Copper Fox Metals Inc.

Canadian NI43-101
Technical Report

Preliminary Economic Assessment on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada

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1.0 Executive Summary

1.1 Overview of the Study

Copper Fox Metals Inc. (Copper Fox) commissioned a Preliminary Economic Assessment (PEA) for its Schaft Creek project in 2006 in an effort to assist the management team make decisions regarding the potential development of the project. Site and investigative work continue with the intention to follow up on this report and produce a prefeasibility study (2Q2008) and a feasibility study (4Q2008). The scoping study benefits from two years of site work by numerous companies and consultants and was prepared to define the overall scope of the Schaft Creek project, perform preliminary mine planning, report on metallurgical testwork and process design, estimate capital and operating costs and determine the economics to develop the project as an open pit mine and mill facility.

This report is a preliminary economic assessment (PEA), by which meaning the report is a preliminary assessment 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 prefeasibility study.

By the CIM Definition Standards on Mineral Resources and Mineral Reserves, a mineral reserve has to be supported by at least a prefeasibility study demonstrating economic viability of the project. It is recognized that the term “ore” cannot be used unless it is associated with a mineral reserve, however, the word “ore” is used throughout this document to refer only to mineralized material within the resource and mill feed that would be delivered to and processed in the proposed concentrator.

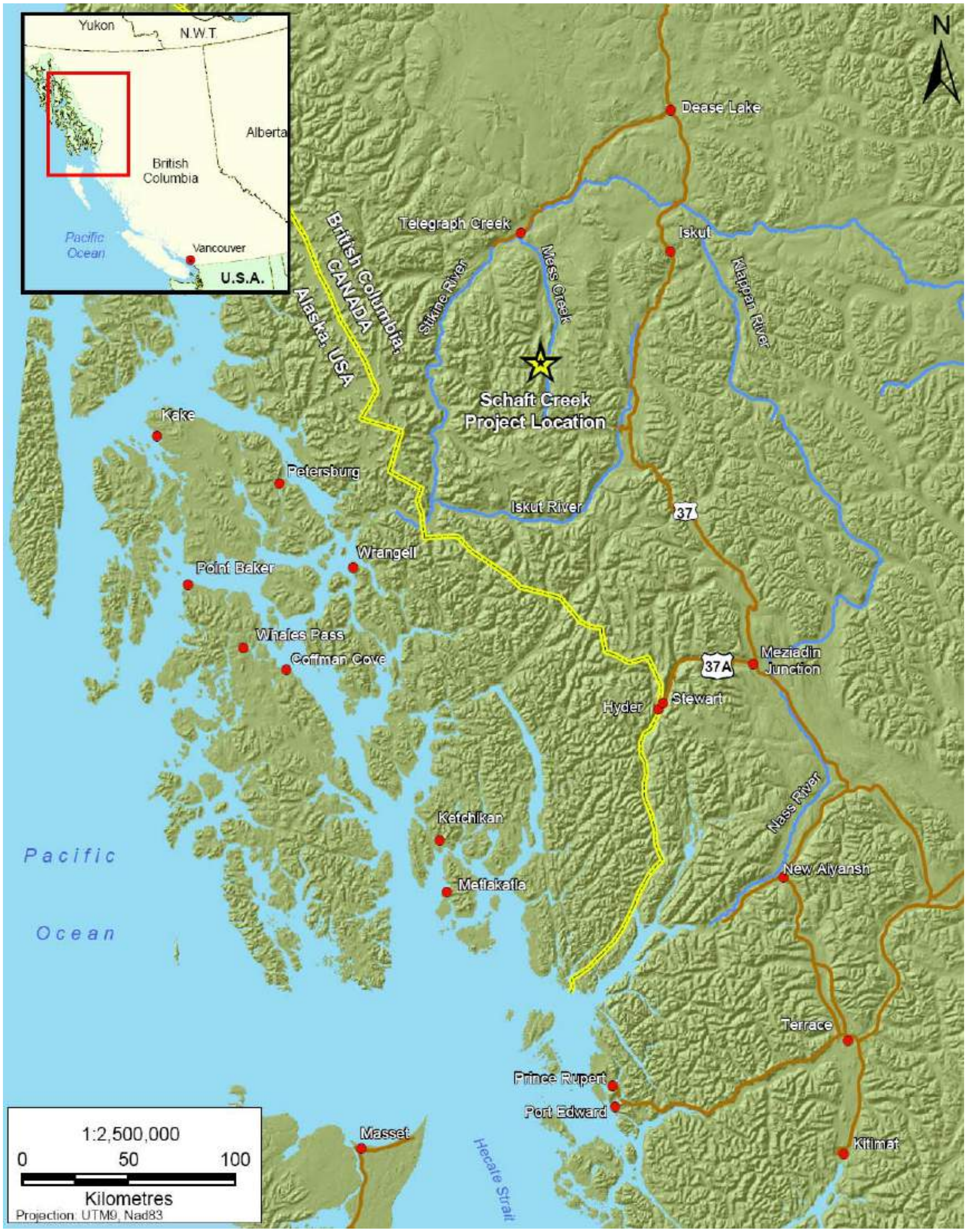
1.2 Schaft Creek Property Location, Description and History

1.2.1 Property Location

The Schaft Creek property is comprised of an area totaling approximately 20,932 ha within the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP) area, located in northwestern British Columbia. The property is positioned within the upper source regions of Schaft Creek, which drains northerly into Mess Creek and onwards into the Stikine River. Located within the Boundary Range of the Coast Mountains, the elevation of the valley at the Schaft Creek campsite is 866 m with nearby mountains exceeding 2,400 m.

The Schaft Creek property is approximately 60 km south of the village of Telegraph Creek, 45 km due west of Highway 37, and approximately 375 kilometres northwest of the town of Smithers.

Smithers is the closest supply center with the capacity to service the project during construction and operation. The property also falls within the traditional territory of the Tahltan Nation. Three predominantly Tahltan communities are within 125 km of the property; Telegraph Creek, Dease Lake and Iskut. All three of these communities will provide labour during construction and operation of the mine and are accessible via Highway 37.


Yukon
N.W.T.
Alberta
British Columbia
Pacific Ocean
Vancouver
U.S.A.
N
Dease Lake
Telegraph Creek
Iskut
British Columbia, CANADA
Alaska, USA
Stikine River
Mess Creek
Klappan River
Schaft Creek Project Location
Kake
Petersburg
Iskut River
37
Wrangell
Point Baker
Whales Pass
Coffman Cove
37A
Meziadin Junction
Hyder
Stewart
Nass River
Ketchikan
Pacific
Metlakatla
New Aiyansh
Ocean
Terrace
Prince Rupert
Port Edward
Masset
Kitimat
Hecate Strait
1:2,500,000
0
50
100
Kilometres
Projection: UTM9, Nad83

1.2.2 Property Description

The Schaft Creek property is a remote 'greenfield' site with no developed roads leading into it and is best accessed by helicopter from Bob Quinn, a small outpost located 80 km southeast of the property on Highway 37. The Burrage airstrip, situated 37 km east of Schaft Creek on Highway 37, also provides a means of access by helicopter and fixed wing. Alternatively, fixed wing aircraft can be chartered from Smithers, B.C. and flown directly to the Schaft Creek camp, utilizing an existing gravel airstrip at the site.

Drill roads have been established within the immediate project area and total approximately ten km of gravel and mud trails. Original construction of the camp facilities at Schaft Creek commenced circa 1965 and in 1967. During the interval from 1968 to 1981, when Hecla Mines and subsequently Teck Corporation aggressively explored the property, most of the site infrastructure was established. Copper Fox re-built the camp to include, a fuel storage depot, two bunk houses accommodating 32-personnel, a new kitchen and dining facility with a 42-person capacity, a new shower and laundry facility attached to the lavatory building, mechanic's shop, generator shack, core shack, log assay shack, recreation hall, sleep cabins, office and first-aid buildings, and a small, pre-fabricated cedar log cabin. The 750 m long airstrip system includes two aforementioned gravel strip runways, one oriented in a general north-south direction was established immediately west of the camp, adjacent to the eastern bank of Schaft Creek, while the second is oriented in a northeast-southwest direction.

Since the Schaft Creek site is located in an alpine environment, the climate is characterized by mild summers and cold winters. The mean monthly temperatures typically remain above freezing from April to October and drop below freezing from November through March. Annual precipitation averages between 700 to 1100 mm. Approximately 60% of the precipitation occurs as snow, which can reach a depth greater than 2 m and persist into June.

1.2.3 History

The history of the Schaft Creek property is summarized below:

- 1957, discoveries nearby spurred exploration northward into the Schaft Creek-Mess Creek areas, leading to the discovery of mineralization at Schaft Creek.
- Area staked in 1957 for the BIK Syndicate; subsequently completed 3,000 ft (914.4 m) of hand trenching.
- 1956, mapping, IP survey and 3-holes were drilled by Silver Standard Mines Ltd., totaling 2, 063 ft (629 m).
- 1966, Liard Copper Mines Ltd. was formed to consolidate area land holdings.
- 1966, Asarco options the property; a 4,000 ft (1,219.2 m) airstrip was constructed, a camp was built and 24 holes were drilled, totaling 11,000 ft (3,352.8 m).
- 1967, in mid-spring of the year, a D6 Cat walked from Telegraph Creek. A second 4,000 ft (1,219.2 m) airstrip was built and construction of the camp

continued. Asarco initially drills 2 holes and continues to complete 22 additional holes for a program total of 24 holes, amounting to 11,000 ft (3,352.8 m). Paramount Mining drills 1 hole.

- 1968, Asarco drops option and Hecla Mining acquires the property. The airstrip was extended to 5,280 ft (1,609.3 m).
- 1968, Hecla drills 9 holes, totaling 13,095 ft (3,991.4 m) 3 of the holes were drilled in the Paramount Zone.
- 1969, Hecla drills 9 holes, totaling 15,501 ft (4,724.7 m).
- 1970, Hecla drills 26 holes, totaling 32,575 ft (9,928.9 m). 5 of the holes were drilled in the Paramount Zone.
- 1971, Hecla drills 25 holes, totaling 22,053 ft (6,721.8 m). 3 of the holes were drilled in the Paramount Zone.
- Total Hecla footage; 83,224 ft (25,366.7 m) of which 8,610 (2,624.3) m were drilled on the Paramount Property and 74,614 were drilled on the Schaft Creek Property.
- 1972-1977, Hecla drilled 35 holes, totaling 38,386 ft (11,700.1 m).
- 1977, 104 holes drilled on the properties held by Hecla, totaling 113,000 ft (34,442.4 m). A reserve of 505 Mt with 0.38% Cu and 0.039% MoS2 delineated.
- Between 1978 and 1979, Hecla Mining forfeits option and Teck Corp. acquires the property.
- 1980, Teck Corp. drilled 47,615 ft (14,513.1 m) in 45 holes, between mid-May to mid-November. The drill sites were prepared with a D6 Caterpillar bulldozer. Assaying of core on 10 ft (3.05 m) sample intervals, by Afton Mines Ltd. in Kamloops.
- 1981, between June and September, Teck Corp. drilled 33,315 ft (10,154.4 m) in 73 holes, and 3,503 ft (1,067.7 m) of condemnation drilling for a tailings pond and mill sites.
- Resource expanded to a global estimate of 1 Gt with 0.30% Cu and 0.034% MoS2.
- Total property drilling is 197,500 ft (60,198 m), in 230 holes.
- 2002, Mr. G. Salazar acquired the right to secure a significant ownership of the property.
- 2005, Mr. G. Salazar incorporated the Schaft Creek property into the holdings of Copper Fox Metals Inc. Copper Fox Metals Inc. then proceeded to obtain the necessary funding to undertake the 2005 program.

1.3 Geology and Resources

1.3.1 Regional Geology

The Schaft Creek copper porphyry (Cu±Mo, Au, Ag) deposit is one of a number of porphyry deposits of similar age and affinity distributed throughout the Intermontane belt of the Canadian Cordillera. The Schaft Creek deposit is located in the Stikina Terrain, which is the westernmost and most aerially extensive terrain of the three known to host significant porphyry copper mineralization within the Intermontane belt. A large number of porphyry copper deposits occur in this terrain, particularly in the north-central portion. Besides the

Schaft Creek deposit, other significant deposits within the Stikina Terrain include the Red-Chris, Galore Creek, Kerr, Kemess, and Huckleberry deposits.

1.3.2 Property Geology

The Schaft Creek deposit is situated in the valley of Schaft Creek along the western slope of Mount LaCasse. The deposit is bounded to the west by the Hickman batholith and to the east by volcanic rocks of the Mess Lake facies. The valley floor exposes the Stuhini group volcanics and conforms to the contact zone of these volcanics with the east margin of the Hickman batholith. Topography within the valley floor is very subdued and largely covered by glacio-fluvial gravels. Bedrock exposures are very scarce in the lower elevations of the valley floor.

The deposit is hosted by north striking, steep, easterly dipping volcanic rocks comprised of a package of: andesitic pyroclastics ranging from tuff to breccia tuff; and aphanitic to augite-feldspar-phyric andesite. The deposit is elongated in a general north-south direction.

Narrow, discontinuous feldspar porphyry and quartz feldspar porphyry dikes, related to the Hickman batholith, intrude the volcanic package. The batholith is considered to be the source of the magmatic-hydrothermal fluids, which ultimately formed the mineralized breccias, veins and stockworks of the deposit.

Three geologically distinct spatially separate zones, representing distinct porphyry environments constitute the Schaft Creek deposit. The largest of these zones is the Liard/Main zone, which is characterized by syn-intrusive poly-phase quartz-carbonate veins and stockworks, and mineralized with variable amounts of chalcopyrite, bornite and molybdenite and late fracture molybdenite.

The second largest zone is the Paramount zone, which is characterized by; primary sulphide mineralization associated with an intrusive breccia phase, containing chalcopyrite, bornite and molybdenite; quartz-carbonate stockworks; and late fracture molybdenite mineralization.

The smallest of the zones is the West Breccia zone. It is characterized by quartz tourmaline veining, pyrite and a hydrothermal breccia.

1.3.3 Resources

The Schaft Creek deposit is a large, multi-phase, complex, porphyry copper-molybdenum-gold-silver system consisting of three distinct, semi-continuous, and structurally modified zones genetically related to the Hickman batholith. The individual zones represent differing levels within the porphyry and correspond with increasing depth in the following order; the West Breccia zone occupies the high level, the Liard/Main zone occupies the medium level and the Paramount zone the deepest level. All of the zones have been structurally controlled, with the earliest mineralizing event strongly influenced by syn-intrusive fracturing and faulting; while, post formational faulting associated with accretionary tectonics modified the deposit considerably.

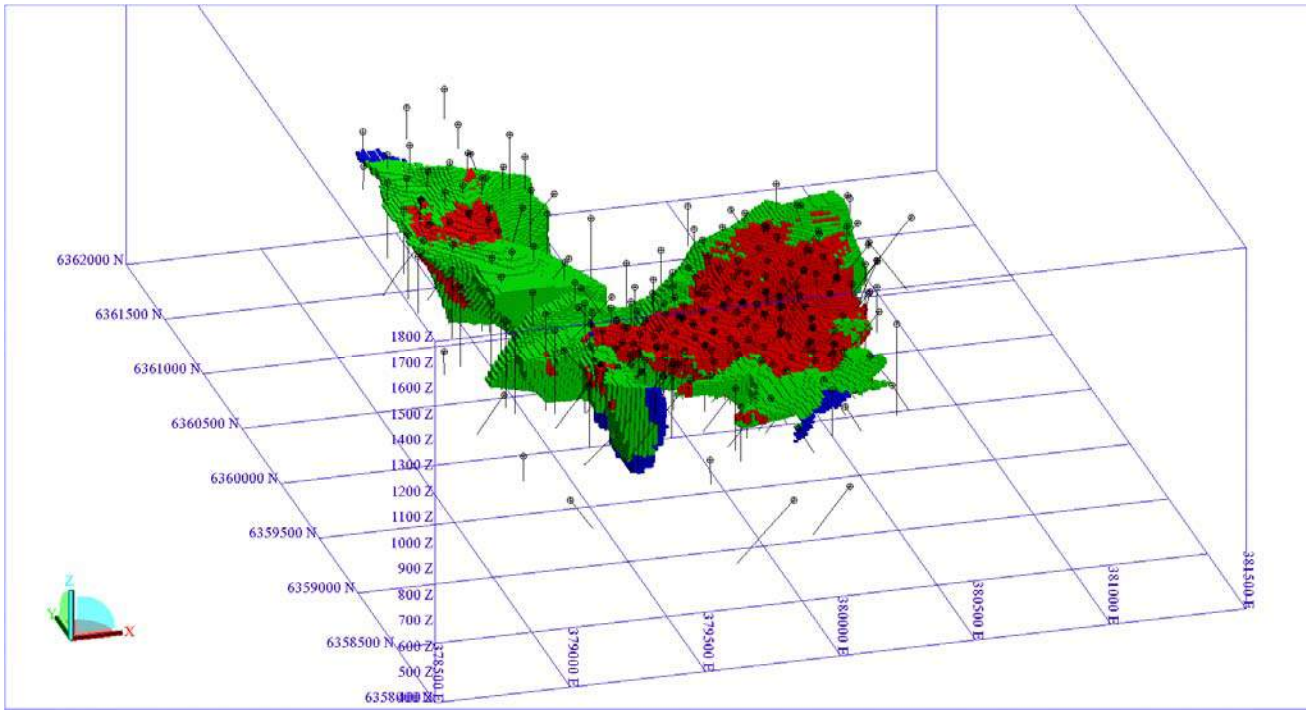


Table 1.1 Schaft Creek Mineral Resource Estimate Summary ≥0.20 % Copper Equivalent Cut-Off						
	Tonnes	**Cu (%)**	**Mo (%)**	**Au (g/t)**	**Ag (g/t)**	**CuEq Grade (%)**
Measured Mineral Resources **(Red)**	463,526,579	0.30	0.019	0.23	1.55	0.46
Indicated Mineral Resources **(Green)**	929,755,592	0.23	0.019	0.15	1.56	0.36
Measured + Indicated Mineral Resources	1,393,282,171	0.25	0.019	0.18	1.55	0.39
Inferred Mineral Resources **(Blue)**	186,838,848	0.14	0.018	0.09	1.61	0.25

The deposition of sulphides at Schaft Creek is the result of a complex polyphase series of mineralizing events. Macroscopic determinations on the Copper Fox drill core define the deposit's sulphide mineral composition as: chalcopyrite (50%), pyrite (22.8%), bornite (14.2%) and molybdenite (13%). Chalcopyrite and bornite, the most essential copper ore minerals, occur in stockworks, as disseminations, and in breccias. Less commonly, chalcopyrite is observed as very thin (10-100 micron) partial coatings on ubiquitous, dm spaced fractures and joints. Molybdenite is also a critical sulphide component of the ore. It occurs as disseminated blebs and stringers in stockworks and veins and is quite common in the breccia zones. Quite often it forms thin coatings on slickensides and fractures.

Hydrothermal breccia matrix is the infilling of inter-clast space for hydrothermally deposited chlorite, carbonate, quartz, tourmaline and sulphides. This style of mineralization is an important but volumetrically smaller ore type in the West Breccia and Paramount zones. Chalcopyrite, bornite, minor molybdenite and trace pyrite are the dominant sulphides and are generally coarse-grained, ranging from 1 to 10 mm.

Two of these zones are dominated by breccia facies, namely the West Breccia zone and the Paramount zone; the third, the Main zone, is characterized by stockworks and structurally controlled vein system. Veining and stockworks at Schaft Creek cover an area 1,400 m long by 300 m wide and form a complex system. Various terminologies are used to refer to and describe veining. Information on veining is derived from all three zones, the Main, West Breccia, and Paramount.

1.4 Metallurgy

1.4.1 Historical Test Programs

The Schaft Creek resource has been given considerable study starting from the early studies by Hecla Mining Company (Hecla) in 1970 and 1971. The following Table 1.2 summarizes the historical test work:

Table 1.2
Historical Test Work Summary

Test Type	Laboratory	Mining Company
Preliminary Flotation Tests	Lakefield Research	Hecla Mining Company - 1970-71
Preliminary Flotation Tests	Lakefield Research	Tech Mining Group - 1981-82
Sample Validation	Process Research Assoc.	Copper Fox Metals, Inc. - 2004
Laboratory Flotation Test	Process Research Assoc.	Copper Fox Metals, Inc. - 2005
Laboratory Flotation Tests	Process Research Assoc.	Copper Fox Metals, Inc. - 2006
Laboratory Flotation Tests	Process Research Assoc.	Copper Fox Metals, Inc. - 2007

1.4.2 Review of the 2006 Drill Core Tests at PRA

1. PRA started testing the 2006 core samples in February 2007. Of a total of 50,000 kg of drill core, approximately 725 kg were used by PRA for the laboratory program and 6,000 kg for pilot plant testing. Vandan Suhbatar and Raymond Hyyppa visited the Schaft Creek property from October 13 through October 16, 2006 to review the property and drill core and to select the samples of PQ drill core from the 2006

drilling season for additional testing. Each of three resource areas - Liard (LZ), West Breccia (WBZ) and Paramount (PZ) were drilled during the 2006 season and representative samples identified and collected (384 kg of Liard, 169 kg of Paramount and 172 kg of West Breccia) for the PRA laboratory test program.

2. Drill location maps, including holes drilled by Tech and Hecla were provided and the anticipated pit limits for each area were outlined. Material was selected from areas within the anticipated pit limits. The drill logs and 3 metre interval assay data were evaluated to determine the drill intervals that would provide an average copper assay of 0.35 to 0.40 % copper from each resource area. Interval assays were not available for all of the selected holes. Approximately 90 kg of samples from each of three zones were sent to Hazen Research for comminution testing. The comminution samples were selected from the same holes as used for the samples sent to PRA. Also, 2,000 kg of each zone (6,000 kg total) were crushed to minus 10 mesh in preparation for pilot plant testing.
3. PRA prepared a composite for each of the three resource areas and a fourth composite (Master) - was prepared with equal portions of the three zones. Since test results indicated the optimum primary grind to be P_{80} = 100 microns, this was the size selected for all Rougher and Scavenger Flotation locked cycle tests. (Figure 16.13). Rougher and Scavenger concentrates were reground to P_{80} = 20 to 25 microns. A grind size versus time calibration was made for lab rod mill for each individual composite. The results of the four locked cycle tests (Table 16.7) indicate a need to regrind the feed to the cleaner circuit to 15 microns in order to achieve both high concentrate grades and recoveries. Higher concentrate grades were achieved for the 2005 drill core tests using a P_{80} of 15 to 20 microns. Comparisons of locked cycle test data for the 2005 (Table 16.5) and 2006 drill (Table 16.7) core, shows that flotation feed ground to a P_{80} = 100 microns results in higher metal recoveries and a regrind of Rougher and Scavenger Flotation Concentrate to P_{80} = 15 microns results in higher metal grades.
4. The locked cycle test data for 2005 (average of MLS, WLZ and NLZ samples) and the 2006 drill core for the Liard Zone also indicate that for similar head grades (0.38 % $Cu_{2005\text{-}LiardZone}$ and 0.33% Cu_{2006}), the finer primary grind of P_{80} = 109 microns results in higher 3rd Cleaner copper recovery by 6.7% (84.10 % Cu_{2006} vs. 77.40 % Cu_{2005}). It is believed that the finer regrind size used for the 2005 samples ($P80_{2006}$ = 20 microns vs. $P80_{2005}$ = 16 microns) was due to their difference in head grade as these samples appear to be very sensitive to grind size. Reagent selection can play a major part in influencing concentrate grade. It is possible that the lower 3rd Cleaner Concentrate grade for the 2006 core samples can be increased with a different reagent selection and dosage. Finally, the 2005 core samples may have a larger proportion of secondary copper minerals that would facilitate higher copper concentrate grades.
5. Information received to date indicates that the life of mine resource would comprise approximately 60% Liard Zone, 25% Paramount Zone and 15% West Breccia Zone materials. Using this assumption with a primary grind size of 80%, passing 100 microns and a regrind size of 80% passing 15 to 20 microns, the following average metal grades and assays can be expected in a Bulk Copper/Molybdenum/Gold/Silver Concentrate.

	Concentrate Recovery	Bulk Concentrate Grade	Copper Concentrate Grade	Moly Concentrate Grade
Copper	90.0%	26.03%	26.5%	0.42%
Molybdenum	72.0%	1.20%	0.27%	54.0%
Gold	82.0%	24.0 g/t	18.4 g/t	-
Silver	72.0%	114 3 g/t	113.2 g/t	-

These values were used for the METSIM mass balance. The molybdenum balance was prepared assuming that 90% of the molybdenum contained in the Bulk Copper/Molybdenum/Gold/Silver Concentrate would report to the Molybdenum Concentrate at a grade of 54% Mo. These assumptions are currently being tested at G & T laboratory.

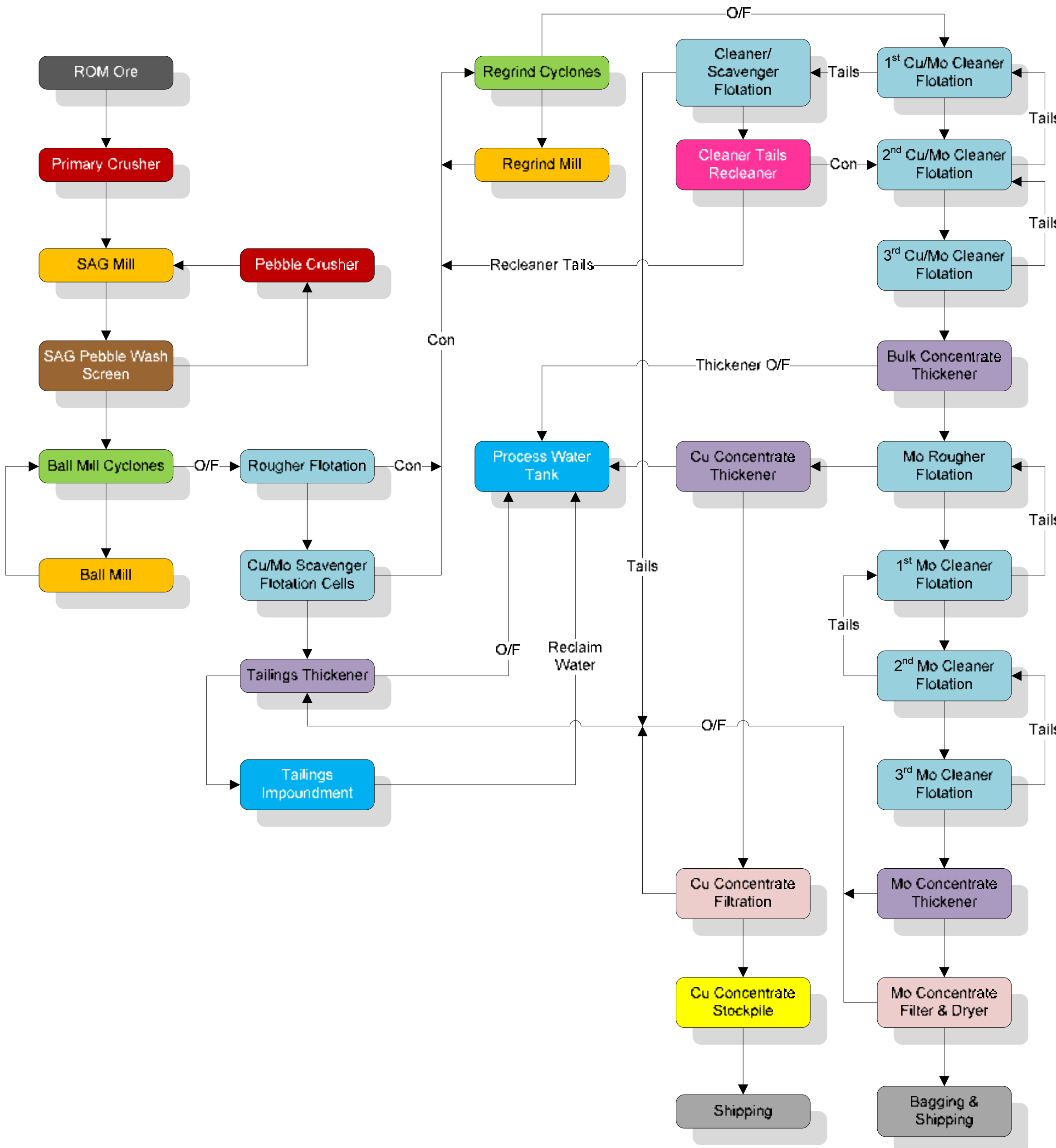
1.5 Process

The Schaft Creek concentrator will have an annual throughput of 23,400,000 tonnes. Copper Fox will construct the concentrator on site which will include a typical comminution (SABC) circuit followed by a flotation circuit and a copper circuit with thickener, filtration and concentrate loadout and shipping. The mill includes a dedicated molybdenum circuit with thickener, filtration circuit, drying and bagging. Tailings thickeners, tailings facility and water reclaim are part of the tailings facilities. This circuit will have a design capacity of 70,652 tonnes per day and a nominal capacity of 65,000 tonnes per day.

It should be noted here that the 65,000 tpd milling rate translates into a 31 year mine life. While a 31 year mine life is not typical in the industry at this time, the 65,000 tpd milling rate was selected based soley on power supply limitations to the area. It is recommended that Copper Fox push the milling rate to 100,000 tpd to bring the mine life to a ‘reasonable’ period and as such are in discussions with BC Hydro about the NW Transmission Extension project that would bring a new 287 kV line to Bob Quinn. This would provide the necessary power supply for the increased milling rate.

1.5.1 Block Process Flow Diagram

A simplified block process flow diagram and the basic design criteria are presented on the following pases. Complete Process Flow Diagrams were developed for the project as well as mechanical and electrical equipment lists, load analysis and single line diagrams.


ROM Ore
Primary Crusher
SAG Mill
Pebble Crusher
SAG Pebble Wash Screen
Ball Mill Cyclones
Ball Mill
O/F
Rougher Flotation
Con
Cu/Mo Scavenger Flotation Cells
Tailings Thickener
Tailings Impoundment
Regrind Cyclones
Regrind Mill
Recleaner Tails
Process Water Tank
O/F
Reclaim Water
Cleaner/ Scavenger Flotation
Cleaner Tails Recleaner
Tails
1st Cu/Mo Cleaner Flotation
2nd Cu/Mo Cleaner Flotation
3rd Cu/Mo Cleaner Flotation
Bulk Concentrate Thickener
Thickener O/F
Cu Concentrate Thickener
Mo Rougher Flotation
1st Mo Cleaner Flotation
2nd Mo Cleaner Flotation
3rd Mo Cleaner Flotation
Mo Concentrate Thickener
Mo Concentrate Filter & Dryer
Bagging & Shipping
Cu Concentrate Filtration
Cu Concentrate Stockpile
Shipping

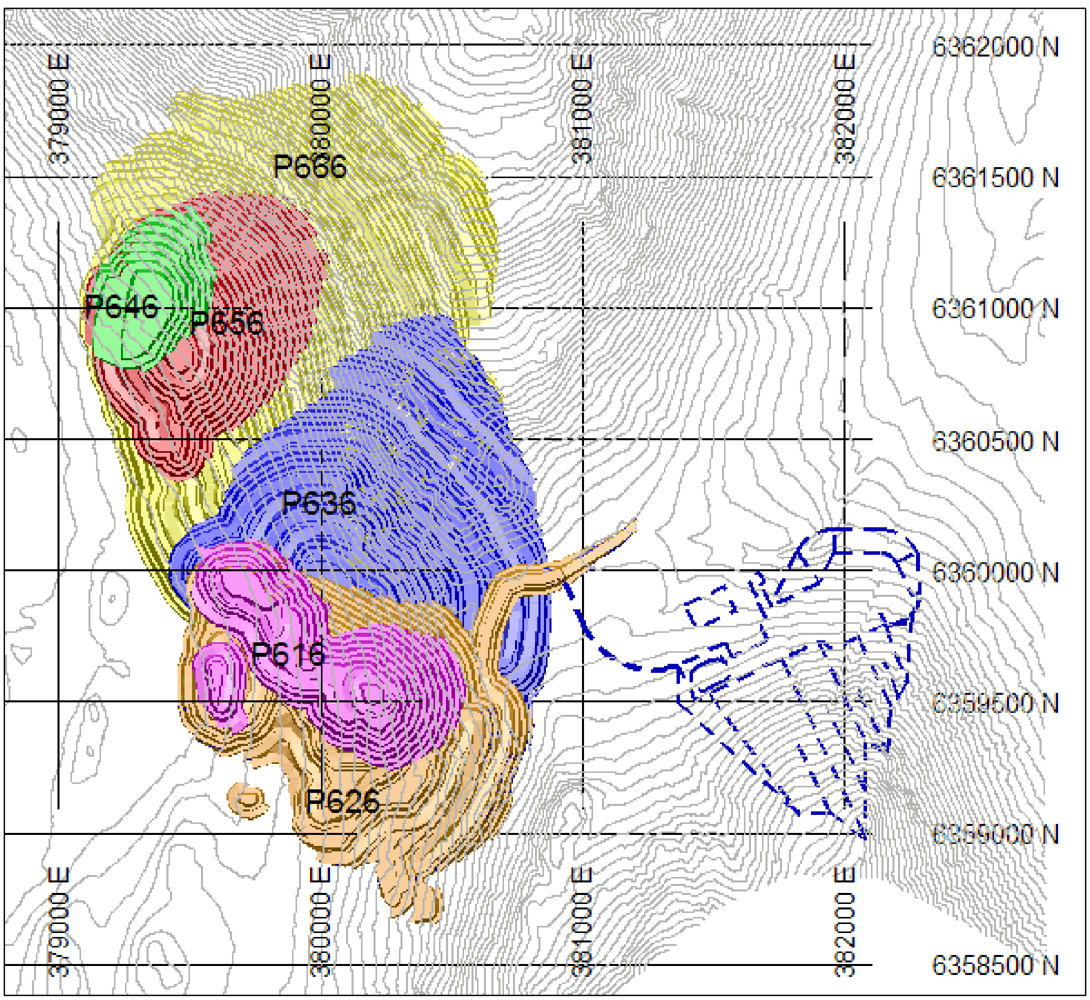
1.5.2 Basic Design Criteria

Table 1.3 Schaft Creek Basic Design Criteria				
	Units	**Balance**	**Design**	**Source**
General Site Information				
Location				
Latitude - Approximate	angular		N57° 32' 18"	
Longitude - Approximate	angular		W131° 01' 09"	
Elevation				
Air Strip	masl		1,040	SE
Plant	masl		1,230	SE
Pit Bottom	masl		600	MMTS
Ambient Air Temperature				
Average Monthly Minimum	°C		-30	
Average Monthly Maximum	°C		28	
Average Annual Precipitation	mm/y		640	
General Project Information				
Reported Resource	tonnes (t)		1,393,282,000	AGL
Cutoff Grade Used	CuEq (%)		0.20	AGL
Estimated Mineable Resources				
Starter Pit (5 Year)	tonnes (t)		117,050,000	MMTS
Life of Mine (includes subgrade to waste)	tonnes (t)		719,091,000	MMTS
Operating Schedule				
Hours per Day	h	24	24	MMTS
Days per Year	d	360	360	MMTS
Hours per Year	h	8,640	8,640	MMTS
Plant Capacity (at 92% availability)	dmtpd	65,000	70,652	MMTS
Plant Capacity (at 92% availability)	dmtph	2,944	2,944	
Annual Ore Processed per Year	t	23,400,000	23,400,000	MMTS
Mineable Resource to Mill	t	713,387,500	713,387,500	MMTS
Estimated Project Life @ 65,000 tpd	y	30.5	30.5	MMTS
Life of Mine Plant Head Grade Estimates				
Estimated Copper Grade	%		0.303	MMTS
Estimated Molybdenum Grade	%		0.020	MMTS
Estimated Gold Grade	g/t		0.217	MMTS
Estimated Silver Grade	g/t		1.761	MMTS
Plant Design (First 5 Years) Head Grade Est				
Estimated Copper Grade	%		0.350	MMTS
Estimated Molybdenum Grade	%		0.018	MMTS

Table 1.3 Schaft Creek Basic Design Criteria				
Estimated Gold Grade	g/t		0.268	MMTS
Estimated Silver Grade	g/t		1.869	MMTS
Design ROM Ore Dry Solids Sp Gr	g/cc		2.69	CFMI
Design ROM Ore Moisture (for material handling)	%		3	CFMI
First Five-year Average Copper Conc Production				
Copper Recovery to Copper Concentrate	%		90.3	PRA
Copper Grade in Copper Concentrate	%		26.5	PRA
Moly Recovery to Copper Concentrate	%		18.2	PRA
Moly Grade in Copper Concentrate	% Mo		0.27	PRA
Gold Recovery to Copper Concentrate	%		82.0	PRA
Gold Grade in Copper Concentrate	g/t		18.40	PRA
Silver Recovery to Copper Concentrate	%		72.13	PRA
Silver Grade in Copper Concentrate	g/t		113.10	PRA
Copper Concentrate Production	dmtph		35.1	HE
Copper Concentrate Production	dmtpy		278,987	HE
First Five-year Average Moly Conc Production				
Moly Recovery to Moly Concentrate	%		72.0	PRA
Moly Grade in Moly Concentrate	% Mo		54.0	PRA
Copper Recovery to Moly Concentrate	%		0.028	PRA
Copper Grade in Moly Concentrate	%		0.42	PRA
Rhenium Grade in Moly Concentrate	ppm		530	PRA
Moly Concentrate Production	dmtph		0.70	HE
Moly Concentrate Production	dmtpy		5,596	HE

1.6 Mine Plan

The Schaft Creek deposits are to be mined with large truck/shovel operations, and an ore mining rate of 65,000 tpd feeding a conventional copper concentrator. The mining is described as typical hard rock bulk mining method. Large equipment will be used and high mining rates are planned to ensure the lowest possible unit costs for mine operations. Selective mining methods will not be used. The waste and ore will require blasting and typical grade control methods using blasthole sampling and possibly blasthole Kriging will be used to determine cut-off grades and digging control limits for the mining shovels. Blast heave, the lack of loading selectivity, haul back in the trucks, and stockpile reclaim will create some ore loss (mining recovery) and dilution as the material moves from In-Situ modeled resource to ROM mill feed. Since the ROM mill feed determines the production schedule and revenue stream for the project, proper evaluation of the mining loss and dilution is required. The definition of the mining parametres used in the reserves calculations are also a NI 43-101 reporting requirement.



The 3D Block Model (3DBM) for Schaft Creek, updated for this study, is based on separate Lithological / Geostatistical domains, There are two ore zones per block with two Copper (Cu), Gold (Au), Silver (Ag), and Molybdenum (Mo) grade values for each block. As such the grade values in each block are not 'whole block diluted'.

With the planned bulk mining method, a means of determining the mining loss and dilution applicable to the Schaft Creek Resource model is needed that will reflect the ROM production from the mining operations. Mineralized zones in the 3DBM are made up of relatively large contiguous blocks of 'ore' above the cutoff grade. There are areas however where isolated blocks of ore are surrounded by waste and also isolated blocks of waste that are surrounded by ore. Higher cutoff grades will result in fewer contiguous blocks and more isolated blocks. Conversely lower cutoff grades will merge more of the indicated isolated blocks into close-by contiguous blocks.

Mining operations will use blasthole samples on 6 to 8 metre spacing to determine the cutoff boundaries for shovel dig limits. "Included" ore and waste blocks on the small blasthole sampling grid will be too small to separate from the shovel face especially after being displaced by blasting. This inclusion of isolated blasthole blocks is handled as the larger blocks in the 3D block model are averaged in to larger 3DBM.

The 3DBM uses 25m x 25m x 15m blocks for this stage of long range planning. Each block represents 25,031 tonnes which is 4 to 5 hours of digging for the shovels, and the plant feed will be approximately 2.4 blocks per day. With blocks of this magnitude, it can be assumed that isolated blocks from the larger 3DBM will be selectively mined and will not be lost or included in the ore. However bulk mining will cause dilution to the blocks, either ore into waste or waste into ore by neighboring blocks, where contact is made between ore grade material and waste.

Other mining losses are also noted in mining operations mainly due to misdirected loads, haul back in frozen truck boxes, and stockpile cleanup. These types of losses are small but need to be accounted for.

The mining resources will be calculated from the Resource model, within an economic pit limit using the applicable mining recovery and dilution parametres. The mining recovery and dilution parametres, in effect, convert the in place "pit delineated resource" to ROM resource tonnes. As stated above it is the ROM tonnes that are required for the production schedule which in turn is used to develop the project cashflows; therefore, the tonnes used in calculating the economic pit limit needs to be based on the ROM. The resources in the model are quantified as ore or waste based on a NSR cutoff.

Mining recovery and dilution parametres are required to account for the following:

- Dilution of waste into ore where blasting "throws" waste into ore at ore/waste boundaries.
- Loss of ore into waste where blasting "throws" ore into waste diluting the mix below cut off grade.
- General mining losses due to haul back from frozen or sticky material in truck boxes, misdirected loads, and repeated handling such as stock pile reclaim.

For this Preliminary Economic Assessment (PEA) an allowance has been made for a mining dilution of 5% applied at the contact between ore and waste dilution and a 10% mining loss.

Since the dilution material on the contact edge of the blocks described above is mineralized, it will have some grade value. The dilution grades are estimated by determining the grades of the envelope of waste in contact with ore blocks inside the pit delineated area. This is estimated by statistical analysis of grades in blocks below the design basis cutoff of $4.25/t. The dilution grade was estimated at 3.49 $/t NSR, 0.060 % Cu, 0.080 g/t Au, 2.150 g/t Ag and 0.004 % Mo representing the average grade of material below the incremental cut-off grade. The results determined the pit delineated resources for the project, as shown below.

Table 1.4
Summarized MII Pit Delineated Resource for Schaft Creek

PHASE	RUN OF	WASTE	ROM	DILUTED GRADES				
	MINE	TOTAL	S/R	NSR	Cu	Au	Ag	Mo
	(mT)	(mT)	(t/t)	$/t	%	g/t	g/t	%
P616	68.8	40.1	0.6	16.6	0.377	0.299	2.03	0.017
P626i	105.2	122.3	1.2	14.4	0.340	0.213	1.61	0.017
P636i	252.0	330.3	1.3	12.6	0.272	0.220	1.54	0.017
P646	15.7	17.1	1.1	15.6	0.314	0.279	2.44	0.023
P656i	82.8	111.5	1.3	14.0	0.298	0.196	1.98	0.024
P666i	194.6	571.1	2.9	13.9	0.302	0.191	1.93	0.024
Total	719.1	1,192.4	1.7	13.8	0.304	0.217	1.77	0.020

A figure depicting the ultimate pit:

1.7 Operating Costs

The operating cost estimate for the Schaft Creek Project Preliminary Economic Assessment has been developed to support a greenfield base case plant capable of processing a 65,000 MTPD copper (gold, silver and molybdenum) porphyry deposit located in the Liard Mining Division of northwestern British Columbia at the conceptual level of analysis. The operating costs have been estimated in Q2/Q3, 2007 Canadian dollars and do not include allowances for escalation. Where source information was provided in other currencies, these amounts have been converted at rates of 1 US$ = 1 $CD.

Unit rates for power costs are based on current knowledge of rates in the area, some earlier meetings with BC Hydro in British Columbia and recent estimates from other developing operations in the area. A rate of $0.050/kWh is used. Power costs are based on the unit rates for power and the electrical load analysis developed for the project.

A summary of the operating costs (based on 23,400,000 ore tonnes per year) are shown in the table below.

Table 1.5
Summary of Operating Cost – Life of Mine Average

Description	Annual Cost	Cost/Tonne Ore	Cost/Tonne Mined
Mining	$92,151,433	$3.94	$1.47
Processing	$91,484,032	$3.91	
General & Admin	$17,008,500	$0.73	
Subtotals	**$200,643,964**	**$8.58**	
Conc Handling & Transport	$62,596,253	$2.68	
Totals	**$263,240,217**	**$11.25**	

Other qualifications, assumptions, and exclusions that are relevant to the operating cost estimate are addressed in Section 23.9 of this report.

1.8 Capital Costs

The capital cost estimate for the Schaft Creek Project Preliminary Economic Assessment has been developed to support the evaluation and assessment of the engineering, procurement and construction of a greenfield base case plant capable of processing a 65,000 MTPD copper (gold, silver and molybdenum) porphyry deposit located in the Liard Mining Division of northwestern British Columbia at the conceptual level of analysis. The capital costs have been estimated in Q2/Q3, 2007 Canadian dollars and do not include allowances for escalation. Where source information was provided in other currencies, these amounts have been converted at rates of 1 US$ = 1 $CD.

While the estimate is not sufficient for final decision making, it will help to further evaluate the Project's viability with respect to capital cost by establishing parametres from which further financial analysis and future funding may be based. The capital cost estimate can be found in Table 1.6 below.

The requirement for this capital cost estimate is intended to have an accuracy of ± 35 percent. In light of recent industry activities, Copper Fox Metals has elected to add a project reserve provision of $300 million to the estimate. Working capital, sustaining capital and reclamation & closure are shown at the end of the table but are not included in the total.

Table 1.6
Breakdown of Capital Cost

	Total ($Ms)
Mine Area Facilities	31.2
Ore Storage & Handling and Crushing	53.5
Grinding and Concentrating	256.9
Tailings	45.6
Concentrate Filtration & Loadout	6.9
Buildings and Ancillary Facilities	26.9
Site Development	29.5
Direct Cost	***450.5***
Frieght	19.0
Contractor Construction	32.3
Construction Camp	30.1
EPCM Services	37.8
Testing, QA/QC, Vendors, Commissioning	9.5
Contracted Cost	***128.8***
Mining & Ancilliary Equipment	184.6
Mine Development	44.5
Spares, Rolling Stock, Initial Fills	16.9
Admin, Shop, Warehouse, Medical, Security, Safety, Camp, Communications	6.2
Transmission Line	38.5
Site Access Road	43.4
Helicopter Support Services	30.0
Owner Indirects	20.7
Owner's Cost	***385.3***
Subtotal	***964.7***
Contingency	163.7
Project Reserve Provision	300.0
Total	**1,428.4**
Working Capital (not included in total)	49.8
Sustaining Capital (not included in total)	200.6
Reclamation & Closure (not included in total)	87.0

Other qualifications, assumptions, and exclusions that are relevant to the capital cost estimate are addressed in Section 23.10 of this report.

1.9 Project Economics

A financial model was created utilizing the mine production schedule, the associated metal grades based on the geological resource estimate, metal recoveries from the Phase I test metallurgical program, capital and operating costs as set out herein. Key model assumptions and inputs can be found in Table 23.45. Modeling was then done utilizing four different metal pricing strategies as described below:

- Base Case – Conservative metal pricing;
- Case 2 – Trailng three year average metal pricing;
- Case 3 – 2 year staggered pricing strategy using 5 year forecast and base case metal pricing;
- Case 4 – 7 year staggered pricing strategy using 5 year forecast, declining price trend and base case metal pricing.

Results of this modeling exercise are shown below in Table 1.7 with the corresponding metal prices.

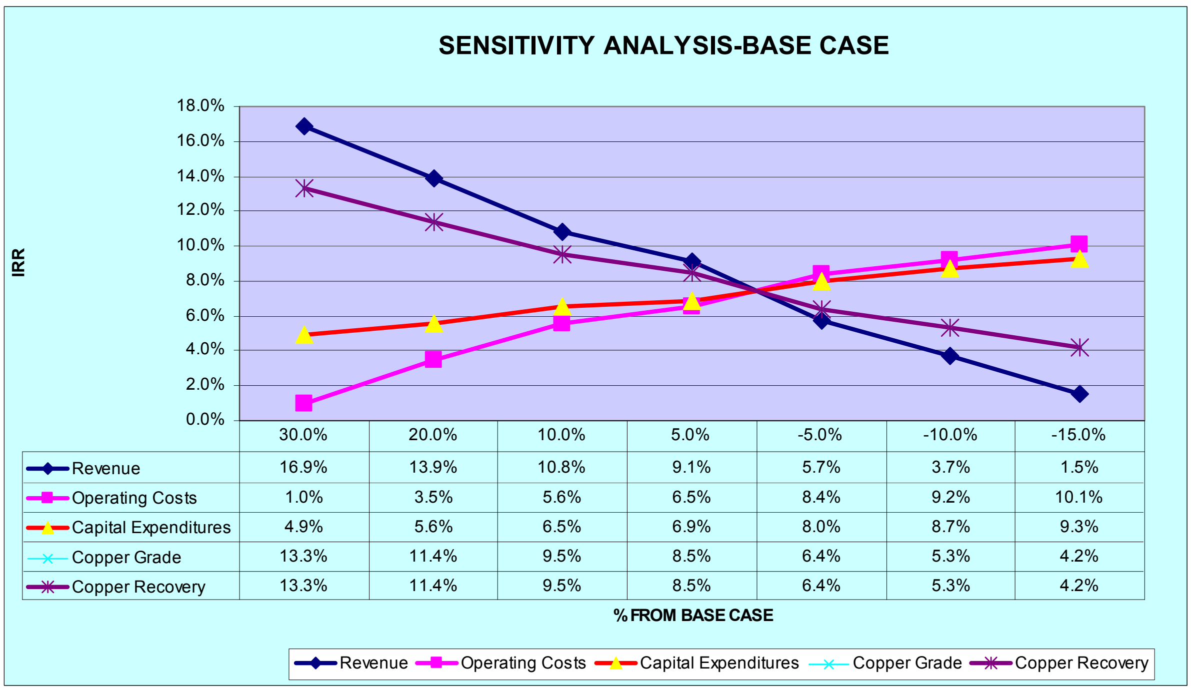
Table 1.7
Summary of Before Tax Economic Modeling Results

	IRR	NPV @ 5% ($million)	Project Profit ($million)
Base Case	**7.5%**	**$380**	**$2,047**
Cu ($/lb) = 1.50			
Mo ($/lb) = 10.00			
Au ($/oz) = 550			
Ag ($/oz) = 10.00			
Case 2 (Trailing 3 Year Average)	**32.7%**	**$5,347**	**$12,357**
Cu ($/lb) = 2.66			
Mo ($/lb) = 27.00			
Au ($/oz) = 564			
Ag ($/oz) = 10.40			
Case 3 (2 Year Staggered Pricing)	**13.9%**	**$976**	**$2,720**
Cu ($/lb) = 2.76 Yrs1&2, 1.50 Yrs 3-31			
Mo ($/lb) = 22.38 Yrs 1&2, 10.00 Yrs 3-31			
Au ($/oz) = 700 Yrs1&2, 550 Yrs 3-31			
Ag ($/oz) = 12.00 Yrs1&2, 10.00 Yrs 3-31			
Case 4 (7 Year Staggered Pricing)	**22.2%**	**$1,618**	**$3,550**
Cu ($/lb) = 2.76 Yrs1-2, 2.55 Yr3, 2.13 Yr4, 2.13 Yr5, 1.92 Yr6, 1.71 Yr7, 1.50 Yrs 8-31			
Mo ($/lb) = 22.38 Yrs 1-2, 20.32 Yr3, 18.25 Yr4, 16.19 Yr5, 14.13 Yr6, 12.06 Yr7,10.00 Yrs 8-31			
Au ($/oz) = 700 Yrs1&2, 675 Yr3, 650 Yr4, 625 Yr5, 600 Yr6, 575 r7, 550 Yrs 8-31			
Ag ($/oz) = 12.00 Yrs1&2, , 11.67 Yr3, 11.33 Yr4, 11.00 Yr5, 10.67 Yr6, 10.33 Yr7, 10.00 Yrs 8-31			

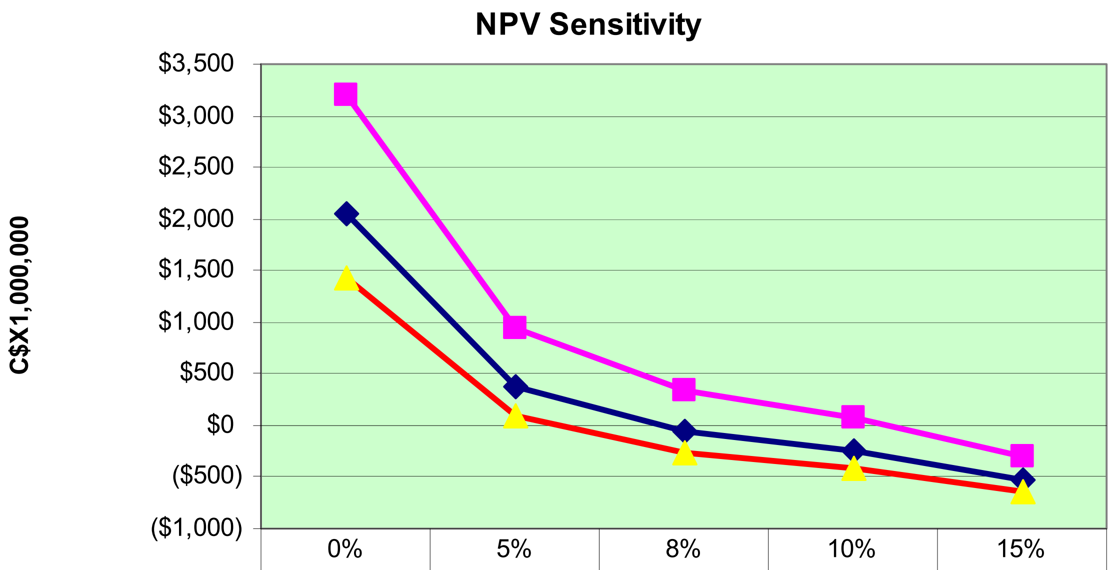
At this early stage of project development, financial results reported herein are prior to both taxation and any underlying agreements (as reported in Section 4.3 of this report). The nature and timing of expenditures as well as the corporate structure of Copper Fox Metals will have a direct bearing on the cash taxes that will be incurred on the project.

1.9.1 Base Case Project Sensitivity Analysis

Sensitivity calculations were performed on the project cash flow by applying factors ranging from -15% to +30% against initial capital, annual operating costs, annual net revenue, copper grade and copper recovery. The effects on IRR and NPV are shown graphically in the following figures. The project is moderately sensitive to changes in capital and operating costs and highly sensitive to changes in revenue (metal pricing) and metal recovery.



	30.0%	20.0%	10.0%	5.0%	-5.0%	-10.0%	-15.0%
Revenue	16.9%	13.9%	10.8%	9.1%	5.7%	3.7%	1.5%
Operating Costs	1.0%	3.5%	5.6%	6.5%	8.4%	9.2%	10.1%
Capital Expenditures	4.9%	5.6%	6.5%	6.9%	8.0%	8.7%	9.3%
Copper Grade	13.3%	11.4%	9.5%	8.5%	6.4%	5.3%	4.2%
Copper Recovery	13.3%	11.4%	9.5%	8.5%	6.4%	5.3%	4.2%



	0%	5%	8%	10%	15%
Base Case	$2,047	$380	($63)	($253)	($531)
Revenue +10%	$3,193	$940	$338	$78	($308)
Op Cost +10%	$1,434	$84	($273)	($425)	($646)

Discount Factor

More detailed information that are relevant to the project economics are addressed in Section 23.11 of this report.

1.10 Conclusions and Recommendations

It is recommended by the authors of this report that Copper Fox Metals advance their Schaft Creek project to the prefeasibility stage. There are numerous opportunities to improve the project economics through optimizations of the mine plan, processing flowsheet, method of tailings disposal, plant throughput rate, power supply, concentrate handling and treatment, operating costs, capital costs and metal pricing strategies.

Copper Fox Metals has budgeted C$16 million for the advancement of this project. This includes monies for resource development, exploration, geotechnical, metallurgical testwork, access road, product marketing, etc.

Key Results

Key results of this Preliminary Economic Assessment include:

- Measured & Indicated mineral resource: 1,393.3 million tonnes at a ≥0.20% copper cut-off grade;
- Inferred mineral resource: 186.8 million tonnes at a ≥0.25% copper cut-off grade;
- Measure, indicated and inferred pit delineated resource of 719.1 million tonnes
- LOM waste material of1,192.4 tonnes;
- LOM head grades: Cu = 0.304%, Mo = 0.020%, Au = 0.217 g/t, Ag = 1.761 g/t;

- 31 year mine life at a milling rate of 65,000 tonnes per day;
- Life of mine stripping ratio of 1.7:1;
- Preproduction capital cost of C$1,428.4 million;
- Total LOM capital cost of C$1,765.7 million;
- Operating cost of C$8.58 per tonne milled over the life of the project – includes mining, milling and G&A;
- Concentrate handling and treatment costs of C$2.68 per tonne milled over the life of the project;
- Metal recoveries: Cu = 90%, Mo = 72%, Au = 82%, Ag = 72%;
- Copper concentrate grades: Cu = 26.5%, Au = 18.4 g/t, Ag = 113.2 g/t, Mo = 0.27%;
- Moly concentrate grades: Mo = 54%, Cu = 0.42%;
- LOM copper production of 1,861.8 million tonnes;
- LOM moly production of 231.5 million pounds;
- LOM gold production of 3.9 million ounces;
- LOM silver production of 27.8 million ounces;
- Base case metal pricing: Cu = $1.50/lb, Mo = $10.00/lb, Au = $550/oz, Ag = $10.00/oz;
- Trailing three year average metal pricing: Cu = $2.66/lb, Mo = $27.00/lb, Au = $569/oz, Ag = $10.50/oz;
- Five year forecast metal pricing: Cu = $2.76/lb, Mo = $22.38/lb, Au = $700/oz, Ag = $12.00/oz;
- Base case pre-tax IRR of 7.5% with a 12 year payback;
- Base case pre-tax NPV of C$380 million at a 5% discount rate;
- Mine site production costs of C$0.57 per pound copper - net moly, gold, and silver credits;
- Trailing three year average (Case 2) pre-tax IRR of 32.7% with a 3 year payback;
- Trailing three year average (Case 2) pre-tax NPV of C$5,347 million at a 5% discount rate.

Risks

It is expected that there will be a relatively low degree of political, legal, or regulatory risk associated with the project. A project of this nature is also sensitive to several project risk factors that would be expected to potentially impact any major project of a similar size:

- Adverse weather conditions;
- Force majeure events;
- Late deliveries;
- Availability of equipment;
- Availability of materials;
- Availability of construction labour;
- Poor performance of contractors;
- Disputes with local residents;
- Disputes with NGO's;
- Escalation of costs;
- Foreign currency exchange rate fluctuations.

Copper Fox Metals has taken the proactive approach and elected to account for these risks with a Project Reserve Provision of C$300 million in the cost estimate and economic analysis for the project.

Other qualifications, assumptions, and exclusions that are relevant to the conclusions and recommendations are addressed in Sections 19 and 20 of this report.

2.0 Introduction

2.1 Purpose

Copper Fox Metals Inc. (Copper Fox) commissioned a Preliminary Economic Assessment (PEA) for its Schaft Creek project in 2006. The purpose of the study was to assist management of Copper Fox in making decisions with respect to the potential development of the Schaft Creek project. The Preliminary Economic Assessment (PEA) was prepared to define the overall scope of the Schaft Creek project, perform preliminary mine planning, report on metallurgical testwork and process design, estimate capital and operating costs and determine the economics to develop the project as an open pit mine and mill facility. The Preliminary Economic Assessment (PEA) follows two years of site work by numerous companies and consultants. This resultant technical report is a compilation of the results of the Schaft Creek study up to this point in time. Site and investigative work continues with the intention to produce a prefeasibility study and a feasibility study.

2.2 Sources of Information

Various Copper Fox personnel including Mr. Guillermo Salazar, Mr. Cam Grundstrom, Mr. Frank Agar, Mr. Michael Smith, Mr. Murray Hunter and Dr. Adrian Mann, along with Copper Fox's consultants provided key input, background information and data for the study. Special thanks go to Mr. Joe Mattson and Mr. Dave Mullen for their valuable contributions as well.

This report is the product of technical contributions from the consultants listed below. Consultants were retained by Copper Fox and Samuel Engineering. Samuel Engineering compiled all contributions provided by other contributors to this report.

- Matt Bender, P.E. – IQP, Study Manager, Samuel Engineering
- Nils von Fersen, P. Geo. – QP, Consulting Geologist, Nomad Exploration Services Inc.
- Daniel Beauchamp, P. Geo. – QP, Consulting Geologist
- John Bothwell, P. Eng. – QP, Consulting Mining Engineer
- Keith M^cCandlish, P. Geo. - IQP, Resource Estimate, Associated Geosciences Ltd.
- Jim Gray, P. Eng. – IQP, Mining, Moose Mountain Technical Services
- Shane Uren, RP Bio. – IQP, Permits and Environmental, Rescan Environmental Services Ltd.
- Mike Fabius, P. Eng. – IQP, Geotechnical, DST Consulting Engineers Inc.
- Ray Hyyppa, P.E. – QP, Study Metallurgist and Process Engineer, Hyyppa Engineering
- Vandan Suhbatar, P. Eng. – IQP, Study Metallurgist and Process Engineer, Metallurgical Consultant
- David Pow, P. Eng. – IQP, Access Road, McElhanney Consulting Services Ltd.
- Hugh Hamilton - Marketing, HM Hamilton & Associates Inc.
- Robert Simpson, President – Public Relations, PR Associates

2.3 Site Visit

Matt Bender, Director of Process, Mining & Metals, of Samuel Engineering visited the project site July 16 - 18, 2007 and again on September 13, 2007. The primary purpose of the site visits were to evaluate site layout options for the mine shop, primary crusher, conveyor, mill site, tailings pond, waste rock, topsoil material, airstrip and mancamp. In addition, observe first hand the project site and drilling activities and to talk with various site personnel.

Keith M^{c}Candlish, Vice President and General Manager, of Associated Geosciences visited the project site on two separate occasions to observe first hand the project site, observe drilling/sampling/logging practices, and to examine available drill core. In addition, available reports, cross sections, geologic interpretations and other relevant geologic data were examined at the Schaft Creek camp.

Jim Gray, Principal Mining Engineer, of Moose Mountain Technical Services visited the project site on September 13, 2007. The primary purpose of the site visit was to observe first hand the project site and the layout and design issues for the proposed open pit mine.

Ray Hyyppa, Metallurgical Engineer, of Hyyppa Engineering visited the project site during the period of October 13 – 16, 2006. The primary purpose of the site visit was to observe first hand the project site and to coordinate with the on-site geologist for the selection of drill core for metallurgical testwork.

Vandan Suhbatar, Consulting Metallurgist, visited the project site during the period of October 13 – 16, 2006. The primary purpose of the site visit was to observe first hand the project site and to coordinate with the on-site geologist for the selection of drill core for metallurgical testwork.

Shane Uren, Manager of Projects, of Rescan Environmental Services visited the project site on July 26, 2007 abd again on October 15th & 16th, 2007. The primary purpose of the site visit was to observe first hand the project site for environmental baseline plans and issues.

Mike Fabius, President, of DST Consulting Engineers visited the project site on on July 27 to 29, 2007. The primary purpose of the site visit was to observe first hand the project site and the geotechnical design issues for the proposed open pit mine and tailings dam.

3.0 Reliance on Other Experts

3.1 Disclaimer

This report is directed solely for the development, presentation of data and recommendations to allow Copper Fox Metals to make informed decisions for the development of their Schaft Creek project. With the exception for provincial securities law, any use of this report by third parties is at their sole risk, and none of the contributors to this report nor any of their respective directors, officers, or employees shall have any liability to any third party for any such use for any reason whatsoever, including negligence, and (b) the contributors to this report disclaim responsibility for any indirect or consequential loss arising from any use of this report or the information contained herein.

This report is intended to be read as a whole, and sections should not be read or relied upon out of context. This report contains the expression of the professional opinions of the contributors to this report and other consultants, based upon information available at the time of preparation. The quality of the information, conclusions and estimates contained herein is consistent with the intended level of accuracy as set out in this report, as well as the circumstances and constraints under which the report was prepared which are also set out herein.

The contributors to this report have, in the preparation of this report, relied upon certain reports, opinions and statements of certain experts. Each of the contributors to this report hereby disclaims liability for such reports, opinions and statements to the extent that they have been relied upon in the preparation of this report.

The contributors to this report have, in the preparation of this report, relied upon certain data provided to them by Copper Fox Metals and certain other parties.

None of the contributors to this report accept any responsibility or liability for the information in this report that was prepared by other contributors.

This report is a preliminary economic assessment (PEA), by which meaning the report is a preliminary assessment 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 prefeasibility study.

By the CIM Definition Standards on Mineral Resources and Mineral Reserves, a mineral reserve has to be supported by at least a prefeasibility study demonstrating economic viability of the project. It is recognized that the term “ore” cannot be used unless it is associated with a mineral reserve, however, the word “ore” is used throughout this document to refer only to mineralized material within the resource and mill feed that would be delivered to and processed in the proposed concentrator.

3.2 Reliance on other Experts

This report was prepared for Copper Fox Metals Inc. (Copper Fox) and is based in part on information not within the control of either Copper Fox or the author. It is believed that the underlying information contained in this report is reliable based on data review and verification performed by Copper Fox and the author.

The author has not independently verified the legal status or ownership of the Schaft Creek property or examined any underlying option agreements. The author has relied on Copper Fox personnel to provide various lists of mining claims, claim maps and option agreements.

The results and opinions expressed in this report are conditional upon the technical and legal information being current, accurate, and complete as of the date of this report, and the understanding that no information has been withheld that would affect the conclusions made herein.

Numerous authors have contributed to the preparation of this technical report including staff and consultants from Copper Fox Metals Inc., Associated Geosciences Ltd., Moose Mountain Technical Services, Rescan Environmental Services Ltd., DST Consulting Engineers Inc., Hyyppa Engineering, LLC, McElhanney Consulting Services Ltd., HM Hamilton & Associates Inc., PR Associates, and Samuel Engineering Inc.

Several sections of this report have been summarized, with express permission from the authors, from a previous report recently filed on SEDAR (June 22, 2007) titled,

> "Updated Resource Estimate for the Schaft Creek Deposit, Northwest British Columbia, Canada, Technical Report on a Mineral Property" prepared by Keith M[c]Candlish, P. Geo., Associated Geosciences Ltd., Calgary, Canada.

The geological model and resource estimate were completed by Riaan Herman, a sub-consultant to AGL with assistance from Susan O'Donnell, Geol.I.T.

The in-pit mineral resource and preliminary mine plan were completed by Jim Gray of Moose Mountain Technical Services.

A summary of the status of the permitting and environmental scoping process has been provided by Shane Uren with Rescan Environmental Services Ltd.

DST Consulting and Associated Geosciences provided geotechnical information and design for the mine pit and tailings dam.

Hyyppa Engineering and Vandan Suhbatar, a sub-consultant to Copper Fox, provided the metallurgical testwork reviews and recommendations, mass balance, design criteria and process design.

The access road alignment and design was completed by McElhanney.

HM Hamilton provided a preliminary copper and moly marketing report which formed the basis for concentrate handling and treatment terms complete with freight estimates and smelter terms.

PR Associates provided the guidance with the First Nations.

This report was prepared under the direction of Matt Bender, P.E., Director of Process, Mining & Metals, Samuel Engineering, Inc., as an independent "Qualified Person" as defined in the National Instrument 43-101.

4.0 Property Description and Location

4.1 Location

The Schaft Creek property is situated in northwestern British Columbia, approximately 60 km south of the village of Telegraph Creek, within the upper source regions of Schaft Creek, which drains northerly into Mess Creek and onwards into the Stikine River. Located within the Boundary Range of the Coast Mountains, the elevation of the valley at the Schaft Creek camp site is 866 m with nearby mountains exceeding 2,400 m. The property lies in proximity to the southwest corner of Mount Edziza Provincial Park, and is located 45 km due west of Highway 37 (Figure 4.1).

Referenced to Energy, Mines, and Resource Canada topographic sheet 104G, Telegraph Creek, the geographic co-ordinate at the campsite is 57º21' north latitude, 130º 59' west longitude. In terms of UTM co-ordinates, NAD 27, the location is Zone 9, 378700m E, 6358600m N. The actual deposit is situated 2 km east of the camp.

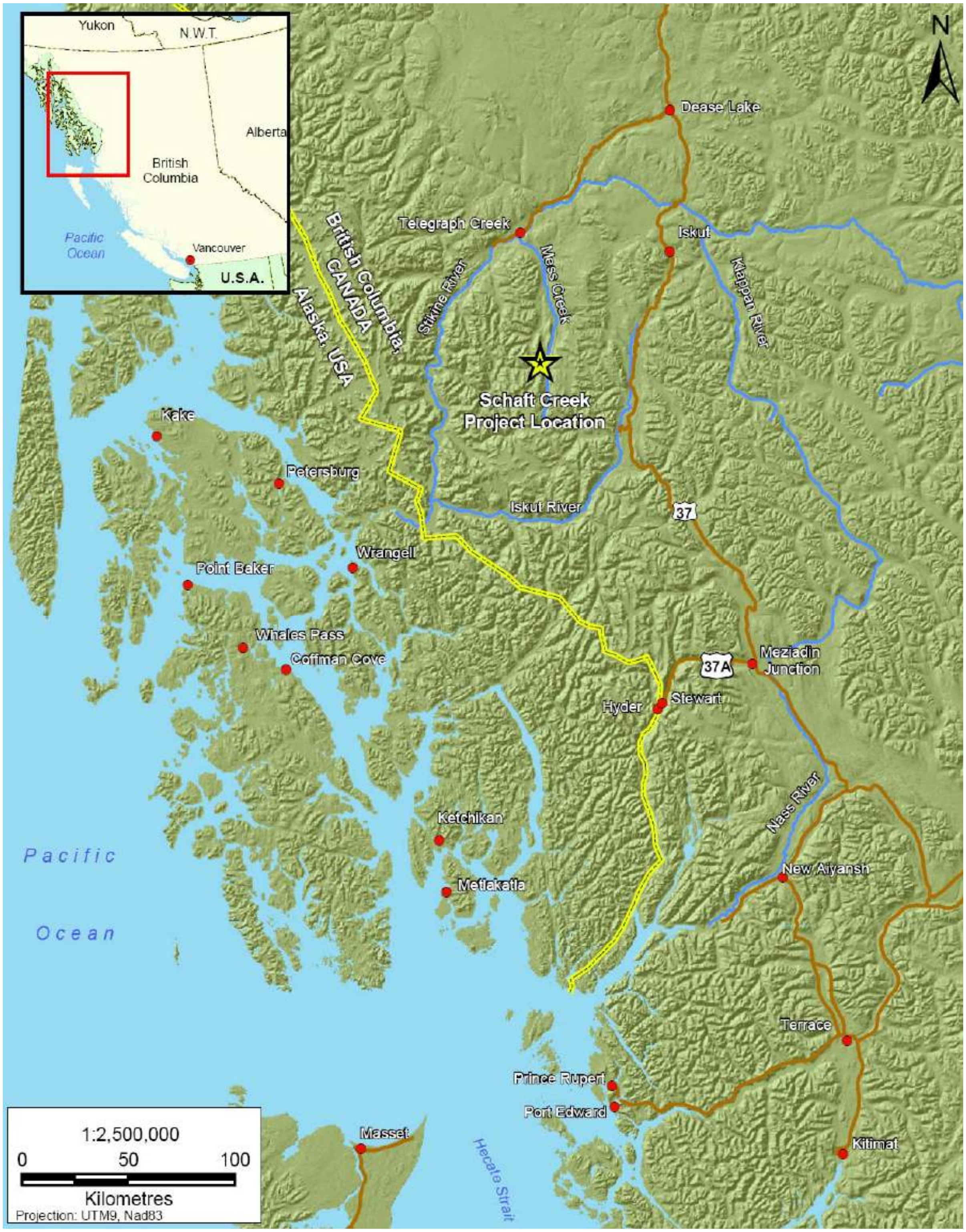


Figure 4.1 Schaft Creek Location Map

4.2 Mineral Tenure

The Schaft Creek property consists of 12-contiguous claims staked in accordance with British Columbia Energy Mines and Resources regulations. The claims encompass an area totaling approximately 20,932 ha. The deposit is situated on claims 514603 and 614637, straddling the south and north boundaries respectively, see Figure 4.2. The claims and their current status are listed in Table 4.1

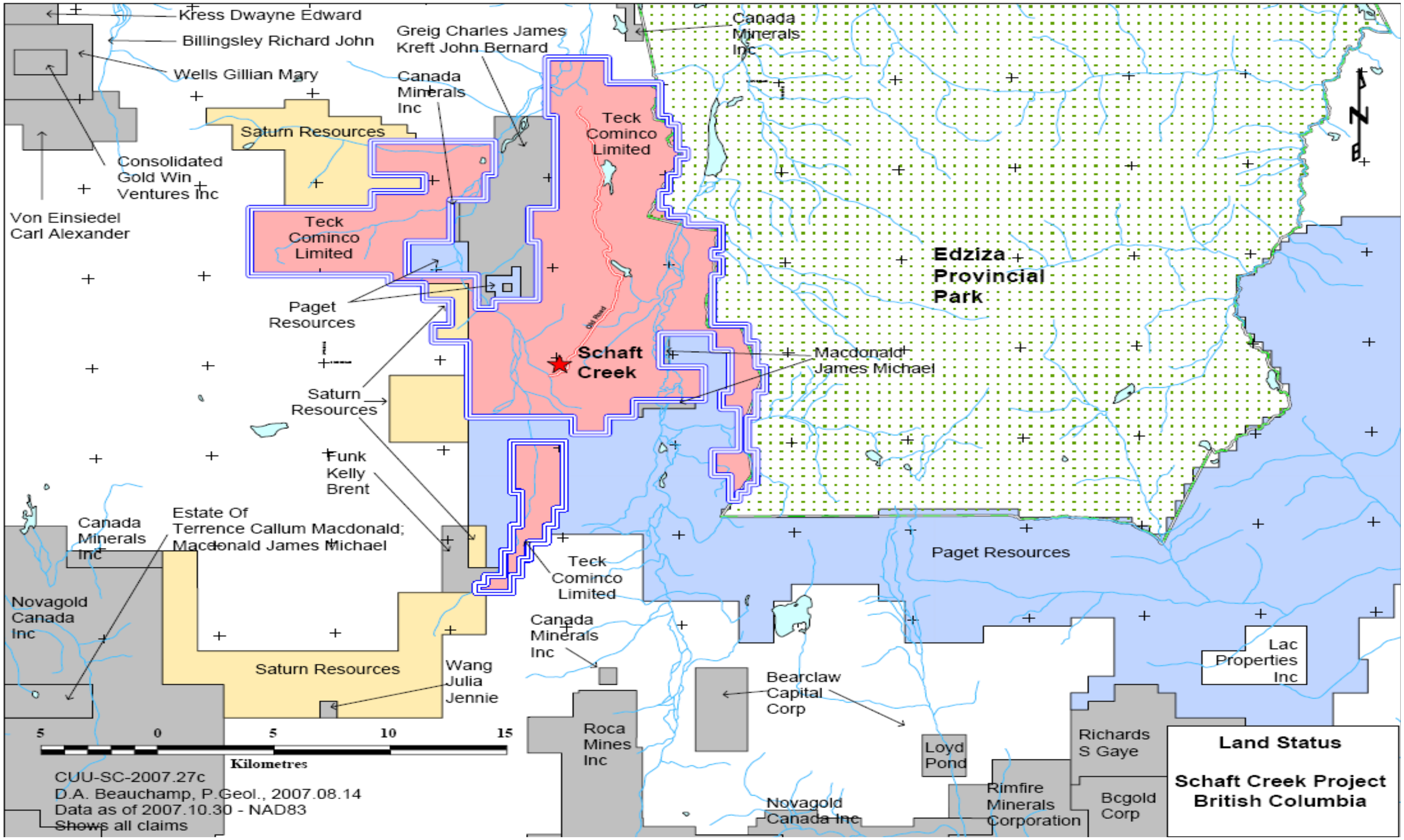


Figure 4.2 Schaft Creek Project Claims

Table 4.1
Property Claims

Title Number	Claim Owner	Percent Interest	Claim Name	Recorded Date	Renewal Date
512807	Bearclaw Capital Corp	100		17-May-05	7-Aug-08
548843	Canada Minerals Inc	100	COMINCO FRACTION IN	7-Jan-07	7-Jan-08
548896	Canada Minerals Inc	100	BIK-ROC	8-Jan-07	8-Jan-08
516787	Greig Charles James; Kreft John Bernard	50; 50		11-Jul-05	11-Nov-07
517462	Greig Charles James; Kreft John Bernard	50; 50		12-Jul-05	11-Nov-07
545064	Greig Charles James; Kreft John Bernard	50; 50	KOPPER NORTH	9-Nov-06	11-Nov-07
545065	Greig Charles James; Kreft John Bernard	50; 50	KOPPER EAST	9-Nov-06	11-Nov-07
560692	Loyd Pond	100		15-Jun-07	15-Jun-08
504843	Macdonald James Michael	100	JAMAC 6	25-Jan-05	25-Jan-08
549077	Macdonald James Michael	100	JIMMYMAC	10-Jan-07	25-Jan-08
501401	Novagold Canada Inc	100	SPC07	12-Jan-05	12-Jan-15
501905	Novagold Canada Inc	100	SPC08	12-Jan-05	12-Jan-15
509261	Novagold Canada Inc	100	ng 01	18-Mar-05	18-Mar-15
509262	Novagold Canada Inc	100	ng 02	18-Mar-05	18-Mar-15
509886	Novagold Canada Inc	100	NR 1	30-Mar-05	30-Sep-11
509889	Novagold Canada Inc	100	NR 2	30-Mar-05	30-Sep-11
516285	Novagold Canada Inc	100		7-Jul-05	1-Dec-15
516286	Novagold Canada Inc	100		7-Jul-05	1-Dec-15
516839	Novagold Canada Inc	100	NR 4	11-Jul-05	30-Sep-11
516903	Novagold Canada Inc	100	NR 06	11-Jul-05	30-Sep-11
517018	Novagold Canada Inc	100	NR 06	12-Jul-05	30-Sep-11
501238	Paget Resources Corporation	100	DA1	12-Jan-05	12-Jan-13
510372	Paget Resources Corporation	100	DA 3	8-Apr-05	8-Apr-13
514952	Paget Resources Corporation	100	DA 4	22-Jun-05	22-Jun-13
521312	Paget Resources Corporation	100	SCHAFT 1	18-Oct-05	11-Jul-11
525712	Paget Resources Corporation	100	MESS 1	17-Jan-06	17-Jan-13
525713	Paget Resources Corporation	100	MESS 2	17-Jan-06	17-Jan-13
525715	Paget Resources Corporation	100	MESS 3	17-Jan-06	17-Jan-13
526100	Paget Resources Corporation	100	SCHAFT 666	23-Jan-06	23-Jan-13
526287	Paget Resources Corporation	100	SCHAFT 667	25-Jan-06	25-Jan-13
526294	Paget Resources Corporation	100	SCHAFT 668	26-Jan-06	26-Jan-13
526295	Paget Resources Corporation	100	SCHAFT 669	26-Jan-06	26-Jan-13
526490	Paget Resources Corporation	100	SCHAFT 670	27-Jan-06	27-Jan-13
526726	Paget Resources Corporation	100	MESS 4	30-Jan-06	30-Jan-13
527394	Paget Resources Corporation	100	MESS 5	10-Feb-06	10-Feb-13
527395	Paget Resources Corporation	100	MESS 6	10-Feb-06	10-Feb-13
530660	Paget Resources Corporation	100	MESS_RUN	28-Mar-06	28-Mar-13
532722	Paget Resources Corporation	100	MESS WEST EXT.	20-Apr-06	20-Apr-13

Table 4.1
Property Claims

533216	Paget Resources Corporation	100		30-Apr-06	30-Apr-13
535835	Paget Resources Corporation	100	NM_W06-1	17-Jun-06	17-Jun-13
535836	Paget Resources Corporation	100	NM_W06-2	17-Jun-06	17-Jun-13
535986	Paget Resources Corporation	100	MESS 44	20-Jun-06	20-Jun-13
537690	Paget Resources Corporation	100	MESS S EXT 1	23-Jul-06	23-Jul-13
537691	Paget Resources Corporation	100	MESS S EXT 2	23-Jul-06	23-Jul-13
537692	Paget Resources Corporation	100	ARCTIC 1	23-Jul-06	23-Jul-13
537693	Paget Resources Corporation	100	ARCTIC 2	23-Jul-06	23-Jul-13
537724	Paget Resources Corporation	100	MESS E	24-Jul-06	24-Jul-13
537725	Paget Resources Corporation	100	ARCTIC 3	24-Jul-06	24-Jul-13
537973	Paget Resources Corporation	100	MESS S 3	27-Jul-06	27-Jul-13
537974	Paget Resources Corporation	100	MESS S 4	27-Jul-06	27-Jul-13
537976	Paget Resources Corporation	100	ARCTIC 4	27-Jul-06	27-Jul-13
537978	Paget Resources Corporation	100	FLATS 1	27-Jul-06	27-Jul-13
537979	Paget Resources Corporation	100	FLATS 2	27-Jul-06	27-Jul-13
537980	Paget Resources Corporation	100	FLATS 3	27-Jul-06	27-Jul-13
538376	Paget Resources Corporation	100	LADYTRON 1	31-Jul-06	31-Jul-13
548806	Paget Resources Corporation	100	WHITTLES	6-Jan-07	6-Jan-08
548807	Paget Resources Corporation	100	COURTNEY LOVE	6-Jan-07	6-Jan-08
548880	Paget Resources Corporation	100	TORI 1	8-Jan-07	8-Jan-08
548881	Paget Resources Corporation	100	AMOS 1	8-Jan-07	8-Jan-08
548882	Paget Resources Corporation	100	BJORK	8-Jan-07	8-Jan-08
548883	Paget Resources Corporation	100	DAFT PUNK	8-Jan-07	8-Jan-08
548884	Paget Resources Corporation	100	FISHERSPOONER	8-Jan-07	8-Jan-08
551352	Paget Resources Corporation	100	MESS 6	7-Feb-07	7-Feb-08
551358	Paget Resources Corporation	100	MESS 7	7-Feb-07	7-Feb-08
551495	Paget Resources Corporation	100	MESS 8	9-Feb-07	9-Feb-08
551510	Paget Resources Corporation	100	MESS 9	9-Feb-07	9-Feb-08
553811	Paget Resources Corporation	100	FLATS 4	7-Mar-07	7-Mar-08
553812	Paget Resources Corporation	100	FLAT 5	7-Mar-07	7-Mar-08
538540	Raven Alan Robert	100	BA-6	2-Aug-06	2-Aug-07
538542	Raven Alan Robert	100	BA-7	2-Aug-06	2-Aug-07
400294	Roca Mines Inc	100	ROC 8	13-Feb-03	30-Sep-07
400295	Roca Mines Inc	100	ROC 9	13-Feb-03	30-Sep-07
400296	Roca Mines Inc	100	ROC 10	13-Feb-03	30-Sep-07
400297	Roca Mines Inc	100	ROC 11	13-Feb-03	30-Sep-07
400298	Roca Mines Inc	100	ROC 12	13-Feb-03	30-Sep-07
400299	Roca Mines Inc	100	ROC 13	13-Feb-03	30-Sep-07
400300	Roca Mines Inc	100	ROC 14	13-Feb-03	30-Sep-07
537207	Roca Mines Inc	100	ROCATOWN	14-Jul-06	30-Sep-07
537208	Roca Mines Inc	100	ROCATOWN	14-Jul-06	30-Sep-07
540082	Roca Mines Inc	100	ROCA FLATS #1	29-Aug-06	30-Sep-07
540083	Roca Mines Inc	100	ROCA FLATS #2	29-Aug-06	30-Sep-07
551325	Salazar Guillermo	100	AREA D1	6-Feb-07	6-Feb-08
551326	Salazar Guillermo	100	AREA D2	6-Feb-07	6-Feb-08
551328	Salazar Guillermo	100	AREA D3	6-Feb-07	6-Feb-08

Table 4.1
Property Claims

560203	Saturn Minerals Inc	100	HUSKY	7-Jun-07	7-Jun-08
560208	Saturn Minerals Inc	100	HUSKY 1	7-Jun-07	7-Jun-08
560214	Saturn Minerals Inc	100	HUSKY 2	7-Jun-07	7-Jun-08
560218	Saturn Minerals Inc	100	HUSKY 3	7-Jun-07	7-Jun-08
560275	Saturn Minerals Inc	100	TOBY 1	7-Jun-07	7-Jun-08
560276	Saturn Minerals Inc	100	TOBY 2	7-Jun-07	7-Jun-08
560278	Saturn Minerals Inc	100	TOBY 3	7-Jun-07	7-Jun-08
560279	Saturn Minerals Inc	100	TOBY 4	7-Jun-07	7-Jun-08
560283	Saturn Minerals Inc	100	TOBY 5	7-Jun-07	7-Jun-08
560284	Saturn Minerals Inc	100	TOBY 6	7-Jun-07	7-Jun-08
560285	Saturn Minerals Inc	100	TOBY 7	7-Jun-07	7-Jun-08
560286	Saturn Minerals Inc	100	TOBY 8	7-Jun-07	7-Jun-08
560287	Saturn Minerals Inc	100	TOBY 9	7-Jun-07	7-Jun-08
560290	Saturn Minerals Inc	100	TOBY 10	7-Jun-07	7-Jun-08
560291	Saturn Minerals Inc	100	TOBY 11	7-Jun-07	7-Jun-08
560292	Saturn Minerals Inc	100	TOBY 12	7-Jun-07	7-Jun-08
560316	Saturn Minerals Inc	100	TOBY 13	8-Jun-07	8-Jun-08
560317	Saturn Minerals Inc	100	TOBY 14	8-Jun-07	8-Jun-08
560318	Saturn Minerals Inc	100	TOBY 15	8-Jun-07	8-Jun-08
560319	Saturn Minerals Inc	100	TOBY 16	8-Jun-07	8-Jun-08
560320	Saturn Minerals Inc	100	TOBY 17	8-Jun-07	8-Jun-08
560321	Saturn Minerals Inc	100	TOBY 18	8-Jun-07	8-Jun-08
560322	Saturn Minerals Inc	100	TOBY 19	8-Jun-07	8-Jun-08
560323	Saturn Minerals Inc	100	TOBY 20	8-Jun-07	8-Jun-08
560324	Saturn Minerals Inc	100	TOBY 21	8-Jun-07	8-Jun-08
560325	Saturn Minerals Inc	100	TOBY 22	8-Jun-07	8-Jun-08
514595	Teck Cominco Ltd	100		16-Jun-05	30-Oct-15
514596	Teck Cominco Ltd	100		16-Jun-05	30-Oct-15
514598	Teck Cominco Ltd	100		16-Jun-05	30-Oct-15
514603	Teck Cominco Ltd	100		16-Jun-05	30-Oct-15
514637	Teck Cominco Ltd	100		17-Jun-05	30-Oct-15
514721	Teck Cominco Ltd	100		17-Jun-05	30-Oct-15
514723	Teck Cominco Ltd	100		17-Jun-05	30-Oct-15
514724	Teck Cominco Ltd	100		17-Jun-05	30-Oct-15
514725	Teck Cominco Ltd	100		17-Jun-05	30-Oct-15
514728	Teck Cominco Ltd	100		17-Jun-05	30-Oct-15
515035	Teck Cominco Ltd	100		22-Jun-05	30-Oct-15
515036	Teck Cominco Ltd	100		22-Jun-05	30-Oct-15
548487	Teck Cominco Ltd	100	BLOCK B1	2-Jan-07	2-Jan-08
548488	Teck Cominco Ltd	100	BLOCK B2	2-Jan-07	2-Jan-08
548489	Teck Cominco Ltd	100	BLOCK B3	2-Jan-07	2-Jan-08
548490	Teck Cominco Ltd	100	BLOCK B4	2-Jan-07	2-Jan-08
548492	Teck Cominco Ltd	100	BLOCK C1	2-Jan-07	2-Jan-08
548493	Teck Cominco Ltd	100	BLOCK C2	2-Jan-07	2-Jan-08
548494	Teck Cominco Ltd	100	BLOCK C3	2-Jan-07	2-Jan-08
548495	Teck Cominco Ltd	100	BLOCK C4	2-Jan-07	2-Jan-08

Table 4.1 Property Claims					
548496	Teck Cominco Ltd	100	BLOCK C5	2-Jan-07	2-Jan-08
548498	Teck Cominco Ltd	100	BLOCK C6	2-Jan-07	2-Jan-08
548759	Teck Cominco Ltd	100	AREA A	5-Jan-07	5-Jan-08
548760	Teck Cominco Ltd	100	AREA C1	5-Jan-07	5-Jan-08
548761	Teck Cominco Ltd	100	AREA C2	5-Jan-07	5-Jan-08
548762	Teck Cominco Ltd	100	AREA C3	5-Jan-07	5-Jan-08
548763	Teck Cominco Ltd	100	AREA C4	5-Jan-07	5-Jan-08
548764	Teck Cominco Ltd	100	AREA B1	5-Jan-07	5-Jan-08
548766	Teck Cominco Ltd	100	AREA B2	5-Jan-07	5-Jan-08
548767	Teck Cominco Ltd	100	AREA B3	5-Jan-07	5-Jan-08
548768	Teck Cominco Ltd	100	AREA B4	5-Jan-07	5-Jan-08
548769	Teck Cominco Ltd	100	AREA B5	5-Jan-07	5-Jan-08
548770	Teck Cominco Ltd	100	AREA B6	5-Jan-07	5-Jan-08
548771	Teck Cominco Ltd	100	AREA B7	5-Jan-07	5-Jan-08
548772	Teck Cominco Ltd	100	AREA B8	5-Jan-07	5-Jan-08
534991	Wang Julia Jennie	100	GALORE GOLD	6-Jun-06	6-Jun-08
547789	Warren Christopher Ian	100		21-Dec-06	21-Dec-07
547798	Warren Christopher Ian	100		21-Dec-06	21-Dec-07

The Schaft Creek mineral claims are contained within the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP) area. The Cassiar Iskut-Stikine LRMP encompasses a total of 5.2 million hectares. The LRMP supports opportunities for mineral and energy exploration and development, including roads for resource development, in all zones outside of Protected Areas subject to standard regulatory approval processes and conditions and consistent with the management direction in the LRMP.

Existing mineral tenure rights are upheld by the Cassiar Iskut-Stikine LRMP, with the exception of two tenures within the Chukachida portion of the Upper Stikine Spatsizi Extension Protected Area. New mineral tenures can be staked and recorded on all mineral lands outside of Protected Areas according to the Mineral Tenure Act and Regulations.

The Cassiar Iskut-Stikine LRMP outlines three categories of management direction for the LRMP area:

- General Management Direction;
- Area-Specific Management;
- Protected Areas.

General Management Direction represents a baseline for resource activities on all Crown land outside of Protected Areas. The General Management Direction applies in all geographic zones, except where different objectives and strategies were developed for certain resource values or activities, outside of Protected Areas. Area-Specific Management refers to geographic resource management zones with distinct biophysical characteristics and resource issues.

The LRMP includes fifteen geographic resource management zones which are distinct with respect to biophysical characteristics and resource issues:

- Hottah-Tucho Lakes
- McBride
- Klappan
- Iskut Lakes
- Mount Edziza
- Kakkidi/Mowdada/Nuttlude Lakes
- Todagin
- Middle Iskut
- Lower Iskut
- Unuk River
- Lower Stikine-Iskut Coastal Grizzly/Salmon
- Telegraph Creek Community Watershed
- Chutine
- Tuya
- Metsantan

The Schaft Creek project is part of the Telegraph Creek Community Watershed and therefore falls under Area-Specific Management requirements stipulated in the LRMP. This zone includes the domestic water supply for the community of Telegraph Creek and is formally designated as a Community Watershed. The objective of the management approach is: “To maintain the quality and quantity of community water supply and to maintain natural stream flow regimes within the natural range of variability”. The LRMP states that mineral exploration, including road construction, maintenance and deactivation, is to be conducted according to the guidelines for community watersheds outlined in the Mineral Exploration Code.

The Schaft Creek mineral claims are located in traditional lands that Tahltan Nation have occupied and used. Copper Fox Metals Inc. has initiated discussions with Tahltan Nation Development Corporation, which represents the economic arm of the Tahltan Nation, to set out the joint understanding and intention of both parties to cooperate in carrying out the work at the Schaft Creek project.

On May 4, 2007, Copper Fox and the Tahltan Nation announced that they had completed a “Memorandum of Understanding”. The agreement defines the scope of work, program commitments, cooperation, and communication that Copper Fox will follow at Schaft Creek, and recognizes that the Tahltan Nation Development Corporation will be a “preferred contractor”.

4.3 Teck Cominco Option Agreement

Terms of the Teck Cominco option agreement to which the property is subject to are summarized as follows:

- 70% interest is directly owned by Teck Cominco and 30% interest is owned by Liard Copper Mines (a private company);
- Once Copper Fox Metals spends more than $15 million on the Schaft Creek project, Copper Fox will earn Teck Cominco's direct interest. **This threshold has been reached**;
- Teck Cominco is to provide the funding for Liard's 30% interest and hence Liard is shielded from the costs of carrying its share of the costs of the project and putting it into production. Teck Cominco's costs of the carrying Liard's 30% interest are to be recovered with interest before Liard's full 30% entitlement to cash generation sharing is to occur;
- Teck Cominco owns 78.1% of the shares of Liard and thus in addition owns 78.1% of 30% = 23.4% indirect interest;
- Copper Fox, by funding a positive feasibility study before December 31, 2011, can earn Teck Cominco's indirect 23.4% interest;
- The option agreement provides that at any time up to 120 days after receiving a copy of a qualifying positive feasibility study from Copper Fox, Teck Cominco has the option of backing into the interest that Copper Fox has earned or is entitled to earn.
- If Teck Cominco backs in before the deadline for the positive feasibility study then Copper Fox is deemed to have earned both the direct and indirect interests at that time;
- Teck Cominco can back in at any time up to 120 days after receiving a positive feasibility study to the extent of 20%, 40% or 75% of the Teck Cominco direct and indirect interests earned or deemed to have been earned by Copper Fox by funding the next costs of the project to the extent of 100%, 300% or 400% respectively of Copper Fox's qualifying costs into the project at the time of back in;
- Teck Cominco, if it elects to back in for 75% of Copper Fox's earned or deemed interest, must use its best efforts to arrange a minimum of 60% of project development costs in the form of debt and if it cannot, but elects to put the project into production, must provide Copper Fox's 25% share of such costs as a subordinated loan recoverable preferentially from Copper Fox's share of cash generation;
- Teck Cominco acquired its direct and indirect interests in the project under a three way agreement with Hecla Mining and Liard Copper Mines, by purchasing Hecla's 70% direct interest for cash and a reserved 5% net proceeds interest to be applied as we interpret it to the direct 70% interest share of cash flow after Teck Cominco has recovered its designated costs. This net proceeds interest was subsequently sold to International Royalties.

The above complex cash flow divisions were not considered for this Preliminary Economic Assessment's (PEA) requirements.

4.4 Environmental Liabilities

Copper Fox has posted a reclamation bond with the Ministry of Energy Mines and Petroleum Resources (MEMPR) to reclaim the Schaft Creek property. This includes removing all surface facilities and reclaiming areas of disturbance. The bond has been deemed sufficient by the MEMPR to reclaim the property in the event that Copper Fox abandons the Schaft Creek property.

With the exception of the above stated requirements to reclaim the Schaft Creek property, there are no known environmental liabilities associated with the property.

4.5 Permits

The Schaft Creek project will require a British Columbia Environmental Assessment Certificate as well as provincial permits, authorizations and licenses to construct and operate the Schaft Creek mine. The project may also require a federal decision on the likelihood of environmental impacts if the Canadian Environmental Assessment Act (CEAA) applies to the Schaft Creek project.

The Schaft Creek project constitutes a reviewable project pursuant to Part 3 of the Reviewable projects Regulations (British Columbia Reg. 370/02) of the British Columbia Environmental Assessment Act (BCEAA). The Schaft Creek project was launched in the BC environmental assessment process on August 14, 2006, with the issuing of Order under Section 10(1)(c) of the BCEAA.

The Canadian environmental assessment process is governed by the Canadian Environmental Assessment Act (CEAA). At this time, it is not known if the CEAA will apply to the Schaft Creek project. CEAA applies when a federal department or agency is required to make a decision on a proposed project. Federal regulatory agencies require specific project details to determine if and how the CEAA will apply.

Upon receipt of the British Columbia Environmental Assessment Certificate, permits, authorizations and licenses will be sought to construct and operate the Schaft Creek mine.

5.0 Accessibility, Climate, Local Resources, Infrastructure & Physiography.

5.1 Physiography

The Schaft Creek property is situated in northwestern British Columbia, approximately 60 km south of the village of Telegraph Creek, within the upper source regions of Schaft Creek, which drains northerly into Mess Creek and onwards into the Stikine River. The property lies in proximity to the southwest corner of Mount Edziza Provincial Park, and is located 45 km due west of Highway 37.

Physiographically, the Schaft Creek area is located within the Boundary Range of the Coast Mountains. The Schaft Creek valley area, at an elevation of 866 m, is the up-stream extension of the Telegraph Creek Lowlands. The immediate area of the Schaft Creek property is approximately 3 x 3 km in size rising rapidly eastward from the valley bottom to near-tree line elevation at the saddle in the vicinity of Snipe Lake, and towards Mess Creek to the east. The surrounding mountain to the south and west of the deposit is steep and rugged; rising to above 2,000 m from the valley floor to snow capped mountain peaks and ice fields within a few kilometres of the camp. To the east, the elevation drops from the Snipe Lake saddle to Mess Creek. To the north of the deposit is the west-facing slope of Mount LaCasse, 2,200 m above sea level. The broad, 1 km wide, north-south trending valley of Schaft Creek to the west of the camp site is a braided stream plain made up of thick, glaciofluvial and fluvial deposits. The gradient of Schaft Creek upstream of the campsite is fairly steep, causing high water velocities and strong erosional forces rapidly changing the multiple creek channels during early summer melting and run-off.

The valleys and associated tributaries are typical alpine and subalpine glaciated valleys that exhibit broad U-shaped cross sections and steep valley slopes. The elevation of the tree line is variable but alpine vegetation predominates above the 1100 m level. Below that, forests are made up of balsam fir, sitka spruce, alder, willow, devils club and cedar. Higher up the valleys, glacial moraines are bare to sparsely overgrown by sub alpine vegetation.

5.2 Accessibility

The Schaft Creek property is a remote ‘greenfield’ site with no developed roads leading into it. The property is best accessed by helicopter from Bob Quinn, a small outpost located 80 km southeast of the property on Highway 37. Bob Quinn serves as a base for several helicopter companies. The Burrage airstrip, situated 37 km east of Schaft Creek, located on Highway 37 also provides a means of access by helicopter and fixed wing, although the government does not sanction its use and there is no supporting infrastructure for aircrafts at this location. Alternatively, fixed wing aircraft can be chartered from Smithers, B.C. and flown directly to the Schaft Creek camp, utilizing an existing gravel airstrip at the camp.

5.3 Proximity of Property to a Population Center

The Schaft Creek property is approximately 375 kilometres northwest of the town of Smithers. Smithers is the closest supply centre with the capacity to service the project during construction and operation.

The Schaft Creek property is within the traditional territory of the Tahltan Nation. Three predominantly Tahltan communities are within 125 km of the property; Telegraph Creek, Dease Lake and Iskut. These three communities will provide labour during construction and operation of the mine. All three communities are accessible via Highway 37.

