Copper Fox Metals, Inc.
Canadian National Instrument 43-101

Amended Technical Report:
Preliminary Feasibility Study on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada

Effective Date: September 15, 2008

Qualified Persons
Matt Bender, P.E.
Keith McCandlish, P.Geo., P.Geol.

Prepared By:
Samuel Engineering, Inc.
8450 E. Crescent Parkway, Suite 200
Greenwood Village, CO 80111
303.714.4840

Samuel Engineering, Inc.
We Provide Solutions

Table of Contents

List of Tables

List of Figures

3.0 Summary

3.1 Introduction

This Preliminary Feasibility Study (PFS) builds upon an earlier Preliminary Economic Assessment (PEA) commissioned by Copper Fox Metals Inc. (Copper Fox) to investigate the viability of developing the Schaft Creek project. Schaft Creek is a large porphyry copper, molybdenum, gold and silver deposit located on the eastern edge of the Coastal Mountain Range in north central British Columbia, Canada. This PFS focused on the design of a 100,000 ore tonnes per day (tpd) open pit operation producing on average 282,882 tonnes of high grade copper, gold and silver concentrate per year, 10,262 tonnes of high grade molybdenum concentrate per year, 35.7 million tonnes of tailings material per year and 68.6 million tonnes of waste rock per year for an expected mine life of 23 years. The project capital expenditure is projected to be US$2.95 billion and is expected to yield a before tax internal rate of return (IRR) of 18.6%. All monetary amounts presented in this report are in U.S. dollars unless specified otherwise.

This PFS is a comprehensive study of the viability of the Schaft Creek mineral project that has advanced to a stage where the mining method and pit configuration has been established and an effective method of mineral processing has been determined. In addition the study includes a financial analysis based on reasonable assumptions of technical, engineering, legal, operating, economic, social, and environmental factors and the evaluation of other relevant factors which are sufficient for a qualified person, acting reasonably, to determine if all or part of the mineral resource may be classified as a mineral reserve.

3.2 Property Description

3.2.1 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 on the eastern edge of the Coastal Mountain Range in north central 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. Figure 3.1 below shows the general project location.

Figure 3.1 Project Location

3.2.2 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.

An exploration camp and drill roads have been established within the immediate project area and total approximately 15 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 exploration camp was again expanded during the 2007 exploration program to accommodate 80+ personnel. A 750 m long gravel airstrip oriented in a general north-south direction is established immediately west of the camp, adjacent to the eastern bank of Schaft Creek.

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.

3.2.3 Geohazards

The Schaft Creek project access route and mine-site areas are located within the Mess Creek Watershed, which drains an area of 2,306 km^2 and is a major tributary of the Stikine River. The mine-site area and Tailings Storage Facility (TSF) site options are located in upper Schaft and Hickman Creeks which are tributaries to Mess Creek.

Due to the location of the Schaft Creek project, consideration of geohazards is particularly important. For this reason, specialists were retained to investigate potential geohazards to support the Tailings Options Study, the selection of the most appropriate Site Access Road and other site related issues. Geohazards were evaluated and classified for each area investigated with no project fatal flaws identified.

3.2.4 Mineral Tenure

The Schaft Creek property is composed of two claim blocks owned by Teck Cominco Ltd. The North Block is composed of 36 contiguous claims and covers an area of 19,560 hectares. The Schaft Creek deposit is located on the south boundary of claim 514603 and on the north boundary of claim 514637. The South Block is located about 600 m to the south and is composed of four claims that cover 1,358 hectares.

About 6 km further south, Copper Fox Metals Inc. owns nine claims along Mess Creek covering an area of about 3,925 hectares, see Figure 3.2.

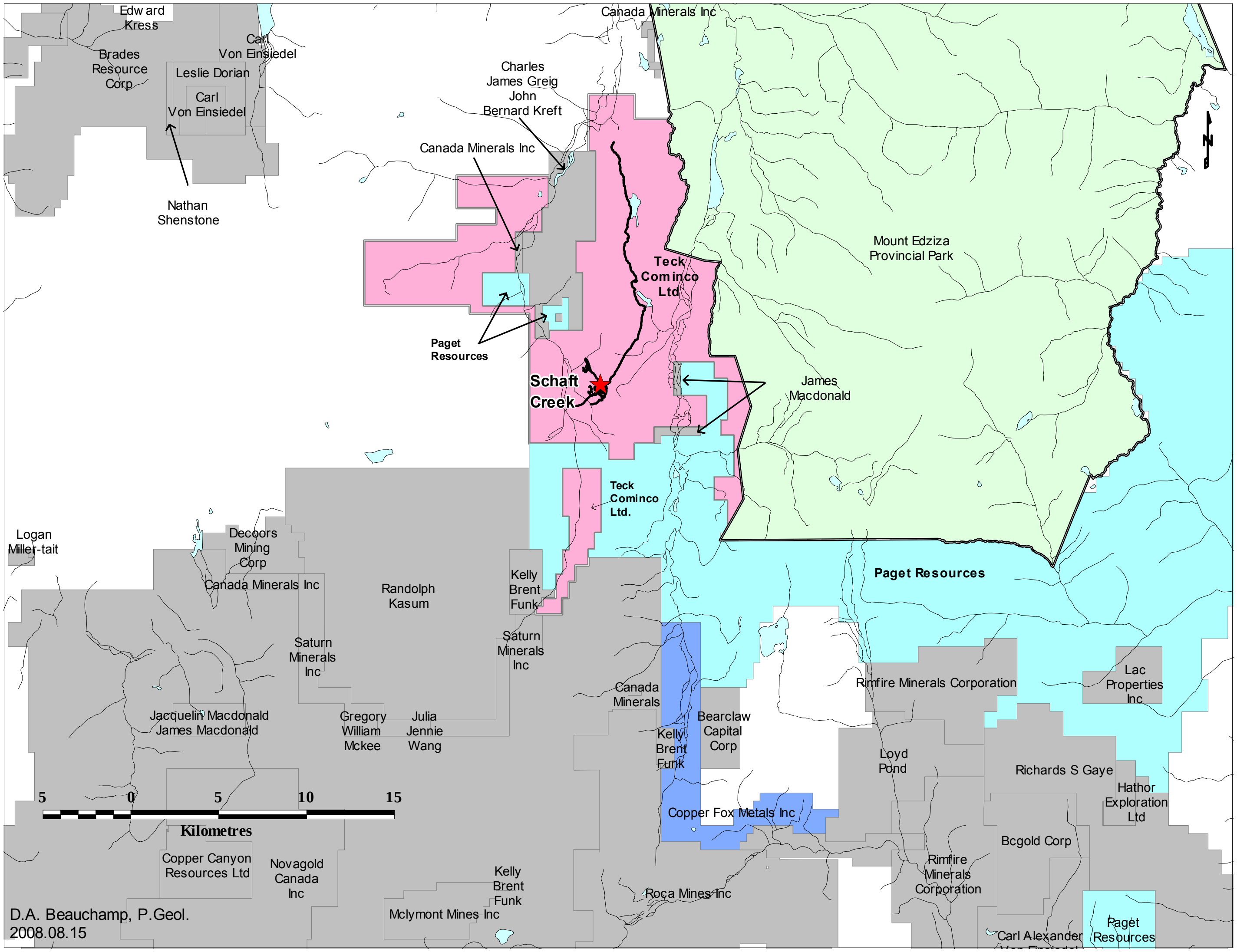


Figure 3.2 Schaft Creek Project Claims

3.2.5 History and Exploration Activity

Schaft Creek was the subject of intense and extensive exploration since mineralization was first discovered in 1957. The culmination of this exploration lead Teck Corp. to commission a PFS 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 in 2002, acquired the right to secure a significant ownership of the property 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 exploration program and subsequent programs in 2006, 2007 and 2008. Camp was put on winter maintenance on October 21, 2008.

The history of the property is summarized below.

- 1957, discoveries nearby spurred exploration northward into 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 feet of hand trenching;
- 1965, mapping, IP survey and 3 holes 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 property and carried out an extensive exploration program. A D6C bulldozer was 'walked in' from Telegraph Creek during the early spring, the airstrip was extended to 4,000 ft and a permanent camp erected. There were 24 holes drilled totaling 10,939 ft (3,334.2 m);
- 1967, Asarco drilled two holes totaling 1,001 ft (305.1 m). Paramount Drilling Limited drilled one hole to 501 ft (152.7 m);
- 1968, Asarco drops option and Hecla Mining acquires properties, airstrip extended to 5,280 ft;
- 1968, Hecla drills 9 holes, totaling 13,095 ft (3,991.3 m);
- 1969, Hecla drills 9 holes, totaling 15,501 ft (4,724.7 m);
- 1970, Hecla drills 26 holes, totaling 32,575 ft (9,928.8 m);
- 1971, Hecla drills 25 holes, totaling 22,053 ft (6,721.7 m);
- 1972, Hecla drills 10 holes totaling 8,950 ft (2,727.9 m);
- 1974, Hecla drills 6 holes totaling 7,062.5 ft (2,152.6 m);
- 1977, Hecla drills 1 hole totaling 2,113 ft (644 m);
- Reserve of 505 Mt, 0.38% Cu, 0.039% MoS_2, delineated;
- Between 1978 and 1979 Hecla Mining forfeits option and Teck Corp. acquires property;
- 1980, Teck Corp. drilled 47,452 ft (14,463.3 m) in 45 holes, between mid-May to mid-November, drill sites prepared with a D6 Caterpillar bulldozer. Assaying of core on 10-foot sample intervals, by Afton Mines Ltd. in Kamloops;
- 1981, between June and September, Teck Corp. drilled 33,315 ft (10,154.4 m), in 74 holes, and 3,503 feet of condemnation drilling for tailings pond and mill sites;
- Resource expanded to 1 billion t, 0.30% Cu, 0.034% MoS_2;
- 2002, Mr. G. Salazar acquires the right to secure a significant ownership of the property and subsequently incorporated it into the holdings of Copper Fox Metals;
- 2005, Copper Fox drills 15 holes totaling 10,367 ft (3,160 m);

- 2006, Copper Fox drills 42 holes totaling 29,547.2 ft (9,006 m);
- 2007, Copper Fox drills 42 holes totaling 20,684 ft (6,304.5 m);
- 2008, Copper Fox drills 48 holes totaling 22,822 ft (6,958 m).

Total property drilling up to October 2008 is 280,041 feet, in 382 holes.

3.3 Geology

3.3.1 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.

3.3.2 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.

3.4 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.

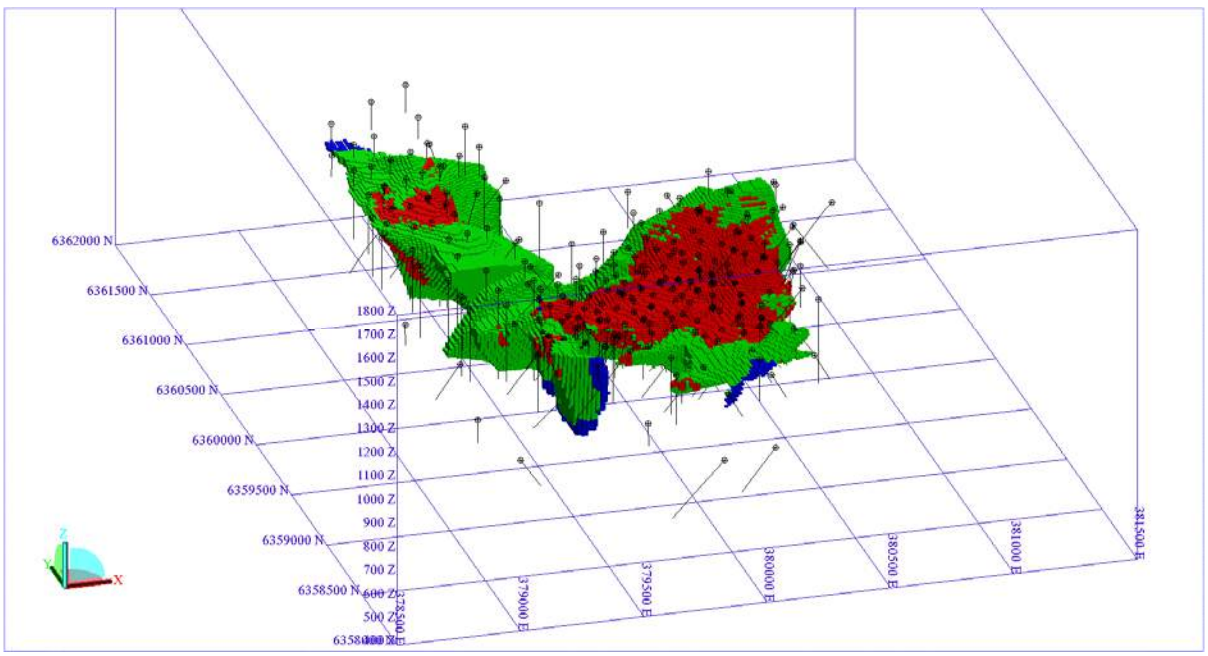


Figure 3.3 Resource Classifications of the Three Main Zones

Table 3.1 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.

Figure 3.3 shows the relative locations of the Measured, Indicated, and Inferred resources within the deposit.

3.5 Reserves

The ore reserves are listed below. Reserves are based on the following mining parameters:

- 10% mining dilution applied at the contact between ore and waste;
- Dilution grades are estimated at 4.63 $/t NSR, 0.076% Cu, 0.088 g/t Au, 1.76 g/t Ag and 0.005% Mo representing the average grade of material below the incremental waste/ore cut-off grade;
- 5% mining loss;
- Waste/Ore Cut-off grade of $5.05/t Net Smelter Return (NSR).

Table 3.2
Proven and Probable Reserves at Shaft Creek

Reserve Category	ROM ORE (Mt)	ROM Diluted Grades				
		NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
Proven	411.1	16.6	0.316	0.236	1.722	0.019
Probable	409.9	15.2	0.283	0.186	1.798	0.020
Total	**821.1**	**15.9**	**0.299**	**0.211**	**1.760**	**0.020**

3.6 Mining Plan

The mine planning work is based on the resource model provided by Associated Geosciences Ltd. (AGL). The 3-dimensional block model (3DBM) from AGL is converted and subsequent mine planning for the Schaft Creek mineral property is based on work done with MineSight®, including 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 parameters, mining cost data derived from supplier estimates and data from other projects in the local area, recommended pit slope angles and anticipated project metallurgical recoveries, plant costs and throughput rates.

The Schaft Creek deposits are to be mined with large truck and shovel operations and an ore mining rate of 100,000 tpd feeding a conventional copper flotation 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 run-of-mine (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 parameters used in the reserves calculations are also a NI 43-101 reporting requirement.

The 3DBM for Schaft Creek that was 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 cut-off grades will result in fewer contiguous blocks and more isolated blocks. Conversely, lower cut-off grades will merge more of the indicated isolated blocks into close-by contiguous blocks.

Mining operations will use blasthole samples on 6 to 8 m spacing to determine the cut-off 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 25 m by 25 m by 15 m blocks for this stage of long-range planning. Each block represents 25,031 t which is 4 to 5 hr of digging for the shovels and the plant feed will be approximately 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 reserve will be calculated from the resource model, within an economic pit limit using the applicable mining recovery and dilution parameters. The mining recovery and dilution parameters, in effect, convert the in-place "pit delineated resource" to ROM reserve 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 cashflow models. Therefore the tonnages 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 Parameters 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 PFS, an allowance has been made for a mining dilution of 10% applied at the contact between ore and waste dilution and a 5% 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 cut-off of $5.05/t. The dilution grade was estimated at 4.63 $/t NSR, 0.076% Cu, 0.088 g/t Au, 1.76 g/t Ag and 0.005% Mo representing the average grade of material below the incremental cut-off grade.

3.6.1 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 to different destinations (mill, stockpiles and waste dumps) by period.

Full production mill feed is expected to commence in 2014. The production schedule uses 'PP' as pre-production, "Year1" as 2013, "Year2" as 2014, etc.

3.6.1.1 *Production Schedule Results*

A summary of the production schedule is shown in Table 3.3.

Table 3.3
Summarized Production Schedule

Material to be Mined	Units	Time (Years)						
		PP	1 to 5	6 to 10	11 to 15	16 to 20	21 to EOM	LOM
Ore Mined To Mill	kt	-	180,050	180,050	180,050	130,575	88,431	759,156
Cu	%	-	0.332	0.323	0.291	0.286	0.326	0.309
Au	g/t	-	0.237	0.235	0.181	0.177	0.267	0.216
Ag	g/t	-	1.59	1.78	1.71	1.78	2.33	1.78
Mo	%	-	0.017	0.018	0.022	0.024	0.026	0.021
ROM ORE to stockpiles	kt	1,378	25,058	15,053	7,254	8167	642	57,551
Total ORE Mined	kt	1,378	205,108	195,103	187,304	138,742	89,072	816,707
ROM ORE retreived from stockpiles	kt	-	-	-	-	49,475	3,599	53,074
Cu	%	-	-	-	-	0.190	0.145	0.187
Au	g/t	-	-	-	-	0.155	0.136	0.153
Ag	g/t	-	-	-	-	1.44	1.70	1.46
Mo	%	-	-	-	-	0.008	0.008	0.008
Total Stockpile Inventory	kt	1,378	26,435	41,488	48,742	7,434	-	-
Total ROM ORE to MILL	kt	-	180,050	180,050	180,050	180,050	92,030	812,230
Cu	%	-	0.332	0.323	0.291	0.260	0.318	0.301
Au	g/t	-	0.237	0.235	0.181	0.171	0.262	0.212
Ag	g/t	-	1.59	1.78	1.71	1.69	2.31	1.76
Mo	%	-	0.017	0.018	0.022	0.020	0.026	0.020

Table 3.3 Summarized Production Schedule								
Material to be Mined	**Units**	**Time (Years)**						
		PP	**1 to 5**	**6 to 10**	**11 to 15**	**16 to 20**	**21 to EOM**	**LOM**
Metal in Process Feed								
Cu	M lb	-	1,278.8	1,282.1	1,154.6	1032.0	646.1	5,389.6
Au	M oz	-	1.37	1.36	1.05	0.99	0.77	5.54
Ag	M oz	-	9.19	10.32	9.88	9.78	6.83	46.00
Mo	M lb	-	65.64	71.74	85.97	77.72	51.75	352.82
Recovered ROM Grades								
RCu	%	-	0.287	0.288	0.259	0.232	0.284	0.269
RAu	g/t	-	0.162	0.161	0.124	0.117	0.180	0.145
RAg	g/t	-	1.28	1.44	1.38	1.36	1.86	1.42
RMo	%	-	0.012	0.013	0.016	0.014	0.018	0.014
Recoverable Metal in Process Feed								
RCu	M lb	-	1,140.7	1,143.6	1,029.9	920.6	576.4	4,811.1
RAu	M oz	-	0.94	0.93	0.72	0.68	0.53	3.80
RAg	M oz	-	7.41	8.33	7.97	7.89	5.52	37.12
RMo	M lb	-	47.26	51.65	61.90	55.96	37.26	254.03
Waste								
Sub-Grade Wasted	kt	-	-	2,107	1,017	1,134	111	4368
Waste Mined	kt s	23,621	339,981	368,876	386,859	385,741	38,113	1,543,191
Total Waste Mined	kt	23,621	339,981	370,983	387,875	386,875	38,224	1,547,559
Waste Types								0
Waste	kt	23,621	339,981	368,876	386,859	385,741	38,113	1,543,191
Strip Ratio (Waste Mined/Ore Milled)		-	1.9	2.0	2.1	2.1	0.4	1.9
Total Material Mined	t	24,999	545,088	566,087	575,179	525,616	127,296	2,364,266
Total Material Moved	t	24,999	545,088	566,087	575,179	525,616	130,895	2,417,340

Figure 3.4 illustrates that significant stockpile reclaim is required in years 15 to 19 when the pre-stripping of the ultimate phase is being completed.

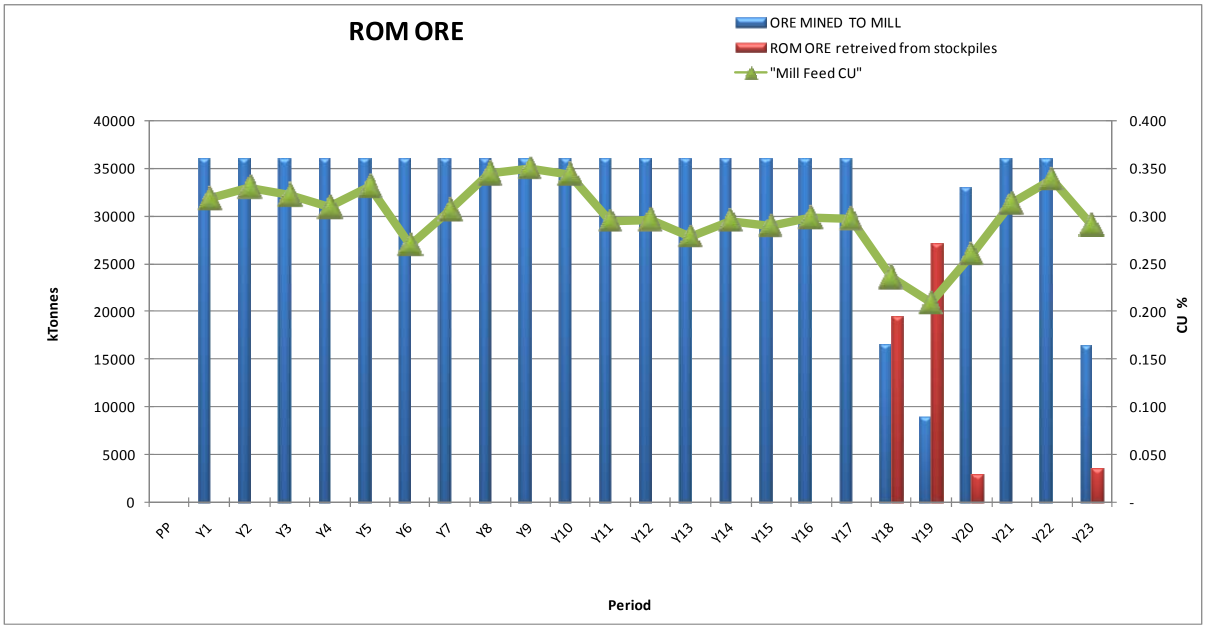


Figure 3.4 Graph Showing ROM Ore Source and Mill Feed Copper grade

If there is insufficient ore stockpiled prior to the pre-stripping of the ultimate phase a substantial increase in fleet will be required in order to meet the material movement requirements.

Figure 3.5 shows a rendering of what the mine pit and dump piles are expected to look like at the end of the mine life. The PAG material location is the magenta colored material and has been backfilled in part of the existing pit and capped with NPAG material.

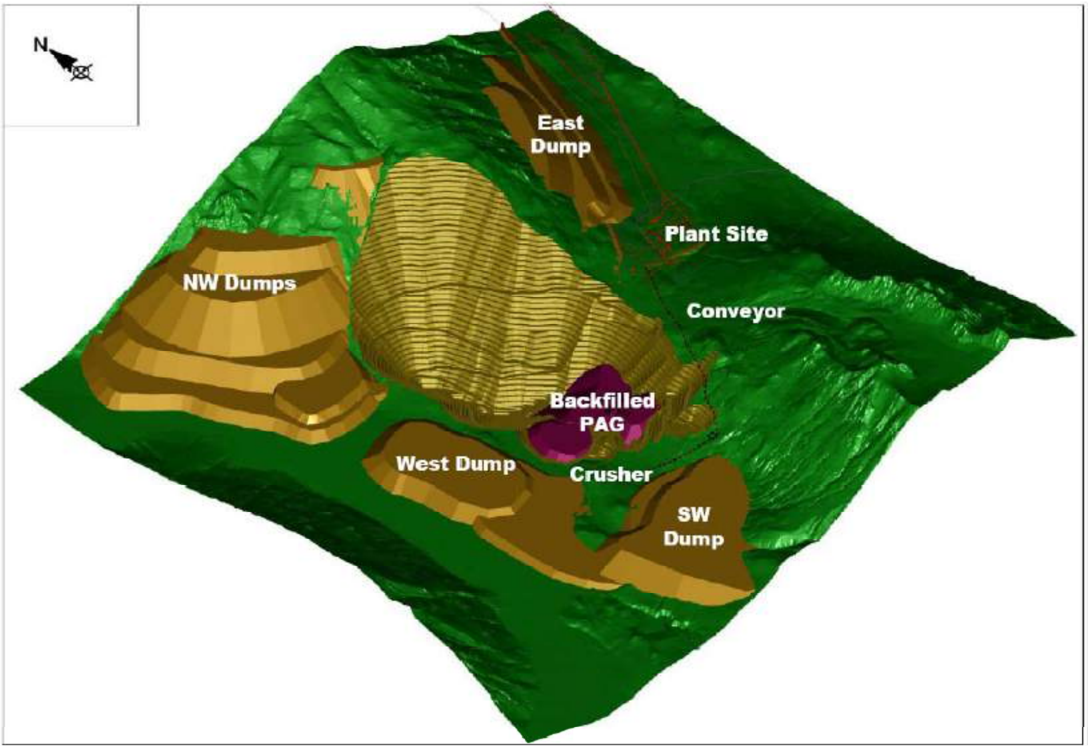


Figure 3.5 End of Life of Mine Orthographic View from the South West

3.6.2 Mine Equipment

Table 3.4
Major Mine Equipment Requirements

	PP	Y1	Y2	Y3	Y4	Y5	Y10	Y20	Y23
Drilling:									
Primary Drill - P&H 120A – 311 mm Electric Drill	2	4	4	4	4	4	4	4	1
Highwall Drill - Sandvik D245S – 150 mm Diesel Drill	1	1	1	1	1	1	1	1	0
Loading:									
P&H4100XPC Cable Shovel - 104 t	2	4	4	4	4	4	4	4	1
Cat 3516 GENSET	1	1	1	1	1	1	1	1	1
Hauling									
Cat 797B Haul Truck - 345 t	8	12	12	17	18	23	28	27	7

3.7 Metallurgy

The metallurgical test program completed for the PFS is a continuation of the test work for the development of the Schaft Creek Project. Some of the previous test work was reported in the report *Preliminary Economic Assessment on the Development of the Schaft Creek*

Project Located in Northwest British Columbia, Canada that was issued December 7, 2007, a Canadian NI43-101 Technical Report.

G&T Metallurgical Services, Ltd. (G&T) subsequently prepared a January 9, 2008 report titled *Floatation Responses of Three ore Sources, Schaft Creek Deposits, Liard District, BC, Canada*, which included some information that was unavailable for the Preliminary Economic Assessment (PEA). Some data used in the PFS is from the earlier G&T report and additional information is from new metallurgical tests that were completed in support of this PFS. The newest data is reported in *Advanced Flowsheet Development Studies Schaft Creek Deposit Liard District, BC Canada* that was issued June 20, 2008.

Samples from 2005 and 2006 drill core were prepared for testing at the following laboratories.

- G&T Metallurgical Services, Ltd. in Kamloops, B.C., Canada;
- Hazen Research, Inc. in Golden, Colorado, USA;
- Polysius AG in Neubeckum, Germany;
- Teck Cominco CESL Technology laboratory in Vancouver, B.C., Canada.

The results of the recent metallurgical testing continue to indicate that the Schaft Creek resource is amenable to the typical conventional flotation methods utilized for copper/molybdenum porphyry deposits. The results indicate that high-grade copper flotation concentrates can be achieved due to the presence of secondary copper minerals such as bornite, covellite and chalcocite. Secondary mineralization appears to be pervasive throughout the resource and occurs even in low-grade areas. Copper concentrates containing 30 to 35% copper are achievable at the average resource grade of 0.30 to 0.35% copper. Copper recoveries of approximately 88 to 90% can be expected at these flotation concentrate grades.

Molybdenum occurs throughout the resource and can be recovered into a saleable concentrate containing 50% molybdenum at an overall recovery of approximately 68 to 71% to the molybdenum concentrate. The molybdenum concentrates should contain approximately 368 ppm rhenium. The mineralogical evaluation of molybdenum concentrates indicates that it should be possible to increase the molybdenum concentrate grade to approximately 54% molybdenum. However, additional studies are needed to optimize the molybdenum separation circuit and achieve these results.

Gold and silver are also pervasive and are recovered with the copper in the copper flotation concentrate. Gold recoveries of approximately 80% appear to be achievable while silver recovery is approximately 71%. Gold and silver grades in the copper concentrate appear to average approximately 22 and 158 g/t respectively.

3.8 Process

For the PFS, the concentrator has an annual throughput of 36,000,000 t, 100,000 tpd, which has increased from 23,400,000 tpa, 65,000 tpd, in the PEA. The concentrator will include an SABC comminution circuit followed by a bulk flotation circuit, a molybdenum separation circuit and a copper circuit. The copper circuit has a thickener, filters and a concentrate stockpile. The molybdenum circuit includes filtration, drying and bagging equipment.

Tailings thickeners, tailings storage facility and water reclaim are part of the tailings management system. The processing circuit will have a design capacity of 108,700 tpd. A block flow diagram of the proposed process can be found below.

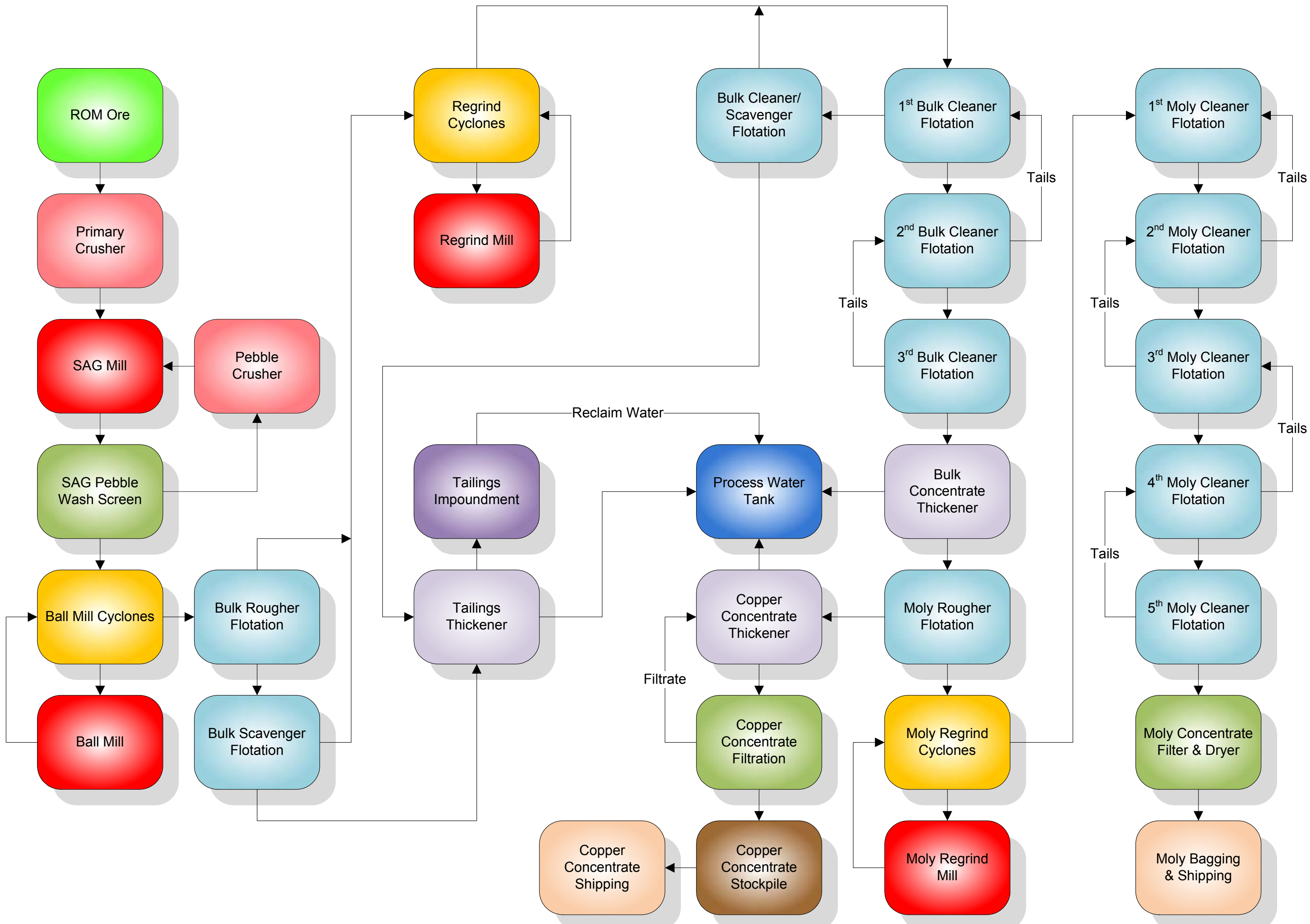


Figure 3.6 Block Flow Diagram

The flotation circuits and other processing circuits were simulated using METSIM. The results of the simulation provided the mass and water balances, which, in turn, were used for equipment sizing. The mass and water balances are shown on the process flowsheets. An abbreviated Process Design Criteria is shown below.

Table 3.5
Abbreviated Process Design Criteria June 26, 2008

	Units	Design	Source
General Site Information			
Location			
Latitude	degrees	N 57° 21' 15”	Copper Fox Metals, Inc.
Longitude	degrees	W 131° 0' 58”	Copper Fox Metals, Inc.
Elevation			
Air Strip	masl	866.00	Samuel Engineering
Plant	masl	1,230.00	Samuel Engineering
Pit Bottom	masl	600.00	Moose Mountain Technical Services
Ambient Air Temperature			
Winter Minimum	°C	-30.00	
Summer Maximum	°C	28.00	
Average Annual Precipitation	mm/y	1,039.00	Knight Piésold
Average Annual Evaporation - Pond Area	mm/y	450.00	Knight Piésold
Maximum Wind Velocity	km/h		
General Project Information			
Reported Resource	tonnes (t)	1,393,282,000	Associated Geosciences Ltd.
Cutoff Grade Used	CuEq (%)	0.20	Associated Geosciences Ltd.
Estimated Mineable Resources			
Starter Pit (5 Year)	tonnes (t)	216,060,000	Moose Mountain Technical Services
Life of Mine (includes subgrade to waste)	tonnes (t)	816,707,000	Moose Mountain Technical Services
Operating Schedule			
Hours per Day	h	24	Samuel Engineering
Days per Year	d	360	Samuel Engineering
Hours per Year	h	8,640	Samuel Engineering
Plant Capacity (design based on 92% grinding circuit operating time)	dmtpd	108,696	Samuel Engineering
Plant Capacity (design based on 92% grinding circuit operating time)	dmtph	4,529	Samuel Engineering
Annual Ore Processed per Year	t	36,000,000	Moose Mountain Technical Services
Mineral Reserves to Mill	t	812,231,000	Moose Mountain Technical Services
Estimated Project Life @ 100,000 tpd	y	22.56	Moose Mountain Technical Services
Life of Mine Plant Head Grade Estimates			
Estimated Copper Grade	%	0.301	Moose Mountain Technical Services
Estimated Molybdenum Grade	%	0.020	Moose Mountain Technical Services

Table 3.5 Abbreviated Process Design Criteria June 26, 2008			
Estimated Gold Grade	g/t	0.212	Moose Mountain Technical Services
Estimated Silver Grade	g/t	1.76	Moose Mountain Technical Services
First 5 Years Plant Head Grade Estimates			
Estimated Copper Grade	%	0.337	Moose Mountain Technical Services
Estimated Molybdenum Grade	%	0.017	Moose Mountain Technical Services
Estimated Gold Grade	g/t	0.249	Moose Mountain Technical Services
Estimated Silver Grade	g/t	1.65	Moose Mountain Technical Services
Plant Design (First 5 Years) Head Grade Estimates			
Estimated Copper Grade	%	0.337	Moose Mountain Technical Services
Estimated Molybdenum Grade	%	0.017	Moose Mountain Technical Services
Estimated Gold Grade	g/t	0.249	Moose Mountain Technical Services
Estimated Silver Grade	g/t	1.649	Moose Mountain Technical Services
Design ROM Ore Dry Solids Sp Gr	g/cc	2.69	Copper Fox Metals, Inc.
Design ROM Ore Moisture (for material handling)	%	3.00	Copper Fox Metals, Inc.
First Five-year Average Copper Concentrate Production			
Copper Recovery to Copper Concentrate	%	89.21	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Grade in Copper Concentrate	%	33.52	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Recovery to Copper Concentrate	%	10.82	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Grade in Copper Concentrate	% Mo	0.21	Hyyppa Engineering, LLC/G&T Metallurgical
Gold Recovery to Copper Concentrate	%	80.71	Hyyppa Engineering, LLC/G&T Metallurgical
Gold Grade in Copper Concentrate	g/t	22.40	Hyyppa Engineering, LLC/G&T Metallurgical
Silver Recovery to Copper Concentrate	%	72.00	Hyyppa Engineering, LLC/G&T Metallurgical
Silver Grade in Copper Concentrate	g/t	132.40	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Concentrate Production	dmtph	40.62	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Concentrate Production	dmtpy	322,848	Hyyppa Engineering, LLC/G&T Metallurgical
First Five-year Average Moly Concentrate Production			
Moly Recovery to Moly Concentrate	%	68.61	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Grade in Moly Concentrate	% Mo	50.00	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Recovery to Moly Concentrate	%	0.07	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Grade in Moly Concentrate	%	1.06	Hyyppa Engineering, LLC/G&T Metallurgical
Rhenium Grade in Moly Concentrate	ppm	368.00	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Concentrate Production	dmtph	1.06	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Concentrate Production	dmtpy	8,397.91	Hyyppa Engineering, LLC/G&T Metallurgical
Life of Mine Average Copper Concentrate Production			

Table 3.5 Abbreviated Process Design Criteria June 26, 2008			
Copper Recovery to Copper Concentrate	%	88.36	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Grade in Copper Concentrate	%	33.85	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Recovery to Copper Concentrate	%	11.24	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Grade in Copper Concentrate	% Mo	0.29	Hyyppa Engineering, LLC/G&T Metallurgical
Gold Recovery to Copper Concentrate	%	81.29	Hyyppa Engineering, LLC/G&T Metallurgical
Gold Grade in Copper Concentrate	g/t	21.90	Hyyppa Engineering, LLC/G&T Metallurgical
Silver Recovery to Copper Concentrate	%	70.69	Hyyppa Engineering, LLC/G&T Metallurgical
Silver Grade in Copper Concentrate	g/t	158.30	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Concentrate Production	dmtph	35.59	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Concentrate Production	dmtpy	282,881.89	Hyyppa Engineering, LLC/G&T Metallurgical
Life of Mine Average Moly Concentrate Production			
Moly Recovery to Moly Concentrate	%	71.29	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Grade in Moly Concentrate	% Mo	50.00	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Recovery to Moly Concentrate	%	0.10	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Grade in Moly Concentrate	%	1.06	Hyyppa Engineering, LLC/G&T Metallurgical
Rhenium Grade in Moly Concentrate	ppm	368.00	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Concentrate Production	dmtph	1.29	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Concentrate Production	dmtpy	10,261.90	Hyyppa Engineering, LLC/G&T Metallurgical

3.9 Permitting and Environmental

3.9.1 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.

3.9.2 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.

3.10 Reclamation and Closure

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.

Although there is little surficial soil in the pre-mining topography, any suitable surficial soils excavated during mining will be stored in stockpiles and used to cap the re-contoured landscape at decommissioning. Whenever possible, direct placement of the suitable topsoil material will be carried out in order to avoid 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 evaluations of the rock. Detailed design criteria will be adjusted based on the results obtained from these future studies and the requirements of all applicable permits.

Copper Fox is planning to complete a soils baseline study by 4th Quarter 2008. This study will provide the information needed to evaluate the types of soils and their availability for reclamation. The information on the soils will provide additional information to be incorporated into the feasibility study with regard to the reclamation design and associated costs.

3.11 Project Execution

A comprehensive plan for the development and implementation of the Schaft Creek project will be developed. The plan will address all aspects of project development, including objectives, scope, and strategies. The plan will provide for engineering, procurement, construction, startup, and commissioning of the plant facilities and infrastructure and addresses the roles, responsibilities, and management plans required to execute and manage the work.

An EPC schedule was developed for the project and summarized below.

- Kick-off feasibility study August 1, 2008
- Environmental assessment submitted March 2, 2009
- Road permit application submitted March 2, 2009
- Feasibility study completed March 31, 2009
- Teck Cominco review period completed July 31, 2009
- Financing period completed October 2, 2009
- Road permit approved October 1, 2009
- Plant permit approved October 1, 2009
- Financing in place October 2, 2009
- Kick-off detailed engineering October 5, 2009
- Place PO for mills October 9, 2009
- Place PO for electric shovels October 5, 2009
- Construction start October 28, 2009
- Place PO for crushers Dec. 18, 2009
- Airfield upgrades complete May 17, 2010
- Access road complete June 30, 2010
- Place PO haul trucks August 30, 2010
- Place PO for electric drills August 30, 2010
- Temporary camps complete October 25, 2010
- Detailed engineering complete February 4, 2011
- Permanent camp complete March 4, 2011
- Begin tailings starter dam June 30, 2011
- Power line complete March 27, 2012
- Begin mine development October 15, 2012
- Complete tailings starter dam Dec. 12, 2012
- Mechanical completion July 18, 2013
- Start-up complete Dec. 5, 2013
- Mine ramp up complete March 12, 2014

3.12 Operating Cost Estimate

The operating costs are estimated to be $12.49 per tonne of ore processed. They are allocated as shown in Table 3.6 and Figure 3.7 below.

Table 3.6 Total Op Cost Estimate LOM Average				
Description	**Fixed or Variable**	**Annual Cost**	**Cost/t Ore**	**Cost /t Tot. Material**
			Ore	
Mining (from MMTS)				
Drilling	Variable	$7,812,461	$0.217	$0.075
Blasting	Variable	$21,742,224	$0.604	$0.208
Loading	Variable	$14,164,770	$0.393	$0.135
Hauling	Variable	$60,543,823	$1.682	$0.579
Mine Maintenance	Variable	$1,859,903	$0.052	$0.018
Mine Operations Support	Variable	$29,305,187	$0.814	$0.280
Snow Removal	Variable	$1,269,140	$0.035	$0.012
Geotech	Variable	$2,642,872	$0.073	$0.025
Unallocated Labor Cost	Variable	$1,869,136	$0.052	$0.018
Mine Ops Salaried Personnel	Fixed	$1,829,645	$0.051	$0.017
Mine Maintenance Salaried Personnel	Fixed	$2,852,992	$0.079	$0.027
Mine Engineering Salaried Personnel	Fixed	$2,022,324	$0.056	$0.019
Technical Services Salaried Personnel	Fixed	$1,057,021	$0.029	$0.010
Mining Total		**$148,971,499**	**$4.1381**	**$1.424**
Processing				
Processing Salaried Personnel	Fixed	$6,603,000	$0.1834	
Hourly Operations Labor	Fixed	$19,317,761	$0.5366	
Hourly Maintenance Labor	Fixed	$5,461,379	$0.1517	
Plant Electrical Power	Variable	$42,088,428	$1.1691	
Reagents	Variable	$16,313,985	$0.4532	
Grinding Steel	Variable	$42,274,930	$1.1743	
Laboratory Supplies	Variable	$1,634,416	$0.0454	
Maintenance Supplies (5% of process ops $)	Variable	$6,602,974	$0.1834	
Misc. Ops Supplies (1% of processing ops $)	Variable	$1,402,969	$0.0390	
Processing Total		**$141,699,842**	**$3.9361**	
General and Administration				
G&A Salaried Personnel	Fixed	$4,470,000	$0.1242	
Hourly Labor	Fixed	$1,492,992	$0.0415	
Power	Fixed	$616,480	$0.0171	
Vehicle Operating & Maintenance	Fixed	$115,385	$0.0032	
Access Road & Powerline Maintenance	Fixed	$1,410,000	$0.0392	
Site Avalanche Control	Fixed	$650,000	$0.0181	

Table 3.6 Total Op Cost Estimate LOM Average				
Description	**Fixed or Variable**	**Annual Cost**	**Cost/t Ore**	**Cost /t Tot. Material**
Communications	Fixed	$75,000	$0.0021	
Camp Operations	Fixed	$8,627,693	$0.2397	
Fly-in, Fly-out Operations & Airfield Operations	Fixed	$10,000,000	$0.2778	
Safety Supplies / Incentives	Fixed	$300,000	$0.0083	
Offsite Training & Conferences	Fixed	$300,000	$0.0083	
Insurance	Fixed	$1,000,000	$0.0278	
Corporate Services and Travel	Fixed	$350,000	$0.0097	
Environmental	Fixed	$1,000,000	$0.0278	
Security & Medical	Fixed	$400,000	$0.0111	
Professional Membership Costs	Fixed	$30,000	$0.0008	
Community Development	Fixed	$1,000,000	$0.0278	
Land Holding	Fixed	$-	$-	
Consultants	Fixed	$500,000	$0.0139	
Misc. Computer Equipment/Software	Fixed	$250,000	$0.0069	
Misc. Office Supplies	Fixed	$200,000	$0.0056	
Misc. Freight & Couriers	Fixed	$100,000	$0.0028	
Recruiting and Relocation	Fixed	$200,000	$0.0056	
Legal, Permits, Fees	Fixed	$500,000	$0.0139	
G&A Total		**$33,587,550**	**$0.9330**	
Total Operations and Administration		**$324,258,890.99**	**$9.0072**	
Concentrate Handling & Treatment		**Annual Cost**	**Cost/Tonne Ore**	**Charge/Tonne Conc**
Copper Concentrate Pipeline to Filter Plant	Variable	$-	$-	
Copper Conc Transportation to Receiving Port	Variable	$34,967,031	$0.9713	$123.61
Copper Concentrate Treatment Charges	Variable	$19,801,733	$0.5500	$70.00
Copper Conc Refining Charges	Variable	$14,777,335	$0.4105	$ 52.24
Gold Refining Charges	Variable	$1,165,189	$0.0324	$4.12
Silver Refining Charges	Variable	$453,511	$0.0126	$1.60
Moly Conc Transportation to Receiving Port	Variable	$2,133,039	$0.0593	$207.86
Moly Concentrate Roasting & Refining	Variable	$52,260,559	$1.4517	$5,092.68
Conc Handling & Treatment Total		**$125,558,396**	**$3.4877**	
Contingency	0.0%	**$**	**$-**	
TOTAL PROJECT COSTS		**$449,817,287**	**$12.49**	

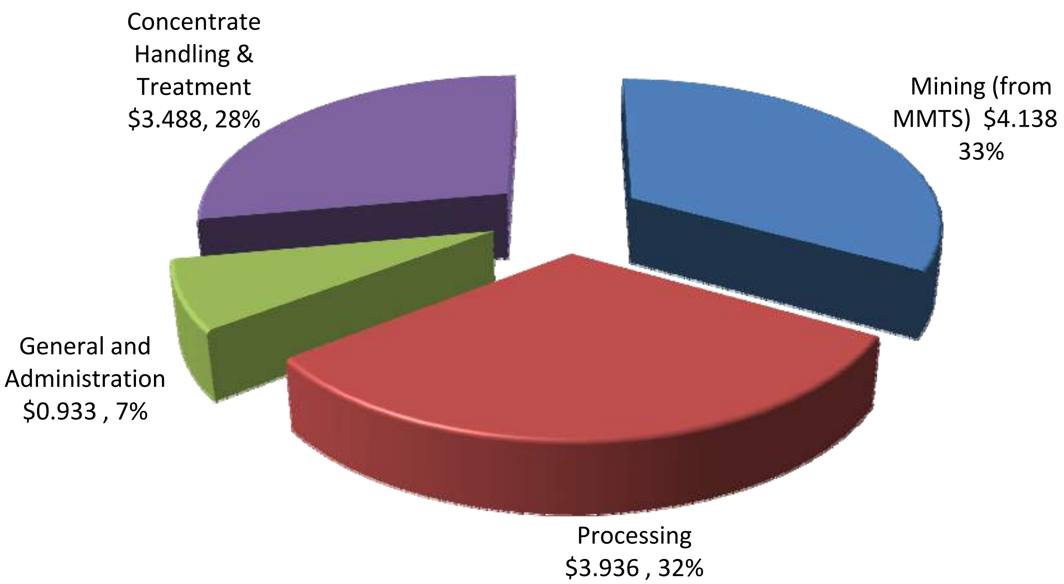


Figure 3.7 Schaft Creek Life of Mine Average Annual Operating Costs

3.13 Capital cost Estimate

The total estimated cost to design, procure, and construct the facilities is $2,950,406,000. Table 3.7 summarizes the capital costs by major area.

Table 3.7
Capital Cost Summary

	Total ($Ms)
Mine Facilities	$50.4
Primary Crushing & Coarse Ore Conveying	$85.8
Coarse Ore Reclaim	$125.9
Grinding & Pebble Crushing	$317.5
Bulk Floatation & Regrind	$133.0
Copper-Molybdenum Separation	$6.0
Copper Concentrate	$14.5
Molybdenum Concentrate	$4.1
Tailings Handling	$35.2
Tailings Pond & Water Reclaim System	$17.3
Tailings Impoundment	$173.3
Reagents	$14.6
Plant Services	$33.8
Buildings & Ancillary Facilities	$39.9
Site Development	$264.2
Contracted Directs	***$1,315.5***
Common Distributables	$399.1

Table 3.7 Capital Cost Summary	
EPCM	$211.0
Contracted Indirects	***$610.1***
Owner's Cost	***$459.8***
Subtotal	***$2,385.3***
PST Taxes	$28.6
Contingency	$536.5
Total	**$2,950.4**

3.14 Economic Analysis

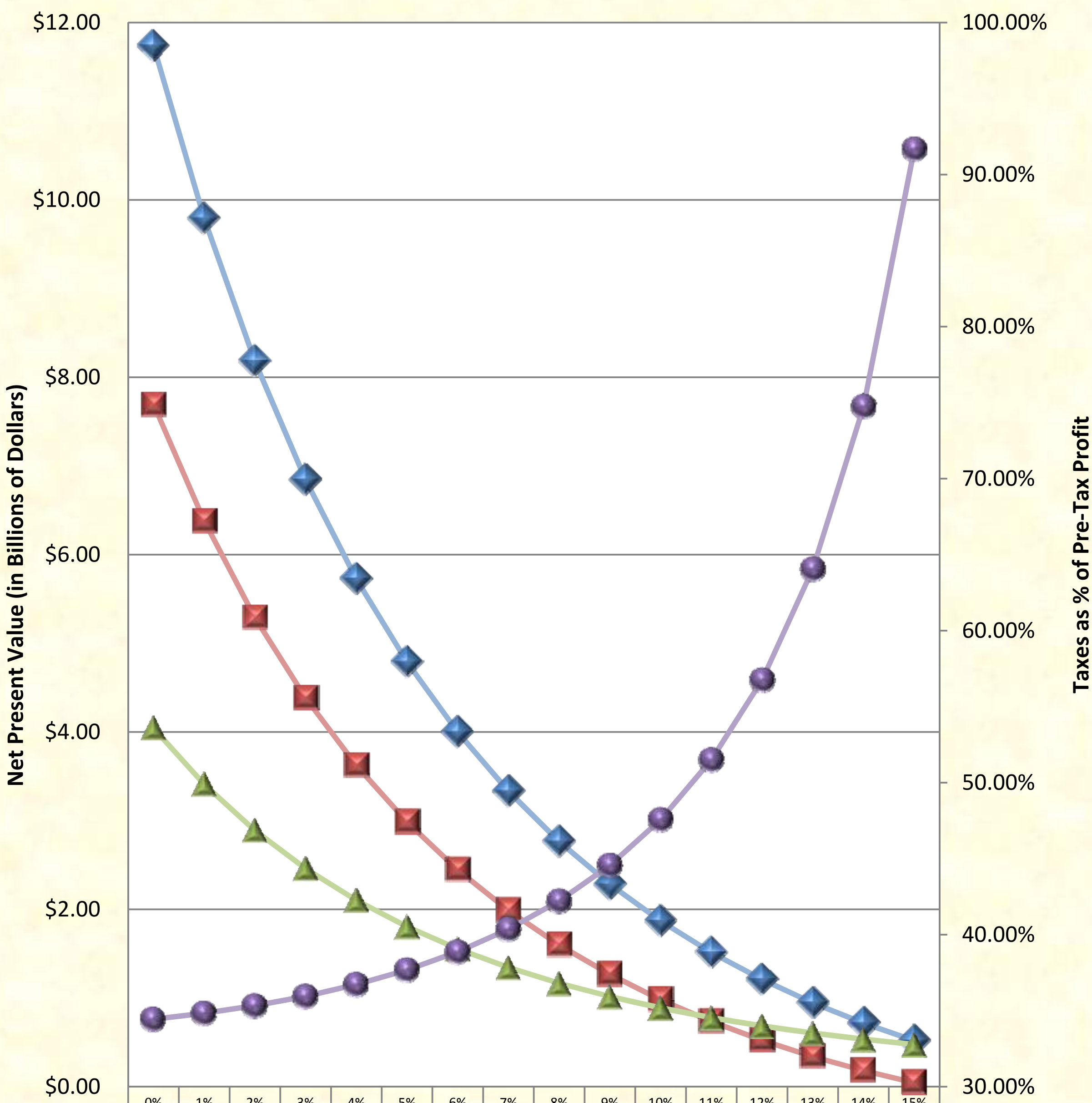
A financial model was created utilizing the mine production schedule, the associated metal grades based on the geological resource estimate, metal recoveries from the ongoing metallurgical test program, capital and operating costs as set out herein and base case metal prices, trailing three year average from August 29, 2008, of copper $3.12/lb, molybdenum $33.00/lb, gold $692.85/oz and silver $13.09/oz.

Modeling at base case metal prices shows that the project could generate a cumulative before tax profit of $11,735 million, with a payback period of 4.7 years, a 18.6% IRR, and a net present value discounted at 10% of $1,868 million, over the 23 year mine life.

Modeling at base case metal prices shows that the project could generate a cumulative after tax profit of $7,393 million, with a payback period of 4.9 years, a 15.3% IRR, and a net present value discounted at 10% of $980 million, over the 23 year mine life.

Below is a graph showing the NPV results and taxation plotted against the corresponding discount rates. The graph suggests that from a strictly optimum PV profit point of view, there may be a higher throughput rate that could increase the PV profitability for the project (note the sharp drop in NPVs).

Schaft Creek Preliminary Feasibility Study Base Case
Net Present Value Profiles

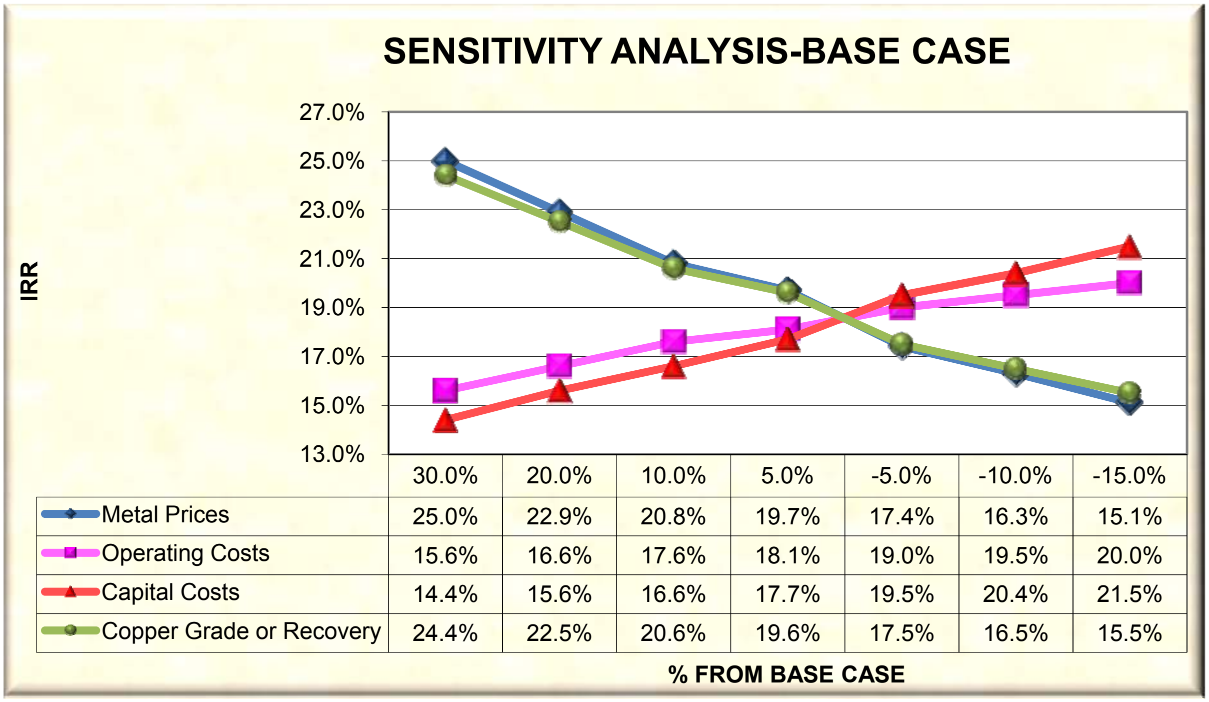


	0%	1%	2%	3%	4%	5%	6%	7%	8%	9%	10%	11%	12%	13%	14%	15%
Before Tax NPV	$11.73	$9.79	$8.18	$6.84	$5.72	$4.79	$4.00	$3.33	$2.76	$2.28	$1.87	$1.51	$1.21	$0.95	$0.72	$0.52
After Tax NPV	$7.69	$6.38	$5.29	$4.38	$3.62	$2.98	$2.44	$1.99	$1.60	$1.26	$0.98	$0.73	$0.52	$0.34	$0.18	$0.04
Taxes	$4.04	$3.41	$2.89	$2.46	$2.10	$1.80	$1.55	$1.34	$1.17	$1.02	$0.89	$0.78	$0.69	$0.61	$0.54	$0.48
Taxes as % of Pre-Tax Profit	34.44%	34.83%	35.33%	35.95%	36.72%	37.68%	38.87%	40.36%	42.21%	44.56%	47.57%	51.49%	56.74%	64.04%	74.73%	91.64%

Present Value Discount Rates (in % Per Year)

3.14.1 Project Sensitivity Analysis

Sensitivity calculations were performed on the project cash flow by applying factors ranging from -15% to +30% against metal prices, initial capital costs, annual operating costs and copper grade. The effects on IRR and NPV are shown graphically in Figure 3.8 and Figure 3.9, respectively. The project is moderately sensitive to changes in capital and operating costs and more sensitive to changes in metal prices and copper grade.



	30.0%	20.0%	10.0%	5.0%	-5.0%	-10.0%	-15.0%
Metal Prices	25.0%	22.9%	20.8%	19.7%	17.4%	16.3%	15.1%
Operating Costs	15.6%	16.6%	17.6%	18.1%	19.0%	19.5%	20.0%
Capital Costs	14.4%	15.6%	16.6%	17.7%	19.5%	20.4%	21.5%
Copper Grade or Recovery	24.4%	22.5%	20.6%	19.6%	17.5%	16.5%	15.5%

Figure 3.8 Base Case IRR Sensitivity Analysis

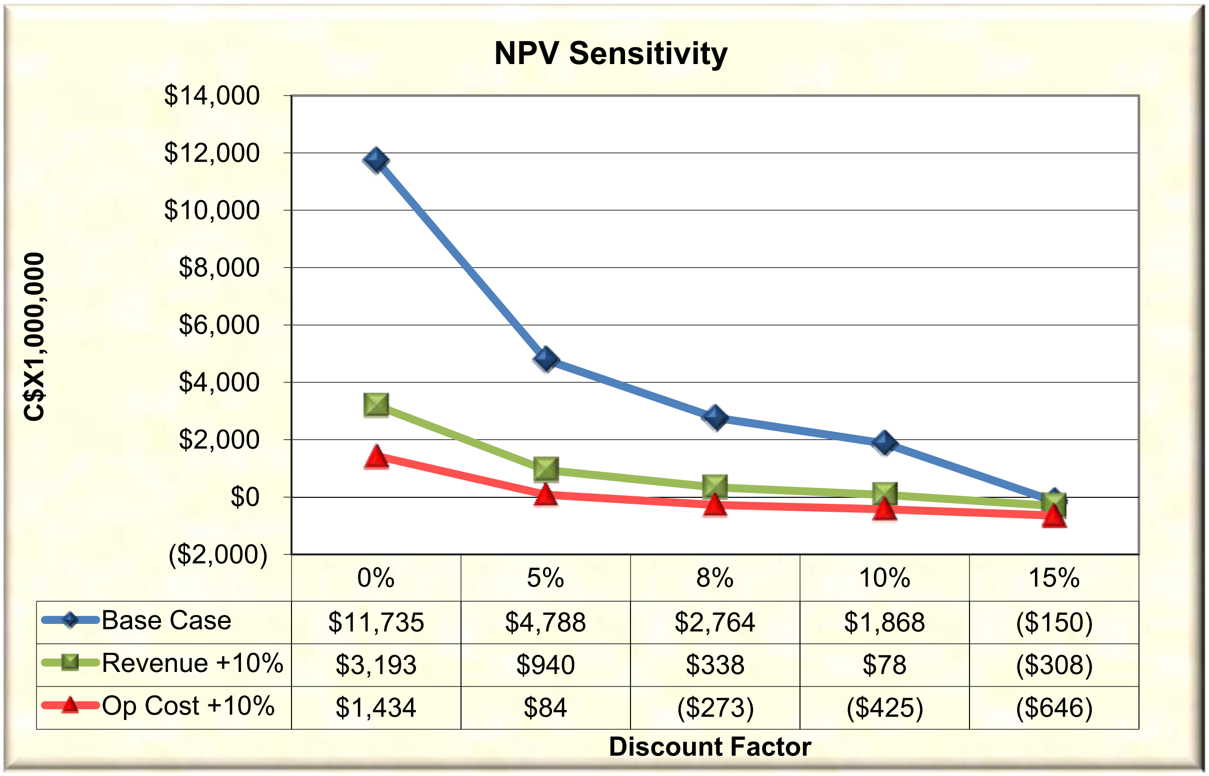


Figure 3.9 Base Case NPV Sensitivity Analysis

3.15 Conclusions & Recommendations

It is recommended that Copper Fox advance their Schaft Creek project to the feasibility stage. There remain opportunities to improve the project economics through optimizations of the mine plan, processing flowsheet, method of tailings disposal, power supply, operating costs, capital costs and metal pricing strategies.

3.15.1 Key Results

Key results of this PFS for the Schaft Creek project are shown in Table 3.8 below.

Table 3.8 Key Project Parameters and Results				
September 9, 2008, Rev. 5a				
Total Resource (M&I)	tonnes	1,393,282,171	@ 0.25% Cu	
Total Reserve	tonnes	816,706,750		
LOM Mill Feed	tonnes	812,230,421		
LOM Waste	tonnes	1,543,190,551		
LOM Strip Ratio		1.88		
Daily Feedrate	tpd	100,000		
Mine Life	yrs	22.6		
Connected Load	MW	146.8		

Table 3.8 Key Project Parameters and Results				
Avg Power Demand	MW	121.4		
Power Cost	$/kWh	0.0469		
Foreign Exchange Rate		US$1 = C$1		
Total Initial Capex	$ (000's)	**2,950,406**		
Directs	$ (000's)	1,315,484		
Indirects	$ (000's)	610,108		
Owner	$ (000's)	459,757		
Taxes	$ (000's)	28,566		
Contingency	$ (000's)	536,490		
Working Capex	$ (000's)	146,420		
Total Sustaining Capex	$ (000's)	797,379		
Mine	$ (000's)	232,893		
Mill	$ (000's)	220,000		
Tailings	$ (000's)	257,486		
Reclamation & Closure	$ (000's)	87,000		
Total LOM Opex	$ (000's)	**10,138,610**		
Total LOM Opex	$/t ore	**12.49**		
Mining	$/t ore	4.14		
Processing	$/t ore	3.94		
G&A	$/t ore	0.93		
Conc Handling & Treatment	$/t ore	3.49		
Contingency (0%)	$/t ore	0.00		
Total LOM Taxes	$ (000's)	4,041,652		
Metrics		**Head Grades**	**Conc Grades**	**Recoveries**
Cu	%	0.301%	33.85%	88.4%
Mo	%	0.020%	50.0%	71.3%
Au	g/t	0.212	21.90	81.3%
Ag	g/t	1.761	158.30	70.7%
Base Case Pricing (Trailing 3 Year Avg - August 29, 2008)				
Cu	$/lb	3.12		
Mo	$/lb	33.00		
Au	$/oz	692.85		
Ag	$/oz	13.09		
Cashflow Results		**Before Tax**	**After Tax**	**Direct Tax Effects**
IRR	%	18.6%	15.3%	-3.21%
NPV @ 0%	$ (000's)	$11,734,537	$7,692,885	($4,041,652)
NPV @ 5%	$ (000's)	$4,787,931	$2,983,847	($1,804,084)
NPV @ 8%	$ (000's)	$2,764,475	$1,597,500	($1,166,974)
NPV @ 10%	$ (000's)	$1,868,441	$979,686	($888,755)
NPV @ 12%	$ (000's)	$1,208,843	$522,898	($685,945)
NPV @ 15%	$ (000's)	($149,721)	($422,853)	($273,132)
Payback Period	yrs	4.7	4.9	
Rock Value *	$/t ore	$31.47		
LOM Recoverable Revenue	$ (000's)	25,559,408		

Table 3.8 Key Project Parameters and Results				
Cu	%	54.4%		
Mo	%	32.2%		
Au	%	12.0%		
Ag	%	1.4%		
Total Metal Production		**LOM**	**Annual**	**Annual Tonnes**
Cu	lbs	4,762,524,025	211,104,788	95,757
Mo	lbs	255,194,418	11,311,809	5,131
Au	ozs	4,493,445	199,178	
Ag	ozs	32,480,015	1,439,717	
CFM Portion of Metal Production		LOM	Annual	Tonnes
Cu	lbs	1,112,049,360	49,292,968	22,359
Mo	lbs	59,587,897	2,641,307	1,198
Au	ozs	1,049,219	46,508	
Ag	ozs	7,584,084	336,174	
Facilities Startup		4 QTR 2013		

4.0 Introduction

4.1 Purpose

Copper Fox Metals Inc. (Copper Fox) continues with site and investigative work with the intention to develop their Schaft Creek project. Copper Fox has commissioned a Preliminary Feasibility Study (PFS) for its Schaft Creek project in 2008 following the successful completion of a Preliminary Economic Assessment (PEA) in 2007. The purpose of this PFS is to advance those concepts and designs developed during the PEA, including metallurgical testwork, resource and reserve estimates, mine design, operating cost estimates, capital cost estimates and project related cash flow analysis. The results of this study will further assist Copper Fox in making decisions in regards with further advancing the project towards eventual development.

4.2 Sources of Information

Various Copper Fox personnel including Mr. Guillermo Salazar, Mr. Cam Grundstrom, Mr. Frank Agar, Mr. Michael Smith, Mr. Murray Hunter Dr. Adrian Mann, Mr. Shane Uren and Mr. Dave Mullen, along with Copper Fox's many consultants provided key input, background information and data for the study. Many of the contributors are Qualified Persons (QPs) who are not independent of Copper Fox Minerals. Therefore, information that was provided by the contributors was verified by the independent QPs, as shown in Table 4.1. Samuel Engineering, Inc. (SE) compiled and reviewed all information provided by other consultants to complete this report. Mr. Matt Bender, the Study Manager and Responsible Author, was employed by Samuel Engineering at the effective date of the report.

Information regarding the geology, deposit, mineralization, exploration, drilling, sampling, sample preparation, analysis and security, and data verification was contributed by others and verified by Mr. Keith McCandlish of Associated Geosciences Ltd.

4.3 Site Visit

- *Matt Bender, Director of Process, Mining & Metals, of Samuel Engineering* (at the time of the study) visited the project site July 16 to 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, he observed the project site and drilling activities and talked with various site personnel.
- *Keith McCandlish, Managing Director, of Associated Geosciences* visited the project site on two separate occasions. His last visit to the site was June 13 to 16, 2008. He observed the project site, drilling/sampling/logging practices, and examined available drill core. In addition, available reports, cross sections, geologic interpretations and other relevant geologic data were examined at the Schaft Creek camp.

Table 4.1 List of Authors			
Section No.	**Section Name**	**Company**	**Responsible QP**
1	Title Page	-	-
2	Table of Contents	-	-
3 except 3.3, 3.4	Summary	SE AGL	Matt Bender, P.E. Keith McCandlish, P.Geo
4	Introduction	SE	Matt Bender, P.E.
5	Reliance on Other Experts	SE	Matt Bender, P.E.
6	Property Description and Location	SE	Matt Bender, P.E.
7	Accessibility, Climate, Local Resources, Infrastructure and Physiography	SE	Matt Bender, P.E.
8	History	SE	Matt Bender, P.E.
9	Geological Setting	AGL	Keith McCandlish, P.Geo
10	Deposit Types	AGL	Keith McCandlish, P.Geo
11	Mineralization	AGL	Keith McCandlish, P.Geo
12	Exploration	AGL	Keith McCandlish, P.Geo
13	Drilling	AGL	Keith McCandlish, P.Geo
14	Sample Method and Approach	AGL	Keith McCandlish, P.Geo
15	Sample Preparation, Analyses and Security	AGL	Keith McCandlish, P.Geo
16	Data Verification	AGL	Keith McCandlish, P.Geo
17	Adjacent Properties	SE	Matt Bender, P.E.
18	Mineral Processing and Metallurgical Testing	SE	Matt Bender, P.E.
19.1 – 19.6 19.7	Mineral Resource and Mineral Reserve Estimates	AGL SE	Keith McCandlish, P.Geo Matt Bender, P.E.
20	Other Relevant Data and Information	SE	Matt Bender, P.E.
21	Interpretation and Conclusions	SE	Matt Bender, P.E.
22	Recommendations	SE	Matt Bender, P.E.
23	References	SE	Matt Bender, P.E.
24	Date and Signature Page	SE AGL	Matt Bender, P.E. Keith McCandlish, P.Geo
25	Additional Requirements for Technical Reports on Development Properties and Production Properties	SE	Matt Bender, P.E.
26	Illustrations	SE	Matt Bender, P.E.

Abbreviations: AGL – Associated Geosciences, Ltd; SE – Samuel Engineering, Inc.

5.0 Reliance on Other Experts

5.1 Reliance on Other Experts

This report was prepared for 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 the responsible Qualified Persons for this technical report.

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 and its Legal Firm to provide various lists of mining claims, claim maps and a copy of the option agreement.

Mr. Darren B. Fach of McLeod & Company, legal counsel for Copper Fox Metals, Inc., stated that Copper Fox satisfied the terms and conditions of the option agreement with Teck Cominco as of the effective date of the report in a letter dated May 17, 2010.

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.

5.2 Contributors

Numerous Persons have contributed the information used in the preparation of this technical report including staff and consultants from Copper Fox, Associated Geosciences Ltd., Moose Mountain Technical Services, Rescan Environmental Services Ltd., DST Consulting Engineers Inc., BGC Engineering, Hyyppa Engineering, LLC, McElhanney Consulting Services Ltd., HM Hamilton & Associates Inc., PR Associates, Walter Hanych, Knight Piésold, G&T Metallurgical Services Ltd., Vandan Suhbatar and Samuel Engineering Inc.

6.0 Property Description and Location

6.1 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, and situated on the eastern edge of the Coastal Mountain Range in north central 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, as shown in Figure 6.1.

Referenced to Energy, Mines, and Resource Canada topographic sheet 104G, Telegraph Creek, the geographic co-ordinate at the campsite is 57º 21' 15" north latitude, 131º 0' 58" 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 northeast of the camp.

Figure 6.1 Schaft Creek Location Map

6.2 Mineral Tenure

The Schaft Creek property is composed of two claim blocks owned by Teck Cominco Ltd. The North Block is composed of 36 contiguous claims and covers an area of 19,560 hectares. The Schaft Creek deposit is located on the south boundary of claim 514603 and on the north boundary of claim 514637. The South Block is located about 600m to the south and is composed of four claims that cover 1,358 hectares.

About 6 km further south, Copper Fox Metals Inc. owns nine claims along Mess Creek covering an area of about 3,925 hectares, see Figure 6.2. The claims and their current status are listed in Table 6.1.

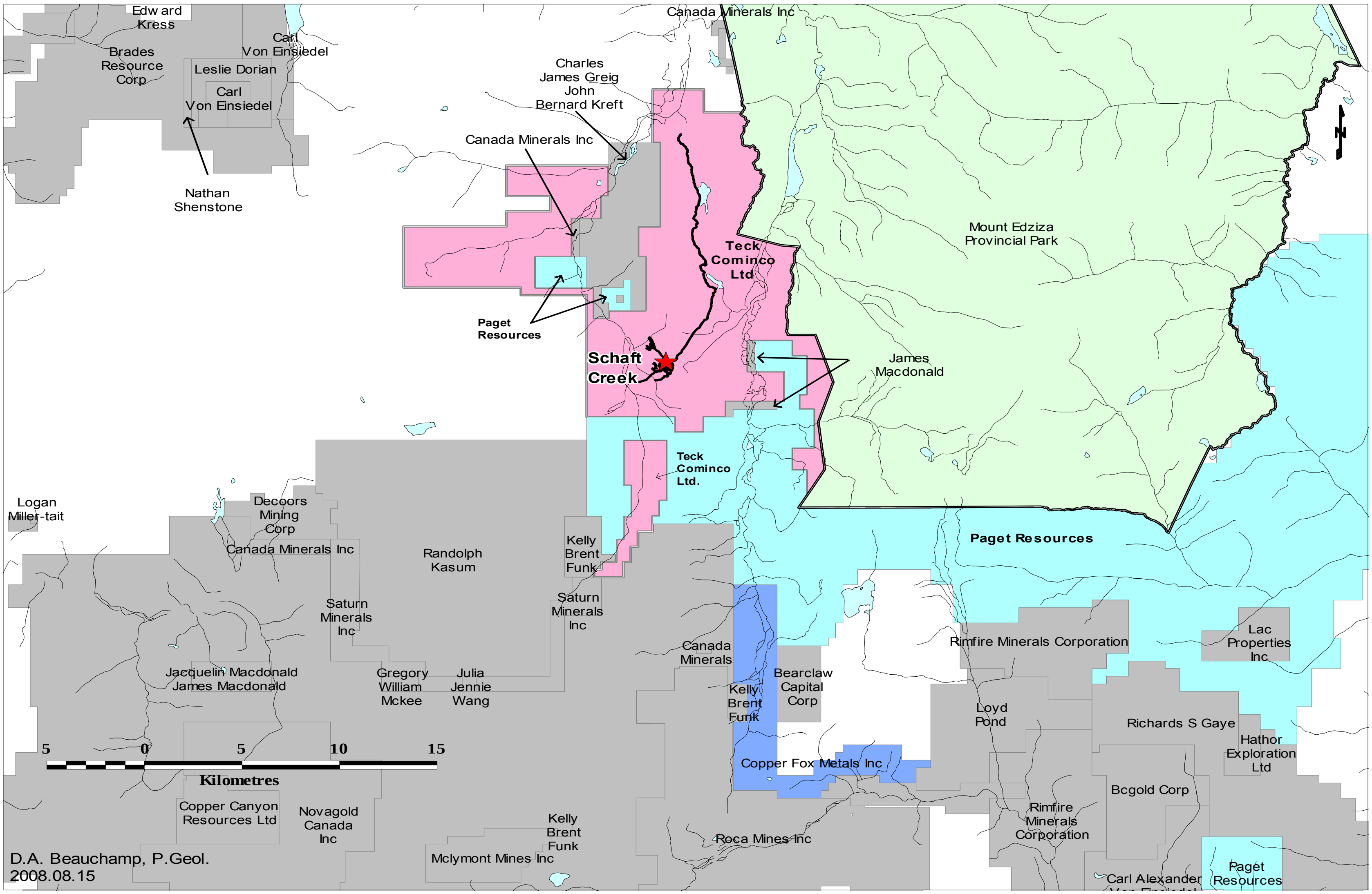


Figure 6.2 Schaft Creek Project Claims

Table 6.1 Property Claims					
Claim Number	**Owner**	**Percent Interest**	**Claim Name**	**Record Date**	**Renewal Date**
566166	1698727 Ontario Inc	100	STIKINE ASIANDA	18-Sep-07	18-Sep-08
515586	Bcgold Corp	100		29-Jun-05	14-Sep-11
516217	Bcgold Corp	100		7-Jul-05	14-Sep-11
516218	Bcgold Corp	100	NICKY EXTENSION	7-Jul-05	14-Sep-11
512807	Bearclaw Capital Corp	100		17-May-05	7-Aug-08
558766	Brades Resource Corp	100		16-May-07	1-Sep-09
558776	Brades Resource Corp	100		16-May-07	1-Sep-09
558777	Brades Resource Corp	100		16-May-07	1-Sep-09
558785	Brades Resource Corp	100		16-May-07	1-Sep-09
548574	Canada Minerals Inc	100	IN 32	4-Jan-07	10-Nov-08
548843	Canada Minerals Inc	100	COMINCO FRACTION IN	7-Jan-07	30-Jul-08
548896	Canada Minerals Inc	100	BIK-ROC	8-Jan-07	8-Jan-09
577025	Copper Fox Metals Inc	100	SC SOUTH 1	23-Feb-08	23-Feb-09
577026	Copper Fox Metals Inc	100	SC SOUTH 2	23-Feb-08	23-Feb-09
577028	Copper Fox Metals Inc	100	SC SOUTH 3	23-Feb-08	23-Feb-09
577031	Copper Fox Metals Inc	100	SC SOUTH 4	23-Feb-08	23-Feb-09
577033	Copper Fox Metals Inc	100	SC SOUTH 5	23-Feb-08	23-Feb-09
577034	Copper Fox Metals Inc	100	SC SOUTH 6	23-Feb-08	23-Feb-09
577037	Copper Fox Metals Inc	100	SC SOUTH 7	23-Feb-08	23-Feb-09
577039	Copper Fox Metals Inc	100	SC SOUTH 8	23-Feb-08	23-Feb-09
577042	Copper Fox Metals Inc	100	SC SOUTH 9	23-Feb-08	23-Feb-09
564056	Funk Kelly Brent	100		3-Aug-07	3-Aug-08
564057	Funk Kelly Brent	100		3-Aug-07	3-Aug-08
564058	Funk Kelly Brent	100		3-Aug-07	3-Aug-08
572459	Funk Kelly Brent	100	150E ROCK GEO 43336	24-Dec-07	24-Dec-08
572460	Funk Kelly Brent	100	ALMOND ROCA	24-Dec-07	24-Dec-08
572480	Funk Kelly Brent	100	GOLDEYE	25-Dec-07	25-Dec-08
586001	Funk Kelly Brent	100		8-Jun-08	8-Jun-09
517462	Greig Charles James; Kreft John Bernard	50; 50		12-Jul-05	11-Sep-08

Table 6.1 Property Claims					
569460	Greig Charles James; Kreft John Bernard	50; 50	GREATER KOPPER	5-Nov-07	11-Sep-08
584475	Kasum Randolph Micheal	100	WLC 4	17-May-08	17-May-09
584505	Kasum Randolph Micheal	100	WLC 5	18-May-08	18-May-09
584513	Kasum Randolph Micheal	100	WLC 6	18-May-08	18-May-09
584517	Kasum Randolph Micheal	100	WLC 7	18-May-08	18-May-09
584519	Kasum Randolph Micheal	100	WLC 8	18-May-08	18-May-09
584528	Kasum Randolph Micheal	100	WLC 12	18-May-08	18-May-09
584529	Kasum Randolph Micheal	100	WCL 13	18-May-08	18-May-09
584530	Kasum Randolph Micheal	100	WLC 14	18-May-08	18-May-09
584531	Kasum Randolph Micheal	100	WLC15	18-May-08	18-May-09
584532	Kasum Randolph Micheal	100	WLC 16	18-May-08	18-May-09
584533	Kasum Randolph Micheal	100	WLC 17	18-May-08	18-May-09
584534	Kasum Randolph Micheal	100	WLC 18	18-May-08	18-May-09
584535	Kasum Randolph Micheal	100	WLC 19	18-May-08	18-May-09
584536	Kasum Randolph Micheal	100	WLC 20	18-May-08	18-May-09
584537	Kasum Randolph Micheal	100	WLC 21	18-May-08	18-May-09
584538	Kasum Randolph Micheal	100	WLC 22	18-May-08	18-May-09
584539	Kasum Randolph Micheal	100	WLC 23	18-May-08	18-May-09
584541	Kasum Randolph Micheal	100	WLC 25	18-May-08	18-May-09
584542	Kasum Randolph Micheal	100	WLC 26	18-May-08	18-May-09
584543	Kasum Randolph Micheal	100	WLC 27	18-May-08	18-May-09
584544	Kasum Randolph Micheal	100	WLC 28	18-May-08	18-May-09
584545	Kasum Randolph Micheal	100	WLC 29	18-May-08	18-May-09
584546	Kasum Randolph Micheal	100	WLC 30	18-May-08	18-May-09
584548	Kasum Randolph Micheal	100	WLC 31	18-May-08	18-May-09
586482	Kasum Randolph Micheal	100	WLC 34	17-Jun-08	17-Jun-09
586483	Kasum Randolph Micheal	100	WLC 36	17-Jun-08	17-Jun-09
586484	Kasum Randolph Micheal	100	WLC 37	17-Jun-08	17-Jun-09
586485	Kasum Randolph Micheal	100	WLC 38	17-Jun-08	17-Jun-09
586486	Kasum Randolph Micheal	100	WLC 39	17-Jun-08	17-Jun-09
577690	Leslie Dorian	100		2-Mar-08	2-Mar-09

Table 6.1
Property Claims

577693	Leslie Dorian	100		2-Mar-08	2-Mar-09
504843	Macdonald James Michael	100	JAMAC 6	25-Jan-05	25-Jan-09
549077	Macdonald James Michael	100	JIMMYMAC	10-Jan-07	25-Jan-09
501238	Paget Resources Corporation	100	DA1	12-Jan-05	12-Jan-13
504761	Paget Resources Corporation	100	Mal 1	25-Jan-05	25-Jan-13
510325	Paget Resources Corporation	100	DA 2	7-Apr-05	7-Apr-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
515037	Paget Resources Corporation	100	CHAIN1	22-Jun-05	22-Jun-13
515038	Paget Resources Corporation	100	CHAIN2	22-Jun-05	22-Jun-13
515039	Paget Resources Corporation	100	CHAIN4	22-Jun-05	22-Jun-13
515040	Paget Resources Corporation	100	CHAIN3	22-Jun-05	22-Jun-13
515050	Paget Resources Corporation	100	GOAT	23-Jun-05	23-Jun-13
515051	Paget Resources Corporation	100	PARIS	23-Jun-05	23-Jun-13
515052	Paget Resources Corporation	100	HILTON	23-Jun-05	23-Jun-13
515053	Paget Resources Corporation	100	VELVET	23-Jun-05	23-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	SHAFT 666	23-Jan-06	23-Jan-13
526287	Paget Resources Corporation	100	SHAFT 667	25-Jan-06	25-Jan-13
526294	Paget Resources Corporation	100	SHAFT 668	26-Jan-06	26-Jan-13
526295	Paget Resources Corporation	100	SHAFT 669	26-Jan-06	26-Jan-13
526490	Paget Resources Corporation	100	SHAFT 670	27-Jan-06	27-Jan-13
526726	Paget Resources Corporation	100	MESS 4	30-Jan-06	30-Jan-13
527062	Paget Resources Corporation	100	HP 1	3-Feb-06	3-Feb-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 6.1
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	30-Jan-09
548807	Paget Resources Corporation	100	COURTNEY LOVE	6-Jan-07	30-Jan-09
548880	Paget Resources Corporation	100	TORI 1	8-Jan-07	10-Jan-13
548881	Paget Resources Corporation	100	AMOS 1	8-Jan-07	10-Jan-13
548882	Paget Resources Corporation	100	BJORK	8-Jan-07	10-Jan-13
548883	Paget Resources Corporation	100	DAFT PUNK	8-Jan-07	10-Jan-13
548884	Paget Resources Corporation	100	FISHERSPOONER	8-Jan-07	10-Jan-13
548889	Paget Resources Corporation	100	FROU FROU	8-Jan-07	10-Jan-13
551352	Paget Resources Corporation	100	MESS 6	7-Feb-07	10-Jan-13
551358	Paget Resources Corporation	100	MESS 7	7-Feb-07	10-Jan-13
551495	Paget Resources Corporation	100	MESS 8	9-Feb-07	10-Jan-13
551510	Paget Resources Corporation	100	MESS 9	9-Feb-07	10-Jan-13
553795	Paget Resources Corporation	100	BALL NX1	7-Mar-07	10-Jan-13
553796	Paget Resources Corporation	100	BALL NX2	7-Mar-07	10-Jan-13

Table 6.1
Property Claims

553798	Paget Resources Corporation	100	BALL NX4	7-Mar-07	10-Jan-13
553799	Paget Resources Corporation	100	BALL NX5	7-Mar-07	10-Jan-13
553811	Paget Resources Corporation	100	FLATS 4	7-Mar-07	10-Jan-13
553812	Paget Resources Corporation	100	FLAT 5	7-Mar-07	10-Jan-13
575411	Paget Resources Corporation	100	BAM NORTH	6-Feb-08	6-Feb-09
575412	Paget Resources Corporation	100	BAM NORTH A	6-Feb-08	6-Feb-09
560692	Pond Loyd	100		15-Jun-07	15-Jun-10
564150	Pond Loyd	100	HHE-2	4-Aug-07	4-Aug-08
564151	Pond Loyd	100	HHE-3	4-Aug-07	4-Aug-08
564152	Pond Loyd	100	HHE-4	4-Aug-07	4-Aug-08
564166	Pond Loyd	100	HHE-5	5-Aug-07	5-Aug-08
564171	Pond Loyd	100	HHE_6	5-Aug-07	5-Aug-08
564173	Pond Loyd	100	HHE-7	5-Aug-07	5-Aug-08
564174	Pond Loyd	100	HHE-8	5-Aug-07	5-Aug-08
564175	Pond Loyd	100	HHE-9	5-Aug-07	5-Aug-08
577970	Richards S Gaye	100	POTENTIAL 1	6-Mar-08	13-Mar-09
577971	Richards S Gaye	100	POTENTIAL 2	6-Mar-08	13-Mar-09
577973	Richards S Gaye	100	POTENTIAL 4	6-Mar-08	13-Mar-09
501122	Rimfire Minerals Corporation	100		12-Jan-05	31-Dec-16
501153	Rimfire Minerals Corporation	100		12-Jan-05	31-Dec-16
504758	Rimfire Minerals Corporation	100	Narby 1	25-Jan-05	31-Dec-16
504799	Rimfire Minerals Corporation	100	KC 2	25-Jan-05	31-Dec-16
510806	Rimfire Minerals Corporation	100	BOGDEN	15-Apr-05	31-Dec-16
535726	Rimfire Minerals Corporation	100	RDN 19	15-Jun-06	31-Dec-16
535728	Rimfire Minerals Corporation	100	RDN 20	15-Jun-06	31-Dec-16
535729	Rimfire Minerals Corporation	100	RDN 21	15-Jun-06	31-Dec-16
569756	Rimfire Minerals Corporation	100	GRIZZLYNORTH1	9-Nov-07	9-Nov-08
569757	Rimfire Minerals Corporation	100	GRIZZLYNORTH2	9-Nov-07	9-Nov-08
569758	Rimfire Minerals Corporation	100	GRIZZLYNORTH3	9-Nov-07	9-Nov-08
569759	Rimfire Minerals Corporation	100	GRIZZLYNORTH4	9-Nov-07	9-Nov-08
569760	Rimfire Minerals Corporation	100	GRIZZLYNORTH5	9-Nov-07	9-Nov-08

Table 6.1
Property Claims

569761	Rimfire Minerals Corporation	100	GRIZZLYNORTH6	9-Nov-07	9-Nov-08
569762	Rimfire Minerals Corporation	100	GRIZZLYNORTH7	9-Nov-07	9-Nov-08
569763	Rimfire Minerals Corporation	100	GRIZZLYNORTH8	9-Nov-07	9-Nov-08
569764	Rimfire Minerals Corporation	100	GRIZZLYNORTH9	9-Nov-07	9-Nov-08
393458	Roca Mines Inc	100	ANT 1	20-May-02	30-Sep-08
393459	Roca Mines Inc	100	ANT 2	20-May-02	30-Sep-08
400294	Roca Mines Inc	100	ROC 8	13-Feb-03	30-Sep-08
400295	Roca Mines Inc	100	ROC 9	13-Feb-03	30-Sep-08
400296	Roca Mines Inc	100	ROC 10	13-Feb-03	30-Sep-08
400297	Roca Mines Inc	100	ROC 11	13-Feb-03	30-Sep-08
400298	Roca Mines Inc	100	ROC 12	13-Feb-03	30-Sep-08
400299	Roca Mines Inc	100	ROC 13	13-Feb-03	30-Sep-08
400300	Roca Mines Inc	100	ROC 14	13-Feb-03	30-Sep-08
406128	Roca Mines Inc	100	DICE 1	8-Oct-03	30-Sep-08
537207	Roca Mines Inc	100	ROCATOWN	14-Jul-06	30-Sep-08
537208	Roca Mines Inc	100	ROCATOWN	14-Jul-06	30-Sep-08
540082	Roca Mines Inc	100	ROCA FLATS #1	29-Aug-06	30-Sep-08
540083	Roca Mines Inc	100	ROCA FLATS #2	29-Aug-06	30-Sep-08
586005	Saturn Minerals Inc	100	TOBY THREE	8-Jun-08	8-Jun-09
586006	Saturn Minerals Inc	100	TOBY FOUR	8-Jun-08	8-Jun-09
586008	Saturn Minerals Inc	100	TOBY FIVE	8-Jun-08	8-Jun-09
514595	Teck Cominco Ltd	100		16-Jun-05	30-Oct-18
514596	Teck Cominco Ltd	100		16-Jun-05	30-Oct-18
514598	Teck Cominco Ltd	100		16-Jun-05	30-Oct-18
514603	Teck Cominco Ltd	100		16-Jun-05	30-Oct-18
514637	Teck Cominco Ltd	100		17-Jun-05	30-Oct-18
514721	Teck Cominco Ltd	100		17-Jun-05	30-Oct-18
514723	Teck Cominco Ltd	100		17-Jun-05	30-Oct-18
514724	Teck Cominco Ltd	100		17-Jun-05	30-Oct-18
514725	Teck Cominco Ltd	100		17-Jun-05	30-Oct-18
514728	Teck Cominco Ltd	100		17-Jun-05	30-Oct-18

Table 6.1 Property Claims					
515035	Teck Cominco Ltd	100		22-Jun-05	30-Oct-18
515036	Teck Cominco Ltd	100		22-Jun-05	30-Oct-18
547789	Teck Cominco Ltd	100		21-Dec-06	21-Dec-18
547798	Teck Cominco Ltd	100		21-Dec-06	21-Dec-18
548487	Teck Cominco Ltd	100	BLOCK B1	2-Jan-07	15-Jan-18
548488	Teck Cominco Ltd	100	BLOCK B2	2-Jan-07	15-Jan-18
548489	Teck Cominco Ltd	100	BLOCK B3	2-Jan-07	15-Jan-18
548490	Teck Cominco Ltd	100	BLOCK B4	2-Jan-07	15-Jan-18
548492	Teck Cominco Ltd	100	BLOCK C1	2-Jan-07	15-Jan-18
548493	Teck Cominco Ltd	100	BLOCK C2	2-Jan-07	15-Jan-18
548494	Teck Cominco Ltd	100	BLOCK C3	2-Jan-07	15-Jan-18
548495	Teck Cominco Ltd	100	BLOCK C4	2-Jan-07	15-Jan-18
548496	Teck Cominco Ltd	100	BLOCK C5	2-Jan-07	15-Jan-18
548498	Teck Cominco Ltd	100	BLOCK C6	2-Jan-07	15-Jan-18
548759	Teck Cominco Ltd	100	AREA A	5-Jan-07	15-Jan-18
548760	Teck Cominco Ltd	100	AREA C1	5-Jan-07	5-Jan-18
548761	Teck Cominco Ltd	100	AREA C2	5-Jan-07	5-Jan-18
548762	Teck Cominco Ltd	100	AREA C3	5-Jan-07	5-Jan-18
548763	Teck Cominco Ltd	100	AREA C4	5-Jan-07	5-Jan-18
548764	Teck Cominco Ltd	100	AREA B1	5-Jan-07	15-Jan-18
548766	Teck Cominco Ltd	100	AREA B2	5-Jan-07	15-Jan-18
548767	Teck Cominco Ltd	100	AREA B3	5-Jan-07	15-Jan-18
548768	Teck Cominco Ltd	100	AREA B4	5-Jan-07	15-Jan-18
548769	Teck Cominco Ltd	100	AREA B5	5-Jan-07	15-Jan-18
548770	Teck Cominco Ltd	100	AREA B6	5-Jan-07	15-Jan-18
548771	Teck Cominco Ltd	100	AREA B7	5-Jan-07	15-Jan-18
548772	Teck Cominco Ltd	100	AREA B8	5-Jan-07	15-Jan-18
551325	Teck Cominco Ltd	100	AREA D1	6-Feb-07	6-Feb-18
551326	Teck Cominco Ltd	100	AREA D2	6-Feb-07	6-Feb-18
551328	Teck Cominco Ltd	100	AREA D3	6-Feb-07	6-Feb-18
564762	Von Einsiedel Carl Alexander	100	WIN AA	17-Aug-07	17-Aug-08

Table 6.1
Property Claims

Claim Number	Owner	Percent Interest	Claim Name	Record Date	Renewal Date
564765	Von Einsiedel Carl Alexander	100	WIN BB	17-Aug-07	17-Aug-08
564767	Von Einsiedel Carl Alexander	100	WIN CC	17-Aug-07	17-Aug-08
534991	Wang Julia Jennie	100	GALORE GOLD	6-Jun-06	6-Oct-08
583973	Wang Julia Jennie	100	MOUNT EDZIZA 1	11-May-08	11-May-09
566166	1698727 Ontario Inc	100	STIKINE ASIANDA	18-Sep-07	18-Sep-08
515586	Bcgold Corp	100		29-Jun-05	14-Sep-11
516217	Bcgold Corp	100		7-Jul-05	14-Sep-11
516218	Bcgold Corp	100	NICKY EXTENSION	7-Jul-05	14-Sep-11
512807	Bearclaw Capital Corp	100		17-May-05	7-Aug-08
558766	Brades Resource Corp	100		16-May-07	1-Sep-09
558776	Brades Resource Corp	100		16-May-07	1-Sep-09
558777	Brades Resource Corp	100		16-May-07	1-Sep-09
558785	Brades Resource Corp	100		16-May-07	1-Sep-09
548574	Canada Minerals Inc	100	IN 32	4-Jan-07	10-Nov-08
548843	Canada Minerals Inc	100	COMINCO FRACTION IN	7-Jan-07	30-Jul-08
548896	Canada Minerals Inc	100	BIK-ROC	8-Jan-07	8-Jan-09
577025	Copper Fox Metals Inc	100	SC SOUTH 1	23-Feb-08	23-Feb-09
577026	Copper Fox Metals Inc	100	SC SOUTH 2	23-Feb-08	23-Feb-09
577028	Copper Fox Metals Inc	100	SC SOUTH 3	23-Feb-08	23-Feb-09
577031	Copper Fox Metals Inc	100	SC SOUTH 4	23-Feb-08	23-Feb-09
577033	Copper Fox Metals Inc	100	SC SOUTH 5	23-Feb-08	23-Feb-09
577034	Copper Fox Metals Inc	100	SC SOUTH 6	23-Feb-08	23-Feb-09
577037	Copper Fox Metals Inc	100	SC SOUTH 7	23-Feb-08	23-Feb-09
577039	Copper Fox Metals Inc	100	SC SOUTH 8	23-Feb-08	23-Feb-09
577042	Copper Fox Metals Inc	100	SC SOUTH 9	23-Feb-08	23-Feb-09
564056	Funk Kelly Brent	100		3-Aug-07	3-Aug-08
564057	Funk Kelly Brent	100		3-Aug-07	3-Aug-08
564058	Funk Kelly Brent	100		3-Aug-07	3-Aug-08
572459	Funk Kelly Brent	100	150E ROCK GEO 43336	24-Dec-07	24-Dec-08

Table 6.1
Property Claims

572460	Funk Kelly Brent	100	ALMOND ROCA	24-Dec-07	24-Dec-08
572480	Funk Kelly Brent	100	GOLDEYE	25-Dec-07	25-Dec-08
586001	Funk Kelly Brent	100		8-Jun-08	8-Jun-09
517462	Greig Charles James; Kreft John Bernard	50; 50		12-Jul-05	11-Sep-08
569460	Greig Charles James; Kreft John Bernard	50; 50	GREATER KOPPER	5-Nov-07	11-Sep-08
584475	Kasum Randolph Micheal	100	WLC 4	17-May-08	17-May-09
584505	Kasum Randolph Micheal	100	WLC 5	18-May-08	18-May-09
584513	Kasum Randolph Micheal	100	WLC 6	18-May-08	18-May-09
584517	Kasum Randolph Micheal	100	WLC 7	18-May-08	18-May-09
584519	Kasum Randolph Micheal	100	WLC 8	18-May-08	18-May-09
584528	Kasum Randolph Micheal	100	WLC 12	18-May-08	18-May-09
584529	Kasum Randolph Micheal	100	WCL 13	18-May-08	18-May-09
584530	Kasum Randolph Micheal	100	WLC 14	18-May-08	18-May-09
584531	Kasum Randolph Micheal	100	WLC15	18-May-08	18-May-09
584532	Kasum Randolph Micheal	100	WLC 16	18-May-08	18-May-09
584533	Kasum Randolph Micheal	100	WLC 17	18-May-08	18-May-09
584534	Kasum Randolph Micheal	100	WLC 18	18-May-08	18-May-09
584535	Kasum Randolph Micheal	100	WLC 19	18-May-08	18-May-09
584536	Kasum Randolph Micheal	100	WLC 20	18-May-08	18-May-09
584537	Kasum Randolph Micheal	100	WLC 21	18-May-08	18-May-09
584538	Kasum Randolph Micheal	100	WLC 22	18-May-08	18-May-09
584539	Kasum Randolph Micheal	100	WLC 23	18-May-08	18-May-09
584541	Kasum Randolph Micheal	100	WLC 25	18-May-08	18-May-09
584542	Kasum Randolph Micheal	100	WLC 26	18-May-08	18-May-09
584543	Kasum Randolph Micheal	100	WLC 27	18-May-08	18-May-09
584544	Kasum Randolph Micheal	100	WLC 28	18-May-08	18-May-09
584545	Kasum Randolph Micheal	100	WLC 29	18-May-08	18-May-09
584546	Kasum Randolph Micheal	100	WLC 30	18-May-08	18-May-09
584548	Kasum Randolph Micheal	100	WLC 31	18-May-08	18-May-09
586482	Kasum Randolph Micheal	100	WLC 34	17-Jun-08	17-Jun-09
586483	Kasum Randolph Micheal	100	WLC 36	17-Jun-08	17-Jun-09

Table 6.1
Property Claims

586484	Kasum Randolph Micheal	100	WLC 37	17-Jun-08	17-Jun-09
586485	Kasum Randolph Micheal	100	WLC 38	17-Jun-08	17-Jun-09
586486	Kasum Randolph Micheal	100	WLC 39	17-Jun-08	17-Jun-09
577690	Leslie Dorian	100		2-Mar-08	2-Mar-09
577693	Leslie Dorian	100		2-Mar-08	2-Mar-09
504843	Macdonald James Michael	100	JAMAC 6	25-Jan-05	25-Jan-09
549077	Macdonald James Michael	100	JIMMYMAC	10-Jan-07	25-Jan-09
501238	Paget Resources Corporation	100	DA1	12-Jan-05	12-Jan-13
504761	Paget Resources Corporation	100	Mal 1	25-Jan-05	25-Jan-13
510325	Paget Resources Corporation	100	DA 2	7-Apr-05	7-Apr-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
515037	Paget Resources Corporation	100	CHAIN1	22-Jun-05	22-Jun-13
515038	Paget Resources Corporation	100	CHAIN2	22-Jun-05	22-Jun-13
515039	Paget Resources Corporation	100	CHAIN4	22-Jun-05	22-Jun-13
515040	Paget Resources Corporation	100	CHAIN3	22-Jun-05	22-Jun-13
515050	Paget Resources Corporation	100	GOAT	23-Jun-05	23-Jun-13
515051	Paget Resources Corporation	100	PARIS	23-Jun-05	23-Jun-13
515052	Paget Resources Corporation	100	HILTON	23-Jun-05	23-Jun-13
515053	Paget Resources Corporation	100	VELVET	23-Jun-05	23-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	SHAFT 666	23-Jan-06	23-Jan-13
526287	Paget Resources Corporation	100	SHAFT 667	25-Jan-06	25-Jan-13
526294	Paget Resources Corporation	100	SHAFT 668	26-Jan-06	26-Jan-13
526295	Paget Resources Corporation	100	SHAFT 669	26-Jan-06	26-Jan-13
526490	Paget Resources Corporation	100	SHAFT 670	27-Jan-06	27-Jan-13
526726	Paget Resources Corporation	100	MESS 4	30-Jan-06	30-Jan-13
527062	Paget Resources Corporation	100	HP 1	3-Feb-06	3-Feb-13

Table 6.1 Property Claims					
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
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	30-Jan-09
548807	Paget Resources Corporation	100	COURTNEY LOVE	6-Jan-07	30-Jan-09
548880	Paget Resources Corporation	100	TORI 1	8-Jan-07	10-Jan-13
548881	Paget Resources Corporation	100	AMOS 1	8-Jan-07	10-Jan-13
548882	Paget Resources Corporation	100	BJORK	8-Jan-07	10-Jan-13
548883	Paget Resources Corporation	100	DAFT PUNK	8-Jan-07	10-Jan-13
548884	Paget Resources Corporation	100	FISHERSPOONER	8-Jan-07	10-Jan-13
548889	Paget Resources Corporation	100	FROU FROU	8-Jan-07	10-Jan-13
551352	Paget Resources Corporation	100	MESS 6	7-Feb-07	10-Jan-13
551358	Paget Resources Corporation	100	MESS 7	7-Feb-07	10-Jan-13

Table 6.1 Property Claims					
551495	Paget Resources Corporation	100	MESS 8	9-Feb-07	10-Jan-13
551510	Paget Resources Corporation	100	MESS 9	9-Feb-07	10-Jan-13
553795	Paget Resources Corporation	100	BALL NX1	7-Mar-07	10-Jan-13
553796	Paget Resources Corporation	100	BALL NX2	7-Mar-07	10-Jan-13
553798	Paget Resources Corporation	100	BALL NX4	7-Mar-07	10-Jan-13
553799	Paget Resources Corporation	100	BALL NX5	7-Mar-07	10-Jan-13
553811	Paget Resources Corporation	100	FLATS 4	7-Mar-07	10-Jan-13
553812	Paget Resources Corporation	100	FLAT 5	7-Mar-07	10-Jan-13
575411	Paget Resources Corporation	100	BAM NORTH	6-Feb-08	6-Feb-09
575412	Paget Resources Corporation	100	BAM NORTH A	6-Feb-08	6-Feb-09
560692	Pond Loyd	100		15-Jun-07	15-Jun-10
564150	Pond Loyd	100	HHE-2	4-Aug-07	4-Aug-08
564151	Pond Loyd	100	HHE-3	4-Aug-07	4-Aug-08
564152	Pond Loyd	100	HHE-4	4-Aug-07	4-Aug-08
564166	Pond Loyd	100	HHE-5	5-Aug-07	5-Aug-08
564171	Pond Loyd	100	HHE_6	5-Aug-07	5-Aug-08
564173	Pond Loyd	100	HHE-7	5-Aug-07	5-Aug-08
564174	Pond Loyd	100	HHE-8	5-Aug-07	5-Aug-08

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 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" (MOU). 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."

