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PRELIMINARY ASSESSMENT UPDATE – BRONSON SLOPE PROPERTY
FOR THE
SKYLINE GOLD CORPORATION

Submitted to:
Skyline Gold Corporation

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3.0 Summary

Skyline Gold Corporation ("SGC") have commissioned Moose Mountain Technical Services ("MMTS") to complete an updated Preliminary Economic Assessment (PEA) with a specific focus on effect of the addition of magnetite to the resource for their Bronson Slope deposit, which is located in North West British Columbia, Canada. The PEA is a follow-up to a previous PEA Technical Report by Leighton Asia Limited ("LAL") and relies on that report for the process plant project capital and operating cost estimates. This technical report has been compiled to disclose the findings of the Preliminary Economic Assessment.

The mineral resource model and resource estimates used in this updated PEA is based on the previous technical reports 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., from Burgoyne Geological Inc., and “Magnetite Mineral Resource Estimate – Bronson Slope Deposit” January 28, 2010, authored by G. H Giroux, P. Eng., A. A. Burgoyne, P.Eng., and A. Burgert, P.Geo., all independent Qualified Persons as defined by NI 43- 101. These Technical reports can be viewed at www.sedar.com.

This report is intended to be read as a whole and 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 and magnetite deposit is hosted within the property with approximate dimensions of 1.5km long and 0.4 to 0.6km wide. The depth of the ore body 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 totaling approximately 186.9 hectares.

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 totaling 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 identifying 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 totaling 3936 metres were drilled in order to increase mineral resource confidence and also to develop additional resource.
- 2008 - Publication of a PA and Technical Report by Leighton, entitled "Preliminary Economic Assessment with Mining Plan and Cost Estimate.
- 2010 - Commissioning of this PEA and other associated development studies.

3.2.2 Mineral Resource and Estimate

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), A. A. Burgoyne and G. H. Giroux (2007-2008), and A. A. Burgoyne, A Burgert, and G. H. Giroux (2010). Refer to A. A. Burgoyne, A Burgert, and G. H. Giroux (2010) for more details. A summary of the 2010 resource estimate has been included 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 utilized during the field exploration season which occurs from late May to early November.

The existing 40km 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 35km segment is followed by a segment of approximately 5km 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. Proposed site layout of mining and site infrastructure and processing infrastructure is summarized 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. 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.

3.4 Geology, Mineralization 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.

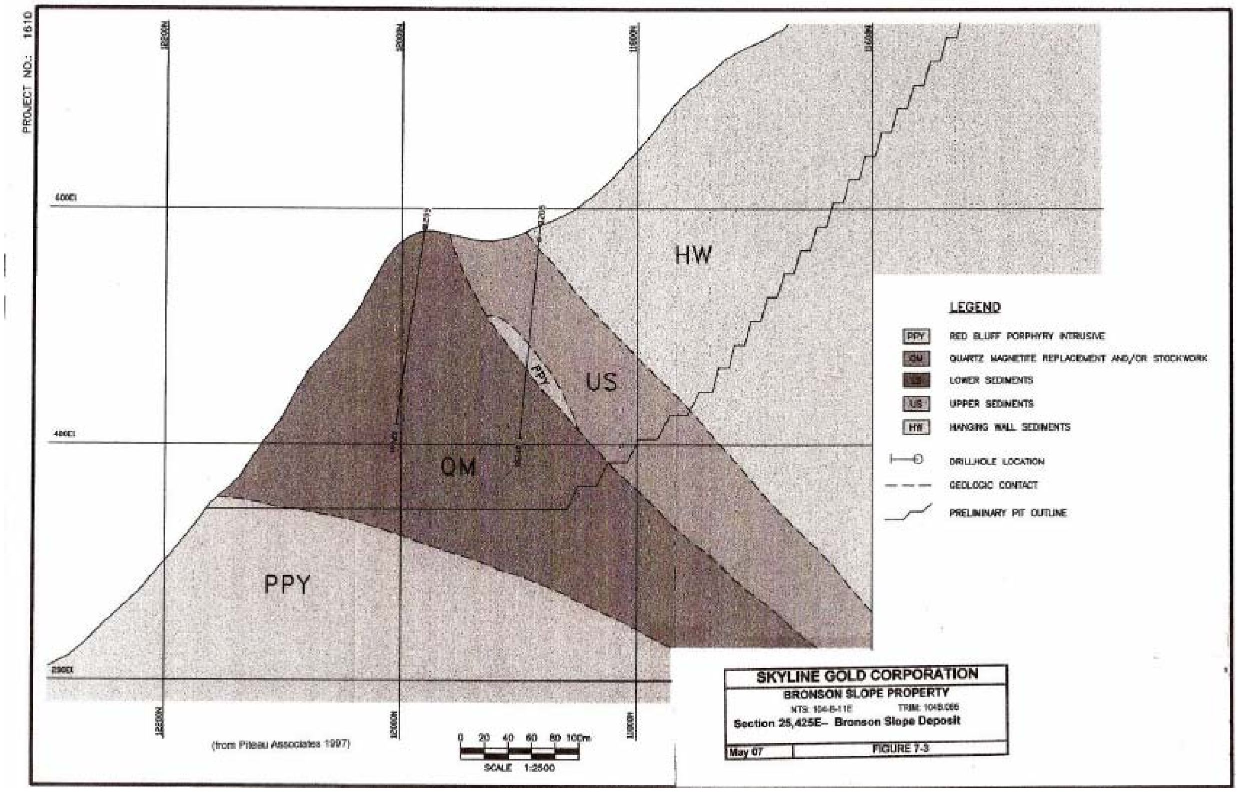


Figure 3-1 Mineralization Section View of the Bronson Slope Deposit

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. Magnetite mineralization is present in a quartz-magnetite-hematite stock work.

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., and “Magnetite Mineral Resource Estimate – Bronson Slope Deposit” January 28, 2010, authored by G. H Giroux, P. Eng., A. A. Burgoyne, P.Eng., and A. Burgert, P.Geo., all independent Qualified Persons as defined by NI 43-101. These Technical reports can be viewed at www.sedar.com. The resource estimate is based on kriging and block modeling.

The updated resource is calculated on updated metal prices and the inclusion of magnetite as a product. A comparison to the resource estimates from 2008 to this study is included in Table 3-1 below.

The 2008 resource metal prices are based on the figures used in the prior 2008 Technical Report by LAL. The 2010 metal prices are updated metal prices and the inclusion of Magnetite as a product. Molybdenum was discounted in the Leighton report and no product value has been included in the updated resource.

The metal prices were used along with metallurgical recoveries to determine individual block values in the 2008 resource estimate. The mineral resource presented in the Table 3-1 below was then determined based on a cut off of $9.00/tonne net recoverable value.

Table 3-1 2008 Mineral Resource Estimate based on 2008 Resource Metal Prices

2008 - Bronson Slope Resource Estimate (Cutoff USD 9/t NSR)					
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
Total Measured + Indicated	**225,100,000**	**0.36**	**2.22**	**0.14**	**0.0077**
Inferred	91,600,000	0.27	1.76	0.13	0.0080

The magnetite mineral resource from "Technical Report, Magnetite Mineral Resource Estimate – Bronson Slope Deposit" by A. A. Burgoyne, P.Eng., M.Sc., Arnd Burgert, P.Geo., B.Sc. and G. H. Giroux P.Eng., MASc**.** dated January 28, 2010 is summarized in Table 3-2 below. It should be noted that the magnetite resource was developed as an independent resource and did not include valuation of other minerals in its determination.

Table 3-2 Bronson Slope – Global Magnetite Mineral Resource

**Cut Off 2% Magnetite*

Category	**Metric kTonnes**	**% Magnetite**	**Contained Magnetite kTonnes**
Measured	66,210	7.58	5,020
Indicated	96,950	7.08	6,860
Total Measured + Indicated	**163,160**	**7.28**	**11,880**
Inferred	6,300	6.92	440

It should be noted that when this global magnetite resource was compared to the preliminary open pit designed for the PEA completed by LAL in March 2009 that 44% of the measured and indicated magnetite tonnage, above a 2% cut-off, was within the pit. The remainder sits below and to the west of the 2008 design pit.

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" was 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.

The previous Leighton report stated;

> *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. indicates in a letter Progress Report dated September 18, 2008, and supported by discussion with the report's 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.*

This statement forms the basis of using a 95% recovery for the magnetite. Further processing and recovery testing is recommended.

A review of current prices and markets for magnetite indicates that western Canadian metallurgical coal markets consume 75 ktpa and a FOB at the mine gate of $130 is competitive. The US market for magnetite is in the order of 300 ktpa with 200 ktpa used in the metallurgical coal market. It is assumed that another 50 ktpa above the Canadian coal market could be achieved for a total of 125 ktpa at USD$130/t. Any production above this would be sold as iron ore on the spot market.

3.6 Mine Plan

Mining Method Selection

The LAL 2008 study recommended using a mobile in-pit conveying system. This method has a lower labour requirement (significant cost in isolated Canadian sites) than truck shovel, however is more complex to schedule. Consideration is made for the inflexibility of the conveying system when developing a LOM production schedule for the project. This mining method is assumed in the update PEA. Future, more detailed studies are recommended to identify the impact of conveyor moves on mine productivity as well as comparing to an alternative truck shovel mining plan.

The HAC option methods of transporting ore and waste is 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.

A conventional conveyor is 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 with a 500m truck haul for final placement in the dump. 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 using haul trucks in preference over using a conveyor system for waste rock transport to storage, is required.

All material below the USD9.00/t NSR cut-off will be taken to the Triangle Lake storage area. An opportunity exists to segregate this storage area further into below cut-off NSR values. This will allow selective re-handling 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 re-handling concept be conducted in the next phase of evaluation.

Available Hours and Utilization

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

Consideration is made for delays that will affect production over and above the equipment planned and corrective maintenance allowances. These delays are summarized in

Table 3-3. 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 are also been included in

Table 3-3. The estimate is likely to be conservative, however due to the relatively variable operating climate in the area it has been decided to use conservative values for this PEA and recommend further data collection and analysis is completed prior to preparation of a prefeasibility study.

Table 3-3 Summary of Planned Delays and Maximum Utilization

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**
Utilization	**90%**		**91.6%**

The physical and financial inputs and drivers used for the pit optimization are provided in Table 3-4.

Table 3-4 Initial Bronson Slope Mine Design Parameters

Parameters Description	
Metal Prices:	
Cu	USD$2.50/lb
Au	USD$900/t oz
Ag	USD$15/t oz
Magnetite	USD$90/tonne
Geotechnical Design Parameters	
Average - All domains	50 degrees --Adapted from Piteau Report
Mining Parameters	
Bench Height (m)	20
Mining cost - Base ($/t)	CAD$1.77/t
Adjustment for depth ($/t/bench)	not considered
Mining dilution (%)	5%
Mining recovery (%)	95%
Concentrator Parameters	
Expected Mill Throughput (t/annum)	5.1Mtpa
Variable processing cost ($/t milled)	$8.89/t
Concentrator Recoveries	
Au	84%
Ag	61%
Cu	86.6%
Magnetite	95%

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)	$50/t of concentrate
Concentrate moisture content (%)	8%
Marketing cost(s)	Nil
Sales cost(s)	
Sales commission(s) (%)	
General and Administration Overhead Costs	
Admin and Overhead unit cost ($/t milled)	$1.00/t milled

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 (mined in 10m benches)
- Batter Angle — 75°
- Berm Width — 11.2 to 11.4m
- Overall Wall Angle — 50°

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 modeling of geotechnical risks is completed prior to commissioning of the project.

The pit size increased sufficiently from the 2008 design parameters to allow for 3 phases instead of 2. Figure 3-2 to Figure 3-4 show the detailed pit designs individually. A series of sections through these pits and the resource model have been provided in Section 25.

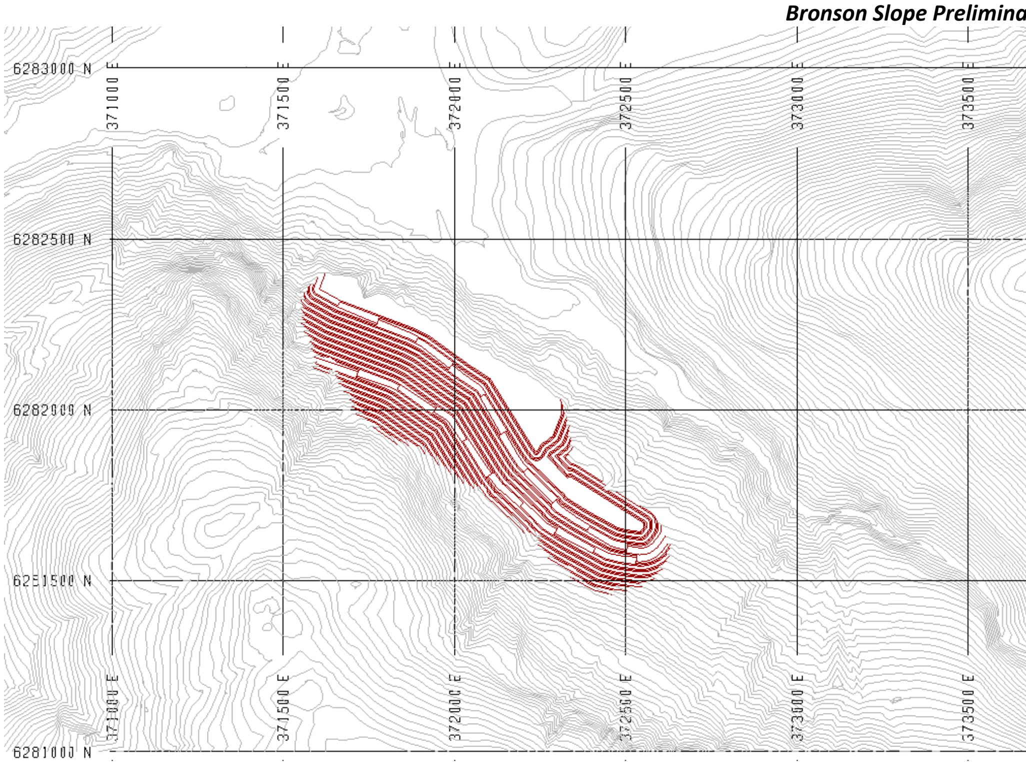


Figure 3-2 Pit 1 Bench Plan

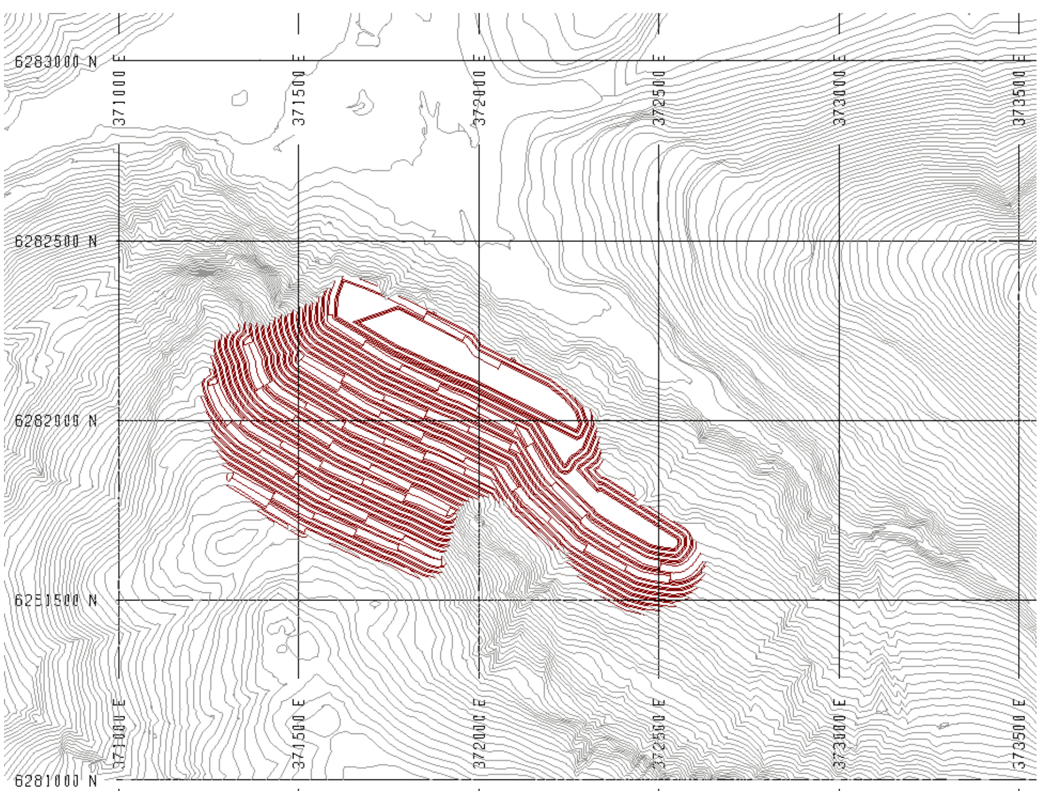


Figure 3-3 Pit 2 Bench Plan

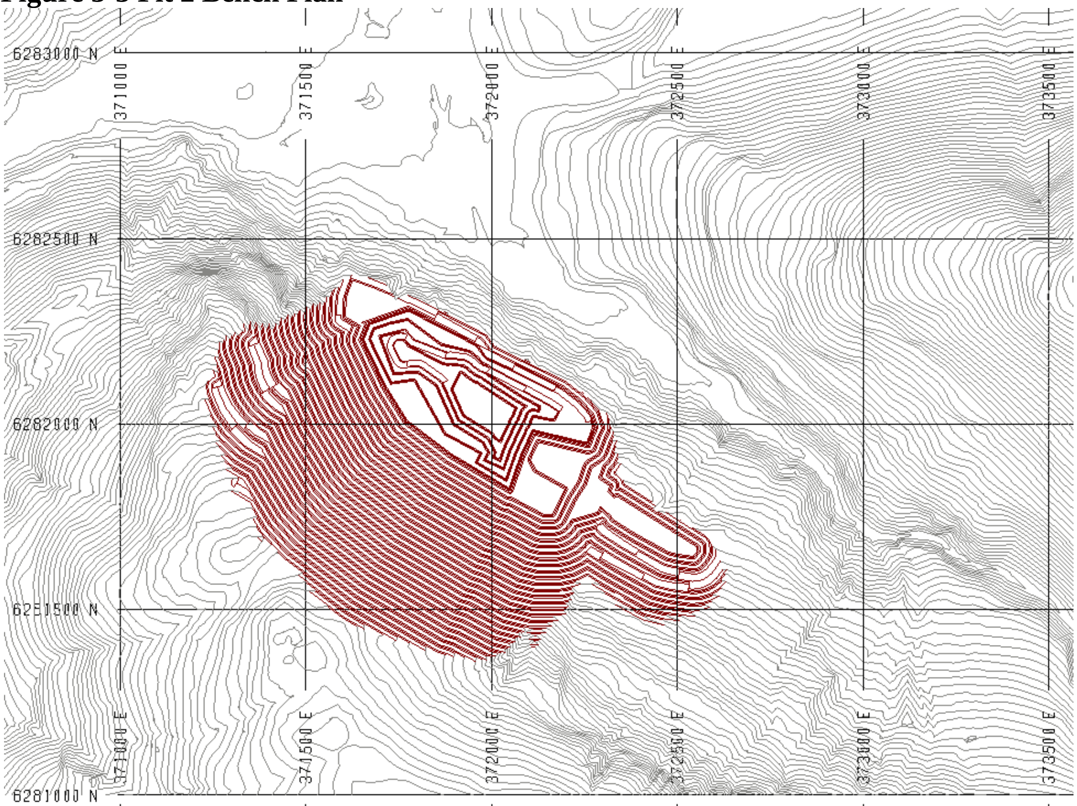


Figure 3-4 Final Detailed Pit Plan

The sum of these phases (total LOM mill feed tonnage and grade) is included in Table 3-5.

Table 3-5 LOM Mill Feed Tonnage and Grade

Phase 1, 2 & 3 (Total) - Tonnes and Grades ($9/t NSR Cut-off)					
Category	**Metric ktonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Mag %**
Total MII	**191,835**	**0.116**	**0.343**	**2.13**	**5.3**
Total Waste	147,499	**Strip Ratio: 0.77 Waste t/Mill Feed t**			
Total Mill Feed and Waste	**339,334**				

LOM Production Schedule

The schedule allows for the progressive sequencing of material movement by phased pit design and bench and assumes that for each bench the material types are mined in weighted equal portions until the bench is 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 is provided in the Figure 3-5 below.

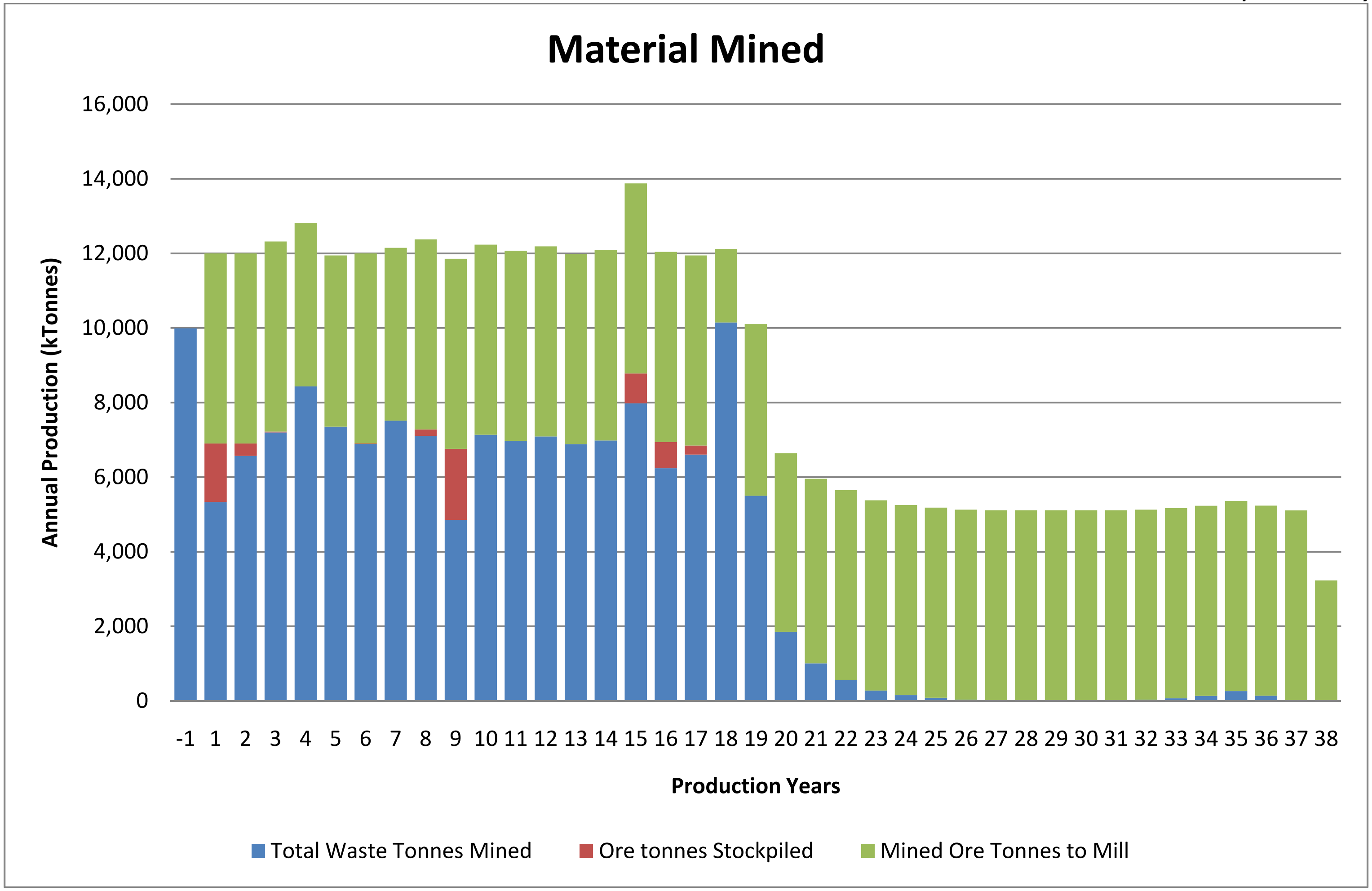


Figure 3-5 Annual Material Movement

A Table containing the metal production by year is provided below.

Table 3-6 Metal Production by Year

Variable	Unit	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11-15	Y16-20	Y21-25	Y26-30	Y31-35	Y36-38	Total
Mill Feed	Ktonne	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	25,490	25,490	25,490	25,490	25,490	13,405	191,835
Cu Grade	%	0.15%	0.15%	0.15%	0.14%	0.15%	0.15%	0.15%	0.15%	0.13%	0.10%	0.08%	0.10%	0.14%	0.12%	0.10%	0.09%	
Recovered Cu	000'slbs	14,697	14,600	14,600	13,432	14,600	14,405	14,502	14,405	12,556	7,446	30,002	43,507	69,689	51,002	36,061	17,121	382,624
Au Grade	g/t	0.54	0.499	0.407	0.378	0.453	0.41	0.425	0.445	0.406	0.409	0.270	0.306	0.371	0.346	0.284	0.252	
Recovered Au	000's oz	78.6	72.6	59.2	55.0	65.9	59.7	61.9	64.8	59.1	50.6	167.1	206.0	269.8	230.0	176.0	81.2	1,757
Ag Grade	g/t	2.54	2.54	2.47	2.40	1.94	2.21	2.10	1.98	1.90	1.88	1.749	2.189	2.280	2.033	1.881	1.937	
Recovered Ag	000's oz	253.8	254.3	246.8	240.0	193.9	220.6	210.1	197.7	189.7	138.9	645.1	940.6	1,139.6	857.2	694.0	387.2	6,809
Magnetite	%	3.14%	4.92%	6.49%	5.61%	1.20%	4.75%	5.01%	6.20%	7.22%	7.41%	6.50%	3.75%	3.75%	5.99%	6.45%	5.65%	
Recovered Magnetite	ktonne	152.1	238.3	314.3	271.7	58.1	230.0	242.6	300.3	349.7	358.9	1,575.0	908.6	909.0	1,451.5	1,560.9	736.0	9,657

Table 3-7 Cumulative Metal Production Comparison

	Leighton 2008	MMTS 2010
Mine Life (years)	19	38
Mill Feed (ktonne)	93,480	191,835
Cu (000's Lbs)	278,962	383,624
Cu%	0.155	0.117
Au (000's Oz)	1,151	1,757
Au (g/t)	0.446	0.342
Ag (000's Oz)	4,510	6,809
Ag (g/t)	2.34	2.05
Magnetite (k tonne)	N/A	9,657
Magnetite (%)	N/A	5.41

The schedule shows that pre-stripping of Pits 1 and 2 for one year is required to provide low copper grade mineralized highwall ore from Pit 2 for blending with the higher copper grade ores in Pit 1 from years 1 to 5. In year 6, Pit 3 mining begins. Between year 6 and 9 mill feed is from pits 1 and 2 with the low grade copper zone in Pit 3 being treated as waste but strategically piled for possible future recovery. This potential mill feed material has a low copper grade but is moderately high in gold grade. Further study needs to be conducted to determine whether the gold recoveries of this material are achievable with very low copper grades. The current plan wastes this material that is currently unable to be blended with higher grade copper ore. It may be possible to progressively blend it into the mill feed to reduce the impact of the reduced copper grades but requires further studies and is addressed as a future opportunity.

Between year 10 and 18 the majority of the mill feed is taken from the lower elevations of Pits 1 and 2 which currently have lower copper grades and therefore recoveries in these years are reduced as an allowance to account for copper feed grades that are almost half the previous years. By year 19, the higher copper grade material in Pit 3 is the primary feed for the mill and the copper grade allows design recoveries to again be applied. At this point the final pit is low enough in strip ratio (year 20 onward) that the production fleet can be reduced to one excavator, crusher combination until the base of the pit is reached in year 38.

The optimum production schedule requires a high strip ratio early in the mine life as a result of pre-stripping requirements for Pits 2 and 3 and the narrow shape of the overall pit and the ore body. Pits 2 and 3 needs to start early in the mine life (Pre-strip and year 6 respectively) in order to ensure consistent mill feed of 5mtpa at the later stages in the mine schedule.

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 a copper gold bulk concentrate and gravity gold transported off site for smelting and refining. A 15,000 tonnes per day (tpd) mill feed processing rate and a concentrator utilization of 93% will produce a total of 29,000 tonne per annum (tpa) of copper concentrate containing both gold and silver. A process flowsheet design from the Leighton Report has been provided in Section 18 and does not show a magnetite recovery circuit.

The concentrator plant design consists 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 is 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 is thickened with the aid of flocculants in a conventional thickener and filtered to reasonable moisture for dispatch off site.

Magnetite recovery process will likely consist of a series of magnetic drums installed at strategic locations throughout the mill. These drums attract the magnetite to the drum which then is scraped off for final collection. Future studies should include incorporating the magnetite recovery system into the flowsheet.

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 it is intended that 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. Future design work is required in these areas.

3.8 Access and Power

The project is 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 30km east of this access road along the Iskut River. Forsite Consultants Ltd was 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 is 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 the site (approximately 25km). 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 updated PEA, the previous 2008 cost estimates for the capital and operating costs expected for the Bronson Slope Project based on the selected mining and processing methods and schedules are reviewed.

This cost estimation review is completed by utilizing a combination of techniques which are summarized below. All cost referred to are in Canadian dollars (CAD) unless otherwise stipulated. Further detail is provided in Section 25.

3.9.1 General Site Infrastructure Capital Estimate

The following

Table 3-8 provides a summary of the expected buildings, services and infrastructure capital costs for the project. These are unchanged from the 2008 study since the throughput capacity of the operation has not changed.

Table 3-8 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 unit rate for the LOM general and administration costs is $0.98/t milled. These costs are also unchanged from the 2008 study.

3.9.3 Processing Capital Cost Estimate

The Processing capital and costs used in this report were taken from the 2008 Leighton report and have not been inflated to 2010 numbers. The additional capital cost of a magnetite circuit is considered to be well within the contingency cost applied to the Leighton numbers.

Table 3-9 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 indirect costs are presented in Table 3-10 below.

Table 3-10 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 plant operating costs have been taken from the 2008 Leighton report without escalation to 2010.

The addition of the magnetite circuit and summary of the costs are shown in Table 3-11 below.

Table 3-11 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		**$5.23**
Magnetite Processing Cost		**$1.10**

An allowance of $1.10/t processing cost for the magnetite recovery has been added to the overall processing cost. This cost is estimated based on a simple series of magnetite drums located to recover the final magnetite particulates. Crushing and conveying costs of the ore has already been accounted for in the above milling process.

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.
All equipment costs are in CAD and primarily based 2008 and 2009 pricing with exchange rates of USD $0.90 applied. No escalation of costs to 2010 is estimated or included.

Replacement capital for equipment is based on 7 year rebuilds and 14 year life for loading and hauling units only while conveyor replacement costs are included in the annual maintenance costs. 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 3-12 summarizes the estimate of overall equipment Capex requirements for the mine operations. Initial capital estimates are provided and an allowance of $5 million capital every 7 years for loading equipment replacement

Table 3-12 Summary Mining Capital Requirements

	Cap Cost $M
Drilling	2.0
Blasting	
Loading	2.8
Mobile Crushing	3.4
In-Pit Conveying	9.5
HAC Transfer areas	2.4
HAC Transfer stations	4.0
HAC	23.5
Overland Conveyor	3.4
Stacker/Hopper	2.0
Waste Rehandle	1.7
Waste Fleet	3.0
Road Construction	5.0
Dump Maintenance	1.5
Total	**64.1**

It is important to note that for this operation the mining costs include the primary 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 primary 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 no escalation of costs is estimated or included. Pricing of all costs in this study also excludes consideration for taxation.

Table 3-13 provides a summary of the total and unit mining costs expected for the life of the Bronson Slope Project.

Table 3-13 Mine Operating Cost Summary

	Total OpEx
	CAD $/t mined
Drilling	0.15
Blasting	0.34
Loading	0.20
Mobile Crushing	0.23
In-Pit Conveying	0.11
HAC loading area	0
Feeders	0.02
HAC	0.11
Overland Conveyor	0.03
Stacker/Hopper	0.01
Waste Rehandle	0.06
Waste Fleet	0.23
Road Maintenance	0.19
Dump Maintenance	0.12
Total	**1.81**

3.9.7 Project Operating Cost Summary

A summary of the site wide operating costs has been included in
Table 3-14. Additional details of the project operating cost estimate are provided in Section 25.

Table 3-14 LOM Project Operating Cost Summary

Cost Category	Unit	LOM
Direct Mining Costs	(CAD,000)	$596.130
	(CAD/tonne Mined)	$1.81
	(CAD/tonne Milled)	$3.11
Overheads and Administration	(CAD,000)	$187,988
	(CAD/tonne Mined)	$0.50
	(CAD/tonne Milled)	$0.98
Processing Costs	(CAD,000)	$1,214,316
	(CAD/tonne Milled)	$6.33
Total Site Operating Costs	(CAD,000)	$1,998,444
	(CAD/tonne Milled)	$10.44
	(USD/tonne Milled)	$9.39

3.10 Project Economics

Economic evaluation for the Project is based on a pre-tax financial model. For the Project as defined in this update of 38 years and 191 Mtonnes mill feed, the following base case financial results are:

- 21.5% IRR
- $330.2 million NPV at 7.5% discount
- 4.8 year back on $241.3 million
- $1,405.6 million NPV at undiscounted cash flow

Constant metal prices and a constant foreign exchange rate are used in the pre-tax model:

- Copper: USD$2.50/lb
- Gold: USD$ 950/t.oz
- Silver: USD$ 15/t.oz for Silver were used for the cash flow analysis
- Magnetite: USD$ 130/t as DMS, USD$ 50/t as iron ore (net price, FOB Stewart or other)
- Exchange rate: USD$ 0.90: CAD$ 1.00

The financial model includes an initial capital of CAD$241.3 million and a sustaining capital of CAD$38.5 million. No allowance has been made for working capital, or salvage value, and no reclamation or closure costs have been estimated or included.

Annual Cash Flow

Production statistics from the mine production schedule are incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tonnage milled, head grades, and recoveries. Market prices for copper, gold, and silver are adjusted to realized price levels by applying smelting, refining, and concentrate transportation charges from mine site to smelter to determine revenue (NSR) from copper concentrate and gravity concentrate sales. Realized price for magnetite are calculated assuming FOB Stewart.

Unit operating costs for mining, milling, and G&A areas are applied to annual mined or milled tonnages to determine the overall mine site operating cost which are deducted from NSR to derive annual Net Revenues.

Initial and sustaining capital costs are incorporated on a year-by-year basis over the mine life and deducted from Net Revenue to determine Net Cash Flow before taxes.

The undiscounted annual cash flow and cumulative cash flow for the first 15 years of operations is shown below.

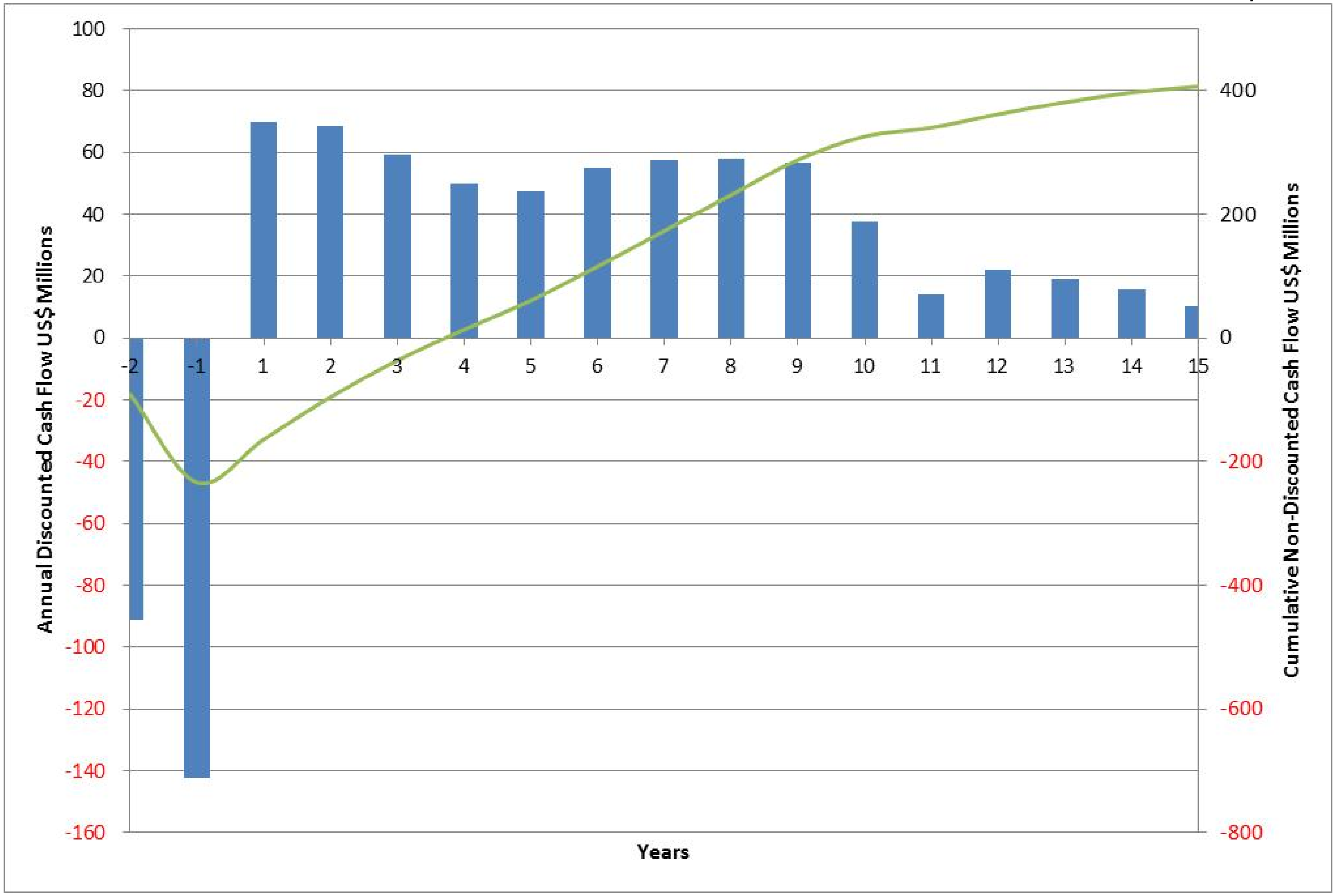


Figure 3-6 Undiscounted Annual and Cumulative Cash Flow

3.11 Sensitivity Analysis

The IRR and NPV has been calculated and summarized in Table 3-15.

Table 3-15 Sensitivity Analysis

	IRR	NPV (CAD Million)
Base Case	**21.5%**	**330.2**
Metal Price		
Current Prices (Sept, 2010)[1]	35.2%	684.2
10% Decrease	16.2%	199.7
10% Increase	26.5%	460.8
Recovery		
10% Decrease	17.6%	232.5
OPEX		
20% increase	16.4%	198.6
10% decrease	23.8%	396.1
CAPEX		
20% Increase	17.4%	280.9
10% Decrease	24.2%	354.9
Exchange Rate (1 CAD=0.9USD)		
1CAD = 0.95 USD	18.9%	264.6
1 CAD= 0.85 USD	24.3%	403.5
Discount Rate (Base Case 7.5%)		
5%	21.5%	514.2
10%	21.5%	215.2

[1]*Current prices used are USD$1250 Au; USD$3.50 Cu; USD$20 Ag; USD$90 Mag.*

3.12 Comparison to Previous Study

This study is an update of the previous 2008 study by Leighton with changes to the effects of higher metal prices and addition of magnetite to the model. The mill feed rate remains unchanged between the two studies. Processing capital and operating costs remain unchanged except for the operating cost of an additional magnetite recovery circuit. Mining costs have been updated based on updated high angle conveyor estimates. The Discount rate remains unchanged at 7.5%. Table 3-16 summarizes the effects on the mine plan.

Table 3-16 Study Comparison of Key Points – Leighton vs MMTS

	Leighton 2008	MMTS 2010
Mine Life (years)	19	38
Mill Feed (ktonne)	93,480	191,835
Cu (k Lbs)	278,962	382,624
Au (k Oz)	1,151	1,757
Ag (k Oz)	4,510	6,809
Magnetite (k tonne)	N/A	9,657
CAPEX ($Million)	236.8	257.6
IRR	11.0%	21.5%
NPV ($Million)	59.3	330.2
Payback (years)	8.2	4.8

**All $ are in CAD*

3.13 Conclusions and Recommendations

The Preliminary Economic Assessment for the Bronson Slope property is 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:

- "Magnetite Mineral Resource Estimate – Bronson Slope Project" dated January 28, 2010, authored by G. H. Giroux, P. Eng. MASc, Arnd Burgert P. Geo. B.Sc, and A. A. Burgoyne, P.Eng., M.Sc

- "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

- "Technical Report Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation Vancouver on the Bronson Slope Property" dated March 6, 2009, authored by J. A. R. Lawrence, MAusIMM and V. Seen, MAusIMM Leighton Asia Limited., independent Qualified Persons as defined by NI 43-101.

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 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 $950/t.oz, Copper price of $2.50/lb, Silver price of $15/t.oz and a weight average magnetite price of $90/tonne and optimizing the mine plan has shown an IRR of 21.3% before tax is achievable based on an economic mine life extending 38 years.

All prices are in USD. Further opportunity exists to enhance the project return by:

1. Extending drilling to the South West to include both the highwall mineralization zone and the main mineralization zone. Both zones currently truncate within the design pit. The extent of the missing mineralization is shown in Figure 3-7 below.

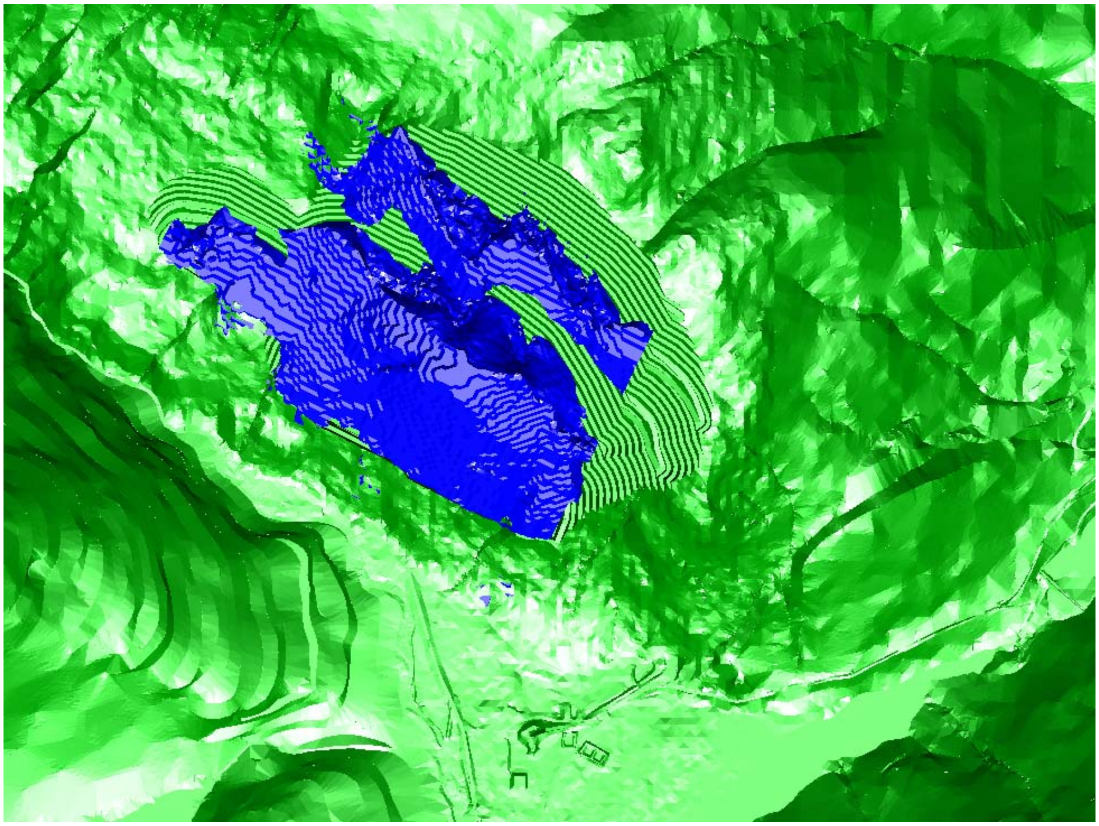


Figure 3-7 Mineralized Zone within the Ultimate Design Pit

2. Review processing opportunities for the high wall gold mineralized material. This material is currently being considered as waste until an economical process can be developed.
3. Drill and assay to upgrade the Inferred resources to at least Indicated levels so they can be included in ore reserve estimates in future Pre-Feasibility studies.
4. After completion of the Preliminary Assessment Pit and conceptual LOM schedule and cost and revenue inputs it was identified that a smaller final pit may result in a higher IRR.

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

- Complete a magnetite study including processing capital and operating costs and market size and prices.
- Complete a review of current recoveries and impact of lower feed grades on these recoveries.
- Complete a study on possible metallurgical recovery of low copper grade material for gold and silver.
- 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 including truck/shovel inpit operations, should also be considered.
- Develop new economic pit shells based on varying economics for the three mineralized zones; mill feed grade copper, magnetite and gold (low grade copper), primarily gold (highwall gold zone)
- Complete a study of crusher sizing and also consider electrical vs diesel crusher economics
- Complete a study to identify the cost savings potential of using used mining and processing equipment now available on the market.
- Complete further studies comparing grinding size versus recovery to identify the optimum grind size.
- Further develop self-generation hydro projects as the basis of power supply.
- 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.
- Complete further geotechnical study of the highwall slope including higher pit wall angle and potential to improve project economics.
- Complete a revised Tailings Storage Facility design and Cost Estimate.