5.4 Climate and Length of Operating Season

The Schaft Creek project is located on the eastern edge of the Coastal Mountains in north central British Columbia. The climate of the project area is characterized by the transition between the coast and interior. The Coast Mountains, with peaks over 3,000 m in elevation lead to lifting of moist air masses moving inland from the Pacific Ocean. Annual precipitation in the Coast Mountains is often above 3,000 mm, while temperatures are mild due to the proximity of the Pacific. The climate of the interior sub-boreal plateau, on the other hand is continental with annual precipitations between 400 and 800 mm with very warm and short summers and cold winters.

Meteorological data has been observed on-site since 2005. This data along with historical data from regional Environment Canada meteorological stations can provide an overview of the climate of the project area. Mean annual air temperature within the project is near 0^{o}C but hourly temperatures can vary between +30^{o}C and -30^{o}C. Mean monthly temperatures typically remain above freezing from April to October and drop below freezing from November through March. Annual precipitation averages between 700 to 1100 mm. Due to the mountainous topography of the area, an orographic precipitation gradient exists that causes precipitation to increase by 5 to 10% per 100 m gain in elevation. Approximately 60% of the annual precipitation occurs as snow. The annual snowpack can reach a depth greater than 2 m and persist into June.

Based on observed on-site data from 2006, the dominate wind direction in the area is from the south and south-east. Wind speed is highly dependant on location. However, monthly average wind speeds were observed to vary between 1.0 and 3.0 m/s in more sheltered areas (Schaft Creek Saddle meteorological station) and up to 7.0 m/s in more exposed areas (LaCasse meteorological station).

5.5 Local Resources

The Schaft Creek project is located entirely within the Tahltan Nation territory. As such, the Tahltan Nation will be consulted individually as project planning proceeds in relation to the potential for project related impacts on their aboriginal interests and as a source for labour and other special project needs. The Tahltan Nation Development Company (TNDC) has determined that their communities are not equipped to accommodate large scale projects.

There are several small communities in the area; Telegraph Creek, Dease Lake, Iskut, Tatogga, Good Hope Lake, Bob Quinn Lake, Bell II, Meziadin Junction, and the larger port town of Stewart. The larger towns of Smithers (located 370 km to the southwest) and Terrace (located 575 km to the south) are the nearest major supply centers.

The hiring of local employees will be a highly competitive market and will be dependent on the development of other projects in the area. It is likely that the towns of Smithers, Stewart and Terrace will receive the most economic benefit from the development of Schaft Creek.

5.6 Infrastructure

5.6.1 General

Infrastructure is all but non-existent in the immediate project area. An old, overgrown and now frequently flooded bulldozer trail exists on the east side of the broad Schaft Creek valley heading north to Telegraph Creek. Drill roads have been established within a 3 x 3 km area and totals approximately 10 km of gravel and mud trails, 4 m in width.

Original construction of the camp facilities at Schaft Creek commenced circa 1965 and in 1967 a D6 Cat bulldozer was walked to the site from Telegraph Creek.

A 1,220 metre runway was constructed, for material handling and personnel transportation by fixed wing aircraft. In 1968 Hecla Mines Ltd. acquired the property, extended the runway to 1,610 metre and erected several new buildings.

During the interval from 1968 to 1981 when Hecla Mines and subsequently Teck Corporation aggressively explored the property, most of the site infrastructure was established. This included: two 9 x 46 metre Quonset style buildings, a fuel storage depot consisting of three 9.1 metre long 3.0 metre diametre tanks, two bunk houses, a kitchen and dining facility, mechanic’s shop, generator shack, core shack, log assay shack, recreation hall, sleep cabins, office buildings, and a small pre-fabricated cedar log cabin owned by a helicopter company. The airstrip system was extended to include two gravel strip runways, one oriented in a general north-south direction was established immediately west of the camp, adjacent to the eastern bank of Schaft Creek, while the second is oriented in a northeast-southwest direction and effectively bisects the camp compound.

The project was shelved by Teck Corporation in 1982 and the camp site was abandoned. Precautions were taken to ensure the survivability of the buildings against weather and rodent damage. Nevertheless, the prolonged disuse took its toll on some of the structures and with the initiation of exploration in the summer of 2005; some of the structures were assessed for demolition.

During the 2005 program a band-aid approach was implemented to re-establish the camp for human occupation, as the main focus was on a general site clean-up. During 2006, the camp was re-built to accommodate in excess of 35-personnel.

Itemized below are the clean-up and construction activities that took place during the course of the 2006 program:

- General clean-up of the camp grounds and sorting of debris and refuse into metal and wood/burnable piles;
- Demolition and burning of the old recreation building;
- Construction of two bunkhouses accommodating 32-personnel in total;
- Construction of a new kitchen and dining facility with a 42-person capacity;
- Construction of a new shower and laundry facility attached to the lavatory building;
- Establishment of a new office and first-aid facility by renovating last years core processing facility;
- Equipping the camp with two high-speed satellite internet systems;
- Relocation of an existing bunkhouse for future use as a recreation facility.

5.6.2 Power

There is no power available to site, the closest major power line being located approximately 150 km south in Meziadin Junction. It is estimated that the Schaft Creek project will require an average power draw of 92 MW with a maximum draw of 103 MW. While generation of power on-site is a consideration, it is felt that it would have serious implications to the financial viability of the project. Therefore at this point of the project development, it is assumed that power will be supplied from the BC Hydro grid. Copper Fox will construct a powerline from the Schaft Creek site to join the BC Hydro grid from Highway 37 near Bob Quinn. The powerline alignment will follow the selected access corridor.

The prospect of mineral development in the region has again made the potential for the BC Government to extend the power grid north from Meziadin Junction a reality. The BC Government (BC Hydro) has initiated an environmental assessment study into a power line through the northwest corridor that would have the capacity to service the Schaft Creek project and other mineral projects in the area. Furthermore, Copper Fox has contracted BC Hydro to conduct a power supply study for the Schaft Creek project which also takes into account the other developing projects in the region. The power study is expected to be completed by January 2008.

As is the case for major remote mining operations, stand-by diesel generators will be maintained for times when the grid power is temporarily unavailable.

5.6.3 Water

Ample water supply is available from surface and subsurface sources. Potable water will be supplied from wells located on the property. Process water for the mill will be supplied from pit dewatering wells and reclaimed from the tailings pond.

6.0 History

Schaft Creek was the subject of intense and extensive exploration since mineralization was first discovered on the property in 1957. The culmination of this exploration lead Teck Corporation to commission a prefeasibility study which included condemnation drilling in the early 1980's. Prevailing economic conditions for the next 20 years prevented the deposit from advancing. Realizing its potential, Mr. G. Salazar, acquired the right to secure a significant ownership of the property in 2002 and subsequently incorporated it into the holdings of Copper Fox Metals Inc. in 2005. Copper Fox Metals Inc. then obtained the necessary funding to undertake the 2005 program.

The history of the property is summarized below:

- 1957, discoveries nearby spurred exploration northward into the Schaft Creek-Mess Creek areas, leading to the discovery of mineralization at Schaft Creek.
- Area staked in 1957 for the BIK Syndicate; subsequently completed 3,000 ft (914.4 m) of hand trenching.
- 1956, mapping, IP survey and 3-holes were drilled by Silver Standard Mines Ltd., totaling 2, 063 ft (629 m).
- 1966, Liard Copper Mines Ltd. was formed to consolidate area land holdings.
- 1966, Asarco options the property; a 4,000 ft (1,219.2 m) airstrip was constructed, a camp was built and 24 holes were drilled, totaling 11,000 ft (3,352.8 m).
- 1967, in mid-spring of the year, a D6 Cat walked from Telegraph Creek. A second 4,000 ft (1,219.2 m) airstrip was built and construction of the camp continued. Asarco initially drills 2 holes and continues to complete 22 additional holes for a program total of 24 holes, amounting to 11,000 ft (3,352.8 m). Paramount Mining drills 1 hole.
- 1968, Asarco drops option and Hecla Mining acquires the property. The airstrip was extended to 5,280 ft (1,609.3 m).
- 1968, Hecla drills 9 holes, totaling 13,095 ft (3,991.4 m) 3 of the holes were drilled in the Paramount Zone.
- 1969, Hecla drills 9 holes, totaling 15,501 ft (4,724.7 m).
- 1970, Hecla drills 26 holes, totaling 32,575 ft (9,928.9 m). 5 of the holes were drilled in the Paramount Zone.
- 1971, Hecla drills 25 holes, totaling 22,053 ft (6,721.8 m). 3 of the holes were drilled in the Paramount Zone.
- Total Hecla footage; 83,224 ft (25,366.7 m) of which 8,610 (2,624.3) m were drilled on the Paramount Property and 74,614 were drilled on the Schaft Creek Property.
- 1972-1977, Hecla drilled 35 holes, totaling 38,386 ft (11,700.1 m).
- 1977, 104 holes drilled on the properties held by Hecla, totaling 113,000 ft (34,442.4 m). A reserve of 505 Mt with 0.38% Cu and 0.039% MoS_2 delineated.
- Between 1978 and 1979, Hecla Mining forfeits option and Teck Corp. acquires the property.

- 1980, Teck Corp. drilled 47,615 ft (14,513.1 m) in 45 holes, between mid-May to mid-November. The drill sites were prepared with a D6 Caterpillar bulldozer. Assaying of core on 10 ft (3.05 m) sample intervals, by Afton Mines Ltd. in Kamloops.
- 1981, between June and September, Teck Corp. drilled 33,315 ft (10,154.4 m) in 73 holes, and 3,503 ft (1,067.7 m) of condemnation drilling for a tailings pond and mill sites.
- Resource expanded to a global estimate of 1 Gt with 0.30% Cu and 0.034% MoS_2.
- Total property drilling is 197,500 ft (60,198 m), in 230 holes.

7.0 Geological Setting

7.1 Regional Geological Overview

The Schaft Creek copper porphyry (Cu±Mo, Au, Ag) deposit is one of a number of porphyry deposits of similar age and affinity distributed throughout the Canadian Cordillera. The Canadian Cordillera is comprised of a number of disparate tectonic terrains that have been accreted to the western margin of North America. These terrains are organized into a number of super terrains based upon a common assemblage prior to accretion to the craton. Five super terrains exist in the Canadian Cordillera, the most important of which with respect to porphyry copper formation is the Intermontane belt.

The Intermontane belt includes three terrains which are known to host significant porphyry copper mineralization. East to west, these are the Quesnellia, Cache Creek, and Stikina terrains. These terrains were amalgamated prior to accretion to ancestral North America, an event which is believed to have occurred sometime during the mid to late Jurassic. The majority of porphyry mineralization in these terrains occurred prior to the major accretionary event, and many of these pre-accretionary deposits are associated with island arc settings.

The Schaft Creek deposit is located in the Stikina terrain, which is the westernmost and most aerially extensive terrain of the Intermontane belt. A large number of porphyry copper deposits occur in this terrain, particularly in the north-central portion. The Stikina terrain is composed of Devonian to Jurassic arc-related volcanic and sedimentary rocks with coeval plutons. The Stikina terrain is the largest of the allochthonous terrains and bears a unique pre-Jurassic geological history, paleontological, and paleomagnetic signature, all indicating an origin spatially separated from the paleomargin of North America. The terrain was amalgamated with the Cache Creek, Quesnellia, and Slide Mountain terrains at some time prior to final accretion with the North American craton. The terrain is made up of a number of assemblages, two of the most significant of which are the Stikine group of Devonian to Permian age, and the Stuhini group of Triassic age.

Besides the Schaft Creek deposit, other significant deposits within the Stikina terrain include the Red-Chris, Galore Creek, Kerr, Kemess, and Huckleberry deposits. The Kemess deposit is calc-alkaline in affinity and has been dated at ~202 Ma. Published dates for Red-Chris, Kerr, Galore Creek, and Schaft Creek are ~210 Ma, ~197 Ma, ~210 Ma, and ~220 Ma respectively, although new geochronological data with respect to the Schaft Creek deposit is currently in preparation. This close clustering both spatially and temporally indicates very favorable local conditions for porphyry copper formation at this time prior to the accretion of Stikina to western North America.

The Schaft Creek deposit is hosted within the intermediate rocks of the Stuhini group. This group is comprised of a package of volcanic and sedimentary rocks that becomes dominated by sedimentary rocks eastwards, a trend which is consistent with the presence of a westerly volcanic arc. The Mess Lake facies hosts the Schaft Creek deposit and includes the most westerly volcanic rocks of the Stuhini group, which are predominantly made up of basaltic andesitic to andesitic volcanic flows and subaerial tuffs, representing a proximal volcanic facies.

The rocks of the Mess Lake facies unconformably overlie the Stikine Assemblage limestones of Lower Permian age to the northwest, and are unconformably overlain by Lower Jurassic conglomerates both to the west of Mess Creek and at their eastern margin. To the west, the rocks of the Mess Lake facies are bounded by the Hickman batholith. To the south, they are in fault contact with Paleozoic rocks of various affinities.

The Hickman batholith is a complexly-zoned intrusive body associated with the Middle to Late Triassic Stikine plutonic suite. Historical work indicated the presence of a cross-cutting intrusive body believed to be associated with the Three Sisters plutonic suite. This was the Yehiniko intrusive; however, recent U-Pb zircon dating supports a single zoned Triassic-aged intrusive rather than two distinct intrusive bodies. It is believed that it is this body which provided the mineralizing fluids that formed the Schaft Creek deposit.

7.2 Property Geology

The Schaft Creek deposit is in part situated in the valley of Schaft Creek and in part along the western slope of Mount LaCasse. The deposit is bounded to the west by the Hickman batholith and to the east by volcanic rocks of the Mess Lake facies. The valley floor exposes the Stuhini group volcanics and conforms to the contact zone of these volcanics with the east margin of the Hickman batholith. Topography within the valley floor is very subdued and largely covered by glaciofluvial gravels. Bedrock exposures are very scarce in the lower elevations of the valley floor.

The deposit is hosted by north striking, steep, easterly dipping volcanic rocks comprised of a package of: andesitic pyroclastics ranging from tuff to breccia tuff; and aphanitic to augite-feldspar-phyric andesite. The deposit is elongated in a general north-south direction, as a result of being modified by regional stress regimes and has been structurally transformed by post formation faulting.

Narrow, discontinuous feldspar porphyry and quartz feldspar porphyry dikes, genetically related to the Hickman batholith, intrude the volcanic package, occupying structural planes of weakness. The orientation of the mineralizing structures, originally related to local stress fields, is associated with the emplacement of the batholith. Potassic alteration envelopes are associated with the dykes. Besides the genetic association of the dykes with the Hickman batholith, the batholith is also considered to be the source of the magmatic-hydrothermal fluids, which ultimately formed the mineralized breccias, veins and stockworks of the deposit.

Although the deposit is spatially related to the Hickman batholith, its exact position with respect to the batholith remains uncertain. The draping of the host volcanic rocks along the intrusion's eastern margin suggests that the deposit flanks the contact zone, but is related to one or more apophyses stemming from the main body of the Hickman batholith. This relationship is further complicated by structural modification associated with accretionary tectonics.

Three geologically distinct, but not necessarily disparate, spatially separate zones, representing distinct porphyry environments constitute the Schaft Creek deposit. The largest of these zones is the Main zone, which is characterized by syn-intrusive poly-phase quartz-carbonate veins and stockworks, and mineralized with variable amounts of chalcopyrite, bornite and molybdenite and late fracture molybdenite.

The second largest zone is the Paramount zone, which is characterized by; primary sulphide mineralization associated with an intrusive breccia phase, containing chalcopyrite, bornite and molybdenite; quartz-carbonate stockworks; and late fracture molybdenite mineralization. This zone represents a deeper cupola environment.

The smallest of the zones is the West Breccia zone. It is characterized by quartz tourmaline veining, pyrite and a hydrothermal breccia. This zone represents a low temperature epizonal environment. Feldspar porphyry, in part, propagated a fault and breccia network that allowed the introduction of hydrothermal fluids and a volatile phase. Eventually this process created a breccia-pipe.

7.2.1 Lithology

In term of the deposit as whole, 17 rock types were observed and recorded. Table 7.1 lists those rock types and the percentage of mass they represent within the Schaft Creek deposit. The most common rock type observed is andesitic lapilli tuff, representing 16% of the total rock types. The majority of the rock types are characteristic of a volcano-sedimentary basin, representing 67% of the total rocks observed. Felsic intrusive rocks genetically related to the Hickman batholith constitute 13% of the total. The degree and intensity of faulting and to a lesser extent shearing, represented by 5.0% of the total rock types, reflects a tectonic setting that structurally modified the basin and the deposit's gross geometry.

Mineralization related lithologies for the West Breccia and Main zones amount to 12% and 10% respectively, while the host volcanics for these zones amount to 68% and 77% respectively. These observations are in sharp contrast to the Paramount zone where mineralization related lithologies represent 61% of the total and host volcanics represents 16% of the total. These differences between the West Breccia and Main zones with the Paramount zone demonstrates the distal or high level environment of the former zones in comparison to the proximal or lower level intrusive related environment of the Paramount zone. Despite the Main zone hosting 10% of the mineralization related lithologies, it contains the largest mineral resource of the deposit, reflecting a uniform distribution of metals within a large volume of genetically unrelated rock.

Table 7.1 Legend and Table of Lithologies, In Order of Decreasing Abundance		
Lithology	**Abundance**	**Rock Code**
Andesitic lapilli tuff	16.0%	ANLP
Feldspar-augite phyric andesite	16.0%	ANAP
Feldspar phyric andesite	12.0%	ANPF
Andesitic breccia	8.0%	ANBX
Andesite	8.0%	ANDS
Andesitic tuff	6.0%	ANTF
Granodiorite	5.0%	GRDR
Fault zone	5.0%	SHER/FAUL
Feldspar porphyry	5.0%	PPPL
Volcanic breccia	4.0%	BRVL
Augite-phyric andesite	3.0%	ANAU
Basic dyke	3.0%	D/BS
Hydrothermal vein breccia	2.0%	HVBX
Alteration zone	2.0%	ANXX
Other	2.0%	OTHR
Feldspar quartz-porphyry	2.0%	PPFQ
Intrusive breccia	1.0%	BRIG

These percentages vary considerably on a zone basis.

Interestingly, the degree of post formational faulting is reflected by the amount of observed fault zones; 6% and 4% for the West Breccia zone and Main zone respectively and 9% for the Paramount zone.

The most abundant rock types at Schaft Creek are andesitic volcanics, which constitute 73% of the 2006 core.

7.2.2 Alteration

Alteration is the process of partial or total replacement of primary igneous silicate minerals by secondary, often hydrous, lower temperature minerals, i.e. chlorite, sericite, carbonate, epidote, hematite, magnetite, quartz, tourmaline and biotite. The term ‘pervasive’ is commonly used to describe core that exhibits significant alteration effects over a considerable amount of intervals. The term “alteration” can also describe millimetre to centimetre halos associated with veins, stockworks, crackle breccia and dykes.

Various alteration types occur at Schaft Creek, including potassic alteration, phyllic alteration, propylitic alteration, epidote alteration, silicification, hematite alteration, and supergene alteration.

7.2.2.1 *Potassic Alteration*

Potassic alteration is a hydrothermal alteration characterized by the presence of potassium feldspar, minor sericite and to lesser extent biotite. The outstanding visual feature of this alteration is its pink to orange colour. It forms pervasive zones as well as millimetre to decimetre halos associated with quartz-carbonate veins and feldspar porphyry. Commonly, disseminated chalcopyrite occurs with the presence of potassic alteration. This alteration is usually the earliest.

In plan view, the distribution of potassic alteration at Schaft Creek is atypical of a “normal” porphyry system in that it occurs as three distinct linear zones 100 to 300 m in width and 1,000 to 1,200 m in length. This suggests that hydrothermal solutions and associated feldspar porphyry were channeled in a complex system of conduits controlled by north-south structures.

7.2.2.2 *Phyllic Alteration*

Phyllic alteration is a hydrothermal alteration, characterized by the assemblage quartz-sericite-pyrite. It occurs as a late overprinting, imparting a yellowish tinge to the rock. It is much more pervasive in its distribution but appears to have been controlled by the same ‘plumbing’ system as the potassic alteration. In plan view, it forms a linear, continuous zone, 200 - 300 m in width, stemming from the Paramount zone in a general south direction. In the vicinity of the Main zone it curves northeastward forming a “U” shape. Normally the phyllic zone is the next outward zone or layer in a “conventional” porphyry system.

7.2.2.3 *Propylitic Alteration*

Propylitic alteration is a low temperature, low pressure event, characterized by the assemblage of chlorite-epidote-carbonate and delineates the outer margins of a porphyry system. At Schaft Creek it forms an extensive zone hundreds of miles in width, loosely conforming, but extending well beyond the zones of potassic and phyllic alteration.

7.2.2.4 *Epidote Alteration*

Epidote alteration is locally abundant in the outer fringes of the West Breccia zone. It may overlap with the deposit scale propylitic zone.

7.2.2.5 *Silicification*

Silicification occurs as decimetre to decametre sections of quartz flooding and stockworks. Bornite and chalcopyrite mineralization in the form of disseminations and stringers are commonly associated with it. Silicification typically overprints the host rocks, imparting a hard glossy luster.

7.2.2.6 *Hematite*

Hematite alteration forms extensive zones, imparting a reddish tinge to the rocks. It is a late alteration, commonly affecting the volcanics. In the past, rocks that were recognized to be hematized were termed ‘purple volcanics’.

7.2.2.7 *Supergene Alteration*

Supergene alteration oxidized copper and iron minerals, forming malachite and limonite. Extensive areas in the vicinity of the Saddle contain fractures painted and disseminated with malachite. In drill core, open vuggy quartz veins and fractures exhibit the effects of oxidizing conditions up to 30 m depths.

7.2.3 Veining

Veining and stockworks at Schaft Creek cover an area 1,400 m long by 300 m wide and form a complex system. Various terminologies are used to refer to and describe veining. Information on veining is derived from all three zones, the Main, West Breccia, and Paramount. As a sulphide carrying geological feature, veining is most prevalent in the Main zone and less so in the two other zones. Veining at Schaft Creek has been recognized as a multiphase, complex, hydrothermal feature which was active during a long time interval and interspersed with deformation events. Considerably more work has to be done to sort out the age sequence and mineralogy of veins in the three zones.

7.2.3.1 *Liard/Main Zone*

Veining in the Liard/Main zone is ubiquitous and abundant; it is the primary sulphide carrier. The largest ore reserve and the highest grades at Schaft Creek are the result of a high concentration of mineralized veins. Seven mineralized vein types have been recognized; veins sensu-stricto, stockwork, crackle-breccia, hairline, breccia, sheeted, and stringer. Vein widths vary from less than 1/10 mm to greater than 20 cm. The most common widths are 2 to 10 mm.

Mineralogy of the veins is variable but is dominated by quartz and carbonate in varying proportions, while the crystallinity of veins is mostly fine-grained. Wider veins, 2 to >10 cm display centers with 1 to 3 mm euhedral quartz and carbonate crystals, suggesting decompression. Ribbon veins are uncommon, but do occur, indicating continued distension of vein walls while gangue and minor sulphide minerals are being deposited.

The position of sulphides within veins varies; commonly sulphides occur in the center but are also concentrated along a margin of a vein, possibly indicating topping direction during crystallization. Sulphide species are dominated by, in order of decreasing abundance, chalcopyrite, pyrite, bornite and molybdenite. Other minerals that have been observed include sphalerite, galena, native copper and rarely cuprite. Malachite is most common in the oxidizing environment, usually associated with fractures.

The relative sulphide abundance in veins varies strongly. Most commonly, total sulphides range from 1 - 10%, the remaining balance is usually quartz, carbonate and chlorite. Chalcopyrite stringers, 0.5 - 2 mm wide, are widespread and most commonly occur as sub-parallel clusters within the propylitic zone. Totally sulphide free veins are uncommon and restricted to late veins of carbonate and gypsum.

Vein density is generally in the order of 10 - 20 veins per metre; however, high densities ranging from 100 - 200 veins per metre do occur. At the other end of the spectrum, low densities ranging from 5 - 10 veins per metre are also present.

The orientation of veins is generally assumed to be random. Commonly, wider veins of 10 to 20 cm of quartz-carbonate have steep to vertical orientations relative to the core axis.

In summary, the following veins have been recognized with the Liard/Main zone and arranged from early to late:

i. Early quartz veins with molybdenite and no carbonate;
ii. Early quartz veins with high bornite;
iii. Late quartz-carbonate-veins with minor chlorite, containing chalcopyrite, bornite and trace molybdenum. These are the most common veins;
iv. Late barren carbonate veins;
v. Late carbonate-gypsum veins.

7.2.3.2 *West Breccia Zone*

Veining in hydrothermal and intrusive breccias is much less prevalent than in andesitic volcanic rocks of the West Breccia zone. The veins are mineralogically composed of varying amounts of quartz-carbonate-chlorite. These veins are usually a late phase and sulphide-poor. The dominant vein assemblage is mono-mineralic and usually carbonate, varying in widths from 1 to 3 mm and commonly vuggy. Rare quartz-molybdenite-chalcopyrite veins occur in breccia rocks, preferentially within a few miles of the contact with volcanic rocks.

7.2.3.3 *Paramount Zone*

Veining in the Paramount zone exhibits a spatial preference to granodiorite and is commonly associated with quartz flooding. Sulphide mineralized stockworks are rare. These veins often display diffuse wall boundaries and within the zone of flooding may contain millimetre to centimetre wide chalcopyrite and molybdenite stringers. Chlorite veinlets form a coalescing network resulting in a crackle breccia mineralized with molybdenite, chalcopyrite and tourmaline.

A summary of the significant features of veining within the Paramount zone is listed below:

- Have variable densities, from millimetre to metre spacing;
- Have variable vein-widths, from <1-10 mm to 50 cm;
- Dips are generally steep, but horizontal dips also exist. Scattered, 1 mm wide, parallel chalcopyrite stringers commonly have a shallow dip relative to the horizontal;
- The strikes of major veins most likely conform to regional trends, stockworks and major vein sets. They are probably controlled by local stress fields, but may have concentrated along specific lithologic horizons, contacts or bedding planes;
- The Hickman batholith was the source of hydromagmatic and hydrothermal fluids from which the veins were generated.

7.2.4 Structure

The Schaft Creek deposit is spatially and genetically associated with the east contact of the Hickman batholith. The three zones that constitute the deposit occur within a north-south trending volcano-sedimentary package that was tilted to form a steep, easterly dipping succession, which controlled ascending hydrothermal solutions. Accretionary tectonics modified the succession by longitudinal block faulting and uplift, resulting in a bowl shaped mineralized zone, with respect to the Main zone.

The West Breccia zone is fault controlled, but is thought to connect with the Paramount zone via a fault feeder channel. Similar fine-grained felsic igneous rocks occur in both zones, despite being separated by 1000 m. The Main zone mineralization is controlled by syn-intrusive overpressure fractures and faults that propagated along bedding and lithologic discontinuities and also formed regional scale longitudinal faults. The ground preparation served to accommodate the intrusion of feldspar porphyry dykes, hydrothermal veins, stockworks, vein sets and sheeted veins.

The Paramount zone is the most proximal zone to the magmatic hydrothermal system, from which the mineralized solutions emanated. The Paramount zone is characterized by intrusive breccias, granodiorite and intense quartz flooding, associated with quartz veins hosted by the granodiorite.

Some of the salient structural features associated with the deposit as a whole are outlined below:

- The deposit is situated east of and in proximity to the contact of the Hickman batholith;
- The eastern limit of the mineralization is recognized as a series of strong faults;
- In part, the known western boundary of the mineralization at present coincides with the West Breccia zone;
- The volcanic succession has an approximate north-south strike;
- Intrusive felsic dykes for a generally north-south trending network.

Fracturing and faulting are ubiquitous and generally strong, in all zones. Various structural features are discussed below along with relevant attributes.

7.2.4.1 *Fracturing*

- Generally moderate to high density, commonly centimetre, occasionally decimetre spacing, resulting in low RQD numbers;
- Several conjugate fracture directions;
- Steep and moderate dip angles relative to the core axis

7.2.4.2 *Microfaulting*

- Microfaulting is defined as thin fractures that visibly offset a vein or other litholigical features in one core sample;
- Microfaulting is common, often occurring as groups of parallel, centimetre spaced microfaults showing several en echelon offsets of a vein. Each offset is 5 mm to 1 cm, which would add up to 5 cm over 10 cm, or 1 m offset over 2 m. If the same amount of deformation is carried through a consistent off-set, it can be extrapolated to tens of metres over 100 m.

7.2.4.3 *Slickensides*

- Very common;
- Decimetre to metre spacing;
- Fairly random dips, including horizontal;
- Unknown strike;
- Striations are common, both in the vertical as well as in the horizontal component (relative to the horizontal plane);
- Commonly coated with either molybdenite or specular hematite.

7.2.4.4 *Crushed zones*

- Uncommon although exists in several holes, both in the Liard zone but especially in the Paramount zone;
- Occurs particularly in granodiorite, which is permeated by tens to >100 per miles of a random or weakly oriented, dense net of fractures, often lined with a thin clay film;
- Commonly dark gray and with a minor coating of molybdenite;
- Interpreted as a result of strong compression, with little to no lateral translatory movement;
- Resulting in rubble, 1 to 5 cm size.

7.2.4.5 *Faulting*

- Faulting is common with spacing at metre to 10 metre intervals;
- Generally the dips are steep with variable strike, assumed to be preferentially north-south;

- Fault gouges are 1 cm to 20 cm wide, with comminuted rock particles and clay;
- Strongly fractured rock portion (30 - 50 fractures per metre) are commonly logged as ‘fault zones’.

7.2.4.6 *Foliation, Shears*

- Foliation is defined as a rock unit showing a distinct fabric or foliation, which is fairly rare;
- Foliated units are 1 to 10 m wide, generally with steep dips and an unknown strike orientation;
- Foliation generally includes brecciation and an introduction of chlorite;
- Some foliated rocks exhibit strongly oriented, eye-shaped relics (2 x 10 mm) of a felsic protolith, enveloped by 1 mm wide, sub-parallel epidote-chlorite stringers. This is interpreted as an oriented, hydrothermally overprinted, barren assemblage;
- Minor, strongly foliated, feldspar porphyry, associated with several mylonite units, fault gouges and diabase dykes, indicates zones of structural weakness and strong deformation. This is associated with an epidote-chlorite-hematite breccia matrix and oriented quartz veins.

8.0 Deposit Type

By the CIM Definition Standards on Mineral Resources and Mineral Reserves, a mineral reserve has to be supported by at least a prefeasibility study demonstrating economic viability of the project. It is recognized that the term “ore” cannot be used unless it is associated with a mineral reserve, however, the word “ore” is used throughout this document to refer only to mineralized material within the resource and mill feed that would be delivered to and processed in the proposed concentrator.

8.1 Porphyry Copper Definition

Porphyry copper deposits are large, low grade, intrusion related deposits which provide the major portion of the world’s copper and molybdenum and to a minor degree, gold. The deposits are formed by a shallow magma chamber of hydrous, intermediate composition at depths of <5 km. When the magma crystallizes, fluids are released; the fluids’ movement upwards through overlying rocks results in hydrothermal alteration and deposition of sulphide minerals both as disseminations and as stockwork mineralization. There is a clear spatial and genetic association between the intrusion and the alteration zones at a regional and local scale.

The defining characteristics that distinguish porphyry deposits are:

- Large size;
- Widespread alteration;
- Structurally controlled ore minerals superimposed on pre-existing host rocks;
- Distinctive metal associations;
- Spatial, temporal, and genetic relationships to porphyritic intrusions.

The Schaft Creek deposit possesses all of these salient features and based on its economic mineral content is considered to be a porphyry copper-molybdenum-gold deposit.

8.2 Schaft Creek Porphyry System

The Schaft Creek deposit is a complex, low grade porphyry system consisting of three distinct, structurally modified zones, genetically related to the Hickman batholith. The three zones appear to be associated with a multi-phase magmatic-hydrothermal system related to either; one northerly plunging apophyses, or; several temporally discrete, smaller dykes and apophyses, stemming from a cupola(s) linked to the main body of the Hickman batholith. Dykes and sheeted veins are controlled by a regional fracture pattern, while mineralized stockworks, crackle veins and breccias are related to high local overpressure. Disseminated mineralization is associated with dykes and their accompanying alteration envelopes.

The Paramount zone, which is the most northerly of the three, represents the deeper portion of the epizone of the porphyry. Characteristics of this zone suggesting proximity to the cupola are: extensive igneous brecciation of the earlier feldspar porphyry intrusion, primary igneous zoned sulphides associated with the breccia matrix; and a higher abundance of chalcopyrite and molybdenite mineralization.

The Main zone represents the mid-level of the epizonal environment of the porphyry and is largely structurally controlled. In this zone: quartz-carbonate veins; sheeted veins; and stockworks, mineralized with chalcopyrite, bornite and molybdenite, were generated by a multi-phase overpressure event resulting from increasing hydrothermal fluid pressures, stemming from the Hickman batholith. This multi-phase event produced several generations of veining, representing different thermal regimes, overprinting alteration, fracturing and faulting within the host volcanic rocks.

Hydrothermal fluids preferentially formed veins and stockworks along shallow and steeply dipping planes of weakness within a homoclinal volcanic succession, dominated by a regional easterly dip.

Later, post formational overpressure and upward doming associated with a postulated, additional intrusive phase of the Hickman batholith structurally modified the Main zone, producing a pseudo-synclinal mineralized cross-section. Late stage mineralization associated with this event is reflected by fracture associated molybdenite. Concomitant with all the events, feldspar porphyry dykes intruded into the volcanic pile.

The West Breccia zone occurs immediately west of the Main zone and represents a high level of epizonal environment to the deposit. A poly-phase system commencing with the intrusion of feldspar porphyry along a pre-existing plane of weakness, indicated a rapidly expanding hydrothermal phase and then continued to self propagate more fractures. Eventually, both phases contributed to the formation of a breccia pipe. This breccia pipe features low temperature mineral assemblages, which are exhibited by propylylitic alteration and high pyrite content. The boron-rich nature of the volatiles in this zone is reflected by the presence of tourmaline in quartz veins. Ascending solutions affected the wall rock of this zone to varying degrees and the complexity of the system is highlighted by the overprinting of the following alterations; potassic, epidote, chloritic, silicic and hematitic. A very limited, late, high pressure gas and fluid event is evident by millimetres to decimetres wide, flow-textured pneumatolytic breccia veins and dykes.

In summary, the Schaft Creek deposit is a large, multi-phase, complex, porphyry system, genetically related to the Hickman batholith. The individual zones represent differing levels within the porphyry and correspond with increasing depth in the following order; the West Breccia zone occupies the high level, the Main zone occupies the medium level and the Paramount zone the deepest level. All of the zones have been structurally controlled, with the earliest mineralizing event strongly influenced by syn-intrusive fracturing and faulting; while, post formational faulting associated with accretionary tectonics modified the deposit considerably.

9.0 Mineralization

By the CIM Definition Standards on Mineral Resources and Mineral Reserves, a mineral reserve has to be supported by at least a prefeasibility study demonstrating economic viability of the project. It is recognized that the term “ore” cannot be used unless it is associated with a mineral reserve, however, the word “ore” is used throughout this document to refer only to mineralized material within the resource and mill feed that would be delivered to and processed in the proposed concentrator.

9.1 Mineralization

9.1.1 Associated Occurrences

Within a 20 km north-south trend, marginal to the eastern contacts of the Yehiniko and Hickman Plutons, 6-mineral showings occur in addition to the Schaft Creek deposit. They are summarized below in Table 9.1

Table 9.1
Summary of Mineral Occurrences Proximal to Schaft Creek

Minfile No.	Name	NTS Map	UTM	Minerals	Description
104G63	Late	104G/06	378850E, 6368000N	cp, bn, py	Sheared contact of the Yehiniko pluton with Stuhini volcanics
104G78	Arc, Post	104G/06	376800E, 6366000N	cp, bn, cc, py	Mineralization with purple volcanics of the Stuhini along shears within the Yehiniko pluton
104G30,31,32	Nabs 13, 21,30	104G/06	378600E, 6362500N	cp, bn	Chloritized quartz monzonite of the Yehiniko pluton at contact with the Stuhini volcanics
104G37	Hicks	104G/06	378400E, 6356200N	bn, cp, mo, py	Mineralization in the Stuhini volcanics near the east margin of the Hickman pluton

9.1.2 Styles of Mineralization

The deposit is defined by three distinct zones that appear to be semi-continuous, and are related genetically. The source of mineralizing fluids stems from one or several cupolas associated with the Hickman batholith. The Paramount zone is considered to be at the deepest level of the porphyry system, while the Main zone and the West Breccia zone represent higher levels. Two of these zones are dominated by breccia facies, namely the West Breccia zone and the Paramount zone; the third, the Main zone, is characterized by stockworks and structurally controlled vein system. Macroscopic determinations on the Copper Fox drill core defines the deposit’s sulphide mineral composition as: chalcopyrite (50%), pyrite (22.8%), bornite (14.2%) and molybdenite (13%).

Copper sulphide mineralogy is dominated by chalcopyrite and bornite, the most essential copper ore minerals, which occur in stockworks, as disseminations, and in breccias. Less commonly, chalcopyrite is observed as very thin (10-100 micron) partial coatings on ubiquitous, decimetre spaced fractures and joints.

Molybdenite is also a critical sulphide component of the ore. It occurs as disseminated blebs and stringers in stockworks and veins and is quite common in the breccia zones. Quite often it forms thin coatings on slickensides and fractures.

Rare accessory ore minerals observed are sphalerite, galena, native copper and possibly tetrahedrite.

Stockwork and vein associated mineralization form the largest component of the ore. A wide range of widths of quartz-carbonate-sulphide veins exists; from 0.1 to 1.0 mm to the most common width of 1 to 10 mm, while rare 5 to 20 cm veins exist. 0.5 to 3 mm wide chalcopyrite stringers and crackle breccia veinlets of millimetre to centimetre spacing, 0.5 to 1 mm wide, randomly oriented, sulphide filled, distensional vein system are also common sulphide bearing veins. The orientation of sulphide bearing veins is considered random, but with a preference for being steep dipping.

Medium- to fine-grained disseminated chalcopyrite, bornite, and pyrite are a common type of mineralization associated with feldspar porphyry dykes and their centimetre to decimetre wide potassic alteration halos. Disseminated sulphides also occur in the millimetre to centimetre potassic halos around veins.

Very fine disseminated sulphides of chalcopyrite and pyrite, 20 to 200 microns in size are observed in polished and thin section samples of weakly altered andesitic volcanic rocks. These sulphide grains are dispersed throughout the rock and are associated with <1 mm clusters of quartz-chlorite-sericite.

Very thin sulphide coatings on fractures are common. These coatings are commonly very thin chalcopyrite or minor molybdenite film. The estimated thickness of the coatings is on the order of 20 - 100 microns. This feature differs from molybdenite coated slickensides as it lacks striations.

Hydrothermal breccia matrix is the infilling of inter-clast space for hydrothermally deposited chlorite, carbonate, quartz, tourmaline and sulphides. This style of mineralization is an important but volumetrically smaller ore type in the West Breccia and Paramount zones. Chalcopyrite, bornite, minor molybdenite and trace pyrite are the dominant sulphides and are generally coarse-grained, ranging from 1 to 10 mm.

The deposition of sulphides at Schaft Creek is the result of a complex polyphase series of mineralizing events.

9.2 Description of Mineralized Zones

Three distinct mineralized zones are recognized at Schaft Creek: the Liard Main zone, the West Breccia zone, and the Paramount zone. All three outline an elongated shape in the north-south direction.

9.2.1 Main Zone

The Main zone has currently defined dimensions of 1,000 x 700 x 300 m depth. It has a 20º northerly plunge and is U-shaped in cross section, with the west boundary dipping 45º east and the east boundary dipping 80º west. Fracture, vein, sheeted vein and stockwork-controlled mineralization are hosted mainly by andesite flows. This zone presently hosts the largest volume of mineralized material. Chalcopyrite is the dominant sulphide, followed by bornite, pyrite and molybdenite.

The overall geometry of the zone in cross section, defined by metal distribution, is bowl or “U”-shaped. This suggests modification by late structural events. Initially, steep, easterly dipping, volcanic successions influenced the distribution of upwardly migrating hydrothermal solutions that originate from an apophyses of the Hickman batholith. Subsequent to this, the lower portion of the zone was block faulted and rotated westerly by an ascending intrusion related to a later phase of the Hickman batholith.

Higher gold values are associated with higher temperatures and bornite mineralization, whereas phyllic overprinting reflects lower temperatures, producing the pyrite-chalcopyrite association.

9.2.2 West Breccia Zone

The West Breccia zone has currently defined dimensions of 500 x 100 x >300 m depth, and lies immediately west of the Main zone. Mineralization is contained within fault controlled tourmaline and sulphide rich hydrothermal breccia and feldspar porphyry. Chalcopyrite is the dominant sulphide, followed by pyrite, bornite, and molybdenite.

The breccia of the zone exhibit multi-phase brecciation, heating and sulphide mineralization. Initially, an early phase of ghost-like brecciation of a fine-grained felsic rock deposited fine sulphide disseminations, resulting in a polygonal pattern. Subsequent to this, an igneous phase brecciated the protolith and formed a matrix of fine-grained, flow oriented lath-like feldspar rock. This was followed by a hydrothermal breccia phase that precipitated coarse sulphides, chalcopyrite and molybdenite. The last event was another hydrothermal phase that is sulphide deficient but rich in tourmaline and quartz. The margins of the zone exhibit late phase, metal deficient, intense, pervasive sericitic and carbonate alterations.

9.2.3 Paramount Zone

The Paramount zone is the most northerly of the zones and has currently defined dimensions of 700 x 200 x 500 m depth. This east-dipping zone is situated north of the Main zone. The mineralization is contained in an intrusive breccia within altered andesite and granodiorite. Chalcopyrite is the dominant sulphide, followed by molybdenite, pyrite and bornite.

The zone is characterized by a large volume of granodiorite, exhibiting a complex multi-phase intrusive, thermal and metasomatic history. The early granodiorite was brecciated by an overpressure event that intruded feldspar-quartz porphyry, which formed the matrix of the breccia. Subsequently, concentrically zoned sulphides exhibiting a core of pyrite, and successively rimmed by chalcopyrite and molybdenite were deposited by a hydrothermal fluid along with disseminated sulphides. This hydrothermal fluid metasomatically replaced potassic feldspar with plagioclase feldspar. The recrystallization of feldspar produced a fine grained, hornfelsic, mosaic rock. Late pervasive silica flooding introduced and remobilized sulphides, forming quartz veins high in pyrite, chalcopyrite and molybdenite. In comparison to the other zones, the feldspars exhibit little to no alteration and are remarkably fresh. The fine-grained mosaic texture of the matrix feldspar is interpreted to be a result of high temperature thermal metamorphism.

10.0 Exploration

10.1 Recent Exploration Summary

The 2005 diamond drill campaign conducted by Copper Fox Metals Inc. ended with the completion of 15-PQ diamond drill holes, totaling 3,160 m. During the period from August 11th to September 30th, a total of 1,089 core samples were collected and submitted for assaying and 782 core metallurgical samples were collected. The 782 core samples collected for a metallurgical bulk sample represent a total combined weight exceeding 17,690 kg.

The 2006 drill campaign ended with the completion of 42-holes, totaling 9,007 m of drilling. Of this drilling, 5,300 m included 25-PQ holes and 3,707 m included 17-HQ holes. During the period from July 12th to October 23rd, a total of 2,107 samples were submitted for assaying, and 896 samples were collected for the metallurgical composites sample. The total of the metallurgical samples collected represents a combined weight of 20,321 kg.

The two campaigns produced a total of 3,196 assay samples, 1,678 metallurgical samples and 12,167 m of core.

10.2 2006 Exploration Program

Field preparation for the 2006 program began on May 30th, while diamond drilling commenced on July 10th. The drill equipment was airlifted by a Bell 205 and a Chinook helicopter transported construction materials, Kubotas and a D5 dozer from Burrage Creek and Bob Quinn to the camp in the initial airlift. Coring commenced on July 10th, and the drill program was terminated on October 23rd, after having completed 42 holes totaling 9,007 m. The Lyncorp drill was stored on the property in the eastern Quonset hut. The two Hytech drills were mobilized off the property on October 26th.

10.2.1 Program Objectives

The 2006 drill program, designed by Associated Mining Consultants Ltd. (AMCL) and G. Salazar to twin historical drill holes, had a three-fold purpose: to confirm the integrity of the archival database derived from earlier drilling, to check the assay results in this database, and to provide a sufficient amount of higher grade material for floatation tests. Time constraints allowed the completion of 9,007 m of drilling in 42-holes, coming very close to completing the original validation program of 43-holes and exceeding the original planned mage of 5,053 m. Due to the limitations of the drill to bore large dia m shallow, angled holes, three of the planned shallow dipping holes had to be re-positioned and in fact did not twin their historical counterpart, but rather intercepted the zone at a steeper angle in the immediate intersection of interest. Two of the original PQ-holes were downgraded to HQ-holes, while two of the HQ-holes were upgraded to PQ-holes.

10.2.2 Field Protocol

The field protocol established for the 2006 program was the same as that for the 2005 program, but with the addition of infill drilling which recovered HQ-dia m core. The program's protocols are as outlined below.

10.2.2.1 *PQ Core Protocol:*

- 25-archival holes were selected to be 'twinned' in order to validate a large, archival database. The old collars were established by GPS-mapping of old drill roads, spotting casings and matching the resulting coordinate points with archival drill plans.
- New 'twin' holes were drilled within a few metres from old casings with the same azimuth, dip and length. Only a few holes had to be drilled from new locations, due to equipment limitations.
- Inclined holes were down-hole surveyed by Reflex instrument. Normally in holes less than 100 m in length, a reading was taken just beyond the bedrock interface and near the bottom of the hole. Deeper holes had additional readings taken at midpoints between bedrock and the bottom.
- All PQ and HyTech's allocation of HQ holes were cored using metric rods (1.5 m and 3.0 m lengths), while Lyncorp's allocation of HQ-holes were cored utilizing imperial length rods of 10 ft.
- All new core was photographed and the photos digitally archived. Core recovery was noted and RQD (rock quality designation) measurements were recorded as the cumulative length of intact core greater than two times its dia m (16 m within a core run). Sample numbers were assigned along 3.05 m intervals for the entire core length for assay samples, as well as for metallurgical samples (MET), using the same fixed 3.05 m intervals. MET samples; however, were taken only along AMCL's pre-defined ore intervals, utilizing the old database.
- The core was sawed twice: the whole core was cut in half, and then one of the halved sections was halved once more, resulting in one half and two quarter sections of core.
- The core was logged before sampling in metric units, recorded first in tabular form, employing historical lithological codes and nomenclature with strict adherence to 3.05 m sample runs, and secondly in descriptive format, respecting lithologic breaks.
- The core was sampled using; a) the ¼ sections for assay samples, b) the ½ sections for metallurgical samples. Both assay and metallurgical samples were placed in separate, numbered plastic pails with security lids.
- ¼ of the core is stored on site as reference material in the original, labeled core trays.
- All core data was entered into Excel spreadsheets by field geologists. Assay samples were shipped to IPL Lab in Vancouver, British Columbia, using bonded trucking firm, locked containers, observing all security precautions to maintain a continuous, intact 'chain of custody'.
- MET samples were shipped to Process Research Associates Lab in Vancouver, British Columbia, adhering to the same chain of custody as for the assay samples.

10.2.2.2 *HQ Core Protocol*

- The treatment of HQ-size core was similar to PQ-size core with the exception that no metallurgical samples were obtained from it. Therefore, this core only required one cut that produced two halves. One half was sent for assay analysis, while the other half was retained for archiving.

10.2.3 Overburden Sampling

The eastern portions of the Main zone and the Paramount zone are covered by overburden that has partially incorporated locally derived talus material. The bedrock exposures of this material along the west slope of Mount LaCasse exhibit malachite stained fractures. During the coring process, metre lengths of intact overburden with fragments of talus were recovered by the drilling. To determine to what extent this material is mineralized, the overburden from the 2005 and 2006 drilling was sampled and analyzed for Cu, Mo, Au, and Ag. Overburden includes glacial till, glaciofluvial material, locally derived purple clay and local bedrock talus.

Overburden material was collected in fixed 10 ft (3.05 m) intervals. The amount of overburden material retrieved varies greatly for each drill hole. The best recovery of 50 - 90% is experienced with purple, clay-rich, consolidated, local, volcanic talus material intermixed with minor foreign boulders.

Coarse, boulder glacial till is comprised of centimetre to decimetre foreign material and a small proportion of fine-grained sandy fractions. This till had a poor recovery, ranging from 10 - 30%.

Fine-grained, sandy and clay-rich glacial material had the poorest recovery of 1 - 20%, as most of the fine-grained material was washed away during the drilling process.

Consequently, with recovery rates of overburden spanning a large range, the assay results of low recovered portions is not representative. Assay results reflect only the chemical characteristics of the small, coarser fraction which was preserved; however, the exception to this is the purple clay-rich talus.

In a few cases, the overburden sample immediately above the bedrock contains a variable percentage of broken in-situ bedrock, mineralized with traces of sulphides or malachite and intermixed with overburden material.

10.2.3.1 *Overburden Sample Results*

Overburden material, although representing a very heterogeneous sample population is not entirely barren and not entirely below the detection limits for the metals tested. Mineralized broken bedrock understandably exhibits anomalous values for all the metals. Approximately ¼ to ½ of all samples are foreign material, transported pebbles and boulders. These exhibit Cu values of >1,000 ppm and ¼ of the samples returned values of >0.1 g/t Au. Samples exhibiting anomalous molybdenum and silver values were the fewest.

Of these, half a dozen samples had 100 ppm Mo, and half a dozen had >1.0 g/t Ag, with three samples between 3.0 and 12.0 g/t Ag.

Further work is recommended to determine the possible recovery of these metals from an economic perspective, and whether or not they are present as recoverable sulphides or tied-in with silicates.

10.3 Proposed Exploration Program

The 2007 exploration program started up in May 2007. Table 10.1outlines the proposed drill holes for the program. Depending on weather and permitting approvals, the following is a list of proposed work to take place throughout the program:

- Supplies, materials and equipment for the 2007 exploration program will be mobilized during the season from an expediting camp located on the Burrage Creek airstrip;
- All drilling for this program is being proposed to be helicopter supported;
- There will be thirty-four (34) geotechnical holes drilled at the potential dam sites for the three (3) tailings management facilities (TMP) and waste dump;
- There will be sixteen (16) exploration holes, located in the West Liard/Main zone and along the ridge;
- The existing camp will continue to be utilized with the addition of eight (8) new tent frames and a fully equipped tool crib in 2007. The camp is being expanded from a 30 to 60 person infrastructure;
- There is a proposal to expand the core racks for the additional 20,000 m of core;
- As part of this expansion a new septic system and electrical system will be installed, and two (2) - 20,000 l Enviro tanks will be brought in 2007;
- Additional equipment on-site will include a helicopter, two Kubota mules, a 315 Cat excavator and 50 kW generator.

Table 10.1
Proposed 2007 Drill Holes

Zone	Access	Zone	Depth (m)	Type of Work	Core	Easting NAD 83	Northing NAD 83
Zone C	Helicopter	W	300	Geotechnical	HQ	367890	6366643
Zone C	Helicopter	W	300	Geotechnical	HQ	367885	6366493
Zone C	Helicopter	W	300	Geotechnical	HQ	367881	6366343
Zone C	Helicopter	W	300	Geotechnical	HQ	367874	6366143
Zone C	Helicopter	W	300	Geotechnical	HQ	367869	6365994
Zone C	Helicopter	E	300	Geotechnical	HQ	374708	6369152
Zone C	Helicopter	E	300	Geotechnical	HQ	374905	6368926
Zone C	Helicopter	E	300	Geotechnical	HQ	375036	6368775
Zone C	Helicopter	E	300	Geotechnical	HQ	375200	6368586
Zone C	Helicopter	E	300	Geotechnical	HQ	375352	6368337
Zone A	Helicopter	N	300	Geotechnical	HQ	381794	6375314

Table 10.1 Proposed 2007 Drill Holes							
Zone	**Access**	**Zone**	**Depth (m)**	**Type of Work**	**Core**	**Easting NAD 83**	**Northing NAD 83**
Zone A	Helicopter	N	300	Geotechnical	HQ	381950	6375118
Zone A	Helicopter	N	300	Geotechnical	HQ	382137	6374883
Zone A	Helicopter	N	300	Geotechnical	HQ	382292	6374688
Zone A	Helicopter	N	300	Geotechnical	HQ	382448	6374492
Zone A	Helicopter	S	300	Geotechnical	HQ	383020	6367229
Zone A	Helicopter	S	300	Geotechnical	HQ	382770	6367229
Zone A	Helicopter	S	300	Geotechnical	HQ	382520	6367229
Zone A	Helicopter	S	300	Geotechnical	HQ	382320	6367229
Zone A	Helicopter	S	300	Geotechnical	HQ	382020	6367229
Zone A	Helicopter	W	300	Geotechnical	HQ	380417	6373342
Zone A	Helicopter	W	300	Geotechnical	HQ	380417	6373192
Zone A	Helicopter	W	300	Geotechnical	HQ	380417	6373042
Zone B	Helicopter		300	Geotechnical	HQ	379283	6355126
Zone B	Helicopter		300	Geotechnical	HQ	378983	6355126
Zone B	Helicopter		300	Geotechnical	HQ	378683	6355126
Zone B	Helicopter		300	Geotechnical	HQ	378383	6355126
Zone B	Helicopter		300	Geotechnical	HQ	378083	6355126
Connector	Helicopter		1,000	Exploration	NQ	380654	6367030
Connector	Helicopter		1,000	Exploration	NQ	380654	6366030
Connector	Helicopter		1,000	Exploration	NQ	380654	6365030
Connector	Helicopter		1,000	Exploration	NQ	380654	6364030
Connector	Helicopter		1,000	Exploration	NQ	380654	6363030
Connector	Helicopter		1,000	Exploration	NQ	380654	6362030
Waste Dump	Helicopter		300	Geotechnical	HQ	377965	6358641
Dirth	Helicopter		300	Exploration	HQ	379004	6360680
Dirth	Helicopter		300	Exploration	HQ	378889	6360372
Dirth	Helicopter		300	Exploration	HQ	378465	6360795
Dirth	Helicopter		300	Exploration	HQ	378350	6360487
Dirth	Helicopter		300	Exploration	HQ	377926	6360910
Dirth	Helicopter		300	Exploration	HQ	377811	6360602
Dirth	Helicopter		300	Exploration	HQ	377387	6361025
Dirth	Helicopter		300	Exploration	HQ	377272	6360717
Dirth	Helicopter		300	Exploration	HQ	376848	6361140
Dirth	Helicopter		300	Exploration	HQ	376733	6360832
Waste Dump	Helicopter		300	Geotechnical	HQ	377474	6359049
Waste Dump	Helicopter		300	Geotechnical	HQ	376983	6359457
Waste Dump	Helicopter		300	Geotechnical	HQ	377613	6358264

Table 10.1 Proposed 2007 Drill Holes							
Zone	**Access**	**Zone**	**Depth (m)**	**Type of Work**	**Core**	**Easting NAD 83**	**Northing NAD 83**
Waste Dump	Helicopter		300	Geotechnical	HQ	377122	6358672
Waste Dump	Helicopter		300	Geotechnical	HQ	376631	6359080
			19200	**34 - Geotechnical**		**16 - Exploration**	

10.3.1 2007 Geophysical Program

At the request of Copper Fox Metals Inc., Associated Geosciences Ltd. (AGL) has proposed an outline for conducting geophysical surveys to map the potential porphyry copper resource at the Schaft Creek Project located north of Stewart, British Columbia.

There are two main objectives for the geophysical surveys:

- To map mineralization to depths of 300 to 800 m in a region to the north of an existing camp;
- To determine if known mineralization, occurring at relatively shallow depth, is continuous beneath an existing swamp.

A brief review of the existing litreature has revealed that the Schaft Creek Project copper-gold-molybdenum mineralized zones are related to quartz feldspar porphyry dykes and breccias. Three individual mineralized zones have been identified; the Liard, West Breccia and Paramount Zones. The three zones are distinguished one from another by sulphide mineral types and alteration mineral suites. Specifically:

- Within the Liard Zone, a pyrite halo surrounds chalcopyrite, bornite and molybdenite mineralization in altered and faulted andesite.
- Within the West Breccia Zone, pyrite is the principle sulphide mineral, with lesser quantities of chalcopyrite and molybdenum.
- Pyrite, bornite and chalcopyrite are present in equal proportions within the Paramount Zone.

Discussions with Copper Fox Metals Inc. personnel suggest that total sulphide mineralization may range between 1% and 5%.

Induced polarization (IP), electromagnetic and magnetic methods are expected be sensitive in varying degrees to the geological conditions apparent at the Schaft Creek Project.

To map structure and to determine if known mineralization is continuous beneath an existing swamp IP, magnetometre and very low frequency electromagnetic (VLF-EM) surveys are recommended. As access may be problematic, a pole-dipole IP configuration may be preferred to minimize movement of the transmitter and transmitting power source. Magnetic susceptibility and VLF-EM response would be measured continuously as the operator walks along survey lines perpendicular to the geologic strike.

To investigate the mineralized zone immediately to the east and northeast of the swamp where surface topography increases substantially, a combination of IP, magnetics/VLF-EM and either controlled source audio magnetotellurics (CSAMT), magnetotellurics (MT) or magnetic IP (MIP) methods is recommended.

It is expected that portions of the proposed survey grid will be inaccessible due to the severe variations in surface topography. Also, as the depth of the swamp to the west and southwest is unknown, full coverage of this area may not be possible.

11.0 Drilling

11.1 Drill Program

Hytech Diamond Drilling Ltd. of Smithers, B.C., was contracted to undertake the drilling of the PQ portion of the program as well as a segment of the HQ-portion. Lyncorp International Ltd. of Calgary, Alberta was also commissioned to complete a portion of the HQ program. Helicopter air support for the program was provided by Quantum Helicopters Ltd., of Terrace B.C., while fixed wing air support was supplied by Northern Thunderbird Air, of Prince George B.C. and Tsayta Air Ltd. of Fort St. James, B.C.

A total of 42 holes were completed, 17 HQ holes and 25 PQ holes, totaling 9,007.6 m. Hytech drilled 34 holes: 25 PQ holes and 9 HQ holes; while Lyncorp drilled 8 HQ holes.

11.1.1 Core Recovery and RQD

Routinely, core recovery and rock quality designation (RQD) were determined for each 3.05 m core run. The RQD was determined by cumulatively adding intact core greater than 16 cm in length for PQ core and greater than 12 cm for HQ core, expressed as a percentage of the run. The intact lengths are derived as two different lengths; PQ-core diametre which is 8 cm and 6 cm for the HQ core.

The results of these measurements are recorded in Table 11.11.1 and separated on a zone basis.

Table 11.11.1
Estimates of RQD and Core Recovery, 2006 Drill Program

Drill Hole ID	Hole RQD	RQD Rating	Core Recovery
West Breccia Zone			
05CF234	61.7	Fair	98
05CF235	51.5	Fair	94
06CF249	22.2	Very Poor	96
06CF250	19.1	Very Poor	95
06CF252	20.2	Very Poor	97
06CF253	27.0	Poor	97
06CF254	28.3	Poor	94
06CF279	15.2	Very Poor	95
06CF280	32.7	Poor	97
06CF281	12.3	Very Poor	93
06CF282	29.9	Poor	100
06CF283	18.4	Very Poor	95
Zone average	28.2	Poor	95.9
Liard/Main Zone			
05CF236	20.6	Very Poor	97
05CF237	32.3	Poor	98
05CF238	31.4	Poor	97
Drill Hole ID	**Hole RQD**	**RQD Rating**	**Core Recovery**

Table 11.11.1
Estimates of RQD and Core Recovery, 2006 Drill Program

05CF239	43.2	Poor	98
05CF240	24.3	Very Poor	98
05CF241	51.0	Fair	98
05CF242	52.5	Fair	98
05CF243	57.9	Fair	95
05CF244	74.3	Fair	98
05CF245	17.9	Very Poor	97
05CF246	40.7	Poor	97
05CF247	74.8	Fair	99
05CF248	59.4	Fair	97
06CF251	28.8	Poor	99
06CF255	35.4	Poor	97
06CF256	35.8	Poor	97
06CF257	38.6	Poor	98
06CF258	32.2	Poor	97
06CF259	22.3	Very Poor	93
06CF260	24.5	Very Poor	99
06CF261	28.4	Poor	93
06CF262	19.9	Very Poor	96
06CF263	31.9	Poor	99
06CF264	29.9	Poor	98
06CF265	29.5	Poor	99
06CF266	18.4	Very Poor	86
06CF267	16.0	Very Poor	90
06CF268	15.4	Very Poor	85
06CF269	18.3	Very Poor	91
06CF270	29.8	Poor	100
06CF271	30.5	Poor	99
06CF272	47.2	Poor	96
06CF273	49.2	Poor	96
06CF274	49.0	Poor	100
06CF275	40.1	Poor	96
06CF276	37.8	Poor	99
06CF277	37.9	Poor	97
06CF278	25.1	Poor	99
06CF284	21.0	Very Poor	93
06CF285	29.1	Poor	97
Zone Average	35.1	Poor	96.4
Drill Hole ID	**Hole RQD**	**RQD Rating**	**Core Recovery**

Table 11.11.1 Estimates of RQD and Core Recovery, 2006 Drill Program			
Paramount Zone			
06CF286	11.0	Very Poor	79
06CF287	18.0	Very Poor	85
06CF288	14.7	Very Poor	92
06CF289	15.0	Very Poor	90
06CF290	2.5	Very Poor	47
Zone Average	12.2	Very Poor	78.6
Drill Hole ID	**Hole RQD**	**RQD Rating**	**Core Recovery**
RQD Rating			
0-25%	Very Poor	75-90%	Good
25-50%	Poor	90-100%	Excellent
50-75%	Fair		

Core recovery of holes drilled in all of the zones is excellent with the exception of two holes drilled in the Paramount zone. Core from the Paramount zone can be highly fractured, crumbly, and broken, displaying decimetre to decametre of gouge and rubble, and hence, the very low RQD value of 12.2% and lower recoveries averaging 78.6% for the zone. In comparison to the West Breccia and Main zones, which have recoveries of 95.9% and 96.4% respectively, the core recovery from the Paramount zone is substantially lower. However, like the Paramount zone, the RQD value for the West Breccia zone averaging at 28.2% and 35.1% for the Main zone falls into the poor zone.

RQD is a function of fracture and fault density, while recovery is the ability of the drilling process to extract core. The large diametre core allows for high recovery rates in generally moderate to highly fractured ground and through gouge zones. Low recoveries were experienced in section of grit and rubble filled faults, where this material was washed out by the drilling process. Under extremely repetitive caving conditions, the drill string would freeze-up as the annulus collapsed in the grit and rubble sections, resulting in extremely slow drilling and in abandoning of the hole in two instances.

12.0 Sampling Method and Approach

12.1 Drilling Methods

A total of 287 drill holes have been captured in the database for the purposes of resource modeling and estimation. Several of the last holes from the 2006 drilling program have not been included in the current resource estimate due to time constraints.

Drilling procedures at the Schaft Creek project are as follows:

- The geologist sets out the holes in an area accessible to the drill rigs (the planned positions are typically marked by a steel peg);
- Drill pads are subsequently prepared by bulldozing, with the supervision of the geologist and the drill operator, in order to obtain a level platform on which to position the drill rig;
- The boreholes are drilled according to the geologist's instructions. Core or chips are laid out for the geologist to inspect;
- The geologist instructs the drill operator when to shut down the holes;
- Hole positions are surveyed by ground survey methods. Dip and azimuth angles are measured, and for some deeper holes a down hole survey is taken;
- The geologist logs the core, prepares and dispatches samples for analysis.

Table 12.1 summarizes various drilling campaigns done on the project area.

Table 12.1
Summary of Drilling Campaigns

Series	Holes	Total Length (m)	Average Length (m)	Minimum Length (m)	Maximum Length (m)
06CopperFox	42	9,066.80	215.88	78.00	351.00
05CopperFox	15	3,158.72	210.58	49.07	341.99
ASARCO	23	3,181.52	138.33	98.45	351.00
SILVER STD	3	628.81	209.60	189.89	221.29
HECLA	75	27,863.77	371.52	29.11	911.96
HECLA Paramount	10	2,923.95	292.39	140.21	477.62
TECK	119	24,804.34	208.44	89.00	593.45
	287	71,627.90	249.57	29.11	911.96

As can be seen from the table, drill hole length is not necessarily related to the drilling campaign. In total, 287 holes were drilled with an average length of 250 m, yielding a total of 71.6 km of drilling. Figure 12.1 Distribution of drill holes depicts the distribution of drill holes throughout the project area.

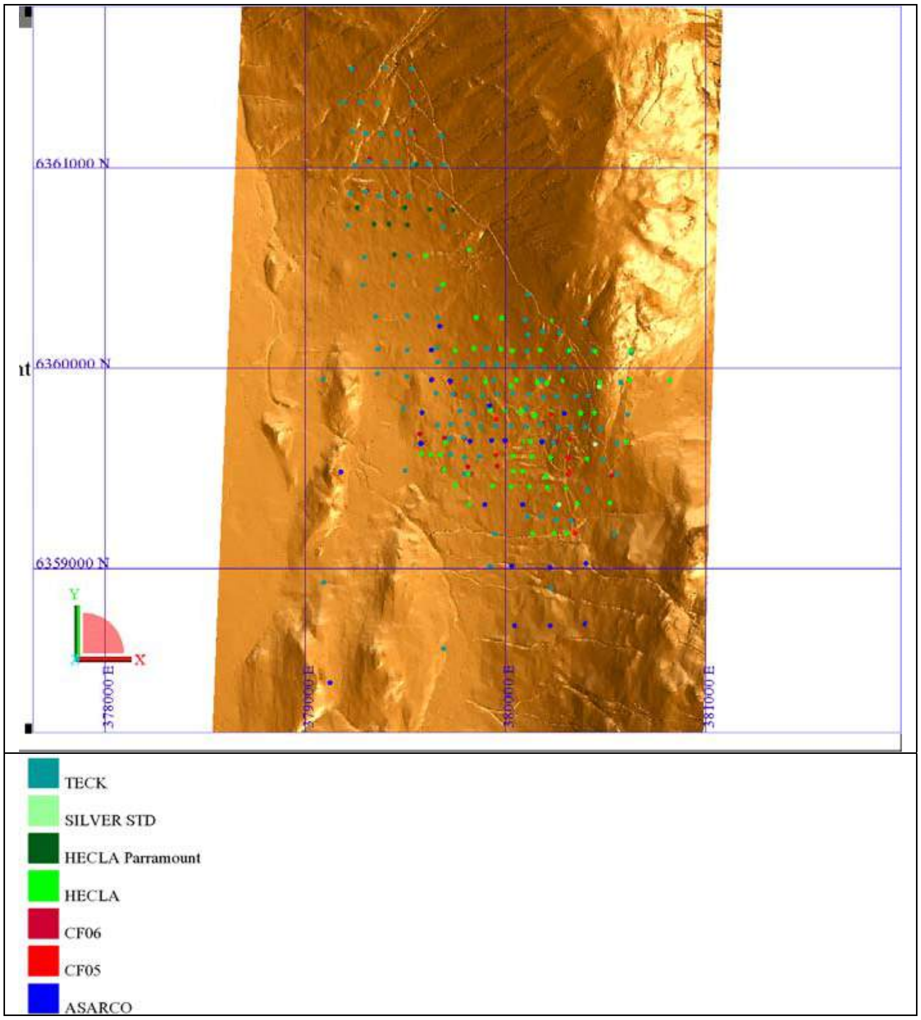


Figure 12.1 Distribution of drill holes

12.2 Analysis of Down Hole Surveys

Of the total 287 holes drilled, 217 (76%) have more than one down hole survey.

A chart plotting the difference between measured and planned position of the holes is depicted in Figure 12.2 and Figure 12.3. Total deviation is calculated using the first survey on the collar as the planned position and calculating the cumulative deviations on all subsequent down hole surveys.

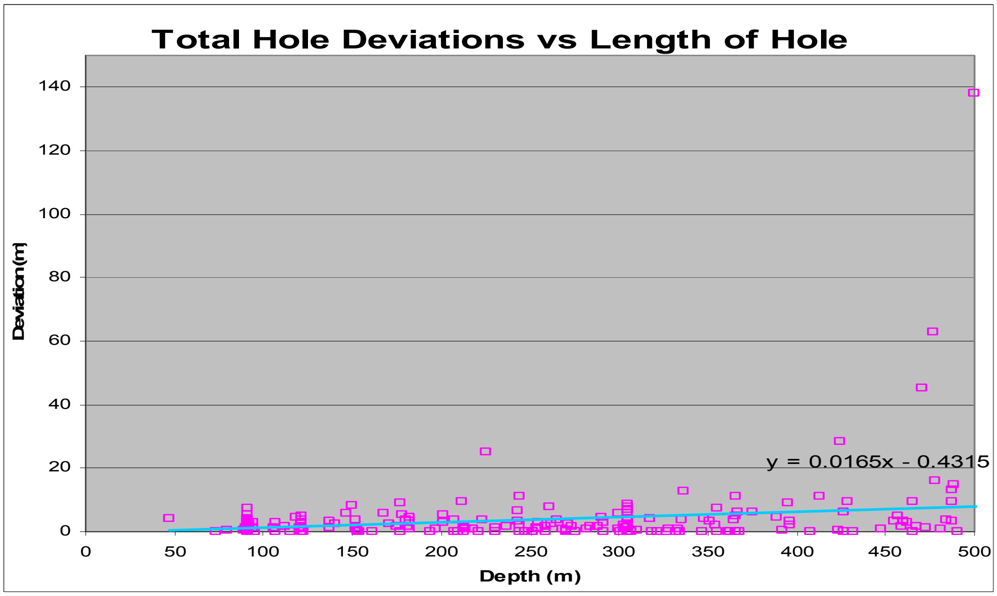


Figure 12.2 Deviations between measured and planned drill holes

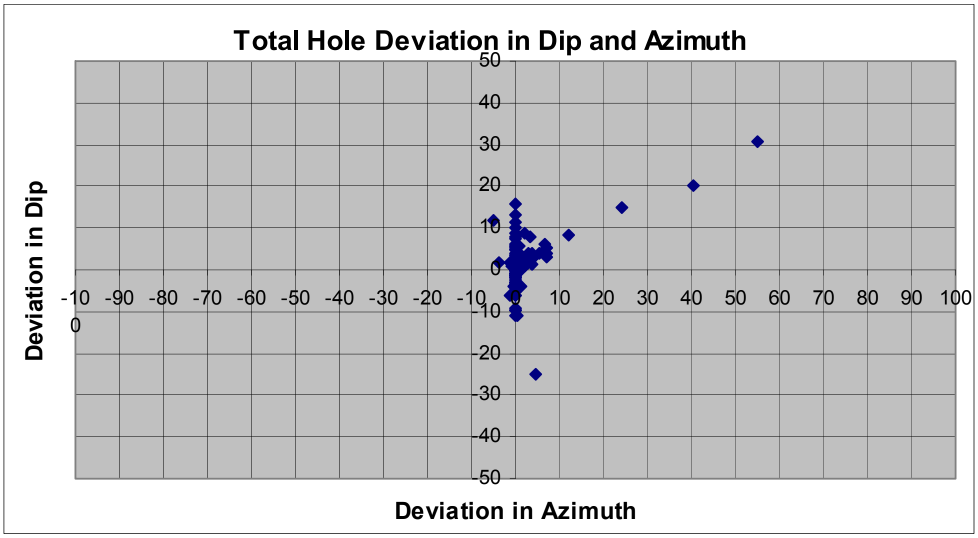


Figure 12.3 Deviations between measured and planned drill holes

From this chart it is clear that holes drilled did not deviate by much, as the average deviation was 3.92 m; however there were five holes that deviated by more than 20 m. Four of the five holes were drilled by TECK and one was from the 2006 Copper Fox drilling program. These holes are listed in Table 12.2. Thirteen holes deviated by more than 10 m from the planned position. The deviations that have occurred are a result of both dip and azimuth changes.

Table 12.2:Drill holes deviating by more than 20 m from the planned position

Table 12.2 Deviating Drill Holes		
Hole ID	Drill Depth (m)	Total Deviation (m)
T80CH145	499.87	138.19
T80CH139	477.01	62.78
T80CH146	470.92	45.19
T80CH148	424.43	28.33
06CF261	225.00	25.23

In the opinion of the authors, these deviations are not excessive; however, this leads to the recommendation that holes drilled to depths greater than 200 m should be routinely down hole surveyed and that the resulting deviations should be no more than 5 m from the planned positions.

12.3 Drill Hole Logging

Core logging of diamond drill core was performed by a geologist and recorded onto a log sheet. Core logging hinges around identifying lithological units. Once identified, the lithological units were put into a rock-TCC field in the database. A Copper Fox geologist worked through this list and broke these codes up into the rock-CCU1 field for modeling purposes.

Core description was done by identifying minerals, grain sizes (where applicable), mineral assemblages, colour, and lastly by giving a rock identifier code. Log sheets were then captured into excel.

12.4 Sampling Protocol, 2006 Drilling Program

A sampling protocol conforming to the Canadian National Securities Administrator's 43-101 requirements were implemented for both the 2005 and the 2006 exploration program. Great care was taken to ensure sample integrity, quality and chain of custody. PQ and HQ core were drilled for different purposes, therefore requiring different handling procedures.

A summary of procedures employed in the 2006 program is as follows:

- All PQ core, for the purpose of twinning and verifying archival results and obtaining material for metallurgical testing, was sawed in half and one half quartered. As the core was broken, the rubble was scooped out and divided according to samples. Pieces larger than 10 cm were sawed. Continuous sampling for assay samples was done in fixed 3.05 m intervals, for the purpose of matching samples of previous archival sampling.
- For PQ core, assigning two sets of sample numbers for: a) Assays, taking ¼ of the core approximately 35 lbs, and b) 'Metallurgical' (MET) samples for selected intervals, taking ½ of the core, weighing 70 lbs. The remaining quarter of the core is retained as a reference sample in the core boxes on site.
- For HQ core, no MET sampling was required. The core is sawed in half and one half is sampled for assay, while the other half is kept as a reference in the core box on site.
- Assay samples were placed in numbered 5-gal plastic pails and MET samples in numbered 10-gal pails, both with security lids. The sample tag for each pail is inserted into a small zip lock plastic bag and affixed to the inside of the pail's rim. Each sample pail carries a shipping tag fixed to the outside of the pail with the laboratory's address.
- Assay samples were shipped to International Plasma Labs Ltd. (IPL) in Richmond, B.C., and MET samples were sent to Process Research Assoc. Ltd (PRA) in Richmond, B.C. For this purpose both sample groups were air lifted to a strip at the road and stored in a locked Seacan container. At weekly intervals, a bonded trucking firm retrieves both sample groups and delivers them directly to the laboratories.

13.0 Sample Preparation, Analysis and Security

13.1 Sample Preparation, at IPL Labs

- Blind duplicates, standards and blanks are inserted, in the field, into the sample stream at a 40 sample interval, for quality control;
- Assay samples are analyzed for:
- Cu %, Mo %, Au g/t (2 assay tons), Ag g/t
- Multi-element spectral analysis;
- Sample preparation for assay samples: A 4 to 5 kg portion of the core sample is crushed to 2 mm size and homogenized. A split of approximately 300 g is pulverized to minus 150 mesh and homogenized by rolling.

13.2 Ore Grade Elements by Multi-Acid Digestion/ICP or ASS

- 0.25 to 1.0 g of sample is weighed and transferred into a 150 mL beaker. HCl, HNO_3, $HClO_4$, and HF acid solutions are added and digested on a hot plate until dry. The sample is boiled again with 80 mL of 25% HCl for 10 min, cooled, bulked up to a fixed volume with distilled H_2O and thoroughly mixed;
- Cu, Mo, and Ag are determined using an inductively coupled plasma emission spectrometre. All elements are corrected for inter-element interference and all data are stored onto a computer diskette.
- Quality control: the spectrophotometre is first calibrated using three known standards and a blank. The samples to be analyzed are then run in batches of 38 or fewer samples. Two tubes with an in-house standard and an acid blank are digested with the samples. A known standard with characteristics best matching the samples is chosen and inserted after every 15th sample. Every 20th sample is re-weighed and analyzed at the end of the batch. The blank used at the beginning of the run is analyzed again. The readings of the control samples are compared with the ‘pre-rack known’ to detect any calibration drift.

13.3 Fire Assay Gold Assay

- Duplicates of 50 g (2 assay tons) are weighed into fusion pots together with various flux materials, including lead oxide. After thorough mixing of silver inquart, a thin borax layer is added.
- The sample is placed into a fire assay furnace at 2,000 ºF (1,093ºC) for 1 hr. Elemental lead, from lead oxide, collects the gold and silver.
- After 1 hr fusion, the sample is poured into a conical cast iron mold. The gold and silver bearing lead button/bead at the bottom is separated from the slag.
- The lead button is placed in a preheated cupel into the furnace for a second separation at 1,650ºF (898.9 ºC). Lead is absorbed by the cupel, whereas gold and silver remain on the surface of the cupel.
- After 45 min of cupellation, the cupel is removed from the furnace and cooled. The doré bead containing the precious metals is transferred to a test tube (sample duplicates are combined) and dissolved in hot aqua regia.

- The gold in solution is determined with an atomic absorption spectrophotometre. The gold value in ppb or g/t is calculated by comparing the reading with that of a standard.
- Fire assay quality control: every group of 24 fusion pots contains 22 samples, one internal standard or blank, and a re-assay of every 20th sample. Samples with gold >1,000 ppb are automatically checked by fire assay/AA. Samples with gold >10,000 ppb are automatically checked by fire assay/gravimetric methods.

14.0 Data Verification

The Schaft Creek deposit has been the object of detailed exploration since the late 1960's to 1981 by various operators. A significant database was developed during that period that included more than 18,000 samples with analyses.

D. Beauchamp, P. Geol and consultant to Copper Fox reviewed these historic sample analyses. The historic database was found to include assay results from at least six different laboratories. Many of thelaboratories did not describe the analytical methods in any detail that were used on the samples. There are no records of any field-based quality control program or any indication of the quality control practices by thelaboratories.

Comparative analyses between several of the labs indicated that the reproducibility for copper was satisfactory, although several of the labs show a distinct bias one to the other. Comparison of Mo, Au, and Ag analyses were more problematic.

A small re-assay program of 160 samples was undertaken in 2004 by Process Research Associates of Vancouver using their affiliate, IPL as the analytical laboratory. These repeat analyses were done on the other half of drill core from four holes. The Cu data showed a scatter around the 1:1 line when compared to the original analyses, but there was little overall bias. The scatter is not unusual when comparing two halves of core. The Mo, Au, and Ag did not repeat as well. This is a clear indication that the original assay data is valid for at least Cu, but additional confirmatory analysis is required for other elements.

Hence, an essential component of the 2005 and 2006 field program was the twinning of historical drill holes to verify the reliability of the archival database; this was accomplished by duplicating the original assay intervals with new core samples along the same specified intervals. The databases generated from the two sets of records were then statistically compared to provide a level of confidence in the incorporation of the historic results to future ore reserves and ore modeling.

14.1 2006 Quality Assurance and Quality Control

In 2006, a program of Quality Assurance and Quality Control (QA/QC) was implemented as part of the diamond drilling and sampling program at the Schaft Creek Project operated by Copper Fox Metals Inc. The purpose of this program was to ensure that reliable and dependable assay results were reported for samples that were sent to the laboratory.

This report summarizes the QA/QC procedures that were used in the field and laboratory. The field procedures are based on a guidebook that describes in detail the sampling procedures and data recording methods so that consistent methods are applied throughout the program.

14.1.1 Field Procedures - Core Sampling

The drill core was sampled at intervals of 3.05 m so that the results would be comparable with historical data that was carried out at 10 ft intervals. The core was split with a rock saw, and half was submitted for analysis and the other half of the core was returned to the core box.

As part of the QA/QC program, a blank sample, a standard sample and a duplicate sample were submitted in each batch of 40 samples.

The purpose of the blank sample is to determine whether there is any contamination in the laboratory from one analysis to another. If the blank sample returns values that are at or below detection limit, we can be confident that there is no contamination.

The purpose of the standard sample is to determine whether the laboratory is providing accurate results. The recommended value for the standard samples was determined by carrying out several assays on the sample, and the results provided by the laboratory should be within a known range. Consulting Geochemist, Barry Smee, has suggested that results should be within ±3 standard deviations of the recommended value for each element. A total of six standard samples were used during the 2006 program. Two of these were from Canadianlaboratories, and four were prepared by International Plasmalaboratories (IPL) for Copper Fox from samples from the property.

Duplicate samples were submitted by further splitting the half core in half. The two samples are numbered sequentially and submitted for analysis. The duplicated are two different samples and results of the assays could vary if mineralized veins are unevenly distributed in the two adjoining samples.

14.1.2 Laboratory Procedures

The laboratory procedures included discussions with laboratory personnel to establish a consistent and routine protocol for the assays and reporting of the results, and an unannounced visit to the laboratory to examine procedures, cleanliness and set-up in late September. The samples were processed and assayed for gold using two-assay ton samples and for a 30-element ICP package.

In total, 77 blanks, 77 duplicates and 78 standards were analyzed. Limits of detection are of 0.01% Cu, 0.01% Mo and 0.5 g/t Ag by ICP method and 0.01 g/T Au by assay.

14.1.3 Evaluation of the Field Procedures

The field procedures were followed meticulously and few problems occurred. Overall, the field personnel carried out the splitting and numbering of more than 3,000 samples in a remarkably consistent manner.

The few variations in procedures that were recognized included the following:

- One shipment of metallurgical samples was sent to the assay laboratory instead of the metallurgical laboratory;
- On two occasions a standard sample was not included in the sample pails;
- Once the sample sequence between the standard and blank sample was inverted.

Neither of these lapses was of any significance in the QA/QC process.

14.1.4 Evaluation of the Laboratory Procedures

The laboratory provided results within 10 - 14 days of receiving the samples in Vancouver. Management personnel were very attentive and helpful in answering questions or queries on results and on the procedures used.

An unannounced visit to the laboratory in late September revealed that the facility is clean, well organized and is operated in an efficient manner. In-house clerical procedures ensure that the laboratory personnel are unaware whose samples are being processed, providing an additional level of security for the company's results.

Several minor procedural issues were noticed during the course of the program:

- In drill hole 06CF255, the first sample of the drill hole reported a value of 95.0 g/t Au and a re-assay by IPL on the same sample gave 93.1 g/t Au. Elements in the ICP package that were also elevated include 14.7 ppm Ag, 327 ppm As, 776 ppm Pb and 73 ppm W.
- After several discussions the sample was sieved to 150 mesh. Both fractions were assayed separately, giving results of 83.95 in the coarse fraction and 0.42 g/t Au in the finer one, confirming the presence of the gold nugget effect in this sample.
- Results were originally reported to an accuracy of two decimals in percent for copper and molybdenum. Only later in the program was it suggested that all results should be to three decimals. IPL reprocessed the results recorded during the analysis and provided results to three decimals for copper and molybdenum.
- In two standard samples, assay results for gold were about 50% of the recommended value. Upon questioning, IPL inquired and reported that the two halves of the two assay ton analysis had not been added, but only for these two samples. Two assay ton samples are analyzed in two 30 g crucibles and the quantities reported are added. The larger crucibles available have a capacity of only 50 g.

14.1.5 Statistical Analysis of the Results

Graphical representation of the blank samples show that they are at the level of detection or below for each of the four metals of interest.

- Results for standard samples show that samples plot within its respective mean ±3 standard deviations of the expected results for their group and element.
- Statistical analysis of all duplicate samples by ANOVA (Analysis of Variance) shows that there is no significant difference between the results of the paired duplicates samples for copper, molybdenum, gold and silver.

- In order to verify the accuracy of the ICP analysis, ten samples were submitted for copper, molybdenum, gold and silver assay. The results show a correlation of 98.5 to 99.9% and a study by ANOVA shows that the samples are well within the same probability distribution.

14.1.6 2006 Quality Assurance and Quality Control Conclusions

The core splitting and sampling in the field was carried out according to the requirements. The minor variations in the procedures or slip-ups were of no consequence on the results or on the reliability of the results.

Laboratory procedures were followed correctly, and the occasional change in procedure was detected and corrected without any consequences to the analytical results or trustworthiness of the outcome.

Data verification using different statistical methods on the blank, duplicate and standard samples show that the data is dependable and that all evidence supports the conclusion that the results are reliable and accurate.

For these reasons the assay results for the 2006 drilling program are deemed to be up to standards and within acceptable limits.

14.2 Review of Analytical Database for Resource Modeling

14.2.1 Sampling

A total of 22,148 samples were submitted for analysis. Table 14.1 provides an overview of the samples sent for analysis relative to the drilling campaign.

Table 14.1
Overview of Samples Relative to Drilling Campaign

	Holes	Geological	Samples (#)
06CopperFox	42	2,910	2,932
05CopperFox	15	1,023	1,034
ASARCO	23	1,155	986
SILVER STD	3	239	194
HECLA	75	10,483	8,730
HECLA Paramount	10	1,033	872
TECK	119	10,043	7,400
Total	287	26,886	22,148

Sampling for both sets of drilling were submitted to the same laboratory (IPS) and the results were entered into spreadsheets for both the RC and diamond drilling holes. The major fields captured in the spreadsheet included Cu%, Mo%, Au(g/t), Ag(g/t) along with their respective duplicates and check samples which had been submitted to a different laboratory (Chemex).

14.2.1.1 *Validation of Sampling and Check Sampling*

Only the recent drill holes done by Copper Fox were subjected to reliable quality control. Out of the current database used for modeling purposes, a total of 1.8% of the samples were directed toward quality control. Table 14.2 demonstrates the amount of quality control samples examined.

Table 14.2
Quality Control Samples

	Sampling	Duplicate	Chemex
06CopperFox	2932	19	-
05CopperFox	1034	27	26
	3966	46	26

Looking at the individual statistics we see that there is a good correlation between the original samples and check samples submitted for analysis. This correlation is depicted in Table 14.3, below, where the average grade, minimum and maximum grades for the two data sets as well as the Correlation Coefficient and R squared values are given.

Table 14.3
Correlation Between Original and Check Samples (1st Laboratory)

Duplicates	46 samples	Cu%	Mo%	Au (g/t)	Ag (g/t)
Original Sample	Minimum	0.01	0.001	0.01	0.00
	Maximum	0.92	0.065	1.02	6.90
	Average	0.34	0.018	0.23	1.75
Check Sample	Minimum	0.01	0.001	0.01	0.00
	Maximum	1.00	0.080	1.09	7.60
	Average	0.36	0.019	0.25	1.65
Correlative Statistics	Correlation Coefficient	0.933	0.826	0.807	0.933
	R-squared	0.870	0.682	0.652	0.870

Table 14.4
Correlation Between Original and Check Samples (2nd Laboratory)

2nd Laboratory	26 samples	Cu%	Mo%	Au (g/t)	Ag (g/t)
Original Sample	Minimum	0.05	0.00	0.02	0.00
	Maximum	2.31	0.25	1.02	14.40
	Average	0.52	0.03	0.23	3.05
Check Sample	Minimum	0.04	0.00	0.02	1.00
	Maximum	2.20	0.25	1.50	11.00
	Average	0.49	0.03	0.27	3.96
Correlative Statistics	Correlation Coefficient	0.997	0.992	0.966	0.997
	R-squared	0.995	0.985	0.932	0.995

For a perfect correlation the correlation coefficient and R-squared would be equal to 1.00. Figure 14.1 below depict the statistics for the 46 check samples compared with the original sample results, plotted on a Q-Q plot.

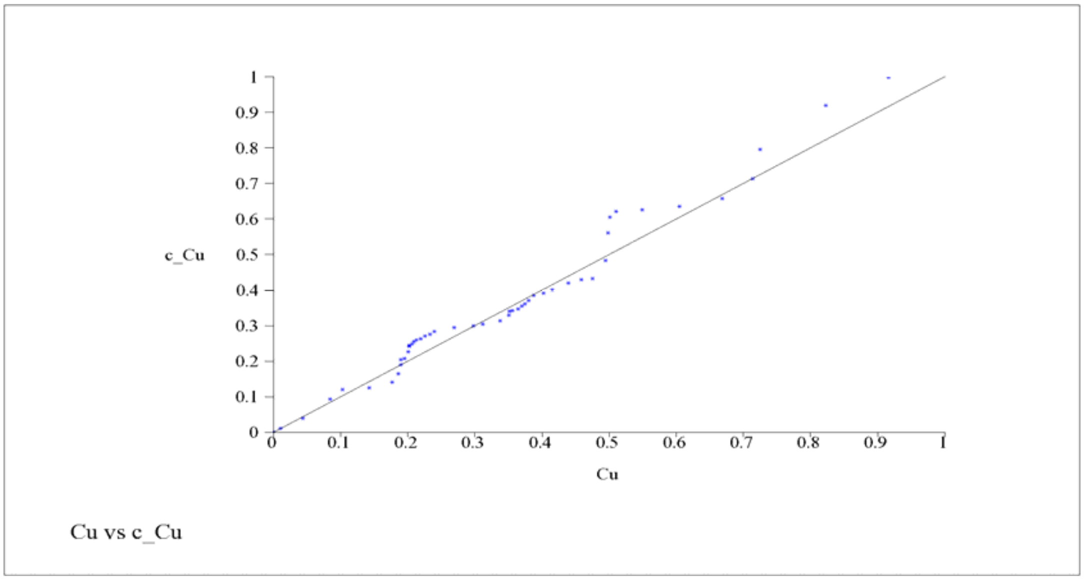


Figure 14.1: Statistics for check samples (1st laboratory), Q-Q plot of Cu

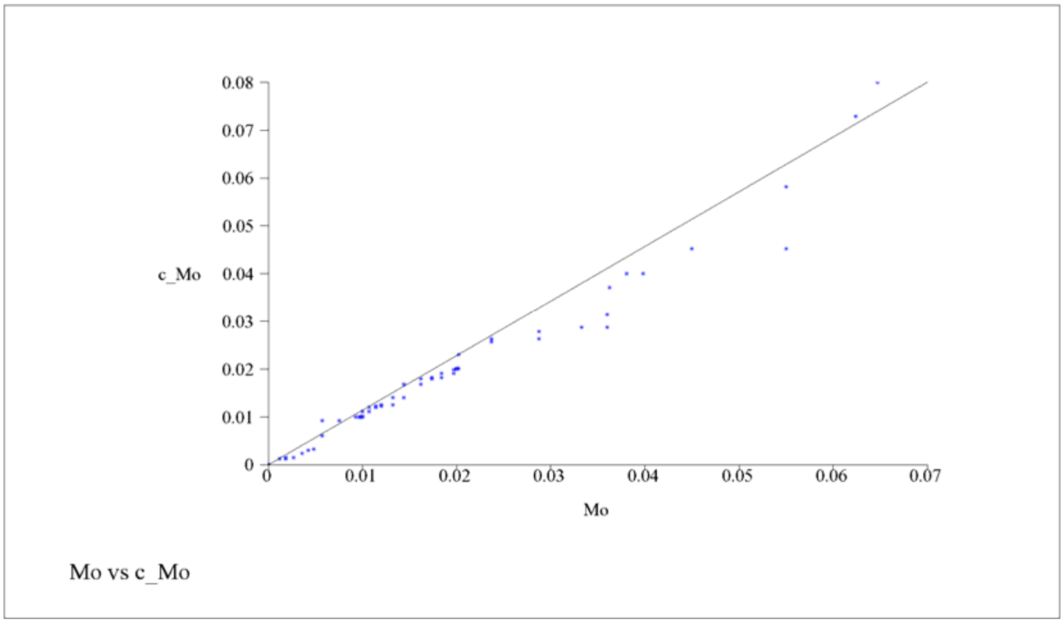


Figure 14.2: Statistics for check samples (1st laboratory), Q-Q plot of Mo

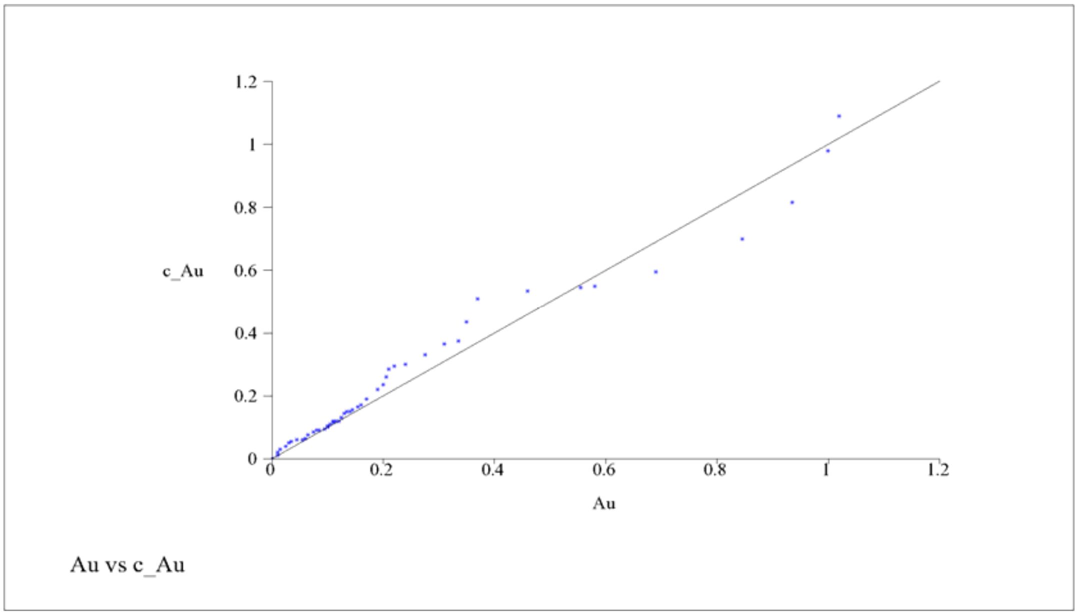


Figure 14.3: Statistics for check samples (1st laboratory), Q-Q plot of Au

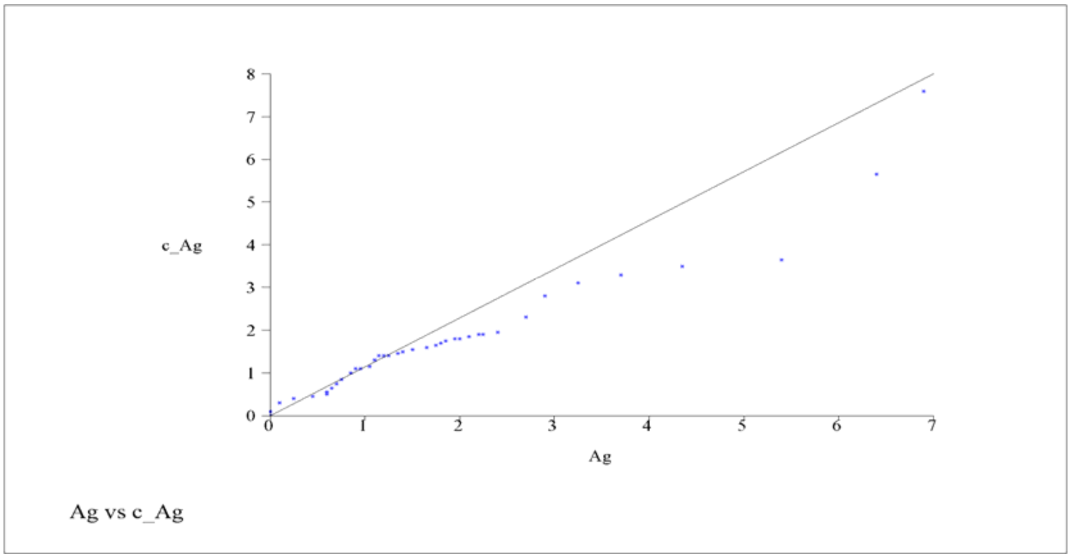


Figure 14.4: Statistics for check samples (1st laboratory), Q-Q plot of Ag

Good correlation exists for the Cu and Au sample populations whereas the Mo shows slightly lower grades in the check samples than the original samples above 0.025%. The Ag shows lower grades in the check samples for the Ag population above a grade of 1.5 g/t.

Figure 14.5: below depict the statistics for the 26 samples put into the second laboratory for check analysis plotted on a Q-Q plot.