6.3 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 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 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 PFS requirements.

6.4 Environmental Liabilities

A review of the permitting and environmental activities for the Schaft Creek project has been provided by Copper Fox Metals Inc. (Copper Fox). Copper Fox is undertaking exploration and geotechnical investigations and environmental baseline studies at the Schaft Creek property. These activities are supported by a 60-person camp and an existing airstrip. Reclamation of the property is required once these activities are complete in the event that no further development is planned. Copper Fox has posted a reclamation bond with the British Columbia (BC) 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 BC 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. A general reclamation plan has been provided to the BC MEMPR.

6.5 Permits

Copper Fox is currently working under an amended Notice of Work (NoW) and Reclamation Program from BC MEMPR under the BC Mines Act (Permit #: MX-1-647, Amendment #: 07-0100455-0530; Mine #: 01400455). Use of the Schaft Creek airstrip is also covered under the NoW and Reclamation Program. The amended NoW was submitted to BC MEMPR on April 11, 2008. The BC MEMPR acknowledged receipt of Copper Fox's amended NoW on April 22, 2008. The BC MEMPR sends the permit amendment application to various government agencies and First Nations for review. These parties have 30 days to review the amendment application.

Copper Fox has also obtained an Occupant License to Cut from the BC Ministry of Forests (Permit #: L47555). This permit allows Copper Fox to cut trees for the purpose of exploration activities. The permit is valid until June 14, 2009.

Copper Fox has posted an additional $35,000 security deposit prior to undertaking surface work for 2008. Copper Fox's bond with the BC Government total $235,000 to date.

In addition to the BC Mines Act, other provincial legislation applies to the work program planned for 2008 at Schaft Creek property. Relevant BC legislation relates to drinking water and the discharge of waste water on site. Through the course of the 2008 work program Copper Fox may require a license to divert water from a stream under the Water Act. If required, this license is readily obtained from the BC Ministry of Environment and does not pose a risk to the 2008 work program.

Copper Fox will also be upgrading its water supply system at the Schaft Creek existing camp. This may require a Construction Permit (section 7(2)) and an Operating Permit (section 8(1)) under the Drinking Water Protection Act. Both permits are readily obtained. The Drinking Water Protection Regulation (section 12) also applies to the distribution of potable water. Details of the Regulation are understood by Copper Fox and will be followed.

6.6 Permitting

The permitting and consultation process in BC is well defined and transparent. From early days, the process takes approximately two to three years before the first permits are granted to begin construction of a mining project subject to the BC Environmental Assessment Act (BCEAA) and the Canadian Environmental Assessment Act (CEAA). Prior to submitting applications for approval under CEAA and BCEAA, baseline environment studies are required. Depending on the nature of the project, a minimum of one year of data is needed prior to submitting an EA Application. Copper Fox began the environmental and social baseline programs in 2005.

The Schaft Creek project will require a BC Environmental Assessment Certificate as well as provincial permits, authorizations and licenses to construct and operate the Schaft Creek mine. The project will require a federal decision on the likelihood of environmental impacts as per 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 BC Environmental Assessment Act.

The Canadian environmental assessment process is governed by the CEAA. At this time, federal regulatory agencies have not officially stated that the CEAA applies 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. Copper Fox is confident that Natural Resources Canada, Department of Fisheries and Oceans, Environment Canada and Transport Canada will have the information necessary to determine if the CEAA applies and how the project will be reviewed under CEAA once this Preliminary Feasibility Study (PFS) is made public.

6.6.1 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 for identified impacts.

Copper Fox is currently reviewing an Order under section 11 of the BCEAA that outlines the scope of the project to be reviewed. The Order also outlines commitments to the First Nations in terms of reviewing available information prior to submitting the EA Application.

Subsequent to finalizing the Order under section 11, 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 Q4-2008/Q1-2009. The review period for the EA Application is approximately six months. During the review period, Copper Fox will complete applications for necessary permits, authorizations and licenses necessary to meet the project development timeline.

6.6.2 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.

The Schaft Creek project will likely be triggered given that a permit, license and/or authorization will be required from at least one federal regulatory agency. Copper Fox has developed an environmental and social baseline program and is in the process of preparing an EA Application that will optimistically lead to federal approval of the project under CEAA.

That is, Copper Fox has structured their environmental work plans to meet the expectations of a federal environmental assessment.

The federal agencies cannot make a decision on the above triggers until project details are available. The details required for federal regulatory agencies to determine if CEAA applies are presented in this PFS report. The information will be subsequently formatted for federal agencies to assist them in their decision-making process.

Under CEAA, projects receive a given level of environmental assessment that is specific to the size of the project and potential for environmental impacts.

There are four review levels under CEAA:

- Screening;
- Comprehensive study;
- Mediation;
- Panel review.

Based on Copper Fox's review of the CEAA and discussions with the Canadian Environmental Assessment Agency, the Schaft Creek project will require a comprehensive study.

Copper Fox is anticipating the following federal triggers:

- Transport Canada – Navigable Waters Act;
- Department of Fisheries and Oceans – Fisheries Act; Metal Mining Effluent Regulations;
- Natural Resources Canada – Explosives Act.

Environment Canada will be involved in the review of the project through the Harmonization Agreement between the federal government and the BC government.

6.6.3 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. 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. This list should not be considered comprehensive.

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);
- Construction Permit (Drinking Water Protection Act);
- Operation Permit (Drinking Water Protection Act);
- Occupant License to Cut (Forest Act);
- Special Use Permit (Forest Act);

- License of Occupation (Land Act);
- Investigative Permit (Land Act);
- Surface Lease (Land Act);
- Right of Way (Land Act);
- Highway Access Permit (Transportation Act);
- Permit to Construct a Water Works (Drinking Water Protection 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);
- Tailings Dam Permit (Letter of Application) (Navigable Waters Works Regulations);
- Explosives Factory License (Explosives Act);
- Explosives Magazine License (Explosives Act);
- Ammonium Nitrate Storage Facilities (Canada Transportation Act);
- Radio Licenses (Radio Communication Act).

Through the Concurrent Permit process established by the BC government, Copper Fox will seek concurrent permitting status for permits and authorizations needed to construction the access road. The advantage of submitting the permit applications for the access road concurrently with the EA Application is that provincial agencies have 60 days to issue concurrent permits after a decision has been made on the EA Application.

Other provincial permits and authorizations will be sought during the 180 day review period for the EA Application. This approach will ensure the timely advancement of the Schaft Creek project. No permits can be issued while the EA Application is under review.

Typically, federal agencies will not advance permit and/or license applications and authorizations until there is assurance of a positive outcome from the EA review. This is anticipated by Copper Fox and will not negatively impact the project development schedule.

Copper Fox is currently in discussions with Transport Canada regarding the permitting of the tailings storage facility (TSF) in the Skeeter Lake watershed. Skeeter Lake is potentially a navigable lake. Thus, depositing tailings and waste rock into Skeeter Lake and portions of Skeeter Creek requires an exemption to sections 22 and 23 of the Navigable Waters Protection Act (NWPA) through an Order-in-Council.

The process to obtain an exemption through an Order-in-Council takes between six and nine months. This process cannot be initiated until the EA review is underway. However, this process under NWPA will not impact the overall project timeline as tailings construction is not scheduled until after the access road is constructed and construction of the access road is estimated to take one year.

7.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

7.1 Access

7.1.1 Access

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, British Columbia and flown directly to the Schaft Creek camp, utilizing an existing gravel airstrip at the camp.

Access to the site will be via the Mess Creek access route based on preliminary studies that have been completed for the PFS. This access route extends north from More Creek along Upper Mess Creek, entering the mine-site area and Schaft Creek drainage near Snipe Lake. More information about the Access Road is provided in Section 25 of this report.

7.1.2 Proximity of Property to a Population Center

The Schaft Creek property is approximately 375 km northwest of the town of Smithers, BC. Smither 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. They are 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.

Figure 7.1 Schaft Creek Location Map

7.2 Local Resources

The Schaft Creek project is located entirely within the Tahltan Nation territory. As such, the Tahltan Nation will be consulted 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, including 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, which is located 370 km to the southwest and Terrace which is 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 the Schaft Creek project.

7.3 Existing Infrastructure

7.3.1 General

Infrastructure is all but non-existent in the immediate project area. An old, overgrown 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 by 3 km area and total approximately 10 km of gravel trails that are 4 m in width.

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

A 1,220 m 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 m and erected several new buildings. This airstrip is still in use today.

During the interval from 1968 to 1981 when Hecla Mines and subsequently Teck Corporation explored the property, most of the site infrastructure was established. This included:

- Two 9 by 46 m Quonset-style buildings;
- A fuel storage depot consisting of three 9.1 m long 3.0 m 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;
- A small pre-fabricated cedar log cabin owned by a helicopter company.

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; 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 35-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.

7.3.2 Power

There is no power available to site. The closest major power line is located approximately 150 km south in Meziadin Junction. It is estimated that the Schaft Creek project will require an average power draw of 121 MW with a maximum draw of 140 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 power line from the Schaft Creek site to join the BC Hydro grid from Highway 37 near Bob Quinn. The power line 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 (EA) 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.

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.

7.3.3 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.

7.4 Climate and Length of Operating Season

The Schaft Creek project is located on the eastern edge of the Coastal Mountain Range 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°C but hourly temperatures can vary between +30°C and -30°C. Mean monthly temperatures typically remain above freezing from April to October and drop below freezing from November through March. Annual precipitation averages from 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 dependent 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).

7.5 Physiography

The Schaft Creek property is situated on the eastern edge of the Coastal Mountain Range in north central 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 by 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.

It rises to above 2,000 m from the valley floor to snow-capped mountain peaks and ice fields within a few km 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 masl. 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, glacio-fluvial 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 sub-alpine 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.

7.6 Geohazards

7.6.1 Introduction

Due to the location of the Schaft Creek project, consideration of geohazards is particularly important. For this reason, BGC Engineering (BGC) of Vancouver, British Columbia was retained to investigate potential geohazards to support the Tailings Options Study and the selection of the most appropriate Site Access Road. For the purposes of this PFS only the selected options are explained in this report.

The Schaft Creek project access route and mine-site areas are located within the Mess Creek Watershed, which drains an area of 2,306 km^2 and is a major tributary of the Stikine River. The mine-site area and Tailings Storage Facility (TSF) site options are located in upper Schaft and Hickman Creeks which are tributaries to Mess Creek.

7.6.2 Tailings Storage Facility

7.6.2.1 *Terms of Reference and Scope of Work*

Copper Fox Metals Inc. (Copper Fox) identified three primary TSF site options in the vicinity of Schaft Creek, as shown in Figure 7.2. The most favourable option was selected based on a variety of factors, such as capital and operating costs, water management, and potential for expansion. The evaluation of geohazards included assessment of landslide and snow avalanches.

The results of the preliminary geohazards study for the TSF options are reported in *Schaft Creek Tailings Option Study, Geohazards.*

The work plan for the geohazards evaluation included:

- Terrain analysis and identification of landslide and snow avalanche geohazards at an overview level of detail;
- Comparison of relative geohazard levels for each tailings option;

- Identification of the most favourable tailings site with respect to geohazards.

7.6.2.2 *Terrain Analysis Methods*

Terrain analysis was done by Kris Holm, M.Sc., P.Geo. (KH) and reviewed by Matthias Jakob, Ph.D., P.Geo. (MJ) using aerial photographs with a scale of approximately 1:10,000 and 1:20,000. Air photo interpretations were digitized onscreen in ArcGIS. Maps and terrain descriptions provide a general overview of the distribution of geohazards. The scale of analysis completed in this work is intended for preliminary development planning and early identification of any major issues related to geohazards. In the case of snow avalanches and rockfall, the arrows on the figure shows general areas of hazard potential but do not indicate individual paths. The level of detail completed for this PFS is not appropriate for detailed site selection or design and does not replace more detailed terrain analysis and field investigation that must be completed during later phases of the project.

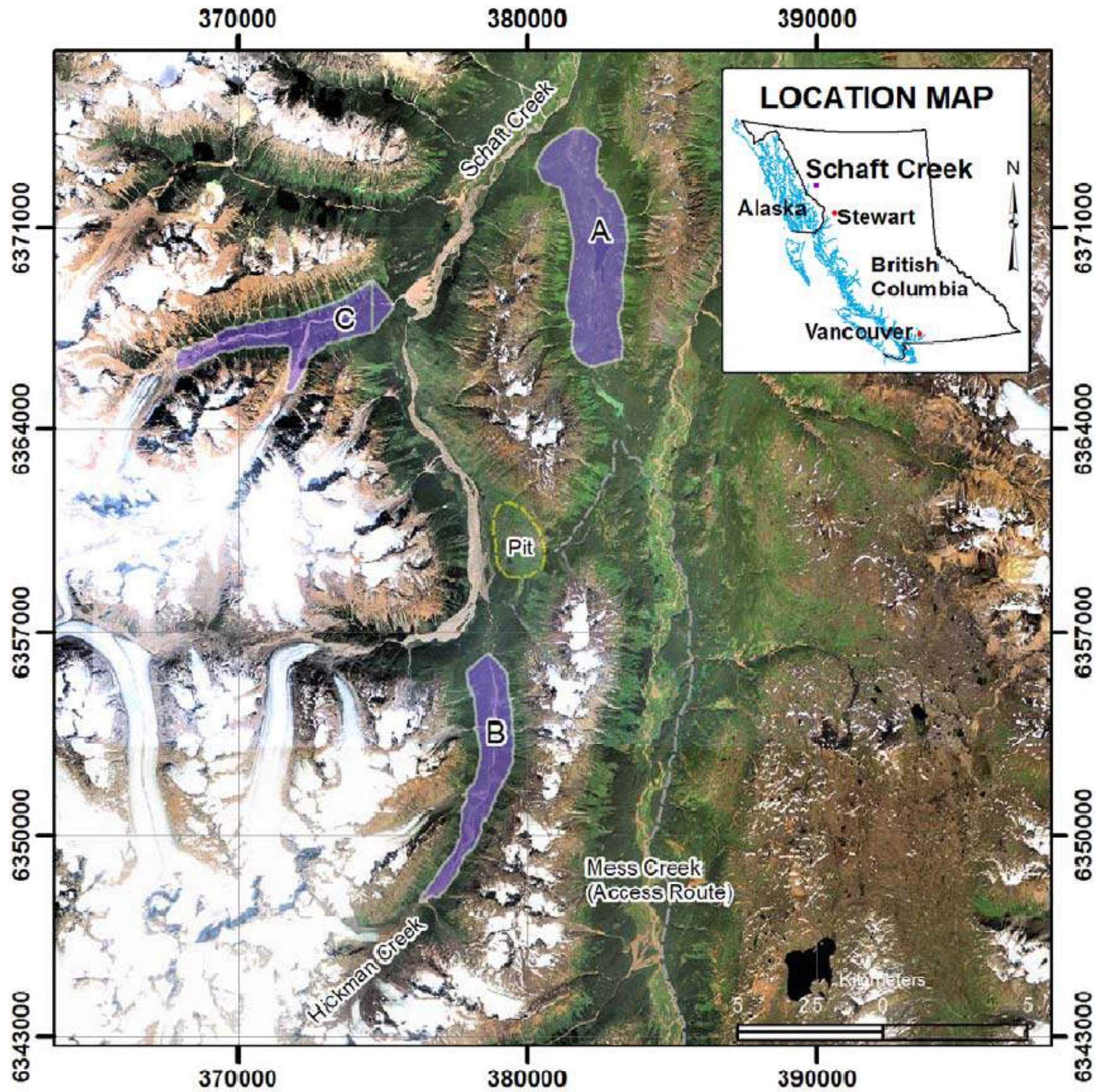


Figure 7.2 Tailings Storage Facility Locations

7.6.2.3 *TSF Site Terrain*

Tailings Option A is located in a valley trough extending north from the northeast side of Mt. LaCasse, and encompasses a 15 km^2 area surrounding Skeeter Lake as shown in Figure 5.2. The valley head is at an elevation of approximately 900 m and has not been glaciated in Holocene time, i.e. the past 10,000 years. Existing glaciers are limited to three cirque glaciers on the upper northeast slopes of Mt. LaCasse. Upper slopes contain gullied bedrock partially overlain by colluvial veneers that increase in thickness towards the valley bottom and within colluvial fans at the outlet of debris flow channels. Holocene glacial till exists in upper cirques on the west side of the valley, north of Mt. LaCasse. The valley bottom contains hummocky bedrock partially overlain by till and organic veneers in depressions. The valley follows an extensional fault zone and fault contact between Stuhini

Group volcanic rock on the west valley side and Stikine Assemblage dolomitic limestones and volcanic rock in the valley bottom and east valley side (Logan et al. 1997).

7.6.2.4 *TSF Site Identification of Geohazards*

Geohazards with the potential to impact the tailings footprint are shown in Figure 7.3. They primarily include run-out zones of snow avalanches and debris flows around the footprint perimeter. Gentle terrain in the middle of the tailings footprint is considered to have very low levels of geohazards.

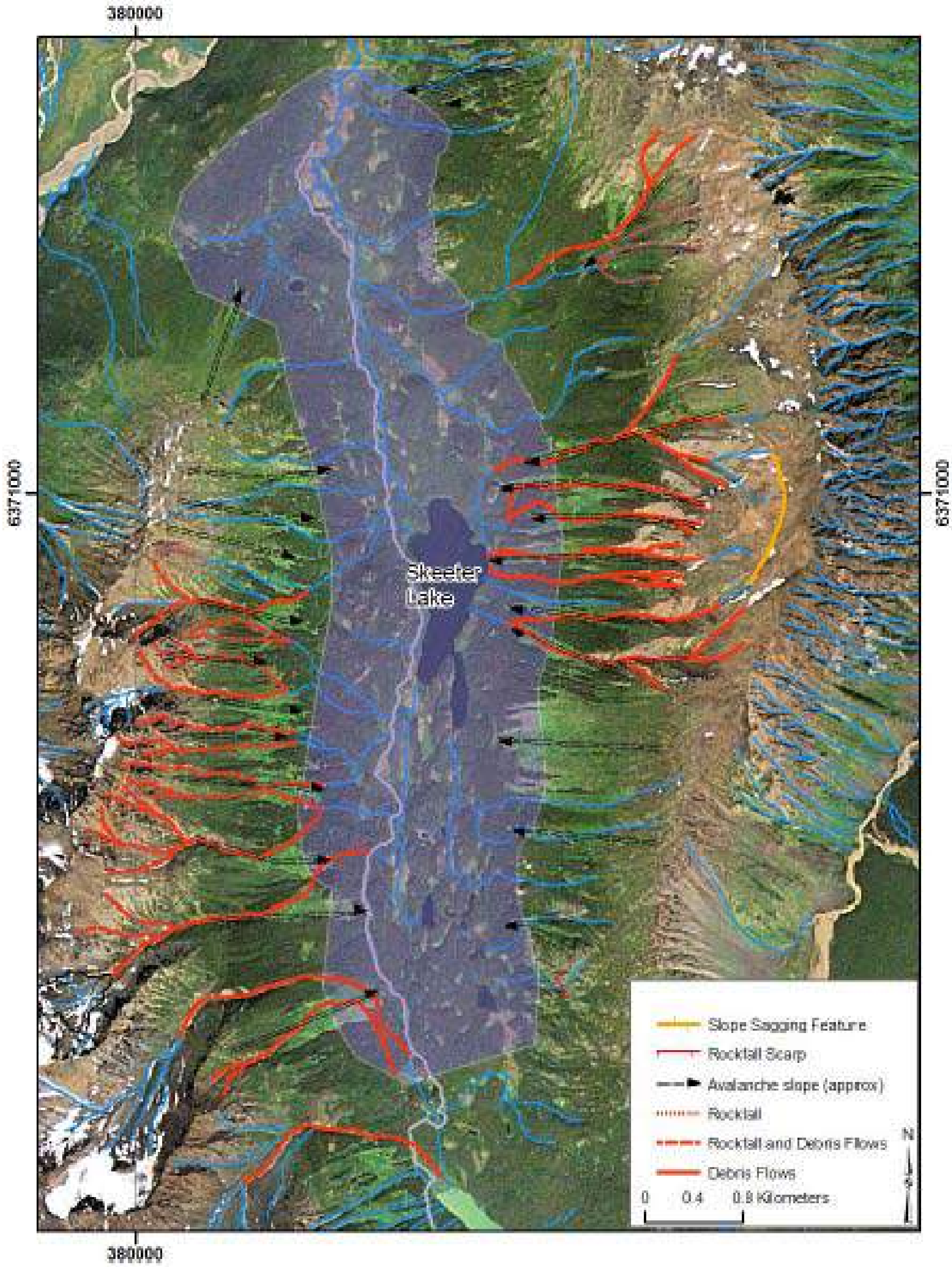


Figure 7.3 Tailing Storage Facility Hazards

A linear bedrock feature that is approximately 2,000 m wide exists on the east side of the valley; it is shown by the orange lines in Figure 7.3 and Figure 7.4. Based on the position of the feature with respect to adjacent slopes, it is interpreted to have sagged several tens of metres relative to slopes above. No signs of recent movement are visible on the photos so it is possible that the feature is very old, i.e. on the order of thousands of years and possibly associated with glacial debuttressing. Implications of rapid failure would likely include wave triggering and could be factored into tailings dam design. However, rapid failure of the block is considered very unlikely (less than 1/1,000). Additional, more detailed geological mapping and monitoring of slope movements, e.g. Interferometric Synthetic Aperture Radar (INSAR) and/or Differential GPS, are required to test this hypothesis.

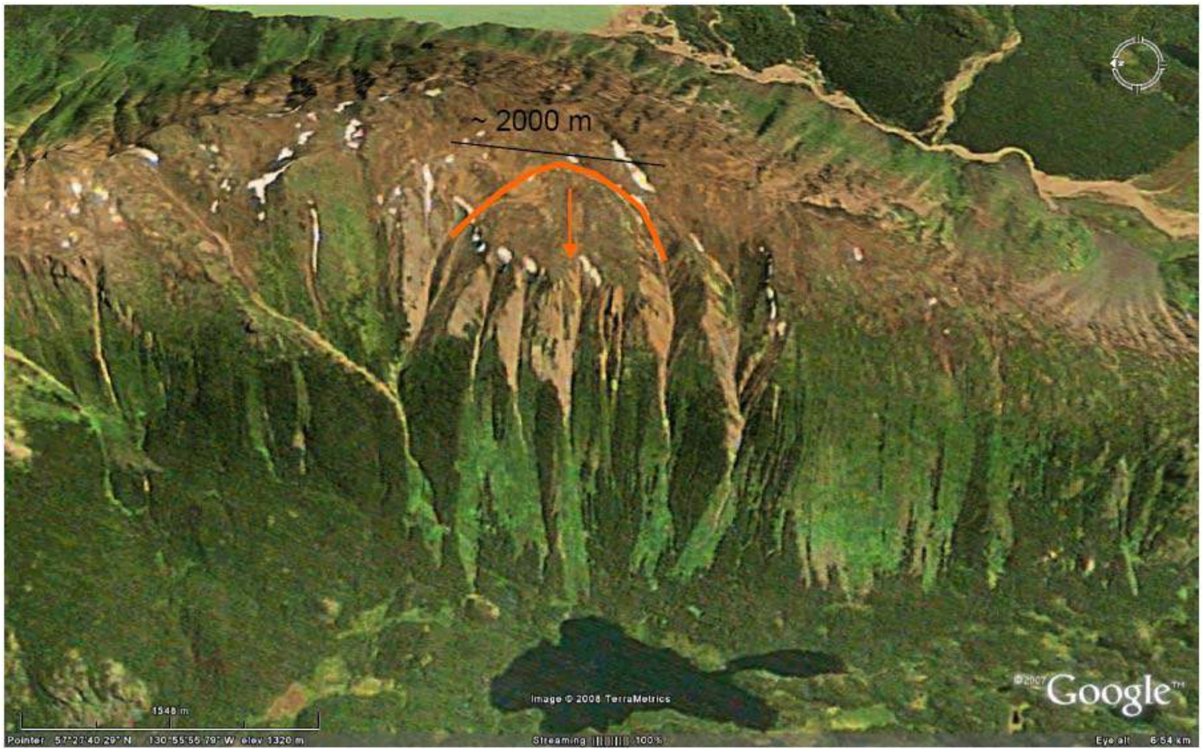


Figure 7.4 Tailing Storage Facility Sagged Slope Feature

Hummocks and numerous disconnected depressions exist in the valley bottom throughout the footprint of the tailings area. Some depressions may be kettles formed from melting of stagnant glacial ice. However, discontinuous bedrock outcrops imply thin till cover and suggest the presence of sinkholes and karst terrain in underlying carboniferous bedrock. These landforms were also noted by Spooner (2007b). Combined with the presence of extensional faults, this may represent a very significant hydro-geotechnical challenge for tailings design related to groundwater flow.

7.6.3 Site Access Road

Two access route options exist for the Schaft Creek project. The first option (Mess Creek access route) extends north from More Creek along Upper Mess Creek, entering the mine-site area and Schaft Creek drainage near Snipe Lake.

A second option (Tahltan Highland route) traverses a high-elevation plateau south of Mess Creek and descends slopes on the east side of Mess Creek to intersect the first road option at km 25.5. A partially-constructed access road parallel to More Creek extends to Highway 37 east of the Iskut River. The study area considered by BGC Engineering includes the entire length of the Mess Creek access route and the south and north most ends of the Tahltan Highland where the alignment ascends and descends from the high plateau.

Information about the geohazards studies by BGC Engineering that compare the two access routes is reported in *Schaft Creek Project Access Route Terrain and Geohazards Mapping*. The selected option is the Mess Creek access route, as explained in the following sections.

7.6.3.1 *Terms of Reference*

Copper Fox retained BGC to prepare terrain and geohazard maps for the access route. The initial, office-based phase of work was completed in early 2008 and a second phase of work commenced in Spring/Summer 2008. The second phase includes fieldwork and a more detailed geohazard assessment. The second phase of work includes more detailed geohazard mapping and field assessments and expansion of the study area to include the mine site.

7.6.3.2 *Scope of Work*

The following preliminary Scope of Work was performed within the study area, which is contoured in white as shown in Figure 7.5. The study area encompasses slopes adjacent to the Mess Creek access route between More Creek and Snipe Lake.

- Preparation of a terrain map showing surficial materials and geohazards, including the initiation and run-out zones of existing landslides, and zones subject to snow avalanche hazard;
- Overview description of terrain and geohazards.

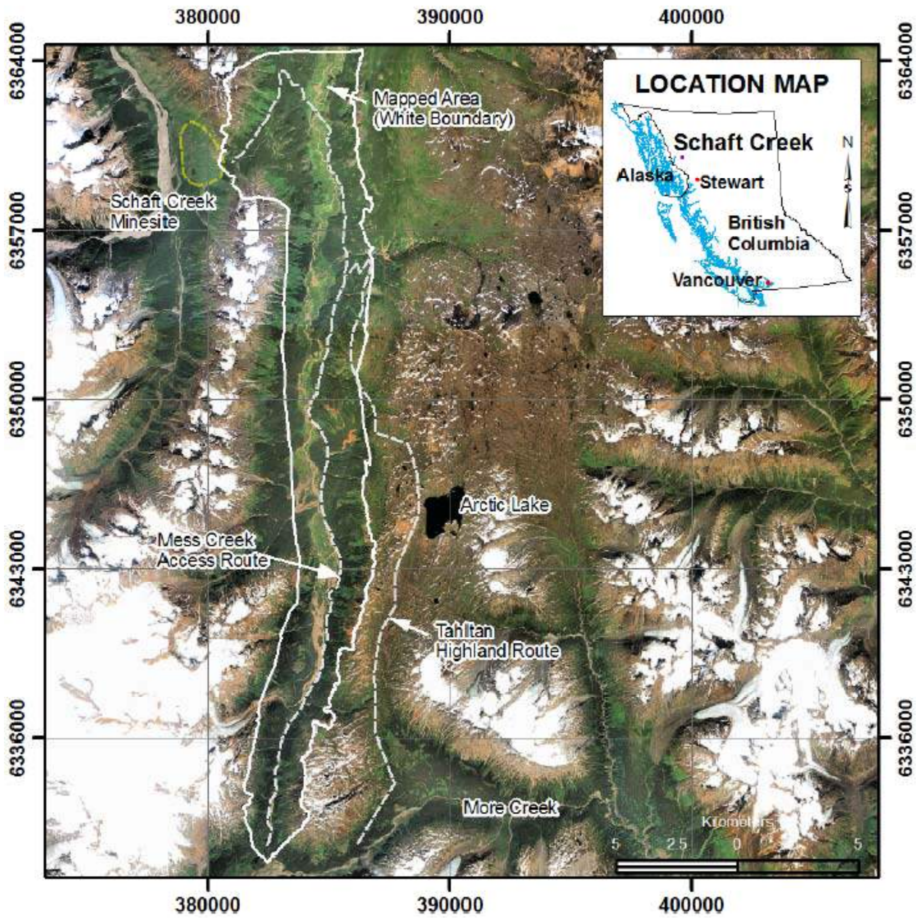


Figure 7.5 Study Area

Scope of Work for phase one did not include:

- Site-specific assessments of geohazard frequency and magnitude;
- Assessment of hydrologic (flood) hazards;
- Mapping of individual snow avalanche paths;
- Assessment of geohazards related to construction;
- Assessment of geohazards related to seismic activity;
- Recommendations for geohazard mitigation.

7.6.3.3 *Terrain Mapping Methods*

Terrain mapping methods are based on the guidelines and standards set by the Resources Inventory Committee (1996) and the Mapping and Assessing Terrain Stability Guidebook (Ministry of Forests 1999). Terrain classification followed the provincial system (Howes and Kenk, 1997). Mapping symbols are defined on the terrain maps and in Appendix I and a list of airphotos is provided in Appendix II of the BGC report.

Terrain mapping was done by Kris Holm, M.Sc., P.Geo. (KH) of BGC. Matthias Jakob Ph.D., P.Geo. (MJ) reviewed the mapping.

Work was based on air-photo interpretation with field checking anticipated during the second phase of this project. Symbols describing material types and drainage classes were added to all terrain polygons. Polygons intersected by the proposed road alignment were also assigned terrain stability classes. Atticus Spatial Information Management Ltd. (Atticus) transferred the polygon boundaries from the air photos to the digital trim base by mono-restitution methods.

7.6.3.4 *Mapping Reliability*

The accuracy of terrain mapping depends on numerous factors, such as the skill and experience of the mapper, the scale and quality of air photos that are used, the type and density of vegetation, field access and length of time spent in the field, quality of base maps and type and complexity of terrain and surficial materials. For this project, the accuracy is considered to be relatively high because the work was carried out by an experienced mapper (KH) under the review of an experienced geoscientist (MJ) and the scale and quality of the air photos is appropriate. Factors limiting the accuracy of mapping include dense forest cover in Mess Creek and lack of field checking for this initial project phase since field checking to be conducted during phase 2 of the geohazard assessment.

The minimum size of terrain polygons for 1:20,000 scale terrain mapping is about 2 ha. Thus local variations in terrain conditions over areas of 2 to 3 ha, or over distances of less than about 150 m, were not mapped. As a result, there may be considerable within-polygon variation in slope steepness, material characteristics and soil moisture. In addition, terrain-stability ratings assigned to terrain polygons intersecting the proposed road alignment are representative of the entire polygon and may not reflect detailed site conditions on the alignment. This implies that detailed planning of construction will require further ground checking to identify sites that may be more sensitive to disturbance than the average conditions mapped for an individual polygon.

7.6.3.5 *Drainage Interpretations*

Drainage classes rate the potential for materials within a polygon to be saturated and take into account both material permeability and the potential for water to drain into a polygon. Examples for each class are shown in Table 7.1. For example, a slope-containing coarse, highly-permeable material may be assigned an imperfect (i) drainage class if it is located in an area where abundant water drained into the polygon.

Table 7.1
Example Materials and Locations for Each Drainage Class

Drainage Class	Description	Example Materials and Locations
Rapid (r)	Water is removed from the soil rapidly in relation to supply	Exposed rock
Well (w)	Water is removed from the soil readily but not rapidly	Thin colluvium, bedrock partially covered in colluvium, till, upper slopes
Moderate (m)	Water is removed from the soil somewhat slowly in relation to supply	Thicker colluvium, till, mid-lower slopes
Imperfect (i)	Water is removed from the soil sufficiently slowly in relation to supply to keep the soil wet for a significant part of the growing	Lowermost slopes, gully bottoms, moist areas of floodplains

Table 7.1 Example Materials and Locations for Each Drainage Class		
	season	
Poor (p) Very Poor (vp)	Water is removed so slowly in relation to supply that the soil remains wet for a comparatively large part of the time the soil is not frozen	Bogs in bedrock depressions, marshy or wet areas of floodplains

7.6.3.6 *Terrain Stability Interpretations*

Terrain stability refers to mass movements such as debris avalanches, debris flows and rock fall. Terrain stability ratings range from class **I** (stable) to class **V** (unstable). The classes indicate the likelihood of instability resulting from road construction, or clear-cut logging in the case of forestry applications. Table 7.2 shows the terrain stability guidelines for road construction. Terrain stability ratings were assigned to polygons intersecting or adjacent to the proposed road alignment.

Table 7.2 Terrain Stability Ratings for Road Construction	
Terrain Stability Class	**Interpretation**
I	No significant stability problems exist
II	There is a very low likelihood of landslides following road construction. Minor slumping is expected along road cuts, especially for 1 to 2 years following construction.
III	There is a low likelihood of landslide initiation following road construction. Minor slumping is expected along road cuts, especially for 1 or 2 years following construction.
IV	Expected to contain areas with a moderate likelihood of landslide initiation following road construction.
V	Expected to contain areas with a high likelihood of landslide initiation following road construction.

The general criteria for assigning terrain stability classes are based primarily on slope steepness, material type and texture and geo-morphological processes. In addition, ratings were adjusted based on site-specific factors such as slope morphology and soil drainage. For example, a slope morphology that includes irregular, near-surface bedrock would typically be rated as more stable than a similar slope with a smooth profile because bedrock irregularities tend to hold surficial materials in place. Relatively poorly-drained or wet slopes may be prone to slope failures through a reduction in shear strength due to high soil pore water pressure and, consequently, they may be assigned a more conservative terrain stability rating. Details of the terrain stability classes are provided in the BGC report.

7.6.3.7 *Study Area Description*

Bedrock Geology

The Schaft Creek deposit and proposed access route are located in the northern part of the Intermontane Belt of the Canadian Cordillera, in Stikine assemblage rocks on the eastern boundary of the Coast Plutonic Complex (Logan et al. 1997, Giroux 2004). Much of Schaft Creek is located within Stuhini group volcanic and arcderived sedimentary rocks. The porphyry mineral deposit is hosted in three main zones by hydrothermally-altered volcaniclastic rocks, felsic porphyritic dykes and breccias overlain by relatively unaltered andesitic volcanic rocks. Northeast of the proposed open pit footprint and Mt. LaCasse, limestone bedrock underlies terrain in the vicinity of Skeeter Lake.

Granitic Coast intrusions occur to the northwest and south of the Schaft Creek deposit, including extensive exposures on the west side of Mess Creek and areas west of Schaft Creek. On the east side of Mess Creek in the vicinity of the access road, intrusives are also exposed within the monzonite Loon Lake Stock and in less-extensive granitic Coast intrusion outcrops. Further north and south of the Loon Lake Stock, terrain is underlain by Devonian to Jurassic age, low-grade (sub-greenschist), metamorphosed volcanic, volcanoclastic and sedimentary rock. Appendix III of the BGC report provides a list of bedrock types and faults that intersect the Schaft Creek access road, based on Logan et al. (1997).

Bedrock structure reflects complex, multiple phases and styles of deformation and faulting from Devonian to Late-Tertiary time. The overall intensity of regional-scale deformation increases northward, from weakly-deformed bedrock in the Forrest Kerr area to more intensely deformed Devonian and Early Carboniferous rock in Mess Creek. Angular unconformities in the Mess Creek area appear to reflect several phases of deformation, including an early-contractional phase and two additional phases in Late Jurassic – Early Cretaceous and Late Cretaceous – Tertiary time that correspond with development of the Skeena Fold and Thrust Belt (Evenchick 1991). Both Mess Creek and Schaft Creek follow extensional fault zones comprised of several fault-bounded blocks (grabens). North-northeast-trending listric normal faults associated with the graben-structure form steep escarpments on the east side of Mess Creek.

Climate

Schaft Creek and the proposed access route are located in a transition zone between wetter conditions of the British Columbia coast and drier conditions in the interior. Most of the study area lies within the Northern Coastal Mountains hydrological zone (9A; Coulson and Obedkoff 1998), characterized at higher elevations by cold winters, short cool summers and high annual precipitation. The major climatic processes during the fall and winter months include frontal cyclones arriving from the Pacific Ocean, resulting in precipitation as moist air masses are forced upwards over the Coast Mountains. Most precipitation from October through May falls as snow.

Topography, Glacial History and Deposition of Surficial Materials

The portion of Mess Creek traversed by the access route follows a broad, U-shaped valley with elevations ranging from 720 –1030 m along the valley bottom to 2300 m on adjacent ridge tops. Lower valley slopes are overlain by glacial till and colluvium with sporadic bedrock outcrops in steeper areas and at channel crossings. Along the valley bottom, extensive sand and gravel fluvial deposits occur on a broad floodplain up to 1 km in width. Upper hill slopes are steep, gullied rock slopes partially overlain by thin rubble colluvium. Surficial materials within terrain polygons intersecting the proposed access route are listed in Appendix V of the BGC report.

The topography within the study area reflects burial beneath the Cordilleran ice sheet during the Late Wisconsinan Fraser Glaciation (ca. 25-10 ka) followed by Holocene alpine glaciation and erosion due to fluvial and landslide processes. McCuaig (2002) notes two main phases of ice flow that are thought to have occurred during the Fraser Glaciation. In earlier- to middle-stages of glaciation the extent of the ice and its thickness increased to form the continental-scale Cordilleran Ice Sheet with south-westward flow towards the Pacific Ocean. During the later stages of glaciation, ice flow became confined to major valleys and fijords and retreated primarily by frontal retreat and downwasting. The broad U-shape of Mess Creek reflects preferential glacial scour of the weaker, fractured rocks along the Mess Creek fault zone.

During Holocene time, several episodes of glacial advance and retreat have occurred since about 7700 calendar years before present (cal-YBP) most recently during the “Little Ice Age” (LIA) from the 12th to 19th centuries (Ryder and Thomson 1986, Desloges and Ryder 1990, Luckman and Villalba 2001, Larocque and Smith 2003, Lewis and Smith 2004, Thompson et al. 2006, Koch 2006). In most areas, the largest Holocene glacial advance culminated in the early- to mid-1800s. Since then, many glaciated areas have decreased by over 30% with over 200 m of associated loss in ice thickness in some basins (Bovis and Evans 1996, Ryder and Thompson 1986).

The valley bottom of Mess Creek was not covered in ice since the Fraser Glaciation. However, several creeks on the west side of Mess Creek have increased sediment supply from areas subject to recent glacial retreat (e.g. polygons 269, 417, 429) and have glacially over-steepened slopes prone to rock fall or rock slides (e.g. polygon 434). In these locations sediment from upper basins provides material subject to entrainment during debris flows, debris floods or floods and is a contributing factor for tributary fan development and channel aggradation in Mess Creek.

7.6.3.8 *Mess Creek Access Route Identification of Geohazards*

Geohazards within terrain polygons intersecting the Mess Creek access route are listed in Appendix V of the BGC report. These include debris flows, debris floods, floods, rock fall, rock slides and snow avalanches. The road alignment traverses 35 polygons rated terrain stability class IV; all contain till or colluvium with moderately steep, approximately 30+ degree slopes. Eight polygons rated stability class V are intersected by the road alignment. In these cases, the stability rating refers to landslides initiating upslope of the road within the polygon. No cases were identified where the road alignment crosses existing terrain instabilities.

Debris flow hazard was identified at 10 channel crossings, including three locations where the road crosses a major fan (polygon nos. 246 at approximately km 17, 618 at approximately km 17.3, and 134 at approximately km 26.1). Rock-fall hazard exists where the road traverses below steep rock slopes (e.g. polygon 611); however, exposure to natural rock-fall hazard is uncommon along the access route and of lower frequency than would likely occur on artificial rock cuts. One landslide interpreted as a rock slide was mapped at km 13.27 in terrain underlain by sandstone (Logan et al. 1997). The failure occurred within an approximately 100 m wide embayment above a section of the proposed road that is approximately 80 m wide.

Snow avalanche hazard exists where the access road crosses avalanche vegetation scars below the tree line in the lower part of the run-out zone. Flood hazard exists at creek crossings and from km 31.7 to 32.9 where the access route crosses the Mess Creek floodplain.

The access road crosses several tributaries (e.g. polygons 134, 156, 246, 618) with upper channels underlain by highly-fractured bedrock. Failures in upper portions of these channels have triggered debris-flow activity and increased sediment supply and have resulted in large fans at the confluence with Mess Creek. These channels are subject to the largest debris flows along the access route, particularly in the basin upstream of polygon no. 246 where slope sagging features (polygons 254, 256) exist above fractured volcaniclastic and granitic slopes undercut by gully erosion. More detailed field assessment of debris flow hazard and implications for bridge design and road alignment is warranted at these sites.

Extensive slope sagging[1] features were identified in polygons 491 and 501, about 700 m west and approximately 100 m upslope of road km 2.4. The upper part of the feature which is approximately 1300 to 1500 m elevation is underlain by quartz diorite and the lower part approximately 1200 to 1300 m elevation is underlain by well-foliated sericite schist (Logan et al. 1997). Slow, deep-seated sagging may be ongoing in this area and is interpreted as possibly associated with gravitational loading and deformation of weaker underlying schist. However, rates of movement are unknown. Field surface investigation of this feature is recommended.

[1] Slow, deep-seated, gravitational slope deformation

8.0 History

8.1 History

Schaft Creek was the subject of intense and extensive exploration since mineralization was first discovered in 1957. The culmination of this exploration lead Teck Corp. to commission a PFS 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 in 2002, acquired the right to secure a significant ownership of the property 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 exploration program and subsequent programs in 2006, 2007 and 2008. Camp was put on winter maintenance on October 21, 2008.

The history of the property is summarized below.

- 1957, discoveries nearby spurred exploration northward into 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 feet of hand trenching;
- 1965, mapping, IP survey and 3 holes 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 property and carried out an extensive exploration program. A D6C bulldozer was 'walked in' from Telegraph Creek during the early spring, the airstrip was extended to 4,000 ft and a permanent camp erected. There were 24 holes drilled totaling 10,939 ft (3,334.2 m);
- 1967, Asarco drilled two holes totaling 1,001 ft (305.1 m). Paramount Drilling Limited drilled one hole to 501 ft (152.7 m);
- 1968, Asarco drops option and Hecla Mining acquires properties, airstrip extended to 5,280 ft;
- 1968, Hecla drills 9 holes, totaling 13,095 ft (3,991.3 m);
- 1969, Hecla drills 9 holes, totaling 15,501 ft (4,724.7 m);
- 1970, Hecla drills 26 holes, totaling 32,575 ft (9,928.8 m);
- 1971, Hecla drills 25 holes, totaling 22,053 ft (6,721.7 m);
- 1972, Hecla drills 10 holes totaling 8,950 ft (2,727.9 m);
- 1974, Hecla drills 6 holes totaling 7,062.5 ft (2,152.6 m);
- 1977, Hecla drills 1 hole totaling 2,113 ft (644 m);
- Reserve of 505 Mt, 0.38% Cu, 0.039% MoS_2, delineated;
- Between 1978 and 1979 Hecla Mining forfeits option and Teck Corp. acquires property;
- 1980, Teck Corp. drilled 47,452 ft (14,463.3 m) in 45 holes, between mid-May to mid-November, drill sites prepared with a D6 Caterpillar bulldozer. Assaying of core on 10-foot sample intervals, by Afton Mines Ltd. in Kamloops;
- 1981, between June and September, Teck Corp. drilled 33,315 ft (10,154.4 m), in 74 holes, and 3,503 feet of condemnation drilling for tailings pond and mill sites;
- Resource expanded to 1 billion t, 0.30% Cu, 0.034% MoS_2;
- 2002, Mr. G. Salazar acquires the right to secure a significant ownership of the property and subsequently incorporated it into the holdings of Copper Fox Metals;
- 2005, Copper Fox drills 15 holes totaling 10,367 ft (3,160 m);

- 2006, Copper Fox drills 42 holes totaling 29,547.2 ft (9,006 m);
- 2007, Copper Fox drills 42 holes totaling 20,684 ft (6,304.5 m);
- 2008, Copper Fox drills 48 holes totaling 22,822 ft (6,958 m).

Total property drilling up to October 2008 is 280,041 feet, in 382 holes.

9.0 Geological Setting

9.1 Introduction

Information about the geology of the Schaft Creek geological resource and the surrounding area was provided by Walter Hanych, Sheena Ewanchuk and Dr. Peter Fisher in a report titled *2006 Diamond Drill Report Schaft Creek Property Northwestern British Columbia* in March 2007. Additional information about the geology can be found in a M.S. thesis that was completed in 2008. It is *The Schaft Creek Cu-Au-Mo-Ag Deposit, Northwestern British Columbia* which was completed by James Edward Scott.

9.2 Regional Geology

The Schaft Creek copper porphyry (copper, molybdenum, gold, silver) 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.

9.3 Local Geology

The Schaft Creek deposit is in part situated in the valley of Schaft Creek and in part along the western slope of Mt. 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 dykes, 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 (also known as the Liard 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.

9.3.1 Lithology

In term of the deposit as whole, 17 rock types were observed and recorded. Table 9.1 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% 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 9.1 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.

9.3.2 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.

9.3.3 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.

9.3.3.1 *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.

9.3.3.2 *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.

9.3.3.3 *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.

9.3.3.4 *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.

9.3.3.5 *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’.

9.3.3.6 *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.

9.3.4 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.

9.3.4.1 *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 over 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% to 10%, the remaining balance is usually quartz, carbonate and chlorite. Chalcopyrite stringers, 0.5 to 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 to 20 veins per m; however, high densities ranging from 100 - 200 veins per m do occur. At the other end of the spectrum, low densities ranging from 5 - 10 veins per m 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.

9.3.4.2 *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.

9.3.4.3 *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 mm to m spacing;
- Have variable vein-widths from less than 1 to 10 mm up 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.

9.3.5 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.

9.3.5.1 *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.

9.3.5.2 *Microfaulting*

- Microfaulting is defined as thin fractures that visibly offset a vein or other lithological features in one core sample;
- Microfaulting is common, often occurring as groups of parallel, cm-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.

9.3.5.3 *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.

9.3.5.4 *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 over 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.

9.3.5.5 *Faulting*

- Faulting is common with spacing at 1 m to 10 m 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 to 50 fractures per m) are commonly logged as “fault zones.”

9.3.5.6 *Foliation, Shears*

- Foliation is defined as a rock unit showing a distinct fabric or foliation, which is fairly rare;
- Foliated units are 1 m 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 by 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.

10.0 Deposit Types

10.1 Deposit Types

10.1.1 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 and silver.

The deposits are formed by a shallow magma chamber of hydrous, intermediate composition at depths of less than 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-silver deposit.

10.1.2 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 apophysis, 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.

11.0 Mineralization

11.1 Alteration and Mineralization

By the CIM Definition Standards on Mineral Resources and Mineral Reserves, a mineral reserve has to be supported by at least a preliminary feasibility 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.

11.1.1 Mineralization

11.1.1.1 *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 4.1

Table 11.1
Summary of Mineral Occurrences Proximal to Schaft Creek

Minfile No.	Name	NTS Map	UTM	Minerals[1]	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

Abbreviations: cp – chalcopyrite, bn – bornite, py – pyrite, cc – chalcocite, mo – molybdenite.

11.1.1.2 *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 define 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 to 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 mm to cm 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 cm- to dm-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 less than 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 to 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.

11.1.2 Description of Mineralized Zones

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

11.1.2.1 *Main Zone*

The Main zone has currently defined dimensions of 1,000 m by 700 m by 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 apophysis 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.

11.1.2.2 *West Breccia Zone*

The West Breccia zone has currently defined dimensions of 500 m by 100 m by over 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.

11.1.2.3 *Paramount Zone*

The Paramount zone is the most northerly of the zones and has currently defined dimensions of 700 m by 200 m by 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.

12.0 Exploration

12.1 Exploration and Drilling Program

12.1.1 Recent Exploration Summary

The 2005 diamond drill campaign conducted by Copper Fox 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 metallurgical composite samples. The total combined weight of the metallurgical samples collected is a total 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.

12.1.2 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 were transported from Burrage Creek and Bob Quinn to the camp in the initial airlift. Core drilling commenced on July 10th and the drill program was terminated on October 23rd after completing 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.

12.1.2.1 *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 flotation 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-diameter, 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 and two of the HQ-holes were upgraded to PQ-holes.

12.1.2.2 *Field Protocol*

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

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 diameter (16 cm 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 3.0 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 (PRA) Lab in Vancouver, British Columbia, adhering to the same chain of custody as for the assay samples.

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 chemical analysis, while the other half was retained for archiving.

12.1.2.3 *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 Mt. LaCasse exhibit malachite-stained fractures. During the coring process, m-lengths of intact overburden with fragments of talus were recovered by the drilling. To determine the extent to which 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.

12.1.2.4 *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 one quarter to one half of all samples consist of foreign material, transported pebbles and boulders. These samples exhibit Cu values of over 1,000 ppm and one quarter of the samples returned values of less than 0.1 g/t Au. Samples exhibiting anomalous molybdenum and silver values were the fewest number of samples.

Of these, half a dozen samples had 100 ppm Mo and half a dozen had over 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.

12.1.3 2007 Exploration Program

The 2007 exploration program started up in May 2007. A total of 42 holes were drilled in 2007 for a total depth of 6,275 m. Drilling was allocated according to the information provided in Table 12.1.

Table 12.1
2007 Exploration Drilling Campaign

Purpose	Number of Holes	Total Metres
Extensions to Mineralization – NE Pit Slope and Snipe Lake	11	2,357.8
Proposed Tailings Storage Facility and Mill	14	1,405.2
Proposed Waste Disposal Areas	17	2,412.0
TOTALS	42	6,275.0

12.1.3.1 *Extensions to Mineralization*

NE Pit Slope – Snipe Lake

Drilling into the north eastern portions (the Mt. LaCasse ridge area) of the deposit has been difficult at best. Low core recoveries and high winds are the main difficulties encountered. However, good mineralization was found by previous operators and was confirmed in 2007. Of special note are the results from DDH 07CF313 which was drilled in an area between the Paramount and the Liard zones that is heavily weathered, leached and shattered. The drill-hole location is along the higher road west of Mt. LaCasse heading north from Snipe Lake. The results from the hole are provided in Table 12.2

Table 12.2
DDH 07CF313 Results

Hole No.	Total Depth	From	To	Weighted Avg, % Cu	Weighted Avg, % Mo	Weighted Avg, Au g/t	Weighted Avg, Ag g/t	Equiv % Cu
07CF313	421.8	47.5	110.0	0.439	0.015	0.208	9.6319	0.765
07CF313	421.8	114.6	129.8	0.394	0.015	0.345	0.0001	0.725
07CF313	421.8	382.8	407.2	0.359	0.039	0.320	2.9898	1.053

This mineralization extends the Main or Liard zone north eastwards under the LaCasse ridge and correlates to two mineralized intervals found in drill hole T81CH198, one with

0.582% Cu, 0.01% Mo and 0.62 g/t Au over 36.6 m and another 6.1 m interval grading 1.17% Cu, 0.03% Mo and 0.62 g/t Au.

These are higher-grade sections of mineralization that extend north easterly away from the Main or Liard zone and under the LaCasse ridge into an area of the property not previously explored but within the claims subject to the Teck Cominco option. The ridge rises steeply to the north and east from the drilled area. Attempts at testing possible extensions to this zone at holes 07CF316A/B (636.1 m deep) and 07CF317A/B (122.0 m deep) failed due to very difficult drilling and working conditions that were found.

Drill holes 07CF301, 07CF303, 07CF308 and 07CF309 were drilled on the north and west edges of an intriguing swamp located immediately south of the Main or Liard Zone. The best mineral intersection here was found in hole number 07CF303 which found a 3.1 m interval assaying 1.28% Cu, 0.001% Mo, 0.020 g/t Au and no silver values at a depth of 94.2 to 97.2 m.

Gold values of 0.60 and 0.68 g/t over 3.1 and 23.5 m, respectively, were found in holes 07CF308 and 07CF309. Hole 07CF301 found a 12.2 m long intersection that assayed 0.40% Cu, 0.002% Mo, 0.020 g/t Au and 0.5 g/t Ag.

Table 12.3 outlines the results of the drilling in the Liard South and NE Pit Slope – Snipe Lake Area.

Table 12.3
Liard South and NE Pit Slope – Snipe Lake Area

Hole No.	Total Depth	From	To	Weighted Average, % Cu	Weighted Average, % Mo	Weighted Average, Au g/t	Weighted Average, Ag g/t	Equiv. % Cu
Liard South - Extension								
07CF301	192.7	9.7	30.2	0.221	0.014	0.057	0.0001	0.448
07CF301		48.5	54.6	0.236	0	0.02	0.0001	0.242
07CF301		109.4	121.6	0.401	0.002	0.02	0.5337	0.439
07CF301		143.0	146	0.169	0	0.03	0.0001	0.178
07CF303	138	8.8	18	0.324	0.001	0.02	0.0001	0.345
07CF303		48.5	60.7	0.454	0.001	0.055	0.0001	0.486
07CF303		75.9	85	0.201	0.001	0.007	0.0001	0.218
07CF303		94.2	97	1.279	0.001	0.02	0.0001	1.300
07CF303		136.3	138	0.197	0	0.03	0.0001	0.206
07CF308	104.9	25.6	31.7	0.236	0.001	0.021	0.8263	0.261
07CF308		43.9	47	0.192	0.001	0.02	0.0001	0.213
07CF308		77.4	80.5	0.367	0.003	0.06	0.0001	0.430
07CF309	111.3	9.5	32.9	0.232	0.014	0.068	0.0001	0.463
NE Pit Slope - Snipe Lake								
07CF312	156.7	48.2	57.3	0.19	0.002	0.326	0.0001	0.320
07CF312		66.4	69.5	0.165	0.001	0.03	0.0001	0.189
07CF312		72.5	81.7	0.175	0.001	0.057	0.7318	0.210
07CF312		124.4	133.5	0.278	0.003	0.053	0.0001	0.339

Table 12.3 Liard South and NE Pit Slope – Snipe Lake Area								
Hole No.	**Total Depth**	**From**	**To**	**Weighted Average, % Cu**	**Weighted Average, % Mo**	**Weighted Average, Au g/t**	**Weighted Average, Ag g/t**	**Equiv. % Cu**
07CF313	421.8	11.0	14	0.475	0.002	0.02	0.0001	0.511
07CF313		47.5	110	0.439	0.015	0.208	9.6319	0.765
07CF313		114.6	129.8	0.394	0.015	0.345	0.0001	0.725
07CF313		148.1	184.7	0.242	0.039	0.165	0.0001	0.877
07CF313		190.8	215.2	0.236	0.014	0.193	0.0001	0.505
07CF313		248.7	257.9	0.279	0.006	0.123	0.0001	0.407
07CF313		303.6	328	0.181	0.011	0.195	2.0004	0.413
07CF313		382.8	407.2	0.359	0.039	0.32	2.9898	1.053
07CF314	256.7	No significant mineralization found						
07CF315	149.9	No significant mineralization found						
07CF316	636.1	499.1	505.2	0.155	0.021	0.06	0.0001	0.488
07CF316		529.0	532.0	0.159	0.005	0.06	0.0001	0.252
07CF316		557.0	560.1	0.352	0.002	0.11	0.0001	0.416
07CF316		569.2	572.3	0.24	0.007	0.05	0.0001	0.360
07CF316		599.5	602.6	0.371	0.009	0.03	0.0001	0.515
07CF317	122	No significant mineralization found						
07CF319	167.7	102.7	108.8	0.187	0.002	0.055	0.0001	0.234
07CF319		145.4	154.3	0.531	0.003	0.044	0.0001	0.589
07CF320A	36.3	NO ASSAYS						
07CF321	0	NO ASSAYS						

Drill Testing of Waste Disposal Areas

A total of 31 drill holes and 3,817.2 meters were dedicated to testing potential Waste Disposal Areas near the deposit. Table 12.4 displays and summarizes the results from this testing.

Table 12.4 Condemnation Drilling of the Waste Disposal Areas									
Hole No	**Total Depth (m)**	**From**	**To**	**Interval**	**Weighted Average % Cu**	**Weighted Average % Mo**	**Weighted Average Au g/t**	**Weighted AverageAg g/t**	**Equiv. % Cu**
Waste Disposal Area									
07CF291	122.5	No significant mineralization found							
07CF292	136.9	No significant mineralization found							
07CF293	125	No significant mineralization found							
07CF294	151.4	No significant mineralization found							
07CF295	120	36.1	39.2	3.05	0.242	0.013	0.05	0.0001	0.452
07CF295		75.55	78.6	3.05	0.165	0.001	0.06	0.0001	0.198
07CF296	185.4	52.3	58.3	6	0.226	0.005	0.115	0.0001	0.336
07CF296		76.6	82.7	6.1	0.225	0.002	0.085	0.0001	0.281

Table 12.4 Condemnation Drilling of the Waste Disposal Areas									
07CF296		104.05	107.1	3.05	0.209	0.005	0.02	0.0001	0.290
07CF296		128.45	131.5	3.05	0.159	0.006	0.02	0.0001	0.255
07CF296		140.65	146.8	6.1	0.274	0.003	0.04	0.0001	0.331
07CF297	154	No significant mineralization found							
07CF298	153.3	No significant mineralization found							
07CF299	110.3	No significant mineralization found							
07CF300	119.2	No significant mineralization found							
07CF302	149.1	No significant mineralization found							
07CF304	139.9	4.6	139.9	135.3	0.296	0.011	0.107	1.5534	0.500
07CF305	129.5	No significant mineralization found							
07CF306	128	No significant mineralization found							
07CF307	136.6	No significant mineralization found							
07CF310	150.3	100	103	3.1	0.248	0.002	0.03	0.0001	0.287
Paramount - Waste									
07CF311	200.6	No significant mineralization found							

Hole number 07CF304 found 139.9 m (from bottom of casing) assaying 0.296% Cu, 0.011% Mo, 0.107 g/t Au and 1.55 g/t Ag with a copper equivalent grade of 0.500% to bottom of hole. This hole extended the West Breccia zone to the south a further 400 m.

Then, holes 07CF301 and 07CF303 were drilled further south towards the swamp. In addition, drill holes 07CF295 and 07CF296 displayed evidence of up to 6.1 m of drill depth with assays of up to 0.274% Cu and up to 0.115, 0.085 and 0.107 g/t Au. Although the intervals are narrow, these drill-hole intersections prove that the mineralization extends into these new areas.

12.1.3.2 *2007 Geophysical Results*

Condemnation testing of one of the proposed mill sites found an induced polarization – chargeability anomaly immediately east of the Liard Zone. It is 250 to 300 m wide and is located in an area never touched by drilling or geophysical surveys. The coincident chargeability and resistivity anomalies have a similar signature to that of the Main/Liard Zone, as shown on the same survey line further west. This anomaly is directly south of the mineralization found under the LaCasse ridge. Plans for summer 2008 include collecting additional information from this area using additional geophysical surveys prior to drilling.

12.1.3.3 *Resource Comparison*

The Preliminary Economic Assessment (PEA) that was completed in 2007 showed that the Schaft Creek property compares favourably to other large projects that have been studied recently including the Galore Creek (B.C.), Quebrada Blanca (Chile) and Pebble Creek (Alaska) deposits. Using Case 2 metal prices as presented on the PEA, a 0.61% Cu Equivalent is calculated for the Measured and Indicated Resources for the Schaft Creek, Galore Creek and Pebble deposits. The calculations for Schaft Creek are based on recoverable metal, as shown by Copper Fox and historical recoveries determined by flotation tests. The presence and recovery of molybdenum at Schaft Creek is a major component of the potential economics of the deposit.

Table 12.5
Resource Comparison

Deposit	Million Tonnes	Cu Equiv, % Cu (Note 1)
Schaft Creek (M&I)	1,393.2	0.61
Galore Creek (M&I)	928.4	0.61
Quebrada Blanca, Chile (Inferred)	1,030.0	0.50
Pebble Creek, Alaska (M&I)	17,716.4	0.61

Note1: Copper Equivalent Calculations based on following prices (in US $): Copper: $2.66/lb; Gold: $563.77/oz.; Silver: $10.40/oz, Molybdenum: $27.00/lb

12.2 Activities Planned to Expand Mineralized Zones and Explore Prospects

This year's diamond drilling program is intended to further define the first five to six years of mine production when Schaft Creek goes into production in an effort to shorten the payback period.

Work on-site will include 36 drill holes for approximately 9,000 metres:

- 3,000 m oriented core drilling – slope stability, NE wall, etc.;
- 3,000 m of geotechnical drilling – tailings, waste and slope stability;
- 1,000 m of exploration/geology drilling.

Additionally, further geophysics will be performed to investigate the IP resistivity anomaly defined during the 2007 season.

The geophysical program is designed to complete approximately 54 line kilometers of IP surveying. The access and road survey work will include extensive archaeological studies.

13.0 Drilling

13.1 Drill Program

Hytech Diamond Drilling Ltd. of Smithers, BC, 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 BC and Tsayta Air Ltd. of Fort St. James, BC.

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.

13.1.1 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 13.1 and separated on a zone basis.

Table 13.1
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

Table 13.1 Estimates of RQD and Core Recovery, 2006 Drill Program			
Drill Hole ID	**Hole RQD**	**RQD Rating**	**Core Recovery**
05CF238	31.4	Poor	97
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

Table 13.1 Estimates of RQD and Core Recovery, 2006 Drill Program			
Drill Hole ID	**Hole RQD**	**RQD Rating**	**Core Recovery**
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
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.

14.0 Sampling Method and Approach

14.1 Sampling, Assaying and Quality Control

14.1.1 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 14.1 summarizes various drilling campaigns done on the project area.

Table 14.1
Summary of Drilling Campaigns

Series	Holes	Total Length (m)	Average Length (m)	Minimum Length (m)	Maximum Length (m)
07 Copper Fox	42	6,275.0			
06 CopperFox	42	9,066.80	215.88	78.00	351.00
05 CopperFox	15	3,158.72	210.58	49.07	341.99
ASARCO	23	3,181.52	138.33	98.45	351.00
Silver Standard	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
	329	77,902.9			

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 14.1 depicts the distribution of drill holes throughout the project area.

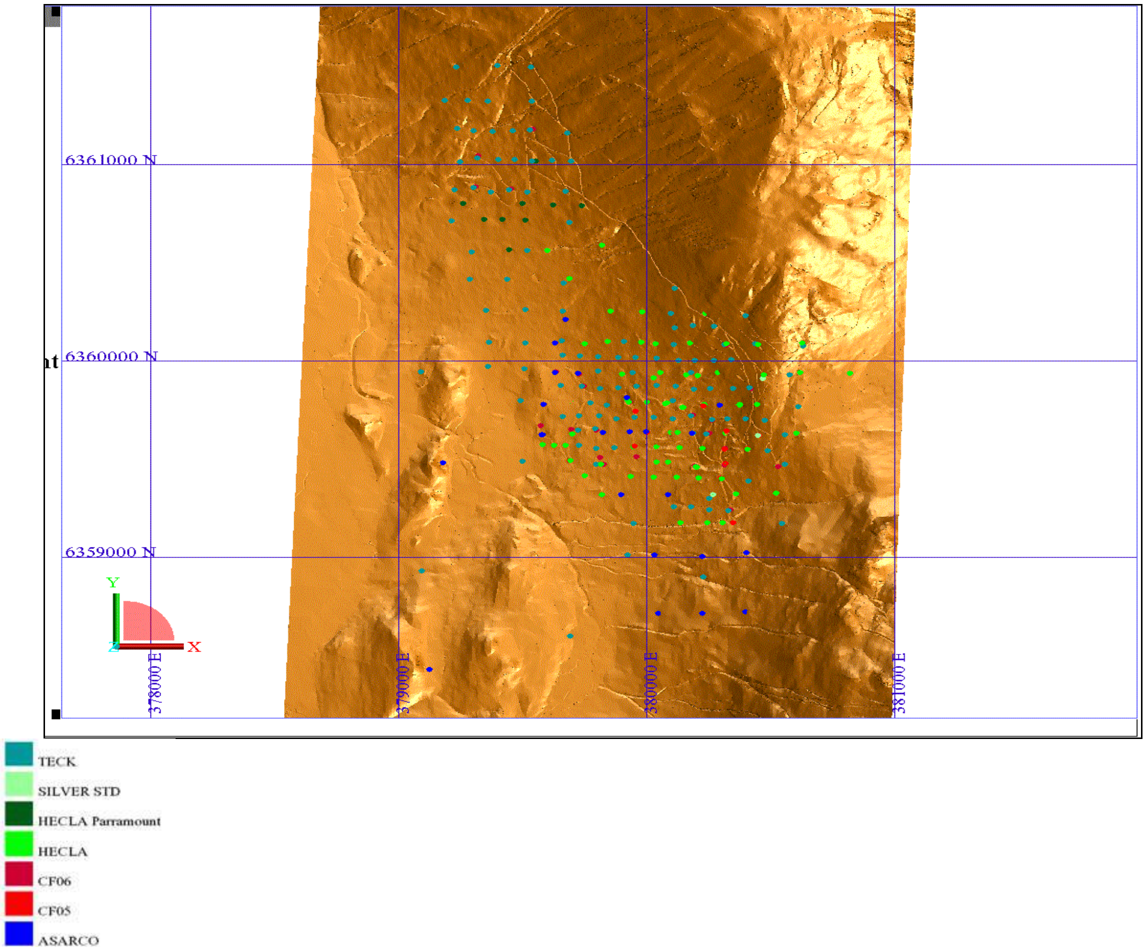


Figure 14.1 Distribution of Drill Holes

14.1.2 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 14.2 and Figure 14.3 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.

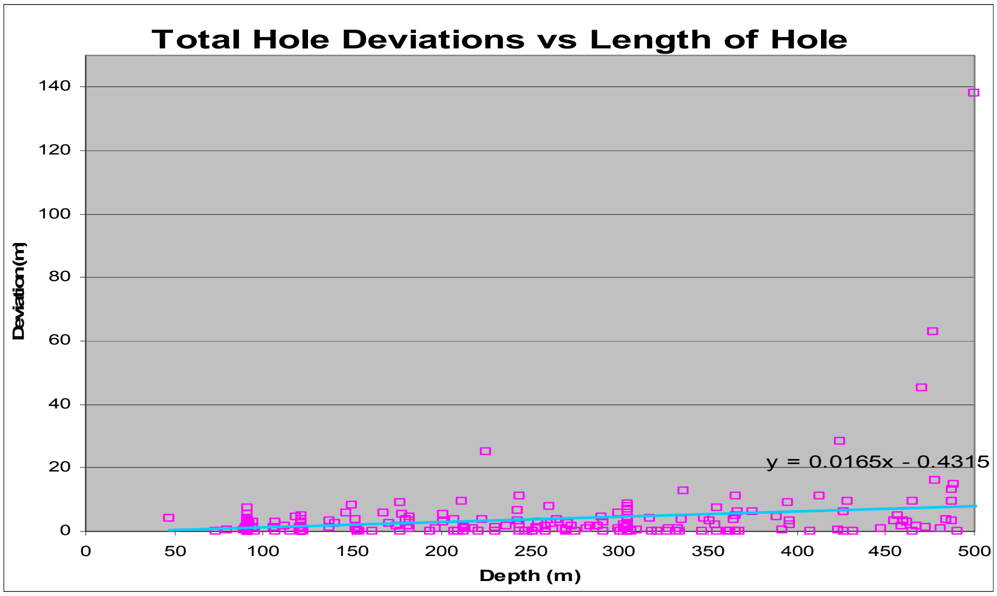


Figure 14.2 Deviations Between Measured and Planned Drill Holes

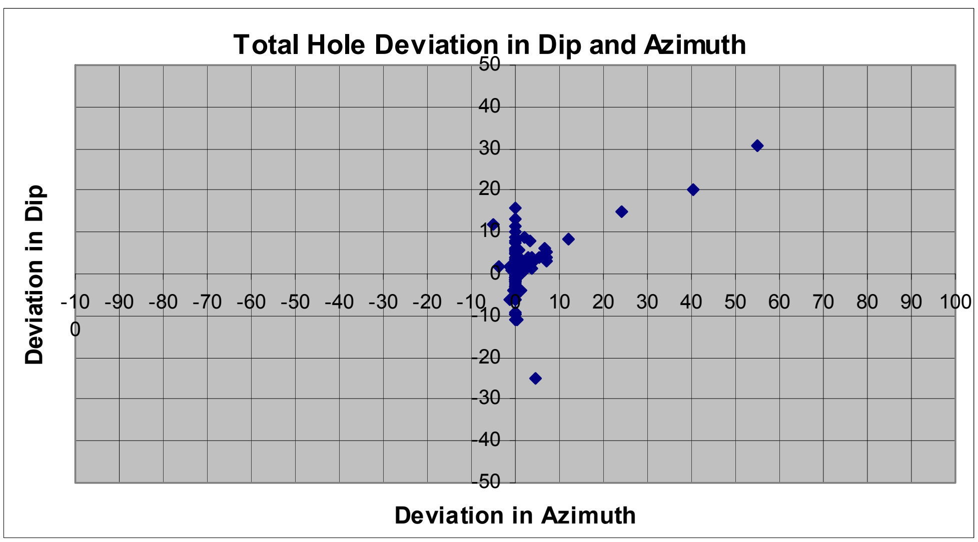


Figure 14.3 Deviations Between Measured and Planned Drill Holes

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 14.2. 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.

Drill holes deviating by more than 20 m from the planned position.

Table 14.2 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.

14.1.3 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 spreadsheets.

14.1.4 Sampling Protocol, 2006 Drilling Program

A sampling protocol conforming to the Canadian Securities Administrators (CSA) NI 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 approximately 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, BC, and MET samples were sent to Process Research Assoc. Ltd (PRA) in Richmond, BC. 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.

15.0 Sample Preparation, Analyses and Security

15.1 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.

15.2 Ore-Grade Elements by Multi-Acid Digestion and 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 spectrometer (ICPES). All elements are corrected for inter-element interference and all data are stored on 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.

15.3 Fire Assay Gold Assay

- Duplicates of 50 g (2 assay tons) are weighed into fusion crucibles together with various flux materials, including lead oxide. After thorough mixing with a 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 which forms at the bottom of the conical mold is separated from the slag;
- The lead button is placed in a pre-heated cupel and placed in 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 solution;
- The gold in solution is determined using an atomic absorption spectrophotometer (AAS). 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 crucibles contains 22 samples, one internal standard or blank, and a re-assay of every 20th sample. Samples with gold concentrations over 1,000 ppb are automatically checked by fire assay with an AA finish, as described above. Samples with gold concentrations over 10,000 ppb are automatically checked by fire assay and a gravimetric finish where the bead is weighed on a micro-balance instead of being dissolved in acid where the gold and silver concentrations are then determined using AAS.

16.0 Data Verification

16.1 Data Verification

The Schaft Creek deposit has been the object of detailed exploration from 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 the laboratories did not describe the analytical methods that were used on the samples in any detail. There are no records of any field-based quality control programs or any indication of the quality control practices by the laboratories.

Comparative analyses between several of the labs indicated that the reproducibility for copper was satisfactory, although several of the labs show a distinct bias from one to the other. Comparison of molybdenum, gold and silver 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 copper 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 molybdenum, gold and silver analyses did not repeat as well.

This is a clear indication that the original assay data is valid for at copper 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.

16.1.1 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. 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.

16.1.1.1 *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, 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 Canadian laboratories, and four were prepared by International Plasma laboratories (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 duplicates are two different samples and results of the assays could vary if mineralized veins are unevenly distributed in the two adjoining samples.

16.1.1.2 *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 fire assay.

16.1.1.3 *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.

16.1.1.4 *Evaluation of the Laboratory Procedures*

The laboratory provided results within 10 to 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 at 150 mesh.
  Both size fractions, i.e. plus 150 mesh and minus 150 mesh were assayed separately, giving results of 83.95 g/t Au 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 largest crucibles available have a capacity of only 50 g.

16.1.1.5 *Statistical Analysis of the Results*

Graphical representations 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.

16.1.1.6 *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.

16.1.2 Review of Analytical Database for Resource Modeling

16.1.2.1 *Sampling*

A total of 22,148 samples were submitted for analysis. Table 16.1 provides an overview of the samples sent for analysis relative to the drilling campaign.

Table 16.1
Overview of Samples Relative to Drilling Campaign

	Holes	Geological	Samples (#)
06 Copper Fox	42	2,910	2,932
05 Copper Fox	15	1,023	1,034
ASARCO	23	1,155	986
Silver Standard	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 reverse circulation 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).

16.1.2.2 *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 16.2 demonstrates the amount of quality control samples examined.

Table 16.2
Quality Control Samples

	Sampling	Duplicate	Chemex
06 Copper Fox	2932	19	-
05 Copper Fox	1034	27	26
Total	**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 16.3 and Table 16.4 for the in-house duplicate assays and the second lab duplicate assays, respectfully. The tables show 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 16.3
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 16.4
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 16.1, Figure 16.2, Figure 16.3 and Figure 16.4 show the statistics for the 46 check samples compared with the original sample results, plotted on Q-Q plots for copper, molybdenum, gold and silver.

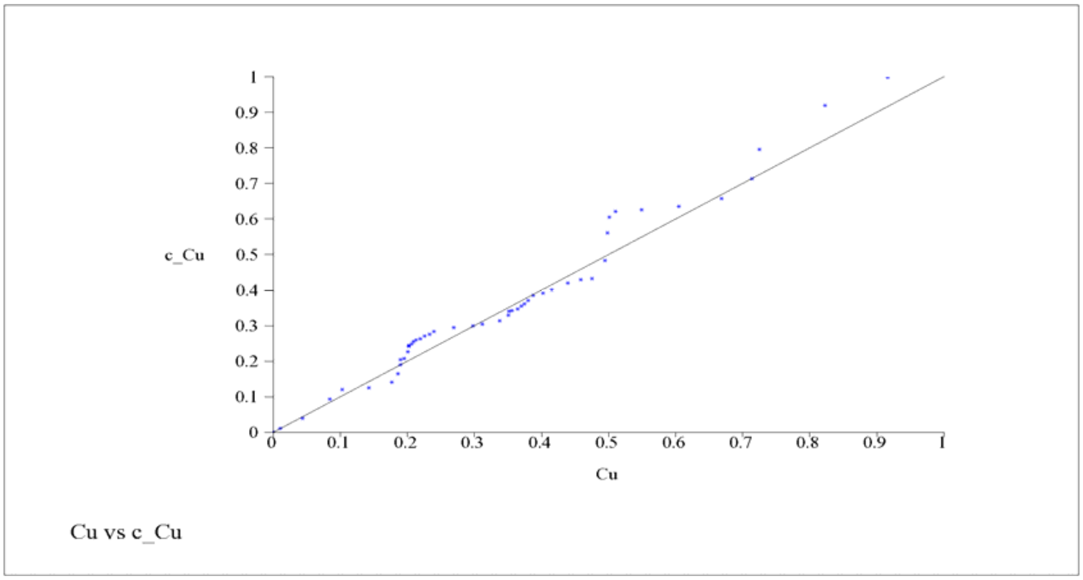


Figure 16.1 Statistics for Check Samples (1st laboratory), Q-Q Plot of Cu

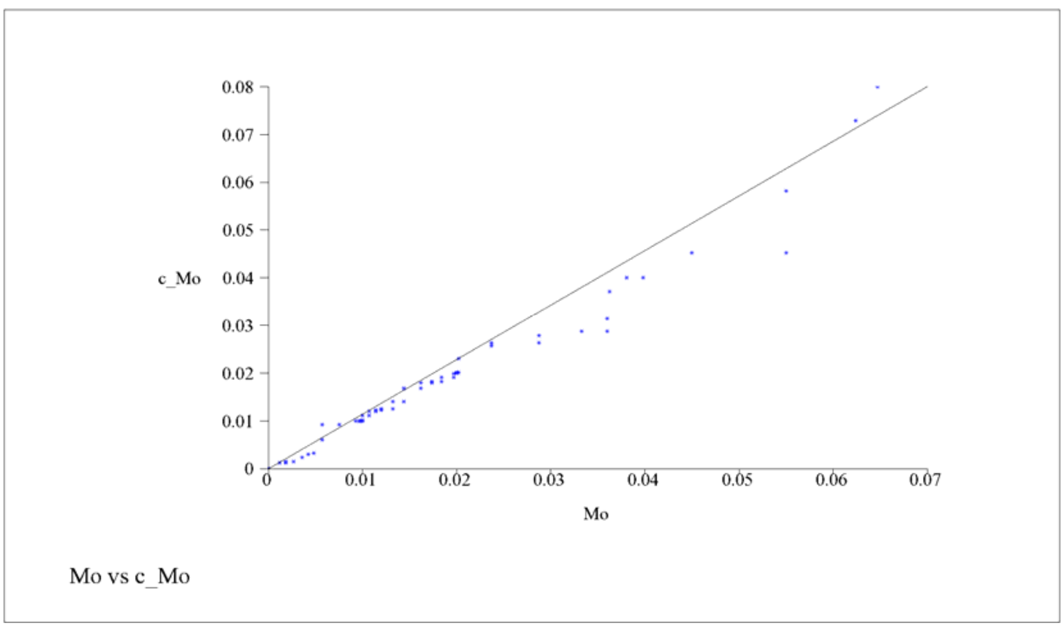


Figure 16.2 Statistics for Check Samples (1st laboratory), Q-Q Plot of Mo

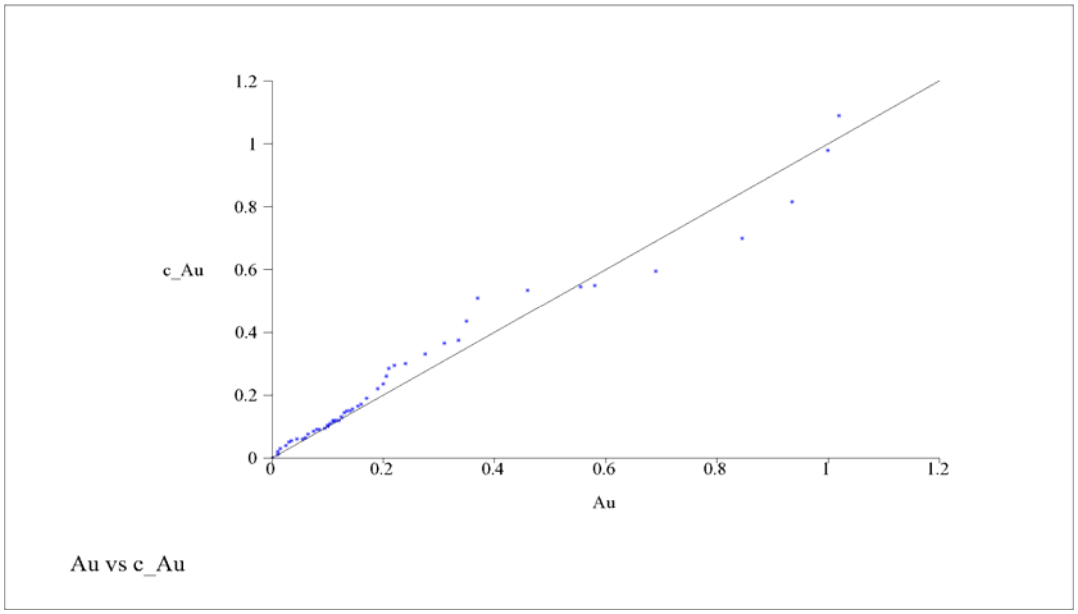


Figure 16.3 Statistics for Check Samples (1st laboratory), Q-Q Plot of Au

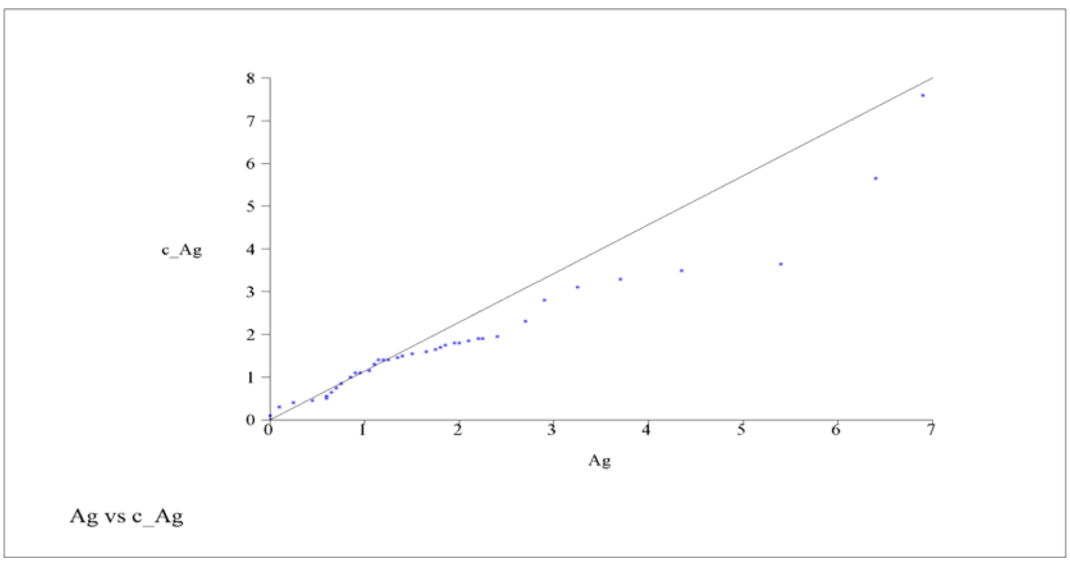


Figure 16.4 Statistics for Check Samples (1st laboratory), Q-Q Plot of Ag

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

Figure 16.5 depicts the statistics for the 26 samples put into the second laboratory for check analyses of copper plotted on a Q-Q plot. Figure 16.6 depicts the same statistics for molybdenum analyses. Figure 16.7 shows the statistics for gold analyses and Figure 16.8 shows the statistics for silver analyses.

