Report to:

SILVER STANDARD RESOURCES INC.

Technical Report and Preliminary Assessment on the Snowfield Property

Document No. 1053750101-REP-R0001-02

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TECHNICAL REPORT AND PRELIMINARY ASSESSMENT ON THE SNOWFIELD PROPERTY

EFFECTIVE DATE: JUNE 1, 2010

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REVISION HISTORY

REV. NO	ISSUE DATE	PREPARED BY AND DATE	REVIEWED BY AND DATE	APPROVED BY AND DATE	DESCRIPTION OF REVISION
00	April 26/10	Anna Siepka April 26/10	John Huang April 26/10	Grant Bosworth April 26/10	Draft report issued.
01	May 20/10	Anna Siepka May 20/10	John Huang May 20/10	Grant Bosworth May 20/10	Final report issued.
02	June 1/10	Anna Siepka June 1/10	John Huang June 1/10	Grant Bosworth June 1/10	Final report to SEDAR.

NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Silver Standard Resources Inc. (Silver Standard) by Wardrop Engineering Inc. (Wardrop), P&E Mining Consultants Inc. (P&E), BGC Engineering Inc. (BGC), Rescan Environmental Services Ltd. (Rescan), and SJA Canada Ltd. (SJA). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Wardrop's, P&E, BGC, Rescan, and SJA, based on (i) information available at the time of preparation, (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use subject to the terms and conditions of Silver Standard's contract with Wardrop, P&E, BGC, Rescan, and SJA. This contract permits Silver Standard to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, *Standards of Disclosure for Mineral Projects*.

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UNITS OF MEASURE

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3

Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	C$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t

Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM

Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Total	T
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

ABBREVIATIONS AND ACRONYMS

Absolute Relative Difference	ARD
Alpine Tundra	AT
ALS Chemex Lab Ltd.	ALS Chemex
Assayers Canada Ltd.	Assayers Canada
Assessment Report Indexing System	ARIS
Atomic Absorption Spectrophotometer	AAS
Au-Equivalent Grade	AuEq
BC Ministry of Energy, Mines, and Petroleum Resources	MEMPR
BGC Engineering Inc.	BGC
Black Hawk Mining Inc.	Black Hawk
British Columbia	BC
British Columbia Environmental Assessment Act	BCEAA
British Columbia Environmental Assessment Office	BCEAO
Canadian Dam Association	CDA
Canadian Environmental Assessment Act	CEAA
Canadian Environmental Assessment Agency	CEAA
Canadian Institute of Mining	CIM
Canadian National Railway	CNR
Carbon-in-Leach	CIL
Closed-Circuit Television	CCTV
Consensus Economics Inc.	Consensus Economics
Counter-Current Decantation	CCD
Direct Leach	DL
Distributed Control System	DCS
Drilling and Blasting	D&B
Energy Metals Consensus Forecast	ECMF
Engelmann Spruce – Subalpine Fir	ESSF
Environmental Management System	EMS
Esso Minerals Canada Ltd.	Esso
Fixed Exchange Rate	FXR
Free Carrier	FCA
General & Administration	G&A

Granduc Mines Ltd.	Granduc
Heating, Ventilating, and Air Conditioning	HVAC
High Pressure Grind Rolls	HPGR
Interior Cedar – Hemlock	ICH
International Congress on Large Dams	ICOLD
International Plasma Labs	IPL
Inverse Distance Squared	ID^2
Internal Rate of Return	IRR
Kerr-Sulphurets-Mitchell	KSM
Land and Resource Management Plan	LRMP
Lerch-Grossman	LG
Life of Mine	LOM
Light Detection and Ranging	LIDAR
Load-Haul-Dump	LHD
Loss on Ignition	LOI
Maximum Design Earthquake	MDE
Metal Mining Effluent Regulations	MMER
Meteorological Service of Canada	MSC
Methyl Isobutyl Carbinol	MIBC
National Instrument 43-101	NI 43-101
Nearest Neighbour	NN
Net Smelter Prices	NSP
Net Smelter Return	NSR
Neutralization Potential	NP
Newhawk Gold Mines Ltd.	Newhawk
Newhawk International Corona Corp.	Newhawk Interational Corona
Northwest Transmission Line	NTL
Official Community Plans	OCPs
Operator Interface Station	OIS
P&E Mining Consultants Inc.	P&E
Potassium Amyl Xanthate	PAX
PRA Metallurgical Division, Inspectorate America Corporation	PRA
Predictive Ecosystem Mapping	PEM
Preliminary Assessment	PA
Preliminary Economic Assessment	PEA
Probable Maximum Flood	PMF
Probable Maximum Precipitation	PMP
Process Research Associates Ltd.	PRA
Qualified Persons	QPs
Rescan Environmental Services Ltd.	Rescan
Resource Modeling Inc.	Resource Modeling
Rock Quality Designation	RQD
Seabridge Gold Inc.	Seabridge
Silver Standard Resources Inc.	Silver Standard
Social and Community Management System	SCMS
Standards Council of Canada	SCC

Tailings Storage Facility	TSF
Terrestrial Ecosystem Mapping	TEM
Thompson-Howarth	T-H
Total Suspended Solids	TSS
Traditional Knowledge/Traditional Use	TK/TU
Tunnel Boring Machine	TBM
Valued Ecosystem Components	VEC
Wardrop Engineering Inc., A Tetra Tech Company	Wardrop
Waste Rock Facility	WRF
Water Balance Model	WBM
Work Breakdown Structure	WBS
Workplace Hazardous Materials Information System	WHIMIS
X-Ray Fluorescence Spectrometer	XRF

1.0 SUMMARY

1.1 INTRODUCTION

In January 2010, Silver Standard Resources Inc. (Silver Standard) commissioned Wardrop Engineering Inc., A Tetra Tech Company (Wardrop) to conduct a preliminary assessment (PA) of the Snowfield deposit.

The following consultants were commissioned to complete the component studies for the National Instrument 43-101 (NI 43-101) Technical Report:

- Wardrop: mining, processing, infrastructure, capital and operating cost estimates, and financial analysis
- P&E Mining Consultants Inc. (P&E): mineral resource estimate
- Rescan Environmental Services Ltd. (Rescan): environmental aspects, waste and water treatment
- BGC Engineering Inc. (BGC): tailings impoundment facility, waste rock and water management, and geotechnical design for the open pit slopes.

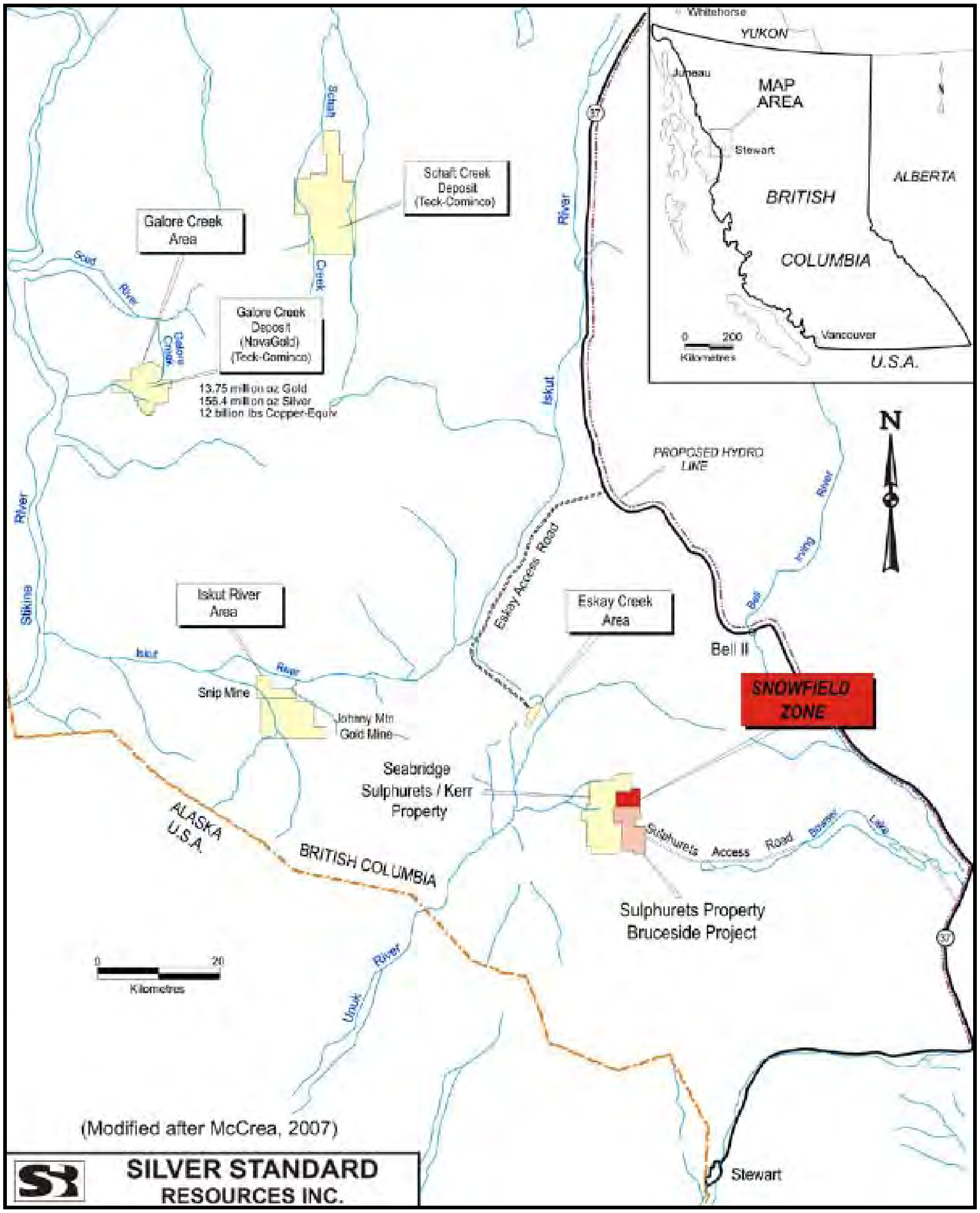
1.2 PROPERTY DESCRIPTION AND LOCATION

In 1999, Silver Standard acquired the Sulphurets claim group, including the Snowfield deposit, through the acquisition of all Newhawk Gold Mines Ltd. (Newhawk) mining shares. Subsequent to the acquisition of Newhawk, Silver Standard reorganized the claim ownership. All of the associated mineral claims are now held by 0777666 B.C. Ltd., a wholly-owned subsidiary of Silver Standard.

The Snowfield property consists of a single mineral claim (509216) totalling 1,267.43 ha and two overlapping placer claims totalling 874.78 ha. Silver Standard is the operator of the property.

The Snowfield property is situated within the Sulphurets District in the Iskut River region, approximately 20 km northwest of Bowser Lake or 65 km north-northwest of the town of Stewart, British Columbia (BC) (Figure 1.1).

Figure 1.1 Location Map of Snowfield Property



1.3 HISTORY

The exploration history of the Sulphurets-Mitchell Creek area dates back to 1933, when placer gold miners worked on Sulphurets Creek. Early work between 1935 and 1959 led to the discovery of several small copper and gold-silver showings in the

Sulphurets-Mitchell Creek and Brucejack Lake areas. In 1959, Granduc Mines Ltd. (Granduc) staked the original Sulphurets claim group.

Between the early 1960s and the late 1990s, when Silver Standard acquired the Snowfield property, the general area was intensely explored by companies such as Granduc, Esso Minerals Canada (Esso), Newhawk, and Newhawk International Corona Corp. (Newhawk International Corona). These companies actively explored the region identifying over 50 mineralized showing including several large mineralized deposits such as the Kerr, Mitchel, Sulphurets, and Snowfield deposits.

Subsequent to acquiring the Snowfield and adjacent Brucejack properties from Black Hawk Mining Inc. (Black Hawk) in 1999, Silver Standard has drilled in excess of 55,500 m of core in approximately 129 holes (2006 through 2009) on the Snowfield property.

1.4 Accessibility, Climate, Local Resources, and Physiography

The property is accessible by helicopter from the town of Stewart, or seasonally from the settlement of Bell II. The flight time from Stewart is approximately 30 minutes and slightly less from Bell II; however, Stewart has an established year-round helicopter base.

The climate is typical of north-western BC with cool, wet summers and relatively moderate but wet winters. Annual temperatures range from approximately +25°C to approximately -25°C. Snowfall accumulations ranging from 10 m to 15 m are common at higher elevations while the accumulations range from 2 m to 3 m along the lower river valleys. The optimum field season is from late June to mid-October.

There are no local resources other than abundant water for any drilling work. The nearest infrastructure is Stewart, BC, which has a minimum of supplies and personnel. BC Hydro is evaluating plans to bring power within approximate 40 km of the Snowfield project site.

Elevations within the property range from 1000 masl along the Mitchell Glacier to 1960 masl along the ridge between the Mitchell and Hanging Glaciers. Local relief, such as at the gossanous Snowfield deposit, is relatively low to moderate elevation.

1.5 Geological Settings

The Snowfield property and the surrounding Sulphurets district are underlain by the Upper Triassic and Lower to Middle Jurassic Hazelton Group of volcanic, volcaniclastic, and sedimentary rocks.

Locally, the Snowfield deposit is hosted by Lower Jurassic andesitic volcanic rocks that correlate with the Upper Andesite unit of the Unuk River formation from the lower portion of the Hazelton Group. The rocks that host the gold mineralization at Snowfield have undergone pervasive hydrothermal alteration resulting in the formation of a moderate to strong foliation that makes identification of protoliths difficult. Margolis (1993) interpreted the mineralized rocks as representing a marine volcanic back-arc environment that consisted of moderately north-westerly-dipping sequences of predominantly andesitic autochthonous breccia flows, lithic, crystal, and lapilli tuffs.

The Snowfield Deposit is a near-surface, low grade, bulk tonnage, porphyry-style, gold deposit that has the additional potential of copper-gold + molybdenum mineralization at depth and west of the Snowfield Fault. The gold mineralization at the Snowfield Deposit is interpreted to be genetically related to one or more Jurassic-age alkaline intrusions.

Gold mineralization occurs as microscopic grains (<30 µm) of electrum encased within 1% to 5% fine-grained, disseminated pyrite that is hosted within schistose, pervasively altered (quartz-sericite-chlorite) volcanic and volcaniclastic rocks. Associated minerals include: galena and sphalerite, tetrahedrite-tennantite, barite, acanthite, minor Mn-rich calcite, and rare chalcopyrite.

1.6 RESOURCES

The 2009 Snowfield drill program extended the known mineralization to the northwest and southeast and, as a result, increased the estimated resources at the Snowfield Deposit. The 2009 updated resource estimate incorporated results from the autumn 2009 diamond drilling program bringing the resource estimate to a total of 19.77 M AuEq ounces in the measured and indicated categories, at a cut-off grade of 0.35 g/t AuEq. A 10.05 M AuEq ounce resource is contained in the Inferred category as shown in Table 1.1.

Table 1.1 December 1, 2009 P&E Snowfield Mineral Resource Estimate

Class	Tonnes (M)	Au (g/t)	Contained Au oz (M)	Ag (g/t)	Contained Ag oz (M)	Cu (%)	Mo (ppm)
Measured	136.9	0.94	4.14	1.7	7.7	0.11	99
Indicated	724.8	0.67	15.63	1.9	43.2	0.12	91
Measured +Indicated	861.7	0.71	19.77	1.8	50.9	0.12	92
Inferred	948.9	0.33	10.05	1.4	43.7	0.07	81

Note: At a 0.35 g/t AuEq cut-off.

1.7 MINING OPERATIONS

The Snowfield project will be an open pit operation with 23 years life of mine (LOM) and a total of 966 Mt of mineralization. Mining will be undertaken using two 45 m^3-electric cable shovels, up to two 39 m^3 diesel hydraulic shovels, and up to thirty 363 t haul trucks with related support equipment over the life of the mine. Benches are planned to be 15 m in height and double benched to a total vertical height of 30 m between catch benches for the final pit.

Benches will be drilled on a 9.3 m x 9.3 m drill pattern to a depth of 16.9 m, including sub-drill. All blast holes will be sampled and assayed. The holes will be loaded and shot with a combination of ANFO and emulsion.

Assay analyses will provide grade control for mineralization. The primary crusher will be located at the pit, which will shorten haul distances of the crushed materials.

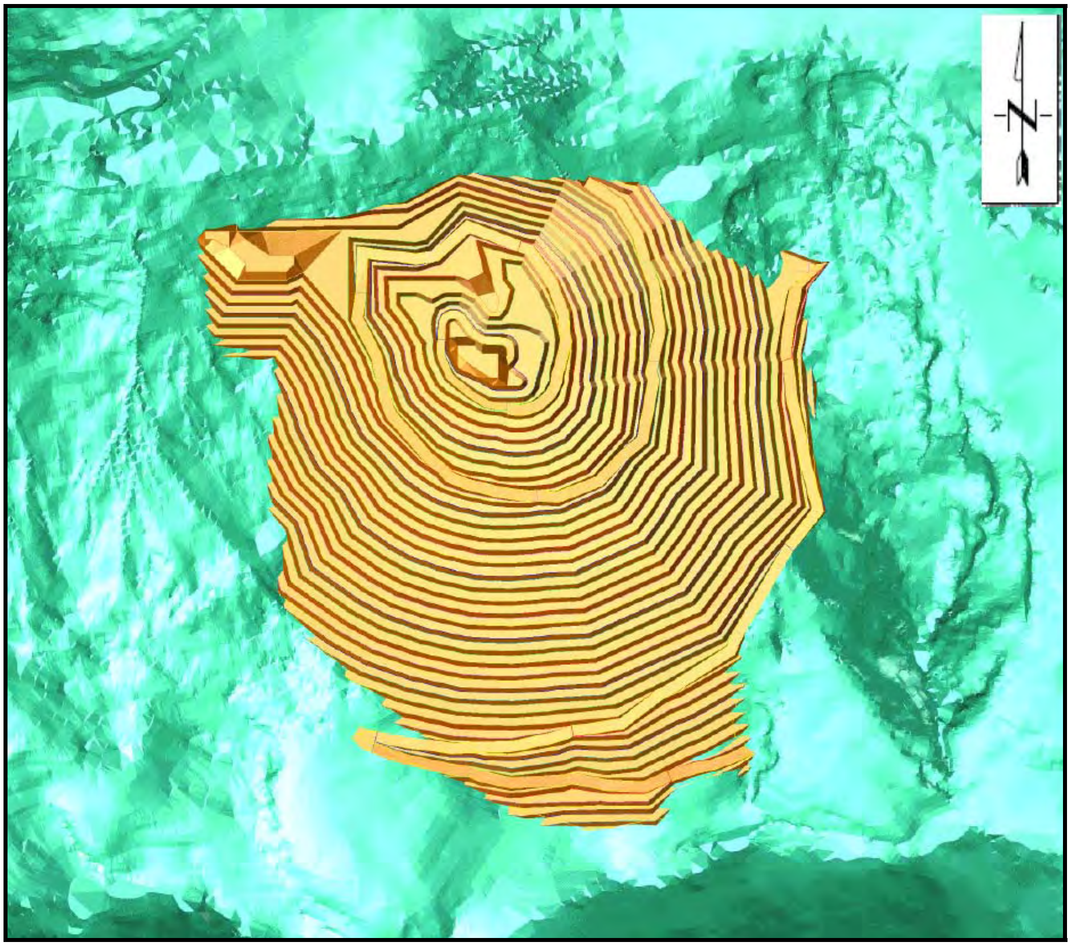
The scoping-level mine plan will be implemented by mining high NSR value material during the first six years of production. The mining of low NSR value material will be deferred to Years 13 to 17.

The mining production schedule is presented in Table 1.2. The final pit is depicted in Figure 1.2.

Table 1.2 Production Schedule

Mining Period	Mine Production (000t)	Waste (000t)	Total Mined (000t)	Strip Ratio	Grade				NSR (US$/t)
					Ag (g/t)	Au (g/t)	Cu (%)	Mo (%)	
-1		7,000	7,000	-	-	-	-	-	-
1	43,800	11,878	55,678	0.27	1.4175	1.3863	0.0362	0.0117	26.2907
2	43,800	1,712	45,512	0.04	1.8153	0.7617	0.0716	0.0115	15.6706
3	43,800	4,938	48,738	0.11	2.0420	0.6017	0.1171	0.0106	14.9305
4	43,800	5,676	49,476	0.13	1.8661	0.6518	0.1192	0.0107	15.9440
5	43,800	6,203	50,003	0.14	1.8162	0.6765	0.1166	0.0101	16.1961
6	43,800	5,625	49,425	0.13	1.8736	0.6822	0.1116	0.0101	16.1201
7	43,799	8,189	51,989	0.19	1.8644	0.7250	0.1190	0.0099	17.2822
8	43,800	18,361	62,161	0.42	1.7684	0.7096	0.1364	0.0095	17.9416
9	43,799	26,649	70,449	0.61	1.4688	0.6406	0.0863	0.0106	13.9221
10	43,799	25,352	69,152	0.58	1.5488	0.6254	0.0870	0.0104	13.6310
11	43,799	28,233	72,033	0.64	1.7065	0.6385	0.1075	0.0090	14.8495
12	43,800	34,924	78,724	0.80	1.6407	0.6357	0.1191	0.0085	15.4426
13	43,788	56,211	100,000	1.28	1.4175	0.5645	0.0943	0.0095	12.7159
14	43,799	56,200	100,000	1.28	1.5274	0.5695	0.0990	0.0092	13.0907
15	43,800	52,962	96,762	1.21	1.4392	0.5179	0.0873	0.0088	11.5837
16	43,800	43,388	87,188	0.99	1.6572	0.5636	0.1146	0.0076	13.8118
17	43,799	48,892	92,692	1.12	1.5135	0.4883	0.1068	0.0074	11.9260
18	43,800	34,776	78,576	0.79	1.4235	0.4775	0.0944	0.0082	11.0801
19	43,799	15,255	59,055	0.35	1.4667	0.5255	0.1044	0.0083	12.4511
20	43,800	12,007	55,807	0.27	1.6887	0.5768	0.1219	0.0072	14.2373
21	43,800	14,468	58,268	0.33	1.8764	0.5356	0.1256	0.0064	13.6194
22	43,800	682	44,482	0.02	1.9116	0.6166	0.1384	0.0081	15.9781
23	2,627	109		0.04	2.3739	0.5912	0.1669	0.0053	16.7600
Total	**966,214**	**519,700,206**	**1,485,915,094**	**0.54**	**1.6720**	**0.6440**	**0.1050**	**0.0090**	**14.9500**

Figure 1.2 Final Pit



1.8 METALLURGICAL TESTWORK REVIEW

Preliminary metallurgical test work, including two locked cycle tests, were carried out to investigate mineralization characteristics, copper/gold/molybdenum bulk flotation, gold bearing pyrite flotation, and gold cyanide leach for individual and composite samples generated from the North and the Upper zones of the Snowfield deposit. The test results appear to indicate that a combination of flotation and cyanidation can be used to recover gold, copper, silver, and molybdenum from the mineralization. The grindability test results showed that the mineralization is moderately hard with an average bond ball mill work index of 16.0 kWh/t. Further testwork is recommended to optimize the flotation and cyanidation flowsheet.

1.9 MINERAL PROCESSING

The proposed Snowfield concentrator will process the gold-copper-molybdenum porphyry mineralization at a nominal rate of 120,000 t/d, operating 365 d/a at an availability of 92%. The concentrator will produce a copper concentrate containing gold and silver, gold-silver doré, and a by-product molybdenum concentrate.

The process plant will consist of crushing and primary grinding, flotation process to recover copper, gold, and molybdenum from the feed material, and cyanidation process to recover gold and silver from the gold-bearing pyrite products.

The crushing will include primary crushing by two gyratory crushers, secondary crushing by four cone crushers, and tertiary crushing by four high pressure grinding rolls (HPGR). Primary crushing will be carried out at the mine site and crushed material will be conveyed to the plant site via a 26 km long tunnel. The crushed material will be further reduced to 80% passing 125 μm prior to the metal recovery by flotation and leaching.

The rougher flotation concentrate will be reground and cleaned to produce a copper-gold-molybdenum concentrate. The rougher flotation tailings will be further floated to recover gold bearing pyrite. A copper-gold and molybdenum separation circuit is proposed to separate molybdenite from copper minerals, to produce a molybdenum concentrate and a copper-gold concentrate. The copper-gold concentrate will be thickened, filtered, and sent to the concentrate stockpile. The molybdenum concentrate will be thickened, filtered, dried, and bagged. Both concentrates will be stored in the plant prior to subsequent shipping to smelters.

The gold bearing pyrite materials will be cyanide leached to recover gold and silver, which will be refined on site to produce gold-silver doré.

The final flotation tailings and leach residues will be transferred to and stored in a conventional tailings impoundment.

1.10 TAILINGS AND WASTE MANAGEMENT

All tailings will be contained within the Scott Creek Valley, located approximately 30 km east-southeast of the pit. A tailings storage facility (TSF) was designed in this valley to contain 966 Mt of tailings based on a mill throughput of 120,000 t/d for the 23-year mine life. During the LOM, tailings will be deposited within the valley and retained by three cross-valley tailings dams to be constructed over the mine life. A 176 m high starter dam will be constructed initially at the south end of the impoundment and raised in stages to an approximate height 283 m above centreline. The two additional dams must be constructed at the northern end of the impoundment during operations to provide containment.

In addition to the tailings dams, the following auxiliary structures will be required for the TSF:

- Spillways – A series of spillways on the right abutment will be constructed over the LOM to protect the integrity of the main tailings dam.
- Operations Diversion Channels – Diversion channels will be constructed above the west and east sides of the ultimate tailings pond to divert non-contact water around the impoundment during the LOM.
- Seepage Recovery Facilities – Seepage recovery systems will be constructed at the toe of each dam to collect potential seepage out of the dam.
- Construction Diversion Tunnel – A diversion tunnel through the right abutment of the main starter tailings dam is required to convey flows from Scott Creek around the starter dam footprint during its construction.

Approximately 520 Mt of waste will be stripped over the LOM, and hauled to two potential waste dumps. The majority of the waste rock will be placed in the East dump, which will contain approximately 90% of the total waste rock. The remainder will be placed in the Southwest dump. Waste rock segregation is assumed to be accomplished depending on the potential of the rock to generate acid and other metals. For the PAG waste dump, steps will be implemented to divert groundwater and surface run-off away from the dump.

1.11 ENVIRONMENTAL CONSIDERATIONS

An initial review of environmental conditions and planned project features indicates that proactive design and mitigation can successfully address environmental impacts associated with developing, operating, and closing the proposed Snowfield project.

As with other projects in the northern Coast Range of BC, water management is a key issue. A suitable location at the Scott Creek valley, with a reasonably small catchment for the tailings storage facility, greatly aids in water management. Diversion channels upslope of the TSF will divert most natural run-off flows around the main dam.

Drainage originating from waste rock, dewatering wells, and the pit itself will be pumped to the upper tunnel portal and piped through the 26 km long access tunnel to the process plant near the TSF. This flow will eventually report to the TSF either directly as liquid or indirectly contained within the tailings slurry.

Discharge from the TSF during operations will be accomplished with a floating decant structure. Installed floating clarifiers will be utilized if suspended solids concentrations are in excess of the mandated value. It is not anticipated that additional water treatment will be required.

Upon closure, the pit will be flooded and excess water will be pumped to the TSF via the tunnel. The diversion channels at the TSF will be breached and discharge will be via a spillway. Protection of stream water quality and fisheries will be a key guiding principle from the earliest planning stages through closure.

Throughout the project, Silver Standard will strive to involve first nations in environmental plans to gain from their knowledge of the region, as well as to keep them informed of project goals.

1.12 INFRASTRUCTURE

The Snowfield site will be accessible by a planned permanent road constructed between a junction with Hwy 37 and the plant site. Hwy 37, a major road access to northern BC, passes approximately 24 km from the Snowfield project plant site (Figure 1.3)

The plant site is located 26 km south east of the open pit area. Twin tunnels constructed with crosscuts will connect the plant site and the mine site. One of the tunnels will be used for conveying the crushed material from the mine site to the 30,000 t live capacity coarse stockpile at the plant site, and the second tunnel will provide a year-round access to the mine site for the transport of the materials and workers.

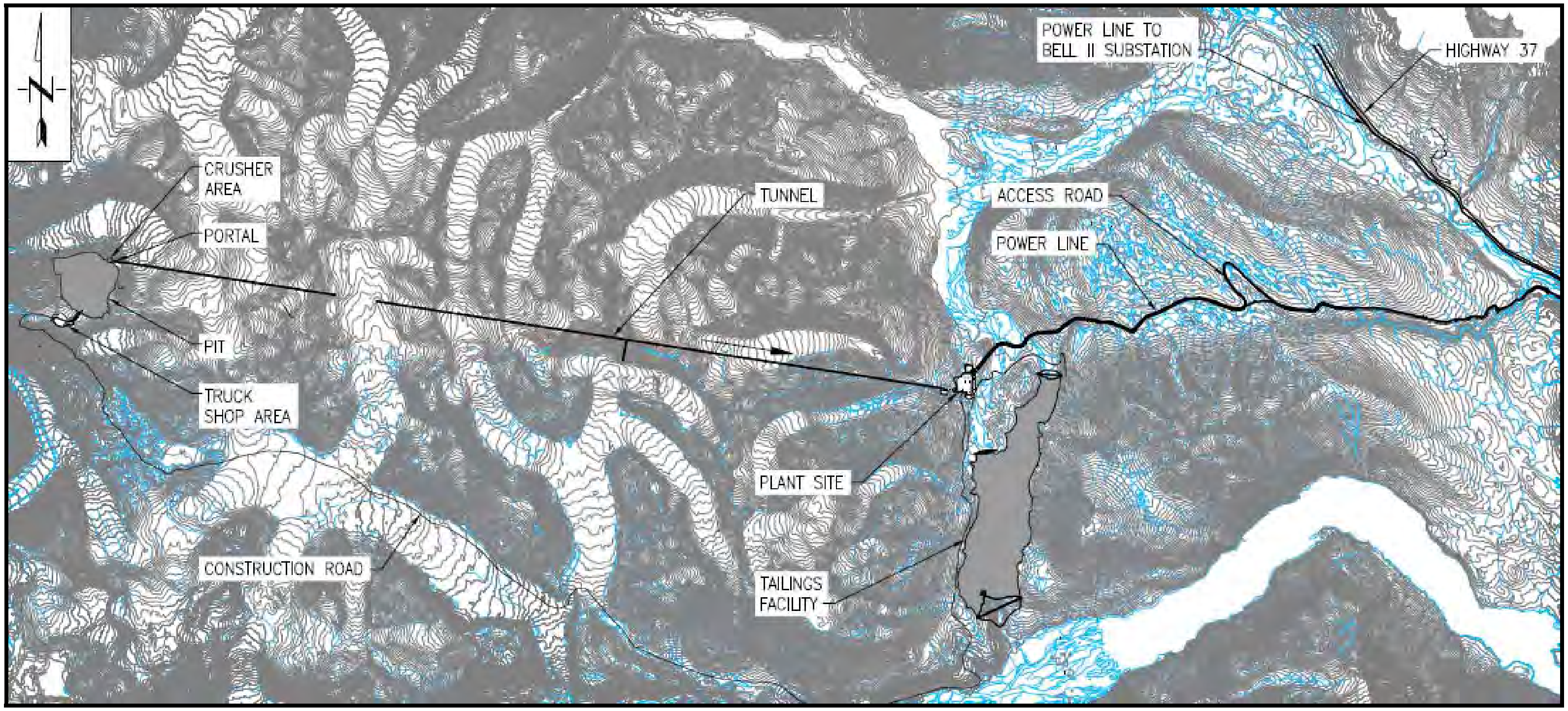
At the mine site, a conventional crushing facility housing two 60’ x 89’ gyratory crushers will be designed to crush the mineralization materials from the proposed mine. The crushing facility will be located northeast of the proposed mine site.

The plant site area will consist of the following facilities:

- 30,000 t live coarse material stockpile (covered) and reclaim
- secondary crushing
- 120,000 t fine material stockpile (covered) and reclaim
- tertiary crushing
- primary grinding and classification, flotation and regrinding
- cyanide leaching
- concentrate dewatering and handling
- maintenance building
- maintenance shop and warehouse
- water services.

The tailings storage facility is located approximately 30 km southeast of the pit and 5 km south of the mill site within the Scott Creek Valley.

Figure 1.3 Snowfield Overall Site Plan



1.13 POWER SUPPLY AND DISTRIBUTION

Electrical power will be supplied from the proposed new Northwest Transmission Line (NTL). The NTL will be a 287 kV line running between Terrace and Bob Quinn Lake, a distance of approximately 335 km. The line to Snowfield from the Bell II substation will be approximately 45 km long, and terminate at a distribution substation at the Snowfield plant site.

There will be four main transformers feeding the plant site. The transformers will be sized to allow the plant to run with one transformer out of service.

Power will be distributed around the site using cables and overhead lines, at 25 kV and additional step-down transformers will be located near remaining loads.

Two additional transformers will be provided at the Snowfield substation to step back up to 69 kV. This will be a suitable voltage to feed via cable through the tunnel to the pit, where it will be further stepped down to 25 kV, 4 kV and 600 V to feed shovels, drills, and the primary crushers.

The tunnel conveyors will be fed from 25 kV cables from at each end of the tunnel. As this is a downhill conveyor, the conveyor drives will be arranged to serve as generators, generating up to 3 MW to 4 MW of power.

1.14 CAPITAL COST ESTIMATE

The estimated initial capital cost of this project, based on the information available at this time, is US$3.378 B (C$3.672 B). This includes a contingency amount 16.3% or US$474 M (C$522 M), which in turn is based on a project contingency risk analysis. The capital cost summary is shown in Table 1.3.

Table 1.3 Capital Cost Summary

Description	Cost (US$)
Direct Works	
Mine Area	728,588,135
Mill Area	591,040,690
Tailing Management, Reclaim Systems, Water Turbidity Control & Closure	433,466,736
Utilities	118,786,094
Site General	173,430,053
Temporary Facilities	92,992,187
Plant Mobile Equipment	7,471,367
Subtotal	**2,145,775,262**
Indirects	
Project Indirects	660,427,128
Contingencies	474,060,369
Owner's Costs	97,722,402
Subtotal	**1,232,209,898**
Total Capital Cost	**3,377,985,160**

1.15 OPERATING COST ESTIMATE

Total operating cost for the project is estimated at C$9.26/t milled. The estimate includes operating costs for mining, process, general and administration (G&A) and surface services. Tailings and water treatment operating costs are included in the sustaining capital costs for the project. A total of 545 personnel are projected for the operation, including 248 personnel for mining, 228 personnel for process, and 69 personnel for general management.

1.16 ECONOMIC EVALUATION

An economic evaluation of the Snowfield project was prepared by Wardrop based on a pre-tax financial model. For the 23 year LOM and 966 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 8.1% IRR
- 7.4 years payback on US$3,378 M capital
- US$877 M NPV at 5% discount rate.

The base case metal prices used for this analysis are as follows:

- silver – US$14.50/oz
- gold – US$878/oz
- copper – US$2.95/lb
- molybdenum – US$17.00/lb.

Sensitivity analyses were carried out on the following parameters:

- copper price
- gold price
- silver price
- molybdenum price
- exchange rate
- copper grade
- gold grade
- silver grade
- molybdenum grade
- operating cost
- capital cost.

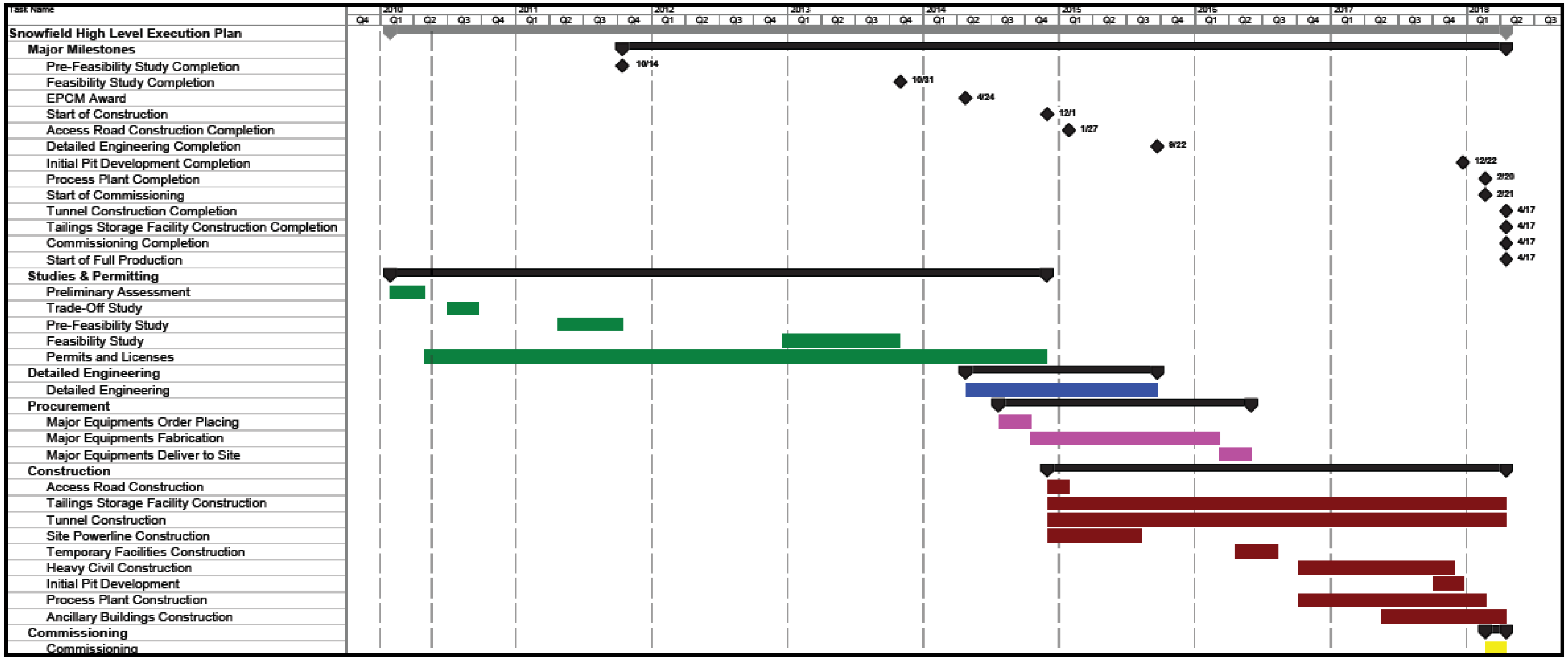
The analyses are presented graphically as financial outcomes in terms of NPV and IRR in section 18.12.6 of this report. The project NPV (at 5% discount rate) is most sensitive to the exchange rate, gold price, and gold grade.

Similarly, the project IRR is most sensitive to the fixed exchange rate (FXR) followed by gold grade and gold price.

1.17 Project Development Plan

The project will take approximately 4 years to complete from the time board approval is received, through construction to introduction of first material in the mill. A further 6 to 8 months is planned for commissioning and ramping of production. The project execution schedule was developed to provide a high level overview of all activities required to complete the project and is summarized in Figure 1.4.

Figure 1.4 Snowfield High Level Execution



1.18 RECOMMENDATIONS AND CONCLUSIONS

Based on the results of the project PA, it is recommended that Silver Standard should continue with the next phase of the project, a pre-feasibility study, in order to identify opportunities and further assess viability of the project.

A detailed list of recommendations, along with the estimated costs to execute each recommendation, is provided in Section 19.0 of this report.

2.0 INTRODUCTION

Silver Standard retained Wardrop to conduct a PA on the Snowfield deposit.

This technical report has been prepared to comply with the standards outlined in the NI 43-101 report. Wardrop compiled this report based on work by the following independent consultants:

- P & E
- Rescan
- BGC.

A summary of qualified persons (QPs) responsible for each section of this report is detailed in Table 2.1.

Table 2.1 Summary of QPs

Report Section	Company	QP
1.0 – Summary	All	Signed-off by Section
2.0 – Introduction	Wardrop	John Huang, P.Eng.
3.0 – Reliance on Other Experts	Wardrop	John Huang, P.Eng.
4.0 – Property Description and Location	P&E	Tracy Armstrong, P.Geo.
5.0 – Accessibility, Climate, Local Resources, Infrastructure and Physiography	P&E	Tracy Armstrong, P.Geo.
6.0 – History	P&E	Tracy Armstrong, P.Geo.
7.0 – Geological Setting	P&E	Tracy Armstrong, P.Geo.
8.0 – Deposit Types	P&E	Tracy Armstrong, P.Geo.
9.0 – Mineralization	P&E	Tracy Armstrong, P.Geo.
10.0 – Exploration	P&E	Tracy Armstrong, P.Geo.
11.0 – Drilling	P&E	Tracy Armstrong, P.Geo.
12.0 – Sampling Method and Approach	P&E	Tracy Armstrong, P.Geo.
13.0 – Sample Preparation, Analysis and Security	P&E	Tracy Armstrong, P.Geo.
14.0 – Data Verification	P&E	Tracy Armstrong, P.Geo.
15.0 – Adjacent Properties	P&E	Tracy Armstrong, P.Geo.
16.0 – Mineral Processing and Metallurgical Testing	Wardrop	John Huang, P.Eng.
17.0 – Mineral Resource	P & E	Fred Brown, CPG Pr.Sci.Nat.

table continues…

Report Section	Company	QP
18.0 – Other Relevant Data and Information		
18.1: Mining Operations	Wardrop	Nory Narciso, P.Eng.
18.2: Infrastructure	Wardrop	John Huang, P.Eng.
18.2.6: Roads and Access	SJA	Mike Boyle, P.Eng.
18.2.7: Site Roads/Earthworks	SJA	Mike Boyle, P.Eng.
18.2.8: Tunnel Development	Wardrop	Iouri Iakovlev, P.Eng.
18.2.11: Power/Electrical	Wardrop	Malcolm Cameron, P.Eng.
18.3: Tailings, Waste Rock, and Water Management	BGC	Lori-Ann Wilchek, P.Eng.
18.3.3: Waste Dump and Open Pit Water Management	BGC	Warren Newcomen, P.Eng.
18.4: Geotechnical	BGC	Lori-Ann Wilchek, P.Eng.
18.4.1: Waste Dumps	BGC	Warren Newcomen, P.Eng.
18.4.2: Pit Slope Angles	BGC	Warren Newcomen, P.Eng.
18.5: Project Execution Plan	Wardrop	Nory Narciso, P.Eng.
18.6: Markets and Contracts	Silver Standard	N/A
18.7 Environmental	Rescan	Pierre Pelletier, P.Eng.
18.8: Taxes	Silver Standard	N/A
18.9: Capital Cost Estimate	Wardrop	Grant Bosworth, P.Eng.
All costs as they relate to the environmental section	Rescan	Pierre Pelletier, P.Eng.
18.10: Operating Cost Estimate	Wardrop	John Huang, P.Eng.
All costs as they relate to the environmental section	Rescan	Pierre Pelletier, P.Eng.
18.10.2: Mining Operating Costs	Wardrop	Nory Narciso, P.Eng.
18.11: Financial Analysis	Wardrop	Nory Narciso, P.Eng.
19.0 – Conclusions and Recommendations	Wardrop	Grant Bosworth, P.Eng.
20.0 – References	Wardrop	Grant Bosworth, P.Eng.
21.0 – Certificates of Qualified Person	All	N/A

3.0 RELIANCE ON OTHER EXPERTS

The authors wish to make clear that they are qualified persons only in respect of the areas in this report identified in their "Certificates of Qualified Persons" submitted with this report to the Canadian Securities Administrators.

A draft copy of the report has been reviewed for factual errors by Silver Standard. Hence, the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are neither false nor misleading at the date of this report.

Silver Standard's employees, who are not QPs, provided additional information on taxes and marketing.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 DESCRIPTION AND TENURE

In 1999, all shares (100%) of Newhawk, a junior resource company, were acquired by Silver Standard under a plan of arrangement. At the time, Newhawk owned the Snowfield property and adjacent Brucejack property (previously referred to as the Sulphurets property). Subsequent to the acquisition of Newhawk, the entire Snowfield and Brucejack mineral claims were reorganized and are now held by 0777666 B.C. Ltd., a wholly-owned subsidiary of Silver Standard, who remains operator of the property.

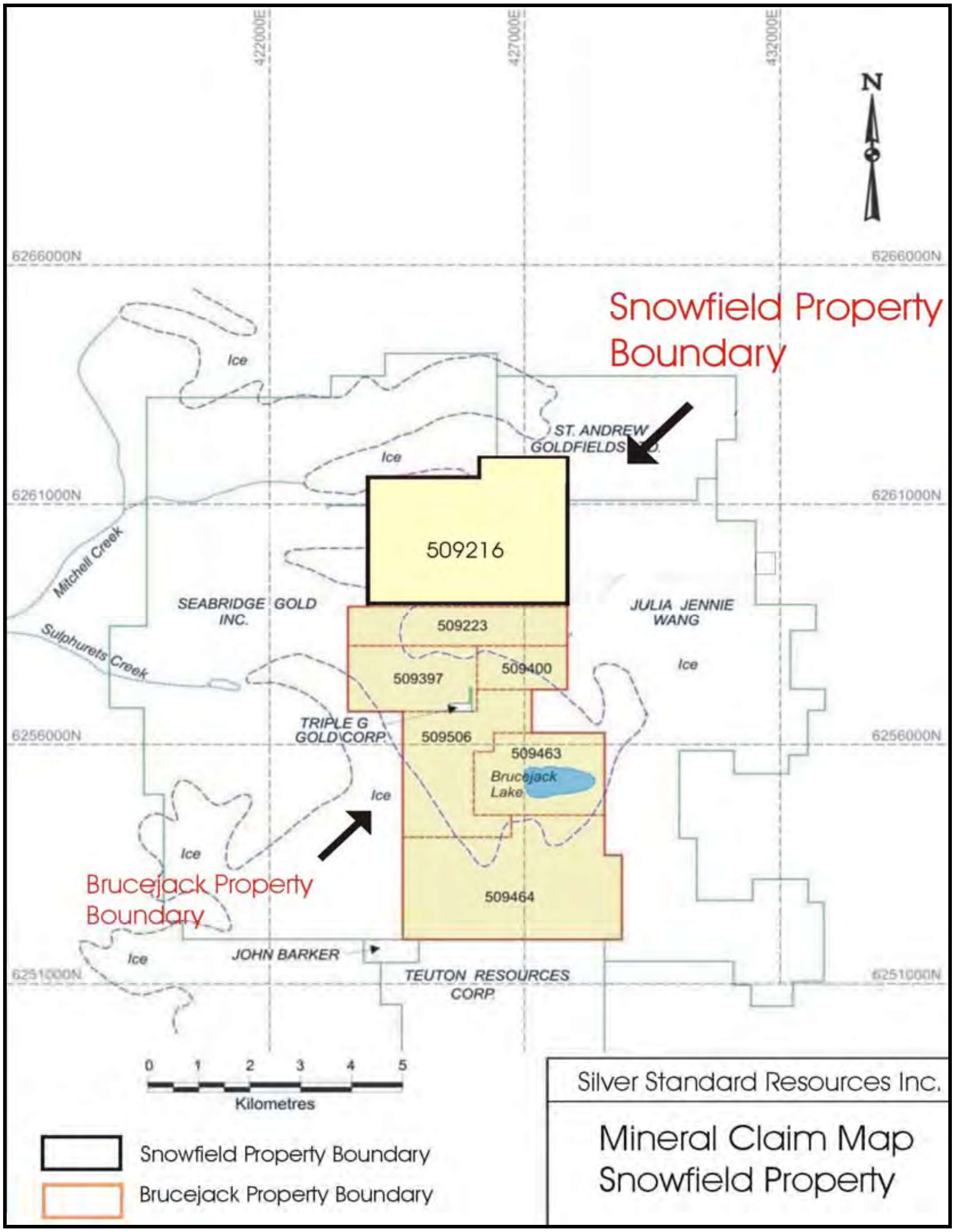
The Snowfield property is composed of one mineral claim (509216) and two placer claims, totalling 2,142.2 ha. The two placer claims overlap the mineral claims. There is one small internal mining lease owned by Triple G Gold Corp. within the claim holdings.

The list of claims is presented in Table 4.1 and the location and configuration of the subject claims and the third-party internal mining lease are shown in Figure 4.1.

Table 4.1 Claims Listing for the Snowfield Property

Tenure No.	Type	ha	Map	Expiry	Status	Owner
509216	Mineral	1267.43	104B	Jan. 31, 2017	Good	0777666 B.C. Ltd.
594266	Placer	428.39	104B	Jan. 31, 2011	Good	0777666 B.C. Ltd.
594267	Placer	446.39	104B	Jan. 31, 2011	Good	0777666 B.C. Ltd.

Figure 4.1 Claims Location Map



Note: After Blanchflower, 2008.

4.2 LOCATION

The Snowfield property is situated at an approximate latitude of 56°31'5″N by Longitude 130°12'18″W. The property is situated approximately 950 km northwest of Vancouver, 65 km northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine. The geographic centre of the property is at UTM coordinates 6,264,193 m north by 434,777 m east, Zone 09 (NAD 83), within NTS map sheet 104B/9 east.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 LOCATION AND ACCESS

The Snowfield property is located in the Boundary Range of the Coast Mountain physiographic belt along the western margin of the Intermontane tectonic belt. The local terrain is generally steep with local reliefs of 1000 m from valleys occupied by receding glaciers, to ridges at elevations of 1200 masl. Elevations within the property range from 1000 m along the Mitchell Glacier to 1960 masl along the ridge between the Mitchell and Hanging Glaciers. However, within several areas of the property, such as at the gossanous Snowfield Deposit, the relief is relatively low to moderate.

The property is easily accessible with the use of a chartered helicopter from the town of Stewart, or seasonally from the settlement of Bell II. The flight time from Stewart is approximately 30 minutes and slightly less from Bell II; however, Stewart has an established year-round helicopter base.

Heavy exploration equipment, fuel, and camp provisions can be transported along a good gravel road from Stewart to the Granduc staging site and then flown by helicopter to the property. This combined truck and helicopter transportation method cuts the more expensive helicopter flight time in half from Stewart.

5.2 CLIMATE AND PHYSIOGRAPHY

The climate is typical of northwestern BC with cool, wet summers, and relatively moderate but wet winters. Annual temperatures range from +20°C to -20°C. Precipitation is high with heavy snowfall accumulations ranging from 10 m to 15 m at higher elevations and 2 m to 3 m along the lower river valleys. Snow packs cover the higher elevations from October to May. The optimum field season is from late June to mid-October.

The tree line is at approximately 1200 m elevation. Sparse fir, spruce, and alder grow along the valley bottoms with only scrub alpine spruce, juniper, alpine grass, moss, and heather covering the steep valley walls. The Snowfield deposit, at an elevation above 1500 m, has only sparse mosses along drainages. Rocky glacial moraine and polished glacial-striated outcrops dominate the terrain above tree line.

The Snowfield deposit is centered between the Mitchell Glacier to the north and the Hanging Glacier to the south. The southern limit of the zone is covered by a small snowfield, which has been rapidly receding.

5.3 INFRASTRUCTURE

The Snowfield property lies immediately east of Seabridge Gold Inc.'s (Seabridge) Kerr-Sulphurets-Mitchell (KSM) property and could be influenced by future access plans for that area as outlined within the Preliminary Economic Assessment (PEA) by Seabridge.

All essential services are available in Stewart, BC, approximately 65 km to the southwest. The town supplies most of the commonly needed materials and services for local camps. The towns of Terrace and Smithers, BC, are also located in the same general region as the Snowfield property. Both towns are directly accessible by daily air service from Vancouver, BC.

6.0 HISTORY

The region surrounding the Snowfield property has a history rich in exploration for precious and base metals dating back to the late 1800s. A detailed summary of the historical exploration of the area is available in Ewert et al. (2009) and Armstrong et al. (2009). This summary has been compiled mostly from various assessment reports available through the BC Ministry of Energy, Mines, and Petroleum Resources (MEMPR) online Assessment Report Indexing System (ARIS).

6.1 SUMMARY OF HISTORICAL EXPLORATION

The exploration history of the Sulphurets-Mitchell Creek area dates back to 1933 when placer gold miners worked on Sulphurets Creek. In 1959, Granduc staked the original Sulphurets claim group (McCrea, 2007) starting the era of modern exploration as briefly outlined below:

- **1960-1980** – Granduc carried out regional reconnaissance prospecting, mapping, and rock sampling over the entire Sulphurets area resulting in the discovery of several porphyry copper-molybdenum and copper-gold occurrences.
- **1980** – Esso optioned the Sulphurets property and conducted detailed geological mapping, trenching, and rock geochemical sampling. The results of this work led to the discovery of the Snowfield, Quartz Stockwork, and Moly zones.
- **1981-1983** – Esso continued exploring the Snowfield zone which appeared to have the potential for a large, low grade gold deposit.
- **1983** – Esso excavated and sampled 24 trenches, totalling 192 m, in the Snowfield zone outlining a 240 m by 120 m area of gold mineralization with an average grade of 0.088 oz/t gold (McCrea, 2007). Their work also discovered the Josephine zone with vein-hosted gold-silver mineralization.
- **1985** – Esso terminated their option of the Sulphurets property. Newhawk and Granduc entered into a 60:40 joint venture agreement with Newhawk operating.
- **1985-1988** – Newhawk tested the Snowfield zone with five diamond drill holes totalling 740 m. At the time, the mineralization was interpreted to be a tabular, shallow, southwardly dipping body averaging 70 m thick. Preliminary metallurgical testing was carried out on the drill core and prospecting continued on the property until 1989.

- **1989** – Newhawk-International joint venture established a property-wide control grid (8 line-km) and conducted a rock sampling program including further rock sampling and trenching on the Snowfield zone.
- **1991** - Two drill holes, totalling 350 m, tested the Snowfield zone with additional rock sampling along its eastern exposed limits. The Newhawk-International joint venture also funded a doctoral thesis on the property by Jake Margolis, which was published in 1993.
- **1993** – Three deep diamond drill holes, totalling 1,164 m, tested the southern extension of the Snowfield zone and another three drill holes, totalling 295 m, tested the nearby Josephine Vein zone.
- **1999** - Silver Standard acquired the Sulphurets claim through the acquisition of all of the shares of Newhawk, including the subject claims.
- **2006** – Silver Standard evaluated the Snowfield zone with 27 diamond drill holes, totalling 6,141 m, and rock sampling to test the lateral and vertical limits of the gold mineralization.
- **2007** – Silver Standard drilled 29 NQ-2 size diamond drill holes, totalling 8,666.29 m. Twenty-one drill holes tested the Snowfield zone, six drill holes tested the nearby Coffeepot zone situated immediately west of the Snowfield zone, and one drill hole tested the Mitchell East zone (now recognized to be the northern extension of the Snowfield zone). A total of 5,484 samples were collected from the 2007 drill core.

7.0 GEOLOGICAL SETTING

The following description of the regional and local geology of the Snowfield property is drawn heavily from the Technical Report titled, “Technical Report on the Snowfield Property, Skeena Mining Division, British Columbia, Canada”, by Minorex Consulting Ltd., dated April 21, 2008.

The Sulphurets district is situated along the western margin of the Intermontane Tectonic Belt, underlain by Stikine Terrane. This district has been the subject of several geological studies since the mid-1980s when it was actively explored for porphyry copper-molybdenum and copper-gold (i.e. Kerr), exhalative volcanogenic (i.e. Eskay Creek), and lode gold-silver vein deposits (i.e. Snip). Researchers include scientists from the Geological Survey of Canada, the British Columbia Geological Survey, the University of British Columbia, and the University of Oregon. The following discussion of the regional geology is a brief summary of their findings. Figure 7.1 shows the geology of the Sulphurets area.

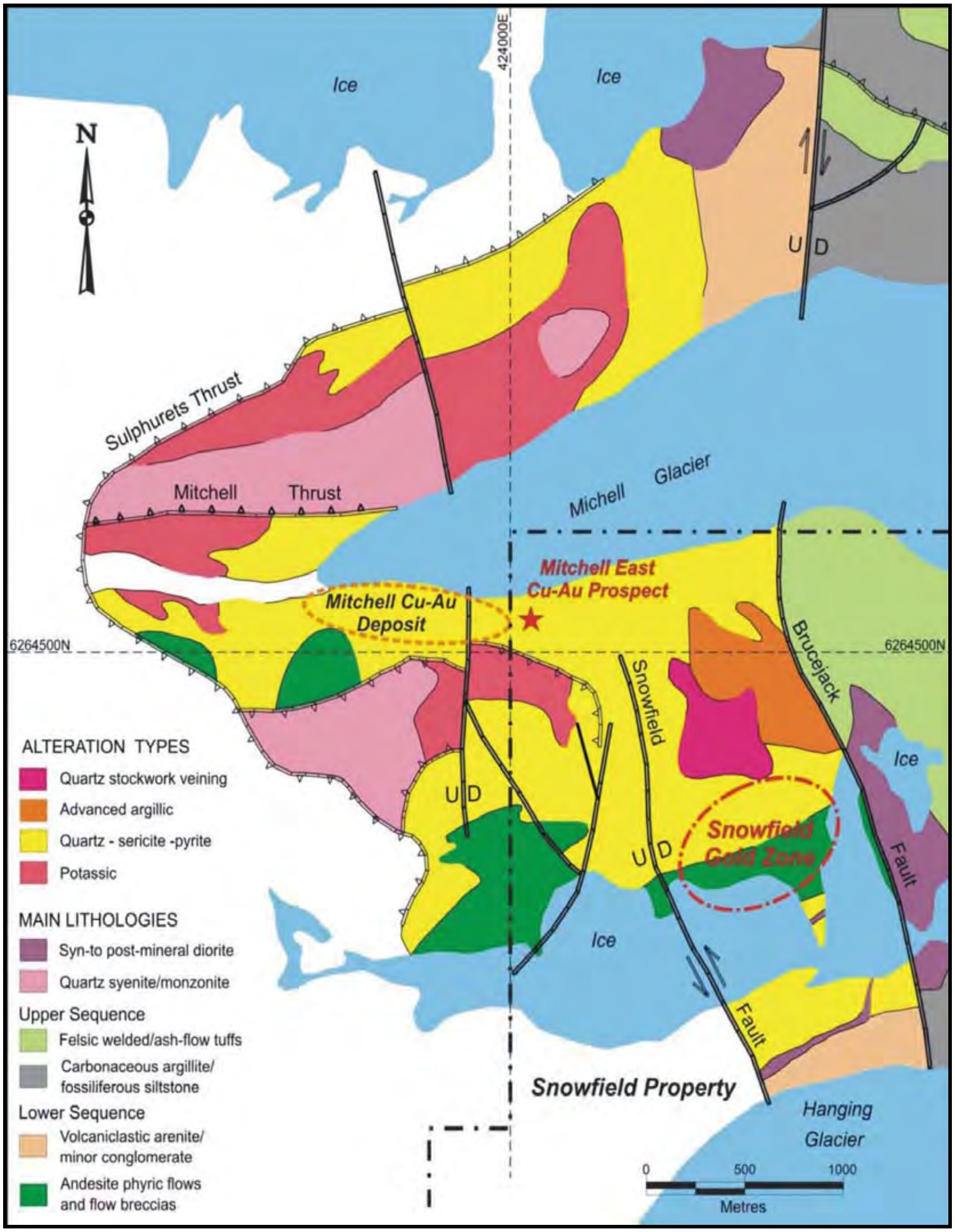
7.1 REGIONAL GEOLOGY

The Snowfield property and the surrounding Sulphurets district are underlain by the Upper Triassic and Lower to Middle Jurassic Hazelton Group of volcanic, volcaniclastic, and sedimentary rocks. According to Roach and MacDonald (1992), the stratigraphic assemblage comprises a package, from oldest to youngest, of:

- Lower Unuk River Formation: alternating siltstones and conglomerates
- Upper Unuk River Formation: alternating intermediate volcanic rocks and siltstones
- Betty Creek Formation: alternating conglomerates, sandstones, and intermediate to mafic volcanic rocks
- Mount Dilworth Formation: felsic pyroclastic tuffaceous rocks and flows
- Salmon River and Bowser Formations: alternating siltstones and sandstones.

Britton and Alldrick (1988) have described three intrusive episodes in the area including intermediate to felsic plutons that are probably coeval with volcanic and volcaniclastic supracrustal rocks, small stocks related to the Cretaceous Coast Plutonic Complex, and minor tertiary dykes and sills.

Figure 7.1 Geology of the Sulphurets Area (After Blanchflower,2008)



Note: Modified after McCrea, 2007

The Hazelton Group lithologies display fold styles ranging from gently warped to tight disharmonic folds. Northerly striking, steep normal faults are common and syn-volcanic, syn-sedimentary, and syn-intrusive faults have been inferred in the region. Minor thrust faults, dipping westerly, are common in the region and are important in the northern and western parts of the Sulphurets area in regard to the interpretation of mineralized zones. Metamorphic grade throughout the area is, at least, lower greenschist.

There are more than seventy documented mineral occurrences and showings in the Sulphurets area. Copper, molybdenum, gold, and silver mineralization found within gossans have affinities to both porphyry and mesothermal to epithermal types of vein deposits. Most mineral deposits occur in the upper members of the Unuk River Formation or the lower members of the Betty Creek Formation.

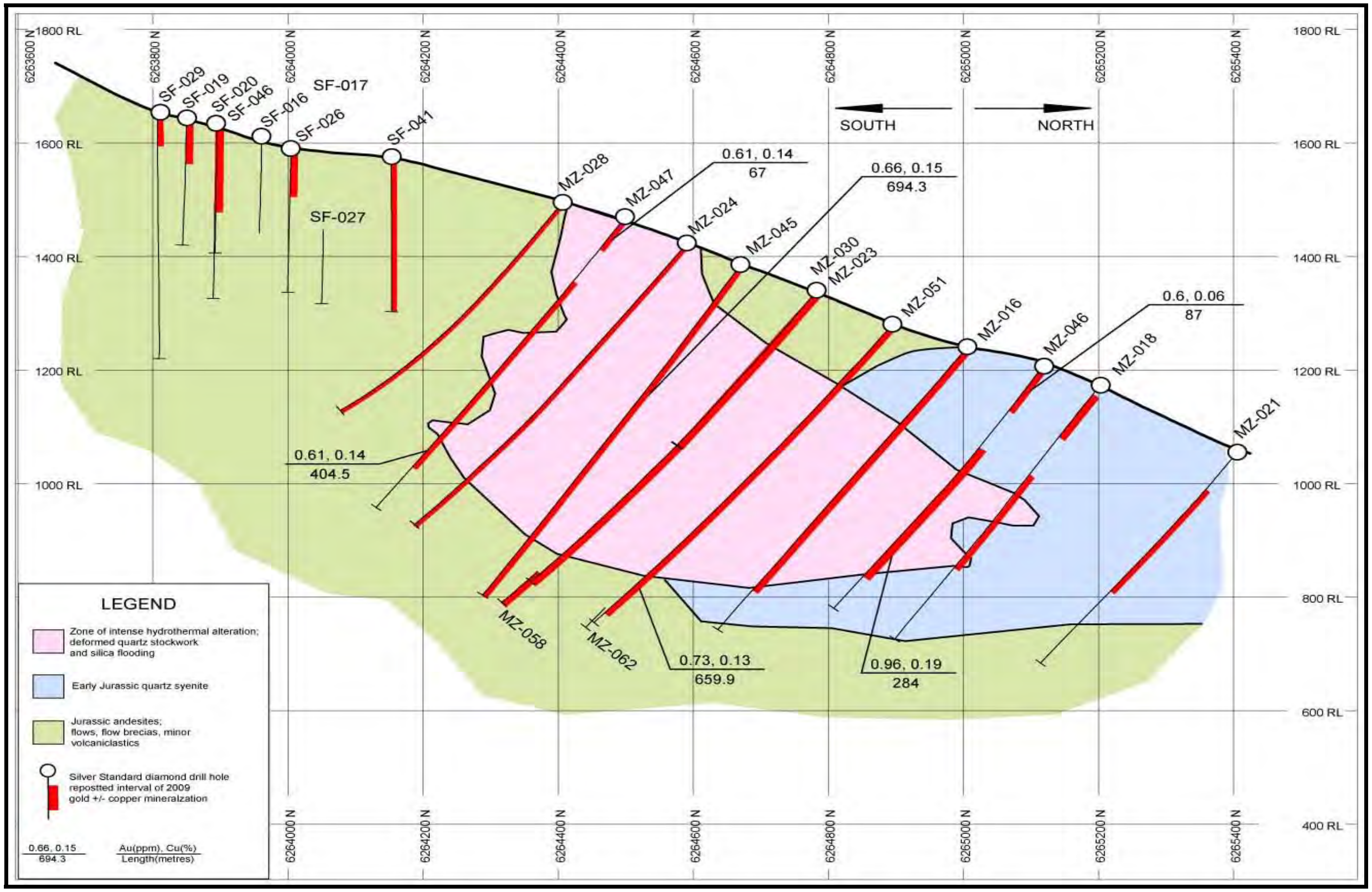
7.2 Structure and Alteration

The Snowfield zone (Figure 7.1) is underlain by Lower Jurassic andesitic volcanic rocks that correlate with the 'Upper Andesite' unit of the Unuk River formation from the lower portion of the Hazelton Group (Alldrick and Britton 1991).

