Report to:

SILVER STANDARD RESOURCES INC.

Technical Report and Preliminary Assessment of the Snowfield-Brucejack Project

Document No. 1053750400-REP-R0001-03

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TECHNICAL REPORT AND PRELIMINARY ASSESSMENT OF THE SNOWFIELD-BRUCEJACK PROJECT

EFFECTIVE DATE: SEPTEMBER 10, 2010

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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 AMC Mining Consultants Ltd. (AMC). 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 AMC, 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 AMC. 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*.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

GLOSSARY

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 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
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

ABBREVIATIONS AND ACRONYMS

Absolute Relative Difference	ABRD
acid base accounting	ABA
acid rock drainage	ARD
Aero Geometrics Ltd.	Aero Geometrics
Alpine Tundra	AT
ALS Chemex Laboratories Ltd.	ALS Chemex
AMC Mining Consultants Ltd.	AMC
Assayers Canada Ltd.	Assayers Canada
Atomic Absorption Spectrophotometer	AAS
atomic absorption	AA
BGC Engineering Inc.	BGC
Black Hawk Mining Inc.	Black Hawk
British Columbia Environmental Assessment Act	BCEAA
British Columbia Environmental Assessment Office	BCEAO
British Columbia Environmental Assessment	BCEA
British Columbia Transmission Corp.	BCTC
British Columbia	BC
Canadian Dam Association	CDA
Canadian Environmental Assessment Act	CEA Act
Canadian Environmental Assessment Agency	CEA Agency
Canadian Institute of Mining, Metallurgy, and Petroleum	CIM
Canadian National Railway	CNR
carbon-in-leach	CIL
Caterpillar's® Fleet Production and Cost Analysis software	FPC
closed-circuit television	CCTV
coefficient of variation	CV
Consensus Economics Inc.	Consensus Economics
counter-current decantation	CCD
cyanide soluble	CN
digital elevation model	DEM
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	Esso
flocculant	floc
Free Carrier	FCA
Gemcom International Inc.	Gemcom
general and administration	G&A
Geospark Consulting	Geospark
gold-equivalent	Au-Eq
Granduc Mines Ltd.	Granduc

heating, ventilating, and air conditioning	HVAC
high pressure grinding rolls	HPGR
Indicator Kriging	IK
inductively coupled plasma atomic emission spectroscopy	ICP-AES
inductively coupled plasma	ICP
Inspectorate America Corp.	Inspectorate
Interior Cedar – Hemlock	ICH
internal rate of return	IRR
International Congress on Large Dams	ICOLD
International Plasma Labs	IPL
Inverse Distance Cubed	ID3
Kerr-Sulphurets-Mitchell	KSM
Lacana Mining Corp.	Lacana
Land and Resource Management Plan	LRMP
Lerchs-Grossman	LG
life-of-mine	LOM
load-haul-dump	LHD
locked cycle tests	LCTs
Loss on Ignition	LOI
McElhanney Consulting Services Ltd.	McElhanney
Metal Mining Effluent Regulations	MMER
Methyl Isobutyl Carbinol	MIBC
metres East	mE
metres North	mN
Mineral Deposits Research Unit	MDRU
Mineral Titles Online	MTO
National Instrument 43-101	NI 43-101
Nearest Neighbour	NN
net invoice value	NIV
net present value	NPV
Net Smelter Prices	NSP
Net Smelter Return	NSR
Neutralization Potential	NP
Newhawk Gold Mines Ltd.	Newhawk
Newhawk International Corona Corp.	Newhawk International
Newhawk, Lacana, and Granduc joint venture	Newcana JV
Northwest Transmission Line	NTL
official community plans	OCPs
operator interface station	OIS
ordinary kriging	OK
organic carbon	org
P&E Mining Consultants Inc.	P&E
Palisade Corp.	Palisade
Pincock Allen & Holt Ltd.	PA&H
Placer Dome Inc.	Placer Dome
potassium amyl xanthate	PAX

Predictive Ecosystem Mapping	PEM
Preliminary Assessment	PA
Preliminary Economic Assessment	PEA
Process Research Associates Ltd.	PRA
Qualified Persons	QPs
quality assurance	QA
quality control	QC
Rescan Environmental Services Ltd.	Rescan
rhenium	Re
rock mass rating	RMR '76
rock quality designation	RQD
SAG mill/ball mill/pebble crushing	SABC
Seabridge Gold Inc.	Seabridge
semi-autogenous grinding	SAG
Silver Standard Resources Inc.	Silver Standard
Social and Community Management System	SCMS
Standards Council of Canada	SCC
Stanford University Geostatistical Software Library	GSLIB
tailings storage facility	TSF
Terrestrial Ecosystem Mapping	TEM
Thompson-Howarth	T-H
total dissolved solids	TDS
Total Suspended Solids	TSS
Traditional Knowledge/Traditional Use	TK/TU
tunnel boring machine	TBM
underflow	U/F
Valued Ecosystem Components	VECs
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	WHMIS
X-Ray Fluorescence Spectrometer	XRF

1.0 SUMMARY

1.1 INTRODUCTION

In June 2010, Silver Standard Resources Inc. (Silver Standard) commissioned Wardrop Engineering Inc. (Wardrop) to conduct a preliminary assessment (PA) of the Snowfield-Brucejack deposits.

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

- Wardrop: processing, infrastructure, capital and operating cost estimates, and financial analysis
- AMC Mining Consultants (Canada) Ltd. (AMC): mining
- 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

The Snowfield and Brucejack properties together make up the Snowfield-Brucejack Project. Due to their unique histories, dates of acquisition, and property boundaries, each property is best described individually.

1.2.1 SNOWFIELD PROPERTY

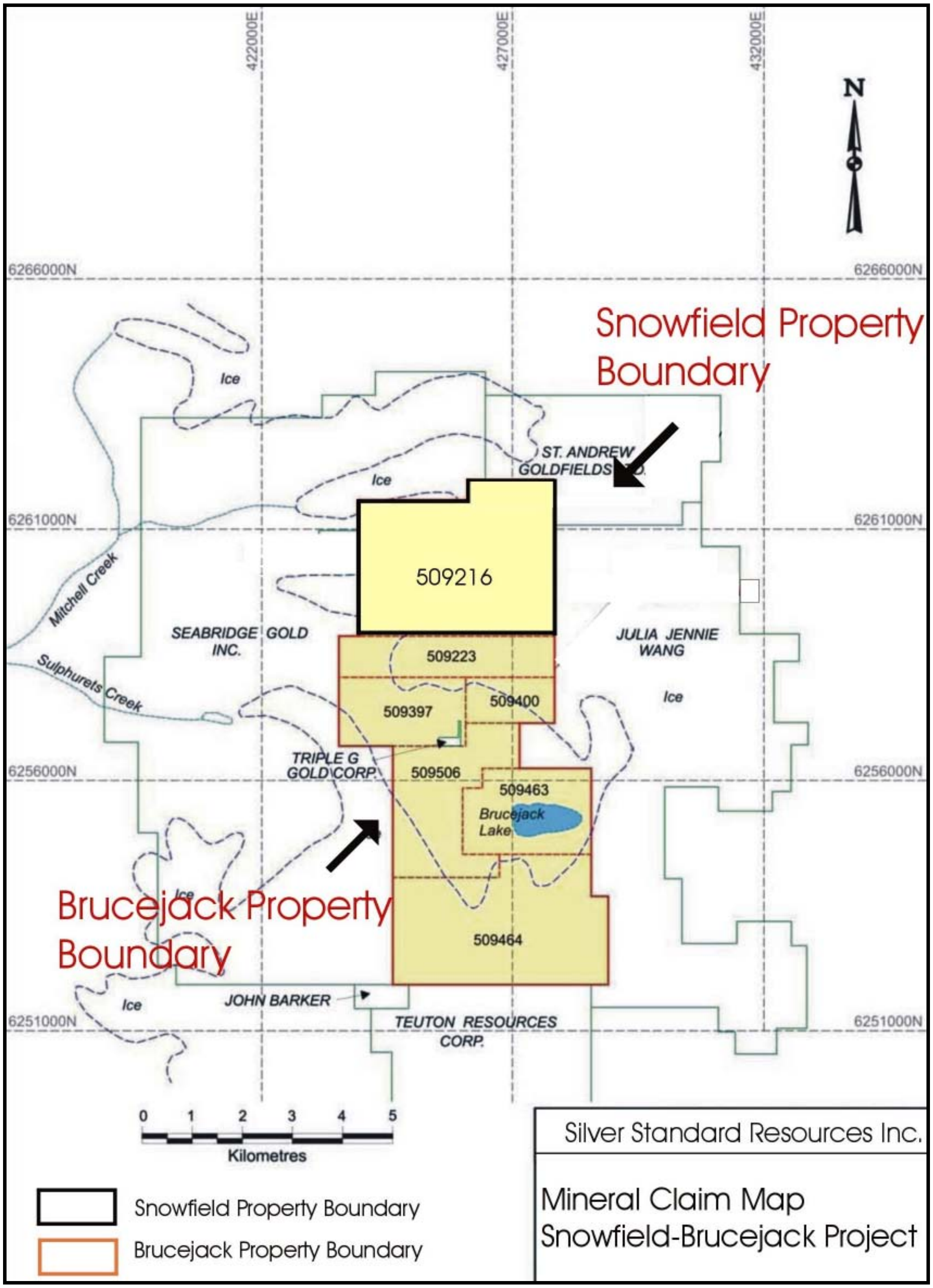
In 1999, Silver Standard acquired the Sulphurets claim group, including the Snowfield deposit, through the acquisition of Newhawk Gold Mines Ltd. (Newhawk). Subsequent to the acquisition of the shares of Newhawk, Silver Standard reorganized the claim ownership. All of the associated mineral claims are now held by 0777666 BC 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-Brucejack Project 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 Regional Map of BC with Location of Snowfield-Brucejack Project

Figure 1.2 Detailed Location Map of the Snowfield-Brucejack Project



Note: modified after Blanchflower, 2008.

1.2.2 BRUCEJACK PROPERTY

The Brucejack property consists of 6 mineral claims totalling 3,199.28 ha in area; all claims are in good standing until January 31, 2017. In 2001, Silver Standard purchased Black Hawk Mining Inc.'s (Black Hawk) 40% interest in the Brucejack property, resulting in 100% interest. Claim ownership is registered to 0777666 BC Ltd., a wholly-owned subsidiary of Silver Standard.

The 6 above-mentioned mineral claims were converted from 28 older legacy claims to BC's new Mineral Titles Online (MTO) system in 2005.

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 1999, when Silver Standard acquired the Brucejack property, the general area was intensely explored by companies such as Granduc, Esso Minerals Canada (Esso), Newhawk, and Newhawk International Corona Corp. (Newhawk International). These companies actively explored the region identifying over 50 mineralized showings including several large mineralized deposits such as the Kerr, Mitchell, Sulphurets, and Snowfield deposits.

Subsequent to acquiring the Snowfield and adjacent Brucejack properties from Newhawk in 1999 and Blackhawk in 2001, Silver Standard has drilled in excess of 55,500 m of core in approximately 129 holes (2006 through 2009) on the Snowfield property and 17,800 m in 37 holes on the Brucejack property (2009).

1.4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, AND PHYSIOGRAPHY

The Project 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 +20°C to approximately -20°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-Brucejack 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.

1.5 GEOLOGICAL SETTING

While deposits such as Snowfield, Kerr, and Mitchell are probably best described as gold-enriched copper porphyry systems, most (if not all) of the mineralization on the Brucejack property (West, Bridge, Galena Hill, Shore, SG, Gossan Hill, and Mammoth zones) has been classified as an epithermal Au-Ag-Cu, low-sulphidation deposit. It is possible that some of the mineralization also displays characteristics of intrusion-related vein systems that fall within the Intermediate-Sulphidation epithermal subtype of Hedenquist et al. (2000).

The Snowfield-Brucejack Project area 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.

The deposits on the Brucejack property are classed as epithermal deposits of Au (±Ag) which are a type of lode gold deposit that comprises veins and disseminations near the Earth's surface (≤1.5 km), in volcanic and volcaniclastic sedimentary rocks, sediments, and, in some cases, also in metamorphic rocks. The deposits may be found in association with hot springs and frequently occur at centres of young volcanism. The ores are dominated primarily by precious metals (Au, Ag) but some deposits may also contain variable amounts of base metals such as Cu, Pb, and Zn.

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 SNOWFIELD AND BRUCEJACK RESOURCES

The resource estimate at Snowfield is based on a conceptual Lerchs-Grossman optimized pit shell developed with inclusion of all available mineral resources (Measured, Indicated, and Inferred).

The results from an optimized pit-shell are used solely for the purpose of reporting Mineral Resources that have reasonable prospects for economic extraction.

All Snowfield Mineral Resources were tabulated against a 0.30 g/t Au-Eq cut-off, as constrained within the optimized pit shell. Gold, silver, copper, and molybdenum block grades were estimated using ordinary kriging (OK) of capped composite values. Rhenium block grades were estimated using co-kriging based on the observed correlation between molybdenum and rhenium.

Table 1.1 P&E Snowfield Mineral Resource Estimate (July 27, 2010) [1,2,3]

Class	Mt	Au (g/t)	Au (M oz)	Ag (g/t)	Ag (M oz)	Cu (%)	Mo (ppm)	Re (g/t)
Measured	143.7	0.83	3.85	1.57	7.27	0.08	100	0.62
Indicated	951.6	0.60	18.19	1.78	54.38	0.11	87	0.47
Measured + Indicated	1095.3	0.63	22.04	1.75	61.65	0.11	89	0.49
Inferred	847.2	0.40	10.99	1.53	41.62	0.07	82	0.33

[1] Mineral Resources 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.

The Brucejack Mineral Resource estimate encompasses six distinct modelled mineralization zones, namely the West Zone, Shore Zone, Gossan Hill Zone, Galena Hill Zone, SG Zone, and Bridge Zone. All Brucejack Mineral Resources were tabulated against a 0.35 g/t Au equivalent cut-off, as constrained within an optimized pit shell.

Table 1.2 P&E Brucejack Combined Mineral Resource Estimate (Dec. 1, 2009)

Class	Mt	Au (g/t)	Ag (g/t)	Au (M oz)	Ag (M oz)
Measured	9.9	2.06	75.0	0.66	23.8
Indicated	110.7	0.95	11.7	3.38	41.6
Measured + Indicated	120.5	1.04	16.9	4.04	65.4
Inferred	198	0.76	11.2	4.87	71.5

***Notes:**
– at a 0.35 g/t Au-Eq cut-off.
– Resource sensitivities are accumulated within an optimized pit shell.
– 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.
– 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.

1.7 Mining Operations

The Snowfield-Brucejack Project will be an open pit operation with a 27-year mine life and a total of 1,172 Mt of mineralization. Mining will be undertaken using 45 m^3 electric cable shovels, 39 m^3 diesel hydraulic shovels, 311 mm blasthole drills, and 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.

At Snowfield, 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. At Brucejack, the mining configuration is dependent upon the size of each final pit. The smaller pits are mined in 10 m benches with 20 m between catch benches. Pits deeper than 200 m are mined as per Snowfield.

Benches will be drilled on an 8.9 m x 10.2 m drill pattern to a depth of 16.8 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. Primary crushers will be located at the Snowfield and Brucejack pits, which will shorten haul distances of the crushed materials.

The scoping-level mine plan will be implemented by mining high net smelter return (NSR) value material during the early years of production. The mining of low NSR value material will be deferred to the later years of mine operations as per the optimized production schedule. The mining production schedule is presented in Table 18.11.

1.8 Metallurgical Testwork Review

Preliminary metallurgical testwork, including locked cycle tests, was carried out on Snowfield mineralization and Brucejack mineralization separately. The testing programs investigated mineralization characteristics and potential process technologies for the recovery of valuable elements from the two deposits and also determined some of process related data. The testwork was focused on the zone composite samples; however, the testwork was also conducted on the sub-zone samples or drill interval samples. The testwork results show:

- A combination of flotation and cyanidation can be used to recover gold, copper, silver, and molybdenum from the Snowfield mineralization. Gold recovery by gravity concentration from the reground rougher and scavenger concentrates may benefit downstream leaching process. It appears that rhenium can be recovered together with molybdenum into molybdenum concentrate.
- A combination of flotation, gravity concentration, and cyanidation can be used to recover gold and silver from the Brucejack mineralization.

The grindability test results showed that the mineralization is moderately hard, with an average Bond ball mill work index of approximately 16.0 kWh/t for both the deposits. Further testwork is recommended to optimize the flotation, gravity, and cyanidation flowsheet.

1.9 Mineral Processing

The proposed concentrator will process the gold/copper/molybdenum mineralization from the Snowfield deposit and the gold-silver mineralization from the Brucejack deposit. The concentrator will be fed at a nominal rate of 120,000 t/d and with an availability of 92% (365 d/a). The feed materials from the two deposits will be processed separately in different time periods according to the mining schedule. The concentrator will produce:

- a marketable copper concentrate containing gold and silver, a by-product molybdenum concentrate, and gold-silver doré from the Snowfield mineralization
- a gold-silver doré only from the Brucejack mineralization.

The process plant will consist of three stages of crushing, primary grinding, followed by flotation processes to recover copper, gold, silver, and molybdenum from the Snowfield material, or gold and silver only from the Brucejack material. The resulting bulk rougher/scavenger concentrates will be reground and gravity concentrated to recover free metallic gold.

Due to a difference in the mineralization, the downstream processes for the Snowfield mineralization and Brucejack mineralization are slightly different:

- For the Snowfield mineralization: a copper-gold-silver and molybdenum bulk cleaner flotation for the reground rougher concentrate and a copper-molybdenum separation circuit are proposed to produce a molybdenum concentrate and a copper concentrate containing gold and silver. The cleaner flotation tailing together with the reground rougher/scavenger concentrate will be cyanide leached to recover gold and silver. The recovered gold and silver will be refined on site to gold-silver doré. If gravity concentration is in operation, the gravity concentrate will be processed in an intensive leach circuit to recover gold and silver.
- For the Brucejack mineralization: a conventional cyanidation will be used to leach the reground rougher and scavenger concentrates (after gravity concentration) to recover gold and silver and an intensive leach to recover gold and silver from the gravity concentrate. The recovered gold and silver will be refined on site to gold-silver doré.

The copper-gold concentrate from Snowfield mineralization 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.

There will be two separate primary crushing systems for the Snowfield site and the Brucejack site. Primary crushing at the Snowfield site will include two fixed gyratory crushers while two semi-mobile gyratory crushers will be installed at Brucejack site. Crushed material from the Snowfield site will be conveyed to the plant site via the main tunnel conveying system in a 26 km-long tunnel (main tunnel). Crushed material from the Brucejack site will be conveyed to the transfer point within the main tunnel. The crushed Brucejack mineralization will be transferred onto the main tunnel conveying system at the transfer point.

Secondary crushing by four cone crushers and tertiary crushing by four high pressure grinding rolls (HPGR) will be located at plant site to reduce the mill feed to a particle size suitable for ball mill milling. The crushed material will be further reduced to 80% passing 125 µm prior to the metal recovery by flotation, gravity concentration, and leaching.

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 of the pits. A tailings storage facility (TSF) was designed in this valley to contain 1,172 Mt of tailings based on a mill throughput of 120,000 t/d for the 27-year mine life. During the life-of-mine (LOM), tailings will be deposited within the valley and retained by four cross-valley tailings dams to be constructed over the mine life. A 177 m high starter dam will be constructed initially at the south end of the impoundment and raised in stages to an approximate height of 300 m above centreline. The three 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 544 Mt of waste will be stripped at the Snowfield pit over the LOM, and hauled to two potential waste dumps. Approximately 645 Mt of waste will be stripped at the Brucejack pits over the LOM and handled as described in Section 18.3.3. 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-Brucejack 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 pits will be piped through the 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 pits 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-Brucejack site will be accessible by a planned permanent road constructed between a junction with Highway 37 and the plant site. Highway 37, a major road access to northern BC, passes approximately 24 km from the Snowfield-Brucejack Project plant site (Figure 1.3).

The plant site is located 26 km east of the open pits area. Twin tunnels constructed with crosscuts will connect the plant site and the mine sites. One of the tunnels will be used for conveying the crushed material from the mine sites 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 sites for the transport of the materials and workers.

At the Snowfield and Brucejack mine sites, two crushing facilities each housing two 60' x 89' gyratory crushers (fixed crushing station at the Snowfield site and semi-mobile crushing station at the Brucejack site) will be designed to crush the mineralization materials from the proposed mine.

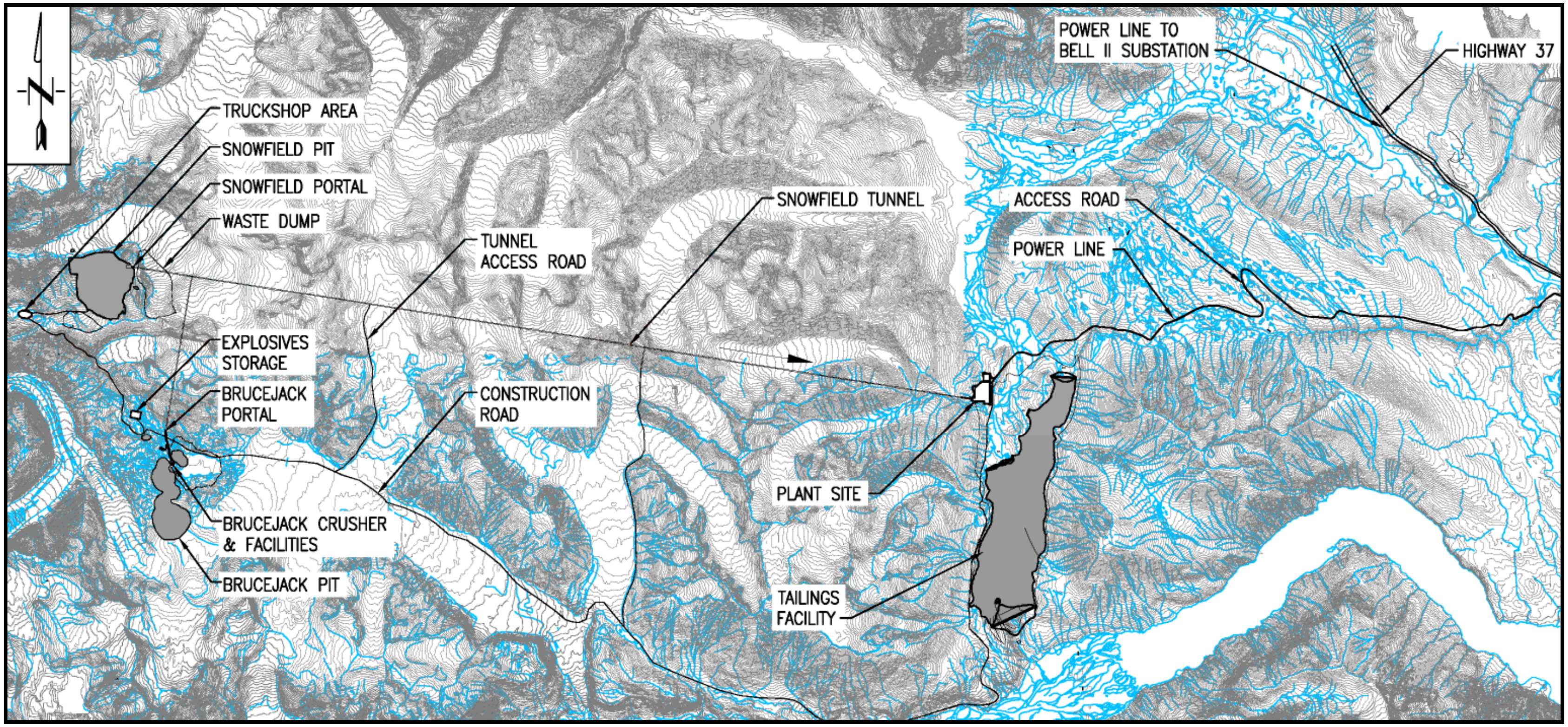
The plant site area will consist of the following facilities:

- 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 and gold recovery
- concentrate dewatering and handling

- maintenance building
- maintenance shop and warehouse
- water services.

The TSF is located approximately 5 km south of the mill site within the Scott Creek Valley.

Figure 1.3 Snowfield-Brucejack 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-Brucejack Project from the Bell II substation will be approximately 45 km long, and terminate at a distribution substation at the Snowfield-Brucejack 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 sites 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-Brucejack substation to step back up to 69 kV. This will be a suitable voltage to feed via cable through the tunnel to the pits, 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$3.465 B. This includes a contingency amount of US$454 M, which 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	713,543,403
Mill Area	583,760,234
Tailing Management, Reclaim Systems, Water Turbidity Control & Closure	473,247,267
Utilities	122,284,321
Site General	228,462,152
Temporary Facilities	93,130,187
Plant Mobile Equipment	7,471,367
Subtotal	**2,221,898,930**
Indirects	
Project Indirects	709,062,327
Contingencies	454,542,568
Owner's Costs	79,747,019
Subtotal	**1,243,351,913**
Total Capital Cost	**3,465,250,843**

1.15 OPERATING COST ESTIMATE

The operating cost for the project is estimated at C$10.20/t milled The estimate includes operating costs for mining, process, general and administration (G&A), water treatment, and surface services. Tailings operating costs are included in the sustaining capital costs for the project. A total of 617 personnel are projected for the operation, including 309 personnel for mining, 228 personnel for process, and 80 personnel for general management and surface services.

1.16 ECONOMIC EVALUATION

An economic evaluation of the Snowfield-Brucejack Project was prepared by Wardrop based on a pre-tax financial model. For the 27-year LOM and 1,172 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 12.4% internal rate of return (IRR)
- 5.3-year payback on US$3,465 M capital
- US$2.30 billion net present value (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
- rhenium – US$7,809/kg.

Metal revenues included in the Snowfield-Brucejack cash flow model are based on the average metal production, as presented in Table 1.4.

Table 1.4 Snowfield-Brucejack Project Metal Production

	Average Annual Production		Total Production	
Metal	**Years 1 to 8**	**LOM**	**Years 1 to 8**	**LOM**
Gold (000 oz)	960	700	7,679	18,910
Silver (000 oz)	7,855	4,162	62,838	112,364
Copper (000 lb)	39,531	44,582	316,245	1,203,715
Molybdenum (000 lb)	3,514	3,668	28,115	99,042
Rhenium (kg)	9,379	9,011	75,029	243,305

Sensitivity analyses were carried out on the following parameters:

- copper price
- gold price
- silver price
- molybdenum price
- rhenium 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 18.11 of this report. The project NPV (at 5% discount rate) is most sensitive to the exchange rate, gold price, and mill feed gold grade.

Similarly, the project IRR is most sensitive to the fixed exchange rate followed by mill feed 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 Section 18.5.

1.18 RECOMMENDATIONS AND CONCLUSIONS

Based on the results of the 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.

2.0 INTRODUCTION

Silver Standard retained Wardrop to conduct a PA on the Snowfield-Brucejack Project. This technical report has been prepared in general accordance with the guidelines provided in NI 43-101 “Standards of Disclosure for Mineral Projects”. Wardrop compiled this report based on work by the following independent consultants:

- P&E
- AMC
- Rescan
- BGC.

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

Table 2.1 Summary of QPs

Report Section	Company	QP
1.0 – Summary	All	sign 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, Analyses, 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 Estimate	P&E	Fred Brown, CPG Pr.Sci.Nat.

table continues…

Report Section	Company	QP
18.0 – Other Relevant Data and Information		
18.1: Mining	AMC	Greg Hollett, P.Eng.
18.2: Infrastructure	Wardrop	John Huang, P.Eng.
18.2.6: Roads and Access	Wardrop	Mike Boyle, P.Eng.
18.2.7: Site Roads/Earthworks	Wardrop	Mike Boyle, P.Eng.
18.2.8: Tunnel Development	Wardrop	Dan Sweeney, P.Eng.
18.2.11: Power/Electrical	Wardrop	Malcolm Cameron, P.Eng.
18.3: Waste and Water Management	BGC	Lori-Ann Wilchek, P.Eng.
18.3.3: Snowfield Waste Dump and Open Pit Water Management	BGC	Warren Newcomen, P.Eng.
18.3.4: Brucejack Waste Dump and Open Pit Water Management	BGC	Warren Newcomen, P.Eng.
18.4: Preliminary Geotechnical Design	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	Paul Greisman, P.Eng.
18.8: Taxes	Silver Standard	N/A
18.9: Capital Cost Estimate	Wardrop	Hassan Ghaffari, P.Eng.
All Costs Relating to Environmental	Rescan	Paul Greisman, P.Eng.
18.10: Operating Cost Estimate	Wardrop	John Huang, P.Eng.
All Costs Relating to Environmental	Rescan	Paul Greisman, P.Eng.
18.10.2: Mining Operating Cost	Wardrop	Nory Narciso, P.Eng.
18.11: Financial Analysis	Wardrop	Scott Cowie, MAusIMM
19.0 – Conclusions and Recommendations	All	sign off by section
20.0 – References	All	N/A

3.0 RELIANCE ON OTHER EXPERTS

The authors wish to make clear that they are QPs 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.

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 SNOWFIELD PROPERTY 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 BC 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; 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	1,267.43	104B	Jan. 31, 2017	Good	0777666 BC Ltd.
594266	Placer	428.39	104B	Jan. 31, 2011	Good	0777666 BC Ltd.
594267	Placer	446.39	104B	Jan. 31, 2011	Good	0777666 BC Ltd.

4.2 SNOWFIELD PROPERTY 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.

4.3 BRUCEJACK PROPERTY DESCRIPTION AND TENURE

The Brucejack property consists of six mineral claims totalling 3,199.28 ha in area (Table 4.2 and Figure 4.1) and all claims are in good standing until January 31, 2017. In 2001, Silver Standard purchased Black Hawk's 40% interest in the Brucejack property resulting in 100% interest in the Brucejack property. As part of the transaction, Silver Standard agreed to pay Black Hawk a NSR royalty of 1.2% on production in excess of the then current resources of silver and gold already outlined on the Brucejack property. Claim ownership is registered to 0777666 BC Ltd., a wholly-owned subsidiary of Silver Standard.

Information relating to tenure was verified by means of the public information available through the Mineral Titles Branch of the BC Ministry of Energy, Mines, and Petroleum Resources MTO land tenure database. The six above-mentioned mineral claims were converted from 28 older legacy claims to BC's new MTO system in 2005. P&E has relied upon this public information, as well as information from Silver Standard, and has not undertaken an independent verification of title and ownership of the Brucejack property claims.

A legal land survey of the claims has not been undertaken.

Table 4.2 Brucejack Property Mineral Claims

Tenure No.	Tenure Type	Map No.	Owner	Silver Standard Interest	Status	In Good Standing To	Area (ha)
509223	Mineral	104B	0777666 BC Ltd.	100%	Good	Jan. 31, 2017	428.62
509397	Mineral	104B	0777666 BC Ltd.	100%	Good	Jan. 31, 2017	375.15
509400	Mineral	104B	0777666 BC Ltd.	100%	Good	Jan. 31, 2017	178.63
509463	Mineral	104B	0777666 BC Ltd.	100%	Good	Jan. 31, 2017	482.57
509464	Mineral	104B	0777666 BC Ltd.	100%	Good	Jan. 31, 2017	1,144.53
509506	Mineral	104B	0777666 BC Ltd.	100%	Good	Jan. 31, 2017	589.78
Total							**3,199.28**

There are no annual holding costs for any of the six mineral claims at this time.

Figure 4.1 illustrates the six Brucejack property claims in relation to the Snowfield property, which adjoins the Brucejack property to the north. Both the Brucejack and the Snowfield properties are 100% owned by Silver Standard. The current report is focused on the six highest priority mineralized zones of the Brucejack property.

The majority of the Brucejack property falls within the boundaries of the Cassiar-Iskut-Stikine Land and Resource Management Plan (LRMP) area, with only a minor south-eastern segment of Mineral Claim No. 509506 falling outside this area. All claims located within the boundaries of the LRMP are considered as areas of

General Management Direction, with none of the claims falling inside any Protected or Special Management Areas.

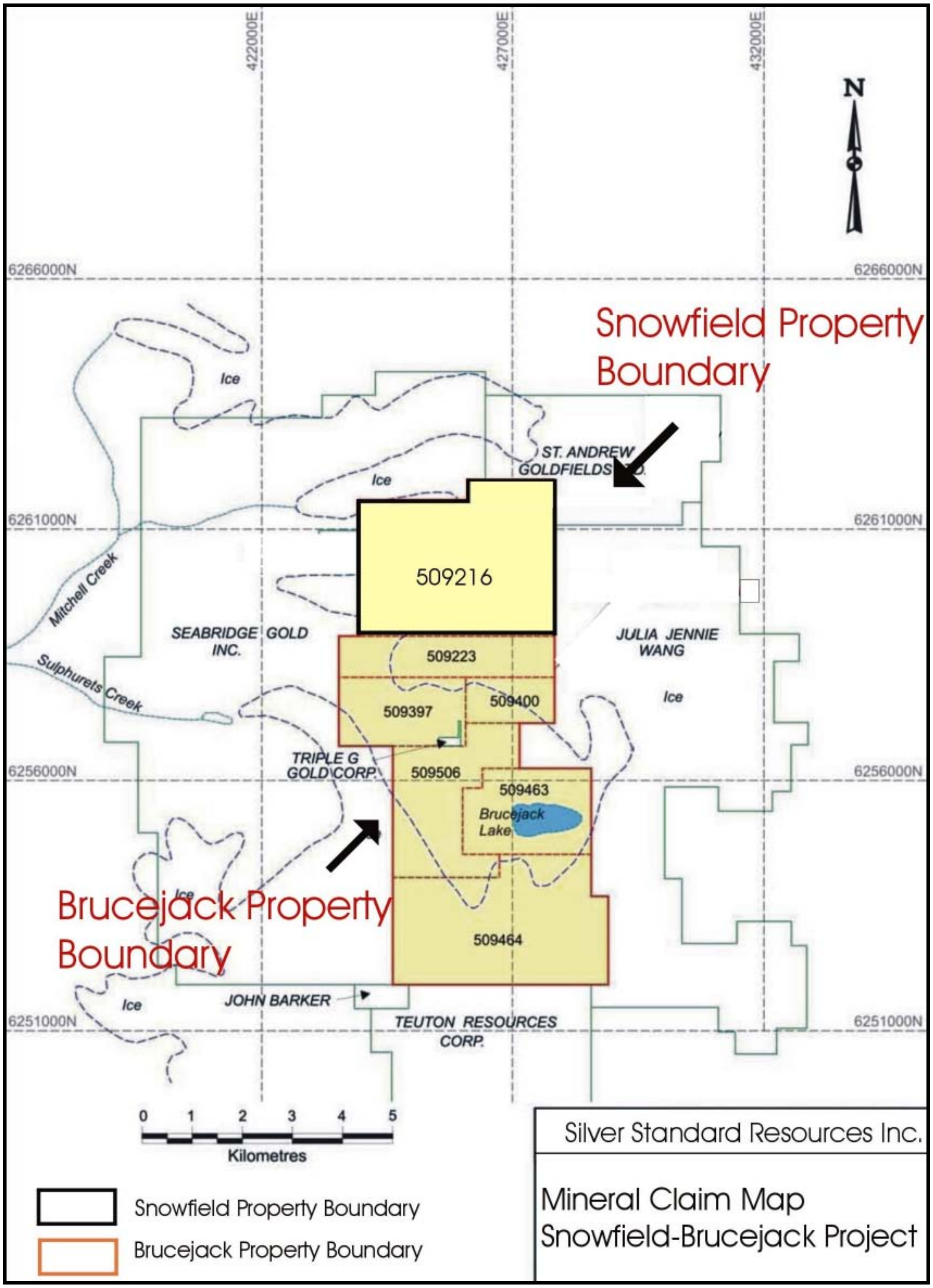
At present the land claims in the area are in review and subject to ongoing discussions between various native groups and the Government of BC. Silver Standard has specified that it maintains good relationships with all native groups.

4.4 Brucejack Property Location

The Brucejack property is situated at an approximate latitude of 56°28'20″N by longitude 130°11'31″W, a position approximately 950 km northwest of Vancouver, 70 km north-northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine. The Brucejack property coordinates used in this report are located relative to the NAD83 UTM coordinate system.

There are six separate mineralized zones within the Brucejack property that are the focus of this report, as summarized in Table 4.2 and shown in Figure 4.2.

Figure 4.1 Snowfield-Brucejack Property Mineral Claim Map

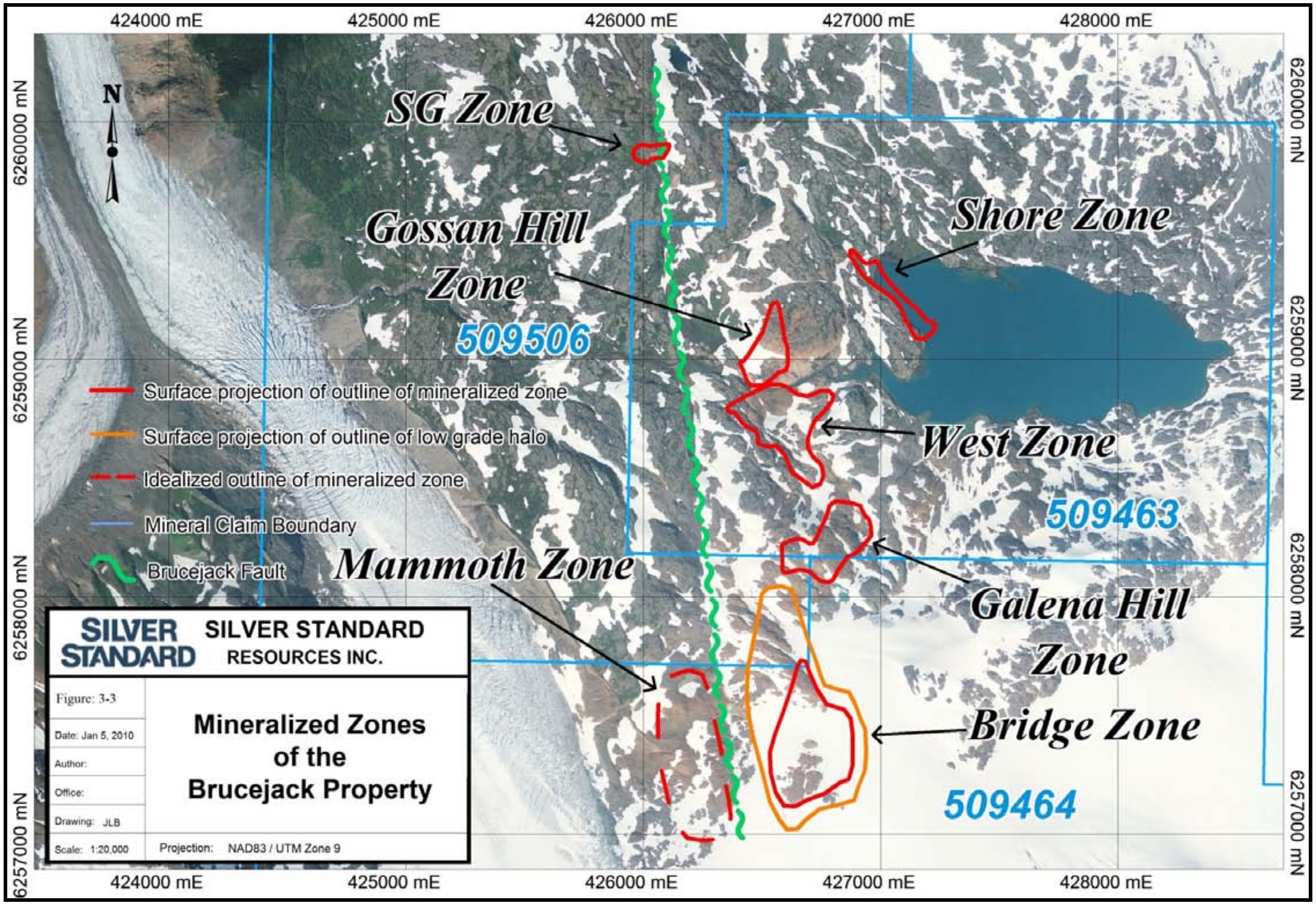


Note: modified after Blanchflower, 2008.

Table 4.3 Brucejack Property Mineralized Zones

Zone	Mineralization Type	Location	Historical Names
West	Gold-Silver	Located entirely within Mineral Claim No. 509463	Previously termed "Sulphurets" and has also been referred to as the "West Brucejack Zone"
Bridge	Gold-Silver	Overlaps Mineral Claim No. 509506 and 509464	Incorporates an older zone previously reported as the "Electrum Zone" (forming the northern extent of the Bridge Zone), as well as a relatively newer zone that forms the southern extension
Galena Hill	Gold-Silver	Overlaps Mineral Claim No. 509506, 509463, and 509464	N/A
Shore	Gold-Silver	Located entirely within Mineral Claim No. 509463	N/A
SG	Gold-Silver	Located entirely within Mineral Claim No. 509506	Incorporates an older zone previously reported as the “Maddux Zone”
Gossan Hill	Gold-Silver	Located entirely within the central western section of Mineral Claim No. 509463	Incorporates a previously separate zone historically known as the "Tommyknocker Zone", which forms the southern-most portion of this zone

Figure 4.2 Mineralized Zones of the Brucejack Property



Source: P&E.

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

5.1 LOCATION AND ACCESS

The Snowfield-Brucejack Project 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 Project area 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 Project, such as at the gossanous Snowfield deposit, the relief is relatively low to moderate.

The Project 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 Project. 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-Brucejack Project, 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.

5.3 Infrastructure and Local Resources

The Snowfield-Brucejack Project lies immediately east of Seabridge Gold Inc.'s (Seabridge's) KSM Project and would likely be influenced by future access plans for that area, as outlined within the Preliminary Economic Assessment (PEA) study by Seabridge (McElhanney Consulting Services Ltd. [McElhanney], 2008; Wardrop, 2009a). The proposed development activities for the KSM Project call for a combined 23 km tunnel for slurry delivery to the processing plant site located at the upper reaches of the Tiegen Creek Valley and a 14 km gravel road that would allow material to be trucked to the paved Cassiar highway (Highway 37). In addition, road access to Mitchell Creek itself would be provided by a 34 km continuation of the Eskay Creek Mine access road (Figure 5.1).

There are no local resources other than abundant water for any drilling work. The nearest infrastructure is the town of Stewart, approximately 65 km to the south, which has a minimum of supplies and personnel. The towns of Terrace and Smithers are also located in the same general region as the Project. Both are directly accessible by daily air service from Vancouver.

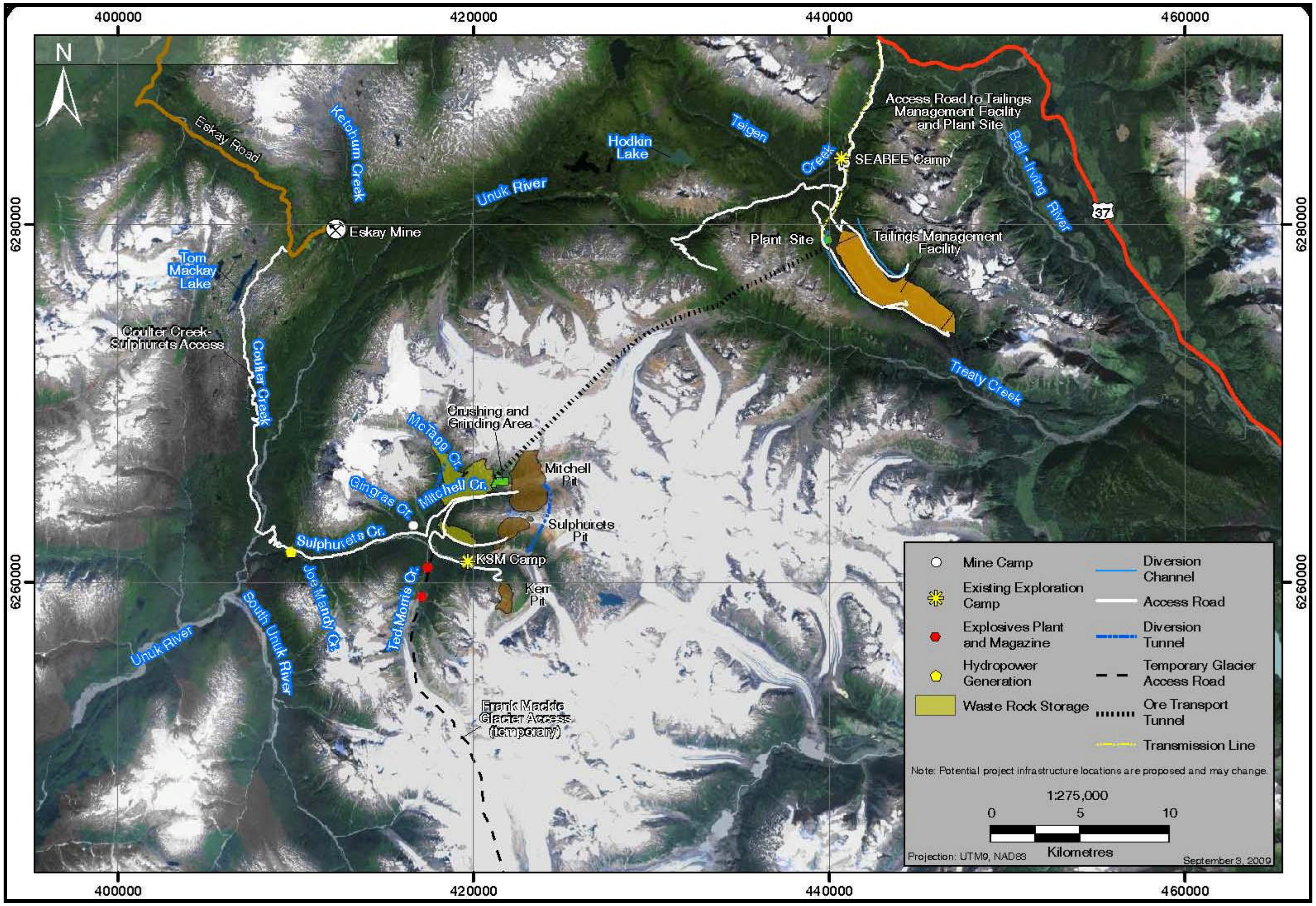
The nearest railway is the Canadian National Railway (CNR) Yellowhead route, which is located approximately 220 km to the southeast. This line runs east-west and terminates at the deep water port of Prince Rupert on the west coast of BC.

The most northerly ice-free shipping port in North America is accessible to store and ship concentrates. Such material is currently being shipped from the Eskay Creek and Huckleberry mines via this terminal.

A proposal to have a high voltage power line run parallel with existing lines along Highway 37 is currently under review (www.highway37.com).

The initial plan calls for the new 287-kV line that would extend from the community of Terrace to the beginning of the Galore Creek access road at Bob Quinn Lake providing access for the project to the BC Hydro electric grid (Figure 5.2). The final capacity of this transmission line has yet to be determined and may be increased due to projected demand.

Figure 5.1 KSM Project Planned Road Access



Note: after Seabridge; Wardrop, 2009a.

Figure 5.2 Proposed High Voltage Northwest Transmission Line

Map by Northwest Powerline Coalition
Map for reference only. Scale 1:4,200,000

Source: www.highway37.com

6.0 HISTORY

The Snowfield-Brucejack Project and the surrounding region have a history rich in exploration for precious and base metals dating back to the late 1800s. This section describes the mineral exploration, including the historical drilling carried out prior to Silver Standard's acquisition of the separate Brucejack and Snowfield properties, within the Brucejack portion of the Project itself and the surrounding region. The historical data have been summarized mostly from various Assessment Reports available through the BC Ministry of Energy, Mines and Petroleum Resources.

6.1 SNOWFIELD PROPERTY

6.1.1 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. There were 21 drill holes tested at the Snowfield zone, 6 drill holes tested the nearby Coffeepot zone situated immediately west of the Snowfield zone, and 1 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.

6.2 BRUCEJACK PROPERTY

6.2.1 SUMMARY OF HISTORICAL EXPLORATION

The exploration history of the area dates back to the 1880s when placer gold was located at Sulphurets and Mitchell Creeks. Placer mining was intermittently undertaken throughout the early 1900s and remained the main focus of prospecting until the mid-1930s.

In 1935, prospectors discovered Cu-Mo mineralization on the Sulphurets property in the vicinity of the Main Copper zone, approximately 6 km northwest of Brucejack Lake; however, these claims were not staked until 1960.

From 1935 to 1959, the area was relatively inactive with respect to prospecting; however, it was intermittently evaluated by a number of different parties and several small Cu and Au-Ag occurrences were made in the Sulphurets-Mitchell Creek area.

In 1960, Granduc and Alaskan prospectors staked the main claim group covering the known Cu and Au-Ag occurrences, which collectively became known as the Sulphurets property, starting the era of modern exploration, outlined as follows:

- **1960-1979** – Granduc continued exploration, conducting further geological mapping, lithogeochemical sampling, trenching, and diamond drilling on known base and precious metal targets north and northwest of Brucejack Lake resulting in the discovery of Au-Ag mineralization in the Hanging Glacier area and Mo on the south side of Mitchell.
- **1980** – Esso optioned the property from Granduc and subsequently completed an extensive program consisting of mapping, trenching, and geochemical sampling that resulted in the discovery of several showings including the Snowfields, Shore, West, and Galena zones. Au was discovered on the peninsula at Brucejack Lake near the Shore Zone.
- **1982-1983** – Exploration was confined to Au and Ag-bearing vein systems in the Brucejack Lake area at the southern end of the property from 1982 to 1983. Drilling was concentrated in 12 Ag and Au-bearing structures including the Near Shore and West zones, located 800 m apart near Brucejack Lake. Drilling commenced on the Shore Zone.
- **1983** – Esso continued work on the property and (in 1984) outlined a deposit on the west Brucejack Zone.
- **1985** – Esso dropped the option on the Sulphurets property.
- **1985** – The property was optioned by Newhawk and Lacana Mining Corp. (Lacana) from Granduc under a three-way joint venture (the Newcana JV). Since then, the Newcana JV has completed work on the Snowfields, Mitchell, Golden Marmot, Sulphurets Gold, and Main Copper zones, along with lesser known targets.
- **1986-1991** – Between 1986 and 1991, the Newcana JV spent approximately $21 M developing the West Zone and other smaller precious metal veins on what would later become the Bruceside property.
- **1991-1992** – Newhawk officially subdivided the Suphurets claim group into the Sulphside and Bruceside properties and optioned the Sulphside property (including Sulphurets and Mitchell Zones) to Placer Dome Inc. (Placer Dome). Throughout the period from 1991 to 1994, joint venture exploration continued on the Sulphurets-Bruceside property including property-wide scaled trenching, mapping, airborne surveys, and surface drilling evaluating various surface targets including the Shore, Gossan Hill, Galena Hill, Maddux, and SG zones. Newhawk purchased Granduc's interest in the Snowfield property in early 1992.
- **1991** – Six holes were drilled at the Shore Zone, totalling 1,200 m, to test its continuity and to determine its relationship to the West and R-8 zones. Results varied from 37 g/t Au over 1.5 m to 13 g/t Au over 4.9 m (www.infomine.com).

- **1994** – Exploration in the Brucejack area consisted of detailed mapping and sampling in the vicinity of the Gossan Hill Zone, and 7,351.5 m of diamond drilling (over 20 holes), primarily on the West, R8, Shore, and Gossan Hill zones. Mapping, trenching, and drilling of the highest priority targets were conducted on 10 of the best deposits (including the West Zone).
- **1996** – Granduc merged with Black Hawk to form the new Black Hawk Mining Inc.
- **1997-1998** – No exploration or development work was carried out in the Brucejack property (Budinski et al., 2001).
- **1999** – Silver Standard acquired Newhawk and, with it, Newhawk's 60% interest and control of the Brucejack property (www.infomine.com).
- **2001** – Silver Standard entered into an agreement with Black Hawk whereby Silver Standard acquired Black Hawk's 40% direct interest in the Brucejack property, resulting in 100% interest in the property.
- **1999-2008** – No exploration or development work was carried out in the Brucejack property during the period from 1999 to 2008.

7.0 GEOLOGICAL SETTING

The following description of the regional and local geology of the Snowfield-Brucejack Project 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.

Because the deposits associated with Snowfield and Brucejack differ from one another, the regional geology of the Snowfield-Brucejack Project is presented as one; however, the local geology, structure and alteration are treated separately for the Snowfield and Brucejack properties.

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 BC Geological Survey, the University of BC, 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.

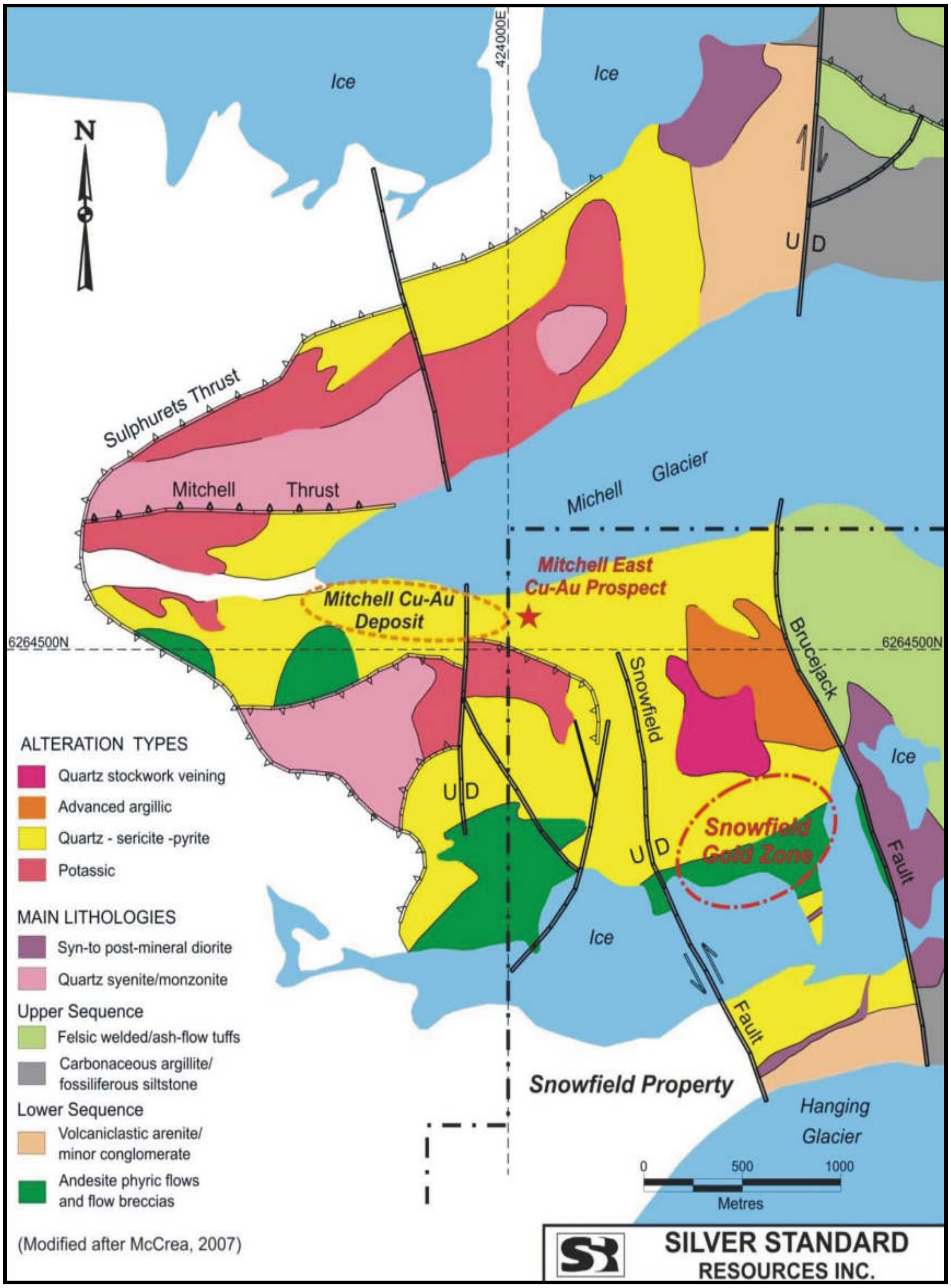
7.1 REGIONAL GEOLOGY

The Snowfield-Brucejack Project 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



Source: after Blanchflower, 2008.

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.

Regional geologic mapping has been completed by the Geological Survey of Canada, the BC Ministry of Energy, Mines and Resources, and the Mineral Deposits Research Unit (MDRU) at the University of BC. A regional geology map is depicted in Figure 7.3.

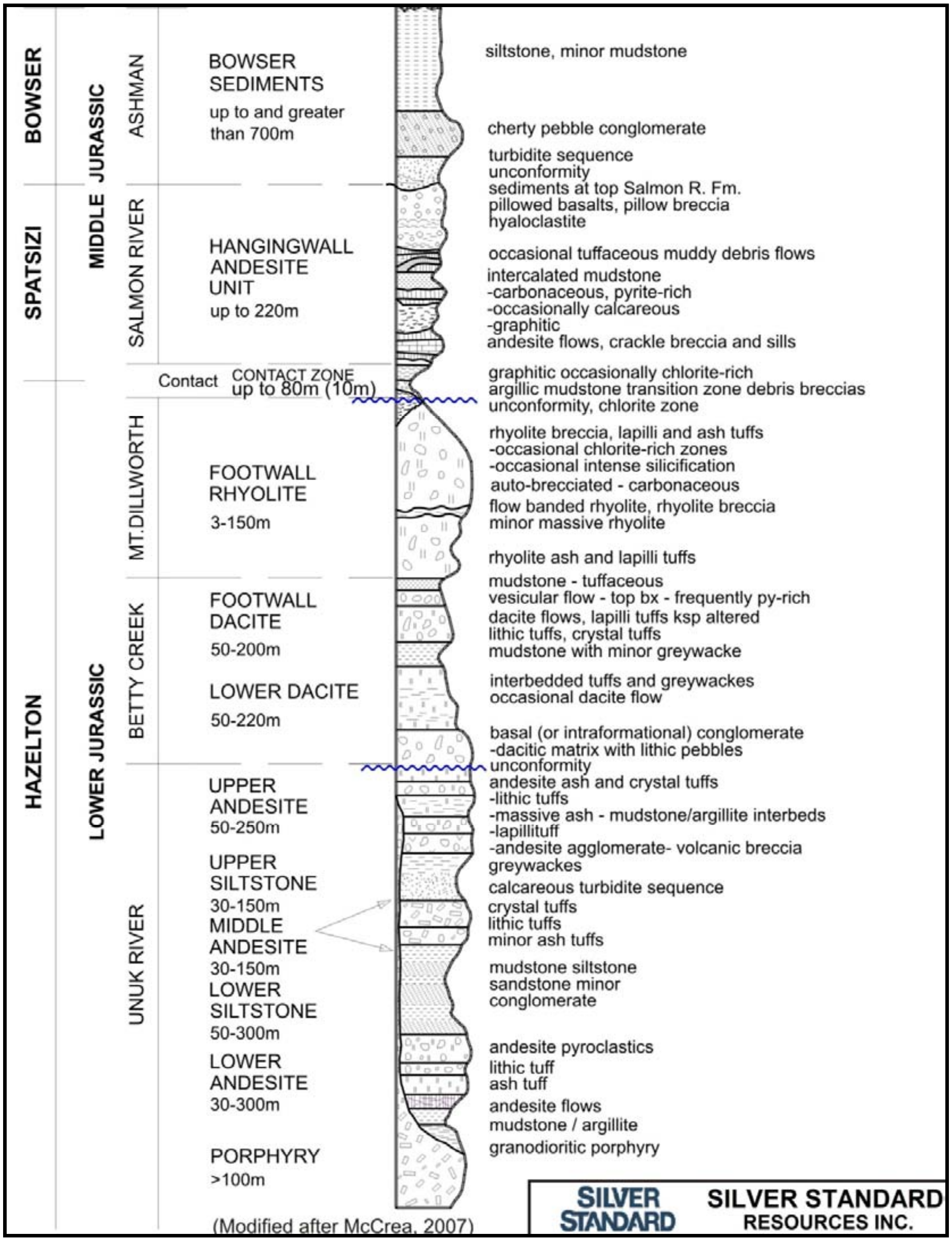
The regional stratigraphic assemblage as originally compiled by Kirkham (1963) and later modified by Britton and Alldrick (1988), Alldrick and Britton (1991), McCrea (2007) and Blanchflower (2008), is illustrated in Figure 7.2 and has been summarized in Table 7.1.

Table 7.1 Summary of Regional Stratigraphy – Oldest to Youngest

Formation	Stage (Triassic – Jurassic)	Description
Lower Unuk River	Norian to Hettangian	Alternating siltstones and conglomerates
Upper Unuk River	Hettangian to Pliensbachian	Alternating intermediate volcanic rocks and siltstones
Betty Creek	Pliensbachian to Toarcian	Alternating conglomerates, sandstones, intermediate and mafic volcanic rocks
Mount Dilworth	Toarcian	Felsic pyroclastic rocks and flows, including tuffaceous rocks ranging from dust tuff to tuff breccias and localized welded ash tuffs
Salmon River & Bowser	Toarcian to Bajocian	Alternating siltstones and sandstones

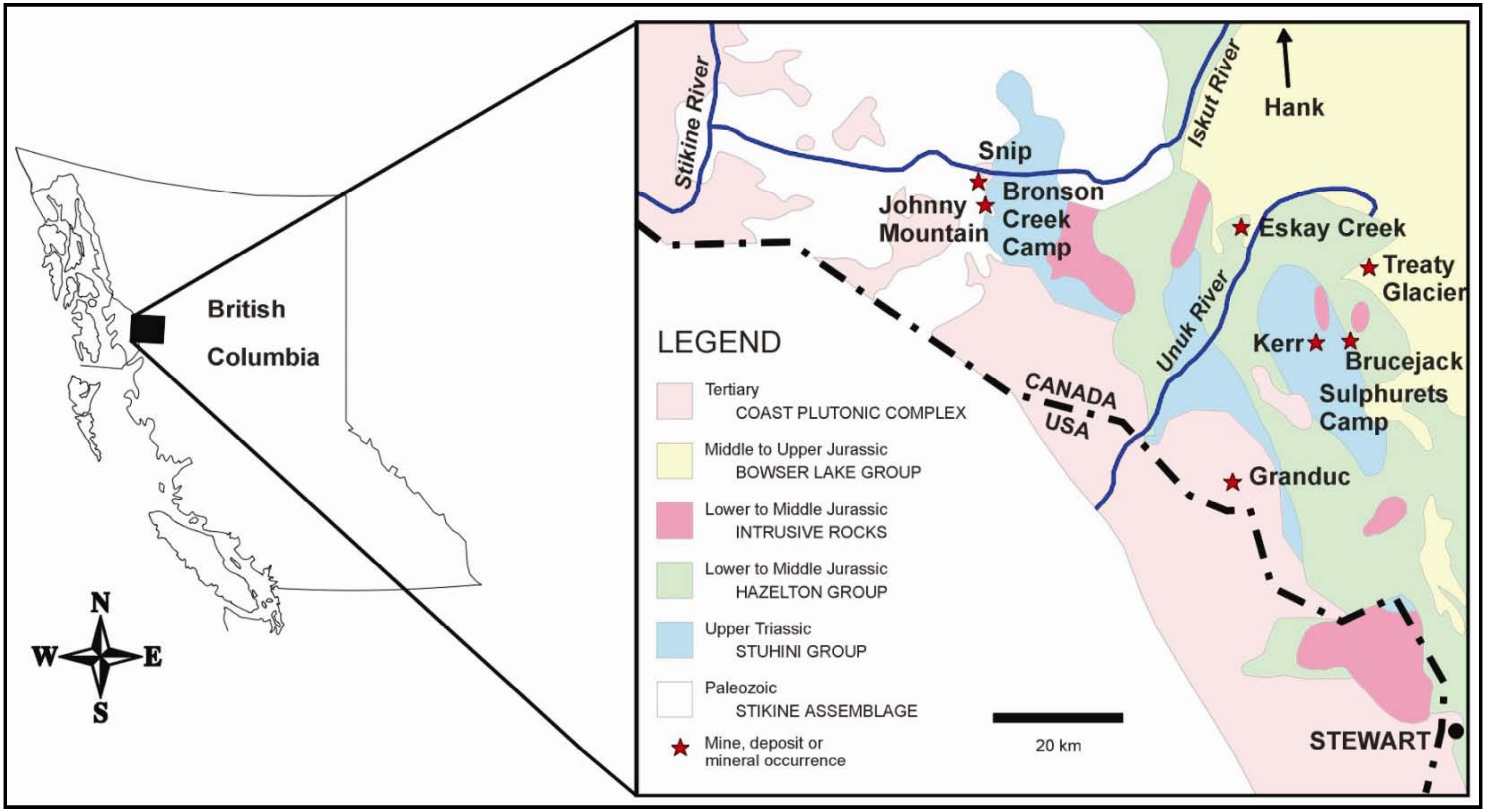
Source: after Blanchflower, 2008.

Figure 7.2 Regional Stratigraphic Column



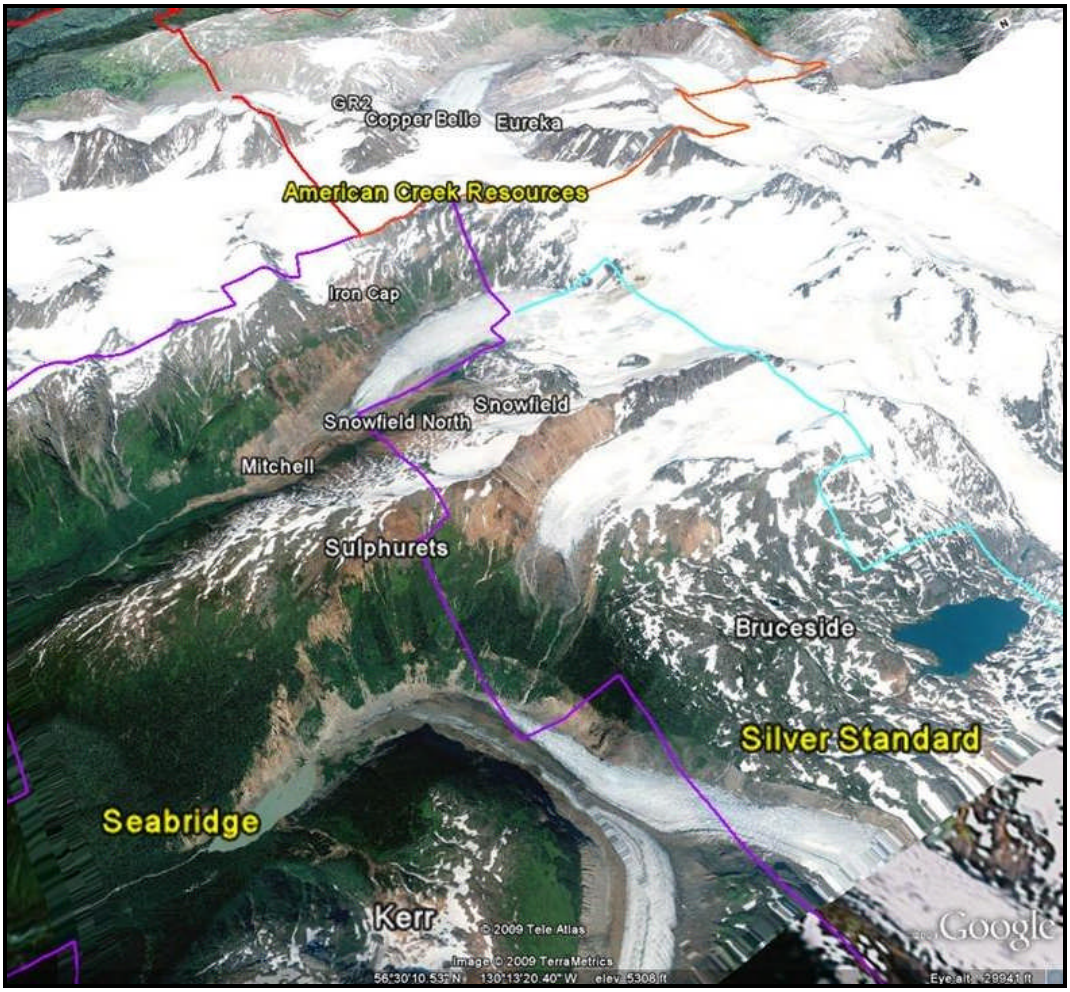
Source: after Blanchflower, 2008.