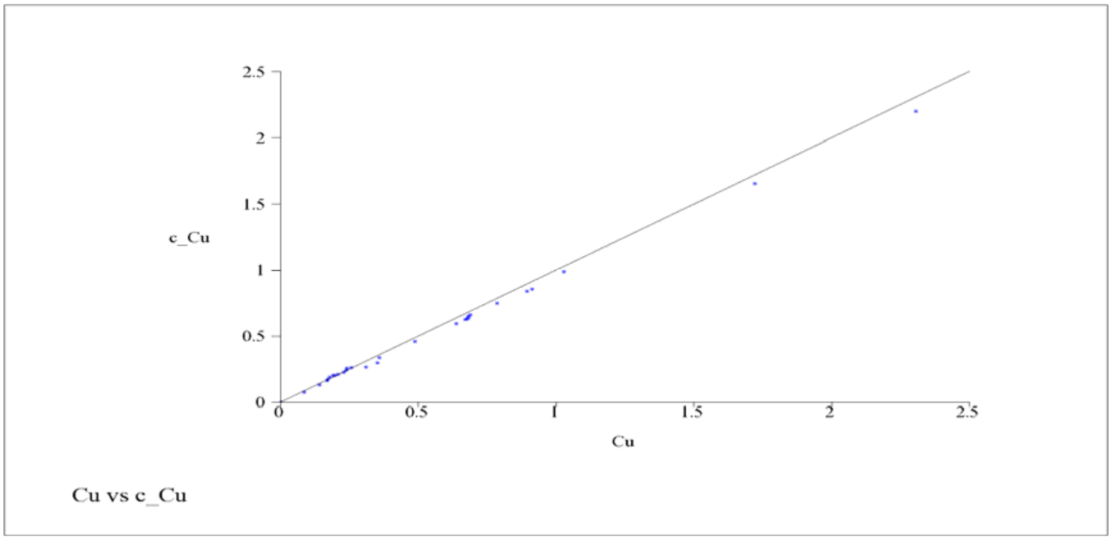


Figure 16.5 Statistics for Check Samples (2nd laboratory), Q-Q Plot of Cu

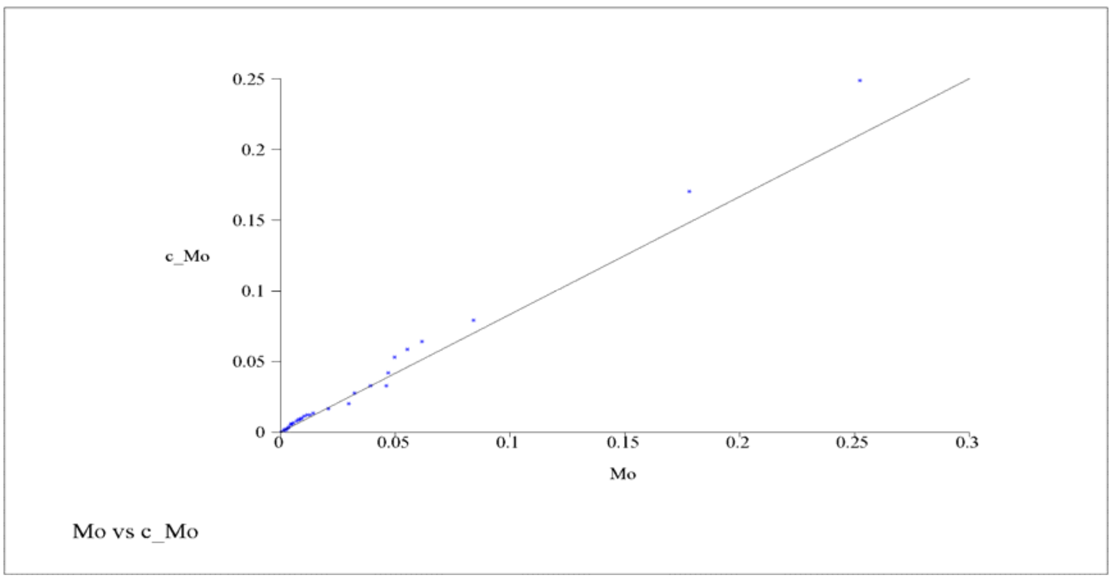


Figure 16.6 Statistics for Check Samples (2nd laboratory), Q-Q Plot of Mo

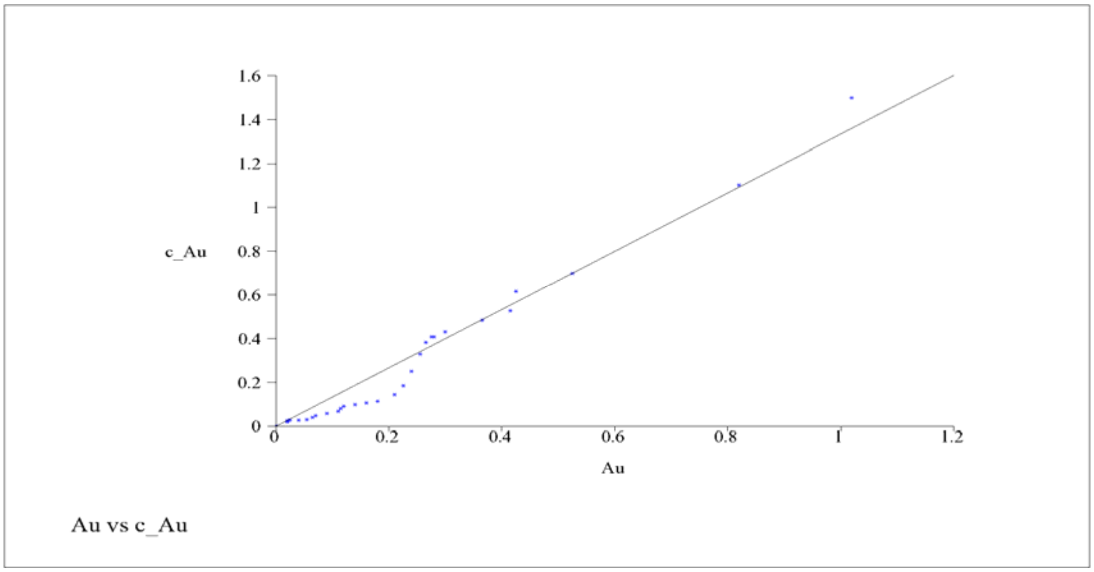


Figure 16.7 Statistics for Check Samples (2nd laboratory), Q-Q Plot of Au

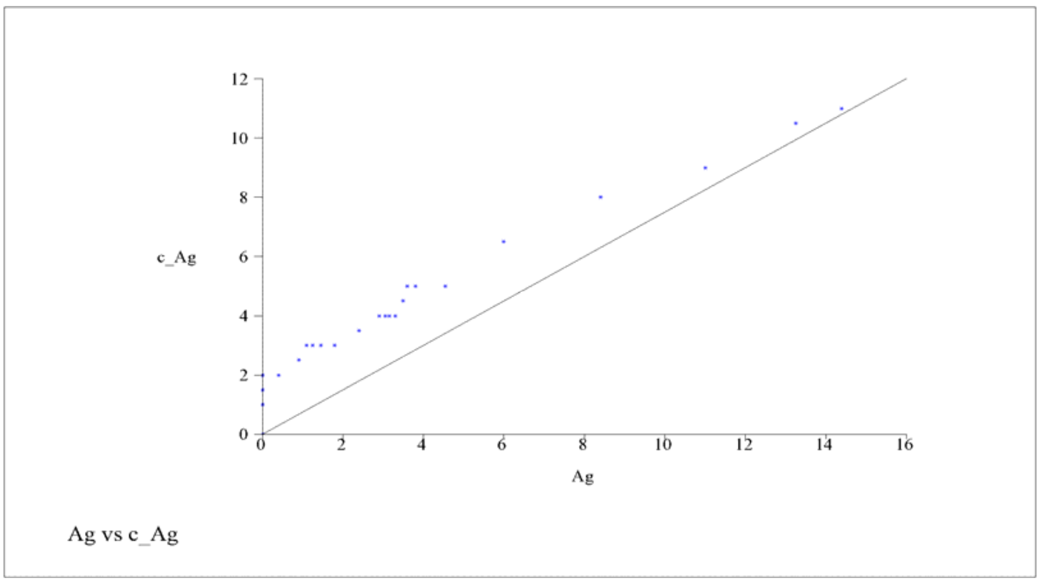


Figure 16.8 Statistics for Check Samples (2nd laboratory), Q-Q Plot of Ag

Good correlation exists for the copper and molybdenum sample populations whereas the silver shows higher grades for the check samples throughout the grade ranges. The gold shows lower grades in the check samples up to a concentration of about 0.3 g/t gold. Thereafter, good correlation exists between the two labs.

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 (about 1.8%) of the total sample population.

17.0 Adjacent Properties

17.1 Adjacent Properties

Figure 17.1 shows the minerals occurrences that are immediately adjacent to the Schaft Creek mineral property. None of these mineral occurrences are currently in or near production, although a few have exploration activities ongoing.

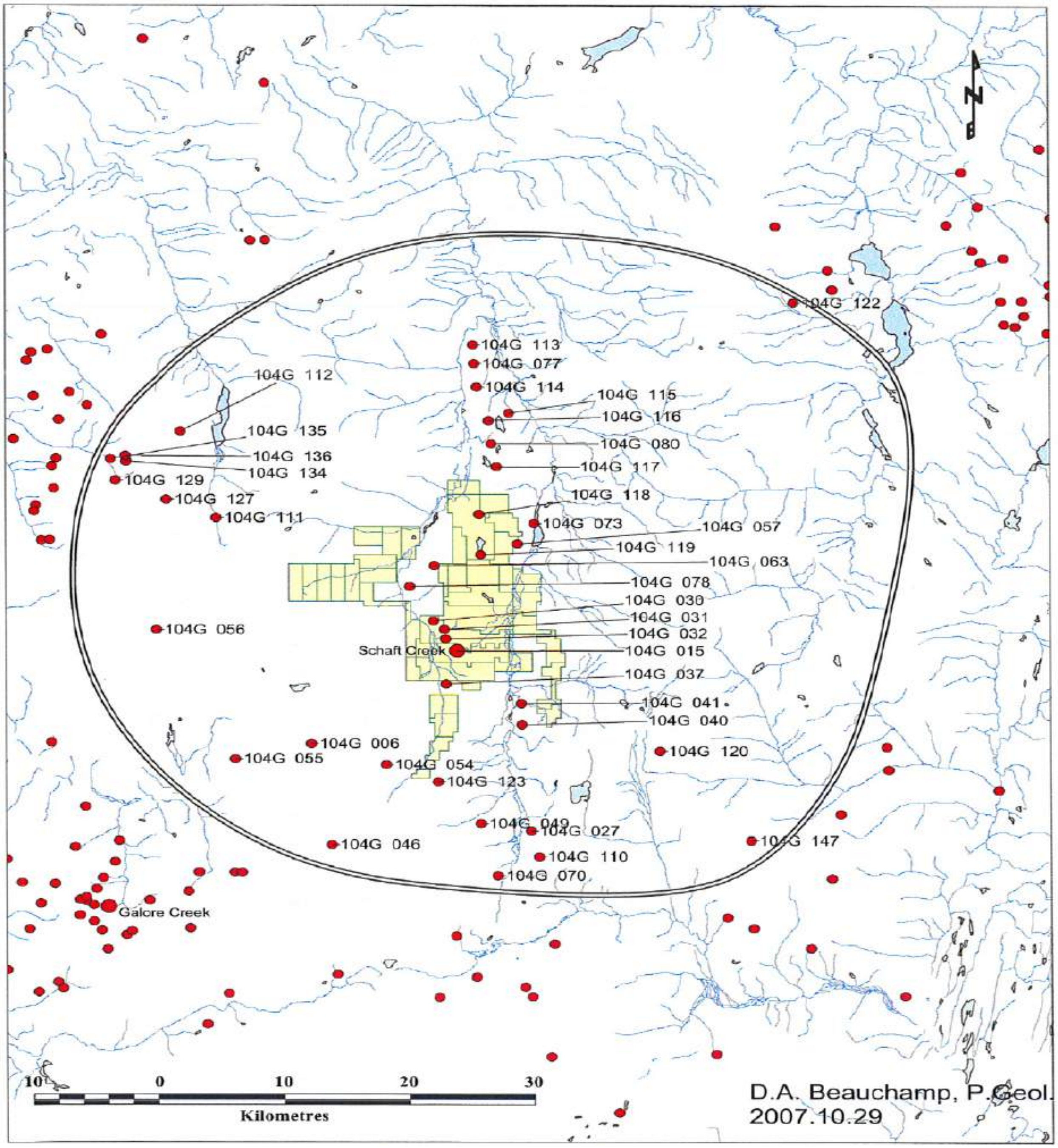


Figure 17.1 Mineral Occurrences Within 20 km of Schaft Creek

18.0 Mineral Processing and Metallurgical Testing

18.1 Description of the Metallurgical Test Program

The metallurgical test program completed for the Preliminary Feasibility Study (PFS) is a continuation of the test work for the development of the Schaft Creek Project. Some of the previous test work was reported in the report *Preliminary Economic Assessment on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada* that was issued December 7, 2007, a Canadian NI43-101 Technical Report. G&T Metallurgical Services, Ltd. (G&T) subsequently prepared a January 9, 2008 report titled *Floatation Responses of Three Ore Sources, Schaft Creek Deposits, Liard District, BC, Canada*, which included some information that was unavailable for the Preliminary Economic Assessment (PEA). Some data used in the PFS is from the earlier G&T report and additional information is from new metallurgical tests that were completed in support of this PFS. The newest data is reported in *Advanced Flowsheet Development Studies Schaft Creek Deposit Liard District, BC Canada* that was issued June 20, 2008.

Samples from 2005 and 2006 drill core were prepared for testing at the following laboratories:

- G&T Metallurgical Services, Ltd. in Kamloops, B.C., Canada;
- Hazen Research, Inc. in Golden, Colorado, USA;
- Polysius AG in Neubeckum, Germany;
- Teck Cominco CESL Technology laboratory in Vancouver, B.C., Canada.

18.2 Description of Metallurgical Samples

Vandan Suhbatar and Raymond R. Hyyppa, metallurgical consultants to Copper Fox Metals, Inc. (Copper Fox) prepared a list of samples for the metallurgical test program. G&T prepared the samples for testing and sent some of the samples to the other test facilities. All of the test work was conducted on drill core from the Schaft Creek 2005 and 2006 drilling programs. The Preliminary Economic Assessment (PEA) indicated that the Schaft Creek resource contained three mineral zones which have been described as Liard (Main) Zone, West Breccia Zone and the Paramount Zone. Approximately 85% of the resource is contained within the Liard Zone, 14% in the Paramount Zone and about 1% in the West Breccia Zone. Therefore the emphasis for testing was placed on the Liard Zone samples. Four types of metallurgical tests were implemented for this program. They are:

- Optimization Testing;
- Variability Testing;
- Comminution Testing;
- High Pressure Grinding Roll (HPGR) Testing.

A series of Optimization Tests were completed at G&T. While all of the previous test work indicated that the Schaft Creek resource performed well in a standard porphyry copper flotation flow sheet, these tests were designed to optimize the flow sheet and design parameters for the largest (Liard) Zone. The PEA indicated that the average grade for the first 5 years of production was approximately 0.35% Cu.

Therefore samples from a total of 42 three m intervals were selected from 12 PQ drill holes from the 2006 drill program for these tests.

The selected drill holes were uniformly distributed over the entire Liard Zone and the hole intervals were selected to represent the upper, middle and lower sections of the hole. The assays of the individual intervals ranged from less than 0.2% Cu to over 0.5% copper for each hole; the average grade for samples from each hole was approximately 0.35% Cu. G&T prepared composite samples from the drill-hole intervals into one 300 kg sample which was used for the Optimization Testing. It assayed approximately 0.36% Cu.

Next, a series of samples were selected based on grade and spatial variability. They were used to test the metallurgical variability of the Schaft Creek resource using the standard test conditions and flow sheet that was determined by the Optimization Tests. Samples from a total of 11 drill holes were selected for these tests. Ten of these holes were from the 2006 drill program and one sample was taken from the 2005 drill program. Ten of the holes were in the Liard Zone and one hole was selected from the Paramount Zone. A total of 34 three m drill-hole intervals were selected. The samples ranged in grade from less than 0.2% Cu to over 1.0% Cu. The total weight of this sample was approximately 250 kg.

A series of tests were conducted to complete the design of the comminution circuit. Five HQ drill holes were selected from around the Liard Zone for testing by Hazen Research, Golden, Colorado, USA. Hazen is licensed to conduct the tests required to determine the JKTech comminution parameters that are used by JKTech for computer simulations of grinding circuits. Following receipt of the data from these tests, the computer simulations were conducted by Mark Richardson, JKTech Support services, Red Bluff, CA, USA. Nine (45 total) three m intervals were selected from the 5 drill holes to represent the upper, middle and lower sections of each hole. Samples from each hole were composited individually and each composite weighed approximately 90 kg. The total weight of samples was approximately 450 kg.

The Schaft Creek resource is a potential candidate for High Pressure Grinding Roll (HPGR) technology. The material is hard and only moderately abrasive. A total of 35 drill-hole intervals were selected from 7 drill holes in the Liard Zone for the HPGR tests. Each interval was crushed to approximately 25 mm and composited into one sample weighing approximately 250 kg.

Additional information about the samples that were selected and prepared for the metallurgical testing that was conducted to support the Preliminary Feasibility Study is provided in Table 18.1

Table 18.1
Sample Selection For G&T 2008 Schaft Creek Metallurgical Testing

Block	Ore Zone	Hole #	From	To	Met Sample #	Assay Sample #	Grade, % Cu
OPTIMIZATION TESTING							
Block 1	**Liard**	05CF236	210.00	220.00	15,089	14,181	0.366
			320.00	330.00	15,100	14,159	0.367
			540.00	550.00	15,122	14,145	0.360
						Average	**0.364**

Table 18.1 Sample Selection For G&T 2008 Schaft Creek Metallurgical Testing							
Block	**Ore Zone**	**Hole #**	**From**	**To**	**Met Sample #**	**Assay Sample #**	**Grade, % Cu**
Block 1	**Liard**	06CF250	45.75	48.80	126,564	125,109	0.240
			70.15	73.20	126,572	125,117	0.540
			57.95	61.00	126,568	125,113	0.280
						Average	**0.353**
Block 1	**Liard**	06CF251	36.60	39.65	126,693	125,289	0.460
			76.25	79.30	126,706	125,305	0.350
			97.60	100.65	126,713	125,312	0.220
						Average	**0.343**
Block 2	**Liard**	06CF255	167.25	170.80	126,721	125,366	0.350
			192.50	195.20	126,729	125,374	0.400
			234.85	237.90	126,743	125,391	0.310
						Average	**0.353**
Block 4	**Liard**	06CF256	137.25	140.30	126,904	125,745	0.310
			244.00	247.05	126,939	125,783	0.410
			259.25	262.30	126,944	125,788	0.200
						Average	**0.307**
Block 6	**Liard**	06CF258	94.55	97.60	126,771	125,443	0.460
			131.15	134.20	126,783	125,455	0.260
			128.10	131.15	126,782	125,454	0.350
						Average	**0.357**
Block 2	**Liard**	06CF259	115.90	118.95	127,205	126,225	0.270
			131.15	134.20	127,210	126,230	0.350
			198.25	201.30	127,232	126,252	0.420
						Average	**0.347**
Block 7	**Liard**	06CF262	143.35	146.40	127,327	145,696	0.230
			155.55	158.60	127,331	145,703	0.460
			183.00	186.05	127,340	145,712	0.370
			176.90	179.95	127,338	145,710	0.280
			122.00	125.05	127,320	145,689	0.200
			128.10	131.15	127,322	145,691	0.450
						Average	**0.310**
Block 4	**Liard**	06CF265	39.65	42.70	127,515	146,956	0.270
			88.45	91.50	127,531	146,972	0.470
			179.95	183.00	127,561	147,005	0.350
						Average	**0.363**
Block 5	**Liard**	06CF268	186.05	189.10	127,388	145,937	0.380

Table 18.1 Sample Selection For G&T 2008 Schaft Creek Metallurgical Testing							
Block	**Ore Zone**	**Hole #**	**From**	**To**	**Met Sample #**	**Assay Sample #**	**Grade, % Cu**
			94.55	97.60	127,358	145,907	0.520
			143.35	146.40	127,374	145,923	0.230
						Average	**0.377**
Block 3	**Liard**	06CF284	24.35	27.40	126,627	125,193	0.290
			45.75	48.80	126,634	125,200	0.400
			67.10	70.15	126,641	125,207	0.170
			195.20	198.25	126,669	125,252	0.500
			170.80	173.85	126,661	125,244	0.180
			73.15	76.20	126,643	125,209	0.520
						Average	**0.400**
Block 4	**Liard**	06CF285	103.70	106.75	126,863	125,630	0.240
			179.95	183.00	126,888	125,655	0.330
			225.70	228.75	126,903	125,673	0.520
						Average	**0.363**
			OPTIMIZATION TESING			Number	42
						Average	0.353
						Maximum	0.540
						Minimum	0.170
VARIABILITY TESTING							
Block 1	**Liard**	06CF251	94.55	97.60	126,712	125,311	0.160
			73.20	76.25	126,705	125,304	0.420
			88.50	91.05	126,710	125,309	0.560
			39.65	42.70	126,694	125,290	0.890
			64.05	67.10	126,702	125,301	1.100
Block 2	**Liard**	06CF255	33.55	36.60	126,720	125,319	0.180
			179.95	183.00	126,725	125,317	0.400
			216.55	219.60	126,737	125,385	0.620
Block 3	**Liard**	06CF284	134.20	137.25	126,649	125,232	0.200
			76.20	79.25	126,644	125,210	0.370
			51.80	54,85	126,636	125,202	0.620
Block 4	**Liard**	06CF285	109.80	112.85	126,865	125,632	0.220
			143.25	146.60	126,876	125,643	0.420
			161.65	164.70	126,882	125,649	0.590
			158.60	161.65	126,881	125,648	0.790

Table 18.1
Sample Selection For G&T 2008 Schaft Creek Metallurgical Testing

Block	Ore Zone	Hole #	From	To	Met Sample #	Assay Sample #	Grade, % Cu
Block 4		06CF265	216.55	219.60	127,573	147,017	0.770
Block 4		06CF256	207.40	210.45	126,927	125,768	1.030
Block 5	**Liard**	06CF268	64.05	67.10	127,348	145,894	0.220
			76.25	79.30	127,352	145,901	0.410
			155.55	158.60	127,378	145,927	0.630
			149.45	152.50	127,376	145,925	0.860
			152.50	155.55	127,377	145,926	1.000
Block 6	**Liard**	06CF258	85.40	88.45	126,768	125,440	0.220
			112.85	115.90	126,777	125,449	0.410
			137.25	140.30	126,785	125,457	0.580
			143.35	146.40	126,787	125,462	0.820
			179.95	183.00	126,799	125,474	1.130
Block 7	**Liard**	06CF262	125.05	128.10	127,321	145,690	0.220
			177.85	176.90	127,337	145,709	0.330
Block 8	**Paramount**	06CF287	70.15	73.20	127,121	126,056	0.240
			85.40	88.45	127,126	126,064	0.360
			161.65	164.70	127,151	126,089	0.660
			143.35	146.40	127,145	126,083	0.990
Block 8		06CF289	112.85	115.90	127,197	126,158	0.950
				VARIABILITY TESING		**Number**	34
						Average	0.570
						Maximum	1.130
						Minimum	0.160
COMMINUTION TESTING BY HAZEN RESEARCH							
Block 1	**Liard**	CH276	161.65	164.70		146,558	0.310
			164.70	167.75		146,559	0.230
			167.75	170.80		146,560	0.190
			170.80	173.85		146,561	0.390
			173.85	176.90		146,562	0.220
			176.90	179.95		146,563	0.320
			179.95	183.00		146,564	0.380
			183.00	186.05		146,565	0.150
			185.05	189.10		146,566	0.140

Table 18.1 Sample Selection For G&T 2008 Schaft Creek Metallurgical Testing							
Block	**Ore Zone**	**Hole #**	**From**	**To**	**Met Sample #**	**Assay Sample #**	**Grade, % Cu**
						Average	**0.259**
Block 2	**Liard**	CH274	70.15	73.20		146,025	0.460
			73.20	76.25		146,026	0.200
			76.25	79.30		146,027	0.440
			79.30	82.35		146,028	0.380
			82.35	85.40		146,029	0.330
			85.40	88.45		146,030	0.230
			88.45	91.50		146,031	0.150
			91.50	94.55		146,032	0.210
			94.55	97.60		146,033	0.350
						Average	**0.306**
Block 3	**Liard**	CH278	48.80	51.85		146,640	0.450
			51.85	54.90		146,641	0.310
			54.90	57.95		146,642	0.270
			57.95	61.00		146,643	0.230
			61.00	64.05		146,644	0.160
			64.05	67.10		146,645	0.200
			67.10	70.15		146,646	0.200
			70.15	73.20		146,647	0.180
			73.20	76.25		146,648	0.100
						Average	**0.233**
Block 4	**Liard**	CH267	9.15	12.20		126,397	0.520
			12.20	15.25		126,398	0.350
			15.25	18.30		126,399	0.410
			18.30	21.35		126,400	0.220
			21.35	24.40		126,401	0.480
			24.40	27.45		126,402	1.000
			27.45	30.50		126,403	0.400
			30.50	33.55		126,404	0.280
			33.55	36.60		126,405	0.650
						Average	**0.479**
Block 7	**Liard**	CH270	100.65	103.70		146,897	0.300
			103.70	106.75		146,901	0.280
			106.75	109.80		146,902	0.680

Table 18.1 Sample Selection For G&T 2008 Schaft Creek Metallurgical Testing							
Block	**Ore Zone**	**Hole #**	**From**	**To**	**Met Sample #**	**Assay Sample #**	**Grade, % Cu**
			109.80	112.85		146,903	0.500
			112.85	115.90		146,904	0.540
			115.90	118.95		146,905	0.820
			118.95	122.00		146,906	0.650
			122.00	125.05		146,907	0.460
			125.05	128.10		146,908	0.480
						Average	**0.523**
HIGH PRESSURE GRINDING ROLL TESTING (POLYSIUS AG)							
Block 1	**Liard**	06CF251	42.70	45.75	126,695		0.630
			45.75	48.80	126,696		0.560
			48.80	51.85	126,697		0.540
			51.85	54.90	126,698		0.480
			54.90	57.95	126,699		0.690
					Average		0.580
Block 2	**Liard**	06CF255	219.60	222.65	126,738		0.380
			222.65	225.70	126,739		0.380
			225.70	228.75	126,740		0.310
			228.75	231.80	126,741		0.570
			231.80	234.85	126,742		0.420
					Average		0.412
Block 3	**Liard**	06CF284	27.40	30.45	126,628		0.290
			30.45	33.50	126,629		0.300
			33.50	36.55	126,630		0.250
			39.60	42.65	126,632		0.490
			42.65	45.70	126,633		0.360
					Average		0.338
Block 4	**Liard**	06CF285	115.90	118.95	126,867		0.340
			118.95	122.00	126,868		0.420
			122.00	125.05	126,869		0.540
			125.05	128.10	126,870		0.340
			128.10	131.15	126,871		0.240
					Average		0.376
Block 5	**Liard**	06CF268	100.65	103.70	127,360		0.360
			103.70	106.75	127,361		0.360

Table 18.1 Sample Selection For G&T 2008 Schaft Creek Metallurgical Testing							
Block	**Ore Zone**	**Hole #**	**From**	**To**	**Met Sample #**	**Assay Sample #**	**Grade, % Cu**
			106.75	109.80	127,362		0.460
			109.80	112.85	127,363		0.360
			112.85	115.90	127,364		0.250
					Average		0.358
Block 6	**Liard**	06CF258	152.50	155.55	126,790		0.690
			155.55	158.60	126,791		0.920
			158.60	161.65	126,792		1.080
			161.65	164.70	126,793		0.830
			164.70	167.75	126,794		0.950
					Average		0.894
Block 7	**Liard**	06CF262	131.15	134.20	127,323		0.220
			134.20	137.25	127,324		0.230
			137.25	140.30	127,325		0.170
			186.50	189.10	127,341		0.290
			189.10	192.15	127,342		0.220
					Average		0.226

18.3 Mineralogical Assessment

Detailed mineralogical studies on process streams from locked-cycle tests conducted in the earlier test program were completed in order to help determine the efficiency of the process and indicate possible areas of improvement. The results of this assessment were reported by G&T in *Advanced Flowsheet Development Studies Schaft Creek Deposit Liard District, BC Canada*.

For the earlier phase of testing, samples from the three main geographical zones were tested. They were:

- Paramount (PZ);
- Liard (LZ);
- West Breccia (WZ).

The exit streams and feed from one of the locked-cycle tests conducted on each composite were subjected to modal analysis. The streams from one copper-molybdenum locked-cycle test were also analyzed mineralogically. Table 18.2 summarizes the results from the locked-cycle tests.

Table 18.2 Schaft Creek Locked-Cycle Bulk Flotation Test Data Summary											
Product	**Mass %**	**Assay, % or g/t**					**Distribution, %**				
		Cu	**Mo**	**S**	**Ag**	**Au**	**Cu**	**Mo**	**S**	**Ag**	**Au**
PZ Composite											
Flotation Feed	100	0.27	0.016	0.4	3	0.22	100	100	100	100	100
Bulk Concentrate	0.8	26.2	1.52	34.1	190	19.4	78	75	74	48	70
Bulk Final Tail	99.2	0.06	0.004	0.1	2	0.07	22	25	26	52	70
LZ Composite											
Flotation Feed	100.0	0.30	0.015	0.3	2	0.23	100	100	100	100	100
Bulk Concentrate	0.9	29.8	1.48	32.0	194	20.8	85	86	82	74	76
Bulk Final Tail	99.1	0.05	0.002	0.1	1	0.06	15	14	18	26	24
WZ Composite											
Flotation Feed	100.0	0.69	0.026	0.8	6	0.38	100	100	100	100	100
Bulk Concentrate	2.0	29.9	0.80	32.9	252	16.2	87	61	86	81	85
Bulk Final Tail	98.0	0.09	0.010	0.1	1	0.06	13	39	14	19	15

In reviewing the data, it can be seen that the sample from the Paramount Zone (PZ) performed the worst with only 78% copper recovery to a bulk concentrate that only assayed 26.2% copper. The samples from the Liard Zone (LZ) and West Breccia (WZ) performed similarly with 85% and 87% copper recovery, respectively, into bulk concentrates with grades of approximately 30%.

Bulk concentrate and rougher tailings samples from cycles IV and V of the locked-cycle tests for each of the three samples were analyzed by modal techniques. These techniques measure the mineral content and fragmentation by mineral class and particle-size range. Detailed test results can be found in the G&T report, Advanced Flowsheet Development Studies Schaft Creek Deposit Liard District, BC Canada. The data is summarized as follows:

- For all ore composite samples, the majority of the copper sulphide losses occurred as binary particles interlocked with non-sulphide gangue minerals. Small quantities of liberated copper sulphides were lost to tailings;
- In the tailings most of the molybdenite occurred as unliberated particles locked with non-sulphide gangue minerals except the sample from the Liard zone where significant liberated grains of molybdenite were detected;
- Further comminution improvements are necessary to improve copper sulphide liberation;
- For all samples the majority of the copper losses occurred in fine-grained particles where copper sulphides were interlocked with non-sulphide gangue. Because of the fine particle sizes, it is not anticipated that the copper recovery can be improved significantly;
- The tailings samples from the Paramount and West Breccia zone composite samples showed molybdenite losses in the coarser-sized fractions which were comprised of

unliberated molybdenite grains locked in binary form with non-sulphide gangue minerals. This indicates that there may be room to improve molybdenum recovery;
- The majority of the molybdenite loss for the Liard zone composite sample occurred as liberated molybdenite grains that were finely sized;
- The bulk concentrates contained 10 to 21% pyrite and significant quantities of non-sulphide gangue minerals, much of which was liberated. This observation indicated a high potential to improve the quality of the bulk concentrate;
- Secondary copper sulphide minerals were present in the concentrates which indicated that a copper concentrate grade of 38 to 40% copper can theoretically be produced.

Modal analyses were also used to evaluate the efficiency of the copper-molybdenum separation process. A summary of the observations from this analysis are as follows:

- The molybdenum concentrate grades were about 50%. The molybdenum recovery from the bulk concentrate averaged about 70%;
- The cleaner tailings contained the majority of the molybdenite losses and additional molybdenite was found in the copper concentrate and the molybdenum rougher tailings. Most of these losses occurred as finely-sized, liberated grains of molybdenite. It is difficult to recover fine-grained particles of molybdenite because the collectors are non-selective collectors that also increase the recover of gangue and pyrite to the molybdenum concentrate.

The Automated Digital Imaging System (ADIS) was used to scan the copper concentrate for gold particles. A summary of the observations is as follows:

- The average size of the gold particles was 11 microns which is too small to be recovered by gravity concentration;
- Most of the gold was liberated and the gold that was not liberated was locked with copper sulphide minerals or multiphase particles. These gold occurrences should be recovered in the copper concentrate.

18.4 Optimization Testing

G&T was directed to prepare a Master Composite of the Liard Zone according the sample descriptions shown in Table 5.1. Two sets of three locked-cycle tests were conducted on the Master Composite. These tests were used to confirm the most appropriate primary grind size for rougher flotation. Tests were conducted at primary grind sizes of approximately 80% passing 109, 142 and 173 microns. Rougher flotation concentrates were reground to approximately 80% passing 20 microns prior to cleaning in a conventional 3-stage cleaner flotation circuit to produce a bulk copper/molybdenum concentrate. A summary of the results of the locked-cycle tests is given in Table 18.3. These data indicate that relatively high copper concentrate grades were achievable at all grind sizes. Copper recovery in the bulk concentrate appeared to be optimum at the 109 micron primary grind size. However, molybdenum, gold and silver recoveries improved at coarser grind sizes.

Therefore, an estimate of the overall optimum grind size was made based on the recovered value of the copper and molybdenum concentrates. The three-year trailing average metal prices were used to estimate the value of the concentrates produced by each grind size.

The cost of grinding power for each of the three grind sizes was deducted to estimate the net value of the concentrate. Figure 18.1 shows the optimum primary grind size to be approximately 150 microns, although the mathematical correlation used to make this determination is weak. This grind size is a change from the Scoping Study which was based on a primary grind size of 80% passing 100 microns.

Table 18.3
Schaft Creek Locked-Cycle Metallurgical Test Data

Product	Mass, %	Assays				Distribution			
		Cu, %	Mo, %	Ag, g/t	Au, g/t	Cu	Mo	Ag	Au
Master Composite (K_{80} = 109µm)									
Flotation Feed	100	0.32	0.013	2.1	0.26	100%	100%	100%	100%
Bulk Concentrate	0.90	32.0	1.040	142.0	19.0	89%	71%	60%	67%
Cleaner Tails	9.3	0.14	0.020	3.0	0.30	4%	14%	11%	9%
Rougher Tails	89.8	0.03	0.002	0.7	0.07	7%	15%	29%	24%
Master Composite (K_{80} = 142µm)									
Flotation Feed	100	0.33	0.014	2.1	0.27	100%	100	100%	100%
Bulk Conc.	0.89	32.3	1.210	135.0	21.1	87%	78%	59%	71%
Cleaner Tails	9.9	0.15	0.018	2.0	0.30	4%	11%	11%	9%
Rougher Tails	89.2	0.03	0.002	0.7	0.06	9%	11%	30%	20%
Master Composite (K_{80} = 173µm)									
Flotation Feed	100	0.33	0.012	2.4	0.24	100%	100%	100%	100%
Bulk Concentrate	0.92	30.9	1.120	143.0	18.8	86%	87%	55%	72%
Cleaner Tails	9.1	0.15	0.009	3.0	0.20	4%	7%	9%	9%
Rougher Tails	90.0	0.04	0.001	0.9	0.05	10%	6%	36%	19%

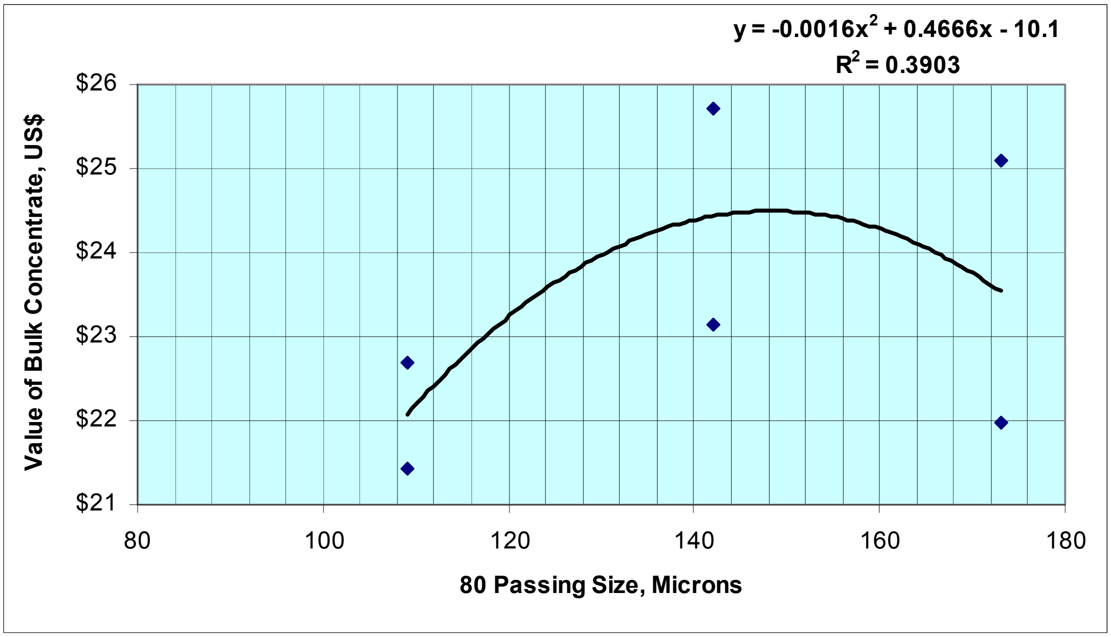


Figure 18.1 Schaft Creek Optimum Primary Grind

The locked-cycle test data and test data from previous metallurgical testing programs were used to predict the metal recoveries to copper and molybdenum concentrates at the 150-mesh grind. The 1st Cleaner Tailings from the Bulk Copper/Molybdenum Circuit were sent to Final Tailings during the G&T locked-cycle tests.

The PFS is based on scavenging the 1st Cleaner Tailings and recycling the Scavenger Conc. to the 1st Cleaners. Thus, additional metal values will be recovered. METSIM was used to simulate the Schaft Creek flowsheet utilizing the G&T test date to estimate the copper and molybdenum concentrate grades and recoveries that are shown in Table 18.4 and Table 18.5.

Table 18.4
Schaft Creek Estimated Flotation Concentrate Grades

Metal	Copper Concentrate Grade, % Copper	Molybdenum Concentrate Grade, % Molybdenum
Copper	33.52%	1.06%
Molybdenum	0.21%	50.00%
Gold	22.4 g/t	N/A
Silver	134.2 g/t	N/A
Rhenium	N/A	368 ppm

Table 18.5 Schaft Creek Estimated Recoveries in the Flotation Concentrates		
Metal	**Copper Concentrate Recovery, % Copper**	**Molybdenum Concentrate Recovery, % Molybdenum**
Copper	89.21%	0.07%
Molybdenum	10.82%	68.61%
Gold	80.71%	N/A
Silver	72.00%	N/A
Rhenium	N/A	N/A

Additional details regarding the details of the tests are shown in Table 18.10 and Table 18.11.

18.5 Variability Testing

A total of 30 three metre PQ drill-hole sections were selected from 10 holes in the Liard Zone for variability testing. And a total of 4 three metre PQ drill hole sections were selected from one drill hole in the Paramount Zone. The samples weighed approximate 250 kg. The drill-hole sections were selected to test samples scattered uniformly over the resource zone and at varying depths. The samples were also selected on the basis of copper head grades, which ranged form approximately 0.20% to 1.0% copper. In addition to the chemical analyses that were completed on the variability samples, mineralogical estimates of the sulphide mineral contents were made using optical mineralogical methods, as discussed in the latest G&T report. A summary of the mineralogical composition of the variability samples is shown in Table 18.6.

Table 18.6 Mineral Composition of the Variability Samples									
Sample	Metal Assay		Weight Percent						
	Cu	Fe	Cp	Bn	Ch/Cv	Py	Ma	He	Gn
126636	0.60	1.95	0.36	0.75	0.00	0.02	0.45	0.16	98.57
126644	0.32	3.00	0.00	0.35	0.13	0.00	0.00	0.36	99.35
126649	0.21	1.22	0.50	0.05	0.00	0.02	0.32	0.00	99.27
126694	0.77	2.81	0.15	0.98	0.13	0.03	0.50	0.15	98.41
126702	0.90	1.70	0.00	1.24	0.15	0.00	0.32	0.12	98.40
126705	0.36	2.86	0.01	0.56	0.00	0.01	0.38	1.17	98.68
126710	0.48	2.36	0.03	0.69	0.04	0.01	0.50	0.50	98.75
126712	0.16	1.57	0.00	0.24	0.00	0.03	1.19	0.63	98.86
126720	0.15	4.67	0.10	0.18	0.00	0.02	6.06	0.93	96.35
126725	0.27	4.41	0.33	0.24	0.00	0.00	1.39	1.16	98.21
126737	0.63	4.32	1.55	0.14	0.00	0.34	1.42	0.43	97.08
126768	0.20	4.97	0.03	0.30	0.00	0.01	2.75	0.89	97.92
126777	0.37	3.80	0.97	0.05	0.00	0.37	1.18	0.65	97.73
126785	0.49	3.02	0.93	0.27	0.00	0.01	0.31	1.13	98.12
126787	0.76	3.67	1.03	0.63	0.00	0.00	1.04	1.31	97.22
126799	0.90	1.29	1.04	0.23	0.00	0.00	0.33	1.28	97.96

Table 18.6
Mineral Composition of the Variability Samples

Sample	Metal Assay		Weight Percent						
	Cu	Fe	Cp	Bn	Ch/Cv	Py	Ma	He	Gn
126865	0.21	0.69	0.60	0.00	0.00	0.05	0.36	0.00	99.18
126876	0.38	3.51	0.47	0.34	0.00	0.00	0.24	0.09	99.03
126881	0.52	3.53	0.37	0.42	0.16	0.00	1.53	0.13	98.25
126882	0.51	3.62	0.17	0.69	0.02	0.01	1.10	0.29	98.45
126927	0.98	2.44	1.91	0.50	0.00	0.08	0.48	0.00	97.29
127121	0.17	2.83	0.43	0.03	0.00	0.37	0.46	0.11	98.89
127126	0.27	2.71	0.49	0.00	0.00	0.00	0.65	0.05	99.17
127145	1.17	1.68	3.37	0.00	0.00	0.05	0.24	0.00	96.47
127151	0.78	1.56	2.25	0.00	0.00	0.41	0.47	0.04	97.10
127197	1.05	2.64	1.94	0.58	0.01	0.01	0.09	0.01	97.41
127321	0.26	4.82	0.74	0.00	0.00	0.03	0.18	0.03	99.12
127337	0.33	3.73	0.95	0.00	0.00	0.53	1.91	0.16	97.53
127348	0.16	3.88	0.37	0.05	0.00	0.02	0.41	0.43	99.16
127352	0.33	2.49	0.95	0.00	0.00	0.57	0.18	0.14	98.32
127376	0.78	5.28	2.25	0.00	0.00	0.16	1.03	1.02	96.61
127377	0.91	5.74	2.59	0.00	0.00	0.14	1.27	0.14	96.59
127378	0.63	6.30	1.81	0.00	0.00	0.36	0.59	0.14	97.48
127573	0.38	4.14	0.24	0.43	0.03	0.01	1.61	1.18	97.97

Notes: a) Cp-Chalcopyrite, Bn-Bornite, Ch/Cv-Chalcocite/Covellite, Py-Pyrite, Ma-Magnetite, He-Hematite, Gn-Non-sulphide minerals.
b) Weight percent are based on mineral occurrences that are sized greater than C6 size (approximately 5µm).

G&T conducted one open-circuit test for each sample using the test conditions from the locked-cycle tests. While the tests were not optimized, the results are very encouraging. Even at very low copper head grades, the copper grade in the bulk concentrate was unexpectedly high. A copper flotation concentrate grade of 51.9% copper was achieved for a sample with a copper head grade of 0.38%, indicating that there are significant quantities of secondary copper minerals present in the samples that were tested. This was substantiated by a mineralogical evaluation of the metallurgical samples. The copper recoveries to the bulk concentrate were also consistently high for non-optimized open circuit tests. Copper recoveries increased as the Head Grade increased, but relatively high copper recoveries were achieved even at low head grades as indicated in Figure 18.2. Figure 18.3 indicates the copper grade of the bulk concentrate even at low head grades.

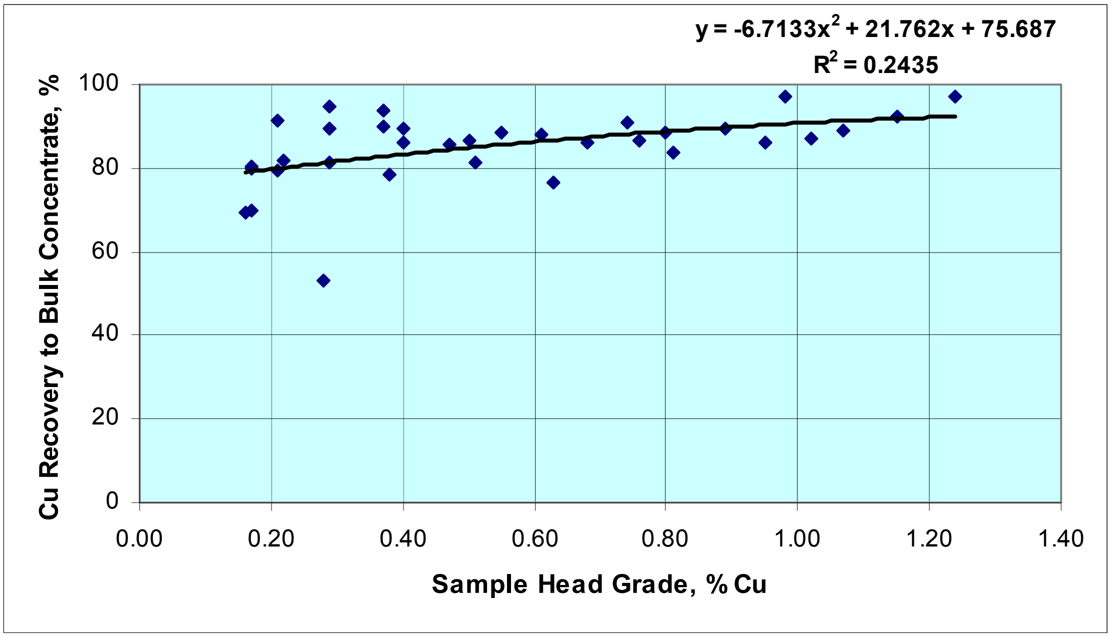


Figure 18.2 Copper Recovery to Bulk Concentrate as a Function of Copper Head Grade

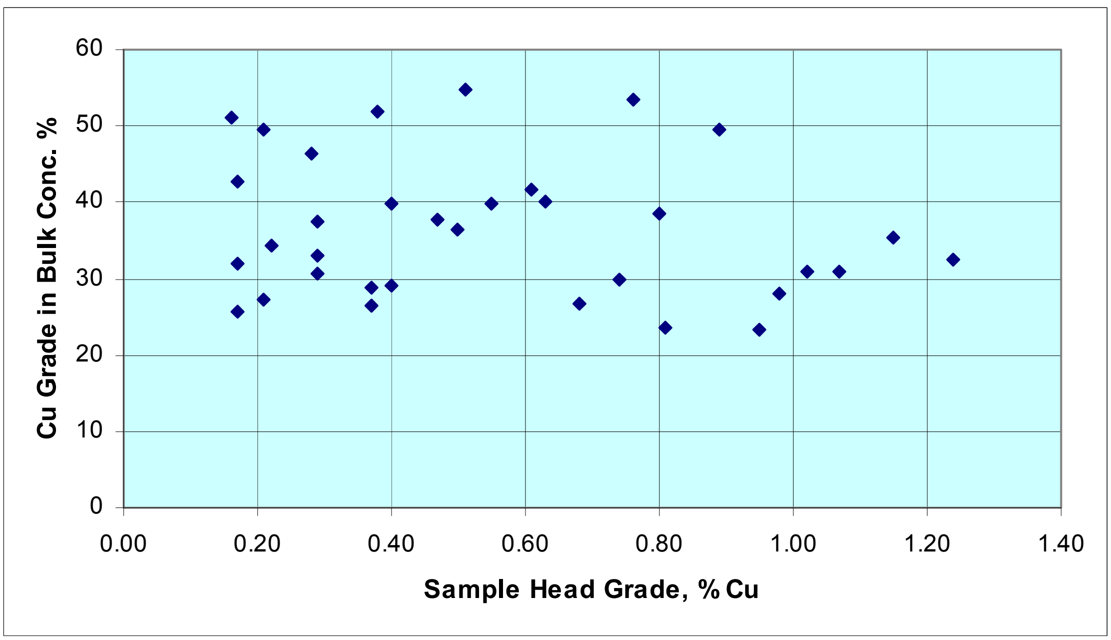


Figure 18.3 Copper Grade of the Bulk Concentrate as a Function of the Copper Head Grade

Sample hardness was evaluated by calculating the Comparative Bond Work Index (BW_i) values of each sample. G&T used the sample grinding time to estimate the work index using to the standard Comparative Work Index procedure.

While this is not an accurate measurement of the sample hardness, it provides an idea of the variability of the hardness of the deposit. These values range in hardness from a low of 19.11 to a high of 44.54, as shown in Table 18.7 and in Figure 18.4. The ore is consistently hard to very hard. Previous Bond Work Index tests of sample composites average approximately 22.9. While there is a lot of scatter in the data, it appears that the sample hardness may possibly be related to the copper grade. The higher-grade samples tend to exhibit a lower Comparative Hardness than the lower grade samples.

Table 18.7
Schaft Creek Comparative Bond Work Indices

Met Sample Number	Comparative Bond Work Index
126,636	29.78
126,644	22.62
126,649	35.36
126,694	25.29
126,702	23.63
126,705	27.55
126,710	35.46
126,712	27.34
126,720	38.68
126,725	35.83
126,737	29.78
126,768	38.13
126,777	29.15
126,785	35.68
126,787	35.46
126,799	34.29
126,865	21.93
126,876	29.15
126,881	31.97
126,882	28.99
126,927	25.78
127,121	28.99
127,126	26.10
127,145	22.79
127,151	22.96
127,197	22.10
127,321	21.24
127,337	34.29
127,348	44.54
127,352	40.27
127,376	20.01
127,377	25.78
127,378	19.11
127,573	28.35

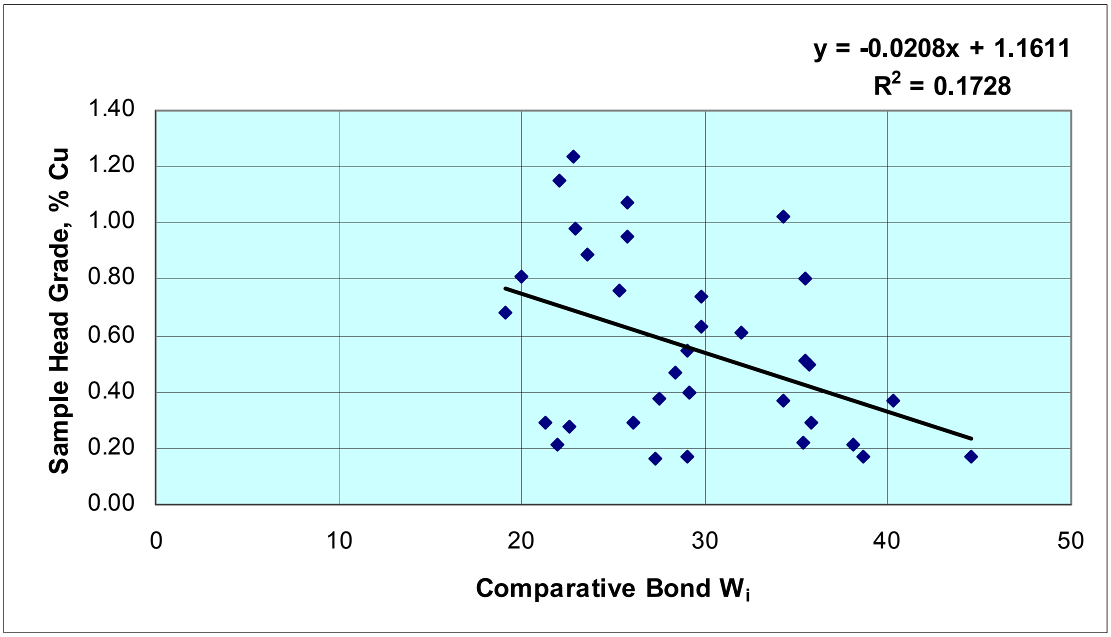


Figure 18.4 Schaft Creek Variability Tests Comparative Work Indices

Molybdenum, gold and silver recovery into the bulk concentrate increases with copper recovery, as has been seen in all the previous Schaft Creek tests. Please refer to Figure 18.5, Figure 18.6 and Figure 18.7, respectively. Based on these results, the best way to ensure high recoveries of molybdenum, gold and silver is to achieve high copper recoveries.

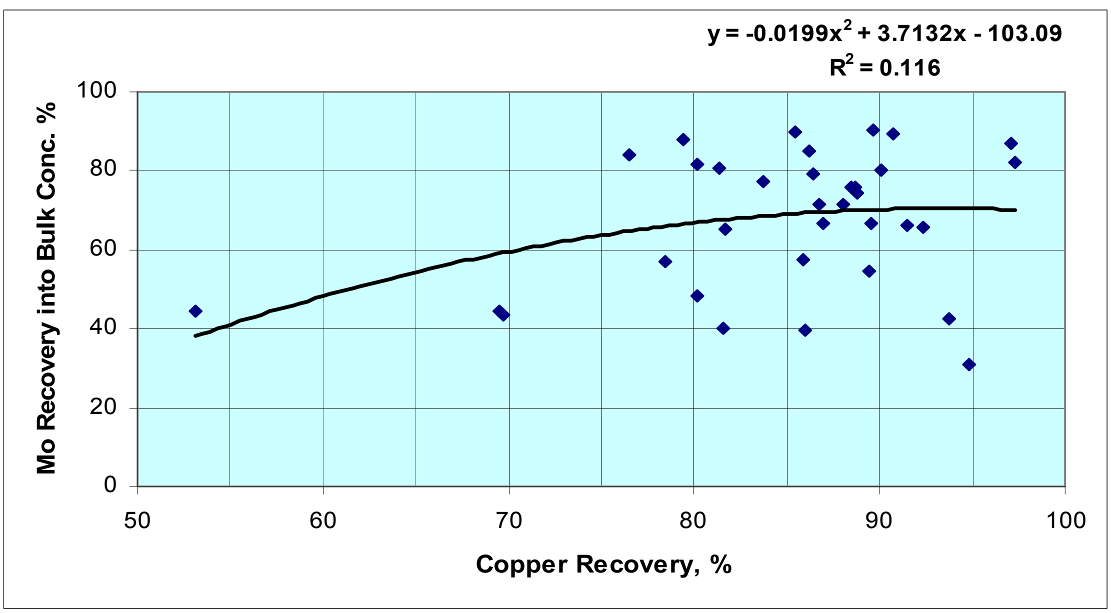


Figure 18.5 Molybdenum as a Function of Copper Recovery in the Bulk Concentrate

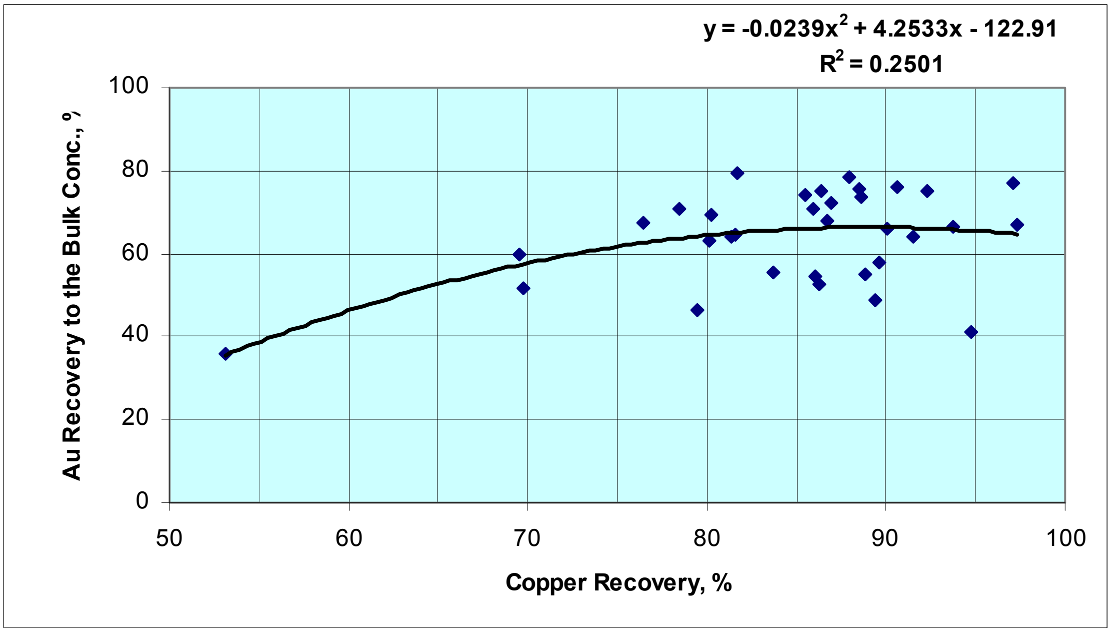


Figure 18.6 Gold Recovery as a Function of Copper Recovery in the Bulk Concentrate

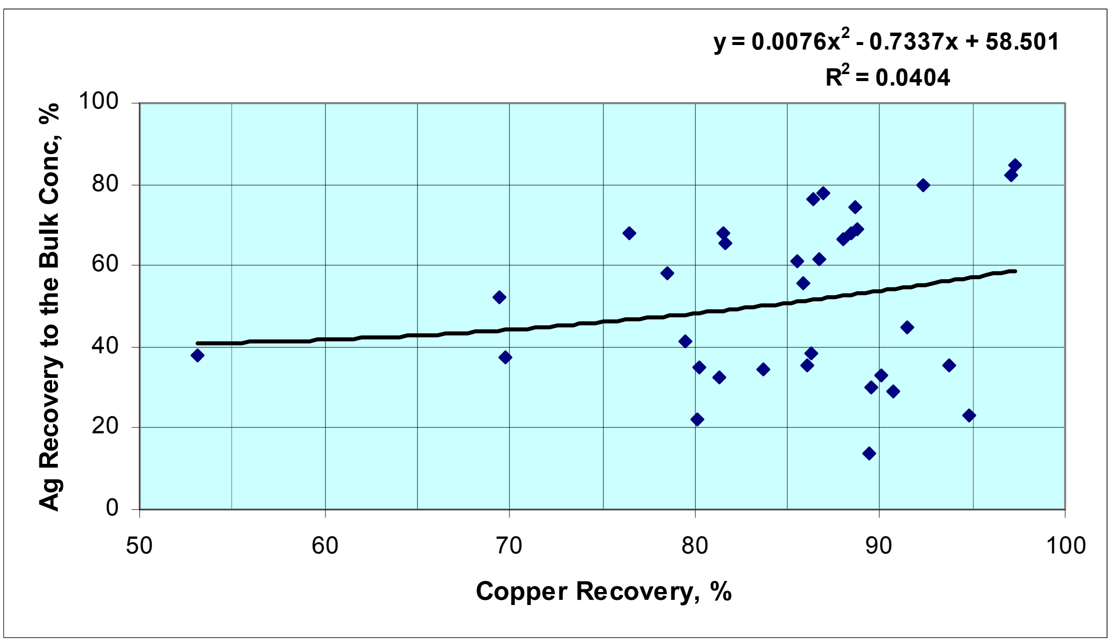


Figure 18.7 Silver Recovery as a Function of Copper Recovery in the Bulk Concentrate

18.6 Comminution Testing

Samples were prepared by G&T and sent to Hazen Research in Golden, Colorado, USA, for characterization using the JKTech procedures to determine the parameters needed to size a semi-autogenous grinding (SAG) comminution circuit.

Hazen is a licensed laboratory used by many companies to provide the parameters required to complete computer simulations of grinding circuits using the JKTech technique. Hazen had conducted these test procedures on one sample from each of the three mineral zones for the PEA. The data that was reported in a February 15, 2007 report is shown in Table 18.8.

Table 18.8
Schaft Creek JKTech Comminution Parameters by Mineralized Zone

Parameter	Liard Zone	Paramount Zone	West Breccia Zone
Specific Gravity	2.73	2.70	2.74
A	47.6	54.3	49.6
b	0.94	0.72	0.86
A x b	44.7	39.1	42.7
Ta	0.64	0.35	0.71
BW_i	21.9	20.1	19.8
RW_i	20.1	21.4	19.6
CW_i	8.93	6.71	6.31
A_i	0.198	0.380	0.186

The JKTech tests were repeated on the five additional samples as shown in Table 18.9. The samples weighed approximately 90 kg each for a total weight of approximately 450 kg. Hazen reported the values in a March 19, 2008 report.

Table 18.9
Schaft Creek JKTech Comminution Parameters for Five Samples

Parameter	Samples				
	1	2	3	4	5
Specific Gravity	2.72	2.73	2.68	2.72	2.70
A	49.64	46.02	60.23	50.17	50.01
b	0.81	0.86	0.58	0.73	0.82
A x b	40.0	39.5	35.1	36.4	41.0
Ta	0.41	0.45	0.35	0.45	0.43
BW_i	22.6	24.0	20.5	24.4	23.8
RW_i	21.6	22.5	19.6	21.8	22.2
CW_i	11.86	11.31	8.59	7.96	10.20
A_i	0.175	0.174	0.569	0.315	0.183

Mark Richardson, JKTech Support Services, Red Bluff, CA, USA, was contracted by Copper Fox Metals to complete JK SimMet computer simulations of a SAG/Ball Mill/Pebble Crusher (SABC) circuit in order to predict the optimum grinding mill sizes and circuit conditions for the comminution circuit. Mr. Richardson was also asked to model the circuit using primary grind sizes of 80% passing 100 microns and 150 microns.

The PFS is based on a primary grind of 80% passing 150 microns. This requires two parallel grinding lines for a 100,000 tpd operation. Each grinding line includes one each 11.58 m diameter x 6.10 m long (EGL) semi-autogenous (SAG) mill and two each 7.93 m diameter x 12.20 m long (EGL) ball mills. The SAG mill and ball mill motor sizes required are 17.5 MW and 12.0 MW respectively.

18.7 High Pressure Grinding Roll (HPGR) Testing

Approximately 260 kg of sample was crushed to 25 mm for HPGR testing by Polysius in Germany. Testing began in early June 2008 so the results of these tests are pending. However, it is anticipated that Schaft Creek materials should be good candidates for HPGR grinding because the material is relatively hard but not overly abrasive. Polysius will provide a report indicating the applicability of HPGR grinding for the Schaft Creek resource and, if appropriate, the approximate equipment sizes and wear rates for a 100,000 tonnes per day operation. If the results are encouraging, additional tests will be conducted on a larger sample. Samples of HPGR ground material will be returned to G&T for additional metallurgical testing to determine if there is any metallurgical benefit associated with HPGR grinding of Schaft Creek materials.

18.8 Copper-Molybdenum Separation

G&T conducted a series of molybdenum separation tests on bulk copper/molybdenum concentrates and reported these tests and test results in their January 9, 2008 report. Bulk concentrates were approximately 80% passing 20 microns and no additional regrinding was done prior to the molybdenum separation tests. Approximately 94% of the chalcopyrite, 91% of the bornite, 88% of the chalcocite and 86% of the molybdenite were liberated in the bulk concentrates. A 20 micron grind appears to be satisfactory for molybdenum separation, as well.

Approximately 35 kg of bulk concentrate was prepared by treating approximately 6.0 t of crushed drill core in a flotation pilot plant. The purpose of the pilot plant was only to prepare a sufficient quantity of bulk concentrate to conduct bench-scale molybdenum separation testing. The separation process used sodium hydrosulphide (NaHS) and nitrogen to depress the copper sulphide minerals. Standard molybdenum separation conditions were employed. The bulk concentrates were processed in rougher flotation cells and, then, four stages of cleaner flotation were run in locked-cycle tests. The 1st Cleaner Tailings were returned to the rougher feed. The 2nd, 3rd and 4th Cleaner Tailings were all returned to the feed of the 1st Cleaner stage of flotation. The average grade of the 4th Cleaner Concentrate was approximately 47% molybdenum. Molybdenum recovery averaged approximately 75% from the bulk concentrate. These tests were not optimized due to the limited quantity of bulk concentrate. Since the major diluents in the molybdenum concentrate are liberated materials, it is anticipated that molybdenum concentrates containing a grade of at least 50% molybdenum can be realized at an 85% molybdenum recovery from the bulk concentrate.

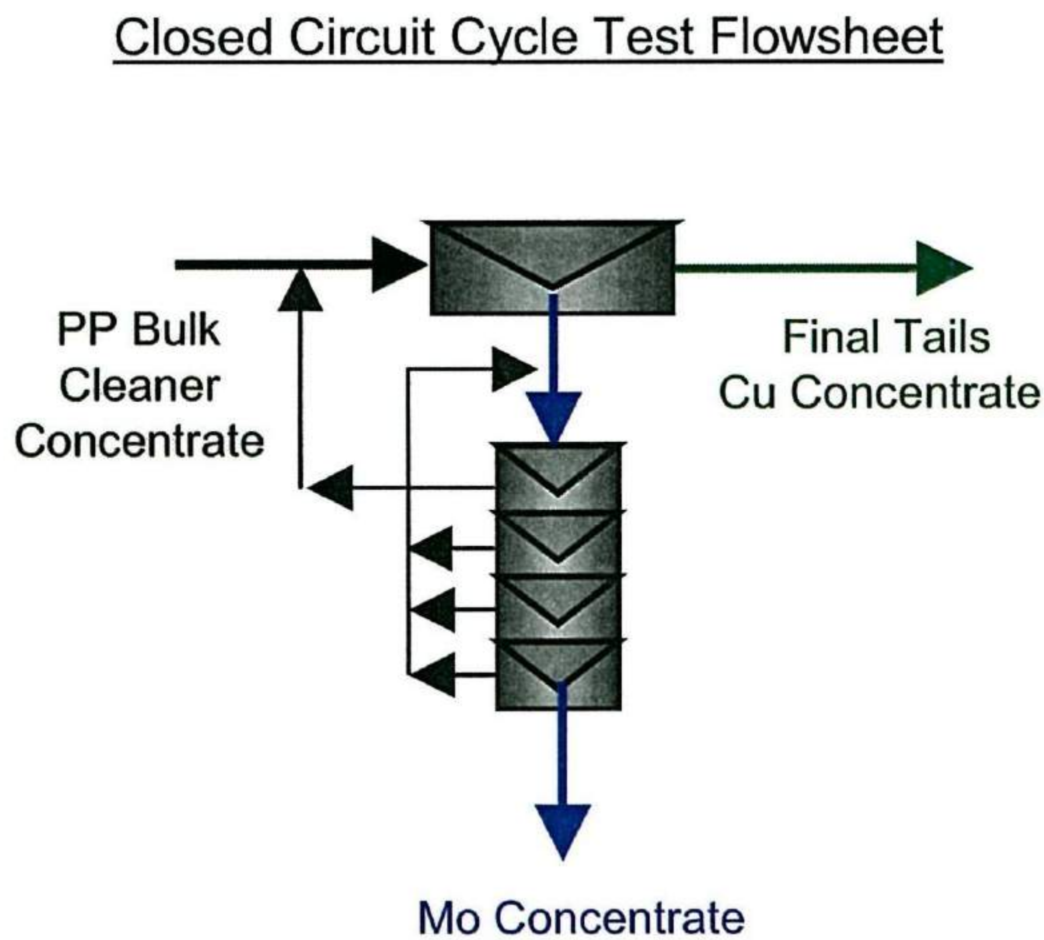


Figure 18.8 Locked-Cycle Test Flowsheet

18.9 Teck Cominco CESL Technology Tests

Teck Cominco tested two Schaft Creek concentrate samples in 2007 and 2008. Approximately 3.6 kg of Bulk Concentrate from the Process Research Associates (PRA) Vancouver laboratory was sent to Teck Cominco for preliminary CESL testing using their proprietary hydrometallurgical leaching technology. Teck Cominco reported these results in an August 24, 2007, report titled *Bench Testwork Report Schaft Creek*. They reported that "Results for copper and gold results were both extraordinarily good, with 96% - 98% copper extraction in 15 to 30 minutes (normal is 60 minutes) and 90% - 92% gold extraction for the residues from these tests." They also indicated that the bench-scale extractions are typically lower than continuous plant operation.

Approximately 5.0 kg of copper concentrate was sent to Teck Cominco from the G&T molybdenum separation tests for additional CESL testing.

Teck Cominco reported the results in a March 3, 2008, File Note with the subject as *Supplemental Testwork on Schaft Creek Concentrate*. These test results confirmed the previous studies of bulk concentrate that indicate that Schaft Creek copper concentrates are amenable to the CESL with very high copper extractions (97.0%) in very short retention times (about 30 minutes). The gold and silver extractions from the CESL sludge are high and using moderate reagent consumptions. The gold extraction from this sample was 87% compared to the 89% extraction from the bulk concentrate.

Table 18.10
Summary of 2008 G & T Locked-Cycle Test Data on Liard Zone Samples

Liard Zone Test Number	Primary Grind Size P_{80}	Flotation Feed % Wt.	Flotation Feed % Cu	Flotation Feed % Fe	Flotation Feed % Mo	Flotation Feed g/t Au	Flotation Feed g/t Ag	Total Tails % Wt.	Total Tails % Cu	Total Tails % Fe	Total Tails % Mo	Total Tails g/t Au	Total Tails g/t Ag	Regrind Size P_{80}	Bulk Concentrate Metal Grade % Wt.	Bulk Concentrate Metal Grade % Cu	Bulk Concentrate Metal Grade % Fe	Bulk Concentrate Metal Grade % Mo	Bulk Concentrate Metal Grade g/t Au	Bulk Concentrate Metal Grade g/t Ag	Bulk Concentrate Metal Distribution % Wt.	Bulk Concentrate Metal Distribution % Cu	Bulk Concentrate Metal Distribution % Fe	Bulk Concentrate Metal Distribution % Mo	Bulk Concentrate Metal Distribution % Au	Bulk Concentrate Metal Distribution % Ag
KM2136-11	109	100	0.325	3.43	0.014	2.110	0.262	99.10	0.04	3.27	0.005	0.854	0.109	21	0.90	32.69	22.20	1.076	17.583	144.000	0.90	88.31	5.67	66.58	58.88	59.89
KM2136-12	142	100	0.329	6.75	0.014	0.245	2.112	99.10	0.05	6.61	0.005	0.077	1.095	19	0.90	32.98	22.68	1.059	19.563	119.058	0.90	86.38	2.90	65.26	68.75	48.59
KM2136-13	109	100	0.325	3.72	0.012	0.251	2.183	99.10	0.03	3.55	0.003	0.063	0.889	19	0.90	31.40	21.31	0.995	20.324	140.509	0.90	89.53	5.31	75.68	75.21	59.67
KM2136-14	173	100	0.327	3.30	0.011	0.229	2.506	99.10	0.05	3.14	0.002	0.067	1.235	18	0.90	31.20	21.70	1.000	17.976	141.517	0.90	86.40	5.90	84.26	71.08	51.17
KM2136-15	142	100	0.330	3.40	0.014	0.287	1.989	99.10	0.04	3.20	0.001	0.080	0.661		0.90	31.60	20.30	1.354	22.657	150.951	0.90	87.00	5.50	90.20	72.40	69.60
KM2136-16	173	100	0.337	3.35	0.013	0.251	2.296	99.10	0.05	3.19	0.001	0.068	0.947		0.90	30.59	20.34	1.245	19.622	144.492	0.90	85.43	5.70	90.54	73.33	59.14

Table 18.11
Summary of G&T Schaft Creek Variability Tests on 2006 Core Samples - May 25 2008

Mineral Zone	G&T Test No.	Met Sample Number	Hole Number	From Meters	To Meters	Elevation At Top of Interval	Primary Grind P_{80}	Comparative Bond Wi	Regrind P_{80}	Rougher Flotation Feed %Cu	Rougher Flotation Feed %Mo	Rougher Flotation Feed g/t Au	Rougher Flotation Feed g/t Ag
Liard	KM2136-51	126,636	06CF284	51.80	54.85	905.75	152	29.78	23	0.63	0.004	0.639	4.676
Liard	KM2136-52	126,644	06CF284	76.20	79.25	881.35	147	22.62	16	0.28	0.005	0.328	2.657
Liard	KM2136-53	126,649	06CF284	134.20	137.25	823.35	137	35.36	15	0.22	0.003	0.131	2.378
Liard	KM2136-54	126,694	06CF251	39.65	42.70	911.91	132	25.29	22	0.76	0.011	1.080	6.759
Liard	KM2136-55	126,702	06CF251	64.05	67.10	887.51	114	23.63	28	0.89	0.026		
Liard	KM2136-56	126,705	06CF251	73.20	76.25	878.36	152	27.55	19	0.38	0.001	0.559	3.385
Liard	KM2136-57	126,710	06CF251	88.50	91.05	863.06	141	35.46	18	0.51	0.002	0.771	4.261
Liard	KM2136-58	126,712	06CF251	94.55	97.60	857.01	143	27.34	17	0.16	0.002	0.280	1.799
Liard	KM2136-59	126,720	06CF255	33.55	36.60	968.98	144	38.68	13	0.17	0.002	0.200	1.725
Liard	KM2136-60	126,725	06CF255	179.95	183.00	822.58	155	35.83	16	0.29	0.018	0.207	1.797
Liard	KM2136-61	126,737	06CF255	216.55	219.60	785.98	199	29.78	25	0.74	0.012	0.304	5.027
Liard	KM2136-62	126,768	06CF258	85.40	88.45	916.13	118	38.13	14	0.21	0.014	0.169	1.899
Liard	KM2136-63	126,777	06CF258	112.85	115.90	888.68	137	29.15	19	0.40	0.008	0.143	1.382
Liard	KM2136-64	126,785	06CF258	137.25	140.30	864.28	153	35.68	19	0.50	0.013	0.598	2.958
Liard	KM2136-65	126,787	06CF258	143.35	146.40	858.18	141	35.46	19	0.80	0.008	0.356	6.003
Liard	KM2136-66	126,799	06CF258	179.95	183.00	821.58	158	34.29	36	1.02	0.016	0.715	6.116
Liard	KM2136-67	126,865	06CF258	109.80	112.85	891.73	143	21.93	20	0.21	0.027	0.299	0.986
Liard	KM2136-68	126,876	06CF285	143.25	146.60	885.36	153	29.15	20	0.40	0.032	0.493	3.499
Liard	KM2136-69	126,881	06CF285	158.60	161.65	870.01	143	31.97	16	0.61	0.021	0.751	5.065
Liard	KM2136-70	126,882	06CF285	161.65	164.70	866.96	162	28.99	21	0.55	0.059	0.963	5.203
Liard	KM2136-71	126,927	06CF256	207.40	210.45	829.95	136	25.78	34	1.07	0.032	0.804	5.510
Paramount	KM2136-72	127,121	06CF287	70.15	73.20	890.19	141	28.99	21	0.17	0.006	0.058	1.219
Paramount	KM2136-73	127,126	06CF287	85.40	88.45	874.94	135	26.10	16	0.29	0.005	0.044	0.956
Paramount	KM2136-74	127,145	06CF287	143.35	146.40	816.99	147	22.79	29	1.24	0.018	0.096	6.310
Paramount	KM2136-75	127,151	06CF287	161.65	164.70	798.69	135	22.96	28	0.98	0.013	0.111	10.510
Liard	KM2136-76	127,197	06CF289	112.85	115.90	845.55	162	22.10	42	1.15	0.005	0.711	7.281
Liard	KM2136-77	127,321	06CF262	125.05	128.10	999.26	157	21.24	21	0.29	0.003	0.069	1.418
Liard	KM2136-78	127,337	06CF262	173.85	176.90	950.46	158	34.29	19	0.37	0.002	0.160	1.459
Liard	KM2136-79	127,348	06CF268	64.05	67.10	1058.14	142	44.54	13	0.17	0.011	0.186	1.615
Liard	KM2136-80	127,352	06CF268	76.25	79.30	1045.94	142	40.27	20	0.37	0.016	0.175	1.394
Liard	KM2136-81	127,376	06CF268	149.45	152.50	972.74	151	20.01	44	0.81	0.018	0.351	2.836
Liard	KM2136-82	127,377	06CF268	152.50	155.55	969.69	133	25.78	42	0.95	0.018	0.364	2.937
Liard	KM2136-83	127,378	06CF268	155.55	158.60	966.64	104	19.11	28	0.68	0.050	0.314	3.394
Liard	KM2136-84	127,573	06CF265	216.55	219.60	835.23	146	28.35	17	0.47	0.112	0.467	3.531

G&T Test No.	Rougher Concentrate Metal Grade Wt. %	Rougher Concentrate Metal Grade %Cu	Rougher Concentrate Metal Grade %Mo	Rougher Concentrate Metal Grade g/t Au	Rougher Concentrate Metal Grade g/t Ag	Rougher Tailing Metal Grade Wt. %	Rougher Tailing Metal Grade %Cu	Rougher Tailing Metal Grade %Mo	Rougher Tailing Metal Grade g/t Au	Rougher Tailing Metal Grade g/t Ag
KM2136-51	9.62	5.40	0.039	5.232	37.33	90.38	0.12	0.001	0.150	1.20
KM2136-52	8.10	2.18	0.039	2.184	16.93	91.90	0.11	0.002	0.165	1.40
KM2136-53	5.51	3.50	0.041	2.139	31.14	94.49	0.03	0.001	0.014	7.00
KM2136-54	6.37	10.89	0.140	13.992	87.04	93.63	0.07	0.002	0.202	1.30
KM2136-55	8.00	10.55	0.300			92.00	0.05	0.002		
KM2136-56	3.71	8.68	0.024	11.562	60.10	96.29	0.06	0.001	0.135	1.20
KM2136-57	3.84	11.49	0.021	14.677	83.33	96.20	0.07	0.001	0.215	1.10
KM2136-58	2.47	5.43	0.043	7.647	45.18	97.53	0.03	0.001	0.093	0.70
KM2136-59	2.85	4.81	0.046	4.359	29.87	97.15	0.03	0.001	0.078	0.90
KM2136-60	3.92	6.52	0.404	3.774	21.34	96.08	0.03	0.002	0.062	1.00
KM2136-61	6.39	10.93	0.172	3.991	25.95	93.61	0.05	0.001	0.053	3.60
KM2136-62	4.17	4.37	0.305	2.361	24.89	95.83	0.02	0.001	0.074	0.90
KM2136-63	7.39	5.04	0.088	1.400	7.42	92.61	0.03	0.002	0.043	0.90
KM2136-64	7.41	6.23	0.155	3.192	28.66	92.59	0.04	0.002	0.066	0.90
KM2136-65	6.79	11.15	0.099	4.247	71.95	93.21	0.05	0.001	0.072	1.20
KM2136-66	9.97	9.38	0.136	5.734	52.30	90.03	0.10	0.003	0.159	1.00
KM2136-67	5.80	3.57	0.371	3.903	10.51	94.20	0.01	0.006	0.077	0.40
KM2136-68	9.19	4.12	0.284	4.510	26.21	90.81	0.02	0.006	0.087	1.20
KM2136-69	6.10	9.43	0.279	10.531	61.47	93.90	0.04	0.004	0.116	1.40
KM2136-70	10.78	4.77	0.469	7.330	36.70	89.22	0.04	0.010	0.194	1.40
KM2136-71	12.37	8.02	0.211	3.959	33.90	87.63	0.09	0.007	0.359	1.50
KM2136-72	9.77	1.62	0.046	0.453	5.09	90.23	0.01	0.002	0.015	0.80
KM2136-73	6.15	4.60	0.036	0.480	4.87	93.85	0.01	0.003	0.015	0.70
KM2136-74	9.22	13.31	0.186	0.886	60.57	90.78	0.02	0.002	0.015	0.80
KM2136-75	11.68	8.29	0.099	0.726	82.41	88.32	0.01	0.002	0.030	1.00
KM2136-76	10.10	11.07	0.038	5.818	63.19	89.90	0.04	0.001	0.137	1.00
KM2136-77	4.37	6.35	0.046	0.907	6.18	95.63	0.02	0.001	0.031	1.20
KM2136-78	9.04	3.95	0.010	1.328	8.08	90.96	0.01	0.001	0.04	0.80
KM2136-79	4.47	3.35	0.217	3.204	16.90	95.53	0.02	0.001	0.05	0.90
KM2136-80	8.49	4.03	0.168	1.563	7.79	91.51	0.03	0.002	0.05	0.80
KM2136-81	10.79	6.70	0.138	2.257	12.23	89.21	0.10	0.003	0.12	1.70
KM2136-82	11.57	7.45	0.137	2.031	12.40	88.43	0.10	0.002	0.15	1.70
KM2136-83	11.52	5.36	0.339	1.837	14.11	88.48	0.07	0.012	0.12	2.00
KM2136-84	6.64	6.45	1.613	5.715	36.30	93.36	0.05	0.005	0.09	1.20

G&T Test No.	Rougher Concentrate Metal Distributions %Cu	Rougher Concentrate Metal Distributions %Mo	Rougher Concentrate Metal Distributions g/t Au	Rougher Concentrate Metal Distributions g/t Ag	Bulk Concentrate Metal Grade Wt. %	Bulk Concentrate Metal Grade %Cu	Bulk Concentrate Metal Grade %Mo	Bulk Concentrate Metal Grade g/t Au	Bulk Concentrate Metal Grade g/t Ag	Bulk Concentrate Metal Distributions %Cu	Bulk Concentrate Metal Distributions %Mo	Bulk Concentrate Metal Distributions g/t Au	Bulk Concentrate Metal Distributions g/t Ag
KM2136-51	82.73	89.24	78.82	76.81	1.19	40.20	0.295	36.090	266.00	76.47	83.92	67.50	67.94
KM2136-52	63.62	63.10	53.89	51.57	0.30	46.50	0.690	36.341	310.80	53.08	44.66	35.71	37.71
KM2136-53	87.18	70.76	89.69	72.19	0.53	34.20	0.398	19.778	296.00	81.66	65.00	79.39	65.69
KM2136-54	91.82	82.65	82.49	81.99	1.22	53.50	0.700	66.364	422.00	86.38	79.09	74.93	76.13
KM2136-55	94.86	92.92			1.60	49.60	1.450			89.67	90.27		
KM2136-56	85.23	65.14	76.80	65.87	0.57	51.90	0.138	69.150	343.00	78.47	57.13	70.77	57.92
KM2136-57	87.44	46.11	73.19	75.18	0.75	54.80	0.095	66.060	385.00	81.57	40.05	64.46	67.97
KM2136-58	81.62	52.03	67.59	62.05	0.22	51.20	0.403	75.180	420.00	69.48	44.24	60.04	52.12
KM2136-59	82.93	57.26	62.09	49.30	0.27	42.60	0.364	38.28	240	69.76	43.27	51.74	37.66
KM2136-60	89.55	89.18	71.28	46.53	0.62	37.50	2.310	21.440	95.00	81.34	80.47	63.93	32.70
KM2136-61	93.95	92.15	83.70	32.96	2.25	30.00	0.475	10.310	65.60	90.68	89.49	76.04	29.30
KM2136-62	88.79	93.00	58.10	54.59	0.33	49.50	3.680	24.080	240.00	79.43	87.80	46.47	41.27
KM2136-63	94.15	77.77	72.20	39.68	1.21	29.20	0.459	6.854	34.00	89.57	66.83	58.05	29.85
KM2136-64	93.27	86.15	79.48	71.83	1.18	36.30	0.810	17.140	154.00	86.76	71.70	68.15	61.63
KM2136-65	94.20	87.87	81.02	81.37	1.85	38.50	0.315	14.200	242.00	88.66	75.87	73.85	74.60
KM2136-66	91.55	83.39	79.98	85.28	2.88	30.90	0.378	17.957	165.00	86.96	66.90	72.27	77.62
KM2136-67	97.34	79.19	75.80	61.79	0.72	27.20	2.510	26.779	61.90	91.51	66.20	64.22	44.96
KM2136-68	95.43	82.73	84.04	68.86	0.92	39.90	1.960	37.795	211.00	85.89	57.40	70.82	55.73
KM2136-69	94.02	81.93	85.54	74.05	1.29	41.70	1.150	45.620	261.00	87.99	71.46	78.44	66.56
KM2136-70	93.20	84.99	82.06	75.99	1.22	39.90	3.690	59.538	289.00	88.48	75.94	75.65	67.93
KM2136-71	92.79	80.98	60.92	76.14	3.09	30.80	0.780	14.347	123.00	88.81	74.62	55.04	68.89
KM2136-72	93.12	76.86	76.67	40.81	0.53	25.60	0.531	6.849	51.10	80.13	48.11	63.26	22.35
KM2136-73	97.73	48.74	67.61	31.29	0.83	32.90	0.171	2.168	26.80	94.81	31.11	41.37	23.35
KM2136-74	98.69	92.64	85.41	88.49	3.70	32.60	0.435	1.993	140.00	97.07	87.12	77.17	82.18
KM2136-75	98.83	86.77	76.19	91.60	3.41	28.00	0.321	2.192	261.00	97.34	81.94	67.06	84.64
KM2136-76	97.04	81.21	82.65	87.65	3.00	35.50	0.105	17.804	194.00	92.33	65.80	75.07	79.87
KM2136-77	94.46	67.79	57.37	19.06	0.86	30.70	0.190	3.939	22.80	89.39	54.71	48.75	13.75
KM2136-78	97.27	49.42	74.83	50.11	1.2	28.7	0.064	8.888	42.9	93.73	42.64	66.38	35.24
KM2136-79	87.68	91.01	76.89	46.74	0.43	32	2.030	30.096	133	80.20	81.69	69.14	35.21
KM2136-80	93.74	88.64	75.87	47.48	1.34	26.5	1.040	9.32	37.2	90.07	80.20	66.18	33.15
KM2136-81	88.72	84.73	69.34	46.52	2.89	23.6	0.469	6.775	34	83.71	77.29	55.73	34.64
KM2136-82	90.77	89.99	64.60	48.81	3.51	23.3	0.428	5.426	32.1	86.26	85.11	52.42	38.39
KM2136-83	91.12	78.60	67.33	47.86	2.18	26.7	0.900	7.88	55	86.02	39.59	54.73	35.36
KM2136-84	91.06	95.82	81.23	68.27	1.07	37.7	9.400	32.518	2023	85.49	89.65	74.19	61.28

18.10 Process Introduction

The process design used to support this PFS has been completed by Samuel Engineering, Inc. (SE).

For the Preliminary Feasibility Study (PFS) the Copper Fox Metals, Inc. (Copper Fox) Schaft Creek concentrator has an annual throughput of 36,000,000 t. This has increased from 23,400,000 tpa in the Preliminary Economic Assessment (PEA). Copper Fox will construct the concentrator on site. It will include an SABC comminution circuit followed by a bulk flotation circuit, a molybdenum separation circuit and a copper circuit. The copper circuit has a thickener, filters and a concentrate stockpile. The molybdenum circuit includes filtration, drying and bagging equipment. Tailings thickeners, tailings storage facility and water reclaim are part of the tailings management system. The processing circuit will have a design capacity of 108,700 tpd and a nominal capacity of 100,000 tpd.

18.11 Process Flowsheet Development

The process flowsheets used as the design basis in the Preliminary Feasibility Study are based on the results of on-going metallurgical testwork.

18.12 Site Layout Considerations

The site for the processing plant and ancillary facilities has been selected based on optimization of the following key considerations:

- Gravity deposition of tailings;
- Consideration of environmental and social impacts;
- Proximity to the mine to minimize ore transportation costs.

The Overall Site Plan drawing that supports the process design for the PFS is available in Section 26 (Illustrations) of this report.

18.13 Process Description

A block flow diagram is provided as Figure 18.9 for reference purposes.

18.13.1 Design Criteria

The process design criteria and equipment sizing are based on a nominal feed rate of 100,000 tpd that will operate with a mechanical availability of 92%. This results in a design feed rate of 108,700 tpd. The feed grade of the material to the processing plant is based on the mine production schedule, as discussed in Section 25 (Additional Requirements for Technical Reports on Development Properties and Production Properties) of this report.

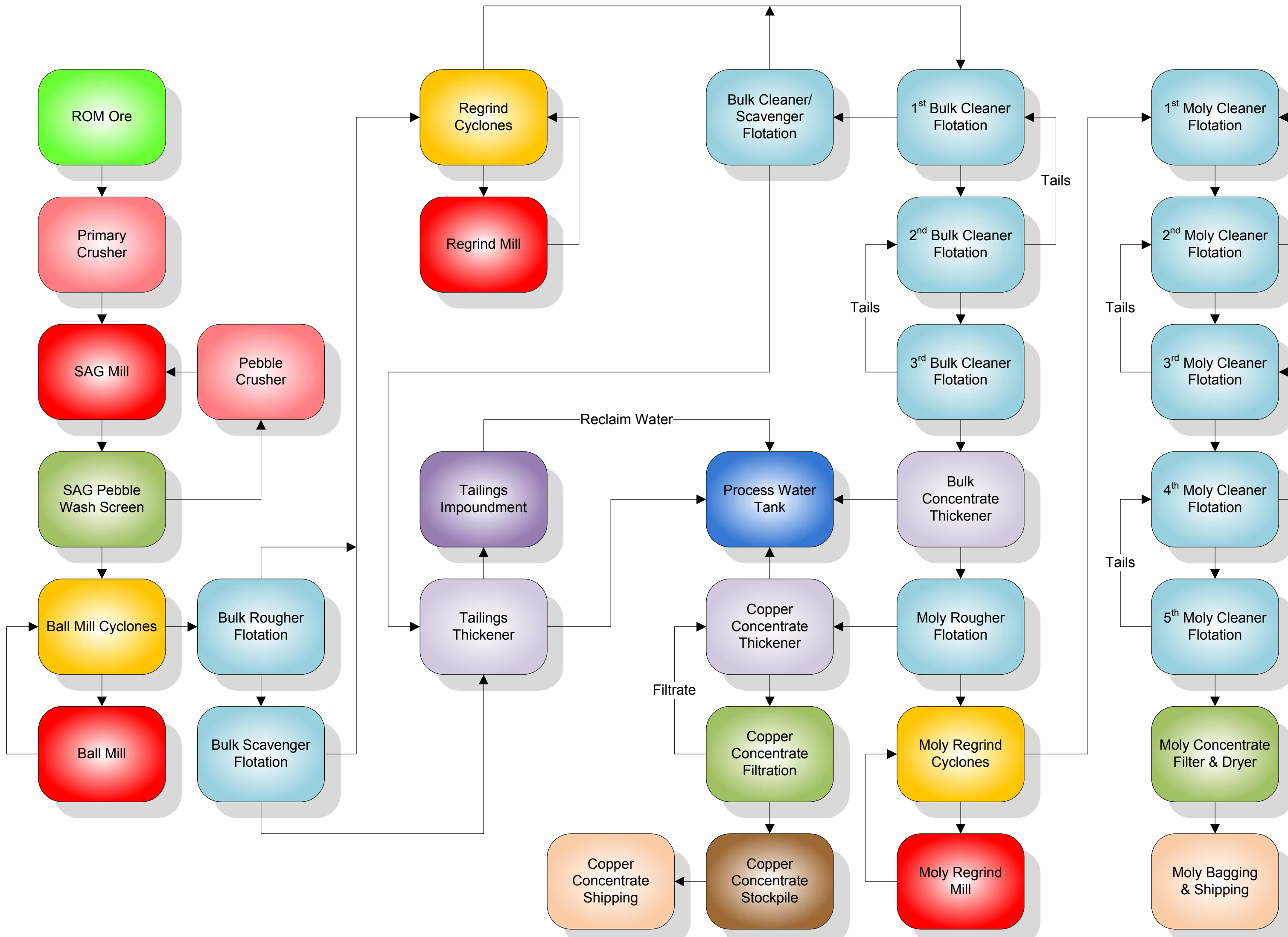


Figure 18.9 Block Flow Diagram

The flotation circuits and other processing circuits were simulated using METSIM. The results of the simulation provided the mass and water balances, which, in turn, were used for equipment sizing. The mass and water balances are shown on the process flowsheets. Table 18.12 shown here is an abridged version of the Process Design Criteria.