The rocks that host the gold mineralization at the Snowfield zone have been subjected to a lower greenschist facies grade of metamorphism with subsequent pervasive hydrothermal alteration, making the identification of protoliths difficult (Figure 7.2). Based upon geological mapping, petrographic studies, and recent drilling results, the mineralized rocks are interpreted to be a marine volcanic back-arc sequence forming a moderate north-westerly-dipping sequence of predominantly andesitic autochthonous breccia flow, lithic, crystal, and lapilli tuff.

Porphyritic quartz-syenite is exposed approximately 3 km west of the Snowfield zone where it occurs in the upper plate of the Sulphurets thrust fault. A U-Pb age date of 192.7 + 5.4/-3.6 Ma was obtained for this felsic intrusive, which is believed to underlie the Snowfield zone and surrounding area to the west and north at depth.

Figure 7.2 Cross Section 424800E Through the Snowfield Deposit Showing 2009 (MZ series) Holes



7.2.1 STRUCTURE

The Sulphurets thrust fault, situated approximately 1 km west of the property, is a west-dipping, northerly-striking structure that places Triassic Stuhini Group over the Lower Jurassic Hazelton Group rocks, part of the regional Late Mesozoic Skeena fold and thrust belt (Margolis 1993).

The Mitchell Thrust Fault, located on the south side of the Mitchell Valley, separates potassically-altered quartz-syenite and other rocks above it from dominantly sericitically altered rocks and the Mitchell quartz stockwork beneath. This low-angle thrust fault appears to have been transferred to a higher-angle, oblique-slip movement along the Snowfield Fault thus, producing a horst within the Snowfield zone.

Two northerly-striking, post-mineralization high-angle faults occurring east and west of the Snowfield zone are called the Brucejack and Snowfield Faults respectively (Figure 7.1). The left-lateral and eastside-down, vertical Snowfield Fault was apparently formed during southeast-directed thrusting which produced the Mitchell and Sulphurets thrusts (Margolis 1993). The Brucejack Fault is a more regional northerly-striking structure that transects the Sulphurets district, truncating geological features and influencing topography.

7.2.2 ALTERATION

The Snowfield zone is situated within the eastern of two structural blocks separated by the northerly-trending Snowfield Fault. The eastern, down-dropped block of volcanic rocks has been pervasively altered to advanced argillic facies, has a quartz stockwork zone, and is rarely affected by potassic alteration east of the fault. In contrast, the western block which has been uplifted has potassic, sericitic and rare advanced argillic alteration accompanying the quartz-syenite intrusion.

According to Margolis (1993), chlorite-rich quartz-sericite-pyrite alteration of the andesitic volcanic rocks is pervasive east of the Snowfield Fault and throughout the Snowfield zone, in contrast to the chlorite-poor alteration west of the fault. The altered host rocks contain abundant disseminations and fracture filling molybdenite and tourmaline which are cut by pyrophyllite veins in the advanced argillic zone and by massive pyrite veins elsewhere in the area. There is evidence that the quartz-sericite-pyrite-chlorite alteration replaced potassic alteration which was rich in hydrothermal biotite, magnetite, and chalcopyrite (Margolis 1993). Beyond the known limits of the Snowfield zone, the quartz-sericite-pyrite-chlorite altered rocks are poorly mineralized, except for molybdenite.

8.0 DEPOSIT TYPES

A complete discussion of the Deposit Model and its related characteristics is given by Armstrong et al. (2009). A brief summary is given below:

The Snowfield deposit is a near-surface, low grade, bulk tonnage, and porphyry-style gold deposit that has the additional potential of copper-gold + molybdenum mineralization at depth and west of the Snowfield Fault. The gold mineralization at the Snowfield deposit, as well as the copper-gold + molybdenum porphyry-style mineralization of the Mitchell deposit that is currently being tested by Seabridge Gold Inc. (Seabridge) on the adjacent Kerr-Sulphurets property to the north and west, is interpreted to be genetically related to one or more Jurassic-age alkaline intrusions (Alldrick and Britton 1991; Margolis 1993).

The following deposit description is taken from the Geology of Canadian Mineral Deposit Types edited by O.R. Eckstrand, W.D. Sinclair, and R.I. Thorpe.

8.1 PORPHYRY DEPOSITS

8.1.1 GEOLOGICAL FEATURES

The following features serve to distinguish porphyry deposits from other types of deposits:

- large size
- widespread alteration
- structurally controlled ore minerals superimposed on pre-existing host rocks
- distinctive metal associations
- spatial, temporal, and genetic relationships to porphyritic epizonal and mesozonal intrusions.

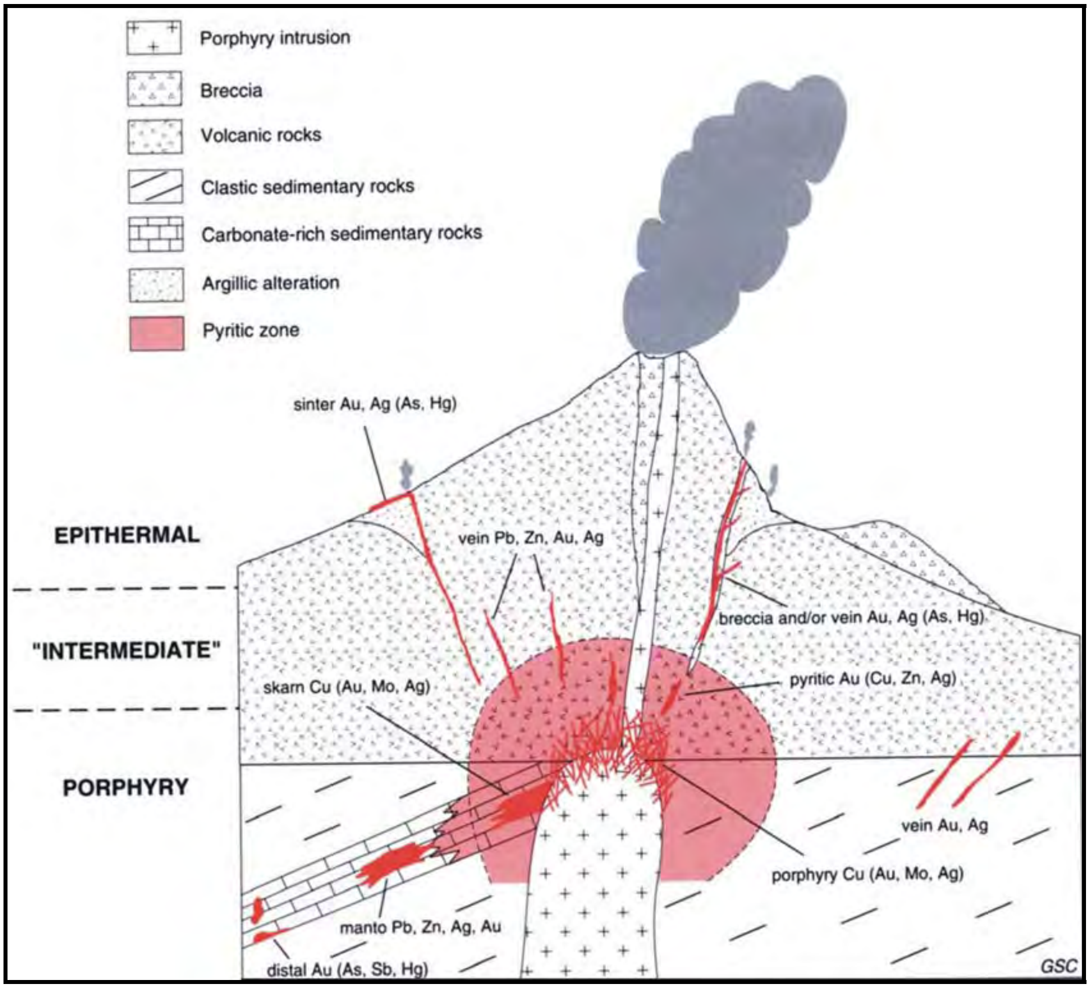
8.1.2 GENETIC MODEL

The most applicable model for porphyry deposits is a magmatic hydrothermal one, or variations thereon, in which the ore metals were derived from temporally and genetically related intrusions (Figure 8.1). Large polyphase hydrothermal systems developed within and above genetically related intrusions and commonly interacted with meteoric fluids (and possibly seawater) on their tops and peripheries. During the waning stages of hydrothermal activity, the magmatic-hydrothermal systems collapsed inward upon themselves and were replaced by waters of dominantly meteoric origin. Redistribution, and possibly further concentration of metals, occurred in some deposits during these waning stages.

Variations of the magmatic-hydrothermal model for porphyry deposits, commonly referred to as the orthomagmatic model, have been presented by numerous authors such as Burnham (1967, 1979), Phillips (1973), and Whitney (1975, 1984). These authors envisaged felsic and intermediate magma emplacement at high levels in the crust and border zone crystallization along the walls and roof of the magma chamber. As a consequence of this crystallization, supersaturation of volatile phases occurred within the magma, resulting in separation of volatiles due to resurgent or second boiling (Figure 8.2). Ore metals and many other components were strongly partitioned into these volatile phases, which became concentrated in the carapace of the magma chamber. When increasing fluid pressures exceeded lithostatic pressures and the tensile strength of the overlying rocks, fracturing of these rocks occurred, permitting rapid escape of hydrothermal fluids into newly created open space. A fundamental control on ore deposition was the pronounced adiabatic cooling of the ore fluids due to their sudden expansion into the fracture and/or breccia systems, thus the importance of structural control on ore deposition in porphyry deposits.

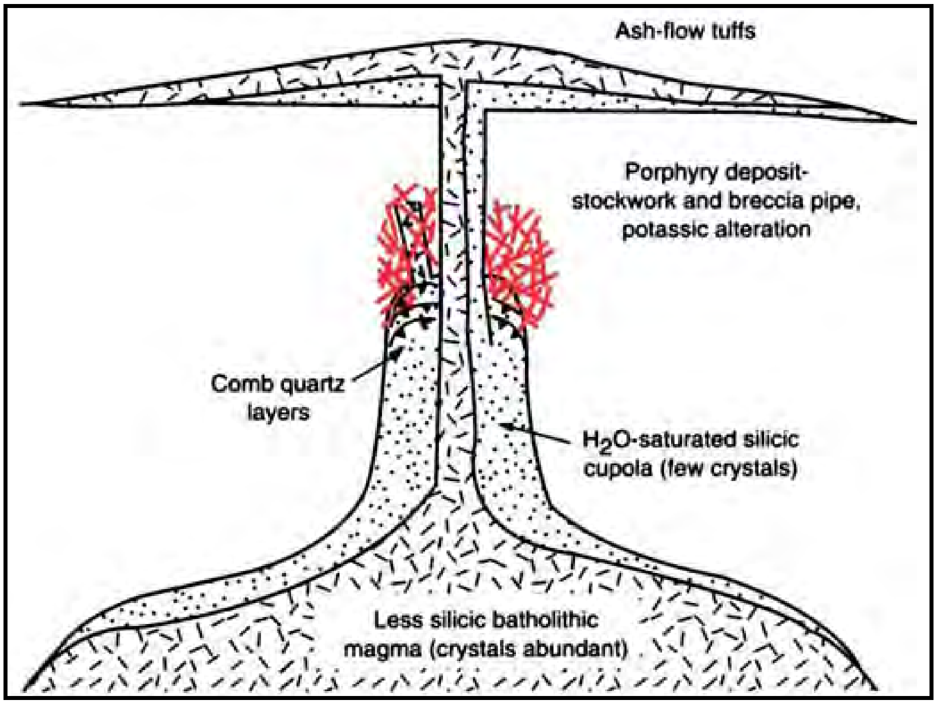
Some modification of the orthomagmatic model is likely required for at least some, if not most, porphyry deposits, in view of studies by Shannon et al. (1982), Carten et al. (1988a), and Kirkham and Sinclair (1988). These authors concluded that, in several deposits, the underlying genetically related intrusions were largely liquid in their carapaces until ore formation was essentially complete. According to this model, volatiles that streamed through large volumes of magma, stripping it of its metal content, accumulated in small cupolas at the top of the magma chambers. Wall rocks of the intrusions and deposits are not considered to be viable sources for the metals in porphyry deposits.

Figure 8.1 Schematic Diagram of a Porphyry Copper System



Note: The diagram illustrates the root zone of an andesitic stratovolcano showing mineral zonation and possible relationship to skarn, manto, "mesothermal" or "intermediate" precious metal, base metal vein and replacement, and epithermal precious-metal deposits.

Figure 8.2 Schematic Diagram of a Crystallizing Batholithic Mass



Note: The diagram shows an overlying volatile-saturated cupola and related ash-flow tuffs illustrating the environment of formation of porphyry deposits (modified from Kirkham and Sinclair, 1988).

9.0 MINERALIZATION

The gold mineralization at the Snowfield deposit is hosted by schistose, pervasively altered (quartz-sericite-chlorite) volcanic and volcaniclastics that contain 1% to 5% disseminated pyrite, minor disseminations, veinlets of tourmaline, molybdenite, and abundant younger calcite veinlets.

Gold mineralization occurs as microscopic grains (less than 30 µm) of electrum that are encased within fine-grained, pervasively disseminated pyrite in close association with trace amounts of galena and sphalerite (Margolis, 1993). Other associated minerals within the gold-mineralized zone include: tetrahedrite-tennantite, barite, acanthite, minor Mn-rich calcite, and rare chalcopyrite. Minute clusters, approximately 75 µm, of pyrite and rutile (+ barite) are also observed within the gold-bearing mineralization (Margolis, 1993).

Molybdenite mineralization appears to have been emplaced during an earlier hydrothermal event. Pyrite-tetrahedrite veinlets from the gold-bearing mineral assemblage are observed cutting molybdenite veinlets. Weakly disseminated and minor fracture filling molybdenite mineralization is widespread and common throughout the Snowfield Deposit and nearby area. Fine-grained tourmaline crystals are often associated with molybdenite in quartz veinlets (Margolis 1993).

Hydrothermal alteration within the Snowfield Deposit includes quartz-sericite-pyrite with varying amounts of chlorite, calcite, and garnet. The dark reddish-brown, rounded garnets are less than 7 mm and appear to have been crystallized during the gold mineralizing event(s). They are probably of hydrothermal origin as they are well fractured and exhibit deformational features consistent with the tectonic event that caused the deformation, alteration, and schistosity of the host rocks (Margolis, 1993).

Chalcopyrite mineralization with minor sphalerite and galena increases at depth coincident with a change in lithology from the medium-grained andesitic tuffs to fine-grained ash-crystal-lithic tuffs (McCrea, 2007). Increasing base metal mineralization with depth may indicate possible porphyry-style copper mineralization associated with the cupola of a buried alkalic intrusion (Margolis, 1993).

10.0 EXPLORATION

There was no other exploration work undertaken on the Snowfield property in 2009 apart from diamond drilling, which is described in detail in Section 11.0.

11.0 DRILLING

For a complete account of diamond drilling prior to the 2008 program, the reader is referred to the technical report on the Snowfield property (J.D. Blanchflower, 2008).

11.1 2008 Snowfield Diamond Drill Program

At the end of the 2007 field season, Silver Standard had completed 29 NQ-2 size diamond drill holes, totalling 8,666 m. Twenty-one drill holes tested the Snowfield zone, six drill holes tested the nearby Coffeepot zone, and one drill hole tested the Mitchell East zone (now recognized to be the northern extension of the Snowfield zone). The focus of the Snowfield zone drilling was to test the lateral limits of the gold-molybdenum mineralization and infill drill hole spacing for mineral resource estimation.

The most significant result from the 2007 exploration drilling was the discovery of the northern extension of the Snowfield zone on trend with Seabridge's Mitchell copper-gold deposit, which is situated immediately east of and contiguous to the Snowfield property and south of Mitchell Creek. The one drill hole that was targeted in this area, (MZ-001) intersected 259 m of 0.71 g/t Au and 0.14% copper. The hole ended in mineralization with the bottom 31 m grading 1.38 g/t Au and 0.31% copper.

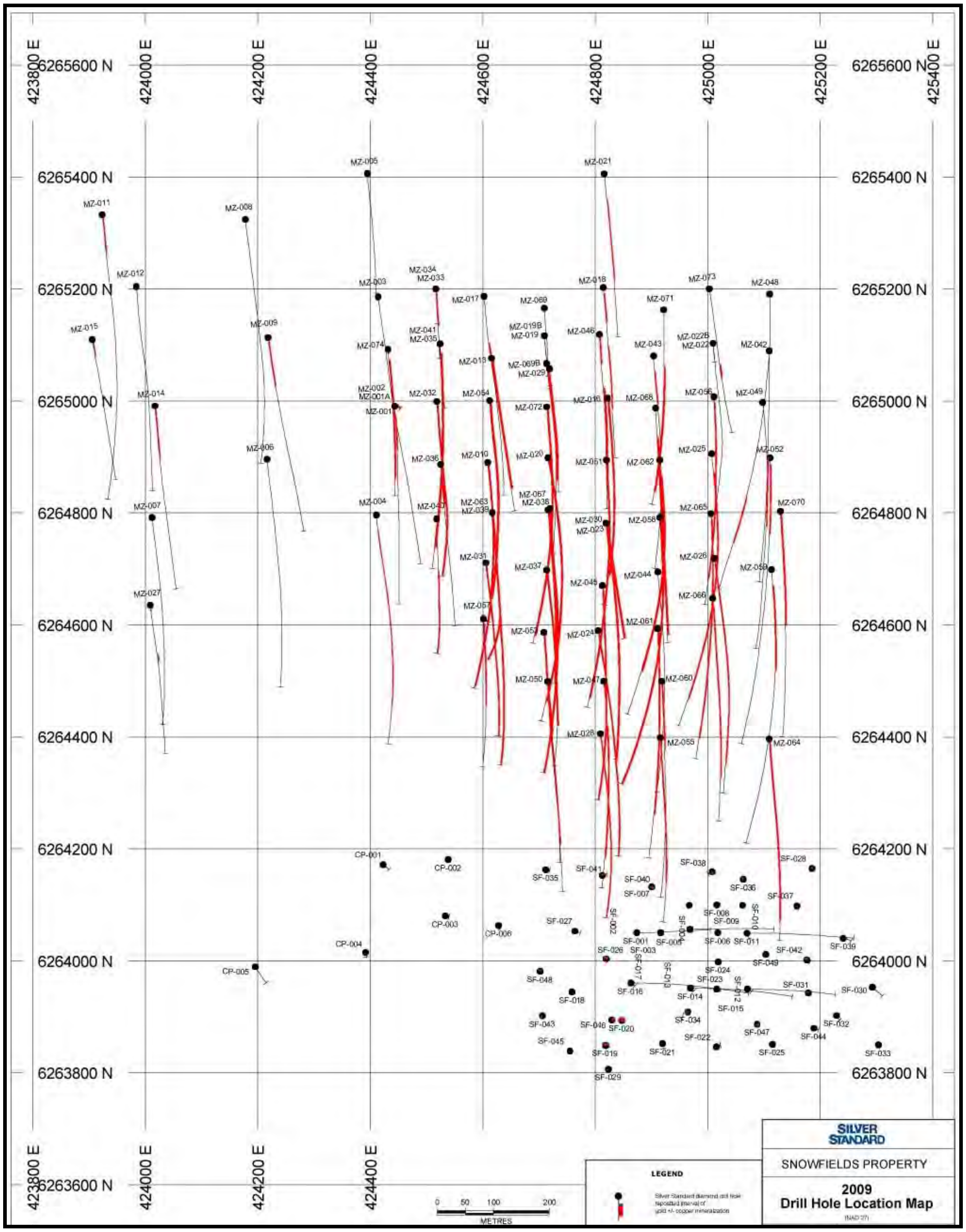
The 2008 drill program consisted of 6,945 m of drilling in 31 holes. Down-hole, E-Z shot surveying of all holes showed that deviation on azimuths was a maximum of 15° for a 700 m long hole, with little movement on dip. Core recovery was excellent at ±95%.

Drill hole collars were surveyed toward the end of the drilling campaign by McElhanney Consulting Services using a differential GPS.

11.2 2009 Snowfield Diamond Drill Program

The 2009 Snowfield drill program which included 23,778 m in 42 drill holes increased the drill density to 100 m centres in the main body of the inferred resources outlined in 2008, and extended the known mineralization to the northwest and southeast. A higher grade gold-copper core with silver and molybdenum credits was defined, and continuity of grade in the northern half of the zone was proven. A plan map of the drill hole locations is shown in Figure 11.1. The drilling and surveying contractors remained the same for the 2008 program.

Figure 11.1 Surface Drill Hole Plan Showing 2009 Drilling



12.0 SAMPLING METHOD AND APPROACH

At the end of each drill shift all core was transported by helicopter to the handling, logging, and storage facility on site. Prior to any geotechnical and geological logging, the entire drill core was photographed in detail with the digital colour photographic images for each interval of core filed with the digital geological logs.

A trained geo-technician recorded the core recovery and rock quality data for each measured drill run. All lithological, structural, alteration, and mineralogical features of the drill core were observed and recorded during the geological logging procedure. This information was later transcribed into the computer using a program that was compatible with Gemcom software.

The geologist responsible for logging assigned drill core sample intervals with the criteria that the intervals did not cross geologic contacts and the maximum sample length was 2 m. Within any geologic unit, sample intervals of 1.5 m long could be extended or reduced to coincide with any geologic contact. Sample lengths were rarely greater than 2 m or less than 0.5 m, averaging 1.52 m long.

Upon completion of the geological logging, the samples were sawn in half lengthwise. One-half of the drill core was placed in a plastic sample bag and the other half was returned to its original position in the core box. The sample bags were consolidated into larger shipping containers and delivered to the assay laboratory.

It is the author's opinion that the core logging procedures employed are thorough and provide sufficient geotechnical and geological information. There is no apparent drilling or recovery factor that would materially impact the accuracy and reliability of the drilling results.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The 2009 program used ALS Chemex Lab Ltd. (ALS Chemex) as the principal laboratory, with approximately 5% to 10% of pulps forwarded to Assayers Canada Ltd. (Assayers Canada) in Vancouver, BC for secondary checks.

The samples that were originally sent to ALS Chemex in Terrace, BC, for sample preparation were then forwarded to the ALS Chemex facility in Vancouver, BC, for analysis.

13.1 ALS CHEMEX LABORATORY

ALS Chemex is an internationally recognized minerals testing laboratory operating in 16 countries and has an ISO 9001:2000 certification. The laboratory in Vancouver has also been accredited to ISO 17025 standards for specific laboratory procedures by the Standards Council of Canada (SCC).

Samples at ALS Chemex were crushed to 70% passing 2 mm. Samples were riffle split and 1,000 g were pulverized to 85% passing 75 µm. The remaining coarse, reject material was returned to Silver Standard for storage in their Smithers warehouse for possible future use.

Gold was determined using fire assay on a 30 g aliquot with an AA finish. Copper was determined using four acid digest with either ICP-AES or AA analysis. In addition, a 33 element package was completed using a four acid digest and ICP-AES analysis, which included the silver and molybdenum.

13.2 ASSAYERS CANADA

Assayers Canada has consistently achieved Certificates of Laboratory Proficiency from the SCC for precious and base metal analysis. The laboratory is steadily working towards ISO 17025 Certification (the new ISO standard specifically for testing and calibration laboratories).

Samples at Assayers Canada were crushed to 60% passing 2 mm. They were riffle split and 250 g was pulverized to 90% passing 150 mesh (approximately 95 µm).

Gold was determined using fire assay on a 30 g aliquot with an AA finish. Copper was determined using four acid digest with either ICP-AES or AA analysis.

It is the author's opinion that the sample preparation, security, and analytical procedures are satisfactory.

14.0 DATA VERIFICATION

14.1 SITE VISIT AND INDEPENDENT SAMPLING 2009

The Snowfield property was visited by Mr. Fred Brown, CPG, Pr.Sci.Nat., from September 9 to 13, 2009. Independent verification sampling was done on diamond drill core, with four samples distributed in four holes collected for assay. An attempt was made to sample intervals from a variety of low and high-grade material. The chosen sample intervals were then sampled by taking quarter splits of the remaining half-split core. The samples were then documented, bagged, and sealed with packing tape and were brought by Mr. Brown to ALS Chemex in Terrace, BC for analysis.

At no time, prior to the time of sampling, were any employees or other associates of Silver Standard advised as to the location or identification of any of the samples to be collected.

A comparison of the P&E independent sample verification results versus the original assay results produced acceptable correlation factors (Armstrong et al., 2009).

14.2 SILVER STANDARD QUALITY CONTROL

The QA/QC program was maintained throughout the 2009 drilling. Certified reference material standards for both copper and gold were purchased from CDN Resource Laboratories Ltd. in Delta, BC. Both of these standards were certified for copper; however, values for gold in both of the standards were provisional only. One standard sample, one blank sample, and one field duplicate sample (1/4 split core) were inserted in every 20 samples. In addition, the lab inserted their own internal QC, which included standards, blanks, and both coarse reject and pulp duplicates.

14.3 2009 DATA VERIFICATION RESULTS

The QC program was monitored on a real-time basis by Silver Standard throughout 2009 and any standards failing the Silver Standard QC protocols were re-run. The author received all the data for the 2009 drilling and verified the performance of the standards, blanks, and duplicates.

14.3.1 Performance Of Certified Reference Material

Both standards performed very well for Au and Cu. In spite of the fact that the Au values were provisional only, the values almost always fell within ±2 standard deviations from the mean. The occasional value falling outside ±3 standard deviations from the mean was flagged by Silver Standard and the work order was re-run. Copper performed extremely well for both standards.

14.3.2 Performance of Blank Material

There were 907 blank samples analyzed during the 2009 program. The author considers that none of the occasional gold or copper failures had any impact on the metal value of the deposit.

14.3.3 2009 Duplicate Statistics

For the 2009 drill program, there were 852 field core duplicate pairs and 567 pulp duplicate pairs graphed for gold and copper. There were no coarse reject duplicates done.

Data for the gold duplicate types were graphed in two different manners. A graph of the sample pair mean versus the Absolute Relative Difference (ARD) of the sample pairs and a Thompson-Howarth (T-H) precision plot were both created.

The gold field duplicates had very good precision, which is not surprising for a porphyry deposit. The ARD demonstrated a precision of 10% for the core duplicates and the T-H yielded a precision of 18%.

The pulp duplicate pairs yielded an ARD value of 5% and a T-H precision value of 6.7% for gold.

The copper field duplicates yielded a T-H precision of 10%, and the copper pulp duplicates yielded a T-H precision of 4%.

14.3.4 External Checks at Assayers Canada

Silver Standard sent between 5% to 10% of pulps to Assayers Canada for checks on gold and copper. All check samples were graphed with a simple scatter graph, and apart from a very rare outlier, the pairs fell along a 1:1 line.

The author considers that the data used in this resource estimate are of excellent quality.

15.0 ADJACENT PROPERTIES

Within the adjacent KSM property there are three notable copper-gold mineral deposits, namely the Kerr, Mitchell, and Sulphurets. All of these occurrences are situated within the claim holdings currently owned and operated by Seabridge.

Seabridge acquired the property from Placer Dome in June 2000. In 2009, Resource Modeling Inc. (Resource Modeling) completed updated NI 43-101 compliant resource estimates for the Kerr, Sulphurets, and Mitchell zones. The Mitchell Resource was reported in a news release dated March 11, 2009, and the Kerr and Sulphurets Resources were reported in a March 25, 2009 news release. The current estimated mineral resources for the Mitchell, Kerr, and Sulphurets zones at 0.50 g/t equivalent gold cut-off grades are shown in Table 15.1 and Table 15.2.

In June 2009, an updated PA estimated a 30 year mine life recovering 19.3 M ounces of gold, 5.3 B pounds of copper, 2.8 M ounces of silver, and 1.9 M pounds of molybdenum. In April 2010, Seabridge published the results of a subsequent pre-feasibility study. These results indicate an estimated reserve statement as shown in Table 15.3. All information for this section has been taken from the Seabridge website at www.seabridgegold.net.

The qualified persons for this report have not verified the above information concerning Seabridge, and the information is not necessarily indicative of the mineralization on the Snowfield property.

Table 15.1 Seabridge 2009 Mitchell Resources

	Measured Mineral Resources					Indicated Mineral Resources					Inferred Mineral Resources				
Zone	t (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	t (000)	Au (g/t)	Cu (%)	Au (oz)	Cu) (M lb)	t (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
Mitchell	579,272	0.66	0.18	12,292	2,298	930,603	0.62	0.18	16,287	2,913	514,878	0.51	0.14	8,442	1,589

Note: At 0.5 g/t AuEq cut-off.

Table 15.2 Seabridge 2009 Kerr and Sulphurets Resources

	Measured Mineral Resources					Inferred Mineral Resources				
Zone	t (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	t (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
Kerr	225,300	0.23	0.41	1,666	2,036	69,000	0.18	0.39	405	601
Sulphurets	87,3000	0.72	0.27	2,021	520	160,9000	0.63	0.17	3,259	603
Total	**312,600**	**0.61**	**0.24**	**3,687**	**5,338**	**230,800**	**0.59**	**0.18**	**3,664**	**1,204**

Note: At 0.5 g/t AuEq cut-off.

Table 15.3 Seabridge 2010 KSM Proven and Probable Reserves

			In Situ Average Grades				Contained Metal			
Zone	Reserve	Mt	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Au (M oz)	Cu (M lb)	Ag (M oz)	Mo (M lb)
Mitchell	Proven	570.6	0.64	0.17	2.95	58.0	11.7	2,101	54.1	73.0
	Probable	764.8	0.59	0.16	2.93	62.3	14.5	2,722	72.0	105.0
	Total	**1,335.4**	**0.61**	**0.16**	**2.93**	**60.4**	**26.3**	**4,823**	**126.1**	**178.0**
Sulphurets	Probable	142.2	0.61	0.28	0.44	101.8	2.8	883	2.0	31.9
Kerr	Probable	125.1	0.28	0.48	1.26	Nil	1.1	1,319	5.1	Nil
Totals	Proven	570.6	0.64	0.17	2.95	58.0	11.7	2,101	54.1	73.0
	Probable	1,032.1	0.56	0.22	2.38	60.2	18.4	4,924	79.1	137.0
	Total	**1,602.7**	**0.59**	**0.20**	**2.58**	**59.4**	**30.2**	**7,024**	**133.1**	**209.9**

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 METALLURGICAL TESTWORK REVIEW

16.1.1 INTRODUCTION

Preliminary metallurgical testwork, investigating the Snowfield mineralization metallurgical performances, has been carried out since early 2009 by Process Research Associates Ltd. (PRA), a Metallurgical Division of Inspectorate America Corporation. PRA is an industrial research laboratory established in 1992 and specializes in metallurgical process development and research, from bench scale testing to pilot plant testing. The chemical analysis of the metallurgical test samples were conducted by International Plasma Labs (IPL), a geochemical laboratory of Inspectorate America Corporation. IPL is an ISO 9001:2000 certified company. The testwork was conducted under the supervision of Frank Wright, P.Eng.

The testing program consisted of preliminary mineralization characteristic determination, copper/gold/molybdenum bulk flotation, gold bearing pyrite flotation, gold cyanide leach, and related ancillary testing of individual drill core interval samples and composite samples collected from the North and the Upper zones. The test results and procedures, including sample preparation and analysis, are presented in two data reports by PRA released in March 2010.

Wardrop has reviewed the testwork data reports and summarized the results in the following sections.

16.1.2 SAMPLE DESCRIPTION

The metallurgical samples were collected from two major mineralization zones, North zone (main zone) and Upper (molybdenum) zone. The samples used for the metallurgical testing program were collected from the following drill holes: MZ 013, MZ 016, MZ 030, MZ 031, SF 002, SF 004, SF 013, SF 016, and SF 023. The drill hole distribution is presented in Section 11.0, shown in Figure 11.1.

NORTH (MAIN) ZONE SAMPLES

A total of approximately 367 kg drill core samples were collected from four drill holes from the North (main) zone. Six dill core interval samples identified as MZ 13A, MZ 13B, MZ 16B, MZ 30B, MZ 31A, and MZ 31B were prepared from the drill core samples for the preliminary metallurgical testing program. Table 16.1 lists the sample identification, drill hole identification, and drill core interval.

Table 16.1 North Zone Drill Core Interval Sample Information

Sample ID	Drill Hole ID	Depth (m)	
		From	To
MZ 13A	MZ-013	75	105
MZ 13B	MZ-013	250	281
MZ 16A	MZ-016	110	140
MZ 16B	MZ-016	350	380
MZ 30A	MZ-030	219	249
MZ 30B	MZ-030	390	420
MZ 31A	MZ-031	120	150
MZ 31B	MZ-031	310	340

Four composite samples were constructed for the preliminary metallurgical testwork. Composite 1 (Comp 1) was composed from samples MZ 13B and MZ 16B while Composite 2 (Comp 2) was from samples MZ 13A, MZ 30B, and MZ 31A. The distributions are based on calculated sample weight. The blending ratios are shown in Table 16.2.

Table 16.2 Blend Sample Composition

Sample ID	Weight (kg)	Distribution
Comp 1		
MZ 13B	8.0	50.0%
MZ 16B	8.0	50.0%
Total	**16.0**	**100.0%**
Comp 2		
MZ 13A	6.0	33.3%
MZ 30B	6.0	33.3%
MZ 31A	6.0	33.3%
Total	**18.0**	**100.0%**
Comp 4		
MZ 13B	21.2	63.9%
MZ 16B	12.0	36.1%
Total	**33.2**	**100.0%**
Comp 5		
MZ 13A	14.1	18.7%
MZ 30B	11.5	15.2%
MZ 31A	23.8	31.4%
MZ 31B	26.0	34.5%
Total	**75.3**	**100.0%**

UPPER (MOLYBDENUM) ZONE SAMPLES

A total of 609 kg drill core samples were collected from the Upper (molybdenum) zone. Five different drill core interval samples were prepared and labelled as SF 02, SF 04, SF 13, SF 16, and SF 23.

A composite sample, labelled as Comp 3, was also prepared by blending four drill core interval samples of SF 02, SF 04, SF 13, and SF16 at the same weight ratio.

16.1.3 SAMPLE HEAD ANALYSES

The results of multiple elemental analyses for the drill core interval samples indicate that the contents of the main value elements, copper (Cu), gold (Au), and molybdenum (Mo) vary significantly from sample to sample.

For the North zone samples, gold grade varied from 0.57 g/t to 1.41 g/t; copper grade ranged between 0.11% and 0.35%; and molybdenum contents fluctuated from 12 ppm to 92 ppm. The drill core interval samples from the Upper zone were found to contain between 0.42 g/t to 2.68 g/t Au, 0.04% to 0.14% Cu, and 104 ppm to 178 ppm Mo. The assay data also indicate that the mineralization contains approximately 0.394 ppm to 1.268 ppm rhenium (Re).

The key assay results for each composite sample are shown in Table 16.3.

Table 16.3 Metal and Sulphur Concentrations of the Blended Samples

Sample ID	Au (g/t)	Cu (%)	Mo (ppm)	S (T) (%)	Fe (%)	Re (ppm)	Ag* (ppm)	Sb (%)	As (%)	Hg (ppm)	Pb* (ppm)	Zn*
Comp 1	1.26	0.26*	11*	5.1	4.9	N/A	<0.5	9*	87*	<3*	40	139
Comp 2	0.69	0.12*	64*	2.9	4.3	N/A	<0.5	<5*	16*	<3*	24	152
Comp 3	2.44	0.03	181	2.9	4.8	0.79	0.9	0.001	0.01	1.6	9	529
Comp 4	0.67	0.16	101	2.4	4.1	0.41	1.6	<0.001	0.001	1.0	3	118
Comp 5	0.90	0.14	65	3.8	5.2	0.39	<0.5	0.001	0.006	0.8	19	228

*ICPM data.

16.1.4 GRINDABILITY TESTWORK

PRA conducted preliminary grindability testwork to determine the bond ball mill work index on each of the samples from the North zone. Table 16.4 presents the work index. It appears that on average, the mineralization is moderately hard.

Table 16.4 Bond Ball Mill Work Index – North Zone

Sample ID	Bond Ball Mill Work Index (kWh/t)
MZ13A	13.6
MZ 16B	16.6
MZ 30B	15.7
MZ 31A	17.0
MZ 31B	17.2

16.1.5 SAMPLE SPECIFIC GRAVITY

Drill core interval samples from both mineralization zones were tested for specific gravity (SG). The results are shown in Table 16.5. The SG was found to be between 2.76 and 2.82 for the North zone samples and between 2.76 and 2.91 for the Upper zone samples.

Table 16.5 Specific Gravity – North and Upper Zones

Sample ID	SG
North Zone	
MZ13A	2.82
MZ 13B	2.76
MZ 16B	2.81
MZ 30B	2.79
MZ 31A	2.77
MZ 31B	2.78
Upper Zone	
SF 02	2.84
SF 04	2.91
SF 13	2.84
SF 16	2.85
SF 23	2.76

16.1.6 FLOTATION TESTWORK

Preliminary flotation testwork was performed on the drill core interval samples and on the blended composite samples. The flotation testwork investigated:

- copper/gold/molybdenum bulk rougher flotation kinetic characteristic
- primary grinding particle size
- cleaner flotation
- gold bearing pyrite flotation.

Two locked cycle flotation tests were also conducted on the composites (Comp 3 and Comp 4) generated from the two major mineralization zones.

FLOTATION TESTWORK ON DRILL CORE INTERVAL SAMPLES

The testing included a copper/gold/molybdenum flotation (bulk flotation) consisting of rougher flotation, rougher concentrate regrinding and subsequent cleaner flotation, and a gold bearing pyrite flotation including rougher/scavenge flotation, pyrite concentrate regrinding, and cleaner flotation.

The target primary grinding particle size was set at 80% passing 74 µm. The regrind of the rougher concentrate and the rougher scavenger concentrate was completed in a ceramic mill.

The collectors used in the bulk flotation circuit consisted of 3418A (mainly dialkyl dithiophosphinates), A208 (mainly dithiophosphates), and potassium amyl xanthate (PAX). The pyrite flotation used PAX, together with copper sulphate ($CuSO_4$), to float gold bearing pyrite. Methyl isobutyl carbinol (MIBC) was used as frother for both the flotation circuits.

Bulk Rougher Flotation

Table 16.6 shows the metal recoveries to the bulk rougher concentrates from both mineralization zones.

Table 16.6 Recoveries to Bulk Rougher Concentrate

Sample ID	Metal Distribution (%) Au	Cu	Mo	Fe	Weight Recovery (%)
North Zone					
MZ 13A	76	81	n/a	n/a	7.2
MZ 13B	87	88	39	82	8.8
MZ 16B	84	81	41	85	12.1
MZ 30B	86	89	71	85	9.7
MZ 31A	69	76	82	40	5.0
MZ 31B	68	78	80	35	4.3
Upper Zone					
SF 02	63	64	64	30	5.1
SF 04	51	59	66	40	5.7
SF 04*	46	51	73	27	4.6
SF 13	58	54	63	23	4.0
SF 16	68	73	71	N/A	8.5
SF 23	69	69	80	N/A	5.5

*Test on SF 04 using collector 3926A instead of 3418A.

At the tested conditions, the samples from the North zone produced higher gold and copper recoveries to the bulk concentrate compared with the samples from the Upper zone. Approximately 76% to 89% of the copper and 68% to 87 % of the gold were recovered into the bulk rougher concentrate from the North zone samples. These recoveries were only approximately 53% to 73% for copper and 51% to 69% for gold for the Upper zone samples. The lower copper and gold recoveries from the Upper zone samples are possibly due to much lower copper head grades, excluding SF23 sample. In addition, molybdenum recovery to the bulk concentrate varied from 39% to 82% for the North zone samples and from 63% to 80% for the North zone samples.

Gold Bearing Pyrite Rougher Flotation

After the bulk flotation, the bulk flotation tailings were further floated to recover the gold associated with pyrite. The gold, copper, and molybdenum recoveries to the gold bearing pyrite concentrate are listed in Table 16.7.

Table 16.7 Metal Recoveries to Gold Bearing Pyrite Concentrates

Sample ID	Metal Distribution (%)			Weight Recovery (%)
	Au	Cu	Mo	
North Zone				
MZ 13A	6.7	5.7	N/A	6.4
MZ 13B	5.6	8.2	23.2	2.4
MZ 16B	9.5	13.7	23.5	3.3
MZ 30B	4.1	6.2	12.3	3.0
MZ 31A	9.6	11.0	5.1	3.0
MZ 31B	9.8	8.9	5.5	2.9
Upper Zone				
SF 02	17.8	17.1	13.9	6.7
SF 04 (1)	22.9	22.2	19.3	5.7
SF 04 (2)*	28.5	22.4	11.9	6.8
SF 13	18.7	16.8	21.5	4.6
SF 16	11.2	11.6	12.2	6.2
SF 23	9.9	8.4	7.6	3.8

*Test on SF 04(2) using collector 3926A instead of 3418A.

Approximately 5% to 29% of the gold in the mineral samples were further recovered into the gold bearing pyrite concentrate. The gold recoveries from the Upper zone samples to the concentrate were significantly higher than the samples from the North zone.

Cleaner Flotation

The bulk rougher concentrate was reground and upgraded by cleaner flotation. The copper grades of the final cleaner concentrates are listed in Table 16.8.

Table 16.8 Metal Grades of Cleaner Concentrate

Sample ID	Metal Grade			
	Au (g/t)	Cu (%)	Mo (%)	Re (ppm)
North Zone				
MZ 13A	44.8	15.6	N/A	N/A
MZ 13B	43.1	16.1	0.02	1.33
MZ 16B	58.5	9.5	0.07	4.20
MZ 30B	43.5	14.3	0.82	40.2
MZ 31A	125.5	28.9	3.1	129
MZ 31B	65.3	28.9	1.61	105
Upper Zone				
SF 02	351.8	13.3	4.3	294.5
SF 04 (Test F2)	450.3	10.4	3.9	186.7
SF 04 (Test F6)*	539.5	10.4	8.9	68.1
SF 13	860.7	15.3	6.3	614.8

*Test on SF 04(2) using collector 3926A instead of 3418A.

The data show that most of the cleaner concentrates contained less than 20% Cu, except for the samples MZ 31A and MZ 31B, which produced the concentrates having approximately 29% Cu. The Upper zone samples produced much lower copper grade concentrates compared to the North zone samples however, gold, molybdenum, and rhenium grades were much higher.

KINETIC FLOTATION ON COMPOSITE SAMPLES

Bulk rougher kinetic flotation tests with a retention time up to 12 min were carried out on the composite samples Comp 1, Comp 2, Comp 3, and Comp 4 at various primary grinding particle sizes. The test results are plotted in Figure 16.1, Figure 16.2, Figure 16.3, and Figure 16.4.

Figure 16.1 Flotation Kinetics at Varied Primary Grinding Particle Size - Comp 1

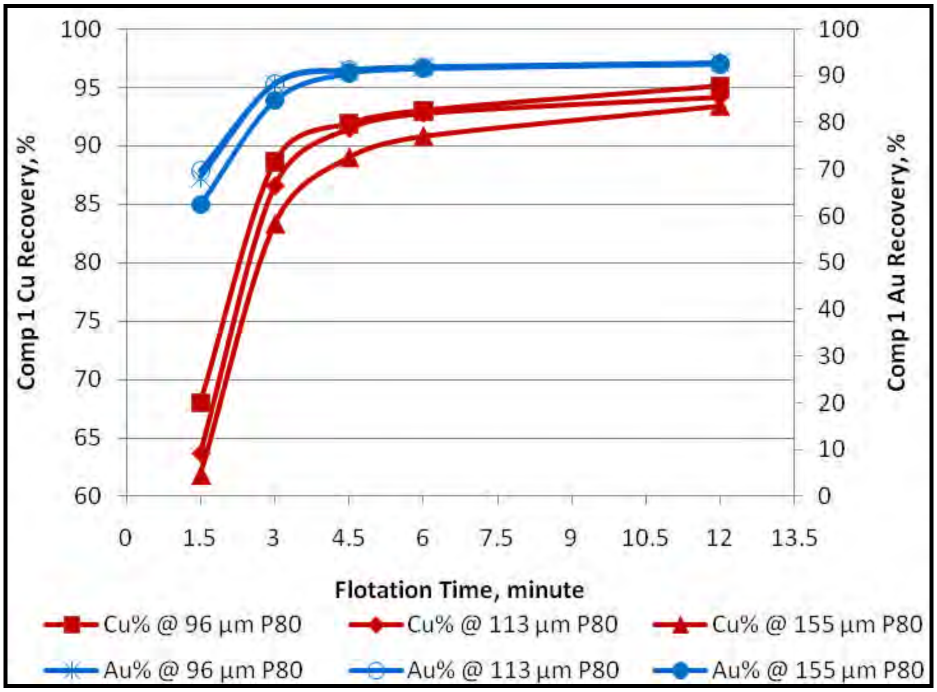


Figure 16.2 Flotation Kinetics at Varied Primary Grinding Particle Size - Comp 2

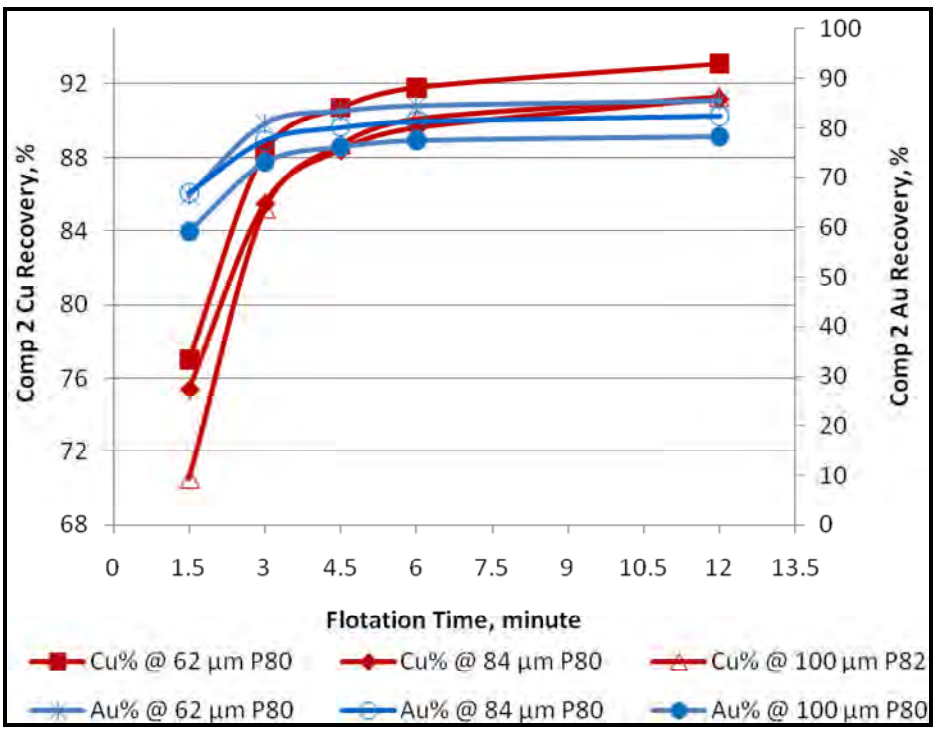


Figure 16.3 Flotation Kinetics at Varied Primary Grinding Particle Size - Comp 3

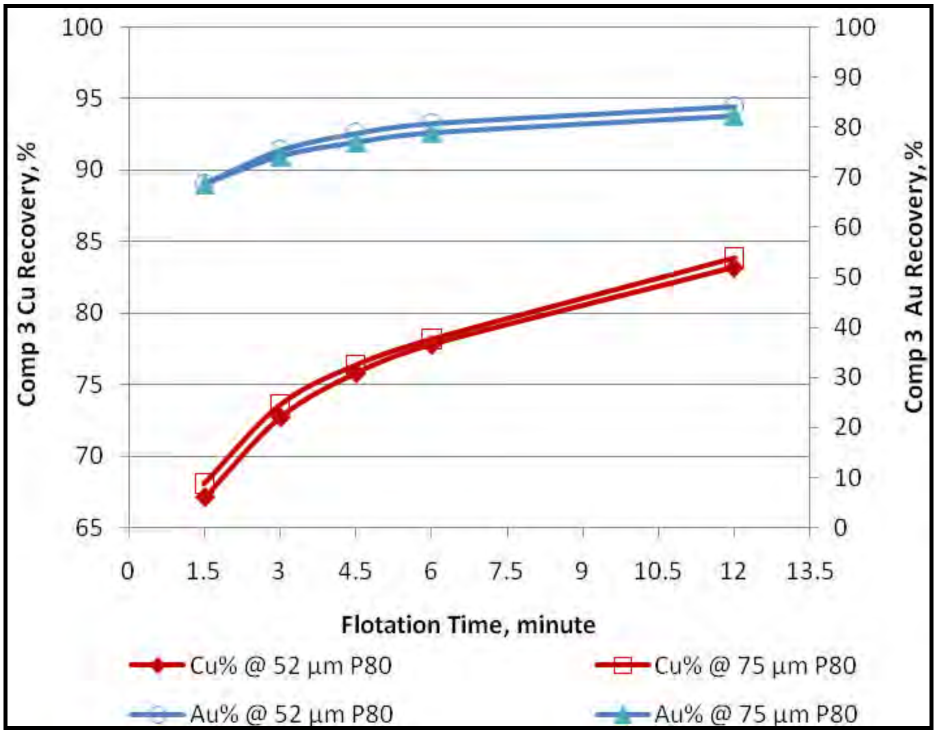
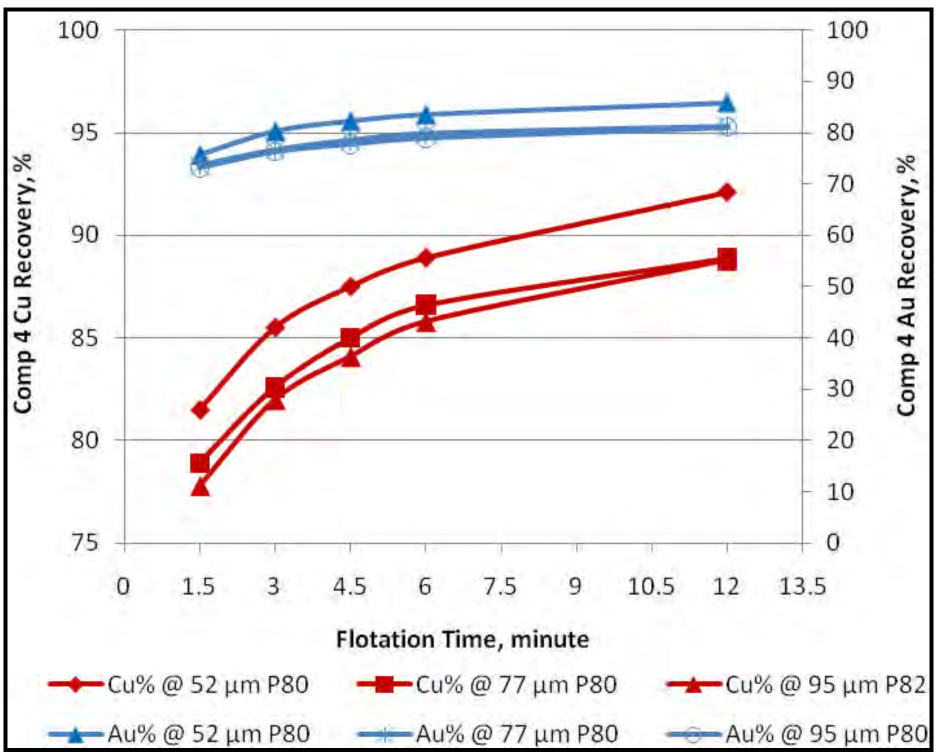


Figure 16.4 Flotation Kinetics at Varied Primary Grinding Particle Size - Comp 4



The test results showed that gold and copper recoveries increased rapidly during the initial three minutes of flotation.

Figure 16.5 to Figure 16.7 highlight the effect of primary grinding particle size on the metal recovery.

Figure 16.5 Copper Recovery vs. Primary Grinding Particle Size

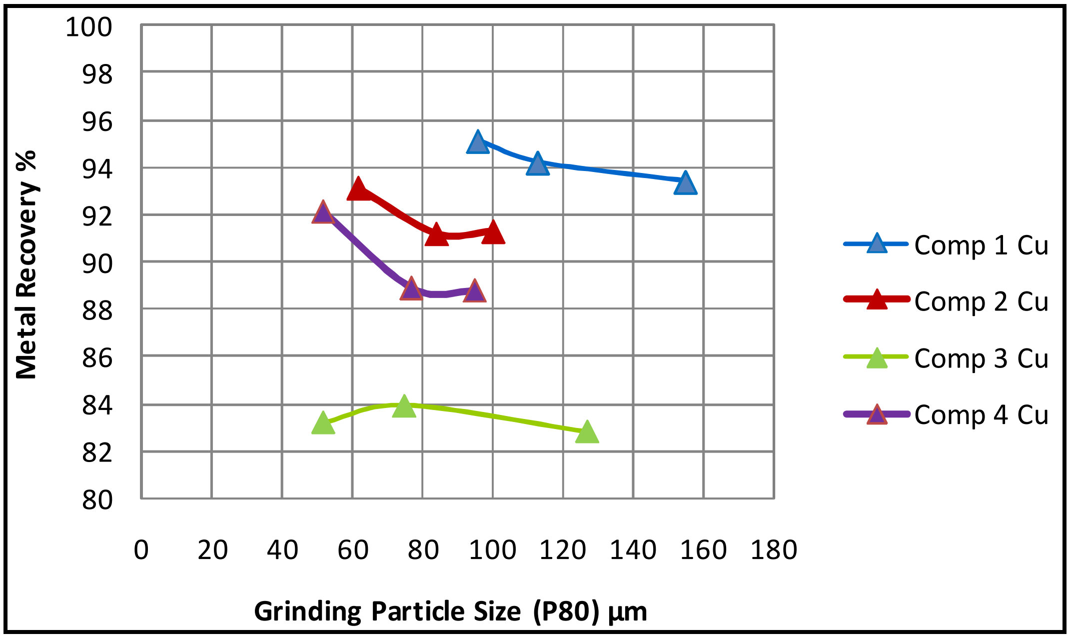


Figure 16.6 Gold Recovery vs. Primary Grinding Particle Size

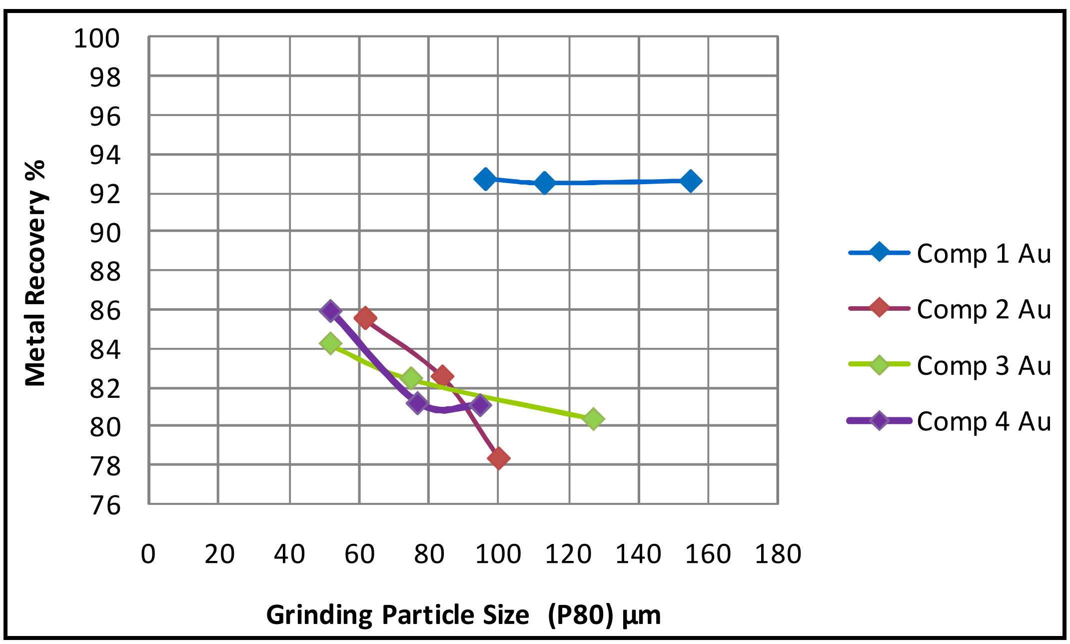
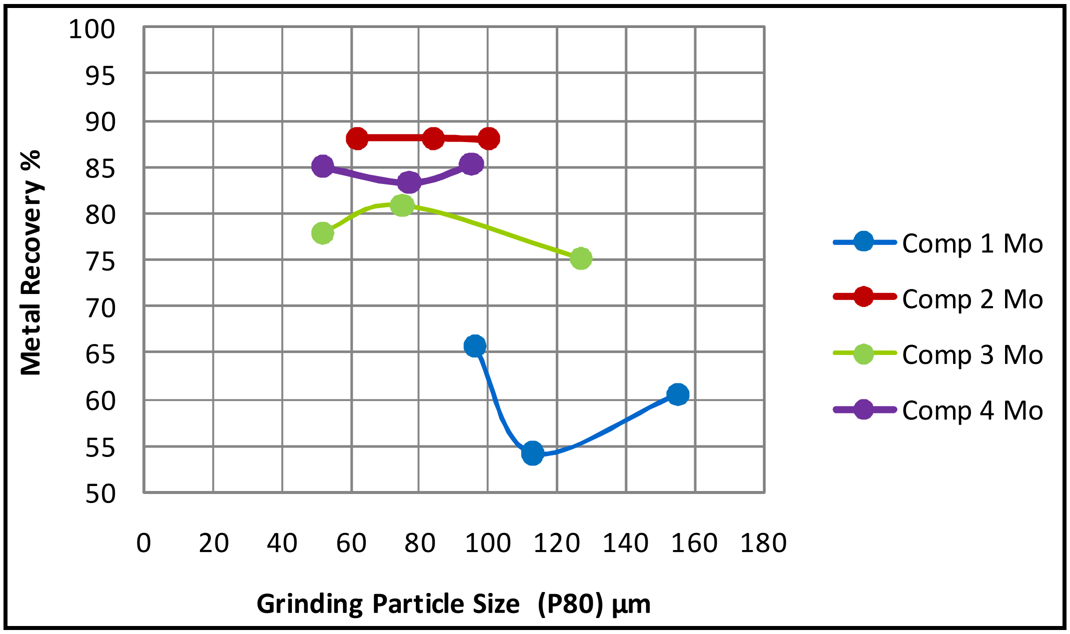


Figure 16.7 Molybdenum Recovery vs. Primary Grind Size



In the tested grinding particle size range, from 80% passing from 52 µm to155 µm, finer primary grinding particle size produced higher metal recoveries for all the samples. The increase in recoveries, however, was different from sample to sample. It appears that gold recovery is more sensitive to the change of primary grinding particle size compared to copper and molybdenum recoveries, in particular for the Comp 2 sample. The Comp 1 sample showed less sensitive to the primary grinding particle size variation. At a coarse primary grinding particle size of 80% passing 155 µm, both the copper and gold recoveries of the samples were more than 92%. Similar to the metallurgical performance of the drill core interval samples from the Upper zone, Comp 3 showed a much lower copper recovery compared with the rest of the composite samples.

CLEANER FLOTATION ON COMPOSITE SAMPLES

PRA further carried out open batch cleaner flotation tests on the various composite samples at various test conditions. The target regrinding particle size for the bulk rougher flotation concentrate was approximately 80% passing 20 µm.

Table 16.9 shows the test results in terms of metal recoveries to bulk rougher concentrates and bulk rougher scavenger concentrates (pyrite concentrates), and final bulk concentrate grades.

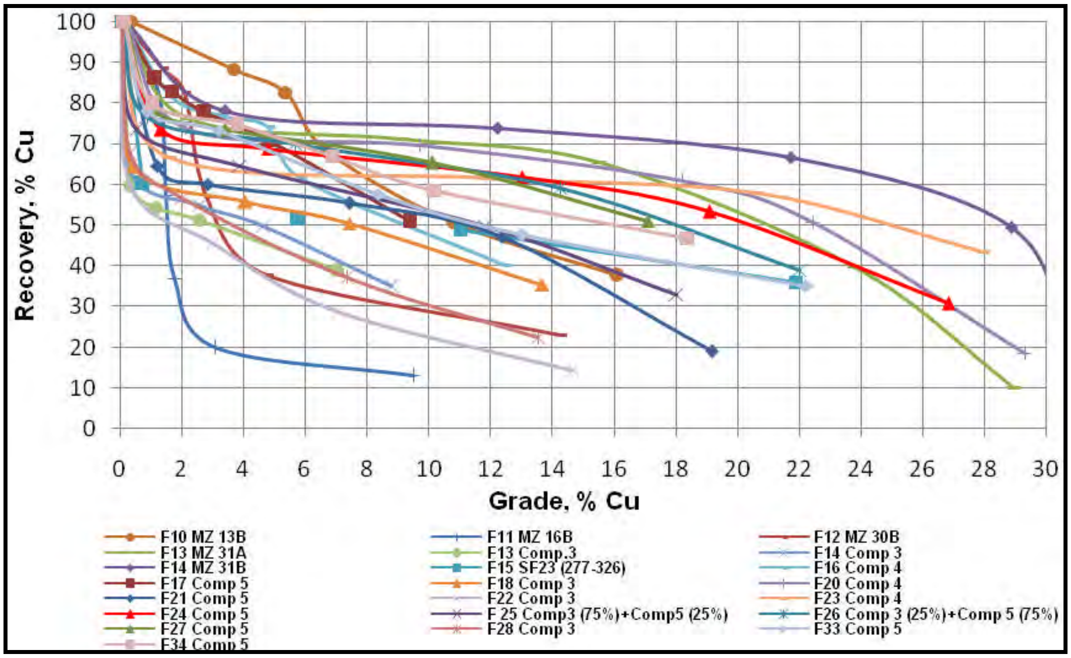
Table 16.9 Metal Recoveries to Different Flotation Concentrates

Sample ID/Primary Grinding Particle Size	Test ID	Metal Recovery to Rougher Concentrate				Metal Recovery to Rougher Scavenger Concentrate				Cleaner Concentrate Metal Grade			
		Au (%)	Cu (%)	Mo (%)	Fe (T) (%)	Au (%)	Cu (%)	Mo (%)	Fe (T) (%)	Au (g/t)	Cu (%)	Mo (%)	Re (ppm)
Comp 3													
52 µm[1]	F13	69.3	59.8	79.0	44.2	6.4	4.6	4.5	4.6	320.4	7.0	3.2	94.8
52 µm	F14	67.4	63.3	79.0	39.1	9.1	6.7	7.8	8.3	468.2	8.8	6.0	204.2
74 µm	F18	62.0	64.4	73.0	45.0	11.2	8.1	11.6	5.1	600.2	13.7	7.3	570.2
125 µm	F22	48.4	59.8	68.9	39.3	20.2	19.0	13.4	13.5	N/A	14.7	15.9	102.9
125 µm[2]	F28	70.3	67.4	74.8	46.6	9.3	14.1	6.3	8.7	986.4	13.5	13.3	963.5
SF 23													
125 µm	F15	68.4	66.6	61.5	46.9	7.5	11.8	4.4	7.1	80.7	27.9	2.1	63.3
Comp 4													
74 µm[1]	F16	78.8	81.7	77.9	49.0	1.4	1.8	2.0	1.9	38.3	12.5	0.8	34.9
74 µm	F20	68.4	73.7	77.8	48.4	10.1	13.1	6.4	8.2	67.5	29.3	1.9	92.9
125 µm	F23	68.4	66.6	61.5	46.9	7.5	11.8	4.4	7.1	80.7	27.9	2.1	879.5
Comp 5													
74 µm[1]	F17	84.5	86.1	82.3	58.6	3.4	4.5	3.9	4.6	43.4	9.4	0.4	19.3
74 µm	F21	68.2	64.6	68.9	49.5	18.2	20.5	12.2	14.1	76.9	19.2	1.1	80.1
120 µm[3]	F33	76.9	77.9	68.5	57.7	8.8	11.2	8.1	8.3	112.9	22.2	1.3	132.3
120 µm[4]	F34	82.2	79.8	79.5	56.8	4.1	9.0	7.3	7.7	91.7	18.4	1.2	
125 µm	F24	75.1	73.4	76.8	51.5	10.4	14.0	7.7	9.8	139.4	26.8	1.7	137.7
125 µm[2]	F27	78.3	78.3	58.0	58.0	7.4	10.7	5.0	7.1	73.2	17.1	0.9	81.2
Comp 3 + Comp 5													
125 µm[5]	F25	73.1	73.7	72.9	49.8	8.3	12.4	6.6	7.8	426.6	18.0	6.1	445.4
125 µm[6]	F26	74.2	76.4	63.7	56.3	8.6	11.8	6.7	7.2	175.6	22.2	2.1	163.4

[1] With 3418A replacing 3926A.
[2] Increased consumption of 3926A and A208, compared with test F22.
[3] Including fuel oil to reduce collector consumption, with 3418A replacing 3926A.
[4] Same as 3 but rougher flotation pH increase from natural to 9.5.
[5] 75% Comp 3 + 25% Comp 5.
[6] 25% Comp 3 + 75% Comp 5.