Figure 7.3 Map of Regional Geology – Simplified Geology and Location Map of the Iskut River Region



Source: Silver Standard.

Figure 7.4 Regional Trendline Showing the Seabridge and American Creek Resources Ltd. Properties in Relation to the Snowfield-Brucejack Project



Note: Figure 7.4 refers to the Bruceside property, which is a historical term for the Brucejack Property. Source: http://www.americancreek.com/images/trendline%208.jpg.

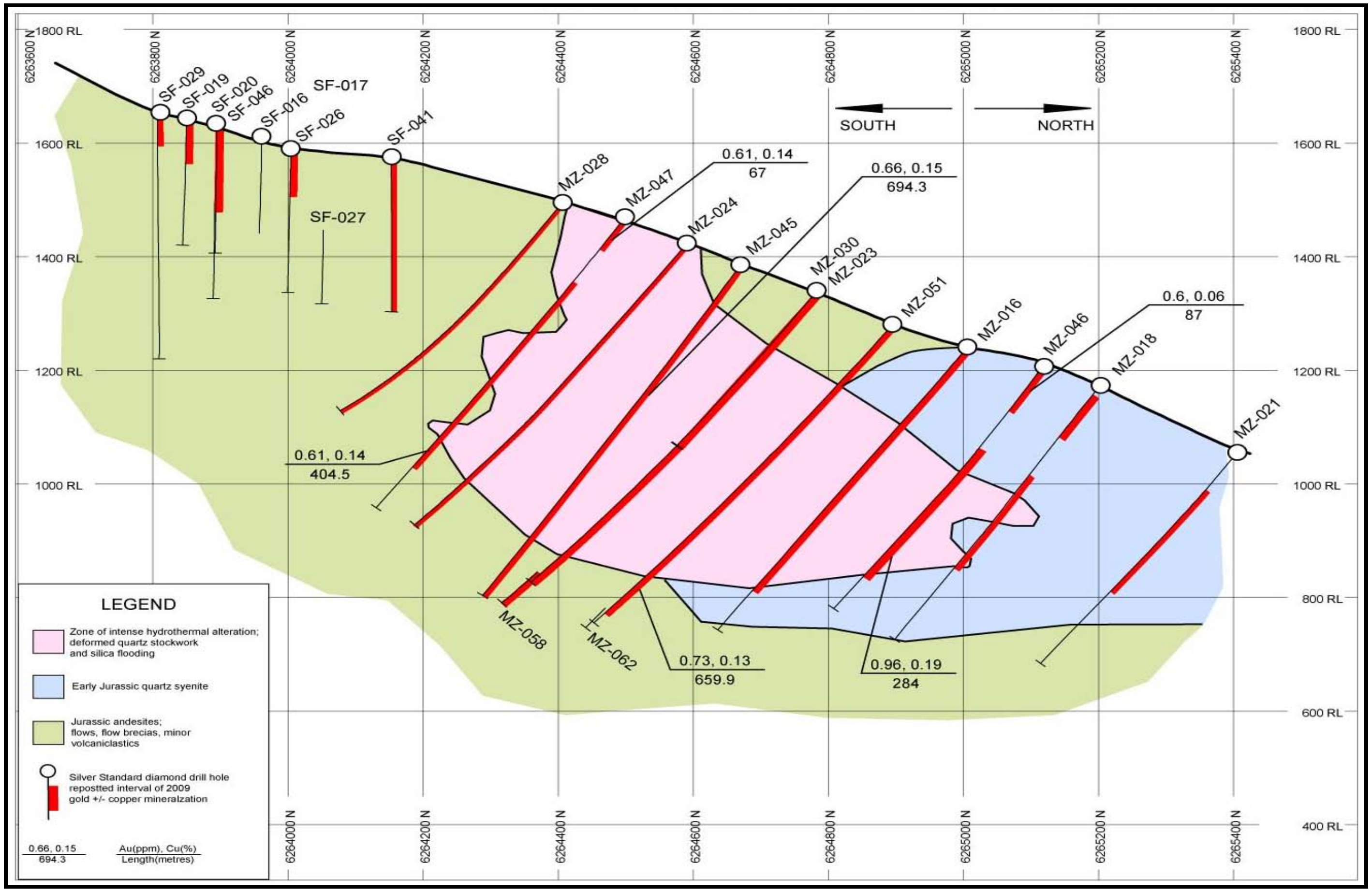
7.2 Geology, Structure, and Alteration of the Snowfield Deposit

The Snowfield deposit (Figure 7.5) 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 Project have been subjected to a lower greenschist facies grade of metamorphism with subsequent pervasive hydrothermal alteration, making the identification of protoliths difficult. 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 7.5 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.

7.3 Geology, Alteration, and Structure of the Brucejack Property

The following description of the geology of the Brucejack property was provided by Mr. Ron Burk, Chief Geologist of Silver Standard, in the form of an internal company report, dated December 9, 2009.

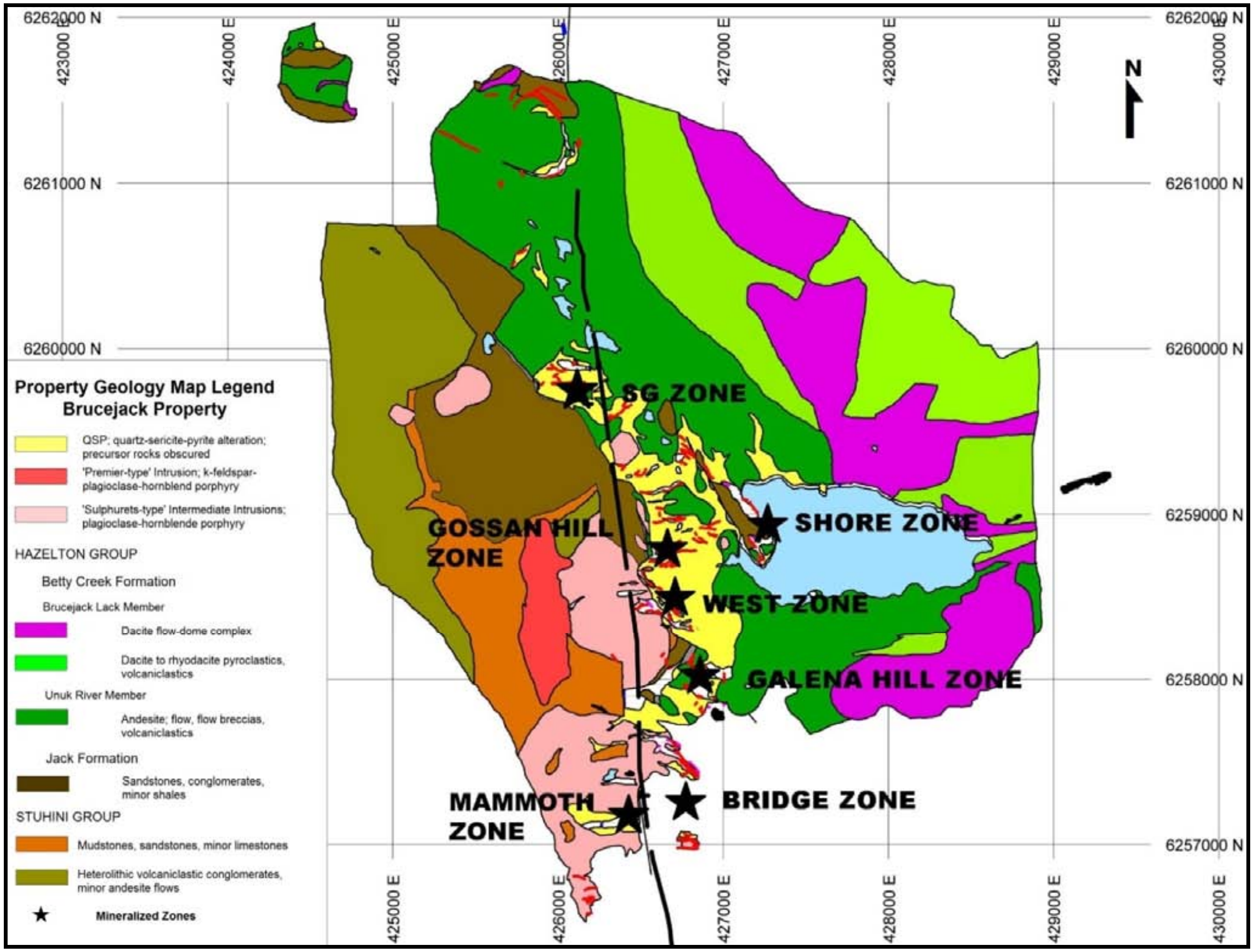
Published descriptions of the regional geology of the Sulphurets Creek-Brucejack Lake area have been presented by the Geological Survey of Canada (Henderson et al., 1992; Kirkham, 1991; Anderson, 1989), geologists working for the BC government (Britten and Alldrick, 1988; Alldrick et al., 1987; Grove, 1986) and by the MDRU at the University of BC (Lewis et al., 2001; Lewis, 2001). This body of work shows that the Brucejack property is underlain by Upper Triassic volcaniclastic and epiclastic sedimentary rocks of the Stuhini Group and Lower to Middle Jurassic volcanic, volcaniclastic, and sedimentary rocks of the Hazelton Group.

Since the Brucejack property occurs within the eastern limb of the McTagg anticlinorium, the stratigraphic sequences recognized on the Snowfield-Brucejack Project overall become younger to the east (Figure 7.6). The oldest rocks, found at lower elevations immediately east of the Sulphurets glacier, consist of heterolithic volcaniclastic conglomerate that is conformably overlain by a sequence of interbedded mudstone, sandstone, and thin limestone units of the Stuhini Group. An angular unconformity marks the contact between the Stuhini Group sedimentary rocks and medium- to coarse-grained sandstones of the Jack Formation, which is the basal formation of the Hazelton Group roughly and is dated at about 196 Ma.

Open folding and probable thrust faulting has also placed a wedge of Jack Formation sandstones and conglomerates at the western end of Brucejack Lake where these rocks are well exposed on a peninsula known as Windy Point. Using the revised Hazelton Group stratigraphy presented in MDRU's Special Publication Number 1 (Lewis et al., 2001), the Jack Formation sedimentary rocks are overlain by a 10 to 50 m-thick unit of mudstone/argillite and cherty argillite that belongs to the Unuk River Member of the Betty Creek Formation. This argillaceous unit is exposed along the southwest side of the West Zone deposit of shear-hosted, Au-Ag quartz veins and stockwork and has been traced southwards through the western part of the Galena Hill Au-Ag prospect.

Overlying the argillite unit is a greater than 500 m-thick package of hornblende and plagioclase-phyric andesitic flows, flow breccias and intermediate tuffaceous rocks intercalated with volcaniclastic conglomerates, sandstones and siltstones. These rocks form the bulk of the Unuk River Member in the Brucejack Property and outcrop extensively within a northwest-trending belt that passes beneath Brucejack Lake.

Figure 7.6 Brucejack Property Geology Map Showing Simplified Geology and Mineralized Zones



Source: Silver Standard.

The andesites of the Unuk River Member are the most important host rocks to Au- and Ag-bearing quartz veins discovered in the Brucejack area and have been affected by widespread hydrothermal alteration, mainly quartz-sericite-pyrite (e.g. Gossan Hill, Galena Hill). U-Pb geochronology and biochronology done by MDRU geoscientists have determined the age of the Unuk River Member volcanics to be in the range of 196 to 194 Ma.

Still higher in the Hazelton Group stratigraphy is a thick sequence of mainly dacitic pyroclastic rocks (tuff-breccia, lapilli tuff, crystal-lithic tuff, minor ash tuff) and flows with thin argillite interbeds that are well exposed on the mountainside north of Brucejack Lake. Based on MDRU's studies and mapping, this predominantly felsic to intermediate volcanic package has been assigned to the Brucejack Lake Member of the Betty Creek Formation. (Prior to MDRU's project in the Iskut River region, these rocks were mapped as belonging to the Betty Creek Formation).

A possible vent area for the tuffs and flows is a flow-dome complex identified just south of the east end of Brucejack Lake (Macdonald, 2001). Here, well developed subvertical flow-banding can be observed along with megacrystic flow-banded dacite, autobrecciated dacite and clast-supported blocky breccia with a hematitic mudstone matrix. Two U-Pb age dates have been obtained from flow-banded dacite and these show the flow-dome was emplaced 185.7 Ma. Several other U-Pb age dates obtained during the MDRU Iskut River Project for rocks assigned to the Brucejack Lake Member indicate that the episode of intermediate to felsic volcanism in the Hazelton Group spanned 8 to 10 million years.

Supracrustal rock units younger than the Brucejack Lake Member have not been reported from the Brucejack property, although they could exist at the top of Mount John Walker on the north side of Brucejack Lake. In the area of the Eskay Creek Au-Ag mine, the youngest member of the Betty Creek Formation is the Treaty Creek Member which is a mixed sequence of sedimentary strata including sandstone, conglomerate, turbiditic siltstone, and limestone. More importantly, the high-grade exhalative Au-Ag sulphide-sulphosalt deposits are associated with or hosted by units belonging to the Salmon River Formation which directly overlies the Treaty Creek Member. To date, rhyolite flows and carbonaceous mudstones that characterize the Salmon River Formation have not been identified in the Brucejack property but should be explored for at the highest elevations of Mount John Walker.

Apart from the high-level, synvolcanic intrusive dacite of the flow-dome complex mapped southeast of Brucejack Lake, there are three types of intrusions recognized in the Brucejack property. The most common intrusive rock in the area consists of plagioclase- and hornblende-phyric to porphyritic rock of diorite to tonalite composition that forms two stocks found in the southern half of the claim group. Each intrusion has surface dimensions of roughly 700 m east-west by 700-1,000 m north-south. A number of smaller bodies of the same rock are scattered around these two main intrusions. These intrusions have been referred to as "Sulphurets-type" intrusions and are considered to be broadly coeval with the andesite volcanics of the Unuk River Member in the Hazelton Group.

A second type of intrusive rock forms an elongate body of about 700 m length, aligned north-south, that was emplaced along the western margin of one of the Sulphurets stocks. This intrusion is best described as potassium feldspar-plagioclase-hornblende porphyry and earlier workers have referred to it as a “two-feldspar” or “Premier-type” porphyry. Based on contact relationships, it would appear that this intrusion is younger than the Sulphurets-type intrusions. The youngest intrusive rocks observed consist of medium to dark green, fine-grained andesite to basaltic andesite dikes that are generally less than 2 m in thickness. These dikes tend to be north to northeast striking.

In terms of structural geology, the lithologies found at the Brucejack property display evidence of both ductile and brittle deformation. The oldest rocks, belonging to the Stuhini Group, are well exposed along the steep ravine of Brucejack Creek and are strongly folded with axial traces trending just west of north.

The overlying Jack Formation epiclastic units are less intensely folded, with an open syncline being the dominant fold to have affected these rocks. A second syncline defined by units of the Jack Formation lies further to the east, with its NNW-SSE axial trace passing through the West Zone Au-Ag deposit.

The Unuk River and Brucejack Lake Member lithologiesof the Hazelton Group predominate in the eastern half of the Snowfield-Brucejack Project and form a homoclinal rock package that dips moderately to steeply in an east to northeast direction. Penetrative fabrics are commonly developed in most lithologies. Rocks that appear to have experienced hydrothermal alteration prior to folding are generally the most intensely foliated. Shearing also appears to have occurred along structures that developed at relatively low angles to stratigraphic layering, with one example being the 140°-trending shear zone that hosts the mineralized quartz veins and stockworks of the West Zone deposit.

Post-dating the folds and the development of penetrative fabrics are numerous brittle-ductile faults with different strike orientations and variable displacements. These structures can be readily observed as lineaments in aerial photographs of the Snowfield-Brucejack Project. One of the most prominent of these late structures is the northerly trending Brucejack Fault which bisects the two main Sulphurets-type intrusions and continues for kilometres to the north crossing the entire project area. Mapping of contact displacements suggests that right-lateral movement of about 150 m has occurred along this major structure; an unknown but probably minor amount of vertical displacement has likely also occurred. Other well-defined lineaments/faults tend to strike northwest or, as seen on the southern slope of Mount John Walker, have north-easterly alignments.

8.0 DEPOSIT TYPES

8.1 INTRODUCTION

While deposits such as Snowfield, Kerr, and Mitchell are probably best described as gold-enriched copper porphyry systems, most (if not all) of the mineralization in the Brucejack property (West, Bridge, Galena Hill, Shore, SG, Gossan Hill, and Mammoth zones) has been classified as an epithermal Au-Ag-Cu, low-sulphidation deposit (UBC deposit model No. H04). It is possible that some of the mineralization also displays characteristics of intrusion related vein systems that fall within the Intermediate-Sulphidation epithermal subtype of Hedenquist et al. (2000).

A complete discussion of the Snowfield deposit model and its related characteristics is given by Armstrong et al. (2009). The following is a brief summary:

> 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 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.

8.2 PORPHYRY DEPOSITS – SNOWFIELD

8.2.1 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.

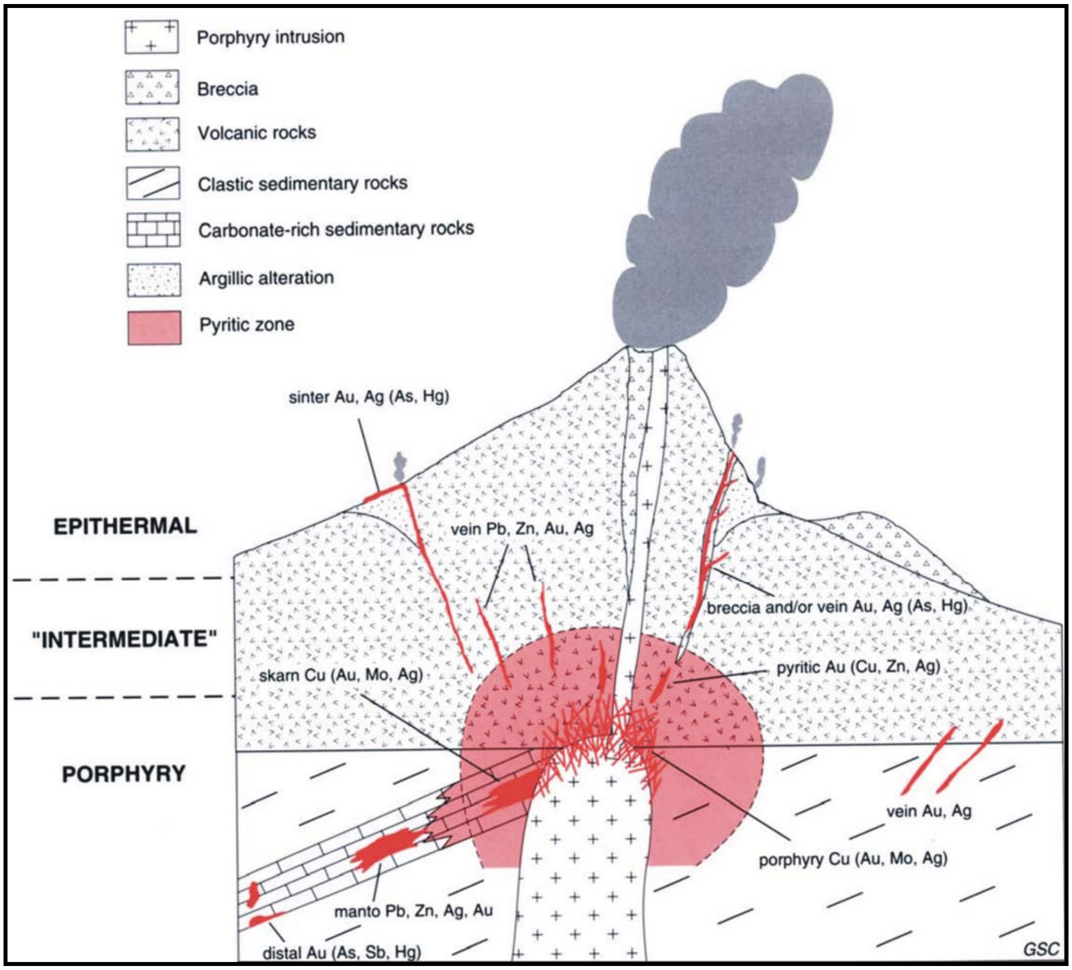
8.2.2 GENETIC MODEL

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.

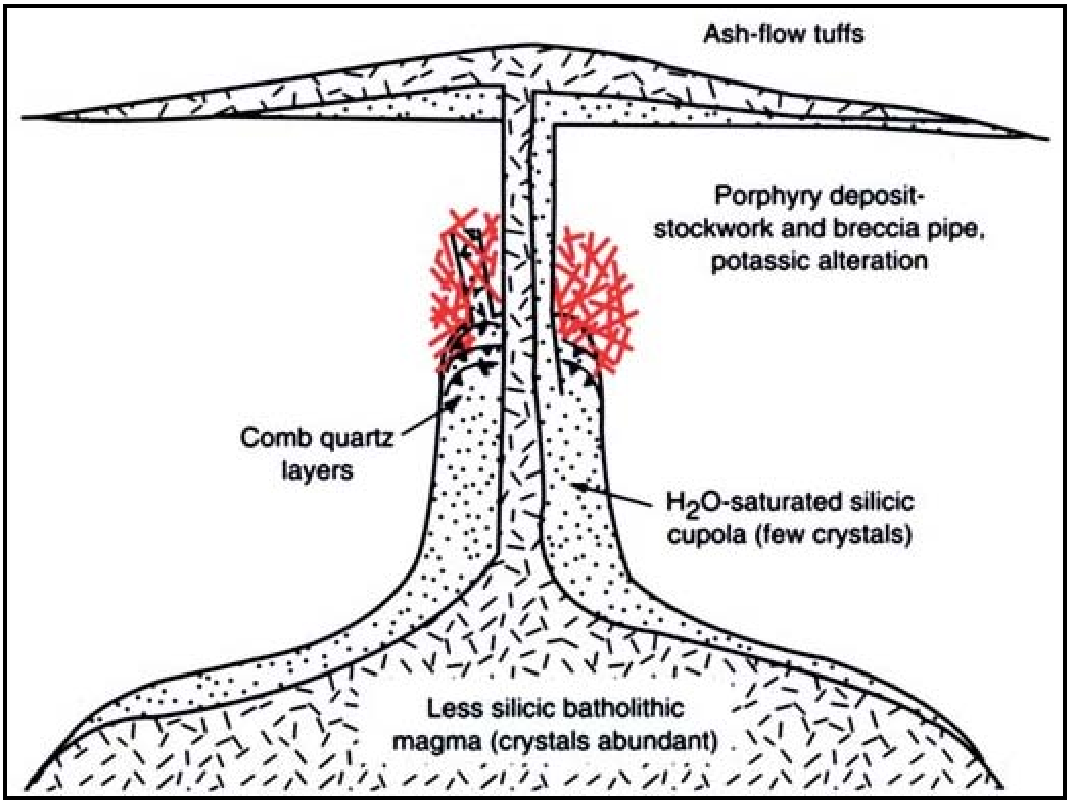
Some modification of the orthomagmatic model is likely required for at least some, if not most, porphyry deposits, in view of studies by Shannon et al. (1982), Carten et al. (1988a), and Kirkham and Sinclair (1988). These authors concluded that, in several deposits, the underlying genetically related 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).

8.3 EPITHERMAL AU-AG-CU, LOW-SULPHIDATION DEPOSITS: BRUCEJACK

A detailed description of epithermal mineralizing systems is provided by Taylor (2007) as his contribution to the most recent edition of the “Mineral Deposits of Canada”, Special Volume 5 published jointly by the Geological Association of Canada-Mineral Deposits Division and the Geological Survey of Canada. Much of the following material in this report section provides a brief overview of the subject that is synthesized from that publication.

Lindgren (1933) divided hydrothermal ore deposits, including those of gold and silver, into thermal types such as epithermal, mesothermal, and hypothermal. Lindgren fully recognized that his scheme also applied in a qualitative way to the depths in the Earth's crust at which various types of deposits form and it is this aspect of his classification scheme that has persisted to the present day. Thus, epithermal gold deposits are those for which there is evidence of a shallow crustal origin (less than 1 or 2 km), mesothermal deposits are those inferred to have formed at 1 to 3 km, and hypothermal deposits at 3 km to more than 5 km. The depth ranges implied for each of the three types are not firmly fixed but are guidelines that reflect variations in lithostatic pressure, fluid pressure, crustal temperature and metamorphic facies transitions, availability of meteoric fluids, and the vertical extent of brittle and ductile fields of deformation and seismicity (Poulsen, 1995).

Deep epithermal (or shallow mesothermal) veins ("transitional" deposits of Panteleyev, 1986) provide an example of the extended depth of formation currently included in the broad sense of epithermal. These transitional deposits are often referred to as intrusion-related vein deposits and occur in the Sulphurets, Mt. Washington, and Zeballos camps, all in BC (Anon., 1992 BC MINFILE; Margolis, 1993).

The Brucejack and Snowfield properties and surrounding properties in the Kerr-Sulphurets region host extensive mineralization and associated alteration systems that were undoubtedly developed as a result of hydrothermal activity focused on hypabyssal, Early Jurassic intermediate, porphyritic intrusions.

Amongst the Brucejack gold and silver deposits, the West Zone has received the most exploration work to-date and accordingly can be considered somewhat typical of the general style of mineralization displayed by the various mineralizing systems comprising the Brucejack property. Budinski et al. (2001) characterize the mineralization in the West Zone as a structurally controlled, complex vein/breccia system related to the Brucejack Fault lying to the immediate west. Like the other Brucejack property deposits it is considered to fit the epithermal high-grade, intermediate to low-sulphidation, Au-Ag model. Other examples in BC include the Blackdome and Silbak-Premier Mines.

8.4 EPITHERMAL GENETIC MODEL

8.4.1 INTRODUCTION

SIMPLIFIED DEFINITION

Epithermal deposits of Au (±Ag) are a type of lode gold deposit that comprises veins and disseminations near the Earth’s surface (≤1.5 km), in volcanic and volcaniclastic sedimentary rocks, sediments, and, in some cases, also in metamorphic rocks. The deposits may be found in association with hot springs and frequently occur at centres of young volcanism. The ores are dominated primarily by precious metals (Au, Ag) but some deposits may also contain variable amounts of base metals such as Cu, Pb, and Zn.

EPITHERMAL SUB-SYSTEMS

Epithermal Au deposits are distinguished on the basis of the sulphidation state of the sulphide mineralogy as belonging to one of three sub-types (Hedenquist et al., 2000):

- High sulphidation: previously called quartz-(kaolinite)-alunite, alunite-kaolinite, enargite-Au, or high-sulphur deposits (Ashley, 1982; Hedenquist, 1987; Bonham, 1988), these highly acidic deposits usually occur close to magmatic sources of heat and volatiles and form from acidic hydrothermal fluids containing magmatic S, C, and Cl.
- Intermediate sulphidation: some deposits with mostly low-sulphidation characteristics have sulphide ore mineral assemblages that represent a sulphidation state between that of high-sulphidation and low-sulphidation deposits. Such deposits tend to be more closely spatially associated with intrusions and Hedenquist et al. (2000) suggest the term ‘intermediate sulphidation’ for these deposits.
- Low sulphidation: previously called adularia-sericite, these low-sulphidation subtype deposits are thought to have a near-neutral pH as a result of being dominated by meteoric waters but containing some magmatic C and S.

8.4.2 *EPITHERMAL MINERALIZATION CHARACTERISTICS*

Lindgren (1922, 1933) suggested that degassing magmas are sources of many ore-forming constituents in epithermal Au deposits, and this supposition appears to be essentially correct for magmatic-hydrothermal high-sulphidation deposits (Stoffregen, 1987; Rye et al., 1992). However, for many deposits (e.g. the majority of low-sulphidation subtypes), O and H isotope data permit only a very small fraction (i.e. <10%) of the hydrothermal water to be of magmatic origin, despite the close association of some deposits with cooling magmatic rocks; whereas, C and S isotope studies indicate a significant magmatic contribution in many cases. Thus, a mineralizing fluid can have a complex origin, involving links to degassing magmas as well as the dominance of local recharge waters to fuel the hydrothermal system.

The two principal (end-member) geochemical environments of epithermal mineralization and alteration are determined largely by the dominance in each case of two different fluids. On the one hand, magmatic-hydrothermal environments that are dominated (buffered) by acidic, magmatic fluids produce high-sulphidation mineral assemblages characterized by base leaching of wall rocks leaving marked (residual) silica enrichment. This environment may overlie porphyry systems (Sillitoe and Bonham, 1984). On the other hand, near neutral, more reduced, meteoric-dominated waters containing Cl, H_2S, and CO_2, yield low-sulphidation (adularia/sericite) mineral assemblages through hydrolysis reactions involving feldspar in the wall rocks. The chemical state of these fluids becomes largely wall-rock buffered.

SOURCES OF GOLD

Two fundamentally different hypotheses regarding the source of Au in epithermal deposits are as follows:

1. The metals are supplied directly by actively or passively degassing magma (e.g. Taylor, 1987 and 1988) that also provides heat to the paleo-hydrothermal system.
2. The metals are leached from the rocks that host the geothermal system.

On the one hand, isotopic confirmation of the importance of meteoric waters has encouraged proponents of the second hypothesis. On the other hand, isotopic data also indicate that S and C are of magmatic origin in certain deposits. Alteration mineral assemblages are characteristic of two end-member chemical environments of alteration and mineralization:

- low to very low pH, oxidized fluids (high-sulphidation subtype)
- near neutral, more reduced fluids (low and intermediate-sulphidation subtypes).

Boiling and chemical fractionation of the hydrothermal fluid provides an explanation for the separation of precious and base metals. This separation results in a vertical zoning where fluids are upwardly flowing (Clark and Williams-Jones, 1990), or in relative temporal stages such as at Silbak-Premier, BC, and El Indio, Chile.

Geological, mineralogical, and geochemical features of epithermal Au deposits are listed for each of three deposit subtypes in Table 8.1.

8.4.3 *DIAGNOSTIC CHARACTERISTICS OF EPITHERMAL SUBTYPES*

GRADE AND TONNAGE CHARACTERISTICS

The size and grade of the principal Canadian epithermal Au vein deposits and selected ‘type’ deposits elsewhere in the world are shown in Figure 8.1. The estimated sizes give an order of magnitude basis for comparison; definition of size depends on cut-off grades and economics.

Canadian epithermal Au deposits are comparable in size and grade to many global deposits (Taylor, 2007) although the largest epithermal deposits (in tonnes of ore) and the richest deposits (in g/t) are found outside of Canada.

In the Sulphurets district in BC, epithermal mineralization tends to comprise disseminated Au in silicified and/or finely veined rocks. Grades are typically lower, but tonnages larger, than in other more typical vein-type epithermal deposits.

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Table 8.1 Summary of Geological Setting, Definitive Characteristics[1] and Examples of Typical Epithermal Au Deposit Subtypes (After Taylor, 2007)

	HIGH-SULPHIDATION subtype Hosted in volcanic rocks	LOW-SULPHIDATION subtype Hosted in volcanic and plutonic rocks	LOW-SULPHIDATION subtype Hosted in sedimentary and mixed host rocks
Geological setting	volcanic terrane, often in caldera-filling volcaniclastic rocks; hot spring deposits and acid lakes may be associated	Spatially related to instrusive centre; veins in major faults, locally ring fracture type faults; hot springs may be present	In calcareous to clastic sedimentary rocks; may be at depth by magma; can form at variety of depths
Ore mineralogy	native gold, electrum, tellurides; magmatic-hydrothermal: (+bn), en, tennantite, cv, sp, gn; Cu typically > Zn, Pb; Au-stage may be distinct, base-metal poor; steam-heated: base-metal poor; gangue: quartz (vuggy silica), barite	electrum (lower Au/Ag with depth), gold; sulphides include: sp, gn, cpy, ss); sulphosalts; gangue: quartz, adularia, calcite, chlorite; ± barite, anhydrite in deeper deposits variable metal content, high sulphide veins closer to intrusions	gold (micrometre): within or on sulphides (e.g. pyrite unoxidized ore), native (in oxidized ore), electrum, Hg-Sb-sulphides, pyrite, minor base metals; gangue: quartz, calcite
Alteration mineralogy	advanced argillic + alunite, kaolintie, pyrophyllite (deeper); ± sericite (illite); adularia, carbonate absent; chlorite and Mn-minerals rare; no selenides; barite with Au; steam-heated: vertical zoning	sericitic replaces argillic facies (adularia ± sericite ± kaolinite); Fe-chlorite, Mn-minerals, selenides present; carbonate and/or rhodochrosite) may be abundant, lamellar if boiling occurred; quartz-kaolinite-alunite-subtype minerals possible steam-heated zone; clays	silicification, decalcification, sericitization, sulphidation; alteration zones may be controlled by stratigraphic permeability rather than by faults and fractures; quartz (may be chalcedonic)-sericite (illite)-montmorillonite
Host rocks	silicic to intermediate (andesite)	intermediate to silicic intrusive/extrusive rocks	felsic intrusions; most sedimentary rocks except massive carbonates (hosts to mantos and skarns)
$^{18}O/^{16}O$ - shift in wall rocks	may be less pronounced, or superposed on earlier high-^{18}O alteration	moderate to large; pronounced in and immediately adjacent to veins	very limited ^{18}O-shift of altered rocks, if present at all
C-H-S isotopes	magmatic fluids indicated ($\delta^{13}C_{CO_2} \equiv -5\pm2$; $\delta D_{H_2O} \equiv -35\pm10$; $\delta^{18}O_{H_2O} \equiv +7\pm2$; $\delta^{34}S_{\Sigma S} \equiv 0$); magmatic-hydrothermal alunite $\delta^{34}S$>sulphide minerals; $\delta D \equiv -35\pm10$; steam-heated alunite $\delta^{34}S \equiv$ sulphides, $\delta^{18}O$ data indicate hydrothermal origin	magmatic water (H_2O) may be obscured by mixing; surface waters dominate; C, S typically indicate a magmatic source, but mixtures with wall rock derived C, S possible	hydrogen isotope data (sericite, clays, fluid inclusions) in some cases indicate presence of evolved surface waters; organic carbon ($\delta^{13}C \equiv -26\pm2$) may be derived from wall rocks
Ore fluids (examples from fluid inclusion studies)	160-240°C; ≤1 wt.% NaCl (late fluids); possibly to 30 wt.% NaCl in early fluids; boiling common; (Nansatsu district, Japan; Hedenquist et al., 1994)	sulphide-poor: 180-31°C, ≤1 wt.% NaCl, about 1.0 molal CO_2 (Mt. Skukum: McDonald, 1987) sulphide-rich: ave. 25°C, <1 to 4 wt.% NaCl (Silbak-Premier: McDonald, 1990)	bimodal: 150-160 (most); 270-280°C, ≤15 wt.% NaCl; nonboiling: (Cinola: Shen et al., 1982); 230-250°C, ≤1 wt.% NaCl; nonboiling (Dusty Mac: Zhang et al., 1989)
Age of mineralization and host rocks	host rocks and mineralization of similar age	mineralization variably younger (>1 Ma) than host rocks	mineralization variably younger (>1 Ma) than host rocks
Deposit size	small areal extent (e.g. 1 km^2) and size (e.g. 2500-3500 kg Au)	may occur over large area (e.g. several tens of km^2); may be large (e.g. 100 000 kg Au).	may have large areal extent (e.g. >>1 km^2), large size (e.g. 58 000 kg Au), low grades (e.g. 2.5 g/t)
Examples Canadian	Equity Silver, B.C.; Mt. Skukum, Yukon (only: alunite 'cap') Al deposit, Toodoggone River, B.C.	Blackdome, B.C.; Mt. Skukum, Yukon (Cirque vein) Silbak-Premier, B.C. (intermediate sulphidation)	Cinola, B.C.
Foreign	Summitville, Colorado Kasuga, Japan	Creede, Colorado (intermediate sulphidation)	Hishikari, Japan
Modern analogues:	Matsukawa, Japan[2]	Broadlands, New Zealand[3]	Salton Sea geothermal field, California[4]

1) based, in part, on Heald et al., 1987; Taylor, 1987; Berger and Henley, 1989; Panteleyev, 1991; Rye et al., 1992; Sillitoe, 1993; Hedenquist et al., 2000; Izawa et al. 1990, 1993; and data reported for Canadaian deposits and other examples cited in the text; 2) Nakamura et al., 1970; 3) Browne in Henley and Hedenquist, 1986; 4) Williams and McKibben, 1989, but analogy not complete.
Abbreviations: bn = bornite; cpy = chalcopyrite; cv = covellite; en = energite; gn = galena; py = pyrite; sp = sphalerite; ss = sulphosalts.

PHYSICAL CHARACTERISTICS

The mineralogy, textural features, host rocks, morphology, and selected chemical properties found typically in epithermal Au deposits are summarized in this section and shown in Table 8.1 (Taylor, 2007).

Mineralogy

Quartz is the predominant gangue mineral in all epithermal Au deposits, whereas distinctive ore and gangue minerals characterize high-sulphidation and low-sulphidation deposit subtypes. Mineralogical zoning around veins or replacement zones may be present in both subtypes, recording chemical and/or thermal gradients.

Low-sulphidation

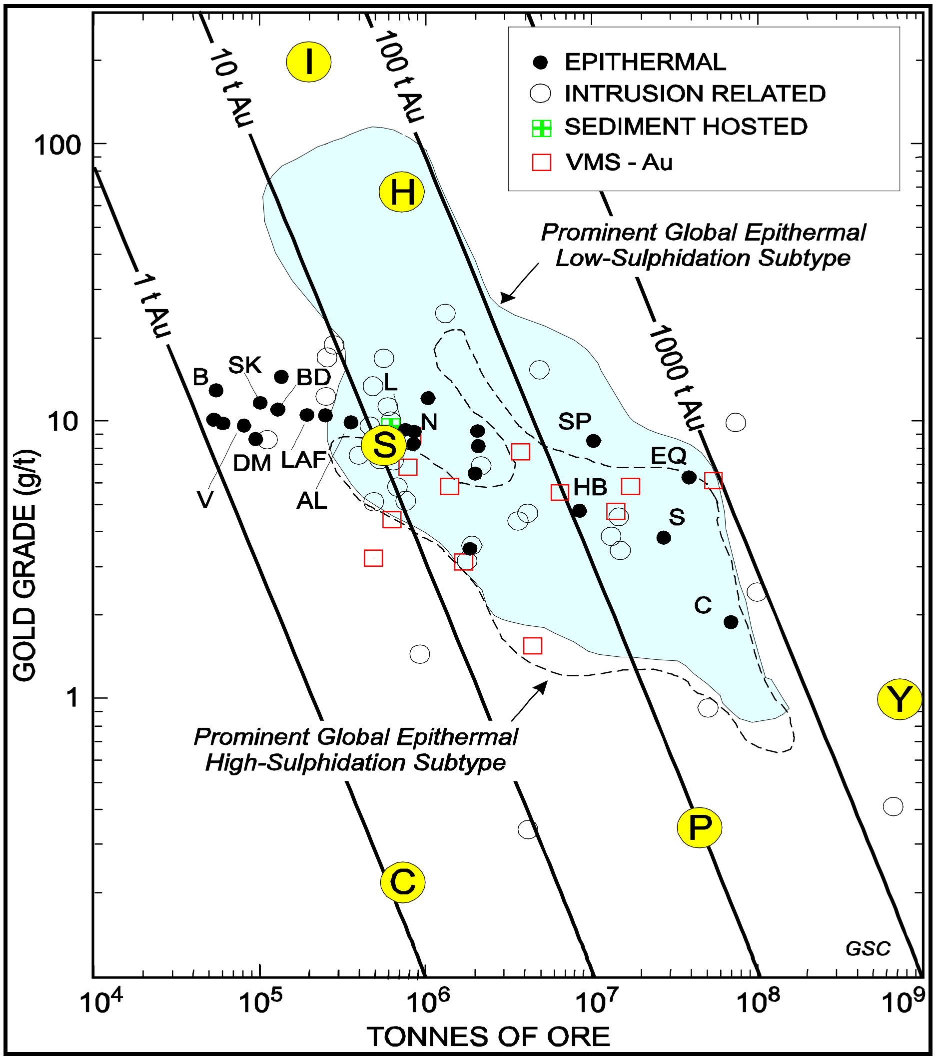
Native Au and electrum occur in low-sulphidation subtype vein deposits that often contain only a few percent or less of sulphides (usually pyrite; e.g. Blackdome, BC). In deposits in which sulphide minerals are abundant (e.g. Venus; Silbak-Premier: sulphide-rich stage), sulphides commonly include chalcopyrite, tetrahedrite, galena, sphalerite, and arsenopyrite in addition to pyrite. The principal gangue minerals include calcite, chlorite, adularia, barite, rhodochrosite, fluorite, and sericite.

In sediment-hosted low-sulphidation deposits, the characteristic assemblage of gangue minerals commonly includes cinnabar, orpiment-realgar, and stibnite, in addition to jasperoid, quartz, dolomite, and calcite. Chalcedonic quartz veins and jasperoid are typically associated with ore, whereas calcite veins are often more common further from ore, or are paragenetically late.

High Sulphidation

In high-sulphidation subtype deposits, native Au and electrum are typically associated with pyrite, enargite, covellite, bornite and chalcocite. In addition to sulphosalts and base metal sulphides, tellurides and bismuthinite are present in some deposits. Total sulphide contents are generally higher in high-sulphidation than low-sulphidation subtype deposits but high sulphide contents may also characterize transitional polymetallic low-sulphidation deposits (e.g. Silbak Premier, BC). Where base metals are present in high-sulphidation deposits, the Cu abundance can vary significantly (Sillitoe, 1993) and typically dominate that of Zn. Principal gangue minerals include quartz (“vuggy silica”), alunite, barite (especially associated with Au). Calcite is not characteristic of high-sulphidation subtype deposits due to the high acidity of the hydrothermal fluids.

Figure 8.3 Plot of Au Grade (g/t) vs. Tonnage (Economic, or Reserves + Production) for Selected Canadian Epithermal Au Deposits & Prominent Examples Worldwide (after Taylor, 2007)



Notes:

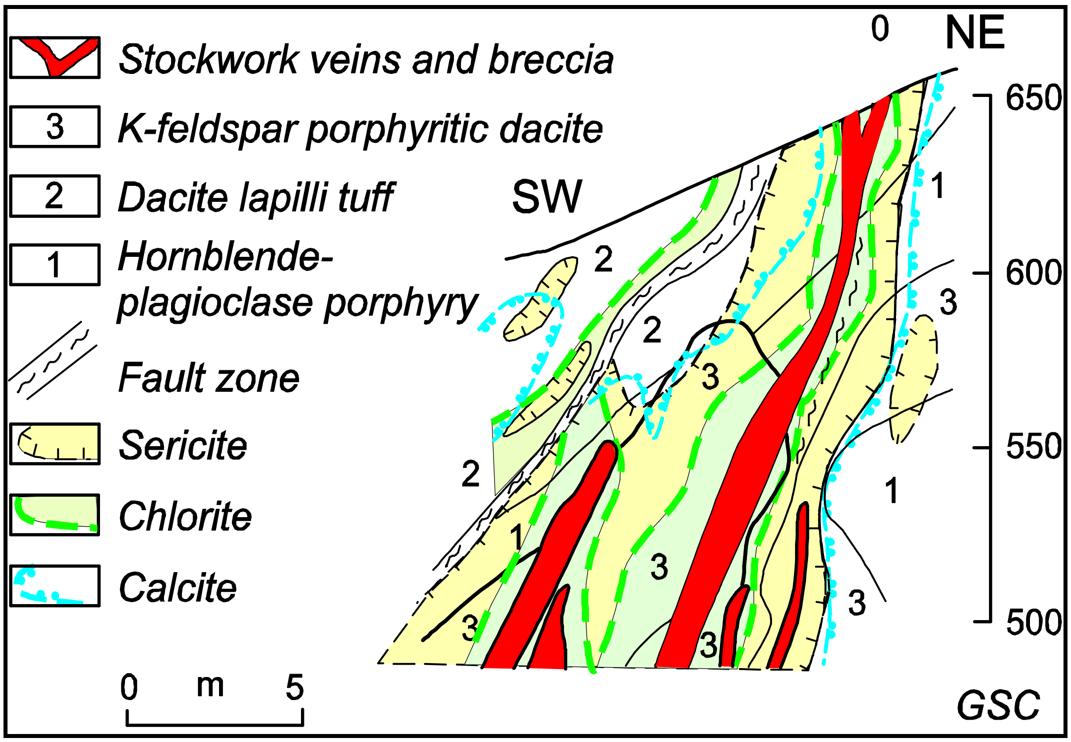
- Canadian epithermal deposits (**filled circles**) include: **Al** = Al, **B** = Baker, **BD** = Blackdome, **C** = Cinola, **DM** = Dusty Mac, **EQ** = Equity Silver, **L** = Lawyers, **LAF** = Laforma, **N** = Mt. Nansen, **SK** = Mt. Skukum, **SP** = Silbak Premier, **S** = Sulphurets, and **V** = Venus.
- Hydrothermal vein deposits of a possible ‘transitional’ or ‘deep epithermal’ deposits are represented by **open circles**, sediment-hosted deposits by a **green square with cross**, and Au-bearing VMS deposits (marine epithermal) by **open red squares**.
- The median grades and tonnages for several comparable types of deposits (**yellow-filled circles**) from Cox and Singer (1986) include porphyry Cu-Au (**P**), low-sulphidation Creede-type (**C**), and high-sulphidation: Summitville deposit (**S**); and Lawyers deposit, Toodoggone River district, BC (**L**) [similar to the ‘Comstock-type’, Nevada (no symbol) of Cox and Singer, 1986].
- Median values for the low-sulphidation Hishikari, Japan vein deposit [**H**], and for the high-sulphidation El Indio, Chile, deposit [**I**] are from Hedenquist et al. (2000).
- Fields for prominent low-sulphidation (**blue shading**) and high-sulphidation (**dashed line**) epithermal Au deposits worldwide (global) are based on data in Hedenquist et al. (1996, 2000).

System Dimensions

High-sulphidation deposits of magmatic hydrothermal origin are typically of smaller dimension than low-sulphidation subtype deposits, and are found in close proximity to and often topographically above, a related source of magmatic heat and volatiles.

Low-sulphidation subtype deposits in most cases cover larger areas than typical high-sulphidation deposits, even though alteration mineral assemblages are restricted to generally narrow zones enclosing veins and breccias. At the Blackdome Mine, BC, quartz veins are contained within an area approximately 2 km by 5 km. Veins and breccia zones as wide as 40 m and as long as 1,200 m comprise the Main Zone of the Silbak-Premier deposit (Figure 8.4) (McDonald, 1990).

Figure 8.4 Geological Cross-section of a Representative Canadian Epithermal Deposit Illustrating Alteration Mineral Zoning and Selected Features



Note: after Taylor, 1996.

Figure 8.4 is a cross-section through a portion of the Silbak-Premier deposit (intermediate sulphidation; after McDonald, 1990), which illustrates hydrothermal propylitic, sericitic, and potassic alteration mineral assemblages in relation to fault-controlled vein stockwork and breccia, and to porphyritic dacite.

Morphology

The morphology of epithermal vein-style deposits can be quite variable. Deposits may consist of roughly tabular lodes controlled by the geometry of the principal faults they occupy (e.g. Cirque vein, Mt. Skukum), or comprise a host of interrelated fracture fillings in stockwork, breccia, lesser fractures, or, when formed by replacement of rock or void space, they may take on the morphology of the lithologic unit or body of porous rock (e.g. irregular breccia pipes and lenses) replaced.

Brecciation of previously emplaced veins (e.g. Mt. Skukum, Yukon) can form permeable zones along irregularities in fault planes: vertically plunging ore zones in faults with strike-slip motion and horizontal ore zones in dip-slip faults. Topographic (i.e. paleosurface) control of boiling by hydrostatic pressure can also result in horizontal or sub-horizontal mineralized zones, limiting the vertical distribution of ore.

Host Rocks

Nearly any rock type, even metamorphic rocks, may host epithermal Au deposits, although volcanic, volcaniclastic and sedimentary rocks tend to be more common. Typically, epithermal deposits are younger than their enclosing rocks, except in the cases where deposits form in active volcanic settings and hot springs. Here, the host rocks and epithermal deposits can be essentially synchronous with spatially associated intrusive or extrusive rocks, within the uncertainty of the determined ages in some cases.

Chemical Characteristics

Ore Chemistry

Gold:silver ratios of epithermal Au deposits may vary widely both between and within deposits ranging from lows of around 0.5 for the high-sulphidation type deposit as typified by the Kasuga deposit in Japan (Hedenquist et al., 1994) to >500 in the Cerro Rico de Potosi deposit in Peru (Erickson and Cunningham, 1993). Differing magmatic metal budgets (Sillitoe, 1993) and depths of formation (Hayba et al., 1985) have been suggested to influence this ratio.

Typically, Ag:Au ratios for epithermal deposits, though variable, tend to be higher in low-sulphidation subtype deposits than in high-sulphidation subtype deposits. The deep epithermal (mesothermal) Equity Silver deposit in BC (e.g. Cyr et al., 1984; Wojdak and Sinclair, 1984) has the highest Ag:Au ratio (approximately 128) among Canadian epithermal deposits.

Alteration Mineralogy and Chemistry

Hydrothermal alteration mineral assemblages are commonly regularly zoned about vein- or breccia-filled fluid conduits in both high and low-sulphidation deposit subtypes. Characteristic alteration mineral assemblages in both deposit subtypes can give way to propylitically altered rocks containing quartz + chlorite + albite + carbonate - sericite, epidote, and pyrite. The distribution and formation of the earlier formed propylitic mineral assemblages generally bears no obvious direct relationship to ore-related alteration mineral assemblages.

Altered rocks in low-sulphidation deposits generally comprise two mineralogical zones:

- an inner zone of silicification (replacement of wall rocks by quartz or chalcedonic silica)
- an outer zone of potassic-sericitic (phyllic) alteration [adularia is the typical K-feldspar but its prominence varies greatly and it may be absent altogether; argillic alteration (kaolinite and smectite) occurs still farther from the vein].

Silicified rocks are common in epithermal deposits, as is quartz gangue in veins. The silicified and decarbonatized host rocks that characterize Carlin type Au deposits in Nevada (e.g. Bagby and Berger, 1986) was apparently controlled by available primary permeability of bedding planes or rock fabric. Secondary permeability can also be produced by physical and chemical processes involving the hydrothermal fluids themselves. The sudden release of pressure on hydrothermal fluid (e.g. by faulting) can cause brecciation, creating pore space permeability. Dissolution of carbonate upon reaction between hydrothermal fluids and wall rocks also can produce secondary permeability.

Advanced argillic alteration mineral assemblages that characterize high-sulphidation deposits include quartz + kaolinite + alunite + dickite + pyrite in and adjacent to veins or zones of replacement in the magmatic-hydrothermal environment. Pyrophyllite occurs in place of kaolinite at the higher temperatures and pressures of deeper deposits. These alteration minerals indicate a very low pH hydrothermal environment of high oxidation state. Zones of silica replacement and 'vuggy silica' are characteristic, and carbonates are absent. Topaz and tourmaline in high-temperature zones indicate the presence of F and B in the acidic hydrothermal fluids.

Acid-sulphate (high-sulphidation) type alteration fluids form by the dissolution of large amounts of magmatic SO_2 in high-temperature hydrothermal systems, and also by reaction of host rocks with steam-heated meteoric waters acidified by oxidation of H_2S (probably of magmatic origin: e.g. Rye et al., 1992; Bethke et al., 2005), or by dissolution of CO_2. Lower acidity, highly saline fluids are thought responsible for intermediate sulphidation deposits typically rich in base metal and Fe sulphide minerals (Hedenquist et al., 2000).

Fluids attributed to low-sulphidation hydrothermal systems are typically less saline than those in high-sulphidation systems, although fluids of two different salinities are also common. The primary fluids in low-sulphidation subtype deposits are commonly inferred to have largely evolved from meteoric rather than magmatic water, or comprise some mixture of the two (e.g. Hishikari, Japan: Faure et al., 2002).

The hydrothermal fluids responsible for alteration and mineralization largely represent altered or 'evolved' meteoric waters whose isotopic compositions have been shifted to higher $^{18}O/^{16}O$ and D/H (deuterium-to-hydrogen) ratios than those of pure local meteoric waters (compare with present day meteoric water). Such isotopic alteration or evolution of the fluids occurs during chemical, isotopic, and mineralogical hydrothermal alteration of the host rocks.

Margolis (1993) inferred progressive mixing of magmatic water and seawater during potassic, sericitic, and advanced argillic alteration at Sulphurets, BC, on the basis of isotopic data and water-rock reaction modeling.

Fluid inclusions typically have been shown to contain predominantly fluids of low salinity and have filling temperatures of 150°C to 300°C, with maxima in the range of approximately 260°C to 280°C. Vapour -dominated systems at or near a boiling water table tend to evolve toward a rather uniform temperature of about 240°C due to the limitation imposed by a maximum in the enthalpy of steam + liquid (e.g. White et al., 1971).

Some deep epithermal (transitional) environments close to genetically related intrusions are characterized by higher temperatures, salinities, and CO_2 contents (e.g. Baker, 2002).

8.5 SUMMARY – EPITHERMAL MINERALIZING SYSTEMS

The geological settings of low-, intermediate- and high-sulphidation subtype epithermal deposits are illustrated schematically in Figure 8.5.

The locations of epithermal Au deposits are typically determined by those features that define the hydrothermal system “plumbing”. Extensional faults are especially important, whether due to local, volcanic-related features or to regional tectonism (e.g. rifting zones, or pull-apart basins associated with strike-slip faults). Fault intersections and fault plane inflections provide zones for vein thickening and zones of brecciation during synchronous movement and vein growth.

8.5.1 HIGH-SULPHIDATION EPITHERMAL DEPOSIT CHARACTERISTICS

High-sulphidation deposits are typically associated with andesitic to rhyolitic rocks and with geologic features associated with sites of active volcanic venting and doming, including among others ring fractures, caldera fill breccias, hot springs, and acidic crater lakes. It is the dominance of directly derived or evolved magmatic fluids that buffer the hydrothermal fluids to low pH and result in the distinct character of the high-sulphidation subtype. Orebodies primarily consist of zones of silica-rich replacement. Bodies of massive ‘vuggy silica’ and marked advanced argillic alteration mineral assemblages are typical.

8.5.2 Low-Sulphidation Epithermal Deposit Characteristics

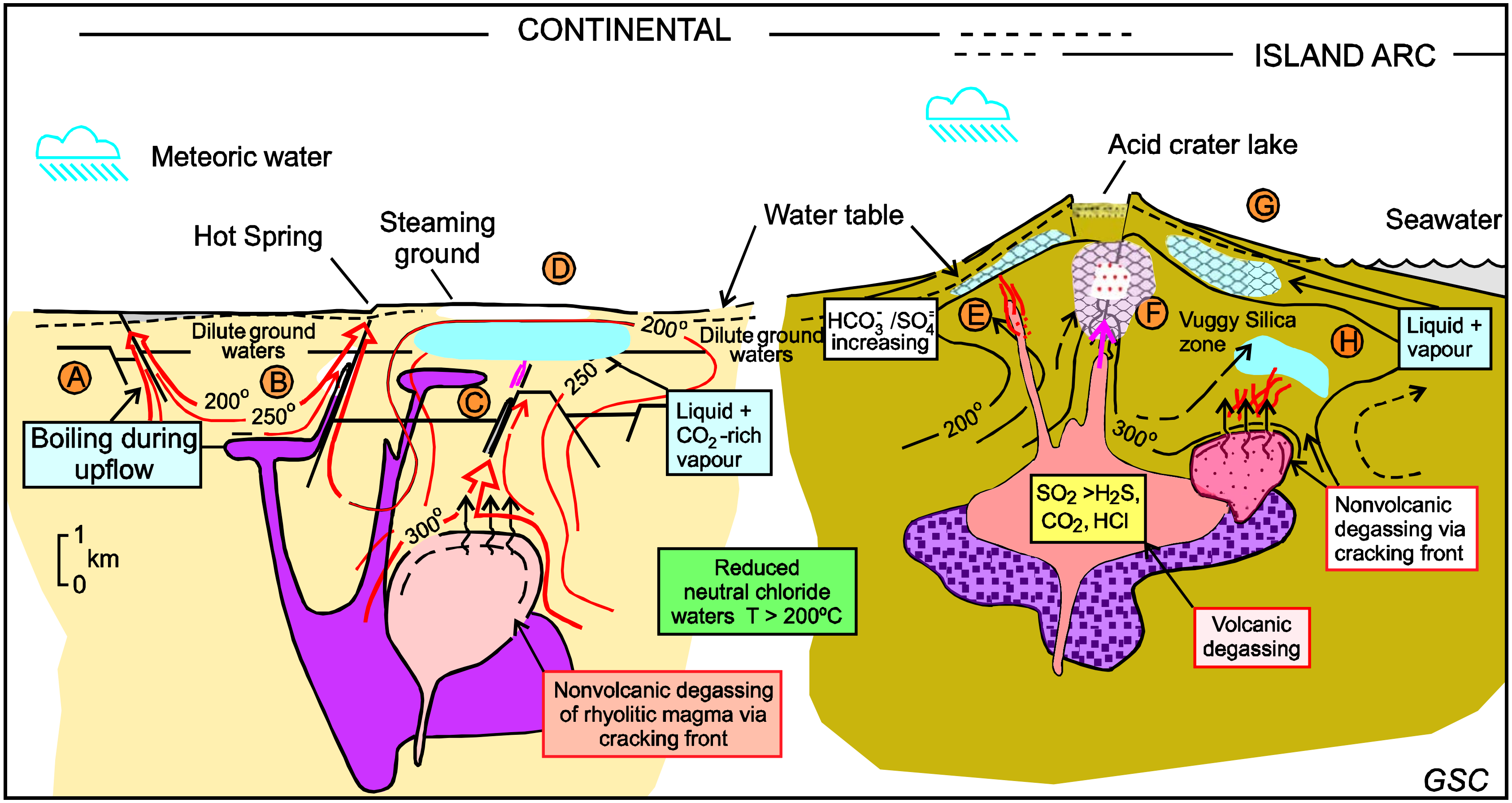
Low-sulphidation deposits that occur further removed from active magmatic vents may be more apparently controlled by structural components, zones of fluid mixing, and emplacement of smaller magmatic bodies (e.g. dykes). Meteoric waters dominate the hydrothermal systems, which are more nearly pH neutral in character. Low-sulphidation related geothermal systems are more closely linked to passive rather than to active magmatic degassing (if at all), and sustained by the energy provided by cooling, sub-volcanic intrusions or deeper sub-volcanic magma chambers.

8.5.3 Transitional-Sulphidation Epithermal Deposit Characteristics

Some deposits with mostly low-sulphidation characteristics with respect to their alteration mineral assemblages have sulphide ore mineral assemblages that represent a sulphidation state between that of high-sulphidation and low-sulphidation deposits. Such deposits tend to be more closely spatially associated with intrusions, and Hedenquist et al. (2000) suggest the term ‘intermediate sulphidation’ for these deposits.

The various Brucejack property mineralized zones that are the subjects of the current report, are considered similar to the Silbak-Premier Mine which, as shown in Figure 8.5, is classified as a transitional to low sulphidation epithermal deposit.

Figure 8.5 Schematic Cross-section – General Geological & Hydrological Settings of Quartz-(Kaolinite)-Alunite & Adularia-Sericite Deposits



Note: from Taylor, 1996; partially adapted from Henley and Ellis, 1983, and Rye et al., 1992).

Characteristics shown in Figure 8.5 evolve with time; all features illustrated are not implied to be synchronous.

Local environments and examples of low-sulphidation deposits, as illustrated in Figure 8.5, include:

- (**A**) basin margin faults; Dusty Mac
- (**B**) disseminated ore in sedimentary rocks; Cinola
- (**C**) veins in degassing, CO_2-rich, low sulphide content, low-sulphidation systems; Blackdome, Mt. Skukum
- (**E**) porphyry-associated vein-stockwork, sulphide-rich (intermediate sulphidation) and sulphide-poor stages; Silbak-Premier
- (**H**) disseminated replacement associated with porphyry-type and stockwork deposits, involving seawater; Sulphurets.

Examples of high-sulphidation environments, as illustrated in Figure 8.5, include:

- (**D** and **G**) steam-heated advanced argillic alteration (quartz-kaolinite-alunite) zone; Toodoggone River district, BC
- (**F**) magmatic-hydrothermal, high-sulphidation vuggy quartz zone (± aluminosilicates, corundum, alunite); Summitville, Colorado, or Nansatsu district, Japan.

The following notes also apply to Figure 8.5:

- Fluid flow parallels isotherms. Up-flow zones are shown schematically by arrowhead-shaped isotherms.
- Volcanic degassing refers to magmatic degassing driven by depressurization during emplacement (‘first boiling’).
- Non-volcanic degassing refers to vapour exsolution during crystallization (‘second boiling’).
- The SO_2 disproportionates to H_2S and H_2S0_4 during ascent beneath environment (**F**).
- Note that free circulation occurs only in crust above about 400°C.
- All temperatures are shown in degrees Celsius.

9.0 MINERALIZATION

9.1 SNOWFIELD DEPOSIT

The gold mineralization at the Snowfield deposit is hosted by schistose, pervasively altered (quartz-sericite-chlorite) volcanic and volcaniclastic rocks 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).

9.2 BRUCEJACK PROPERTY

9.2.1 *INTRODUCTION*

There are more than 70 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 Unuk River Formation or the lower members of the Betty Creek Formation (Britton and Alldrick, 1988).

Early Jurassic sub-volcanic intrusive complexes are common in the Stikinia terrane, and several host well-known precious and base metal rich hydrothermal systems. These include copper-gold porphyry deposits such as Galore Creek, Red Chris, Kemess, Mt. Milligan, and KSM. In addition, there are a number of related polymetallic deposits including skarns at Premier, epithermal veins and subaqueous vein and replacement sulphide deposits at Eskay Creek, Snip, Brucejack, and Granduc (Savell, 2008).

Within the Kerr-Sulphurets area, two basic styles of mineralization have been documented:

- Porphyry-type gold mineralization associated with fine grained syenite to syenodiorite intrusive rocks intrusive breccias and pyritization
- Silver-gold-base metal epithermal veins occurring within or adjacent to fine grained syenodiorite intrusions and associated with large area of intense sericite, quartz, pyrite alteration; these structurally controlled veins may or may not have significant sulphide contents.

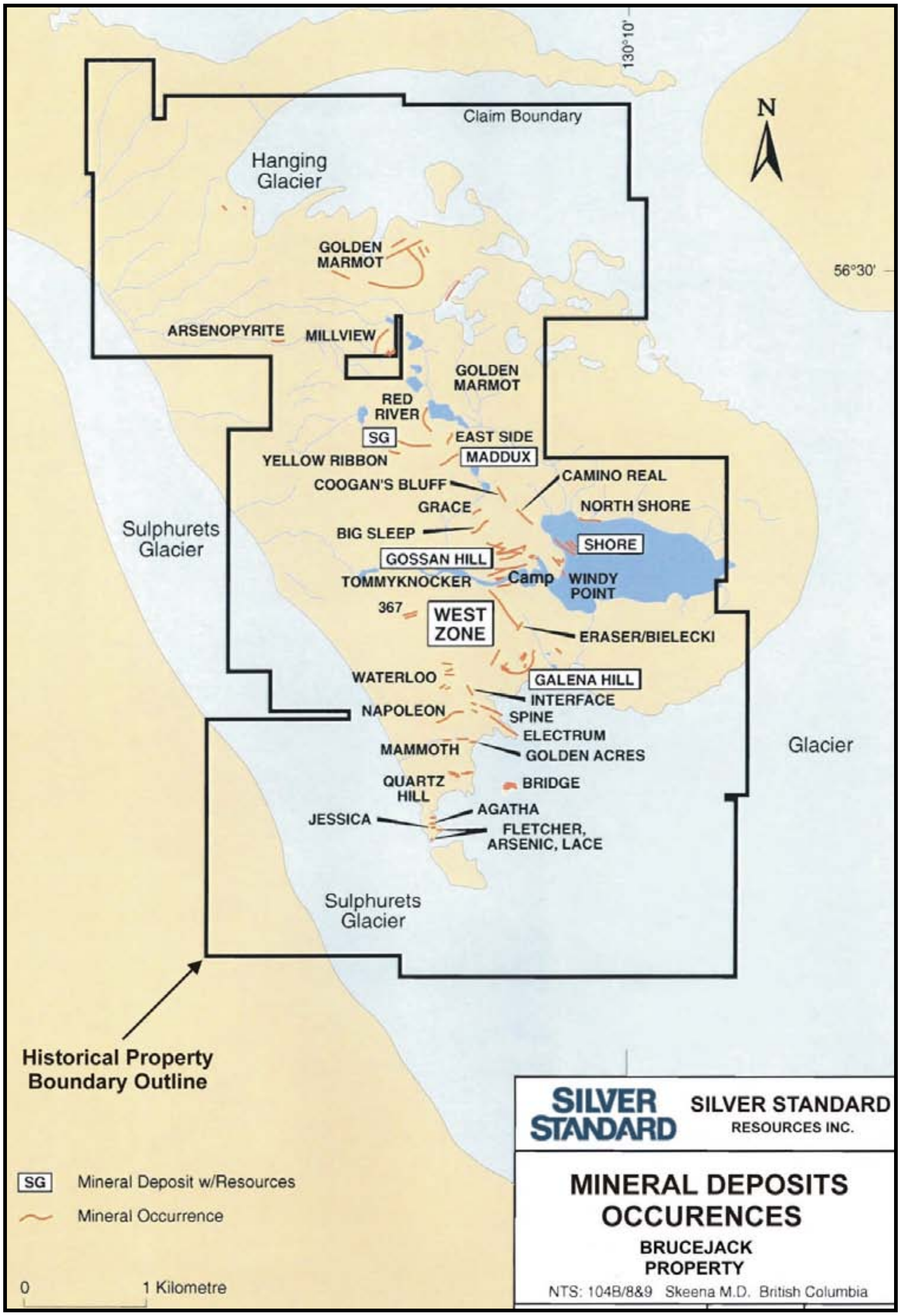
The Brucejack area is dominated by structurally controlled silver-gold-base metal bearing epithermal veins as described by Alldrick and Britton (1991).

