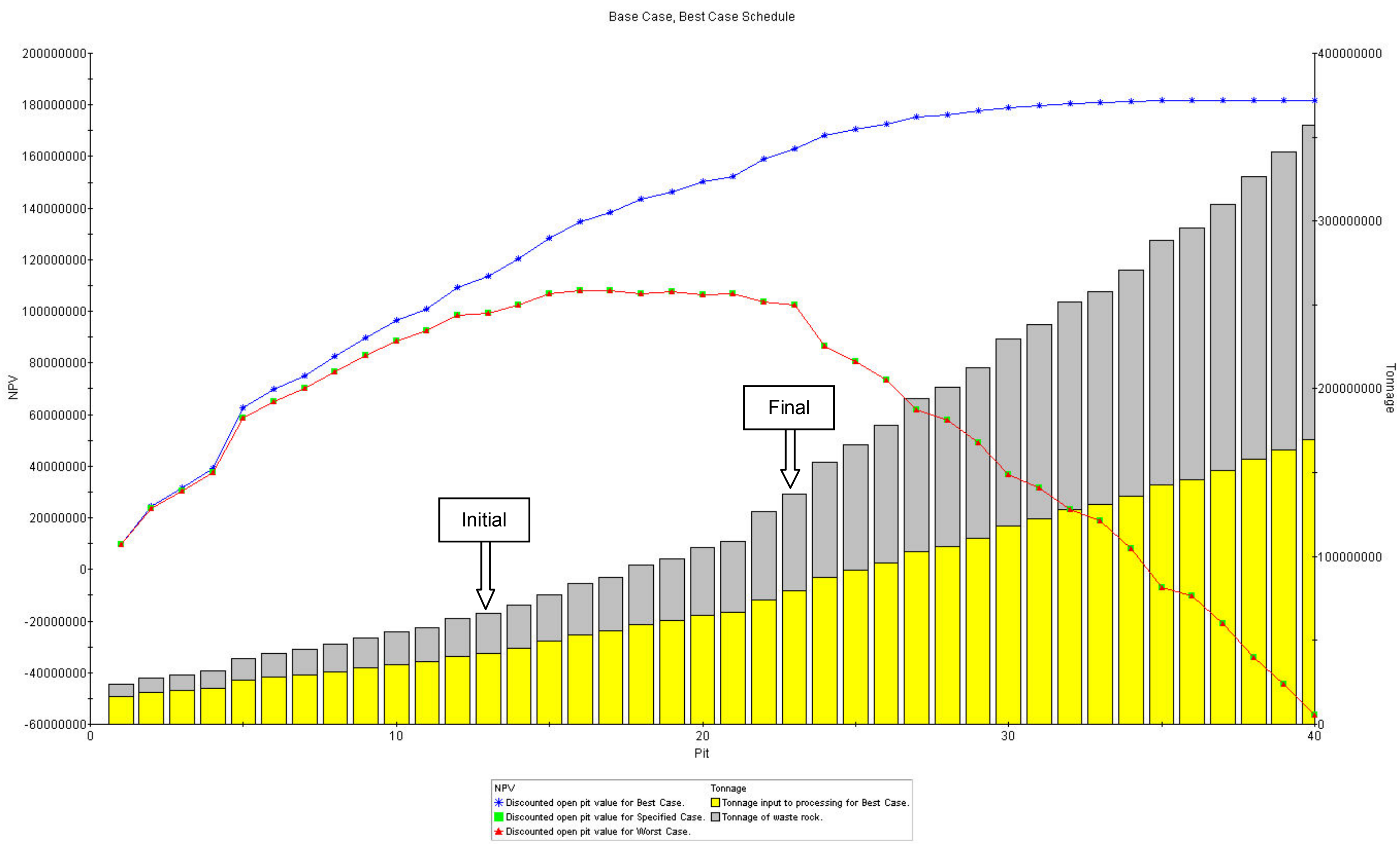


Figure 25-4: Base Case Pit by Pit Graph

The pit optimisation completed for this Preliminary Assessment used NPV as the key performance measure. IRR, payback period and mine life were secondary measures.

With appropriate scheduling in most cases an NPV can be generated somewhere close to the average of the worst case and best case scenario NPV's. This is dependent however on the number of push backs, and the mining sequence that is possible given the selection of mining equipment for the project, amongst other factors.

More detailed performance evaluation was completed considering a mining phase variants of 0, 1 and 2 push backs. The results of these scenarios indicated the most appropriate number of pit shells is 2 (starter pit and final pit pushback). 3 phases resulted in marginally better financial performance however after reviewing the pit shells more closely the mining width between the 2nd and 3rd phase is too narrow to achieve realistically using the mining equipment selected so therefore the two phase scenario was adopted for the Bronson slope project. A strategic life of mine schedule was generated and the financial performance indicators reviewed to determine the most optimum pit. A summary for the optimum pit has been included in *Table 25-4*below:

Table 25-4: Optimum Pit Sequence Summary

Measure	Value
Mill feed (t)	87,342,491
Waste (reject) (t)	2,265,513
Waste (other) (t)	66,848,726
Total (t)	156,456,730
Life (yr)	17.9

25.2.3.4 Pit Shell and Strategic Shell Progression

Figure 25-5 to *Figure 25-6* shows the starter pit and the final pit shell that were the selected optimum shells.

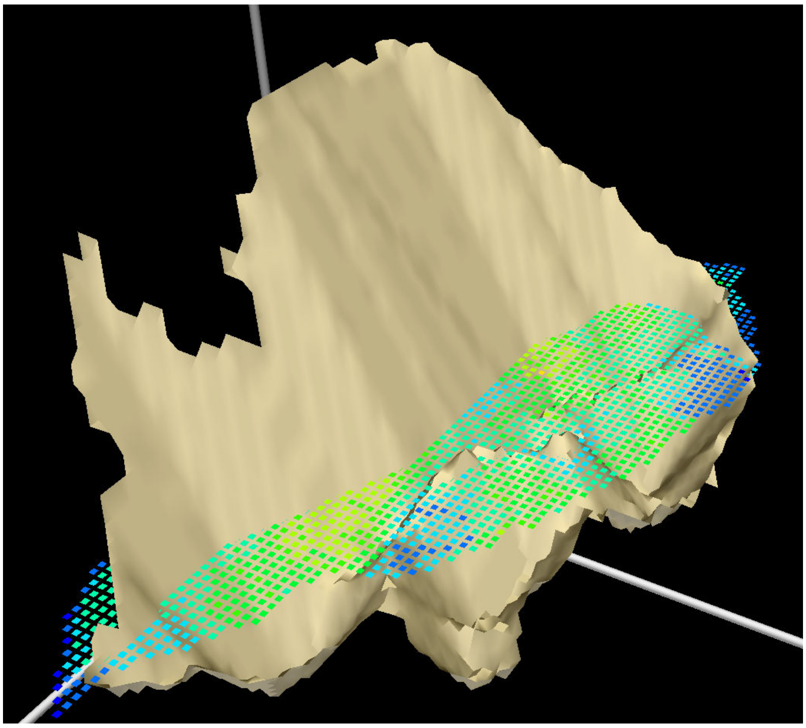


Figure 25-5: Starter Pit

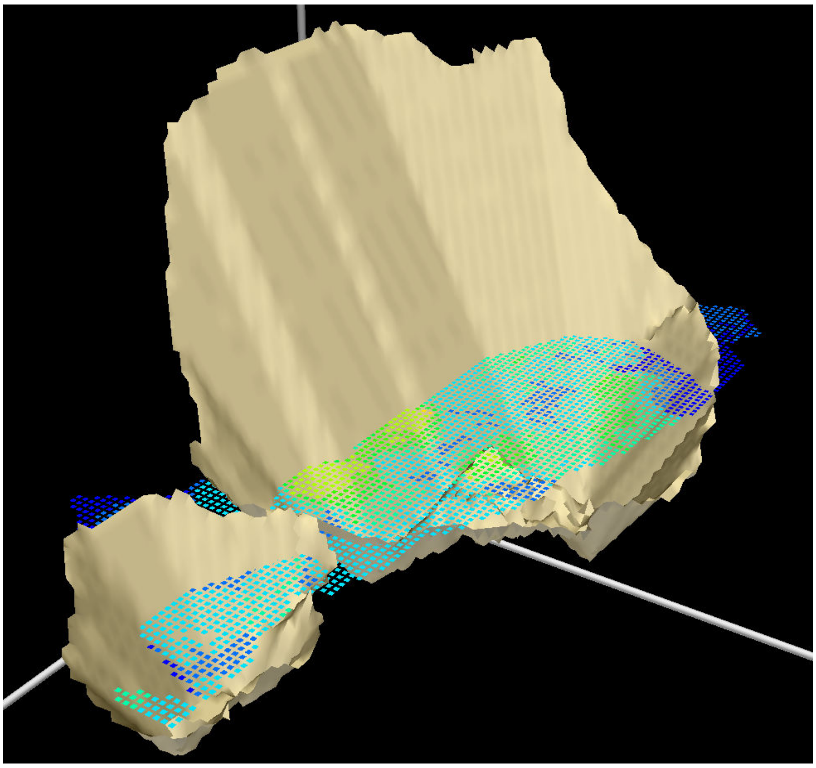


Figure 25-6: Final Optimum Pit

25.2.4 Detailed Pit Design

The results from the pit optimisation are used as guidelines to generate starter and final detailed pit designs. A number of factors are evaluated such as mining width, location of copper and gold grades, continuity of mill feed supply, geotechnical constraints and proposed mining sequence. Pits are then designed with consideration for this detailed analysis and within the limitations and constraints identified.

25.2.4.1 Geotechnical Design Constraints

The geotechnical constraints were taken from the report titled "Preliminary Geotechnical Assessments and Slope Design Studies for the Bronson Open Pit" completed by Piteau Associates in March 1997. Based on this review a representative overall pit wall angle of 50 degrees has been used for the pit design. However it is important to note that this geotechnical study was completed more than 10 years ago and further exploration drilling and core logging has taken place. Further review of the geotechnical conditions within and surrounding the revised pit limits is essential to ensure that the most accurate modelling of geotechnical risks is completed prior to further development and commissioning of the project.

25.2.4.2 Pit Design Graphics

Figure 25-7 shows the 3D view of the 2 pit shells.

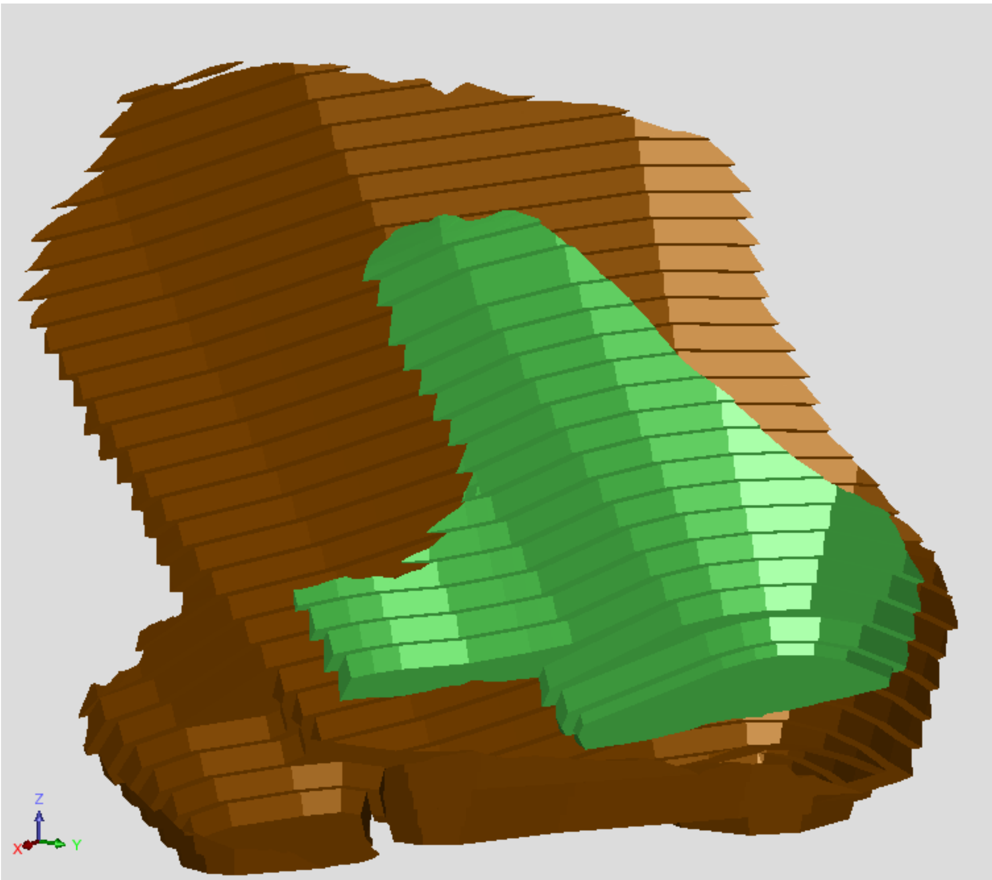


Figure 25-7: Stacked Detailed Pit Designs

Figure 25-8 to Figure 25-9 show the detailed pit designs individually. A series of representative cross sections showing the aggregate Cu and Au grades within the pit designs have been provided in Figure 25-10 to Figure 25-20.

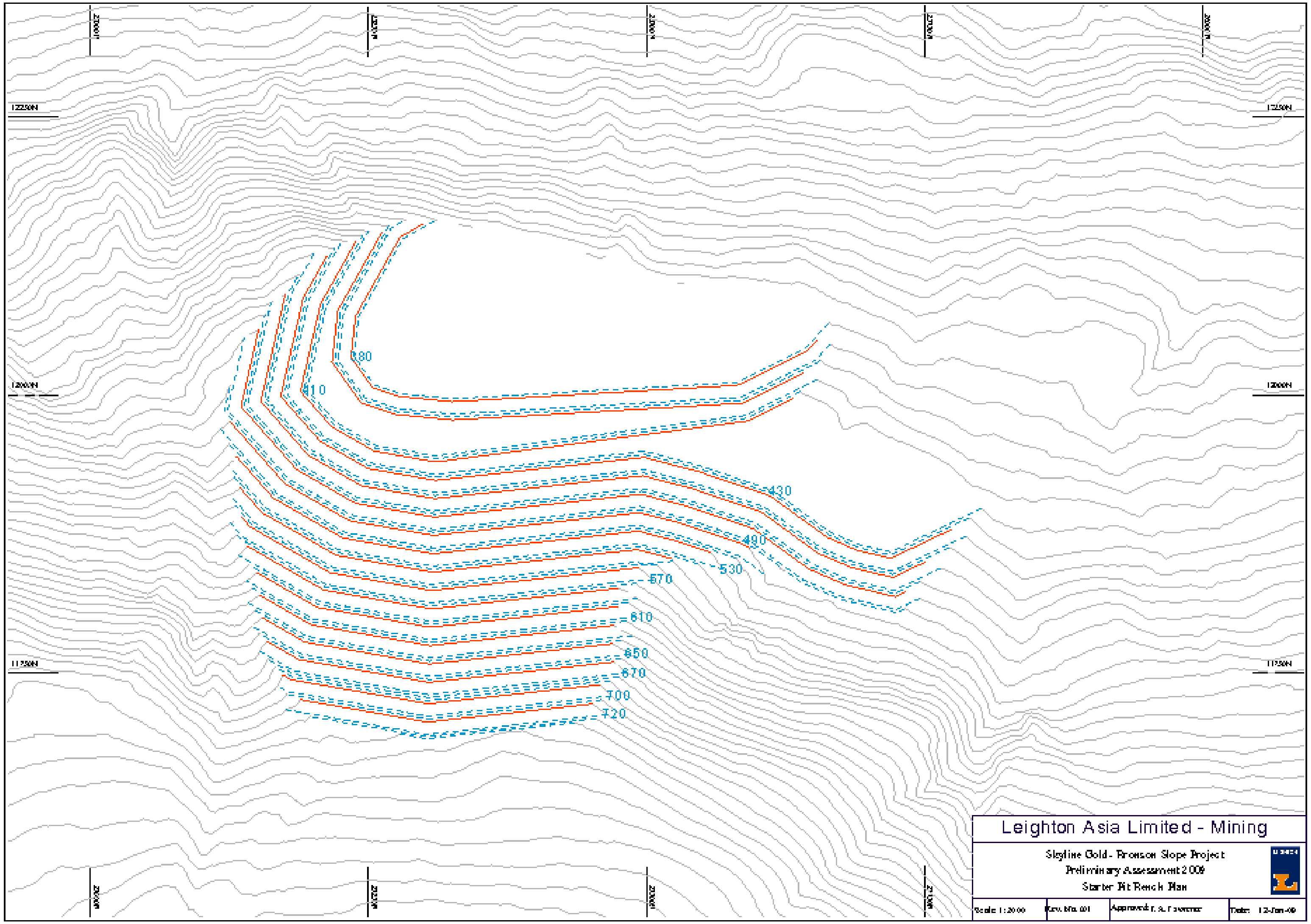


Figure 25-8: Starter Pit Bench Plan

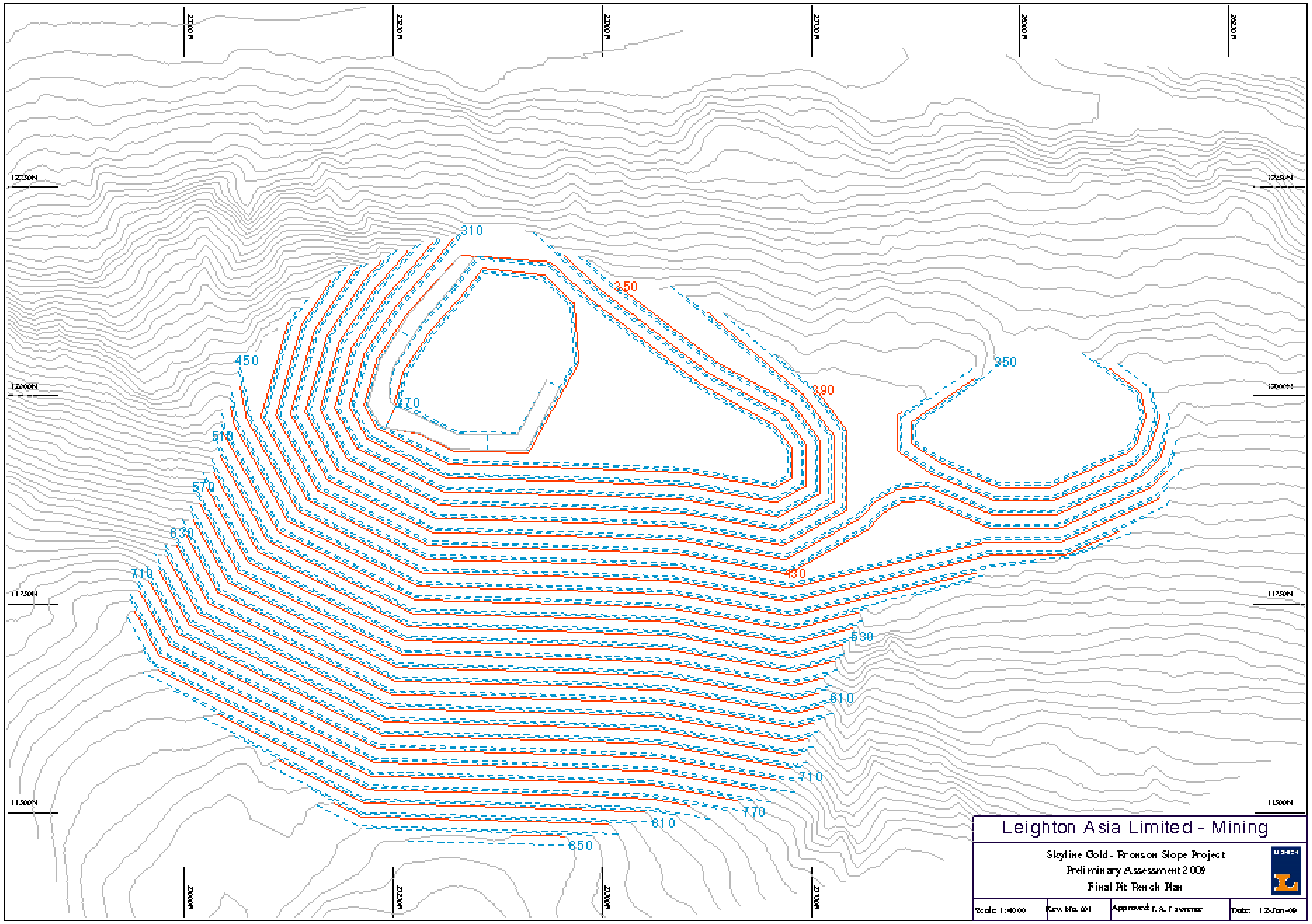


Figure 25-9: Final Detailed Pit Plan

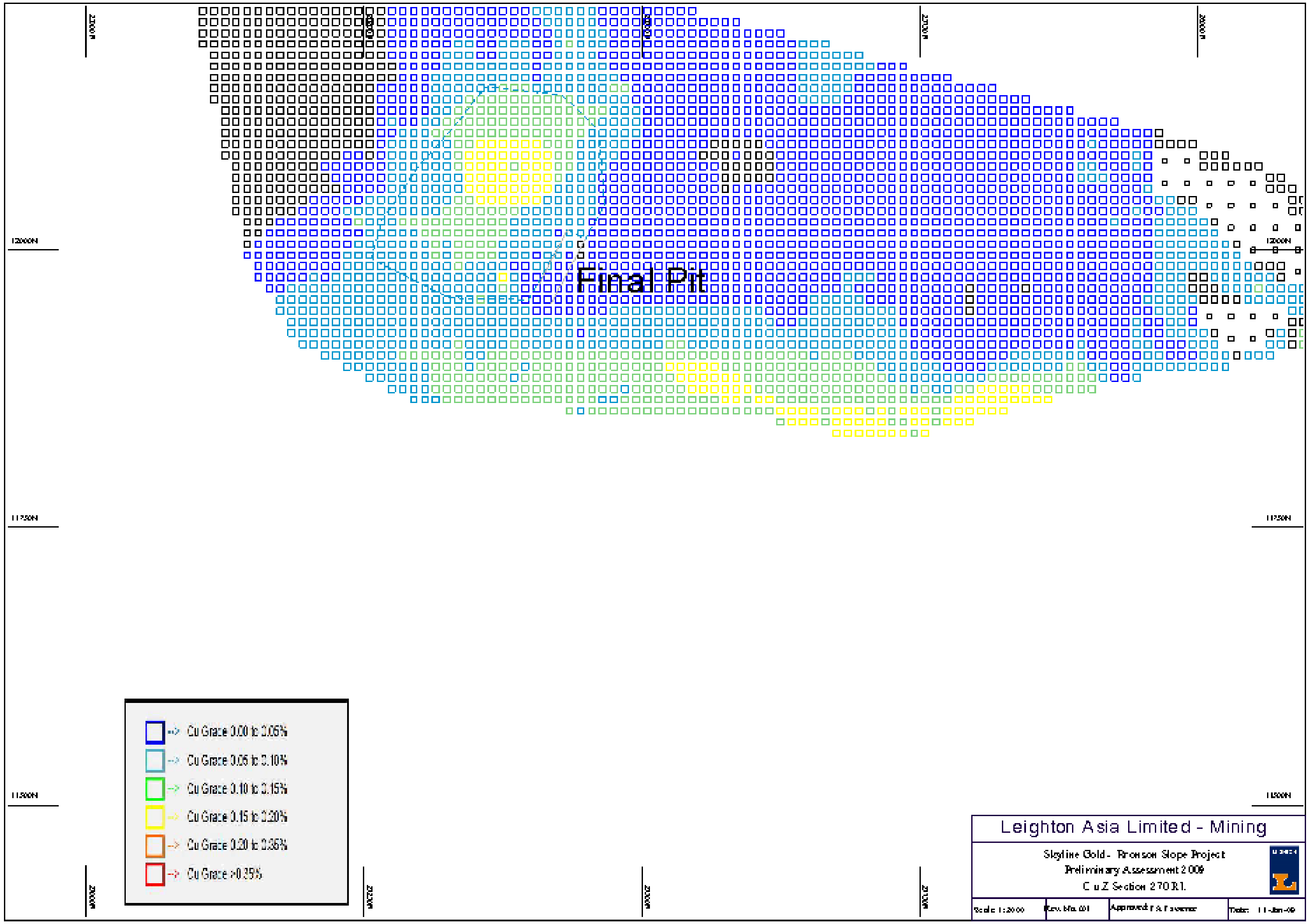


Figure 25-10: Cu Z Section (270RL – Final pit base)

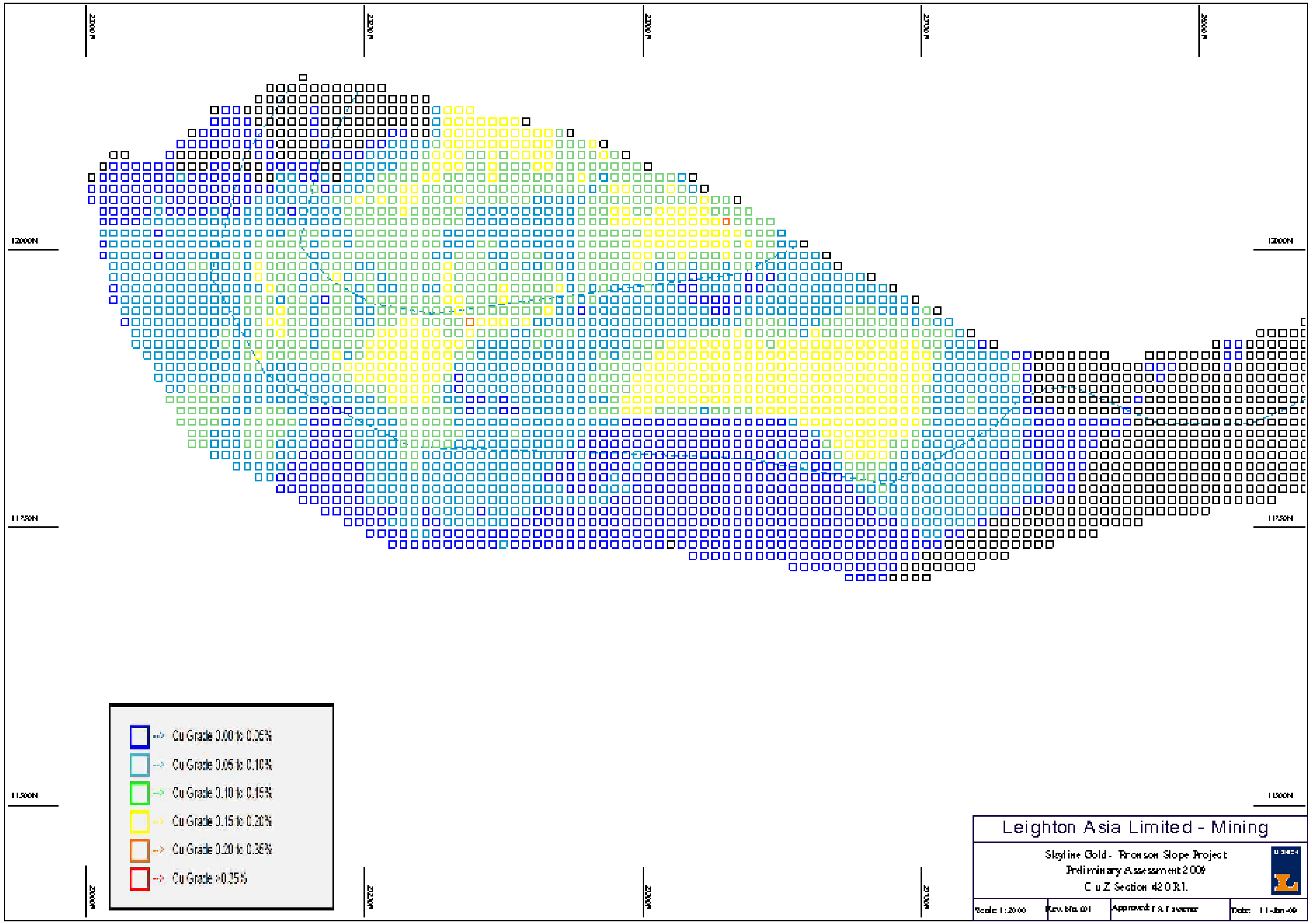


Figure 25-11: Cu Z Section (420RL)