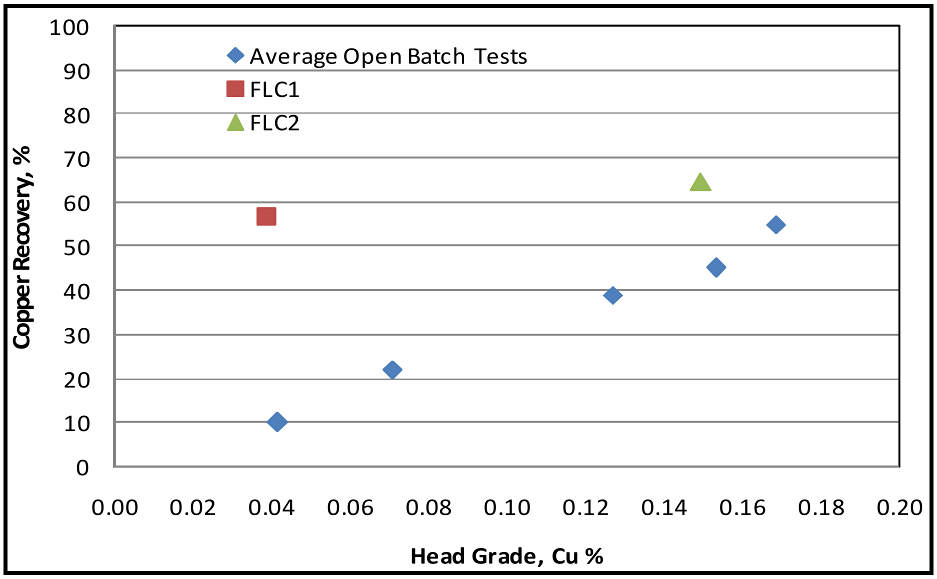
Figure 16.8 presents the relationship between copper grade and copper recovery of the copper/gold/molybdenum bulk concentrate obtained from the drill core interval samples and the composite samples as discussed earlier.

Figure 16.8 Copper Recovery vs. Copper Grade - Open Batch Tests



The data from the open batch flotation tests indicates that six of the samples were able to produce concentrates with grade greater than 22% Cu. Comp 4 sample had the best metallurgical performance, while Comp 3 sample responded poorly to the test conditions. The poor metallurgical responses were possibly caused by a low head copper grade. The projected copper recovery (open batch flotation tests) at the concentrate grade of 22% Cu from the five composite samples, are presented in Figure 16.9. The results show that copper recovery reporting to copper-gold concentrate is closely related to copper head grade.

Figure 16.9 Projected Copper Recovery vs. Head Grade – Open Batch Tests



Note: FLC1: locked cycle test results (Comp 3), concentrate grade 6.4% Cu.

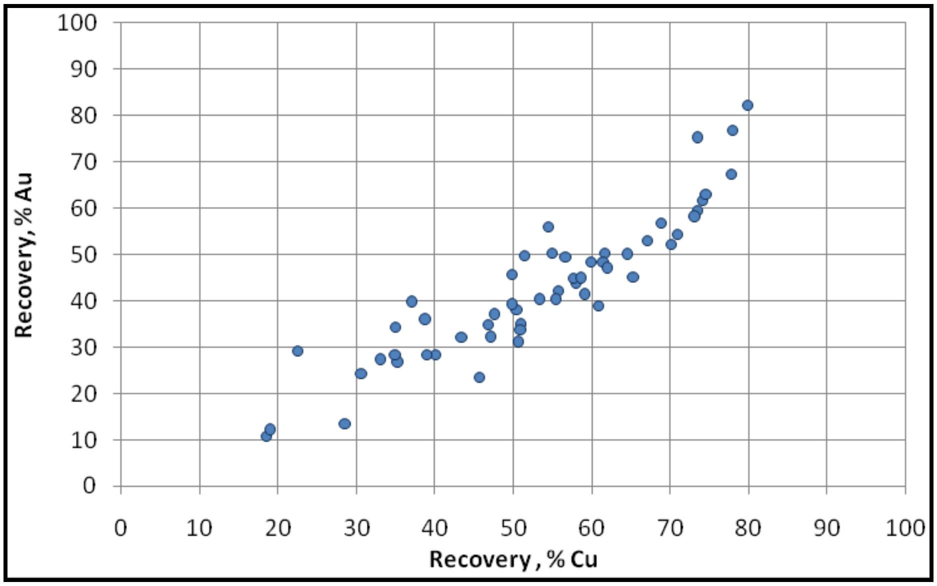
FLC2: locked cycle test results (Comp 4), concentrate grade 25.8% Cu.

The data in

Table 16.9 shows that molybdenum grade of the cleaner concentrates is high, ranging from more than 1% Mo for the Comp 4 and Comp 5 samples to 16% Mo for the Comp 3 sample. On average, approximately 72% of the molybdenum was recovered to the bulk rougher flotation concentrate from the composite samples.

The test results also indicate that most of the gold in the mineralization is closely related with copper minerals. The relationship is illustrated in Figure 16.10.

Figure 16.10 Gold Recovery vs. Copper Recovery – Open Batch Tests



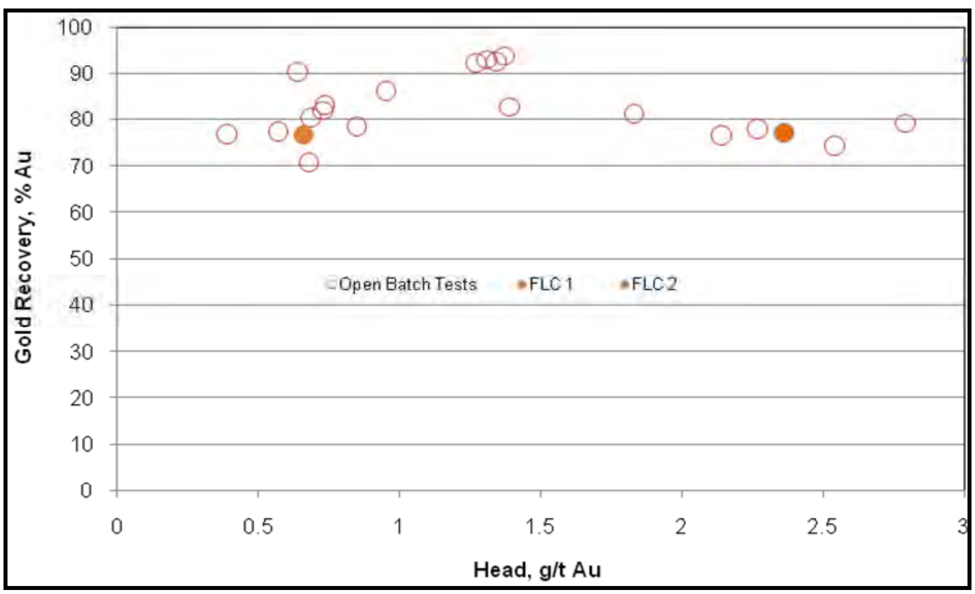
Gold Bearing Pyrite Flotation Test Results

As shown in

Table 16.9, on average, the gold bearing pyrite flotation further recovered approximately 10% of the gold from the head samples.

The total gold recovery by flotation including the bulk concentrate and the pyrite concentrate is presented in Figure 16.11.

Figure 16.11 Total Gold Recovery vs. Head Gold Grade



It appears that the total gold recovery into the flotation concentrates may relate to gold content in head samples, except for the samples containing lower than 1.5 g/t Au. The high gold samples appeared to produce lower gold flotation recoveries. The reasons for the low gold flotation recovery are not clear. It is possibly related to mineralogy.

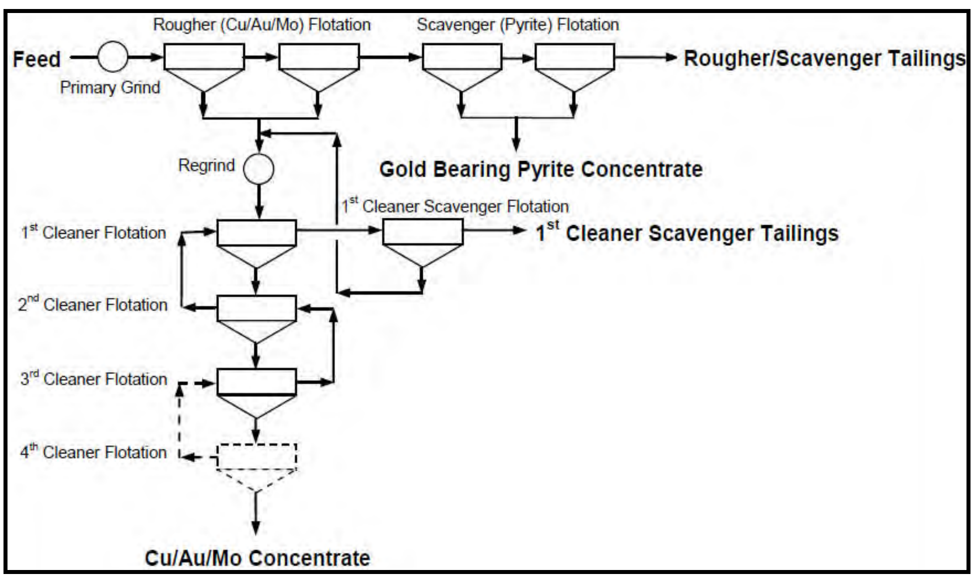
The exploratory tests on the flotation tailings using intensive flotation, including regrinding, appeared to be able to further recover some of the gold which initially lost into the flotation tailings.

LOCKED CYCLE FLOTATION TESTWORK

Two locked cycle flotation tests were carried out on Comp 3 and Comp 4 samples using the same reagent scheme. The flotation flowsheet, as shown in Figure 16.12 included:

- copper/gold/molybdenum bulk rougher flotation
- bulk rougher concentrate regrinding
- reground concentrate cleaner flotation
- gold bearing pyrite flotation.

Figure 16.12 Locked Cycle Test Flowsheet



The targeted primary grinding particle size was 80% passing approximately 125 µm. The targeted bulk rougher concentrate regrinding particle size was 80% passing 20 µm. The test results are listed in Table 16.10.

Table 16.10 Locked Cycle Test Results

Sample ID/Test ID	Recovery Au (%)	Recovery Cu (%)	Recovery Mo (%)	Recovery Ag (%)	Grade Au (g/t)	Grade Cu (%)	Grade Mo (%)	Grade Ag (g/t)	Grade Re* (ppm)
Comp 3/FLC 1									
Calculated Feed	100	100	100	100	2.36	0.04	0.02	1.3	N/A
3rd Cu Concentrate	49.6	56.7	73.0	45.3	341	6.41	4.0	174	313.4
Gold Pyrite Concentrate	8.4	10.0	6.7	8.0	2.84	0.06	0.02	1.5	1.4
1st Cu Cleaner Scavenger Tailings	19.1	13.9	4.5	30.8	5.00	0.06	0.009	4.5	1.3
Rougher Scavenger Tailings	22.9	19.5	15.8	15.9	0.65	0.01	0.003	0.3	<0.1
Comp 4/FLC 2									
Calculated Feed	100	100	100	100	0.66	0.15	0.01	1.7	N/A
4th Cu Concentrate	50.7	64.7	70.7	33.5	88.7	25.8	1.9	154	107.5
Gold Pyrite Concentrate	3.6	8.3	3.1	9.3	0.3	0.2	0.005	2.5	0.3
1st Cu Cleaner Scavenger Tailings	22.3	8.8	3.4	27.7	1.8	0.16	0.004	5.8	0.3
Rougher Scavenger Tailings	23.3	18.2	22.7	29.6	0.18	0.03	0.003	0.6	<0.04

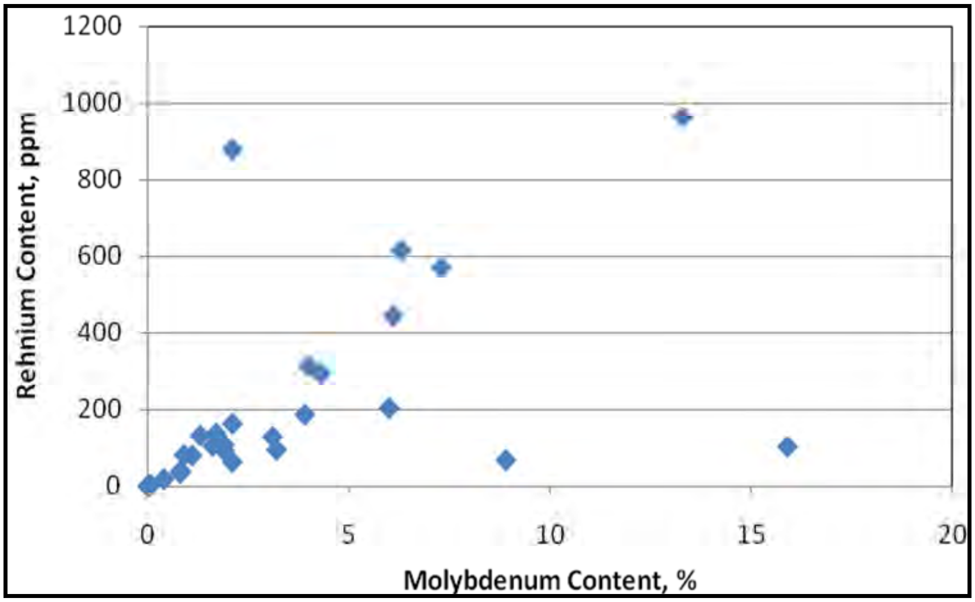
*Average Re grade of cycle 5 and cycle 6.

At the applied test conditions, Comp 4 produced a 25.8% Cu concentrate while the cleaner concentrate from Comp 3 contained only 6.4% Cu. This result is in agreement with the previous open cycle test results.

Approximately 50% of the gold was recovered into the copper/gold/molybdenum bulk concentrate. The total gold reporting the flotation concentrates including gold bearing pyrite concentrate was approximately 77%. Over 70% of the molybdenum was floated with the copper minerals into the bulk concentrate.

It appears that the rhenium in the mineralization was concentrated to the bulk concentrate as well. The close relationship between rhenium and molybdenum, as shown in Figure 16.13, may suggest that rhenium would be able to recover together with molybdenum minerals.

Figure 16.13 Relationship between Rhenium and Molybdenum



Multi-element assay on the concentrates produced from the locked cycle tests are summarized in Table 16.11.

Table 16.11 Multi-Element Assay on Concentrates from Locked Cycle Tests

Element	Unit	FLC 1 Cycle 6/Comp 3	FLC 2 Cycle 6/Comp 4
S	%	42	33.2
Sb	ppm	2,903	<5
As	ppm	15,890	235
Co	ppm	95	25
Cd	ppm	512	<0.2
Bi	ppm	36	<2
Hg	ppm	9.9	1.4
Ni	ppm	275	144
Pb	ppm	2,689	2,907
Zn	ppm	44,587	1,182
Se	ppm	<100	62
Al_2O_3	%	1.80	1.06
BaO	%	0.19	0.05
CaO	%	1.79	0.80
Fe_2O_3	%	41.8	38.9
K_2O	%	0.33	0.25
MgO	%	0.27	0.24
MnO	%	0.05	0.01
Na_2O	%	0.08	0.06
P_2O_5	%	0.02	0.01
SiO_2	%	3.94	3.20
TiO_2	%	0.11	0.17
LOI*	%	28.8	18.7

*Note: Loss on Ignition (LOI).

It appears that the impurity levels of the concentrate produced from Comp 4 are lower than the smelting penalty thresholds set by most smelters. However, arsenic (As), antimony (Sb), mercury (Hg), and zinc (Zn) in the concentrate produced from Comp 3 may attract a smelting penalty.

Further flotation tests (F29 to F32) were performed on the final tailings obtained from the locked cycle test (FLC 1) of Comp 3. Approximately 66% of the gold lost in the flotation tailings of the locked cycle test was recovered by refloating the reground final tailings (80% passing 26 µm).

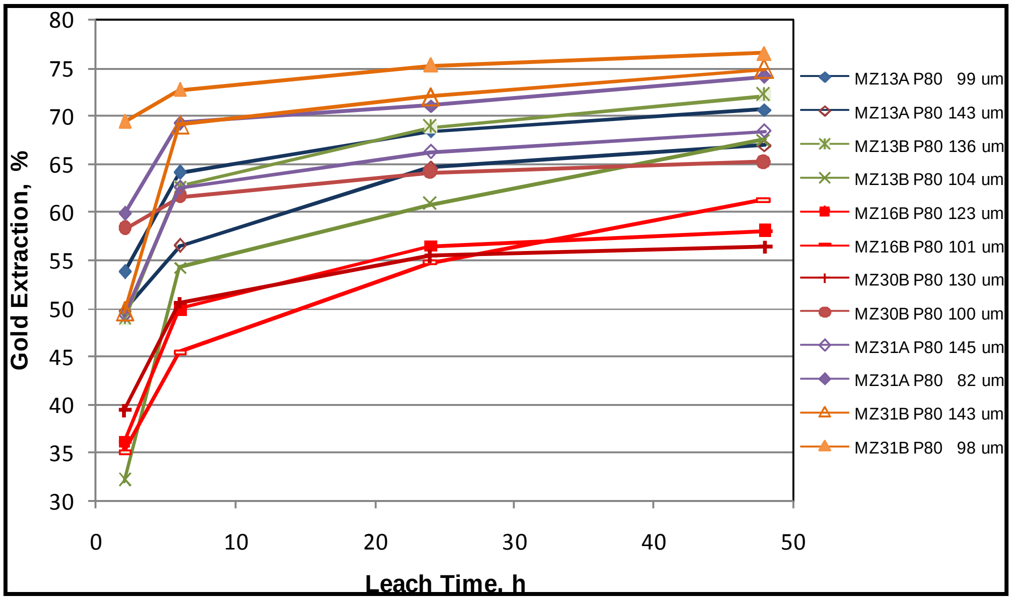
16.1.7 CYANIDE LEACHING TESTS

Preliminary cyanide leaching testwork was performed on head sample and various flotation products, including rougher scavenger flotation tailings, gold bearing pyrite concentrate, and 1st cleaner flotation tailings. All the leaching tests were carried out at a pH value of approximately 10.5.

Cyanide Leaching Tests on Head Samples

The cyanide leaching tests were performed on the drill core interval samples from the North zone at varied particle sizes. The initial sodium cyanide (NaCN) concentration was 2 g/L and the leach pulp density was 40% solids by weight. The grinding particle sizes ranged from 80% passing 82 µm to 145 µm. The test results are shown in Figure 16.14.

Figure 16.14 Direct Cyanide Leach Test Results on Head Samples



After 48 h leaching, the gold extraction rate of the samples varied from 56% to 76% and the silver extraction rate was between 29% and 60%. The results indicate that a finer grinding particle size produced a higher gold extraction rate, excluding the MZ 13B sample, which produced a higher gold extraction at a coarser grinding particle size. The tests also indicated that leach kinetics was fast. On average, approximately 88% of the leachable gold was extracted within 6 h.

Cyanide Leaching Tests on Open Cycle Flotation Products

Flotation Rougher Scavenger Tailings

Five leach tests were carried out on the rougher scavenger flotation tailings from the North zone drill core interval samples (Samples MZ13B, MZ16B, MZ30B, MZ31A, and MZ31B). The leaching retention time was 24 h and the initial NaCN strength was 2.0 g/L. The leach pulp density was 40% solids by weight. The feed particle size was between 80% passing 63 µm and 78 µm. The results showed that approximately 36% to 62% of the gold and 49% to 65% of the silver in the tailings were extracted. Another leaching test on the Comp 3 rougher scavenger flotation tailings (80% passing 53 µm) showed that 78.7% of the gold and 51.5% of the silver were cyanide leachable.

Cleaner Flotation and Gold Bearing Pyrite Concentrate

PRA performed a leaching test on the combined product of the bulk cleaner flotation tailings and the gold bearing pyrite concentrate produced from the Comp 3 sample. The leaching retention time was 48 h and the initial NaCN strength was 2.0 g/L. The leach pulp density was 30% solids by weight. Approximately 53% of the gold and 91% of the silver were extracted from the combined gold bearing product.

The bulk cleaner tailings and the upgraded gold bearing pyrite concentrate produced from the flotation test on Comp 5 were combined and subjected to a 48 h cyanide leach test with an initial NaCN strength of 2.0 g/L. The test extracted 82% of the gold and 77% of the silver from the leach head.

CYANIDE LEACHING TESTS ON LOCKED CYCLE FLOTATION PRODUCTS

Twelve cyanide leach tests were performed on the products from the two locked cycle flotation tests. The varied leach conditions including initial cyanide strength, leaching retention time, and with or without pre-aeration prior to leaching process, were tested.

FLC1 Locked Cycle Flotation Products

The products produced from the locked cycle test FLC 1 (Comp 3) were tested for gold and silver extraction. The results are shown in Table 16.12.

Table 16.12 Leach Test Results

Test ID	Products	Test Condition	Calc. Head (g/t)		Extraction (%)	
			Au	Ag	Au	Ag
C1	FLC1 Flotation Tailings-Cycle 6	Direct Leach (DL)[1]	0.54	0.5	74.1	51.8
C2	FLC 1 Cl Tailings-Cycle 6	CIL[2]	3.11	4.71	35.0	55.4
C3	FLC 1 Au-Pyrite Conc.-Cycle 6	Regrind/CIL[2]	0.98	3.43	75.6	92.7
C4	FLC 1 Au-Pyrite Conc.-Cycle 5	Regrind/CIL[2]	2.49	4.24	72.3	85.9
C9	FLC 1 Cl Tailings - Composite	Regrind/CIL[2,3]	3.61	8.2	60.5	90.3
C10	FLC 1 Cl Tailings - Composite	Regrind/$Pb(NO_3)_2$/CIL[2]	4.08	3.2	62.7	41.1
C11	FLC 1 Cl Tailings - Composite	Regrind/DL[3,4]	3.03	6.8	65.2	41.4
C12	FLC 1 Cl Tailings - Composite	Regrind/Roasting/DL[2]	4.81	8.1	67.6	29.3

[1] 3.0 g/L NaCN
[2] 5.0 g/L NaCN
[3] No pre-aeration (all other tests with 4 h pre-aeration)
[4] 20.0 g/L NaCN

The test results indicated that the bulk cleaner scavenger tailings with higher gold and silver grades responded poorly to the cyanide leaching. Test C2 only extracted approximately 35% of the gold from the cleaner scavenger tailings. However, with pre-treatment by regrinding, the gold leach extraction from the cleaner scavenger tailings was significantly improved to 61%. With regrinding and adding lead nitrate ($Pb(NO_3)_2$), or pre-treatment by roasting, or adding 20.0 g/L NaCN, the gold extraction from the cleaner scavenger tailings was further improved.

The gold bearing pyrite flotation concentrate performed much better than the bulk cleaner tailings. Approximately 76% of the gold and 93% of the silver were extracted from Test C3. These results were obtained by regrinding the leach feed to a particle size of 80% passing 12 µm.

The tests also indicated that 74% of the gold was extractable from the flotation tailings with a particle size of 80% passing 107 µm.

FLC2 Locked Cycle Flotation Products

The cyanide leaching tests were also conducted on the products from the FLC2 test (Comp 4). The test results are presented in Table 16.13.

Table 16.13 Leach Test Results

Test ID	Products	Test Condition	Calc. Head (g/t)		Extraction (%)	
			Au	Ag	Au	Ag
C5	FLC 2 Flotation Tailings - Cycle 6	Direct Leach	0.14	0.5	42.8	51.4
C6	FLC 2 Cl Sc Tailings - Cycle 6	CIL	1.19	5.75	28.8	91.3
C7	FLC 2 Au-Pyrite Conc. - Cycle 6	Regrind/CIL	0.28	3.2	61.0	50.0
C8	FLC 2 Au-Pyrite Conc. - Cycle 5	Regrind/CIL	0.36	2.9	77.7	91.5

The tests produced similar results which were obtained from the leach tests on the FLC 1 products. The results indicate that the bulk cleaner tailings did not respond well to the cyanidation. With 4 h pre-aeration and 24 h leaching, the procedure only extracted 29% of the gold from the cleaner tailings with a particle size of 80% passing 23 µm, however, the silver extraction was high.

It appears that the gold bearing pyrite concentrate responded well to the cyanide leach procedure, a gold extraction of 78% was obtained from the pyrite concentrate which was reground to a particle size of 86% passing 38 µm.

About 43% of the gold was able to extract from the bulk rougher scavenger tailings only containing 0.18 g/t Au.

16.1.8 OTHER TESTS

SETTLING TESTS

Four settling tests were carried out on the final cycle copper rougher scavenger tailings (final tailings) from the locked cycle flotation tests. The test results are presented in Table 16.14. The results show that the addition of flocculant would significantly improve settling rate.

Table 16.14 Settling Test Results

Sample ID	Test	Feed Solids Density (%)	Floc* (g/t)	pH	Required U/F Solids Density	Unit Thickening Area m²/t/d solids	Supernatant	
							TSS (mg/l)	TD (mg/l)
FLC1 Final Tailings	ST-1	18.2	N/A	8.1	40.0	3.31	114	156
	ST-2	18.3	20	10.0	40.0	1.40	6.4	144
FLC2 Final Tailings	ST-3	17.9	N/A	8.5	50.0	8.09	300	224
	ST-4	17.9	20	10.0	40.0	0.44	28.8	176

*Flocculant.

ACID BASE ACCOUNTING TESTS

Preliminary acid base accounting tests were performed on cyanide leaching residues and on locked cycle flotation final tailings. The test results are presented in Table 16.15.

Table 16.15 Acid Accounting Test Results*

Sample ID	S (T) (%)	Acid Potential	Neutralization Potential (NP)		
			Actual	Ratio	Net
C2 Residue	21.9	684.4	39.6	0.1	-644.7
C3 Residue	1.9	60.9	57.6	0.9	-3.3
C6 Residue	24.4	762.5	26.8	0.0	-735.7
C7 Residue	1.3	40.0	39.8	1.0	-0.2
FLC1 Tails	0.2	1.6	63.34	40.5	61.8
FLC2 Tails	0.1	1.3	41.8	33.4	40.5

***Notes:**

- Analytical procedures from “Field and Laboratory Methods Applicable to Overburden and Minesoils.” EPA 600/2-78-054, 1978. pp. 45-55.
- Actual NP = Neutralization potential as determined by Sobek acid consumption test.
- Acid potential = (% total sulphur-% sulphate sulphur) X 31.25.
- NP Ratio = Actual NP / Acid potential.
- Net NP = Actual NP - Acid potential.
- The acid potential and the neutralizing potentials are expressed in Kg CaC03 equivalent per tonne of sample.

The leach residues presented acid generating potential, particularly the residues from Tests C2 and C6. Further detailed testwork is recommended to investigate the effect of the flotation tailings and leach residues disposal on the environment.

16.1.9 Conclusions and Recommendations

Conclusions

The preliminary testwork reviews resulted in the following conclusions:

- The mineralization is moderately hard.
- Two main flowsheets — direct cyanide leach of the head samples to recover gold and silver, and a combination of flotation and cyanide leach to recover gold, copper and molybdenum — were investigated. It appears that the combined process should be more amendable for the mineralization.
  - The flotation process will include copper/gold/molybdenum bulk flotation, including cleaner flotation, followed by copper/molybdenum separation.
  - The separation will produce a copper/gold concentrate and a molybdenum concentrate.
  - The copper/gold/molybdenum bulk flotation tailings will be further floated to produce a gold bearing pyrite concentrate.
  - The gold bearing pyrite concentrate together with the cleaner tailings from the bulk concentrate cleaner flotation will be cyanide leached to produce gold/silver doré.
- The test results indicate that the mineralization of the Upper zone was less amendable to the flotation flowsheet compared with the Main zone mineralization.
- The process conditions from the testwork have not yet been optimized.
- It appears that the contents of arsenic (As), antimony (Sb), mercury (Hg), and zinc (Zn) in the concentrate produced from the Upper zone mineralization may be higher than the smelting thresholds set out by most of the smelters.
- The gold bearing pyrite flotation concentrates responded reasonably well to cyanide leach. However, it appears that the copper/gold/molybdenum bulk cleaner flotation tailings produced lower gold extractions in comparison to the gold bearing pyrite flotation concentrates.

Recommendations

Further testwork is recommended to confirm the findings of the testwork completed to date, optimize process flowsheet, investigate metallurgical performances, and determine engineering related data. For detailed recommendations, refer to Section 19.0 of this report.

16.1.10 Metallurgical Performance Projection

According to the preliminary metallurgical test results and the proposed annual mining schedule, the metallurgical performances of the mineralization are projected in Table 16.16.

Table 16.16 Projected Metallurgical Performances

						Copper Concentrate												Doré						Molybdenum Concentrate			
		Mill Feed Grade				Tonnage	Recovered Metals					Grade			Recovery			Recovered Metals				Recovery		Tonnage	Rec'd Metal	Grade	Recovery
	Annual Process Rate						Cu	Au		Ag								Au		Ag							
Year	(t) (000's)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	(t)	(t)	(kg)	(oz) (000's)	(kg)	(oz) (000's)	Cu (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (%)	Ag (%)	(kg)	(oz) (000's)	(kg)	(oz) (000's)	Au (%)	Ag (%)	(t)	(t)	Mo (%)	Mo (%)
1	43,800	0.036	1.39	1.42	0.012	11,070	2,440	11,270	362	12,150	390	22.0	1,018	1,097	15.4	18.6	19.6	31,950	1,027	30,170	970	52.6	48.6	5,130	2,560	50	50
2	43,800	0.072	0.76	1.82	0.012	58,760	12,930	12,240	394	29,980	964	22.0	208	510	41.2	36.7	37.7	9,300	299	20,070	645	27.9	25.2	5,040	2,520	50	50
3	43,800	0.117	0.60	2.04	0.011	139,580	30,710	13,120	422	45,420	1,460	22.0	94	325	59.9	49.8	50.8	4,550	146	13,750	442	17.2	15.4	4,640	2,320	50	50
4	43,800	0.119	0.65	1.87	0.011	143,690	31,610	14,350	461	41,900	1,347	22.0	100	292	60.5	50.3	51.3	5,000	161	12,770	411	17.5	15.6	4,690	2,340	50	50
5	43,800	0.117	0.68	1.82	0.010	138,610	30,490	14,720	473	40,310	1,296	22.0	106	291	59.7	49.7	50.7	5,390	173	12,930	416	18.2	16.3	4,420	2,210	50	50
6	43,800	0.112	0.68	1.87	0.010	128,970	28,370	14,490	466	40,630	1,306	22.0	112	315	58.0	48.5	49.5	5,700	183	13,700	450	19.1	17.1	4,420	2,210	50	50
7	43,800	0.119	0.73	1.86	0.010	143,290	31,530	15,950	513	41,820	1,345	22.0	111	292	60.5	50.2	51.2	5,800	186	13,320	428	18.3	16.3	4,340	2,170	50	50
8	43,800	0.136	0.71	1.77	0.010	178,300	39,230	16,740	538	42,480	1,366	22.0	94	238	65.7	53.8	54.8	4,850	156	10,740	345	15.6	13.9	4,160	2,080	50	50
9	43,800	0.086	0.64	1.47	0.011	82,990	18,260	11,690	376	27,450	883	22.0	141	331	48.3	41.7	42.7	6,550	211	13,530	435	23.3	21.0	4,640	2,320	50	50
10	43,800	0.087	0.63	1.55	0.010	84,190	18,520	11,470	369	29,090	935	22.0	136	346	48.6	41.9	42.9	6,310	203	14,060	452	23.0	20.7	4,560	2,280	50	50
11	43,800	0.108	0.64	1.71	0.009	121,200	26,660	13,290	427	36,260	1,166	22.0	110	299	56.6	47.5	48.5	5,390	173	12,900	415	19.3	17.3	3,940	1,970	50	50
12	43,800	0.119	0.64	1.64	0.009	143,490	31,570	13,990	450	36,820	1,184	22.0	97	257	60.5	50.2	51.2	4,830	155	11,120	357	17.4	15.5	3,720	1,860	50	50
13	43,788	0.094	0.56	1.42	0.010	96,960	21,330	10,880	350	27,950	899	22.0	112	288	51.7	44.0	45.0	5,130	165	11,560	372	20.8	18.6	4,160	2,080	50	50
14	43,800	0.099	0.57	1.53	0.009	105,460	23,200	11,300	363	30,990	996	22.0	107	294	53.5	45.3	46.3	4,970	160	11,950	384	19.9	17.9	4,030	2,020	50	50
15	43,800	0.087	0.52	1.44	0.009	84,710	18,640	9,520	306	27,090	871	22.0	112	320	48.7	42.0	43.0	4,890	157	12,200	392	21.5	19.4	3,850	1,930	50	50
16	43,800	0.115	0.56	1.66	0.008	134,740	29,640	12,150	391	36,450	1,172	22.0	90	271	59.1	49.2	50.2	4,230	136	11,090	357	17.1	15.3	3,330	1,660	50	50
17	43,800	0.107	0.49	1.51	0.007	119,880	26,370	10,120	325	32,040	1,030	22.0	84	267	56.4	47.3	48.3	3,720	120	10,280	331	17.4	15.5	3,240	1,620	50	50
18	43,800	0.094	0.48	1.42	0.008	97,170	21,380	9,210	296	28,090	903	22.0	95	289	51.7	44.1	45.1	4,080	131	10,900	350	19.5	17.5	3,590	1,800	50	50
19	43,800	0.104	0.53	1.47	0.008	115,400	25,390	10,760	346	30,660	986	22.0	93	266	55.5	46.7	47.7	4,220	136	10,530	338	18.3	16.4	3,640	1,820	50	50
20	43,800	0.122	0.58	1.69	0.007	149,000	32,780	12,850	413	38,350	1,233	22.0	86	257	61.4	50.9	51.9	4,090	131	10,640	342	16.2	14.4	3,150	1,580	50	50
21	43,800	0.126	0.54	1.88	0.006	156,360	34,400	12,120	390	43,270	1,391	22.0	77	277	62.5	51.6	52.6	3,540	114	10,980	353	15.1	13.4	2,803	1,400	50	50
22	43,800	0.138	0.62	1.91	0.008	182,430	40,140	14,650	471	46,240	1,487	22.0	80	253	66.2	54.2	55.2	3,880	125	10,620	341	14.4	12.7	3,550	1,770	50	50
23	2,627	0.167	0.59	2.37	0.005	14,610	3,210	920	30	3,760	121	22.0	63	257	73.3	59.2	60.2	160	5	570	18	10.6	9.2	139	70	50	50
Total	966,215	0.105	0.64	1.67	0.009	2,630,840	578,780	277,790	8,931	769,180	24,730	22.0	106	292	56.9	44.6	47.6	138,520	4,454	290,660	9,345	22.3	18.0	89,180	44,590	50	50
Average	43,800	0.105	0.64	1.67	0.009	119,260	26,237	12,593	405	34,870	1,121	22.0	106	292	56.9	44.6	47.6	6,280	202	13,180	424	22.3	18.0	4,040	2,020	50	50

16.2 Mineral Processing

16.2.1 *Introduction*

The proposed Snowfield concentrator will process the gold-copper-molybdenum porphyry mineralization at a nominal rate of 120,000 t/d and with an availability of 92% (365 d/a). The concentrator will produce a marketable copper concentrate containing gold and silver, gold-silver doré and a by-product molybdenum concentrate.

16.2.2 *Summary*

The process is developed to produce three products: a copper-gold concentrate, a molybdenum concentrate, and gold-silver doré. The process plant will consist of three stages of crushing, primary grinding, followed by flotation processes to recover copper, gold, and molybdenum from the feed material. An additional process of cyanidation will be used on the gold-bearing pyrite products to recover gold which will be refined on site to gold-silver doré.

A copper-gold and molybdenum separation circuit is proposed to separate molybdite from copper minerals to produce a molybdenum concentrate and a copper-gold concentrate. The copper concentrate will be thickened and filtered and sent to the concentrate stockpile while the molybdenum concentrate will be thickened, filtered, dried, bagged, and stored for subsequent shipping to smelters.

The final flotation tailings and leach residues will be stored in a conventional tailings pond. Process water will be recycled from the tailings pond. Fresh water will be used for mill cooling, gland seal service, and reagent preparation.

16.2.3 *Flowsheet Development*

The mill flowsheet design is based on the results of grinding, flotation and leach testwork carried out by PRA, in 2009 and 2010, together with engineering experience.

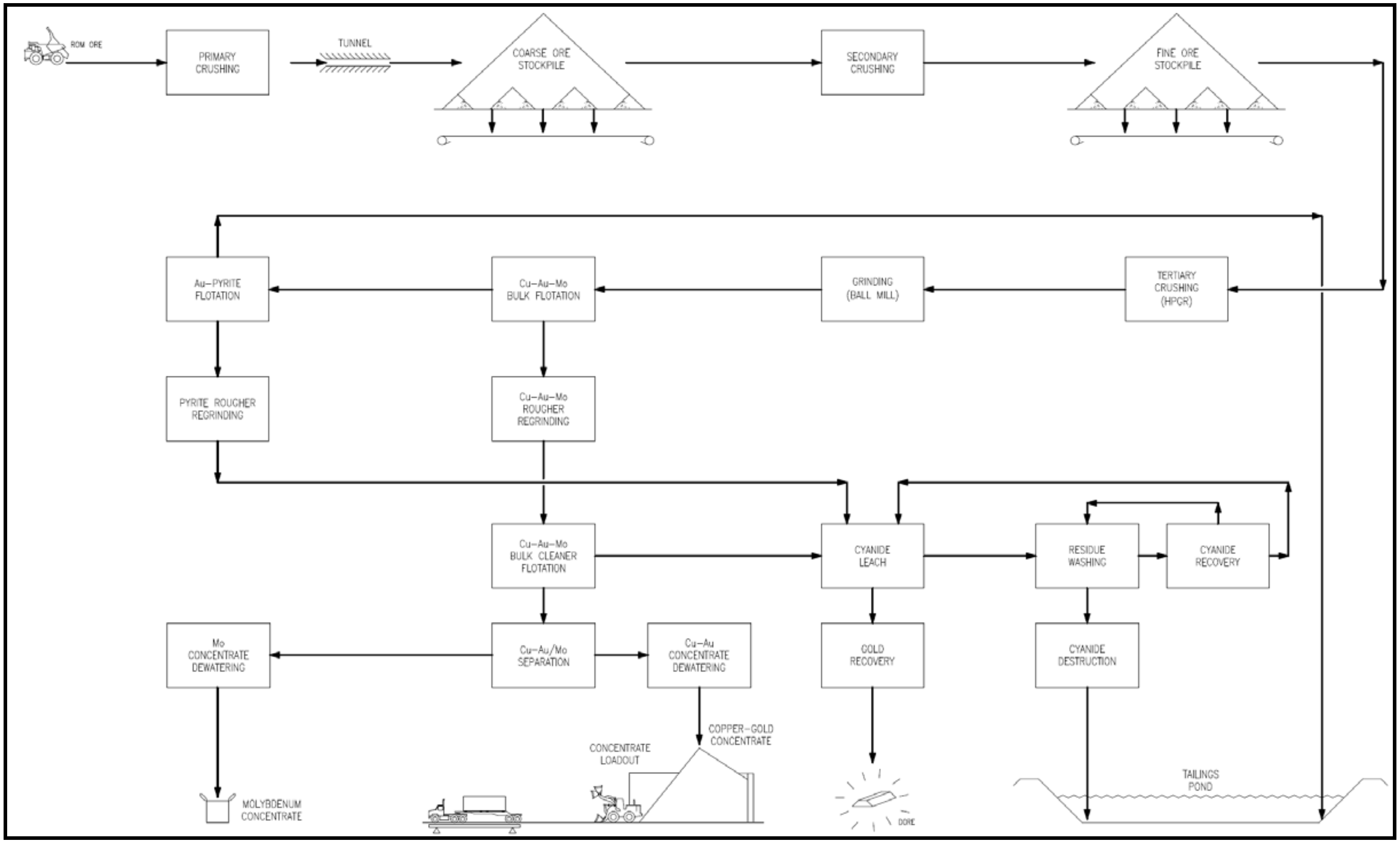
The process plant will consist of the following unit operations:

- primary crushing
- conveying system
- coarse material stockpile and reclaim
- mill feed receiving
- secondary and tertiary crushing
- grinding

- flotation
- concentrate thickening, filtration, and dispatch
- cyanide leaching by CIL
- gold recovery from gold loaded carbon and doré production
- cyanide recovery, destruction, and related processes
- tailing disposal to a tailing impoundment.

The simplified flowsheet is shown in Figure 16.15.

Figure 16.15 Simplified Process Flowsheet



16.2.4 PLANT DESIGN

MAJOR DESIGN CRITERIA

The concentrator has been designed to process 120,000 t/d, equivalent to 43,800,000 t/a. The major criteria used in the design are outlined in Table 16.17.

Table 16.17 Major Design Criteria

Criteria	Unit	
Daily Processing Rate	t/d	120,000
Operating Days per Year	d	365
Primary Crushing		
Crushing Availability	%	70
Primary Crushing Rate	t/h	7,143
Primary Crushing Product Particle Size, P_{80}	µm	150,000
Secondary Crushing		
Crushing Availability	%	75
Secondary Crushing Rate	t/h	6,667
Secondary Crushing Product Particle Size, P_{80}	µm	40,000
Tertiary Crushing (HPGR)/Grinding/Flotation/Leach		
Availability	%	92
Milling & Flotation Process Rate	t/h	5,435
HPGR Crusher Feed Size, F_{100}	mm	50
HPGR Crusher Product Size, P_{80}	mm	3
Ball Mill Grinding Particle Size, P_{80}	µm	125
Ball Mill Circulating Load	%	300
Bond Ball Mill Work Index	kWh/t	16.6
Copper Concentrate Regrinding Particle Size, P_{80}	µm	20
Gold-bearing Pyrite Concentrate Regrinding Particle Size, P_{80}	µm	20
Leach method		CIL
Feed Rate to Leach Circuit	t/d	17,655

OPERATING SCHEDULE AND AVAILABILITY

The primary crushing and process plant will be designed to operate on the basis of two 12 h shifts per day, for 365 d/a.

The primary crusher overall availability will be 70% and will be located at the mine site. Secondary crusher availability will be 75% and this circuit will be located at the plant site. Tertiary crushing (HPGR), grinding, flotation and leach circuit availability will be 92%. The availabilities will allow for a potential increase in crushing rate, and will allow sufficient downtime for scheduled and unscheduled maintenance of the crushing and process plant equipment and potential weather interruptions.

16.2.5 PROCESS PLANT DESCRIPTION

PRIMARY CRUSHING

A conventional gyratory crushing facility will be designed to crush mineralization materials from the proposed mine to reduce the size of the rocks in preparation for the grinding process at an average rate of 7,143 t/h.

The major equipment and facilities in this area include:

- two dump pockets
- two hydraulic rock breakers
- two gyratory crushers, each1,525 mm x 2,261 mm (60″ x 89″), 600 kW each
- apron feeders
- conveyor belts, metal detectors, self-cleaning magnets, and belt tear detectors
- belt scales
- dust collection system.

The mill feed will be trucked from the proposed open pit to the primary crushers by 363-t haul trucks. The mineralization will be reduced to 80% passing 150 mm using two gyratory crushers. Two rock breakers will be installed to break any oversize rocks.

The crusher product will be discharged onto an overland conveyor using an apron feeder. The apron feeder discharge will be conveyed via an overland conveyor through a 26 km long tunnel to the crushed material stockpile at the Snowfield project plant site.

The crushing facility will be equipped with a dust collection system to control fugitive dust generated during crushing and conveyor loading.

COARSE MATERIAL STOCKPILE AND RECLAIM

The coarse material stockpiling and reclaim system will include:

- coarse material stockpile, 30,000 t live capacity
- reclaim apron feeders
- conveyors, metal detectors, self-cleaning magnets, and belt tear detectors
- dust collection system.

The coarse material stockpile will have a live capacity of 30,000 t. The crushed materials will be reclaimed from this stockpile by apron feeders at a nominal rate of

6,667 t/h. The apron feeders will feed two 2,134 mm wide conveyors which in turn feed the secondary crushing circuits.

The crushing facility and the stockpile will be equipped with a dust collection system to control fugitive dust that will be generated during conveyor loading and the transportation of the crushed materials.

SECONDARY CRUSHING

The secondary circuit will consist of two crushing trains for a combined secondary crushing circuit capacity of 6,667 t/h. The secondary crushing circuits will be operated in closed-circuit with dry screens.

The secondary crushing facility will include:

- four double-deck vibratory screens: each 3.7 m wide x 7.3 m long, 75/50 mm apertures
- four cone crushers each with 750 kW installed power
- conveyor belts, metal detectors, self-cleaning magnets, and belt tear detectors
- belt scales
- dust collection system.

Reclaimed materials from the coarse material stock pile will be conveyed on two conveyors to the secondary crushing facility. The secondary crushers will consist of two trains each containing a splitter chute, two vibrating dry double-deck screens in closed circuit with two cone crushers. The cone crusher product will return to the screen feed conveyor to combine with fresh reclaimed materials as feeds to the vibratory double deck screens. The screened product, finer than 50 mm, will be delivered to the fine material stockpile by conveyor.

FINE MATERIAL STOCKPILE AND RECLAIM

The fine material stockpiling and re-handling system will include:

- two fine material stockpiles, each having 60,000 t live capacity
- reclaim apron feeders
- conveyor belts, metal detectors, self-cleaning magnets, and belt tear detectors
- dust collection system.

Two fine material stockpiles will have a combined live capacity of 120,000 t. The crushed material will be reclaimed from these stockpiles by apron feeders at a nominal rate of 6,667 t/h. Apron feeders will reclaim material to feed two 400 t live capacity HPGR surge bins. The surge bins will feed the tertiary crushing circuit.

The tertiary crushing facility and the fine material stockpiles will be equipped with a dust collection system to control fugitive dust that will be generated during conveyor loading and the transportation of the crushed materials.

TERTIARY CRUSHING

Tertiary crushing will be done using HPGR's to reduce the crushed material to a product size of P80 3 mm prior to entering the grinding circuit. The facility will include:

- four belt feeders
- four HPGR crushers with two-2,600 kW motors each
- four 3.7 m wide x 7.3 m long double-deck vibratory screens, with 15/6 mm apertures.

There will be four HPGR crushers, each fed independently via belt feeders from the HPGR surge bins. Each HPGR discharge will be in closed circuit with a vibrating double deck screen. HPGR product will be wet-screened at a cut size of 6 mm. Screening in this area will be fed independently from each HPGR with screen oversize returning to the HPGR feed bin. Screen undersize will leave the crushing circuit and report to the primary grinding circuit at a process flow rate of 5,435 t/h, or 1,359 t/h per line.

PRIMARY GRINDING AND CLASSIFICATION

The grinding circuit will consist of a ball mill circuit. The ball mills will be arranged in a closed circuit with the classifying cyclones. The grinding will be conducted as a wet process at a nominal rate of 1,359 t/h of material per ball mill.

The grinding circuit will include:

- four 15.0 MW ball millls (7.93 m diameter x 12.3 m long (26 ft x 40.5 ft)
- cyclone feed slurry pumps
- four cyclone clusters, each with eight 710 mm diameter cyclones
- particle size analyzers
- samplers.

Each ball mill will be operated independently in closed circuit with a cyclone cluster. The screened HPGR product will enter the grinding circuit via the cyclone feed pumpbox. The product from each ball mill will be discharged into its separate cyclone feed pumpbox combining with the HPGR screen undersize. The slurry in each mill discharge pumpbox will be pumped to a cyclone cluster for classification with the cyclone underflow returning by gravity to the ball mill. The cut size for the cyclones will be a P80 of 125 µm, and the circulating load to the individual ball mill circuits will be 300%.

The new feed to each ball mill circuit will be 1,359 t/h and the combined total of the four mills, 5,435 t/h, will constitute the feed rate to the copper/gold/molybdenum flotation circuit. Dilution water will be added to the grinding circuit as required.

Provision will be made for the addition of lime to the ball mills for the adjustment of the pH of the slurry in the grinding circuit prior to the flotation process if necessary.

The cyclone overflow from each classification circuit will be discharged into the head end of a flotation train. The pulp density of the cyclone overflow slurry will be approximately 35% solids.

FLOTATION CIRCUIT

The milled pulp will be subjected to flotation to recover the targeted minerals. Three products will be created in the flotation circuit, a copper concentrate containing gold and silver, a molybdenum concentrate, and a gold bearing pyrite concentrate.

Copper/Gold/Molybdenum Flotation Circuit

The copper/gold/molybdenum flotation circuit will include the following equipment:

- flotation reagent addition facilities
- four rougher flotation tank cells (four trains, six 200 m^3 cells per train)
- four 1119 kW regrind tower mills
- two cyclone clusters, each with sixteen 250 mm diameter cyclones
- five 100 m^3 first cleaner flotation tank cells
- two 100 m^3 first cleaner scavenger flotation tank cells
- five 50 m^3 second cleaner flotation tank cells
- third cleaner flotation column
- slurry pumps
- sampling system.

The cyclone overflows from the grinding circuit will gravity flow to the copper/gold/molybdenum rougher flotation tank cells, which will be arranged in four parallel trains. The feed to each flotation train will enter the circuit at the feed rate of 1,359 t/h. Flotation reagents will be added to the flotation circuit as defined through testing. The flotation reagents added will be the collectors, PAX, 3926A and A208 and the frother, MIBC. Lime will be used as a pH modifier throughout the process as required. Provision will be made for the staged addition of the reagents in the cleaner stage of the flotation circuit.

The conditioned slurry will overflow the feed box into the rougher flotation tank cells. Rougher flotation concentrates with a mass recovery of approximately 4% of the flotation feed will be discharged into the regrind cyclone feed pumpbox where the concentrate joins with the regrinding mill discharges. The combined products will be pumped to the two regrinding classification cyclone clusters.

The rougher regrinding circuit cyclone will separate the fine fraction with a particle size of 80% passing 20 µm into a cyclone overflow product. The coarser cyclone underflow will be split into four and will be the feed for the rougher regrinding tower mills.

The cyclone overflow from the rougher regrinding circuit will combine with the second cleaner tailings and the first cleaner-scavenger concentrate as feed to the first cleaner stage.

The concentrate from the first cleaner stage will feed the second cleaner flotation stage with the second cleaner concentrate reporting to the third cleaner flotation stage. The concentrate from the third cleaner flotation stage will be a bulk copper/gold/molybdenum concentrate and will feed the copper/molybdenum concentrate thickener in the copper/molybdenum separation circuit. The tailings from the third cleaner stage will be returned to the second cleaner stage. Tailings from the first cleaner-scavenger flotation stage will report to the carbon-in-leach (CIL) feed thickener of the gold cyanide leach circuit.

Conventional tank flotation cells will be used for the bulk flotation circuit up to the second cleaner flotation. Third cleaner flotation will take place in a flotation column to improve copper concentrate grade.

Copper / Molybdenum Separation

The final copper/gold/molybdenum bulk concentrate will be processed to produce a copper/gold concentrate and a molybdenum concentrate.

The bulk concentrate will feed into the copper/molybdenum separation circuit where conventional copper/molybdenum separation will take place.

The circuit will include bulk concentrate thickening, separation of the molybdenum from the copper concentrate through copper depression, and molybdenum rougher flotation concentrate regrinding and upgrading. The molybdenum rougher flotation concentrate will be upgraded through stage-wise cleaning. Column flotation will be incorporated in this area. The final molybdenum concentrate will be dewatered using a pressure filter. The dewatered concentrate will be further dried prior to bagging and storage for shipment.

The tailings from the molybdenum circuit will constitute the copper concentrate and will be directed to the copper concentrate thickener.

Gold-Bearing Pyrite Flotation Circuit

The tailings from the copper rougher flotation circuit will be the feed to the gold-bearing pyrite flotation circuit.

The flotation circuit will include the following equipment:

- flotation reagent addition facilities
- four rougher flotation tank cells, four trains, each train nine 200 m^3 tank cells
- ten 1,119 kW regrinding tower mills
- two cyclone clusters (eighteen 250 mm diameter cyclones per cluster)
- slurry pumps

The pyrite circuit will consist of four parallel lines following on from the copper flotation circuit. The targeted mineral in this circuit is the gold bearing pyrite. The flotation reagents added will be the collectors, PAX, 3418A and A208, and the frother, MIBC.

Tailings from the pyrite flotation circuit will be sampled automatically prior to discharge into one of two final tailings pumpboxes. These streams will constitute the final tailings leaving the plant and will gravity flow or be pumped to the tailings storage facility.

The rougher pyrite concentrate will be reground to a particle size of 80% passing 20 µm in 10 regrinding tower mills. The tower mills will be in closed circuit with two hydrocyclone clusters consisting of a total of 36 cyclones. The cyclone overflow will report to the CIL feed thickener in the gold leach circuit.

The scavenger tailings will be sampled automatically prior to discharge into the final tailings pumpbox. This stream will constitute the final tailings leaving the plant.

GOLD RECOVERY FROM FLOTATION PRODUCTS

CIL Leaching

The re-ground gold bearing pyrite product together with the first cleaner scavenger tailings from the copper/gold/molybdenum bulk flotation circuit will form the feed to the gold leach circuit.

The key equipment in the leach circuit will include:

- a CIL feed thickener
- two aeration tanks (15.0 m diameter x 15.0 m high)
- ten CIL leach tanks with in-tank carbon transferring pumps and screens (15.0 m diameter x 15.0 m high)
- a loaded carbon screen
- a carbon safety screen
- a slurry pumps.

Feed will enter the circuit via the CIL feed thickener where the solids will be thickened to a density of 60% solids. The thickener underflow will be pumped to the head of the two aeration tanks where the slurry will be diluted and aerated prior to entering the leach circuit. Lime will be added to adjust pH.

Sodium cyanide will be used to recover gold from the slurry in a conventional CIL circuit. The CIL leach circuit will consist of ten agitated tanks equipped with in-tank carbon transferring systems. The circuit will run in a counter-current arrangement with carbon advancing to the feed cell prior to discharge.

Loaded carbon will be washed on a loaded carbon screen before it is advanced to the subsequent gold stripping circuit.

Slurry will leave the final leach tank and go over a carbon safety screen. CIL tailings will be pumped to the cyanide recovery/destruction plant for treatment prior to disposal in the tailings pond.

Carbon Stripping and Regeneration

The loaded carbon will be treated by acid washing and the Zadra pressure stripping process for gold desorption to create a gold-rich solution for electrowinning. Carbon stripping will be done as a batch process with a design of one elution per day.

The main process equipment includes:

- acid wash vessel
- two elution columns
- loaded and barren solution tanks
- acid wash reagent tank
- heating systems
- heat exchanger systems
- pumps.

The loaded carbon will be acid washed prior to transfer to the two elution columns.

Barren strip solution will be pumped through a heat recovery heat exchanger and a solution heater. The solution will then flow up through the bed of carbon in the elution column and overflow near the top of the stripping vessel. The solution will flow back through the heat exchange system where it will be cooled by exchanging heat with barren solution and flow through a back pressure control valve, to the pregnant solution holding tank. Pregnant solution will be pumped from the pregnant solution tank to the electrowinning cells for subsequent gold recovery. Barren solution created in the electrowinning circuit will then be returned to the barren solution tank for recycle.

Eluted carbon can be re-used upon reactivation. The process will require the following major items of equipment:

- reactivation kiln
- carbon quench tank
- carbon attrition tank
- washing/dewatering screen
- carbon storage bin
- fine carbon filter press
- associated pumps.

Eluted carbon will be reactivated in the reactivation kiln and combined with fresh carbon, the carbon will be treated and screen sized to remove fines prior to its re-introduction to the CIL circuit.

Gold Electrowinning and Refining

The pregnant gold solution will be pumped from the pregnant solution tank to the electrowinning cells. Gold will be deposited on stainless steel cathodes. Barren solution will be returned to the barren solution tank.

The precious metal sludge will be removed from the electrowinning cells on a batch basis and will be dewatered in a pressure filter. The filter cake will be transferred to the gold room for drying and smelting. An electric induction furnace will be used for the gold refining. The electrowinning and refinery area will be in a secure area with a security surveillance system in operation.

Cyanide Destruction and Recovery

- two 45.0 m diameter counter-current decantation (CCD) thickeners
- cyanide recovery system
- three cyanide destruction tanks (one 6.0 m diameter x 6.0 m high, two 11.0 m diameter x 12.0 m high)
- pumps.

The residue slurry from the CIL circuit will be pumped to a two-stage conventional CCD washing circuit. The first stage CCD thickener overflow will return to the CIL aeration tank or alternatively to the cyanide recovery system. The thickener overflow from the second CCD thickener is in a closed circuit and feeds the first stage CCD thickener. The first stage CCD thickener underflow will enter the second CCD thickener for further washing before it discharges to the agitated cyanide destruction tanks.

The remaining cyanide in the washed leach residues will be destroyed by the sulphur dioxide/air oxidation destruction method.

CONCENTRATE HANDLING

The copper/gold cleaner flotation concentrate will be thickened, filtered and stored prior to shipment to the smelter. The concentrate handling circuit will have the following concentrate equipment:

- thickener
- slurry pumps
- stock tank
- pressure filter
- storage and dispatch facility.

The concentrate produced will be pumped from the final copper/ molybdenum separation stage to the concentrate thickener. Flocculant will be added to the thickener feed to aid the settling process. The thickened concentrate will be pumped to the concentrate stock tank. The underflow density will be 60% solids. The concentrate stock tank will be an agitated tank which will serve as the feed tank for the concentrate filter. The concentrate filter will be a filter press unit. Since filtration with a filter press unit will be a batch process, the concentrate stock tank will also act as a surge tank for the filtration operation. The filter press will dewater the concentrate to produce a final concentrate with a moisture content of about 9%. The filtrate will be returned to the concentrate thickener. The filter press solids will be discharged to the concentrate stockpile. The dewatered concentrate will be stored in a designated storage facility. The concentrate will be loaded into trucks for dispatch off the property.

The thickener overflow solution from the concentrate thickener will be collected for recycling.

The molybdenum flotation concentrate will be thickened, filtered, dried, bagged and stored prior to shipment to the smelter. The concentrate handling circuit will have the following equipment for the concentrate:

- a thickener
- slurry pumps
- a stock tank
- a filter press
- a dryer
- a bagging system and storage.

The molybdenum concentrate will be dewatered using a similar process to the copper concentrate. The filtered concentrate will be further dewatered by an indirect heat dryer to reduce the moisture to 5% before being bagged and transported to processors.

TAILINGS HANDLING

The pyrite flotation tailings and the CIL residues will form the final plant tailings, which will be sent to a tailings impoundment. The tailings handling circuit will include the following systems:

- slurry transfer pumping system
- reclaim water barge and pumping system.

The CIL residues will be sent to the tailings facility separately from the flotation tailings. The residues will be deposited near the centre of the tailings impoundment facility to be covered with tailings pond water to aid in the prevention of sulphide mineral oxidation.

Water will be reclaimed from the tailings impoundment area to the process water tank by two stages of pumping.

Reagent Handling and Storage

Various chemical reagents will be added to the process slurry stream to facilitate the processes.

Reagents used in the process will include:

- flotation: PAX, 3926A, A208, fuel oil, sodium sulphide, lime, MIBC, and sodium silicate
- CIL and gold recovery: lime, sodium cyanide, activated carbon, sodium hydroxide, and hydrochloric acid
- cyanide recovery and destruction: metabisulphite, copper sulphate, sulphuric acid, lime, and sodium hydroxide
- others: flocculant and anti-scalant.

The preparation of the various reagents will require:

- a bulk handling system
- mix and holding tanks
- metering pumps
- a flocculant preparation facility
- a lime slaking and distribution facility
- eye-wash and safety showers
- applicable safety equipment.

Various chemical reagents will be added to the grinding, flotation and leaching circuits to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles into the various concentrate products.

Fresh water will be used for the making up or for the dilution of the various reagents that will be supplied in powder/solid form, or which require dilution prior to the addition to the slurry. The strength of the diluted reagent solutions will range between 10% and 25%. These solutions will be stored in separate holding tanks and added to the addition points of the flotation circuits, the CIL circuit and related circuits using metering pumps.

The liquid reagents (including fuel oil, A208, 3926A, MIBC, hydrochloric acid, sulphuric acid and anti-scalant) will not be diluted and will be pumped directly from the bulk containers to the points of addition using metered pumps.

Flocculant will be prepared in the standard manner as a dilute solution of less than 1% solution strength. This will be further diluted in the thickener feed well.

Lime will be delivered in bulk and will be off-loaded pneumatically into a silo. The lime will then be prepared in a lime slaking system as a 15% concentration slurry. This lime slurry will be pumped to the points of addition using a closed loop system.

The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire and safety protection and Material Safety Data Sheet stations will be provided at the facility.

ASSAY AND METALLURGICAL LABORATORY

The assay laboratory will be equipped with the necessary analytical instruments to provide all routine assays for the mine, the concentrator, and the environmental departments. The most important of these instruments includes:

- fire assay equipment
- atomic absorption spectrophotometer (AAS)
- X-ray fluorescence spectrometer (XRF)
- Leco furnace.

The metallurgical laboratory will undertake all necessary testwork to monitor metallurgical performance and, more importantly, to improve process flowsheet unit operations and efficiencies. The laboratory will be equipped with laboratory crushers, ball and stirred mills, particle size analysis sieves, flotation cells, filtering devices, balances, and pH meters.

WATER SUPPLY

Two separate water supply systems for fresh water and process water will be provided to support the operation.

Fresh Water Supply System

Fresh and potable water will be supplied to a fresh/fire water storage tank from wells and rivers. Fresh water will be used primarily for the following:

- fire water for emergency use
- cooling water for mill motors and mill lubrication systems

- gland service for the slurry pumps
- reagent make-up
- potable water supply.

The fresh/fire water tank will be equipped with a standpipe which will ensure that the tank is always holding at least a 6 h supply of fire water.

The potable water from the fresh water source will be treated (chlorination and filtration) and stored in the potable water storage tank prior to delivery to various service points.

Process Water Supply System

Concentrate thickener overflow solution will be re-used in process circuit. The majority of the process water will be reclaimed water from the tailings pond. All process water required will be distributed to the plant site from the process water tank.

AIR SUPPLY

Air service systems will supply air to the following service areas:

- crushing circuit - high-pressure air will be provided by dedicated air compressors for dust suppression
- flotation circuits - low-pressure air for flotation cells will be provided by air blowers
- leach circuits - high-pressure air will be provided by dedicated air compressors
- cyanide recovery and destruction circuits - high-pressure air will be provided by dedicated air compressors
- filtration circuit - high-pressure air will be provided by dedicated air compressors for filtration and drying
- plant air service - high-pressure air will be provided by dedicated air compressors for the various services
- instrument air - will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

PROCESS CONTROL AND INSTRUMENTATION

The plant control system will consist of a Distributed Control System (DCS) with PC-based Operator Interface Stations (OIS) located in the following two control rooms:

- primary crusher control room
- plant site control room.

The plant control room will be staffed by trained personnel 24 h/d.

In addition to the plant control system, a closed-circuit television (CCTV) system will be installed at various locations throughout the plant including the crushing facility, the stockpile conveyor discharge point, the tailings facility, the concentrate handling building and the gold recovery facilities. The cameras will be monitored from the local control room and central control room.

The plant will rely on the on-stream analyzer for process control. An on-line analyser will analyse each flotation stage for the circuit. A sufficient number of samples will be taken for on-line control and metallurgical accounting. Shift samples will be assayed in the assay laboratory.

An on-stream particle size monitor will determine the particle sizes of the primary cyclone overflow and the regrinding circuit products.

For the protection of operating staff, cyanide monitoring/alarm systems will be installed at the cyanide leaching area, cyanide recovery area and cyanide destruction areas. An SO_2 monitor/alarm system will also be used to monitor the cyanide destruction area.

17.0 MINERAL RESOURCE ESTIMATE

17.1 INTRODUCTION

The mineral resource estimate presented herein is reported in accordance with the Canadian Securities Administrators NI 43-101 and has been estimated in compliance with Canadian Institute of Mining's (CIM) estimation of mineral resource and mineral reserves best practices guidelines. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the mineral resource will be converted into mineral reserve. The quantity and grade of reported inferred resources in this estimate are conceptual in nature.

All resource estimation work reported here was done by FH Brown, MSc (Eng) CPG Pr.Sci.Nat., and Eugene Puritch, P.Eng.; both are independent QPs as defined by NI 43-101, from information and data supplied by Silver Standard. The effective date of this estimate is December 1, 2009. A draft copy of this report was reviewed by Silver Standard for factual errors.