More detailed interpretations and conclusions for the Bronson Slope property is 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.

4.0 Introduction

4.1 Terms of Reference

Moose Mountain Technical Services (MMTS) has been commissioned by Skyline Gold Corporation (SGC) to update the Preliminary Assessment (PA) for their Bronson Slope property, which is located in North West British Columbia, Canada. The previous PA, produced by Leighton Asia Limited, was based on a 2008 Resource model and Resource Estimate Technical Report by Giroux and Burgoyne. These Technical Reports evaluated the gold, silver, copper, and molybdenum resources of the deposit. After this work, SGC commissioned evaluation of the magnetite resources within the same deposit. Giroux and Burgoyne produced a new resource estimate for the magnetite. The combined resource model is now the basis of this updated PA by MMTS. A summary of the pertinent previous Technical Reports follows, with other earlier reports listed in the References section. These reports have been filed on SEDAR.

- "Mineral Resource Estimate — Bronson Slope Deposit", April 30, 2008, G. H. Giroux, P. Eng. MASc, and A. A. Burgoyne, P.Eng., M.Sc., .
- "Magnetite Mineral Resource Estimate – Bronson Slope Project" January 28, 2010, G. H. Giroux, P. Eng. MASc, Arnd Burgert P. Geo. B.Sc, and A. A. Burgoyne, P.Eng., M.Sc
- "Preliminary Economic Assessment with Mine Plan and Cost Estimate for Skyline Gold Corporation Vancouver, BC on the Bronson Slope Property"- Leighton Asia Limited, 6 March 2009

This Preliminary Assessment update is intended to update the basic mining and processing conceptual design developed in the Leighton report and add incremental processing infrastructure, costs and revenues for the magnetite portion of the project. This report combines components of the two above mentioned Resource Estimate - Technical reports and utilizes much of the Leighton study and report. MMTS has updated the mine plan, processing, infrastructure, production schedule, cost estimates and financial model to suit the combined mineral resources for the project. This Technical Report has been compiled to be NI 43-101 compliant at a Preliminary Assessment or scoping level of accuracy. The results are an estimate of the project's potential, and indicates areas where more work is required to refine the estimate. The estimate is based on the parameters and assumptions as listed at the time of this report. Costs are based on typical values for similar projects, and market prices and volumes are speculative and based on the assumptions and projections as noted. The forward looking degree of accuracy of these parameters, assumptions, and projections is beyond the ability of MMTS and should not be inferred.

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. Inferred resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Measured or Indicated Resources or as Mineral Reserves.

4.2 Site Visits

Site visits for Qualified Persons for the previous Technical Reports are documented in the respective Technical Reports. For this report Jim Gray P Eng visited the site on July 5th, 2010.

5.0 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, Indicated and Inferred Resources** however inferred resources are minor.

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.*

Future Pre-Feasibility or Feasibility level studies, if warranted will use more comprehensive information and as such will be a more accurate measure of a project's economic viability than this 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 and are deemed to be typical and suitable for the Bronson Slope Project. 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 realized or that the economic results projected herein will be achieved.

Information on the items set out in sections 6 to 19 of this Report can be found in two previous independent Technical Reports 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 second technical report is titled "Magnetite Mineral Resource Estimate – Bronson Slope Deposit For Skyline Gold Corporation, Vancouver, BC on the Bronson Slope Property" dated January 28, 2010, authored by and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc. and G. H Giroux and Arnd Burgert, P.Geo., B.Sc. of Arnd Burgert Consulting Ltd, all three independent Qualified Persons as defined by NI 43-101. This Technical Report was posted to SEDAR on March 5, 2010 (www.sedar.com). As such Burgoyne and Burgert are considered to be 'Other experts' to this report. Section 19 from these two reports has been combined in Section 19 of this report. Giroux has reviewed this edited information and has signed as a QP to this report.

Information on the metallurgy, processing, site infrastructure, and conceptual mining method can be found in "Technical Report – Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation Vancouver, BC on the Bronson Slope Property", dated March 6, 2009 and posted to SEDAR. This report was prepared by J. A. R. Lawrence, MAusIMM (#209746) and V. Seen, MAusIMM of Leighton Asia Limited ("LAL"). This Technical Report can be viewed at www.sedar.com. This work was accepted in this previous study and MMTS has acceptable it at a scoping level of study and has adapted it where applicable in this work. As such Leighton Asia Limited is considered 'Other Experts' to this study. To MMTS'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.

Guidance on 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.

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.

These reports were deemed suitable by Leighton in their previous PA and are utilized in this PA update. A detailed review of these reports has not been completed but is recommended prior to higher levels of study. As such "Piteau Engineering Ltd' is considered 'Other experts' to this study.

While MMTS has reviewed all of the information provided by SGC, and their consultants, particularly on land tenure, metal pricing, marketing, transportation and local infrastructure and believes the information to be reliable, MMTS has not conducted an independent in-depth investigation to verify its accuracy and completeness, MMTS feels it is reasonable at a scoping level at the time of this study. Much of this information is time sensitive and should be updated as required, and reviewed in more detail, for higher levels of study.

The land tenure position, as described in the above Technical Report of May 10, 2007 and authored by G. H Giroux, and A. A. Burgoyne, P.Eng., has not been updated for this report. Land tenure can change over time, and MMTS recommends that any evaluations of this property should verify the land tenure position at the time of the evaluation.

6.0 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. 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 centered on 13105' West Longitude and 5640' 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.

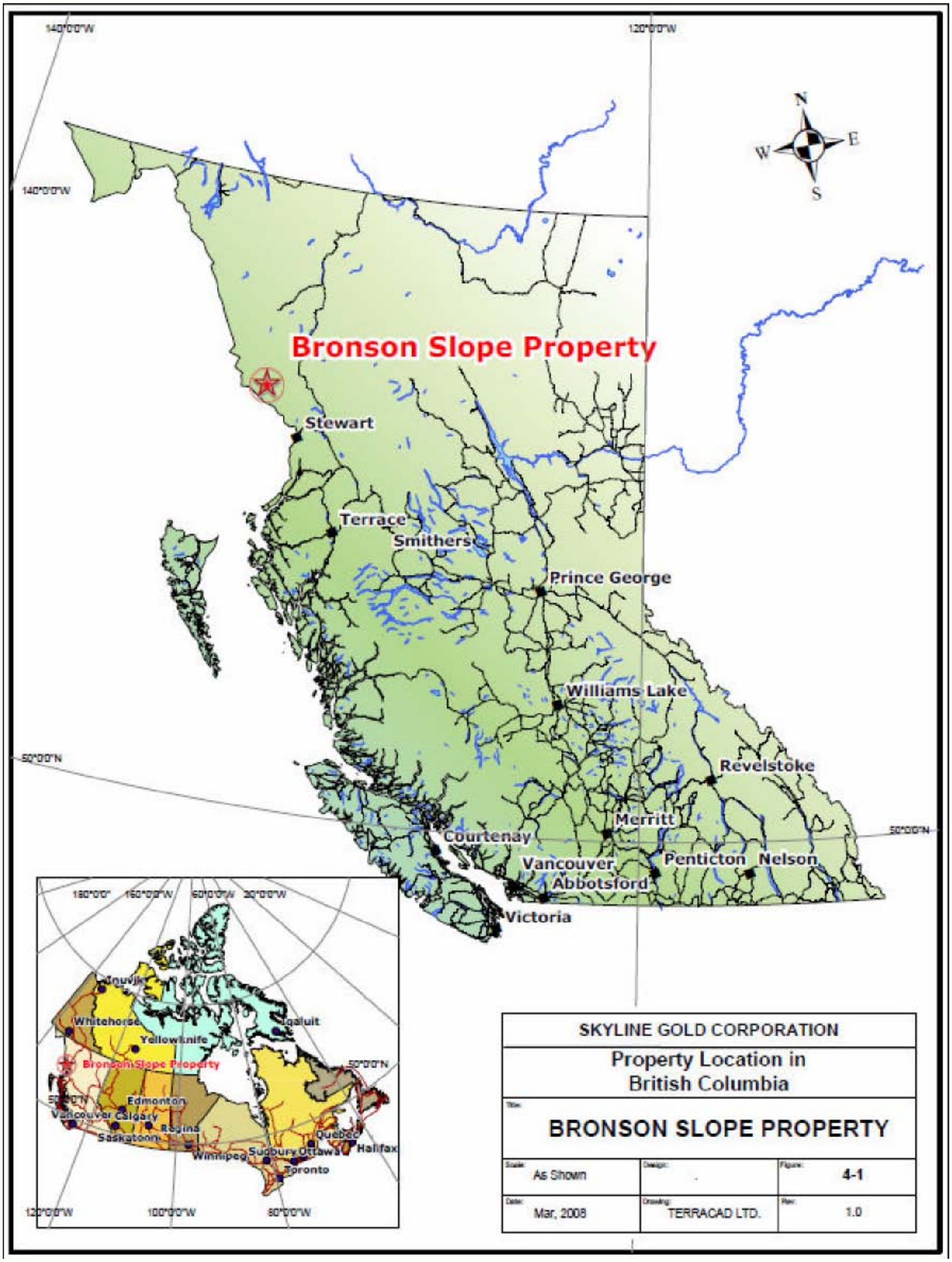


Figure 6-1 Bronson Slope Location

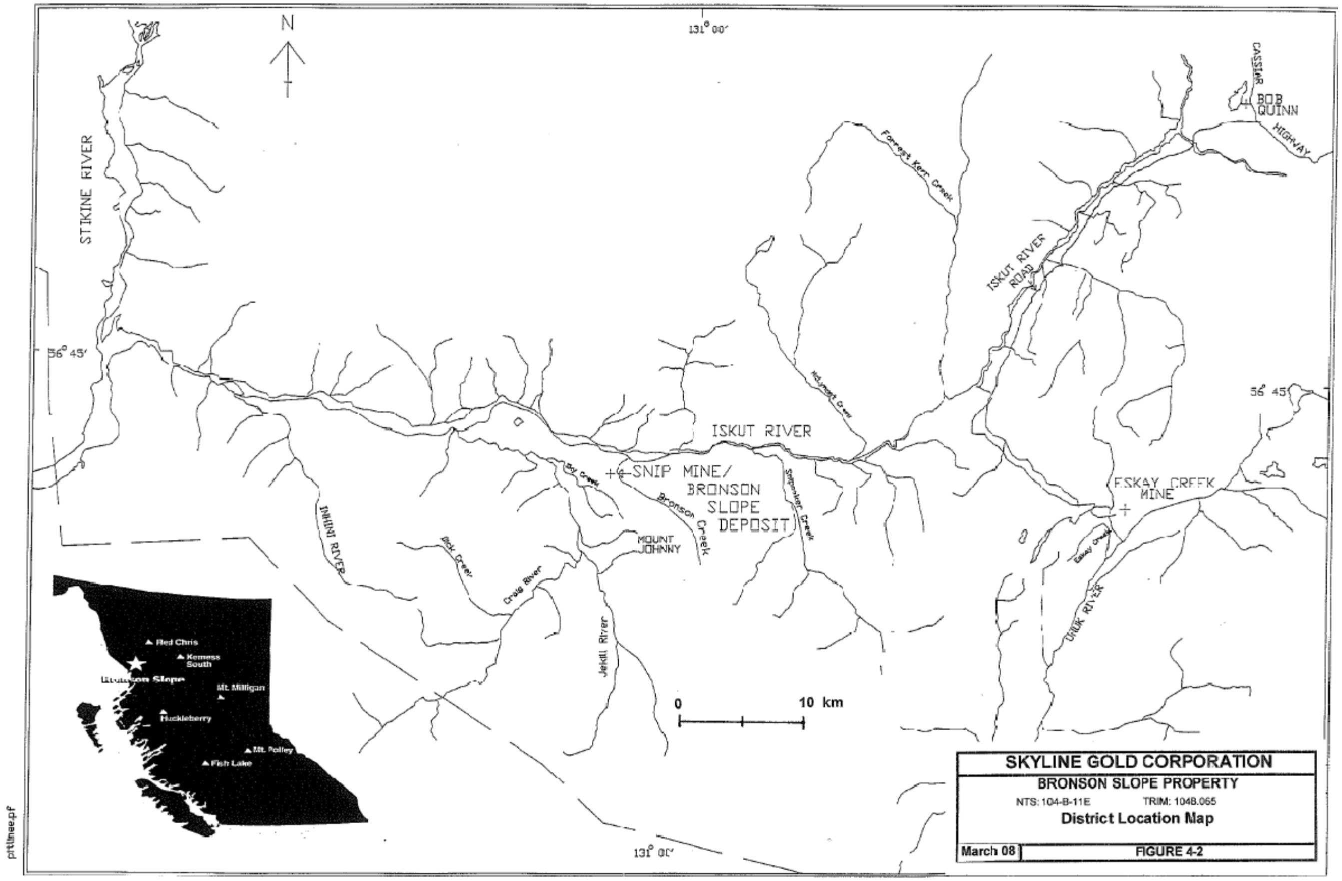


Figure 6-2 Bronson Slope Location

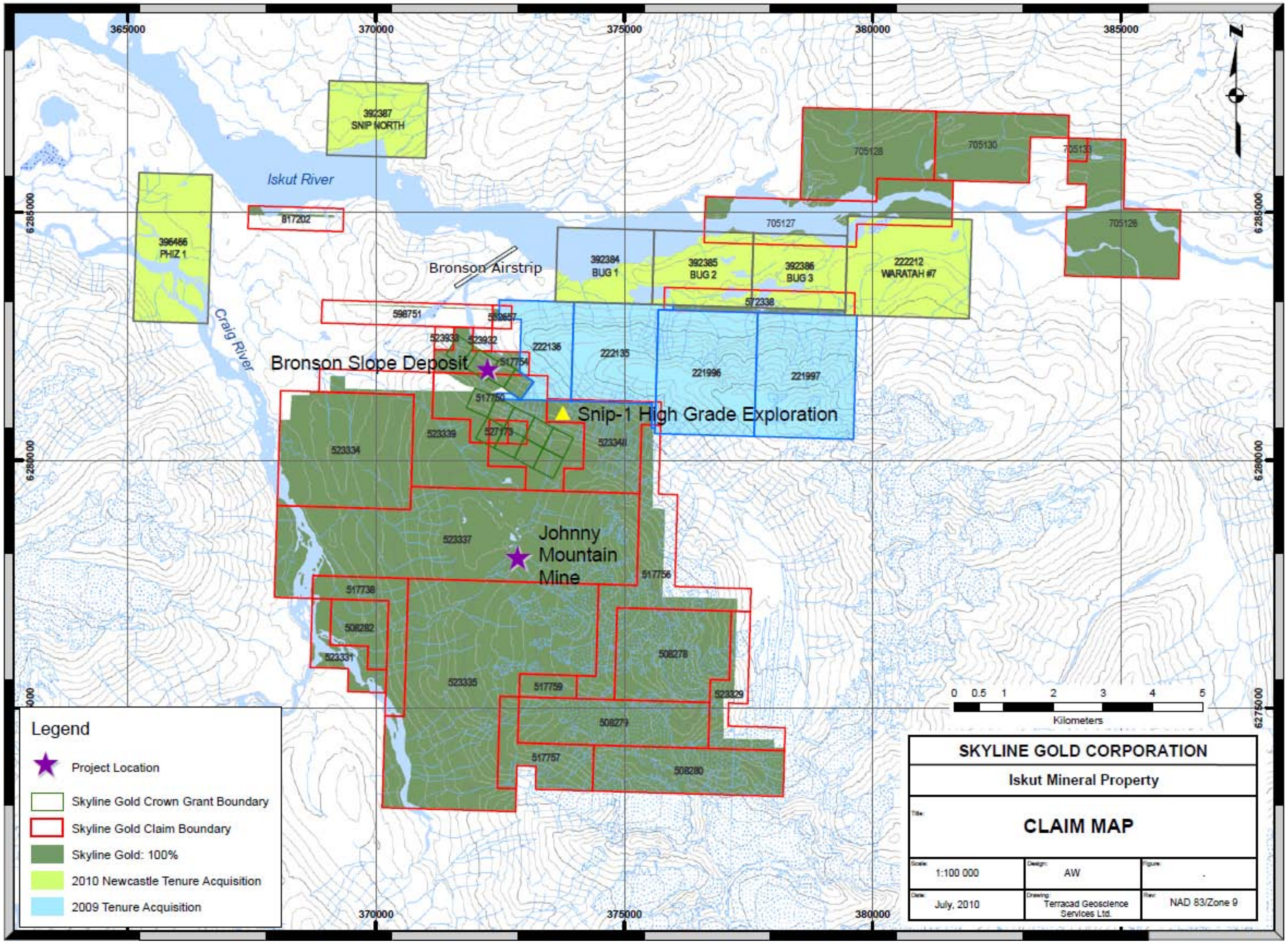


Figure 6-3 Property Boundaries

The property consists of BC Mineral Claim Tenures 517750, 517754, 523932, and 523933, and 6 Crown Granted Mineral Claims, totaling 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
HANDEL	221996	October 19, 2011
RAVEL	221997	October 19, 2011
CHOPIN I	222135	October 19, 2013
CHOPIN II	222136	October 19, 2013
jmx	508278	December 31, 2015
jmx2	508279	December 31, 2015
jmx3	508280	December 31, 2015
jmx4	508282	December 31, 2015
BURNIE2	517738	December 31, 2015
BRONSON	517750	December 31, 2015
BRONSON2	517754	December 31, 2010
SKYFILL1	517756	December 31, 2015
BURNIEADD	517757	December 31, 2015
BURNIEADD1	517759	December 31, 2015
HIGHADD	523329	December 31, 2015
JEKYLLADD	523331	December 31, 2015
	523334	December 31, 2015
	523335	December 31, 2015
	523337	December 31, 2015
	523339	December 31, 2015
SNIP 1	523348	March 1, 2020
KATYADD	523932	December 31, 2010
CGADD	523933	December 31, 2010
CG1	527173	December 31, 2010
BRONSON SLOPE FRACTION	552657	March 1, 2020
RIVER	572338	December 21, 2010
SNIPPED	598751	February 5, 2011
SNIPPAKER-1	705126	February 1, 2011
GOLD COUNTRY	705127	February 1, 2011
FINAL APPROACH	705128	February 1, 2011
DESCENT	705130	February 1, 2011
BLOCK	705133	February 1, 2011
ISKUT GOLD	817202	July 12, 2011
Crown Grants	**Lot Number**	**Taxes Due Date**
Red Bluff	2857	July 2, 2011
Homestake	2858	July 2, 2011
Red Bird	2859	July 2, 2011
Mermaid	2860	July 2, 2011
El Oro	2862	July 2, 2011
Golden Pheasant	2864	July 2, 2011

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.

6.2 Environmental Issues

The authors are not aware of any environmental issues or liabilities that affect the property and have 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 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.

7.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

Reference is made to "Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation on the Bronson Slope Property", dated March 6, 2009 and posted to SEDAR on March 6, 2009. This report was prepared by J. A. R. Lawrence, MAusIMM (#209746) and V. Seen, MAusIMM of Leighton Asia Limited.

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 oC, and lowest in January falling to -15 oC. The highest temperature recorded on site over the 5 year period was 31 oC and the lowest temperature recorded was -32 oC. 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.

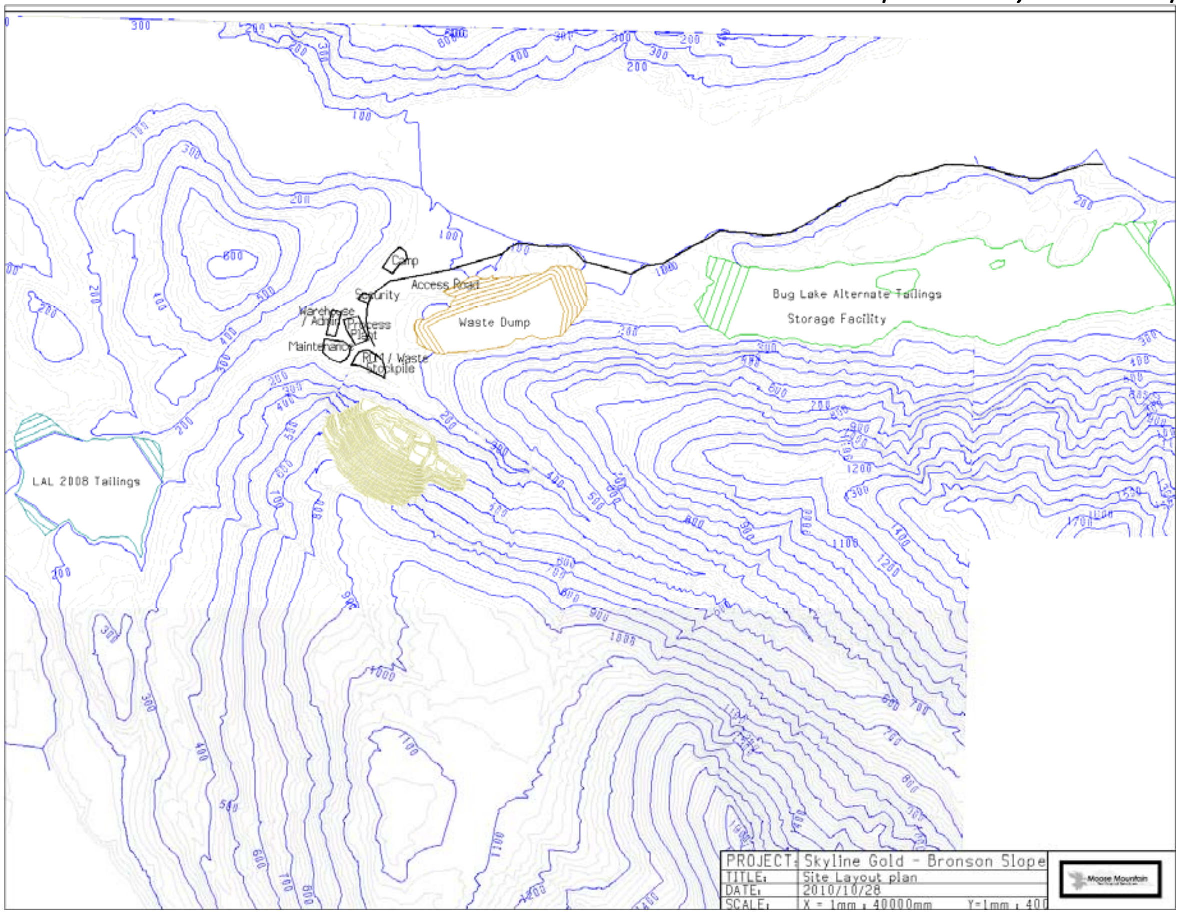


Figure 7-1 Site Layout at End of Mine

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

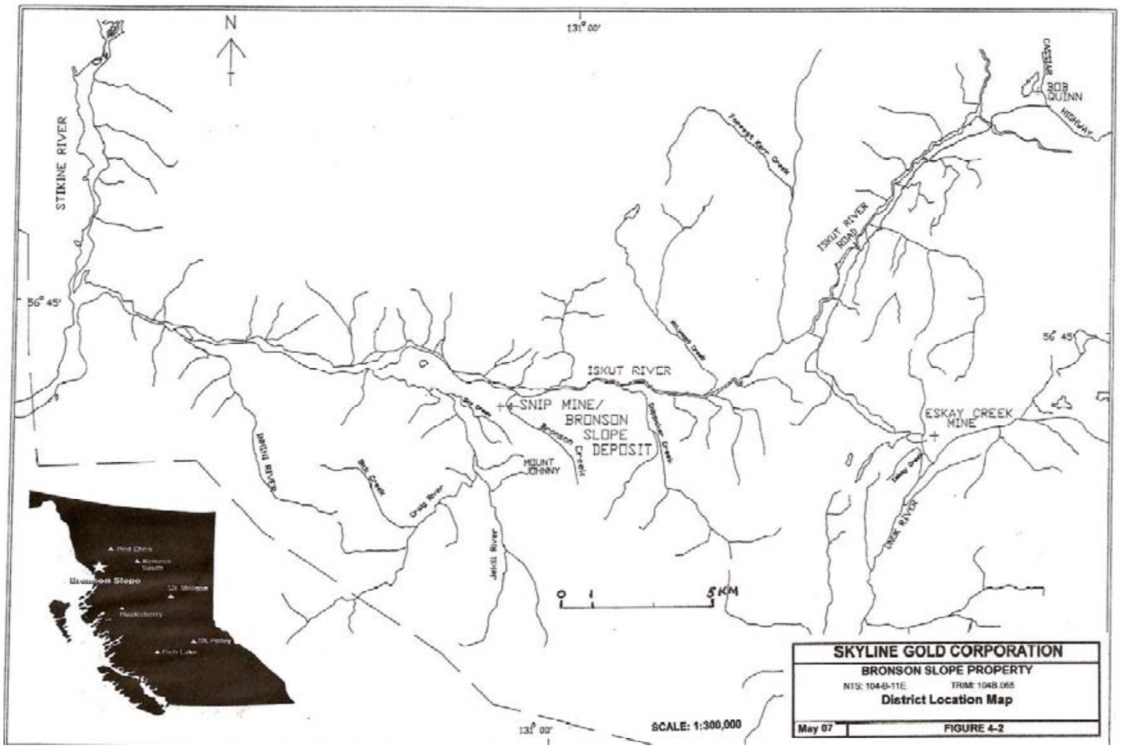


Figure 7-2 Site Location With Respect to the Iskut River

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 summarized 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	lm	$10	$128,000
6	Drainage Applicances				
	600 mm CMP	1,400	lm	$150	$210,000
	800 mm CMP	150	lm	$275	$41,250
	1000 mm CMP	70	lm	$375	$26,250
	1200 mm CMP	50	lm	$480	$24,000
	1600 mm CMP	40	lm	$670	$26,800
	2200 mm CMP	20	lm	$1,420	$28,400
	2400 mm CMP	20	lm	$1,650	$33,000
7	Open Bottom Arches				
	1.8m - 3.0m	60	lm	$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	lm	$10,000	$120,000
	15 m Bridge - 4 each	60	lm	$10,500	$630,000
	24 m Bridge -1 each	24	lm	$11,000	$264,000
	35 m Bridge -1 each	35	lm	$9,000	$315,000
	42 m Bridge -1 each	42	lm	$9,500	$399,000
	65 m Bridge -1 each	65	lm	$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 maneuvering 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 (100W MT lots), the trucking costs are CAD40.00/t to Port Stewart.

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 previous Leighton Report used labour force statistics for February 2006 which 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 from 2009 have shown an increase in the unemployment rate in BC to 7.2%, mostly due to the current financial downturn.

In the mining industry there is a general lack of skilled personnel. Direct employment in BC's mining industry totaled 7,345 during 2006, compared to 7,071 in 2005. Salary and benefits totaled $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 7-4 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 7-4 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.

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 previous 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 previous tailings area and the water diversion structure diagram are illustrated in Figure 7-4.

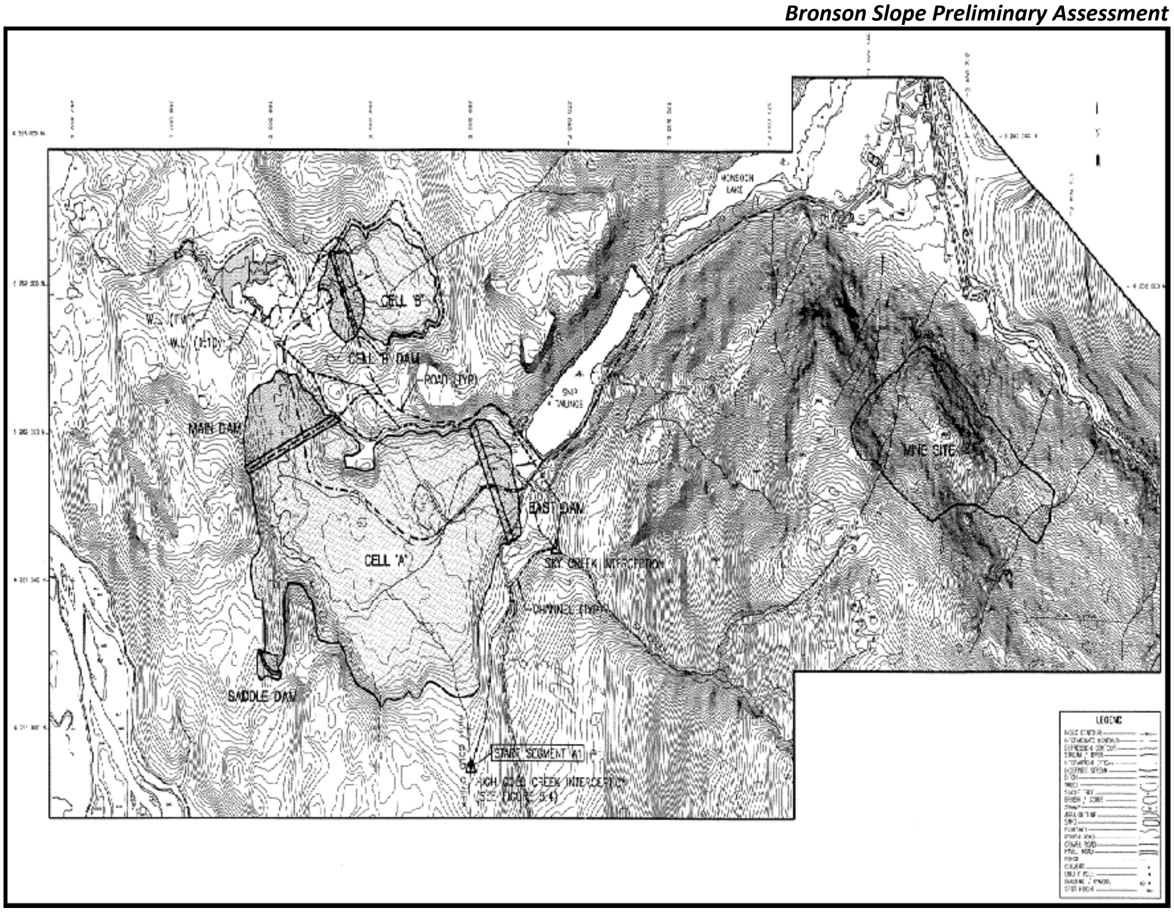


Figure 7-4 Previous Proposed Tailing Storage Facility Area

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.*

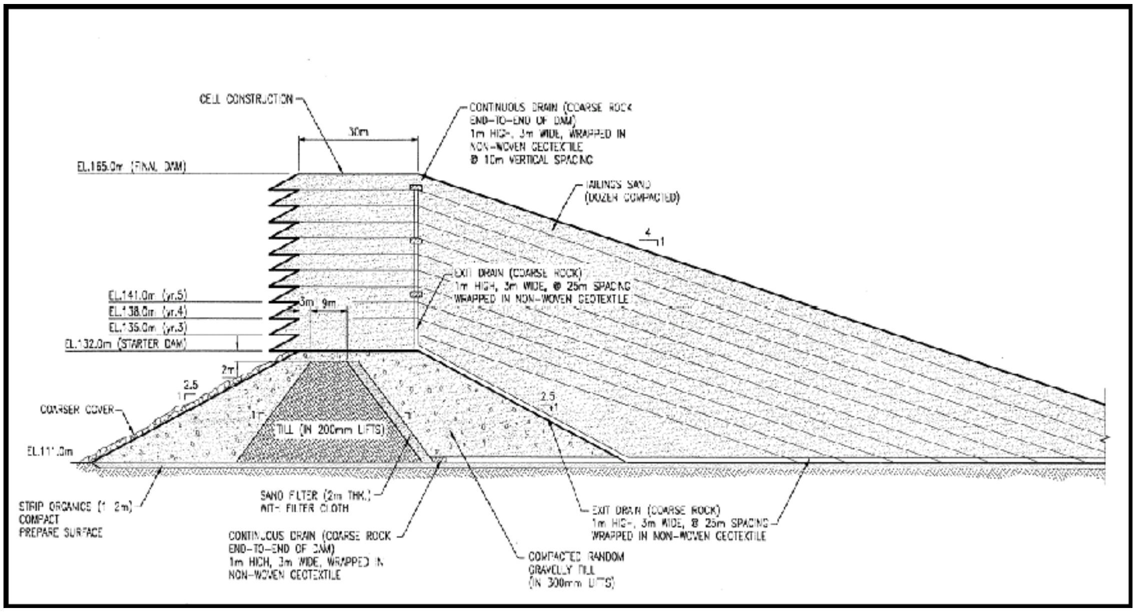


Figure 7-5 A Cross Section of the Main Dam (Piteau 1997)

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 behavior will be used to confirm assumptions about tailings behavior 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 Leighton PEA proposed tailing storage in the Sky Creek valley which is approximately 2.5km over relatively flat ground southwest of the plant site. However due to the increased tonnages, from the introduction of magnetite and the updating of commodity prices, it is necessary to re-evaluate locations for tailing storage for the PEA update for Bronson Slope. Based on an average tailings density of 1.75t/m³, the estimated volume required for tailing storage is 125 million cubic meters. The Sky Creek valley tailings storage was designed to accommodate 51.7 million cubic meters, and with a footprint of roughly 1 million square meters which requires the dam heights being raised by over 70m. This would result in dam crests surpassing the elevation of the surround topography on all but the southeastern corner of the tailings area and therefore the Sky Creek valley will not accommodate the full volume of the tailings from the expanded pit.

The primary alternative site for tailing storage is the Bug Lakes area, approximately 4km northeast of the plant site, beyond the proposed Triangle Lake waste dump. The Bug Lakes area is naturally constrained on two sides, with a high mountain ridge on the South (see left side of **Error! Reference source not found.**Figure 7-6 below) and rugged hills to the North, which separate it from the main flow channel of the Iskut River.

Figure 7-6 View of Bug Lakes Area; Looking West

For full capacity the tailing storage site requires two earthworks embankment dams at the western and eastern ends of the Bug Lakes area. Each dam has a 4:1 downstream slope and is built in raises, with a final elevation of 235m above sea level. The Western dam has a maximum height of 110m, averaging approximately 100m in height along its 750m crest width, while the eastern dam has maximum height of 85m, averaging approximately 50m in height along its 950m crest width. A rough schematic on the proposed Bug Lakes Tailing Storage can be seen in Figure 7-7 below.

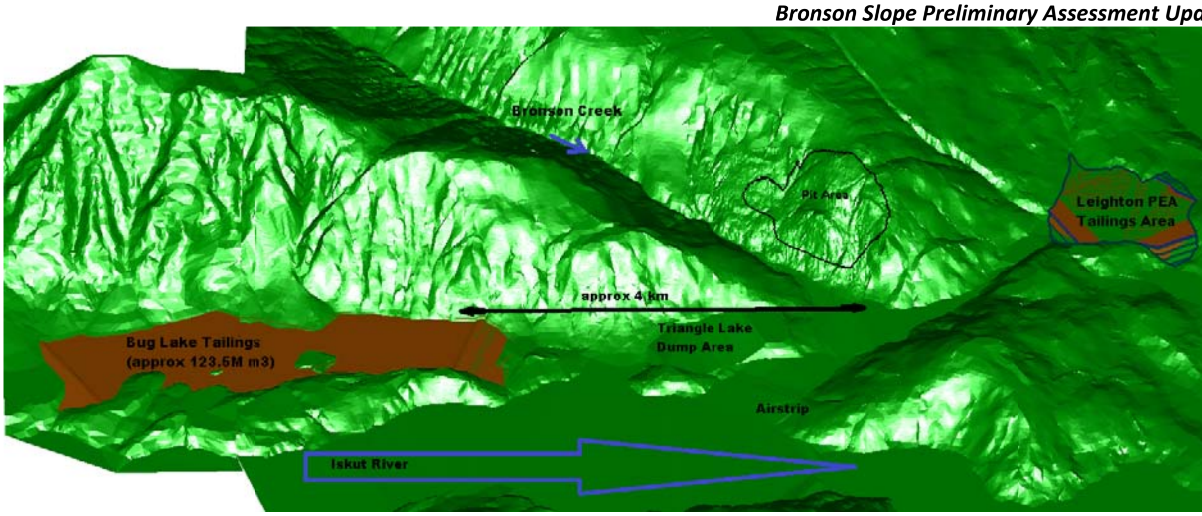


Figure 7-7 Alternate Tailing Pond Location

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 re-handled 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 Leighton report dump design has been raised by 40 meters to a final elevation of 240m to accommodate the increased waste from the larger mine design. A conceptual waste dump design has been provided in Figure 7-8.

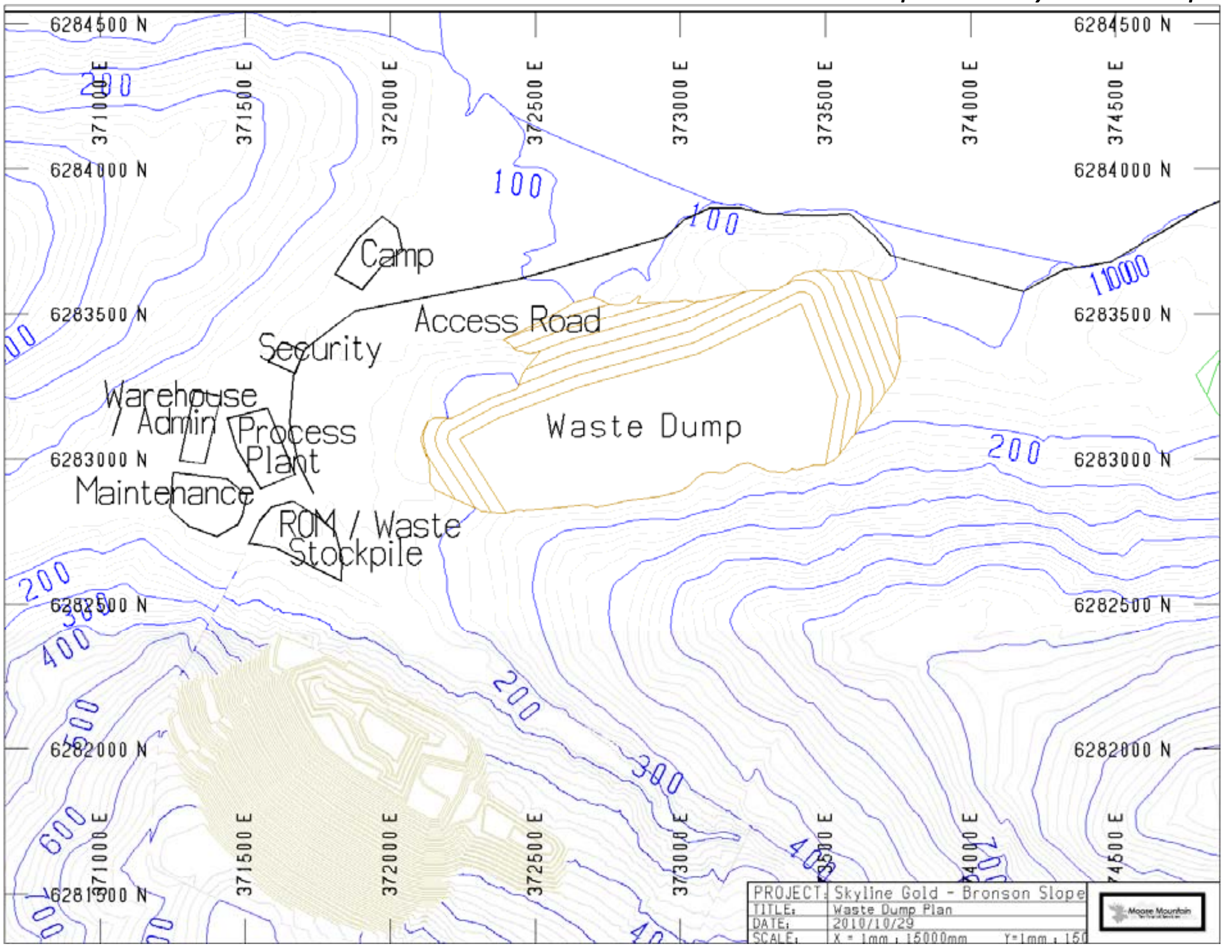


Figure 7-8 Conceptual Waste Dump

The waste storage facility will also be segregated into below cut-off grade mineralized rock (selected based on a marginal NSR cutoff and stockpiled as low grade for potential future processing) and waste (with no potential economic value). It is anticipated the marginal mineralized 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 re-handle. Based on a NRS marginal grade cutoff of USD 7.00/t approximately 40% of the waste storage facility may be considered for re-handle at a later date. No economic consideration for below cut-of low grade re-handle 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 7-9. The surface tenure at this site is currently held by Barrick Gold Corp.*

Figure 7-9 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.

8.0 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.. References contained within the excerpt are as given in the original report.

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 meter HQ diameter core drilling program. The total 2006 exploration expenditures were $1.45 million*

8.3 Historical Mineral Resource Estimate

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 offour 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 was 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 modeling 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 modeling 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**	Tonne s	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**
			In.	**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	Dec 16 96	Kriging100 m	Meas + Ind	74.5	0.559	2.65	0.198		9.1
56 ddh			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	May 1 97	Kriging100m	Meas + Ind	85.9	0.59	3.05	0.163		8.91
63 ddh			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 modeling 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.

9.0 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.. References contained within the excerpt are as given in the original report.

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).