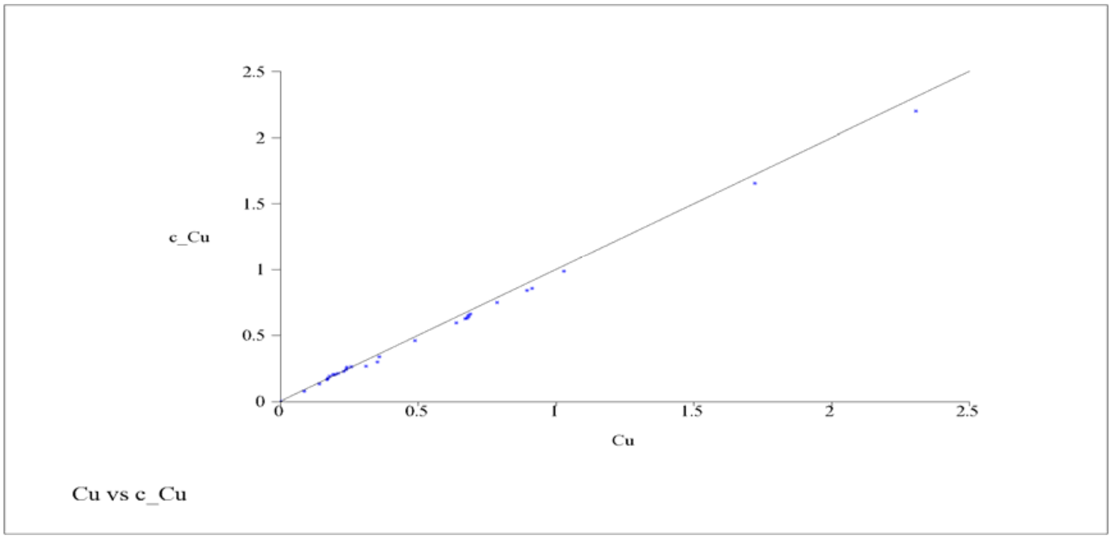


Figure 14.5: Statistics for check samples (2nd laboratory), Q-Q plot of Cu

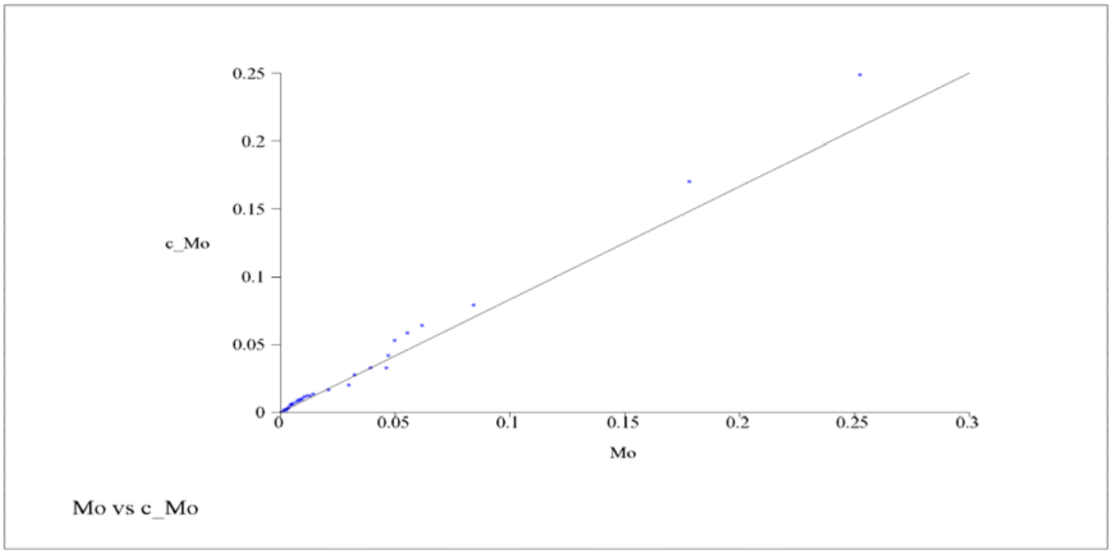


Figure 14.6: Statistics for check samples (2nd laboratory), Q-Q plot of Mo

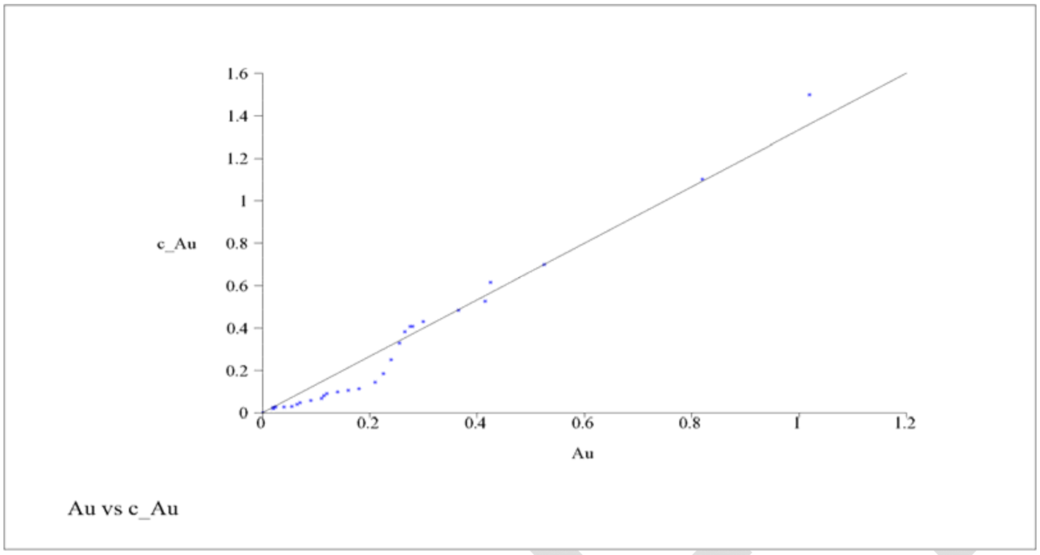


Figure 14.7: Statistics for check samples (2nd laboratory), Q-Q plot of Au

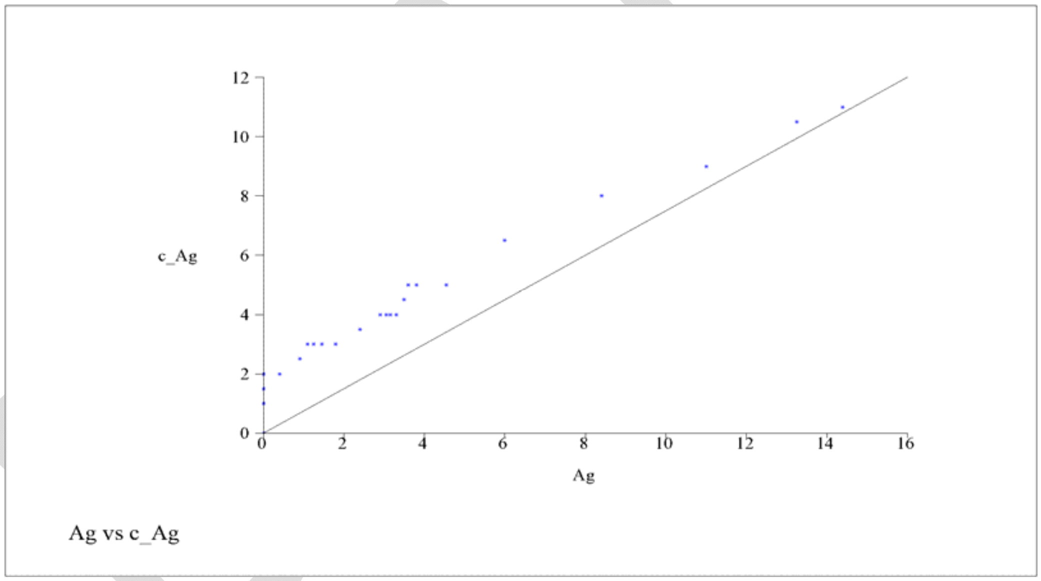


Figure 14.8: Statistics for check samples (2nd laboratory), Q-Q plot of Ag

Good correlation exists for the Cu and Mo sample populations whereas the Ag shows higher grades for the check samples throughout the grade ranges. The Au shows lower grades in the check up to 0.3 g/t- thereafter good correlation exists.

It is in the author's opinion that the check samples show a good repeatability and have a spread that is to be expected for detailed statistical and spatial statistical analysis; this is also evident by the good correlation coefficient and R squared values. It has to be noted though that the check samples represent only a small portion (1.8%) of the total sample population.

15.0 Adjacent Properties

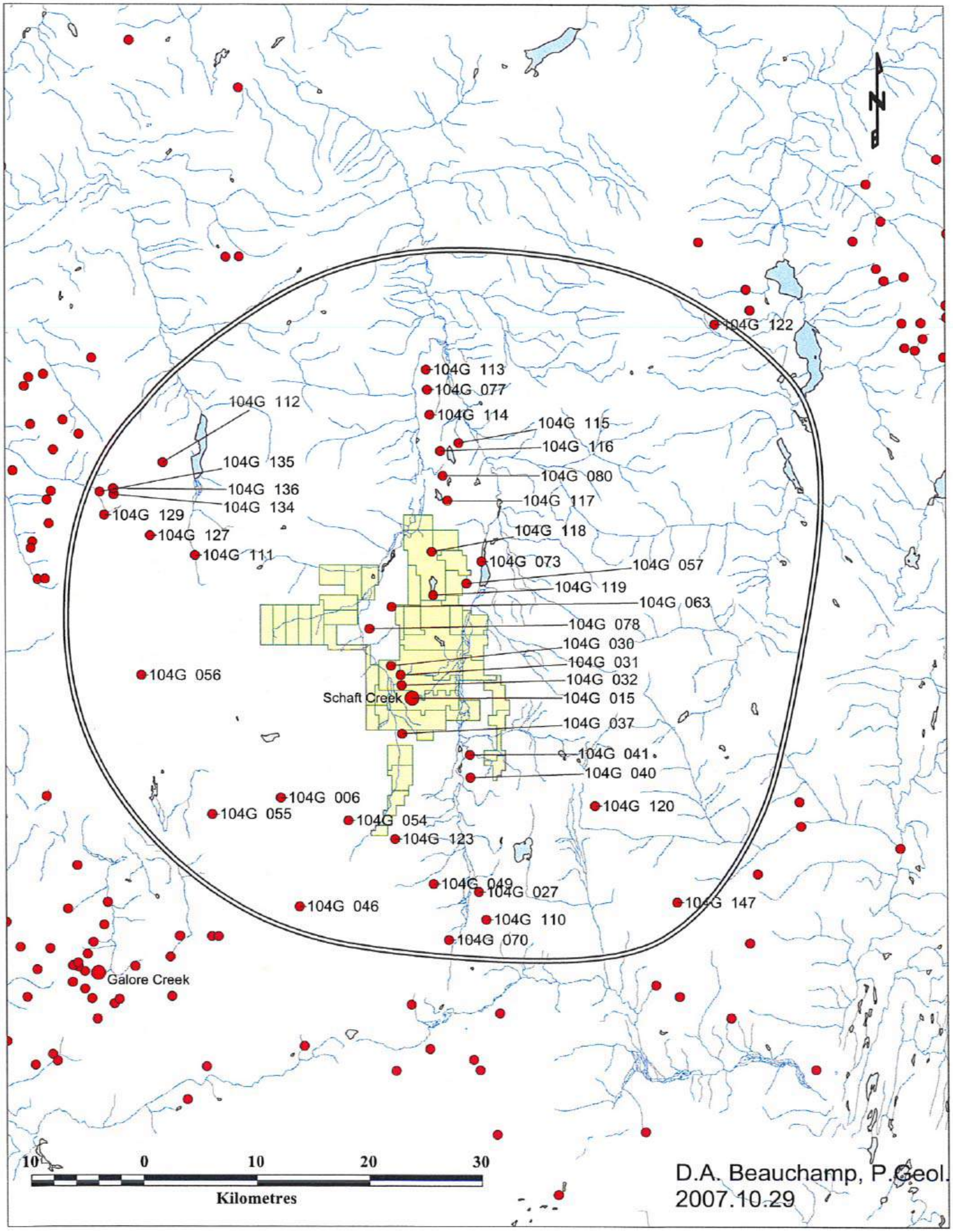


Figure 15.1 Mineral Occurrences within 20 km of Schaft Creek

Figure 15.1 above shows the minerals occurances that are immediately adjacent to the Schaft Creek mineral property. None of these mineral occurances are currently in or near production, although a few have exploration activities ongoing.

16.0 Mineral Processing and Metallurgical Testing

This section was authored by Raymond Hyyppa, a metallurgical consultant retained by Copper Fox Metals.

16.1 Historical Test Programs

The Schaft Creek resource has been given considerable study starting from the early studies by Hecla Mining Company (Hecla) in 1970 and 1971. The following Table 16.1 summarizes the historical test work:

Table 16.1
Summarizes the Historical Test Work

Test Type	Laboratory	Mining Company
Preliminary Flotation Tests	Lakefield Research	Hecla Mining Company - 1970-71
Preliminary Flotation Tests	Lakefield Research	Tech Mining Group - 1981-82
Sample Validation	Process Research Assoc.	Copper Fox Metals, Inc. - 2004
Laboratory Flotation Test	Process Research Assoc.	Copper Fox Metals, Inc. - 2005
Laboratory Flotation Tests	Process Research Assoc.	Copper Fox Metals, Inc. - 2006
Laboratory Flotation Tests	Process Research Assoc.	Copper Fox Metals, Inc. - 2007

The metallurgical tests of Hecla's 1970-1971 programs at Lakefield Research have not been reviewed by Mr. Hyyppa.

In 1981-82, Tech Mining Group tested the Schaft Creek resource and investigated the effect of primary grind size; regrind size, reagent types and reagent dosages to optimize the metal grade and recovery in a combined Copper/Molybdenum Bulk Concentrate. Tests for molybdenum separation were not performed during this period. The resource grades of the core used by Lakefield Research was similar to the one used in the recent tests at 0.36% Cu and 0.0211% Mo. Lakefield was able to produce a 3rd cleaner Copper/Molybdenum Concentrate with metal grades of 28.5% Cu and 1.68% Mo at metal recoveries of 85.1% and 73.5% respectively by using a primary grind of P80 = 100 microns. The rougher concentrate was reground to approximately a P80 = 15 to 20 microns.

In February 2004, Process Research Associates (PRA) performed follow-up metallurgical work. The initial work consisted of testing old core for assay validation of the copper and molybdenum values with addition of iron and sulfur analyses. The PRA assays showed adequate correspondence to the historical database. Scoping flotation tests were conducted on these core samples to determine the suitability of the old core for additional testing. While the results were similar to those achieved by Lakefield in 1981-82, it was nevertheless decided to conduct all future tests on fresh core in an attempt to improve results. Acid Base Accounting (ABA) of the flotation tailings also indicated an excess of neutralizing potential, a property that needs to be considered when planning the tailings impoundment design.

16.2 Review of 2005 Drill Core Tests at PRA

The author reviewed available PRA Schaft Creek test data and a draft of the partially completed report. Testing involved ½ splits of PQ (83.1 mm) core from the 2005 drilling season. Since a final report had not been issued, the following comments are based on the information available to the reviewer.

- Testing was done on 5 sample composites (from a total of approximately 27,000 kg of material) selected from PQ Schaft Creek core higher grade intervals from each of the Liard and West Breccia Zones

The samples were grouped into the following four zones:

- Main Liard Zone (MLZ)
- West Liard Zone (WLZ)
- North Liard Zone (NLZ)
- West Breccia Zone (WBZ)

An additional composite of equal portions of all four samples was made, the "Pit Composite" where most of the early test work was conducted to establish the optimum rougher flotation grind size for both copper and molybdenum recovery. PRA's selected grind of approximately 160 microns (P80 = 160 micron) did not achieve the highest copper recovery but it was reported to maximize the recovery of molybdenum. Analysis of the PRA 2005 test data indicated to the author that copper recovery could be enhanced at grinds finer than 160 microns but decreased molybdenum recoveries should be expected below 140 microns. It was therefore decided to investigate the response of the core from the 2006 drilling program at grind sizes of 80% passing 75, 100 and 150 microns to fine-tune the optimum grinding parametres. The PRA 2006 core tests confirmed that higher recoveries could be achieved at P80 = 100 microns.

- The flowsheet used by PRA for the locked cycle tests on the 2005 drill core is shown in Figure 16.1. Table 16.5 summarizes the locked-cycle test data for the five composites prepared. The data was analyzed and recoveries plotted for each of the four metals, Cu, Mo, Au, and Ag for a range of rougher flotation feed size distributions. Figures 16.2 and 16.3 show the variation of copper, molybdenum, gold and silver recovery as a function of rougher flotation feed size. While PRA's intention was to grind all composites to a uniform size of P80 = 160 by fixing the grind time, the variation of the individual samples hardness produced grind sizes ranging from 132 microns to 165 microns. The resulting data was therefore used to evaluate the effect of grind size on recoveries.
- The PRA locked-cycle test results compare favorably to the historical test work conducted by Tech Corporation in 1981-82.
- The 3rd Cleaner Bulk Concentrate for the 2005 core locked cycle test for the Pit Composite indicates no rejection of elements such as Sb (at 95 ppm), As (at 18 ppm), Bi (at 344 ppm) Pb (at 1,282 ppm) or Zn (at 2,400 ppm). This may have been due to limited number of cycles utilized (six for all tests). Additional (10 to 12) and longer cycles may show an increase in the levels of these metals. Impurity levels of the Copper Concentrate for the 2006 core

will be evaluated in the up-coming test program. While penalty elements and the actual penalty levels vary depending on the smelter, according to the Model Smelter Schedule for Copper Concentrates from Mining Cost Service, Schaft Creek concentrates are well below rejection levels.

- A pilot plant test was run on 1,600 kg of assay reject and core samples from the 2005 drill core to produce a sample of Bulk Concentrate for Molybdenum separation tests (Figure 16.4). The products of five tests were analyzed and summarized in Table 16.6. The Bulk Concentrate contained high levels of carbon identified as graphite, whose source was established in a prior pilot plant test. Since Graphite behaves very similarly to molybdenum, it cannot be depressed without also depressing molybdenum. The molybdenum concentrate assay averaged 12.48% Mo with a recovery of 32.97%, with a carbon (graphite) content of 35.78% equivalent to 42.92% of total carbon contained in the bulk concentrate. Graphite is a major diluent in the molybdenum concentrate. It should be noted however that recent pilot plant tests at G & T Metallurgical Services, Inc., Kamloops, B.C. were completed without any evidence of graphite contamination. Molybdenum separation tests are still in progress.
  Rhenium (Re) was found in the 11th cleaner concentrate (test F75) in concentrations of 180 ppm (Re) as compared to an assay of 18.44% Mo. A mineralogical examination of the concentrate from test F75 indicates that Rhenium occurs as discrete particles intimately associated with the molybdenite. If a more typical molybdenum concentrate of around 54% Mo were achieved, the equivalent rhenium assay would be 530 ppm, assuming that all of the rhenium is associated with molybdenite. If confirmed by additional testing Schaft Creek may have significant rhenium values, improving thereby the revenue stream. All of the 2006 core test final molybdenum cleaner concentrates is being analyzed by mass spectrometry to obtain a better estimate of the Re content of the Schaft Creek resource.
- Samples of each four zones composite were sent to Hazen Research for Bond rod mill work index (RMi), Bond ball mill work index test (BWi) and Bond abrasion resistance tests (Ai). These are reported in Table 16.2 below.

Table 16.2
RMi, BWi, and Ai Tests

Composite	Ai	RMi	BWi
MLS	0.25	24.0	22.4
WBZ	0.30	21.2	20.7
WLZ	0.27	23.7	24.5
NLZ	0.18	24.1	24.1
Avg.	0.25	23.3	22.9

This indicates that the samples are moderately hard and abrasive. The range of hardness value from 20.7 to 24.1 indicates that the rod mill grinding time must be adjusted for each type of material in order to achieve a consistent grind size.

16.3 Mineralogical Evaluation of 2005 Samples by Lehne & Associates

Lehne & Associates, of Germany, conducted a mineralogical evaluation of representative samples from the five sample composites, from selected flotation products (2005 drill core) and from the three zone composites from 2006 drill core. It was concluded that there are no mineralogical differences among the composites and therefore the 2005 and 2006 drill core should behave in similar ways during processing and should require similar grinding size distributions.

The main constituents of the mineralization are chalcopyrite, pyrite and bornite as well as magnetite and hematite; molybdenite is less common. In a few samples, minor chalcocite, often associated with bornite, can be observed. Tennantite-tetrahedrite has been found in association with chalcopyrite. Sphalerite and galena appear to be of a very sporadic occurrence. Digenite and covellite represent secondary copper sulfides that formed during an incipient supergene alteration which is only weakly developed in the investigated composites. Proustite-pyrargyrite, an important contributor of silver was observed in tow of the Bulk Copper/Molybdenum flotation concentrates.

The flotation concentrates for the 2005 drill core test program were reground to approximately 80% passing 15 microns. Lehne & Associates determined that complete mineral liberation was achieved at this grind and the diluents, primarily pyrite and gangue, were liberated. Copper Concentrate grades of greater than 30% should be possible.

16.4 Mineralogical Evaluation of 2005 Moly Samples by Dr. Peter Fischer

- Dr. Peter Fischer (a geological consultant to Copper Fox) also analyzed the 11th stage molybdenum concentrate (PRA test 75) and indicated an unconfirmed presence of graphite, because the grains observed dominated over molybdenite.
- Dr. Fischer also was observed the presence of rhenium in one molybdenite grain.

16.5 2005 Core Grade Variability Testing

25 open circuit tests were performed to investigate the flotation response of a wide range of copper head assays (ranging from 0.146 to 1.253 % Cu). The results showed that higher head grades, achieved higher 3rd Cleaner Concentrate grade. Similarly, the higher the head grade, the higher the metal recovery into the third Cleaner Concentrate (Figures 16.5 through 16.12).

16.6 Hydrometallurgical Treatment of Copper Concentrates

Approximately 3.5 kg (2005 Core) of 3rd Cleaner Bulk Copper/Molybdenum Concentrate were tested by Cominco Engineering Services Ltd. (CESL) for amenability to its proprietary leaching technology. The CESL Process allows lower grade copper concentrates to be converted to high purity cathode copper without the conventional smelter. Preliminary testing of the sample was very encouraging and has indicated that Schaft Creek copper concentrates assaying 26.3% Cu are amenable to the CESL process with yields of 98.7 to 98.8% copper extraction. Additional confirmatory testing is in progress.

16.7 Flotation Tailings Thickener Tests

A sample of the pilot plant flotation tailings was tested to determine the settling characteristics for thickener sizing. Only a single settling test, without flocculants, was conducted. The results are shown in Table 16.8. However, since no flocculants were used, the information was not used for thickener sizing.

16.8 Review of the 2006 Drill Core Tests at PRA

6. PRA started testing the 2006 core samples in February 2007. Of a total of 50,000 kg of drill core, approximately 725 kg were used by PRA for the laboratory program and 6,000 kg for pilot plant testing. Vandan Suhbatar and Raymond Hyyppa visited the Schaft Creek property from October 13 through October 16, 2006 to review the property and drill core and to select the samples of PQ drill core from the 2006 drilling season for additional testing. Each of three resource areas - Liard (LZ), West Breccia (WBZ) and Paramount (PZ) were drilled during the 2006 season and representative samples identified and collected (384 kg of Liard, 169 kg of Paramount and 172 kg of West Breccia) for the PRA laboratory test program.

7. Drill location maps, including holes drilled by Tech and Hecla were provided and the anticipated pit limits for each area were outlined. Material was selected from areas within the anticipated pit limits. The drill logs and 3 metre interval assay data were evaluated to determine the drill intervals that would provide an average copper assay of 0.35 to 0.40 % copper from each resource area. Interval assays were not available for all of the selected holes.

   Approximately 90 kg of samples from each of three zones were sent to Hazen Research for comminution testing. The comminution samples were selected from the same holes as used for the samples sent to PRA. Also, 2,000 kg of each zone (6,000 kg total) were crushed to minus 10 mesh in preparation for pilot plant testing.

8. PRA prepared a composite for each of the three resource areas and a fourth composite (Master) - was prepared with equal portions of the three zones. Since test results indicated the optimum primary grind to be P_{80} = 100 microns, this was the size selected for all Rougher and Scavenger Flotation locked cycle tests. (Figure 16.13). Rougher and Scavenger concentrates were reground to P_{80} = 20 to 25 microns. A grind size versus time calibration was made for lab rod mill for each individual composite. The results of the four locked cycle tests (Table 16.7) indicate a need to regrind the feed to the cleaner circuit to 15 microns in order to achieve both high concentrate grades and recoveries. Higher concentrate grades were achieved for the 2005 drill core tests using a P_{80} of 15 to 20 microns. Comparisons of locked cycle test data for the 2005 (Table 16.5) and 2006 drill (Table 16.7) core, shows that flotation feed ground to a P_{80} = 100 microns results in higher metal recoveries and a regrind of Rougher and Scavenger Flotation Concentrate to P_{80} = 15 microns results in higher metal grades.

9. The locked cycle test data for 2005 (average of MLS, WLZ and NLZ samples) and the 2006 drill core for the Liard Zone also indicate that for similar head grades (0.38 % $Cu_{2005\text{-}LiardZone}$ and 0.33% Cu_{2006}), the finer primary grind of P_{80} = 109 microns results in higher 3rd Cleaner copper recovery by 6.7% (84.10 % Cu_{2006} vs. 77.40 % Cu_{2005}). It is believed that the finer regrind size used for the 2005 samples ($P80_{2006}$ = 20 microns vs. $P80_{2005}$ = 16 microns) was due to their difference in head grade as these samples appear to be very sensitive to grind size. Reagent selection can play a major part in influencing concentrate grade. It is possible that the lower 3rd Cleaner Concentrate grade for the 2006 core samples can be increased with a different reagent selection and dosage. Finally, the 2005 core samples may have a larger proportion of secondary copper minerals that would facilitate higher copper concentrate grades.

10. Information received to date indicates that the life of mine resource would comprise approximately 60% Liard Zone, 25% Paramount Zone and 15% West Breccia Zone materials. Using this assumption with a primary grind size of 80%, passing 100 microns and a regrind size of 80% passing 15 to 20 microns, the following average metal grades and assays can be expected in a Bulk Copper/Molybdenum/Gold/Silver Concentrate.

	Concentrate Recovery	Bulk Concentrate Grade	Copper Concentrate Grade	Moly Concentrate Grade
Copper	90.0%	26.03%	26.5%	0.42%
Molybdenum	72.0%	1.20%	0.27%	54.0%
Gold	82.0%	24.0 g/t	18.4 g/t	-
Silver	72.0%	114 3 g/t	113.2 g/t	-

These values were used for the METSIM mass balance. Due to graphite contamination, the molybdenum balance was prepared assuming that 90% of the molybdenum contained in the Bulk Copper/Molybdenum/Gold/Silver Concentrate would report to the Molybdenum Concentrate at a grade of 54% Mo. These assumptions are currently being tested at G & T laboratory.

16.9 2006 Core Comminution Testing

90 kg of each of the three Schaft Creek resource zones – Liard, West Breccia and Paramount – were sent to Hazen Research in Golden, Colourado for comminution testing. Specifically, Hazen conducted the following tests on each of the composite samples to develop the data required to size the crushing and grinding circuit for Schaft Creek:

- JKTech Drop Weight tests;
- JKTech SMC tests;
- JKTech Abrasion tests (Ai);
- Bond Crushing Index tests (CWi);
- Bond Rod Mill Index tests (RMi);

- Bond Ball Mill Index tests (BWi); and
- Bond Abrasion Index tests.

Table 16.3 Data for Sizing the Crushing and Grinding Circuit for Schaft Creek				
Composite	**Ai**	**RMi**	**BWi**	**CWi**
LZ	0.198	20.1	2139	8.93
WBZ	0.186	19.6	19.8	6.31
PZ	0.380	21.4	20.1	6.71
Avg.	0.186	20.4	20.6	7.32

JKTech Contract Support Services, California, conducted a series of computer simulations using its proprietary software to size the crushing and grinding circuits to treat 65,000 and 100,000 tonnes per day and produce a flotation feed having a size of 80% passing 100 microns. A SABC (Semi-Autogenous Mill + Ball Mill + Pebble Crusher) circuit was selected.

The 100,000 tonne per day case is not included in this study.

Additional Tables and Figures

Table 16.4
PRA Locked-Cycle Test Results on Core Samples Drilled in 2005

Sample Composite Identification	Test Number	Rougher Flotation Feed						Bulk Concentrate Metal Assays							Bulk Concentrate Metal Distributions				
		g/t	g/t	%	%	%	Primary	Regrind	%	g/t	g/t	%	%	%	%	%	%	%	%
		Au	Ag	Cu	Mo	S	Microns	Microns	Wt.	Au	Ag	Cu	Mo	S	Au	Ag	Cu	Mo	S
PIT	**F39LC**	0.30	2.8	0.457	0.026	0.59	132	16	1.2	15.93	112.1	25.54	1.359	30.36	78.40	60.60	84.80	78.40	78.50
MLS	**F40LC**	0.27	1.8	0.435	0.017	0.55	145	19	1.4	15.56	81.4	25.38	0.948	29.22	84.50	66.30	84.20	79.60	77.40
WBZ	**F41LC**	0.19	2.3	0.421	0.027	0.62	147		1.4	10.30	122.1	24.60	1.547	30.48	78.00	74.80	82.70	79.80	69.40
WLZ	**F42LC**	0.33	1.5	0.355	0.015	0.36	166	15	0.7	33.44	129.6	34.21	1.240	28.83	75.90	62.50	71.80	61.10	59.70
NLZ	**F43LC**	0.29	1.9	0.344	0.022	0.36	165	14	0.8	27.43	149.0	32.08	1.912	28.98	76.10	63.10	76.20	71.80	65.30

Table 16.5
PRA Moly Separation Test Data on Core Samples Drilled in 2005

		Bulk Concentrate Assay				Moly Concentrate Assay								Moly Concentrate Distribution			
		%	%	%	%	%	ppm	%	%	%	%	%	%	%	%	%	%
Sample	**Test No.**	**Cu**	**Mo**	**Fe**	**S**	**Wt.**	**Re**	**C**	**Cu**	**Mo**	**Fe**	**S**	**C**	**Cu**	**Mo**	**Fe**	**S**
PPF1	**F70**	21.33	1.71	22.90	24.89	4.84		30.40	1.71	18.12	8.35	18.68		0.36	53.97	1.69	3.62
PPF1	**F74**	21.33	1.71	22.90	24.89	8.48		17.75	3.28	6.81	30.33	51.81	63.72	1.26	39.60	11.77	16.71
PPF1	**F75**	21.33	1.71	22.90	24.89	3.75	1551 (1)	40.60	1.30	18.44	4.47	19.40	67.27	0.21	52.74	0.71	3.04
PPF1	**F76**	21.33	1.71	22.90	24.89	0.13		59.73	0.88	9.17	3.14	11.25	2.42	0.01	0.49	0.02	0.06
PPF1	**F77**	21.33	1.71	22.90	24.89	2.86		30.43	0.53	9.88	23.35	32.43	38.25	0.07	18.07	2.97	3.86
	Avg					4.01		35.78	1.54	12.48	13.93	26.71	42.92	0.38	32.97	3.43	5.46

Note 1: This assay was by ICP and Mass Spectrometrey is considered to be a more accurate method
The assay using Mass Spectrometry was 181.4 ppm. This is considered to be a reliable value.

Table 16.6
PRA Locked-Cycle Test Results on Core Samples Drilled in 2006

Sample		Rougher Flotation Feed						Bulk Concentrate Metal Assays							Bulk Concentrate Metal Distributions				
Composite	PRA	g/t	g/t	%	%	%	Primary	Regrind	%	g/t	g/t	%	%	%	%	%	%	%	%
Identification	Test No.	Au	Ag	Cu	Mo	S	Microns	Microns	Wt.	Au	Ag	Cu	Mo	S	Au	Ag	Cu	Mo	S
Liard Zone	F38LC	0.43	2.90	0.33	0.010	0.43	109	20	1.150	30.29	135.3	25.09	0.649	27.47	80.40	84.10	87.10	75.40	73.16
Paramount Zone	F39LC	0.27	2.10	0.46	0.031	0.43	101	25	1.530	13.80	87.2	28.00	1.787	25.22	78.00	64.10	93.55	88.12	88.74
West Breccia Zone	F40LC	0.26	3.00	0.41	0.020	0.42	107	20	1.450	15.55	75.4	26.50	1.170	25.98	88.30	36.40	93.37	84.83	89.62
Master Composite	F41LC	0.29	2.10	0.38	0.018	0.39	101	20	1.290	18.46	107.3	26.02	0.993	26.34	82.00	66.90	88.16	79.02	87.14
Average		0.32	2.67	0.40	0.020	0.43	105	22	1.377	19.88	99.30	26.53	1.202	26.22	82.23	61.53	91.34	82.78	83.84
Sample		0.43	2.90	0.33	0.010	0.43	109	20	1.150	30.29	135.3	25.09	0.649	27.47	80.40	84.10	87.10	75.40	73.16

Note 1: The "Master Composite" is an equal blend of the three individual Zone Composites
Note 2: The "Average" is the arithmetic average of the three individual Zone Composites

Table 16.7
Summary of PRA Variability Tests on 2005 Core Samples

			Rougher Flotation Feed						Final Flotation Tails					Rougher Concentrate Metal Grade						Rougher Concentrate Metal Distributions					3rd Cleaner Concentrate Metal Assays					3rd Cleaner Concentrate Metal Distributions				
Sample	Test No.	Microns	g/t Au	g/t Ag	%Cu	%Mo	%S		g/t Au	g/t Ag	%Cu	%Mo	%S	Wt. %	g/t Au	g/t Ag	%Cu	%Mo	%S	%Au	%Ag	%Cu	%Mo	%S	%Au	%Ag	%Cu	%Mo	%S	%Au	%Ag	%Cu	%Mo	%S
WB-4	F59 Var. 17	125	0.24	2.90	0.411	0.022	0.20		0.05	0.80	0.052	0.005	0.10	18.70	1.06	12.10	1.969	0.10	0.20	83.00	78.80	89.70	83.40	80.40	12.82	138.30	24.970	1.214	25.40	72.40	64.90	82.00	73.80	68.90
WB-2	F57 Var. 15	129	0.13	1.90	0.268	0.019	0.30		0.04	0.50	0.032	0.005	0.20	19.60	0.51	7.80	1.239	0.08	0.30	75.80	79.10	90.50	80.10	72.50	8.12	128.50	21.046	1.284	31.50	63.50	69.50	81.60	66.70	48.70
WB-5	F60 Var. 18	131	0.20	2.80	0.383	0.022	0.20		0.06	0.80	0.065	0.004	0.20	18.70	0.78	11.40	0.001	0.10	0.20	75.00	77.90	86.30	83.40	88.30	9.20	127.60	22.013	1.239	24.60	63.70	62.70	77.60	77.70	79.30
WB-1	F56 Var. 14	132	0.16	1.00	0.171	0.029	0.80		0.02	0.50	0.019	0.005	0.10	16.60	0.85	3.80	0.933	0.15	1.10	89.40	60.20	90.70	85.80	89.50	8.24	32.90	8.861	1.396	40.70	83.70	50.10	82.80	77.90	81.40
LS-1	F44 Var. 1	135	0.05	2.40	0.156	0.007	0.13		0.01	2.00	0.032	0.002	0.05	10.90	0.35	5.60	1.172	0.05	0.43	81.20	25.50	81.90	71.30	85.50	7.15	77.40	24.262	0.887	30.76	66.70	14.30	66.90	54.90	44.20
WB-3	F58 Var. 16	140	0.18	1.90	0.340	0.019	0.30		0.05	0.50	0.050	0.005	0.20	18.70	0.73	7.80	1.598	0.08	0.30	77.10	78.20	88.00	80.40	80.30	8.88	90.00	20.205	1.012	31.40	67.70	65.20	80.30	71.60	68.50
LS-4	F47 Var. 4	142	0.27	2.00	0.441	0.017	0.30		0.05	0.80	0.070	0.003	0.10	11.70	1.97	11.30	3.235	0.12	0.30	83.90	66.60	86.10	85.20	86.50	19.00	90.20	29.778	1.173	30.20	77.00	50.60	75.20	78.30	64.10
LS-6	F49-2 Var. 7	143	0.53	2.40	0.860	0.033	0.40		0.10	0.50	0.071	0.004	0.10	24.30	1.88	8.40	3.312	0.12	0.40	85.80	84.30	93.80	90.40	93.30	14.04	59.30	24.853	0.949	26.50	77.70	72.40	85.20	84.60	85.90
WL-6	F67 Var. 25	143	0.37	5.20	0.893	0.047	0.10		0.04	0.50	0.032	0.005	0.20	24.10	1.41	19.20	3.416	0.17	0.10	91.60	89.00	92.20	87.60	92.40	13.16	157.90	29.407	1.433	23.00	91.80	78.50	85.00	78.90	84.40
WL-2	F63 Var. 21	144	0.17	1.20	0.256	0.016	0.20		0.03	0.50	0.025	0.003	0.20	18.60	0.76	4.10	1.265	0.07	0.30	85.30	65.00	92.10	85.80	84.50	15.90	83.80	27.161	1.477	23.80	73.50	55.20	81.50	76.20	69.60
LS-2	F45 Var. 2	145	0.10	1.90	0.285	0.010	0.22		0.01	1.30	0.049	0.003	0.07	10.90	0.85	7.40	2.211	0.07	0.44	91.30	42.10	84.70	76.80	85.10	10.90	61.60	27.375	0.899	30.40	81.50	24.40	73.40	67.40	55.50
LS-3	F46 Var. 3	149	0.22	1.60	0.328	0.012	0.30		0.03	0.80	0.060	0.003	0.30	13.50	1.45	7.20	2.046	0.07	0.40	88.30	60.00	84.10	80.90	83.50	20.36	73.60	26.104	0.937	29.20	81.00	40.00	70.00	71.50	54.40
LS-6	F49-1 Var. 6	150	0.55	3.00	0.860	0.028	0.90		0.06	0.80	0.087	0.004	0.10	21.80	2.22	11.00	3.635	0.12	3.90	88.60	79.30	92.10	89.00	92.40	15.20	65.90	24.267	0.811	26.50	80.10	62.90	81.40	83.10	82.70
WL-4	F65 Var. 23	152	0.27	2.10	0.452	0.025	0.20		0.04	0.80	0.056	0.003	0.10	17.40	1.33	8.60	2.329	0.13	0.30	87.60	70.70	89.80	89.30	88.00	13.16	82.00	23.296	1.320	23.60	79.00	61.80	82.20	84.00	77.60
LS-5	F47 Var. 5	153	0.60	3.00	0.747	0.028	0.20		0.01	0.50	0.059	0.003	0.10	29.20	2.02	9.00	2.410	0.09	0.10	99.30	88.20	94.40	92.50	94.00	21.66	88.30	25.004	0.962	24.10	92.00	74.00	83.90	85.10	84.80
WL-1	F62 VR. 20	157	0.33	1.00	0.146	0.004	0.30		0.03	0.50	0.018	0.002	0.10	18.40	1.68	3.10	0.709	0.01	0.50	92.70	58.70	89.70	59.00	80.80	38.00	53.30	15.547	0.251	23.20	86.00	40.80	80.80	44.40	51.70
LN-1	F50 Var. 8	158	0.22	1.10	0.152	0.014	0.10		0.07	0.50	0.023	0.003		17.60	0.95	3.80	0.758	0.07	0.10	74.30	62.00	87.70	84.90	80.30	36.93	139.30	28.670	2.555	21.20	63.60	49.70	73.00	72.30	65.60
WL-5	F66 Var. 24	159	0.31	2.70	0.606	0.028	0.20		0.04	0.50	0.055	0.004	0.10	20.40	1.36	11.50	2.753	0.12	0.20	89.70	85.50	92.90	89.60	91.50	13.16	101.20	24.083	1.141	21.60	89.70	77.60	83.90	85.00	81.30
WB-6	F61 Var. 19	163	0.21	3.40	1.253	0.074	0.30		0.03	0.80	0.090	0.006	0.10	25.70	0.74	11.20	4.615	0.27	0.40	89.50	83.80	94.70	93.90	92.00	4.24	61.80	26.358	1.611	23.20	83.70	75.40	88.10	91.00	87.10
LN-3	F52 Var. 10	170	0.31	1.60	0.349	0.025	0.40		0.05	0.50	0.036	0.003	0.10	16.80	1.61	7.40	1.897	0.13	1.90	86.60	74.80	91.30	88.60	84.30	24.66	101.60	28.966	1.996	28.10	78.90	61.30	82.70	79.90	75.40
LN-4	F53 Var. 11	176	0.56	2.50	0.466	0.036	0.20		0.09	0.80	0.049	0.004	0.10	15.90	3.01	11.60	2.664	0.21	0.20	86.40	74.50	91.20	90.90	87.10	35.80	119.70	30.187	2.417	28.20	80.70	60.60	81.20	83.60	77.40
LN-5	F54 Var. 12	179	0.80	3.50	0.642	0.032	0.10		0.14	0.80	0.057	0.003	0.10	17.00	4.03	17.00	3.507	0.17	0.10	85.50	82.20	92.70	91.30	88.90	41.40	159.60	35.197	1.793	23.50	78.70	69.40	83.40	86.60	79.40
LN-6	F55 Var. 13	179	0.99	4.10	0.783	0.064	0.10		0.14	0.80	0.074	0.003	0.10	16.70	5.27	20.60	4.327	0.37	0.10	88.30	84.60	92.10	95.60	88.50	47.30	174.50	38.036	3.419	23.80	81.60	73.80	83.40	91.90	80.10
LN-2	F51 Var. 9	193	0.22	2.50	0.258	0.021	0.10		0.04	0.80	0.028	0.003	0.10	14.40	1.30	12.90	1.626	0.13	0.20	84.60	74.40	90.80	88.80	85.40	19.28	188.60	23.824	1.929	30.30	77.70	67.50	82.70	80.70	77.20
WL-3	F64 Var. 22	200	0.22	1.60	0.332	0.016	0.20		0.03	0.50	0.038	0.002	0.10	16.00	1.24	6.80	1.874	0.09	0.20	88.80	72.30	90.50	88.20	84.40	18.20	100.90	27.425	1.299	23.60	80.40	65.90	81.90	82.40	72.30

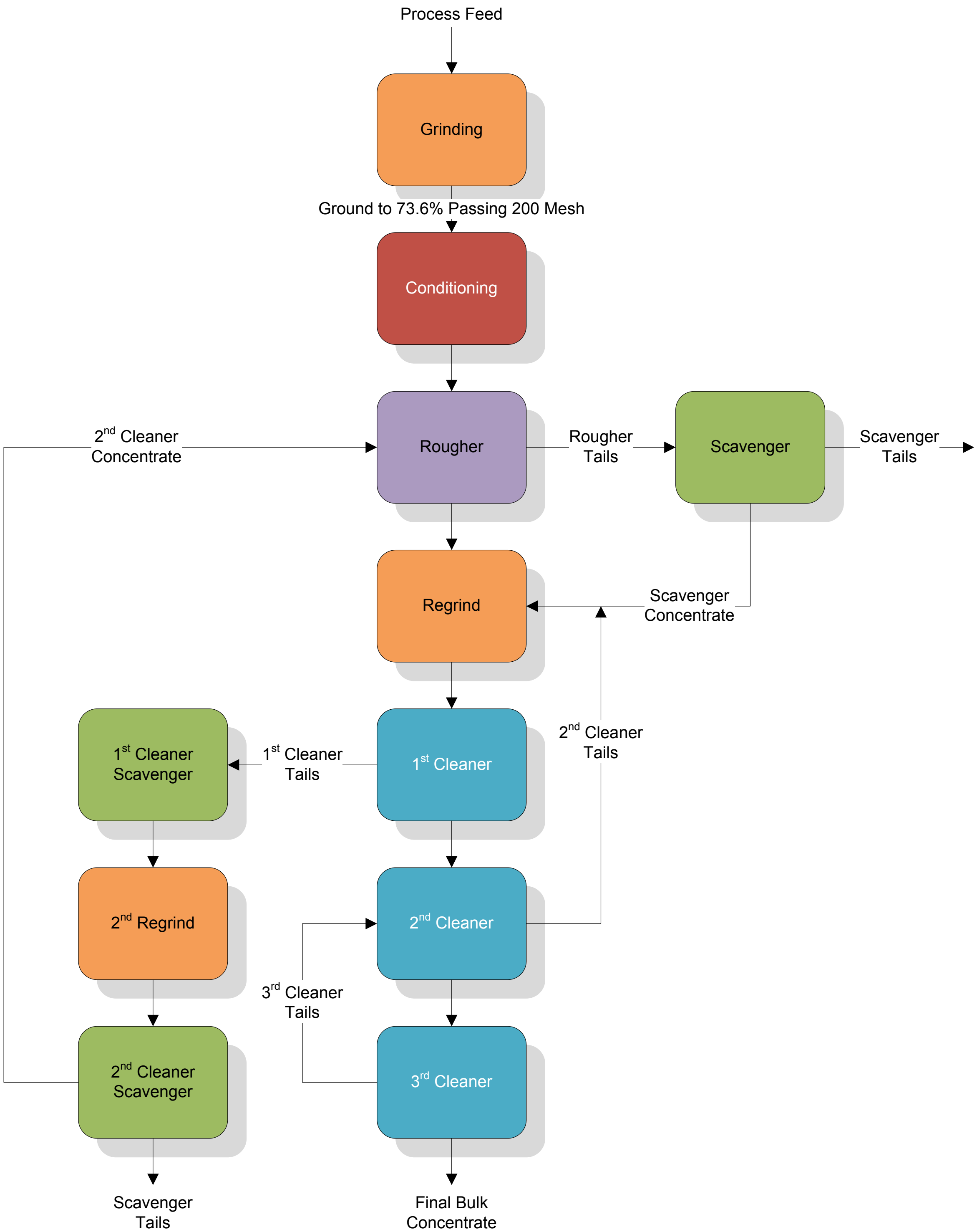


Figure 16.1 PRA 2005 Flowsheet for Locked Cycle Tests

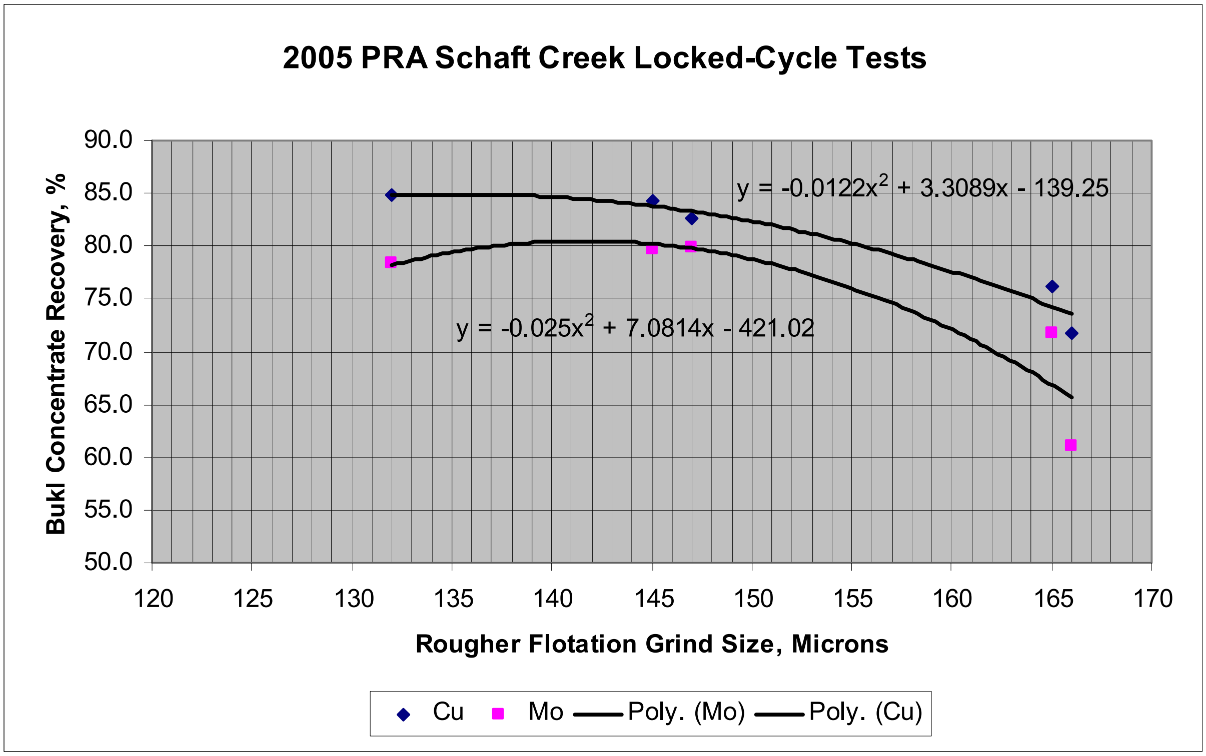


Figure 16.2 Grind Site vs. Recovery - Poly

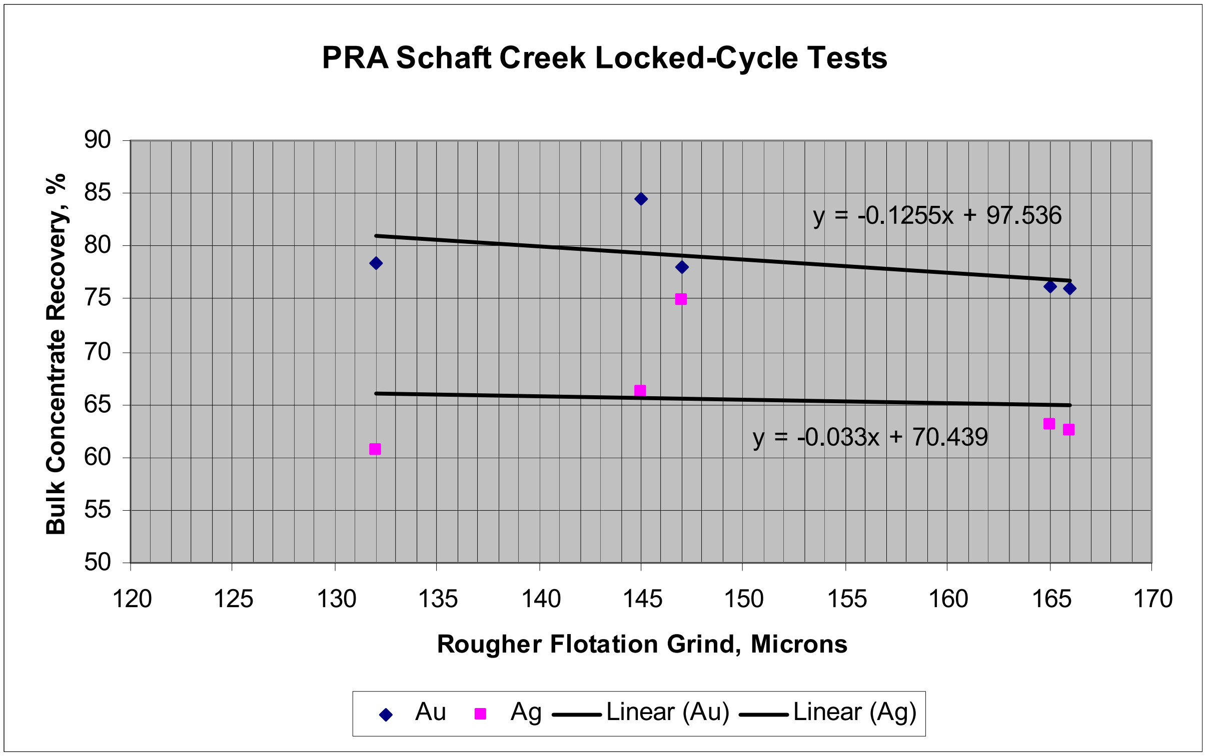


Figure 16.3 Grind Site vs. Recovery - Linear

PRA

FLOTATION TEST FLOWSHEET

Client: Copper Fox Metals Inc
Test: F75 - PPF1 Cu/Mo separation
Sample: Cut of Cu-Mo Concentrate from PPF1

Date: 31-May-07
Project: 0603303

Objective: Cu-Mo separation test on ~1.5kg of PP1 4th Cl concentrate using NaHS to depress Cu, flotation in inert (N2)atmosphere

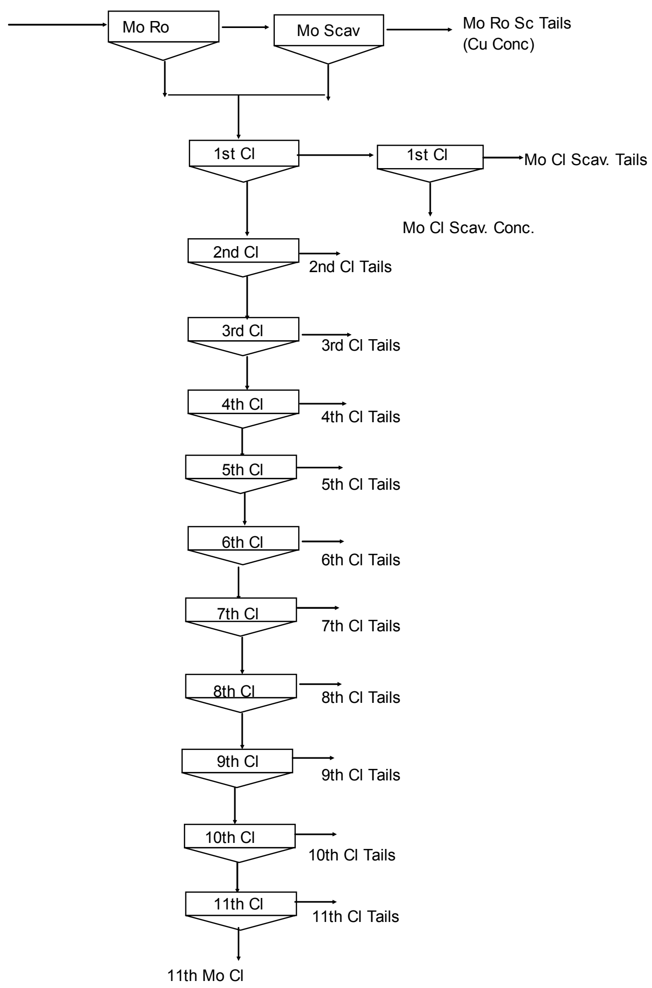


Figure 16.4 Flowsheet for Flotation Testwork

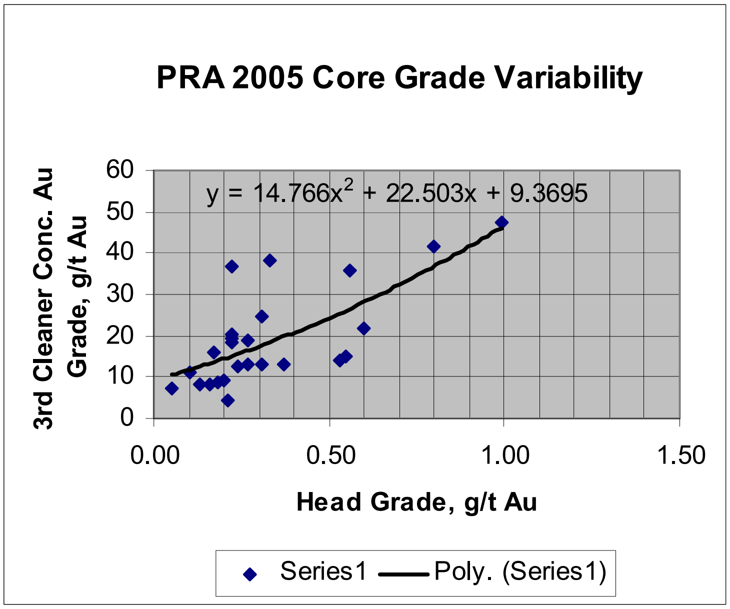


Figure 16.5 Au Head Grade vs. 3rdClnr. Conc. Grade - Poly

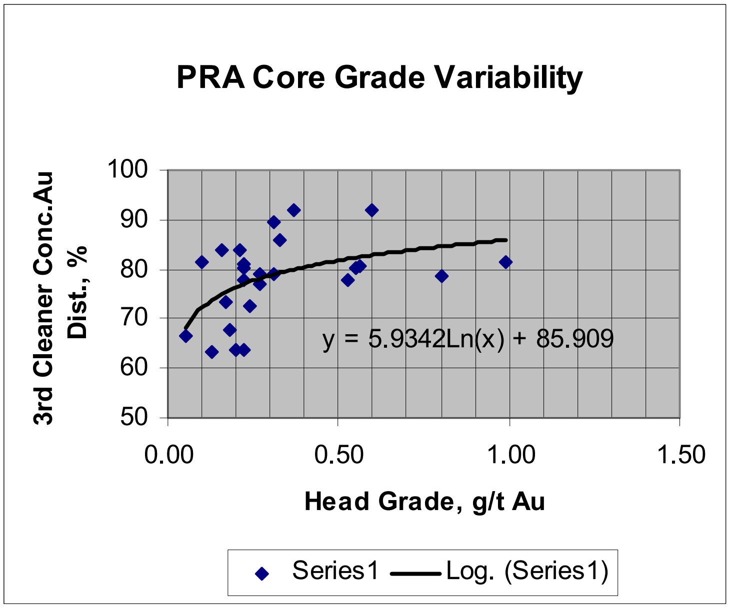


Figure 16.6 Au Head Grade vs. 3rd Clnr. Conc. Grade Distribution - Log

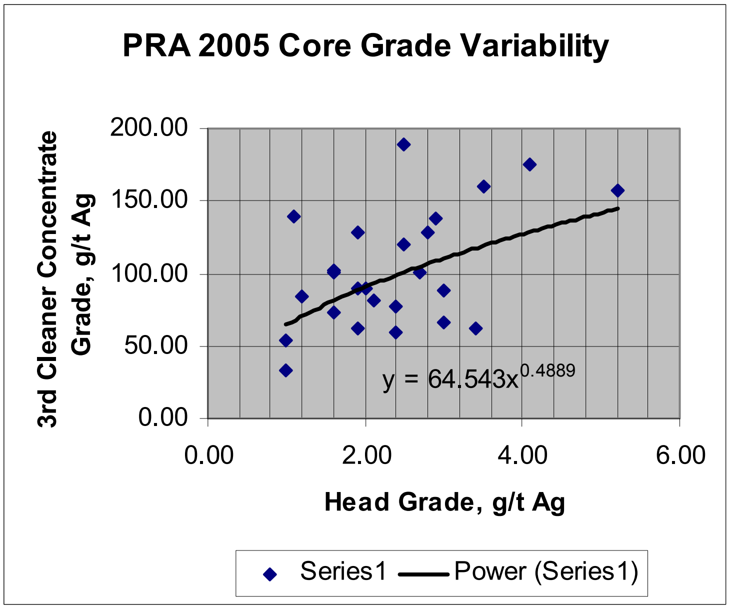


Figure 16.7 Ag Head Grade vs. 3rd Clnr. Conc. Grade - Power

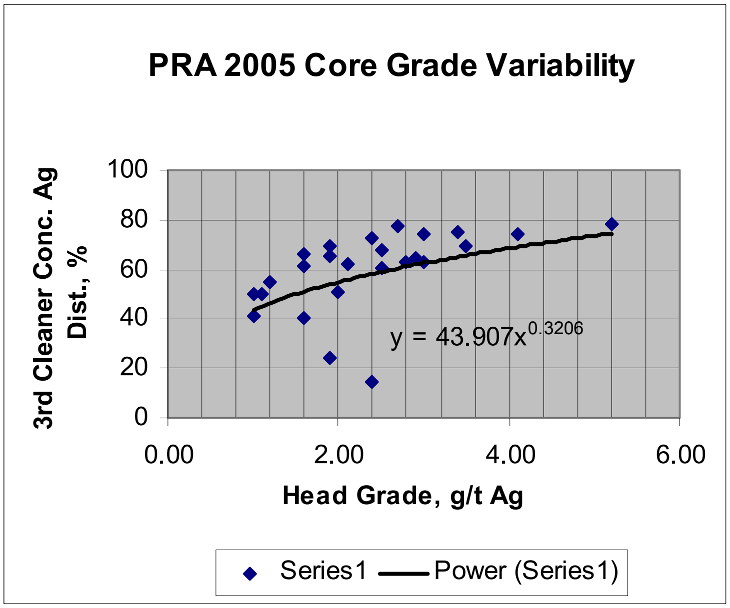


Figure 16.8 Ag Head Grade vs. 3rd Clnr. Conc. Distribution - Power

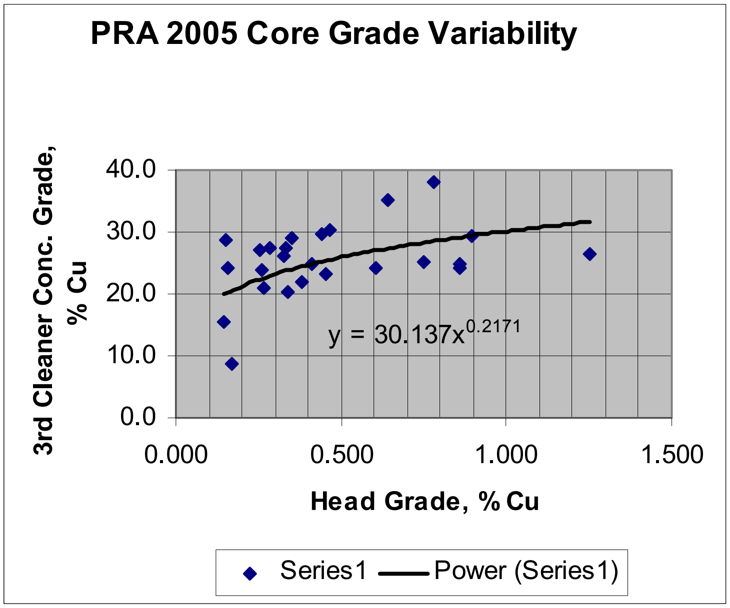


Figure 16.9 Cu Head Grade vs. 3rd Clnr Conc. Grade - Power

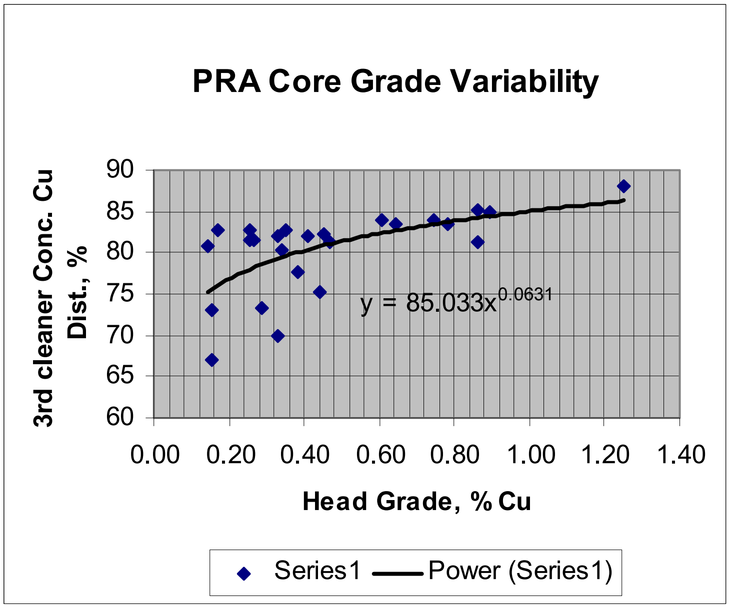


Figure 16.10 Cu Head Grade vs. 3rd Clnr Conc. Distribution - Power

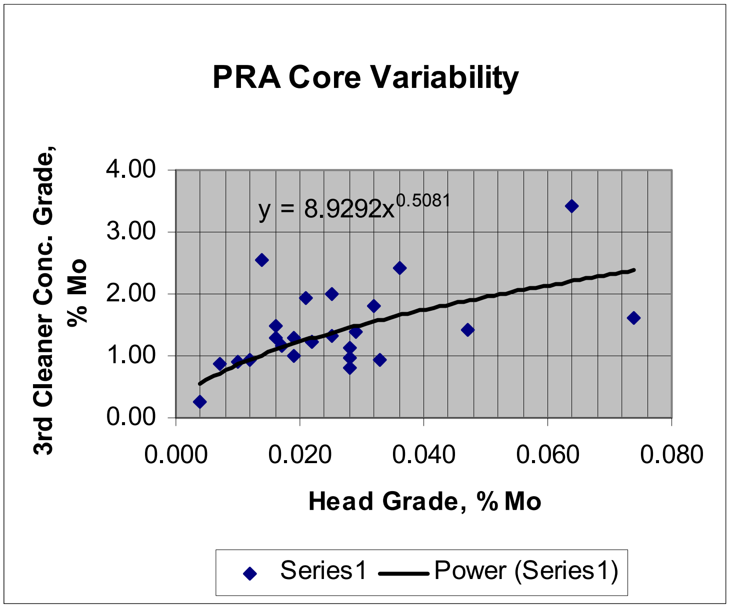


Figure 16.11 Mo Head Grade vs. 3rd Clnr. Conc. Grade - Power

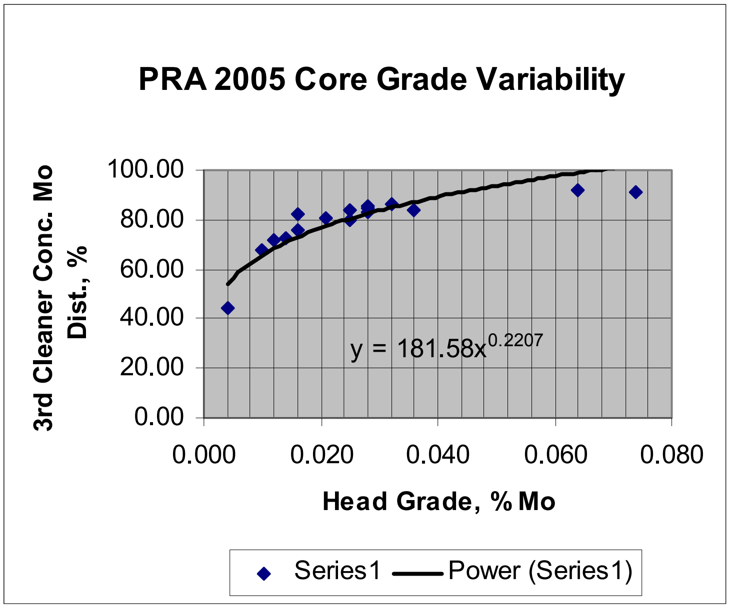


Figure 16.12 Mo Head Grade vs. 3rd Clnr. Conc. Distribution - Power

PRA

FLOTATION TEST FLOWSHEET

Client:	Copper Fox Metals Inc	**Date:**	30-May-07
Test:	F40LC	**Project:**	0701301
Sample:	06-WBZ Composite	**Operator:**	JP

Objective: Locked cycle test on individual zone composite samples to investigate impact of recycling streams on grades and recovery

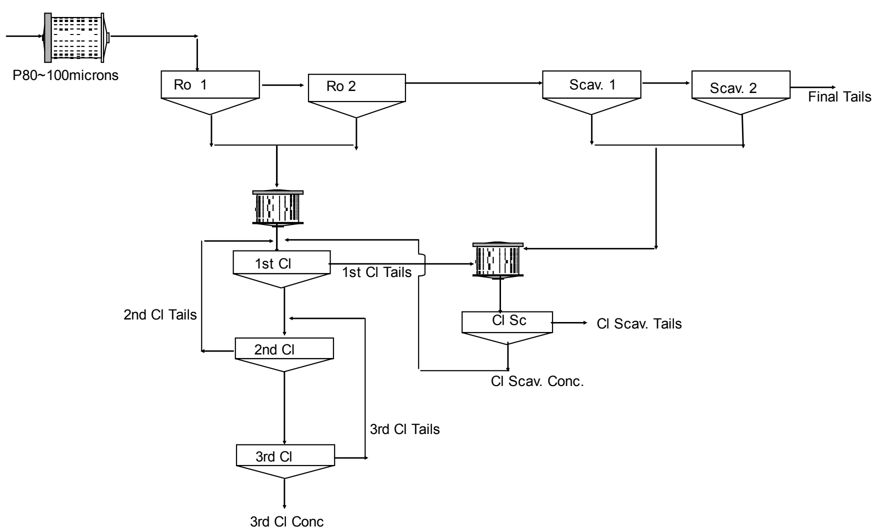


Figure 16.13 Locked Cycle Testing Flowsheet on Zone Composites

17.0 Mineral Resource and Mineral Reserve Estimates

17.1 Mineral Resource Estimate

AGL has reported a mineral resource estimate for the Schaft Creek Deposit which has been classified in the measured, indicated and inferred categories of mineral resources based on the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves. It is recognized that the term “ore” cannot be used unless it is associated with a mineral reserve, however, the word “ore” is used throughout this document to refer only to mineralized material within the resource and mill feed that would be delivered to and processed in the proposed concentrator.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the earth’s crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

This definition suggests that it is necessary to apply an economic cut-off grade even at an early stage of resource estimation. It is the considered opinion of AGL that mineralized material below a copper equivalent cut-of grade of 0.20% at Schaft Creek can not be considered as mineral resources as they are potentially uneconomic. The CIM definitions for resource categories are defined following:

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

An “Indicated Mineral Resource” is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parametres, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such outcrops, trenches, pits, workings, and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A “Measured Mineral Resource” is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parametres, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

A summary of the mineral resources of the Schaft Creek Deposit follows (Table 17.1):

Table 17.1
Schaft Creek Mineral Resource Estimate Summary
≥0.20 % Copper Equivalent Cut-Off

	Tonnes	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq Grade (%)
Measured Mineral Resources	463,526,579	0.30	0.019	0.23	1.55	0.46
Indicated Mineral Resources	929,755,592	0.23	0.019	0.15	1.56	0.36
Measured + Indicated Mineral Resources	1,393,282,171	0.25	0.019	0.18	1.55	0.39
Inferred Mineral Resources	186,838,848	0.14	0.018	0.09	1.61	0.25

The mineral resource estimate has also been reported at various copper equivalent cut-off grades from ≥0.20 % CuEq for different mineral resource categories in Table 17.2-Table 17.5.

Table 17.2
Measured Mineral Resources (≥0.20 CuEq % Cut-Off)

CuEq Cut-Off (%)	Tonnes	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq Grade (%)
0.200	463,526,579	0.30	0.019	0.23	1.55	0.46
0.250	427,185,355	0.32	0.019	0.23	1.48	0.48
0.300	406,104,927	0.33	0.020	0.24	1.48	0.49
0.350	366,510,032	0.34	0.021	0.24	1.50	0.51
0.400	308,920,880	0.36	0.022	0.25	1.53	0.53
0.450	237,822,543	0.37	0.024	0.27	1.59	0.56
0.500	160,958,217	0.40	0.026	0.29	1.70	0.61
0.550	100,681,743	0.43	0.028	0.32	1.85	0.65
0.600	60,312,284	0.46	0.030	0.35	2.07	0.71
0.650	36,461,242	0.51	0.031	0.38	2.38	0.76
0.700	23,605,744	0.54	0.031	0.41	2.65	0.81
0.750	15,877,150	0.58	0.031	0.43	2.92	0.86
0.800	10,557,072	0.60	0.032	0.47	3.16	0.90
0.850	6,933,279	0.63	0.032	0.49	3.39	0.94
0.900	4,246,088	0.66	0.032	0.53	3.64	0.98
0.950	2,520,582	0.68	0.032	0.57	4.02	1.019

Table 17.3 Indicated Mineral Resources (≥0.20 CuEq % Cut-Off)						
CuEq Cut-Off (%)	Tonnes	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq Grade (%)
0.200	929,755,592	0.23	0.019	0.15	1.56	0.36
0.250	619,933,986	0.27	0.023	0.18	1.44	0.43
0.300	508,789,414	0.30	0.024	0.20	1.41	0.46
0.350	416,625,183	0.32	0.026	0.22	1.38	0.50
0.400	326,015,999	0.33	0.028	0.25	1.34	0.53
0.450	234,441,849	0.35	0.030	0.29	1.28	0.57
0.500	161,657,679	0.37	0.032	0.32	1.22	0.61
0.550	108,335,598	0.40	0.034	0.36	1.19	0.66
0.600	74,247,442	0.42	0.036	0.38	1.21	0.69
0.650	51,100,769	0.44	0.036	0.39	1.21	0.73
0.700	31,393,004	0.47	0.037	0.40	1.24	0.76
0.750	13,509,785	0.51	0.038	0.39	1.43	0.80
0.800	5,427,378	0.56	0.042	0.36	1.59	0.85
0.850	2,104,031	0.59	0.046	0.37	1.61	0.89
0.900	627,451	0.63	0.057	0.26	1.92	0.94
0.950	183,848	0.61	0.079	0.19	1.79	0.978

Table 17.4 Measured + Indicated Mineral Resources (≥0.20 CuEq % Cut-Off)						
CuEq Cut-Off (%)	Tonnes	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq Grade (%)
0.200	1,393,282,171	0.25	0.019	0.18	1.55	0.39
0.250	1,047,119,341	0.29	0.022	0.20	1.46	0.45
0.300	914,894,341	0.31	0.022	0.22	1.44	0.48
0.350	783,135,215	0.33	0.023	0.23	1.44	0.50
0.400	634,936,879	0.34	0.025	0.25	1.43	0.53
0.450	472,264,392	0.36	0.027	0.28	1.44	0.57
0.500	322,615,896	0.39	0.029	0.31	1.46	0.61
0.550	209,017,341	0.41	0.031	0.34	1.51	0.66
0.600	134,559,726	0.44	0.033	0.36	1.59	0.70
0.650	87,562,011	0.47	0.034	0.39	1.69	0.74
0.700	54,998,748	0.50	0.034	0.41	1.84	0.78
0.750	29,386,935	0.55	0.034	0.41	2.24	0.83
0.800	15,984,450	0.59	0.035	0.43	2.62	0.88
0.850	9,037,310	0.62	0.035	0.46	2.97	0.93
0.900	4,873,539	0.66	0.036	0.49	3.42	0.97
0.950	2,704,430	0.68	0.035	0.54	3.87	1.02

Table 17.5 Inferred Mineral Resources (≥0.20 CuEq % Cut-Off)						
CuEq Cut-Off (%)	**Tonnes**	**Cu (%)**	**Mo (%)**	**Au (g/t)**	**Ag (g/t)**	**CuEq Grade (%)**
0.200	186,838,848	0.14	0.018	0.09	1.61	0.25
0.250	75,777,298	0.17	0.026	0.08	1.56	0.30
0.300	19,312,810	0.22	0.034	0.10	1.55	0.39
0.350	8,321,377	0.29	0.040	0.11	1.49	0.49
0.400	4,381,796	0.41	0.033	0.14	1.70	0.59
0.450	3,096,952	0.49	0.032	0.16	1.91	0.66
0.500	2,546,608	0.53	0.034	0.15	2.08	0.70
0.550	2,419,163	0.53	0.035	0.15	2.11	0.71
0.600	2,065,526	0.55	0.036	0.15	2.06	0.73
0.650	1,405,022	0.58	0.040	0.16	1.99	0.77
0.700	1,065,789	0.60	0.043	0.17	1.96	0.81
0.750	852,113	0.62	0.043	0.15	2.06	0.83
0.800	496,784	0.66	0.044	0.14	2.09	0.87
0.850	379,508	0.69	0.045	0.11	2.21	0.88
0.900	39,039	0.74	0.043	0.11	2.84	0.92
0.950	0	0	0	0	0	0

The distribution of mineral resource by zone is broken down in Table 17.6

Table 17.6 Distribution of Mineral Resource by Zone			
≥0.20% CuEq cutoff	**Measured**	**Indicated**	**Inferred**
West Breccia Zone	1%	6%	6%
Main Zone	73%	42%	1%
Paramount Zone	20%	31%	3%
Low Grade Zone	6%	23%	86%

17.2 Metal Equivalents

A recoverable copper equivalent (CuEq) grade has been estimated for the polymetallic Schaft Creek deposit at the request of Copper Fox Metals Inc. Form 43-101F1 states that:

"When the grade for a polymetallic mineral resource or mineral reserve is reported as a metal equivalent, report the individual grades of each metal, and consider and report the recoveries, refinery costs and all other relevant conversion factors in addition to metal prices and the date and sources of such prices."

Information or data regarding recoveries, mining costs, treatment charges, refining costs, etc. does not, generally, become available until a project has been the subject of at least a scoping level study or, more generally, a prefeasibility assessment. While AGL accepts that metal equivalents can be a useful tool for the mining professional in assessing the comparative merits of different projects, the reader is cautioned that metal equivalent grades calculated as part of a resource assessment can be misleading unless all of the relevant data used in the calculations are fully understood and that the calculations are on an equivalent basis. AGL has reported all of the individual metal grades.

Metal price data used in the recoverable copper equivalent calculation have been provided by Copper Fox. Metal recoveries are preliminary estimates from metallurgical testing forming part of the on-going Schaft Creek Preliminary Economic Assessment (PEA).

The formula used to estimate recoverable copper equivalent grades for the Schaft Creek deposit is as follows:

$$CuEq\% = \frac{\left(Cu\%*10*\frac{lb}{kg}*P_{Cu}*R_{Cu}\right)+\left(Mo\%*10*\frac{lb}{kg}*P_{Mo}*R_{Mo}\right)+\left(M_{Au}\%*\frac{ozt}{g}*P_{Au}*R_{Au}\right)+\left(M_{Ag}\%*\frac{ozt}{g}*P_{Ag}*R_{Ag}\right)}{\left(10*P_{Cu}*\frac{lb}{kg}\right)}$$

Variable		Description		Value (For This Report)
CuEq%		Copper Equivalence in Percent		*(Variable we are solving for)*
Cu%		Copper Grade in Ore (%)		*Variable* in kg / tonne ore
Mo%		Molybdenum Grade in Ore (%)		*Variable* in kg / tonne ore
M_{Au}		Gold Grade in Ore (g/t)		*Variable*
M_{Ag}		Silver Grade in Ore (g/t)		*Variable*
lb/kg		Pounds (Avoir) per kilogram		2.2046
ozt/g		Ounces (troy) per gram		0.03215
P_{Cu}		Market price of Copper Metal		\$1.50 / lb
P_{Mo}		Market price of Molybdenum Metal		\$10 / lb
P_{Au}		Market price of Gold Metal		\$550 / ozt
P_{Ag}		Market price of Silver Metal		\$10 / ozt
R_{Cu}		Recovery Rate of Copper from Ore		0.91
R_{Mo}		Recovery Rate of Molybdenum from Ore		0.63
R_{Au}		Recovery Rate of Gold from Ore		0.76
R_{Ag}		Recovery Rate of Silver from Ore		0.80

The formula incorporates the four principal elements of economic interest; copper, molybdenum, gold and silver. The assumptions used in the formula are as follows:

Table 17.7
Assumptions used in the Copper Equivalent Estimation

Metal	Commodity Prices	Metallurgical Recoveries
Copper (Cu)	US$1.50/lb	91%
Molybdenum (Mo)	US$10.00/lb	63%
Gold (Au)	US$550/oz	76%
Silver (Ag)	US$10/oz	80%

The conversion factors used were; 1 kilogram (kg) =2.2046226 avoir pounds (lb) and 1 Troy ounce (ozt) = 31.10348 gram (g).

As an example, using a 0.3% copper equivalent cut-off in the measured mineral resource category, the input grades to the formula would be 0.33% Cu, 0.020% Mo, 0.24 g/t Au and 1.48 g/t Ag yielding a recovered copper equivalent grade of 0.49%.

On May 09, 2007 (filed on SEDAR) Copper Fox Metals Inc. released a resource estimate for Schaft Creek prepared by Associated Geosciences Ltd.. The public disclosure of a resource estimate on a material property where there has been >100% change from a previously reported mineral resource estimate triggers a requirement within National Instrument 43-101 to complete and file an independent technical report in support of the resource estimate within 45 days.

During the preparation of this report a number of errors were identified in the copper equivalent formula including an incorrect conversion factor where a conversion of kilogram to pounds (Troy) was used instead of pounds (Avoir), an incomplete term in the denominator and the misplacement of a bracket. The impact of the error affected the contribution of metal values to the copper equivalent grade.

The formula was subsequently corrected and the geological resource model and methodology independently peer reviewed by Gilles Arseneau, Ph.D., P.Geo., Manager Geology of Wardrop Engineering Inc.

The mineral resources presented in this report have been updated to reflect the corrected formula. Where a final mineral resource estimate supported by a technical report differs from a previously disclosed estimate, NI 43-101 requires that the two estimates be reconciled. While the overall tonnes and grades at a 0% copper equivalent cut-off for the individual elements in all mineral resource categories has not changed there has been a considerable rearrangement of the tonnes and grades assigned at various copper equivalent cut-offs (particularly above a 0.20% CuEq). This has the effect of increasing the tonnage at any particular copper equivalent cut-off while raising the copper equivalent grade.

17.3 Discussion

The low grade zone dominates the mass below a 0.20% CuEq cut-off. In the measured category the majority of material comes from the Main Zone, whereas the majority of material in the Indicated category is obtained from the Main and Paramount Zone above a CuEq cutoff of 1%.

The following figures illustrate the percentages of materials according to resource category:

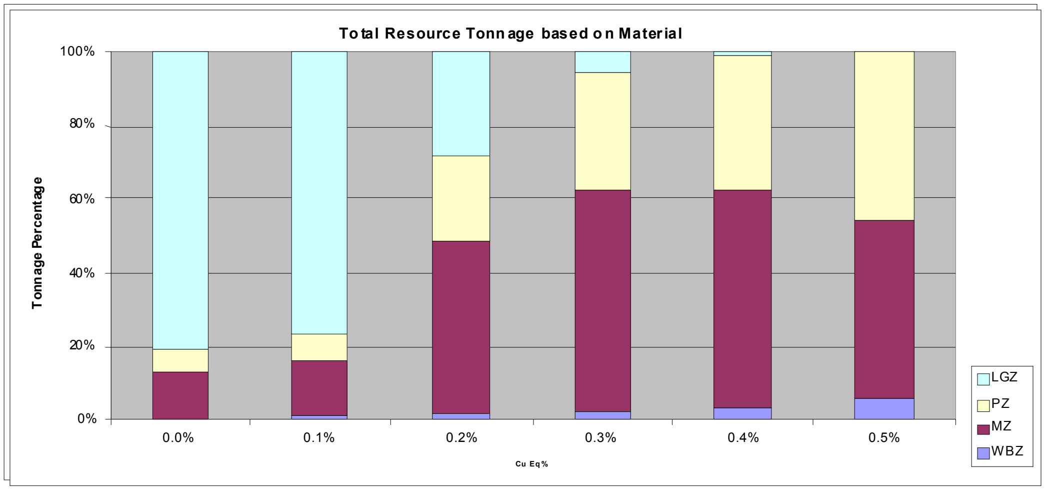


Figure 17.1 Percentages of total resource tonnage based on various zones

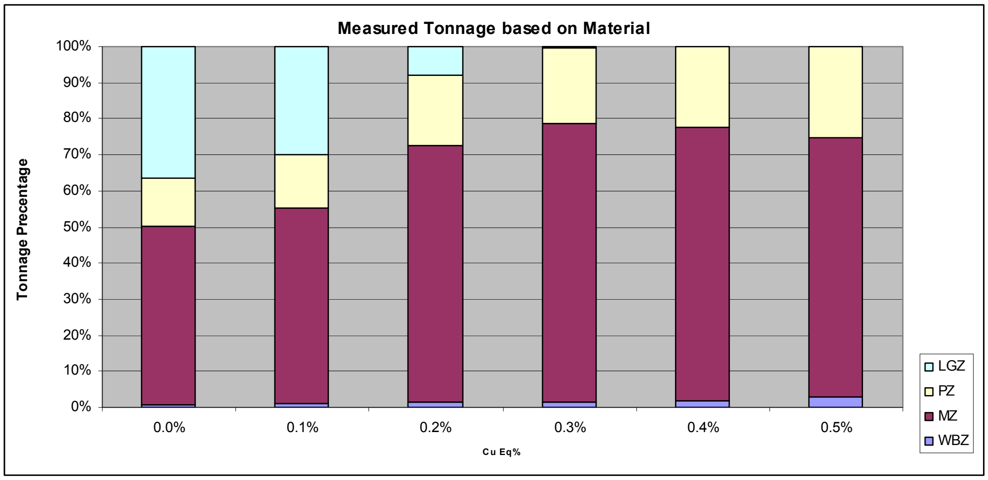


Figure 17.2 Percentages of measured resource tonnage based on various zones

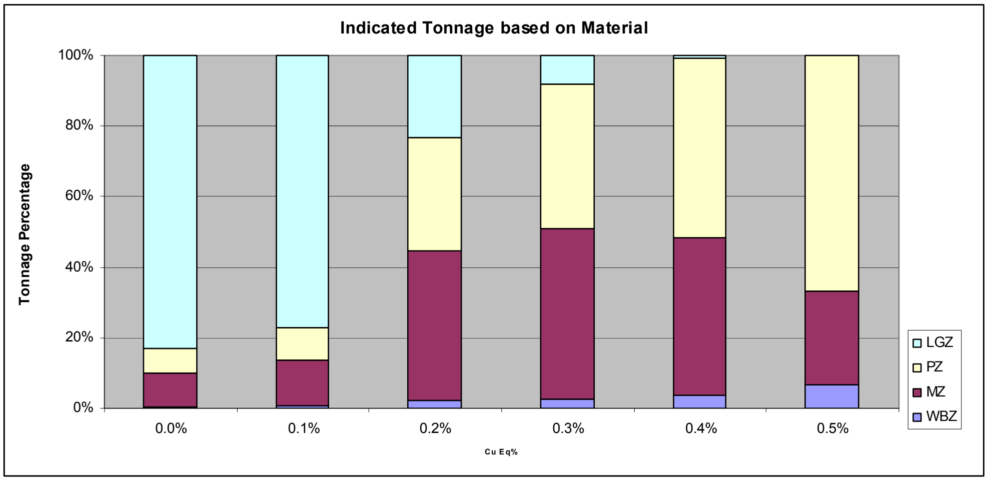


Figure 17.3 Percentages of indicated resource tonnage based on various zones

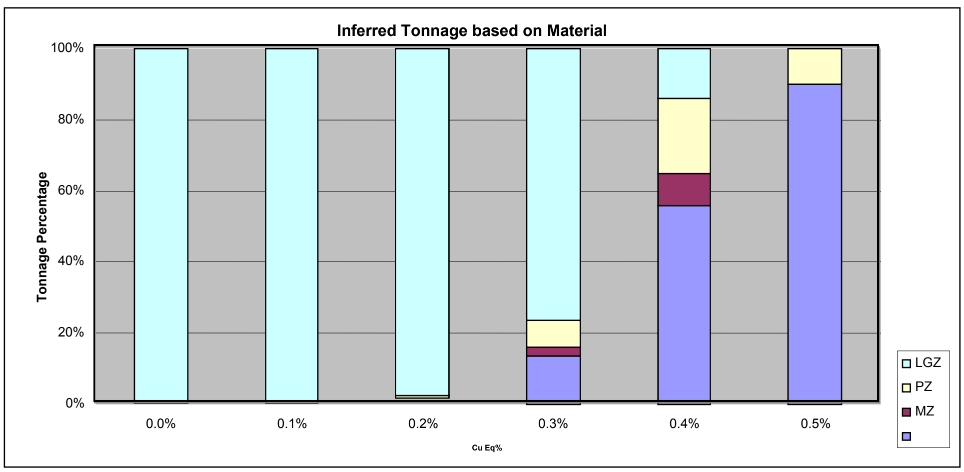


Figure 17.4 Percentages of inferred resource tonnage based on various zones

17.4 Grade-Tonnage Curves

Grade tonnage curves for the measured, indicated, measured & indicated and inferred resources categories are displayed in Figure 17.5 to Figure 17.8

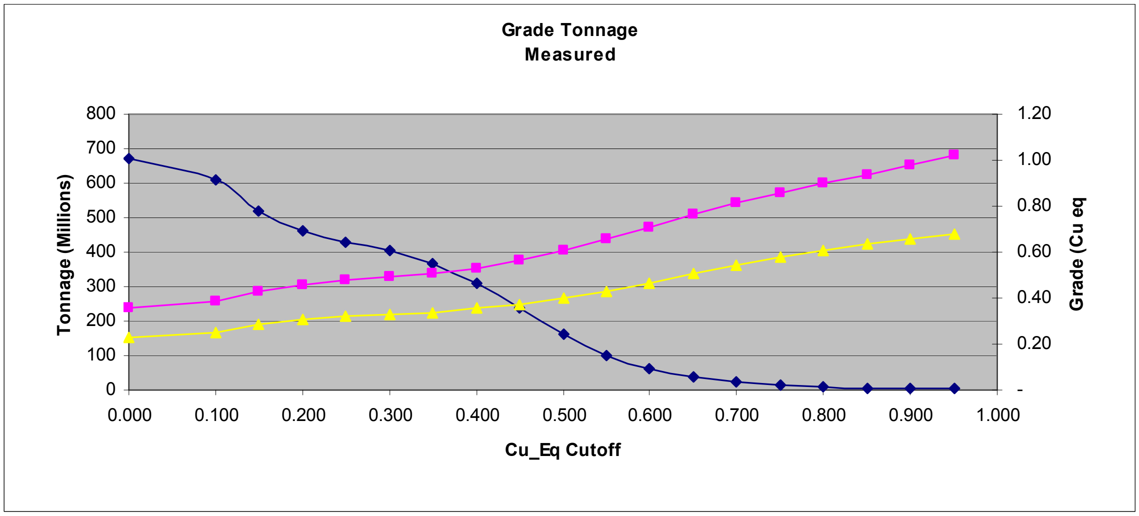


Figure 17.5: CuEq Grade Tonnage curve, measured resource category

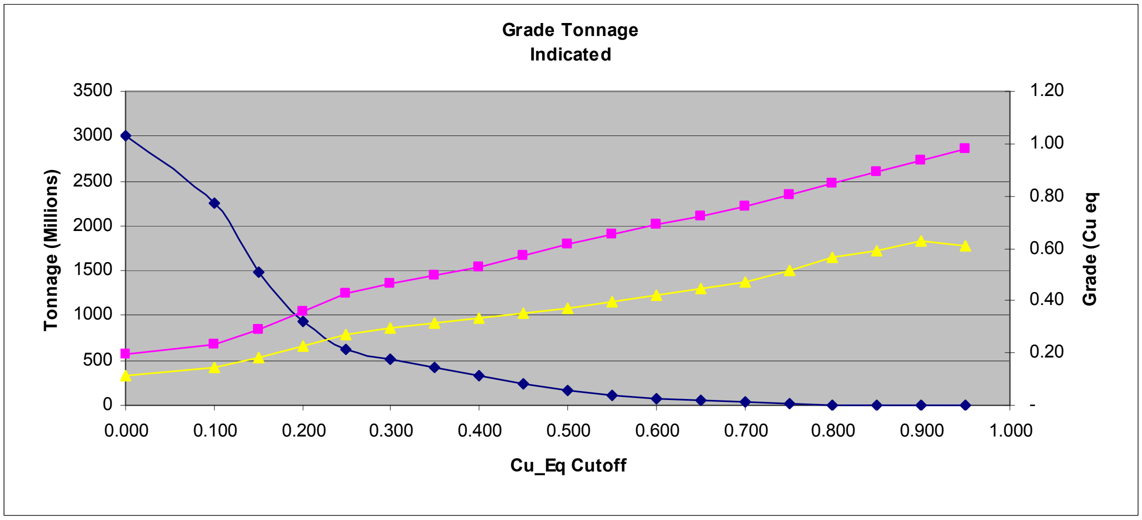


Figure 17.6: CuEq Grade Tonnage curve, indicated resource category

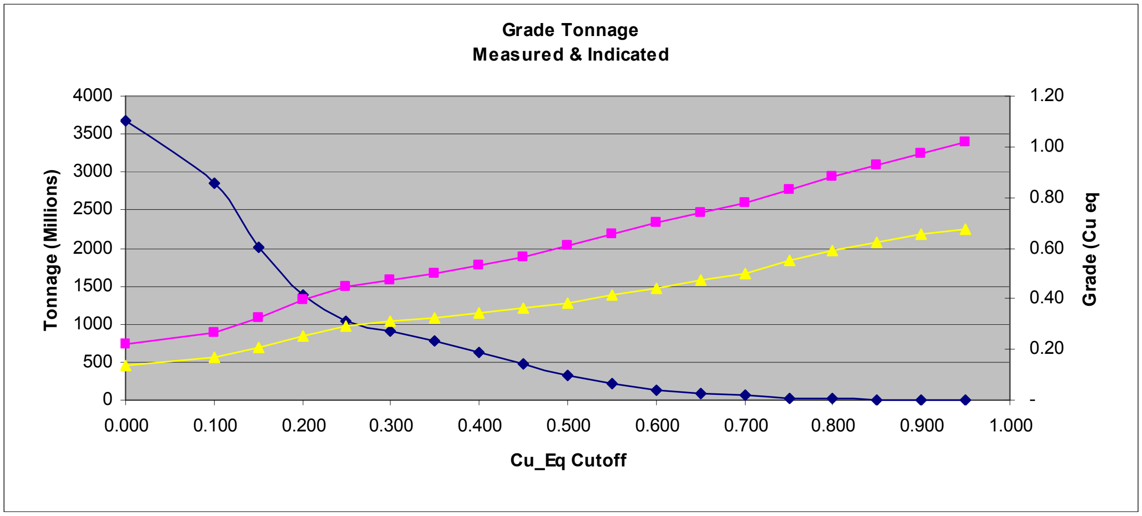


Figure 17.7 CuEq Grade Tonnage curve, MI resource categories

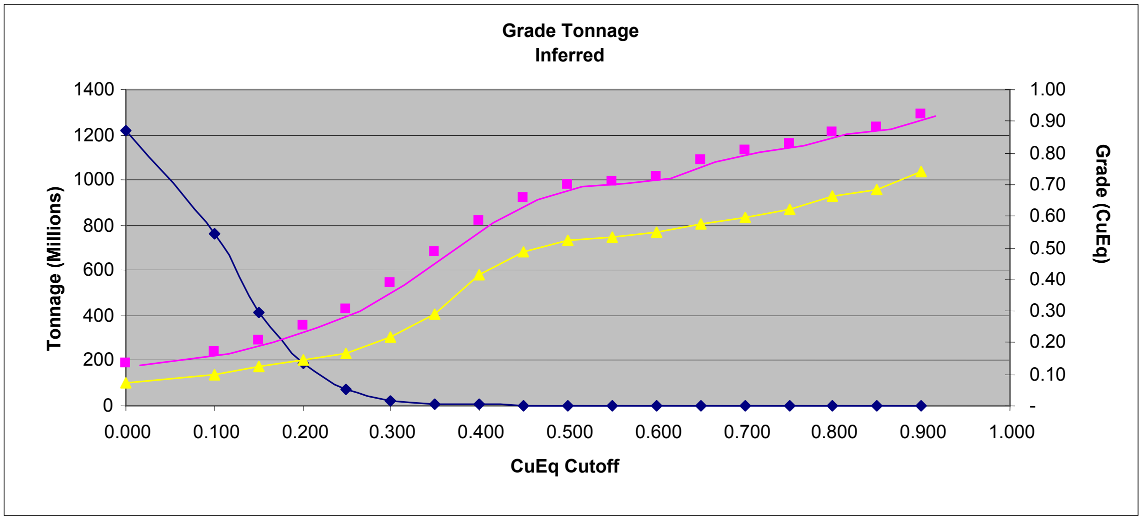


Figure 17.8 CuEq Grade Tonnage curve, inferred resource category

17.5 Copper Cutoff Grade-Tonnage Curves

Grade Tonnage curves at a Cu% cutoff for the various zones are depicted below (Figure 17.9 through Figure 17.13)

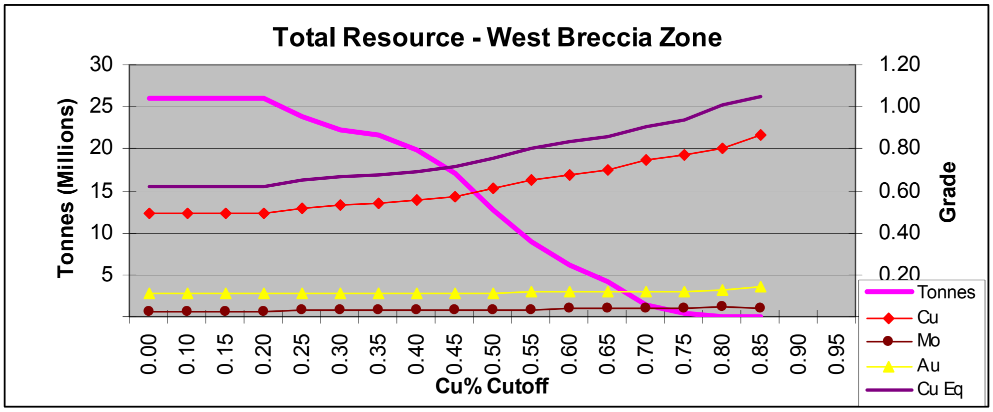


Figure 17.9: Cu% cutoff Grade Tonnage Curve, West Breccia zone

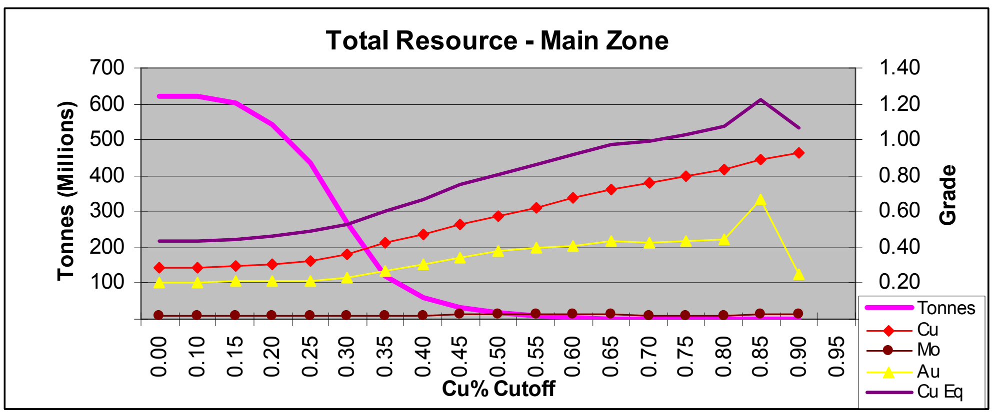


Figure 17.10: Cu% cutoff Grade Tonnage Curve, Main zone

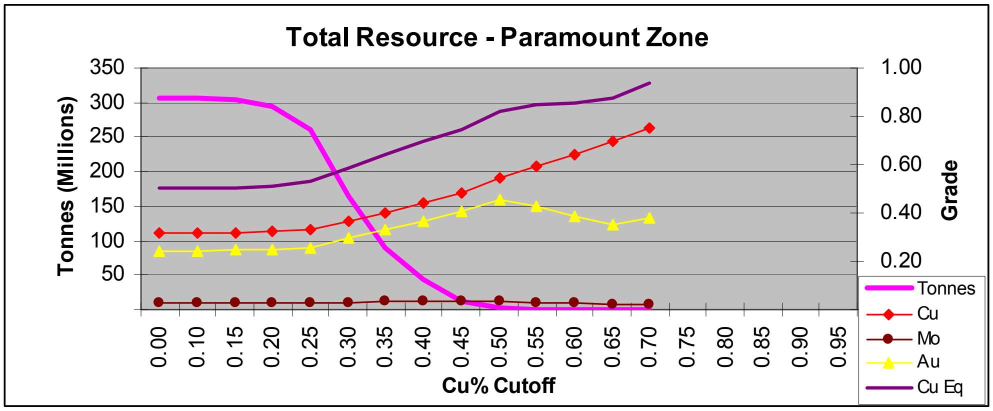


Figure 17.11: Cu% cutoff Grade Tonnage Curve, Paramount zone

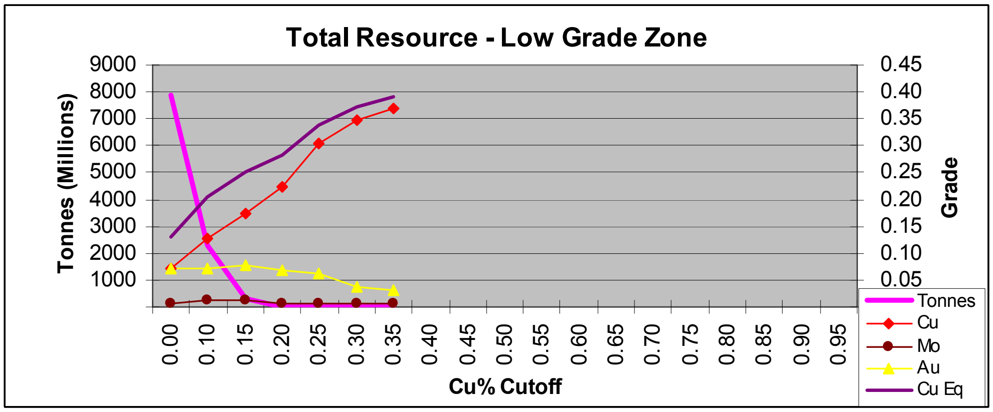


Figure 17.12: Cu% cutoff Grade Tonnage Curve, Low Grade zone

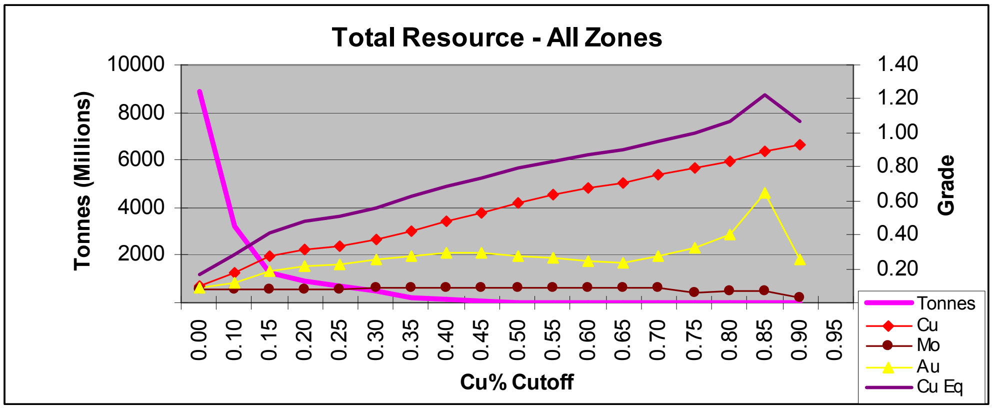


Figure 17.13: Cu% cutoff Grade Tonnage Curve, all zones

17.6 Resource Classification

The resource was classified into measured, indicated and inferred mineral resource categories. This categorization was based on various factors. These factors included the variogram ranges obtained, sample to block estimation distances and number of samples used in the estimation process.