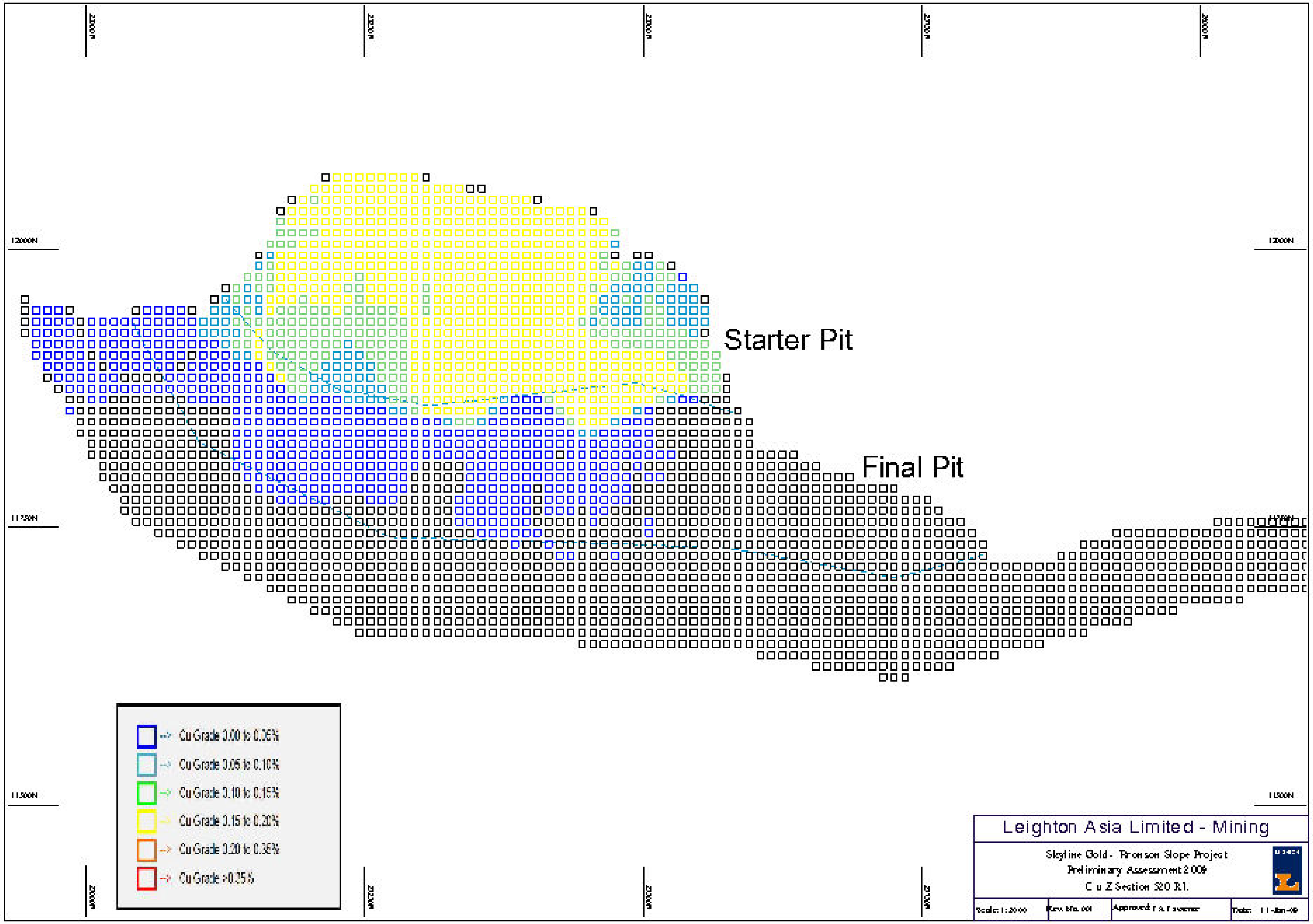


Figure 25-12: Cu Z Section (520RL)

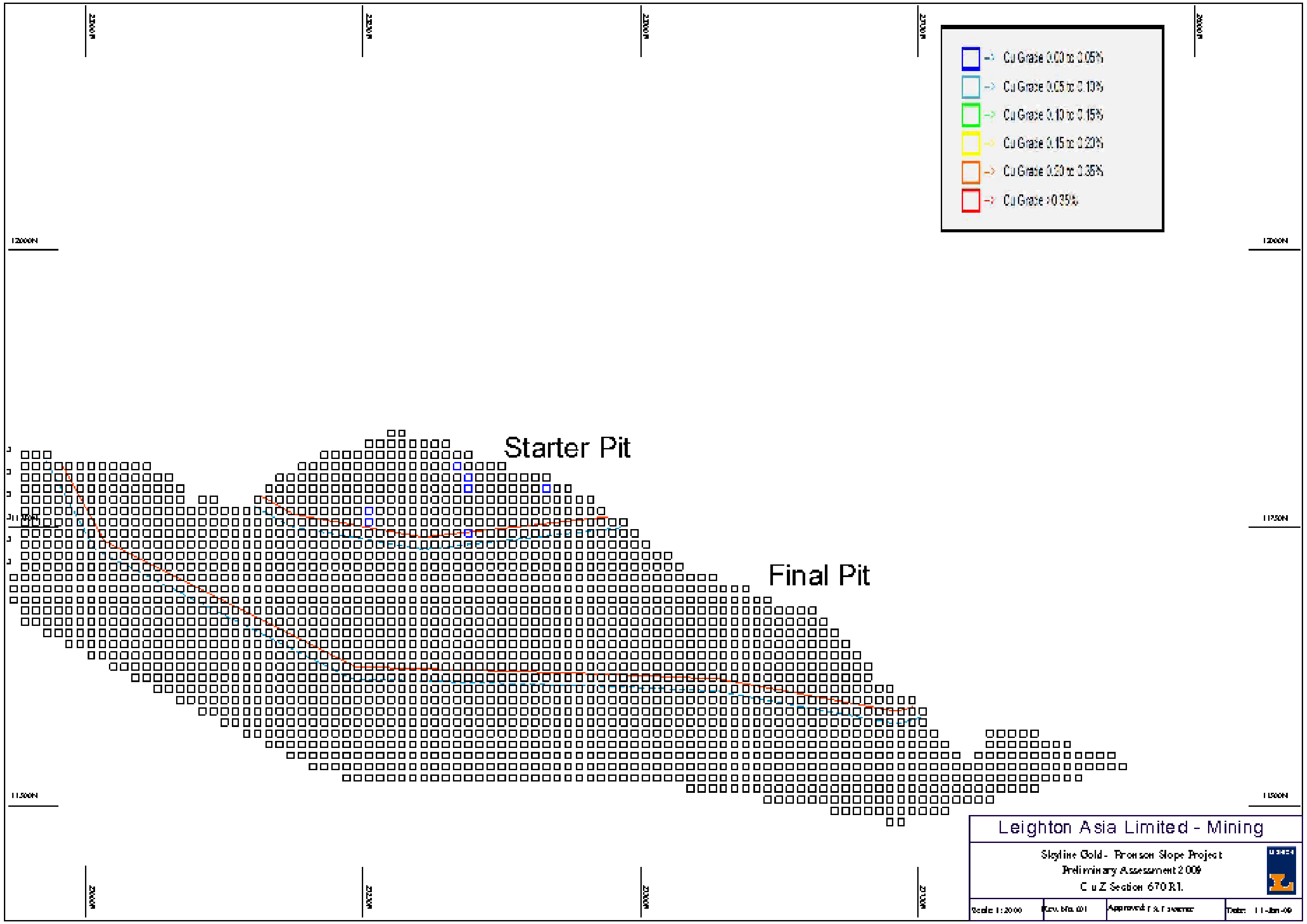


Figure 25-13: Cu Z Section (670RL)

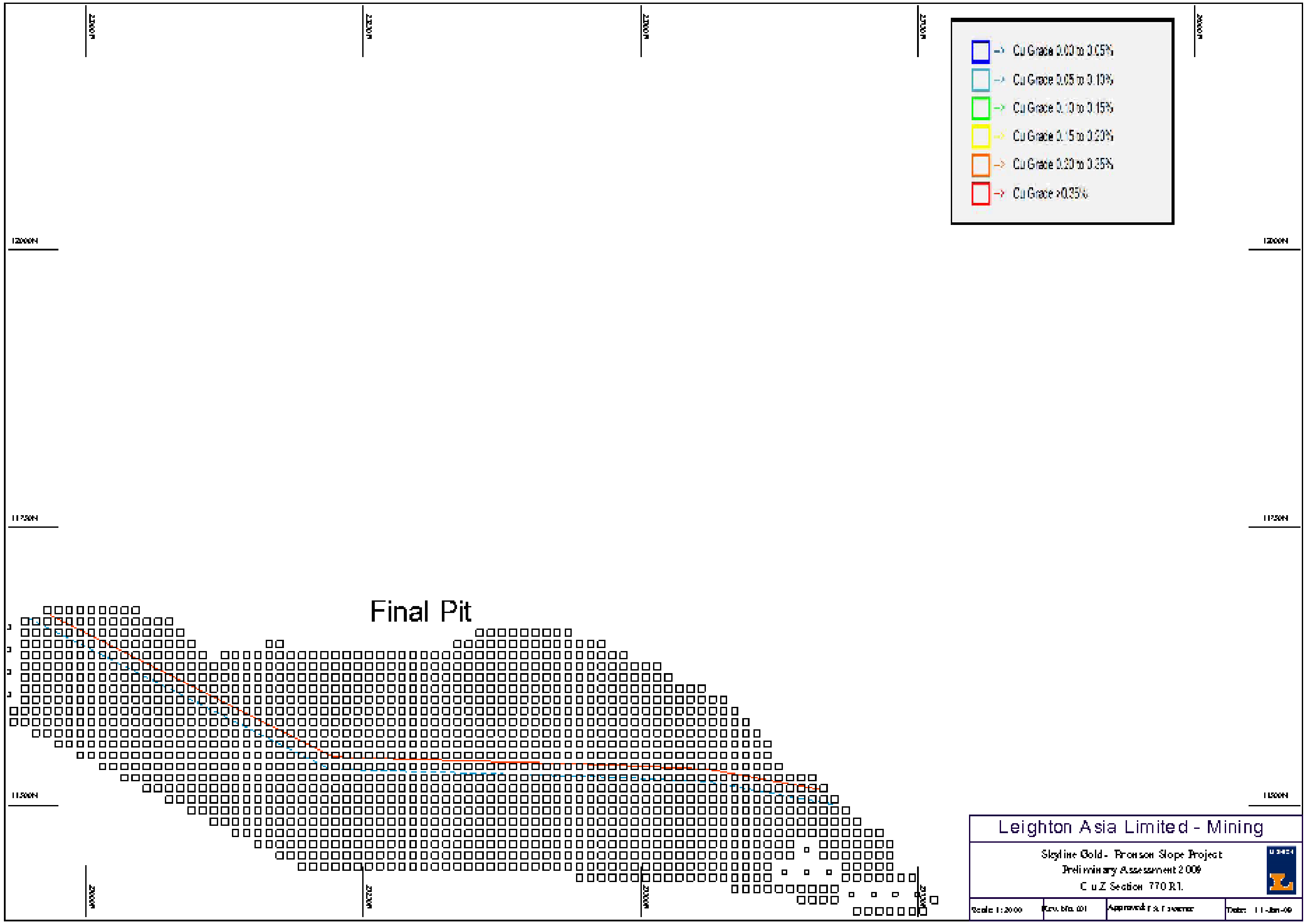


Figure 25-14: Cu Z Section (770RL)

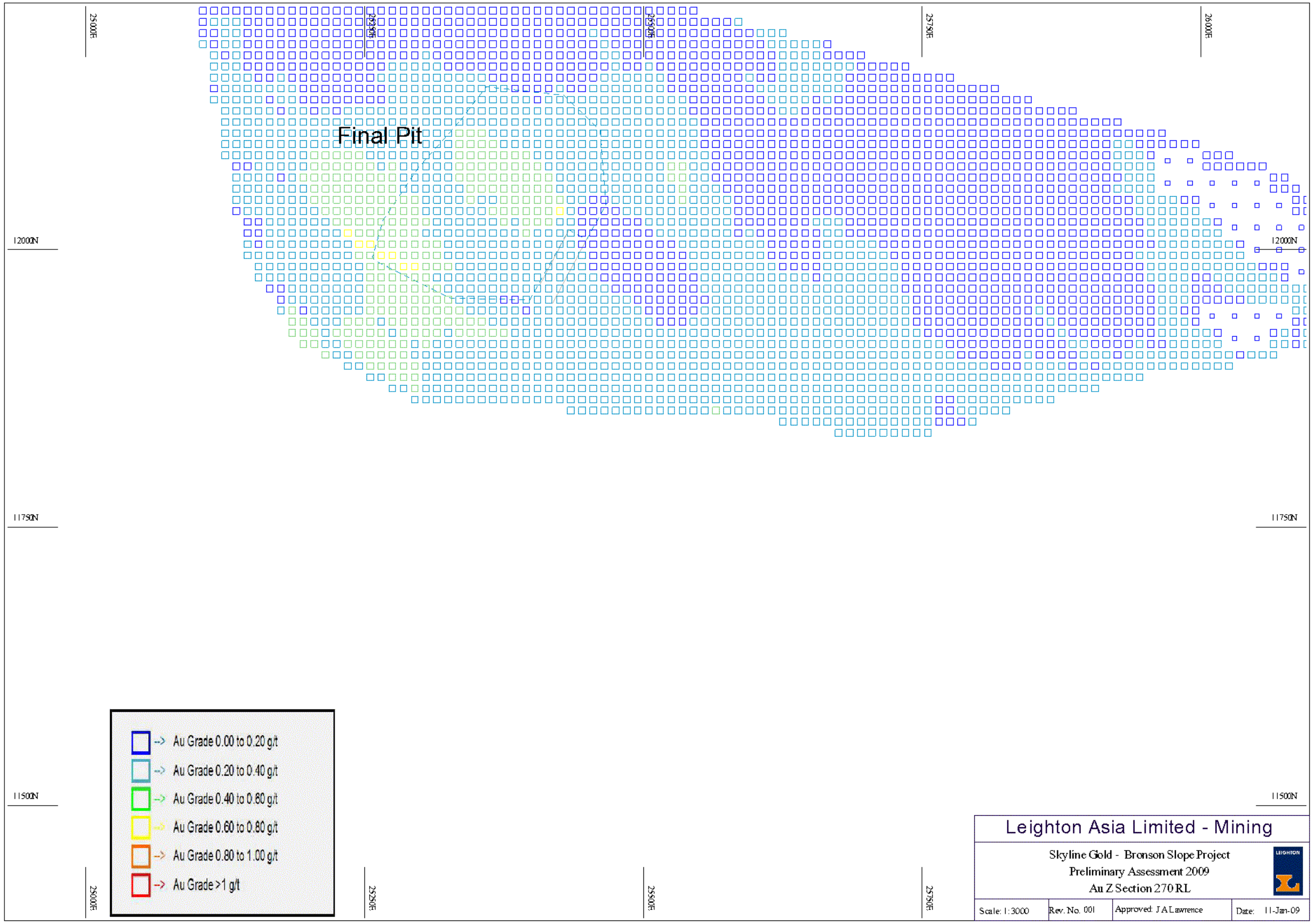


Figure 25-15: Au Z Section (270RL – Final pit base)

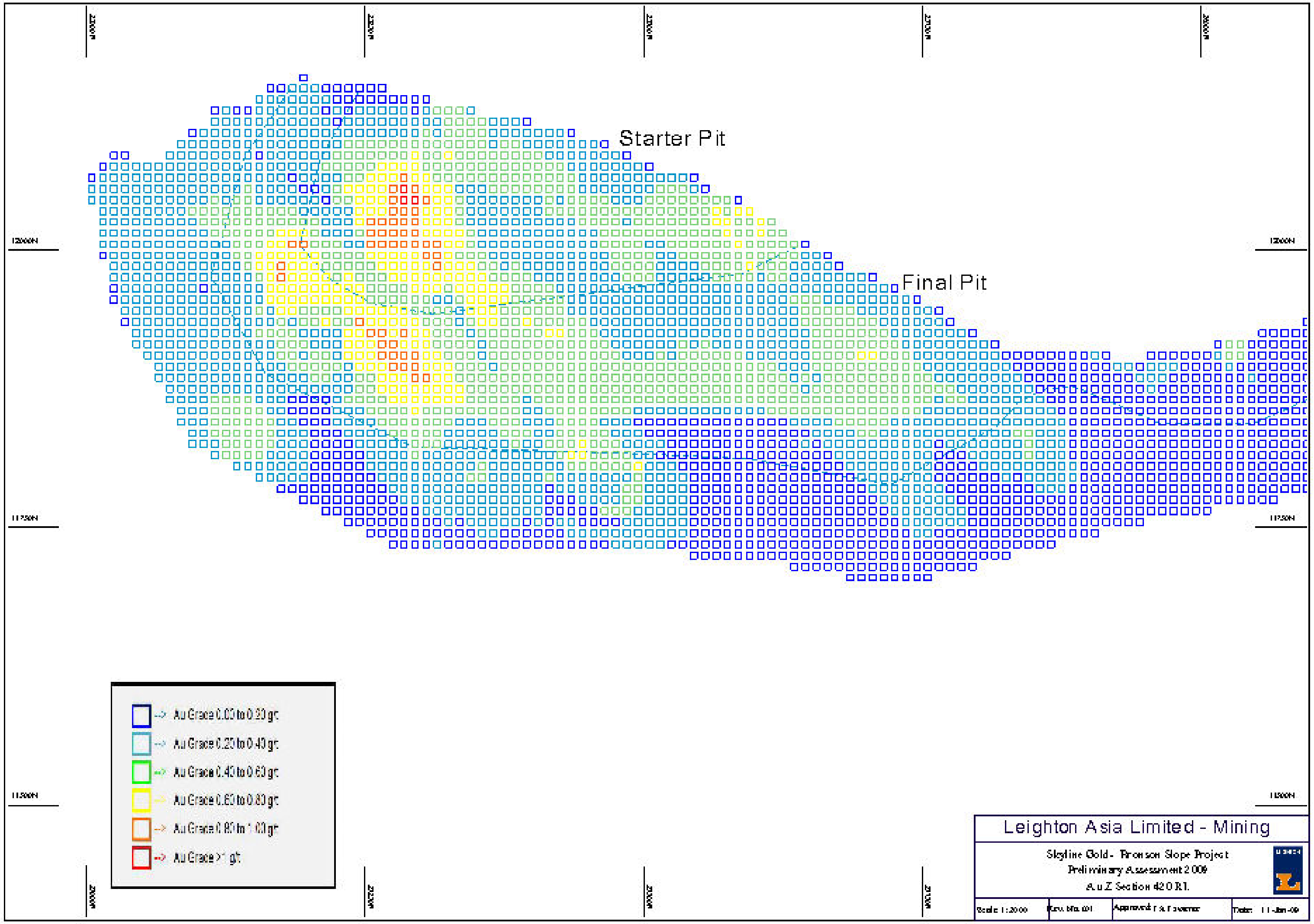


Figure 25-16: Au Z Section (420RL)

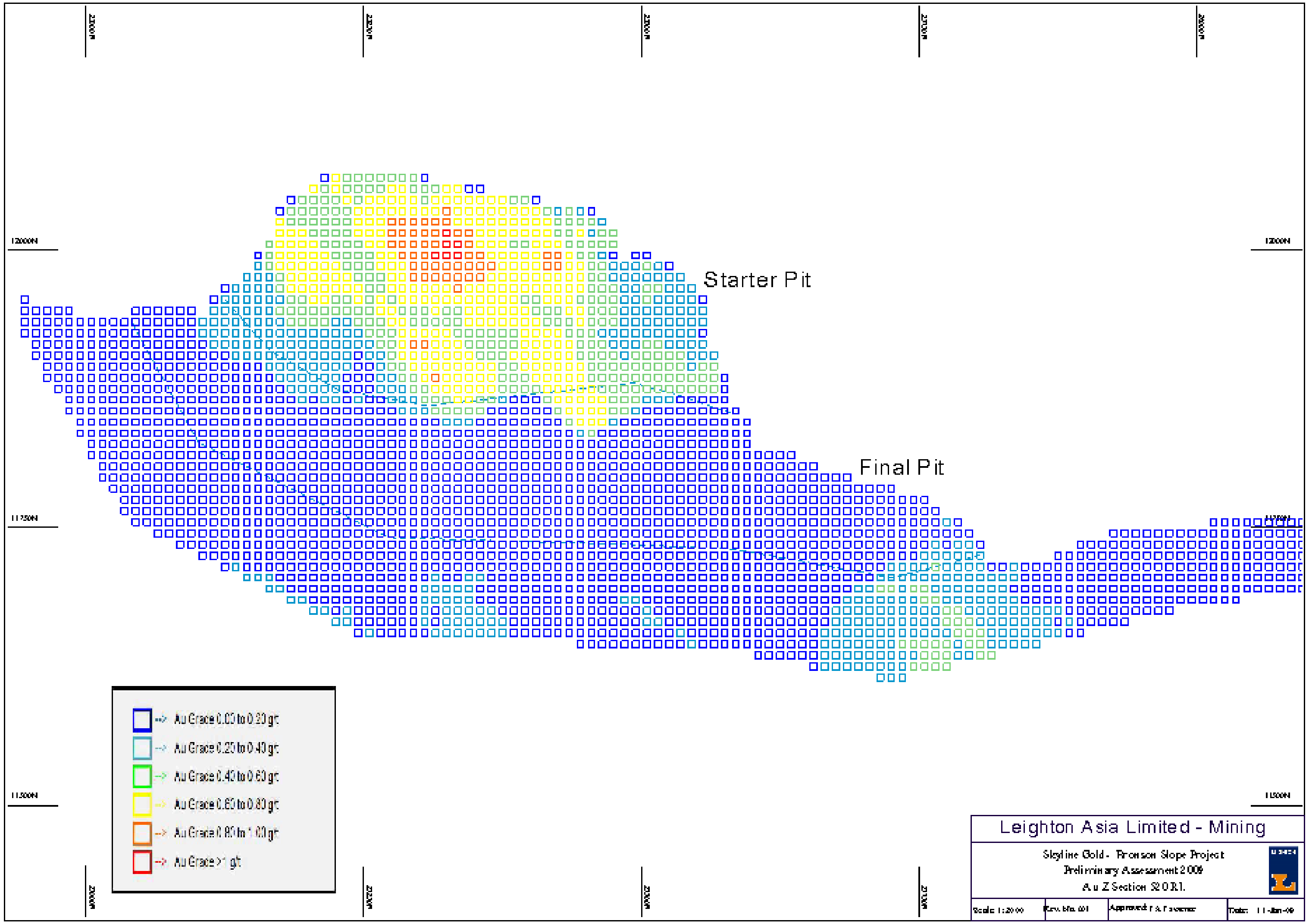


Figure 25-17: Cu Z Section (520RL)