Mineral resource modeling and estimation were carried out using the commercially available GEMS Gemcom (v5.23) and Snowden Supervisor (v7.10.11) software programs. Pit shell optimization was carried out using Whittle Four-X Single Element (v1.10).

17.2 PREVIOUS RESOURCE ESTIMATES

A previous resource estimate dated April 21, 2008 for the Snowfield deposit was prepared by Minorex. The mineral resource estimate reported a measured and indicated resource of 3.08 M ounces Au and an inferred resource of 0.47 M ounces Au in-situ (Table 17.1). The estimate was based on the results of 51 drillholes, 15 sample trenches, and used a global density of 2.82 t/m^3.

Table 17.1 Mineral Resource Estimate at 0.1 g/t Au Cut-Off[1], April 21, 2008

Class	Tonnes (x M)	Au (g/t)	Au (oz) (x 1000)
Measured	1.5	2.18	101.5
Indicated	77.1	1.20	2,975.6
Measured + Indicated	78.6	1.22	3,077.1
Inferred	14.4	1.01	466.2

[1] The above mineral resource estimate was prepared under the supervision of a QP as defined by NI 43-101. P&E have not independently verified the mineral resource estimate.

A mineral resource estimate dated January 31, 2009 for the Snowfield deposit was prepared by P&E. The mineral resource estimate reported a measured and indicated mineral resource of 4.36 M ounces Au and an inferred mineral resource of 14.28 M ounces Au (Table 17.2) using cut-off of 0.5 g/t AuEq. The estimate was based on the results of 113 drillholes and constrained within an optimized conceptual pit shell.

Table 17.2 Mineral Resource Estimate at 0.5 g/t AuEq Cut-off, January 31, 2009

Class	Tonnes (x M)	Au (g/t)	Au ozs (x M)	Ag (g/t)	Ag ozs (x M)	Cu (%)	Mo (ppm)
Measured	31.9	1.49	1.53	1.4	1.47	0.03	140
Indicated	102.8	0.86	2.83	1.6	5.21	0.07	110
Measured + Indicated	134.7	1.01	4.36	1.5	6.68	0.06	120
Inferred	661.8	0.67	14.28	1.8	39.00	0.12	80

17.3 Sample Database

Sample data were provided by Silver Standard in the form of ASCII text files, Excel spreadsheets, and Access databases.

P&E prepared a Gemcom format access database from the data supplied by Silver Standard. One drillhole was identified as a wedged drillhole from a parent drillhole, which no downhole survey data were available. The wedged drillhole was not used for mineral resource estimation. The remaining 141 drillhole and 15 sampling trench records contain collar, survey, and assay data (Table 17.3). Assay data fields consist of drillhole ID, downhole interval distances, sample number; and Au, Ag, Cu and Mo grade fields. All data are in metric units and grid coordinate are in the UTM NAD27 system.

Table 17.3 Database Records

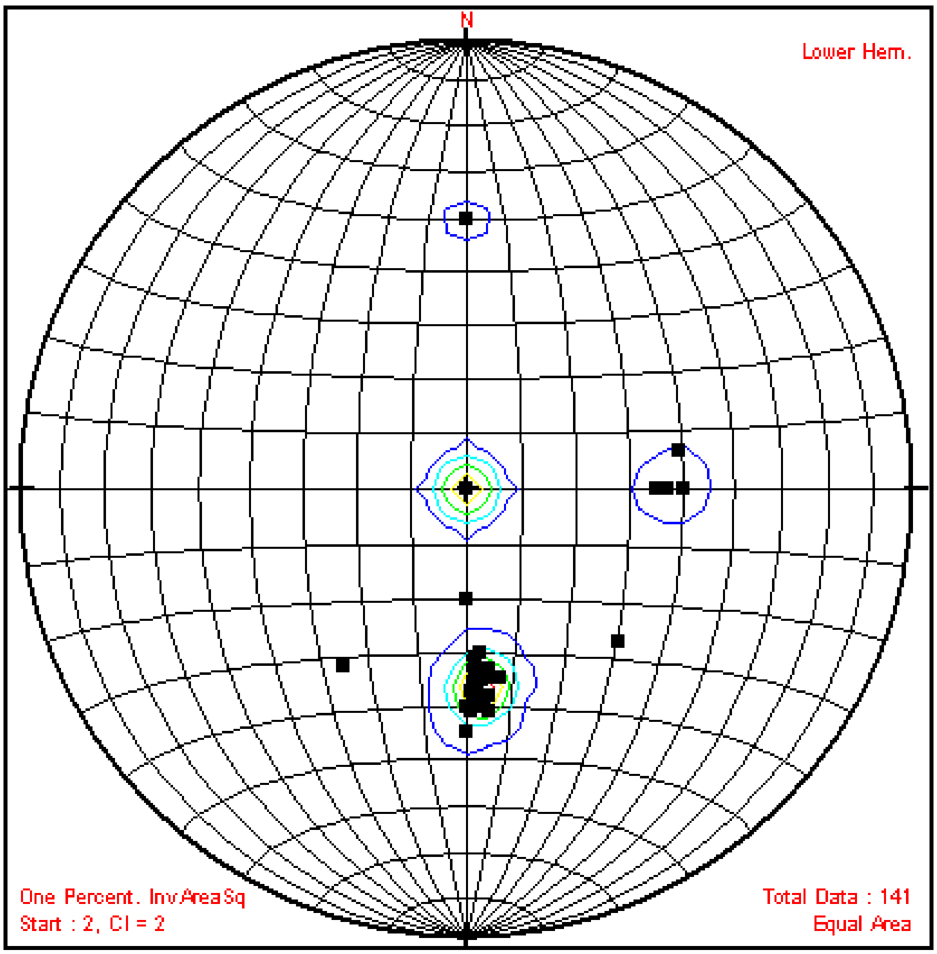
Data Type	Record Count
Collars	157
Survey Records	98972
Assay Records (Au)	35,937

17.4 Database Validation

Industry standard validation checks were completed on the supplied database and minor corrections made. P&E validates a mineral resource database by checking for inconsistencies in naming conventions or analytical units, duplicate entries, interval, length or distance values less than or equal to zero, blank or zero-value assay results, out of sequence intervals, intervals or distances greater than the reported drillhole length, inappropriate collar locations and missing interval, and coordinate fields. No significant discrepancies with the supplied data were noted.

Downhole surveys were completed by Silver Standard with a Reflex EZ-Shot® magnetic instrument. Measurements were taken every 100 m unless drastic deviations occurred, in which case additional measurements were taken every 50 m to eliminate error. Downhole survey data were examined by P&E for significant deviations. Of the 141 drillholes in the database, 20 drillholes displayed one or more downhole survey deviations from the previous measurement of greater than 5°. Drillhole orientations were also examined and appear appropriate for the local geology (Figure 17.1).

Figure 17.1 Drillhole Orientation



17.5 TOPOGRAPHIC CONTROL

Silver Standard contracted McElhanney to produce a detailed topographic plan of the Snowfield project area. This plan, drafted at 1:2,000 scale and displayed with the North American Datum 1927 (NAD27) Zone 9 UTM grid, covers an area of 2.85 km², with the northwest corner of the area located at 423,900 m E, 6,265,500 m N and the southeast corner having coordinates 425,400 m E, 6,263,600 m N. To generate the topographic contours for this area, with the contour interval being at 1 m, McElhanney used a Light Detection and Ranging (LIDAR) satellite image of the area along with 36 field control points that were surveyed with a Leica 500 GPS instrument during the summer months of 2006 through 2009. In addition, the locations of 101 diamond drillhole collars that were surveyed by McElhanney field crews during the four field seasons were incorporated into the database, which generated the topographic contours. The McElhanney topography map was originally produced with a NAD83, Zone 9 UTM grid system, which was then converted to the NAD27 system using national Transformation Model NTv2Points.

17.6 BULK DENSITY

A total of 439 bulk density measurements were provided by Silver Standard, with an average bulk density of 2.78 t/m^3. Density measurements were obtained from core samples by ALS Chemex. Bulk density measurements were back-tagged to the relevant domain and assigned as bulk density values for mineral resource estimation (Table 17.4).

Table 17.4 Bulk Density Statistics

	Mineralized Domains	Non-Mineralized Zones
Count	371	68
Minimum	2.34	2.50
Maximum	3.17	3.02
Average	2.78	2.80

17.7 DOMAIN MODELING

In conjunction with Silver Standard, P&E modeled a primary 0.5 g/t Au mineralization domain for mineral resource estimation. The mineralization domain was created by computer screen digitizing of successive polylines on 90 drillhole sections spaced 25 m a part. The outlines of the polylines were defined by the selection of mineralized material at/or above 0.5 g/t Au, with demonstrated continuity along strike and down dip. In some cases, mineralization below 0.5 g/t Au was included for the purpose of maintaining continuity. Sectional polyline interpretations were digitized from drillhole to drillhole but typically not extended more than the distance between two sections. A three-dimensional model of the mineralization domain was then created by combining successive polylines into a wireframe.

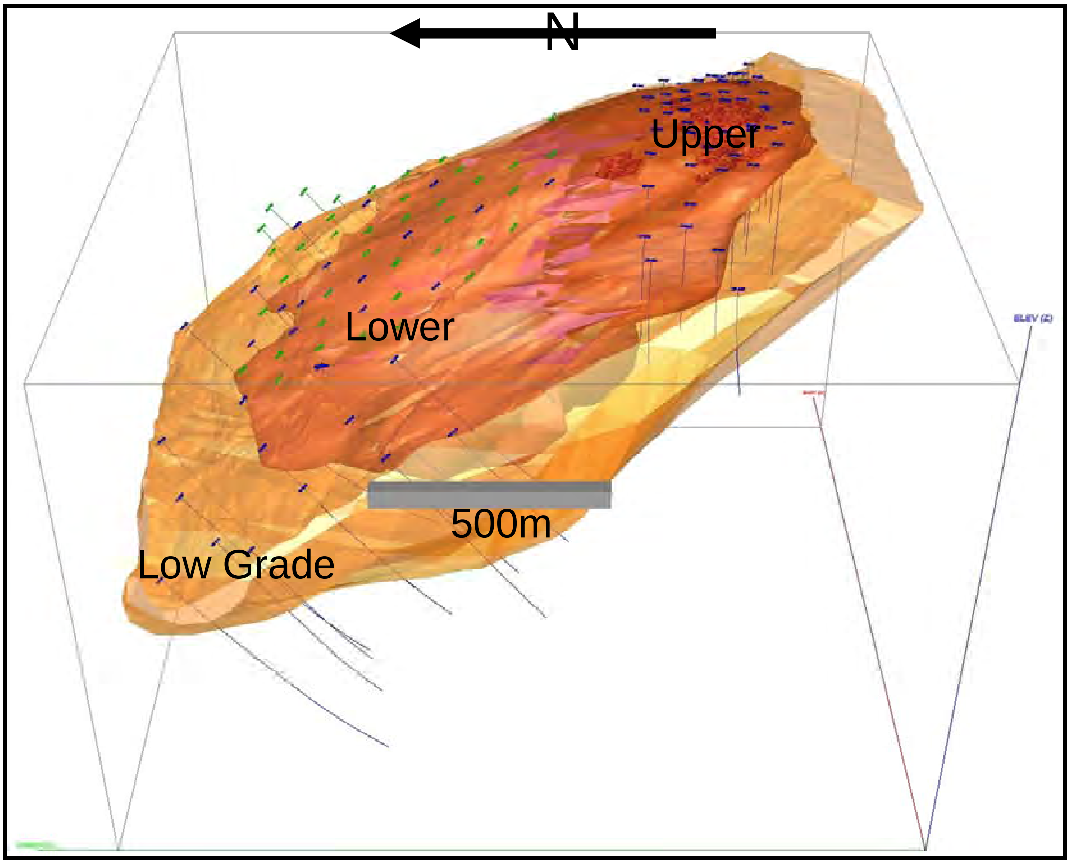
In order to ensure that all potential economic mineralization was captured for mineral resource estimation, a secondary low grade mineralization domain was subsequently modeled using a 0.2 g/t Au-equivalent cut-off value based on the parameters listed in Table 17.5.

Table 17.5 Au-Equivalent Parameters

Commodity	Price (US$)	Recovery (%)	Au Equivalency
Au	800.00/oz	75%	1.00
Ag	12.00/oz	73%	0.015
Cu	2.50/lb	85%	2.43
Mo	10.00/lb	60%	6.86

Examination of the assay data indicates the presence of two distinct domains within the defined primary mineralization domain; an upper Au-rich/Cu-poor zone and a lower Au-poor/Cu-rich zone (Figure 17.2). Therefore, P&E further divided the primary mineralization wireframe into two domains, designated as the upper and lower domains, respectively. The resulting domains were treated as hard boundaries for mineral resource estimation and used for statistical analysis, grade interpolation, and rock coding.

Figure 17.2 Isometric View of the Snowfield Domains, Topography Removed



17.8 COMPOSITING

Assay sample lengths for the database range from 0.22 m to 13.51 m, with an average sample length of 1.53 m. A compositing length of 1.50 m was, therefore selected for use during estimation.

Length-weighted composites were calculated for Au, Ag, Cu, and Mo within the defined mineralization domains. The compositing process started at the first point of intersection between the drillhole and the domain, intersected and halted upon exit from the domain wireframe. Composites that were less than 0.5 m in length were discarded so as to not introduce a short sample bias into the estimation process. The wireframes that represented the interpreted mineralization domains were also used to back-tag a rock code field into the drillhole workspace. Assays and composites were assigned a domain rock code value based on the domain wireframe that the interval midpoint fell within. The composite data were then exported to Gemcom extraction files for grade estimation.

17.9 Exploratory Data Analysis

Summary assay statistics (Table 17.6) and composite statistics (Table 17.7) were calculated by domain for each commodity. Comparison of the data sets indicates that no significant bias was introduced from the compositing process. A comparison of the data sets also demonstrates the difference in grade distributions within the domains (Figure 17.3).

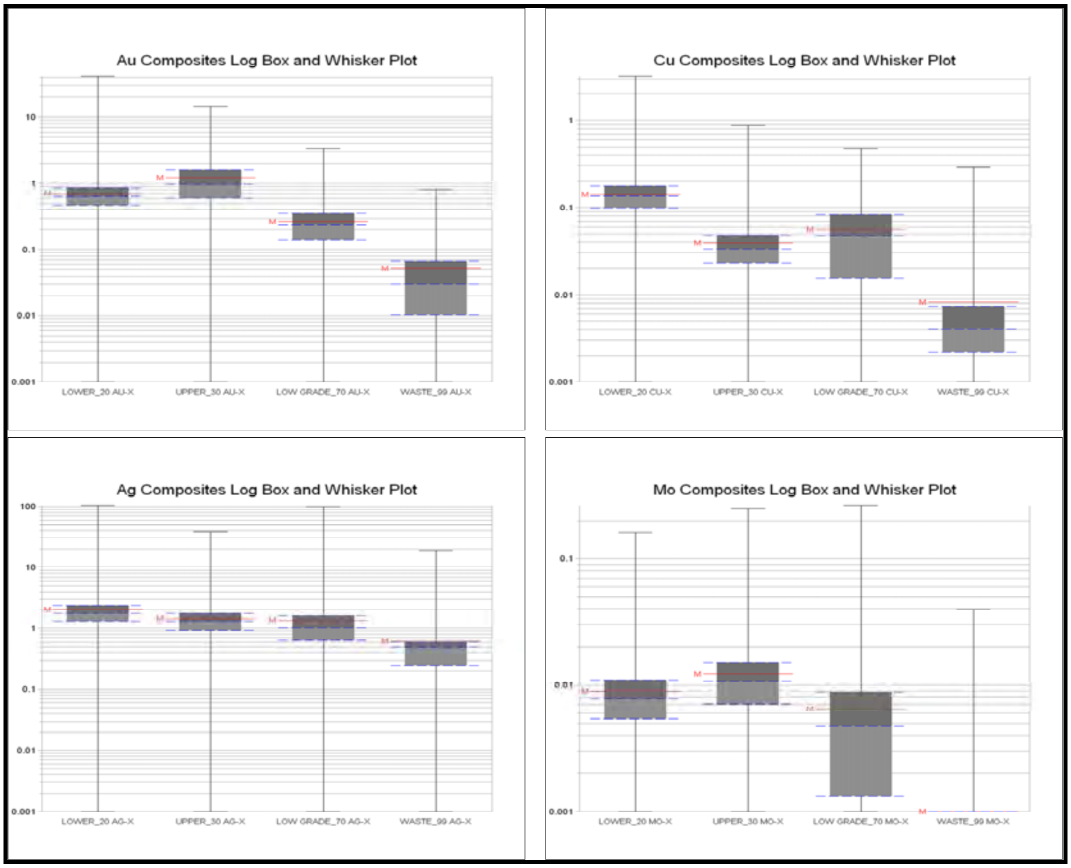
Table 17.6 Summary Assay Statistics by Domain

	Total	Lower Domain	Upper Domain	Low Grade	Waste
Au (g/t)					
Samples	35,585	16,660	5,415	10,565	2,945
Minimum	0.0025	0.0025	0.0030	0.0025	0.0025
Maximum	53.8000	53.8000	14.5500	4.2600	0.6350
Mean	0.6057	0.7075	1.2076	0.2923	0.0474
Standard Deviation	0.6575	0.5993	0.9362	0.2298	0.0534
CV	1.0856	0.8470	0.7753	0.7865	1.1250
Ag (g/t)					
Samples	35,588	16,660	5,418	10,565	2,945
Minimum	0.2500	0.2500	0.3000	0.2500	0.2500
Maximum	110.0000	110.0000	38.1000	100.0000	21.5000
Mean	1.6402	2.0216	1.4191	1.4279	0.6508
Standard Deviation	1.9187	2.0123	0.9680	2.1795	0.8738
CV	1.1698	0.9954	0.6821	1.5264	1.3426
Cu (%)					
Samples	35,588	16,660	5,418	10,565	2,945
Minimum	0.0001	0.0006	0.0008	0.0001	0.0001
Maximum	4.0900	4.0900	0.8750	0.4860	0.3330
Mean	0.0912	0.1421	0.0392	0.0609	0.0075
Standard Deviation	0.0776	0.0755	0.0305	0.0493	0.0120
CV	0.8512	0.5314	0.7798	0.8105	1.5953
Mo (%)					
Samples	35,584	16,660	5,416	10,563	2,945
Minimum	0.0001	0.0001	0.0002	0.0001	0.0001
Maximum	0.2900	0.1620	0.2570	0.2900	0.0574
Mean	0.0081	0.0088	0.0121	0.0070	0.0004
Standard Deviation	0.0080	0.0067	0.0094	0.0084	0.0016
CV	0.9848	0.7603	0.7760	1.1931	3.9491

Table 17.7 Summary Composite Statistics by Domain

	Total	Lower Domain	Upper Domain	Low Grade	Waste
Au (g/t)					
Samples	37,166	17,202	5,667	11,265	3,032
Minimum	0.0010	0.0010	0.0010	0.0010	0.0010
Maximum	42.0283	42.0283	14.4872	3.4435	0.8155
Mean	0.5964	0.7022	1.2218	0.2668	0.0516
Standard Deviation	0.6189	0.5164	0.9019	0.2088	0.0672
CV	1.0377	0.7355	0.7382	0.7827	1.3028
Ag (g/t)					
Samples	37,166	17,202	5,667	11,265	3,032
Minimum	0.0010	0.0010	0.0010	0.0010	0.0010
Maximum	104.2340	104.2340	38.100	100.000	19.0080
Mean	1.6105	2.0073	1.4356	1.3608	0.6141
Standard Deviation	1.8296	1.7644	0.9801	2.2427	0.8463
CV	1.1361	0.8790	0.6827	1.6481	1.3781
Cu (%)					
Samples	37,166	17,202	5,667	11,265	3,032
Minimum	0.0001	0.0010	0.0010	0.0001	0.0001
Maximum	3.2329	3.2329	0.8744	0.4802	0.2862
Mean	0.0886	0.1406	0.0389	0.0559	0.0082
Standard Deviation	0.0749	0.0706	0.0279	0.0477	0.0149
CV	0.8450	0.5020	0.7169	0.8519	1.8106
Mo (%)					
Samples	37,166	17,202	5,667	11,265	3,032
Minimum	0.0001	0.0001	0.0002	0.0001	0.0001
Maximum	0.2645	0.1614	0.2510	0.2645	0.0396
Mean	0.0080	0.0089	0.0122	0.0064	0.0006
Standard Deviation	0.0074	0.0063	0.0086	0.0074	0.0017
CV	0.9262	0.7056	0.7054	1.1483	2.9006

Figure 17.3 Box and Whisker Plots of Composite Statistics by Domain



Sample assay populations drawn from the trenching data and the drillhole data were also examined by commodity for the upper domain. The trenching assay data show a positive bias for Au and Ag when compared to the drillhole data. A bias of this type often occurs in trenching data, and is typically the result of weathering, preferential sampling by the geologist, over-collection of softer mineralized material during sampling, or any combination of the above. Therefore, the trenching data were used while defining the extent of the mineralization domains, but were not used for mineral resource estimation.

17.10 TREATMENT OF EXTREME VALUES

The presence of high-grade outliers was evaluated by examining composite CV cutting graphs, histograms, and log-probability graphs for the defined mineralization domains.

Cutting graphs indicate inflection points where a rapid change in the standard deviation or the mean is occurring.

Threshold values were selected that minimize changes in the composite sample distribution (Table 17.8). The influence of composite samples equal to or higher than the threshold value selected was restricted during estimation to 30 m. The use of a threshold strategy for a low-grade deposit honours the true distribution of the sample values while restricting the influence of high-grade outliers during linear estimation.

Table 17.8 Threshold Values

Commodity	Lower	Upper	Low Grade
Ag	9.0g/t	5.0g/t	11.0g/t
Au	3.0g/t	5.0g/t	2.0g/t
Cu	0.14%	0.14%	0.25%
Mo	0.04%	0.14%	0.04%

17.11 Variography

For the upper and lower domains, anisotropy was determined for each commodity from ellipsoids fitted to directional exponential correlograms oriented along azimuths spaced 30° apart and calculated at dips of 0°, 30°, 60°, and 90°. The resulting ellipsoid axes were used as the basis for estimation search ranges, distance calculations, and mineral resource classification (Table 17.9). The correlograms represent ranges of continuity, which somewhat commensurate with the drilling spacing and, therefore, cannot be considered to be truly representative of the underlying mineralization.

Table 17.9 Upper and Lower Domain Anisotropy Definitions

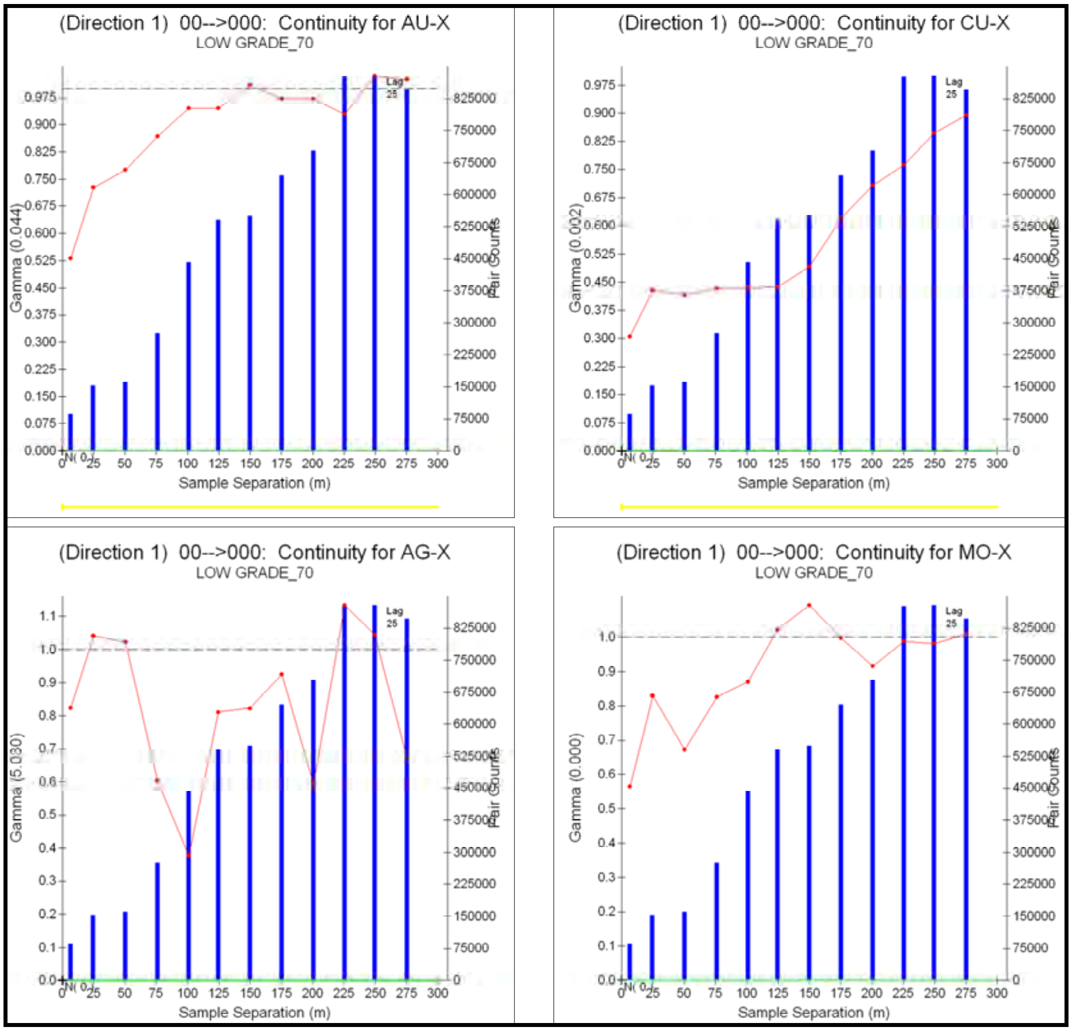
		Upper Domain			Lower Domain		
		Range (m)	Azimuth (°)	Dip (°)	Range (m)	Azimuth (°)	Dip (°)
Au	Z	198	299	55	323	314	33
	Y	333	347	-26	1077	332	-55
	X	651	065	23	203	050	9
	CO	0.341			0.787		
	C1	0.659			0.213		
Ag	Z	265	260	50	150	298	68
	Y	437	299	-34	862	317	-21
	X	181	015	20	162	045	7
	CO	0.581			0.735		
	C1	0.419			0.265		
Cu	Z	1,077	196	66	273	297	39
	Y	278	000	23	1187	325	-47
	X	1298	093	6	231	039	14
	CO	0.486			0.431		
	C1	0.514			0.569		
Mo	Z	270	298	76	195	281	50
	Y	467	267	-12	500	327	-30
	X	166	358	-7	231	043	23
	CO	0.767			0.529		
	C1	0.233			0.471		

For the low grade domain, isotropic semi-variograms were calculated from uncapped composite data in order to determine applicable estimation search ranges and mineral resource classification criteria (Table 17.10 and Figure 17.4).

Table 17.10 Low Grade Domain Ranges

Commodity	Range (m)
Au	150 m
Ag	200 m
Cu	300 m
Mo	125 m

Figure 17.4 Low Grade Domain Semi-Variograms



17.12 BLOCK MODEL

An orthogonal block model was established across the property (Table 17.11) consisting of separate models for Au estimated grades, Ag estimated grades, Cu estimated grades, Mo estimated grades, associated rock codes, percent, density and classification attributes, and a calculated Au-equivalent grade (AuEq). A percent block model was used to accurately represent the volume and tonnage that was contained within the constraining low grade mineralized domain. As a result, domain boundaries were properly represented by the percent model's capacity to measure infinitely variable inclusion percentages within a specific domain.

Table 17.11 Block Model Setup

	Origin (m)	No. of Blocks	Size (m)
X	423,600	90	25
Y	6,263,500	90	25
Z	1,850	135	10
Rotation	None		

17.13 Estimation and Classification

Inverse Distance Squared (ID²) linear weighting of composite values was used for the estimation of block grades. Composite data used during estimation were limited to samples located within their respective mineralization domain. Individual block grades were then used to calculate an Au-equivalent grade model.

For the upper and lower domains a three-pass series of expanding search ellipses with varying minimum sample requirements was used for sample selection and classification. Sample distances were adjusted by the defined anisotropy:

- During the first pass, twelve composite values from three or more drillholes within a search ellipse corresponding to 15% of the defined variogram ranges were required for estimation. All block grades estimated during the first pass were classified as measured, with a total of 7,962 blocks estimated.
- During the second pass, blocks not populated during the first pass were estimated. Twelve composite values from three or more drillholes within a search ellipse corresponding to 100% of the defined variogram ranges were required for estimation. All block grades estimated during the second pass were classified as indicated, with a total of 39,992 blocks estimated.
- During the third pass, blocks not populated during the first or second pass were estimated. Three to twelve composite values within a search ellipse corresponding to 200% of the defined variogram ranges were required for estimation. All block grades estimated during the third pass were classified as Inferred, with a total of 2,591 blocks estimated.

For the low grade domain, two passes were used for sample selection and estimation:

- During the first pass, twelve composite values from three or more drillholes within 75 m of the block centroid were required for estimation. All block grades estimated during this pass were classified as indicated, with a total of 2,567 blocks estimated.

- During the second pass, blocks not populated previously were estimated. Three to twelve composite values from one or more drillholes within 450 m of the block centroid were required for estimation. All block grades estimated during this pass were classified as Inferred, with a total of 91,242 blocks estimated.

17.14 MINERAL RESOURCE ESTIMATE

Mineral resources were classified in accordance with guidelines established by CIM November 11, 2005:

- Inferred Mineral Resource: An inferred mineral resource is part of a mineral resource for which quantity, grade, or quality can be estimated on the basis of geological evidence, 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 drillholes.
- Indicated Mineral Resource: An indicated mineral resource is 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 drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.
- Measured Mineral Resource: A measured mineral resource is 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 drillholes that are spaced closely enough to confirm both geological and grade continuity.

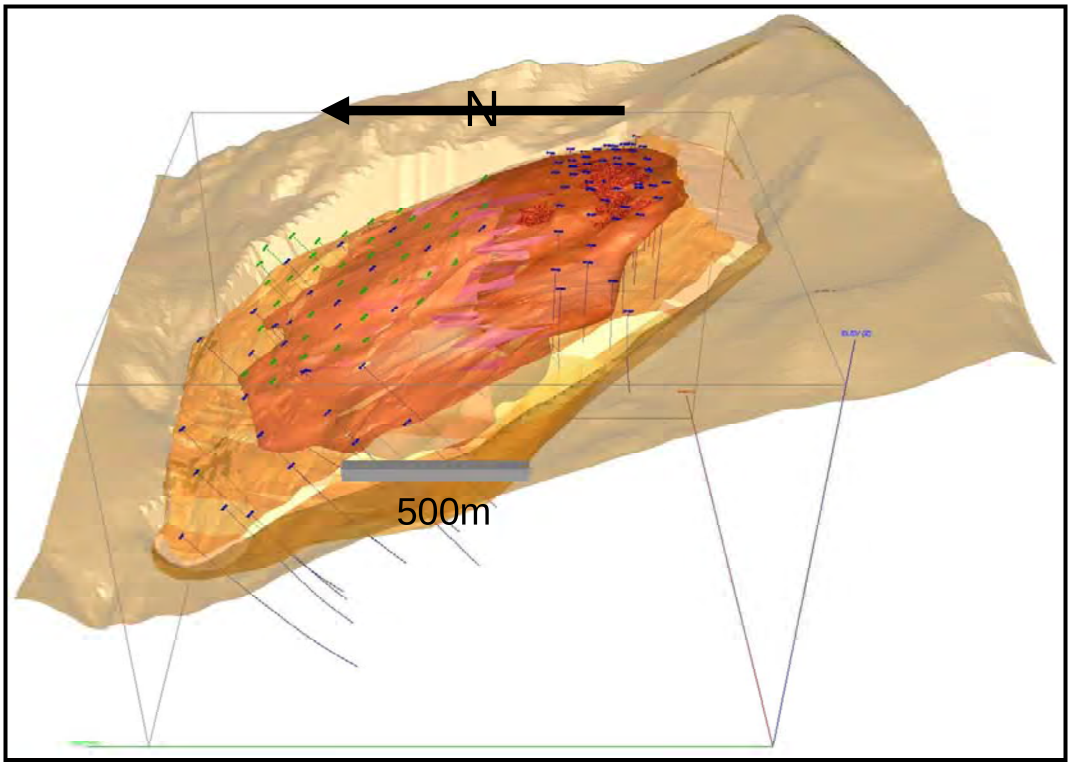
In order to ensure that the reported mineral resources meet the CIM requirement for “reasonable prospects for economic extraction” a conceptual Lerchs-Grossman (LG) optimized pit shell was developed based on all available mineral resources (measured, indicated, and inferred), using the economic parameters listed in Table 17.12 and Figure 17.5.

Based on knowledge of mineral resource projects in the vicinity of Snowfield, Silver Standard mandated the use of a 0.35 g/t AuEq cut-off for the reporting of mineral resources at Snowfield. An average recovery value for each commodity was applied across the block model. The results from the optimized pit-shell are used solely for the purpose of reporting mineral resources that have reasonable prospects for economic extraction.

Table 17.12 Optimized Pit-Shell Parameters

Optimized Pit Shell Parameters	
Tailings & Water	$0.80/rock tonne
Mining Cost	$1.75/rock tonne
Processing Cost	$5.00/ore tonne
Process Recovery	AU 75%, Ag 73%, Cu 85%, Mo 60%
G & A	$1.00/ore tonne
Pit Wall Slope Angle	50°

Figure 17.5 Conceptual Optimized Pit Shell



All mineral resources were tabulated against a mandated 0.35 g/t Au equivalent cut-off, as constrained within the optimized pit shell (Table 17.13).

Table 17.13 Mineral Resource Estimate at a 0.35 g/t AuEq Cutoff[1,2,3]

Class	Mt	Au (g/t)	Au (M oz)	Ag (g/t)	Ag (M oz)	Cu (%)	Mo (ppm)
Measured	136.9	0.94	4.14	1.7	7.7	0.11	99
Indicated	724.8	0.67	15.63	1.9	43.2	0.12	91
Measured + Indicated	861.7	0.71	19.77	1.8	50.9	0.12	92
Inferred	948.9	0.33	10.05	1.4	43.7	0.07	81

[1] Mineral resource sensitivities are accumulated within an optimized pit shell.
[2] Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
[3] The quantity and grade of reported inferred resources in this estimation are conceptual in nature. There is no guarantee that all or any part of the mineral resource will be converted into mineral reserve.

To demonstrate the economic sensitivity of the deposit, estimated mineral resources were also tabulated using a cut-off grade of 1.0 g/t Au equivalent. Total estimated mineral resources at this cut-off are comprised of measured and indicated Au mineral resources of 13.51 M ounces and inferred Au mineral resources of 0.59 M ounces (Table 17.14).

Table 17.14 Mineral Resource Sensitivity Demonstrated at 1.0 g/t AuEq Cut-off[1,2,3]

Class	Tonnes (x M)	Au (g/t)	Au (oz x M)	Ag (g/t)	Ag (oz x M)	Cu (%)	Mo (ppm)
Measured	97.9	1.10	3.45	1.8	5.8	0.12	104
Indicated	371.9	0.84	10.06	2.1	24.5	0.15	90
Measured + Indicated	469.8	0.89	13.51	2.0	30.3	0.14	93
Inferred	25.8	0.71	0.59	2.0	1.7	0.12	100

[1] Mineral resource sensitivities are accumulated within an optimized pit shell.
[2] Mineral resources which are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
[3] The quantity and grade of reported inferred resources in this estimation are conceptual in nature. There is no guarantee that all or any part of the mineral resource will be converted into mineral reserve.

17.15 VALIDATION

The block model was validated visually by the inspection of successive section lines in order to confirm that the block model correctly reflects the distribution of high-grade and low-grade samples. An additional validation check of the mineral resource estimate was completed by comparing average composite grades to the grade of the block containing the composites (Figure 17.6). The observed differences in grades suggest a minimal conditional bias, and are deemed acceptable for mineral resource estimation.

An additional validation check for global bias was also completed by comparing the ID2 block model estimates to a Nearest Neighbour (NN) block model estimate generated using the same search criteria and tabulated at a zero cut-off within the constraining pit-shell. Results demonstrate a minimal global bias and very slight smoothing for the ID^2 estimate as compared to the NN estimate (Figure 17.7) and correctly duplicate grade trends.

Figure 17.6 Block Grades vs. Average Composite Grades

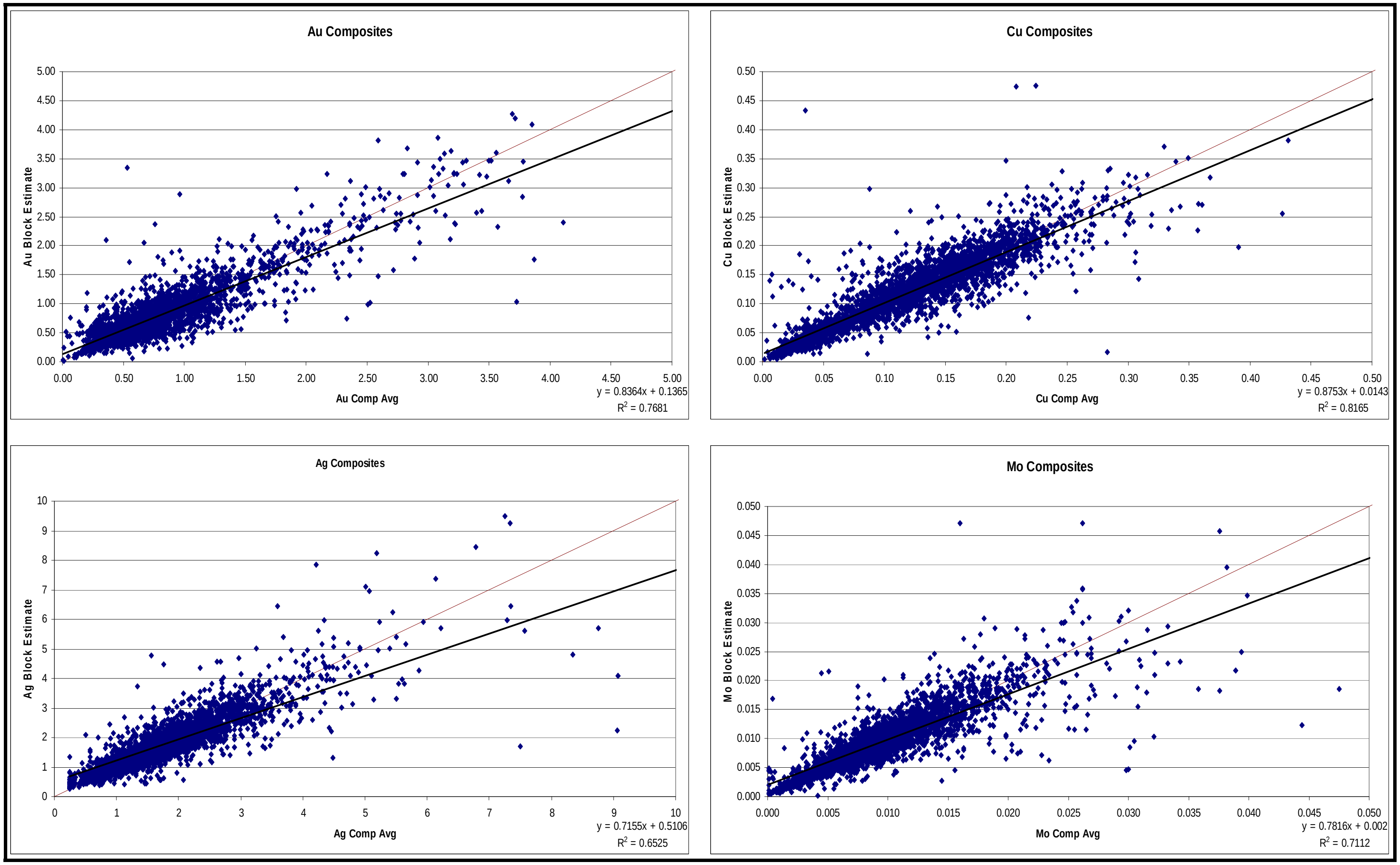
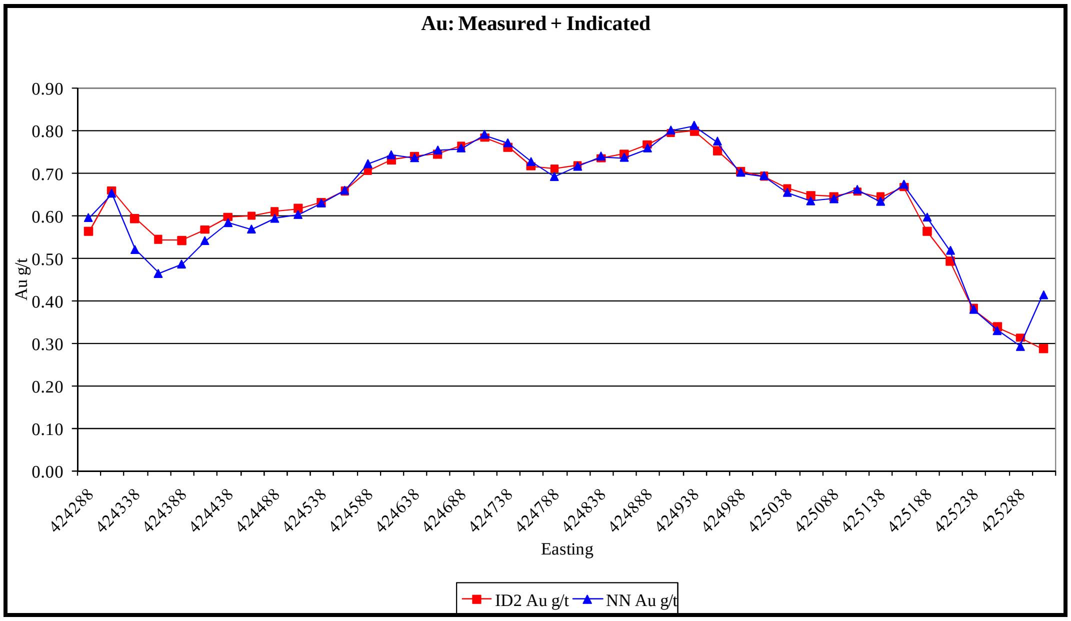


Figure 17.7 Section and Plan Swath Plots



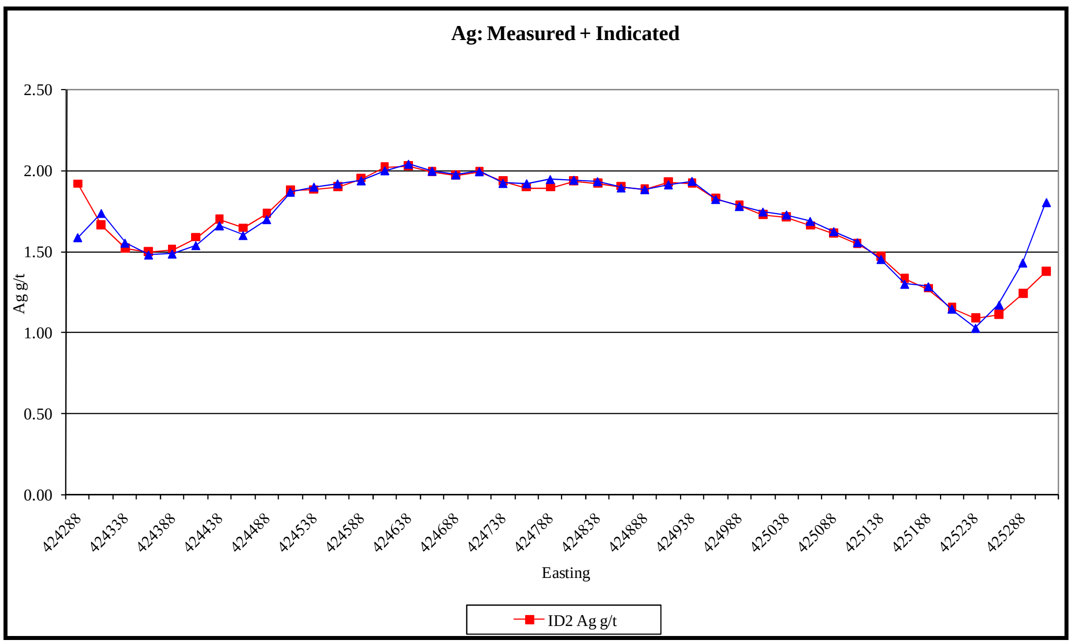
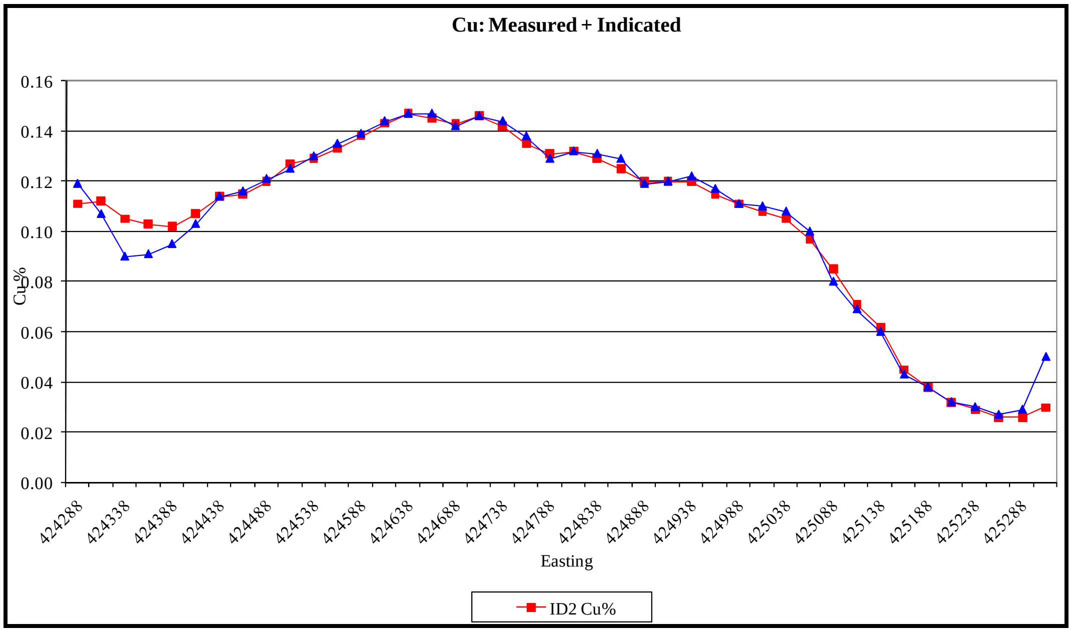


Figure 17.7 (con't) Section and Plan Swath Plots



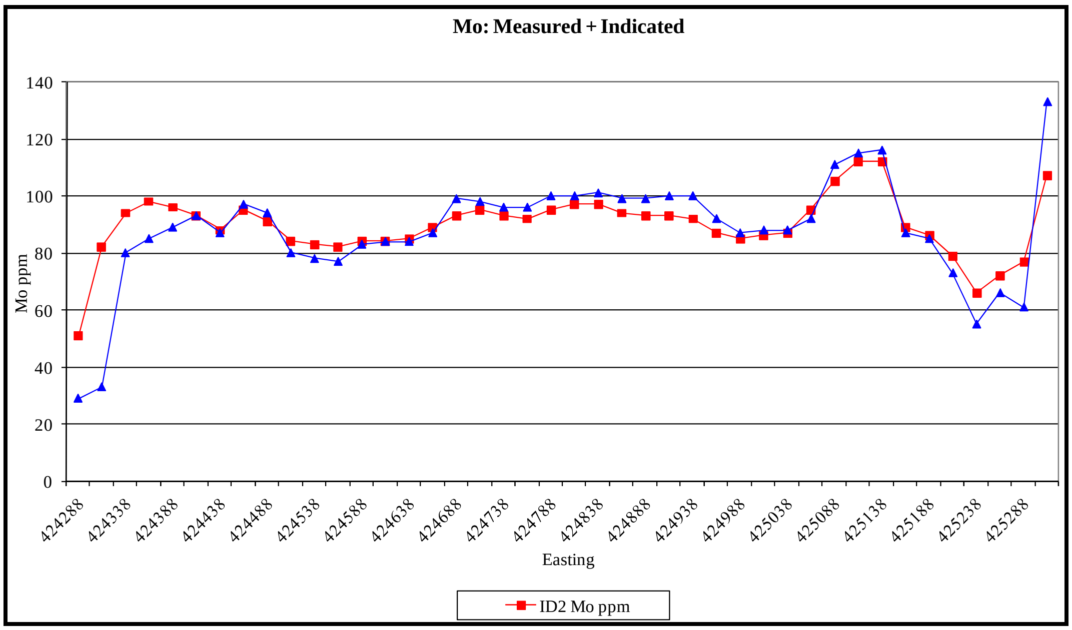
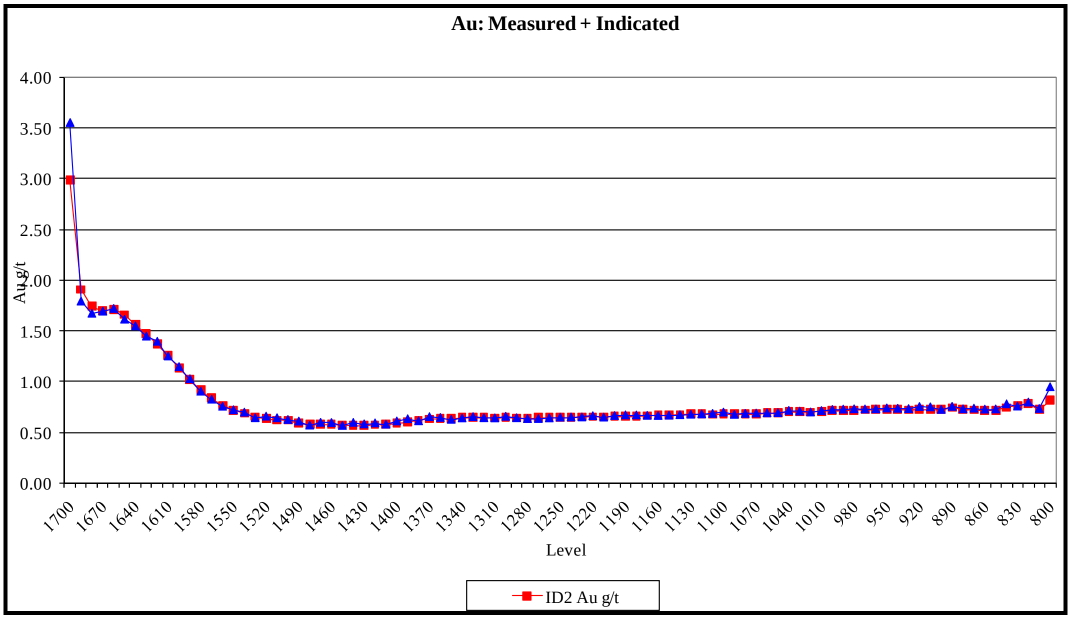


Figure 17.7 (con't) Section and Plan Swath Plots



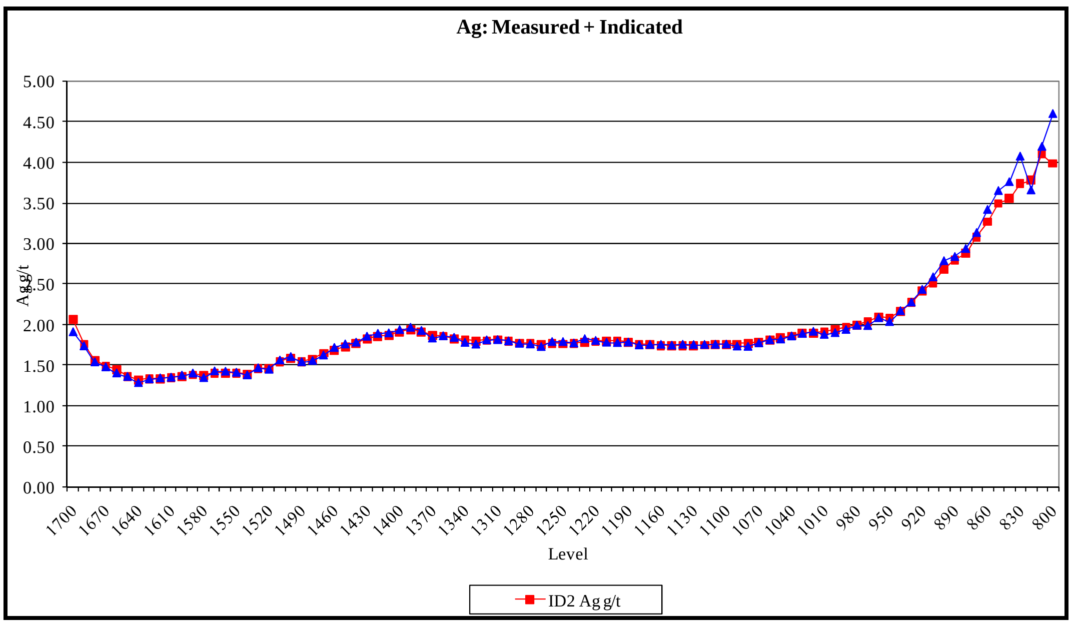
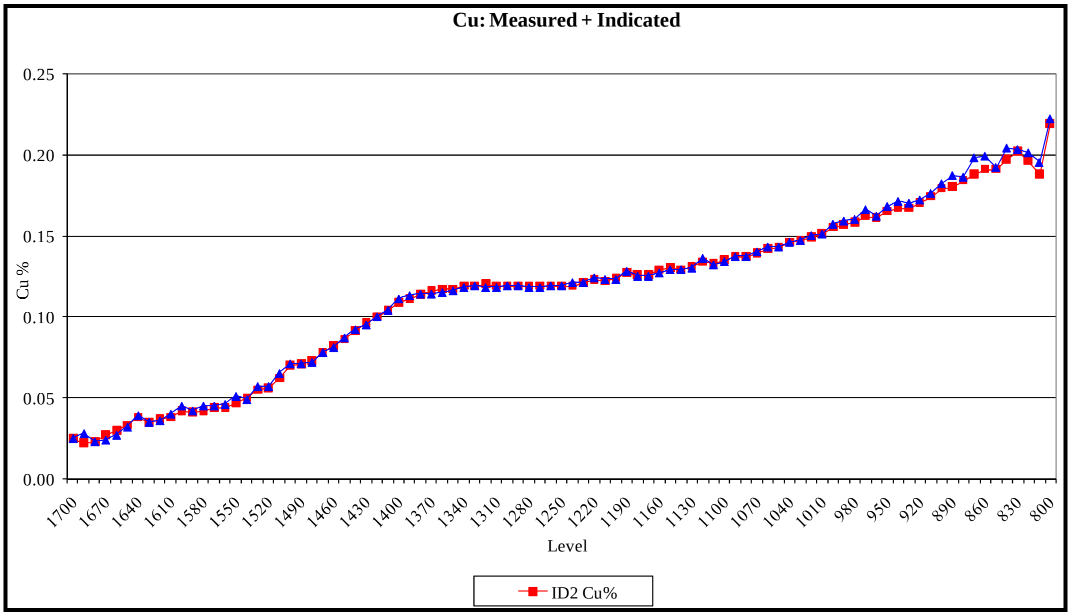
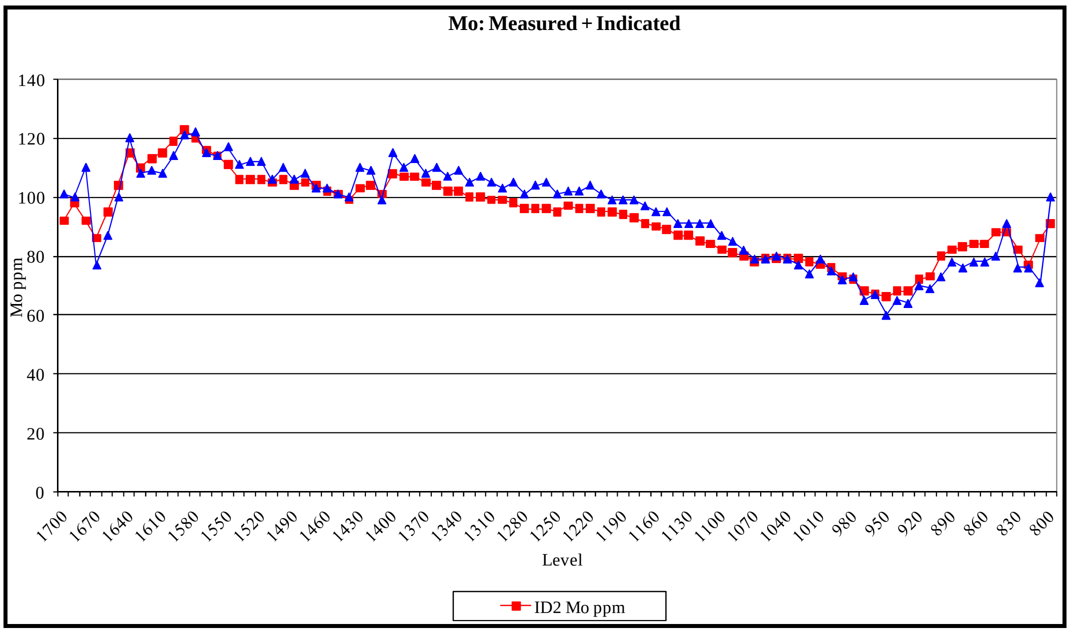


Figure 17.7 (con't) Section and Plan Swath Plots





18.0 OTHER RELEVANT DATA AND INFORMATION

18.1 MINING

The Snowfield property is located within the Sulphurets District in the Iskut River region, approximately 20 km northwest of Bowser Lake or 65 km north-northwest of the town of Stewart, BC.

The deposit will be mined using a bulk open pit mining method to provide a mill feed at a nominal rate of 120,000 t/d (43.8 Mt/a). At the mine's operating peak, 100.0 Mt/a of material will be mined, with a LOM waste-to-mineralized material ratio of approximately 0.54:1.

The operation will focus on application of the largest available mining equipment to reduce unit cost. The primary equipment fleet will consist of three 311 mm blasthole drills, two 45 m^3 electric cable shovels, three diesel hydraulic shovels, and thirty 363 t haul trucks.

The primary equipment will be supported by track- and rubber-tired bull dozers, motor graders, a compactor, a water truck, a small excavator, and other ancillary equipment.

The mineralized material and waste material will be mined in 15 m benches A double-bench configuration was assumed for the final pit walls, resulting in 30 m vertical height between catch benches. The overall mining sequence was developed through a series of five scoping level mining pushbacks. The sequence was designed to:

- gain early access to the higher grade mineralized material in the upper mineralized material zone to facilitate early capital recovery
- defer waste stripping to later years by positioning haul roads along topography to minimize initial capital investment
- develop an ore blending strategy to maximize the NPV by mining higher Au grade ore during the initial years and mining higher Cu grade ore to constantly maintain high NSR values
- locate primary crusher to as low an elevation as possible to reduce haulage distance and consequently reduce initial capital and mine operating costs

- develop a conceptual integration of the operation of peripheral wells and gravity drainage of surface run-off to minimize surface water from entering the open pit.

18.1.1 INTRODUCTION

The mine planning work for the scoping study of the Snowfield property was based on NI 43-101 published resource models dated January 14, 2010.

Mine planning was conducted through the application of Whittle® and Surpac® software packages. This includes block model manipulations, pit optimization, conceptual planning, and preliminary assessment level production scheduling.

In addition to the block model, other data used for the mine planning includes the base economic parameters, mining and milling cost data derived from other projects in northern BC, recommended preliminary pit slope angles by BGC, and estimated project metallurgical recoveries.

18.1.2 3D BLOCK MODEL

The original Snowfield block model does not extend far enough into the southern direction. To accommodate the size of the optimal pit shell generated in the optimization, it was necessary to extend the block model. The text file was imported to a Surpac software block model. Details of the extended block model are shown in Table 18.1.

Table 18.1 Details of the Block Model

Type	Y	X	Z
Minimum Coordinates	6,262,550	423,250	500
Maximum Coordinates	6,266,200	426,450	2,000
User Block Size	25	25	10
Minimum Block Size	25	25	10
Rotation	0	0	0

The block model contained rock type, density for Au, Ag, Cu, Mo, class AuEq, and percent of mineralized material inside the low-grade mineralization halo.

RESOURCE VALIDATION

In order to compare P&E's 2009 published resource estimate with the current model, the original Au equivalent cut-off was used to query the block model within the P&E optimized pit shell. The original Au equivalent was calculated using the metal prices, recoveries and other parameters listed in Table 18.2, which was obtained from the

Technical Report and Updated Resource Estimate on the Snowfield Property dated December 1, 2009.

Table 18.2 Mineral Resource Estimate at a 0.35 g/t AuEq Cut-off

Class	Mt	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Au (M oz)	Ag (M oz)	Cu (M lb)	Mo (M lb)
P&E									
Measured	136.9	0.94	1.70	0.11	0.0099	4.14	7.70	0.15	0.014
Indicated	724.8	0.67	1.90	0.12	0.0091	15.63	43.20	0.87	0.066
Measured + Indicated	861.7	0.71	1.80	0.12	0.0092	19.77	50.90	1.02	0.080
Inferred	948.9	0.33	1.40	0.07	0.0081	10.05	43.70	0.66	0.077
Total Resource	1,810.6	0.51	1.59	0.09	0.0086	29.82	94.60	1.68	0.156
Wardrop									
Measured	137.3	0.94	1.75	0.11	0.0099	4.15	7.71	0.15	0.014
Indicated	726.4	0.67	1.85	0.12	0.0091	15.67	43.28	0.89	0.066
Measured + Indicated	863.6	0.71	1.84	0.12	0.0092	19.83	50.99	1.04	0.080
Inferred	960.8	0.33	1.43	0.07	0.0081	10.20	44.31	0.69	0.078
Total Resource	1,824.4	0.51	1.62	0.09	0.0086	30.02	95.30	1.73	0.158
Percent Difference (%)									
Measured	-0.3	0.1	-2.8	3.5	0.0	-0.3	-0.2	3.3	-0.3
Indicated	-0.2	-0.2	2.5	-2.1	0.0	-0.3	-0.2	-2.3	-0.2
Measured + Indicated	-0.2	-0.6	-2.0	0.1	0.0	-0.3	-0.2	-1.5	-0.2
Inferred	-1.3	0.0	-2.5	-3.0	0.0	-1.5	-1.4	-4.3	-1.3
Total Resource	-0.8	-0.2	-2.2	-1.0	0.0	-0.7	-0.7	-2.6	-0.7

Table 18.2 compares the resource tonnage, metal grades and metal values for the different resource classes. The comparison for the total resource between P&E and Wardrop indicates that:

- P&E's total resource tonnage is about 0.8% lower than that of Wardrop's.
- P&E's metal grades in the total resource is about 0.2% lower for Au, 2.2% lower for Ag, and 1.0% for Cu and 0.0% for Mo, than that of Wardrop's.
- P&E's contained metal values in the total resource is about 0.7% lower for Au, 0.7% lower for Ag, 2.6% lower for Cu and 0.7% for Mo, than that of Wardrop's.

P&E and Wardrop's resource estimates are reasonably close, and within the accuracy of the estimates. Wardrop therefore concludes that the two models are similar.

18.1.3 WHITTLE PARAMETERS

Wardrop used the LG algorithm application in Gemcom Software International Inc.'s Whittle® application, supplemented by GEMSTM mine planning software, to perform the pit optimization. The Whittle® input parameters are explained below.

PRODUCTION RATE

A production rate of 120,000 t/d mill feed was confirmed by Silver Standard as the basis for the pit optimization. This rate was selected based on the preliminary consideration of various factors such as resource tonnage and power supply, and confirmed as a favourable production rate for the project in the course of this study (Table 18.3).

METAL PRICES AND EXCHANGE RATE

Metal prices to be used as pit optimization parameters for this project were provided by Silver Standard and are shown in Table 18.3.

Table 18.3 Metal Prices

Commodity	Metal Price (US$)
Cu	2.35/lb
Au	800/oz
Ag	12.55/oz
Mo	13.91/lb
Exchange Rate	0.92

Note: The metal prices shown are used as a pit optimization input parameters only. Different metal prices were used for project economic evaluation.

PROCESS RECOVERIES

Metal recoveries were projected according to available metallurgical test results. They were provided by the project metallurgical engineer and are presented in Table 16.15.

SMELTER TERMS AND DEDUCTIONS

Smelter terms and deductions used as pit optimization input parameters are shown in Table 18.4.