Table 17.8 depicts the process used in the resource classification. This classification was based around the major element Cu%.

Table 17.8
Process used in the Resource Classification Process

Cu% Estimation		Estimation Pass	Number of Samples	Average distance of Samples
West Breccia Zone	Measured	1	10	64
	Indicated	1	5	127
	Inferred	2	5	190.5
Main Zone	Measured	1	10	103
	Indicated	1	5	205
	Inferred	2	5	307.5
Paramount Zone	Measured	1	10	113
	Indicated	1	5	250
	Inferred	2	5	375
Low Grade Zone	Measured	1	10	128
	Indicated	1	5	256
	Inferred	2	5	384

The following figures depict the resource classification for the Schaft Creek deposit. The red depicts the measured resource, green the indicated and blue the inferred resources. The first image depicts the West Breccia, Main and Paramount zones; the next image includes the Waste Zone; and the last the block model illustrates the total model excluding the Air. Similarly, Figure 17.14 through Figure 17.19 depict the interpolated element grades converted to a copper-equivalent value.

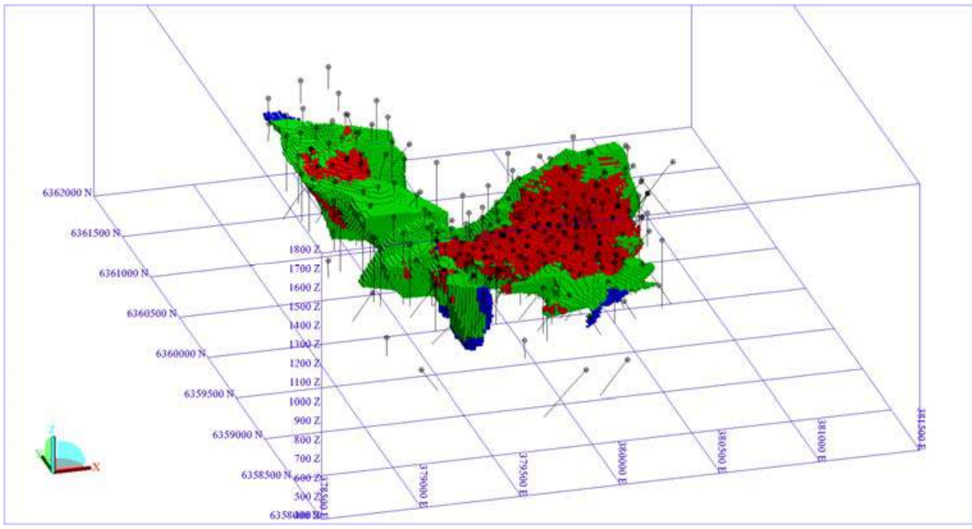


Figure 17.14 Resource Classification of the Three Main Zones

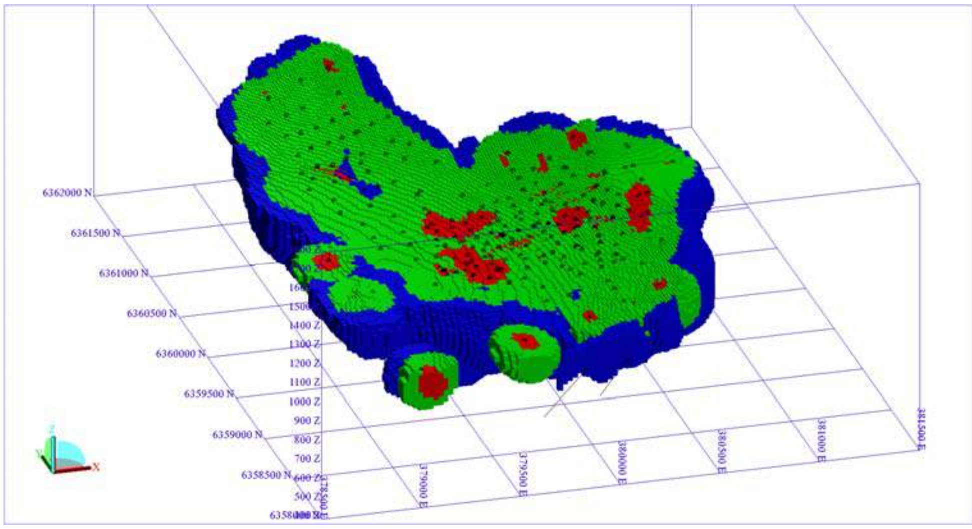


Figure 17.15 Resource Classification of the Three Main Zones, Waste Included

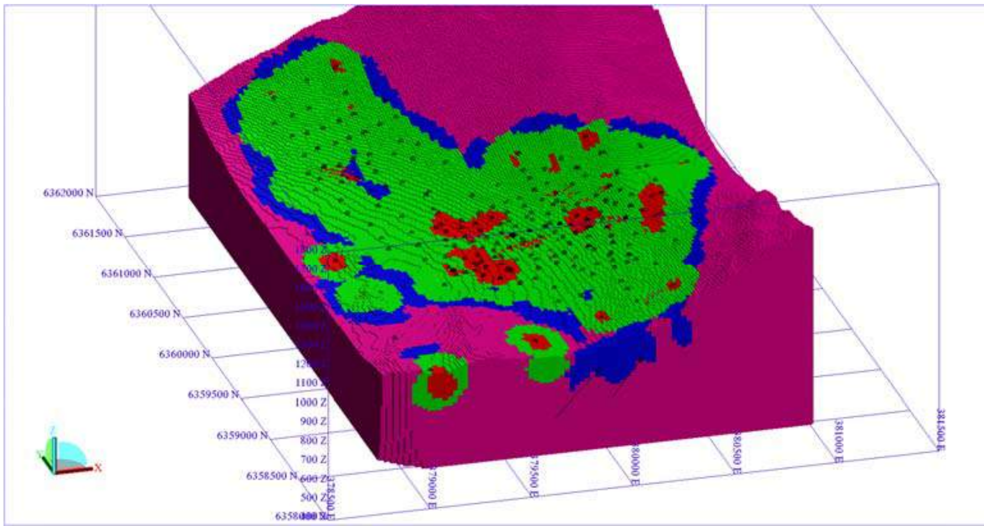


Figure 17.16 **Resource Classification of the three main zones, excluding air**

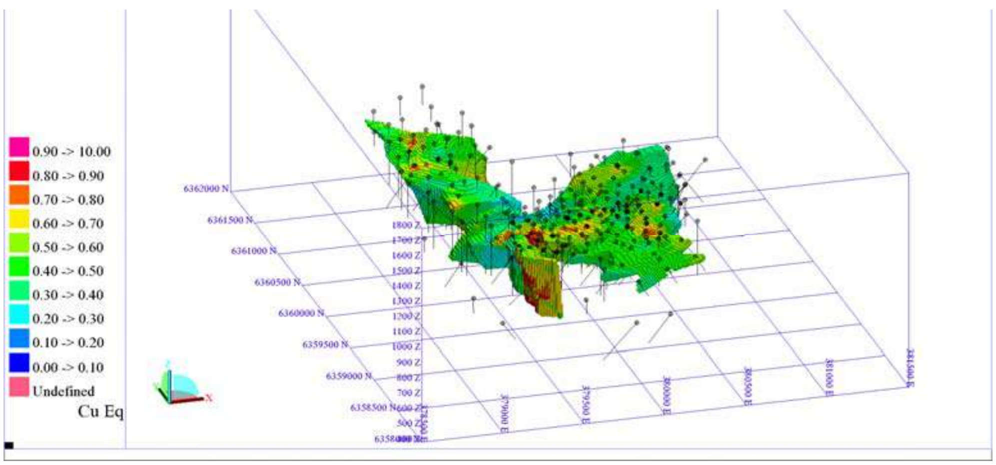


Figure 17.17 Interpolated element grades converted to a Cu equivalent value

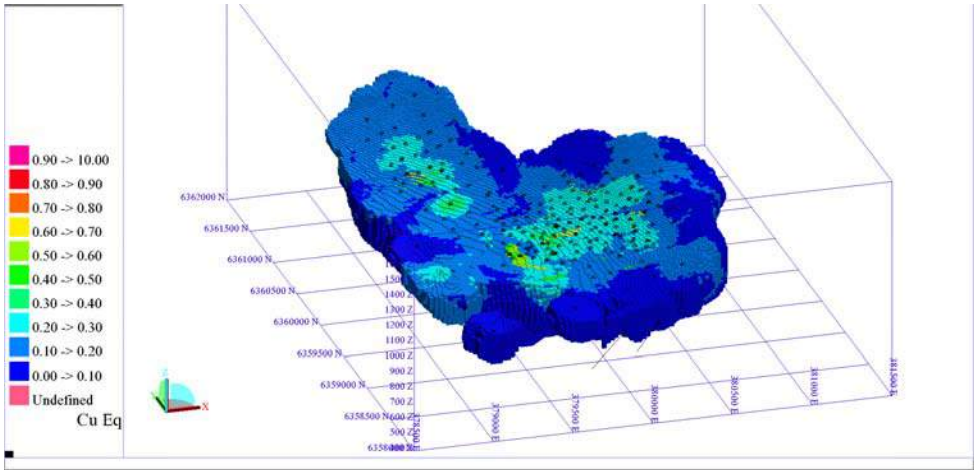


Figure 17.18 Interpolated element grades converted to a Cu equivalent value

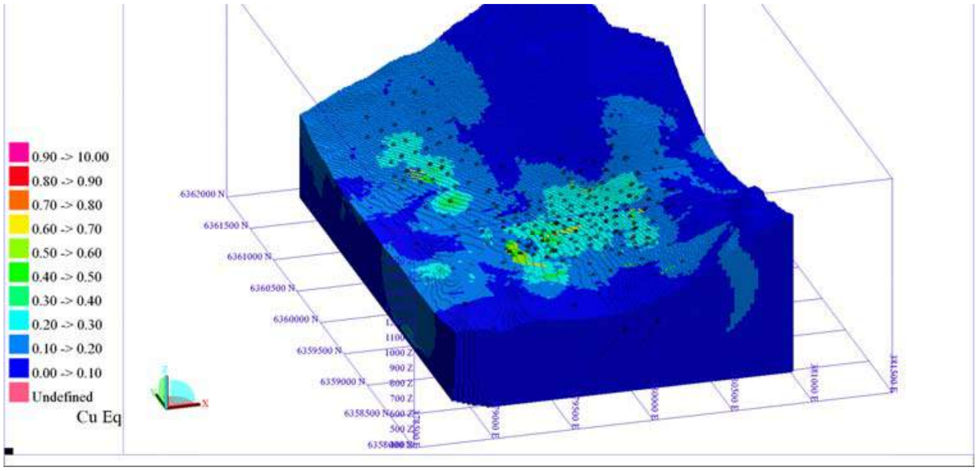


Figure 17.19 Interpolated element grades converted to a Cu equivalent value

17.7 Mineral Reserve Estimates

No mineral reserves are being reported for the property at this time.

18.0 Other Relevant Data

18.1 Permitting

18.1.1 Permits

The Schaft Creek project will require a British Columbia Environmental Assessment Certificate as well as provincial permits, authorizations and licenses to construct and operate the Schaft Creek mine. The project may also require a federal decision on the likelihood of environmental impacts if the Canadian Environmental Assessment Act (CEAA) applies to the Schaft Creek project.

The Schaft Creek project constitutes a reviewable project pursuant to Part 3 of the Reviewable projects Regulations (British Columbia Reg. 370/02) of the British Columbia Environmental Assessment Act (BCEAA). The Schaft Creek project was launched in the BC environmental assessment process on August 14, 2006, with the issuing of Order under Section 10(1)(c) of the BCEAA.

The Canadian environmental assessment process is governed by the Canadian Environmental Assessment Act (CEAA). At this time, it is not know if the CEAA will apply to the Schaft Creek project. CEAA applies when a federal department or agency is required to make a decision on a proposed project. Federal regulatory agencies require specific project details to determine if and how the CEAA will apply.

Upon receipt of the British Columbia Environmental Assessment Certificate, permits, authorizations and licenses will be sought to construct and operate the Schaft Creek mine.

18.1.2 British Columbia Environmental Assessment Process

Once the BCEAA applies, the environmental assessment process that follows includes the collection of environmental and social baseline information, identification of potential project effects, provincial and federal regulatory review, First Nation participation, community consultation and measures for managing potential project effects.

The intent of the BCEAA process is to identify any foreseeable adverse impacts through the project's lifecycle, including: construction, start-up, operation and closure; and to determine ways to eliminate, minimize (mitigate) or compensate identified impacts.

Through the BC environmental assessment process, a Terms of Reference for the EA Application for an EA Certificate will be developed. The Terms of Reference will define the requirements of the EA Application that will be submitted to the Environmental Assessment Office pursuant to an EA Certificate. The EA Certificate provides approval in principle for the project to be developed. Permits, licenses and authorizations will then be sought to begin construction of the Schaft Creek project.

The Schaft Creek EA Application for an EA Certificate is estimated to be submitted for review by forth quarter 2008. The Application review period is approximately six months.

18.1.3 Canadian Environmental Assessment Act

CEAA 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. CEAA is triggered by federal involvement in a project. CEAA applies when a federal department or agency is required to make a decision on a proposed project. Under CEAA's "triggering" provisions, an assessment is required if a federal authority exercises or performs one or more of the following powers, duties or functions relating to a project:

- Proposing a project;
- Granting money or any other form of financial assistance to the proponent;
- Granting an interest in land to enable a project to be carried out;
- Exercising a regulatory duty in relation to a project, such as issuing a permit or license, that is included in the law list prescribed in CEAA's regulations. This includes various federal licenses and authorizations.

If CEAA applies to the Schaft Creek project, it will be triggered by the last bullet item above. The federal agencies can not make a decision on the above triggers until project details are available. It is estimated that this information will be provided to federal regulators by end of the second quarter 2008.

18.1.4 Permits, Licenses and Authorizations

In addition to a BC Environmental Assessment Certificate, the Schaft Creek project will require various permits, licenses and authorizations from both the provincial and federal governments if CEAA applies. The following sections list the major permits, licenses, approvals, consents and material authorizations which are required to occupy, use, construct and operate the proposed Schaft Creek mine. At this time, the lists are not comprehensive as details of the project are still being developed.

The following is a list of provincial permits, licenses and authorizations likely required to develop the Schaft Creek project:

- Permit Approving Work System and Reclamation Program (Mines Act);
- Water License (Water Act);
- Occupant License to Cut (Forest Act);
- Special Use Permit (Forest Act);
- License of Occupation (Land Act);
- Surface Lease (Land Act);
- Right of Way (Land Act);
- Waste Management Permit (Environmental Management Act);
- Camp Operations Permit (Environmental Management Act).

The following is a list of federal approvals and licenses likely required to develop the Schaft Creek project:

- Metal Mining Effluent Regulations (Fisheries Act/Environment Canada);
- Fish Habitat Compensation Agreement (Fisheries Act);
- Section 35(2) Authorization for harmful alteration, disruption or destruction of fish habitat (Fisheries Act);
- Navigable Water: Stream Crossings Authorization (Navigable Waters Protection Act);
- Explosives Factory License (Explosives Act);
- Explosives Magazine License (Explosives Act);
- Ammonium Nitrate Storage Facilities (Canada Transportation Act);
- Radio Licenses (Radio Communication Act).

18.1.5 Environmental and Social Baseline

The environmental and social baseline programs for the Schaft Creek project began in the fall of 2005 and are currently ongoing. The baseline programs are designed to characterize current conditions in the project area. The baseline reports will serve as the basis for the overall environmental and social impact assessment of the Schaft Creek project. The baseline programs cover multiple disciplines and include multiple years of data collection. Baseline studies in 2006 were undertaken for the following disciplines:

- Meteorology;
- Hydrology;
- Wildlife (birds and moose);
- Fisheries and aquatic biology;
- Water quality;
- Archaeology;
- Socio-economics.

The program has been expanded for 2007 to build upon the studies from 2006 and address new project developments:

- Tahltan (traditional) knowledge;
- Vegetation;
- Wetlands;
- Ecosystem mapping;
- Soils;
- Wildlife (ungulates);
- Amphibians;
- Fisheries;
- Aquatic biology;
- Water quality;
- Metal leaching and acid rock drainage;
- Hydrogeology;
- Hydrology;
- Meteorology;
- Air quality;
- Noise;
- Archaeology;

- Country foods;
- Socio-community;
- Navigable waters.

Additional studies will be designed for 2008 as required to complete the EA Application. These programs will be presented to government regulators and the Tahltan Nation for review and comment.

18.2 Environmental

A preliminary environmental assessment study was completed by J.B. Krusche, P.Eng., for the Schaft Creek deposit in 2005. The report recommended several phases of environmental assessment, as outlined in Table 18.1. The phases were intended to provide a guideline for the environmental assessment process.

Schaft Creek environmental baseline studies began in October 2005 and are currently ongoing. Baseline studies completed in 2006 included wildlife (moose, goats and bird studies), water quality, aquatic biology, fisheries, hydrology, meteorology, archaeology and metal leaching and acid rock drainage (ML/ARD). These studies were reviewed by federal and provincial regulators and the Tahltan Nation.

The scope of work for the 2007 baseline studies has increased relative to 2006. The broad scope is aimed at fulfilling requirements of both a federal and provincial environmental assessment process. In addition, the 2007 work includes specific studies requested by the Tahltan Nation. Recently, the 2007 environmental baseline studies were presented and approved by the government and First Nations, these studies include: socio-economics, traditional knowledge, country foods, wetlands, hydrogeology, soils, ecosystem mapping, vegetation, archaeology, human health, fisheries and aquatics, wildlife, hydrology and Metal Leaching / Acid Rock Drainage (ML/ARD).

The baseline studies from 2005 through 2007 will form the basis of the Schaft Creek environmental assessment application. Work on the application begin in 2007 and will continue through 2008. The anticipated completion date of the application for an environmental assessment certificate is fourth quarter 2008.

Table 18.1 Environmental Assessment, Preliminary Phases			
Phase I	Defining Issues of Concern	Preliminary Term of Reference (TOR)	
		Preliminary Valued Ecosystem Components (VEC)	
	Litreature Review	Review past assessments and reports on Schaft Creek for data useful to the EA process	
	LMRP Review	Review of Cassiar Iskut – Stikine Land and Resource Management Plan (LRMP) and LRMP Monitoring report	
	Regulatory Review	Identify preliminary issues of concern	Environmental regulatory acts, regulations, and guides
			First Nations documents
			Non-Governmental Agencies (NGOs) and Public Groups
Phase II	Preliminary Engineering	Siting	
		Timing	
		Tonnage	
		Process	
		Geotechnical and metallurgical	
		ARD assessment	
		Life expectancy	
		Reclamation	
		Access	
Phase III	Identify Project Issues	Compare preliminary engineering results to regulations	
		Review engineering data collection requirements	
		Review environmental baseline information needed	
		Design program to collect environmental and engineering baseline information	
		Regulator review of aquatics and terrestrial field program (i.e. DFO, MEM, WLAP)	
		Review past EA process for key issues to avoid or improve	
		Review concerns of NGOs and Public for issues of concern	
		Design program for open house presentations	
		Organize meetings with regulators and First Nations	
Phase IV	Baseline collection and Preliminary Project Mitigation	Collection of baseline information	
		Study impact potentials	
		Design preliminary mitigative steps	
Phase V		Detailed Mine Planning	
		Establish approximate EA Schedule	
		Compilation of environmental baseline information	
		Summary of impact assessment and mitigation plans	
		First Nations issues summary and mitigation plans	
		Public concerns summary and mitigation plans	
		Established preliminary TOR	
Phase VI	Official EA Process begins – project submittal.		

18.3 Access Road

18.3.1 Introduction

18.3.1.1 *Background Information*

McElhanney Consulting Services Ltd. (McElhanney) was retained by Copper Fox Metals Inc. (CFMI) to complete a prefeasibility study of the proposed access road to the Schaft Creek deposit. Two routes, the Mess Creek route and the Raspberry Pass route, were identified and described in McElhanney's "Scoping Study Report" dated December 2005. The Mess Creek route was chosen for further study in this prefeasibility report. See Figure 18.1 on following page.

An initial investigation was carried out to identify feasible road corridors, which would meet geotechnical and engineering constraints. The assessment was based on reconnaissance level review of air photos, low level flights and TRIM mapping. This prefeasibility study advanced the engineering understanding of the Mess Creek route by field checking earlier assumptions and quantifying road construction parametres.

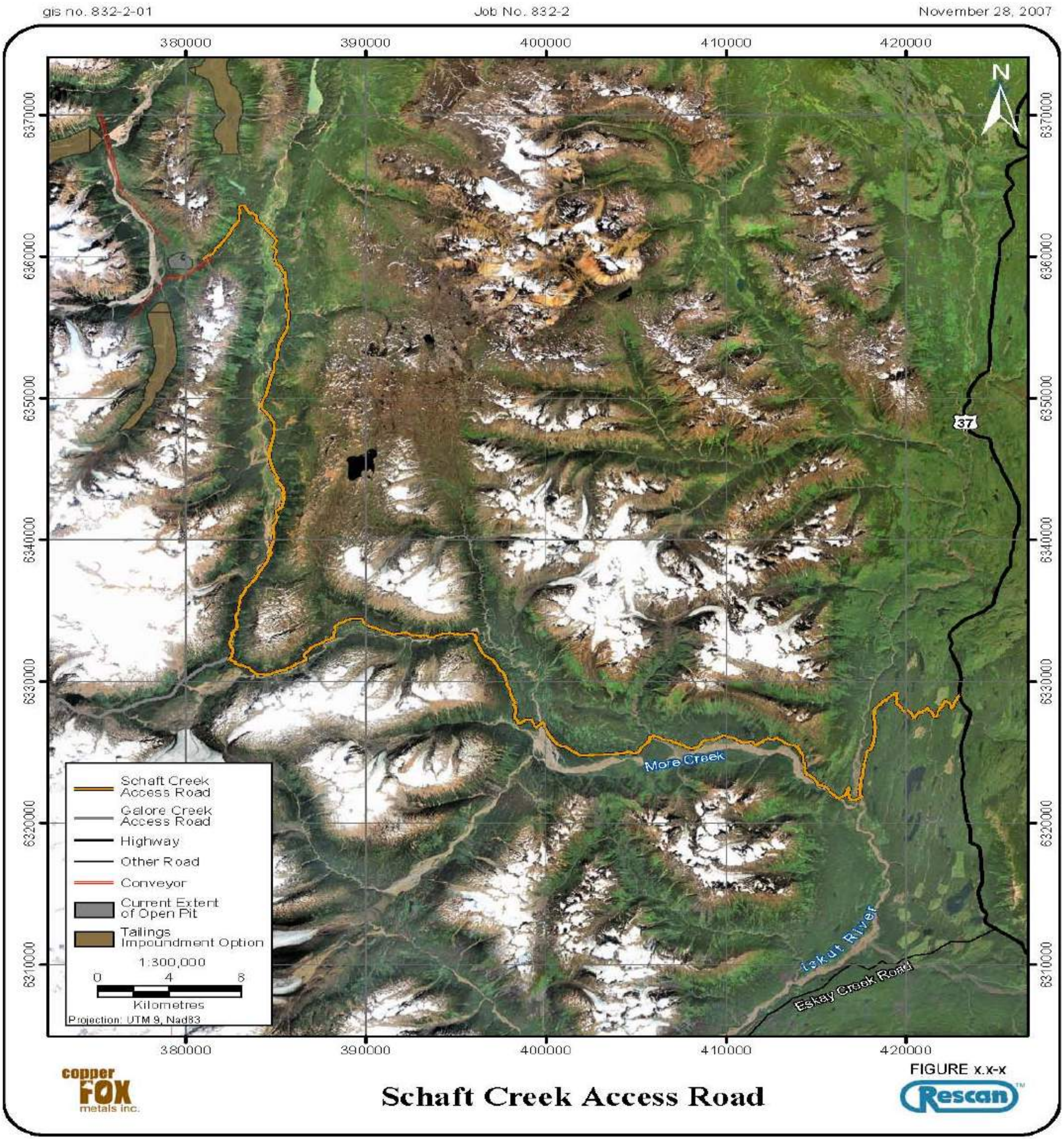


Figure 18.1 Schaft Creek Access Road

18.3.1.2 *Project Objectives*

This Schaft Creek access road prefeasibility study encompasses route selection, road location, road and bridge design constraints, construction recommendations and a construction cost estimate for the Mess Creek route. The field reconnaissance was conducted with geotechnical and environmental parametres in mind but separate detailed studies are still required.

Primary objectives of this study were to:

- Explore possible alternative access routes;
- Locate and GPS (hand held) a preliminary road centerline between control points;
- Locate and mark all bridge and major culvert crossings;
- Record soil types, rock outcrops, % sideslope and other topographic features;
- Note any evidence of unstable slopes;
- Note proximity to wetlands;
- Update construction categories and revise cost estimates and submit prefeasibility report.

18.3.2 Road Engineering

18.3.2.1 *Route Selection*

Using the available 1:20,000 TRIM mapping and 1:50,000 topographic maps the potential access routes to the Schaft creek deposit were identified. A comprehensive review of these resources, complemented by an air reconnaissance provided importance guidance for the ground work which comprises the basis for this study.

The following road corridors were examined but eliminated from further study.

- A route north down Mess Creek to the Telegraph Creek road was flown and although technically feasible it does not meet operational constraints as this route adds over 200 km to the haul distance to Stewart BC. There are also major construction issues relative to deep rock canyons along the lower Mess Creek.
- The Ball Creek route via the Arctic Lake Plateau was also flown and rejected due to the unstable slopes along Ball Creek and the large expanse of exposed terrain around Arctic Lake. Approximately 5 km of this route lies within Mount Edziza Provincial Park.
- The most direct route to Hwy 37 goes through Raspberry Pass and connects to the Willow Creek FSR. This route is feasible from an engineering perspective but approximately 31 km of new construction would be necessary within Mount Edziza Provincial Park.

The route south up the Mess Creek valley connecting to More Creek at Three Valleys and then east along More Creek to Iskut river was selected for a detailed ground reconnaissance.

This involved walking the route using hand held GPS units to record control points, terrain features and major creek crossings. Often several preliminary routes were investigated in order to achieve a feasible road location.

18.3.2.2 *Route Description*

Mess Creek

From Snipe Lake at km 0 the road location proceeds north along the side of Mount LaCasse for 4.5 km. Typically there are cold wet glacial till soils with numerous cross drains and small stream crossings. Potentially unstable Class IV terrain was noted at km 1.6 and km 2.0, an avalanche path at km 2.5 and bedrock outcrops near km 1.6 and km 3.5. The proposed location continues to drop at 5-10% favorable grade to the Mess Creek valley floor at km 6.2.

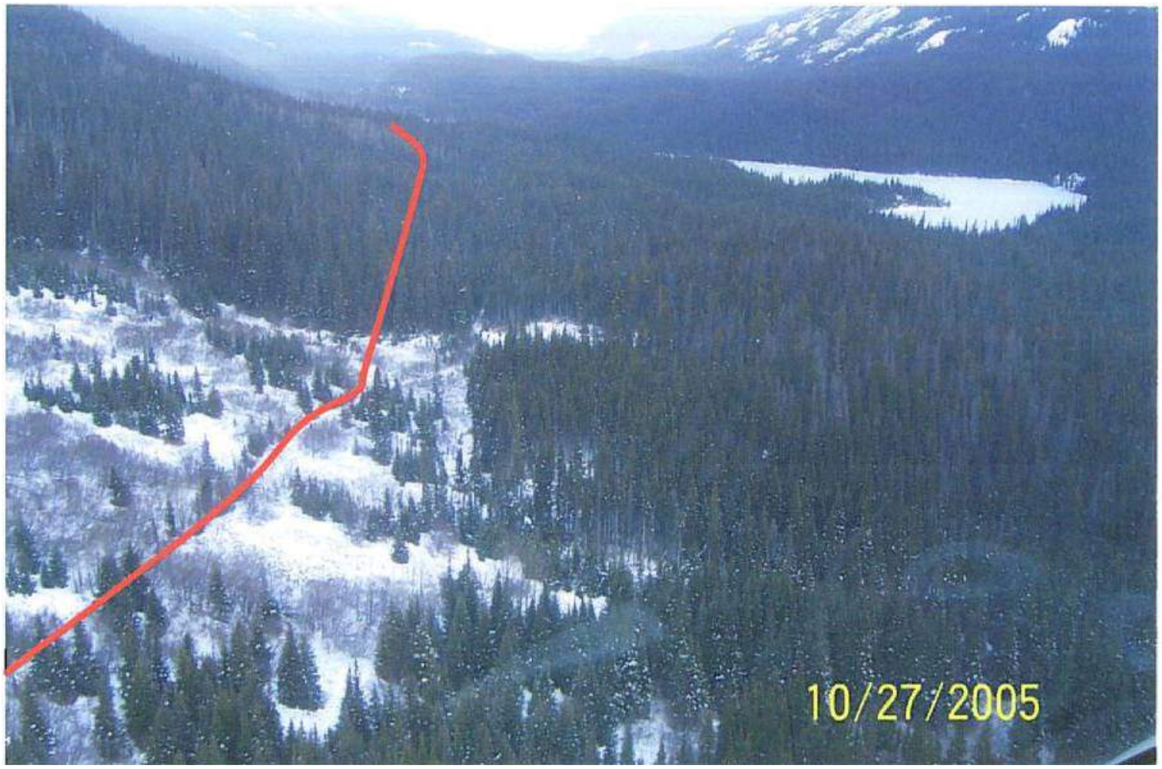


Figure 18.2 km 0 Looking Northeast

From km 6.5 to km 8.0 the route crosses the Mess Creek flood plain and two main channels requiring bridges of between 30-40 m each. A rock fill causeway is needed across the wetlands for 500 m before reaching drier sandy gravel soils on a 40% sideslope.

Figure 18.3 Mess Creek Flood Plain Road Centerline Crossing

The next 4 km are located on primarily a west aspect with well drained granular soils with 25-30% coarse fragments. Several rock outcrops were noted and a 40 m wide rock face requiring a 6 m cut on centerline was traversed at km 9.8.

Figure 18.4 Typical Angular Broken Rock Outcrop

km 12 to km 13 is relatively flat terrain with granular soils. km 13 marks the start of the Little Arctic Creek fluvial fan which extends for approximately 500 m. The creek crossing is estimated to require a 27 m span bridge. (Figure 18.5)

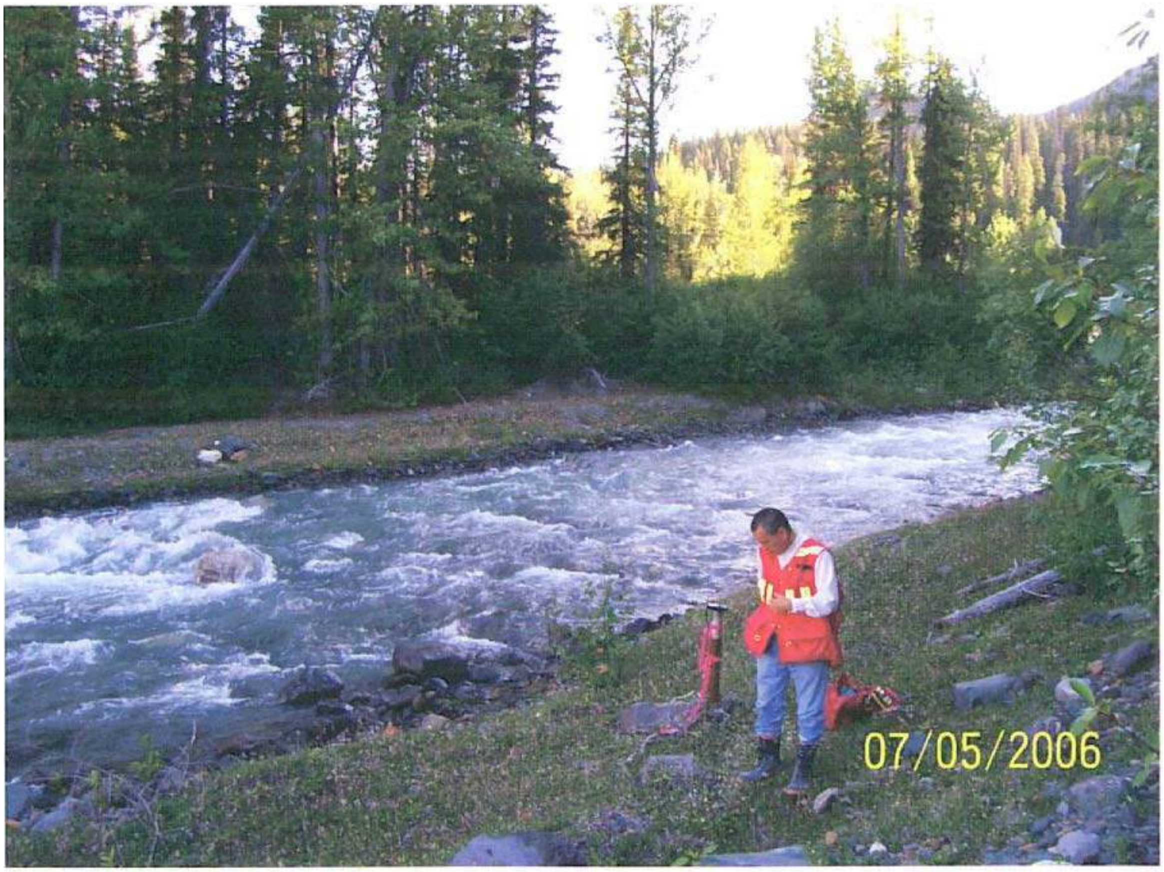


Figure 18.5 Little Arctic Creek Bridge Crossing

The route continues south along the east side of the Mess Creek valley over undulating ground with major stream crossings at km 14.2, km 15.2 and km 16.9. Other important features include an avalanche chute at km 17.3 and a rock bluff requiring drilling and blasting at km 18.5.

Figure 18.6 Talus Slope along lake shore near km 17.5

Gentle slopes and average construction conditions allowing sidecasting of silty sandy gravel soils were encountered up to the 15 m bridge crossing at km 23.1. Beyond this point all the way to Arctic Creek at km 33.5 the road location follows along the sidehill on 40-60% sideslopes. Soils consist of broken rock and a thin veneer of gravelly silt overlying bedrock requiring endhaul of earthworks on the steeper slopes.

The Mess Creek valley narrows abruptly as the route climbs up from Arctic Creek and through the pass between Mess Creek and More Creek drainages. Special features include a 200 m long boulder field of 2-3 m diametre boulders at km 34.6 and extensive avalanche hazard openings between km 36 and km 37 and again for 200m from km 37.3.

The proposed road location ends at km 40, where it then ties into km 65 of the designed Galore Creek Mine access road.

Figure 18.7 Avalanche hazard at km 37.5

18.3.2.3 *Road Design*

Design Requirements

The Schaft Creek access road is classified as a resource development road. The design criteria specified for the road called for a single lane (6 m) radio controlled road capable of carrying the legal Axle loading for trucks on British Columbia highways on a year round basis. The road is required to provide vehicle access; for development of the mine site and to provide year round road access for supplies, equipment, crew transport, and once operations commence the road will be used for continuous concentrate hauling.

Alignment controls such as maximum 10% sustained grades and 50 m minimum radius horizontal curves were used in the ground reconnaissance to locate a feasible route. The typical road cross sections (Appendix I) depict the approximate range of construction procedures for varying terrain conditions. They also correspond to the construction categories defined in the Preliminary Economic Assessment (PEA) report.

Table 18.2
Construction Categories

Category	Description
1	Existing Road / Upgrade
2	Other Material (O.M.) or Fluvial Fan/ 0-30% Sideslope / South Aspect
3	Other Material (O.M.) of Fluvial Fan/ 0-30% Sideslope / North Aspect / Sidecast
4	Other Material (O.M.) & some Solid Rock or Talus Slope/>50% Sideslope / short End haul
5	Solid rock / Drill & Blast / End Haul
6	Wetlands/Overland Construction/End Haul Rock Ballast/Geotextiles

Design Specifications

Preliminary plans and profiles (Appendix II) were prepared based on the following design specifications and field observations.

Table 18.3
Design Specifications and Field Observations

Classification	Single Lane
Average Daily Traffic (ADT)	≥30
Design Speed (km/hr)	60
Maximum Grades (%)	10
Road Width (m)	6.0
Pull-out Width (m)	4.0 m
Right-of-way (m)	≈30
Min. SSD (m)	70
Min. Radius (m)	55
Min. K. Factor – Sag	9
Min. K. Factor - Crest	10

The design criteria established for the stream crossings include:

- All bridges to be designed to pass the 100 year flood and maintain 1.5 m debris clearance;
- All fish bearing streams shall be bridged or have open bottomed arches installed to protect the channel;
- Culverts shall be sized for the 100 year flood and cross drain culverts are to be installed not more than 250 m apart;
- Major bridge crossings shall meet the requirements of the Navigable Waters Protection Act.

The horizontal and vertical alignment is shown along with road grades and GPS coordinates for terrain features and construction constraints. An estimate of road sections requiring drilling and blasting is listed in Table 18.4.

Table 18.4
Mess Creek Rock Work Areas

Location	Estimated Length
km 1.6	40m
km 3.5	50m
km 8.3	50m
km 9.8	40m
km 18.5	50m
km 23.5	200m
km 24.8	50m
km 27.9	30m
km 28.3	150m
km 33.2	30m
km 34.9	40m
km 35.5	80m
km 36.7	50m

The bridges on the Mess Creek route listed in Table 18.5 are simple clear spans ranging from 9 m to 40 m in length. A typical bridge general arrangement and a fish passage culvert are shown in Appendix I.

Table 18.5
Mess Creek Bridge Locations

Location	Estimated Length
km 2.1	10m
km 6.8	10m
km 6.9	30m
km 7.2	40m
km 13.5	27m
km 15.2	20m
km 16.5	15m
km 16.9	9m
km 18.5	15m
km 22.2	30m
km 23.0	15m
km 26.7	10m
km 26.9	18m
km 27.4	12m
km 28.2	12m
km 33.5	27m
km 34.9	9m
km 35.6	12m

18.3.3 Construction Cost Estimate

18.3.3.1 *Schedule of Quantities and Unit Prices*

The Mess Creek route was divided into logical construction categories by map sheet and a construction cost estimate prepared. Unit costs are consistent with those in the previous Scoping Study report (2005) but quantities and construction categories have been adjusted to reflect the results of the ground reconnaissance. An estimate of risk is included for most items as an indication of the certainty of the quantity estimates.

An allowance of $20,000,000 is included in the project capital cost estimate for the access road from Bob Qiunn to Three Valleys, along the previously designed route.

Table 18.6
Mess Creek Route

Drawing	Section	Description	Construction Category					Bridges	
			2 km	3 km	4 km	5 km	6 km	#	Total Length m
1242-1-01	0-6	Snipe Lake to Mess Creek Valley bottom along side of Mount LaCasse		4.5	1.25	0.25		1	10
M-02	6-12	Across Mess Creek to the east side and along the hillside	2.0	1.6	0.6	.2	1.6	3	80
M-03	12-18	Continuing along the east side of Mess Creek	1.2	3.0	1.8			4	71
M-04	18-24	Continuing along hillside gently rising	0.5	3.5	1.75	0.25		3	60
M-05	24-30	Rising up the hillside	0.5	1.75	3.5	0.25		4	52
M-06	30-36	Continuing up the hillside to headwaters of Mess Creek		1.0	4.8	0.2		3	48
M-07	36-40	Connection to Three Valleys			3.5	0.5			
		Total	4.2	15.35	17.2	1.65	1.6	18	321

Table 18.7
Mess Creek Route Cost Estimate

Schaft Creek Access Road
Mess Creek Route
km 0.0 to km 40.0

40.0 kilometres section length

Cost Estimate (Class C)

Item No.	Description of Work	Unit of Meas.	Neat Line Quantity	Est. of Risk	Approx. Quantity	Unit Price	Extended Amount
	Part A – Road Construction						
1.0	***Site Preparation***						
1.1	Mobilization (5% of Part A – total)	L.S.	1	0%	1	722,000	$722,000
1.2	Logging	m^3	26250	15%	30,200.0	32	$966,400
1.3	Clearing & Grubbing	Ha	120	10%	138.0	12,000	$1,656,000
1.4	Stripping	m^3	273600		301,000.0	2	$602,000
2.0	***Construction Category (Primary)***						
2.1	(1) Existing Road / Upgrade (0.0 – 6.0 km)	km	0.0	0%	0.00	0	$0
2.2	(2) O.M. or Fluvial Fan / 0-30% Sideslope / South Aspect	km	4.2	10%	4.60	75,000	$345,000
2.3	(3) O.M. & Rippable Rock / 30-50% Sideslope / North Aspect / Sidecast	km	15.4	10%	16.90	125,000	$2,112,500
2.4	(4) O.M. & Some Solid Rock or Talus Slope / >50% Sideslope / Short End	km	17.2	10%	18.90	175,000	$3,307,500
2.5	(5) Solid Rock / End Haul / > 10% road Grade	km	1.7	10%	1.82	250,000	$453,750
2.6	(6) Wetlands / Overland Construction / End Haul Rock Ballast / Geotextiles	km	1.6	10%	1.80	160,000	$288,000
3.0	***Road Base & Surfacing (Secondary)***						
3.1	Select Granular Base (Surfacing)	m^3	84,000	15%	96,600	12	$1,159,200
3.2	Geotextiles	m^2	320,000	10%	352,000	3	$880,000
4.0	***Drainage Culverts***						
4.1	<1000 mm dia. CSP	crossing	160	15%	184	1,100	$202,400
4.2	1000-<2000 mm dia. CSP	crossing	5	10%	6	2,500	$15,000

Table 18.7
Mess Creek Route Cost Estimate

4.3	Fish Passage Culvert	crossing	5	10%	6	75,000	$450,000
4.4	Rip Rap 10kg class	m^3	3,400	5%	3,600	18	$64,800
4.5	Rip Rap 50kg class	m^3	175	5%	185	18	$3,330
5.0	***Avalanche and Rock Fall Protection***						
5.1	No-Post Barriers	m	1,500	15%	1,725	120	$207,000
5.2	Lock Blocks 2m High Wall	m	600	15%	690	600	$414,000
5.3	Earth Berm 5m High	m	200	15%	230	2,100	$483,000
5.4	Snow Shed	m	50	10%	55	15,000	$825,000
	Part A - Total						**15,156,880**
Item No.	**Description of Work**	**Unit of Meas.**	**Neat Line Quantity**	**Est. of Risk**	**Approx. Quantity**	**Unit Price**	**Extended Amount**
	Part B – Bridge Construction						
6.0	***Bridges***						
6.1	9 m length – 40 m length	m	321	10%	353.0	10,000	$3,530,000
	Part B - Total						**$3,530,000**
	Contingency (15%)						$2,803,032
	Design engineering (5%)						$934,344
	Construction Supervision (5%)						$934,344
	Estimate Total						**$23,359,000**

18.3.4 Operating and Maintenance Requirements

18.3.4.1 *Regular Maintenance*

The road and bridge design concepts have been chosen to provide a low maintenance transportation system with adequate structural strength to accommodate maximum legal Axle loading year round. It is normal for road maintenance costs to be higher initially as the ground and streams adjust to the new configuration. Eventually the cost of stream bank protection, rock scaling, ditch stabilization and settlements will diminish; however gravelling will be an ongoing program, because the road loses gravel through use and maintenance.

Regular maintenance includes those activities that can be planned in advance such as:

- Snow removal;
- Sanding;
- Water/ice control;
- Removal of fallen trees, rocks, slides;
- Sign repairs;
- Ditch clearing;
- Subgrade repairs;
- Rock scaling; and
- Gravelling and grading.

Bridge maintenance is also necessary and includes such items as:

- Riprap replacement;
- Clearing log jams;
- Semi-annual inspections
- Repairing scour damage;
- Replacing curbs, deck and delineators when needed; and
- Sign maintenance.

The estimated cost of regular maintenance is $8000/km annually.

18.3.4.2 *Unplanned Maintenance*

Unplanned maintenance is, by definition, difficult to budget for, but contingency plans and resources need to be available to repair damage caused by unexpected occurrences.

These events could include:

- Debris flows;
- Landslides, rock falls and avalanches;
- Earthquakes;
- Major flooding;
- Fuel spills, traffic accidents/ road closures.

In some cases it may be necessary to construct, and subsequently remove, temporary by-pass roads and bridges used while permanent repairs are completed. An annual contingency fund of $200,000 for the Mess Creek Route is recommended to cover passive avalanche control and unexpected repairs.

18.3.4.3 *Avalanche Forecasting and Control*

Avalanche forecasting and control must be an integral part of the Road Maintenance program for the Mess Creek Route as it passes through high risk avalanche prone terrain and continuous road access is a priority.

Passive avalanche control by monitoring snow conditions and proper signage will suffice for the majority of the road network however avalanche control using explosives in conjunction with control structures will be required for some sections.

A range of structures and the associated unit costs per lineal metre are as follows:

- Low risk - no-post concrete barriers $120.00/lm installed
- Moderate risk -concrete lock block retaining wall (3 m high) $600/lm
- High risk - 5m high earth berm $2100/lm

The above unit costs are factored into the capital costs for road construction.

18.3.4.4 *Vehicle Loading Restrictions*

Full legal highway hauling from the mine access road along Highway 37 and 37A to Stewart BC typically would consist of B-Train configurations with a Maximum Gross Combined Vehicle Weight (GCVW) of 63,500 kg. This is comparable to the proposed BCFS L-75 design vehicle (68,040 kg GVW) for the bridge structures along the mine access road.

It currently appears that there will be no restrictions in using the legal vehicle configurations up to 63,500 kg for highway hauling between the mine access road and Stewart BC. Specific construction and operational equipment overloads will have to be assessed on an individual basis as the design progresses.

18.4 Geotechnical Investigations

18.4.1 Introduction

DST has been retained by Copper Fox Metals to carry out a preliminary assessment of the geotechnical aspects of tailing management options at their Schaft Creek project. This work will form part of the Scoping Study for the project. The purpose of this report is to assess in a preliminary way three options for tailings and waste rock disposal locations, as identified by Samuel Engineering, Inc.

The three tailings storage locations are identified on the Samuel Drawing No 100-GA-052 Rev A dated October 4, 2006. These are illustrated on the figure below and are described as follows:

- Option A, located at Skeeter Lake
- Option B, located on Hickman Creek
- Option C, located east of Mt. LaCasse (on a Schaft Creek tributary)

Waste rock disposal options are near the proposed open pit and various locations along the Schaft Creek and Hickman Creek flood plains.

The scope of work involved the following tasks:

- visit site;
- review available information;
- identify preliminary management parametres;
- review the proposed options and geotechnical issues;
- establish plan for pre-feasibility geotechnical assessment.

The results of this work are discussed in the foregoing section.

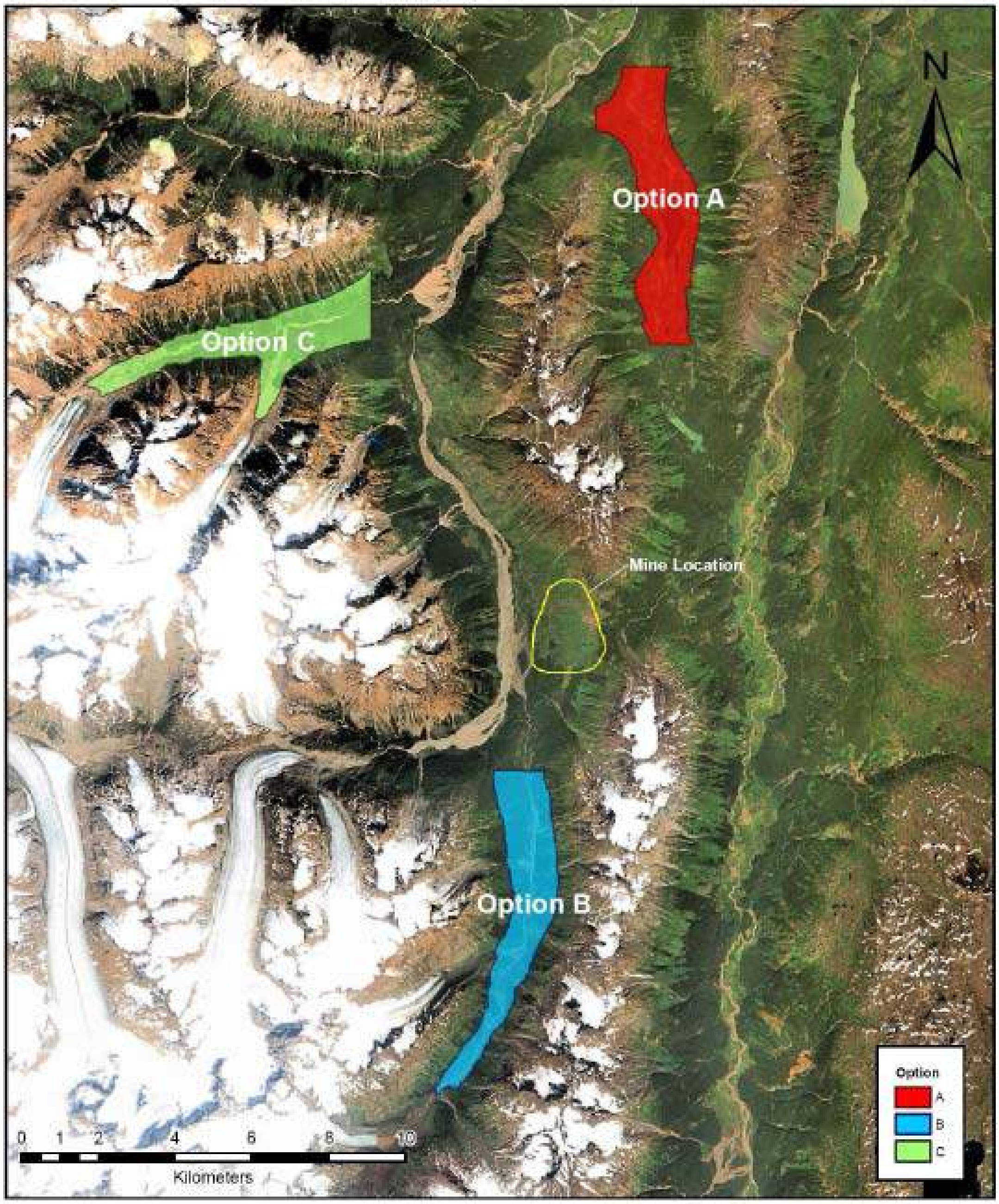


Figure 18.8 Tailings Storage Locations

18.4.2 Assessment Program

The site was visited by the Author on July 27 to 29, 2007. At this time the camp was fully functional and exploration drilling was underway for the season. The three tailings disposal locations and potential waste rockfill dump areas were viewed by helicopter. In addition, an area uphill of the proposed pit area was partially toured by foot and vehicle.

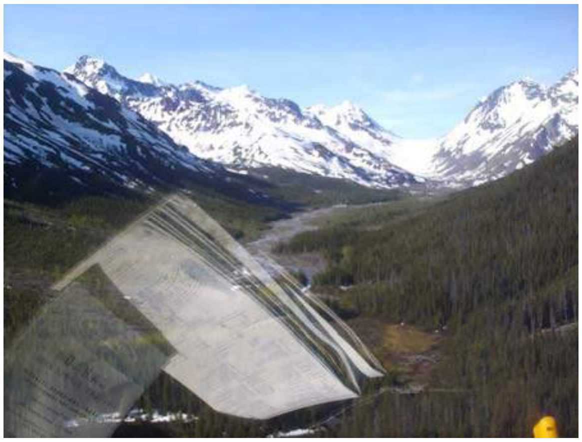
Several photos are attached below. A brief description of each location is provided as follows.

Pit Area

The open pit will be located in what is known as 'The Saddle' immediately south of Mount LaCasse and near Schaft Creek. Exploration drilling is not yet complete and the open pit location has not been finalized. However it is likely that the pit wall will encroach on the toe of Mount LaCasse.

Tailings Option 'A'

This option lies in a valley immediately east of Mount LaCasse. Several small surface water ponds as well as areas of swampy terrain are found on this site. Topographic information indicates that the site currently drains both to the north and south. The main dam will be located at the north end, with an adjacent low saddle dam. A lower dam will be needed at the south end.

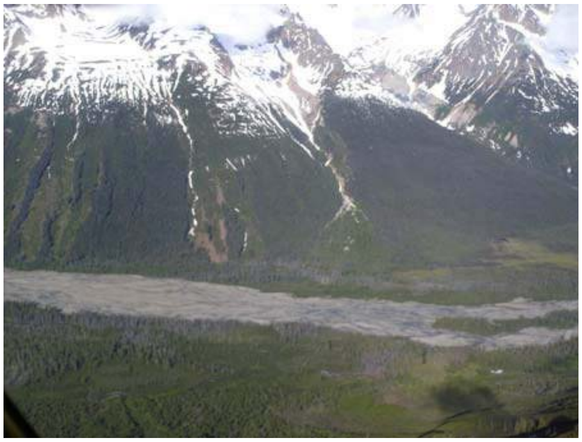


Tailings Option 'B'

This option lies in a valley immediately west of Mount Hickman. The valley has a glacier at its head. Topographic information indicates that the site currently drains to the north as Hickman Creek. The only dam needed will be located at the north end.

Tailings Option 'C'

This option lies in a valley immediately west of Mount LaCasse and east of Schaft Creek. The valley has a glacier at its head. Topographic information indicates that the site currently drains to the east into Schaft Creek. The only dam needed will be located at the east end.



Waste Rock Disposal Areas

The waste rock will be disposed of in the general vicinity of the open pit. With mine development, it will extend easterly, where it will encroach onto the flood plain of Schaft Creek and possibly Hickman Creek, as required.

Borrow Sources

To date, no borrow areas in the immediate vicinity of the Mine have been identified.

18.4.3 Information Review

The following geological references are available for the general area of the site, albeit at a small scale.

- J.G. Souther. MAP 11-1971, Telegraph Creek. Preliminary Series. 1:250,000. Geological Survey of Canada, 1972.
- J.M. Logan, J.R. Drobe, V.M. Koyanagi, D.C. Elsby. Map: Geology of the Forest Kerr – Mess Creek Area, Northwestern British Columbia (104B/10, 15 & 104G/2 & 7W). 1:100,000. B.C. Ministry of Employment and Investment, 1997.
- J.M. Logan, J.R. Drobe, W.C. McClelland. Geology of the Forest Kerr – Mess Creek Area, Northwestern British Columbia (104B/10, 15 & 104G/2 & 7W), Bulletin 104. British Columbia Geological Survey, 2000.

The geology of the Schaft Creek area, including the tailings and waste rock disposal options as well as the pit location, is described therein. References indicate that the site generally has an overburden consisting of glacial outwash, till, colluvium and alpine moraine of varying thicknesses. Furthermore, bedrock in the central, northern and southern areas of the site (Tailings Options 'A' and 'C', Pit Area) consists of extrusive igneous augite-andesite (porphyritic basalt flows). Thinly bedded sedimentary dolomitic limestone with minor interbeds of extrusive igneous tuff, as well as sedimentary cherty siltstone, are also found in parts of the northern areas of the site (Tailings Option 'A'). In the western areas of the site (Tailings Option `C' and Waste Rock Storage) intrusive igneous monozonite predominates.

An extensive array of holes have been drilled in the proposed mine pit area; however, these have little geotechnical data other than the depth of overburden. In this area, overburden depth is highly variable, from nil to about 60 m (records provided by Copper Fox). No such data is available for the tailings or waste rock disposal areas, although drilling at these locations is in progress (summer 2007).

A preliminary assessment of hazards associated with the proposed mine pit and disposal areas is available (A Preliminary Assessment of Mass Wasting and Glacial Outburst Flood Risk at the Schaft Creek Project Site; Spooner, 2007). General hazards associated with the site were noted as follows:

- Snow avalanches;
- Debris flows;
- Shallow landslides;
- Large scale rock creep.

In addition, site specific risks were noted as follows:

- Open Pit
    - Hazards related to 'The Bench' on Mount LaCasse above the pit are likely to be shallow debris flows. Possible glacial outburst flooding with associated debris flows were identified south of the pit.

- Tailings Option 'B'
    - Glacial outburst flooding from the head of Hickman Creek into the tailings disposal site was found to represent a minor risk.

An assessment of metal leaching (ML) and acid rock drainage (ARD) is available. (Draft, Schaft Creek Project - Prediction of Metal Leaching and Acid Rock Drainage, Phase 1; Rescan, 2007). This initial phase indicates that nil to 2% (of 59 samples) were net acid generating, 5-14% were "uncertain", and the remainder were net neutralizing. Further testing is underway to assess uncertainties.

18.4.4 Management Parametres

The following guidelines and regulations will guide the geotechnical requirements for the mine:

- Mining Association of Canada (MAC) Guidelines;
- Canadian Dam Safety Guidelines;
- British Columbia provincial legislation;
- Canadian federal legislation;
- General state of the practice methods in geotechnical engineering;
- National Instrument 43-101 Standards for Disclosure of Mineral Projects.

Of particular importance in the above is the need to establish suitable design criteria for natural events, both climate and seismic related.

At this stage, the following operational parametres, provided by Samuel Engineering, have been applied:

- A 65,000 tonne per day mill operation;
- Production for a period of more than 30 years

18.4.5 Options Review

Based on the above information, the options have been reviewed for preliminary geotechnical assessment purposes.

In general, based on the above assessment, all of these options are considered feasible at this stage, as detailed below.

Several significant issues have been identified and need to be assessed at the prefeasibility stage in order to provide geotechnical input for screening of the options. These are also discussed below.

18.4.5.1 *Open Pit*

The open pit will encroach onto a localized area of the lower slopes of Mount LaCasse. The exact extent will depend on the configuration of the deposit and open pit. The preliminary assessment indicates that cutting into the slope as required is feasible.

Significant issues for the site surrounding the proposed open pit location are expected to be as follows:

- potential unstable slopes of Mount LaCasse;
- geohazards, including snow avalanches and debris flows;
- the design of the pit slopes, particularly the high wall against Mount LaCasse.

18.4.5.2 *Tailings Disposal*

The assessment has indicated that each of the three tailings disposal sites is suitable for containing the tailings. Furthermore, the sites are suitable for containment by dams.

Significant issues for the tailings disposal sites are expected to be as follows:

1. the structural geology is complex and requires assessment;
2. dams will be very high, in the 120 to 200 m range, requiring comprehensive geotechnical investigations and design;
3. potential impacts of earthquakes on dam foundations, particularly overburden, will have a significant impact on dam configurations;
4. weak overburden under dam footprints may require removal;
5. bedrock and overburden permeability under dams require assessment (will dictate requirements for dam cores, liners, grouting);
6. seepage control through permeable dam foundation materials will be a significant issue, particularly if waste is potentially acid generating;
7. surface water routing away from the disposal sites would involve large perimetre ditches with associated maintenance issues (access roads, erosion control, geohazard impacts);
8. impacts of geohazards on dam stability and freeboard will need to be assessed and controlled;
9. there may be a shortage of suitable nearby borrow materials for dam core construction. This issue becomes particularly important if further testing indicates that tailings need to be saturated for long term ARD control.

18.4.5.3 *Rock Waste Disposal*

Potential waste rock dump areas near the mine pit, and along Schaft and Hickman Creeks, are viable. Suitable locations designed for the rockfill will be constrained by the need to pass creek flood flows and control rock dump instabilities, both internally and in the foundation soils.

Significant issues for the rock waste disposal sites are expected to be as follows:

1. Seismic impacts on the stability of overburden materials under the disposal area;
2. Routing of existing surface water (including Hickman and Schaft Creeks) where the flood plain is encroached upon by the waste rock;
3. The need for addressing ARD concerns, if any.

18.4.5.4 *Borrow*

There is very little information on potential borrow supplies for dam core construction.

Significant issues for a borrow supply are expected to be as follows;

1. The need to control seepage with a dam core, particularly in the case of acid generating tailings;
2. The location, quantity and quality of borrow sources.

Given that at this point ARD is not seen as a concern, the need for core materials will likely be minimal, given the availability of synthetic lining systems and the seepage control provided by the tailings themselves.

18.4.5.5 *Environmental and Operational Issues*

Other non-geotechnical issues that will come into play and will need to be considered in selection of options will include:

- Environmental assessment results;
- Effluent treatment considerations;
- Groundwater and surface water baseline and assimilative data;
- Accessibility;
- Distance from the mill;
- Relative elevation of the mill;
- Land and resources uses;
- Ownership and mineral rights;
- First Nations' land claims;
- Transportation corridors;
- Infrastructure requirements;
- Potential deposits;
- Construction material availability;
- Reclamation considerations;
- Operations and maintenance considerations, including costs.

18.5 Mass Wasting and Glacial Outburst Flood Risk

18.5.1 Introduction

Dr. Ian Spooner, P. Geo, of Acadia University was retained by Copper Fox Metals Inc. to complete a preliminary qualitative assessment of risk of mass wasting and glacial outburst floods at the proposed Schaft Creek development site. This assessment will serve as a guide to more detailed studies associated with a geotechnical risks.

Schaft Creek is located in a region of northwestern British Columbia characterized by moderate to extreme relief in which active glaciers, moderate to high precipitation rates, and high relief result in an environment in which landscape instability is common (Spooner, 1994). Though mass wasting and geomorphic hazard studies are rare in the region that work that has been done indicates that landslides, avalanches and, less commonly glacial outburst floods do occur.

The study site is typical of alpine glaciated terrain in northwestern British Columbia. Schaft Creek, Hickman Creek (Tailings Option B) and Fletcher Creek (Tailings Option C) valleys and associated tributaries are typical alpine and subalpine glaciated valleys that exhibit broad U-shaped cross sections and steep valley slopes. Active glaciers are found in each of these valley systems (Figure 18.9). If a glacier exists in the valley or tributary the valley bottom is typically flat bottomed with extensive proglacial outwash sedimentation. Where hanging valleys exist debris fans are common.

Skeeter Lake Valley (Tailings Option A) is somewhat anomalous as glacial and fluvial activity is a relatively minor landscape modifier. This valley may have once been a distributary corridor that was beheaded by headward erosion of Mess Creek. River gradients are low and both longitudinal and cross sectional valley profiles are more subdued than in adjacent valleys.

Valley slopes in the study region are composed of varying amounts of outcrop, and coarse grained colluvium or till; lateral moraines are locally important sediment sources. Where slopes exceed 25° landslides occur. The most active slides in the study region are found on the southwestern slope of Mt. LaCasse. Surface (mantle or soil) creep is common on most slopes but is especially prevalent above treeline. Deep seated creep in bedrock (sakung) is likely occurring in the region especially where high angled slope parallel structures exist. Detailed air photo analysis is required to delineate these features. The vegetation assemblage on the slopes reflects these dynamic conditions (as well as climate) and is comprised primarily of grasses, various shrub types (willow, poplar, ash) as well as subalpine fir. Slopes that experience repeated avalanche activity are characterized by the dominance of shrubs, herbs and grasses and have often been partially stripped of the overburden cover.

After an initial site reconnaissance the focus of this preliminary assessment was defined as the following:

1. Potential for glacial outburst floods from glaciers in the study region, particularly in the Hickman Valley and Fletcher Valley corridors;
2. Potential for landslide and debris fan activity, particularly at the proposed mine site, tailings impoundment options and immediately adjacent regions;
3. Potential for avalanche activity and avalanche recurrence particularly in the proposed mine site region.

All sites referred to in the text are referenced on a satellite composite (Figure 18.9).

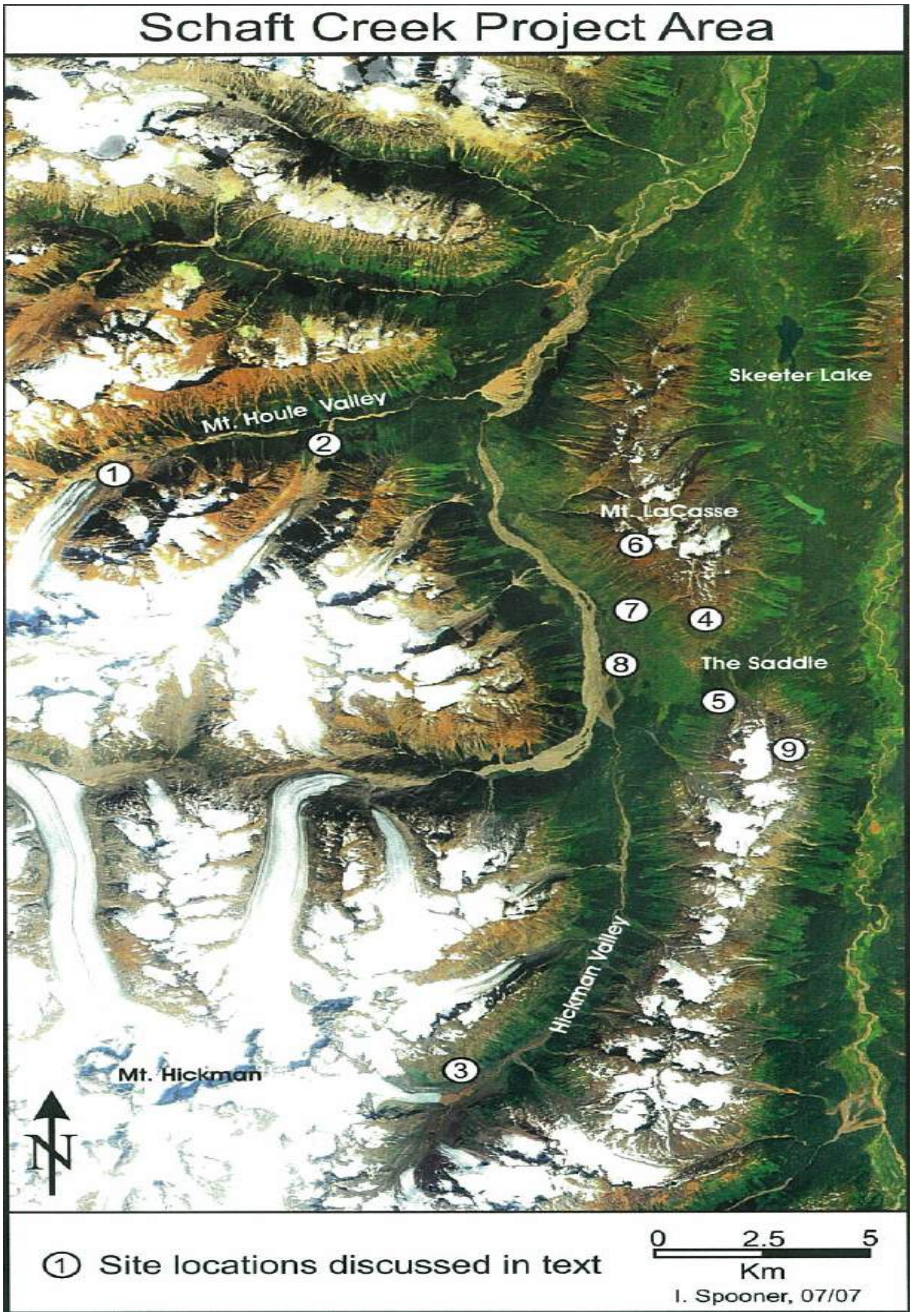


Figure 18.9 Location of Sites Referred to in Text

18.5.2 Glacial Outburst Floods

Catastrophic outbursts most commonly occur when water that pools in front of a glacier is rapidly released. Water can also collect under glaciers, especially those that exhibit complex bed morphology. A flood occurs when the glacier detaches itself from its bed allowing meltwater to drain, often rapidly. This type of flooding is most often associated with large valley glaciers though floods have also been known to originate underneath cirque and alpine glaciers (McKillop and Clague 2006).

As McKillop and Clague (2006) have indicated, outburst flood hazard is greatest when moderately large pro glacial lakes that are impounded by large, narrow, ice-free moraine dams composed of sedimentary rock debris drain into steep, sediment-filled gullies above major river valleys. This scenario does not exist in the study area. However, at the head of Hickman Creek, a moraine-dammed lake has formed on the low gradient outwash plain in front of the Hickman Glacier (Site 3). As well, a series of debris fans on the south side of The Saddle (Site 5) are of indeterminate origin. Ponded water at the toe of the up-valley glacier may be a possibility.

Outburst floods can develop into highly destructive debris flows. Evidence of past catastrophic flood events most commonly occurs as debris fields and scour located in the glacial forefield exposed by recent ice retreat. As well, outburst floods can result in the transportation of debris above the normal terrace level of the outlet streams.

Risk to infrastructure from glacial outburst floods was assessed both by helicopter reconnaissance and site assessment of glacier forefields within the watershed of the three impoundment options. Risk of flooding from glacial drainages that have the potential to impact the proposed conveyor corridor to Option A was also assessed.

Head of Hickman Creek

A moderately sized moraine-dammed pro glacial lake exists at the head of Hickman Creek (Figure 18.10 & Figure 18.11). The lake is approximately 800 m long, 500 m wide and is at least 12 m deep (as indicated by large icebergs floating in the lake). The moraine dam has been breached in the past and parallel terraces observed along the outlet creek may be an indication that the moraine breech was accompanied by flooding. Moderate outlet flow is present at the site. The moraine is post - Little Ice Age (LIA) in age and most likely was deposited less than 50 years ago (Ryder 1987).

This site represents an unpredictable hydraulic feature in the Hickman Creek watershed. If the lake basin was formed by over deepening during LIA advance appreciable amounts of water could be stored in the lake below the level of the sill that impounds the lake. Flooding could take place if moraine sediment forms the sill that impounds the lake at present and a rapid increase in input water volume leads to erosion of this sill. It is unlikely that appreciable amounts of water are presently accumulating underneath Hickman Glacier given the apparent gradient of the bed. If possible, this pro glacial lake should be sounded to determine water volume stored.

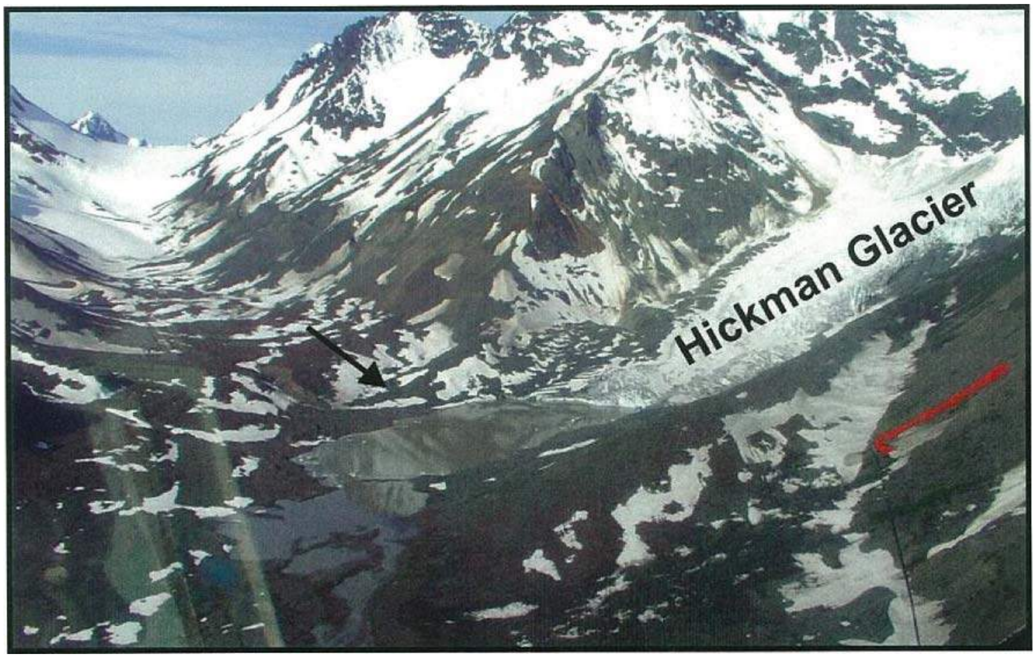


Figure 18.10 Proglacial Lake located at the base of Hickman Glacier.

Figure 18.11 Recessional moraines that impedes drainage.

Hickman Glacier Proglacial Lake. Dashed line denotes crest of the recessional moraine that impedes drainage. Two other older recessional moraines (1 & 2) are also evident.

The Saddle

A prominent debris fan occurs on the southwest slope of the The Saddle (Site 5). This debris fan is active at present and impacts an old access road. The presence of matrix supported sediment, deep lateral erosion terraces, and oversized bedload suggests that rapid release of water stored upstream of the fan may have occurred.

18.5.3 Landslides and Debris Flows

Landslides (and landslide scars) along the sides of the Mess Creek Valley and its tributaries are relatively common. They occur as thin translational slides that rapidly transform into debris flows with debris often reaching the valley floors (Spooner 1994). Most of these landslides occur where there are benches of unconsolidated sediment that may represent remnants of lateral moraines or glacial lake deposits (Spooner and Osborn 2000). A number of debris flows can be observed on the southwest flank of Mt. LaCasse above the Paramount Zone (also referred to as the West Breccia Zone; Site 4).

Debris flows are common at the toes of steep gradient glaciers where recently exposed moraine material has been mobilized by focused meltwater flow. A prominent series of proglacial debris flows (and associated debris fans) have occurred on the south side of The Saddle and have intersected an access road (Site 5; see Figure 18.12 – Figure 18.14 below). Debris fans are also common at the bottoms of gullies and ravines. The debris fans grow rapidly in regions where wet avalanches are abundant as various types of rocks and organic matter are transported down slope by the avalanches.

A preliminary assessment of landslide risk in the Schaft Creek Site was completed by helicopter reconnaissance and on-site evaluation (LaCasse Debris Flows, Saddle Debris Fans). In this assessment particular attention was focused on debris flow potential above development locations on southwest Mt. LaCasse and in the vicinity of Mill Option A in The Saddle.

Mt. LaCasse Debris Flows

A series of debris flows are located on the southwest side of Mt. LaCasse (Figure 18.12 & Figure 18.13). The initiation zone is located directly beneath a distinct geomorphic feature (called The Bench) of indeterminate origin that appears to be formed of glacial till and bedrock (Figure 18.14) with a thick talus veneer (Figure 18.15). The bedrock occurs as a number of knobs or protuberances with till plastered in between. Though not investigated in detail initial image analysis indicates that these bedrock "knobs" are likely a consequence of varying bedrock composition and structure. As well, the concentration of structural lineaments in the area indicates that The Bench itself may be structurally controlled. The debris flow scars are typically long, narrow, and relatively straight, initiating between bedrock knobs near the top of The Bench where the slope gradient is generally the steepest and till is thickest. The slopes where failures occur range from 30° to 50°. The flow scars tend to be from 200 m to 300 m in length though they can be shorter (<150 m in length). They are relatively narrow (5 m to 20 m) and shallow, removing 0.5 m to 1.5 m of surface sediment. These failures may be called thin skinned debris flows (Wahl et al. 2007).

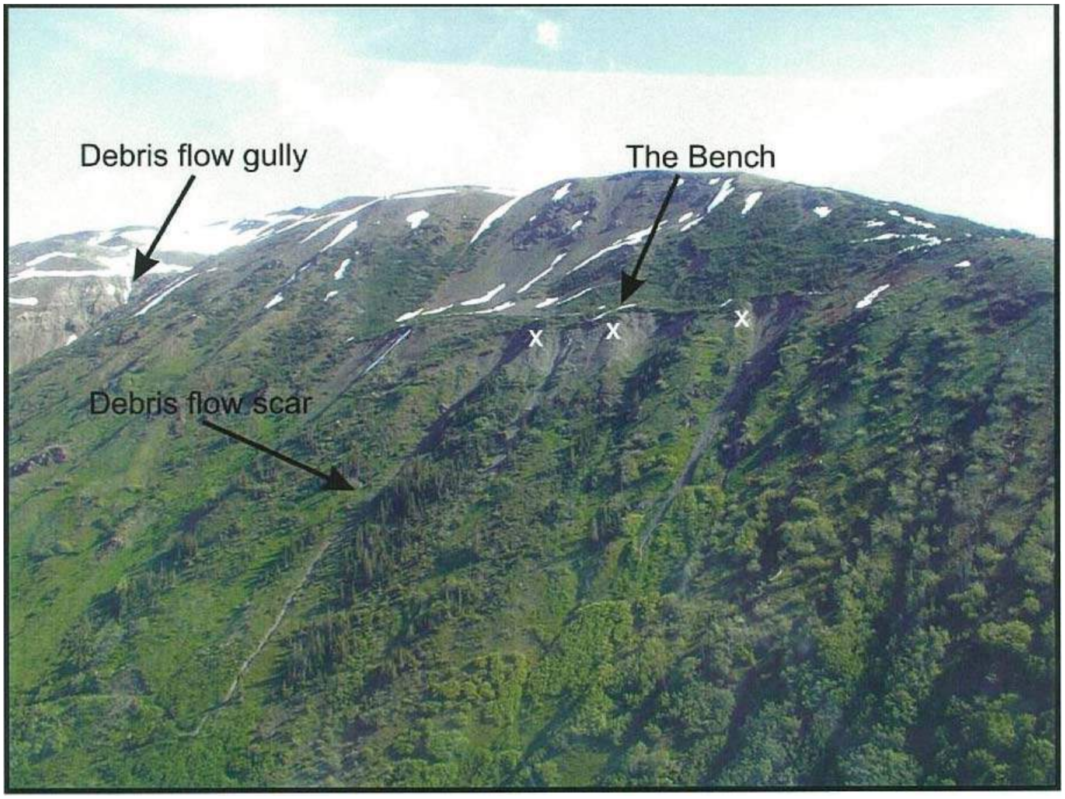


Figure 18.12 Debris flow scars on Mt. LaCasse.

Five of the scars are associated with an enigmatic geomorphic feature called The Bench. Bedrock (x) forms the ridges between scars. A debris flow gully is located to the north and is most likely fault controlled.

The remnant debris deposited at the base of the slide scar is primarily cobble to boulder sized, with the fine fraction most likely removed by consequent rainfall and snowmelt. This debris is consistently angular and unsorted. The presence of inverted trees of varying size, shattered rock, and scouring to bedrock indicate that failure is rapid and turbulent, and falls broadly within the classification scheme of debris flow (Varnes, 1978).

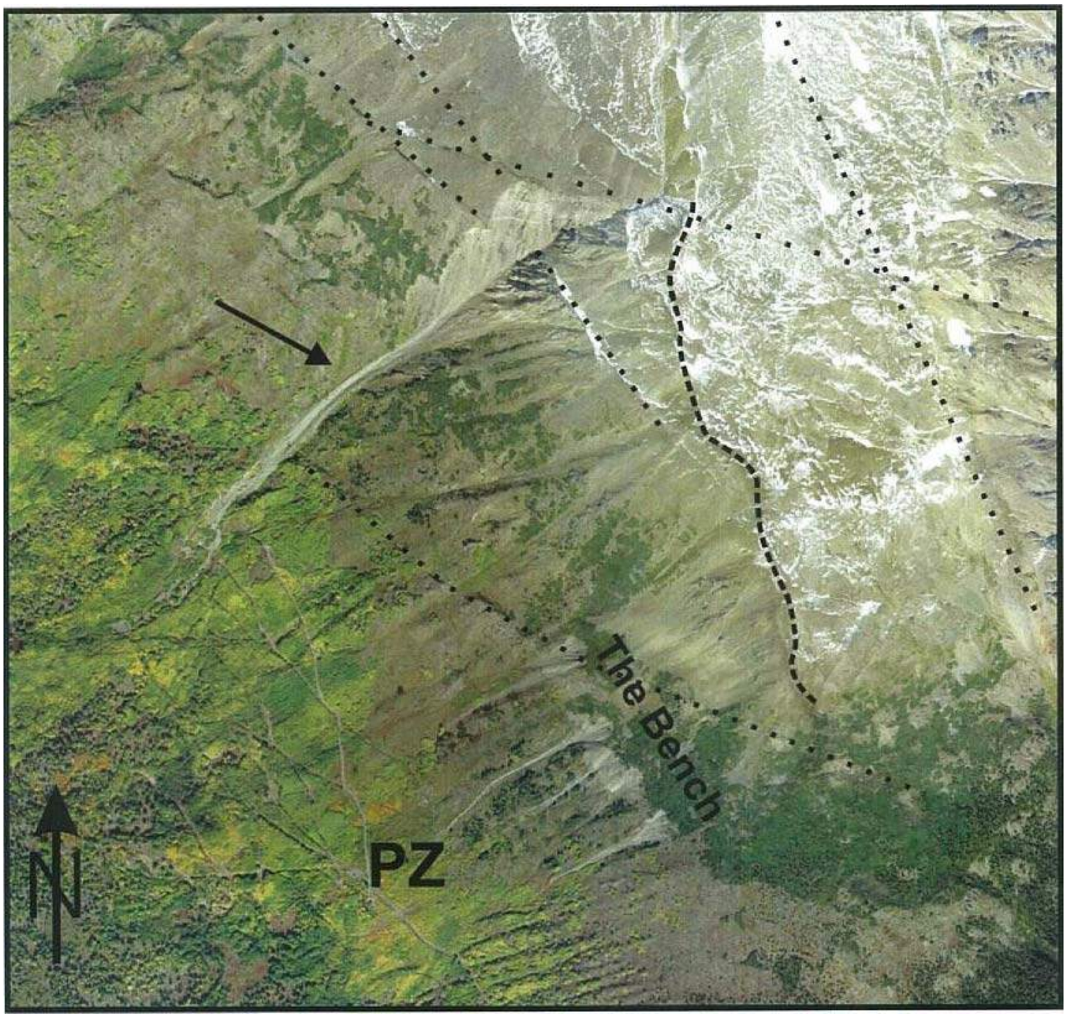


Figure 18.13 Air photo of the southwest slope of Mt. LaCasse.

Five debris flow scars can be seen below The Bench. Arrow points to a prominent structurally controlled debris flow scar. Dashed line indicates the top of the southwest slope. Dotted lines indicate lineaments. PZ stands for Paramount Zone.

A much larger debris flow scar occurs about 300 m to the north of The Bench (Figure 18.13). The site is active but is located to the north of the Paramount Zone and at present does not represent a significant hazard to future mining operations. This debris flow scar is most likely structurally controlled.

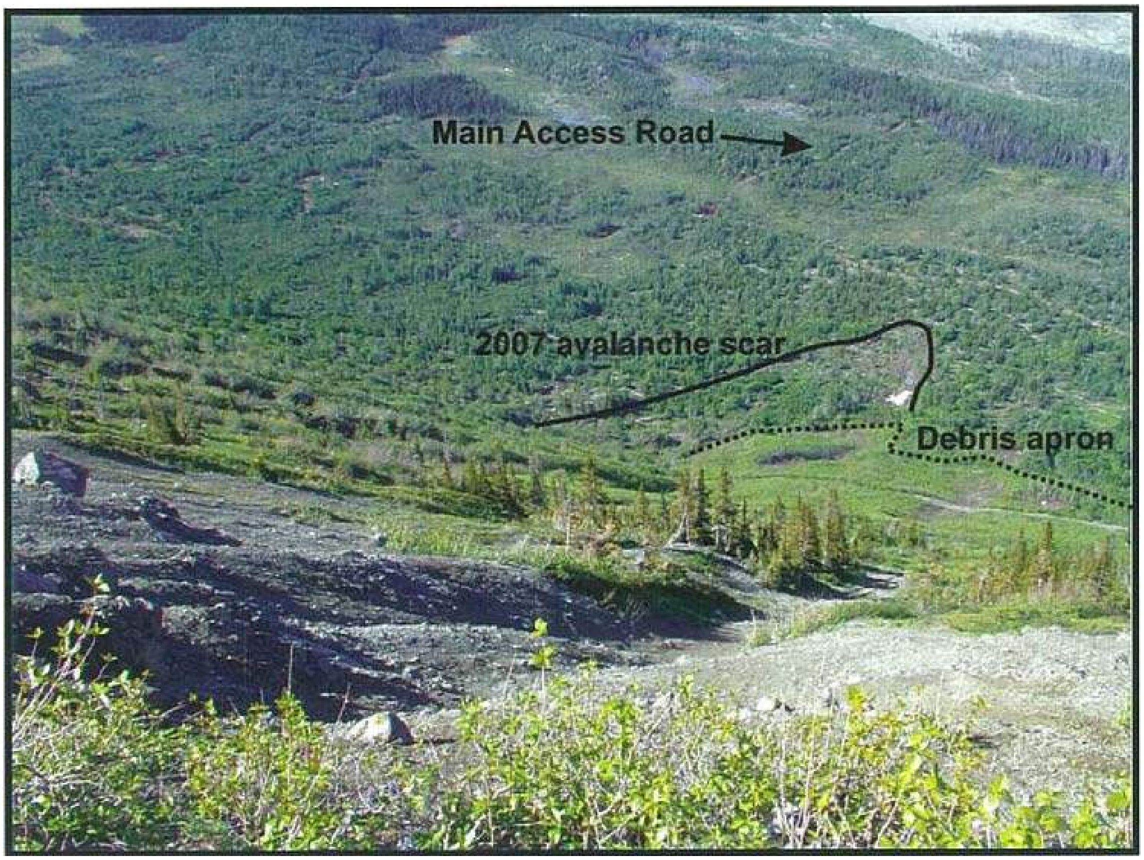


Figure 18.14 Looking down one of the scars from The Bench.

A debris apron formed of debris flow and rockfall material is located at the base of the slope. This is also a site of avalanche activity.

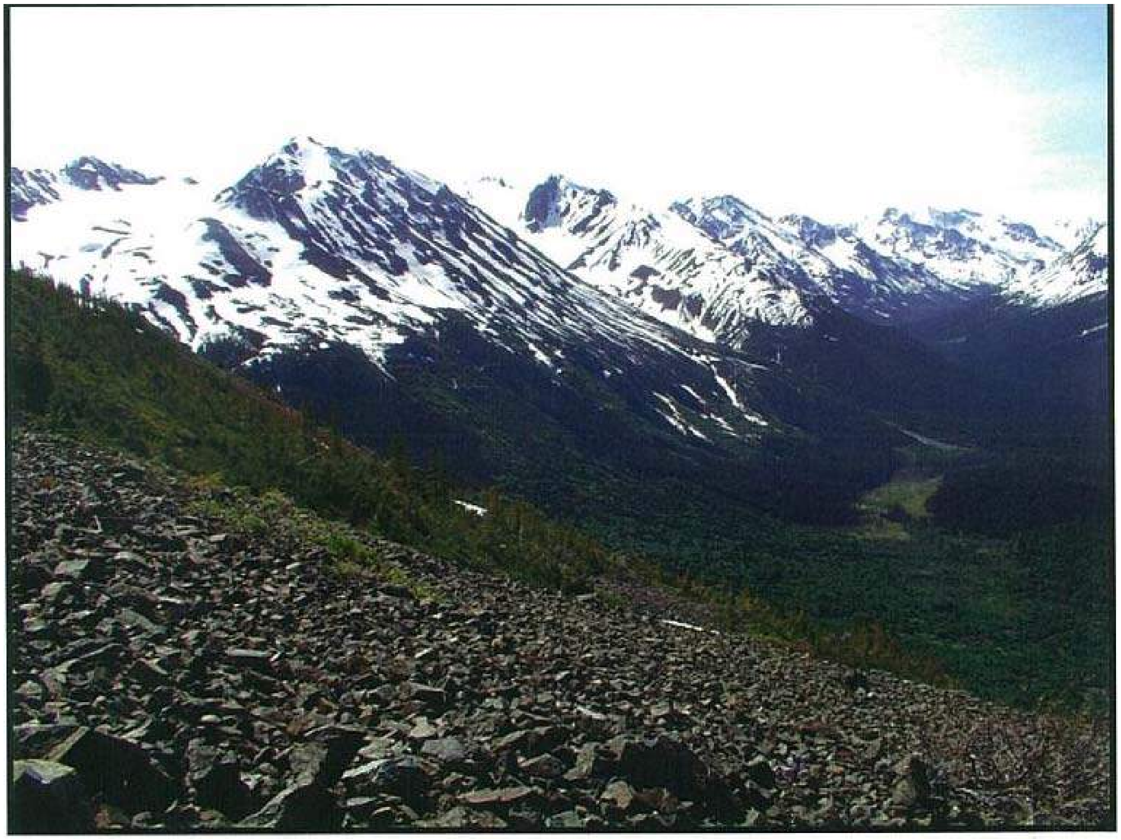


Figure 18.15 Talus slope above The Bench.

This talus feeds into debris flow scars and is a contributor to rockfall hazard at the site.

Saddle Debris Flows

At least two ages of debris fans are evident on the south side of The Saddle (Figure 18.16). The sediment that makes up these fans was deposited by debris flows that originated upslope in an alpine glaciated valley south of the fan. The older fan is presently inactive and was formed by a number of high energy flows (most likely debris flows) at least 70 years ago as indicated by one preliminary tree ring date obtained from a subalpine fir tree located on this fan. Detailed dendrochronology on the fan surface has not been attempted.

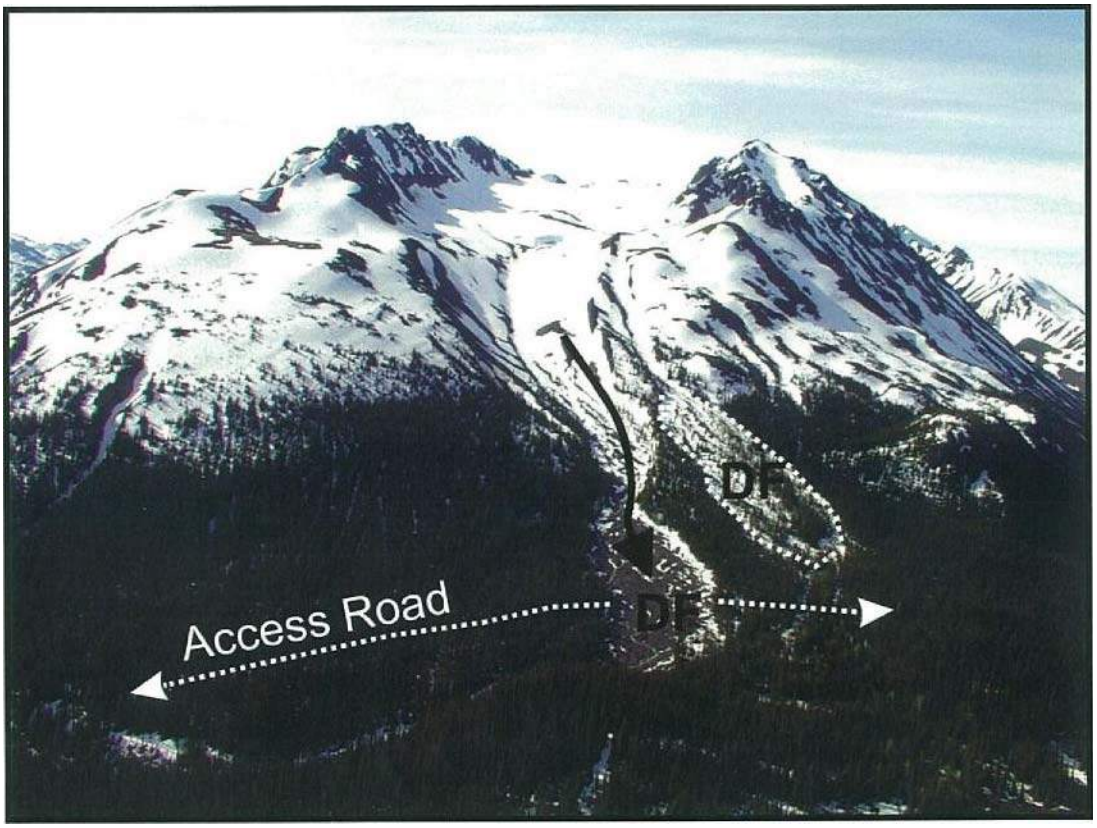


Figure 18.16 Saddle debris flows looking south.

An older, inactive debris fan (outlined in white) is located to the west of the main debris fan. The arrow indicates the active debris transportation corridor

The younger fan is active; annual vegetation and occasional willow (Salix sp.) are present on the surface (Figure 18.17). The fan is approximately 130 m in width and clasts up to 1.3 m in diametre are evident. The fan sediment is poorly sorted and moderately to poorly rounded. Little fabric that might be indicative of bedload transport is evident. Some crude, matrix-supported fabric is evident in sections exposed along the fan margins. Fluvial processes are modifying sediment on the surface of the fan. An old access road that once cut across the fan is now completely obscured.

Figure 18.17 Saddle debris fan looking south.

This debris fan is active as indicated by freshly eroded banks and lack of perennial vegetation. The source of the sediment is a Little Ice Age lateral moraine located at the tree line.

The mechanism for sediment transport to the fan is not clear. Spring freshets may account for much of the incision channel development on the present fan. However, the sediment is very coarse and angular. Exposures in the cut banks show little structure (imbrication, stratification) and the sediment is, in places, matrix supported. These data suggest that at least some of the sediment has been deposited at the site by viscous flow (debris flow). Given the modest slope of the fan (±10°) one of the few possible mechanisms for debris flow would be rapid release of stored water. As the glaciated basin and source of the sediment was still snow covered at the time of this investigation geomorphic evidence for water storage was not observed. However air photos show interesting features near the toe of the glacier that may indicate that meltwater could be stored and/or focused creating viscous flow conditions (Figure 18.18).

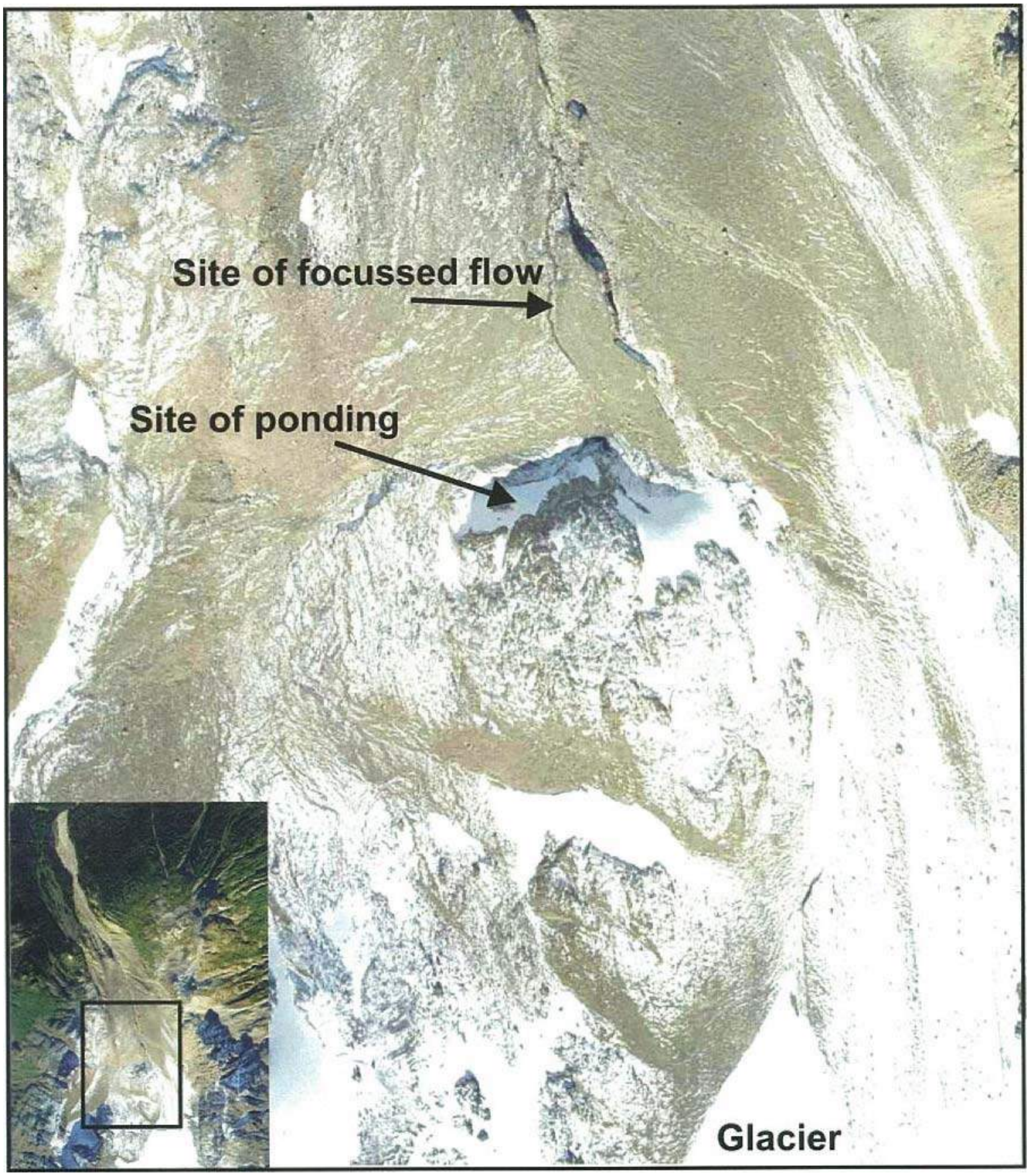


Figure 18.18 Toe of the glacier that provides meltwater to The Saddle debris fan.

High resolution photograph of the toe of the glacier that provides meltwater to The Saddle debris fan. Arrows point to features that may indicate that meltwater pools in front of the glacier and is released rapidly. Inset shows location of photo, north is up.

18.5.4 Avalanche Hazards

Snow avalanches occur along almost all glacial scoured valley walls within the study site. Large, sparsely vegetated avalanche paths are common and the scoured, slightly concave alpine valley slopes are efficient accumulation and release zones. Optimal conditions occur on open 30° - 50° slopes with well established tracks mantled by loose debris (McClung 2001). A number of large avalanches that had the potential to damage development infrastructure occurred in spring 2007 on the southwest side of Mt. LaCasse immediately above the Paramount Zone and the potential waste rock site. These avalanches were most likely wet as they exhibited a tendency to closely follow gullies and ravines (Luckman 1977). The avalanche transported considerable debris including talus and deadfall that resulted from an extensive fire in the mid-1970's (Grundstrom, per comm.).

A preliminary examination of the recurrence interval of avalanches within the potential mine site was conducted through a detailed study of the Paramount Zone avalanche. Sections of mature coniferous trees damaged in the 2007 avalanche were investigated for the presence of tree ring reaction wood and scarring, both indicators of past avalanche activity along the same slide path. An inventory of the oldest standing trees within the high velocity zone of the avalanche was also conducted to derive a minimum time since the last large avalanche.

2007 Paramount Zone Mt. LaCasse Avalanche

The winter of 2006 - 2007 in northwestern British Columbia was characterized as having one of the heaviest snowfalls on record (F. Day, per comm.). At the Schaft Creek exploration camp buildings that had been standing since the 1970's collapsed under the weight of snow. Though not observed directly, the size, amount, and chaotic arrangement of the debris entrained by the Paramount Zone avalanche suggests that the avalanche was wet and dense.

The terrain characteristics in starting zone which influenced avalanche formation included low ground surface roughness and low vegetation density and height. At the study site the northern starting zone is typified by a slightly concave slope with thin soil (primarily colluviated rockfall, and felsenmeer) and about 40% bedrock outcrop. Vegetation cover is sparse and consists of grasses, stunted fir and the occasional shrub willow.

The southern starting zone is somewhat more complex as a rock/sediment bench is present that may hinder the release of avalanches (Figure 18.19). The avalanche toe extends well into access road and drill pads and is primarily defined by oriented debris and accumulations of rafted deadfall (Figure 18.20).