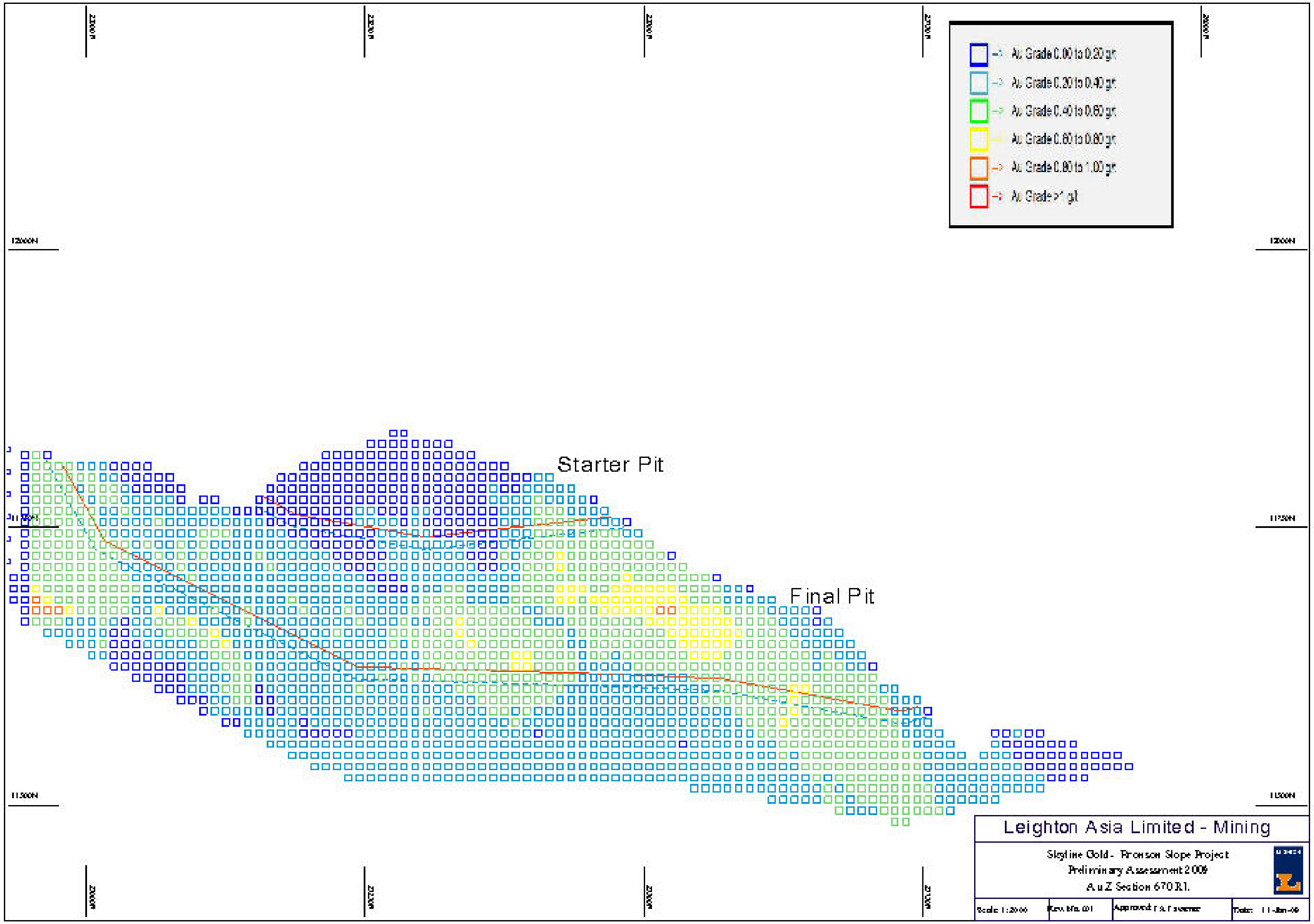


Figure 25-18: Au Z Section (670RL)

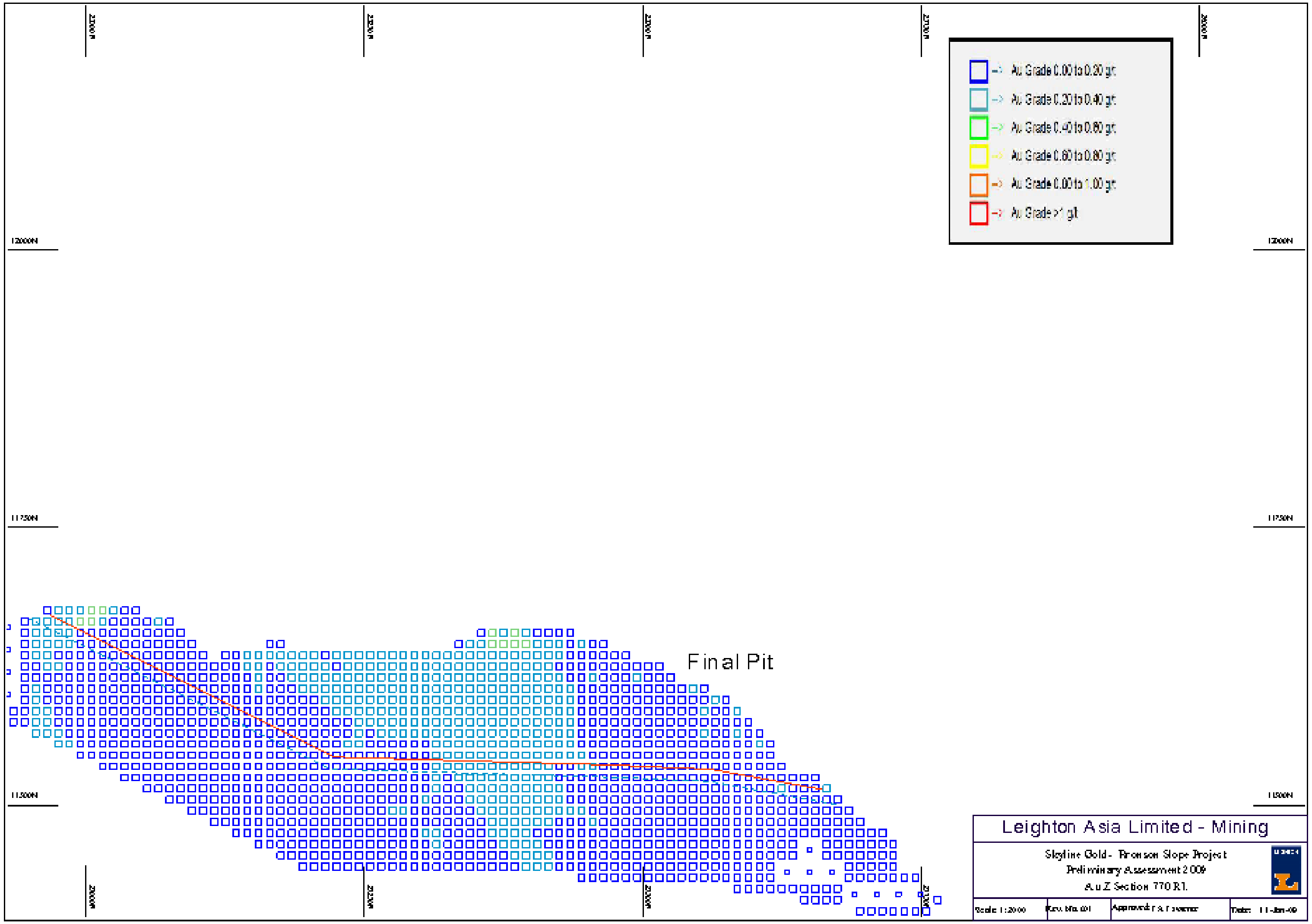


Figure 25-19: Au Z Section (770RL)

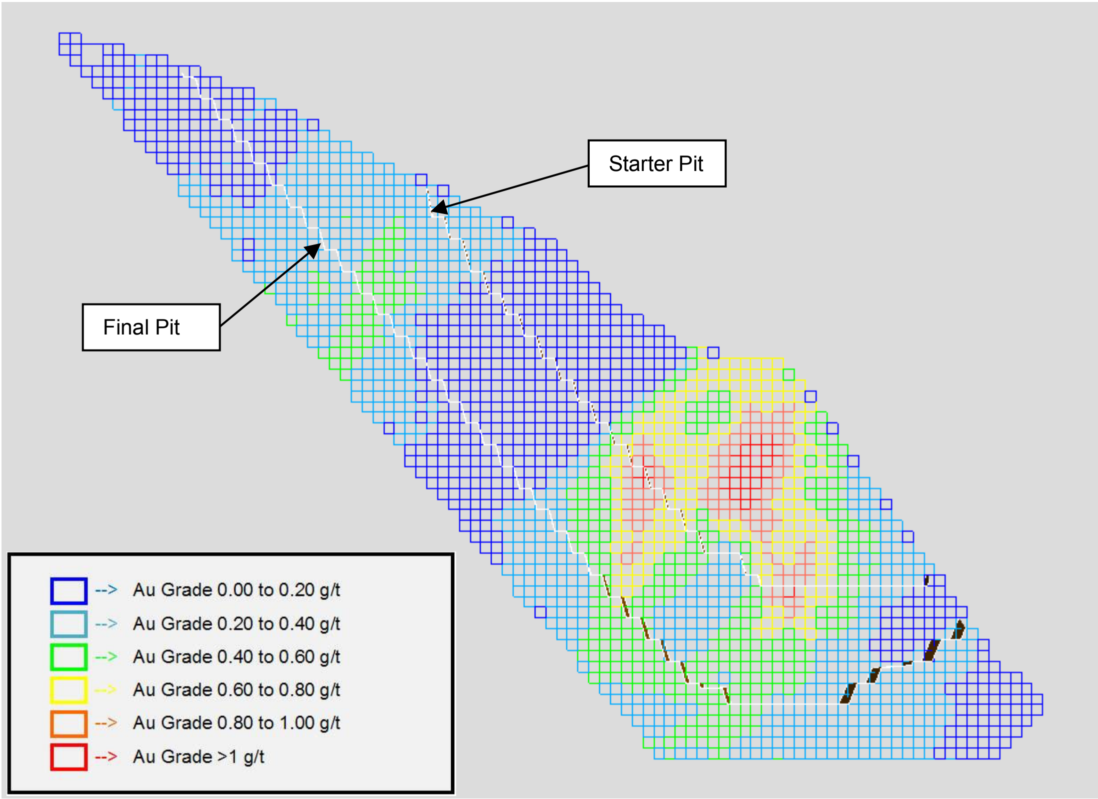


Figure 25-20: Au Section through E25300 (not to scale)

The variability in Cu and Au grade within the pit limits is worth noting at this point. It can be seen from the above sections there are certain benches where Au grade is high but Cu grade is low (particularly above 650RL), and vice versa. It is recommended that further segregation of high and low grade Cu and Au is evaluated at block model level. Separate material types should be defined and considered individually within a more detailed pit shell in order to achieve the highest possible project return through selective mining. For the purposes of this assessment the consolidation of all mineralization materials into one mineralization type is considered acceptable and will result in a more conservative estimate of the financial performance for the project.

Figure 25-21 is an Easting section that demonstrates the location of the different material types within the pit limits.

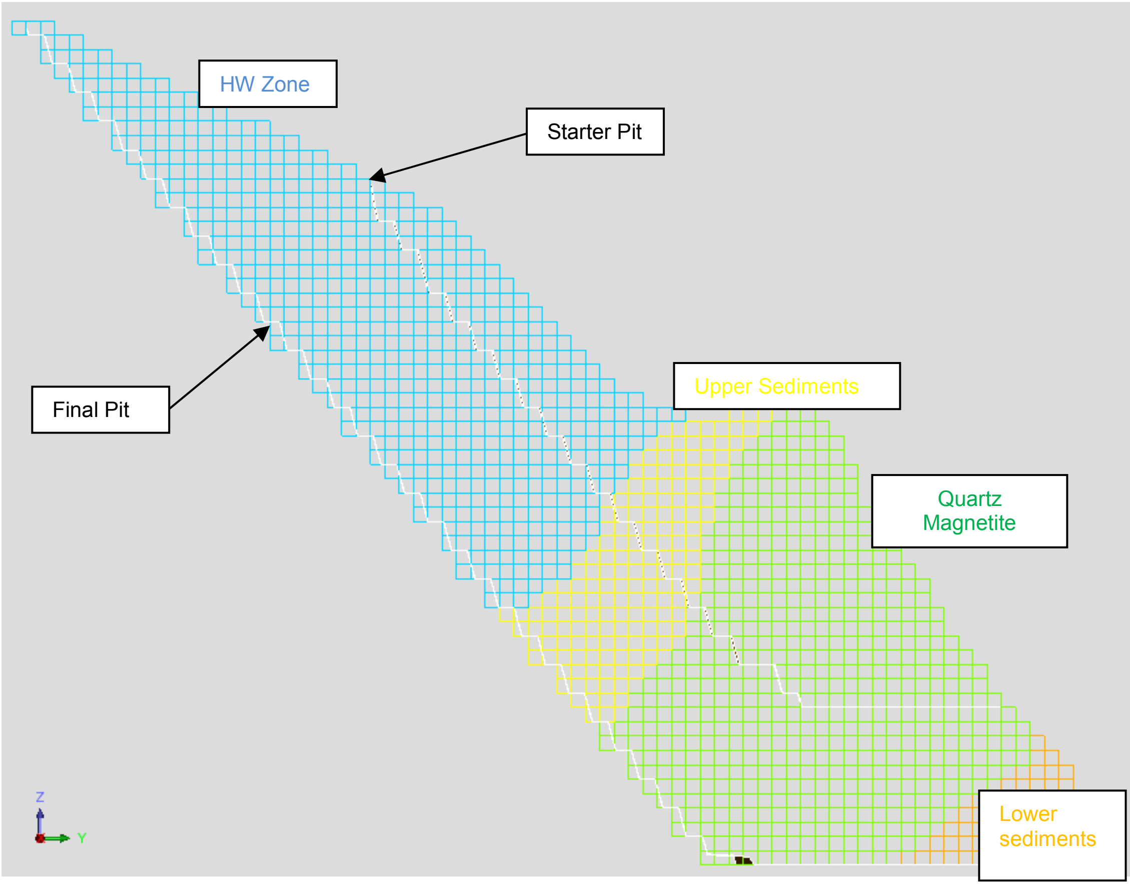


Figure 25-21: Easting Section of Material Types (not to scale)

25.2.5 Mining Areas and Contained Resource

Following the design of the different pit stages, the volume, tonnes and grades for each of the benches in each of the pit phases is calculated based on Measured and Indicated classifications only. Any inferred material is considered as waste for the purposes of the mine schedule completed for this study.

The full details of the bench grades and tonnages are included in Appendix 1. Summaries for each of the pit phases are included in *Table 25-5* and *Table 25-6* below.

Table 25-5: Phase I Tonnes and Grades

Phase I (Starter) - Tonnes and Grades (USD9/t NRV Cut-off)					
Category	**Metric Tonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Mo %**
Measured	30,657,643	0.195	0.539	2.552	0.004
Indicated	2,352,991	0.150	0.415	2.423	0.009
Inferred	0	0.000	0.000	0.000	0.000
Total Measured + Indicated	**33,010,634**	**0.192**	**0.530**	**2.543**	**0.005**
Total Waste (including inferred)	12,282,066	**Strip Ratio:**		**0.37**	
Total Mill feed and Waste	45,292,700				

Table 25-6: Phase II Tonnes and Grades

Phase II (Final) - Tonnes and Grades (USD9/t NRV Cut-off)					
Category	**Metric Tonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Mo %**
Measured	31,421,748	0.153	0.417	2.009	0.006
Indicated	29,037,532	0.118	0.391	2.502	0.007
Inferred	473,839	0.046	0.417	3.892	0.003
Total Measured + Indicated	**60,459,280**	**0.135**	**0.401**	**2.228**	**0.007**
Total Waste (incl Inferred)	66,242,641	**Strip Ratio:**		**1.08**	
Total Mill feed and Waste	126,701,921				

The sum of both of these phases (total LOM mill feed tonnage and grade) is included in *Table 25-7*.

Table 25-7: LOM Mill feed Tonnage and Grade

Phase I&II (Total) - Tonnes and Grades (USD9/t NRV Cut-off)					
Category	**Metric Tonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Mo %**
Measured	62,079,392	0.174	0.477	2.277	0.005
Indicated	31,390,522	0.120	0.393	2.497	0.007
Inferred	473,839	0.046	0.417	3.892	0.003
Total Measured + Indicated	**93,469,914**	**0.155**	**0.446**	**2.34**	**0.0058**
Total Waste (incl Inferred)	78,524,707	**Strip Ratio:**		**0.83**	
Total Mill feed and Waste	**171,994,621**				

25.2.6 LOM Production Schedule

The bench by bench tonnes and grades were built into a production scheduling system generated in Microsoft Excel. The schedule allowed for the progressive sequencing of material movement by phased pit design and bench and assumed that for each bench the material types were mined in weighted equal portions until the bench was completely mined out before progressing to the next bench.

The full detailed production schedule is presented in Table 25-8 and Table 25-9 and Figure 25-22 and Figure 25-23.

Table 25-8: LOM Production Schedule

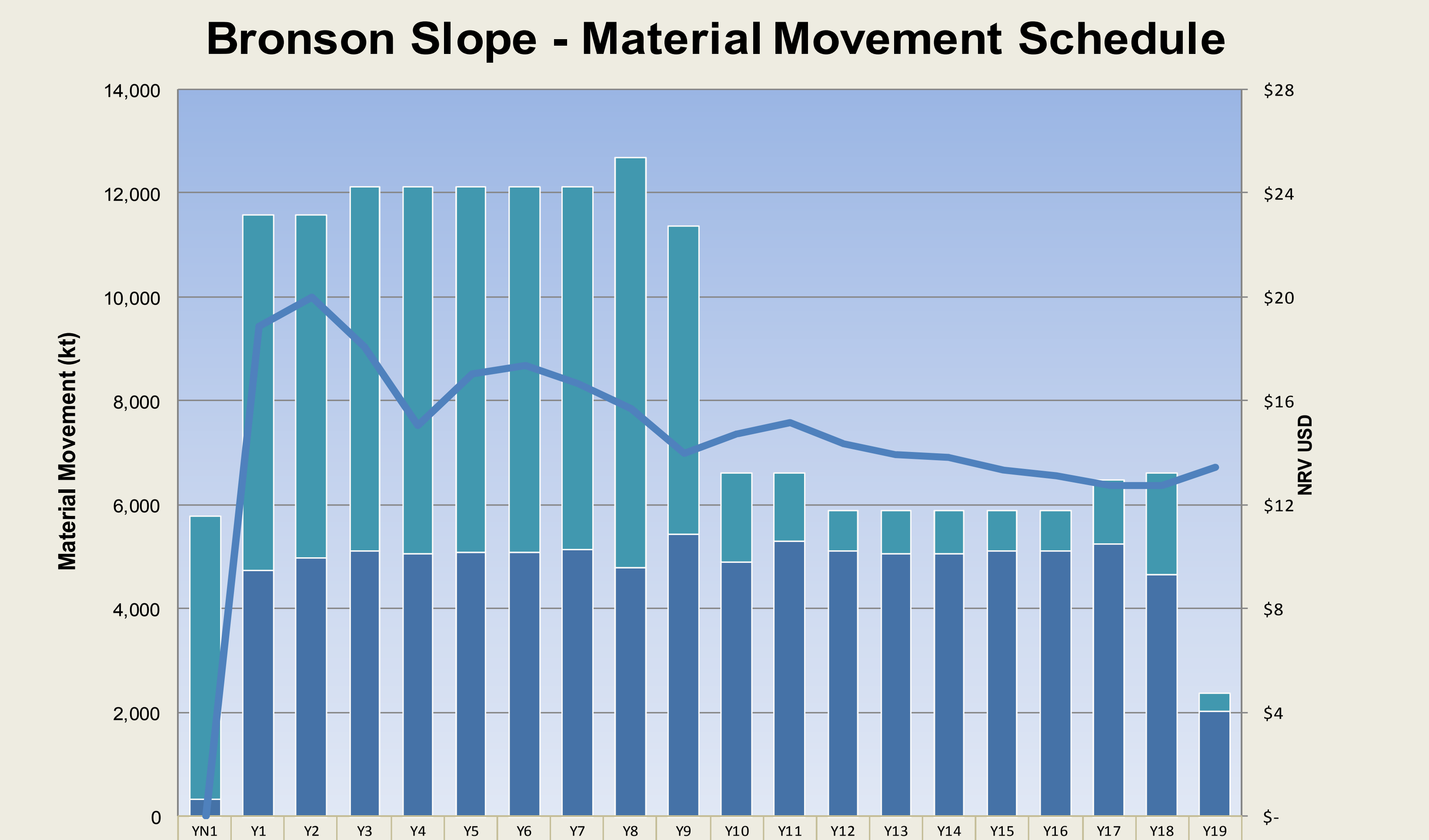
Prepared for 2009 Preliminary Economic Assessment

	YN1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Total
Material Movement (kBCM)																					
Waste	1,981	2,478	2,396	2,541	2,560	2,551	2,556	2,538	2,857	2,148	625	478	282	307	308	282	287	448	707	119	**28,449**
Bronson Ore	119	1,722	1,804	1,859	1,840	1,849	1,844	1,862	1,743	1,974	1,775	1,922	1,858	1,833	1,832	1,858	1,853	1,902	1,693	741	**33,882**
Total Material	**2,100**	**4,200**	**4,200**	**4,400**	**4,400**	**4,400**	**4,400**	**4,400**	**4,600**	**4,121**	**2,400**	**2,400**	**2,140**	**2,140**	**2,140**	**2,140**	**2,140**	**2,350**	**2,400**	**859**	**62,331**

	YN1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Total
Material Movement (ktonnes)																					
Waste (kt)	5,464	6,836	6,611	7,010	7,064	7,038	7,053	7,003	7,881	5,926	1,726	1,319	779	846	850	778	793	1,236	1,951	328	**78,490**
Bronson Ore (kt)	330	4,752	4,977	5,130	5,075	5,102	5,087	5,137	4,810	5,445	4,896	5,303	5,126	5,058	5,054	5,126	5,111	5,248	4,671	2,043	**93,480**
Total Material	**5,794**	**11,588**	**11,588**	**12,140**	**12,140**	**12,140**	**12,140**	**12,140**	**12,691**	**11,371**	**6,622**	**6,622**	**5,904**	**5,904**	**5,904**	**5,904**	**5,904**	**6,484**	**6,622**	**2,371**	**171,970**

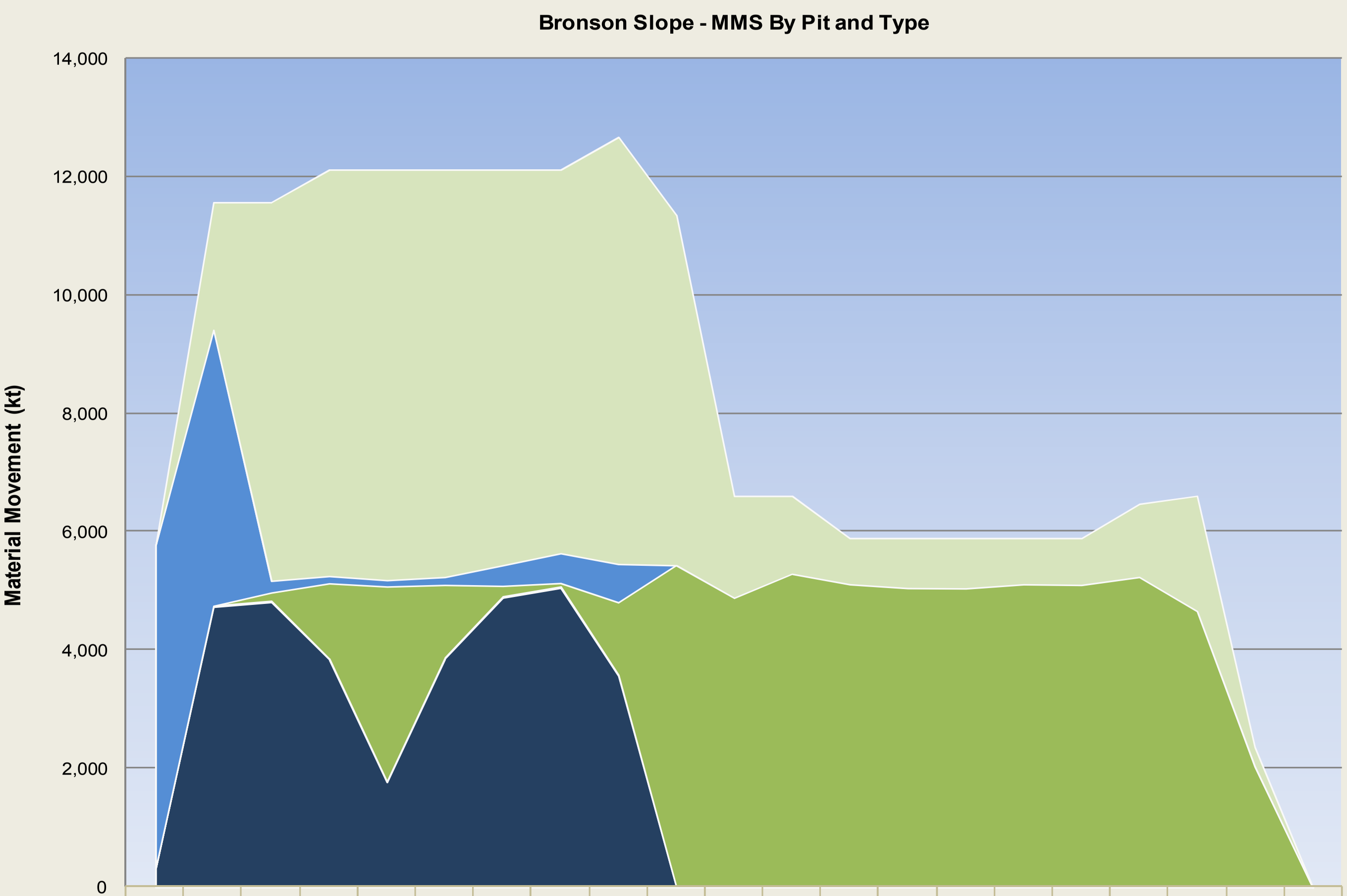
	YN1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Total
Grade Bronson																					
Cu (%)	0.03	0.21	0.22	0.16	0.09	0.15	0.17	0.17	0.16	0.17	0.16	0.17	0.15	0.15	0.14	0.14	0.14	0.13	0.14	0.15	0.16
Au (g/t)	0.52	0.52	0.57	0.57	0.56	0.54	0.52	0.49	0.46	0.37	0.40	0.42	0.41	0.40	0.40	0.38	0.38	0.37	0.36	0.36	0.45
Ag (g/t)	1.25	2.46	2.65	2.77	3.02	2.77	2.57	2.45	2.37	2.05	2.46	2.35	2.09	1.91	2.06	2.12	2.13	2.14	2.17	2.03	2.35
Mo (%)	0.001	0.009	0.006	0.003	0.002	0.003	0.004	0.003	0.004	0.012	0.009	0.008	0.008	0.008	0.007	0.007	0.006	0.005	0.004	0.003	0.006
Total Milled (kt)	**0**	**4779**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**5098**	**2035**	**93,480**

	YN1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Total
Contained metal (conc)																					
Cu (klb)	0	19,171	21,122	15,941	8,410	14,870	16,953	16,558	15,787	16,139	15,911	16,227	14,879	14,351	14,046	13,599	13,216	12,637	13,172	5,973	278,962
Au (t.oz)	0	68,836	79,400	79,737	78,985	75,421	72,343	68,750	63,799	51,284	56,236	59,005	57,244	56,081	55,871	53,475	52,865	51,806	50,218	20,117	1,151,474
Ag (t.oz)	0	240,495	276,813	288,945	315,630	289,386	267,864	255,499	247,932	214,514	256,489	245,526	218,362	199,102	215,146	221,382	222,291	223,614	226,365	84,786	4,510,140
NRV (USD/t ore)	$ -	$ 18.86	$ 20.00	$ 18.05	$ 15.07	$ 17.04	$ 17.37	$ 16.69	$ 15.68	$ 14.00	$ 14.72	$ 15.19	$ 14.34	$ 13.92	$ 13.82	$ 13.33	$ 13.10	$ 12.73	$ 12.73	$ 13.42	$ 15.30