9.2.2 *GENERAL BRUCEJACK PROPERTY MINERALIZATION*

The Brucejack area has been the focus of periodic exploration over the past several decades resulting in the discovery of at least 40 gossanous zones of gold, silver, copper and molybdenum–bearing quartz/carbonate veining, stockwork and breccia hosted mineralization (Figure 9.1). Typically, these gossanous showings reflect the weathering of disseminated pyrite in argillic and phyllic alteration zones. The size of these gossans, their tectonic fabric, intensity of alteration and metallogenesis make them attractive exploration targets (Alldrick and Britton, 1991) and most have been extensively sampled and/or drill tested.

The mineralization on the Brucejack property typically consists of structurally controlled, intrusive related quartz-carbonate, gold-silver bearing veins, stockwork and breccia zones. The veins are hosted within a broad zone of potassium feldspar alteration, overprinted by sericite-quartz-pyrite ± clay. Structural style and alteration geochemistry indicates the deposits were formed in a near surface epithermal style environment.

Figure 9.1 Historical Map with Mineral Deposits and Occurrences



Note: modified after Budinksi, 1995.

Mineralization was likely a three-stage process as envisioned by Lewis (1994) in the summary below:

- **Stage 1** is interpreted as an initial episode of fault-development and ground preparation. Pre-cursor structures to the West, Shore, and Electrum zones likely formed at this time, as steep northwest trending normal faults with limited displacement, cutting all rock types.
- **Stage 2** involved development of syntectonic mineralization and alteration. Massive and stockwork vein systems were emplaced within an east-west compressional stress field. The main vein orientations resulting from this stress are:
  - (**i**) east-west dilational veins
  - (**ii**) northwest trending veins localized along pre-existing structures such as the West, Shore, and Bridge (Electrum) zones

  Underground mapping at the West Zone indicates that the northwest trending structures have been brecciated, while east-west trending structures have not. This would support the theory of reactivation along pre-existing northwest structures. Reactivation was probably sinistral in movement. The localization of major vein systems within the volcanic rocks as opposed to the sedimentary rocks is likely the results of preferential ground preparation.
- **Stage 3** was marked by the development of northwest trending cleavage and local warping of smaller veins as a result of northeast-southwest shortening.

Silver Standard has reviewed all of the historical and ongoing exploration results, allowing it to identify seven zones of potentially near term economically viable mineralization. This is in addition to the Snowfield Zone of porphyry-type mineralization to the north.

The following seven high–priority zones of mineralization presently comprise the Brucejack property:

1. West Zone
2. Bridge Zone
3. Galena Hill Zone
4. Shore Zone
5. SG Zone
6. Gossan Hill Zone
7. Mammoth Zone.

Zones 1 through 6 are the focus of Mineral Resource Estimates outlined in Section 17.0 of this report and are discussed individually in Sections 9.2.3 through 9.2.8.

Further drilling is planned to increase the density of holes on the Mammoth showing, and thus allow definition of a Mineral Resource.

VEIN MINERALIZATION

The zones of gold-silver-copper-molybdenum mineralization comprising the Brucejack area are, for the most part, considered the product of fault and fracture-controlled hydrothermal activity related to local intrusive activity.

In general, the vein mineralization appears to represent a complex system of structurally controlled overprinting of ore types and multiple generations of alteration and vein assemblages. Veins can be classified on the basis of metal content and gangue mineralogy. Typically the exposed veins are thin (1 m) and short (<50 m). Individual veins may coalesce into more densely packed vein systems, especially in more intensely altered areas, and locally often represent in excess of 25% of the outcrop. Such vein systems typically grade imperceptibly into the strongly silicified host rocks.

Base metal bearing quartz veins consist primarily of thin stringers of quartz ± carbonate which locally contain zones of disseminated to massive sulphides with varying amounts of pyrite, galena, and/or sphalerite. They are found locally around the Brucejack Plateau outside the main areas of alteration. Individual veins may be strongly gossanous.

Precious and base metal veins (e.g. West Zone) are polymetallic stockworks of thin veins and fracture fillings. Tension gash structures are common. The veins show complex crosscutting relationships that indicate repeated fracturing and filling as the host rocks underwent brittle deformation.

Precious metal mineralization may be confined to one particular episode of veining, which is not necessarily the same episode as base metal mineralization. The gold is associated with pyrite + electrum in quartz ± calcite veins. Arsenopyrite may occur peripherally in the host rocks.

Barite veins were first discovered by Bruce and Jack Johnson in 1935 near the outflow of Brucejack Lake. They consist of coarsely crystalline barite with minor quartz, carbonate, and sulphides

PORPHYRY-TYPE MINERALIZATION

Porphyry-type disseminated pyrite-chalcopyrite-molybdenite mineralization occurs on the Snowfield and KSM properties immediately adjacent to the north and west of the

Brucejack property. Such mineralization occurs within sub alkaline porphyritic intrusions, including monzodiorite, monzonite, syenite, and granite.

The porphyry-type gold and copper deposits (e.g. Mitchell, Sulphurets, and Snowfield Zone) usually have a higher-grade central or core area surrounded by lower-grade mineralization that is dispersed over a very large area and is related to very fine grained disseminated chalcopyrite.

Within the higher grade core area, gold and copper grades correlate closely with one another. The Cu /Au ratio tends to be slightly higher closer to the phyllic-propylitic transitional areas. In the low-grade peripheral shells, the Cu /Au grades tend to be the highest. The gold and copper distribution is remarkably smooth and continuous with grades decreasing very gradually outwards from the higher grade core. These observations suggest that the deposit was generated by a large, stable hydrothermal system with a low thermal gradient within homogeneous host rocks. The distribution was minimally disrupted by late faulting with only minor offsets.

9.2.3 WEST ZONE

The following descriptions (Sections 9.2.3 through 9.2.8) of the mineralization of the West, Bridge, Galena Hill, Shore, SG, and Gossan Hill zones of the Brucejack property were provided by Mr. Ron Burk, Chief Geologist at Silver Standard, in the form of an internal company report, dated December 9, 2009.

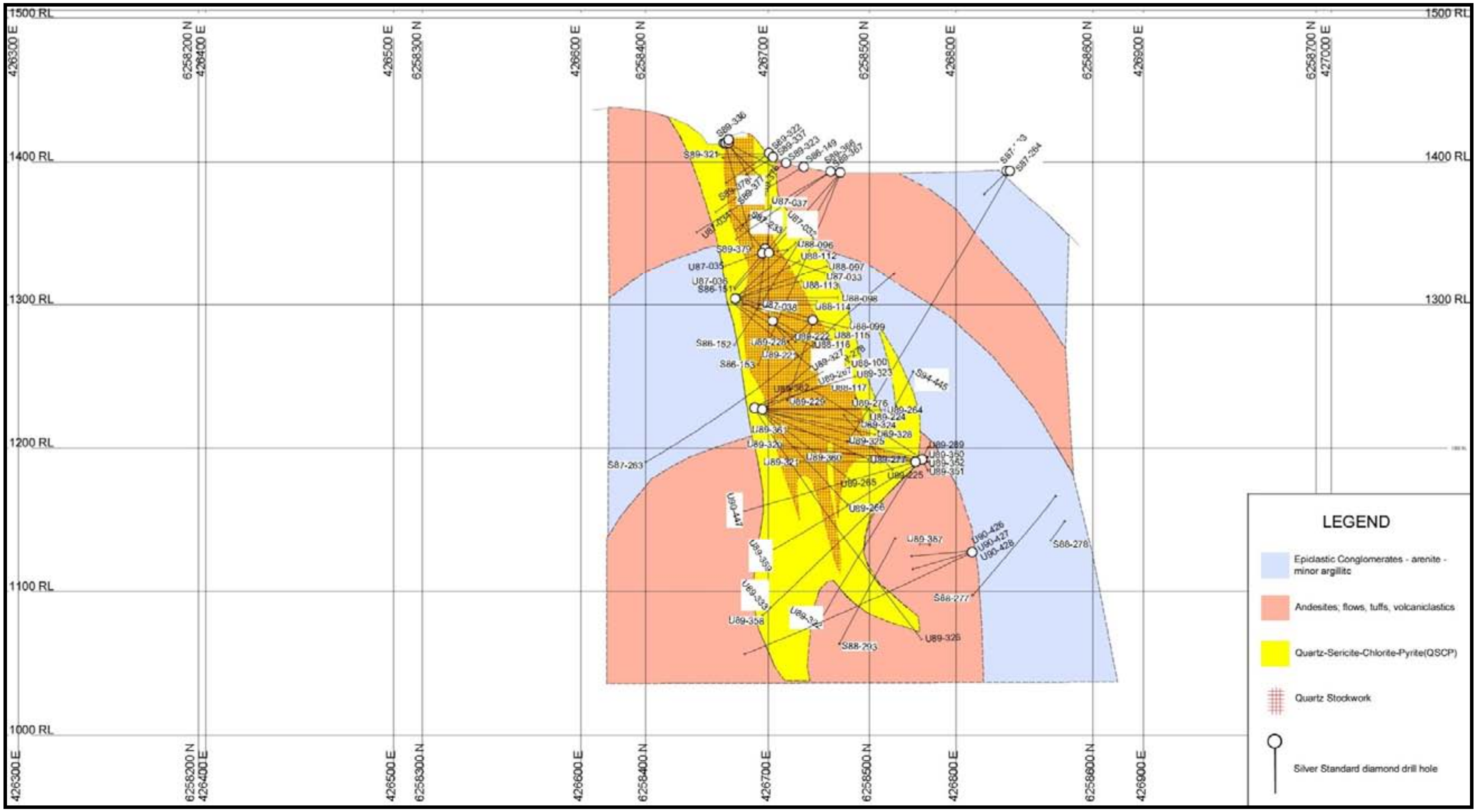
The West Zone gold-silver deposit is hosted by a north-westerly trending band of lower Jurassic (Unuk River member, Hazelton Group) andesitic and lesser sedimentary rocks (Figure 9.2), 400 m to 500 m wide, that passes between two intrusive bodies of plagioclase-hornblende porphyry. The supracrustal rocks are steeply inclined to the northeast and display varying degrees of brittle-ductile deformation and moderate to intense hydrothermal alteration, particularly where the precious metal deposit has been outlined.

The deposit itself comprises at least 10 quartz veins and quartz stockwork shoots, the longest of which has a strike length of 250 m and a maximum thickness of about 6 m. Most mineralized shoots have vertical extents that are greater than their strike lengths. Geometries of the main veins suggest they represent central and oblique shear veins which developed in response to transpressional strain and resulting sinistral, mainly ductile deformation (Roach and Macdonald, 1991). Crack-seal features shown by most of the veins are evidence of brittle deformation overlapping with some crystallization of gangue minerals. Thus, at the West Zone, it appears that ductile shearing generated the dilatant structures that served as conduits for the hydrothermal fluids, which deposited silica and precious metals, but hydrostatic overpressures within the conduits intermittently caused brittle failure along these structures.

In terms of hydrothermal alteration, the West Zone is marked by a central silicified zone that passes outwards to a zone of sericite ± quartz ± carbonate and then an outer zone of chlorite ± sericite ± carbonate. The combined width of these alteration zones across the central part of the deposit is 100 m to 150 m.

Gold in the West Zone occurs principally as electrum and in quartz veins and is associated with, in decreasing order of abundance, pyrite, sphalerite, chalcopyrite, and galena. Besides being found with gold in electrum, silver occurs in tetrahedrite, pyrargyrite, polybasite, and rarely stephanite and acanthtite. Gangue mineralogy of the veins is dominated by quartz, with accessory K-feldspar, albite, sericite, and minor carbonate and barite.

Figure 9.2 Section 5160N of the West Zone, Brucejack Property– Looking Northwest



Source: Silver Standard.

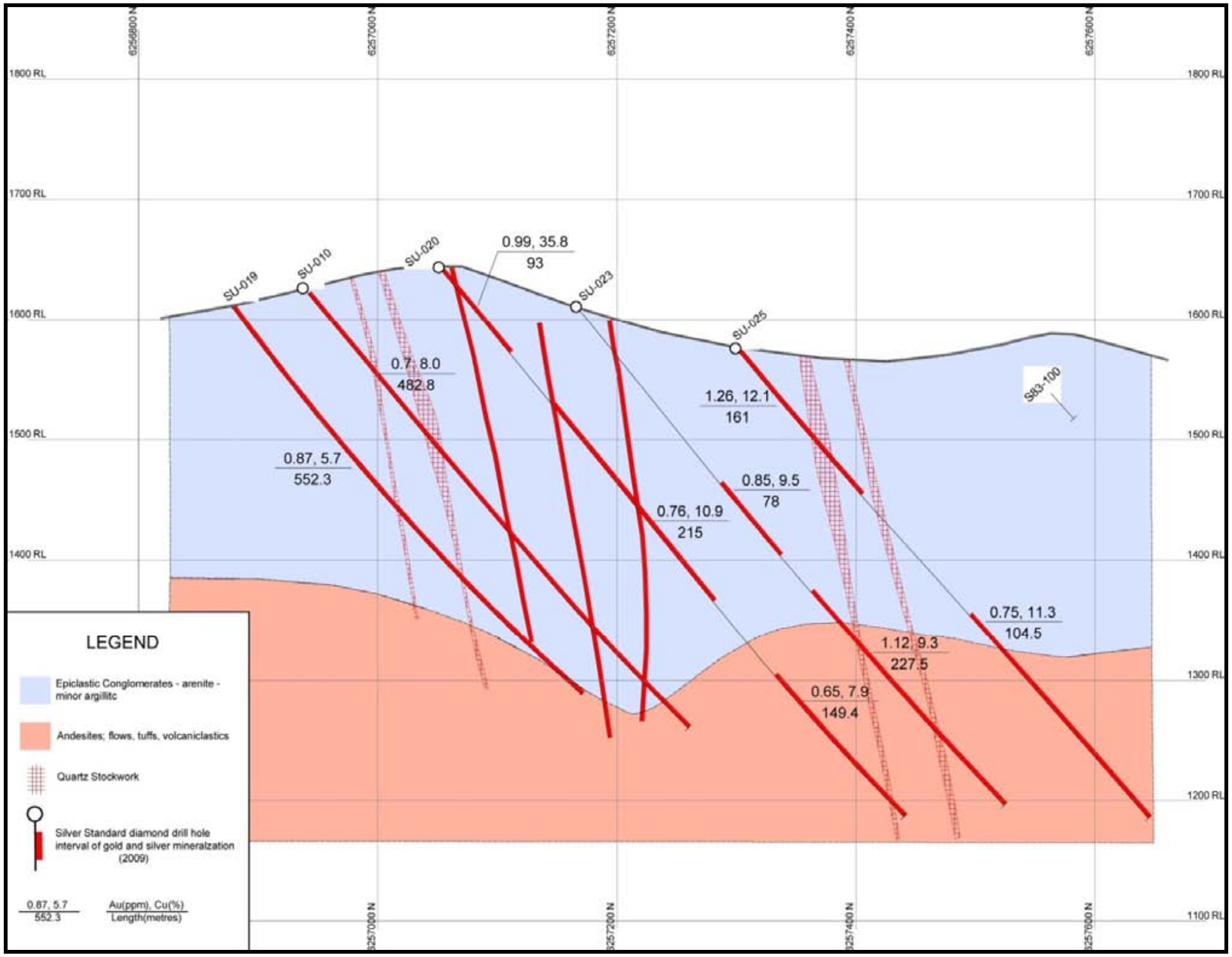
9.2.4 BRIDGE ZONE

The Bridge Zone is located about 1,500 m north of the southern Brucejack property boundary and is centred on a 3-ha nunatak outcrop that is surrounded by ice of the eastern arm of the Sulphurets glacier. Geologists working for Newhawk and the Geological Survey of Canada had previously mapped and sampled this outcrop, recognizing that it displayed strong sericite-pyrite alteration and was transected by a number of discontinuous mineralized quartz veins. Based on the encouraging gold assays obtained in these historical rock-chip samples, Silver Standard decided to test the prospect with a single drill hole, SU-10 (Figure 9.3). Assay results for this drill hole showed that it intersected a broad zone of low-grade gold mineralization of possible economic significance.

The mineralized intercept in SU-10 was reported as being 483 m averaging 0.70 g/t Au and extended from surface. The discovery of potentially bulk-mineable gold at the Bridge Zone prompted Silver Standard to drill another 12 diamond bore holes to probe for the limits of this mineralization.

These drill holes determined that the bulk of the gold mineralization is hosted by plagioclase-hornblende porphyry intrusive rock that in general is moderately sericite-chlorite altered, with disseminated and stringer pyrite making up a few percent of the rock by volume. Quartz ± chlorite ± sericite veins, 20 cm to 200 cm in thickness, were intermittently intersected by the drill holes, and these commonly contain minor to trace amounts of pyrite, sphalerite, galena, molybdenite, and unknown dark grey, silver-bearing sulfosalt(s).

Figure 9.3 Section 426775E of the Bridge Zone, Brucejack Property – Looking West



Source: Silver Standard.

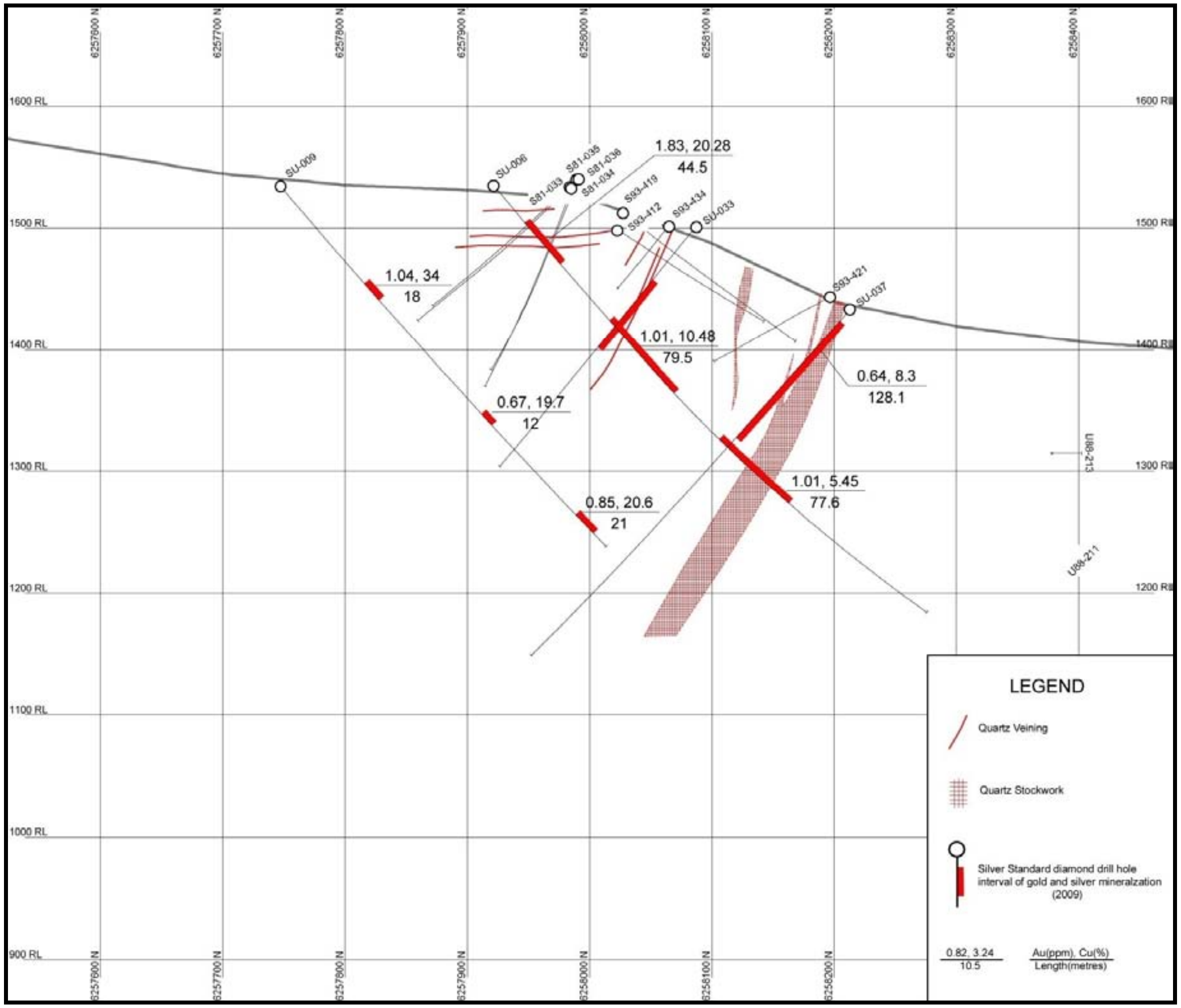
9.2.5 GALENA HILL ZONE

The prospect area known as Galena Hill is situated between the West Zone and Bridge Zone gold deposits on a prominent hill marked by widespread iron oxide staining of altered meta-andesites. The Galena Hill Zone had been previously tested with 27 bore holes belonging to a number of different drilling campaigns, with half of the holes being less than 100 m in length. Assays from these holes, together with detailed geological mapping and channel rock-sampling, indicate that there is a system of E-W and NE-SW-trending quartz veins and quartz stockworks at Galena Hill that, as a whole, define a zone of hydrothermal alteration and mineralization that is at least 400 m long and 200 m wide.

Rather than target the larger quartz veins, which locally contain high-grade gold + silver mineralization on surface, Silver Standard decided to test for the potential of a low-grade, bulk-mineable deposit. This was done with 8 relatively long (>400 m) drill holes completed during the 2009 exploration program. The majority of these bore holes passed through amygdaloidal and massive andesite flows, volcaniclastic deposits rich in lapilli-sized andesitic clasts and thin units of carbonaceous and cherty mudstones. A few holes intersected rhyolitic dikes and one hole (SU-005) yielded a 50 m-long quartz vein intercept enriched in gold and silver along its margins, though it is likely that this intercept is at a low angle to the dip of the vein.

As in the West Zone, gold mineralization at the Galena Hill Zone is preferentially associated with quartz veins (Figure 9.4), although the sericite-altered, andesitic host rocks are typically mineralized with disseminated pyrite and have geochemically anomalous gold contents, generally in the 100-500 ppb Au range. In some veins, trace amounts of native gold and electrum are accompanied by minor to occasionally substantial amounts of sphalerite, chalcopyrite, and galena. Two of the drill holes drilled in the 2009 drill program intersected spectacularly rich gold mineralization. A 1.5 m-long intercept in SU-012 gave impressive assays of 16.95 kg/t Au and 8.95 kg/t Ag, where the precious metals occurred as a centimetre-wide band of electrum within a quartz vein only a few centimetres wide itself.

Figure 9.4 Section 426925E of the Galena Hill Zone, Brucejack Property – Looking West



Source: Silver Standard.

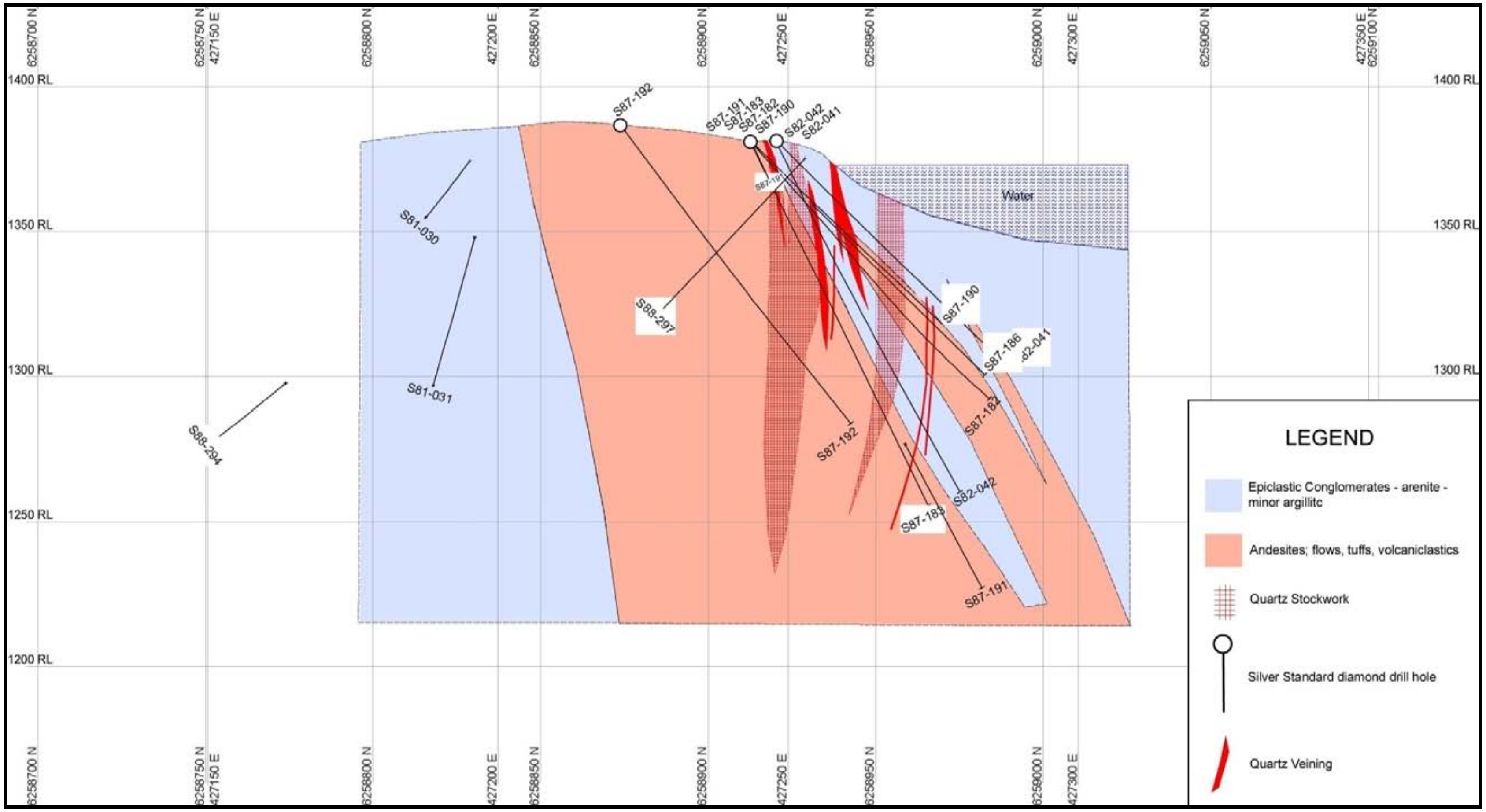
9.2.6 SHORE ZONE

A small gold-silver Resource was identified by Newhawk along the north-eastern shore of the peninsula that extends into the west end of Brucejack Lake. Referred to as the Shore Zone, it is a zone of quartz veining hosted by foliated, sericite-altered andesites with a strike length of roughly 500 m and a maximum width of 50 m (Figure 9.5). The NW-SE trend of the zone is coincident with a pronounced structural lineament (likely a shear fault) that extends from the Brucejack Fault south-eastwards beneath Brucejack Lake.

Several discrete quartz veins and quartz stockworks were traced along the zone, with historical drilling being concentrated on the southern end of the zone. The veins occur as 'stacked', en echelon, sigmoidal lenses up to 100 m in length and 1.5 m wide, although they are typically 20-40 m long. Predominantly composed of quartz with minor carbonate and barite, the veins contain podiform sulphide mineralization consisting of varying amounts of pyrite, tetrahedrite, sphalerite, galena, and arsenopyrite. Electrum has been observed in trace amounts. Silver is present in some of the highest concentrations observed in the Brucejack area.

Silver Standard has not drill-tested the Shore Zone since acquiring the Brucejack property; the gold and silver Resources calculated in 2009 for this zone were based on historical assay data from approximately 50 diamond bore holes drilled by the previous property owners.

Figure 9.5 Section 427250E of the Shore Zone, Brucejack Property – Looking West-Northwest



Source: Silver Standard.

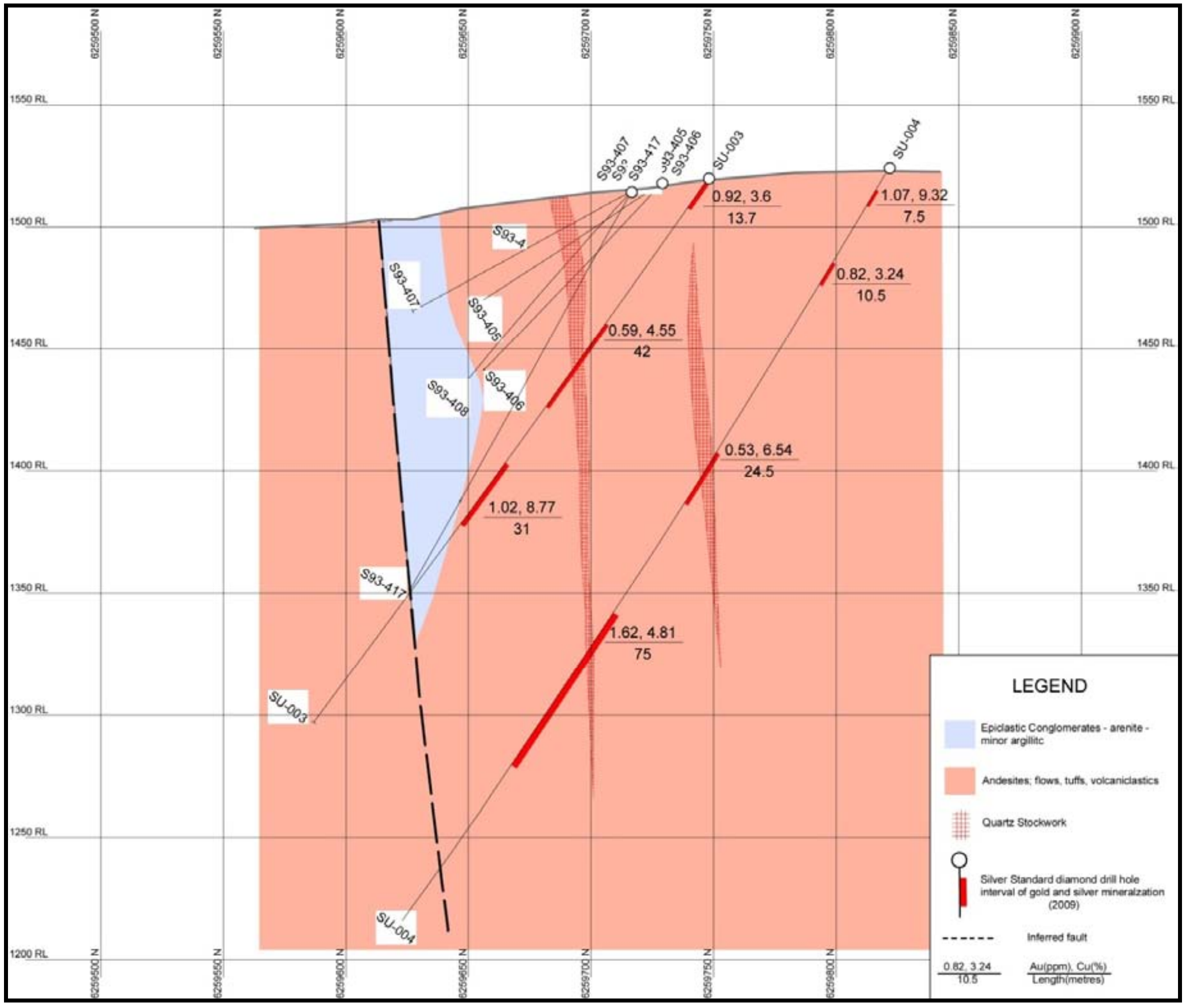
9.2.7 SG ZONE

The SG Zone is located in the north-central part of the Brucejack property and is represented by an area of iron oxide-stained, sericite-altered rocks that occur adjacent to the northerly striking Brucejack Fault. Channel rock sampling done by Silver Standard and earlier workers tested a restricted zone of quartz stockwork veining close to the major fault as well as an east-striking, 150 m-long and 20 to 80 cm-wide quartz vein that extends westwards from the stockwork.

In addition, seven historic and four Silver Standard diamond drill holes tested for gold mineralization in this area. The Silver Standard boreholes passed through a sequence of mainly clastic andesitic rocks (Figure 9.6) – likely redeposited tuff and lapilli tuff – that are intercalated with quartzo-feldspathic sandstone and minor siltstone units.

SU-004 yielded the best mineralized intersection of the four Silver Standard drillholes; 75 m averaging 1.62 g/t Au, including 27 m at 2.57 g/t Au. This intersection contains surprisingly minor quartz veining; instead, the mineralized lapilli tuff hosts minor quartz-carbonate stockwork veinlets and trace amounts of fine, acicular arsenopyrite in addition to 1-3% disseminated pyrite.

Figure 9.6 Section 426125E of the SG Zone, Brucejack Property – Looking West



Source: Silver Standard.

9.2.8 GOSSAN HILL ZONE

The mineralized zone known as Gossan Hill is a circular area, about 300 m in diameter, of intense quartz-sericite-pyrite alteration developed in Jurassic andesites of the Unuk River member of the Betty Creek formation. This visually impressive alteration zone is host to at least eleven quartz vein and quartz stockwork structures most of which trend east-west and dip steeply to the north. Individual structures are up to 250 m-long and 20 m-wide.

Historical work done at Gossan Hill consisted of rock-chip sampling, hand trenching and diamond drilling, with a few +400-m holes passing through the central part of the mineralized area. Precious metal mineralization at the Gossan Hill Zone is sporadic but generally best developed in the larger quartz lenses, particularly where these contain minor aggregates of pyrite, tetrahedrite, sphalerite, and galena. Electrum is rarely observed, while silver also occurs in tetrehedrite, pyrargyrite, and polybasite.

Silver Standard only drilled two holes at the Gossan Hill Zone, with the objective of finding a broad zone of low-grade gold mineralization that may enclose or exist between a few of the more discrete structures tested by the historical surface sampling and drilling.

10.0 EXPLORATION

10.1 SNOWFIELD PROPERTY

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.

10.2 BRUCEJACK PROPERTY

Silver Standard did not undertake any exploration or development work on the Brucejack property since acquiring the Snowfield and Brucejack properties in 1999 until June of 2009, when it commenced its 2009 field program. The 2009 program included drilling, rock-chip and channel sampling, and re-sampling of historical drill core. All drilling completed by Silver Standard is outlined in Section 11.0 of this report; all other exploration work carried out by Silver Standard is outlined below.

All historical exploration work carried out by previous owners and/or operators prior to Silver Standard's acquisition of the Brucejack property (including drilling) has been summarized in Section 6.0 of this report.

During the 2009 Brucejack property field program, Silver Standard collected a total of 1,940 drill core samples from 25 historical drill holes stored onsite and sent them for analysis to ALS Chemex Laboratories Ltd. (ALS Chemex). The samples were sent to the ALS Chemex assay laboratory in Terrace for preparation and then forwarded to the Chemex facility in Vancouver for analysis. Samples were analyzed for gold (fire assay with atomic absorption finish) as well as 33 other elements by inductively coupled plasma (ICP) analysis. The 2009 program also included re-analysis of 941 pulp samples derived from historical drill core samples. These samples were also analyzed for gold, plus 33 other elements at the Chemex facility in Vancouver.

Field work undertaken throughout the 2009 program included the collection of 2,739 rock-chip and channel samples from surface outcrops. This sampling work was mostly done at target areas that were drilled by the company in 2009, with samples generally collected along north-south oriented lines that corresponded to the surface traces of some of the 2009 drill holes. Specifically, rock-chip and channel sampling was completed at the Galena Hill, Bridge, SG, and Mammoth zones (where drilling was carried out in 2009), as well as at the Hanging Glacier Zone, where historical surface sampling had identified rocks enriched in gold and silver. The surface samples were analyzed for gold plus 33 other elements. To ensure the integrity of the analytical data, 430 quality control samples were also included (on top of the 2,739 field samples) in the field sampling program (Burk, 2009b).

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, Skeena Mining Division, British Columbia, Canada" (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.

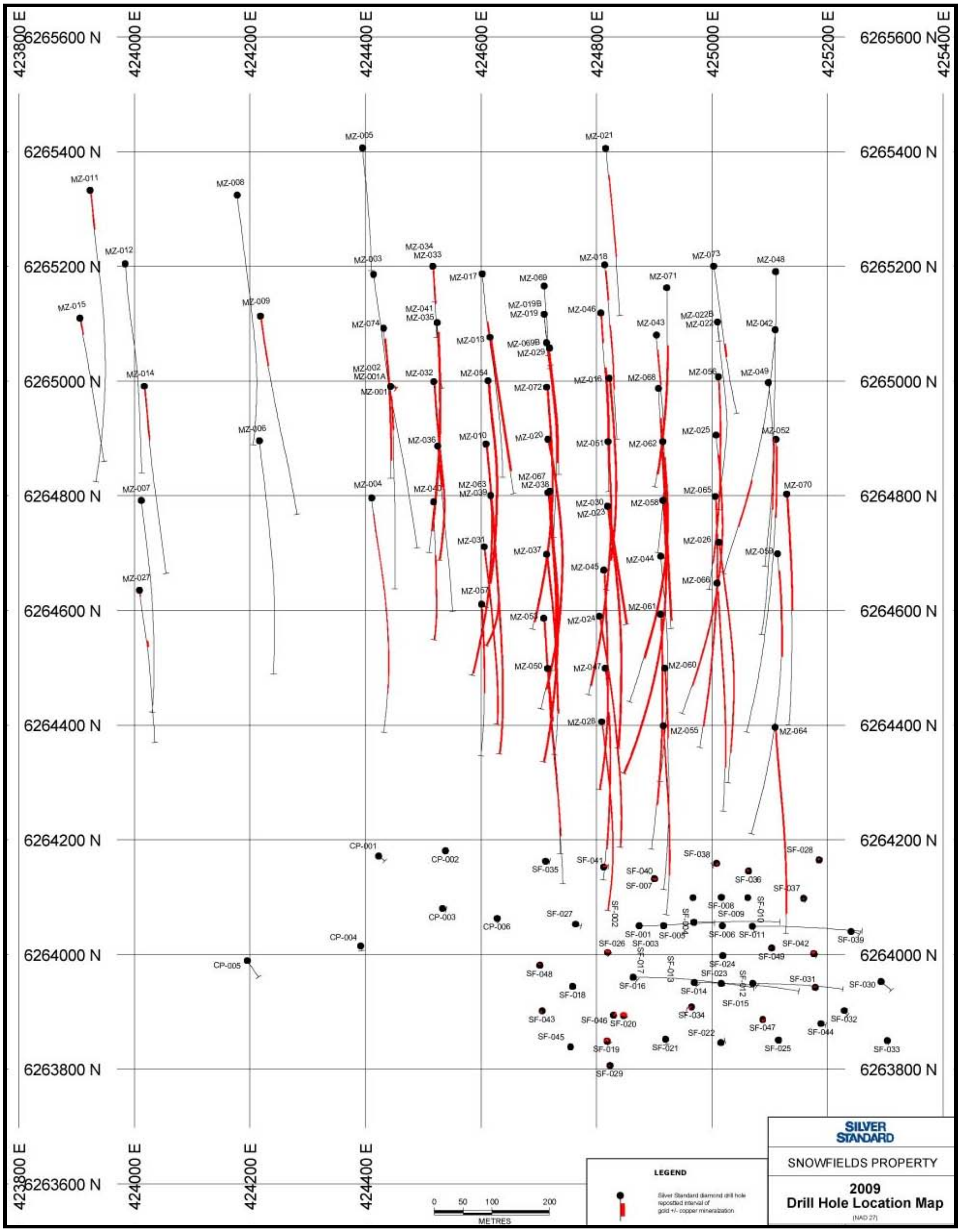
The 2008 drill program consisted of 6,945 m of drilling in 31 holes. Down-hole, E-Z shot surveying of all holes showed that deviation on azimuths was a maximum of 15° for a 700 m long hole, with little movement on dip. Core recovery was excellent at ±95%.

Drill hole collars were surveyed toward the end of the drilling campaign by McElhanney 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 Snowfield Drilling



11.3 2009 Brucejack Diamond Drilling Program

11.3.1 *Introduction*

More than 900 surface and underground diamond drill holes were drilled in the Brucejack area prior to Silver Standard's involvement commencing September of 1999. Drilling within the Brucejack property prior to this date has been summarized in Section 6.0 of this report. Of the historical holes, 432 underground and 333 surface drill holes were incorporated into the current Resource Estimates by P&E (Section 17.0), which serve as the basis for this PA.

No drilling was carried out on the Brucejack property by Silver Standard from the time of acquisition in September 1999 until 2009. In 2009, Silver Standard opted to focus its drilling program at the Galena Hill Zone, as well as on the newly discovered Bridge Zone, located in the southern part of the Brucejack property. Other targets tested by the 2009 campaign included the previously drilled Gossan Hill and SG zones as well as two areas of hydrothermal alteration and sporadic gold mineralization situated west and north of the Bridge Zone (Mammoth and Electrum prospects).

Matrix Diamond Drilling of Kimberly, BC, was commissioned to drill a minimum of 8,000 m of diamond drilling to test several gold-silver targets within the Brucejack area. Helicopter supported drilling commenced in July 2009 with two drills in the area. The number of drills later increased to three in August 2009 after continued success of the program and the discovery of the Bridge Zone (an expansion of previously defined gold mineralization known as the Electrum Zone).

The 2009 Brucejack property drilling program comprised 37 surface diamond drill holes (mainly HQ- and some NQ-diameter), SU-01 to SU-37, totalling 17,845.71 m in length, all of which intersected gold-silver mineralization. Out of these 37 holes, 35 were used in the current Resource Estimates by P&E (Section 17.0). The 2009 drill program succeeded in identifying and defining previously undefined gold targets, as well as intersecting gold mineralization over significant intervals, with some intersections exceeding 500 m. Drilling results have been summarized by zone in Sections 11.4 to 11.10.

From the 37 drill holes, 14,085 drill core samples were sent to ALS Chemex Laboratories for analytical testing. The samples were sent to the ALS Chemex assay laboratory in Terrace for preparation and then forwarded to the Chemex facility in Vancouver for analysis. Samples were analyzed for gold by fire assay method with an atomic absorption finish and the samples were also analyzed for 33 other elements by ICP analysis. There were also 80 samples analyzed by metallics methodology (to test for the presence and amount of coarser grained gold) and 584 samples analyzed for specific gravity values.

The sampling program also included an additional 2,544 quality control samples made up of 823 standards, 888 blanks, and 833 duplicate samples (Burk, 2009b).

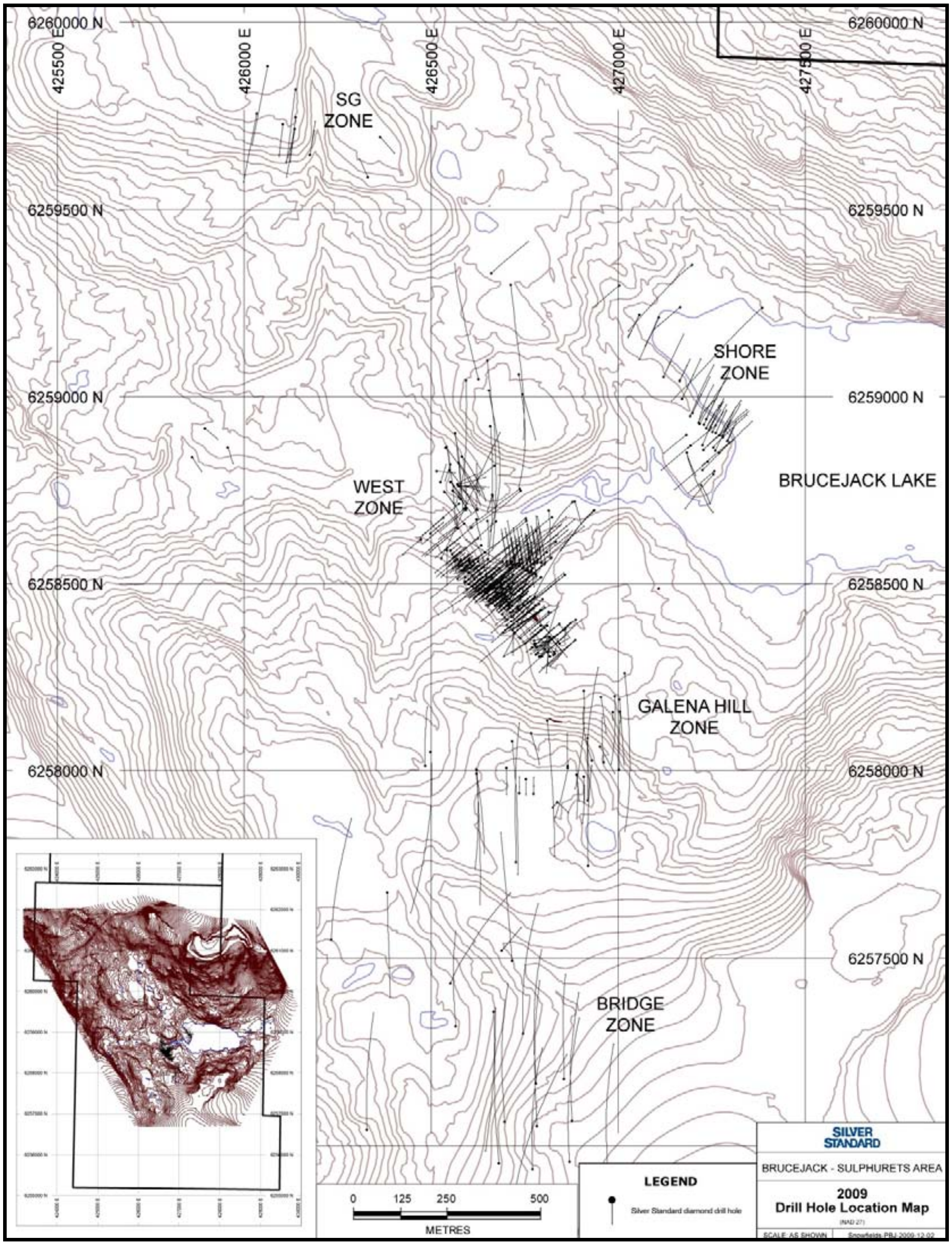
Silver Standard's quality control program for the 2009 drill program is discussed in Sections 12.0 through 14.0.

Drill hole collars were surveyed by McElhanney, based in Smithers, BC, using a Leica 500 GPS.

Down hole digital core orientation surveys were undertaken at the end of all holes and at approximate 50-m intervals on the trip out of the hole using a Reflex E-Z Shot instrument.

Figure 11.2 shows the locations of the Brucejack property diamond drill holes.

Figure 11.2 2009 Brucejack Property Drilling Program Drill Hole Layout Map



Source: Silver Standard

11.4 WEST ZONE

The West Zone, previously termed "Sulphurets", is located entirely within the south-western portion of mineral claim number 509463, approximately 500 m northwest of the Galena Hill Zone (Figure 4.1). Historically, it is the most important zone within the Brucejack property and has an extensive history of exploration and underground development. Figure 11.2 is a plan map of the drill hole layout.

A total of 1,253.94 m were drilled over two holes at the West Zone deposit during the 2009 drilling program (SU-32 and SU-36). These holes tested the southwest-trending structurally controlled vein system for extensions to the north and to depth. Drilling extended the historically defined mineralized zone by approximately 80 m to the north-west and defined an area approximately 500 m long, 75 m wide, and 450 m deep.

Of the two holes, SU-32 contained numerous intersections, the best of which was 147.5 m of 1.32 g/t Au, including 64.5 m of 2.03 g/t Au. Hole SU-32 also ended in mineralization. Select intersections from the West Zone drilling program have been summarized in Table 11.1.

Table 11.1 2009 West Zone Mineralized Intersections (Average Grades)

Hole No.	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)
SU-32 [1, 2]	138.5	161.0	22.5	0.58	7.2
	194.0	214.0	20.0	0.70	22.4
	327.0	474.5	147.5	1.32	37.4
incl.	336.5	401.0	64.5	2.03	19.3
	526.5	566.5	40.0	3.00	9.1
SU-36	49.5	69.5	20.0	1.57	12.9
	89.0	114.5	25.5	0.97	4.5
	153.5	338.0	184.5	0.65	5.9
	630.5	650.0	19.5	1.65	7.3

Note: true thickness to be determined.
[1] ended in mineralization.
[2] for the quoted average gold assays, any assay in excess of 31.1 g/t Au was cut to 31.1 g/t Au.

11.5 BRIDGE ZONE

The Bridge Zone is located approximately 1,200 m south of the West Zone, overlapping mineral claim numbers 509506 and 509464 (Figure 4.1). This zone incorporates the older known Electrum Zone, forming its northern extent, plus the newly discerned southern extension, recently discovered by Silver Standard during the 2009 drilling program.

A total of 8,616.13 m of drilling was carried out in the Bridge Zone over 16 drill holes, with the majority of holes being drilled to the north. Drilling was designed at approximately 100 m centres to help define mineralization within the zone. The 16 mineralized drill intercepts identified a 780 m-long, 400 m-wide and 800 m-deep zone of gold-silver mineralization primarily associated with moderate sericite-chlorite alteration of the plagioclase hornblende intrusive host rock. This zone is open to the south and to the east.

Select intersections from the 2009 Bridge Zone drilling program have been summarized in Table 11.2.

Table 11.2 2009 Bridge Zone Mineralized Intersections (Average Grades)

Hole No.	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)
SU-10 [1]	7.0	489.8	482.8	0.70	8.0
incl.	346.0	395.5	49.5	1.26	16.1
incl.	457.0	489.8	32.8	1.25	7.8
SU-11	18.0	72.0	54.0	1.51	11.1
	108.0	124.5	16.5	1.05	13.3
	345.0	387.0	42.0	0.52	3.4
SU-19 [1]	4.0	556.3	552.3	0.87	5.7
	296.5	556.3	259.8	1.19	6.8
SU-20	0.0	93.0	93.0	0.99	35.8
	145.5	360.5	215.0	0.76	10.9
	441.0	590.4	149.4	0.63	7.9
SU-21 [1, 2]	22.2	611.4	589.2	0.99	12.4
incl.	356.7	591.9	235.2	1.43	12.0
SU-22	31.5	368.6	337.1	0.78	19.7
incl.	267.5	279.5	12.0	2.25	164.5
	396.5	456.5	60.0	0.80	13.3
SU-23	190.5	268.5	78.0	0.85	9.5
	308.5	536.0	227.5	1.12	9.3
SU-24 [1]	146.0	350.5	204.5	0.58	10.9
	389.5	508.5	119.0	0.75	15.5
SU-25	0.0	161.0	161.0	1.26	12.1
incl.	101.1	128.1	27.0	3.09	20.6
	296.0	400.5	104.5	0.73	11.3
SU-26	38.5	108.5	70.0	0.52	7.0
	220.8	269.7	48.9	0.63	9.4
SU-27	0.9	58.0	57.1	0.86	4.2
	119.0	168.0	49.0	0.49	4.5
	226.7	411.0	184.3	0.67	4.0

table continues...

Hole No.	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)
SU-28(1)	0.0	23.5	23.5	3.92	71.1
	164.0	512.0	348.0	0.70	3.3
incl.	222.5	512.0	289.5	0.76	3.5
	557.0	595.9	38.9	0.74	5.8
SU-29 [2]	0.0	28.5	28.5	1.55	3.0
	211.5	270.5	59.0	1.58	14.1
SU-30 [1]	0.9	514.0	513.0	0.99	4.1
incl.	10.5	49.7	39.2	1.40	7.9
incl.	401.1	478.5	77.4	1.38	5.0
	607.0	661.6	54.6	0.52	5.3
SU-31	52.0	77.5	25.5	1.08	4.4
	123.6	140.5	16.9	0.83	21.5

Note: true thickness to be determined.
[1] ended in mineralization.
[2] for the quoted average gold assays, any assay in excess of 31.1 g/t Au was cut to 31.1 g/t Au.

11.6 Galena Hill Zone

This previously identified 200 m-wide by 400 m-long north-east to south-west striking zone of mineralization hosts a series of at least eight steeply dipping sulphide-bearing quartz veins up to 285 m in length and up to 8 m wide. The objective of the 2009 drilling program undertaken at the Galena Hill Zone was to test for gold mineralization and further define this zone.

A total of 5,238.27 m of drilling was carried out in the Galena Hill Zone over a total of 12 drill holes, with all holes being drilled either in a northerly or southerly direction. All 12 drill holes intercepted significant gold-silver mineralization with some holes intersecting quartz veins containing visible gold. Three of the 12 final holes ended in mineralization.

Select intersections from the drilling carried out at the Galena Hill Zone have been summarized in Table 11.3.

Table 11.3 2009 Galena Hill Zone Mineralized Intersections (Average Grades)

Hole No.	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)
SU-05	39.2	51.5	12.3	0.61	13.3
	323.5	478.5	155.0	1.26	20.4
incl.	466.5	478.5	12.0	5.37	26.3
SU-06	40.0	84.5	44.5	1.83	20.3
incl.	56.7	68.0	11.3	4.65	55.2
	146.0	225.5	79.5	1.01	10.5
	278.9	356.5	77.6	1.01	5.5
SU-07	162.5	207.5	45.0	0.61	6.1
SU-08	100.0	133.5	33.5	0.85	6.6
	202.5	286.0	83.5	0.76	11.5
SU-09	106.5	124.5	18.0	1.04	34.0
	250.5	262.5	12.0	0.67	19.7
	363.5	384.5	21.0	0.85	20.6
SU-12 [2]	258.0	278.6	20.6	5.33	158.8
incl.	273.0	274.5	1.5	16,949.00	8,696.0
	301.0	323.8	22.8	1.02	10.2
	354.4	373.5	19.1	2.64	9.7
	460.0	502.0	42.0	1.59	8.4
SU-17 [1]	113.0	203.4	90.4	1.13	12.6
SU-29 [2]	430.1	473.0	42.9	0.80	9.2
	530.0	571.5	41.5	1.72	56.5
incl.	560.8	561.3	0.5	5,344.00	3,740.0
SU-33	57.1	128.1	71	2.17	25.2
	95	110	15	6.27	66.4
SU-34 [1]	269.5	290.5	21	0.89	7.6
	312.5	350	37.5	1.23	5.6
SU-35 [1]	247	261	14	1.69	38.8
	282.5	305.1	22.6	2.11	5.4
incl.	292.4	305.1	12.7	3.26	6.23
SU-37	12.9	141	128.1	0.64	8.3

Note: true thickness to be determined.
[1] ended in mineralization.
[2] for the quoted average gold assays, any assay in excess of 31.1 g/t Au was cut to 31.1 g/t Au.

11.7 SHORE ZONE

No drilling was carried out at the Shore Zone during 2009; all drilling within this zone was undertaken prior to Silver Standard's involvement in the Brucejack property.

11.8 SG Zone

The SG Zone, which is located approximately 1,200 m north of the West Zone, was also tested with four drill holes over 1,271.97 m.

Select intersections from the 2009 drilling at the Galena Hill Zone have been summarized in Table 11.4.

Table 11.4 2009 SG Zone Mineralized Intersections (Average Grades)

Hole No.	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)
SU-01	1.5	40.5	39.0	1.06	3.5
	150.0	178.5	28.5	0.88	1.5
Incl.	150.0	162.0	12.0	1.28	2.2
SU-03	1.3	15.0	13.7	0.92	3.6
	73.5	115.5	42.0	0.58	4.6
	144.0	175.0	31.0	1.02	8.8
SU-04	11.0	18.5	7.5	1.07	9.3
	45.5	56.0	10.5	0.82	3.2
	137.0	161.5	24.5	0.53	6.5
	215.0	290.0	75.0	1.62	4.8
Incl.	228.0	255.5	27.0	2.57	6.2

Note: true thickness to be determined.

11.9 Gossan Hill Zone

No drilling was carried out at the Gossan Hill Zone during 2009; all drilling within this zone was undertaken prior to Silver Standard's involvement in the Brucejack property.

11.10 Mammoth Zone

The 2009 drilling program also included three drill holes (totalling 1,543.57 m) testing for mineralization at the Mammoth Zone, located on the western side of the Brucejack Fault at approximately the same northing as the Bridge Zone (Figure 4.1). Table 11.5 shows select mineralized intersections from the 2009 drilling program at the Mammoth Zone.

Table 11.5 2009 Mammoth Zone Mineralized Intersections (Average Grades)

Hole No.	From (m)	To (m)	Interval (m)	Gold (g/t)	Silver (g/t)
SU-13	147.0	161.0	14.0	0.98	1.7
	238.5	295.5	57.0	1.21	3.0
SU-14	41.0	45.5	4.5	5.42	5.8
	304.4	323.0	18.6	1.98	2.0
SU-15	407.5	449.0	41.5	1.01	19.1
	493.0	518.0	25.0	1.24	4.1

Note: true thickness to be determined.

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 on the Snowfield-Brucejack Project used 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 atomic absorption (AA) finish. Copper was determined using four acid digest with either inductively coupled plasma atomic emission spectroscopy (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, molybdenum, and rhenium.

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-Brucejack Project 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 from both the Snowfield and Brucejack properties.

14.2 SNOWFIELD SITE VISIT

During the Snowfield site visit, four samples distributed in four holes were 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).

14.3 SILVER STANDARD QUALITY CONTROL

The quality assurance/quality control (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 (¼ split core) were inserted in every 20 samples. In addition, the laboratory inserted their own internal QC, which included standards, blanks, and both coarse reject and pulp duplicates.

14.4 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.

14.4.1 *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.

14.4.2 *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.

14.4.3 *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 (ABRD) 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 ABRD demonstrated a precision of 10% for the core duplicates and the T-H yielded a precision of 18%.

The pulp duplicate pairs yielded an ABRD 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%.

14.4.4 *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.

14.5 Brucejack Site Visit

The Brucejack property was visited by Mr. Fred Brown (CPG, Pr.Sci.Nat.) from September 9 to September 13, 2009. Independent verification sampling was done on diamond drill core for the current 2009 program, with eight samples distributed in eight holes collected for analysis. An attempt was made to sample intervals around a reported grade of 0.50 g/t Au in each of the defined zones. The chosen sample intervals were then sampled by taking quarter splits of the remaining half-split core. The samples were then documented, bagged, sealed with packing tape, and brought by Mr. Brown to the ALS Chemex laboratory in Terrace, BC.

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 (Ewert et al., 2009).

14.6 Pre-Silver Standard Historical Data & Quality Control

The Brucejack area has been the subject of intense exploration from surface and underground from 1962 through 1994, though work was not continuous throughout this period. Four different companies explored the Brucejack property, and an underground development program was completed on the West Zone by Newhawk. The total number of surface drill holes is 458, while an additional 443 holes were drilled from underground. Table 14.1 shows the number of holes drilled per zone.

Table 14.1 Historical Drill Holes by Zone

Zone	No. of Drill Holes
West	736
Galena Hill	28
SG	9
Shore	56
General Exploration	72
Total	**901**

Silver Standard retained the services of Geospark Consulting (Geospark) for the purposes of verifying all historical data to ensure its integrity for use in the Resource Estimate. All hard copies of drill logs and analytical certificates were entered in digital format in a database, and a thorough verification with respect to collar coordinates, down hole surveys, azimuths, dips, sampling, analytical methods and results was completed. Historical drill core was stored on site; however, it was impossible to resample due to distance markers in the boxes no longer being legible.

Most of the pulps from the historical drilling were available and intact and it was therefore decided to reanalyze approximately 10% of them, pro-rata to the number of holes per zone. A total of 941 pulps were sent to ALS Chemex in Vancouver, BC, for Au and Ag analysis.

Geospark ran a series of statistics on the historic versus 2009 pulp reruns in order to determine the precision between the pairs, and therefore the integrity of the historical data. Of the 901 holes drilled, 849 passed the QC review and were deemed acceptable for use in the Resource Estimate; the remaining 52 holes were excluded from the database. Complete results are presented in a report prepared by Geospark and listed in the References section of this report.

14.7 P&E Independent Data Review

P&E obtained the database from Geospark and completed an independent data verification that included employing essentially the same statistical methods used by Geospark. Keeping in mind that data from two different laboratories cannot be expected to demonstrate ideal precision, P&E agrees with Geospark's conclusions regarding the use of 849 historical drill holes in the Resource database.

14.8 2009 Data Verification Results

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

14.8.1 Performance of Certified Reference Material

For the Brucejack area drill program, two certified reference materials were purchased from CDN Resource Laboratories Ltd., which were both certified for Au, Ag, Cu, Pb, and Zn. Both the standards performed very well for Au and Ag (the other metals were not monitored). The occasional very rare failure was noted by Silver Standard, and a re-run was completed of the samples surrounding the failed standard. All data used for the Resource Estimation have passed the QC protocol.

14.8.2 Performance of Blank Material

The blank material used for the 2009 drill program was ¾″ crushed granite sold by Imasco Minerals as landscape material.

There were 879 blank sample results for both gold and silver. For gold, the blanks were generally below three times detection limit, with 29 values (3%) greater than this threshold. The highest value was 0.072 g/t Au.

For silver, there were 9 values (1%) greater than 3 times the detection limit, with a high value of 8.5 g/t Ag.

All of the failures were investigated, and many of them turned out to be misallocations (not blank samples). The actual blank failures were reviewed for their impact to the Resource and no action was necessary.

14.8.3 2009 Duplicate Statistics

For the 2009 drill program, there were 446 field core duplicate pairs and 341 pulp duplicate pairs graphed for gold, and 446 field and 231 pulp duplicate pairs graphed for silver. There were no coarse reject duplicates done.

Data for the two duplicate types were graphed and three types of graphs were created. Scatter plots were made, as well as a graph of the sample pair mean versus the ABRD of the sample pairs, and a T-H precision plot.

The ABRD and the T-H precision were in close agreement for the gold field and pulp duplicates. The ABRD for the field duplicates yielded a value of approximately 30%, and the T-H value was also 30%. The ABRD for the pulp duplicates was approximately 8%, and the T-H also yielded a value of 8%.

The ABRD and the T-H precision were in close agreement for the silver field and pulp duplicates. The ABRD for the field duplicates yielded a value of approximately 43%, and the T-H value was 46%. The ABRD for the pulp duplicates was approximately 5%, and the T-H yielded a value of 7%.

14.8.4 2009 External Checks

Silver Standard sent approximately 10% of the pulps to Assayers Canada laboratory for checks on the principal laboratory (ALS Chemex). Simple scatter plots were created for 336 pairs for gold and silver. With the exception of a rare outlier, all data fell on a 1:1 line, indicating excellent precision.

P&E concludes that the data are of good quality for use in the Snowfield and Brucejack current Resource Estimates.

15.0 ADJACENT PROPERTIES

Within the adjacent KSM property there are three notable copper-gold mineral deposits, namely 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. 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 oz of gold, 5.3 B lb of copper, 2.8 M oz of silver, and 1.9 M lb 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 QPs for this report have not verified the information concerning Seabridge, and the information is not necessarily indicative of the mineralization on the Snowfield property.

Table 15.1 Seabridge 2009 Mitchell Resources

Zone	Measured Mineral Resources					Indicated Mineral Resources					Inferred Mineral Resources				
	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

Zone	Measured Mineral Resources					Inferred Mineral Resources				
	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

Zone	Reserve	Mt	In Situ Average Grades				Contained Metal			
			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

Process Research Associates Ltd. (PRA), the Metallurgical Division at Inspectorate America Corp. (Inspectorate), carried out preliminary metallurgical testwork investigating the metallurgical performance on the Snowfield and Brucejack mineralization since early 2009 and late 2009, respectively. PRA is an industrial research laboratory established in 1992 that 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. IPL is an ISO 9001:2000 certified company. The testwork was conducted under the supervision of Frank Wright, P.Eng.

The mineralization from the Snowfield deposit contains gold, copper, silver, molybdenum, and rhenium recoverable metals. The testing program consisted of preliminary mineralization characteristic determination, copper/gold/molybdenum bulk flotation and copper/molybdenum separation 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. There were three testing programs conducted on the Snowfield mineralization. The test results and procedures, including sample preparation and analysis, are presented in three data reports by PRA released in March and July 2010.

The key valuable metals in the mineralization from the Brucejack deposit are gold and silver. The testwork conducted on the Brucejack mineralization was similar to that on the Snowfield mineralization. The testing focuses more on the cyanidation and gravity concentration. The test results and procedures, including sample preparation and analysis, are presented in the data reports by PRA released in July 2010.

Wardrop has reviewed the testwork data reports and summarized the results in the following sections based on mineralization deposits.

16.1.2 *SNOWFIELD MINERALIZATION*

SAMPLE DESCRIPTION

The metallurgical samples were collected from two major mineralization zones: the North Zone (Main Zone) and the Upper (Molybdenum) Zone. The drill holes included MZ 013, MZ 016, MZ 030, MZ 031, MZ 038, MZ 041, MZ 051, MZ 054, MZ 058, MZ 068, MZ 070, SF 002, SF 004, SF 013, SF 016, and SF 023. The drill hole distribution is presented in Section 11.0 (Figure 11.1).

North (Main) Zone Samples

A total of approximately 367 kg of assay reject samples were collected from four drill holes from the North (Main) Zone for two early testing programs. Six intervals of assay reject 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 Assay Reject Interval Sample Information – Snowfield

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. The blending ratios of composite samples are shown in Table 16.2.

Table 16.2 Blend Sample Composition – Snowfield

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 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**
Comp 5		
MZ 13B	21.2	63.9
MZ 16B	12.0	36.1
Total	**33.2**	**100.0**

In December 2009, a total of approximately 2,400 kg assay reject samples from the Snowfield North Zone were composed into 18 composite samples for additional testing. The sample identification, drill hole identification, and sample elevation are shown in Table 16.3. Four master composite samples (identified as Comp NZ1, Comp D, Comp M, and Comp S) were further generated from most of the 18 composite samples for pilot plant tests and bench scale tests. The blending details for the master composites are shown in Table 16.4.