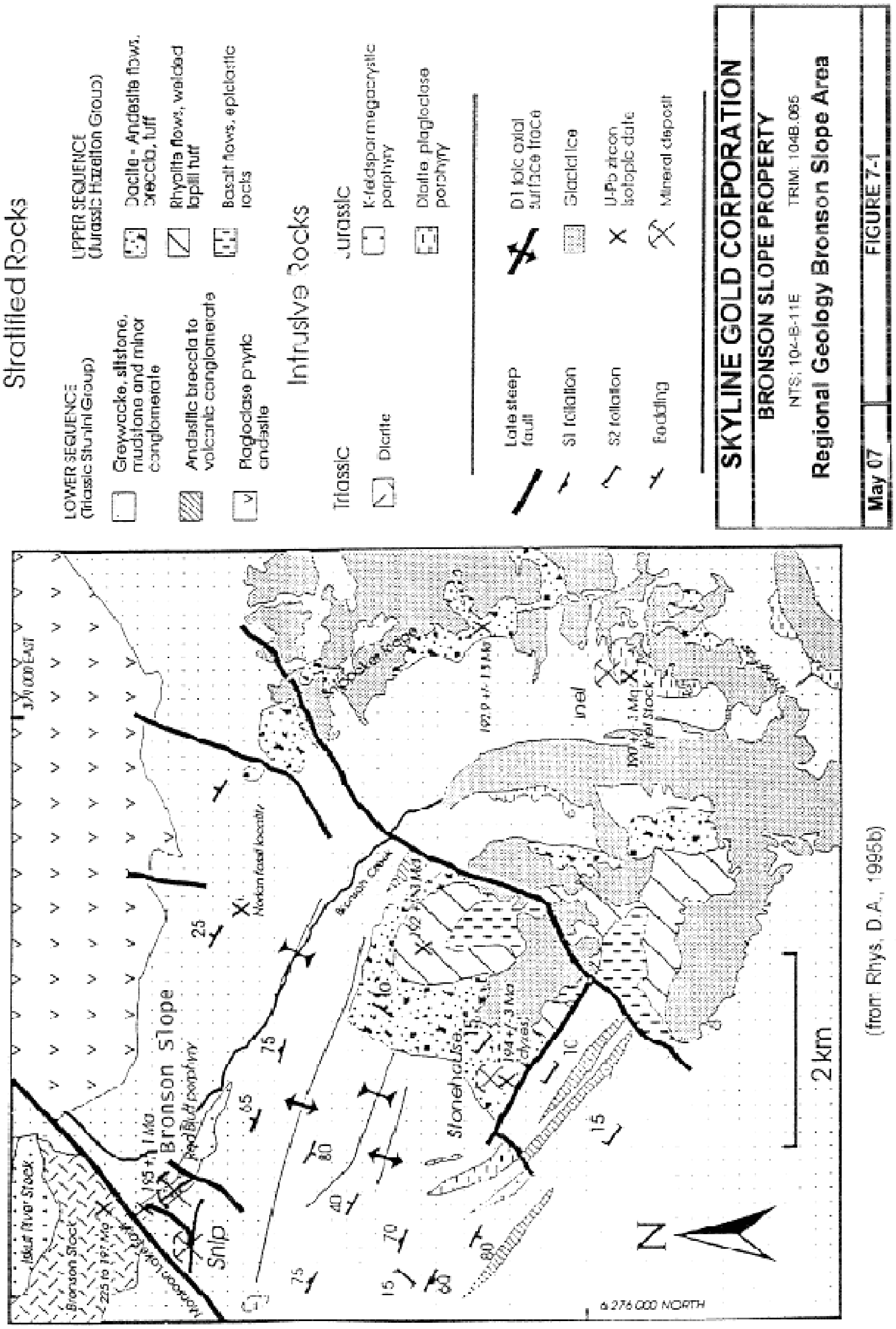


Figure 9-1 Regional Geology

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).*

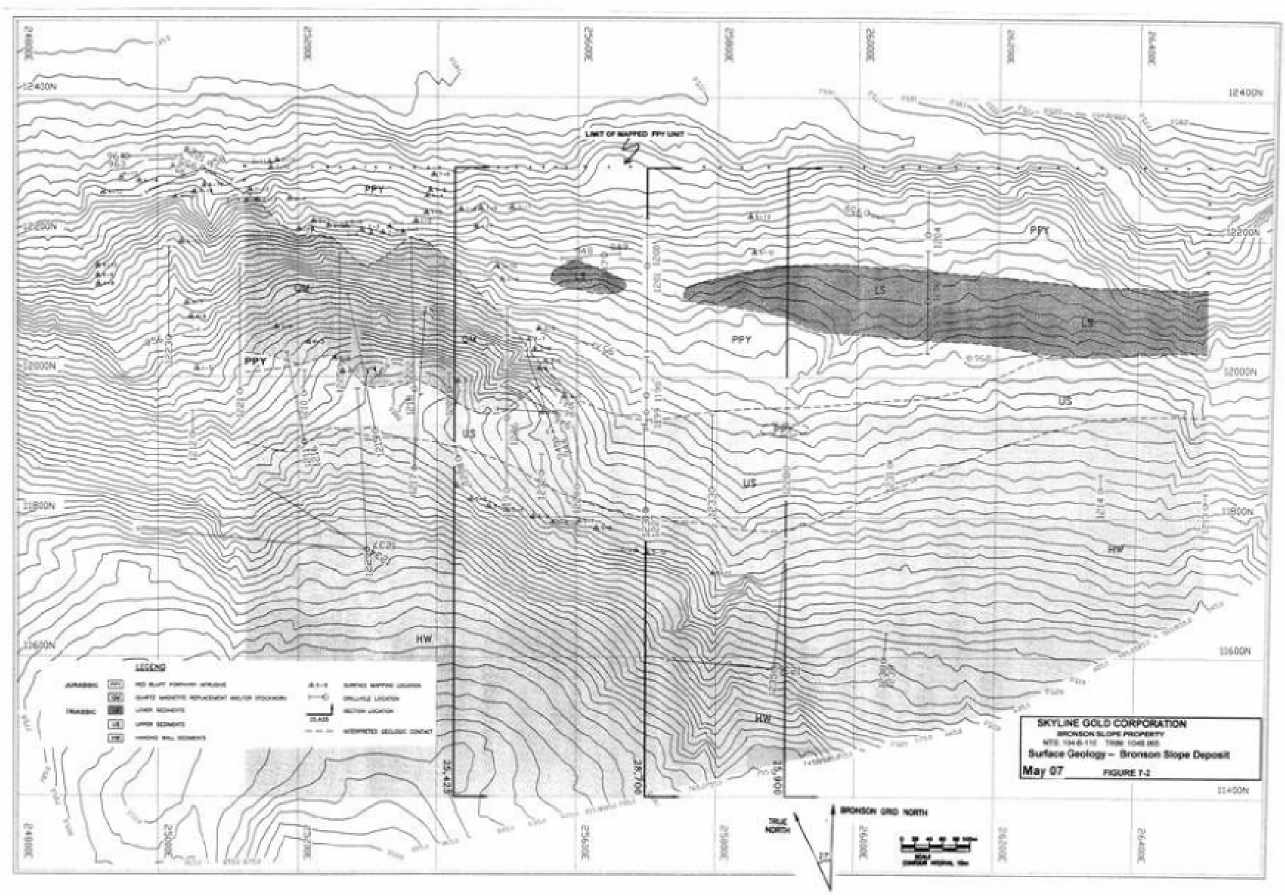


Figure 9-2 Surface Geology (Piteau, 1997)

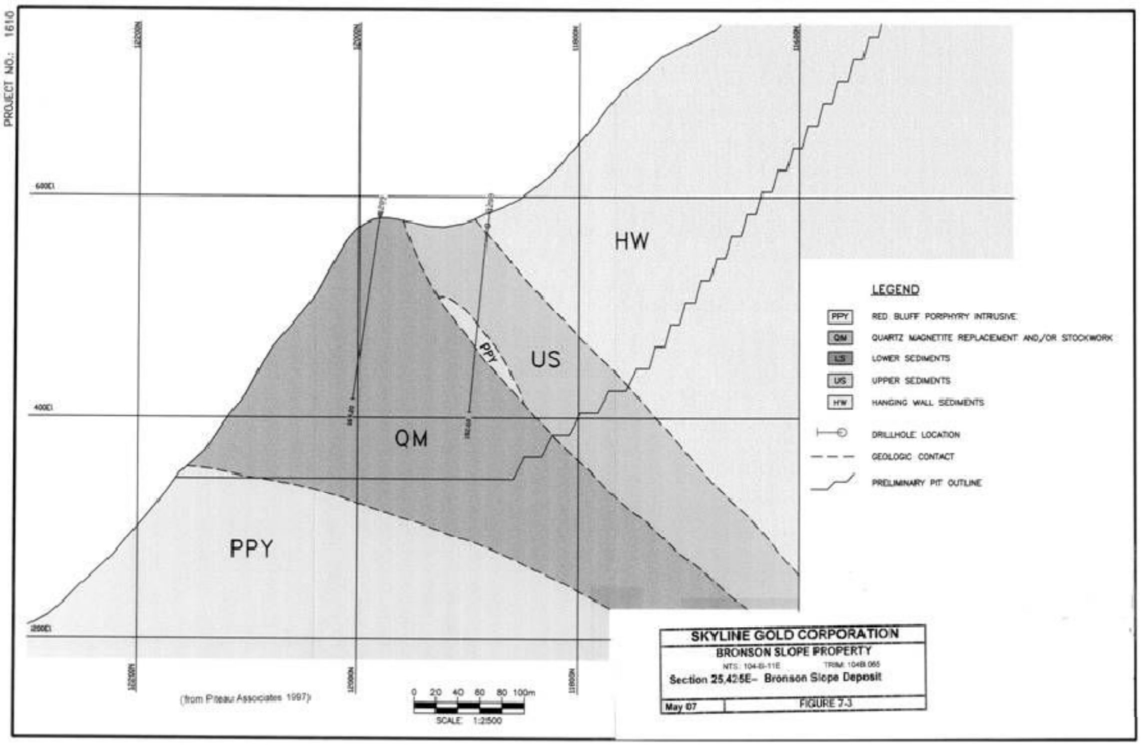


Figure 9-3 Deposit Section 25425E (Piteau, 1997)

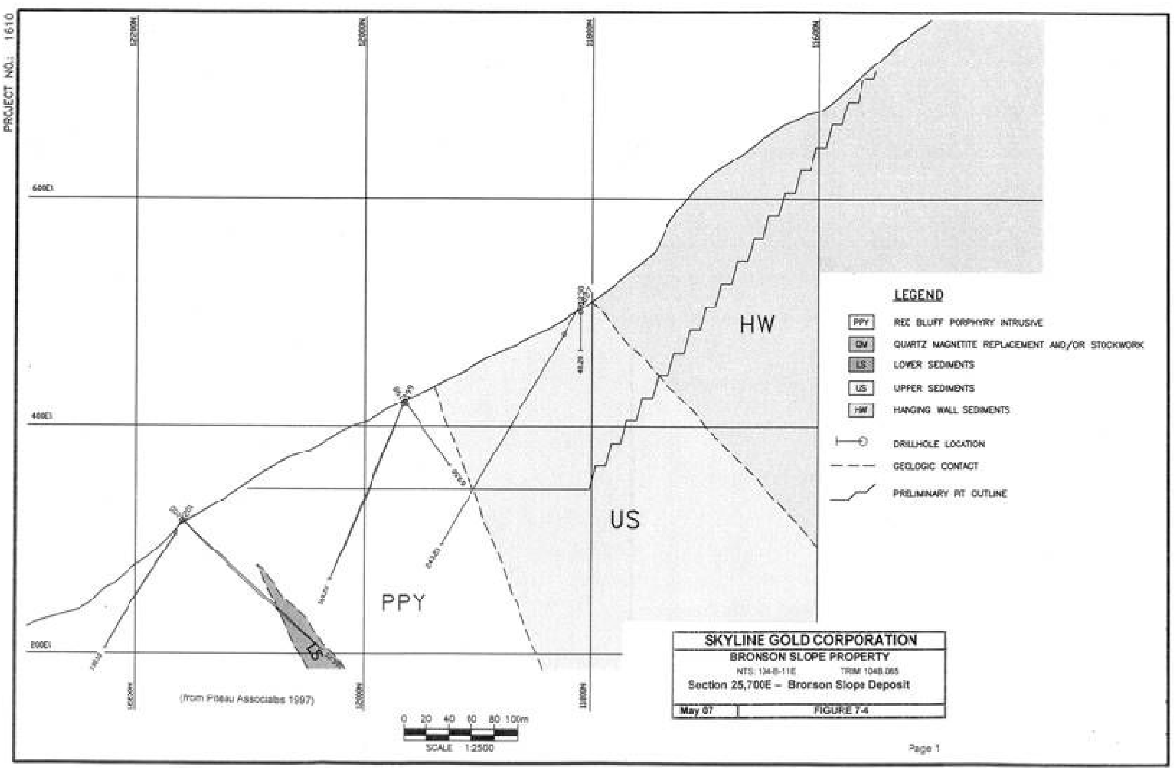


Figure 9-4 Deposit Section 25700E (Piteau, 1997)

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(of the referenced report)** 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 through time.

A 25 to 50 meter 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°/70° NW) e.g., Stonehouse Gold Deposit, Johnny Mountain*
- *southwest dipping shears (120°/45° SW) - e.g., former mined Snip Gold Deposit, and,*
- *northeast dipping shears (130°/45° 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.

10.0 Deposit Type

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.

11.0 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.

The following Item 11 — Mineralization has been extracted from Section 9 — Mineralization within the previous Technical Report titled "Magnetite Mineral Resource Estimate – Bronson Slope Deposit For Skyline Gold Corporation,Vancouver, BC on the Bronson Slope Property" dated January 28, 2010, authored by and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc.and G. H Giroux and Arnd Burgert, P.Geo.,B.Sc. of Arnd Burgert Consulting Ltd, all three independent Qualified Persons as defined by NI 43-101. This Technical Report was posted to SEDAR on March 5, 2010 (www.sedar.com).

*The quantity and tenure of magnetite mineralization defined by the 2009 core-sampling program in certain drill holes of the Bronson Slope deposit is discussed in **Item 11**. Magnetite mineralization is spatially associated with the Quartz Magnetite Breccia and the Red Bluff Porphyry Intrusive units. Magnetite mineralization also occurs adjacent to the Quartz magnetite unit within the Lower Sediments unit.*

12.0 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. There has been no exploration on the Bronson slope deposit since the 2008 Resource estimate. References contained within the excerpt are as given in the original report.

*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 meters 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 meters 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 meters. 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 kilometer 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. 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 totaling 2332 meters, 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 meters and 19 drill holes over 1494.5 meters 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 meters 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 meters 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.

13.0 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. There has been no exploration on the Bronson slope deposit since the 2008 Resource estimate. References contained within the excerpt are as given in the original report.

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.

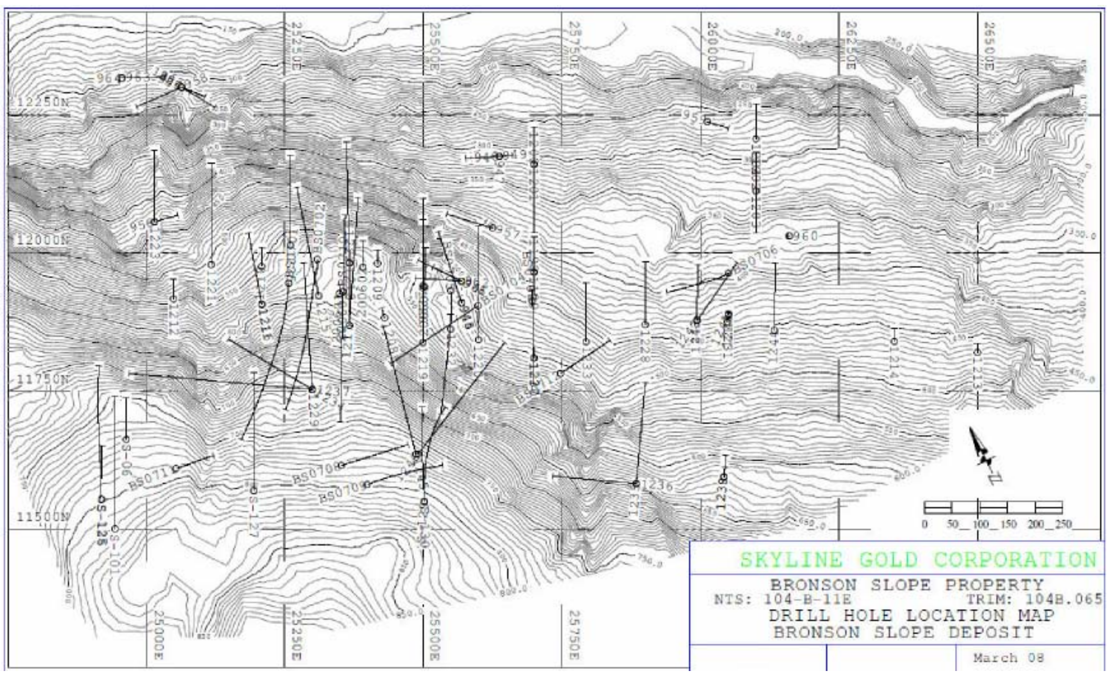


Figure 13-1 Drill Hole Locations

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 meters over 92 core drill holes. Drilling by SGC in 1988 and 1993 through 1997 involved a total of 12,153 meters over 63 core drill holes. Drilling in 2006 involved a total of 562 meters over 4 holes. Drilling in 2007 involved a total of 3936 meters 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 Deposit

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.

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 Pam icon 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 totaling 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.

2006 SGC Program

During September and October 2006 SGC completed 561.6 meters 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 meters, was re drilled by hole BS0603 from essentially the same drill site. BS0603, drilled to a depth of 270 meters 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 meters is an in fill hole between holes 1209 and 1222 again drilled in 1994 and 1995.

BS0604, drilled to a depth of 122 meters, is a near twin to hole 1226 drilled in 1996. In hole 1226 the top 117.5 meters weight averages 0.42 g/t gold and 0.208 % copper versus the top 117 meters 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 meters 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

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
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
*BS060	31.0	25348.9	11926.4	594.2	3.5	-60
*BS060	138.6	25391.1	11974.0	598.6	0.5	-70
*BS060	270.0	25355.7	11929.8	590.4	1.5	-59
*BS060	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"*

14.0 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. There has been no sampling on the Bronson Slope deposit that would affect the resource estimate since the above mentioned 2008 report.

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.

15.0 Sample Preparation, Analysis and 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. References contained within the excerpt are as given in the original report. 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 analyzed in percent by Group 7AR (aqua regia) (HCL-HNO3-H2O) method using 1 gram of sample pulp, diluted to100ml and analyzed 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 labeled with aluminum 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**.*

The following section 13 is taken from the technical report titled "Magnetite Mineral Resource Estimate – Bronson Slope Deposit For Skyline Gold Corporation,Vancouver, BC on the Bronson Slope Property" dated January 28, 2010, authored by and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc.and G. H Giroux and Arnd Burgert, P.Geo.,B.Sc. of Arnd Burgert Consulting Ltd, all three independent Qualified Persons as defined by NI 43-101. References within the excerpt are as given in the original report. This Technical Report was posted to SEDAR on March 5, 2010 (www.sedar.com).

13.1 Drill Core

All drill core samples were submitted to the Assayers Canada sample preparation lab in Telkwa, BC, where samples were logged according to the security protocol described in Item 13.3.1. Samples were then dried, weighed, crushed, and split using a riffle splitter. The coarse reject fraction was placed into secure storage at the Assayers Canada Telkwa facility, while the split fraction was forwarded to the Assayers Canada facility in Vancouver for pulverization.

Once crushed splits were received at Assayers Canada's Vancouver facility, the routine procedure involved pulverizing for 90 seconds and sieving through a 150-mesh size screen. A riffle splitter was then used to obtain a representative 200g split. The split pulp samples were forwarded to Met-Solve Metallurgical Laboratories in Burnaby, BC (Met-Solve) for analysis.

An orientation program was performed by Assayers Canada and Met-Solve to determine the pulverization time for optimal magnetite liberation and the sub sample size for optimal magnetite recovery. The orientation study is described in Item 13.1.1.

The magnetic separator used for the magnetite analyses is a Davis Tube apparatus. The analytical procedure involves first weighing out a 20-gram dry pulp sample into a beaker and adding enough water to wet and cover the dry pulp sample. The discharge end of the Davis Tube is clamped off, and the Davis Tube filled with enough water to cover the magnet poles. The magnet is turned on and the slurry sample washed into the Davis Tube. The discharge hose clamp is unclamped to regulate a steady flow of water through the Davis Tube, and the tube is then agitated for two minutes, allowing the slimes or cloudiness to be washed out of the tube. Magnetic material is attracted and held fast in the magnetic zone between the two magnet poles. After two minutes the agitator is stopped, the magnet turned off, and the magnetic fraction flushed to the bottom of the tube. The magnetic material is then collected through the discharge tube, dried, and weighed to determine the percentage of magnetic material.

13.1.1 Magnetite Analyses Orientation Study

Met-Solve prepared laboratory standard material from a sample of Bronson Slope drill core that had been collected for a metallurgical study in 2008. The material was used to prepare fifty-six standard samples to determine the pulverization time for optimal magnetite liberation and the sub sample size for optimal magnetite recovery. Of the fifty-six standard samples, half were pulverized for each of 90 seconds and 120 seconds. Each of these two subsets was then split into three additional subsets for analysis of 10g, 15g, and 20g samples.

The shorter (90 second) pulverization time consistently gave very slightly higher assay values. The standard deviation among assay values of 10g sub samples was higher than the standard deviation of the 1 5g sub sample set. The standard deviation among the 20g set was similar to that among the 15g set.
Based on these data, the 90-second pulverization and 15g sub sample were selected as the routine parameters. However, review of the initial batch of 56 core pulp samples revealed lower than expected repeatability among laboratory duplicates and laboratory repeats, so further analyses used 20g sub samples. Repeatability improved markedly using 20g sub samples, and the larger sub sample size was adopted for all remaining analyses.

The series of ten standard samples from the orientation study subjected to 90-second pulverization of 20g samples is included in the data set used for statistical analysis of laboratory standards, described in Item 14.3.6.

13.2 Pulps

The pulps obtained from Acme had been prepared from drill core in 2006 and 2007 according to the following methodology. The diamond drill core and the respective standards and duplicates were delivered to Acme in Vancouver, BC where laboratory staff assumed the chain of custody of the samples. Acme is an ISO 9001:2000 accredited company using accepted and good quality analytical technology and protocol with respect to current industry standards. The samples were recorded, dried, crushed, and split. The split portion was then ground or pulverized. Standard sample procedure during this process involved pulverization *of split core so that 1 kg is crushed to 70% passing a No. 10 mesh size sieve screen with a 250 gram split pulverized to 95% passing a No. 150 mesh.*

During the 2006 and 2007 programs, Skyline had used a blank standard for gold and five separate standards for gold, copper, molybdenum, and silver that were inserted as rock pulps into the sample chain in the field. Before submitting the series of 2007 pulps for magnetite analyses, Skyline supplied Met-Solve with a series of standard QA/QC pulps for magnetite, including a supply of magnetite blank and two magnetite standards. Assayers Canada then removed all 2007 gold, copper, and molybdenum standards from the sample stream, replacing each with one of the new magnetite standards.

The new series of pulps was then submitted for magnetite analysis following the procedure described in Item 13.1.

13.3 Sample Security

13.3.1 Core Sample Security

For all core samples processed during 2009, Skyline implemented a chain of custody procedure to ensure sample security. Drill core was handled at Skyline's core facility at Bronson Camp where doors were locked when unattended. Once packed in rice sacks, completed samples were stored in a locked sample storage shed behind the core shack. Following sample transport by air, samples were either taken into possession by trusted personnel or held in secure storage until delivered to Assayers Canada where laboratory staff assumed custody. Assayers Canada is an ISO 9001:2008 certified laboratory. Samples were submitted to Assayers Canada in rice sacks sealed with numbered locking security (NLS) ties. As laboratory personnel unpacked each rice sack, the NLS serial number was recorded along with the sample sequence therein contained. This information was sent by Email to the Project Geologist for verification against the data in the sample log.

13.3.2 Pulp Security

Core and sample security, up to the point of the 2007 analyses, is described by Burgoyne and Giroux (2008) as noted in Item 13.4. After the 2007 analyses were performed, secure custody of the sample pulps was retained by Acme until they were shipped to Met-Solve in 2009, where Assayers Canada staff took custody of the pulps.

13.4 Sample Preparation & Analyses 2007, 2006, and Historical Drilling. This is detailed in Burgoyne and Giroux (2008) at www.sedar.com.

The authors are of the opinion that the techniques and analytical methods used by the Skyline 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.

13.5 Site Security and Chain of Custody for 2007, 2006, and Historical Drilling

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). This is detailed in Burgoyne and Giroux (2008).

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 12.3.*

16.0 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. References within this excerpt are as given in the original report. This Technical Report can be viewed at www.sedar.com.

16.1 Quality Control and Quality Assurance Program 2007Program

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, totaling 65 samples, were inserted into the sample chain at different times that were analyzed for the four elements named above. All of these samples were analyzed 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

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 16-2*** *and* ***Figures 16-1 to 16-3****.*

For gold duplicate sample results there is obviously some natural variance as given in

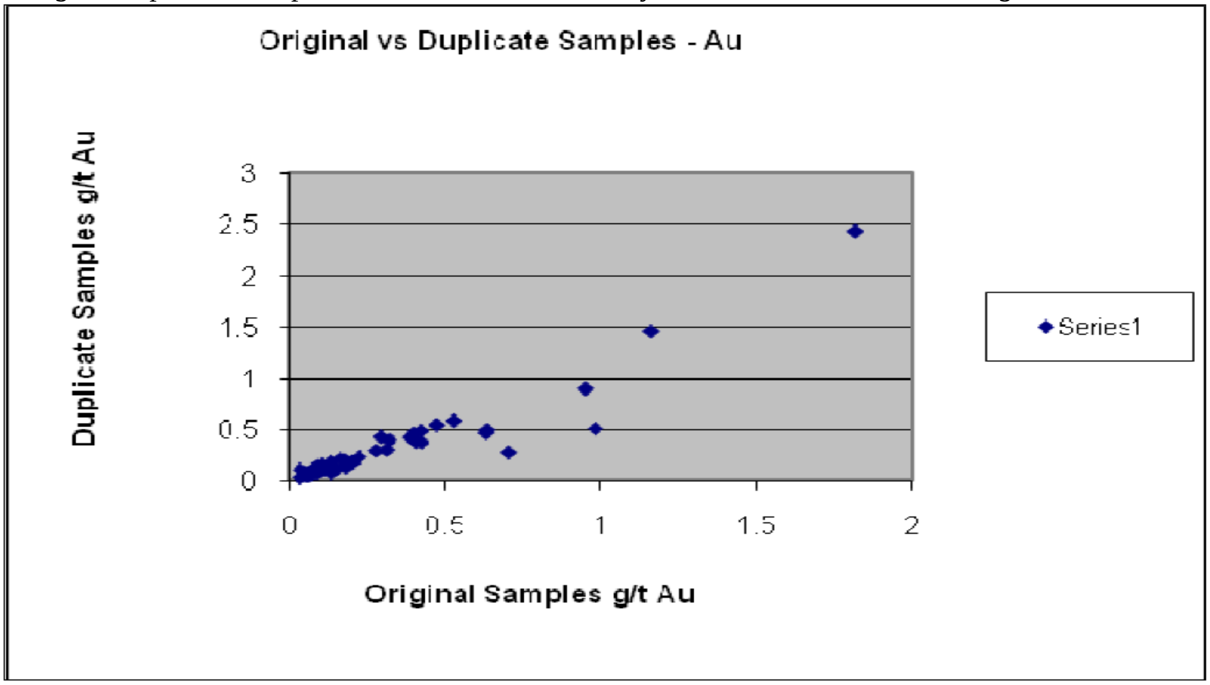


Figure 16-1 Original vs. Duplicate Samples - Au

Table 16-2***;*** *this is expected at gold contents at the concentrations analyzed. 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.*

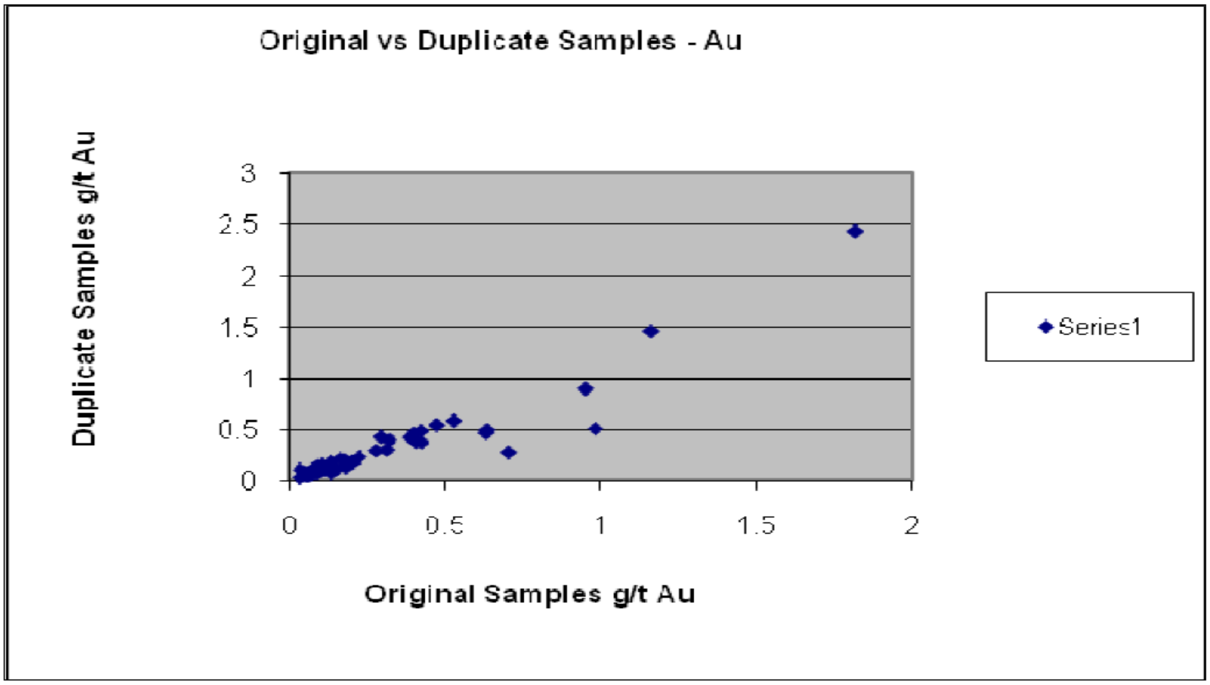


Figure 16-1 Original vs. Duplicate Samples - Au

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
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.

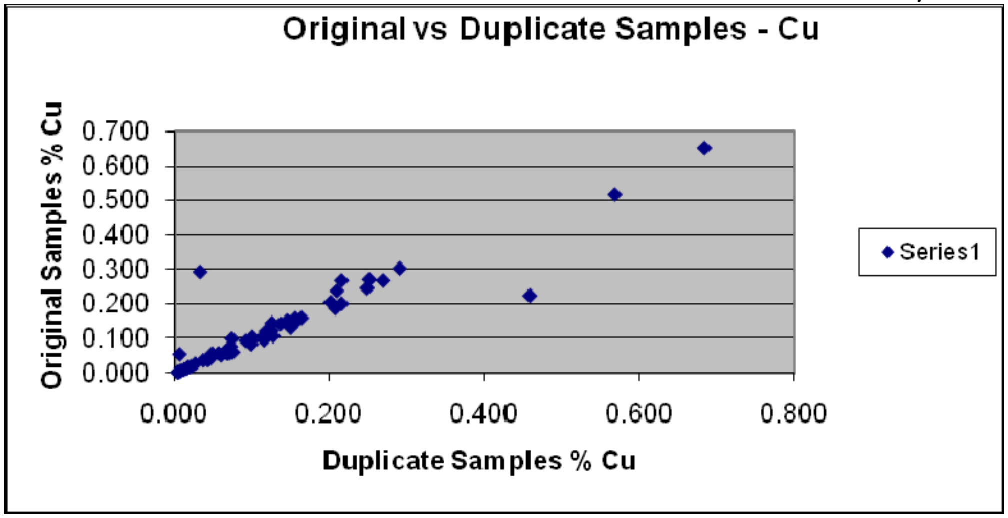


Figure 16-2 Original vs. Duplicate – Cu

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 16-4 illustrates 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 analyzed. 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.

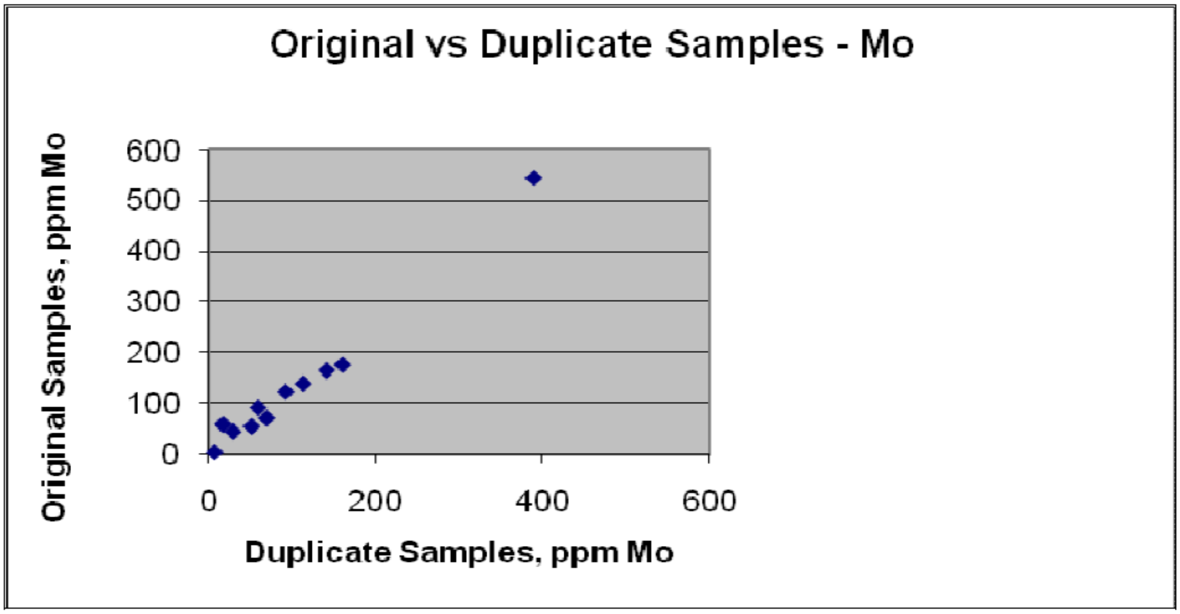


Figure 16-3 Original vs. Duplicate – Mo

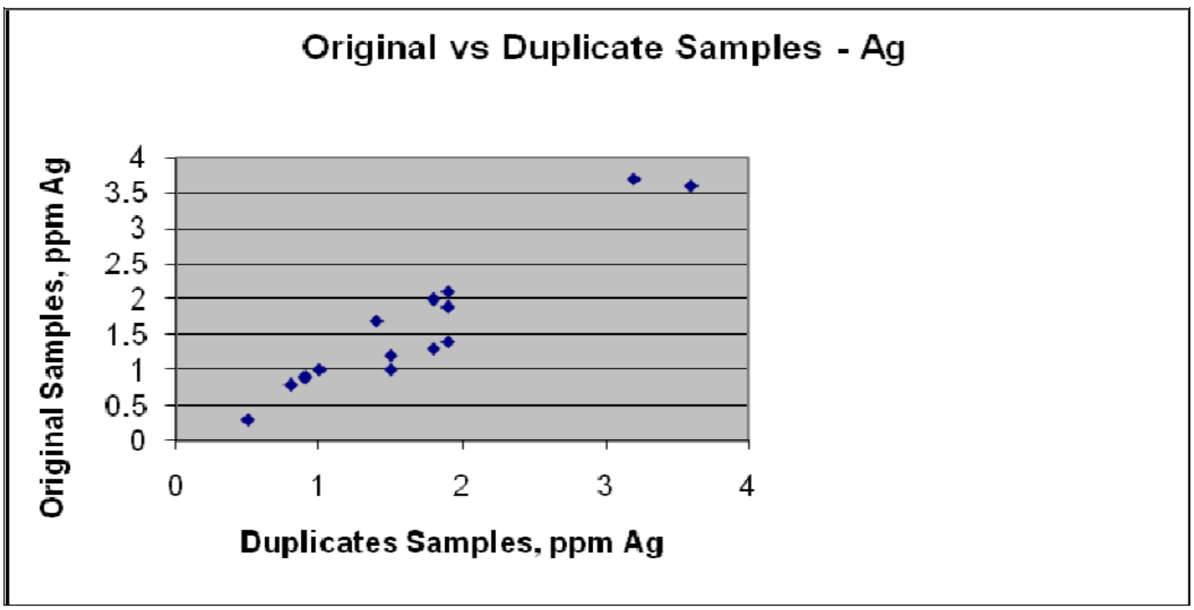


Figure 16-4 Original vs. Duplicate Samples – Ag

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, totaling 13 samples, were inserted into the sample chain at different times that were analyzed 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 reanalyzes ("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 reanalyzed, 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 following section has been taken from the LAL 2009 technical report.

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.

The following section 14 is taken from the technical report titled "Magnetite Mineral Resource Estimate – Bronson Slope Deposit For Skyline Gold Corporation,Vancouver, BC on the Bronson Slope Property" dated January 28, 2010, authored by and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc.and G. H Giroux and Arnd Burgert, P.Geo.,B.Sc. of Arnd Burgert Consulting Ltd, all three independent Qualified Persons as defined by NI 43-101. References within this excerpt are as given in the original report. This Technical Report was posted to SEDAR on March 5, 2010 (www.sedar.com).

Quality Control and Quality Assurance Programs for the 2007 and 2007 drilling which were directed toward the definition of gold-silver-copper-molybdenum mineralization grade are detailed in Burgoyne and Giroux (2008) and given on www.sedar.com and are not repeated here. These programs included inserted standards, duplicate sample insertion, check analyses, and laboratory internal standards. Likewise the Quality Assurance Programs for pre 2006 (historical) drilling are detailed in Burgoyne and Giroux (2007) and given on www.sedar.com and are not repeated here.

14.1 Quality Control and Quality Assurance Program for 2009 Magnetite Sampling Program

During the course of Skyline Gold's Magnetite Sampling Program 2,255 samples were analyzed, including 1,232 diamond drill core samples, 745 pulp samples, and 278 samples comprising the Quality Control and Quality Assurance (QA/QC) Program. Protocols followed for the QA/QC program are described in Item 14.2, and results are presented in Item 14.3. The following six types of samples were prepared and analysed as QA/QC checks, by either Skyline field personnel or the assay lab:

- *Field blanks;*

- *Field duplicates;*
- *Field standards;*
- *Laboratory duplicates; - Laboratory repeats;*
- *Laboratory standards.*

The QA/QC program indicates that no significant analytical problems were encountered, and no contamination of samples occurred. The assay values as reported are considered reliable.

14.2 QA/QC Protocols

14.2.1 Field Blank Protocol

Field blanks were inserted into the sample stream to test for contamination of samples at any stage of handling and analysis. Blank samples are meant to contain none of the commodity being analyzed, and the analysis values returned by the lab are expected to be below the detection limit. An assay value significantly higher than the detection limit would suggest an analytical blunder or sample contamination. Different blanks were used for pulp samples and core samples.

Blank samples inserted into the pulp sample stream were prepared by packaging 70g packs of blank rock pulp supplied by ALS Chemex Laboratory in North Vancouver, BC. These packs were inserted into the sample stream at an interval of approximately every 40th sample, for a total of eleven blank pulps. Since the blank rock pulp was intended for use as a blank for base and precious metals assays, it turned out to contain some magnetite. Accordingly, it did not return values below the detection limit, and the data are not useful as blanks. The data are treated as an additional field standard for the pulp sample set, and results are discussed as Standard 3 in Item 14.3.3.

Blanks inserted into the core sample stream were prepared by packaging 200g samples of coarse crushed, non-magnetic, white marble in standard rock sample bags. These packages were inserted into the sample stream at an interval of approximately every 40th sample, for a total of thirty-three blank rock samples. The samples were submitted blind to the lab, where they were prepared and analyzed the same as every other sample.

14.2.2 Field Duplicate Protocol

The purpose of field duplicate samples is to test for analytical precision and repeatability. The value of a duplicate sample analysis is expected to be close to the original analysis value. Due to natural variability of the magnetite concentration throughout the rock formation (and hence drill core), small disparities between initial and duplicate assay values are expected, and not indicative of an analytical problem. A large disparity between initial and duplicate analysis values would suggest an analytical problem.

No field duplicates were available for the pulp series. For the drill core series, field duplicate samples were prepared by sampling half of the available drill core as usual, but rather than replacing the remaining core portion in the core box as a reference, the remaining portion was bagged and prepared as an additional (duplicate) sample. The duplicate sample was submitted blind to the laboratory, where it was prepared and analyzed the same as every other sample. Duplicate samples were prepared approximately every 40th sample, for a total of thirty-five duplicate core samples.

14.2.3 Field Standard Protocol

Field standards were inserted into the sample streams for both pulps and drill core to test for analytical accuracy and repeatability of assays and systemic analytical deviations. Standards of three magnetite concentrations were used.

No prepared standard material for magnetite was commercially available. Material for standards was obtained from two sources. Standards 1 and 2 were prepared before the start of fieldwork by Skyline personnel from rock chips of a rock formation containing fine-grained magnetite. Two batches of magnetite-bearing rock chips were prepared, one of which was blended in a desired proportion with chips of a non-magnetic sandstone formation. The two rock chip batches were then submitted to Acme Analytical Laboratories (Canada) Ltd. in Vancouver, where the entire batches were crushed and pulverized to pass a No. 150 mesh size sieve screen. These were then homogenized by tumbling each batch in a rock sample bag. Standard 3 was in the form of rock pulp obtained from ALS Laboratory Group in North Vancouver, BC (ALS). The ALS standard was intended for use as blank pulp in the stream of pulp samples, but turned out to contain some magnetite, and has therefore been treated as an additional standard.

Standard samples were prepared by packaging 70g packs of the prepared standard material in small paper envelopes. These were then placed in standard rock sample bags and inserted into the sample sequence approximately every 30th sample, for a total of seventy-seven standard samples.
At the sample preparation laboratory (Assayers Canada in Telkwa, BC), the standard samples were crushed, sieved, and transferred into the laboratory's pulp envelopes the same as every other sample. They were submitted blind to the analytical lab.

14.2.4 Laboratory Duplicate Protocol

Laboratory duplicates were not available for the sequence of pulp samples. For the core sample sequence, laboratory duplicates were inserted into the sample stream by the sample preparation laboratory. The purpose of laboratory duplicates is to test for repeatability of assays. The value of a duplicate sample analyses is expected to be close to the original analysis value. Some variability is expected due to inherent in homogeneity in mineral distribution within the drill core (and hence within the coarse crushed core from which duplicate samples are prepared). A large disparity between initial and duplicate analysis would suggest an analytical problem.

At the crushing stage of sample preparation, a duplicate pulp sample was produced from approximately every 20th sample, for a total of eighty-five laboratory duplicate samples. The laboratory duplicate samples were identified by the original sample number plus the suffix "DP".

14.2.5 Laboratory Repeat Protocol

The purpose of laboratory repeats is to test for repeatability of analyses without the influence of inhomogeneous mineral distribution in the rock formation (and hence drill core and crushed core). The value of a repeat sample assay is expected to be close to the original assay value. Small disparities between initial and repeat analysis values can be used to judge the precision of the analytical technique. A large disparity between initial and repeat analysis would suggest an analytical problem.

Laboratory repeat analyses were performed by analyzing a pulp sample as usual, and then performing a second analysis using a fresh sub sample of pulp from the same pulp *sample. Laboratory repeat analyses were performed by the assay laboratory on approximately every 50th sample, for a total of twenty-four repeat analyses.*

14.2.6 Laboratory Standard Protocol

Laboratory standards were used to test for analytical accuracy and repeatability of analyses and systemic analytical deviations. Met-Solve prepared laboratory standard material from a sample of Bronson Slope drill core that was collected for a metallurgical study in 2007. The drill core was crushed, pulverized, and homogenized for use as a laboratory standard. An initial batch of samples from the standard material was analyzed as part of an orientation study described in Item 13.1.1. Laboratory standard samples were inserted into the core and pulp sample streams at an interval of approximately every 100th sample, for a total of fourteen laboratory standards.

14.3 QA/QC Results

14.3.1 Field Blank Results

Analyses results from the blank rock sample analyses are presented in Table 14-1. Of the thirty- three blank rock samples submitted, thirty-two returned values below the detection limit, while a single sample returned a value of the detection limit (0.1%). These results are consistent with what is expected for blank samples, and are not indicative of contamination at any stage of sampling, sample transport, sample preparation, or analyses.

TABLE 14-1 FIELD BLANK ANALYSES RESULTS

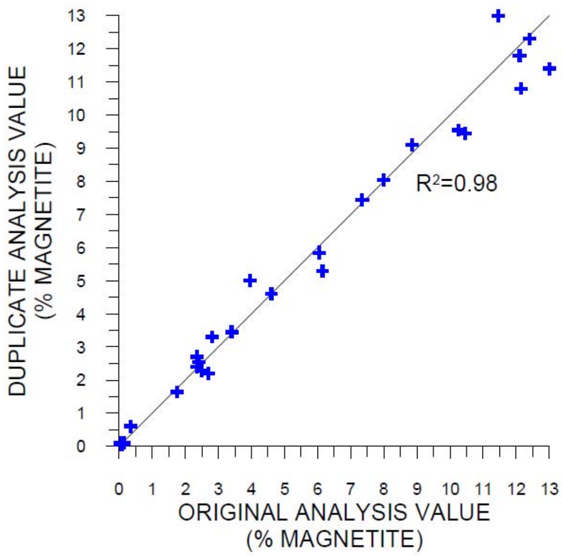
Sample No.	Magnetite Assay (%)	Sample No.	Magnetite Assay (%)	Sample No.	Magnetite Assay (%)
126547	BDL	127049	BDL	127571	BDL
126744	BDL	127089	BDL	127602	BDL
126791	BDL	127128	BDL	127634	BDL
126816	BDL	127250	BDL	127674	BDL
126866	BDL	127290	BDL	127714	BDL
126916	BDL	127330	BDL	127754	BDL
126578	BDL	127376	BDL	127814	BDL
126628	BDL	127416	BDL	127854	BDL
126678	BDL	127460	BDL	127894	BDL
126981	0.1	127502	BDL	127926	BDL
127019	BDL	127529	BDL	127966	BDL

Note: BDL means Below Detection Limit.

14.3.2 Field Duplicate Results

Analyses results from the thirty-five field duplicate samples are plotted on Figure 14-1. Perfect results in which assays of initial and duplicate samples returned identical values would plot on the diagonal line in the figure. The distribution of the value pairs (points) about the line can be described by the coefficient of determination (R2), where 0.0 would describe a random scatter of points, and 1.0 would describe a perfect fit to the line. The R2 for the field duplicate data set is 0.98. This degree of scatter is expected, and not indicative of any analytical problem.

FIGURE 14-1 FIELD DUPLICATE ANALYSES RESULTS



14.3.3 Field Standard Results

*Statistics for the set of analysis results for standards 1 through 3 are summarized in **Table 14-2**, and the data are presented graphically in **Figures 14-2** through **14-4**, respectively. A sample falling outside the second deviation (SD) from the mean is generally considered a failed standard analysis.*

TABLE 14-2 STATISTICAL SUMMARY OF FIELD STANDARD RESULTS

Standard 1		Standard 2		Standard 3	
n	33	n	32	n	11
Mean	1.84	Mean	0.62	Mean	1.24
Median	1.85	Median	0.70	Median	1.25
SD	0.20	SD	0.24	SD	0.10
2 SD	0.40	2 SD	0.48	2 SD	0.21

The mean value of the thirty-three Standard 1 assays is 1.84%. As shown on Figure 14-2, all Standard 1 analyses are within the 2 SD limit, except Sample 127829. The initial Sample No. 127829 assay, at 0.5% magnetite, lies well outside the expected range, and is considered a failed standard analysis. Met-Solve performed a re-analysis of the failed sample plus three other samples from the same sample sequence. The re-analysis of the failed standard returned 1.4% magnetite, while the re-analyses of the three remaining samples returned values similar to the respective initial values. A summary of the four re-analyses values is given in Table 14-3. Although the re-analysis value of 1.4% remains slightly below the 2 SD cut-off, it does appear to belong within the dataset, and does not appear to indicate a systematic analytical problem. Met- Solve could offer no explanation for the variable result. The failed standard assay (0.5%) has been omitted from statistical calculations, but the re-analysis value (1.4%) is included.