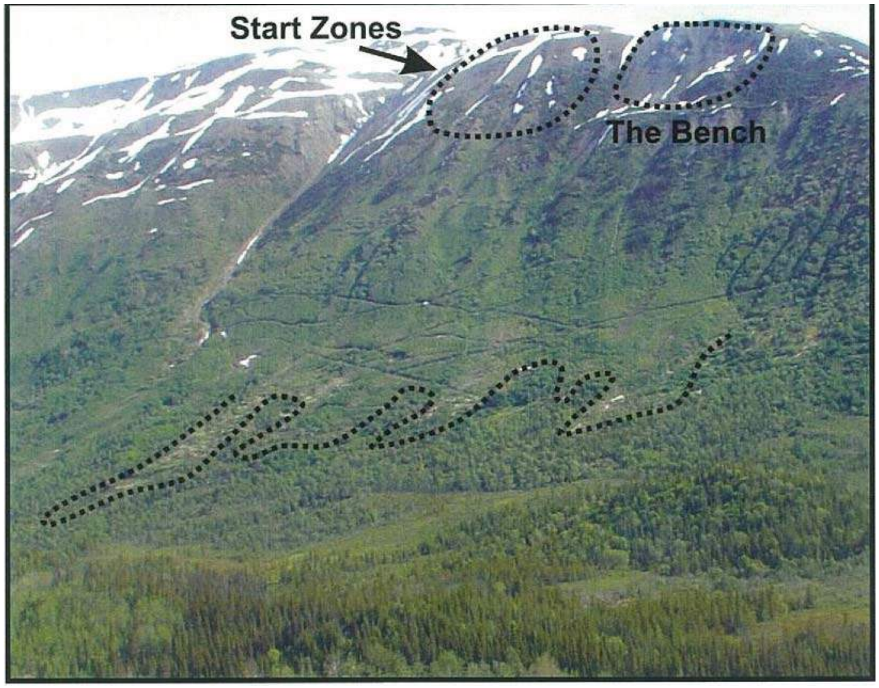


Figure 18.19 Avalanche scars and start zones for 2007 Paramount Zone Avalanches.

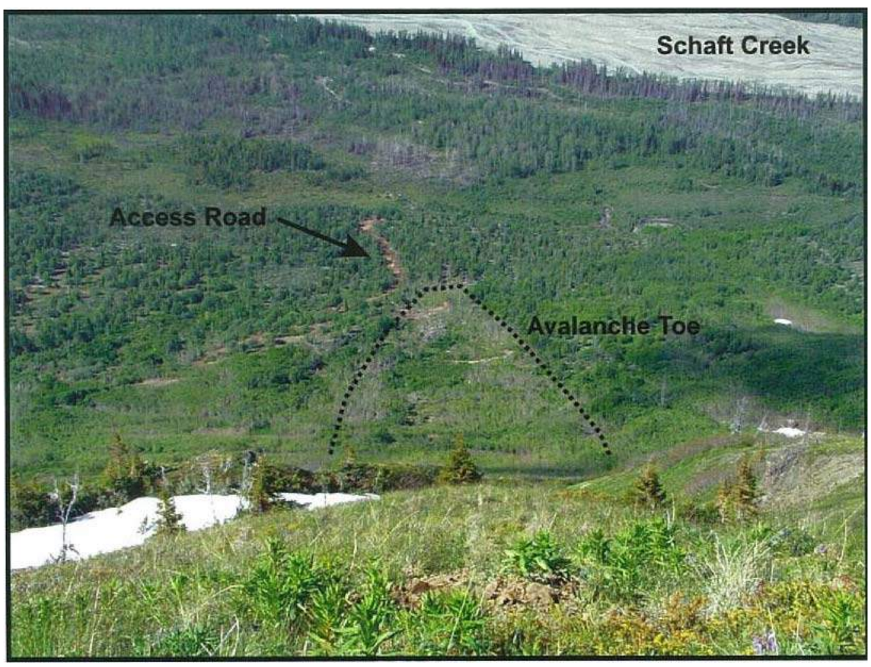


Figure 18.20 Paramount Zone avalanche toe scar as seen from the start zone.

Results of Tree Ring Analyses

The data collected suggest four periods of major avalanche activity since the trees that were sample became established. Almost all the trees began to grow in the early to mid-1970's, a consequence of re-establishment after a forest fire or after a particularly devastating avalanche.

The accuracy of this assessment is limited by the number of subalpine fir tree sections that were obtained in the study (nine) and the inherent error in using ring counting to determine both age and the time of the event that lead to the development of reaction wood. Both false rings and locally absent rings are known to occur in subalpine firs and can lead to some error.

18.6 Air Photo Interpretation of the Schaft Creek – Mt. LaCasse Region

18.6.1 Introduction

Dr. Ian Spooner, P. Geo, of Acadia University was retained by Copper Fox Metals Inc. to complete a preliminary qualitative air photo interpretation of the Schaft Creek development site and adjoining areas. Black and white air photos taken in 1965 at 1:32,000 scale were viewed in stereo in order to delineate structural and surficial features in the study area. The interpreted features were plotted on a mosaic of high resolution air photos obtained exclusively for Copper Fox Metals Inc.

Air photo interpretation using stereo photo pairs (with 60% common overlap incoverage) is recognized as one method that can be used to elucidate structural and geomorphic features that may not be evident during a field mapping program. In particular the exaggerated topography produced by the parallax effect heightens the topographic contrast that occurs along faults and stratigraphic boundaries. The scale of the photos used is dependent on availability as well as the size of the study area. In this study the 1965 coverage provided the largest scale coverage (1:32.000) of the study area. Black and white air photos are often desirable as the tonal contrast between features is often more evident and easier to interpret than in colour air photos. However, subtle colour differences that may be associated with boundaries, mineralization or surface cover change may not be evident.

18.6.2 Faulting

Faults evident through air photo analysis have been subdivided into major and minor faults on the basis of both persistence and fidelity. Both types of faults are common in the Skeeter Lake Valley, a relationship that was also noted by Logan and Drobe (1993). The larger faults are > 4 km in length and are steeply dipping as they comprise strong curvilinear lineaments. The BB deposit (Logan and Drobe, 1993) west of Skeeter Lake appears to be related to a cluster of particularly prominent faults and, perhaps, the proximity to Late Triassic intrusive rocks. This site in particular may warrant further investigation.

Three prominent faults on the south shoulder of Mt. LaCasse may be of particular interest as they cut across several prominent contacts. Their surface geometry also suggests that they are near vertical and may have a direct affect on the geometry of the deposit.

18.6.3 Lineaments and Bedding

Lineaments are defined as linear topographic features on the earth's surface. They may be positive (such as dike) or negative (such as a fault controlled valley). Lineaments are very common in the study area and may represent one of many features including minor faults, contacts, dykes, bedding, and surficial features related to glaciation or mass wasting where it is fairly certain that they represent bedding the lineament has been colour coded as such (see accompanying map; Figure 18.21). Cryptic lineaments to the south of the Schaft Creek deposit most likely represent major faults that are obscured by the relatively thin overburden cover in the region (Figure 18.21 & Figure 18.22).

Bedding attitude is highly variable and may represent the prevalence of tight folds particularly in volcanic units. Bedding on the southeastern slope of Mt. LaCasse is generally steeply dipping (eastwards and westwards) whereas bedding on the southwestern slope in the vicinity of the Schaft Creek deposit displays more moderate dips and appears broadly correlative to contact trends. These observations suggest a broadly synclinal structure. Lineaments near the summit of Mt. LaCasse likely represent both minor (or obscured) faults and dikes.

Figure 18.21 High resolution air photo (2006) of Schaft Creek deposit site.

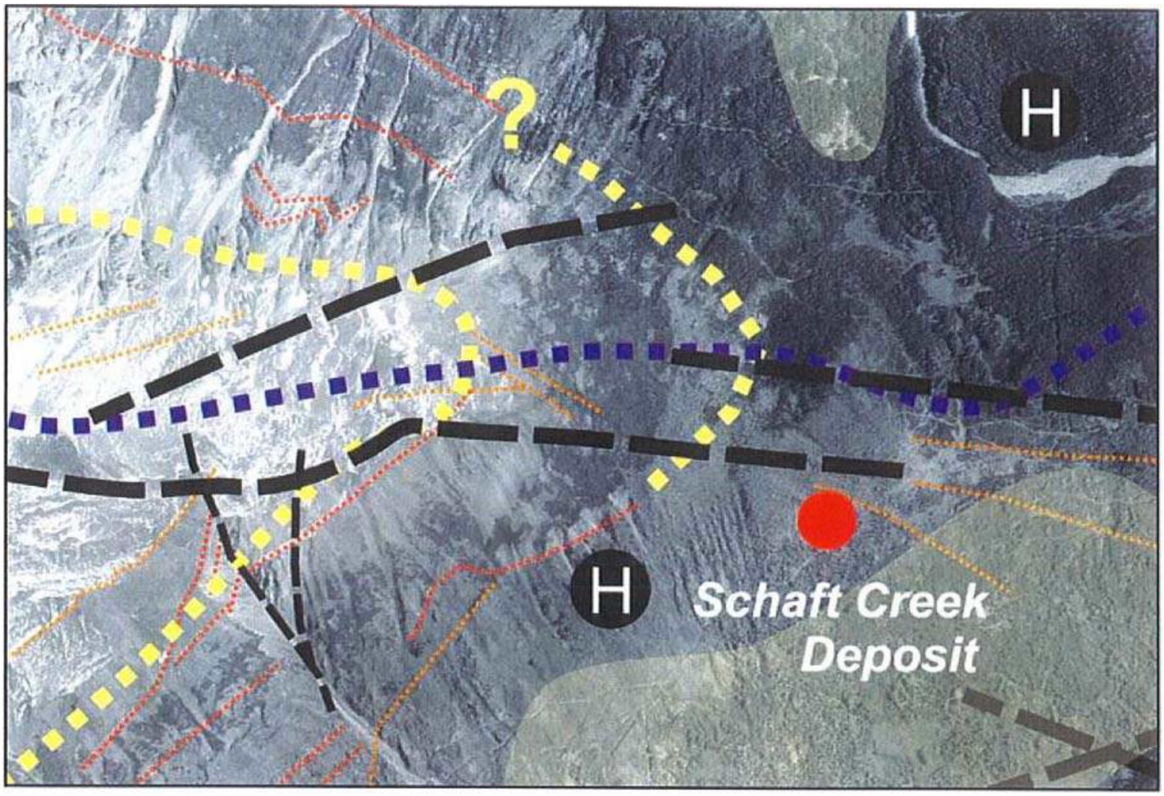


Figure 18.22 1965 air photos overlaid on 2006 composite air photo image.

18.6.4 Contact Relationships and Unit Morphology

The contact between the intrusive rocks in the western portion of the study region and the volcanic rocks to the east is evident where slopes are steep and overburden cover is thin. In general the intrusive rocks appear to be more homogeneous and do not display the bedding or layering characteristic of the various volcanic facies. This contact is obscured in the vicinity of the SCD due to significant overburden cover. Contacts between the various volcanic facies are, for the most part cryptic though contact relationships noted by Logan and Drobe (1993) on the ridge above the SCD appear also to be evident as distinct changes in lithological character. On the basis of both the contact and lineament geometry it appears the southern portion of Mt. LaCasse displays strongly layered stratigraphy.

A significant contact between the Triassic volcanic rocks and carboniferous rocks of the Stikine Assemblage is evident in the eastern portion of the study area (Figure 18.23 & Figure 18.24).

This contact is evident both as a strong lineament {most likely a fault) and also as a distinct change in lithological character. The Carboniferous rocks display knob and kettle topography that is poorly drained, an indication that these rocks may be susceptible to solution weathering and the development of karst topography. Consequently, the valley between Skeeter Lake and Start Lake to the south displays a low gradient and a contorted watershed divide. The carboniferous rocks appear to be only mildly deformed into gentle folds.

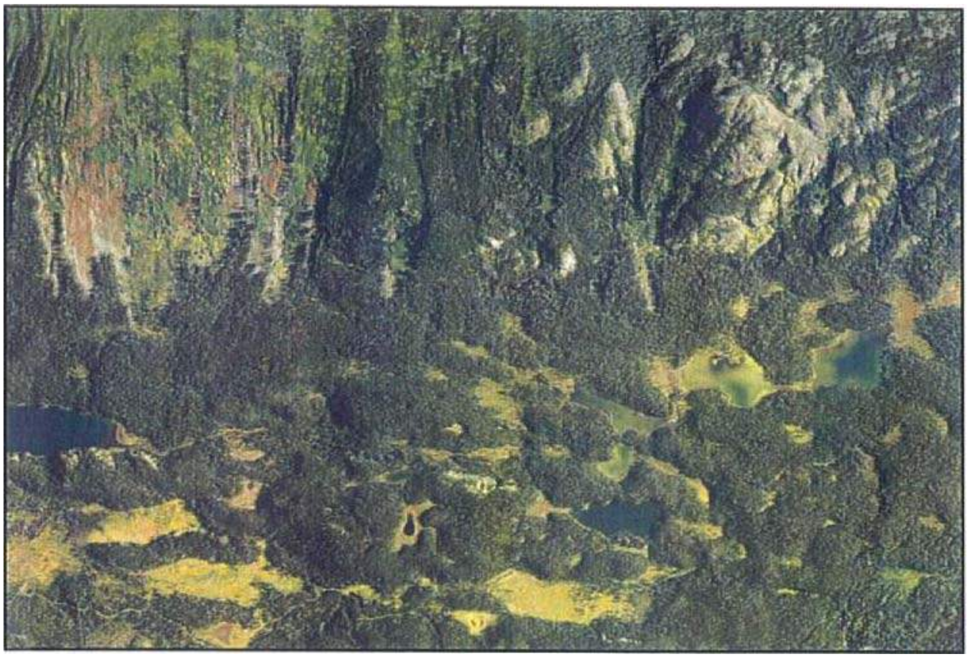


Figure 18.23 Terrain between Skeeter Lake and Start Lake.

High resolution colour air photo of the terrain between Skeeter Lake and Start Lake. Solution weathering is evident as lakes and fens with disconnected drainage. The knob and kettle topography may be accentuated by stagnant ice meltout till deposition. Ground truthing is required.

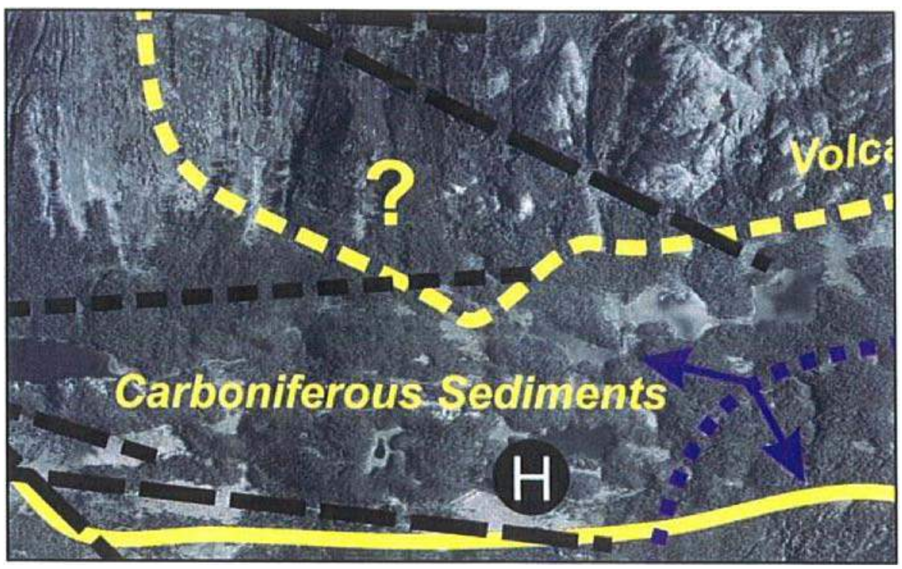


Figure 18.24 Carboniferous sediments between Skeeter Lake and Start Lake.

Interpreted extent of carboniferous sediments (karst susceptible terrain) between Skeeter Lake and Start Lake (north is to the left). Note the east-west orientation of both faults and contacts. The dashed line represents a local watershed boundary.

18.6.5 Surficial / Glacial Geology

Three sites in which glacial, alluvial or colluvial sedimentation is extensive have been noted and arc outlined in yellow on the accompanying map. These regions generally show weak lineament development and lack bedding or contact indicators. The terrain east of Start Lake is most likely overlain by glacial deposits whereas strata on the southwestern slope of Mt. LaCasse are covered by colluviated and alluvial sediments.

There are few directional indicators of Wisconsinan (Fraser) glaciation in the study area. The Skeeter Lake Valley may have been isolated from the major trunk glaciers occupying the Schaft Creek and Mess Creek Valleys and, as such was subjected to limited basal erosion. Glacial sediment in the valley floor may be largely dominated by meltout or stagnation facies.

Mass wasting (debris flows, rock and soil creep, rotation failure, sakung) is common in the study are and has resulted in both the exposure and concealment of stratigraphy. Potential hazard sites that were noted by Spooner (2007) have also been noted in the air photo analyses.

18.6.6 High Resolution Air photo Collage (Fall, 2006)

An interpretation of the high resolution airphotos obtained in 2006 was not the focus of this study (as stereo pairs of this imagery were not available); however, a black and white composite of these photos was used as the map base for the air photo interpretation. As such, a few comments regarding the high resolution image have been included.

Several sites were noted on the image that displayed anomalous colouring of indeterminate origin. Especially interesting were areas near intrusive contacts that display subtle blue or orange shading (Figure 18.25). Theses locales may represent areas of alteration worthy of future study. In some cases lineaments that were not evident on the 1965 stereo pairs were noted. These lineaments may be evident due to changes in vegetation cover or due to colour variation too subtle to be recognized in the black and white stereo pairs. An in-depth analysis of the high resolution colour imagery may be of some value.

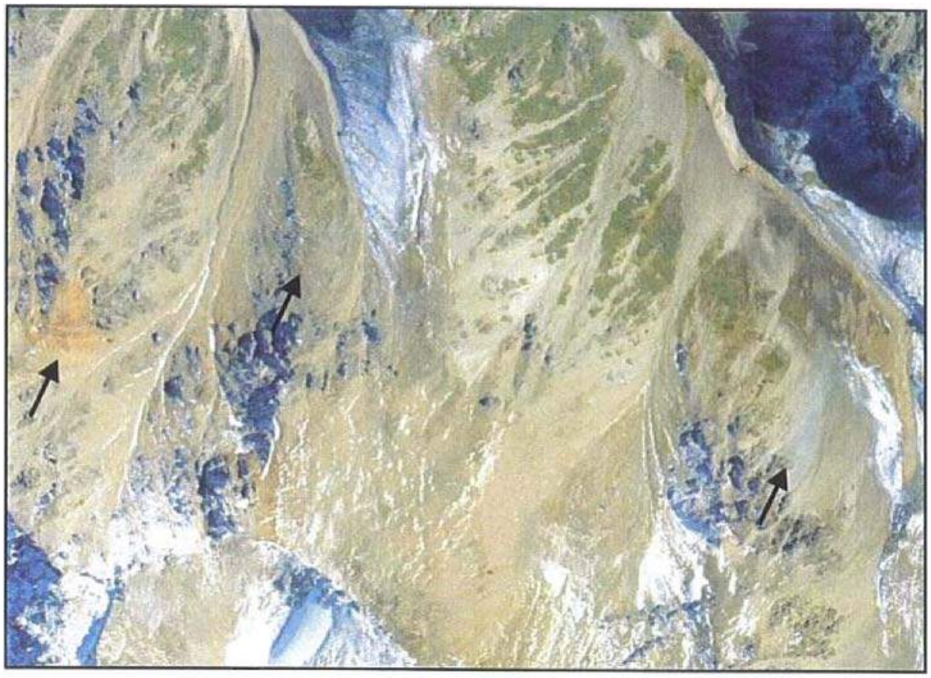


Figure 18.25 A portion of the colour air photos obtained in Fall, 2006.

A portion of the colour air photos obtained in Fall, 2006. The arrows point to terrain that displays anomalous colouring (gossan) and may be worthy of future study.

19.0 Interpretations and Conclusions

This report is a preliminary economic assessment (PEA), by which meaning the report is a preliminary assessment 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 prefeasibility study.

By the CIM Definition Standards on Mineral Resources and Mineral Reserves, a mineral reserve has to be supported by at least a prefeasibility study demonstrating economic viability of the project. It is recognized that the term “ore” cannot be used unless it is associated with a mineral reserve, however, the word “ore” is used throughout this document to refer only to mineralized material within the resource and mill feed that would be delivered to and processed in the proposed concentrator.

19.1 Resource

The Schaft Creek Deposit has been explored extensively prior to its acquisition by Copper Fox Metals Inc. In order to validate the historic drilling database, a large component of the 2005 and 2006 drilling programs was to twin older drill holes. Analyses of the twinned holes have yielded satisfactory results, and AGL is relatively confident in the accuracy of the historic database.

The Quality Assurance/Quality Control (QA/QC) procedures currently being practiced by Copper Fox Metals Inc. at Schaft Creek are well within industry recognized standards.

The current mineral resource estimate has been prepared according to the CIM Definition Standards on Mineral Resources and Mineral Reserves. A substantial resource base has been identified and classified in the measured, indicated and inferred mineral resource categories.

Table 19.19.1
Schaft Creek Mineral Resource Estimate Summary
≥0.20 % Copper Equivalent Cut-Off

	Tonnes	Cu (%)	Mo (%)	Au (g/t)	Ag (g/t)	CuEq Grade (%)
Measured Mineral Resources	463,526,579	0.30	0.019	0.23	1.55	0.46
Indicated Mineral Resources	929,755,592	0.23	0.019	0.15	1.56	0.36
Measured + Indicated Mineral Resources	1,393,282,171	0.25	0.019	0.18	1.55	0.39
Inferred Mineral Resources	186,838,848	0.14	0.018	0.09	1.61	0.25

It is the considered opinion of AGL that mineralized material below a copper equivalent cut-off grade of 0.20% at Schaft Creek cannot be considered as mineral resources as they are potentially uneconomic; therefore, only mineral resources ≥0.20% copper equivalent cut-off have been reported.

Overburden material has been found to contain metal values. Further work is warranted to determine the possible recovery of these metals from an economic perspective, and whether or not they are present as recoverable sulphides or tied-in with silicates.

Very little tonnage is attributed to the resource base from the Low Grade Zone above a 0.20% CuEq cut-off. By definition, this zone sits outside the 0.2% Cu cut-off modeled for the three zones and therefore should not contain much tonnage.

Several of the last holes from the 2006 drilling program have not been included in the current resource estimate due to time constraints. The current resource estimate has already delineated a large quantity of measured, indicated, and inferred material. As such, the remaining holes should be added to the model but are not expected to materially alter the results of the resource estimate.

19.2 Mining

A valid production schedule based on 65,000 tpd mill feed schedule at a Preliminary Economic Assessment (PEA) level is developed for the Schaft Creek mine. Detailed pit phases are engineered from the results of a Lerchs-Grossman (LG) sensitivity analysis. Pit delineated resources include a 5% mining dilution applied at the contact between ore and waste and 10% Mining Losses.

Table 19.19.2
Schaft Creek Pit Delineated Resource

Classification	RUN OF MINE	DILUTED GRADES			
	Tonnes	CU %	AU g/t	AG g/t	MO %
Measured	379,900,000	0.317	0.238	1.72	0.019
Indicated	337,900,000	0.289	0.195	1.83	0.020
Inferred	1,300,000	0.265	0.101	2.05	0.018
Total	719,100,000	0.304	0.217	1.77	0.020

The mine planning work is based on the Resource model provided by Associated Geosciences Ltd. The 3D Block model from AGL is converted and subsequent mine planning for the Schaft Creek mineral property is based on work done with MineSight® a suite of software well proven in the Industry. This includes the resource model, pit optimization (Minesight Economic Planner, MS-EP), detailed pit design, and optimized production scheduling (Minesight Strategic Planner, MS-SP).

In addition to the geological information used for the block model, other data used for the mine planning includes the base economic parametres, mining cost data derived from supplier estimates and data from other projects in the local area, conservative slope angles, and anticipated project metallurgical recoveries, plant costs and throughput rates.

19.3 Metallurgy & Process

While the Schaft Creek resource is geologically unique, the mineralized material may be classified as typical copper/moly porphyry with significant gold and silver values. The results of all of the recent and historical metallurgical testing indicate that the Schaft Creek resource responds positively to the typical grinding and conventional flotation method for copper/molybdenum porphyries. The results, depending on metallurgical test conditions, are consistently in the range of 25 to 32% copper concentrate grades at attendant recoveries of 85 to 93%. Acid Base Accounting (ABA) on the flotation tailings indicates an excess of neutralization potential. These results form a solid basis for flowsheet development. Process optimization and metallurgical efficiency improvement are an on-going process and are in progress.

The process plant, as designed, will handle 65,000 mtpd through a typical crushing and two stage grinding circuit. The primary flotation circuit will produce a bulk concentrate which will be reground and further upgraded in a four stage cleaner circuit. Final bulk concentrate will then be further separated into a copper concentrate and a molybdenum concentrate.

Almost 100% of the gold and silver values in the Bulk Concentrate should report to the Copper Concentrate.

The final Bulk Concentrate indicates no deleterious levels of antimony, arsenic, bismuth, lead or Zinc.

The occurrence of rhenium (Re) in the Schaft Creek ore has been established by previous owners. Recent assays and mineralogical observation of one moly concentrate sample indicate that the moly concentrate may contain a significant amount of Re.

19.4 Access Road

The 2006 field reconnaissance provided a better understanding of the road design and construction constraints on the Mess Creek access route which resulted in a preliminary road corridor. All bridge and major culvert crossings were inspected and the most suitable stream crossings located using hand held GPS units. Terrain features such as rock outcrops, talus slopes, and wetlands were recorded and geohazards such as rock falls, landslides and avalanche paths noted.

19.5 Environmental

Work on the Schaft Creek EA Application is progressing on schedule; submission is set for end of 2008. Copper Fox has signed a number of agreements relating to the environmental assessment process with the Tahltan Nation and is working with them to integrate comments and identify potential environmental and social issues.

The Schaft Creek project is advancing through the BC environmental assessment process and has engaged local, regional, provincial and federal regulators since the outset of the EA process. Regulator comments have been incorporated into 2007 baseline studies to ensure all regulatory requirements will be met through the EA Application.

The Schaft Creek project is part of the Telegraph Creek Community Watershed and therefore all mineral exploration, including road construction, maintenance and deactivation, is to be conducted according to the guidelines for community watersheds outlined in Mineral Exploration Code. Copper Fox has met or exceeded all of its environmental obligations to date.

The Memorandum of Understanding between Tahltan Nation Development Corporation and Copper Fox Metals Inc. is the successful first step towards a full Participation Agreement with the Tahltan First Nation.

19.6 Geotechnical

Based on the preliminary geotechnical assessment, the options identified for tailings disposal and rockfill disposal are considered feasible. Similarly, it is feasible to construct an open pit encroaching on the toe of Mount LaCasse. Significant issues have been identified.

Mass Wasting and Glacial Risks

Glacial outburst floods appear to represent a minor risk at the development site. A small proglacial pond at the head of Hickman Creek (Option B) may be worthy of extra study though an initial investigation did not indicate that large floods are likely from this site or have occurred in the recent past. An examination of glaciers and glacier forefields in other valleys indicated that outburst floods were unlikely in the past.

Debris flow corridors are common in the study area. Those located above the Main and Paramount Zones on the southwest slope of Mt. LaCasse represent a significant rockfall and debris flow hazard to pit development. Many large scale scarps were noted and some may represent sakung (large scale rock creep). A debris fan on the south side of The Saddle represents an active slope feature that should be avoided.

Avalanches are common in the study area. The steep alpine slopes near the summit of the southwest shoulder of Mt. LaCasse serve as excellent avalanche initiation zones. Deadfall that is prevalent on the lower slopes is readily incorporated into landslides increasing destructive potential. Preliminary dendrochronological analysis of reaction wood in subalpine fir tree sections in the main slide path indicates that significant avalanches occur every 10 years on average.

19.7 Key Results

Key results of this Preliminary Economic Assessment include:

- Measured & Indicated mineral resource: 1,393.3 million tonnes at a ≥0.20% copper cut-off grade;
- Inferred mineral resource: 186.8 million tonnes at a ≥0.25% copper cut-off grade;
- Measure, indicated and inferred pit delineated resource of 719.1 million tonnes
- LOM waste material of1,192.4 tonnes;
- LOM head grades: Cu = 0.304%, Mo = 0.020%, Au = 0.217 g/t, Ag = 1.761 g/t;
- 31 year mine life at a milling rate of 65,000 tonnes per day;
- Life of mine stripping ratio of 1.7:1;
- Preproduction capital cost of C$1,428.4 million;
- Total LOM capital cost of C$1,765.7 million;
- Operating cost of C$8.58 per tonne milled over the life of the project – includes mining, milling and G&A;
- Concentrate handling and treatment costs of C$2.68 per tonne milled over the life of the project;
- Metal recoveries: Cu = 90%, Mo = 72%, Au = 82%, Ag = 72%;
- Copper concentrate grades: Cu = 26.5%, Au = 18.4 g/t, Ag = 113.2 g/t, Mo = 0.27%;
- Moly concentrate grades: Mo = 54%, Cu = 0.42%;

- LOM copper production of 1,861.8 million tonnes;
- LOM moly production of 231.5 million pounds;
- LOM gold production of 3.9 million ounces;
- LOM silver production of 27.8 million ounces;
- Base case metal pricing: Cu = $1.50/lb, Mo = $10.00/lb, Au = $550/oz, Ag = $10.00/oz;
- Trailing three year average metal pricing: Cu = $2.66/lb, Mo = $27.00/lb, Au = $569/oz, Ag = $10.50/oz;
- Five year forecast metal pricing: Cu = $2.76/lb, Mo = $22.38/lb, Au = $700/oz, Ag = $12.00/oz;
- Base case pre-tax IRR of 7.5% with a 12 year payback;
- Base case pre-tax NPV of C$380 million at a 5% discount rate;
- Mine site production costs of C$0.57 per pound copper - net moly, gold, and silver credits;
- Trailing three year average (Case 2) pre-tax IRR of 32.7% with a 3 year payback;
- Trailing three year average (Case 2) pre-tax NPV of C$5,347 million at a 5% discount rate.

19.8 Project Risks

The project is located in the northern province of British Columbia, Canada and it is expected that there will be a relatively low degree of political, legal, or regulatory risk associated with the project. An assessment of this risk is beyond the scope of expertise of the authors of this report and accordingly no allowance for such risk has been included in the cost estimates or economic analysis for the project.

A project of this nature is also sensitive to several project risk factors that would be expected to potentially impact any major project of a similar size:

- Adverse weather conditions;
- Force majeure events;
- Late deliveries;
- Availability of equipment;
- Availability of materials;
- Availability of construction labour;
- Poor performance of contractors;
- Disputes with local residents;
- Disputes with NGO's;
- Escalation of costs;
- Foreign currency exchange rate fluctuations.

Copper Fox Metals has taken the proactive approach and elected to account for these risks with a Project Reserve Provision of C$300 million in the cost estimate and economic analysis for the project.

20.0 Recommendations

20.1 Resources

Associated Geosciences Ltd. recommends that Copper Fox continue with its plans to bring the project to a prefeasibility stage. Several of the last holes from the 2006 drilling program have not been included in the current resource estimate due to time constraints. As such, the remaining holes should be added to the model but are not expected to materially alter the results of the resource estimate.

20.2 Mining

Moose Mountain Technical Services recommends that Copper Fox continue with its plans to bring the project to a prefeasibility stage. The following items should be addressed in more detail as the project moves forward:

- Control surveys need to be completed;
- The current short supply with long delivery times for tires mean that supply contracts are required from tire manufactures before the end of 2007 to ensure that tires will be available in time for project start up. Tire supply remains a project risk;
- Backfill opportunities have been ignored in this study to avoid placing waste on areas where ongoing exploration may increase the future pit limit. Backfilling will significantly reduce haul costs and should be investigated further;
- Geotechnical analysis of the pit slopes is a critical path with significant impact to the outcome of this project. A detailed stability study should be carried out with recommendations for future prefeasibility studies;
- Hydro-Geology evaluation of the area is needed;
- More detailed ARD evaluation is needed;
- Metallurgical recovery assumptions need to be verified. Future mine studies should use recovered grades related to in-situ block grades. There is a chance that a variable metallurgical recovery will significantly alter the mine plan shown in this schedule.

20.3 Metallurgy and Process

Ray Hyyppa recommends that Copper Fox continue with its plans to bring the project to a prefeasibility stage. Additional metallurgical testing is recommended prior to completion of a prefeasibility study. These recommendations are as follows:

- While existing data indicates that the three resource zones have an average Ball Mill Work Index of approximately 20.6 (19.8 to 21.9), additional testwork on new samples should be done to further evaluate the hardness of the ore from each zone and to determine whether specific areas of each zone vary significantly;
- The current flowsheet is based on a traditional grinding circuit consisting of a Primary Crusher, a single Semi-Autogenous (SAG) Mill feeding three parallel Ball Mills. The SAG Mill is closed with two MP1000 pebble crushers. However, laboratory scale tests should be conducted to determine the benefit

of High Pressure Grinding Rolls (HPGR). These crushers are said to be well suited to hard ores such as those found at Schaft Creek;

- Pilot scale column flotation tests should be conducted to more closely size the Bulk Copper/Moly/Gold/Silver assumed in this study. All lab and pilot scale flotation machines simulated conventional flotation cells, but cleaner flotation cells used in this flowsheet have assumed the use of column flotation cells. Standard industry "rules of thumb", appropriate for a PEA level of study, were used to estimate the column flotation cell sizes;
- Additional lab scale flotation tests should be conducted on PQ core from the 2005 and 2006 drilling season to conduct locked cycle tests on feed samples of a wide range of copper grades. This will allow a more thorough understanding of the moly, gold and silver recovery relationships as a function of copper recovery;
- Additional lab scale flotation tests and mineralogical investigations are needed to understand the nature of a slime interference which occurs when floating samples, especially with the Paramount Zone. A clay-like mineral slows the flotation kinetics;
- It is likely that the 1st cleaner tailings need to be scavenged and the scavenger concentrate recycle back to the circuit. The scavenger tailings need to be combined with the primary scavenger tails as Final Tailings. This has not been adequately tested in the lab, but is the circuit assumed for this study;
- Lab scale moly separation tests need to be completed to better define the expected metal grades and recoveries to the Copper and Moly Concentrates;
-
- The moly concentrates produced from the 2006 core Liard, West Breccia and Paramount Zones should be analyzed by mass spectrometry to provide a better estimate of the revenue potential from rhenium;
- Additional thickener and filter sizing tests are required;
- Paste thickener tests should be conducted on final tailing to determine the applicability of this technology to Schaft Creek. Paste tailings would reduce the required tailings storage capacity. In addition, the possibility of co-mingling paste tailings with mine waste material offers the opportunity to completely eliminate the tailings impoundment;
- Additional lab scale tests need to be conducted to study at primary grinds coarser than 100 microns and the resultant recoveries. Coarser grinds would also greatly reduce the power requirements in the grinding circuit. Paste thickeners are more typically suited to slurries with an 80% passing size of 120 microns or coarser;
- Investigate increasing mill throughput to optimize costs and reduce mine life;
- Continue the investigation of alternative concentrate transportation methods. A concentrate slurry pipeline from the project site to a filter plant located in the Bob Quinn area would reduce the truck traffic on the access road;
- Continue the investigation of supplying diesel fuel to the project site via a pipeline from the Bob Quinn area. This would reduce the truck traffic on the access road;
- Evaluate the potential to produce copper cathode on-site vs. shipping copper concentrate off-site for treatment. Some preliminary testwork has been performed by Cominco Engineering Services Ltd. (CESL) with encouraging

results. Other options to treat copper concentrate that may warrant investigation include pressure oxidation, AARL/UBC, Dynatec, Activox, MIM Albion, BioCop and Galvanox;
- Trade-off study comparing truck hauling of ROM to the mill vs. a pit rim crusher and conveying to the mill.

20.4 Environmental

The environmental data collection for baseline studies and the subsequent environmental assessment of the Schaft Creek project is currently ongoing. Additional studies will need to be undertaken as required to complete the EA Application. Additional studies that will likely be required once the TOR is finalized include:

- Traffic study for Highway 37;
- Visual effects assessment;
- Water quality modeling;
- Hydrogeologic and hydrologic modeling;
- Human health;
- Terrestrial ecosystem mapping;
- Geohazard and terrain stability mapping.

2007 studies that should be continued into 2008 include:

- Archaeology;
- Water quality;
- Fisheries and aquatic biology;
- Hydrology;
- Hydrogeology;
- Metal leaching and acid rock drainage.

20.5 Access Road

A preliminary Mess Creek route is now available for further study. Future work necessary to advance this route should include detailed Environmental, Geotechnical, and Archaeological studies; consultation with First Nations and other stakeholders and an engineered design from a road centerline location survey.

20.6 Geotechnical

In order for the project to advance to the prefeasibility and feasibility stage, Copper Fox Metals needs to complete a program of geotechnical investigations related to tailings impoundment design, waste dump stability and open pit wall stability. Currently fieldwork for advanced level design is being completed before winter conditions set in. The program components include:

- Establish Tailings Management Parametres including, but not limited to:
  - design criteria including hydrologic and seismic criteria;
  - tailings water balance;
  - slurried tailings characteristics;

  - freeboard requirements;
  - reclaim/discharge plan;
  - requirement to submerge tailings/waste rock (Acid Rock Drainage (ARD) and potential;
  - concomitant metals leaching assessment;
  - spillway sizing;
  - timing of excess water release;
  - freshwater diversion system.
- All tailings dam design work should be conducted with regard to the Canadian Dam Safety Guidelines. The establishment of suitable management parametres would be undertaken concurrently with the overall tailings storage assessment;
- Assess Storage Capabilities and perform a screening level geo-hazard assessment for each proposed tailings site;
- Geotechnical Assessment;

  - Geotechnical drilling of impoundment foundations, containment areas and potential borrow sources;
  - Geological mapping of drill core and any associated fracture zones;
  - Geological outcrop mapping in areas not previously mapped;
  - Standard Penetration Testing (SPT) of overburden to assess soil density and collect samples;
  - Hydrogeological investigations including packer testing during drilling to assess bedrock hydraulic conductivities;
  - Install piezometres to assess groundwater elevations through conducting falling/rising head tests;
  - Seismic refraction transverses around the preferred tailings dam footprint are recommended in the spring of 2008 in support of feasibility-level design.

- Condemnation drilling of the ground below the proposed tailings impoundments to confirm the absence of mineralization;
- Geotechnical investigation of the proposed waste dump site foundations;
- Development of design criteria for pit wall stability.

20.7 General

Other areas that require further work and investigation as the project advances include:

- Investigate critical long lead items that could negatively impact the development schedule. These include:
  - Electric shovels;
  - Electric drills;
  - Haul trucks;
  - Large grinding mills;
  - Large crushers;
  - Regrind mills;
  - Large flotation cells;
  - Transformers;

- Switch gear.

- Labour resource for both construction and operation
- Power supply options, BC Hydro vs. on-site generation

A detailed scope of work and schedule has been developed for the environmental assessment process. It is recommended that this scope of work and schedule be implemented and progress in conjunction with the Schaft Creek feasibility studies.

As more project information is available, the current environmental and social baseline studies should be reviewed to ensure they adequately address potential impacts of the Schaft Creek project. This review should include input from government regulators, the Tahltan Nation and various stakeholders.

Copper Fox Metals has budgeted C$16 million for the advancement of this project. This includes monies for resource development, exploration, geotechnical, metallurgical testwork, access road, product marketing, etc.

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21.2 Glossary

Acid Generating Material (AGM): Materials that react with water and oxygen to form acids, such as mine tailings containing sulfides (*ex.* Pyrite) that react to form sulfuric acid.

Acid Rock Drainage (ARD): A natural occurance within some environments as part of the rock weathering process but is exacerbated by large-scale earth disturbances characteristic of mining and other large construction activities, usually within rocks containing an abundance of sulfide minerals. (See **Acid Generating Material**).

Adit: A nearly horizontal passage leading from the surface into a mine.

Allochthonous: An adjective for rocks, deposits, etc.; that are found in a place other than where they and their constituents were formed.

Anticline: A fold of rock layers that slope downward on both sides of a common crest. Anticlines form when rocks are compressed by plate-tectonic forces. They can be as small as a hill or as large as a mountain range.

Apophysis: A branch from a dike or vein.

ASTM: American Society for Testing and Materials.

Aqua Regia: a yellow, fuming liquid composed of one part nitric acid and three to four parts hydrochloric acid: used chiefly to dissolve metals as gold, platinum, or the like.

Batholith: A large mass of igneous rock that has intruded and melted surrounding strata at great depths. Batholiths usually have a surface area of over 100 km^2 (38 mi^2).

BCM: Bank Cubic Metre. One cubic metre of material as it lies in the natural state.

Bornite: (*a.k.a.* peacock ore) An important brownish-bronze, lustrous copper ore with the composition Cu_5FeS_4 that tarnishes to purple when exposed to air.

Breccia: A rock composed of angular fragments embedded in a fine-grained matrix. Breccias form from explosive volcanic ejections, the compaction of talus, or plate tectonic processes. Breccias are different from conglomerates in that the fragments they contain are angular instead of rounded.

Chalcopyrite: A brassy yellow, metallic, tetragonal mineral, usually occurring as shapeless masses of grains. Chalcopyrite is found in igneous rocks and copper-rich shales, and it is an important ore of copper. Because of its shiny look and often yellow colour, it is sometimes mistaken for gold, and for this reason it is also called fool's gold. Chemical formula: $CuFeS_2$.

Colluvium: Loose earth material that has accumulated at the base of a hill, through the action of gravity, as piles of **talus**, avalanche debris, and sheets of detritus moved by soil creep or frost action.

Comminution: To reduce to powder; pulverize.

Culvert: A drain or channel crossing under a road, sidewalk, etc.

Cyanidation: A highly controversial, though most commonly used, metallurgical technique for extracting gold from low-grade ore.

Dendrochronology: The science dealing with the study of the annual rings of trees in determining the dates and chronological order of past events.

Dip: The angle at which a stratum is inclined from the horizontal, measured perpendicular to the **strike** and in the vertical plane.

Drift: An approximately horizontal passageway in underground mining.

Drill Hole: A circular hole made by drilling either to explore for minerals or to obtain geological information.

Epizone: The zone of metamorphism characterized by moderate temperature, low hydrostatic pressure, and powerful stress. The outer depth zone of metamorphic rocks.

En Echelon: Describing parallel or subparallel, closely-spaced, overlapping or step-like minor structural features in rock, such as faults and tension fractures, that are oblique to the overall structural trend.

Exploration: The search for economic mineral by geological surveys, prospecting or use of tunnels, **drifts** or **drill holes**.

Facies: The appearance and characteristics of a sedimentary deposit, esp. as they reflect the conditions and environment of deposition and serve to distinguish the deposit from contiguous deposits.

Fault: A fracture in the continuity of a rock formation caused by a shifting or dislodging of the earth's crust, in which adjacent surfaces are displaced relative to one another and parallel to the plane of fracture.

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.

Fluvial: Features created by the actions of a river. Also called “glaciofluvial” when originating from the meltwater rivers of a glacier.

FOB: The acronym for “free on board”. The FOB price is the sales price of product loaded in a vessel at the port and excludes freight or shipping cost.

Freeboard: The height of the watertight portion of a structure (*ex.* tailings dam) above a given level of water in a river, lake, etc.

Front End Loader: A tractor or wheeled type loader having a shovel or bucket that dumps at the end of an articulated arm located at the front of the vehicle.

Geophysical Log: A graphic record of the measured or computed physical characteristics of the rock section encountered by a probe or sonde in a drill hole, plotted as a continuous function of depth. Also commonly referred to as an e-log.

Geohazards: Naturally occuring destructive forces such as volcanoes, earthquakes, landslides, avalanches, and tsunamis.

Geotextiles: Permeable fabrics which, when used in association with soil, have the ability to separate, filter, reinforce, protect, or drain. Applications include roads, airfields, railroads, embankments, retaining structures, reservoirs, canals, dams, bank protection and coastal engineering.

Glacial Outburst Flood: a sudden and often catastrophic flood that may occur during a volcanic eruption, or when a lake contained by a glacier or a terminal **moraine** dam fails. This can happen due to erosion, a buildup of water pressure, an avalanche of rock or heavy snow, an earthquake or cryoseism, or if a large enough portion of a glacier breaks off and massively displaces the waters in a glacial lake at its base.

Gossan: An exposed, oxidized portion of a mineral vein, especially a rust-coloured deposit of mineral matter at the outcrop of a vein or orebody containing iron-bearing materials.

Greenfield: A project which lacks any constraints imposed by prior work, with no need to demolish or remodel any existing structures (*i.e.* new construction).

Highwall: The unexcavated face of exposed overburden and ore in an opencast mine or the face or bank of the uphill side of a contour strip-mine excavation.

Imbrication: A sedimentary structure in which flat pebbles are uniformly tilted in the same direction.

Isopach: A line drawn on a map connecting all points of equal thickness of a particular geologic formation.

LCM: Loose Cubic Metre. One cubic metre of material as it lies in a post-disturbed state, such as a stockpile.

Lease: A contract between a landowner and a lessee, granting the lessee the right to search for and produce ore upon payment of an agreed rental, bonus and/or royalty.

Little Ice Age (LIA): The period from about 1400-1900 a.d., characterized by expansion of mountain glaciers and cooling of global temperatures, especially in the Alps, Scandinavia, Iceland, and Alaska. The Little Ice Age followed the Medieval Warm Period.

Mass Wasting: (See **Slope Creep**)

Mineable: Capable of being mined profitably under current mining technology, environmental, and legal restrictions, rules and regulations.

ML: Metal Leaching.

Molybdenite: A soft, lead-gray hexagonal mineral that is the principal ore of molybdenum. It occurs as sheetlike masses in pegmatites and in areas where contact metamorphism has taken place.

Moraine: A mass of **till** (boulders, pebbles, sand, and mud) deposited by a glacier, often in the form of a long ridge. Moraines typically form because of the plowing effect of a moving glacier, which causes it to pick up rock fragments and sediments as it moves, and because of the periodic melting of the ice, which causes the glacier to deposit these materials during warmer intervals. A moraine deposited in front of a glacier is a *terminal moraine*. A moraine deposited along the side of a glacier is a *lateral moraine*. A moraine deposited down the middle of a glacier is a *medial moraine*. Medial moraines are actually the combined lateral moraines of two glaciers that have merged.

Ore: A mineral, rock, or natural product serving as a source of some metallic substance (ex. copper, gold, etc.), nonmetallic substance (ex. Sulfur), or a native metal, that can be mined at a profit. The term “ore” cannot be used unless it is associated with a mineral reserve, however, the word “ore” is used throughout this document to refer only to mineralized material within the resource and mill feed that would be delivered to and processed in the proposed concentrator.

Orography: The study of the physical geography of mountains and mountain ranges.

Outcrop: Economic mineral, which appears at or near the surface; the intersection of ore with the surface.

Overburden: Waste earth and rock covering a useful or economic mineral deposit.

Permeability: The capability of a porous rock or sediment to permit the flow of fluids through its pore spaces.

Preliminary Economic Assessment (PEA): A preliminary assessment 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 prefeasibility study

pH: The potential of hydrogen. Numerically, it is the logarithm of the reciprocal of hydrogen ion concentration in gram atoms per litre of solution. Qualitatively, this is a measure of the acidity or alkalinity of a solution, numerically equal to 7 for neutral solutions, increasing with increasing alkalinity and decreasing with increasing acidity. The pH scale commonly in use ranges from 0 (highly acidic) to 14 (highly alkaline, or basic).

Physiography: The study of the natural features of the earth's surface, especially in its current aspects, including land formation, climate, currents, and distribution of flora and fauna.

Porosity: The ratio, expressed as a percentage, of the volume of the pores or interstices of a substance, as a rock or rock stratum, to the total volume of the mass.

Porphyry: An igneous rock containing the large crystals known as phenocrysts embedded in a fine-grained matrix.

Reaction Wood: Formed by a woody plant in response to mechanical stress, and helps to position newly formed parts of the plant in an optimal position. This stress may be the result of wind exposure, excess of snow, soil movement, avalanches, etc. The reaction wood appears as asymmetric growth. The cambium in the affected part of the trunk is more active on one side, leading to thicker growth rings.

Reclamation: The restoration of land at a mining site after the ore has been 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 drill pipe and drill collars 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 (ROM): The ore produced from the mine before it is separated and any impurities removed.

Slope Creep: (*a.k.a.* Downhill creep, or commonly just creep) The slow downward progression of rock and soil down a low grade slope; it can also refer to slow deformation of such materials as a result of prolonged pressure and stress. Creep may appear to an observer to be continuous, but it really is the sum of numerous minute, discrete movements of slope material caused by the force of gravity. Friction being the primary force to resist gravity is produced when one body of material slides past another offering a mechanical resistance between the two which acts on holding objects (or slopes) in place. As slope on a hill increases, the gravitational force that is perpendicular to the slope decreases and results in less friction between the material that could cause the slope to slide.

Stockwork: A metalliferous deposit characterized by the impregnation of the mass of rock with many small veins or nests irregularly grouped. Such deposits are typically worked in floors or stories.

Strike: The direction of the line formed by the intersection of the bedding plane of a bed or stratum of sedimentary rock with a horizontal plane.

Strip Ratio: The overburden material (tonnes) that must be removed to provide a unit weight of ore (tonne). In general, the lower the strip ratio, the more likely an ore body is to be mined by open pit methods.

Surface Mining: Methods of mining at or near the surface. Includes mining and removing ore from open cuts with mechanical excavating and transportation equipment and the removal of capping overburden to uncover the ore.

Syncline: A fold of rock layers that slope upward on both sides of a common low point. Synclines form when rocks are compressed by plate-tectonic forces. They can be as small as the side of a cliff or as large as an entire valley.

Tahltan Nation: (*a.k.a.* Nahanni) refers to a Northern Athabaskan people that live in northern British Columbia around Telegraph Creek, Dease Lake, and Iskut.

Tailings: Waste that has been separated from the ore in the metallurgical processing plant.

Tailings Impoundment: a body of tailings confined within an enclosure or behind a dam.

Talus: Sharp, irregular rock fragments that have accumulated at the base of a cliff or slope. The concave slope formed by such an accumulation of rock fragments is called a talus slope.

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.

Till: Unconsolidated, unstratified, and heterogeneous mixture of soil deposited by a glacier; consists of sand and clay and gravel and boulders mixed together.

Vug: A small cavity in a rock or vein, often with a mineral lining of different composition from that of the surrounding rock.

21.3 SI Prefixes

Power	Prefix	Symbol	Decimal Equivalent (in SI Writing Style)
10^{24}	yotta-	Y	1 000 000 000 000 000 000 000 000
10^{21}	zeta-	Z	1 000 000 000 000 000 000 000
10^{18}	exa-	E	1 000 000 000 000 000 000
10^{15}	peta-	P	1 000 000 000 000 000
10^{12}	tera-	T	1 000 000 000 000
10^{9}	giga-	G	1 000 000 000
10^{6}	mega-	M	1 000 000
10^{3}	kilo-	k	1 000
10^{2}	hecto-	h	100
10^{1}	deca-	da	10
10^{0}			1
10^{-1}	deci-	d	0.1
10^{-2}	centi-	c	0.01
10^{-3}	milli-	m	0.001
10^{-6}	micro-	µ	0.000 001
10^{-9}	nano-	n	0.000 000 001
10^{-12}	pico-	p	0.000 000 000 001
10^{-15}	femto-	f	0.000 000 000 000 001
10^{-18}	atto-	a	0.000 000 000 000 000 001
10^{-21}	zepto-	z	0.000 000 000 000 000 000 001
10^{-24}	yocto-	y	0.000 000 000 000 000 000 000 001

21.4 List of Abbreviations

Above mean sea level .... amsl
Ampere .... A
Annum (year) .... a
Bank cubic metre .... BCM
Cubic metre .... m^3
Day .... d
Days per week .... d/wk
Days per year (annum) .... d/a
Degree .... °
Degrees .... deg
Degrees Celsius .... °C
Diametre .... ø
Dry metric tonne .... dmt
Gram .... g
Grams per cubic centimetre .... g/cc
Grams per litre .... g/L
Grams per tonne .... g/t
Greater than .... >
GR Technical Services .... GR
Hectare (10,000 m^2) .... ha
Hertz .... Hz

Horsepower .... hp
Hour .... hr
Hours per day .... h/d
Hours per week .... h/wk
Hours per year .... h/a
Inch .... "
Joule (Newton-metre) .... J
Kilowatt-hour .... kWh
Kilowatt-hours per short ton (US) .... kWh/st
Kilowatt-hours per tonne (metric tonne) .... kWh/t
Kilowatt-hours per year .... kWh/a
Kilowatts adjusted for motor efficiency .... kWe
Less than .... <
Litre L
Litres per minute .... L/m
Loose cubic metres .... LCM
Megabytes per second .... Mb/s
Metre .... m
Metres above sea level .... masl
Metric tonne .... t
Metric tonne .... mt
Micrometre (micron) .... µm
Microsiemens (electrical) .... µS
Miles per hour .... mph
Million .... M
Million metric tonnes (megatonne) .... mmt
Minute (plane angle) .... '
Minute (time) .... min
Month .... mo
Newton .... N
Ohm (electrical) .... Ω
Ounce (troy) .... ozt
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
Potential of Hydrogen (*i.e.* acidity or alkalinity level) .... 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 metre .... m^2
Tonne (1,000 kg) .... t
Tonnes per annum .... tpa
Tonnes per day .... tpd

Tonnes per hour .... tph
Total dissolved solids .... TDS
Total suspended solids .... TSS
Volt V
Volt-Ampere .... VA
Watt (Joules per second) .... W
Week .... wk
Weight/weight .... w/w
Wet metric ton .... wmt
Yard .... yd
Year (annum) .... a
Year (US) .... y

21.5 List of Acronyms

Bond Abrasion Resistance Test .... Ai
Average Daily Traffic .... ADT
Ammonium Nitrate .... AN
Ammonium Nitrate and Fuel Oil .... ANFO
Acid Rock Drainage .... ARD
American Society of Testing and Materials .... ASTM
All-terrain vehicle .... ATV
B.C. Environmental Assessment Act .... BCEAA
B.C. Environmental Assessment Office .... BCEAO
B.C. Forest Service .... BCFS
Bank Cubic Metre Waste .... BCMW
B.C. Utilities Commission .... BCUC
Brazil, Russia, India, China (and Chile) .... BRIC
Bond Ball Mill Work Index .... BWi
Construction cost index CCI
Canadian Council of Ministers of the Environment .... CCME
Canadian Environmental Assessment Act .... CEAA
Copper Fox Metals, Inc. .... CFM
Carrier, Insurance, and Freight .... CIF
Canadian Institute of Mining, Metallurgy, and Petroleum .... CIM
Canadian Pacific Railway .... CPR
Controlled Source Audio Magnetotellurics .... CSAMT
Copper Fox Metals, Inc. .... CUU
Bond Crushing Work Index .... CWi
Department of Fisheries and Oceans .... DFO
Diamond drill hole .... DDH
Discounted cash flow .... DCF
Environmental Assessment Office .... EAO
Floating Cone .... FC
Free On Board .... FOB
Free-swelling indices .... FSI
Gross Combined Vehicle Weight .... GCVW
Gross Domestic Product .... GDP
General Mine Expense .... GME
Global Positioning System .... GPS
Geological Survey of Canada .... GSC
Gridded Surface File .... GSF
Hardgrove indices .... HGI
High-Pressure Grinding Rollers .... HPGR
International Monetary Fund .... IMF
Independent Qualified Person .... IQP
Internal Rate of Return .... IRR
International Organization for Standardization .... ISO
Japanese Industrial Standards .... JIS
Little Ice Age .... LIA
Land and Resource Management Plan .... LRMP
Lands and Water B.C. .... LWBC
Life of Mine .... LOM
Loose Cubic Metres .... LCM
Liard Zone .... LZ
Migratory Bird Convention Act .... MBCA
Ministry of Energy Mines and Petroleum Resources .... MEMPR

A Windows-Based Process Simulator .... METSIM
Magnetic IP .... MIP
Metal Leaching .... ML
Main Liard Zone .... MLZ
Minesight® Economic Planner .... MS-EP
Minesight® Strategic Planner .... MS-SP
Magnetotellurics .... MT
North American Datum .... NAD
Non-Governmental Organization .... NGO
North Liard Zone .... NLZ
National Topographic System .... NTS
Paramount Zone .... PZ
Organization for Economic Cooperation and Development .... OCED
Qualified Person .... QP
Bond Rod Mill Work Index .... RWi
Rock Quality Designation .... RQD
Semi-Autogenous Mill + Ball Mill + Pebble Crusher .... SABC
System for Electronic Document Analysis and Retrieval .... SEDAR
Standard Penetration Testing .... SPT
Sub-Surface Deposition .... SSD
Transportable Moisture Limit .... TML
Tahltan Nation Development Company .... TNDC
Universal Transverse Mercator .... UTM
Valued Ecosystem Component .... VEC
Very Low Frequency Electromagnetic .... VLF-EM
West Breccia Zone .... WBZ
Ministry of Water, Land and Air Protection .... WLAP
West Liard Zone .... WLZ

22.0 Date and Signature Pages

CERTIFICATE OF QUALIFICATION

I, Matt R. Bender, P.E., QP (Metallurgy), Director of Process, Mining & Metals, employed by Samuel Engineering Inc., 8450 East Crescent Pkwy. Suite 200, Denver, CO., 80111 do hereby certify:

- I am a full time employee of Samuel Engineering, Inc.;
- I am a registered Professional Engineer (P.E.), Metallurgical;
- I am a graduate of the Colorado School of Mines;
- I am a member in good standing of; Mining and Metallurgical Society of America (MMSA); and Society for Mining, Metallurgy and Exploration (SME);
- I have practiced my profession since 1987;
- 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;
- I am responsible for coordinating the study and the primary author of this report: "Technical Report: Preliminary Economic Assessment on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada";
- I visited the Schaft Creek site on July 16th, 17th, and 18th, 2007 and again on September 13th, 2007;
- I have had no prior involvement with the property that is the subject of the Technical Report;
- I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101;
- 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;
- As of the date of this certificate, to best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading;

Dated this 7th day of December 2007 in Denver, Colorado, USA

Original signature on file

____
Signature of Qualified Person

Matt R. Bender
Director of Process, Mining & Metals

CERTIFICATE OF QUALIFICATION

I, Keith M^{c}Candlish, P.Geo., QP, Director of Process, Vice President & General Manager, employed by Associated Geosciences Ltd., Suite 415, 708-11th Avenue S.W., Calgary, Alberta, do hereby certify:

- I am a full time employee of Associated Geosciences Ltd.;
- I am a registered Professional Geologist (P.Geol.) and a registered Professional Geoscientist (P.Geo.);
- I am a member in good standing of the Asscocitation of Professional Engineers, Geologists and Geophysicists of Alberta and the Association of Professional Engineers and Geoscientists of British Columbia;
- 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;
- I am the primary author of the geology and resource setions of this report: "Technical Report: Preliminary Economic Assessment on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada";
- I visited the Schaft Creek site on two separate occasions;
- I have had prior involvement with the property that is the subject of this Technical Report (Updated Resource Estimate for the Schaft Creek Deposit, dated June 22, 2007);
- I am independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101;
- 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;
- As of the date of this certificate, to best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading;

Dated this 7th day of December 2007 in Calgary, Alberta, Canada

Original signature on file

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Signature of Qualified Person

Keith M^{c}Candlish
Vice President & General Manager

Moose Mountain Technical Services

CERTIFICATE of QUALIFICATION

I, James H Gray P.Eng., a Principal of Moose Mountain Technical Services, 1584 Evergreen Hill SW Calgary Alberta T2Y 3A9 do hereby certify that:

1. I am Principal Engineer – Mining for Moose Mountain Technical Services;
2. I am a graduate of the University of British Columbia with a Bachelor of Applied Science Degree in Mineral Engineering in 1975;
3. I am a registered Professional Engineer (PEng) with APPEGA and APEGBC;
4. I am a member of the Canadian Institute of Mining and Metallurgy;
5. I have been practicing as a Professional Engineer for over 30 years with relevant experience for the Technical Report including:
   1975 to 1978 Underground stope mining, Mine Supervision, and Mine Engineering positions in operations in Canada and Australia
   1978 to 1989, mine site engineering, operations and management positions, costing, evaluating new mineral projects and development properties.
   1989 to present, mine engineering consultant work on assessment and feasibility studies of numerous coal, base metal, industrial mineral, and precious metal deposits in Canada, United States, Mexico, Bolivia, Chile, Argentina, Peru, Turkey, Iran, Greenland, and Australia.
6. I have read the definition of a "Qualified Person" as set out in National Instrument 43-101 of the Canadian Securities Administrators and certify that by reason of education, experience, independence, and affiliation with a professional association, I meet the requirements of a "Qualified Person";
7. I am responsible for the Mine Engineering aspects of the study including pit designs, dump designs, and mine production scheduling based on information supplied by others and certain assumptions as stated in the report "Technical Report: Preliminary Economic Assessment on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada" dated November 30, 2007;
8. I have visited the Schaft Creek site on September13th, 2007;
9. I have no prior involvement with the property that is the subject of the Technical Report;
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. 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;

Dated this 7th day of December 2007 in Calgary, Alberta, Canada

Original signature on file

J.H. Gray P.Eng

CERTIFICATE OF QUALIFICATION

I, Raymond R. Hyyppa, P.E., QP (Metallurgy), consulting metallurgical engineer for Hyyppa Engineering, LLC with an office located at 5887 West Atlantic Place, Lakewood, Colourado, 80227 do hereby certify:

- I am a full time employee of Hyyppa Engineering, LLC;
- I am a registered Professional Engineer (P.E.), Wyoming (#2900) and Colourado (#17314);
- I am a graduate of the Montana School of Mines with a B.S. Degree in Mining Engineering, 1965, a B.S. Degree in Mineral Dressing Engineering, 1965 and Montana College of Mineral Science & Technology with a B.S. Degree in Mineral Dressing Engineering, 1968;
- I am a member in good standing of; The Society for Mining, Metallurgy and Exploration (SME) and the Extractive Metallurgy Chapter of Denver;
- I have practiced my profession since 1965;
- 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 Qualified Person as defined in National Instrument 43-101;
- I am responsible for the metallurgical testwork and process sections of this report: “Technical Report: Preliminary Economic Assessment on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada”;
- I visited the Schaft Creek site on October 16th, 17th, 18th and 19th, 2006;
- I have performed consulting services to Copper Fox Metals, Inc since August 17, 2006, the company that is the subject of the Technical Report;
- I am not independent of the issuer applying all of the tests in Section 1.4 of National Instrument 43-101;
- 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;
- As of the date of this certificate, to best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical report not misleading;

Dated this 7th day of December 2007 in Denver, Colorado, USA

Original signature on file

____
Signature of Qualified Person

Raymond R. Hyyppa
Hyyppa Engineering, LLC

23.0 Additional Requirements for Technical Reports on Development Properties & Production Properties

This report is a preliminary economic assessment (PEA), by which meaning the report is a preliminary assessment 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 prefeasibility study.

By the CIM Definition Standards on Mineral Resources and Mineral Reserves, a mineral reserve has to be supported by at least a prefeasibility study demonstrating economic viability of the project. It is recognized that the term “ore” cannot be used unless it is associated with a mineral reserve, however, the word “ore” is used throughout this document to refer only to mineralized material within the resource and mill feed that would be delivered to and processed in the proposed concentrator

23.1 Mining Section

Section 23.1 was prepared by Mr. Jim Gray, P. Eng. – Moose Mountain Technical Services

23.1.1 Summary

Unless specified otherwise, $ are Canadian dollars (C$).

A production schedule based on 65,000 tpd mill feed schedule at a Preliminary Economic Assessment (PEA) level, is developed for the Schaft Creek mine. Detailed pit phases are engineered from the results of a Lerchs-Grossman (LG) sensitivity analysis. Pit delineated resources are tabulated below. The pit delineated resources in the table below include a 5% mining dilution applied at the contact between ore and waste and 10% Mining Losses. Dilution grades are estimated at 3.49 $/t NSR, 0.060 % Cu, 0.080 g/t Au, 2.150 g/t Ag and 0.004 % Mo representing the average grade of material below the incremental cut-off grade.

Cut-off grade for the Phase reserves in the below Table 23.1 is $4.25/t Net Smelter Return (NSR).

Table 23.1
Summarized MII Pit Delineated Resource for Schaft Creek

PHASE	RUN OF MINE	WASTE	ROM	DILUTED GRADES				
			S/R	NSR	CU	AU	AG	MO
	(mT)		(t/t)	$/t	%	g/t	g/t	%
P616	68.8	40.1	0.6	16.6	0.377	0.299	2.03	0.017
P626i	105.2	122.3	1.2	14.4	0.340	0.213	1.61	0.017
P636i	252.0	330.3	1.3	12.6	0.272	0.220	1.54	0.017
P646	15.7	17.1	1.1	15.6	0.314	0.279	2.44	0.023
P656i	82.8	111.5	1.3	14.0	0.298	0.196	1.98	0.024
P666i	194.6	571.1	2.9	13.9	0.302	0.191	1.93	0.024
Total	719.1	1,192.4	1.7	13.8	0.304	0.217	1.77	0.020

Measured, Indicated and Inferred pit delineated resources at Schaft Creek are summarized in the below Table 23.2.

Table 23.2 Pit Delineated MII Resource at Schaft Creek					
Classification	RUN OF MINE	DILUTED GRADES			
	(kT)	CU %	AU g/t	AG g/t	MO %
Measured	379.9	0.317	0.238	1.72	0.019
Indicated	337.9	0.289	0.195	1.83	0.020
Inferred	1.3	0.265	0.101	2.05	0.018
Total	719.1	0.304	0.217	1.77	0.020

23.1.2 Introduction

The mine planning work is based on the Resource model provided by Associated Geosciences Ltd. (AGL). The 3D Block model from AGL is converted and subsequent mine planning for the Schaft Creek mineral property is based on work done with MineSight® a suite of software well proven in the Industry. This includes the resource model, pit optimization (Minesight Economic Planner, MS-EP), detailed pit design, and optimized production scheduling (Minesight Strategic Planner, MS-SP).

In addition to the geological information used for the block model, other data used for the mine planning includes the base economic parametres, mining cost data derived from supplier estimates and data from other projects in the local area, conservative slope angles, and anticipated project metallurgical recoveries, plant costs and throughput rates.

23.1.3 Mining Datum

The historical drill hole information and topography are based on various surveys with different sets off control. Effort has been made to ensure the Mine design is using the most up to date topography, in conjunction with the infrastructure planning and that the drill hole data base is congruent with the topography surface.

23.1.3.1 *Grade Model Topography Surface*

The historical geology and topographic data for Schaft Creek has used various survey grids which have been converted to a single basis.

23.1.4 Mine Planning 3D Block Model and MineSight Project

The Resource model is converted to MineSight from AGL's Surpac 3D Block model used for the Resource Statement in the Technical Report dated June 22, 2007. The Surpac model has two mineralized zones and uses sub-blocking to increase the ore resolution. The two zones are for the separate modeling of the higher grade and deeper lower grade zones in the deposit. It is noted by AGL that material in the low grade zone below 0.2% Cu should be considered waste.

The MineSight 3D block model converts the Sub-blocked model to full blocks (25m x 25m x 15m) and retains the ore resolution by storing an ORE% for each full block based on the sub-block splits. The ORE% method allows a smaller 3DBM while retaining the resolution. This is important for the ultimate economic pit runs since LG (& Whittle in Surpac) has to use whole blocks anyway and with out ORE% the resolution would be lost. The MineSight 3DBM contains the following Block items:

Table 23.3
3D Block Model Setup

ITEM	DESCRIPTION	
TOPO	Percent of block below topography	
ZONE1 & 2	Mineral Zone1 = Higher Grade	West Breccia = 1 Main = 2 Paramount = 3
	Mineral Zone2 = Lower Grade	If CU>= 0.2% = 4 If CU < 0.2% = 5
ORE1 & 2	Percent of whole block that is ore (Grade assigned)	
AG1 & 2	Interpolated Silver Grade (g/t)	
AU1 & 2	Interpolated Gold Grade (g/t)	
CU1 & 2	Interpolated Copper Grade (%)	
Mo1 & 2	Interpolated Molybdenite Grades (%) as Mo	
CUEQ1 & 2	Calculated Copper Equivalent grade (%)	
SG1 & 2	Interpolated SG	
RES1 & 2	Resource Class	Measured = 1 Indicated = 2 Inferred = 3
NET	Net dollar value of the block based on NSR	Re-calculated each LG run
NSR	Net Smelter Return on whole block basis	Wt'd avg. NSR1 + NSR2
NSR1 & 2	Net Smelter Return including NSP & Rec for each metal	Sum (NSPx X Recx)

The project and model dimensions are:

Model Limits

	Minimum	Maximum	Size	Number
X	378900	381500	25	104
Y	6358400	6361900	25	140
Z	430	1705	15	85

23.1.4.1 *Net Smelter Return (NSR)*

Cutoff grades are determined using the Net Smelter Return (NSR) in $/tonne which is calculated using Net Smelter Prices (NSP). The NSR (Net of offsite concentrate and smelter charges and On-Site mill recovery) is used as a cutoff item for break-even ore/waste selection and for the grade bins for cashflow optimization. The net smelter price is based on base case metal prices, US$ exchange rate, and Off-Site transportation, smelting, and refining charges, etc. (see Appendix D5-3). The metal prices and resultant NSP's used are:

	Metal Price (US$)	NSP (C$)
Cu	1.50 $/lb	1.30 $/lb
Au	550 $/oz	16.48 $/g
Ag	10 $/oz	0.269 $/g
Mo	10 $/lb	7.57 $/g

Metallurgical recoveries used for the NSR calculation are:

- Cu Recovery = 90%
- Au Recovery = 81%
- Ag Recovery = 72%
- Mo Recovery = 80%

The NSR formula is:

$$NSR = \left(\frac{Cu\%}{100} * \frac{R_{Cu}}{100} * P_{Cu} * \frac{lb}{kg}\right) + \left(M_{Au} * \frac{R_{Au}}{100} * P_{Au}\right) + \left(M_{Ag} * \frac{R_{Ag}}{100} * P_{Ag}\right) + \left(\frac{Mo\%}{100} * \frac{R_{Mo}}{100} * P_{Mo} * \frac{lb}{kg}\right)$$

NSR = CU(%) x 0.90 x $1.30/lb x 22.046 + AU(g/t) x 0.81 x $16.48/g + AG(g/t) x 0.72 x $0.26/g + MO(%) x 0.80 x $7.57/lb x 22.046

23.1.4.2 *Mining Loss and Dilution*

The Schaft Creek deposits are to be mined with large truck/shovel operations, and an ore mining rate of 65,000 tpd feeding a conventional copper concentrator. The mining is described as typical hard rock bulk mining method. Large equipment will be used and high mining rates are planned to ensure the lowest possible unit costs for mine operations. Selective mining methods will not be used. The waste and ore will require blasting and typical grade control methods using blasthole sampling and possibly blasthole Kriging will be used to determine cut-off grades and digging control limits for the mining shovels. Blast heave, the lack of loading selectivity, haul back in the trucks, and stockpile reclaim will create some ore loss (mining recovery) and dilution as the material moves from In-Situ modeled resource to ROM mill feed.

Since the ROM mill feed determines the production schedule and revenue stream for the project, proper evaluation of the mining loss and dilution is required. The definition of the mining parametres used in the reserves calculations are also a NI 43-101 reporting requirement.

The 3D Block Model (3DBM) for Schaft Creek, updated for this study, is based on separate Lithological / Geostatistical domains, There are two ore zones per block with two Copper (Cu), Gold (Au), Silver (Ag), and Molybdenum (Mo) grade values for each block. As such the grade values in each block are not 'whole block diluted'.

With the planned bulk mining method, a means of determining the mining loss and dilution applicable to the Schaft Creek Resource model is needed that will reflect the ROM production from the mining operations. Mineralized zones in the 3DBM are made up of relatively large contiguous blocks of 'ore' above the cutoff grade. There are areas however where isolated blocks of ore are surrounded by waste and also isolated blocks of waste that are surrounded by ore. Higher cutoff grades will result in fewer contiguous blocks and more isolated blocks. Conversely lower cutoff grades will merge more of the indicated isolated blocks into close-by contiguous blocks.

Mining operations will use blasthole samples on 6 to 8 metre spacing to determine the cutoff boundaries for shovel dig limits. "Included" ore and waste blocks on the small blasthole sampling grid will be too small to separate from the shovel face especially after being displaced by blasting. This inclusion of isolated blasthole blocks is handled as the larger blocks in the 3D block model are averaged in to larger 3DBM.

The 3DBM uses 25m x 25m x 15m blocks for this stage of long range planning. Each block represents 25,031 tonnes which is 4 to 5 hours of digging for the shovels, and the plant feed will be approximately 2.4 blocks per day. With blocks of this magnitude, it can be assumed that isolated blocks from the larger 3DBM will be selectively mined and will not be lost or included in the ore. However bulk mining will cause dilution to the blocks, either ore into waste or waste into ore by neighboring blocks, where contact is made between ore grade material and waste.

Other mining losses are also noted in mining operations mainly due to misdirected loads, haul back in frozen truck boxes, and stockpile cleanup. These types of losses are small but need to be accounted for.

The mining resources will be calculated from the Resource model, within an economic pit limit using the applicable mining recovery and dilution parametres. The mining recovery and dilution parametres, in effect, convert the in place "pit delineated resource" to ROM resource tonnes. As stated above it is the ROM tonnes that are required for the production schedule which in turn is used to develop the project cashflows; therefore, the tonnes used in calculating the economic pit limit needs to be based on the ROM. The resources in the model are quantified as ore or waste based on a NSR cutoff.

Mining recovery and dilution parametres are required to account for the following:

- Dilution of waste into ore where blasting “throws” waste into ore at ore/waste boundaries.
- Loss of ore into waste where blasting “throws” ore into waste diluting the mix below cut off grade.
- General mining losses due to haul back from frozen or sticky material in truck boxes, misdirected loads, and repeated handling such as stock pile reclaim.

For this Preliminary Economic Assessment (PEA) an allowance has been made for a mining dilution of 5% applied at the contact between ore and waste dilution and a 10% mining loss.

Since the dilution material on the contact edge of the blocks described above is mineralized, it will have some grade value. The dilution grades are estimated by determining the grades of the envelope of waste in contact with ore blocks inside the pit delineated area. This is estimated by statistical analysis of grades in blocks below the design basis cutoff of $4.25/t. The dilution grade was estimated at 3.49 $/t NSR, 0.060 % Cu, 0.080 g/t Au, 2.150 g/t Ag and 0.004 % Mo representing the average grade of material below the incremental cut-off grade.

23.1.5 Economic Pit Limits, Pit Designs

23.1.5.1 *Introduction*

The economic pit limit is determined using the MS-EP optimization routines in MineSight which are based on the Lerchs Grossman (LG) algorithm. The LG algorithm runs against the 3D Block model, evaluating the costs and revenues of the blocks within potential pit shells. The routine uses input costs, net smelter prices, plant recoveries, and overall slope angles, and expands downwards and outwards from previous interim economic 3D surfaces, until the last increment is at break-even economics. Additional cases are included in the analysis to evaluate the sensitivities of prices, and slope angles. Block discounting for time value is also evaluated to determine the NPV effect of the delay between earlier stripping costs to the revenue released from deeper ore.

At this stage of the project (Scoping) typical pit slopes, and mining costs are used. metallurgical recoveries are based on initial metallurgical test work.

23.1.5.2 *Pit Slopes*

The rock is competent and an overall pit slope angle of 50° might be achievable; however, the ultimate east wall will be more than 1000 m high. This in the top 10 percentile in the world, and at this height the rock stress could become high enough to fail the rock, not just slip on weakness planes.

The Rock Mass Rating (RMR) of deep rocks, and detailed failure mode analysis should be determined and examined to determine safe pit slope constraints.

Hoek & Bray (1981) apparently use approx 45 deg as the safe 'Stability' line at 500m depth and an inferred extension to higher slopes to drop to 42 deg.

The design basis pit slope angle for the eastern ultimate wall is 42°.

Final highwall ramps will be on the west wall which is lower height on the ultimate pit limit. This will reduce stripping requirements. The LG design basis pit slope for the western ultimate wall is 40° to allow for the ramps.

23.1.5.3 *Mining Costs*

Mining unit costs have been estimated from similar large open pit mining projects in British Columbia. The unit mining cost assumptions are shown in the table below.

Table 23.4
Economic Pit Limit Estimated Unit Mining Costs

Area	**$C/t**
Drilling	0.100
Blasting	0.180
Loading	0.200
Hauling Waste	0.650
Dewatering	0.006
Roads & Dumps	0.200
Other Dir Mining	0.050
Sub Tot	**1.386**
Mine G&A	0.200
GME (Eng,Geol, Supervision)	0.020
Grand Total	**1.606**

23.1.5.4 *Sensitivity Cases*

The economic pit limits are based on the current cost and metal price assumptions. Since these economic parametres are estimates, the sensitivity of the ultimate economic pit limits need to be evaluated. This is done by varying the economic parametres in series of cases. The pit shells from these cases are also used to select pit pushbacks or phases. For each case being tested the series of LG pit shells are determined by keeping mining costs constant and varying the estimated net smelter metal prices (NSP).

Costs used for the LG series are:

- Unit Mining costs discussed above;
- Process + G&A Cost of $ 4.25 $/t ore.

The LG revenue also uses metallurgical recoveries described above.

23.1.5.5 *Sensitivity Cases*

The base case (100% case in the tables below) uses market prices of US$ 1.50/lb for copper, US$ 550/oz for gold, US$10.00/oz for silver and US$10/lb for molybdenum. The pit expansion cases are determined by varying the market price for all metals and the subsequent Net Smelter Price (NSP).

NSP deducts all off site costs including Smelting & Refining costs and concentrate transportation to smelter. The net result is revenue dollars available for the operation including mining, processing, general and administration costs.

The NSP for each LG Price Case is listed in the table below. The **Boldfaced** 100% case below is the base case for the Preliminary Economic Assessment (PEA) estimate. Other cases are variations on the base case.

Table 23.5
NSP for each LG case

Case	Market Prices				Net Price for Mine, Plant, & O/H			
	Copper	Gold	Silver	Moly	Copper	Gold	Silver	Moly
	US$/lb	US$/oz	US$/oz	US$/lb	$C/lb	$C/g	$C/g	$C/lb
45%	$0.68	$247.50	$4.50	$4.50	$0.43	$5.85	$0.09	$1.53
50%	$0.75	$275.00	$5.00	$5.00	$0.51	$6.82	$0.11	$2.08
55%	$0.83	$302.50	$5.50	$5.50	$0.59	$7.80	$0.12	$2.63
60%	$0.90	$330.00	$6.00	$6.00	$0.66	$8.76	$0.14	$3.18
65%	$0.98	$357.50	$6.50	$6.50	$0.75	$9.74	$0.16	$3.73
70%	$1.05	$385.00	$7.00	$7.00	$0.82	$10.70	$0.17	$4.28
80%	$1.20	$440.00	$8.00	$8.00	$0.98	$12.63	$0.21	$5.38
90%	$1.35	$495.00	$9.00	$9.00	$1.14	$14.56	$0.24	$6.48
100%	**$1.50**	**$550.00**	**$10.00**	**$10.00**	**$1.30**	**$16.48**	**$0.26**	**$7.57**
110%	$1.65	$605.00	$11.00	$11.00	$1.46	$18.41	$0.30	$8.67
130%	$1.95	$715.00	$13.00	$13.00	$1.78	$22.25	$0.37	$10.87
150%	$2.25	$825.00	$15.00	$15.00	$2.10	$26.09	$0.43	$13.07

23.1.5.6 *Economic Pit Limit Sensitivity to Low Grade Ore*

Sensitivity of the economic pit limit to the low grade ore in the Low Grade zone has been tested. As described above AGL has designated a low grade zone and has recommended that any material with copper grade < 0.2% within this zone be designated as waste. This zone is designated as Zone2 in the MineSight version of the resource model and tagged =4 if Cu >= 0.2% or =5 if Cu <0.2%. The two cases to test the economics of the low grade zone are:

1. The first case (A24) uses only the revenues from mineralized material from all ZONE1 and ZONE2 = 4.
2. The second case (A15) uses revenues from all mineralized material, including all of ZONE1 and 2 based on their economic merits.

A comparison of the two cases is presented below. The table below shows In-situ tonnages and waste for Pit A24.

Table 23.6 Pit A24 (Wasting Low Grade zone when Cu<0.2%)						
CLASS	**kTonnes**	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Mo (%)**	**CuEQ (%)**
Measured	432,027	0.319	0.237	1.719	0.019	0.482
Indicated	457,742	0.292	0.198	1.849	0.021	0.450
Inferred	2,595	0.312	0.113	1.957	0.180	0.422
Total	**892,363**	**0.305**	**0.217**	**1.787**	**0.020**	**0.465**
Waste	1,055,139	S/R (t/t)	1.18			

The table below shows In-situ Resource tonnages and waste for Pit A15. This indicates the pit is larger than A24 but only marginally.

Table 23.7 Pit A15 (Including all of the low grade zone in economic calculation)						
CLASS	**kTonnes**	**CU (%)**	**AU (g/t)**	**AG (g/t)**	**MO (%)**	**CUEQ (%)**
Measured	432,678	0.319	0.237	1.719	0.019	0.4818
Indicated	482,637	0.292	0.197	1.857	0.021	0.4490
Inferred	3,233	0.315	0.126	1.980	0.019	0.4322
Total	**918,547**	**0.305**	**0.2157**	**1.792**	**0.020**	**0.4644**
Waste	1,346,482	S/R (t/t)	1.47			

The sections below show Pit A24 in green and the A15 pit in purple. There is only a marginal pit expansion by including the low grade zone.

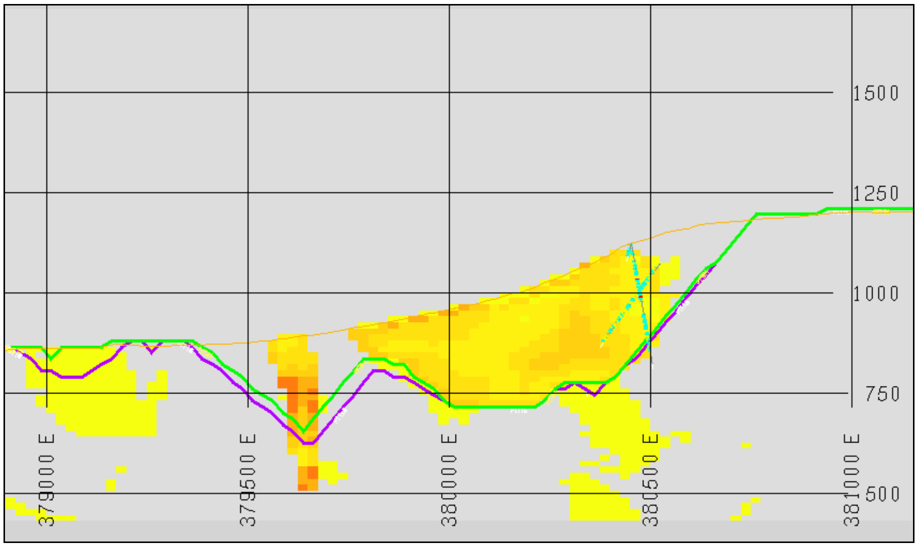


Figure 23.1 EW Cross Section at 6359612.5 N Showing LG pits

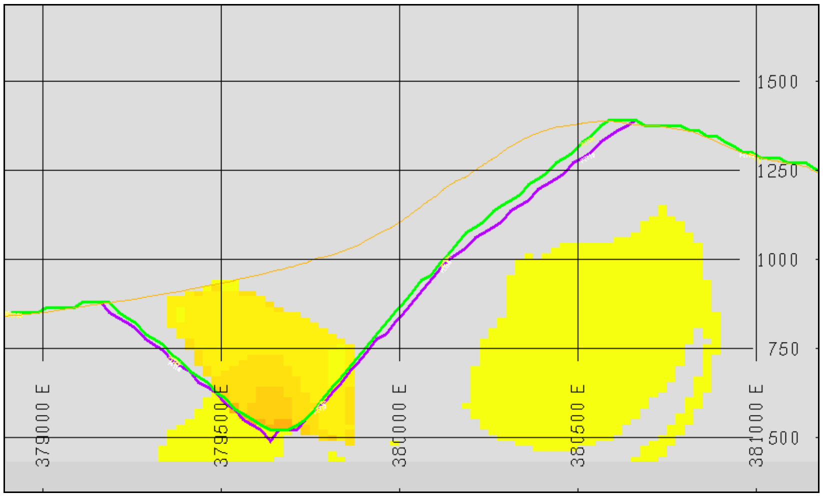


Figure 23.2 EW Cross Section at 6360612.5 N Showing LG pits

The material between the A24 and A15 pits is lower grade and at high strip ratio as shown in the table below verifying what is shown on the above sections.

Table 23.8
Incremental Pit Resource Pit A15 – Pit A24

CLASS	kTonnes	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	CuEQ (%)
Measured	651	0.302	0.133	1.509	0.017	0.412
Indicated	24,895	0.287	0.177	2.009	0.022	0.440
Inferred	638	0.329	0.182	2.075	0.021	0.475
Total	**26,184**	**0.288**	**0.176**	**1.991**	**0.022**	**0.440**
Waste	291,344	S/R (t/t)	11.13			

The comparison shows that the economic pit limit is not sensitive to the inclusion of ore < 0.2 % Cu in the Low Grade ZONE2. The economic pit limit design basis therefore omits material < 0.2 % Cu in the Low Grade ZONE2. At this stage of the project assessment this is a valid conservative assumption since the effect of low grade on the metallurgical recoveries of all the metals in consideration. Further met testing will allow better evaluation of this material at more advanced levels of study.

23.1.5.7 *East Highwall Pit Slope Sensitivity*

As discussed above the ultimate east wall may be more than 1000 m high. In the absence of a detailed geotechnical analysis the sensitivity of the economic pit limit to pit slope requires examination.

Two cases are examined

1. **Highwall pit slope = 50^o**
2. **Highwall pit slope = 42^o**

In-situ Resource at a waste/ore cutoff grade of $ 4.25/t NSR for each pit case for these two pit slope cases are compared in the figure below. These cases treat ZONE2 as waste.

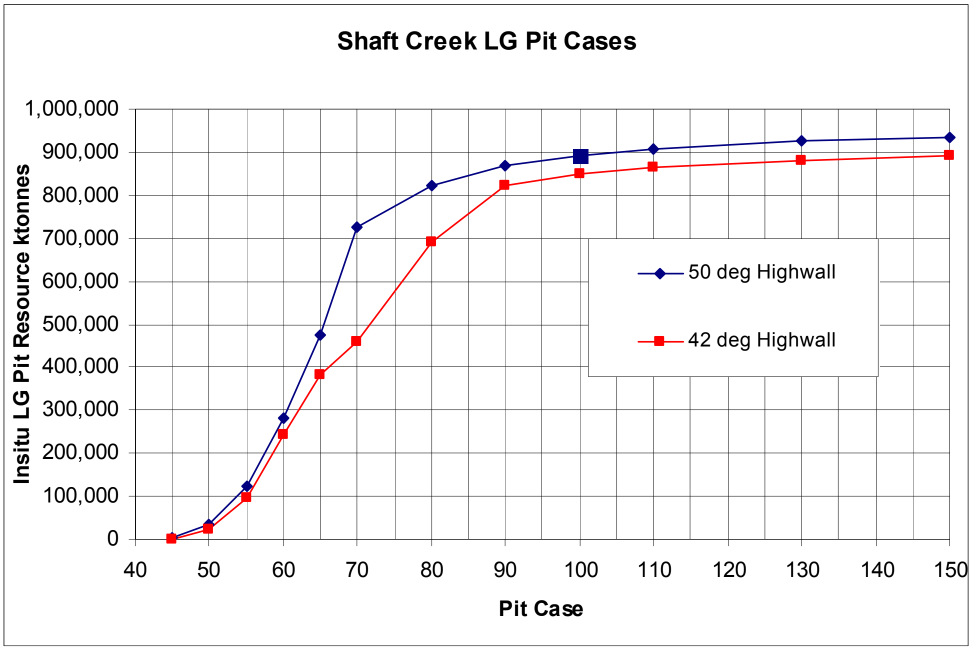


Figure 23.3 LG sensitivity to East Highwall pit slope

The pit slope sensitivity shows that the economic pit limit is marginally sensitive to pit slope, and as a result the more conservative 42^{o} highwall pit slope assumption should be used until recommendations are obtained from a detailed geotechnical study.

23.1.5.8 *Economic Pit Limits*

The pit resources for the Schaft Creek sensitivity cases are shown in the graph below. Each pit shell is cumulative of the previous case shown before it.

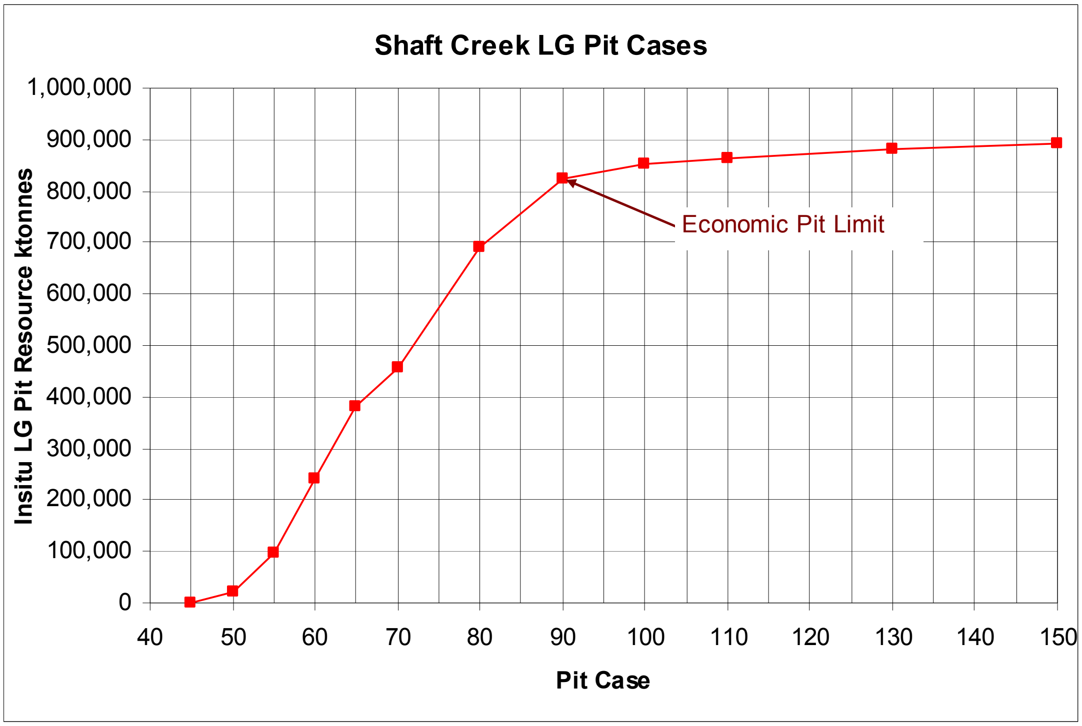


Figure 23.4 LG pit cases using a 42° pit slope angle

Table 23.9 Cumulative LG Insitu Pit Resources									
Pit	**% Price**	**ORE**	**ORE**	**ORE**	**ORE**	**ORE**	**ORE**	**Waste**	**Stripping**
Shell	**Case**	**kT**	**NSR**	**Cu**	**Au**	**Ag**	**Mo**	**kT**	**Ratio**
pt216	45	1,554	19.979	0.4478	0.4545	3.0560	0.0134	312	0.20
pt217	50	23,332	19.409	0.4285	0.4019	2.6248	0.0187	14,704	0.63
pt218	55	95,950	16.581	0.3832	0.2945	2.0406	0.0179	47,782	0.50
pt219	60	242,686	14.969	0.3438	0.2545	1.7777	0.0178	107,649	0.44
pt220	65	381,816	14.279	0.3244	0.2421	1.6940	0.0177	198,947	0.52
pt221	70	458,379	14.035	0.3179	0.2330	1.6803	0.0180	281,241	0.61
pt222	80	691,157	13.457	0.3043	0.2094	1.6677	0.0187	662,197	0.96
pt223	90	824,145	13.588	0.3037	0.2108	1.7373	0.0196	1,117,018	1.36
pt224	100	850,726	13.588	0.3035	0.2107	1.7466	0.0196	1,231,865	1.45
pt225	110	863,428	13.581	0.3034	0.2104	1.7502	0.0196	1,296,400	1.50
pt226	130	882,229	13.587	0.3038	0.2099	1.7561	0.0196	1,427,195	1.62
pt227	150	890,810	13.578	0.3036	0.2096	1.7586	0.0196	1,497,681	1.68

Price Case 90 (90% of base case metal prices) using a 42^{o} East Highwall pit slope has been chosen as the economic pit limit for Schaft Creek. There is no opportunity to significantly increase the economic pit limit at higher prices.

The figures below show the plan view of the 90% LG Pit Case relative to the Plant site for orientation, and selected cross sections showing the 90% LG Pit case.