Table 18.4 Smelter Terms and Deductions

Items	Units	US$	C$	Value
CONCENTRATE				
Copper Concentrate Grade	%			22.0
Molybdenum Concentrate Grade	%			50.0
Moisture Content	%			9.0
METAL PAYABLE				
Copper Concentrate				
Copper	%			99.0
Gold	%			97.5
Silver	%			90.0
Molybdenum Concentrate				
Molybdenum including losses	%			97.5
Gold and Silver Doré				
Gold	%			99.8
Silver	%			99.8
CONCENTRATE TREATMENT TERMS				
Smelting	$/dmt conc.	85.00		
Refining				
Copper	$/acc lb	0.085		
Silver	$/acc oz	0.450		
Price Participation – above Base Cu Price	%			1.5
Base Copper Price	$/lb	1.500		
Capped	$/lb	0.040		
Roasting				
Molybdenum	$/lb	1.500		
CONCENTRATE TRANSPORTATION				
Truck	$/wmt		$25.00	
Port	$/wmt		$25.00	
Ocean	$/wmt	65.00		
Moisture	%			9.0
Concentrate losses (during transport & Rehandle)	%NIV			0.50
Insurance	%NIV			0.15
Representation	$/wmt	0.50		
GOLD & SILVER DORÉ				
Combined smelting &transport costs	$/oz	2.00		
Insurance	%NIV			0.15
Representation (based on copper concentrate ratio	%NIV			0.02

MINERALIZED MATERIAL DILUTION AND MINING RECOVERY

Mining activities will cause dilution to the blocks (either mineralized material into waste or waste into mineralized material) where contact is made between mineralized material and waste, depending on the cut-off grade. In addition, misdirected loads and haul-back in frozen truck boxes will cause mining losses and dilution as material is moved from the mine site to the conveying system.

Internal dilution refers to waste material within the orebody that, due to mining constraints, cannot be physically separated from the mineralized material. It is typically included in the mineralized material grade estimates. External dilution relates to the material outside of the in-place, pre-blasted mineralized material block boundaries; it is not included in the mineralized material grade estimates. Typically, external dilution can be tracked by the reconciliation of truck counts and average truck tonnage factors to the in-place ore block tonnes. The ability of a large shovel to mine precisely along the limits of the mineralized material zone is a trade-off between minimizing dilution and increasing operating costs.

A sensitivity analysis on mineralized material dilution and mining recovery is provided in Table 18.5.

Table 18.5 Dilution Sensitivity

mineralized material Dilution (%)	Mining Recovery (%)	Change in NPV (%)	Comments
0	95.0	-4.9	
2.5	97.5	-5.1	
3	97.0	-5.8	Base Case
5	95.0	-9.8	

For this scoping level of study, a preliminary allowance was made for an internal mining dilution of 3% and an external mining loss of 3%. The diluting mineralized material grades were assumed to be zero for this stage of the project.

The analysis shows that for a mineralized material dilution of 5% and mining loss of 5%, the change in NPV at a 5% discount rate can be as high as 10%.

OPERATING COSTS

Mining and milling operating costs used as pit optimization parameters for this project were based on approximate costs developed during this study. These costs are shown in Table 18.6.

Table 18.6 Operating Costs

Project Area	US$/t	US$
Mining (mineralized material or waste)	Mined	1.58
Stockpile Rehandling (mineralized material)	Mined	0.50
Process, G&A, & Others	Milled	6.70
Total		**8.78**

PIT SLOPE ANGLES

Overall slope angles for the different sectors of the pit were provided by BGC, as shown in Table 18.7.

Table 18.7 Preliminary Open Pit Slope Angles

	Slope Azimuth		Assumed Overall Slope Height	Bench Height	Bench Face Angle	Max. Overall Slope Angle
Design Sector	**Start (°)**	**End (°)**	**Oh (m)**	**Bh (m)**	**Ba (°)**	**Oa (°)**
North Wall	317	037	510	30	65	43
East Wall	037	102	800	30	65	39
South Wall	102	225	1080	30	65	36
West Wall	225	317	800	30	65	39

18.1.4 NET SMELTER RETURN

The Net Smelter Return (NSR) is calculated in US$/t using Net Smelter Prices (NSP), as calculated in Section 18.11. The NSP is based on base case metal prices, currency exchange rate, offsite transportation, smelting and refining charges, and other factors. The metal prices and NSP used in the optimization are shown in Table 18.8.

Table 18.8 Metal Prices & NSP

	Metal Price (US$)	NSP (US$)
Cu	2.35/lb	2.01/lb
Au	800/oz	743.2/oz
Ag	12.55/oz	10.26/oz
Mo	13.91/lb	11.91/lb

The NSR formula is:

$$NSR = Au(g/t) * 0.032151 * AuRec * NSPAu + Ag(g/t) * 0.032151 * AgRec * NSPAg + Cu(\%) * 22.046 * CuRec * NSPCu + Mo(\%) * 22.046 * MoRec * NSPMo$$

Where:

- Cu = copper grade (%)
- Au = gold grade (g/t)
- Mo = molybdenum grade (%)
- Ag = silver grade (g/t)
- CuRec = copper recovery (<1)
- AuRec = gold recovery (<1) (doré+concentrate)
- MoRec = molybdenum recovery (<1)
- AgRec = silver recovery (<1) (doré+concentrate)
- NSPCu = NSP for copper ($/lb)
- NSPAu = NSP for gold ($/oz)
- NSPMo = NSP for molybdenum ($/lb)
- NSPAg = NSP for silver ($/oz).

The NSR formula includes offsite concentrate handling and doré refining.

18.1.5 PIT OPTIMIZATION ANALYSIS

Pit optimization determines the optimum pit limits and the economically mineable mineralized material inventories that are estimated to generate a maximum NPV. To this end, a 3D geological block model and other economic and operational variables were loaded into Whittle®. The variables included mining and milling parameters, product grades, costs, metal prices, and smelter terms.

18.1.6 PIT OPTIMIZATION RESULTS

The pit optimization was conducted using the LG algorithm. Each block is assigned a value that essentially shows the net cash flow that would result from mining that block. This value is calculated as the sale price minus the costs of mining and milling; blocks that return a zero or negative are considered waste blocks or air blocks.

The optimization determines which combination of blocks should be mined to yield the highest possible total economic value. It expands downwards and outwards from interim shells until the last increment reaches the breakeven point.

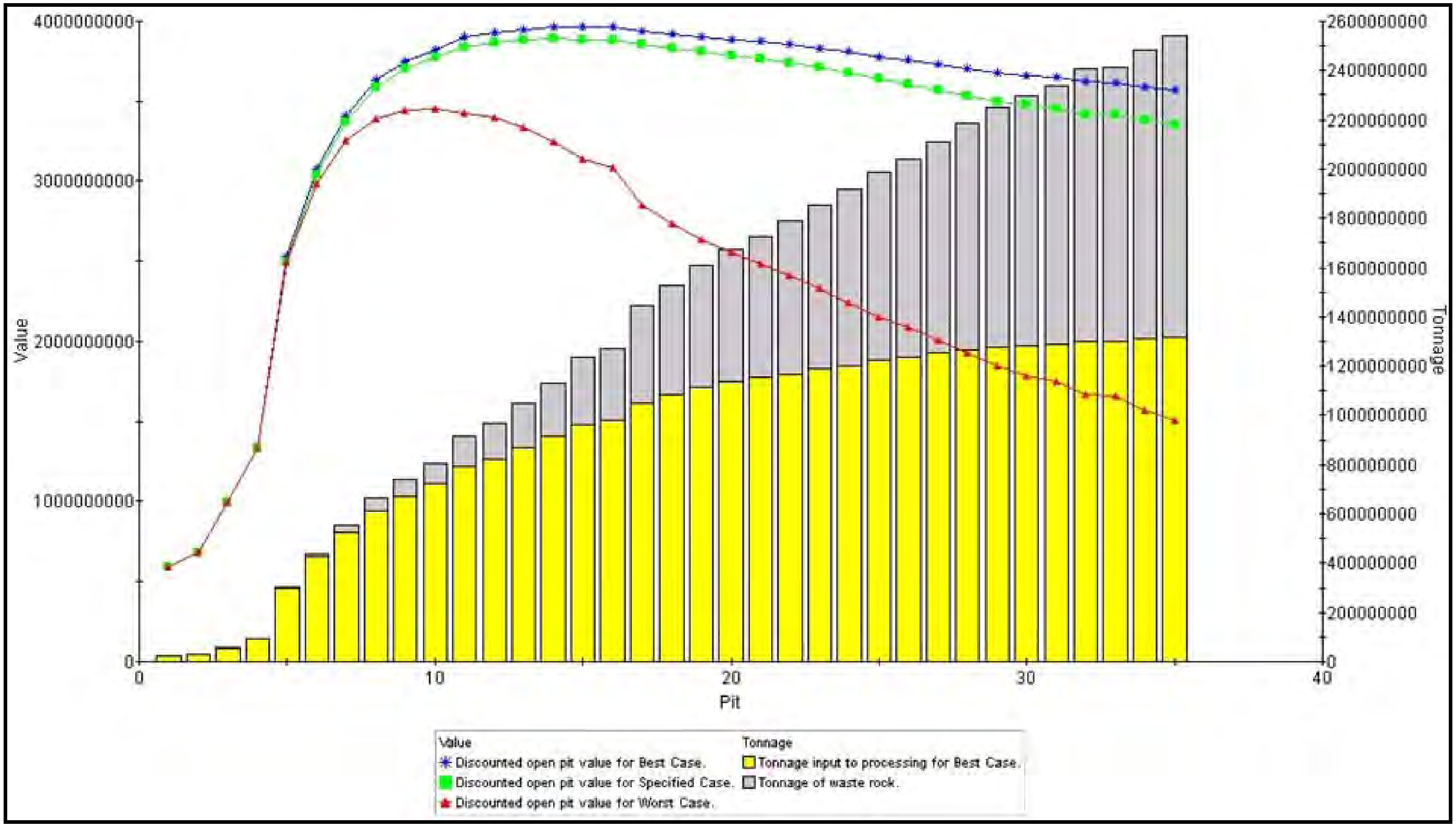
The routine uses input costs, metal prices, deductions, plant recoveries, and overall slope angles to expand downwards and outwards from previous interim economic 3D surfaces, until the last increment arrives is at a "break-even" point. Time-value discounting of each block is applied to account for the delay in mining the shallow mineralized material, and for mining of mineralized material at depth. The result is a 3D surface that represents an optimal pit that maximizes the total value of the mineralized material deposit. Further extension of the pit from this optimal shape will not increase the total economic return.

The three curves shown in Figure 18.1 are defined in the Whittle® manual as follows:

- Blue curve: discounted open pit value for Best Case. The best case schedule consists of mining out the smallest pit, and then mining out each subsequent pit shell from the top down, before starting the next pit shell. This schedule is seldom feasible because the pushbacks are usually much too narrow. Its usefulness lies in setting an upper limit to the achievable NPV.
- Red curve: discounted open pit value for Worst Case. The worst case schedule consists of mining each bench completely before starting on the next bench. This schedule or one very close to it is usually feasible. It also sets a lower limit to the NPV.
- Green Curve: discounted open pit value for Specified Case. The difference between the worst case and best case is significant. The Specified case schedule is an approximation of a more realistic mining schedule between the two extremes by specifying the sequence of pit outlines to push back to. This can be generated by:
  - specifying the number of benches by which the mining of each pushback is to lag behind the previous one
  - specifying the order in which the benches of pushbacks are mined as the algorithm seeks to maximize NPV
  - specifying the algorithm to find a schedule with improved throughput balance.

The specified case schedule was selected as the basis for the preliminary economic assessment study. As shown in Figure 18.1, Pit Shell No.14 generates the highest economic value relative to the other pit shells in the specified case schedule.

Figure 18.1 Pit Value Graph



18.1.7 *MINE PRODUCTION SCHEDULE*

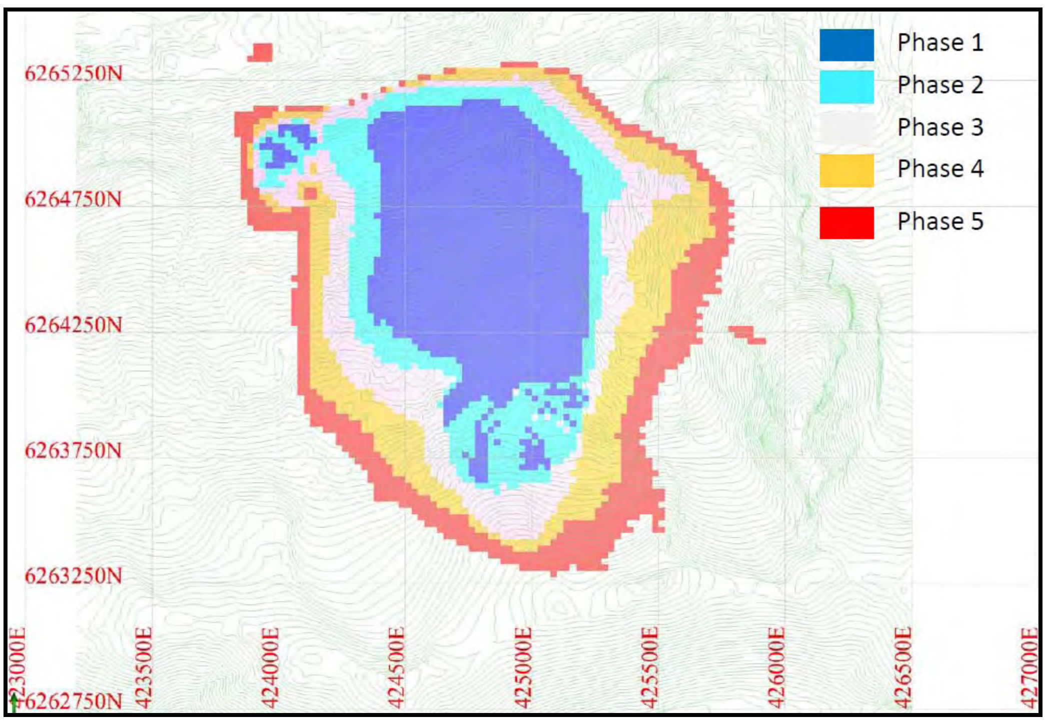
SCHEDULING CONCEPT

The pit optimization program takes no account of bench operating width in the generation of pits. Consequently, an optimized final pit which is used as a starting point for the design of a final pit may have a floor that is too narrow, and/or show irregularities in the pit wall that cannot be easily followed in practice.

Similarly, the optimized internal pit shells used in the design framework for pushbacks can have similar problems as those that may be encountered in the design of the final pit. Mining width problems can also arise if the wall of a pushback is too close to that of a subsequent pushback or the final pit.

In order to produce pushbacks that are operationally feasible, a mining width module of Whittle® was applied in the optimization process. Pushbacks were specified with a mining width of 60 m to accommodate large mining equipment that will operate on a bench. Five pit shells that are conceptually equivalent to five push-backs were selected for the development of the mine production schedule (Figure 18.2).

Figure 18.2 Mine Phases



Whittle® provides various scheduling algorithms.

- Milawa NPV finds a schedule with improved NPV. The order in which the benches are mined is determined by the Milawa algorithm, as it seeks to maximize NPV.
- Milawa Balanced finds a schedule with improved throughput balance. In this mode, the Milawa algorithm seeks to maximize the use of production facilities early in the life of the mine, instead of maximizing NPV. This option will only be of use if at least two of the possible mining, processing, or selling limits are specified.
- With Fixed Lead a number of benches are specified, by which the mining of each pushback is to lead the next push back. If, for example, a lead of four benches, mining would commence on the first pushback and proceed until four benches were mined. Mining would then commence on the second pushback and would also continue at the first pushback. Work would commence on the third pushback when the fourth bench of the second pushback has been completed, etc.

In scheduling the Snowfield pit, all three algorithms with various raised cut-off grades were evaluated, until the maximum NPV was reached. The algorithm that produced the best schedule with the highest NPV was selected. In doing so, various stockpiling options, with low and moderate grade mineralized material, were considered to establish an early cash-flow; however, the lack of available space prevented any consideration of low grade mineralized material stockpiling for this project.

PRODUCTION SCHEDULE

A production schedule, based on 120,000 t/d mill feed schedule, has been developed and is shown in Figure 18.3 and Table 18.9.

Figure 18.3 Production Schedule Graph

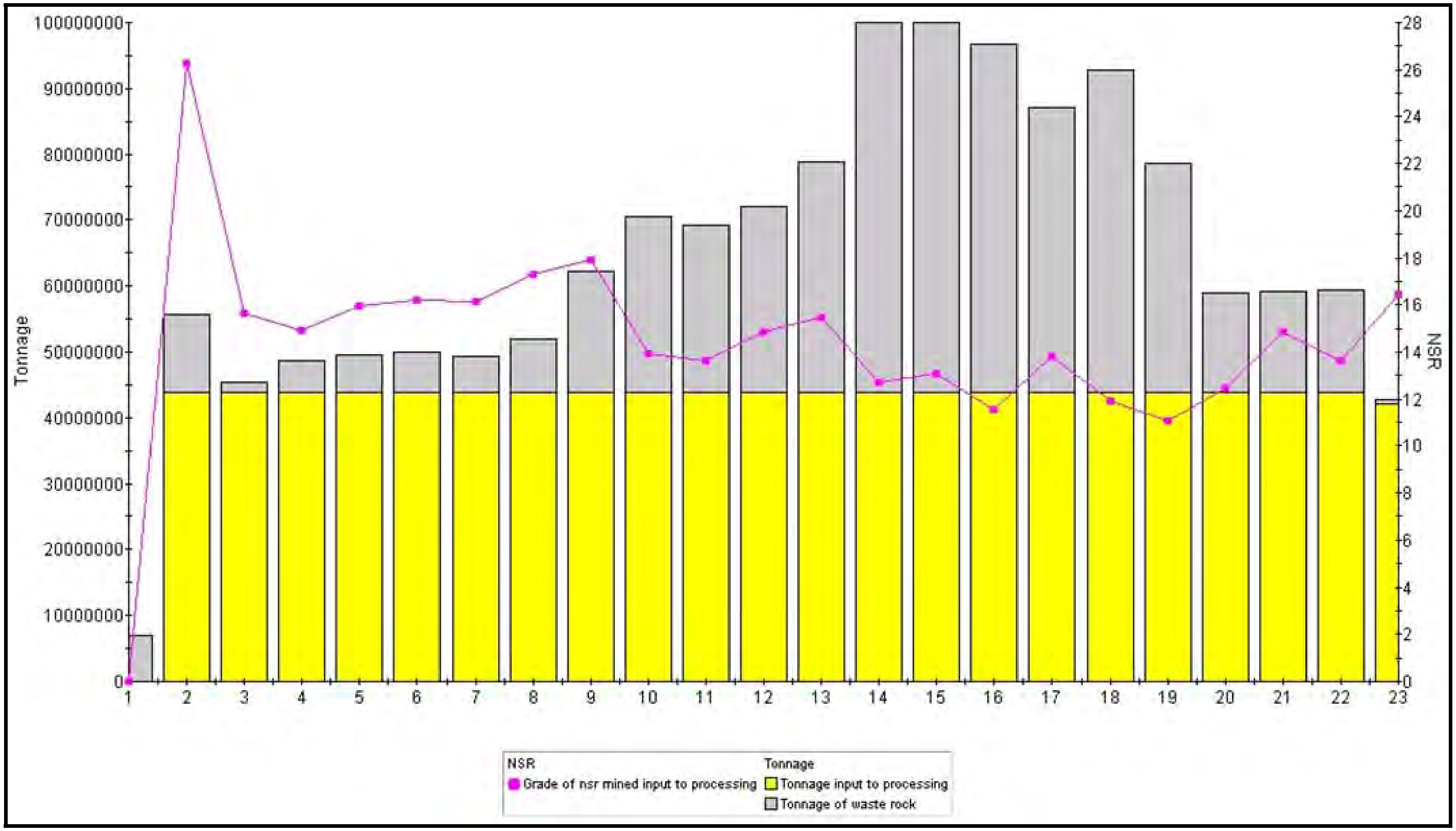


Table 18.9 Production Schedule

Mining Period	Mine Production (000 t)	Waste (000 t)	Total Mined (000 t)	Strip Ratio	Grade Ag (g/t)	Au (g/t)	Cu (%)	Mo (%)	NSR (US$/t)
-1		7,000	7,000	-	-	-	-	-	-
1	43,800	11,878	55,678	0.27	1.4175	1.3863	0.0362	0.0117	26.2907
2	43,800	1,712	45,512	0.04	1.8153	0.7617	0.0716	0.0115	15.6706
3	43,800	4,938	48,738	0.11	2.0420	0.6017	0.1171	0.0106	14.9305
4	43,800	5,676	49,476	0.13	1.8661	0.6518	0.1192	0.0107	15.9440
5	43,800	6,203	50,003	0.14	1.8162	0.6765	0.1166	0.0101	16.1961
6	43,800	5,625	49,425	0.13	1.8736	0.6822	0.1116	0.0101	16.1201
7	43,799	8,189	51,989	0.19	1.8644	0.7250	0.1190	0.0099	17.2822
8	43,800	18,361	62,161	0.42	1.7684	0.7096	0.1364	0.0095	17.9416
9	43,799	26,649	70,449	0.61	1.4688	0.6406	0.0863	0.0106	13.9221
10	43,799	25,352	69,152	0.58	1.5488	0.6254	0.0870	0.0104	13.6310
11	43,799	28,233	72,033	0.64	1.7065	0.6385	0.1075	0.0090	14.8495
12	43,800	34,924	78,724	0.80	1.6407	0.6357	0.1191	0.0085	15.4426
13	43,788	56,211	100,000	1.28	1.4175	0.5645	0.0943	0.0095	12.7159
14	43,799	56,200	100,000	1.28	1.5274	0.5695	0.0990	0.0092	13.0907
15	43,800	52,962	96,762	1.21	1.4392	0.5179	0.0873	0.0088	11.5837
16	43,800	43,388	87,188	0.99	1.6572	0.5636	0.1146	0.0076	13.8118
17	43,799	48,892	92,692	1.12	1.5135	0.4883	0.1068	0.0074	11.9260
18	43,800	34,776	78,576	0.79	1.4235	0.4775	0.0944	0.0082	11.0801
19	43,799	15,255	59,055	0.35	1.4667	0.5255	0.1044	0.0083	12.4511
20	43,800	12,007	55,807	0.27	1.6887	0.5768	0.1219	0.0072	14.2373
21	43,800	14,468	58,268	0.33	1.8764	0.5356	0.1256	0.0064	13.6194
22	43,800	682	44,482	0.02	1.9116	0.6166	0.1384	0.0081	15.9781
23	2,627	109		0.04	2.3739	0.5912	0.1669	0.0053	16.7600
Total	**966,214**	**519,700,206**	**1,485,915,094**	**0.54**	**1.6720**	**0.6440**	**0.1050**	**0.0090**	**14.9500**

Several schedule iterations were run that deferred waste stripping to later mining years. The low waste cover for the initial push backs allowed low strip ratios from Years 2 to 8, with total materials averaging about 51.6 Mt. In subsequent years, the volume of moved materials trends as follows:

- gradually increasing strip ratios from Years 9 to 13 with total materials moved averaging approximately 72.6 Mt;
- high strip ratios from Years 14 to 18 with total materials moved averaging approximately 92.5 Mt; and
- moderate strip ratios from Years 19 to 22, with total materials moved averaging approximately 54.4 Mt. Only 2.6 Mt of mineralized material will remain to be mined in Year 23.

The limited available space for waste rock dumps created the need to send some below-cut-off grade mineralized materials to the mill towards the end of mine life. The impact of this approach on project economics is insignificant, since the low-grade mineralized material is mined close to the end of LOM. Space limitations prevented consideration of mineralized material stockpiling and blending to further enhance project economics.

A schedule of mineralized material and waste mined from each phase is presented in Figure 18.4, Figure 18.5, and Figure 18.6. The production schedule was created to target high grade mineralized material in the early years of mining and defer the stripping to the later production years. Beginning in Year 8, stripping gradually increases and peaks around Year 14, and gradually drops after Year 15.

Figure 18.4 Mill Feed Schedule by Pushback

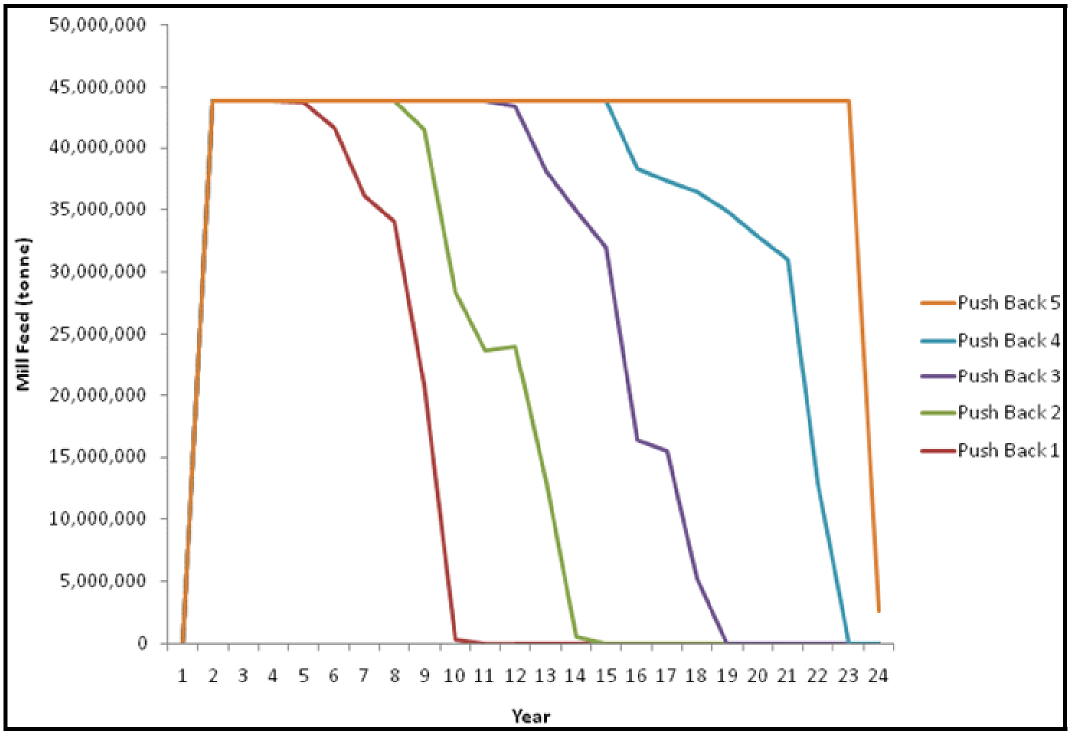


Figure 18.5 Waste Rock Schedule by Pushback

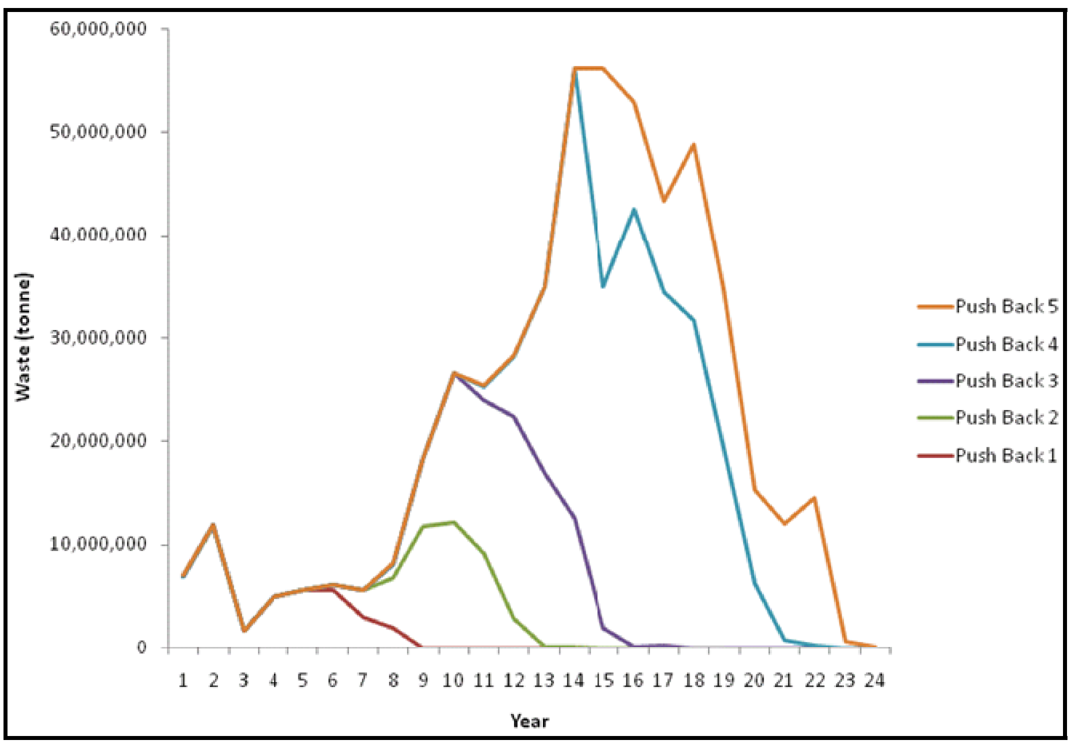


Figure 18.6 Projected Mill Feed Grade Schedule by Pushback

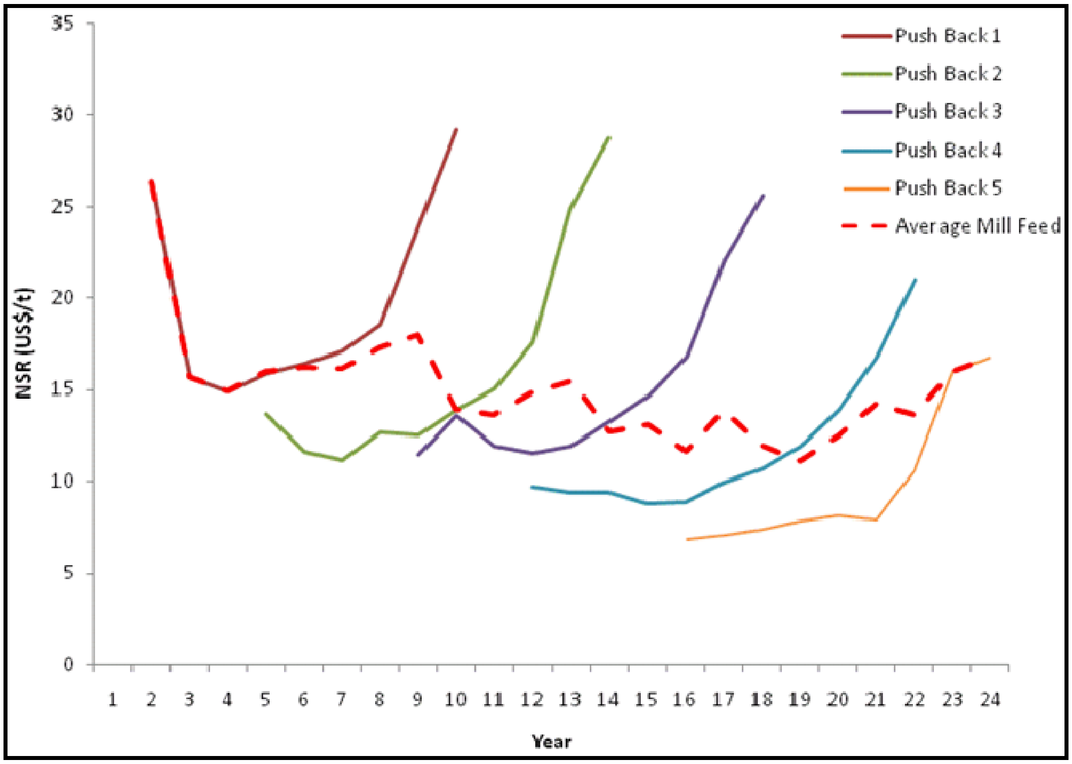
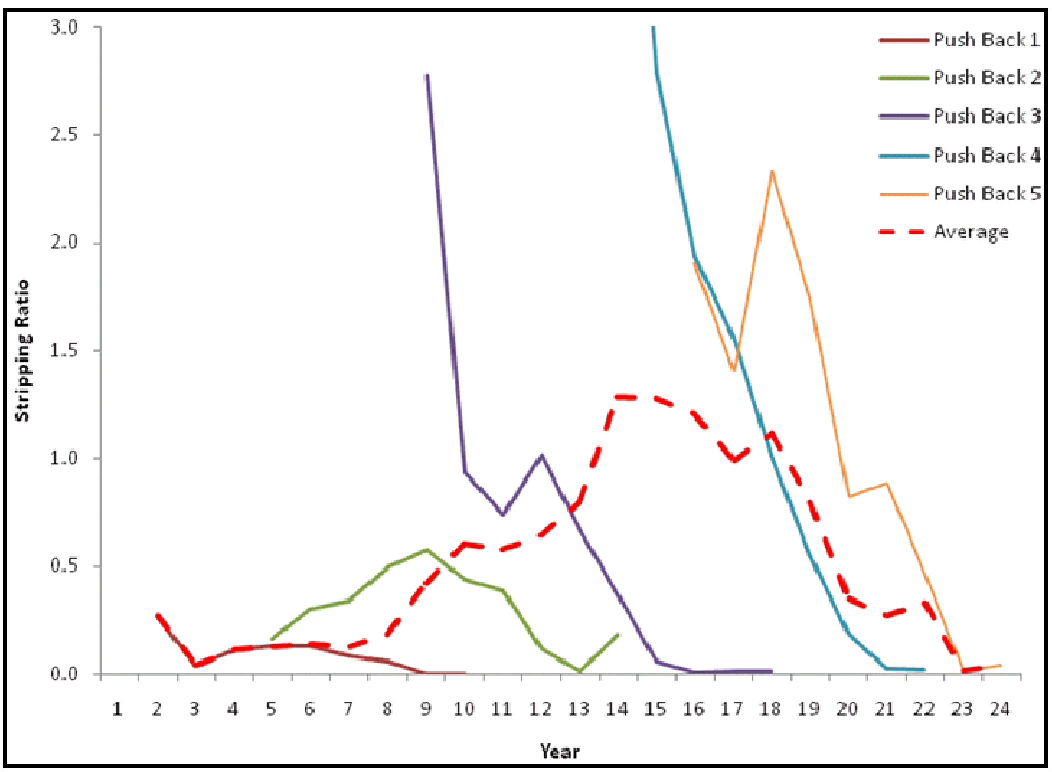


Figure 18.7 Stripping Ratio Schedule by Pushback



MINERALIZED MATERIAL DELIVERY TO THE PRIMARY CRUSHER

Because of the low overall strip ratio of the deposit, the impact of mineralized material haulage to the mining operating costs will be more significant than that of waste haulage.

The location of the primary crusher facility will reduce the number of trucks required and minimize mineralized material haulage costs. In the conceptual plan developed, the highest mineralized material bench is at 1690 masl and the lowest mineralized material bench is at 780 masl. A crusher site location was identified at the northeast portion of the pit with the truck dump at 1450 masl. Although further trade-off studies will evaluate the location of the crusher, the selected location for the preliminary assessment provides an initial concept with low haulage time and costs.

WASTE ROCK DELIVERY TO THE WASTE DUMP

As indicated in the production schedule (Table 18.9 and Figure 18.3), over 520 Mt of waste will be stripped over the LOM, and hauled to two potential waste dumps. Waste rock segregation is assumed to be accomplished depending on the potential of the rock to generate acid and other metals. For the PAG waste dump, steps will be implemented to divert groundwater and surface run-off away from the dump.

Wardrop proposes placing approximately 520 Mt of waste rock in the east and southwest dumps. The majority of the waste materials will be placed in the east dump, which will contain approximately 90% of the total waste rock. The remainder will be placed in the southwest dump.

BGC provided the following design parameters for the waste dump layouts to Wardrop:

- a 37°angle of repose for dump faces
- a swell factor of 30%
- overall dump slopes of 2:1
- no restrictions on free dumping height.

In determining dump geometry, an average quality rock was assumed to achieve a 37° angle of repose. The blending of average and poor quality rock is not shown in the dump layout and will be addressed in the next phase of study.

It has also been assumed that the dumps will be built by free dumping from top to bottom. For the east dump, a recommended overall dump slope of 2:1 applies on the northwest, north, and east side of the layout. This will allow road access to haul waste rocks from the lower benches to the 1570 m lift of the East Dump. Most of the waste rocks from the upper benches will be dumped to the 1870 m lift of the East Dump. The other waste rocks from the upper benches will be dumped to the Southwest Dump.

The dump locations and configurations were selected for a scoping level of study, based on space constraints. These selections will require future confirmation in a future study.

CONCEPTUAL MINE PLANNING

During pre-production, the main haul road will be developed along the east side of the proposed open pit to connect the truck shop down to the primary crusher location and to the north side of the open pit at 1190 masl. The approximate length of the main haul road is estimated to be about 7.5 km. It is estimated that only one 39.0 m^3 hydraulic shovel and four 363 t haul trucks will be required for pre-production stripping.

Site preparation for the primary crusher, located northeast of the open pit, will be conducted early in the construction period, and in conjunction with the construction of the conveyor tunnel. The truck dump hopper will be located at about 1450 masl, in the vicinity of coordinates 425,700 easting, 626,500 northing, as shown in Figure 18.8.

Pre-production stripping as shown in Year-1 of the production schedule will commence on bench 1790 m of the south wall of Pushback 4. A total of 7.0 Mt will be stripped down to 1720 m.

Full production will commence in Year-1. Pushback 1 will first encounter mineralized material at bench 1,690 m and will provide a total of 43.8 Mt of mineralized material inventory at a waste-to-mineralized material ratio of about 0.27. High-grade mineralized material will be mined from this initial pit and the mineralized material will have an average Au grade of approximately 1.39 g/t for an NSR value of approximately C$26.3/t for the year. The corresponding copper grade will be low, averaging approximately 0.036% Cu. During the year, it is expected that two electric cable shovels will undertake the primary production at low operating costs. One diesel hydraulic shovel will be assigned to mainly mineralized material and tight working areas.

Pushback 1 will be the source of mineralized material until Year 4. Mineralized material delivered to the mill will average:

- approximately 0.76 g/t Au and Cu grade of approximately 0.072% in Year 2
- approximately 0.60 g/t Au and Cu grade of approximately 0.11% in Year 3
- approximately 0.65 g/t Au and Cu grade of approximately 0.12% in Year 4.

In Year-5, Pushback 2 will commence, and will contribute approximately 5% of the total mineralized material inventory delivery to the mill. In Years 2 to 5, approximately 175.2 Mt of mineralized material inventory will be delivered to the mill. Au grade will decrease to an average of approximately 0.67 g/t while Cu grade will increase to an average of approximately 0.11%. A total of 18.5 Mt of waste will be removed for a waste to mineralized material ratio of 0.11. During these years, the same number of shovels, and a maximum of 16 haul trucks, will be required.

During Years 6 to10, approximately 92.8 Mt of mineralized material will be mined from Pushback 1, 87.2 Mt from Pushback 2 and 39.0 Mt from Pushback 3, for a total of 219.0 Mt of mineralized material. The Au grade will increase slightly to an average of 0.68 g/t and Cu grade will be maintained at an average of 0.11%. A total of 84.2 Mt of waste will be removed for a waste to mineralized material ratio of 0.38. The same number of shovels, and a maximum of 21 haul trucks, will be required.

During Years 11 to 18, approximately 37.7 Mt of mineralized material will be mined from Pushback 2, 147.9 Mt from Pushback 3, 136.8 Mt of mineralized material from Pushback 4 and 28.0 Mt from Pushback 5 for a total of 350.4 Mt of mineralized material inventory. The Au grade will decrease to an average of 0.56 g/t and Cu grade will slightly decrease to an average of 0.10%. A total of 355.6 Mt of waste will be removed for a waste-to-mineralized material ratio of 1.01. Another hydraulic shovel is planned to be added in Year 13 when the total materials mined will reach 100.0 Mt. A maximum of 30 haul trucks will be required to haul the material.

From Years 19 to 23, approximately 76.7 Mt of mineralized material will be mined from Pushback 4 and 96.8 Mt from Pushback 5 for a total of 173.5 Mt of mineralized material. A total of 42.5 Mt of waste will be removed, for a waste-to-mineralized material ratio of 0.24. A total of three shovels and a maximum of 23 haul trucks will be required. The Au grade will stay at an average of 0.56 g/t and Cu grade will increase to an average of 0.12%.

The concept of accessing the upper benches from each pushback involved driving haul ramps from the main haul road located east of the proposed open pit.

18.1.8 *Mineral Resource Estimate in the Final Pit Shell*

Based on the production schedule, mineral resource and diluted grades in each of the pit phases are provided in Table 18.10.

Table 18.10 Mineral Inventory

Pit Phases	Mineral Resources (t)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (%)	NSR (US$/t)
1	298,272,258	1.863	0.803	0.109	0.010	18.411
2	126,037,670	1.710	0.656	0.113	0.010	15.577
3	189,264,672	1.580	0.632	0.104	0.009	14.513
4	201,812,960	1.497	0.543	0.101	0.008	12.600
5	151,523,250	1.614	0.471	0.100	0.008	11.281
Total	**966,910,809**	**1.672**	**0.644**	**0.105**	**0.009**	**14.948**

18.1.9 PROJECT OPPORTUNITIES

Given the optimized geometry for the base case prices, the pit optimization was re-run to determine the impact of metal prices on the size of the pit and the mineral resource inventory. All other input parameters remained identical to those of the base case. The higher metal price are provided in Table 18.11.

Table 18.11 Metal Prices and NSP

	Metal Price (US$)	NSP (US$)
Cu	2.95/lb	2.55/lb
Au	850.0/oz	793.71/oz
Ag	14.5/oz	12.04/oz
Mo	17.0/lb	14.9/lb

The optimization results are provided in Table 18.12.

Table 18.12 Pit Optimization Results from Increased Metal Prices

Comparison with the Base Case	Mineralized Material (kt)	Waste (kt)	Total Materials (kt)	Strip Ratio	Au (k oz)	Ag (k oz)	Cu (k lb)	Mo (k lb)
Base Case vs. Large Pit								
Base Case Pit	966,215	519,700	1,485,915	0.54	20,005	51,952	2,244,363	196,614
Potential Pit A – High Price	1,105,724	728,625	1,834,349	0.66	22,610	59,191	2,535,187	219,391
Difference	139,509	208,925	348,434	1.50	2,605	7,239	290,824	22,777
Difference (%)	14.4	40.2	23.4		13.0	13.9	13.0	11.6
Base Case vs. Comparable Pit								
Base Case Pit	966,215	519,700	1,485,915	0.54	20,005	51,952	2,244,363	196,614
Potential Pit A - High Price	998,269	533,729	1,531,998	0.53	20,413	53,439	2,288,814	198,070
Difference	32,054	14,029	46,083	0.44	407	1,487	44,451	1,456
Difference (%)	3.3	2.7	3.1		2.1	2.9	2.0	0.7

Base Case Pit versus Large Pit

Potential Pit A

High Price is the optimal pit shell generated in the pit optimization using high metal prices. It is a larger pit compared to the Base Case Pit. It contains approximately 14.4% more mineralized material and 40.2% more waste materials. The pit contains approximately 13.0% more Au and 13.0% more Cu. However, the additional mineralized material inventory is available to be mined at a high incremental waste-to-mineralized material ratio of 1.50.

Base Case Pit versus Comparable Pit

Potential Pit B

High Price is comparable in size to the Base Case Pit and it represents an internal pit shell of Potential Pit A. The pit contains approximately 2.1% more mineralized material and 2.7% more waste materials. It contains approximately 2.1% more Au and 2.0% more Cu. The additional mineralized material inventory is available to be mined at a low incremental waste to mineralized material ratio of 0.44.

The higher metal values contained in the large pit provide a potential opportunity to improve the economics of the property. However, the disposal of the increased waste materials will require further assessment.

18.1.10 Mining Operations

General Comments

Large-scale mining equipment was selected to match the 365 d/a mine production schedule. Crews will work in two 12-h shifts, 4 days on and 4 days off. Equipment sizes were not optimized for this study. Equipment selection, sizing, and fleet requirements were based on planned operating conditions, long haulage profiles, production cycle times, mechanical availability, and overall utilization. To determine the number of units for each equipment type (drills, shovels, haulers, etc.), annual operating hours were calculated and compared to the available annual equipment hours.

Mobile mine support equipment, such as front-end loaders, track and rubber-tired dozers, graders, water, lube, and fuel trucks were matched with the major mining units. Ancillary and maintenance equipment was assigned to haul road maintenance, snow removal, mechanical and electrical servicing of the mining fleet.

Equipment additions were estimated over the life of the mine, while sustaining equipment replacements were estimated based on the operating life of each class of equipment item.

MINE EQUIPMENT OPERATING SCHEDULE

The equipment calendar operating schedule is shown in Table 18.13.

Table 18.13 Total Schedule for Mining Equipment

Loading Parameters	Units	Operating Time
Calendar Days	d/a	365
Work Days	d/a	365
Shifts per Day	shifts/d	2
Hours per Day	h/d	12
Total Hours	**h/a**	**8,760**

BLASTHOLE DRILL – NET PRODUCTIVE OPERATING TIME

The initial drill requirements will consist of two blasthole drills capable of drilling 311 mm diameter blastholes. A 9.3 m x 9.3 m mineralized material and waste rock drilling pattern was selected.

The mechanical availability of the blasthole drill was applied on a sliding scale based on the age of the unit. The overall mechanical availability averaged approximately 83.7%. The use of available hours was assumed to be 75.0% for each year of operation. The estimated effective utilization of the drills over the LOM is approximately 62.8%.

BLASTHOLE DRILL PRODUCTIVITY

Drill productivities were based on a bit penetration rate of 0.60 m/min for both mineralized material and waste rock. Total estimated drill time per hole including penetration and move time is based on Wardrop's other open pit projects in northern BC, as shown in Table 18.14.

Table 18.14 Blasthole Drill Productivity

Item Description	Units	Mineralized Material & Waste Rock
Hole Diameter	mm	311
Bench Height	m	15.0
Subgrade	m	1.9
Hole Length	m	16.9
Penetration Rate	m/min	0.60
- Penetration Time per Hole	min	28.1
- Move/Align Time	min	2.0
- Hole Collaring	min	1.0
- Grade Control Sampling	min	1.5
Total Time Per Hole	min	32.6
Holes Per Hour	holes	1.84
Average Drilling Rate	m/h	31.1
Spacing & Burden	m	9.3
Material Weight	t/m^3	2.78
Rock Mass per Hole	t	3620
Availability	%	83.7
Use of Availability	%	75
Effective Utilization	%	62.8
Maximum Operating Hours	h	5,501
Hourly Maximum Drill Productivity	**t/h**	**6,661**
Yearly Maximum Drill Productivity	**t/a**	**36,641,000**

General Blasting Conditions for Production Holes

Overall explosive consumption has been estimated based on 30% wet holes using 70% ANFO and 30% Emulsion.

An explosive supplier will erect an on-site bulk explosives plant, bulk product storage facility, and explosives magazines. The supplier will be contracted to supply, deliver, and load explosives into the blastholes. The supplier will also provide the blasting crew. The drill and blast foreman will oversee the contractor's blasting crew who will prime, stem, and tie-in blastholes. The contractor will also dewater wet blastholes.

Table 18.15 shows the blasting parameters that were used to estimate explosives consumption.

Table 18.15 Blasting Parameters for 311 mm Production Blastholes

Blasting Parameters	Unit	Mineralized Material & Waste Rock
Burden & Spacing	m	9.3
Hole Diameter	mm	311.0
Bench Height	m	15.0
Subgrade	m	1.9
Rock Mass per Hole	t	3620.0
Explosives Bulk Density	t/m^3	0.98
Percent of Hole Charged	%	70.0
Column Charge per Hole	kg	878.0

General Loading Conditions

The loading fleet consist of two 44.7 m³ electric cable shovels and one 39.0 m³ diesel hydraulic face shovel. A second hydraulic shovel is planned in Year 13 when the total materials moved reaches 100 Mt. The electric shovels will be matched with 363 t trucks to load mineralized material and waste materials in four bucket passes. These shovels were assigned a digging cycle of 35 seconds. The hydraulic shovels will also be matched with the 363 t trucks, which will be loaded in five passes. These shovels were assigned a digging cycle of 42 seconds.

A large front-end loader is assigned to load residual materials from the shovels and perform various functions in the pit areas. The back-up loader is matched to load the 363 t trucks in handling mineralized material and waste rock materials.

Shovel Loading Productivity

The estimated average loading productivities for the two types of shovels in loading the 363 t haulers are shown in Table 18.16.

Table 18.16 Average Estimated Productivity of Shovels in Mineralized Material and Waste Rock

Loading Parameters	Units	Electric Hydraulic Shovel*	Hydraulic Face Shovel*
Total Hours	h/a	8,760	8,760
Mechanical Availability	%	80.8	75.1
Available Hours	h/a	7,078	6,579
Standby/Idle	%	5	5
Gross Operating Hours	h	6,724	6,250
Total Operating Delays	h/shift	2.58	2.58
Total Operating Delays	h/a	1,883	1,883
Net Operating Hours	h/a	4,842	4,367
Use of Availability	%	66.9	66.4
Effective Utilization	%	55.3	49.9
Bucket Capacity (Heaped)	lcm	44.7	39.0
Dry Material Weight	dmt/lcm	2.78	2.78
Swell Factor	%	30	30
Dry Material Weight	dmt/lcm	2.14	2.14
Moisture Content	%	3.0	3.0
Wet Material Weight	wmt/lcm	2.20	2.20
Bucket Fill Factor	%	95	90
Effective Bucket Capacity	lcm	42.5	35.1
Tonnes/Pass	wmt	93.6	77.4
Truck Capacity	wmt	363	363
Average Passes/Truck	passes	3.9	4.7
Truck Load for Productivity	wmt	363.0	363.0
Truck Spot Time	seconds	30	30
First Bucket Cycle Time	seconds	35	42
Subsequent Bucket Cycle Time	seconds	35	42
Load Time/Truck	min	2.76	3.78
Maximum Productivity	trucks/h	21.7	15.9
	wmt/h	7,886	5,756
Truck Availability to Shovel	%	80	80
Production Hours	h/a	3,873	3,494
Annual Production	wmt/a	30,545,307	20,111,413
Productivity	wmt/noh	6,309	4,605
Truck Specifications			
Gross Vehicle Weight	kg	623,690	623,690
Empty Vehicle Weight	kg	260,690	260,690
Truck Rated Payload	t	363.0	363.0
Truck Body Capacity	lcm	267	267

*Based on loading 363 t trucks.

Note: Equipment Specification Basis:

Electric Cable Shovel – P&H4100XPC; Diesel Hydraulic Face shovel – PC8000; Haul Truck – Cat 797F Mechanical Truck.

GENERAL HAULING CONDITIONS

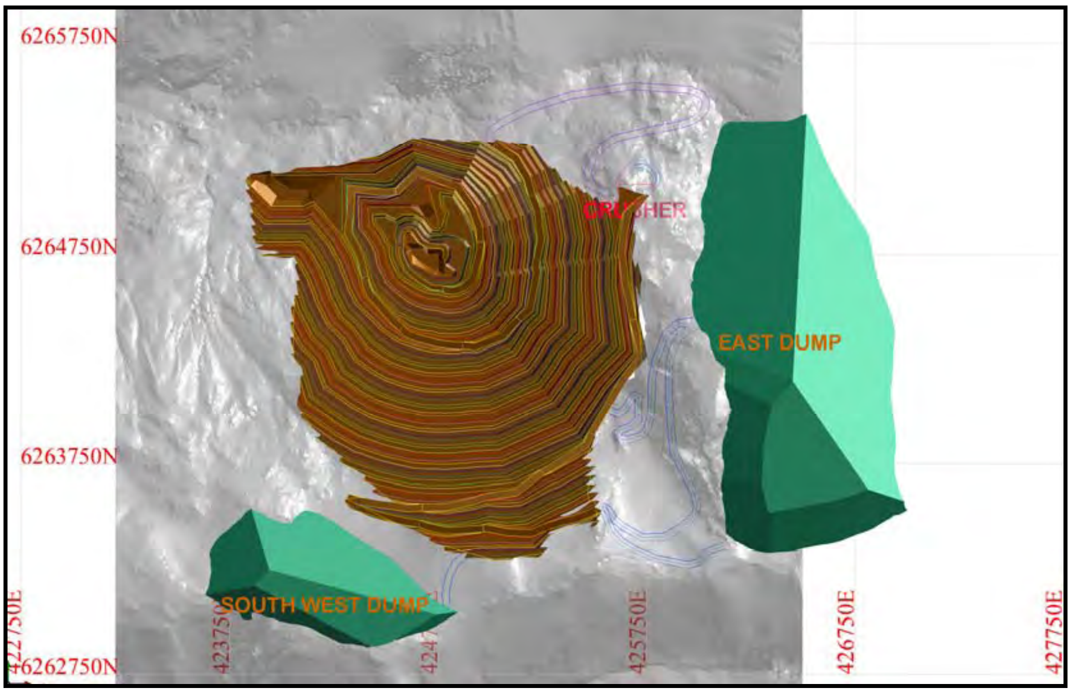
The 363 t haul truck was selected to match the 44.7 m³ electric cable shovel and the 39.0 m³ hydraulic shovel.

The number of trucks required to deliver 120,000 t/d mineralized material to the primary crusher and haul the corresponding waste rock to the waste dumps are based on the approximate locations of the key mine facilities shown in Table 18.17 and Figure 18.8.

Table 18.17 Facility Locations

Facilities	Easting (m)	Northing (m)	Top Elevation (m)	Approx. Avg. Distance from Pit Centroid (m)
Primary Crusher	425,700	6,265,000	1450	3,899
East Dump	426,507	6,263,8000	1,960	4,595
Southwest Dump	424,346	6,263,052	1894	2,000

Figure 18.8 Final Pit and Waste Dump Layout



HAUL TRUCK PRODUCTIVITY

From computer haul simulations, total cycle times and annual haulage hours were calculated for the various dumps. The following sections outline information that was used in the simulation analysis.

Production Schedule by Phases and by Bench

A Whittle® production schedule was developed showing the mineralized material, waste rock mined, and material by destination. The schedule was used to estimate the location of the centroid of each type of material mined in each period, phase, and bench. The centroid was estimated to be the mid-point of total tonnages mined between the top and bottom benches for each phase and each period when the material is mined out. From this centroid, the mineralized material and waste materials are hauled to the respective destinations.

Truck Haulage Profiles

Haul profiles for each period were conceptually estimated for the pit mineralized material destined to the primary crusher and for the waste rock sent to the waste dumps. The ex-pit haul roads to the primary crusher and waste dumps were approximated.

Each bench in the pit is planned to be accessed from the east and west. These access roads are further linked to the ex-pit haul roads that are connected to the primary crusher and the waste dumps. Haul profiles were then developed to define the haul distances and gradients from period and phase centroid to each destination.

Truck Simulation with Fleet Production and Cost (FPC) Program

Caterpillar's Truck Simulation Program FPC was used to calculate truck cycle times for each material destination based on the Cat 797F truck rimpull performance. Higher rolling resistances and lower speeds are imposed on trucks that are destined to the waste dumps due to variable soft ground foundation conditions. Lower speeds are imposed for trucks that are destined to the crusher due to high traffic density.

The program calculated the haul and return cycle times. The dump and manoeuvre delay (estimated at 30 seconds) and the shovel load time were added to the travel times to complete the total haul cycle.

Total Cycle Time by Period by Phase

Using the estimated cycle time by phase by centroid for each material, the weighted average cycle times by phase and by destination was calculated for the respective period.

Typical haulage simulation parameters for the 363 t Cat 797F haulers are shown in Table 18.18.

Table 18.18 Cat 797F Haul Truck Productivity

Haulage Simulation Parameters – Cat 797F Hauler			
Truck Size – Nominal	363 t		
Engine Gross Power Rating	2,983 kW		
Empty Vehicle Weight (Estimated)	260,690 kg		
Tire Size	59/80R 63		
Rolling Resistance			
General Applications	3%		
At Waste Dumps	5%		
Rock			
Moisture	3.0%		
Fill factor	95%		
Payload (dmt)	363.0		
Net Operating Time per Shift			
	Electric Cable Shovel	**Hydraulic Diesel Shovel**	
Spot and Load Times (min)	3.26	4.28	
Turn and Dump Times (min)	0.50	0.50	
Speed Limit (km/h)	**Grade (%)**	**Loaded**	**Empty**
	+3 to +10	30	30
	-10 to -3	30	30

Water Management for the Open Pit and Waste Dumps

The in-pit dewatering, surface water management around the open pit and the East and Southwest waste dumps, are discussed in Section 18.4.3.

Run-off into the open pit will be managed by the use of a combination of sumps and pumps. During high run-off periods from May to October, the pit bottom will be used as a primary sump. The pumps will be sized to dewater the pit bottom, which will be dry at the beginning of the winter months.

All flows from the mine, waste dumps and dewatering wells along the periphery of the pit will be pumped to the upper tunnel portal and piped 26 km to the process plant, and eventually discharged to the TSF.

The assumptions used to calculate water flows into the open pit are also discussed in Section 18.4.3. In summary, a 200-year return period was assumed for the open pit water management, given the high precipitation levels experienced in this region. Most pit sumps are designed for 10 to 25-year return periods.

18.1.11 MINE CAPITAL COST

Mine capital costs are derived from a combination of supplier quotes, historical data and from InfoMine USA.

The estimated mine equipment capital costs include basic equipment capital, assembly, and commissioning. Costs for delivery to site (excluding federal and provincial taxes or duties) are included as capital costs. The estimated mine capital costs are summarized in Table 18.19.

Table 18.19 Estimated Mine Capital Costs

	C$ (000)
Pre-Production Stripping	15,346
Mobile & Support Equipment	205,317
Explosives Storage	244
Fuel Storage & Delivery	230
Dewatering (In-Pumping only)	5,199
Electrical & Distribution	43,000
Communication	1,032
Safety	122
Engineering Equipment	2,023
Other Mining Costs	-
Total Mine Capital	**272,513**

MINING BASIS OF ESTIMATE

The magnitude of consumables and labour required are determined for each specific activity from similar projects in the area.

Currencies are expressed in Canadian dollars. All costs in this section were calculated in Q2 2010 Canadian dollars. A conversion to US$ was implemented at an exchange rate of 0.92. No allowance is included for costs escalation beyond this quarter.

The unit costs are based on the following information.

Salaries for the supervisory and administrative job category are based on Wardrop's experience of similar functions in BC mines. An average burden rate of 39% has been applied to base salaries to include all statutory Canadian and BC, social insurance, medical and insurance costs, pension, and vacation costs.

For hourly employees, general labour rates expected in BC mines and proposed projects were used. An average burden rate of 46% has been applied to base wages to include all statutory Canadian and BC, social insurance, medical and insurance costs, pension, and vacation costs.

Conceptual mine plans to determine the size and makeup of the mine fleet as well as fuel requirements which is affected by distance from the pit to the various destinations over the existing and future topography.

Budgetary quotations, including freight for all consumables, tires, and fuel as well as assembly and commissioning. Mining equipment consumables, major equipment replacements, sustaining capital, labour loading factors, equipment life, and costs are based on a combination of vendor information, InfoMine USA's 2008 Mine and Mill Equipment Costs, and Wardrop's data base from similar mining operations.

The estimated initial capital costs include the following:

- major mine equipment
- services and infrastructure
- pre-production tasks
- support and ancillary equipment.

Initial mine mobile equipment costs are shown in Table 18.20.

Table 18.20 Mine Mobile and Support Equipment Capital

Fleet Capital Cost	Task Description	PP/Period 1 C$ (000)
Diesel Rotary Drill – 311 mm	Primary Drill	10,476
Geotechnical Drill	Establishing Intermediate/Final Slopes	1,061
Electric Cable Shovel – 44.7 m^3	Loading Mineralized Material & Waste	59,132
Diesel Hydraulic Shovel – 39.0 m^3	Loading Mineralized Material & Waste	13,992
Front End Loader 1091 kW	Loader Backup in Pit/Road Maintenance	4,935
Haul Trucl – 363 t	Hauling Ore/Waste	66,847
Portable Generator for Electric Shovel	Moving Electric Shovels	75
Track Dozer – 634 kW	Shovel/Dump Support	6,300
Track Dozer – 433 kW	Shovel/Dump Support	4,437
Rubber Tired Dozer – 372 kW	Pit Clean Up	2,448
Fuel/Lube Truck – 18,000 L	Shovel Fuelling & Lube	445
Tire Handler	Tire Handling	2,252
Water Truck – 20,000 gal	Haul Roads Water Truck	2,400
Track Dozer – 433 kW	Dump/Road/Stockpile Maintenance	-
Motor Grader – 397 kW	Road Grading	7,344
Vibratory Compactor	Road Construction/Maintenance	450
Snow Plow/Sanding Truck	Road Maintenance	348
Screening Plant	Crushed Rock for Road Maintenance	331
Scrapers 37.0 t	Road Maintenance	6,000
Multi-purpose Front End Loader (Cat 988H)	Miscellaneous Pit Assignment	2,112
Hydraulic Backhoe – 4 m^3	Utility Excavator	1,300
Tractor & Lowboy – 189 t	Utility Tractor Transport In-pit	600
Heavy Lift Crane – 250 t	Field Maintenance	3,570
Heavy Lift Crane – 100 t	Utility Crane	2,040
Hydraulic Crane with Telescopic Boom – 40 t	Utility Crane	1,112
Mechanic Service Truck	Field Mechanical Maintenance	383
Welder Truck	Field Welding	383
Electrical Truck c/w Manlift Basket	Installing Electrical Lines	255
Maintenance Truck c/w Hydraulic Lift	Field Mechanical Maintenance	306
Integrated Tool Carrier/Cable Reeler	Field Mechanical Maintenance	600
Crew Bus	Crew Transportation	-
Crew Cab Pickup – 4x4	Supervision & Crew Transportation	696
Field Forklift – 30.0 t	Field/Shop Maintenance	250
Shop Forklift – 10.0 t	Shop Maintenance	108
Ambulance Including Equipment	Pit Emergencies	124
Fire Truck	Fire Suppression	450
Mine Rescue Truck	Safety/Mine Rescue	325
Light Plant/Towers	In-Pit/Dump Lighting	186
Mine Pumps	In-Pit Sump Dewatering	-
Shop Tools	Shop/Field Maintenance	1,245
Total Capital Cost		**205,317**

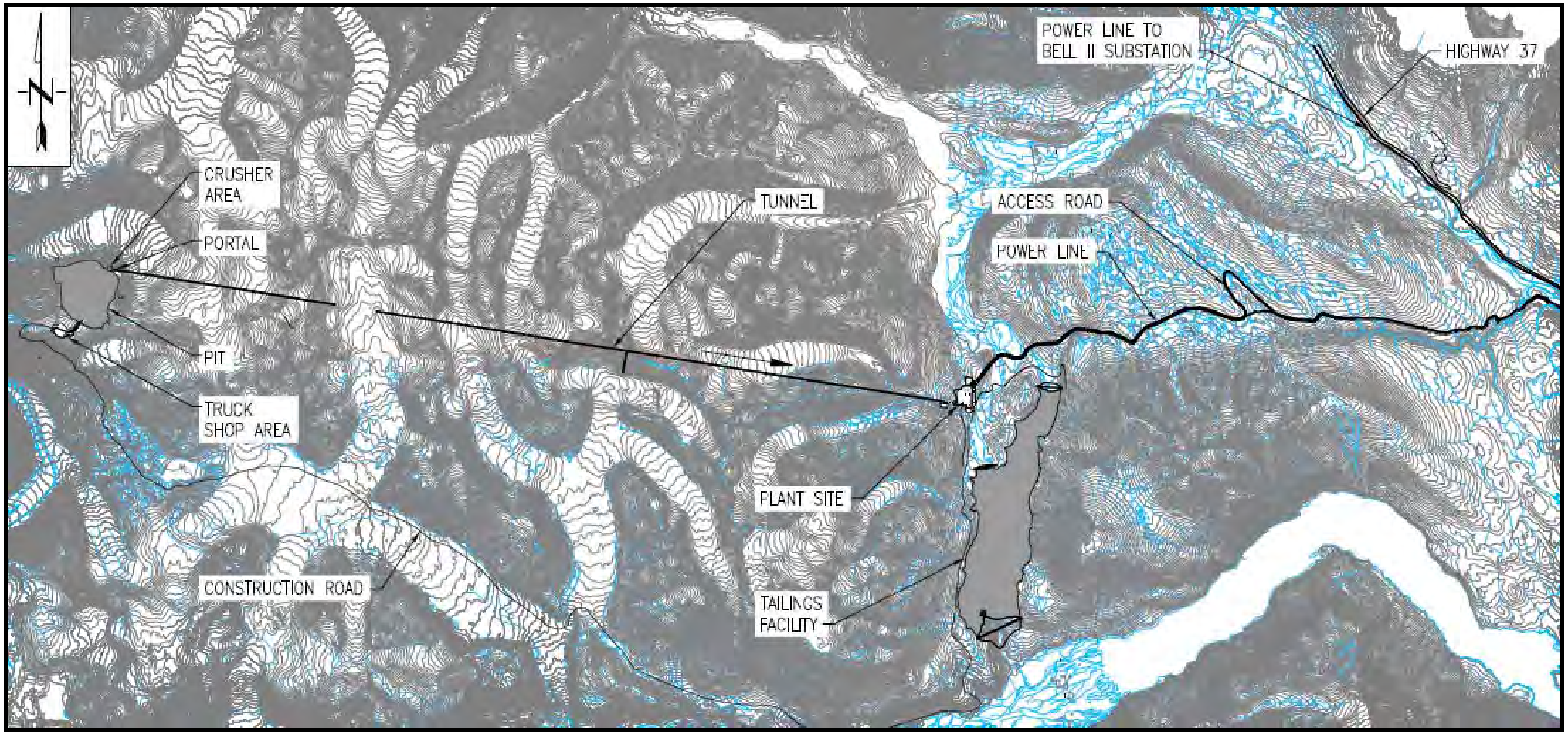
* PP = Pre-Production.