Table 18.12
Abridged Process Design Criteria June 26, 2008

	Units	Design	Source
General Site Information			
Location			
Latitude	degrees	N 57° 21'15″	Copper Fox Metals, Inc.
Longitude	degrees	W 131° 0′58″	Copper Fox Metals, Inc.
Elevation			
Air Strip	masl	866.00	Samuel Engineering
Plant	masl	1,230.00	Samuel Engineering
Pit Bottom	masl	600.00	Moose Mountain Technical Services
Ambient Air Temperature			
Winter Minimum	°C	-30.00	
Summer Maximum	°C	28.00	
Average Annual Precipitation	mm/y	1,039.00	Knight Piésold
Average Annual Evaporation - Pond Area	mm/y	450.00	Knight Piésold
Maximum Wind Velocity	km/h		
General Project Information			
Reported Resource	tonnes (t)	1,393,282,000	Associated Geosciences Ltd.
Cutoff Grade Used	CuEq (%)	0.20	Associated Geosciences Ltd.
Estimated Mineable Resources			
Starter Pit (5 Year)	tonnes (t)	216,060,000	Moose Mountain Technical Services
Life of Mine (includes subgrade to waste)	tonnes (t)	816,707,000	Moose Mountain Technical Services
Operating Schedule			
Hours per Day	h	24	Samuel Engineering
Days per Year	d	360	Samuel Engineering
Hours per Year	h	8,640	Samuel Engineering
Plant Capacity (design based on 92% grinding circuit operating time)	dmtpd	108,696	Samuel Engineering
Plant Capacity (design based on 92% grinding circuit operating time)	dmtph	4,529	Samuel Engineering
Annual Ore Processed per Year	t	36,000,000	Moose Mountain Technical Services
Mineral Reserves to Mill	t	812,231,000	Moose Mountain Technical Services
Estimated Project Life @ 100,000 tpd	y	22.56	Moose Mountain Technical Services
Life of Mine Plant Head Grade Estimates			
Estimated Copper Grade	%	0.301	Moose Mountain Technical Services
Estimated Molybdenum Grade	%	0.020	Moose Mountain Technical Services
Estimated Gold Grade	g/t	0.212	Moose Mountain Technical Services
Estimated Silver Grade	g/t	1.76	Moose Mountain Technical Services

Table 18.12 Abridged Process Design Criteria June 26, 2008			
First 5 Years Plant Head Grade Estimates			
Estimated Copper Grade	%	0.337	Moose Mountain Technical Services
Estimated Molybdenum Grade	%	0.017	Moose Mountain Technical Services
Estimated Gold Grade	g/t	0.249	Moose Mountain Technical Services
Estimated Silver Grade	g/t	1.65	Moose Mountain Technical Services
Plant Design (First 5 Years) Head Grade Estimates			
Estimated Copper Grade	%	0.337	Moose Mountain Technical Services
Estimated Molybdenum Grade	%	0.017	Moose Mountain Technical Services
Estimated Gold Grade	g/t	0.249	Moose Mountain Technical Services
Estimated Silver Grade	g/t	1.649	Moose Mountain Technical Services
Design ROM Ore Dry Solids Sp Gr	g/cc	2.69	Copper Fox Metals, Inc.
Design ROM Ore Moisture (for material handling)	%	3.00	Copper Fox Metals, Inc.
First Five-year Average Copper Concentrate Production			
Copper Recovery to Copper Concentrate	%	89.21	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Grade in Copper Concentrate	%	33.52	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Recovery to Copper Concentrate	%	10.82	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Grade in Copper Concentrate	% Mo	0.21	Hyyppa Engineering, LLC/G&T Metallurgical
Gold Recovery to Copper Concentrate	%	80.71	Hyyppa Engineering, LLC/G&T Metallurgical
Gold Grade in Copper Concentrate	g/t	22.40	Hyyppa Engineering, LLC/G&T Metallurgical
Silver Recovery to Copper Concentrate	%	72.00	Hyyppa Engineering, LLC/G&T Metallurgical
Silver Grade in Copper Concentrate	g/t	132.40	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Concentrate Production	dmtph	40.62	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Concentrate Production	dmtpy	322,848	Hyyppa Engineering, LLC/G&T Metallurgical
First Five-year Average Moly Concentrate Production			
Moly Recovery to Moly Concentrate	%	68.61	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Grade in Moly Concentrate	% Mo	50.00	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Recovery to Moly Concentrate	%	0.07	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Grade in Moly Concentrate	%	1.06	Hyyppa Engineering, LLC/G&T Metallurgical
Rhenium Grade in Moly Concentrate	ppm	368.00	Hyyppa Engineering, LLC/G&T

Table 18.12 Abridged Process Design Criteria June 26, 2008			
			Metallurgical
Moly Concentrate Production	dmtph	1.06	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Concentrate Production	dmtpy	8,397.91	Hyyppa Engineering, LLC/G&T Metallurgical
Life of Mine Average Copper Concentrate Production			
Copper Recovery to Copper Concentrate	%	88.36	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Grade in Copper Concentrate	%	33.85	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Recovery to Copper Concentrate	%	11.24	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Grade in Copper Concentrate	% Mo	0.29	Hyyppa Engineering, LLC/G&T Metallurgical
Gold Recovery to Copper Concentrate	%	81.29	Hyyppa Engineering, LLC/G&T Metallurgical
Gold Grade in Copper Concentrate	g/t	21.90	Hyyppa Engineering, LLC/G&T Metallurgical
Silver Recovery to Copper Concentrate	%	70.69	Hyyppa Engineering, LLC/G&T Metallurgical
Silver Grade in Copper Concentrate	g/t	158.30	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Concentrate Production	dmtph	35.59	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Concentrate Production	dmtpy	282,881.89	Hyyppa Engineering, LLC/G&T Metallurgical
Life of Mine Average Moly Concentrate Production			
Moly Recovery to Moly Concentrate	%	71.29	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Grade in Moly Concentrate	% Mo	50.00	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Recovery to Moly Concentrate	%	0.10	Hyyppa Engineering, LLC/G&T Metallurgical
Copper Grade in Moly Concentrate	%	1.06	Hyyppa Engineering, LLC/G&T Metallurgical
Rhenium Grade in Moly Concentrate	ppm	368.00	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Concentrate Production	dmtph	1.29	Hyyppa Engineering, LLC/G&T Metallurgical
Moly Concentrate Production	dmtpy	10,261.90	Hyyppa Engineering, LLC/G&T Metallurgical

18.13.2 Crushing and Coarse Ore Stockpile

Run-of-Mine ore will be transported from the open pit mine using 345 t haul trucks. It will be dumped into a 720 t live-capacity feed pocket, which feeds a 1,524 by 2,794 mm (60 by 110 in) primary gyratory crusher powered by a 1.0 MW drive motor. The primary crusher has a design feed size (F_{80}) of 600 mm (24 in) and a product size (P_{80}) of 150 mm (6 in).

The crushed ore will be conveyed by the crushed ore stacking belt conveyor to a covered coarse ore stockpile, which will have a minimum live capacity of 100,000 t and a total capacity of 290,000 t.

18.13.3 Grinding

The comminution circuit for the Schaft Creek concentrator will be conventional semi-autogenous (SAG) primary grinding mills followed by secondary ball mills with pebble crushers to break down critical-sized material that discharges from the SAG mill. This circuit is commonly referred to as an “SABC” comminution or grinding circuit.

Ore from the stockpile is reclaimed onto two semi-autogenous grinding (SAG) mill feed belt conveyors by six of eight stockpile reclaim belt feeders below the stockpile and discharged into two SAG mill feed belt discharge chutes. Dual grinding lines, each comprised of one 11.58 m diametre by 5.79 m long SAG mill (38 ft x 19 ft) with a 17.5 MW (23,500 hp) motor, one 750 kW (1,000 hp) MP1000 pebble cone crusher and two 7.32 m diametre by 12.19 m long ball mills (24 ft x 40 ft) with 12.0 MW (16,000 hp) motors, operating in parallel in order to provide the grinding capacity for the comminution circuit. The SAG mill product (P_{80} of 6.5 mm) discharges through the trommel screen. The trommel screen undersize (P_{80} of 2.0 mm) will flow through the SAG mill discharge launder that is equipped with a chute magnet to remove tramp metal. It will flow by gravity to a ball mill cyclone feed sump along with the ball mill discharge that correlates with the specific grinding circuit sump. Feed to each of the ball mill cyclone clusters will be provided by one of two ball mill cyclone feed pumps (one operating and one in standby mode). The four ball mill cyclone clusters will each be comprised of thirteen 660 mm (24 in) cyclones (11 operating and two spares) per cluster. They will operate in closed circuit with the respective grinding circuit. The cyclone overflow, (P_{80} of 150 microns) is the product from the grinding circuit and the feed to the flotation circuit. The cyclone underflow returns by gravity to the respective ball mill.

Critical-size pebbles will be removed by the SAG mill trommel screen. They will be washed on the SAG mill pebble wash screen, the two pebble streams will be combined 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 also operating in parallel.

Crushed product from the pebble crushers is then recombined for a uniform product and transferred back to the SAG mill feed belt conveyor by way of the pebble crusher transfer belt conveyor from the pebble crusher belt conveyor via another split surge bin thus ensuring even redistribution back to the two SAG mills.

18.13.4 Flotation

The Schaft Creek concentrator uses a conventional copper/molybdenum flotation circuit. A rougher/scavenger flotation circuit is used to produce bulk rougher flotation concentrate that is reground to a finer size and cleaned in a three-stage cleaner flotation circuit. The cleaned bulk flotation concentrate is thickened and pumped to the copper/molybdenum separation circuit. The molybdenum is recovered in the rougher flotation circuit. The tailings from the copper/molybdenum separation circuit are the copper concentrate, which is the primary product of the concentrator.

The molybdenum concentrate is cleaned in a five-stage cleaner flotation circuit. The tailings from the first molybdenum cleaner flotation circuit are sent back to the rougher flotation circuit, while each subsequent stage of cleaner tailings are returned to the stage preceding that cleaner flotation circuit for incremental recovery of additional molybdenum concentrate.

18.13.4.1 *Bulk Rougher/Scavenger Flotation Circuit*

The cyclone overflow from the grinding lines will combine for redistribution to the four flotation lines at the rougher flotation feed distributor. Feed from the distributor will report to each of four Rougher Flotation Cell banks ("A", "B", "C" and "D"). Each Rougher Flotation Cell bank will consist of six 160 m^3 (5,650 ft^3) tank cells which will provide 19 minutes of residence time. The total rougher concentrate from processing 100,000 tpd of ore will be transported to the regrind cyclone feed pump box. Tailings from the four 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 slurry to each of the four scavenger flotation cells consisting of three 160 m^3 tank cells per line. Flotation concentrate produced in 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 be transported through the tails disposal launder and on to the tailings thickener.

18.13.4.2 *Bulk Cleaner Flotation Circuit*

The product from the bulk rougher/scavenger flotation circuit will have a nominal size that has an F_{80} of 150 microns. This slurry will be pumped from the concentrate regrind cyclone feed pump box via two of four slurry pumps to two separate regrind cyclone clusters (cyclone banks "A" and "B"). The cyclone overflow has a nominal product size (P_{80}) of 20 microns.

The 1st cleaner flotation circuit will receive the regrind cyclone overflow. The 1st cleaner flotation circuit consists of a feed distribution box and four 5.49 m (18 ft) diametre by 10.97 m (36 ft) tall column-type flotation cells operating in parallel and having a volume, including an allowance for the froth factor, of 207.53 m^3 (7,329 ft^3) each. The 1st cleaner tails are transferred to the cleaner/scavenger circuit consisting of three 160 m^3 (5,650 ft^3) tank cells to reclaim missed concentrate which is transferred via the recleaner feed pump back to the 1st cleaner feed distributor. The cleaner/scavenger tails are combined with the tails from the bulk scavenger flotation circuit in the tails disposal launder that are destined for the tailings thickener.

The 2nd cleaner flotation circuit will consist of the 2nd cleaner flotation feed distributor and two 5.49 m (18 ft) diametre by 10.97 m (36 ft) tall column-type flotation cells operating in parallel and having a volume, including an allowance for the froth factor, of 207.53 m^3 (7,329 ft^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 4.88 m (16 ft) diametre by 10.97 m (36 ft) tall column-type flotation cells operating in parallel and having a volume, including an allowance for the froth factor, of 163.97 m^3 (5,790 ft^3) each.

Tails from the 3rd cleaner circuit report to the 3rd cleaner tails pump box and are pumped back to the 2nd cleaner flotation feed distributor by the 3rd cleaner tails recirculating pump. Concentrate from the 3rd cleaner circuit reports to the bulk concentrate thickener.

18.13.4.3 *Molybdenum Separation Flotation Circuit*

After dewatering in a 30 m (98.43 ft) diametre thickener from a slurry density of 25% solids to a slurry density of 60% solids, the slurry is pumped via one of two bulk concentrate thickener underflow pumps to the conditioning tank serving the bulk molybdenum rougher flotation circuit. The rougher flotation circuit will consist of six standard (enclosed) 10 m^3 (353 ft^3) rougher flotation cells. The tails from the molybdenum rougher flotation circuit are the copper concentrate, which 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 regrind circuit and ultimately the molybdenum cleaner circuit.

18.13.4.4 *Molybdenum Cleaner Flotation Circuit*

The molybdenum cleaner flotation circuit consists of the molybdenum cleaner flotation feed distributor which is divided into five sections by baffles, the first of which will direct initial feed to the 1st molybdenum cleaner flotation column. The molybdenum flotation cells and distributor will be covered or enclosed to minimize NaSH oxidation by contact with ambient air.

The 1st molybdenum cleaner flotation cell will be a 2.44 m (8 ft) diametre by 4.88 m (16 ft) tall column-type flotation cell having a volume, including an allowance for the froth factor, of 18.22 m^3 (643.4 ft^3). The 1st molybdenum cleaner flotation concentrate reports to the second section of the molybdenum cleaner flotation feed distributor via the 1st cleaner concentrate pump. Tails from the 1st molybdenum cleaner reports to the molybdenum rougher flotation circuit along with the thickened concentrate from the bulk concentrate thickener via the 1st cleaner tailings pump.

Feed to the 2nd molybdenum cleaner flotation cell is from the second section of the molybdenum cleaner flotation feed distributor and will consist predominantly of concentrate from the 1st molybdenum cleaner flotation and recycled tailings from the 3rd molybdenum cleaner cell. The 2nd molybdenum cleaner flotation cell will be a 1.83 m (6 ft) diametre by 4.88 m (16 ft) tall column-type flotation cell having a volume, including an allowance for the froth factor, of 10.25 m^3 (362.0 ft^3).

Tails from the 2nd molybdenum cleaner will be pumped to the first section of the molybdenum cleaner flotation feed distributor by the 2nd cleaner tailings pump. Concentrate from the 2nd molybdenum cleaner flotation cell will be pumped to the third section of the molybdenum cleaner flotation feed distributor by the 2nd cleaner concentrate pump.

Feed to the 3rd molybdenum cleaner flotation cell is from the third section of the molybdenum cleaner flotation feed distributor and will consist predominantly of concentrate from the 2nd molybdenum cleaner flotation and recycled tailings from the 4th molybdenum cleaner cell. The 3rd molybdenum cleaner flotation cell will be a 1.83 m (6 ft) diametre by 4.88 m (16 ft) tall column-type flotation cell having a volume, including an allowance for the froth factor, of 10.25 m^3 (362.0 ft^3).

Tails from the 3rd molybdenum cleaner will be pumped to the second section of the molybdenum cleaner flotation feed distributor by the 3rd cleaner tailings pump. Concentrate from the 3rd molybdenum cleaner will be pumped to the fourth section of the molybdenum cleaner flotation feed distributor by the 3rd cleaner concentrate pump.

Feed to the 4th molybdenum cleaner flotation cell is from the fourth section of the molybdenum cleaner flotation feed distributor and will consist predominantly of concentrate from the 3rd molybdenum cleaner flotation and tails from the 5th molybdenum cleaner cell. The 4th molybdenum cleaner flotation cell will be a 1.83 m (6 ft) diametre by 4.88 m (16 ft) tall column-type flotation cell having a volume, including an allowance for the froth factor, of 10.25 m^3 (362.0 ft^3). Tails from the 4th molybdenum cleaner will be pumped to the third section of the molybdenum cleaner flotation feed distributor by the 4th cleaner tailings pump. Concentrate from the 4th molybdenum cleaner will be pumped to the fifth section of the molybdenum cleaner flotation feed distributor by the 4th cleaner concentrate pump.

Feed to the 5th molybdenum cleaner flotation cell is from the fifth section of the molybdenum cleaner flotation feed distributor and will consist predominantly of concentrate from the 4th molybdenum cleaner flotation. The 5th molybdenum cleaner flotation cell will be a 1.83 m (6 ft) diametre by 4.88 m (16 ft) tall column-type flotation cell having a volume (with froth factor) of 10.25 m^3 (362.0 ft^3). Tails from the 5th molybdenum cleaner will be pumped to the fourth section of the molybdenum cleaner flotation feed distributor by the 5th cleaner tailings pump. Concentrate from the 5th molybdenum cleaner flotation cell reports to the molybdenum concentrate thickener via the 5th cleaner concentrate pump.

18.13.4.5 *Final Tailings and Tailings Storage Facility*

The final tailings consist predominately of the bulk scavenger tailings and the bulk cleaner/scavenger tailings. They will be gravity fed to the Tailings Storage Facility (TSF) located in the Skeeter Lake Valley, north of the open pit. The TSF has been designed to permanently store 812 M t of tailings (570 M m^3 at an overall average dry density of 1.4 t/m^3). Tailings from the mill will be discharged to the TSF at an average slurry solids content of 55% at a throughput of 98,900 t/d, or approximately 33 M tpa. The final tailings settled solids content is anticipated to be 80%.

Additional details about the TSF can be found in Section 25 (Additional Requirements for Technical Reports on Development Properties and Production Properties) of this report.

Embankments

There will ultimately be three embankments to contain the 1 G t of tailings. These will be constructed in stages throughout the life of the project using a combination of Non-Potentially Acid Generating (NPAG) waste rock materials, local borrow and cycloned tailings. The starter facility, that is designed to accommodate tailings production for a period of two years, will consist of a single 60 m high embankment constructed at the northern end of the facility with fill that is sourced from local borrow pits. This embankment will be raised progressively throughout the life of the facility using cycloned tailings, to an ultimate height of 125 m. Later in the life of the facility, the West and South embankments will be constructed and raised progressively to ultimate heights of 60 m and 45 m, respectively.

The West embankment will be constructed in a similar manner to the North embankment, using local borrow and cycloned sand, while the South embankment, being relatively close to the pit, will be constructed largely from NPAG mine waste.

Tailings Distribution

The discharge of tailings into the TSF will initially be from a series of large-diametre valved off-takes located along the North embankment. Later in the operational life of the facility, tailings distribution will also take place from the South and West embankments and from the western side of the facility. The coarse fraction of the tailings is expected to settle rapidly and will accumulate closer to the discharge points, forming a gently sloping beach while the finer tailings particles will travel further and settle remote from the discharge point. The proposed deposition pattern will result in a dish shaped deposit with the supernatant pond located centrally within the facility, remote from all confining embankments. Tailings deposition will be managed to maintain the supernatant pond away from the embankments, in order to reduce seepage and to ensure that reclaimed water is clear and suitable for reuse in the milling process.

Reclaim Water

Construction of the Starter facility will commence some time prior to mill start-up. Water impounded within the starter facility will therefore be available for mill commissioning and early operations. Mill process water for ongoing operations will be reclaimed from the TSF supernatant pond. The annual water balances completed for the PFS using average precipitation conditions indicate that the facility water balance will be slightly positive during the operational life of the facility. Releases of water from the TSF may be required later in the operational life of the facility but under average conditions a make-up water system should not be required for the milling process other than for the supply of clean water in order to meet potable and process water requirements. More detailed water balances will be completed for the Feasibility Study of the Schaft Creek project in order to more definitively estimate the water supply and water discharge requirements.

Water will be reclaimed from the tailings pond by a barge-mounted pump station. The water will consist of supernatant from the settled tailings and runoff from precipitation and snowmelt within the TSF catchment area. A dedicated steel/HDPE pipeline located along the main western access road will convey the reclaimed water to the process water tank.

The floating reclaim pump station in the TSF will be located in the south-central part of the facility.

A temporary reclaim pump station will initially be required in the northern part of the facility. This pump station will pump water from the initial supernatant pond location to the location of the permanent decant. As the facility is filled, the supernatant water pond will migrate southwards towards the location of the main floating reclaim pump station. The barge pumps will be controlled from the mill control room based on the water level in the process water tank. The barge will be fitted with vertical turbine pumps including standby pumping capacity and all necessary control, check, drainage and isolation valves. Under normal operating conditions one pump will be operated at all times during winter to reduce the potential for freezing of the water in the reclaim pipeline.

Reclaimed water will be pumped from the reclaim barge to a process water tank at the mill. The operational storage capacity of this tank will be approximately 100,000 m^3. The reclaim pipelines will be graded to minimize high or low sections and to allow for gravity drainage back into the TSF or the process water tank.

The reclaim pipeline from the TSF will consist of sections of large-diametre HDPE and steel pipe. Steel pipe will be used only for the initial high-pressure sections of the pipeline and HDPE pipe will be used for the remainder of the pipeline. During the early stages of facility development an HDPE pipeline will be laid from the temporary pond location at the northern part of the facility to the temporary water storage facility located in the vicinity of the permanent reclaim pump system.

18.13.5 Copper Concentrate Thickening, Filtration, Storage

Final copper concentrate from the molybdenum rougher flotation tailings stream with a slurry density of approximately 25% solids will be thickened to 60% solids in a 30 m (98.43 ft) diametre thickener. The thickened copper concentrate will be pumped to one of two pressure belt-type dewatering filters which will reduce the moisture content of the copper concentrate to approximately 8.5%.

Copper concentrate filter cake reports to the covered copper concentrate stockpile via the copper concentrate stacking belt conveyor. The copper concentrate will be loaded into transport trucks with a front-end loader for shipment.

18.13.6 Molybdenum Concentrate

The concentrate from the 5th molybdenum cleaner flotation circuit reports to a molybdenum concentrate surge tank, which in turn, is pumped to a disc-type molybdenum concentrate dewatering filter. Filtrate from the molybdenum dewatering filter is returned to the molybdenum concentrate filtrate storage tank via filtrate receivers for reuse as molybdenum process water in the molybdenum circuit. At this point in the process, the molybdenum concentrate has a moisture content of approximately 9%.

The cake will be transported by a molybdenum concentrate transfer screw conveyor from the molybdenum filter press to the concentrate dryer where its moisture content is reduced to approximately 3%. The concentrate is then discharged through a retractable sack fill chute into bulk bags for shipping.

18.13.7 Reagents

18.13.7.1 *Lime*

Burnt or quick lime (CaO) will be added to the crushed ore feeding the SAG mills in order to reduce the quantity of slaked lime required to increase the pH of the bulk rougher banks to 9.0. Slaked lime [$Ca(OH)_2$] will be used to continue increasing the pH levels at various points in the process. For example, the bulk concentrate regrind sump will operate at a pH of 10.0, and the cleaner flotation steps and copper concentrate thickener will operate at a nominal pH of 11.0.

18.13.7.2 *Flocculant*

Flocculant distribution pumps will deliver the flocculant to the tailings thickeners, the copper concentrate thickener and the bulk concentrate thickener.

18.13.7.3 *Sodium Hydrosulfide*

A sodium hydrosulfide (NaSH) transfer pump will deliver the NaSH to all five stages of molybdenum cleaning flotation cells, the molybdenum cleaner/scavenger flotation bank and the molybdenum rougher flotation bank.

18.13.7.4 *Collectors*

One collector, fuel oil No. 2 or equivalent, will be delivered by trucks and unloaded to a storage tank via an unloading pump. Two feed pumps (one operating and one standby) will feed the bulk rougher, rougher/scavenger, cleaner, and cleaner/scavenger circuits, as well as the molybdenum rougher and cleaner flotation circuits.

A transfer pump will deliver sodium ethyl xanthate (SEX) to the storage tank. Two distribution pumps (one operating and one standby) will feed a head tank. A control valve system will feed reagent to the rougher flotation feed distributor, the first bulk cleaner flotation feed distributor and the molybdenum rougher conditioning tank.

18.13.7.5 *Frother*

The frother, methyl isobutyl carbinol (MIBC), will be delivered by trucks and unloaded to a storage tank via an unloading pump. Two transfer pumps (one operating and one standby) will feed a head tank. The frother will then be fed at controlled rates to the bulk rougher and cleaner flotation cells and the first bulk cleaner feed distributor, the cleaner/scavenger flotation circuit, the molybdenum rougher conditioning tank and the molybdenum cleaner feed distributor.

18.13.7.6 *Sulphuric Acid*

The sulphuric acid (H_2SO_4) will be delivered by trucks and unloaded to a storage tank via an unloading pump. Two transfer pumps (one operating and one standby) will feed the acid to the molybdenum rougher flotation bank at a controlled rate to reduce the slurry pH to 10.5.

18.13.7.7 *Utilities*

Potable Water

The source of potable water will be a series of fresh water wells located in the Schaft Creek valley tapping into the upper aquifer at an average depth of 200 to 300 m. This water will be pumped to a fresh water tank located near the concentrator building. From the fresh water tank it will be distributed to the mancamp and the other buildings on site.

Process Water

Process water will predominately consist of supernatant water from the copper concentrate thickener and tailings thickeners, mine pit dewatering wells and make-up/dilution water as needed from the fresh water tank. Any overflow of process water will be sent to the TSF.

Air

Air for the bulk flotation circuit is supplied by air blowers. Compressed air is supplied to various areas in the processing plant by the air compression plant that is located in the concentrator building. Additional users of the compressed air include the truck shop, the process equipment instrumentation and valve controls, and pneumatic HVAC controls throughout the mine site.

Compressed Nitrogen

Due to the usage of NaSH (sodium hydrosulfide) as a reagent in the molybdenum flotation circuit, air is an unsuitable flotation medium; therefore, in the molybdenum rougher cells and the molybdenum cleaner cells, compressed nitrogen will be used in lieu of compressed air. These molybdenum flotation cells will be covered or enclosed as well to minimize NaSH oxidation by contact with ambient air. Because of the remote location of the mine, the nitrogen for the molybdenum circuit will be generated on site by a Pressure Swing Adsorption (PSA) nitrogen generation plant.

18.14 Control Philosophy

18.14.1 Process Control Philosophy

Plant process control and monitoring functions will be performed by a microprocessor-based distributed control system (DCS). Microprocessor-based controllers, process-interface stations and operator stations, connected by a common data bus enable operators to centrally control all plant facilities. Color video display terminals with keyboards, graphics pointing devices, and alarm and report printers constitute the operator/process control interface. Equipment can normally be controlled and/or monitored from a common central control room. A local control room located at the primary crusher will be included for the mine.

A redundant deterministic common data bus will be interconnected to all equipment, including all servers, the process controllers, data link devices to packaged equipment programmable logic controllers and other gateways and communication devices.

An interface device, residing on the data bus, will be connected to an open ethernet/token ring TCP/IP network. Operator terminals, engineering workstations, a supervisory computer and an historical trend system are connected to this network. The network allows integration with management information systems. Fiber optic cable data paths are utilized for process communications between each process facility.

18.14.2 Control Systems

18.14.2.1 *General*

Plant process control and monitoring functions will be performed by an advanced microprocessor-based distributed control system (DCS) such as a Delta V. Microprocessor-based controllers, process interface stations and operator stations, connected by a common data bus, will enable operators to centrally control all plant facilities.

Color video display terminals (VDT's) with keyboards, graphics pointing devices and alarm and report printers will constitute the operator/process control interfaces. All facility equipment will normally be controlled and/or monitored from a common central control room.

18.14.2.2 *Control Room*

A central control room (CCR) will be provided in the concentrator facility core, which will be the main operating center for the complex. From the CCR control consoles, primary crushing, material handling systems, grinding and flotation, molybdenum plant, reagents, tailings, filter plant, and utility systems will be monitored and/or controlled.

A crusher control room will be provided in the primary crusher building at the mine and will be the operating center for the crusher and coarse ore transport conveyors. From the crusher operator interfaces, the primary crusher, overland conveyors and coarse ore stockpile shuttle conveyor systems will be monitored and controlled.

A molybdenum control room will be provided in the molybdenum plant and will be the operating center for molybdenum production. From the molybdenum operator interfaces, molybdenum flotation, regrind, leaching, drying, storage and loadout systems will be monitored and controlled.

18.14.2.3 *Computer Room*

A computer room adjacent to the CCR will contain engineering workstations (EWS), a supervisory computer, historical trend system, management information system (MIS) server, programming terminal, network and telecommunications equipment and documentation printers. This will primarily be used for DCS development and support activities by plant and control system engineers.

18.14.2.4 *Local Control*

Although the facilities will normally be controlled from the central control rooms, local VDT control stations will be selectively provided on the plant floor for occasional local monitoring and control of certain process areas.

These will provide all of the functions of the central control room console stations. Any local control panels that are supplied by others will be interfaced with the DCS for remote monitoring and/or control from the related control room.

18.14.2.5 *X-ray Analyzer*

An x-ray analyzing and monitoring system (XAMS) in the CCR will be interfaced to the DCS. The XAMS data will be integrated into the DCS database for data logging and reporting.

18.14.2.6 *Closed Circuit Television*

Closed circuit television (CCTV) video cameras will be used for viewing mine facilities and coarse ore stockpiles and coarse ore conveyor flights transfer stations. The CCTV monitors will be mounted in the central control room and crusher control room.

18.14.2.7 *Instrumentation Satellite Rooms*

Distributed field equipment associated with the control system (integrated function controllers, data processing units, data communications equipment and multiplexers) will be located in environmentally controlled instrument satellite rooms and electrical rooms throughout the facility. Marshaling cabinets for field termination of input and output signals will be located in the instrument satellite rooms for the DCS and electrical rooms for motor control.

18.14.2.8 *Packaged Equipment*

Control system devices supplied by packaged equipment vendors will be interfaced to the data bus for the purpose of monitoring/control of those systems from the DCS consoles.

Examples include the primary crusher, overland conveyor systems, SAG mill drives, x-ray analyzer system, molybdenum loadout packaging system, baghouse dust collectors, pressure filters, water treatment systems and truck weigh scale.

18.14.2.9 *Fault Tolerance*

CMS component redundancy will be selectively applied to achieve a high degree of system availability. The control system will continuously monitor itself for failure of various components and will advise the operator when and where a failure occurs. All redundant hardware will be removable for repair or replacement while the system is on line without any loss of control functions or data.

18.15 Plant Services

18.15.1 Mobile Equipment

The process facilities will be equipped with typical mobile equipment required for the operations, to include a mobile crane, man lift, forklifts and a small front-end loader.

18.15.2 Assay/Metallurgical Laboratories

A dedicated assay/metallurgical laboratory is located close to the processing facilities. The facilities will contain analytical, fire assay, AA/ICP, sample prep area and a large wet lab area. The facilities will be fully equipped to perform ongoing testwork to support the process plant, to include crushing, grinding, flotation, thickening and filtration lab scale equipment.

19.0 Mineral Resource and Mineral Reserve Estimates

19.1 Mineral Resource Estimate

Associated Geosciences Ltd. (AGL) has reported a mineral resource estimate for the Copper Fox Metals Inc. (Copper Fox) Schaft Creek project. The mineral resource estimate 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. The AGL report, *Updated Mineral Resource Estimate for the Schaft Creek Deposit, Northwest British Columbia, Canada Technical Report on a Mineral Property*.

It is noted here that this resource estimate does not include any information from the 2007 drilling program.

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. The CIM Standards provide specific definitions for Mineral Resources and three categories of Mineral Resources, i.e. Inferred Mineral Resources, Indicated Mineral Resources and Measured Mineral Resources. The definitions, taken directly from the standards, follow.

> "A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals 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-off grade of 0.20% at Schaft Creek can not be considered as mineral resources as they are potentially uneconomic. The CIM definitions for the resource categories are:

> "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 parameters, to support mine planning and evaluation of the economic viability of the deposit.

The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

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

A summary of the mineral resources of the Schaft Creek Deposit follows (Table 19.1):

Table 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

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 19.2 to Table 19.5.

Table 19.2
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
CuEq Cut-Off (%)	**Tonnes**	**Cu (%)**	**Mo (%)**	**Au (g/t)**	**Ag (g/t)**	**CuEq Grade (%)**
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

Table 19.2 Measured Mineral Resources (≥0.20 CuEq % Cut-Off)						
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 19.3 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 19.4 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

Table 19.4 Measured + Indicated Mineral Resources (≥0.20 CuEq % Cut-Off)						
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 19.5
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 19.6

Table 19.6
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%

19.2 Metal Equivalents

A recoverable copper equivalent (CuEq) grade has been estimated for the polymetallic Schaft Creek deposit at the request of Copper Fox. 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 Preliminary Feasibility Study (PFS). 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) and Preliminary Feasibility Study (PFS).

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

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

The values of the variables shown in the copper equivalent grade equation are shown in Table 19.7.

Table 19.7 Assumptions Used in the Copper Equivalent Estimation		
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 (Avoirdupois) per kilogram	2.2046
ozt/g	Ounces (troy) per gram	0.03215

Table 19.7 Assumptions Used in the Copper Equivalent Estimation		
Variable	**Description**	**Value (for this report)**
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

As an example, using a 0.3% copper equivalent cut-off grade 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 greater than 100% change from a previously reported mineral resource estimate triggers a requirement within National Instrument (NI) 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 (avoirdupois), 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, NI43-101 requires that the two estimates be reconciled.

While the overall tonnes and grades at a 0% copper equivalent cut-off grade 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-off grades (particularly above a 0.20% CuEq).

This has the effect of increasing the tonnage at any particular copper equivalent cut-off grade while raising the copper equivalent grade.

19.3 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%.

Figure 19.1 through Figure 19.4 illustrate the percentages of materials according to each resource category.

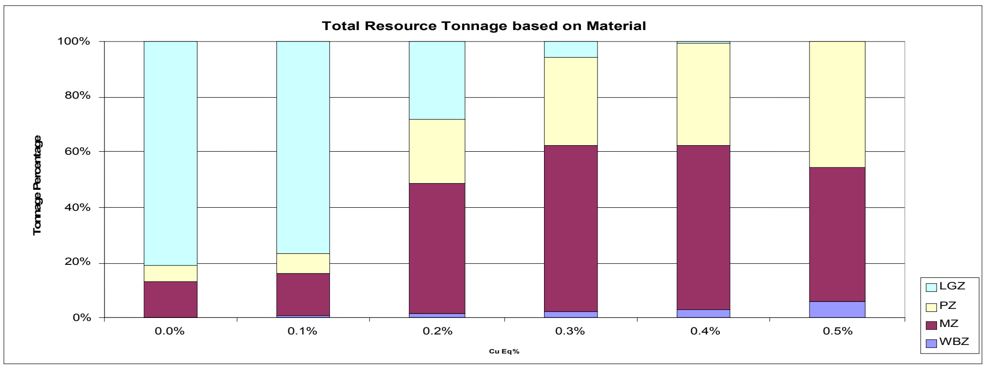


Figure 19.1 Percentages of Total Resource Tonnage Based on Various Zones

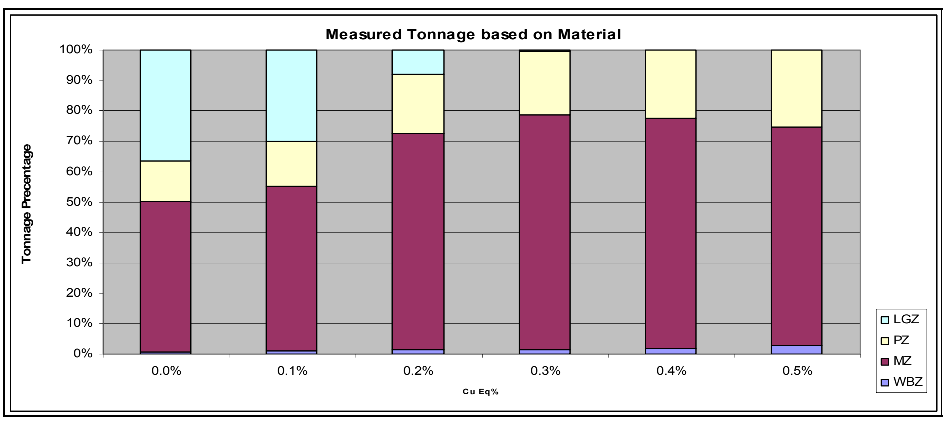


Figure 19.2 Percentages of Measured Resource Tonnage Based on Various Zones

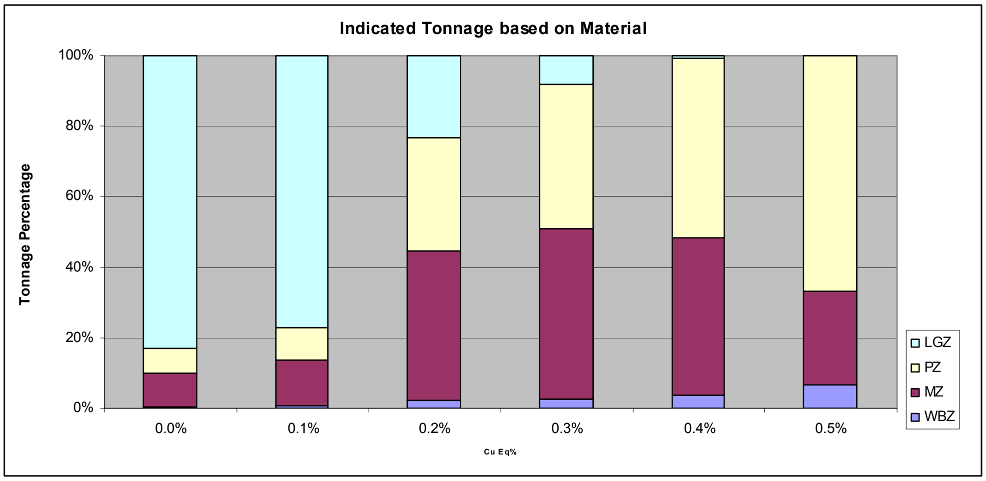


Figure 19.3 Percentages of Indicated Resource Tonnage Based on Various Zones

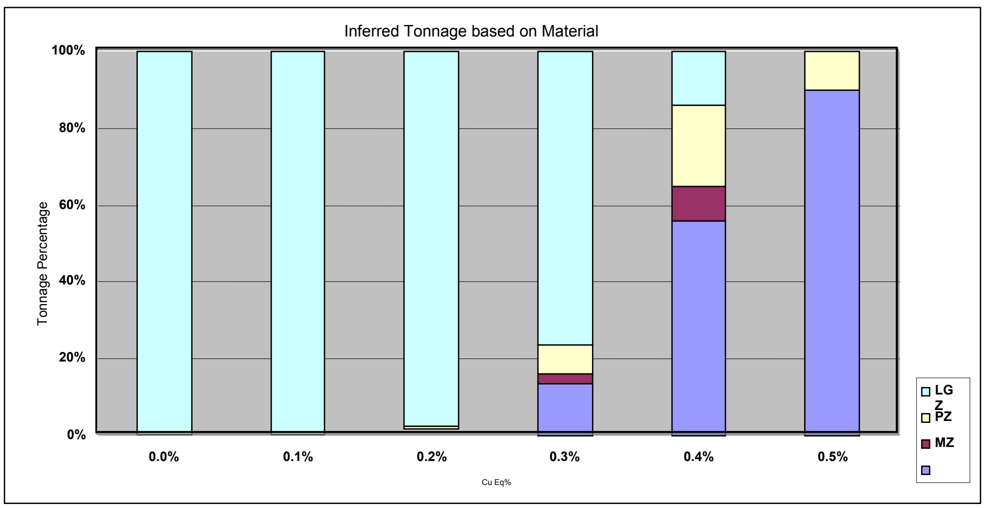


Figure 19.4 Percentages of Inferred Resource Tonnage Based on Various Zones

19.4 Grade Tonnage Curves

Grade tonnage curves for the measured, indicated, measured & indicated and inferred resources categories are displayed in Figure 19.5 to Figure 19.8

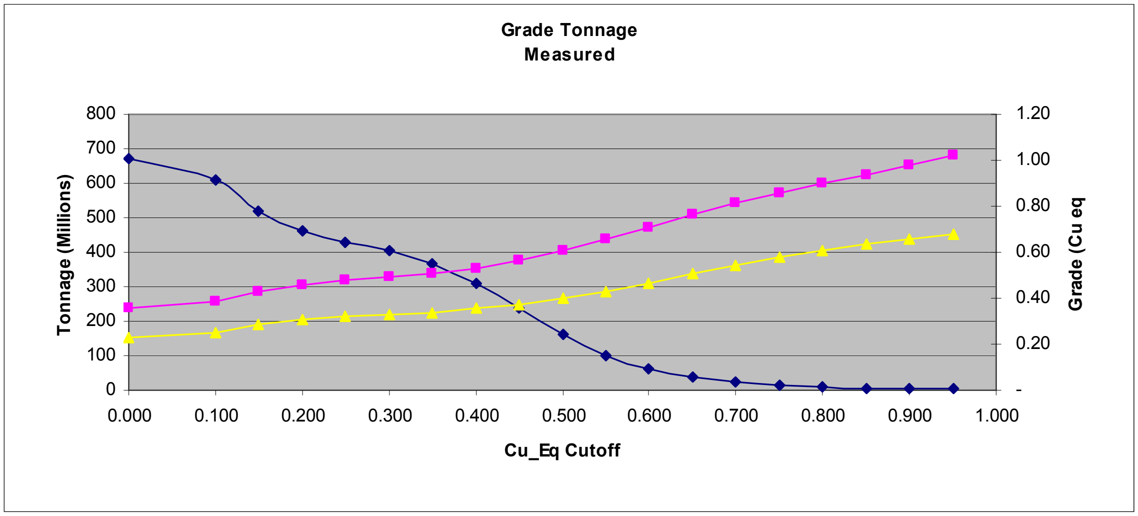


Figure 19.5 Copper Equivalent Grade Tonnage Curve for the Measured Resource Category

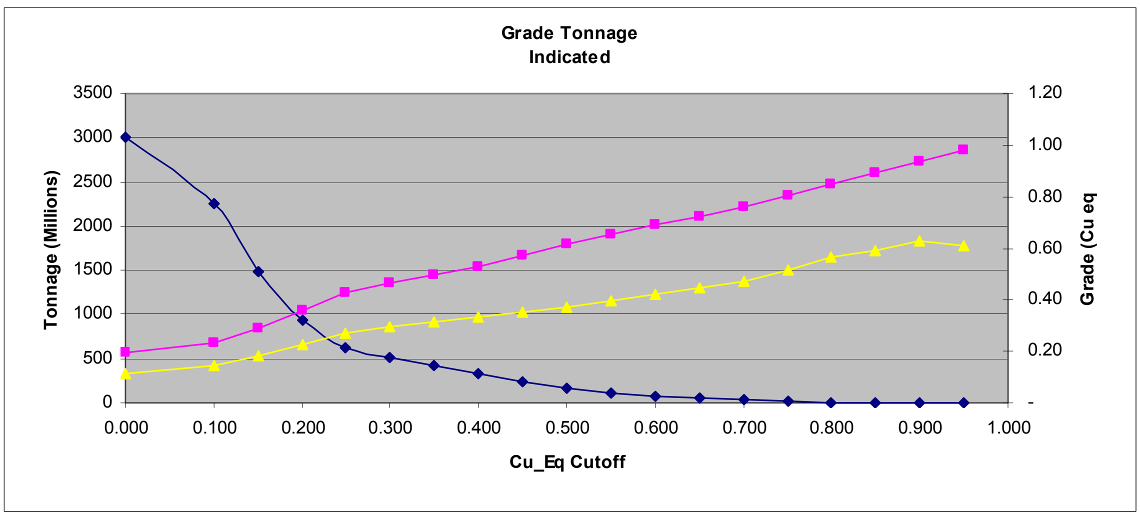


Figure 19.6 Copper Equivalent Grade Tonnage Curve for the Indicated Resource Category

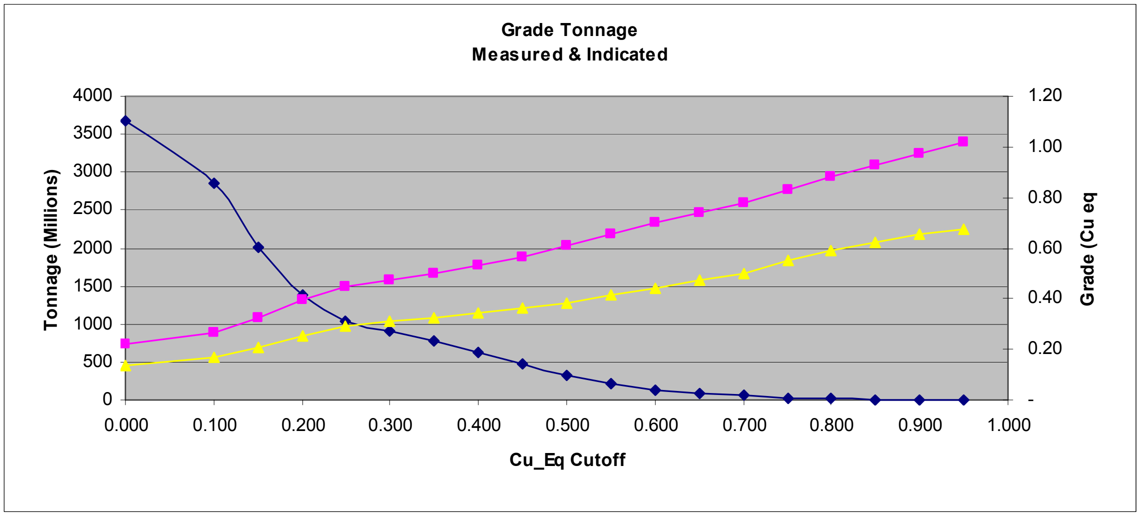


Figure 19.7 Copper Equivalent Grade Tonnage Curve for the Measured and Indicated Resource Categories

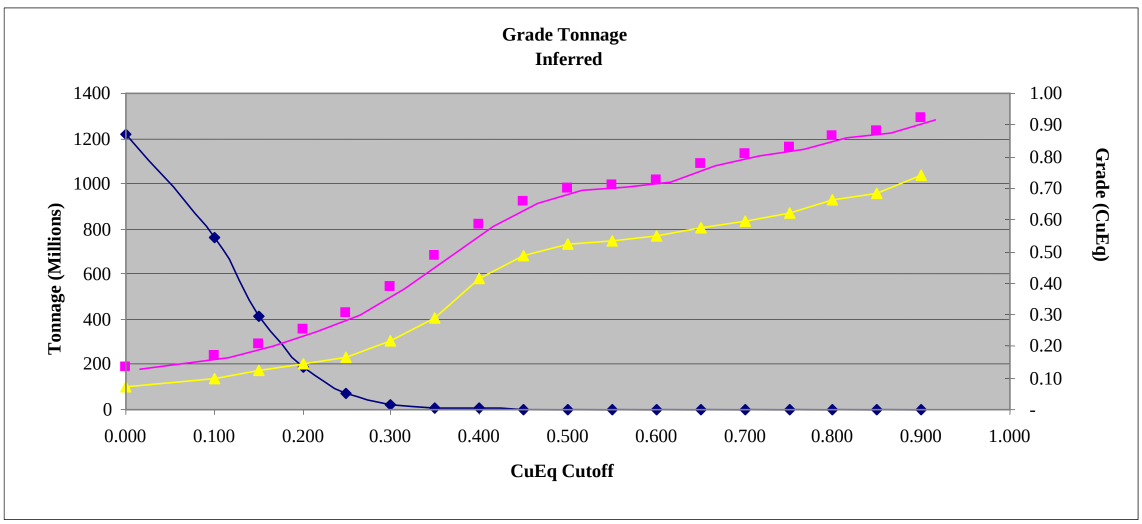


Figure 19.8 Copper Equivalent Grade Tonnage Curve for the Inferred Resource Category

19.5 Copper Cut-off Grade-tonnage Curves

Grade-Tonnage curves at a percent copper cutoff for the various zones are depicted in Figure 19.9 through Figure 19.13.

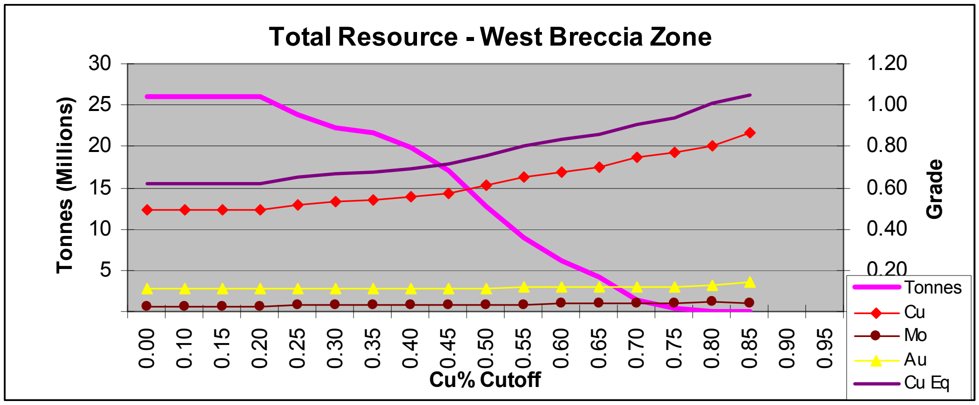


Figure 19.9 Percent Copper Cut-off Grade-Tonnage Curve for the West Breccia Zone

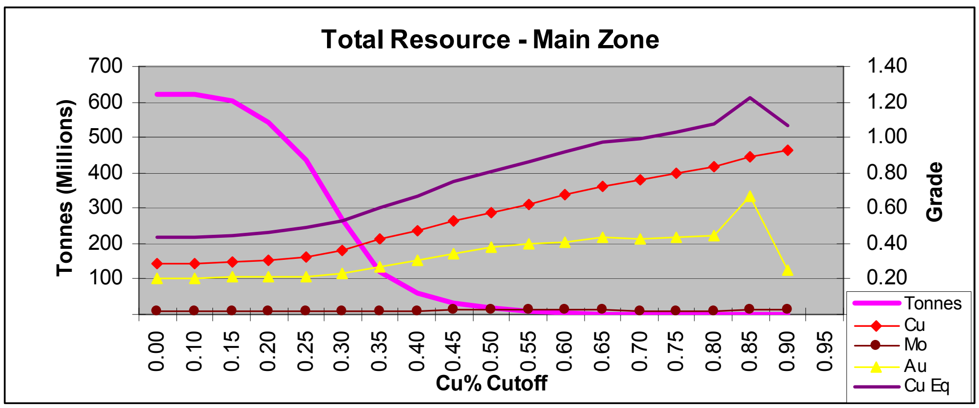


Figure 19.10 Percent Copper Cut-off Grade-Tonnage Curve for the Main Zone

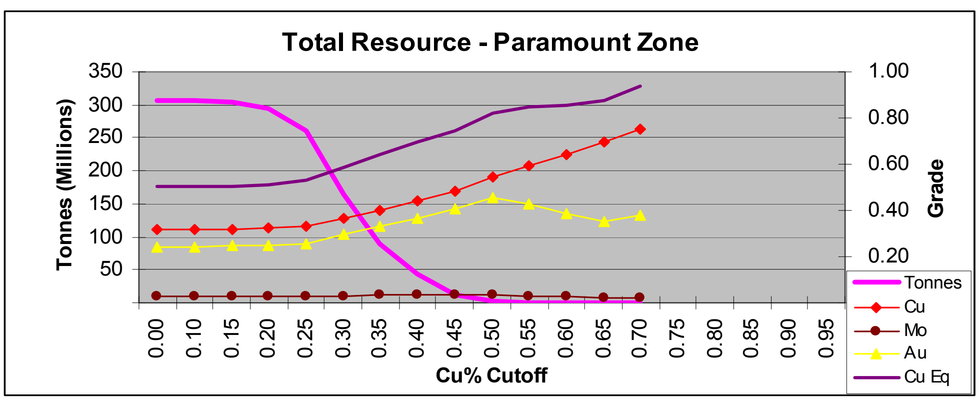


Figure 19.11 Percent Copper Cut-off Grade-Tonnage Curve for the Paramount Zone

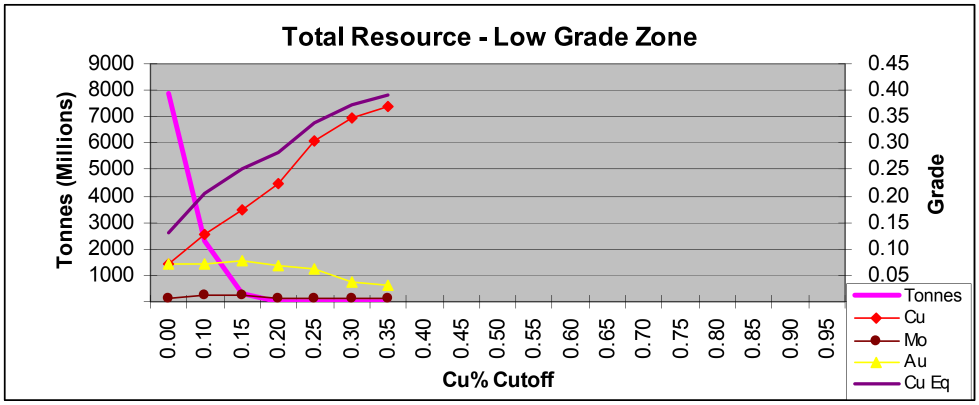


Figure 19.12 Percent Copper Cut-off Grade-Tonnage Curve for the Low-Grade Zone

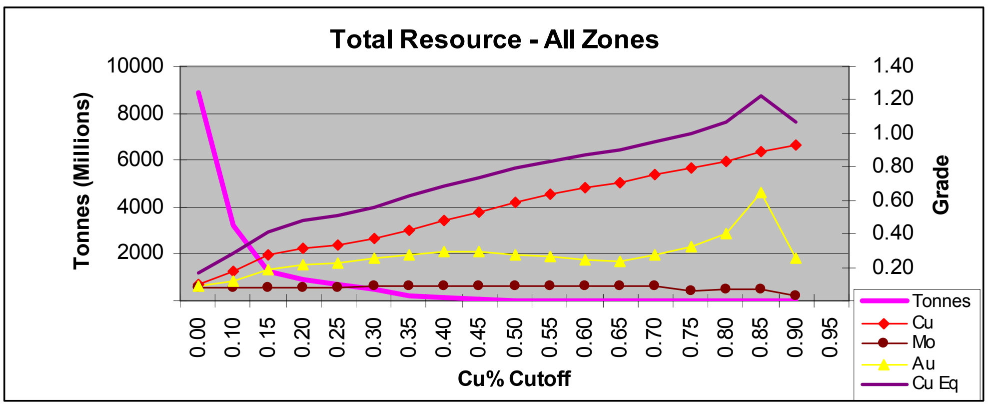


Figure 19.13 Percent Copper Cut-off Grade-Tonnage Curve for All Zones

19.6 Resource Classification

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

Table 19.8 depicts the process used in the resource classification. This classification was based around the major element Cu%.

Table 19.8 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

Figure 19.14 through Figure 19.16 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 19.17, Figure 19.18 and Figure 19.19 depict the interpolated element grades converted to a copper-equivalent value.

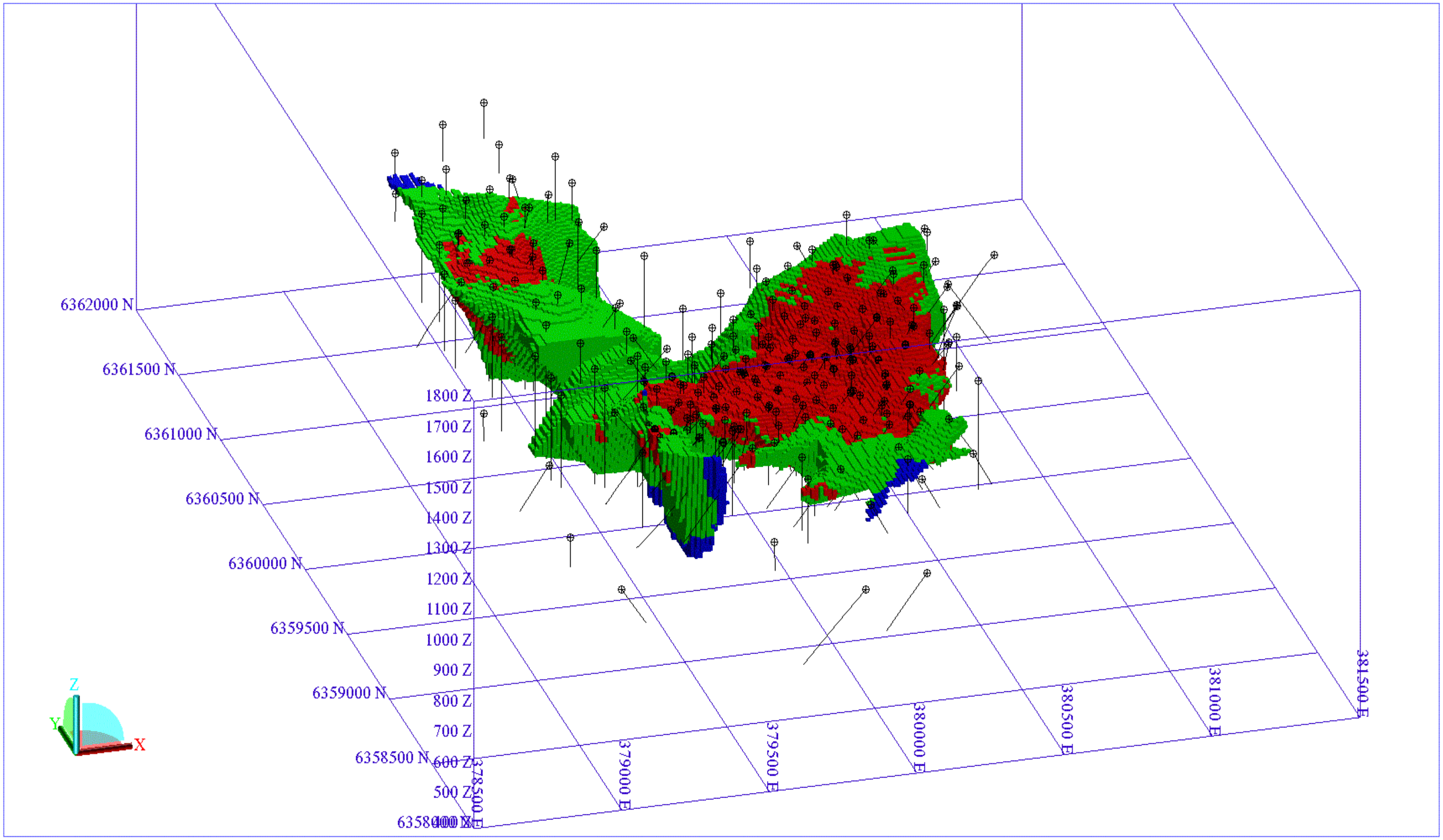


Figure 19.14 Resource Classifications of the Three Main Zones

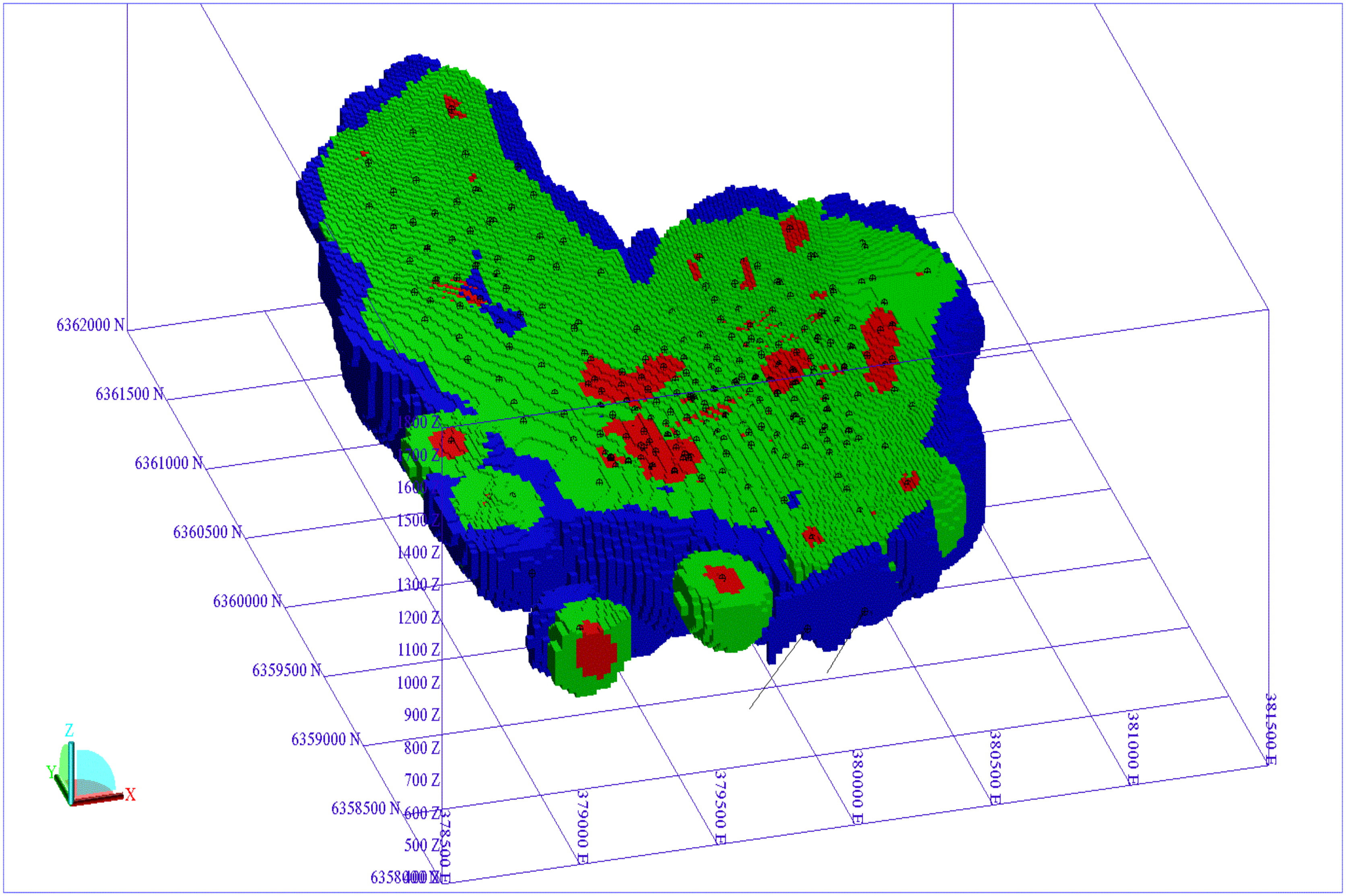


Figure 19.15 Resource Classifications of the Three Main Zones, Waste Included

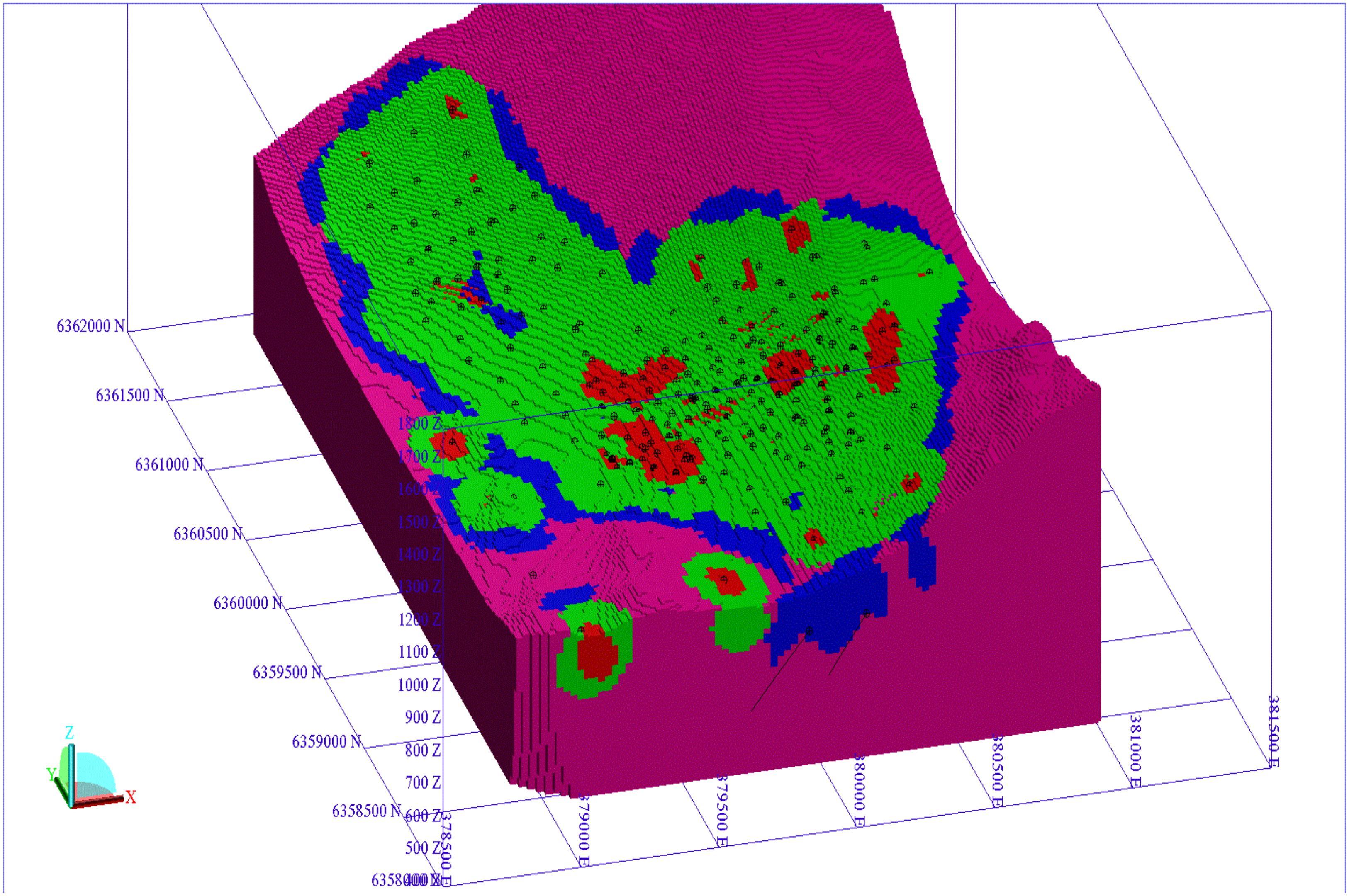


Figure 19.16 Resource Classifications of the Three Main Zones, Excluding Air

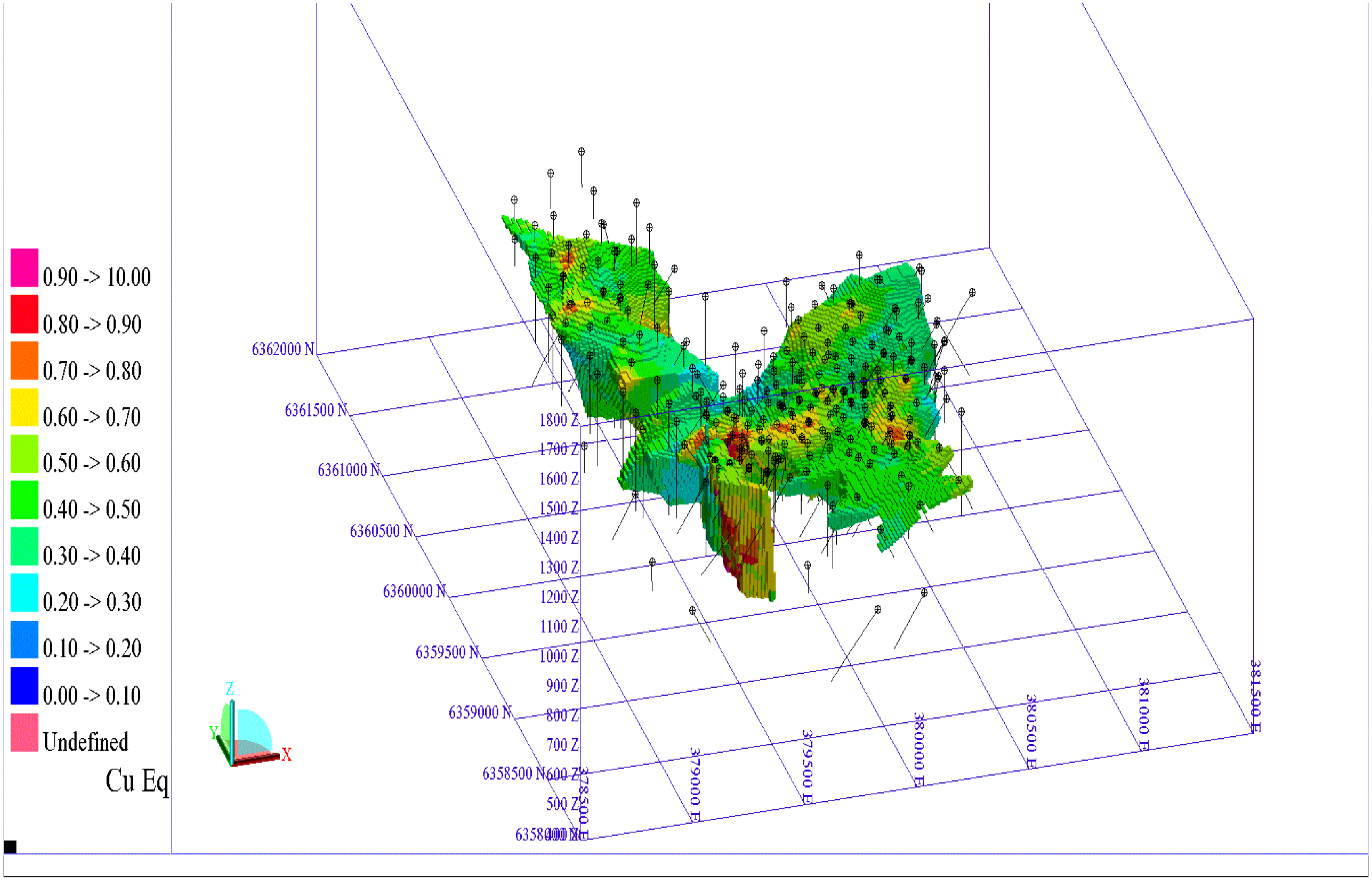


Figure 19.17 Interpolated Element Grades Converted to a Copper-Equivalent Value

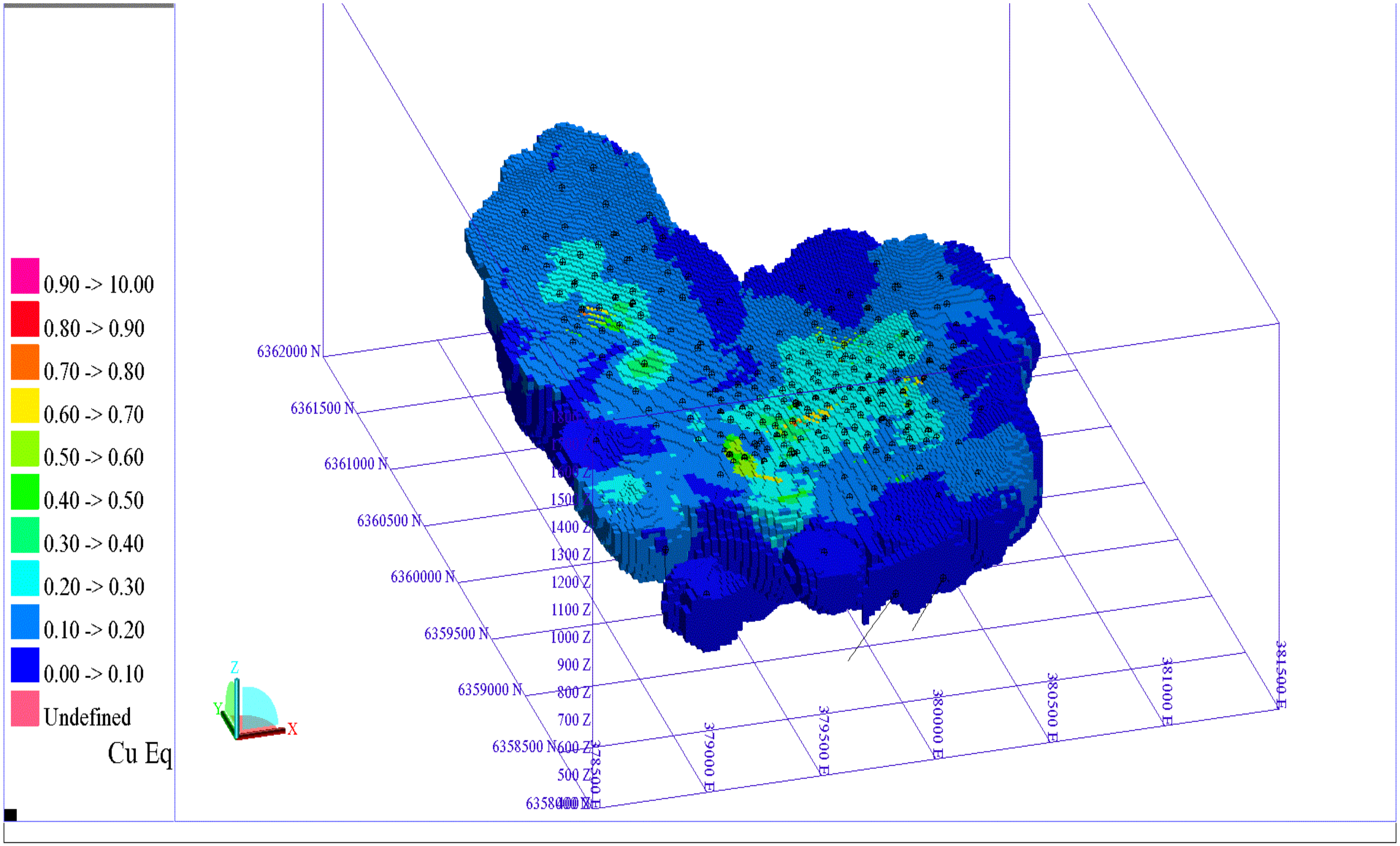


Figure 19.18 Interpolated Element Grades Converted to a Copper-Equivalent Value

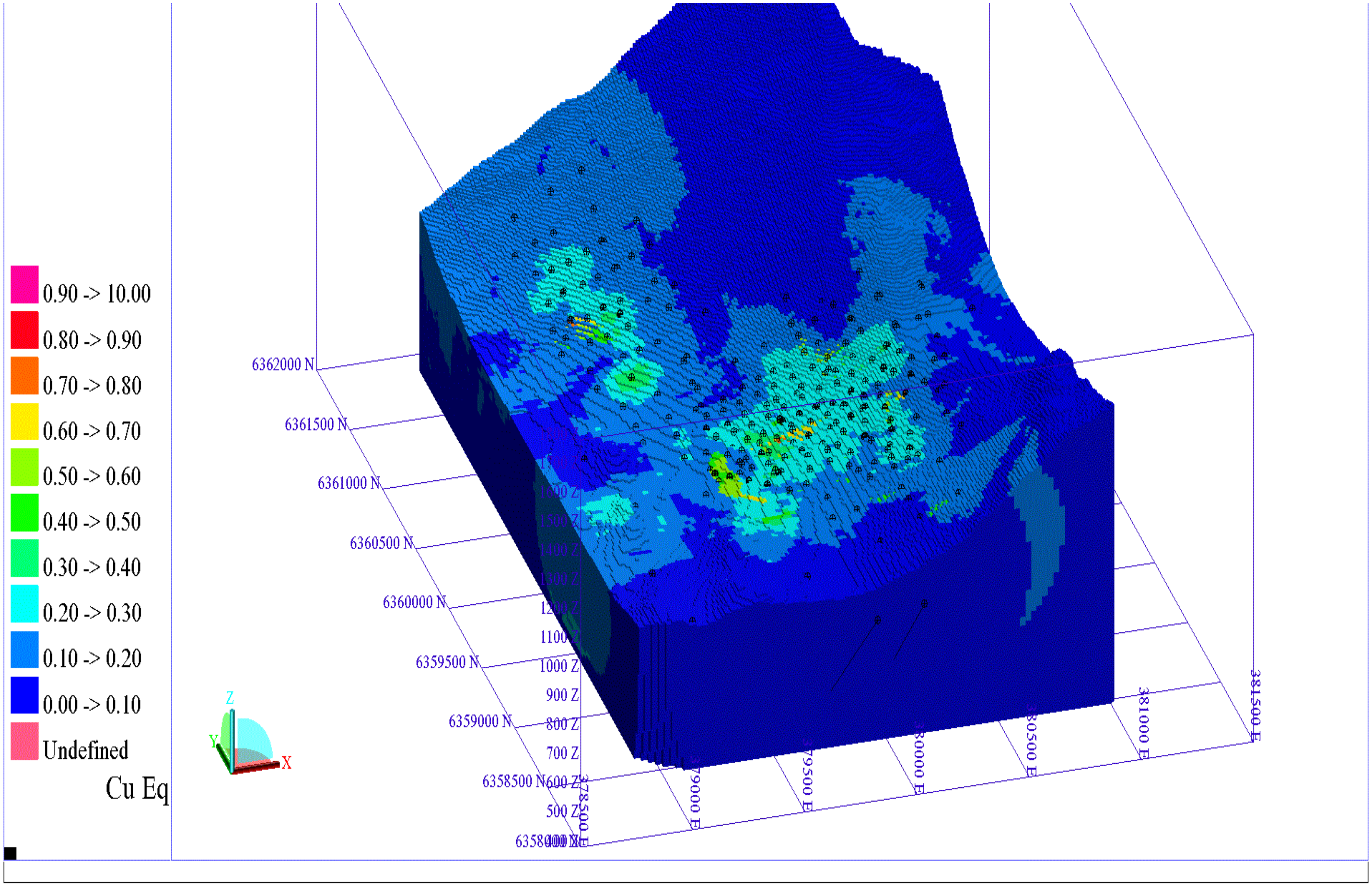


Figure 19.19 Interpolated Element Grades Converted to a Copper-Equivalent Value

19.7 Mineral Reserve Estimates

Mineral reserves for the Schaft Creek property were developed by Moose Mountain Technical Services.

Based on the Economic Lerchs-Grossman pit limits, detailed pit phases have been designed, as described in more detail in Section 25 (Additional Requirements for Technical Reports on Development Properties and Production Properties) of this report. The waste and ore reserves for the material within the Ultimate pit limit (P651) and for each incremental pit phase are listed below. Reserves are based on the following mining parameters:

- 10% mining dilution applied at the contact between ore and waste.
- Dilution grades are estimated at 4.63 $/t NSR, 0.076% Cu, 0.088 g/t Au, 1.76 g/t Ag and 0.005% Mo representing the average grade of material below the incremental waste/ore cut-off grade.
- 5% mining loss.
- Waste/Ore Cut-off grade of $5.05/t Net Smelter Return (NSR).

Table 19.9
Summarized Measured & Indicated Reserves for Schaft Creek

Pit	Description	ROM ORE (Mt)	ROM Diluted Grades					Waste (kt)	Stripping Ratio (t/t)
			NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)		
P611	South Starter	297.8	16.7	0.330	0.239	1.685	0.017	241.2	0.81
P621i	South Incremental	136.1	14.3	0.266	0.207	1.509	0.018	178.6	1.31
P631	North Starter	52.9	16.2	0.291	0.185	1.940	0.024	45.4	0.86
P641i	Intermediate	208.1	14.3	0.277	0.148	1.674	0.020	476.1	2.29
P651i	Final	125.9	18.1	0.305	0.263	2.275	0.027	601.9	4.78
Total		821.1	15.9	0.299	0.211	1.760	0.020	1 543.2	1.88

Proven and Probable Reserves at Shaft Creek are summarized in the table below.

Table 19.10
Proven and Probable Reserves at Schaft Creek

Reserve Category	ROM ORE (Mt)	ROM Diluted Grades				
		NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
Proven	411.1	16.6	0.316	0.236	1.722	0.019
Probable	409.9	15.2	0.283	0.186	1.798	0.020
Total	821.1	15.9	0.299	0.211	1.760	0.020

20.0 Other Relevant Data and Information

20.1 Additional Information

At the time of printing, it is the opinion of the author that there is no other relevant data or information required to make the report understandable and not misleading.

Please reference Section 25 (Additional Requirements for Technical Reports on Development Properties and Production Properties) for other pertinent information regarding the development of the Schaft Creek project.

21.0 Interpretation and Conclusions

21.1 Introduction

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, published in December 2007, is a compilation of the results of the Schaft Creek study up to that point in time.

Due to the results of the PEA and the potential nature of the project, Copper Fox continues with site and investigative work with the intention to develop the project. Copper Fox has commissioned a Preliminary Feasibility Study (PFS) for its Schaft Creek project in 2008. The purpose of this study is to advance those concepts and designs developed during the PEA, including metallurgical testwork, resource and reserve estimates, mine design, operating cost estimates, capital cost estimates and project related cash flow analysis. The results of this study will further assist management of Copper Fox in making decisions in regards with further advancing the project towards eventual development.

21.2 Conclusions

This report is a Preliminary Feasibility Study (PFS), by which meaning the report is a comprehensive study of the viability of a mineral project that has advanced to a stage where the mining method and pit configuration has been established and an effective method of mineral processing has been determined. In addition the study includes a financial analysis based on reasonable assumptions of technical, engineering, legal, operating, economic, social, and environmental factors and the evaluation of other relevant factors which are sufficient for a qualified person, acting reasonably, to determine if all or part of the mineral resource may be classified as a mineral reserve.

This PFS is complete based upon the criteria specified. Certain assumptions have been made to allow for typical conditions in the area of the project, these assumptions were based on industry acceptable standards and sound engineering practice.

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, Associated Geosciences Ltd., Moose Mountain Technical Services, Rescan Environmental Services Ltd., DST Consulting Engineers Inc., BGC Engineering, Hyyppa Engineering, LLC, McElhanney Consulting Services Ltd., HM Hamilton & Associates Inc., PR Associates, Walter Hanych, Knight Piésold, G&T Metallurgical Services Ltd., Vandan Suhbatar and Samuel Engineering Inc.

21.2.1 Resource

The Schaft Creek Deposit has been explored extensively prior to its acquisition by Copper Fox. 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 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 21.1
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 0.20% 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.

21.2.2 Reserves

The ore reserves are listed below. Reserves are based on the following mining parameters:

- 10% mining dilution applied at the contact between ore and waste;
- Dilution grades are estimated at 4.63 $/t NSR, 0.076% Cu, 0.088 g/t Au, 1.76 g/t Ag and 0.005% Mo representing the average grade of material below the incremental waste/ore cut-off grade;
- 5% mining loss;
- Waste/Ore Cut-off grade of $5.05/t Net Smelter Return (NSR).

Table 21.2
Proven and Probable Reserves at Schaft Creek

Reserve Category	ROM ORE (Mt)	ROM Diluted Grades				
		NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
Proven	411.1	16.6	0.316	0.236	1.722	0.019
Probable	409.9	15.2	0.283	0.186	1.798	0.020
Total	**821.1**	**15.9**	**0.299**	**0.211**	**1.760**	**0.020**

21.2.3 Mining

A comprehensive production schedule was produced based on 100,000 tpd mill feed. Based on the Economic Lerchs-Grossman pit limits, detailed pit phases have been designed, as described in more detail in Section 25 (Additional Requirements for Technical Reports on Development Properties and Production Properties) of this report. The waste and ore reserves for the material within the Ultimate pit limit (P651) and for each incremental pit phase are listed below.

Table 21.3
Summarized Measured & Indicated Reserves for Schaft Creek

Pit	Description	ROM ORE (Mt)	ROM Diluted Grades					Waste (kt)	Stripping Ratio (t/t)
			NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)		
P611	South Starter	297.8	16.7	0.330	0.239	1.685	0.017	241.2	0.81
P621i	South Incremental	136.1	14.3	0.266	0.207	1.509	0.018	178.6	1.31
P631	North Starter	52.9	16.2	0.291	0.185	1.940	0.024	45.4	0.86
P641i	Intermediate	208.1	14.3	0.277	0.148	1.674	0.020	476.1	2.29
P651i	Final	125.9	18.1	0.305	0.263	2.275	0.027	601.9	4.78
Total		821.1	15.9	0.299	0.211	1.760	0.020	1 543.2	1.88

21.2.4 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 are consistently in the range of 33% copper concentrate grades and 50% moly concentrate grades with recoveries of 88% for copper and 71% for moly. 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 100,000 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 all 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. Preliminary testwork indicate that the moly concentrate will contain a significant amount of Re.

21.2.5 Key Project Resutls

Key results from this Preliminary Feasibility Study are summarized in the following table.

Table 21.4
Key Project Parameters and Results

September 9, 2008, Rev. 5a				
Total Resource (M&I)	tonnes	1,393,282,171	@ 0.25% Cu	
Total Reserve	tonnes	816,706,750		
LOM Mill Feed	tonnes	812,230,421		
LOM Waste	tonnes	1,543,190,551		
LOM Strip Ratio		1.88		
Daily Feedrate	tpd	100,000		
Mine Life	yrs	22.6		
Connected Load	MW	146.8		
Avg Power Demand	MW	121.4		
Power Cost	$/kWh	0.0469		
Foreign Exchange Rate		US$1 = C$1		
Total Initial Capex	$ (000's)	**2,950,406**		
Directs	$ (000's)	1,315,484		
Indirects	$ (000's)	610,108		
Owner	$ (000's)	459,757		
Taxes	$ (000's)	28,566		
Contingency	$ (000's)	536,490		

Table 21.4 Key Project Parameters and Results				
Working Capex	$ (000's)	146,420		
Total Sustaining Capex	$ (000's)	797,379		
Mine	$ (000's)	232,893		
Mill	$ (000's)	220,000		
Tailings	$ (000's)	257,486		
Reclamation & Closure	$ (000's)	87,000		
Total LOM Opex	$ (000's)	**10,138,610**		
Total LOM Opex	$/t ore	**12.49**		
Mining	$/t ore	4.14		
Processing	$/t ore	3.94		
G&A	$/t ore	0.93		
Conc Handling & Treatment	$/t ore	3.49		
Contingency (0%)	$/t ore	0.00		
Total LOM Taxes	$ (000's)	4,041,652		
Metrics		**Head Grades**	**Conc Grades**	**Recoveries**
Cu	%	0.301%	33.85%	88.4%
Mo	%	0.020%	50.0%	71.3%
Au	g/t	0.212	21.90	81.3%
Ag	g/t	1.761	158.30	70.7%
Base Case Pricing (Trailing 3 Year Avg - August 29, 2008)				
Cu	$/lb	3.12		
Mo	$/lb	33.00		
Au	$/oz	692.85		
Ag	$/oz	13.09		
Cashflow Results		**Before Tax**	**After Tax**	**Direct Tax Effects**
IRR	%	18.6%	15.3%	-3.21%
NPV @ 0%	$ (000's)	$11,734,537	$7,692,885	($4,041,652)
NPV @ 5%	$ (000's)	$4,787,931	$2,983,847	($1,804,084)
NPV @ 8%	$ (000's)	$2,764,475	$1,597,500	($1,166,974)
NPV @ 10%	$ (000's)	$1,868,441	$979,686	($888,755)
NPV @ 12%	$ (000's)	$1,208,843	$522,898	($685,945)
NPV @ 15%	$ (000's)	($149,721)	($422,853)	($273,132)
Payback Period	yrs	4.7	4.9	
Rock Value *	$/t ore	$31.47		
LOM Recoverable Revenue	$ (000's)	25,559,408		
Cu	%	54.4%		
Mo	%	32.2%		
Au	%	12.0%		
Ag	%	1.4%		
Total Metal Production		**LOM**	**Annual**	**Annual Tonnes**
Cu	lbs	4,762,524,025	211,104,788	95,757
Mo	lbs	255,194,418	11,311,809	5,131
Au	ozs	4,493,445	199,178	

Table 21.4 Key Project Parameters and Results				
Ag	ozs	32,480,015	1,439,717	
CFM Portion of Metal Production		LOM	Annual	Tonnes
Cu	lbs	1,112,049,360	49,292,968	22,359
Mo	lbs	59,587,897	2,641,307	1,198
Au	ozs	1,049,219	46,508	
Ag	ozs	7,584,084	336,174	
Facilities Startup		4 QTR 2013		

21.3 Risks

- Some of the major risks associated with this project are due to the size of the planned operations, the remote location of the geologic deposit and the current state of the mining industry which means a shortage of virtually every resource that will be needed to develop the Schaft Creek project.
- Personnel that have the required knowledge to design and operate a mine of this size are in short supply all around the world. There are simply not enough mining engineers and mineral processors to meet the current needs of the global mining industry. This is due to a combination of insufficient numbers of new graduates and an aging workforce.
- To complicate matters further, there is also a limited supply of engineers from most disciplines so it difficult to find engineering companies that are capable of meeting the demands of designing a project as large as the Schaft Creek project. All engineering companies that have the expertise to work on mining projects are overworked. This means that no project receives the attention that is required to minimize mistakes and complete the most optimum designs.
- Workers are either aging, i.e. in many cases they have come out of retirement to enjoy the boom or they are young, inexperienced and, although eager, not always the most competent without significant need for mentoring and oversight.
- Equipment to support the development of the project, both mining equipment and processing equipment, is in high demand and short supply. This equates to long lead times for delivery of major equipment and premium, constantly-escalating prices.
- These types of pressures make it very difficult to complete accurate studies at the Scoping, Preliminary Feasibility Study and Feasibility Study stages of project development. Of course, the less-accurate studies that are done with information that is limited and conceptual at best, e.g. Preliminary Economic Assessments and Preliminary Feasibility Studies have a high risk that the costs will not be accurate by the time a project is constructed and commissioned.

21.3.1 Mining Risks

- The current short supply with long delivery times for tires mean that supply contracts are required from tire manufactures early in 2009 to ensure that tires will be available in time for project start up. Tire supply remains a project risk.

- A detailed hydro-geological evaluation of the area is needed to improve the accuracy of pit dewatering design. Vertical dewatering wells have not been included as part of the required PFS activities.
  Results from the 2008 field program may show this to be a future requirement to lower the water table within the pit prior to mining. Although not included in the current estimate, it may be necessary to prevent water inflow from the nearby glaciers through the glacial sediment. The 2008 field program needs to identify whether this is an issue that needs to be addressed.
- A 3-dimensional Acid Base Accounting (ABA) model is required to classify waste geochemistry. The spatial distribution of potentially acid generating (PAG) and non-potentially acid generating (NPAG) rock might significantly alter the haulage phase mining sequences waste destination requirements and ultimately haulage costs. For this study the PAG material is estimated at ten percent of total material mined. Only after the appropriate ABA geochemistry model is completed can the distribution of PAG and NPAG material be incorporated into the geological block model. The quantity and spatial distribution of PAG material will directly affect the truck haulage hours for this material, but at this time it is not known the extent of this impact.
- 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.

21.3.2 Economic Risks

Predicting future commodity prices is and will always remain a big part of economic risk associated with any large project in the development stage. The same increases in commodity prices that drive the development of numerous new projects also mean high prices for the materials that are required to develop the new facilities. Add to this the major increases in iron ore and steel prices over the last several years and a major economic risk is trying to determine what the cost of the equipment will be by the time it is delivered two or three years in the future. Add to this the foreign currency exchange rate fluctuations and the true costs for future development is hard to predict.

21.3.3 Operational Risks

The project is located in the northern region 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.

21.3.4 Closure and Reclamation Risks

- The largest risk of reclamation and closure at a sulphide mining operation in an area of relatively-high precipitation is the risk of developing an Acid Rock Drainage (ARD) problem, particularly in the waste rock disposal or storage facilities. ARD is caused by sulphidic minerals, such as pyrite, reacting with water and oxygen. ARD is a significant problem in legacy mining operations that operated before modern-day understandings of the problems and how to mitigate them were established. The best ways to mitigate ARD are to measure the acid generating potential (AGP) and the acid neutralizing potential (ANP) of waste rock and to manage the disposal such that sufficient neutralizing potential is available to counteract the acid generation of the material. It is also helpful to isolate the potentially acid generating (PAG) material from oxygen by burying it within the waste dumps and capping them or by submerging the PAG material under water.
- A secondary concern for ARD is the tailings that are placed in the tailings storage facility (TSF). In a flotation concentrator it is assumed that many of the sulphide minerals will be recovered in the flotation concentrates that are sold. However, in a facility that contains a large number of cleaner flotation circuits, such as the proposed Schaft Creek concentrator, it is possible that a significant amount of pyrite and other sulphide minerals that are not of economic value will be rejected from the final concentrate in order to increase the concentrate grade and thereby improve its marketability and minimize transport costs. The residual sulphide minerals in the tailings are likely to be acid-generating and they must also be managed to mitigate the generation of the acid. Again, the tailings must be mixed with an excess of lime or other neutralizing materials and/or isolated from oxygen by capping with an impervious material or submersion under water in order to stop the generation of acid.
- The primary risk of acid generation is the potential to affect streams in the area and to kill the fish that may be living in them by exposure to sulphuric acid and the heavy metals that are released during the oxidation process. Once an ARD problem develops at a mine, the water must be treated in perpetuity in order to mitigate the problems. This is expensive. It also requires on-going operations and particular attention to the disposal of sludge and other materials that are byproducts of the water treatment operations.
- Other risks include the disruption of wildlife in the area of the mine and the potential for creating a long-term impact on the area.
- The Environmental Assessment (EA) that is part of the permitting process will ensure that all possible risks to the environment are identified and that mitigating steps are implemented into the design and operation of the Schaft Creek project.
- This, as well as the experience of the design engineers and the project and operations managers, will enable the Schaft Creek project to be designed and

operated such that environmental, reclamation and closure risks are both identified and resolved prior to the development of significant problems.