Table 16.3 Pilot Plant Test and Further Test Samples – Snowfield

Sample ID	Drill Hole ID	Drill Hole Interval
MZ 38D	MZ-038	Lower
MZ 38M	MZ-038	Middle
MZ 38S	MZ-038	Upper
MZ 41M	MZ-041	Middle
MZ 41S	MZ-041	Upper
MZ 51D	MZ-051	Lower
MZ 51M	MZ-051	Middle
MZ 51S	MZ-051	Upper

table continues…

Sample ID	Drill Hole ID	Drill Hole Interval
MZ 54D	MZ-054	Lower
MZ 54M	MZ-054	Middle
MZ 54S	MZ-054	Upper
MZ 58D	MZ-058	Lower
MZ 58M	MZ-058	Middle
MZ 58S	MZ-058	Upper
MZ 68D	MZ-068	Lower
MZ 68M	MZ-068	Middle
MZ 68S	MZ-068	Upper
MZ 70S	MZ-070	Upper

Table 16.4 Blend Sample Composition – Snowfield

Master Composite ID	Sample Source	Blending Ratio
Comp S	MZ-38S, MZ-51S, MZ-54S, MZ-58S, MZ-68S	Variable
Comp M	MZ-38M, MZ-51M, MZ-54M, MZ-58M, MZ-68M	Variable
Comp D	MZ-38D, MZ-51D, MZ-54D, MZ-58D, MZ-68D	Variable
Comp NZ1	Comp S, Comp M, Comp D	1: 1: 1

Upper (Molybdenum) Zone Samples

A total of 609 kg of assay reject 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 SF 16 at an equal weight ratio.

SAMPLE HEAD ANALYSES

The key assay results for each composite sample are shown in Table 16.5, which indicates that the contents of the main value elements (copper, gold, and molybdenum) 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 0.05% and 0.35%, and molybdenum contents fluctuated from 12 ppm to 194 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 indicated that the mineralization contains approximately 0.39 ppm to 1.27 ppm rhenium.

Table 16.5 Metal and Sulphur Concentrations of the Blended Samples – Snowfield

Sample ID	Au (g/t)	Cu (%)	Mo (ppm)	S (T) (%)	Fe (%)	Re (ppm)	Ag* (ppm)	Sb (%)	As (%)	Hg (ppm)	Pb* (ppm)	Zn* (ppm)
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
Comp NZ1	0.93	0.19	90	3.8	4.6	0.53	2.7	<5*	21*	<3*	37	258
Comp S	0.82	0.15	80	3.4	5.0	0.55	4.2	<5*	13*	<3*	53	392
Comp M	0.97	0.18	90	4.0	4.3	0.53	3.9	<5*	23*	<3*	39	228
Comp D	0.92	0.19	80	3.5	4.3	0.39	5.0	<5*	25*	<3*	20	142

*by inductively coupled plasma (ICP).

GRINDABILITY TESTWORK

PRA conducted preliminary grindability testwork to determine the Bond ball mill work index on the samples from the North Zone of Snowfield deposit. Table 16.6 presents the grinding work index. It appears that on average, the mineralization of the Snowfield North Zone is moderately hard.

Table 16.6 Bond Ball Mill Work Index – North Zone

Sample ID	Bond Ball Mill Work Index (kWh/t)
MZ 13A	13.6
MZ 16B	16.6
MZ 30B	15.7
MZ 31A	17.0
MZ 31B	17.2

SAMPLE SPECIFIC GRAVITY

Drill core interval samples from both mineralization zones were tested for specific gravity (SG). The results are shown in Table 16.7. 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 16.7 Specific Gravity – Snowfield North and Upper Zones

North Zone						Upper Zone	
Sample ID	SG	Sample ID	SG	Sample ID	SG	Sample ID	SG
MZ13A	2.82	MZ 38S	2.83	MZ 54S	2.81	SF 02	2.84
MZ 13B	2.76	MZ 41S	2.83	MZ 58D	2.80	SF 04	2.91
MZ 16B	2.81	MZ 41M	2.80	MZ 58M	2.78	SF 13	2.84
MZ 30B	2.79	MZ 51D	2.81	MZ 58S	2.78	SF 16	2.85
MZ 31A	2.77	MZ 51M	2.75	MZ 68D	2.77	SF 23	2.76
MZ 31B	2.78	MZ 51S	2.77	MZ 68M	2.79		
MZ 38D	2.81	MZ 54D	2.79	MZ 68S	2.82		
MZ 38M	2.78	MZ 54M	2.77	MZ 70S	2.88		

FLOTATION TESTWORK

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

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

Three locked cycle flotation tests were also conducted on the composites (Comp 3, Comp 4, and Comp NZ1) generated from the two major mineralization zones.

Flotation Testwork on Drill Core Interval Samples

The testing included copper/gold/molybdenum flotation (bulk flotation) consisting of rougher flotation, rougher concentrate regrinding, and subsequent cleaner flotation, and 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 16.8 shows the metal recoveries to the bulk rougher concentrates from both mineralization zones.

Table 16.8 Recoveries to Bulk Rougher Concentrate – Snowfield

Sample ID	Metal Distribution (%)				Mass Recovery (%)
	Au	Cu	Mo	Fe	
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
MZ 41S	65	77	77	47	9.6
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 65% 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 sample SF23. 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 Upper 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 16.9.

Table 16.9 Metal Recoveries to Gold Bearing Pyrite Concentrates – Snowfield

Sample ID	Metal Distribution (%)			Mass 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
MZ 41S	14.8	11.3	9.0	14.2
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 4% 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 and the other major valuable element concentrations of the final cleaner concentrates are listed in Table 16.10.

Table 16.10 Metal Grades of Cleaner Concentrate – Snowfield

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
MZ 41S	76.7	25.6	2.72	N/A

table continues…

Sample ID	Metal Grade			
	Au (g/t)	Cu (%)	Mo (%)	Re (ppm)
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, MZ 31B, and MZ 41S, which produced the concentrates with a copper grade higher than 25%. 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 minutes 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 to Figure 16.4.

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

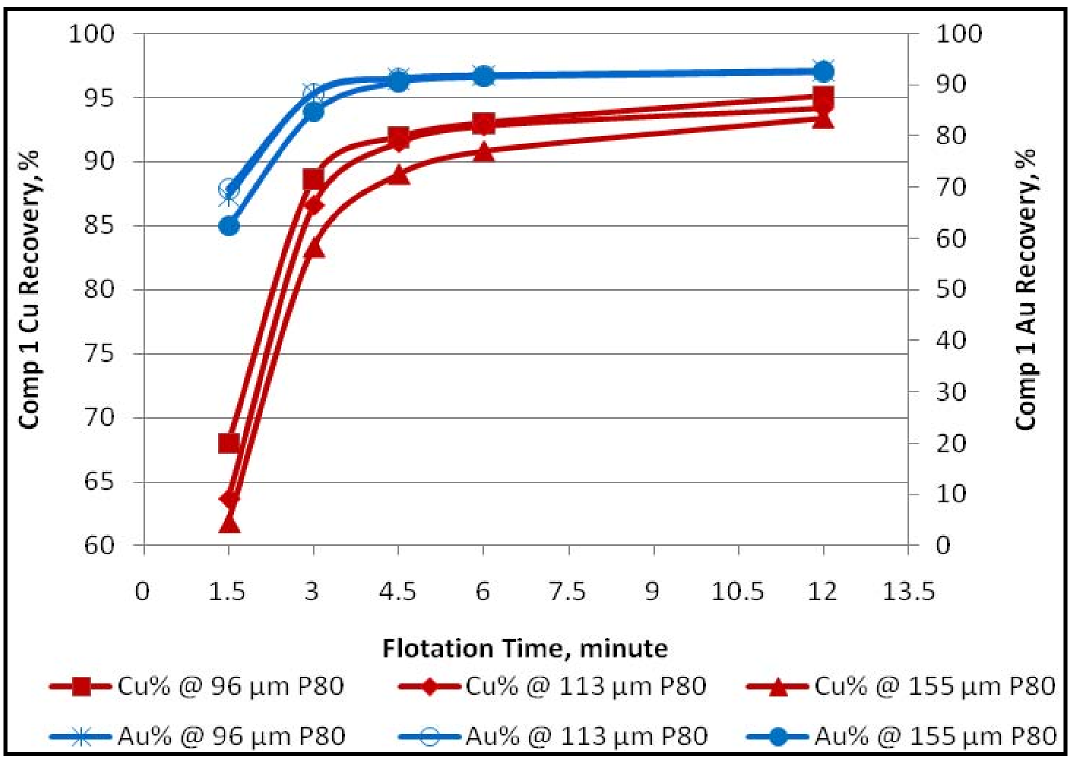


Figure 16.2 Flotation Kinetics at Varied Primary Grinding Particle Size – Comp 2

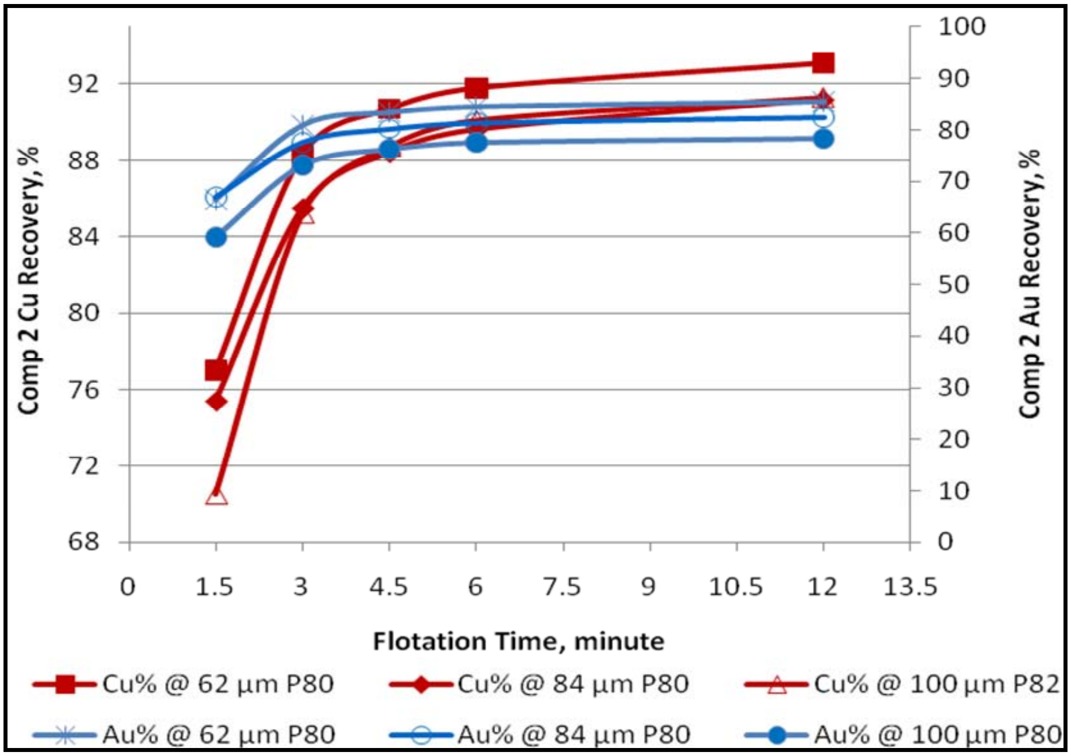


Figure 16.3 Flotation Kinetics at Varied Primary Grinding Particle Size – Comp 3

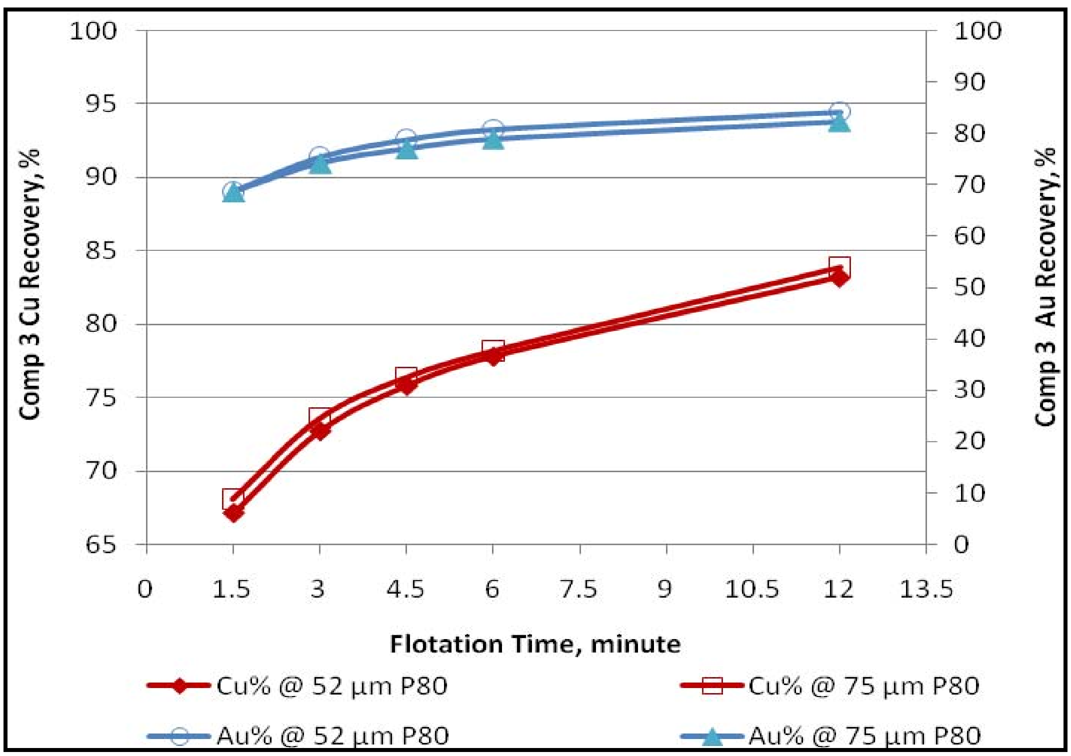
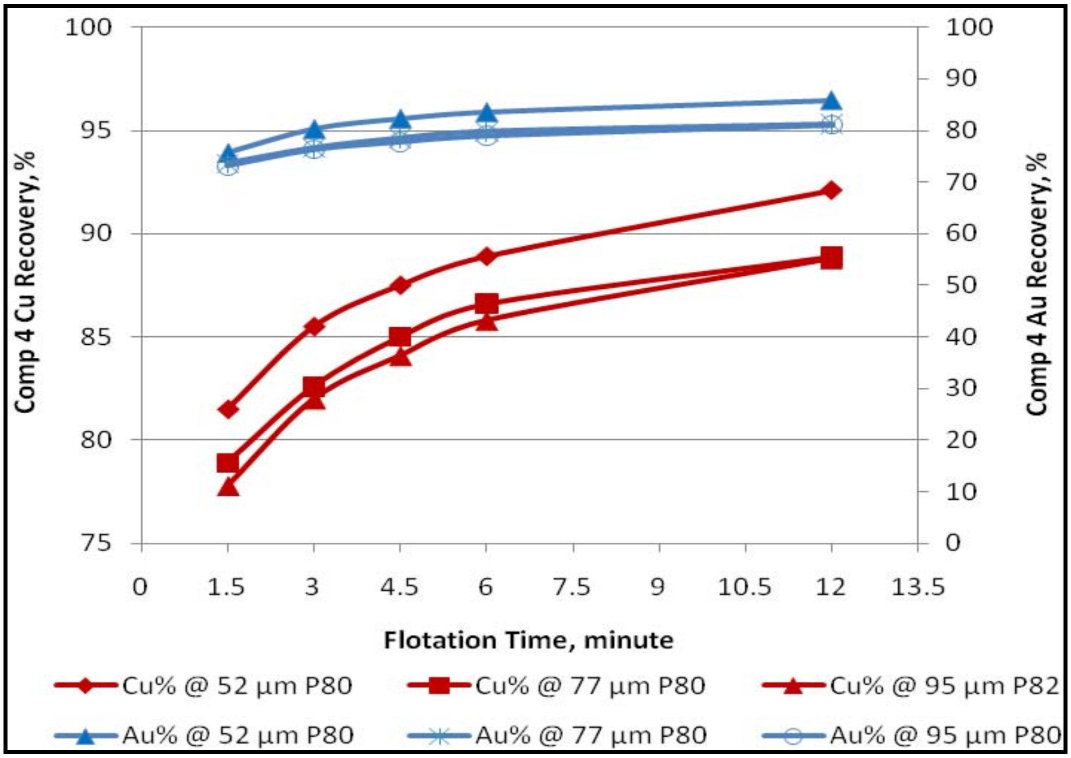


Figure 16.4 Flotation Kinetics at Varied Primary Grinding Particle Size – Comp 4



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

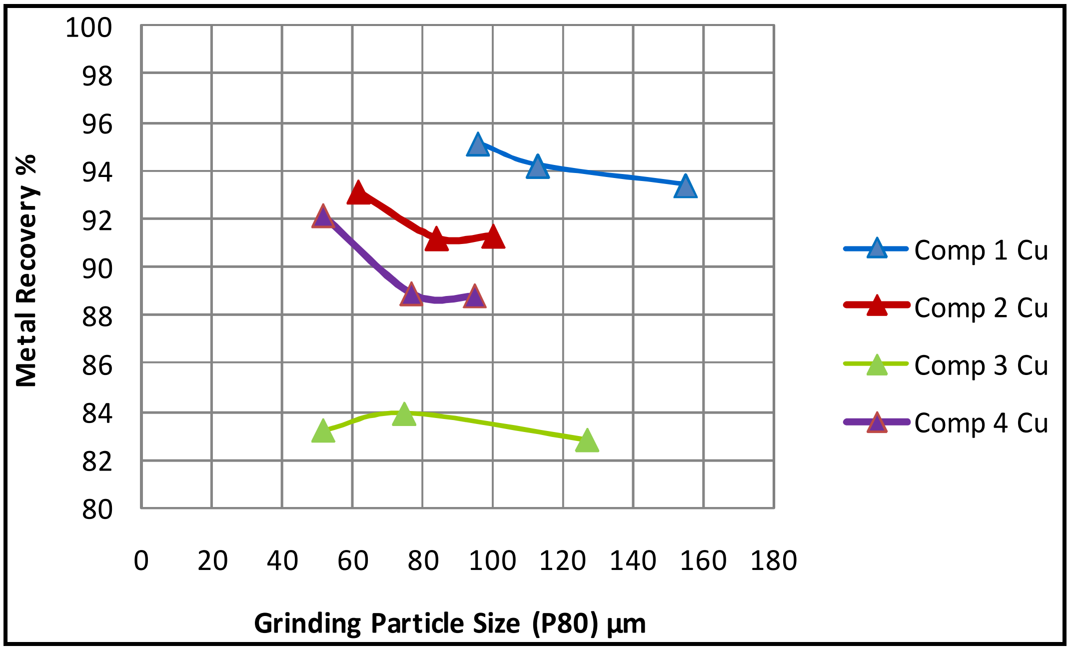


Figure 16.6 Gold Recovery vs. Primary Grinding Particle Size

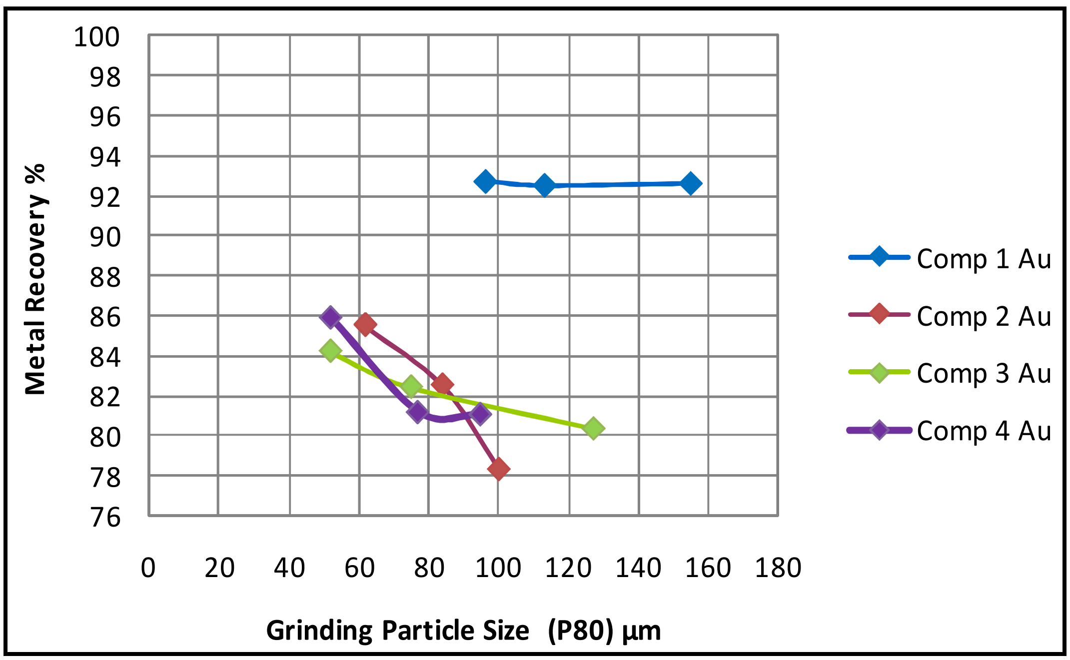
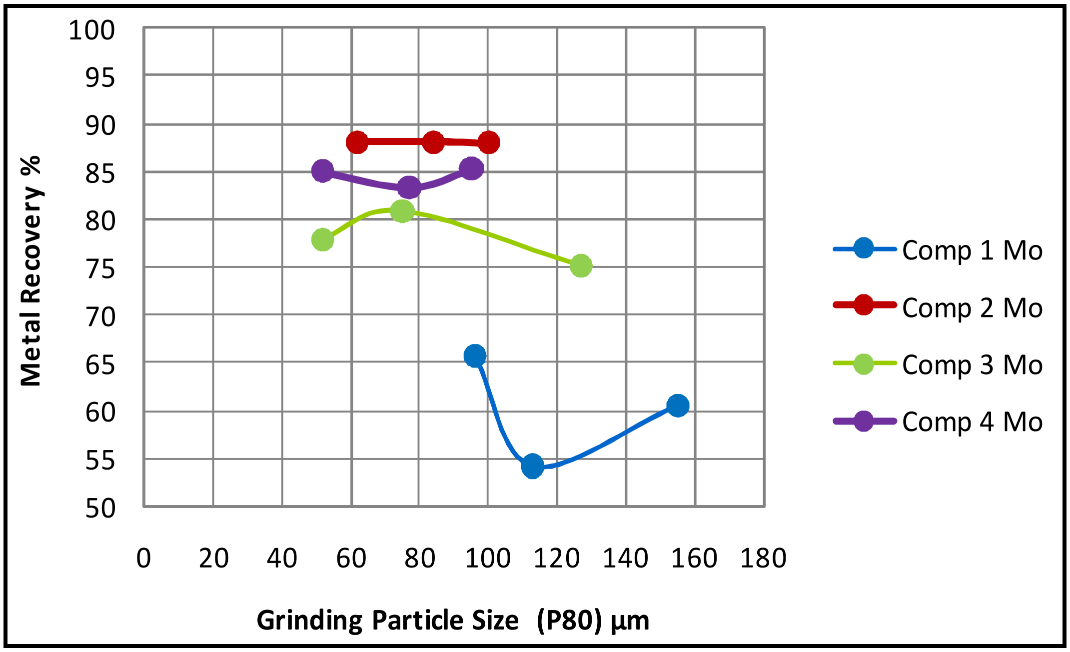


Figure 16.7 Molybdenum Recovery vs. Primary Grind Size



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 16.11 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. The results were obtained from Comp 3, Comp 4, Comp 5, and SF 23 composites.

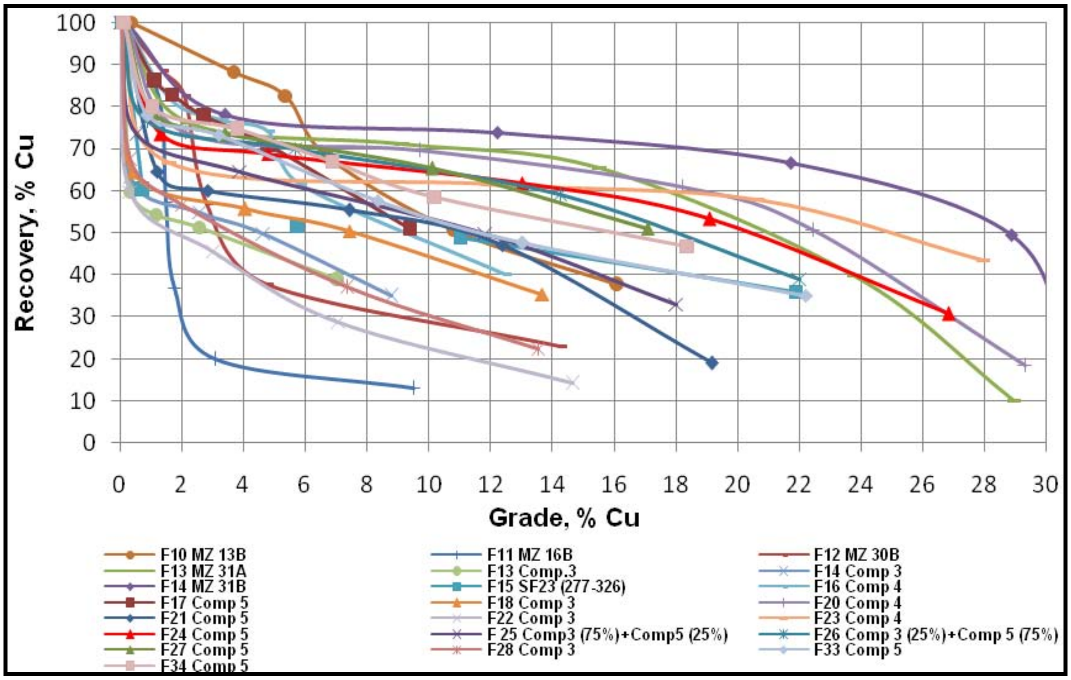
Table 16.11 Metal Recoveries to Different Flotation Concentrates – Snowfield

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.

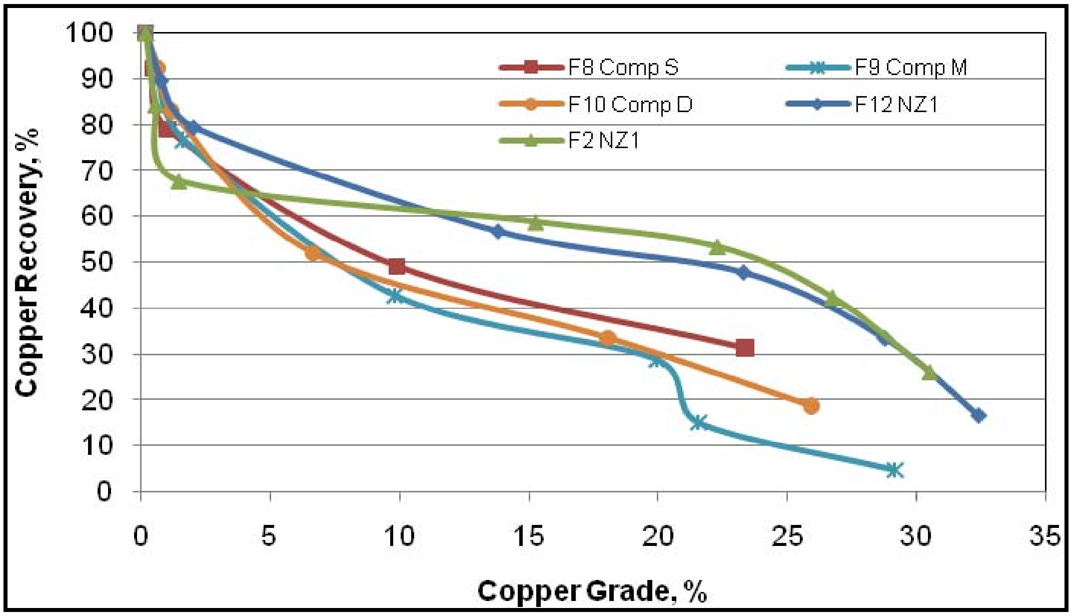
Figure 16.8 Copper Recovery vs. Copper Grade – Open Batch Tests



The data from the open batch flotation tests indicates that six of the samples were able to produce concentrates with grades greater than 22% Cu. The Comp 4 sample had the best metallurgical performance while the Comp 3 sample responded poorly to the test conditions. The poor metallurgical responses were possibly caused by a low head copper grade.

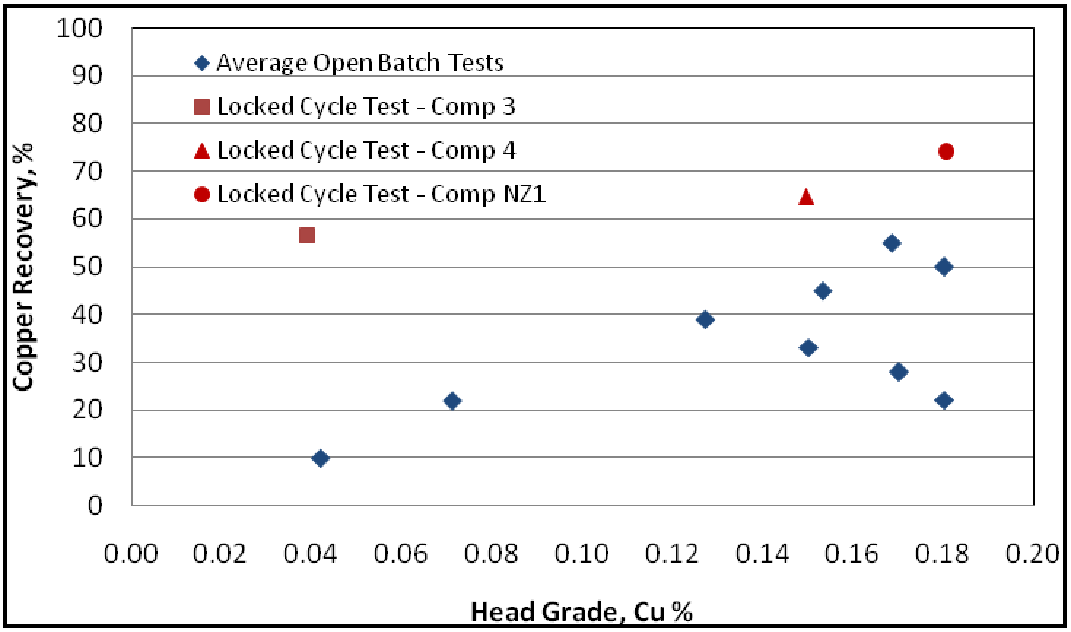
The latter test results, which were obtained from Comp NZ1, Comp D, Comp M, and Comp S samples of the North Zone of Snowfield deposit, are shown in Figure 16.9. The data show that all the samples were able to produce a copper concentrate of higher than 22% Cu. The copper grade of the concentrate from Comp NT1 was higher than 30% Cu.

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



According to the open cycle tests, the copper recovery (open batch flotation tests) from the composite samples at the concentrate grade of 22% Cu is projected and shown in Figure 16.10. The results show that copper recovery reporting to copper-gold concentrate is closely related to copper head grade, except for Comp M and Comp D samples. The copper recoveries from locked cycle tests (LCTs) are shown in Figure 16.10.

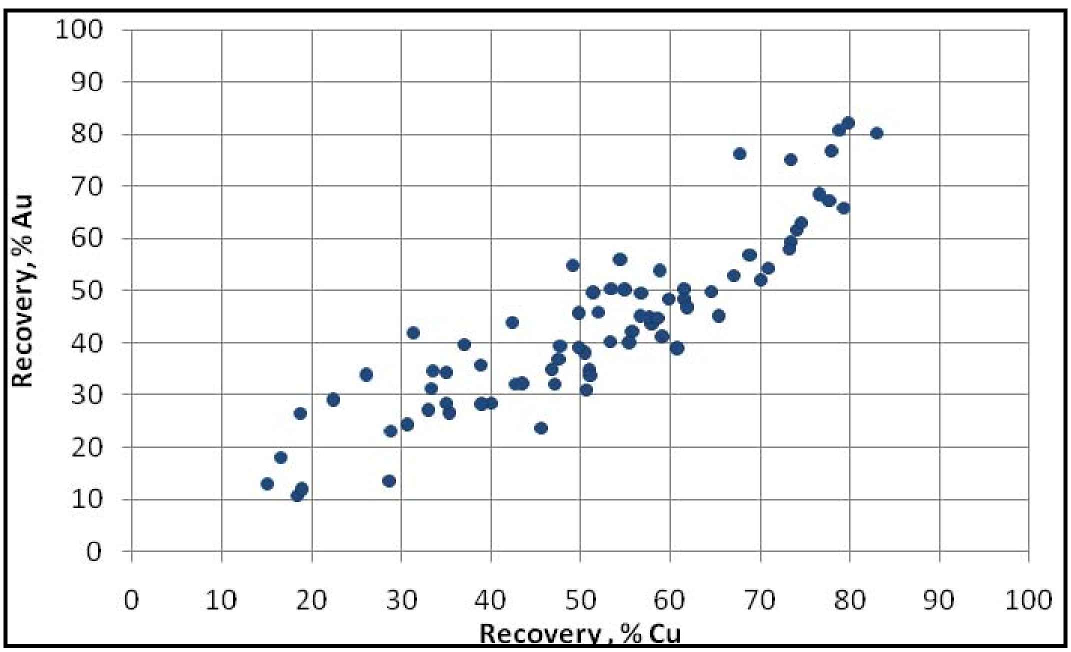
Figure 16.10 Projected Copper Recovery vs. Head Grade – Open Batch Tests



Note:
LCT Comp 3: 6.4% Cu concentrate grade.
LCT Comp 4: 25.8% Cu concentrate grade.
LCT Comp NZ1: 29.6% Cu concentrate grade.

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 16.11.

Figure 16.11 Gold Recovery vs. Copper Recovery – Open Batch Tests



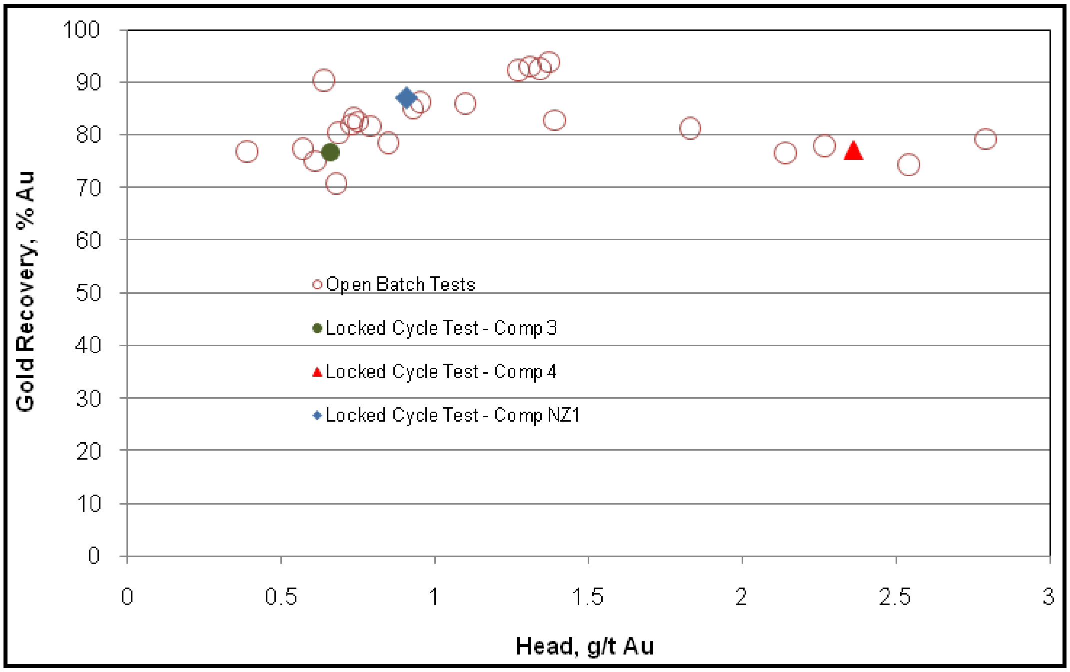
The results were obtained from Comp 3, Comp 4, Comp 5, and SF 23 composites. Table 16.11 also shows that molybdenum grade of the cleaner concentrates is high, ranging from more than 1% Mo for the Comp 4 and Comp 5 samples to 16% Mo for the Comp 3 sample. On average, approximately 72% of the molybdenum was recovered to the bulk rougher flotation concentrate from the composite samples.

Gold Bearing Pyrite Flotation Test Results

As shown in Table 16.11, 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 16.12. The total gold recoveries from locked cycle tests are also plotted in Figure 16.12.

Figure 16.12 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 higher 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, although 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.

Cu-Mo Separation Testwork

One copper-molybdenum separation test was conducted using a middling sample (0.057% Mo) generated from a pilot plant test (no data available for review). The test included two stages:

- upgrading the concentrate from the pilot plant tests
- copper-molybdenum separation.

Sodium hydrosulphide (NaHS) (45 g/t to 90 g/t at each cleaner stage) was used to suppress the copper minerals. The test only produced a low grade molybdenum concentrate containing 15.2% Mo after three stages of cleaner flotation. The molybdenum recovery is low as well, only 2.2% of the molybdenum in the middling sample reported to the final concentrate.

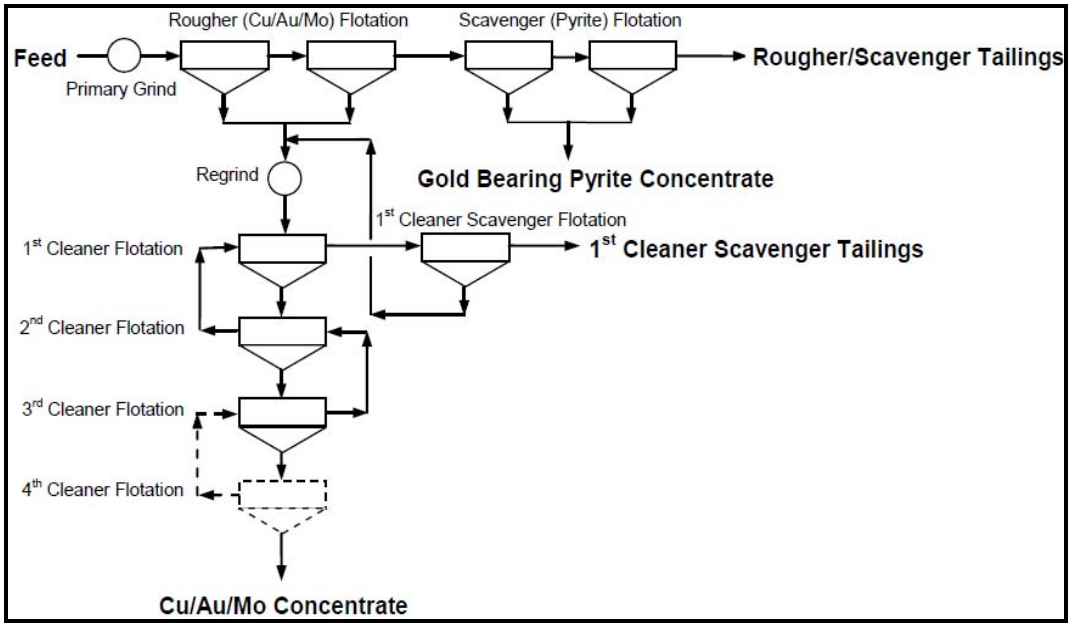
The copper-molybdenum separation testing program is ongoing.

Locked Cycle Flotation Testwork

Three locked cycle flotation tests were carried out on Comp 3, Comp 4, and Comp NZ1 samples using the same reagent scheme that was developed from the open batch tests. The flotation flowsheet, as shown in Figure 16.13, included:

- copper/gold/molybdenum bulk rougher flotation
- bulk rougher concentrate regrinding
- reground concentrate cleaner flotation
- gold bearing pyrite flotation (there was one cleaner flotation on reground pyrite concentrate (three scavenger flotation) for Comp NZ1 sample).

Figure 16.13 Locked Cycle Test Flowsheet – Snowfield



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 16.12.

Table 16.12 Locked Cycle Test Results – Snowfield

	Recovery				Grade				
Sample ID/Test ID	**Au (%)**	**Cu (%)**	**Mo (%)**	**Ag (%)**	**Au (g/t)**	**Cu (%)**	**Mo (%)**	**Ag (g/t)**	**Re* (ppm)**
Comp 3/FLC 1 (PRA Test Program 0904606)									
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 (PRA Test Program 0904606)									
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

table continues…

Sample ID/Test ID	Recovery Au (%)	Cu (%)	Mo (%)	Ag (%)	Grade Au (g/t)	Cu (%)	Mo (%)	Ag (g/t)	Re* (ppm)
Comp NZ1/FLC 1 (PRA Test Program 0907510)									
Calculated Feed	100	100	100	100	0.91	0.18	40	2.2	0.16
4th Cu Concentrate	46.4	74.1	25.8	30.1	93.3	29.6	0.208	146	8.09
Gold Pyrite Cleaner Concentrate	3.8	4.3	7.7	4.8	0.7	0.16	0.006	2.1	0.29
1st Cu Cleaner Scavenger Tailings	36.9	9.5	41.3	56.3	2.06	0.11	0.009	7.6	0.42
Gold Pyrite Cleaner Tailings	1.4	2.5	5.5	3.0	0.15	0.05	0.002	0.8	0.09
Rougher Scavenger Tailings	12.9	12.1	25.2	8.9	0.15	0.03	0.001	0.3	0.06

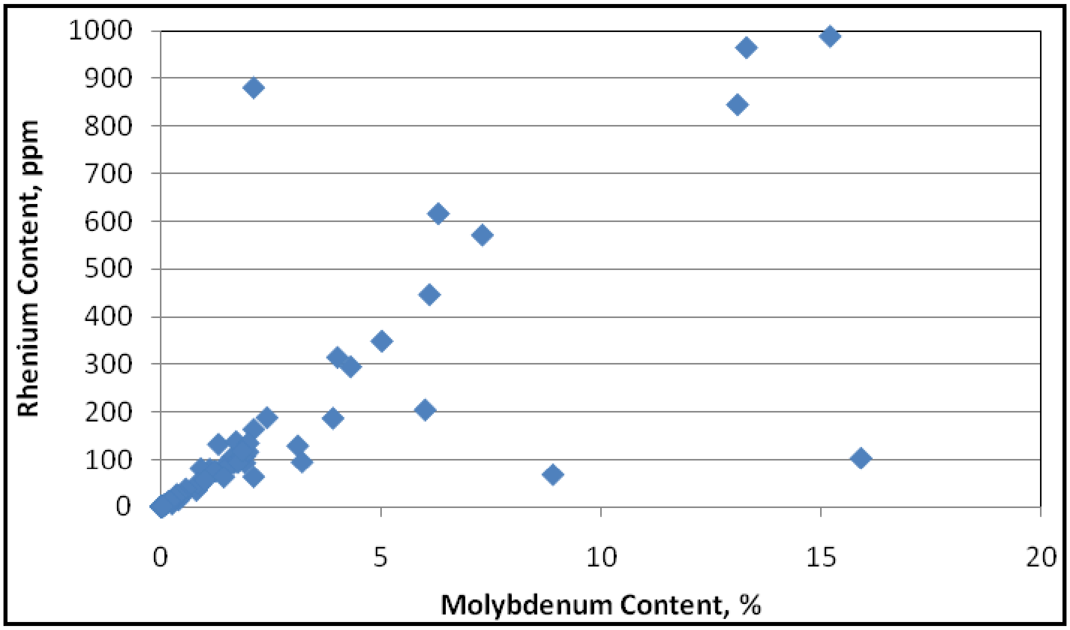
* average Re grade of Cycle 5 and Cycle 6.

At the applied test conditions, Comp NZ1 and Comp 4 samples produced 29.6% Cu and 25.8% Cu concentrate respectively, 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 between 46% and 51% 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% for Comp 3 and Comp 4 samples and 87% for Comp NZ1 sample. Over 70% of the molybdenum was floated with the copper minerals into the bulk concentrate from Comp 3 and Comp 4 samples. But only 26% of the molybdenum was recovered to the copper-molybdenum concentrate for Comp NZ1 sample.

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 16.14, suggests that rhenium would be recoverable together with molybdenum minerals.

Figure 16.14 Relationship between Rhenium and Molybdenum – Snowfield



Multi-element assay on the concentrates produced from the locked cycle tests are summarized in Table 16.13.

Table 16.13 Multi-element Assay on Concentrates from Locked Cycle Tests – Snowfield

Element	Unit	LCT/Comp 3 Cycle 6	LCT/Comp 4 Cycle 6	LCT/ Comp NZ1 Cycle 6
S	%	42	33.2	31.6
Sb	ppm	2,903	<5	<5
As	ppm	15,890	235	95
Co	ppm	95	25	8
Cd	ppm	512	<0.2	43
Bi	ppm	36	<2	<2
Hg	ppm	9.9	1.4	3.0
Ni	ppm	275	144	83
Pb	ppm	2,689	2,907	4800
Zn	ppm	44,587	1,182	8694
Se	ppm	<100	62	73
Al_2O_3	%	1.80	1.06	1.1
BaO	%	0.19	0.05	0.06
CaO	%	1.79	0.80	3.38
Fe_2O_3	%	41.8	38.9	30.0
K_2O	%	0.33	0.25	0.17

table continues…

Element	Unit	LCT/Comp 3 Cycle 6	LCT/Comp 4 Cycle 6	LCT/ Comp NZ1 Cycle 6
MgO	%	0.27	0.24	0.14
MnO	%	0.05	0.01	0.01
Na_2O	%	0.08	0.06	0.04
P_2O_5	%	0.02	0.01	0.04
SiO_2	%	3.94	3.20	3.38
TiO_2	%	0.11	0.17	0.11
Loss on Ignition	%	28.8	18.7	14.1

It appears that the impurity levels of the concentrate produced from Comp 4 and Comp NZ1, which were composed from the North zone, 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, which was generated from South the zone, 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).

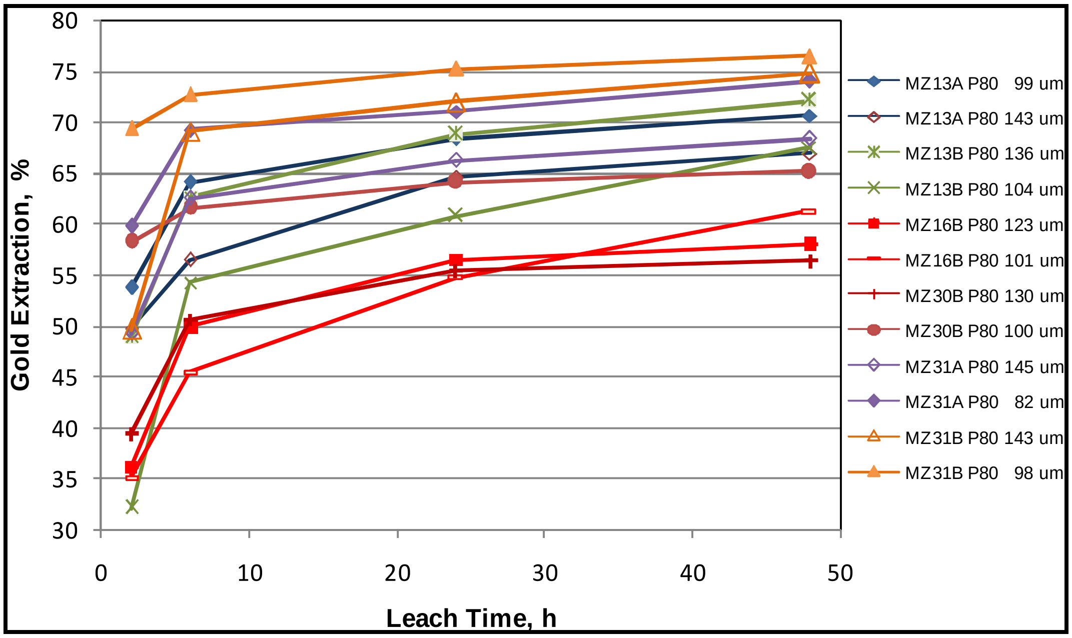
CYANIDE LEACHING TESTWOK

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 16.15.

Figure 16.15 Direct Cyanide Leach Test Results on Head Samples – Snowfield



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 Tailings 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.

The latter testwork conducted additional three cyanide leach tests using a cyanide concentration of 5 g/L. The tests were conducted on the cleaner scavenger tailing and scavenger flotation concentrate. The test results are presented in Table 16.14.

Table 16.14 Cyanide Leach Test Results – Snowfield

	Head (g/t)		Extraction (%) and Leach Retention Time					
			6 h		25 h		72 h	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag
1st Cleaner Scavenger Tailing & Scavenger Concentrate (Reground)	1.50	7.1	71.0	36.3	69.9	48.4	80.7	64.5
Scavenger Concentrate (No Reground)	0.70	3.8	48.5	28.0	56.1	42.7	58.5	52.9
1st Cleaner Scavenger Tailing (Reground)	1.80	6.5	36.8	35.5	38.2	44.1	36.0	45.7

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 16.15.

Table 16.15 Leach Test Results – Snowfield

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 Cleaner Tailings - Cycle 6	Carbon-in-Leach (CIL)[2]	3.11	4.7	35.0	55.4
C3	FLC 1 Au-Pyrite Conc. - Cycle 6	Regrind/CIL[2]	0.98	3.4	75.6	92.7
C4	FLC 1 Au-Pyrite Conc. - Cycle 5	Regrind/CIL[2]	2.49	4.2	72.3	85.9
C9	FLC 1 Cleaner Tailings - Composite	Regrind/CIL[2,3]	3.61	8.2	60.5	90.3
C10	FLC 1 Cleaner Tailings - Composite	Regrind/$Pb(NO_3)_2$/CIL[2]	4.08	3.2	62.7	41.1
C11	FLC 1 Cleaner Tailings - Composite	Regrind/DL[3,4]	3.03	6.8	65.2	41.4
C12	FLC 1 Cleaner 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 16.16.

Table 16.16 Leach Test Results – Snowfield

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	5.8	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 as 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 be extracted from the bulk rougher scavenger tailings, only containing 0.18 g/t Au.

GRAVITY CONCENTRATION

Two preliminary gravity concentration (centrifugal concentration + panning) tests were conducted on the reground gold bearing flotation products, one on the blend of 1st cleaner flotation tailings and the scavenger flotation concentration and the other on the rougher flotation concentrate. It appears that the mineralization does not respond well to the gravity concentration. Only less than 10% of the gold and silver were recovered into the paned concentrates. The test results are summarized in Table 16.17.

Table 16.17 Gravity Concentration Test Results

Product	Grade (g/t)		Recovery (%)	
	Au	Ag	Au	Ag
Reground 1st Cleaner Flotation Tailing and Scavenger Flotation Concentrate				
Pan Concentrate	93.0	250.1	9.0	8.9
Gravity Concentration Tailing	2.0	5.4	91.0	91.1
Feed	2.2	5.9	100.0	100.0
Reground Rougher Flotation Concentrate				
Pan Concentrate	114.0	53.9	8.7	1.6
Gravity Concentration Tailing	5.0	14.2	91.3	98.4
Feed	5.4	14.4	100.0	100.0

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 16.18. The results show that the addition of flocculant would significantly improve settling rate.

Table 16.18 Settling Test Results – Snowfield

Sample ID	Test	Feed Solids Density (%)	Floc (g/t)	pH	Required U/F Solids Density	Unit Thickening Area (m^2/t/d Solids)	Supernatant	
							TSS (mg/L)	TDS (mg/L)
PRA Test Program 0904606								
FLC1 Final Tailings/ Comp 3	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/ Comp 4	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
PRA Test Program 0907510								
FLC1 Final Tailings/ Comp NZ1	ST-1	19.6	N/A	7.1	50	7.86	-	-
	ST-2	19.6	20	10.4	50	2.08	-	-

Notes: Floc = flocculant; U/F = underflow; TSS = total suspended solids; TDS = total dissolved solids.

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 16.19.

Table 16.19 Acid Accounting Test Results* – Snowfield

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 potential are expressed in kg $CaCO_3$ 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.

CONCLUSIONS

The reviews of the preliminary testwork on Snowfield mineralization samples resulted in the following conclusions:

- The mineralization is moderately hard.
- Two main flowsheets were investigated — 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. It appears that the combined process should be more amendable for the mineralization. The combined process should include:
  - 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, antimony, mercury, and zinc 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 the process flowsheet
- investigate metallurgical performances
- determine engineering related data.

Detailed recommendations are specified in Section 19.0 of this report.

16.1.3 BRUCEJACK MINERALIZATION

SAMPLE DESCRIPTION

Two batches of assay reject samples were received by PRA in October and November 2009. The first batch had 378 samples with a total weight of 1,695 kg (including packing bag weight), while the second weighed 950 kg (including packing bag weight) with 198 samples.

The samples were grouped into 16 composite samples which were labelled as: SU-4, SU-5, SU-6A, SU-6B, SU-10, SU-19, SU-21A, SU-21B, SU-25, SU-27, SU-032A, SU-032B, SU-032C, SU-033, SU-036A, and SU-036B. The drill hole distribution is presented in Section 11.0 (Figure 11.2).

The composite samples were further composed into zone composite samples representing the West Zone (Composite R8), the Gossan Hill and Bridge zones (Composite BZ), and the Galena Hill Zone (Composite GH).

SAMPLE HEAD ANALYSES

The head assay on the composites is summarized in Table 16.20. It appears that the gold grades of the zone composites are higher than the average gold grades of the mineralization zones where the samples were collected.

Table 16.20 Metal and Sulphur Contents of Composite Samples – Brucejack

Sample ID	Au (g/t)	Au(CN) (g/t)	Ag[1] (g/t)	Au(CN[2]) (g/t)	S(-2) (%)	C(org[3]) (%)	Cu[1] (ppm)	As (ppm)
SU-4	1.86		3.9		2.67	0.22	57	0.113
SU-5	0.99		34.8		1.58	0.10	235	0.026
SU-6A	1.36		67.3		3.63	0.06	101	0.020
SU-6B	1.05		12.9		3.79	0.19	90	0.029
SU-10	0.71		8.3		1.89	0.13	77	0.011
SU-19	1.35		6.6		2.03	0.25	133	0.010
SU-21A	0.62		10.3		2.39	0.14	70	0.026
SU-21B	5.23		12.3		2.07	0.18	96	0.031
SU-25	1.64		11.4		1.86	0.22	34	0.025
SU-27	0.64		4.0		1.21	0.15	23	0.033
SU-032A	2.46	1.70	13.3	11.7	3.50	0.11	66	0.016
SU-032B	0.84	0.78	71.1	73.8	3.11	0.35	57	0.007
SU-032C	1.90	1.62	1.9	4.0	2.93	0.29	27	0.024
SU-033	2.17	2.10	24.5	29.8	3.08	0.21	63	0.018
SU-036A	1.40	0.68	10.2	8.8	3.23	0.22	104	0.046
SU-036B	0.64	0.41	3.8	3.0	3.56	0.33	26	0.028
Comp R8	1.14						60	0.022
Comp GH	1.65						131	0.022
Comp BZ	1.53						77	0.020

[1] by ICP
[2] CN = cyanide soluble
[3] org = organic carbon.

GRINDABILITY TESTWORK

Table 16.21 presents the Bond ball mill work index obtained from the Brucejack mineralization. It appears that on average, the mineralization is moderately hard. The ball mill grinding work index is very comparable to the ones obtained from the Snowfield mineralization.

Table 16.21 Grindability Test Results – Brucejack

Sample ID	Bond Ball Mill Work Index (kWh/t)
BZ	16.4
GH	15.6
R8	16.2

Sample Specific Gravity

The SG of the Brucejack mineral samples are shown in Table 16.22. The SG data varied narrowly from 2.71 to 2.84.

Table 16.22 Sample Specific Gravity – Brucejack

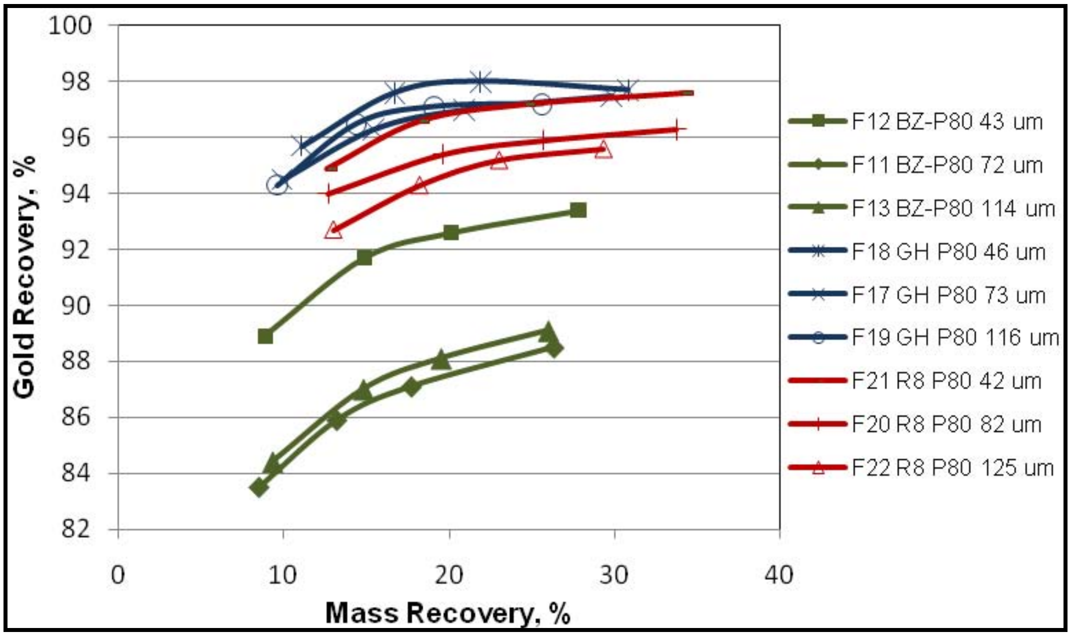
Sample ID	SG
SU-4	2.79
SU-5	2.74
SU-6A	2.82
SU-6B	2.84
SU-10	2.76
SU-19	2.76
SU-21A	2.75
SU-21B	2.77
SU-25	2.71
SU-27	2.74
SU-032A	2.73
SU-032B	2.73
SU-032C	2.72
SU-033	2.78
SU-036A	2.82
SU-036B	2.78

Flotation Testwork

Primary Grind Size

Three different primary grind sizes were tested on the BZ, GH, and R8 composite samples. PAX and A208 were used as collectors, MIBC as the frother, and copper sulphate as the activator (at scavenger flotation only). The test results are shown in Figure 16.16. The data indicate that gold recovery improves when the primary grind size is finer than 70 µm. The improvement becomes much less significant at the grind size between 80% passing 70 µm and 80% passing 125 µm. Also the test results show that gold recovery increases with concentrate mass pull, in particular, when the mass pull is less than 15% to 20%.

Figure 16.16 Effect of Primary Grind Size on Gold Recovery – Brucejack

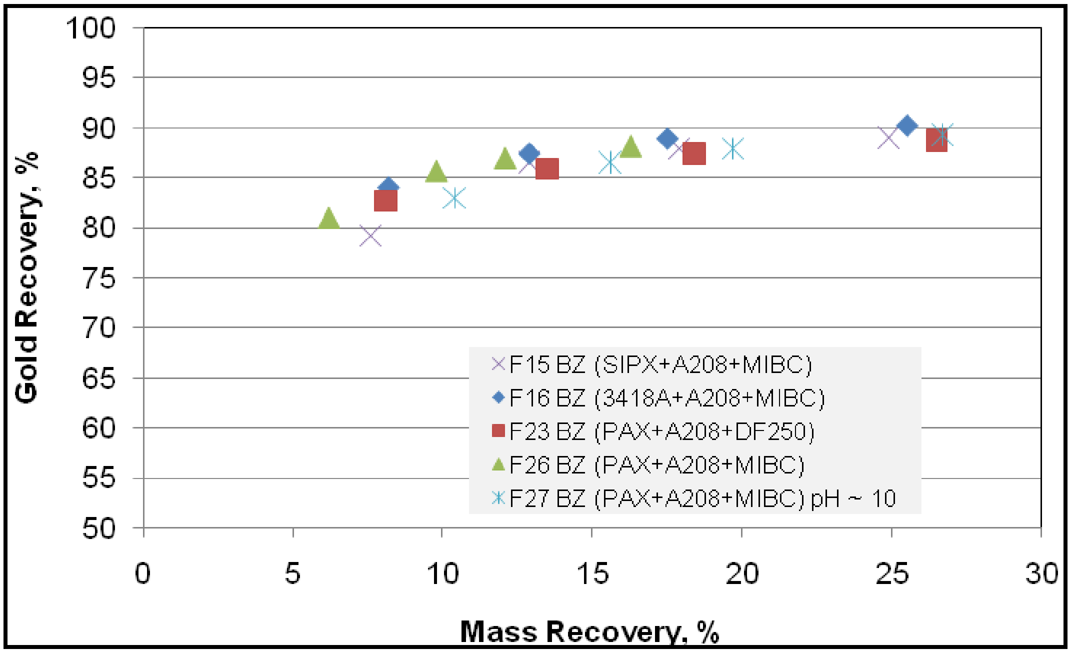


There is a substantial difference in metallurgical response between the Bridge Zone mineralization and the other mineralization (Galena Hill, West, and Gossan Hill zones). The gold recovery of the BZ sample is approximately 87% at a primary size of 80% passing 114 µm and a mass recovery of 15%; however, the GH sample produces a higher than 96% gold recovery at the similar test conditions.

Reagents and Slurry pH

The testing program also investigated the effect of flotation reagents and slurry pH on the metallurgical performance. The test results of the Bridge Zone composite sample are summarized in Figure 16.17. It appears that the effect of the reagents and slurry pH on the gold recovery was not significant.

Figure 16.17 Effect of Reagent and Slurry pH on Gold Recovery – Brucejack

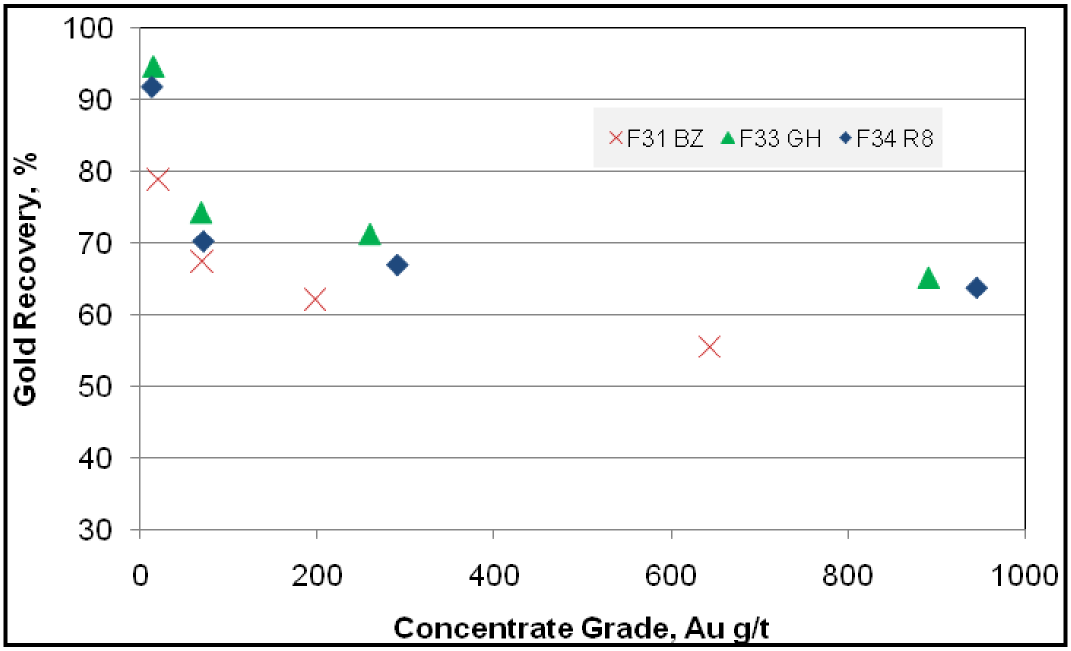


* Test F27 was conducted at a higher pH, the others at natural pH.

Cleaner Flotation Testwork

The testing program also studied the possibility of upgrading the rougher flotation concentrates. The tests indicated that the cleaner flotation was able to substantially upgrade the concentrates from the Brucejack mineral samples. However, the gold recovery reduced significantly at the 1st cleaner flotation stage.

Figure 16.18 Effect of Cleaner Flotation on Gold Recovery – Brucejack



Gravity Concentration Testwork

Metallic gold determination tests and gravity concentration tests showed that Brucejack mineralization contains a significant amount of fine grain nugget gold. The metallic gold determination test results are shown in Table 16.23 and Table 16.24.

Table 16.23 Metallic Gold Test Results – Individual Samples (Brucejack)

Screen Mesh*	Sample ID	Grade (g/t)		Distribution (%)			Sample ID	Grade (g/t)		Distribution (%)		
		Au	Ag	Au	Ag	Mass		Au	Ag	Au	Ag	Mass
+150	SU-4	1.91	1.0	9.4	4.1	8.6	SU-25	2.63	15.0	9.0	10.7	7.3
-150		1.74	2.2	90.6	95.9	91.4		2.08	9.8	91.0	89.3	92.7
Total		**1.75**	**2.1**	**100.0**	**100.0**	**100.0**		**2.12**	**10.2**	**100.0**	**100.0**	**100.0**
+150	SU-5	2.99	29.3	11.5	3.8	4.2	SU-27	2.70	0.5	7.5	2.5	2.5
-150		1.02	32.7	88.5	96.2	95.8		0.86	0.5	92.5	97.5	97.5
Total		**1.10**	**32.6**	**100.0**	**100.0**	**100.0**		**0.91**	**0.5**	**100.0**	**100.0**	**100.0**
+150	SU-6A	9.25	50.6	21.8	4.2	4.7	SU-32A	6.49	15.1	14.2	4.7	4.9
-150		1.62	56.9	78.2	95.8	95.3		2.02	15.7	85.8	95.3	95.1
Total		**1.98**	**56.6**	**100.0**	**100.0**	**100.0**		**2.24**	**15.7**	**100.0**	**100.0**	**100.0**
+150	SU-6B	100.1	94.0	73.7	27.1	3.8	SU-32B	8.28	51.0	38.1	4.7	6.5
-150		1.43	10.1	26.3	72.9	96.2		0.94	73.1	61.9	95.3	93.5
Total		**5.23**	**13.3**	**100.0**	**100.0**	**100.0**		**1.42**	**71.7**	**100.0**	**100.0**	**100.0**
+150	SU-10	2.11	2.1	11.4	2.0	4.1	SU-32C	10.9	9.0	37.1	22.0	10.4
-150		0.70	4.3	88.6	98.0	95.9		2.15	3.7	62.9	78.0	89.6
Total		**0.76**	**4.2**	**100.0**	**100.0**	**100.0**		**3.06**	**4.2**	**100.0**	**100.0**	**100.0**
+150	SU-19	1.65	3.0	4.6	3.2	4.4	SU-33	22.6	29.6	59.6	7.8	9.0
-150		1.57	4.2	95.4	96.8	95.6		1.52	34.9	40.4	92.2	91.0
Total		**1.57**	**4.1**	**100.0**	**100.0**	**100.0**		**3.42**	**34.4**	**100.0**	**100.0**	**100.0**
+150	SU-21A	0.64	4.3	3.7	2.0	3.7	SU-36A	2.12	9.5	15.4	7.9	9.4
-150		0.64	8.2	96.3	98.0	96.3		1.21	11.4	84.6	92.1	90.6
Total		**0.64**	**8.1**	**100.0**	**100.0**	**100.0**		**1.30**	**11.2**	**100.0**	**100.0**	**100.0**
+150	SU-21B	22.0	2.5	34.8	3.0	8.0	SU-36B	0.69	7.9	12.3	20.4	9.9
-150		3.58	6.9	65.2	97.0	92.0		0.54	3.4	87.7	79.6	90.1
Total		**5.05**	**6.5**	**100.0**	**100.0**	**100.0**		**0.55**	**3.8**	**100.0**	**100.0**	**100.0**

* Tyler Mesh.