FIGURE 14-2
STANDARD 1 ANALYSES RESULTS

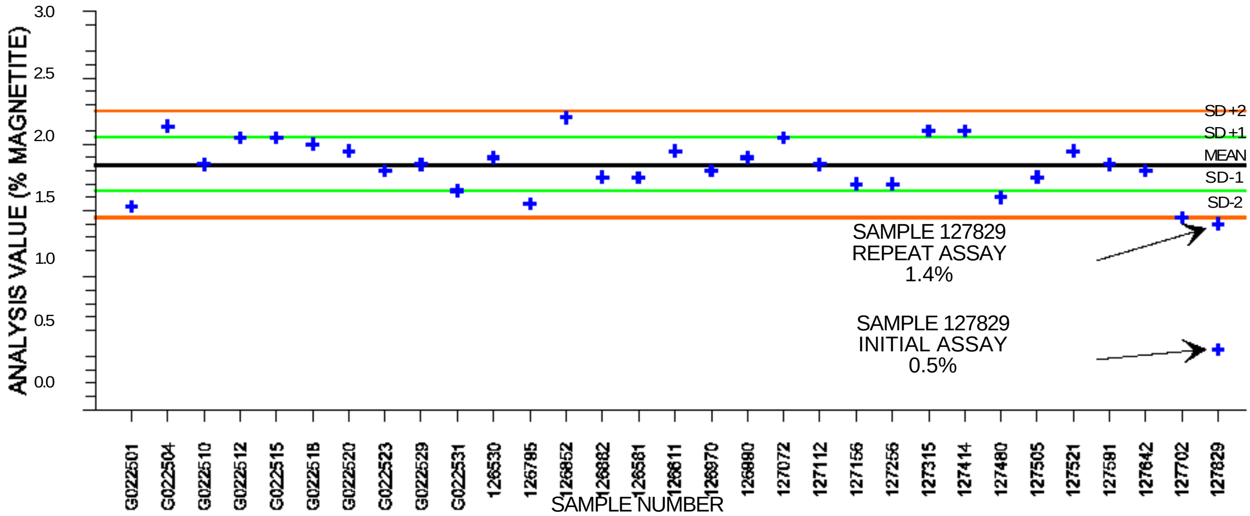


TABLE 14-3
SUMMARY OF FAILED STANDARD RE-ANALYSES

Sample No.	Magnetite (%)	
	Initial Analysis	Re-analysis
127829 (STA 1)	0.5	1.4
127828	BD	BD
127830	L	L
127839	0.1	0.1
	0.7	0.8

*The mean value of the thirty-two Standard 2 analyses is 0.62%. As shown on **Figure 14-3**, all Standard 2 analyses are within the 2 SD limit*

.

*The mean value of the eleven Standard 3 analyses is 1.24%. As shown on **Figure 14-4**, all Standard 2 assay values are within the 2 SD limit except one. Sample G022502 is slightly below the 2 SD limit. This sample was analysed with the initial batch of samples to be assayed at the lab. For the initial batch, sub samples of 15 g were analysed in the magnetic separator. An assessment of the duplicate and repeat analysis values for this sample batch determined that analytical precision and repeatability were relatively low, so for all subsequent analyses, 20g sub samples were analysed. The analysis value of Sample G022502 confirms the relatively low precision of this first sample batch, and the problem was corrected by using 20g sub samples for all subsequent analyses.*

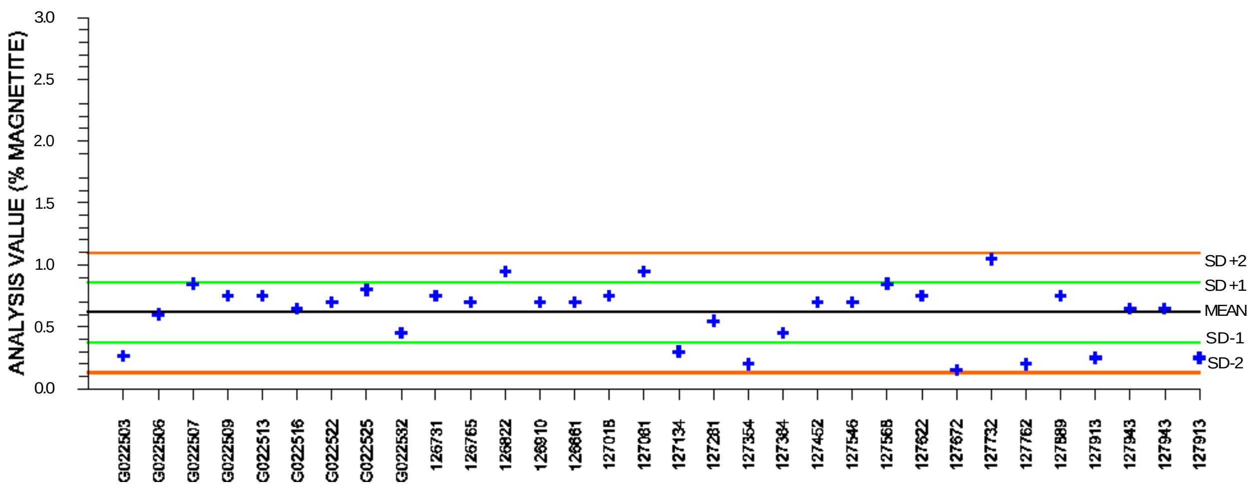
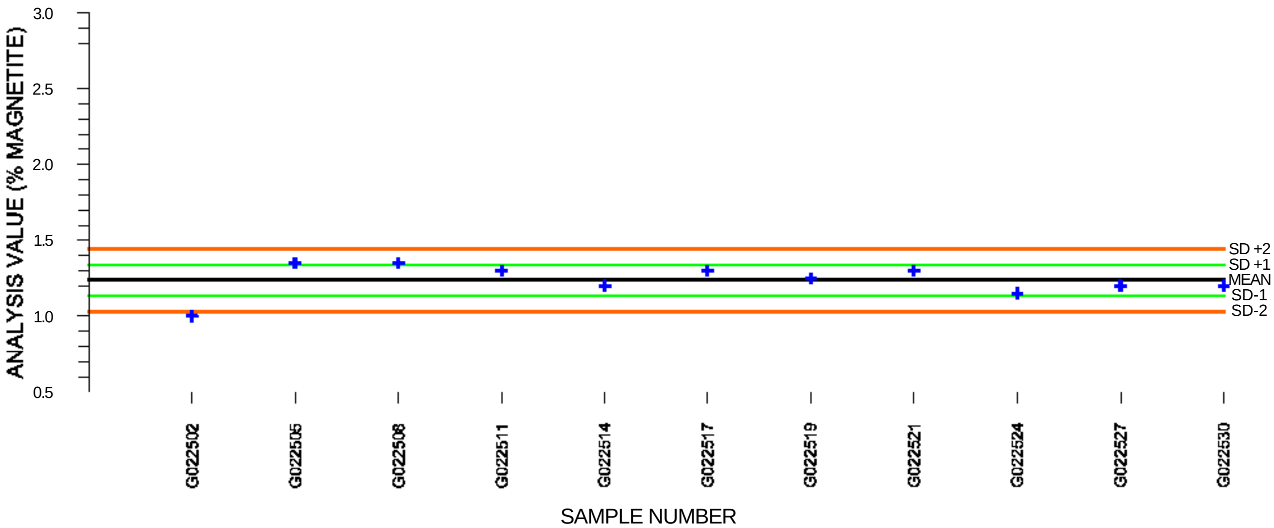


FIGURE 14-3
STANDARD 2 ANALYSES RESULTS

SAMPLE NUMBER

FIGURE 14-4

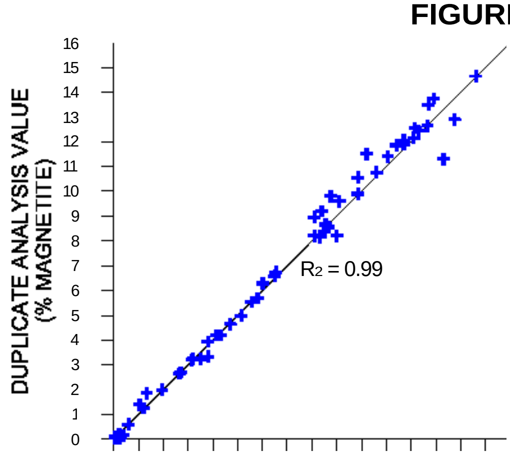


STANDARD 3 ANALYSES RESULTS

14.3.4 Laboratory Duplicate Results

*Analysis results of the eighty-five laboratory duplicate samples are presented graphically in **Figure 14-5**. The correlation is very good, with an R^2 of 0.99. No analytical problem is indicated.*

FIGURE 14-5



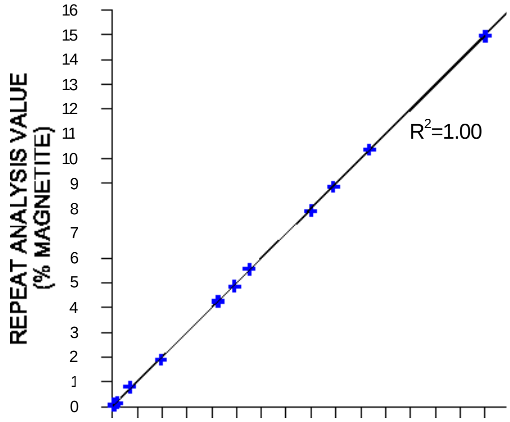
LABORATORY DUPLICATE ANALYSES RESULTS

0 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
ORIGINAL ANALYSIS VALUE
(% MAGNETITE)

14.3.5 Laboratory Repeat Results

Analysis results of the twenty-four laboratory repeat samples are presented graphically in Figure 14-6. The correlation is almost perfect, with an R^2 of 1.00. No analytical problem is indicated.

FIGURE 14-6



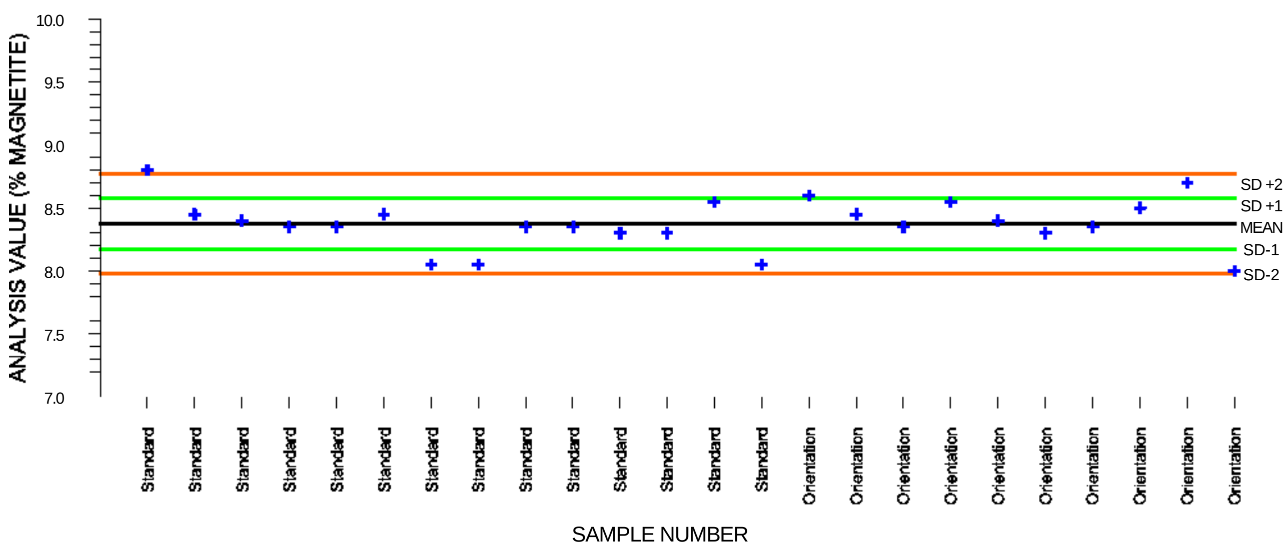
LABORATORY REPEAT ANALYSIS RESULTS

0 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
ORIGINAL ANALYSIS VALUE
(% MAGNETITE)

14.3.6 Laboratory Standard Results

*The twenty-four laboratory standard samples comprising the statistical population include ten samples from the orientation study described in **Item 13.1.1.** These ten orientation samples are identified on **Figure 14-7** as "Orientation". The mean value of the twenty-four laboratory standard analyses is 8.38% magnetite. As shown on **Figure 14-7**, all Laboratory Standard analysis values are within the 2 SD limit except one. At 8.8%, the first sample at the left side of the plot is slightly above the 2 SD limit of 8.77. However, the data point does appear to belong with the data set, and is suggestive of variability in the analytical precision rather than any analytical problem.*

FIGURE 14-7
LABORATORY STANDARD ANALYSIS RESULTS



14.4 Technical Review by Authors

The 2009 drill-sampling program 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.

All of the exploration work conducted on the Bronson Slope property from 1988 through 2007 was performed by competent, professionally qualified persons.

Due diligence studies by the writers include those completed during the site visits and review of the data on this property during 2006, 2007, and 2009. During 2009, Mr. Burgoyne was at the property from October 1 through 3, 2009, and Mr. Burgert was on site through the complete sampling program from October 1 through 22, 2009. This evaluation work in 2009 and 2010 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.*
- *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 analysis 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 through 2009.*

17.0 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. References contained within the excerpt are as given in the original report.

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.

18.0 Mineral Processing and Metallurgical Testing

The following section 18 is taken from "Technical Report – Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation Vancouver, BC on the Bronson Slope Property", dated March 6, 2009 and posted to SEDAR on March 6, 2009. This report was prepared by J. A. R. Lawrence, MAusIMM (#209746) and V. Seen, MAusIMM of Leighton Asia Limited ("LAL"). This Technical Report can be viewed at www.sedar.com.

18.1 Metallurgical Testing

18.2 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 characterization, 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.

18.2.1 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 Oxidized - 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 minimize 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.*

18.2.2 **Metallurgical Test Work Details**

18.2.2.1 **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

18.2.2.2 **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.

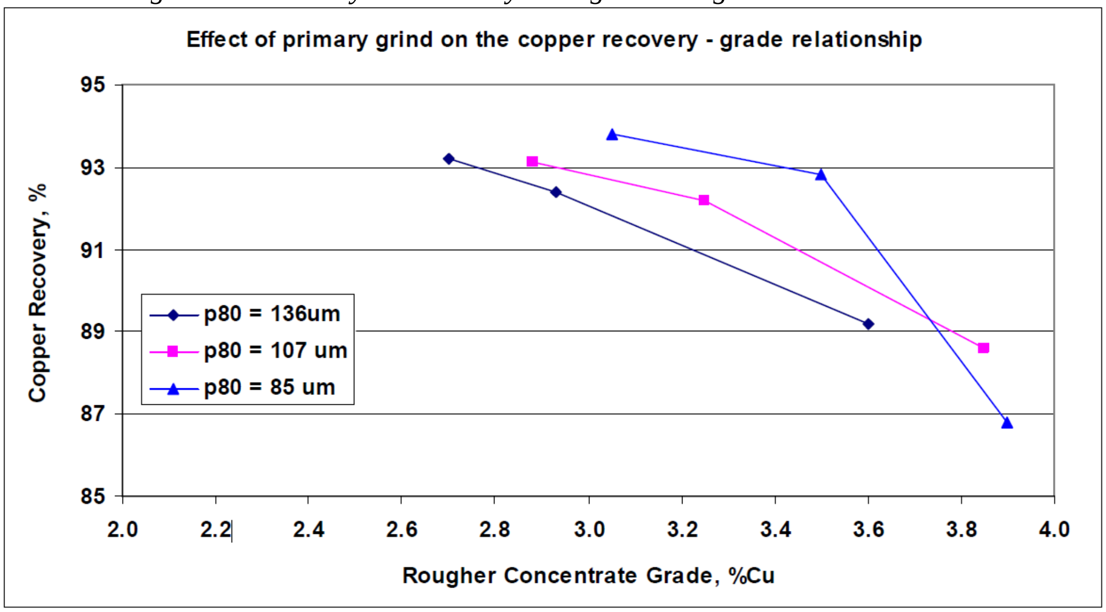


Figure 18-1 Effect of Primary Grind on the Copper Recovery and Grade

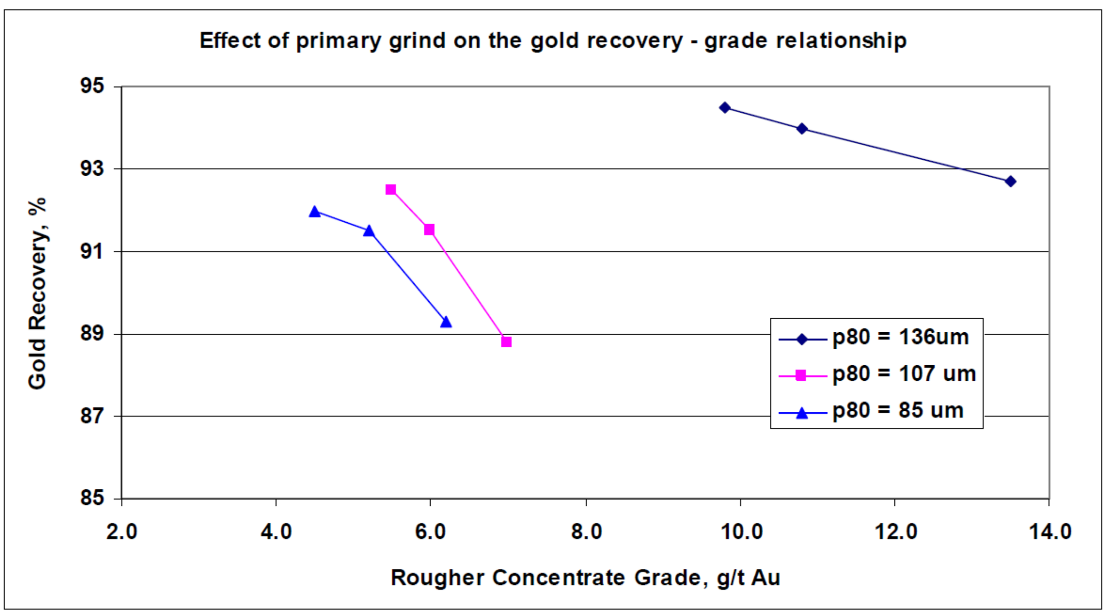


Figure 18-2 Effect of Primary Grind on the Gold Recovery and Grade

18.2.2.3 **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= 136u

Conditions	Weight %	Grade				Recovery			
			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 = 136u

Gravity Concentrate			Rougher flotation concentrate									
Weight	Recovery, %		Weight	Grade				Recovery				Total
				Cu	Mo	Au	Ag	Cu	Mo	Au	Ag	Au Rec
%	Au	Ag	%	%	%	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

18.2.2.4 **Cleaner Flotation Test**

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 = 18.5u

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 git	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

18.2.2.5 **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	25.2	0.500	24.6	15.8	77.2	34.6	19.1	2.3
	Cleaner Conc.	0.6			36.3	197			51.6	51.8
US	Gravity Conc.	0.5	26.4	0.642	21.3	32.2	80.6	50.7	21.3	6.2
	Cleaner Conc.	0.6			39.8	196			55.6	52.3
USO	Gravity Conc.	0.5	23.2	0.723	53.5	73.1	71.0	37.7	32.7	11.6
	Cleaner Conc.	0.7			43.8	174			40.7	42.1
PPY	Gravity Conc.	0.4	25.6	0.948	28.8	26.7	66.9	41.5	28.3	3.9
	Cleaner Conc.	0.4			44	236			40.3	31.9
QM	Gravity Conc.	0.4	19.2	0.228	52.5	25.9	80.5	29.9	36.7	3.9
	Cleaner Conc.	0.7			30.3	174			40.8	50.5
QM *	Gravity Conc.	0.5	25.6	0.323	22.6	26.7	77.8	27.5	23.9	4.9
	Cleaner Conc.	0.5			40.2	242			44.6	46.2
SP	Gravity Conc.	0.2	24.8	0.330	21.1	20.8	75.7	30.7	8.3	1.4
	Cleaner Conc.	0.7			33.6	182			54.9	49.7
HG	Gravity Conc.	0.2	26.2	0.361	95.4	42.7	82.4	48.9	23.8	2.2
	Cleaner Conc.	1.2			31.5	167			49.7	54.6

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

18.2.2.6 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 indicates 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	A_u *	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**				

18.2.2.7 **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	Up.per Sediment Oxidized	Porphyry	Quartz Magnetite	Starter Pit	High Grade
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
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
Si (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_20_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)*

18.2.2.7.1 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 summarized 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

18.2.2.8 **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		

18.2.2.9 **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

18.2.2.10 **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

18.2.2.11 **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^{2}h 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

18.2.2.12 **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 CaCO3 equivalent per tonne, but the cleaner-scavenger tails reported an average of -526.1 kg CaCO3 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 CaCO3 equivalent per tonne. This indicates that the combined final tailings have a slight acid generating potential.

The following comments on magnetite in section 16 is taken from the technical report titled "Magnetite Mineral Resource Estimate – Bronson Slope Deposit For Skyline Gold Corporation, Vancouver, BC on the Bronson Slope Property" dated January 28, 2010, authored by and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc.and G. H Giroux and Arnd Burgert, P.Geo.,B.Sc. of Arnd Burgert Consulting Ltd, all three independent Qualified Persons as defined by NI 43-101. Eferences in this excerpt are as given in the original report. This Technical Report was posted to SEDAR on March 5, 2010 (www.sedar.com).

16.0 Mineral Processing & Metallurgical Testing

Magnetite

Klein (2008) completed metallurgical and marketing studies with respect magnetite that could be recovered from the Bronson Slope deposit. The market study estimated the amount of magnetite used for BC and Alberta Coal industries for dense media separation and the metallurgical study assessed the properties of the Bronson Slope magnetite use in dense media. Dense media separation is a process in which finely ground magnetite is mixed with water to create a medium that has properties of a dense liquid. Specifically when coal and rock particles are added to the medium, the low-density coal particles will float while the high-density rock particles will sink thereby facilitating separation of coal from waste rock. The dense media is used in two types of separators referred to static separators, such as the dense media drum or dynamic separators such as the dense media cyclone. The specifications for magnetite used in dense media applications is as follows:

- *Particle size: 90% passing 325 mesh Density: >4.7 g/cm3*
- *Magnetics content: >93% magnetics*
- *Based on the study by Klein (2008) the estimated usage of magnetite by BC and Alberta Coal Mines for 2007 was 52,743 tonnes.*

Klein (2008) completed metallurgical testing from bulk samples obtained from drill core from the Quartz Magnetite Unit of the Bronson Slope deposit. The testing confirmed the metallurgical process, determined the grinding work index for regrinding and characterize the magnetite product with respect to the specifications. The testing involved grinding and flotation to recover copper sulphides. The flotation tailings were used and for magnetic separation testing.

The quartz magnetite sample used for the testing graded 8.94% Fe, 0.27% Cu, and 0.91 g/t Au. From flotation, the copper concentrate contained 79.1% of the copper and the combined flotation- gravity concentration gold recovery was 81.7%.

The flotation tailings were subjected to three stages of magnetic separation. The final product had a density of 4.97 g/cm3 and magnetic content of 99.9%. These specifications exceed those required for dense media confirming that a high-grade magnetite product can be produced that is suitable for dense media separation. The product accounted for 3.68% of the feed mass and contained 28% of the total iron. Based on rougher mass it is expected these values could be increased to close to 10% and 55%, respectively.

Klein (personal communications, 2010) estimates an approximate 95% recovery of the magnetite can be possible although this must be confirmed in further metallurgical studies.

This report by Klein (2008) gives details on Magnetite Usage, Metallurgical Balance Test Report, Metallurgical Balance Flow sheet, Flotation Test Report, Davis Test Tube Report, and Particle Size Analysis and is given as ***Appendix D****.*

Bronson Slope Cu-Au-Ag Porphyry Deposit – A Summary

The following discussion of copper, gold, silver, and molybdenum metallurgical studies and recoveries are detailed in Lawrence and Seen (2009) and Burgoyne and Giroux (2008). Detailed discussions are given for the 1994, 1995, 1996 and 1997 programs, which are detailed in Burgoyne and Giroux (2008). These reports can be found on www.sedar.com. The following is a summary of this information.

Skyline Gold 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.

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 millwork 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.

TABLE 16-1
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 16-2
COMPARISON OF ASSAY GRADES OF
COMPOSITED METALLURGICAL TEST SAMPLES (1996, 1997

METALLURGICAL ASSAYING									
Comp. Sample Name	**BC**		**US**	**USO**	**PPY**	**QM**	**SP**	**HG**	Average of
No. Of Tests	28		3	2	2	4	2	2	All Comps
Average Metallurgical	Au g/t	0.472	0.446	0.776	0.369	0.518	0.517	0.724	0.546
Calculated Head Grades	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									
Comp. Sample Name	**BC**		**US**	**USO**	**PPY**	**QM**	**SP**	**HG**	Average of
No. Of Core Samples	1488		462	195	199	817	115	145	All Comps
Unweighted Average	Au g/t	0.50	0.49	0.9	0.4	0.54	0.58	0.66	0.58
Assay Grades	Ag g/t	2.60	2.7	3.3	2.5	2.6	2.9	3.1	2.8
	Cu %	0.18	0.19	0.23	0.14	0.18	0.19	0.24	0.19
	Mo %	0.005	0.008	0.013	0.006	0.003	0.004	0.007	0.01
Comp. Sample Name	**BC**		**US**	**USO**	**PPY**	**QM**	**SP**	**HG**	Average of
No. Of Core Samples	1488		462	195	199	817	115	145	All Comps
Weighted Average	Au g/t	0.47	0.47	0.99	0.4	0.49	0.54	0.63	0.57
Assay Grades	Ag g/t	2.3	2.2	3.3	2.2	2.4	2.3	3.1	2.5
	Cu %	0.16	0.18	0.23	0.13	0.16	0.17	0.23	0.18
	Mo %	0.004	0.007	0.013	0.006	0.003	0.003	0.007	0.006

No. = Number
Comp. = Composite
BC = Bulk Composite 350 m Pit (1996)
US = Upper Sedimentary Rock (1997)
USO = Oxidized Upper Sedimentary Rock (1997)
PPY Porphyry (1997)
SP Starter Pit (1997)
HG High Grade Sample

The following section 18 is taken from "Technical Report – Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation Vancouver, BC on the Bronson Slope Property", dated March 6, 2009 and posted to SEDAR on March 6, 2009. This report was prepared by J. A. R. Lawrence, MAusIMM (#209746) and V. Seen, MAusIMM of Leighton Asia Limited ("LAL"). This Technical Report can be viewed at www.sedar.com.

18.3 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.

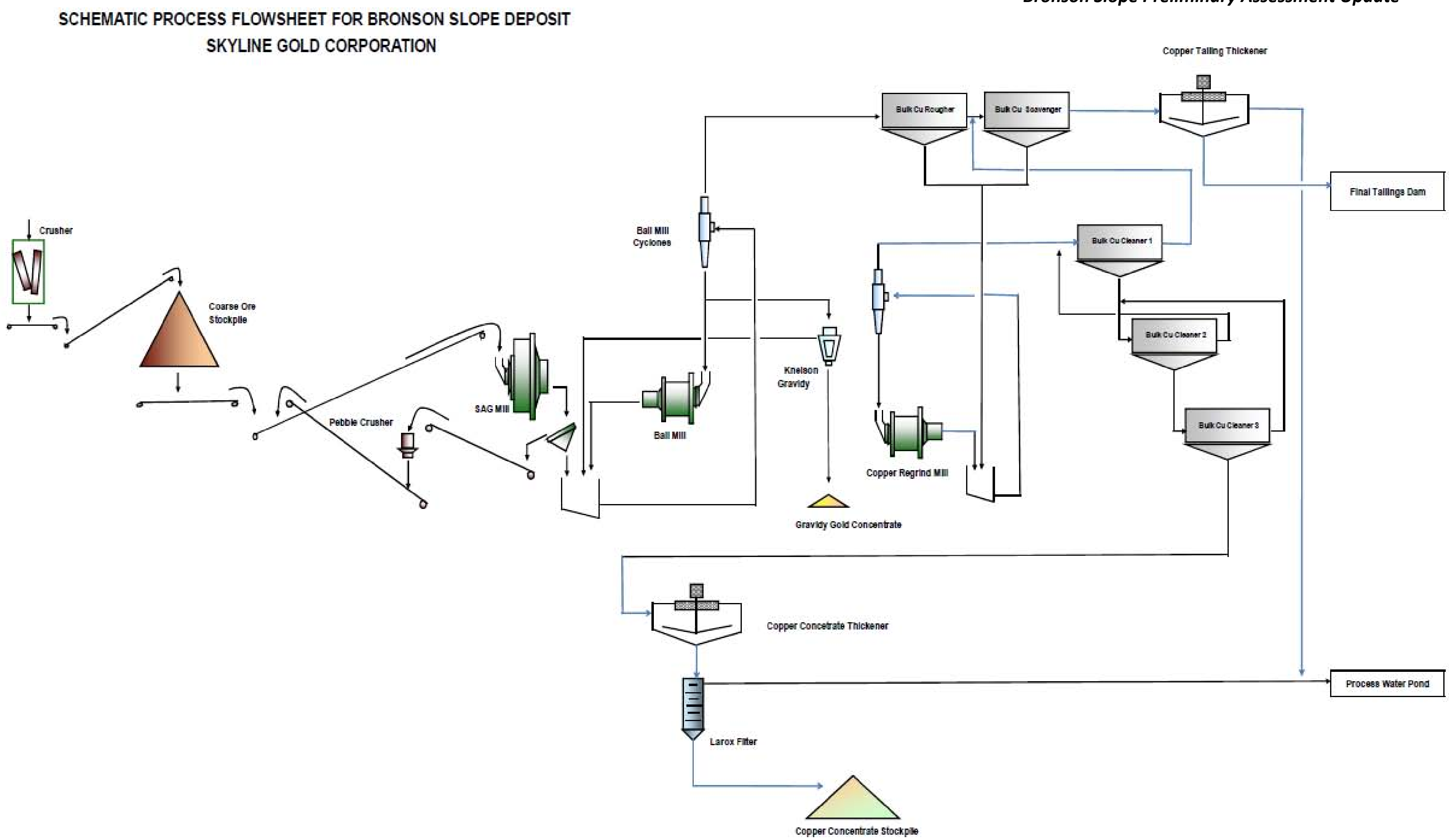


Figure 18-3 Proposed Process Flowsheet

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*

18.3.1.1 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.

18.3.1.2 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.

18.3.1.3 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.*

18.3.1.4 **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.

18.3.1.5 **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.

18.3.1.6 **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.

18.3.1.7 **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.

18.3.2 **Magnetite Process**

The magnetite for the PA (scoping) level of study has been added as an allowance on the assumption that magnetic separators can be added instream with the currently designed process. The size of the circuits is assumed to fit within the existing plant infrastructure. Detailed process design is recommended for higher levels of study.

19.0 Mineral Resource and Reserve Estimates

19.1 Mineral Resource

The Bronson Slope Property hosts a porphyry gold-copper-silver-molybdenum-magnetite 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 modeling and kriging is considered reliable and relevant. The resource estimate is presented based on updated cases of metal prices. These are presented in Table 19-1below:

Table 19-1 Metal Prices Used for Resource Estimate

Metal	Prices
Cu	$2.00/lb
Au	$650/t.oz
Ag	$10/t.oz
Magnetite	$90/ton

*nb - all figures are in USD

The metal prices were used along with block based metallurgical recoveries to determine individual block values. The mineral resources presented in Table 19-2 below were then determined based on a cut off of USD 9.00 per tonne net recoverable value.

Table 19-2 2008 Bronson Slope Resource Estimate

2008 - Bronson Slope Resource Estimate (Cutoff USD 9/t NSR)					
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
Total Measured + Indicated	**225,100,000**	**0.36**	**2.22**	**0.14**	**0.0077**
Inferred	91,600,000	0.27	1.76	0.13	0.0080

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).

The magnetite mineral resource is summarized in table below. The details of the magnetite resource calculation are included in the following section.

Table 19-3 Bronson Slope – Global Magnetite Mineral Resource (Cut-off 2% Magnetite)

Category	Metric kTonnes	% Magnetite	Contained Magnetite kTonnes
Measured	66,210	7.58	5,020
Indicated	96,950	7.08	6,860
Total Measured + Indicated	**163,160**	**7.28**	**11,880**
Inferred	6,300	6.92	440

The resources reported above came from two different reports and studies. The initial estimate for Cu, Ag, Au and Mo was completed in 2008 while a second resource estimate was completed for the Magnetite content in 2010.

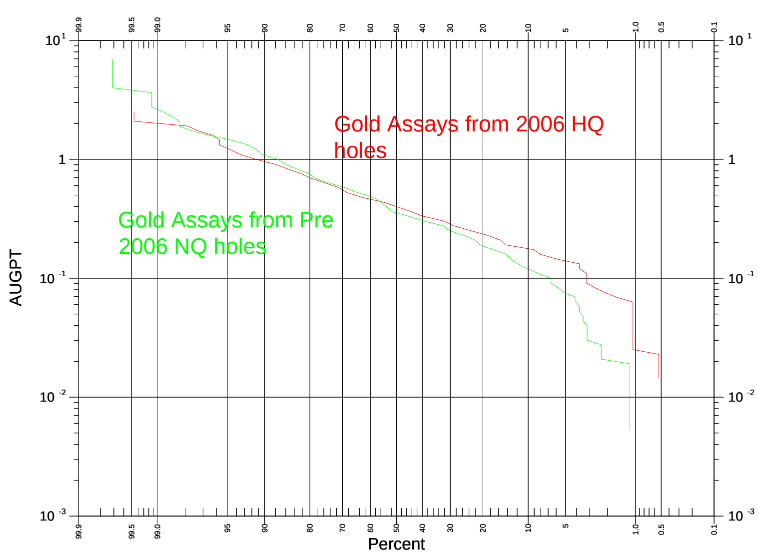
The following section 17 is taken from the report “Mineral Resource Estimate – Bronson Slope Deposit For Skyline Gold Corporation, Vancouver, BC on the Bronson Slope Property” dated April 30, 2008, authored by and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc. and G. H Giroux from Giroux Consultants, both independent Qualified Persons as defined by NI 43-101. This Technical Report was posted to SEDAD in 2010 (www.sedar.com). References contained within the excerpt are as given in the original report.

17.0 Mineral Resource Estimation

17.1 Data Analysis

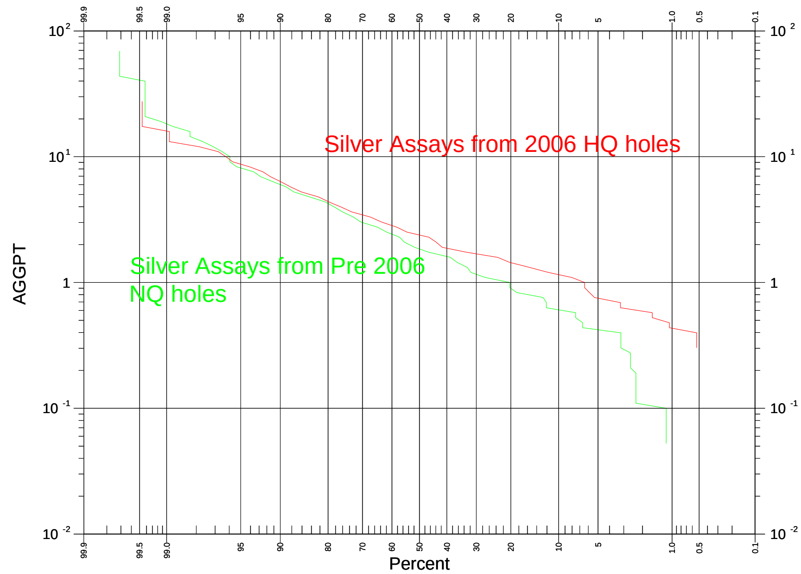
At the request of Skyline Gold Corporation, an updated resource estimation was completed on the Bronson Slope Deposit. Data for the project was provided by Skyline with information on 11 diamond holes drilled in 2007 added to the data base. A total of 85 drill holes were used to complete the resource estimate and totalled 18,985 m.

A total of 5,413 samples were assayed for gold and most were also assayed for Ag, Cu and Mo. The drill holes represent two main campaigns with different diameter drill holes. The Pre 2006 drill holes use NQ core while the 2006 drilling used larger diameter HQ core. To test the effects of a different core size and hence a different sample size a comparison for each variable was made within the volume of rock tested by the 2006 drill holes. Cumulative frequency plots for each variable are shown below with the Pre 2006 NQ assays shown in green and the 2006 HQ assays shown in red.



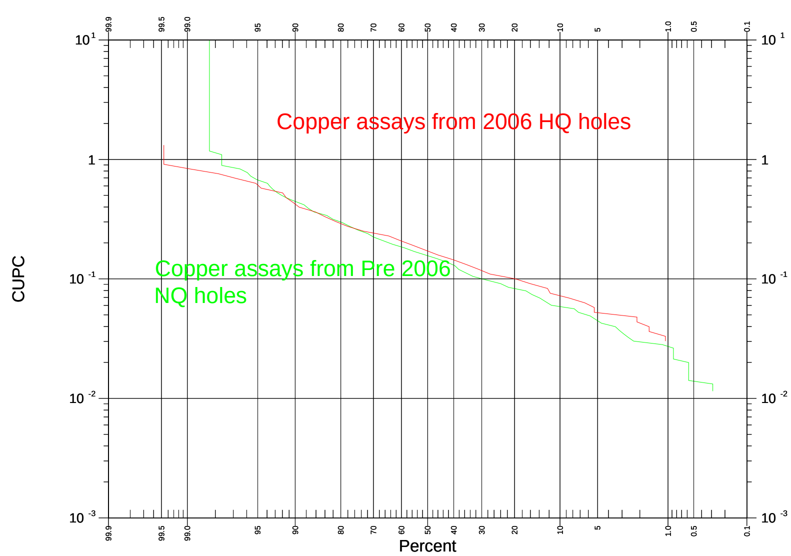
Cumulative Frequency Plot for Gold within a Similar Volume

FIGURE 17-1: CUMULATIVE FREQUENCY PLOT FOR AU IN 2006 HQ SAMPLES VS. PRE 2006 NQ SAMPLES



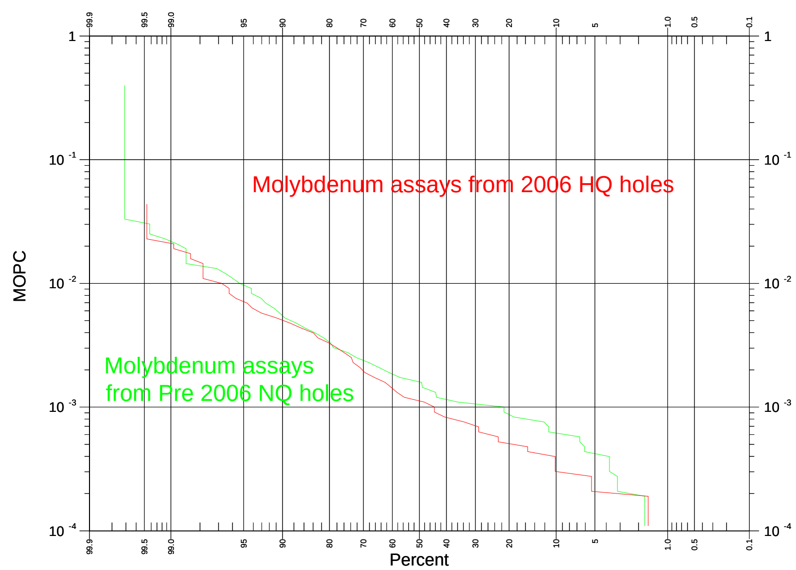
Cumulative Frequency Plot Silver within Similar Volume

FIGURE 17-2: CUMULATIVE FREQUENCY PLOT FOR AG IN 2006 HQ SAMPLES VS. PRE 2006 NQ SAMPLES



Cumulative Frequency Plot for Copper within Similar Volume

FIGURE 17-3: CUMULATIVE FREQUENCY PLOT FOR CU IN 2006 HQ SAMPLES VS. PRE 2006 NQ SAMPLES



Cumulative Frequency Plot Molybdenum within Similar Volume

FIGURE 17-4: CUMULATIVE FREQUENCY PLOT FOR MO IN 2006 HQ SAMPLES VS. PRE 2006 NQ SAMPLES

For each variable there is no indication of bias present with both grade distributions within a similar volume very similar and almost overlapping. There is no reason to suspect the larger volume of sample from the HQ drilling is giving different results from the smaller NQ core.

A total of 5 lithologic domains were modelled by Skyline.

- *HW - Hanging wall Sediments*
- *LS - Lower Sediments*
- *PPY - Porphyry*
- *QM - Quartz magnetite*
- *US - Upper Sediments*

Individual assays were tagged with geology and the statistics for each variable are shown below.

TABLE 17-1
SUMMARY OF ASSAY STATISTICS FOR GEOLOGIC DOMAINS.

Variable	Domain	Number	Mean	S.D.	Minimum	Maximum	C.V.
Au (g/t)	HW	2,014	0.293	0.999	0.001	39.22	3.41
	LS	234	0.192	0.141	0.001	0.90	0.73
	PPY	744	0.265	0.293	0.001	3.63	1.10
	QM	1,311	0.467	0.708	0.001	13.75	1.52
	US	1,612	0.380	0.355	0.001	4.97	0.94
Ag (g/t)	HW	1,755	2.93	15.03	0.01	489.00	5.12
	LS	234	2.11	2.44	0.01	20.40	1.15
	PPY	744	1.98	4.54	0.01	79.80	2.29
	QM	1,311	2.47	3.60	0.05	70.00	1.46
	US	1,595	3.12	25.40	0.01	999.00	8.13
Cu (%)	HW	1,755	0.035	0.056	0.001	1.560	1.60
	LS	234	0.120	0.069	0.001	0.463	0.57
	PPY	744	0.108	0.360	0.001	9.400	3.33
	QM	1,311	0.159	0.217	0.001	4.520	1.37
	US	1,595	0.186	0.139	0.001	1.486	0.75
Mo (%)	HW	1,315	0.003	0.005	0.0001	0.062	1.97
	LS	234	0.014	0.012	0.0001	0.079	0.85
	PPY	744	0.005	0.008	0.0001	0.112	1.74
	QM	1,311	0.003	0.012	0.0001	0.400	3.73
	US	1,572	0.010	0.017	0.0001	0.400	1.66

Note. S.D. is Standard Deviation and C.V. is coefficient of variation and is equal to S.D. / Mean

While the mean grades for each element are somewhat similar in each geologic domain the grade distributions are different as is shown by ***Figures 17-5 to 17-8*** *in Burgoyne and Giroux (2007).*

As a result of the different grade distributions within individual lithologies, each lithology was evaluated independently for each of the four variables by producing cumulative lognormal probability plot. In each case the grade distribution shows multiple overlapping populations. In almost all cases the highest grade population was considered to be erratic high grade. The *locations of these high samples were checked and in almost all cases they were shown to be isolated high values. A capping procedure was used to minimize the effects of these isolated high assays. In almost all cases the capping threshold was set at 2 standard deviations above the mean of the next lowest population. The capping intervals and numbers of samples capped are shown in* ***Table 17-2.***

TABLE 17-2
SUMMARY OF CAPPING BY DOMAIN

Lithology	Variable	Level Capped	Cap Value	Number Capped
HW	Ag	2SDAM2	153.0 g/t	3
	Au	2SDAM2	5.74 g/t	3
	Cu	2SDAM2	0.286 %	6
	Mo	2SDAM2	0.038 %	2
LS	Ag	2SDAM2	13.9 g/t	1
	Au	2SDAM2	0.85 g/t	2
	Cu	2SDAM2	0.305 %	4
	Mo	2SDAM2	0.044 %	7
PPY	Ag	2SDAM2	45.5 g/t	2
	Au	2SDAM2	2.56 g/t	1
	Cu	2SDAM2	1.060 %	2
	Mo	2SDAM2	0.071 %	1
QM	Ag	2SDAM2	20.5 g/t	8
	Au	2SDAM2	5.3 g/t	5
	Cu	2SDAM1	1.910 %	1
	Mo	2SDAM2	0.040 %	10
US	Ag	2SDAM2	74.0 g/t	3
	Au	2SDAM2	2.21 g/t	7
	Cu	2SDAM2	1.100 %	4
	Mo	2SDAM2	0.083 %	6

Note: 2SDAM2 refers to 2 Standard Deviations Above the Mean of Population 2

The effects of capping are shown in ***Table 17-3*** *comparing uncapped mean values with capped mean values for each lithology.*

TABLE 17-3
SUMMARY OF ASSAY STATISTICS FOR GEOLOGIC DOMAINS

Variable	Domain	Number	Mean Uncapped	Mean Capped
Au (g/t)	HW	2,014	0.293	0.276
	LS	234	0.192	0.192
	PPY	744	0.265	0.264
	QM	1,311	0.467	0.456
	US	1,612	0.380	0.375
Ag (g/t)	HW	1,755	2.93	2.72
	LS	234	2.11	2.08
	PPY	744	1.98	1.91
	QM	1,311	2.47	2.41
	US	1,595	3.12	2.52
Cu (%)	HW	1,755	0.035	0.034
	LS	234	0.120	0.119
	PPY	744	0.108	0.095
	QM	1,311	0.159	0.157
	US	1,595	0.186	0.185
Mo (%)	HW	1,315	0.003	0.003
	LS	234	0.014	0.014
	PPY	838	0.005	0.005
	QM	1,311	0.003	0.003
	US	1,572	0.010	0.010

17.2 Composites

Drill holes were "passed through" the geologic solids with the point at which each hole entered and left each solid recorded. Uniform down hole 10 m composites were formed that honoured the solid boundaries. Composites at the boundaries less than 5 m were combined with adjoining samples while those greater than 5 but less than 10 were left as is. The result is a composite file of uniform support with all composites 10 ±5 m in length. The statistics for composites within each domain is shown in ***Table 17-4.***

TABLE 17-4
SUMMARY OF 10 M COMPOSITE STATISTICS FOR GEOLOGIC DOMAINS

Variable	*Domain*	*Number*	*Mean*	*S.D.*	*Minimum*	*Maximum*	*C.V.*
Au (g/t)	*HW*	*504*	*0.226*	*0.234*	*0.001*	*1.759*	*1.04*
	LS	*74*	*0.182*	*0.108*	*0.001*	*0.469*	*0.59*
	PPY	*236*	*0.270*	*0.205*	*0.001*	*1.586*	*0.76*
	QM	*380*	*0.415*	*0.358*	*0.027*	*2.096*	*0.86*
	US	*479*	*0.360*	*0.232*	*0.001*	*1.273*	*0.64*
Ag (g/t)	*HW*	*492*	*2.01*	*3.89*	*0.01*	*47.64*	*1.93*
	LS	*74*	*1.94*	*1.71*	*0.01*	*8.10*	*0.88*
	PPY	*236*	*1.78*	*1.96*	*0.01*	*15.98*	*1.10*
	QM	*380*	*2.22*	*1.86*	*0.19*	*15.31*	*0.84*
	US	*479*	*2.33*	*2.42*	*0.01*	*26.39*	*1.04*
Cu (%)	*HW*	*492*	*0.033*	*0.035*	*0.001*	*0.348*	*1.05*
	LS	*74*	*0.114*	*0.056*	*0.001*	*0.225*	*0.49*
	PPY	*236*	*0.095*	*0.078*	*0.001*	*0.741*	*0.82*
	QM	*380*	*0.142*	*0.120*	*0.001*	*0.604*	*0.85*
	US	*479*	*0.178*	*0.102*	*0.001*	*0.559*	*0.58*
Mo (%)	*HW*	*394*	*0.003*	*0.006*	*0.0001*	*0.071*	*2.10*
	LS	*74*	*0.014*	*0.009*	*0.0001*	*0.031*	*0.63*
	PPY	*236*	*0.005*	*0.006*	*0.0001*	*0.038*	*1.27*
	QM	*380*	*0.003*	*0.004*	*0.0001*	*0.031*	*1.32*
	US	*472*	*0.010*	*0.008*	*0.0001*	*0.065*	*0.87*

Note. S.D. is Standard Deviation and C.V. is coefficient of variation and is equal to S.D. / Mean

17.3 Variography

Since the data within any one of the domains was limited all composites were modelled using pairwise relative semivariograms. Models were created for gold, silver, copper and molybdenum. In all cases the horizontal plane was modelled using a number of different directions to find the direction of maximum continuity. Once found, the vertical plane perpendicular to this direction was modelled. The parameters for each model are summarized below in ***Table 17.5****.*

TABLE 17-5
SUMMARY OF SEMIVARIOGRAMS FOR AU, AG, CU AND MO - BRONSON SLOPE DEPOSIT

Variable	Azimuth	Dip	C_0	C_1	C_2	Short Range (m)	Long Range (m)
Au	130	0	0.10	0.12	0.33	20	300
	40	-20	0.10	0.12	0.33	30	140
	220	-70	0.10	0.12	0.33	20	300
Ag	90	0	0.05	0.40	0.20	15	300
	0	-45	0.05	0.40	0.20	20	200
	180	-45	0.05	0.40	0.20	10	100
Cu	90	0	0.10	0.15	0.20	10	240
	0	0	0.10	0.15	0.20	40	200
	0	-90	0.10	0.15	0.20	60	200
Mo	90	0	0.10	0.18	0.39	10	280
	0	-45	0.10	0.18	0.39	25	200
	180	-45	0.10	0.18	0.39	20	140

17.4 Bulk Density

A total of 88 specific gravity measurements have been made on crushed core at Bronson Slope by Acme Analytical Laboratories. The results are presented in Table 17-6. Three of the five geologic domains were sampled US, PPY and QM. Average values for these three domains are 2.74, 2.76 and 2.77 respectively. The average of these three or 2.76 is used for the other 2 Domains not sampled, namely HW and LS.