18.2 INFRASTRUCTURE

18.2.1 MINE AND SITE LAYOUT

The general arrangement of the Snowfield project mine and plant sites is presented in Figure 18.9.

Figure 18.9 Snowfield Overall Site Plan



The Snowfield project site will be accessible by a permanent road to be constructed southwest from Hwy 37 to the plant site. Hwy 37, a major road access to northern BC, passes approximately 24 km from the Snowfield project plant site. A 45 km construction road from the plant site location to the open pit area will be used to mobilize equipment and supplies.

The plant site is located 26 km southeast of the open pit area. Twin tunnels will connect the plant and the mine site. One tunnel is designated for conveying the mineralized material from the pit to the plant site. The second tunnel will provide a reliable year-round route between the plant and mine sites for workers and materials transportation.

The tailings storage facility is located approximately 5 km southeast from the plant site within the Scott Creek valley.

18.2.2 Ancillary Buildings

The pre-engineered and stick-built structures will be constructed for the Snowfield project and will include the following:

- administration building
- warehouse and maintenance building
- assay and metallurgical laboratory
- first aid building
- fuel storage facility and fuel station
- concentrate storage building
- maintenance shop and truck wash
- sewage treatment plant
- 500-person modular camp
- 1000-person construction camps at the plant and the mine sites.

18.2.3 Truck Shop/Warehouse

The principal function of the truck shop/warehouse complex is to provide servicing facilities for mine equipment and warehousing. The facility will be constructed of structural steel with metal clad wall and roof systems. The truck shop will include the following:

- four heavy duty repair bays
- one weld bay
- two light vehicle repair bays

- maintenance workshops
- truck wash/tire change bay
- emergency response facility
- warehouse
- offices.

18.2.4 FUEL STORAGE

Diesel fuel for the mining, process and ancillary facilities will be supplied from above-ground diesel fuel storage tanks located near the truck shop. The diesel fuel storage tank will have a capacity sufficient for approximately seven days of operation. Diesel storage will consist of above-ground tanks and will include loading and dispensing equipment. A dedicated service truck will transport diesel to the mining equipment operating in the pit.

18.2.5 CONCENTRATE STORAGE

Copper-gold and molybdenum concentrates will be stored in an on-site facility capable of storing a week of concentrate production at a time. On-site, the concentrates will be loaded into trucks and transported by contract trucking firms along Highway 37 to the port at Stewart, BC.

18.2.6 ROADS AND ACCESS

The plant site will be accessible via a new road from Highway 37. In addition, a temporary 45 km construction road from the plant site to the pit will be provided. Both the main access road and the construction road will approximate the path of the old Newhawk Gold Ltd. exploration access track.

18.2.7 SITE ROADS/EARTHWORKS

The earthworks portion of the infrastructure development will consist of:

- a 24-km main access road from Highway 37 to the plant site
- grading of the plant site
- a 45-km construction road from the plant site to the pit
- miscellaneous roads for use in the tailing storage facility construction and operation.

The main access road route roughly follows an access road that was reportedly utilized by Newhawk Gold Ltd. during their exploration activities. The main access road grades are limited to 10% and the travelled surface width is specified as 8 m. There is little geotechnical information currently available with respect to this route;

further physical investigation of this route will be required at the next stage of the project.

The plant site area will require a detailed geotechnical investigation to determine the suitability of the proposed location and the types of material that will be encountered. For this study, it has been assumed there is 300 mm of topsoil and, that 50% of the remaining material is rock. Approximately 50% of that rock is assumed rippable; the rest will require drill and blast methods.

The 45-km construction road extends from the proposed plant site, and will be used for construction traffic accessing the pit area. The proposed construction road route travels south from the plant site and parallels the tailing pond along the route of the track developed by Newhawk Gold Ltd.; the road then turns west and traverses the glaciers leading to the pit area. Where possible, the construction access road will be combined with other uses such as the west tailing pipeline/diversion ditch maintenance road and a haul road from a rock quarry to the southern tailing dam. It is believed that the Newhawk Gold track will need major up-grading/re-routing in order to haul major mining components and large quantities of construction materials to the pit crusher and additional facilities.

Construction road grades are limited to 10%; the travelled surface width varies according to its usage. This road will be an all-year usage road in order to accommodate construction schedule requirements.

About 14 km of the proposed construction road route passes along and across a glacier. Though apparently feasible, the concept of a year-round road across a glacier requires further investigation, especially with respect to the physical properties of the glacier and the method of construction. Budget allowances for each section vary greatly depending upon their width and traffic usage.

Also included in the initial construction estimate is a 9 km of maintenance roads for the east tailing pipeline/reclaim water pipelines and other minor roads.

In order to shorten the construction time for the twin tunnels (from the pit to the plant site), access to two intermediate tunnel construction sites from the plant site-pit construction road is provided. Access to these sites requires a total of 19 km of road over glaciers.

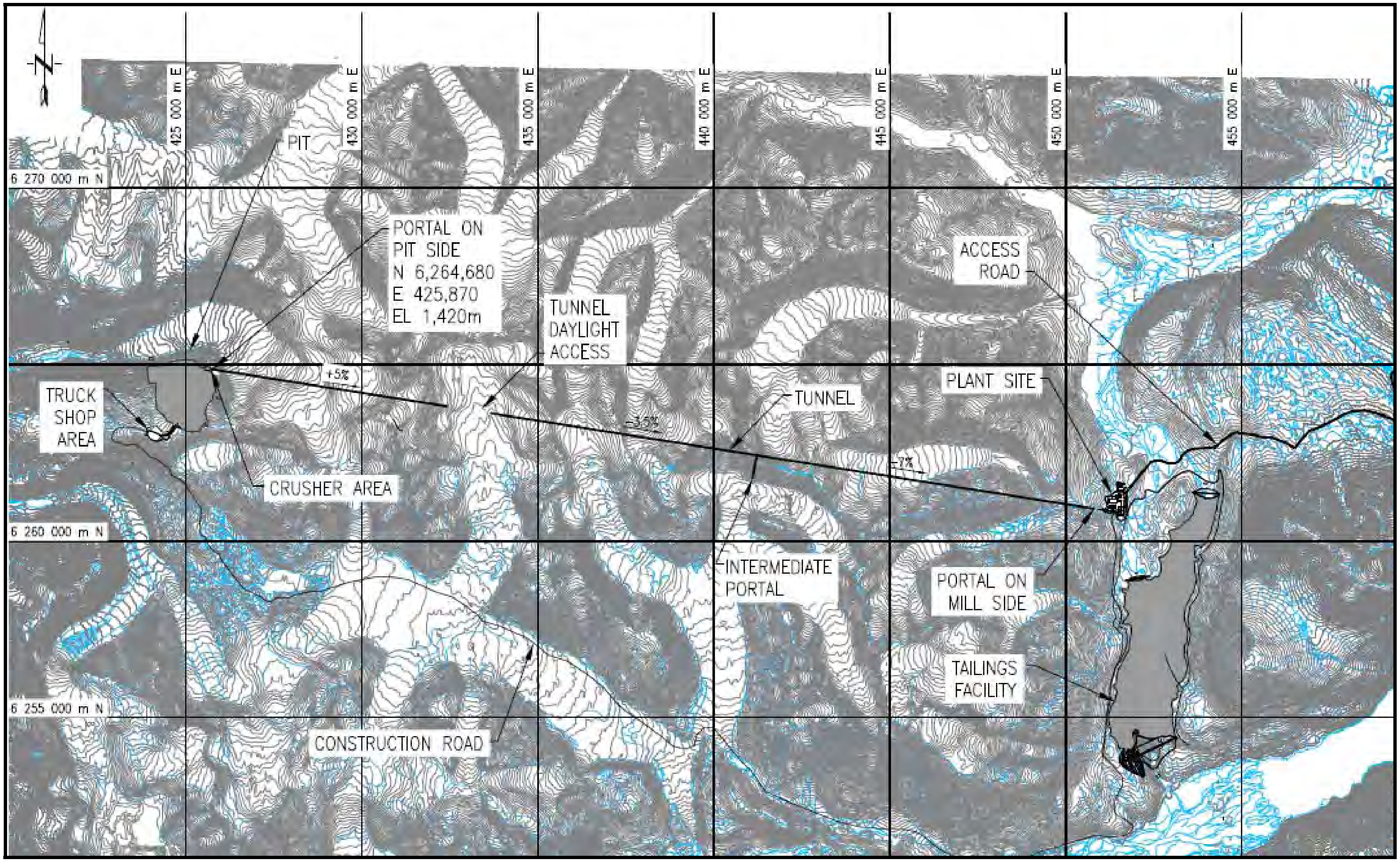
No allowance has been provided for hazard control (e.g. avalanche, landslide, etc.) or hazard avoidance. An assessment of the risks and mitigations with respect to these hazards is required.

18.2.8 Tunnel Development

The process plant facilities and the mine camp will be located approximately 26 km east of the open pit. A twin tunnel will connect the mill and the mine area. One tunnel will be used for conveying the mineralized material from the pit to the processing facilities, and one tunnel will provide a reliable year-round route to the mine site for materials and workers.

The location of the proposed tunnel access is shown in Figure 18.10.

Figure 18.10 Tunnel Development Plan View



TUNNEL DEVELOPMENT CONCEPTUAL DESIGN

Since there is no geotechnical information currently available, Wardrop assumed that tunnels will be developed using a mechanized drilling and blasting (D&B) method. The use of a tunnel boring machine (TBM) for tunnel development was not considered for this study because geological uncertainty fractured rock conditions, ground instability, rock falls, and possible caving may make the use of a TBM difficult or impossible for the 26 km long tunnel.

The major disadvantage of the TBM method is its up-front cost. TBMs are expensive to construct and they can be difficult to transport. However, the TBM method tunnels much more efficiently than the D&B method, which would lead to a shorter project duration and potentially lower costs. Reducing pre-production development time will have positive impact on overall project economics.

The overall tunnel is divided into three sections so that the tunnel can be developed simultaneously from various portals, which would reduce the overall development time. The tunnel is designed to have an intermediate access 6,758 m from the pit side. Another portion of the tunnel from the mill side will be 17,355 m long. This latter portion will be divided into two development sections by intermediate decline access. That decline will be 620 m long and will be developed at -10% from the side of the tunnel, perpendicular to the tunnel direction. This design will provide an opportunity to develop the tunnel from six working faces, each working towards another.

The tunnel dimensions were determined by the stationary and mobile equipment and by their required clearances. Based on the 1,600 mm belt conveyor, the size of the conveyor tunnel will be 6.5 m wide and 4.8 m high, to allow use of rubber-tired equipment to carry equipment parts along the tunnel and to provide required clearances between the conveyor, the rubber-tired equipment and the walls for the conveyor maintenance.

An access tunnel will be 4.5 m wide and 5.5 m high to provide clearances for the equipment during development. The tunnel will be used for delivery of the materials and supplies, and serve as a second exit from the conveyor gallery during production.

The twin tunnels will run in parallel 20 m apart from each other with crossover connections at 450 m intervals to reduce auxiliary ventilation requirements.

During development, one of the tunnels will be used for fresh air intake and another for exhaust. This will eliminate useof duct ventilation for the entire length of the tunnel. The only dead-end portion of the tunnel after a crosscut will require auxiliary ventilation through the duct. This will minimize time to clear smoke after each blast at the face, which is important when faces advance several kilometres from the portal. The twin tunnel will provide a second egress in case of fire or blockage in one of the tunnels. There will be a second exit for the conveyor tunnel during production, in the event of a fire on the conveyor belt.

All underground development will be on a grade to provide drainage to the portals and eliminate the need for pump and sump development. No permanent dewatering pumps will be required after the underground development is completed; therefore there will be no risk of pump failure and flooding if a water pipe breaks.

The cross sections of twin tunnel with equipment arrangement during development and production are shown in Figure 18.11 and Figure 18.12.

Figure 18.11 Conveyor Tunnel

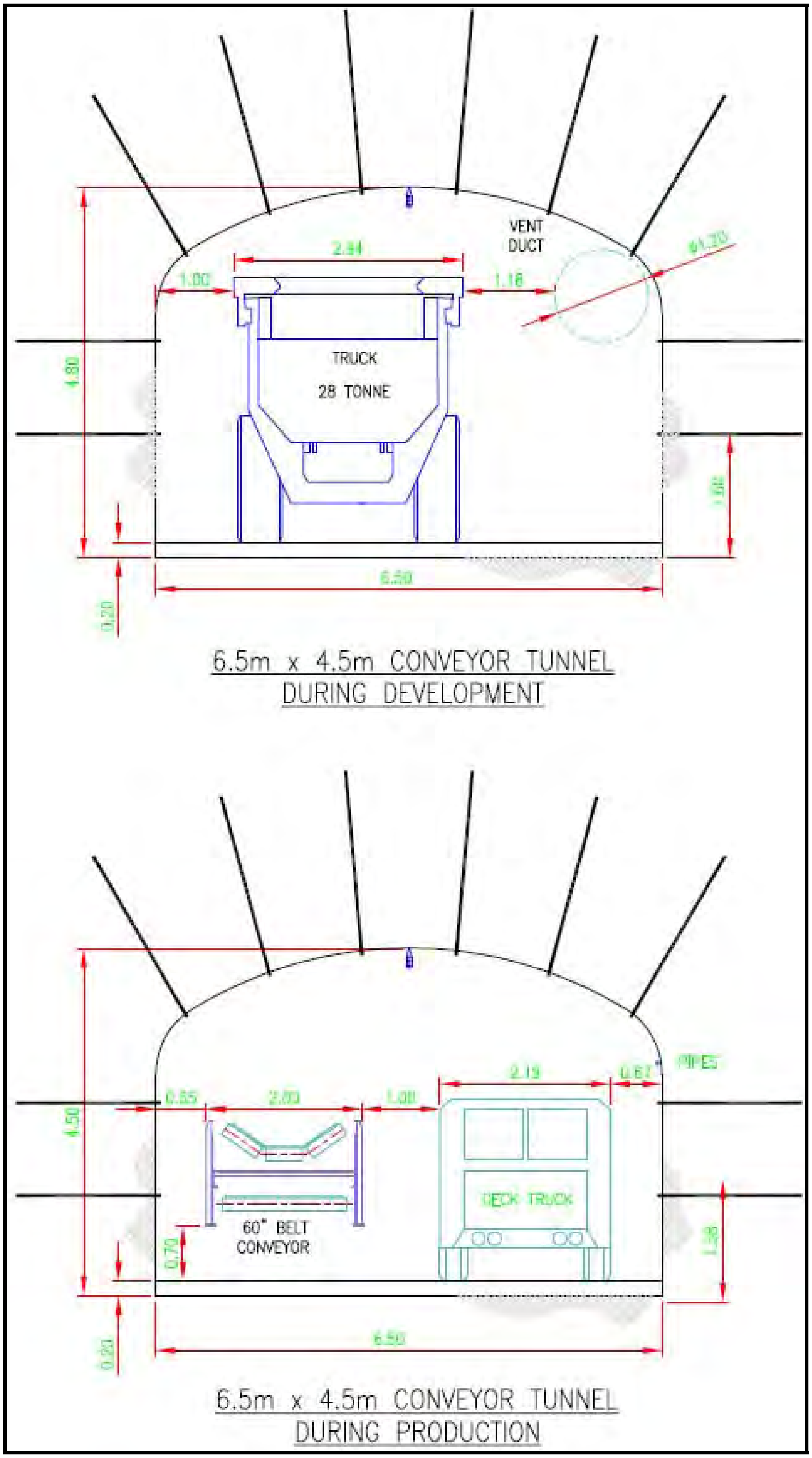
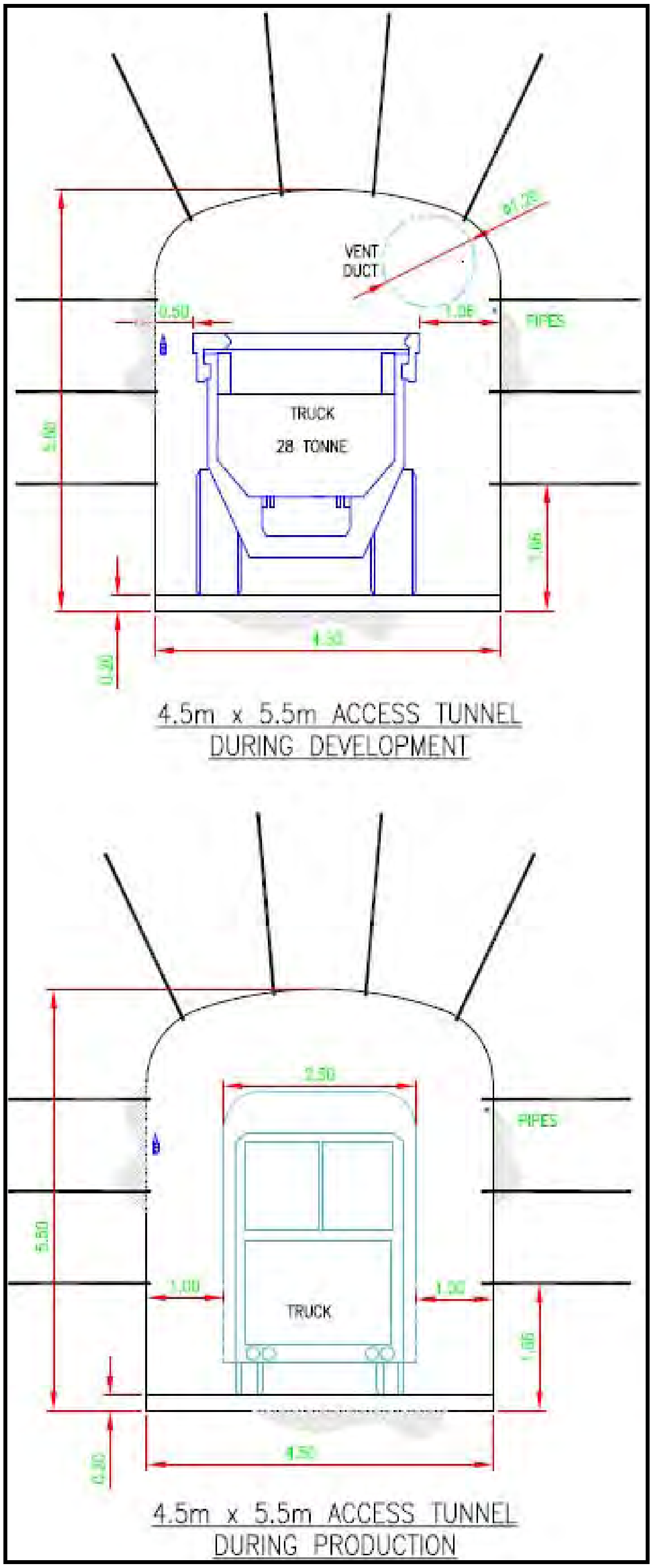


Figure 18.12 Main Access Tunnel Cross Section



GROUND SUPPORT

Because no geotechnical assessment has been conducted, the ground conditions and the ground support requirements have not yet been determined. A geotechnical and hydrogeological evaluation is recommended to advance the project to the next stage.

Regular geotechnical ground control must be provided during development to control ground conditions and to monitor support requirements.

TUNNEL DEVELOPMENT EQUIPMENT SELECTION

Electro-hydraulic double-boom jumbos will be used to drill the face of the twin tunnel. A rockbolter will be used for rockbolt drilling and installation of grouted rebars and mesh.

The 4.6 m^3 LHDs will be used to muck the broken waste rock from the face; 28-t surface trucks will haul the waste to surface.

PERSONNEL

Contractor employees at the Snowfield tunnel development were divided into two personnel categories as follows:

- indirect personnel requirements including administrative, technical, and supervisory staff
- direct labour including mechanics, jumbo operator, miners, blaster, service equipment operators, electricians, welders, and tunnel supervisor.

Hourly personnel were estimated based on operation productivities, maintenance and services requirements. Personnel productivities were estimated for all main activities by developing cycle times for each operation.

DEVELOPMENT CYCLE

A jumbo crew will drill 4.0 m rounds with 45 mm holes. The holes will be loaded with ANFO from a pneumatic loader and blasting initiated with nonel caps. Smooth perimeter drilling and blasting techniques will be used to reduce damage to the walls and back, and to minimize ground support requirements.

The broken rock will be mucked from the face of the underground opening by 4.6 m^3 load-haul-dump (LHD) and hauled to the remuck bays located at 150 m intervals to clear the face as quickly as possible. When the face of the development opening is clean and ready for bolting, the waste will be mucked from the remuck bays and hauled to surface by 28-t trucks.

Remuck bays could subsequently act as temporary sumps, and as spaces for electrical substations and as material storage space.

The back and walls of the headings will be scaled and ground support will be installed. The pipelines, ventilation ducts and power cables will be installed as the heading advances.

The estimated jumbo development cycle time is shown in Table 18.21.

Table 18.21 Estimated Jumbo Crew Development Cycle Time

	Unit	Conveyor Tunnel	Main Access Tunnel
Width	m	6.5	4.5
Height	m	4.8	5.5
Gradient	%	5.0	5.0
Summary Cycle Times			
Drilling	h	4.58	3.96
Blasting	h	2.08	1.80
Re-Entry	h	0.50	0.50
Mucking	h	2.63	2.20
Support	h	6.07	5.22
Services	h	0.80	0.80
Secondary Mucking	h	7.34	8.15
Trucking	h	18.79	15.59
Single Heading			
Critical Path Cycle Time	h	15.85	13.68
Advance Per Shift	m	2.15	2.5
Advance Per Day	m	4.3	5.0

VENTILATION OF HEADINGS DURING DEVELOPMENT

The development headings will be ventilated by auxiliary fans and vent ducts, initially from the portal. When the first crosscut between the tunnels is developed, flow-through ventilation will be established using surface fan and airlock at the portal. The auxiliary fans will be replaced closer to the faces to the intersection with crosscut. When the next crosscut will be developed, the previous crosscut will be bulkheaded to provide flow-through ventilation closer to the face, and the auxiliary fans will be replaced again. Only about 500 metres of the development heading will require auxiliary ventilation by vent duct.

The ventilation system designed for the twin tunnel development is a forced-air system delivering approximately 110 m^3/s. A main intake fan located on surface and two underground auxiliary fans will control the primary ventilation circuit. Bulkheads and ventilation doors will be used to control air flow.

The portal will be equipped with airlock-type double-doors to allow vehicle passage without interrupting mine ventilation. The ventilation system designed for the twin tunnel development is consistent with regulations applied by the Canadian Occupational Health and Safety Standards.

Ventilation of Headings During Production

Each tunnel will have a completely independent ventilation system during the operation phase. The twin tunnels will utilize a ventilation system developed for construction of the tunnels; however, the conveyor tunnel will require the installation of airlock doors at the portal. The flow direction of the ventilation air in the twin tunnels will be from the pit to the mill, the same direction as the conveyor belt. There will be no air leakage through the bulkheads and ventilation doors between the tunnels during the operation phase.

Tunnel Development Schedule

The tunnel development will be performed by a contractor. It is assumed that access roads and power will be completed before the contractor mobilizes to the site.

Jumbo crews will develop tunnels from five portals simultaneously to reduce the construction time.

In the development schedule it was assumed that remuck bays and crosscut development will not affect the tunnel advancement rates.

18.2.9 *Communications*

The project telecommunications design will incorporate proven and reliable systems to ensure that personnel at the mine site have adequate data, voice, and other communications channels available. The telecommunications system will be supplied as a design-build package.

The base system will be installed during the construction period then expanded to encompass the operating plant.

The major features of the communication system will include:

- a satellite communications for voice and data
- ethernet cabling for site infrastructure
- provision for two-way radio communications at site.

A variety of communications media (copper and wireless during the construction phase and fibre optic during the operating phase) will be incorporated in the overall design.

18.2.10 Tailings Storage Facility

The TSF is located within the Scott Creek Valley, approximately 30 km from the open-pit and 6 km from the process plant. The tailings delivery system was designed to transport 966 Mt of tailings to the tailings deposition area.

18.2.11 Power / Electrical

Plant Load

The mill throughput is 120,000 t/d. At this production level, the plant load is estimated to be approximately 140 MW ± 10%.

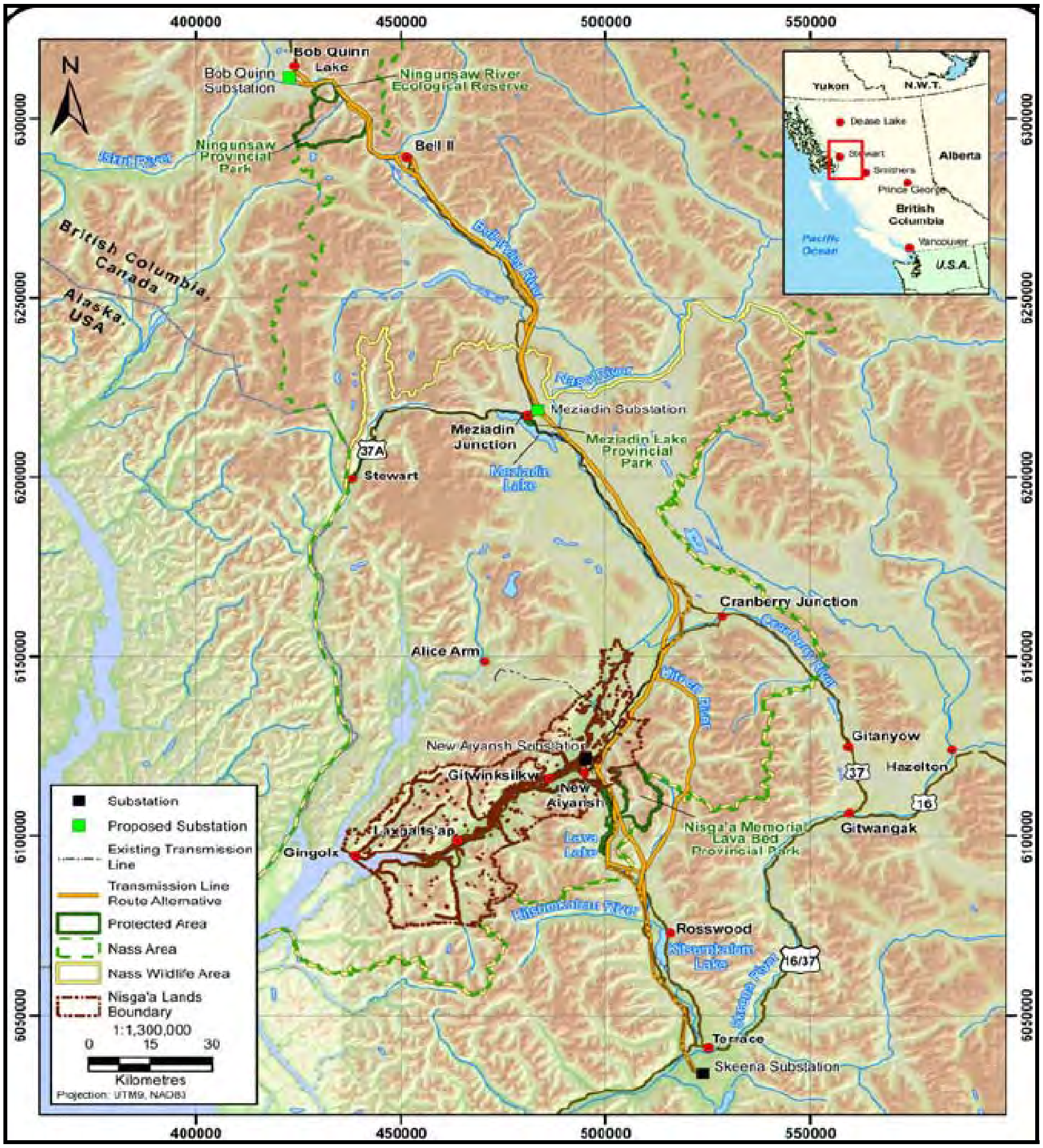
Power Source – Northwest Transmission Line

Electrical power will be supplied from the proposed Northwest Transmission Line (NTL). The NTL will be a 287 kV line between Terrace, BC and Bob Quinn Lake.

The line will be built by winter 2012.

A map of the proposed line to Bob Quinn Lake is shown in Figure 18.13:

Figure 18.13 Proposed Northwest Transmission Line Route



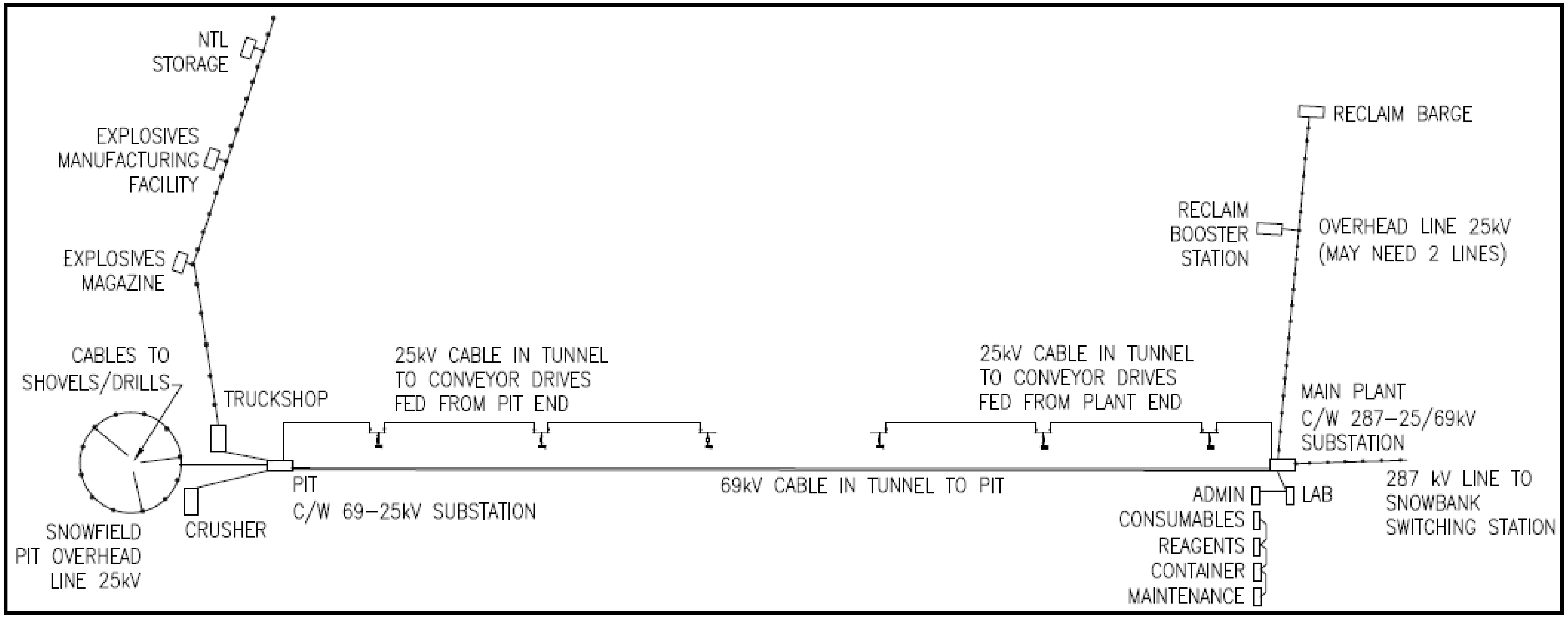
Note: British Columbia Transmission Corp.

SERVICE INTO SNOWFIELD

The most economical way to serve the Snowfield load is to establish a transmission line from the Bell II substation.

The line to Snowfield will be approximately 45 km long, and terminate at a distribution substation at the Snowfield mill site. A conceptual one-line diagram for the Snowfield project is shown in Figure 18.14.

Figure 18.14 Conceptual One Line Diagram



There will be four main transformers feeding the mill site. Each transformer will be base-rated at 60 MVA, with additional fan cooled ratings of 80 MVA and 100 MVA. The transformers will be sized to allow the plant to run with one transformer out of service. Transformers of this size are in the 100 tonne range and will be one of the largest loads to transport into site.

Each transformer will feed its own bus at the 25 kV level. Large motor loads (e.g. ball mills) will be served at 13.8 kV via dedicated step-down transformers. Power will be distributed around the site using cables and overhead lines, at 25 kV and additional step-down transformers will be located near remaining loads. Medium sized motor loads (250 HP to 5,000 HP) will be served at 4.16 kV. Smaller motor loads will be served at 600 V.

Two additional transformers will be provided at the Snowfield main substation to step back up to 69 kV. This will be a suitable voltage to feed via cable through the tunnel to the pit, where it will be further stepped down to 25 kV, 4 kV and 600 V to feed the shovels, drills, and primary crusher.

The tunnel conveyors will be fed from 25 kV cables from both ends of the tunnel. The feed from each end will cover one half of the tunnel. Dry-type transformers will be used to step-down from 25 kV to 4 kV to feed the conveyor motors. As this is a downhill conveyor, the conveyor drives will act as generators and up to 3 MW to 4 MW of power will be generated.

All of the main transformers will be equipped with on-load tapchangers to help maintain voltage levels as the load on NTL changes. Shunt capacitors and reactors may also be required to help regulate the voltage.

CONCLUSIONS

Electrical service into the Snowfield project is feasible. The immediate concerns include:

- access to the limited power supply
- uncertainty of costs associated with participating in the NTL installation expenditure
- reliability of a relatively long transmission line in a harsh environmental setting.

18.3 WASTE AND WATER MANAGEMENT

The conceptual schemes for the waste and water management for the Snowfield project have been prepared by BGC and Rescan.

18.3.1 TAILINGS MANAGEMENT

To ensure that the TSF continuously meets its objectives, a tailing management plan was developed during this study. The goals of this management plan are to:

- provide a guide or framework to manage the TSF structures in a safe and environmentally responsible manner throughout all stages of the Snowfield project
- provide a means to manage the TSF itself (managing substances going in to and out of the facility)
- manage the discharge from the TSF to ensure that all effluent meets and/or exceeds the permitted water quality levels and guidelines.
- provide continual improvement in the environmental safety and operational performance of the TSF structures
- provide environmental and performance monitoring and reporting
- provide an organizational structure to ensure accountability and responsibility to manage the implementation and maintenance of obligations under Silver Standard's environmental policy.

At the next phase of design, tests will be undertaken to characterize the tailing and supernatant to allow estimation of the rate of oxidation and resulting water quality. This information will guide planning for tailing water management.

At present, it is assumed that the high sulphide content of the pyrite tailing from the process plant will cause this material to quickly oxidize and generate acid if exposed to air. The proposed solution to this acid generation, and potential subsequent metal leaching, is to store the tailing permanently under water where oxidation is vastly reduced or eliminated. The TSF is designed to isolate the pyrite tailing in a stable subaqueous environment in perpetuity. Where feasible, ditches will be constructed on both sides of the TSF to divert surface flows.

The proposed Scott Creek tailings impoundment has been sized to store mill tailings production, a 200-year annual run-off (50 Mm^3 assuming channel diversions are in place), a minimum operating pond (5 Mm^3), plus an appropriate freeboard (emergency freeboard + height required to pass the probable maximum flood through the stage 1 spillway).

Seepage from the TSF will be collected in purpose-built ponds or wells and pumped back to the TSF.

(Water management for the TSF as well as the pit and waste storage facilities are discussed in section 18.3.2)

At closure, the TSF will be configured with minimal pond/wetland area, and re-vegetated with grasses and trees. Surface drainage within the impoundment will be directed towards a closure spillway. No discharge will be permitted until water quality meets discharge standards. The water will be treated prior to release if it does not initially meet discharge standards. Treatment will continue as long as necessary to ensure that all discharges to the receiving environment meet permit requirements.

18.3.2 WATER MANAGEMENT

Water management will be a critical component of the project design in this high precipitation environment. The most likely avenue for transport of contaminants into the natural environment will be through surface or ground water.

As such, Silver Standard will develop through its consultants a comprehensive water management plan that applies to all mining activities undertaken during all phases of the Snowfield project. The main objectives of this water management plan will be to divert non-contact water from the TSF and regulate the movement of water in and around the mine site.

The goals of this management plan will be to:

- provide a basis for management of the freshwater on the site, especially with the changes to flow pathways and drainage areas
- protect ecologically sensitive sites and resources, and avoid harmful impacts on fish and wildlife habitat
- provide and retain water for mine operations
- define required environmental control structures
- manage water to ensure that any discharges meet and/or exceed the permitted water quality levels and guidelines.

Strategies for water management include:

- diverting surface water from disturbed areas, protecting disturbed areas from water erosion, collecting surface water from disturbed areas and treating to meet discharge standards prior to release
- minimizing the use of fresh water and recycling water wherever possible to minimize the amount of water released
- monitoring the composition of release water and treating it to remove or control contaminants as required to meet discharge standards
- diversion channels or tunnels will be constructed to direct run-off away from disturbed areas.

18.3.3 *WASTE DUMP AND OPEN PIT WATER MANAGEMENT*

Run-off into the open pit and the waste dumps is assumed to be contact water that needs to be contained without any uncontrolled discharge to the environment. All run-off will be pumped up to a surge pond located near the tunnel portal that leads to the process plant 26 km to the east. Water in the surge pond will then be directed into a gravity pipeline that extends the full length of the tunnel and feeds into the plant for use in process. Pumping activities will be limited from the May to October period, when significant run-off is expected.

The overall water management strategy for the open pit and dumps is as follows:

- Run-off into the open pit will be managed with a combination of sumps and pumps. Given that a high percentage of run-off is expected to occur during snowmelt, the open pit bottom will be used as a primary sump during the "summer" season. With this strategy, the pumps would be operated from May to October leaving the pit bottom inaccessible during this period. The pumps would be sized appropriately so that the sump was dry by the start of winter. The coldest winter months would then be used to advance the open pit bottom by 1 or 2 benches below the main active mining bench each year.
- Pit dewatering groundwater will be directed to the surge pond and process plant pipeline.
- A total of six monitoring wells will be established down gradient of both interception trenches.
- Water collected at the toe of the east dump may be suitable for release (dependent on water quality) and may be discharged back into Mitchell Creek.

A preliminary water balance for the open pit and waste dumps was constructed using a monthly time-step. An average annual precipitation of 2,033 mm was assumed. Annual lake evaporation and sublimation was estimated at around 215 mm. Given that potential evaporative losses are very low, resulting run-off coefficients are very high. Based on average precipitation conditions, an average annual run-off volume of 8.4 Mm^3 (956 m^3/h) has been estimated for the life-of-mine. Early in the mine life, there will be opportunities to minimize run-off inflows by constructing diversion ditches upslope of the open pit and East dump. Annual run-off volumes will therefore be at a minimum for the first several years of operations and at a maximum later in the mine life when the open pit and waste dumps footprints are fully developed.

All flows from the mine, waste rock storage areas, and dewatering wells will be pumped to the upper tunnel portal and piped 26 km to the process plant and eventually be discharged to the TSF.

Mine water as well as run-off from the waste dumps is assumed to be contact water and is planned to be contained without any uncontrolled discharge to the environment. The overall water management strategy for the open pit and waste rock dumps is outlined below.

An interception trench will be constructed at the toe of the Southwest Waste Rock Facility (WRF). Run-off from the Southwest WRF could be directed to the tunnel portal by gravity pipeline.

An interception trench will also be constructed at the toe of the East WRF. Run-off from the East WRF will be pumped to the tunnel portal. Water collected at the toe of the East WRF may be suitable for release so that arrangements will be made to allow for discharge into Mitchell Creek.

OPEN PIT

In order to minimize run-off inflows and depending on the final pit sequencing, there will likely be opportunities for run-off diversion ditches to be constructed at the pit crest, particularly during the initial phases (Yrs -1 to +8) when up to three separate pits will be developed. The ultimate pit footprint reaches the watershed divide, so there is no need for upslope surface diversions later in the mine life. Also, the northern extent of the pit daylights above Mitchell Creek, so no diversion is needed at the downslope end either. The open pit has an ultimate footprint of approximately 272 ha (or 2.7 km^2). The final pit footprint will also capture an additional undisturbed area of 50 ha in the southeast corner. This area does not include run-off from the Southwest WRF, which will be diverted to the upper tunnel portal.

EAST AND SOUTHWEST WASTE ROCK FACILITY

Interception trenches will be located at the toe of both waste rock facilities to collect both surface run-off and seepage. The collection trench at the base of the East WRF will have the assumed following dimensions:

- 300 m in length
- 5 m in depth consisting primarily of bedrock
- 0.5H:1V side slopes in bedrock (assume blasting required)
- a bottom width of 4 m.

In addition, three monitoring wells will be located down gradient of the interception trench. Water will be pumped directly from the collection trench to the upper tunnel portal. It is assumed that the East WRF will be constructed from the bottom up and that there will be opportunities to construct upslope diversion ditches (that divert the water to the east) to minimize run-off volumes captured by the interception trench at the toe of the dump. A more detailed analysis will be conducted during the next level of design when end-of-period maps are available.

The interception trench for the Southwest WRF will capture a much smaller drainage area and flows will be transported via a gravity pipeline to the upper tunnel portal. As with the East WR, three monitoring wells will be located down gradient of the interception trench.

CALCULATED FLOWS

Figure 18.15 shows average annual flows in m^3/h averaged over the LOM for average precipitation conditions. As the pits and waste dumps develop, increased run-off volumes will need to be handled. This increase is illustrated in Figure 18.16, which shows average run-off volumes for the final year of mining (again with average precipitation conditions). Average run-off is shown to increase approximately 50% with the increased footprint.

Figure 18.15 Average Annual Flows LOM, Average Precipitation Conditions

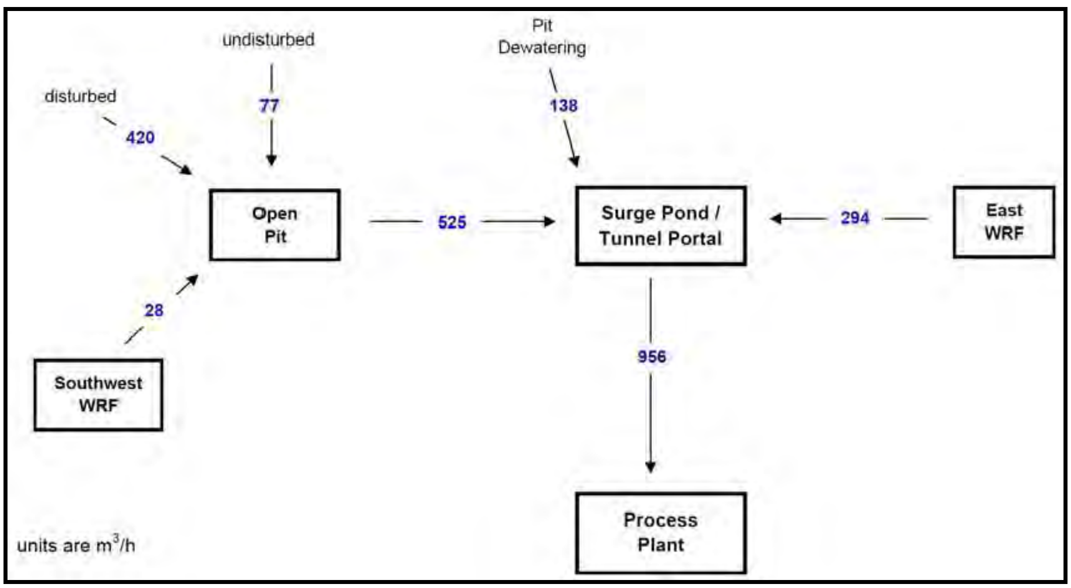
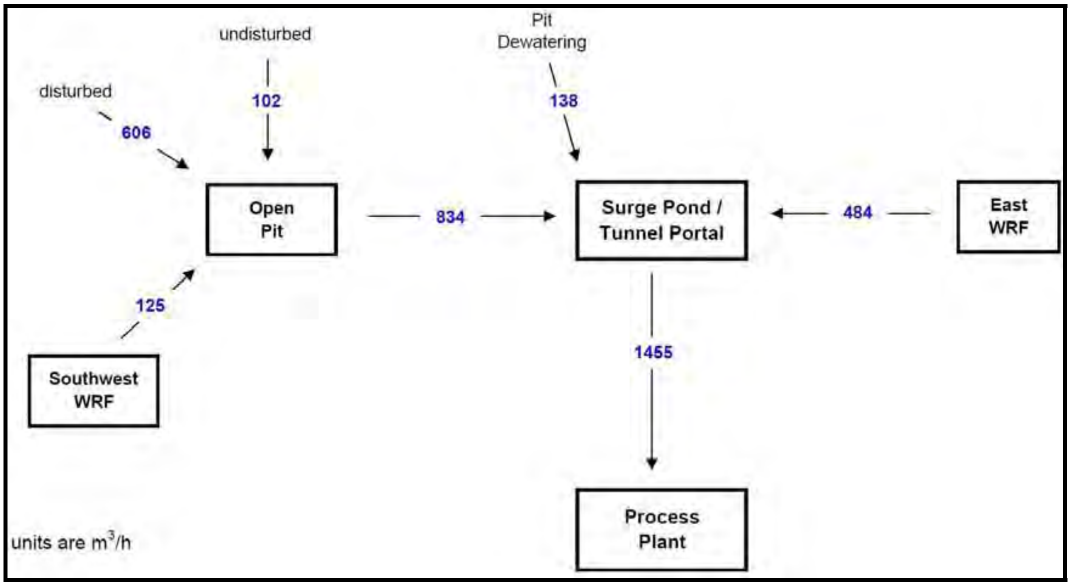


Figure 18.16 Average Annual Flows at End of Mine Life, Average Precipitation Conditions



Depending on the risk management and mining strategy employed by Silver Standard, there are a number of ways in which the run-off could be handled. The recommended strategy is as follows.

Given that a high percentage of run-off is expected to occur during snowmelt, the open pit bottom would be used as a sump during snowmelt and the remainder of summer/early fall. Using the bottom of the pit as a sump would allow a lower pumping rate to be used through the six-month warm period (May to October). With this strategy, the pumps would be operated from May to October leaving the pit bottom inaccessible during this period. The pumps would be sized appropriately so that the sump will be dry by the start of winter. The coldest winter months would then be used to advance the open pit bottom by one or two benches below the main active mining bench each year. This advanced open pit development would provide additional storage capacity and allow a reduced pumping capacity. For example, the average annual pumping rate out of the open pit is approximately 834 m^3/h by the end of mine life, for average precipitation conditions (Figure 18.16). If this volume of water was pumped out over the course of six months (May to October), the average pumping rate would be 1,670 m^3/h. However, the pumping system should be sized to accommodate years with above average precipitation as the pit bottom must be accessible for a portion of the year. Annual precipitation with a 200-year return period has been conditionally adopted as the design standard for the pumps and pipeline. Therefore, the maximum pumping rates required during mine life are as follows (assuming a six-month pumping period):

- 3,100 m^3/h from the open pit sump
- 1,100 m^3/h from the east WRF interception trench.

Accounting for pit dewatering flows, the pipeline from the tunnel portal to the process plant would be sized for approximately 4,500 m^3/h or 1.25 m^3/s. If the bottom of the open pit could not be used as a sump for more than one or two months, significantly higher pumping rates would be required.

Reduced maximum pumping requirements would apply earlier in the mine life, when the mine footprint is significantly reduced. Also when the pits are initially developed, there are three satellite pits, each of which would require their own system of sumps and pumps. These design elements will need to be assessed in more detail during the next level of engineering design.

A 200-year return period is a conservative assumption for open pit design, but may be warranted given the extreme climatic conditions experienced in this region (high annual precipitation and minimal evaporative losses due to high humidity and low temperatures). Most pit sumps are designed for 10 to 25-year return period events. If a 10-year return period criterion was applied to the open pit water management, then the pipeline to the process plant would be designed for a peak flow of 3,400 m^3/h. The return period ultimately adopted for final design should reflect the level of risk that Silver Standard is willing to accept. A lower pumping rate could also be adopted if pumping extended beyond the six-month "summer" period into the winter.

18.3.4 TAILINGS STORAGE FACILITY WATER MANAGEMENT

GENERAL

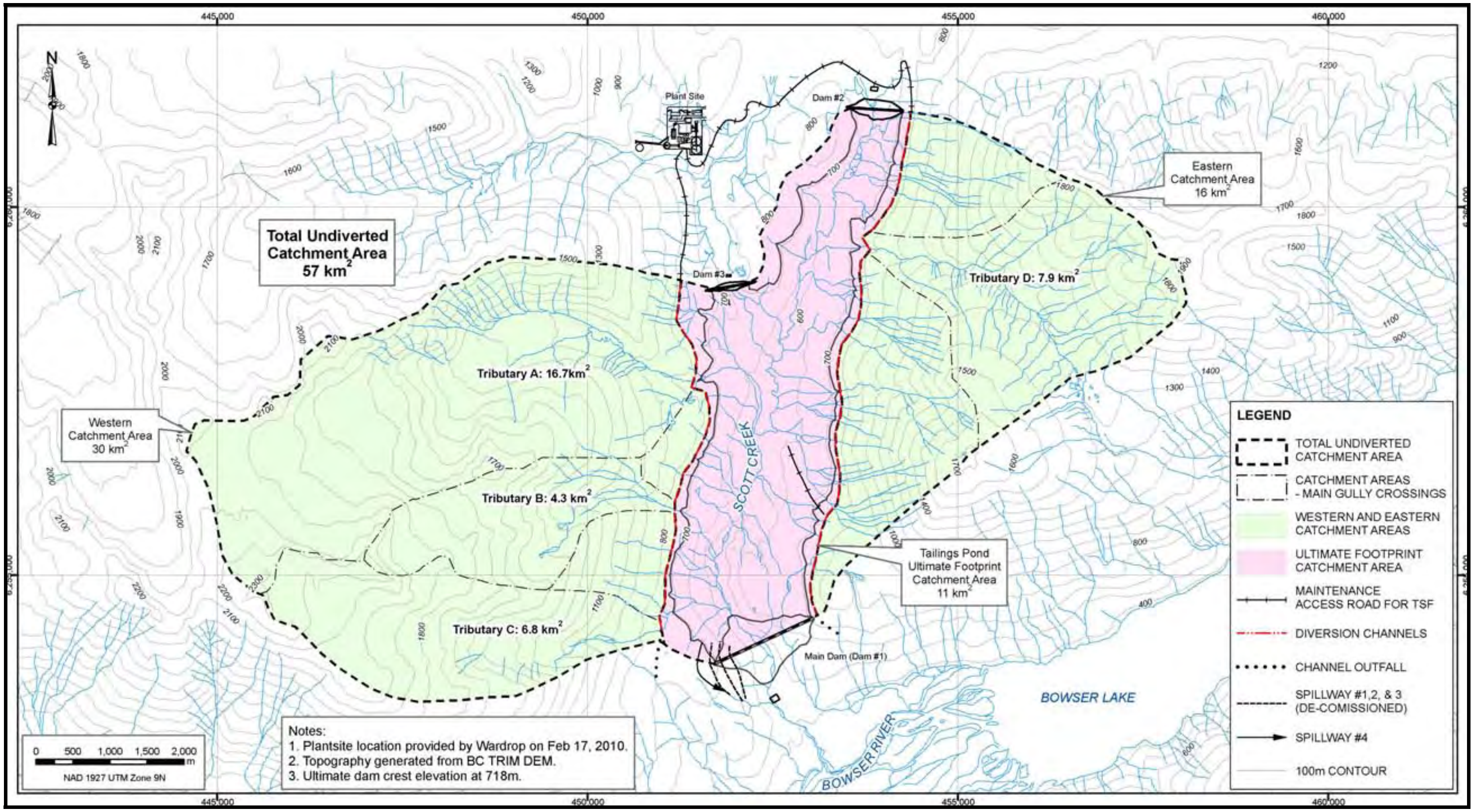
The catchment area reporting to the TSF is approximately 57 km^2 (Figure 18.17). Diversion channels are proposed on both the east and west sides of the valley to limit inflows, as the facility will be operating with a surplus of water given the high annual precipitation and low evaporation rates that characterize the region. The area diverted around the TSF is approximately 46 km^2. At capacity, the footprint of the tailings impoundment will be about 11 km^2.

The proposed TSF will occupy the valley bottom of Scott Creek: the base of Dam #1 is at an approximate elevation of 421 m. The headwaters of Scott Creek are located on the west side of the TSF, where maximum of 2300 masl are attained. Here, glaciers occupy a significant proportion of the upper watershed and feed three principal tributaries that discharge into Scott Creek.

Lower terrain is present on the east side of the TSF. Maximum are about 1900 masl and only one significant tributary discharges into Scott Creek.

Downstream of the TSF, Scott Creek discharges into the Bowser River just upstream of Bowser Lake.

Figure 18.17 Scott Creek – Proposed TSF – Catchment Areas Plan



Diversion structures will be designed to manage freshet flows and 1-in-200-year storm events. Greater capacity will be provided if required based on an assessment of the consequences of failure. Lesser capacity may be provided where overflows can be stored and managed by other downstream structures, such as the TSF. Disturbed areas such as overburden storage sites will be vegetated or otherwise protected from erosion. Run-off from these areas will be directed to settling ponds with sufficient capacity to provide the retention time required to achieve discharge standards. The Metal Mining Effluent Regulations (MMER) limits total suspended solids to 15 mg/L. Flocculation may be required to meet discharge standards in some instances. Where possible, reclaim water will be used in preference to fresh water for makeup purposes in order to minimize the withdrawal of fresh water from natural systems and reduce the volume of contact water discharged to the environment. Contact water may require treatment. The quality of water in streams affected by the project, and of all discharges, will be monitored on a regular basis.

The proposed Scott Creek TSF will have a net annual surplus of water based on the preliminary monthly water balance. The TSF provides storage up to a 200-year annual run-off and diversion channels will be constructed to reduce the catchment area reporting to the TSF. Water will be reclaimed from the pond for use in the process plant; however, an average annual water treatment volume of 14.1 Mm^3 will be required, although this will vary significantly due to natural variations in annual precipitation. Installed capacity for treatment and release of excess water from the impoundment will be approximately 3500 m^3/h.

SPILLWAYS

To protect the integrity of the main tailings dam, flows in excess of the 200 year event a run-off will pass through one of the four staged spillways excavated into bedrock in the East abutment during the mine life. These spillways have been designed to pass the routed flow from a Probable Maximum Flood (PMF = 744 m^3/s) from the entire catchment area 56.7 km^2. Based on this discharge, the spillways will have an 8 m (minimum) wide channel invert and surcharge depths[1] of 2 m to 5 m.

Spillways 1 through 3 are designed as unlined emergency or temporary structures. Spillway 4 will be the closure or permanent structure. Flows from these spillways would discharge into a tributary of the Bowser River via erosion resistant outlet channels.

Spillway 1 will be constructed prior to the start of operations and will be operational for a couple of years of operations. Spillways 2 and 3 will be constructed during operations. In order to use Spillways 2 and 3 for more than one year, a portion of the spillway invert must be raised incrementally using grouted gabion baskets. The final closure spillway (Spillway 4) will be constructed in Year 25.

Flows from these spillways would discharge into a tributary of the Bowser River via erosion resistant outlet channels. During the mine life, pond water will be reclaimed from the TSF to the process plant via a floating reclaim barge located near the eastern side of the impoundment. A pump barge will pump any excess water to the crest of the main dam where it will be piped down and released into the Bowser River tributary.

[1]Surcharge depth is a function of the construction stage with increasing tailings footprints leading to reduced spillway discharge due to flood attenuation.

Diversion Channels

Diversion channels will be constructed above the west and east sides of the ultimate tailings pond to divert fresh water (or non-contact water) around the Scott Creek impoundment during the entire mine life. These channels are essential to maintain water balance given the large catchment area and wet climate. Approximately 13 km of channel are proposed around the impoundment and are designed to pass flows from a 200-year flood event. Both diversion channels will be lined with HDPE to increase their efficiency and also rip rapped for channel maintenance (i.e. channel clean out). Flows from these channels discharge into tributaries of Bowser River by way of erosion resistant outlet channels. The diversion channels will be breached and reclaimed for closure.

Major stream crossings are at risk from geomorphic events such as debris flows, debris floods, and snow avalanches. For this study, it has been assumed that debris barriers will be required at four of the major stream crossings. Further studies are required to determine the level of geomorphic risk at all diversion channel/tributary channel junctions.

Blockage of the channels could occur due to soil and rock falls from the upper slopes as well as snow avalanches. Regular channel inspections and maintenance will be required to keep the channels operational year-round.

Seepage Recovery Facilities

Foundation treatment for the tailings dams (Dam #1, 2 & 3) will be designed to minimize seepage out the tailings impoundment; however seepage recovery systems will be constructed at the toe of each dam to collect potential seepage out of each dam and foundation. Any seepage water collected will be pumped back to the tailings impoundment unless it meets the specified water quality guidelines for discharge. The seepage recovery system for each dam will include a seepage collection trench and pond, and interceptor wells located immediately downstream of the toe. Monitoring wells will be located farther downstream of the toe for groundwater sampling and testing.

Diversion Tunnel (during construction)

A 1.2 km long lined diversion tunnel through the right abutment of the main tailings dam is proposed (Figure 18.20). The purpose of this tunnel is to convey water from behind a 24 m high temporary diversion dam around the proposed main starter dam footprint, in order to build the starter dam in the dry. The tunnel must be completed prior to any dam earthworks placement in the valley bottom. The tunnel will be in use for two to three years (approximately the entire construction period of the main starter dam). Prior to the start of operations, the tunnel will be closed with a permanent concrete plug after construction of the starter dam.

The tunnel is designed to pass flows from a 200-year return period flood event from the entire catchment area. Based on this criterion, a 4 m wide by 4.6 m high horseshoe shaped tunnel is required. No site specific data regarding the rock conditions along the proposed tunnel alignment was available for this study. As such, the tunnel was assumed to be fully lined with shotcrete and pattern bolting.

The diversion dam, located at the upstream portal location of the tunnel, is designed as a compacted rockfill dam with an upstream low-permeability facing. The dam has a 20 m wide and 165 m long crest. The upstream and downstream slopes are assumed to be 3H:1V.

PROCESS WATER REQUIREMENTS

Water requirements for the process plant will be met from two primary water sources:

- reclaim from the TSF pond
- seasonal run-off from the open pit and WRF that will be piped to the process plant through the proposed 26 km tunnel.

Pond water will be reclaimed from the TSF to the process plant via a floating reclaim barge located on the east side of the impoundment. Because the mine site water will only be available from May to October, TSF reclaim rates will be at a minimum during the summer months and at a maximum during the winter (when reclaim will provide almost all of the process plant water requirements). This strategy will require that there is a sufficiently large supernatant pond volume in October so that ongoing void losses in the winter are offset by the pond volume, which would gradually deplete until the following spring freshet.

PRELIMINARY WATER BALANCE

A preliminary water balance model (WBM) for the TSF and process plant was constructed using a monthly time-step. The following assumptions were used as input to the WBM:

- A final tailings settled dry density of 1.3 t/m^3 and a solids specific gravity of 2.7.
- Tailings production of 120,000 tpd at 35% solids by weight. This results in a tailings slurry water requirement of 9,286 m^3/h.
- The process plant has a minimum freshwater requirement of 300 m^3/h.

- An average annual precipitation of 1,525 mm and evaporative/sublimation losses of 374 mm for open water.
- Run-off co-efficients of 0.75 to 1 were assumed for the various land surfaces (i.e. undisturbed ground, active tailings beach, inactive tailings beach, and pond).
- Diversion channels on the west and east sides of the TSF have an assumed diversion efficiency of 80%.
- All of the contact water collected at the mine site (open pit run-off, seepage and surface run-off from the dumps, and pit dewatering groundwater) will be used in the process plant.

Because of high annual precipitation, minimal evaporative losses, and the footprints of the various facilities, the TSF is expected to operate with a net annual surplus of water in most years. However, it is currently expected that surplus water will be of suitable water quality for discharge to Scott Creek. Parameters of immediate concern, ammonia and cyanide, are expected to naturally degrade given suitable residence time in the supernatant pond. Surplus tailings water would be discharged during the May to October period. A pump barge will pump the excess water to the crest of the main dam where it will be piped down and released into the Bowser River tributary.

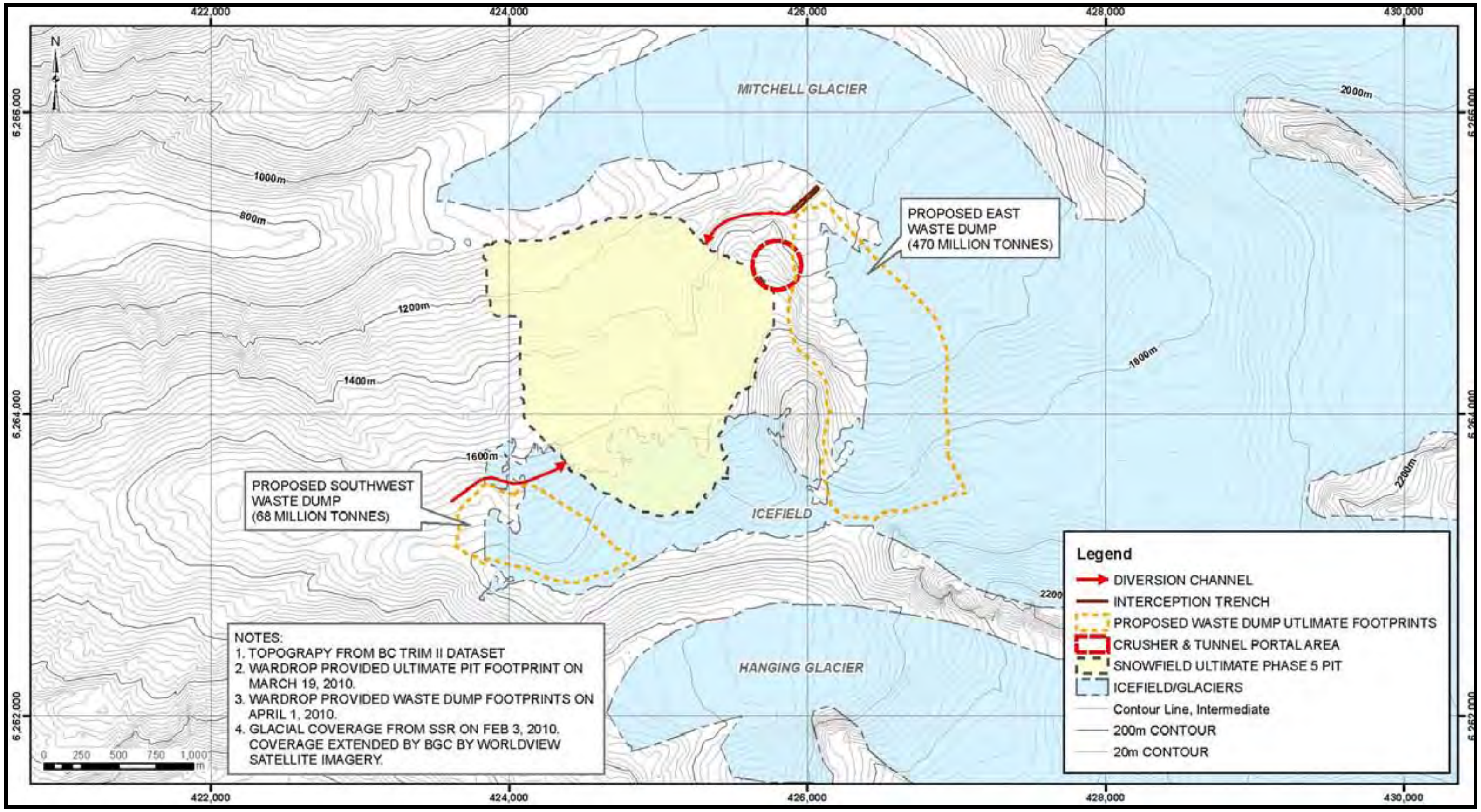
Based on average precipitation conditions, the supernatant pond is estimated to an average annual surplus volume of 14.1 Mm^3 (1,616 m^3/h) over the life-of-mine. Assuming that the discharge would be compressed into a 6 month period (or potentially even less), the average discharge rate of the pumps would be about 3,230 m^3/h. Surplus volumes are expected to vary significantly due to natural variations in annual precipitation and the extent of development. As the pits and waste dumps develop, increased run-off volumes are pumped to the process plant from the mine site, thereby reducing TSF reclaim volumes and increasing seasonal discharge volumes.