Table 16.24 Metallic Gold Test Results – Composite Samples (Brucejack)

Sample ID	Screen Mesh	Grade (Au g/t)	Distribution (%)	
			Mass	Au
Composite R8	+150	6.95	4.8	23.1
	-150	1.16	95.2	76.9
	Total	**1.44**	**100.0**	**100.0**
Composite GH	+150	6.66	7.9	30.3
	-150	1.31	92.1	69.7
	Total	**1.73**	**100.0**	**100.0**
Composite BZ	+150	3.89	5.4	12.6
	-150	1.54	94.6	87.4
	Total	**1.67**	**100.0**	**100.0**

As indicated from the test results in Table 16.23, the free gold occurrence changes substantially from sample to sample. The SU-6B, SU-6A, SU-21B, SU-32B, SU-32C, and SU-33 samples may contain significant amount of free gold. Compared to Composite BZ in Table 16.24, more gold may be present in the form of free gold in Composite R8 and Composite GH.

PRA conducted gravity concentration tests on the head composite samples (ground to approximately 80% passing 116 to 131 µm) and flotation concentrate samples (reground to 80% passing 25 µm). Two stages of gravity concentration were conducted – the first stage by centrifugal concentration, and the second stage by panning. The test results, shown in Table 16.25, indicated that most of the samples responded well to the gravity concentration, especially the reground concentrates. Approximately 29% to 45% of the gold in the concentrates of the zone composite samples was recovered into the gravity concentrates containing over 1,000 g/t Au. However, silver did not show similar metallurgical responses to gold. Again, the test results indicated that some of the samples (such as the SU-36B sample) were less amendable to the gravity concentration.

Table 16.25 Gravity Concentration Test Results – Brucejack

Test ID	Sample ID	Primary Grind/ Regrind Size	Grade (g/t)		Recovery (%)	
			Au	Ag	Au	Ag
GF35	BZ	P80 131 um	685	428	17.0	4.4
GF37	R8	P80 116 um	70.5	677	2.7	1.8
GF36	GH	P80 116 um	158	495	11.0	1.8
GF41	GH	P80 116 um	331	339	25.7	1.4
GF38	R8	P80 <25 um	1,081	1,222	35.6	2.6
GF39	GH	P80 <25 um	1,918	3,103	44.8	4.5
GF40	BZ	P80 <25 um	1,079	984	29.3	5.9
GF42	SU-32B	P80 <25 um	801	4,193	22.6	1.4
GF43	SU-33	P80 <25 um	5,810	8,341	43.9	4.9
GF44	SU-36A	P80 <25 um	3,337	1,653	42.3	4.0
GF45	SU-36B	P80 <25 um	217	337	10.6	2.4

Cyanide Leach Testwork

PRA conducted cyanide leach tests on various samples to investigate the gold extraction from various samples including head samples, flotation concentration samples, and flotation tailing samples.

The head sample leaching test results are summarized in Table 16.26. The tests were conducted at a pH of 10.5 and a sodium cyanide concentration of 3 g/L with three different primary grind sizes.

Table 16.26 Head Sample Cyanidation Test Results – Brucejack

Test No	Sample ID	Grind Size (P80 µm)	Calculated Head (g/t)		Extraction (%)		Residue Grade (g/t)		Consumption (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
C1	BZ	71	1.79	9.68	81.0	59.7	0.34	3.9	2.09	0.28
C2	BZ	40	2.01	10.3	85.1	63.0	0.30	3.8	2.17	0.37
C3	BZ	127	2.35	10.6	84.7	57.5	0.36	4.5	1.97	0.23
C4	GH	72	1.41	40.2	77.9	67.6	0.31	13.0	1.91	0.24
C5	GH	42	1.35	38.3	76.3	72.0	0.32	10.8	1.94	0.23
C6	GH	119	1.49	36.6	72.4	68.6	0.41	11.5	1.77	0.23
C7	R8	78	1.37	26.4	75.9	65.2	0.33	9.2	1.71	0.32
C8	R8	44	1.24	24.5	75.0	68.2	0.31	7.8	2.02	0.32
C9	R8	131	1.34	25.2	73.8	63.2	0.35	9.3	1.85	0.33

At the leach retention time of 48 hours, the gold extractions ranged from 72% to 85%; silver extraction was lower, ranging from 58% to 72%. The influence of primary grind size on the gold and silver recoveries was relatively insignificant. The test results indicated that the gold extraction of Composite BZ was better than Composites GH and R8. This may result from a higher gold head grade of Composite BZ, compared to the other two samples. It appears that the samples need a longer leach retention time because the leach was not complete when the tests were terminated. Sodium cyanide consumption varied from 1.7 kg/t to 2.2 kg/t.

Further tests were conducted on the flotation concentrates that were reground to 90% passing 25 µm. The sodium cyanide concentration was high at 5 g/L NaCN. The leach retention time was increased to 96 hours. The test results are summarized in Table 16.27.

Table 16.27 Concentrate Cyanidation Test Results – Brucejack

Test No.	Sample ID*	Pre-treatment	Calculated Head (g/t)		Extraction (%)		Residue Grade (g/t)		Consumption (kg/t)	
			Au	Ag	Au	Ag	Au	Ag	NaCN	Lime
C10/ F24	R8	regrind	8.0	125	86.0	86.7	1.13	16.6	13.8	0.55
C11/ F25	GH	regrind	8.6	203	79.4	87.3	1.77	25.8	15.4	1.08
C12/ F26	BZ	regrind	11.6	56	82.6	79.7	2.02	11.3	15.6	0.61
C13/ F24	R8	$KMnO_4$ to regrind	8.1	123	82.7	85.5	1.40	17.9	13.7	0.41
C14/ F25	GH	regrind + oxygen in leach	9.2	129	72.5	81.2	2.54	24.3	16.0	1.77
C15/ F26	BZ	$Pb(NO_3)_2$ to regrind	10.7	55	69.9	73.2	3.22	14.8	14.5	1.53

* rougher + scavenger concentrate.

The test results appear to indicate that approximately between 79% and 86% of the gold can be leached from the reground concentrates. The addition of the $KMnO_4$, $Pb(NO_3)_2$ and oxygen did not improve gold extraction. It appears that the required gold leach retention time ranged from approximately 48 hours to 72 hours but silver required a longer leach retention time compared with gold. Cyanide consumption was high, ranging from 13.7 kg/t NaCN to 16.0 kg/t NaCN. The high cyanide consumption was possibly due to a high cyanide dosage (5 g/L NaCN).

Gravity + Flotation + Cyanidation Testwork

According to the finding of the preliminary testwork, PRA conducted further testing using a combination of flotation, gravity concentration, and cyanidation to recover gold and silver from the Brucejack mineralization. There were two different process combinations:

- primary grind, gravity concentration, rougher/scavenger flotation, and regrind on the flotation concentrate, followed by cyanidation on the reground concentrate (Flowsheet A)
- primary grind, rougher/scavenger flotation, regrind on the flotation concentrate, and gravity concentration on the reground concentrate, followed by cyanidation on gravity tailings (Flowsheet B).

The test results are presented separately in Table 16.28 and Table 16.29 for the two different combinations.

Table 16.28 Gravity Concentration + Flotation + Cyanide Leach Test Results (Flowsheet A) – Brucejack

Test ID/Sample ID	Primary Grind/ Regrind Sizes	Grade (g/t)		Recovery/Extraction*	
		Au	Ag	Au (%)	Ag (%)
GF35/Composite BZ					
Gravity Concentrate	P80 131 um	685	428	17.0	4.4
Flotation Concentrate		18.6	45.2	77.1	77.3
Leach on Flotation Concentrate	P90 <25 um			93.7	86.2
Head		4.5	10.9		
GF37/Composite R8					
Gravity Concentrate	P80 116 um	70.5	677	2.7	1.8
Flotation Concentrate		11.5	158	94.4	91.0
Leach on Flotation Concentrate	P90 <25 um			91.5	93.7
Head		2.8	39.5		

table continues…

Test ID/Sample ID	Primary Grind/ Regrind Sizes	Grade (g/t)		Recovery/Extraction*	
		Au	Ag	Au (%)	Ag (%)
GF36/Composite GH					
Gravity Concentrate	P80 116 um	158	495	11.0	1.8
Flotation Concentrate		9.6	200	85.5	92.5
Leach on Flotation Concentrate	P90 <25 um			84.9	89.6
Head		1.9	36.3		
GF41/Composite GH					
Gravity Concentrate	P80 116 um	331	339	25.7	1.4
Flotation Concentrate		7.7	186	71.8	92.8
Leach on Flotation Concentrate	P90 <25 um			83.0	89.9
Head		1.8	34.5		

* Extraction refers to flotation concentrate.
- Leach retention time: 96 hours.
- Cyanide concentration: 5 g/L.

As shown in Table 16.28, the flotation and gravity concentration recovered approximately 84% of the gold from the BZ sample, and 97% of the gold from the R8 and GH samples. The gold leaching extraction rates from the flotation concentrates were higher than 91% for the BZ and R8 samples. Compared to the other two samples, the GH sample showed a lower gold cyanide extraction rate at approximately 84% on average.

Table 16.29 Flotation + Gravity Concentration + Cyanide Leach Test Results (Flowsheet B) – Brucejack

	Primary Grind & Regrind Sizes	Concentrate Grade(g/t)		Recovery/Extraction	
		Au	Ag	Au (%)	Ag (%)
GF38/Composite R8[1]					
Flotation Concentrate	P80 128 um	7.51	106	94.1	88.6
Gravity Concentrate	P94 33 um	1081	1222	35.6	2.6
Gravity Tailing		4.68	103	58.5	86.0
Leach on Gravity Tailing				91.8	83.6
Head		2.03	26.5		
GF39/Composite GH[1]					
Flotation Concentrate	P80 141 um	12.9	212.1	97.1	98.7
Gravity Concentrate	P90 <25 um	1918	3103	44.8	4.5
Gravity Tailing		4.68	103.2	52.3	94.2
Leach on Gravity Tailing				86.2	68.7
Head		1.99	32.1		

table continues…

	Primary Grind & Regrind Sizes	Concentrate Grade(g/t)		Recovery/Extraction	
		Au	Ag	Au (%)	Ag (%)
GF40/Composite BZ[1]					
Flotation Concentrate	P80 133 um	8.60	44.4	85.1	97.3
Gravity Concentrate	P90 <25 um	1079	984	29.3	5.9
Gravity Tailing		4.68	103	55.7	91.4
Leach on Gravity Tailing				80.9	68.7
Head		1.70	7.68		
GF42/Composite SU-32B[2]					
Flotation Concentrate	P80 109 um	4.71	382	93.1	90.8
Gravity Concentrate	P80 <25 um	801	4193	22.6	1.4
Gravity Tailing		3.57	376	70.5	89.3
Leach on Gravity Tailing				78.7	78.6
Head		0.99	82.3		
GF43/Composite SU-33[2]					
Flotation Concentrate	P80 92 um	13.5	164	98.5	93.3
Gravity Concentrate	P80 <25 um	5810	8341	43.9	4.9
Gravity Tailing		7.50	156	54.6	88.4
Leach on Gravity Tailing				87.6	78.2
Head		2.32	29.7		
GF44/Composite SU-36A[2]					
Flotation Concentrate	P80 138 um	8.95	45.7	97.0	94.5
Gravity Concentrate	P80 <25 um	3337	1653	42.3	4.0
Gravity Tailing		5.05	43.8	54.7	90.5
Leach on Gravity Tailing				61.5	66.2
Head		2.12	11.1		
GF45/Composite SU-36B[2]					
Flotation Concentrate	P80 96 um	2.71	19.3	91.5	95.0
Gravity Concentrate	P80 <25 um	217	337	10.6	2.4
Gravity Tailing		5.05	43.8	80.9	92.6
Leach on Gravity Tailing				56.9	63.3
Head		0.58	4.0		

[1] Extraction is referred to gravity concentration tailings; leach retention time = 25 hours; direct cyanide leach; cyanide concentration = 5 g/L;
[2] Extraction is referred to gravity concentration tailings; leach retention time = 24 hours; CIL; cyanide concentration = 3 g/L.

As shown in Table 16.29, Flowsheet B produced a much higher gold gravity concentration recovery from the BZ, GH, and R8 samples when compared to Flowsheet A. Also, the tests indicated that the leach retention time for the gravity concentration tailings reduced significantly. It appears that most of the leachable gold in the gravity concentration tailings were extracted within 25 hours (approximately 90% or more of the leachable gold was extracted within 6 hours).

Flowsheet B was also used to test the SU-32B, SU-33, SU-36A, and SU-36B samples. Gold and silver flotation recoveries obtained from these samples were similar to that achieved from three zone composite samples; however, the gold and silver leach extraction rates were lower.

The SU-32B and SU-36B samples also produced lower gold recoveries at the gravity concentration stage.

CONCLUSIONS

The review of preliminary testwork on the Brucejack mineralization resulted in the following conclusions:

- Brucejack mineralization is moderately hard.
- The test results suggest that the combined process should be more amendable for the mineralization. The process should include:
  - flotation to produce rougher and scavenger concentrates
  - regrinding on the rougher and scavenger concentrates
  - gravity concentration to recover free gold and silver
  - cyanide leaching on gravity concentration tailings and concentrate separately to produce gold/silver doré.
- The test results indicate that there is significant variation in metallurgical performance between the mineralization samples.
- The process conditions from the testwork have not yet been optimized.

RECOMMENDATIONS

Further testwork is recommended to:

- confirm the findings of the testwork completed to date
- optimize the process flowsheet,
- investigate metallurgical performances
- determine engineering related data.

Section 19.0 provides more detailed recommendations.

16.1.4 *METALLURGICAL PERFORMANCE PROJECTION*

According to the preliminary metallurgical test results and the proposed annual mining schedule, the metallurgical performance of mineralization from the Brucejack and Snowfield deposits are projected in Table 16.30 and Table 16.31. The total projected metal productions, including concentrate productions, are summarized in Table 16.32.

Table 16.30 Projected Metallurgical Performance – Brucejack

Year	Annual Process Rate (t x 000)	Mill Feed Grade		Doré					
				Recovered Metals				Recovery	
		Au (g/t)	Ag (g/t)	Au		Ag		Au (%)	Ag (%)
				kg	oz (000)	kg	oz (000)		
1	0								
2	10,580	1.12	13.8	9,180	295	104,580	3,360	77.4	71.7
3	18,000	1.41	14.3	20,780	668	190,270	6,120	82.0	73.8
4	12,190	1.16	34.2	11,100	357	355,970	11,440	78.2	85.4
5	15,980	1.60	46.2	21,450	690	612,520	19,690	84.0	83.0
6	10,950	0.95	15.2	7,210	232	121,950	3,920	69.2	73.1
7	17,780	0.98	14.4	11,380	366	178,960	5,750	65.0	70.0
8	10,950	1.01	11.6	8,210	264	88,690	2,850	74.0	70.0
9	10,950	0.72	10.4	5,130	165	79,600	2,560	65.0	70.0
10	22,550	0.76	10.7	11,200	360	169,480	5,450	65.0	70.0
11	10,950	0.82	10.2	5,810	187	78,520	2,520	65.0	70.0
12	11,060	0.86	10.7	6,180	199	82,640	2,660	65.0	70.0
13	10,950	0.90	9.3	6,420	206	62,360	2,000	65.0	61.0
14	10,950	1.08	10.4	8,710	280	79,710	2,560	74.0	70.0
15	32,850	0.88	9.0	18,690	601	181,070	5,820	65.0	61.0
16	5,030	1.04	17.1	3,890	125	65,280	2,100	74.0	76.0
17-27	0								
Total	**211,710**	**1.02**	**15.6**	**155,320**	**4,994**	**2,451,570**	**78,820**	**72.2**	**74.5**
Average		**1.02**	**15.6**					72.2	74.5

Table 16.31 Projected Metallurgical Performance – Snowfield

							Copper Concentrate													Doré						Molybdenum Concentrate				
		Mill Feed Grade						Recovered Metals						Grade			Recovery			Recovered Metals				Recovery			Molybdenum			
								Copper		Gold		Silver								Au		Ag					Recovered Metal		Grade	Recovery
Year	Annual Process Rate (t x 000)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Re (ppm)	Tonnes	t	lb (000)	kg	oz (000)	kg	oz (000)	Cu (%)	Au (g/t)	Ag (g/t)	Cu (%)	Au (%)	Ag (%)	kg	oz (000)	kg	oz (000)	Au (%)	Ag (%)	Tonnes	Mo (t)	Re (kg)	Mo (%)	Mo (%)
1	39,550	1.56	0.032	1.45	124	0.87	6,570	1,440	3,180	9,640	310	9,580	308	22.0	1,469	1,459	11.2	15.7	16.7	33,770	1,086	29,140	937	54.9	50.7	4,900	2,450	17,230	50	50
2	32,850	0.78	0.101	1.96	116	0.62	81,840	18,000	39,690	11,750	378	30,170	970	22.0	144	369	54.3	45.9	46.9	5,550	178	12,530	403	21.6	19.5	3,800	1,900	10,180	50	50
3	25,810	0.70	0.129	2.06	100	0.59	96,150	21,150	46,630	9,460	304	28,420	914	22.0	98	296	63.5	52.4	53.4	2,990	96	7,850	252	16.6	14.7	2,580	1,290	7,600	50	50
4	31,200	0.70	0.117	1.91	100	0.52	99,280	21,840	48,150	10,810	347	30,230	972	22.0	109	305	59.8	49.8	50.8	3,980	128	9,750	314	18.3	16.4	3,120	1,560	8,110	50	50
5	27,820	0.68	0.116	2.00	100	0.53	87,310	19,210	42,350	9,400	302	28,050	902	22.0	108	321	59.5	49.5	50.5	3,480	112	9,100	293	18.3	16.4	2,780	1,390	7,350	50	50
6	32,850	0.70	0.131	1.90	90	0.51	125,430	27,600	60,840	12,190	392	33,490	1,077	22.0	97	267	64.1	52.8	53.8	3,770	121	9,030	290	16.3	14.5	2,960	1,480	8,370	50	50
7	26,020	0.95	0.163	1.94	80	0.28	139,600	30,710	67,710	14,540	468	30,000	964	22.0	104	215	72.4	58.6	59.6	3,330	107	5,940	191	13.4	11.8	2,080	1,040	3,640	50	50
8	32,850	0.75	0.045	1.47	100	0.76	15,860	3,490	7,690	5,980	192	12,270	394	22.0	377	773	23.6	24.3	25.3	8,920	287	16,020	515	36.3	33.1	3,290	1,640	12,540	50	50
9	32,850	0.62	0.074	1.71	110	0.80	46,930	10,320	22,760	7,690	247	21,660	696	22.0	164	462	42.5	37.6	38.6	5,320	171	13,180	424	26.0	23.5	3,610	1,810	13,060	50	50
10	21,250	0.59	0.092	1.76	120	0.51	45,080	9,920	21,870	5,470	176	16,550	532	22.0	121	367	50.7	43.4	44.4	2,720	87	7,240	233	21.6	19.4	2,550	1,280	5,360	50	50
11	32,850	0.58	0.093	1.61	120	0.55	71,010	15,620	34,440	8,290	267	23,630	760	22.0	117	333	51.1	43.7	44.7	4,020	129	10,070	324	21.2	19.0	3,940	1,970	9,030	50	50
12	32,740	0.61	0.104	1.52	110	0.60	85,700	18,860	41,570	9,330	300	23,770	764	22.0	109	277	55.4	46.6	47.6	3,910	126	8,740	281	19.6	17.5	3,600	1,800	9,740	50	50
13	32,850	0.66	0.124	1.71	100	0.62	114,880	25,270	55,720	11,140	358	29,330	943	22.0	97	255	62.0	51.3	52.3	3,670	118	8,440	271	16.9	15.0	3,290	1,640	10,240	50	50
14	32,850	0.80	0.144	1.81	80	0.42	145,600	32,030	70,620	14,460	465	33,540	1,078	22.0	99	230	67.7	55.3	56.3	3,970	128	8,020	258	15.2	13.5	2,630	1,310	6,880	50	50
15	10,950	0.69	0.035	1.06	90	1.09	2,450	539	1,190	1,330	43	2,160	69	22.0	543	881	14.1	17.7	18.7	3,060	98	4,280	138	40.5	37.0	990	490	5,980	50	50
16	38,770	0.55	0.079	1.46	100	0.54	62,570	13,770	30,350	8,320	268	22,790	733	22.0	133	364	44.9	39.3	40.3	5,030	162	12,110	389	23.8	21.4	3,880	1,940	10,360	50	50
17	43,800	0.55	0.099	1.49	100	0.50	105,460	23,200	51,150	10,980	353	30,170	970	22.0	104	286	53.5	45.3	46.3	4,770	153	11,490	370	19.7	17.6	4,380	2,190	11,030	50	50
18	43,630	0.62	0.107	1.62	80	0.48	119,790	26,360	58,100	12,740	409	34,290	1,102	22.0	106	286	56.5	47.4	48.4	5,130	165	12,100	389	19.1	17.1	3,490	1,750	10,430	50	50
19	43,800	0.80	0.126	1.81	80	0.38	157,160	34,580	76,230	18,060	581	41,690	1,340	22.0	115	265	62.6	51.7	52.7	6,160	198	12,450	400	17.6	15.7	3,500	1,750	8,240	50	50
20	43,800	0.53	0.073	1.40	90	0.53	60,970	13,410	29,570	8,590	276	23,350	751	22.0	141	383	42.0	37.2	38.2	5,760	185	13,770	443	25.0	22.5	3,940	1,970	11,490	50	50
21	43,800	0.50	0.097	1.53	90	0.44	101,830	22,400	49,390	9,790	315	30,580	983	22.0	96	300	52.7	44.8	45.8	4,220	136	11,560	372	19.3	17.3	3,940	1,970	9,550	50	50
22	43,800	0.57	0.121	1.53	90	0.40	147,220	32,390	71,410	12,600	405	34,710	1,116	22.0	86	236	61.1	50.7	51.7	4,030	130	9,680	311	16.2	14.4	3,940	1,970	8,640	50	50
23	43,800	0.73	0.126	1.84	70	0.26	157,160	34,580	76,230	16,450	529	42,380	1,363	22.0	105	270	62.6	51.7	52.7	5,470	176	12,330	397	17.2	15.3	3,070	1,530	5,600	50	50
24	43,020	0.60	0.060	1.38	90	0.58	40,490	8,910	19,640	8,200	264	19,580	629	22.0	203	483	34.5	32.0	33.0	7,560	243	15,870	510	29.5	26.7	3,870	1,940	12,390	50	50
25	43,800	0.48	0.076	1.54	90	0.34	65,790	14,470	31,910	8,080	260	26,520	853	22.0	123	403	43.5	38.3	39.3	4,980	160	14,340	461	23.6	21.3	3,940	1,970	7,460	50	50
26	43,800	0.50	0.107	1.60	70	0.34	120,260	26,460	58,330	10,360	333	33,870	1,089	22.0	86	282	56.5	47.4	48.4	3,830	123	10,940	352	17.5	15.6	3,070	1,530	7,400	50	50
27	38,800	0.66	0.148	2.04	70	0.28	179,440	39,480	87,030	14,340	461	45,170	1,452	22.0	80	252	68.8	56.0	57.0	3,490	112	9,510	306	13.6	12.0	2,720	1,360	5,420	50	50
Total	**959,900**	**0.68**	**0.101**	**1.66**	**94**	**0.51**	**2,481,830**	**546,000**	**1,203,730**	**279,980**	**9,001**	**737,920**	**23,725**	**22.0**	**113**	**297**	**56.3**	**43.2**	**46.2**	**152,880**	**4,915**	**305,440**	**9,820**	**23.6**	**19.1**	**89,850**	**44,930**	**243,310**	**50**	**50**
Average	**35,550**	**0.68**	**0.101**	**1.66**	**94**	**0.51**	**91,920**	**20,220**	**44,580**	**10,370**	**333**	**27,330**	**879**	**22.0**	**113**	**297**	**56.3**	**43.2**	**46.2**	**5,660**	**182**	**11,310**	**364**	**23.6**	**19.1**	**3,330**	**1,660**	**9,010**	**50**	**50**

Table 16.32 Projected Metal Production – Snowfield and Brucejack

	Mill Feed – Snowfield						Mill Feed – Brucejack			Copper Concentrate							Dore				Molybdenum Concentrate				
		Grade						Grade			Recovered Metals						Recovered Metals					Recovered Metals			
	Annual Process						Annual Process				Copper		Gold		Silver		Gold		Silver			Molybdenum		Rhenium	
Year	Rate (t x 000)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Re (ppm)	Rate (t x 000)	Au (g/t)	Ag (g/t)	Tonnes	t	lb (000)	kg	oz (000)	kg	oz (000)	kg	oz (000)	kg	oz (000)	Tonnes	t	lb (000)	kg	lb
1	39,550	1.56	0.032	1.45	124	0.87	0			6,570	1,440	3,180	9,640	310	9,580	308	33,770	1,086	29,140	937	4,900	2,450	5,410	17,230	37,990
2	32,850	0.78	0.101	1.96	116	0.62	10,580	1.12	13.8	81,840	18,000	39,690	11,750	378	30,170	970	14,720	473	117,110	3,765	3,800	1,900	4,180	10,180	22,450
3	25,810	0.70	0.129	2.06	100	0.59	18,000	1.41	14.3	96,150	21,150	46,630	9,460	304	28,420	914	23,780	764	198,120	6,370	2,580	1,290	2,850	7,600	16,750
4	31,200	0.70	0.117	1.91	100	0.52	12,190	1.16	34.2	99,280	21,840	48,150	10,810	347	30,230	972	15,080	485	365,720	11,758	3,120	1,560	3,440	8,110	17,880
5	27,820	0.68	0.116	2.00	100	0.53	15,980	1.60	46.2	87,310	19,210	42,350	9,400	302	28,050	902	24,930	802	621,620	19,986	2,780	1,390	3,070	7,350	16,210
6	32,850	0.70	0.131	1.90	90	0.51	10,950	0.95	15.2	125,430	27,600	60,840	12,190	392	33,490	1,077	10,970	353	130,980	4,211	2,960	1,480	3,260	8,370	18,450
7	26,020	0.95	0.163	1.94	80	0.28	17,780	0.98	14.4	139,600	30,710	67,710	14,540	468	30,000	964	14,710	473	184,900	5,945	2,080	1,040	2,300	3,640	8,030
8	32,850	0.75	0.045	1.47	100	0.76	10,950	1.01	11.6	15,860	3,490	7,690	5,980	192	12,270	394	17,130	551	104,710	3,366	3,290	1,640	3,620	12,540	27,660
9	32,850	0.62	0.074	1.71	110	0.80	10,950	0.72	10.4	46,930	10,320	22,760	7,690	247	21,660	696	10,440	336	92,780	2,983	3,610	1,810	3,980	13,060	28,780
10	21,250	0.59	0.092	1.76	120	0.51	22,550	0.76	10.7	45,080	9,920	21,870	5,470	176	16,550	532	13,920	448	176,710	5,681	2,550	1,280	2,810	5,360	11,820
11	32,850	0.58	0.093	1.61	120	0.55	10,950	0.82	10.2	71,010	15,620	34,440	8,290	267	23,630	760	9,840	316	88,590	2,848	3,940	1,970	4,350	9,030	19,900
12	32,740	0.61	0.104	1.52	110	0.60	11,060	0.86	10.7	85,700	18,860	41,570	9,330	300	23,770	764	10,090	324	91,370	2,938	3,600	1,800	3,970	9,740	21,470
13	32,850	0.66	0.124	1.71	100	0.62	10,950	0.90	9.3	114,880	25,270	55,720	11,140	358	29,330	943	10,090	324	70,790	2,276	3,290	1,640	3,620	10,240	22,570
14	32,850	0.80	0.144	1.81	80	0.42	10,950	1.08	10.4	145,600	32,030	70,620	14,460	465	33,540	1,078	12,680	408	87,730	2,820	2,630	1,310	2,900	6,880	15,170
15	10,950	0.69	0.035	1.06	90	1.09	32,850	0.88	9.0	2,450	540	1,190	1,330	43	2,160	69	21,740	699	185,350	5,959	990	490	1,090	5,980	13,190
16	38,770	0.55	0.079	1.46	100	0.54	5,030	1.04	17.1	62,570	13,770	30,350	8,320	268	22,790	733	8,920	287	77,390	2,488	3,880	1,940	4,270	10,360	22,850
17	43,800	0.55	0.099	1.49	100	0.50	0			105,460	23,200	51,150	10,980	353	30,170	970	4,770	153	11,490	370	4,380	2,190	4,830	11,030	24,320
18	43,630	0.62	0.107	1.62	80	0.48	0			119,790	26,360	58,100	12,740	409	34,290	1,102	5,130	165	12,100	389	3,490	1,750	3,850	10,430	22,990
19	43,800	0.80	0.126	1.81	80	0.38	0			157,160	34,580	76,230	18,060	581	41,690	1,340	6,160	198	12,450	400	3,500	1,750	3,860	8,240	18,160
20	43,800	0.53	0.073	1.40	90	0.53	0			60,970	13,410	29,570	8,590	276	23,350	751	5,760	185	13,770	443	3,940	1,970	4,350	11,490	25,330
21	43,800	0.50	0.097	1.53	90	0.44	0			101,830	22,400	49,390	9,790	315	30,580	983	4,220	136	11,560	372	3,940	1,970	4,350	9,550	21,040
22	43,800	0.57	0.121	1.53	90	0.40	0			147,220	32,390	71,410	12,600	405	34,710	1,116	4,030	130	9,680	311	3,940	1,970	4,350	8,640	19,050
23	43,800	0.73	0.126	1.84	70	0.26	0			157,160	34,580	76,230	16,450	529	42,380	1,363	5,480	176	12,330	396	3,070	1,530	3,380	5,600	12,340
24	43,020	0.60	0.060	1.38	90	0.58	0			40,490	8,910	19,640	8,200	264	19,580	629	7,560	243	15,870	510	3,870	1,940	4,270	12,390	27,310
25	43,800	0.48	0.076	1.54	90	0.34	0			65,790	14,470	31,910	8,080	260	26,520	853	4,980	160	14,340	461	3,940	1,970	4,350	7,460	16,450
26	43,800	0.50	0.107	1.60	70	0.34	0			120,260	26,460	58,330	10,360	333	33,870	1,089	3,830	123	10,940	352	3,070	1,530	3,380	7,400	16,320
27	38,800	0.66	0.148	2.04	70	0.28	0			179,440	39,480	87,030	14,340	461	45,170	1,452	3,490	112	9,510	306	2,720	1,360	2,990	5,420	11,950
Total	**959,900**	**0.68**	**0.101**	**1.66**	**94**	**0.51**	**211,709**	**1.02**	**15.6**	**2,481,830**	**546,000**	**1,203,730**	**279,980**	**9,001**	**737,920**	**23,725**	**308,200**	**9,909**	**2,757,010**	**88,640**	**89,850**	**44,930**	**99,040**	**243,310**	**536,400**

16.2 MINERAL PROCESSING

16.2.1 INTRODUCTION

The proposed concentrator will process the gold/copper/molybdenum porphyry mineralization from the Snowfield deposit, and the gold/silver mineralization from the Brucejack deposit. The concentrator will be fed at a nominal rate of 120,000 t/d and with an availability of 92% (365 d/a). The feed materials from the two deposits will be processed separately in different time periods according to the mining schedule. The concentrator will produce a marketable copper concentrate containing gold and silver, a by-product molybdenum concentrate, and gold-silver doré during processing of the Snowfield mineralization, and only gold-silver doré during processing of the Brucejack mineralization.

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 flowsheets for both deposits are similar except the Brucejack flowsheet does not include copper cleaner flotation and copper/molybdenum separation.

The process plant will consist of three stages of crushing, primary grinding, followed by flotation processes to recover copper, gold, silver, and molybdenum from the Snowfield material, and only gold and silver from the Brucejack mineralization. The resulting bulk rougher/scavenger concentrates will be reground and gravity concentrated to recover free metallic gold. Due to a difference in the mineralization, the downstream processes for Snowfield mineralization and Brucejack mineralization are slightly different:

- For the Snowfield mineralization: a copper-gold-silver and molybdenum bulk cleaner flotation for the reground rougher concentrate and a copper-molybdenum separation circuit are proposed to produce a molybdenum concentrate and a copper concentrate containing gold and silver. The cleaner flotation tailing together with the reground rougher/scavenger concentrate will be cyanide leached to recover gold and silver. The recovered gold and silver will be refined on site to gold-silver doré. If gravity concentration is in operation, the gravity concentrate will be processed in an intensive leach circuit to recover gold and silver.
- For the Brucejack mineralization: a conventional cyanidation process will be used to leach the reground rougher and scavenger concentrates (after gravity concentration) to recover gold and silver; an intensive leach process will be used to recover gold and silver from the gravity concentrate. The recovered gold and silver will be refined on site to gold-silver doré.

The copper concentrate from Snowfield mineralization 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

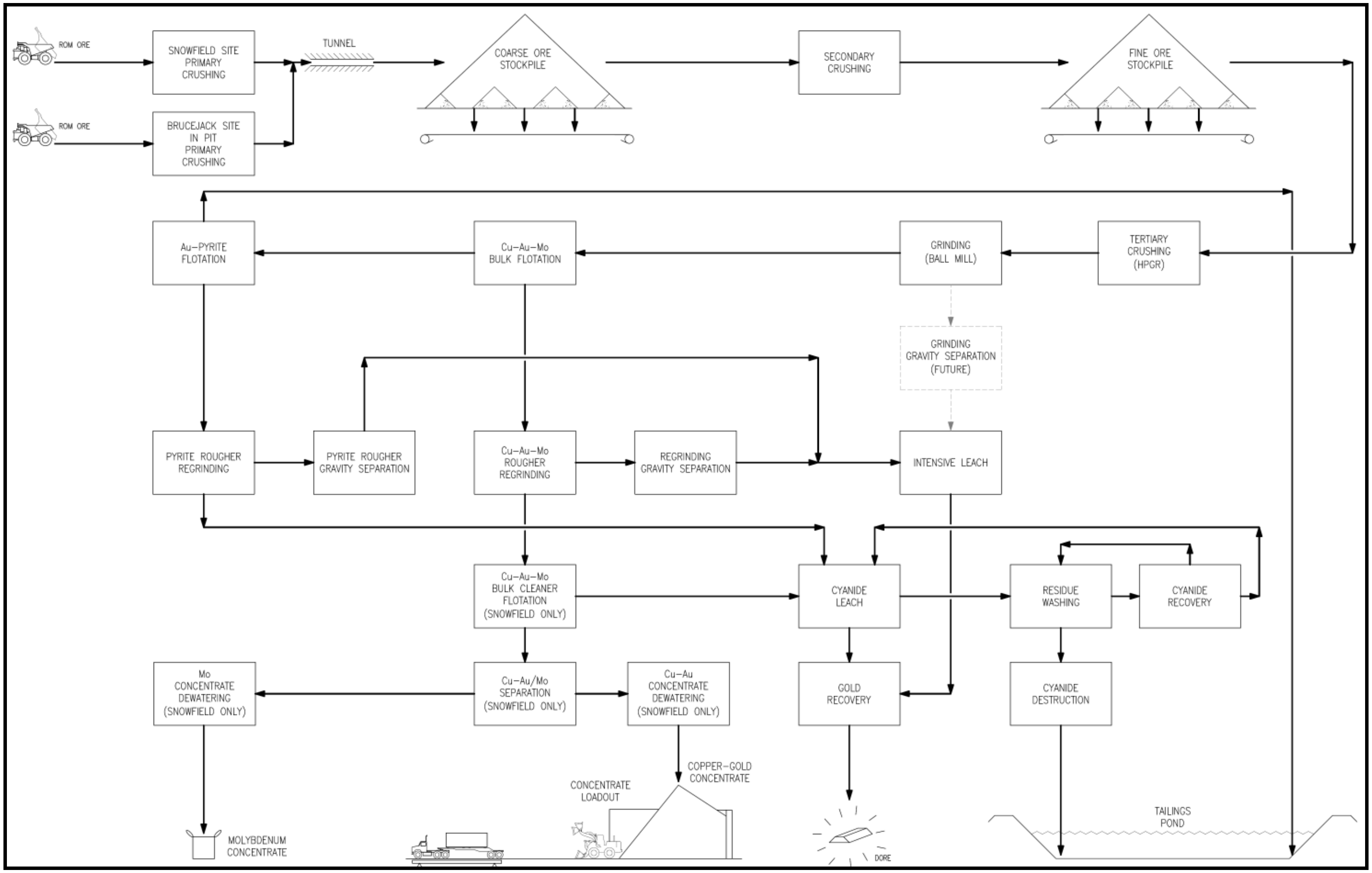
The mill flowsheet design is based on the results of grinding, flotation and leach testwork carried out by PRA, in 2009 and 2010, together with engineering experience.

The process plant will consist of the following unit operations:

- primary crushing
- conveying system
- coarse material stockpile and reclaim
- secondary and tertiary crushing
- grinding
- rougher/scavenger flotation
- rougher/scavenger concentrate regrinding and gravity concentration
- copper/gold-silver/molybdenum cleaner flotation and separation (Snowfield mineralization only) conventional cyanide leaching by carbon-in-leach (CIL) on the reground rougher/scavenger flotation concentrate for Brucejack mineralization, or on the copper and molybdenum flotation cleaner flotation tailing and scavenger concentrate for Snowfield mineralization
- intensive cyanide leach on the gravity concentrates
- gold recovery from gold loaded carbon and doré production
- cyanide recovery, destruction, and related processes
- copper and molybdenum flotation concentrate thickening, filtration, and dispatch
- tailing disposal to the tailing impoundment.

The simplified flowsheet is shown in Figure 16.19.

Figure 16.19 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 16.33.

Table 16.33 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/Gravity Concentration		
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
Free Gold Recovery from Reground Concentrate		Gravity Concentration
Leach Method – Reground Rougher/Scavenger Concentrates		CIL
Leach method – Gravity Separation Concentrates		Intensive Cyanide leach
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, regrinding, gravity concentration, 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.

According to the mining schedule, the mineralization from Snowfield and Brucejack sites will be processed separately. Each mineralization will be fed to the process plant for a period of no shorter than three months.

16.2.5 PROCESS PLANT DESCRIPTION

PRIMARY CRUSHING

Two separate primary crushing systems have been proposed for the Snowfield and Brucejack sites to crush mineralized materials from the proposed mines in order to reduce the size of the rocks in preparation for the grinding process. The crushing facilities will have the same average process rate of 7,143 t/h. A fixed conventional gyratory crushing facility consisting of two gyratory crushers is proposed for the Snowfield site, while two semi-mobile crushing stations are proposed for the Brucejack site.

The major equipment and facilities at each site include:

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

The primary crusher feed will be trucked from the proposed open pits 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 from the Snowfield site will be conveyed onto an overland conveying system, consisting of a series of overland belt conveyors, and transported to the crushed material stockpile at the plant site through a 26 km-long main tunnel.

The crushed materials from the Brucejack site will be conveyed to the transfer point within the 26-km main tunnel. The transfer point is approximately 4.8 km away from the Snowfield end of the main tunnel. There is a 5 km-long tunnel connecting the Brucejack site and the main tunnel.

The primary crushing facilities 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 stockpile will also 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 crushing will be conducted in 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.

The secondary crushing area will be equipped with a dust collection system to control fugitive dust that will be generated during crushing and transporting the crushed materials.

FINE MATERIAL STOCKPILE AND RECLAIM

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

- two covered 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 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 80% passing 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.

A dust collection system will be installed in the areas to control fugitive dust.

PRIMARY GRINDING AND CLASSIFICATION

The primary grinding circuit will consist of four ball mill circuits. The ball mills will be arranged in a closed circuit with 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.

The screened HPGR product will enter the grinding circuit via the cyclone feed pumpbox. Each ball mill will be operated independently in closed circuit with a cyclone cluster. 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 80% passing 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 AND REGRINDING CIRCUITS

The milled pulp will be subjected to flotation to recover the targeted minerals. For the Snowfield mineralization, three products will be produced in the flotation circuit: a copper concentrate containing gold and silver, a molybdenum concentrate, and a gold bearing pyrite concentrate. For the Brucejack mineralization, two gold bearing pyrite flotation concentrate products will be produced: a rougher flotation concentrate and a scavenger flotation concentrate.

Bulk Rougher Flotation/Regrinding/Gravity Separation

The bulk rougher flotation circuits will recover copper, gold, silver, and molybdenum bearing minerals from the Snowfield mineralization, or gold and silver bearing minerals from the Brucejack mineralization. The flotation circuit will include 24 rougher flotation tank cells (4 parallel trains, six 200 m^3 cells per train). The feed to each flotation train will be at a rate of 1,359 t/h. Flotation reagents will be added to the flotation circuits as defined through testing. The flotation reagents added will be the collectors (potassium amyl xanthate [PAX] and A208) and the frother (methyl isobutyl carbinol [MIBC]). When processing the Snowfield mineralization, 3926A and fuel oil will be added to improve molybdite recovery. Lime will be used as a pH modifier throughout the process as required. The concentrates produced from the rougher flotation circuits will be sent to the regrinding and gravity separation circuits. The mass recovery of the rougher concentrate is approximately 4% of the flotation feed. The tailings will flow to the subsequent scavenger floatation.

The concentrates from the four trains of the rougher flotation circuits will be reground to 80% passing 20 µm by four tower mills. The tower mills will be in closed circuit with two hydrocyclone clusters consisting of a total of 18 cyclones. A portion of the hydrocyclone underflow will be sent to a centrifugal concentrator to recover any coarse free gold nuggets. The gravity separation tailings will return to the hydrocyclone feed pumpbox. The centrifugal gravity concentrate is upgraded by tabling, together with the centrifugal gravity concentrates from the scavenger concentrate regrinding circuit. The table concentrate reports to a separate intensive leach circuit while the table tailing returns to the centrifugal gravity concentrator.

When processing the Snowfield mineralization, the hydrocyclone overflow will report to the copper and molybdenum bulk flotation circuits. If the rougher flotation feed is from the Brucejack deposit, the hydrocyclone overflow will report to the gold CIL circuit.

The rougher flotation, regrind and gravity separation circuits will consist of:

- rougher flotation tank cells (4 parallel trains, six 200 m^3 cells per train)
- four 1,119 kW regrind tower mills
- two cyclone clusters, each with eighteen 250 mm diameter cyclones
- one centrifugal gravity concentrator
- one gravity separation table
- slurry pumps
- sampling system
- particle size analyzer.

Bulk Scavenger Flotation/Regrinding/Gravity Separation

The rougher flotation tailings from both the Snowfield and Brucejack mineralization will be further floated to recover the gold and silver bearing minerals, mainly pyrite.

The flotation circuits will include the following equipment:

- scavenger flotation tank cells (4 trains, nine 200 m^3 tank cells per train)
- ten 1,119 kW regrinding tower mills (5 mills per train)
- two cyclone clusters (eighteen 250 mm diameter cyclones per cluster)
- particle size analyzer
- slurry pumps.

The pyrite flotation circuit will consist of four parallel lines following on from the rougher flotation circuit. 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 TSF.

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. A portion of the hydrocyclone underflow from each hydrocyclone cluster will be sent to a centrifugal concentrator to recover any coarse free gold nuggets. The gravity separation tailings will return to the hydrocyclone feed pumpbox. The centrifugal gravity concentrates will be upgraded by tabling in the rougher regrinding circuit.

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.

Copper/Gold/Molybdenum Cleaner Flotation – Snowfield Mineralization Only

When processing the Snowfield mineralization, the hydrocyclone overflow of the rougher concentrate regrinding circuit will gravity flow to the copper and molybdenum cleaner flotation circuits. The cleaner flotation will include the following equipment:

- 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 flotation reagents will be added in stage in the cleaner flotation and will include PAX, A208, 3926A and fuel oil as collectors, and MIBC as frother. Lime will be used as a pH modifier throughout the process as required. The hydrocyclone overflow together with the second cleaner flotation tailings will feed to the first cleaner flotation. The concentrate from the first cleaner flotation will feed the second cleaner flotation, with the second cleaner concentrate reporting to the third cleaner flotation. The concentrate from the third cleaner flotation 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 CIL feed thickener of the gold cyanide leach circuit.

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

Copper/Molybdenum Separation – Snowfield Mineralization Only

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 for conventional copper/molybdenum separation.

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 for shipment.

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

GOLD RECOVERY FROM FLOTATION/GRAVITY SEPARATION PRODUCTS

Conventional CIL Leaching – Flotation Products

The reground gold-bearing rougher and scavenger concentrates from the Brucejack mineralization, or the reground gold-bearing scavenger concentrate together with the first cleaner scavenger tailings from the Snowfield mineralization 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 slurry pH.

Sodium cyanide will be used to leach gold and silver 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.

The CIL leaching residue will leave the final leach tank and go over a carbon safety screen. The residue 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 and silver 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 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. 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.

Gold Electrowinning and Refining

The pregnant gold solution will be pumped from the pregnant solution tank to the electrowinning cells. Gold and silver 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 Recovery and Destruction

Prior to disposal in the TSF, the leach residue will undergo cyanide recovery and cyanide destruction processes. The circuits will include the following equipment:

- 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 first stage CCD thickener underflow will enter the second CCD thickener for further washing before it discharges to the agitated cyanide destruction tanks. The thickener overflow from the second CCD thickener is in a closed circuit and feeds the first stage CCD thickener.

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

CONCENTRATE HANDLING

Copper/Gold Concentrate

The copper/gold 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 approximately 60% solids. The concentrate stock tank will be an agitated tank which will serve as the feed tank for the concentrate filter. The pressure-type filter will be used for further concentrate dewatering. 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.

Molybdenum Concentrate

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

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, or leach gold and silver.

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 2 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 and the water from the proposed mine sites (pit water and runoff water). 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 three control rooms:

- primary crusher control room at the Snowfield site
- primary crusher control room at the Brucejack site
- 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, tunnels, 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 analyzer will analyze the flotation stages. A sufficient number of samples will be taken for on-line control and metallurgical accounting. Shift samples will be assayed in the assay laboratory.

On-stream particle size analyzers 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 conformity with generally accepted Canadian Institute of Mining, Metallurgy, and Petroleum (CIM) "Estimation of Mineral Resource and Mineral Reserves Best Practices" (2005) guidelines:

- **Inferred Mineral Resource:** "An 'Inferred Mineral Resource' is that part of a mineral resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes."
- **Indicated Mineral Resource:** "An 'Indicated Mineral Resource' is that part of a mineral resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings, and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed."
- **Measured Mineral Resource:** "A 'Measured Mineral Resource' is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings, and drillholes that are spaced closely enough to confirm both geological and grade continuity."

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 mineral resources in this estimate are conceptual in nature.

All resource estimation work reported herein was carried out by F.H. Brown (M.Sc. Eng., CPG, Pr.Sci.Nat.) and Eugene Puritch (P.Eng.), both independent QPs in terms of NI 43-101, from information and data supplied by Silver Standard.

Mineral resource modelling and estimation were carried out using the commercially available Gemcom GEMS™ 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 BRUCEJACK MINERAL RESOURCE ESTIMATE

The Brucejack mineral resource estimate encompasses six distinct modelled mineralization zones, namely the West Zone, Shore Zone, Gossan Hill Zone, Galena Hill Zone, SG Zone, and Bridge Zone. The effective date of this estimate is December 1, 2009.

17.2.1 PREVIOUS RESOURCE ESTIMATES

A previous public mineral resource estimate for the Brucejack deposit dated April 16, 2001, was prepared by Pincock Allen & Holt Ltd. [1] (PA&H). The mineral resource estimate reported a total Measured and Indicated mineral resource of 421,400 oz Au and an Inferred mineral resource of 82,000 oz of Au (Table 17.1), based on a gold-equivalent (Au-Eq) cut-off derived from an Ag:Au equivalency ratio of 66:1.

Table 17.1 Brucejack Mineral Resource Estimate – PA&H (April 16, 2001)

Zone	Class	Au-Eq Cut-off (oz/t)	t (000)	Au (g/t)	Ag (g/t)	Au (oz x 000)	Ag (oz x 000)
West	Measured	0.1	144.0	15.09	594	69.8	2,750.4
West	Indicated	0.1	899.5	10.98	482	317.5	13,942.3
Shore	Indicated	0.2	92.3	11.54	143	34.2	424.6
Total	**Indicated**		**991.8**	**11.03**	**451**	**351.8**	**14,366.8**
Total	**M+I***		**1,135.8**	**11.54**	**470**	**421.4**	**17,150.6**
West	Inferred	0.1	51.6	5.82	249	9.6	412.8
SG	Inferred	0.2	46.2	9.21	25	13.7	37.0
Galena Hill	Inferred	0.2	30.9	24.39	271	24.2	268.8
Gossan Hill	Inferred	0.2	22.6	47.34	62	34.4	45.2
Total	**Inferred**		**51.3**	**16.86**	**156**	**82.0**	**756.5**

* Measured + Indicated.

[1] *Sulphurets-Bruceside Property BC Technical Report.* PA&H, dated April 16, 2001.

17.2.2 SAMPLE DATABASE

Sample data were provided by Silver Standard in the form of ASCII text files and Excel spreadsheets. Data included historical surface drilling records, historical underground drilling records, and current Silver Standard drilling records.

The supplied databases contain records for 929 drillholes. Of these, 85 drillholes were outside the block model limits or had no reported assay data.

The 844 drillhole records (Table 17.2) used for this mineral resource estimate contain collar, survey, and assay data. Assay data fields consist of the drillhole ID, downhole interval distances, sample number, Au grades, and Ag grades. All data are in metric units and all collar coordinates were converted by Silver Standard to the UTM NAD27 system.

Table 17.2 Brucejack Drilling Database Records

Data Type	Record Count
Historical Surface Drilling	365
Historical Underground Drilling	442
Silver Standard Surface Drilling	37
Total	**844**

The database contains a total of 51,985 Au assays and 51,049 Ag assays. Due to the varying assay protocols in use during different project phases, the following low grade conversions were used:

- For historical drilling, Au assay grades less than 0.17 g/t were converted to 0.085 g/t, and Ag assay grades less than 1.71 g/t were converted to 0.85 g/t.
- For the current Silver Standard drilling program, Au assay grades less than 0.005 g/t were converted to 0.0025 g/t, and Ag assay grades less than 0.5 g/t were converted to 0.025 g/t.

Silver Standard also provided an AutoCAD format wireframe of the historical underground mining development at the West Zone. Historic mine plans were used to digitize the underground development. Underground workings were digitized on 44 east-west sections in the mine grid coordinate system using AutoCAD software. Section lines were generally spaced every 10 m, with a reduction to 5 m spacing in areas of more complex development (i.e. in areas of multiple tunnels, junctions, etc.). The digitized data were converted to UTM NAD27 coordinates using the McElhanney conversion factors, imported into the Gemcom mining software, and used to generate a single three dimensional solid to represent the underground workings.

17.2.3 DATABASE VALIDATION

Industry standard validation checks were completed on the supplied database, and minor corrections were made. P&E typically 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 drill hole length, inappropriate collar locations, and missing interval and coordinate fields. No significant discrepancies with the supplied data were noted.

Downhole surveys for the current drilling 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 190 survey measurements reported for the current Silver Standard drilling program, three measurements reported a downhole survey deviation from the previous measurement of greater than 5°, with a maximum reported difference of 10.2°.

Surface drillhole orientations were also reviewed by P&E. Eleven historical surface drillholes reported a plunge of less than 30°, and should be reviewed against historical records. P&E notes that the majority of the surface drillholes completed by Silver Standard are sub-parallel to identified structural orientations at Brucejack, and recommends additional drilling moving forward to further define mineralization trends in the project area.

17.2.4 TOPOGRAPHIC CONTROL

Aerial photography specialist Aero Geometrics Ltd. (Aero Geometrics) was contracted by Silver Standard to produce a topographic map of the Brucejack property. Using high-resolution photographs taken from a small airplane in 2008, a photo-mosaic was first made of the Brucejack and adjoining Snowfield properties. Using this photo-mosaic and elevation data obtained from 1:50,000-scale national topographic maps published in 1979 by the Surveys and Mapping Branch of the Department of Energy, Mines, and Resources, Aero Geometrics digitally generated a contoured topographic map with contour lines spaced at 2-m intervals and presented this map as digital elevation model (DEM) in dxf (AutoCAD) format.

In order for this topographic map to be consistent with the NAD27 Zone 9 UTM grid system being used by Silver Standard for the project, it was necessary to make minor adjustments (vertical and lateral shifts) to the positioning of the DEM. These adjustments were carried out by various workers, including geological consultants and McElhanney technicians, and were checked against numerous topographic points (historic and 2009 Brucejack drill hole collars, the western shoreline of Brucejack Lake, and historic mine grid stations) that had been surveyed by McElhanney field crews in 2009.

17.2.5 DENSITY

A total of 312 specific gravity measurements were provided by Silver Standard, with an average specific gravity of 2.73 (Table 17.3). Specific gravity measurements were obtained from assay pulps by ALS Chemex. For the West Zone, a value of 2.75 was used historically [2]. As this value is in good agreement with the average density reported by Silver Standard, a global specific gravity value of 2.75 was assigned to all lithologies for this Mineral Resource estimate.

Table 17.3 Brucejack Specific Gravity Statistics

Count	312
Minimum	2.21
Maximum	3.28
Average	2.73
Standard Deviation	0.14

17.2.6 BRUCEJACK DOMAIN MODELLING

Six mineralization zones at Brucejack have been identified by Silver Standard, with the West Zone and Shore Zone considered by Silver Standard to be predominately structurally controlled vein systems related to the north-trending Brucejack Fault, and the other zones tentatively defined as mineralized stockwork/breccia/vein systems.

The overall trend of the West Zone and Shore Zone mineralization is ~135°, and modelling for these zones was generating from successive polylines spaced every 10 m and oriented perpendicular to the trend of the mineralization. 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. All polyline vertices were snapped directly to drillhole assay intervals, in order to generate a true three-dimensional representation of the extent of the mineralization.

For the Gossan Hill Zone, Galena Hill Zone, SG Zone and Bridge Zone, mineralization models were generated from successive polylines spaced every 25 m and oriented north-south. 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. All polyline vertices were snapped directly to drillhole assay intervals, in order to generate a true three-dimensional representation of the extent of the mineralization.

[2] ibid.

In order to ensure that all potential economic mineralization was captured for mineral resource estimation, a secondary mineralization halo for the Bridge Zone was subsequently modelled using a 0.2 g/t Au-Eq value, based on the parameters listed in Table 17.4. The highest assay sample grades reported for the current Silver Standard drilling program (16,948.5 g/t Au and 8,695.5 g/t Ag) were recovered from inside this halo, highlighting the potential for high-grade precious metal vein systems within the Bridge Zone.

Three-dimensional models of the mineralization domains were then created by combining successive polylines into wireframes. P&E notes that the use of large-scale mineralization domaining is likely to bias the contribution of high grade veins within the various mineralization domains.

Table 17.4 Au-Eq Parameters

Commodity	Price	Recovery	Au Equivalency
Au	US$800/oz	75%	1.00
Ag	US$12/oz	73%	0.015

17.2.7 *COMPOSITING*

Assay sample lengths for the database range from 0.01 m to 48.00 m, with an average sample length of 1.52 m. A compositing length of 1.50 m was therefore selected for use. Length-weighted composites were calculated for Au and Ag within the defined mineralization domains. Missing sample intervals in the historical data were assigned a nominal background grade of 0.001 g/t Au or 0.001 g/t Ag.

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. Each assay and composite 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.

17.2.8 *EXPLORATORY DATA ANALYSIS*

Summary assay statistics (Table 17.5) and summary composite statistics (Table 17.6) were calculated by domain for each commodity. Comparison of the data sets suggests that, moving forward, additional drilling will be required in order to identify individual higher grade vein sets within the defined mineralization domains.

Table 17.5 Brucejack Summary Assay Statistics by Domain

	Total	West Zone	Shore Zone	Gossan Hill	Galena Hill	SG Zone	Bridge Zone	BZ Halo
Ag Assays								
Samples	34,728	22,774	1,523	1,876	2,199	347	3,511	2,498
Minimum	0.25	0.86	0.86	0.70	0.25	0.25	0.25	0.25
Maximum	37,636.40	37,636.40	15,340.50	2,982.86	1,490.00	130.97	774.00	8,695.50
Mean	87.65	124.25	49.05	13.64	19.53	8.81	10.13	12.99
St Dev	595.82	721.08	426.54	77.48	63.20	10.23	34.47	195.93
CV	6.80	5.80	8.70	5.68	3.24	1.16	3.40	15.08
Au Assays								
Samples	34728	22774	1523	1876	2199	347	3511	2498
Minimum	0.002	0.012	0.085	0.002	0.009	0.005	0.002	0.002
Maximum	16948.500	2519.900	9490.870	368.260	135.600	18.510	384.000	16948.500
Mean	3.736	3.305	13.723	1.456	1.290	1.475	0.979	9.632
St Dev	116.878	34.311	293.532	12.293	4.535	2.134	7.087	355.540
CV	31.284	10.381	21.390	8.444	3.515	1.446	7.241	36.912

Table 17.6 Brucejack Summary Composite Statistics by Domain

	Total	West Zone	Shore Zone	Gossan Hill	Galena Hill	SG Zone	Bridge Zone	BZ Halo
Ag Composites								
Samples	33,684	22,014	1,676	1,774	1,990	321	3366	2,543
Minimum	0.001	0.001	0.402	0.001	0.250	0.379	0.250	0.149
Maximum	2,8781.600	2,8781.600	1,410.690	1,493.950	905.188	93.327	653.720	4,572.290
Mean	66.360	94.090	28.231	11.761	17.309	7.977	9.102	11.073
St Dev	373.990	457.055	84.499	53.733	42.963	8.590	23.750	126.506
CV	5.636	4.858	2.993	4.569	2.482	1.077	2.609	11.425
Au Composites								
Samples	33,684	22,014	1,676	1774	1,990	321	3,366	2,543
Minimum	0.001	0.001	0.034	0.001	0.029	0.007	0.040	0.002
Maximum	8,905.310	1,639.000	701.964	173.214	90.694	14.025	97.147	8,905.310
Mean	2.755	2.710	2.849	1.209	1.164	1.294	0.887	8.061
St Dev	68.867	22.597	22.387	6.980	3.069	1.791	3.007	240.831
CV	24.997	8.338	7.857	5.772	2.636	1.384	3.390	29.875

17.2.9 *TREATMENT OF EXTREME VALUES*

The presence of high-grade outliers was evaluated by examining composite cutting graphs, histograms, and log-probability graphs for the defined mineralization domains. For the Bridge Zone, Galena Hill, Gossan Hill, SG Zone, and Shore Zone, threshold values (Table 17.7) were selected that minimize rapid changes in the composite sample distribution. The influence of composite samples equal to or higher than the threshold value selected was restricted during estimation to 30 m, in order to limit the influence of higher grade assay values on the overall stockwork mineralization captured within the defined mineralization domains.

For the West Zone and the Bridge Zone Halo, capping limits were implemented on composites exceeding these values during estimation.

Table 17.7 Brucejack Capping and Threshold Values

Commodity	Au (g/t)	Ag (g/t)
Bridge Zone	8.00	160
Bridge Zone Halo	4.00	160
Galena Hill	16.00	260
Gossan Hill	20.00	160
SG Zone	10.00	40
Shore Zone	80.00	550
West Zone	60.00	2,300

17.2.10 VARIOGRAPHY

For the Bridge Zone, Galena Hill, Gossan Hill, SG Zone, and Shore Zone mineralization 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 correlograms were derived from composite values.

For the West Zone, indicator exponential correlograms based on 8.00 g/t Au and 90 g/t Ag discriminators were generated for azimuths spaced 30° apart and calculated at dips of 0°, 30°, 60° and 90°. The correlograms were derived from composite values.

The resulting ellipsoids were used as the basis for estimation search ranges, distance calculations and mineral resource classification (Table 17.8). The correlograms represent ranges of continuity somewhat commensurate with the drilling spacing and cannot therefore be considered to be truly representative of the underlying mineralization.

Table 17.8 Brucejack Domain Anisotropy Definitions

		Range	Azimuth	Dip	Range	Azimuth	Dip
		Bridge Zone			**Galena Hill**		
Au	**Z**	7	121	30	13	192	45
	Y	59	304	60	94	324	34
	X	387	032	-1	30	073	26
	CO	0.7			0.6		
	C1	0.3			0.4		
Ag	**Z**	559	026	45	221	286	43
	Y	33	299	-3	29	307	-45
	X	62	032	-45	43	026	10
	CO	0.8			0.8		
	C1	0.2			0.2		
		SG Zone			**Shore Zone**		
Au	**Z**	66	197	74	22	304	25
	Y	271	290	1	132	307	-65
	X	19	021	16	18	035	1
	CO	0.4			.9		
	C1	0.6			.1		
Ag	**Z**	18	181	37	15	327	37
	Y	190	266	-45	206	010	-45
	X	25	331	23	26	075	23
	CO	0.5			0.8		
	C1	0.5			0.2		
		Gossan Hill			**West Zone Indicator**		
Au	**Z**	4	159	50	45	281	50
	Y	37	269	17	34	32	17
	X	9	012	35	9	008	35
	CO	0.5			0.7		
	C1	0.5			0.3		
Ag	**Z**	11	17	43	50	254	71
	Y	9	330	-36	41	303	-12
	X	20	80	-26	15	030	14
	CO	0.4			0.6		
	C1	0.6			0.4		

17.2.11 Block Models

An orthogonal block model was established across the property for the Bridge Zone, Galena Hill, Gossan Hill, SG Zone, and Shore Zone mineralization domains (Table 17.9). A separate rotated block model was established for the West Zone (Table 17.10). Each block model consists of separate models for Au estimated grades, Ag estimated grades, associated rock codes, percent, density and classification attributes, and a calculated Au-Eq grade. A percent block model was used to accurately represent the volumes and tonnages that were contained within the respective mineralization domains. As a result, domain boundaries were properly represented by the percent model's capacity to measure infinitely variable inclusion percentages within a specific domain. The volume of the defined historical workings was also calculated for the West Zone and depleted from the model prior to estimation.

Table 17.9 Brucejack Block Model Setup

	Origin	Blocks	Size
X	425,800	80	25 m
Y	6,256,500	140	25 m
Z	2,000	100	10 m
Rotation	None		

Table 17.10 West Zone Block Model Setup

	Origin	Blocks	Size
X	426,800	150	10 m
Y	6,257,600	150	10 m
Z	1,600	90	10 m
Rotation	40°		

17.2.12 *Estimation and Classification*

The mineral resource estimate was constrained by wireframes that form hard boundaries between the respective composite assay data files. Individual block grades were used to calculate an Au-Eq grade model.

For the Bridge Zone, Galena Hill, Gossan Hill, SG Zone, and Shore Zone mineralization domains, block grades were estimated using Inverse Distance Cubed (ID3) linear weighting of composite values. The following two-pass series of expanding search ellipses with varying minimum sample requirements was used for sample selection and classification, and sample distances were adjusted by the defined anisotropy:

- During the first pass, 8 to 12 composite values from 2 or more drillholes within a search ellipse corresponding to the defined ranges were required for estimation. All block grades estimated during the first pass were classified as Indicated, with a total of 6,275 blocks estimated.
- During the second pass, blocks not populated during the first pass were estimated. There were 3 to 12 composite values from 1 or more drillholes

within a search ellipse corresponding to about 200% of the defined range that were required for estimation. All block grades estimated during the second pass were classified as Inferred, with a total of 10,049 blocks estimated.

For the West Zone mineralization domain, the block estimates were calculated using a two-bin Indicator Kriging (IK) partition of each commodity. Based on the defined indicator correlograms, for each block a high-grade probability, a high grade estimate and a low-grade estimate were calculated and then combined into a single block estimate. The following three-pass series of expanding search ellipses with varying minimum sample requirements were used for sample selection and classification:

- During the West Zone first pass, 12 composite values from 1 or more drillholes within a search ellipse corresponding to 15% of the defined range were required for estimation. All block grades estimated during the first pass were classified as Measured, with a total of 4,433 blocks estimated.
- During the West Zone second pass, blocks not populated during the first pass were estimated. There were 8 to 12 composite values from 1 or more drillholes within a search ellipse corresponding to 100% of the defined range that were required for estimation. All block grades estimated during the second pass were classified as Indicated, with a total of 9,500 blocks estimated.
- During the West Zone third pass, blocks not populated during the first or second pass were estimated. There were 3 to 12 composite values from 1 or more drillholes within a search ellipse corresponding to 200% of the defined range that were required for estimation. All block grades estimated during the third pass were classified as Inferred, with a total of 770 blocks estimated.

For the Bridge Zone Halo mineralization domain, block estimates were calculated using ID3 weighting of the nearest six capped composites within the defined Bridge Zone Halo. All blocks estimated during this pass were classified as Inferred, with a total of 18,325 blocks estimated.

17.2.13 Brucejack Mineral Resource Estimate

In order to ensure that the reported mineral resources meet the CIM requirement for “reasonable prospects for economic extraction”, conceptual Lerchs-Grossman optimized pit shells were developed based on all available mineral resources (Measured, Indicated, and Inferred), using the economic parameters listed in Table 17.11.

Based on knowledge of mineral resource projects in the vicinity of Snowfield, Silver Standard mandated the use of a 0.35 g/t Au-Eq cut-off for the reporting of mineral resources at Snowfield. The results from the optimized pit-shells are used solely for the purpose of reporting mineral resources that have reasonable prospects for economic extraction.

Table 17.11 Optimized Pit Shell Parameters

Area	Parameter
Tailings & Water	US$0.80/rock tonne
Mining Cost	US$1.75/rock tonne
Processing Cost	US$5.00/ore tonne
Process Recovery	75%
G&A	US$1.00/ore tonne
Pit Wall Slope Angle	50°

All mineral resources were tabulated against a 0.35 g/t Au-Eq cut-off, as constrained within the optimized pit shell (Table 17.12 and Table 17.13).