TABLE 17-6
SUMMARY OF SPECIFIC GRAVITY DETERMINATIONS

Domain	Number	Minimum SG	Maximum SG	Average SG
US	13	2.65	2.85	2.74
PPY	3	2.75	2.77	2.76
QM	72	2.67	2.89	2.77
TOTAL	88	2.65	2.89	2.76

17.5 Block Model

A geologic model was completed by Skyline geologists using Surpac software. Five geologic solids were created based on sectional interpretations. A block model of 10 x 10 x 10 m blocks was superimposed over these solids. For each block the percentage below topography and inside the respective solids was recorded. The origin of the block model is as follows:

Lower Left Corner	*24800 E*	*10 m wide*	*160 columns*
	11300 N	*10 m long*	*120 rows*
Top of Model	*900*	*10 m high*	*108 levels*
No rotation			

17.6 Grade Interpolation

Each of the five geologic domains were compared to it's neighbours using contact plots. These plots examine the contact between two units and compare the average grade of a variable as a function of distance from the contact. These plots are useful to determine if hard or soft boundaries should be used during estimation of grade. Contact plots were produced for each variable on each domain contact. The only domain showing considerable differences in average grade across a boundary was the Hanging Wall unit (HW As a result of this contact investigation the HW unit was estimated on its own using only composites from within the unit. All other units were combined and estimated using soft boundaries between units.

Grade estimation was completed by ordinary kriging using search ellipses tied to the semivariogram ranges both in distance and orientation. The estimation process was run for each variable in a series of passes. Pass 1 used ¼ of the semivariogram range in the three principal directions of continuity. A minimum of 4 composites was required to estimate the block. For blocks not estimated in Pass 1 a second pass with search ellipse distances equal to ½ the semivariogram ranges was completed. A third pass using the full semivariogram range was used for blocks still not estimated. As gold is the principal commodity a fifth pass with larger search ellipses was completed for Cu, Ag and Mo to make sure all blocks with a gold grade had the other three variables estimated. In all cases if more than 16 composites were found the closest 16 to the block centroid were used. The various search parameters and distances are shown below in Table 17-7 along with the number of blocks estimated in each pass.

TABLE 17-7
SUMMARY OF SEARCH DIRECTIONS AND DISTANCES FOR RESOURCE ESTIMATION

Domain	Variable	Pass	Number	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)	Az/Dip	Dist. (m)
HW	Au	1	37,410	130/0	75	40/-20	35	220/-70	75
		2	65,613	130/0	150	40/-20	70	220/-70	150
		3	67,969	130/0	300	40/-20	140	220/-70	300
	Ag	1	22,497	90/0	75	0/-45	50	180/-45	25
		2	52,142	90/0	150	0/-45	100	180/-45	50
		3	58,717	90/0	300	0/-45	200	180/-45	100
		4	37,345	90/0	700	0/-45	500	180/-45	300
	Cu	1	30,561	90/0	60	0/0	50	0/-90	50
		2	59,470	90/0	120	0/0	100	0/-90	100
		3	70,458	90/0	240	0/0	200	0/-90	200
		4	10,503	90/0	480	0/0	400	0/-90	400
	Mo	1	25,172	90/0	70	0/-45	50	180/-45	35
		2	58,672	90/0	140	0/-45	100	180/-45	70

		3	70,915	90/0	280	0/-45	200	180/-45	140
		4	16,233	90/0	660	0/-45	500	180/-45	380
OTHER DOMAINS	Au	1	68,125	130/0	75	40/-20	35	220/-70	75
		2	80,518	130/0	150	40/-20	70	220/-70	150
		3	92,486	130/0	300	40/-20	140	220/-70	300
	Ag	1	45,522	90/0	75	0/-45	50	180/-45	25
		2	68,104	90/0	150	0/-45	100	180/-45	50
		3	75,482	90/0	300	0/-45	200	180/-45	100
		4	52,021	90/0	700	0/-45	500	180/-45	300
	Cu	1	59,714	90/0	60	0/0	50	0/-90	50
		2	71,304	90/0	120	0/0	100	0/-90	100
		3	83,383	90/0	240	0/0	200	0/-90	200
		4	26,728	90/0	480	0/0	400	0/-90	400
	Mo	1	54,253	90/0	70	0/-45	50	180/-45	35
		2	68,052	90/0	140	0/-45	100	180/-45	70
		3	77,643	90/0	280	0/-45	200	180/-45	140
		4	41,181	90/0	660	0/-45	500	180/-45	380

17.7 Classification

Based on the study herein reported, delineated mineralization of the Bronson Slope Deposit is classified as a resource according to the definitions supplied in National Instrument 43-101 and from CIM (2005).

For the Bronson Slope deposit geological continuity has been established though surface mapping and diamond drill hole interpretation. Grade continuity can be quantified by semivariogram analysis. By tying the classification to the semivariogram ranges through the use of various search ellipses the resource is broken into classes based on grade continuity.

- *Measured - Blocks estimated during pass 1 using search ellipses set to ¼ of the semivariogram range and located within the central core of well drilled information*
- *Indicated - Blocks estimated during pass 1 and outside the central core or blocks estimated during pass 2 using search ellipses set to ½ the semivariogram range*
- *Inferred - All remaining estimated blocks.*

17.8 Results

As this is a multi element deposit some form of combining of variables to produce a listing by cutoff is required. For the Bronson slope a Net Recoverable Value (NRV) was produced that takes into account reasonable metal prices and metal recoveries. Not considered in this value are mining costs, smelter costs, transportation and other economic factors yet to be determined. The metal prices used are based on various other deposits completing pre-feasibility and feasibility studies in north western B.C. The metal recoveries are based on studies completed (See Metallurgy Section 16) for each of the PPY, US and QM geologic Domains. An average set of recovery numbers were used for the untested HW and LS domains. Future metallurgical work should be completed on these two domains prior to economic studies.

CASE 1

Case 1 uses prices and recoveries used in the 2007 report and serves as a comparison showing the effects of the additional drilling.

Metal Prices used in NRV calculation (all in US $):
Gold - $525/oz Silver - $8.00/oz Copper - $1.50/lb Molybdenum - $10/lb
Recoveries used in NRV calculation

Domain	Au Recovery	Ag Recovery	Cu Recovery	Mo Recovery
PPY	82.9 %	72.5 %	83.4 %	53.2 %
HW	84.0 %	67.4 %	86.8 %	46.5 %
LS	84.0 %	67.4 %	86.8 %	46.5 %
US	82.2 %	72.0 %	88.8 %	58.4 %
QM	87.6 %	70.1 %	87.1 %	33.5 %

The NRV value is calculated in US$.

The results of the additional drilling at Bronson Slope can be shown by comparing the 2007 resource with the 2008 results using an additional 11 diamond drill holes.

TABLE 17-16
BRONSON SLOPE MINERAL RESOURCE
BASE CASE 2007 VS CASE 1 2008

2007 Base Case Resource Estimate **2008 Case 1 Resource Estimate**

Category	NRV Cutoff US$	Metric Tonnes	Au g/t	Ag g/t	Cu %	Mo%	Metric Tonnes	Au g/t	Ag g/t	Cu %	Mo%	Tonnage Difference
Measured	$5.00	84,000,000	0.42	2.38	0.15	0.005	95,700,000	0.4	2.26	0.15	0.006	13.90%
Indicated	$5.00	258,900,000	0.30	2.28	0.09	0.008	269,200,000	0.29	2.04	0.10	0.006	4.00%
M+I	$5.00	**342,900,000**	**0.33**	**2.3**	**0.11**	**0.007**	**364,900,000**	**0.32**	**2.1**	**0.11**	**0.006**	6.40%
Inferred	$5.00	238,800,000	0.28	2.25	0.08	0.007	179,400,000	0.26	1.65	0.09	0.006	-24.90%
Measured	$9.00	54,500,000	0.51	2.47	0.19	0.005	58,700,000	0.5	2.45	0.18	0.006	7.70%
Indicated	$9.00	75,400,000	0.39	2.41	0.14	0.011	80,800,000	0.36	2.38	0.15	0.009	7.20%
M+I	$9.00	**129,800,000**	**0.44**	**2.44**	**0.16**	**0.008**	**139,500,000**	**0.42**	**2.41**	**0.17**	**0.008**	7.50%
Inferred	$9.00	45,200,000	0.37	1.92	0.16	0.011	30,200,000	0.34	1.89	0.15	0.007	-33.20%

CASE 2

Case 2 uses more current metal prices to demonstrate the price sensitivity and recoveries based on additional work in 2007 and 2008.

Metal Prices used in CASE 2 NRV calculation (all in US $):
Gold - $650/oz Silver - $10.00/oz Copper - $2.00/lb Molybdenum - $12/lb

Recoveries used in NRV calculation

Domain	*Au Recovery*	*Ag Recovery*	*Cu Recovery*	*Mo Recovery*
PPY	*82.9 %*	*72.5 %*	*83.4 %*	*53.2 %*
HW	*84.0 %*	*71.5 %*	*93.9 %*	*46.5 %*
LS	*84.0 %*	*71.5 %*	*86.8 %*	*46.5 %*
US	*82.2 %*	*72.0 %*	*88.8 %*	*58.4 %*
QM	*87.6 %*	*70.1 %*	*87.1 %*	*33.5 %*

The NRV value is calculated in US$.

BRONSON SLOPE MINERAL RESOURCE – CASE 2 METAL PRICES
Cut Off US $ 9.00 /Tonne Net Recoverable Metal Value

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*
Total Measured + Indicated	*225,100,000*	*0.36*	*2.22*	*0.14*	*0.0077*
Inferred	*91,600,000*	*0.27*	*1.76*	*0.13*	*0.0080*

The following section 17 is taken from the report "Magnetite Mineral Resource Estimate – Bronson Slope Deposit For Skyline Gold Corporation, Vancouver, BC on the Bronson Slope Property" dated January 28, 2010, authored by and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc. and G. H Giroux and Arnd Burgert, P.Geo, B.Sc. of Arnd Burgert Consulting Ltd, all three independent Qualified Persons as defined by NI 43-101. This Technical Report was posted to SEDAR on March 5, 2010 (www.sedar.com). References contained within the excerpt are as given in the original report.

17.0 Mineral Resource Estimation
MAGNETITE MINERAL RESOURCE
17.1 Data Analysis

A total of 22 diamond drill holes totaling 7,521 m were re-sampled for magnetite content. The drill holes were compared to the domain solids used to estimate the Cu-Au Resource in 2008. Individual magnetite measurements, as determined by the Davis Tube method and discussed in ***Item 13*** *were tagged with a Domain Code. The statistics for each Domain are tabulated below.*

TABLE 17-1
SAMPLE STATISTICS FOR MAGNETITE IN ANALYSES
(Weight Percent)

	US	**QM**	**PPY**	**HW**
Number	*928*	*691*	*225*	*113*
Mean	*0.86*	*7.22*	*6.05*	*0.06*
Standard Deviation	*1.97*	*4.09*	*3.56*	*0.02*
Minimum	*0.05*	*0.05*	*0.05*	*0.05*
Maximum	*12.95*	*16.55*	*17.85*	*0.20*
Coefficient of Variation	*2.29*	*0.56*	*0.59*	*0.41*

No capping was required in any of the lithologies.

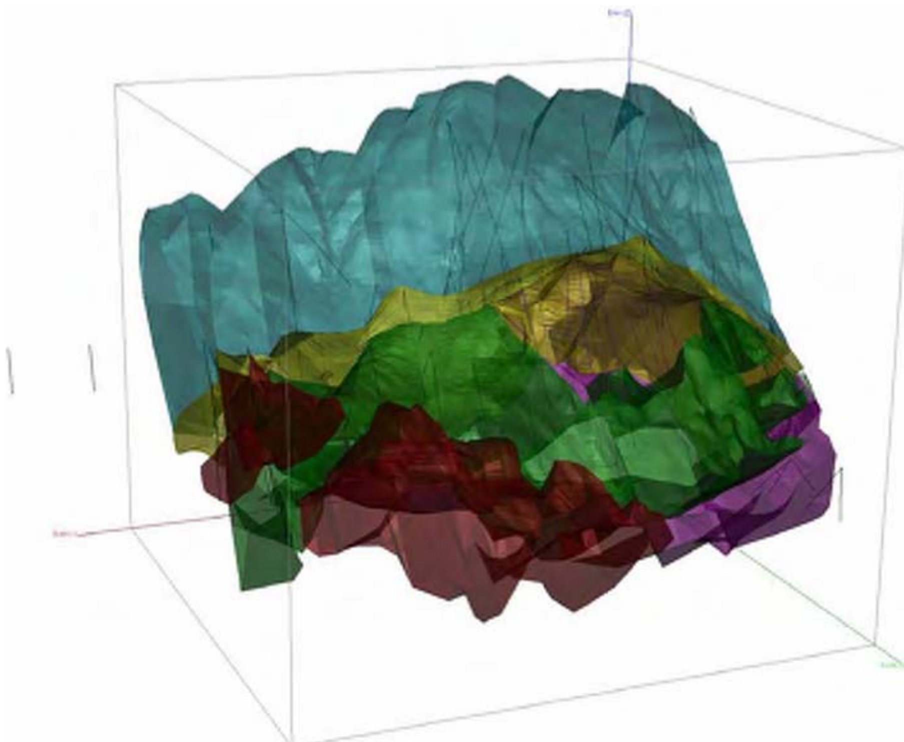


FIGURE 17-1
ISOMETRIC VIEW LOOKING SOUTH SHOWING DRILL HOLES AND THE VARIOUS DOMAINS, HW IN BLUE, PPY IN MAGENTA, QM IN GREEN, US IN YELLOW-GREEN AND LS IN RED.

17.2 Composites

Uniform down hole composites were produced at 5 m intervals within each of the four domains. Intervals less than 2.5 m at the domain boundaries were combined with the adjoining assay to produce a uniform support of 5 ± 2.5 m.

TABLE 17-2
SAMPLE STATISTICS FOR MAGNETITE IN 5 M COMPOSITES
(Weight Percent)

	US	QM	PPY	HW
Number	551	417	141	74
Mean	0.84	7.26	5.98	0.06
Standard Deviation	1.85	3.77	3.38	0.04
Minimum	0.05	0.05	0.05	0.05
Maximum	12.44	14.94	14.06	0.33
Coefficient of Variation	2.20	0.52	0.56	0.58

Contact plots, showing the changes in average grade taken at different distances from a contact, between QM and PPY show similar grades on both sides indicating a soft boundary could be used. There are however roughly three times as many measurements in the Quartz Magnetite as in the Intrusive (PPY) and as a result a hard boundary was used to avoid smoothing higher QM grades into PPY blocks where there were gaps in the PPY coverage. The contact plot between QM and US shows that while the QM has significantly higher grades on average the contact area of up to 20 m on each side shows very similar grades in the 3 to 4 % range. Again, however, in areas where there are no US data to mitigate an estimate the higher grade QM could *overestimate the US grades, so a hard boundary was placed between these two domains for interpolation.*

17.3 Semivariograms

Due to the limited amount of composite data a single set of semivariograms was run using all composites containing Magnetite assays. Pair-wise relative semivariograms were run in the four principal horizontal directions; E-W, N-S, SW-NE and NW-SE. Geometric anisotropy was demonstrated with the longest continuity in the E-W direction. The second longest range was in the vertical direction. Nested spherical models were fit to all the semivariograms.

TABLE 17-3
SUMMARY OF SEMIVARIOGRAM PARAMETERS

Variable	Az	Dip	C_o	C_1	C_2	Short Range (m)	Long Range (m)
Magnetite	90	0	0.05	0.30	0.50	20	200
	0	0	0.05	0.30	0.50	20	50
	0	-90	0.05	0.30	0.50	40	60

17.4 Bulk Density

A total of 88 specific gravity measurements have been made on crushed core at Bronson Slope by Acme Analytical Laboratories. The results are presented in ***Table 17-4.*** *Three of the five geologic domains were sampled US, PPY and QM. Average values for these three domains are 2.74, 2.76 and 2.77 respectively. The average of these three or 2.76 is used for the other two Domains not sampled, namely HW and LS.*

TABLE 17-4
Summary of Specific Gravity Determinations

Domain	Number	Minimum SG	Maximum SG	Average SG
US	13	2.65	2.85	2.74
PPY	3	2.75	2.77	2.76
QM	72	2.67	2.89	2.77
TOTAL	88	2.65	2.89	2.76

17.5 Block Model

A geologic model was completed by Skyline geologists using Surpac software. Five geologic solids were created based on sectional interpretations. A block model of 10 x 10 x 10 m blocks was superimposed over these solids. For each block the percentage below topography and inside the respective solids was recorded. The origin of the block model is as follows:

Lower Left Corner	*24900 E*	*10 m wide*	*160 columns*
	11300 N	*10 m long*	*115 rows*
Top of Model	*895*	*10 m high*	*80 levels*
No rotation			

17.6 Grade Interpolation

Magnetite grades were interpolated into blocks containing some percentage of QM, PPY, US or HW domains using ordinary kriging. The kriging was done first for blocks containing some percentage of QM using only QM composites. A second kriging run was completed for blocks containing some percentage of PPY and only PPY composites. Finally a third kriging run was *completed for blocks containing some percentage of US or HW using composites from within these two domains.*

In all runs the kriging was completed in four passes with the search ellipse for each pass a function of the semivariogram ranges. For Pass 1 a minimum of 4 composites were required within a search ellipse with dimensions equal to 1/4 the semivariogram range. For blocks not estimated a second pass using search ellipse dimension equal to 1/2 the range were used. A third pass using the full range and a fourth pass using twice the range completed the exercise. In all cases a maximum of 12 composites were used and if more than 12 were found within any search, the closest 12 were used. In all cases a maximum of 3 composites from any single hole were allowed insuring each block was estimated with at least two drill holes.

17.7 Results

*Blocks were classified during the resource estimation for Au and Cu completed in **Item 17.7** of Burgoyne and Giroux (2008) found at www.sedar.com The tonnage was established for each block during this estimation. For the magnetite estimation the block classifications and tonnages were taken from the 2008 block model. The results are tabulated below.*

TABLE 17-5
MEASURED MAGNETITE GLOBAL RESOURCE

Magnetite Cut-off (%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off Magnetite (%)	Contained Tonnes Magnetite
1.00	71,280,000	7.15	5,100,000
2.00	66,210,000	7.58	5,020,000
3.00	62,900,000	7.86	4,940,000
4.00	59,550,000	8.10	4,820,000
5.00	55,680,000	8.35	4,650,000
6.00	51,020,000	8.61	4,390,000
7.00	43,570,000	8.97	3,910,000
8.00	32,790,000	9.44	3,100,000
9.00	20,300,000	10.01	2,030,000
10.00	8,730,000	10.74	940,000

TABLE 17-6
INDICATED MAGNETITE GLOBAL RESOURCE

Magnetite Cut-off (%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off Magnetite (%)	Contained Tonnes Magnetite
1.00	108,680,000	6.47	7,030,000
2.00	96,950,000	7.08	6,860,000
3.00	88,500,000	7.52	6,660,000
4.00	83,620,000	7.76	6,490,000
5.00	76,160,000	8.07	6,150,000
6.00	67,200,000	8.42	5,660,000
7.00	57,080,000	8.75	4,990,000
8.00	39,890,000	9.26	3,690,000
9.00	22,580,000	9.84	2,220,000
10.00	7,230,000	10.71	770,000

TABLE 17-7
MEASURED PLUS INDICATED MAGNETITE GLOBAL RESOURCE

Magnetite Cut-off (%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off Magnetite (%)	Contained Tonnes Magnetite
1.00	179,960,000	6.74	12,130,000
2.00	163,160,000	7.28	11,880,000
3.00	151,400,000	7.66	11,600,000
4.00	143,170,000	7.90	11,310,000
5.00	131,840,000	8.19	10,800,000
6.00	118,220,000	8.50	10,050,000
7.00	100,640,000	8.84	8,900,000
8.00	72,680,000	9.34	6,790,000
9.00	42,880,000	9.92	4,250,000
10.00	15,960,000	10.73	1,710,000

TABLE 17-8
INFERRED MAGNETITE GLOBAL RESOURCE

Magnetite Cut-off (%)	Tonnes > Cut-off (tonnes)	Grade > Cut-off Magnetite (%)	Contained Tonnes Magnetite
1.00	13,040,000	4.01	520,000
2.00	6,300,000	6.92	440,000
3.00	5,710,000	7.38	420,000
4.00	5,320,000	7.66	410,000
5.00	4,980,000	7.88	390,000
6.00	4,390,000	8.21	360,000
7.00	3,500,000	8.60	300,000
8.00	2,510,000	9.00	230,000
9.00	1,110,000	9.58	110,000
10.00	240,000	10.40	20,000

It should be noted that when this global magnetite resource is compared to the preliminary open pit designed for the Preliminary Economic Assessment completed by Asia Leighton (Lawrence and Seen, 2009) in March 2009 that 44% of the measured and indicated magnetite tonnage, at a 2% cut-off is within the pit. The remainder sits below and to the west. As a result it is our recommendation that a new PEA be completed using the magnetite results to demonstrate the changes in project economics along with magnetite marketing considerations as noted below.

Since magnetite is a speciality industrial mineral, the amount that can be marketed and its price per metric tonne can only be determined by definition of sales agreements and marketing studies. Consequently the actual economic cut off for magnetite at the Bronson Slope cannot quantitatively be defined at this time. In the marketing study by Klein (2008), he reports that magnetite for one operation, Craigmont, was sold at $211 per tonne delivered to coal operations at Elkford, BC. Klein further indicates that the net cost of magnetite, less freight delivery costs, is in the order of $127 per tonne. Klein also notes that magnetite prices, as iron ore, have been up to $200 per tonne.

The 2% magnetite cut off highlighted in the above tables and used as a base case for resource estimation is based on flotation milling of tails from the Bronson Slope copper-gold-silver operation and truck transportation cost information supplied by Skyline Gold (Jensen 2010a) balanced against what is believed projected conservative sale price of $130 per tonne net of milling and transportation charges. This price is based upon a western Canada magnetite market analysis (Klein 2008) – as discussions with Craigmont Mines that produces magnetite powder for dense media separation coal cleaning. The estimated break-even cut-off grade is less than 2% magnetite on a by-product basis. It must be stressed that the economics are also dependent on the ability to market and sell magnetite. No metallurgical recoveries have been applied to the resource.

Skyline Gold's internal analysis (Jensen, 2010b) estimate that a 7% weight magnetite cut-off based on the 2009 Bronson slope PEA (Lawrence and Seen 2009) and the 2008 magnetite study (Klein 2008) is potentially viable on a stand-alone production basis and assuming that magnetite is the only mineral produced at Bronson Slope. At this cut-off Bronson Slope Measured and Indicated magnetite resource is 100,640,000 tonnes grading 8.8% magnetite for a total Measured and Indicated estimate of 8,900,000 tonnes of contained magnetite. It is reasonably expected that, on a by-product production basis, the magnetite economic cut-off grade would be lower and the magnetite resource estimate at lower cut-off grades is shown in the above tables.

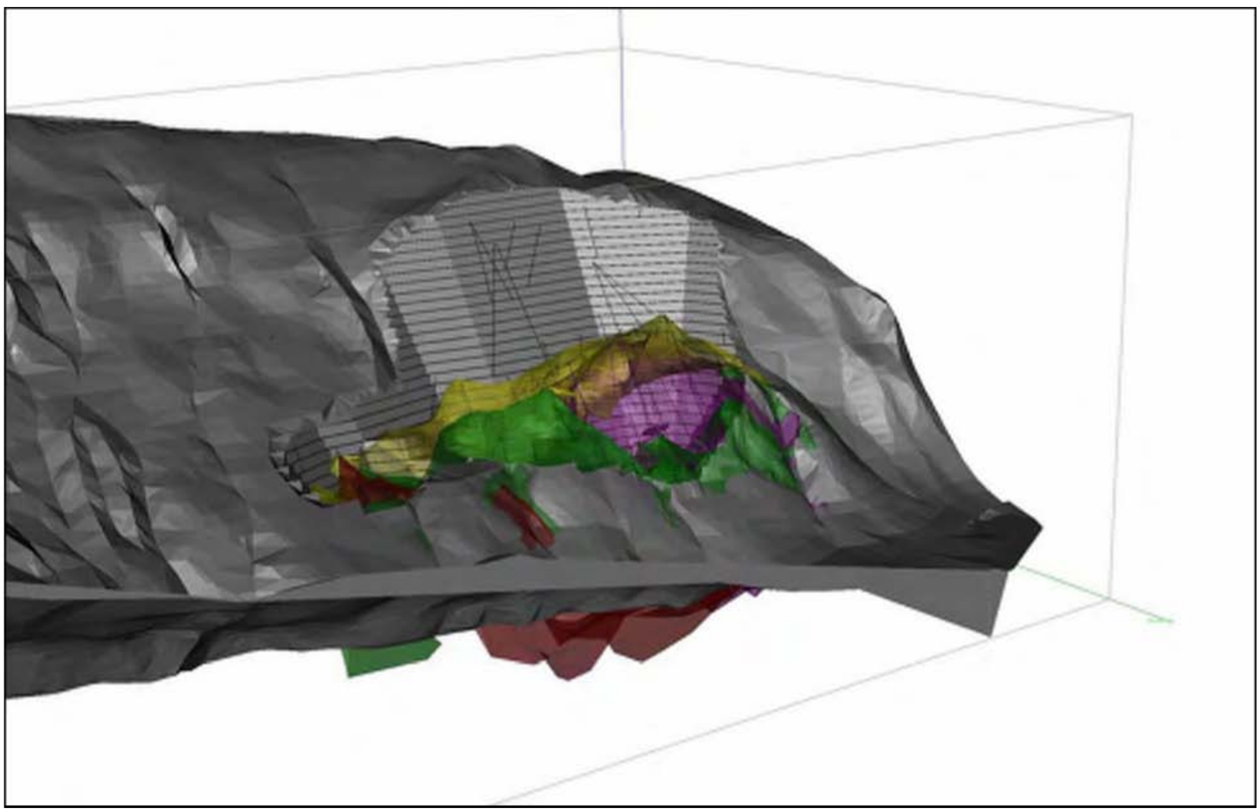


Figure 17-5
Isometric View Looking Southwest Showing Topography and Proposed Open Pit as Designed by Leighton Asia (Lawrence and Seen 2009) – Note Figure 17-1 for Rock Codes

*The relative proportions of the total magnetite resource present within the various rock domains is tabulated below In **Table 17-9** at a 2.0% magnetite cut-off for measured plus indicated blocks.*

TABLE 17-9
GLOBAL MAGNETITE RESOURCE (M + IND) IN VARIOUS ROCK DOMAINS

Domain	Classification	Magnetite Cut-off %	Tonnes above Cut-off	% Magnetite	Tonnes* Magnetite	% of Total Magnetite
QM	M+I	2.00	72,240,000	8.45	6,100,000	51.52
PPY	M+I	2.00	81,830,000	6.71	5,490,000	46.37
US	M+I	2.00	6,120,000	3.39	210,000	1.77
HW	M+I	2.00	1,520,000	2.57	40,000	0.34

Rounding has been applied to the measured plus indicated resource tonnages; consequently the total of all Domain types does not equal the measured plus indicated resource of **Table 17-7*

19.2 Pit Delineates Resources

A review of the annual quantities of Magnetite that are scheduled for production appears to be in excess of what the market can absorb at the premium magnetite price. The assumption is that excess magnetite product can be sold as high grade iron ore. This value is estimated at $50/ tonne at the mine gate and will represent 50% of the overall magnetite production. The average value for Magnetite over life of mine is estimated at $90/tonne.

Pit delineated resources are based on the following mineral prices to develop a pit shell which determined the outline of the design pit for reserve determination. **Error! Reference source not found.** summarizes the total mineral prices used in the LG calculation of the economic ultimate pit shell.

Table 19-4 Mineral prices for Reserve Shell

Metal	Prices
Cu	$2.50/lb
Au	$900/t.oz
Ag	$15/t.oz
Magnetite	$90/ton

The Mineral reserves are based on the ultimate design pit and are summarized in **Error! Reference source not found.** below.

Table 19-5 LOM Reserves

Phase 1, 2 & 3 (Total) - Tonnes and Grades ($9/t NSR Cut-off)					
Category	**Metric ktonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Mag %**
Total MII	**191,835**	**0.116**	**0.343**	**2.13**	**5.3**
Total Waste	147,499	**Strip Ratio: 0.77 Waste t/Mill Feed t**			
Total Mill Feed and Waste	**339,334**				

20.0 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. The description below reviews the previous studies referenced in the Leighton 2009 study as well as other work used in this PA update. Table 20-1 details a brief description of what each of the reports referenced by Leighton includes.

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.

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.

20.1 Cost Estimation Accuracy

This report provides a revised capital and operating cost estimate for the project based on more up to date operational constraints, and revised unit cost estimates for mining operations. The LAL 2008 plant capital and operating costs have not been updated. 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.*

MMTS 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.*

This PA Update has increased the mill processed tonnage from 87,342 ktonnes to 191,835 ktonnes, an increase of 120%. The tailing storage sight described above by Leighton does not have sufficient capacity to accommodate all the tailings. An alternate site at Bug Lake has been identified as having the capacity requirements. The dam construction is of the same order of magnitude as the LAL 2008 sights and there have been no adjustment to tailings impoundment construction costs in the 2008 report. MMTS recommends further review of this alternate location and the design concepts for the tailings facility to ensure adequacy of the design with consideration for current environmental and geotechnical standards.

20.3 Mining Rate

Throughout the LAL 2009 study, the optimization process proved 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. This rate of mining was maintained within this report in order to determine the impact of the addition of magnetite to the mine plan with a minimum of other changes. Operationally, the 12mtpa continued to make sense as this will mean 2 equal sized mining fleets

MTTS recommends proceeding with a re-design of the pit stages, re-run the production schedule, and revise the equipment selection, capital and operating cost estimates, and the project cash flow model, based on varying mining and mill throughput rates during the next phase of engineering studies.

20.4 Magnetite Recovery

The following is taken from the LAL 2008 report.

> *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 report's 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.*

The price for magnetite used in this study assumed 50% of the magnetite production could be sold as magnetite product, primarily to the coal mining industry and some specialty product, with the remaining product being sold as high grade iron ore. The weighted average price was estimated to be $90/tonne at the mine gate. A marketing study should be undertaken prior to the next phase of engineering studies.

21.0 Interpretation and Conclusions

MMTS has completed a Preliminary Assessment for the Bronson Slope property based on a wide variety of data, observations and previous technical reports. This report is an update of the LAL 2009 Technical Report with the addition of a magnetite product and the associated review of mining costs and schedules.

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 Northwestern 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

“Technical Report, Magnetite Mineral Resource Estimate – Bronson Slope Deposit” dated January 28, 2010, authored by A. A. Burgoyne, P.Eng., M.Sc., Arnd Burgert, P.Geo., B.Sc. and G. H. Giroux P.Eng., MASc**.** independent Qualified Persons as defined by NI 43-101.

“Preliminary Economic Assessment with Mining Plan and Cost Estimate for Skyline Gold Corporation on the Bronson Slope Property”, dated March 6, 2009 and authored by J. A. R. Lawrence, MAusIMM and V. Seen, MAusIMM of Leighton Asia Limited Leighton Asia Limited both independent Qualified Persons as defined by NI 43-101. This Technical report was posted to SEDAR on March 6, 2009.

These Technical Reports have provided a technical review of the Bronson Slope property including a detailed review and evaluation of the historical resource estimations and mine plans and schedules for the Bronson Slope Au-Cu-Ag-Magnetite 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 and their contractors, is of good quality and meets good practice industry standards.

Capital and operating cost estimates for the mining and processing of the Bronson Slope orebody have been reviewed and updated where noted with budget prices and quotations from various suppliers for the major conveyor and crushing equipment items. Other costs including major mining, milling and processing components, major infrastructure development such as access roads and other consumables such as power and diesel were reviewed for accuracy and consistency. MTSS have reviewed these 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 MMTS's intention the report will be read in full to ensure full comprehension of all relevant interpretations and conclusions.

22.0 Recommendations

At the completion of the mine development and scheduling, several opportunities have been identified to enhance the project return. A smaller pit or an increased mining and milling rate would improve the project return. The current 38 year life returns little NPV value in the last half of the mine life.

The following section has been taken from the LAL 2009 technical report and is included here, with MMTS recommendations, to provide a comprehensive list.

> *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 Optimization 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 modeling 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*

The following is a summary of MMTS recommendations for the Project. There is some overlap with LAL recommendations and are stated to provide increased clarity of the issues.

- Complete a magnetite study including processing capital and operating costs along with the possible market size and prices.
- Complete a review of current recoveries and the impact of lower feed grades on these recoveries.
- Initiate a study on possible metallurgical recovery of low copper grade material for gold and silver.
- 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 including truck/shovel in-pit operations, should also be considered.
- Model the three mineralized zones as individual zones; mill feed grade copper, magnetite and gold (low grade copper), primarily gold (highwall gold zone)
- Develop new economic pit shells based on varying economics for the three mineralized zones.
- Complete a study of crusher sizing including consideration of electrical vs. diesel crusher economics
- Complete a study to identify the cost savings potential of using used mining and processing equipment now available on the market

- Complete further studies comparing grinding size versus recovery to identify the optimum grind size.
- Further develop self-generation hydro projects as the basis of power supply
- 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.
- Complete further Geotechnical studies of the highwall slope including the potential for steeper pit wall angles to improve project economics
- Complete a revised Tailings Storage Facility site design, cost estimate and geotechnical viability.

MMTS recommends that the Bronson Slope project proceed to higher levels of study. This should be a progressive approach, where initial studies will develop the scope for more detailed work. The progress should start with the optimization studies already indicated above which will lead to Pre-Feasibility and Feasibility level of studies. The priority and costs of these this future work needs to be determined. At this time and estimated \$0.5 million should be budgeted for the initial optimization studies. Concurrently, \$1.5 million should be budgeted for exploration drilling, targeting higher grade gold zones. The results of these efforts will determine the scope and cost of work beyond that.

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

23.0 References

The following section has been taken from LAL 2009 technical report.

The following is a list of references utilized 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.

Burgoyne, A.A. and Giroux, G.H., 2008. Technical Report, Mineral Resource Estimate – Bronson Slope Deposit, dated April 30, 2008. – see www.sedar.com.

Burgoyne, A.A. and Giroux, G.H., 2007. Technical Report, Mineral Resource Estimate for Skyline Gold Corporation On The Bronson Slope Property, Northwestern British Columbia dated May 10, 2007 — see <www.sedar.com>.

Burgoyne, A.A., 2006. Technical Report for Skyline Gold Corporation On The Bronson Slope Property, Northwestern British Columbia dated June 1, 2006 — see <www.sedar.com>

Burgoyne, A.A., 1992. An Evaluation of the Iskut Property Northwestern British Columbia for Skyline Gold Corporation dated October 1992.

Burgoyne, A.A., 1993a: Bronson Creek IP Surveys — Memo for Skyline Gold Corporation dated August 20, 1993.

Burgoyne, A.A., and 1993b: Bronson Creek Porphyry Target Proposed Drill Locations IP Surveys — Memo for Skyline Gold Corporation dated August 1993.

Dick, R. C, 1997: Engineering Report dated 1997, R.C. Dick (Geotechnical Engineering Consultant)

Program at Bronson Slope — Internal report for Skyline Gold Corporation Forsite Engineering Ltd. (Forsite) 2008. Bronson Creek Access Road, Bridges and Environmental Considerations. Schedule K SUP Application. Prepared for Skyline Gold Corporation, Richmond, BC.

Forsite Engineering Ltd. (Forsite) 2006. Bronson Creek Access Road and Bridge Pre-Feasibility Study. Unpublished report Prepared for Skyline Gold Corporation Richmond, BC.

Ganshorn, A. J.,1997: Marketing Report with appendix dated May 1, 1997, Ganshorn, Arthur J.

Ganshorn, A. J., 2007: Revised Marketing Costs — Transportation, dated March 12, 2007, J. Arthur Ganshorn (Marketing Consultant).

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

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

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).

Klein, B., 1997: Acid Base Accounting (ABA) Test Results", dated April 29, 1997, Bern Klein (Process Research Associates).

Knight Piesold Ltd, 1997: Long Term Water Treatment Options dated July 1997, Hallam Knight Piesold Ltd.

Liu, Q., 1997: Metallurgical Report dated July 18, 1997, Qi Liu (Senior Metallurgist) Process Research Associates.

MacDonald, A. J., Van Der Heyden, P., Alldrick, D.J., and Lefeubre, D. 1992: Geochronology of the Iskut River Area — an update. In Geological Fieldwork 1991. British Columbia Ministry of energy, Mines and petroleum Resources, Paper 1992-1, p495-501.

MINFILE Record Summary No 104B Bronson Slope, Red Bluff, Ministry of Energy, Mines and Petroleum Resources. The Map Place web page:
http://www.em.gov.bc.ca/Mining/Geolsurv/Minfile/App/Summary.aspx?minfilno=104B%

MINFILE Record Summary No 104B 008, Eskay Creek, Mackay, Eskay, Ministry of Energy, Mines and Petroleum Resources. The Map Place web page:
http://www.em.gov.bc.ca/Mining/Geolsurv/Minfile/App/Summary.aspx?minfilno=104B%

MINFILE Record Summary No 104B 107, Johnny Mountain. Ministry of Energy, Mines and Petroleum Resources. The Map Place web page:

http://www.em.gov.bc.ca/Mining/Geolsurv/Minfile/App/Summary.aspx?minfilno=104B%

MINFILE Record Summary No 104B 250, Snip, Twin... Ministry of Energy, Mines and Petroleum Resources. The Map Place web page: http://www.em.gov.bc.ca/Mining/Geolsurv/Minfile/App/Summary.aspx?minfilno=104B%.

MINFILE Record summary No 104B 054, Premier, Silbak Premier, Premier Gold, Silbak Premier Mine, Salmon-Bear River Mining, Salmon Gold: http://www.em.gov.bc.ca/Mining/geolsurv/minfile/App/Summary.aspx?minfilno=104B++054

Moore, M. J., 1997b: Bronson Slope Drilling Assessment Report for International Skyline Gold Corporation dated January 1997.

Parsons, G., 1965: Geological Report on Bronson Creek No. 1-3 Claim Group, Assessment

Report 00769 Piteau Associates, 1997: Bronson Slope Project Preliminary Geotechnical Assessments and Slope Design Studies for the Bronson Slope Open Pit dated March 1997

Process Research Associates Ltd., 1997: Metallurgical Study On The Bronson Slope Samples by Qi Liu, Ph.D. and Bern Klein, Ph.D., dated July 18, 1997

Raymond, G. P., 1997: Geostatistical Study of Drilling to May, 1997 and Recommended Drill

Rescan, 1995: Preliminary Feasibility Study - Mill Facilities dated March 27, 1995, Rescan Engineering Limited.

Rhys, D., 1995a: Memo to D. Yeager re Red Bluff Porphyry (Bronson Slope Project) — geology, core library and core logging procedures for International Skyline Resources dated May 1995.

Rhys, D.A., 1995b: The Red Bluff gold-copper porphyry and associated precious and base metal veins, north-western British Columbia in Porphyry Deposits of the Northwestern Cordillera of North America, pages 838 — 850, editor T. G. Schroeter, Special volume 46, Canadian Institute of Mining, Metallurgy.

Yeager, D.A., 2006: Personal Communications

Yeager, D. A., 2003: Summary of Exploration Activities and Recommended Exploration Programs On the Iskut River Property of Skyline Gold Corporation, September 17, 2003

Yeager, D. A., 1998b: Geology of the Bronson Slope Porphyry Gold, Copper Deposit for International Skyline Gold Corporation dated February 1998.

Yeager, D., 1997a: Bronson Slope Deposit, In-fill and Exploration Drilling, Excel Spreadsheet, PLAN97.XLS

Yeager, D. A., 1997b: Compilation of Drill Holes Assays, Drill Hole Assay Composite Table and Metallurgical Composite Sample Listings from the Bronson Slope Gold, Copper, Silver,

Yeager, D. A., 1994: Summary of Exploration Work on the Red Bluff Area, Bronson Slope Project of International Skyline Gold Corporation dated October 20, 1994 Woznow, D.P. and Yeager, D.A., 1999: Closure Plan for The Johnny Mountain Gold Mine, Reclamation Permit No. M —178 for International Skyline Gold Corporation dated October 13, 1999

Yeager, D., 1995: Mine Plan dated September 22, 1995, Christopher Turek (P.Eng) & David Yeager (P.Geo).

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

Costmine 2008: Costmine — Mining Cost Service, InfoMine USA Inc., 2008 Costmine 2007: Costmine — Mining Cost Service, InfoMine USA Inc., 2007

"Galore Creek Project Feasibility Study", Nova Gold Canada Inc., NWBC, Oct 31, 2006 "Hard Rock Miner's Handbook", Edition 3, McIntosh Engineering, May 2003

Mining Association of British Columbia, http://www.mining.bc.ca/, 2007, 2008

British Columbia Ministry of Transportation and Infrastructure, http://www.gov.bc.ca/tran/, 2007, 2008

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 separatorConveyor 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

MMTS has contacted various vendors, and used internal information to update some of the mining components of this study. At a Preliminary Assessment (scoping) level of study, these updates have not been referenced as formal reference documents.

24.0 Date and Signature Pages

I, G H Giroux, of 982 Broadview drive, North Vancouver, British Columbia, do hereby certify that:

1. I am a consulting engineer with an office at #1215 – 675 West Hastings Street, Vancouver, British Columbia,.
2. I am a graduate of the University of British Columbia in 1970 with a B.A. Sc. and in 1984 with a M. A. Sc., both in Geological Engineering..
3. I am a member in good standing of the Association of Professional and Geoscientists of the Province of British Columbia.
4. I have practiced my profession continuously since 1970. I have had over 30 years experience calculating mineral resources. I have previously completed resource estimations on a wide variety of deposits many similar to Bronson Slope.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by reason of education, experience, independence, and affiliation with a professional association, I meet the requirements of an Independent Qualified Person as defined in National Instrument 43-101.
6. I am responsible for the mineral resource estimates for this report titled “Preliminary Assessment Update – Bronson Slope Property, for Skyline Gold Corporation” dated November 5th, 2010, as stated in Section 19. This is based on a study of the data and literature available on the Bronson Slope Property. I have not visited the property.
7. I have previously completed resource estimates on this property in 1996, 2007, 2008, and 2010. The Technical Reports for the most recent of these estimates are listed in this report.
8. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.
9. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

Dated November 5, 2010

“Signed”

Signature of Qualified Person

G. H Giroux P.Eng., MASc.
Print Name of Qualified Person

I, James H Gray. P.Eng., do hereby certify that:

1. I am a Principal of Moose Mountain Member Corp., 1584 Evergreen Hill SW Calgary, Alberta Canada T2Y 3A9.
2. I graduated with a Bachelor of Applied Science in Mining Engineering from the University of British Columbia in 1975.
3. I am registered by The Association of Professional and Geoscientists of the Province of British Columbia, registration number 11,919, and the Association of Professional Engineers, Geologists and Geophysicists of Alberta (M47177).
4. I have worked as a Professional Engineer for over 25 years since my graduation from university.
5. 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.
6. I am responsible for the technical report “Preliminary Assessment Update – Bronson Slope Property, for Skyline Gold Corporation” November 5^{th}, 2010.
7. I have visited the Bronson Slope Property on July 5^{th} 2010.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in section 1.4 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated November 5, 2010

“Signed”

____
Signature of Qualified Person

James H Gray P.Eng.
Print Name of Qualified Person

25.0 Additional Requirements for Technical Reports on Development Properties and Production Properties

25.1 Mining Operations

25.1.1 Available Hours and Utilization

The following section 25.1.1 is taken from the LAL 2008 report and has been reviewed and is considered accurate and applicable in its entirety.