	YN1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
Waste (kt)	5,464	6,836	6,611	7,010	7,064	7,038	7,053	7,003	7,881	5,926	1,726	1,319	779	846	850	778	793	1,236	1,951	328
Bronson Ore (kt)	330	4,752	4,977	5,130	5,075	5,102	5,087	5,137	4,810	5,445	4,896	5,303	5,126	5,058	5,054	5,126	5,111	5,248	4,671	2,043
NRV (USD/t ore)	$-	$18.86	$20.00	$18.05	$15.07	$17.04	$17.37	$16.69	$15.68	$14.00	$14.72	$15.19	$14.34	$13.92	$13.82	$13.33	$13.10	$12.73	$12.73	$13.42

Figure 25-22: Material Movement by Type



	YN1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20
Final Pit Waste (kt)	0	2159	6407	6876	6944	6891	6695	6490	7225	5926	1726	1319	779	846	850	778	793	1236	1951	328	0
Starter Pit Waste (kt)	5464	4677	204	134	121	146	358	513	656	0	0	0	0	0	0	0	0	0	0	0	0
Final Pit Ore (kt)	0	0	146	1271	3295	1228	181	63	1227	5445	4896	5303	5126	5058	5054	5126	5111	5248	4671	2043	0
Starter Pit Ore (kt)	330	4752	4831	3859	1780	3874	4906	5074	3583	0	0	0	0	0	0	0	0	0	0	0	0

Figure 25-23: Material Movement by Pit and Type

Table 25-9: Metal Production Schedule

Variable	Unit	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y 10
Tonnes Ore Mined & Milled	000's t	4,779	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098
Copper Grade	%	0.21%	0.22%	0.16%	0.09%	0.15%	0.17%	0.17%	0.16%	0.17%	0.16%
Recovered Copper (86.6% Recovery)	000's lbs	19,171	21,122	15,941	8,410	14,870	16,953	16,558	15,787	16,139	15,911
Gold Grade	g/t	0.525	0.567	0.570	0.564	0.539	0.517	0.491	0.456	0.366	0.402
Recovered Gold (85.4% Recovery)	000's oz	68.8	79.4	79.7	79.0	75.4	72.3	68.7	63.8	51.3	56.2
Silver Grade	g/t	2.46	2.65	2.77	3.02	2.77	2.57	2.45	2.37	2.05	2.46
Recovered Silver	000's oz	240.5	276.8	288.9	315.6	289.4	267.9	255.5	247.9	214.5	256.5
CONCENTRATE GRADE											
Copper	%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%
Gold	g/t	62.04	64.95	86.43	162.28	87.65	73.74	71.75	69.83	54.91	61.07
Silver	g/t	216.77	226.45	313.21	648.50	336.29	273.02	266.63	271.38	229.68	278.55
Concentrate Moisture Content	%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%
WET CONCENTRATE TONNAGE	wmt	37,268	41,062	30,989	16,349	28,906	32,957	32,188	30,689	31,373	30,931

Variable	Unit	Y 11	Y 12	Y 13	Y 14	Y 15	Y 16	Y 17	Y 18	Y 19	AVG	TOTAL
Tonnes Ore Mined & Milled	000's t	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	2,035	4,920	93,480
Copper Grade	%	0.17%	0.15%	0.15%	0.14%	0.14%	0.14%	0.13%	0.14%	0.15%	0.156%	
Recovered Copper (86.6% Recovery)	000's lbs	16,227	14,879	14,351	14,046	13,599	13,216	12,637	13,172	5,973	14,682	278,962
Gold Grade	g/t	0.422	0.409	0.401	0.399	0.382	0.378	0.370	0.359	0.360	0.449	
Recovered Gold (85.4% Recovery)	000's oz	59.0	57.2	56.1	55.9	53.5	52.9	51.8	50.2	20.1	61.9	1,151
Silver Grade	g/t	2.35	2.09	1.91	2.06	2.12	2.13	2.14	2.17	2.03	2.356	
Recovered Silver	000's oz	245.5	218.4	199.1	215.1	221.4	222.3	223.6	226.4	84.8	242.4	4,510
CONCENTRATE GRADE												
Copper	%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	
Gold	g/t	62.83	66.48	67.52	68.73	67.95	69.12	70.84	65.88	58.20	73.81	
Silver	g/t	261.45	253.59	239.73	264.68	281.31	290.65	305.76	296.96	245.27	249.20	
Concentrate Moisture Content	%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	
WET CONCENTRATE TONNAGE	wmt	31,545	28,925	27,899	27,305	26,436	25,691	24,567	25,606	11,612	29,067	542,298

The schedule demonstrates that the starter pit is mined exclusively until year 2, when the stripping of the waste and mining of the high wall area of the final pit takes place. Between year 3 and 6 mill feed is encountered in the highwall zone of the second pit and therefore is fed through the process. This mill feed material has a low copper grade but is high in gold grade. Further study needs to be conducted to determine whether the gold recoveries of the highwall zone are achievable with very low copper grades. An alternate plan may be required to stockpile the highwall zone material and progressively blend it into the mill feed to reduce the impact of the reduced copper grades.

Between year 6 and 9 the majority of the mill feed is taken from the starter pit and further waste stripping is conducted from the highwall zone of the final pit. At the point when the base of the starter pit is reached the final pit becomes mill feed bound (year 10) and the production fleet is reduced to one excavator, crusher combination. From year 10 onwards a single fleet operates out of the final pit mining bench by bench until the base of the pit is reached in year 20.

Figure 25-23 also shows the optimum production schedule requires a high strip ratio early in the mine life. This is somewhat contradictory to conventional phased pit mining techniques utilised by other mining projects around the world. The high strip ratio requirements are a result of the narrow shape of the pit and the orebody and also the high gold grade material that is present in the high wall zone. Stripping of the final pit needs to start early in the mine life (towards the end of year 1) in order to ensure consistent mill feed of 5.089 mtpa at the later stages in the mine schedule. Also due to the high proportional value of gold as compared to copper, the high grade gold mill feed found in the highwall zone is targeted early in the mine life and cash flow.

25.3 Recoverability

25.3.1 Mining

Due to the bulk consistent low grade nature of the orebody the bench configurations for the pit are relatively simple. Delineation will be controlled in the field by the geology department and will provide guidance for the excavator operators to ensure minimal dilution of waste to mill feed. However waste material will be typically mineralized but will be below the cut-off grade (in this case $9.00/t NRV). A bench height of 10m has been adopted which will not cause any significant issues with grade control delineation bench to bench and will also ensure that blasting movement and direction can be adequately controlled.

Some mill feed loss will be experienced when blasting near the bench limits as some loose face material will fall over the edge, and will not be recovered. However this mill feed loss will be minimal. By utilising the conveying system and having immediate control over the dump location of all excavated material there will be minimal risk with run of mine material being taken to the wrong location (as is the case at times with traditional truck haulage and complex dumping requirements). However close management of the stockpiling of mill feed and waste in the ROM area will be critical to ensure minimal dilution and loss. Clear separation of the mill feed and waste stockpiles with unambiguous delineation will be important to ensure that rehandle activities are conducted from the correct stockpiles.

25.3.2 Processing

Conventional gravity flotation processing of gold and copper is proposed for the Bronson Slope ores, recoverability of gold and copper are 85.4% and 86.6% respectively. Metallurgical test results indicate that the gravity gold can be recovered in the gravity separator using a conventional Knelson gravity circuit. Gold associated with iron minerals can be recovered in the flotation circuit. The variations of gold recoveries among the composites are very low which reveals the similarity of gold mineralization of the mineralization types.

The majority of the copper minerals can be recovered in the flotation circuits equipped with a regrind facility. Copper recoveries are consistent among the mineralization composites, with recovery in the range of 82% and 90% being achieved. Using the proven technology and amenability of the mineralization to flotation technology, it is expected that the recoverability of both the metals, copper and gold, are achievable.

Recoverability of molybdenum minerals is not proven yet. Low head grade and fine disseminate particle characteristics would be a challenge for molybdenum recovery. Low mass recovery and high concentrate grade may encounter some operational challenges for molybdenum recovery. The capital and operating costs estimated for the recovery of molybdenum, coupled with the need for further molybdenum recovery testing has lead to the molybdenum processing circuit being left out of this Preliminary Assessment.

Testing has demonstrated metal recoverability of copper, gold and silver in the copper smelter are 96.5%, 97.5% and 90% respectively. The metal recoverability is mainly attributed to the efficiency of the smelting and refining processes. Losses in dust, evaporative gas and slag are some of the factors that contribute to recoverability of the metals.

25.4 Markets

SGC have advised they do not have any existing agreements for sale of product from the Bronson Slope Property. The following discussion is based on research conducted throughout the course of the study to identify what offsite processing terms and conditions should be considered for the Bronson Slope Project.

25.4.1 Smelter Terms

25.4.1.1 Copper

The following is a series of quotes obtained from publicly available articles:

“American Metal market, Jan 2007. - NEW YORK -- Freeport-McMoRan Copper & Gold Inc. reportedly has settled annual treatment and refining charges (TC/RCs) with Japanese copper smelters. The headline terms of $60 a tonne/6.0 cents a pound with no price participation are in line with earlier benchmark deals between Anglo-Australian miner BHP Billiton and the Japanese.”

“Interfax, China June 2007 - Japanese smelters [including Sumitomo Metals Mining, Mitsubishi Materials Corp. and Pan Pacific Copper] and Jinlong Copper both completed the first round of negotiations with BHP Billiton [NYSE:BHP] for annual TC/RC charges at the end of last month, with no result being reached. BHP offered the Japanese party a TC/RC charge of 40/4.0 with price participation.”

"In China domestic copper smelters are coming under increased pressure from the copper concentrate shortage, especially if they are expanding capacity. They stand to make a loss if the TC/RC charge falls below the current 50/5.0 price and if they don't possess their own copper mines,"

“Reuters, Tokyo June 2007 - Japan's Pan Pacific Copper Co. Ltd. has settled its mid-year copper processing fees with leading miners at $50-$55 per tonne and 5.0-5.5 cents per pound, a company official said on Tuesday.”

“Pan Pacific's deal compares with that signed between Japan's Sumitomo Metal Mining Co. Ltd. and Highland Valley at $45 a tonne and 4.6 cents a pound, which included price participation of 1.5 percent above $1.50 a pound with a 4 cent cap.”

"U.S Geological Survey, May 2006 - The USGS used the relation between per capita income and per capita copper consumption to estimate copper consumption in the 20 most populous countries in 2020. The results suggest that world copper consumption will increase an estimated 3.1 percent per year from 14.9 Mt in 2000 (our base year) to 27 Mt in 2020. Most of the increased consumption will take place in developing countries. For example, copper consumption in the United States and Japan will increase from 3 Mt and 1.3 Mt in 2000 to 3.5 Mt and 1.4 Mt respectively in 2020, while copper consumption in China and India will increase from 2 Mt and 400,000 t in 2000 to 5.6 Mt and 1.6 Mt respectively in 2020."

Based on the current market scenario, the copper smelter treatment and refining charges are most likely negotiated at a range of $50 - $55 per tonne and 5.0 – 5.5 cents per pound. A limited supply of copper concentrate and escalated consumption of copper metals will lead to a temporary shortage of copper in the world. According to the recent pricing trend in the market it is anticipated a downward trend of copper smelting costs will be continue. Copper smelters in the world would be continuing to suffer from a low treatment and refining fees in the coming years.

Figure 25-24 taken from the website: www.economagic.com, illustrates the production cost index of primary smelting and refining of copper from 1990 to 2007, base 2002.

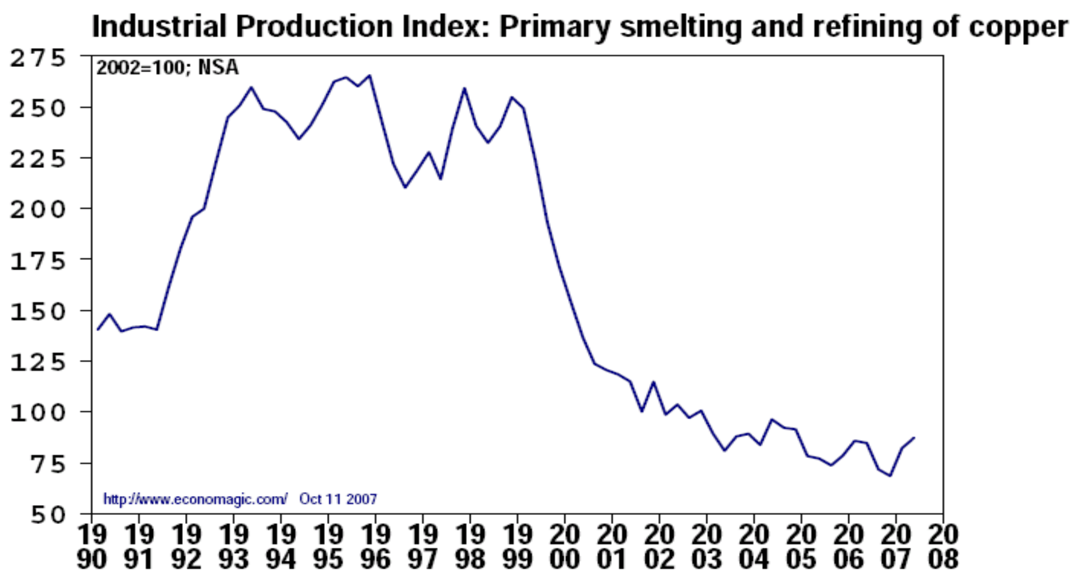


Figure 25-24: Primary Cost Index for Primary Smelting and Refining of Copper

Net smelting payable value of a custom smelter that pays to a miner on a long term contract or spot market is negotiated on the following terms;

- Concentrate treatment charge ($T.C_{cu}$)
- Concentrate refining charge ($R.C_{cu}$).
- Quotation period for payable metals (QP_{cu}, QP_{Au}, QP_{Ag}),
- Payable precious metals and refining charges ($R.C_{Au}$; $R.C_{Ag}$)
- Payment schedule (provision payment, etc)

A summary of the assumption for treatment and refining and other commercial terms for copper concentrate are presented in the next section.

25.4.1.2 Payable Metals

Copper = Unit deduction x copper content in concentrate. Deduct 1 unit and pay for balance of content with refining charges of USD 0.050-0.055/lb.

Silver = Contained Silver x Silver recovery. If over 30g/dmt, 90% pay with a refining charge of USD 0.35/oz.

Gold = Contained Gold x Gold recovery, If less that 1 g/dmt no pay, with a refining charge of USD 5.00/oz.

With a payable gold scale table as below;

Table 25-10: Payable Gold Scale

Gold Grade	Payable %
1 to 3 g/dmt	90.0
3 to 5 g/dmt	93.0
5 to 7 g/dmt	95.0
7 to 10 g/dmt	96.5
10 to 20 g/dmt	97.0
Over 20 g/dmt	97.5

25.4.1.3 Deductions

The following deductions are determined from the current studied market trend data.

- Treatment Charge (TC): USD 50 - $55/dmt
- Price participation (PP): 1.5 percent above $1.50 a pound with a 4 cent cap.
- Penalties:

Table 25-11: Trace Element Penalties

Trace Element	Penalty
Arsenic	USD 3.00 per 0.1% over 0.2%
Antimony	USD 3.00 per 0.1% over 0.1%
Lead	USD 3.00 per 1% over 2%
Zinc	USD 3.00 per 1% over 4%
Mercury	USD 0.200 per ppm over 20ppm
Bismuth	USD 5.00 per 0.1% over 0.05%
Selenium	USD 3.00 per 0.01% over 0.05%
Fluorine	USD 3.00 per 10ppm over 300ppm

25.4.1.4 Payment

Provisional - 90% on arrival of an ocean vessel, which for average tonne deemed to be 45 days after production. This deemed time may vary once production volume and shipment size and frequency is determined. Final 10% balance when all facts known deemed to be 150 days after production.

25.4.1.5 Net Payable Metal Value

Net payable metal value = Net metal value recovered – Treatment cost – Refining cost – Penalty elements cost - Transportation cost

Net metal value recovered = [Rec(cu) * Cu in conc (lb) * Cu metal price ($/lb)] + [Rec(Au) * Au in conc (oz) * Au price ($/oz)] + [Rec(Ag) * Ag in conc (oz) * Ag price ($/oz)]

Treatment cost = Tonnes of concentrate treated * treatment cost ($/t)

Refining cost = [Rec(cu) * Cu in conc (lb) * Refining charge for Cu ($/lb)] + [Rec(Au) * Au in conc (oz) * Refining charge for Au ($/oz)] + [Rec(Ag) * Ag in conc (oz) * Ag refining charge ($/oz)]

(Where: Rec(Cu) = accountability of copper, Rec(Au) = accountability of gold and Rec(Ag) = accountability of silver in the refining process.

Cu in conc = copper grade in concentrate, Au in conc = gold grade in copper concentrate and Ag in conc = silver in copper concentrate.)

The following applies to the Bronson Slope Project concentrate with an expected copper grade of 25.2% (based on the Average sample), 73g/t Au and 249g/t Ag:

- Treatment charge = USD 50/dmt
- Copper accountability = 96.3%
- Gold accountability = 97.5%
- Silver accountability = 90%
- Copper refining charge = USD 0.055/lb
- Gold refining charge = USD 5.0/oz
- Silver refining charge = USD 0.35/oz

25.5 Contracts

SGC have advised they have no existing contracts that would have a significant impact on the results of this Preliminary Assessment.

25.6 Environmental Considerations

As part of the permitting process for a new mine in British Columbia, SGC will need to submit a full environmental impact assessment (EIA) report to the government to facilitate the issuance of relevant permits and licenses required for mining operations. Environmental consultants will need to be employed to provide full environmental data, information, plans and recommendations. Data from past studies (done before 1997) may still be applicable while other baseline studies may have to be revised up to date. This section will briefly outline certain environmental considerations and recommendations in the submission of an EIA.

25.6.1 Regulatory Framework

The federal environmental process, governed by the Canadian Environmental Assessment Agency (CEAA), is the federal measure by which the project's integrity is tested. In a similar mandate to the BCEAA, the CEAA also ensures that the environmental effects of projects are carefully reviewed before federal authorities take action in connection with them so that projects do not cause significant adverse environmental effects. Under CEAA, projects again receive a level of environmental assessment tailored to their impact potential. There are four environmental assessment review options under CEAA: screening, comprehensive study, mediation and panel review. A comprehensive study report / environmental assessment are required to satisfy both the provincial and federal approval requirements. The British Columbia Environmental Assessment Act details the process required for environmental assessment of a mining project in British Columbia. The following is a summary of this process:

Step 1: Determine if the British Columbia Environmental Act Applies

Step 2: Determine the appropriate review path

Step 3: Determine how the assessment will be conducted

Step 4: Develop and approve the application terms of reference

Step 5: Prepare and submit the application

Step 6: Application review

Step 7: Prepare the Assessment Report

Step 8: Issuance of the certificate (upon successful application)

25.6.2 Plans and Descriptions

A series of plans and method statements need to be provided. The Bronson Slope Project will need to initiate the CEAA process, and a comprehensive study report (inclusive of the plans and descriptions) is required for submission to both the BCEAA (provincial) and the CEAA (federal). This environmental assessment is meant to satisfy both the provincial and federal approval requirements. On the matter of public consultation for First Nation groups, government and the community at large, an extensive outreach to interested parties who will become involved in the Bronson Slope Project will be needed. This documentation should include:

- Mining – outline details such as the planned mining method, layout, schedule, LOM, number of pits, planned equipment, locations of waste dumps and stockpiles and the handling of acid generating waste.
- Processing – planned processing rate, mill capacity, location, planned utilization, process description, treatment of residual materials, details of transporting pipe works and filter plant plans.
- Water Supply – supply of potable and non-potable water, treatment and storage process.
- Tailings and Waste – Construction and plans of tailings dam, waste management plans and process and operation procedures. Proposed waste management plan can be found in section 7 of “Bronson Slope Mine Plan” written by Christopher M. Turek and David A. Yeager. The waste management plan here is not definitive since the ARD potential of the rock has not yet been fully established. However, it covers aspects of waste management such as waste rock disposal, tailings disposal, mechanical wastes and site wastes (such as sewage, camp waste, solid waste and other waste). The report titled “Bronson Slope Mine Conceptual Design of Tailings Facility” outlines a proposed tailings facility plan. Further evaluation of ARD is required prior to a full review and update being conducted on the design of the waste rock storage and tailings storage facility.
- Freshwater diversions – runoff diversion plans around infrastructure and facilities, structures required to divert and control flow, diversion creek crossings, debris management and emergency overflow areas.

- Other facilities and access roads – plans and layout with emphasis on effects to sensitive habitats.

25.6.3 Environmental Setting and Impacts

A full description of the mine's environmental setting and the relevant geographical region is required for an impact assessment. Details should include location, climate, topography and studies on biogeoclimatic zones. SGC (or the employed consultants) will also need to determine the boundaries of the project and the appropriate regional 'Land and Resources Management Plan' it falls under.

Different components of the environment require particular consideration during the planning and design of the Bronson Slope Project. These components are sometimes widely termed Valued Ecosystem Components (VECs) and can be identified through a comprehensive consultation with the local communities, federal and provincial regulatory bodies, and other interested parties. Evaluation of the potential environmental effects of the Bronson Slope Project begins with the VECs. They are the most representative aspects of the natural environment deemed susceptible to influence by the wide-reaching scope of the project. As the environmental footprint of the project may be large, the determinations about the future integrity of each VEC over the life of the mine are crucial to the development of initiatives designed to lessen environmental impacts whenever it is reasonable to do so. VECs, to be determined in the future, that may require monitoring include:

- Air quality
- Climate change
- Site noise
- Surface water – protected under the *British Columbia Water Act* and *Canada Water Act*
- Groundwater (quantity and quality)
- Aquatic resources – micro-organism effecting ecosystem, cycling nutrients, photosynthesis and the production and processing of organic matter
- Sediment quality – monitoring required under federal metal mining effluent regulations
- Fish and fish habitat – ecological, economic and cultural health to BC
- Wetlands – minimize impacts due to ecological importance
- Terrestrial ecosystem, vegetation and soil landscapes
- Wildlife and wildlife habitat
- Archaeology – protected in BC under the *Heritage Conservation Act*

25.6.4 Socio-Economic Settings and Impacts

A socio-economic overview assessment of both the current setting and the effects of prospective mine developments in BC will need to be conducted. The report will need to show a general socio-economic benefit from the Bronson Slope project. It is recommended that the assessment address social concerns related to the new developments in the affected communities.

With the exception of Stewart, the residents of northwestern B.C. are largely members of the Tahltan Band and Iskut First Nation living in the communities of Dease Lake, Iskut and Telegraph Creek. The Tahltan and Iskut people claim extensive territorial hunting and fishing grounds. These long-inhabited Tahltan communities are considered to lie within the primary area of socio-economic impact of the Bronson Slope Project.

Northwestern B.C. is relatively remote from the rest of the province and supports a small population generally dependent upon the region's resource base, making land-based economies important for Tahltan people. The nearest large communities to the project site are Terrace and Smithers to the south and southeast; these communities lie within the secondary area of impact of the Bronson Slope Project.

The Tahltan and Iskut have identified the long-term viability of the regional mining sector as an essential driver for the economic, cultural and political advancement of the Tahltan Band and Iskut First Nation. The total population resident along Highway 37 is approximately 1,000, two-thirds of whom are Tahltan and Iskut. The remoteness of Dease Lake, Iskut, Telegraph Creek and Stewart and the limited availability of employment opportunities have contributed to extensive out-migration of residents.

Many Tahltan and Iskut see a long-term sustainable mining industry as providing the means to encourage and sustain their culture; they welcome economic development that will benefit the Tahltan people and culture, and provide an incentive to former residents to return to their home communities. The Tahltan Nation Development Corporation (TNDC) was created through the collaborative efforts of the predominantly Iskut and Tahltan populations of Dease Lake, Iskut and Telegraph Creek. Representing the Iskut First Nation, the Tahltan Band and the Tahltan Central Council (TCC), the TNDC has evolved into a major local and regional employer and a force for Tahltan economic development through its own activities and through joint-venture relationships with other companies. It has established a range of long-term initiatives geared to increasing Tahltan employment, enhancing skill levels and ensuring sustainable economic livelihoods for greater numbers of Tahltan people.

Community-based issues are most likely to generate concern from government and the public. The TCC, elders, leaders and community members of the Tahltan and Iskut Bands, the District of Stewart, local governments of Smithers and Terrace as well as representatives of regional, provincial and federal governments should all be consulted during the environmental assessment. Scope of issues may range from employment and business development opportunities, substance abuse, highway traffic and accidents to the cultural implications of development and the ability of local jurisdictions to be opportunistic.

Another important issue is to determine existing education, social and health programs and capacities. Mining development may stress present communities but the benefit of providing stable employment, training and apprenticeship openings, business supplier opportunities and economic stability for the community needs to be assessed. Plans should be developed to minimize potential adverse impacts from the Bronson Slope project while enhancing opportunities for the local communities.

25.6.5 Environmental Management, Monitoring and Follow Up

SGC (or the employed consultants) will require planned monitoring systems designed to monitor sentinel environmental components or the determined VECs. Procedures and draft plans for the rehabilitation, reclamation and closure of the mine at the end of its life will also be required.

25.6.6 Recommended Baseline Studies

Recommendations regarding environmental baseline studies to be undertaken during the current field season in support of the Bronson Slope Mine are outlined from the report dated 7th August 2007 and written by Neil D. Mallen. Recommendations are made in regards to hardware installation, aquatic sampling, terrestrial surveys and general / design mitigation considerations. Suggested preliminary cost allowances are provided in Table 25-12.

Table 25-12: Recommended Baseline Studies Preliminary Cost Allowances

Recommendation	Qty	Item	Cost	Subtotal
(1) Weather Station	1	Hobo Stn	$3,000	$5,700
	1	Lakewood R-X Ultra-logger or equivalent	$2,000	
		Installation[1]	$700	
(2) Streamflow Dataloggers	3	Instrumentation Northwest PS-9800 Pressure Transducers or equivalent	$3,000	$10,400
	3	Lakewood R-X Ultra-loggers or equivalent	$6,000	
		Installation[1]	$1,400	
(3) Groundwater Monitoring Well		Oversee Drilling, Installation, Purge & Sample[2]	$5,400	$8,400
		Incidental Supplies	$1,000	
		Laboratory Analysis[3]	$2,000	
(4) Water Quality Sampling	1	Handheld water quality sensor	$1,000	$4,200
		Sampling	$700	
		Laboratory Analysis	$2,500	
(5) Sediment & Aquatic Biota Sampling		Sampling[1]	$2,100	$10,600
		Laboratory Analysis (Sediments)	$1,500	
		Laboratory Analysis (Aquatic Biota)	$7,000	
(6) Terrestrial Ecosystem Mapping & Rare Plant Survey		Mapping[4]	$7,000	$8,400
		Field Survey[5]	$1,400	
TOTAL 2007 BASELINE PROGRAM				**$47,700**

1. An environmental technician at $50 per hour and an assistant at $30 per hour. Travel, accommodation and other disbursements are excluded.