Table 17.12 Combined Mineral Resource Estimate – 0.35 g/t Au-Eq Cut-off [1,2,3]

Class	Mt	Au (g/t)	Ag (g/t)	Au (M oz)	Ag (M oz)
Measured	9.9	2.06	75.0	0.66	23.8
Indicated	110.7	0.95	11.7	3.38	41.6
Measured + Indicated	120.5	1.04	16.9	4.04	65.4
Inferred	198	0.76	11.2	4.87	71.5

[1] Resources 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.

Table 17.13 Mineral Resource Estimates by Domain – 0.35 g/t Au-Eq Cut-off [1,2,3]

	Mt	Au (g/t)	Ag (g/t)	Au (M oz)	Ag (M oz)
Bridge Zone					
Measured	0.0	0.00	0.0	0.00	0.0
Indicated	88.2	0.80	7.9	2.27	22.4
Measured + Indicated	88.2	0.80	7.9	2.27	22.4
Inferred	80.4	0.81	12.9	2.09	33.5
Bridge Zone Halo					
Measured	0.0	0.00	0.0	0.00	0.0
Indicated	0.0	0.00	0.0	0.00	0.0
Measured + Indicated	0.0	0.00	0.0	0.00	0.0
Inferred	88.9	0.67	9.5	1.90	27.2
SG Zone					
Measured	0.0	0.00	0.0	0.00	0.0
Indicated	0.0	0.00	0.0	0.00	0.0
Measured + Indicated	0.0	0.00	0.0	0.00	0.0
Inferred	1.1	1.27	7.6	0.05	0.3
Shore Zone					
Measured	0.0	0.00	0.0	0.00	0.0
Indicated	1.6	5.40	26.3	0.27	1.3
Measured + Indicated	1.6	5.40	26.3	0.27	1.3
Inferred	1.6	1.59	12.0	0.08	0.6
Gossan Hill					
Measured	0.0	0.00	0.0	0.00	0.0
Indicated	0.5	2.59	10.6	0.04	0.2
Measured + Indicated	0.5	2.59	10.6	0.04	0.2
Inferred	9.0	0.92	12.6	0.27	3.7
Galena Hill					
Measured	0.0	0.00	0.0	0.00	0.0
Indicated	6.9	1.10	17.8	0.25	4.0
Measured + Indicated	6.9	1.10	17.8	0.25	4.0
Inferred	16.6	0.87	11.0	0.47	5.9
West Zone					
Measured	9.9	2.06	75.0	0.66	23.8
Indicated	13.5	1.27	31.7	0.55	13.8
Measured + Indicated	23.4	1.61	50.0	1.21	37.6
Inferred	0.5	1.01	30.8	0.02	0.5

[1] 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.

17.2.14 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. Visual validation of the block estimates combined with observed differences in the summary statistics suggests that the impact of high-grade vein systems on the mineral resource estimate will need to be further evaluated moving forward, especially in the Shore Zone and Bridge Zone.

A validation check for global bias was completed by comparing the modelled block estimates to a Nearest Neighbour (NN) block estimate generated using the same search criteria and tabulated at a zero grade cut-off within the constraining pit-shells. Results correctly duplicated grade trends and demonstrate a minimal global bias and slight smoothing for the modelled estimates as compared to the NN estimates.

17.3 SNOWFIELD MINERAL RESOURCE ESTIMATE

The effective date of this mineral resource estimate is **July 27, 2010**.

17.3.1 PREVIOUS RESOURCE ESTIMATES

A previous mineral resource estimate dated April 21, 2008 for the Snowfield deposit was prepared by Minorex Consulting Ltd. [3]. The mineral resource estimate reported a Measured and Indicated mineral resource of 3.08 M oz Au and an Inferred mineral resource of 0.47 M oz Au in-situ (Table 17.14) using a cut-off grade of 0.1g/t Au. The estimate was based on the results of 51 drillholes and 15 sample trenches, and used a global specific gravity of 2.82.

Table 17.14 Snowfield Mineral Resource Estimate, April 21, 2008 – 0.1 g/t Au Cut-off [1]

Class	Mt	Au (g/t)	Au (oz x 000)
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] 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.

[3] *Technical Report on the Snowfield Property, Skeena Mining Division, British Columbia, Canada.* Minorex Consulting Ltd., dated April 21, 2008.

P&E prepared a mineral resource estimate for the Snowfield deposit dated January 31, 2009 [4]. The mineral resource estimate reported a Measured and Indicated mineral resource of 4.36 M oz Au and an Inferred mineral resource of 14.28 M oz Au (Table 17.15) using a cut-off of 0.5 g/t Au-Eq. The estimate was based on the results of 113 drillholes and constrained within an optimized conceptual pit shell.

Table 17.15 Snowfield Mineral Resource Estimate, January 31, 2009 – 0.5 g/t Au-Eq Cut-off

Class	Mt	Au (g/t)	Au (M oz)	Ag (g/t)	Ag (M oz)	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

P&E prepared a further mineral resource estimate for the Snowfield deposit dated December 1, 2009 [5]. The mineral resource estimate reported a Measured and Indicated mineral resource of 19.77 M oz Au and an Inferred mineral resource of 10.05 M oz Au (Table 17.16) using a cut-off of 0.35 g/t Au-Eq. The estimate was based on the results of 141 drillholes and constrained within an optimized conceptual pit shell.

Table 17.16 Snowfield Mineral Resource Estimate, December 1, 2009 – 0.35 g/t Au-Eq Cut-off

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

17.3.2 SNOWFIELD 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,

[4] *Technical Report and Resource Estimate on the Snowfield Property, Skeena Mining Division, British Columbia*. P&E, dated January 31, 2009.
[5] *Technical Report and Updated Resource Estimate on the Snowfield Property, Skeena Mining Division, British Columbia*. P&E, dated December 1, 2009.

for 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, lithology, alteration, and assay data (Table 17.17). 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 coordinates are in the UTM NAD27 system. Assay values equal to the lower detection limit were converted to half of the lower detection limit.

Table 17.17 Snowfield Database Records

Data Type	Record Count
Collars	157
Survey Records	989
Assay Records (Au)	35,937

17.3.3 *Database Validation*

Industry standard validation checks were completed on the supplied database, and minor corrections made. P&E typically 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 drill hole 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 1 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.

17.3.4 *Topographic Control*

Silver Standard contracted McElhanney to produce a detailed topographic plan of the Snowfield project area. This plan, drafted at a 1:2,000-scale and displayed with the NAD27 Zone 9 UTM grid, covers an area of 2.85 km^2, with the northwest corner of the area located at 423,900 mE, 6,265,500 mN and the southeast corner having coordinates 425,400 mE, 6,263,600 mN. To generate the topographic contours for this area, with the contour interval being at 1 m, McElhanney used a 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 4 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 and this was then converted to the NAD27 system using national Transformation Model NTv2Points.

17.3.5 *Specific Gravity*

A total of 439 specific gravity measurements were provided by Silver Standard, with an average specific gravity of 2.78 t/m^3. Density measurements were obtained from assay pulps by ALS Chemex. Specific gravity measurements were back-tagged to the relevant domain and assigned as specific gravity values for mineral resource estimation (Table 17.18).

Table 17.18 Snowfield Specific Gravity Statistics

Count	439
Minimum	2.34 t/m^3
Maximum	3.17 t/m^3
Average	2.78 t/m^3

17.3.6 *Snowfield Domain Modelling*

Silver Standard supplied detailed logging information related to lithology and observed alteration types. Statistical analysis and visual examination of the lithology and alteration data indicated that grade distribution is partially related to the recorded alteration type for Au, Ag, and Mo.

Alteration domains as defined by Silver Standard (Table 17.19) were generated by computer screen digitizing of successive polylines on sequential drillhole sections spaced 25 m apart. 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 alteration domains was then created by combining successive polylines into the corresponding wireframes. In order to ensure that all potential economic mineralization was captured for mineral resource estimation, a secondary low-grade halo was subsequently modelled based on the extent of observed low-grade Au mineralization.

Table 17.19 Primary Alteration Domains

Alteration Domain	Assigned Rock Code
Intermediate Argillic	140
Propylitic	160
Distal Potasic	130
Silica	180
Sericite	170

For Cu, the mineral resource model was split into upper and lower domains independent of the defined alteration domains, based on an observed split between low grade and high grade Cu assay values.

17.3.7 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, and Mo within the defined alteration domains and for Cu within the defined upper and lower grade 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 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.

17.3.8 EXPLORATORY DATA ANALYSIS

Summary assay statistics (Table 17.20) and summary composite statistics (Table 17.21) 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 differences in grade distributions within the domains.

Assay sample populations drawn from the trenching data and the drillhole data were also examined by commodity. The trenching assay data show a positive bias for Au and Ag when compared to the local 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. The trenching data were therefore used while defining the extent of the mineralization, but were not used for mineral resource estimation.

Table 17.20 Summary Assay Statistics by Domain

	Total	130	140	160	170	180
Au (ppm)						
Samples	31,560	5,188	10,810	668	9,515	5,379
Minimum	0.0025	0.0025	0.0025	0.0060	0.0025	0.0025
Maximum	53.8000	8.3100	10.0000	1.8300	14.5500	53.8000
Mean	0.6657	0.5511	0.7524	0.2271	0.5832	0.8024
St Dev	0.6685	0.5701	0.6560	0.1855	0.5429	0.9145
CV	1.0043	1.0345	0.8718	0.8168	0.9309	1.1397
Ag (ppm)						
Samples	31,563	5,188	10,811	668	9,517	5,379
Minimum	0.2500	0.2500	0.2500	0.2500	0.2500	0.2500
Maximum	110.0000	91.2000	100.0000	22.7000	90.5000	110.0000
Mean	1.7455	1.5205	1.7456	1.4690	1.6691	2.1318
St Dev	1.9725	2.4455	2.0243	1.5170	1.4351	2.1635
CV	1.1300	1.6083	1.1596	1.0326	0.8598	1.0149
Mo (%)						
Samples	31,559	5,184	10,811	668	9,517	5,379
Minimum	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001
Maximum	0.2900	0.1400	0.2900	0.0621	0.1620	0.1090
Mean	0.0089	0.0082	0.0097	0.0062	0.0095	0.0075
St Dev	0.0079	0.0070	0.0085	0.0062	0.0087	0.0055
CV	0.8841	0.8526	0.8778	1.0022	0.9145	0.7365

	Total	High	Low
Cu (%)			
Samples	31,648	18,538	13,110
Minimum	0.0001	0.0001	0.0001
Maximum	4.0900	4.0900	0.8750
Mean	0.0999	0.1400	0.0432
St Dev	0.0764	0.0726	0.0340
CV	0.7649	0.5185	0.7872

Table 17.21 Summary Composite Statistics by Domain

	Total	130	140	160	170	180
Au (ppm)						
Samples	32,356	5,287	11,049	676	9,922	5,422
Minimum	0.0025	0.0044	0.0043	0.0064	0.0025	0.0057
Maximum	52.9863	7.8833	9.5785	1.4986	9.9701	52.9863
Mean	0.6710	0.5570	0.7570	0.2260	0.5950	0.8020
St Dev	0.6490	0.5590	0.6340	0.1680	0.5380	0.8820
CV	0.9672	1.0036	0.8375	0.7434	0.9042	1.1000
Ag (ppm)						
Samples	32,356	5,287	11,049	676	9,922	5,422
Minimum	0.2500	0.2500	0.2500	0.2500	0.2500	0.2500
Maximum	100.0000	75.7421	100.0000	19.7101	15.4180	91.0362
Mean	1.7530	1.5230	1.7640	1.4660	1.6670	2.1490
St Dev	1.8700	2.2810	2.0020	1.4230	1.0340	2.2770
CV	1.0667	1.4977	1.1349	0.9707	0.6203	1.0596
Mo (%)						
Samples	32,356	5,287	11,049	676	9,922	5,422
Minimum	0.0001	0.0001	0.0001	0.0001	0.0001	0.0001
Maximum	0.2349	0.0896	0.2349	0.0512	0.1301	0.0893
Mean	0.0090	0.0080	0.0100	0.0060	0.0100	0.0080
St Dev	0.0070	0.0060	0.0070	0.0050	0.0080	0.0060
CV	0.7778	0.7500	0.7000	0.8333	0.8000	0.7500

	Total	High	Low
Cu (%)			
Samples	32,323	18,753	13,570
Minimum	0.0007	0.0010	0.0007
Maximum	3.6527	3.6527	0.8744
Mean	0.0990	0.1400	0.0430
St Dev	0.0730	0.0680	0.0320
CV	0.7374	0.4857	0.7442

17.3.9 *Treatment of Extreme Values*

The presence of high-grade outliers was evaluated by examining composite coefficient of variation (CV) cutting graphs, histograms, and log-probability graphs. 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, and composites were capped to this value prior to estimation (Table 17.22).

Table 17.22 Threshold Values

Commodity	Capping Level
Ag	10 g/t
Au	6.00 g/t
Mo	0.10%
Cu	0.80%

17.3.10 VARIOGRAPHY

Contact analysis of the alteration domains indicated little change between alteration domains 130 and 140 and between alteration domains 160 and 170. These domains were therefore combined for variographic analysis and for estimation. Experimental semi-variograms were modelled as isotropic structures using uncapped composites (Table 17.23).

Table 17.23 Snowfield Semi-variogram Definitions

Element	Zone	Domain	Experimental Semi-variogram
Ag	1	130+140	0.20 + SPH (0.10, 130 m) + SPH (0.70, 450 m)
	2	160+170	0.20 + SPH (0.40, 20 m) + SPH (0.40, 340 m)
	3	180	0.20 + SPH (0.10, 170 m) + SPH (0.70, 300 m)
Au	1	130+140	0.10 + SPH (0.20, 70 m) + SPH (0.70, 140 m)
	2	160+170	0.30 + SPH (0.30, 25 m) + SPH (0.40, 250 m)
	3	180	0.40 + SPH (0.40, 50 m) + SPH (0.20, 30 m)
Mo	1	130+140	0.10 + SPH (0.50, 10 m) + SPH (0.40, 50 m)
	2	160+170	0.10 + SPH (0.40, 17 m) + SPH (0.50, 165 m)
	3	180	0.20 + SPH (0.30, 10 m) + SPH (0.50, 130 m)
Cu	1	High Grade	0.10 + SPH (0.90, 160 m)
	2	Low Grade	0.20 + SPH (0.30, 15 m) + SPH (0.50, 150 m)

17.3.11 BLOCK MODEL

An orthogonal block model was established across the property (Table 17.24), consisting of separate models for estimated grades, associated rock codes, percent, density and classification attributes and a calculated Au-Eq grade. A percent block model was used to accurately represent the volume and tonnage that was contained within the constraining mineralization halo. As a result, the mineral resource boundary was properly represented by the percent model’s capacity to measure infinitely variable inclusion percentages. Within the mineralization halo whole blocks were assigned an alteration domain code and a Cu domain code.

Table 17.24 Block Model Setup

	Origin	Blocks	Size
X	423,600	90	25 m
Y	6,263,500	90	25 m
Z	1,850	135	10 m
Rotation	None		

17.3.12 *ESTIMATION AND CLASSIFICATION*

Ordinary Kriging (OK) of capped composite values was used for the estimation of block grades. Block discretization was set at 5 m x 5 m x 2 m to reflect the selected block size.

A three-pass series of expanding search spheres with varying minimum sample requirements were used for sample selection and estimation, with the diameter of the search sphere derived from the Au Zone-1 semi-variogram. Composite data used during estimation were restricted to samples located in their respective zones. Individual block grades were then used to calculate an Au-Eq block model.

During the first pass, 7 to 12 composites from 3 or more drillholes within a search sphere 70 m in diameter were required for estimation. All blocks estimated during the first pass were classified as Measured, creating a series of semi-continuous zones. The continuity over a broader area given the current drillhole spacing is not sufficient to allow a more coherent zone of Measured Resources to be delineated, and additional drilling will be required to define one.

During the second pass, 7 to 12 composites from 3 or more drillholes within a search sphere 140 m in diameter were required for estimation. All blocks estimated during the second pass were classified as Indicated.

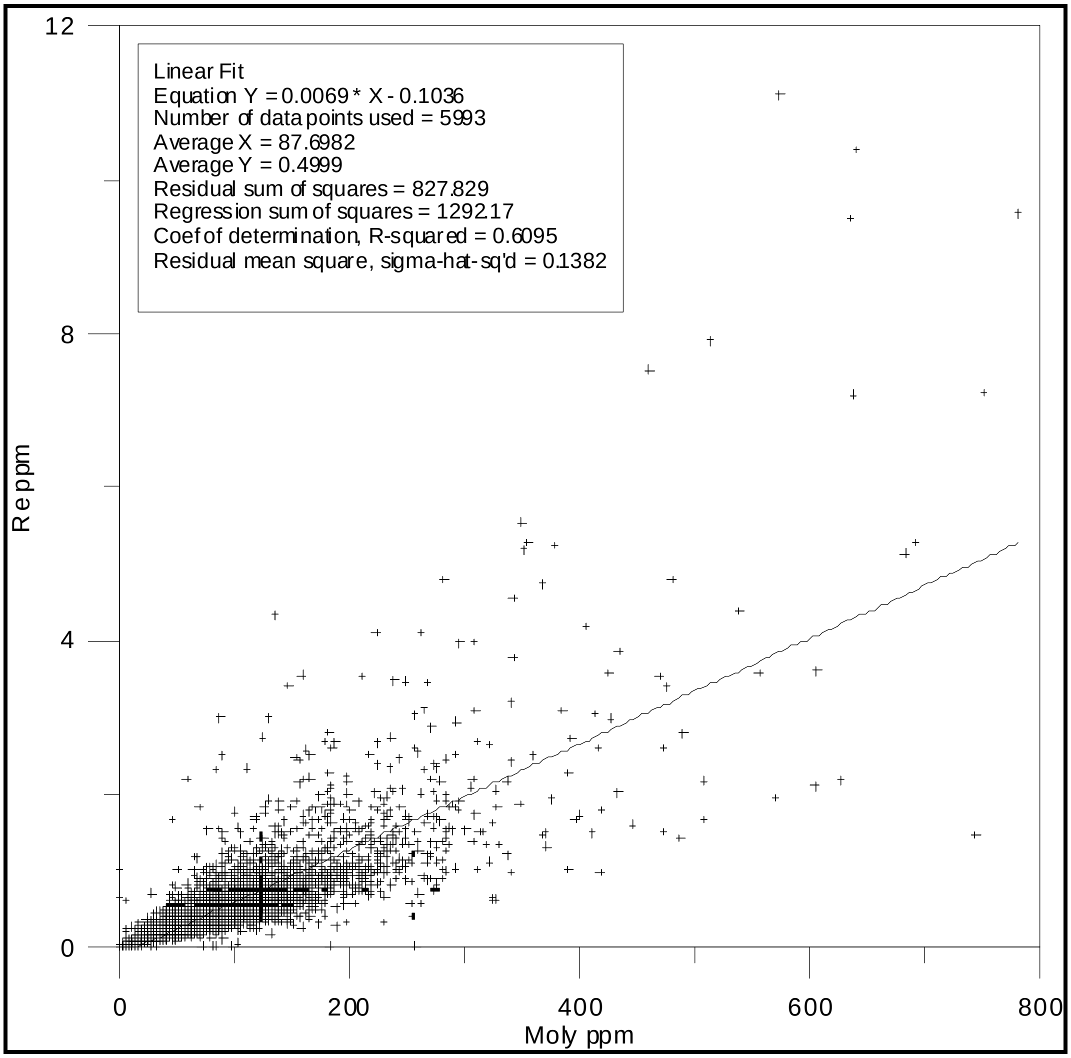
During the third pass, 3 to 12 composites from 1 or more drillholes within a search sphere 280 m in diameter were required for estimation. All blocks estimated during the third pass were classified as Inferred.

17.3.13 *RHENIUM MODEL*

Subsequent to the original sampling program, Silver Standard assayed a sub-set of the stored pulps for rhenium (Re). P&E did not monitor or observe the re-sampling, and all data were provided by Silver Standard.

The supplied Re database contains 6,637 records, including blank, duplicate, standard and assay sample results. Of the total database, 5,993 Re assays are co-located with Mo and display a high degree of correlation with Mo (Figure 17.1).

Figure 17.1 Snowfield Co-located Re and Mo Assay Data



In order to include rhenium in the mineral resource model, co-kriging of Re 1.5 m composite data was used based on the observed correlation between Mo and Re. Summary statistics for the Re assay and composite data indicate that the compositing process did not introduce a significant bias (Table 17.25). Experimental semi-variograms were derived for the total Mo composite data set, the co-located composite data sets for Mo and Re, and a cross-variogram for Mo + Re (Table 17.26). Before estimation the experimental semi-variograms were modified to ensure that the co-variance matrix was positive definite.

Table 17.25 Re Assay and Composite Summary Statistics

	Re Assays	Re Composites
Number	5993	5935
Minimum	0.005	0.005
Maximum	11.100	9.670
Mean	0.4999	0.4939
St Dev	0.5948	0.5484
CV	0.8406	0.9005

Table 17.26 Co-located Semi-variogram Models

Element	Experimental Semi-variogram
Mo	0.1 + SPH (0.4, 20) + SPH (0.1, 100) + SPH (0.4, 500)
Re	0.2 + SPH (0.3, 30) + SPH (0.1, 200) + SPH (0.4, 600)
Re + Mo	0.2 + SPH (0.3, 20) + SPH (0.1, 250) + SPH (0.4, 560)

Re block grades based on the total uncapped Mo data set and the co-located uncapped Re data set were estimated using the Stanford University Geostatistical Software Library (GSLIB) algorithms. As a check of the validity of the model, Re block grades were compared to blocks estimated using only the co-located data sets, as well as with a NN model. No significant discrepancies were noted between the Re model results.

17.3.14 Snowfield Mineral Resource Estimate

In order to ensure that the reported mineral resources meet the CIM requirement for “reasonable prospects for economic extraction” a conceptual Lerchs-Grossman optimized pit shell was developed based on all available mineral resources (Measured, Indicated, and Inferred), using the economic parameters listed in Table 17.27.

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 17.27 Optimized Pit Shell Parameters

Area	Parameter
Tailings & Water	US$0.80/t
Mining Cost	US$1.75/t
Processing Cost	US$5.00/t
G&A	US$1.00/t
Pit Wall Slope Angle	45°
Au Price	US$980.00/oz
Ag Price	US$14.89/oz
Cu Price	US$2.65/lb
Mo Price	US$17.00/lb
Re Price	US$145.00/oz
Au Recovery	71%
Ag Recovery	70%
Cu Recovery	70%
Mo Recovery	60%
Re Recovery	60%

A 0.30 g/t Au-Eq cut-off for the reporting of mineral resources at Snowfield was calculated, based on the economic parameters listed in Table 17.27. All mineral resources were constrained within the optimized pit shell (Table 17.28).

Table 17.28 Mineral Resource Estimate – 0.30 g/t Au-Eq Cut-off [1,2,3]

Class	Mt	Au (g/t)	Au (M oz)	Ag (g/t)	Ag (M oz)	Cu (%)	Mo (ppm)	Re (g/t)
Measured	143.7	0.83	3.85	1.57	7.27	0.08	100	0.62
Indicated	951.6	0.60	18.19	1.78	54.38	0.11	87	0.47
Measured + Indicated	1095.3	0.63	22.04	1.75	61.65	0.11	89	0.49
Inferred	847.2	0.40	10.99	1.53	41.62	0.07	82	0.33

[1] Mineral resources 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.

17.3.15 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. 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 OK block model estimates to a NN block model estimate generated using the same search criteria and tabulated at a zero cut-off within the constraining pit-shell. Results demonstrated a minimal global bias and slight smoothing for the OK estimate as compared to the NN estimate, and correctly duplicate grade trends.

18.0 OTHER RELEVANT DATA AND INFORMATION

18.1 MINING

The Snowfield-Brucejack Project 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 Brucejack deposit area is situated 5 km to the south of the Snowfield deposit.

The deposits will be mined using bulk open pit mining methods to provide a mill feed at a nominal rate of 120,000 t/d (43.8 Mt/a). Primary crushers capable of sustaining the processing rate of 120,000 t/d will be established at both Snowfield and Brucejack. The product from these will feed conveyor systems carrying the mineralized material through a tunnel to the processing plant, located approximately 26 km to the east of the Snowfield and Brucejack deposits. The mineralized material will be deposited on a crushed mineralized material stockpile at the processing plant to allow for a limited decoupling of the mine and plant. Personnel and materials will be transported from the mill to the Brucejack and Snowfield areas through the tunnel system. A road linking the two project areas will be established for transferring heavy equipment between the operations.

Snowfield is a polymetallic deposit containing economically extractable amounts of copper, gold, silver, and molybdenum; the Brucejack deposit contains gold and silver.

At the mine's operating peak, 120.0 Mt/a of material will be mined, with waste-to-mineralized material ratio of approximately 0.57:1 for Snowfield and 2.95:1 for Brucejack, with an average of 1:1 for the combination of both.

The operation will focus on application of the largest available mining equipment to reduce unit cost. The primary equipment fleet will consist of four 311 mm blasthole drills, three 45 m^3 electric cable shovels, five diesel hydraulic shovels, and thirty-one 363 t haul trucks. Initially two of the electric cable shovels and three of the diesel hydraulic shovels will be commissioned at Brucejack with the remaining excavators being used for Snowfield. As the Brucejack Project is completed, those excavators will be decommissioned, transported and recommissioned at Snowfield to account for the higher strip ratios encountered in that pit later in the mine life. The trucks and drills will be transported between the operations as required.

The primary equipment will be supported by track- and rubber-tired bulldozers, motor graders, a compactor, a water truck, a small excavator, and other ancillary equipment. It is assumed that all this equipment is shared between the operations and can be transported from one to the other, as required. Tractor/lowboy units have been included in the ancillary fleet to transport tracked equipment between the operations.

At Snowfield, 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. At Brucejack, the mining configuration is dependent upon the size of each final pit. The smaller pits are mined in 10 m benches with 20 m between catch benches. Pits deeper than 200 m are mined as per Snowfield.

The overall mining sequence was developed through a series of six scoping-level mining pushbacks at Snowfield and six individual pits at Brucejack. The major aim of the sequence was designed to bring forward high grade material from both deposits while deferring waste stripping as late as possible to improve project NPV.

18.1.1 *Introduction*

The mine planning work for the scoping study of the Snowfield property was based on a resource model provided by Fred Brown of P&E on June 6, 2010 (Section 17.0).

Two resource models were used for Brucejack: one incorporated the West Zone, and the other incorporated all the remaining mineralization in the Brucejack deposit. The West Zone and Brucejack models were published in an NI 43-101-compliant Technical Report released on January 14, 2010.

Mine planning was conducted through the application of Gemcom Software International Inc. (Gemcom) Whittle™, Gemcom Surpac™, and MineMax Scheduler 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 supplied by BGC, and estimated project metallurgical recoveries supplied by Wardrop.

18.1.2 *3D Block Models*

Details of the three block models used for this study are shown in Table 18.1 to Table 18.3.

Table 18.1 Details of the Snowfield 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

Table 18.2 Details of the Brucejack Block Model

Type	Y	X	Z
Minimum Coordinates	6,256,500	425,800	1,000
Maximum Coordinates	6,260,000	427,800	2,000
User Block Size	25	25	10
Minimum Block Size	25	25	10
Rotation	0	0	0

Table 18.3 Details of the West Zone Block Model

Type	Y	X	Z
Minimum Coordinates	6,257,600	426,800	700
Maximum Coordinates	6,259,100	428,300	1,600
User Block Size	10	10	10
Minimum Block Size	10	10	10
Rotation	-40° (around Z)		

The Snowfield block model contains rock type, density, grades for Au, Ag, Cu, Mo, class AuEq, and the percent of the block that is within the mineralized zone. Subsequent to the optimization and scheduling processes, a new Snowfield block model was received that included rhenium (Re) grades. Values for rhenium have not been used for optimization or scheduling purposes, but grades have been reported based on the previously derived pit shells and schedules.

The West Zone and Brucejack models contained similar fields with the exception of grade fields for Cu and Mo, as only Au and Ag mineralization has been modelled for these deposits.

The West Zone block model is a rotated model that overlays a portion of the other Brucejack zones block model. There are no overlapping areas of mineralization.

18.1.3 WHITTLE PARAMETERS

AMC used the LG algorithm application in Gemcom's Whittle™ program 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.

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 18.4.

Table 18.4 Metal Prices

Commodity	Metal Price (US$)
Copper	2.25/lb
Gold	850/oz
Silver	12.50/oz
Molybdenum	12.50/lb
Exchange Rate	0.92

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

PROCESS RECOVERIES

Metal recoveries for each project area were estimated according to available metallurgical test results, and were provided by the Wardrop project metallurgical engineer, as presented in Table 16.16.

SMELTER TERMS AND DEDUCTIONS

Smelter terms and deductions used as pit optimization input parameters are shown in Table 18.5.

Table 18.5 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		
Gold	$/acc oz	8.000		
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 and Rehandle)	%NIV*			0.50
Insurance	%NIV			0.15
Representation	$/wmt	0.50		
Gold and Silver Doré				
Combined Smelting and Transport Costs	$/oz	2.00		
Insurance	%NIV			0.15
Representation (Based on Copper Concentrate Ratio)	%NIV			0.02

* NIV = net invoice value.

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 mineralized material 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.

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.

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	Unit	Snowfield	Brucejack
Mining (Mineralized Material or Waste)	US$/t Mined	1.58	1.58
Stockpile Rehandling (Mineralized Material)	US$/t Mined	0.50	0.50
Process, G&A, and Others	US$/t Milled	6.70	6.55

PIT SLOPE ANGLES

Overall slope angles for the two project areas were provided by BGC.

Snowfield was split into sectors by bearing from the approximate centre of the pit, as shown in Table 18.7.

Table 18.7 Snowfield Preliminary Open Pit Slope Angles

Design Sector	Slope Azimuth (°)		Assumed Overall Slope Height (m)	Bench Height (m)	Bench Face Angle (°)	Catch Bench Width (m)	Maximum Overall Slope Angle (°)
	Start	End					
North Wall	317	037	510	30	65	18.5	43
East Wall	037	102	800	30	65	23	39
South Wall	102	225	1080	30	65	20.5	36
West Wall	225	317	800	30	65	18	39

The Brucejack slope angles were broken down into slope regions based on the depth of the final pit, as shown in Table 18.8. This includes the West Zone.

Table 18.8 Brucejack Preliminary Open Pit Slope Angles

Open Pit Size	Assumed Overall Slope Height (m)	Bench Height (m)	Bench Face Angle (°)	Catch Bench Width (m)	Maximum Overall Slope Angle (°)
Small	≤200	20	65	12	45
Medium	200 to 400	30	65	19	42
Large	400 to 600	30	65	19	41

18.1.4 NET SMELTER RETURN

The NSR is calculated in US$/t using Net Smelter Prices (NSP). 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 18.9. Separate NSP calculations were completed for gold and silver in concentrate, and gold and silver in doré, due to the differences in off-site costs of the two delivery methods.

Table 18.9 Metal Prices and NSP

	Metal Price (US$)	NSP (US$)
Cu	2.25/lb	1.69/lb
Au in Concentrate	850/oz	815.41/oz
Au in Doré	850/oz	844.86/oz
Ag in Concentrate	12.50/oz	10.73/oz
Ag in Doré	12.50/oz	10.45/oz
Mo	12.50/lb	10.51/lb

The NSR formula is:

NSR = Au (g/t) * 0.032151 * AuConcRec * NSPAuConc +
Au (g/t) * 0.032151 * AuDoréRec * NSPAuDoré +
Ag (g/t) * 0.032151 * AgConcRec * NSPAgConc +
Ag (g/t) * 0.032151 * AgDoréRec * NSPAgDoré +
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)
- AuConcRec = gold recovery in concentrate (<1)
- AuDoréRec = gold recovery in doré (<1)
- AgConcRec = silver recovery in concentrate (<1)
- AgDoréRec = silver recovery in doré (<1)
- MoRec = molybdenum recovery (<1)
- NSPCu = NSP for copper ($/lb)
- NSPAuConc = NSP for gold in concentrate ($/oz)
- NSPAuDoré = NSP for gold doré ($/oz)
- NSPMo = NSP for molybdenum ($/lb)
- NSPAgConc = NSP for silver in concentrate ($/oz)
- NSPAgDoré = NSP for silver in doré ($/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, the 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 optimizations were conducted using the Lerchs-Grossman (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. Blocks with a zero density are coded as air blocks.

LG optimization is a process used to identify the optimal limit of an open pit. The process considers the potential revenue generated from a block of material, the cost of mining the block, and the cost of mining the blocks above for access. The blocks that must be mined to access a mineralized material block are selected based on an overall slope angle that estimates the final slope including design bench face angle, catch benches, and ramps. If the result of the net revenue minus the cost is positive, the increment, including the mineralized material block and those which must be mined to access it, are added to the shell. The process considers deeper and deeper material until the increment does not add value. This is considered the optimal pit under the financial scenario being tested. The process is run iteratively with increasing commodity prices to generate a suite of shells of increasing size which can be evaluated under a range of financial scenarios. The analysis provides an understanding of the potential return from a shell and the financial risks associated with selecting a particular shell as the basis of design work if the inputs are different to those forecast.

The suite of shells is used as a guide for selecting pit stages as well as the final pit. It should be noted that the LG algorithm does not apply a factor for the time value of money and therefore a schedule needs to be run to assess the effect of time costs and discount rates.

The three curves shown in Figure 18.1 to Figure 18.3 are defined in the Whittle™ manual as follows:

- Black Curve: is the undiscounted open pit value for the 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 identifying the “Optimal Pit” as identified by the LG algorithm.
- Blue Curve: is the discounted open pit value for the 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: is the discounted open pit value for the 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.

For the West Zone (Figure 18.3) the blue and red curves are equivalent due to the contained mineralized material tonnes being less than the proposed processing rate of 43.8 Mt/a, and therefore the discount rate has no effect. In practice, this would not be the case as the West Zone will be mined with the other deposits over a number of years.

One of the major constraints on the Snowfield property is the available area to deposit waste rock. This constraint is based around the topography, the geometry of the pit, and the proximity of the Mitchell Glacier. The waste constraint has limited the size of the pit chosen in this study to Pit 28 as the final pit limit for Snowfield Project.

A comparison of the chosen final pit and the optimal pit as identified by the LG algorithm is shown in Table 18.10. While the Optimal Pit 36 is unlikely to be the most economic once scheduling and discount rates are applied, it does provide evidence that there could be an increase in the size of the pit and mineralized material produced over the life of the mine if alternative waste storage could be identified.

For Brucejack, the Optimal Pit Shell 36 was chosen as the final pit. The selection of final shell was not limited by any constraints. The selection parameters of the final shell should be reviewed in the next level of study.

Figure 18.1 Snowfield Model Pit Value Graph

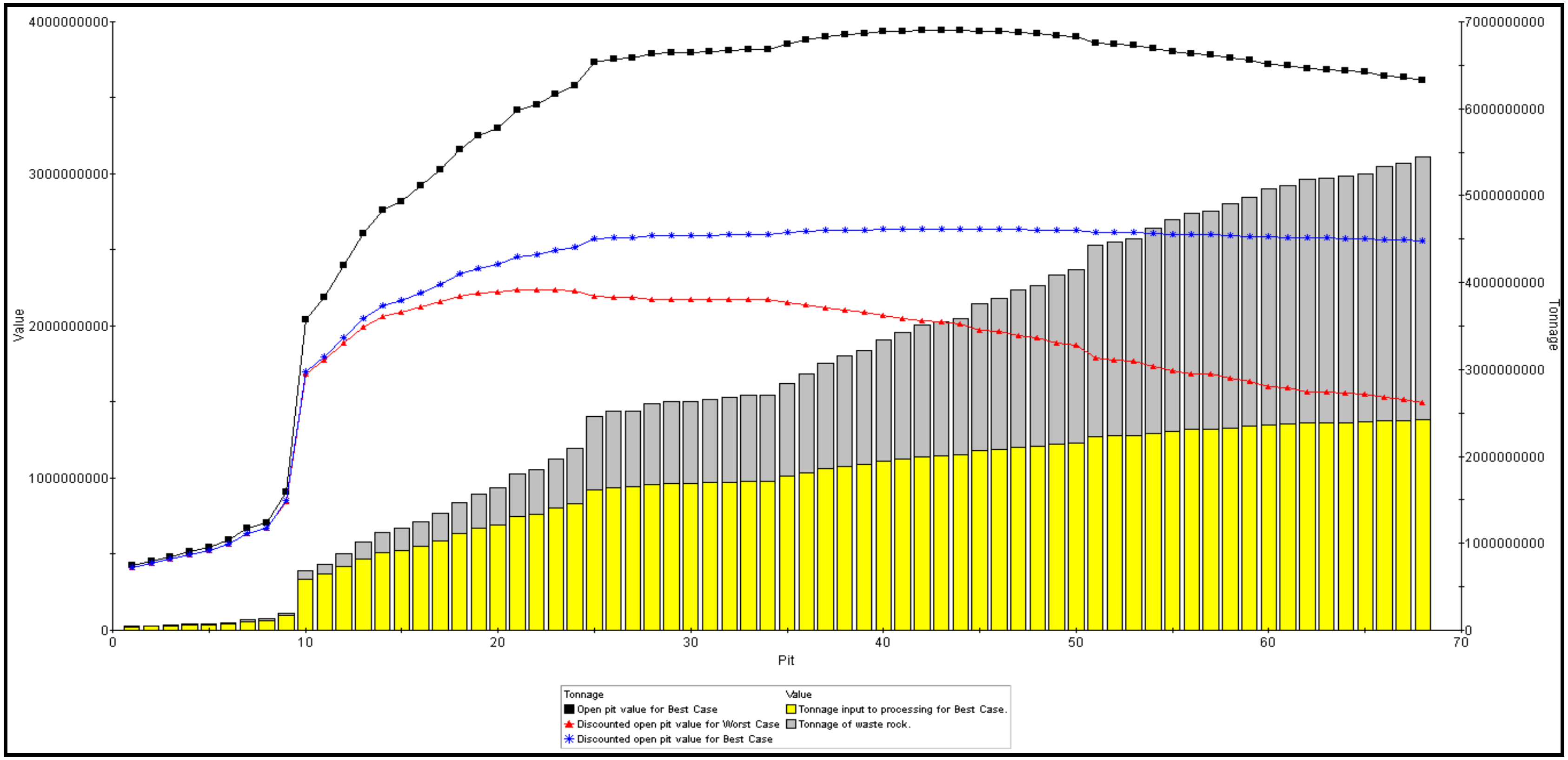


Figure 18.2 Brucejack Model Pit Value Graph

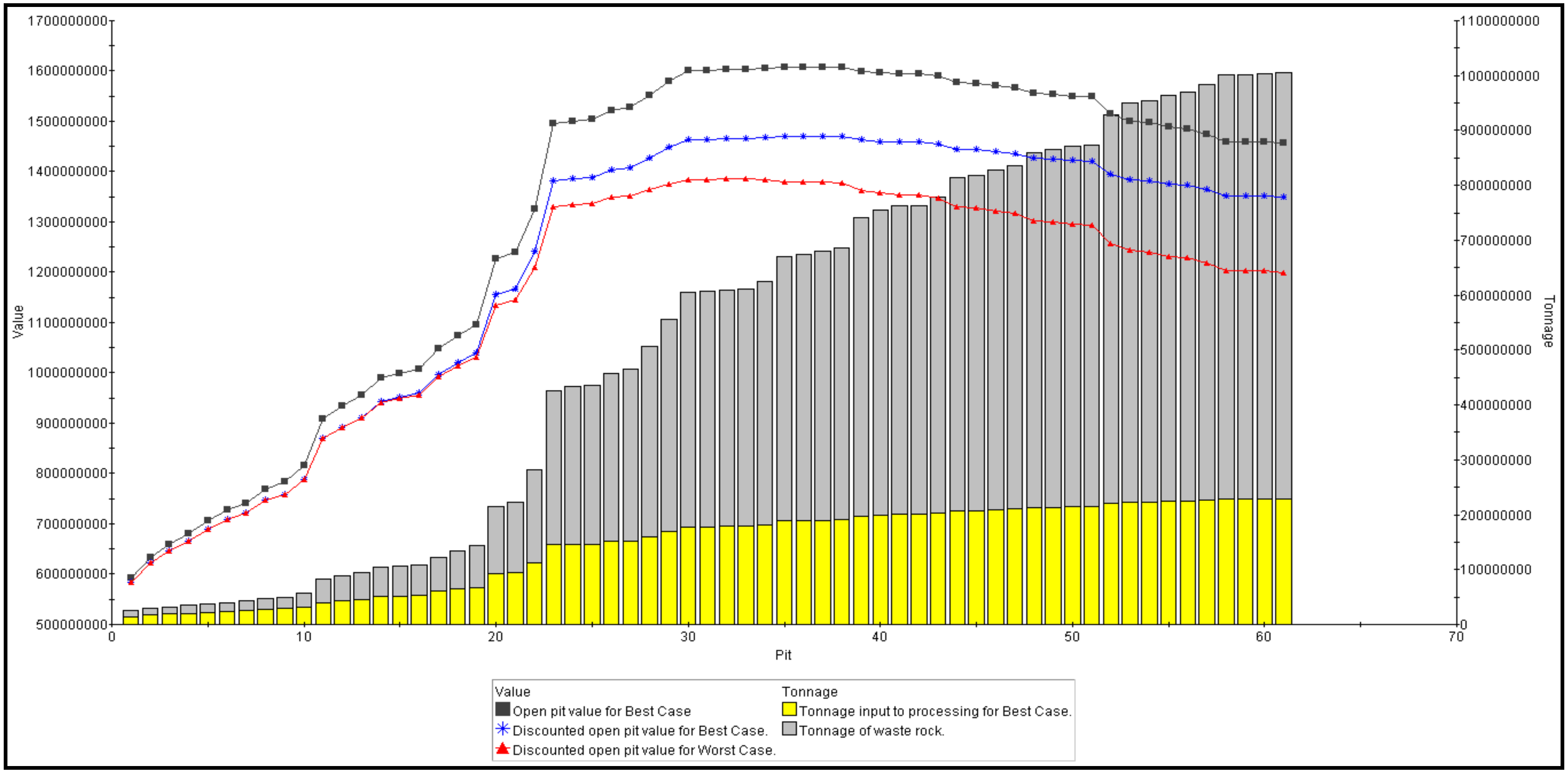


Figure 18.3 West Zone Model Pit Value Graph

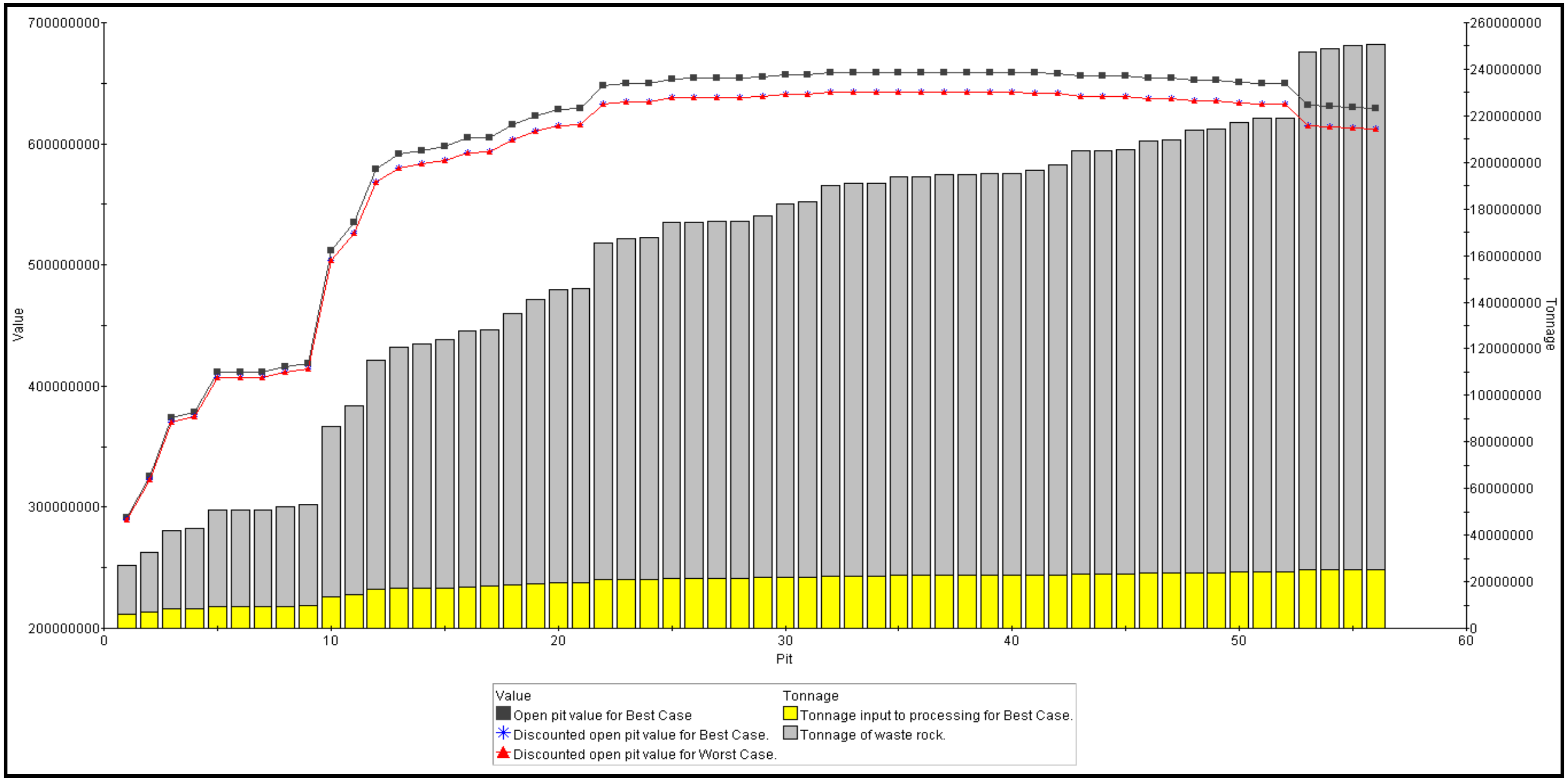


Table 18.10 Chosen Snowfield-Brucejack Final Pit vs. Optimal Pit

Deposit	Pit #	Rev. Factor	Rock (t)	Mineralized material (t)	Waste (t)	Strip Ratio	Grade				
							NSR (US$)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (%)
Snowfield Model	28	0.805	1,504,504,752	959,900,277	544,604,475	0.57	16.07	1.66	0.68	0.10	0.01
Snowfield Model	36	1	2,074,195,772	1,182,463,869	891,731,903	0.75	15.50	1.69	0.65	0.10	0.01

18.1.7 *Mine Production Schedule*

Scheduling Concepts

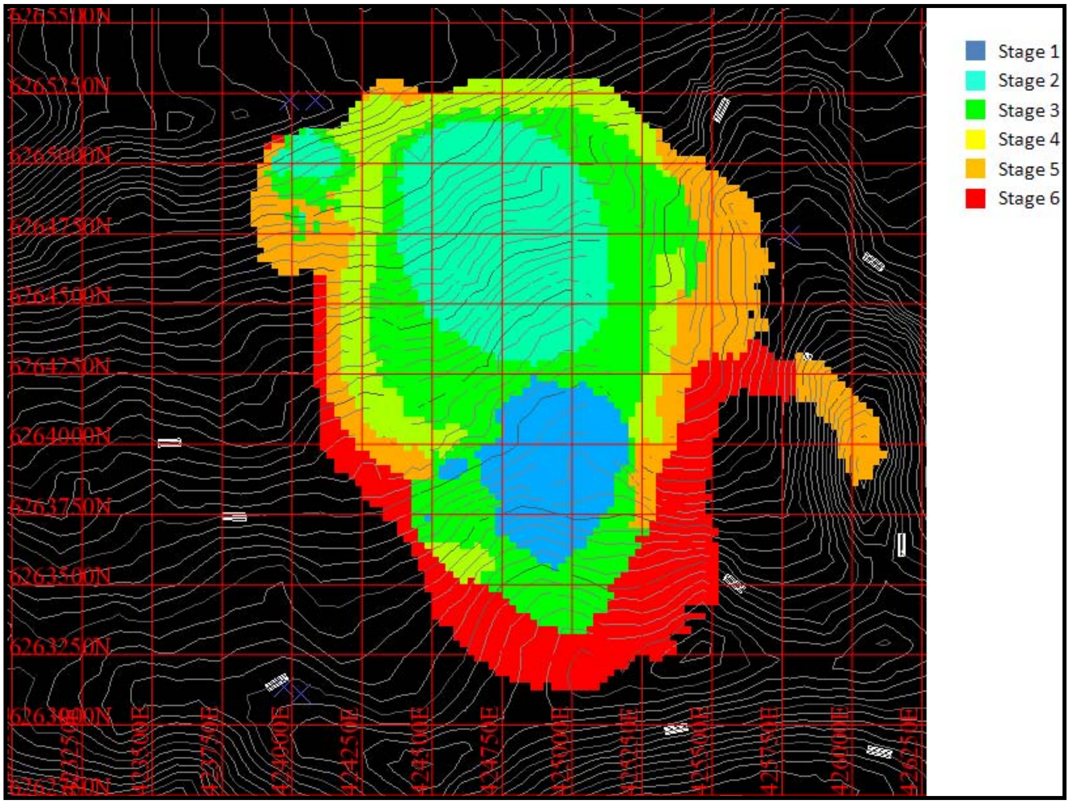
The pit optimization program takes no account of bench operating width in the generation of pits. Consequently, an optimized final pit which is used as a starting point for the design of a final pit may have a floor that is too narrow, and/or show irregularities in the pit wall that cannot be easily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

Similarly, the optimized internal pit shells used in the design framework for pit phases 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 pit phase or the final pit.

To produce pit phases that are operationally feasible, a mining width module of Whittle™ was applied in the optimization process. Pit phases were specified with a mining width of 50 m to accommodate large mining equipment that will operate on a bench. This is the equivalent of two block widths in the Snowfield and Brucejack models, or five for the West Zone.

At Snowfield, five pit shells that are conceptually equivalent to five pit phases were selected for the development of the mine production schedule. The first pit shell consisted of two discreet mining areas in the central south and central north areas of the final pit and, as such, the two were separated for scheduling purposes, resulting in six identified pit phases, as shown in Figure 18.4. Due to the relatively short life of Phases 1 and 2, it was agreed with BGC that an overall wall angle of 45° could be used to improve the stripping ratio and economics of the early years of mining at Snowfield. Subsequent phases have had the standard pit slope angles applied as per Table 18.7.

Figure 18.4 Snowfield Mine Phases



The Brucejack project area consists of six mineralized zones that are mined as individual pits. Some of the zones overlap, such as Gossan Hill, West Zone, and Galena Hill, which has been accounted for within the scheduling sequence. Four of the zones (SG, Shore, Gossan Hill, and Galena Hill) have been assumed to be mined in a single phase, while the largest zones (Bridge Zone and West Zone) are mined in three phases each. The final pits are shown in Figure 18.5, while the detailed phasing of the Bridge Zone and the West Zone are shown in Figure 18.6 and Figure 18.7, respectively.

Figure 18.5 Brucejack Mining Areas

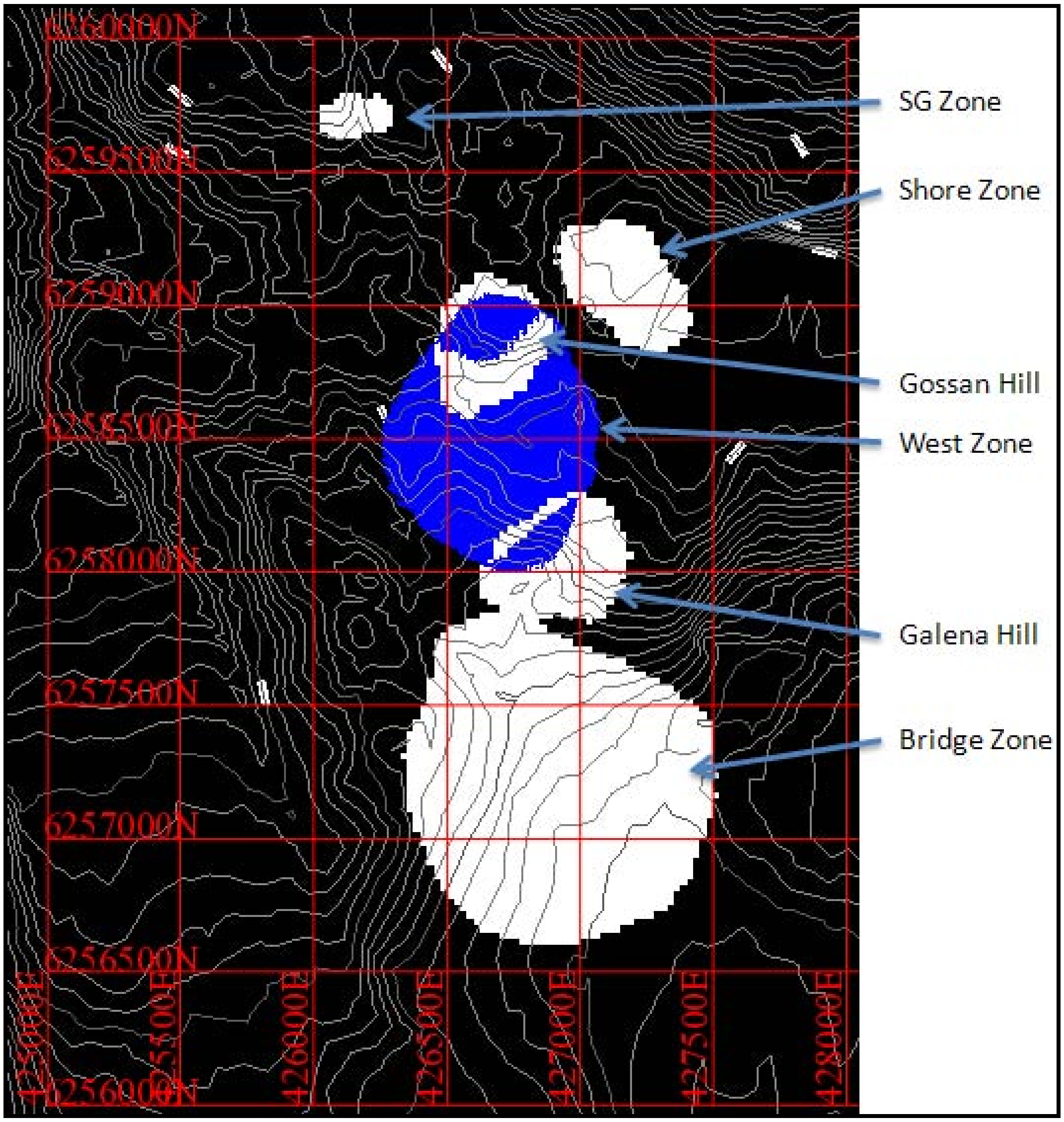


Figure 18.6 Bridge Zone Mine Phases

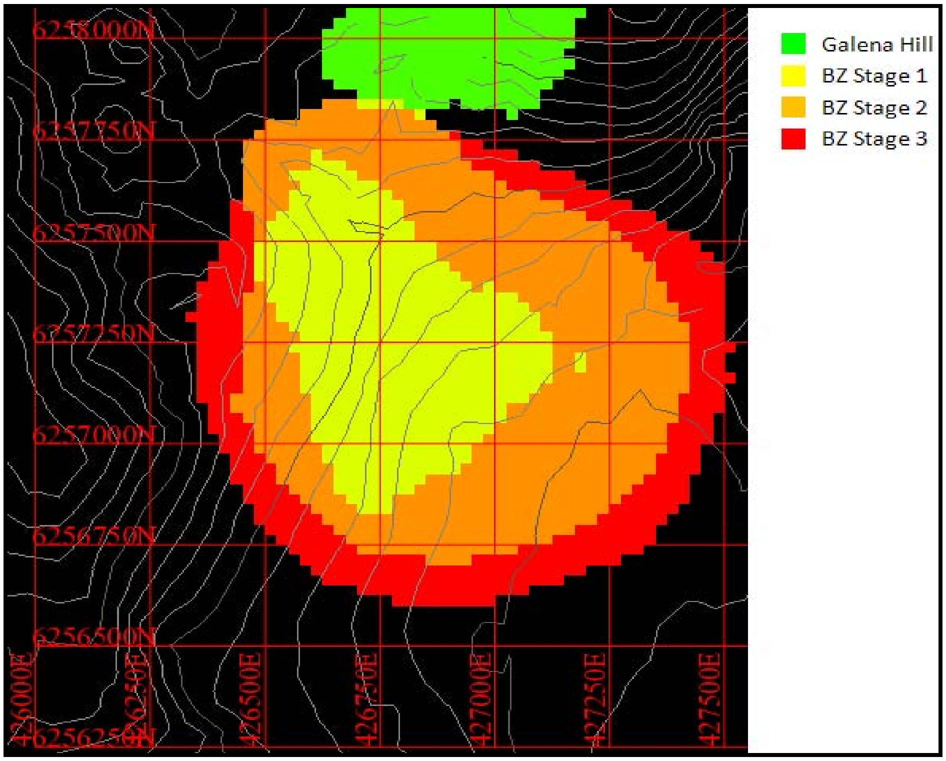
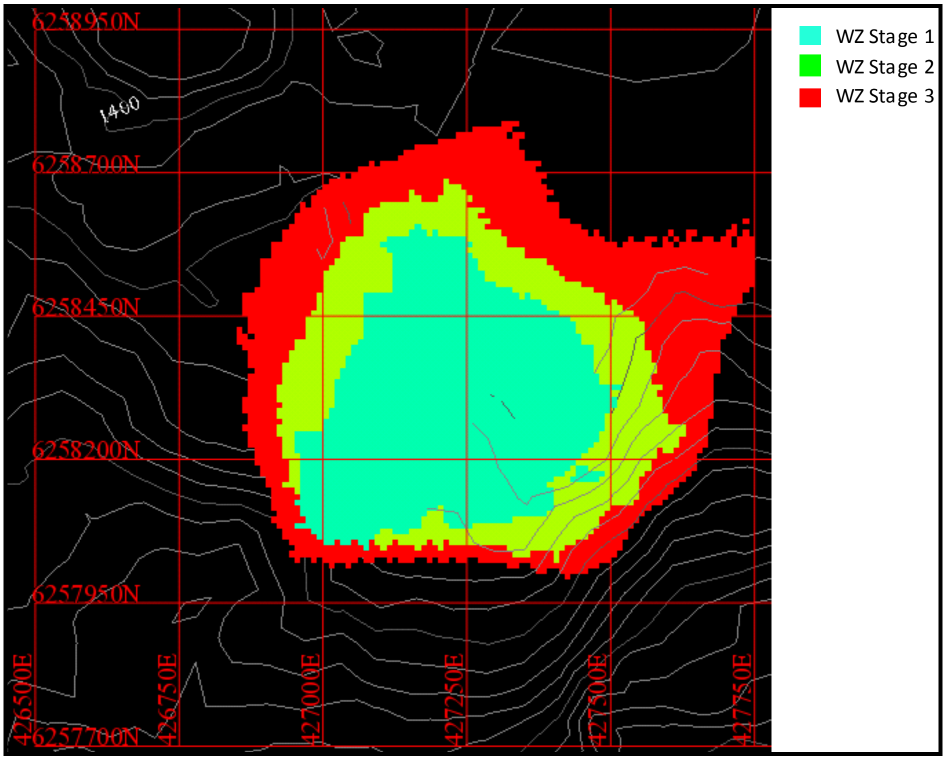


Figure 18.7 West Zone Mine Phases



The truncated appearance on the north-eastern and southern edges of the West Zone pit (Figure 18.7) is caused by the interaction of the West Zone, Gossan Hill, and Galena Hill pits. Gossan Hill and Galena Hill will be mined prior to the West Zone and, as such, this material has been removed from the West Zone pit calculations.

Scheduling for the combined Snowfield and Brucejack areas has been completed using MineMax Scheduler software. Scheduler is a schedule optimization tool that produces schedules based on a framework of user-applied constraints, which can be optimized for a number of given targets. For Snowfield and Brucejack, the processing limit was applied along with a set of practical mining constraints to produce a schedule that optimizes the project NPV.

One of the major constraints placed on the mining schedule is the need for separate batch processing of Snowfield and Brucejack mineralized materials. Therefore, the first two years of processing were scheduled by month, then by year from Year 3 onward.

PRODUCTION SCHEDULE

A production schedule, based on 120,000 t/d mill feed schedule, has been developed and is shown in Figure 18.8 and Table 18.11.

Figure 18.8 Production Schedule Graph

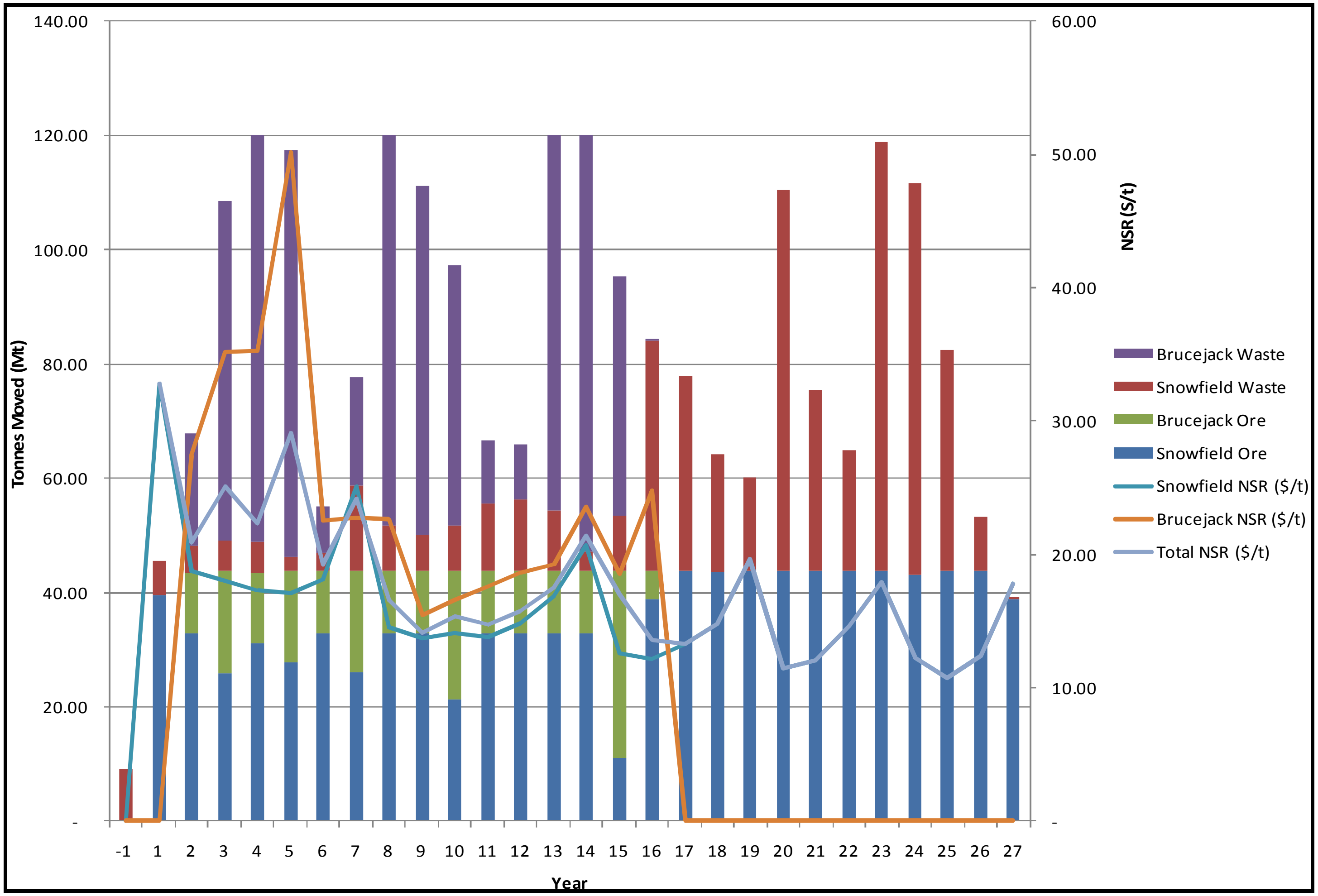


Table 18.11 Production Schedule (Page 1 of 2)

Mining Area	Year	TOTAL	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Snowfield	Total Movement (000t)	1,504,505	9,000	45,608	37,668	31,114	36,645	30,179	36,112	41,042	40,843	39,086	29,140	44,693	45,218	43,490
	Waste (000t)	544,604	9,000	6,059	4,818	5,309	5,448	2,356	3,262	15,022	7,993	6,236	7,887	11,843	12,477	10,640
	Strip Ratio	0.57	-	0.15	0.15	0.21	0.17	0.08	0.10	0.58	0.24	0.19	0.37	0.36	0.38	0.32
	Ore (000t)	959,900	-	39,549	32,850	25,805	31,197	27,823	32,850	26,020	32,850	32,850	21,253	32,850	32,741	32,850
	Au Grade (g/t)	0.68	-	1.56	0.78	0.70	0.70	0.68	0.70	0.95	0.75	0.62	0.59	0.58	0.61	0.66
	Ag Grade (g/t)	1.66	-	1.45	1.96	2.06	1.91	2.00	1.90	1.94	1.47	1.71	1.76	1.61	1.52	1.71
	Cu Grade (%)	0.10	-	0.03	0.10	0.13	0.12	0.12	0.13	0.16	0.05	0.07	0.09	0.09	0.10	0.12
	Mo Grade (%)	0.01	-	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
	Re Grade (g/t)~	0.51	-	0.87	0.62	0.59	0.52	0.53	0.51	0.28	0.76	0.79	0.50	0.55	0.59	0.62
	NSR($/t)	16.07	-	32.83	18.71	18.02	17.28	17.09	18.11	25.12	14.56	13.67	14.12	13.79	14.82	16.91
Brucejack	Total Movement (000t)	836,319	-	-	30,066	77,496	83,355	87,254	18,996	36,637	79,157	72,180	68,258	21,970	20,659	76,510
	Waste (000t)	624,610	-	-	19,489	59,501	71,163	71,277	8,046	18,857	68,207	61,230	45,711	11,020	9,599	65,560
	Strip Ratio	2.95	-	-	1.84	3.31	5.84	4.46	0.73	1.06	6.23	5.59	2.03	1.01	0.87	5.99
	Ore (000t)	211,709	-	-	10,576	17,995	12,192	15,977	10,950	17,780	10,950	10,950	22,547	10,950	11,059	10,950
	Au Grade (g/t)	1.02	-	-	1.12	1.41	1.16	1.60	0.95	0.99	1.01	0.72	0.76	0.82	0.86	0.90
	Ag Grade (g/t)	15.55	-	-	13.79	14.32	34.20	46.19	15.23	14.38	11.57	10.39	10.74	10.24	10.67	9.34
	NSR($/t)	24.55	-	-	27.47	35.19	35.24	50.12	22.57	22.79	22.69	15.50	16.54	17.61	18.61	19.25
TOTAL	Total Movement (000t)	2,340,824	9,000	45,608	67,733	108,610	120,000	117,434	55,108	77,679	120,000	111,265	97,398	66,663	65,876	120,000
	Waste (000t)	1,169,214	9,000	6,059	24,307	64,810	76,611	73,634	11,308	33,879	76,200	67,465	53,598	22,863	22,076	76,200
	Strip Ratio	1.00	-	0.15	0.56	1.48	1.77	1.68	0.26	0.77	1.74	1.54	1.22	0.52	0.50	1.74
	Ore (000t)	1,171,610	-	39,549	43,426	43,800	43,389	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800	43,800
	Au Grade (g/t)	0.74	-	1.56	0.86	0.99	0.83	1.02	0.77	0.97	0.81	0.65	0.68	0.64	0.67	0.72
	Ag Grade (g/t)	4.17	-	1.45	4.84	7.10	10.98	18.12	5.23	6.99	4.00	3.88	6.38	3.77	3.84	3.61
	Cu Grade (%)*	0.08	-	0.03	0.08	0.08	0.08	0.07	0.10	0.10	0.03	0.06	0.05	0.07	0.08	0.09
	Mo Grade (%)*	0.01	-	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
	Re Grade (g/t)*~	0.42	-	0.87	0.47	0.35	0.37	0.34	0.38	0.17	0.57	0.60	0.24	0.41	0.44	0.47
	NSR($/t)	17.61	-	32.83	20.85	25.07	22.33	29.14	19.23	24.17	16.59	14.13	15.37	14.75	15.77	17.49

Notes:

*Average grade for copper, molybdenum, and rhenium not representative of process head grades due to batch processing assumption.