21.4 Opportunities

21.4.1 On-Site Power Generation

- There are several watersheds on the Schaft Creek property that may provide viable opportunities to utilize the elevation difference between the headwaters and the confluence points with other streams for renewable, zero emission power generation. As an example, Hydro microturbines with a low head range up to 30 m are available that generate up to 10 MW of clean, green power with minimal disruption to the landscape and wildlife. Unlike hydroelectric dams, the water is simply diverted through the microturbines on its journey down the mountainside without retention and subsequent lake formation. Since the water simply passes through the turbines and continues on, these technologies may be staged (stacked) in order to take advantage of increased elevation changes and double or triple power production all with the same volumetric flow of water.
- Another opportunity for environmentally friendly onsite power generation is the utilization of solar conversion technologies. Recent advances in Building Integrated Photovoltaic (BIPV) technologies have enabled many of the surfaces already present (siding, windows, etc.) to become silent, clean power generators. For example, there are BIPV technologies that allow a window to retain its translucence but generate power when illuminated by sunlight.
- Numerous other opportunities exist to recover waste heat and turn it into usable energy and thereby further reduce the demand of grid-supplied power and mitigate the power supply risk.
- Wind power generation may offer another option for this site.

21.4.2 Other Opportunities

Other opportunities that have been identified, and to some extent investigated, include:

- High Pressure Grinding Rolls (HPGR) to replace the SAG mills;
- On-site treatment of copper concentrates;
- Cost sharing of infrastructure with other developing projects in the area;
- Paste tailings to reduce the wet tailings footprint;
- Alternative mill site locations;
- Recognizing rhenium as a revenue stream.

22.0 Recommendations

22.1 Recommendations

22.1.1 Mining

- Orders should be placed for long delivery equipment items such as trucks and shovels.
- The results from 2008 exploration drill information to improve the ore classification will need to be added to the drill hole data base and the resource model rebuilt. The 2007 holes can be added as well and the economic pit limits, detailed pit designs, productions schedules, and mining costs updated at the Feasibility Study level.
- Detailed Hydro-Geology evaluation of the area is needed to improve the accuracy of pit dewatering design. Vertical dewatering wells have not been included as part of the required PFS activities. Results from the 2008 field program may show this to be a future requirement to lower the water table within the pit prior to mining. Although not included in the current estimate, it may be necessary to prevent water inflow from the nearby glaciers through the glacial sediment. The 2008 field program needs to identify whether this is an issue that needs to be addressed so Engineering can proceed to mitigate any issues. It is recommended that a plan for the location of the pit water discharge be included in future detailed studies as well.
- Combined with the Hydro-Geology evaluation, a Hydrology assessment is needed so that the diversion and water management plan can be developed for the mining area considering the combined surface and groundwater quantities.
- A 3-dimensional Acid Base Accounting (ABA) Geology model is required to classify waste geochemistry. The spatial distribution of PAG and NPAG rock might significantly alter the haulage from the mining phases and sequences, waste destination requirements and ultimately haulage costs. For this study, the potentially acid generating (PAG) material is being estimated at ten percent of total material mined. Only after the appropriate Acid Based Accounting (ABA) geochemistry model and kinematic test are completed can the distribution of PAG and non-PAG material be incorporated into the geological block model. The quantity and spatial distribution of PAG material will directly affect the truck haulage hours for this material, but at this time it is not known the extent of this impact.
- 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.
- As the project infrastructure locations become more defined a Geohazard assessment is required including snow and avalanche loss control programs.
- More detailed geotechnical data will require redesign of the pit slopes.
- Foundation testing of the waste dump sites will generate foundation preparation requirement for the waste dumps.
- A detailed blasting study is recommended for more advanced project studies. This will assist in determining the most applicable powder factor for the rock types present at Schaft Creek. 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. 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.

- It is recommended that further optimization of the shovel fleet be completed in more advanced studies. Specifically, there are many years where half of the large shovel's production capability is not being used. These optimization studies should evaluate the use of a smaller shovel that is capable of loading the large trucks. Also, optimization studies should evaluate whether the use of a large rubber-tired front end loader would provide economical benefits to the operation, perhaps providing flexibility and mobility that the electric cable shovel can not provide.
- It is recommended that further optimization of the haulage fleet be completed in more advanced studies. In this study, it is assumed that the large off-highway haul trucks are used for all mining requirements. However, there is the potential to use a smaller-sized shovel-truck fleet for such specific activities as the opening up of upper benches where the initial space for mining is limited.
- A detailed drill study is recommended for more advanced project studies. This will help determine the penetration rate that can be expected for the selected drills and the specific rock types that exist within the pit area.
- Waste characterization and water quality prediction work must be carried out in future studies and will be central in determining the actual PAG rock handling requirements.

22.1.2 Metallurgy

- Metallurgical testing should continue to more definitively define the separation characteristics of the Schaft Creek resource.
- If the current HPGR tests indicate that the Schaft Creek resource is amenable to the technology, a more advanced phase of testing should be conducted in order to size the equipment. Products from the HPGR test should also be returned to a metallurgical testing lab to determine if there is any metallurgical benefit if the HPGR technology is used for comminution.
- Additional locked-cycle tests are required using samples of varying copper grades from selected areas in the Schaft Creek resource in order to provide additional data to more accurately predict the metal recoveries from all relevant areas of the resource.
- Additional molybdenum separation tests should be conducted to optimize the molybdenum separation parameters.
- This will require running a flotation pilot plant in order to produce sufficient quantities of bulk concentrate to use for bench-scale molybdenum separation tests. At least 6 t of feed material will be required for the pilot plant.
- Additional samples should be collected from the pilot plant for environmental studies. Samples of the pilot plant tailings should be sent to vendors for paste thickener tests. Also, sedimentation tests should be conducted on tailings and concentrate samples. Filtration tests should be conducted on the copper and molybdenum concentrates.
- Marketing samples of copper concentrate and molybdenum concentrate should be produced from the pilot plant and molybdenum separation tests.
- Additional mineralogical studies should be conducted to assist in understanding the metallurgical treatment characteristics of the Schaft Creek resource.
- It may be necessary to drill additional PQ metallurgical drill holes to test specific areas of the relevant mineral zones.

- Testing of additional samples to determine the hardness is required in order to assure that the grinding circuit is designed to accommodate the material that will be processed from the Schaft Creek resource. This is particularly important given the extreme hardness of the samples that have been tested to date.
- The results of the recent metallurgical testing continue to indicate that the Schaft Creek resource is amenable to the typical conventional flotation methods utilized for copper/molybdenum porphyry deposits.
- The results indicate that high-grade copper flotation concentrates can be achieved due to the presence of secondary copper minerals such as bornite, covellite and chalcocite. Secondary mineralization appears to be pervasive throughout the resource and occurs even in low-grade areas. Copper concentrates containing 30 to 35% copper are achievable at the average resource grade of 0.30 to 0.35% copper. Copper recoveries of approximately 88 to 90% can be expected at these flotation concentrate grades.
- Molybdenum occurs throughout the resource and can be recovered into a saleable concentrate containing 50% molybdenum at an overall recovery of approximately 68% to the molybdenum concentrate. The molybdenum concentrates should contain approximately 368 ppm Rhenium. The mineralogical evaluation of molybdenum concentrates indicates that it should be possible to increase the molybdenum concentrate grade to approximately 54% molybdenum. However, additional studies are needed to optimize the molybdenum separation circuit and achieve these results.
- Gold and silver are also pervasive and are recovered with the copper in the copper flotation concentrate. Gold recoveries of approximately 80% appear to be achievable while silver recovery is approximately 72%. Gold and silver grades in the copper concentrate appear to average approximately 22.4 and 134.2 g/t respectively. The most effective way to maximize the recovery of the by-product metals is to maximize the recovery of copper.
- The copper concentrate produced during the molybdenum separation tests indicate levels of antimony (228 ppm), arsenic (30 ppm), bismuth (238 ppm), lead (0.04 ppm) and zinc (0.03 ppm) that should not result in penalty charges by the smelters that will further process the copper concentrate.

22.1.3 Tailings Storage Facility

- Site-specific hydro-meteorological data are necessary in order to complete feasibility-level design.
  The current design is based on a number of assumptions that significantly influence the water balance for the facility. It is recommended that a detailed hydrometeorology study be completed using the available data collected on site in conjunction with long-term regional records in order to provide estimates of average long-term conditions at the project site. The study should specifically provide water-balance modeling inputs. This study is expected to cost approximately $40,000.
- Geotechnical conditions in the vicinity of the Tailings Storage Facility and Waste Dumps will have a significant influence on the feasibility design and estimated costs, particularly with respect to seepage control measures and construction materials.
- There is an opportunity to significantly reduce the capital costs by identifying sources of low permeability soils for embankment construction and better defining the depth

to bedrock along the embankment footprints. A geotechnical site investigation to support the feasibility design is underway at the time of writing of this report.

- A site-wide water balance model should be completed for the project. This will require detailed input from a Hydrometeorology Study. The water balance model should be capable of tracking water quality parameters in the various flows. It is estimated that developing a detailed water balance model, complete with water quality tracking, will cost approximately $100,000.
- It is recommended that a waste characterization study be completed to better define the quantity of Potentially Acid Generating (PAG) waste rock, the variation in reactivity within the PAG component of the waste rock and recommendations for treatment and storage of the waste rock. A complete waste characterization study, including laboratory testing, is expected to cost approximately $500,000.

22.1.4 Geohazard Assessment

This section of the report provides a preliminary assessment of surficial geology and geohazards for the Mess Creek access route. It is based on Phase 1 of a geohazards and terrain stability investigation program for the Schaft Creek project. The following work is recommended for Phase 2 of geohazard assessment for the Schaft Creek project.

22.1.4.1 *Access Route*

- Field checking to verify terrain stability mapping conducted for Phase 1;
- Addition of potential sediment delivery rating to polygons intersecting the proposed access road (rating of the potential for surface erosion to transport sediment to valley bottom streams);
- Linear geohazard assessment of the proposed access road;
- Terrain stability field assessment (TSFA) including a geotechnical review of the design for road sections with cut and fill slopes ≥ 5 m high in soil[2];
- Preparation of an Avalanche Locator Map (CAA 2002);
- Review of selected sections of the geometric design for the proposed access road;
- Estimation of design flows at channel crossings;
- Identification of geohazard mitigation options;
- Identification of road sections that will require more detailed investigations and/or supervision by a qualified registered professional during construction.

22.1.4.2 *Mine Site and Tailings*

- Expansion of the terrain stability and geohazard mapping study area to include the minesite;
- Field assessment of glacier outburst flood hazard;
- Provision of laser photocopies of terrain-mapped air photos to use as a basis for the surficial geological component of Terrestrial Ecosystem Mapping (TEM);
- Field checking to verify terrain stability mapping and geohazards interpretations;
- Preparation of an Avalanche Atlas (CAA 2002);

[2] To be completed in close communication with McElhanney Consulting Services Ltd. who are responsible for design of the Mess Creek Access Road.

- Overview description of landslide and snow avalanche geohazards with the potential to affect proposed facilities;
- Identification of geohazard mitigation options;
- Identification of any requirements for landslide instrumentation.

22.1.5 Other Recommendations

- Given the local weather conditions at the Schaft Creek site and the dependence of the project on reliable airlift of personnel and critical supplies, it is recommended that a study of the project site be conducted in order to identify the most suitable location for the proposed airfield.
- It is recommended that a cost of risk analysis (CORA) study be conducted to include projected capital cost expenditures and operating cost expenditures together with the cost and likelihood of operational disruptions for each logistical support approach.

23.0 List of References

23.1 References

BGC Engineering Incorporated (5 February 2008) Project Memo: *Schaft Creek Tailings Options Study, Geohazards*

Canadian Dam Association (2007), *Dam Safety Guidelines*

Coulson, C.H. and Obedkoff, W. 1998, *British Columbia Streamflow Inventory*, Water Inventory Section, Resources Inventory Branch, B.C. Ministry of Environment, Lands, and Parks. Victoria, B.C.

DST Consulting Engineers (25 January, 2008), *Preliminary Assessment of Waste Rock Disposal, Schaft Creek Project, British Columbia*

DST Consulting Engineers (27 February, 2008), *Pre-feasibility Geotechnical Assessment of Tailings Dam Options, Schaft Creek Project, B.C.*

Giroux, G., 2004, *Summary Report, Status and Resource Estimate, Schaft Creek Property, Northwestern British Columbia*, 96 pp

Giroux, G., and Ostensoe, E., 2004, *Summary Report, Status and Resource Estimate, Schaft Creek Property, Northwestern British Columbia*, Report prepared for 955528 Alberta Ltd., 96 pp

Hershfield, D.M. (1977), *Some Tools for Hydrometeorologists,* 2nd Conference on Hydrometeorology, American Meteorological Society. pp. 79-82.

Hogg, W.D., Carr, D.A. (1985), *Rainfall Frequency Atlas for Canada*, Canadian Climate Program. pp. A-1 to A-12.

Howes, D.E. and E. Kenk. (eds.) 1997, *Terrain Classification System for British Columbia, Version 2. A system for the classification of surficial materials, landforms and geological processes of British Columbia*, Resource Inventory Branch, Ministry of Environment, Lands and Parks, Province of B.C. Victoria, B.C. 100pp

ICOLD – International Commission on Large Dams (1995), *Tailings Dams and Seismicity: Review and Recommendations*, Bulletin 98.

Koch, J., 2006, *Holocene glacier fluctuations in Garibaldi Provincial Park, southern coast mountains, British Columbia*, PhD. Thesis, Simon Fraser University, 200 pp

Logan, J., Drobe., J., Koyanagi, V., and Elsby, D., 1997, *Geology of the Forest Kerr – Mess Creek Area, Northwestern British Columbia (104B/10, 15, & 104G/2 & 7W)*, Ministry of Employment and Investment, Geoscience Map 1997-3, 1:100,000 scale

McCuaig, S., and Roberts, C., 2002, *Topographically-independent ice flow in northwestern British Columbia: implications for Cordilleran Ice Sheet reconstruction*, Journal of Quaternary Science 17 (4), p. 341-348

Ministry of Forests, 1999, *Mapping and Assessing Terrain Stability Guidebook*, Website URL: www.for.gov.bc.ca/tasb/legsregs/fpc/fpcguide/terrain/index

Moose Mountain Technical Services (19 March, 2008), File Note Re: *Draft of Tailings Site Evaluation*

Obedkoff (June 2001) Water Inventory Section, Resource Inventory Branch, *Streamflow in the Skeena Region*

Rescan Tahltan Environmental Consultants Inc., 2006, *Schaft Creek Project 2006 Hydrology Baseline Report*, Report prepared for Copper Fox Metals Inc., 57 pp

Rescan Tahltan Environmental Group (March 2007), Schaft Creek Project, Meteorology Baseline Report

Resource Inventory Committee, 1996, *Guidelines and Standards to Terrain Mapping in British Columbia*, Guidelines prepared by the Resource Inventory Committee Surficial Geology Task Group. 131 pp

Samuel Engineering Incorporated (7 December, 2007), *Preliminary Economic Assessment on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada*

Spooner, I., 2007, *A preliminary Assessment of Mass Wasting and Glacial Outburst Flood Risk Report,* prepared July 2007 for Copper Fox Metals Inc. 27 pp.

Spooner, I., 2007b, *Airphoto Interpretation of the Schaft Creek – Mt. LaCasse Region. Reportand map,* prepared August 2007 for Copper Fox Metals Inc. 9 pp

Stumpf, A., Broster, B., and Levson, V., 2000, *Multiphase flow of the late Wisconsinan Cordilleran ice sheet in western Canada*, GSA Bulletin, 112 (12), p. 1850-1863.

Thompson, D., Van Hollen, D., Osborn, G., Ryane, C., Menounos, B., Clague, J., Koch, J., Riedel, J., Scout, K., and Reasoner, M., 2006, *Did Neoglaciation begin as early as 6400 years ago?,* Geological Society of America Abstracts with Programs, 38 (7), p. 236

World Meteorological Organization, *Manual for Estimation of Probable Maximum Precipitation, Operational Hydrology Report 1*, WMO No. 332, 2nd ed., Secretariat of the World Meteorological Organization, Geneva, Switzerland, 1986.

23.2 List of Acronyms

23.3 Glossary

Acid-Base Accounting (ABA): Test methods and calculations that predict the balance between acid generating potential and acid neutralizing potential of materials, particularly mine waste materials.

Acid Generating Material (AGM): Materials that react with water and oxygen to form acids and to mobilize metals into the resulting solutions. An example is mine tailings containing sulfides (*ex.* Pyrite) that react to form sulfuric acid.

Acid Generating Potential (AGP): Test methods and calculations used to measure of the potential of material to become acid generating material.

Acid Neutralizing Potential (ANP): Test methods and calculations used to measure of the potential of material to neutralize acid generating material.

Acid Rock Drainage (ARD): A natural occurrence 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**).

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. The mineralogical abbreviation is bn.

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: $CuFeS_2$. The mineralogical abbreviation is cp.

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.

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 occurring destructive forces such as volcanoes, earthquakes, landslides, and avalanches.

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.

Graben: A depressed block of land bordered by parallel faults.

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.

NPAG: Non-Potentially Acid Generating

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.

PAG: Potentially Acid Generating.

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.

Pyrite: The mineral pyrite, or iron pyrite, is an iron sulfide with the formula FeS_2. This mineral's metallic luster and pale-to-normal, brass-yellow hue have earned it the nickname fool's gold due to its resemblance to gold. Pyrite is the most common of the sulfide minerals. The mineralogical abbreviation is py.

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.

23.4 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

Power	Prefix	Symbol	Decimal Equivalent (in SI Writing Style)
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

23.5 List of Abbrviations

Above mean sea level .... amsl
Ampere .... A
Annum (year) .... a
Bank cubic metre .... BCM
Copper .... Cu
Cubic metre .... m^3
Cubic metres per day .... m^3/d
Cubic metres per hour .... m^3/h
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
Gold .... Au
Gram .... g
Grams per cubic centimetre .... g/cc
Grams per litre .... g/L
Grams per tonne .... g/t
Greater than .... >
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 .... in

Joule (Newton-metre) .... J
Kilometre .... km
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
Lead .... Pb
Less than .... <
Litre L
Litres per day .... L/d
Litres per minute .... L/m
Litres per second .... L/s
Loose cubic metres .... LCM
Megawatt .... MW
Megabytes per second .... Mb/s
Metre .... m
Metres above sea level .... masl
Metres per second .... m/s
Metric tonne .... t
Metric tonne .... mt
Micrometre .... micron
Microsiemens (electrical) .... µS
Miles per hour .... mph
Million .... M
Million metric tonnes (megatonne) .... mmt
Minute (plane angle) .... '
Minute (time) .... min
Molybdenum .... Mo
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
Pound (avoirdupois) .... lb
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
Silver .... Ag
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
Uranium .... U
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) .... yr

24.0 Date and Signature Pages

The Certificates of Qualification for Mr. Matt Bender and Mr. Keith McCandlish, the Qualified Persons responsible for this Technical Report, can be found at the end of the document.

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

25.1 Mining Plan

25.1.1 Summary

A production schedule based on 100,000 tpd mill feed schedule at a preliminary feasibility study (PFS) level is developed for the Schaft Creek mine. Detailed pit phases are engineered from the results of a Lerchs-Grossman (LG) sensitivity analysis. Pit reserves are tabulated below. The pit reserves in the table below include a 10% mining dilution applied at the contact between ore and waste and 5% Mining Loss. Dilution grades are estimated at 4.63 $/t NSR, 0.076% Cu, 0.088 g/t Au, 1.76 g/t Ag and 0.005% Mo representing the average grade of material below the incremental cut-off grade.

Cut-off grade for the Phase reserves in Table 25.1 is $5.05/t Net Smelter Return (NSR).

Table 25.1
Summarized Measured, Indicated Reserves for Schaft Creek

Pit	Description	ROM ORE (Mt)	ROM Diluted Grades					Waste (kt)	Stripping Ratio (t/t)
			NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)		
P611	South Starter	297.8	16.7	0.330	0.239	1.685	0.017	241.2	0.81
P621i	South Incremental	136.1	14.3	0.266	0.207	1.509	0.018	178.6	1.31
P631	North Starter	52.9	16.2	0.291	0.185	1.940	0.024	45.4	0.86
P641i	Intermediate	208.1	14.3	0.277	0.148	1.674	0.020	476.1	2.29
P651i	Final	125.9	18.1	0.305	0.263	2.275	0.027	601.9	4.78
Total		**821.1**	**15.9**	**0.299**	**0.211**	**1.760**	**0.020**	**1,543.2**	**1.88**

Proven and Probable Reserves at Schaft Creek are summarized Table 25.2

Table 25.2
Proven and Probable Reserves at Schaft Creek

Reserve Category	ROM ORE (Mt)	ROM Diluted Grades				
		NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)
Proven	411.1	16.6	0.316	0.236	1.722	0.019
Probable	409.9	15.2	0.283	0.186	1.798	0.020
Total	**821.1**	**15.9**	**0.299**	**0.211**	**1.760**	**0.020**

25.1.2 Introduction

The mine planning work for the Preliminary Feasibility Study (PFS) is based on the Resource model provided by Associated Geosciences Ltd. (AGL) reported in the Technical Report dated June 22, 2007. 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 parameters, mining cost data derived from supplier estimates and data from other projects in the local area, recommended pit slope angles, and projected project metallurgical recoveries, plant costs and throughput rates.

25.1.3 Project Production Rate Consideration

A number of factors are considered in establishing an appropriate mining and processing rate, the key ones are discussed below in relation to Schaft Creek:

- Resource size: Typically mine life is set at 12.5 to 20 years; as for anything beyond this, time value discounting shows insignificant contribution to Net Present Value (NPV) of the project and capital investment typically is targeted at projects with payback of 3 to 5 years.
- Unit Capacity: Generally, unit operating costs are lower using the largest possible equipment for a single train in the mill. In the case of Schaft Creek this is two 11.58 m x 6.10 m (38 ft x 20 ft) SAG Mills (17 MW) followed by two 7.32 m x 12.19 m (24 ft x 40 ft) Ball Mills (12 MW). Depending on ore hardness, and considering a single primary mill, throughputs of up to 100,000 tpd are possible depending on final grind size selection.
- Operational Constraints: Power, water or resources for operations support can limit production.
- Construction Constraints: Physical size and weight of equipment and shipping limits can determine the maximum size of available units.
- Project Financial Performance: Generally, economies of scale can be realized at higher production rates and lead to reduced unit operating costs. These are tempered to the above mentioned physical and operational constraints and generally higher capital requirements for higher tonnage throughputs.
- Higher production rates generally pay back fixed capital at a faster rate, thereby improving project NPV.

A throughput of 100,000 tpd sets the mine life is set at 23 years for the estimated 821 Mt pit reserve. Project NPV may be improved by increasing the mill throughput above 100,000 tpd, but this would require the re-engineering of mining phases to provide sufficient working bench widths for additional mine equipment.

25.1.3.1 *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 onsite 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 offsite transportation, smelting, and refining charges, etc. The metal prices and resultant NSPs used are shown in Table 25.3.

Table 25.3 3D Block Model Setup			
	Metal Price ($)	**NSP ($)**	**Recovery, %**
Cu	1.93 $/lb	1.49 $/lb	89.2%
Au	658 $/ozt	17.51 $/g	80.7%
Ag	10.9 $/ozt	0.26 $/g	72.0%
Mo	14.7 $/lb	12.05 $/lb	68.6%

The NSR formula is:

$$NSR = \frac{Cu\%}{100} * \frac{Rec_{Cu}}{100} * NSP_{Cu} * 2204.6 + Au\left(\frac{g}{t}\right) * \frac{Rec_{Au}}{100} * NSP_{Au} + Ag\left(\frac{g}{t}\right) * \frac{Rec_{Ag}}{100} * NSP_{Ag} + \frac{Mo\%}{100} * \frac{Rec_{Mo}}{100} * NSP_{Mo} * 2204.6$$

$$NSR = Cu\% * 0.892 * \frac{\$1.49}{lb} * 22.046 + Au\left(\frac{g}{t}\right) * 0.807 * \frac{\$17.51}{g} + Ag\left(\frac{g}{t}\right) * 0.72 * \frac{\$0.26}{g} + Mo\% * 0.686 * \frac{\$12.05}{lb} * 22.046$$

25.1.3.2 *Mining Loss and Dilution*

The Schaft Creek deposits are to be mined with large truck/shovel operations, and an ore mining rate of 100,000 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 parameters used in the reserves calculations are also a NI 43-101 reporting requirement.

The 3D Block Model (3DBM) for Schaft Creek 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 since the larger blocks in the 3D block model will average in the isolated blocks from any future blast hole models.

The 3DBM uses 25 m X 25 m X 15 m 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 4 blocks per day. With blocks of this magnitude, it can be assumed in the PFS planning, that isolated blocks from the larger 3DBM will be selectively mined on a full block basis 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 neighbouring 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 reserve is calculated from the Resource model, within an economic pit limit using the applicable mining recovery and dilution parameters.

The mining recovery and dilution parameters, in effect, convert the inplace “pit delineated resource” to ROM reserve 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 Parameters 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 feasibility study an allowance has been made for a mining dilution of 10% applied at the contact between ore and waste dilution and a 5% 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 $5.05/t. The dilution grade is estimated at 4.63 $/t NSR, 0.076% Cu, 0.088 g/t Au, 1.76 g/t Ag and 0.005% Mo representing the average grade of material below the incremental cut-off grade.

25.1.4 Economic Pit Limits, Pit Designs

25.1.4.1 *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 smaller high grade pits versus larger lower grade or higher strip ratio pit shells (by varying the Net Smelter Price) and also different slope angles Time value block discounting is also evaluated to determine the NPV effect of the delay between earlier stripping costs to the revenue released from deeper ore.

Pit slopes design parameters are derived from Knight Piésold Preliminary Feasibility Pit Slope Design report of 16 April 2008. Mining costs from the PEA are used. Metallurgical recovery assumptions supplied by Samuel Engineering.

25.1.4.2 *Pit Slopes*

The current geotechnical model incorporates the inferred distribution for the three major geological domains encountered: Overburden, Stuhini Group Volcanics, and Intrusives. Intact rock strengths are generally found to be strong and the rock mass quality is typically FAIR to POOR. Large-scale structural features across the deposit area occur as sets of steeply dipping North-South and East-West trending faults. Discontinuities measured on outcrops across the deposit area reflect the major structural trends shown in the fault system.

This preliminary geotechnical database has been utilized to develop recommendations for pit slope design. Four major pit design sectors, Northeast, Southeast, Southwest and Northwest, were defined based on the spatial distribution of the major structures and the slope orientations. Design methods used to determine appropriate pit slope angles for the Schaft Creek Pit included kinematic stability analyses using stereographic methods and evaluation of the overall stability of the rock mass using limit equilibrium techniques. Recommended pit slope geometries for each design sector have been determined based on minimum acceptable criteria for each of these design methods.

The bench scale slope stability has been assessed using stereographic analyses. Bench geometry is selected to reduce the potential of unstable blocks and wedges from being formed by predominant small-scale structural features. A 30 m high double-bench pit wall is recommended for the Schaft Creek Pit with a bench face angle of 65 degrees due to the fractured nature of the rock. A single bench configuration is considered to be more appropriate for the Lower Southeast Wall where the broken Intrusive rocks are exposed.

The maximum inter-ramp slope angle is typically controlled by bench geometry and/or any large scale structural features such as faults, shear zones and bedding. An inter-ramp slope angle of 45 degrees is recommended for the Southwest and Northwest Sectors where adverse structural features are less significant. An inter-ramp slope angle of 43 degrees is recommended for the Northeast and Southeast Sectors to reduce the potential for wedge failure in the Northeast Wall and toppling failure in the Southeast Wall.

The overall stability of the pit slopes has been evaluated using conventional limit equilibrium analyses. The overall slope angles have been determined to achieve a minimum factor of safety of 1.3 for each design sector based on the assumptions of blasting disturbance and groundwater pressure. It is recognized that the proposed Northeast Wall will reach a maximum slope height of 1,200 m. Some slope degradation resulting in localized ravelling is anticipated due to stress relaxation and the corresponding slope deformation during ongoing pit development.

Additional wider cleanout benches should be placed along the upper third and the lower third of the Northeast Wall and across the upper Southeast Wall to capture ravelling rock debris and to allow for access and removal. The resulting overall slope angle for the Northeast and Southeast Walls will be approximately 40 degrees after allowing for the cleanout benches. A 44 degrees overall pit slope is appropriate for the remaining walls where the maximum slope height is less significant. Carefully controlled blasting practices and sufficient groundwater depressurization measures are also recommended for the final pit wall development.

The overall pit slopes recommended by Knight Piésold are summarized in Table 25.4.

Table 25.4
Knight Piésold Recommended Overall Pit Slope

Pit Design Sector	Nominal Pit Wall Dip Direction	Recommended Overall Pit Slope
Northeast	250	40
Southeast	270	40
Southwest	60	44
Northwest	70	44

25.1.4.3 *Mining Costs*

Mining unit costs have been estimated in the PEA and are used as the bases costs for the PFS LG runs. The unit mining cost assumptions are shown in Table 25.5.

Table 25.5
Economic Pit Limit Estimated Unit Mining Cost

	$/t
Drilling	0.09
Blasting	0.23
Loading	0.11
Hauling	0.58
Mine Maintenance	0.02

Table 25.5 Economic Pit Limit Estimated Unit Mining Cost	
Mine Operations - Support	0.26
Snow Removal	0.02
Geotechnical	0.02
Unallocated Labour Cost	0.01
Direct Costs - Subtotals	1.36
Mine Ops Salaried Labour Costs	0.03
Mine Maintenance Salaried Labour Costs	0.04
Mine Engineering Salaried Labour Costs	0.02
Technical Services Salaried Labour Costs	0.02
TOTAL General Mine Expense Costs	0.11
Total Mine Operating Cost	**1.47**

25.1.4.4 *Economic Pit Limits*

The slope sensitivity graph in Figure 25.1 shows pit resource sensitivity to revenue case for the Schaft Creek pit area using the above design criteria. A cutoff grade of $5.05/t NSR is assumed, and Ore is estimated as in-situ.

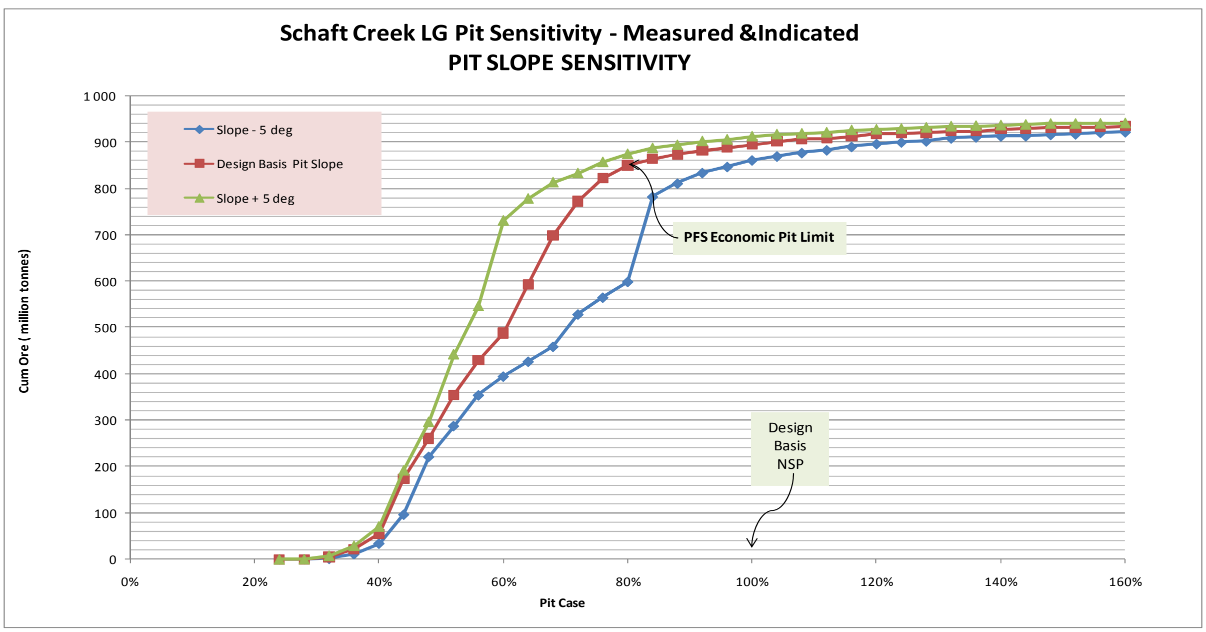


Figure 25.1 Economic Pit Limit Overall Pit Slope Sensitivity

This slope sensitivity graph shows that there is insignificant change to Economic Pit Limit Ore by increasing or decreasing the overall pit slope by 5 degrees. The selection of Pit Case would change with slope assumption changes, but the quantity of the Economic Pit Limit Ore would remain the same.

The economic pit limit for the Schaft Creek Pit is the 80% Revenue case (Pit 15). There is a major inflection just before the 80% Pit Case and no significant increase in pit size at incrementally higher NSP's. A suitably sized starter pit is evident at the 44% case (Pit 06).

The graph in Figure 25.2 compares the Economic Pit Limit selection for PFS and with the PEA (the PEA study used measured, indicated, and inferred ore)

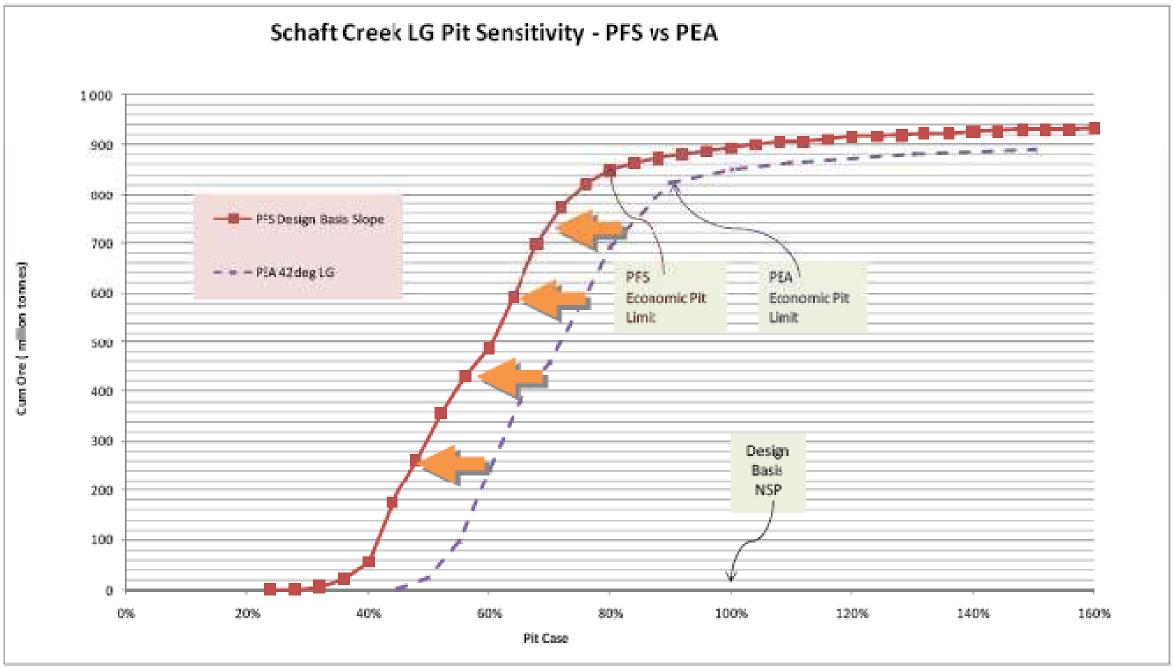


Figure 25.2 PEA Economic Pit Limit vs. PFS Economic Pit Limit

The graph above shows that the PFS Economic Pit Limit Ore is slightly more than the PEA Economic Pit Limit Ore. The PFS curve has shifted left due to:

- Increased metal price assumptions
- Reduced average mining cost assumption ($1.49/t for PFS vs $1.61/t for PEA)

The effect of deeper versus shallow ore on NPV is evaluated with time value discounting where the revenues of deeper ore are later in time than the waste stripping cost to release it. The absolute NPV value for any mining is not known since the years that any blocks are mined is not known at this stage of planning. However the LG economic limit stops mining when the last 'skin' of the expanding pit breaks even so a relative NPV will suffice. If it is assumed that any incremental blocks will be included in a valid mineable pit phase, then the relative effect of time delay of revenues versus costs can be estimated in the LG by including the discount factor in each block. In these runs it is assumed that the phases will be designed to mine 12 benches per year (sinking rate) and a discount rate of 6% per annum is used.

25.1.5 Detailed Pit Designs

MMTS has completed preliminary feasibility 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, recommended geotechnical parameters, regulated standards for road widths, and minimum mining widths based on efficient operation for the size of mining equipment chosen for the project.

25.1.5.1 *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.

25.1.5.2 *Design Standards*

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 PFS, minimum mining width generally conforms to 50 m, 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 to handle material over the crest. Full Truck/shovel excavation of the dozer or trammed material will be done as rehandle 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.

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 more than twice the width of the widest haul vehicle. Two-way roads require a running surface more than 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 the ramp will be removed upon retreat.

Road grades are designed at a maximum grade of 10% to facilitate continuous operations through winter months. 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 feasibility study, 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 stripping. 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 requires 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 not be optimized if ramps are not designed into the east wall. However in the case of Schaft Creek the highwall has been reduced to 40 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 phases it may be necessary to leave a highwall ramp in the upper benches of the phase in order to gain access to intermediate pit phases. These intermediate highwall ramps may not be needed when the final pit phase is completed.

Variable Berm Width

As mentioned in the preceding section, pit designs for Schaft Creek are designed honouring overall pit slope angles, a nominal bench face angle (65°) and variable safety berm widths with a minimum 8 m width. Due to the low overall pit slope angles, berm widths are generally greater than 15 m. 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.

Bench Height

The Schaft Creek pit designs are based on the digging height of the large shovels. A 15 m operating bench with double benching between highwall. Therefore the berms are separated vertically by 30 m. 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.

25.1.5.3 *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 geo-technical recommendations.

The 80% price case LG pit 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 the multiple unit mining operations of drilling, blasting, loading, and hauling
- Bench sizes large enough so the number of benches mined per year is reasonable (sinking rate).
- Not be too narrow so the shovels can load the trucks 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 56% price case LG Pit (Pit 09) in Figure 25.3 illustrates the two separate starter pit areas, one in the North and the other in the South.

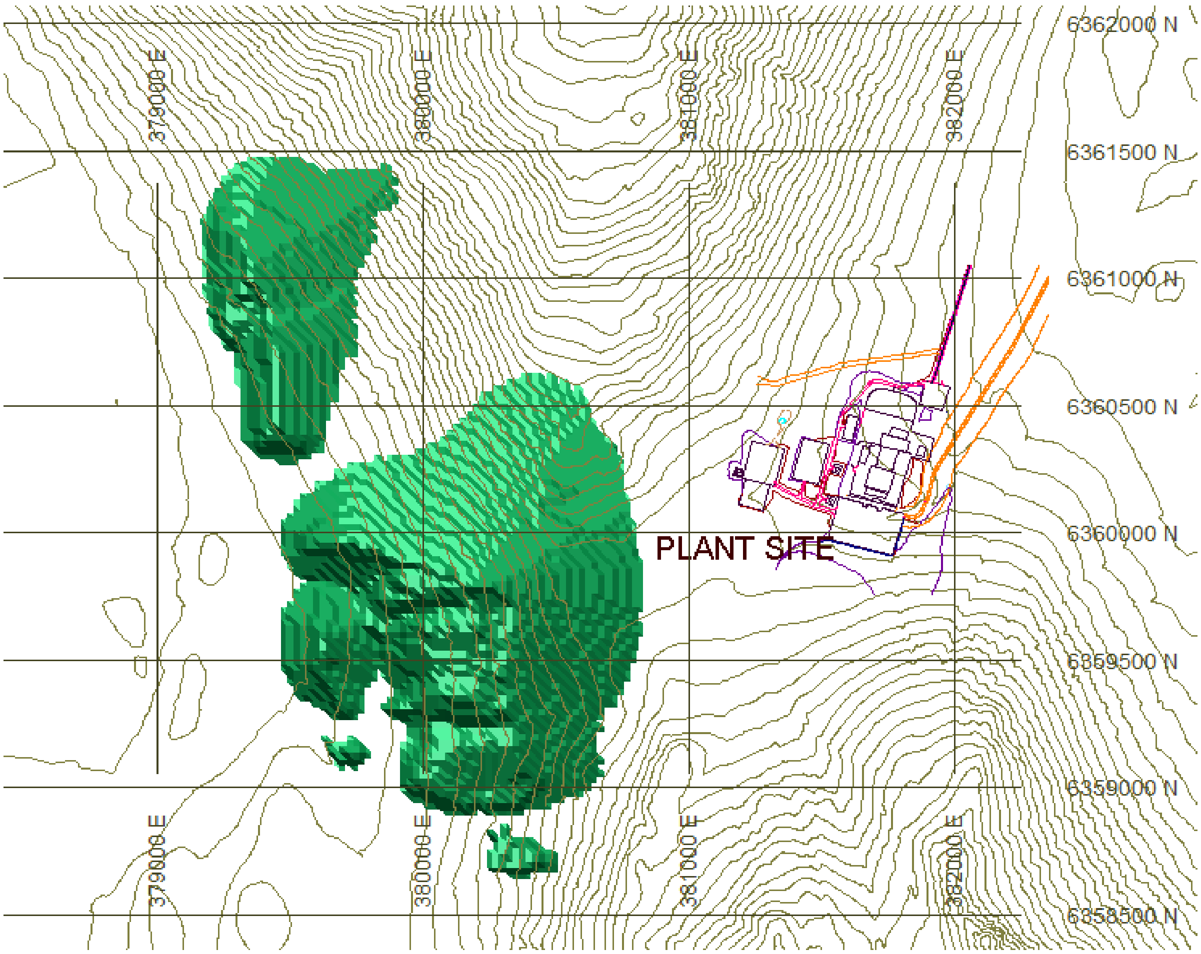


Figure 25.3 Incremental Pushback LG Pit (Pit 09) - Plan View Still Showing Separate North and South Pit Areas

Ultimate Pit

The ultimate pit is guided by LG Pit 15 (80% price case). Ramps are left in the highwall for access to the mid and upper benches and to help achieve the design basis overall pit slope.

The plan view of the Ultimate Pit P651i is shown in Figure 25.4.

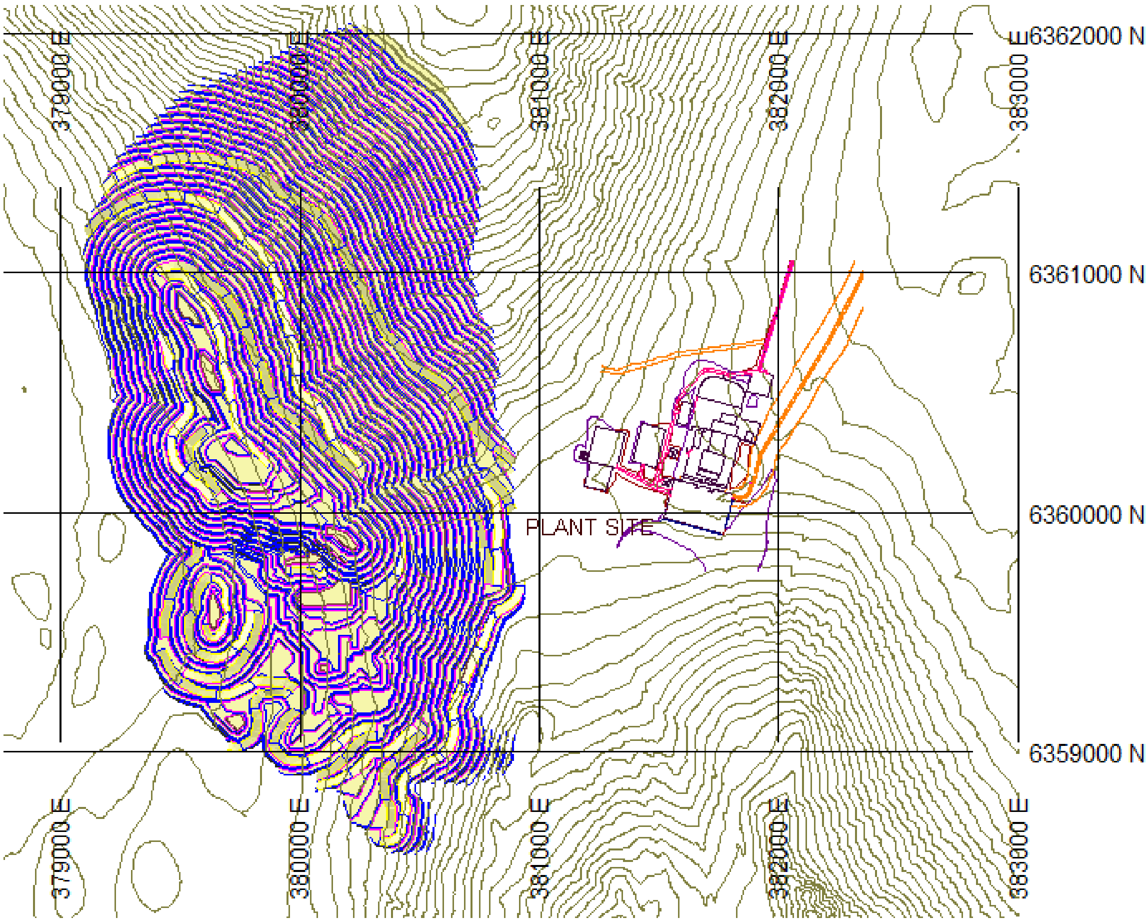


Figure 25.4 Plan View of the Ultimate Pit P651i

All the pit phases are shown in Figure 25.5 and Figure 25.6.

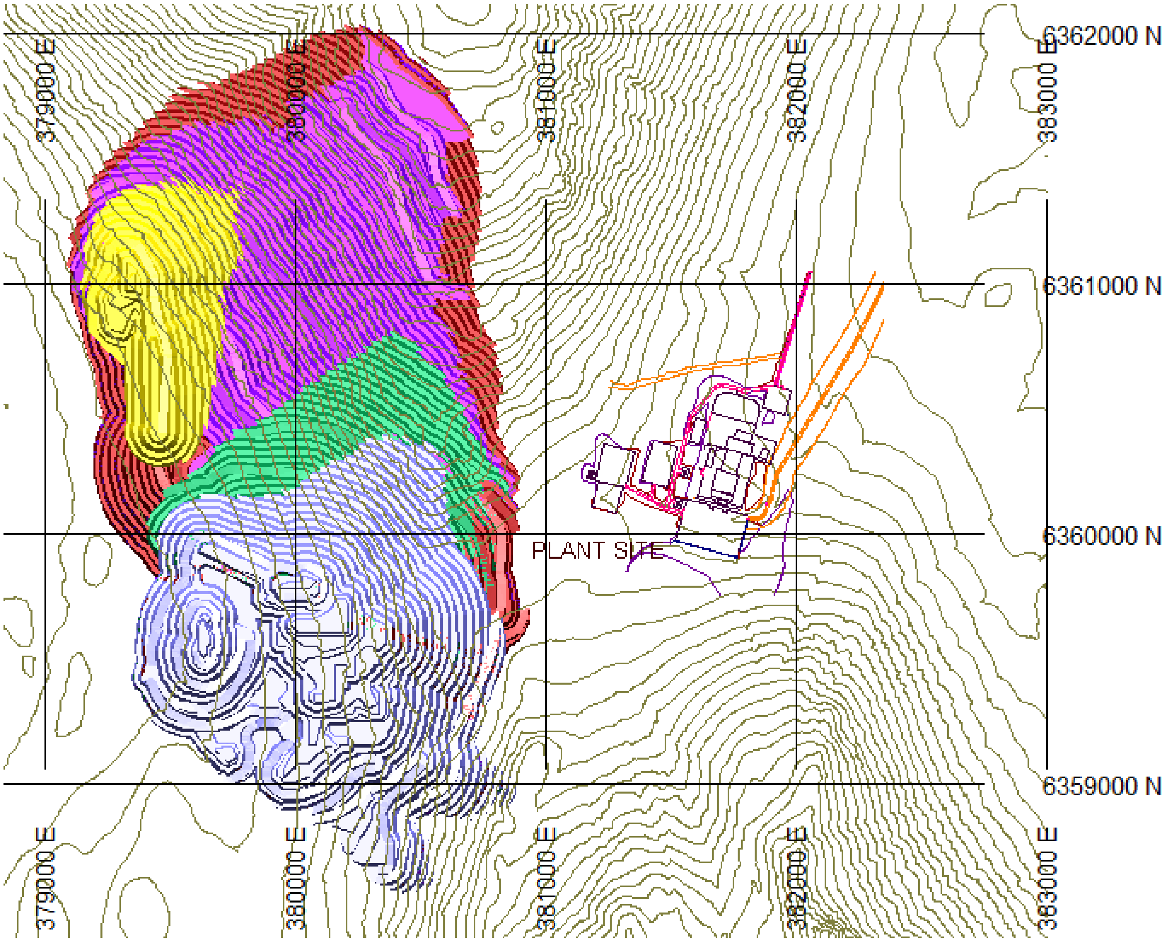


Figure 25.5 Plan View of All Designed Pit Phases

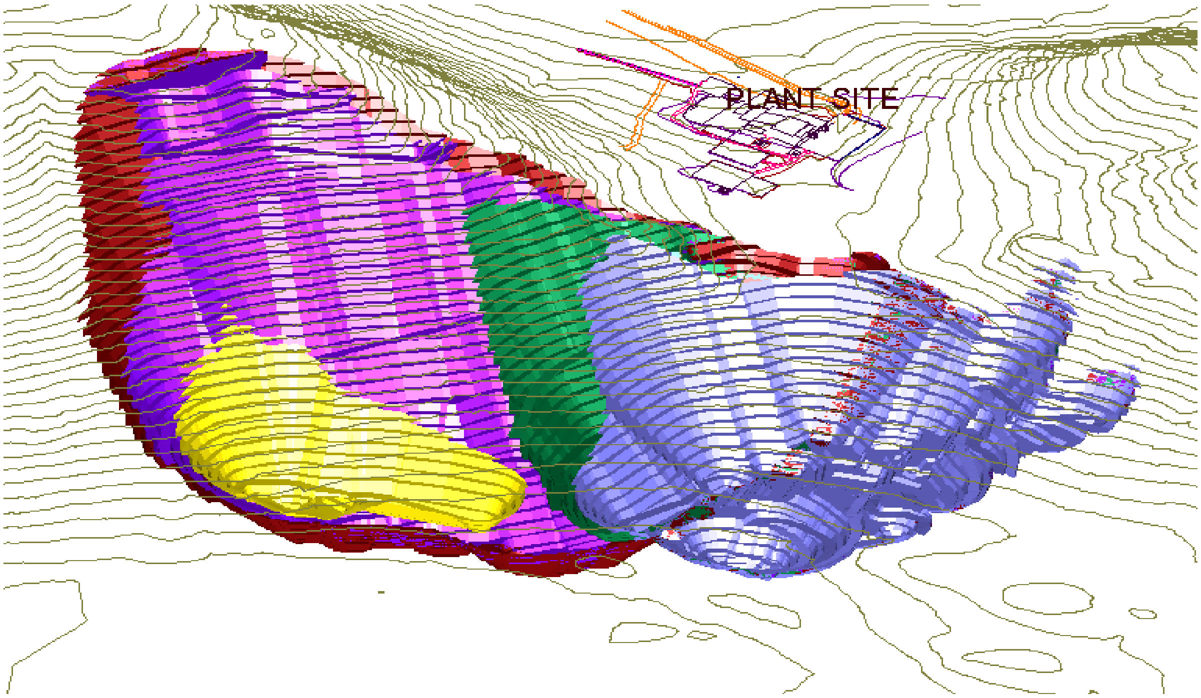


Figure 25.6 Orthographic View from the West of All Designed Pit Phases

25.1.5.4 *Pit Reserves*

Table 25.6 and Table 25.7 list the waste and ore reserves for the material within the Ultimate pit limit (P651).

Pit reserves are estimated in the tables below using the MineSight ® PITRES routine with the following parameters:

- 10% mining dilution applied at the contact between ore and waste;
- Dilution grades are estimated at 4.63 $/t NSR, 0.076% Cu, 0.088 g/t Au, 1.76 g/t Ag and 0.005% Mo representing the average grade of material below the incremental waste/ore cut-off grade (with estimated dilution grades in the $1.00/t to $5.00/t NSR grade bin);
- 5% mining loss;
- Waste/Ore cut-off grade of $5.05t Net Smelter Return (NSR).

Table 25.6 Summarized Measured, Indicated Reserves for Schaft Creek										
Pit	**Description**	**ROM Ore (Mt)**	**ROM Diluted Grades**					**Waste (kt)**	**Stripping Ratio (t/t)**
			NSR($/t)	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Mo (%)**		
P611	South Starter	297.8	16.7	0.330	0.239	1.685	0.017	241.2	0.81
P621i	South Incremental	136.1	14.3	0.266	0.207	1.509	0.018	178.6	1.31
P631	North Starter	52.9	16.2	0.291	0.185	1.940	0.024	45.4	0.86
P641i	Intermediate	208.1	14.3	0.277	0.148	1.674	0.020	476.1	2.29
P651i	Final	125.9	18.1	0.305	0.263	2.275	0.027	601.9	4.78
Total		**821.1**	**15.9**	**0.299**	**0.211**	**1.760**	**0.020**	**1 543.2**	**1.88**

Proven and Probable Reserves at Schaft Creek are summarized in the tables below.

Table 25.7 Proven and Probable Reserves at Schaft Creek						
Reserve Category	**ROM ORE (Mt)**	**ROM Diluted Grades**				
		NSR ($/t)	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**Mo (%)**
Proven	411.1	16.6	0.316	0.236	1.722	0.019
Probable	409.9	15.2	0.283	0.186	1.798	0.020
Total	821.1	15.9	0.299	0.211	1.760	0.020

25.1.6 Mine Plan

25.1.6.1 *LOM 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 2013. The production schedule uses 'PP' as pre-production, 'Year1' as 2013, 'Year2' as 2014, etc.

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 100 tonne class shovel matched with the 360 tonne class truck. Suitable drills to match this size of truck/shovel fleet are indicated in the work below. The performance and costs of a P&H4100XPC cable shovel matched with CAT 797C haulers, P&H 320XPC Electric 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.

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. However 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 maps the grade bins as follows:

- Sub-grade material is wasted;
- Low-grade material is sent to the low-grade stockpile;
- Mid-grade material is sent to the mid-grade stockpile.

To test project NPV sensitivity to mill feed cut-off grade (COG), a set of Long Range production schedules were run using the economic inputs for:

- Revenues, (plant recovery, NSP etc);
- Major mine equipment capital costs;
- Mine operating costs;
- Process operating costs.

The resulting 'relative' NPV @ 6% for each scheduling case is shown in Figure 25.7 (This is a relative project NPV to compare the cases and does not include all the fixed and sustaining capital):

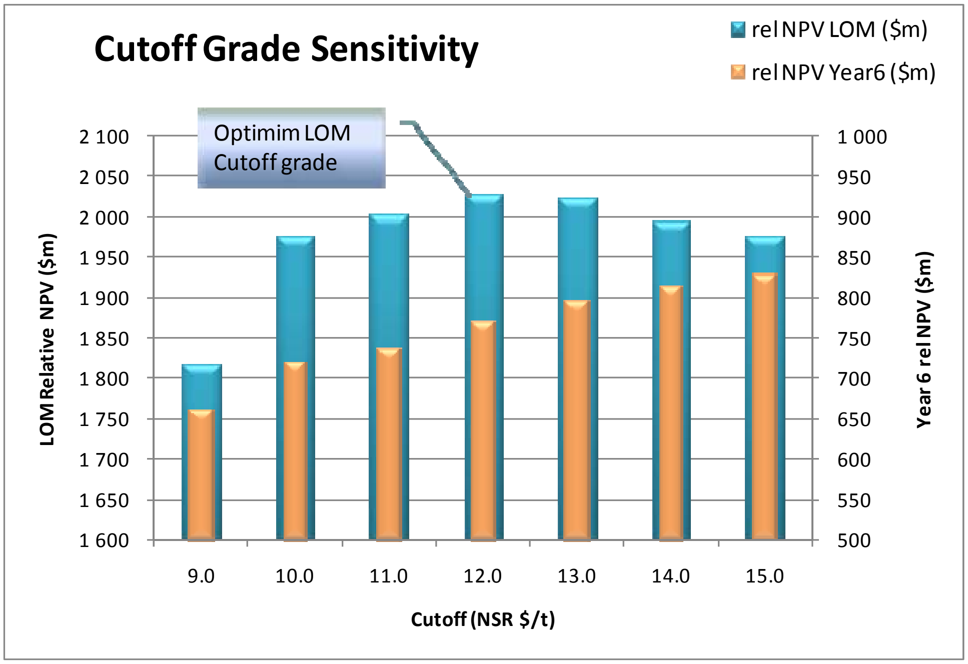


Figure 25.7 Cut-off Grade Optimization

The best case is a Mill Feed COG of $12/t where the relative NPV peaks. Any incremental increase in cut off grade reduces the LOM NPV. There is no value to add by increasing cut-off grade beyond the $12/t case and this is used as the basis of the more detailed PFS schedule cases.

Schedule Results

A summary of the production schedule is shown in Table 25.8.

Table 25.8
Summarized Production Schedule

Material to be Mined	Units	Time (Years)						
		PP	1 to 5	6 to 10	11 to 15	16 to 20	21 to EOM	LOM
Ore Mined To Mill	kt	-	180,050	180,050	180,050	130,575	88,431	759,156
Cu	%	-	0.332	0.323	0.291	0.286	0.326	0.309
Au	g/t	-	0.237	0.235	0.181	0.177	0.267	0.216
Ag	g/t	-	1.59	1.78	1.71	1.78	2.33	1.78
Mo	%	-	0.017	0.018	0.022	0.024	0.026	0.021
ROM ORE to stockpiles	kt	1,378	25,058	15,053	7,254	8,167	642	57,551
Total ORE Mined	kt	1,378	205,108	195,103	187,304	138,742	89,072	816,707
ROM ORE retreived from stockpiles	kt	-	-	-	-	49,475	3,599	53,074
Cu	%	-	-	-	-	0.190	0.145	0.187
Au	g/t	-	-	-	-	0.155	0.136	0.153
Ag	g/t	-	-	-	-	1.44	1.70	1.46
Mo	%	-	-	-	-	0.008	0.008	0.008
Total Stockpile Inventory	kt	1,378	26,435	41,488	48,742	7,434	-	-
Total ROM ORE to MILL	kt	-	180,050	180,050	180,050	180,050	92,030	812,230
Cu	%	-	0.332	0.323	0.291	0.260	0.318	0.301
Au	g/t	-	0.237	0.235	0.181	0.171	0.262	0.212
Ag	g/t	-	1.59	1.78	1.71	1.69	2.31	1.76
Mo	%	-	0.017	0.018	0.022	0.020	0.026	0.020
Metal in Process Feed								
Cu	M lb	-	1,278.8	1,282.1	1,154.6	1,032.0	646.1	5,389.6
Au	M oz	-	1.37	1.36	1.05	0.99	0.77	5.54
Ag	M oz	-	9.19	10.32	9.88	9.78	6.83	46.00
Mo	M lb	-	65.64	71.74	85.97	77.72	51.75	352.82

Table 25.8 Summarized Production Schedule								
Material to be Mined	**Units**	**Time (Years)**						
		PP	**1 to 5**	**6 to 10**	**11 to 15**	**16 to 20**	**21 to EOM**	**LOM**
Recovered ROM Grades								
RCu	%	-	0.287	0.288	0.259	0.232	0.284	0.269
RAu	g/t	-	0.162	0.161	0.124	0.117	0.180	0.145
RAg	g/t	-	1.28	1.44	1.38	1.36	1.86	1.42
RMo	ppm	-	0.012	0.013	0.016	0.014	0.018	0.014
Recoverable Metal in Process Feed								
RCu	M lb	-	1,140.7	1,143.6	1,029.9	920.6	576.4	4,811.1
RAu	M oz	-	0.94	0.93	0.72	0.68	0.53	3.80
RAg	M oz	-	7.41	8.33	7.97	7.89	5.52	37.12
RMo	M lb	-	47.26	51.65	61.90	55.96	37.26	254.03
Waste								
Sub-Grade Wasted	kt	-	-	2,107	1,017	1,134	111	4,368
Waste Mined	kt	23,621	339,981	368,876	386,859	385,741	38,113	1,543,191
Total Waste Mined	kt	23,621	339,981	370,983	387,875	386,875	38,224	1,547,559
Waste Types:								0
Waste	kt	23,621	339,981	368,876	386,859	385,741	38,113	1,543,191
Strip Ratio (Waste Mined/Ore Milled)		-	1.9	2.0	2.1	2.1	0.4	1.9
Total Material Mined	t	24,999	545,088	566,087	575,179	525,616	127,296	2,364,266
Total Material Moved	t	24,999	545,088	566,087	575,179	525,616	130,895	2,417,340

Figure 25.8 illustrates that significant stockpile reclaim is required in years 15 to 19 when the pre-stripping of the ultimate phase is being completed.

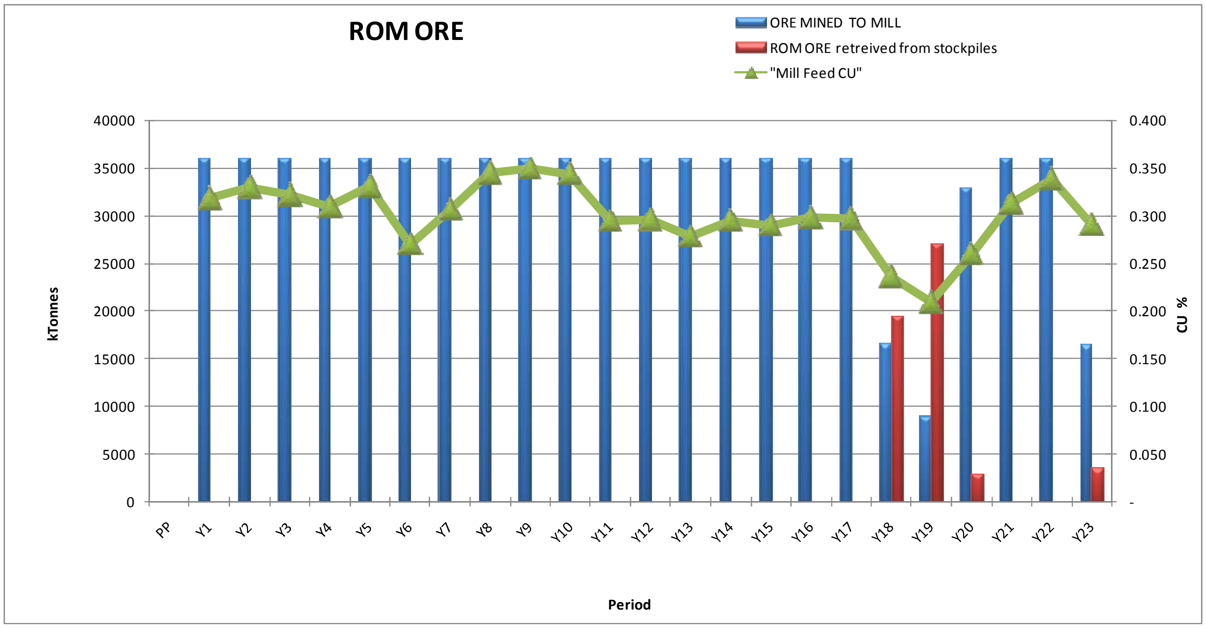


Figure 25.8 Graph Showing ROM Ore Source and Mill Feed Copper Grade

If there is insufficient ore stockpiled prior to the pre-stripping of the ultimate phase a substantial increase in fleet will be required in order to meet the material movement requirements.

The strip ratio, by period, is shown in Figure 25.9. In periods where long hauls are required, the strip ratio is decreased. In periods where short hauls are required, the strip ratio is increased. This is done to smooth out the size of the mining fleet over the life of mine.

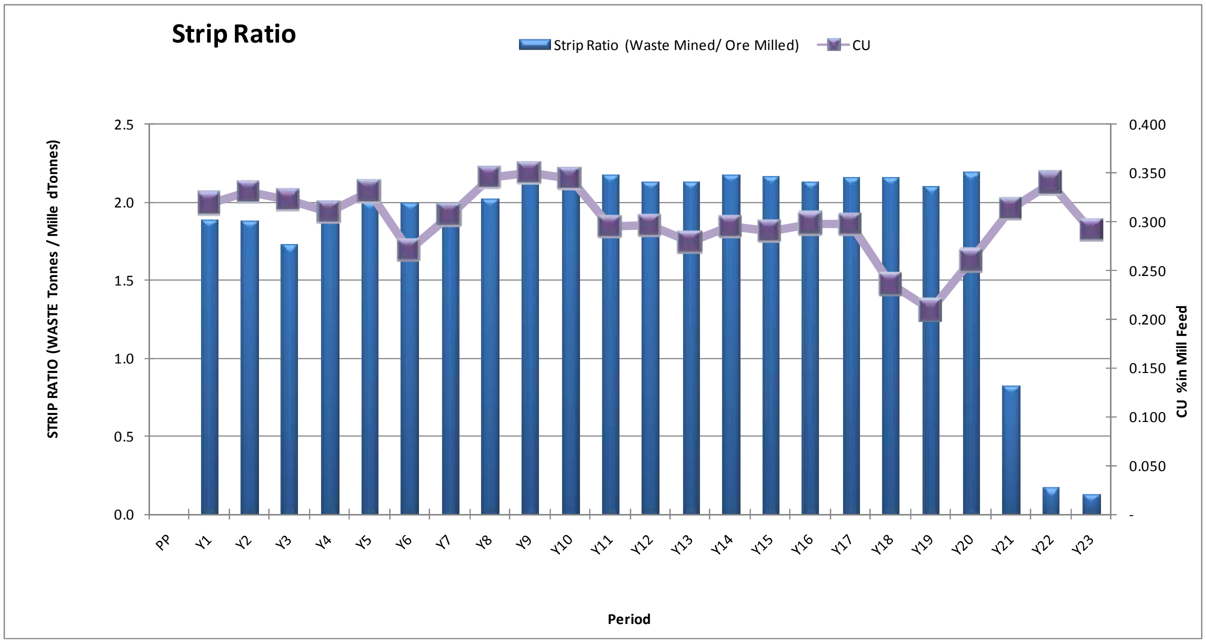


Figure 25.9 Graph Showing Strip Ratio and Mill Feed Copper Grade

25.1.6.2 *Waste Rock Storage*

Design Parameters

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.

PAG Waste Rock

Potentially Acid Generating waste rock requires special handling to minimize the environmental impact from acid rock drainage (ARD) and metal leaching. Preliminary geochemical test work carried out on Schaft Creek rock showed an estimated 4% of the tested material was Potentially Acid Generating (PAG), and 6% was uncertain. Spatial distribution of the PAG rock has not yet been modeled, and results from metal leaching testwork are not yet available. Using the results from preliminary geochemical testwork results, ABA assumptions for the Schaft Creek PFS are:

- 10% of all waste is assumed to be PAG that is uniformly distributed throughout the modeled area.
- PAG waste rock removed from the pit before a backfilling opportunity is available will be placed on dumps with an NPAG base, and will be capped with NPAG rock to seal it off and prevent oxidation; and
- Once P611 pit has been mined out, the PAG waste rock mined will be backfilled to the mined out P611 pit.

In the current production schedule, the south starter pit P611 is available for backfilling from Year 10 onwards and all PAG waste rock is hauled to this pit from Year 10 to the end of LOM.

If necessary, the body of water receiving the backfilled waste rock at closure may require batch treatment for acid neutralization and metals leaching. Treatment may be required for some time after closure until oxidation ceases.

Dump Monitoring and Planning

The high-lift dumping approach is most economical and 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-lift 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 facilitates a lower overall dump slope angle for future reclamation. The wrap-around terraces reduce the amount of material to be moved to re-slope the final reclaimed dump faces upon completion. Where possible dumps will be re-sloped and re-vegetated progressively rather than all dumps at the end of mining activities. This will reduce erosion off the dumps during operations, reduce closure costs at the end of operations, and help progressively establish effective and final reclamation criteria.

Annual Waste Volumes and Placement

Annual waste volumes produced from the 100 ktpd schedule (MS-SP Schedule 9) are shown for selected periods in Table 25.9.

Table 25.9 Waste Production Schedule (LCM)			
Year	**Waste Mined**	**NPAG Waste**	**PAG Waste**
	Mm^3	**Mm^3**	**Mm^3**
Pre-Production	11.4	10.2	1.1
Year 1	32.8	29.5	3.3
Year 2	32.7	29.4	3.3
Year 3 - 5	98.8	88.9	9.9
Year 6 - 10	179.2	161.3	17.9
Year 11 - 20	374.3	336.8	37.4
Year 21 - LOM	18.5	16.6	1.8
Total	**747.6**	**672.9**	**74.8**

25.1.6.3 *Mine Production Detail*

End-of-period mine status maps have been developed for Year 1, Year 2, Year 5, Year 10, Year 20 and Year 23 (Life of Mine).

End of Pre-Production

During pre-production, pit phase P611 is mined to bench 1150 m, and P621i is mined to 1345 m. Waste above 1225 m is hauled to the East Dump and waste below 1225 m is hauled to the SW to fill out a haul road to the crusher and lay down a pad for the stockpile. Ore is hauled to the stockpile to the SW of the pit. The mine layout at the end of the pre-production period is illustrated in Figure 25.10 and Figure 25.11. These figures show the plant site and general infrastructure for orientation purposes.

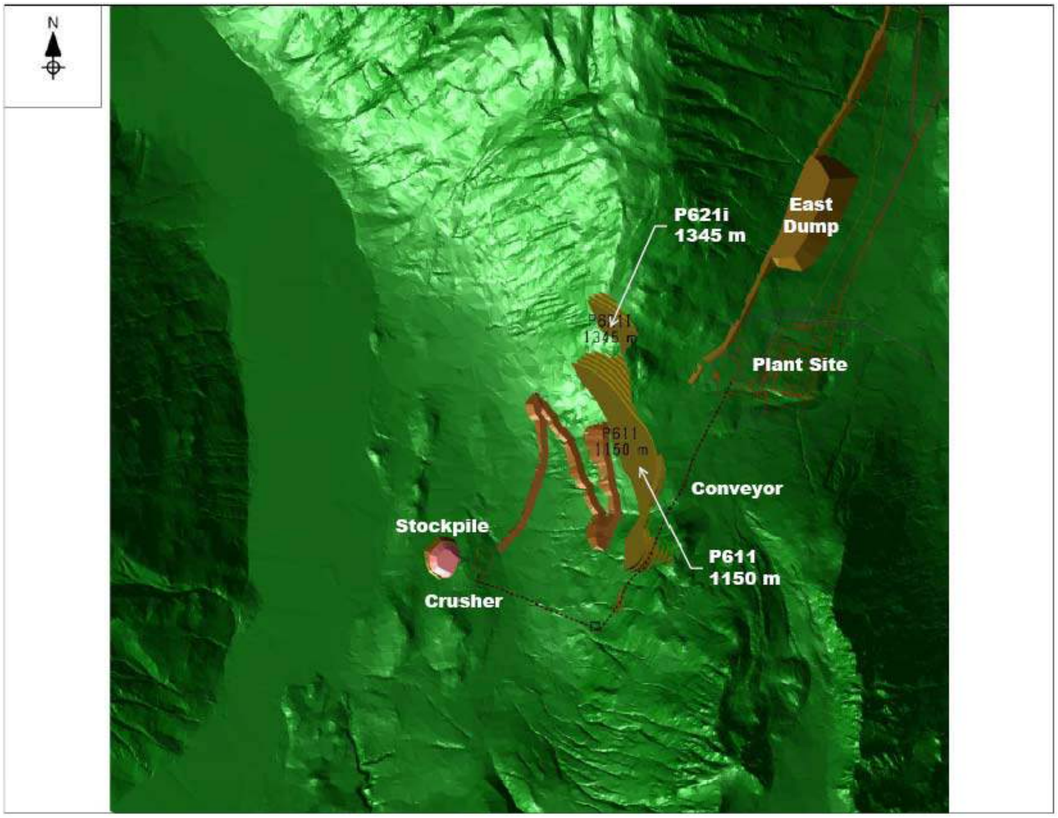


Figure 25.10 End of Pre-Production Plan View

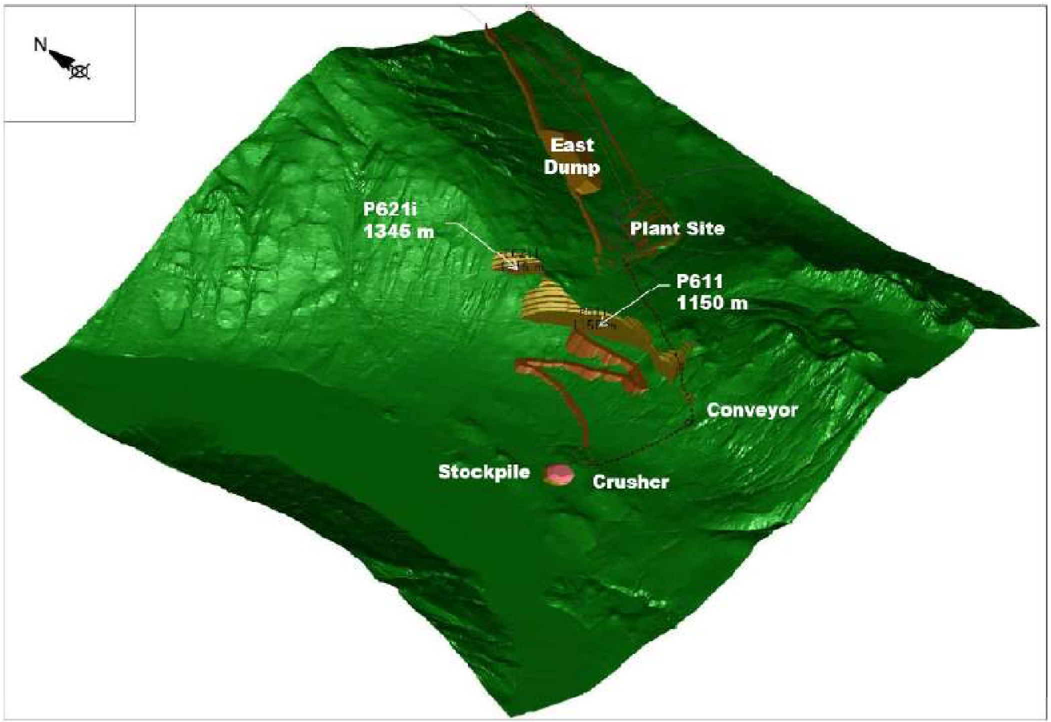


Figure 25.11 End of Pre-Production Orthographic View from the SW

End of Year 1

During year 1, pit phases P611 and P621i are mined to benches 1015 m and 1345 m respectively. Highwall Waste is hauled to the East Dump and lower bench waste is hauled to the SW Dump out of the southwest pit. Ore is hauled to the stockpile and crusher to the SW.

The mine layout at the end of Year 1 is illustrated in

Figure **25.12** and Figure 25.13.

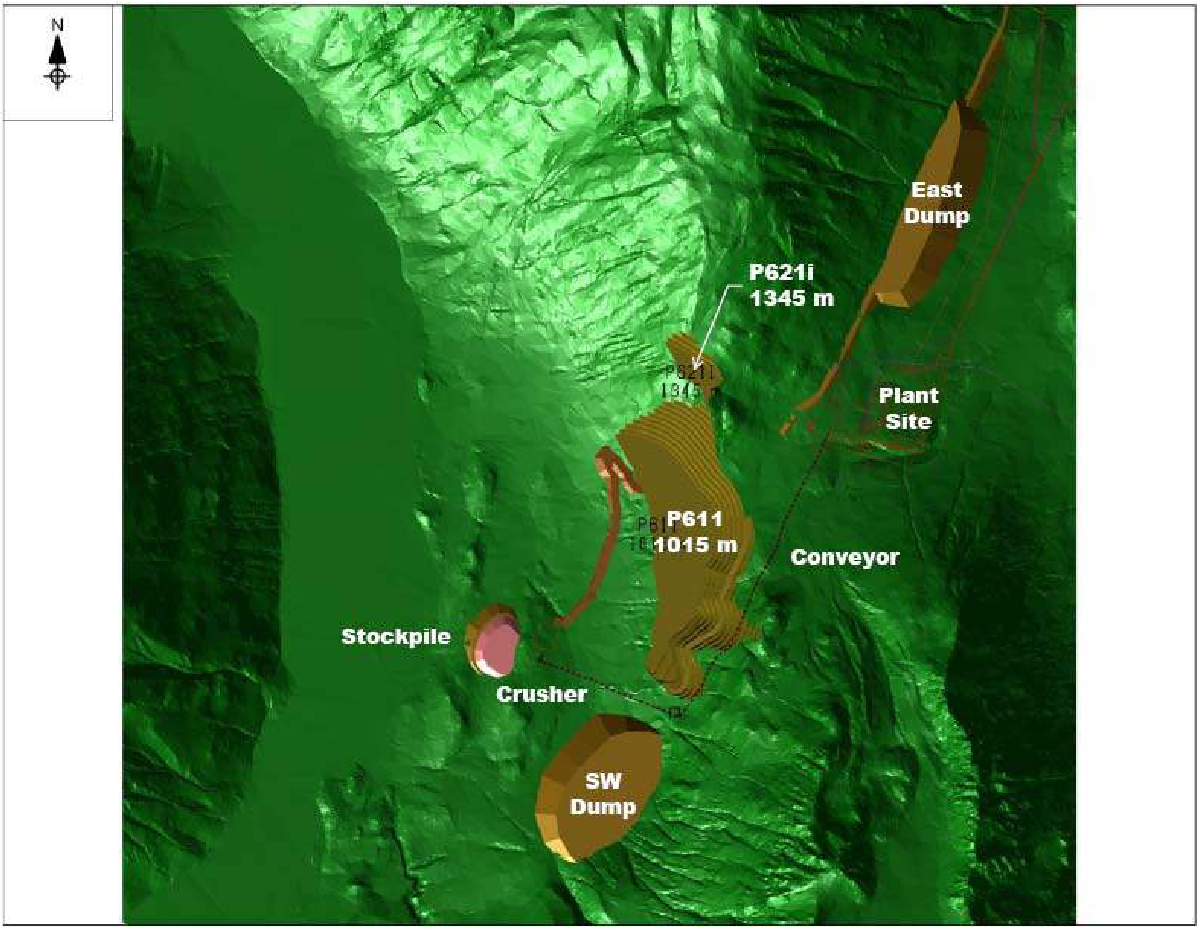


Figure 25.12 End of Year 1 Plan View

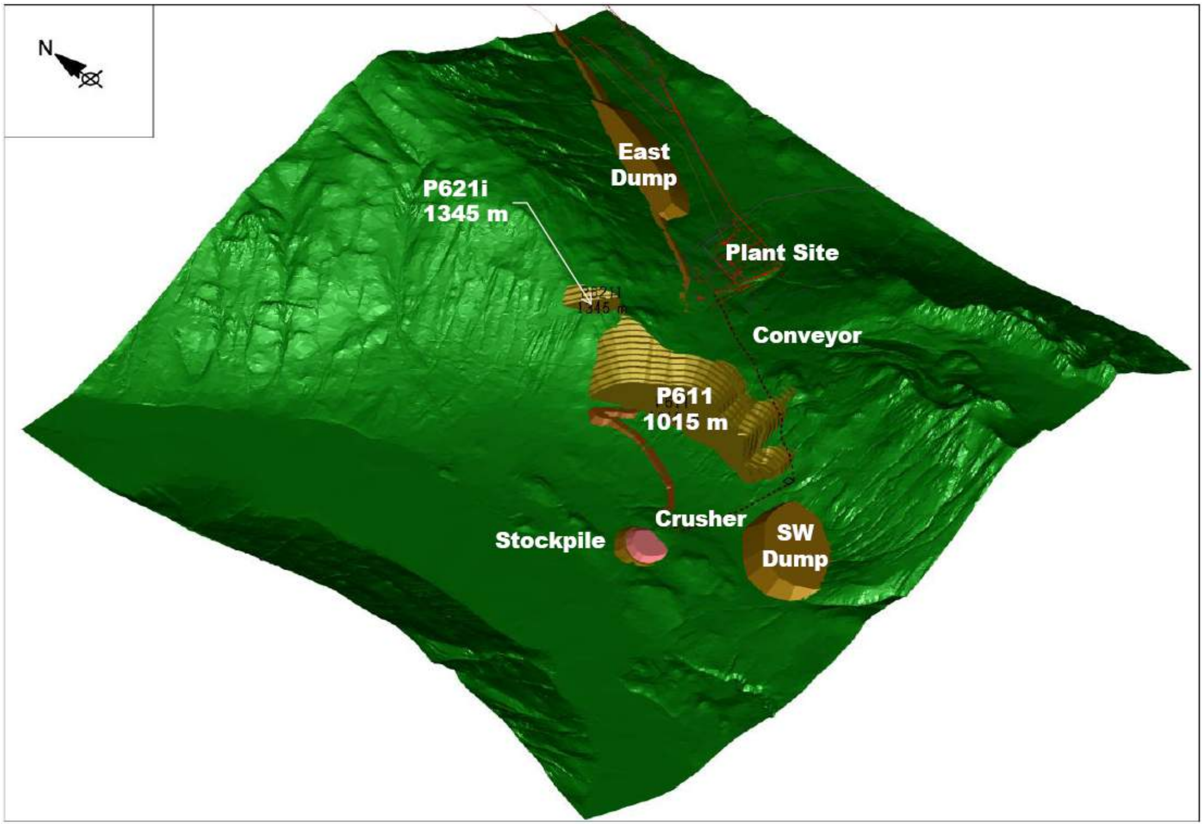


Figure 25.13 End of Year 1 Orthographic View from the SW

End of Year 2

During year 2, pit phase P611 and P621i are mined to 955 m and 1210 m respectively. All Waste is hauled to the SW Dump out of the southern end of the pit. Ore is hauled to the Crusher and Stockpile to the south of the pit. The mine layout at the end of Year 2 is illustrated in Figure 25.14 and Figure 25.15.

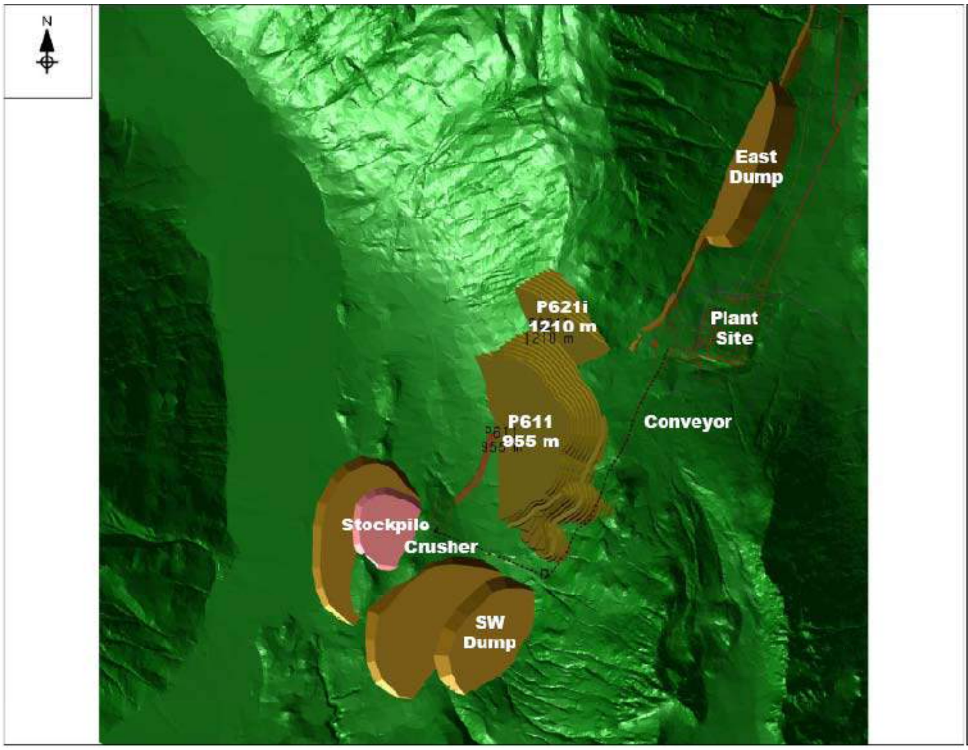


Figure 25.14 End of Year 2 Plan View

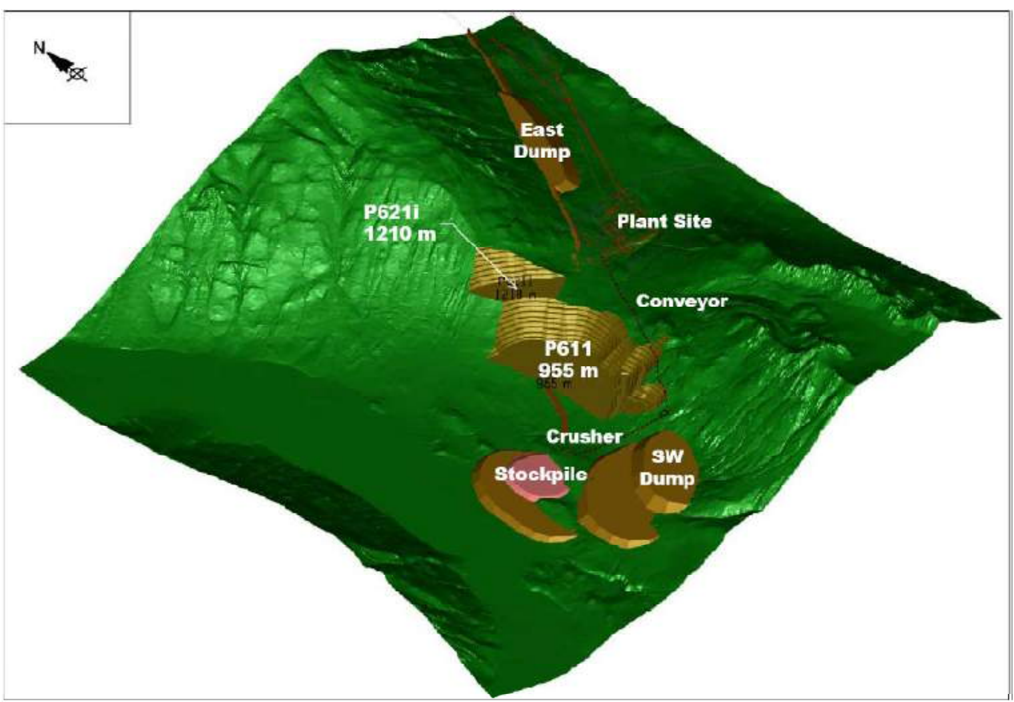


Figure 25.15 End of Year 2 Orthographic View from the SW

End of Year 5

During years 3 to 5, pit phases P611, P621i, P631, P641i, and P651i are mined to 850 m 1015 m, 910 m, 1615 m, and 1720 m respectively. Waste from P621i, P641i, and P651i is hauled to the East Dump and waste from P611 and P631 is hauled to the Lower SW Dump. Ore is hauled to the Crusher and Stockpile to the SW.

The mine layout at the end of Year 5 is illustrated in Figure 25.16 and Figure 25.17.

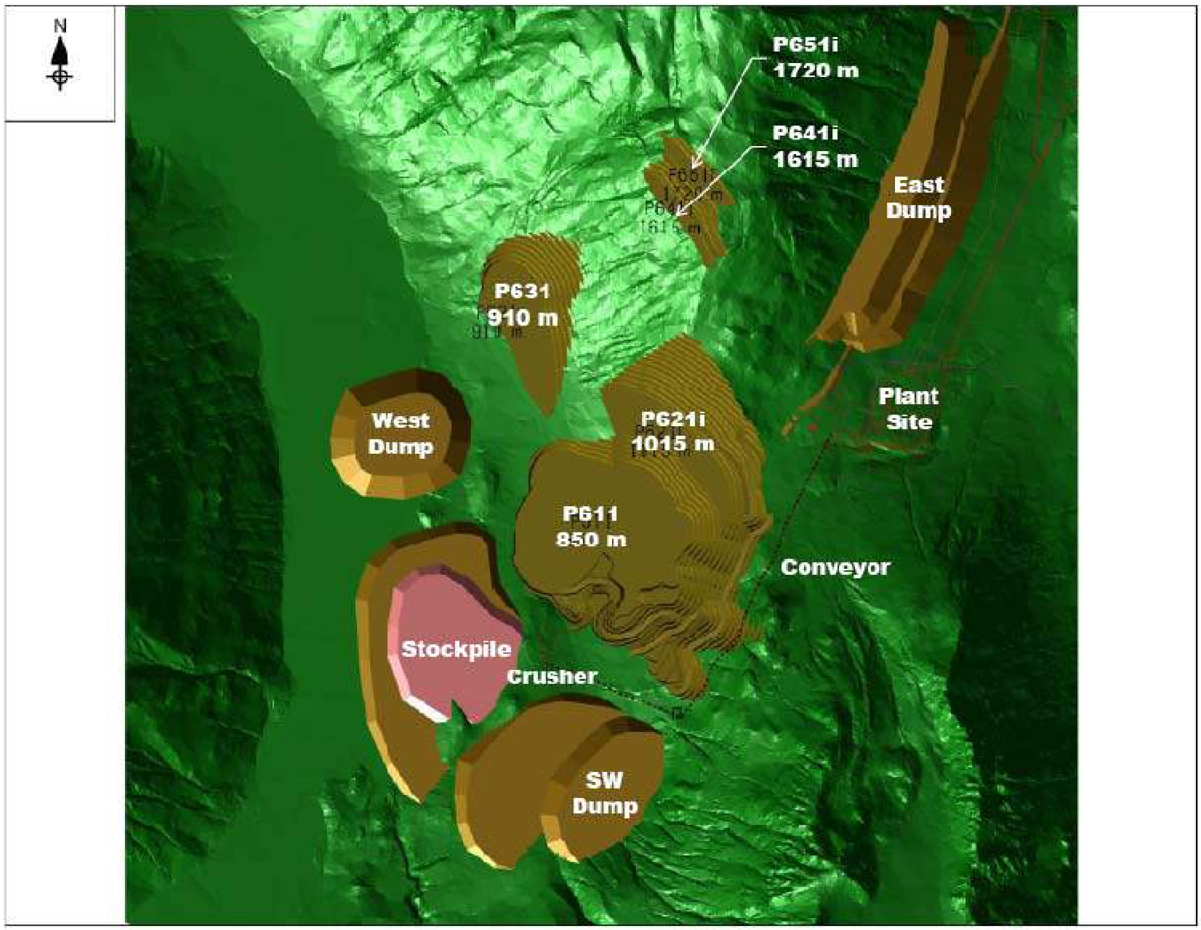


Figure 25.16 End of Year 5 Plan View

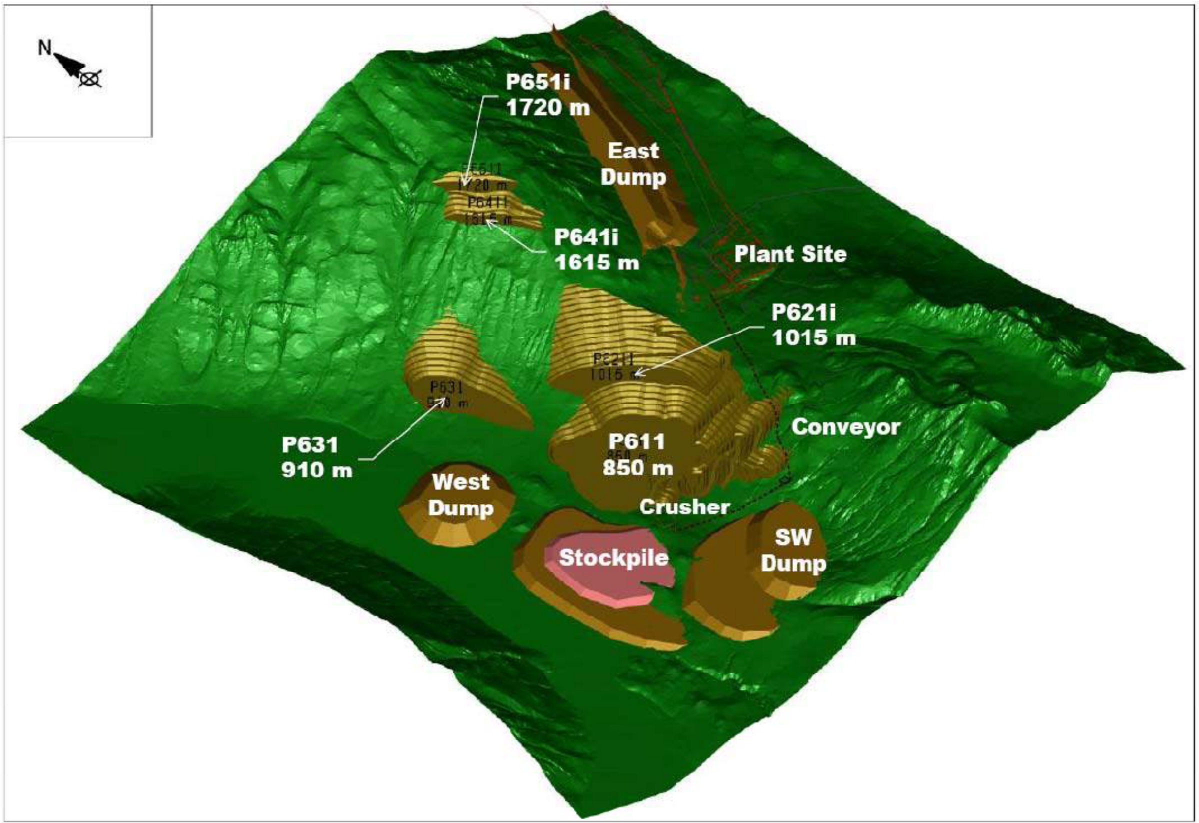


Figure 25.17 End of Year 5 Orthographic View from the SW

End of Year 10

During years 6 to 10, pit phases P611, P621i, P631, and P641i are mined to benches 685 m, 775 m, 880 m, and 1000 m respectively. Waste from the upper P641i is hauled to the East dump and Upper NW Dumps, from the lower benches of pits P631 and P641i is hauled to the lower NW Dump, and NPAG from pits P611 and P621i is hauled to the SW dump. Ore is hauled to the Stockpile and Crusher. PAG rock is mixed with NPAG waste to be capped at the end of the mine's life.