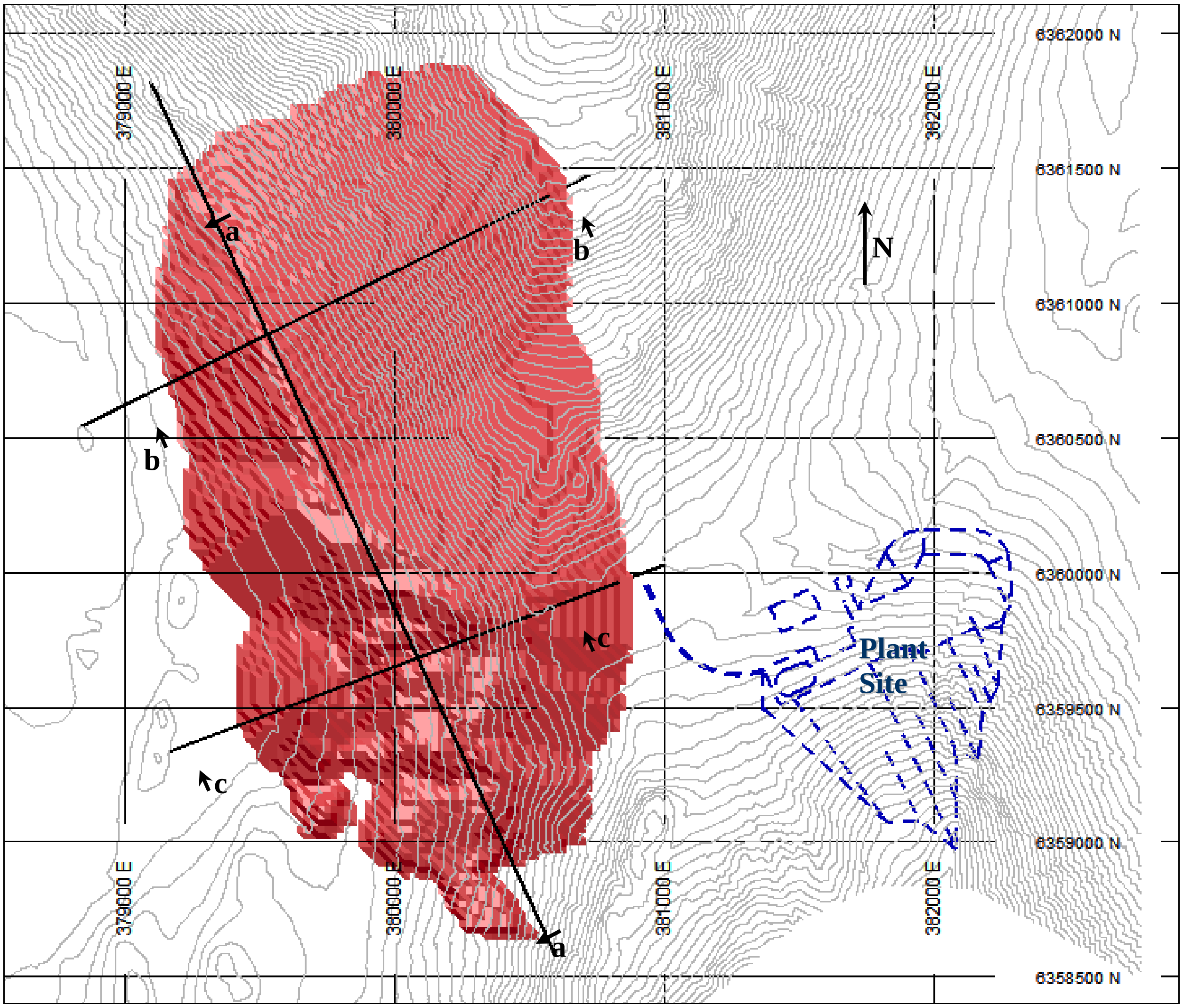


Figure 23.5 LG Pit Limit Plan View

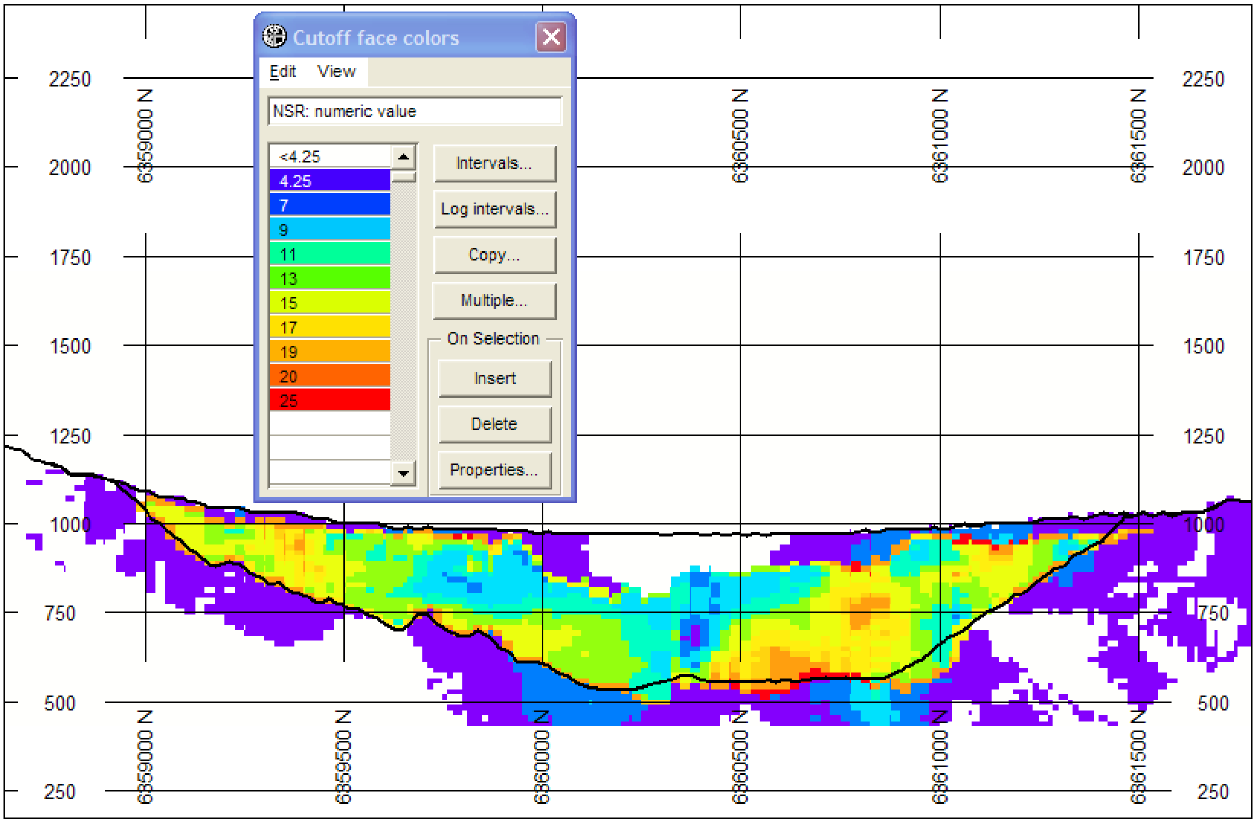


Figure 23.6 LG Pit Limit - Cross Section 'a'

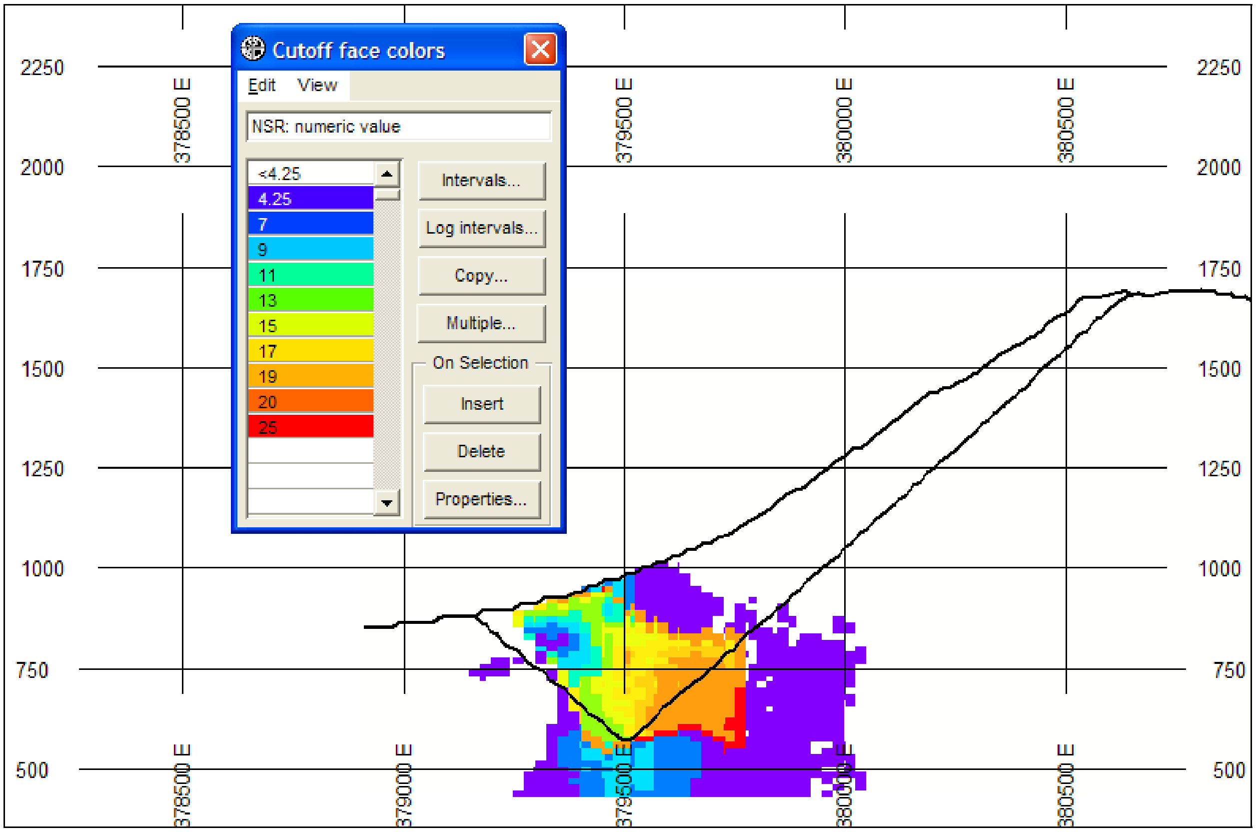


Figure 23.7 LG Pits - Cross Section 'b'

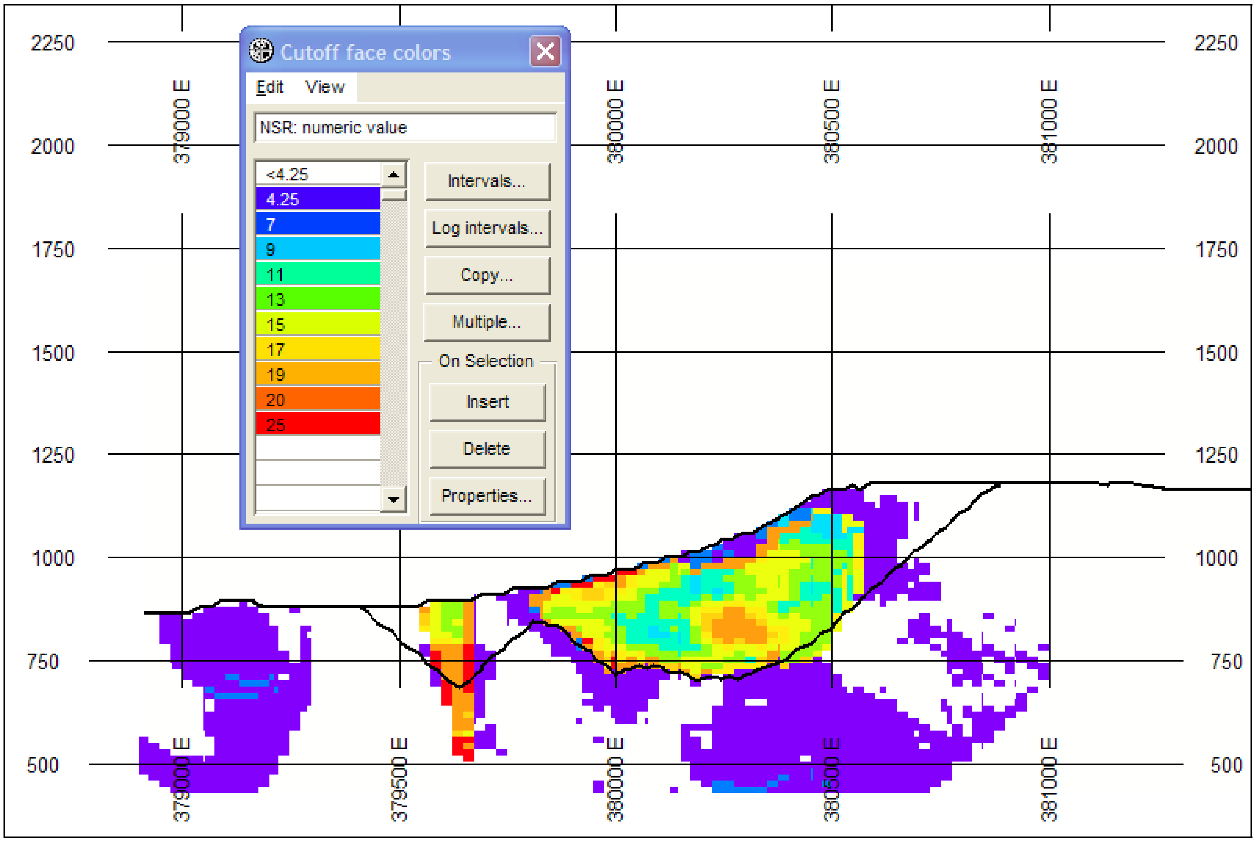


Figure 23.8 LG Pits - Cross Section 'c'

The cross sections 'a' above shows that most grade blocks on the south end of the mineralized area are mined out with at the 90% pit limit case. Cross section 'b' shows a significant volume of high grade mineralized zone on the east side is not mined below the highest part of the east wall.

23.1.6 Detailed Pit Designs

MMTS has completed scoping level pit designs demonstrating the viability of accessing and mining economically mineable resources at the Schaft Creek site. The designs are developed using MineSight® software, estimated geotechnical parametres, regulated standards for road widths, and minimum mining widths based on efficient operation for the size of mining equipment chosen for the project.

23.1.6.1 *Haul Road Widths*

Haul road widths are designed to provide safe and efficient haulage and to comply with the BC Mines Regulations.

For dual lane traffic a travel width of not less than 3 times the width of the widest haulage vehicle used on the road.

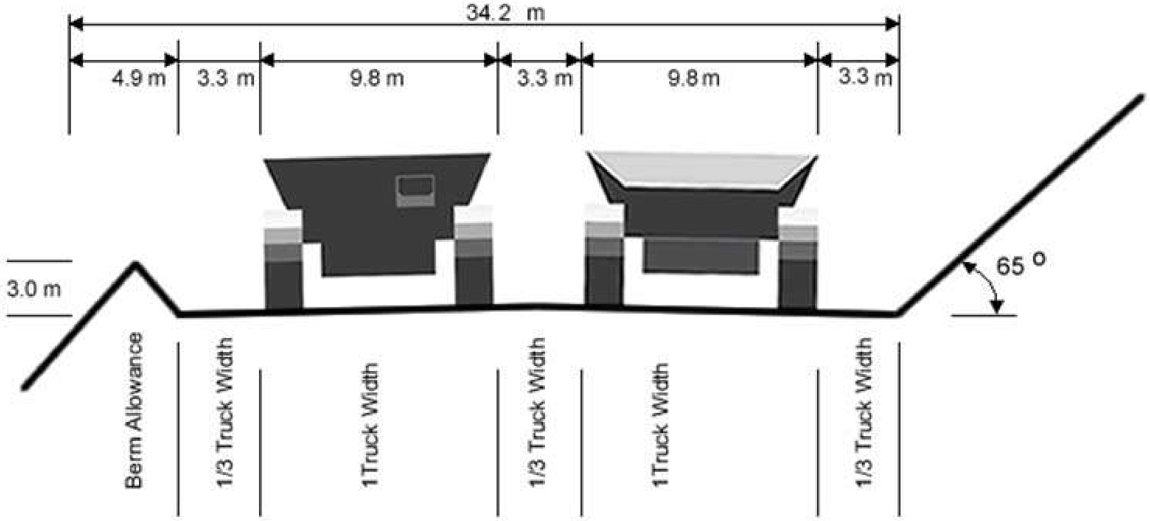
Where single lane traffic exists a travel width of not less than 2 times the width of the widest haulage vehicle used on the road.

Shoulder barriers should be at least 3/4 of the height of the largest tire on any vehicle hauling on the road along the edge of the haulage road wherever a drop-off greater than 3 m exists. The shoulder barriers are designed at 1.5:1 (H:V) side slope. The width of the barrier is excluded from the travel width. The figures below show typical road cross sections for haul roads.

Ditches are included within the travel width allowance. For crowned haul roads, the width of this ditch allowance is 4.5m. Ditches are not added to the in-pit highwall roads as there is adequate water drainage at the edge of the road between the crowned surface and lateral embankments such as highwalls or lateral impact berms. During run off, when water is flowing, this ditch allowance can still be used as lateral clearance for haul trucks and driven on if required to avoid obstructions. In practice, excavated ditches in haul roads quickly get filled in by road grading; and when maintained as open ditches can create a hazard if haul trucks or light vehicles catch a wheel in them. Avoiding the addition of ditch width to the 3-truck travel width on the in-pit high wall roads can significantly reduce the pit waste stripping.

Based on a 345 tonne truck, the haul road design basis is:

Largest Vehicle Overall Width (CAT 797B)	9.8	m
Maximum Tire Height (59/80R63)	4.0	m
Minimum Haul road outside berm height	3.0	m
Berm Width	4.9	m
Ditch Width	4.5	m
Double lane highwall haul road allowance	34.2	m
Double lane external haul road allowance	48.0	m
Single lane highwall haul road allowance	24.4	m
Single lane external haul road allowance	38.3	m



Note: Highwall face slopes will vary by geotechnical design criteria

Figure 23.9 Dual lane highwall haul road cross section

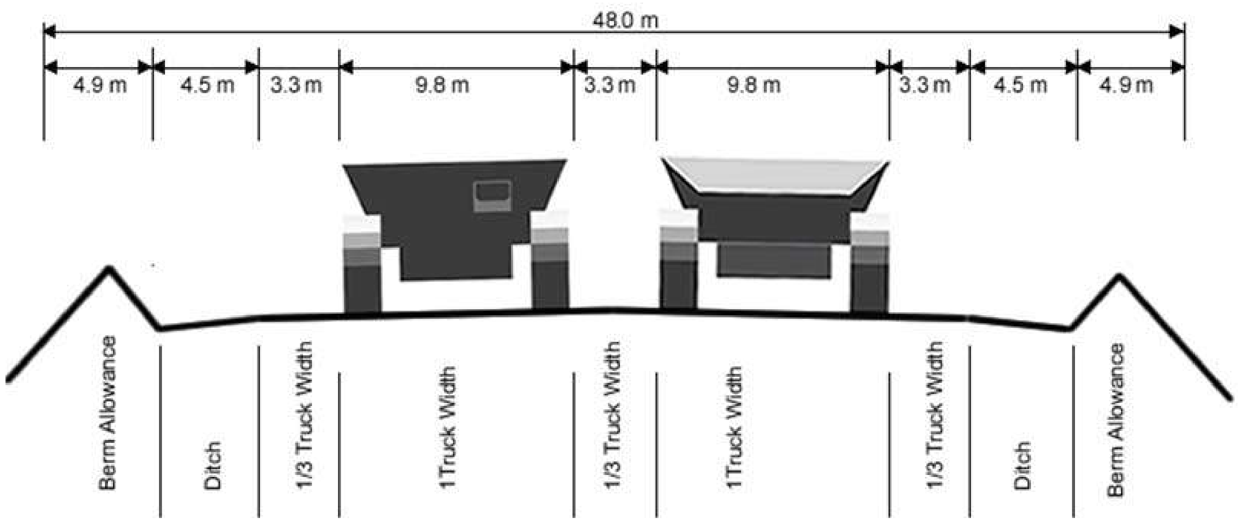
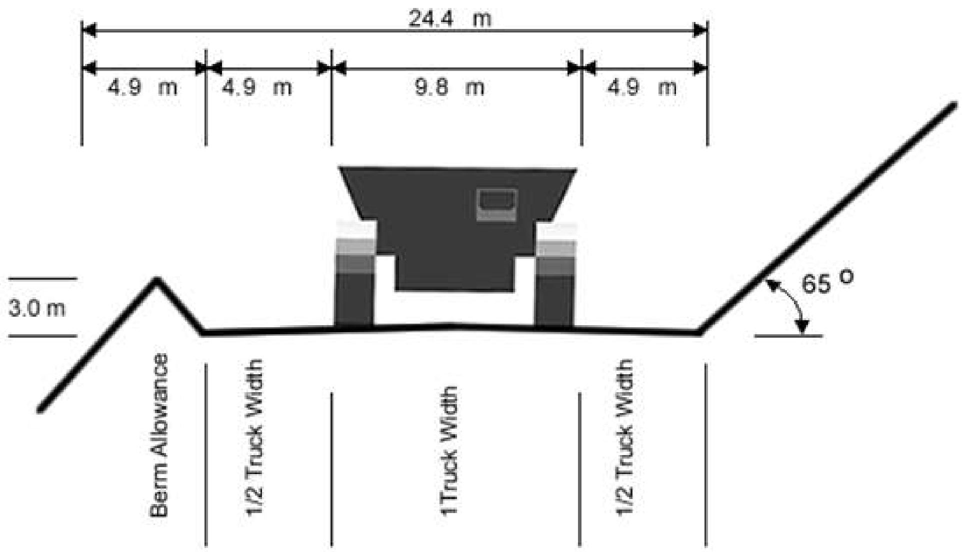


Figure 23.10 Dual lane external haul road cross section



Note: Highwall face slopes will vary by geotechnical design criteria

Figure 23.11 Single lane highwall haul road cross section

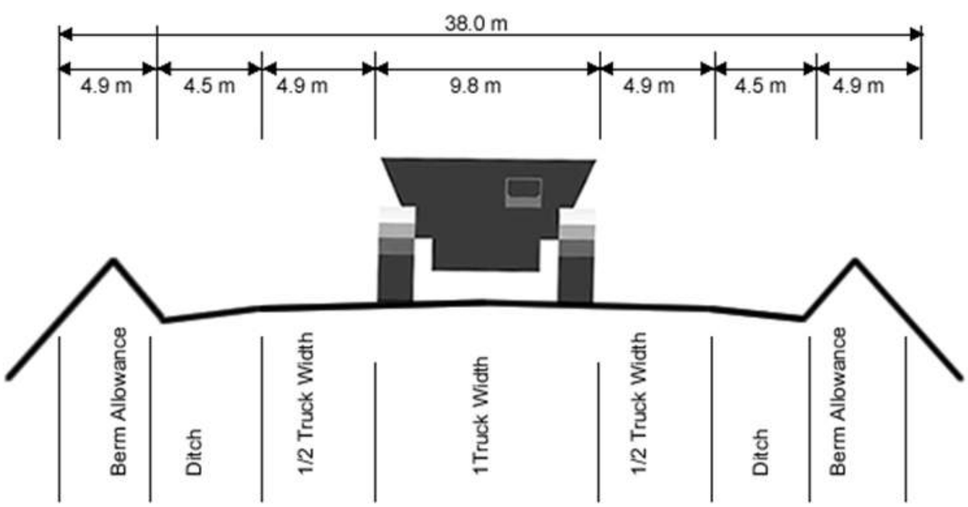


Figure 23.12 Single lane external haul road cross section

23.1.6.2 *Design Standards*

23.1.6.3 *Minimum Mining Width*

As described above, a minimum mining width between pit phases must be reserved to maintain a suitable mining platform for efficient mining operations. This width is established based on equipment size and operating characteristics.

For the Schaft Creek Preliminary Economic Assessment (PEA), minimum mining width generally conforms to 50m, which provides sufficient room for 2-sided truck loading; but, due to the configuration of merging pits, it is sometimes less.

In areas where minimum shovel mining width is not achieved, such as initial outcrop benches, drill and blast ramps will be cut and crawler-dozers or loader tramming will be utilized for excavation. Full truck/shovel excavation of the dozer or trammed material will be done from lower benches where sufficient bench width has been achieved.

For the lowest pit benches at the bottom of the phases, where smaller volumes and bench widths, do not allow for efficient use of the big mining shovels, it is assumed that smaller mining equipment, possibly contractor equipment, or loader tramming will be used to complete small quantities.

23.1.6.4 *Access Considerations*

As stated in the design criteria summary, haul road widths are dictated by equipment size . One-way haul roads must have a travel surface equal to twice the width of the widest haul vehicle. Two-way roads require a running surface equal to three times the width of the widest vehicle planned to use the road. One-way roads are not normally employed for main long term haul routes as they limit the safe passing of trucks and consequently lead to reduced productivity. They are, however, an appropriate option for low volume traffic flow, shorter-term operations where the construction of a two-way road is not warranted. In the current study, the use of one-way haul roads is limited to the bottom two or three benches of some pits. An access ramp is not designed for the very last bench of each pit bottom, on the assumption that it will be removed upon retreat.

Road grades are designed at a maximum grade of 8% to facilitate continuous operations through winter months. Steeper grades (10%) are possible with the current equipment selection but would have to be scheduled for summer only operations for safety and road maintenance considerations. A decision to design steeper roads can be considered after operating experience has been accumulated. Switchbacks are designed flat, with ramps entering and exiting at design grade. In practice however, grades will be transitioned such that visibility and haul speeds are optimized going around the switchback. Where possible, switchbacks are located such that they tie in to future phase access development.

As a Preliminary Economic Assessment (PEA) , ramp optimization in sinking cuts has not been done. With no geotechnical details, ramps in the highwalls are assumed as necessary to meet the conservative low overall Pit slope angles. If future geotechnical studies indicated steeper walls are possible then consideration will be made whether place ramps outside the LG shell to maximize resource recovery or inside to reduce waste strip. The grade of the material being lost or gained and the strip ratio carried to the top of the wall will be considered in future design stages of the project.

In the final pit wall access the lowest pit benches require a spiral ramp designed to exit at the lowest point on the pit rim or joining with infrastructure features (such as the crusher location or previously designed haul road junctions). In the mountainous terrain at Schaft Creek, benches above the lowest point of the pit rims can be accessed by external ramps built on the original hill side slopes, reducing the need for internal ramps in the final wall. Switchbacks and flat grade segments should be minimized. Whether the decline ramp is built inside or outside the LG ultimate pit shell, the amount of ore lost under the ramp or extra waste mined above the ramp is minimized if the ramp is not located on the higher strip ratio wall. In Schaft Creek the topography is rising to the east, making the east wall the highest stripping ratio wall in each case. Ordinarily, the impact on the final pit stripping ratio and net revenue would be optimized if ramps are not designed into the east wall; however, in the case of Schaft Creek the highwall has been reduced to 42 degrees. This enables ramps to be left in the Schaft Creek east highwall without increasing the strip ratio of the ultimate pit if no ramps had been included.

In some intermediate phases it may be necessary to leave highwall ramps in the upper benches of the phase in order to gain access. These intermediate highwall ramps may not be needed when the final pit phase is completed.

23.1.6.5 *Variable Berm width*

As mentioned in the preceding section, pit designs for Schaft Creek are designed honoring overall pit slope angles, a nominal bench face angle (70°) and variable safety berm widths with a minimum 11m width. Where haul roads intersect designed safety benches, the haul road width is counted towards the safety berm width for the purpose of calculating the maximum overall pit slope angle. While this design standard reduces stripping requirements for access construction it may mean an increase in the frequency of clean-up required to keep haul roads free of ravel. Operating experience from the earlier pit phases may justify changing the way berms and ramps are considered in future designs.

23.1.6.6 *Bench height*

The Schaft Creek pit designs are based on the digging height of the large shovels. A 15 metre operating bench with double benching between highwall. Therefore the berms are separated vertically by 30 metres. Single benching is sometimes employed to maximize ore recovery and maintain the safety berm sequence. The berm width is varied to meet the maximum interslope pit slope angle with a minimum of 8 m.

23.1.6.7 *LG Phase Selection*

The LG pits previously discussed are used to evaluate alternatives for determining the economic pit limit and the best pushbacks or phases to begin detailed design work on. LG pits provide a geometrical guide to detailed pit designs. Among the details will be the addition of roads and bench access, removal of impractical mining areas with a width less than the minimum, and insuring the pit slopes meet the detailed geotechnical recommendations.

The 90% price case LG pit with 42^{o} Pit Slope discussed above is the economic pit limit for Schaft Creek. Small pit phases exist within the economic pit limits that are economically mineable at lower metal prices. When considered at Base Case economics these lower price case pits have higher NSR values due to lower strip ratios or better grades than the full economic pit limit. Mining these pits as phases from higher NSR to lower NSR, maximizes revenue and minimizes mining cost at the start of mining operations and thereby shortens the project capital payback and improves the project cash flow.

The selection of LG pit cases to guide the design of starter pits requires the consideration of some practical mining constraints. The starter pits must be:

- Large enough to accommodate mining operations;
- Not be so narrow that large mining equipment cannot operate efficiently.

The pit areas are examined to find the lowest LG Price Case that can sustain mining operations.

Waste from the starter pits is pre-stripped to expose ore for plant start up and can be used for some construction fills for the Schaft Creek project. (It may be more cost effective to do some borrow for construction from other areas to reduce costs if hauls are too long from the starter pit area. A second cost effective alternative is to borrow from upper benches of future pit phases.

A plan view of the 70% price case LG below illustrates the two separate starter pit areas, one in the North and the other in the South.

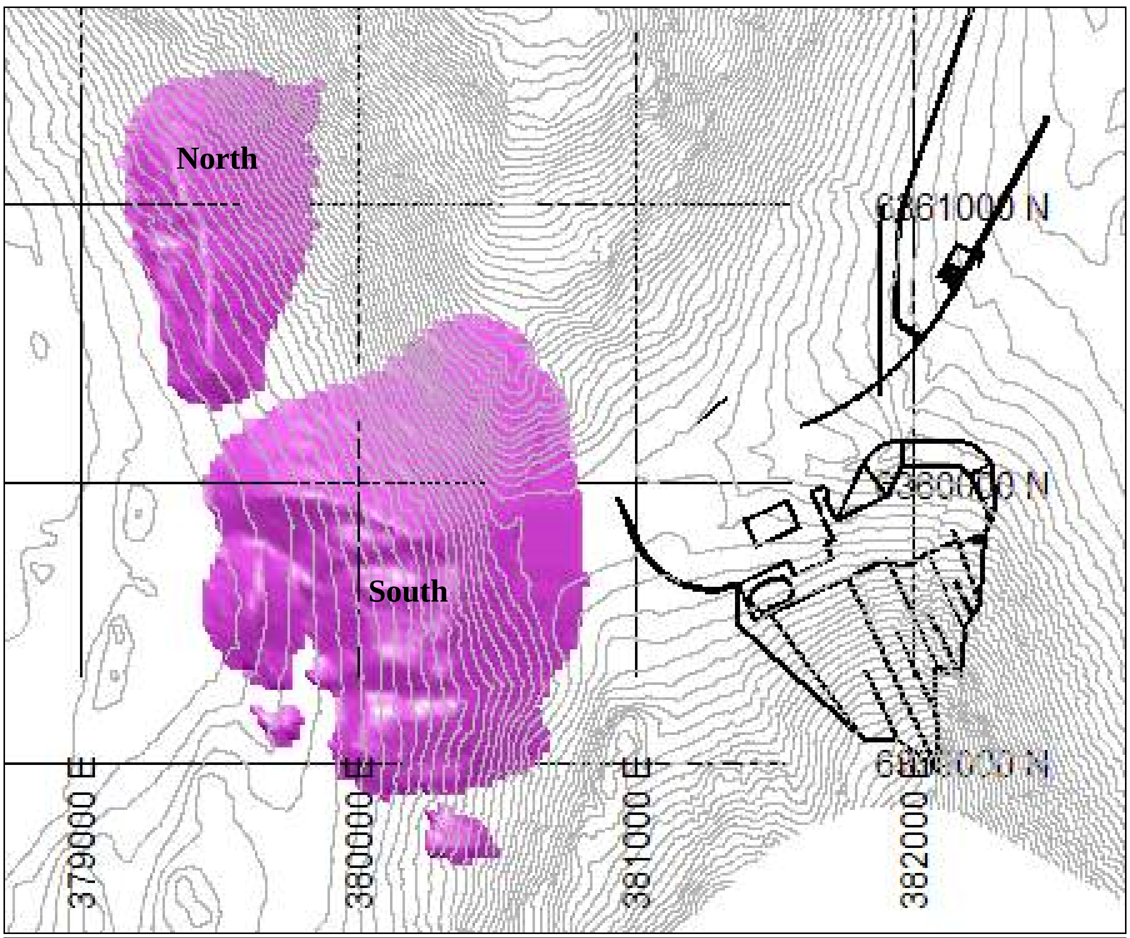


Figure 23.13 Plan View of 70% Price Case LG showing N and S Starter pit areas.

The LG Pit delineated resource table below is analyzed to find the lowest LG price case that meets practical mining requirements.

Table 23.10
Incremental LG Pit Resources

Pit Shell	% Price Case	ORE kT	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste kT	Stripping Ratio
pt217i	50	29,003	18.573	0.4203	0.3580	2.3294	0.0189	14,157	0.49
pt218i	55	89,872	15.619	0.3614	0.2639	1.9191	0.0181	36,930	0.41
pt219i	60	157,385	13.805	0.3131	0.2256	1.5883	0.0181	56,484	0.36
pt220i	65	191,931	13.207	0.2887	0.2124	1.6552	0.0196	134,073	0.70
pt221i	70	254,654	13.250	0.2915	0.1855	1.7607	0.0219	308,848	1.21
pt222i	80	94,461	13.736	0.2868	0.2151	1.9650	0.0232	201,333	2.13
pt223i	90	45,009	12.862	0.2775	0.1944	1.9923	0.0205	127,610	2.84
pt224i	100	26,171	12.896	0.2821	0.1857	1.9579	0.0208	99,256	3.79
pt225i	110	15,135	14.426	0.3111	0.2217	2.2089	0.0227	88,068	5.82
pt226i	130	17,128	13.389	0.3010	0.1869	2.1084	0.0205	115,842	6.76
pt227i	150	7,336	14.332	0.3133	0.2178	2.1831	0.0220	71,047	9.69

Table 23.11
Cumulative LG In-situ Pit Resources

Pit Shell	% Price Case	ORE kT	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste kT	Stripping Ratio
pt216	45	1,554	19.979	0.4478	0.4545	3.0560	0.0134	312	0.20
pt217	50	23,332	19.409	0.4285	0.4019	2.6248	0.0187	14,704	0.63
pt218	55	95,950	16.581	0.3832	0.2945	2.0406	0.0179	47,782	0.50
pt219	60	242,686	14.969	0.3438	0.2545	1.7777	0.0178	107,649	0.44
pt220	65	381,816	14.279	0.3244	0.2421	1.6940	0.0177	198,947	0.52
pt221	70	458,379	14.035	0.3179	0.2330	1.6803	0.0180	281,241	0.61
pt222	80	691,157	13.457	0.3043	0.2094	1.6677	0.0187	662,197	0.96
pt223	90	824,145	13.588	0.3037	0.2108	1.7373	0.0196	1,117,018	1.36
pt224	100	850,726	13.588	0.3035	0.2107	1.7466	0.0196	1,231,865	1.45
pt225	110	863,428	13.581	0.3034	0.2104	1.7502	0.0196	1,296,400	1.50
pt226	130	882,229	13.587	0.3038	0.2099	1.7561	0.0196	1,427,195	1.62
pt227	150	890,810	13.578	0.3036	0.2096	1.7586	0.0196	1,497,681	1.68

23.1.6.8 *South Pits*

The Pit pt218i (55% price case) meets the minimum practical mining constraints. Only the southern portion of pt218i is suitable as the northern portion is too small for practical purposes. The plan view of Pit pt218i is shown below with the south starter phase P616 (purple).

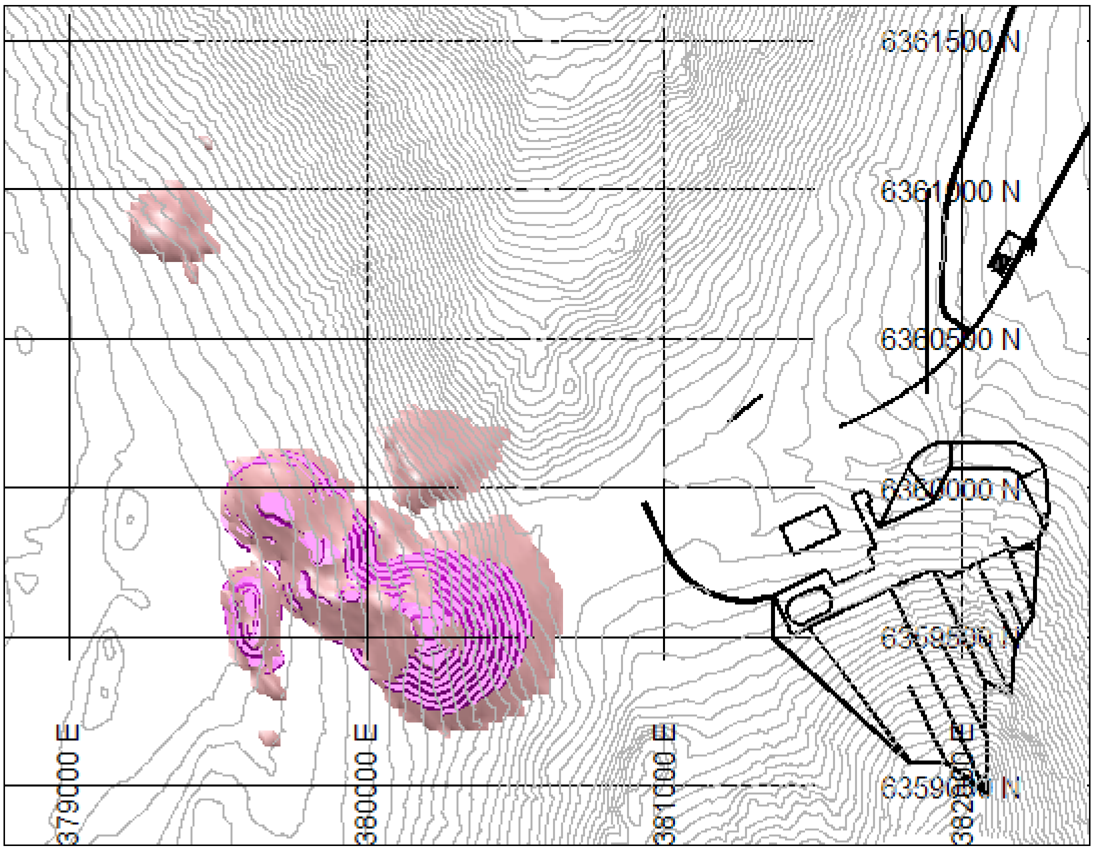


Figure 23.14 Plan view of Pit pt218 used to guide the south starter phase P616

The incremental LG pits expand in all directions from the south starter pit. This is not practical for efficient mining operations. Incremental pits that expand to final pit are therefore designed in two stages. The first intermediate phase generally expands west to final wall and establishes final pit ramps in the west wall as near as possible to the ultimate pit wall. The second incremental pit expands east. Ramps are left in the eastern highwall that help achieve the targeted 42° pit slope while leaving sufficient access to waste and ore destinations from the upper benches.

The South intermediate phases are shown with the ultimate pit LG case below. The intermediate phase (P626i) is orange and the second South Phase (P636i) is blue.

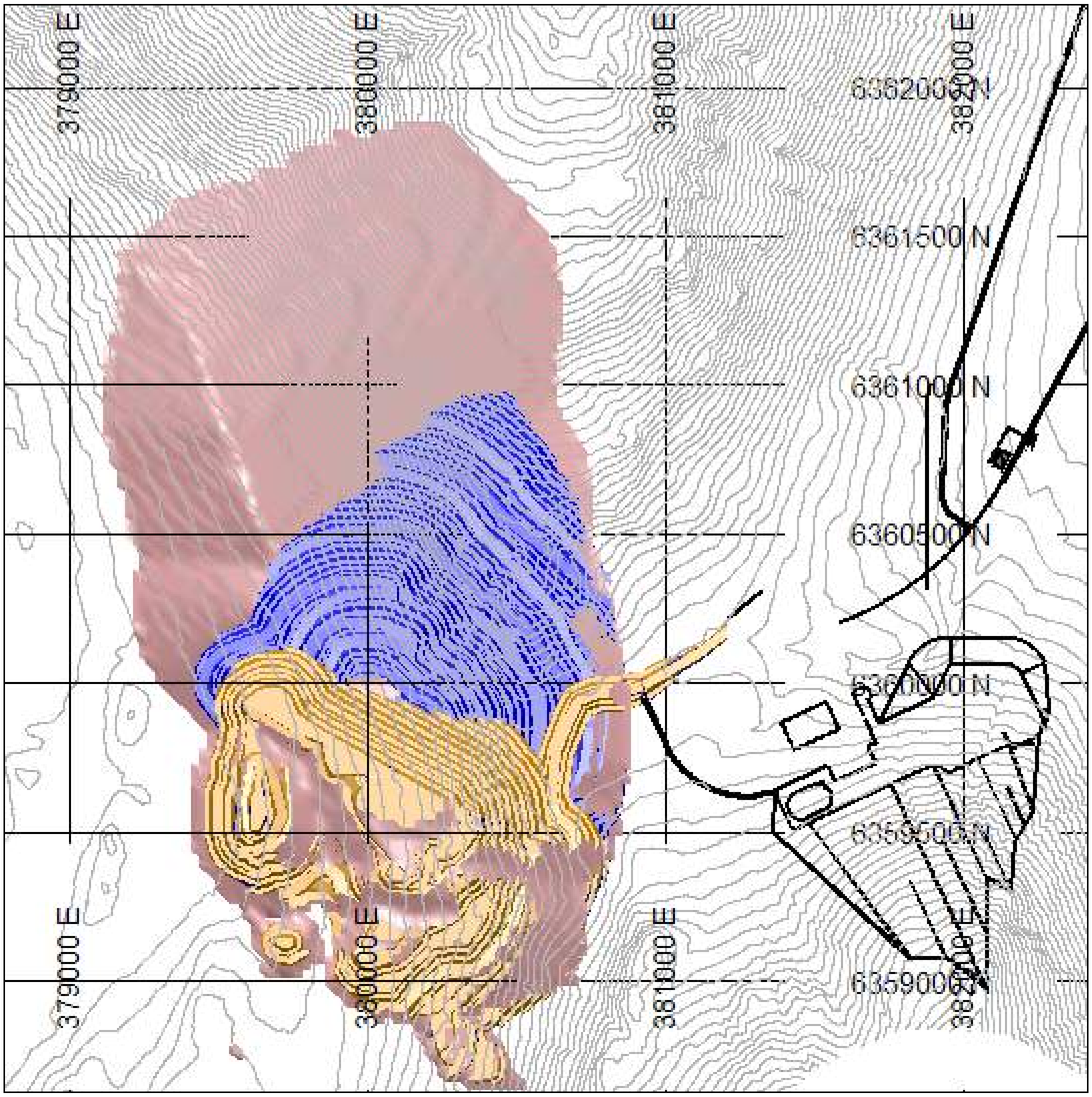


Figure 23.15 Plan view of Pit pt223 used to guide (P626i) and (P626) phases

23.1.6.9 *North Pits*

The first LG to meet minimum feasible mining requirements for a starter pit in the north area is Pit pt219i (60% price case). The plan view of Pit pr219i with the north starter phase P646 (green) is shown below.

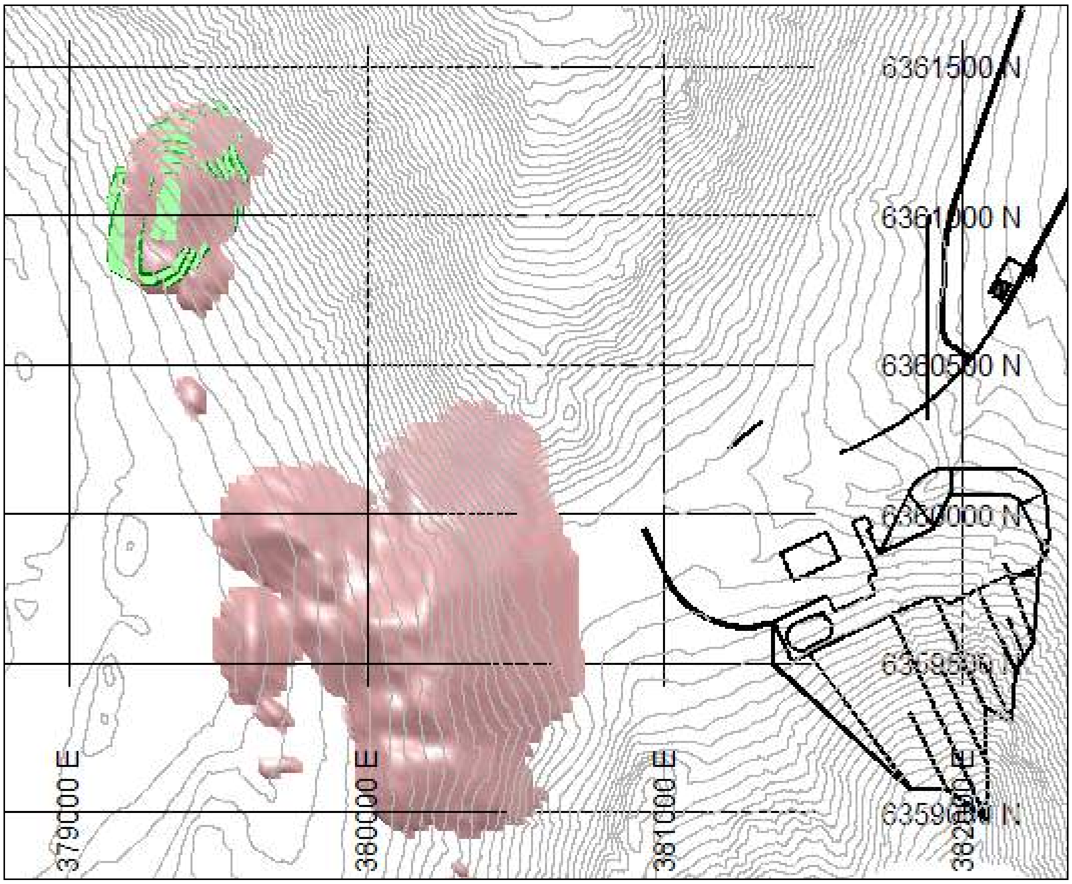


Figure 23.16 Plan view of Pit pt219 used to guide the north starter phase P646

Pit pt221i (70% price case) is used to guide the intermediate north starter phases. This phase has ample ore to sustain mining operations and sufficient bench width for efficient shovel operation and a suitable vertical advance rate. The plan view of Pit pt221i with the north starter phase P646 (green) and the north incremental phase P656i (yellow) is shown below.

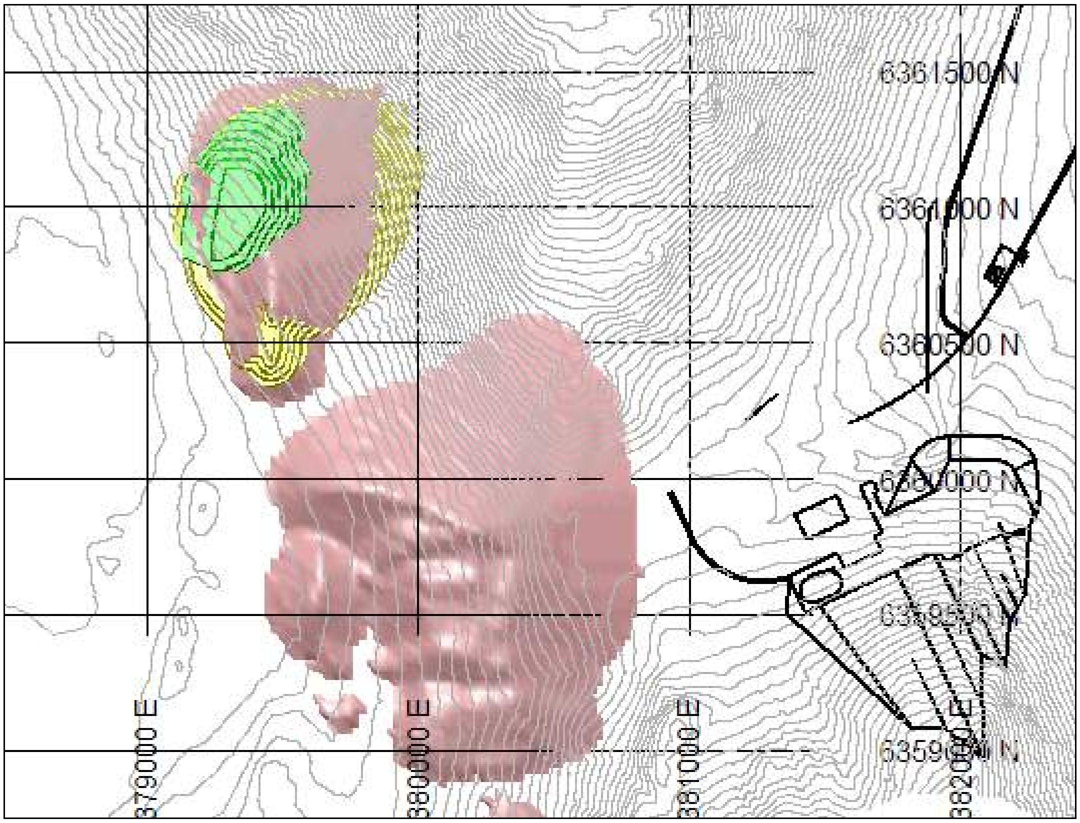


Figure 23.17 Plan view of Pit pt221 used to guide the north incremental phase P656i

23.1.6.10 *Ultimate Pit*

The ultimate pit is guided by pt223 (90% price case). Ramps are left in the highwall for access to the upper benches and to help achieve the estimated 42 degree overall pit slope.

The plan view of Pit pt223i with the Ultimate Pit P666i (Yellow) is shown below.

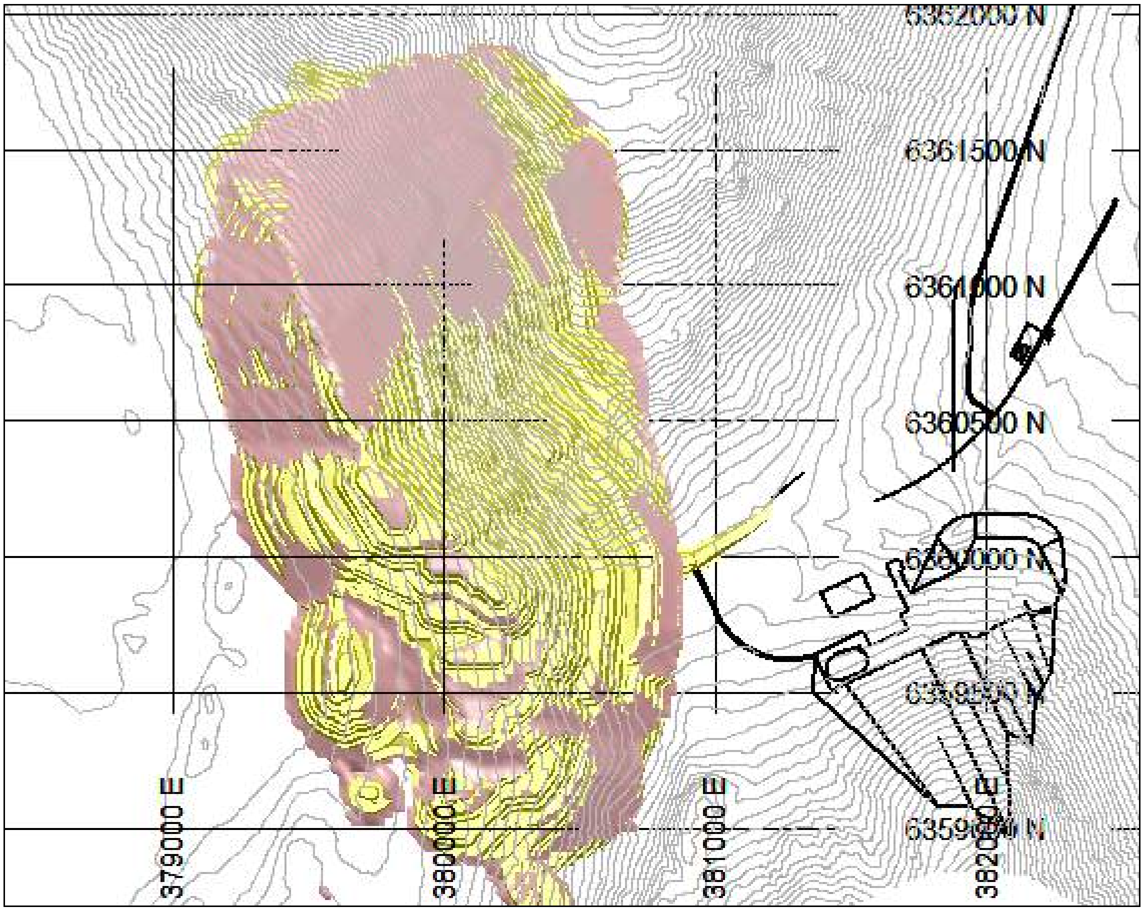


Figure 23.18 Plan view of Pit pt223 used to guide the Ultimate Pit P666i

The following Detailed Pit Designs are developed from the LG pit limits, potential pit phases, and design considerations reviewed above.

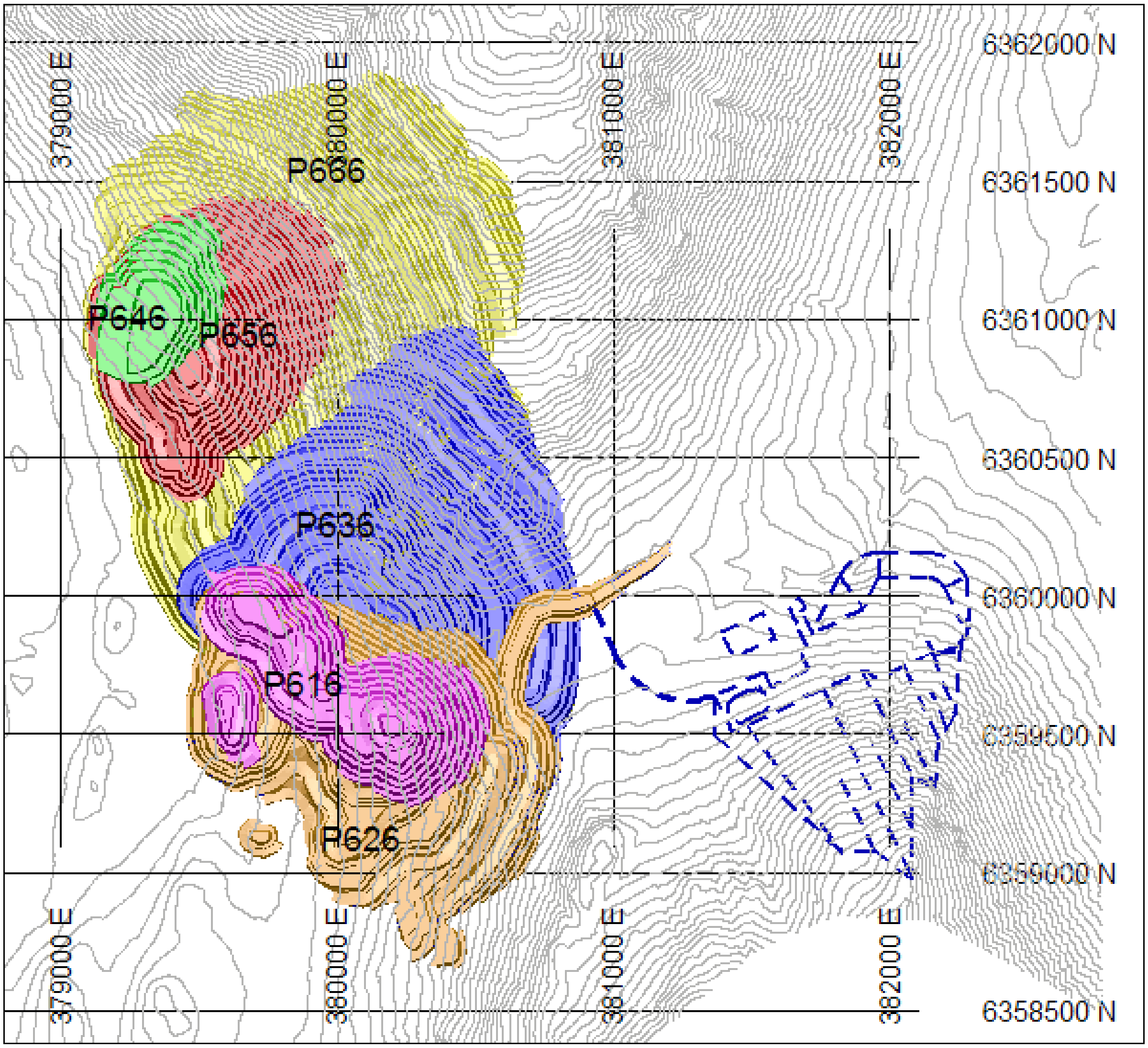


Figure 23.19 Plan view of all designed pit phases

23.1.7 Geotechnical Checks

The above pit designs should be reviewed by Geotechnical professionals for stability and to recommend pit slope design parametres for future studies.

23.1.8 Pit Resources

The following tables list the waste and ore reserves for the material within the Ultimate pit limit (P666) and for each incremental pit phase. Pit delineated resources are estimated in the tables below using the MineSight ® PITRES routine with the following parametres:

- 5% mining dilution applied at the contact between ore and waste. Dilution grades are estimated at 3.49 $/t NSR, 0.060 % Cu, 0.080 g/t Au, 2.150 g/t Ag and 0.004 % Mo representing the average grade of material below the incremental cut-off grade.
- 10% Mining Losses.

Table 23.12
Summarized MII Pit Delineated Resource for Schaft Creek

PHASE	RUN OF	WASTE	ROM	DILUTED GRADES				
	MINE	TOTAL	S/R	NSR	Cu	Au	Ag	Mo
	(mT)	(mT)	(t/t)	$/t	%	g/t	g/t	%
P616	68.8	40.1	0.6	16.6	0.377	0.299	2.03	0.017
P626i	105.2	122.3	1.2	14.4	0.340	0.213	1.61	0.017
P636i	252.0	330.3	1.3	12.6	0.272	0.220	1.54	0.017
P646	15.7	17.1	1.1	15.6	0.314	0.279	2.44	0.023
P656i	82.8	111.5	1.3	14.0	0.298	0.196	1.98	0.024
P666i	194.6	571.1	2.9	13.9	0.302	0.191	1.93	0.024
Total	719.1	1,192.4	1.7	13.8	0.304	0.217	1.77	0.020

Measured, Indicated and Inferred Reserves at Schaft Creek are summarized in the table below.