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 utilization 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 summarized in Table 25-1 Summary of Delays and the Resulting Utilization. It is envisaged that, with appropriate planning, 6 days can be utilized for opportune maintenance on the mining equipment reducing the burden.

Table 25-1 Summary of Delays and the Resulting Utilization

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**
Utilization	**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

The italicized statements in the following section 25.1.2 have been taken from the LAL 2009 report and updates have been inserted where applicable.

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 ore body 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 utilized 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 utilized in a mining environment to date. This conveyor system is being utilized 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.

Correspondence with the manufacturer provided confirmation of the uncertainty of operational capability of the enclosed belt conveying system discussed during cold weather. Therefore the operating costs for in-pit mobile conveying systems coupled with transfer points to more conventional sandwich conveyor technology are developed to compare with the costs in the LAL 2009 report. These unit costs are discussed in the operating cost section.

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 25-1below:

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.*

This report uses the cost of more traditional sandwich conveyors to provide a cost comparison with the LAL 2009 report.

> ***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.*

The above statement is still a reasonable assumption, especially given the high snowfall region and long downhill runs with switchbacks that would require the trucks to operate at slower speeds thus increasing the costs and risks even further.

> ***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.*

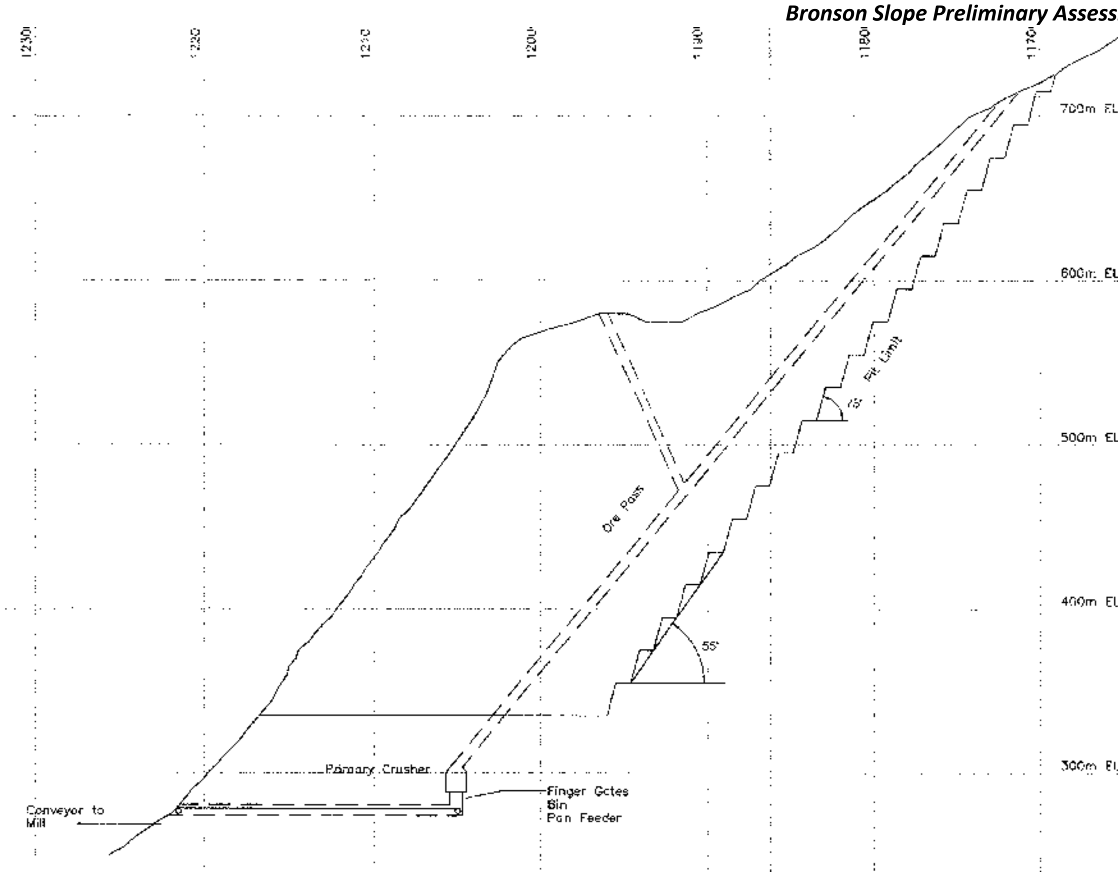


Figure 25-2: Ore Pass Concept

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

To facilitate optimization, mine design and reserve calculations the project was converted into Minesight™ using the following steps. The Project data was received from Gary Giroux on May 25, 2010 as ASCII files (BRONSON SLOPE May25 2010 BLOCK MODEL.CSV) and included recent Magnetite assays and existing gold, silver, molybdenum and copper assays. The Magnetite assays were sampled with different assays sample lengths, and require separate drillhole data files from the Au-Ag-Cu-Mo assays.

The model also required conversion from Mine grid to UTM. The data received from Skyline resources and Giroux was projected and used within a specific MineGrid . End of Mine Life topography was provided by Skyline as determined by the *2008* proposed mine life plan which included mined out area and dumps. However the topography supplied was not available over the expanded mine model. Accurate updated Topography was requested from Eagle Mapping and received on June14, 2010 to construct a starting surface for optimization purposes. It was calculated that the following transposition was necessary to work with the project data within UTM Nad83 grid.

<u>from minegrid to utm</u>
rotation 26.4355 degrees
shift y 6270334.36m
shift x 346283.68m

Table 25-2 Minesight Project Limits for Bronson Slope

	Minegrid	UTM NAD83
EASTING	24800 - 26400	371083 - 372683
NORTHING	11300 - 12500	6281634 - 6282834
ELEVATION	-180 - 900	-180 - 900

The May 25 csv block model was transposed with the East (X) and North(Y) shifts, and loaded into a new transposed MineSight project before the project was rotated. Drillhole data collars were transposed and rotated, then loaded into the rotated MineSight project.

The January 2010 model file included the 3D block information with all the 2008 previous model information for Au, Ag, Cu and Mo, in addition to the recently 2010 interpreted Magnetite by Giroux.

Following the importing of the May25 csv model file into MineSight, a resource based on 2% MAG COG was calculated for reconciliation purposes. The results (difference < 1%) indicate the transfer of data was successful.

25.2.2 Resource Estimate Comparison

The resource estimates used for this study has been taken from two separate technical reports. The first is 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. The second is the Technical report titled “Magnetite Mineral resource Estimate – Bronson Slope Deposit” dated January 12, 2010, authored by G. H Giroux, P. Eng., from Giroux Consultants Ltd, Arnd Burgert, P Geo., B.Sc from Arnd Burgert Consulting Ltd and A. A. Burgoyne, P.Eng., M.Sc. from Burgoyne Geological Inc., independent Qualified Persons as defined by NI 43-101. Both of these technical reports can be viewed at www.sedar.com. These resource estimates have been included in Item 19.

MMTS performed a block model report calculation to determine how the imported block model tonnes and grades compare with these resource estimates. The Resource estimate for Au, Ag and Cu calculated using the newly generated block model has been provided in Table 25-3. This table also summarizes the difference between the new model and the resource estimates from the reports noted above. The comparison table below uses the same imported USD NSR value generated by Giroux so that a direct comparison can be made. Subsequent MMTS work uses a revised CAD NSR.

Table 25-3 Minesight Resource (USD 9.00 NSR cut-off)

MMTS Bronson Slope Resource Estimate (Cutoff USD 9/t NSR)				
Category	**Metric Tonnes**	**Au g/t**	**Ag g/t**	**Cu %**
Measured	74,734,000	0.45	2.31	0.17
Indicated	150,150,000	0.31	2.17	0.13
Inferred	91,590,000	0.27	1.76	0.13
Total Measured + Indicated	**224,893,000**	**0.36**	**2.22**	**0.14**
G Giroux 2008 Resource Estimate				

Measured	74,800,000	0.45	2.31	0.17
Indicated	150,300,000	0.31	2.17	0.13
Inferred	91,600,000	0.27	1.76	0.13
Total Measured + Indicated	**225,100,000**	**0.36**	**2.22**	**0.14**
MMTS variance from 2008 Resource Estimate				
0.0%Measured	0.1%	0.0%	0.0%	0.0%
Indicated	0.1%	0.0%	0.0%	0.0%
Inferred	0.0%	0.0%	0.0%	0.0%
Total Measured + Indicated	**0.1%**	**0.0%**	**0.0%**	**0.0%**

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 Minesight. All pit optimization, design and reporting of grades, volumes and tonnages throughout this study have been completed using the Minesight block model and the base case metal prices. The mill feed cut-off has been calculated based on a USD 9.00/t NSR.

25.2.3 Pit Optimization and Strategic Planning

Pit shells were developed using the Minesight program which uses the Lerchs-Grossmann (LG) algorithm. A net smelter return (NSR) value of each block in the block model is calculated using the minerals expected selling price, with external costs such as smelter, refining and transport costs being deducted from the price, to establish available mining and processing costs at the mine gate. This establishes a Net Smelter Price (NSP) for each metal for use in the LG runs. Defined process recoveries and processing, mining and G&A operation costs are then deducted from the available income to determine the NSR value of the block. Mining constraints such as pit slopes are then applied and the blocks are progressively mined, from the top down. The resulting pit shell will be defined by mining the blocks that adhere to all imposed constraints.

A series of cost sensitivities are then generated to develop a set of nested pit shells based on varying revenue factors. This series of nested pit shells are then used to help identify the most effective sequence to mine the ore body and optimum ultimate pit limit.

The initial step is to understand the impact of the addition of Magnetite to the overall pit shell and shape. The figure below shows the effect of magnetite pricing from $0 at the mine gate to $130. The base case pit shell is shown with the red zone identifying the pit expansion area due to magnetite. All other product prices and operating costs are kept constant and only the value of magnetite was increased. The black pit outline is the crest of the design pit from the Leighton study to illustrate changes to the overall pit expansion.

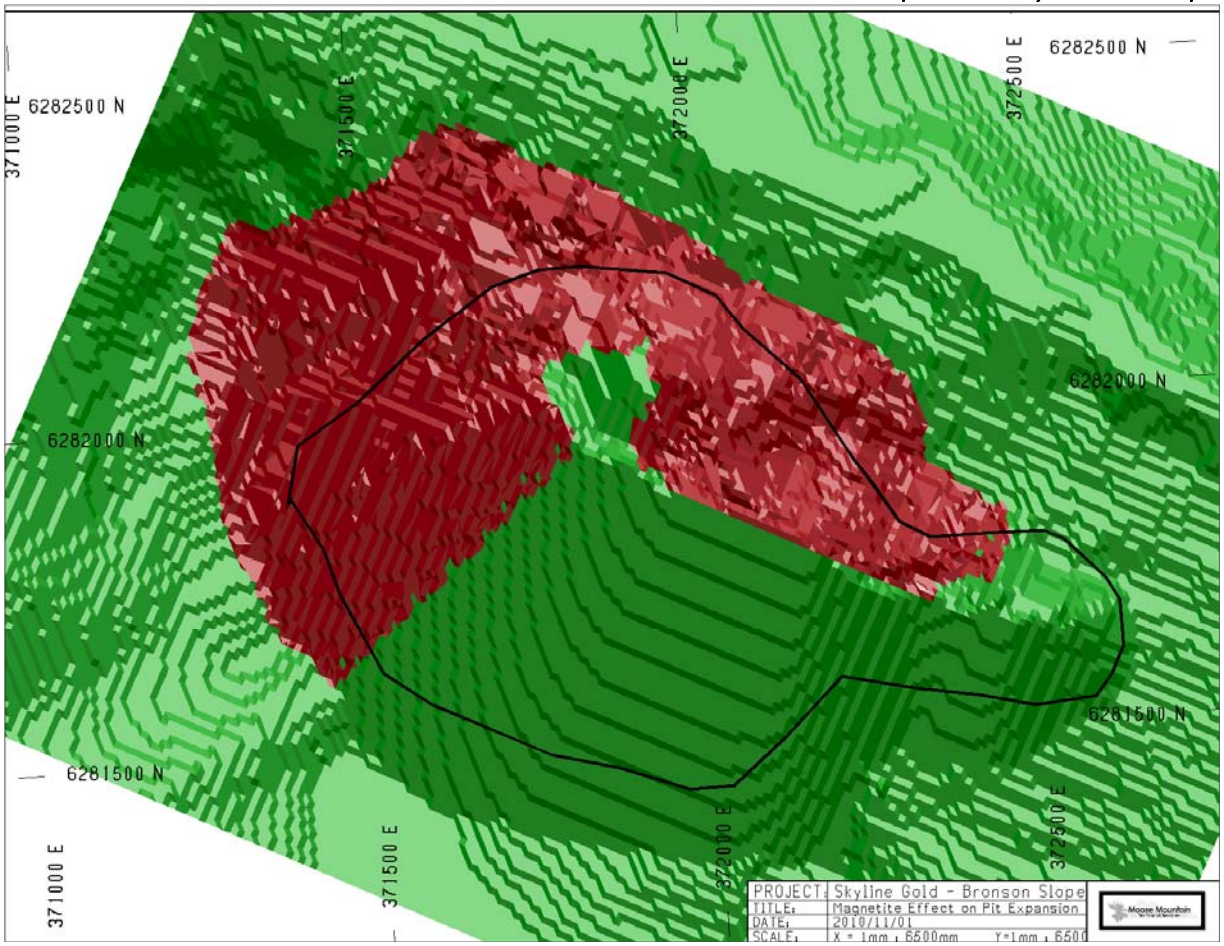


Figure 25-3 Magnetite Effect on Pit Expansion

As shown in the figure above, although the addition of Magnetite has little effect on the south and east pit limits, it is a significant influence on expansion to the north and northwest limits.

25.2.3.1 **Pit Shell Optimization**

The MineSight 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. This base case scenario is then evaluated through a series of iterations to identify the best operational and financial performance for the project. This normally involves evaluating the most appropriate number of phases, mine life, production rates and strategic scheduling of the benches within the mining phases. For the purposes of this report update, the mill throughput rate has been held constant with the LAL 2008 report and the mine life is determined by the projected quantity of milled ore. The most optimum sequence and schedule is then selected and the results output used for further analysis and pit design review.

25.2.3.2 Inputs and Drivers for the Pit Shell Optimization

Various pit optimization 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 initial stages and the most optimum ultimate pit in order to develop a high level strategic schedule representative of the best return for the project that can physically be achieved.

A series of sensitivities on variables are run to determine how the pit shells are affected by changing these variables. The effect of some variables is primarily for discussion, such as the slope sensitivity, which gives an indication of the effect of slope change but is designed at 50 degrees for this study. The operating cost variance can be a significant indicator of optimum pit shells and stages by looking for inflection points on the associated graph. Theses inflection points are indicators of the interaction between topography and the mineral zones. The final pit shell is generally determined by establishing the "best" base costs to use based on the inflection point of the cost sensitivity graph. The phases take into account not only the economic pit shells but also optimize the surrounding topography for both access and size constraints.

The design parameters used for the base case and to establish the NSR value for each block are shown in Table 25-4.

Table 25-4 Base Case Design Parameters

Parameters Description	
Metal Prices:	
Cu	US$2.50/lb
Au	US$900/t oz
Ag	US$15/t oz
Magnetite	USD$90/tonne
Geotechnical design parameters	
Average - All domains	50 degrees --Adapted from Piteau Report
Mining parameters	
Bench Height (m)	20
Mining cost - Base ($/t)	CDN$1.77/t
Adjustment for depth ($/t/bench) Mining	not considered
dilution (%)	5%
Mining recovery (%)	95%
Concentrator parameters	
Expected Mill Throughput (t/annum)	5.1Mtpa
Variable processing cost ($/t milled)	$8.89/t
Concentrator Recoveries	
Au	84%
Ag	61%
Cu	86.6%
Magnetite	95%
Smelting and refining	
Smelting recovery(s) (%)	97%
Refining recovery(s) (%)	100%
Smelting cost ($/t concentrate)	$85/t
Refining cost(s)	$0.075/lb Cu; $6.00/oz Gold; $0.4/oz Silver
Concentrate Transport costs ($/t)	$50/t of concentrate
Concentrate moisture content (%)	8%
Marketing cost(s)	Nil
Sales cost(s)	
Sales commission(s) (%)	
General and Administration Overhead costs	
Admin and Overhead unit cost ($/t milled)	$1.00/t milled

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 (mined in 10m benches)
- Batter Angle — 75°

- Berm Width — 11.2 to 11.4m
- Overall Wall Angle — 50°

25.2.3.3 Pit Shell Sensitivity Runs and Results

Slope sensitivities were run using the base cost operating cost and prices. Slopes of 55, 50 and 45 were used. Table 25-5 below shows the LG pit shell reserves using a $9.00 NSR cutoff.

Table 25-5 Slope Sensitivity Results

Highwall Slope Angle	ORE Tonnage (kTonne)	AU g/t	AG g/t	CU lb/t	MAG %	Total Waste (kTonne)	Stripping Ratio	Total Tonnage (kTonne)
55	276,588	0.341	2.173	0.109	4.16	136,732	0.49	413,320
50	257,892	0.348	2.166	0.105	4.44	145,892	0.57	403,784
45	248,304	0.354	2.159	0.102	4.50	174,074	0.70	422,378

Table 25-5 above is graphed in Figure 25-4 below. There is a 4% loss in ore reserves going from 50 to 45 degree wall and a 19% increase in the waste tonnes.

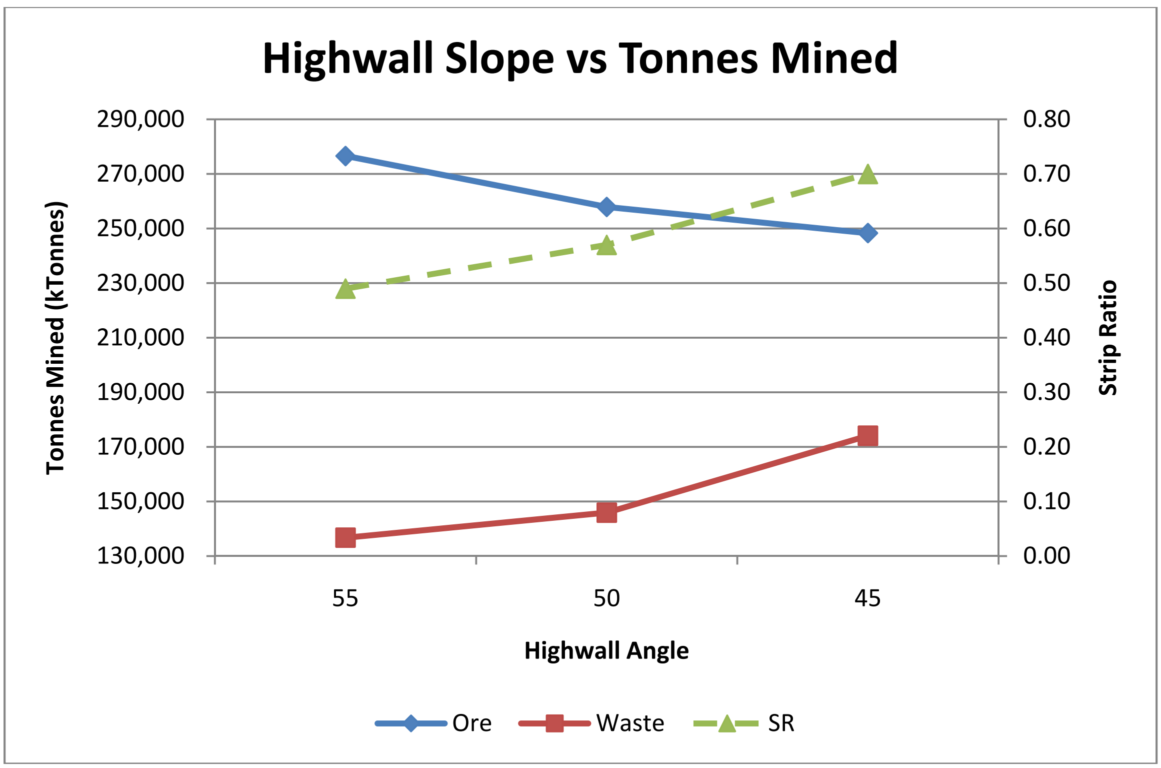


Figure 25-4 Graph of Highwall Slope Sensitivity

Operating cost sensitivity run, in 10% increments from -40% to +40% of base costs uses inputs shown in Table 25-6 below. The base case is variable 6 and the mineral prices are NSP at the mine gate.

Table 25-6 Input Variables for Cost Sensitivity

Variable	1	2	3	4	5	6	7	8	9	10
?01	Au NSPrice	29.88	29.88	29.88	29.88	29.88	29.88	29.88	29.88	29.88
?02	AG NSPrice	.449	.449	.449	.449	.449	.449	.449	.449	.449
?03	CU NSPrice	2.448	2.448	2.448	2.448	2.448	2.448	2.448	2.448	2.448
?04	MAG Price	90	90	90	90	90	90	90	90	90
?05	Slope Angle	50	50	50	50	50	50	50	50	50
?06	Proc & GA Cost	5.33	6.22	7.11	8.00	8.89	9.78	10.67	11.56	12.45
?07	Mining Cost	1.059	1.236	1.412	1.588	1.765	1.942	2.118	2.295	2.471
?08	Resultant Pit	29	28	27	26	25	24	23	22	21
?09	Limit Pit	00	29	28	27	26	25	24	23	22

The following Table summarizes the LG pit shell resources for each variation in operating costs.

Table 25-7 Operating Cost Sensitivity Results

Operating Costs	ORE (kTonne)	AU g/t	AG g/t	CU lb/t	MAG %	Total Waste (kTonne)	Stripping Ratio
40%	149,995	0.392	2.16	0.121	5.41	67,518	0.45
30%	179,029	0.379	2.19	0.115	5.22	87,418	0.49
20%	201,779	0.369	2.19	0.111	5.05	102,218	0.51
10%	225,218	0.359	2.19	0.108	4.81	118,368	0.53
Base	257,892	0.348	2.17	0.105	4.44	145,892	0.57
-10%	299,693	0.339	2.15	0.104	3.93	202,351	0.68
-20%	327,602	0.334	2.12	0.103	3.65	258,056	0.79
-30%	346,649	0.332	2.09	0.103	3.48	319,508	0.92
-40%	361,035	0.330	2.07	0.102	3.37	385,748	1.07

The above Table is graphically shown in Figure 25-5 Operating Cost Sensitivity GraphFigure 25-5 below. There are 2 inflection points to note. From 10% to -10% the steeper line indicates a larger increase in ore tonnage for the incremental changes than above the -10% operating cost. This indicates that using operating costs 10% less than base case would be the optimum pit shell to achieve. Taking into account the size of potential tailing capacity it was decided to use the base price shell as the ultimate pit shell for design purposes.

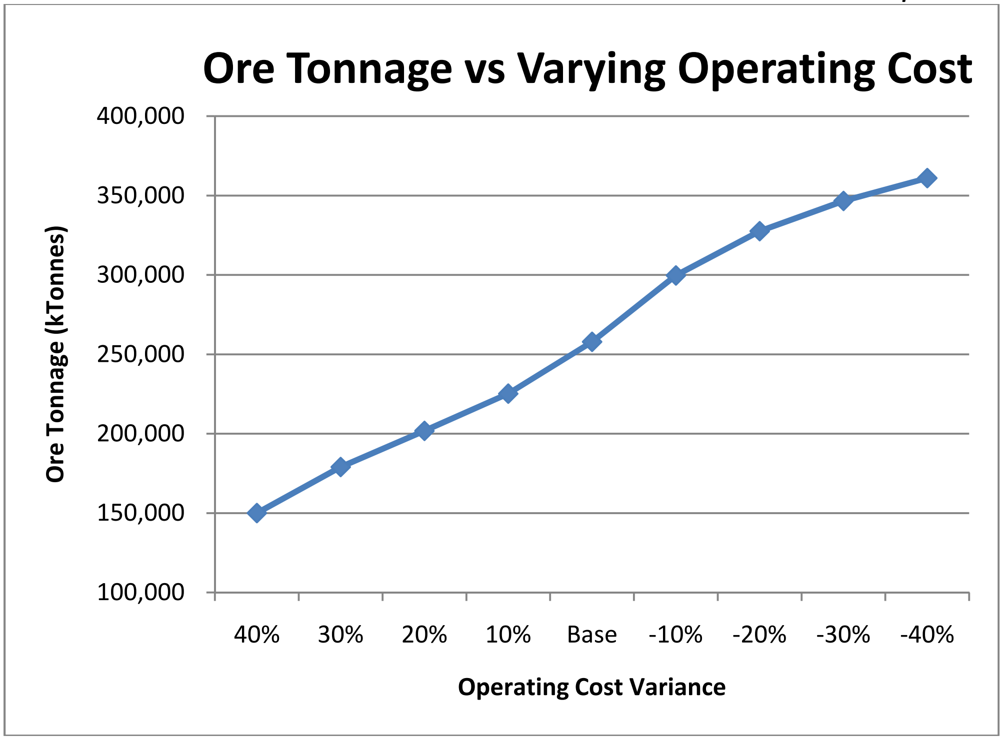


Figure 25-5 Operating Cost Sensitivity Graph

25.2.3.4 **Pit Shell and Strategic Shell Progression**

The base case operating costs use significant in-pit crushing and conveying over more traditional mining methods. A number of production assumptions and scaling up costs from quarry operations were used to forecast these costs. The risk of using the reduced the operating costs of 10% to achieve the inflection point on the graph, was considered too high for the gain in ore. Further review of the limited waste and tailing storage capacities on site was also taken into consideration. The ultimate pit shell chosen is based on the more conservative +10% increase in operating costs case. This pit shell pushes the south highwall back from the LAL 2009 design by, on average, 60 meters

The LAL 2009 report identified that 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 was too narrow to achieve realistically using the mining equipment selected so therefore the two phase scenario was adopted by LAL for the Bronson slope project. The addition of the 60 meters of mining width in this study will allow a 3 phase mine to be a reasonable operating scenario.

The first phase will take advantage of the natural plateau at the 590 elevation, above the mill site as a starting pit. This will provide earliest ore to the mill, and leave highwall access for the subsequent stages so the pits can be linked as soon as possible.

25.2.4 **Detailed Pit Design**

The results from the pit sensitivities are used as guidelines to generate starter and final detailed pit designs. A number of factors are evaluated such as access roads, 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 Topographical and 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. Whenever possible, interim highwalls are designed at the more conservative 45 degree angle by designing access roads in the temporary highwalls. 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 modeling of geotechnical risks is completed prior to further development and commissioning of the project.

Access roads for mine development and ongoing transportation of men and equipment are integrated into the design. Pit 1, the starter pit, is designed to take advantage of starting on the naturally occurring plateau above the proposed mill site and leaves a 13m wide access road in the highwall. Initial access to Pits 2 and 3 will be a 9 kilometer 8% road along the west face as shown below. Possible routes are shown in Figure 25-6.

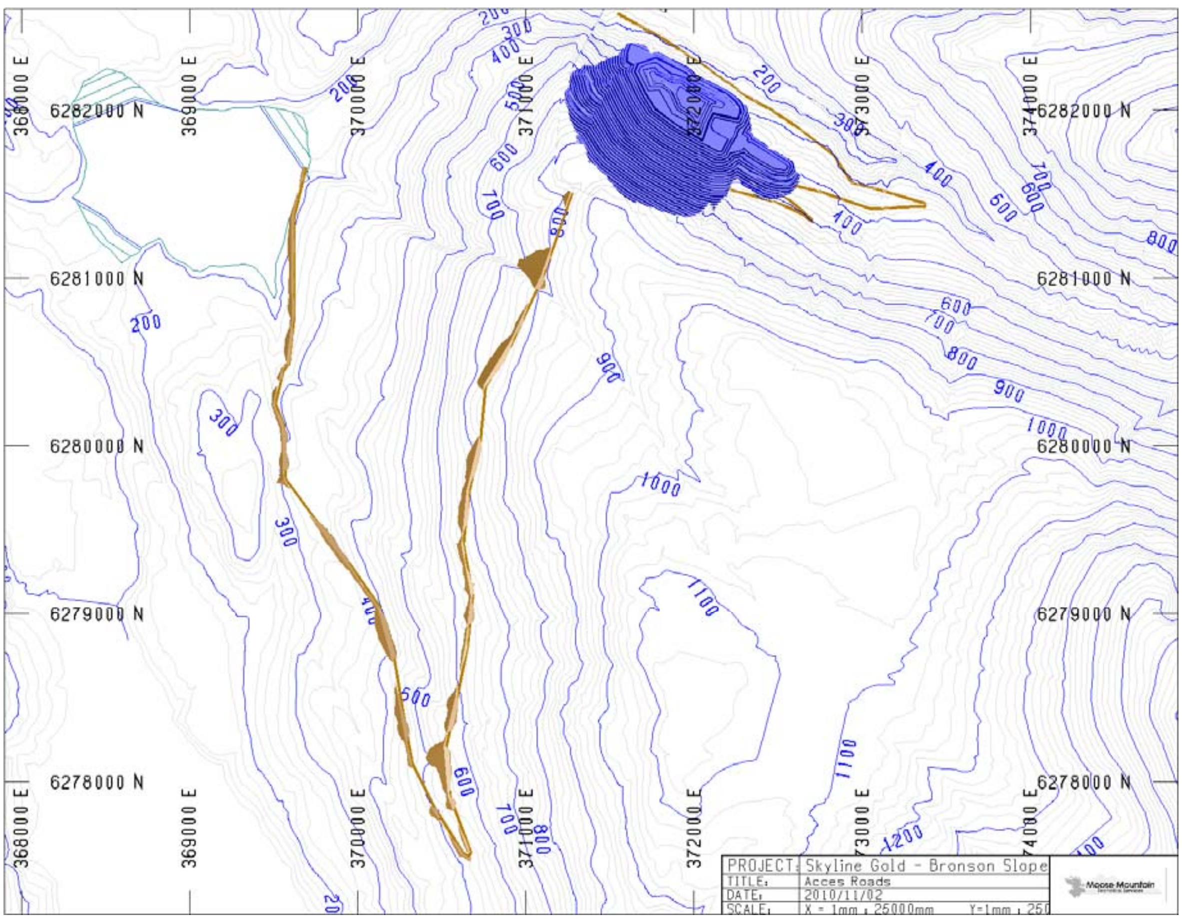


Figure 25-6 Pit 1 Access Road Route (In Relation to Final Pit Limit)

25.2.4.2 Pit Design Graphics

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

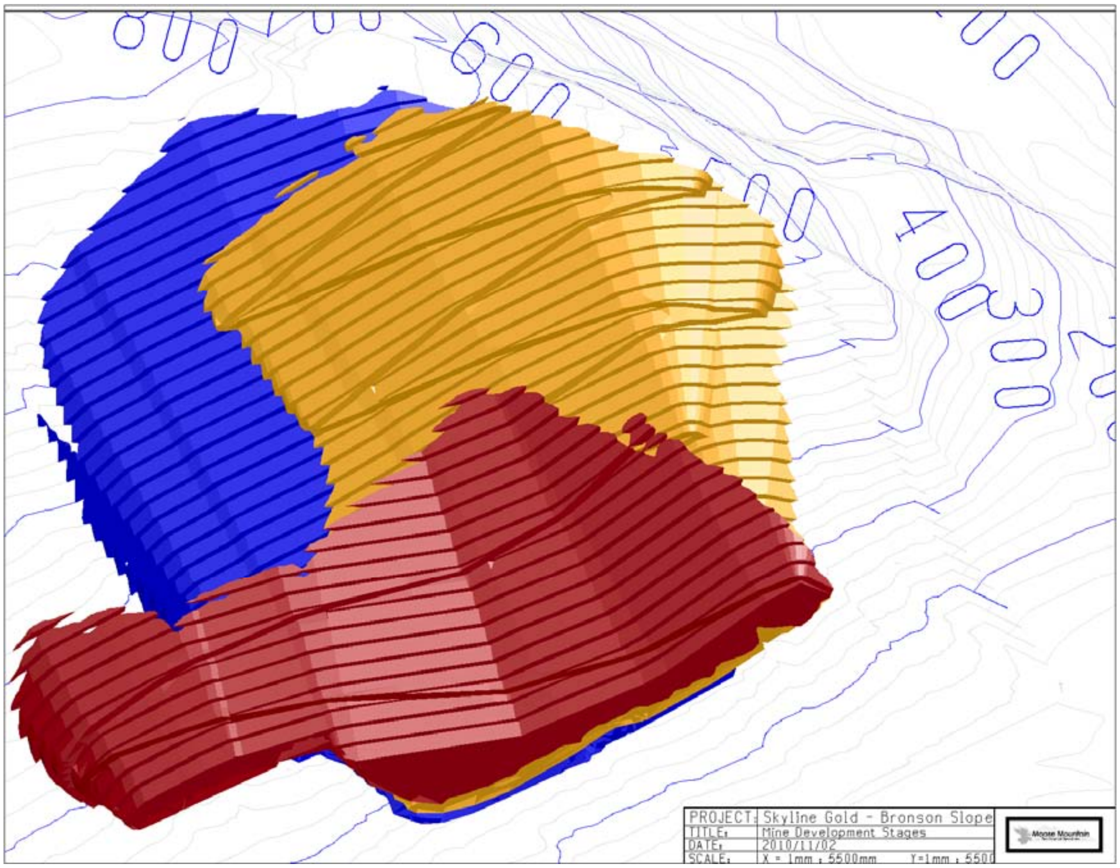


Figure 25-7 Stacked Detailed Pit Designs

Pit 1 is a 70 meter wide pit and requires careful scheduling with Pits 2 and 3 when they are active above. Pit 1 incorporates the small area to the east of the main pit in order to minimize switchbacks on the highwall road and provide approximately 50% of the strike length to be east of Pit 2. Pits 2 and 3 are also staggered to allow areas of mining that are not below the active pit above.

Figure 25-8 to Figure 25-10 show the detailed pit designs individually. A series of representative cross sections showing the aggregate Cu (%) and Au (gpt) grades within the pit designs have been provided in Figure 25-11 to Figure 25-20

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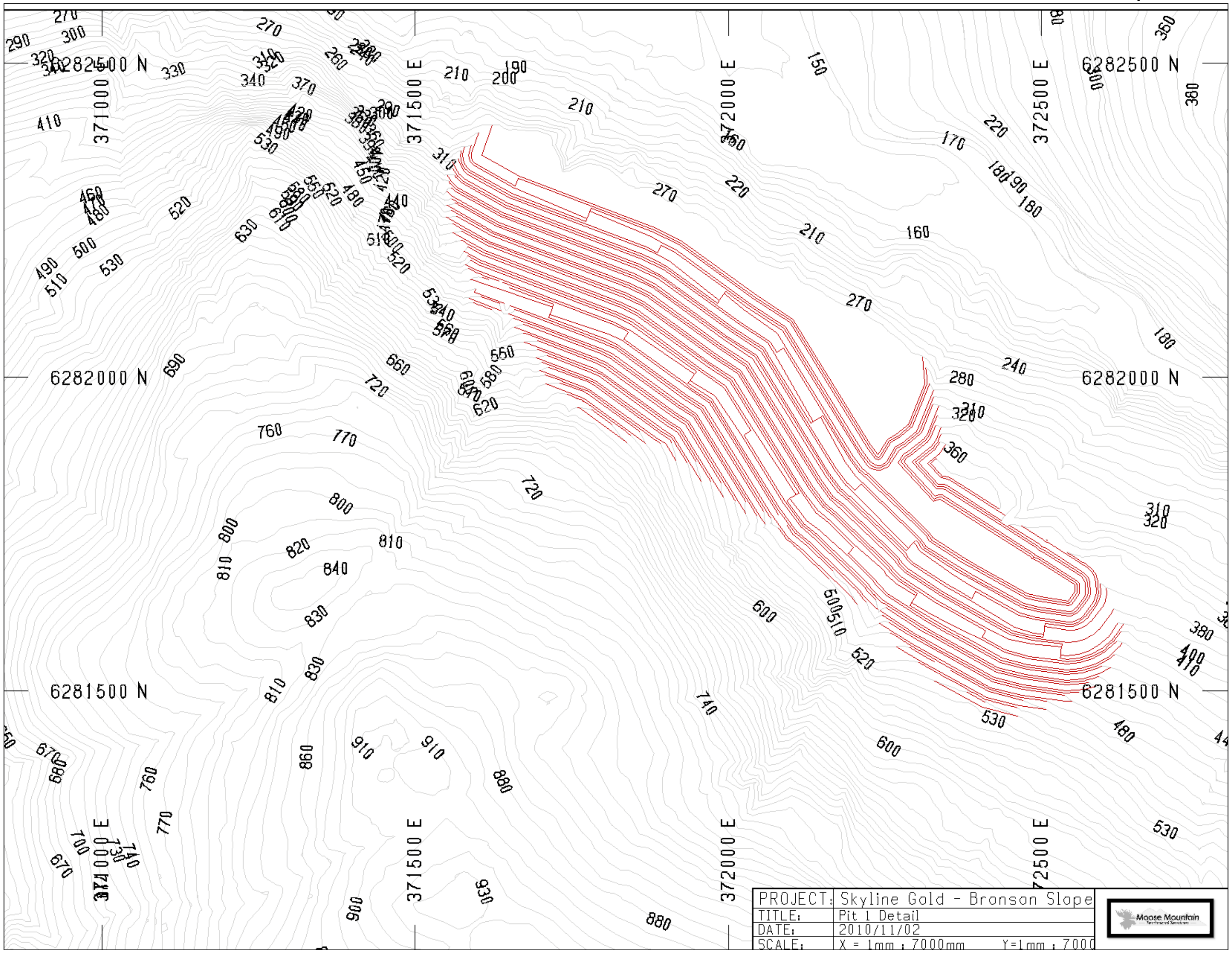