2. One monitoring well with two piezometres is assumed. The allowance is for a professional geoscientist at $150 per hour and a First Nations assistant at $30 per hour. Costs to drill the well are excluded. Travel, accommodation and other disbursements are excluded.

3. The allowance is for analysis of two samples (assuming a shallow and a deep piezometer installed in the well) for typical parameters including physical parameters, nutrients, dissolved anions, total organic carbon, total cyanide, and total and dissolved metals.

4. This is a preliminary assessment based on previous experience with similar-sized study areas.

5. The allowance is for a registered professional biologist at $150 per hour and a First Nations assistant at $30 per hour. Travel, accommodation and other disbursements are excluded.

The summarized cost estimates above excludes professional fees to coordinate the work. Baseline characterization of air quality, ambient noise, terrain stability, soils, potential metal leaching and acid rock drainage (ML/ARD), wetlands, heritage resources and other issues are excluded at this stage.

25.7 Taxes

The Bronson Slope mining project, being operated by a Canadian mining company, is subject to taxation at the federal, provincial and local level. All monetary values in this section are quoted in Canadian dollars unless stated otherwise. The taxes of primary importance for the Bronson Slope Property are as follows:

- Federal taxes – Federal Income Tax, Goods and Services Tax (GST)
- Provincial Taxes – Provincial Income, Capital, Mining and Sales Tax

Other less significant taxes may apply to the project, however for the purposes of this study they have not been considered. A detailed review of taxation requirements should be conducted prior to completion of a detailed feasibility study.

25.7.1 Federal Tax

Between 2003 and 2007 the Federal government incrementally reduced the corporate income tax rate for mining companies to the general 21% rate that applies to other corporations. The general tax rate is to be reduced such that it will be 15% in 2012.

In general terms, federal taxable income for the Bronson Slope project can be defined as mining revenue less the following deductions:

- Operating costs
- Capital cost allowance (CCA)
- Resource allowance
- Canadian exploration expense (CEE)
- Cumulative canadian development expense (CCDE)
- Interest expense
- Crown royalties and provincial mining taxes paid.

Crown royalties and provincial mining taxes became fully deductible for mining firms in 2007 to compensate for elimination of the Resource Allowance.

A GST of 5% of the cost of goods and services, similar to a value added tax (VAT), is collected at each stage of processing or distribution. A mining company recovers the tax it has paid for goods and services through a tax it levies on the sale of its own products.

25.7.2 Provincial Tax

Taxable income derived from operations at Bronson Slope is subject to Corporate Income Tax at the rate 11.0%. Taxable income is the federal taxable income adjusted by deducting the provincial taxes paid under the Mineral Tax Act, and adding back the allowable federal resource allowance.

Under the Mineral Tax Act, mining companies pay Mineral taxes in two stages. The stage I tax is 2% of net current proceeds (defined as the current year's gross revenue less operating costs). Operating costs are all current operating costs, but do not include expenses due to capital investment such as preproduction exploration and development expenses. If the mine has an operating loss, no net current proceeds tax (stage I tax) is payable. After the company's investment and a reasonable return on investment have been recovered, the company must pay the stage II tax of 13% of adjusted net revenue, essentially the net current proceeds from Stage I tax computations from the mine. The stage I tax is deducted from the stage II tax owed, so the maximum tax does not exceed 13%. Any previous stage I tax paid is deductible from the stage II tax owed. It can be carried forward indefinitely.

25.8 Capital and Operating Cost Estimates

25.8.1 General Site Infrastructure Capital Estimate

A detail cost estimate was carried out by Rescan Engineering as part of the draft pre-feasibility report in July 1997. This cost estimate has been used as the basis for the revised cost estimate presented in this report.

The cost estimates presented in this section are derived using the following method:

1. Develop a list of required infrastructure

2. Use infrastructure estimates from recent copper/gold mines in British Columbia with similar production rates (13,000-20,000t per day) and strip ratios (0.3-0.6:1) to the Bronson Slope Project and use Infomine's (2007) Estimator Guide to develop a first pass cost estimate
3. Develop a construction cost index to convert Rescan Engineering's 1997 costs into 2008 costs
4. Compare estimates with converted Rescan Engineering's costs and make adjustments to expectations that have changed since 1997 (such as length of required roads, design aspects, accommodation expectations, etc)
5. Draw up a final estimate costing based on points 1-4

25.8.1.1 List of Required Infrastructure

Proposed access road from Eskay Creek road intersection - The proposed road length from the Forrest Kerr to the mining property is approximately 32km based on Forsite's access road study (2006) and revised cost estimate submitted in 2008.

Power transmission - Length of lines is 60km and uses wooden poles. Costs include mobilization, labour, materials, equipment operation, overhead profit, clearing, pole supply and installation, installation of conductors, overhead ground wires, insulators, clearance poles and surplus material disposal. An allowance is also included for joints and dead ends, clipping and installation of jumpers and spacers. Minimum power line requirements are 138kV (UBC 1996). Power lines will run from Bob Quinn to the mine site.

Site access roads – An allowance has been made for the costs for the proposed access road, initial pit access roads and access for the conveyor systems.

Drainage control – An allowance for drainage is based on drain dimensions of 20m base, 30 degree batter, 5m deep. The cost also includes culvert installations.

Separate permanent lighting – The cost for site development lighting, including additional road and yard lighting.

Fencing and security gatehouses - The cost includes fencing and security gatehouses. Each security gatehouse consists of an aluminium cargo container with modifications that include walk in door, fluorescent lighting, window, electrical outlet and complete wiring.

Power supply - This allowance only includes emergency backup power which consists of an 1100kW genset with all required cabling, lighting, transformers, panels and grounding.

Site administration/engineering office building - The cost includes the construction of a 588m^2 office and office equipment setup such as servers and computers.

Camp facilities and medical post - The camp will include all infrastructures to facilitate a total of 120 people. The medical post will be developed as an annex to the camp building.

General warehouse and site storage facilities - The warehouse will have steel framing and painted steel cladding, and be located beside the mechanical workshop. A secure, open area of at least 550-650m^2 will be incorporated within the warehouse facility for merchandise and goods that may be stored out-of-doors. The enclosed storage area, including office space, will be 300-400m^2 and should have a ceiling clearance of at least 7m so that an overhead crane can be installed. The metallurgical processing reagents storage area will include a perimeter barrier so that any spills are contained and can be easily cleaned up.

Maintenance workshop - The mechanical and electrical sections of the workshop will provide sheltered spaces for the repairs and scheduled maintenance of the mine and plant equipment.

The maintenance workshop will have an overhead clearance of at least 7m and should be fully enclosed and heated.

Assay/sampling laboratory - A chemical laboratory will provide a sheltered and fully-equipped facility for sample preparation, assays, and quality control of the plant production. A separate or adjoining laboratory will be developed for metallurgical testing.

Geology warehouse - The planned facilities will expand upon existing buildings, thus the construction costs will be less than initially estimated. The design includes a sheltered floor area of at least 200 m^2 for sample storage.

Fuel storage and dispensing facility - The fuel storage and dispensing facility is priced with a lined, containment area. Estimation includes three 50,000L and one 25,000L steel saddle mounted fuel tank (located in mine maintenance facilities for in-pit operations). Allowance for pumping equipment and metering devices are also included.

Sewage/waste water treatment - Costing for the construction of a waste water treatment plant where non-process waste water from some of the site facilities, such as the camp and offices, will be treated. Costs also include an engineered facility within the mine concession where sludge material produced by the sewage treatment plant will be stored.

Dust suppression water, potable water and fire water supply - This includes the allowances for the construction of water bores, genset pumps, pipelines (laid and buried), standpipes and storage facilities such as sumps, turkey's nests or above ground tanks. Potable water treatment units will comprise of 1-micron and 10-micron cartridge filters, UV disinfection unit, a hypochlorite addition systems, raw water tank, small mix tank, metering pumps and booster pumps. Flow metres will be installed to monitor fresh water consumption.

Control and communication system - The allowance here will include setting up communications infrastructure and transmission such as phone, fax, radio, fire alarm, etc. The control system will include monitoring systems on operational performances.

Explosives facilities- A quotation has been obtained from Dyno Nobel in Canada for the required explosives loading facilities. The costs are charged as a lease and the initial mobilization capital costs are paid up front as capital.

25.8.1.2 Development of Canadian construction related costs indices

Table 25-13 below demonstrates the cost indices that have been used to scale the costs estimated in 1997 to 2008 prices.

Table 25-13: Development of Canadian Construction Related Cost Index

	1997	2008	% change	Proportion for final index
Canadian Construction Cost Index (Statistics Canada)	100	141.8	42%	0.3
Canadian Unit Labour Cost Index (SC)	100	135.6	36%	0.15
Canadian Construction Machinery/Equipment Cost Index (SC)	116.61	133.22	14%	0.1
Canadian Construction Materials Index (SC)	117.3	176	50%	0.15
Surface Mine Capital cost index - Canada (Infomine)	106.5	125.6	18%	0.3
Av. % index increase in construction related indices			27%	
Av Compound % per year			2.18%	
Canadian CPI av % increase (for reference)			20.70%	

25.8.1.3 Indirect Costs

Engineering, procurement and construction management costs - Costs are included for the engineering, procurement and construction management for the project and have been estimated at 12% of direct costs based on similar projects in BC and Rescan Engineering's estimates.

Construction in-directs - Construction in-directs are based on estimated figures from Rescan Engineering (1997).

Freight and insurance - An allowance for freight, insurance and mobilization is based on Rescan Engineering's estimations.

Start-up and commissioning - An allowance of CAD250,000 has been made to cover the cost of commissioning crews based on typical durations and preproduction employment rates and commissioning material costs.

Contingency - A contingency of 15% has been included in the estimate.

25.8.1.4 Infrastructure Capital Cost Estimate

Table 25-14: Final Estimated Infrastructure Capital Costs (excl. tax considerations)

Item No.	Description	Estimated allowance/cost (CAD 000) 2008
	Off-site Infrastructure	
1	Proposed access road to site from Eskay Creek road intersection (~30km)	$7,576
2	Power distribution (~60km)	$17,018
	Site Development	
3	General	-
4	Site access road, initial pit access roads and access road to conveyor	$3,934
5	Drainage control and culverts installation	$1,368
6	Separate permanent lighting	$91
7	Fencing and gatehouses allowance	$102
	Infrastructure - Utilities	
8	Power supply	$780
9	Power distribution on site	$1,835
10	Fuel supply and distribution	$133
11	Fresh water supply and distribution	$48
12	Water and sewage treatment	$47
13	Fire protection and prevention	$38
14	Waste Disposal	$45
15	Control and communications system	$128
	Infrastructure - Buildings and Facilities	
16	Site administration/Engineering office building	$1,012
17	Camp facility and medical post to facilitate 120	$4,066
18	General Warehouse and site storage facilities	$1,911
19	Maintenance workshop	$706
20	Assay/sampling laboratory	$1,528
21	Expand existing geology warehouse	$397
22	Explosives magazine and AN/emulsion storage	$42
	Estimated initial infrastructure cost (not including processing/tailings facilities)-->	$42,805
	Indirect Costs	
	Engineering, procurement and construction management costs (12%)	$5,137
	Construction indirects	$1,177
	Freight, insurance and mobilization (7%)	$2,996
	Start up commissioning	$3,008
	Contingency (15%)	$6, 420
Total Capex for Site Infrastructure (CAD ,000)		**$61,543**

25.8.2 General and Administration Operating Cost Estimate

Considerations for mining, processing and general overheads and administration costs (overheads) have been made as part of the estimate for site operating costs. The operating cost estimate also includes consideration for a range of ancillary equipment that will provide general support to the operations. The total allowance for the LOM general and administration costs equates to CAD91.6 Million (CAD0.55/t mined, CAD0.98/t milled). The following items, with varying allowances dependent upon the mining stage, have been included:

- Medium and heavy trucks required for general transport around the site and for carting and dispensing fuel and lubricants.
- A front end loader fitted with an integrated tool set including a tyre handler.
- A 20t mobile crane for general purpose site use as well as an allowance for periodic events requiring heavy lifting equipment
- Portable workshop equipment such as compressors, welders, steam cleaners, and maxi heaters
- Key workshop items such as heavy lift jacks, stands and specialist tooling such as torque wrenches.
- Mobile lighting plant for provision of night lighting to working areas.
- A small portable generator for field use.
- A portable pump and standpipe for in pit filling of the watercart when possible.
- The light and medium vehicles including buses for transportation of personnel between camp and the mine site.

In addition to the provision of support equipment allowances have also been made to include for maintenance and replacement of computers and associated hardware, recurring costs for telecommunications and two-way radio communications, mine planning software licenses. Allowances have also been made for Site utilities such as power, sewerage and potable water, security and waste disposal.

Allowances for safety and training in this element include:

- Personal Protective Equipment (PPE) – such as clothing, winter gear, footwear, safety glasses, helmets, dust masks, ear plugs, gloves, torches and batteries, UV protection.
- General safety consumables
- Mine site signage and traffic controls.
- Fire protection – portable fire extinguisher supply and maintenance
- Recurring medical expenses – pre-start medical examinations, first aid kits aid supplies, ongoing alcohol and drug testing.
- Direct training expenses required for key operations and maintenance personnel to ensure a level of proficiency consistent with the estimated productivities.

An allowance has been made for the maintenance of fixed mine infrastructure buildings and facilities. Progressive mobilisation and establishment cost after production start is included in this element.

Mobilisation and establishment of the initial mining fleet and associated personnel and equipment is included in the separate mobilisation and establishment item as part of the pre-production capital estimate.

Allowances have been made for estimated inventory holding costs associated with spare parts for mining equipment, permits, legal fees, contract administration, and community welfare and donations.

Costs associated with maintaining administration and senior management staff onsite have also been included. These staff and costs have been summarised in Table 25-15.

Table 25-15: Administration Manpower Summary

Description	Manpower	Hourly Base rate CAD	Base Salary CAD	Salary Burden	Unit Salaries CAD	Total salaries CAD
Basis:	2 weeks in / 2 weeks out			42% of base		
ADMINISTRATIVE				42%		
General Manager	1		$ 160,000	$ 67,200	$ 227,200	$ 227,200
HR Superintendent	1		$ 85,000	$ 35,700	$ 120,700	$ 120,700
Safety Officer	1		$ 65,000	$ 27,300	$ 92,300	$ 92,300
SUB-TOTAL G&A ADMIN.	**3**		**$ 245,000**	**$ 102,900**	**$ 347,900**	**$ 347,900**
ACCOUNTING						
Chief Accountant	1		$ 85,000	$ 35,700	$ 120,700	$ 120,700
Accounting Assistant	1		$ 55,000	$ 23,100	$ 78,100	$ 78,100
Warehouse Supervisor/Buyer	1		$ 60,000	$ 25,200	$ 85,200	$ 85,200
Secretary/Receptionist	3	$ 20.00	$ 45,000	$ 18,900	$ 63,900	$ 191,700
SUB-TOTAL ACCOUNTING	**6**		**$ 245,000**	**$ 102,900**	**$ 347,900**	**$ 475,700**
Basis:	2 weeks in / 2 weeks out		2016 hrs/yr	42% of base		
OPERATING LABOUR			2016	42%		
Yard Foreman	2		$ 55,000	$ 23,100	$ 78,100	$ 156,200
Warehouse Assistants	4	$ 25.00	$ 50,400	$ 21,168	$ 71,568	$ 286,272
Labourers - Plant/Yard	6	$ 23.00	$ 46,368	$ 19,475	$ 65,843	$ 395,055
Janitors	2	$ 23.00	$ 46,368	$ 19,475	$ 65,843	$ 131,685
SUB-TOTAL G&A OP. LABOUR	**14**		**$ 198,136**	**$ 83,218**	**$ 281,353**	**$ 969,212**
TOTAL G&A MANPOWER	**23**					**$ 1,792,812**
					CAD/tonne	**$ 0.352**

25.8.3 Processing Capital Cost Estimate

The estimated capital cost for the concentrator plant discussed in Item 18 is based on the following:

- Preliminary flow-sheet and equipment requirements.
- Equipment costs from vendors' estimation, suppliers' quotation and Western Infomine's estimator cost data.
- Escalation of the 1997's costs to present value using Western Infomine's capital cost index data.
- Mechanical installation costs for equipment are calculated by multiplying labour cost by the number of labour hours required.
- Other processing plant installation costs are either adjusted by an allowance factor or using an escalated capital cost index.
- Comparison to cost models of similar operations.

Table 25-16: Summary of Processing Direct Capital Costs

Direct Costs	Total CAD 000's
AREA 13 – Primary Crushing and Ore Stockpile*	$6,402
AREA 16 – Grinding	$43,014
AREA 17 – Copper Flotation	$9,730
AREA 18 – Copper Concentrate Dewatering	$2,181
AREA 20 – Reagent Systems	$1,014
AREA 28 – Process Utilities	$3,618
AREA 31 – Tailings	$21,064
Total Direct costs	**$87,023**

*Mobile crusher included in Mining capex

For the purpose of this capital cost estimation, the following assumptions have been made:

- The process design criteria and flow sheet provides a plant capable of processing 15,000tpd of material.
- During the preparation of this estimation, some of the metallurgical information and data are unavailable. To complete this exercise that some of the proposed process equipment selection is based on similar existing processing practice in the industry, or recommendation by equipment supplier.
- All the main process mechanical equipment has been chosen for the plant are captured, included in the estimate and suitable for the process.
- Other provision of infrastructure, quantity of take offs, labour requirements and allowance of pipe work and instrumentation are valid.
- The capital cost estimate has not included contractor engineering management, commissioning, contingency and working capital costs.

The capital estimate cost consists of the following main contributors:

- Process equipment costs
- Labour construction and installation costs
- Allowance costs for process pipe and valves
- Process plant building structure cost
- Electric and instrumentation cost

The estimate proportions of labour, mechanical, structural and electrical of the capital cost are summarized in Table 25-17.

Table 25-17: Processing Capital Costs by Discipline

Direct Costs	CAD 000's
Labour	$8,994
Mechanical	$43,027
Structural	$32,619
Electrical	$2,384
Total	**$87,023**

Other costs that are not included in the capital estimate are tailing embankment construction cost, contingency cost and indirect costs (construction indirect, engineering, procurement and commissioning, working capital). The cost allowance for indirect costs and contingency cost are calculated, based on percentage of the total direct cost. The indirect costs are presented in Table 25-18.

Table 25-18: Summary of Indirect Costs

Indirect Costs (CAD ,000)	CAD 000's
Construction indirect (3% of direct cost)	$2,611
EPCM engineering services (10% of direct cost)	$8,702
First fill inventory (5% of direct cost)	$4,351
Total Indirect Cost	**$15,664**
Contingency (15% of direct cost)	**$13,053**
Total Costs (Direct + Indirect + Contingency)	**$115,740**

Most of the major process mechanical equipment (mills, cyclones, flotation cells, knelson concentrator, magnetite separator and pumps) are quoted from the equipment suppliers. Other minor mechanical equipments are obtained from the Western Infomine's milling equipment cost manual otherwise the 1997's cost is adjusted to current value by a CPI cost index.

Construction labour rate is estimated, based on the average labour rate in the industry as compiled in the Western Infomine manual. The average construction labour hourly rate of CAD 58.00 is used for this study, the cost has included allowances of health, holidays and pension, overtime premium, contractor supervision and overhead cost, and contractors' profit.

Plant building, electrical and instrumentation requirements are based on the Rescan Engineering 1997's study. The estimate costs are adjusted to the present value using the Infomine's cost index. The cost has included material cost, installation cost and construction cost that are required for the work.

Allowance of pipe work and valves are based on an industry rule of thumb of 11% of the mechanical equipment material costs.

The capital cost estimate is then validated with comparison to the cost model developed by the Western Infomine's estimate cost manual.

25.8.4 Concentrator Plant Operation Cost Estimate

The estimated plant operating costs are based on the following:

- Manpower, standard labour rate and salary package compiled by Western Infomine's statistic data.
- Mill consumables and cost are derived from typical consumption rate in the industry.
- Reagent consumables are determined from the preliminary test work indicating typical reagent consumption can be expected.
- Power costs are determined by multiplying equipment power requirement by the commercial power cost supplied in British Columbia.

A summary of the costs are shown in Table 25-19 below.

Table 25-19: Summary of Process Plant Operating Costs

No.	Description	CAD/tonne milled
1	Process Labour	1.10
2	Power cost	1.53
3	Consumable & Maintenance	2.52
4	Surface Equipment	0.08
Total unit direct process operating cost		**$ 5.23**
USD direct process operating cost		**$ 4.45**

Operating cost is estimated from the following components;

Process Labour Requirements - Operating employees required for the plant concentrator. Operating employees required for the concentrator operation is based on general industry work practice. The salary packages and labour wages are estimated with reference to the Western Infomine's labour cost section. A summary of the processing manning requirements has been included in Table 25-20 below:

Table 25-20: Annual Processing Manpower Requirements

Description	Manpower	Hourly Base Rate CAD	Base Salary CAD	On Costs CAD	Overtime Allowance	Shift Premium	Unit Salary CAD	Total Salary CAD
Basis:	12 hrs/shift, 2 in/ 2 out		2016 hrs/yr	42% of base				
SUPERVISION			2016	42%				
Mill Superintendent	1		$ 95,000	$ 39,900			$ 134,900	$ 134,900
Assist. Mill Superintendent	0		$ -	$ -			$ -	$ -
Senior Foreman	1		$ 75,000	$ 31,500			$ 106,500	$ 106,500
Mill General Supervisor	0		$ -	$ -			$ -	$ -
Maintenance Superintendent	1		$ 85,000	$ 35,700			$ 120,700	$ 120,700
Maintenance Foremen	2		$ 75,000	$ 31,500			$ 106,500	$ 213,000
Plant Planner	1		$ 70,000	$ 29,400			$ 99,400	$ 99,400
Mill Clerk	2	$23.40	$ 45,000	$ 18,900			$ 63,900	$ 127,800
SUB-TOTAL SUPERVISION	**8**	**$23.40**	**$ 445,000**	**$ 186,900**	**$ -**	**$ -**	**$ 631,900**	**$ 802,300**
Basis:	12 hrs/shift, 2 in/ 2 out		2016 hrs/yr	28% of base	10% of base	5% of base		
TECHNICAL			2016	28%	10%	5%		
Metallurgical Engineers	2		$ 95,000	$ 26,600			$ 121,600	$ 243,200
Chief Assayer	1		$ 65,000	$ 18,200			$ 83,200	$ 83,200
Metallurgical Technicians	2	$27.58	$ 55,595	$ 15,567	$ 5,560	$ 2,780	$ 79,501	$ 159,002
Environmental Technicians	0		$ 64,350	$ 18,018	$ 6,435	$ 3,218	$ 92,021	$ -
Assayers / Sample Prep	4	$26.91	$ 54,251	$ 15,190	$ 5,425	$ 2,713	$ 77,578	$ 310,313
SUB-TOTAL TECHNICAL	**9**		**$ 334,196**	**$ 93,575**	**$ 17,420**	**$ 8,710**	**$ 453,900**	**$ 795,715**
HOURLY PERSONNEL								
OPERTATING LABOUR								
Senior Operators	4	$37.44	$ 75,479	$ 21,134	$ 7,548	$ 3,774	$ 107,935	$ 431,740
Operators	16	$32.76	$ 66,044	$ 18,492	$ 6,604	$ 3,302	$ 94,443	$ 1,511,090
Labourers	6	$24.32	$ 49,038	$ 13,731	$ 4,904	$ 2,452	$ 70,124	$ 420,744
SUB-TOTAL OPER. LABOUR	**26**		**190,561**	**53,357**	**19,056**	**9,528**	**272,502**	**2,363,575**
REPAIR LABOUR								
Mechanics/Welders	10	$30.00	$ 70,762	$ 19,813	$ 7,076	$ 3,538	$ 101,189	$ 1,011,891
Electricians	4	$32.00	$ 75,479	$ 21,134	$ 7,548	$ 3,774	$ 107,935	$ 431,740
Instrument Technicians	2	$32.00	$ 75,479	$ 21,134	$ 7,548	$ 3,774	$ 107,935	$ 215,870
SUB-TOTAL REPAIR LABOUR	**16**		**$ 221,720**	**$ 62,082**	**$ 22,172**	**$ 11,086**	**$ 317,059**	**$ 1,659,501**
TOTAL	**59**		**1,191,476**	**395,913**	**58,648**	**29,324**	**1,675,361**	**5,621,091**
							CAD/tonne	**1.103**

Power - Total power required for the plant operation. Determination of the total power required for the plant operation is basically to add up all the plant mechanical power requirements and multiple by the unit power cost. The power unit cost of CAD0.055 per kWh is used for this study. The unit rate is provided by SGC. Table 25-21 below provides the total power requirements and annual costs for the project.

Table 25-21: Site Power Requirements

AREA DESCRIPTION	Connected kW	Operating kW	Total Consumption (kWh/year)	Total Cost (CAD/year)	Unit Cost (CAD/t ore)
Area 13C - Secondary Crushing	1313	966	7,879,471	$ 433,371	$ 0.085
Area 16 - Grinding	12990	11950	97,473,786	$ 5,361,058	$ 1.052
Area 17 - Copper Flotation	3046	2390	19,494,757	$ 1,072,212	$ 0.210
Area 18 - Cu Dewatering	134	117	954,346	$ 52,489	$ 0.010
Area 20 - Reagent Systems	97	91	742,269	$ 40,825	$ 0.008
Area 28 - Process Auxillaries	2680	1388	11,321,641	$ 622,690	$ 0.122
Area 51 - Main Substation	33	27	220,234	$ 12,113	$ 0.002
Shops/Warehouse/Accomodation	577	404	3,295,348	$ 181,244	$ 0.036
TOTAL	**20,870**	**17,333**	**141,381,852**	**$ 7,776,002**	**$ 1.525**

CAD/tonne	**$ 1.525**

Consumables - Milling consumable required is based on the general industry consumption rate. Average industry consumable cost and the reagents consumption rates determined by the metallurgical test work are applied for the consumable cost calculations. Table 25-22 below provides a summary of the milling consumables usage and maintenance allowances.