Table 23.13
Pit Delineated MII Resource at Schaft Creek

Classification	RUN OF MINE	DILUTED GRADES			
		Cu	Au	Ag	Mo
	(kT)	%	g/t	g/t	%
Measured	379.9	0.317	0.238	1.72	0.019
Indicated	337.9	0.289	0.195	1.83	0.020
Inferred	1.3	0.265	0.101	2.05	0.018
Total	719.1	0.304	0.217	1.77	0.020

```
==================================================================================================================
 BENCH   ZONE     Class        CUTOFF       INSITU      INSITU      RUN OF      DILUTED GRADES
 TOE     NAME     NO.                      ORE         ORE         MINE        NSR        CU        AU        AG        MO
                                           (kBCMS)     (kTONNES)   (kTONNES)
==================================================================================================================

TOTALS   MEAS       1       6.08-    8.00        9.         25.         23.     7.940    0.1700    0.1600    1.8900    0.0080
                            8.00-   10.00      482.       1299.       1192.     9.470    0.2144    0.1704    1.5904    0.0089
                           10.00-   12.00     1556.       4173.       3828.    11.157    0.2500    0.2085    1.6128    0.0106
                           12.00-   14.00     6364.      17120.      15591.    13.116    0.3025    0.2360    1.5207    0.0129
                           14.00-   16.00     6631.      17805.      16217.    14.946    0.3425    0.2594    1.7772    0.0160
                           16.00-   18.00     4522.      12127.      11090.    16.892    0.3860    0.2934    2.0158    0.0178
                            >=  18.00         7653.      20580.      18731.    22.526    0.4998    0.4307    2.7570    0.0235
                             TOTALS:         27217.      73129.      66672.    16.654    0.3769    0.3032    2.0195    0.0172

         IND        2       6.08-    8.00        6.         16.         15.     7.919    0.1885    0.1278    1.3427    0.0055
                            8.00-   10.00       69.        185.        185.     8.856    0.2017    0.1390    1.4549    0.0072
                           10.00-   12.00       65.        175.        174.    11.064    0.2464    0.1656    1.8059    0.0109
                           12.00-   14.00       86.        233.        241.    12.667    0.2935    0.1561    1.8458    0.0110
                           14.00-   16.00      262.        709.        667.    14.868    0.4221    0.1099    2.0247    0.0123
                           16.00-   18.00      177.        481.        464.    16.723    0.3983    0.1412    2.8321    0.0234
                            >=  18.00          140.        378.        355.    21.271    0.5074    0.3096    3.4256    0.0204
                             TOTALS:           806.       2175.       2100.    15.212    0.3808    0.1631    2.3459    0.0153

         INF        3       8.00-   10.00        2.          6.          7.     8.020    0.1636    0.0736    1.4948    0.0082
                             TOTALS:             2.          6.          7.     8.020    0.1636    0.0736    1.4948    0.0082

         ---------------------------------------------------------------------------------------------------------
TOTALS   SUMMARY            6.08-    8.00       15.         41.         38.     7.931    0.1774    0.1471    1.6707    0.0070
                            8.00-   10.00      554.       1490.       1384.     9.380    0.2124    0.1657    1.5718    0.0087
                           10.00-   12.00     1621.       4348.       4001.    11.153    0.2498    0.2066    1.6212    0.0106
                           12.00-   14.00     6451.      17352.      15832.    13.109    0.3024    0.2348    1.5257    0.0129
                           14.00-   16.00     6893.      18513.      16884.    14.943    0.3456    0.2535    1.7870    0.0159
                           16.00-   18.00     4700.      12607.      11554.    16.886    0.3865    0.2873    2.0486    0.0180
                            >=  18.00         7793.      20958.      19085.    22.503    0.5000    0.4285    2.7694    0.0235
                             TOTALS:         28026.      75310.      68779.    16.609    0.3770    0.2989    2.0294    0.0172

    WASTE       40127.  (kTONNES)   ROM S/R=    0.58
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Table 23.14
P616 South Starter Pit Resource

Bench	ORE Tonnage	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste Tonnage	Stripping Ratio
1150	-	0.00	0.000	0.000	0.00	0.000	-	-1
1135	-	0.00	0.000	0.000	0.00	0.000	161	-1
1120	-	0.00	0.000	0.000	0.00	0.000	384	-1
1105	-	0.00	0.000	0.000	0.00	0.000	796	-1
1090	37	11.58	0.258	0.093	0.46	0.012	914	24.45
1075	307	14.40	0.359	0.148	0.69	0.015	1,099	3.58
1060	623	15.04	0.383	0.164	0.79	0.016	1,018	1.63
1045	1,059	13.68	0.340	0.158	0.89	0.016	1,219	1.15
1030	1,395	14.10	0.347	0.176	1.09	0.017	1,243	0.89
1015	1,873	15.13	0.358	0.214	1.31	0.018	1,538	0.82
1000	2,124	16.04	0.362	0.258	1.57	0.019	1,684	0.79
985	2,737	16.46	0.365	0.291	1.80	0.019	1,979	0.72
970	3,276	16.34	0.350	0.324	1.91	0.017	1,830	0.56
955	4,519	16.22	0.347	0.337	1.90	0.016	1,858	0.41
940	4,815	16.70	0.362	0.350	1.97	0.016	2,123	0.44
925	5,650	16.30	0.357	0.334	1.99	0.016	2,524	0.45
910	5,669	16.41	0.361	0.330	2.06	0.016	2,788	0.49
895	6,223	16.50	0.365	0.327	2.09	0.017	4,002	0.64
880	6,261	16.44	0.372	0.301	2.10	0.017	4,437	0.71

Table 23.14
P616 South Starter Pit Resource

865	6,794	16.42	0.383	0.279	2.13	0.017	3,803	0.56
850	5,684	16.29	0.383	0.264	2.16	0.017	2,129	0.37
835	5,076	17.02	0.405	0.261	2.23	0.019	1,601	0.32
820	2,639	19.58	0.472	0.294	2.62	0.020	652	0.25
805	1,230	22.38	0.531	0.384	3.39	0.020	205	0.17
790	787	23.81	0.560	0.428	3.89	0.020	138	0.18
Total	**68,778**	**16.61**	**0.377**	**0.299**	**2.03**	**0.017**	**40,127**	**0.58**

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=======================================================================================================================
 BENCH   ZONE     ZONE        CUTOFF        INSITU     INSITU     RUN OF      DILUTED  GRADES
 TOE     NAME     NO.                       ORE        ORE        MINE        NSR       CU        AU        AG        MO
                                            (kBCMS)    (kTONNES)  (kTONNES)
=======================================================================================================================

TOTALS   MEAS       1      6.08-    8.00        83.       225.        210.     7.783    0.2236    0.0723    0.7007    0.0047
                           8.00-   10.00      2073.      5585.       5084.     9.188    0.2201    0.1743    1.1960    0.0064
                          10.00-   12.00      5787.     15485.      14148.    11.090    0.2500    0.2155    1.3372    0.0101
                          12.00-   14.00      9950.     26516.      24078.    13.039    0.3055    0.2123    1.4695    0.0145
                          14.00-   16.00      7085.     18917.      17205.    14.862    0.3496    0.2277    1.6763    0.0175
                          16.00-   18.00      3674.      9881.       8974.    16.898    0.3942    0.2525    2.0021    0.0212
                           >=  18.00          4644.     12578.      11404.    21.644    0.5120    0.3008    2.4209    0.0285
                            TOTALS:          33296.     89186.      81104.    14.468    0.3385    0.2303    1.6638    0.0165

         IND        2      6.08-    8.00        27.        71.         69.     7.541    0.1984    0.0953    1.2733    0.0048
                           8.00-   10.00       231.       624.        597.     9.228    0.2293    0.0849    1.7286    0.0113
                          10.00-   12.00      1054.      2784.       2612.    11.384    0.2934    0.1276    1.3931    0.0115
                          12.00-   14.00      4985.     13033.      12079.    12.970    0.3215    0.1331    1.1206    0.0181
                          14.00-   16.00      2071.      5524.       5151.    14.952    0.3411    0.2113    1.3086    0.0201
                          16.00-   18.00       548.      1480.       1407.    16.950    0.4283    0.1440    2.1335    0.0215
                           >=  18.00           770.      2105.       1994.    21.787    0.5467    0.1911    2.8194    0.0275
                            TOTALS:           9687.     25622.      23910.    14.084    0.3450    0.1535    1.4078    0.0186

         INF        3      5.70-    6.08        13.        36.         33.     5.911    0.1924    0.0192    1.0512    0.0020
                           6.08-    8.00        16.        43.         41.     6.325    0.1997    0.0215    1.0329    0.0024
                          12.00-   14.00        16.        44.         46.    12.909    0.2659    0.1562    0.7186    0.0154
                          16.00-   18.00        28.        76.         83.    17.435    0.3829    0.1081    2.2993    0.0208
                            TOTALS:             74.       199.        203.    12.268    0.2880    0.0870    1.4788    0.0128

         ------------------------------------------------------------------------------------------------------------
TOTALS   SUMMARY           5.70-    6.08        13.        36.         33.     5.911    0.1924    0.0192    1.0512    0.0020
                           6.08-    8.00       126.       340.        321.     7.543    0.2151    0.0708    0.8672    0.0044
                           8.00-   10.00      2304.      6209.       5681.     9.192    0.2211    0.1649    1.2520    0.0069
                          10.00-   12.00      6841.     18269.      16760.    11.136    0.2568    0.2018    1.3459    0.0103
                          12.00-   14.00     14951.     39593.      36203.    13.016    0.3108    0.1858    1.3521    0.0157
                          14.00-   16.00      9156.     24440.      22357.    14.883    0.3476    0.2239    1.5915    0.0181
                          16.00-   18.00      4251.     11437.      10463.    16.909    0.3987    0.2367    2.0221    0.0212
                           >=  18.00          5415.     14683.      13399.    21.665    0.5172    0.2844    2.4802    0.0284
                            TOTALS:          43057.    115006.     105217.    14.376    0.3399    0.2125    1.6053    0.0170

   WASTE      122346. (kTONNES)  ROM S/R=    1.16
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Table 23.15
P626i South Intermediate Pit Incremental Resource

Bench	ORE Tonnage	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste Tonnage	Stripping Ratio
1210	-	0.00	0.000	0.000	0.00	0.000	-	-1
1195	-	0.00	0.000	0.000	0.00	0.000	109	-1
1180	-	0.00	0.000	0.000	0.00	0.000	596	-1
1165	-	0.00	0.000	0.000	0.00	0.000	2,710	-1
1150	-	0.00	0.000	0.000	0.00	0.000	4,385	-1
1135	-	0.00	0.000	0.000	0.00	0.000	5,231	-1
1120	4	11.39	0.186	0.126	0.59	0.007	5,070	1370.35
1105	53	10.85	0.214	0.152	0.63	0.009	4,971	94.32
1090	95	10.85	0.242	0.167	0.73	0.009	4,946	52.22
1075	198	10.63	0.234	0.158	0.80	0.010	5,379	27.19
1060	496	11.63	0.259	0.164	1.02	0.013	4,919	9.93
1045	984	12.47	0.283	0.180	1.16	0.015	5,025	5.11
1030	1,709	13.23	0.310	0.186	1.15	0.016	4,524	2.65
1015	2,492	13.57	0.316	0.196	1.17	0.017	4,104	1.65
1000	2,920	13.85	0.328	0.195	1.17	0.017	3,405	1.17
985	3,621	13.59	0.317	0.194	1.24	0.017	3,165	0.87
970	4,971	13.46	0.309	0.202	1.30	0.017	2,436	0.49

Table 23.15 P626i South Intermediate Pit Incremental Resource								
955	6,187	13.23	0.306	0.207	1.32	0.016	1,963	0.32
940	7,081	13.26	0.312	0.205	1.34	0.016	1,953	0.28
925	7,582	13.14	0.312	0.198	1.32	0.016	2,820	0.37
910	7,796	13.30	0.313	0.203	1.40	0.016	3,228	0.41
895	6,939	13.90	0.328	0.215	1.54	0.016	3,841	0.55
880	6,544	15.17	0.355	0.246	1.70	0.017	4,502	0.69
865	6,056	15.56	0.361	0.255	1.76	0.018	6,235	1.03
850	5,740	15.57	0.360	0.257	1.87	0.017	6,588	1.15
835	5,507	14.85	0.346	0.236	1.92	0.017	6,386	1.16
820	6,578	14.36	0.341	0.216	1.86	0.016	6,247	0.95
805	6,650	14.50	0.354	0.201	1.87	0.016	5,327	0.8
790	5,832	14.50	0.355	0.194	1.88	0.017	5,107	0.88
775	4,801	16.18	0.395	0.211	2.08	0.018	3,489	0.73
760	3,210	17.35	0.421	0.215	2.12	0.021	2,889	0.9
745	581	24.35	0.630	0.116	2.44	0.040	612	1.05
730	376	25.18	0.631	0.116	2.49	0.045	145	0.39
715	214	24.15	0.600	0.114	2.66	0.044	37	0.17
Total	**105,217**	**14.38**	**0.340**	**0.213**	**1.61**	**0.017**	**122,346**	**1.16**

BENCH TOE	ZONE NAME	ZONE NO.	CUTOFF	INSITU ORE (kBCMS)	INSITU ORE (kTONNES)	RUN OF MINE (kTONNES)	DILUTED GRADES NSR	CU	AU	AG	MO
TOTALS	MEAS	1	0.0- 5.70	19.	50.	45.	5.429	0.1000	0.1350	1.6951	0.0055
			5.70- 6.08	75.	200.	180.	5.927	0.1212	0.1388	1.6825	0.0047
			6.08- 8.00	1590.	4252.	3829.	7.371	0.1519	0.1717	1.3342	0.0068
			8.00- 10.00	9618.	25743.	23244.	9.161	0.1978	0.1868	1.2101	0.0098
			10.00- 12.00	13481.	36146.	32634.	11.007	0.2371	0.2156	1.2819	0.0130
			12.00- 14.00	13287.	35664.	32211.	13.009	0.2854	0.2293	1.3631	0.0171
			14.00- 16.00	12468.	33410.	30201.	14.958	0.3280	0.2381	1.5302	0.0222
			16.00- 18.00	6533.	17409.	15744.	16.830	0.3750	0.2511	1.6362	0.0256
			>= 18.00	4499.	11976.	10788.	20.006	0.3926	0.3214	1.7063	0.0393
			TOTALS:	61571.	164848.	148876.	13.120	0.2833	0.2288	1.4088	0.0183
	IND	2	0.0- 5.70	923.	2464.	2217.	5.368	0.1120	0.1193	1.6403	0.0043
			5.70- 6.08	695.	1856.	1680.	5.904	0.1234	0.1294	1.7243	0.0049
			6.08- 8.00	4715.	12600.	11428.	7.099	0.1486	0.1505	1.7469	0.0068
			8.00- 10.00	7662.	20492.	18778.	9.087	0.1971	0.1686	1.5624	0.0102
			10.00- 12.00	9644.	25834.	23650.	10.969	0.2432	0.1821	1.5638	0.0138
			12.00- 14.00	5971.	16043.	14594.	12.961	0.2853	0.2204	1.7609	0.0167
			14.00- 16.00	5978.	16114.	14683.	15.066	0.3203	0.2548	1.8713	0.0218
			16.00- 18.00	4549.	12302.	11182.	16.832	0.3474	0.2917	1.9222	0.0261
			>= 18.00	2011.	5402.	4905.	19.259	0.3832	0.3229	2.1297	0.0339
			TOTALS:	42149.	113106.	103118.	11.890	0.2545	0.2083	1.7256	0.0159
TOTALS	SUMMARY		0.0- 5.70	942.	2514.	2262.	5.370	0.1118	0.1196	1.6414	0.0044
			5.70- 6.08	770.	2056.	1860.	5.906	0.1232	0.1303	1.7202	0.0049
			6.08- 8.00	6304.	16852.	15257.	7.168	0.1494	0.1558	1.6433	0.0068
			8.00- 10.00	17280.	46235.	42022.	9.128	0.1975	0.1787	1.3676	0.0100
			10.00- 12.00	23125.	61980.	56284.	10.991	0.2397	0.2016	1.4004	0.0134
			12.00- 14.00	19259.	51706.	46805.	12.994	0.2853	0.2265	1.4872	0.0170
			14.00- 16.00	18447.	49523.	44884.	14.993	0.3255	0.2435	1.6418	0.0221
			16.00- 18.00	11083.	29710.	26926.	16.831	0.3635	0.2680	1.7550	0.0258
			>= 18.00	6510.	17378.	15693.	19.772	0.3897	0.3218	1.8386	0.0376
			TOTALS:	103720.	277954.	251994.	12.617	0.2715	0.2204	1.5384	0.0173

WASTE 330269. (kTONNES) ROM S/R= 1.31

Table 23.16
P636i South Final Pit Incremental Resource

Bench	ORE Tonnage	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste Tonnage	Stripping Ratio
1495	-	0.00	0.000	0.000	0.00	0.000	13	-1
1480	-	0.00	0.000	0.000	0.00	0.000	40	-1
1465	-	0.00	0.000	0.000	0.00	0.000	170	-1
1450	-	0.00	0.000	0.000	0.00	0.000	291	-1
1435	-	0.00	0.000	0.000	0.00	0.000	688	-1
1420	-	0.00	0.000	0.000	0.00	0.000	980	-1
1405	-	0.00	0.000	0.000	0.00	0.000	1,463	-1
1390	-	0.00	0.000	0.000	0.00	0.000	1,729	-1
1375	-	0.00	0.000	0.000	0.00	0.000	2,356	-1
1360	-	0.00	0.000	0.000	0.00	0.000	2,836	-1
1345	-	0.00	0.000	0.000	0.00	0.000	3,734	-1
1330	-	0.00	0.000	0.000	0.00	0.000	4,351	-1
1315	-	0.00	0.000	0.000	0.00	0.000	5,333	-1
1300	-	0.00	0.000	0.000	0.00	0.000	6,056	-1
1285	-	0.00	0.000	0.000	0.00	0.000	6,995	-1
1270	-	0.00	0.000	0.000	0.00	0.000	7,463	-1
1255	-	0.00	0.000	0.000	0.00	0.000	8,237	-1
1240	-	0.00	0.000	0.000	0.00	0.000	8,582	-1

Table 23.16 P636i South Final Pit Incremental Resource								
Bench	ORE	ORE	ORE	ORE	ORE	ORE	Waste	Stripping
	Tonnage	NSR	Cu	Au	Ag	Mo	Tonnage	Ratio
1225	-	0.00	0.000	0.000	0.00	0.000	9,382	-1
1210	-	0.00	0.000	0.000	0.00	0.000	9,564	-1
1195	108	9.45	0.238	0.104	1.22	0.007	10,183	94.28
1180	607	9.84	0.261	0.112	1.08	0.008	10,333	17.03
1165	1,046	10.30	0.273	0.125	1.04	0.009	11,905	11.38
1150	1,421	10.83	0.283	0.138	1.07	0.010	12,908	9.08
1135	1,981	10.61	0.269	0.137	1.10	0.011	12,901	6.51
1120	2,663	10.84	0.270	0.136	1.13	0.012	12,394	4.65
1105	3,555	11.33	0.280	0.142	1.17	0.013	12,273	3.45
1090	4,523	11.63	0.286	0.152	1.22	0.014	11,133	2.46
1075	4,955	11.94	0.291	0.160	1.28	0.014	10,531	2.13
1060	5,625	12.09	0.294	0.163	1.32	0.015	9,658	1.72
1045	6,209	12.28	0.297	0.166	1.38	0.015	9,376	1.51
1030	6,987	12.75	0.307	0.176	1.45	0.016	8,712	1.25
1015	7,256	13.16	0.314	0.191	1.52	0.016	8,560	1.18
1000	7,751	13.26	0.309	0.205	1.56	0.016	8,190	1.06
985	8,101	12.99	0.292	0.215	1.58	0.017	8,610	1.06
970	8,501	12.60	0.276	0.221	1.58	0.016	8,203	0.96
955	8,861	12.10	0.260	0.224	1.59	0.015	8,248	0.93
940	8,840	11.55	0.243	0.227	1.58	0.014	7,062	0.8
925	9,009	11.01	0.231	0.218	1.54	0.014	7,285	0.81
910	9,168	10.50	0.218	0.214	1.52	0.013	7,143	0.78
895	9,319	10.32	0.215	0.211	1.51	0.013	7,290	0.78
880	9,452	10.44	0.215	0.217	1.52	0.013	7,231	0.77
865	9,569	10.56	0.215	0.220	1.51	0.013	7,359	0.77
850	9,730	11.05	0.223	0.229	1.52	0.014	6,406	0.66
835	9,919	11.41	0.229	0.231	1.51	0.016	5,716	0.58
820	10,087	12.02	0.242	0.236	1.51	0.017	4,724	0.47
805	10,395	12.77	0.261	0.240	1.57	0.019	3,432	0.33
790	10,816	13.33	0.276	0.237	1.60	0.020	2,466	0.23
775	10,768	13.72	0.288	0.239	1.67	0.021	1,500	0.14
760	10,029	14.03	0.294	0.240	1.68	0.022	1,191	0.12
745	10,161	14.65	0.312	0.245	1.70	0.022	1,463	0.14
730	8,429	15.02	0.316	0.252	1.70	0.023	1,189	0.14
715	7,478	15.47	0.320	0.263	1.70	0.025	1,317	0.18
700	5,606	16.03	0.326	0.277	1.70	0.026	979	0.17
685	3,769	16.44	0.328	0.287	1.63	0.028	630	0.17
670	3,188	16.47	0.327	0.289	1.65	0.028	601	0.19

Table 23.16 P636i South Final Pit Incremental Resource								
Bench	**ORE**	**ORE**	**ORE**	**ORE**	**ORE**	**ORE**	**Waste**	**Stripping**
	Tonnage	**NSR**	**Cu**	**Au**	**Ag**	**Mo**	**Tonnage**	**Ratio**
655	2,346	16.27	0.322	0.285	1.64	0.028	311	0.13
640	1,899	15.93	0.317	0.271	1.65	0.028	226	0.12
625	1,157	15.55	0.312	0.256	1.62	0.028	123	0.11
610	709	15.01	0.292	0.228	1.55	0.025	243	0.34
Total	**251,994**	**12.62**	**0.272**	**0.220**	**1.54**	**0.017**	**330,269**	**1.31**

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=====================================================================================================================
BENCH  ZONE    ZONE      CUTOFF       INSITU     INSITU     RUN OF      DILUTED  GRADES
TOE    NAME    NO.                    ORE        ORE        MINE        NSR       CU        AU        AG        MO
                                      (kBCMS)    (kTONNES)  (kTONNES)
=====================================================================================================================

TOTALS  MEAS      1     8.00-  10.00      30.        80.        72.     9.877   0.1759   0.1611   2.2369   0.0208
                       10.00-  12.00     437.      1156.      1048.    11.312   0.2270   0.1637   2.0212   0.0211
                       12.00-  14.00     969.      2570.      2324.    13.041   0.2722   0.1902   2.1279   0.0226
                       14.00-  16.00    1348.      3563.      3224.    15.064   0.3197   0.2339   2.3093   0.0238
                       16.00-  18.00    1222.      3228.      2927.    16.822   0.3530   0.2853   2.4768   0.0248
                         >=  18.00       498.      1315.      1194.    20.124   0.4491   0.3399   3.0102   0.0245
                          TOTALS:       4505.     11912.     10789.    15.266   0.3229   0.2429   2.3647   0.0236

        IND       2    10.00-  12.00      43.       113.       113.    11.572   0.2369   0.1441   1.8842   0.0180
                       12.00-  14.00     449.      1178.      1114.    13.024   0.2657   0.2003   2.1404   0.0197
                       14.00-  16.00     511.      1345.      1284.    14.957   0.2926   0.2583   2.4478   0.0213
                       16.00-  18.00     506.      1327.      1255.    16.982   0.3049   0.3789   2.7436   0.0217
                         >=  18.00       482.      1269.      1167.    20.194   0.3181   0.6182   3.1241   0.0211
                          TOTALS:       1991.      5232.      4933.    16.197   0.2944   0.3584   2.6007   0.0209

        -------------------------------------------------------------------------------------------------------------
TOTALS  SUMMARY         8.00-  10.00      30.        80.        72.     9.877   0.1759   0.1611   2.2369   0.0208
                       10.00-  12.00     480.      1269.      1161.    11.337   0.2280   0.1618   2.0079   0.0208
                       12.00-  14.00    1418.      3747.      3438.    13.036   0.2701   0.1935   2.1319   0.0217
                       14.00-  16.00    1859.      4907.      4508.    15.034   0.3120   0.2409   2.3488   0.0231
                       16.00-  18.00    1727.      4555.      4182.    16.870   0.3386   0.3134   2.5568   0.0239
                         >=  18.00       980.      2584.      2360.    20.159   0.3844   0.4775   3.0665   0.0228
                          TOTALS:       6496.     17144.     15722.    15.558   0.3139   0.2791   2.4388   0.0228

    WASTE      17090. (kTONNES)  ROM S/R=   1.09
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Table 23.17
P646 North Starter Pit Resource

Bench	ORE Tonnage	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste Tonnage	Stripping Ratio
1120	-	0.00	0.000	0.000	0.00	0.000	2	-1
1105	-	0.00	0.000	0.000	0.00	0.000	4	-1
1090	-	0.00	0.000	0.000	0.00	0.000	90	-1
1075	-	0.00	0.000	0.000	0.00	0.000	92	-1
1060	-	0.00	0.000	0.000	0.00	0.000	280	-1
1045	-	0.00	0.000	0.000	0.00	0.000	372	-1
1030	-	0.00	0.000	0.000	0.00	0.000	805	-1
1015	-	0.00	0.000	0.000	0.00	0.000	958	-1
1000	64	17.30	0.328	0.272	3.40	0.020	1,406	21.83
985	270	16.14	0.291	0.319	3.16	0.021	1,379	5.11
970	602	16.13	0.293	0.341	3.15	0.021	1,704	2.83
955	820	15.96	0.291	0.351	3.03	0.021	1,762	2.15
940	1,282	15.46	0.275	0.352	2.73	0.021	1,900	1.48
925	1,697	15.73	0.287	0.343	2.63	0.022	1,754	1.03
910	2,362	15.85	0.317	0.295	2.50	0.023	1,605	0.68
895	2,476	15.70	0.333	0.252	2.35	0.023	1,377	0.56
880	2,739	15.35	0.332	0.235	2.24	0.024	1,223	0.45
865	1,790	15.26	0.326	0.237	2.12	0.024	200	0.11
850	1,620	14.93	0.319	0.228	2.02	0.024	178	0.11
Total	**15,722**	**15.56**	**0.314**	**0.279**	**2.44**	**0.023**	**17,090**	**1.09**

BENCH TOE	ZONE NAME	ZONE NO.	CUTOFF	INSITU ORE (kBCMS)	INSITU ORE (kTONNES)	RUN OF MINE (kTONNES)	DILUTED GRADES NSR	CU	AU	AG	MO
TOTALS	MEAS	1	0.0- 5.70	9.	25.	23.	5.580	0.1100	0.0500	0.3700	0.0150
			5.70- 6.08	27.	73.	66.	5.937	0.1200	0.0572	1.1749	0.0139
			6.08- 8.00	1052.	2842.	2580.	7.271	0.1610	0.0821	1.4299	0.0127
			8.00- 10.00	1844.	4936.	4464.	9.063	0.1994	0.1094	1.5475	0.0159
			10.00- 12.00	3725.	9942.	8984.	11.138	0.2540	0.1233	1.6047	0.0194
			12.00- 14.00	4752.	12674.	11448.	13.042	0.2914	0.1644	1.8013	0.0221
			14.00- 16.00	4752.	12608.	11384.	14.934	0.3164	0.2216	2.1952	0.0251
			16.00- 18.00	4135.	11006.	9930.	17.014	0.3564	0.2649	2.2948	0.0285
			>= 18.00	3082.	8289.	7468.	19.283	0.4020	0.3050	2.5553	0.0325
			TOTALS:	23380.	62395.	56346.	14.057	0.3031	0.1974	1.9980	0.0239
	IND	2	6.08- 8.00	87.	236.	220.	7.701	0.1584	0.0935	1.2475	0.0149
			8.00- 10.00	730.	1957.	1824.	9.206	0.2046	0.1009	1.4463	0.0159
			10.00- 12.00	2388.	6390.	5872.	11.013	0.2555	0.0967	1.5003	0.0202
			12.00- 14.00	2898.	7738.	7121.	13.052	0.2811	0.1294	1.6772	0.0266
			14.00- 16.00	1940.	5157.	4759.	14.835	0.3012	0.1944	1.9861	0.0286
			16.00- 18.00	1332.	3533.	3225.	16.908	0.3231	0.3318	2.5101	0.0261
			>= 18.00	1415.	3763.	3412.	19.049	0.3532	0.4199	2.9484	0.0275
			TOTALS:	10790.	28774.	26432.	13.855	0.2872	0.1937	1.9397	0.0248
TOTALS	SUMMARY		0.0- 5.70	9.	25.	23.	5.580	0.1100	0.0500	0.3700	0.0150
			5.70- 6.08	27.	73.	66.	5.937	0.1200	0.0572	1.1749	0.0139
			6.08- 8.00	1140.	3078.	2800.	7.305	0.1608	0.0830	1.4156	0.0129
			8.00- 10.00	2573.	6893.	6287.	9.105	0.2009	0.1069	1.5182	0.0159
			10.00- 12.00	6113.	16332.	14856.	11.089	0.2546	0.1128	1.5635	0.0198
			12.00- 14.00	7651.	20413.	18569.	13.046	0.2875	0.1510	1.7537	0.0238
			14.00- 16.00	6692.	17765.	16143.	14.905	0.3119	0.2135	2.1336	0.0262
			16.00- 18.00	5468.	14539.	13156.	16.988	0.3482	0.2813	2.3476	0.0280
			>= 18.00	4497.	12052.	10880.	19.209	0.3867	0.3411	2.6786	0.0310
			TOTALS:	34170.	91170.	82778.	13.993	0.2980	0.1963	1.9794	0.0242

WASTE 111468. (kTONNES) ROM S/R= 1.35

Table 23.18
P656i North Intermediate Pit Incremental Resource

Bench	ORE Tonnage	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste Tonnage	Stripping Ratio
1345	-	0.00	0.000	0.000	0.00	0.000	3	-1
1330	-	0.00	0.000	0.000	0.00	0.000	2	-1
1315	-	0.00	0.000	0.000	0.00	0.000	58	-1
1300	-	0.00	0.000	0.000	0.00	0.000	63	-1
1285	-	0.00	0.000	0.000	0.00	0.000	284	-1
1270	-	0.00	0.000	0.000	0.00	0.000	317	-1
1255	-	0.00	0.000	0.000	0.00	0.000	563	-1
1240	-	0.00	0.000	0.000	0.00	0.000	564	-1
1225	-	0.00	0.000	0.000	0.00	0.000	857	-1
1210	-	0.00	0.000	0.000	0.00	0.000	859	-1
1195	-	0.00	0.000	0.000	0.00	0.000	1,223	-1
1180	-	0.00	0.000	0.000	0.00	0.000	1,279	-1
1165	-	0.00	0.000	0.000	0.00	0.000	1,708	-1
1150	-	0.00	0.000	0.000	0.00	0.000	1,752	-1
1135	-	0.00	0.000	0.000	0.00	0.000	2,245	-1
1120	-	0.00	0.000	0.000	0.00	0.000	2,302	-1
1105	-	0.00	0.000	0.000	0.00	0.000	2,836	-1
1090	-	0.00	0.000	0.000	0.00	0.000	2,863	-1

Table 23.18 P656i North Intermediate Pit Incremental Resource								
Bench	**ORE Tonnage**	**ORE NSR**	**ORE Cu**	**ORE Au**	**ORE Ag**	**ORE Mo**	**Waste Tonnage**	**Stripping Ratio**
1075	-	0.00	0.000	0.000	0.00	0.000	3,465	-1
1060	-	0.00	0.000	0.000	0.00	0.000	3,464	-1
1045	-	0.00	0.000	0.000	0.00	0.000	4,190	-1
1030	-	0.00	0.000	0.000	0.00	0.000	4,099	-1
1015	-	0.00	0.000	0.000	0.00	0.000	4,908	-1
1000	-	0.00	0.000	0.000	0.00	0.000	4,717	-1
985	19	15.99	0.319	0.273	3.92	0.025	5,543	288.69
970	241	14.99	0.299	0.227	3.46	0.023	5,246	21.79
955	955	15.45	0.305	0.263	3.05	0.023	5,521	5.78
940	1,389	15.33	0.306	0.261	2.77	0.024	5,099	3.67
925	2,147	15.08	0.305	0.256	2.45	0.024	5,578	2.6
910	2,654	14.48	0.302	0.218	2.17	0.024	5,426	2.04
895	4,313	13.85	0.294	0.200	2.06	0.023	5,820	1.35
880	5,691	13.55	0.293	0.173	1.92	0.024	4,840	0.85
865	7,079	13.60	0.292	0.180	1.91	0.024	5,325	0.75
850	6,648	13.52	0.289	0.188	1.91	0.024	3,862	0.58
835	8,089	13.95	0.298	0.200	1.92	0.024	3,563	0.44
820	7,329	13.69	0.290	0.196	1.87	0.024	2,603	0.36
805	6,929	13.63	0.286	0.197	1.86	0.024	2,312	0.33
790	5,887	13.79	0.293	0.200	1.89	0.024	1,588	0.27
775	5,275	14.18	0.304	0.205	1.96	0.024	1,427	0.27
760	4,374	14.05	0.298	0.202	1.93	0.024	955	0.22
745	4,015	14.08	0.299	0.196	1.92	0.025	755	0.19
730	3,088	14.46	0.312	0.190	1.96	0.026	451	0.15
715	2,608	14.77	0.327	0.177	1.97	0.027	417	0.16
700	1,650	14.83	0.333	0.171	2.02	0.027	221	0.13
685	1,310	14.55	0.322	0.166	2.02	0.027	177	0.13
670	718	14.86	0.330	0.161	2.13	0.028	79	0.11
655	370	14.78	0.332	0.144	2.13	0.029	40	0.11
Total	**82,778**	**13.99**	**0.298**	**0.196**	**1.98**	**0.024**	**111,468**	**1.35**

BENCH TOE	ZONE NAME	ZONE NO.	CUTOFF	INSITU ORE (kBCMS)	INSITU ORE (kTONNES)	RUN OF MINE (kTONNES)	DILUTED GRADES NSR	CU	AU	AG	MO
TOTALS	MEAS	1	5.70- 6.08	10.	27.	24.	6.023	0.1292	0.0951	1.4432	0.0087
			6.08- 8.00	400.	1076.	968.	6.953	0.1661	0.0971	1.5306	0.0081
			8.00- 10.00	278.	746.	672.	9.290	0.2443	0.0741	1.4011	0.0129
			10.00- 12.00	726.	1947.	1753.	10.869	0.2601	0.0916	1.5906	0.0197
			12.00- 14.00	667.	1784.	1610.	13.057	0.2898	0.1598	2.0655	0.0226
			14.00- 16.00	793.	2115.	1903.	15.105	0.2973	0.2016	2.2703	0.0323
			16.00- 18.00	1641.	4342.	3907.	16.876	0.3065	0.2497	2.3156	0.0390
			>= 18.00	2203.	5918.	5326.	19.914	0.3802	0.3226	2.7688	0.0396
			TOTALS:	6717.	17953.	16164.	15.710	0.3117	0.2253	2.2697	0.0317
	IND	2	0.0- 5.70	408.	1113.	1038.	5.094	0.1193	0.0603	1.5724	0.0065
			5.70- 6.08	160.	438.	397.	5.939	0.1428	0.0698	1.6709	0.0074
			6.08- 8.00	1549.	4213.	3875.	7.289	0.1794	0.0785	1.6553	0.0091
			8.00- 10.00	9861.	26812.	24349.	9.246	0.2409	0.0801	1.4744	0.0123
			10.00- 12.00	16244.	44148.	40050.	11.035	0.2706	0.1049	1.5402	0.0173
			12.00- 14.00	14837.	40286.	36619.	12.922	0.2931	0.1477	1.6406	0.0224
			14.00- 16.00	9857.	26696.	24226.	14.864	0.3110	0.2090	1.9368	0.0269
			16.00- 18.00	6741.	18138.	16492.	16.921	0.3303	0.2827	2.4376	0.0302
			>= 18.00	12259.	33492.	30339.	20.559	0.3983	0.3861	2.7482	0.0338
			TOTALS:	71916.	195337.	177384.	13.750	0.3009	0.1882	1.8991	0.0227
	INF	3	6.08- 8.00	47.	128.	120.	7.519	0.1862	0.0568	1.8312	0.0098
			8.00- 10.00	112.	306.	279.	9.181	0.2425	0.0616	2.0963	0.0119
			10.00- 12.00	84.	229.	206.	11.193	0.2658	0.0877	2.3433	0.0204
			12.00- 14.00	97.	264.	239.	13.121	0.3075	0.0829	2.2621	0.0269
			14.00- 16.00	78.	207.	208.	14.727	0.2721	0.2258	2.1512	0.0209
			TOTALS:	418.	1134.	1052.	11.375	0.2613	0.1034	2.1629	0.0185
TOTALS	SUMMARY		0.0- 5.70	408.	1113.	1038.	5.094	0.1193	0.0603	1.5724	0.0065
			5.70- 6.08	171.	465.	421.	5.944	0.1421	0.0713	1.6577	0.0075
			6.08- 8.00	1996.	5417.	4963.	7.229	0.1770	0.0816	1.6353	0.0090
			8.00- 10.00	10251.	27864.	25300.	9.246	0.2410	0.0797	1.4793	0.0123
			10.00- 12.00	17053.	46323.	42009.	11.029	0.2701	0.1043	1.5463	0.0174
			12.00- 14.00	15601.	42334.	38467.	12.929	0.2931	0.1478	1.6623	0.0224
			14.00- 16.00	10728.	29018.	26337.	14.880	0.3097	0.2086	1.9626	0.0273
			16.00- 18.00	8381.	22479.	20399.	16.912	0.3257	0.2764	2.4142	0.0319
			>= 18.00	14462.	39410.	35665.	20.463	0.3956	0.3766	2.7512	0.0346
			TOTALS:	79051.	214424.	194600.	13.900	0.3016	0.1909	1.9313	0.0235

WASTE 571117. (kTONNES) ROM S/R= 2.93

Table 23.19
P666i Ultimate Pit Incremental Resource

Bench	ORE Tonnage	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste Tonnage	Stripping Ratio
1690	-	0.00	0.000	0.000	0.00	0.000	810	-1
1675	-	0.00	0.000	0.000	0.00	0.000	1,518	-1
1660	-	0.00	0.000	0.000	0.00	0.000	1,786	-1
1645	-	0.00	0.000	0.000	0.00	0.000	2,524	-1
1630	-	0.00	0.000	0.000	0.00	0.000	2,735	-1
1615	-	0.00	0.000	0.000	0.00	0.000	3,410	-1
1600	-	0.00	0.000	0.000	0.00	0.000	3,586	-1
1585	-	0.00	0.000	0.000	0.00	0.000	4,106	-1
1570	-	0.00	0.000	0.000	0.00	0.000	4,082	-1
1555	-	0.00	0.000	0.000	0.00	0.000	4,612	-1
1540	-	0.00	0.000	0.000	0.00	0.000	4,606	-1
1525	-	0.00	0.000	0.000	0.00	0.000	5,242	-1
1510	-	0.00	0.000	0.000	0.00	0.000	5,301	-1
1495	-	0.00	0.000	0.000	0.00	0.000	5,969	-1

**Table 23.19
P666i Ultimate Pit Incremental Resource**

Bench	ORE Tonnage	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste Tonnage	Stripping Ratio
1480	-	0.00	0.000	0.000	0.00	0.000	5,998	-1
1465	-	0.00	0.000	0.000	0.00	0.000	6,549	-1
1450	-	0.00	0.000	0.000	0.00	0.000	6,558	-1
1435	-	0.00	0.000	0.000	0.00	0.000	7,034	-1
1420	-	0.00	0.000	0.000	0.00	0.000	6,984	-1
1405	-	0.00	0.000	0.000	0.00	0.000	7,443	-1
1390	-	0.00	0.000	0.000	0.00	0.000	7,333	-1
1375	-	0.00	0.000	0.000	0.00	0.000	7,896	-1
1360	-	0.00	0.000	0.000	0.00	0.000	7,872	-1
1345	-	0.00	0.000	0.000	0.00	0.000	8,475	-1
1330	-	0.00	0.000	0.000	0.00	0.000	8,471	-1
1315	-	0.00	0.000	0.000	0.00	0.000	9,041	-1
1300	-	0.00	0.000	0.000	0.00	0.000	9,060	-1
1285	-	0.00	0.000	0.000	0.00	0.000	9,558	-1
1270	-	0.00	0.000	0.000	0.00	0.000	9,429	-1
1255	-	0.00	0.000	0.000	0.00	0.000	9,751	-1
1240	-	0.00	0.000	0.000	0.00	0.000	9,651	-1
1225	-	0.00	0.000	0.000	0.00	0.000	9,918	-1
1210	-	0.00	0.000	0.000	0.00	0.000	9,761	-1
1195	-	0.00	0.000	0.000	0.00	0.000	10,005	-1
1180	-	0.00	0.000	0.000	0.00	0.000	9,932	-1
1165	-	0.00	0.000	0.000	0.00	0.000	10,240	-1
1150	-	0.00	0.000	0.000	0.00	0.000	10,331	-1
1135	-	0.00	0.000	0.000	0.00	0.000	10,711	-1
1120	-	0.00	0.000	0.000	0.00	0.000	10,790	-1
1105	-	0.00	0.000	0.000	0.00	0.000	11,025	-1
1090	-	0.00	0.000	0.000	0.00	0.000	10,960	-1
1075	-	0.00	0.000	0.000	0.00	0.000	11,088	-1
1060	-	0.00	0.000	0.000	0.00	0.000	11,055	-1
1045	-	0.00	0.000	0.000	0.00	0.000	11,220	-1
1030	-	0.00	0.000	0.000	0.00	0.000	11,286	-1
1015	-	0.00	0.000	0.000	0.00	0.000	11,533	-1
1000	-	0.00	0.000	0.000	0.00	0.000	11,511	-1
985	-	0.00	0.000	0.000	0.00	0.000	11,426	-1
970	5	13.94	0.208	0.228	1.80	0.013	11,102	2312.81

Table 23.19
P666i Ultimate Pit Incremental Resource

Bench	ORE Tonnage	ORE NSR	ORE Cu	ORE Au	ORE Ag	ORE Mo	Waste Tonnage	Stripping Ratio
955	258	13.53	0.257	0.237	2.03	0.016	10,720	41.61
940	799	13.76	0.273	0.241	2.08	0.020	10,317	12.91
925	1,259	14.82	0.285	0.251	2.21	0.024	10,123	8.04
910	1,704	15.39	0.292	0.258	2.31	0.028	9,998	5.87
895	1,762	15.68	0.290	0.260	2.31	0.031	10,414	5.91
880	2,445	15.09	0.285	0.230	2.16	0.030	11,294	4.62
865	3,377	13.85	0.274	0.185	1.88	0.028	13,428	3.98
850	4,224	13.16	0.268	0.170	1.76	0.026	13,260	3.14
835	4,897	12.84	0.266	0.162	1.71	0.025	12,673	2.59
820	5,793	12.80	0.273	0.159	1.70	0.024	12,062	2.08
805	6,253	12.73	0.276	0.155	1.70	0.023	11,122	1.78
790	7,549	12.84	0.281	0.158	1.74	0.023	11,296	1.5
775	8,822	12.69	0.285	0.155	1.74	0.021	9,221	1.05
760	10,663	12.84	0.293	0.162	1.79	0.020	8,273	0.78
745	11,305	12.74	0.293	0.164	1.85	0.019	6,327	0.56
730	12,777	12.88	0.294	0.170	1.88	0.020	5,852	0.46
715	12,696	13.16	0.301	0.175	1.90	0.020	4,242	0.33
700	13,739	13.40	0.301	0.183	1.94	0.021	3,679	0.27
685	13,171	13.68	0.306	0.187	1.94	0.022	2,698	0.2
670	13,112	13.84	0.305	0.191	1.96	0.023	2,576	0.2
655	11,611	14.20	0.311	0.195	1.97	0.024	2,086	0.18
640	10,801	14.98	0.326	0.206	2.04	0.026	1,972	0.18
625	9,111	15.54	0.333	0.219	2.08	0.027	1,543	0.17
610	8,556	15.76	0.329	0.231	2.12	0.028	1,362	0.16
595	6,732	16.24	0.334	0.246	2.13	0.029	976	0.15
580	5,322	16.55	0.332	0.266	2.14	0.029	884	0.17
565	3,361	16.71	0.329	0.280	2.21	0.029	825	0.25
550	1,569	15.70	0.316	0.247	2.04	0.027	294	0.19
535	927	14.73	0.294	0.221	2.05	0.024	312	0.34
Total	**194,600**	**13.90**	**0.302**	**0.191**	**1.93**	**0.024**	**571,117**	**2.93**

23.1.9 Mine Production Schedule

23.1.9.1 *Mine Load and Haul Fleet Selection*

The mine load and haul fleet is selected prior to production scheduling. Similar projects in the area have shown that the lowest cost/tonne fleet of cable shovels and haul trucks for large hard rock open pit mines that are currently being used are the 80 Ton class shovel matched with the 360 Ton class truck. A large wheel loader and suitable drills to match this size of truck/shovel fleet front end loaded are indicated in the work below. The performance and costs of a P&H4100XPB cable shovel matched with CAT 797B haulers, a Letourneau L-2350 wheel loader, P&H 120A Drill and P&H 250XP Drill are used in the following work. This is not an endorsement of these brands but is just used for typical performance, productivity, and cost information for this class of equipment. Future studies will include detailed performance and cost evaluations of other brands, including discussions with the OEM's.

23.1.10 Life of Mine Production Schedule

The mine production schedule after pre-stripping is developed with MineSight Strategic Planner (MS-SP), a comprehensive long range scheduling tool for open pit mines. It is typically used to produce a life-of-mine schedule that will maximize the Net Present Value of a property subject to user specified conditions and constraints. Annual production requirements, mine operating considerations, product prices, recoveries, destination capacities, equipment performance and operating costs are used to determine the optimal production schedule. Scheduling results are presented by period as well as cumulatively and include:

- Tonnes and Grade mined by period broken down by material type, bench and mining phase.
- Truck and Shovel requirements by period in number of units and number of operating hours
- Tonnes transported by period to different destinations (mill, stockpiles and waste dumps)

Full production mill feed is expected to commence in January 2011. The production schedule uses 'PP' as pre-production, 'Year1' as 2011, 'Year2' as 2012 etc.

23.1.10.1 *Pre Production Mining*

The planned pre-production allows for a smooth ramp up of the major mining fleet. Pre-production of 4 MT of pit waste is used for an access road external to the pit. Additional pit waste is used for a mill pad and stockpile base. An estimated 12 MT of pre-production waste is available for the above and any other construction fills. This has a positive contribution to the project value by avoiding the high incremental cost of mining the required rock from borrow sources.

23.1.10.2 *Schedule Criteria*

The Schaft Creek schedule setup includes:

- Truck efficiencies are based on the equipment operations efficiency from the design basis.

Table 23.20
Truck Availability and Cost Assumptions for MS-SP

Up to Hours	% Availability
7000	87.3
14000	86.3
21000	83.4
28000	83.4
35000	82.5
42000	82.5
49000	80.5
56000	80.5
63000	79.5
400000	79.5

Table 23.21
P&H4100XPB Availability and Cost Assumptions for MS-SP

Up to Hours	% Availability
7000	85.4
14000	83.4
21000	82.5
28000	81.5
35000	77.6
42000	81.5
49000	81.5
56000	81.5
63000	81.5
70000	77.6
77000	81.5
400000	80.5

Table 23.22 L2350 Availability and Cost Assumptions for MS-SP	
Up to Hours	% Availability
6000	85.4
12000	83.4
18000	81.5
24000	81.5
30000	81.5
36000	80.5
400000	80.5

Load times for the P&H4100XPB, RH400 and L2350 include operator efficiency.

At this time only mill feed material types are defined. In order to optimize the project NPV grade bins have been specified (based on NSR values in each block of the block model). It is assumed that blasthole assays will be suitable for specifying mill feed and low grade stockpile cut off grades for a cut off grade strategy. This is typical for bulk mining in these types of deposits. The specifications in the table below are used within the MS-SP optimized scheduler. Mining operations do not use these many bins in the field.

Table 23.23 Material Types Defined For MS-SP	
NSR Grade Bin	Reserve Class
Sub ($4.25/t – $5.70/t)	Sub Grade
Low ($5.70/t – $6.08/t)	Low Grade
Mid ($6.08/t – $8.00/t)	Mid Grade
Hi1 ($8.00/t – $10.0/t)	High Grade Bin 1
Hi2 ($10.0/t – $12.0/t)	High Grade Bin 2
Hi3 ($12.0/t – $14.0/t)	High Grade Bin 3
Hi4 ($14.0/t – $16.0/t)	High Grade Bin 4
Hi5 ($16.0/t – $18.0/t)	High Grade Bin 5
Hi6 (> $18.0/t)	High Grade Bin 6

Only one waste type is assumed. All waste is treated as Non-Potentially Acid Generating (NPAG).

Mining precedence is required to specify the phase mining order based on relative location of the phases. For example if the phases represent progressive expansions of a single hole in the ground, then the first expansion must stay ahead of the second expansion and so on.

Table 23.24 Schaft Creek Mining Precedence		
Phase A ID	**Constraint**	**Phase B ID**
P626I	After	P616
P636I	After	P626I
P666I	After	P636I
P656I	After	P646
P666I	After	P656I

The primary program objective in each period is to maximize NPV. The MS-SP NPV calculation is guided by following inputs:

- 6% discount rate.
- $0.50/tonne for fixed mining costs.
- $4.25 processing costs
- Net Smelter metal prices of $1.30/lb Cu, $16.48/Au, $0.269/Ag.

There are 360 mine operating days scheduled and 21 hours per day.

A default cycle times of 20 minutes is assumed for most hauls. The default cycle time is only used if a valid calculated cycle time is not available.

Annual mill feed of 23,400 ktpa is targeted based on 65,000 tonnes/day ore milling.

Haul and Return Times are estimated using simulations from CAT's FPC program. Productivity calculations used the following criteria:

- For all benches in all pits, the haul times, return times and fuel burn are linearly interpolated based on the haul and return times.
- Haul and return distances for all pits from the bottom of the pit, a mid point in the pit, the pit rim and a point in the upper benches of the pit.
- The actual haul and return times, as well as the fuel burn on each route were derived from FPC.
- The haul and return times were derated by 90% operator efficiency.
- Dump and Maneuver time of 1.5 minutes was also used including a derate for hauler operator efficiency.
- The derated haul, return, dump and maneuver times are added and used as the cycle time in Strategic Planner. The linear interpolation of truck cycle times is carried out for all phases from all benches to all estimated destinations.

Cut off Grade Optimization

Typically the mill feed grade can be increased by sending low and mid grade classes to stockpiles whilst simultaneously preventing stockpile reclaim. The mill feed rate is maximized and this effectively increases the revenue per tonne milled. Stockpiling also results in increased total mined rock and the mine cost per tonne milled also increases. At some point the cost of mining more material will exceed the incremental revenue from the higher grade milled.

The current schedule routes grade bins as follows:

- Sub grade is wasted.
- Low grade is sent to the Low grade stockpile
- Mid grade is sent to the Mid grade stockpile

The grade distribution in the current model is of such a nature at Schaft Creek that small increases in cutoff grade result in radical increases in stockpiled material. This Preliminary Economic Assessment (PEA) has not extensively tested optimized Cut off Grade strategy and this remains an upside potential for future studies.

Stockpile Grades

Mill Feed grades reported in the production schedule include a loss of 5% points in mill recovery to account for oxidation for an extended period on the pile.

Schedule Results

The summarized production schedule results are shown in the table below.

Table 23.25
Summarized Production Schedule

		PP	Year 1-5	Year 6-10	Year 11-15	Year 16-31	LOM
ROM Mined							
ROM to crusher	kTonnes	-	117,050	106,116	103,712	272,772	599,650
Cu	%	-	0.350	0.342	0.288	0.316	0.322
Au	g/t	-	0.268	0.207	0.245	0.221	0.232
Ag	g/t	-	1.869	1.685	1.665	1.943	1.835
Mo	%	-	0.018	0.017	0.020	0.026	0.022
ORE to stockpiles	kTonnes	10,154	6,157	15,095	44,553	37,779	113,737
Total ROM Mined	kTonnes	10,154	123,207	121,211	148,265	310,551	713,387
from stockpiles	kTonnes	-	-	10,921	13,338	89,478	113,737
Cu	%	-	-	0.27	0.26	0.20	0.215
Au	g/t	-	-	0.17	0.17	0.14	0.147
Ag	g/t	-	-	1.23	1.28	1.50	1.448
Mo	%	-	-	0.013	0.011	0.011	0.011
Stockpile Inventory	kTonnes	10,154	16,311	20,485	51,700	-	-
Total Mill Feed	kTonnes	-	117,050	117,037	117,050	362,250	713,387
Cu	%	-	0.350	0.334	0.283	0.285	0.303
Au	g/t	-	0.268	0.203	0.235	0.199	0.217
Ag	g/t	-	1.869	1.637	1.613	1.815	1.761
Mo	%	-	0.018	0.017	0.019	0.022	0.020
Waste							
Sub Grade to Waste	kTonnes	-	30	3	4,122	1,548	5,703
Waste Mined	kTonnes	12,035	237,803	253,006	203,315	486,257	1,192,417
Total Waste Mined	kTonnes	12,035	237,833	253,010	207,437	487,805	1,198,120
As Mined S/R		1.2	1.9	2.1	1.4	1.6	1.7
Effective S/R		-	3.1	3.2	3.0	2.2	2.7
Total Material *Mined*	kTonnes	22,190	361,040	374,220	355,702	798,355	1,911,507
Total Material *Moved*	kTonnes	22,190	361,040	385,142	369,040	887,834	2,025,245

NOTE:
*As Mined S/R = (Total Waste/ Ore Mined)
**Effective S/R = (Total Waste + Stockpiled/Mill Feed)

23.1.10.3 *Pit End of Period Maps*

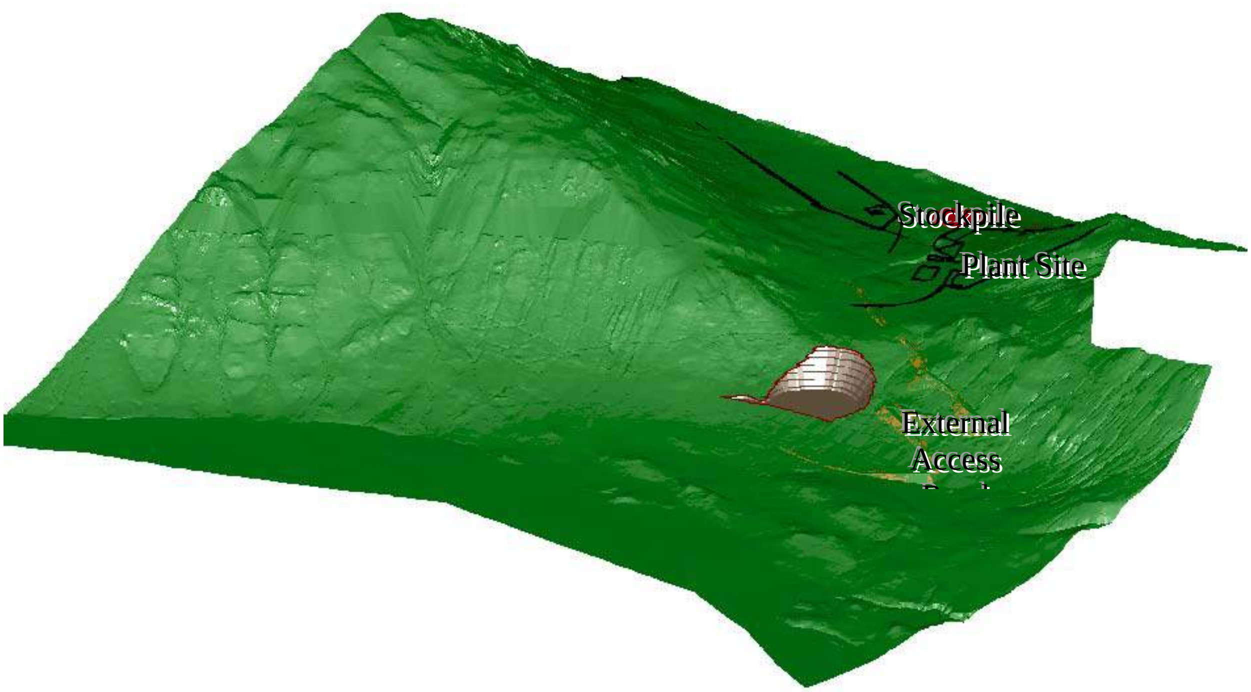
End of period maps for PP, Year 5, and Life of Mine are shown below.

23.1.10.4 *Pre-Production*

Mining is carried out in the South Starter Pit (P616) which is mined down to the 970 m bench. Access to the P616 pit is from external roads on the south end.

Waste is used to fill out the external haul road, and then hauled to the plant site pad, and stockpile pad. All ore above subgrade is placed on the stockpile.

Figure 23.20 End of Pre-Production



23.1.10.5 *End of Year 5*

The plant starts up in Year 1 and ore is processed for the first time. Mining is carried out in pits P616, P626I, P636I, P646 from Year 1 to Year 5.

- P616 Is mined down to the 790 m bench
- P626I Mined to 910 m bench
- P636I Mined to 1225 m bench
- P646 Mined to 865 m bench

The period map below illustrates mining up to Year 5. Waste from the upper benches of the south pits is taken to the East Dump. Waste from the Mid benches is hauled to the South Dump, and waste from the bottom benches is hauled to the West Dump. All waste from the North pits is hauled to the North Dump.

Figure 23.21 End of Year 5

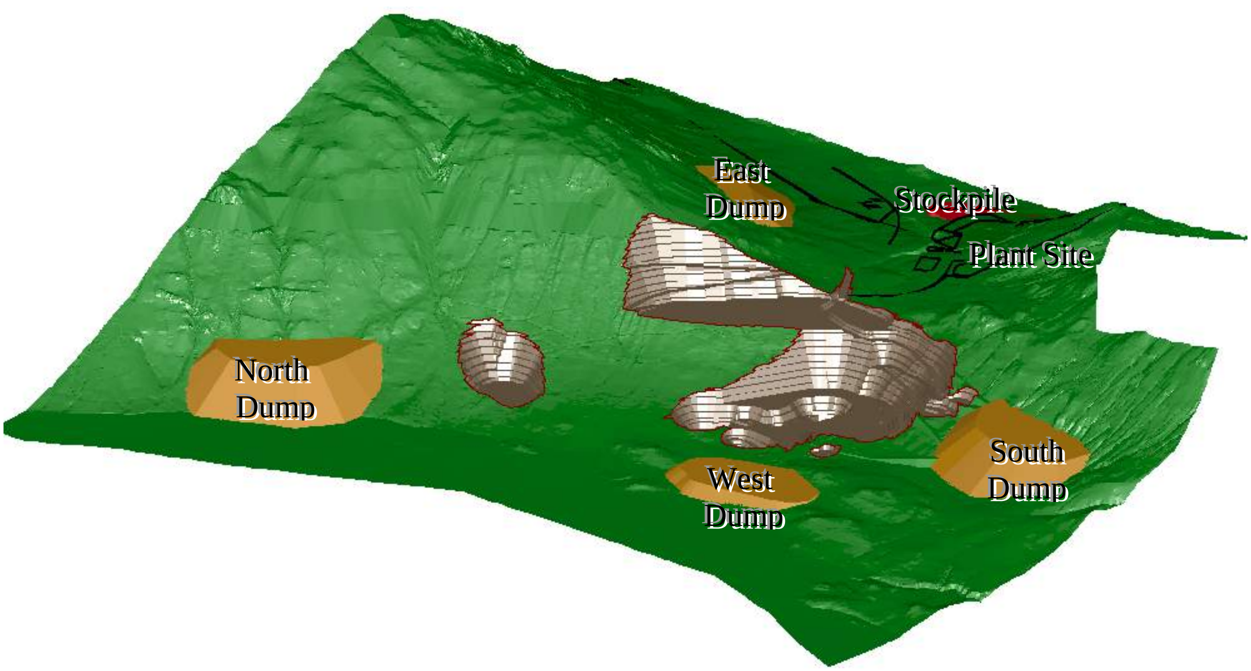
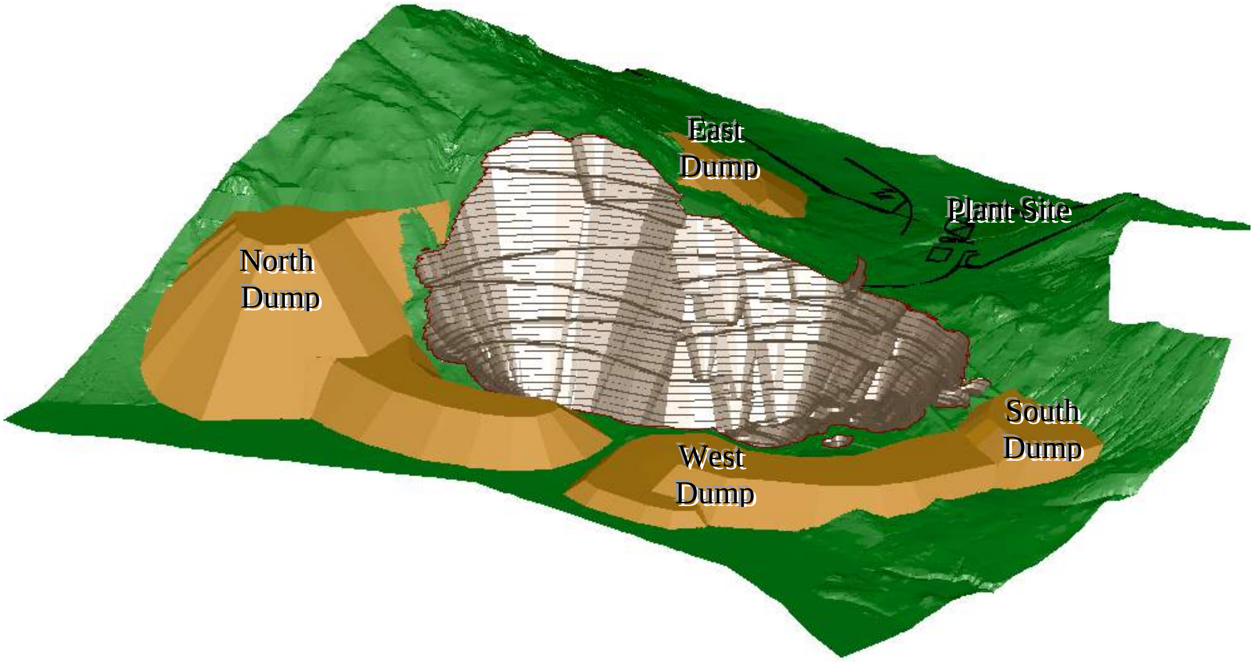


Figure 23.22 End of Life of Mine



A schedule of the total material mined (waste + ore) from each phase is illustrated in the graph below.

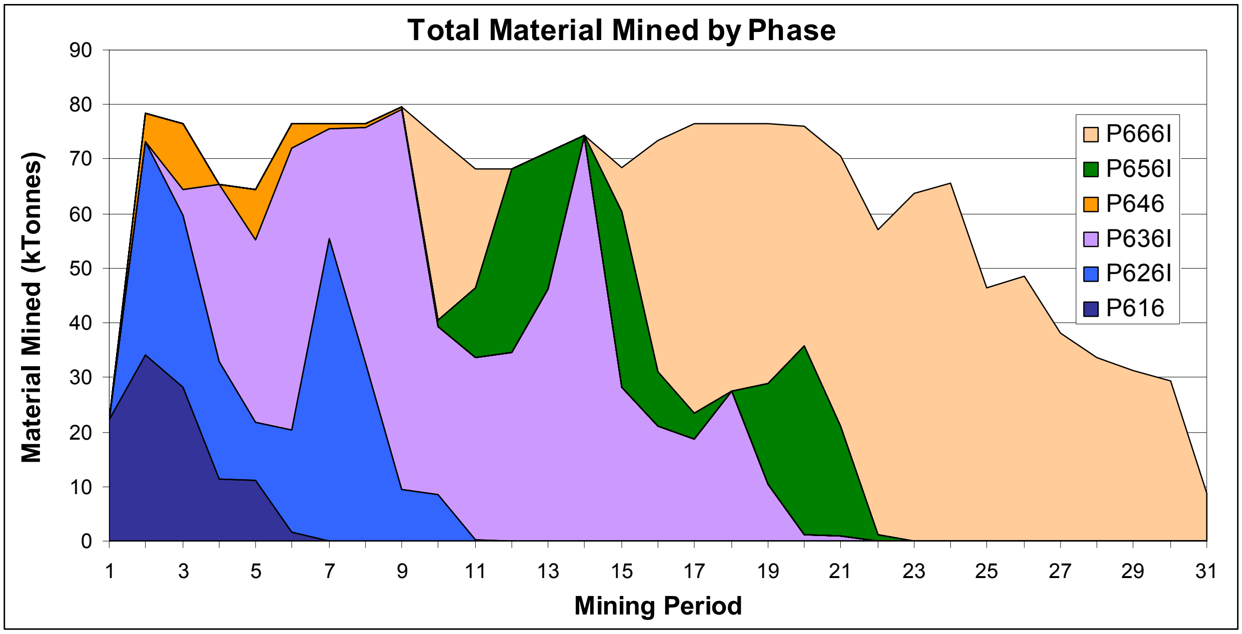


Figure 23.23 Schedule of Material Mined

23.1.10.6 *Waste Rock Storage*

Design Parametres

All dumps are designed with a natural angle of repose of 37°. Output from MS-SP reports tonnages of material by type. Only one type of waste material is scheduled at this time. A 30% swell factor is then applied to these in-situ volumes to calculate the loose dumped volumes that need to be placed in waste storage dumps.

Annual waste volumes produced from Schedule 5b are shown below.

Table 23.26
Waste Production Schedule (LOM)

Period	P616	P626I	P636I	P646	P656I	P666I	TOTAL
PP	5,860	-	-	-	-	-	5,860
Year1	5,390	18,656	-	2,381	-	-	26,427
Year2	7,024	11,033	2,232	4,490	-	-	24,777
Year3	780	3,279	15,788	-	-	-	19,847
Year4	484	1,572	16,271	1,363	-	-	19,691
Year5	-	1,187	23,853	-	1	-	25,042
Year6	-	15,269	9,211	87	-	-	24,567
Year7	-	6,779	16,807	-	-	-	23,587
Year8	-	1,407	23,504	-	-	-	24,911
Year9	-	369	8,180	-	626	16,166	25,341
Year10	-	18	8,010	-	6,227	10,526	24,781
Year11	-	-	6,985	-	16,421	-	23,406
Year12	-	-	8,310	-	11,589	-	19,899
Year13	-	-	13,938	-	-	-	13,938
Year14	-	-	3,687	-	10,425	3,860	17,972
Year15	-	-	1,292	-	1,881	20,604	23,777
Year16	-	-	-	-	-	25,804	25,804
Year17	-	-	2,003	-	-	23,823	25,826
Year18	-	-	554	-	1,626	23,149	25,330
Year19	-	-	60	-	3,970	19,603	23,633
Year20	-	-	118	-	1,449	24,155	25,722
Year21	-	-	-	-	58	26,721	26,779
Year22	-	-	-	-	-	26,385	26,385
Year23	-	-	-	-	-	22,787	22,787
Year24	-	-	-	-	-	12,720	12,720
Year25	-	-	-	-	-	9,113	9,113
Year26	-	-	-	-	-	4,628	4,628
Year27	-	-	-	-	-	2,910	2,910

Table 23.26 Waste Production Schedule (LOM)							
Year28	-	-	-	-	-	2,352	2,352
Year29	-	-	-	-	-	1,975	1,975
Year30	-	-	-	-	-	791	791

The potential to backfill mined out pit phases has been considered but not modeled at this stage. Backfilling is very dependent on mining sequence and advance of the pits. Backfilling should be considered in further detail as the mining progresses. Any realized backfill opportunities will reduce the haul distance required, reduce the area disturbed by mining, and reduce reclamation efforts. This will lead to savings in the over project mining costs.

Dump Monitoring and Planning

The high lift dumping approach that is most economical will be done with due diligence and control. Experience in the Elk Valley mining area of British Columbia has shown that high lift dumps can be built in mountainous terrain with minimal risk if proper practices and procedures are followed. Operating experience will be used to refine the design criteria and operating practices for monitoring and control of dumping activities in the Schaft Creek mining operations. With careful dump planning and a formal monitoring program, safe operation of the dumps can be expected including high dumps.

Dumping during the initial stages of mining will be done with low lifts in areas that are non-critical. As experience is gained and stable foundations are established, dumping can proceed with higher lifts as required.

Dump Reclaim

The dump designs include terraces from wrap around stages. These reduce the uphill haul requirements as the pit benching mines downward and also reduce the amount of material to be moved to reslope the final reclaimed dump faces upon completion. Where possible, dumps will be resloped and revegetated progressively rather than all dumps at the end of mining activities. This will reduce erosion of the dumps during operations, reduce closure costs at the end of operations, and help establish effective and final reclamation criteria.

23.1.11 Mine Operations

The mining operations will be typical of Open pit operations in mountainous terrain in western Canada and will employ tried and true mining methods and equipment. There is a wealth of operating and technical expertise, services and support in western Canada, British Columbia, and in the local area for the proposed operations. As already discussed, a large capacity operation is being designed and large sized equipment is being specified for the major operating areas in the mine to generate high productivities. This will reduce unit mining costs and as a result, provide for the lowest mining costs. Large scale equipment will also reduce the manpower requirement on site and will dilute the fixed overhead costs for the mine operations. Much of the general overhead for the mine operations can be minimized if the number of production fleets and the manpower requirements are minimized.

The Mine Operations are organized into three areas, Direct Mining, Mine Maintenance, and General Mine Expense (GME).

Direct mining includes the equipment operating costs, operating labour, and distributed Mine Maintenance costs for the Drilling, Blasting, Loading, Hauling, and Pit Maintenance activities in the mine. The distributed mine maintenance includes items such as Maintenance labour and repair parts which contribute to the hourly operating cost of the equipment.

Mine Maintenance accounts for the supervision, planning, and general shop costs of the Mine Maintenance activities. The costs in these items are not distributed to the equipment fleets.

GME includes the Supervision and training for the Direct mining activities as well as technical support from Mine Engineering and Geology functions. More detailed descriptions of the mine organization and the unit mining activities follows.

In this study the Direct Mining and Mine Maintenance is planned as an owner's fleet with the equipment ownership and manpower being directly under operations. It may be possible to contract out some of the mining activities under typical mine stripping and Maintenance and Repair Contracts (MARC) as has been done at other operations. The viability and cost effectiveness of contracting can be determined in future detailed and commercial planning. The exception is in blasting, where the typical approach is to have the mine employ the blasters but to contract out the supply and on-site manufacturing of blasting materials due to the specialized expertise required.

23.1.12 Organization

23.1.12.1 *General Organization*

The Schaft Creek Property will be organized in the following general manner with most of the functions on-site; however, some financial and administrative support will be done off-site. This report deals solely with Mine operations indicated in orange in the figure below.

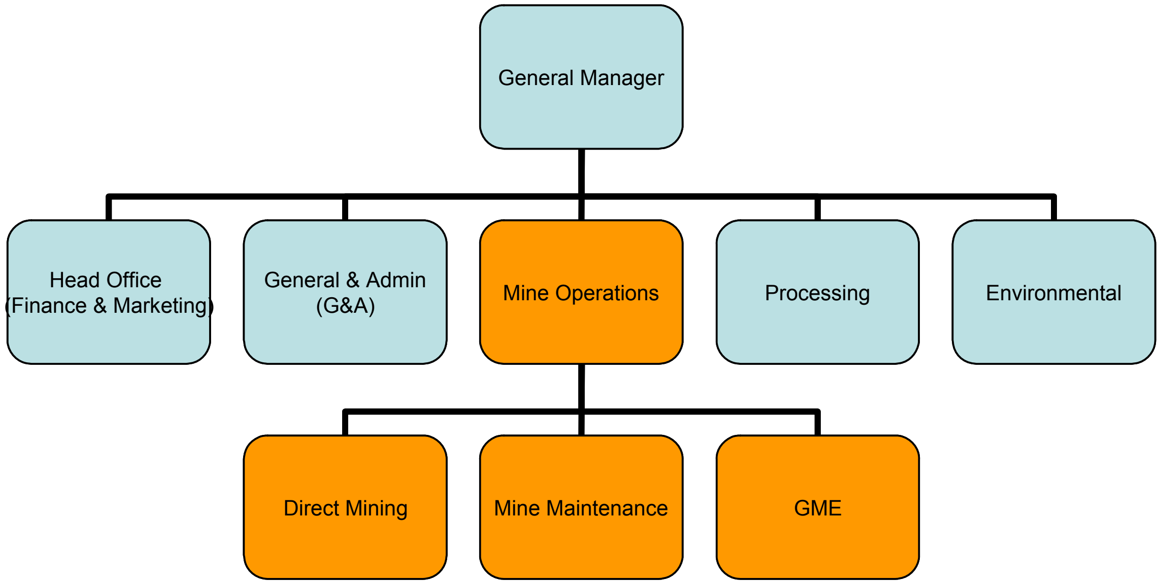


Figure 23.24 General Organization Chart

23.1.12.2 *Mine Operations Organization*

Details of Mine Operations are given below showing the breakdown of the Direct Mining, Mine Maintenance, and GME functions.

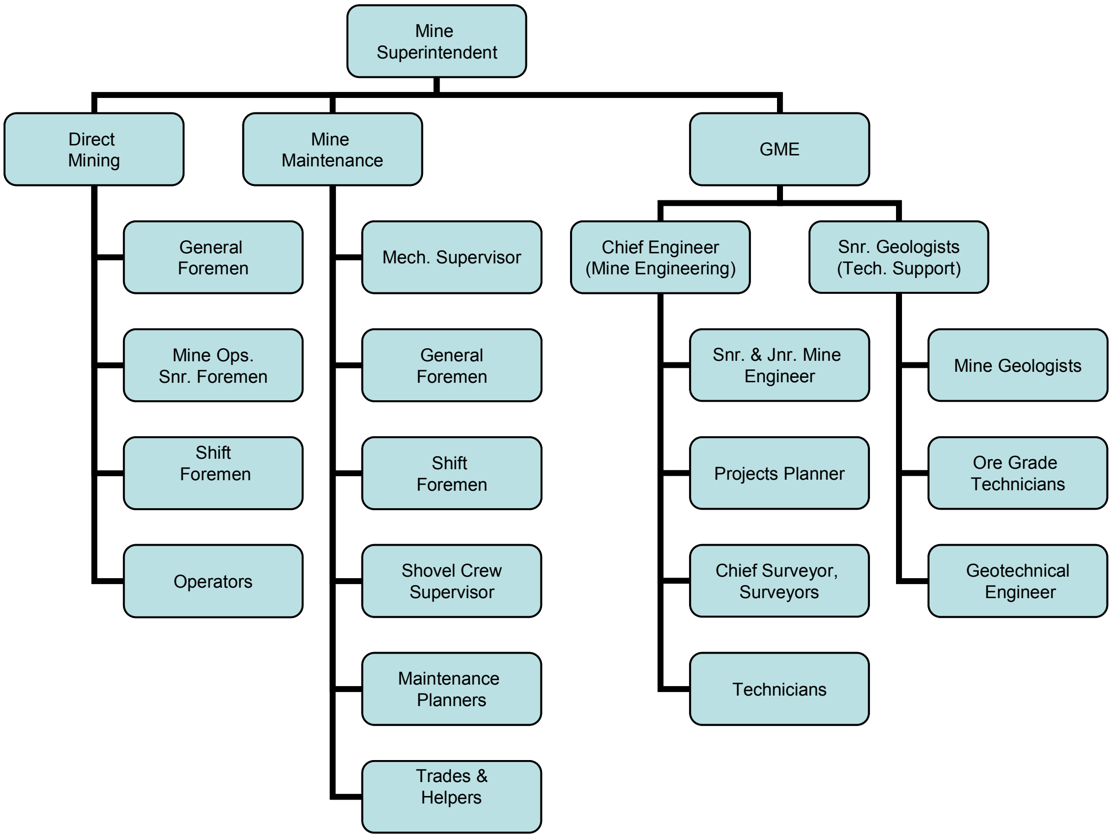


Figure 23.25 Mine Operations Organization Chart

23.1.13 Direct Mining Unit Operations

In-situ rock will require drilling and blasting to create suitable fragmentation for efficient loading and hauling of both waste and ore material. For this study the drilling and blasting design is to provide a particle size distribution and diggability (Looseness in the muck pile) suitable for high productivity from the selected shovel and truck fleet. Some operations also increase the blasting energy in ore to enhance ore comminution (crushing and grinding). Blasting for improved mine to mill performance can be optimized in future studies.

Mill feed and waste will be defined in the blasted muck pile and a fleet management system will assist in optimizing the fleet deployment and utilization to meet the production plan, to track each load to ensure material is hauled to the correct destination, as well as to provide production statistics for management and reconciliation of the mine operations with respect to the mine plan. Descriptions of the unit mining operations follow.

23.1.13.1 *Drilling*

Areas will be prepared on the bench floor to provide rows of holes. The spacing and burden will be varied as required to meet the specified powder factor for the various rock types. Dozers will be used to establish flat working areas in the overburden and weathered slopes of the initial hill side benches. Ramps will be cut on these upper slopes and where the established benches don't provide drill access close enough to the edge to meet the burden and spacing requirement of the pattern for the next bench below.

The blasthole drills will be fitted with GPS navigation and drill control systems to optimize drilling. The GPS navigation will allow for stake-less drilling and is considered a necessity due to high snow levels at the Schaft Creek site. It is a proven technology utilized at most mines in Western Canada. The drills will also be fitted with samplers to provide grade control samples from the drill cutting in the ore zones. The drillers will take the cuttings samples (two to three samples per hole may be required) and bag and tag the samples for the ore control technician to collect each day. These samples will be used for blast hole Kriging to define the ore waste boundaries on the bench as well as stockpile grade bins for the grade control system to the mill.

Two types of drills have been specified, electric rotary drills for the majority of the drilling, and a single more mobile diesel hydraulic drill with a similar hole size. The diesel drill will act as a supplemental drill and will provide flexibility when frequent moves are required. Drilling production assumptions are listed below.

Table 23.27 Drilling Production Assumptions				
Drill Production	**Electric Rotary**		**Diesel Hydraulic**	
Bench Height	15	m	15	m
Subgrade	1.5	m	1.5	m
Holes size	311	mm	311	mm
Penetration Rate	25.2	m/hr	22.7	m/hr
Hole depth	16.5	m	16.5	m
Setup Time	2.0	minutes	2.0	minutes
Drill Time	39.3	minutes	43.6	minutes
Move Time	2.0	minutes	3.0	minutes
Total Cycle Time	43.3	minutes	48.6	minutes
Holes per Hour	1.4		1.2	

A 150 mm diesel highwall drill is also specified to operate in all pits for controlled blasting techniques on high wall rows and development of initial upper benches. The highwall drill and the development drilling requirements have not been detailed in this study. An allowance of 15% of the production drill hours has been used as an allowance for costing purposes.

23.1.14 Blasting

Powder Factor

Blasting will be required to ensure fragmentation of the muckpile and ease of digging or "Diggability". This is a function of various aspects of the rock strength and fabric used in the blast design and affected by the applied powder factor. A "hard digging" muckpile will cause poor productivity in the loading fleet, excessive bucket and tooth wear, increased maintenance costs, poor load profiles, longer loading times for the trucks and will adversely effect areas as diverse as mining plan conformance and mill through put. The most common approach to the provision of a muckpile which is "easy" to dig is the combination of long inter row delay and relatively high powder factor, promoting extensive lateral movement and vertical heave within the rock mass.

A similar large open pit project in the Schaft Creek area uses a powder factor of 0.32 kg/t for competent rock which will achieve a fragmentation adequate for the size shovels to be used at Schaft Creek. Informal discussions with other mines and explosive suppliers in British Columbia confirm a power factor of 0.32 kg/t is suitable in this area. A detailed blasting study should be done in future more advanced studies for the project.

Explosives

A contract explosives supplier will provide the blasting materials and technology for the mine. Because of the remote nature of the operation, an explosives plant will be built on site. The exact location of this plant will be affected by the table of distances that govern the storage of explosives and blasting agents such as ANFO (Ammonium Nitrate Fuel Oil). The nature of the business relationship between the explosives supplier and the mining operator will determine who is responsible for obtaining the various manufacture, storage and transportation permits as well as any necessary licenses for blasting operations. This will be established during commercial negotiations.

Until it is determined the extent of ground water and surface water in the blast holes, it is assumed 70/30 ANFO and emulsion mix explosive will be used since ANFO alone has no water resistance. The inclusion of an emulsion provides water resistance. Higher use of ANFO and possible borehole liners to keep the ANFO dry to prevent incomplete detonations can be investigated in future studies to reduce blasting costs.

Explosives Loading

Loading of the explosives will be done with bulk explosives loading trucks provided by the explosives supplier. The trucks should be equipped with GPS guidance and be able to receive automatic loading instructions for each hole from the engineering office. This practice is common now in Western Canada and the explosives supplier’s truck has this capability already installed. The GPS guidance will be a necessity since the high snow loads will require stakeless patterns. The explosives product that is being used is a mix of ANFO and emulsion; therefore, the container on the truck will have two separate compartments. The separation will be set at the proper ratio so that both compartments will be emptied at the same time. This will minimize trips back and forth from the blast pattern to the explosives storage site.

The holes will also have to be stemmed to avoid fly-rock and excessive air blasts. Crushed rock will be provided for stemming material and will be dumped adjacent to the blast pattern. A loader with a side dump bucket is included in the mine fleet to tram the crush to each hole dump the stemming material into the hole.

From time to time during the high snow fall period it is expected that some of the mining areas will be shutdown and may lose regular road access. If a pattern is partially loaded it will be necessary to tie-in the loaded holes and blast before the snow accumulation gets too high to find the surface lines for tie-in. To blast a partial blast it will be necessary to ‘square-off’ the pattern by loading some holes to complete some rows in the pattern. To this end a specialty loading unit may be required during months of high snow fall. The unit (called a Goat by one explosives supplier) is similar to a skidder used in the logging industry (a large tired 4-wheel drive tractor) or alternatively a highway truck fitted with tracks similar to ‘Nodwell’ units used in the oil and gas industry. This will be fitted with bulk explosives tank and explosives pumping/mixing capabilities. The need for a ‘Goat’ is included as a contingency. This unit may be excluded in future studies as the amount of snow and the impact on operations is quantified.

Blasting Operations

The blasting crew will be mine employees and will be on day shift only. Based on existing mines of similar size and previous experience, the estimated crew size will be four people. The main duties of the blasting crew will include setting up guard fences around the loading area, guiding and directing the explosives loading truck, preparing the boosters and primers ahead of the actual loading of the holes, stemming the blast holes after they are loaded, tie-in of the blast patterns and detonating of the blasts.

The blasting crew will coordinate the drilling and blasting activities to ensure a minimum two weeks of broken material inventory is maintained for each shovel. The drilling areas and ramps for the hillside holes, will be prepared in suitable time for the next pattern and ramps will be surveyed if required. In winter the pattern preparation will also include snow removal.

Due to the high snow load the drilled holes will need to be covered. Also, the blast patterns will not be staked therefore the blasting activities will also have to have GPS control. The blasters will require hand held GPS to identify the holes for the pattern tie-in.

A B-line, Nonel Caps, and Detonating Cord detonation system is used. The blasting crew will place the down hole boosters and down lines and will tie-in the pattern with surface delays. The pattern size may be limited by the rate of snow fall in some months. As the snow depth gets too high it will be problematic to find the holes and the down line making it difficult to tie-in the blast. This may require smaller more frequent blasts to complete smaller patterns before the snow gets too deep.

The blasting contractor’s employees will supply and deliver explosives on the pattern to the hole. The explosives contractor will supply and manufacture bulk explosives on site and deliver them to the holes using a digital controlled ‘Smart’ truck as is common in Western Canadian surface mines.

A 1.5 m subgrade is assumed to ensure there are minimum high spots between holes on the resultant bench floor. The height of the explosives column is calculated from the explosives density and hole diametre to give the required powder factor. The remainder of the hole is backfilled with drill cuttings or crushed rock.

From the powder factor above, the blasting requirements for the Schaft Creek operations have been evaluated based on the large diametre hole size. The blasting specifications are derived below. It has been assumed that all rock will require drilling and blasting.

Table 23.28 Blasting Assumptions		
Blasting Pattern	**Specifications**	
Spacing	8.5	m
Burden	8.5	m
Hole Size	12¼	inch hole
	311	mm
Explosive In-Hole Density	1.25	g/cc
Average down hole loading	95.1	Kg/m
Bench Height	15	m
Sub-drill	1.5	m
Collar	7	m
Charge per hole	903	kg/hole
Rock SG	2.6	Kg/BCM
Yield per hole	2,818	Tonnes/hole
Powder factor	0.32	kg/tonne

23.1.14.1 *Loading*

The design basis assumes three shovels as an optimum fleet size to ensure minimum risk to availability along with minimum capital equipment. Two 100-tonne dipper class electric cable shovels have been selected as the primary digging units plus one rubber tired Front End Loader as a third shovel also specified to rehandle stockpiled material and as a pit clean up, snow removal, or as an alternate to load trucks in the pit if periodic low shovel availability requires it. The use of the FEL should be minimized due to its higher hourly operating cost and longer truck loading times.

The loading units will also be fitted with a GPS based digging monitor which will enable digital dig boundaries from the Ore Control system to define the ore types and waste on the shovel operators graphics screen in the cab. This will be definite requirement since grade control stakes will not be visible during high snow fall periods.

23.1.14.2 *Hauling*

Ore and waste haulage will be handled by large off highway haul trucks with a 345 tonne payload. Haulage profiles have been estimated from pit centroids at each bench to designated dumping points for each time period. These haul profiles are inputs to the truck simulation program and the resulting cycle times are used in MineSight© schedule optimization routine (MS-SP) which is set to maximize project NPV by using the shortest haul to a feasible destination. The payload, loading time and haul cycle then determines the truck productivity. In the production schedule, the truck productivities range from 300 tph to 2400 tph. Depending on the loading equipment and haul distance.

A GPS based Fleet Management/Dispatch system is specified for the trucks, shovels, and the ancillary equipment fleets to ensure coordination and proper management of the fleet over multiple pits in a large mining area. This will be particularly important with the high snow fall at the site. State-of–the-art wireless communication and location systems for management and potential navigation assistance should be considered during the detailed planning and specifications for the project. The capacities, and capabilities of these systems have improved greatly in the last few years and the costs are decreasing.

23.1.14.3 *Pit Services*

Pit services including haul road maintenance, dewatering, transporting operating supplies, and the snow removal crew will be directed by the General Foreman. Manpower and equipment costs are included for these activities. The snow fleet will be manned by mine operations staff in normal winter conditions with operators taken from reduced activities such as dust control and summer field programs. During severe storms additional crew to man the snow fleet will be drawn from truck and shovel operations as the fleets shutdown. As required, some Mine Maintenance personnel from the shovel crew etc. may be required to man additional snow fleet equipment.

23.1.15 Mine Maintenance

Mine maintenance activities will be directed under the Mine Mechanical Supervisor who will assume overall responsibility for mine maintenance and will report to the Mine Superintendent. (In an alternate organization this position may be filled at a Superintendent level reporting to the General Manager.) Maintenance planners will coordinate planned maintenance schedules. The daily maintenance shift coordination will be carried out by Mine Maintenance Foremen and Electrical Foremen.

The Mine Maintenance department will perform break-down and field maintenance and repairs, regular PM maintenance, component change-outs, and in-field fuel and lube servicing, and tire change-outs. Major component rebuilds are done by specialty shops off-site and are costed as sustainable capital repairs.

23.1.16 GME and Technical

Mine GME and technical departments will include Mine operation supervision down to the Foreman level, training, mine engineering and geology.

A mine superintendent will assume responsibility for overall supervision of the mining operation. A general mine foreman will be responsible for overall open pit supervision and equipment coordination. Supervision will also be required for drilling and blasting, training, and dewatering. A mine shift foreman is required on each 12-hour shift, with overall responsibility for the shift operation. Security/First-aid staff and mine clerks will also report to the mine superintendent.

Initial training and equipment operation will be provided by experienced operators. As performance reaches adequate levels, the number of experienced operators can be decreased to an optimal level.

A chief mine engineer will direct the mine engineering department. The mine engineer will coordinate the senior and junior engineers, the mine planning group, geotechnical monitoring, and the surveyors. A senior surveyor will assume responsibility for surveying the entire property and will supervise surveyors. Surveying will use GPS based systems. The Geotechnical Engineer will assume responsibility for all mine geotechnical issues including pit slope stability, hydrogeological studies, and tailings dam construction quality control.

The Geology department will include a Senior Geologist, Junior Geologists and Ore Grade Technicians. This department will be responsible for local step out and infill drill programs for on-site exploration activities and updating the long range mine ore body models. The Geology department will also provide grade control support to mine operations, managing and executing the blast hole sampling and blast hole Kriging of the short range blasthole models for operations planning and ore grade definition.

23.1.17 Mine Fleet Details

The mining equipment descriptions below are intended to provide general specifications so that, dimensions and capacities can be determined from the manufactures specification documents. The list includes specific make and brands; however, does not mean that final equipment selections have been made. The make and model listed is meant to be representative of equipment of that size class. Final brand selection will be subject to a commercial process.

23.1.17.1 *Major Mining Equipment*

A summary of the major mining equipment fleet schedule until is tabled below.

Table 23.29
Major Mining Equipment Schedule

	PP	Y1	Y2	Y3	Y4	Y5	Y10	Y20	Y31
Drilling									
Primary Drill - P&H 120A - Electric Drill	0	3	3	3	3	3	3	3	0
Secondary Drill - P&H 250XP Diesel Drill	1	1	1	1	1	1	1	1	0
Highwall Drill - Sandvik D245S - 150 mm Diesel Drill- Highwall	1	1	1	1	1	1	1	1	0
Loading									
P&H4100XPC Cable Shovel - 104 tonnes	1	2	2	2	2	2	2	2	0
Le Tourneau L-2350 Wheel Loader - 73 tonnes	0	0	0	0	1	1	1	1	1
Cat 3516 GENSET	2	2	2	2	2	2	2	2	0
Hauling									
Cat 797B Haul Truck - 345 tonnes	3	11	12	15	15	15	13	13	3

23.1.17.2 *Drilling*

Drilling will be carried out with electric (primary drill) and diesel (secondary drill) 12¼ " drills. The cost data used in this study is based on the P&H 120A drill and a P&H 250 XP drill.

The result is one diesel drill required at start up and three electric drills required in Year 1.

23.1.17.3 *Blasting*

The blasting activities described above will require an on-site storage and explosives manufacturing plant as well as a management and maintenance facility for the explosives contractor. The location of the plant and the magazines are determined by the table of distance as specified in the Canada's Explosives Act. The Schaft Creek operation will own and build the facilities for use by the contractor. This will include serviced building and power and communication links. The facilities as specified by the explosives contractor are listed below. The contractor will also provide specialty equipment such as the computer controlled bulk loading trucks and any other site specific equipment listed in the table below. The capital and operating costs of these facilities and equipment is part of the contractor's unit rate for supply of the explosives.

23.1.17.4 *Loading and Hauling*

The truck shovel fleet selected for Schaft Creek is the 345 tonne truck (Cat 797 or Equivalent) and 105 tonne dipper shovel (P&H 4100 XPC or equivalent).

The wheeled loader matched capable of reaching the haulers is specified. A wheeled loader will be required as back up for production loaders that are mechanically unavailable, as well as for loading material from ore stockpiles.

The surge in demand for large mining truck tires over the past four years has led to a tire supply shortage. Manufacturers indicate that this shortage has been sustained for the last two years and that new tire production due to come online in 2007 will not alleviate the shortage. The tire shortage applies to all large mine haul trucks and therefore remains a project risk. Tire supply commitments from tire suppliers and manufacturers for all tire types are imperative to the success of this project. Tire suppliers currently recommend that Mines enter a fixed contract to ensure site demand is met. Suppliers from Europe and Eastern Europe are now entering the supply stream and the quality of these new tires is improving. It is assumed that tire issues will be resolved during the life of this project.

23.1.17.5 *De-Watering*

It is important to control the water that is in the active mining areas. In-pit water generally increases the cost of mining and flooded box cuts need to be drained. Rock cuts in tires increase in wet conditions and the presence of water in the shovel digging area can greatly decrease the average tire life of the trucks. Rocks can easily be hidden in puddles that the haul trucks have to drive through and this can lead to instantaneous tire failure. Wet muck that the shovel is digging can easily get frozen to the sides of the truck boxes in the wintertime and this “carry back” results in less material being hauled per truck load (i.e. lower productivities). Water also affects the stability of walls and dumps.

Horizontal drain holes must be established in the final walls as they are exposed. The design and amount required should be determined by geotechnical consultants. On the active bench floor, the water that is collected from the horizontal drain holes will be directed to the sump where it can be removed from the pit. Ditches will be put into the berms to collect the water and direct it to a berm sump in an area where the berm is sufficiently wide. These ditches must be lined and kept clean to avoid water seeping back into the wall. The water from the sump on one berm can be drained down to the next berm below and collect into another berm sump. Actual operating conditions and detailed engineering will determine how many berm sumps can be collected together before putting a pump in place to remove the collected water. It is estimated that as the mining progresses down, the horizontal drains in the wall higher up will be mostly dry except during seasonal times when the water in the wall is recharged by rainfall or snowfall.

Sloped bench floors will also aid in keeping the digging face dry. A gradient of 1% is usually sufficient to collect the water to one area where sump pumps can then be used to pump the water out of the pit. The direction of the slope will have to be determined individually for each pit but generally the floors should slope downwards to the initial starting point of each bench.

That way as the shovels dig outwards and away from this starting point, the water will drain back away from the shovel digging area. A sump can be dug into the bench floor to collect this water and a pump put in to remove the water from the pit. The sloping of the floors will also cause the berms to be sloped and the ditches that are established in the berms will naturally drain to one side of the pit. This side of the pit is where the berm sumps should be established.

In-situ water also reduces drilling productivity and creates many problems in the blasting operations. Large amounts of water can lead to holes caving and under blasting due to incomplete detonation. It can also lead to hard digging, as some holes are unable to be loaded because they don't stay open. This leads to higher maintenance costs on the shovel and lower productivity because of poor digging conditions. Vertical wells should be drilled in advance of pit development and water pumped from these to remove the in-situ water. This will also have to be done on a bench scale, especially as the pits get deeper and more and more water is naturally being collected into the pit. When the amount of water in the digging area gets to be a problem, de-watering wells should start to be established in advance of shovel digging. A hole is drilled into blasted material and then lined with a casing. An instrument is run down the hole that punctures the casing and allows the water to flow into the hole. A pump can then be put at the bottom of the hole to pump out all the water that is collected. The design and amount of vertical wells required should be determined by geotechnical consultants

All surface water and precipitation in the pit will be handled by submersible sump pumps installed in each active pit bottom as part of the flexible and moveable bench scale pumping system. The sump pump will be connected to semi-permanent and permanent piping systems to convey the sump water out of the pits. The sump will be installed with each box cut as the benching is advanced. With the high amount of precipitation it is assumed that the box cuts will have to be made wide enough to facilitate the sump pump and piping as required, as the face advances and until a bench sump can be established on each new bench. The excavation of the sumps is therefore included in the direct mining costs but the pump handling and piping is included in mining support costs.

A plan for the location of the pit water discharge should be included in future detailed studies.

23.1.17.6 *Mine Operations Support*

The following is a summary of the support equipment requirements at Schaft Creek. The summary will describe the fleet estimated and the required tasks of that fleet.

There are a number of pieces described below that are identical to equipment chosen for the fly-in/contractor fleet of equipment. That equipment is considered completely separate from the support fleets described below, but there are opportunities for synergy that have a potential to save money in upfront capital expenditure that should be examined in the detailed design phase of the project.

The following equipment is chosen specifically for support of the mining operations. All equipment is chosen to start operations in the pre-production phase. The equipment is replaced as required and costs for this equipment are applied according to the details included in the cost model.

Dozers

A fleet of two 634 kW dozers (costs and productivities based on CAT D11R), and a fleet of three 433 kW dozers (costs and productivities based on CAT D10T) are chosen to support the mining operations.

The dozers are chosen primarily to support the waste dumps on the minesite and to prepare the initial benches for each pushback including drill ramps. It is not likely that more than two dump areas are utilized at any one time.

The dump dozers maintain the operating surface of the dump and assist in ‘spotting’ the trucks as they back into the dump crest to dump their load. Dozing is required to level any material free dumped on the top surface of the dump to fill slumped or settled areas and to maintain good driving conditions for the trucks.

The section on production scheduling shows that the average waste production in one year is approximately 40,000 ktonnes. Assuming a 360 days/year operation, and 22 hours/day, this breaks down to approximately 5,050 tonnes/hour. The figures below show the productivity of D10 and D11 dozers on the waste dumps using Caterpillar’s DOZ-Sim estimation program.

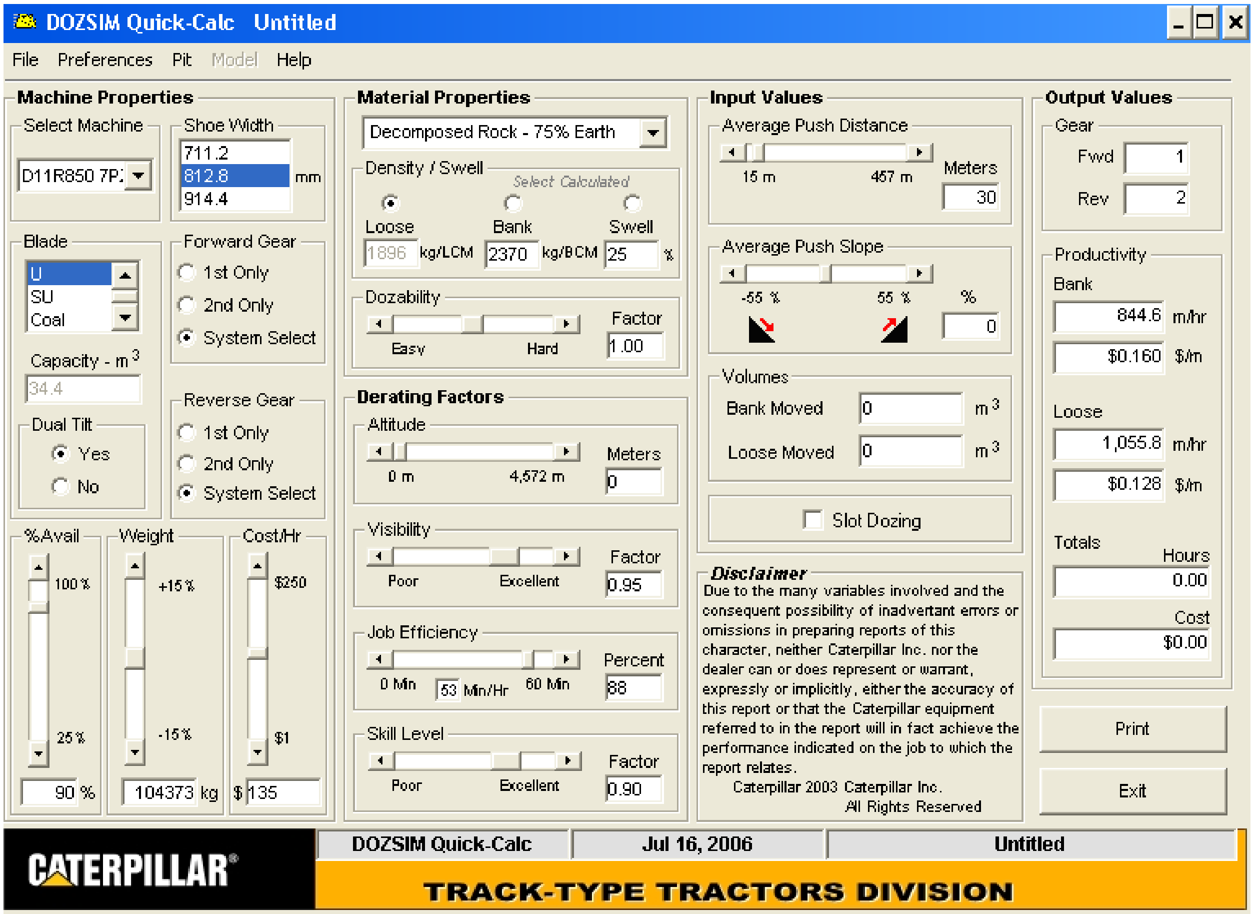


Figure 23.26 Dozsim D11 Productivity on waste dump simulation

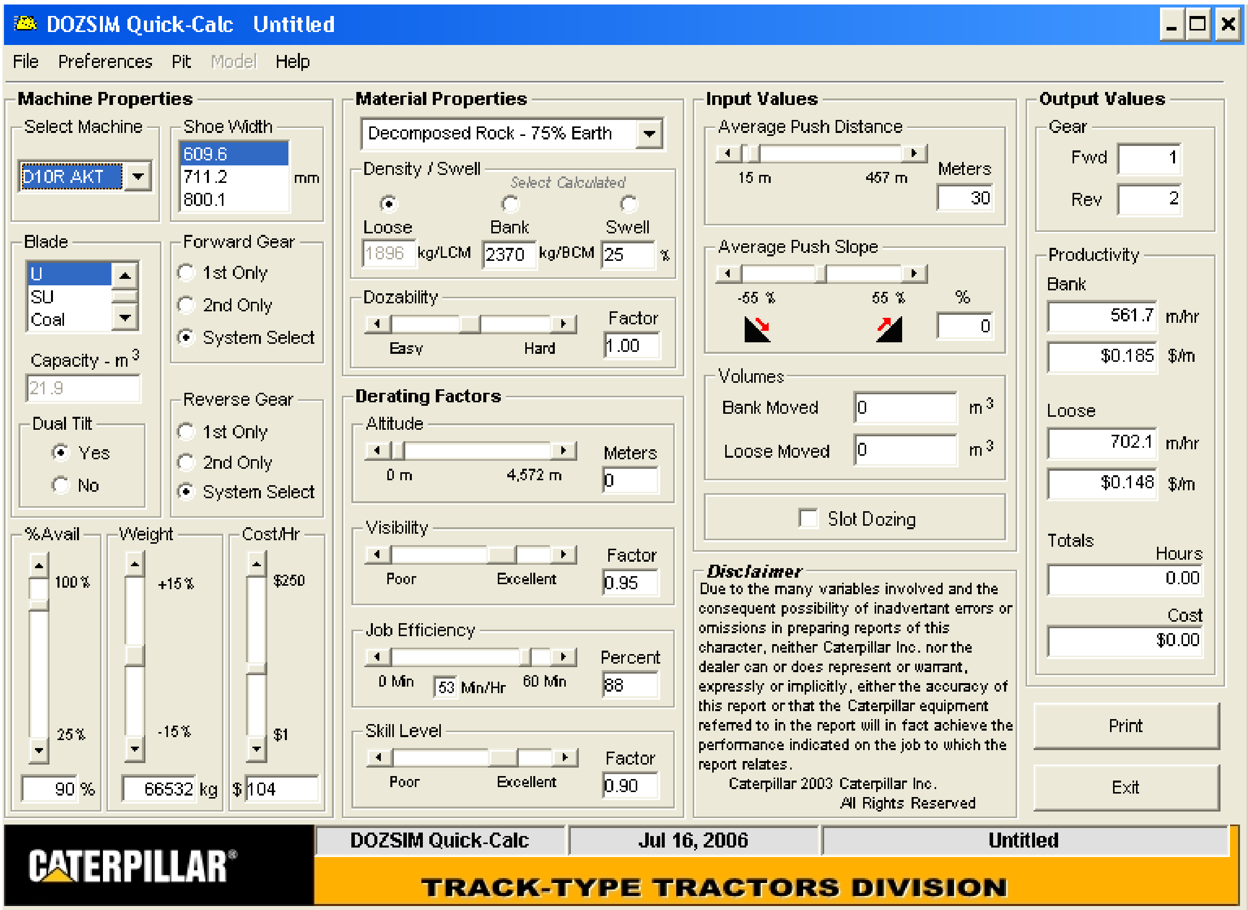


Figure 23.27 D10 Dozsim Productivity on waste dump simulation

Both scenarios include a flat 30m push for the dozers on the dumps. The D11 simulation shows that the larger dozer's productivity is 845 m^3/hour or 2,000 tonnes/hour (at a bank density of 2.37 tonnes/m^3). The D10 Simulation shows that the smaller dozer's productivity is 562 m^3/hour or 1,330 tonnes/hour (at a bank density of 2.37 tonnes/m3). These estimations show that the three larger dozers alone are able to handle 6,000 tonnes/hour of the approximately 5,050 tonnes/hour required from the production schedule at the waste dumps.

The dozers are used in some occasions to trap load to the shovels. From a bench above the shovel's working bench, the dozers push material to the shovels. Under certain circumstances, this facilitates a higher productivity for the shovels.

The dozers are used to push out haul routes for the haul trucks and other equipment within the mining pits, and to assist the graders, described below, to maintain those haul routes.

The dozers rip material that requires ripping before digging or dozing. The productivity and required hours of ripping are not estimated, and it is assumed that the fleet chosen has the ripping hours available for what is required on site.

Intermittently, the dozers assist in other odd jobs in the mining pits, and at the waste dumps; such as towing vehicles, cutting ditches, cleaning shovel work areas, etc.

Motor Graders

A fleet of three 7.32m (CAT 24H) blade-width motor graders are included in the mine operations support fleet. The graders are used to maintain the haul routes for the haul trucks and other equipment within the mining pits, and on all routes to the ore stockpile and waste dumps. The graders ensure the routes are free of debris, and that they conform to the design parametres of the routes for cross-section and grade.

The graders are also used occasionally to level benches and waste dumps that have been excavated or dumped off design targets. With advanced mining support systems described in the occurrence of these situations are minimized.

Other Equipment

The following is a list of equipment chosen and a short description of the tasks that the equipment performs in support of the mining operations:

- Three 463 kW rubber tired dozers are included in the fleet. Three Caterpillar 844H rubber tired dozers are added to the cost model. The rubber tired dozers are used to clean shovel faces and assist in clearing haul routes of large debris. They work primarily at the shovel load faces to keep the floor clean of falling debris. It is important, when considering the longevity of tire life for the haulers, that the possibility of tire cuts is minimized. Most tire cuts occur at the shovel face from falling debris from the face itself, or from material spilled out of overloaded trucks and off of the shovel buckets. Keeping a dozer at each shovel face minimizes the occurrence this issue (maximum of three shovel faces at one time with one hydraulic shovel and two cable shovels in the fleet; so for all faces this fleet requires a third rubber tire dozer, or will be supplemented with a track dozer). Tire cuts are also prevalent for the haulers travelling at high speeds along the haul routes and running over large debris that has fallen off of other haulers. Where required, the rubber tired dozers assist the motorgraders to sweep the haul roads of this debris.
- A hydraulic excavator with the ability to pass approximately 12-13 tonnes per bucket is included in the fleet. A Caterpillar 385C hydraulic excavator is added to the cost model. This equipment digs ditches along the haul routes for dewatering of the routes as described in the section on haul route design; helps to construct small earth structures and ramps within the pits and mine operations areas; and assists with the excavation of sumps and other small excavations where required.
- A hydraulic excavator with the capability to pass approximately 5-7 tonnes per bucket is included in the fleet. A Caterpillar 345C hydraulic excavator is added to the cost model. This equipment is outfitted with a vibratory hammer, and is used to break up oversized material that is handled by the shovel and haul trucks, but cannot be handled by the ore crusher or waste screener. The

excavator is also outfitted with a bucket to dig material for the waste crusher, as well as maintain the water diversion channels and structures.

- A wheeled loader with the capability to pass approximately 14 tonnes per bucket is included in the fleet. A Caterpillar 988H wheeled loader is added to the cost model, as well as various work tools that can be quick-coupled to the machine. The wheeled loader is outfitted with a bucket to assist in earth works where required. It is outfitted with a cable-reeler to move shovel cable in long distance shovel moves. The shovels are outfitted with on-board cable-reelers, but require support for long distance moves. The wheel loader is outfitted with fork tines to support movement of supplies and small components required for both operations and maintenance. The wheeled loader is outfitted with a brush for cleaning work areas at the truck shop and the offices.
- A wheeled loader outfitted with a tire manipulator able to manipulate 59/80R63 tires is included in the fleet. A Caterpillar 988H wheeled loader with an IMAC CR450 tire manipulator is added to the cost model. This tire manipulator is dedicated to moving tires wherever they are needed on site, as well as changing tires on the large rigid frame haul trucks.
- Two articulated trucks outfitted with fuel/lube arrangements are included in the fleet. Two Caterpillar 740 articulated haulers outfitted with Ground Force Fuel/Lube arrangements are added to the cost model. The fuel/lube trucks are used to provide lubrication maintenance to the mining equipment while in the mining pit and other working areas on site. The articulated trucks are chosen for navigation into working areas that may not be possible with standard flatbed semi-trucks. The size of the trucks is dictated by the fuel/lube arrangement that is included to support the large hydraulic and cable shovels, large haul trucks, large track and wheeled dozers and large motorgraders.
- Two Rigid Frame Trucks outfitted with 48,000 Gallon Water bodies are included in the fleet. Two Caterpillar 789C haulers outfitted with MEGA MTT48 water bodies are added to the cost model. The water trucks spray the width of the haul roads with a sheet of water in order to minimize the airborne dust that is created by the passage of equipment along the dirt and gravel based roads. The airborne dust may create both visibility (productivity) and environmental issues that are mitigated by the use of the water trucks. The size of the water bodies are chosen to correspond to the width of the roads, and the distance of the road to the tailings dam.
- A hole-stemmer with the ability to lift approximately 3 tonnes of material is included in the fleet. A Caterpillar IT28G integrated tool carrier is added to the cost model. After the blast holes have been loaded with blast material and charged, the hole-stemmer takes drill cutting from the drilling of the hole and fills the top portion of the blast holes with a capping. The capping contains the blast to within the hole and outward into the ground, rather than having the blasted energy escape out of the top of the hole. It is estimated that one tool carrier is able to keep up with the fleet of drills.
- Twenty ½ ton pickup trucks are included in the fleet. Twenty Ford F-150 pickup trucks are added to the cost model. These pickup trucks are used for transportation of mine maintenance, technical and managerial personal around the mine site. Seven units are used for maintenance, five units for

management, one unit for surveying, one unit for environmental, three units for geotechnical and three units for engineering.

- Four 1 ton pickup trucks are included in the fleet. Four Ford F-550 pickup trucks are added to the cost model. These pickup trucks are used to transport small good around the mine site, primarily for maintenance items, but also for other miscellaneous goods.
- Three maintenance trucks are included in the fleet. Three Kenworth T300 trucks outfitted with service units are added to the cost model. These units include the tooling required for maintenance personnel to perform service of other equipment in the field, and will be used by maintenance personnel to perform service of other equipment in the field.
- A forklift with a 30 tonne capacity and a forklift with a 10 tonne capacity are included in the fleet. A Hyster 620F forklift and a Hyster 210HD forklift are added to the cost model. The forklifts are used in the warehouse and around the maintenance shop to assist in the transportation of machine components and other goods. Many machine components exceed the 10 tonne weight limit of the smaller forklift, thus the need for the larger one. Most stocked items are under the 10 tonne limit of the smaller forklift.
- Two trucks outfitted with picker arms are included in the fleet. Two Kenworth C500 picker trucks are added to the cost model. The picker trucks are used by maintenance to lift components into equipment in the field. They are also used on occasion to lay small pipeline and transport heavier goods into the field that require lifting to larger heights than can be achieved by the wheeled loader.
- One crew bus and two crew vans are included in the fleet. One GMC Guide XL crew bus and two Chevrolet G3500 vans are added to the cost model. The bus and vans are used to transport personnel coming on shift to the working areas, and personnel coming off shift out of the working areas. The bus transports most operators to and from the working areas, and the vans are utilized for inter-shift runs, as well as for the requirements of maintenance and staff for transporting groups into the field.
- One ambulance, one fire truck and one mine rescue truck are included in the fleet and added to the cost model. These three units are used to maintain the safety of personnel and equipment working on site.
- One tractor and flatbed trailer is included in the fleet from period 3, according the production schedule, to the end of mine life. One Caterpillar 789C tractor and one MEGA 240-ton trailer are included in the cost model. This unit is utilized for transporting tracked equipment throughout the various mining pits and working areas. Whenever possible, all tracked equipment long distance movement is accomplished with the trailer.
- One screening plant with the ability to produce ¾” and 3” material is included in the fleet. One Fintec 570 Screening plant is added to the cost model. The screening plant will be fed by the hydraulic excavator described above.

The mine operations support fleet size is listed in the table below.

Table 23.30
Mine Operations Support Fleet

Mine Operations Support Fleet	Fleet Size
Cat 385C Hydraulic Excavator - 12-13 tonne	1
Cat 24H Grader - 7.32 m Blade	3
Cat D11R - 634 kW	2
Cat 988H Multipurpose - Cable Reeler, Forks, Brushes, Bucket	1
Cat 345 Hydraulic Excavator - 5-7 tonnes	1
Cat 844H Rubber Tired Dozer - 463 kW	3
CAT IT28G Hole Stemmer - 3 tonnes	1
Cat 988H Tire Manipulator	1
Cat 789 C Water Truck - 48,000 Gallons	2
Cat 740 Fuel/Lube Truck	2
Kenworth T800 FireTruck	1
Kenworth C500 Picker Truck	2
Chevrolet G3500 Passenger Vans	2
Ambulance	1
Hyster 620F Forklift - 30 tonne	1
Hyster 210HD Forklift - 10 tonne	1
Ford F550 Maintenance Truck - 1 Tonne	4
Kenworth T300 Service Truck	3
Ford 1/2 Ton Pickups – F150	20
Cat D10T - 433kW	3
Cat 789 Float Tractor/Trailer - 240 tonnes	1
Mine Rescue Truck	1
GMC Guide XL Crew Bus	1
Fintec 570 Screening Plant - 12" max	1

23.1.17.7 *Mine Maintenance Support*

The following equipment is chosen specifically for support of the duties of the mine maintenance department.

- One crane with the ability to lift 250-tonne components at least 20m in height and one crane with the ability to lift 100-tonne components at least 20m in height are included in fleet. One Liebherr LTM1250-6.1 crane and one Liebherr LTM1100 are added to the cost model. The cranes are required to lift equipment components in the operation of erecting the equipment in the field, as well as maintaining the equipment in the field. The cranes are also used to lift the equipment itself in order to block it off for maintenance work on ground based components such as tires and tracks.

- One welding truck is included in the fleet. One Kenworth T300 Truck outfitted with a welding unit is added to the cost model. The welding truck is used in support of maintenance personnel's needs for welding equipment, and equipment support items such as truck bodies, dozer blades, shovel buckets, etc.

- Two powerline trucks with a man-lift basket are included in the fleet. The powerline truck is required for the safe movement of all powerlines on site.

- Two 110 kW water sump pumps are included in the fleet at startup. The pumps are ramped up to seven by Year6 Tsurumi LH8110-60 submersible pumps are added to the cost model, the multiples of which are described above. The pumps are required for keeping the mining pits as dry as possible, and to avoid interruption to the production of the mining fleet.

The pit maintenance fleet size is listed in the table below.

Table 23.31
Pit Maintenance Fleet in Year 5

Pit Maintenance Fleet	Fleet Size
LTM1250-6.1 – 250-tonne crane	1
Kenworth T300 Welding Truck	1
PowerLine Truck	2
LTM1100 – 100-tonne crane	1
Tsurumi LH8110-60 Water Pump - 1,400 Gal/min	7

23.1.17.8 *Snow Fleet Support*

The following equipment is chosen specifically for support of the duties of the snow fleet. Because of the potentially large amounts of snowfall on site, a dedicated snow fleet has been specified for this operation. All equipment is chosen to start operation during pre-production and continue to the end of mine life, unless otherwise noted. The equipment is replaced as required and the costs for this equipment are applied according to the details included in the cost model.

- Five scrapers with the ability to haul 37 tonnes are included in fleet. The Caterpillar 637G scrapers are added to the cost model. The scrapers are required to haul crushed rock material along the roads after heavy snowfall activities. The scrapers also remove large amounts of snow from the haul roads and mine working areas as necessary. The scrapers are also used on occasion (less than 5% of the time) for small earthmoving jobs. They may also be used for reclamation projects.

- A wheeled loader with the capability to pass approximately 14 tonnes per bucket is included in the fleet. A Caterpillar 988H wheel loader is added to the cost model. The wheeled loader is utilized during snowfall periods to clear snow from the plant area and truck shop, as well as ancillary routes within the mine. The wheeled loader is also used to load the cone crusher described below.

- A motorgrader with a 4.88 m wide blade is included in the fleet. One Caterpillar 16H motorgrader with various blade types to handle snowfall is added to the cost model. This motorgrader is utilized to clear haul routes of snow where the mining support fleet of graders cannot. It is used as backup during snow fall periods, as well as in areas too tight for the larger graders.

- A cone crusher with the ability to produce 6” minus rock is included in the fleet. One Fintec 1080 cone crusher is added to the cost model. The cone crusher will produce crush material where it is not available in the natural ground to spread onto haul roads by the scrapers during periods of heavy snowfall. The crusher will be fed by the wheel loader described above.

- Two snowcats with the ability to transport 5 passengers at a time. Two LMC 1500 snowcats are added to the cost model. These snowcats are used to transport operators to equipment that is in a location that is inaccessible to the crew bus or vans because of heavy snowfall.

- The Snow Fleet has a low utilization as it is only required in wintertime. Other than the use of the crusher to produce road gravel, operating this equipment outside of wintertime is optional and not necessary. The size of the Snow Fleet is listed in the table below.

Table 23.32 Snow Removal Fleet	
Snow Removal Fleet	**Fleet Size**
Cat 637G Scraper - 37 tonnes	5
Cat 988H Wheel Loader - 14 tonnes	1
Cat 16H Grader - 4.88m blade	1
Fintec 1080 cone crusher - 7.5" max.	1
LMC 1500 Snowcat	2

23.1.17.9 *Mine Fuel Consumption*

Fuel consumption rates are estimated in the mine schedule for each equipment type. These consumption rates are applied to the operating hours of the equipment to estimate the total fuel consumption.

Explosive factory fuel consumption is estimated based on the quantity of explosives used, and an estimated 40 litres diesel fuel consumed per tonne of explosives.

Fuel quantities scheduled for until first 5 years of milling are shown in the table below.

Table 23.33 Mine Fuel Consumption Schedule						
FUEL CONSUMPTION (m^3)	**PP**	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**
DRILLING	668	524	455	76	42	504
BLASTING (Explosives Factory)	316	1,118	1,090	933	919	1,090
LOADING	-	515	489	180	336	807
HAULING	4,284	15,545	17,157	22,177	21,950	23,783
MINE MAINTENANCE	165	165	165	165	165	176
MINE OPERATIONS - SUPPORT	6,305	6,270	6,270	6,270	6,257	6,693
SNOW REMOVAL	568	568	568	568	568	600
Total	**12,306**	**24,705**	**26,194**	**30,369**	**30,236**	**33,652**

23.2 Process Summary

23.2.1 Description

The Copper Fox Metals, Inc. Schaft Creek concentrator will have an annual throughput of 23,400,000 tonnes per year. Copper Fox will construct the concentrator on site which will include a comminution circuit followed by a flotation circuit and a copper circuit with thickener, filtration and concentrate loadout and shipping. The mill includes a dedicated molybdenum circuit with thickener, filtration circuit, drying and bagging. Tailings thickeners, tailings facility and water reclaim are part of the tailings facilities. This circuit will have a design capacity of 70,652 tonnes per day and a nominal capacity of 65,000 tonnes per day.

23.2.1.1 *Crushing*

Run-of-Mine ore will be dumped from the 345 tonne haul trucks out of the pit into a 720 tonne live capacity feed pocket which feeds a 1524 by 2870 millimetre (60 by 113 inch) primary gyratory crusher.

23.2.1.2 *Coarse Ore Stockpile*

The crushed ore will be conveyed by the crushed ore stacking belt conveyor to a covered coarse ore surge pile, which has a minimum live capacity of 65,000 tonnes and a total capacity of 500,000 tonnes.

23.2.1.3 *Grinding*

Ore from the stockpile is reclaimed onto the semi-autogenous grinding (SAG) mill feed belt conveyor by four of five stockpile reclaim belt feeders below the stockpile and discharged into the SAG mill feed belt discharge chute. A single grinding line comprised of one 12.2 metre (m) diameter by 7.31 m long SAG mill (40 ft x 24 ft, 23 MW) and three 7.16 m diametre by 12.04 m long ball mills (24 ft x 39.5 ft, 13.5 MW each) will provide the grinding capacity for the comminution circuit. The SAG mill product discharges through the mill trommel. The trommel undersize will flow through the SAG mill discharge launder with a chute magnet to each of three ball mill cyclone feed sumps along with the ball mill discharge that correlates with the specific sump. Feed to each of the ball mill cyclone clusters will be provided by one of two ball mill cyclone feed pumps (one in standby mode). The three ball mill cyclone clusters will be comprised of nine-26 inch cyclones (two spares) per cluster and will operate in closed circuit with the ball mill. The cyclone overflow, at P_{80} product size of 100 micrometres (µm), will be fed to the flotation circuit. The cyclone underflow returns by gravity to the ball mill.

23.2.1.4 *Pebble Crushing*

Critical size pebbles will be screened out by the SAG mill trommel and the SAG mill pebble wash screen and transferred to the Pebble Crusher surge bin via the SAG mill pebble collection belt conveyor. Pebbles are fed from the bin through the two surge bin discharge chutes onto the two pebble crusher belt feeders which transfer the pebbles to the two short head MP1000 (750 kW) pebble crushers.

Crushed product from the pebble crushers is transferred back to the SAG mill feed belt conveyor by way of the pebble crusher transfer belt conveyor from the pebble crusher belt conveyor.

23.2.1.5 *Flotation*

The cyclone overflow from the grinding lines will combine for redistribution to the three flotation lines at the rougher flotation feed distributor. Feed from the distributor will report to each of three Rougher Flotation Cell banks (“A”, “B”, and “C”). Each Rougher Flotation Cell bank will consist of four 300 m^3 tank cells which will provide 24 minutes residence time. The total rougher concentrate from processing 65,000 tonnes per day of ore will be transported through the rougher concentrate launder to the regrind cyclone feed pump box. Tailings from the three rougher flotation circuits will combine in the rougher tailings collection launder and then onto the scavenger circuit feed pump sump. A scavenger feed pump will deliver slurry to the Scavenger Flotation Feed Distributor. This distributor will feed each of three scavenger flotation cells consisting of six 300 m^3 tank cells per line. Concentrate from the scavenger circuit will report to the rougher flotation feed distributor along with the fresh feed from the grinding circuit. Tails from the scavenger circuit will report to the tailings thickener.

Slurry at F_{80} of 100 µm from the concentrate regrind cyclone feed pump box will be pumped via four slurry pumps to four separate regrind cyclone clusters (cyclone banks ‘A’, ‘B’, ‘C’, or ‘D’).

The 1st cleaner flotation circuit will receive the regrind cyclone overflow at a P_{80} of 15 µm. The 1st cleaner flotation circuit consists of a feed distribution box and two 7 by 12 m column type float cells having a volume (with froth factor) of 282 m^3 each. The 1st cleaner tails are transferred to the cleaner/scavenger circuit consisting of two 200 m^3 tank cells, arranged in two lines of one cell each, to reclaim concentrate which is transferred via the recleaner feed pump to the 1st cleaner tailings recleaner circuit. The 1st cleaner tails scavenger cleaner flotation (the 1st cleaner tailings recleaner) will consist of two 1 by 12 m column type scavenger cells. Concentrate from the recleaners will join the concentrate from the 1st cleaners at the 2nd cleaner feed pump box. The tailings from the recleaners will be transferred via the recleaner tailings pump back to the regrind cyclone feed sump (where it will mix with fresh feed and report to the 1st cleaner in the cyclone overflow).

The 2nd cleaner flotation circuit consisting of the 2nd cleaner flotation feed distributor and two 3 by 12 m column type cleaner cells having a volume of 217 m^3 each.

Tails from the 2nd cleaner circuit will be sent back to the 1st cleaner circuit via the 2nd cleaner tailing pump box and pump. Concentrate from the 2nd cleaner will report to the 3rd cleaner flotation circuit via the 3rd cleaner feed pump. The 3rd cleaner flotation feed circuit consists of the third cleaner flotation feed distributor and two 3rd cleaner flotation columns, each at 2 by 12 m having a volume of 108 m^3. Tails from the 3rd cleaner circuit reports to the 3rd cleaner tails pump box and is pumped back to the 2nd cleaner flotation feed distributor by the 3rd cleaner tail recirculating pump. Concentrate from the 3rd cleaner circuit reports to the bulk concentrate thickener.

After thickening in an 11 m diametre thickener from 25% solids to 60% solids, the slurry is pumped via one of two bulk concentrate thickener underflow pumps to the bulk molybdenum rougher flotation circuit. The rougher flotation circuit will consist of three standard (enclosed) 16 m^3 rougher cells. Tails (copper concentrate) from the molybdenum rougher circuit will report to the copper concentrate thickener pump box and be pumped by the copper concentrate thickener feed pump to the copper concentrate thickener. The concentrate from the molybdenum rougher flotation circuit reports to the molybdenum cleaner circuit. The molybdenum cleaner flotation circuit consists of the molybdenum cleaner flotation feed distributor which will direct initial feed to the 1st molybdenum cleaner flotation cell via baffles. The 1st molybdenum cleaner flotation column will be 1.6 by 12 m high with a volume of 25 m^3. 1st molybdenum cleaner flotation concentrate reports to the second (middle) section of the molybdenum cleaner flotation feed distributor via the 2nd molybdenum cleaner feed pump. Tails from the 1st molybdenum cleaner reports to the molybdenum rougher flotation circuit with the thickened concentrate from the bulk concentrate thickener via the 1st molybdenum cleaner tails pump. Feed to the 2nd molybdenum cleaner flotation cell from the center section of the molybdenum cleaner flotation feed distributor will consist predominantly of concentrate from the 1st molybdenum cleaner flotation and tails from the 3rd molybdenum cleaner flotation. The 2nd molybdenum cleaner flotation cell will be a 1.0 by 12.0 m column type cell with a volume of 10.1 m^3. Tails from the 2nd molybdenum cleaner will be pumped to the first section of the molybdenum cleaner flotation feed distributor by the 2nd molybdenum cleaner tails pump. Concentrate from the 2nd molybdenum cleaner will be pumped to the 3rd section of the molybdenum cleaner flotation feed distributor by the 3rd molybdenum cleaner feed pump. Feed from the 3rd section of the molybdenum cleaner flotation feed distributor reports to the 3rd molybdenum cleaner. The 3rd molybdenum cleaner will be 0.6 by 12.0 m column type cell with a volume of 3.4 m^3. Tails from the 3rd molybdenum cleaner will be pumped to the center section of the molybdenum cleaner flotation feed distributor by the 2nd molybdenum cleaner feed pump. Concentrate from the 3rd molybdenum cleaner flotation cell reports to the molybdenum concentrate thickener.

23.2.1.6 *Copper Concentrate Thickening, Filtration, Storage*

Final copper concentrate at 25% solids from the molybdenum rougher flotation tails stream will be thickened to 60% in a 6 m (20 foot) diametre thickener. The thickened copper concentrate will be pumped to the copper concentrate receiving tank where the slurry is mixed with copper concentrate filtrate clarifier underflow. Copper concentrate slurry is pumped by the filter feed pumps to one of two pressure belt type dewatering filters.

Copper concentrate filter cake reports to the covered copper concentrate stockpile via the copper concentrate stacking belt conveyor for loading by a loader to copper concentrate transport trucks.

23.2.1.7 *Molybdenum Concentrate*

Third molybdenum cleaner concentrate reports to a 10 m diametre molybdenum concentrate thickener. Underflow is pumped to a disc type molybdenum concentrate dewatering filter. Filtrate from the molybdenum dewatering filter is returned to the molybdenum concentrate thickener via filtrate receivers.

The cake will be transported by a molybdenum concentrate transfer screw conveyor from the molybdenum filter press to the concentrate dryer and is subsequently discharged through a retractable sack fill chute into bulk bags for shipping.

23.2.1.8 *Reagents*

Lime:

Milk of lime will be used as pH control to maintain pH levels of 10 to 10.5. Slaked lime will be added to the process at various rates to the SAG and Ball mill feeds, the regrind feed.

Flocculant:

Flocculant distribution pumps will deliver the flocculant to the tailings thickeners, the copper concentrate thickener, the molybdenum concentrate thickener and the bulk concentrate thickener.

Sodium Hydrosulfide (NaSH):

A NaSH transfer pump will deliver the NaSH to the Cu precipitation tank, the 1st molybdenum float cell, the 2nd molybdenum float cell, the 3rd molybdenum float cell, and the molybdenum rougher float bank.

Primary and Secondary Collectors:

Primary collector (fuel oil or equivalent) will be delivered by trucks and unloaded to a storage tank via an unloading pump. Two feed pumps (1 operating, 1 standby) will feed the ball mill as well as the second and third molybdenum cleaner flotation columns.

A transfer pump will deliver the secondary collector (SIPX) to the storage tank. Two distribution pumps (1 operating, 1 standby) will feed a head tank. A control valve system will feed reagent to the rougher flotation cells, the first and second cleaner flotation feed distribution boxes, and the copper cleaner/scavenger flotation cells.

Frother:

Frother (MIBC) will be delivered by trucks and unloaded to a storage tank via an unloading pump. Two transfer pumps (1 operating, 1 standby) will feed two frother head tanks. The frother will then be fed at controlled rates to the rougher flotation cells from one head tank and to the copper cleaner flotation circuit feed from the other head tank.

23.2.2 Block Process Flow Diagram

A simplified block process flow diagram showing the facilities to which these design criteria relate are presented in Figure 23.28. Complete Process Flow Diagrams were developed for the project as well as mechanical and electrical equipment lists, load analysis and single line diagrams.

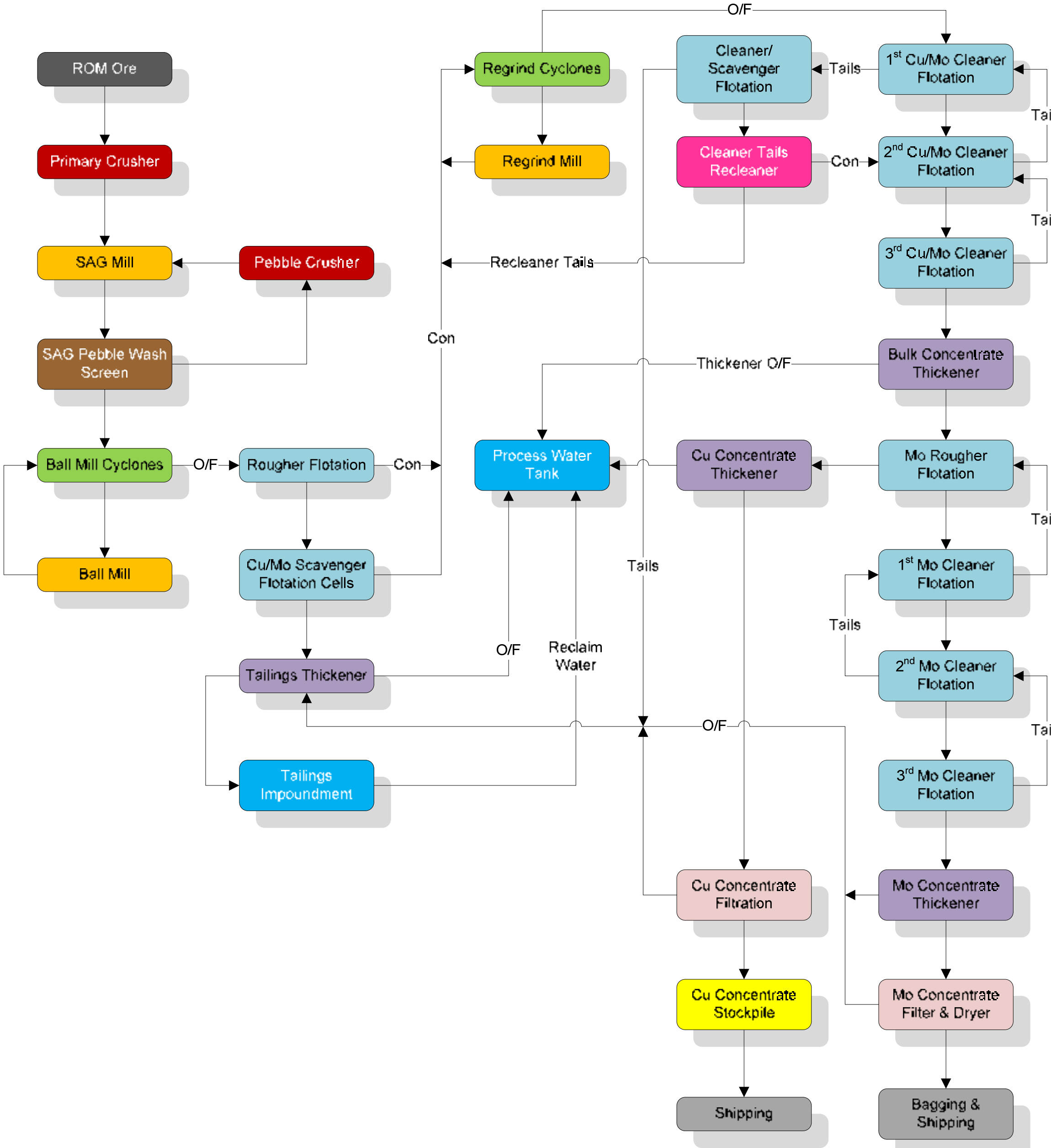


Figure 23.28 Block Process Flow Diagram

23.2.3 Basic Design Criteria

Table 23.34
Schaft Creek Basic Design Criteria

	Units	Balance	Design	Source
General Site Information				
Location				
Latitude - Approximate	angular		N57° 32' 18"	
Longitude - Approximate	angular		W131° 01' 09"	
Elevation				
Air Strip	masl		1,040	SE
Plant	masl		1,230	SE
Pit Bottom	masl		600	MMTS
Ambient Air Temperature				
Average Monthly Minimum	°C		-30	
Average Monthly Maximum	°C		28	
Average Annual Precipitation	mm/y		640	
Average Annual Evaporation - Pond Area	mm/y			
Maximum Wind Velocity	km/h			
General Project Information				
Reported Resource	tonnes (t)		1,393,282,000	AGL
Cutoff Grade Used	CuEq (%)		0.20	AGL
Estimated Mineable Resources				
Starter Pit (5 Year)	tonnes (t)		117,050,000	MMTS
Life of Mine (includes subgrade to waste)	tonnes (t)		719,091,000	MMTS
Operating Schedule				
Hours per Day	h	24	24	MMTS
Days per Year	d	360	360	MMTS
Hours per Year	h	8,640	8,640	MMTS
Plant Capacity (at 92% availability)	dmtpd	65,000	70,652	MMTS
Plant Capacity (at 92% availability)	dmtph	2,944	2,944	