The mine layout at the end of Year 10 is illustrated in Figure 25.18 and Figure 25.19.

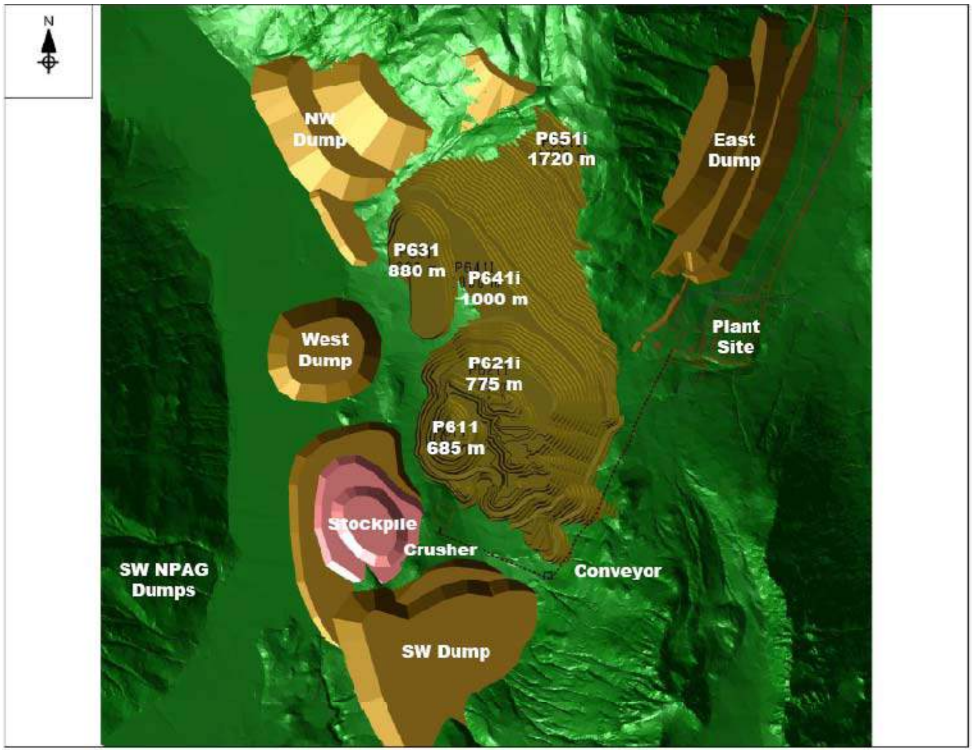


Figure 25.18 End of Year 10 Plan View

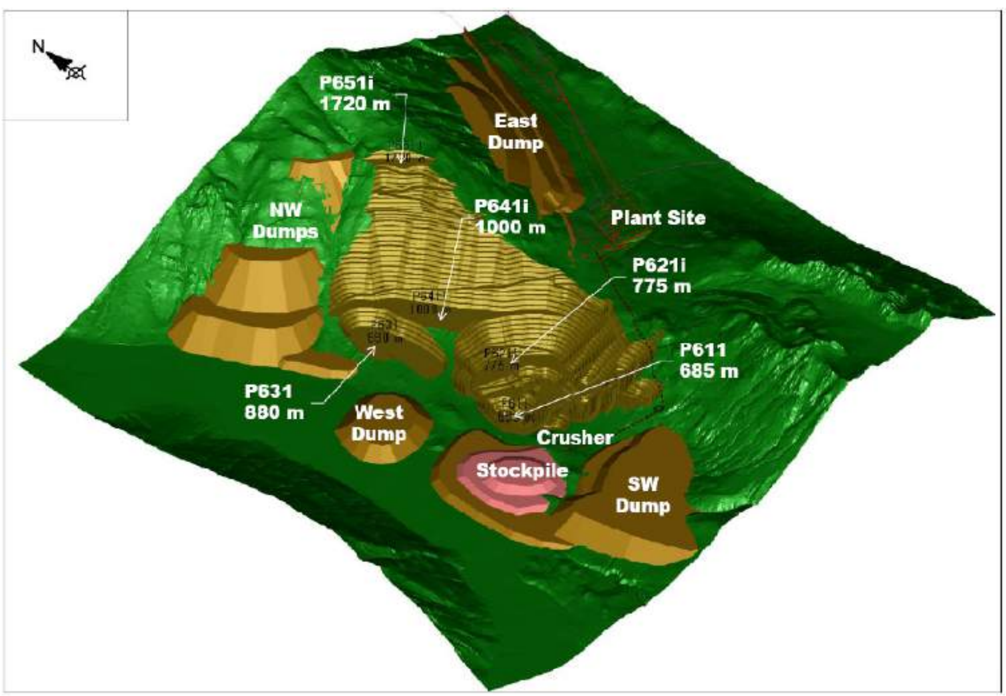


Figure 25.19 End of Year 10 Orthographic View from the SW

End of Year 20

During years 11 to 20, pits P642i, P631, and P641i are mined to bottom benches and P651i is mined to 775 m. NPAG is hauled from the lower benches to the W NPAG Dumps. Highwall waste and excess lower bench waste is taken to the NW Dump. Ore is hauled to the Stockpile and Crusher. PAG is backfilled into the P611 pit.

The mine layout at the end of year 20 is depicted in Figure 25.20 and Figure 25.21.

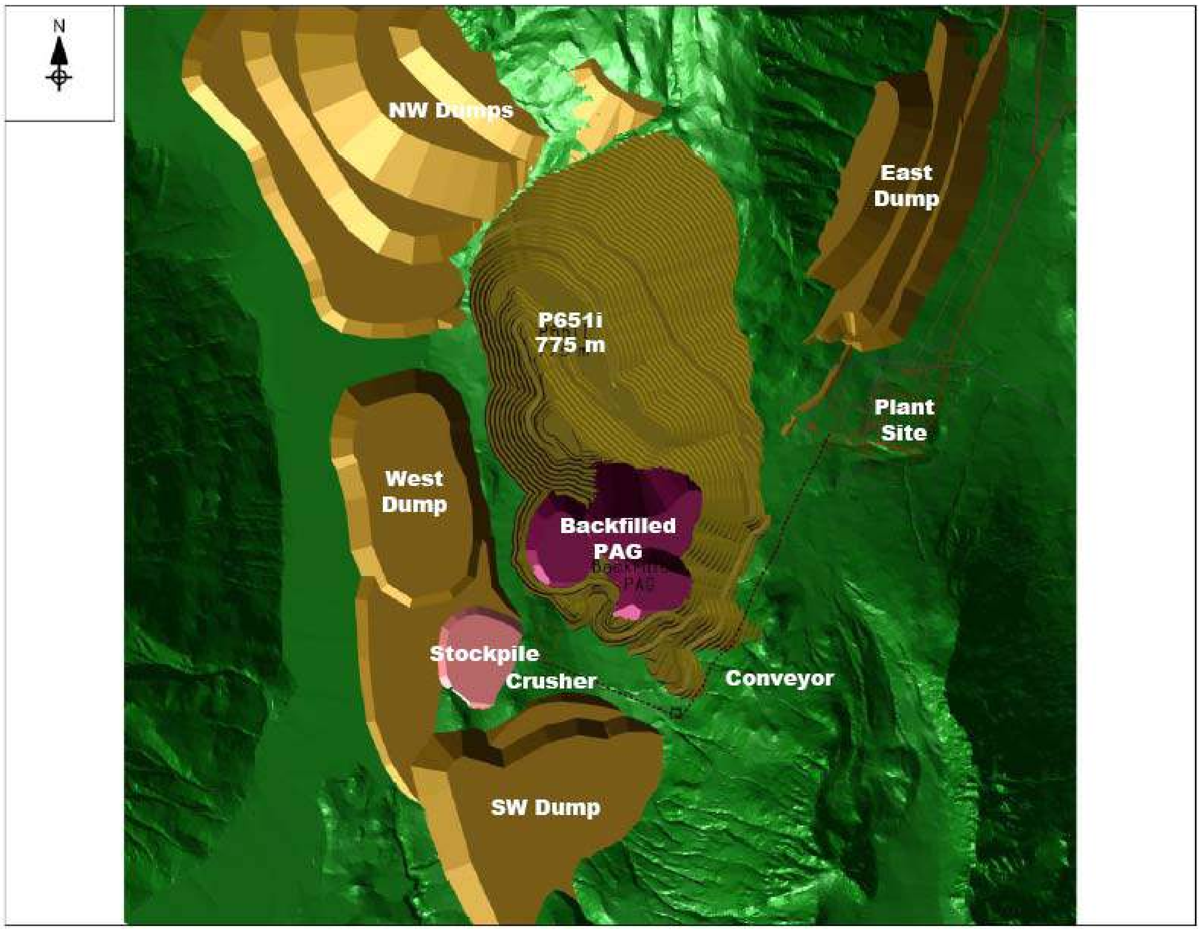


Figure 25.20 End of Year 20 Plan View

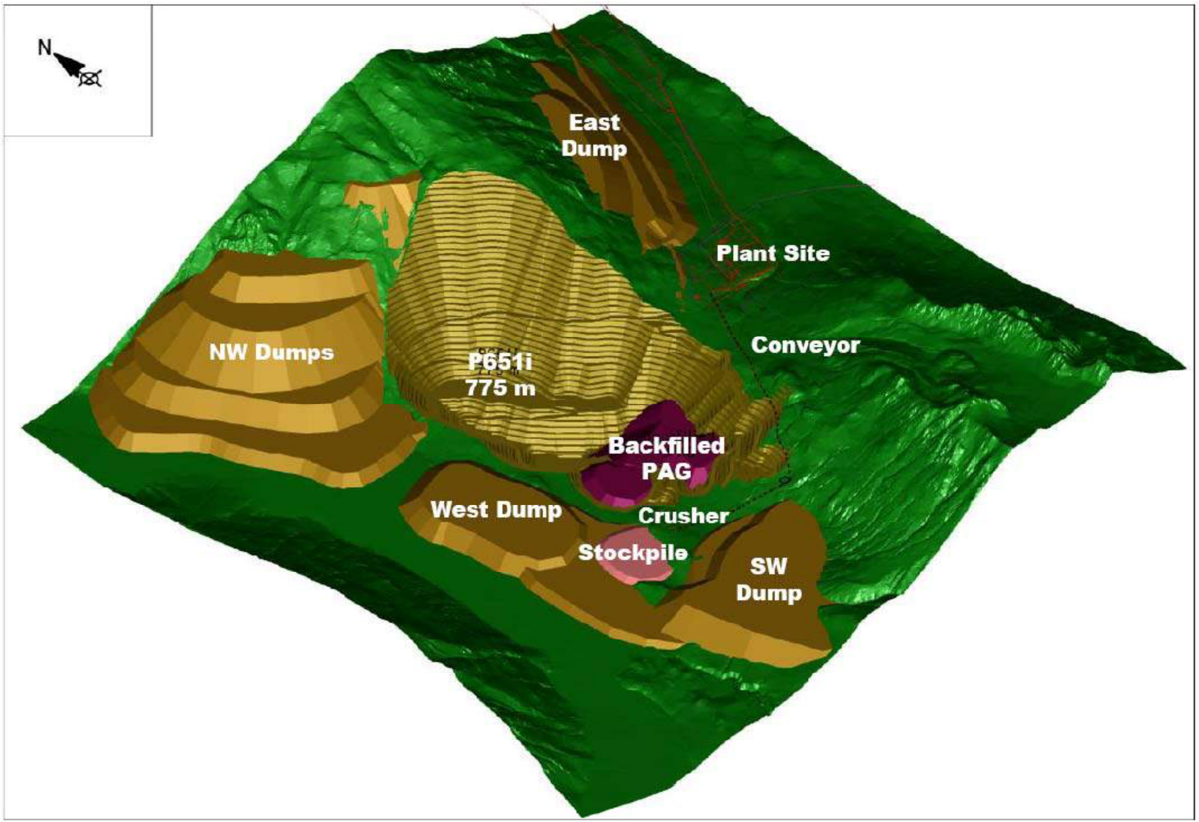


Figure 25.21 End of Year 20 Orthographic View from the SW

End of Life of Mine

During years 21 to 24, phase P651i is mined to bench 520 m. NPAG waste is hauled to the Waste Dump and PAG is backfilled into the P611 pit. Ore is hauled to the Crusher from the pit and the stockpile.

The mine layout at the end of LOM prior to closure is illustrated in Figure 25.22 and Figure 25.23.

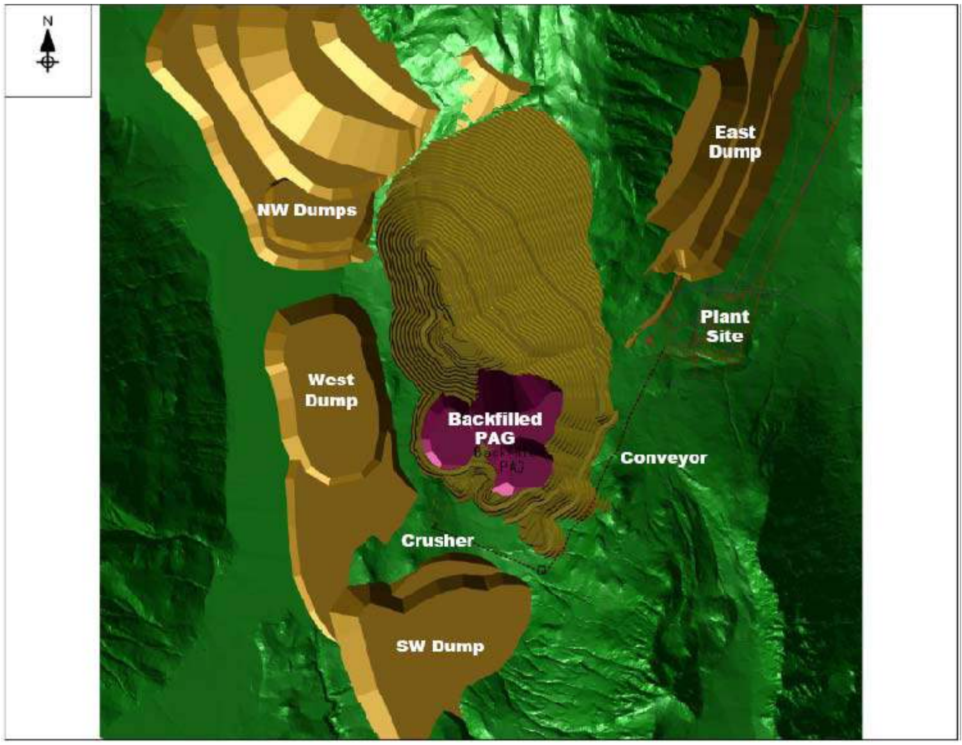


Figure 25.22 End of Life of Mine Plan View

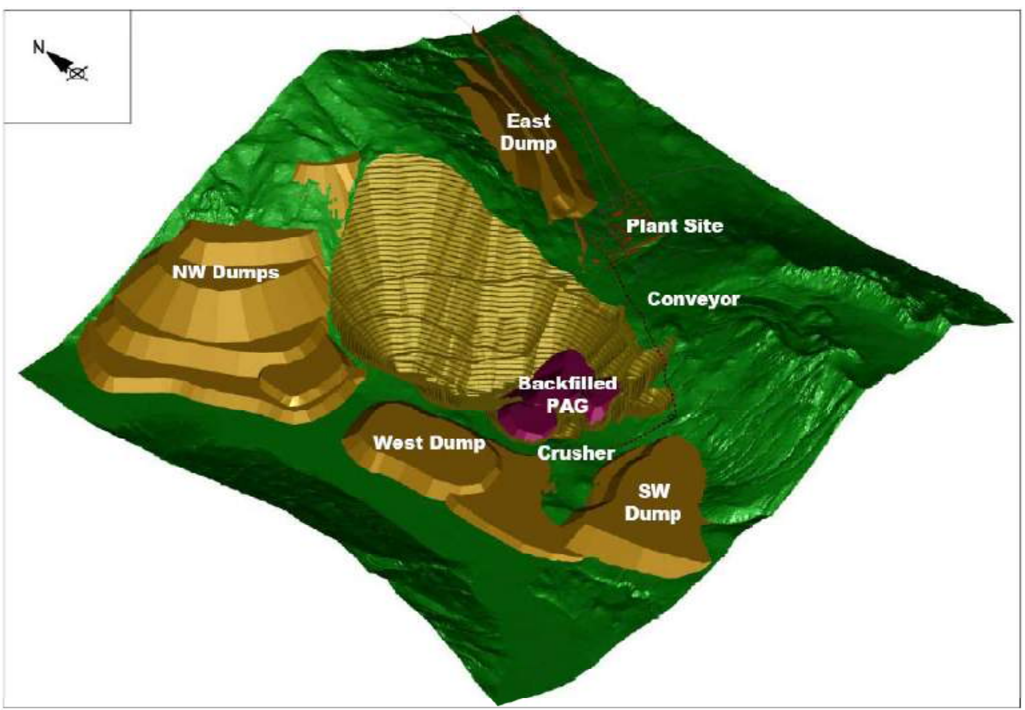


Figure 25.23 End of Life of Mine Orthographic View from the SW

25.1.7 Mine Operations

The mining operations will be typical of open-pit operations in mountainous terrain in western Canada and will employ tried and true bulk 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. A large capacity operation is being designed and large scale equipment is specified for the major operating areas in the mine to generate high productivities which will reduce unit mining costs and will allow the lowest mining cost to be achieved. 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.

25.1.8 Mine Equipment

The mining equipment descriptions below provide general specifications so that dimensions and capacities can be determined from the manufactures specification documents. While the list may include specific manufacturers and models, it does not mean that a final selection of equipment has been made; rather it is meant to be representative of equipment of the size class selected. Final brand selection will be subject to a commercial process.

25.1.8.1 *Major Mine Equipment*

The production requirements for the major mining equipment over the life of the mine are summarized in Table 25.10. According to the current production schedule and the haulage assumptions, the maximum number of trucks required is 31.

For this study, the potentially acid generating (PAG) material is being estimated at ten percent of total material mined. Only after the appropriate Acid Based Accounting (ABA) geochemistry model is completed can the distribution of PAG and non-PAG material be incorporated into the geological block model. The quantity and spatial distribution of PAG material will directly affect the truck haulage hours for this material, but at this time it is not known the extent of this impact.

Table 25.10
Major Mine Equipment Requirements

	PP	Y1	Y2	Y3	Y4	Y5	Y10	Y20	Y23
Drilling:									
Primary Drill - P&H 120A – 311 mm Electric Drill	2	4	4	4	4	4	4	4	1
Highwall Drill - Sandvik D245S – 150 mm Diesel Drill	1	1	1	1	1	1	1	1	0
Loading:									
P&H4100XPC Cable Shovel - 104 t	2	4	4	4	4	4	4	4	1
Cat 3516 GENSET	1	1	1	1	1	1	1	1	1
Hauling:									
Cat 797B Haul Truck - 345 t	8	12	12	17	18	23	28	27	7

25.1.8.2 *Mine Support Equipment*

The mine support equipment fleet requirements over the life of the mine are summarized in Table 25.11.

Table 25.11
Mine Support Equipment Fleet

Equipment	PP	Y1	Y2	Y3	Y4	Y5	Y10	Y20	Y23
Track Dozer – Cat D11 – 634 kW	4	4	4	4	4	4	4	4	3
Track Dozer – Cat D10 – 433 kW	3	3	3	3	3	3	3	3	1
Rubber-Tired Dozer – Cat 844 – 634 kW	2	2	2	2	2	2	2	2	1
Motor Grader – Cat 24 – 7.3 m blade	3	3	3	3	3	3	3	3	3

25.1.8.3 *Mine Ancillary Equipment*

The mine ancillary equipment fleet is listed in Table 25.12.

Table 25.12
Mine Ancillary Equipment Fleet

Equipment	Max. Fleet Size
Cat 385 Hydraulic Excavator - 12-13 t	2
Cat 345 Hydraulic Excavator - 5-7 t	2
Cat 988 Multipurpose - Cable-Reeler, Forks, Brushes, Bucket	1
Cat 988H Tire Manipulator	1
Cat 740 Fuel/Lube Truck	3
Cat 789 Water Truck - 48,000 Gallons	2
CAT IT28 Blasthole Stemmer - 3 t	2
Ford F150 Pickup Truck – ½ st	20
Ford F550 Maintenance Truck - 1 st	4
Kenworth T300 Service Truck	3
Hyster 620F Forklift - 30 t	1
Hyster 210HD Forklift - 10 t	1
Kenworth C500 Picker Truck	2
GMC Guide XL Crew Bus	2
Chevrolet G3500 Passenger Vans	2
Ambulance	1
Kenworth T800 Fire Truck	1
Mine Rescue Truck	1
Cat 789 Float Tractor/Trailer - 189 t	1
Fintec 570 Screening Plant – 12in maximum	1

25.1.8.4 *Mine Maintenance Equipment*

The mine maintenance equipment fleet is listed in Table 25.13.

Table 25.13
Mine Maintenance Equipment Fleet

Equipment	Max. Fleet Size
LTM1250-6.1 – 250 t crane	1
Kenworth T300 Welding Truck	2
Power Line Truck	2
LTM1100 – 100 t crane	1
Tsurumi LH8110-60 Water Pump – 318 m^3/h	8

25.1.8.5 *Snow Removal Equipment*

The snow removal equipment fleet is listed in Table 25.14.

Table 25.14
Snow-Removal Equipment Fleet

Equipment	Fleet Size
Cat 637G Scraper - 37 tonnes	5
Cat 988H Wheel Loader - 14 tonnes	1
Cat 16H Grader - 4.88 m blade	1
Fintec 570 Screening Plant – 30 cm max	1
SnowCat	2

25.1.8.6 *Mine Ancillary Facilities*

Shops and Offices

In addition to providing an area for maintenance bays, tire shops and a wash bay the truck shop will also house the following:

- Welding bay;
- Electrical shop;
- Ambulance;
- First aid room;
- First aid office;
- Machine shop area;
- Mine dry;
- Warehouse;
- Offices for mining and engineering staff;

- Lunch room and the foreman's office.

There will be four truck maintenance bays, which will be suitable for the 345 t haul trucks. The workshop/warehouse/office complex will be located on the north end of the building, along with two small truck service bays, a welding bay, electrical shop, machine shop, ambulance and first aid room. This office complex will include space for mine engineering, maintenance supervision and related support staff.

Mine Electric Power

A substation will be required to drop the power from 23 kV at the plant site to 13.8 kV, which is the required supply voltage for the mine. An overhead powerline is required from the substation at the plant site to the pit. An overhead powerline will be installed around the perimeter of the pit. Initially this will be located around pit phase 621 which will allow mining to proceed until Year 5. At that time, the power line will be relocated to the perimeter of the ultimate pit, pit phase 651. Short stub lines are required to bring the power from the perimeter to the pit edge or into the pit where a substation will be used to drop the power from 13.8 kV to 4160 V which is the power supply required by the electric cable shovels and electric rotary drills.

25.2 Infrastructure

The remote location of the Schaft Creek project site will require development of support facilities for the project. These services will include medical, fuel supply, roads, power, water supply, general maintenance for process equipment and site vehicles. Of critical importance to the success of the project will be reliable year round access to the site by both ground and air.

25.2.1 Site Access Ground

25.2.1.1 Background Information

McElhanney Consulting Services Ltd. (McElhanney) was retained by Copper Fox Metals Inc. (Copper Fox) to complete a preliminary feasibility study (PFS) of the proposed access road to the Schaft Creek project. The Schaft Creek project involves the development and mining of a major copper-molybdenum porphyry deposit located in remote northwest British Columbia, approximately 70 km west of Bob Quinn Lake which is located on the Stewart-Cassier Highway. The McElhanney report, *Schaft Creek Mine Access Road Mess Creek Access Route – Pre-Feasibility Report* was completed in May 2008. This study advances the understanding of the Schaft Creek access routes that has progressed over time. McElhanney completed a scoping study report in December 2005 and an initial pre-feasibility study report in January 2007.

A P-line survey was conducted and site surveys were made on stream crossings to provide additional information on the structures required. The revised study that is used to support this PFS advances the engineering understanding of the Mess Creek route by field checking earlier assumptions and quantifying road construction parameters.

25.2.1.2 *Project Objectives*

This Schaft Creek access road study encompassed 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 parameters in mind but separate additional detailed studies are 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 preliminary feasibility report.

25.2.1.3 *Road Engineering*

Route Selection

Contour data from mapping provided by Eagle Mapping in 2006 and McElhanney in 2007 was utilized in the layout and design of the route to provide access to the Schaft Creek project. A comprehensive on-ground review of the selected route resulted in revision to the original route and is the basis for this study.

To avoid the avalanche area at the south end of the route, alternate routes on both the east and west sides of the valley were traversed. A route along the west side was chosen to provide a safer route to the mine site.

The following route configurations were investigated during the on ground survey:

- Modifications to the road location along the east side of the Mess Creek valley to minimize avalanche impacts presented at the south end of the route;
- Four corridors along the west side of the Mess Creek valley to avoid the avalanche areas on the east side. The route presented here is the favoured road location.

Route Description

The Mess Creek route originates at 65.1 km of the Galore Creek Road and travels along the west side of Little Mess Creek through the pass dropping down to the valley bottom and crosses 400m of wetland. The route then climbs to a bench on the west side of the valley and remains on this bench for 1.0 km above the South Mess Lake. The route crosses the upper crossing of Little Mess Creek with an 18 m bridge. Side slopes from 0.0 km to Little Mess Creek at 1.9 km are between 10% and 30%. From Little Mess Creek the route crosses an old alluvial fan formed by Little Mess Creek then rises up the slope traversing a rock outcrop at 2.3 km for 150 m; cut slopes may be in the order 60 m high.

The route then crosses 400 m requiring blasting and end haul before traversing a 200 m long bench with cut and fill. A large slide is present at 3.3 km and the route passes along the toe of this slide. From here the route traverses slopes between 45% and 70% with boulders over moraine or bedrock. A 100 m rock cut at 3.6 km will be needed with cut slopes up to 20 m high. The route then crosses Little Mess Creek at 4.0 km with a 68 m bridge.

From the crossing on Little Mess Creek the route remains on a lower bench (morainal deposit) and crosses several areas of wet material with signs of soil creep. There is a section of end haul at 5.7 km approximately 200 m long that will require drilling and blasting.

Arctic Creek (6.3 km) requires a 27 m bridge and end haul will be required for 145 m past the bridge, followed by cut and fill for 1 km. There are areas of wet-silt soils that will require 100% fill. Much of the alignment from Arctic Creek to Hits Creek (12.3 km) is located at the toe of the slope as the road passes through avalanche chutes (8.7 km and 10.5 km) for part of this section. Instability may be encountered at locations above the toe.

Many of the streams along this section have multiple channels at the crossings due to the junction of multiple creeks or braiding of the creeks in chutes.

The section from Arctic Creek to Hits Creek has gentle to moderately-steep slopes ranging from 15% to 50% on the uphill side; on average the slopes are 30% to 35%. The grade is gently rolling from -5% to +2%.

At 12.3 km a 7 m through cut is required. A 12 m bridge is planned to cross Hits Creek.

Itsh Creek (15.5 km) is a large incised gully and will require a 33 m bridge. Terrain from Itsh Creek is rolling with slopes between 30% and 55% with no end haul anticipated. The route crosses an alluvial fan from 16.5 km to 16.7 km with a 15 m bridge required at Jori Creek (16.6 km).

The section between Jori Creek and Alexander Creek is a large alluvial deposit where the fans of Jori Creek and Alexander Creek (17.4 km) meet. The section is mostly gravel, sand and flat. Alexander Creek is a large stream with a large active flood plain of gravel, cobble and upstream boulders. A 45 m bridge will be needed to span the entire active channel.

From Alexander Creek the grade rolls for 1100 m with through cuts up to 11 m high. There are two short sections of end haul, 150 m and 60 m in length. A talus slope is crossed at 18.7 km.

The route traverses relatively-gentle terrain ranging from 10% to 35% side slope with no major through cuts to Mikael Creek (21.2 km). This crossing will require a 12 m bridge. From Mikael Creek to 23.3 km there are a series of talus slopes and minor avalanche chutes that require intermittent full-bench construction.

The route from 23.3 km to Alicia Creek (24.5 km) is typical terrain for the area with slopes of 0% to 30% and areas requiring over landing. Alicia Creek is incised into the hillside indicating a morainal or alluvial blanket over bedrock requiring a 21 m bridge.

From Alicia Creek the route traverses a pass between the two creek systems before descending to Little Arctic Creek (24.5 km). Over landing will be needed for 450 m of this section due to wet soil conditions. There are several through cuts required along this section none over 8 m in depth. Side slopes to the crossing reach 60% and end haul is required for 50 m. Little Arctic Creek requires a 40 m bridge structure.

Terrain from Little Arctic Creek is very flat for 1600 m, not exceeding 5% side slope. From 700 m to 1200 m north of Little Arctic Creek the road crosses a very dry 100% sand area.

From 27.7 km to 29.2 km moist- to wet-morainal soils are encountered. There are several small through cuts that appear to be rippable.

The rock cut at 29.2 km may have cut slopes of 15 m to 17 m. A talus slope is crossed at 30.3 km with two more rock cuts at 31.0 km and 31.4 km; the route then drops to the Mess Creek flood plain.

A rock-fill causeway about 500 m long is required for the approach to the first of two bridges (40 m and 30 m) that are required to cross Mess Creek.

From Mess Creek the route rises up the side of Mt. LaCasse crossing Shift Creek (12 m bridge) and Big B Creek (10 m bridge). A switchback is located from 34.7 km to 32.5 km. The rock face at 35.9 km will require drilling and blasting. There are 200 m of wet ground at 36.1 km and avalanche chutes to cross at 36.3 km and 36.7 km. Another area of wet ground is encountered from 37.7 km to 38.5 km. The route terminates at Snipe Lake (39.5 km).

Photographs of the Mess Creek access route are shown in Figure 25.24 to Figure 25.28.

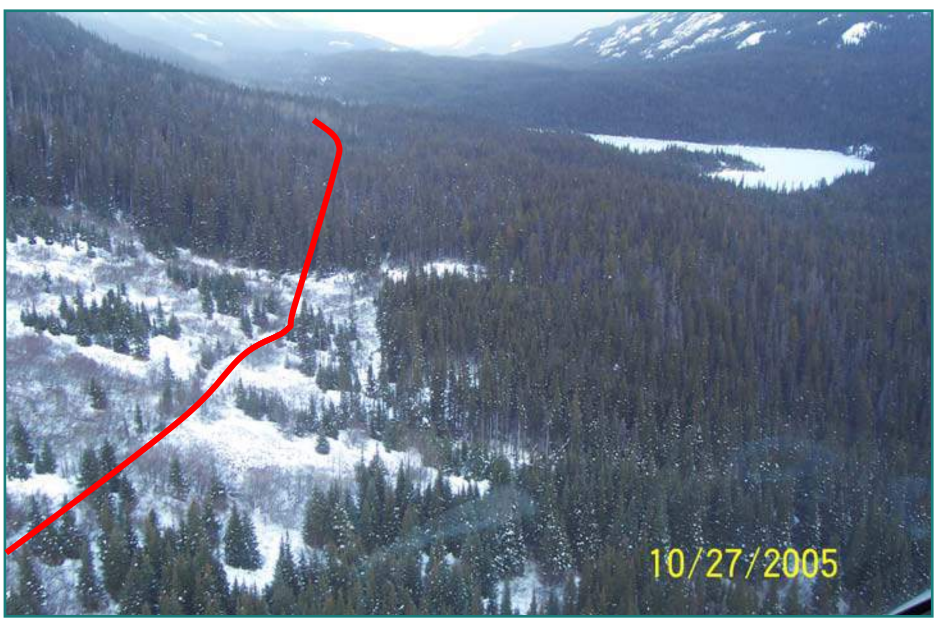


Figure 25.24 km 39 Looking Northeast

Figure 25.25 Mess Creek Flood Plain Road Centreline Crossing

Figure 25.26 Typical Angular Broken Rock Outcrop

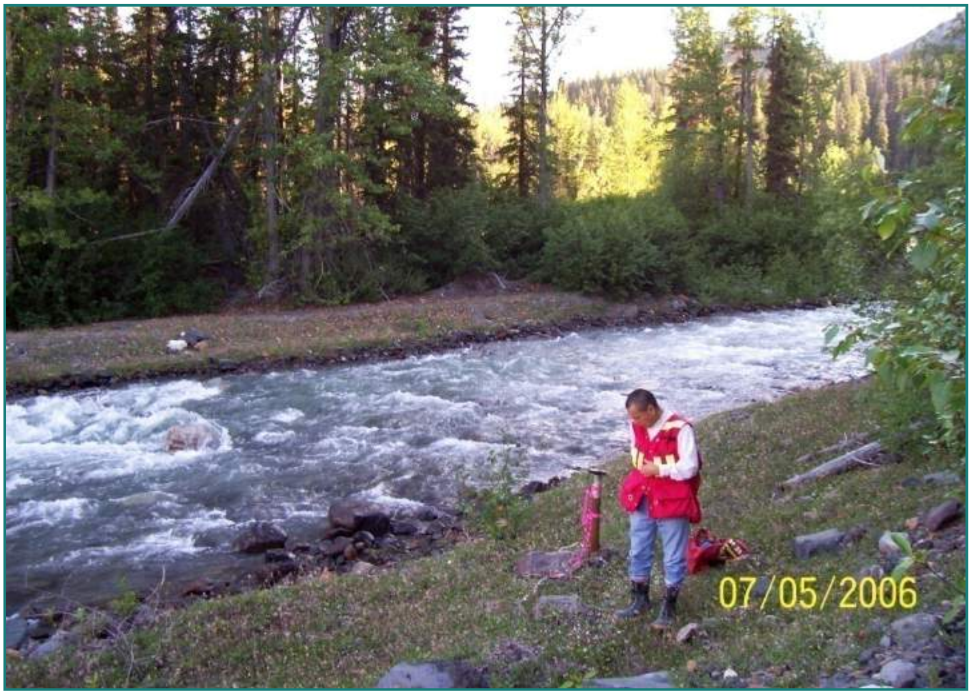


Figure 25.27 Little Arctic Creek Bridge Crossing

Figure 25.28 Tallus Slope along Lake Shore near km 22.0

25.2.1.4 *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 axel loading for trucks on British Columbia highways on a year-around 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. The typical road cross sections are provided in Appendix I of the McElhanney report. They depict the approximate range of construction procedures for varying terrain conditions. They also correspond to the construction categories defined in the Scoping Study report. The construction categories for the Mess Creek Access Road are given in Table 25.15.

Table 25.15
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.) or 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 along with the horizontal and vertical alignment with road grades and GPS coordinates for terrain features and construction constraints are provided in Appendix II of the McElhanney report. They were prepared based on the design specifications shown in Table 25.16 and field observations.

Table 25.16 Design Specifications	
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)	Add 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 yr 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 yr 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.

An estimate of road sections requiring drilling and blasting is listed in Table 25.17

Table 25.17 Mess Creek Route Rock Work Areas	
Location	**Estimated Length**
km 2.3	150 m
km 3.6	120 m
km 5.1	450 m
km 6.7	50 m
km 16.2	100 m
km 29.2	120 m
km 29.8	40 m
km 31.1	50 m
km 31.4	250 m
km 34.5	50 m
km 35.9	50 m
km 36.1	50 m
km 37.5	100 m

The bridges on the Mess Creek route listed in Table 25.18 are simple clear spans ranging from 10 m to 45 m in length. The crossing of Little Mess Creek will require a span of approximately 68 m. A typical bridge general arrangement is shown in Appendix I of the McElhanney report.

Table 25.18
Mess Creek Route Bridge Locations

Location	Crossing	Estimated Length
km 1.9	Little Mess Creek	18 m
km 4.0	Little Mess Creek	68 m
km 4.9	Tish Creek	15 m
km 6.3	Arctic Creek	27 m
km 12.3	Hits Creek	12 m
km 13.0	Unnamed Creek	10 m
km 15.5	Itsh Creek	33 m
km 16.6	Jori Creek	15 m
km 17.4	Alexander Creek	30 m
km 21.2	Mikael Creek	12 m
km 24.5	Alicia Creek	21 m
km 25.1	Little Arctic Creek	27 m
km 32.5	Mess Creek	40 m
km 32.8	Mess Creek	30 m
km 33.2	Shift Creek	10 m
km 34.1	Big B Creek	10 m

25.2.1.5 *Operating and Maintenance Requirements*

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 axel 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;
- Sub-grade repairs;
- Rock scaling;
- 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;
- Sign maintenance.

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.

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.

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.

25.2.1.6 *Summary*

The 2007 field reconnaissance and layout have provided a better understanding of the road design and construction constraints on the Mess Creek access route. Alternate access routes within the Mess Creek Valley were investigated and a preliminary road corridor was located with GPS control points established. All bridge and major culvert crossings and major fish streams were inspected and site surveys were conducted. Terrain features such as rock outcrops, talus slopes, and wetlands were recorded and geohazards such as rock falls, landslides and avalanche paths noted.

The preliminary Mess Creek route is flagged in the field for further detailed study, which should include:

- Survey of the road centreline and side slopes;
- Establishment of survey control along the road corridor;
- Completion of archaeological studies;
- Detailed geotechnical investigations of major crossings, rock cuts and unstable terrain;
- Completion of fisheries and environmental studies;
- Review of the Mess Creek crossings by a fluvial geomorphologist;
- Consultation with First Nations and other stakeholders;
- An engineered design from a road centerline location survey;
- Timber cruise of road right of way for cutting permit application.

25.2.2 Site Access – Air

The Schaft Creek project will be a fly-in, fly-out operation and as such will require a dedicated airfield. The existing airstrip will be upgraded to support the requirements of the Schaft Creek operations. It is anticipated the airfield will be developed early in the project in order to assist with the construction efforts and allow for transport of critical items as needed.

Early in the project development, prior to the site access road completion and upgrading the existing airstrip, helicopter support will be utilized for development/construction support. This will be based out of the Bob Quinn area.

25.2.2.1 *Airfield Terminal*

The airfield terminal will have a 450 m^2 footprint set on a 15 m by 30 m foundation that is nestled into the terrain and set back an appropriate distance from the airstrip itself but on the edge of the apron. In addition to a comfortable waiting area for personnel coming to and going from the project, the terminal will house restrooms, a small kitchen, a small meeting room, air traffic control tower, computer equipment room, and several small offices.

25.2.2.2 *Runway Maintenance Equipment and Emergency Vehicle Storage*

Runway maintenance equipment and emergency vehicle storage will be housed in the drive-out basement underneath the terminal building.

25.2.3 Site Layout

Selection of the location of process plant and ancillary facilities will be based on the following criteria:

- Preference for gravity handling of tailings between the process plant and the tailings storage facility (TSF);
- Potential environmental and social impacts;
- Proximity to the mine to minimize ore transportation cost;
- Favourable topography allowing gravity flow within and between process facilities and to minimize mass earthworks;
- Proximity to reclaim water from tailings;
- Preference of fresh water sources;
- Plant elevation above sea level.

Preference for gravity handling of tailings to the impoundment area was given primary consideration in locating the processing facilities due to the high cost of operating a pumped system and potential environmental impact of a pressurized line. This dictated that the plant be at a higher elevation than the ultimate TSF elevation, conflicting with a desire to minimize the impact of noise, night time light emission and visibility relative to the nearby provincial park. Consideration was also given to the proximity of the plant to potential avalanche chutes, micro-climates and proximity to both Schaft Creek and Mess Creek. The result among these requirements put the plant at an elevation where tailings will be gravity fed for the life of the mining operation.

Final plant site and TSF locations will be determined during the preliminary feasibility study.

Next in importance is minimizing ore transportation distance from the mine to the process plant. The primary crusher will be located as close to the mine as possible to minimize truck haulage distance, dictating the conveying of crushed ore to the concentrator via belt conveyors.

In order to allow gravity flow within and between process facilities and to minimize mass earthworks at the process plant, a 50 m drop in elevation from the base of the crushed ore stockpile to the tailings thickeners is desirable. The topography at the preferred plant location used to develop costs for the PFS allows for this requirement. A final plant site location will be determined during the feasibility study.

The fresh water source is approximately 2 km from the plant site. Reclaim water will be pumped from a floating barge within the tailings storage facility.

The ancillary facilities are located in relation to the main facilities for convenience, such as the truck shop is located adjacent to the pit and the primary crusher. Another consideration in some cases is isolation. For example the permanent camp is located far enough away from the operations to minimize the impact of plant noise.

The plant is located at the terminus of a purpose-built access road. The roads are designed to allow for concentrate truck haulage and access for delivery of reagents with tractor/trailer and associated maneuverability.

Copper Fox is very cognizant of potential concerns about any visual impact the mine site may have from the edge of Mt. Edziza Provincial Park; therefore, the company requested artistic renderings of what the mine site will look like from the perimeter of the park. Renderings were done from two vantage points at the park perimeter as shown in Figure 25.29. The rendering from vantage point two is actually atop the ridge just outside the perimeter as the true boundary lies in the valley to the east of the vantage point and sighting of the mine would be impossible from this boundary. These renderings may be found in Section 26 (Illustrations) of this report.

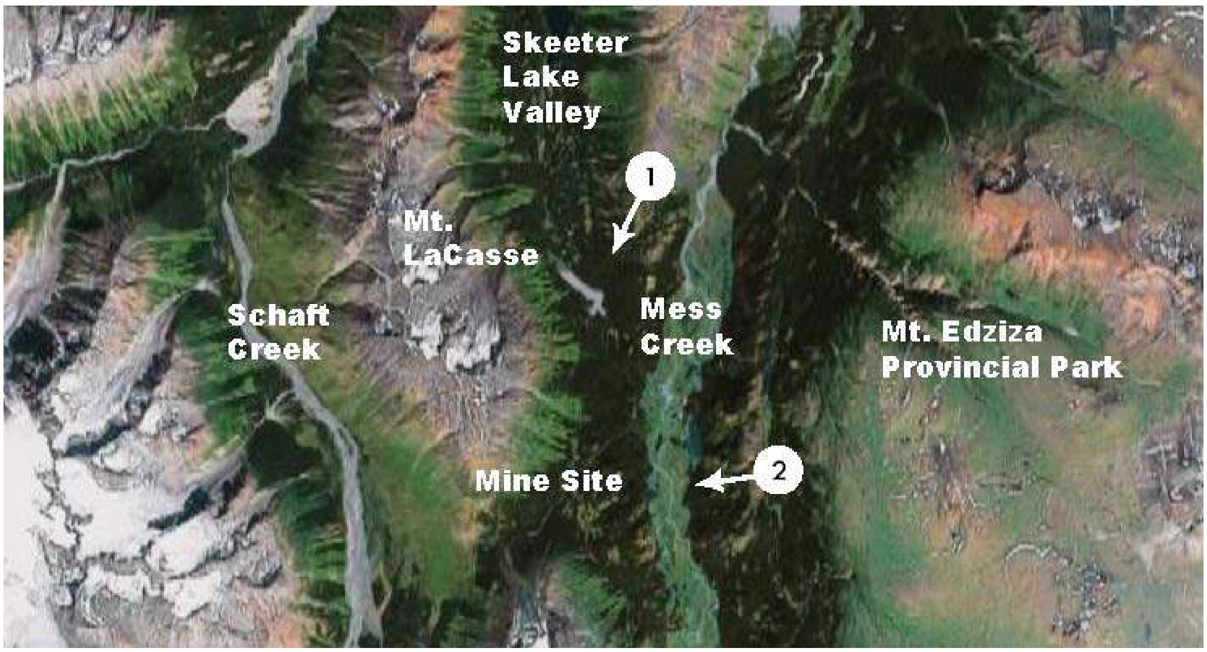


Figure 25.29 Artistic Renderings Vantage Points

25.2.4 Site Roads

The site roads for the project have been designed by Samuel Engineering (SE). A total of four access roadways to and from the four areas of the project are needed. These are the Process Plant, the Administration Building, the Permanent Man Camp, the Construction Man Camp and a portion to the TSF. In addition there are five Service Roads and two driveways in the civil design to allow vehicular access around and into the various areas of the processing facilities. The design parameters used in the roadway design are as follows:

- A crowned, two-way, 10 m roadway with 2 to 3.5 m drive lanes and 2 to 1.5 m shoulders;
- 2% cross slopes on a 200 mm layer of minus 7 mm gravel base;
- A Vee-shaped side drainage ditch for the cut conditions with a 3:1 inside slope and a 1.5:1 outside slope;
- Use of a 1 m high safety berm for any fill areas one m above existing grade;
- Cut and fill slopes at 1.5:1;
- Minimum longitudinal grade of 0.60% and a maximum longitudinal grade of 10.50%.

Throughout the roadway routes, drainage culverts have been placed for the diversion of the intercepted existing, off-site stream flows thereby perpetuating the historic flow paths. With the preliminary drainage study work compiled to date the culvert sizes range from 61 mm to 274 mm in diameter. In a few locations multiple barrel installations have been utilized to keep the sizes smaller and in turn easier to obtain and place. These have been placed with the use of Prefab end sections and rock rip-rap discharge aprons on the outlet ends.

The plant site areas are graded with a minimum 2% slope at all surfaces to provide positive drainage away from all buildings and with 1.5:1 cut and fill slopes for the building pad development. The plant site areas are to be surfaced with a minimum 200 mm layer of gravel as well as being provided with the 1 m high safety berm along all fill slopes.

The Process Plant site grading has been designed to allow for the on-site runoff that could be potentially contaminated, to be intercepted and conveyed via perimeter swales to three containment ponds placed within the plant site. These pond sites were developed from the low areas located at the toe of the embankments inherent from the roadway grading that surrounds the plant site. This approach to containing the on-site runoff shall allow the plant site to be considered a zero discharge facility. In addition to drainage conveyance the swales and ponds are anticipated to be used for snow stockpiling during the winter periods.

25.2.5 Site Buildings

25.2.5.1 *Buildings*

The following buildings will be part of the mine site facilities:

- Primary Crushing;
- Truck Shop;
- Crushed Ore Storage;
- Mill and Concentrator Building;
- Pebble Crushing;
- Reagent Storage and Mixing;
- Mill Maintenance Shops and Warehouse;
- Copper Concentrate Filtration Plant;
- Copper Concentrate Storage;
- Administration Offices;
- Laboratory;
- Security Gate House;
- Access Road Equipment Maintenance and Warehouse;
- Mancamp Facilities;
- Airfield Terminal and Runway Equipment Maintenance and Storage;
- ANFO and Prill Storage;
- Explosives Plant;
- Explosives Storage.

Primary Crushing

The primary crushing building will be structural steel with metal siding and roofing. The building and equipment foundations will be reinforced concrete. This structure will rest above a sub-grade primary gyratory crusher with access from two sides to allow two haul trucks to dump run-of-mine (ROM) ore simultaneously. Below the crusher will be a pocket with the same capacity as the dump pocket providing a continuous head of material to the crusher discharge feeder and conveyor. The remaining above-grade rooms will include a crusher control room and a main crusher shaft repair bay. A rock breaker will be situated at the edge of the dump pocket to split oversize boulders.

Truck Maintenance Facility

The truck shop will be a large complex fabricated from structural steel with metal siding and roofing and will serve multiple purposes.

In addition to providing an area for four 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. The building and equipment foundations will be reinforced concrete.

Crushed Ore Storage

The crushed ore stockpile storage facility will consist of a structural steel A-frame which will cover the stockpile and provide support for the shuttle conveyor that will stack the crushed ore in an elongated stockpile with a total capacity of 290,000 t. Approximately 100,000 t of the total capacity will be "live", which represents the portion of the stock that will flow down by gravity into the eight reclaim hoppers below the pile. The majority of the A-frame will be covered by metal roofing.

Two parallel reinforced concrete reclaim tunnels will be located below the crushed ore stockpile.

Each tunnel will house four reclaim hoppers (three operating and one standby) and will have emergency exits. Structural steel platforms covered with metal grating in the tunnels will provide support and access for the feeders.

Mill and Concentrator Building

The mill and concentrator building will be structural steel with insulated metal siding and roofing, while the building and equipment foundations will be reinforced concrete. Structural steel platforms covered with metal grating will provide support and access for the equipment. The building will be divided into three main sections housing the following functions:

- Grinding;
- Bulk copper/molybdenum flotation, regrind, and concentrate thickening;
- Copper concentrate thickening;

- Molybdenum flotation, regrind, filtration, drying, and truck load out.

The first building section contains the SAG mills, ball mills, and associated equipment such as cyclones. The second section contains the bulk rougher flotation cells, while the third section houses the regrind mills, the cleaner flotation cells and the complete molybdenum circuit. Adjacent to the third section resides the bulk and copper concentrates thickeners.

Pebble Crushers

The pebble crushers building will be located near the mill and concentrator building and will be fabricated from structural steel with non-insulated metal siding and roofing, while the building foundation will be reinforced concrete. Structural steel platforms covered with metal grating will provide support and access for the crushers.

Reagents Storage and Mixing

The reagents storage and mixing building will be a conventional steel-framed structure with insulated metal siding and roofing. The building columns will be supported by spread footings and a reinforced concrete floor will be constructed for the entire building.

Structural steel platforms covered with metal grating will provide support and access for the various tanks and equipment.

Mill Maintenance Shops and Warehouse

The maintenance shops and warehouse building will be a conventional steel-framed structure with non-insulated metal siding and roofing. The building columns will be supported by spread footings and a reinforced concrete floor will be constructed for the entire building. The lower level will have a paint shop, welding shop, small vehicle repair shop, machine shop, carpentry shop, and electrical shop, tool crib, office space, and changing room. The warehouse comprises the remainder of the lower level. A loading dock serves as a shipping/receiving area for the warehouse. The upper level consists of additional office space, a lunch room, and training room. Aprons will be provided at all truck entrances.

Copper Concentrate Filtration Plant

The copper concentrate filtration building will be located near the mill and concentrator building and will be fabricated from structural steel with insulated metal siding and roofing, while the building foundation will be reinforced concrete. Structural steel platforms covered with metal grating will provide support and access for the equipment.

Copper Concentrate Storage

The copper concentrate storage facility will consist of a structural steel A-frame which will cover the stockpile and provide support for the conveyor that will stack the copper concentrate in a conical stockpile with a total capacity of 6,700 t. The majority of the A-frame will be covered by metal roofing.

Administration Offices

The administration building will be a conventional steel-framed structure with insulated metal siding and roofing. The building columns will be supported by spread footings and a reinforced concrete floor will be constructed for the entire building. The building space will be divided into a reception area, numerous offices, open cube farms, conference rooms, men's and women's changing rooms, a lunch room, training room and infirmary with an ambulance receiving area.

Laboratory

The laboratory will be a conventional steel-framed structure with insulated metal siding and roofing. The building columns will be supported by spread footings and a reinforced concrete floor will be constructed for the entire building. The laboratories include a wet lab, a metallurgical lab and a fire assay lab, as well as a sample drying and preparation lab.

Office space, a lunch room, and records room augment the labs. A loading dock receives the samples and services the sample preparation area.

Security Gate House

The security gate house will be located on the plant entrance road and will be a modular construction.

Access Road Equipment Maintenance and Warehouse

The access road equipment maintenance shops and warehouse building will be a conventional steel-framed structure with non-insulated metal siding and roofing. The building columns will be supported by spread footings and a reinforced concrete floor will be constructed for the entire building. The lower level will have a paint shop, welding shop, small vehicle repair shop, machine shop, carpentry shop and electrical shop, tool crib, office space and changing room. The warehouse for sand and de-icer comprises the remainder of the lower level. A lockable vault area will be used for storing an Avalauncher and the associated shells. A storage area outside the building is used for riprap and gravel storage. A loading dock serves as a shipping/receiving area for the warehouse. The upper level consists of additional office space, a lunch room and training room. Aprons will be provided at all truck entrances.

Mancamps, Dormitories, and Kitchen/Infirmary

These facilities will be constructed of pre-fabricated pod-style components branching off a central corridor linking all of the dormitories, the recreation facilities and the kitchen/infirmary. The construction (temporary) camp will be sized for occupancy by 2,100 persons, while the permanent camp size will be designed for a more modest 700-person occupancy; however, it is anticipated that approximately 550 persons will actually inhabit the camp at any given point in time.

ANFO and Prill Storage

The ANFO and prill storage building will be a Type-1 permanent storage facility and shall be bullet resistant, weather-resistant, theft-resistant and well ventilated compliant with 18 U.S.C. Chapter 40 of the Bureau of Alcohol, Tobacco, and Firearms Orangebook.

Explosives Plant

The explosives plant will be constructed within the guidelines given by the Explosive Act – Natural Resources Canada. Furthermore, the plant will be constructed on a prepared flat surface with a perimeter fence, and will contain: Mobile Manufacturing Unit (MMU) truck storage, MMU wash bay, parts storage, raw storage, process area, AN Prill loading/unloading areas, and an office building with washroom and break room.

Explosives Storage

The explosives storage building will be a Type-1 permanent storage facility and shall be bullet resistant, weather-resistant, theft-resistant, and well ventilated compliant with 18 U.S.C. Chapter 40 of the Bureau of Alcohol, Tobacco, and Firearms Orangebook.

25.2.6 Water Supply Distribution

The following water systems are included in the design 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 concentrate thickener overflows and from reclaim water from the tailings storage facility. The remaining water will be obtained from fresh water make-up.

Process water will be supplied from two centrally-located tanks.

Fresh water will be required for mill make-up 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 shall consist of a packaged skid-mounted unit suitable for the number of personnel on site during site development and construction and for the site conditions. The unit shall have high rate settling, adsorption clarification, mixed media filtration and UV disinfection functions well-suited for variable water conditions including high turbidity, high color and cold water. The resulting water shall have a process guarantee to meet all local, provincial, and federal mandates for potable water quality. A potable water storage tank will provide a one-day storage capacity.

The waste water treatment system shall consist of a packaged skid-mounted unit suitable for the site conditions and anticipated flow rate of water released from the tailings storage facility (TSF) back into the environment. The unit shall perform a continuous flow chemical precipitation process capable of handling all potentially generated acids, heavy metals as well as all small suspended solids not precipitated in the TSF itself in a manner compliant with all local, provincial, and federal codes having jurisdiction over the effluent.

Water will be available to the fire-water 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 fire-water reserve will not be accessible to the mill fresh water system. A two-hour fire-water reserve will be ensured by piping the fire water from the bottom of the storage tank and the fresh/potable water systems from higher in the tank in order to provide guaranteed fire-water availability.

25.2.7 Power Distribution

The main substation at the mine will be fed from a 138 kV overhead transmission line. The substation will contain two main transformers to step power down to 23.9 kV to feed the switchgear lineup. 23.9 kV power from the lineup will directly feed the 2 SAG and 4 ball mills.

The 23.9 kV switchgear lineup will also feed several other transformers within the substation that will step down power to either 4160 V or 480 V. Two 4160 V overhead lines will leave the substation to distribute power to remote areas of the plant.

25.2.7.1 *Pit Area*

Power to the pit area will be fed by a 4160 V overhead line which will directly feed a 4160 V switchgear lineup.

25.2.7.2 *Process Area*

Power to the process area will be fed underground from the main substation at 23.9 kV, 4160 V and 480 V depending on the equipment.

25.2.7.3 *Crushing Area*

Power to the crushing area will be fed underground from the main substation at 4160 V and 480 V depending on the equipment.

25.2.7.4 *Tailings Reclaim*

Power to the tailings reclaim area will be fed by a 4160 V overhead line which will directly feed a 4160 V switchgear lineup.

25.2.7.5 *Water Well Location*

Power to the water well location will be fed by a 4160 V overhead line which will directly feed a 4160 V switchgear lineup.

25.2.8 Fire Protection

A dedicated tank will be provided for fresh water and firewater at each of the following three locations: the airfield, the mancamp, the administration building and the plant site. Separate firewater loops will be fed by a diesel-driven pump and an electric jocky pump at each location. The fresh-water discharge connection is at an elevation above the tank bottom and ensures the remaining volume will be available for firewater purposes. Distribution will consist of a buried ring main around major facility buildings with hydrants and stand pipes. Allowances have also been made for portable handheld extinguishers for localized protection.

25.2.9 Sanitary Sewage

The overall Process Plant site infrastructure encompasses a vast amount of area to support the anticipated open pit mining operations. After some research considering the cold-weather effects of the project site, the distance between the four areas of infrastructure, the occupancy load of each area and the selection of suitable locations for septic leach fields considering the topography of the project site; the use of standard septic system for the domestic wastes and sewage for this location is deemed viable. Thus four individual septic disposal systems (ISDS) are planned for this project.

With an assumed percolation rate of 8 minutes per cm, an average daily waste water demand varying between 75 to 189 L/d and a minimum drain pipe slope of 1.4% the four septic systems have been preliminarily designed. Each utilizes a prefabricated septic tank located near the buildings for the primary treatment and a leach field comprised of gravel-filled trenches for the secondary treatment into the sub-grade soils. It is anticipated that each of the four systems will require dosing. The information about each of the individual septic disposal systems is listed below.

1. ISDS #1 serves the Truck Shop and the Grinding Building using a 9.5 m^3 and a 22.7 m^3 septic tank respectively. The tanks discharge into a single septic leach field of an area of 718 m^2 with 10 leach trenches that are each 30 m long.

2. ISDS #2 serves the Permanent Man Camp facilities using a 94.6 m^3 septic tank. The tank discharges into three septic leach fields with a total area of 2,111 m^2 using 9 leach trenches that are each 32 m long for each field.

3. ISDS #3 serves the Lab & Administration Buildings using an 11.4 m^3 septic tank. The tank discharges into a single septic leach field of an area of 253.3 m^2 with 4 leach trenches that are each 26 m long.

4. ISDS #4 serves the Construction Man Camp facilities using four 718 m^2 septic tanks. The tanks discharge into ten septic leach fields with a total area of 8,466 m^2 using 10 leach trenches that are each 35 m long for each leach field.

25.2.10 Communications

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.

25.2.11 Property Security and Medical Facility

Site security and medical services include a medical facility, security building, fire truck and ambulance.

25.3 Tailings, Waste Rock Disposal and Water Management

25.3.1 Introduction

Copper Fox Metals Inc. (Copper Fox) appointed Knight Piésold Ltd. (KPL) to undertake the preliminary feasibility design of the tailings, waste rock and water management components of the project. More detailed information may be found in the *Report on Pre-Feasibility Design of the 100,000 Tonnes per Day Tailings Storage Facility*.

25.3.1.1 *Options Study*

Three potential Tailings Storage Facility (TSF) sites, referred to as Options A, B and C were identified during the scoping stage of the project. As part of the design of the tailings management components of the project for the Preliminary Feasibility Study (PFS), KPL undertook a conceptual-level assessment of these three options in addition to two alternative options within the Option A valley referred to as Option Aa and Option A1. The options study concluded that Option A was the preferred arrangement. Although Options Aa and A1 offered some potential cost savings over Option A, the latter was adopted as the preferred site for the purposes of this study, as requested by Copper Fox.

25.3.1.2 *Waste Rock Dumps*

Up to 1.6 G t of waste rock will be generated during the mining of the Schaft Creek open pit over the 23 year mine life (assuming a strip ratio of 2:1). Potentially Acid Generating (PAG) waste, for the purposes of this study assumed to be 10% of total waste rock, will be appropriately managed at the open pit. Non-Potentially Acid Generating (NPAG) rock will be stored in three conventional surface waste dumps, located west and south of the open pit and east of Schaft and Hickman Creeks. Waste dumps will be constructed using conventional dumping techniques and will be up to 400 m high.

25.3.1.3 *Tailings Storage Facility Design*

Approximately 812 M t of ore will be processed at an envisaged throughput of 100,000 tpd. Tailings will be stored in an engineered storage facility located in the Skeeter Lake Valley, north of the open pit.

The major components of the TSF are listed below:

- Three main earth-rockfill, zoned embankments, referred to as the North, South and West embankments;
- Seepage collection systems;
- Access roads along the eastern and western sides of the valley;
- Surface water management structures;
- Tailings delivery system (by others);
- Tailings distribution system, and;
- Reclaim water system (by others).

The facility embankments will be constructed in stages throughout the life of the project using a combination of NPAG waste rock materials, local borrow and cycloned tailings. The starter facility, designed to accommodate tailings production for a period of two years, will comprise a single 60 m high embankment constructed at the northern end of the facility with fill sourced from local borrow pits. This embankment will be raised progressively throughout the life of the facility using cycloned tailings, to a maximum ultimate height of 125 m. Later in the life of the facility, the West and South embankments will be constructed and raised progressively to ultimate maximum heights of 60 m and 45 m respectively. The West embankment will be constructed in a similar manner to the North embankment, using local borrow and cycloned sand, while the South embankment, being relatively close to the pit, will be constructed largely from NPAG mine waste.

An HDPE geomembrane will be installed on the upstream face of the starter embankments to minimise seepage through the structures. The geomembrane will be tied into a bentonite-cement slurry cut-off wall installed beneath the embankment upstream toe. The slurry cut-off wall will extend down through the permeable sand and gravel foundation and will be keyed into competent bedrock to form a positive hydraulic cut-off. Low permeability core zones will be implemented in the subsequent staged expansions to the tailings embankments.

25.3.1.4 *Tailings Deposition*

The discharge of tailings into the TSF will initially be from a series of large-diameter, valved off-takes located along the North Embankment. Later in the operational life of the facility, tailings distribution will also take place from the South and West embankments and from the western side of the facility. The proposed deposition pattern will result in a dish-shaped deposit with the supernatant pond located centrally within the facility, remote from all confining embankments. Water will be reclaimed from the tailings pond by one or more barge-mounted pump stations.

25.3.1.5 *Surface Water Management*

A peripheral drainage system will be constructed around the facility to minimize the inflow of water to the TSF. The system will comprise two major diversion drains constructed on either side of the valley, immediately above the ultimate facility elevation. Each drain will terminate in a major drop structure which will convey collected water downstream of the North embankment to the Skeeter Creek watercourse.

The model indicates that prior to commissioning the volume of water collected in the starter facility will be in the range of 10 M m^3 to 25 M m^3 which is equivalent to between four and ten months of process water requirements. Based on this model, make-up water from external sources should not be required for start up.

The modeling indicates that during operations under average conditions the facility water balance will be positive, generating on average an annual excess of around 4 M m^3 to 7 M m^3 of water.

The water balance will fluctuate on a monthly basis, being negative for 5 to 6 months of the year during the low precipitation months of the winter and being strongly positive in the spring and summer months.

25.3.1.6 *Facility Water Balance*

A preliminary water balance model was developed for the TSF to assess the various inflows to and outflows from the facility on a monthly basis and to determine the variation in supernatant pond volume and required volumes of make up or discharge.

The model indicates that prior to commissioning the volume of water collected in the starter facility will be in the range of 10 M m^3 to 25 M m^3 which is equivalent to between four and ten months of process water requirements. Based on this model, make-up water from external sources should not be required for start up.

The modeling indicates that during operations under average conditions the facility water balance will be positive, generating on average an annual excess of around 4 M m^3 to 7 M m^3 of water. The water balance will fluctuate on a monthly basis, being negative for 5 to 6 months of the year during the low precipitation months of the winter and being strongly positive in the spring and summer months.

25.3.1.7 *Scope of Work*

This report presents the preliminary feasibility level design of the Tailings Storage Facility (TSF) and includes the following:

- Review of site characteristics including hydrometeorology, regional geology, hydrogeology and seismicity;
- Options study to identify the preferred site of the three selected by Others during earlier studies;
- Results of testwork on representative tailings material as supplied by Copper Fox to determine tailings physical properties;

- Review and evaluation of the results of geotechnical investigations carried out at the TSF site by Others;
- Development of a suitable TSF layout and operating strategy for tailings disposal;
- Development of a preliminary filling schedule for the TSF to determine initial and on-going staged embankment construction requirements;
- Design of TSF embankments, including identification of potential borrow sources and stability analyses;
- Examination of surface drainage and diversion requirements;
- Development of a preliminary water balance for the TSF to provide a running total of the inputs and outputs from the facility and water availability for reclaim;
- Preliminary seepage analyses and consideration of seepage control, recovery, and recycle systems;
- Preliminary layouts and operating requirements for tailings delivery and water reclaim systems (preliminary feasibility design and costing by Others);
- Site water management and sediment control;
- Estimated TSF construction quantities and costs;
- Evaluation of the TSF foundations and identification of potential construction materials.

25.3.2 Site Characteristics

The design of the Tailings Storage Facility (TSF) takes into account the topography, geology and climate. Additionally, the TSF design considers:

- The dam classification;
- Seismicity;
- Storm events.

25.3.2.1 *Dam Classification*

A preliminary dam classification has been carried out to enable appropriate design earthquake and flood events to be determined for the Tailings Storage Facility (TSF). The selection of appropriate design earthquake and flood events has been based on classification of the tailings dam using criteria provided by the Canadian Dam Association's (CDA) *Dam Safety Guidelines* (2007).

The dam classification scheme defined by the CDA Dam Safety Guidelines is reproduced as Table 25.19 of this report. Classification of a tailings dam is carried out by considering the potential incremental consequences of a failure. That is, those consequences of dam failure which are in addition to the impacts that would occur from the earthquake or flood event without the facility being in place. The consequences of failure include loss of life and environmental and economic impacts.

The potential for loss of life is likely minor following a dam failure but cannot be discounted, particularly during operations at the TSF. If failure resulted in the release of tailings and/or process water it may have a significant environmental impact on downstream watercourses. The economic consequences (including clean-up, repair and remedial works) would also be high. Consequently, a VERY HIGH dam classification has been assigned to the proposed

TSF. This dam classification should be reviewed for future design studies and revised if appropriate.

Table 25.19
Dam Classification

		Incremental losses		
Dam Class	**Population at risk [1]**	**Loss of Life [2]**	**Environmental and Cultural Values**	**Infrastructure and Economics**
Low	None	0	Minimal short-term loss No long-term loss	Low economic losses; area contains limited infrastructure or services
Significant	Temporary Only	Unspecified	No significant loss or deterioration of fish or wildlife habitat Loss of marginal habitat only Restoration or compensation in kind highly possible	Losses to recreational facilities, seasonal workplaces, and infrequently used transportation routes
High	Permanent	10 or fewer	Significant loss or deterioration of important fish or wildlife habitat Restoration or compensation in kind highly possible	High economic losses affecting infrastructure, public transportation, and commercial facilities
Very high	Permanent	100 or fewer	Significant loss or deterioration of critical fish or wildlife habitat Restoration or compensation in kind possible but impractical	Very high economic losses affecting important infrastructure or services (e.g., highway, industrial facility, storage facilities for dangerous substances)
Extreme	Permanent	More than 100	Major loss of critical fish or wildlife habitat Restoration or compensation in kind impossible	Extreme losses affecting critical infrastructure or services (e.g. hospital, major industrial complex, major storage facilities for dangerous substances)

Notes:
1. Definitions for population at risk:
   None – There is no identifiable population at risk so there is no possibility of loss of life other than through unforeseeable misadventure.
   Temporary – People are only temporarily in the dam-breach inundation zone, e.g. seasonal cottage use, passing through on transportation routes, participating in recreational activities.
   Permanent – The population at risk is ordinarily located in the dam-breach inundation zone, e.g. as permanent residents; three consequence classes (high, very high, extreme) are proposed to allow for more detailed estimates of potential loss of life to assist in decision-making if the appropriate analysis is carried out.
2. Implications for loss of life:
   Unspecified – The appropriate level of safety required at a dam where people are temporarily at risk depends on the number of people, the exposure time, the nature of their activity and other conditions. A higher class could be appropriate, depending on the requirements. However, the design flood requirement, for example, might not be higher if the temporary populations is not likely to be present during the flood season.

25.3.2.2 *Sesmicity*

Seismic Hazard

A preliminary review of the regional seismicity has been carried out to enable selection of appropriate design earthquake events for the TSF for use in stability assessment.

The region of coastal northwest British Columbia and the southwest Yukon Territory is one of the most seismically-active areas in Canada. The seismic hazard in the region is also influenced by the seismically active region of southeast Alaska. The coastal region has experienced many large earthquakes, including events in the range of Magnitude 7.0 to 8.0. These earthquakes are typically associated with the Queen Charlotte fault, Fairweather fault (the northern extension of the Queen Charlotte fault) and the eastern Denali fault system. In 1958 a Magnitude 7.9 earthquake occurred along the Fairweather fault.

However, the level of seismicity within northwest British Columbia reduces significantly with distance from the coast. The most significant inland zone of seismicity follows the Dalton and Duke River segments of the Denali fault zone through the southwest Yukon.

Review of historical earthquake records and regional tectonics indicates that the Schaft Creek Project site is situated in a region of low seismic hazard. To provide seismic ground motion parameters for design of the TSF a probabilistic seismic hazard analysis has been carried out using the database of Natural Resources Canada (NRC).

The results are summarized in Table 25.20 in terms of earthquake return period, probability of exceedance (for a 23 year design operating life) and the median average maximum acceleration. For a return period of 475 years, the corresponding maximum acceleration is 0.06g, confirming a low seismic hazard for the site. Estimated values of the mean maximum acceleration are also included in Table 25.20.

Table 25.20
Summary of Probabilistic Seismic Hazard Analysis

Return Period (Years)	Probability of Exceedance [1] (%)	Maximum Median Acceleration [2] (g)	Maximum Mean Acceleration [3] (g)
100	21	0.02	0.03
475	5	0.06	0.07
1,000	2	0.07	0.09
2,500	1	0.10	0.12
5,000	0.5	0.15	0.18

Notes:
1. Probability of exceedance calculated for a design operating life of 23 years.
   $q = 1 - \exp(-L/T)$
   where
   q = probability of exceedance
   L = design life in years
   T = return period in years
2. Maximum accelerations are median average values on very dense soil or soft rock. (Site Class C, as defined by National Building Code of Canada, 2005.)
3. Mean values of maximum acceleration estimated as 1.2 x median values.
4. Maximum accelerations provided by the Natural Resources Canada seismic hazard calculation, with the exception of the 5,000 year value which was obtained by extrapolation.

The CDA Dam Safety Guidelines recommend that the mean maximum acceleration value should be used for dam design. This is likely to be similar or slightly higher (by about 10% to 20%) than the median value provided by NRC.

Consequently, estimated mean maximum acceleration values have been adopted for the design earthquake events used in seismic stability analyses.

Design Earthquakes

Consistent with the current design philosophy for geotechnical structures such as dams, two levels of design earthquake have been considered: the Operating Basis Earthquake (OBE) for normal operations, and the Maximum Design Earthquake (MDE) for extreme conditions (ICOLD, 1995). Values of maximum ground acceleration and design earthquake magnitude have been determined for both the OBE and MDE.

The OBE is usually selected from the results of a probabilistic hazard evaluation.

The hazard level selected for the OBE is often chosen as the earthquake with a 10% probability of exceedance in 50 years (with a corresponding return period of 475 years). For design of the TSF, the OBE has been taken as the 1 in 475 year return period event. The probability of exceedance for this event is only 5% for a 23 year operating period. The mean average maximum acceleration is estimated to be 0.07g for the 1 in 475 year earthquake.

A design earthquake magnitude of 7.5 has been selected for the OBE, based on a review of regional tectonics and historical seismicity. The TSF would be expected to function in a normal manner after the OBE.

An appropriate MDE for embankment design has been selected, based on the VERY HIGH dam classification defined for the TSF and the criteria for design earthquakes provided by the CDA Dam Safety Guidelines. The suggested design earthquake events for each dam classification are presented in Table 25.21 (reproduced from the CDA Dam Safety Guidelines). The CDA guidelines require that a VERY HIGH dam classification be designed for a probabilistically derived event (known as the Earthquake Design Ground Motion) having an annual exceedance probability (AEP) of 1/5,000. Consequently, the MDE selected for the TSF is the 1 in 5,000 year earthquake. The mean average maximum acceleration is estimated to be 0.18g for the 1 in 5,000 year earthquake. A design earthquake magnitude of 8.0 has been selected for the MDE, based on a review of regional tectonics and historical seismicity. Limited deformation of the tailings embankment is acceptable under seismic loading from the MDE, provided that the overall stability and integrity of the TSF is maintained and that there is no release of stored tailings or water (ICOLD 1995).

Table 25.21 Suggested Design Flood and Earthquake Levels		
	Annual Exceedance Probability (AEP)	
Dam Class[1]	Inflow Design Flood (IDF[2])	Earthquake Design Ground Motion (EDGM[3])
Low	1/100	1/500
Significant	Between 1/100 and 1/1,000[4]	1/1000
High	1/3 between 1/1,000 and PMF[5]	½,500[6]
Very High	2/3 between 1/1,000 and PMF [5]	1/5,000[6]
Extreme	Probably Maximum Flood (PMF)[5]	1/10,000

Notes:
1. As defined in Table 10.1 Dam Classification
2. Extrapolation of flood statistics beyond 1/1,000 year flood (10^{-3} AEP) is discouraged
3. AEP levels for EDGM are to be used for mean rather than median estimates for the hazard.
4. Selected on the basis of incremental flood analysis, exposure and consequences of failure.
5. PMF has no associated AEP. The flood is defined as “1/3 between 1/1,000 and PMF” or “2/3 between 1/1,000 year and PMF has no defined AEP.
6. The EDGM value must be justified to demonstrate conformance to societal norms of acceptable risk. Justification can be provided with the help of failure initiated by a seismic event. If the justification cannot be provided, the EDGM should be 1/10,000.

25.3.2.3 *Design Storm Events*

Selection of an appropriate Inflow Design Flood (IDF) is required to carry out a safety assessment of the TSF and to determine flood storage requirements. The size of the IDF increases with increasing consequences of failure. Based on a VERY HIGH hazard classification assigned to the TSF, an appropriate IDF is a probabilistically-derived event with a return period of two thirds between the 1/1000 year and the Probable Maximum Flood (PMF). However, for this study, the deterministically-derived 24-hr PMF has been conservatively selected to represent the IDF for design of the TSF. This large storm event has been selected in recognition that the TSF will not have a spillway for discharge of flood flows during operations.

The PMF is the flow resulting from the most severe combination of precipitation and basin hydrological conditions, including snowmelt. The most extreme precipitation is referred to as the Probable Maximum Precipitation (PMP), which is the result of the worst possible meteorological conditions. The PMP estimate is by definition “theoretically the greatest depth of precipitation for a given duration that is physically possible over a given size storm area at a particular geographic location at a certain time of year” (WMO, 1986). There is no probability associated with such an event, but a likely comparison would be a storm with a return period greater than 20,000 years.

The Hershfield (1977) method was used to estimate a PMP of 264 mm from the 24 hr maximum precipitation data available in the Rainfall Frequency Atlas for Canada (RFAC). The mean and standard deviation of the annual (1 in 1-year) 24 hr maximum rainfall were determined for the project area according to the isohyetal maps in the atlas. These are shown in Table 25.22.

Table 25.22 Annual Rainfall Extremes		
Duration	**Mean**	**Standard Deviation**
5 min	3.0	1.0
10 min	4.0	1.0
15 min	6.0	1.0
30 min	8.0	2.0
1 hr	10.0	4.0
2 hr	18.0	4.0
6 hr	20.0	6.0
12 hr	22.0	12.0
24 hr	25.0	8.0

Potential snowmelt during the PMP event also has to be taken into consideration when estimating the resulting PMF. A monthly snowpack depth was estimated from site and regional data. The potential snowmelt for each month that could occur in conjunction with the PMP was then estimated by applying a formula that considered monthly mean maximum daily temperature and monthly average wind speeds recorded on site. The highest potential snowmelt was found to be 83 mm in a 24 hour period during the month of May. Onsite snow surveys indicate that this depth of snow water equivalent is generally present in May.

The total potential runoff depth from combined rainfall and snowmelt is estimated to be 347 mm. This value was applied to a HydroCAD® model to estimate the resultant PMF. The total PMF inflow is dependent on basin parameters while the instantaneous peak flow is related to the time of concentration (t_c). Runoff data from the report Streamflow in the Skeena Region (Obedkoff 2001), a report specific to the site's hydrologic zone, were used to calibrate the rainfall-runoff model.

The design of the TSF includes sufficient capacity and freeboard to store the PMF during operations.

The storm storage volume of the TSF that is required and provided during operations is approximately 12.3 M m^3, corresponding to an equivalent runoff depth of 310 mm, and a peak inflow rate of 1,400 m^3/s.

Return period precipitation values in the form of intensity-duration-frequency (IDF) curves were largely generated from the return period storm event data in the RFAC. The data is shown in Table 25.23. The IDF curves are presented on Figure 25.30.

Table 25.23 Suggested Design Flood and Earthquake Levels										
Duration (years)	**2**	**5**	**10**	**15**	**20**	**25**	**50**	**100**	**200**	**1000**	**PMP**
Return Period Rainfall Amounts											
5 min	3	4	4	5	5	5	6	6	7	8	
10 min	4	5	5	6	6	6	7	7	8	9	
15 min	6	7	7	8	8	8	9	9	10	11	
30 min	8	9	11	11	12	12	13	14	15	18	
1 hr	9	13	15	17	17	18	20	23	25	30	**78**
2 hr	17	21	23	25	25	26	28	31	33	38	
6hr	19	24	28	30	31	32	36	39	42	50	**128**
12 hr	30	46	56	62	67	70	80	89	99	122	
24 hr	**36**	**46**	**53**	**57**	**60**	**62**	**69**	**75**	**82**	**97**	**253**
Rainfall Intensity (mm/hr)											
5 min	34	45	52	56	58	61	67	74	80	95	
10 min	23	28	32	34	35	36	40	43	46	54	
15 min	23	27	29	31	31	32	34	37	39	44	
30 min	15	19	21	23	23	24	26	29	31	36	
1 hr	9	13	15	17	17	18	20	23	25	30	**78**
2 hr	9	10	12	12	13	13	14	15	16	19	
6hr	3.2	4.1	4.6	5.0	5.2	5.4	5.9	6.5	7.0	8.3	**21**
12 hr	2.5	3.8	4.7	5.2	5.5	5.8	6.6	7.5	8.3	10.2	
24 hr	**1.5**	**1.9**	**2.2**	**2.4**	**2.5**	**2.6**	**2.9**	**3.1**	**3.4**	**4.0**	**11**

Notes:

1. Mean annual 24 hour extreme rainfall and standard deviation were estimated using the Rainfall Frequency Atlas of Canada.
2. Durations of 12 hours or more were increased by a factor of 1.5 to account for orographic effects.
3. Return period rainfall amounts computed assuming an Extreme Value Type I (Gumbel) distribution, as shown in Table 23.24.

Table 25.24 Gumbel Frequency Factors										
Return Period	**2**	**5**	**10**	**15**	**20**	**25**	**50**	**100**	**200**	**1000**
K_T	-0.164	0.719	1.305	1.635	1.866	2.044	2.592	3.137	3.679	4.936

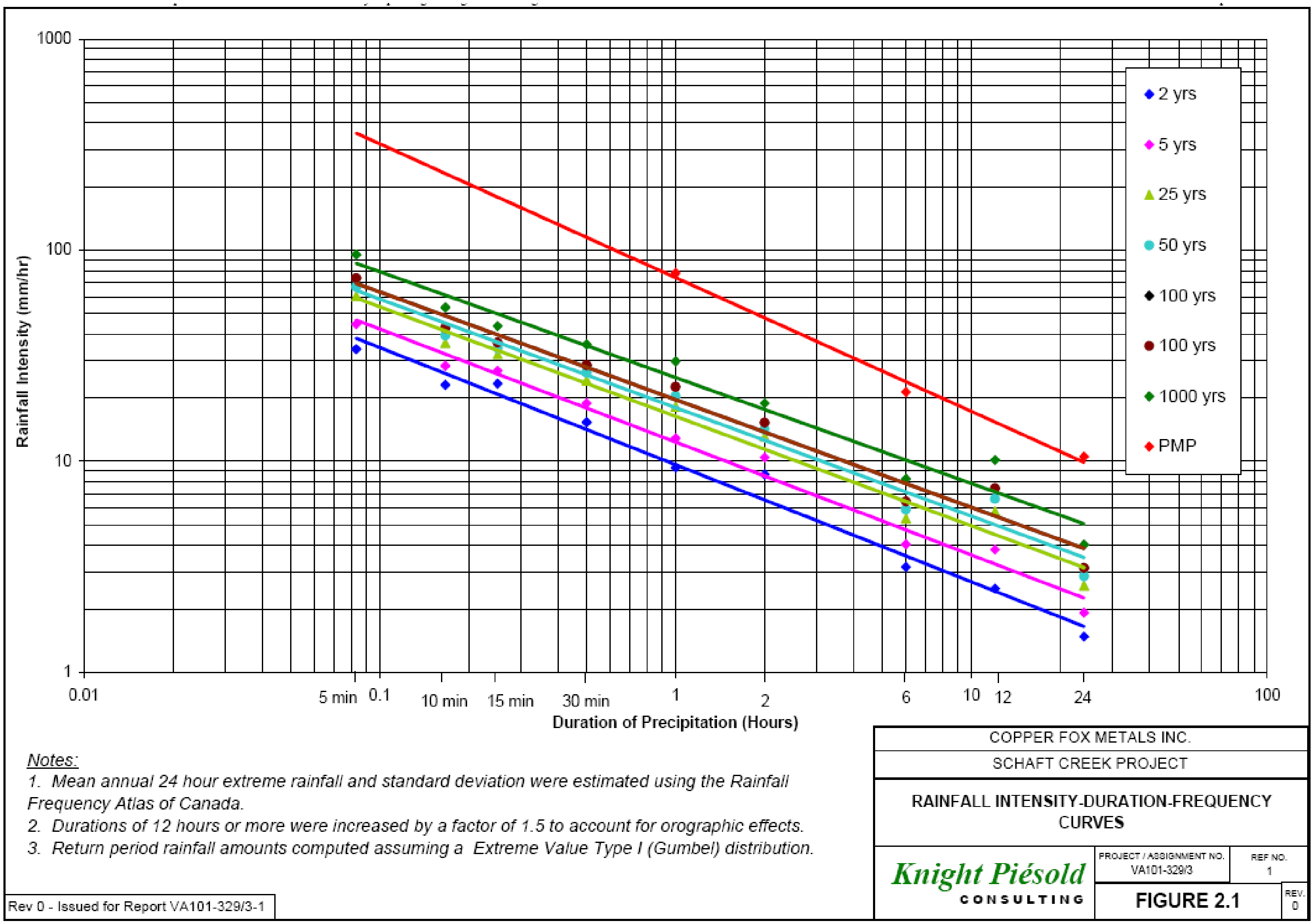


Figure 25.30 Rainfall Intensity-Duration-Frequency Curves

25.3.3 Tailings Characteristics

25.3.3.1 *General*

Tailings from the Schaft Creek Project operation will be produced from conventional milling of ore. Tailings from the mill will be discharged to the TSF at an average slurry solids content of 55% at a throughput of 100,000 tpd or approximately 36 M tpa.

Laboratory testing of representative tailings samples, supplied by Copper Fox in February 2008, was undertaken by KPL at its testing laboratory in Colorado, USA. Bench-scale tests were carried out to determine the physical properties of the tailings, including classification tests and settling tests. These latter tests were undertaken to investigate the amount and rate of supernatant release as well as the tailings settled density under drained and undrained conditions.

Test results are presented in Appendix A of the KPL report. The test results are also summarized in the following sections.

25.3.3.2 *Classification Tests*

Screen and hydrometer particle-size analyses were carried out on the samples in accordance with ASTM D422 procedures. These tests provide a measure of the type and condition of the material, specifically the particle density, composition (size) and plasticity characteristics. The index properties can provide a relationship to material structural properties, including compressibility, permeability and strength.

The size-gradation curves indicate that all of the samples are well-graded consisting predominantly of sandy and silt sized material. The tailings samples consisted of between 49% to 57% silt, with 26% to 36% sand and 15% to 17% clay-sized material. The Schaft Liard sample contains more sand (35%) than the Paramount sample (26%) and the West Breccia sample lies somewhere in between.

The specific gravity of the tailings samples was determined in accordance with the ASTM D854 procedure. The measured values of specific gravity (sg) range from 2.71 to 2.83 with an average of approximately 2.75.

25.3.3.3 *Settling Tests*

Settling tests were undertaken to assess the rate at which supernatant water is released and to provide an estimate of the dry density to which the tailings might settle under undrained and drained (including underdrainage) conditions. These tests provide an estimate of expected tailings densities in a storage facility after settling and before any significant consolidation or air-drying occurs.

Average dry densities achieved in the laboratory were 1.23 t/m^3 in the undrained tests and 1.34 t/m^3 in the drained test.

Measured supernatant water release was 48% and 57% in the undrained and drained tests, respectively. Water release was relatively rapid in both tests with the majority of water being released within the first 24 hours.

25.3.3.4 *Air Drying Tests*

Air-drying tests were carried out to determine the effect of air-drying after initial slurry settling and removal of supernatant water, thereby simulating expected conditions following sub-aerial deposition.

The average dry density achieved in the laboratory following air drying was approximately 1.64 t/m^3.

25.3.3.5 *Conclusions*

The average dry density of the tailings deposit can be predicted from the laboratory testing results. Actual field densities achieved are dependent on the area available for tailings drying and the overall depth of deposited tailings. A suitable deposition plan and efficient operation of the facility can greatly improve tailings density. It is conservatively estimated that the average dry density in the field will be at least 1.4 t/m^3. Supernatant water release is expected to vary between 45% and 55%, for an initial tailings slurry solids content of 55%.

25.3.4 TSF Options Study

25.3.4.1 *Introduction*

Three potential TSF sites referred to as Options A, B and C, as shown on Figure 25.31, were identified during the scoping stage of the project.

In late 2007 an options study comprising a preliminary feasibility level geotechnical assessment of the three sites was undertaken by DST Consulting Engineers Inc. (DST). Details of this work were presented in a report titled, *Pre-Feasibility Geotechnical Assessment of Tailings Dam Options, Schaft Creek Project, British Columbia, DST, February 27, 2008*. The conclusion was that from a geotechnical point of view, Option A was the preferred option with Option B a close second.

In early 2008, BGC Engineering Inc. (BGC) was commissioned by Copper Fox to undertake an overview study to determine the most favourable of the three sites with respect to geo-hazards. The results of the BGC study (reported in a memorandum dated 5 February, 2008) indicated that Option A was the preferred site with respect to geo-hazards with Option B and C being rated equally and less favorably. At the same time Moose Mountain Technical Services (MMTS) was requested to compare the three sites. The results of their work, presented in a memo dated March 2008, concluded that Option A was the preferred site for the TSF.

As part of the preliminary feasibility design of the tailings management components of the project, Knight Piésold Ltd. undertook a conceptual-level assessment of the three previously-identified options and in the process developed a number of other alternatives.

Details and results of this work are presented in Appendix B of the KPL report. The information is summarized in the following sections.

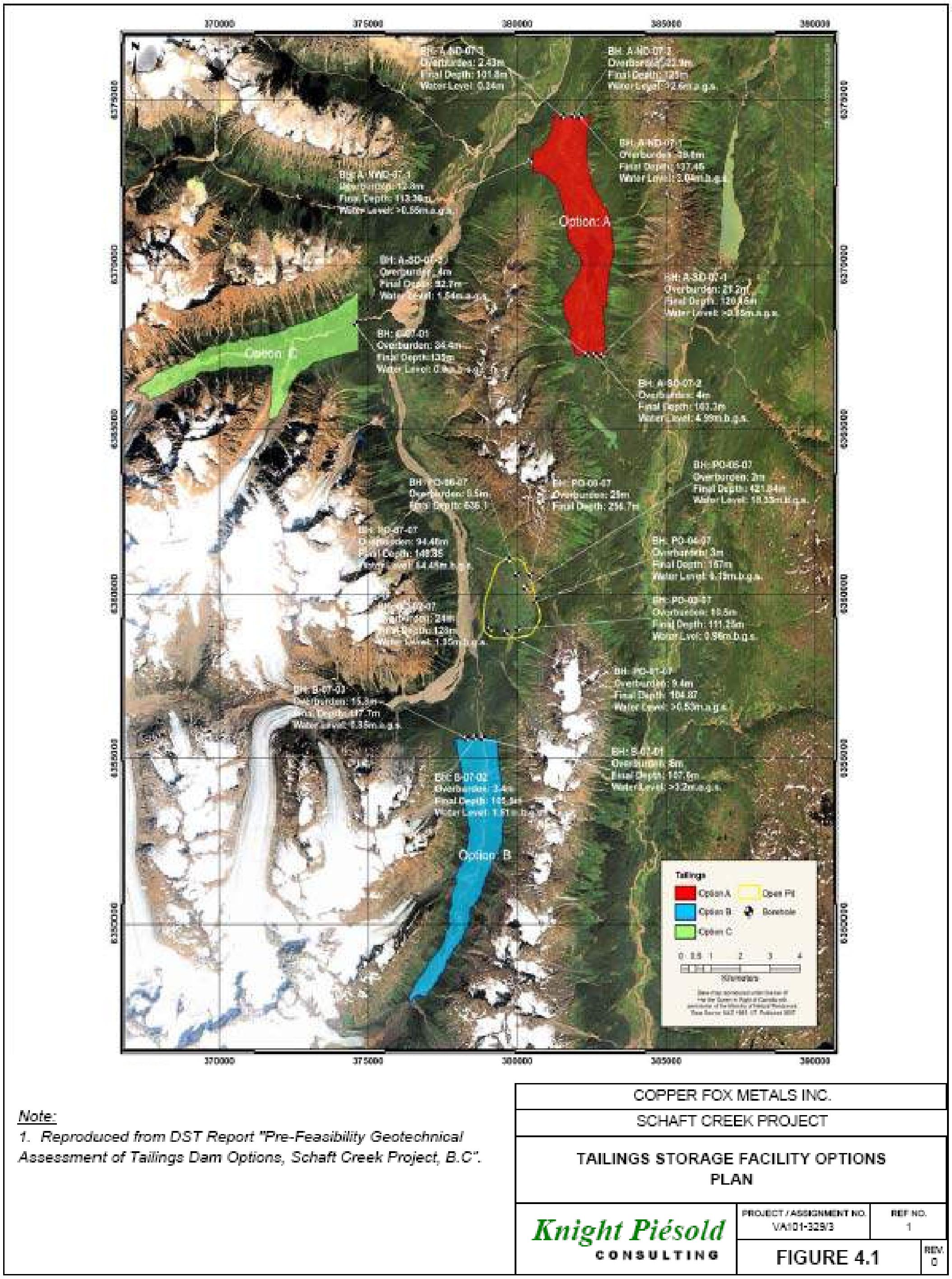


Figure 25.31 Tailings Storage Facility Options Plan

25.3.4.2 *Methodology*

The options study was based on a conceptual-level comparison of the various sites, with respect to the following criteria:

- Capital cost of starter facility – the costs of major capital items associated with the starter facility for each option were estimated using global unit rates for embankment construction, tailings delivery and water reclaim pipelines systems, surface water drainage diversions, access and haul roads;
- Water management considerations – the options were compared on the basis of catchment area and topography;
- Ongoing construction costs – the cost of ongoing embankment raising, road construction and pipeline installation were assessed and compared;
- Operational considerations – these included elevation of the facility relative to the mill, distance to the facility from the mill, ease of operation;
- Potential for expansion – the potential for expansion of the facility capacity beyond the 1 G tonne design case was qualitatively assessed.