Table 25-22: Mill Consumables and Maintenance Costs

Suppliers	Consumption Rate (kg/t ore)	Consumption (kg/yr)	Unit Cost (CAD/kg)	Total Cost (CAD/yr)	Unit Cost (CAD/t ore)
SAG Mill Balls, 5 in.	0.350	1,784,300	$ 1.00	$ 1,784,300	$ 0.350
Bal Mill Balls, 3 in.	0.400	2,039,201	$ 1.00	$ 2,039,201	$ 0.400
Regrind Mill Balls, 1 In.	0.400	2,039,201	$ 1.00	$ 2,039,201	$ 0.400
SAG Mill Liners	0.040	203,920	$ 2.93	$ 597,486	$ 0.117
Ball Mill Liners	0.033	168,234	$ 2.93	$ 492,926	$ 0.097
Regrind Mill Liners	0.020	101,960	$ 2.93	$ 298,743	$ 0.059
PAX (Xanthate)	0.008	40,784	$ 2.95	$ 120,313	$ 0.024
A208 - Promoter	0.008	40,784	$ 3.90	$ 159,058	$ 0.031
Flocculant (Percol 351)	0.010	50,980	$ 4.93	$ 251,331	$ 0.049
MIBC - frother**	0.051	259,998	$ 2.80	$ 727,995	$ 0.143
Lime as $Ca(OH)_2$[#]	0.060	305,880	$ 0.55	$ 168,234	$ 0.033
Maintenance Supplies	allowance			$ 2,500,000	$ 0.490
Assay Supplies	allowance			$ 145,000	$ 0.028
TSF Construction/op	allowance			$ 1,500,000	$ 0.294
TOTAL				**$ 12,823,787**	**$ 2.515**

CAD/tonne	**$ 2.515**

An allowance for running costs and maintenance cost for general surface mobile equipment required for the concentrate plant has been made. This equates to CAD0.089/t milled.

25.8.5 Mining Capital Cost Estimate

25.8.5.1 Mining Equipment Lead-Times

The current global supply/demand for mining equipment has led to extended lead-times for delivery of most items. It is uncertain at this stage if this situation is expected to abate in the next few years. Consideration should be made in procurement planning for a minimum allowance of 9 months lead time on all sizable mining equipment. Large mining equipment such as dozers, loaders, graders and excavators have a likely lead time of 9 to 15 months depending on the make and model.

25.8.5.2 Equipment Capital Pricing

Major production and mining support equipment units were priced using a combination of the following:

- Infomine data (see references)
- Vendor budget pricing
- Reference to recent quotations obtained by LAL for other projects

The estimates for capex were prepared for new equipment supplied on a turn-key basis and account for the anticipated cost of sea and land transportation of each item of equipment, import duties and associated port charges, and erection and commissioning costs for the Bronson Slope site location. VAT is specifically excluded from the pricing.

All equipment costs are in CAD and primarily based on 2nd half 2008 pricing with exchange rates of USD0.85 and AUD1.15 to CAD applied. No escalation of costs is estimated or included.

Replacement life for equipment is based on LAL's extensive experience operating a wide range of mining equipment. The hourly operating costs calculated for these equipment items are based on operating the equipment for its full expected life (average Whole of Life operating costs).

Table 25-23 summarises the estimate of overall equipment capex requirements for the life of mine based from an owner's perspective. Initial and sustaining capital estimates are provided.

Table 25-23 Summary LOM Mining Capital Schedule (CAD 000'S)

Initial and Sustaining Mining Capital Cost Items																			
Initial Items	Unit Price (CAD 000's)	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15-20	Total (CAD 000's)
Pre-Production Mining Costs	$ 18,090	$1,990	$16,100																$ 18,090
Conveying System (2300tph)	$ 11,958	$5,979	$ 5,979																$ 11,958
Rehandle Loader	$ 1,694		$ 1,694	$1,694															$ 3,388
Mobile Crusher (1200tph)	$ 1,684		$ 1,684	$1,684															$ 3,368
90t Class Rear Dump	$ 1,506		$ 3,012	$3,012															$ 6,024
Hyd Excavator 6.7m3	$ 1,365		$ 1,365	$1,365															$ 2,730
Drill DTH 203mm	$ 971		$ 971	$ 971															$ 1,942
Track Dozer 15' Blade	$ 682		$ 682	$ 682															$ 1,364
Watercart 20kL	$ 494		$ 494																$ 494
Grader 12' Blade	$ 341																		$ 341
IT Tool Carrier	$ 265		$ 265																$ 265
25T Exc/R.breaker	$ 175		$ 175																$ 175
Roller	$ 94		$ 94																$ 94
SubTotal - Initial Items (CAD 000's)		**$7,969**	**$32,856**	**$9,408**															**$ 50,233**
Replacement Items	Unit Price (CAD 000's)	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15-20	Total (CAD 000's)
Conveying System (2300tph)	$ 11,958																		$ -
Rehandle Loader	$ 1,694								$1,694	$1,694									$ 3,388
Mobile Crusher (1200tph)	$ 1,684											$1,684							$ 1,684
90t Class Rear Dump	$ 1,506									$3,012									$ 3,012
Hyd Excavator 6.7m3	$ 1,365								$1,365	$1,365							$1,365		$ 4,095
Drill DTH 203mm	$ 971													$ 971					$ 971
Track Dozer 15' Blade	$ 682													$ 682					$ 682
Watercart 20kL	$ 494																		$ -
Grader 12' Blade	$ 341																		$ -
25T Exc/R.breaker	$ 175								$ 175								$ 175		$ 350
Roller	$ 94																		$ -
SubTotal - Replacement Items (CAD 000's)									**$3,234**	**$6,071**		**$1,684**		**$1,653**			**$1,540**		**$ 14,182**
TOTAL MINE CAPITAL COSTS (CAD 000's)		**$7,969**	**$32,856**	**$9,408**	**$ -**	**$ -**	**$ -**	**$ -**	**$3,234**	**$6,071**	**$ -**	**$1,684**	**$ -**	**$1,653**	**$ -**	**$ -**	**$1,540**	**$ -**	**$ 64,415**

25.8.6 Mining Operating Cost Estimate

25.8.6.1 Overheads and Manning Requirements

Overheads for mining were divided into three different stages, depending on the number of excavator fleets working and on organisational changes as time progresses, and spread accordingly.

Theses stages are summarized as;

- Mining Stage 1 1 x Fleet in pre-production phase (Year -2 to -1)
- Mining Stage 2 2 x Fleets production phase (Years 1 – 8)
- Mining Stage 3 1 x Fleet and ramp down strategy (Years 9 to 20)

Mining personnel numbers are detailed for each year in Appendix 2. All staff and labour will work a Travel In and Out roster of 2 weeks on and 2 weeks off. Operation and some supervision will work on a day/night shift rotation.

The summary of total manning (average numbers) required for each of the mining stages is provided in Table 25-24. Note that this table shows the total number of personnel employed which is more than the numbers of personnel on site at any given time due to the roster arrangements.

Table 25-24 Total Manning by Mining Stage

Item	Mine Supervision	Technical Staff	Maintenance Supervision	Mine Operators	Maintenance Technicians	TOTAL
Mining Stage 1	7	7	8	35	21	78
Mining Stage 2	10	14	13	51	31	119
Mining Stage 3	7	10	8	23	14	62

Appendix 2 details the manning included in this element. Periodic use of consultants and expenses for project audits are also included in this element.

The on-costed rates applied for all personnel are consistent with the rosters and conditions in BC, Canada. An allowance for accommodation and travel has also been included in the on-costed rate.

Recruitment expenses for all mining personnel have been made. An employee turn-over rate of 20% per annum has been used. On-site accommodation and messing infrastructure will be provided for all permanent full time employees. Visitors and corporate travel are also included in this element.

25.8.6.2 Direct Operating Costs

The mining cost estimate is based on the premise that all mining equipment will be owned, operated and maintained by the mine owner.

Historical information generated over the many years of operation allows LAL to confidently estimate the overall life cycle owning and operating costs of mining equipment, something that is commonly misunderstood in the industry. However consideration also needs to be given to the local operating conditions at the mine site.

All equipment costs are in CAD and primarily based on 2nd half 2008 pricing with exchange rates of USD$0.85 and AUD$1.15 to CAD applied. No escalation of costs is estimated or included. Pricing of all costs in this study also excludes consideration for taxation.

The equipment selected for costing was based on the mine production forecast and methodology as well as general logistical and scheduling considerations specific for the Bronson Slope Project.

In addition, emphasis was placed on specifying equipment supplied by Caterpillar and Komatsu that met the scheduled production requirements. This was done based on the following considerations:

- Extensive in-house experience with this equipment in a broad range of environments. This would ensure more robust estimates to be made with a higher degree of accuracy.
- Critical equipment performance requirements, based on the mining methodology selected.
- The potential for alternative equipment solutions to provide better efficiencies in terms of overall cost per unit of output may well exist however would not greatly affect the outcome of this study.
- The required degree of project support for the Bronson Slope location is expected to be available from these OEMs.

The equipment fleet chosen and numbers required for mine production plan Years -2 to 20 are listed in Table 25-25.

Table 25-25 Key Mine Equipment List by Period

Description	Model	-1	Yr 1 to 8	Year 9 to 19
100t Excavator	Komatsu PC1250	1	2	1
Mobile Crusher	Nordberg LT140	1	2	1
High Angle Conveyor	ICSI	1	1	1
20T Rockbreaker	Cat 330	1	1	1
Grader 12’ blade	Cat 12M	1	1	1
Water Cart 20 kL	TBA	1	1	1
Loader 8.6m^3	Cat 990H	1	2	1
203mm DTH Drill	TBA	1	2	1
Track Dozer 15’ blade	Cat D8	1	2	1
90t Class RD Truck	Cat 777F	2	4	2
Ancillary Loader IT	Cat 966 IT	1	1	1
Roller		1	1	1

Equipment operating costs were prepared on the basis of Whole of Life (WOL) principles used by LAL. These costs include consideration for the following items consistent with the demands of the mine location and duty cycle:

- Fuel, lubricants and other fluids such as anti-freeze
- Repair and maintenance materials including for major components, minor parts and general consumables over the useful life of the equipment
- Tyre and track components consumed over the useful life
- Direct maintenance labour, including periodic OEM attendance, for all repairs, maintenance and servicing over the useful life of the equipment

- Ground engaging tools and down the hole drilling gear

The following expenses are excluded from the estimated unit operating costs, however are included in the overall mine operating cost estimate:

- Maintenance labour
- Operating labour
- Freight for components, parts and general maintenance materials (except for fuel and lubricants)
- Ancillary cost for labour such as transport, accommodation and messing
- Indirect maintenance support labour and staffing
- Training costs for direct maintenance labour
- Workshop and maintenance support facilities and equipment costs
- Spare parts inventory holding costs

Fuel is a key cost to the mine. The cost of diesel fuel has been estimated at CAD 1.00 per litre delivered to site based on estimated long term market conditions and actual costs reported from nearby minesites.

Cost elements for repair and maintenance were priced by reference to recent vendor indications for equipment specified for operations in similar demanding environments and tempered by Leighton's previous experience with this type of equipment. Consideration of the anticipated duty cycles for the project was also given.

The estimates for repair and maintenance were prepared on a WOL basis and include for provisioning for future repairs. VAT is specifically excluded from the pricing.

The estimate for equipment unit costs has been prepared using a combination of quoted, estimated and factored pricing to a level of accuracy in the order of ±30%.

Contingency for variances in the cost aspects of the entire project are included as a separate item in the overall project financial analysis.

Items included in the direct cost estimate are loading, crushing and conveying (both fixed and mobile sections inclusive of stackers), drill and blast, major support equipment and ROM and Waste rehandle. A summary of the direct mine operating costs has been included in Table 25-26. These costs have been calculated from first principles based on the material movement schedule selected for this project.

Table 25-26: Mine Operating Costs yr -2 to yr 9

Mine Operating Costs	Year-->	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019
	Period-->	-1	1	2	3	4	5	6	7	8	9
Drilling	(CAD,000)	2,207	3,614	3,614	3,614	3,614	3,614	3,614	3,614	3,614	3,614
	(CAD/t Mined)	0.381	0.312	0.312	0.298	0.298	0.298	0.298	0.298	0.285	0.318
	(CAD/t Milled)		0.756	0.709	0.709	0.709	0.709	0.709	0.709	0.709	0.709
Blasting	(CAD,000)	784	1,613	1,616	1,692	1,691	1,691	1,691	1,692	1,762	1,592
	(CAD/t Mined)	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14
	(CAD/t Milled)		0.34	0.32	0.33	0.33	0.33	0.33	0.33	0.35	0.31
Loading, Crushing &Conveying (mill feed and waste)	(CAD,000)	1,959	3,748	3,750	3,871	3,870	3,871	3,870	3,871	3,988	3,707
	(CAD/t Mined)	0.34	0.32	0.32	0.32	0.32	0.32	0.32	0.32	0.31	0.33
	(CAD/t Milled)		0.78	0.74	0.76	0.76	0.76	0.76	0.76	0.78	0.73
Rehandle Load&Haul	(CAD,000)	3,806	7,170	7,146	7,495	7,501	7,498	7,500	7,494	7,895	6,951
	(CAD/t Mined)	0.66	0.62	0.62	0.62	0.62	0.62	0.62	0.62	0.62	0.61
	(CAD/t Milled)		1.50	1.40	1.47	1.47	1.47	1.47	1.47	1.55	1.36
Ancillary - roads & dumps	(CAD,000)	3889	5709	5595	5904	5931	5918	5925	5900	6452	5206
	(CAD/t Mined)	0.67	0.49	0.48	0.49	0.49	0.49	0.49	0.49	0.51	0.46
	(CAD/t Milled)		1.19	1.10	1.16	1.16	1.16	1.16	1.16	1.27	1.02
Total Direct Costs	(CAD,000)	$13,403	$21,820	$21,686	$22,541	$22,574	$22,558	$22,567	$22,537	$23,678	$21,036
	(CAD/t Mined)	2.31	1.88	1.87	1.86	1.86	1.86	1.86	1.86	1.87	1.85
	(CAD/t Milled)		4.57	4.25	4.42	4.43	4.42	4.43	4.42	4.64	4.13

Table 25-27: Mine Operating Costs Yr 10 to 20

Mine Operating Costs	Period-->	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	Total
		10	11	12	13	14	15	16	17	18	19	kt
Drilling	(CAD,000)	946	951	852	852	851	852	852	932	944	342	$24,198
	(CAD/t Mined)	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	0.14	$0.146
	(CAD/t Milled)	0.19	0.19	0.17	0.17	0.17	0.17	0.17	0.18	0.19	0.17	$0.259
Blasting	(CAD,000)	2,373	2,176	2,019	2,018	2,018	2,019	2,019	2,146	2,171	809	$56,273
	(CAD/t Mined)	0.36	0.33	0.34	0.34	0.34	0.34	0.34	0.33	0.33	0.34	$0.339
	(CAD/t Milled)	0.47	0.43	0.40	0.40	0.40	0.40	0.40	0.42	0.43	0.40	$0.602
Loading, Crushing & Conveying	(CAD,000)	3,862	3,818	3,361	3,369	3,369	3,361	3,363	3,732	3,886	1,351	$103,929
	(CAD/t Mined)	0.58	0.58	0.57	0.57	0.57	0.57	0.57	0.58	0.59	0.57	$0.625
	(CAD/t Milled)	0.76	0.75	0.66	0.66	0.66	0.66	0.66	0.73	0.76	0.66	$1.112
Rehandle Load & Haul	(CAD,000)	1684	1438	1043	1084	1086	1043	1051	1375	1819	428	$68,479
	(CAD/t Mined)	0.25	0.22	0.18	0.18	0.18	0.18	0.18	0.21	0.27	0.18	$0.412
	(CAD/t Milled)	0.33	0.28	0.20	0.21	0.21	0.20	0.21	0.27	0.36	0.21	$0.733
Ancillary - Roads & Dumps	(CAD,000)	1,933	1,933	1,933	1,933	1,933	1,933	1,933	1,933	1,933	773	$53,359
	(CAD/t Mined)	0.29	0.29	0.33	0.33	0.33	0.33	0.33	0.30	0.29	0.33	$0.321
	(CAD/t Milled)											
Total Direct Costs	(CAD,000)	$10,798	$10,316	$9,209	$9,256	$9,258	$9,209	$9,219	$10,118	$10,753	$3,704	$292,836
	(CAD/t Mined)	1.63	1.56	1.56	1.57	1.57	1.56	1.56	1.56	1.62	1.56	$1.77
	(CAD/t Milled)	2.12	2.02	1.81	1.82	1.82	1.81	1.81	1.98	2.11	1.82	$3.14

Load, Crush and Convey - The load, crush and conveying costs include the excavator, crusher, rock breaker, mobile conveyor and fixed conveyor inclusive of a stacker.

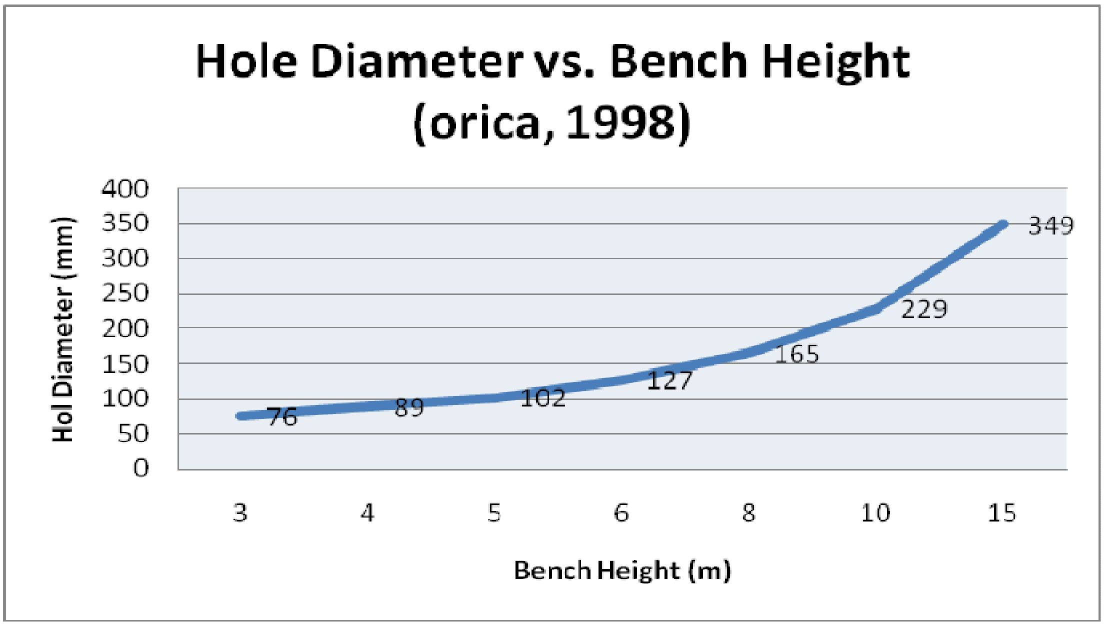
Drill and Blast - The drill and blast costs include:

- The cost for operating the drills
- Blasting fuel (ANFO and emulsion) and accessories
- Fixed costs associated with management and maintenance of a subcontracted explosives facility on site

The fixed costs mentioned above are charged at a fixed monthly rate.

The graph provided in Figure 25-25 demonstrates the theoretical relationship between bench height and blast hole size selection. The Bronson Slope bench height is 10m, and therefore a blast hole size of 203mm has been selected for this project.

Figure 25-25: Bench Height vs. Blasthole Size



The following table shows the drill and blast parameters selected for Bronson slope by material type.

Table 25-28: Drill and Blast Parameters

Blasting Criteria	Measure	Waste	Mill feed
Bench Height	Metres	10	10
Sub-drill	Metres	1.5	1.5
Hole Diameter	Millimetres	203	203
Burden	Metres	5.2	4.9
Spacing	Metres	6	5.6
Yield per Hole	Cu.m	313	273
Stem Height	Metres	4.06	4.06
Explosive Density	kg/m^3	0.85	0.85
Powder Factor - Theoretical	kg/BCM	0.654	0.751
Penetration Rate	Metres/hour	28	28

The drill patterns selected vary by material type. A tighter drill pattern (and higher powder factor) has been adopted for the mill feed material. The crush size for the mill feed material is 150mm whereas the waste only needs to be crushed to 250mm. The higher powder factor is considered to provide more effective fragmentation during the blasting process to improve the productivity of the crusher whilst crushing mill feed. A nominal allowance has also been made for wet holes, which will require emulsion based (water resistant) explosives.

Roads and Dumps – Support Equipment - Some equipment will be required to maintain roads and dumps and perform other ancillary type duties such as digging sumps, preparing and maintaining safety berms and other similar tasks. An allowance for the costs of this equipment has been made in the ancillary roads and dumps section of the mining cost build up. Allowance has been made for a grader, dozers, a water cart, an excavator with a rock breaker, a snow plough, a roller and use of the rehandle dump trucks. These items of equipment will be utilised only for the hours they are required.

25.8.7 LOM Project Cost Summary

A summary of the LOM Capital Costs has been included in Table 25-29. A summary of the site wide operating costs has been included in the following Table 25-30 and Table 25-31.

Table 25-29: Project Capital Cost Summary

Project Capital Cost Summary	Unit	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Processing Direct Costs	CAD 000's	$ 43,512	$ 43,512	$ 400	$400	$400	$400	$400	$ 400	$ 400	$400	$ 400
Infrastructure Directs	CAD 000's	$ 21,403	$ 21,403	$ 100	$100	$100	$100	$100	$ 100	$ 100	$100	$ 100
Mine Pre-production	CAD 000's	$ 878	$ 16,690	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Mining Equipment	CAD 000's	$ 8,960	$ 13,440	$ 9,408	$ -	$ -	$ -	$ -	$3,234	$6,071	$ -	$1,684
Reclamation	CAD 000's			$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Recovery	CAD 000's			$ -	$ -	$ -	$ -	$ -	$ (323)	$ (607)	$ -	$ (168)
Total Directs	**CAD 000's**	**$ 74,753**	**$ 95,044**	**$ 9,908**	**$500**	**$500**	**$500**	**$500**	**$3,411**	**$5,964**	**$500**	**$2,016**
Mill Indirects	CAD 000's	$ 7,832	$ 7,832	$ -								
Infrastructure indirects	CAD 000's	$ 6,159	$ 6,159	$ -								
Total Indirects	**CAD 000's**	**$ 13,991**	**$ 13,991**	**$ -**								
Directs + Indirects	CAD 000's	$ 88,744	$109,035	$ -								
Contingency ==> (15%)	CAD 000's	$ 13,312	$ 16,355	$ -								
Working Capital				$14,126								
LOM Project Capital Schedule	**CAD 000's**	**$102,055**	**$125,391**	**$24,034**	**$500**	**$500**	**$500**	**$500**	**$3,411**	**$5,964**	**$500**	**$2,016**

Project Capital Cost Summary	Unit	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Total
Processing Direct Costs	CAD 000's	$400	$ 400	$400	$400	$ 400	$400	$400	$400	$ 400	$ -	$ 94,223
Infrastructure Directs	CAD 000's	$100	$ 100	$100	$100	$ 100	$100	$100	$100	$ 100	$ -	$ 44,605
Mine Pre-production	CAD 000's	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 17,569
Mining Equipment	CAD 000's	$ -	$1,653	$ -	$ -	$1,540	$ -	$ -	$ -	$ -	$ -	$ 45,990
Reclamation	CAD 000's	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$5,000	$ 5,000	$ 10,000
Capital Recovery	CAD 000's	$ -	$ (165)	$ -	$ -	$ (154)	$ -	$ -	$ -	$ -	$(15,561)	$ (16,980)
Total Directs	**CAD 000's**	**$500**	**$1,988**	**$500**	**$500**	**$1,886**	**$500**	**$500**	**$500**	**$5,500**	**$(10,561)**	**$ 195,407**
Mill Indirects	CAD 000's											$ 15,664
Infrastructure indirects	CAD 000's											$ 12,318
Total Indirects	**CAD 000's**											
Directs + Indirects	CAD 000's											
Contingency ==> (15%)	CAD 000's											$ 29,667
Working Capital												$ 14,126
LOM Project Capital Schedule	**CAD 000's**	**$500**	**$1,988**	**$500**	**$500**	**$1,886**	**$500**	**$500**	**$500**	**$5,500**	**$(10,561)**	**$ 267,183**

Table 25-30: LOM Project Operating Cost Summary Yr-2 to Yr9

	Year→	-2	-1	1	2	3	4	5	6	7	8	9
Direct Mining Costs	(CAD,000)	$0	$13,403	$21,820	$21,686	$22,541	$22,574	$22,558	$22,567	$22,537	$23,678	$21,036
	(CAD/t Mined		2.31	1.88	1.87	1.86	1.86	1.86	1.86	1.86	1.87	1.85
	(CAD/t Milled)			4.57	4.25	4.42	4.43	4.42	4.43	4.42	4.64	4.13
Overheads and Administration	(CAD,000)	$1,422	$5,845	$9,651	$9,923	$9,923	$9,923	$9,923	$9,923	$3,614	$3,614	$3,614
	(CAD/t Mined			0.83	0.86	0.82	0.82	0.82	0.82	0.30	0.28	0.32
	(CAD/t Milled)			2.02	1.95	1.95	1.95	1.95	1.95	0.71	0.71	0.71
Processing Costs	(CAD,000)	$0	$0	$24,996	$26,663	$26,663	$26,663	$26,663	$26,663	$26,663	$26,663	$26,663
	(CAD/t Milled)			5.23	5.23	5.23	5.23	5.23	5.23	5.23	5.23	5.23
Total Costs	(CAD,000)	$1,422	$19,247	$56,467	$58,272	$59,127	$59,160	$59,144	$59,153	$52,814	$53,954	$51,312
	(CAD/t Milled)			11.81	11.43	11.60	11.60	11.60	11.60	10.36	10.58	10.07

Table 25-31: LOM Project Operating Cost Summary Yr 10 to 20

	Year→	10	11	12	13	14	15	16	17	18	19	Total/Avg
Direct Mining Costs	(CAD,000)	$10,798	$10,316	$9,209	$9,256	$9,258	$9,209	$9,219	$10,118	$10,753	$3,704	$292,836
	(CAD/t Mined	1.63	1.56	1.56	1.57	1.57	1.56	1.56	1.56	1.62	1.56	$1.77
	(CAD/t Milled)	2.12	2.02	1.81	1.82	1.82	1.81	1.81	1.98	2.11	1.82	$3.14
Overheads and Administration	(CAD,000)	$2,349	$2,349	$2,349	$2,349	$2,349	$2,226	$2,226	$2,226	$2,226	$890	$91,652
	(CAD/t Mined	0.35	0.35	0.40	0.40	0.40	0.38	0.38	0.34	0.34	0.38	$0.55
	(CAD/t Milled)	0.46	0.46	0.46	0.46	0.46	0.44	0.44	0.44	0.44	0.44	$0.98
Processing Costs	(CAD,000)	$26,663	$26,663	$26,663	$26,663	$26,663	$26,663	$26,663	$26,663	$26,663	$10,642	$488,901
	(CAD/t Milled)	5.23	5.23	5.23	5.23	5.23	5.23	5.23	5.23	5.23	5.23	$5.23
Total Costs	(CAD,000)	$39,810	$39,328	$38,221	$38,268	$38,270	$38,097	$38,107	$39,006	$39,642	$15,236	$873,388
	(CAD/t Milled)	7.81	7.71	7.50	7.51	7.51	7.47	7.47	7.65	7.78	7.49	$9.34

25.9 Economic Analysis

A cashflow model has been generated based on all revenue and expenses presented in this report. The cashflow is an annualised model based on the annual production schedule presented in this Item. Transport, smelting and refining charges have been levied based on the gross value of the metal in the concentrate each year. This is to ensure that the co-product nature of the concentrate is considered for the Bronson Slope project. The resulting net realised metal prices have also been presented in the cashflow model summary included in Table 25-32 to Table 25-34. A single figure containing the LOM cashflow summary has been provided in Appendix 3.