~ Rhenium grade not used for optimization or scheduling.

Table 18.11 (con't) Production Schedule (Page 2 of 2)

Mining Area	Year	TOTAL	14	15	16	17	18	19	20	21	22	23	24	25	26	27
Snowfield	Total Movement (000t)	1,504,505	36,210	20,659	79,064	77,871	64,123	60,125	110,420	75,513	64,924	118,939	111,748	82,427	53,238	39,407
	Waste (000t)	544,604	3,360	9,709	40,297	34,071	20,492	16,325	66,620	31,713	21,124	75,139	68,731	38,627	9,438	611
	Strip Ratio	0.57	0.10	0.89	1.04	0.78	0.47	0.37	1.52	0.72	0.48	1.72	1.60	0.88	0.22	0.02
	Ore (000t)	959,900	32,850	10,950	38,767	43,800	43,631	43,800	43,800	43,800	43,800	43,800	43,017	43,800	43,800	38,797
	Au Grade (g/t)	0.68	0.80	0.69	0.55	0.55	0.62	0.80	0.53	0.50	0.57	0.73	0.60	0.48	0.50	0.66
	Ag Grade (g/t)	1.66	1.81	1.06	1.46	1.49	1.62	1.81	1.40	1.53	1.53	1.84	1.38	1.54	1.60	2.04
	Cu Grade (%)	0.10	0.14	0.04	0.08	0.10	0.11	0.13	0.07	0.10	0.12	0.13	0.06	0.08	0.11	0.15
	Mo Grade (%)	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
	Re Grade (g/t)~	0.51	0.42	1.09	0.53	0.50	0.48	0.38	0.52	0.44	0.39	0.26	0.58	0.34	0.34	0.28
	NSR($/t)	16.07	20.66	12.62	12.19	13.33	14.81	19.70	11.47	12.04	14.59	17.96	12.29	10.69	12.41	17.76
Brucejack	Total Movement (000t)	836,319	83,790	74,614	5,377	-	-	-	-	-	-	-	-	-	-	-
	Waste (000t)	624,610	72,840	41,764	344	-	-	-	-	-	-	-	-	-	-	-
	Strip Ratio	2.95	6.65	1.27	0.07	-	-	-	-	-	-	-	-	-	-	-
	Ore (000t)	211,709	10,950	32,850	5,033	-	-	-	-	-	-	-	-	-	-	-
	Au Grade (g/t)	1.02	1.08	0.88	1.04	-	-	-	-	-	-	-	-	-	-	-
	Ag Grade (g/t)	15.55	10.40	9.04	17.07	-	-	-	-	-	-	-	-	-	-	-
	NSR($/t)	24.55	23.55	18.51	24.75	-	-	-	-	-	-	-	-	-	-	-
TOTAL	Total Movement (000t)	2,340,824	120,000	95,273	84,441	77,871	64,123	60,125	110,420	75,513	64,924	118,939	111,748	82,427	53,238	39,407
	Waste (000t)	1,169,214	76,200	51,473	40,641	34,071	20,492	16,325	66,620	31,713	21,124	75,139	68,731	38,627	9,438	611
	Strip Ratio	1.00	1.74	1.18	0.93	0.78	0.47	0.37	1.52	0.72	0.48	1.72	1.60	0.88	0.22	0.02
	Ore (000t)	1,171,610	43,800	43,800	43,800	43,800	43,631	43,800	43,800	43,800	43,800	43,800	43,017	43,800	43,800	38,797
	Au Grade (g/t)	0.74	0.87	0.83	0.60	0.55	0.62	0.80	0.53	0.50	0.57	0.73	0.60	0.48	0.50	0.66
	Ag Grade (g/t)	4.17	3.96	7.04	3.25	1.49	1.62	1.81	1.40	1.53	1.53	1.84	1.38	1.54	1.60	2.04
	Cu Grade (%)*	0.08	0.11	0.01	0.07	0.10	0.11	0.13	0.07	0.10	0.12	0.13	0.06	0.08	0.11	0.15
	Mo Grade (%)*	0.01	0.01	0.00	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
	Re Grade (g/t)*~	0.42	0.31	0.27	0.47	0.50	0.48	0.38	0.52	0.44	0.39	0.26	0.58	0.34	0.34	0.28
	NSR($/t)	17.61	21.38	17.04	13.63	13.33	14.81	19.70	11.47	12.04	14.59	17.96	12.29	10.69	12.41	17.76

Notes:

* Average grade for copper, molybdenum, and rhenium not representative of process head grades due to batch processing assumption.

~ Rhenium grade not used for optimization or scheduling.

In general terms, mine scheduling tends to defer waste stripping in an attempt to improve NPV by deferring costs. This is achieved through pit staging, where each successive pit stage has a higher strip ratio during the life of the mine.

The combined schedule for Snowfield and Brucejack does not conform to this trend. High NSR grades at Brucejack outweigh the higher stripping ratio for the project creating a period of production peaking at 120 Mt/a from Years 3 to 5.

The total movement schedule shows cyclical movement rates coinciding with the commencement and completion of the different pit stages. Cycles of approximately two to three years are targeted at either peak or trough rates to allow for extended periods of stability for machine and workforce numbers. An exception to this is Year 21, though this peak is likely to be smoothed out using detailed short- to medium-term planning once the mine is in operation.

Cycling production rates allows for the deferral of waste stripping as mentioned above, which results in an NPV improvement. One factor that could influence the success of this method of cycling production rates is the availability of suitable personnel when production rates increase, and the ability to reduce the workforce when production rates drop. This should be considered at the next level of study. Due to the limited space at Snowfield, no stockpiles have been used in determining the schedule.

With the greater amount of available space at Brucejack, some stockpiling has been allowed. Due to the higher strip ratio at Brucejack, achieving the required instantaneous production rate of 120,000 t/d would require an excessively large mining fleet. As such, production at Brucejack has been averaged over up to nine months per year, with mineralized material stockpiles created to supplement feed during batch processing of Brucejack mineralized material.

MINERALIZED MATERIAL DELIVERY TO PRIMARY CRUSHERS

It has been assumed that there will be a crusher installation capable of producing the required 120,000 t/d of mineralized material at both the Snowfield and Brucejack project areas.

At Snowfield, the crusher has been located to the northeast of the pit at approximately 1470 masl. This location was chosen in an attempt to minimize average mineralized material haulage distance, both vertical and horizontal, over the life of the Snowfield operation. Due to limitations on available space and the required instantaneous production rates, most of the mineralized material from Snowfield will be direct tipped by the haulage trucks into the crusher feed bin. A small surge stockpile will be required in case of crusher breakdowns to give an alternative dumping location for trucks hauling mineralized material. Operationally, the size of this stockpile should be kept to a minimum to avoid rehandling costs.

For Brucejack, the initial crusher location is to the east of the West Zone pit and central to the SG, Shore Zone, Gossan Hill, West Zone, and Galena Hill pits. This location was chosen to reduce mineralized material haulage cycles during the early years of Brucejack operation. Each of the pits above is assumed to be backfilled with waste as they are completed. Upon completion of mining and backfilling of the Galena Hill and West Zone pits (currently Year 8), the crusher will be relocated during a Snowfield batch processing period to a flat area created by backfilling in the Galena Hill pit. The crusher conveyor will be realigned to cross the West Zone pit, hence the requirement for that pit to also be backfilled. The new location will reduce the mineralized material haulage distance (and therefore the operating cost) for the Bridge Zone pit, which is the final pit to be mined in the Brucejack project area.

Crushed mineralized material stockpiles have been included at the processing plant with a total live capacity of 150,000 t, allowing approximately one day of decoupling between the mine and the plant. This alleviates some of the risk of the direct mine to crusher scenario identified for this project.

WASTE ROCK DELIVERY TO THE WASTE DUMPS

Topographical and pit layout issues require different approaches to dumping of waste at the Snowfield and Brucejack properties.

For Snowfield, available space is a major limiting factor for the size and layout of the waste dumps. As indicated in the production schedule (Table 18.11and Figure 18.8), approximately 544.6 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. Further investigation into PAG material handling should be undertaken in the next phase of study.

Part of the areas identified for the waste dumps at Snowfield are currently covered in permanent ice. This ice would have to be either moved or melted prior to establishing the waste dumps.

The majority of the waste materials will be placed in the East dump, which will contain approximately 470 Mt of the waste rock. The Southwest dump will contain approximately 68 Mt of the waste rock. There will also be an opportunity late in the mine life for backfilling the northernmost areas of the open pit once these areas are completed.

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

- 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 Brucejack project area does not have the same space limitations as Snowfield; therefore several areas and methods will be used for waste dumping. Initially, all waste material will be dumped into Brucejack Lake, which will be dewatered using a combination of pumping and displacement by waste material. Initial mining at Brucejack will be in a combination of the smaller pits (SG Zone, Shore Zone, Gossan Hill, and Galena Hill). As each pit is completed, it will be backfilled using waste material from the next mining area.

Once Brucejack Lake and all available backfill areas are completed, a waste dump will be constructed using the same design parameters that were used for Snowfield. The waste dump will cover the lake area and part of the West Zone pit; the final dump height will be 1500 masl, or 120 m above the approximate current lake level.

Like the Snowfield waste dump, part of the Bridge Zone pit at Brucejack is currently permanently covered in ice. The depth of this ice is currently unknown. For scheduling and cost purposes, the ice has been treated as waste rock and space for it has been allocated in the Brucejack waste dump. When the Bridge Zone pit is mined, the ice will need to be removed; currently, it is assumed that the ice will be mined using the standard mining equipment and placed on an ice dump to the south of the pit.

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

CONCEPTUAL MINE PLANNING

Pre-production stripping, as shown in Table 18.11 as Year -1, will involve removing approximately 9 Mt of waste material from Stages 1, 2, and 3 at the Snowfield pit. This will be achieved using one 39.0 m^3 hydraulic shovel and associated truck fleet. During pre-production, haul roads will be established that connect the truck shop west of the Snowfield pit with the planned pit exits, waste dumps, and the planned crusher location.

Full production will begin in Year 1; a short ramp-up period has been identified during the first quarter of Year 1, with full production maintained after that point. One of the major concepts for the mine schedule is the requirement to batch process the best available mineralized material from each of the project areas. The minimum period for each batch of the mineralized material was set as one quarter of a year, or 10.95 Mt.

Site preparation for the primary crusher at Snowfield, located northeast of the open pit, will be conducted early in the construction period, in conjunction with the construction of the conveyor tunnel. The truck dump hopper will be located at about 1470 masl, in the vicinity of coordinates 425,700 Easting, 626,500 Northing, as shown in Figure 18.9.

Focusing on achieving the highest NPV, the schedule has identified that the entire feed for the processing plant during Year 1 will be from Snowfield's Stages 1 and 2, both of which target zones of high value material. During this period, there will be no mining activity at Brucejack. The excavator fleet at Snowfield will be upgraded to one electric rope shovel and two diesel hydraulic shovels.

Production will continue from Snowfield only until the last quarter of Year 2, when the SG Zone, Shore Zone and Galena Hill pits will be mined at Brucejack. While the trucks and support machines are assumed to be transported between Snowfield and Brucejack as required, a separate shovel fleet has been allowed for at Brucejack. This will consist of two electric rope shovels and three diesel hydraulic shovels with the same capacities as those at Snowfield. During this period of production at Brucejack, activities at Snowfield will be limited to care and maintenance of roads and pit dewatering.

From Year 3 until the end of the mine life, a minimum of one quarter's production from each project area has been maintained. This assumption was made to limit the maximum idle time of the excavators at each area to less than one year to prevent degradation through disuse.

Major milestones in the mine schedule include the following:

- Year -1:
  - Pre-production stripping and establishment at Snowfield.
- Year 1:
  - Full production rate achieved at Snowfield.
- Year 2:
  - Continued mining at Snowfield Stages 1 and 2 in Q1 to Q3.
  - Dewatering Brucejack lake in Q1 to Q3.
  - Establishment and full production rate achieved at Brucejack in Q4.
  - Brucejack mining commences at SG Zone, Shore Zone, and Galena Hill pits.

- Year 3:
  - 59% of mineralized material produced from Snowfield Stage 2.
  - Remaining mineralized material produced from Brucejack.
  - Mining completed at Brucejack SG Zone and Shore Zone pits, backfilling commences.
  - Mining commences at Brucejack Gossan Hill Pit.
- Year 4:
  - 72% of mineralized material from Snowfield Stages 1 and 2.
  - Mining completed at Brucejack Gossan Hill and Galena Hill Pits, backfilling commences.
  - Mining commences at Brucejack West Zone Pit.
- Year 5:
  - 64% of mineralized material from Snowfield Stage 2.
- Year 6:
  - 75% of mineralized material from Snowfield Stage 1 and 2.
  - Mining completed at Brucejack West Zone pit, backfilling commences.
  - Mining commences at Brucejack Bridge Zone pit.
- Years 7 to 15:
  - Average of 65% of mineralized material from Snowfield.
  - Mineralized material production maximum of 75% Snowfield per year.
  - Mineralized material production minimum of 25% Snowfield per year.
- Year 16:
  - 89% of mineralized material from Snowfield.
  - Mining complete at Brucejack Bridge Zone pit.
  - All mining complete at Brucejack, rehabilitation commences.
  - Remaining mining equipment decommissioned, transported and re-commissioned as needed at Snowfield.
- Years 17 to 27:
  - All mineralized material production from Snowfield.

The mine schedule contains an aggressive bench advance rate exceeding 12 benches per year. While this is achievable with careful scheduling and management, and acceptable for this level of study, it does introduce a risk factor to the schedule that should be further investigated during the next phase of study.

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 18.12.

Table 18.12 Mineral Inventory

Pit Location	Block Model	Pit Name	Pit Phase	Mineral Inventory (t)	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Re (g/t)~	NSR ($/t)
Brucejack	Brucejack	SG Zone	BJ_1	1,203,278	1.25	7.54	-	-		29.59
		Shore Zone	BJ_2	3,604,375	3.14	18.49	-	-		79.98
		Gossan Hill	BJ_3	9,269,424	1.04	12.61	-	-		25.88
		Galena Hill	BJ_4	21,189,222	0.99	13.20	-	-		23.74
		Bridge Zone	BJ_5	35,075,429	0.98	13.45	-	-		22.43
			BJ_6	79,310,274	0.83	10.40	-	-		18.04
			BJ_7	39,376,773	0.90	10.09	-	-		19.32
	West Zone	West Zone	WZ_1	6,108,729	1.72	72.61	-	-		61.26
			WZ_2	7,713,901	1.54	49.26	-	-		49.55
			WZ_3	8,857,891	1.53	34.19	-	-		44.69
Snowfield	Snowfield	Snowfield	SF_1	48,889,376	1.49	1.43	0.03	0.01	0.86	31.35
			SF_2	163,763,010	0.72	2.00	0.13	0.01	0.50	18.62
			SF_3	220,303,353	0.67	1.65	0.10	0.01	0.61	15.62
			SF_4	172,215,796	0.64	1.61	0.11	0.01	0.48	15.35
			SF_5	175,750,916	0.57	1.53	0.10	0.01	0.44	13.70
			SF_6	178,977,826	0.56	1.62	0.09	0.01	0.38	13.15
Subtotals	Brucejack	All		189,028,775	0.95	11.46	-	-		21.40
	West Zone	West Zone		22,680,521	1.58	49.67	-	-		50.81
	Snowfield	Snowfield		959,900,277	0.68	1.66	0.10	0.01	0.51	16.07
Total*	**All**	**All**		**1,171,609,573**	**0.74**	**4.17**	**0.08**	**0.01**	**0.42**	**17.61**

* Average grades for copper, molybdenum, and rhenium not representative of process head grades due to batch assumption.

~ Rhenium grade not used in NSR calculation.

18.1.9 *Project Opportunities*

There are two main areas of opportunity for mining schedule improvement at the Snowfield-Brucejack Project.

As previously stated, one of the major constraints at the Snowfield property is the limited area available for mine waste dumping. This should be a point of focus during the next level of study.

The second area for opportunity is within the mine schedule itself. Currently the schedule has been optimized for NPV, based on production revenue and costs within the given constraints. The other factors such as capital cost expenditures, length of campaign mining periods, and mine operating costs should be considered when developing alternate schedules. It is recommended that alternate schedules are developed and evaluated in the next phase of the study.

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.

All equipment other than the excavator fleet is assumed to be shared between the two operations. Transport will be via a connecting road. The road will be suitable for tramming of haul trucks and other wheeled equipment while the dozers and drills will be transported by low-loader as necessary.

This approach to equipment scheduling will result in extended periods of under-utilization of the excavator fleet. This is an inefficient use of relatively high-capital equipment and will require attention to care and maintenance of the idle equipment to prevent degradation of the equipment through disuse.

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 Shift	h/shift	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, increasing to a maximum of four drills during peak production. An 8.9 m x 10.2 m average pattern size on a 15 m bench was selected for mineralized material and waste rock drilling.

The mechanical availability of the drills was estimated at approximately 80%. The maximum use of available hours was assumed to be 80% for each year of operation. The estimated effective utilization of the drills over the LOM is therefore 64%.

Blasthole Drill Productivity

Drill productivities were based on an instantaneous penetration rate of 32 m/h for both mineralized material and waste rock. Total estimated drill time per hole including penetration and move time is based on general operating experience, as shown in Table 18.14.

Table 18.14 Blasthole Drill Productivity

Drilling Parameters	Unit	All Material
Hole Diameter	mm	311
Hole Depth	m	16.8
Hole Yield	t	3,786
Penetration Rate – Instantaneous	m/h	32
Drilling Time per Hole	min	31.5
Time between Holes	min	2
Hole Collaring	min	1
Grade Control Sampling Delays	min	1.5
Total Time per Hole	min	36
	h	0.6
Penetration Rate per Operating Hour	m/h	28
Availability	%	80
Use of Availability	%	80
Operating Hours	h/a	5,606
Hourly Productivity	t/h	6,309
Yearly Productivity	**t/a**	**35,371,955**

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	All Material
Hole Diameter	mm	311
Burden	m	8.9
Spacing	m	10.2
Bench Height	m	15
Subdrill	m	1.8
Drilling per Hole	m	16.8

table continues...

Blasting Parameters	Unit	All Material
Yield per Hole	m^3	1,362
	t	3,786
Material Density	t/m^3	2.78
Stemming Height	m	4.5
Explosive Density	t/m^3	0.98
Charge Length	m	12.3
Charge Weight	kg	916
Powder Factor	kg/m^3	0.672
	t/m^3	0.24

GENERAL LOADING CONDITIONS

The total loading fleet consists of three 44.7 m^3 electric cable shovels and five 39.0 m^3 diesel hydraulic face shovels. Of these, two electric cable shovels and three diesel hydraulic face shovels will be located at Brucejack. It is assumed that, as the requirement for the shovels at Brucejack decreases, the machines will be decommissioned, transported, and permanently re-assigned to Snowfield.

At Brucejack, the diesel hydraulic face shovels will be used to preferentially mine mineralized material, while the electric shovels mine bulk waste. This is due to the slightly more complex nature of the Brucejack mineralized material, as the hydraulic shovels offer marginally higher selectivity than the electric shovels, and the high strip ratio will allow large working areas in waste for the electric units.

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

	Excavator Type		**Electric Shovel**	**Hydraulic Shovel**
	Truck Type		**363t Class**	**363t Class**
	Material Type		**Ore/Waste**	**Ore/Waste**
	Digger Configuration		**Shovel**	**Shovel**
	Mining Style		**Bulk**	**Bulk**
	Flitch Height	**(m)**	**15**	**15**
Material Detail				
	Dry Density	(t/bcm)	2.78	2.78
	Moisture Content	(%)	3%	3%
	Swell Factor	(%)	30%	30%
	Wet Loose Density	t/m3	2.20	2.20
	Wet Bank Density	t/m3	2.86	2.86
Shovel Details				
	Bucket Heaped Cap.	(m3)	44.70	39.00
	Fill Factor	(%)	95%	85%
	Bkt Cap. Volume	(bcm)	32.7	25.5
	Bkt Cap. Weight	(t)	90.8	70.9
	Bkt Cap. Weight	(bcm)	31.7	24.8
	Bkt Cap. Adopted	(bcm)	31.7	24.8
Truck Details				
	Tray Capacity	(m3)	220.0	220.0
	Trk Fill Factor	(%)	100%	100%
	Volume Limit	(bcm)	169.2	169.2
	Rated Payload	(t)	363.0	363.0
	Assumed Overload	(%)	0%	0%
	Adjusted Payload	(t)	363.0	363.0
	Weight Limit	(bcm)	126.8	126.8
	Adopted Capacity	(bcm)	126.8	126.8
	Min. Bucket Fill	(%)	95%	90%
	Calc Passes Per load		4.0	5.1
	Calc Passes Per load	(rounded)	4.0	5.0
	Actual Trk Load	(bcm)	126.8	123.8
	Actual Trk Load	(t)	363.0	354.5
	Actual Trk Load	(dry t)	352.4	344.1
	Dump Time	(min)	0.75	0.75
Excavator Productivity				
	Cycle Time	(sec)	35	42
	Efficiency Factor	(%)	92%	92%
	1st Pass	(sec)	35	42
	Truck Exchange	(sec)	30	30
	Loading Time	(min)	2.83	4.00
	Max. Productivity	(bcm/OH)	2,461	1,702
	Effective Ut'n of op hours	(%)	75%	75%
	Productivity	**(bcm/OH)**	**1,846**	**1,277**
	Productivity	(t/OH)	**5,285**	**3,549**
	Productivity per eff Hr	**(dry t/OH)**	**5,131**	**3,445**
	Availability	(%)	81%	75%
	Productivity	(dry t/OH)	4,156	2,584
	Hours per day	OH/day	17.01	15.75
	Productivity per day	**(dry t/day)**	**87,277**	**54,266**
	Productivity per Year	**(dry t/a)**	**31,856,119**	**19,806,964**

Note – equipment specification basis:
- Electric Cable Shovel – P&H4100XPC
- Diesel Hydraulic Face shovel – PC8000
- Haul Truck – Cat 797F Mechanical Truck.

GENERAL HAULING CONDITIONS

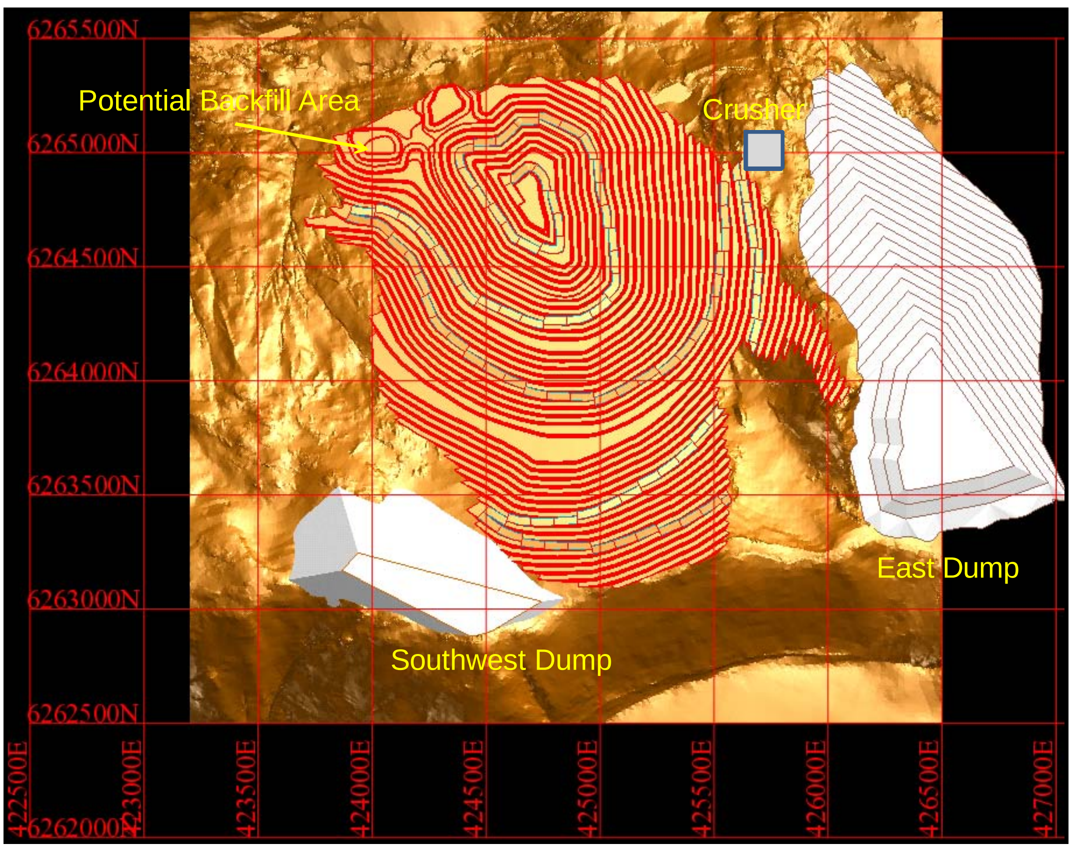
The 363 t haul truck was selected to match the 44.7 m³ electric cable shovel and the 39.0 m³ hydraulic shovel.

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. For Snowfield, these are shown in Table 18.17 and Figure 18.9.

Table 18.17 Snowfield Facility Locations

Facilities	Easting (m)	Northing (m)	Top Elevation (m)	Approximate Average Distance from Pit Centroid (m)
Primary Crusher	425,700	6,265,000	1470	3,899
East Dump	426,507	6,263,800	1960	4,595
Southwest Dump	424,346	6,263,052	1894	2,000

Figure 18.9 Snowfield Final Pit and Waste Dump Layout



Major Brucejack facility locations are shown in Table 18.18. The Brucejack Project will go through a number of phases as pits are completed and backfilled. Firstly, the SG, Shore Zone, Gossan Hill, and Galena Hill pits will be mined and the lake backfilled as shown in Figure 18.10. Once those pits are complete, the West Zone will be mined and the depleted pits backfilled with waste, while the lake continues to be filled and construction of the waste dump commences (Figure 18.11). Finally, the Bridge Zone will be mined with waste going to the West Zone pit and the waste dump (Figure 18.12).

Table 18.18 Brucejack Facility Locations

Facilities	Easting (m)	Northing (m)	Top Elevation (m)
Primary Crusher – Initial	427,080	6,258,780	1370
Primary Crusher – Post Year 8	426,850	6,258,060	1480
Waste Dump	427,700	6,258,800	1500

Figure 18.10 Brucejack First Phase Pit Layout

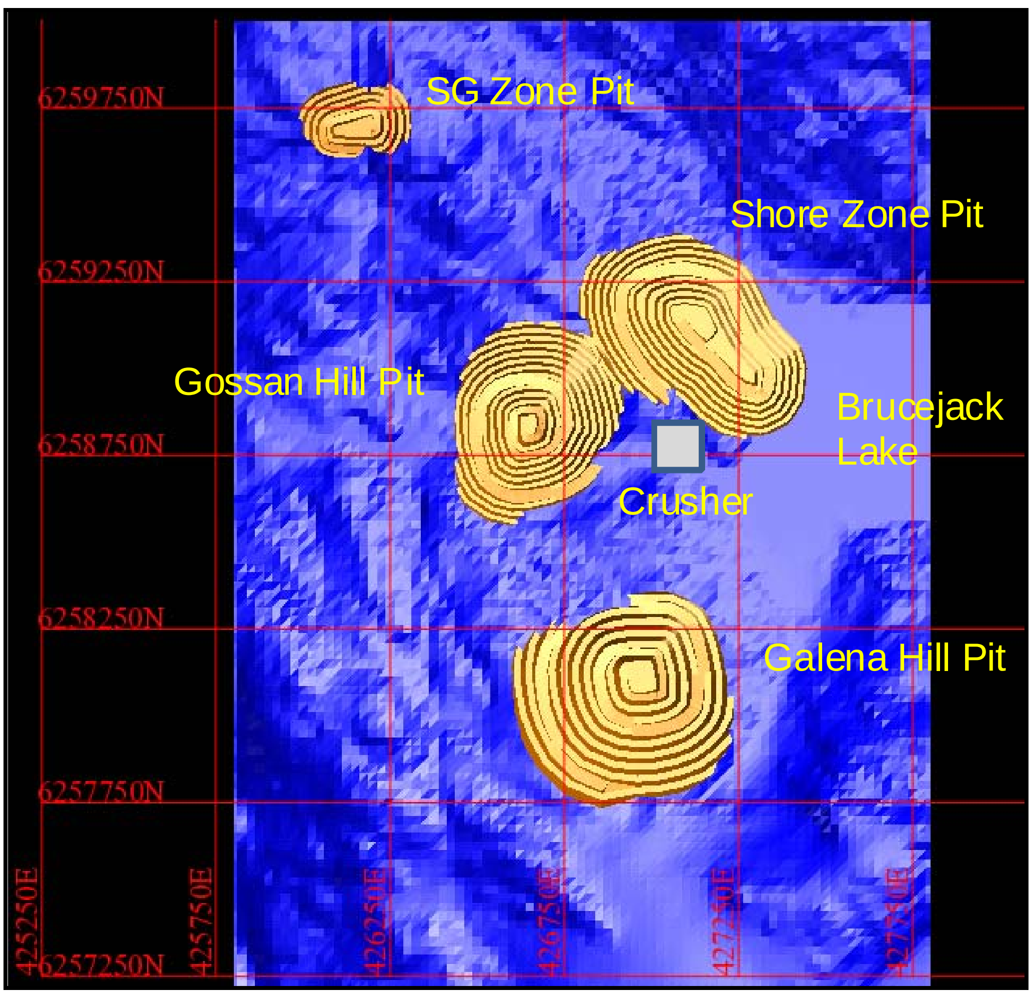


Figure 18.11 Brucejack Second Phase Pit Layout

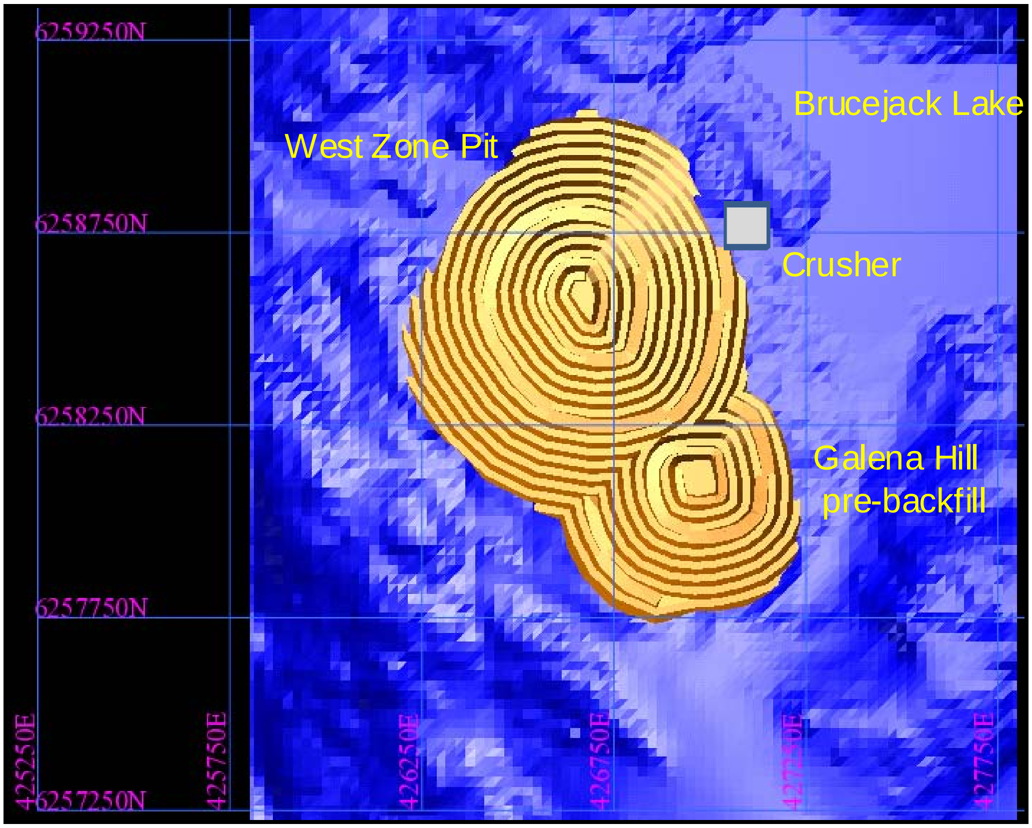
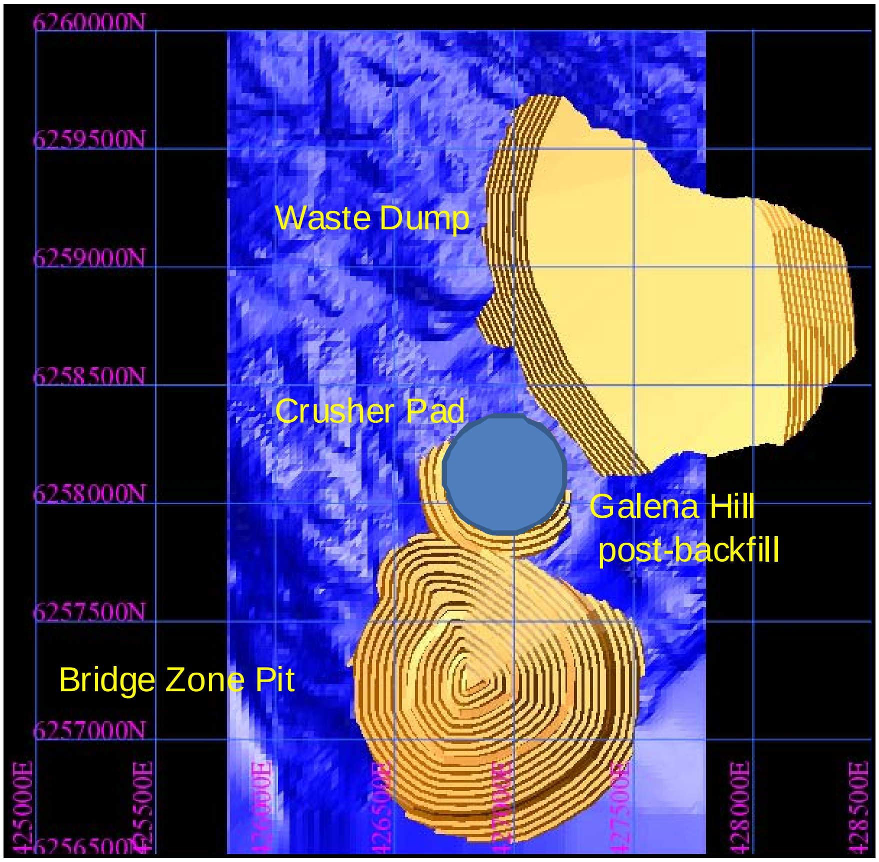


Figure 18.12 Brucejack Final Phase Layout



Haul Truck Productivity

The mining schedule has been completed on a bench by bench basis, which has then been used to estimate truck haulage profiles.

The approximate centroid of each mining bench has been estimated and, in conjunction with identified pit exits and assumed dumping locations, haulage distances and gradients have been calculated. Manual calculations were then completed to estimate truck cycle times. The cycle times included travel time (loaded and empty), loading time, dumping time, and delays.

Finally, the cycle times were applied to the bench schedule and truck requirements on a period by period basis. These requirements were then used to create the mine fleet capital schedule for fleet expansions and replacements.

Water Management for the Open Pit and Waste Dumps

Surface water management around the mining areas and in-pit dewatering are discussed in detail in Sections 18.3.3 and 18.3.4.

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	15,345
Mobile & Support Equipment	205,623
Explosives Storage	488
Fuel Storage & Delivery	460
Electrical & Distribution	41,000
Communication	1,032
Safety	122
Engineering Equipment	2,811
Total Mine Capital	**266,881**

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 cost escalation.

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.

18.2 INFRASTRUCTURE

18.2.1 MINE AND SITE LAYOUT

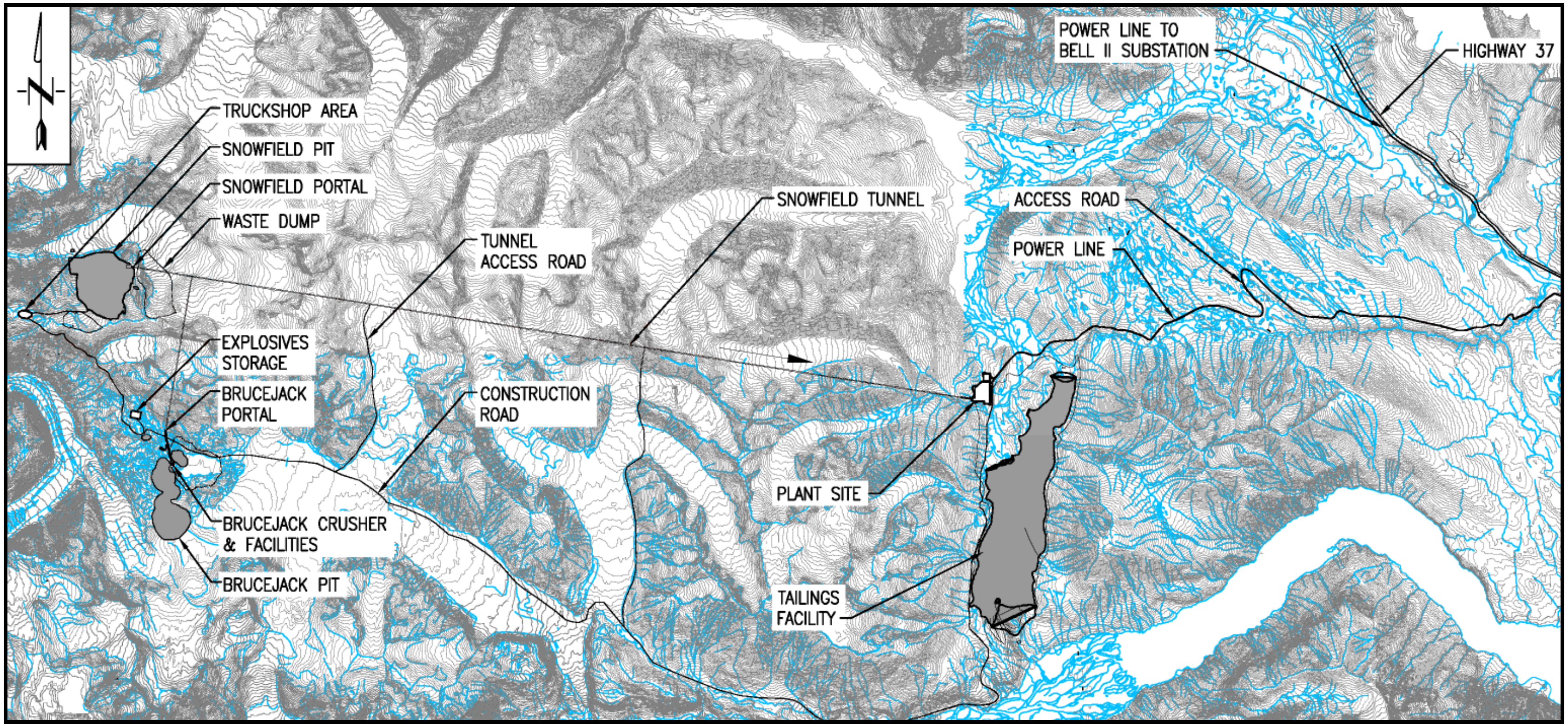
The general arrangement of the mine and plant sites for the Snowfield-Brucejack Project is presented in Figure 18.13.

The Snowfield-Brucejack Project site will be accessible by a permanent road to be constructed southwest from Highway 37 to the plant site. Highway 37, a major road access route to northern BC, passes approximately 24 km from the Snowfield-Brucejack Project plant site. A 45 km construction road from the plant site location to the open pits will be upgraded and used to mobilize equipment and supplies. Part of this road connecting the Brucejack and Snowfield properties will be upgraded to a permanent road to allow for equipment transfers between pits.

The plant site is located 26 km to the east of the Snowfield-Brucejack Project area. Twin tunnels will be developed to connect the plant and the mine sites: 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 TSF is located approximately 5 km southeast from the plant site within the Scott Creek valley.

Figure 18.13 Snowfield-Brucejack Overall Site Plan



18.2.2 ANCILLARY BUILDINGS

The pre-engineered and stick-built structures will be constructed for the Snowfield-Brucejack 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 at the plant site
- 1,000-person in total 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 for the Snowfield and Brucejack operations. 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 at the process plant and Snowfield and Brucejack sites. Each diesel fuel storage tank will have a capacity sufficient for approximately seven days of operation. Diesel storage 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 and Doré Storage

Copper-gold and molybdenum concentrates will be stored in an on-site facility capable of storing a week of 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. Doré will be stored in a secured vault and shipped off-site on a regular basis by specialty service provider contracted by the mine.

18.2.6 Roads and Access

The plant site will be accessible via a new 24 km-long road from Highway 37. In addition, a temporary 45 km construction road from the plant site to the pits will be provided. Both the main access road and the construction road will approximate the path of the old Newhawk 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 Brucejack and Snowfield pits; part of this road will also act as the permanent mining truck access/connecting road between Brucejack and Snowfield pits
- miscellaneous roads for use in the TSF construction and operation
- grading of the pads for the Snowfield truck shop, the Brucejack truck shop, the explosive magazine, and the primary crushers
- the road from the Brucejack pit to the dual connecting tunnel portal, the Brucejack truck shop access road, and the explosive magazine access road.

The main access road route roughly follows an access road that was reportedly utilized by Newhawk 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 that there is 300 mm of topsoil, and that 50% of the remaining material is rock. Approximately 50% of that rock is assumed to be rippable and the remaining rock will require the application of drilling and blasting methods.

The 45-km construction road extends from the proposed plant site, and will be used for construction traffic accessing the Snowfield and Brucejack properties. It follows much of the overland portion of the access road used by Newhawk during their exploration activities. The proposed construction road route travels south from the plant site and parallels the tailings pond along the route of the track developed by Newhawk; the road then turns west and traverses the glaciers leading to the pit areas. Where possible, the construction access road will serve as access to the west tailings pipeline/diversion ditch maintenance areas, a haul road from the rock quarries to the southern tailings dam, and the permanent mining truck access/connecting road between the two pits. It is believed that the Newhawk track will need major up-grading/re-routing in order to allow for the haulage of major mining components and large quantities of construction materials.

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 are a 9 km of maintenance roads for the east tailings pipeline/reclaim water pipelines and other minor roads.

To shorten the construction time for the twin tunnels (from the Snowfield 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.

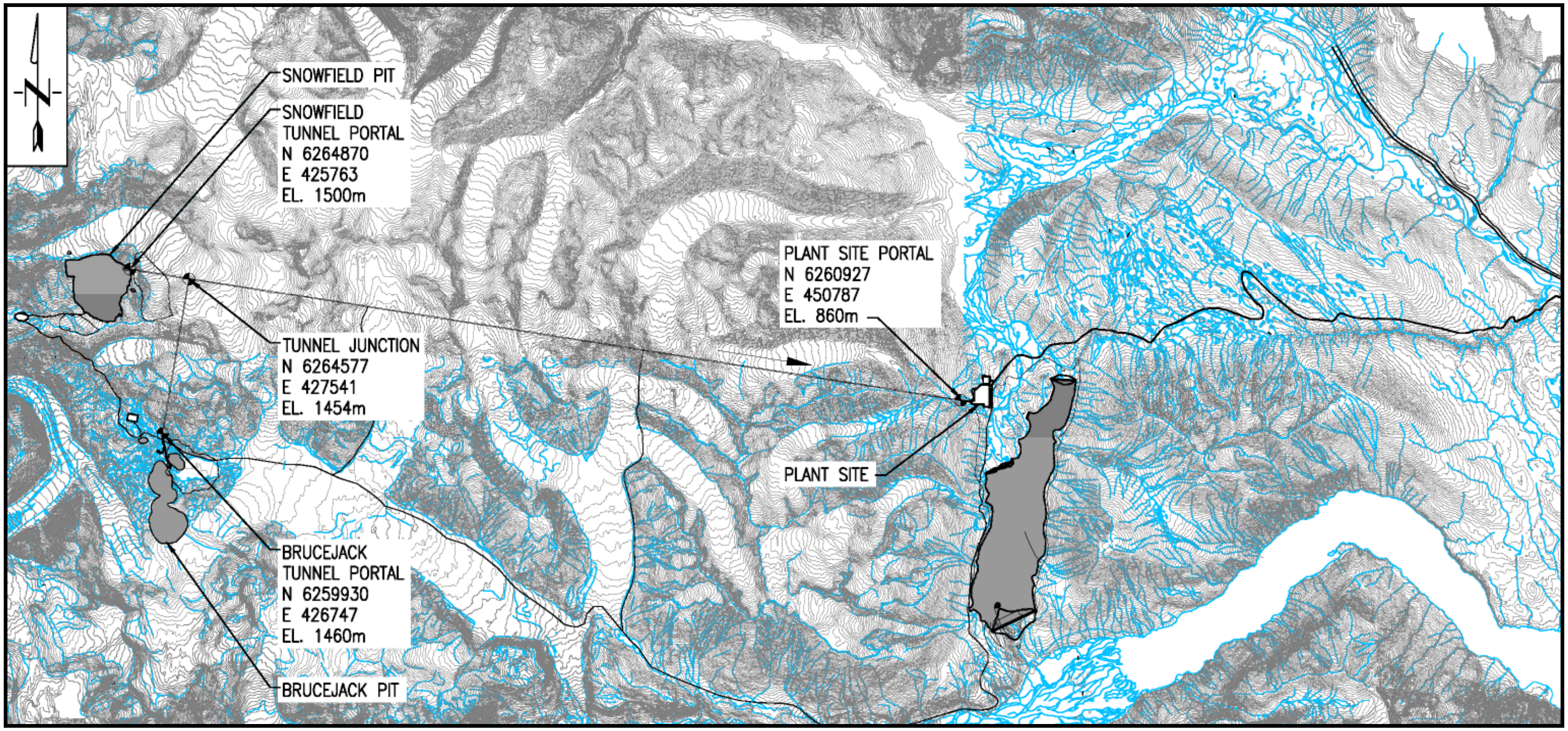
No allowance has been provided for hazard control (e.g. avalanche, landslide, etc.) or hazard avoidance. An assessment of the risks and mitigations with respect to these hazards is required.

18.2.8 TUNNEL DEVELOPMENT

The process plant facilities and the mine camp will be located approximately 26 km east of the Snowfield and Brucejack properties. Twin tunnels will connect the mill and the mine areas. One tunnel will be used mainly for conveying the mineralized material from the pits to the processing facilities, and the other tunnel will provide a reliable year-round route to the mine sites for materials and workers.

The location of the proposed tunnels is shown in Figure 18.14.

Figure 18.14 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 shorter project duration and potentially lower costs.

The 26 km tunnel from the Snowfield pit to the mill 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 6.8 km from the pit side. Another portion of the tunnel from the mill side will be 17.4 km-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 5 km tunnel connecting the Brucejack operation with the Snowfield tunnel will be developed from two working faces – one located at the Brucejack portal, and the other located at the intersection with the Snowfield tunnel.

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 the use of 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 18.15 and Figure 18.16.

Figure 18.15 Conveyor Tunnel

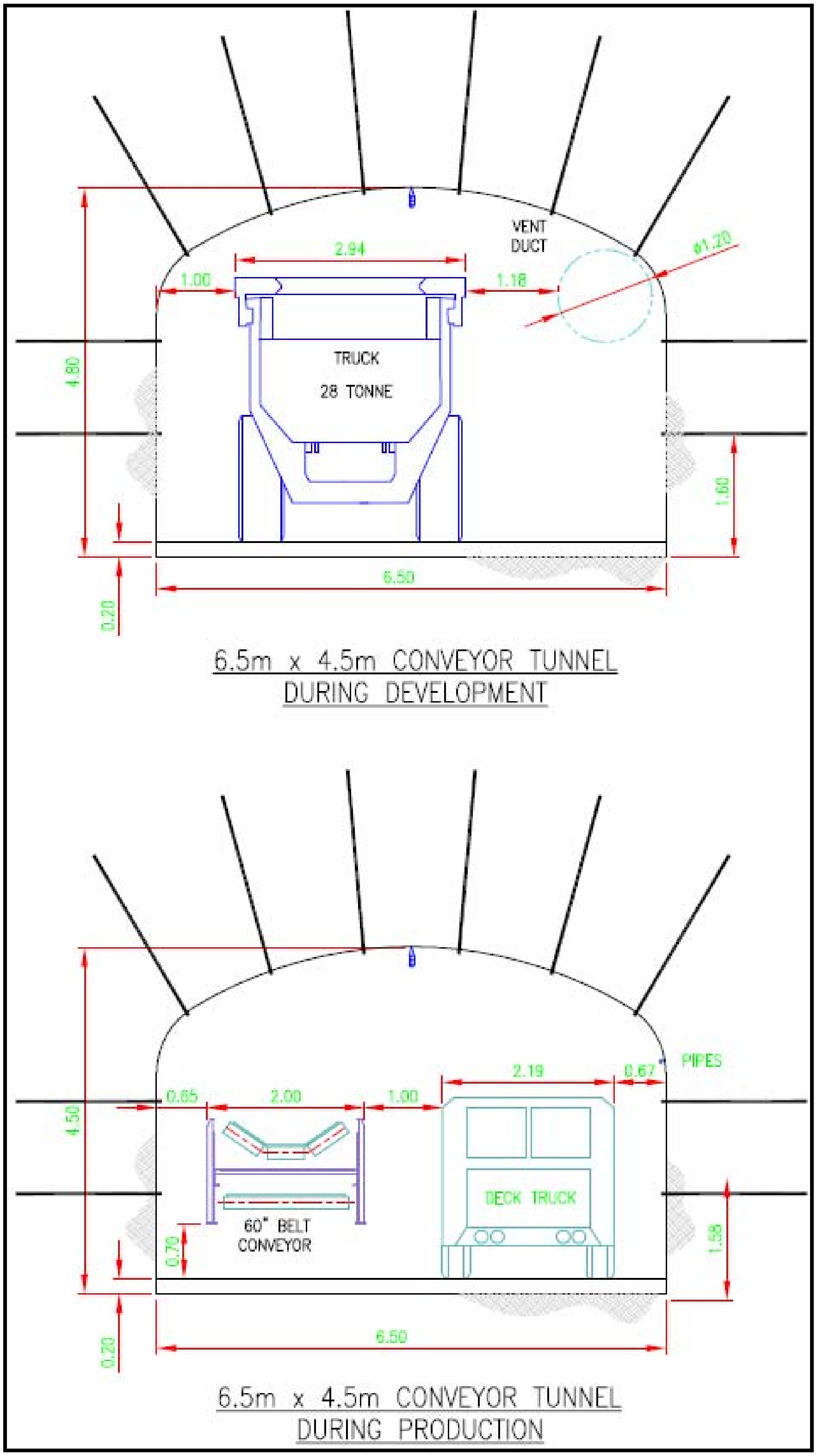
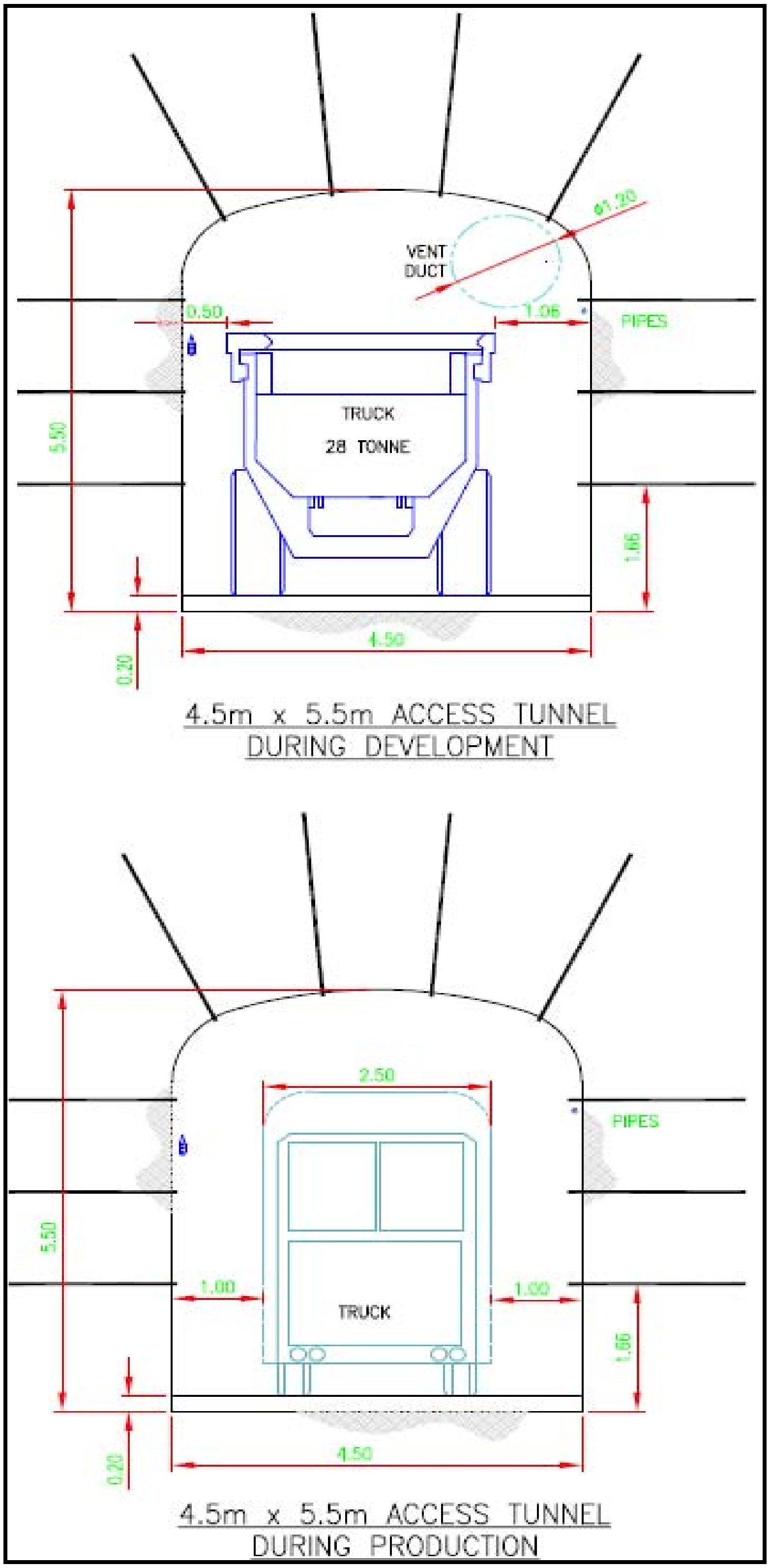


Figure 18.16 Main Access Tunnel Cross Section



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 load-haul-dump (LHD) vehicles 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 for 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 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 18.20.

Table 18.20 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 m 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 pits and plant sites 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 mine operations.

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 all sites.

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 Snowfield and Brucejack properties, and 6 km from the process plant. The tailings delivery system was designed to transport 1,172 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 150 MW ± 10%.

POWER SOURCE – NORTHWEST TRANSMISSION LINE

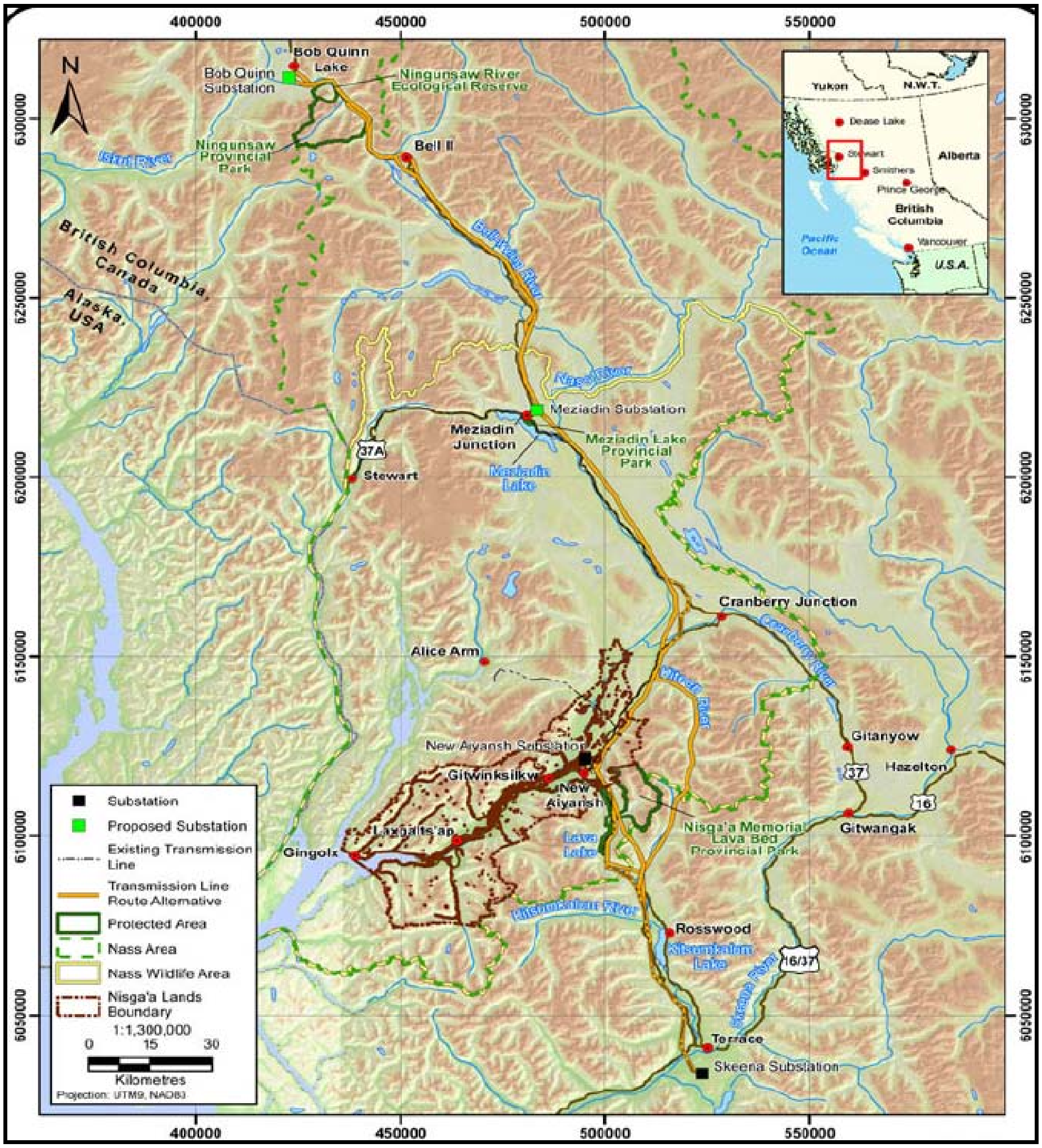
Electrical power will be supplied from the proposed Northwest Transmission Line (NTL), which is to be built by winter 2012. The NTL will be a 287 kV line between Terrace, BC, and Bob Quinn Lake, BC, a distance of approximately 335 km.

The latest British Columbia Transmission Corp. (BCTC) cost estimates indicate that the line will cost C$404 M. The Government of Canada has pledged C$130 M, which leaves a balance of C$274 M to be split between the BC provincial government and the private sector. The provincial government has stated that it expects up to C$90 M from the private sector, so the expectation is that the provincial government will eventually pledge C$184 M.

For the purposes of this study, it is assumed that the Snowfield-Brucejack Project contribution to the NTL is C$20 M.

A map of the proposed line to Bob Quinn Lake is shown in Figure 18.17:

Figure 18.17 Proposed Northwest Transmission Line Route



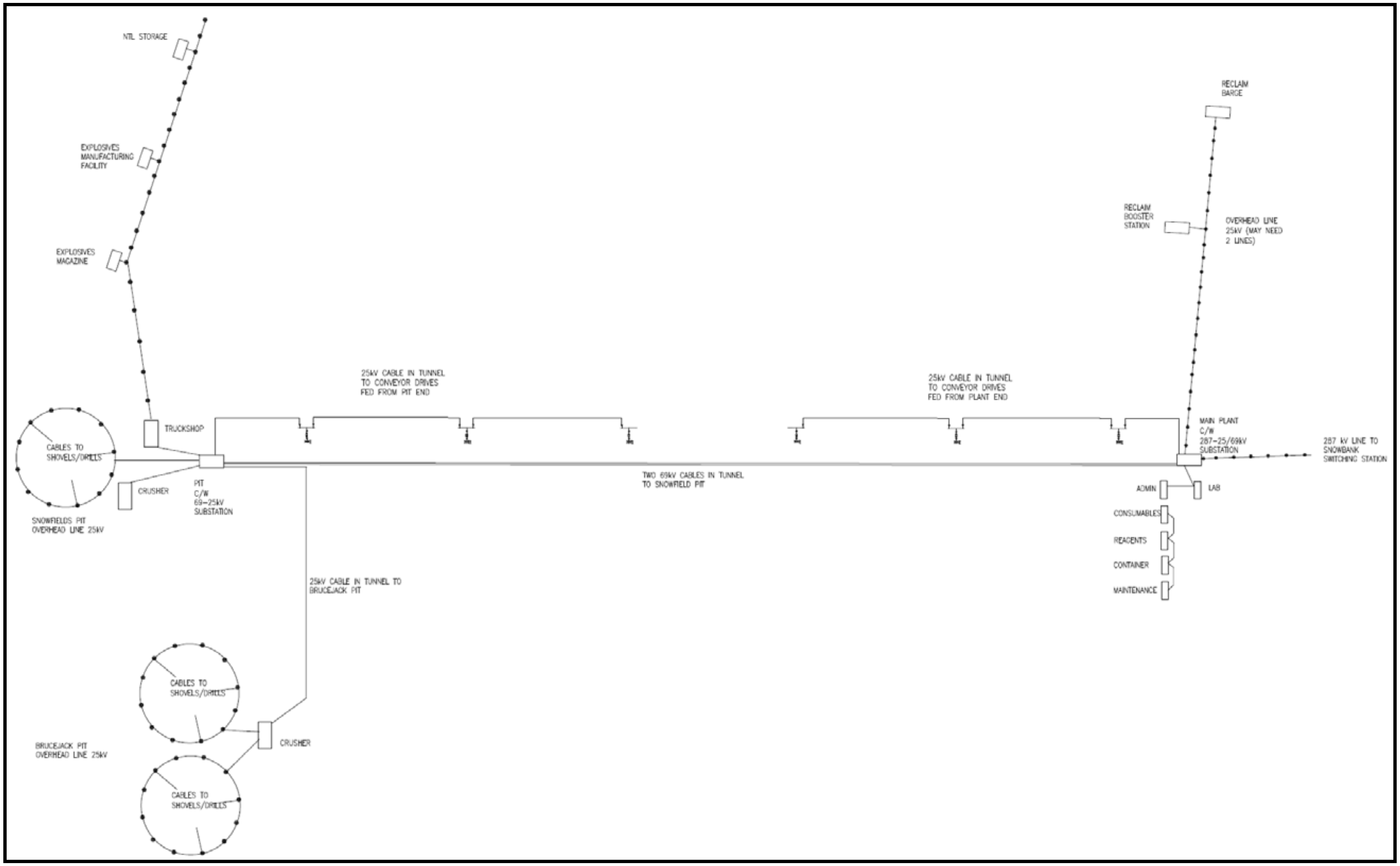
Note: BCTC.

Service to Snowfield and Brucejack Pits

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

The line to the Snowfield-Brucejack Project will be approximately 45 km long, and will terminate at a distribution substation at the mill site. A conceptual one-line diagram for the Snowfield-Brucejack Project is shown in Figure 18.18.

Figure 18.18 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 t 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-Brucejack main substation to step back up to 69 kV. This will be a suitable voltage to feed via cable through the tunnel to the pits, where it will be further stepped down to 25 kV, 4 kV and 600 V to feed the shovels, drills, and primary crushers.

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-Brucejack 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-Brucejack Project have been prepared by BGC and Rescan.

18.3.1 TAILINGS MANAGEMENT

To ensure that the TSF continuously meets its objectives, a tailings 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–Brucejack Project
- provide a means to manage the TSF itself (managing substances going in 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 tailings and supernatant to allow estimation of the rate of oxidation and resulting water quality. This information will guide planning for tailings water management.

At present, it is assumed that the high sulphide content of the pyrite tailings 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 tailings permanently under water where oxidation is vastly reduced or eliminated. The TSF is designed to isolate the pyrite tailings in a stable subaqueous environment in perpetuity. Diversion channels will be constructed on both sides of the TSF to minimize surface runoff to the facility.

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

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, through its consultants, Silver Standard will develop a comprehensive water management plan that applies to all mining activities undertaken during all phases of the Snowfield-Brucejack 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, and collecting surface water from disturbed areas and treating it to meet discharge standards prior to release
- minimizing the use of fresh water; recycling of 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 that will be constructed to direct runoff away from disturbed areas.