25.3.4.3 *Comparison of Options*

The options study concluded that Option C was the least preferred based on nearly all of the points of comparison.

Option B, although located close to the pit and therefore a source of inexpensive embankment fill material, had the largest external catchment area of all sites considered. It was concluded that water management at the site would pose very significant technical and operating challenges which would be very difficult and expensive to overcome. The volume of water to be managed at the Option B site is estimated to be between two and four times greater than for the Option A site due to a combination of large catchment area, higher elevation, steeper valley side slopes and larger glacier fraction. This excess water would require treatment prior to release from the facility, thus adding considerably to the operating costs. Surface water-diversion systems constructed around the facility would also be several times larger for Option B, based on the larger inflows to be diverted. Maintenance of surface water diversion structures in the Option B valley would pose severe operating challenges and hazards, particularly with respect to blockage by snow avalanches and terrain hazards. These diversions would likely operate with a relatively low overall efficiency, further contributing to large inflows to the facility. It would be necessary to maintain adequate storage within the TSF at all times to accommodate the Probable Maximum Flood (PMF). This requirement would necessitate the raising of the facility to a higher elevation than would be required for tailings storage only. For these reasons Option B is not preferred.

Development of a TSF in the Option A valley was considered to provide the best option for tailings storage at the site, although the potential for Karstic formations within the valley needs to be examined further during the next design phase. Within the Option A valley, three different arrangements were examined. Of these, Option A1 offers significant cost savings over Option A, although it was noted that the issue of fish habitat at the southern end of the facility might pose permitting challenges.

Option Aa is broadly similar to Option A in all respects other than the starter facility arrangement, which allows for a reduction in the initial capital cost for development.

On the basis of this options study and using the data currently available, Option A was selected as the preferred option, followed by Option Aa, then Option A1, and lastly Option B. The preliminary feasibility design presented in the following sections is therefore based on the construction of Option A.

25.3.5 Geological and Geotechnical Conditions

25.3.5.1 *Geotechnical Site Investigation*

A geotechnical site investigation of the three potential TSF sites was undertaken by DST in the period July 2007 to October 2007. The investigation, which focused on the Option A site, included the completion of eleven drill holes and twenty-six test pits. In-situ permeability tests and Standard Penetration Tests (SPT) were carried out in the drill holes and all holes were logged by DST personnel with assistance from Copper Fox. A laboratory testing program was also undertaken on selected samples. Complete details of the investigation are presented in the DST *Pre-feasibility Geotechnical Assessment of Tailings Dam Options* report.

The investigation indicated that each of the three TSF valley sites have a variable depth of overburden, comprising alluvium, till and colluvium (up to 40 m deep) overlying bedrock. Overburden is generally comprised of dense sand and gravel. Bedrock is comprised of various formations of limestone and andesite at the Option A site and monzonite at the Option B and C sites.

Diamond drilling was employed using direct circulation water flush and drilling mud where necessary. Holes were drilled using either an NQ or HQ double-tube barrel. Drill-hole locations are shown on Figure 25.31 and geotechnical logs are included in Appendix C of the KPL report.

25.3.5.2 *Subsurface Conditions*

General

Based on the results of the site investigation program, the sub-surface conditions encountered at the three sites are described in the following sections.

Option A Site

Overburden Drilling

Seven drill holes were completed at the Option A site, three at the north dam site (A-ND-07-01 to A-ND-07-03), three at the south dam site (A-SD-07-01 to A-SD-07-03) and one at the northwest dam site (A-NWD-07-01).

At the south dam, drilling encountered overburden to a depth of 4 m on the western side of the valley (A-SD-07-03), 4 m in the centre of the valley (A-SD-07-02) and 20 m on the eastern side (A-SD-07-01). The single hole drilled at the northwest dam location encountered 13 m of overburden.

In all cases overburden consisted predominately of dense well-graded sands and gravels. Grain-size analyses on samples of overburden collected from the north dam site indicated the overburden material typically ranged from well-graded gravel (GW) to well-graded sand with clay and gravel. The maximum particle size was typically in the range 15 mm to 30 mm, the gravel was generally fine grained and the sand fine to coarse grained. Grain size analysis distributions are presented on Figure 25.32.

SPT blow count (N) values measured within the sand and gravel overburden at the north dam site ranged from refusal (taken as a blow count of more than 50) to a minimum value of 13. The average measured SPT N value was 28 in A-ND-07-02 and 44 in A-ND-07-2 indicating the granular material is classified as medium dense to very dense but typically is dense.

Bedrock Drilling

Bedrock at the Option A site comprises various formations of limestone and andesite.

At the north dam site, bedrock on the east side of the valley comprises weakly- to moderately-fractured perodite with an andesite dyke found at the east edge of the valley. In the centre of the valley, strongly-fractured gabbro and leuco gabbro changing to quartz monzonite was encountered, while on the western side of the valley, bedrock was comprised of weak- to moderately-fractured quartz monzonite. All rock is described as being completely to slightly weathered.

At the south dam site, bedrock at the east side of the valley is described as weakly- to moderately-fractured limestone, sandstone and pelite in the upper 40 m with limestone below this depth. In the central part of the valley, bedrock is comprised of weakly- to moderately-fractured volcanic ash with calcareous siltstone and limestone, while in the western side of the valley it is described as being weakly to moderately fractured siltstone, ortho quartzite. Bedrock encountered at the eastern and western sides of the valley was completely to moderately weathered whereas that in the centre of the valley was described as being highly weathered to fresh.

At the northwest dam site, bedrock was comprised of weakly- to highly-fractured augite phyric andesite with some quartz monzonite at lower depths. The rock was described as being completely to moderately weathered.

Hydraulic Conductivity

Hydraulic conductivity was assessed within the overburden and the underlying bedrock by means of variable head permeability test within the drill holes. Measured permeability's are summarized in Table 25.25.

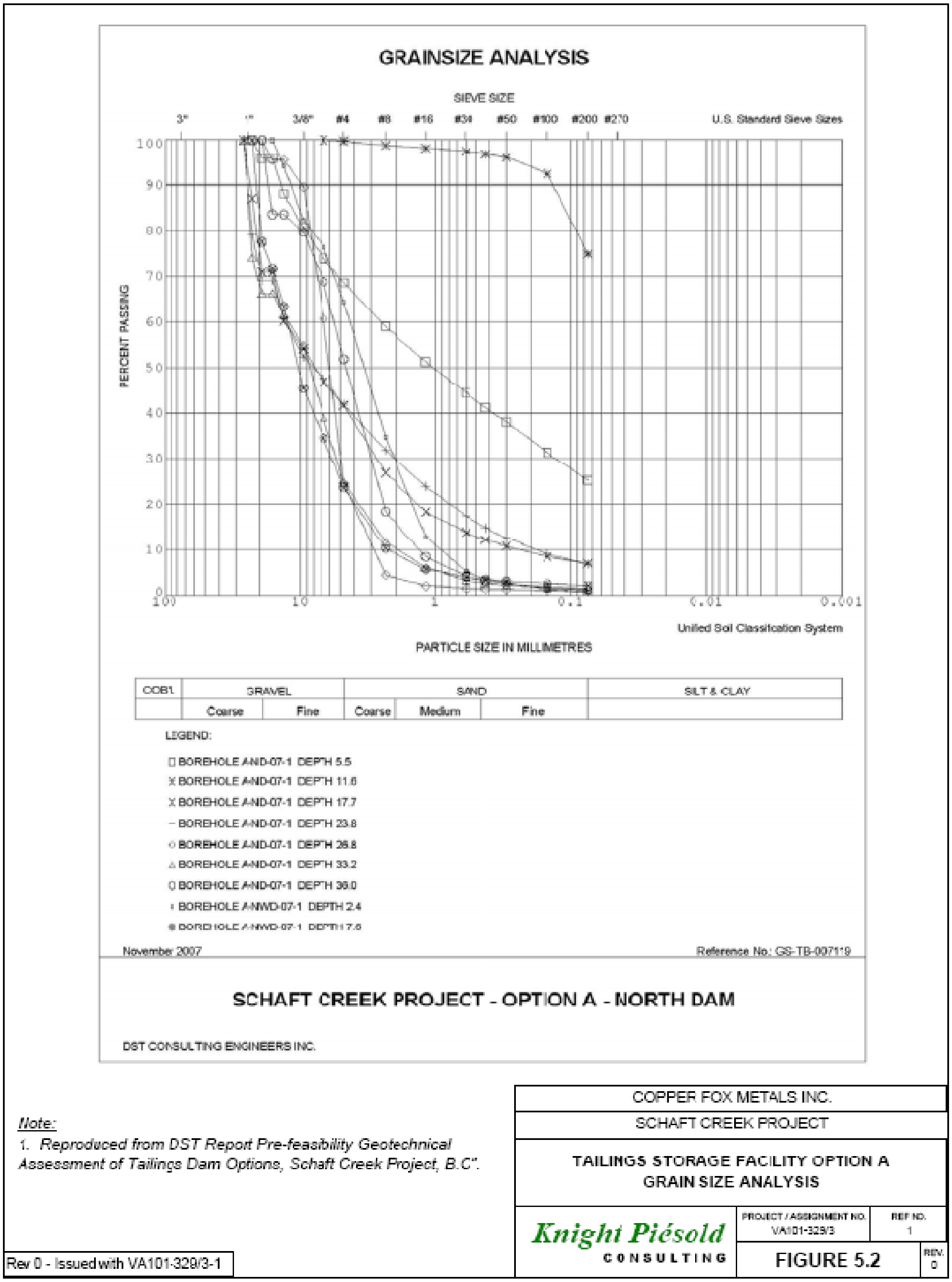


Figure 25.32 TSF Option A Grain Size Analysis

Table 25.25 Summary of In-Situ Measure Permeabilities				
TSF Option	Embankment Dam	Typical Permeability (m/s)		
		Drill Hole	Bedrock	Overburden
Option A	North	A-ND-07-01	2 x 10^{-5}	0 m to 20 m $3x10^{-6}$ 20 m to 40 m $3x10^{-4}$
		A-ND-07-02	$3x10^{-7}$ to $2x10^{-6}$	$1x10^{-6}$
		A-ND-07-03	$1x10^{-7}$ increasing to $1x10^{-5}$ at depth	N/A
	South	A-SD-07-01	$1x10^{-5}$	$1x10^{-5}$ to $1x10^{-3}$
		A-SD-07-02	$5x10^{-6}$ to $5x10^{-5}$	N/A
		A-SD-07-03	10 m to 40 m: $3x10^{-6}$ Below 40 m $3x10^{-5}$	$1x10^{-3}$
	Northwest	A-NWD-07-01	$5x10^{-7}$	$1x10^{-4}$ to $1x10^{-3}$
Option B	North	B-07-01	0 m to 20 m $5x10^{-6}$ Below 20 m $1x10^{-6}$	N/A
		B-07-02	$1x10^{-6}$	N/A
		B-07-03	$3x10^{-7}$ to $2x10^{-6}$	N/A
Option C	East	C-07-01	$1x10^{-8}$	$1x10^{-7}$ to $1x10^{-4}$

Option B Site

Overburden Drilling

Three drill holes were completed at the Option B dam site. Overburden depths as encountered in drill holes B-07-01 (east), B-07-02 (centre), and B-07-03 (west), were 6 m, 3 m and 15 m respectively. Overburden consisted predominately of dense, well-graded sands and gravels. Grain-size analyses on samples of overburden collected from the dam site indicated the material typically ranged from well-graded gravel (GW) to well-graded sand with clay and gravel. The maximum particle size was typically in the range 15 mm to 30 mm. The gravel was generally fine grained and the sand fine to coarse grained.

SPT N values measured within the sand and gravel overburden ranged from refusal (taken as a blow count of more than 50) to a minimum value of 26. The average measured SPT N value was 40 in B-07-01 and B-07-3 indicating that the granular material would be classified as dense.

Bedrock Drilling

Bedrock encountered at the Option B site was comprised of predominately low- to moderately-fractured gabbro, with a felsic intrusive dyke on the eastern side of the valley in B-07-01 and a thin layer of diorite on the western side of the valley in B-07-03. Rock encountered in the eastern and central drill holes was typically completely to moderately weathered, whereas on the western side of the valley the rock encountered was highly to freshly weathered.

Hydraulic Conductivity

Hydraulic conductivity was measured within the drill holes, by means of variable head permeability tests. Measured permeabilities within the bedrock were typically around $1x10^{-6}$ m/s.

Option C Site

Overburden Drilling

A single drill hole, C-07-01, was completed in the centre of the valley at the Option C dam site. The drill hole encountered 34.4 m of overburden comprised of dense to very dense sand and gravel.

SPT N values measured within the sand and gravel overburden ranged from 14 to 30 with an average of 22, indicating that the granular material would be classified as medium dense.

Bedrock Drilling

Bedrock encountered at the single drill hole completed at the Option C site comprised of lowly- to moderately-fractured quartz monzonite with a weathering index ranging from completely to moderately weathered.

Hydraulic Conductivity

Hydraulic conductivity measured within the uppermost 10 m of overburden was typically around 3 x 10^{-5} m/s, falling to 4 x 10^{-7} m/s, below 10 m depth. Bedrock permeability below 40 m depth was typically measured at between 1 x 10^{-7} m/s and 1 x 10^{-8} m/s.

25.3.5.3 *Test Pitting*

Twenty-six test pits were excavated in the vicinity of the proposed open pit to identify potential sources of low permeability fill, filter and bedding material. Pit depths varied from 1.0 m to 5.1 m with an average excavated depth of 3.0 m. Materials encountered comprised sand and gravel. Test pit geotechnical logs are presented in Appendix C of the KPL report.

25.3.5.4 *Embankment Foundation*

The geotechnical investigative work undertaken to date indicates that the embankments will be founded primarily on sand and gravel deposits of variable depth with thin layers of silty clay and clay till and bedrock.

In-situ testing indicates that the sand and gravel deposits are generally classified as dense and of moderate to high permeability.

25.3.5.5 *Construction Materials*

Test pitting and drilling undertaken at the TSF site encountered very limited amounts of low-permeability material.

On the basis of the logs provided it appears that borrow areas developed within the facility basin may provide sources of bedding and filter material in addition to general granular fill. Future site investigations will aim to identify, characterise and delineate local sources of low permeability fill material for construction of the embankment core zone.

It is envisaged that the North and West starter embankments will be constructed using fill sourced from local borrow and ongoing staged raises will be constructed using cycloned sand. The South embankment will be constructed using primarily NPAG mine waste. The low permeability core zone for each embankment will likely be constructed using till from local borrow pits. In the event that low-permeability till is not readily available within a reasonable distance from each respective embankment, a low-permeability soil will be manufactured onsite using cycloned sand and bentonite powder.

25.3.6 Waste Rock Dumps

25.3.6.1 *General*

Up to 1.6 G t of waste rock will be generated during the mining of the Schaft Creek open pit over the 23-year mine life, assuming an open pit strip ratio of 2:1. For the purposes of this study it was estimated that up to ten percent (200 M t) of the waste may be Potentially Acid Generating (PAG), while the balance will be Non-Potentially Acid Generating (NPAG).

PAG waste rock will be appropriately managed at the open pit. NPAG waste rock will be stored in the three conventional surface waste dumps located to the west and south of the open pit, and east of Schaft and Hickman Creeks. These waste dump locations were identified during the scoping stage of the project and presented in the Preliminary Economic Assessment (PEA) for the project (Samuel, 2007).

The proposed waste rock storage area is generally characterized as a flat to undulating valley floor, located immediately adjacent to the northward flowing Schaft Creek. The dump area east of the Schaft Creek floodplain, surrounding the open pit, is at a higher elevation with ground sloping downwards towards the creek. Immediately south of the dump area is Hickman Creek.

25.3.6.2 *Geotechnical Conditions*

DST completed a preliminary geotechnical assessment of the proposed waste rock dumps in the February 2008 report *Pre-Feasibility Geotechnical Assessment of Waste Rock Disposal*. Selected information from that report is summarized below.

Typical valley bottoms in the project area are flat-bottomed with extensive pro-glacial outwash sedimentation and a high water table. Although no site investigations have been undertaken at the actual location of the dumps, site investigations at the nearby dam site for the Option B TSF indicated the presence of three inter-bedded overburden deposits: coarse-granular soils, very stiff cohesive tills, and compact to dense sands. The thickness of the overburden observed at the dam site varies from approximately 3 to 16 m.

In addition to the data from the Option B TSF site investigation, considerable data on overburden thickness are available from exploration drill holes completed in and around the proposed pit area. Typically these holes indicate overburden depths of up to 60 m.

25.3.6.3 *Waste Rock Dump Construction*

Waste dumps will be constructed using conventional dumping techniques, with lift heights in the range of 15 m to 30 m. During operations the dumps will be constructed with an overall slope of 22° and single-lift slopes at the angle of repose of the waste material. For closure the overall slope will be 22°, with single-lift slopes of no greater than 27°. Certain localized areas may require 22° single-lift slopes at closure. More detail is presented in the DST waste dump geotechnical assessment report. All waste dumps will need to be constructed with suitable surface water diversions, foundation preparation and drainage, and seepage control systems.

25.3.7 Tailings Storage Facility Design

25.3.7.1 *General*

The principal objectives and requirements for the design for the Tailings Storage Facility (TSF) are as follows:

- Permanent, secure and total confinement of all solid waste materials within an engineered disposal facility;
- Control, collection and removal of free-draining liquids from the tailings surface during operations for recycling as process water to the maximum practical extent;
- Diversion of surface-water runoff from the valley sides above the envisaged elevation of the facility;
- The inclusion of monitoring features for all aspects of the facility to ensure performance goals are achieved and design criteria and assumptions are met;
- Staged development of the facility over the life of the project.

The overall general arrangement of the TSF is shown on Figure 25.33. General arrangements illustrating the starter facility and ultimate facility are shown on Figure 25.34 and Figure 25.35, respectively.

25.3.7.2 *Design Basis*

The mine will operate at a 100,000 tpd mill throughput and the open pit will yield 812 M t of ore and 1.6 G t of waste rock over a mine life of 23 years. Local borrow areas developed within the facility basin area will provide fill material for starter facility embankment construction. Thereafter, ongoing construction of the facility embankments will be undertaken using waste rock materials from the open pit mining operations and cycloned sand generated from the mill tailings.

The TSF has been designed to permanently store 812 M t of tailings (570 M m^3 at an overall average dry density of 1.4 t/m^3).

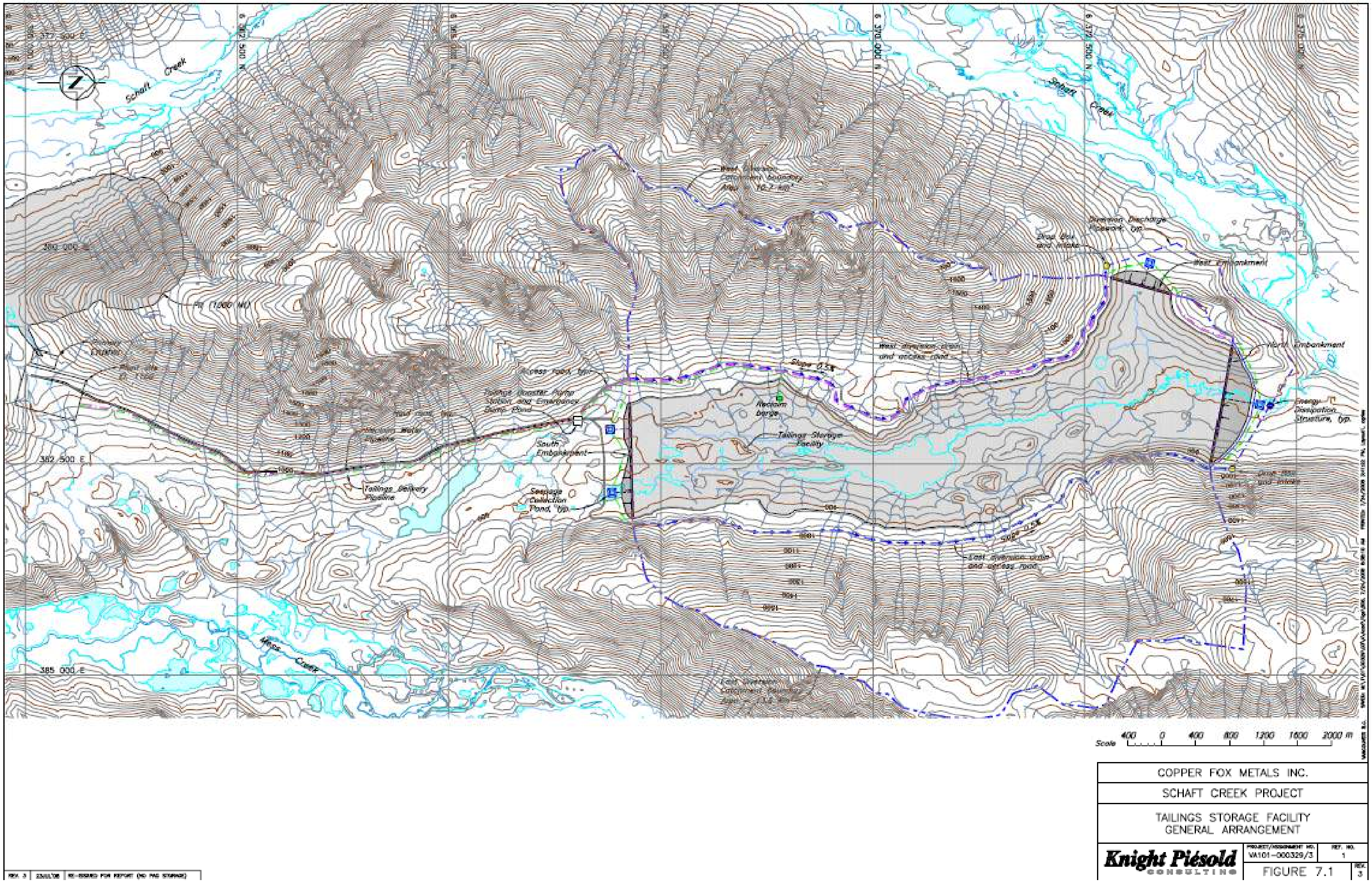


Figure 25.33 General Arrangement

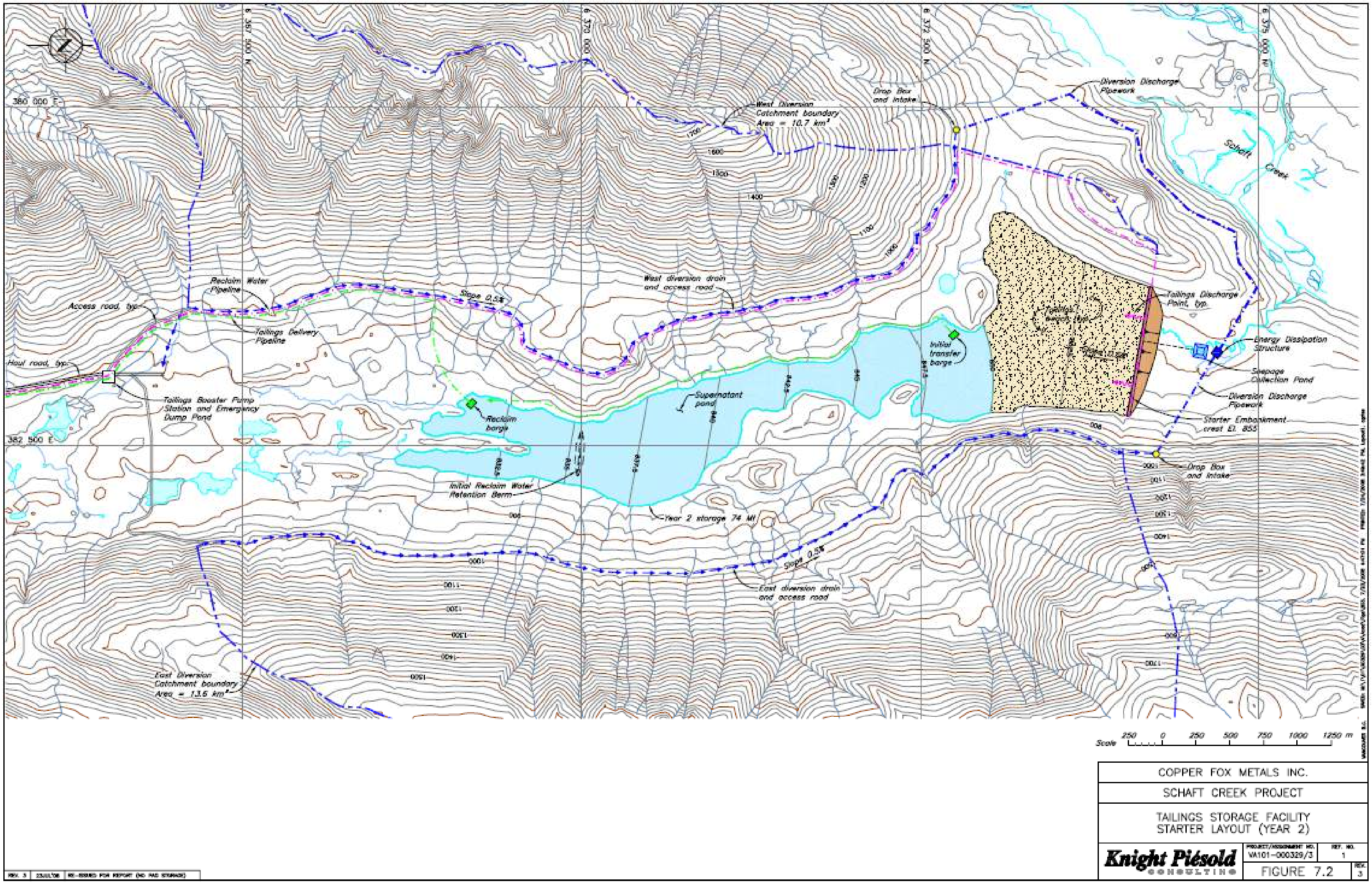


Figure 25.34 Starter Layout (Year 2)

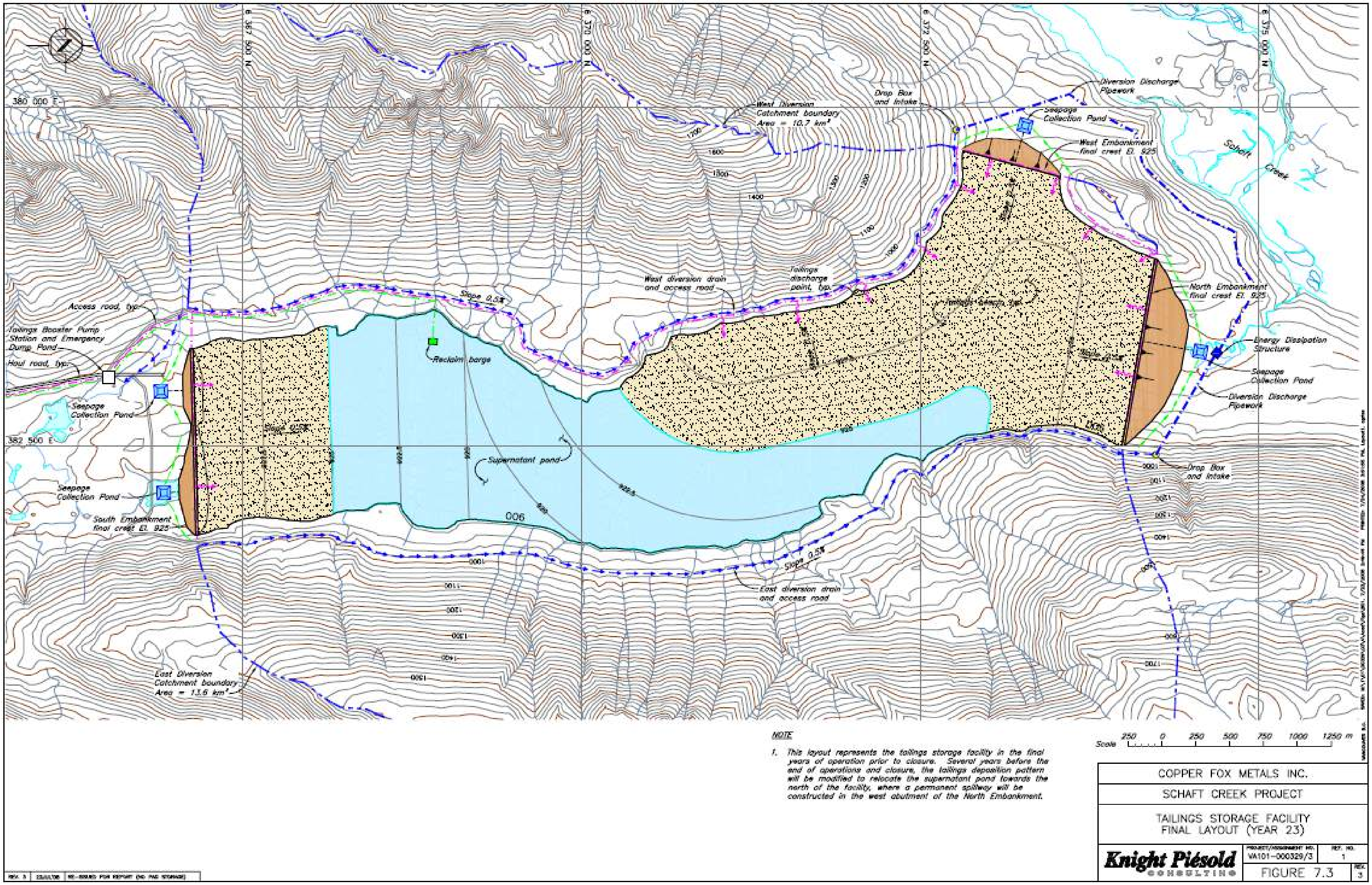


Figure 25.35 Final Layout (Year 23)

The Depth-Area-Capacity (DAC) relationship for the TSF is shown on Figure 25.36. For development of the TSF filling schedule, a tailings average dry density of 1.3 t/m³ was assumed during the first 2 years of production, rising to 1.4 t/m³ in subsequent years.

25.3.7.3 *Layout and Operating Strategy*

General

The major components of the TSF are illustrated on Figure 23.34. The major components are:

- Three main earth-rockfill, zoned embankments, referred to as the North, South and West embankments;
- Seepage collection systems;
- Access roads along the eastern and western sides of the valley;
- Surface water management structures;
- Tailings distribution system, and;
- Reclaim water system.

Embankments

The embankments will be constructed in stages throughout the life of the project using a combination of NPAG waste rock materials from the open pit, local borrow and cycloned tailings. The starter facility, designed to accommodate tailings production for a period of two years, will include a single embankment constructed at the northern end of the facility with fill sourced from local borrow pits. This embankment will be raised progressively throughout the life of the facility using cycloned tailings. Later in the life of the facility, the West and South embankments will be constructed and raised progressively. The West embankment will be constructed in a similar manner to the North embankment, using local borrow and cycloned sand, while the South embankment, being relatively close to the pit, will be constructed largely from mine waste.

Seepage Management

All seepage losses through the embankments will be collected in seepage interception systems constructed downstream of the embankments. Intercepted seepage will be fed to collection sumps from where it will be pumped back to the facility.

Special design provisions to minimize seepage losses include the installation of geomembrane liners on starter embankments, the installation of seepage cut-offs beneath the embankments, inclusion of low-permeability core zones in ongoing embankment raises and the development of extensive low-permeability tailings beaches that isolate the supernatant pond from the embankments.

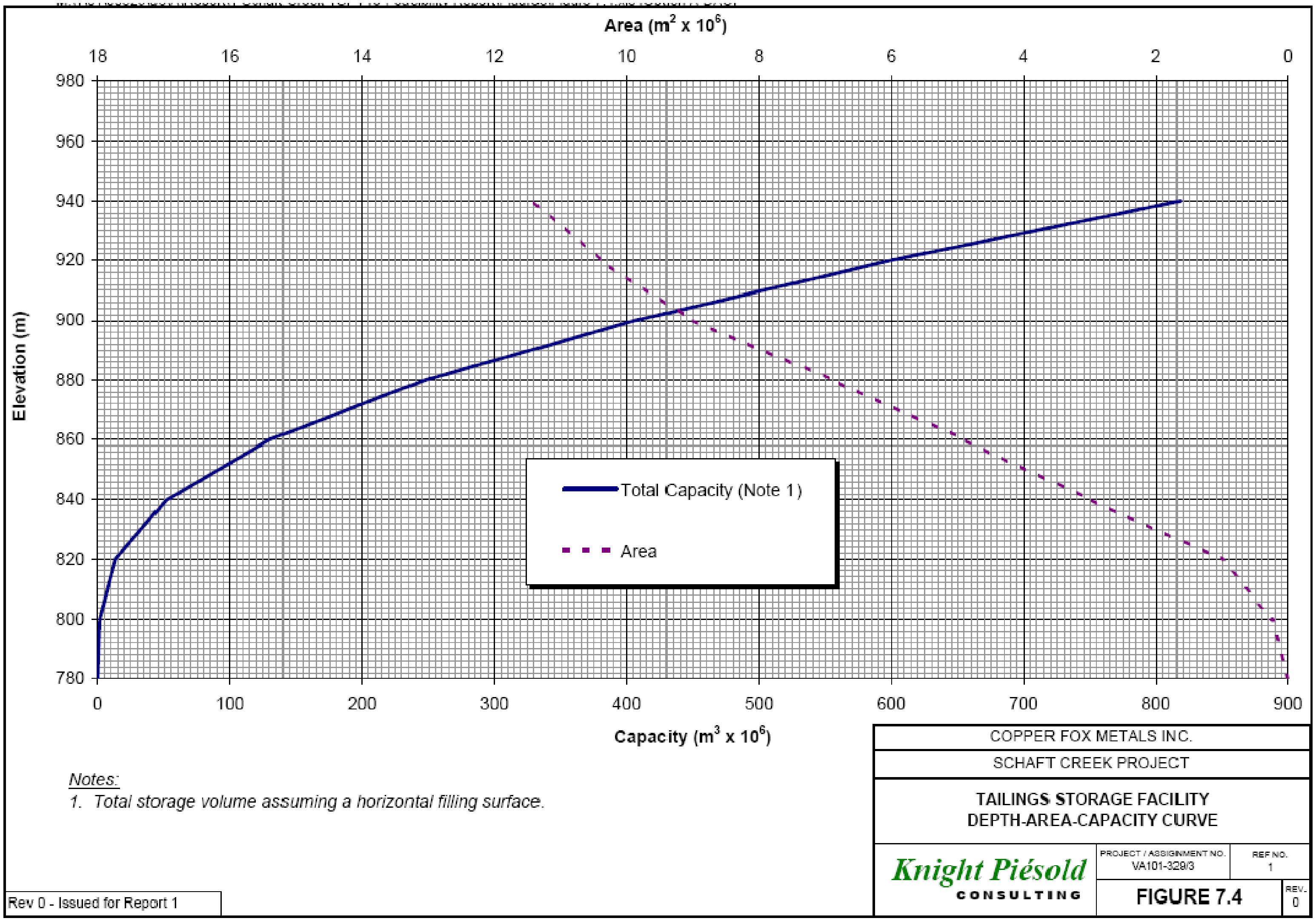


Figure 25.36 Depth-Area-Capacity Curve

Access Roads

The primary access to the TSF will be along the main access road on the western side of the valley above the ultimate-facility level. This road will extend from the mill along the western side of the facility to the West embankment and onto the North embankment. Short spur roads off the main road will provide access to the South embankment and reclaim barge. A second access road will run along the eastern side of the valley above the ultimate facility level. This road will also extend to the North embankment.

Both roads will provide access to the surface-water diversion structures to be constructed on either side of the facility.

Surface Water Management

Permanent surface-drainage channels will be constructed around the periphery of the facility to intercept and safely discharge surface-water runoff from the upper catchment.

These drainage channels will be installed on both sides of the valley and will discharge surface runoff to Skeeter Creek downstream of the North embankment.

Tailings Distribution

The discharge of tailings into the TSF will initially be from a series of large-diameter valved off-takes located along the North embankment. Later in the operational life of the facility, tailings distribution will also take place from the South and West embankments and from the western side of the facility.

The coarse fraction of the tailings is expected to settle rapidly and will accumulate closer to the discharge points, forming a gently-sloping beach while the finer tailings particles will travel further and settle remote from the discharge point. The proposed deposition pattern will result in a dish-shaped deposit with the supernatant pond located centrally within the facility, remote from all confining embankments. Tailings deposition will be managed to maintain the supernatant pond away from the embankments, in order to reduce seepage and to ensure that reclaimed water is clear and suitable for reuse in the milling process.

Reclaim Water

Construction of the Starter facility will commence some time prior to mill start-up. Water impounded within the starter facility will therefore be available for mill commissioning and early operations. Mill process water for ongoing operations will be reclaimed from the TSF supernatant pond.

Project annual water balances completed for average precipitation conditions indicate that the facility water balance will be slightly positive during the operational life of the facility.

Releases of water from the TSF may be required later in the operational life of the facility but under average conditions a make-up water system should not be required for the milling process other than for the supply of clean water requirements for potable and process water.

25.3.7.4 *Embankment Construction*

Main North Embankment

The main North embankment will comprise an approximately 60-m high starter embankment constructed to elevation 855 masl. Geotechnical investigations undertaken to date at the site encountered no sources of potential low-permeability fill material that might be suitable for construction. The starter embankment will therefore be constructed using granular material sourced from local borrow areas within the facility basin area. An HDPE geomembrane will be installed on the upstream face of the embankment to minimize seepage through the structure. The geomembrane will be tied into a bentonite-cement slurry cut-off wall installed beneath the embankment upstream toe. The slurry cut-off wall will extend down through the permeable sand and gravel foundation and will be keyed into competent bedrock to form a positive hydraulic cut-off. The depth to bedrock is variable along the length of the embankment, and will be more accurately defined in future geotechnical site investigations. If the depth to bedrock in certain areas exceeds the maximum reasonable constructible depth of a slurry wall, a combination of excavation, slurry cut-off wall, and low permeability backfill may be used.

Above the elevation of the starter-wall crest, the embankment will be raised using centreline construction and cycloned tailings, to an ultimate elevation of 925 masl, resulting in a final maximum embankment height of approximately 125 m. Tailings will be passed through a number of cyclones installed on the embankment. The finer tailings fraction will emerge with the cyclone overflow which will be discharged into the facility. The coarser tailings fraction will emerge with the cyclone underflow and will be spread and compacted using dozers and rollers to form the embankment raise. A low-permeability core zone, initially keyed into the impermeable crest zone of the starter facility, will be extended vertically upwards through each successive raise. The core zone will be constructed using low permeability glacial till from local borrow. If a suitable source of borrow material is not available, a low-permeability soil will be manufactured onsite using cycloned sand and bentonite powder. A typical section through the main north embankment is shown on Figure 25.37.

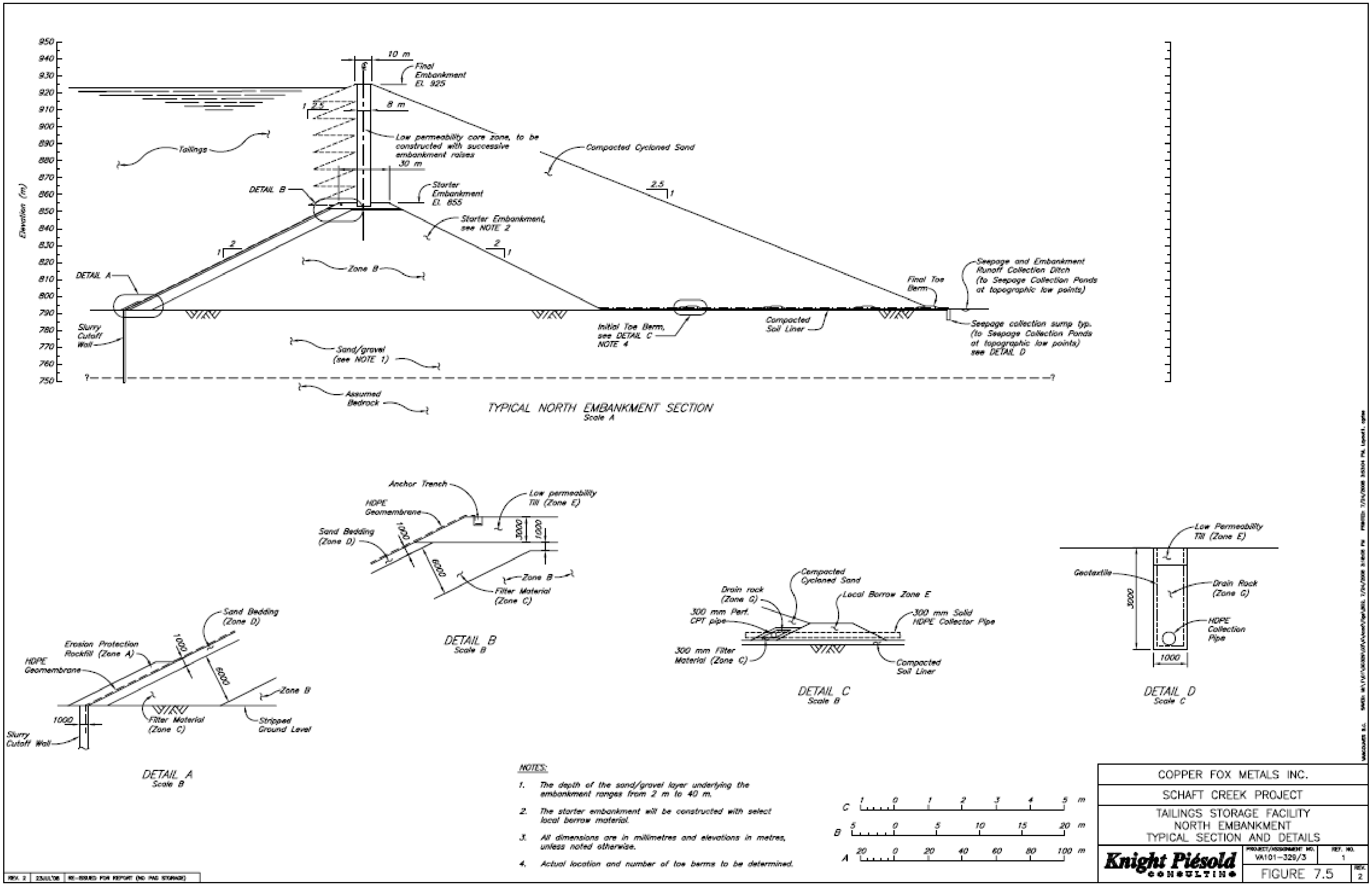


Figure 25.37 North Embankment Typical Section and Details

West Embankment

The West embankment will comprise an approximately 20-m high starter embankment constructed to an elevation of 880 masl. The embankment will be constructed in a similar manner to the North starter embankment using similar materials. An HDPE geomembrane will be installed on the upstream face of the embankment to minimize seepage through the structure.

The geomembrane will be tied into a bentonite-cement slurry cut-off wall installed beneath the embankment upstream toe. The slurry cut-off wall will extend down through the permeable sand and gravel foundation and will be keyed into competent bedrock to form a positive hydraulic cut-off. The depth to bedrock is variable along the length of the embankment, and will be more accurately defined in future geotechnical site investigations. If the depth to bedrock in certain areas exceeds the maximum reasonable constructible depth of a slurry wall, a combination of excavation, slurry cut-off wall, and low-permeability backfill may be used. A low-permeability core zone, initially keyed into the impermeable-crest zone of the starter facility, will be extended vertically upwards through each successive raise. The core zone will be constructed using low permeability glacial till from local borrow.

If a suitable source of borrow material is not available, a low-permeability soil will be manufactured onsite using cycloned sand and bentonite powder.

Above the elevation of the starter wall crest, the embankment will be raised using centreline construction and cycloned tailings to an ultimate elevation of 925 masl resulting in a final maximum embankment height of 60 m. Tailings will be passed through cyclones installed on the embankment. The finer tailings fraction will emerge with the cyclone overflow which will be discharged into the facility. The coarser tailings fraction will emerge with the cyclone underflow and will be spread and compacted using dozers and rollers to form the embankment raise.

A typical section through the West embankment is shown on Figure 25.38.

South Embankment

The South embankment will comprise an approximately 20 m high starter embankment constructed to an elevation of 900 masl. The starter embankment will be constructed using select mine waste sourced from mining operations at the open pit. An HDPE geomembrane will be installed on the upstream face of the embankment to minimize seepage through the structure. The geomembrane will be tied into a bentonite-cement slurry cut-off wall installed beneath the embankment upstream toe. The slurry cut-off wall will extend down through the permeable sand and gravel foundation and will be keyed into competent bedrock to form a positive hydraulic cut-off. The depth to bedrock is variable along the length of the embankment and will be more accurately defined in future geotechnical site investigations. If the depth to bedrock in certain areas exceeds the maximum reasonable constructible depth of a slurry wall a combination of excavation, slurry cut-off wall, and low-permeability backfill may be used.

Above the elevation of the starter wall crest, the embankment will be raised using centreline construction and mine waste, to an ultimate elevation of 925 masl, resulting in a final maximum embankment height of 45 m.

A low-permeability core zone, initially keyed into the impermeable crest zone of the starter facility, will be extended vertically upwards through each successive raise. The core zone will be constructed using low permeability glacial till from local borrow. If a suitable source of borrow material is not available, a low permeability soil will be manufactured onsite using cycloned sand and bentonite powder. Filter and transition zones will be required between the low permeability core and the rockfill comprising the downstream embankment shell.

The filter and transition zones will be constructed using local borrow material. A typical section through the South embankment similar to the section for the West embankment as shown on Figure 25.38.

25.3.7.5 *Tailings Storage Facility Seepage Analyses*

Steady-state seepage analyses for the TSF were carried out to estimate the amount of seepage through the embankments and foundation materials. The analyses were conducted using the finite element computer program SEEP/W©. Seepage rates through the North, West and South embankments were examined.

The seepage rate through foundation materials and embankment fill zones is influenced by the following factors:

- Permeability of the embankment zones;
- Permeability of the foundation materials;
- The thickness and permeability of the tailings stored within the TSF;
- Seepage gradients in the embankment and foundation zones;
- The seepage area (increases during operations).

The seepage flow rate is expected to vary over the life of the TSF as it is gradually filled with tailings and supernatant water. During operation of the TSF, the tailings deposit will increase in thickness and decrease in permeability due to ongoing consolidation.

Foundation conditions incorporated into the seepage analyses were based on information provided by the geotechnical site investigation programs.

Seepage analyses indicate total seepage through the embankment and foundations of around 400 L/s of which approximately 50% is expected to be intercepted by the seepage collection systems and returned to the facility.

A detailed discussion of the TSF seepage analyses and results is presented in Appendix D of the KPL report.

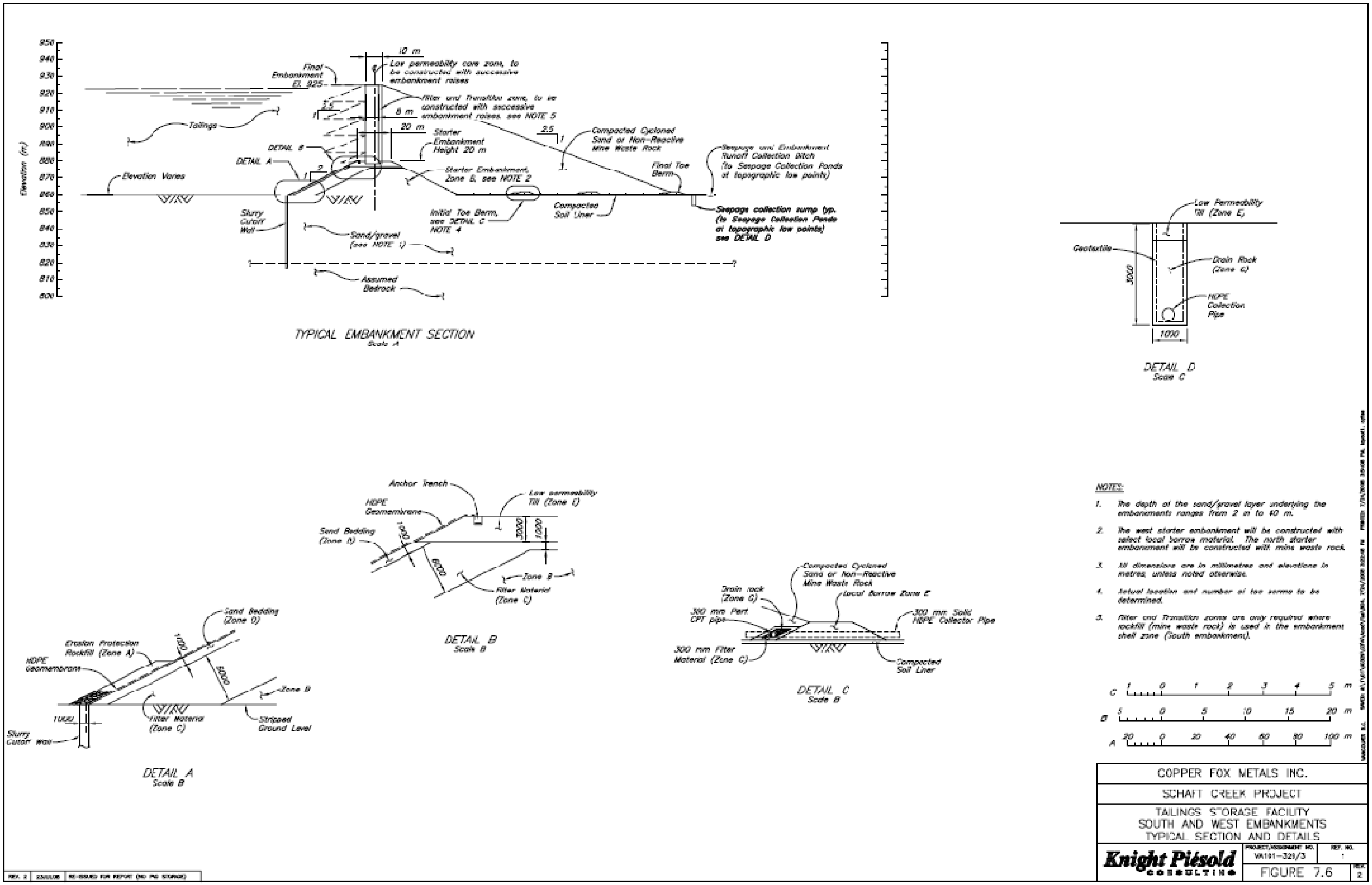


Figure 25.38 South and West Embankments Typical Section and Details

25.3.7.6 *Seepage Collection System*

Seepage from the TSF will result from infiltration of ponded water directly through the embankment fill and the natural ground and pore water released from the tailings mass as it consolidates. The seepage will largely be controlled by the low-permeability HDPE liner installed on the upstream face of the starter embankments and the bentonite-cement slurry cut-off wall installed through the embankment foundations.

Seepage interception systems will be installed at all three embankment locations. A drainage blanket will be installed over a low-permeability soil liner within the downstream shoulder of the starter embankment. Seepage collected in the drainage blanket will be directed to an outfall pipe which will convey the flow to a sub-surface, reinforced concrete, seepage sump located downstream of the ultimate embankment downstream toe. The downstream drainage blanket will be extended progressively as the embankment is raised.

A seepage interception trench will be excavated downstream of the ultimate embankment downstream toe.

The trench will be lined with geotextile and backfilled with drainage aggregate. A perforated drainage pipe installed at the base of the trench will discharge intercepted water to the sub-surface seepage sump.

Ongoing water-quality monitoring will be used to assess the effectiveness of the seepage collection system. In the unlikely event that the seepage collection system is found to not effectively recover seepage it will be necessary to install additional seepage control provisions.

The locations of these seepage intercept systems are indicated on the typical section for the North embankment (Figure 25.37) and for the South and West embankments (Figure 25.38). Figure 25.39 shows cross sections of the seepage collection systems.

25.3.7.7 *Embankment Stability Analyses*

Stability analyses were carried out to investigate the stability of the embankments under both static and seismic-conditions. These involved checking the stability of the embankment arrangement for each of the following cases:

- Static conditions during operations and post-closure;
- Earthquake loading from the Operating Basis Earthquake (OBE) and the Maximum Design Earthquake (MDE);
- Post-earthquake conditions using residual (post-liquefaction) tailings strengths.

The stability analyses were carried out using the limit equilibrium computer program SLOPE/W©. In this program a systematic search is performed to obtain the minimum factor of safety from a number of potential slip surfaces. Factors of safety were computed using Spencer’s Method.

In accordance with international recommendations (ICOLD, 1995) and standard industry practice, the minimum acceptable factor of safety for the tailings embankment under static conditions is 1.3 for short-term operating conditions and 1.5 after closure of the TSF. A factor of safety of less than 1.0 is acceptable for earthquake loading conditions provided that calculated embankment deformations resulting from seismic loading are not significant and that the post earthquake stability of the embankment maintains a factor of safety greater than 1.2.

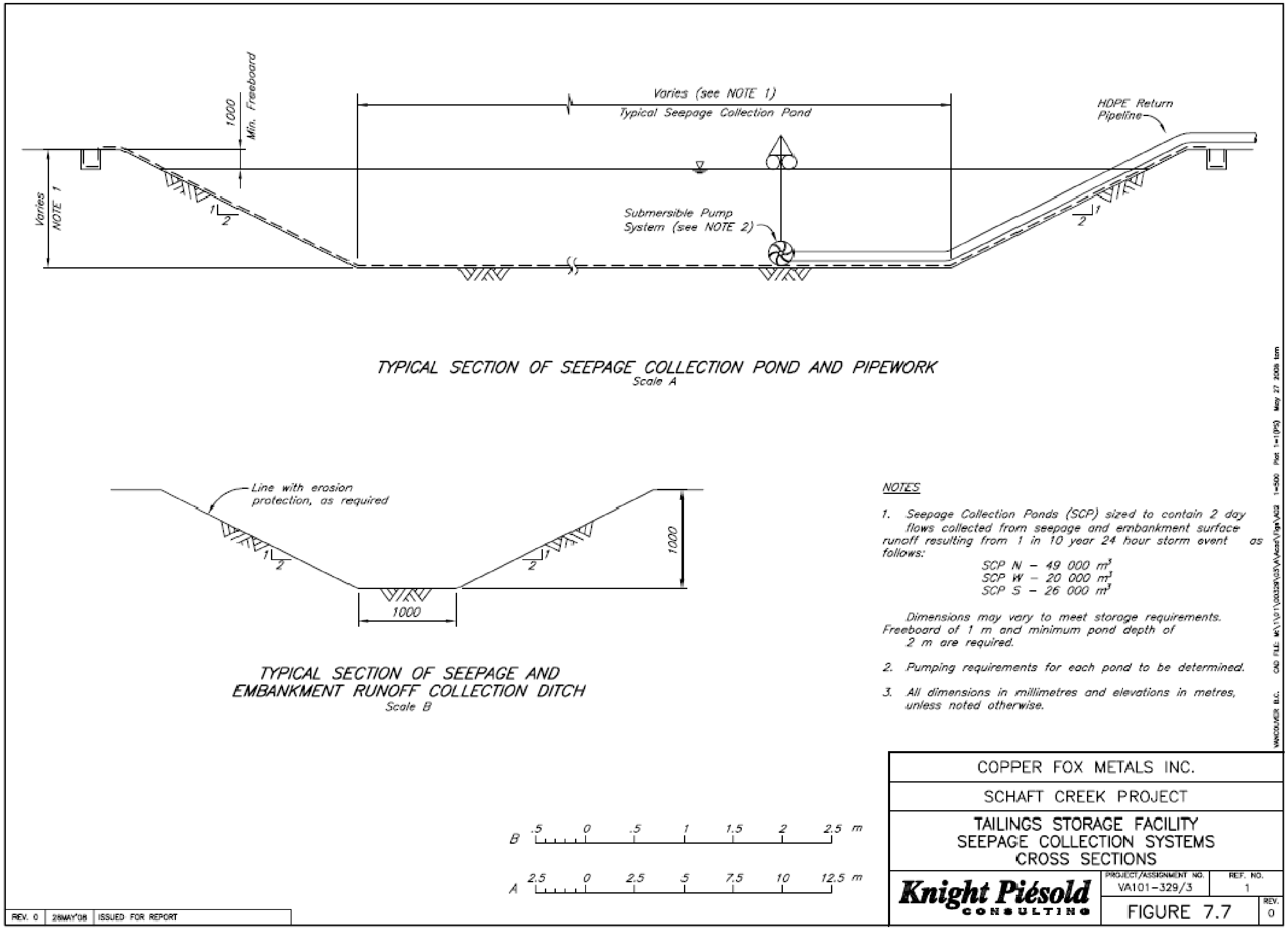


Figure 25.39 Seepage Collection Systems Cross Sections

The results of the stability analyses satisfy the minimum requirements for factors of safety and indicate that the proposed design is adequate to maintain both short-term (operational) and long-term (post-closure) stability.

The seismic analyses indicate that any embankment deformations during earthquake loading from the OBE or MDE would be minor and would not have any significant impact on embankment freeboard or result in any loss of embankment integrity. An analysis of post-earthquake conditions using residual (post-liquefaction) tailings strengths indicated that the stability conditions are the same as under static conditions as the embankment is not dependant on the tailings strength for stability.

Details of the embankment stability analyses are presented in Appendix E of the KPL report.

25.3.7.8 *Access Roads*

The primary access road to the facility will run from the mill area to the North embankment along the western side of the valley.

The road will be positioned alongside the TSF, above the ultimate facility elevation and will be constructed with a minimum running width of 9 m and an overall downward gradient of 0.5% to the north. In addition to carrying the TSF power line, tailings delivery pipe work and reclaim water lines, the road will incorporate the main surface water diversion drain on the western side of the facility, which will intercept surface water runoff from above the facility and convey it around the facility periphery.

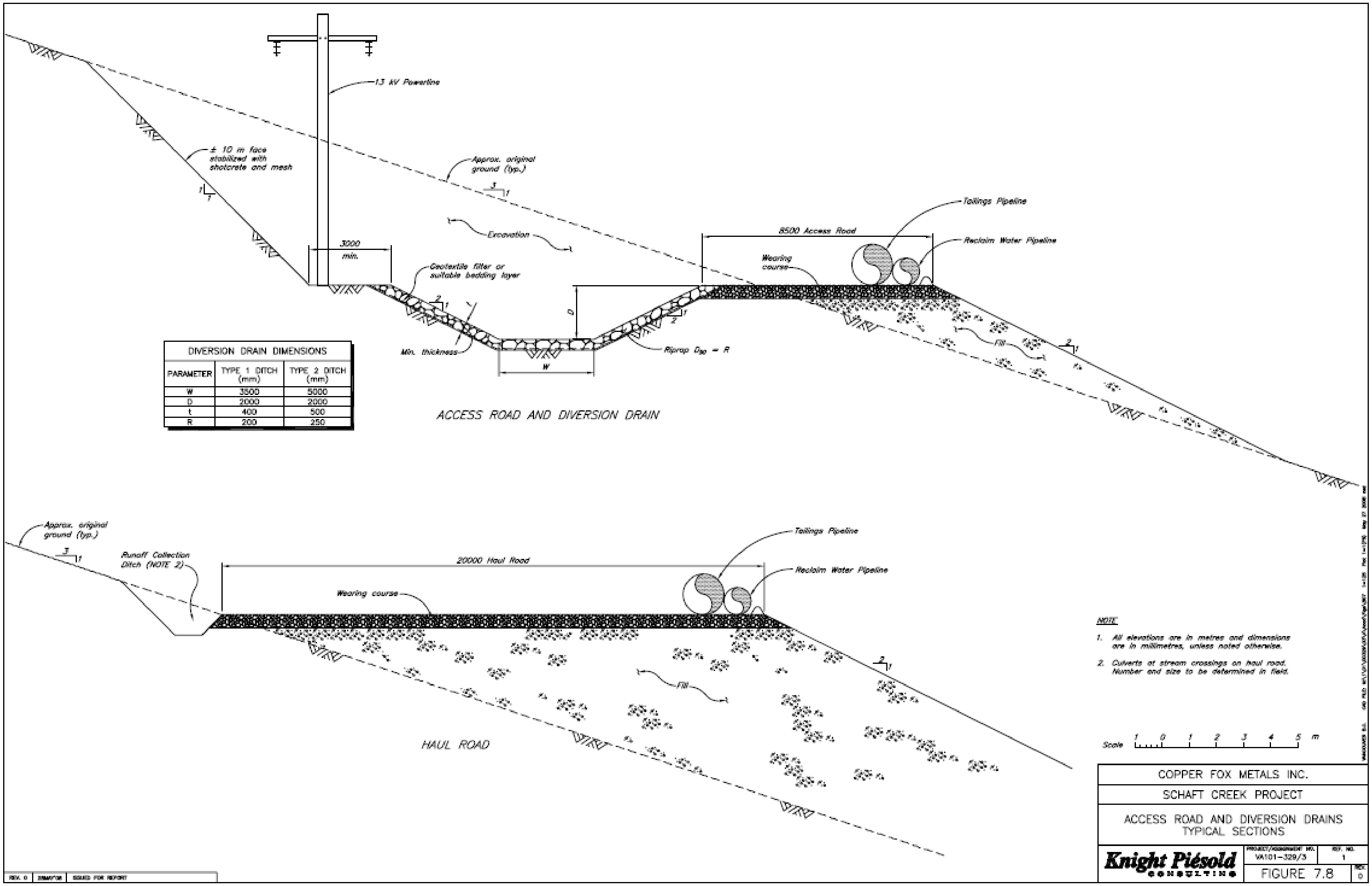
A second access road will be constructed on the eastern side of the valley. It will be constructed above the ultimate facility elevation in a similar manner to the main access road and will also incorporate surface water drainage.

Details of the access roads are shown on Figure 25.40.

25.3.7.9 *Surface Water Mangement*

To minimize the inflow of water into the TSF a peripheral drainage system will be constructed around the facility. The system will comprise two major diversion drains constructed on either side of the valley, immediately above the ultimate facility elevation. The trapezoidal drains will be excavated within the access road cuts and will be sized to carry the peak flow from a 1 in 10 year event. Diversions will be constructed at an overall gradient of 0.5% and will be rock lined. Each drain will terminate in a major drop structure, one located on the southern side of the West embankment and the other on the eastern side of the North embankment. Water collected in the cast-in-place concrete drop structures will be drawn into two 2,134-mm (84”) steel pipes. The flow will be carried to Skeeter Creek downstream of the North embankment and ancillary structures (seepage control works). An energy dissipation structure, comprising a large rock lined stilling basin, will be constructed at the outlet of the conveyance pipework to ensure acceptable flow velocities entering the natural stream channel downstream. A schematic plan and profile are shown on Figure 25.41 Details of the drop structure, pipeworks, and energy dissipation structure are shown on Figure 25.42.

Lined ditches will be excavated along the toe of each of the three embankments to collect surface water runoff from the embankment faces. The approximate alignments of the ditches are shown on the Final Layout Drawings (Figure 25.35). The ditches will report to the seepage collection ponds which are also shown on Figure 25.35. The ponds are located at the topographic low points downstream of the confining embankments.



DIVERSION DRAIN DIMENSIONS

PARAMETER	TYPE 1 DITCH (mm)	TYPE 2 DITCH (mm)
W	3500	5000
D	2000	2000
t	400	500
R	200	250

Figure 25.40 Access Road and Diversion Drains Typical Sections

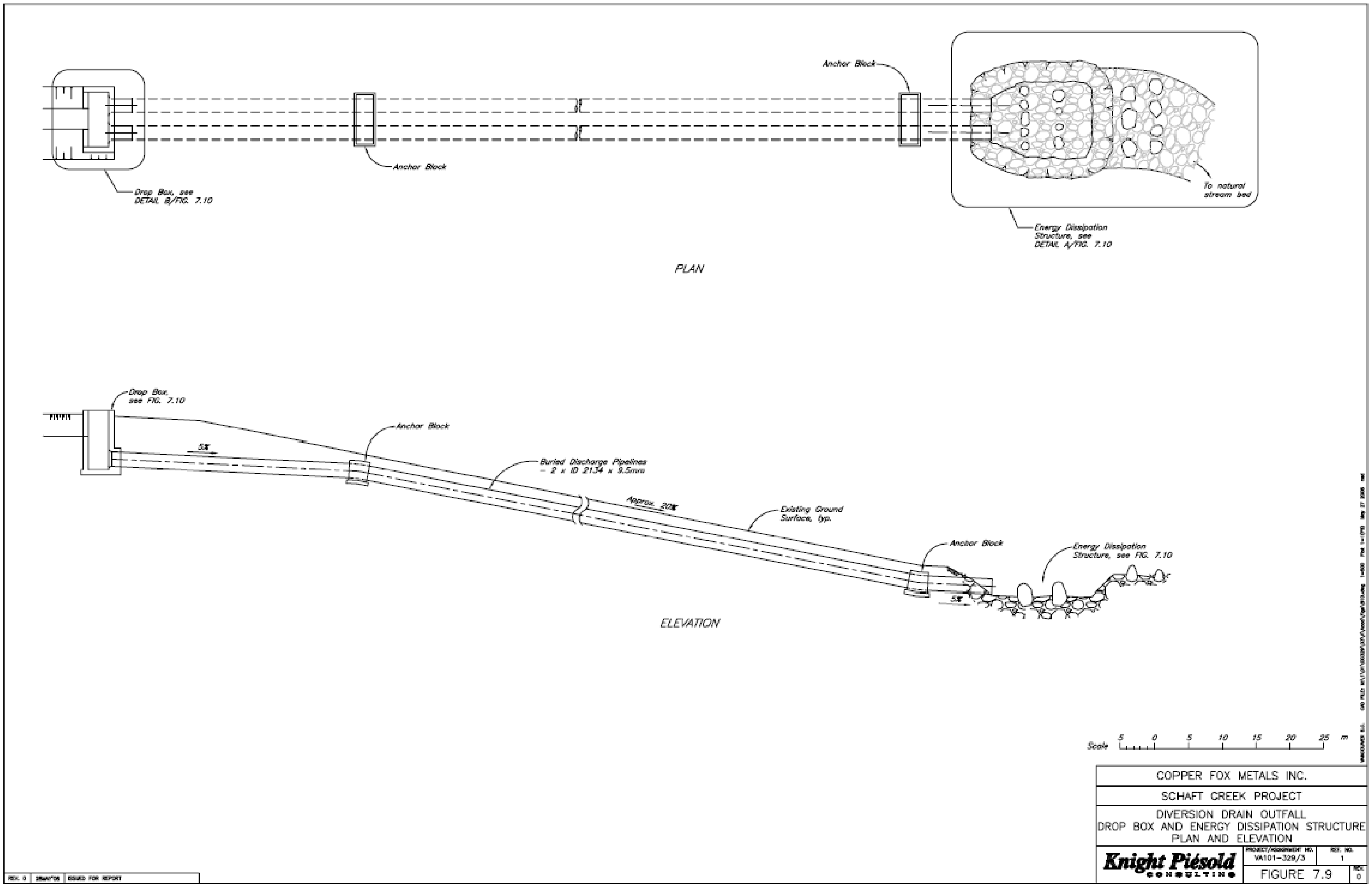


Figure 25.41 Schematic Plan and Profile

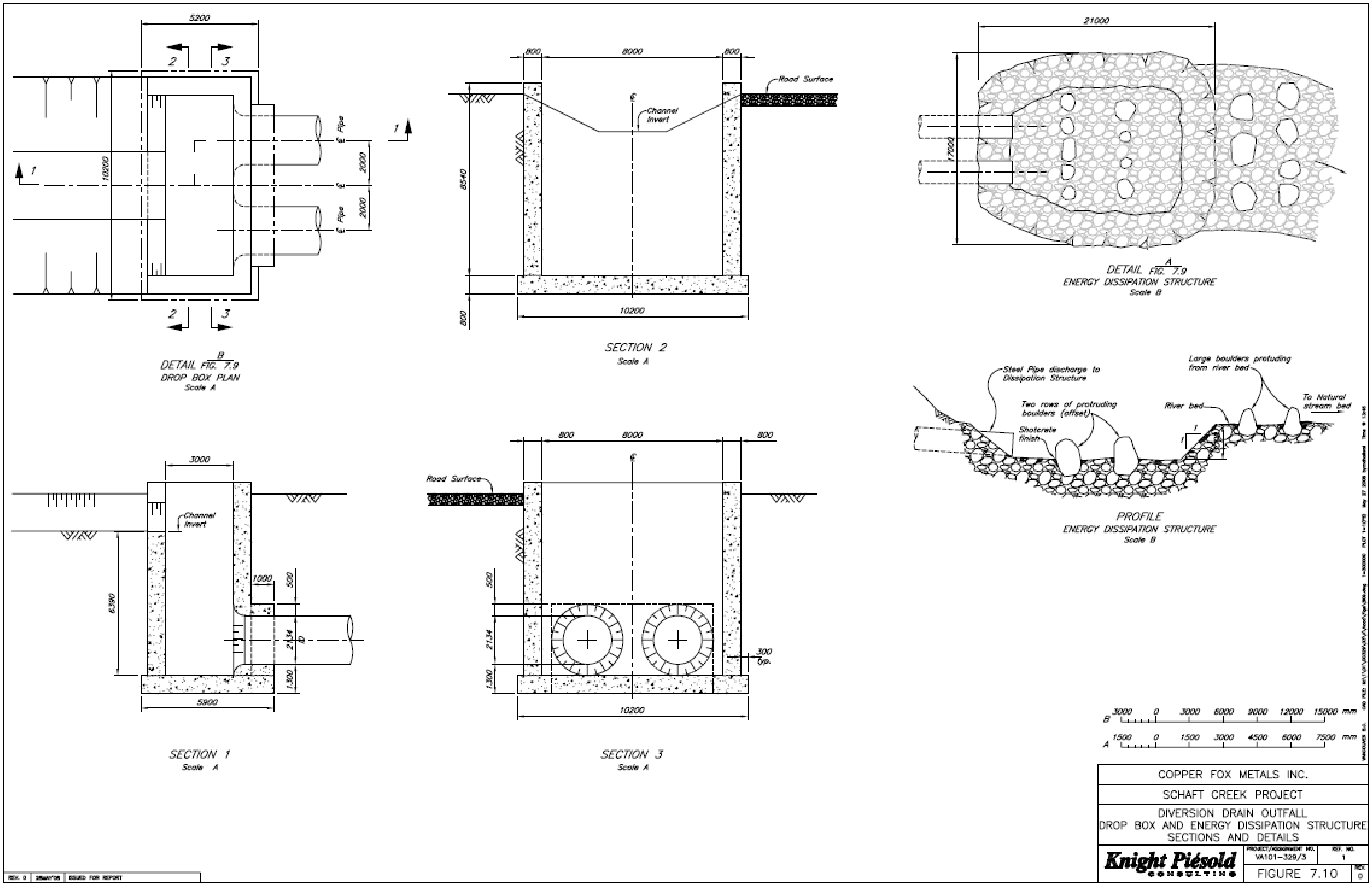


Figure 25.42 Drop Structure, Pipeworks, and Energy Dissipation Structure

The seepage collection ponds have been sized to store the runoff resulting from a 1 in 10 year storm event over a 24-hr period, combined with the steady-state seepage collected in the seepage collection system. Suitable pumping capacity will be provided to recycle the collected water to the TSF.

25.3.7.10 *Tailings Deposition*

Delivery Pipeline

The tailings delivery system will be designed by others. The following provides a brief description of the system.

Tailings from the mill will be delivered to the TSF through a tailings pipeline comprising either HDPE or HDPE lined steel pipe. The pipeline will be positioned along the downhill side of the main western access road.

Tailings will be discharged from the delivery pipeline into the TSF from large-diameter valved off-takes that include rubber-lined steel tees, with appropriate valves and HDPE discharge piping. Inline valves installed at intervals along the delivery pipelines will allow the tailings discharge locations to be moved as appropriate for controlled beach development.

Tailings Deposition

The tailings pipeline will initially extend from the mill to the eastern side of the North embankment. Spigot off-takes will be located along the North embankment crest in the area between the future West embankment and the North embankment and in the area to the south of the future West embankment as shown in the Final Layout Drawing (Figure 25.35).

Once construction of the West starter embankment is complete, the pipeline will be relocated onto the West embankment crest. The pipeline along the North embankment crest will then be removed to facilitate the initial raising of this embankment and will be reinstated on completion. At this point tailings deposition will take place from both embankments and the area north and south of the West embankment.

On completion of the South embankment starter facility a spur line will be installed along the embankment crest extending eastwards from the main tailings delivery line. This will enable tailings deposition to take place from either end of the facility.

Tailings will be deposited from all three embankments and the northwestern side of the impoundment during the operational life of the facility, resulting in the formation of a dish-shaped tailings surface centered about the permanent decant location as shown on the final Layout Drawing (Figure 25.35).

It is envisaged that a booster-pump system may be required to discharge tailings at the far northern end of the impoundment in the later years of operation, as the level of the facility rises.

25.3.7.11 *Reclaim Water*

General

Water will be reclaimed from the tailings pond by a barge-mounted pump station. The water will consist of supernatant from the settled tailings and runoff from precipitation and snowmelt within the catchment area.

A dedicated steel/HDPE pipeline, located along the main western access road will convey the reclaimed water to the process water pond, which is located adjacent to and upgradient of the mill.

Reclaim Barge

The floating reclaim pump station in the TSF will be located in the south-central part of the facility. A temporary reclaim pump station will initially be required in the northern part of the facility. This pump station will pump water from the initial supernatant pond location to the location of the permanent decant.

As the facility fills the supernatant pond will migrate southwards towards the location of the main floating reclaim pump station.

The barge pumps will be controlled from the mill control room, based on the water level in the process water pond. The barge will be fitted with vertical turbine pumps, including standby pumping capacity and all necessary control, check, drainage and isolation valves. One pump will normally be operated at all times during winter to reduce the potential for freezing of the water in the reclaim pipeline.

Reclaim Pipelines

Reclaimed water will be pumped from the reclaim barge to a process water pond at the mill. The operational storage capacity of this pond will be approximately 100,000 m^3. The reclaim pipelines will be graded to minimize high or low sections and to allow for gravity drainage back into the TSF or the process water pond.

The reclaim pipeline from the TSF will consist of sections of large-diameter HDPE and steel pipe. Steel pipe would be used only for the initial high-pressure sections of the pipeline, while HDPE pipe will be used for the remainder of the pipeline.

During the early stages of facility development an HDPE pipeline will be laid from the temporary pond location at the northern part of the facility to the temporary water-storage facility located in the vicinity of the permanent reclaim pump system.

25.3.8 Water Balance

25.3.8.1 *General*

A preliminary water balance model was developed for the TSF to assess the various inflows to and outflows from the facility on a monthly basis, and to determine the variation in supernatant pond volume and required volumes of make up or discharge.

The determination of an accurate water balance for the operating facility is a key part of the facility design process. Due to the limited data presently available, the water balance prepared for this study should be considered as preliminary only. Considerable refinement of the model will be required during the feasibility-design stage when additional data are available.

The present model considers the effects of supernatant release, rainfall runoff and evaporation and is based on the tailings production schedule outlined previously in this report. The water balance models and results are discussed in the following sections.

25.3.8.2 *Parameters and Assumptions*

The preliminary water balance model is based on a number of parameters and assumptions, summarized as follows:

- Tailings throughput of 100,000 tpd at 55% solids;
- Supernatant release of between 45 and 55%;
- Water losses in tailings voids determined for tailings dry density of 1.3 t/m^3 for the first 2 years then increasing to 1.4 t/m^3 by Year 9;
- Fresh water make-up requirement of 6%;
- Average annual precipitation and evaporation of 1,039 mm and 450 mm, respectively;
- Runoff coefficients of 0.8, 0.9 and 1.0 for undisturbed catchment, tailings beach and supernatant pond areas, respectively, and;
- Peripheral diversion drain efficiency of 75%.

Other than slurry water, rainfall runoff from the facility external catchment area and direct precipitation on the facility, the water balance does not consider or make allowance for any other sources of inflow to the facility, e.g. pit dewatering. During the feasibility-design stage, a site-wide water balance incorporating all other site facilities will be required.

The primary inputs and outputs for the water balance model are shown schematically in Appendix F of the KPL report.

25.3.8.3 *Water Balance Modeling*

The TSF water balance model was developed using average climatic conditions and the assumptions listed above. The results of the water balance as discussed in Appendix F of the KPL report are summarized in the following sections.

Pre-Production

The main starter-facility embankment will be constructed prior to the commencement of milling. The volume of water impounded prior to commissioning will be determined by the timing of facility construction both seasonally and with respect to commencement of milling. Assuming that impoundment commences twelve months before first tailings production, the volume of water in the facility at start up will be in the range of 10 Mm^3 to 25 Mm^3. This is equivalent to between four and ten months of total process water requirements. Make-up water from external sources should therefore not be required for start up.

Operations

Based on the design assumptions, the modeling indicates that the facility water balance will be positive under average climatic conditions, generating on average an annual excess of around 4 M m^3 to 7 M m^3 of water. On a monthly basis the water balance will fluctuate, being negative for 5 to 6 months of the year during the low precipitation months of the winter and being strongly positive in the spring and summer months.

As previously noted, the water balance model has been based on a number of assumed values, small changes in which have the potential to significantly impact the overall results of the water balance, particularly given the very large volumes of water passing through the facility. During the feasibility-design stage, significant effort will need to be directed towards better quantifying these assumptions and more accurately defining the facility and site water balances.

To assess the significance of changes to some of the major assumed values a sensitivity analysis was undertaken, as discussed below.

Sensitivity Analysis

Rainfall

No long-term historic, site-specific data are available for the project site. The baseline meteorology report for the project prepared by Rescan, *Schaft Creek Project 2006 Meteorology Baseline Report*, describes attempts to estimate the mean annual precipitation for the site using three different methods. The results obtained varied widely.

Rescan proposes the use of the precipitation data collected for 2006 (1,039 mm) as being representative, but highlights the fact that the monthly variation in the collected data does not reflect the trends in monthly data observed in other regional data.

The 2006 annual precipitation figure is also considerably greater that the precipitation estimated by other methods.

Given the uncertainty associated with both the mean annual precipitation value and the monthly distribution of this precipitation, the water balance was run with a range of mean annual precipitation values as follows: 520 mm (50% of 2006 value), 780 mm (75% of 2006 value), 1,300 mm (125% of 2006 value), and 1,560 mm (150% of 2006 value).

The results indicate that with 520 mm of annual rainfall the facility water balance will be negative with an average annual make-up requirement of between 3 M m^3 to 5 M m^3. With 780 mm of annual rainfall the facility water balance is largely neutral. Annual rainfall higher than the 780 mm will result in a positive water balance with the annual quantity of excess water generated increasing with increasing rainfall.

As noted above the annual excess is around 5 M m^3 for average rainfall of 1,039 mm, increasing to 10 M m^3 for 1,300 mm, and 15 M m^3 for 1,560 mm.

Diversion Drain Efficiency

The above modeling was based on the assumption that the peripheral diversion drains around the facility captured and safely diverted 75% of the surface water runoff from the upper catchment. To assess the impact of drain efficiency the model was also run under average climatic conditions with diversion drain efficiencies of 25% and 50%. The results indicate that the average annual water surplus of 5 M m^3 with 75% drainage efficiency increases to 10 M m^3 and 15 M m^3 for drainage efficiencies of 50% and 25%, respectively.

Conclusions

The water balance should be considered as preliminary only due to the limited data presently available. The water balance model has been based on a number of assumed values. Small changes in the assumptions have the potential to significantly impact the overall results of the water balance. During the feasibility design stage, significant effort will need to be directed towards better quantifying these assumptions and more accurately defining the facility and site water balances.

Two parameters of particular significance are rainfall and diversion drain efficiency. The model was initially run using an assumed average annual rainfall value of 1,039 mm and a drainage efficiency of 75%.

To assess the sensitivity of the water balance to changes in the value of these parameters the model was run with varying rainfall values and diversion efficiencies. The results of the sensitivity analyses indicate that for rainfall of approximately 75% of the assumed average value or less, the facility will operate with a negative water balance, requiring a permanent source of make-up water. For rainfall in excess of this value the facility will operate with a positive water balance generating excess water each year.

Based on the design parameters and assumptions made, the model indicates that the facility will operate with a positive water balance under average climatic conditions, generating between 4 M m^3 and 7 M m^3 of excess water annually. For example the model indicates an annual excess of water in the facility of some 5 M m^3 under average precipitation conditions and diversion efficiency of 75%. This excess increases to 10 M m^3 and 15 M m^3 for drainage efficiencies of 50% and 25%, respectively. Over the life of the facility the total excess volume is estimated to be in the range of 90 M m^3 to 160 M m^3.

25.3.9 Instrumentation and Monitoring

25.3.9.1 *General*

Extensive monitoring of all aspects of the Tailings Storage Facility (TSF) operations should be undertaken. Complete details of the monitoring program will be included in the Operations, Maintenance, and Surveillance (OMS) manual that will be produced for the facility during the final-design stage.

Geotechnical instrumentation will be installed in the tailings embankment and foundation during construction and over the life of the project.

The instrumentation will be monitored during the construction and operation of the TSF to assess embankment performance and to identify any conditions different to those assumed during the design phase. Amendments to the ongoing designs and/or remediation work can be implemented to respond to the changed conditions, should the need arise.

Geotechnical instrumentation, including piezometers and movement monuments will be installed at selected planes on the embankments. Groundwater monitoring wells will be installed at suitable locations downstream of each embankment.

25.3.9.2 *Instrumentation*

Vibrating wire type piezometers will be installed in the embankment foundation, fill and tailings materials to measure pore water pressures during initial placement and during operations. The piezometers will be distributed throughout the various foundation and fill zones to provide a spectrum of monitoring data. The piezometer leads will be appropriately routed from the fill to readout panels for ease of monitoring.

Movement monuments will be installed on the embankment crest following the completion of selective embankment raises to monitor deflections along the slope and crest of the embankments. Periodic surveying of the monument locations will provide early warning of movements and possible acceleration of movement which often occurs prior to failure.

Groundwater monitoring/recovery wells will be installed at appropriate locations along the downstream toe. The wells will be used to recover samples for water quality monitoring.

25.3.9.3 *Monitoring Program*

Monitoring of the facility will be undertaken routinely both during operations and post-closure, in accordance with the facility operations manual. Monitoring falls into two basic types:

- Short-term operations monitoring – this includes items such as off-take location, condition of pipe for tailings delivery and water reclaim, thickness of deposited tailings etc. which are part of ensuring the facility is operating as designed;
- Performance monitoring – this includes items such as checking monitoring wells for water quality, checking levels in the piezometers within embankments, tailings level surveys and water flow measurements etc., which are used to monitor the

performance of the facility and refine future embankment lift levels and final tailings extent and to ensure that the project is meeting all its commitments in regard to a safe and secure operation.

25.3.9.4 *Facility Inspection*

In addition to the routine inspections carried out by mine personnel on a shift, daily, weekly and monthly basis, the facility should be regularly audited by a suitably-qualified geotechnical engineer to ensure it is operating in a safe and efficient manner. Such audits should be conducted annually.

25.4 Environmental Considerations

25.4.1 Environmental Liabilities

A review of the permitting and environmental activities for the Schaft Creek project has been provided by Copper Fox Metals Inc. (Copper Fox). Copper Fox is undertaking exploration and geotechnical investigations and environmental baseline studies at the Schaft Creek property. These activities are supported by a 60-person camp and an existing airstrip. Reclamation of the property is required once these activities are complete in the event that no further development is planned. Copper Fox has posted a reclamation bond with the British Columbia (BC) 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 BC 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. A general reclamation plan has been provided to the BC MEMPR.

25.4.2 Environmental Baseline Studies

The Schaft Creek project environmental baseline studies (Schaft Creek Environmental and Social Work Plans, April 2008) have been reviewed and approved by various regulatory agencies, local and municipal government representatives and the First Nations. These studies set the foundation of the environmental assessment applications to be made to the BC and federal government.

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 was expanded in 2007 to build upon the studies from 2006 and address new project developments:

- Tahltan (traditional) knowledge;
- Vegetation;
- Wetlands;
- Ecosystem mapping;
- Soils;
- Wildlife;
- 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.

The 2008 environmental and social programs were designed to build upon the information gathered previously and address details of the project design as new information becomes available. New studies for 2008 include:

- Geohazards;
- Geofluviomorphology (Mess Creek).

Additional investigations are planned to support the EA Application:

- Traffic study for Highway 37;
- Visual effects assessment;
- Water quality modeling;
- Hydrogeologic and hydrologic modeling.

25.5 Reclamation and Closure

25.5.1 Introduction

For the Preliminary Feasibility Study (PFS) the primary concepts for reclamation and closure of the Schaft Creek project have been provided by Knight Piésold Ltd. (KPL) and by Moose Mountain Technical Services (MMTS) in their respective reports on the Tailings Storage Facility (TSF) and the mining plan.

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.

Although there is little surficial soil in the pre-mining topography, any suitable surficial soils excavated during mining will be stored in stockpiles and used to cap the re-contoured landscape at decommissioning. Whenever possible, direct placement of the suitable topsoil material will be carried out in order to avoid stockpile losses and re-handling costs.

Post closure landform, reclamation and acid rock drainage (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 evaluations of the rock. Detailed design criteria will be adjusted based on the results obtained from these future studies and the requirements of all applicable permits.

Copper Fox Metals Inc. (Copper Fox) is planning to complete a soils baseline study by 4th Quarter 2008. This study will provide the information needed to evaluate the types of soils and their availability for reclamation. The information on the soils will provide additional information to be incorporated into the feasibility study with regard to the reclamation design and associated costs. At this PFS stage of the project, neither the reclamation schedule nor definitive reclamation costs have been determined.

25.5.2 Reclamation and Closure by Facility

25.5.2.1 *Tailings Storage Facility*

The primary objective of the closure and reclamation initiatives will be to eventually return the Tailings Storage Facility (TSF) site to a self-sustaining system. The TSF will be required to maintain long-term stability, protect the downstream environment and manage surface water. Activities that will be carried out during operations and at closure to achieve these objectives are discussed in the following sections.

Decommissioning and Closure

Upon mine closure, surface facilities will be removed in stages and full reclamation of the TSF will be initiated. General aspects of the closure plan include:

- Selective discharge of tailings around the facility during the final years of operations to establish a final tailings beach that will facilitate surface water management and reclamation;
- Capping exposed tailings beaches with a layer of rockfill, followed by placement of suitable growth medium and topsoil layer. A pond and wetlands system will be maintained over much of the tailings surface;
- Dismantling and removal of the tailings and reclaim delivery systems and all pipelines, structures and equipment not required beyond mine closure;
- Construction of an outlet channel/spillway at the North Embankment to enable discharge of surface water from the TSF to Skeeter Creek;
- Removal of the seepage collection system at such time that suitable water quality for direct release is achieved;
- Removal and re-grading of all access roads, ponds, ditches and borrow areas not required beyond mine closure;
- Long-term stabilization of all exposed erodible materials.

Ongoing Monitoring Requirements

The seepage collection ponds and recycle pumps at the TSF will be retained until monitoring results indicate that any seepage from the TSF is of suitable quality for direct release to downstream waters. The groundwater monitoring wells and all other geotechnical instrumentation will be retained for use as long-term monitoring devices.

Post-closure requirements will also include an annual inspection of the TSF and an on-going evaluation of water quality, flow rates and instrumentation records to confirm design assumptions for closure.

25.5.2.2 *Open Pit Mine*

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 direct it to the watercourses as established by 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 will not be re-sloped.

25.5.2.3 *Waste Rock Facilities*

Mine waste dumps will be comprised of non-reactive waste rock and will be constructed in series of lifts at 37° inter-slope angles with appropriate berm widths. This will effectively result in an overall slope angle of 26°. The overall 26° 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° including the 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.

25.5.2.4 *Roads and Drainages*

Decommissioned mine roads will be scarified and capped with suitable surficial soils. Dykes and dams that are exposed above the water line will also be scarified and capped with suitable soils. The surfaces will then be seeded to establish vegetation.

25.5.2.5 *Exploration Activities*

All surface facilities related to exploration activities will be removed and all areas of disturbance will be reclaimed.

A reclamation bond has been posted with the British Columbia Ministry of Mines, Energy and Petroleum Resources (BC MEMPR) based on the current exploration and development activities. The bond has been deemed sufficient by the BC MEMPR to reclaim the property from its current state. It is anticipated that the bond will be increased as the areas of disturbance are expanded.

25.5.2.6 *Buildings and Infrastructure*

All buildings and associated infrastructure will be decommissioned and materials will be recycled where practical. The areas that have been leveled for the plant sites will be scarified and capped with suitable soils. The surfaces will then been seeded to establish vegetation.

25.5.3 Monitoring and Reporting

Monitoring and reporting will meet the requirements of all permits that are obtained for the Schaft Creek project. It is likely that the monitoring requirements will be expanded beyond the monitoring requirements for the TSF alone.

25.5.4 Reclamation Schedule and Cost

The information required to develop a reclamation schedule and the associated costs for reclamation is not currently available. The information that is required, such as a baseline study on soils is in progress. The details of the reclamation design including the schedule and cost will be developed during the feasibility study for the Schaft Creek project and presented in the feasibility study report.

25.6 Socioeconomics

25.6.1 Introduction

This section was provided by Robert Simpson, President of PR Associates.

25.6.2 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 Metals Inc. (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.

25.6.3 First Nations

An examination of asserted traditional territory, Treaty Boundaries, Statement of Intent boundaries and known consultation boundaries primarily identified by the British Columbia 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, 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.

25.6.4 Public Consultation

PR Associates' general approach for notifying and consulting with public stakeholders includes the following principles:

25.6.4.1 *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.

25.6.4.2 *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.

25.6.4.3 *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.

25.6.4.4 *Consultation, not Consensus*

Input from the public, along with information gathered through technical, environmental and social impact studies, will inform Copper Fox decisions.

25.6.5 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;
- 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 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.

25.6.6 Public Consultation Activities

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 mail outs;

Providing comprehensive reporting of the process and results of the consultation process, including consultation summaries to support the EAC and CPCN applications.

25.7 General Services and Administration (G&A)

25.7.1 Introduction

Samuel Engineering, Inc. (SE) has produced an updated estimate for the Schaft Creek general and administrative (G&A) costs. These costs include everything that is not directly attributed to operations, as outlined in Table 25.26. The cost estimates have been revised for the Preliminary Feasibility Study (PFS) based on more recent and more reliable information. They are significantly higher than the costs estimated for the Preliminary Economic Assessment (PEA) primarily due to the costs associated with operation of the camp and fly-in, fly-out transportation costs. These costs have risen to over $18 million per year from an original estimate of $7 million per year, which increases the total G&A costs from about $17 million per year to approximately $33 million per year. However, the basis for changing the costs indicates that the new costs are more realistic, as outlined in the following sections of this report. Of course some of the increases in costs are due to the fact that the operation has been increased to 100,000 tpd from the previous 65,000 tpd.

Table 25.26
Summary of Operating Cost – G&A (Life Of Mine Average Annual Costs)

Description	Type	Annual Cost	Cost/Tonne Ore
Salaried Personnel	Fixed	$ 4,470,000	$ 0.1242
Hourly Labour	Fixed	$ 1,492,992	$ 0.0415
Power	Fixed	$ 616,480	$ 0.0171
Vehicle Operating & Maintenance	Fixed	$ 115,385	$ 0.0032
Access Road & Power line Maintenance	Fixed	$ 1,410,000	$ 0.0392
Site Avalanche Control	Fixed	$ 650,000	$ 0.0181
Communications	Fixed	$ 75,000	$ 0.0021
Camp Operations	Fixed	$ 8,627,693	$ 0.2397
Fly-in, Fly-out Operations & Airfield Operations	Fixed	$ 10,000,000	$ 0.2778
Safety Supplies / Incentives	Fixed	$ 300,000	$ 0.0083
Offsite Training & Conferences	Fixed	$ 300,000	$ 0.0083
Insurance	Fixed	$ 1,000,000	$ 0.0278
Corporate Services and Travel	Fixed	$ 350,000	$ 0.0097
Environmental	Fixed	$ 1,000,000	$ 0.0278
Security & Medical	Fixed	$ 400,000	$ 0.0111
Professional Membership Costs	Fixed	$ 30,000	$ 0.0008
Community Development	Fixed	$ 1,000,000	$ 0.0278
Land Holding	Fixed	$ -	$ -
Consultants	Fixed	$ 500,000	$ 0.0139
Misc. Computer Equipment/Software	Fixed	$ 250,000	$ 0.0069
Misc. Office Supplies	Fixed	$ 200,000	$ 0.0056
Misc. Freight & Couriers	Fixed	$ 100,000	$ 0.0028
Recruiting and Relocation	Fixed	$ 200,000	$ 0.0056
Legal, Permits, Fees	Fixed	$ 500,000	$ 0.0139
General and Administrative Costs Total Life of Mine Average Annual Costs		**$ 33,587,550**	**$ 0.9330**

25.7.2 G&A Personnel and Hourly Labour

Labour costs include the costs for both salaried and hourly personnel. The positions allocated to G&A expenses include:

- Site management;
- Marketing and sales;
- Environmental management;
- Human resources;
- Community relations;
- Purchasing and warehousing;
- Safety and security;
- Accounting;
- Access road maintenance;
- General site labour requirements.