Table 25-32: Cash Flow Model Year -2 to 5

Bronson Slope Cashflow Model	Year		-2	-1	1	2	3	4	5
Metal Prices									
Copper	USD/lb				2.00	2.00	2.00	2.00	2.00
Gold	USD/oz				700	700	700	700	700
Silver	USD/oz				15	15	15	15	15
Exchange Rate	CAD:US				0.85	0.85	0.85	0.85	0.85
Production Data									
Tonnes Cu/Au Ore Mined	kt				4,779	5,098	5,098	5,098	5,098
Tonnes Waste Mined	kt				6,836	6,611	7,010	7,064	7,038
Total Ore and Waste Mined	kt				11,616	11,709	12,108	12,162	12,136
Strip Ratio (waste t : ore t)					1.43	1.30	1.37	1.39	1.38
Ore Milled	kt				4,779	5,098	5,098	5,098	5,098
Grade - Copper	%				0.210	0.217	0.164	0.086	0.153
Grade - Gold	g/t				0.525	0.567	0.570	0.564	0.539
Grade - Silver	g/t				2.46	2.65	2.77	3.02	2.77
Concentrate Data									
Copper Recovery	%				86.60	86.60	86.60	86.60	86.60
Gold Recovery	%				85.40	85.40	85.40	85.40	85.40
Silver Recovery	%				63.70	63.70	63.70	63.70	63.70
Copper Recovered to Concentrate	million lbs				19.2	21.1	15.9	8.4	14.9
Gold Recovered to Concentrate	koz				68.8	79.4	79.7	79.0	75.4
Silver Recovered to Concentrate	koz				240.5	276.8	288.9	315.6	289.4
Concentrate Production									
tonnes (dry)	kdmt				34.5	38.0	28.7	15.1	26.8
Copper Grade (dry)	%				25.20	25.20	25.20	25.20	25.20
Gold Grade	g/dmt				62.0	65.0	86.4	162.3	87.6
Silver Grade	g/dmt				216.8	226.5	313.2	648.5	336.3
Total Contained Gold Metal					80.6	93.0	93.4	92.5	88.3
TC/RC Terms					95.3	95.8	100.1	115.3	100.5
Concentrate Transportation	USD/dmt				129.0	129.4	132.3	142.5	132.5
Net Realized Prices									
Copper	USD/lb				1.72	1.73	1.75	1.79	1.75
Gold	USD/oz				629	630	637	651	638
Silver	USD/oz				12.23	12.25	12.39	12.66	12.40
CASH FLOW									
Net Revenue									
Copper - 28.2%	CAD 000's				38,890	42,930	32,775	17,664	30,594
Gold - 43.3%	CAD 000's				50,953	58,882	59,800	60,479	56,601
Silver - 3.2%	CAD 000's				3,459	3,989	4,212	4,699	4,221
Net Revenue					**93,302**	**105,801**	**96,787**	**82,842**	**91,416**
NSR / t Milled	CAD / t Milled				19.52	20.75	18.99	16.25	17.93
Operating Costs	CAD								
Mining	3.14	per t milled			21,872	21,913	22,482	22,616	22,551
Milling	5.23	per t milled			24,982	26,647	26,647	26,647	26,647
G&A	0.98	per t milled			9,651	9,923	9,923	9,923	9,923
Total Direct Operating Costs	**9.35**	**per t milled**			**56,505**	**58,484**	**59,052**	**59,187**	**59,121**
Direct Operating Costs per t Ore Milled					11.82	11.47	11.58	11.61	11.60
Operating Pretax Cash Flow					36,797	47,317	37,735	23,656	32,295
Capex			102,055	125,391	24,034	500	500	500	500
Pre-tax Cash Flow			**(102,055)**	**(125,391)**	**12,763**	**46,817**	**37,235**	**23,156**	**31,795**
	Cumulative		(102,055)	(227,446)	(214,683)	(167,865)	(130,630)	(107,475)	(75,680)
Taxation									
	Federal Income		-	-	-	-	-	-	-
	Provincial Income		-	-	-	-	-	-	-
	Mining Tax		2,041	2,508	(255)	(936)	(745)	(463)	(636)
	Total Taxes		**2,041**	**2,508**	**(255)**	**(936)**	**(745)**	**(463)**	**(636)**
Net After Tax Cash Flow			**(100,014)**	**(122,883)**	**12,508**	**45,881**	**36,490**	**22,693**	**31,159**

Table 25-33: Cash Flow Model Year 6 to 13

Bronson Slope Cashflow Model	Year	6	7	8	9	10	11	12	13
Metal Prices									
Copper	USD/lb	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00
Gold	USD/oz	700	700	700	700	700	700	700	700
Silver	USD/oz	15	15	15	15	15	15	15	15
Exchange Rate	CAD:US	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
Production Data									
Tonnes Cu/Au Ore Mined	kt	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098
Tonnes Waste Mined	kt	7,053	7,003	7,881	5,926	1,726	1,319	779	846
Total Ore and Waste Mined	kt	12,151	12,101	12,979	11,024	6,824	6,417	5,877	5,944
Strip Ratio (waste t : ore t)		1.38	1.37	1.55	1.16	0.34	0.26	0.15	0.17
Ore Milled	kt	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098
Grade - Copper	%	0.174	0.170	0.162	0.166	0.163	0.167	0.153	0.147
Grade - Gold	g/t	0.517	0.491	0.456	0.366	0.402	0.422	0.409	0.401
Grade - Silver	g/t	2.57	2.45	2.37	2.05	2.46	2.35	2.09	1.91
Concentrate Data									
Copper Recovery	%	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60
Gold Recovery	%	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40
Silver Recovery	%	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70
Copper Recovered to Concentrate	million lbs	17.0	16.6	15.8	16.1	15.9	16.2	14.9	14.4
Gold Recovered to Concentrate	koz	72.3	68.7	63.8	51.3	56.2	59.0	57.2	56.1
Silver Recovered to Concentrate	koz	267.9	255.5	247.9	214.5	256.5	245.5	218.4	199.1
Concentrate Production									
tonnes (dry)	kdmt	30.5	29.8	28.4	29.0	28.6	29.2	26.8	25.8
Copper Grade (dry)	%	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20
Gold Grade	g/dmt	73.7	71.7	69.8	54.9	61.1	62.8	66.5	67.5
Silver Grade	g/dmt	273.0	266.6	271.4	229.7	278.5	261.4	253.6	239.7
Total Contained Gold Metal		84.7	80.5	74.7	60.1	65.8	69.1	67.0	65.7
TC/RC Terms		97.7	97.3	97.0	94.3	95.7	95.8	96.3	96.4
Concentrate Transportation	USD/dmt	130.6	130.4	130.1	128.2	129.1	129.2	129.7	129.8
Net Realized Prices									
Copper	USD/lb	1.74	1.73	1.73	1.72	1.72	1.73	1.73	1.73
Gold	USD/oz	634	633	632	626	629	630	631	631
Silver	USD/oz	12.32	12.30	12.29	12.17	12.23	12.24	12.26	12.27
CASH FLOW									
Net Revenue									
Copper - 28.2%	CAD 000's	34,642	33,797	32,191	32,587	32,282	32,952	30,278	29,217
Gold - 43.3%	CAD 000's	53,929	51,196	47,464	37,789	41,632	43,720	42,501	41,656
Silver - 3.2%	CAD 000's	3,881	3,698	3,585	3,071	3,690	3,535	3,151	2,874
Net Revenue		**92,452**	**88,691**	**83,239**	**73,448**	**77,605**	**80,208**	**75,930**	**73,747**
NSR / t Milled	CAD / t Milled	18.13	17.40	16.33	14.41	15.22	15.73	14.89	14.47
Operating Costs	CAD								
Mining	3.14 per t milled	22,587	22,465	24,215	20,394	11,127	9,997	9,166	9,318
Milling	5.23 per t milled	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647
G&A	0.98 per t milled	9,923	3,614	3,614	3,614	2,349	2,349	2,349	2,349
Total Direct Operating Costs	**9.35 per t milled**	**59,158**	**52,726**	**54,476**	**50,655**	**40,124**	**38,994**	**38,163**	**38,315**
Direct Operating Costs per t Ore Milled		11.60	10.34	10.69	9.94	7.87	7.65	7.49	7.52
Operating Pretax Cash Flow		33,294	35,965	28,764	22,793	37,481	41,214	37,768	35,432
Capex		3,411	5,964	500	2,016	500	1,988	500	500
Pre-tax Cash Flow		**29,883**	**30,001**	**28,264**	**20,777**	**36,981**	**39,226**	**37,268**	**34,932**
	Cumulative	(45,796)	(15,796)	12,468	33,245	70,226	######	146,720	181,652
Taxation									
	Federal Income	-	-	-	-	(631)	(4,763)	(4,651)	(4,440)
	Provincial Income	-	-	-	-	(463)	(3,493)	(3,411)	(3,256)
	Mining Tax	(598)	(600)	(565)	(416)	(740)	(785)	(745)	(699)
	Total Taxes	**(598)**	**(600)**	**(565)**	**(416)**	**(1,833)**	**(9,041)**	**(8,807)**	**(8,394)**
Net After Tax Cash Flow		**29,286**	**29,401**	**27,698**	**20,362**	**35,147**	**30,186**	**28,460**	**26,538**

Table 25-34: Cash Flow Model Year 14 to 19

Bronson Slope Cashflow Model		Year	14	15	16	17	18	19	TOTAL
Metal Prices									
Copper	USD/lb		2.00	2.00	2.00	2.00	2.00	2.00	2.00
Gold	USD/oz		700	700	700	700	700	700	700
Silver	USD/oz		15	15	15	15	15	15	15
Exchange Rate	CAD:US		0.85	0.85	0.85	0.85	0.85	0.85	0.85
Production Data									
Tonnes Cu/Au Ore Mined	kt		5,098	5,098	5,098	5,098	5,098	2,035	93,480
Tonnes Waste Mined	kt		850	778	793	1,236	1,951	328	73,026
Total Ore and Waste Mined	kt		5,948	5,876	5,891	6,334	7,049	2,362	166,506
Strip Ratio (waste t : ore t)			0.17	0.15	0.16	0.24	0.38	0.16	0.78
Ore Milled	kt		5,098	5,098	5,098	5,098	5,098	2,035	93,480
Grade - Copper	%		0.144	0.140	0.136	0.130	0.135	0.154	0.156
Grade - Gold	g/t		0.399	0.382	0.378	0.370	0.359	0.360	0.449
Grade - Silver	g/t		2.06	2.12	2.13	2.14	2.17	2.03	2.356
Concentrate Data									
Copper Recovery	%		86.60	86.60	86.60	86.60	86.60	86.60	86.60
Gold Recovery	%		85.40	85.40	85.40	85.40	85.40	85.40	85.40
Silver Recovery	%		63.70	63.70	63.70	63.70	63.70	63.70	63.70
Copper Recovered to Concentrate	million lbs		14.0	13.6	13.2	12.6	13.2	6.0	279
Gold Recovered to Concentrate	koz		55.9	53.5	52.9	51.8	50.2	20.1	1,151
Silver Recovered to Concentrate	koz		215.1	221.4	222.3	223.6	226.4	84.8	4,510
Concentrate Production									
tonnes (dry)	kdmt		25.3	24.5	23.8	22.7	23.7	10.8	502.1
Copper Grade (dry)	%		25.20	25.20	25.20	25.20	25.20	25.20	25.2
Gold Grade	g/dmt		68.7	67.9	69.1	70.8	65.9	58.2	71.3
Silver Grade	g/dmt		264.7	281.3	290.6	305.8	297.0	245.3	279.4
Total Contained Gold Metal			65.4	62.6	61.9	60.7	58.8	23.6	72.5
TC/RC Terms			96.8	96.8	97.1	97.5	96.7	94.9	97.4
Concentrate Transportation	USD/dmt		130.0	129.9	130.1	130.3	129.7	128.6	130.3
Net Realized Prices									
Copper	USD/lb		1.73	1.73	1.73	1.73	1.73	1.72	1.73
Gold	USD/oz		632	632	632	633	631	628	632.29
Silver	USD/oz		12.28	12.28	12.29	12.30	12.27	12.20	12.29
CASH FLOW									CAD 000's
Net Revenue									
Copper - 28.2%	CAD 000's		28,623	27,704	26,944	25,793	26,807	12,091	568,762
Gold - 43.3%	CAD 000's		41,539	39,748	39,323	38,575	37,290	14,860	857,936
Silver - 3.2%	CAD 000's		3,109	3,198	3,213	3,236	3,267	1,217	65,306
Net Revenue			**73,270**	**70,649**	**69,480**	**67,604**	**67,364**	**28,168**	**1,492,004**
NSR / t Milled	CAD / t Milled		14.37	13.86	13.63	13.26	13.21	13.84	16.22
Operating Costs	CAD								
Mining	3.14	per t milled	9,327	9,165	9,198	9,884	11,447	3,691	293,414
Milling	5.23	per t milled	26,647	26,647	26,647	26,647	26,647	10,636	488,621
G&A	0.98	per t milled	2,349	2,226	2,226	2,226	2,226	890	91,652
Total Direct Operating Costs	**9.35**	**per t milled**	**38,324**	**38,039**	**38,071**	**38,757**	**40,320**	**15,217**	**873,686**
Direct Operating Costs per t Ore Milled			7.52	7.46	7.47	7.60	7.91	7.48	9.35
Operating Pretax Cash Flow			34,946	32,611	31,409	28,847	27,044	12,951	618,317
Capex			1,886	500	500	500	5,500	(10,561)	267,183
Pre-tax Cash Flow			**33,060**	**32,111**	**30,909**	**28,347**	**21,544**	**23,512**	**351,135**
	Cumulative		214,712	246,823	277,732	306,079	327,623	351,135	
Taxation									
	Federal Income		(4,263)	(4,235)	(4,180)	(3,894)	(2,962)	(3,303)	(37,322)
	Provincial Income		(3,126)	(3,106)	(3,065)	(2,856)	(2,172)	(2,422)	(27,370)
	Mining Tax		(661)	(642)	(618)	(567)	(431)	(470)	(7,023)
	Total Taxes		**(8,051)**	**(7,983)**	**(7,863)**	**(7,317)**	**(5,566)**	**(6,195)**	(71,715)
									-
Net After Tax Cash Flow			**25,009**	**24,127**	**23,046**	**21,030**	**15,978**	**17,317**	279,420

A summary of the financial performance of the project can be seen in Table 25-35 below.

Table 25-35: Financial Performance Measure Summary

Financial Performance Summary				
			Gross	**Net Tax**
		Project IRR	**11.0%**	**10.0%**
	Disc Rate	**Unit**		
Project NPV-->	10.0%	million CAD	14.0	0.0
	7.5%	**million CAD**	**59.3**	**38.3**
	5.0%	million CAD	123.5	92.0
	0.0%	million CAD	351.1	279.4
Payback		Years	8.2	
MineLife		Years	18.4	

The pro-rata cash costs and net gold cash cost after co-product and by-product credits have been calculated within the financial model and have been provided in Table 25-36.

Table 25-36: Project Cash Cost Summary

	Gold (USD/t.oz)	Copper (USD/lb)	Silver (USD/t.oz)
Pro Rata Cash Cost	$ 428.46	$ 1.24	$ 9.28
Net Cash Cost after credits	$ 231.86		

A sensitivity analysis has also been completed on the pre tax cashflow to demonstrate the sensitivity of the financial performance of the project to major inputs such as metal prices, capital costs, operating costs, exchange rate and mill feed grade. The results of this analysis are presented in Table 25-37to Table 25-40 and Figure 25-26 and Figure 25-27.

It can be seen that NPV and IRR are most sensitive to the USD:CAD exchange rate, followed by gold price, gold grade, operating costs, copper price, copper grade and finally least sensitive to changes in the Initial capital costs.

Table 25-37: Gross NPV Sensitivty

NPV Value Gross Sensitivity					
Variant	**-20%**	**-10%**	**0%**	**+10%**	**+20%**
Cu Price	0.53	29.90		88.64	118.01
Au Price	-30.29	14.49		104.05	148.83
US/C Exch	242.92	140.89		-7.51	-63.16
OpCost	147.43	103.35	59.27	15.19	-28.89
Initial Capex	99.96	79.61		38.93	18.58
Cu Grade	10.77	35.02		83.52	107.77
Au Grade	-29.65	14.81		103.73	148.19

Table 25-38: Gross IRR Sensitivity

IRR Value Gross Sensitivity					
Variant	-20%	-10%	0%	+10%	+20%
Cu Price	7.5%	9.3%	11.0%	12.6%	14.2%
Au Price	5.6%	8.4%		13.5%	15.9%
US/C Exch	20.7%	15.4%		7.0%	3.4%
OpCost	15.9%	13.5%		8.4%	5.7%
Initial Capex	14.4%	12.6%		9.6%	8.5%
Cu Grade	8.2%	9.6%		12.3%	13.6%
Au Grade	5.7%	8.4%		13.5%	15.8%

Table 25-39: Net NPV Sensitivity

NPV Value Net Sensitivity					
Variant	-20%	-10%	0%	+10%	+20%
Cu Price	-8.89	14.92	38.26	60.86	82.44
Au Price	-34.57	2.27		72.30	104.96
US/C Exch	170.84	98.97		-15.61	-62.96
OpCost	104.19	71.92		2.70	-33.75
Initial Capex	77.94	58.20		18.32	-1.61
Cu Grade	-0.55	18.95		56.91	74.85
Au Grade	-34.01	2.52		72.06	104.51

Table 25-40: Net IRR Sensitivity

IRR Value Net Sensitivity					
Variant	-20%	-10%	0%	+10%	+20%
Cu Price	6.9%	8.5%	10.0%	11.5%	12.9%
Au Price	5.2%	7.6%		12.2%	14.4%
US/C Exch	18.6%	13.9%		6.5%	3.3%
OpCost	14.4%	12.2%		7.7%	5.3%
Initial Capex	13.5%	11.6%		8.6%	7.4%
Cu Grade	7.5%	8.7%		11.2%	12.4%
Au Grade	5.3%	7.7%		12.2%	14.3%

Net Tax NPV Sensitivity Analysis

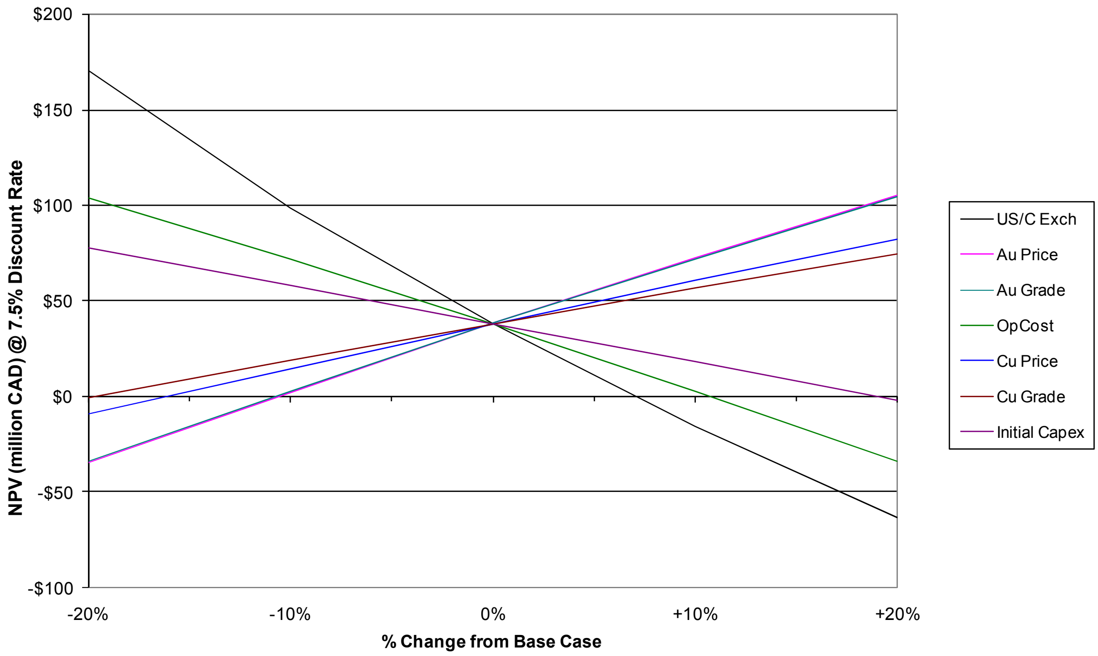


Figure 25-26: Net NPV Sensitivty Graph

Net Tax IRR Sensitivity Analysis

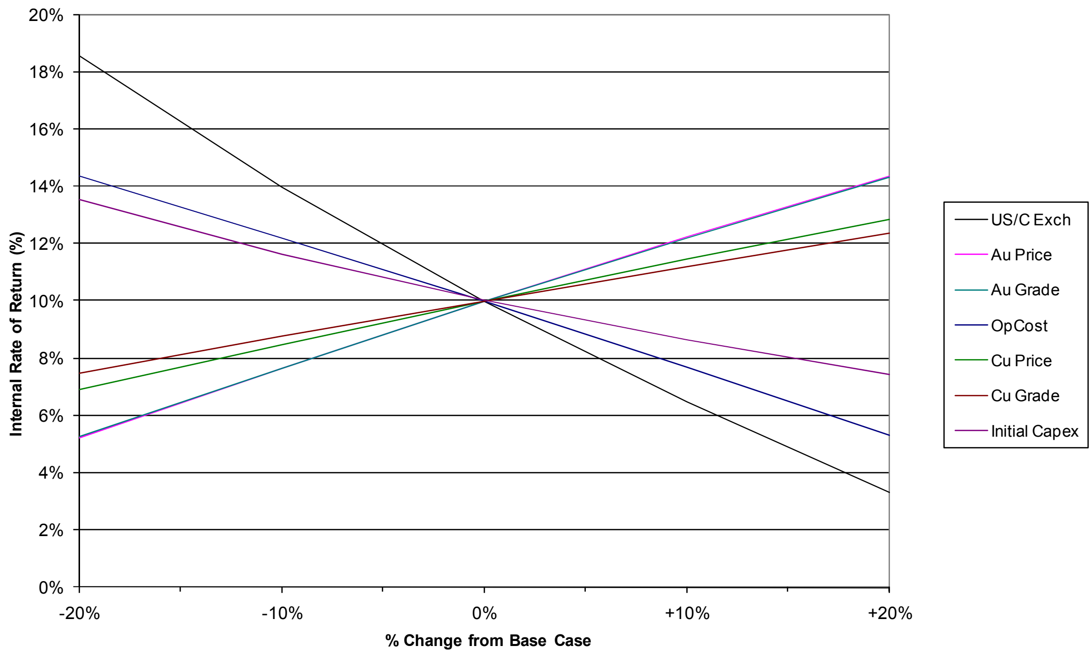


Figure 25-27: Net IRR Sensitivity Analysis

25.9.1 Payback

Based on the cash flow model provided payback occurs in the first half of year 8. It may be possible to reduce the payback period by utilising a smaller starter and final pit (reducing the total resource size). This will eliminate some of the need for waste stripping up front and will help push more of the mining costs towards the end of the project. Further review is required to fully understand what impact this pit shell and schedule adjustment will have on payback life.

25.9.2 Mine Life

Based on the LOM schedule and the optimum pits selected for this project the mine life is approximately 18.5 years. This schedule is based on maximising the NPV, which has the affect of maximising the mine life. A secondary optimisation has been completed targeting the highest IRR for the project. This scenario results in the most optimum final pit providing a mine life closer to 10 years. Further study is required to identify what is the preferred case for the Bronson Slope project.

26 Illustrations

A number of Illustrations have been provided throughout the report. Please refer to the appropriate section to view these illustrations.