18.4 Preliminary Geotechnical Design

18.4.1 Waste Dumps

Mine design and scheduling results in placement of approximately 520 Mt of waste rock in the East and Southwest dumps, which refers to their relative location to the open pit, respectively (Figure 18.18). The majority of the waste materials will be placed in the East dump, which is proposed to contain approximately 90% of the total waste rock. The remainder will be placed in the Southwest dump.

Figure 18.18 Snowfield Open Pit and Proposed Waste Dumps



The following parameters were provided to Wardrop to design the waste dumps:

- 37° angle of repose for dump faces
- a swell factor of 30%
- overall dump slopes of 2:1
- no restrictions on free dumping height.

The assumed angle of repose for the waste rock assumes that it will generally consist of “fair” quality rock, consistent with the majority of the rock observed in the Snowfield pit area. Poor quality rock which will be mined from the landslide area is not desirable in the foundation of the waste dump. If possible, these materials should be mixed with better quality waste rock to avoid zones of weakness within the waste dump. This material will not likely be suitable as rock drain construction material.

The swell factor assumed is appropriate for the waste rock but will vary somewhat based on the construction sequence of the dump. Dumps built from the bottom up could be denser and may have a slightly lower swell factor. The recommended overall dump slopes of 2:1 are at the upper end of those suitable for reclamation; however, slopes at these angles can still be re-graded. Re-vegetation needs and long term stability requirements need to be considered in selecting the final overall waste dump slope.

Free dumping height constraints are contingent on the absence of weak materials in the foundation. There are some advantages to using free dumping methods to constructing waste dumps, as rock drains can be developed by segregation of the rock. However, heights such as those proposed for the Snowfield waste dumps are well beyond those required to achieve adequate segregation.

The scoping level waste dump design provided by Wardrop has the East dump proposed for an area immediately adjacent to the Mitchell Glacier (Figure 18.18). The East dump toe is located El. 1270 m and the crest is located at El. 1960 m, resulting in a repose angle waste dump 690 m high. At the end of the mine life, this dump will be approximately 2.1 km long and 0.8 km wide. The Southwest dump is located along the ridgeline. The slopes of this smaller dump vary from El. 1650 m at the toe to El. 1890 m at the crest of the dump. At the end of the mine life, this dump will be approximately 1.2 km long, 0.5 km wide, and 240 m high.

The dump locations and configurations are suitable for preliminary planning purposes but will not meet long term stability requirements. For the scoping study, it has been assumed that the dumps will not be founded on weak materials or active portions of the glaciers in the area, and any ice that currently exists under the existing footprints will have been removed or melted prior to the start of operations. For this scoping level of study the locations selected are a reasonable starting point given the space constraints on this project; however, the above assumptions will need to be confirmed during the next stage of project study.

At the next phase of design the foundations beneath the proposed dump areas should be characterized so that the depth, extent and strength of any soil, rock, and ice that these dumps could be founded on are understood. Areas will need to be defined where surface water will come into contact with the waste material and ways to contain and manage this contact water will need to be developed. If significant drainages or discharge areas are to be covered with waste rock, there will be constraints on the average grain size (D_{50}) of the waste rock to convey the creek flows. This will require an estimate of in-situ block size from the pit area, the typical block shape, and an idea of which rock type the rock drain might be constructed out of. Geotechnical stability analyses of the dumps have not been conducted because there is insufficient foundation information available to date for the dump sites. Geotechnical site investigations and stability analyses are appropriate at the next phase of design. A detailed set of recommendations for further work required for the next stage of design is outlined in Section 19.

18.4.2 *Pit Slope Angles*

BGC compiled data from available reports, databases, and geological models to support preliminary open pit slope design criteria estimates for the proposed Snowfield open pit. It is important to note that the ultimate pit would include a south highwall approximately 1100 m high, which is near the maximum slope height achieved by any existing open pit mine. In addition, the development of the proposed pit requires mining of the “Snowfield Landslide”, a large-scale slope deformation which occurs on the south side of the Mitchell Valley.

Preliminary design criteria estimates are based on a review of rock mass properties, major geological structures, and possible structural domain boundaries. Available geotechnical and geological data has been used to estimate bench, interramp, and overall slope scale design criteria. Geotechnical core logging completed by Silver Standard on exploration core obtained in 2007, 2008, and 2009 has been heavily relied upon for these designs. BGC also drew upon its experience with other porphyry deposits and their associated rocks within BC. Data used appears to be appropriate for a preliminary or scoping-level design.

The geotechnical core logging data available for the Snowfield deposit includes rock quality designation (RQD) and fracture intercept (average distance between adjacent discontinuities). BGC made conservative estimates of intact rock strength and joint (i.e. discontinuity) condition to develop a preliminary rock mass rating (RMR ’76) for the Snowfield deposit. Rock mass strength estimates have been developed for slope stability analyses and the assessment of open pit slope angles. At the current level of design, the rock within the Snowfield deposit is treated as a single geotechnical unit. The majority of the rock mass, including the expected rock of the ultimate pit slopes, is estimated to be fair (RMR ’76 of 41 – 60), with some poor (21 – 40) zones expected in the near-surface deformation zone of the Snowfield landslide.

The study area of the Snowfield deposit includes significant large-scale geological structures, including the west dipping Mitchell Thrust Fault (24° – 309° ; dip – dip direction) and steeply dipping to vertical Brucejack and Snowfield Faults (~ 75° – 070°). In addition, the rock mass of the Snowfield deposit has a schistose (foliation) fabric (70° – 005°). Due to the limited data available at the PA stage has been combined into a single structural domain.

The PA level open pit design criteria developed for the proposed open pit of the Snowfield deposit are presented by design sector (Table 18.22). Design sectors are defined by ranges of slope azimuths and roughly correspond to the expected north, east, south, and west walls of the pit. The blending of slope angles between adjacent design sectors must be accomplished so that the maximum slope angles are not exceeded within any sector. This requires blending of steeper sections into less steep sections to be completed within the steeper sector.

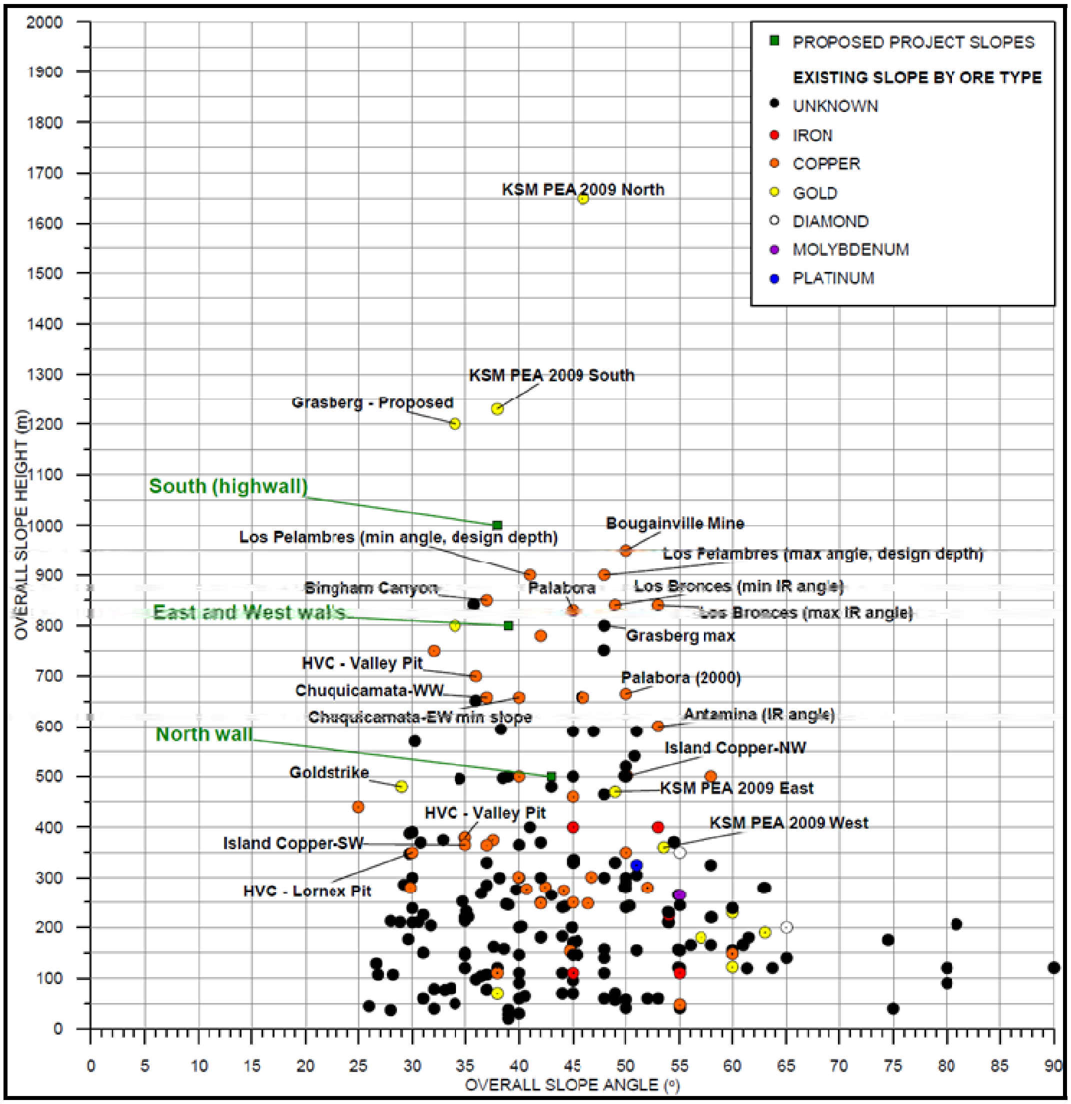
A double bench (2 m x 15 m) configuration was assumed for the final walls of the PA level open pit, resulting in a vertical distance of 30 m between catch benches. Based on industry experience, a 65° bench face angle is generally achievable in porphyry deposits using traditional production drill and blast methods, with trim and buffer blasts on the final pit walls. This was assumed to be applicable for all domains.

The overall slope design criteria recommended are within the range of those achieved for similar open pit scale designs in other parts of the world (Figure 18.19). Where overburden is encountered, slopes should be benched with bench height limited to 15 m (single benches). BGC recommends that bench face angles be limited to 45° (1H:1V); catch benches should be a minimum of 8 m wide.

Table 18.22 Preliminary Open Pit Slope Design Criteria

	Slope Azimuth		Assumed Overall Slope Height	Bench Height	Bench Face Angle	Catch Bench Width	Max. Interramp Angle	Max. Interramp Height	Max. Overall Slope Angle
Design Sector	**Start (°)**	**End (°)**	**Oh (m)**	**Bh (m)**	**Ba (°)**	**Bw (m)**	**Ia (°)**	**Ih (m)**	**Oa (°)**
SF-357	317	037	510	30	65	18.5	43	600	43
SF-069	037	102	800	30	65	23.0	39	600	39
SF-163	102	225	1080	30	65	20.5	41	600	36
SF-271	225	317	800	30	65	18.0	43	600	39

Figure 18.19 Comparison of Proposed Overall Slopes with Industry Experience



Note: Slope Height Slope Angle Data have been collected from a number of published and unpublished sources.

18.4.3 TAILINGS STORAGE FACILITY

SUMMARY

A TSF is designed for 966 Mt of mineralized material based on a mill throughput of 120,000 t/d for the 23 year LOM. During the mine life, mineralized material will be extracted from the Snowfield open pit. The mineralized material will be processed, generating approximately 966Mt of tailings and 520 Mt of waste rock.

All tailings will be contained within the Scott Creek Valley, located approximately 30 km east-southeast of the pit. The tailings will be deposited within the valley and retained by three cross-valley tailings dams to be constructed over the mine life. The main tailings dam (Dam #1), located furthest north and approximately 2.5 km upstream of the confluence with Bowser River, will be constructed in stages to an ultimate crest elevation of 704 masl, with an ultimate dam height of approximately 283 m above centreline. Two additional tailings dams (Dam #2 and Dam #3) must be constructed at the south end of the impoundment during operations to provide containment. The ultimate dam heights for Dam 2 & 3 are 61 m and 16 m (above centreline) respectively.

Tailings will be transported hydraulically to the tailings deposition area where they will be spigotted off the main tailings dam crest and valley slopes. During operations, an operating pond will be created to allow water to be reclaimed from the pond back to the plant. This pond will facilitate settling of suspended solids and natural degradation of cyanide and ammonia. At the end of the mine life, the tailings impoundment will be approximately 7 km long and 1.5 km wide. The tailings will be flooded during operations and for perpetuity at closure.

TAILINGS DAM DESIGN

All three tailings dams are designed as compacted rockfill dams with a central low-permeability (i.e. clay till) core and filters immediately downstream of the core. They will all be raised via downstream construction method during operations to an ultimate crest elevation of 704 masl.

The dam designs provided for this study are at a scoping level. No site investigations (i.e. mapping, drilling, geophysics, or test pits excavations) were completed as part of this work. As well, no seismic hazard assessment, stability analyses, or seepage analyses were completed.

DESIGN CRITERIA

Table 18.23 summarizes the design criteria applicable to the tailings dam. Standard procedures from the Canadian Dam Association (CDA) and International Congress on Large Dams (ICOLD) were applied for these scoping level designs. The design criteria were established in discussion with Silver Standard.

Table 18.23 Tailings Dam Design Criteria

Criteria	Description/Comments
Total Mineralized Material	966 Mt
Mill Throughput	120,000 t/d
Mine Life	23 a
Tailings Dry Density	1.3 t/m³
Total Tailings	966 Mt (or 743 Mm³)
Capacity – Starter Dam	TSF to store:2 years tailings (88 Mt or 67 Mm³) + 200/a run-off + 5 Mm³ operating pond + 5 m (freeboard)
Capacity – Ultimate Dam	TSF to store: 966 Mt (or 743 Mm³) of tailings + 200/a run-off + 5 Mm³ operating pond + 5 m (freeboard)
Maximum Design Earthquake (MDE)	1 in 10,000 earthquake with a peak ground acceleration of 0.2 g
Design Flood	Store 200/a run-off
Operating Pond	5 Mm³
Spillway Design Capacity	run-off from 24 h Probable Maximum Precipitation (PMP)
Design Flood Freeboard	5 m above maximum pond level

MAIN TAILINGS DAM (DAM #1)

The main starter dam will be constructed to a crest elevation of 597 masl (176 m dam height above centerline) and has an approximately 720 m crest length. The rockfill shells will be constructed with compacted quarried rock with 1.7H:1V side slopes. The central low-permeability core is 100 m wide at the base with 1H:7V slopes. Immediately downstream of the core, are two 4 m wide granular filters zones (fine filter & coarse filter) and one 4 m wide zone of transition rockfill.

The main starter dam has been sized to store two years of mill tailings production, a 200 a run-off, an operating pond, plus 5 m of freeboard (emergency freeboard + height required to pass the probable maximum flood through the Stage 1 spillway). Figure 18.20 shows the proposed main starter dam in plan. During operations, the main tailings dam will be raised to an ultimate dam crest elevation of 704 masl (283 m high above centreline). Figure 18.21 shows the proposed ultimate dam in plan. A typical cross-section through the ultimate main tailings dam is shown in Figure 18.22.

Figure 18.20 Scott Creek – Proposed TSF – Starter Dam Layout Plan

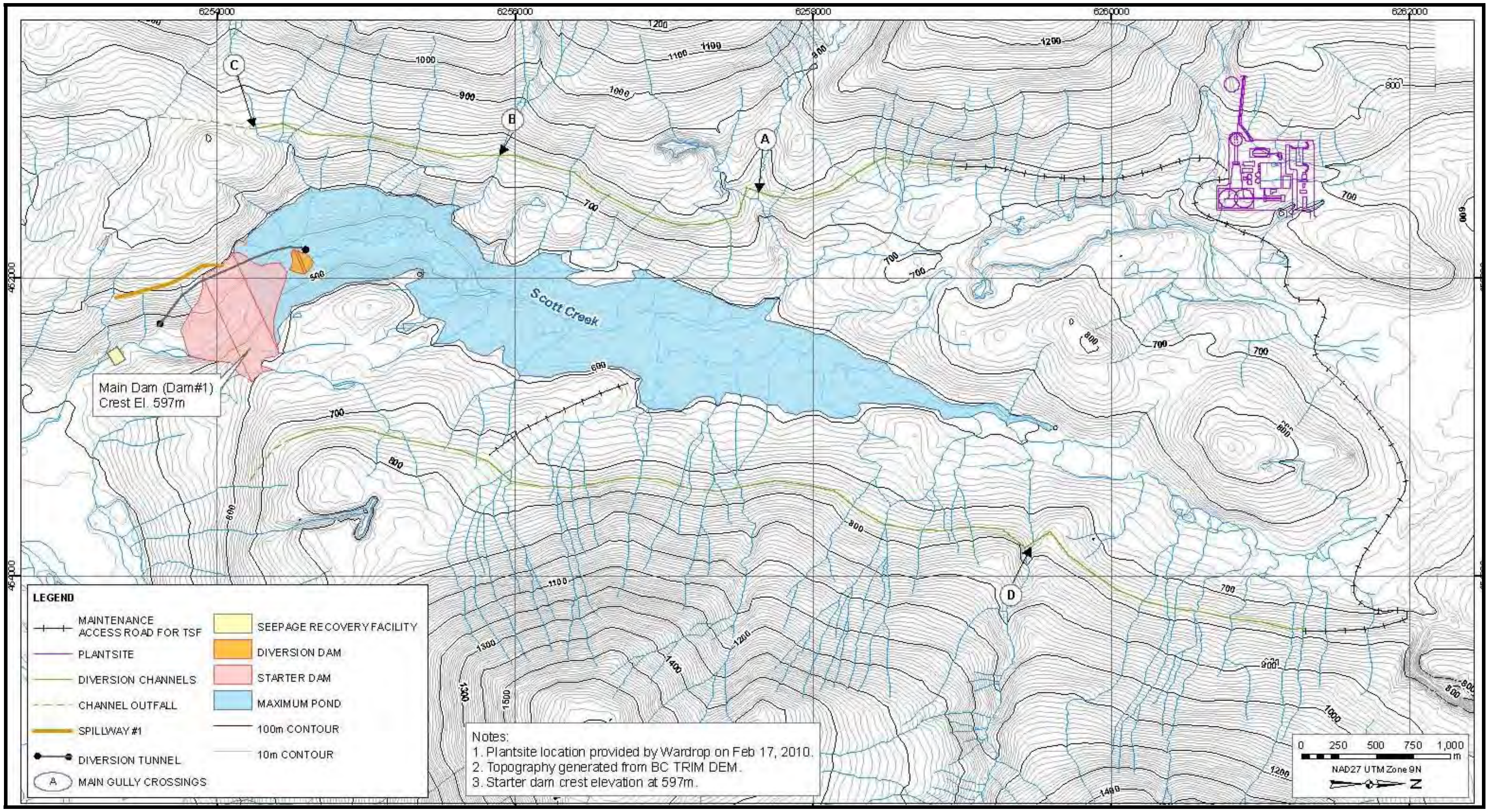


Figure 18.21 Scott Creek – Proposed TSF – Ultimate Dam Layout Plan

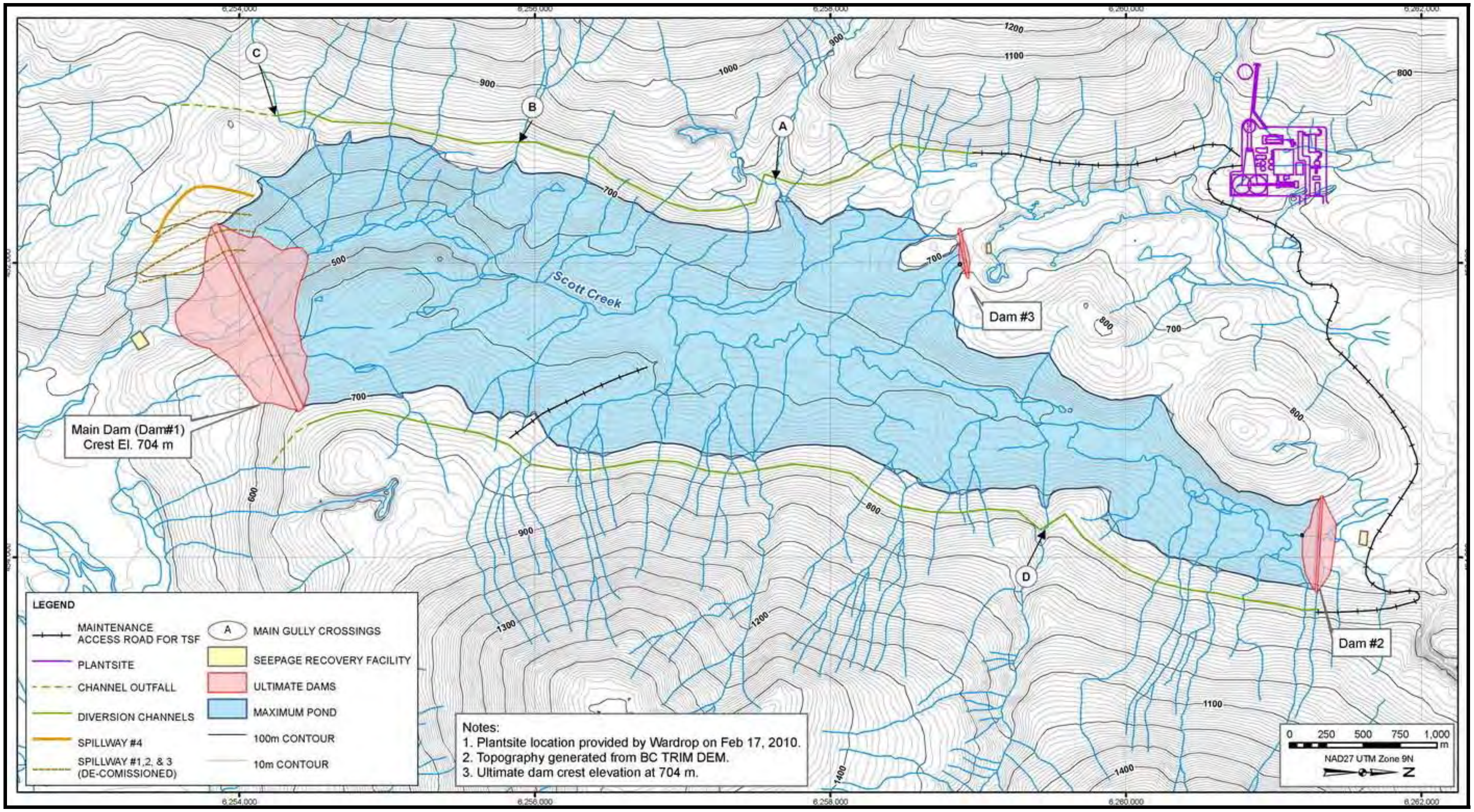
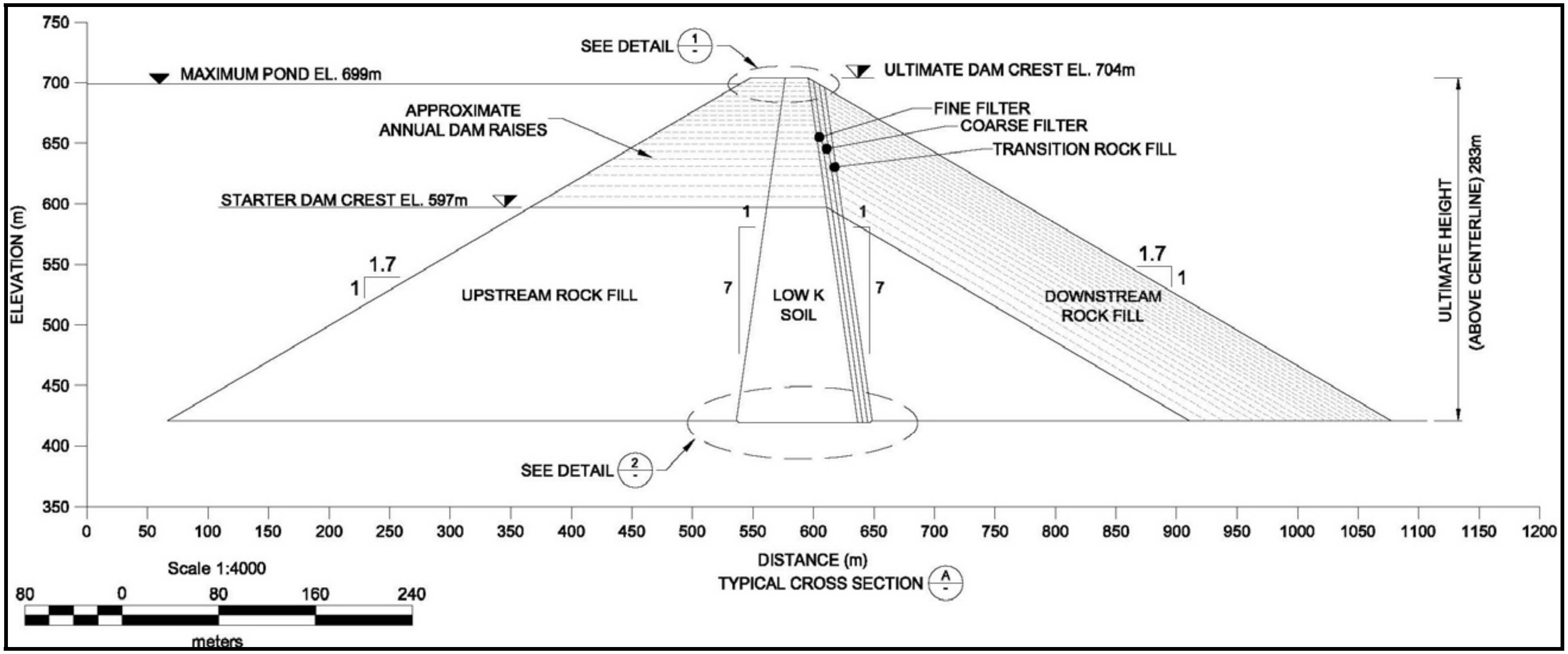


Figure 18.22 Scott Creek – Proposed TSF – Typical Main Dam Section



DAM #2

Dam #2, located at the northern end of the impoundment, is designed as a compacted rockfill dam with a central low-permeability (i.e. clay till) core and filters immediately downstream of the core. It is the second largest dam required for the Scott creek TSF. This dam is sized to provide containment of the impoundment; the ultimate dam crest elevation is the same as the main dam – El. 704 masl (ultimate dam height of 61 m above centreline). The starter dam will be built to dam crest El. 655 masl (11 m high) in Year 8. Downstream dam raises have been assumed throughout the remainder of the mine life.

DAM #3

Dam #3, located in the northwest corner of the impoundment, is designed as a compacted rockfill dam with a central low-permeability (i.e. clay till) core and filters immediately downstream of the core. It is the smallest of the three dams required for the Scott creek TSF. This dam is sized to provide containment of the impoundment; the ultimate dam crest elevation is the same as the main dam – El. 704 masl, with an ultimate dam height of 16 m. The starter dam will be built to dam crest El. 700 masl (12 m high) in Year 18. One downstream dam raise is required during the remainder of the mine life.

DAM FOUNDATIONS

No site specific data regarding the subsurface stratigraphy and engineering characteristics under each of the three dam footprints was available for this study. Based on a review of some high resolution satellite imagery, all three dams are assumed to be founded on less than a few metres of glacial till, alluvium and/or colluvium overlying bedrock. From regional geology mapping (Groves, 1983), the bedrock is assumed to be of sedimentary origin from the Salmon River Formation.

Foundation preparation will consist of clearing and grubbing followed by a nominal stripping depth over the entire dam footprint. Within the core key trench, scaling and cleaning must be completed followed by some dental concrete and slush grouting. Due to the lack of site specific data on the dam foundations, no significant foundation grouting (i.e. grout curtain) or trimming of the rock abutments has been assumed. These assumptions will have to be checked at the next phase of design.

AUXILIARY STRUCTURES

In addition to the three tailings dams, the following auxiliary structures are required for the TSF:

- Spillways – A series of four spillways on the right abutment will be constructed over the mine life to protect the integrity of the Main Tailings Dam.

- Operations Diversion Channels – Approximately 13 km of diversion channels will be constructed above the west and east sides of the ultimate tailings pond to divert fresh water (or non-contact water) around the Scott Creek impoundment during the entire mine life.
- Seepage Recovery Facilities – Seepage recovery systems will be constructed at the toe of each tailings dam to collect potential seepage out of each dam and foundation.
- Construction Diversion Tunnel – A 1.2 km long lined diversion tunnel through the right abutment of the main starter tailings dam is required to convey flows from Scott Creek around the starter dam footprint during its construction.

18.5 PROJECT EXECUTION PLAN

The preliminary project execution schedule was developed to provide a high level overview of all activities required to complete the project and is summarized in Figure 18.23.

Upon receipt of construction and operating permits, the project will take approximately 4 years to complete, from project release through to the introduction of first mineralized material and commissioning.

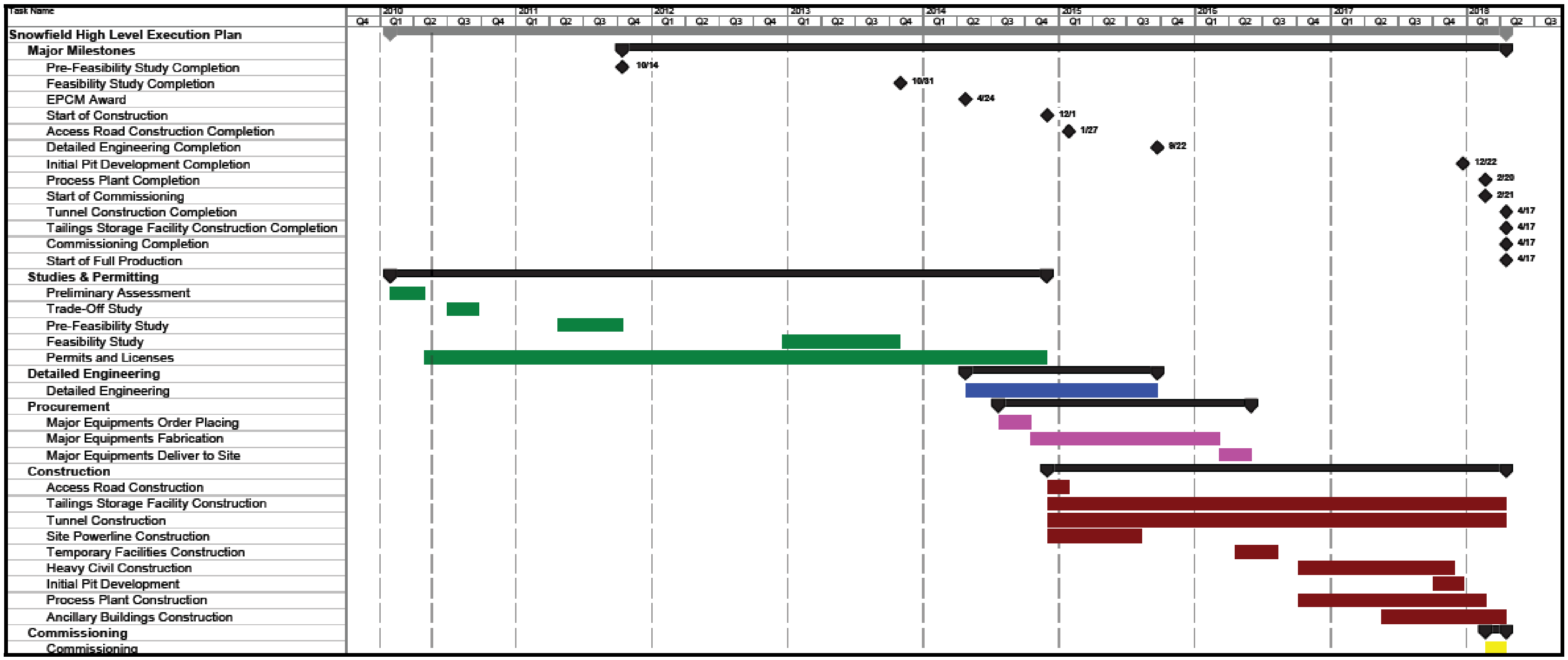
The project will be executed and constructed in accordance with appropriate national, provincial and local requirements, as well as international practices and standards.

The critical path of the project schedule is composed of activities related to:

- project economic assessment
- baseline studies and environmental application
- permitting and licensing
- detailed engineering
- construction
- commissioning

Additional activities such as pre-feasibility and feasibility studies, additional drilling programs, metallurgical testing, as well as major equipment fabrication can proceed in parallel to the critical path activities.

Figure 18.23 Snowfield High Level Execution



18.6 Markets and Contracts

The final products to be produced by the Snowfield project are a gold-silver doré, a copper concentrate, and a molybdenum concentrate. The gold-silver doré will likely be transported to a North American-based precious metals refinery. The copper concentrate will be sold to international smelting companies and metals traders most likely located in Asia, Europe, and North America depending on buyer terms and product quality. The molybdenum concentrate will be sold to international smelting companies and metals traders most likely located in Asia, Europe, and North America.

A more precise projection of marketing terms will be prepared during the prefeasibility study phase of this project.

18.7 Environmental

18.7.1 Introduction

The Snowfield property is situated within the Sulphurets District in the Iskut River region. The property is located in the Boundary Range of the Coast Mountain physiographic belt along the western margin of the Intermontane tectonic belt.

The climate is typical of north-western British Columbia with cool, wet summers, and relatively moderate but wet winters. The optimum field season is from late June to mid-October.

Tree line is at approximately 1200 masl. The Snowfield deposit is centred between the Mitchell Glacier to the north and the Hanging Glacier to the south.

The area is remote, and undeveloped. The widely varying terrain hosts a broad range of ecosystems. Its rivers are home to all five species of pacific salmon, as well as trout and dolly varden char. Black and grizzly bears frequent the forests and moose and migratory birds can be found in the wetlands. Mountain goats are common in the alpine areas.

18.7.2 Environmental Setting

The Snowfield project is located in a remote area for which little baseline environmental data are publically available. Silver Standard has engaged Rescan, a Vancouver-based consulting firm with extensive mining-related environmental assessment experience in BC, to undertake the baseline studies required for an environmental assessment of the project. Baseline studies for the Snowfield project will be initiated in summer 2010, following issuance of the Section 10 order from the British Columbia Environmental Assessment Office (BCEAO).

TERRAIN, SOILS, AND GEOLOGY

The Snowfield project is located in a rugged area with elevations ranging from about 420 m at the outlet to the planned tailings storage facility to over 2000 m at the top of the ridge above the deposit. Surrounding peaks are up to 2200 m in elevation. Glaciers and ice fields surround the mineral deposits to the north, south, and east. The Snowfield deposit is a near-surface, low grade, bulk tonnage, porphyry-style, gold deposit that has the additional potential of copper-gold + molybdenum mineralization at depth and west of the Snowfield fault. The gold mineralization at the Snowfield deposit is interpreted to be genetically related to one or more Jurassic-age alkaline intrusions. Gold mineralization is hosted by schistose, pervasively altered (quartz-sericite-chlorite) volcanic and volcaniclastics that contain 1% to 5% disseminated pyrite, minor disseminations and veinlets of tourmaline and molybdenite, and abundant younger calcite veinlets. Recent and rapid deglaciation has resulted in over-steepened and unstable slopes in many areas. Recently deglaciated areas typically have limited soil development, consisting of glacial till and colluvium. Lower elevation areas with mature vegetation may have a well developed organic soil layer. Avalanche chutes are common throughout the area and management of snow avalanches will be a concern for the development and operation of the project. Similarly, project design may have to consider the potential for debris flows in some areas.

ACID ROCK DRAINAGE

Baseline sampling has not yet been undertaken for the project but some prognostications can be based on general knowledge of the region. It is probable that there will be a reasonably strong chemical signature characteristic of acidic drainage resulting from the oxidation of naturally occurring sulphide minerals. The drainage would likely include elevated concentrations of sulphate, iron, and copper. Elsewhere in the region, seeps around natural gossans indicate natural acid conditions with pH in the 2.5 to 3.0 range. In water with near-neutral pH, evidence of precipitation, such as white aluminum oxyhydroxide and iron staining, are likely to be found from processes which have been occurring naturally over a geological time scale. Baseline ABA and metals analyses for various rock types will be undertaken to evaluate potential ARD concerns. Pending more detailed assessment, it is difficult to predict the ratio of net acid neutralizing to net acid generating rock. The net acid generating rock will also be evaluated for kinetic rate of reaction which will give an indication of the type of management strategy required.

CLIMATE, AIR QUALITY, AND NOISE

The climate of the region is relatively extreme and daily weather patterns in the Iskut region are unpredictable. Prolonged clear sunny days can prevail during the summers. Precipitation in the region is about 1,600 mm to 2,000 mm annually. The majority of precipitation is received in the fall and winter from September through to February. Annually, Stewart receives 70% of its yearly precipitation during this time.

October tends to have the highest or second highest precipitation levels for the year. Stewart regularly receives 30% of its precipitation as snow that falls from November to March. In October, when Stewart typically has its heaviest precipitation, 97% of it falls as rain. Late spring or early summer months typically receive the least amount of rainfall on an annual basis. Snow pack ranges from 1 m to 2 m but high winds can create snowdrifts up to 10 m deep. Silver Standard established a full meteorological station to collect site specific weather data near the Brucejack Lake camp in mid October 2009 (Figure 18.24). The station measures wind speed and direction, air temperature and pressure, rainfall, snowfall, relative humidity, solar radiation, net radiation, and snow depth.

Figure 18.24 Met Station Installation Near the Brucejack Lake Camp.

Assumed climate data for the Scott Creek TSF and the mine site are shown in Table 18.24 below. The climate station installed at Brucejack Lake has only been in operation since mid-October 2009 so that average climate data were sourced from the Meteorological Service of Canada (MSC) climate station, Unuk River Eskay Creek (#1078L3D). Data from this station are available for the period September 1989 to February 2007. The Unuk River station is located approximately 45 km northwest of Scott Creek at an elevation of 887 m. Temperature data summarized in Table 18.24 are based on scaling the Eskay Creek data (887 m) to the mine site (~ 1400 m) and TSF (~ 600 m) assuming an adiabatic lapse rate of -0.6°C per 100 m.

Table 18.24 Average Monthly Climate Data for Snowfield Project

Month	Mine Site		Scott Creek TSF		
	Average Temperature (°C)	Average Precipitation (mm)	Average Temperature (°C)	Average Precipitation (mm)	Average Evaporation/ Sublimation (mm)
January	-11.3	253	-6.5	190	5
February	-9.1	207	-6.1	155	5
March	-7.2	169	-4.1	126	7
April	-2.6	93	0.5	70	12
May	1.1	93	4.2	70	30
June	5.1	68	8.2	51	81
July	7.3	82	10.3	62	88
August	7.3	142	10.4	107	76
September	2.7	215	5.8	161	52
October	-2.4	243	0.7	182	7
November	-7.9	214	-4.9	160	6
December	-9.8	256	-6.7	192	5
Average/Total	**-2.2**	**2,033**	**1.0**	**1,525**	**374**

Note: From BGC

Precipitation at the mine site is currently assumed to be similar to that observed at Eskay Creek given their close proximity (19 km) and similar basin physiography. However, the Scott Creek TSF is approximately 30 km east-southeast of the mine site and located behind a range of glaciated mountains with peak elevations of up to 2300 m. This range is expected to have a rain shadow effect with reduced precipitation in its lee. Therefore, average annual precipitation at Scott Creek is expected to be about 75% (1,525 mm) of that recorded at Eskay Creek.

WATER RESOURCES

Flow Volumes

Most of the project area including the location of the planned TSF in the Scott Creek valley, drains to the Bowser River. The Bowser River enters Bowser Lake approximately 3 km downstream of its confluence with Scott Creek. The Bowser Lake outflow, in turn, joins the Bell-Irving River which eventually flows into the Nass River before reaching the Pacific Ocean. Parts of the proposed crusher and pit areas drain westward into the Unuk which enters Alaska within 30 km of the project area. Proximity to the coast, relatively high precipitation rates, mountainous terrain, and the presence of glaciers result in large run-off flows within the project area. Some hydrometric data is available for this region from the Water Survey of Canada, including flow data from the Bell-Irving River. However, most of the regional data are historical (the Bell-Irving River data collection sites were decommissioned in 1996) and from relatively large watersheds; therefore, the data may not represent current hydrological conditions of the sites of interest.

The proposed location for the TSF and associated dam structures impact on the drainages of the small tributaries to Scott Creek. Water would be diverted from these creeks to the Bowser River to minimize flows of contact water and thus the flows requiring treatment. However, after further analysis, it may prove advantageous to allow the creeks to enter the TSF thereby decreasing concentrations of regulated parameters within the TSF and minimizing or obviating the need for treatment. A Rescan hydrological station was installed on Scott Creek near the confluence with Bowser Lake in October 2009 and is shown in Figure 18.25.

Figure 18.25 Hydrological Station on Scott Creek

Water Quality

Little historical baseline water quality information is available for the Snowfield area. Silver Standard has plans to initiate a comprehensive assessment of water and sediment quality and related aquatic ecology. The very sparse water quality data collected to date at Brucejack Lake indicate that the concentrations of metals are naturally elevated by drainage from the mineralized zones. Naturally-occurring seeps in the nearby mineralized zones may have pH values in the range of 2.5 to 3.0 and exhibit elevated levels of sulphate, iron, and copper characteristic of metal leaching/ARD caused by the oxidation of naturally occurring sulphide minerals.

FISHERIES

The Bell-Irving River is a large river system provides important spawning routes for the five species of Pacific salmon and anadromous steelhead trout, as well as habitat for resident trout (cutthroat, rainbow), resident char (e.g. Dolly Varden and/or bull trout), and whitefish. The fisheries resources and fish habitat of the Bowser River and potentially affected tributaries of the Bell-Irving River will be assessed as part of a baseline program. Mitigation measures and any compensation that may be due as a result of fisheries impacts related to the project will be discussed and developed in consultation with the appropriate agencies and relevant Aboriginal groups.

ECOSYSTEMS AND VEGETATION

The Snowfield project is located in the humid environment of the Coast Mountain Range and comprised largely of Interior Cedar – Hemlock (ICH), Engelmann Spruce – Subalpine Fir (ESSF), and Alpine Tundra (AT) biogeoclimatic classifications. Silver Standard will undertake a systematic mapping of the project area using both Predictive Ecosystem Mapping (PEM) and Terrestrial Ecosystem Mapping (TEM) methods. The PEM method will be used over the whole of the project area; whereas, the more intensive TEM method will be restricted to areas of disturbance such as access roads, pits, plant site, and the TSF. The PEM product will show the distribution and classification of forested and non-forested ecosystems in the study area, using provincially mandated standards so that wildlife habitat ratings can be applied. The TEM product will provide similar information at a higher level of detail in the project footprint area. Concurrent with the PEM and TEM mapping, Silver Standard will map plant communities and plant species of conservation concern to aid environmental impact assessment.

WETLANDS

The project encompasses areas of wetland along the proposed access routes and in the proposed TSF location. Wetlands in Canada are valued ecosystem components under the Canadian Environmental Assessment Act (CEAA). They are conserved and managed through federal initiatives, such as the Federal Policy on Wetland Conservation. Baseline studies will include mapping of wetland ecosystems to allow for the identification of areas where project modification may limit negative impacts. Water quality, aquatic biology, fisheries, and hydrology data will also be collected from potentially affected wetland sites.

WILDFIRE

The region encompassing the proposed project is likely home to many terrestrial wildlife species including black and grizzly bears, mountain goats, moose, birds of prey, migratory songbirds, waterfowl, western toads and small mammals. Comprehensive baseline surveys will be initiated to characterize the wildlife populations and distribution and to understand their significance to the area. Habitat suitability mapping for several species will be conducted in parallel with the PEM and TEM work. Silver Standard will evaluate the potential impacts on species, especially listed species, which could occur in the area. A number of listed species are expected to occur in the proposed project area based on past work on other mining projects in the region; viz, wolverine, fisher, tailed frogs, western toad and rusty blackbird. Species of concern include those that may not be of conservation concern but are of regional importance for other reasons identified in the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP), e.g. moose, mountain goat, marmot/arctic ground squirrel, and grizzly bear, amongst others. Grizzly bears have been observed close to the project study area. These bears feed on salmon during the spawning season and on vegetation and small mammals during the rest of the year. Black bears are ubiquitous throughout the area. Moose are important in the region from both ecosystem and socioeconomic (i.e., hunting) perspectives. Low elevation and wetland areas are important moose habitat in the study area. Mountain goat usage of the project area is likely and will be documented. They are important from both ecosystem and socioeconomic (i.e., hunting) perspectives and are especially sensitive to development. Aerial surveys following government protocols will be used to assess mountain goat populations to aid in the development of appropriate mitigation techniques. Breeding birds and raptors will be documented in the project areas and will be given special attention due to statutory protection and conservation concerns.

TRADITIONAL KNOWLEDGE AND TRADITIONAL LAND USE

The Snowfield project site is located on Crown land in an area historically used by several First Nations groups. The project lies within the boundaries of the Nass Area, as defined in the Nisga'a Final Agreement. Traditional Knowledge/Traditional Use (TK/TU) studies will be undertaken and will involve the potentially affected First Nations and Treaty Nations. It is anticipated that these studies will identify areas and seasons where aboriginal groups have traditionally engaged in hunting, fishing, gathering, and spiritual activities. The outcomes of these studies will be used to inform the overall design and operation of the project.

NON-ABORIGINAL LAND USE

The western part of the Snowfield project area is included in the Cassiar Iskut-Stikine Land and Resource Management Area which was approved by the province in 2000. The LRMP is a sub-regional integrated resource plan that establishes the framework for land use and resource management objectives and strategies, and provides a basis for more detailed management planning. The LRMP outlines the management direction, research and inventory priorities, and economic strategies for the Cassiar Iskut-Stikine area, and presents an implementation and monitoring plan to reach the established objectives. Detailed planning initiatives and resulting products are expected to be guided by, and be consistent with, the LRMP management direction. Part of the project area lies within the boundaries of the South Nass Sustainable Resource Management Plan area, currently in the planning process. The Snowfield project area has been the focus of mineral exploration for many years. There are indications that prospectors explored the area for placer gold in the late 1800s and early 1900s. Placer gold production has been reported from Sulphurets Creek in the 1930s and a large log cabin near the confluence of Mitchell and Sulphurets creeks was reportedly used by placer miners until the late 1960s. The whole region surrounding the project is heavily staked and several other mining companies have active exploration programs nearby. The Kerr and Sulphurets deposits have been extensively explored on an intermittent basis since the 1960s. Intensive underground exploration adjacent to Brucejack Lake in the 1990s was supported by a temporary road from Bowser Lake and over Knipple Glacier. The nearby Bell II Lodge on Highway 37 has a successful heli-ski operation that covers a very broad area. Guide outfitter territories and trap-lines exist in the project area and commercial recreational and fishing guide territories also exist there. The relative remoteness of the site suggests that recreational hunting and fishing is fairly limited in the immediate project area. Commercial timber harvesting has occurred near Highway 37, about 10 km to the east of the project site. Further timber harvesting in the project area is possible subject to a viable market for the timber.

VISUAL AND AESTHETIC RESOURCES

The Snowfield project is located in a relatively remote and undisturbed area characterized by rugged mountains, glaciers, untouched forest, and wild rivers. The nearest road is Highway 37, about 10 km to the east of the proposed TSF. The TSF will not be visible from the highway. The controlled-access Eskay Mine road terminates about 20 km to the north of the proposed pit. The mine will be located in an isolated area that is not visible from the Eskay Mine road.

ARCHAEOLOGY AND HERITAGE RESOURCES

Archaeological assessments will determine the presence of artefacts or sites and conduct required mitigations prior to major project related disturbances.

WATER MANAGEMENT

Water management will be a critical component of the project design in this high precipitation environment. The most likely avenue for transport of contaminants into the natural environment will be through surface or ground water.

As such, Silver Standard will develop a comprehensive water management plan that applies to all mining activities undertaken during all phases of the Snowfield project. The main objectives of this water management plan will be to divert non-contact water from the TSF and regulate the movement of water in and around the mine site.

The goals of this management plan will be to:

- provide a basis for management of the freshwater on the site, especially with the changes to flow pathways and drainage areas
- protect ecologically sensitive sites and resources, and avoid harmful impacts on fish and wildlife habitat
- provide and retain water for mine operations
- define required environmental control structures
- manage water to ensure that any discharges meet and/or exceed the permitted water quality levels and guidelines.

Strategies for water management include diverting surface water from disturbed areas, protecting disturbed areas from water erosion, collecting surface water from disturbed areas and treating to meet discharge standards prior to release, minimizing the use of fresh water, recycling water wherever possible to minimize the amount of water released, and monitoring the composition of release water and treating it to remove or control contaminants as required to meet discharge standards. Diversion channels or tunnels will be constructed to direct run-off away from disturbed areas. Channels will likely be constructed to collect surface run-off above all pit high walls, waste rock dumps, the plant site, and the TSF, where permitted by terrain characteristics. These diversions will isolate surface water from exposed metal rich rock and tailing and allow the run-off to be released with little or no treatment.

WATER TREATMENT

Approximately 14.1 Mm^3 per year of water from the TSF will be treated to meet receiving water quality requirements and discharged around Dam #1 to the Bowser River. Discharge will occur only during the ice-free season or approximately six months per year. Decant, pumping and treatment will be designed for approximately 1,600 m^3/h to accommodate the yearly volume over the period of discharge. Adequate excess storage capacity will be provided in the TSF to accommodate extreme precipitation years allowing treatment capacity to be sized for the average year. It is anticipated that water impounded within the TSF near Dam #1 will meet all receiving water quality requirements with the possible exception of total suspended solids (TSS). Withdrawal of water from the 10 cm thick surface layer of the impoundment will likely be sufficient to maintain TSS below the mandated 15 mg/L. However, in the event that this low level of TSS cannot be met with decantation alone, clarification with floating clarifiers, employing flocculants, will be utilized.

A floating decant structure will be moored in the TSF near the upstream face of Dam #1. This structure will accommodate a weir box and pumps. The weir crest will be positioned approximately 10 cm below the water surface, requiring a weir length of approximately 16 m. Three vertical turbine pumps, of approximately 120 kW power each, will be mounted on the decant structure and withdraw water from the weir box.

Each pump will be capable of lifting 1800 m³/h, 30 m over the dam. A floating high density polyethylene pipeline (approximately 36" DR 17) will convey the decanted water from the pumps over and around the dam.

As the dam is raised, the floating structure will rise with it with adjustment to the mooring lines and the floating pipeline. Water lift requirements will decrease as the dam rises because of the increasing horizontal area of the flooded valley and the diminishing need for freeboard to accommodate extreme precipitation year flow.

Also moored in the TSF will be three floating clarifiers. These draw water radially inward over the circumferential weir toward a central tube packed core. Solids settle onto the conical shell as well as within the core and are transported downward while clear water is decanted at the top of the central core. Test work to size these clarifiers is essential, but experience at other sites suggests that each of the three may require a diameter of 55 m.

Clarifiers will only operate when required by elevated TSS concentrations.

Floating structures will be accessible via a floating walkway and power provided by a submersible power cable run along the walkway or the floating pipeline.

WATER SUPPLY

Process water will be obtained from the TSF.

Potable water will likely be sourced from water diversions constructed around the perimeter of the plant site and the waste rock dump. During the winter months, well water from a field of wells near the plant site may be needed to supply fresh water for process make up and domestic use at the plant and camp facility.

INTERNAL RECYCLES STRATEGIES

Process water will be recycled where feasible to reduce the volumes of water released to the environment. Tailing supernatant will be recovered from the TSF using barge mounted pumps and returned to the plant. Storm Water Management

Storm water will be managed throughout the construction and operation of the project to minimize erosion and transport of contaminants. Diversion structures and collection and treatment facilities will be designed to handle 1-in-200-year storm events, as projected using available historic hydrological and meteorological data. Greater capacity will be provided if required based on an assessment of the consequences of failure.

DISCHARGE STRATEGY AND QUALITY

Discharges from the TSF will be controlled where feasible to mimic natural flows in order to minimize adverse effects on local hydrological regimes, for example discharge during only six months of the year. Some modification of natural flows will be required from time to time to avoid disturbed areas and to optimize dilution in order to consistently meet discharge standards. Discharges from the TSF will be managed to meet the federal government MMERs and negotiated provincial water quality objectives.

CONSTRUCTION WATER MANAGEMENT

Silver Standard will place a high priority on early and effective application of water management systems during the construction period using lessons learned from similar projects in the region.

18.7.3 *WASTE MANAGEMENT*

TAILING MANAGEMENT

The TSF is designed to isolate the pyrite tailing in a stable subaqueous environment in perpetuity. To ensure that the TSF continuously meets its objectives, Silver Standard will develop and implement a tailing management plan. The goals of this management plan are to:

- Manage the TSF structures in a safe and environmentally responsible manner.
- Manage the discharge from the TSF to ensure that all effluent meets and/or exceeds the permitted water quality levels and guidelines.
- Provide a framework for continual improvement in the environmental safety and operational performance of the TSF structures.
- Define environmental and performance monitoring and reporting.

Tests will be undertaken to characterize the tailing and supernatant to allow estimation of the rate of oxidation and resulting water quality. This information will guide planning for tailing water management.

At closure, the TSF will be configured with minimal pond/wetland area, and revegetated with grasses and trees. Surface drainage within the impoundment will be directed towards a closure spillway. No discharge will be permitted until water quality meets discharge standards. The water will be treated prior to release if it does not initially meet discharge standards. Treatment will continue as long as necessary to ensure that all discharges to the receiving environment meet permit requirements.

WASTE ROCK AND OVERBURDEN MANAGEMENT

The Snowfield project will potentially generate 520 Mt of waste rock over the anticipated LOM. Waste rock will be segregated according to its potential to generate acid and leach metals. A comprehensive testing program using blast hole cuttings will be established to characterize all rock removed from the pits. This program will be integrated with the mineralized material control program to ensure that rock is correctly directed to the process plant, the NPAG dump, or the PAG storage area. Location of the PAG waste rock dump is currently under study. It will be located with a view to isolate the PAG waste rock from ground water and surface run-off.

HAZARDOUS WASTE MANAGEMENT

Hazardous waste materials, such as spoiled reagents and used batteries, will be generated throughout the life of the project, from construction to decommissioning. Silver Standard will incorporate a comprehensive management plan for hazardous wastes. These materials will be anticipated in advance, segregated, inventoried, and tracked in a manner consistent with federal and provincial legislation and regulations such as the Federal Transportation of Dangerous Goods Act. A separate secure storage area will be established with appropriate controls to manage spillages. Hazardous wastes will be labelled and stored in appropriate containers for shipment to approved off-site disposal facilities.

NON-HAZARDOUS WASTE MANAGEMENT

Silver Standard will initiate a comprehensive waste management program prior to start of construction of the project. The Program will minimize potential adverse effects to the environment, including wildlife and wildlife habitat and will ensure compliance with regulatory requirements, permit and licence obligations and Silver Standard Environmental Policy. Waste management will involve segregation of wastes into appropriate management channels. Project waste collection/disposal facilities will include one or more incinerators, a permitted landfill, waste collection areas for recyclable and hazardous waste, and sewage effluent/sludge disposal. Most facilities will be duplicated at the mine and plant sites. Waste collection areas will have provisions to segregate waste according to disposal methods and facilities to address spillage and fire.

18.7.4 AIR EMISSION AND DUST CONTROL

Air emissions can represent a significant component of contaminant dispersion for a mining project. Baseline studies, utilizing on-site meteorological stations and wind monitoring stations, will collect atmospheric data in the Snowfield project area to allow air dispersion modelling to be undertaken. Mitigation procedures will then be developed to minimize adverse impacts from emissions. Regular monitoring of emissions will assess the success of the mitigation methods and warn of any requirement to adjust the current approach.

Silver Standard will implement an air emissions plan to ensure that the levels of air emissions generated by project activities are below the regulatory requirements of the Canada and BC Ambient Air Quality Objectives.

Adverse effects from air emissions and fugitive dust will be minimized through the implementation of mitigation measures such as:

- the use of clean, high-efficiency technologies for diesel mining equipment
- the use of appropriate emissions control equipment such as scrubbers
- the use of low-sulphur diesel fuel when practical
- the use of a vehicle fleet powered by diesel engines with low emissions of nitrous oxide and hydrocarbons (greenhouse gases)
- preventative maintenance to ensure optimum performance of light-duty vehicles, diesel mining equipment, and incinerators
- the use of large haul trucks for mineralized material and waste transport to minimize the number of trips required between the source and destination
- the use of slurry pipelines for moving crushed and ground mineralized material and a pipeline for diesel fuel to reduce the number of haul truck trips and the consequent amount of diesel emissions and fugitive dust
- the implementation of a recycling program to reduce the amount of incinerated wastes and hence CO2 emissions
- the segregation of waste prior to incineration to minimize toxic air emissions.

Dust is generated at mining sites by many common activities including blasting, rock excavation, haulage and stockpiling, crushing and screening operations, mineralized material and waste conveying, and vehicle travel on gravel roads. Silver Standard will use a range of control and mitigation measures to reduce dust creation and dispersion. Some of these measures include the following:

- Blasting will be designed with appropriate delays and blast hole stemming to direct energy into rock breaking rather than dust creation.
- Loader and shovel operators will be instructed to minimize drop distances when moving rock to reduce dust creation.
- Crushing and screening operations will be enclosed and equipped with bag houses to collect dust.
- Conveyor transfer points will be enclosed and equipped with dust control systems such as water sprays or bag houses.
- Conveyors will incorporate wind covers where required.
- Haul roads and access roads will be treated for dust control. The selection of dust control methods will consider the need to avoid the use of products that may attract wildlife to roads.

18.7.5 *Design Guidance*

Project Development Philosophy

Every reasonable effort will be made to minimize long-term environmental impacts and to ensure that the project provides lasting benefits to local communities while generating substantial economic and social advantages for shareholders, employees, and the broader community.

Precautionary Principle

The 1992 Rio Declaration on Environment and Development defined the precautionary principle as: "Where there are threats of serious or irreversible damage, lack of full scientific certainty shall not be used as a reason for postponing cost-effective measures to prevent environmental degradation." Silver Standard will use appropriate and cost-effective actions to prevent serious or irreversible damage. The lack of full scientific certainty regarding the probability of such effects occurring will not be used as a reason for postponing such mitigation.

Integration of Traditional Knowledge

Silver Standard respects the traditional knowledge of the Aboriginal peoples who have historically occupied or used the project area. Silver Standard recognizes that it has significant opportunity to learn from people who may have generations of accumulated experience regarding the character of the plants and animals and the spiritual significance of the area. Traditional knowledge will guide aspects of the project, including any future changes once the mine is approved. Silver Standard anticipates changes as part of its commitment to continual improvements, based on ongoing monitoring and research. This approach will ensure the most beneficial environmental, social, and economic outcomes for the project. Silver Standard is committed to a process that invites and considers input from people with traditional knowledge of the project area towards the environmental assessment and design of the Snowfield project. Silver Standard is striving to establish a cooperative working relationship with all relevant Treaty and First Nations people to ensure opportunities to gather traditional knowledge.

Baseline Research

Silver Standard will initiate comprehensive baseline studies of the regional project area's atmosphere/climate, surface hydrology, aquatics, water and sediment, limnology, fish habitat and community, rock geochemistry, soils, vegetation, and wildlife to characterize the local and regional ecosystem prior to major disturbances. Archaeology, heritage, land use, cultural, Traditional Knowledge, and socioeconomic baseline studies will also be carried out to characterize the regional human environment. The methodologies for the baseline studies will be developed in consultation with regulatory agencies and Treaty and First Nations peoples of the area.

VALUED ECOSYSTEM COMPONENTS

Silver Standard recognizes that different components of the natural and socioeconomic environments will be of special importance to local communities and other stakeholders, based upon scientific concern or cultural values. These components are widely termed valued ecosystem components (VECs) and will be given particular consideration during project assessment, planning, and design. VECs applicable to the project will be identified through a comprehensive issues scoping exercise, which will include consultation with federal and provincial regulatory bodies, local Treaty and First Nations, and other stakeholders.

ENVIRONMENTAL ASSESSMENT STRATEGY AND SCOPE

The environmental assessment of the Snowfield project that is required under federal and provincial legislation will focus on the identified VECs to ensure the primary concerns of all stakeholders are addressed. The methodology to be applied has been developed to ensure a comprehensive, logical, and transparent assessment and involves examination of the potential effects of each mine component through all project stages. Silver Standard will use the environmental assessment process as an opportunity to refine project design to minimize long-term environmental impacts and to identify appropriate mitigation and management procedures.

ECOSYSTEM INTEGRITY

The project area ecosystem is relatively undisturbed by human activities, although it is not static. Glacier retreat and relatively recent (within the last 10,000 years) volcanoes, along with frequent landslides, debris flows, and snow avalanches, continue to modify the landscape. Silver Standard's objective is to retain the current ecosystem integrity as much as possible during the construction and operation of the project. This objective will be met first by avoiding adverse impacts where feasible, second by mitigating unavoidable adverse impacts, and third by compensating for adverse impacts that cannot be mitigated. Upon closure and reclamation of the project, the intent will be to return the disturbed areas to a level of productivity equal to or better than that which existed prior to project development and for the end configuration to be consistent with pre-existing ecosystems to the extent possible.

BIODIVERSITY AND PROTECTED SPECIES

Silver Standard is committed to making every reasonable effort toward maintaining biodiversity in the project area. Biodiversity is defined by the BC Ministry of Forests and Range as "the diversity of plants, animals and other living organisms in all their forms and levels of organization, and includes the diversity of genes, species and ecosystems, as well as the evolutionary and functional processes that link them".

Species diversity refers to the variety and abundance of different types of organisms within a region. Ecosystem diversity refers to the variety of ecosystems or habitats within a region. For the purpose of the environmental assessment of the Snowfield project, biodiversity will be considered at the species and ecosystem (habitat) levels.

The Canadian Species at Risk Act was created to protect wildlife species from becoming extinct in two ways: by providing for the recovery of species at risk due to human activity and by ensuring through sound management that species of special concern do not become endangered or threatened. It includes prohibitions against killing, harming, harassing, capturing or taking species at risk, and against destroying their critical habitats.

The objectives of the “Convention on Biological Diversity” (the Convention) signed by Canada in 1992 are to conserve biodiversity, to use biological resources in a sustainable manner, and to share benefits resulting from the use of genetic resources. The Convention recognizes environmental assessment as an important decision-making tool for the protection of biodiversity. Canada issued the Canadian Biodiversity Strategy in 1995 in response to the Convention. Although the Biodiversity Strategy does not explicitly recommend a strategic plan or program for the mining sector, it does address associated issues such as ecosystem rehabilitation, reduction or elimination of harmful substance release to the environment, improving methods for monitoring ecosystems, and identifying mechanisms to use traditional knowledge.

Biodiversity is not an isolated concept but a part of project planning (mitigations and monitoring), environmental effects analysis, and consideration of sustainability. Silver Standard is applying this concept in carrying out scoping, effects analysis, project design and mitigation, determination of effects significance, and monitoring. This concept will be integrated both implicitly and explicitly throughout the environmental assessment.

Environmental Standards

Silver Standard will design, construct, operate, and decommission the Snowfield project to meet all applicable BC and Canadian environmental and safety standards and practices. Some of the pertinent federal and provincial legislation that establish or enable these standards and practices are outlined below:

- Environment and Land Use Act (BC)
- Environmental Management Act (BC)
- Health Act (BC) Forest Act (BC)
- Forest and Range Practices Act (BC)
- Fisheries Act (BC)
- Land Act (BC)

- Mines Act (BC)
- Soil Conservation Act (BC)
- Water Act (BC)
- Wildlife Act (BC)
- Canadian Environmental Protection Act
- Canada Transportation Act Fisheries Act
- Transportation of Dangerous Goods Act
- Workplace Hazardous Materials Information System (WHIMIS) Safety Act.

A key commitment in meeting these standards will be the development and implementation of an Environmental Management System (EMS). The EMS will define the process by which compliance will consistently be met and demonstrated, and will include ongoing monitoring and reporting to relevant parties.

Design for Social and Community Requirements

Silver Standard will strive to establish collaborative and cooperative relationships with relevant Treaty and First Nations people, other communities and interested stakeholders. Silver Standard recognizes that its social licence to operate is dependent on being a good corporate citizen and neighbours to all groups with interests in the region.

Silver Standard is committed to a process that ensures communities benefit from employment, training, and contracting opportunities, that potential negative impacts are mitigated, and that any commitments and benefit agreements are respected. Silver Standard will meet its requirements through the development and implementation of a Social and Community Management System (SCMS). The SCMS will define the process by which the company will maintain its involvement and on-going commitments to communities and stakeholders.