Annual Ore Processed per Year	t	23,400,000	23,400,000	MMTS
Mineable Resource to Mill	t	713,387,500	713,387,500	MMTS
Estimated Project Life @ 65,000 tpd	y		30.5	MMTS
Life of Mine Plant Head Grade Estimates				
Estimated Copper Grade	%		0.303	MMTS
Estimated Molybdenum Grade	%		0.020	MMTS
Estimated Gold Grade	g/t		0.217	MMTS
Estimated Silver Grade	g/t		1.761	MMTS

Table 23.34
Schaft Creek Basic Design Criteria

First 5 Years Plant Head Grade Estimates				
Estimated Copper Grade	%		0.350	MMTS
Estimated Molybdenum Grade	%		0.018	MMTS
Estimated Gold Grade	g/t		0.268	MMTS
Estimated Silver Grade	g/t		1.869	MMTS
Years 6 - 10 Plant Head Grade Estimates				
Estimated Copper Grade	%		0.334	MMTS
Estimated Molybdenum Grade	%		0.017	MMTS
Estimated Gold Grade	g/t		0.203	MMTS
Estimated Silver Grade	g/t		1.637	MMTS
Years 11 - 15 Plant Head Grade Estimates				
Estimated Copper Grade	%		0.283	MMTS
Estimated Molybdenum Grade	%		0.019	MMTS
Estimated Gold Grade	g/t		0.235	MMTS
Estimated Silver Grade	g/t		1.613	MMTS
Years 16 - 31 Plant Head Grade Estimates				
Estimated Copper Grade	%		0.285	MMTS
Estimated Molybdenum Grade	%		0.022	MMTS
Estimated Gold Grade	g/t		0.199	MMTS
Estimated Silver Grade	g/t		1.815	MMTS
Plant Design (First 5 Years) Head Grade Est				
Estimated Copper Grade	%		0.350	MMTS
Estimated Molybdenum Grade	%		0.018	MMTS
Estimated Gold Grade	g/t		0.268	MMTS
Estimated Silver Grade	g/t		1.869	MMTS
Design ROM Ore Dry Solids Sp Gr	g/cc		2.69	CFMI
Design ROM Ore Moisture (for material handling)	%		3	CFMI
First Five-year Average Copper Conc Production				
Copper Recovery to Copper Concentrate	%		90.3	PRA
Copper Grade in Copper Concentrate	%		26.5	PRA
Moly Recovery to Copper Concentrate	%		18.2	PRA
Moly Grade in Copper Concentrate	% Mo		0.27	PRA
Gold Recovery to Copper Concentrate	%		82.0	PRA
Gold Grade in Copper Concentrate	g/t		18.40	PRA
Silver Recovery to Copper Concentrate	%		72.13	PRA
Silver Grade in Copper Concentrate	g/t		113.10	PRA
Copper Concentrate Production	dmtph		35.1	HE

Table 23.34 Schaft Creek Basic Design Criteria				
Copper Concentrate Production	dmtpy		278,987	HE
First Five-year Average Moly Conc Production				
Moly Recovery to Moly Concentrate	%		72.0	PRA
Moly Grade in Moly Concentrate	% Mo		54.0	PRA
Copper Recovery to Moly Concentrate	%		0.028	PRA
Copper Grade in Moly Concentrate	%		0.42	PRA
Rhenium Grade in Moly Concentrate	ppm		530	PRA
Moly Concentrate Production	dmtph		0.70	HE
Moly Concentrate Production	dmtpy		5,596	HE
Year 6 - 10 Average Copper Conc Production				
Copper Recovery to Copper Concentrate	%		90.3	PRA
Copper Grade in Copper Concentrate	%		26.5	PRA
Moly Recovery to Copper Concentrate	%		18.2	PRA
Moly Grade in Copper Concentrate	% Mo		0.27	PRA
Gold Recovery to Copper Concentrate	%		82.0	PRA
Gold Grade in Copper Concentrate	g/t		14.60	PRA
Silver Recovery to Copper Concentrate	%		72.13	PRA
Silver Grade in Copper Concentrate	g/t		103.80	PRA
Copper Concentrate Production	dmtph		33.5	HE
Copper Concentrate Production	dmtpy		266,285	HE
Year 6 - 10 Average Moly Conc Production				
Moly Recovery to Moly Concentrate	%		72.0	PRA
Moly Grade in Moly Concentrate	% Mo		54.0	PRA
Copper Recovery to Moly Concentrate	%		0.028	PRA
Copper Grade in Moly Concentrate	%		0.42	PRA
Rhenium Grade in Moly Concentrate	ppm		530	PRA
Moly Concentrate Production	dmtph		0.67	HE
Moly Concentrate Production	dmtpy		5,286	HE
Year 11 - 15 Average Copper Conc Production				
Copper Recovery to Copper Concentrate	%		83.4	PRA
Copper Grade in Copper Concentrate	%		28.3	PRA
Moly Recovery to Copper Concentrate	%		18.8	PRA
Moly Grade in Copper Concentrate	% Mo		0.43	PRA
Gold Recovery to Copper Concentrate	%		76.1	PRA
Gold Grade in Copper Concentrate	g/t		21.40	PRA
Silver Recovery to Copper Concentrate	%		67.19	PRA

Table 23.34 Schaft Creek Basic Design Criteria				
Silver Grade in Copper Concentrate	g/t		129.70	PRA
Copper Concentrate Production	dmtph		24.6	HE
Copper Concentrate Production	dmtpy		195,493	HE
Year 11 - 15 Average Moly Conc Production				
Moly Recovery to Moly Concentrate	%		74.1	GTMS
Moly Grade in Moly Concentrate	% Mo		54.0	GTMS
Copper Recovery to Moly Concentrate	%		0.026	PRA
Copper Grade in Moly Concentrate	%		0.43	PRA
Rhenium Grade in Moly Concentrate	ppm		530	PRA
Moly Concentrate Production	dmtph		0.77	HE
Moly Concentrate Production	dmtpy		6,105	HE
Year 16 - 31 Average Copper Conc Production				
Copper Recovery to Copper Concentrate	%		83.4	PRA
Copper Grade in Copper Concentrate	%		28.4	PRA
Moly Recovery to Copper Concentrate	%		18.8	PRA
Moly Grade in Copper Concentrate	% Mo		0.49	PRA
Gold Recovery to Copper Concentrate	%		76.1	PRA
Gold Grade in Copper Concentrate	g/t		18.10	PRA
Silver Recovery to Copper Concentrate	%		67.19	PRA
Silver Grade in Copper Concentrate	g/t		145.60	PRA
Copper Concentrate Production	dmtph		24.7	HE
Copper Concentrate Production	dmtpy		195,954	HE
Year 16 - 31 Average Moly Conc Production				
Moly Recovery to Moly Concentrate	%		74.1	PRA
Moly Grade in Moly Concentrate	% Mo		54.0	PRA
Copper Recovery to Moly Concentrate	%		0.026	PRA
Copper Grade in Moly Concentrate	%		0.25	PRA
Rhenium Grade in Moly Concentrate	ppm		530	PRA
Moly Concentrate Production	dmtph		0.89	HE
Moly Concentrate Production	dmtpy		7,066	HE

23.3 Concentrate Transportation

The basis for this study was a filter plant on the Schaft Creek site and trucking of copper concentrate to the port of Stewart. Pipeline Systems Incorporated (PSI) was retained to perform a scoping level study for the investigation of pumping a concentrate slurry from the site to a filter plant located in the Bob Quinn area. In addition, PSI was asked to include a diesel pumping station and supply pipeline to the site. This study resulted in capital and operating costs as shown below.

Table 23.35 Copper Concentrate & Diesel Pipelines	
Capital Cost	
Slurry Pump Station	$17,303,000
Slurry Terminal Station	$1,741,000
Diesel Unloading & Pipeline Pump Station	$635,000
Mine Site Stations – Receiving	$291,000
Pressure Monitoring Stations	$486,000
Pipelines	$56,260,000
Electrical, Telecommunications, SCADA	$4,096,000
Subtotal Capital Cost	$80,812,000
EPCM (12%)	$9,697,000
Contingency (20%)	$18,102,000
Total Capital Cost	**$108,611,000**
Annual Operating Cost	
Labour	$996,971
Power	$182,735
Maintenance Materials	$281,481
Contract Services	$60,000
Misc.	$199,005
G&A	$30,000
Subtotal Operating Cost	$1,750,000
Contingency (15%)	$263,000
Total Operating Cost	$2,013,000
Cost per Tonne Pumped	$7.06

This option needs to be further investigated as the project advances into the next stage.

All concentrates will be trucked to the port terminal at Stewart. Trucking of concentrate from site out to the Bob Quinn area is assumed to be conventional 30 tonne trucks due to access road restrictions. From Bob Quinn to Stewart, conventional B-train style commercial truck haulage can then be utilized, similar in nature to that currently in operation at Eskay Creek, Kemess and Highland Valley mines.

The concentrate haul for the route is largely a highway haul, with the exception of the Schaft Creek access road portion. The round trip time for this route is approximately 10 to 12 hours.

Concentrate that will be marketed overseas will be transported by truck to Stewart, British Columbia. Stewart is BC's most northerly ice free port and is capable of accommodating large ocean going vessels.

Concentrate production is expected to be about 279,000 tonnes per year. Sales contract quantities and the number of buyers and required delivery frequency will determine parcel size. It is likely that sales contracts will be around the 50,000 tonne size. Generally the intent will be to spread deliveries to a smelter evenly over the year. Given these volumes, one to two ships per month will likely have to be scheduled in order to move production.

At Stewart, there is an existing concentrate loader and at present, two sheds capable of holding over 30,000 tonnes. Presently the storage is used for Eskay Creek and Huckleberry productions, but both mines have limited lives remaining. It is probable that such storage may be available for Schaft Creek. Keep in mind that other mines in the area are planning to use Stewart as the concentrate shipping port. There is enough room to construct additional warehouse facilities capable of handling the volume needed for the Schaft Creek project if required. One caveat is the current environmental state at the present facilities.

If additional warehouse facilities were necessary, Stewart Bulk Terminals may finance construction of the shed at its cost and amortize against throughput. Space and plans for an additional 50,000 tonnes of storage are in place at the facility.

23.4 Tailings

Three options for tailings disposal sites were selected based on location, proximity to the concentrator, dam constructability, elevation, visual capacity, and groundwater and glacial meltwater drainage patterns. Each site was reviewed both from the ground (limited basis) and by air.

The three tailings storage locations are described as follows and illustrated on the figure below:

- Option A, located at Skeeter Lake;
- Option B, located on Hickman Creek;
- Option C, located on Fletcher Creek.

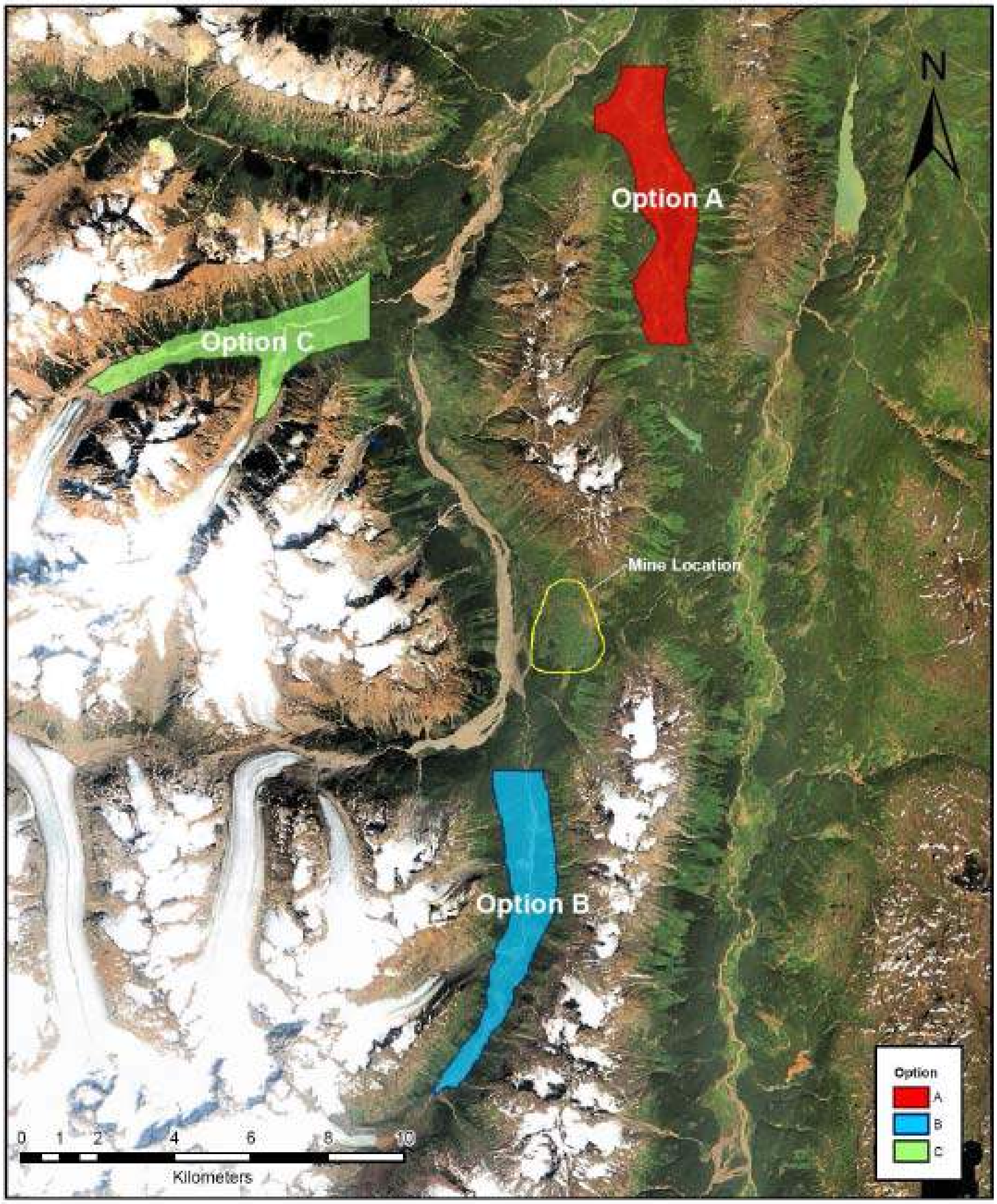


Figure 23.29 Potential Tailings Storage Locations

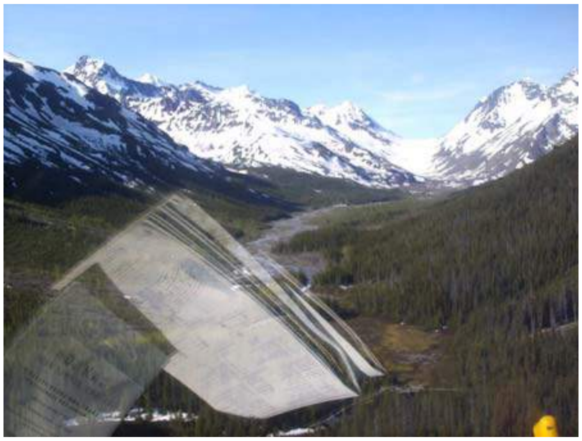
A brief description of each location is provided as follows:

Tailings Option 'A'

This option lies in a valley immediately east of Mount LaCasse. Several small surface water ponds as well as areas of swampy terrain are found on this site. Topographic information indicates that the site currently drains both to the north and south. The main dam will be located at the north end, with an adjacent low saddle dam. A lower dam will be needed at the south end.

Tailings Option 'B'

This option lies in a valley immediately west of Mount Hickman. The valley has a glacier at its head. Topographic information indicates that the site currently drains to the north as Hickman Creek. The only dam needed will be located at the north end.



Tailings Option 'C'

This option lies in a valley immediately west of Mount LaCasse and east of Schaft Creek. The valley has a glacier at its head. Topographic information indicates that the site currently drains to the east into Schaft Creek. The only dam needed will be located at the east end.

A preliminary assessment of glacial outburst floods, mass wasting and avalanches risk was undertaken in which impacts on the potential tailing impoundments options were evaluated. The assessment process identified possible sites of risk and sites that might be worthy of future study. This report can be found in Section 18.5 of this report.

While each site is exposed to one or more of the risks discussed above, it was determined that all three options are considered feasible at this stage. Preliminary 'opinions' of those involved are generally leaning toward Option A (Skeeter Lake) as the best choice for this project.

23.4.1 Mill Tailings

For scoping level design, a total tailings of 705 million metric tonnes, or megatonnes (Mt) was assessed for a 30.5 year mine life, based on a mill production of 23.4 Mt per year and concentrate production of 279,000 tonnes per year.

The proposed mill process will involve grinding so that the tailings material will consist of 80% passing 100 microns. The ore will be processed by conventional flotation methods without the use of cyanidation. Tailings material are expected to have an average settled dry density of 1.35 t/m^3 over the entire mine life.

The tailings material will be entirely contained within the tailings impoundment. Tailings will be transported hydraulically to the designated tailings deposition area where it will be spigotted off the crest of the dam and/or nearby valley slopes. During operations, an operating pond (and settling basin) will be created to allow water to be reclaimed to the plant. This pond will facilitate settling of suspended solids.

All three tailings options are more than capable of holding the entire projected mine life tailings volume. Option A, as shown currently, has a total capacity of 1.3 billion cubic metres which translates into 1.7 billion tonnes of tailings material on a dry basis. This offers the opportunity to reduce the tailings impoundment structure(s).

23.4.2 Tailings Dam

To date, no borrow areas in the immediate vicinity of the mine or tailing impoundments have been identified. It is assumed that the starter dam will be constructed from borrow material within the selected impoundment area. At this point the tailings dam will be a rockfill structure with an impervious (i.e. clay till) central core. The internal core of clay till is an impermeable barrier to control seepage through the dam. The core will be built of compacted local glacial deposits such as a clay rich till and glaciolacustrine clays and silts.

Foundation preparation prior to fill placement may require a shear key and grouting under the footprint of the impervious core to minimize the potential for seepage, piping and core cracking, highly dependent on future geotechnical work under the selected option.

Immediately downstream of the ultimate toe, a seepage collection system will be constructed to intercept seepage out of the pond. This system comprises a surface water diversion berm, a seepage recovery ditch and interceptor wells.

The dam will be expanded in stages as required. Inbetween main dam expansions, tailings material will be delivered by a series of pipelines with valved offtakes located along the top edge of the upstream dam face. The coarse fraction of the tailings will settle rapidly and accumulate closer to the discharge points, forming a gentle beach with an expected slope of one percent.

To protect the integrity of the tailings dam, a series of emergency spillways will be constructed on the abutment during the mine life. All spillways will be designed to pass the routed flow from a Probable Maximum Flood (PMF). The spillway control section will either be excavated into rock or riprap and slush grout will be used to armor the control section of the spillway.

Multiple snow avalanche hazards exist on the sides of the valleys for all options. This poses the potential to run out into the footprint of the proposed tailings and waste impoundment, and intersect and possibly block any diversion channels. A snow avalanche management program will be instituted during construction and operations. In addition, the potential of snow avalanches impacting the tailings pond and any waves generated by an avalanche will be accounted for in the final design.

23.4.3 Tailings Acid Generations

A preliminary report (Schaft Creek Project – Prediction of Metal Leaching and Acid Rock Drainage, Phase 1; March 2007) was completed on the acid generation and metal leaching potential of the Schaft deposit. Recommendations for additional studies presented in the report are currently underway. The preliminary report included the analysis of 59 samples of core rejects from the 2005 drilling program for expanded Sobek acid-base accounting (ABA) and for total-element analyses. The analysis of these samples revealed that only 0 to 2% of the samples were net acid generating and 5 to 14% were “uncertain” and thus will require additional test work to confirm their acid generating potential. These preliminary findings indicate that the Schaft Creek property will have limited to no acid generating tailings or waste rock. However, additional work is being undertaken to comprehensively characterize the Schaft Creek deposit.

Phase two of the metal leaching and acid rock drainage program includes a few hundred more samples for ABA and total-element testing, kinetic testing and on-site leach pad testing. This work is being conducted in conjunction with government regulators and the Tahltan Nation.

23.5 Infrastructure

23.5.1 Power Supply

The project site is currently isolated from power. A 138 kV transmission line has been proposed to connect the minesite. It has been determined that the estimated load at the mine site will have an average demand load of 92 MW and a maximum peak load of 103 MW. Electrical power will be supplied from British Columbia Transmission Corp's (BCTC) new 287 kV Northwest Transmission Line from a point near Bob Quinn Lake.

The study considers that the route for the 138 kV transmission line to the Schaft Creek project will be from Bob Quinn Lake along the proposed mine access road corridor. The proposed access road is discussed in more detail in Section 18.3.

Two separate standby generation systems will serve the site. The first system is at 4160 V and will provide standby power to the plant operations for safe shutdown and to provide power for essential services. The second system is dedicated to providing standby power to the camp facilities and administrative facilities.

23.5.2 Airfield

The Schaft Creek project will be a fly-in, fly-out operation and as such will require a dedicated airfield. A preliminary design for the airfield has included a compacted gravel landing strip capable of handling a Boeing 737, terminal building, fueling facilities, a small maintenance facility and complete control and lighting systems.

It is anticipated the airfield will be developed early in the project in order to assist with construction efforts.

23.5.3 Permanent Camp

It is anticipated a labour force of 1000 to 1200 will be required for the construction phase while the operations phase will require a permanent labour force of approximately 500 to 525.

Table 23.36
Permanent Labour Force Requirements

Function	Salaried	Hourly
G&A	55	-
Process	35	115
Mining	40	265
Total	**130**	**380**

The permanent camp facilities will be built first and used during the construction phase and then converted/refurbished to accommodate the permanent labour force. The design of the camp facilities is based on Atco structures, utilizing their standard sleeper modules, wash car modules, kitchen/diner modules and recreation hall modules.

The layout was configured for a 49 person barrack arrangement – each barrack having 7 sleeper modules and one wash car module. There will be a total of 12 barracks, two kitchen/diner modules and two recreation halls. This arrangement will accommodate a total of 588 personnel.

The work schedule will be 12 hour shifts for two shifts per day, rotating in for two weeks and out for two weeks. The assumption at this point is half of the hourly work force is needed on-site at the same time – 190 total, 95 personnel per shift.

All sleeper modules for both the hourly crews and salaried personnel will be converted from double occupancy rooms to single occupancy rooms following the construction phase.

Eight-49 person barracks will be dedicated for hourly personnel, thus allowing an extra 12 rooms for 'float'. Four-49 person barracks will be dedicated for salaried personnel, thus allowing an extra 66 rooms for visitors, consultants, vendors and 'float'.

During the construction phase all rooms will be double occupied, thus allowing a maximum of 1176 personnel.

The kitchen/diner modules are designed to handle 138 personnel each, thus a total of 276 personnel at one time.

The above arrangement is set up as single story structures in order to reduce visual impacts from Mount Edziza Provincial Park across the Mess Creek Valley. If footprint is an issue, then the 49 person barracks can be double stacked and even triple stacked.

23.5.4 Communication

The communication system for the mine site will include a satellite telephone system, PC LAN and fiber optic cabling connecting the various sites.

The fiber optic cable between the plant's buildings will be strung along the electrical transmission lines. The fiber optic cable is primarily used by the control system to link its components.

In the plant's buildings, phones will be available for internal site communications. Phone communication will be carried over the Ethernet network (voice-over-IP technology). Phones will be distributed over the site. The camp will offer phone, ethernet and satellite service.

A satellite dish will be installed on-site to provide external voice and data links.

23.5.5 Buildings

The following buildings will be part of the mine site facilities:

- Mill
- Truck shop
- Warehouse
- Administration
- Maintenance
- Laboratory

- Camp facilities
- Airfield facilities
- Explosives storage
- Filter plant
- Water treatment plant
- Potable water

The mill building will be divided into two sections, one for the grinding section and one for the flotation section. There will be a small machine and welding shop to be used for minor maintenance. A shower and locker facility will be provided for the mill operators. Offices for the mill staff are included and a control room to run the operations.

The truck shop will be a large complex which serves multiple purposes. In addition to providing an area for maintenance bays, tire shops and a wash bay, the truck shop complex will also house a small welding bay, electrical shop, first aid room, machine shop area, mine dry, warehouse, offices for mining and engineering staff, lunch room and the foreman's office.

A single level building will house the main administrative functions for the operation with offices and open work areas for senior management and administration. There is also a small lunch room, mud and storage room, meeting rooms and an electrical/mechanical room.

A maintenance shop will be located adjacent to the concentrator which will house maintenance shops, offices, storage and working space required for maintaining the equipment in the concentrator.

The assay laboratory will be located adjacent to the mill building. The assay lab will provide equipment for sample preparation and assaying as well as support services for the mill.

23.5.6 Water Systems

The following water systems are included and have been accounted for in the early design phase for this project:

- Process water
- Fresh water
- Potable water
- Waste water
- Fire water

Process water is primarily required for the purposes of ore dilution for milling and flotation. The main source of process water will be recycled water from the tailings and concentrator thickener overflows. This typically constitutes about 70 - 75% of the process water requirements. The remaining water will be obtained from reclaim water from the tailings dam and fresh water makeup.

Process water will be supplied from two centrally located tanks.

Fresh water will be required for mill makeup water, gland seal water, fire water and potable water. Fresh water will be obtained from groundwater wells in the vicinity of the mine and mine pit dewatering wells.

The potable water system will comprise of standard water treatment units and will provide the daily water requirements at approximately 3.5 – 4.0 m^3 per hour. A potable water storage tank will provide a one day storage capacity.

There will be two domestic waste water treatment units on site. One unit will serve the main industrial facilities and camp and the other will be dedicated to the truckshop and explosive facilities due to their remote locations.

Water will be available to the firewater main from the fresh/fire water tank. The storage tank will have a firewater reserve that will supply two hours of firewater in the event of an emergency. The firewater reserve will not be accessible to the mill fresh water system. A two hour firewater reserve will be ensured by piping the firewater from the bottom of the storage tank, and the fresh/potable water systems from approximately 8.5 metres higher in the tank in order to guaranteed water availability.

23.6 Reclamation & Closure

23.6.1 Mine Closure and Reclamation

When the mine reaches the end of its life, it will embark on a mine closure and reclamation plan that will meet its end land use objectives and satisfy its regulatory commitments.

Ultimately, the goal is to re-establish the land to a productive environment that will be compatible to its natural surroundings. Restoration of terrestrial and aquatic life will be primary objectives. Stable, re-shaped landforms will be created to ensure self maintenance capability in perpetuity.

Progressive reclamation in conjunction with on-going mining activities will be practiced where applicable to minimize overall mine closure costs. This approach will also allow early monitoring of reclamation activities and advance closure to certain mine areas.

Suitable surficial soils excavated during mining will be stockpiled and used to cap re-contoured landscape at decommissioning. Where possible, direct placement of the suitable topsoil material will be carried out thereby avoiding stockpile losses and re-handling costs.

Post closure landform, reclamation, and ARD/Heavy metal impacts of the project will be the subject of extensive work in future studies. The following general design aspects for post mining considerations are based on typical considerations for other projects in this area and specific early evaluation of the rock. Detailed design criteria will be adjusted based on these future studies and requirements.

23.6.1.1 *Mine Waste Dump Reclamation*

Mine waste dumps comprised of non-reactive waste rock will be constructed in series of lifts at 37 degree inter-slope angles, with appropriate berm widths that will effectively result in an overall slope angle of 26 degrees. The overall 26 degree slope angle is the recommended maximum angle that will sustain long term vegetation. Research and testing during the life of the operations may show that a steeper reclaimed slope is more suitable for the end land use.

At decommissioning, the surfaces of the waste dumps will be re-contoured and scarified. The crests of the berms will be rounded and inter-slope angles will be reduced to provide overall slopes of 26 degrees including berms. Low gradient drainage ditches will be established across the slopes to collect surface runoff. The surfaces will be capped with suitable surficial soils and seeded to establish vegetation that will minimize erosion.

23.6.1.2 *Mine Roads and Dykes*

Decommissioned mine roads will be scarified and capped with suitable surficial soils. Dykes and dams that are exposed above the water line will be also scarified and capped with suitable soils. The surfaces will then be seeded to establish vegetation.

23.6.1.3 *Pit Areas*

Generally the mined out pits will naturally be filled with water from surface runoff and groundwater forming lakes. Spillways will be constructed to manage the overflow and directed to the watercourses as established in the mine closure water management plan. Ditches will be constructed on the pit berms to manage the runoff from the pit walls. Typically the pit walls are not planned to be re-sloped.

23.7 Socioeconomics

This section provided by Robert Simpson, President of PR Associates.

23.7.1 First Nations Consultation Plan

PR Associates will lead the First Nations consultation activities in respect of the Project and will lead the broader public consultation program and ensure that First Nations are informed of the broader public engagement process and opportunities for input regarding the Project.

PR Associates will ensure that its First Nation consultation activities meet all legal requirements relating thereto, including assisting any Crown decision-makers in carrying out their respective consultation requirements.

PR Associates consultation plan includes implementation of the following initiatives with the potentially affected First Nations:

- Providing timely information and updates regarding the Project on an ongoing basis;
- Providing timely information and updates regarding the environmental and regulatory approval processes associated with the Project on an ongoing basis;
- Responding to First Nations' questions and/or information requests regarding the Project in a timely manner;
- Seeking to understand First Nations' concerns or issues respecting the Project and consider such issues or concerns in the Project's final design and delivery;
- Engaging in discussions with First Nations to further identify means, where appropriate, to mitigate, minimize or otherwise accommodate First Nations' concerns or issues relating to the Project;
- Documenting all consultation activities with First Nations and providing a record of such documentation to all appropriate Crown decision-makers and regulators; and
- Providing regular opportunities for First Nations to meet with Copper Fox, and their representatives to exchange information regarding the Project.

First Nation consultation activities will be undertaken throughout the pre-application stage of the Environmental Assessment (EA) process (including the preparation of the EAC Application), and subsequently during the application review and public comment period (following submission of the Application and in conjunction with Technical Working Group review activities). The need for post-Certification consultation initiatives will be assessed on an ongoing basis leading up to EA Certification, should the project proceed.

23.7.2 First Nations

An examination of asserted traditional territory, Treaty Boundaries, Statement of Intent boundaries and known consultation boundaries primarily identified by the B C Treaty Commission (BCTC), has resulted in the Tahltan nation being identified as the only group with historical claim over the project area.

As such, the Tahltan Nation will be consulted individually as project planning proceeds in relation to the potential for project related impacts on their aboriginal interests.

PR associates initiated contact with the Tahltan Nation by phone call, letter, and fax on November 9th, 2006. The Tahltan Nation was provided with a letter of introduction that outlined the respective roles of PR Associates and Copper Fox Metals, identified an internal point of contact for further communication, and requested an opportunity to meet with the Tahltan Central Council and their representatives to discuss the potential project.

The Tahltan Central Council was also provided with a project information handout, and map showing the mine site and proposed plans.

Two initial meetings were held in December 2006 and January 2007 and included PR Associates, Copper Fox Metals and members of the Tahltan Central Council. Copper Fox provided the Tahltan Central Council with a project description to facilitate a discussion of preliminary issues, interest and impacts within the proposed mine site.

Copper Fox emphasized their desire to work with First Nations to design a consultation process that was participatory and reflective of their interests and expectations The Tahltan also provided guidance with respect to future consultations and clarified which level of First Nations government would represent their Traditional Territory interests.

As a result of these initial meetings, capacity funding was offered to each First Nation's respective representative to compensate them for their attendance during the first round of meetings, and to provide resources for future participation during this introductory phase of discussions.

Comprehensive capacity and participation agreements with the Tahltan are expected as the project approval and development process moves forward, and discussions are currently underway in anticipation of this next step.

23.7.3 Public Consultation

PR Associates' general approach for notifying and consulting with public stakeholders includes the following principles:

Open public process:

Interested parties will be encouraged to participate throughout the planning process and regulatory review, with effective two-way communication. The consultation process is viewed as an opportunity for constructive dialogue with an informed audience;

Meaningful consultation:

Interested parties can expect that consultation will be real and meaningful from their perspective. Specific expectations may vary among the people being consulted and may change over time;

Transparent and accountable:

Interested parties can expect that they will be provided with access to all relevant information and that they will be informed about changes in the Project and decisions made by Copper Fox; and

Consultation, not consensus:

Input from the public, along with information gathered through technical, environmental and social impact studies, will inform Copper Fox decisions.

23.7.4 Public Consultation Objectives

Copper Fox has a responsibility to communicate Project intent, respond to public issues and concerns, track commitments, and gather suggestions with regard to its construction, schedule and operations.

The objectives of the public consultation plan are:

- Project Justification: to provide information to interested and affected residents and stakeholders about the need for the Project;
- Public Education: to explain the rationale for the Project;
- Issues Identification and Management: to identify and address potential issues and concerns from a community and stakeholder perspective; and
- Public Input: to identify opportunities for individuals to have input into BCTC decisions that may affect them and, where feasible and appropriate, make adjustments to the plan based on their input.

As the environmental assessment process for the project proceeds, Copper Fox Metals will undertake public notification and consultation activities throughout the preparation of the Environmental Assessment Certificate (EAC) and Certificate of Public Convenience and Necessity (CPCN) applications to engage the public with meaningful input and feedback. Copper Fox will also undertake formal public consultation activities, such as scheduled public meetings and information sessions, during the public comment period following submission of the Application for regulatory and public review. The scope and delivery of the events will be arranged to best engage input from communities.

23.7.5 Public Consultation

Consultation activities that will be undertaken through all stages of the Project include the following:

- Public issues scoping and community profiling;
- Website development and printed materials;
- Meetings with media in the project area;
- Meetings with key stakeholder groups;
- Open houses, information sessions and meetings to raise awareness, and to identify and address issues and concerns;
- Ongoing issues tracking and proactive response;

- Public notification of events, meetings and the status of the project using a variety of media (predominantly advertising, and both hardcopy and electronic mailouts); and
- Providing comprehensive reporting of the process and results of the consultation process, including consultation summaries to support the EAC and CPCN applications.

23.8 Project Development

A preliminary development schedule is provided in Section 24.0. Key milestones for the project development consist of:

• Preliminary Economic Assessment completed	Nov. 30, 2007
• Prefeasibility Study completed	Apr. 30, 2008
• Place PO for mills	May 1, 2008
• Place PO for electric shovels	May 1, 2008
• Environmental Assessment Submitted	Nov. 14, 2008
• Feasibility Study completed	Dec. 31, 2008
• Place P for crushers	Jan. 1, 2009
• Place PO haul trucks	Jan. 1, 2009
• Place PO for electric drills	Jan. 1, 2009
• Place PO for regrind mills	Jan. 1, 2009
• Permit approval	Jul. 15, 2009
• Start construction	Jul. 16, 2009
• Detailed engineering completed	Dec. 31, 2009
• Begin tailings starter dam	Jul.1, 2010
• Begin mine development	Jan. 3, 2011
• Mechanical completion	Jul. 4, 2011
• Complete tailings starter dam	Jul. 4, 2011
• Mine development completed	Oct. 3, 2011
• Begin production	Dec. 1, 2011

23.9 Operating Cost Estimate

23.9.1 Summary

The operating costs for the Schaft Creek Project have been estimated in Q2/Q3, 2007 Canadian dollars and do not include allowances for escalation or exchange rate fluctuations. The requirement for this operating cost estimate is intended to be at a scoping level with an accuracy of ± 35 percent.

Where source information was provided in other currencies, these amounts have been converted at rates of 1 US$ = 1 C$.

Operating costs were assembled for the mine and concentrator separately; however, both estimates were built up from the organizational chart and unit prices for commodities delivered to the mine site.

Unit rates for power costs are based on current knowledge of rates in the area, some earlier meetings with BC Hydro in British Columbia and recent estimates from other developing operations in the area. A rate of $0.050/kWh is used. Power costs are based on the unit rates for power and the electrical load analysis developed for the project.

General and Administrative costs include:

- Salaried and hourly labour for administrative personnel
- Operating and maintenance costs for support vehicles
- Access road and powerline maintenance
- Site avalanche control
- Communications expenses
- Camp Operations
- Fly-in, Fly-out Operations and Airfield Operations
- Safety supplies/incentives
- Off-site training and conferences
- Insurance
- Corporate Services and Travel
- Environmental monitoring
- Security
- Medical services
- Professional Membership Costs
- Community development
- Land Holding
- Consultants
- Computer equipment and software
- Miscellaneous Office Supplies
- Miscellaneous Freight and Couriers
- Recruiting and Relocation
- Legal, permits and fees

A concentrate marketing study was commissioned to confirm the marketability of the copper and molybdenum concentrates. This included the cost for smelting and refining.

A summary of the operating costs (based on 23,400,000 ore tonnes per year) by area are shown in the table below.

Table 23.37
Summary of Operating Cost – Life of Mine Average

Description	Annual Cost	Cost/Tonne Ore	Cost/Tonne Mined
Mining	$92,151,433	$3.94	$1.47
Processing	$91,484,032	$3.91	
General & Admin	$17,008,500	$0.73	
Subtotals	**$200,643,964**	**$8.58**	
Conc Handling & Transport	$62,596,253	$2.68	
Totals	**$263,240,217**	**$11.25**	

23.9.2 Mining Costs

This section provided by Jim Gray of Moose Mountain Technical Services (MMTS).

Unit costs for consumable and labour rates were estimated from sources listed below while the magnitude of consumables and labour required are determined for each specific activity from experience and first principles.

The unit costs are based on the following data:

- Salaries for each job category are based on MMTS's experience of similar operations in BC mines. An average burden rate of 40% has been applied to base salaries to include all statutory Canadian and BC, social insurance, medical and insurance costs, pension and vacation costs. For hourly employees, the labour rates for mines in BC were used.
- Mine designs to determine the size and makeup of the mine fleet as well as fuel requirements which is affected by distance from the pit to the various destinations and topography.
- Budgetary quotations, including freight, for all consumables including tires and fuel. Fuel is estimated at a delivered cost to site of C$ 0.80 /litre.
- Mining equipment consumables, major equipment replacements, labour loading factors, equipment life and costs are based on vendor information and MMTS's experience in similar mining operations.

The current fleet hourly operating costs are used as a constant basis over the schedule periods.

The basic operating capacities of the equipment, the travel speed characteristics of the trucks, and the haul road profiles, the equipment productivities for the shovels and trucks are calculated from the MineSight production scheduling program.

The truck speeds and cycle times for the various haul cycles were calculated by using CAT's FPC simulation program. The equipment productivity and the scheduled production are used in the scheduling program to calculate the required equipment operating hours. These are multiplied by the hourly consumables consumption rates and unit operating costs to calculate the total equipment operating costs for each time period.

Each major part replacement was calculated from the expected life of the major part, the cost of the part, and the fleet size for that equipment. The same parametres were used for equipment replacement cost calculations.

Blasting costs were based on studies from similar projects and historical blasting costs.

Geotechnical costs are based on historical data collected by MMTS.

Labour factors in Man Hours/operating hour were also assigned to each of the equipment. Labour costs were calculated by multiplying the labour factor by the equipment operating hours, and labour costs are allocated to the equipment where labour has been assigned. The total hours required for each job type on all the equipment are summed and any additional labour required to complete a crew is assigned to unallocated labour. Some trades in Mine Operations (Grader Operator, Track Dozer Operator, Scraper Operator, Water Truck Operator, and Fuel Truck Operator) and Mine Maintenance (Crane Operator, Welder, Tireman, Labourer and Service man) are treated as shared labour during the unallocated labour assignment. The mine hourly and salaried labour schedules were generated as well to support the operating cost estimate.

Cost estimates for the base case mining operations are broken down by function in the table below.

Table 23.38
Summary of Operating Cost - Mining (Life Of Mine Average)

Description	Type	Annual Cost	Cost/Tonne Ore	Cost/Tonne Total Mat'l
Drilling	Variable	$5,625,255	$0.24	$0.09
Blasting	Variable	$14,671,753	$0.63	$0.23
Loading	Variable	$7,184,462	$0.31	$0.11
Hauling	Variable	$36,114,715	$1.54	$0.58
Mine Maintenance	Variable	$1,495,191	$0.06	$0.02
Mine Operations Support	Variable	$16,328,033	$0.70	$0.26
Snow Removal	Variable	$1,222,700	$0.05	$0.02
Geotech	Variable	$1,451,602	$0.06	$0.02
Unallocated Labour Cost	Variable	$927,328	$0.04	$0.01
Mine Ops G&A	Fixed	$2,115,608	$0.09	$0.03
Mine Maintenance G&A	Fixed	$2,377,343	$0.10	$0.04
Mine Engineering G&A	Fixed	$1,477,182	$0.06	$0.02
Technical Services G&A	Fixed	$1,160,260	$0.05	$0.02
Mining Total		**$92,151,433**	**$3.9381**	**$1.4701**

23.9.3 Processing Costs

This section provided by Ray Hyyppa of Hyyppa Engineering, LLC and Matt Bender of Samuel Engineering.

Processing costs are a summation of administrative, operating and maintenance labour, power, consumables including reagents, grinding liners and media, and an estimate for maintenance and miscellaneous operating supplies.

Labour costs are based on the organizational chart and Canadian labour rates. Salaries for each job category are based on experience of similar operations in BC mines. An average burden rate of 40% has been applied to both salaried labour and hourly labour to include all statutory Canadian and BC, social insurance, medical and insurance costs, pension and vacation costs. For hourly employees, the published labour rates for BC mines were used.

Power costs are based on the electrical load analysis for the entire site operation and a unit rate of $0.050/kWh.

Reagent costs are based on projected consumptions from metallurgical testwork and quotations for unit reagent prices.

Grinding steel consumption is an estimate based on previous project experience. Liner and media prices are based on current markets.

Maintenance and miscellaneous operating supply costs are a percentage factor based on previous project experience.

A small amount of ore will be run through the concentrator during the pre-operational phase of startup. It is recognized that a ramp-up period for production will be experienced at startup but the duration has not been accounted for at this stage of the project.

Operating costs are summarized below.

Table 23.39
Summary of Operating Cost - Processing (Life Of Mine Average)

Description	Fixed / Variable	Annual Cost	Cost / Tonne Ore
Salaried Personnel	Fixed	$2,156,000	$0.09
Hourly Operations Labour	Fixed	$7,486,335	$0.32
Hourly Maintenance Labour	Fixed	$3,447,360	$0.15
Site Plant Electrical Power	Variable	$34,559,300	$1.48
Reagents	Variable	$12,753,423	$0.55
Grinding Steel	Variable	$25,730,663	$1.10
Maintenance Supplies (5% of process ops $)	Variable	$4,306,654	$0.18
Misc. Ops Supplies (1% of processing ops $)	Variable	$1,044,296	$0.04
Processing Total		**$91,484,032**	**$3.91**

23.9.4 General and Administrative Costs

General and administrative costs include costs for everything that is not directly attributed to operations, as outlined in the table below.

Table 23.40
Summary of Operating Cost – G&A (Life Of Mine Average)

Description	Type	Annual Cost	Cost/Tonne Ore
Salaried Personnel	Fixed	$3,661,000	$0.16
Hourly Labour	Fixed	$892,500	$0.04
Vehicle Operating & Maintenance	Fixed	$75,000	$0.003
Access road & powerline maintenance	Fixed	1,700,000	$0.07
Site Avalanche Control	Fixed	650,000	$0.03
Communications	Fixed	$75,000	$0.003
Camp Operations	Fixed	$2,000,000	$0.09
Fly-in, Fly-out Operations & Airfield Operations	Fixed	$5,000,000	$0.21
Safety Supplies / Incentives	Fixed	$150,000	$0.01
Offsite Training & Conferences	Fixed	$200,000	$0.01
Insurance	Fixed	$850,000	$0.04
Corporate Services and Travel	Fixed	$200,000	$0.01
Environmental	Fixed	$200,000	$0.01
Security & Medical	Fixed	$200,000	$0.01
Professional Membership Costs	Fixed	$30,000	$0.001
Community Development	Fixed	$200,000	$0.01
Land Holding	Fixed	$-	$-
Consultants	Fixed	$200,000	$0.01
Misc. Computer Equipment/Software	Fixed	$100,000	$0.004
Misc. Office Supplies	Fixed	$100,000	$0.004
Misc. Freight & Couriers	Fixed	$75,000	$0.003
Recruiting and Relocation	Fixed	$200,000	$0.01
Legal, Permits, Fees	Fixed	$250,000	$0.01
General and Administration Total		**$17,008,500**	**$0.7269**

Salaried Personnel and Hourly Labour

Labour costs are based on the organizational chart and Canadian labour rates. Salaries for each job category are based on experience of similar operations in BC mines. An average burden rate of 40% has been applied to both salaried labour and hourly labour to include all statutory Canadian and BC, social insurance, medical and insurance costs, pension and vacation costs. For hourly employees, the published labour rates for BC mines were used.

Vehicle Operating and Expense

This includes the annual cost for fuel, lubricants, spare parts, and maintenance for pickup trucks, vans, busses, maintenance vehicles, forklifts, and other vehicles required to support the overall operation. All vehicles are assumed to be diesel. An annual allowance was used based on experience with similar operations.

Communication Expenses

This includes monthly fees for landline and mobile telephone services, satellite transmission services, and Internet services. An annual allowance was used based on experience with similar operations.

Camp Operations

This includes the cost for operating and maintaining the permanent camp facilities. An annual allowance was used based on experience with similar operations.

Fly-in, Fly-out Operations and Airfield Operations

This includes the cost to operate and maintain the airfield operations and the contract cost of flying the operations work force in and out. It is anticipated the operations will be on a two week in, two week out rotation. An annual allowance was used.

Safety Supplies

A total of $300 per man per year was allowed for safety support.

Off-site Training and Conferences

A total of $16,667 per month was allowed for personnel to attend off-site training, conferences and seminars.

Insurance

This includes general liability, risk, and vehicle insurance policies. An annual allowance was used based on experience with similar operations.

Corporate Services and Travel

An annual allowance was used based on experience with similar operations.

Environmental

This includes costs associated with quality sampling and monitoring, analysis of surface and ground water, as well as surface flow measurement. An annual allowance was used based on experience with similar operations.

Security and Medical

This includes costs for contracted security and medical services. An annual allowance was used based on experience with similar operations.

Professional Membership Costs

An annual allowance was used based on experience with similar operations.

Community Development

An allowance was assumed for costs associated with public disclosure and information programs, assistance programs for communities, and other related programs. This allowance was based on Copper Fox Metals current community development program. This does not include staff labour.

Land Holding

No allowance was made.

Consultants

An annual allowance was used based on experience with similar operations.

Computer Equipment/Software

An annual allowance was made to cover replacement and/or addition of computer system peripherals and standard software packages. This does not include replacement of computer workstations or other major hardware, which are included in sustaining capital.

Miscellaneous Office Supplies

An annual allowance was used based on experience with similar operations.

Miscellaneous Freight and Couriers

An annual allowance was used based on experience with similar operations.

Recruiting, Hiring and Relocation

An annual allowance was used based on experience with similar operations to account for the costs involved in recruiting and hiring new staff due to normal turnover. Allowances were also made to cover the cost of providing relocation assistance to a certain percentage of new employees.

Legal, Permits and Fees

This includes costs for permits and fees required by various governmental entities. Outside legal counsel has also been identified. An annual allowance was used based on experience with similar operations.

23.9.5 Concentrate Handling and Treatment

This section was assembled from a combination of sources: Hugh Hamilton of H.M. Hamilton & Associates Inc; Ray Hyyppa of HEL; Matt Bender of Samuel Engineering; Jim Gray of Moose Mountain, Dave Mullen of Cordy Oilfield Services (Copper Fox consultant for trucking) and other industry experts as required.

Table 23.41
Concentrate Handling and Treatment

Description of Charges	Type	Annual Cost	Cost/Tonne Ore	Cost/DMT Concentrate
Cu Conc Transport to Receiving Port	Variable	$25,281,075	$1.0804	$90.62
Copper Concentrate Treatment	Variable	$16,739,219	$0.7154	$60.00
Copper Concentrate Refining	Variable	$9,779,470	$0.4179	$35.05
Gold Refining	Variable	$800,451	$0.0342	$2.87
Silver Refining	Variable	$337,310	$0.0144	$1.21
Mo Conc Transport to Receiving Port	Variable	$406,012	$0.0174	$72.55
Moly Conc Roasting & Refining	Variable	$9,252,716	$0.3954	$1,653.47
Totals		**$62,596,253**	**$2.6751**	

Copper concentrate transportation includes: the cost of trucking bulk copper concentrate from the mine site to the port of Stewart ($45/wmt); port handling, storage, assaying, ship loading, and insurance ($11.40/wmt); and ocean freight ($40/wmt). It is assumed the copper concentrate will be shipped to a smelter located in China or Japan.

Copper concentrate treatment and refining charges of $60/dmt and $0.06/lb copper. The TC & RC's are current pricing from major smelters in Asia.

Gold and silver refining charges are $5.00/oz and $0.35/oz respectively.

Molybdenum concentrate transportation includes: the cost of trucking bagged moly concentrate from the mine site to the port of Stewart ($45/wmt); port handling, storage, assaying, ship loading, and insurance ($11.37/wmt); and ocean freight ($20/wmt). It is assumed the moly concentrate will be shipped to the Molymet smelter located in Santiago, Chile.

Moly concentrate roasting and refining charges are $0.75/lb moly.

23.10 Capital Cost Estimate

23.10.1 Summary

The capital costs for the Schaft Creek Project have been estimated in Q3/Q4 2007 Canadian dollars and do not include allowances for escalation.

The requirement for this capital cost estimate is intended to be at an accuracy of ± 35 percent.

Where source information was provided in other currencies, these amounts have been converted at rates of 1 US$ = 1 C$.

The rate of foreign currency exchange could have a serious impact on the value of labour and materials obtained in the local market (including freight, duties and taxes). In addition, the value of the dollar against other worldwide currencies, such as the Euro, could also influence future Project cost. No additional funds have been allocated in the estimate to offset any currency fluctuations.

At this stage, much of the direct costs have been derived using historical data from similar existing facilities and applying factors and escalation as required for size and other project specific requirements.

The capital cost estimate for the Schaft Creek Project has been developed to support the evaluation and assessment of a base case plant capable of processing 65,000 MTPD of copper-bearing material at the conceptual level of analysis.

The estimate is not sufficient for final decision making. However, it will help to further evaluate the Project's viability with respect to capital cost by establishing parametres from which further financial analysis and future funding may be based.

The estimated cost to construct and commission the facilities is $570 million, while the the owner's costs are estimated to be $385 million. The Contingency has been estimated at $164 million. In light of recent industry activities, Copper Fox Metals has elected to also add a project reserve provision of $300 million to the estimate. This brings the total project capital cost to an estimated $1,428 million.

A more detailed beakdown of the estimate is provided in the table below.

Table 23.42 Breakdown of Capital Cost	
	Total ($Ms)
Mine Area Facilities	31.2
Ore Storage & Handling and Crushing	53.5
Grinding and Concentrating	256.9
Tailings	45.6
Concentrate Filtration & Loadout	6.9
Buildings and Ancillary Facilities	26.9
Site Development	29.5
Direct Cost	***450.5***
Frieght	19.0
Contractor Construction	32.3
Construction Camp	30.1
EPCM Services	37.8
Testing, QA/QC, Vendors, Commissioning	9.5
Contracted Cost	***128.8***
Mining & Ancilliary Equipment	184.6
Mine Development	44.5
Spares, Rolling Stock, Initial Fills	16.9
Admin, Shop, Warehouse, Medical, Security, Safety, Camp, Communications	6.2
Transmission Line	38.5
Site Access Road	43.4
Helicopter Support Services	30.0
Owner Indirects	20.7
Owner’s Cost	***385.3***
Subtotal	***964.7***
Contingency	163.7
Project Reserve Provision	300.0
Total	**1,428.4**
Working Capital (not included in total)	49.8
Sustaining Capital (not included in total)	200.6
Reclamation & Closure (not included in total)	87.0

23.10.2 Scope

The estimate addresses the engineering, procurement and construction of a greenfield 65,000 MTPD copper (gold, silver and molybdenum) porphyry deposit located in the Liard Mining Division of northwestern British Columbia. The deposit is located 1,040 km north of Vancouver, 100 km by road northwest of Bob Quinn Camp, on the Stewart-Cassier highway.

The Scope of Facilities addressed in this estimate are those related to the development of the mining facilities and the copper concentrator plant facilities including all major mining and process equipment, materials of construction, construction and associated indirect costs such as EPCM, freight and start-up.

The scope of this estimate includes off-site infrastructure such as roads and power lines.

The capital cost estimate is based on the following information:

- Mine Design & Fleet Selection
- Design Criteria
- Process Flow Diagrams
- Mechanical Equipment List
- Electrical Single Lines
- Plot Plans and General Arrangement Drawings
- Electrical Equipment List
- Budget quotations from vendors
- In-house historical data and database information

Samuel Engineering was responsible for the preparation and assembly of the capital cost estimate with supporting data provided by others for the key areas noted below:

Mine Equipment	by Moose Mountain Technical Services
Access Road	by McElhanney
Transmission Line	by Copper Fox
Process Equipment	by Samuel Engineering
Contracted Cost	by Samuel Engineering and Copper Fox
Owner's Cost	by Samuel Engineering and Copper Fox
Contingency	by Samuel Engineering
Working Capital	by Samuel Engineering and Copper Fox
Sustaining Capital	by Samuel Engineering and Moose Mtn
Reclamation & Closure	by Samuel Engineering

23.10.2.1 *Exclusions*

Items not included in the capital estimate are:

- Land Acquisition and Rights-Of-Way
- Sunk Costs
- Permitting
- Environmental Studies
- Reclamation Costs (Included in Financial Analysis)
- Escalation Beyond Fourth Quarter 2007
- Foreign Currency Exchange Rate Fluctuations
- Interest

- Financing Cost
- Sales Taxes
- Risk due to political upheaval, government policy changes, labour disputes, permitting delays, weather delays or any other force majeure occurrences Accuracy

Payroll and fuel taxes are included in this capital cost estimate.

23.10.3 Accuracy

The requirement for this capital cost estimate is intended to be at a preliminary assessment level with a minus 35% plus 35% accuracy.

At this stage, much of the direct costs have been derived based on rough material take-off quantities for mass excavation, fill, concrete and structural steel. Labour hours were then applied to these areas at prevailing labour rates.

Overall, there is expected to be approximately 3.1 million cubic metres of earthwork, 55,500 cubic metres of concrete, 10,500 tonnes of steel in this facility, and 6.8 million labour hours.

23.10.4 Contingency

Contingency is an allowance included in the capital cost to cover unforeseeable costs within the project's scope of work, but which cannot be explicitly defined or described at the time of the estimate due to lack of information. It is assumed that contingency will be spent, however it is does not cover scope changes or project exclusions.

The contingency allowance has been evaluated by considering the estimating uncertainties for the various elements of the estimate and assigned respective percentage rates to cover the uncertainties based on the best judgment of the Project Team.

The resulting contingency amount for this estimate is an overall 17% which totals about $164 M.

No external risk factors (examples: escalation, currency exchange rates, political, weather, force majeure, etc.) have been addressed in this analysis. A project reserve provision of $300 million has been included by Copper Fox Metals to account for elements of risk outside of the defined variables.

23.10.5 Contracted Directs

Freight

Freight has been calculated as an allowance of 8% on equipment and materials costs. This is intended to cover all freight requirements from factory to jobsite, including packaging if required, factory loading, insurance, and inland trucking to site. No allowance has been made for air freight deliveries.

Construction Contractor's Indirect Costs

The Construction Contractor's indirect costs for the process facilities construction have been added at 20% of the direct construction labour.

Items in the contractor's indirect costs include:

- Contractor's mobilization and demobilization
- Supervision and administrative support costs
- Temporary construction facilities (offices, etc)
- Warehousing and lay down area cost
- Temporary toilets
- Construction vehicles, fuel and maintenance
- Weather protection and dust suppression
- Craft training and testing
- Clean-up and waste removal
- Bonds and insurances
- Utilities hook-up
- Temporary communications
- Construction surveying

Construction Power/ Utilities

Construction power/utilities have been added based on the schedule and the anticipated needs of construction. This includes power, fresh water and communications (phone, internet, fax, etc.).

Cost for hook-up to the utilities has been allowed for in the contractors' indirect costs.

Construction Man Camp

Construction man camp installation cost was provided by Copper Fox.

Camp Operations includes catering, housekeeping, maintenance and recreation.

23.10.6 Contracted Indirect

EPCM Services

EPCM services for the plant facilities have been included at 8% of the direct costs.

No allowance has been included for third party Construction Management. It has been assumed that this would be by the Owners personnel and included in "Owners Cost".

Surveying and Testing Services

Surveying and testing services have been added as a lump sum allowances to cover the cost of non-contractor surveying and QA/QC verifications.

An allowance has been used for independent testing and QA/QC services. This would include soils compaction, concrete sampling, pipe/tank welding, liner systems weld tests, etc.

Commissioning and Startup Services

For the period following mechanical completion, costs have been developed based on assumptions for craft labour, non-operations group commissioning team (CM contract), vendor representatives and power consumption. In addition, these costs include an allowance for the overall plant operations manuals and training materials preparation.

Vendor Representatives

Vendor Representatives have been added as a percentage of mechanical equipment based on historical data. The number used is 5%. This cost includes both labour and travel expenses.

23.10.7 Owner's Direct

Mining and Related Equipment

The initial and first year mining equipment requirements have been included in the capital cost estimate and are summarized below.

Table 23.43
New Capital Schedule ($000)

Description	Initial Capital Requirement
Drilling	18,080
Loading	44,950
Hauling	65,533
Mine OPS Support	41,434
Mine OPS Maintenance	5,989
Mine OPS Removal	8,590
Totals	**184,576**

Preproduction Mine Development

All mining related preproduction efforts have been included ($44.5 million).

Misc.

Communications, camp facilities, admin office equipment and warehouse items have been included.

Spare Parts

Spare parts have been allowed for at 5% of the total equipment costs. This is based on historical data from another project and includes commissioning and first year spares. Capital spares are not included.

Initial Fills

Initial fills of chemicals and reagents are included in the estimate. Each initial fill was identified, quantified and priced with phone quotes. The quantities used are based on the size of the tank or container required to be filled.

Site Security and Medical Services

Site security and medical services have been included as an allowance for facilities based on historical data which includes a medical facility, security building, firetruck and ambulance. Services have been added also as a monthly allowance to include a security service, medical doctors/nurses and supplies.

Power Transmission Line

An allowance of $350,000/km has been included for the new transmission line from Bob Quinn to site – a total of 110 km.

Site Access Road

A total of $43.9 million has been allocated for the access road from the mouth of More Creek in the Bob Quinn area to the mine site – a total of 110 km.

Helicopter Support Services

An allowance of $30 million has been included to support construction efforts.

23.10.8 Owner's Indirect

Owner's cost, including preproduction employment training, client management, camp operations, project administration, permitting, corporate services, insurances, environmental, security, medical, goodwill, and other expenses have been included.

23.10.9 Working & Sustaining Capital

Working capital, sustaining capital and reclamation & closure are included in the economic analysis for the project and are over and above the total capital cost estimate shown in Table 23.43. These include the following:

Working Capital

An allowance for four months of operating expenses, $49.8 million, is included to account for the delay between initial startup and first production until actual posting of revenue to Copper Fox's account.

Sustaining Capital

Sustaining capital for the mine, mill and tailings dam are included. The totals for each are; mining = $107.6 million, milling = $78 million, tailings dam = $15 million.

Reclamation & Closure

An estimate of $87 million is included to cover the cost for activities associated with the mine reclamation and closure. This cost is spread over the last five years of the mine life.

23.11 Economic Analysis

The economic analysis for the Schaft Creek Project are reported in Q3/Q4 2007 Canadian dollars and do not include allowances for escalation or foreign currency exchange flutuations.

Where source information was provided in other currencies, these amounts have been converted at rates of 1 US$ = 1 C$.

Below are the key assumptions and inputs used to generate the model results.

Table 23.44
Important Financial Model Assumptions and Inputs

Currency Exchange Rate	US$1.00 = C$1.00
Initial capital requirements	$1,428,356,000
Working capital	$49,761,000
Sustaining capital	$200,566,500
Reclamation & Closure	$87,000,000
Average mining rate	62 Mt/y or 170,000 tpd
Average mining cost	$3.94/tonne ore or $1.47/tonne total mat'l
Ore processing rate	65,000 tpd = 23,400,000 tpy
Average processing cost	$3.91/tonne ore
Annual G&A expenses	$17,008,500 ($0.73/tonne ore)
Average LOM Cu grade	0.303%
Cu recovery	90%
Cu concentrate grade	26.5%
Cu conc transport, handling and ocean freight	$90/tonne
Cu smelting	$60/tonne
Cu refining	$0.06/lb Cu
Average LOM Mo grade	0.02%
Mo recovery	72%
Mo concentrate grade	54%
Mo conc transport, handling and ocean freight	$72/tonne
Mo roasting & refining	$0.75/lb Mo
Average LOM Au grade	0.22 g/t
Au recovery	82%
Au grade in copper concentrate	18.4 g/t
Au refining	$5.00/oz
Average Ag grade	1.76 g/t
Ag recovery	72%
Ag grade in copper concentrate	113 g/t
Ag refining	$0.35/oz
Total LOM Cu production	1,861,825 tonnes
Total LOM Mo production	231,507,963 lbs
Total LOM Au production	3,890,181 oz
Total LOM Ag production	27,809,579 oz

Taxes

At this early stage of project development, financial results reported herein are prior to both taxation and the Teck Cominco Option Agreements (as reported in Section 4.3 of this report). The nature and timing of expenditures as well as the corporate structure of Copper Fox Metals will have a direct bearing on the cash taxes that will be incurred on the project. Some of the information that is required to make a more precise tax calculation include:

- The corporate structure that will be established when the Schaft Creek project is put into production. For example, if the project is in a joint venture company, limited partnership or new corporation will have a bearing on the calculation and distribution of after tax earnings;
- The timing and nature of expenditures and their deductibility for tax purposes;
- The applicable rates of mineral tax, before and after project costs are recovered, differ and will be better known when the project is closer to production;
- The applicable rates of federal and provincial income tax to be used will differ from existing rates when the project becomes taxable;
- Copper Fox has raised and may continue to raise additional capital through the issuance of flow through shares that flow through the tax deductions available to Copper Fox.

The current rates of taxation in Canada and British Columbia are as follows:

- Mineral taxes on production income are 2% before the cost recovery of the project and then 13% thereafter;
- Federal taxes are currently 22.12% of taxable income and are proposed to be reduced to 15% by 2012;
- British Columbia Provincial taxes are currently 12%;
- The Federal Government has said recently that it will seek to collaborate with the provinces to reach a combined federal-provincial rate of 25% by 2012.

23.11.1 Base Case Economics

A financial model was created utilizing the mine production schedule, the associated metal grades based on the geological resource estimate, metal recoveries from the Phase I test metallurgical program, capital and operating costs as set out herein and base case metal prices of copper US$1.50/lb, molybdenum US$10.00/lb, gold US$550/oz and silver US$10.00/oz.

Modeling at base case metal prices shows that the project could generate a cumulative before tax profit of C$2,047 million, with a payback period of 12 years, a 7.5% IRR, and a net present value discounted at 5% of C$380 million, over the 31 year mine life. Project net cash costs after byproduct credits are estimated to average C$0.57/lb Cu over the project life

23.11.2 Case 2 Economics

The same model was used to generate project economics utilizing the trailing three year average metal prices (end of October 2007) of copper US$2.66/lb, molybdenum US$27.00/lb, gold US$564/oz and silver US$10.40/oz.

Model results at these trailing three year metal prices shows that the project could generate a cumulative before tax profit of C$12,357 million, with a payback period of 3 years, a 32.7% IRR, and a net present value discounted at 5% of C$5,347 million, over the 31 year mine life.

23.11.3 Case 3 Economics

The same model was used again to generate project economics utilizing a staggered pricing concept that is becoming more prevalent in the industry. This concept assumes the short term (5 year) forecast is fairly predictable and accurate. In this case, a very recent (Nov. 2007) marketing study from a similar project was utilized for the pricing structure which is summarized below. The model takes advantage of the forecast pricing for the first two years of production and then reverts back to the base case pricing for the remainder of the mine life.

Table 23.45
Case 3 – 2 Year Staggered Pricing Strategy

	Year 1	Year 2	Years 3 – 31
Copper $/lb	2.76	2.76	1.50
Moly $/lb	22.38	22.38	10.00
Gold $/oz	700	700	550
Silver $/oz	12.00	12.00	10.00

Model results using this metal pricing strategy shows that the project could generate a cumulative before tax profit of C$2,720 million, with a payback period of 6 years, a 13.9% IRR, and a net present value discounted at 5% of C$976 million, over the 31 year mine life.

23.11.4 Case 4 Economics

The Case 3 model was used again to generate project economics utilizing a slightly different staggered pricing concept that is also becoming prevalent in the industry. This concept takes advantage of the same short term (5 year) forecast metal pricing for the first two years of production and then experiences a declining pricing trend over the next five years of production back to the base case pricing for the remainder of the mine life.

Table 23.46 Case 4 – 7 Year Staggered Pricing Strategy							
	Yrs 1-2	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yrs 8-31**
Copper $/lb	2.76	2.55	2.34	2.13	1.92	1.71	1.50
Moly $/lb	22.38	20.32	18.25	16.19	14.13	12.06	10.00
Gold $/oz	700	675	650	625	600	575	550
Silver $/oz	12.00	11.67	11.33	11.00	10.67	10.33	10.00

Model results using this metal pricing strategy shows that the project could generate a cumulative before tax profit of C$3,550 million, with a payback period of 4 years, a 22.2% IRR, and a net present value discounted at 5% of C$1,618 million, over the 31 year mine life.

23.11.5 Base Case Project Sensitivity Analysis

Sensitivity calculations were performed on the project cash flow by applying factors ranging from -15% to +30% against initial capital, annual operating costs, annual net revenue, copper grade and copper recovery. The effects on IRR and NPV are shown graphically in Figures 23-30 and 23-31, respectively. The project is moderately sensitive to changes in capital and operating costs and highly sensitive to changes in revenue (metal pricing) and metal recovery.

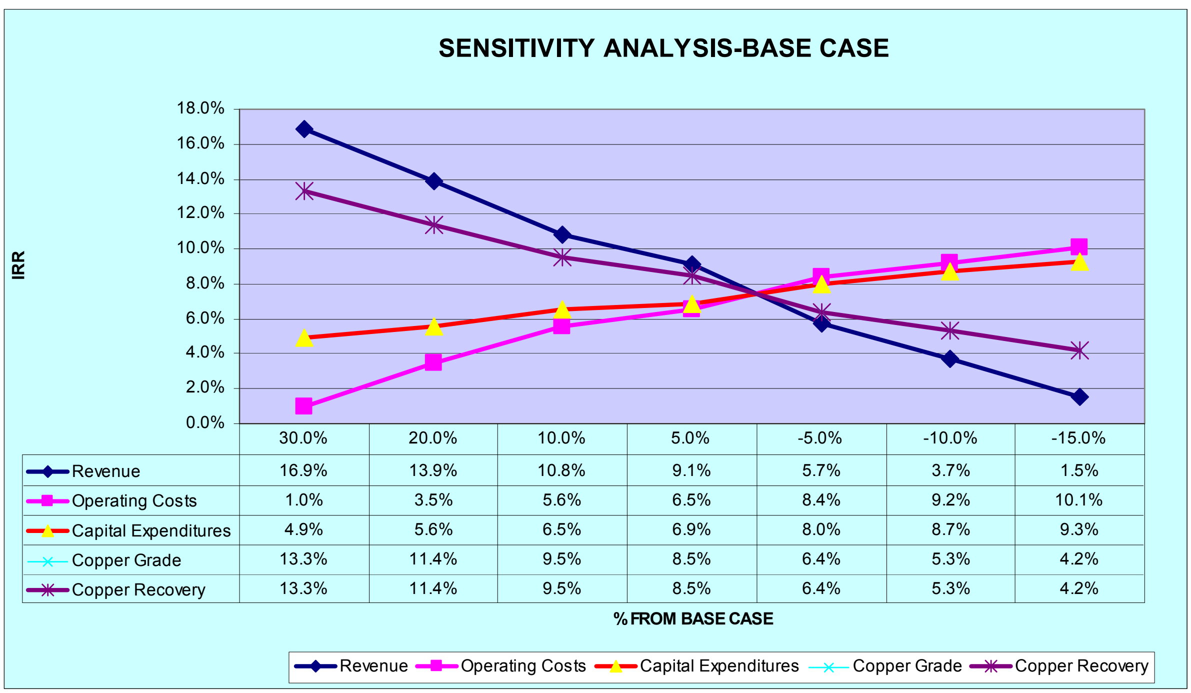
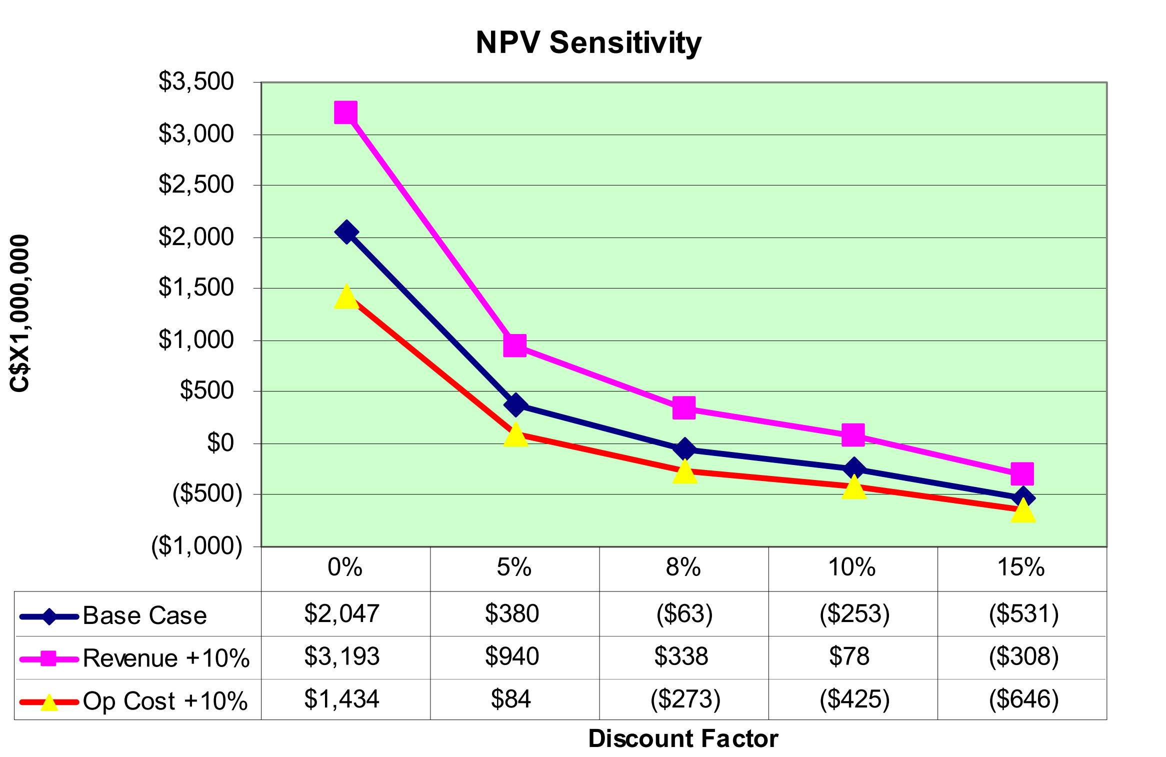


Figure 23.30 Base Case IRR Sensitivity Analysis



	0%	5%	8%	10%	15%
Base Case	$2,047	$380	($63)	($253)	($531)
Revenue +10%	$3,193	$940	$338	$78	($308)
Op Cost +10%	$1,434	$84	($273)	($425)	($646)

Figure 23.31 Base Case NPV Sensitivity Analysis

23.11.6 Project Economics Summary

Table 23.47 below is a summary of the economic results for the Schaft Creek project.

Table 23.47
Summary of Before Tax Economic Modeling Results

	IRR	NPV @ 5% ($million)	Project Profit ($million)
Base Case	**7.5%**	**$380**	**$2,047**
Cu ($/lb) = 1.50			
Mo ($/lb) = 10.00			
Au ($/oz) = 550			
Ag ($/oz) = 10.00			
Case 2 (Trailing 3 Year Average)	**32.7%**	**$5,347**	**$12,357**
Cu ($/lb) = 2.66			
Mo ($/lb) = 27.00			
Au ($/oz) = 564			
Ag ($/oz) = 10.40			
Case 3 (2 Year Staggered Pricing)	**13.9%**	**$976**	**$2,720**
Cu ($/lb) = 2.76 Yrs1&2, 1.50 Yrs 3-31			
Mo ($/lb) = 22.38 Yrs 1&2, 10.00 Yrs 3-31			
Au ($/oz) = 700 Yrs1&2, 550 Yrs 3-31			
Ag ($/oz) = 12.00 Yrs1&2, 10.00 Yrs 3-31			
Case 4 (7 Year Staggered Pricing)	**22.2%**	**$1,618**	**$3,550**
Cu ($/lb) = 2.76 Yrs1-2, 2.55 Yr3, 2.13 Yr4, 2.13 Yr5, 1.92 Yr6, 1.71 Yr7, 1.50 Yrs 8-31			
Mo ($/lb) = 22.38 Yrs 1-2, 20.32 Yr3, 18.25 Yr4, 16.19 Yr5, 14.13 Yr6, 12.06 Yr7,10.00 Yrs 8-31			
Au ($/oz) = 700 Yrs1&2, 675 Yr3, 650 Yr4, 625 Yr5, 600 Yr6, 575 r7,550 Yrs 8-31			
Ag ($/oz) = 12.00 Yrs1&2, , 11.67 Yr3, 11.33 Yr4, 11.00 Yr5, 10.67 Yr6, 10.33 Yr7,10.00 Yrs 8-31			

23.12 Marketing

This section was provided by Hugh Hamilton of H.M. Hamilton & Associates Inc.

23.12.1 Introduction

Copper Fox Metals (CFM) is considering putting into production its Schaft Creek copper-gold-molybdenum-silver property located in the Province of British Columbia. The principal value from the production will be the copper in the concentrates, but the contained gold, and molybdenum will also make significant contributions to the cash flow.

H.M.Hamilton & Associates (HMH) have been commissioned to provide a provisional marketing report. In developing this report, some components of previous projects completed by HMH (marketing studies and marketing communications) were reviewed. Qualities and production tonnages provided by CFM were used in accordance with the Scope below.

The prime objectives of this marketing review are to:

Provide:

a. Comments on the Copper market including a forward price forecast,
b. Marketability of the Copper concentrates,
c. Estimates of what the contract terms may be for the sale of these concentrates to custom Smelters,
d. Estimates of the transportation cost to possible markets and
e. A formatted Net Smelter Return (NSR) to the mine for the Copper concentrates.

Comments on the terms and markets are derived from HMH's knowledge of and involvement in the market and generally represent industry consensus.

The transportation costs from the mine to the smelters were reviewed in detail with the transporters and they are close estimations. The marketability comments are derived from a combination of HMH's market involvement and forecast production data supplied by CFM.

The Net Smelter Return does include transportation estimates, but not insurance or bonuses/charges for lot or delivery size.

All references herein are to US dollars, whether or not specifically stated.

This report is intended as a partial marketing review since smelters were not asked to generate a letter of interest/intent to purchase in order to establish the indicated terms.

23.12.2 Mine Production

According to the International Copper Study Group (ICSG), world copper mine production rose 34% in the 10-year period from 1996 to 2006. Copper mine production reached 14.9 million metric tonnes, or megatonnes (Mt), in 2005 from 11.1 Mt in 1996. Production of copper in concentrates rose by 26% while solvent extraction–electrowinning (SX-EW) production rose by 86%.

Chile continues to dominate mined copper production with a growing share of global output. During the past ten years, production in Chile rose 71% or 2.2 Mt. In 2005, Chile supplied almost 36% of global copper mine production. During the last ten-years supply between other copper producing countries became more evenly shared, on a closer hierarchical scale, thus exhibiting an even more distinct dominance by Chile.

During this period, US production fell by 41%, or 796,000 metric tonnes. (Figure 23.32)

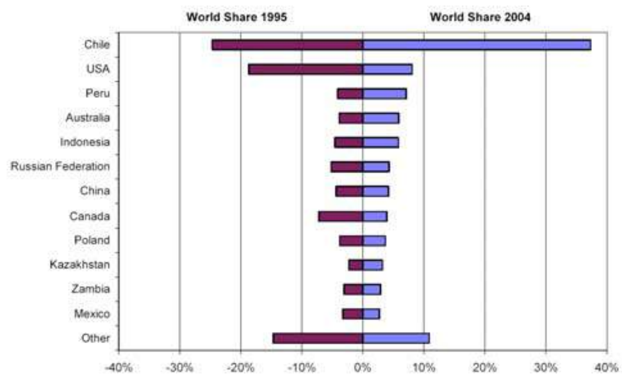


Figure 23.32 Copper Mined Production

Notable is the revival of production in Zambia and Congo; the entry of Argentina and Laos as copper-mine producing countries; and significant growth of production from expansions and new projects in Australia, Brazil, China, Indonesia, Kazakhstan, Myanmar, Peru and Russia that together added more that 2 Mt to world mine production. Apart from the United States, significant decreases in mine production during this period occurred in Canada, Philippines and South Africa.

23.12.2.1 *Refined Copper Production*

Over the last ten years, world refined production rose by 30% to 16.45 Mt from 12.68 Mt. Primary refined production rose 34% to 14.34 Mt, while secondary refined production increased 5% to 2.103 Mt from 2 Mt. The share of secondary production in total refined production decreased to 13% from 16%.

Chile, where production rose by 62% to 2.8 Mt, retained its position as the largest world producer; China's refined production more than doubled to 2.6 Mt. China’s refined copper capacity is growing rapidly. One source reports China’s copper refining capacity expanded by more than 70% during the past five years.

This along with plans for large capacity increases in the future, are believed to set China up to replace Chile as the world’s largest producer of refined metal.

The growth of China's ability to turn copper from its raw form into a metal for pipes, wires and electrical components is set to change the dynamics of the market. Contrary to this trend, copper concentrate imports into China were reported to be down on a year-on-year basis in the first six months of 2006.

Around the world, South Korea's production of refined copper doubled to more than 500,000 metric tonnes, while India experienced a ten-fold increase in production to become a major producer.

Indonesia, Laos, Myanmar and Thailand became new producers of refined copper, according to the ICSG. The United States, the leading world producer in 1996, saw its production decrease by 1.1 Mt during the ten-year period, and its world position as a producer of refined copper slip to fourth. Production declines also occurred in Canada, and France, while the United Kingdom stopped producing refined copper entirely.

23.12.3 Mine Capacity

The ICSG expects annual mine capacity over the 2005 to 2009 period to grow at an average rate of 4.3% per year to reach 19.6 Mt in 2009, an increase of about 3 metric tonnes (18%) from that in 2005. (Figure 23.33) Of the total increase, copper in concentrate production is expected to increase by 1.7 metric tonnes (3.1%/yr) and SX-EW production by 1.3 metric tonnes (9.3%/yr). Included in the projected mine growth is 1.1 metric tonnes of capacity that is still in the exploration and/or feasibility stage and has not been finally committed to development.

Projected World Capacities until 2009

000's metric tonnes Cu	*2005*	*2006*	*2007*	*2008*	*2009*	% change 2005-2009
SX-EW	3,117	3,283	3,705	4,141	4,437	42.3%
Concentrates	13,484	13,602	13,971	14,475	15,212	12.8%
Mines Total	16,601	16,884	17,676	18,616	19,649	18.4%
Smelters	16,244	16,602	16,990	17,217	17,476	7.6%
Electrolytic Refineries	16,425	16,731	17,176	17,473	17,828	8.5%
Refineries Total	20,268	20,711	21,584	22,324	22,980	13.4%
Year on Year Changes (tonnage)		*2005/06*	*2006/07*	*2007/08*	*2008/09*	*Accumulated 2005-2009*
SX-EW		166	423	436	296	1,320
Concentrates		118	370	504	737	1,728
Mines Total		283	792	940	1,033	3,048
Smelters		358	388	227	259	1,232
Electrolytic Refineries		306	445	297	355	1,403
Refineries Total		443	873	740	656	2,712

Figure 23.33 Projected World Capacities until 2009

23.12.4 Smelter Capacity

The projected average annual smelter capacity growth rate for the period 2005-2009 (1.8%/yr) is 1.3%/yr lower than the projected growth in concentrate capacity. During the first two years (2006-2007), however, the smelter growth rate (4.6%/yr) is projected to exceed the corresponding annual concentrate growth rate of 3.6%/yr. The situation will be reversed in 2008-2009, when concentrate capacity growth will exceed smelter capacity growth.

Assuming smelter capacity utilization rates rise from the current low level to their historical average, smelter capacity on average should be sufficient to treat any additional concentrate production. There could, however, be short-term shifts in the concentrate supply-demand balance owing to the unequal distribution of growth.

23.12.5 Demand

23.12.5.1 *Ten Year History*

During the past ten years refined copper usage increased by 31% to 16.51 Mt from 12.64 Mt. Asia, excluding Japan, is responsible for virtually all of the growth in world refined copper consumption in the past 20 years, driven by Korea and Taiwan, and more recently China.

China became the leading world copper user in 2002. China's rapid industrialization is creating massive new copper demand. Usage over the ten-year period increased by around 2.3 Mt (+182%), and China's share of total world usage rose from 10% to 22%. (ICSG) In the past six years, China's compound annual growth in copper demand was 14.5%. Expectations are this will not be maintained; long-term growth rates of greater than 5-6% are expected as China develops.

Complementing China, the rest of Asia – particularly India – is expected to remain a core component of future copper demand in the coming decade. Chinese demand also contributes to copper consumption growth in neighboring countries, most notably Taiwan, South Korea and Japan, as these nations consume copper to produce manufactured goods for export to China. This region is expected to be a principal engine of growth, resulting in higher average future capacity utilization rates and lower relative inventory levels, driving considerable production capacity expansions.

In the last ten years usage in the United States and Japan fell by 13% and 17%, respectively, while their world share decreased to 15% from 21%, and 7% from 12%, respectively. Their position as global users of copper fell to two and three, respectively, behind China. Russia's use increased by about 450,000 metric tonnes (270%) and India doubled its usage to 400,000 metric tonnes. The European Union-15 increased usage by 2%. (ICSG)

23.12.5.2 *Projected Future Demand*

AME and BHP Billiton each project a 3.5% mid-term (four year) growth rate in refined copper consumption. Global long-term copper consumption growth is expected to be in the region of 2.1% per annum.

- AME's analysis of the copper market forecasts global refined copper consumption growth towards the end of the decade to be at a significantly higher rate than the 2.1% per year long-term trend. AME forecasts world refined copper consumption will top 20 metric tonnes by 2010, which implies a healthy compound growth rate of over 3.5%.
- According to market analysts, until 2010 the average annual growth in refined copper demand will account for 631,000 tonnes of copper (3.5% per annum). (BHP Billiton)

Strong demand growth is expected in China, South Korea, India, and South East Asian nations. Japan's decade-long contraction is expected to reverse. Long-term growth will be powered by BRIC countries (Brazil, Russia, India and China), and the economies that support their development through the supply of manufactured goods. (Table 23.48)

Table 23.48
Copper Growth Consumption Scenario Million Metric Tonnes

	2002	2015	2025	2050
BRICs (Brazil, Russia, India & China)	3.5	18.6	23.6	37.2
% of 2002 world	23%	124%	158%	249%

Source: BHP Billiton, derived from Goldman Sachs, CRU, AKME, UN, BP et al, May 2006
www.bhpbilliton.com/bbContentRepository/Presentations/060327CWGML2006ConfMiami.pdf

23.12.6 Global Economic Growth

The lead indicators currently point to a slowing in OECD industrial growth and this clearly has implications for copper demand in the short-to-medium term. The U.S. National Association of Realtors report sales of previously owned homes dropped in August to the lowest level since early 2004. The visibility of a sharp and relatively broad-based US slowdown is poised to increase in coming months. Housing is set to collapse further, pulling down consumer spending growth. The only bright spot on the US domestic front is business investment, set to rise 5.5% in 2007

In the long term, the Organization for Economic Co-Operation and Development (OECD) forecasts OECD countries will realize significant net gains from the rapid economic development of non-OECD economies, in particular, the big five non-OECD countries (Brazil, China, Chile, India, and Russia). In coming decades, the interrelated factors of greater integration of the world's economies, improvements in human capital and technological change are expected to be key in providing the potential for continued growth in world prosperity. Advances in information and production technologies, developments in new materials and biotechnology and improvements in transportation have already begun to have significant effects and contributed to changes in the structure and distribution of international trade and investment.

The potential for closer integration amongst economies is likely to increase and, with falling communications costs, new technologies will become more accessible to developing countries. Fully realizing the potential benefits of these developments over the next 25 years is expected to depend in large part on government policies.

If governments make only slow progress in liberalization of international trade and finance, fiscal consolidation, or reforms to product and labour markets, or a "business-as-usual" scenario, OECD and non-OECD countries might experience a productivity performance no better than that of the past 25 years, implying average annual economic growth rates falling to 2 per cent or below in the OECD and 4 per cent in the non-OECD. Against a background in which the non-OECD share of world population rises from 81 per cent to about 85 per cent, its share of world GDP would increase from 40 per cent to around 50 per cent in 1992 PPP terms. However, virtually all this increase would be expected to accrue to Asia, with the shares for Latin America, Russia and Africa stagnating or decreasing.

Much worse scenarios could be envisaged, for example, resulting from a reversal of the current process of globalization or disruption resulting from military conflicts.

23.12.7 Supply - Demand Balance

23.12.7.1 *Overview*

Generally, copper demand and supply have maintained a reasonable balance for the past twenty years with brief periods of surplus or deficit. (Figure 23.34) The past ten years (1997-2006) have exhibited the widest gaps between supply and demand. Table 23.49 presents a summary of industry expectations for the mid-term (three years). Industry analysts expected copper to be in deficit in 2006, moving to a small surplus in 2007. The overall expectation for 2007 is for an average of over 4% growth in global concentrates translating to an average 198,000 tonne surplus in 2007. With a slowing US housing market and the expectation China's State Reserve Bureau will continue to release additional metal, the copper market is expected to turn to surplus in 2007. [1]

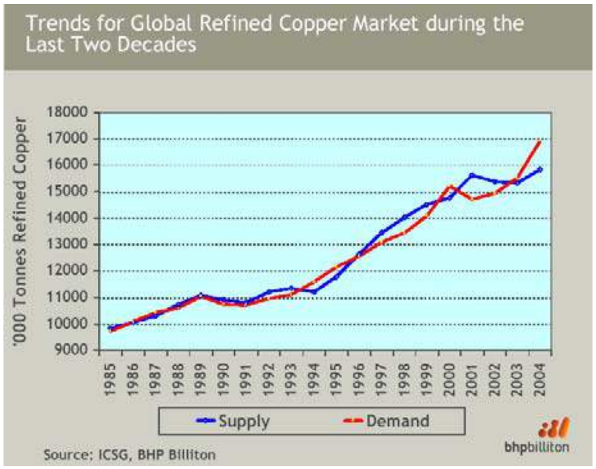


Figure 23.34 Trends for Global Refined Copper during the Last Two Decades

Table 23.49 Canaccord Adams & Consensus Copper Forecasts ('000 tonnes)						
	2005A		**2006E**		**2007E**	
	surplus/ (deficit)	global cons. growth	surplus/ (deficit)	global cons growth	surplus/ (deficit)	global cons. growth
Brook Hunt (Oct-06)	-292	-0.6%	-66	4.7%	357	4.5%
AME (Oct-06)	-32	-1.3%	-289	5.0%	-63	2.3%
Metal Bulletin (western world, Sep-06)	-38	N.A.	-23	N.A.	395	N.A.
ABARE (Oct-06)	-152	-0.1%	51	2.2%	152	0.0%
ICSG (Oct-06)	-102	-0.7%	239	3.3%	176	4.2%
Phelps Dodge (Jul-06)	-	-	"modest deficit"	5.0%	-	-
Teck Cominco (Apr-06)	-380	0.2%	-36	6.0%	306	4.5%
Cohilco, Chile state copper commission (Apr-06)	-	-	-209	5.1%	39	4.9%
Falconbridge (Jul-06)	-	-	-102		"small deficit"	
Canaccord Adams	-170	-0.4%	-49	4.4%	220	4.8%
Average forecast surplus/(deficit)	**-167**		**-54**		**198**	**4.2%**
Canaccord Adams Metals and Mining, Q4/06 Commodity Price Review, Daily Letter, October 12, 2006						

23.12.7.2 *Mine Supply Requirements*

Westhouse Securities published the following supply related forecasts in a presentation titled *Copper Market, An Overview*, December 2005.

Based on demand projections, the Mining Industry needs to bring on stream the equivalent of:

- 2 major mines (200,000 tpa of contained copper mine equivalent) each year until 2008
- 4 major mines each year between 2009 – 2012
- 5 major mines each year between 2013-2016

42 new major mine equivalents needed on stream by 2016 to bridge the gap between copper mine supply and demand

- “possible” and “probable” brownfields expansions could account for 1/3rd ONLY of this gap
- 28 new major greenfield project developments are needed to bridge the gap

If these supply requirements are not achieved, there will be a significant upward pressure on the long-term price as a result of the supply demand imbalance.

23.12.7.3 *Inventory Effect*

Total world copper stocks gradually increased from 820,000 metric tonnes in 1996 to a peak of 2.050 Mt in 2002. Stocks began to decline in 2003, and by year-end 2005 had fallen by around 1.2 Mt to 850,000 metric tonnes. The International Monetary Fund (IMF) predicts global inventories will remain at historically low levels, believing the introduction of new capacity has been delayed because of high energy and equipment costs, and labour unrest and shortages. Looking forward, the IMF analysis predicts, “despite an expected capacity increase in metals this year (2006), the tight market situation will probably continue into late 2007–early 2008, until sufficient new capacity comes into operation.”

23.12.7.4 *Long Term*

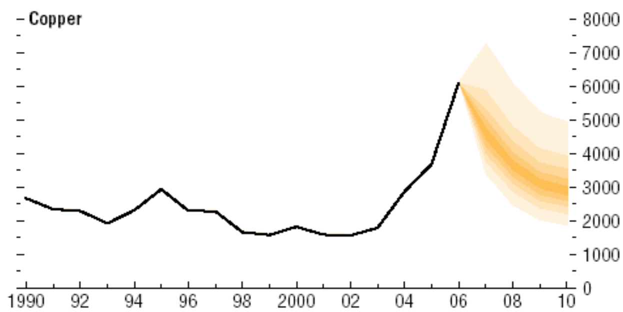
In the long-term it is thought many conditions, unique to the current industry, will work to keep copper supply and demand in a respectable balance. These include most notably consolidation of supply, growth of global GDP and substitution in periods of high prices. According to the IMF, “Looking ahead, rapid industrial output growth, construction activity, and infrastructure needs could sustain the growth of demand of emerging markets for metals at high rates in the medium term”. However, IMF economists warned “some of the current demand strength could be temporary especially as the Chinese government is aiming at a rebalancing of growth from investment to consumption over the medium term.”

23.12.8 Price Forecast

23.12.8.1 *Mid-Term (2007- 2010)*

Analysis of the copper market suggests the price of copper is above sustainable levels under various assumptions about global growth, additions to productive capacity, and price responsiveness of supply and demand.

Looking forward, metals prices are expected to retreat from their current highs. Demand and supply analysis suggests copper prices should moderate by the end of the decade. The price of copper is forecast to fall relatively more than the price of replacement potential aluminum. This is consistent with prices in the futures markets and the fact that the market-price-to-cost ratio is currently much higher for copper than for aluminum.



Sources: IMF, Commodity Price System database; and IMF staff estimates.
[1]The fan chart corresponds to a 95 percent probability band for future metal prices. Each shade represents a 10 percent likelihood with the exception of the central band (represented by the darkest shade in the fan), which represents a 15 percent likelihood. See Appendix 5.1 for details.

Figure 23.35 Price of Copper

The estimated price range is very wide, reflecting uncertainties about global growth, capacity expansion in the metal industry, and the econometric model. (Figure 23.35) In the baseline scenario, the real price of copper is forecast to decline by 57 percent by 2010. The price decline is generated by a combination of factors:

1. recent accumulated price increases will have some dampening impact on demand;
2. considerable supply expansion is projected in the next five years; and
3. some additional supply is expected to come on stream as the current metals prices are higher than projections.

The IMF forecasts copper prices should average $1.50 per pound by 2010, while Chilean copper experts are predicting a price of $1.21/lb. Table 23.50 contains mid-term price decline expectations leading to the 2010 predicted price levels.

Table 23.50 Canaccord Adams' and First Call consensus Copper Forecast				
	2006E	2007E	2008E	2009E
Canaccord Adams forecast US cents/lb	311	303	230	175
First Call consensus forecast US cents/lb	292	251	220	175

Source: Canaccord Adams Metals and Mining, Q4/06 Commodity Price Review, Daily Letter, October 12, 2006

23.12.8.2 *Long Term*

Considering price developments beyond 2010, the key issue is whether metals supply will be able to meet rising demand in an environment of continued strong growth. While investment gestation lags can reach three to five years in the sector (or more in case of greenfield investments), they are generally shorter than in the oil industry. These supply-side factors tilt the long-term price risks for metals to the downside and clearly differentiate the metals sector from the oil market where prices are expected to remain high in the foreseeable future.

The historical experience is metals prices tend to converge to production costs in the medium term. The current prices are well above production costs (the ratios of market prices to costs are 1½-2¾ for the main metals). For copper, futures markets similarly predict a gradual decline in the price during the next five years. (Figure 23.36)

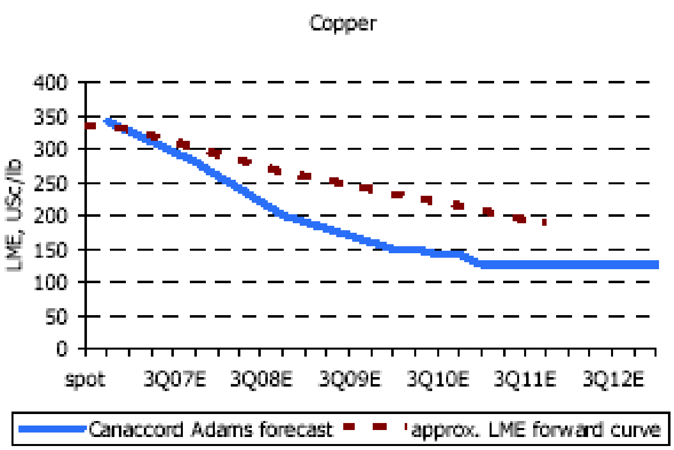


Figure 23.36 Copper

However, the futures markets also reflect an increase in production costs indicating future prices will not return to historic ninety cents per pound levels. In addition, if the mine supply requirements referred to earlier are not achieved, there will be significant upward pressure on the price due to the shortfall in supply.

Table 23.51 presents Canaccord Adams' copper price expectations for the mid-and long-term. Essentially, the industry expects long-term copper prices to range from a low of $1.21 to $1.50 per pound.

Table 23.51 Canaccord Adams' Copper price forecast, US cents/lb				
	2006E	**2007E**	**2008E**	**Long Term**
New forecast	3.11	3.03	2.30	1.25
Old forecast	2.96	2.83	2.10	1.15
Change	5%	7%	10%	9%
Source: Canaccord Adams Estimates, October 12, 2006				

23.12.9 Quality and Quantity

23.12.9.1 *QUALITY*

In the absence of full concentrate assays from the Schaft Creek project ore body, (presently being developed) the following analysis will be used as reference assays. (Table 23.48)

Table 23.52 Schaft Creek Concentrate Forecast Assays	
Element	**Assay**
Cu	25.00%
Au	15 g/t
Ag	95 g/t
No other deleterious elements	

Because of the incomplete assays reported for the concentrates, smelters were only able to provide a preliminary interest in the product. CFM will need a more complete assay of the concentrates in order to obtain a Letter of Interest or a Letter of Intent from any smelter.

It is assumed that the moisture content will range from 7 % to 9 %.

23.12.9.2 *QUANTITIES*

Production is forecast to commence in 2011. The annual production rates are expected to be in the order of 300,000 wet metric tonnes (wmt) of Copper concentrates plus 10,000 wmt of Molybdenum concentrates assaying 50% Molybdenum.

23.12.10 Marketability

23.12.10.1*General*

Generally the Schaft Creek concentrates would be readily marketed. The annual volume would dictate the placement into four to five different smelters. (60,000 – 100,000 metric tonnes to each location). If the copper grade of the concentrates was above 30%, the concentrates would command a premium position in the market place since it would allow the receiving smelters to use the Schaft Creek concentrates as a diluent and thereby acquire additional lower grade materials.

23.12.10.2 *Details*

I. The higher copper content would allow the smelters to blend with lower grade concentrates.
II. If the precious metals are shown to be high, this would be to the liking of the Japanese smelters.
III. Placing the CFM concentrates into China would be more difficult since the majority of Chinese copper smelters do not have high recoveries of the contained precious metals. This situation is gradually being remedied and by 2008 some of these smelters may have improved their precious metal recoveries
IV. At present, while India is looking for copper units, the majority of their smelters do not want high precious metal concentrates.
V. Based on the concentrate assays presently available for review, a low level of penalties are anticipated,

It is assumed that the moisture content of the concentrates will be below the Transportable Moisture Limit (TML) required for ocean vessel transport and yet to be established for the Schaft Creek concentrates. It should be noted that this TML may be below the trigger point for a smelter moisture penalty.

23.12.11 Smelter Terms

23.12.11.1*Overview*

In the absence of letters of interest or letters of intent from potential smelters to further define the potential terms for the concentrates, HMH have generated assumptions for smelter terms with respect to treatment charges, penalties. These were reviewed in the light of the current market, as well as historic and future expected trends. In addition, there were discussions with smelters in order to determine current trends in terms and the long-term outlook on capacity availability and the minor element impact.

It must be recognized that smelters in different market areas may use different formulae with respect to metal accountability and charges and this is reflected in the presented terms.

It is also expected that, on final negotiations for concentrate contracts, the terms could be expected to vary due to shifts in the world market conditions.

Samples might be requested for testing prior to acceptance.

23.12.11.2 *Impurity Thresholds for Processing Smelters*

While CFM have assays on the various ores plus some analyses on probable concentrate, it is difficult to fully translate these analyses into complete concentrate qualities, the following are some of the penalty thresholds usually encountered in copper concentrate terms. (Table 23.49)

Table 23.53
Penalty Thresholds for Copper Concentrates

Element	Penalty Threshold
As	0.1%
Sb	0.1%
Zn	2.5%
Pb	0.5%
Zn + Pb	2.5%
Bi	0.1%
Ni + Co	0.5%
Cl	0.05%
F	0.03%
Al_2O_3	3.0%
Se	0.03%
H_2O	8-10%
Hg	20ppm%

With possible copper smelter outlets in Japan, Canada and Europe, the terms for Dowa in Japan have been used as a base case scenario.

23.12.11.3 *Copper Concentrate Terms*

Table 23.54
Processing Smelter Copper Concentrate Terms

Component Metal	Grade	Payable Content	Deductions	Refining Charge(1)	Treatment Charge(2)	Transportation Charge(3)
Copper	< 30%	96.5%	Minimum 1.0%	US $0.06 per payable lb	US $60 per dmt of Concentrate	US $71.0654 per dmt of Concentrate
	30-35%	96.65%	Minimum 1.1%			
	> 35%	96.7%	Minimum 1.1%			
Silver	< 30 g/t	0%	1.0 ozt / dmt of Concentrate	US $0.35 per payable ozt		
	> 30 g/t	90%				
Gold	1-3 g/t	90%	0.04 ozt / dmt of Concentrate	US $5.00 per payable ozt		
	3-5 g/t	94%				
	5-10 g/t	95%				
	10-15 g/t	96%				
	15-20 g/t	97%				
	> 20 g/t	97.5%				

Notes: Figures in this table were used in the financial model contained within this report. Due to more recent information, HMH's Marketing Report dated December 2006 had the following differences from the table above:

(1) RC for Copper of US $0.09 per payable pound plus a price participation of 10% in access of the RC to a maximum of US $0.10 per payable pound. RC for Gold of US $6.00 per payable ozt.
(2) TC of US $90 per dmt of concentrate CIF Free Out Japan
(3) Transportation charge of US $77.46 per dmt of concentrate

23.12.11.4 *Molybdenum Concentrate Terms*

Table 23.55
Processing Smelter Molybdenum Concentrate Terms

Component Metal	Grade	Payable Content	Deductions / Discount	Refining Charge(1)	Treatment Charge(1)	Transportation Charge(2)
Molybdenum	Any	99%	16% Below Market Price of Metal	US $0.75 per payable lb		US $52.7245 per dmt of Concentrate

Notes: Figures in this table were used in the financial model contained within this report. Due to more recent information, HMH's Marketing Report dated December 2006 had the following differences from the table above:

(1) RC/TC for Molybdenum were not supplied
(2) The transportation charge was assumed to be the same as for copper concentrate plus a US $25 per dmt for one metric tonne supersacks.

23.12.11.5 *Transportation*

The copper concentrate will be delivered to Asian ports, most likely to Japan, South Korea or northern China; while, the molybdenum concentrate will be delivered to South American ports, most likely Chile.

For these movements there will be three elements of cost, i.e. a truck haul from the mine to the export port of Stewart B.C., storage and vessel loading at Stewart, and ocean transportation in bulk carrier vessels to the receiving ports. All of the receiving smelters bear the cost of unloading at the receiving ports plus any inland transportation costs to the smelter. The port of Stewart has a dock with sufficient draft to receive and load bulk carriers of up to 40,000 tonnes of carrying capacity. It is presently being used to load materials generated by the Huckleberry and Eskay Creek mines. The owner of the dock and the storage sheds has expressed a desire to handle additional concentrates. Despite its northern location, Stewart is an ice free port and operates year around. The distance from the Schaft Creek property to the port of Stewart is 278 kilometres.

For another northern project, an average indication of the trucking rate from the property to the port was $0.09 Cdn per kilometre per wmt. Assuming an 8% moisture content and an 88 cent Canadian dollar, this translates to $23.93 U.S. per dmt. Storage and loading at Stewart has been offered by Stewart Bulk Terminals at $10.00 Cdn per wmt. This is line with the charges being paid by others. The average of two Vancouver-based shipping brokers estimates for long-term rates for shipment of concentrates from Stewart to Asian ports (in US dollars) are:

To Japanese ports	$43.96 per dmt
To South Korean ports	$45.05 per dmt
To North Chinese ports	$47.25 per dmt

These take into account that tugs probably will continue to be required to position bulk carriers at the dock in Stewart and that these tugs will be brought in from Prince Rupert. The rates assume a loading of 10,000 wmt each.

The totals of these projected movements are, in US$/ dry metric tonne:

	To Japan	**To South Korea**	**To North China**
Truck	$23.93	$23.93	$23.93
Storage and Loading	$9.57	$9.57	$9.57
Ocean Freight	$43.96	$45.05	$47.25
Totals	**$77.46**	**$78.55**	**$80.75**

23.12.12 Net Smelter Returns

The Net Smelter Returns (NSR) were established on the basis of the following:

i) Average prices (In US funds):
   a. Copper $2755/mt ($1.25/lb)
   b. Silver $7.00/ozt
   c. Gold $480.00/ozt
   d. Molybdenum $8.00/lb
ii) A US/CDN exchange rate of 0.88
iii) Average concentrate grades as per CFM
iv) The in transit financing assumes no provisional payments until vessel loading.
v) Freight, Representation and Payment Financing is included in the calculations.

The transportation costs were developed on the following basis:

- Mine direct to Stewart B.C. $0.09 per kilometre per wmt.
- Load-out facility costs at $10.00 Cdn/wmt.
- Estimated ocean freight rates to Japan at $43.96 US/dmt
- For the Molybdenum concentrates the transportation is assumed to be the same as for Copper concentrates plus a $25 US/dmt charge for one metric tonne bags.
- An overall handling loss of 0.3% is assumed for Copper concentrates.
- No capital costs were added to any of the rates.

The following table shows the metal prices used in the financial model contained within this report:

Table 23.56
Financial Model Base-Case & 3yr Trailing Average Metal Pricing

Metal	Base-Case Price	3-Year Trailing Average Price
Copper	$1.50 / lb	$ / lb
Silver	$10.00 / ozt	$ / ozt
Gold	$550 / ozt	$ / ozt
Molybdenum	$10.00 / lb	$ / lb

Again, once a full suite of assays is established and the freight rates are more definitive, the detailed estimate of the Net Smelter Returns can be refined.

The NSR and freight calculations are in US$ per dry metric tonne (dmt) since all the smelter contracts will be written in US$/dmt.

Appendix A Net Smelter Return (US per dmt) – Molybdenum Concentrates

Contained Molybdenum @ 50%Mo	1102.3 lbs/dmt
Content Deduction @ 1%	11.0 lbs/dmt
Net Payable	1091 3 lbs/dmt
Metal Price	$8.00 US/lb
Price Discount 16% * Price	$1.28 US/lb
Payable Metal Price	$6.72 US/lb
Smelter Payment	$7333.54/dmt
Less Freight	$23.93/dmt
Truck	$ 9.57/dmt
Port	$43.96/dmt
Ocean	$ 0.27/dmt
Representation	$25.00/dmt
Bags	$77.73/dmt
Total	$7333.54/dmt
Net Smelter Return	$7255.81 / dmt

Note: This NSR does not assume containerization for the bags.

NET SMELTER RETURN - SCHAFT CREEK COPPER CONCENTRATES
DETAILS - JAPAN - STEWART

				US$/DMT	US$/YR	C$/YR
Gross Value				**US$/DMT**	**US$/YR**	**C$/YR**
Copper				$688.93	**$190,143,300**	$216,071,932
Gold				$231.50	**$63,895,104**	$72,608,073
Silver				$21.38	**$5,901,101**	$6,705,796
Total gross value		per dmt		**$941.81**	**$259,939,505**	$295,385,801
Payable Metal Deductions						
Copper	lbs	3.5 min 1.0		22.04		
gold	g/mt			0.45		
silver	g/mt		10%	9.50		
Payable Deductions-$Value						
Copper				($27.55)	**($7,603,593)**	($8,640,447)
Gold				($6.95)	**($1,916,853)**	($2,178,242)
Silver				($2.14)	**($590,110)**	($670,580)
Value- Total Deductions				**($36.63)**	**($10,110,556)**	($11,489,268)
Payable Value						
Copper				$661.38	**$182,539,707**	$207,431,485
Gold				$224.56	**$61,978,251**	$70,429,831
Silver				$19.24	**$5,310,991**	$6,035,217
Net Payable Value				**$905.18**	**$249,828,949**	$283,896,532
Charges and Penalties						
Refining charge Gold				($2.81)	**($3,199,964)**	($3,636,323)
Refining charge Silver				($0.96)	**($1,096,835)**	($1,246,403)
Refining charge Copper				($47.62)	**($13,143,217)**	($14,935,473)
Price Participation Cu Refining				($18.52)	**($5,110,754)**	($5,807,675)
Treatment Charge		Base charge		($90.00)	**($24,840,000)**	($28,227,273)
Penalties				$0.00		
Total Penalties+Charges				**($159.91)**	**($44,134,248)**	($50,152,555)
Total GSR value				**$745.27**	**$205,694,700**	$233,743,977
Other Costs:						
Truck Freight				($23.93)	**($6,606,000)**	($7,506,818)
Terminal storage and Ship loading				($9.57)	**($2,640,000)**	($3,000,000)
Ocean Freight				($43.96)	**($12,133,500)**	($13,788,068)
Loss factor (0.30%)				($2.24)	**($617,084)**	($701,232)
Representation				$(0.27)	**($75,000)**	($85,227)
Total Other Costs				**($79.97)**	**($22,071,584)**	($25,081,346)
Net value FOB end mine road				**$665.30**	**$199,590,344**	$226,807,209
In Transit Financing				($5.79)	**($1,597,656)**	($1,815,518)
Net Smelter Return				**$659.51**	**$182,025,460**	$206,847,114

NET SMELTER RETURN - SCHAFT CREEK COPPER CONCENTRATES
DETAILS - JAPAN - STEWART

Assumptions:

Grades		opt	
Gold g/dmt		0.48	15.00
Silver g/dmt		3.05	95.00
Lead %			0
Zinc %			0
Copper %			25
Iron %			15
Antimony %			0
Arsenic %			0
Silica %			0
Cadmium %			0
Alumina %			0
Nickel	ppm		0
Bismuth %			0
Mercury ppm			0
Sulphur %			16
H2O			8

Freight to Japan

Vol. Wmt		\$/ dmt	\$/ wmt
300000	Truck	\$ 23.93	\$22.02
	Port	\$ 9.57	\$8.80
	Ocean	\$ 43.96	\$40.45
	Rep'n	\$ 0.27	\$ 0.25
	Total	\$ 77.73	

Exchange Rate US\$/C\$

0.88

Payment delay cost

	% amount	Days after production
provisional	85	30
final	15	100

Interest rate % 7

Price

Copper	MT/ lb	\$2,755.70	\$1.2500
Gold	per oz	\$480.00	
Silver	per oz	\$7.00	

Treatment Charge

Base charge \$90.00

Refining charges			**Basis**
gold	per oz	\$6.00	
silver	per oz	\$0.35	
copper	per lb	\$0.09	90c+10% to max 10c

Penalties

Lead	over %	0.5
	amount	\$2.50
	per %	1
Arsenic	over %	0.1
	amount	\$3.00
	per %	0.1
Zinc	over %	2.5
	amount	\$3.00
	per%	1
Bismuth	over %	0.1
	amount	\$2.00
	per %	0.01
Antimony	over %	0.1
	amount	\$2.00
	per %	0.1

24.0 Illustrations