25.7.3 Power Costs

Power costs are based on the Electrical Load Study that was completed by SE electrical department. The cost of the power has been calculated using the BC Hydro rates for maximum power draw and average power draw. The costs allocated to G&A expenses include the power demand required for operation of the equipment associated with the airfield, the administration building, laboratory, the main camp, site lighting requirements and pumping costs associated with the fresh water supply. The average power demand for these areas is approximately 13 M kWh/y.

25.7.4 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.

25.7.5 Access Road and Power Line Maintenance

25.7.5.1 *Access Road Maintenance*

Costs for maintenance and operation of the Access Road were provided by McElhanney Consulting Services Ltd. (McElhanney).

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 axel 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;
- Sub-grade repairs;
- Rock scaling;
- 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;
- Sign maintenance.

The estimated annual cost of regular maintenance is $8,000 per km. The length of the access road is 110 km so the cost of regular maintenance is estimated at $880,000.

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.

This brings the total cost for maintenance of the access road to $1,080,000 per year.

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.

Powerline Maintenance Costs

The cost for maintenance of the power line is based on $3,000 per km. The powerline is the same length as the access road, i.e. 110 km. Therefore, the total allocation for powerline maintenance is $330,000.

25.7.6 Site Avalanche Control

Due to the location of the Schaft Creek site and the climate, it is important to include an estimate of the cost associated with avalanche control. An annual allowance was used based on experience with similar operations.

25.7.7 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.

25.7.8 Camp Operations

This includes the cost for operating and maintaining the permanent camp facilities. The camp will be operated by a contractor. The cost is based on a quotation provided by CJL Enterprises Ltd. for a 450-person man camp. Their quotation has been modified to account for the size of permanent camp that is anticipated at Schaft Creek. A summary of the costs for the operation of the camps is provided in Table 25.27.

Table 25.27
Summary of Operating Cost – G&A (Life Of Mine Average Annual Costs)

Description	Persons	Annual Cost
Annual Food Cost at $22.22 per day, 365 d/y	462	$ 3,972,173
Annual Manpower Cost to Operate Camp		$ 4,655,520
Total Cost		**$ 8,627,693**

The number of people is based on the total number of people who will be working at the site at any given time.

Due to the working schedule of two weeks on site followed by two weeks off site, it is assumed that a little over half of the total employees will be on site at any given time which is 311.

In addition to the direct employees of Schaft Creek an allowance of 50 visitors and 50 people to be housed at the Bob Quinn camp was made.

Finally, food costs for the 51 people that work at the camp were also included in the cost estimate. This means that the total number of people who will be fed each day is 462.

25.7.9 Fly-in, Fly-out Operations and Airfield Operations

This includes the cost to operate and maintain the airfield operations and the contracted 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. The costs are estimated on the basis of quotations received from contract air service companies.

Due to the unavoidable uncertainties relating to the actual hiring and staffing demographics, the annual passenger airlift operating expenditure is given in terms of average annual expenditures per employee working at the site.

Assuming a biweekly staffing rotation, the average expense per employee is $200 per week, $400 round trip for each biweekly rotation and $10,000 per year for a 50-wk year that assumes two weeks of vacation per year per employee, i.e. one missed rotation. From an overall project perspective, this equates to $6.22 M annually for 622 employees, which is the life-of-mine average annual number of employees that will be required during operations.

As part of this Preliminary Feasibility Study (PFS), input from many Canadian and U.S. air carriers that operate appropriate aircraft for serving the airfield at the Schaft Creek mine (N57 32 18 W131 01 09) was solicited. A spreadsheet was created that assumes a Dash 8 aircraft (100 series with 37 seats) operating at a cost of $0.052 (Canadian) per revenue passenger kilometer with a 100% load factor. The average per flight cost of $7,401 (Canadian) and the average cost of $200.02 (Canadian) per passenger between the Schaft Creek mine airfield and Smithers, BC.

The allocated budget for passenger airlift operating expenditures is $10 million annually in order to make allowance for more distant recruiting than Smithers, to allow for transport of cargo in addition to the personnel and to make allowance for maintenance, snow removal, etc. from the site airfield. Recruiting from Juneau and other closer airports could be used to mitigate other inflationary influences and to avoid cost overruns. The cost projections for air transportation assume that management is able to hire and schedule employees in such a way that aircraft load factors are close to or at 100%.

25.7.10 Safety Supplies and Incentives

A total of $300 per man per year was allowed for safety support.

25.7.11 Off-site Training and Conferences

A total allowance of $25,000 per month is included for personnel to attend off-site training, conferences and seminars.

25.7.12 Insurance

This includes general liability, risk, and vehicle insurance policies. An annual allowance was used based on experience with similar operations.

25.7.13 Corporate Services and Travel

An annual allowance was used based on experience with similar operations.

25.7.14 Environmental

This includes costs associated with environmental sampling and monitoring, analysis of surface and ground water, as well as surface flow measurements. An annual allowance was used based on experience with similar operations.

25.7.15 Security and Medical

This includes costs for contracted security and medical services. An annual allowance was used based on experience with similar operations.

25.7.16 Professional Membership Costs

An annual allowance was used based on experience with similar operations.

25.7.17 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 Inc. current community development program. The allowance does not include the cost of staff labour. The labour costs are included in the Salaried Personnel costs for HR/Community Relations.

25.7.18 Land Holding

No allowance was made for the cost of land holding.

25.7.19 Consultants

An annual allowance was used based on experience with similar operations.

25.7.20 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.

25.7.21 Miscellaneous Office Supplies

An annual allowance was used based on experience with similar operations.

25.7.22 Miscellaneous Freight and Couriers

An annual allowance was used based on experience with similar operations.

25.7.23 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.

25.7.24 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.

25.8 Project Execution

25.8.1 Overview

The purpose of the Plan of Execution (PE) is to provide a comprehensive plan for the development and implementation of the Schaft Creek project. The PE addresses all aspects of project development, including objectives, scope, and strategies. The PE provides a plan for engineering, procurement, construction, startup, and commissioning of the plant facilities and infrastructure and addresses the roles, responsibilities, and management plans required to execute and manage the work.

25.8.2 Project Schedule

An EPC schedule was developed for the project and summarized below.

- Kick-off feasibility study .... August 1, 2008
- Environmental assessment submitted .... March 2, 2009
- Road permit application submitted .... March 2, 2009
- Feasibility study completed .... March 31, 2009
- Teck Cominco review period completed .... July 31, 2009
- Financing period completed .... October 2, 2009

- Road permit approved .... October 1, 2009
- Plant permit approved .... October 1, 2009
- Financing in place .... October 2, 2009
- Kick-off detailed engineering .... October 5, 2009
- Place PO for mills .... October 9, 2009
- Place PO for electric shovels .... October 5, 2009
- Construction start .... October 28, 2009
- Place PO for crushers .... Dec. 18, 2009
- Airfield upgrades complete .... May 17, 2010
- Access road complete .... June 30, 2010
- Place PO haul trucks .... August 30, 2010
- Place PO for electric drills .... August 30, 2010
- Temporary camps complete .... October 25, 2010
- Detailed engineering complete .... February 4, 2011
- Permanent camp complete .... March 4, 2011
- Begin tailings starter dam .... June 30, 2011
- Power line complete .... March 27, 2012
- Begin mine development .... October 15, 2012
- Complete tailings starter dam .... Dec. 12, 2012
- Mechanical completion .... July 18, 2013
- Start-up complete .... Dec. 5, 2013
- Mine ramp-up complete .... March 12, 2014

25.8.3 Objectives

The PE is designed to achieve the following objectives:

- Uncompromised safety;
- Conformance to the budget;
- On-schedule completion;
- Compliance with project quality standards and expectations;
- Maximization of participation by Tahltan's;
- Environmental compliance.

25.8.4 Project Management

The project management team will be an integrated organization of globally sourced management professionals. This team will develop and implement a project procedures manual that will describe all key areas of project development and completion:

- Project management;
- Engineering management;
- Procurement;
- Logistics and transportation;
- Construction;
- Commissioning and startup;
- Quality assurance;

- Health and safety;
- Communications;
- Project controls;
- Project schedule;
- Project closeout.

The management team will provide critical project execution guidance and oversight to ensure timely project completion. Key activities will include:

- Organize and conduct a comprehensive project kickoff meeting;
- Ensure that project operating procedures reflect the requirements of the quality management system and are adequate and up to date;
- Manage engineering, procurement, and construction in accordance with approved budgets and schedules;
- Ensure timely processing of budget and schedule change requests and revisions to approved budgets;
- Ensure that engineering deliverables (e.g., drawings, specifications, requisitions, etc.) comply with applicable government regulations and sound engineering practices. Coordinate with the discipline chief engineers to conduct peer reviews of deliverables as required;
- Ensure that required reviews and approvals are provided and documented;
- Analyze financial and execution risks to project performance;
- Conduct risk review meetings with key project and management personnel, and develop and implement action items;
- Oversee the development of an integrated activity schedule that will plan and prioritize all the major activities;
- Establish an execution technology implementation plan with regard to automation tools, software, work processes, procedures, and design execution centers.; and
- Ensure the project is appropriately staffed with competent personnel.

25.8.5 Engineering

All engineering work and detailed design packages such as roads, infrastructure, and tailings containment will be executed in North America.

25.8.5.1 *Codes and Standards*

Design of the Schaft Creek project processing facility and infrastructure requirements will be done in accordance with all applicable and acceptable Canadian design and construction codes and standards. Where no applicable Canadian codes or standards exist, North American codes and standards will be applied.

25.8.5.2 *Electrical, Control Systems and Instrumentation*

To the greatest extent possible, major equipment components, i.e. motor control centers (MCCs), switchgear, etc., will be bundled into pre-engineered and prefabricated containerized units or power distribution centers (PDCs) to reduce the number of onsite craft man hours.

25.8.5.3 *Mechanical and Piping*

To the greatest extent possible, equipment and piping systems will be skidded, preassembled, or modularized by the manufacturer to reduce onsite craft man hours. A modularization/preassembly study will be performed early in the project to determine the feasibility of modularization and preassembly design considerations.

25.8.5.4 *Civil and Structural*

All site-specific conditions will be incorporated into the design; examples may include soil conditions and other key design criteria outlined in geotechnical reports and provided in site design criteria.

25.8.6 Procurement

25.8.6.1 *Purchasing*

Purchasing activities include:

- Contracts for supply of critical-path, long-lead items of equipment, particularly crushing and grinding equipment, will be awarded early and at risk (prior to final approval of project funding);
- Other long-lead equipment will require early issue of request for quotation (RFQs) to allow award immediately after project funding is approved;
- A comprehensive expediting and inspection program will be implemented to ensure timely delivery of vendor data and equipment;
- Delivered goods will be stored in a secure and protected lay-down area;
- Special attention will be given to the expedited award of purchase orders for classification equipment, electrical transformers, switchgear, PDC modules, conveying equipment, and major pumps;
- Most of the major equipment and material is expected to be sourced in the United States, Canada, China, Germany and South America. Prefabricated, skid-mounted packages will be considered to the greatest extent possible. Prefabricated modules will be equipped with piping, valves, wiring, and instrumentation to expedite erection on site.

25.8.6.2 *Contracting*

- Bid documents for construction contracts will be issued in accordance with the project contracting plan;
- Early issue of bid documents for construction contracts will be performed based on the schedule. The access road will be completed early to meet the critical path;
- A detailed contracting plan, indicating scope breakdown and contract type, will be developed.

25.8.6.3 *Expediting*

Expediting activities will include both telephone contacts and visits to vendors' manufacturing facilities. Expediting reports will be fed into the material control reporting system to provide accurate, updated, and easily accessible online reports on the status of equipment and materials.

25.8.6.4 *Inspections*

A detailed plan for inspection will be developed to ensure that materials and equipment are supplied in accordance with the purchase order and contract specifications. Global shop inspection service companies will be used.

25.8.6.5 *Traffic and Logistics*

A traffic and logistics plan will be developed for traffic services for the project, including the selection and negotiation of all agreements with freight forwarding agents, customs agents, and inland transportation companies. Material delivery status will be tracked from the points of shipment until receipt.

25.8.7 Construction

- The management team will manage the site activities of all onsite general contractors and specialty construction contractors;
- Construction of the process plant and infrastructure facilities will be performed by contractors specialized in the scope of work as described in the request for proposal (RFP) documents. Contracts will be awarded to major construction contractors following competitive bidding based on bid documents prepared by the engineer. The bid documents will clearly identify the limits of the scope of work and contractor-supplied items as well as the engineering work packages developed to support the scope of work;
- Specific timing for all engineering work packages and construction RFP packages will be included in the project master schedule;
- Initial site activities will include:
    - Geotechnical investigations;
    - Construction of overhead power lines; and
    - Construction of the access road between Bob Quinn and the site.
- Once the access road construction is complete, the following activities will proceed:
    - Upgrading of the airstrip;
    - Mine prestripping and initial mine development;
    - Construction of the tailings dam;
    - Development of the water supply;
    - Establishment of temporary security facilities;
    - Establishment of communications systems;
    - Mobilization and establishment of warehousing and temporary construction facilities;

- Installation of the construction mancamp; and
- Construction of the administration building and permanent ancillary facilities that may be used during construction and startup.

25.8.8 Commissioning and Startup

Plant startup is defined as when feed is introduced to the primary crusher and the plant. Commissioning and startup will be carried out by a team of plant startup professionals. The sequencing and scheduling of mine development, and the completion of the process plant and ancillary facilities will be developed in detail to ensure a timely startup of the project.

Plant startup will be initiated with the preparation of an overall plan for acceptance testing, safety, lock-out tag-out, compilation of instruction manuals, and supply of reagents, spare parts, and supplies. Process control system checkout will also be included.

Training of operational personnel will include onsite instruction, training at similar existing facilities, training at vendor facilities, and in-plant training when the facilities are completed. Maintenance personnel will obtain similar training and will participate in the commissioning of the plant.

25.9 Operating Cost Estimate

25.9.1 Introduction

Mining-related operating costs for the Schaft Creek project have been developed by Moose Mountain Technical Services (MMTS) and Samuel Engineering, Inc. (SE) estimated the remaining operating costs. All of the operating costs have been completed in collaboration with the management of Copper Fox Metals, Inc. (Copper Fox).

The Schaft Creek operation will be organized in the general manner illustrated in Figure 25.43. Most of the functions will be performed on site. However some financial and administrative support will be provided from locations off site.

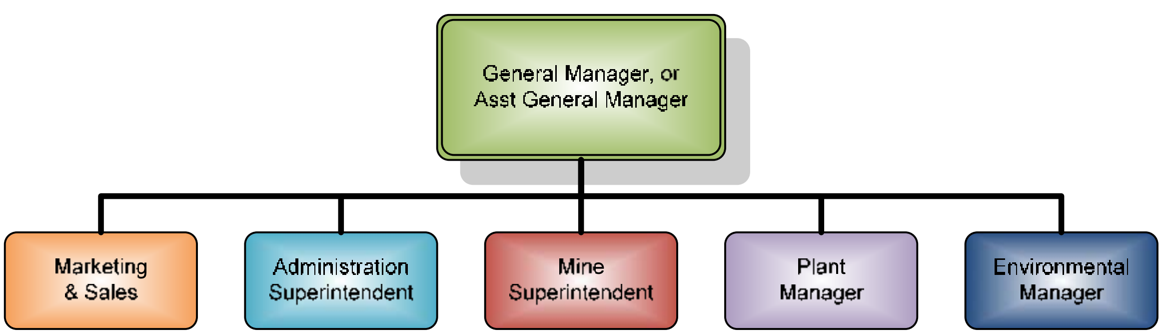


Figure 25.43 Schaft Creek General Organizational Structure

25.9.1.1 *Basis of Operating Cost Estimate*

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 experience with similar functions in British Columbia mines. An average burden rate of 50% has been applied to base salaries to include all statutory Canadian and British Columbia social insurance, medical and insurance costs, pension and vacation costs;
- For hourly employees, general labour rates expected in British Columbia mines were used for the mining positions. For the processing positions the costs are based on information taken from the April 2008 update of the *Mining Cost Service*. An average burden rate of 80% has been applied to base salaries to include all statutory Canadian and British Columbia social insurance, medical and insurance costs, pension and vacation costs, as well as the cost of shift differential and overtime;
- The normal operating schedule is 2,160 hr/yr which is based on 12 hr shifts for one half of the 360 d/yr that the mine will be operating.

25.9.1.2 *Operating Cost Overview*

The operating costs are estimated to be $12.49 per tonne of ore processed. They are allocated as shown in Figure 25.44. At 33% of the total operating cost, the mining costs are slightly higher than the processing costs. However, if the costs associated with handling and treating the concentrates are included in the processing costs, the total jumps from 32% to 60% of the total operating costs at $7.42 per tonne of ore.

Table 25.28
Schaft Creek Average Annual Life of Mine Operating Cost Summary

Description	Fixed or Variable	Annual Cost	Cost/t Ore	Cost/t Total Material or Charge/t Concentrate
Mining (from MMTS)				
Drilling	Variable	$7,812,461	$0.217	$0.075
Blasting	Variable	$21,742,224	$0.604	$0.208
Loading	Variable	$14,164,770	$0.393	$0.135
Hauling	Variable	$60,543,823	$1.682	$0.579
Mine Maintenance	Variable	$1,859,903	$0.052	$0.018
Mine Operations Support	Variable	$29,305,187	$0.814	$0.280
Snow Removal	Variable	$1,269,140	$0.035	$0.012
Geotech	Variable	$2,642,872	$0.073	$0.025
Unallocated Labor Cost	Variable	$1,869,136	$0.052	$0.018
Mine Ops Salaried Personnel	Fixed	$1,829,645	$0.051	$0.017
Mine Maintenance Salaried Personnel	Fixed	$2,852,992	$0.079	$0.027
Mine Engineering Salaried Personnel	Fixed	$2,022,324	$0.056	$0.019

Table 25.28
Schaft Creek Average Annual Life of Mine Operating Cost Summary

Description	Fixed or Variable	Annual Cost	Cost/t Ore	Cost/t Total Material or Charge/t Concentrate
Technical Services Salaried Personnel	Fixed	$1,057,021	$0.029	$0.010
Mining Total Costs		**$148,971,499**	**$4.1381**	**$1.424**
Processing				
Processing Salaried Personnel	Fixed	$6,603,000	$0.1834	
Hourly Operations Labor	Fixed	$19,317,761	$0.5366	
Hourly Maintenance Labor	Fixed	$5,461,379	$0.1517	
Plant Electrical Power	Variable	$42,088,428	$1.1691	
Reagents	Variable	$16,313,985	$0.4532	
Grinding Steel	Variable	$42,274,930	$1.1743	
Laboratory Supplies (1% of total process op costs)	Variable	$1,634,416	$0.0454	
Maintenance Supplies (5% of process op costs)	Variable	$6,602,974	$0.1834	
Misc. Ops Supplies (1% of processing op costs)	Variable	$1,402,969	$0.0390	
Processing Total Costs		**$141,699,842**	**$3.9361**	
General and Administration				
G&A Salaried Personnel	Fixed	$4,470,000	$0.1242	
Hourly Labour	Fixed	$1,492,992	$0.0415	
Power	Fixed	$616,480	$0.0171	
Vehicle Operating & Maintenance	Fixed	$115,385	$0.0032	
Access Road & Powerline Maintenance	Fixed	$1,410,000	$0.0392	
Site Avalanche Control	Fixed	$650,000	$0.0181	
Communications	Fixed	$75,000	$0.0021	
Camp Operations	Fixed	$8,627,693	$0.2397	
Fly-in, Fly-out Operations & Airfield Operations	Fixed	$10,000,000	$0.2778	
Safety Supplies / Incentives	Fixed	$300,000	$0.0083	
Offsite Training & Conferences	Fixed	$300,000	$0.0083	
Insurance	Fixed	$1,000,000	$0.0278	
Corporate Services and Travel	Fixed	$350,000	$0.0097	
Environmental	Fixed	$1,000,000	$0.0278	
Security & Medical	Fixed	$400,000	$0.0111	
Professional Membership Costs	Fixed	$30,000	$0.0008	
Community Development	Fixed	$1,000,000	$0.0278	
Land Holding	Fixed	$-	$-	
Consultants	Fixed	$500,000	$0.0139	
Misc. Computer Equipment/Software	Fixed	$250,000	$0.0069	
Misc. Office Supplies	Fixed	$200,000	$0.0056	
Misc. Freight & Couriers	Fixed	$100,000	$0.0028	
Recruiting and Relocation	Fixed	$200,000	$0.0056	
Legal, Permits, Fees	Fixed	$500,000	$0.0139	

Table 25.28 Schaft Creek Average Annual Life of Mine Operating Cost Summary				
Description	**Fixed or Variable**	**Annual Cost**	**Cost/t Ore**	**Cost/t Total Material or Charge/t Concentrate**
G&A Total Costs		**$33,587,550**	**$0.9330**	
Total Operations Costs and Administration		**$324,258,890.99**	**$9.0072**	
Conc Handling & Treatment				
Copper Conc Pipeline to Filter Plant	Variable	$-	$-	
Copper Conc Transport to Receiving Port	Variable	$34,967,031	$0.9713	$123.61
Copper Concentrate Treatment Charges	Variable	$19,801,733	$0.5500	$70.00
Copper Conc Refining Charges	Variable	$14,777,335	$0.4105	$52.24
Gold Refining Charges	Variable	$1,165,189	$0.0324	$4.12
Silver Refining Charges	Variable	$453,511	$0.0126	$1.60
Moly Conc Transportation to Receiving Port	Variable	$2,133,039	$0.0593	$207.86
Moly Conc Roasting & Refining	Variable	$52,260,559	$1.4517	$5,092.68
Concentrate Handling & Treatment Total		**$125,558,396**	**$3.4877**	
TOTAL PROJECT COSTS		**$449,817,287**	**$12.49**	

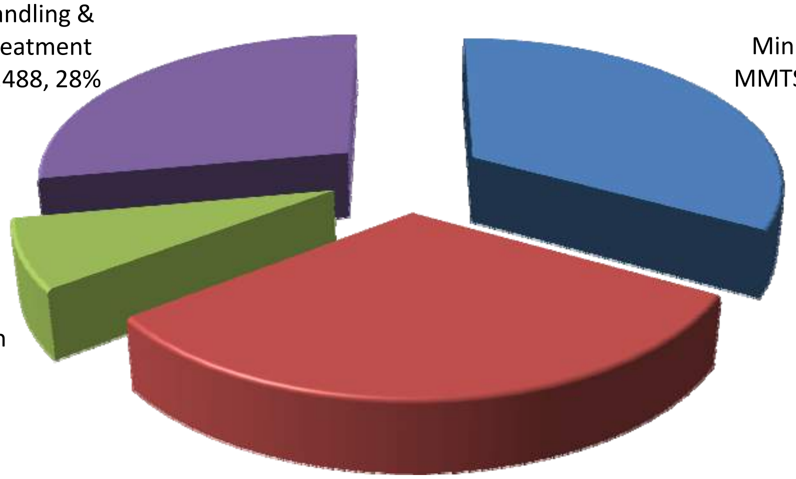


Figure 25.44 Schaft Creek Life of Mine Average Annual Operating Costs

25.9.2 General and Administrative (G&A)

General and administrative (G&A) costs include everything that is not directly attributed to operations. This includes costs of labour to perform administrative functions such as accounting, human resources and safety and security.

G&A is organized into four basic areas: HR/Community Relations, Purchasing, Accounting and Warehousing. In addition to these small departments, the Access Road Maintenance Chief, Safety and Security Manager and contractors' representatives will report to the Administrative Superintendent.

Details of G&A organization are illustrated in Figure 25.45 which shows the breakdown of the Direct Mining, Mine Maintenance and GME functions.

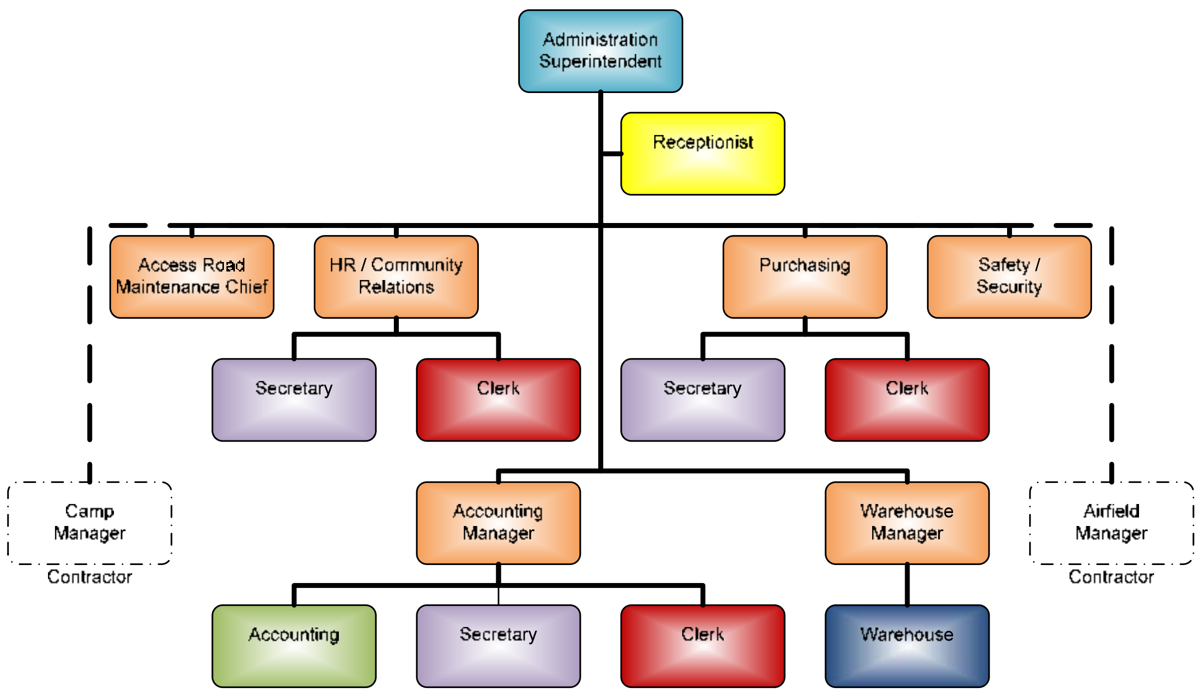


Figure 25.45 Schaft Creek G&A Organizational Structure

Based on the organizational chart shown above, the numbers of personnel required in each position were determined. Since the operating schedule is based on a fly-in, fly-out schedule of two weeks on site and two weeks off site for each position, the number of people required is double the number required for each position in most cases. The General Manager will have total responsibility for all site-related functions including G&A, mining, processing, environmental and marketing and sales activities. Due to the very large size of the operation and the anticipated work schedule, an Assistant General Manager is included in the organizational structure. When the General Manager is off site the Assistant General Manager assumes the overall responsibilities for all of the site activities.

The operating costs allocated to General and Administrative functions are provided in Table 25.29.

Table 25.29 G&A Fixed Operating Costs		
Item	**Average Life of Mine Annual Cost, $**	**Average Life of Mine Annual Cost, $/t**
G&A Salaried Personnel	$4,470,000	$0.1242
Hourly Labour	$1,492,992	$0.0415
Power	$616,480	$0.0171
Vehicle Operating & Maintenance	$115,385	$0.0032
Access Road & Power-line Maintenance	$1,410,000	$0.0392
Site Avalanche Control	$650,000	$0.0181
Communications	$75,000	$0.0021
Camp Operations	$8,627,693	$0.2397
Fly-in, Fly-out Operations & Airfield Operations	$10,000,000	$0.2778
Safety Supplies / Incentives	$300,000	$0.0083
Offsite Training & Conferences	$300,000	$0.0083
Insurance	$1,000,000	$0.0278
Corporate Services and Travel	$350,000	$0.0097
Environmental	$1,000,000	$0.0278
Security & Medical	$400,000	$0.0111
Professional Membership Costs	$30,000	$0.0008
Community Development	$1,000,000	$0.0278
Land Holding	-	-
Consultants	$500,000	$0.0139
Misc. Computer Equipment/Software	$250,000	$0.0069
Misc. Office Supplies	$200,000	$0.0056
Misc. Freight & Couriers	$100,000	$0.0028
Recruiting and Relocation	$200,000	$0.0056
Legal, Permits, Fees	$500,000	$0.0139
G&A Total	**$33,587,550**	**$0.9330**

25.9.3 Mining

The mining operating costs were developed by Moose Mountain Technical Services (MMTS). Since the mining operating costs are variable, the life of mine average annual mining operating costs for the Schaft Creek project have been estimated in 3rd quarter 2008 Canadian dollars. The estimate does not include allowances for escalation or exchange rate fluctuations.

25.9.3.1 *Basis of Mining Operating Cost Estimate*

Unit costs for consumables 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 of salaried employees are based on MMTS experience of similar functions in British Columbia mines. An average burden rate of 50% has been applied to base salaries to include all statutory Canadian and British Columbia social insurance, medical and insurance costs, pension and vacation costs;
- For hourly employees, general labour rates expected in British Columbia mines were used. An average burden rate of 80% has been applied to base salaries to include all statutory Canadian and British Columbia social insurance, medical and insurance costs, pension and vacation costs, as well as costs for shift differential and overtime;
- Mine designs to determine the size and makeup of the mine fleet as well as fuel requirements which are affected by distance from the pit to the various destinations and topography;
- Budgetary quotations, including freight for all consumables, tires and fuel. The long-term fuel price is estimated at a delivered cost to site of $1.00 per litre;
- Power costs were estimated as the sum of consumption charges and demand charges and estimated at $0.0469/kWh;
- Mining equipment consumables, major equipment replacements, sustaining capital, labour-loading factors, equipment life and costs are based vendor information and MMTS's experience in similar mining operations.

25.9.3.2 *Mine Operating Costs*

Mine operating costs are derived from a combination of supplier quotes and historical data collected by MMTS. This includes the labour, maintenance, major component repairs, fuel and consumables costs.

The fleet hourly operating costs are used as a constant basis over the schedule periods.

From 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 Caterpillar's Fleet Production and Cost Analysis (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.

Blasting costs were based on studies from similar projects and historical blasting costs.

Geotechnical costs are based on historical data collected by MMTS.

The mine operating costs have been broken down into three components in the production and cost schedules. These are: salaried-employee labour costs, hourly employee labour costs and equipment operating costs.

A tabular and graphical summary of the mine operating costs is provided in Table 25.30 and Figure 25.46 respectively. By far the largest mine operating cost is the cost for hauling which totals 41% of the total mine operating costs.

Table 25.30 Mining Operating Costs			
	Life of Mine Average Annual Cost	**Life of Mine Average Cost per t Material Moved**	**Life of Mine Average Cost per t Ore**
Drilling	$7,812,461	$0.075	$0.217
Blasting	$21,742,224	$0.208	$0.604
Loading	$14,164,770	$0.135	$0.396
Hauling	$60,543,823	$0.579	$1.869
Mine Maintenance	$1,859,903	$0.018	$0.054
Mine Operations - Support	$29,305,187	$0.280	$0.895
Snow Removal	$1,269,140	$0.012	$0.039
Geotech	$2,642,872	$0.025	$0.073
Unallocated Labour Cost	$1,869,136	$0.018	$0.052
Direct Cost Subtotals	*$141,209,516*	*$1.350*	*$3.922*
Mine Ops Salaried Labour	$1,829,645	$0.017	$0.051
Mine Maintenance Salaried Labour	$2,852,992	$0.027	$0.079
Mine Engineering Salaried Labour	$2,022,324	$0.019	$0.056
Technical Services Salaried Labour	$1,057,021	$0.010	$0.029
Total General Mine Expense Costs	$7,761,983	$0.074	$0.216
Total Estimated Mine Op Costs	**$148,971,499**	**$1.424**	**$4.138**

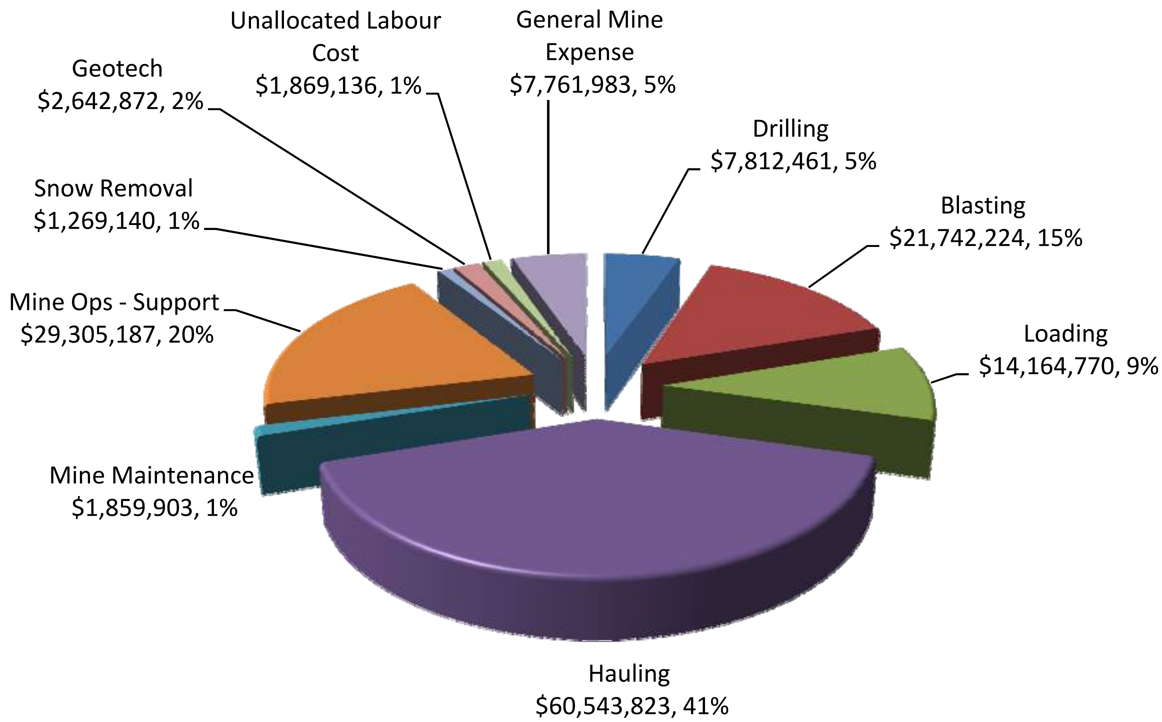


Figure 25.46 Mine Operating Costs

25.9.4 Processing

Process operating costs are derived from a combination of supplier quotes and historical data collected by Samuel Engineering, Inc. (SE). Mill labour costs are based on costs for mines in British Columbia taken from the Mining Cost Service, April 2008 update.

Table 25.31 and Figure 25.47 provide a tabular and graphical representation of the process operating costs respectively. The cost of grinding steel and power, together consume 60% of the process operating costs followed by the aggregated labour costs at 22% and reagents at 11%. Various supplies consume the remaining 7% of operating costs.

Table 25.31
Plant Operating Cost Summary

Cost Category	Cost Annual US$	US$/t Milled
Labour		
Salaried Personnel	$6,603,000	$0.183
Hourly Operations Personnel	$19,317,761	$0.537
Hourly Maintenance Personnel	$5,461,379	$0.152
Total Labour	*$31,382,140*	*$0.872*
Consumables		
Power	$42,088,428	$1.169
Reagents	$16,313,985	$0.462

Table 25.31 Plant Operating Cost Summary		
Cost Category	**Cost Annual US$**	**US$/t Milled**
Grinding Steel	$42,274,930	$1.174
Laboratory Supplies	$1,634,416	$0.045
Maintenance Supplies	$6,602,974	$0.184
Miscellaneous Operating Supplies	$1,402,969	$0.039
Total Consumables	*$110,317,702*	*$3.064*
Total Estimated Plant Operating Cost	**$141,699,842**	**$3.945**

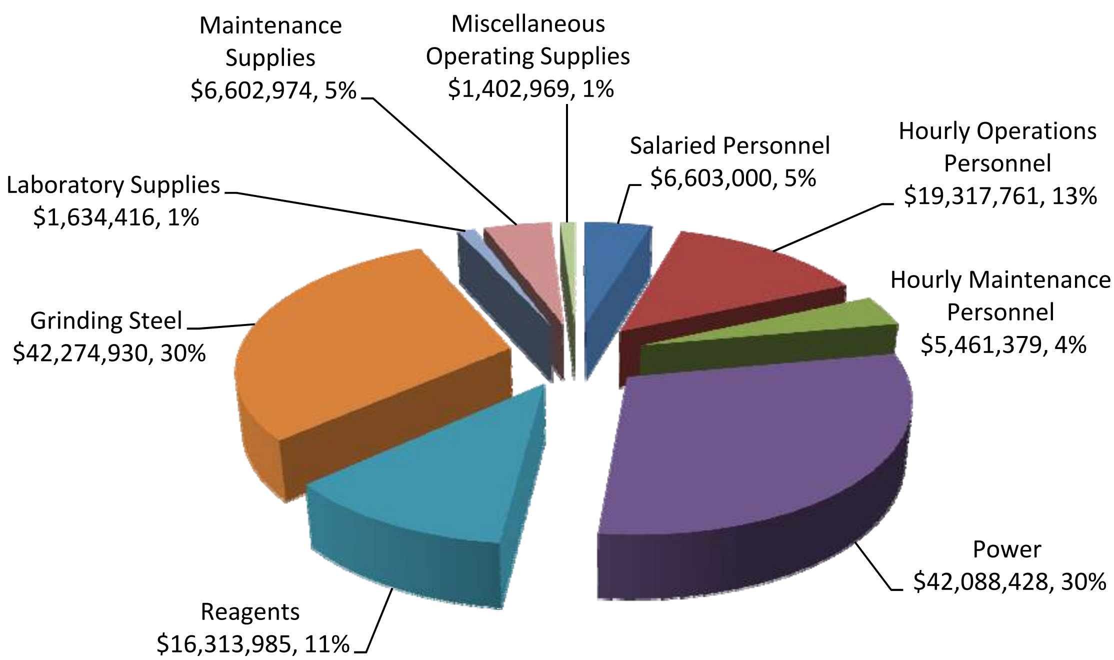


Figure 25.47 Process Operating Costs

25.9.4.1 *Basis of Process Operating Costs*

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, metallurgical testing and first principles.

The unit costs are based on the following data:

- Salaries for each salaried job category general labour rates expected in British Columbia mines are used. An average burden rate of 50% has been applied to base salaries to include all statutory Canadian and British Columbia, social insurance, medical and insurance costs, pension and vacation cost. For hourly employees, the rates that were used were taken from the April 2008 update of the *Mining Cost Service.* An average burden rate of 80% has been applied to base salaries to include all statutory Canadian and British Columbia, social insurance, medical and insurance costs, pension and vacation costs, as well as the cost of shift differential and overtime;

- Budgetary quotations, including freight for reagents, were solicited for consumables, grinding balls, steel liners;
- Power costs were estimated using BC Hydro rates for demand charges, energy consumption charges, a rate rider and taxes. The estimated cost is $0.0469/kWh;
- Allocations have been made for analytical and metallurgical laboratory supplies, maintenance supplies and general operating supplies based on experience at similar operations.

25.9.5 Concentrate Handling and Treatment

The cost of shipping and processing concentrates for this project amounts to approximately 28% of the total operating costs. Table 25.32 shows the various categories that have been estimated and the costs associated with each of them. The costs are based on the average annual amounts of concentrates that will be produced. This is 282,882 t copper concentrate and 10,262 t molybdenum concentrate.

Table 25.32
Copper and Molybdenum Concentrate Handling and Treatment Charges

Concentrate Handling & Treatment	Annual Cost	Cost/t Ore	Charge/t Concentrate
Copper Conc Transport to Receiving Port	$34,967,031	$0.9713	$123.61
Copper Concentrate Treatment Charges	$19,801,733	$0.5500	$70.00
Copper Conc Refining Charges	$14,777,335	$0.4105	$52.24
Gold Refining Charges	$1,165,189	$0.0324	$4.12
Silver Refining Charges	$453,511	$0.0126	$1.60
Moly Conc Transport to Receiving Port	$2,133,039	$0.0593	$207.86
Moly Concentrate Roasting & Refining	$52,260,559	$1.4517	$5,092.68
Concentrate Handling & Treatment Total	**$125,558,396**	**$3.4877**	

Molybdenum concentrate roasting and refining charges are taken as a price discount. Molybdenum payable is 100 percent of the contained moly in the concentrate at quoted price less a discount. The discount for this economic analysis is 14% of the price subject to the following:

- Maximum of $5.50;
- Minimum of $2.50 per pound of contained moly.

The moly discount (roasting and refining charges) were calculated to be $4.62/lb Mo or $1.45/t ore, based on a Moly price of $33.00/lb.

25.10 Capital cost Estimate

25.10.1 Introduction

The capital cost estimate for the Schaft Creek Project Preliminary Feasibility Study addresses a base case plant capable of processing 100,000 metric tonnes per day (mtpd) of copper bearing material.

The estimate has been prepared to support the economic evaluation and assessment of the project as well as to assist with raising funds for further project development.

The key objectives of the overall study are to:

- Assess the economic evaluation of the project;
- Support the identification and assessment of the processes and facilities that will provide the most favourable return on investment;
- Provide guidance and direction for the next phase of more detailed studies;
- Assist with raising funds for further project development.

The estimate for process facilities has been prepared by Samuel Engineering (SE). The following basis of estimate (BOE) is an itemization of the methodology, assumptions, rates and criteria utilized in the preparation of the estimate.

The total estimated cost to design, procure, and construct the facilities described in this section is $2,950,406,000. Table 25.33 summarizes the capital costs by major area.

Table 25.33 Capital Cost Summary	
Mine Facilities	$50.4
Primary Crushing & Coarse Ore Conveying	$85.8
Coarse Ore Reclaim	$125.9
Grinding & Pebble Crushing	$317.5
Bulk Flotation & Regrind	$133.0
Copper-Molybdenum Separation	$6.0
Copper Concentrate	$14.5
Molybdenum Concentrate	$4.1
Tailings Handling	$35.2
Tailings Pond & Water Reclaim System	$17.3
Tailings Impoundment	$173.3
Reagents	$14.6
Plant Services	$33.8
Buildings & Ancillary Facilities	$39.9
Site Development	$264.2
Contracted Directs	***$1,315.5***
Common Distributables	$399.1
EPCM	$211.0
Contracted Indirects	***$610.1***
Owner's Cost	***$459.8***
Subtotal	***$2,385.3***
PST Taxes	$28.6
Contingency	$536.5
Total	**$2,950.4**

25.10.2 Currency

The estimate is expressed in second-quarter 2008 United States dollars. No provision has been included to offset future escalation.

Where source information was provided in Canadian currency, these amounts have been converted at a rate of $1.00 CAD = $1.00 USD.

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 world currencies could also influence future project cost if equipment is purchased in Europe or elsewhere. No funds have been allocated in the estimate to offset potential currency fluctuations.

25.10.3 Scope

The estimate addresses the engineering, procurement and construction of a greenfield 100,000 mtpd copper, molybdenum, gold and silver concentrator located in the Liard Mining Division on the eastern edge of the Coastal Mountain Range in north central British Columbia. The deposit is located 1040 km north of Vancouver, 63 km northwest of Bob Quinn Camp, off the Stewart-Cassier highway at an altitude of approximately 1200 metres above sea level (masl).

The plant is situated at approximately 1230 masl and will include an SABC comminution circuit followed by a bulk flotation circuit, a molybdenum separation circuit and a copper circuit. The copper circuit has a thickener, filters and a concentrate stockpile. The molybdenum circuit includes filtration, drying and bulk bagging equipment. Tailings thickeners, tailings storage facility and water reclaim are part of the tailings management system. The processing circuit will have a design capacity of 108,700 tonnes per day (tpd) and a nominal capacity of 100,000 tpd.

The scope of facilities addressed by the estimate are those relating to the development of the mine, concentrator plant facilities, on-site ancillary facilities, off-site infrastructure (access roads and power lines), tailings impoundment and owner's cost.

The estimate is based on the scope of work as outlined in the facilities description and Work Breakdown Structure (WBS), and as defined by the following:

- Process design criteria;
- Process flow diagrams;
- Mechanical equipment list;
- Electrical single-line diagrams;
- Plot plans and general arrangement (GA) drawings;
- Electrical equipment list;
- Civil, structural, mechanical, and electrical design criteria;
- Control philosophy;
- Quotations from vendors;
- Material takeoffs (MTOs) generated from GA models and layouts;
- In-house historical data and database information, and;
- Geotechnical reports GS-TB-007119 and GS-TB-007675, from DST Consulting Engineers, Inc. covering Tailings Dam Options and Potential Rock Disposal Sites respectively.

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 Costs	by Moose Mountain Technical Services
Access Road	by McElhanney
Transmission Line	by Copper Fox
Tailings Facility	by Knight Piésold
Process Facility	by Samuel Engineering
Airfield	by Copper Fox
Construction Indirect Cost	by Samuel Engineering and Copper Fox
Owner's Cost	by Samuel Engineering and Copper Fox
Contingency	by Samuel Engineering and Copper Fox

25.10.4 Estimate Exclusions

Items not included in the capital estimate are as follows:

- Sunk Costs;
- Environmental Studies;
- Sustaining Capital;
- Working Capital;
- Reclamation Costs (Included in Financial Analysis);
- Escalation Beyond Second Quarter 2008;
- Foreign Currency Exchange Rate Fluctuations;
- Interest;
- Financing Cost;
- Goods and Services Tax (GST), and;
- Risk due to political upheaval, government policy changes, labour disputes, permitting delays, weather delays or any other force majeure occurrences.

25.10.5 Accuracy

The capital cost estimate for the Schaft Creek Project has been developed to a level sufficient to assess/evaluate the project concept and its overall viability. The estimate has an intended level of accuracy of minus 25% plus 25%, after the inclusion of the recommended contingency.

The various cost components have been reviewed with regard to the forces that may influence their costs. An assessment was made as to their variability and contingency applied based on the assessment. We believe the accuracy objective has been met.

25.10.6 Quantities and Pricing Methodology

The estimate is built up by cost centers as defined by the project Work Breakdown Structure (WBS) and by prime commodity accounts, which include earthwork, concrete, structural steel, mechanical equipment (including platework), piping, electrical and instrumentation.

Pricing was derived through various sources including vendor budgetary quotations, contractor feedback, in-house historical data, published databases (Richardson's, RS

Means, Page & Nation, etc.), factors from similar projects, and estimators' judgments. The application of these methodologies within each commodity group is described below.

Due to the limited scope of engineering at this stage, complete material quantity takeoffs are not possible. Therefore, costs for ancillary mechanical equipment, minor electrical equipment and most bulk materials for facilities or portions of facilities, for which MTOs are unavailable, have been allocated using factors.

The capital cost estimate is based on the assumption that equipment and materials will be purchased on a competitive basis and that installation contracts will be awarded in defined packages also on a competitive basis.

25.11 Economic Analysis

25.11.1 Economic Analysis

The economic analysis for the Schaft Creek Project are reported in Q2 2008 US dollars and do not include allowances for escalation or foreign currency exchange fluctuations.

Where source information was provided in other currencies, these amounts have been converted at rates of US$1.00 = CAN$1.00.

Below are the key assumptions and inputs used to generate the model results.

Table 25.34 Important Financial Model Assumptions and Inputs	
Currency Exchange Rate	US$1.00 = CAN$1.00
Initial capital requirements	$2,950,406,000
Working capital	$146,420,000
Sustaining capital	$797,379,000
Reclamation & Closure	$87,000,000
Average mining rate	190,550 tpd = 68,597,000 tpy
Average mining cost	$4.14/tonne ore or $1.42/tonne total mat'l
Ore processing rate	100,000 tpd = 36,000,000 tpy
Average processing cost	$3.94/tonne ore
Annual G&A expenses	$33,587,550 ($0.93/tonne ore)
Average LOM Cu grade	0.301%
Cu recovery	88.4%
Cu concentrate grade	33.9%
Cu conc transport, handling and ocean freight	$123.61/tonne conc
Cu treatment charges	$70/tonne conc
Cu refining charges	$0.07/lb Cu
Average LOM Mo grade	0.02%
Mo recovery	71.3%
Mo concentrate grade	50%
Mo conc transport, handling and ocean freight	$207.86/tonne conc
Mo roasting & refining	$4.62/lb Mo
Average LOM Au grade	0.212 g/t
Au recovery	81.3%
Au grade in copper concentrate	21.9 g/t
Au refining	$6.00/oz
Average LOM Ag grade	1.76 g/t
Ag recovery	70.1%
Ag grade in copper concentrate	158.3 g/t
Ag refining	$0.35/oz
Total LOM Cu production	4,762,524,025 lbs
Total LOM Mo production	255,194,418 lbs
Total LOM Au production	4,493,445 oz
Total LOM Ag production	32,480,015 oz

25.11.2 Taxes

Financial results reported herein include British Columbia and Canadian taxation but do not include any royalty payments generated from the Teck Cominco Option Agreements. 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.

The following assumptions were used in determining the taxation calculations for the modeling:

- The model does not have a tax loss carryback or forward mechanism as all years are income;
- Cash taxes are payable when incurred (should usually be a bit of a lag);
- Investment tax credits are not considered;
- Benefits for tax pools that Copper Fox may currently have are not considered;
- Interest expense have not been considered, while taxes on cash flow from operations are;
- Current enacted tax rates are used for taxation calculations, even though British Columbia may reduce future income tax rates from 11% to 10%;
- Flow-through shares will be issued to fund the pre-stripping;
- All G&A costs are mine site related and not head office related;
- Hedging is not included in the assumed price received for the metals as hedging would change the amount of BC Mineral tax.

Life of mine total project related taxes were calculated to be $4,041,652,000.

25.11.3 Capital Costs

25.11.3.1 *Initial Capex*

The initial capital costs for this project were determined by Samuel Engineering to be $2,950,406,000. For the economic analysis, the initial capex was spread over a three year period, the construction period, prior to concentrate production. The spread for the three year period are as follows:

Year -3	**Year -2**	**Year -1**
$295,041,000	$1,180,162,000	$1,475,203,000
10%	40%	50%

25.11.3.2 *Working Capex*

Working capital costs were taken as four months of the annual operating expense recognized in Year -1 for the economic analysis.

25.11.3.3 *Sustaining Capex*

Moose Mountain Technical Services provided a complete spread of replacement capital for the mining fleet for the duration of the mine life.

Samuel Engineering allowed for an annual sustaining cost of $10,000,000 for the milling facilities.

Knight Pièsold provided a complete spread of sustaining cost for the tailings facility for the duration of the mine life.

The reclamation and closure cost were estimated from a combination of mining, tailings, mill and site wide considerations.

Below is a summary of the total project related sustaining capex:

Mine	$232,893,000
Mill	$220,000,000
Tailings	$257,486,000
Reclamation & Closure	$87,000,000
Total	**$797,379,000**

25.11.4 Base Case Economics

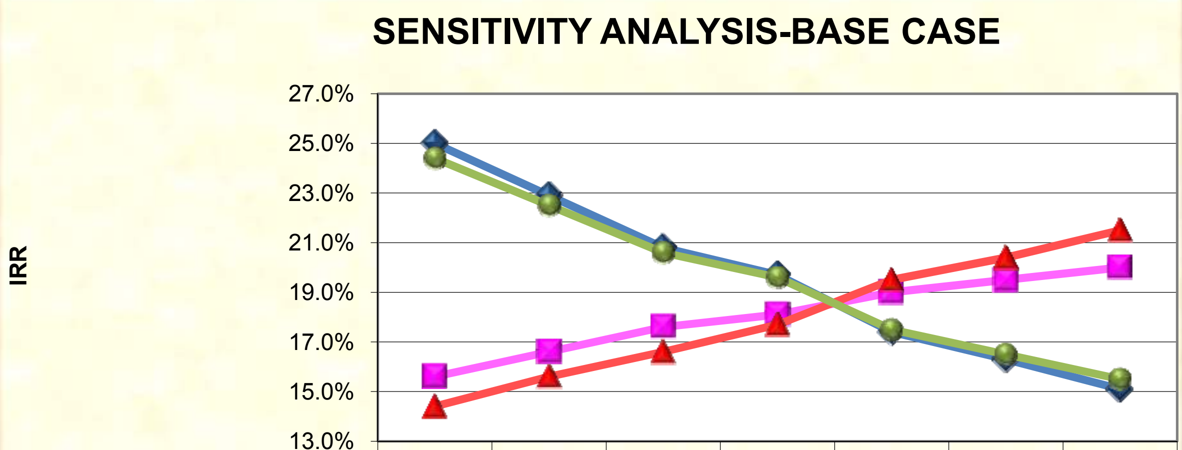
A financial model was created utilizing the mine production schedule, the associated metal grades based on the geological resource estimate, metal recoveries from the ongoing metallurgical test program, capital and operating costs as set out herein and base case metal prices, trailing three year average from August 29, 2008, of copper US$3.12/lb, molybdenum US$33.00/lb, gold US$692.85/oz and silver US$13.09/oz.

Modeling at base case metal prices shows that the project could generate a cumulative before tax profit of C$11,735 million, with a payback period of 4.7 years, a 18.6% IRR, and a net present value discounted at 10% of C$1,868 million, over the 23 year mine life.

Modeling at base case metal prices shows that the project could generate a cumulative after tax profit of C$7,693 million, with a payback period of 4.9 years, a 15.3% IRR, and a net present value discounted at 10% of C$980 million, over the 23 year mine life.

25.11.5 Base Case Project Sensitivity Analysis

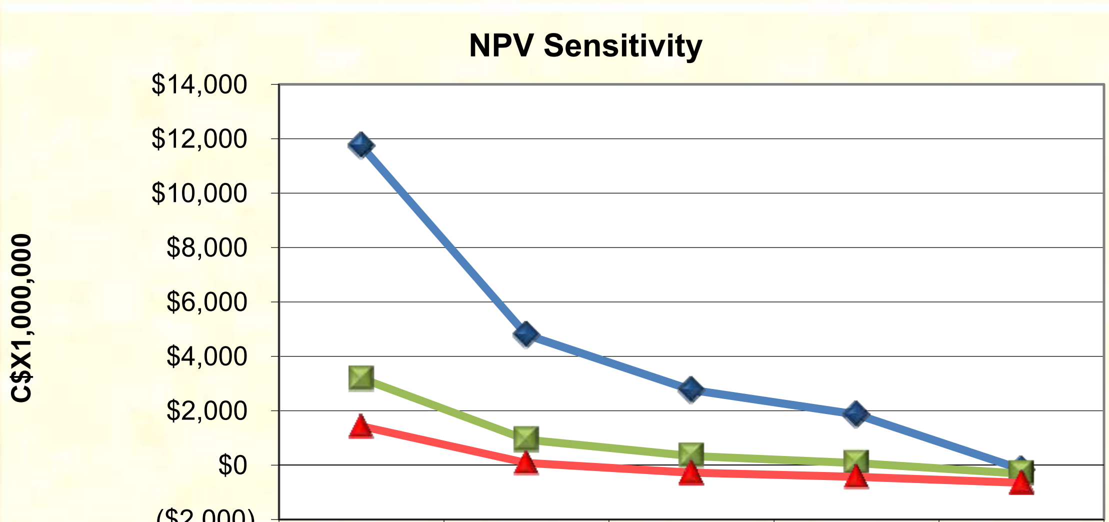
Sensitivity calculations were performed on the project cash flow by applying factors ranging from -15% to +30% against metal prices, initial capital costs, annual operating costs and copper grade. The effects on IRR and NPV are shown graphically in Figure 25.48 and Figure 25.49, respectively. The project is moderately sensitive to changes in capital and operating costs and more sensitive to changes in metal prices and copper grade.



	30.0%	20.0%	10.0%	5.0%	-5.0%	-10.0%	-15.0%
Metal Prices	25.0%	22.9%	20.8%	19.7%	17.4%	16.3%	15.1%
Operating Costs	15.6%	16.6%	17.6%	18.1%	19.0%	19.5%	20.0%
Capital Costs	14.4%	15.6%	16.6%	17.7%	19.5%	20.4%	21.5%
Copper Grade or Recovery	24.4%	22.5%	20.6%	19.6%	17.5%	16.5%	15.5%

% FROM BASE CASE

Figure 25.48 Base Case IRR Sensitivity Analysis



	0%	5%	8%	10%	15%
Base Case	$11,735	$4,788	$2,764	$1,868	($150)
Revenue +10%	$3,193	$940	$338	$78	($308)
Op Cost +10%	$1,434	$84	($273)	($425)	($646)

Discount Factor

Figure 25.49 Base Case NPV Sensitivity Analysis

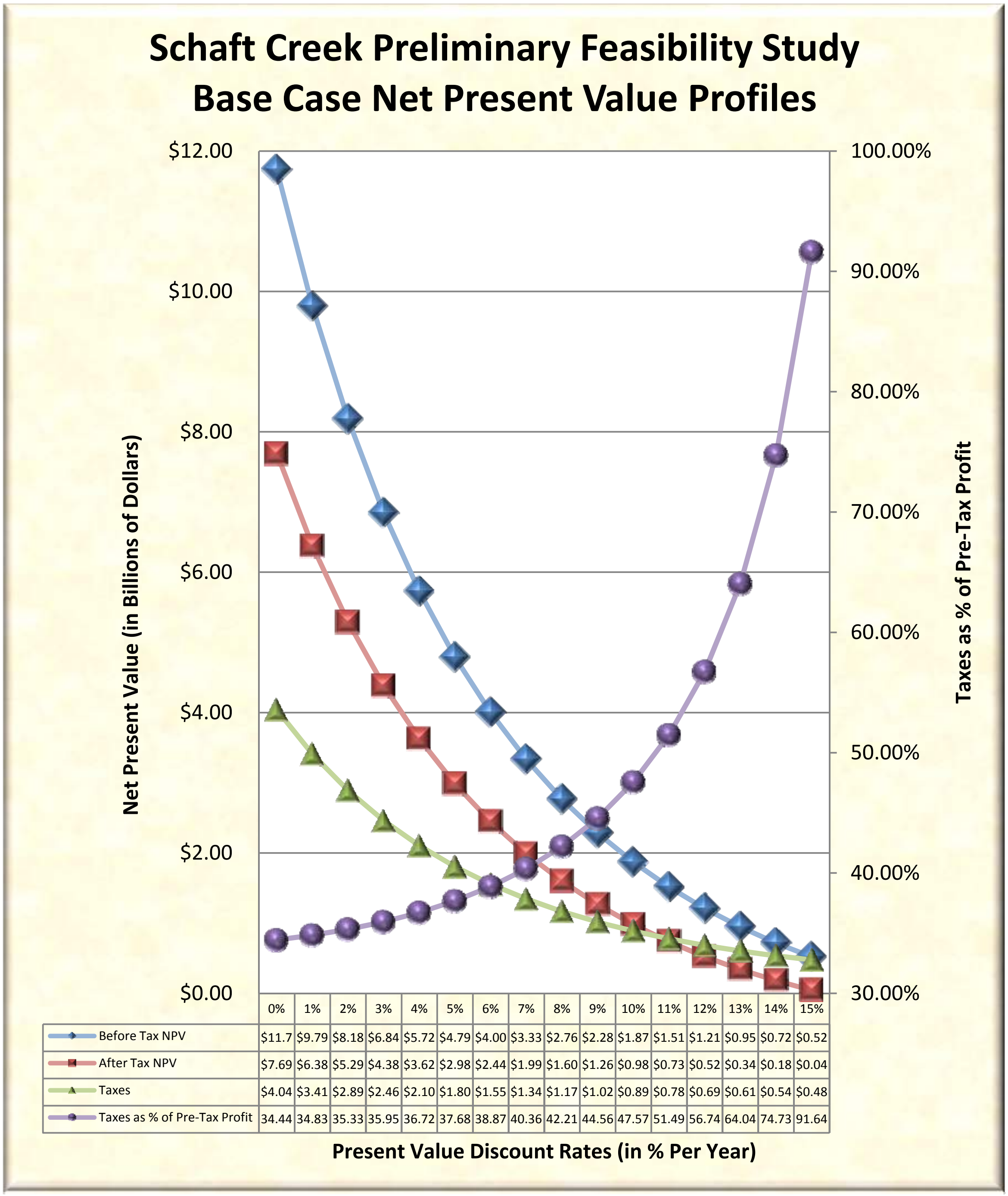
25.11.6 Project Economics Summary

Table 25.35 below is a summary of the economic results for the Schaft Creek project.

Table 25.35 Key Project Parameters and Results				
September 9, 2008, Rev. 5a				
Total Resource (M&I)	tonnes	1,393,282,171	@ 0.25% Cu	
Total Reserve	tonnes	816,706,750		
LOM Mill Feed	tonnes	812,230,421		
LOM Waste	tonnes	1,543,190,551		
LOM Strip Ratio		1.88		
Daily Feedrate	tpd	100,000		
Mine Life	yrs	22.6		
Connected Load	MW	146.8		
Avg Power Demand	MW	121.4		
Power Cost	$/kWh	0.0469		
Foreign Exchange Rate		US$1 = C$1		
Total Initial Capex	$ (000's)	**2,950,406**		
Directs	$ (000's)	1,315,484		
Indirects	$ (000's)	610,108		
Owner	$ (000's)	459,757		
Taxes	$ (000's)	28,566		
Contingency	$ (000's)	536,490		
Working Capex	$ (000's)	146,420		
Total Sustaining Capex	$ (000's)	797,379		
Mine	$ (000's)	232,893		
Mill	$ (000's)	220,000		
Tailings	$ (000's)	257,486		
Reclamation & Closure	$ (000's)	87,000		
Total LOM Opex	$ (000's)	**10,138,610**		
Total LOM Opex	$/t ore	**12.49**		
Mining	$/t ore	4.14		
Processing	$/t ore	3.94		
G&A	$/t ore	0.93		
Conc Handling & Treatment	$/t ore	3.49		
Contingency (0%)	$/t ore	0.00		
Total LOM Taxes	$ (000's)	4,041,652		
Metrics		**Head Grades**	**Conc Grades**	**Recoveries**
Cu	%	0.301%	33.85%	88.4%
Mo	%	0.020%	50.0%	71.3%
Au	g/t	0.212	21.90	81.3%
Ag	g/t	1.761	158.30	70.7%
Base Case Pricing (Trailing 3 Year Avg - August 29, 2008)				
Cu	$/lb	3.12		
Mo	$/lb	33.00		
Au	$/oz	692.85		

Table 25.35 Key Project Parameters and Results				
Ag	$/oz	13.09		
Cashflow Results		**Before Tax**	**After Tax**	**Direct Tax Effects**
IRR	%	18.6%	15.3%	-3.21%
NPV @ 0%	$ (000's)	$11,734,537	$7,692,885	($4,041,652)
NPV @ 5%	$ (000's)	$4,787,931	$2,983,847	($1,804,084)
NPV @ 8%	$ (000's)	$2,764,475	$1,597,500	($1,166,974)
NPV @ 10%	$ (000's)	$1,868,441	$979,686	($888,755)
NPV @ 12%	$ (000's)	$1,208,843	$522,898	($685,945)
NPV @ 15%	$ (000's)	($149,721)	($422,853)	($273,132)
Payback Period	yrs	4.7	4.9	
Rock Value *	$/t ore	$31.47		
LOM Recoverable Revenue	$ (000's)	25,559,408		
Cu	%	54.4%		
Mo	%	32.2%		
Au	%	12.0%		
Ag	%	1.4%		
Total Metal Production		**LOM**	**Annual**	**Annual Tonnes**
Cu	lbs	4,762,524,025	211,104,788	95,757
Mo	lbs	255,194,418	11,311,809	5,131
Au	ozs	4,493,445	199,178	
Ag	ozs	32,480,015	1,439,717	
CFM Portion of Metal Production		LOM	Annual	Tonnes
Cu	lbs	1,112,049,360	49,292,968	22,359
Mo	lbs	59,587,897	2,641,307	1,198
Au	ozs	1,049,219	46,508	
Ag	ozs	7,584,084	336,174	
Facilities Startup		4 QTR 2013		

Below is a graph showing the NPV results and taxation plotted against the corresponding discount rates. The graph suggests that from a strictly optimum PV profit point of view, there may be a higher throughput rate that could increase the PV profitability for the project (note the sharp drop in NPVs).



	0%	1%	2%	3%	4%	5%	6%	7%	8%	9%	10%	11%	12%	13%	14%	15%
Before Tax NPV	$11.7	$9.79	$8.18	$6.84	$5.72	$4.79	$4.00	$3.33	$2.76	$2.28	$1.87	$1.51	$1.21	$0.95	$0.72	$0.52
After Tax NPV	$7.69	$6.38	$5.29	$4.38	$3.62	$2.98	$2.44	$1.99	$1.60	$1.26	$0.98	$0.73	$0.52	$0.34	$0.18	$0.04
Taxes	$4.04	$3.41	$2.89	$2.46	$2.10	$1.80	$1.55	$1.34	$1.17	$1.02	$0.89	$0.78	$0.69	$0.61	$0.54	$0.48
Taxes as % of Pre-Tax Profit	34.44	34.83	35.33	35.95	36.72	37.68	38.87	40.36	42.21	44.56	47.57	51.49	56.74	64.04	74.73	91.64

25.11.7 Marketing Study

Copper Fox retained H.M.Hamilton & Associates Inc. to perform a marketing study for copper, molybdenum, gold and silver. The basis of this study was used for the copper and molybdenum concentrate handling and treatment charges in the financial analysis. Following is a summary of that report.

25.11.7.1 *Transportation*

Copper Concentrates

The majority (if not all) of the Copper concentrates will be delivered to Asian ports, most likely to Japan, South Korea or northern China. The export port will most probably be Stewart B.C. 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 kilometers. The indicated trucking rate from the property to the port is $0.1515 Cdn per kilometer per swt.

(This does not include the fuel surcharge). Assuming an 8% moisture content and a Canadian dollar at par, this translates to $50.46 U.S. per dmt. Storage and loading at Stewart has been offered by Stewart Bulk Terminals at $12.00 Cdn per wmt. This is line with the charges being paid by others.

A Vancouver-based shipping broker estimates for long-term rates for shipment of concentrates from Stewart to Asian ports (in US dollars) to be:

To Japanese ports.. $50.00 per wmt

To South Korean ports .... $51.00 per wmt

To North China ports .... $53.00 per wmt

These rates assume a vessel loading of 10,000 wmt each.

The totals of these projected movements are shown in Table 25.32, in $US/ dry metric ton:

Table 25.36
Copper Concentrate Handling Charges From Stewart, BC to Various Asian Ports

	To Japan	To South Korea	To North China
Truck	$50.46	$50.46	$50.46
Storage and Loading	$13.04	$13.04	$13.04
Ocean Freight	$54.34	$55.43	$57.61
Totals	**$107.84**	**$118.87**	**$121.11**

As an alternative, some of the copper concentrates could be delivered to eastern Canadian smeltesr (Flin Flon Manitoba and The Horne in Quebec). For these deliveries the concentrates would need to be trucked to the load-out facility at Kitwanga B.C. and then loaded onto CNRail.

This facility is very basic and would need to be upgraded if any significant tonnage movement were envisaged. In addition, the availability of additional CN covered gondola railcars is suspect.

Molybdenum Concentrates

For the Molybdenum concentrates, the transportation is assumed to be through Kitwanga B.C.and Montreal to Antwerp using 1.8 metric ton bags in containers

Truck to Kitwanga.. $50.00

Reloading & Rail to Montreal.. $90.00

Ocean Freight to Antwerp $44.00

Total CIF Antwerp .... $184.00 (Excluding the cost of containerization)

25.11.7.2 *Pricing*

HMHamilton & Associates recommends the pricing reflected in Table 25.33:

Table 25.37 Metal Pricing Forecasts				
Years	Molybdenum ($/lb)	Copper ($/lb)	Gold ($/oz)	Silver ($/oz)
1 - 2	$22.38	$3.00	$700.00	$14.00
3 – 4	$20.00	$2.80	$675.00	$12.00
5 – 10	$15.00	$2.00	$650.00	$11.00
11 - 20	$12.00	$1.50	$650.00	$10.00
Average	**$14.74**	**$1.93**	**$657.50**	**$10.90**

Quantities

Production is forecast to commence in 2013. The annual production rates are expected to be in the order of 334,000 wet metric tons (wmt) of Copper concentrates plus 9,300 wmt of Molybdenum concentrates.

Marketability

Copper Concentrates

Generally the Shaft Creek Copper concentrates would be readily marketed. The annual volume would dictate the placement into four to five different smelters. (60,000 – 100,000 metric tons to each location).

If the copper grade of the concentrates was maintained at or above 30%, the concentrates would command a premium position in the market place since it would allow the receiving smelters to use the Shaft Creek concentrates as a diluent and thereby acquire additional lower grade materials.

Details:

- The higher copper content would allow the smelters to blend with lower grade concentrates.
- If the gold content proves to be in the order of 0.5 opt this would be to the liking of the Japanese smelters.
- Placing the CFM concentrates into China would be more difficult since the majority of Chinese copper smelters do not have high recoveries of the precious metals. This situation is gradually being remedied and by 2009 some of these smelters may have improved their precious metal recoveries
- At present, while India is looking for copper units, the majority of their smelters do not want high precious metal concentrates.
- Based on the concentrate assays presented, 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.

Molybdenum Concentrates

The recent rapid growth in demand for molybdenum has led to a world wide shortage of roasting capacity. The current roaster shortage may last for 2-4 more years and more capacity will be required as the demand expands.

The quality of the concentrate produced may affect its marketability and the terms of sale. In particular, the level of copper impurity in the concentrate will be important and efforts should be made to keep it below 0.5%. Below 0.2% would be even more desirable. Unless special efforts are made to sharply reduce the level of copper in the molybdenite concentrate to be produced, it may be present at levels which, after conversion to oxide, are unacceptable for most uses. After roasting it may constitute 1-2% in the oxide concentrate.

For the Molybdenum concentrates, indications are that there would not be a specific payment for the contained Rhenium. However, in a tight market for available roasting space, the Rhenium content may dictate that the Schaft creek concentrates would have preference.

Deliveries are assumed to be to Europe (Antwerp) via Kitwanga B.C .and Montreal using bags.

25.11.7.3 *Smelter Terms*

Overview

While the present copper concentrate market is favoring the mines with low treatment and refining charges and no price participation, over the longer term these charges will return to more normal levels as the supply/demand position equalizes. With the long term forecast price for copper in the $1.50/lb range we can expect the price participation to return but at a new trigger level approximately equal to this long term forecast price.

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.

With possible copper smelter outlets in Japan, Korea, China, Eastern Canada and Europe, the terms for Japan have been used as a base case scenario.

Copper Concentrate Terms

Table 25.38
Processing Smelter Copper Concentrate Terms

Component Metal	Grade	Payable Content	Deductions	Refining Charge(1)	Treatment Charge	Penalties
Copper	< 30%	96.5%	Minimum 1.0%	US $0.09 per payable lb	US $90 per dmt of Concentrate	Arsenic 0.10 % free US $ 3.00 for each 0.1 % thereafter Antimony 0.10 % free US $ 3.00 for each 0.1 % thereafter Zinc + Lead 3.0 % free US $ 2.50 for each 1 % thereafter Bismuth 0.1 % free US $ 2.00 for each 0.01% thereafter Mercury 20 ppm free US $2.00 for each 100 ppm thereafter Water 10% free US$1.00 for each 1% thereafter
	30 – 35%	96.65%	Minimum 1.1%			
	> 35%	96.7%	Minimum 1.1%			
Silver	< 30 g/t	0%		US $0.35 per payable ozt		
	> 30 g/t	90%				
Gold	1 – 3 g/t	90%		US $6.00 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:

(1) RC for Copper of US $0.09 per payable pound plus a price participation of 10% up to US $0.10 per payable pound when the Copper metal price exceeds $1.50 per pound.

Molybdenum Concentrate Terms

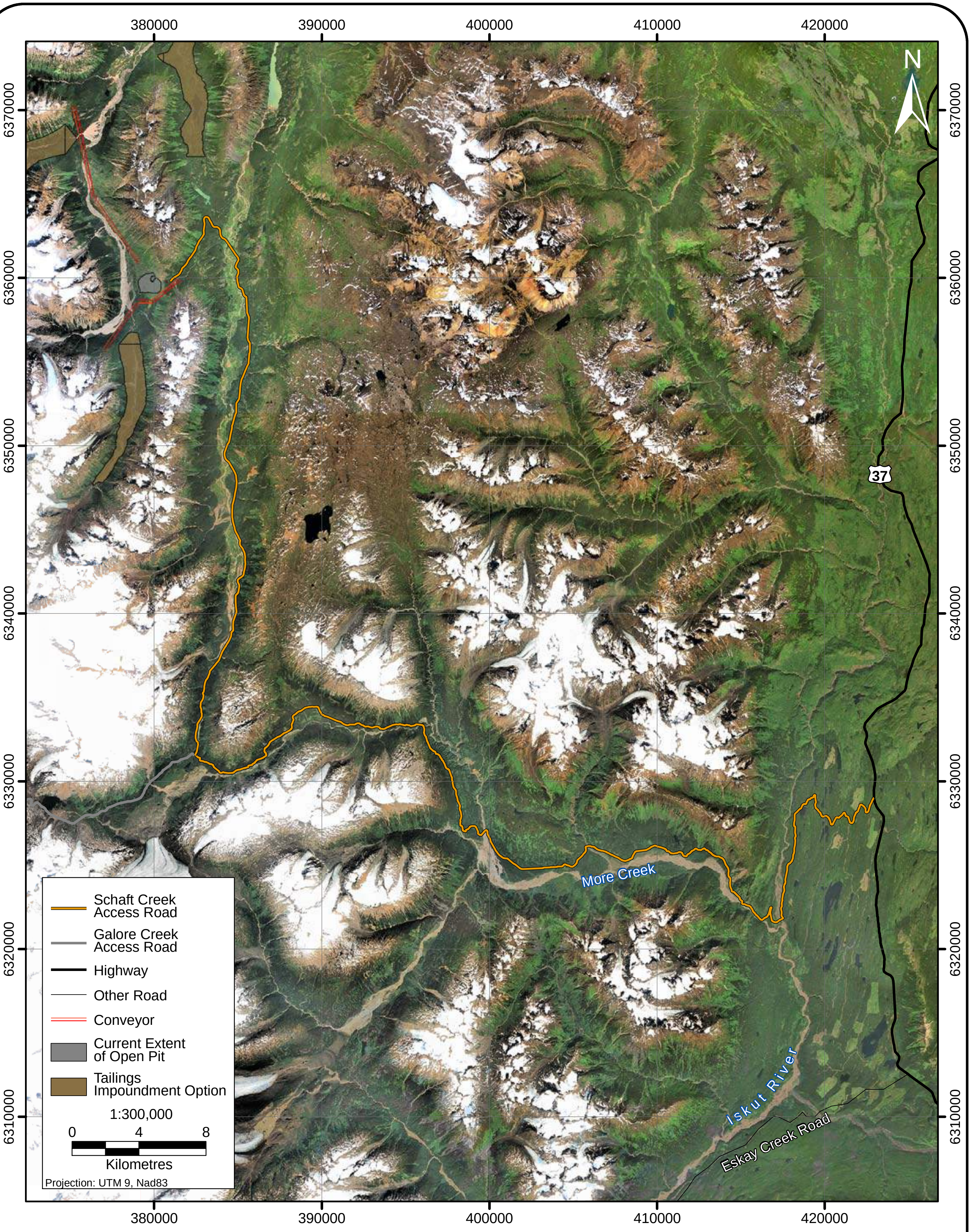
Table 25.39
Processing Smelter Molybdenum Concentrate Terms

Component Metal	Grade	Payable Content	Deductions(1)
Molybdenum	Any	100%	14% of Molybdenum Metal price

Notes:

(1) Deduction subject to a maximum of US $5.50 and a minimum of US $2.50 per pound of contained molybdenum; this also includes any roasting and refining charges.

26.0 Illustrations

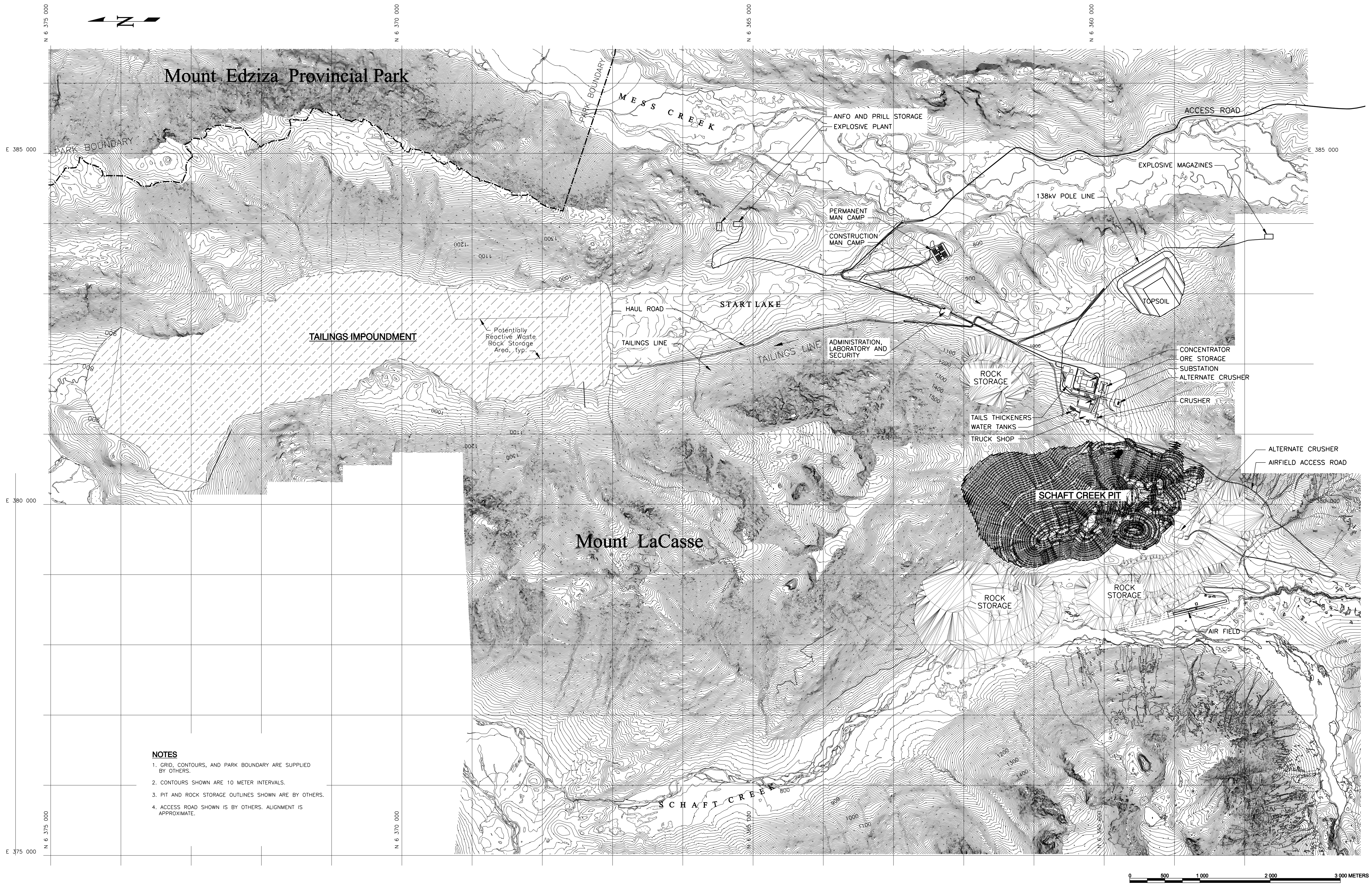


copper FOX metals inc.

FIGURE x.x-x

Schaft Creek Access Road

Rescan™



NOTES

1. GRID, CONTOURS, AND PARK BOUNDARY ARE SUPPLIED BY OTHERS.
2. CONTOURS SHOWN ARE 10 METER INTERVALS.
3. PIT AND ROCK STORAGE OUTLINES SHOWN ARE BY OTHERS.
4. ACCESS ROAD SHOWN IS BY OTHERS. ALIGNMENT IS APPROXIMATE.

NO.	DESCRIPTION	DATE	BY
C	RELOCATED AIRFIELD, ADDED ALTERNATE CRUSHER REISSUED FOR PRE-FEASIBILITY STUDY		
B	REISSUED FOR PRE-FEASIBILITY STUDY	27JUN08	ABE
A	ISSUED FOR PRE-FEASIBILITY STUDY	12 JUN 08	ABE

REVISIONS

DWG NO.	DESCRIPTION

REFERENCE DRAWINGS

		DATE
SCALE:	1 : 25 000	
DESIGNED:	A. EDMONSTONE	MAR. 08
DRAWN:	A. EDMONSTONE	01APR08
CHECKED:		
APPROVED:		
APPROVED		

CLIENT: COPPER FOX METALS, INC.

LOCATION: CALGARY, ALBERTA, CANADA

copper FOX metals inc.

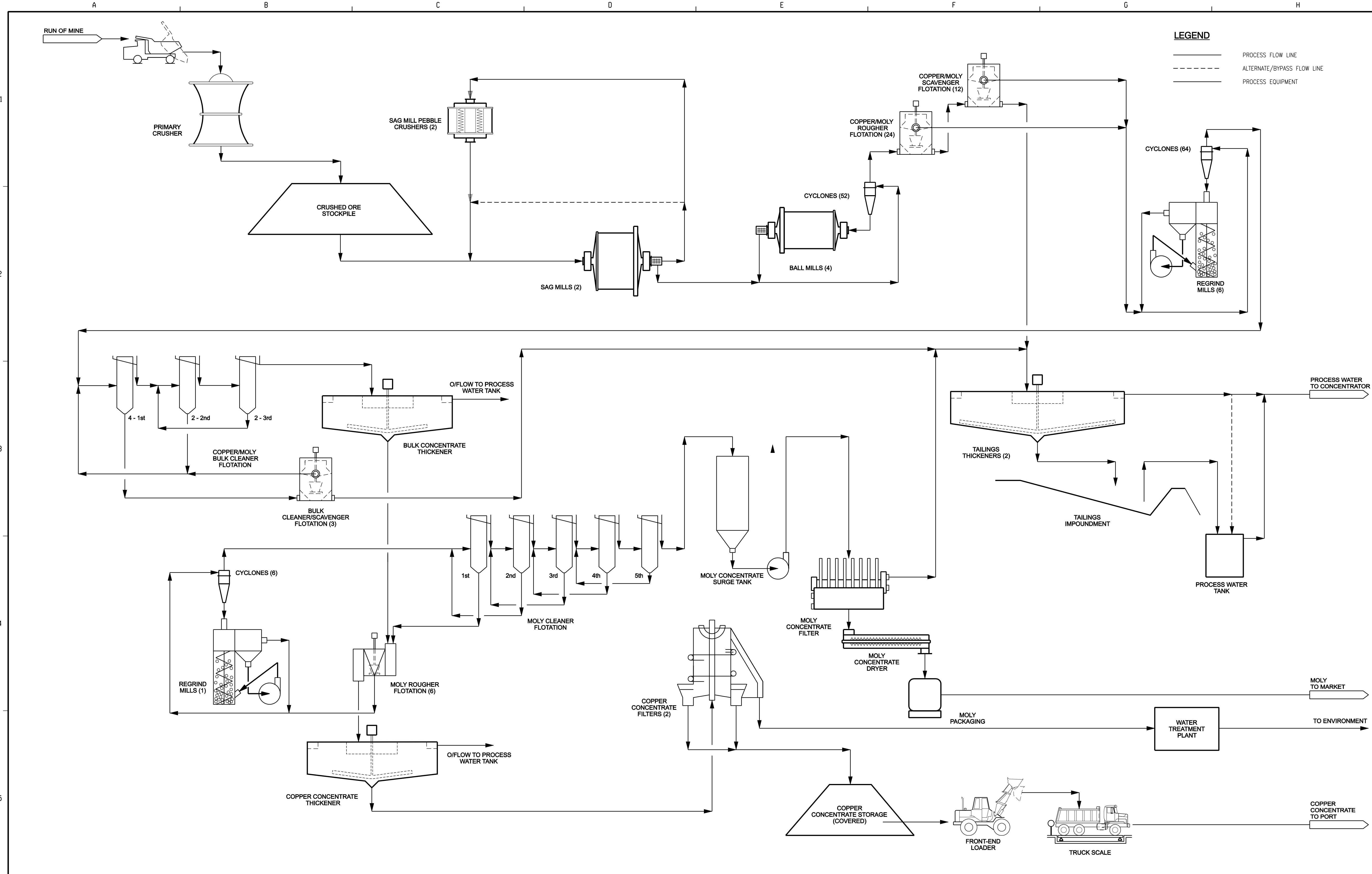
Samuel Engineering, Inc.
We Provide Solutions
8450 E. CRESCENT PARKWAY
GREENWOOD VILLAGE, CO 80111
Phone: 303.714.4840
Fax: 303.714.4800

TITLE
SCHAFT CREEK PREFEASIBILITY STUDY
100,000 DMTPD COPPER CONCENTRATOR
GENERAL ARRANGEMENT
OPERATIONS AREA
SITE PLAN

PROJECT NO. **7148-01**

DRAWING NUMBER 050-GL-051

REV. C



DWG NO.	DESCRIPTION
REFERENCE DRAWINGS	

NO.	DESCRIPTION	DATE	BY
REVISIONS			

NO.	DESCRIPTION	DATE	BY
B	ISSUED FOR PRE-FEASIBILITY STUDY	03 JUN 08	
A	ISSUED FOR REVIEW AND COMMENT	5/28/08	AE

REVISIONS

		DATE
SCALE:	N.T.S.	
DESIGNED:	S. BAUER	JAN.08
DRAWN:	C. SOUKUP	2/6/08
CHECKED:	S. GREEN	FEB.08
APPROVED:		
APPROVED		

CLIENT: COPPER FOX METALS, INC.

LOCATION: CALGARY, ALBERTA, CANADA

copper FOX metals inc.

SE Samuel Engineering, Inc.
We Provide Solutions
8450 E. CRESCENT PARKWAY
GREENWOOD VILLAGE, CO 80111
Phone: 303.714.4840
Fax: 303.714.4800

TITLE
SCHAFT CREEK PRE-FEASIBILITY STUDY
100,000 DMTPD COPPER CONCENTRATOR
PROCESS
SIMPLIFIED PROCESS FLOW DIAGRAM

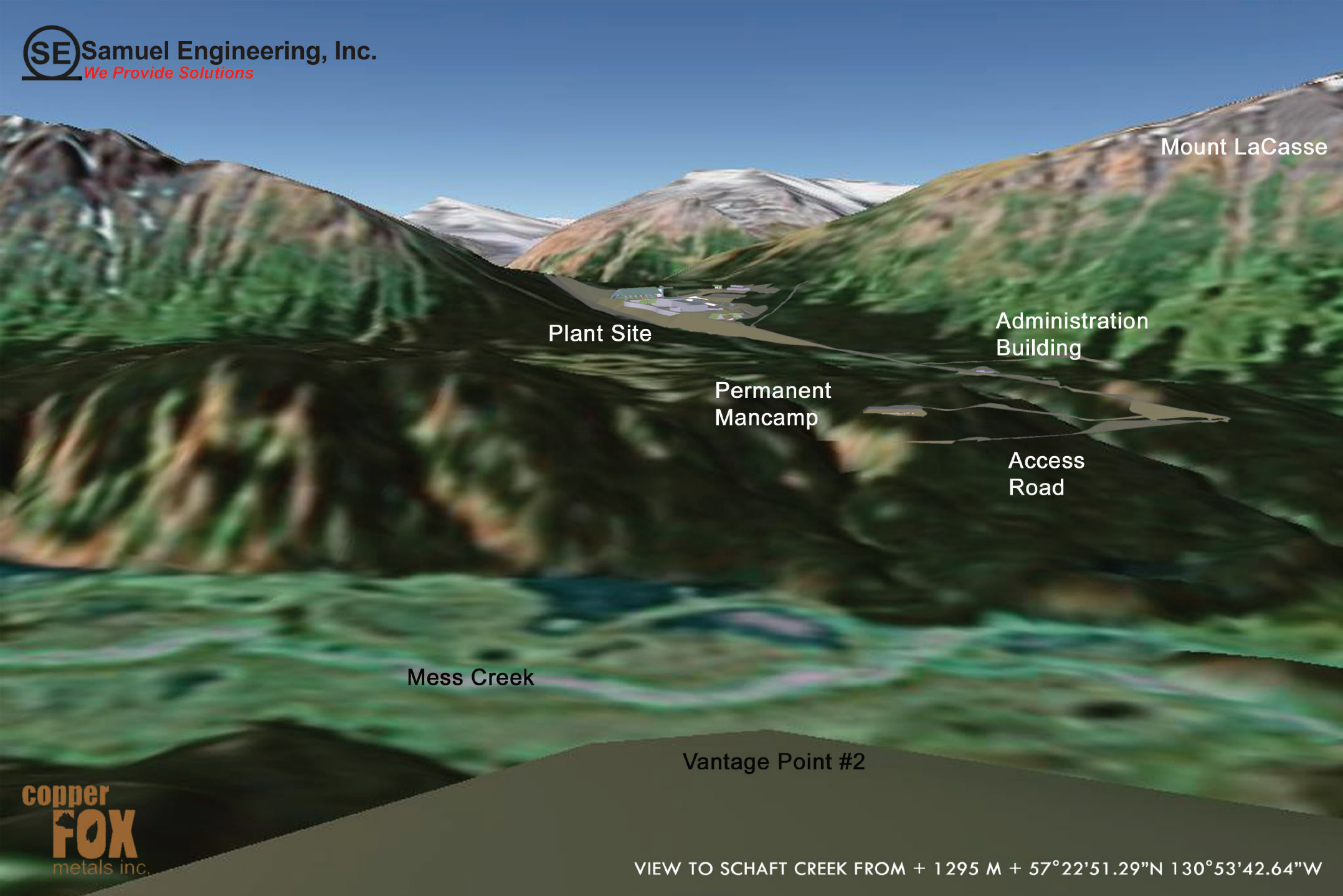
PROJECT NO.
7148-01

DRAWING NUMBER
100-PF-100

REV.
B

...\cad\Flowsheets\100-PF-100.dgn 6/11/2008 9:39:36 AM


Samuel Engineering, Inc.
We Provide Solutions
Mount Edziza
Provincial Park
Plant Site
Mount LaCasse
Skeeter Lake
Valley
Mess Creek
copper
FOX
metals inc.
Vantage Point #1
VIEW TO SCHAFT CREEK FROM + 1641 M + 57°24'27.87"N 130°57'28.66"W


Samuel Engineering, Inc.
We Provide Solutions
Mount LaCasse
Plant Site
Administration Building
Permanent Mancamp
Access Road
Mess Creek
Vantage Point #2
copper FOX metals inc.
VIEW TO SCHAFT CREEK FROM + 1295 M + 57°22'51.29"N 130°53'42.64"W

27.0 Certificate of Qualifications

CERTIFICATE OF QUALIFICATION

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

- Was a full time employee of Samuel Engineering, Inc. at the time of subject technical report;
- 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 responsible Qualified Person for sections 3, 4, 5, 6, 7, 8, 17, 18, 19.7, 20, 21, 22, 23, 25, and 26 of this report: “Amended Technical Report: Preliminary Feasibility Study on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada,” effective September 15, 2008;
- I visited the Schaft Creek site on July 16th, 17th, and 18th, 2007 and again on September 13th, 2007;
- I have had prior involvement with the property that is the subject of the Technical Report (Preliminary Economic Assessment on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada, dated December 7, 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 25th day of May 2010 in Denver, Colorado, USA

(Original Signature on File)

Matt R. Bender, P.E.

Samuel Engineering, Inc.
We Provide Solutions

CERTIFICATE OF QUALIFICATION

I, Keith M[c]Candlish, P.Geo., P.Geol., Managing Director, 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.); a member in good standing of the Association of Professional Engineers, Geologists and Geophysicists of Alberta; and a Registered Professional Geoscientist; a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia;
- I am and 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 sections 3.3, 3.4, 9, 10, 11, 12, 13, 19.1-19.6, and have reviewed and take responsibility for sections 14, 15, and 16 of this report entitled "Amended Technical Report: Preliminary Feasibility Study on the Development of the Schaft Creek Project Located in Northwest British Columbia, Canada," effective date September 15, 2008;
- I last visited the Schaft Creek site from June 13th-16th, 2008;
- 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 19th day of May 2010 in Calgary, Alberta, Canada

(Original Signature on File)

____

Keith McCandlish, P.Geo., P.Geol.

Associated Geosciences