18.7.6 *Socioeconomic Setting*

Northwestern BC is a sparsely populated area with a number of small, predominantly Aboriginal communities and the larger centres of Smithers, Terrace and Stewart, which provide services and supplies to much of the region. It is characterized by its inherent remoteness; communities within the region are generally dispersed and isolated from one another. Transportation and communication options are limited with the region intersected by Highways 37 (north to south) and 16 (east to west).

The region has suffered from declining population and weakening economic prospects, particularly among the Highway 37 communities. The regional population declined by 5.9% between 2001 and 2006, in contrast with a 5.3% population increase in the province over the same period.

The region has a strong dependence on primary resource industries principally mining and forestry. Mineral exploration activity has in recent years grown and the mining industry represents a significant source of employment. Due to the strong dependence on the resource sector, the economy is typified by "boom and bust" patterns. Mining is anticipated to continue to form the basis of the regional economy.

Community and socioeconomic impacts of a project such as Snowfield can potentially be very favourable for the region as new, long term opportunities are created for local and regional workers. Such opportunities would reduce and possibly reverse the out-migration to larger centres. Silver Standard is working with and intends to continue to work with Treaty Nation and First Nations groups, and members of local communities to maximize benefits through employment and business opportunities, training and skills development programs.

The following northwestern BC socioeconomic setting is compiled from the Northwest BC Mining Projects Socio Economic Impact Assessment, prepared in 2005 for the Ministry of Small Business and Economic Development, updated using data from the 2006 Census of Canada. Silver Standard is conducting further socioeconomic baseline studies to provide current information for the environmental assessment required by the province.

Highway 16 Corridor

Highway 16 extends from the Prince Rupert port eastwards to Terrace, Hazelton, Smithers, and Prince George. The Canadian National Railway (CNR) also follows this corridor. Most of the communities along this corridor are discussed in this section. The Highway 16 corridor is recovering from the economic downturn of the 1990s and has excess capacity with respect to social service infrastructure. The respective communities are incorporated providing a framework and capacity to:

- plan for, finance, and deliver services that might be required
- meet incremental growth from new mine developments.

Highway 37 Corridor

Highway 37 connects with Highway 16 at Kitwanga and runs northwards to the Yukon border. At Meziadin Highway 37A branches off Highway 37 and connects to the Port of Stewart. Highway 37 communities include Iskut, Dease Lake, and Good Hope Lake.

With the exception of Stewart, the majority of the population belongs to First Nations (e.g. Good Hope Lake). These communities rely heavily on public sector and mining employment. Since 1996, Highway 37 communities have experienced an overall decline in population. Stewart is located 60 km west of Meziadin junction on the west coast of BC, at the head of the 145 km long Portland Canal and the terminus of Highway 37A. The Stewart Bulk Terminals are used by the mining and forestry industries to ship products from northern BC and the Yukon to international destinations. Much of the town of Stewart was built for the development of the Granduc mine. The town's population has fallen dramatically in the past 20 years, coinciding with the closure of the Granduc and Premier mines.

NORTHWEST TRANSMISSION LINE

In 2007, the province of BC announced that a new 287 kV transmission line would be constructed from near Terrace to Bob Quinn Lake following the Highway 37 corridor. This line would replace the existing 128 kV transmission line between Terrace and Meziadin and extend the electricity grid northwards into a previously unserviced area. The transmission line will provide high voltage electricity to within 10 km to 15 km of the Snowfield project site.

The environmental assessment for the proposed extension of the provincial electricity grid to Bob Quinn Lake is ongoing, with the BC government acting as the proponent.

18.7.7 *CONSULTATION ACTIVITIES*

Community engagement and consultation is fundamental to the success of the proposed Snowfield project and will take place during the project's planning and regulatory review, construction, and operations phases. Prior to beginning the BCEAA process, Silver Standard will initiate project and company introductions with the potentially affected Treaty and First Nation groups. Subsequent consultation activities in the form of information sharing will occur during the planning and regulatory review, construction, and operations phases. These consultations will include BCEAO technical working group meetings (with government agency, Treaty, and First Nations participation), leadership meetings, community meetings, project information distribution, focus groups and workshops, communication tracking, and issue identification and resolution.

CONSULTATION POLICY REQUIREMENTS

The BCEAA and the CEAA contain provisions for consultation with Treaty Nations, First Nations, and the public as a component of the environmental assessment process. Public consultation measures will comply with the *Public Consultation Policy Regulation*, *BC Reg. 373/2002*.

CONSULTATION GROUPS

Treaty and First Nations

Silver Standard may be delegated the responsibility of information sharing with potentially affected Treaty and First Nations. If this comes to pass, the process will be initiated with the potentially affected Treaty and First Nations, as identified by the provincial Crown, and will continue.

Government

Silver Standard will engage and collaborate with the federal, provincial, Treaty Nations, Regional and Municipal government agencies as required with respect to topics such as: land and resource management protected areas official community plans (OCPs) environmental and social baseline studies and effects assessment mitigation, management, monitoring, and reclamation plans.

Public and Stakeholders

Silver Standard will consult with the public and relevant stakeholder groups [1], including: land tenure holders trappers guide outfitters recreation and tourism businesses economic development organizations businesses and contractors (e.g. suppliers and service providers) special interest groups (e.g. environmental, labour, social, health, and recreation).

CONSULTATION ACTIVITIES AND APPROACH

Silver Standard will initiate a consultation program relevant and useful to each consultation group. The proposed Snowfield project consultation program will include: government agency, Treaty Nation, and First Nations participation in the BCEAO technical working group meetings, leadership meetings, community meetings, information distribution, focus groups and workshops.

Consultation activities will reflect the BCEAO and CEAA consultation requirements, as well as Silver Standard's goals for meaningful and sustainable relationships with the leaders and community members affected by and involved in the Snowfield project.

18.7.8 LICENSING AND PERMITTING

Mining projects in BC are subject to regulation under federal and provincial legislation to protect workers and the environment. This section discusses the principal licences and permits required for the Snowfield project.

The schedule is based on the provincial and federal approval process as it stands today. The schedule as outlined suggests complete approval with necessary permits, licences and authorizations to start construction late in the fourth quarter of 2014.

[1] The public, in this context, pertains to the communities of Smithers, Terrace, Stewart, and Dease Lake. Stakeholders are individuals or groups of people with potential interests or issues with the Snowfield project.

Some key milestones for Silver Standard are:

- Preliminary Economic Evaluation: May 2010
- Project Description to BC EAO: June 2010
- Prefeasibility completed: June 2012
- Submission of EA: October 2012
- Complete Feasibility Study: August 2013

This schedule is based on starting baseline studies in the spring of 2010 and completing them in March 2012, culminating with the submission of the EA in October 2012. A number of steps in the environmental approval process have to be completed in a timely manner to accomplish this schedule.

British Columbia Environmental Assessment Act Process

The British Columbia Environmental Assessment Act (BCEAA) requires that certain large-scale project proposals undergo an environmental assessment and obtain an Environmental Assessment Certificate before they can proceed. Proposed mining developments that exceed a threshold criterion of 75,000 t/a, as specified in the Reviewable Project Regulations, are required under the Act to obtain an Environmental Assessment Certificate from the Ministers of Environmental and Energy, Mines and Petroleum Resources before the issuance of any permits to construct or operate. The Snowfield project will thus require an Environmental Assessment Certificate because its proposed production rate exceeds the specified threshold.

Regulatory Review and Approval Schedule Process

The Canadian Environmental Assessment Agency (CEAA) has advised Silver Standard that the Snowfield project will require an environmental assessment under the Canadian Environmental Assessment Act.

Authorizations Required

Lists of the major federal and provincial licences, permits, and approvals required to construct, operate, decommission, and close the Snowfield project are summarized in the following sections. The lists cannot be considered comprehensive due to the complexity of government regulatory processes, which evolve over time and the large number of minor permits, licences, approvals, consents and authorizations, and potential amendments that will be required throughout the life of the mine.

British Columbia Authorizations, Licences and Permits

Provincial permitting, licensing and approval processes (statutory permit processes) may proceed concurrently with the BCEAA review or may, at the proponent's option, follow the Environmental Assessment Certificate. At this time, it is too early to ascertain whether Silver Standard will seek concurrent approvals under the BCEAA process. However, no statutory permit approvals may be issued before an Environmental Assessment Certificate is obtained. Statutory permit approval processes are normally more specific than the environmental assessment level of review, and for example, will require detailed and possibly final engineering design information for certain permits such as the TSF structures and others. Table 18.25 presents a list of provincial authorizations, licences, and permits required to develop the Snowfield project. The list includes only the major permits and is not intended to be comprehensive.

Federal Approvals and Authorizations

Federal approvals include an authorization from the Federal Minister of Environment approving the combined Application/Comprehensive Study Report for the Snowfield project. Major stream crossing authorizations will be required from Fisheries and Oceans under the Fisheries Act. Approvals for water crossings will also be required under the Navigable Waters Protection Act. An explosive factory licence will be required under the Explosives Act. The MMER under the Fisheries Act, administered by Environment Canada, may require a Schedule 2 authorization because the area proposed for the TSF contains fish habitat. Other activities under federal jurisdiction, such as radio communication and aviation, will require licensing.

Table 18.26 lists some of the federal approvals required.

Table 18.25 List of British Columbia Authorizations, Licences, and Permits Required to Develop the Snowfield Project

BC Government Permits & Licences	Enabling Legislation
Environmental Assessment Certificate	BCEAA
Permit Approving Work System & Reclamation Program (Minesite – Initial Development)	Mines Act
Amendment to Permit Approving Work System & Reclamation Program (Pre-production)	Mines Act
Reclamation Program (Bonding)	Mines Act
Amendment to Permit Approving Work System & Reclamation Program (Mine Plan-Production)	Mines Act
Approvals to Construct & Operate TSF Dam	Mines Act
Permit Approving Work System & Reclamation Program (Gravel Pit/Wash Plant/Rock Borrow Pit)	Mines Act
Water Licence – Notice of Intention (Application)	Water Act
Water Licence – Storage & Diversion	Water Act
Water Licence – Use	Water Act
Licence to Cut – Mine Site/TSF	Forest Act
Licence to Cut – Gravel Pits and Borrow Areas	Forest Act
Licence to Cut – Access Road	Forest Act
Licence to Cut – Transmission Line	Forest Act
Special Use Permit – Plant Access Road, Extension of Eskay Road	Forest Act
Road Use Permit – Eskay Road	Forest Act
Licence of Occupation – Borrow/Gravel Pits	Land Act
Licence of Occupation/Statutory Right of Way - Transmission Line	Land Act
Pipeline Permit – Diesel Pipeline	Pipeline Act
Surface Lease – Mine Site Facilities	Land Act
Waste Management Permit – Effluent (Tailing & Sewage)	Environmental Management Act
Waste Management Permit – Air (Crushers, concentrator)	Environmental Management Act
Waste Management Permit – Refuse	Environmental Management Act
Camp Operation Permits (Drinking Water, Sewage, Disposal, Sanitation and Food Handling) Management Act	Health Act/Environmental
Special Waste Generator Permit (Waste Oil)	Environmental Management Act (Special Waste Regulations)

Table 18.26 List of Federal Approvals and Licences Required to Develop the Snowfield Project

Federal Government Approvals & Licences	Enabling Legislation
CEAA Approval	Canadian Environmental Assessment Act
MMER	Fisheries Act/Environment Canada
Fish Habitat Compensation Agreement	Fisheries Act
Section 35(2) Authorization	Fisheries Act
Navigable Water: Stream Crossings Authorization	Navigable Waters Protection Act
Explosives Factory Licence	Explosives Act
Ammonium Nitrate Storage Facilities	Canada Transportation Act
Radio Licences	Radio Communication Act
Radioisotope Licence (Nuclear Density Gauges/X-ray analyzer)	Atomic Energy Control Act
Dam Licence	International River Improvements Act

18.8 TAXES

18.8.1 CORPORATION TAXES - FEDERAL

A rate of 15% will be assessed on taxable income.

Accelerated provisions apply in determining taxable income. These include deductions for:

- exploration and pre-production development expenditures at 100%
- Class 41 (b) – ongoing capital expenditures at 25% declining balance
- Class 41 (a.1) – accumulating ongoing capital expenditures at 100%
- Class 41 (a) – initial capital expenditures at 100% and claimed up to income from mine operating profit
- CEE – initial mine pre-strip capital expenditures at 100% and claimed up to income from mine operating profit
- loss carry forward provision – 20 years
- provincial resource taxes (see below).

18.8.2 CORPORATION TAXES - PROVINCIAL

The provincial corporate taxable income base is the same as the federal tax base. Similar write-off deductions apply (excluding resource taxes). A tax rate of 10% applies.

18.8.3 Mining Taxes - Provincial

Two taxes apply:

- Provincial net current proceeds – at 2% on net revenue less operating cost
- Net Provincial revenue tax – at 13%.

For financial modelling, these taxes have been applied strictly on a project basis, 100% equity funding, without debt financing charges.

18.9 Capital Cost Estimate

The initial capital cost for the Snowfield project was estimated at US$3.4 B with an expected accuracy range of ±35%.

The estimate was developed by Wardrop with inputs from the following consultants:

- BGC – Material take-offs for tailings management facilities and water management
- Rescan – water turbidity control and environmental costs
- Silver Standard – owner's costs.

The capital cost estimate consists of four main parts:

- direct costs
- indirect costs
- contingency
- owner's costs.

The capital cost summary and its distribution by area is shown in Table 18.27.

Table 18.27 Capital Cost Summary

Description	Labour Cost (US$)	Material Cost (US$)	Construction Equipment Cost (US$)	Process Equipment Cost (US$)	Total Cost (US$)	%
Direct Works						
Mine Area	164,338,747	111,170,648	122,423,492	330,655,248	728,588,135	21.6
Mill Area	126,634,988	137,177,571	12,231,162	314,996,969	591,040,690	17.5
Tailing Management, Reclaim Systems, Water Turbidity Control & Closure	76,255,62	198,653,80	138,714,05	19,843,250	433,466,736	12.8
Utilities	38,108,788	26,168,245	29,217,427	25,291,633	118,786,094	3.5
Site General	58,980,321	51,121,730	58,384,235	4,943,767	173,430,053	5.1
Temporary Facilities	6,134,985	86,857,202	0	0	92,992,187	2.8
Plant Mobile Equipment	146,106	0	0	7,325,261	7,471,367	0.2
Subtotal	**470,599,558**	**611,149,202**	**360,970,373**	**703,056,129**	**2,145,775,262**	**63.5**
Indirects						
Project Indirects	7,471,872	648,815,255	0	4,140,000	660,427,128	19.6
Contingencies	0	474,060,369	0	0	474,060,369	14.0
Owner's Costs	0	97,722,402	0	0	97,722,402	2.9
Subtotal	**7,471,872**	**1,220,598,026**	**0**	**4,140,000**	**1,232,209,898**	**36.5**
Total Capital Cost	**478,071,431**	**1,831,747,228**	**360,970,373**	**707,196,129**	**3,377,985,160**	**100.0**

18.9.1 ESTIMATE BASE DATE AND VALIDITY PERIOD EXCHANGE RATE

Wardrop has prepared this preliminary assessment estimate with a base date of Q2-2010. No escalation beyond Q2 2010 was applied to the estimate.

The budget quotes used in this estimate were obtained in the Q1-2010 and have a validity period of 90 days.

The estimate was prepared in C$ and then converted into US$ using a currency exchange rate of C$1.00 to US$0.92.

18.9.2 ESTIMATE APPROACH

The capital cost estimate was structured as per the project work breakdown structure (WBS) consisting of the following main areas as shown in Table 18.28.

Table 18.28 Project WBS

Area	Description
1000	Mine Area
2000	Mill Area
3000	Tailing management, Water Reclaim Systems, water turbidity Control
4000	Utilities
5000	Site General
6000	Temporary Facilities
7000	Plant Mobile Equipment
9000	Indirects
9800	Owner's Costs
9600	Contingencies

The capital cost estimate was developed based on the following:

- budget quotations were obtained for the supply of tailing pipelines, barge, pumps, covered tunnel, starter tailings dam material and haulage, ball mills, and mill drives. An in-house database was used for the balance of the equipment
- tunnelling costs were developed from quotations received from tunnelling contractors
- preliminary material quantity estimates were provided by in-house disciplines for mining, earthworks, concrete, steel, architectural, and tailings pipelines. BGC provided the material quantities for the construction of the tailings facilities. Rescan provided the details for the water turbidity plant
- power supply and distribution costs were developed based on information for electrical components from recent similar projects completed by Wardrop

- instrumentation, piping, and heating, ventilating, and air conditioning (HVAC) were expressed as a percentage for process equipment cost based on similar recent projects and in-house experience
- the estimated installation hours were based on in-house experience and cost book references.

All equipment and material costs were based on free carrier (FCA) manufacturer plant (INCOTERMS 2000) and are exclusive of spare parts, taxes, duties, freight, and packaging.

The freight costs and spares costs were covered in the indirect section of the estimate as an allowance, based on a percentage of the value of materials and equipment. The mining equipment costs are inclusive of freight.

Wardrop has assumed the construction man-hours/workweek to be 10 h/d with 3 wk on and 1 wk off rotation, with a land accessible construction camp.

18.9.3 ELEMENTS OF COSTS

DIRECT COSTS

LABOUR RATES, PRODUCTIVITY, AND TRAVEL ALLOWANCES

A blended labour rate of US$70.43/h was used throughout the estimate. This Labour rate was based on guidelines and requirements of the Construction Labour Relation Agreement BC 2010.

The labour rates include:

- vacation and statutory holiday pay
- fringe benefits and payroll burdens
- overtime and shift premiums
- small tools
- consumables
- Personal protection equipment
- contractor's overhead and profit.

Wardrop has assumed that the labour source is available as follows:

- 10% locally
- 25% regionally
- 65% out of town.

The source and availability of labour should be verified in the next phase of the study.

Travel allowances of US$60 M are included in the construction indirect section.

A productivity factor of 1.15 was applied to the labour portion of the estimate to allow for the inefficiency of long work hours, climatic conditions, and due to the 3 wk in 1 wk out rotation. This was based on in-house data supplied by contractors on previous similar projects in northern BC projects.

COST BASIS BY DISCIPLINE

Bulk Earthworks Including Site Preparation, Access and Haul Roads

Bulk earthwork quantities were generated from preliminary grading designs. Excavation of top soil and allowance for rock excavation were based on assumptions made at the time of the estimate preparation. Structural fill pricing was based on aggregates being produced at site utilizing a portable crushing and screening plant. The mobilization and set-up costs of the aggregate plant are included in the unit rates. The actual cost of aggregate production is included in the earthwork unit rates. Earthwork quantities do not include any allowance for bulking or compaction of materials; these allowances were included in the unit prices.

In the absence of geotechnical information, Wardrop has made the following assumptions:

- Topsoil, 300 mm average, was stripped and stockpiled on site.
- 5% of excavated material is deemed to be unsuitable.
- Depending on location, an average of 50% of the excavated material is deemed to be excavation in rock, of which 50% is rippable rock and the balance requires drilling and blasting. Surplus excavated material is stockpiled at a location within 5 km site.
- Allowable ground bearing pressure is assumed to be minimum 400 kPa at the plant site location. Equipment foundations may require greater ground bearing capacity (to be confirmed by selected vendors and a geotechnical engineer in the next phase of the project).
- The mine site primary crushers are assumed to be located adjacent to the pit on rock.
- Rock slope is assumed to be cut at a 1:1 slope.
- Allowable ground bearing pressure for structures located at the mine site is assumed to be a minimum of 600 kPa.
- The borrow pit for the construction of the tailings dam is assumed to be 10 km away.

- The possible need for soil remediation, or special sub-surface measures, or the need for piling is excluded.

Access roads are based on 8 m wide with 200 mm thick surfacing material (minus 50 mm) and 300 mm thick base (-150 mm).

Haul roads were based on a 30 m wide road, complete with 200 mm thick surfacing material (-50 mm), 300 mm thick base (-300 mm), and 500 mm thick sub-base.

Safety berms have been included as required and built from excavated materials.

Concrete

Concrete quantities are developed from preliminary engineering designs and sketches with no allowance included for overpour and wastage.

Typically all concrete is based on a 28 d compressive strength of 30 MPa. Wardrop used a concrete price of US$320/m³, to supply and deliver to the point of placement at site. The average installed concrete unit rates for 30 MPa concrete used in the estimate are US$660/m³. Concrete unit rates include for formwork, reinforcing steel, placement, and finishing of concrete.

Structural Steel

Structural steel quantities are based on quantities developed from preliminary engineering design and sketches with no allowance made for growth and wastage. Allowances are included for cut-offs, bolts, and connections.

An average supply unit rate of US$2,950/t for fabricated steel based on quotations from recent similar projects, was used in this estimate.

Craneage was included for all tonnages at a rate of US$230/t.

Platework and Liners

Preliminary quantities for platework and metal liners for tanks, launders, pump-boxes, and chutes were estimated using recent similar projects and in-house data.

Mechanical

The equipment estimate was prepared based on the project process flow diagrams and equipment list. The mechanical pricing is based on budgetary quotes obtained for the following major equipment:

- Ball mills
- Tailings pumps

- Tailings pipes
- Tailing pump barges
- The HPGR grinding equipment costs was estimated from recent similar projects.

All other mechanical equipment were based on information from recent quotes on similar projects.

HVAC and Fire Protection

HVAC and Fire protection is included as a percentage of the process equipment cost and is based on experience with similar recent projects.

Dust Collection

The dust collection equipment is included as a percentage of the process equipment based upon the process flow sheets and similar recent projects. Major dust collection equipment is covered in the mechanical section.

Piping and Valves

Piping and Valve costs were estimated as a percentage of process equipment, based on experience with recent similar projects.

Electrical

The mechanical equipment list was used to Estimate loading and site power requirements. Some non-mechanical loads were added to the equipment list to identify all known electrical loads for the study.

The power related equipment cost was estimated based on in-house data and experience with recent similar projects.

The equipment list in conjunction with the site plan was used to determine electrical building locations by centralizing electrical infrastructure to minimize cable runs.

Instrumentation

Instrumentation was estimated as a percentage of the equipment list allowance assigned to each area and based on experience with recent similar projects. The percentage varies between the different areas.

Plant control system costs are based on the installation of a Distributed Control System (DCS). Cost of the DCS was based on pricing received for a similar recent project.

Buildings

The estimate for the engineered steel framed, pre-engineered, and modular buildings is based on complete buildings with roofing, cladding, door, and architectural finishes. An in-house data base and experience with similar recent projects was uses as a base for the cost estimate. The major structures and buildings were identified from general arrangement drawings.

These structures and buildings include:

- primary crushing
- secondary crushing
- tertiary crushing
- mill building
- maintenance building
- truck shop
- administration and mine dry
- assay and met lab
- warehouse
- 500 person permanent camp
- emergency response building, including medical clinic
- gatehouse
- construction camps.

The modular construction camp will be expanded to accommodate the increase in labour force during construction. The approximate maximum size of the construction camp will be a 1,500 person camp.

18.9.4 PERMANENT ACCOMMODATION AND CONSTRUCTION CAMPS

There is a permanent and construction camp included in the estimate. Modular camp included in the estimate will be expanded to accommodate increasing labour force during construction. On completion it will be refurbished for owners use.

18.9.5 TAXES AND DUTIES

The estimate was prepared with taxes and duties on materials (HST, PST, and GST) excluded.

18.9.6 FREIGHT AND LOGISTICS

A freight allowance of 8% was provided for materials and the process equipment, except mining mobile equipment.

18.9.7 OWNER'S COSTS AND PERMIT ALLOWANCES

The Owner has provided an allowance of US$86 M for owner's costs and US$11 M for permits and licence fees.

18.9.8 EXCLUSIONS

The following are not included in the capital cost estimate:

- force majeure
- schedule delays such as those caused by:
  - major scope changes
  - unidentified ground conditions
  - labour disputes
  - abnormally adverse weather conditions
- receipt of information beyond the control of the EPCM contractors
- cost of financing (including interests incurred during construction)
- royalties
- schedule acceleration costs
- working capital
- cost of this study
- sunk costs.

18.9.9 ASSUMPTIONS

The following assumptions have been made in the preparation of this estimate:

- All material and installation subcontracts will be competitively tendered on an open shop, lump sum basis.
- Site work is continuous and is not constrained by the owner or others.
- Skilled tradespersons, supervisors, and contractors are readily available.
- The geotechnical nature of the site is assumed to be sound, uniform, and able to support the intended structures and activities. Adverse or unusual geotechnical conditions requiring piles or soil densification have not been allowed for in this estimate.

18.9.10 Contingency

A contingency allowance was developed using Monte Carlo simulations. It is considered that this contingency will adequately cover minor changes to the current scope to be expected during the next phase of the project.

The weighted contingency for the project is calculated to be 16.3% of direct costs.

18.10 Operating Cost Estimate

18.10.1 Summary

Total operating cost for the project is estimated at C$9.26/t milled. The estimate includes mining, process operating, G&A cost, and surface service costs (tailings dam construction and water treatment costs are included in sustaining capital costs). A total of 545 personnel are projected for the operation, including 248 for mining operation, 228 personnel for process, and 69 personnel for surface service and general management.

The unit costs are based on an average annual ore production of 43,800,000 t/a, or 120,000 t/d throughput and 365 d of operation per year. The currency exchange rate used for the estimate is 1:0.92 (C$:US$). The operation cost for the Snowfield project has been estimated in C$ within an accuracy range of ±35%. The breakdown of the estimated preliminary operating cost in C$ and US$ is presented in Table 18.29.

Table 18.29 Overall Operating Cost*

Area	Staffing	Unit Operating Cost (C$/t Milled)	Unit Operating Cost (US$/t Milled)
Mining	248	2.48	2.28
Processing	228	5.90	5.43
General & Administrative	48	0.67	0.62
Plant Services	21	0.21	0.19
Total	**545**	**9.26**	**8.52**

* Tailings dam construction costs are included in sustaining capital costs.

18.10.2 Mining Operating Cost

All mining operating costs are shown in C$, unless otherwise specified. Mine operating costs are derived from a combination of sources. The operating cost includes the labour, maintenance, major component repairs, fuel, and consumables costs.

Estimated costs for consumables such as tires, explosives, and drill accessories were obtained from previous Wardrop database and published reports on similar projects in northern BC. Maintenance, major component, and wear item repairs were estimated using Western Mining Database and from Wardrop's recent project data. The equipment fleet hourly operating costs were used to calculate the total equipment costs for each year.

Blasting costs are based on studies from similar projects and historical blasting costs. In this cost estimate, a total blasting scenario is assigned to the blasting contractor with the exception of the Wardrop blaster.

Staff and hourly operating rates are based on current salary and wage levels in similar mines operating in BC. A total benefit package was applied to the base rate consisting of vacation, statutory holidays, medical and health insurance, employment insurance, long term disability insurance, Canada Pension Plan, and Worker's Compensation Board.

From the basic operating capacities of the equipment, the travel speed of the trucks and cycle times for the various haul profiles were estimated using Caterpillar's FPC Version 3.05 program. The haul roads were laid out in MineSight® and the haul profiles entered into the FPC program. The program determines the haul truck speed based on the rimpull curves of a 363 t truck but is constrained by maximum operating speed criteria. The equipment productivity and operating hours are calculated separately in a spreadsheet scheduling program based on the total equipment cycle times derived from FPC. The calculated operating hours were multiplied by the hourly labour, maintenance, major component repairs, fuel, and consumable costs to arrive at the total operating costs.

Labour requirements were determined for each labour category. In the case of operators, labour requirements were estimated based on the amount of equipment on duty. Maintenance labour was estimated based on historical ratio between equipment operators and maintenance mechanics and electricians. All other labour and staff are estimated from experience with existing mines and anticipated operating conditions for the project.

The average hourly labour and salaried personnel for the first five years are summarized in Table 18.30.

Table 18.30 Average Labour Summary for Years 1 to 5

Labour Summary	No.
Hourly Personnel	
Mine Operations	
Shovel Operator	10
Loader Operator	4
Haul Truck Operator	51
Drill Driver	12
Dozer Operator	24
Grader Operator	13
Water/Sand Truck Operator	-
Dispatch Operator	4
Equipment Trainee	4
Mine Labourer	4
Mine Maintenance	
Mechanic - Heavy Duty	26
Mechanic – Light Dity	13
Electrician	10
Serviceman	13
Welder	13
Tireman	8
Maintenance Labour/Trainee	5
Total Hourly	**214**
Salaried Personnel	
Mine Operations	
Mine Superintendent	1
General Mine Foreman	1
Drill & Blast Foreman	4
Mine Foreman	4
Training Coordinator	2
Blaster	2
Dispatch Engineer	1
Mine Clerk	1
Maintenance Superintendent	1
Maintenance General Foreman	1
Maintenance Foreman	4
Mechanical Foreman	1
Electrical Foreman	2
Maintenance Planner	2
Maintenance Clerk	1
Technical Services	
Chief Mine Engineer	1
Senior Mine Engineer	2
Drilling / Blasting Engineer	1
Senior Surveyor	2
Survey Technician	4
Senior Geologist	1
Senior Geotechnical Engineer	1
Geologist – Grade Control	3
Technician- Grade Control	4
Total Salaried	**47**
Total Overall Personnel	**261**

Mine hourly and salary staff operating rates shown in Table 18.31 and Table 18.32 are based on current salary and wage levels at similar mines operating in BC.

Table 18.31 Mine Hourly and Staff Rates

Position	Base Rate (C$/h)	Payroll Burden (C$/h)	Total Pay (C$/h)
Mine Operations			
Shovel Operator	31.0	14.1	45.1
Loader Operator	30.0	13.7	43.7
Haul Truck Operator	27.9	12.9	40.8
Drill Operator	29.6	13.5	43.1
Dozer Operator	28.8	13.2	42.0
Grader Operator	28.8	13.2	42.0
Water/Sand Truck Operator	27.9	12.9	40.8
Blaster	30.7	14.0	44.7
Blaster Helper	26.4	12.3	38.7
Equipment Trainee	27.0	12.6	39.6
Mine Labourer	22.4	10.8	33.2
Mine Maintenance			
Mechanic – Heavy Duty	31.9	14.6	46.5
Machinist – Light Duty	29.3	13.6	42.9
Electrician	31.9	14.6	46.5
Serviceman	29.7	13.7	43.4
Welder	31.9	14.6	46.5

Table 18.32 Mine Salary Staff Rates

Position	Base Salary (C$)	Payroll Burden (C$)	Salary with Burden (C$)
Mine Operations			
Mine Superintendent	140,000	51,000	191,000
General Mine Foreman	94,000	35,000	129,000
Drill & Blast Foreman	81,000	31,000	112,000
Mine Foreman	62,000	25,000	87,000
Training Coordinator	81,000	31,000	112,000
Mine Clerk	44,000	20,000	64,000
Maintenance Superintendent	128,000	47,000	175,000
Maintenance General Foreman	98,000	37,000	135,000
Maintenance Foreman	83,000	32,000	115,000
Mechanical Foreman	83,000	32,000	115,000
Electrical Foreman	83,000	32,000	115,000
Maintenance Planner	81,000	32,000	113,000
Maintenance Clerk	44,000	20,000	64,000
Technical Services			
Chief Mine Engineer	118,000	43,000	161,000
Senior Mine Engineer	93,000	36,000	129,000
Drilling/Blasting Engineer	76,000	30,000	106,000
Senior Surveyor	64,000	26,000	90,000
Survey Technician	60,000	25,000	85,000
Senior Geologist	95,000	36,000	131,000
Senior Geotechnical Engineer	93,000	36,000	129,000
Geologist – Grade Control	81,000	32,000	113,000

LOM unit operating costs are listed in Table 18.33.

Table 18.33 Average Mining Costs per Tonne

	LOM Cost (C$/t Milled)	LOM Cost (C$/t Mined)
Drilling	0.07	0.05
Blasting	0.28	0.19
Loading	0.17	0.11
Hauling	0.97	0.63
Support Equipment	0.23	0.15
Dewatering	0.01	0.01
Labour	0.74	0.48
Total Mining Cost	**2.48**	**1.62**

18.10.3 *Plant Operating Costs*

The estimated process operating cost is approximately C$5.90/t milled or C$259 M/a. The estimate is based on an annual process rate of 43,800,000 t at an operation availability of 92%.

A summary of the plant operating cost is shown in Table 18.34. All the costs are exclusive of taxes, permitting costs, or other government imposed costs unless otherwise noted. The following aspects have been included in the estimate:

- manpower requirement including supervision, operation, and maintenance; salary/wage levels based on current labour rates in comparable operations in British Columbia. Benefit burden of 40% including holiday and vacation payment, pension plan, various benefits, northern allowance and tool allowance costs
- power supply from potential local electric grid line
- liner and grinding media consumption estimated from the Bond ball mill work index and the Wardrop database
- maintenance supplies cost, including building maintenance cost, based on approximately 5% of major equipment capital costs
- laboratory supplies, service vehicles consumables and other costs based on Wardrop's in-house database and industry experience
- reagent costs based on the consumption rates from test results and quoted budget prices or Wardrop database.

Table 18.34 Summary of the Process Operating Cost

Description	Staffing	Annual Cost (C$)	Unit Cost (C$/t Milled)
Operating Staff	33	$3,981,000	$0.091
Operating Labour	112	$10,026,000	$0.229
Maintenance	83	$8,301,000	$0.190
Sub-total Labour	**228**	**$22,308,000**	**$0.509**
Metal Consumables		$66,444,000	$1.517
Reagent Consumables		$94,003,000	$2.146
Maintenance Supplies		$26,630,000	$0.608
Operating Supplies		$2,530,000	$0.058
Sub-total Consumables and Supplies		**$189,606,000**	**$4.329**
Power Supply		$46,683,000	$1.066
Sub-total Power		**$46,683,000**	**$1.066**
Total (Process)		**$258,598,000**	**$5.904**

The estimated manpower cost is C$0.51/t milled. A total of 228 persons are estimated for the process operation, including 33 staff for management and professional services, 112 operators for operating and assaying, and 83 personnel for maintenance. The estimate is based on 12 h/shift, 10 d in and 10 d out, 24 h/d and 365 d/a.

Major metal consumables are estimated at C$1.52/t milled. The metal consumables include mill and crusher liners and grinding media.

The estimated reagent cost is C$2.15/t milled. Reagent consumptions were estimated from laboratory test results and comparable operations. The reagent costs were from the current budget prices from potential suppliers or Wardrop database.

The maintenance supplies are estimated at C$0.61/t milled. The power cost is estimated based on the average power requirement of 107 MW and a unit electric energy price of C$0.046/kWh.

The operating cost breakdowns for flotation plant, cyanide leach plant, and tailing delivery and reclaim water are detailed below:

Operating Cost - Crushing, Grinding, Flotation, and Concentrate Dewatering

The estimate operating cost for flotation plant including crushing, grinding, copper pyrite and molybdenum flotation, and flotation concentrate dewatering is shown in Table 18.35. Total cost for the process is estimated at C$4.24/t milled. A total of 161 personnel are required to operate the plant.

Table 18.35 Operating Cost – Comminution, Flotation, and Concentrate Dewatering

Description	Staffing	Annual Cost (C$)	Unit Cost (C$/t Milled)
Operating Staff	22	2,566,000	0.059
Operating Labour	72	6,473,000	0.148
Maintenance	67	6,749,000	0.154
Sub-Total Labour	**161**	**15,788,000**	**0.360**
Metal Consumables		66,444,000	1.517
Reagent Consumables		33,980,000	0.776
Maintenance Supplies		23,397,000	0.534
Operating Supplies		2,255,000	0.051
Power		43,920,000	1.003
Sub-Total Supplies		**169,996,000**	**3.881**
Total	**161**	**185,784,000**	**4.242**

OPERATING COST - GOLD LEACH, GOLD RECOVERY AND CYANIDE HANDLING

The operating costs for cyanide leaching plant including gold leach, gold recovery, cyanide recovery and leach residues cyanide destruction are estimated at C$1.56/t milled or C$10.61/t leach feed. The costs are shown in Table 18.36 . The leach and cyanide handling operations will be operated by their designated personnel including staff, labour, and maintenance. The labour cost is estimated at C$0.13/t milled. The reagent consumption is the major cost which is estimated to be C$1.37/t milled. The estimated maintenance supply cost is C$0.03/t milled.

Table 18.36 Operating Cost - Gold Leach, Gold Recovery, and Cyanide Handling

Description	Staffing	Annual Cost (C$)	Unit Cost (C$/t CIL Feed)	Unit Cost (C$/t Milled)
Operating Staff	11	1,415,000	0.220	0.032
Operating Labour	32	2,842,000	0.441	0.065
Maintenance	16	1,552,200	0.241	0.035
Sub-total Labour	**59**	**5,809,000**	**0.901**	**0.133**
Reagent Consumables		60,022,000	9.31	1.370
Maintenance Supplies		1,498,000	0.233	0.034
Operating Supplies		135,000	0.021	0.003
Power Supply		891,000	0.138	0.020
Sub-total Supplies		**62,547,000**	**9.706**	**1.428**
Total	**59**	**68,355,000**	**10.608**	**1.561**

OPERATING COST – TAILING DELIVERY AND WATER RECLAIM

The total operating cost for tailings delivery and water reclaim is estimated at C$0.10/t milled. The operating cost estimates includes only the costs to deliver tailings to the TSF and to reclaim the water from the TSF. The tailings dam construction and operation costs are included in project sustaining capital.

The breakdown unit costs for labour, maintenance supplies, operating suppliers and power supply are given in Table 18.37.

Table 18.37 Operating Cost - Tailing Delivery and Water Reclaim

Description	Staffing	Annual Cost (C$)	Unit Cost (C$/t Milled)
Labour	8	711,000	0.016
Supplies		3,746,000	0.086
Maintenance Supplies		1,734,500	0.040
Operating Supplies		140,000	0.003
Power Supply		1,872,000	0.043
Total	**8**	**4,458,000**	**0.102**

18.10.4 General and Administration and Surface Services

G&A and surface services costs are estimated to be C$0.67/t milled and C$0.21/t milled respectively. The costs are developed by Wardrop and Silver Standard.

The G&A costs include:

- labour cost for administrative personnel
- expense and services related to general administration, travel, human resources, safety and security
- allowances for insurance, regional taxes, and licenses
- sustainability, including environment, community liaison, and engineering consulting
- transportation of personnel, including air and road transportation
- camp accommodation costs.

A summary of the G&A costs are provided in Table 18.38.

Table 18.38 General and Administrative Operating Cost

G & A	Staffing	Total Cost (C$/a)	Unit Cost (C$/t Milled)
General & Administrative Labour			
General & Administrative	36	3,773,000	0.086
G & A Hourly Personnel	12	1,080,000	0.025
Sub-Total Man Power	48	4,853,000	0.111
General & Administrative Expenses			
General Office Expense		250,000	0.006
Computer Supplies inc Software		200,000	0.005
Communications		275,000	0.006
Travel		200,000	0.005
Audit		100,000	0.002
Consulting/External Assays		600,000	0.014
Head Office Allowance: Marketing		200,000	0.005
Environmental		3,000,000	0.068
Insurance		2,000,000	0.046
Regional Taxes & Licenses Allowance		1,000,000	0.023
Legal Services		200,000	0.005
Warehouse		1,150,000	0.026
Recruiting		100,000	0.002
Entertainment/Memberships		150,000	0.003
Employee Communications		100,000	0.002
Medicals & First Aid		150,000	0.003
Relocation Expense		100,000	0.002
Training/Safety		1,000,000	0.023
Accommodation/Camp Costs		6,500,000	0.148
Crew Transportation (Flight+Bus)		5,500,000	0.126
Liaison Committee/Sustainability		200,000	0.005
Small Vehicles		140,000	0.003
Others		200,000	0.005
Sub-Total Expenses		**24,465,000**	**0.559**
Total	**48**	**29,318,000**	**0.670**

The surface service cost estimates are shown in Table 18.39 and include:

- labour costs for surface service personnel
- surface mobile equipment and light vehicle operations
- portable water and waste management
- general maintenance including yards, roads, fences, and building maintenance
- off-site operation expense
- building heating
- avalanche control.

Table 18.39 Surface Services Operating Cost

Surface Service	Man Power	Total Cost (C$/a)	Unit Cost (C$/t Milled)
Surface Service Personnel	21	1,989,000	0.045
Surface Service Expenses		7,075,000	0.162
Small Vehicles/Equipment		300,000	0.007
Potable Water & Waste Management		425,000	0.010
Supplies		200,000	0.005
Building Maintenance		950,000	0.022
Building Heating		2,000,000	0.046
Road Maintenance		2,000,000	0.046
Avalanche Control		1,000,000	0.023
Off-Site Operation Expenses		200,000	0.005
Total	**21**	**9,064,000**	**0.207**

18.11 FINANCIAL ANALYSIS

18.11.1 INTRODUCTION

An economic evaluation of the Snowfield project was prepared by Wardrop based on a pre-tax financial model. For the 23-year LOM and 966.2 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 8.1% IRR
- 7.4 years payback on US$3,378 M capital
- US$877 M NPV at 5% discount rate.

The base case prices, using Wardrop adopted consensus forecast metal prices from the Energy Metals Consensus Forecast (ECMF) are as follows:

- Silver – US$14.50/oz
- Gold – US$878/oz
- Copper – US$2.95/lb
- Molybdenum – US$17.00/lb
- Exchange rate – 0.92:1.00 (US$:C$).

Sensitivity analyses were carried out to evaluate the project economics for several metal prices scenarios.

18.11.2 Pre-Tax Model

Financial Evaluations

The production schedule, based on a mineable inventory with an NSR cut-off grade of US$6.70/t, has been incorporated into the pre-tax financial model to develop annual recovered metal production. Market prices for gold, silver, copper, and molybdenum have been adjusted to realized price levels by applying smelting, refining, and concentrate transportation charges from mine site to smelter in order to determine the NSR contributions for each metal.

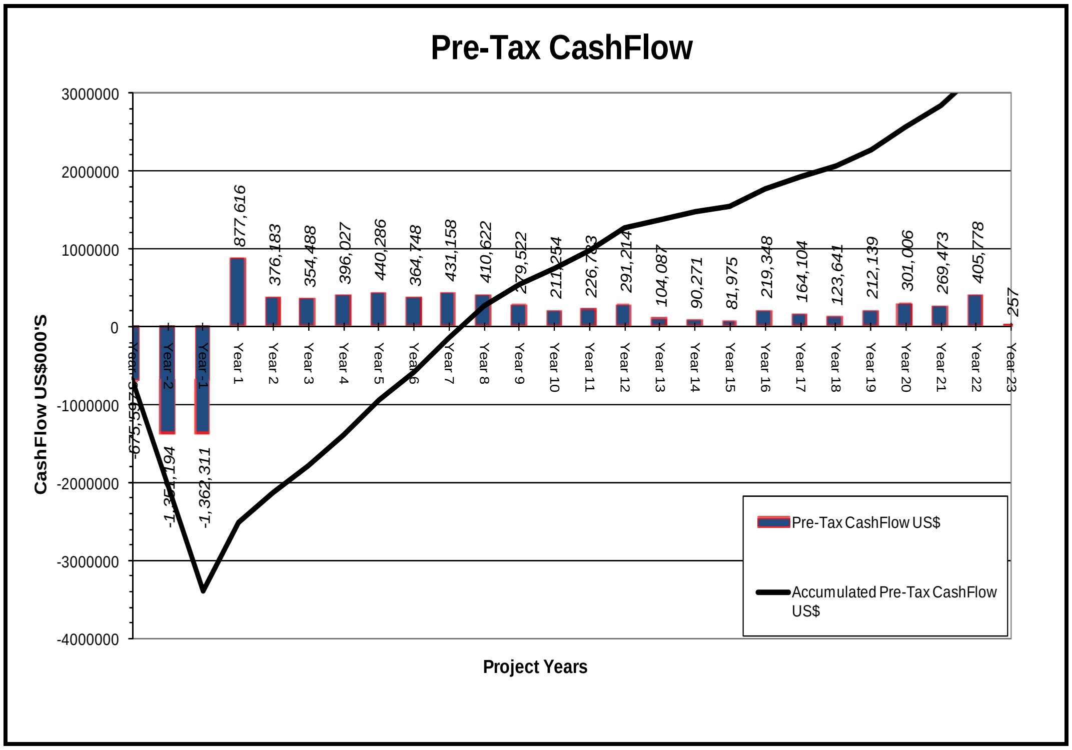
Unit operating costs of US$8.52 were multiplied by annual milled tonnages to determine the total mine operating costs. The total mine operating costs were then deducted from NSRs to derive annual mine income.

Initial and sustaining capital costs have been incorporated on a year-by-year basis over the mine life and deducted from the net revenue to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of concentrate; sustaining capital includes expenditures for mining and milling additions, replacement of equipment, and tailing embankment construction.

Working capital has been calculated on the basis of three months mine site operating costs and applied to the first year of expenditures. It will be recovered at the end of the mine life and aggregated with the salvage value contribution and applied towards reclamation during closure.

The annual pre-tax cash flow is presented in Figure 18.26.

Figure 18.26 Pre-Tax Cash Flow



METAL PRICES SCENARIOS

The commodity prices are within one standard deviation of the consensus forecast from the EMCF. The EMCF is published by Consensus Economics Inc. (Consensus Economics) of London. Consensus Economics provide quarterly forecast (the EMCF) for a variety of metals prices based on an average price from long term projections of 20 analysts representing international banks.

The financial outcomes for four metal price scenarios have been tabulated for NPV, IRR, and payback of capital. Discount rate of 5% was applied to all cases identified by the following metal price scenarios:

- Base case
- Alternate Case
- Spot metal prices as of April, 8, 2010.

The summary of the project economic evaluation is presented in Table 18.40.

Table 18.40 Summary of Pre-tax NPV, IRR, and Payback by Metal Price Scenario

Economic Returns	Unit	Base Case	Alternate Case	Current Prices (April 8, 2010)
Project IRR	%	8.1	3.4	16.3
NPV at 5.0% Discount Rate	M US$	877	-339	3,593
Payback	Years	7.4	11.7	4.5
Exchange Rate	US$:C$	0.92	0.92	0.92
Mine Life	Years	23	23	23
Au Price	(US$/oz)	878	800	1150
Ag Price	(US$/oz)	14.50	12.55	18.10
Mo	(US$/lb)	17.00	13.91	19.54
Cu	(US$/lb)	2.95	2.35	3.50

ROYALTIES

There are no royalties on the project.

18.11.3 SMELTER TERMS

In the absence of letters of interest or letters of intent from potential smelters or buyers of concentrate, in-house database numbers were used to benchmark the terms supplied by Silver Standard.

Contracts will generally include payment terms as follows:

- Copper Concentrate:
  - Silver – pay 90% of silver content; a refining charge of US$0.45/accountable troy oz will be deducted from the metal price.
  - Gold – pay 97.5% of Gold content; a refining charge of US$8.00/accountable troy oz will be deducted from the metal price.
  - Copper – Deduct 1 unit of the copper concentrate content; a refining charge of US$0.09/accountable lb will be deducted from the metal price.
  - Treatment and Smelting Charge – US$85/dmt of concentrate delivered will be deducted. The treatment charge might be subject to both positive and negative price escalation.
  - Impurities –no penalties are applied due to insufficient assay data for impurity elements.

- Doré:
  - Gold – pay 99.8% of Gold content; a smelting and transport charge of $2.00/troy oz will be deducted from the metal price.
  - Silver– pay 99.8% of Silver content; a smelting and transport charge of $2.00/troy oz will be deducted from the metal price.
- Molybdenum Concentrate:
  Contracts will generally include payment terms for molybdenum as follows:
  - There will be 2.5% deduction from the recovered molybdenum by the smelter; therefore, the mine will receive 97.5% of the recovered molybdenum.
  - There is a roasting charge of US$1.50 per accountable pound of molybdenum.
  - Impurities –no penalties are applied due to insufficient assay data for impurity elements.

18.11.4 *Markets and Contracts*

Markets

The project will produce a copper concentrate containing the majority of the recovered gold, silver, and copper as well as a separate molybedenum concentrate which contains rhenium. In addition gold and silver doré will be produced.

Contracts

There are no established contracts for the sale of concentrate currently in place for this project.

18.11.5 *Concentrate Transport Logistics*

Concentrate from the mine site will be truck transported to a port facility where it gets transferred onto ships. Transportation charges were prepared by Wardrop for truck, port and ocean freight.

- truck transport – C$25.00/wmt
- port storage and handling – C$25.00/wmt
- ocean transport – US$65.00/wmt
- moisture content – 9%.

CONCENTRATE TRANSPORT INSURANCE

An insurance rate of 0.15% will be applied to the provisional invoice value of the concentrate to cover transport from the mine site to the smelter.

OWNERS REPRESENTATION

An Owners representation rate of US$0.50/wmt will be applied to the provisional invoice value of the concentrate to cover attendance during unloading at the smelter, supervising the taking of samples for assaying, and determining moisture content.

CONCENTRATE LOSSES

Concentrate losses are estimated at 0.5% of the provisional invoice value during shipment from the mine to smelter.

18.11.6 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- copper price
- gold price
- silver price
- molybdenum price
- exchange rate
- copper grade
- gold grade
- silver grade
- molybdenum grade
- operating cost
- capital cost.

The analyses are presented as financial outcomes in terms of NPV in Table 18.41 and Figure 18.27 and internal rate of return (IRR) in Table 18.42 and Figure 18.28. The project NPV (at 5% discount rate) is most sensitive to the exchange rate, gold price, and gold grade.

Similarly, the project IRR is most sensitive to the fixed exchange rate (FXR) followed by gold grade and gold price.

Table 18.41 Output Variable Values for NPV

	Sensitivity (%) US$ (M)				
	-20.0	**-10.0**	**0.0**	**10.0**	**20.0**
Cu Price	493	685	877	1,070	1,262
AU Price	-511	183	877	1,572	2,266
Ag Price	827	852	877	903	928
Mo Price	685	781	877	974	1,070
Exchange Rate	2,635	1,756	877	-1	-880
Cu Grade	198	537	877	1,221	1,569
Au Grade	-622	125	877	1,629	2,374
Ag Grade	830	854	877	901	925
Mo Grade	704	791	877	964	1,051
Operating Cost	1,121	999	877	756	634
Capital Cost	1,540	1,209	877	546	215

Note: US$ M

Figure 18.27 NPV Sensitivity Analysis

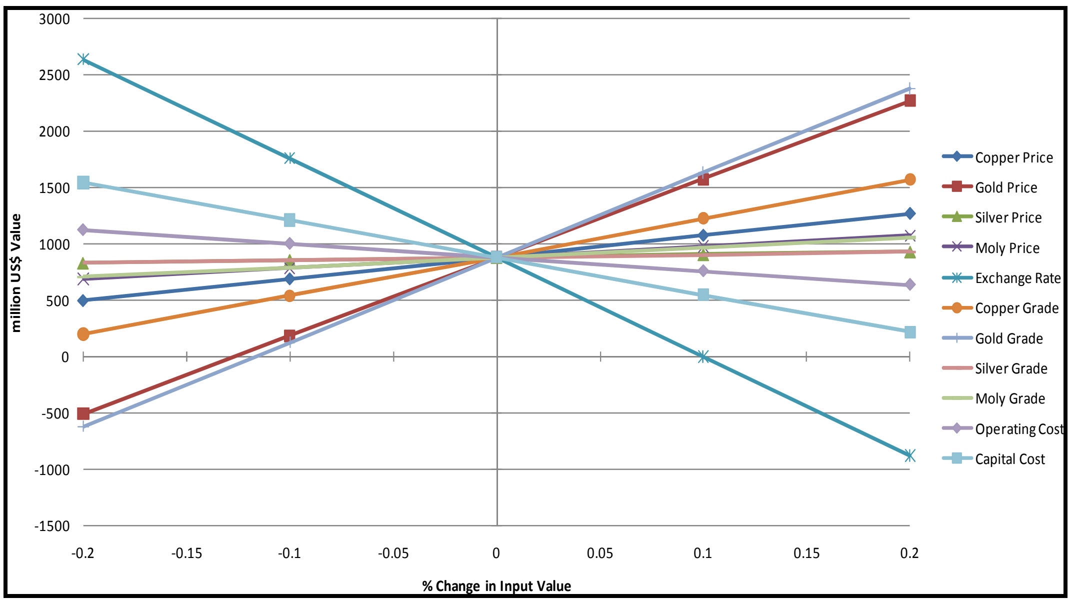
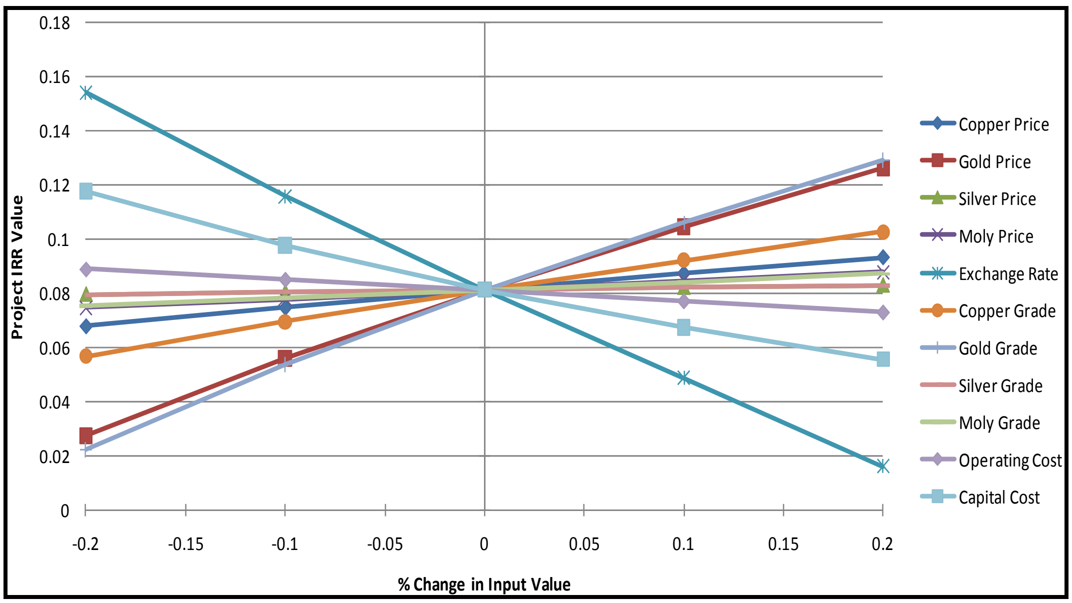


Table 18.42 Output Variable Values for Project IRR

	Sensitivity (%)				
	-20.0	-10.0	0.0	10.0	20.0
Cu Price	6.8	7.5	8.1	8.7	9.3
AU Price	2.7	5.6	8.1	10.4	12.6
Ag Price	7.9	8.0	8.1	8.2	8.3
Mo Price	7.4	7.8	8.1	8.4	8.8
Exchange Rate	15.4	11.6	8.1	4.9	1.6
Cu Grade	5.7	6.9	8.1	9.2	10.2
Au Grade	2.2	5.4	8.1	10.6	12.9
Ag Grade	8.0	8.0	8.1	8.2	8.3
Mo Grade	7.5	7.8	8.1	8.4	8.7
Operating Cost	**8.9**	**8.5**	**8.1**	**7.7**	**7.3**
Capital Cost	**11.7**	**9.8**	**8.1**	**6.7**	**5.6**

Figure 18.28 IRR Sensitivity Analysis



19.0 CONCLUSIONS AND RECOMMENDATIONS

19.1 CONCLUSIONS

Based on the results of the project PA, Wardrop recommends that Silver Standard proceed with the next phase of the project, in order to identify opportunities and further assess viability of the project.

Based on these conclusions and recommendations, the pre-feasibility phase of work for this project is expected to include additional in-fill drilling to complete reserve definition, geotechnical studies, and hydrogeologic investigations. On a preliminary basis, the drilling and associated studies are estimated to cost approximately US$7 M and production of the subsequent pre-feasibility report is projected to cost approximately US$5 M for a total of US$12 M.

19.2 RECOMMENDATIONS

19.2.1 GEOLOGY

P&E is of the opinion that Silver Standard should continue with a comprehensive exploration program in 2010, with the main focus being to:

- attempt to convert a large portion of the inferred resources to measured and indicated
- test for extensions of the known mineralization
- prospect, map, and trench numerous other showings, which were located as part of historical programs.

A 16,000 m diamond drilling program is recommended to potentially upgrade the inferred resources to the measured and indicated categories. A portion of the drilling should be used to test possible deposit extensions.

In addition to the drilling programs, a portion of the budget should be allocated to prospecting in the area.

19.2.2 GEOTECHNICAL/HYDROGEOLOGICAL

Recommendations for the next phase of design (prefeasibility level) to confirm the geotechnical assumptions for the tailings storage facility, waste dumps, and water balances are summarized in the following section.

RECOMMENDATIONS FOR OPEN PIT DESIGN

- Five to eight geotechnical core holes should be drilled, with lengths sufficient to provide samples from the proposed ultimate PA pit walls. This will require approximately 3,500 m to 4,000 m of drilling.
- Geotechnical logging, point load testing, and sampling of the geotechnical drill holes should be conducted.
- Acoustic and optical televiewer surveys of the geotechnical drill holes should be conducted, to provide data on the orientation of discontinuities (joints, faults, etc.) as well as the density of the features along the holes and the thickness or in-situ aperture of individual features.
- Packer testing of the geotechnical drill holes should be conducted to provide data for the estimates of hydraulic conductivity of the Snowfield rock mass.
- Installation of piezometers and data loggers in the geotechnical drill holes should be conducted to provide data on current hydrogeological conditions.
- Laboratory testing of rock core samples should be conducted to determine the uniaxial compressive strength, Brazilian tensile strength, and direct shear strength.
- Field mapping of outcrops should be conducted for rock mass properties and structural geology.
- Field mapping of geomorphic features related to the Snowfield Landslide should be conducted.
- Monitoring stations should be installed on the landslide, and a monitoring program should be established.
- Interpretation and analysis of the Snowfield Landslide should be conducted to estimate the extents and failure mode.
- The available data and design of the open pit slopes should be analyzed.
- A 3D model of lithologic units, alteration zones, and major geological structures should be developed.
- A preliminary 3D hydrogeological model of the pit area should be developed to support dewatering and depressurization estimates.
- A preliminary review and identification of geohazards in and around the pit area via aerial photographs, satellite imagery, and field mapping should be conducted.

- With respect to pit slope angles, reasonable assumptions have been used to estimate the rock mass strength of the Snowfield deposit. If site investigations show higher intact rock strengths than currently assumed, it may be possible to steepen some pit slopes (or parts of slopes) if the geological structure is favourable.

RECOMMENDATIONS FOR WASTE DUMP DESIGN

- Geotechnical and hydrogeological site investigations (i.e. mapping, drilling, geophysics, and/or test pits excavations) will need to be completed.
- Characterize glaciers and icefields in the vicinity of the proposed waste dumps. The extent and thickness of all glaciers and icefields in the vicinity of the proposed East dump and Southwest dump must be determined.
- Geotechnical stability analyses of the dumps should be conducted once the site investigations have been completed.

RECOMMENDATIONS FOR TSF DESIGN

- High resolution topographic data with elevation precision of ±1 m should be obtained for the entire catchment area of the proposed Scott Creek TSF.
- Stereoscopic aerial photographs should be obtained for the Scott Creek TSF study area and Snowfield open pit and waste dumps area to allow review of the geomorphology of these areas to support geohazard assessments, field mapping, and borrow assessments. Photos should be at a 1:15,000 to 1:20,000 scale.
- Geotechnical and hydrogeological site investigations (i.e. mapping, drilling, geophysics, and test pits excavations) will need confirm the assumptions used to develop the preliminary designs presented in this report. Collection of baseline surface water, groundwater quantity, and quality data at the three proposed seepage recovery facilities should be completed.
- Borrow studies to identify specific locations and characterize potential areas for rockfill, granular filters, and low permeability soils must be completed. Dam slope stability and preliminary seepage analyses should be done once geotechnical site investigations and laboratory testing are completed.
- A probabilistic and deterministic seismic hazard assessment should be completed for the proposed TSF site.
- Laboratory testing should be completed on representative samples of tailings from the Snowfield deposit.
- A snow avalanche hazard assessment should be completed for the proposed Scott Creek impoundment (including the diversion channels and maintenance access roads). This will require a combination of desk study and field assessment by a specialist snow avalanche sub-consultant.

- A geohazard assessment must be completed to identify and characterize potential geohazards impacting the TSF and auxiliary facilities.
- Once the tailings have been characterized, cyclone sand could be considered as a construction material for dam raises during operations.

RECOMMENDATIONS FOR TSF, PIT & WASTE DUMP WATER BALANCES

- All existing hydrometric stations and flow monitoring sites (i.e. Brucejack Lake) must continue to be monitored and maintained with an appropriate level of quality control.
- Meterological station(s), rain gauge(s), and flow monitoring site(s) should be installed in Scott Creek to confirm assumptions on the water balance and 200 a run-off values. An automated hydrometric system should be installed on Mitchell Creek, or alternatively, weekly manual flow measurements should be conducted on this creek.
- Snowpack surveys should be conducted throughout the Scott Creek watershed prior to snowmelt to quantify snowfall distribution and confirm precipitation measurements at the Brucejack Lake climate station.
- A freshwater source for the process plant needs to be identified for the winter months (300 m^3/h).
- The water balance model and water management strategy needs to be refined to account for staging of the various mine facilities (i.e. pit and dump staging over the life of mine). This work may include probabilistic water balance modeling.
- Acceptable risk tolerance criteria must be established for water management (i.e. confirm the adoption of a 200-year return period, 1-year duration as the design standard for the open pit sumps, pumps and pipeline to the process plant).
- It is currently assumed that the build-up of surplus water in the supernatant pond will have suitable water quality for seasonal discharge. This assumption needs to be rigorously tested in the next stage of engineering design.

19.2.3 ENVIRONMENTAL

It is recommended that Silver Standard proceed with a standard environmental assessment study. During the course of this study, baseline information will be collected which will aid in the environmentally sensitive design of certain project facilities, such as the waste rock facilities for which glaciological studies will be conducted.

Mine water and waste rock flows will be geochemically characterized to ensure that adequate water treatment is provided during operations and at closure.

19.2.4 MINING

The following are recommendations for the next phase of study.

- A trade-off study is recommended to determine whether the 120,000 t/d throughput rate is optimal. This can be accomplished by conducting an economic evaluation of tonnage increments above and below the 120,000 t/d rate.
- Detailed mining simulations and cut-off grade optimization is required to optimize the ore blending strategy in the Upper and Lower ore zones, in order to maximize the NPV.
- The impact of ore dilution and ore loss on project economics should be evaluated in detail.
- Optimization of the size of the shovel and truck fleet should be conducted during the next phase of study. The optimization should include an economic evaluation of the use of electric-driven shovels and drills.
- An evaluation of locating a second primary crusher further down the pit, closer to the mining activities, compared with using trucks to haul ore to a single crusher, should be conducted.
- A detailed hydrogeological evaluation of the pit area should be conducted, in order to determine the design of overall dewatering systems in and around the open pit.
- Detailed drilling and blasting studies should be conducted in order to map water contacts and rock hardness from specific rock types. The information will help determine the powder factor and explosives mix for each rock type.
- An economic analysis to compare the efficiency and cost-effectiveness of owner-run blasting versus full-contractor blasting should be conducted.
- An evaluation of possible use of a mining contractor when material volumes peak in later years of mine operations, to moderate capital investment in the number of large haul trucks close to the end of LOM, should be conducted.
- An additional study on channelling the surface water from the waste dumps and vertical wells through specific bench elevations to assess the impact on additional waste removal should be conducted. The bench that includes the diversion channel, service road, and berm is expected to be wide to accommodate the high volume of water run-off during the snow melt season. The addition of water channels will flatten the high wall slope and increase waste stripping.
- An evaluation of the cost-effectiveness of using in-pit haul roads using switchbacks versus the use of outside-pit haul roads should be conducted.

19.2.5 PROCESS AND METALLURGY

The following are recommendations for the next phase of study:

- Initial metallurgical tests suggest that the deposit may contain significant amount of rhenium. Investigation of potential rhenium value recovery from molybdenum concentrate should be conducted, and its impact on improved project economics should be evaluated. The project geological block model should be updated to include rhenium.
- Further testwork is required to confirm the previous testwork findings, optimize process flowsheet, and investigate metallurgical performances. The testwork should include:
  - mineralogical analysis
  - mineralization hardness determination and grinding circuit simulation
  - flotation, including copper and molybdenum separation, and the effect of raw water from the proposed pit and waste rock storage site on flotation
  - cyanidation, including cyanide solution handling
  - ancillary tests, including settling and filtration tests
  - copper recovery by hydrometallurgical processes
  - pilot plant scale tests
- Optimization of primary grinding circuit, including SAG milling, ball mill milling, and pebble crushing (SABC) circuit, should be conducted.
- The mill throughput should be optimized further.
- The potential energy recovery for the tunnel conveyor system should be investigated.
- The potential energy saving opportunities, including processes and equipment for the project, should be investigated.

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21.0 DATE AND SIGNATURE PAGE

The effective date of this Technical Report, titled "Technical Report and Preliminary Assessment of the Snowfield Property", is June 1, 2010.

Signed,

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