Figure 25-8: Pit 1 Details

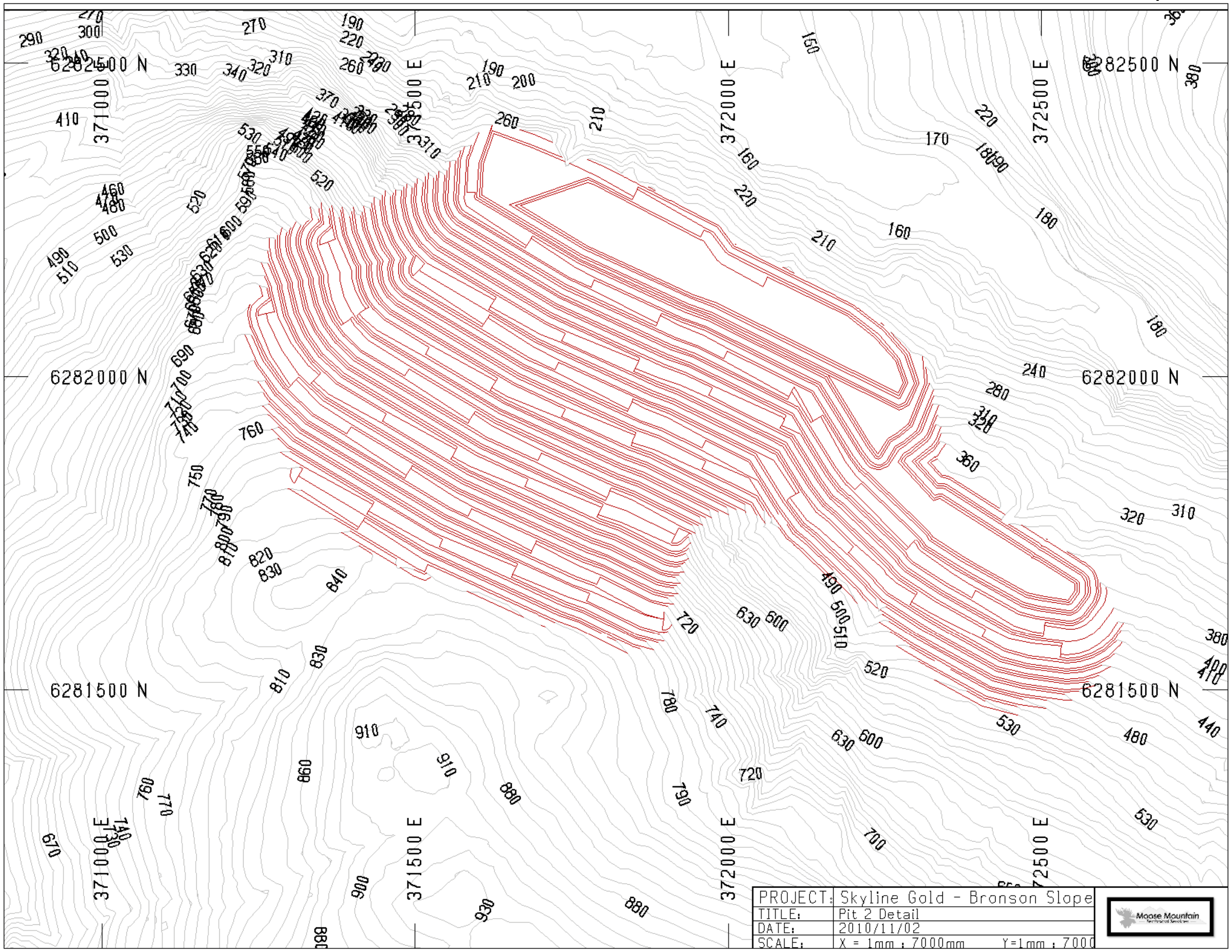

Figure 25-9: Pit 2 Details

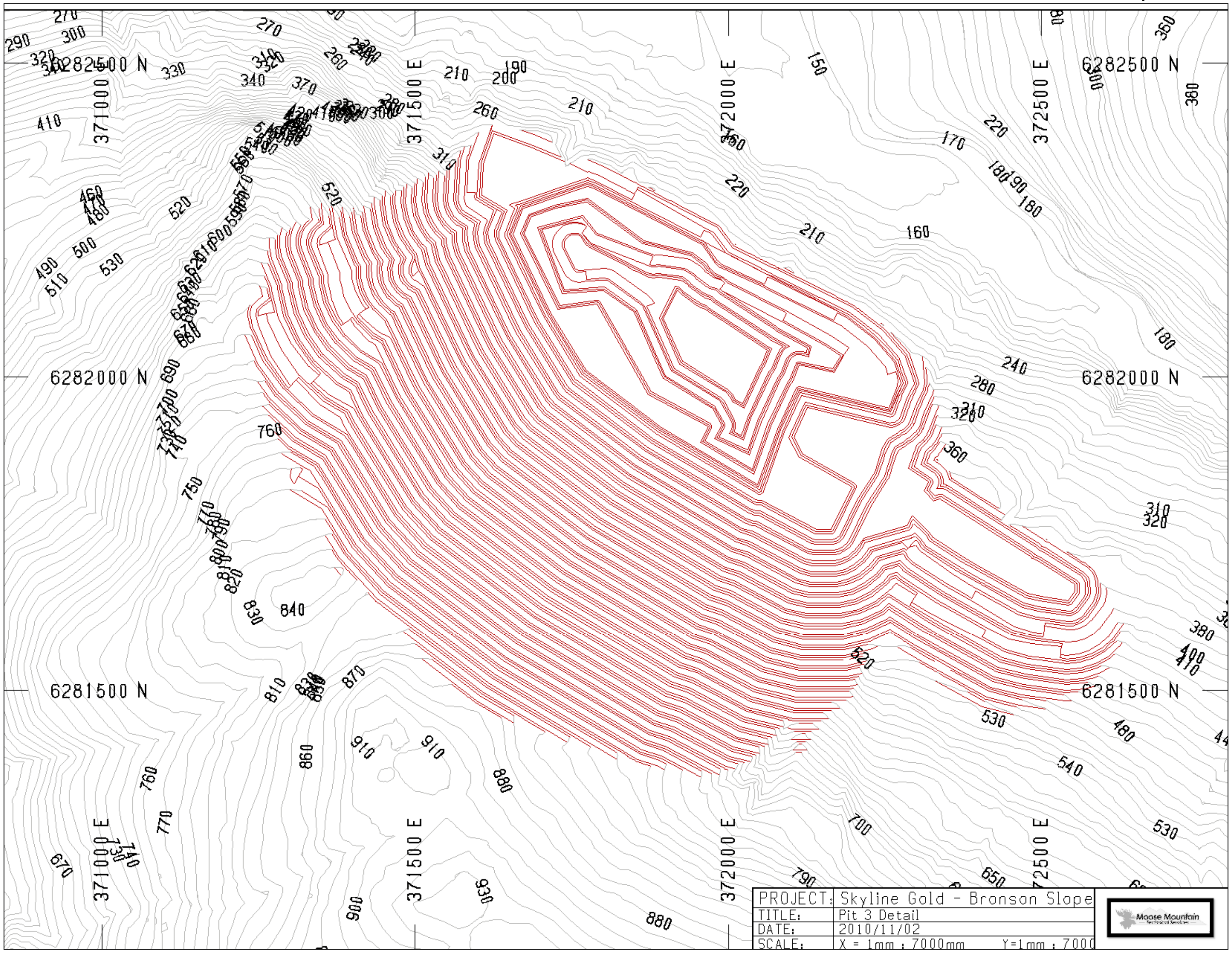

Figure 25-10: Pit 3 Details

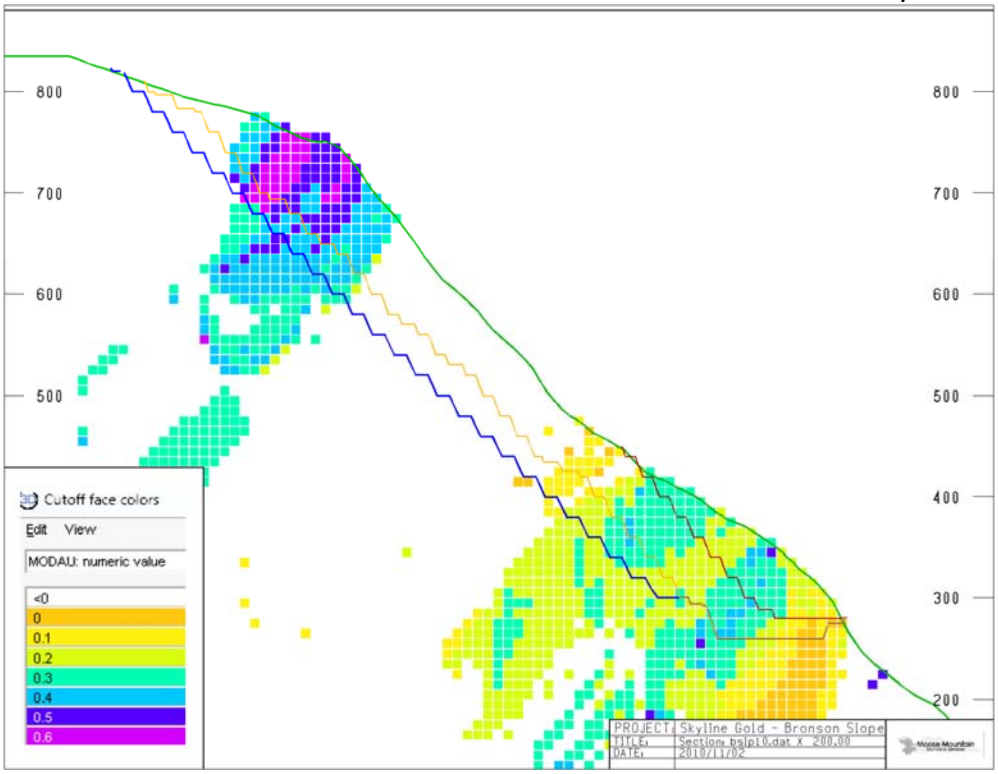

Figure 25-11 Gold Grades Section 200

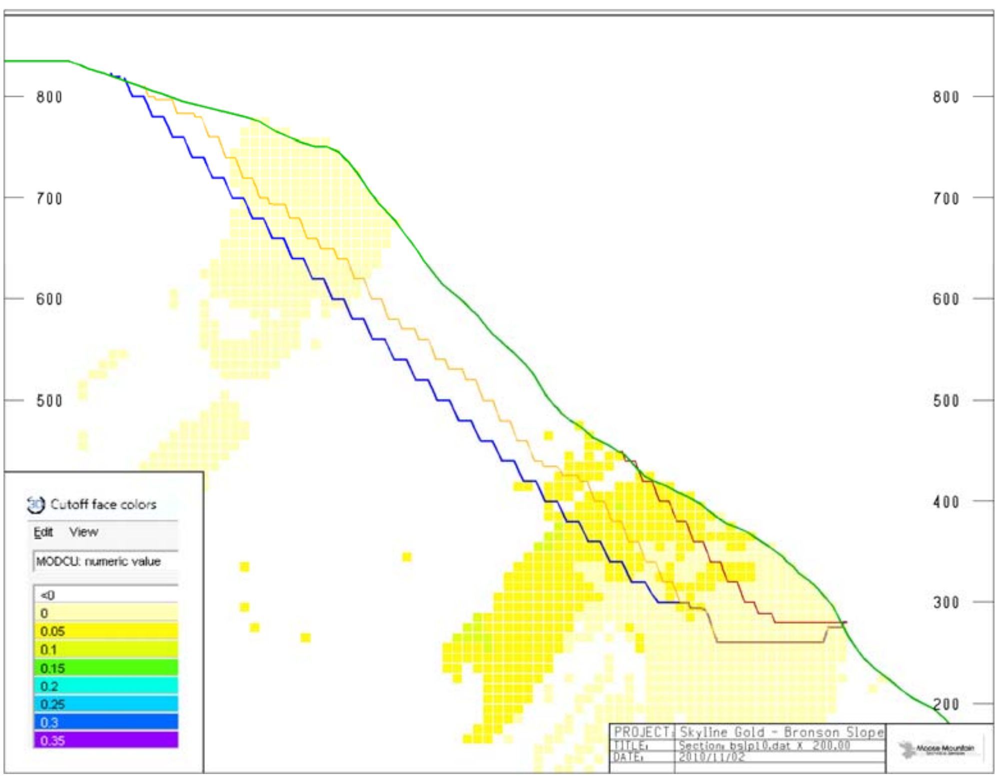

Figure 25-12 Copper Grades Section 200

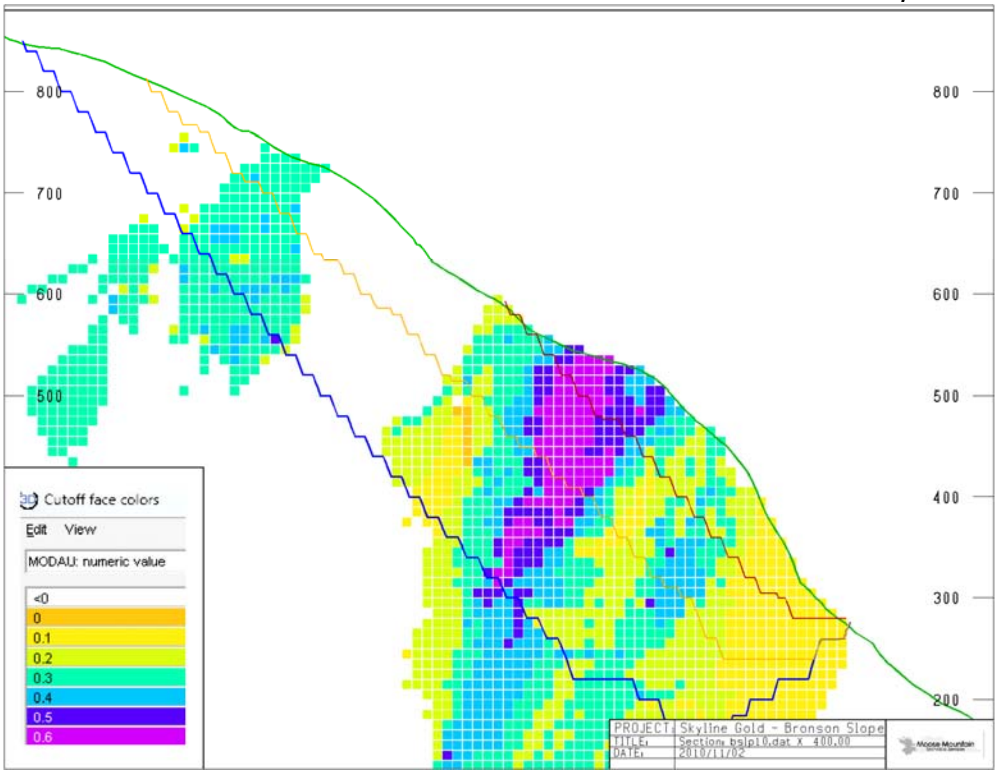

Figure 25-13 Gold Grades Section 400

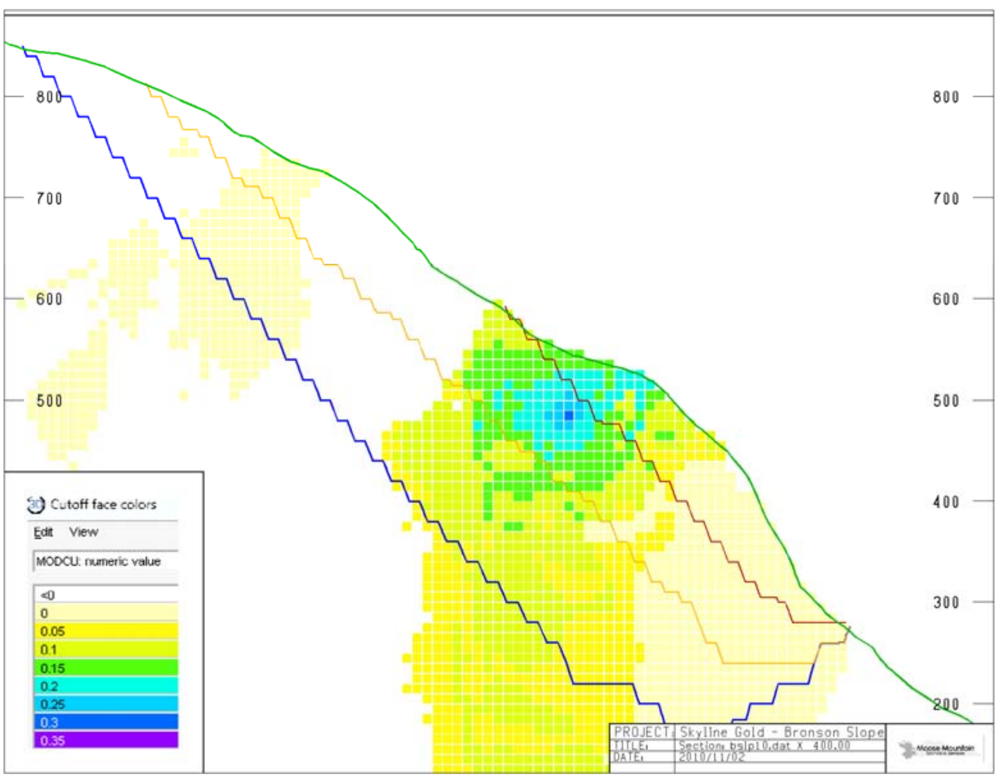

Figure 25-14 Copper Grades Section 400

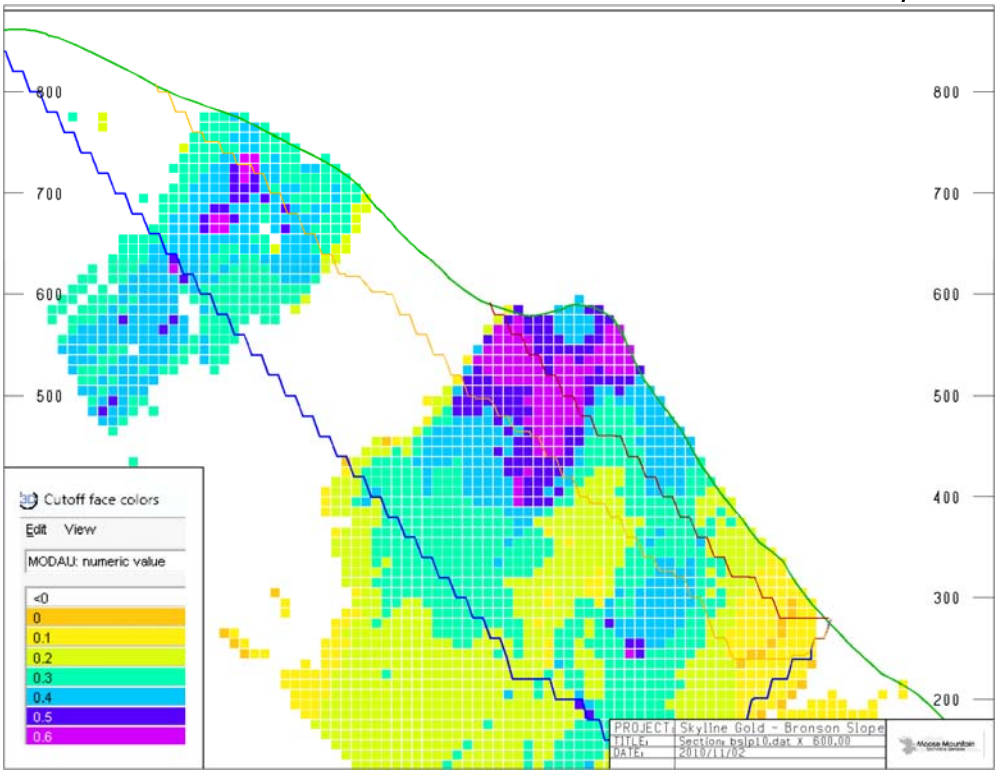

Figure 25-15 Gold Grades Section 600

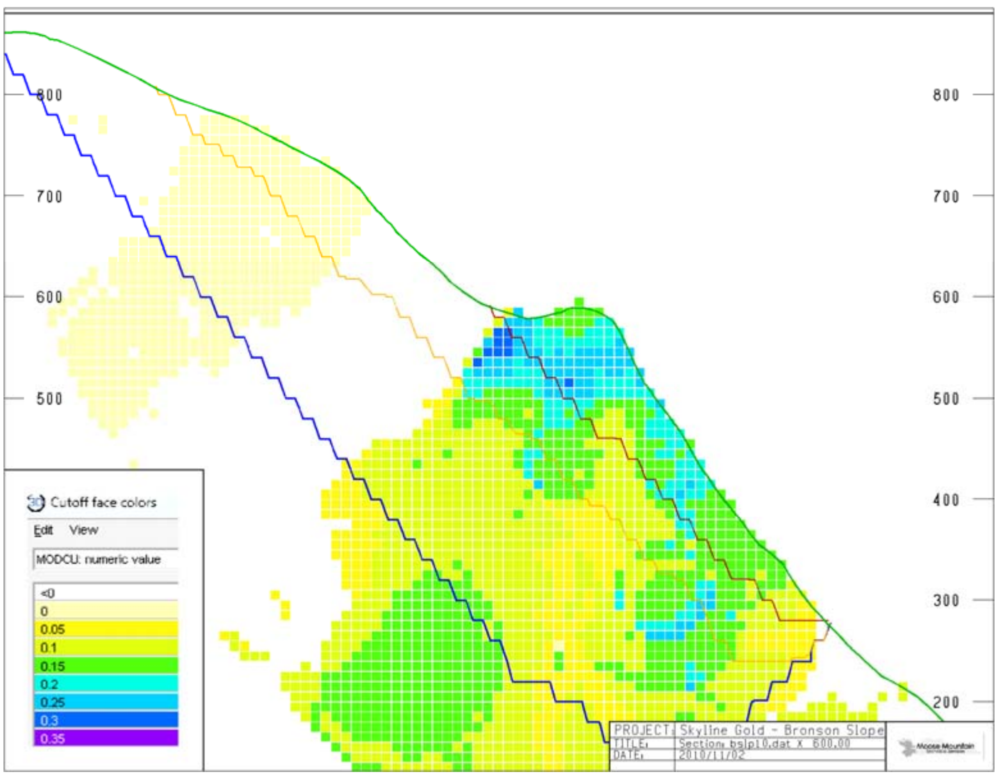

Figure 25-16 Copper Grades Section 600

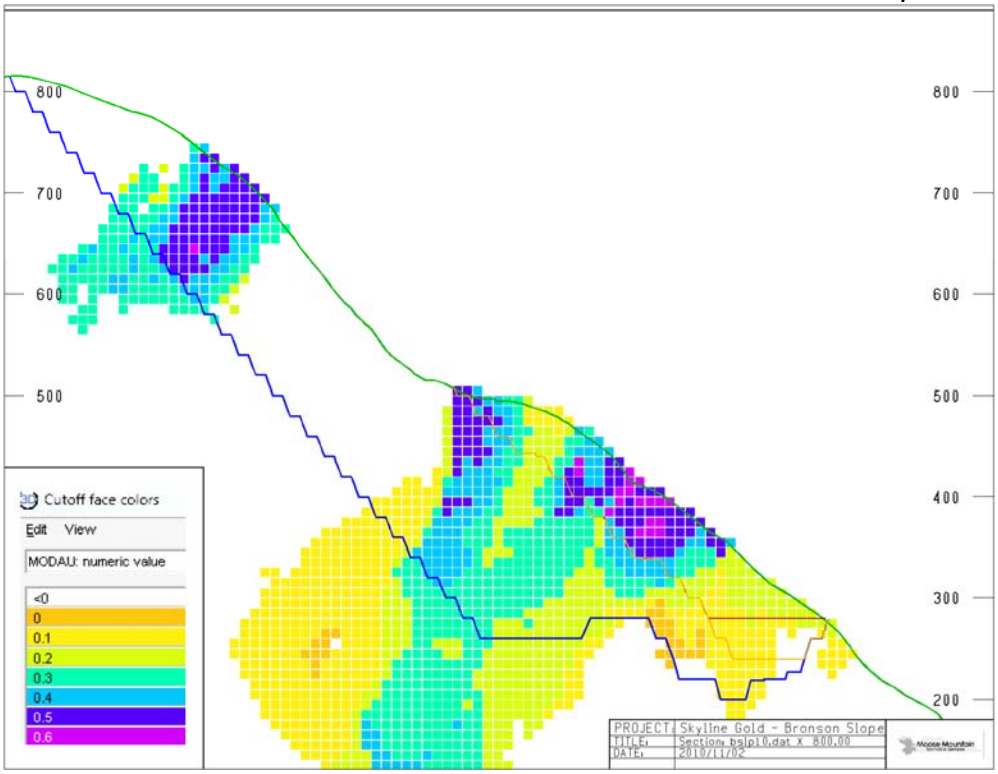

Figure 25-17 Gold Grades Section 800

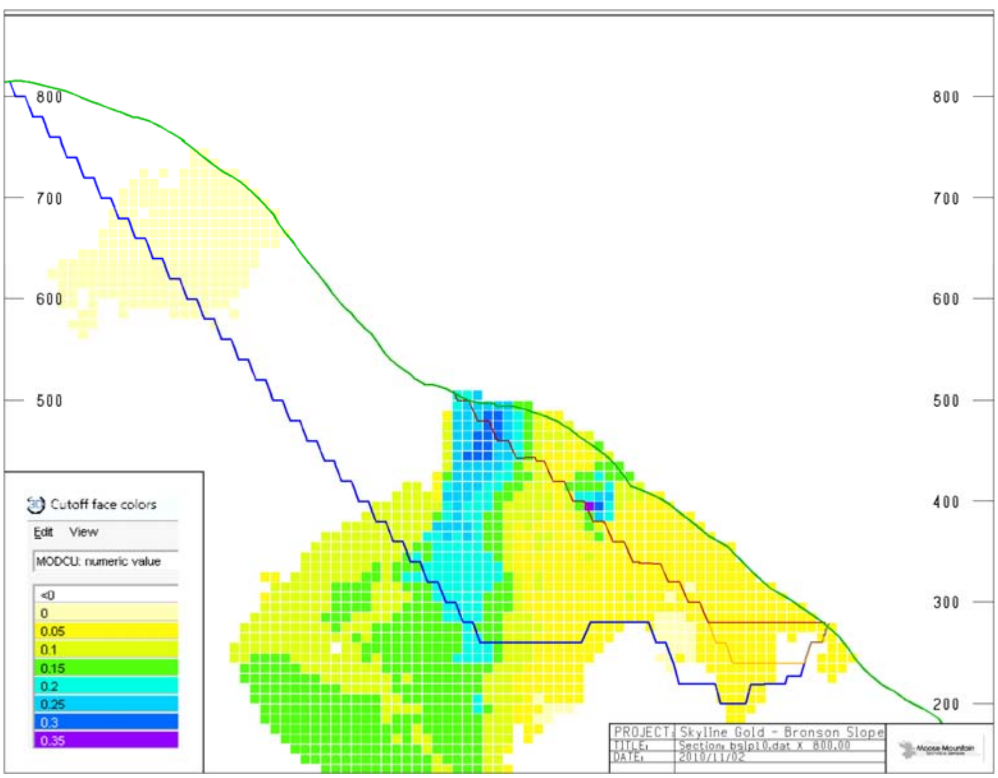

Figure 25-18 Copper Grades Section 800

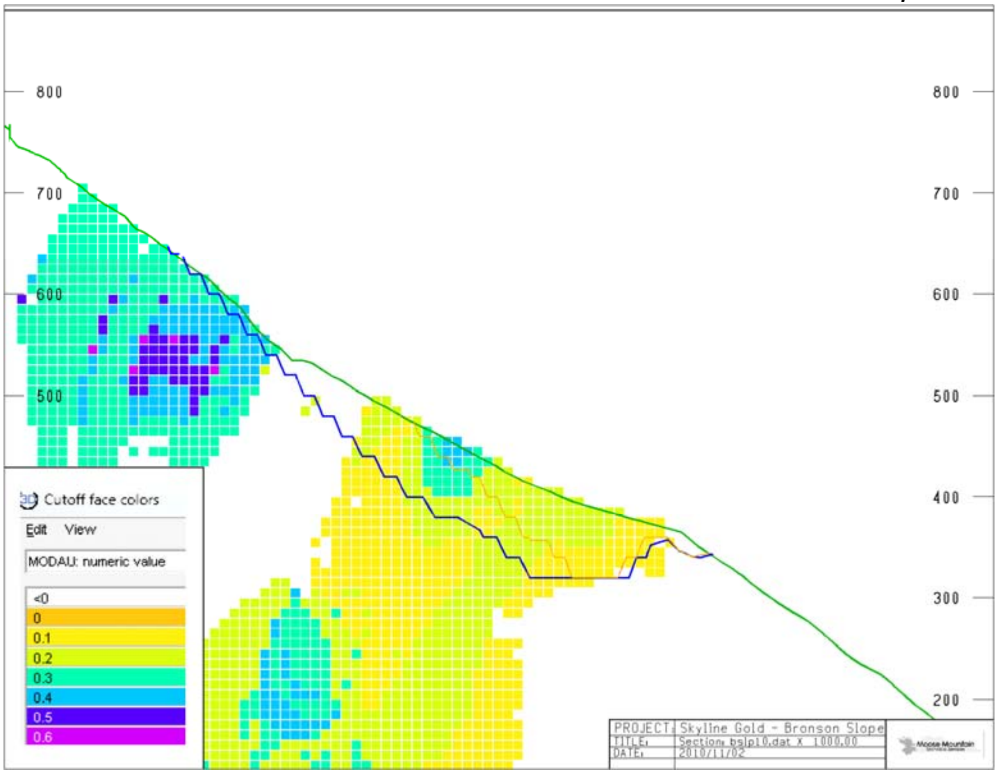

Figure 25-19 Gold Grades Section 1000

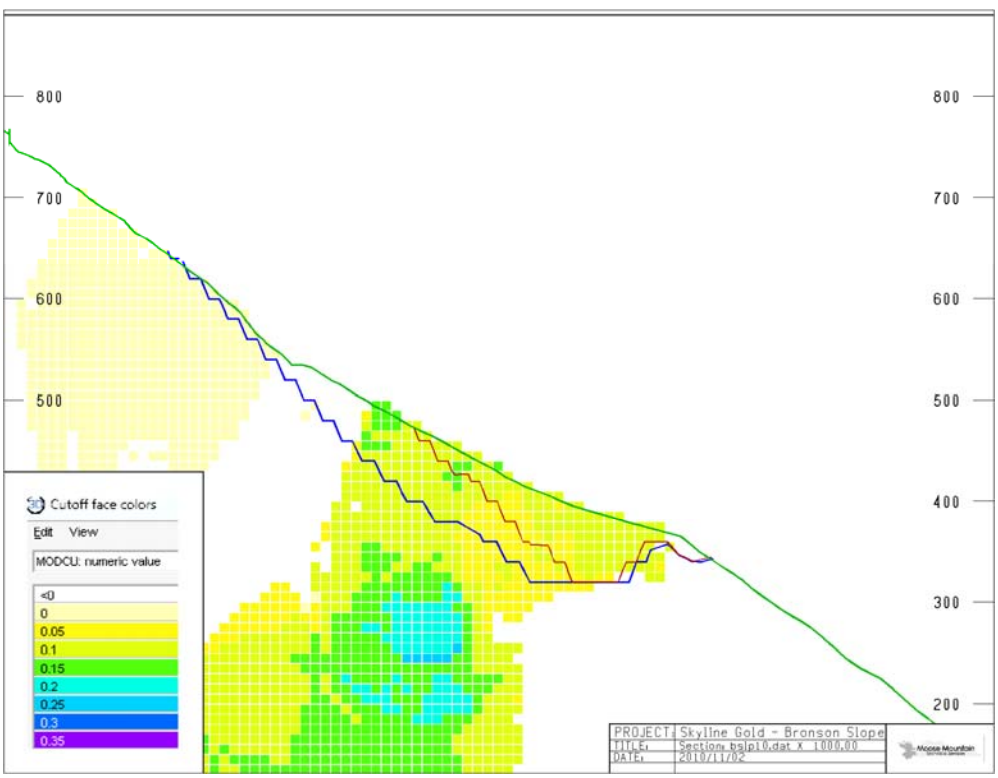

Figure 25-20 Copper Grades Section 1000

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.

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. Summaries for each of the pit phases and LOM are included in Table 25-8 to Table 25-11 below based on 5% dilution and 95% mining recovery.

Table 25-8 Pit 1 Tonnes and Grades

Pit 1 - Tonnes and Grades (USD9/t NSR Cut-off)					
Category	**Metric kTonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Magnetite %**
Measured	20,027	0.151	0.421	2.341	6.7
Indicated	20,758	0.093	0.248	1.824	4.7
Inferred	240	0.085	0.208	1.931	4.5
Total Measured + Indicated	**40,785**	**0.121**	**0.333**	**2.078**	**5.7**
Total Waste (including inferred)	10,820	**Strip Ratio:**	**0.27**		
Total Mill feed and Waste	51,605				

Table 25-9 Pit 2 Tonnes and Grades

Pit 2 - Tonnes and Grades (USD9/t NSR Cut-off)					
Category	**Metric kTonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Magnetite %**
Measured	25,610	0.146	0.434	1.936	6.8
Indicated	21,679	0.049	0.296	1.464	4.6
Inferred	1,429	0.035	0.321	1.458	1.9
Total Measured + Indicated	**47,289**	**0.102**	**0.321**	**1.720**	**5.8**
Total Waste (including inferred)	38,233	**Strip Ratio:**	**0.81**		
Total Mill feed and Waste	85,522				

Table 25-10 Pit 3 Tonnes and Grades

Pit 3 - Tonnes and Grades (USD9/t NSR Cut-off)					
Category	**Metric kTonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Magnetite %**
Measured	38,516	0.154	0.410	2.348	4.9
Indicated	60,301	0.117	0.336	2.539	4.9
Inferred	3,275	0.090	0.329	2.525	4.4
Total Measured + Indicated	**98,817**	**0.132**	**0.365**	**2.465**	**4.9**
Total Waste (including inferred)	103,390	**Strip Ratio:**	**1.05**		
Total Mill feed and Waste	202,207				

The sum of these pits (total LOM mill feed tonnage and grade) is included in Table 25-11.

Table 25-11 LOM Mill Feed Tonnes and Grade

Pits 1, 2 and 3 (Total) - Tonnes and Grades (USD9/t NSR Cut-off)					
Category	**Metric kTonnes**	**Cu%**	**Au g/t**	**Ag g/t**	**Magnetite %**
Measured	84,153	0.151	0.420	2.221	5.9
Indicated	102,738	0.098	0.310	2.168	4.8
Inferred	4,944	0.074	0.321	2.187	3.7
Total Measured + Indicated	**186,891**	**0.122**	**0.360**	**2.192**	**5.3**
Total Waste (incl Inferred)	152,443	**Strip Ratio:**	**0.82**		
Total Mill feed and Waste	**339,334**				

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 25-12 and Table 25-13. At a PA level of study, the inferred resources are included as mill feed.

Table 25-12 Production by Year 1-19

Variable	Unit	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
Mill Feed	000's t		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	5,098	5,098
Copper Grade	%		0.15%	0.15%	0.15%	0.14%	0.15%	0.15%	0.15%	0.15%	0.13%	0.10%	0.10%	0.09%	0.08%	0.07%	0.07%	0.06%	0.06%	0.07%	0.15%
Copper Recovery	%		86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	86.6%
Recovered Copper	M lbs		14.70	14.60	14.60	13.43	14.60	14.41	14.50	14.41	12.56	7.45	7.23	6.72	5.99	5.11	4.97	4.68	4.46	5.19	14.60
Gold Grade	g/t		0.54	0.499	0.407	0.378	0.453	0.41	0.425	0.445	0.406	0.409	0.263	0.305	0.284	0.261	0.237	0.21	0.183	0.381	0.381
Gold Recovery Gravity Conc	%		30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%
Gold Recovery Flotation Conc	%		59%	59%	59%	59%	59%	59%	59%	59%	59%	46%	46%	46%	46%	46%	46%	46%	46%	46%	59%
Cumulative Gold Recovery	%		89%	89%	89%	89%	89%	89%	89%	89%	89%	76%	76%	76%	76%	76%	76%	76%	76%	76%	89%
Recovered Gold	oz		78,595	72,628	59,238	55,017	65,933	59,674	61,858	64,769	59,092	50,613	32,546	37,743	35,144	32,298	29,328	25,987	22,646	47,148	55,453
Silver Grade	g/t		2.54	2.54	2.47	2.40	1.94	2.21	2.10	1.98	1.90	1.88	1.78	1.73	1.71	1.74	1.79	1.69	1.58	2.60	2.40
Silver Recovery	%		61%	61%	61%	61%	61%	61%	61%	61%	61%	45%	45%	45%	45%	45%	45%	45%	45%	45%	61%
Recovered Silver	oz		253,754	254,254	246,755	239,956	193,865	220,560	210,062	197,664	189,665	138,884	131,583	127,673	125,903	128,042	131,656	124,428	116,315	191,621	240,256
Magnetite %	%		3.14%	4.92%	6.49%	5.61%	1.20%	4.75%	5.01%	6.20%	7.22%	7.41%	4.71%	6.45%	6.73%	7.20%	7.43%	7.56%	7.63%	0.80%	0.92%
Mag Recovery	%		95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%
Recovered Magnetite	tonne		152,073	238,281	314,317	271,698	58,117	230,047	242,639	300,272	349,672	358,874	228,110	312,380	325,941	348,703	359,842	366,138	369,529	38,745	44,557
Concentrate Grade																					
Copper	%		25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%
Gold	g/t		61.2	56.9	46.4	46.9	51.7	47.4	48.8	51.4	53.9	70.7	46.9	58.5	61.1	65.8	61.5	57.9	52.9	94.7	43.5
Silver	g/t		298.4	300.9	292.1	308.7	229.5	264.6	250.3	237.1	261.0	322.1	314.4	328.3	363.2	432.7	458.0	459.9	451.0	638.4	284.4
Concentrate Moisture Content	%		8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%
Wet Concentrate	tonne		28,571	28,381	28,381	26,111	28,381	28,003	28,192	28,003	24,408	14,486	14,060	13,065	11,645	9,941	9,657	9,089	8,663	10,083	28,381
Gravity Concentrate																					
Gold Content	g/t		162	149.7	122.1	113.4	135.9	123	127.5	133.5	121.8	122.7	78.9	91.5	85.2	78.3	71.1	63	54.9	114.3	114.3
GRG Concentrate Moisture	%		5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
Wet Concentrate	tonne		5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353	5,353
Magnetite Concentrate																					
Magnetite Grade	%		98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%
Dry Magnetite Concentrate	tonne		155,177	243,143	320,732	277,243	59,303	234,742	247,591	306,400	356,808	366,198	232,765	318,755	332,592	355,820	367,186	373,611	377,070	39,536	45,466

Table 25-13 Production by Year 20-38

Variable	Unit	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27	Y28	Y29	Y30	Y31	Y32	Y33	Y34	Y35	Y36	Y37	Y38	Total
Mill Feed	000's t	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	5,098	3,209	191,835
Copper Grade	%	0.15%	0.15%	0.15%	0.14%	0.14%	0.14%	0.13%	0.13%	0.12%	0.12%	0.11%	0.11%	0.10%	0.10%	0.10%	0.09%	0.09%	0.09%	0.09%	
Copper Recovery	%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	
Recovered Copper	M lbs	14.60	14.60	14.41	13.92	13.53	13.24	12.95	12.65	8.99	8.40	8.04	7.67	7.52	7.31	7.01	6.57	6.57	6.28	4.28	383
Gold Grade	g/t	0.376	0.378	0.369	0.369	0.371	0.367	0.36	0.356	0.35	0.339	0.327	0.309	0.295	0.291	0.285	0.242	0.237	0.251	0.267	
Gold Recovery Gravity Conc	%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	
Gold Recovery Flotation Conc	%	59%	59%	59%	59%	59%	59%	59%	59%	46%	46%	46%	46%	46%	46%	46%	46%	46%	46%	46%	
Cumulative Gold Recovery	%	89%	89%	89%	89%	89%	89%	89%	89%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	76%	
Recovered Gold	oz	54,726	55,017	53,707	53,707	53,998	53,416	52,397	51,815	43,312	41,951	40,466	38,238	36,506	36,011	35,268	29,947	29,328	31,061	20,798	1,757,377
Silver Grade	g/t	2.68	2.66	2.46	2.25	2.03	2.00	2.04	2.04	2.04	2.04	2.01	1.96	1.86	1.82	1.91	1.86	1.92	1.93	1.96	
Silver Recovery	%	61%	61%	61%	61%	61%	61%	61%	61%	51%	51%	51%	51%	51%	51%	51%	51%	51%	51%	51%	
Recovered Silver	oz	267,851	265,751	246,355	224,959	202,963	199,564	203,963	204,363	170,108	170,443	167,935	164,173	155,229	151,969	159,576	155,229	160,746	160,997	103,236	6,998,305
Magnetite %	%	1.85%	2.50%	3.03%	3.77%	4.47%	5.00%	5.39%	5.75%	6.09%	6.27%	6.47%	6.57%	6.49%	6.58%	6.45%	6.14%	6.05%	6.15%	4.76%	
Mag Recovery	%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	95%	
Recovered Magnetite	tonne	89,597	121,078	146,746	182,585	216,487	242,155	261,043	278,478	294,945	303,662	313,349	318,192	314,317	318,676	312,380	297,366	293,008	297,851	145,111	9,656,959
Concentrate Grade																					
Copper	%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	25.2%	
Gold	g/t	42.9	43.1	42.7	44.2	45.7	46.2	46.3	46.9	50.2	52.0	52.4	51.9	50.5	51.3	52.4	47.4	46.5	51.5	50.6	
Silver	g/t	317.0	314.5	295.5	279.3	259.2	260.5	272.3	279.1	327.1	350.6	361.1	369.8	356.5	359.5	393.2	408.0	422.5	442.8	417.1	
Concentrate Moisture Content	%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	8%	
Wet Concentrate	tonne	28,381	28,381	28,003	27,057	26,300	25,732	25,165	24,597	17,468	16,332	15,622	14,912	14,628	14,202	13,634	12,781	12,781	12,213	8,314	
Gravity Concentrate																					
Gold Content	g/t	112.8	113.4	110.7	110.7	111.3	110.1	108	106.8	105	101.7	98.1	92.7	88.5	87.3	85.5	72.6	71.1	75.3	80.1	
GRG Concentrate Moisture	%	5%	5%	105%	205%	305%	405%	505%	605%	705%	805%	905%	1005%	1105%	1205%	1305%	1405%	1505%	1605%	1705%	
Wet Concentrate	tonne	5,353	5,353	10,451	15,549	20,647	25,745	30,843	35,941	41,039	46,137	51,235	56,333	61,431	66,529	71,627	76,725	81,823	86,921	57,922	949,308
Magnetite Concentrate																					
Magnetite Grade	%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	98%	
Dry Magnetite Concentrate	tonne	91,426	123,548	149,741	186,311	220,905	247,097	266,371	284,161	300,964	309,860	319,743	324,685	320,732	325,180	318,755	303,435	298,987	303,929	148,072	9,854,040

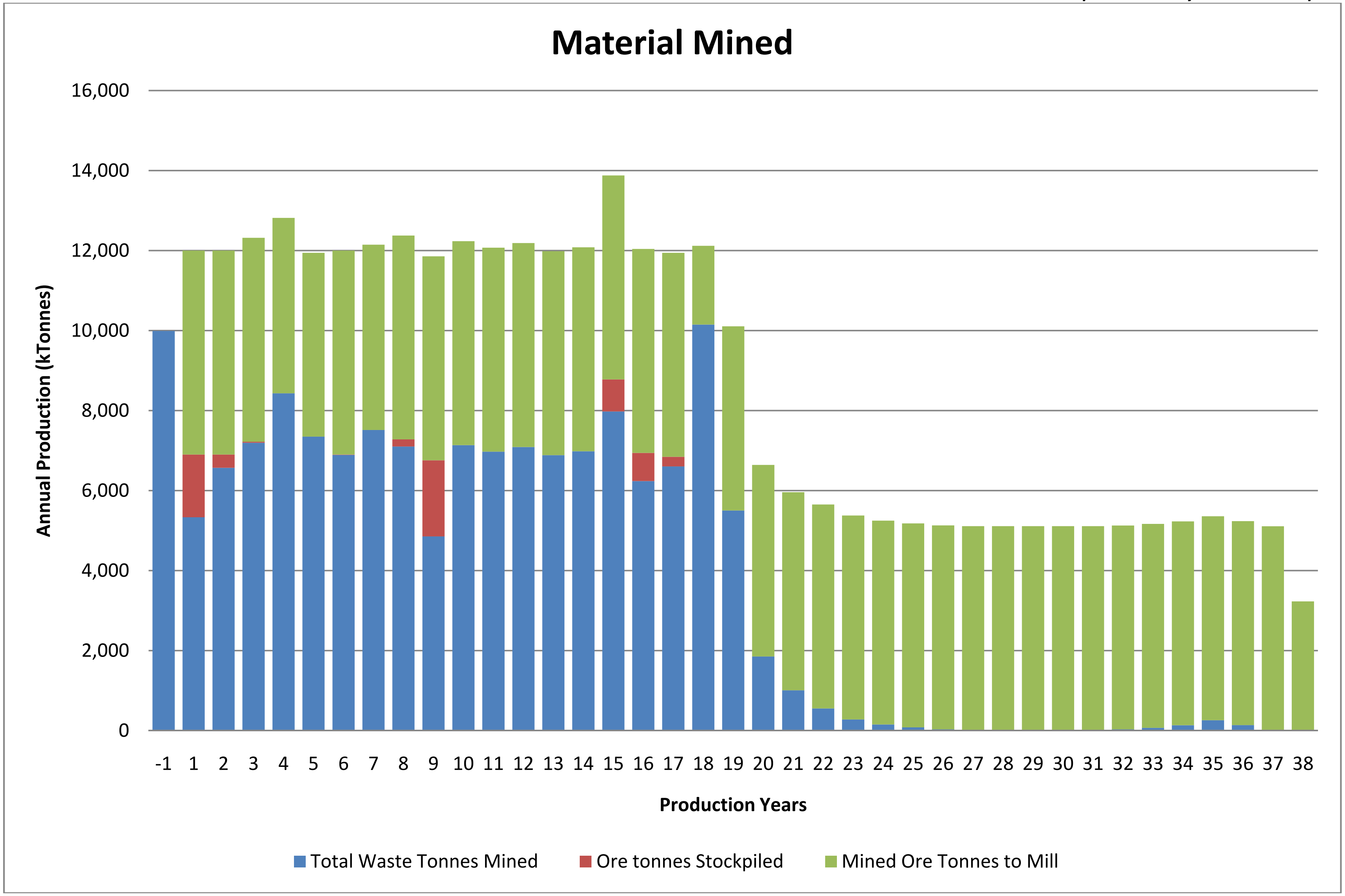

Figure 25-21 Material Movement by Type

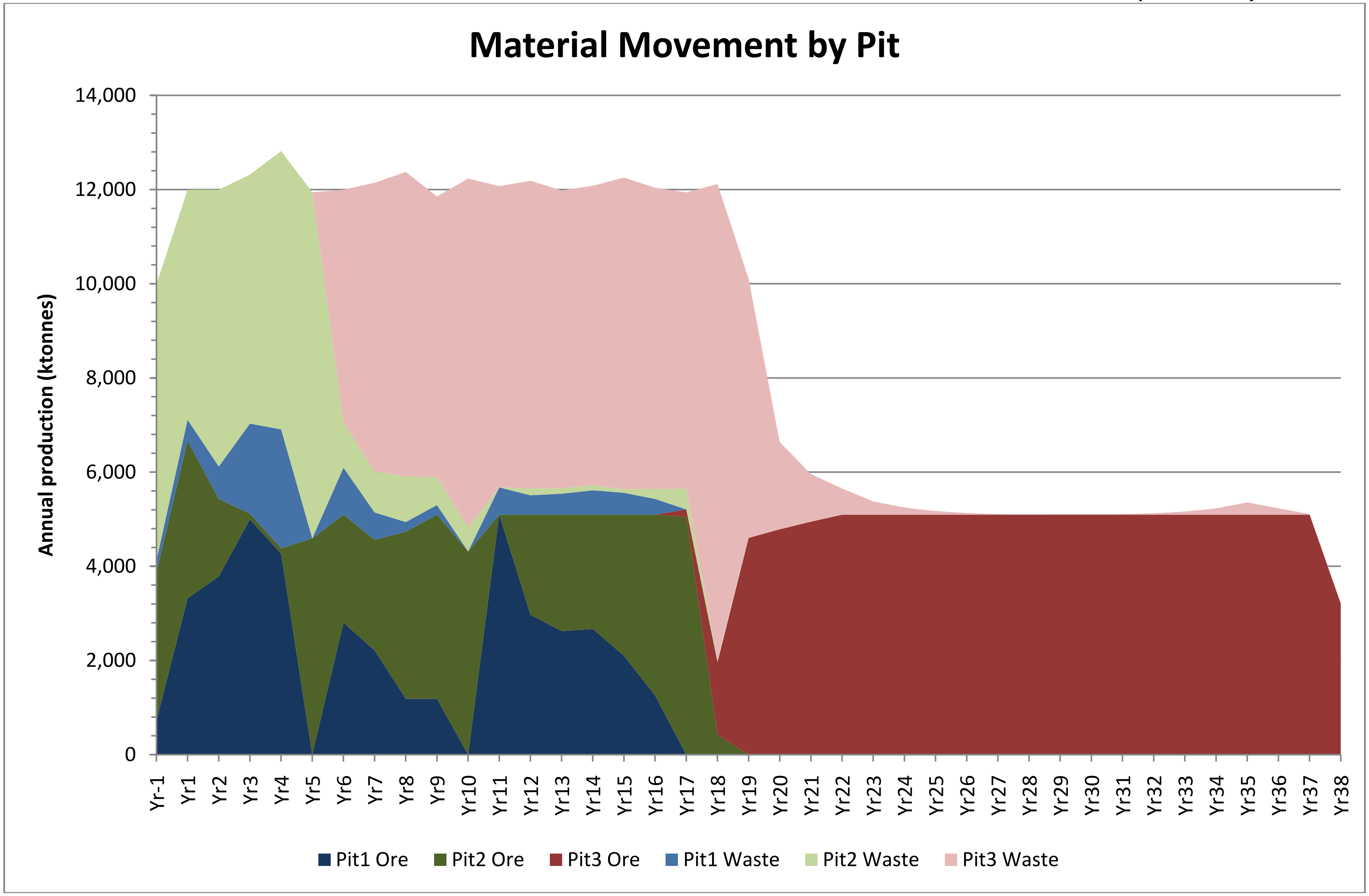

Figure 25-22 Material Movement by Pit

Table 25-14 Pit Separations by Year

	Pit 1-Pit 2 Bench Separations	Pit 2 – Pit 3 Bench Separations
Yr-1	15	15
Yr1	15	20
Yr2	16	23
Yr3	17	26
Yr4	11	36
Yr5	10	32
Yr6	9	32
Yr7	9	30
Yr8	7	30
Yr9	4	29
Yr10	1	30
Yr11	3	28
Yr12	2	28
Yr13	2	27
Yr14	1	27
Yr15	0	27
Yr16	0	28
Yr17	0	28
Yr18	0	25

The schedule and table above demonstrates that Pit 1 and Pit 2 are mined concurrently from Year -1 (Pre-strip) until year 18. The schedule is based on Pit 2 accelerated to catch up to Pit 1 by Year 9 and then Pit1 and Pit 2 are essentially mined as one large pit until completion of both pits in year 18. Stripping of Pit 3 begins in year 6 and is primarily waste until year 17. The highwall gold from this pit is treated as waste for the purposes of this study and should be strategically placed in the waste dump for future recovery options. Between year 1 and 3 mill feed is encountered in the highwall zone of Pit 2 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.

At the point when the base of Pits 1 and 2 is reached, Pit 3 becomes mill feed bound (year 20) and the production fleet is reduced to one excavator, crusher combination. From year 20 onwards a single fleet operates out of the final pit mining bench by bench until the base of the pit is reached in year 38. Figure 25-21 also shows the production schedule requires a high strip ratio early in the mine life. 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 Pit 3 needs to start early in the mine life (towards the end of year 5) 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

MMTS has not updated the metallurgy or processing aspects of the project. It is assumed that the LAL process and mill configuration is suitable. At scoping level it is assumed that a Magnetite recovery circuit will be added within the existing mill infrastructure. The following description is taken from the LAL March 6th 2009 Technical Report.

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 NSR). 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 utilizing 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.

The recovery process for Magnetite will consist of strategically placed magnetic drums to attract the mineral from the crushed and/or slurried process material. The magnetite will then be scraped from the drum onto a belt for drying, if necessary, and packaging. Further work on the magnetite recovery needs to be completed before the next series of engineering studies.