In Particular please refer to:

- Figure 6-1 and Figure 6-2 for maps showing the location
- Figure 7-1 for the site layout plan
- *Figure 25-8* to *Figure 25-19* for pit plans and sections

APPENDIX 1

Detailed Bench Tonnes and Grades

Total Pit Reserve - Based on Measured & Indicated

Pit	RL	SG-W	SG-O		Waste (tonnes)	ORE					Total Tonnes	BCM		Totals BCM
						Tonnes	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)		Waste	Ore	
Starter Pit	730	2.76	2.76		157,549	10,182	0.02	0.50	1.12	0.00	167,731	57,083	3,689	60,772
Starter Pit	720	2.76	2.76		221,758	41,022	0.02	0.54	1.20	0.00	262,780	80,347	14,863	95,210
Starter Pit	710	2.76	2.76		258,808	43,644	0.02	0.55	1.19	0.00	302,452	93,771	15,813	109,584
Starter Pit	700	2.76	2.76		358,386	60,615	0.03	0.56	1.24	0.00	419,001	129,850	21,962	151,812
Starter Pit	690	2.76	2.76		365,628	55,586	0.03	0.54	1.28	0.00	421,215	132,474	20,140	152,614
Starter Pit	680	2.76	2.76		446,698	60,576	0.03	0.49	1.28	0.00	507,274	161,847	21,948	183,795
Starter Pit	670	2.76	2.76		542,343	38,640	0.03	0.47	1.30	0.00	580,983	196,501	14,000	210,501
Starter Pit	660	2.76	2.76		522,462	11,040	0.04	0.45	1.34	0.00	533,502	189,298	4,000	193,298
Starter Pit	650	2.76	2.76		589,564	8,280	0.04	0.43	1.34	0.00	597,844	213,610	3,000	216,610
Starter Pit	640	2.76			558,185	0					558,185	202,241	0	202,241
Starter Pit	630	2.76			639,362	0					639,362	231,653	0	231,653
Starter Pit	620	2.76			612,375	0					612,375	221,875	0	221,875
Starter Pit	610	2.76			724,061	0					724,061	262,341	0	262,341
Starter Pit	600	2.76	2.75		709,535	187	0.07	0.15	2.59	0.02	709,722	257,078	68	257,146
Starter Pit	590	2.76	2.75		807,339	78,799	0.15	0.35	2.44	0.01	886,138	292,519	28,692	321,211
Starter Pit	580	2.76	2.75		702,644	365,566	0.18	0.44	2.53	0.01	1,068,210	254,581	133,077	387,658
Starter Pit	570	2.76	2.75		685,625	622,050	0.20	0.48	2.44	0.01	1,307,675	248,420	226,272	474,692
Starter Pit	560	2.76	2.75		516,780	856,791	0.21	0.52	2.39	0.01	1,373,571	187,239	311,498	498,737
Starter Pit	550	2.76	2.75		442,713	1,097,521	0.22	0.55	2.42	0.01	1,540,235	160,404	398,934	559,338
Starter Pit	540	2.76	2.75		247,824	1,347,045	0.22	0.56	2.50	0.01	1,594,868	89,790	489,577	579,367
Starter Pit	530	2.76	2.75		158,441	1,632,504	0.22	0.55	2.55	0.01	1,790,945	57,412	593,249	650,661
Starter Pit	520	2.76	2.75		33,873	1,772,160	0.23	0.57	2.67	0.01	1,806,033	12,270	643,743	656,013
Starter Pit	510	2.75	2.75		51,916	1,931,559	0.22	0.57	2.73	0.01	1,983,475	18,847	701,259	720,106
Starter Pit	500	2.75	2.76		56,670	1,914,155	0.22	0.58	2.78	0.00	1,970,825	20,609	694,142	714,751
Starter Pit	490	2.75	2.76		83,489	2,068,482	0.21	0.57	2.73	0.00	2,151,971	30,348	749,548	779,896
Starter Pit	480	2.75	2.76		140,686	1,941,916	0.20	0.58	2.62	0.00	2,082,602	51,090	702,714	753,804
Starter Pit	470	2.76	2.76		54,303	2,185,801	0.19	0.55	2.59	0.00	2,240,104	19,672	790,869	810,541
Starter Pit	460	2.76	2.76		98,877	2,051,802	0.19	0.53	2.56	0.00	2,150,679	35,790	742,119	777,909
Starter Pit	450	2.76	2.76		144,092	2,149,214	0.18	0.52	2.56	0.00	2,293,306	52,127	777,402	829,529
Starter Pit	440	2.77	2.77		176,740	1,981,442	0.18	0.51	2.53	0.00	2,158,182	63,917	716,462	780,379
Starter Pit	430	2.76	2.77		189,484	2,068,999	0.17	0.50	2.46	0.00	2,258,483	68,542	748,094	816,636
Starter Pit	420	2.77	2.77		220,183	1,426,648	0.17	0.50	2.49	0.00	1,646,831	79,553	515,168	594,721
Starter Pit	410	2.77	2.77		104,739	1,592,938	0.17	0.47	2.43	0.00	1,697,677	37,812	575,343	613,155
Starter Pit	400	2.77	2.77		204,963	1,219,967	0.17	0.48	2.44	0.00	1,424,931	74,042	440,689	514,731
Starter Pit	390	2.77	2.77		233,725	1,237,419	0.16	0.47	2.40	0.00	1,471,144	84,408	447,136	531,544
Starter Pit	380	2.77	2.77		220,247	1,138,082	0.16	0.48	2.40	0.00	1,358,329	79,525	411,299	490,824
Final Pit	850	2.76			20,366	0					20,366	7,379	0	7,379
Final Pit	840	2.76			71,401	0					71,401	25,870	0	25,870
Final Pit	830	2.76			224,937	0					224,937	81,499	0	81,499
Final Pit	820	2.76			315,446	0					315,446	114,292	0	114,292
Final Pit	810	2.76			509,979	0					509,979	184,775	0	184,775
Final Pit	800	2.76			601,305	0					601,305	217,864	0	217,864
Final Pit	790	2.76			877,059	0					877,059	317,775	0	317,775
Final Pit	780	2.76			1,023,019	0					1,023,019	370,659	0	370,659
Final Pit	770	2.76	2.76		1,375,187	1,140	0.02	0.48	1.44	0.00	1,376,326	498,256	413	498,669
Final Pit	760	2.76	2.76		1,458,006	15,053	0.02	0.50	2.00	0.00	1,473,059	528,263	5,454	533,717
Final Pit	750	2.76	2.76		1,686,851	79,096	0.02	0.58	2.40	0.00	1,765,947	611,178	28,658	639,836
Final Pit	740	2.76	2.76		1,648,838	152,763	0.02	0.59	2.39	0.00	1,801,601	597,405	55,349	652,754
Final Pit	730	2.76	2.76		1,766,254	209,271	0.02	0.59	2.37	0.00	1,975,525	639,947	75,823	715,770
Final Pit	720	2.76	2.76		1,583,975	251,041	0.02	0.55	2.59	0.00	1,835,016	573,904	90,957	664,861
Final Pit	710	2.76	2.76		1,601,222	439,919	0.02	0.54	2.87	0.00	2,041,141	580,153	159,391	739,544
Final Pit	700	2.76	2.76		1,368,113	525,211	0.02	0.55	3.31	0.00	1,893,324	495,693	190,294	685,987
Final Pit	690	2.76	2.76		1,447,794	628,090	0.02	0.56	3.61	0.00	2,075,884	524,563	227,569	752,132
Final Pit	680	2.76	2.76		1,247,335	673,305	0.02	0.57	3.48	0.00	1,920,640	451,933	243,951	695,884
Final Pit	670	2.76	2.76		1,240,253	757,551	0.02	0.56	3.07	0.00	1,997,804	449,367	274,475	723,842
Final Pit	660	2.76	2.76		1,286,615	651,233	0.03	0.55	2.86	0.00	1,937,848	466,165	235,954	702,119
Final Pit	650	2.76	2.76		1,472,598	554,760	0.03	0.54	3.07	0.00	2,027,358	533,550	201,000	734,550
Final Pit	640	2.76	2.76		1,570,186	391,920	0.03	0.53	3.18	0.00	1,962,106	568,908	142,000	710,908
Final Pit	630	2.76	2.76		1,712,045	322,031	0.03	0.52	3.51	0.00	2,034,076	620,306	116,678	736,984
Final Pit	620	2.76	2.76		1,771,467	191,936	0.03	0.50	3.77	0.00	1,963,403	641,836	69,542	711,378
Final Pit	610	2.76	2.76		1,883,642	180,518	0.03	0.48	3.98	0.00	2,064,160	682,479	65,405	747,884
Final Pit	600	2.76	2.76		1,945,687	45,383	0.03	0.48	2.92	0.00	1,991,070	704,959	16,443	721,402
Final Pit	590	2.76	2.76		2,034,131	41,345	0.02	0.50	1.30	0.00	2,075,476	737,004	14,980	751,984
Final Pit	580	2.76	2.76		2,003,766	20,559	0.02	0.54	1.17	0.00	2,024,325	726,002	7,449	733,451
Final Pit	570	2.76	2.76		2,077,090	18,288	0.02	0.55	1.16	0.00	2,095,378	752,569	6,626	759,195
Final Pit	560	2.76	2.76		2,006,031	24,801	0.02	0.49	1.50	0.00	2,030,833	726,823	8,986	735,809
Final Pit	550	2.76	2.76		2,092,041	19,320	0.02	0.49	1.72	0.00	2,111,361	757,986	7,000	764,986
Final Pit	540	2.76	2.76		2,022,197	11,040	0.02	0.51	1.56	0.00	2,033,237	732,680	4,000	736,680
Final Pit	530	2.76	2.75		2,089,190	24,790	0.10	0.48	2.00	0.01	2,113,980	756,954	9,000	765,954
Final Pit	520	2.76	2.74		1,924,856	163,251	0.17	0.37	1.81	0.01	2,088,107	697,438	59,493	756,931
Final Pit	510	2.76	2.74		1,848,958	283,946	0.17	0.37	1.76	0.01	2,132,904	669,960	103,535	773,495
Final Pit	500	2.76	2.74		1,508,261	624,683	0.17	0.36	1.73	0.01	2,132,944	546,618	227,731	774,349
Final Pit	490	2.76	2.74		1,418,162	816,615	0.17	0.35	1.76	0.01	2,234,776	514,162	297,662	811,824
Final Pit	480	2.76	2.74		1,092,451	1,212,643	0.17	0.37	2.11	0.01	2,305,094	396,256	442,034	838,290
Final Pit	470	2.76	2.74		1,029,916	1,420,991	0.16	0.36	2.18	0.01	2,450,907	373,557	517,806	891,363
Final Pit	460	2.76	2.75		747,149	1,749,642	0.16	0.38	2.22	0.01	2,496,791	270,892	636,722	907,614
Final Pit	450	2.76	2.75		756,653	1,876,872	0.16	0.39	2.38	0.01	2,633,525	274,267	682,496	956,763
Final Pit	440	2.76	2.75		569,878	2,032,332	0.17	0.41	2.53	0.01	2,602,210	206,585	738,511	945,096
Final Pit	430	2.76	2.75		681,019	2,089,158	0.16	0.42	2.58	0.01	2,770,177	246,928	759,032	1,005,960
Final Pit	420	2.76	2.75		600,241	2,494,716	0.17	0.43	2.33	0.01	3,094,958	217,649	905,646	1,123,295
Final Pit	410	2.76	2.75		617,321	2,697,950	0.16	0.41	2.29	0.01	3,315,271	224,029	979,340	1,203,369
Final Pit	400	2.76	2.76		411,156	3,037,297	0.15	0.41	2.10	0.01	3,448,454	149,117	1,102,008	1,251,125
Final Pit	390	2.76	2.76		482,032	3,212,237	0.15	0.40	1.98	0.01	3,694,269	174,711	1,165,498	1,340,209
Final Pit	380	2.76	2.76		536,976	3,021,658	0.15	0.40	1.85	0.01	3,558,634	194,525	1,096,105	1,290,630
Final Pit	370	2.76	2.76		757,724	4,409,808	0.15	0.40	2.05	0.01	5,167,531	274,172	1,597,978	1,872,150
Final Pit	360	2.77	2.76		627,674	4,132,278	0.14	0.39	2.09	0.01	4,759,952	226,973	1,496,998	1,723,971
Final Pit	350	2.77	2.76		623,970	4,088,774	0.14	0.37	2.17	0.01	4,712,745	225,520	1,481,074	1,706,594
Final Pit	340	2.77	2.76		469,517	2,979,130	0.13	0.38	2.10	0.00	3,448,647	169,593	1,078,169	1,247,762
Final Pit	330	2.77	2.76		543,274	2,849,346	0.13	0.37	2.13	0.00	3,392,621	196,382	1,031,245	1,227,627
Final Pit	320	2.76	2.76		665,796	2,272,598	0.13	0.37	2.16	0.00	2,938,394	240,868	822,170	1,063,038
Final Pit	310	2.76	2.76		803,421	2,069,029	0.13	0.36	2.16	0.00	2,872,450	290,743	748,478	1,039,221
Final Pit	300	2.76	2.76		699,050	1,595,270	0.14	0.36	2.17	0.00	2,294,320	253,000	577,000	830,000
Final Pit	290	2.76	2.76		654,750	1,481,710	0.14	0.35	2.19	0.00	2,136,460	237,000	536,000	773,000
Final Pit	280	2.77	2.76		83,060	857,010	0.15	0.36	2.14	0.00	940,070	30,000	310,000	340,000
Final Pit	270	2.77	2.76		55,350	759,970	0.16	0.36	1.83	0.00	815,320	20,000	275,000	295,000
Summary Information					**78,524,707**	**93,469,914**	**0.156**	**0.449**	**2.35**	**0.0059**	**171,994,621**	**28,448,827**	**33,881,897**	**62,330,724**

APPENDIX 2

Detailed Life of Mine Manning Requirements

Leighton Asia

SGC002 Mining Plan and Cost Estimate – Bronson Slope Project

Skyline Gold Corporation - Bronson Slope Project
Preliminary Economic Assessment
Excavators, Crusher and Conveyors
MINE OPERATION MANNING REQUIREMENTS

Date: 3-Feb-09
By: JAL
Ore Processing Production: 15,000 t/day
Production Period: 12 mths

	Period-->	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029
		-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
MINE SUPERVISION																						
Mining Manager	(#)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Superintendent	(#)	-	-	1	1	1	1	1	1	1	1	1	-	-	-	-	-	-	-	-	-	-
Mine Production Foreman	(#)	-	2	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	2	2	2	2
Drill & Blast Foreman	(#)	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Safety/Training Officer	(#)	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Training Officer	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Leading Hand	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL MINE SUPERVISION	(#)	1	7	10	10	10	10	10	10	10	10	10	7	7	7	7	7	7	7	7	7	7
TECHNICAL STAFF																						
Mine Technical Services Manager	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Senior Engineer	(#)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Planning Engineer	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Planning Technician	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Engineer	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining Engineers	(#)	-	-	1	1	1	1	1	1	1	1	1	-	-	-	-	-	-	-	-	-	-
Senior Geologist	(#)	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-	-	-	-	-	-
Geologist	(#)	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Surveyor	(#)	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surveyors	(#)	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Systems Administrator	(#)	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Office Assistant	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Clerk	(#)	-	-	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1
Senior Geotechnical Engineer	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydrological/Geotech Engineers	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Enviromental Engineers	(#)	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-	-	-	-	-	-
Grade Control/Sampling Technicians	(#)	-	2	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
TOTAL TECHNICAL STAFF	(#)	3	7	14	14	14	14	14	14	14	14	14	10	10	10	10	10	10	10	10	10	10
MINE MAINTENANCE SUPERVISION																						
Mine Maintenance Superintendent	(#)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-
Electrical Supervisor	(#)	-	2	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	2	2	2	2
Shovel/Drill Supervisor	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Equipment Maintenance Supervisor	(#)	-	2	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	2	2	2	2
Mine Maintenance General Forman	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Shift Supervisor	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Maintenance Engineer	(#)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Maintenance Planner	(#)	-	1	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1
Maintenance Clerk	(#)	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
TOTAL MAINTENANCE SUPERVISION	(#)	1	8	13	13	13	13	13	13	13	13	13	8	8	8	8	8	7	7	7	7	7
TOTAL MINE SUPERVISION	(#)	1	7	10	10	10	10	10	10	10	10	10	7	7	7	7	7	7	7	7	7	7
TOTAL TECHNICAL STAFF	(#)	3	7	14	14	14	14	14	14	14	14	14	10	10	10	10	10	10	10	10	10	10
TOTAL MAINTENANCE SUPERVISION	(#)	1	8	13	13	13	13	13	13	13	13	13	8	8	8	8	8	7	7	7	7	7
TOTAL OVERHEAD MANPOWER	(#)	4	22	37	37	37	37	37	37	37	37	37	25	25	25	25	25	24	24	24	24	24
CALCULATED LABOUR REQUIREMENTS																						
OPERATIONS																						
Drilling	(#)	-	2	4	4	4	4	4	4	4	4	4	2	2	2	2	2	2	2	2	2	1
Loading&Conveying	(#)	-	7	13	13	14	14	14	14	14	14	13	8	8	7	7	7	7	7	7	8	3
Rehandle (Load and Haul)	(#)	-	12	17	17	18	18	18	18	18	20	16	5	4	3	3	3	3	3	4	5	1
Ancillary	(#)	-	15	17	17	17	17	17	17	17	17	17	9	9	9	9	9	9	9	9	9	4
Total Operating Labour	(#)	0	35	51	51	53	53	53	53	53	55	50	24	23	21	21	21	21	21	23	25	9
MAINTENANCE																						
Drilling	(#)	-	1	2	2	2	2	2	2	2	3	2	1	1	1	1	1	1	1	1	1	0
Loading&Conveying	(#)	-	4	8	8	8	8	8	8	8	9	8	4	4	4	4	4	4	4	4	4	2
Rehandle (Load and Haul)	(#)	-	7	11	10	11	11	11	11	11	12	10	3	3	2	2	2	2	2	2	3	1
Ancillary	(#)	-	9	10	10	10	10	10	10	10	10	10	5	5	5	5	5	5	5	5	5	2
Total Maintenance Labour	(#)	0	21	31	31	32	32	32	32	32	33	30	14	14	13	13	13	13	13	14	15	5
TOTAL MINING PERSONNEL	(#)	4	78	119	118	121	121	121	121	121	125	116	63	62	58	58	58	57	57	60	63	37

APPENDIX 3

Cash Flow

Leighton Asia

SGC002 Mining Plan and Cost Estimate – Bronson Slope Project

Bronson Slope Cashflow Model	Year	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	TOTAL
Metal Prices																							
Copper	USD/lb			2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00	2.00
Gold	USD/oz			700	700	700	700	700	700	700	700	700	700	700	700	700	700	700	700	700	700	700	700
Silver	USD/oz			15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15	15
Exchange Rate	CAD:US			0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85	0.85
Production Data																							
Tonnes Cu/Au Ore Mined	kt			4,779	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	2,035	93,480
Tonnes Waste Mined	kt			6,836	6,611	7,010	7,064	7,038	7,053	7,003	7,881	5,926	1,726	1,319	779	846	850	778	793	1,236	1,951	328	73,026
Total Ore and Waste Mined	kt			11,616	11,709	12,108	12,162	12,136	12,151	12,101	12,979	11,024	6,824	6,417	5,877	5,944	5,948	5,876	5,891	6,334	7,049	2,362	166,506
Strip Ratio (waste t : ore t)				1.43	1.30	1.37	1.39	1.38	1.38	1.37	1.55	1.16	0.34	0.26	0.15	0.17	0.17	0.15	0.16	0.24	0.38	0.16	0.78
Ore Milled	kt			4,779	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	2,035	93,480
Grade - Copper	%			0.210	0.217	0.164	0.086	0.153	0.174	0.170	0.162	0.166	0.163	0.167	0.153	0.147	0.144	0.140	0.136	0.130	0.135	0.154	0.156
Grade - Gold	g/t			0.525	0.567	0.570	0.564	0.539	0.517	0.491	0.456	0.366	0.402	0.422	0.409	0.401	0.399	0.382	0.378	0.370	0.359	0.360	0.449
Grade - Silver	g/t			2.46	2.65	2.77	3.02	2.77	2.57	2.45	2.37	2.05	2.46	2.35	2.09	1.91	2.06	2.12	2.13	2.14	2.17	2.03	2.356
Concentrate Data																							
Copper Recovery	%			86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60	86.60
Gold Recovery	%			85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40	85.40
Silver Recovery	%			63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70	63.70
Copper Recovered to Concentrate	million lbs			19.2	21.1	15.9	8.4	14.9	17.0	16.6	15.8	16.1	15.9	16.2	14.9	14.4	14.0	13.6	13.2	12.6	13.2	6.0	279
Gold Recovered to Concentrate	koz			68.8	79.4	79.7	79.0	75.4	72.3	68.7	63.8	51.3	56.2	59.0	57.2	56.1	55.9	53.5	52.9	51.8	50.2	20.1	1,151
Silver Recovered to Concentrate	koz			240.5	276.8	288.9	315.6	289.4	267.9	255.5	247.9	214.5	256.5	245.5	218.4	199.1	215.1	221.4	222.3	223.6	226.4	84.8	4,510
Concentrate Production																							
tonnes (dry)	kdmt			34.5	38.0	28.7	15.1	26.8	30.5	29.8	28.4	29.0	28.6	29.2	26.8	25.8	25.3	24.5	23.8	22.7	23.7	10.8	502.1
Copper Grade (dry)	%			25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.20	25.2
Gold Grade	g/dmt			62.0	65.0	86.4	162.3	87.6	73.7	71.7	69.8	54.9	61.1	62.8	66.5	67.5	68.7	67.9	69.1	70.8	65.9	58.2	71.3
Silver Grade	g/dmt			216.8	226.5	313.2	648.5	336.3	273.0	266.6	271.4	229.7	278.5	261.4	253.6	239.7	264.7	281.3	290.6	305.8	297.0	245.3	279.4
Total Contained Gold Metal				80.6	93.0	93.4	92.5	88.3	84.7	80.5	74.7	60.1	65.8	69.1	67.0	65.7	65.4	62.6	61.9	60.7	58.8	23.6	72.5
TC/RC Terms				95.3	95.8	100.1	115.3	100.5	97.7	97.3	97.0	94.3	95.7	95.8	96.3	96.4	96.8	96.8	97.1	97.5	96.7	94.9	97.4
Concentrate Transportation	USD/dmt			129.0	129.4	132.3	142.5	132.5	130.6	130.4	130.1	128.2	129.1	129.2	129.7	129.8	130.0	129.9	130.1	130.3	129.7	128.6	130.3
Net Realized Prices																							
Copper	USD/lb			1.72	1.73	1.75	1.79	1.75	1.74	1.73	1.73	1.72	1.72	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.73	1.72	1.73
Gold	USD/oz			629	630	637	651	638	634	633	632	626	629	630	631	631	632	632	632	633	631	628	632.29
Silver	USD/oz			12.23	12.25	12.39	12.66	12.40	12.32	12.30	12.29	12.17	12.23	12.24	12.26	12.27	12.28	12.28	12.29	12.30	12.27	12.20	12.29
CASH FLOW																							CAD 000's
Net Revenue																							
Copper - 28.2%	CAD 000's			38,890	42,930	32,775	17,664	30,594	34,642	33,797	32,191	32,587	32,282	32,952	30,278	29,217	28,623	27,704	26,944	25,793	26,807	12,091	568,762
Gold - 43.3%	CAD 000's			50,953	58,882	59,800	60,479	56,601	53,929	51,196	47,464	37,789	41,632	43,720	42,501	41,656	41,539	39,748	39,323	38,575	37,290	14,860	857,936
Silver - 3.2%	CAD 000's			3,459	3,989	4,212	4,699	4,221	3,881	3,698	3,585	3,071	3,690	3,535	3,151	2,874	3,109	3,198	3,213	3,236	3,267	1,217	65,306
Net Revenue				**93,302**	**105,801**	**96,787**	**82,842**	**91,416**	**92,452**	**88,691**	**83,239**	**73,448**	**77,605**	**80,208**	**75,930**	**73,747**	**73,270**	**70,649**	**69,480**	**67,604**	**67,364**	**28,168**	**1,492,004**
NSR / t Milled	CAD / t Milled			19.52	20.75	18.99	16.25	17.93	18.13	17.40	16.33	14.41	15.22	15.73	14.89	14.47	14.37	13.86	13.63	13.26	13.21	13.84	16.22
Operating Costs	CAD																						
Mining	3.14 per t milled			21,872	21,913	22,482	22,616	22,551	22,587	22,465	24,215	20,394	11,127	9,997	9,166	9,318	9,327	9,165	9,198	9,884	11,447	3,691	293,414
Milling	5.23 per t milled			24,982	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	26,647	10,636	488,621
G&A	0.98 per t milled			9,651	9,923	9,923	9,923	9,923	9,923	3,614	3,614	3,614	2,349	2,349	2,349	2,349	2,349	2,226	2,226	2,226	2,226	890	91,652
Total Direct Operating Costs	**9.35 per t milled**			**56,505**	**58,484**	**59,052**	**59,187**	**59,121**	**59,158**	**52,726**	**54,476**	**50,655**	**40,124**	**38,994**	**38,163**	**38,315**	**38,324**	**38,039**	**38,071**	**38,757**	**40,320**	**15,217**	**873,686**
Direct Operating Costs per t Ore Milled				11.82	11.47	11.58	11.61	11.60	11.60	10.34	10.69	9.94	7.87	7.65	7.49	7.52	7.52	7.46	7.47	7.60	7.91	7.48	9.35
Operating Pretax Cash Flow				36,797	47,317	37,735	23,656	32,295	33,294	35,965	28,764	22,793	37,481	41,214	37,768	35,432	34,946	32,611	31,409	28,847	27,044	12,951	618,317
Capex		102,055	125,391	24,034	500	500	500	500	3,411	5,964	500	2,016	500	1,988	500	500	1,886	500	500	500	5,500	(10,561)	267,183
Pre-tax Cash Flow		**(102,055)**	**(125,391)**	**12,763**	**46,817**	**37,235**	**23,156**	**31,795**	**29,883**	**30,001**	**28,264**	**20,777**	**36,981**	**39,226**	**37,268**	**34,932**	**33,060**	**32,111**	**30,909**	**28,347**	**21,544**	**23,512**	**351,135**
	Cumulative	(102,055)	(227,446)	(214,683)	(167,865)	(130,630)	(107,475)	(75,680)	(45,796)	(15,796)	12,468	33,245	70,226	######	146,720	181,652	214,712	246,823	277,732	306,079	327,623	351,135	
Taxation																							
	Federal Income	-	-	-	-	-	-	-	-	-	-	-	(631)	(4,763)	(4,651)	(4,440)	(4,263)	(4,235)	(4,180)	(3,894)	(2,962)	(3,303)	(37,322)
	Provincial Income	-	-	-	-	-	-	-	-	-	-	-	(463)	(3,493)	(3,411)	(3,256)	(3,126)	(3,106)	(3,065)	(2,856)	(2,172)	(2,422)	(27,370)
	Mining Tax	2,041	2,508	(255)	(936)	(745)	(463)	(636)	(598)	(600)	(565)	(416)	(740)	(785)	(745)	(699)	(661)	(642)	(618)	(567)	(431)	(470)	(7,023)
	Total Taxes	**2,041**	**2,508**	**(255)**	**(936)**	**(745)**	**(463)**	**(636)**	**(598)**	**(600)**	**(565)**	**(416)**	**(1,833)**	**(9,041)**	**(8,807)**	**(8,394)**	**(8,051)**	**(7,983)**	**(7,863)**	**(7,317)**	**(5,566)**	**(6,195)**	**(71,715)**
																							-
Net After Tax Cash Flow		(100,014)	(122,883)	12,508	45,881	36,490	22,693	31,159	29,286	29,401	27,698	20,362	35,147	30,186	28,460	26,538	25,009	24,127	23,046	21,030	15,978	17,317	279,420