25.4 Markets

MMTS has not updated the marketing assumptions since the previous PA Technical Report. MMTS considers that at PA (scoping) level of study, the LAL marketing assumptions are suitable. The following description is taken from the LAL March 6th 2009 Technical Report.

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 25-23 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.

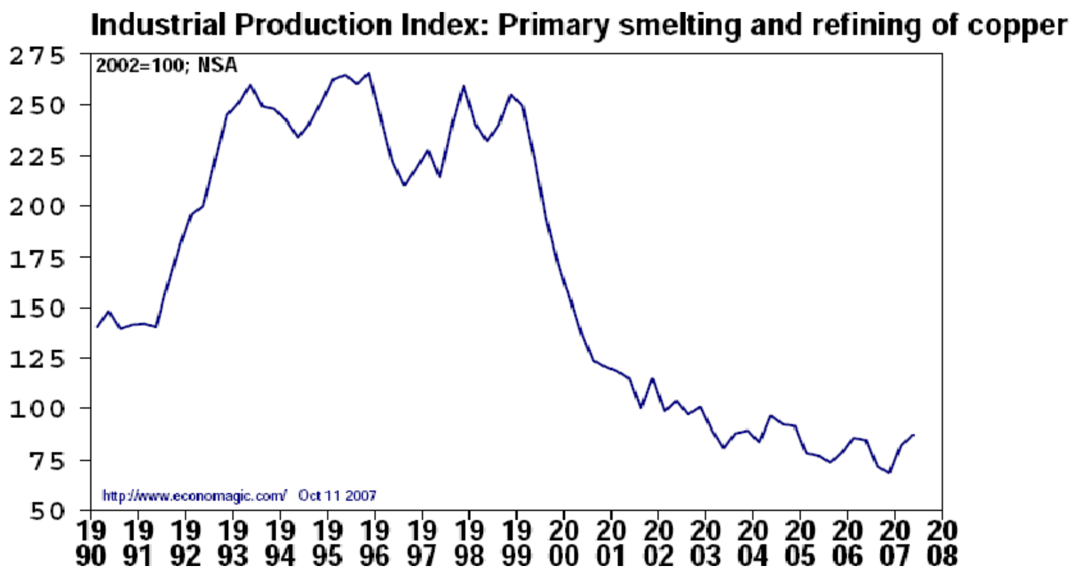

Figure 25-23 Primary Cost Index for Primary Smelting and Refining of Copper

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.0 cu)*
- *Concentrate refining charge (R.0 cu).*
- *Quotation period for payable metals (QP_{cu}, QPA_u, QPA_g),*
- *Payable precious metals and refining charges (R.0 A_u; R.0 A_g)*
- *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 25-15.

Table 25-15 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 25-16 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

MMTS has not updated the Environmental Considerations since the previous PA Technical Report. MMTS considers that at PA (scoping) level of study, the LAL Environmental Considerations are suitable. The following description is taken from the LAL March 6th 2009 Technical Report.

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.

25.7 Taxes

MMTS has not updated the Taxes assumptions since the previous PA Technical Report. MMTS considers that at PA (scoping) level of study, the LAL Taxes assumptions are suitable. The following description is taken from the LAL March 6th 2009 Technical Report.

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.

In 2010 the British Columbia government implemented the Harmonized Sales Tax (HST) which is a merely a combination of the Provincial Sales tax and the GST already considered in these evaluations.

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

MMTS has not updated the overall Capital and Operating Cost Estimates since the previous PA Technical Report with the exception of some of the Mine Operations components. MMTS considers that at PA (scoping) level of study, the LAL capital and Operating assumptions are suitable. The following description is taken from the LAL March 6th 2009 Technical Report. The MMTS inputs are noted.

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 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.*

25.8.1.3 Infrastructure Capital Cost Estimate

Table 25-17 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 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 includingprocessing/tailings facilities)-->	$42,805
	Indirect Costs	
	Engineering, procurement and construction management (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 mobilization and establishment cost after production start is included in this element.

Mobilization and establishment of the initial mining fleet and associated personnel and equipment is included in the separate mobilization and establishment item as part of the preproduction 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 summarized in

Description	Manpower	Hourly Base rate	Base Salary	Salary Burden	Unit Salaries	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**

.

Table 25-18 Administration Manpower Summary

Description	Manpower	Hourly Base rate	Base Salary	Salary Burden	Unit Salaries	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 25-19 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 25-20.

Table 25-20 Processing Capital Cost 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 25-21.

Table 25-21 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 25-22 below.

Table 25-22 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*

Description	Manpower	Hourly Base Rate CAD	Base Salary	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	5% of		
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	3$2.00	$ 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**

below:

Table 25-23 Annual Processing Manpower Requirements

Description	Manpower	Hourly Base Rate CAD	Base Salary	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	5% of		
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	3$2.00	$ 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 25-24* Site Power Requirements

AREA DESCRIPTION	Connected kW	Operating kW	Total Consumption (kWh/year)	Total Cost (CAD/year)	Unit Cost (CAD/tore)
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**

below provides the total power requirements and annual costs for the project.

Table 25-24 Site Power Requirements

AREA DESCRIPTION	Connected kW	Operating kW	Total Consumption (kWh/year)	Total Cost (CAD/year)	Unit Cost (CAD/tore)
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 25-25* 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**

below provides a summary of the milling consumables usage and maintenance allowances.

Table 25-25 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 returned to a relatively constant compared to the LAL 2009 report period during which there were extended lead-times for delivery of most items. Consideration should be made in procurement planning for a minimum allowance of 6-9 months lead time on all sizable mining equipment. Large mining equipment such as dozers, loaders, graders and excavators have a likely lead time of 6 to 12 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*
- *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).

Following a review of LAL study cost estimates, it was determined that most capital costs used are reasonable for this study. MMTS undertook a more detailed costing of all conveying including the High Angle Conveyor (sandwich style conveyor) option after discussions with the previously recommended vendor indicated concerns of reliability in this operating environment. The capital equipment requirements are summarized in Table 25-26. The Pre-production capital of $16.3 Million is not included in this table.

Table 25-26 Mine Equipment Capital Costs

Item	MMTS Estimates (k$CAN)
Drilling	2,000
Loading	2,800
In-Pit Crushing	3,400
In-Pit Conveying	6,300
HAC Transfer	5,400
HAC Conveyor	19,300
Overland Conveyor & Stacker	5,400
Rehandle Fleet	4,700
Road Construction	5,000
Support Equip	1,500
Total	**55,700**
15% Contingency	8,355
Total Mining Equip	**64,055**
Sustaining Capital (every 8 years)	5,000

25.8.6 Mining Operating Cost Estimate

25.8.6.1 Overheads and Manning Requirements

Overheads for mining will vary over time depending on the number of excavator fleets working. For the purpose of this study a LOM average has been applied.

Theses stages are summarized as;

- Pre-Strip — 2 x Fleet in pre-production phase (Year -1)
- Mining Stage 1 — 2 x Fleets production phase (Years 1 to 19)
- Mining Stage 2 — 1 x Fleet and ramp down strategy (Years 20 to 38)

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 25-27. 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 25-27 Total Manning by Mining Stage

Item	Mine Supervision	Technical Staff	Maintenance Supervision	Mine Operators	Maintenance Technicians	TOTAL
Pre-Strip	10	14	13	51	31	119
Mining Stage 1	10	14	13	51	31	119
Mining Stage 2	7	10	8	23	14	62

The 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 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 following section has been taken from the LAL 2009 study and is considered accurate and relevant for this study.

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 25-28.

Table 25-28 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*
- *Tire 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 liter 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 re-handle. A summary of the direct mine operating costs has been included in Table 25-29. These costs have been taken from LAL report, reviewed for accuracy and changes made where appropriate. The MMTS assumed waste rock conveyance to the dump which increases the conveying aspect and lowers the re-handle cost.

Table 25-29 Mine Operating Costs

Mine Operating Costs	MMTS	LAL Study
Drilling	$0.15	$0.146
Blasting	$0.34	$0.339
Loading, Crushing & Conveyor	$0.72	$0.625
Rehandle, Load & Haul	$0.29	$0.412
Ancillary – Roads & Dumps	$0.31	$0.321
Total Direct Costs	**$1.81**	**$1.843**

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 25-24 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.

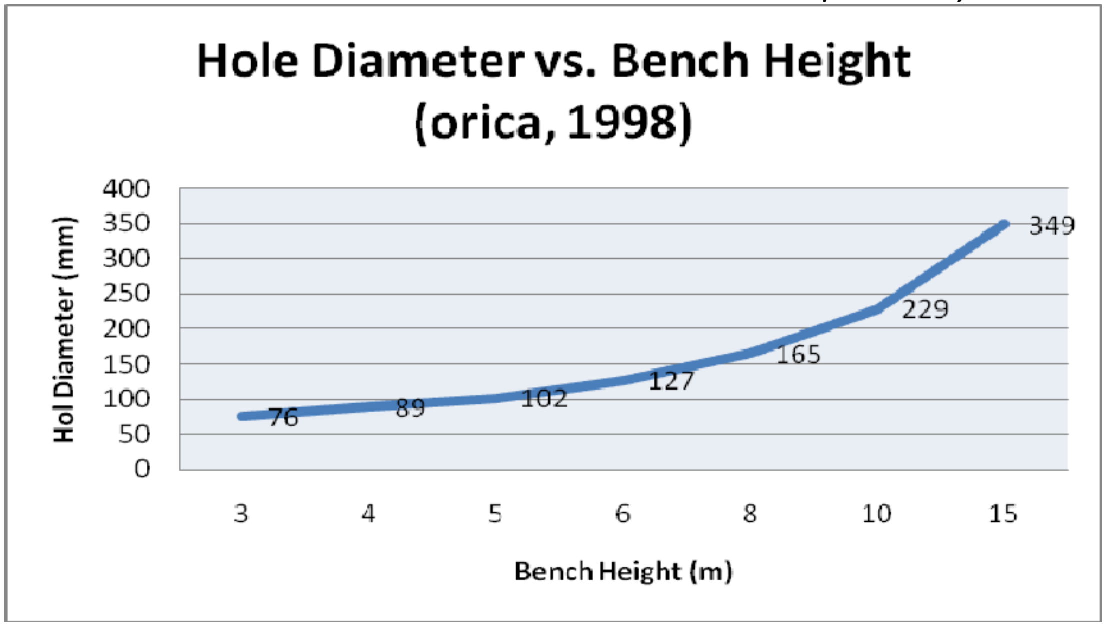

Figure 25-24 Bench Height vs Blasthole Size

The following Table shows the drill and blast parameters selected for Bronson slope by material type.

Table 25-30 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 utilized 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 25-31.

Table 25-31 LOM Capital Cost Expenditures by Period

Capital Item (Million $)	Yr-2	Yr -1	Yr1-8	Yr9-16	Yr16-24	Yr24-32
Infrastructure	30.8	30.8				
Processing	57.9	57.9	4.0	4.0	4.0	4.0
Mining	12.8	67.5	5.0	5.0	5.0	5.0
Total	**101.5**	**156.2**	**9.0**	**9.0**	**9.0**	**9.0**

25.9 Economic Analysis

A cashflow model has been generated based on all revenue and expenses presented in this report. The cashflow is an annualized 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 realized metal prices have also been presented in the cashflow model summary included in Table 25-32 to Table 25-34.

25.9.1 Project Operating Cost Summary

A summary of the site wide operating costs has been included in Table 25-32.The annual operating cost by process is shown in Table 25-33 on the following page.

Table 25-32 Summary of Operating Costs

Cost Category	Unit	LOM
Direct Mining Costs	(CAD,000)	$596.130
	(CAD/tonne Mined)	$1.81
	(CAD/tonne Milled)	$3.11
Overheads and Administration	(CAD,000)	$187,998
	(CAD/tonne Mined)	$0.50
	(CAD/tonne Milled)	$0.98
Processing Costs	(CAD,000)	$1,214,316
	(CAD/tonne Milled)	$6.33
Total Site Operating Costs	(CAD,000)	$1,998,444
	(CAD/tonne Milled)	$10.44
	(USD/tonne Milled)	$9.39

Table 25-33 Annual Operating Costs

(CAN$,000)	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13
Direct Mining Costs	21,720	21,720	22,299	23,201	21,620	21,720	21,988	22,404	21,461	22,147	21,852	22,062	21,695
Processing Costs	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270
G&A	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996
Total Operating Costs	58,986	58,986	59,566	60,467	58,887	58,986	59,254	59,671	58,728	59,414	59,119	59,328	58,961

(CAN$,000)	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21	Yr22	Yr23	Yr24	Yr25	Yr26
Direct Mining Costs	21,868	25,119	21,794	21,619	21,935	18,294	12,022	10,786	10,234	9,732	9,504	9,378	9,285
Processing Costs	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270
G&A	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996
Total Operating Costs	59,135	62,386	59,061	58,885	59,202	55,560	49,288	48,052	47,500	46,999	46,771	46,644	46,552

(CAN$,000)	Yr27	Yr28	Yr29	Yr30	Yr31	Yr32	Yr33	Yr34	Yr35	Yr36	Yr37	Yr38	Total
Direct Mining Costs	9,255	9,253	9,251	9,253	9,253	9,283	9,356	9,470	9,700	9,479	9,249	5,850	**596,130**
Processing Costs	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	32,270	20,313	**1,214,316**
G&A	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	4,996	3,145	**187,998**
Total Operating Costs	46,521	46,519	46,517	46,519	46,519	46,550	46,622	46,736	46,966	46,745	46,515	29,308	**1,998,444**

25.9.2 Project Economics

Economic evaluation for the Project is based on a pre-tax financial model. For the Project as defined in this update of 38 years and 191 Mtonnes mill feed, the following base case financial results are:

- 21.5% IRR
- $330.2 million NPV at 7.5% discount
- 4.8 year back on $257.6 million
- $1,405.6 million NPV at undiscount cashflow

Constant metal prices and a constant foreign exchange rate is used in the pre-tax model:

- Copper: US$ 2.50/lb
- Gold: US$ 950/t.oz
- Silver: US$ 15/t.oz for Silver were used for the cash flow analysis
- Magnetite: USD$ 130/t as DMS, US$ 50/t as iron ore (net price, FOB Stewart or other)
- Exchange rate: US$ 0.90: CAD$ 1.00

The financial model includes an initial capital of C$241.3 million, pre-strip capital of C$16.3 million and a sustaining capital of C$38.5 million. No allowance has been made for working capital, or salvage value, and no reclamation or closure costs have been estimated or included.

Annual Cash Flow

Production statistics from the mine production schedule are incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tonnage milled, head grades and recoveries. Market prices for copper, gold, and silver are adjusted to realized price levels by applying smelting, refining, and concentrate transportation charges from mine site to smelter to determine revenue (NSR) from copper concentrate and gravity concentrate sales. Realized price for magnetite are calculated assuming FOB Stewart.

Unit operating costs for mining, milling, and G&A areas are applied to annual mined or milled tonnages to determine the overall mine site operating cost which are deducted from NSR to derive annual Net Revenues.

Initial and sustaining capital costs are incorporated on a year-by-year basis over the mine life and deducted from Net Revenue to determine Net Cash Flow before taxes.

The cost model is summarized in Table 25-34 and

Table 25-35 below. The cost model used is a generic model modified for this project and US currency is used extensively throughout to align with the product price base. Refer to the units when comparing numbers from this table to other tables that are in Canadian currency.

Table 25-34 Summarized Cashflow to Year -2 to 18

SKYLINE GOLD CORPORATION -- BRONSON SLOPE																						
SELECTED CASE:	1																					
SELECTED ASSUMPTIONS:	1																					
	Units	**Base**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**
MINING																						
Mined Ore Tonnes to Mill	k dmt		0	0	5,098	5,098	5,098	4,386	4,594	5,098	4,632	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	5,098	1,969
Ore tonnes Stockpiled				0	1569	333	24	0	0	10	0	178	1905		0	0	0	0	800	703	241	0
Total Waste Tonnes Mined	k dmt		0	9,992	5,333	6,569	7,198	8,432	7,351	6,892	7,516	7,102	4,854	7,138	6,975	7,091	6,888	6,984	7,980	6,240	6,605	10,150
Total Tonnes Mined	k dmt		0	9,992	12,000	12,000	12,320	12,818	11,945	12,000	12,148	12,378	11,857	12,236	12,073	12,189	11,986	12,082	13,878	12,041	11,944	12,119
Stockpile to Mill					0	0	0	712	504	0	466	0	0	0	0	0	0	0	0	0	0	3129
MILLING																						
Total Ore Tonnes Milled	000 dmt		0	0	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	5,098
Cu Grade Mill Feed	%		0.00%	0.00%	0.15%	0.15%	0.15%	0.14%	0.15%	0.15%	0.15%	0.15%	0.13%	0.10%	0.10%	0.09%	0.08%	0.07%	0.07%	0.06%	0.06%	0.07%
Cu to Cu Conc Recovery	%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%
Ag Grade Mill Feed	g/t		0.00	0.00	2.54	2.54	2.47	2.40	1.94	2.21	2.10	1.98	1.90	1.88	1.78	1.73	1.71	1.74	1.79	1.69	1.58	2.60
Ag to Cu Conc Recovery	%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%
Au Grade Mill Feed	g/t		0.000	0.000	0.378	0.349	0.285	0.265	0.317	0.287	0.298	0.312	0.284	0.286	0.184	0.214	0.199	0.183	0.166	0.147	0.128	0.267
Au to Cu Conc Recovery	%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%
Gravity Concentrate																						
Au Grade Gravity Feed	g/t		0.00	0.00	0.54	0.50	0.41	0.38	0.45	0.41	0.43	0.45	0.41	0.41	0.26	0.31	0.28	0.26	0.24	0.21	0.18	0.38
Au to Au Gravity Conc Recovery	%	30.0%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%
Magnetite Concentrate																						
Mag Grade Mill Feed	%		0.00%	0.00%	3.14%	4.92%	6.49%	5.61%	1.20%	4.75%	5.01%	6.20%	7.22%	7.41%	4.71%	6.45%	6.73%	7.20%	7.43%	7.56%	7.63%	0.80%
Mag to Mag Conc Recovery	%	95%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%
PAYABLE METALS																						
Cu Metal Payable	M lbs				14.11	14.02	14.02	12.90	14.02	13.83	13.93	13.83	12.06	7.15	6.94	6.45	5.75	4.91	4.77	4.49	4.28	4.98
Ag Metal Payable	k oz				203	203	197	192	155	176	168	158	152	111	105	102	101	102	105	100	93	153
Au Metal Payable	oz				50,742	46,889	38,244	35,519	42,567	38,526	39,936	41,815	38,150	29,747	19,128	22,183	20,655	18,983	17,237	15,273	13,310	27,710
Gravity Au Metal Payable	oz	100%			26,553	24,537	20,013	18,587	22,275	20,160	20,898	21,881	19,964	20,111	12,932	14,997	13,965	12,834	11,654	10,326	8,998	18,734
Magnetite Payable	k mt	100%			152.07	238.28	314.32	271.70	58.12	230.05	242.64	300.27	349.67	358.87	228.11	312.38	325.94	348.70	359.84	366.14	369.53	38.74
REVENUES																						
Cu Sales	US$M		0.00	0.00	35.28	35.05	35.05	32.25	35.05	34.58	34.82	34.58	30.14	17.88	17.35	16.12	14.37	12.27	11.92	11.22	10.69	12.44
Silver Credit	US$M		0.00	0.00	3.05	3.05	2.96	2.88	2.33	2.65	2.52	2.37	2.28	1.67	1.58	1.53	1.51	1.54	1.58	1.49	1.40	2.30
Gold Credit	US$M		0.00	0.00	48.20	44.54	36.33	33.74	40.44	36.60	37.94	39.72	36.24	28.26	18.17	21.07	19.62	18.03	16.38	14.51	12.64	26.32
Total Cu Sales	**US$M**		**0.00**	**0.00**	**86.53**	**82.65**	**74.34**	**68.87**	**77.82**	**73.83**	**75.28**	**76.68**	**68.66**	**47.80**	**37.10**	**38.73**	**35.50**	**31.84**	**29.87**	**27.22**	**24.73**	**41.07**
Gravity Gold Sales	US$M		0.00	0.00	25.22	23.31	19.01	17.66	21.16	19.15	19.85	20.79	18.97	19.11	12.29	14.25	13.27	12.19	11.07	9.81	8.55	17.80
Magnetite Sales	US$M		0.00	0.00	17.60	21.91	25.72	23.58	7.56	21.50	22.13	25.01	27.48	27.94	21.41	25.62	26.30	27.44	27.99	28.31	28.48	5.04
Total Gross Revenues	**US$M**		**0.00**	**0.00**	**215.90**	**210.52**	**193.42**	**178.98**	**184.35**	**188.31**	**192.54**	**199.16**	**183.77**	**142.65**	**107.89**	**117.33**	**110.57**	**103.30**	**98.81**	**92.56**	**86.49**	**104.97**
Revenues (Net of TCRC)																						
Cu Conc Sales - Net	US$M		0.00	0.00	80.71	76.88	68.62	63.60	72.09	68.19	69.60	71.03	63.73	44.83	34.26	36.07	33.12	29.79	27.89	25.36	22.96	38.92
Grav Conc Sales - Net	US$M		0.00	0.00	25.09	23.19	18.91	17.56	21.05	19.05	19.75	20.68	18.87	19.00	12.22	14.17	13.20	12.13	11.01	9.76	8.50	17.70
Mag Sales - Net	US$M		0.00	0.00	17.60	21.91	25.72	23.58	7.56	21.50	22.13	25.01	27.48	27.94	21.41	25.62	26.30	27.44	27.99	28.31	28.48	5.04
Total Net Revenues	**US$M**		**0.00**	**0.00**	**123.41**	**121.98**	**113.25**	**104.75**	**100.70**	**108.74**	**111.48**	**116.72**	**110.07**	**91.78**	**67.89**	**75.86**	**72.62**	**69.35**	**66.89**	**63.42**	**59.94**	**61.67**
OPERATING COSTS																						
Mining	US$M		0.00	16.28	19.55	19.55	20.07	20.88	19.46	19.55	19.79	20.16	19.32	19.93	19.67	19.86	19.53	19.68	22.61	19.61	19.46	19.74
Milling	US$M		0.00	0.00	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04
G&A	US$M		0.00	0.00	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50
Total Operating Costs	**US$M**		**0.00**	**0.00**	**53.09**	**53.09**	**53.61**	**54.42**	**53.00**	**53.09**	**53.33**	**53.70**	**52.85**	**53.47**	**53.21**	**53.40**	**53.06**	**53.22**	**56.15**	**53.15**	**53.00**	**53.28**
CAPITAL EXPENDITURES (in US$M)																						
Capital Expenditures - Sustaining	US$M		0.00	0.00	0.45	0.45	0.45	0.45	0.45	0.45	0.45	4.95	0.45	0.45	0.45	0.45	0.45	0.45	0.45	4.95	0.45	0.45
Capital Expenditures - Initial	US$M		91.31	140.56	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total Capital Expenditures	**US$M**		**91.31**	**140.56**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**4.95**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**4.95**	**0.45**	**0.45**
CASHFLOW and NPV (before taxes)																						
Net Revenue	US$M		0.0	0.0	123.4	122.0	113.3	104.7	100.7	108.7	111.5	116.7	110.1	91.8	67.9	75.9	72.6	69.4	66.9	63.4	59.9	61.7
Total Operating Costs	US$M		0.0	0.0	-53.1	-53.1	-53.6	-54.4	-53.0	-53.1	-53.3	-53.7	-52.9	-53.5	-53.2	-53.4	-53.1	-53.2	-56.1	-53.2	-53.0	-53.3
Capital Expenditures	US$M		-91.3	-140.6	-0.5	-0.5	-0.5	-0.5	-0.5	-0.5	-0.5	-5.0	-0.5	-0.5	-0.5	-0.5	-0.5	-0.5	-0.5	-5.0	-0.5	-0.5
CASHFLOW (before taxes)	US$M		-91.3	-140.6	69.9	68.4	59.2	49.9	47.3	55.2	57.7	58.1	56.8	37.9	14.2	22.0	19.1	15.7	10.3	5.3	6.5	7.9
Cumulative Cashflow	US$M		-91.3	-231.9	-162.0	-93.6	-34.4	15.5	62.8	118.0	175.7	233.7	290.5	328.4	342.6	364.6	383.7	399.4	409.7	415.0	421.5	429.4
NPV (and yearly PV of Cashflow)	297.2	US$	-91.3	-130.8	60.5	55.1	44.3	34.7	30.6	33.3	32.4	30.3	27.5	17.1	6.0	8.6	6.9	5.3	3.2	1.6	1.8	2.0
	330.2	C$																				
IRR	21.5%																					

Table 25-35 Summarized Cashflow Year 19 to LOM

SKYLINE GOLD CORPORATION -- BRONSON SL																							
SELECTED CASE:	1																						
SELECTED ASSUMPTIONS:	1																						
	Units	**Base**	**19**	**20**	**21**	**22**	**23**	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**	**32**	**33**	**34**	**35**	**36**	**37**	**38**	**Total**
Mined Ore Tonnes to Mill	k dmt		4,605	4,787	4,951	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	3,209	**191,835**
Ore tonnes Stockpiled			0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
Total Waste Tonnes Mined	k dmt		5,502	1,855	1,008	556	279	153	83	32	15	14	13	14	14	31	71	134	261	139	12	23	**147,499**
Total Tonnes Mined	k dmt		10,107	6,642	5,959	5,654	5,377	5,251	5,181	5,130	5,113	5,112	5,111	5,112	5,112	5,129	5,169	5,232	5,359	5,237	5,110	3,232	**339,335**
Stockpile to Mill			493	311	147	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	
MILLING																							
Total Ore Tonnes Milled	000 dmt		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	5,098	5,098	3,209	
Cu Grade Mill Feed	%		0.15%	0.15%	0.15%	0.15%	0.14%	0.14%	0.14%	0.13%	0.13%	0.12%	0.12%	0.11%	0.11%	0.10%	0.10%	0.10%	0.09%	0.09%	0.09%	0.09%	
Cu to Cu Conc Recovery	%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	86.6%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	
Ag Grade Mill Feed	g/t		2.40	2.68	2.66	2.46	2.25	2.03	2.00	2.04	2.04	2.04	2.04	2.01	1.96	1.86	1.82	1.91	1.86	1.92	1.93	1.96	
Ag to Cu Conc Recovery	%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	61.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	51.0%	
Au Grade Mill Feed	g/t		0.267	0.263	0.265	0.258	0.258	0.260	0.257	0.252	0.249	0.245	0.237	0.229	0.216	0.207	0.204	0.200	0.169	0.166	0.176	0.187	
Au to Cu Conc Recovery	%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	84.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	65.0%	
Gravity Concentrate																							
Au Grade Gravity Feed	g/t		0.38	0.38	0.38	0.37	0.37	0.37	0.37	0.36	0.36	0.35	0.34	0.33	0.31	0.30	0.29	0.29	0.24	0.24	0.25	0.27	
Au to Au Gravity Conc Recovery	%	30.0%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	30%	
Magnetite Concentrate																							
Mag Grade Mill Feed	%		0.92%	1.85%	2.50%	3.03%	3.77%	4.47%	5.00%	5.39%	5.75%	6.09%	6.27%	6.47%	6.57%	6.49%	6.58%	6.45%	6.14%	6.05%	6.15%	4.76%	
Mag to Mag Conc Recovery	%	95%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	
PAYABLE METALS																							
Cu Metal Payable	M lbs		14.02	14.02	14.02	13.83	13.37	12.99	12.71	12.43	12.15	8.62	8.06	7.71	7.36	7.22	7.01	6.73	6.31	6.31	6.03	4.10	**367**
Ag Metal Payable	k oz		192	214	213	197	180	162	160	163	163	120	120	119	116	110	107	113	110	114	114	83	**5,447**
Au Metal Payable	oz		35,801	35,331	35,519	34,674	34,674	34,862	34,486	33,828	33,452	25,456	24,655	23,783	22,474	21,455	21,164	20,728	17,601	17,237	18,255	12,223	**1,094,272**
Gravity Au Metal Payable	oz	100%	18,734	18,488	18,587	18,144	18,144	18,243	18,046	17,702	17,505	17,210	16,669	16,079	15,194	14,506	14,309	14,014	11,899	11,654	12,342	8,264	**635,150**
Magnetite Payable	k mt	100%	44.56	89.60	121.08	146.75	182.58	216.49	242.16	261.04	278.48	294.94	303.66	313.35	318.19	314.32	318.68	312.38	297.37	293.01	297.85	145.11	**9,657**
REVENUES																							
Cu Sales	US$M		35.05	35.05	35.05	34.58	33.42	32.48	31.78	31.08	30.38	21.56	20.15	19.28	18.40	18.05	17.53	16.82	15.77	15.77	15.07	10.26	
Silver Credit	US$M		2.88	3.21	3.19	2.96	2.70	2.44	2.39	2.45	2.45	1.80	1.81	1.78	1.74	1.64	1.61	1.69	1.64	1.70	1.71	1.24	
Gold Credit	US$M		34.01	33.56	33.74	32.94	32.94	33.12	32.76	32.14	31.78	24.18	23.42	22.59	21.35	20.38	20.11	19.69	16.72	16.38	17.34	11.61	
Total Cu Sales	**US$M**		**71.95**	**71.83**	**71.98**	**70.48**	**69.05**	**68.03**	**66.94**	**65.66**	**64.61**	**47.54**	**45.38**	**43.65**	**41.49**	**40.08**	**39.24**	**38.21**	**34.14**	**33.85**	**34.12**	**23.11**	
Gravity Gold Sales	US$M		17.80	17.56	17.66	17.24	17.24	17.33	17.14	16.82	16.63	16.35	15.84	15.28	14.43	13.78	13.59	13.31	11.30	11.07	11.72	7.85	
Magnetite Sales	US$M		5.79	11.65	15.74	17.34	19.13	20.82	22.11	23.05	23.92	24.75	25.18	25.67	25.91	25.72	25.93	25.62	24.87	24.65	24.89	17.26	
Total Gross Revenues	**US$M**		**167.48**	**172.87**	**177.36**	**175.53**	**174.48**	**174.22**	**173.12**	**171.19**	**169.77**	**136.18**	**131.78**	**128.24**	**123.32**	**119.65**	**118.01**	**115.34**	**104.45**	**103.42**	**104.86**	**71.33**	
Revenues (Net of TCRC)																							
Cu Conc Sales - Net	US$M		66.24	66.11	66.27	64.85	63.61	62.75	61.76	60.60	59.65	44.01	42.08	40.48	38.47	37.12	36.37	35.44	31.55	31.26	31.64	21.42	
Grav Conc Sales - Net	US$M		17.70	17.47	17.56	17.15	17.15	17.24	17.05	16.73	16.54	16.26	15.75	15.19	14.36	13.71	13.52	13.24	11.24	11.01	11.66	7.81	
Mag Sales - Net	US$M		5.79	11.65	15.74	17.34	19.13	20.82	22.11	23.05	23.92	24.75	25.18	25.67	25.91	25.72	25.93	25.62	24.87	24.65	24.89	17.26	
Total Net Revenues	**US$M**		**89.73**	**95.23**	**99.57**	**99.33**	**99.89**	**100.81**	**100.92**	**100.38**	**100.12**	**85.02**	**83.01**	**81.35**	**78.74**	**76.54**	**75.82**	**74.30**	**67.66**	**66.93**	**68.19**	**46.48**	
OPERATING COSTS																							
Mining	US$M		16.46	10.82	9.71	9.21	8.76	8.55	8.44	8.36	8.33	8.33	8.33	8.33	8.33	8.36	8.42	8.52	8.73	8.53	8.32	5.26	**596.111**
Milling	US$M		29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	29.04	18.28	**1,214.316**
G&A	US$M		4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	4.50	2.83	**187.998**
Total Operating Costs	**US$M**		**50.00**	**44.36**	**43.25**	**42.75**	**42.30**	**42.09**	**41.98**	**41.90**	**41.87**	**41.87**	**41.87**	**41.87**	**41.87**	**41.89**	**41.96**	**42.06**	**42.27**	**42.07**	**41.86**	**26.38**	**1,998.425**
CAPITAL EXPENDITURES (in US$M)																							
Capital Expenditures - Sustaining	US$M		0.45	0.45	0.45	0.45	0.45	4.95	0.45	0.45	0.45	0.45	0.45	0.45	0.45	4.95	0.45	0.45	0.45	0.45	0.45	0.45	
Capital Expenditures - Initial	US$M		0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	
Total Capital Expenditures	**US$M**		**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**4.95**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**4.95**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	**0.45**	
CASHFLOW and NPV (before taxes)																							
Net Revenue	US$M		89.7	95.2	99.6	99.3	99.9	100.8	100.9	100.4	100.1	85.0	83.0	81.3	78.7	76.5	75.8	74.3	67.7	66.9	68.2	46.5	
Total Operating Costs	US$M		-50.0	-44.4	-43.2	-42.8	-42.3	-42.1	-42.0	-41.9	-41.9	-41.9	-41.9	-41.9	-41.9	-41.9	-42.0	-42.1	-42.3	-42.1	-41.9	-26.4	
Capital Expenditures	US$M		-0.5	-0.5	-0.5	-0.5	-0.5	-5.0	-0.5	-0.5	-0.5	-0.5	-0.5	-0.5	-0.5	-5.0	-0.5	-0.5	-0.5	-0.5	-0.5	-0.5	
CASHFLOW (before taxes)	US$M		39.3	50.4	55.9	56.1	57.1	53.8	58.5	58.0	57.8	42.7	40.7	39.0	36.4	29.7	33.4	31.8	24.9	24.4	25.9	19.7	
Cumulative Cashflow	US$M		468.7	519.1	575.0	631.2	688.3	742.1	800.6	858.6	916.4	959.1	999.8	1,038.8	1,075.2	1,104.9	1,138.3	1,170.1	1,195.1	1,219.5	1,245.4	1,265.0	
NPV (and yearly PV of Cashflow)	**297.2**	US$	9.2	11.0	11.4	10.6	10.1	8.8	8.9	8.2	7.6	5.2	4.6	4.1	3.6	2.7	2.9	2.5	1.8	1.7	1.7	1.2	
	330.2	C$																					
IRR	**21.5%**																						

The undiscounted annual cash flow and cumulative cash flow for the first 15 years of operations is shown below.

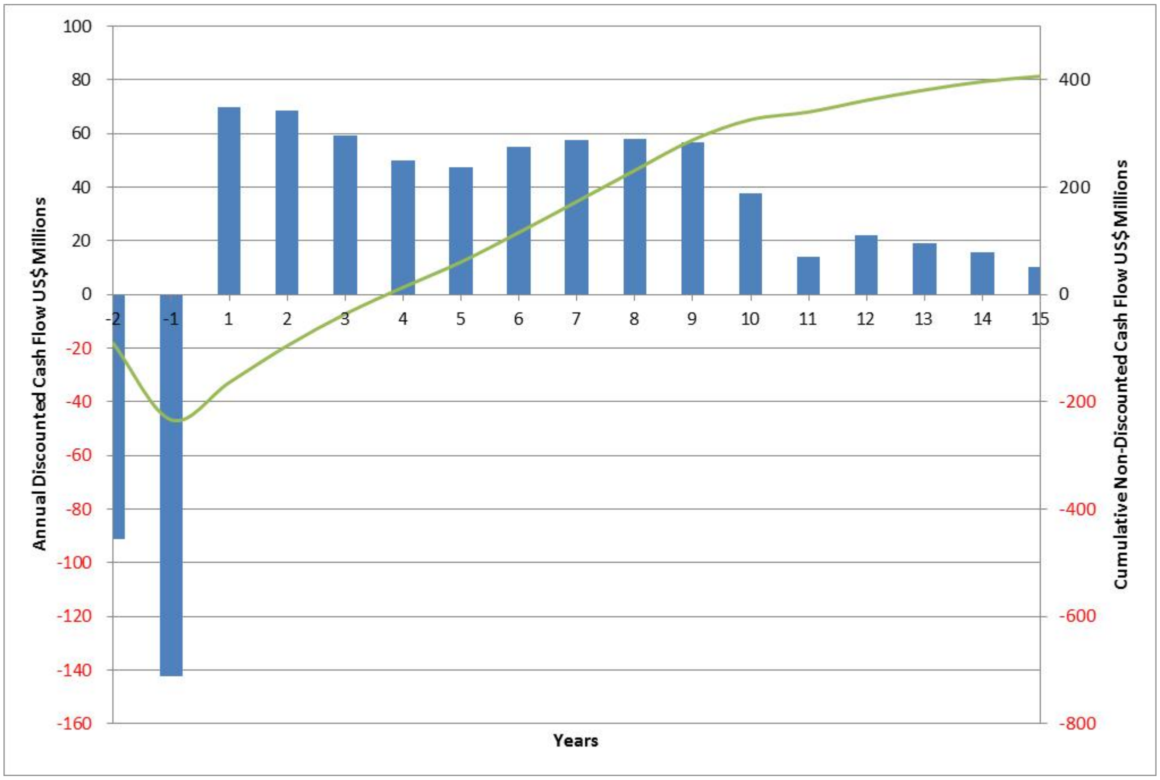

Figure 25-25 Undiscounted Annual Cashflow

25.10 Sensitivity Analysis

A series of sensitivities varying metal prices, recovery, operating costs and capital cost has been developed. The resultant IRR and NPV has been calculated and summarized in Table 25-36.

Table 25-36 Sensitivity Analysis

	IRR	NPV (CDN$Million)
Base Case	**21.5%**	**330.2**
Metal Price		
Current Prices (Sept, 2010)[1]	35.2%	684.2
10% Decrease	16.2%	199.7
10% Increase	26.5%	460.8
Recovery		
10% Decrease	17.6%	232.5
OPEX		
20% increase	16.4%	198.6
10% decrease	23.8%	396.1
CAPEX		
20% Increase	17.4%	280.9
10% Decrease	24.2%	354.9
Exchange Rate (1 CAD=0.9USD)		
1CAD = 0.95 USD	18.9%	264.6
1 CAD= 0.85 USD	24.3%	403.5
Discount Rate (Base Case 7.5%)		
5%	21.5%	514.2
10%	21.5%	215.2

[1]*Current prices used are USD$1250 Au; USD$3.50 Cu; USD$20 Ag; USD$90 Mag.*

26.0 List of Acronyms

ARD – Acid Rock Drainage
ASTM – American Society for Testing and Materials.
BCMW Bank Cubic Meter Waste
BCMRC Bank Cubic Meter Raw Coal
BCR – BC Rail
CAD – Canadian Dollars
FOB – The abbreviation for "free on board". The FOB price is the sales price of coal loaded in a vessel at the port and excludes freight or shipping cost
FSI (Free Swelling Index) - A number assigned to particular coal used in determining its suitability for coke making or other uses. The index, from zero to nine, is determined by tests established by ASTM standards
GSC – Geologic Survey of Canada
MEMPR – Ministry of Mines and Petroleum Resources
ML – Metal Leaching
MSEP – MineSight Economic Planner used to produce economic pit limits
MSSP – MineSight Strategic Planner used to produce an economic production schedule
MTCC – Metric Tonne Clean Coal
MTRC – Metric Tonne Raw Coal
NTS –
QP – Qualified Person
USD – US Dollars
UTM –

Above mean sea level	amsl
Ampere	A
Annum (year)	a
Bank cubic metre	bcm
Cubic metre	m3
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Degree	°
Degrees	deg
Degrees Celsius	°C
Diameter	ø
Dry metric ton	dmt
Gram	g
Grams per cubic centimetre	g/cc
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m2)	ha
Hertz	Hz
Horsepower	hp
Hour (not hr)	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a

Inch	"
Joule	J
Joules per kilowatt-hour	J/kWh
Kelvin	K
Kilo (thousand)	k
Kilocalorie	kcal
Kilogram	kg
Kilograms per cubic metre	kg/m3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m2
Kilojoule	kJ
Kilometre	km
Kilometres per hour	km/h
Kilonewton	kN
Kilopascal	kPa
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per short ton (US)	kWh/st
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Kilowatts adjusted for motor efficiency	kWe
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere.	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres per hour	m/hr
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Micrometre (micron)	µm
Microsiemens (electrical)	µs
Miles per hour	mph
Milliamperes	mA
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million tonnes	Mt
Minute (plane angle).	’
Minute (time)	min

Month	mo
Newton	N
Newtons per metre	N/m
Ohm (electrical)	Ω
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Pascal (newtons per square metre)	Pa
Pascals per second	Pa/s
Percent	%
Percent moisture (relative humidity)	% RH
Phase (electrical)	Ph
Power factor	pF
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Short ton (2,000 lb)	st
Short ton (US)	st
Short tons per day (US)	stpd
Short tons per hour (US)	stph
Short tons per year (US)	stpy
Specific gravity	SG
Square kilometre	km2
Square metre	m2
Thousand tonnes	kt
Tonne (1,000 kg)	t
Tonnes per annum	tpa
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Total dissolved solids	TDS
Total suspended solids	TSS
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Yard	yd
Year (annum)	a
Year (US)	y

27.0 Glossary

Air Dried Basis (adb) Coal that has been left to dry in air and has an approximate 'dry' moisture of 1%
Ash - Impurities consisting of silica, iron, alumina and other incombustible matter that are contained in coal. As increases the weight of coal and adds to the cost of handling. Ash content is measured as a percentage by weight of coal on an "as received" or a "dry" (moisture-free) basis.
As Received Basis (arb) Coal as received with in-situ/drained moisture content assumed to be 8%.
Coal Washability - The analysis of the specific gravity distribution of chemical and physical characteristics of coal.
Drillhole - A circular hole made by drilling either to explore for minerals or to obtain geological information.
Dip - The angle at which a stratum is inclined from the horizontal, measured perpendicular to the strike and in the vertical plane.
Dry Basis (db) - Coal that has moisture removed by prescribed laboratory procedure or excluded by calculation.
Exploration - The search for coal by geological surveys, prospecting or use of tunnels, drifts or drillholes.
Fault - A fracture in rock along which the adjacent rock surfaces are differentially displaced.
First Nations - An aboriginal governing body organized and established by aboriginal people within their traditional territory in British Columbia, which has been mandated by its constituents to enter into treaty negotiations on their behalf with Canada and British Columbia.
Fixed Carbon - The solid residue, other than ash, remaining after the volatile matter and moisture have been liberated from coal during combustion.
Float/Sink - A laboratory procedure, which measures the floating and sinking of particles of material of various size fractions in heavy liquids at various specific gravities.
Front End Loader - A tractor or wheel type loader with a digging bucket mounted on the front end that dumps.
Geophysical Log - A graphic record of the measured or computed physical characteristics of the rock section encountered by a probe or sonde in a drillhole, plotted as a continuous function of depth. Also commonly referred to as an e-log.
Highwall - The unexcavated face of exposed overburden and coal or ore in an opencast mine or the face or bank of the uphill side of a contour strip-mine excavation.
Interburden - Waste material located between economically recoverable resources.
Isopach - The areal extent and thickness variation of a stratigraphic unit in geology.
Lease - A contract between a landowner and a lessee, granting the lessee the right to search for and produce coal upon payment of an agreed rental, bonus and/or royalty.
Metallurgical - Coal with characteristics making it suitable for production of coke that can be used by the iron and steel industry.
Mineable - Capable of being mined under current mining technology and environmental and legal restrictions, rules and regulations.
Out-of-Seam Dilution (OSD) - The contamination of mined coal with rock outside of the coal seam being mined.
Outcrop - Coal, which appears at or near the surface; the intersection of a coal seam with the surface.
Overburden - The rock, earth or other material lying over the coal.
Proximate Analysis - Laboratory analysis to determine the percentage by prescribed methods of moisture, volatile matter, fixed carbon and ash.

Pulverized Coal Injection (PCI) - Low-grade metallurgical coking coal.
Raw Coal - The coal that remains after oversized OSD material has been removed in the breaker station and which is the feedstock for the preparation plant.
Reclamation - The restoration of land at a mining site after the coal is extracted. Reclamation operations are usually conducted as production operations are taking place elsewhere at the site. This process commonly includes re-contouring or reshaping the land to its approximate original appearance, restoring topsoil and planting native grasses, trees and ground covers.
Rotary Drill - A drill machine that rotates a rigid, tubular string of rods to which is attached a bit for cutting rock to produce boreholes.
Royalty - A share of the product or profit reserved by the owner for permitting another to use the property. A lease by which the owner or lessor grants to the lessee the privilege of mining and operating the land in consideration of the payment of a certain stipulated royalty on the mineral produced.
Run-of-Mine Coal (ROM) - The coal produced from the mine before it is separated and any impurities removed.
Saleable Coal - The shippable product of a coal mine or preparation plant. Depending on customer specifications, saleable coal may be run-of-mine, crushed-and-screened (sized) coal, or the clean coal from a processing plant.
Strip Ratio - The volume of overburden material (bank cubic meters) that must be removed to provide a unit weight of coal (tonne).
Surface Mining - Methods of mining at or near the surface. Includes mining and removing coal from open cuts with mechanical excavating and transportation equipment and the removal of capping overburden to uncover the coal.
Syncline - A fold in which the core contains the stratigraphically younger rocks; it is generally concave upward.
Tailings - Fine refuse material or waste that has been separated from the fine clean coal in the froth flotation cells in the coal processing plant.
Thermal Coal - Coal with characteristics making it suitable for burning to produce steam for generating electricity.
Thrust Fault - A fault with a dip of 45 degrees or less over much of its extent, on which the hanging wall appears to have moved upward relative to the footwall.
Train Loadout - A facility to load coal in rail cars.
Volatile Matter - Those products, exclusive of moisture, given off by a material such as gas or vapour, determined by definite prescribed methods, which may vary according to the nature of the material.
Yield - The ratio of the clean coal product to the raw coal plant feed, expressed as a percentage.