18.3.3 *SNOWFIELD WASTE DUMP AND OPEN PIT WATER MANAGEMENT*

GENERAL

The proposed Snowfield open pit is located downstream of the Mitchell Glacier on the south side of the Mitchell Creek valley. Elevations for the final pit footprint range from approximately 1020 m to 1900 m. The south end (crest of the highwall) of the pit daylights very close to the drainage divide between Mitchell and Sulphurets valleys. At the north wall, the pit crest daylights above the elevation of the Mitchell Glacier and Mitchell Creek (Figure 18.19).

The Snowfield waste rock facility (WRF) will consist of two dumps. The East WRF (470 Mt) is located to the immediate east of the open pit and to the west of the Mitchell Glacier. A smaller waste dump is located near the watershed divide on the southwest side of the pit perimeter. The Southwest WRF will contain approximately 68 Mt of waste rock.

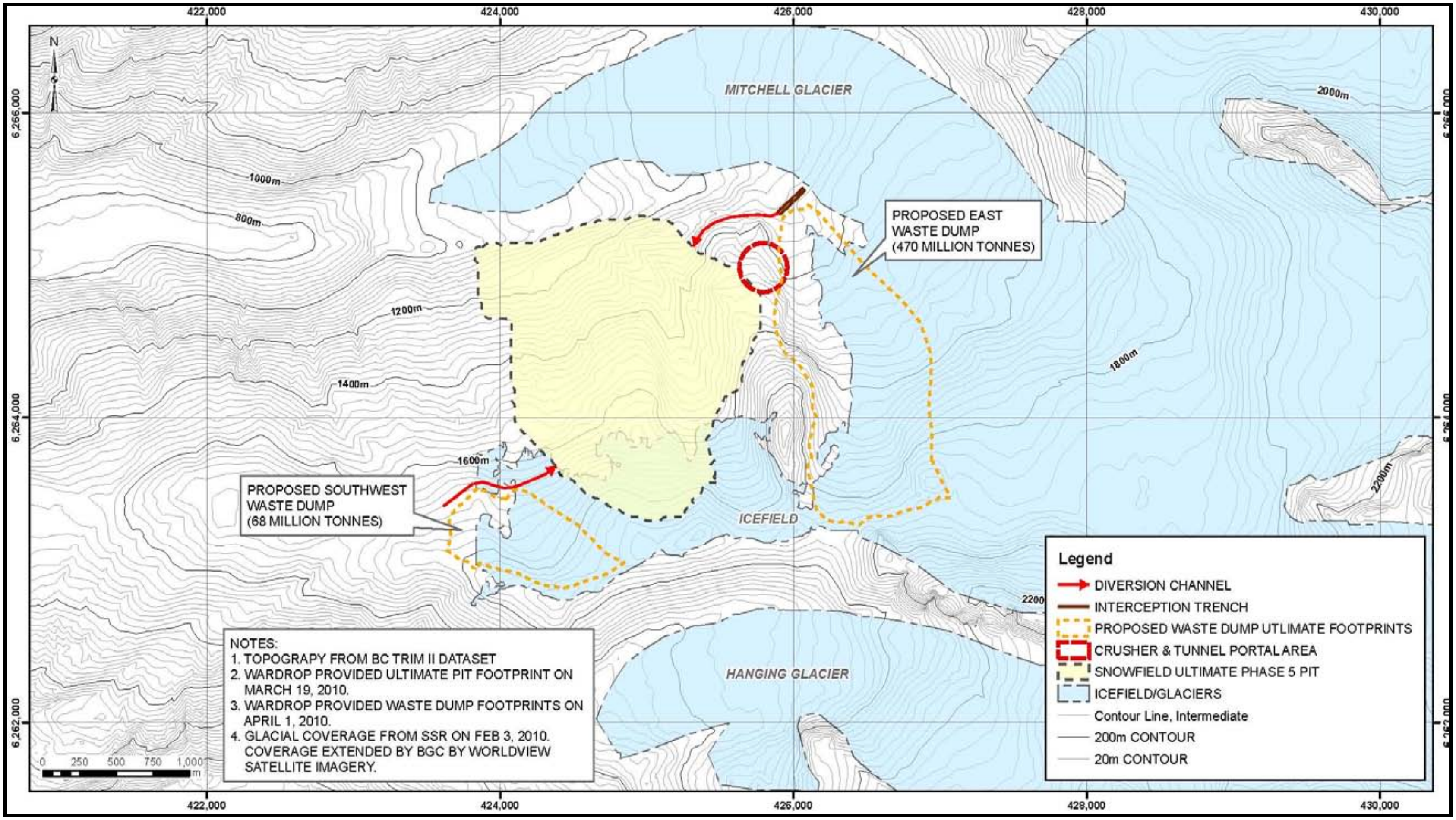
WATER MANAGEMENT STRATEGY

Runoff 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.

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

- Runoff into the open pit will be managed with a combination of sumps and pumps. This runoff will be pumped up to a surge pond located near the tunnel portal that leads to the process plant. 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.
- Pit dewatering groundwater will also be directed to the surge pond and process plant pipeline.
- Interception trenches will be constructed down gradient of both the East WRF and Southwest WRF to collect both surface runoff and seepage. Runoff collected by the southwest interception trench will discharge directly into the open pit, while a majority of runoff collected by the east interception trench will be pumped directly up to the tunnel portal. In the event that inflows to the east trench exceed the pumping and storage capacity, excess flows will be directed to the open pit. A total of six monitoring wells will be established down gradient of both interception trenches.
- Water collected at the toe of the East WRF may be suitable for release (dependent on water quality) and may be discharged back into Mitchell Creek.

Figure 18.19 Snowfield Open Pit and Proposed Waste Dumps



PRELIMINARY WATER BALANCE

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, while annual lake evaporation and sublimation was estimated at around 215 mm. Given that potential evaporative losses are very low, resulting runoff coefficients are very high. Based on average precipitation conditions, an average annual runoff volume of 8.4 Mm^3 (953 m^3/h) has been estimated for the life-of-mine. This volume represents about 10% of the annual process plant requirements, although a majority of this volume will only be available for about half the year. As the pits and waste dumps develop, increased runoff volumes will need to be handled. Average runoff volumes for the final year of mining are approximately 50% greater than the life-of-mine average (based on average precipitation conditions).

Depending on the risk management and mining strategy employed by Silver Standard, there are a number of ways in which the runoff could be handled. The following is a description of the proposed strategy.

Given that a high percentage of runoff is expected to occur during snowmelt, the open pit bottoms would be used as a sump during snowmelt and the remainder of summer/early fall. Using the bottom of the pits as a sump would allow a lower pumping rate to be used through the 6-month warm period (May to October). With this strategy, a significant portion of the pit bottoms may be inaccessible during this period. The pumps would be sized appropriately so that the sump was dry for a portion of the winter, except for years with well above average annual precipitation. The coldest winter months would then be used to advance the open pit bottoms 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 by the end of mine life for average precipitation conditions out of the Snowfield open pit is approximately 1,440 m^3/h. However, the pumping system should be sized to accommodate years with above average precipitation as the pit bottoms need to be accessible for a portion of the year. Annual precipitation with a 200-year return period and a 12-month pumping period has been conditionally adopted as the design standard for the pumps and pipelines. Based on these criteria, the maximum pumping rates required during mine life are:

- 1,600 m^3/h from the open pit sump
- 600 m^3/h from the East WRF interception trench.

Accounting for pit dewatering flows, the pipeline from the Snowfield tunnel portal to the process plant would then be sized for approximately 2,300 m^3/h, or 0.64 m^3/s. Using this maximum pumping rate, approximately seven months would be required to dewater the maximum open pit footprint under average precipitation conditions.

Higher pumping rates will be required if a dry sump is required for a longer period within the year.

A 200-year return period is a conservative assumption for open pit design, but is likely warranted given the extreme climatic conditions experienced in this region (high annual precipitation and minimal evaporative losses due to high humidity and low temperatures).

18.3.4 BRUCEJACK WASTE DUMP AND OPEN PIT WATER MANAGEMENT

GENERAL

The Brucejack deposit, located west of Brucejack Lake on the east side of the Sulphurets Glacier valley, is proposed to have six open pits (Figure 18.20). The SG and Shore Zone pits will be mined first followed by concurrent mining of the Gossan Hill and Galena Hill pits. When these pits are depleted, mining of the West Zone pit will occur, eventually merging with the Gossan Hill pit. Finally, the Bridge Zone pit will be mined, which eventually merges with the Galena Hill Zone after several phases of mining. Roughly two-thirds of the Bridge Zone pit will be mined through glacial ice.

As mining progresses, 120 Mt of waste rock will be backfilled into dewatered Brucejack Lake and SG, Gossan Hill, and West Zones. A 430 Mt waste dump will be constructed over backfilled Brucejack Lake and the surrounding natural surface. The ultimate footprint of the Brucejack waste dump is 172 ha.

WASTE MANAGEMENT STRATEGY

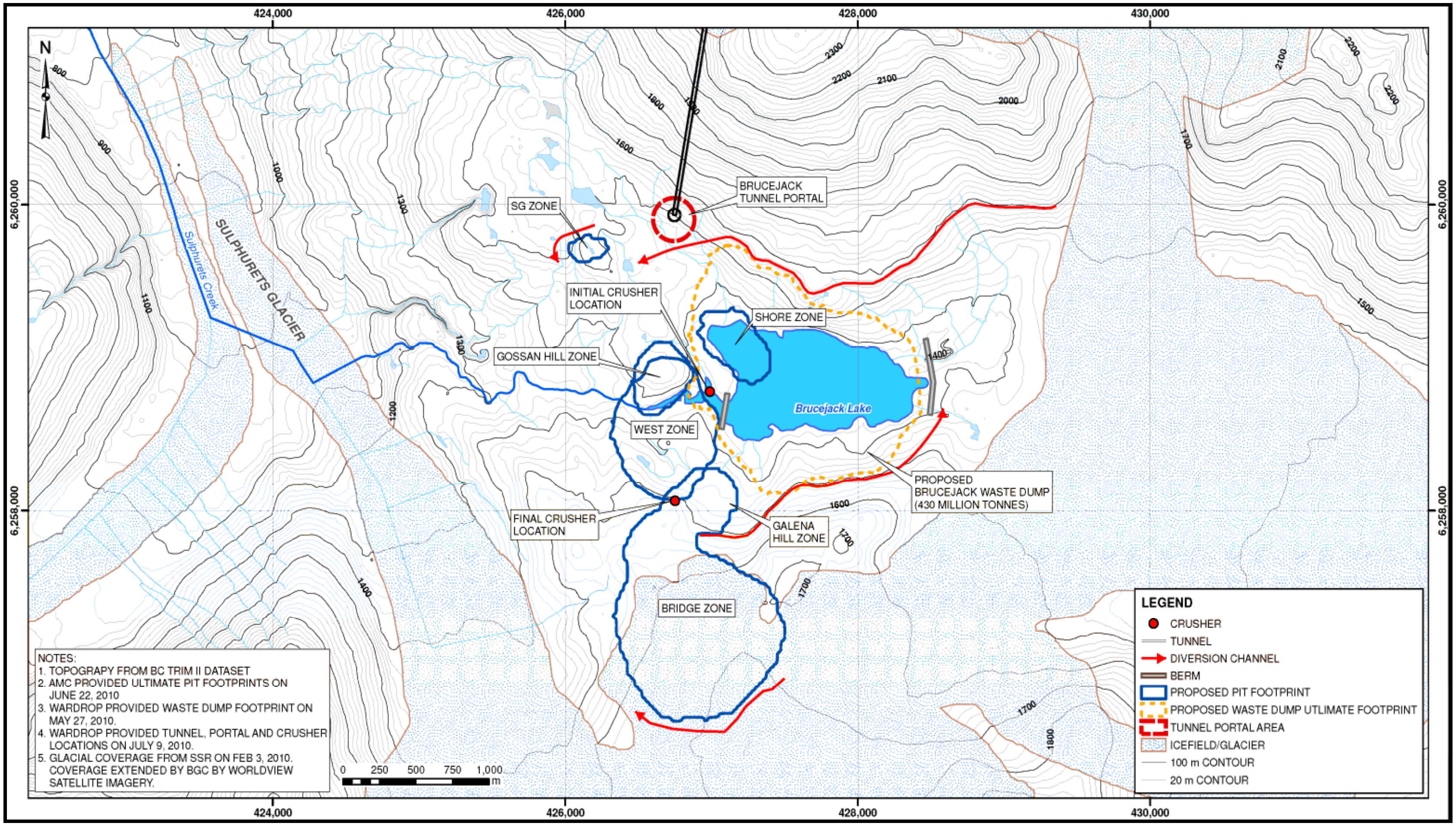
Similar to Snowfield, runoff into the open pits and waste dump at Brucejack is assumed to be contact water that needs to be contained without any uncontrolled discharge to the environment. The water management strategy for the Brucejack area is as follows:

- In order to mine the Shore Zone, Brucejack Lake (28.8 Mm^3) will be dewatered prior to the start of operations. Accounting for concomitant runoff from the adjacent slopes, an average pumping rate of 3,900 m^3/h would be required to dewater the lake over a 1-year period.
- Freshwater diversion channels will be constructed on the north (3.3 km) and south (2.1 km) sides of the waste dump. The North diversion channel will discharge to a small tributary of Sulphurets Creek downstream of mining activities, while the South channel will discharge into a small impoundment located immediately upstream of Brucejack Lake. The combined diversion area is approximately 520 ha.
- Runoff to the open pits will be managed with a combination of sumps and pumps. This runoff will be pumped up to a surge pond located near the

Brucejack tunnel portal that leads to the Snowfields tunnel, 4.7 km to the north. Water in the surge pond will then be directed into a pipeline and pumped up to a junction in the Snowfields tunnel, 1.8 km from the Snowfields portal, from where it will be gravity-fed to the plant for use in process.

- A berm (Lower Berm) will be constructed near the current Brucejack Lake outlet to prevent runoff to the Gossan Hill and West zones open pits and will contain runoff coming into contact with the waste rock. Ponded water will be pumped to the Brucejack surge pond and portal for transport to the process plant.
- A second berm (Upper Berm) will be constructed at the upstream end of the lake. The Upper Berm will limit the volume of water coming into contact with the waste rock by intercepting freshwater runoff from a watershed area of approximately 500 ha, as well as receiving runoff from the South diversion channel. Water impounded by the Upper Berm will be pumped up to the North diversion channel for discharge to the environment.
- Pit groundwater will also be directed to the surge pond and process plant pipeline.
- Freshwater runoff to the south of the Bridge Zone will be diverted to the east margin of the Sulphurets Glacier, and freshwater runoff to the north of the SG Zone will be diverted to natural drainage to the west of the SG Zone open pit.
- A significant volume of ice will need to be removed to mine the Bridge Zone pit; approximately 90 ha of the pit footprint is covered by the Sulphurets Glacier. All of this ice will need to be removed prior to mining, including a 100 m buffer to the west and south in order to construct an upslope diversion channel. Removed ice will be deposited downslope of mining activities on the east margin of the Sulphurets Glacier.

Figure 18.20 Brucejack Open Pits and Proposed Waste Dump



PRELIMINARY WATER BALANCE

A preliminary water balance for the Brucejack open pits and waste dump was constructed using similar inputs as described for Snowfield. Based on average precipitation conditions, an average annual runoff volume of 10.7 Mm^3 (1,222 m^3/h) has been estimated for the life-of-mine. This volume represents about 13% of the annual process plant requirements, although a majority of this volume will only be available for about half the year. As the pits develop, increased runoff volumes will need to be handled (i.e. approximately 40% greater for the final year of mining over the life-of-mine average).

Using the same strategy for the Brucejack deposit (200-year return period design standard and a 12-month pumping period), the maximum pumping capacities are as follows:

- Upper Berm to North diversion channel: 1,800 m^3/h
- Lower Berm to Brucejack tunnel portal: 1,100 m^3/h
- open pits to Brucejack tunnel portal: 900 m^3/h.

To account for pit dewatering flows (a maximum of about 600 m^3/h during mining of the Bridge Zone), the pipeline from the Brucejack tunnel portal to the main tunnel are sized for approximately 2,600 m^3/h, or 0.72 m^3/s.

18.3.5 TAILINGS STORAGE FACILITY WATER MANAGEMENT

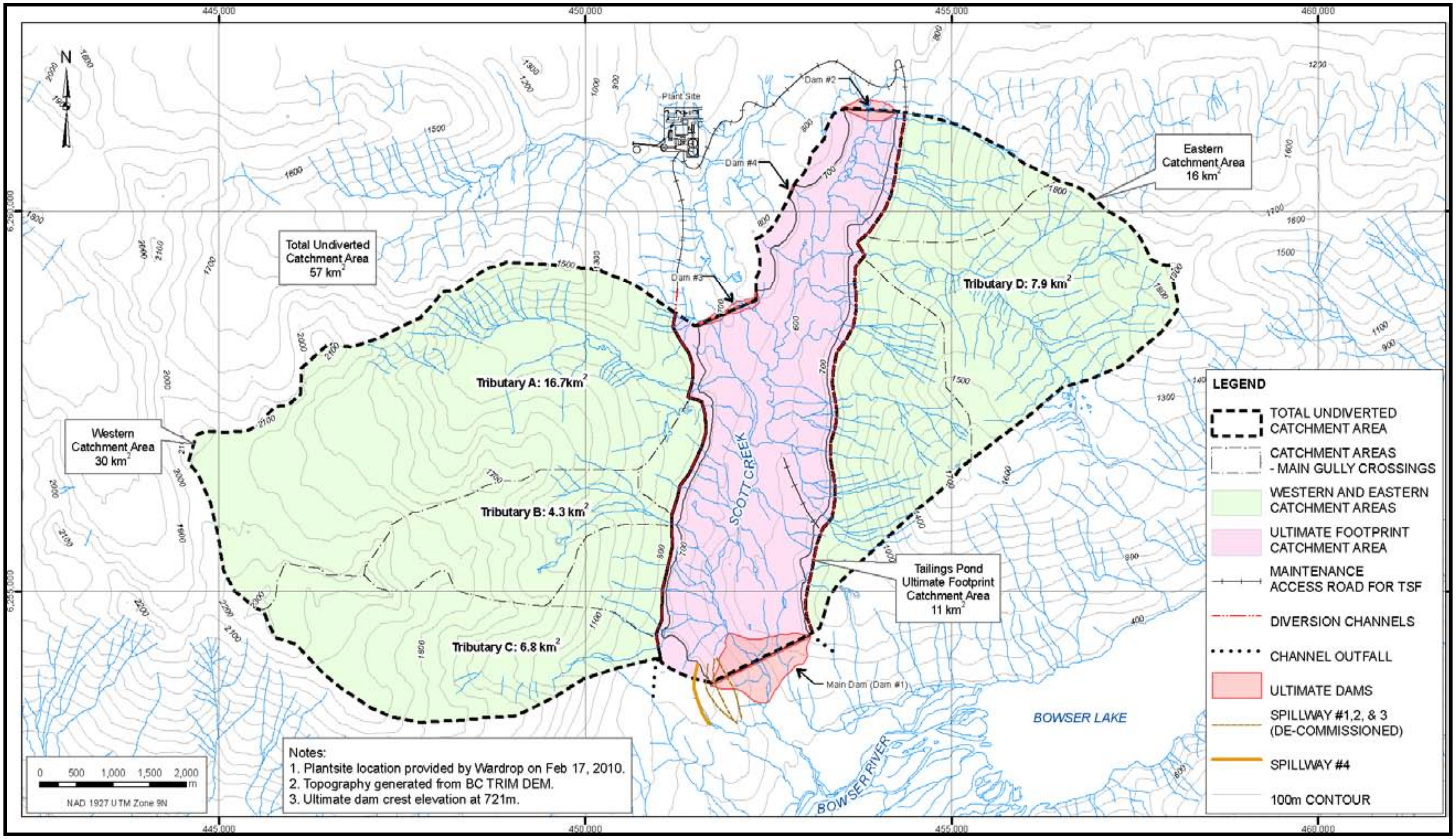
GENERAL

The catchment area reporting to the TSF is approximately 57 km^2 (Figure 18.21). 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. Assuming a diversion efficiency of 80%, the total area reporting to the TSF is estimated at approximately 20 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 a maximum of 2300 masl is 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.

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

Figure 18.21 Scott Creek Proposed TSF – Catchment Areas Plan



Disturbed areas such as overburden storage sites will be vegetated or otherwise protected from erosion. Runoff 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. Only reclaim water and water from mining areas will be used in process to minimize impacts to the environment. The quality of water in streams affected by the project, and of all discharges, will be monitored on a regular basis.

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 a manageable water balance given the large catchment area and wet climate. Approximately 13 km of channel is proposed around the impoundment and is designed to pass peak flows from a 200-year flood event.

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.

Spillways

To protect the integrity of the main tailings dam, flows in excess of the 200-year return period event 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 from the entire catchment area 56.7 km^2.

Seepage Recovery Systems

Foundation treatment for the tailings dams (Dams #1, #2, #3, and #4) 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.

PROCESS WATER REQUIREMENTS

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

- reclaim from the TSF pond
- seasonal runoff from the open pits and WRFs that will be piped to the process plant through the proposed 26 km tunnel
- pit dewatering groundwater from the open pits.

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. Runoff and pit dewatering groundwater from the Snowfield and Brucejack open pit and WRF areas will be piped to the process plant through the proposed tunnels. Variable amounts of process water will come from the mine site throughout the year; therefore, 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 t/d at 35% solids by weight
- an average annual precipitation of 1,525 mm and evaporative/sublimation losses of 374 mm for open water
- runoff co-efficients of 0.75 to 1 for the various land surfaces (i.e. undisturbed ground, active tailings beach, inactive tailings beach, and pond)
- all of the contact water collected at the Snowfield and Brucejack mine site areas (open pit runoff, seepage and surface runoff from the dumps, and pit dewatering groundwater) will be pumped to and used in the process plant.

Because of high annual precipitation and minimal evaporative losses, the TSF is expected to operate with a net annual surplus of water. 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 either Scott Creek or Bowser River.

Based on average precipitation conditions, the supernatant pond is estimated to an average annual surplus volume of 27.1 Mm^3 (3,100 m^3/h) over the LOM. 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 6,200 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 runoff 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 538 Mt of waste rock in the East and Southwest dumps, which refers to their relative location to the Snowfield open pit (Figure 18.19).

Approximately 430 Mt of waste rock will also be placed in the Brucejack waste dump, constructed over backfilled Brucejack Lake and the surrounding natural surface (Figure 18.20).

The following parameters were provided to AMC 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 dump assumes that it will generally consist of “fair” quality rock, consistent with the majority of the rock observed in the Snowfield and Brucejack pit areas. Poor quality rock, which will be mined from the Snowfield landslide area, is not desirable in the foundation of any of the waste dumps. If possible, the poor quality materials should be mixed with better quality waste rock to avoid zones of weakness within the waste dump. The material excavated from the landslide will not likely be suitable as rock drain construction material, should a rock drain be required.

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 more dense and may have a slightly lower swell factor. The recommended overall dump slopes of 2:1 are likely at the upper end of those suitable for reclamation; however, slopes at these angles can still be re-graded with bulldozers. However, re-vegetation needs and long term stability requirements may still need to be considered when 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 for the Snowfield East dump is proposed for an area immediately adjacent to the Mitchell Glacier (Figure 18.19). The East dump toe is located at 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 Snowfield 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 scoping-level waste dump design for the Brucejack dump consists of placing waste rock into Brucejack Lake. Once the lake is full, waste will be stacked approximately 120 to 140 m above the original lake level, with waste rock dump faces extending to the east and southwest-west-northwest. The elevation of the top of the dump is 1600 m. The dump is approximately 1.5 km long and 1.1 km wide.

The dump locations and configurations are suitable for preliminary planning purposes but will not likely 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 that 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 19.0.

18.4.2 *Pit Slope Angles*

Overview

BGC compiled data from available reports, databases, and geological models to support preliminary open pit slope angle design criteria estimates for the proposed open pits of the Snowfield and Brucejack properties. The ultimate Snowfield pit would include a south highwall approximately 1100 m high, which is close to the maximum slope height achieved by any existing open pit mine. In addition, development of the proposed pit requires mining of the “Snowfield Landslide”, a large-scale slope deformation that occurs on the south side of the Mitchell Valley. BGC understands that the open pit mining plan for the Brucejack property includes two main open pits targeting the Bridge Zone and West Zone, with the potential to mine two to four smaller targets. The depths of the open pits vary from less than 200-400 m in the West Zone, and up to approximately 600 m in the Bridge Zone.

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 copper and gold deposits within BC. The data used appears to be appropriate for preliminary or scoping-level designs.

The geotechnical core logging data available for the Snowfield and Brucejack properties 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 rock masses at the two properties. 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 masses within the Snowfield and Brucejack properties have been treated as separate geotechnical units. The majority of the rock mass of the Snowfield property, including the expected rock of the ultimate pit slopes, is estimated to be “fair” (41 < RMR ’76 < 60) and “medium strong” (R3). Some “poor” (21 < RMR ’76 < 40) zones are expected in the near-surface deformation zone of the Snowfield Landslide. The rock mass of the Brucejack property is estimated to range from “fair” to “very good” (RMR ’76 > 81) and the rock is interpreted to be “medium strong” (R3) to “strong” (R4.5). At this stage of study, the rock mass character of the Brucejack property is assumed to be uniform with depth below ground surface.

The Snowfield property 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, the Snowfield property is assumed to represent a single structural domain.

The Brucejack property has rock mass fabrics associated with bedding, foliation, and faults. The strongest concentration of bedding orientations in the data compiled by BGC suggests that the bedding is predominantly steeply east dipping. Foliation (mapped as "schistosity") is found and best developed in sericite altered rocks of the project area; the foliation dips steeply to the north. The Brucejack Fault is also observed at the Brucejack property; however, the orientation of this fault may be different than observed at the Snowfield property. Less prominent faults have also been mapped at the Brucejack property. North, northeast and northwest striking faults are inferred based on surface lineaments and topographic lows. These faults are inferred to be steeply dipping. At the PA stage, the Brucejack property is assumed to represent a single structural domain.

SNOWFIELD OPEN PIT SLOPE DESIGN CRITERIA

The PA-level open pit design criteria developed for the proposed open pit of the Snowfield deposit are presented by design sector in Table 18.21. Design sectors are defined by ranges of slope azimuths and roughly correspond to the expected north, east, south, and west walls of the proposed 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 steeper sections into less steep sections to be completed within the steeper sector.

A double bench (2 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. Controlled blasting of the final walls has been assumed, including buffer rows, trim shots, and/or pre-split blasting due to the double bench configuration. Based on the bench design criteria and consideration of rock mass stability on the overall slope scale, the recommended overall slope design criteria are within the range of those achieved for similar open pit scale designs in other parts of the world (Figure 18.22).

Where overburden is encountered, slopes should be benched with bench heights 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.

BRUCEJACK OPEN PIT SLOPE DESIGN CRITERIA

BGC has estimated preliminary slope design criteria (Table 18.22) for three general sizes of open pit expected at the Brucejack property: <200 m deep, 200–400 m deep, and 400–600 m deep. Designs have not been presented by sectors due to the predominance of steep geological structures and overall limited structural orientation data.

BGC understands that the mined bench height (Bh) may be 10 m for the 'small' pits and 15 m for the 'medium' to 'large' pits (G. Hollett, pers. comm.). At this PA level of design, BGC has assumed that final walls will be double-benched with two mining lifts separating each catch bench, resulting in bench heights of 20 m for the 'small' pits and 30 m for the 'medium' to 'large' pits. A 65° bench face angle (Ba) is assumed for all pits and bench heights. Controlled blasting of the final walls has been assumed, including buffer rows, trim shots, and/or pre-split blasting due to the double bench configuration.

It is noteworthy that the steep geological structures inferred at this stage of study could result in localized toppling failures, if the spacing and continuity of these structural sets is high. Other factors that may increase the likelihood of toppling include poor rock mass quality or high water pressures in the pit walls. Due to the presence of these steep structures, BGC has limited the maximum angle of any bench stack at the interramp scale to 45°. Slopes steeper than this angle are more likely to develop toppling failures due to the wide range of expected dips (60° to 90°) for the geological structures of the site. Depressurization will be required to mitigate toppling if the steeply dipping structures are continuous. If toppling is initiated during mining, industry experience suggests that the bench stack angles may need to be reduced to 38° and the slope depressurization efforts would have to be increased. Based on experience with toppling failures, the most effective means of depressurization will likely be horizontal drains; however; these will have to be supplemented with vertical wells if the groundwater is compartmentalized between the sub-vertical structures.

The recommended interramp slope height (i.e. the height between ramps or geotechnical berms wider than standard berms) has been limited to 210 m for the 'medium' and 'large' pits. Rock mass controlled failure is not likely at these interramp heights, based on the rock mass quality estimated. These ramps or wide benches provide operational flexibility in case mitigations for toppling are needed as well as adequate space for dewatering/depressurization wells.

The recommended overall slope angles at this PA level of study for the 'small', 'medium', and 'large' open pits have been estimated with consideration of the inferred rock mass quality, proposed final wall heights, and bench and interramp scale geometry. The recommended overall slopes assume some residual pore pressures in the final pit walls (Ru = 0.09). Recommended overall slope design criteria are within the range of those achieved for similar open pit scale designs in other parts of the world (Figure 18.22).

Where overburden is encountered, slopes should be benched with bench heights 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.

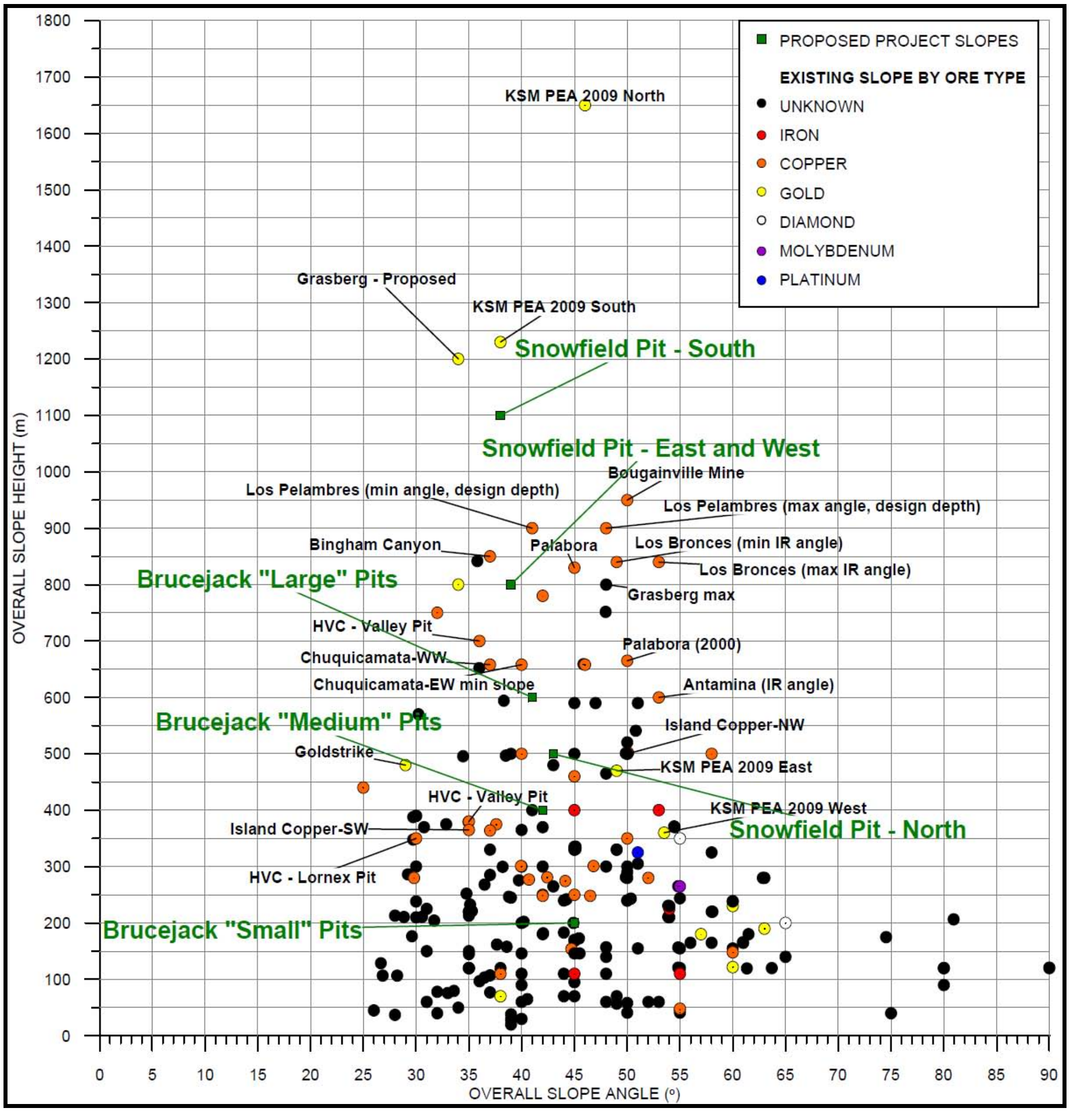
Table 18.21 Preliminary Open Pit Slope Design Criteria for Snowfield

Design Sector	Slope Azimuth Start (°)	Slope Azimuth End (°)	Assumed Overall Slope Height (m)	Bench Height (m)	Bench Face Angle (°)	Catch Bench Width (m)	Maximum Interramp Height (m)	Maximum Overall Slope Angle (°)
SF-357	317	037	510	30	65	18.5	600	43
SF-069	037	102	800	30	65	23.0	600	39
SF-163	102	225	1080	30	65	20.5	600	36
SF-271	225	317	800	30	65	18.0	600	39

Table 18.22 Preliminary Open Pit Slope Design Criteria for Brucejack

Open Pit Size	Assumed Overall Slope Height (m)	Bench Height (m)	Bench Face Angle (°)	Catch Bench Width (m)	Maximum Interramp Height (m)	Maximum Overall Slope Angle (°)
'Small'	<200	20	65	12	200	45
'Medium'	200–400	30	65	19	210	42
'Large'	400–600	30	65	19	210	41

Figure 18.22 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 1,172 Mt of mineralized material based on a mill throughput of 120,000 t/d for the 27-year LOM. During the mine life, mineralized material will be extracted from the Snowfield and Brucejack open pits. The mineralized material will be processed, generating approximately 1,172 Mt of tailings and 1,170 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 four cross-valley tailings dams to be constructed over the mine life. The main tailings dam (Dam #1), located furthest south and approximately 2 km upstream of the confluence with Bowser River, will be constructed in stages to an ultimate crest elevation of 721 masl, with an ultimate dam height of approximately 300 m above centreline. Three additional tailings dams (Dam #2, #3, and #4) must be constructed at the north end of the impoundment during operations to provide containment. The ultimate dam heights for Dams #2, #3, and #4 are 77 m, 42 m, and 8 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 four 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 721 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	1,172 Mt
Mill Throughput	120,000 t/d
Mine Life	27 years
Tailings Dry Density	1.3 t/m³
Total Tailings	1,172 Mt (or 903 Mm³)
Capacity – Starter Dam	2 years tailings (88 Mt or 67 Mm³) + 200-year runoff (61 Mm³)+ 5 Mm³ operating pond + 5 m (freeboard)
Capacity – Ultimate Dam	1,172 Mt (or 903 Mm³) of tailings + 200-year runoff (61 Mm³) + 5 Mm³ operating pond + 5 m (freeboard)
Maximum Design Earthquake	1-in-10,000 earthquake with a peak ground acceleration of 0.2 *g*
Design Flood	store 200-year runoff* = 61 Mm³
Operating Pond	5 Mm³
Spillway Design Capacity	runoff from 24 h Probable Maximum Precipitation
Design Flood Freeboard	5 m above maximum pond level

* 200-year annual runoff volume is approximately 61 Mm³, assuming an 11 km² catchment area for the TSF and a diversion efficiency of 80% for upstream reaches.

MAIN TAILINGS DAM (DAM #1)

The main starter dam will be constructed to a crest elevation of 599 masl (177 m dam height above centerline) and has an approximately 715 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 101 m wide at the base with 1H:7V slopes. Immediately downstream of the core are two 4 m-wide granular filters zones (fine filter and 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-year runoff, an operating pond, plus 5 m of freeboard (emergency freeboard plus the height required to pass the Probable Maximum Flood through the Stage 1 spillway). Figure 18.23 shows the proposed main starter dam in plan. During operations, the main tailings dam will be raised to an ultimate dam crest elevation of 721 masl (300 m high above centreline). Figure 18.24 shows the proposed ultimate dam in plan. A typical cross-section through the ultimate main tailings dam is shown in Figure 18.25.

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Figure 18.23 Scott Creek Proposed TSF – Starter Dam Layout Plan

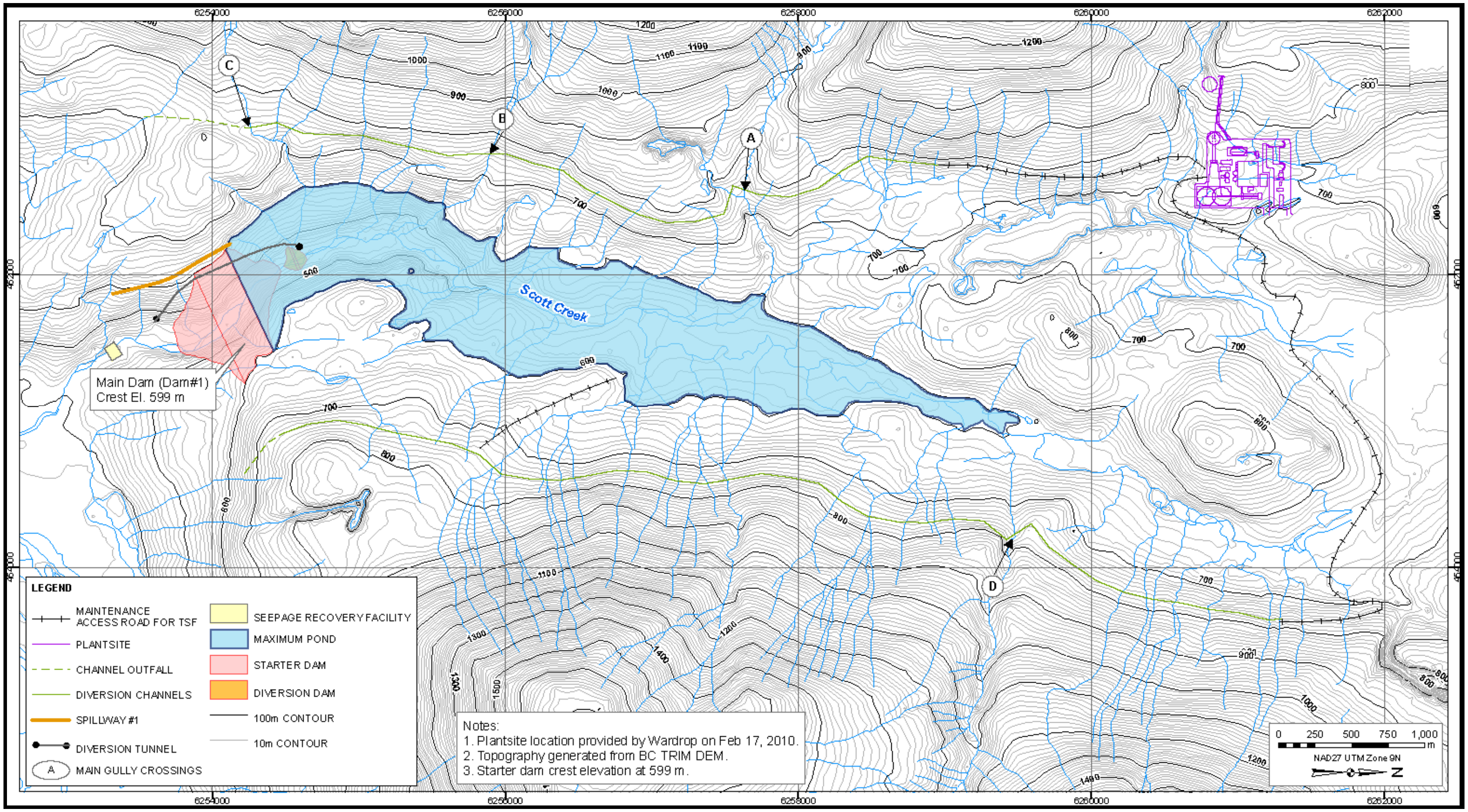


Figure 18.24 Scott Creek Proposed TSF – Ultimate Dam Layout Plan

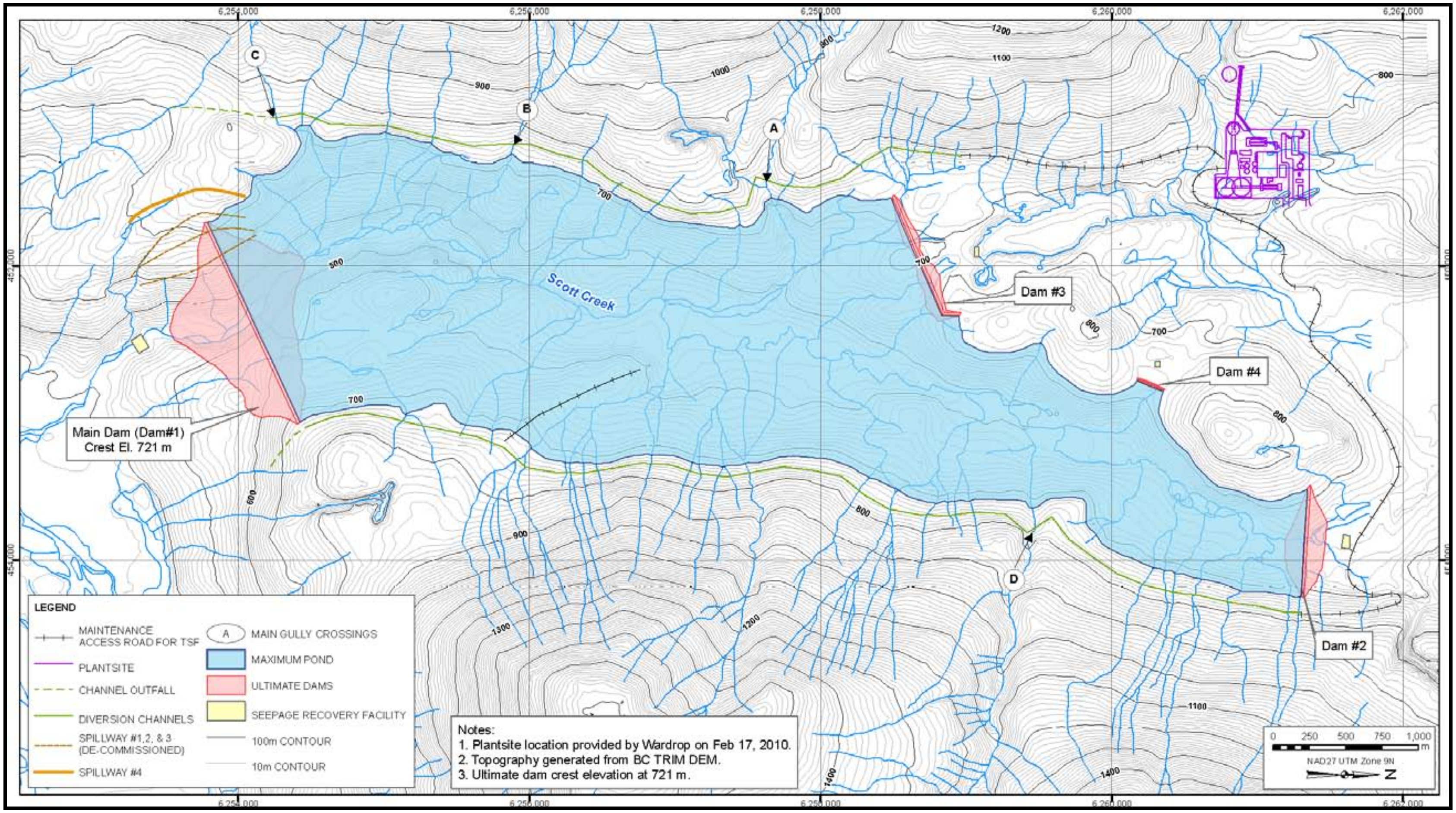
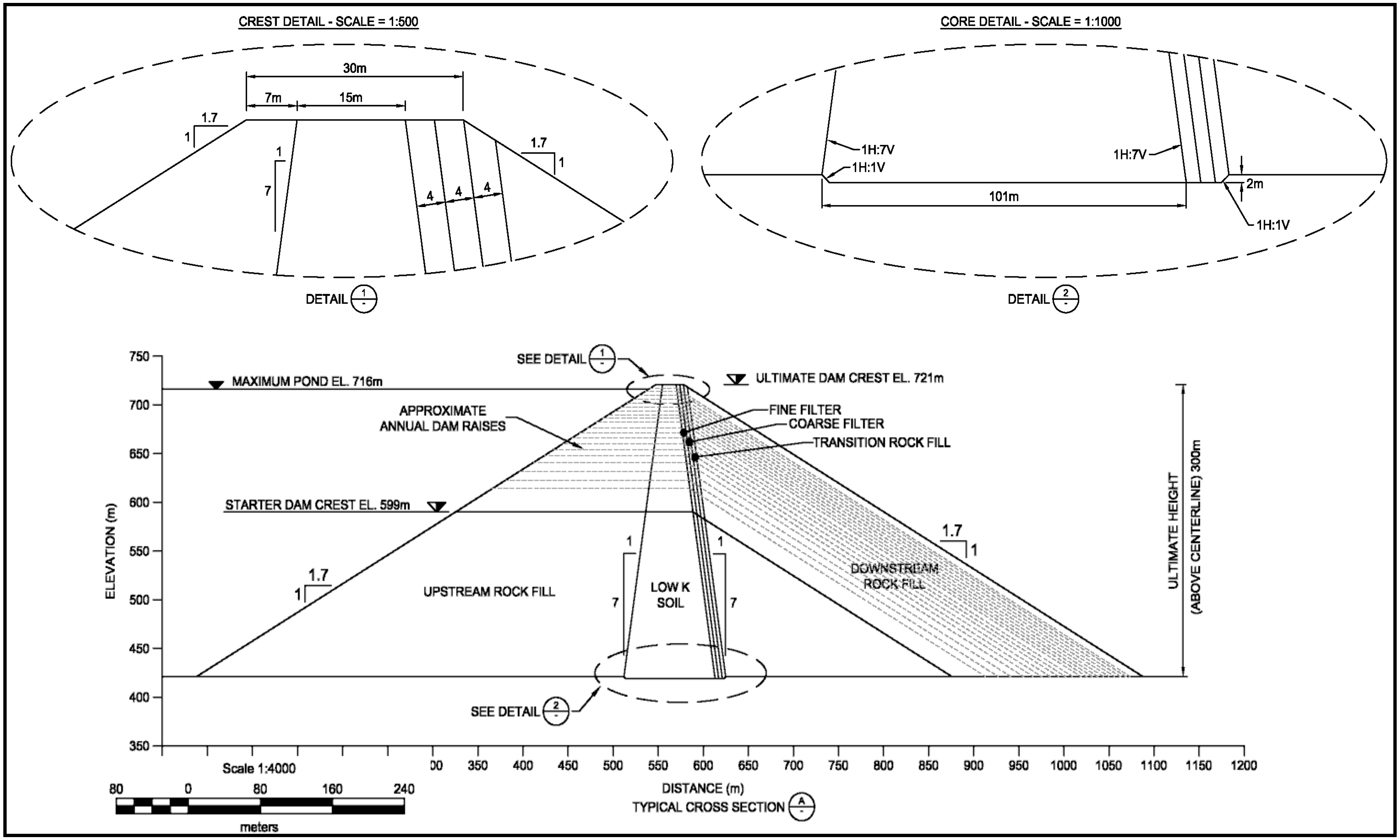


Figure 18.25 Scott Creek Proposed TSF – Typical Main Dam Section



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 (721 masl, with an ultimate dam height of 77 m above centreline). In Year 7, Dam #2 is required to maintain containment. 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. This dam is sized to provide containment of the impoundment; the ultimate dam crest elevation is the same as the main dam (721 masl, with an ultimate dam height of 42 m above centreline). In Year 15, Dam #3 is required to maintain containment. Downstream dam raises have been assumed throughout the remainder of the mine life.

DAM #4

Dam #4, located in the northwest corner of the impoundment (between Dam #2 and Dam #3), is designed as a compacted rockfill dam with a central low-permeability (i.e. clay till) core and filters immediately downstream of the core. This dam is sized to provide containment of the impoundment; the ultimate dam crest elevation is the same as the main dam (721 masl, with an ultimate dam height of 8 m above centreline). In Year 24, Dam #4 is required to maintain containment. Downstream dam raises have been assumed throughout the remainder of the mine life.

DAM FOUNDATIONS

No site specific data regarding the subsurface stratigraphy and engineering characteristics under each of the four dam footprints was available for this study. Based on a review of some high resolution satellite imagery, all four dams are assumed to be founded on 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 four 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.

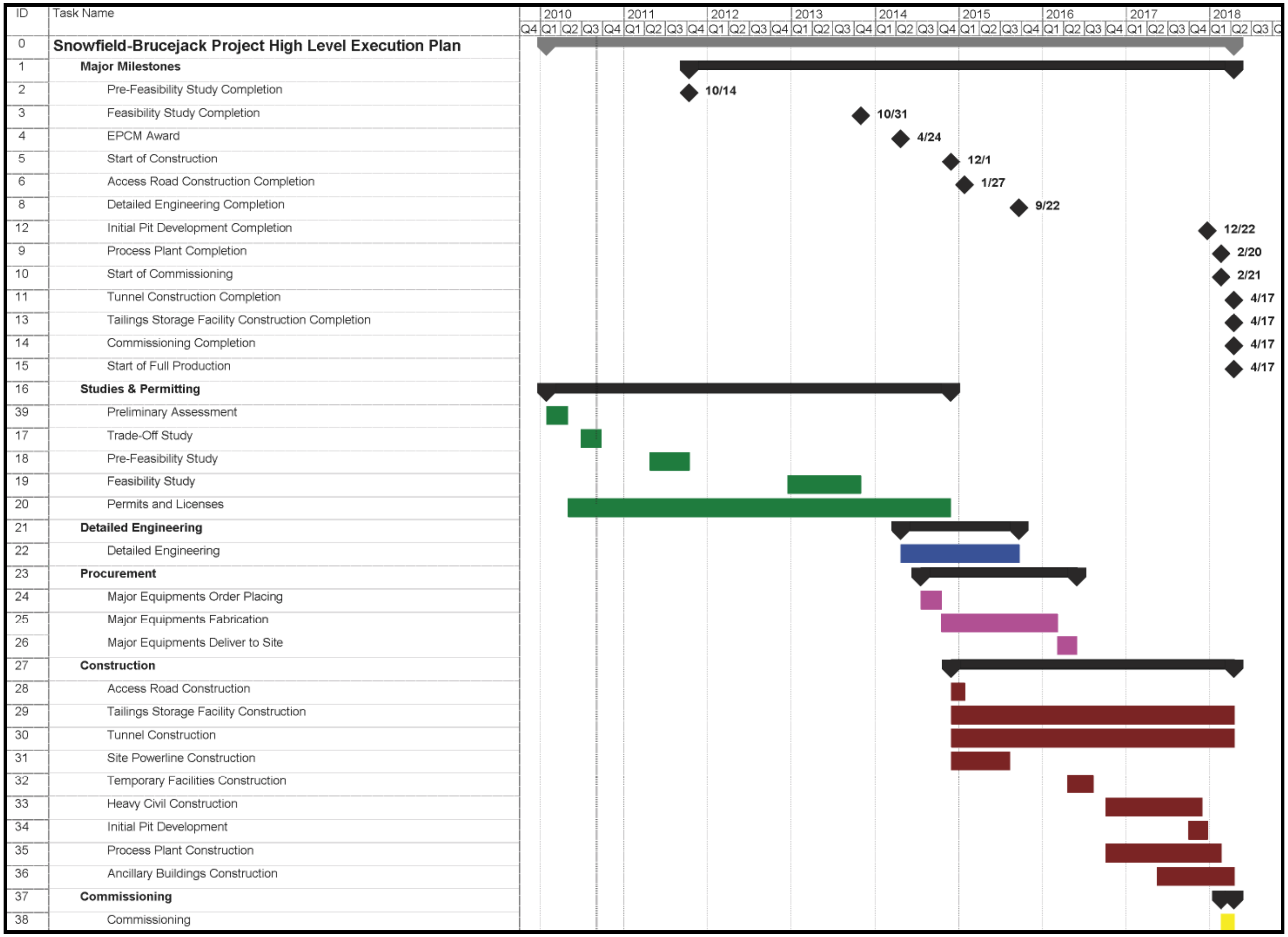
18.5 PROJECT EXECUTION PLAN

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 18.26. 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 18.26 Snowfield-Brucejack High Level Execution Plan



18.6 Markets and Contracts

The final products to be produced by the Snowfield-Brucejack Project are a gold and silver doré, a copper concentrate, and a molybdenum concentrate. The gold and 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 pre-feasibility phase of this project.

18.7 Environmental

18.7.1 Introduction

The Snowfield and Brucejack properties are situated within the Sulphurets District in the Iskut River region. The properties are 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 BC 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 and Brucejack deposits are centred between the Mitchell Glacier to the north and the Knipple 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-Brucejack 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. At the time of writing this report, baseline studies for the Snowfield-Brucejack Project have been initiated.

TERRAIN, SOILS, AND GEOLOGY

The Snowfield-Brucejack Project is located in a rugged area with elevations ranging from about 500 m at the planned TSF 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.

The Brucejack area has been the focus of periodic exploration over the past several decades resulting in the discovery of at least 40 gossanous zones of gold, silver, copper, and molybdenum–bearing quartz/carbonate veining, stockwork and breccia hosted mineralization. Typically, these gossanous showings reflect the weathering of disseminated pyrite in argillic and phyllic alteration zones. The size of these gossans, their tectonic fabric, intensity of alteration, and metallogenesis make them attractive exploration targets (Alldrick and Britton, 1991) and most have been extensively sampled and/or drill tested.

The mineralization on the Brucejack property typically consists of structurally controlled, intrusive related quartz-carbonate, gold-silver bearing veins, stockwork and breccia zones. The veins are hosted within a broad zone of potassium feldspar alteration, overprinted by sericite-quartz-pyrite ± clay. Structural style and alteration geochemistry indicates the deposits were formed in a near surface epithermal style environment.

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 just begun for the project but some prognostications can be based on general knowledge of the region. Although the exploration adit and Brucejack Lake water show no discernable acid rock drainage (ARD) signature, it is probable that elsewhere 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, is likely to be found from processes that have been occurring naturally over a geological time scale. Baseline acid base accounting (ABA) and metal 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; 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 18.27). The station measures wind speed and direction, air temperature and pressure, rainfall, snowfall, relative humidity, solar radiation, net radiation, and snow depth. Another meteorological station has recently been established near the junction of Scott Creek and the Bowser River (August 2010).

Figure 18.27 Meteorological 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. 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 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 masl) to the mine site (~1400 masl) and TSF (~600 masl) assuming an adiabatic lapse rate of -0.6°C per 100 m.

Table 18.24 Average Monthly Climate Data for the Snowfield-Brucejack 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 Scott Creek, drains to the Bowser River. The exception is the Brucejack Lake catchment and parts of the proposed crusher and pit areas which drain into Sulphurets Creek, which flows into the Unuk River toward Alaska. 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. The Unuk enters Alaska within 30 km of the project area and eventually flows through Misty Fjords National Monument in Alaska and finally into Behm Canal on the Pacific coast. Proximity to the coast, relatively high precipitation rates, mountainous terrain, and the presence of glaciers result in large runoff 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 (Figure 18.28) near the confluence with Bowser Lake in October 2009.

Figure 18.28 Hydrological Station on Scott Creek

At the time of writing this report, it is planned to pipe all contact water from the Brucejack catchment to the process plant adjacent to the TSF for use in the process and subsequently to be discharged to the TSF. This approach has the twin advantages of concentrating water treatment at one location and providing hydroelectric power (there is a 650 m elevation difference between Brucejack Lake and the TSF).

Water Quality

Little historical baseline water quality information is available for the Snowfield and Brucejack areas. Silver Standard has initiated an assessment of water and sediment quality and related aquatic ecology. The sparse water quality data collected to date at Brucejack Lake indicate that the concentrations of metals are only slightly elevated above background at the portal of the old exploration adit. Additionally, water quality in the lake itself appears not to be measurably affected so that it is of high enough quality to discharge directly to the environment without treatment. Water quality through the deeper layers of the lake will be established during the ongoing field program.

Naturally-occurring seeps in the nearby mineralized zones, however, 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 that 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 are being assessed as part of the 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-Brucejack 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 intends to 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 (CEA Act). 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.

WILDLIFE

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 Predictive Ecosystem Modelling (PEM) and the field-work-intensive Terrestrial Ecosystem Mapping (TEM) work. Silver Standard will evaluate the potential impacts on species, especially listed species, which could occur in the area. Based on past work on other mining projects in the region, listed species expected to occur in the project area include 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, among 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. Mountain goats 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-Brucejack 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-Brucejack 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-Brucejack 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-Brucejack 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.

18.7.3 SOCIOECONOMIC SETTING

North-western 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. The area 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 Highway 37 (north to south) and Highway 16 (east to west).

The region has suffered from declining populations 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-Brucejack 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 sections provide details on the Highway 16 and Highway 37 corridors, and are 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).

HIGHWAY 16 CORRIDOR

Highway 16 extends from the Prince Rupert port eastwards to Terrace, Hazelton, Smithers, and Prince George. The CNR also follows this corridor. 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 Terrace to Bob Quinn Lake following the Highway 113 and Highway 37 corridors. This line would run parallel to the existing 138 kV transmission line between Terrace and Meziadin Junction 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-Brucejack Project site.

The application for an Environmental Assessment Certificate for the proposed extension of the provincial electricity grid to Bob Quinn Lake has been submitted to the BC Environmental Assessment Office (BCEAO) with BC Hydro acting as the proponent.

18.7.4 WATER SUPPLY, TREATMENT AND RECYCLE

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
- 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 runoff away from disturbed areas. Channels will likely be constructed to collect surface runoff 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 tailings and allow the runoff to be released with little or no treatment.

Water management is described in detail in Section 18.3.

WATER SUPPLY

All flows originating from the mining area will be used in the process. In addition, water from the TSF will be reclaimed as necessary.

Use of mine area water will ensure that any potential acidity is neutralized. It is anticipated that pH adjustment will precipitate dissolved metals to meet environmental standards. In addition, the mine water flow will provide a source of hydroelectric power to the project.

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 RECYCLE STRATEGIES

Process water will be recycled where feasible to reduce the volumes of water released to the environment. Tailings supernatant will be recovered from the TSF using barge mounted pumps and will be 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.

WATER TREATMENT (SUSPENDED SOLIDS REDUCTION)

It is anticipated that water discharged from the TSF will not require treatment to reduce metals concentrations or to adjust pH. The only water quality parameter that may require attention is the suspended solids concentration, which may exceed the 15 mg/L total suspended solids (TSS) discharge criterion within the TSF on occasion.

On average, approximately 27.1 Mm^3/a of water from the TSF will be decanted and the suspended solids concentrations reduced to meet receiving water quality requirements before discharge around Dam #1 to the Bowser River. Discharge will occur only during the ice-free season or approximately six months per year. Decantation, pumping, and treatment will be designed for approximately 6,200 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.

As stated above, it is anticipated that water impounded within the TSF near Dam #1 will meet all receiving water quality requirements with the possible exception of the TSS. Withdrawal of water from the 10 cm-thick surface layer of the impoundment will likely be sufficient to maintain the 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 27 m. Three vertical turbine pumps, approximately 200 kW each, will be mounted on the decant structure and withdraw water from the weir box. Each pump will be capable of lifting 2,100 m^3/h, 30 m over the dam. A floating high density polyethylene pipeline (approximately 40″ DR 17) will convey the decanted water from the pumps over and around the dam.

As the dam is raised, the floating structure and floating pipeline will rise with increasing water level and the mooring lines will be adjusted appropriately. Pumping 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; power will be provided by a submersible power cable run along the walkway or the floating pipeline.

DISCHARGE STRATEGY AND QUALITY

Discharges from the TSF will be controlled, where feasible, to mimic natural flows to minimize adverse effects on local hydrological regimes (e.g. 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 MMER 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.

18.7.5 *Waste Management*

Tailings management

The TSF is designed to isolate the pyrite tailings in a stable subaqueous environment in perpetuity. To ensure that the TSF continuously meets its objectives, Silver Standard will develop and implement a tailings 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 tailings and supernatant to allow estimation of the rate of oxidation and resulting water quality. This information will guide planning for tailings water management.

At closure, the TSF will be configured with minimal pond/wetland areas, 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 discharge to the receiving environment meets permit requirements.

Waste Rock and Overburden Management

The Snowfield-Brucejack Project will potentially generate 1,170 Mt of waste rock over the anticipated LOM. A comprehensive testing program using blast hole cuttings will be established to characterize all rock removed from the pits.

In the Snowfield area, waste rock will be deposited in two dumps, one to the east of the pit and a smaller dump to the southwest. Drainage from these dumps will be collected and pumped to the Snowfield tunnel portal for use in the process. This will ensure that acidity is neutralized.

In the Brucejack area, all the pits will be backfilled with waste rock with the exception of the Bridge Zone pit, which will be flooded at closure. In addition, Brucejack Lake will be filled with waste rock and a dump constructed above it. As for the Snowfield-Brucejack area, all drainage from the dumps will be collected and pumped to the Brucejack tunnel portal from where it will flow to the Snowfield tunnel and to the process plant.

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's 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.

18.7.6 *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-Brucejack 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 CO_2 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.

18.7.7 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-Brucejack 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 has begun baseline studies of the regional project area's atmosphere/climate, surface hydrology, aquatics, water and sediment, limnology and fish habitat, and will initiate comprehensive baseline studies of 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.

Archaeology and Heritage Resources

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

Environmental Assessment Strategy and Scope

The environmental assessment of the Snowfield-Brucejack Project that is required under federal and provincial legislation will focus on the identified VECs to ensure that 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) volcanic activity, 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".

Maintenance of biodiversity is not an isolated effort but an integral part of project planning (mitigations and monitoring), environmental effects analysis and achievement of sustainability goals. This approach will be implemented throughout project development and the environmental assessment process.

Ecosystems and Vegetation

Silver Standard will undertake a systematic mapping of the project area using both PEM and 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.

Environmental Standards

Silver Standard will design, construct, operate, and decommission the Snowfield-Brucejack 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.

18.7.8 CONSULTATION ACTIVITIES

Silver Standard will initiate a consultation program relevant and useful to each consultation group. The proposed Snowfield - Brucejack 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 Canadian Environmental Assessment Agency (CEA Agency) 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-Brucejack Project.

Community engagement and consultation are fundamental to the success of the proposed Snowfield - Brucejack Project and will take place during the project's planning and regulatory review, construction, and operations phases. Prior to beginning the British Columbia Environmental Assessment (BCEA) 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 British Columbia Environmental Assessment Act (BCEAA) and the CEA Act 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, and regional and municipal government agencies as required, with respect to topics such as land and resource management, protected areas, official community plans (OCPs), and environmental and social baseline studies.

Public and Stakeholders

Silver Standard will consult with the public and relevant stakeholder groups [1], including: land tenure holders, trappers, guides, outfitters, recreation and tourism businesses, economic development organizations, businesses and contractors (e.g.

[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-Brucejack Project.

suppliers and service providers), and special interest groups (e.g. environmental, labour, social, health, and recreation).

18.7.9 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-Brucejack 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 as early as the first quarter of 2015.

Some key milestones for Silver Standard are:

- PA: August 2010
- Project Description to BCEAO: on hold; typically one month after PA
- Pre-feasibility Study: on hold; typically two years after submission of the Project Description
- Submission of Environmental Assessment: to be determined; typically four months after completion of the Pre-feasibility Study
- Feasibility Study: to be determined; typically 14 months after completion of the Pre-feasibility Study.

British Columbia Environmental Assessment Act Process

The 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 BCEAA 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-Brucejack Project will thus require an Environmental Assessment Certificate because its proposed production rate exceeds the specified threshold.

Regulatory Review and Approval Schedule Process

The CEA Agency has advised Silver Standard that the Snowfield-Brucejack Project will require an environmental assessment under the CEA Act.

AUTHORIZATIONS REQUIRED

Lists of the major federal and provincial licences, permits, and approvals that are required to construct, operate, decommission, and close the Snowfield-Brucejack 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 due to the large number of minor permits, licences, approvals, consents, 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 BCEAO 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 BCEA 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-Brucejack 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-Brucejack Project. Major stream crossing authorizations will be required from Fisheries and Oceans under the Fisheries Act. Approvals for navigable water crossings will also be required under the Navigable Waters Protection Act by Transport Canada. An explosive factory licence will be required under the Explosives Act by National Resources Canada. MMER, under the Fisheries Act administered by Environment Canada, may require a Schedule 2 amendment 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 BC Authorizations, Licences, and Permits Required to Develop the Snowfield-Brucejack Project

BC Government Permits & Licences	Enabling Legislation
Environmental Assessment Certificate	BCEAA
Permit Approving Work System & Reclamation Program (Mine Site – 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 (Tailings & 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 Federal Approvals and Licences Required to Develop the Snowfield-Brucejack Project

Federal Government Approvals & Licences	Enabling Legislation
CEA Agency Approval	CEA 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 (Section 18.8.2).

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-Brucejack Project was estimated at US$3.47 B with an expected accuracy range of ±35%.

The estimate was developed by Wardrop, with input from the following consultants:

- BGC – material take-offs for tailings management facilities and water management
- Rescan – water turbidity control and environmental costs
- AMC – mine development
- 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	US$ Labour Cost	Material Cost	Construction Equipment Cost	Process Equipment Cost	Total Cost
Direct Works					
Mine Area	182,111,893	112,662,701	140,550,627	278,218,182	713,543,403
Mill Area	126,421,542	133,837,679	12,088,325	311,412,687	583,760,234
Tailings Management, Reclaim Systems, Water Turbidity Control & Closure	99,946,21	202,783,83	150,121,97	20,395,250	473,247,267
Utilities	38,937,350	26,427,327	29,353,164	27,566,480	122,284,321
Site General	105,465,112	53,918,001	63,923,681	5,155,358	228,462,152
Temporary Facilities	6,134,985	86,857,202	138,000	0	93,130,187
Plant Mobile Equipment	146,106	0	0	7,325,261	7,471,367
Direct Works Subtotal	**559,163,201**	**616,486,742**	**396,175,770**	**650,073,218**	**2,221,898,930**
Indirects					
Indirects	6,027,840	699,372,547	0	4,140,000	709,540,388
Contingency	0	454,542,568	0	0	454,542,568
Owner's Costs	0	79,747,019	0	0	79,747,019
Indirects Subtotal	**6,027,840**	**1,233,184,073**	**0**	**4,140,000**	**1,257,453,104**
Total	**565,191,041**	**1,849,670,815**	**396,175,770**	**654,213,218**	**3,465,250,843**

18.9.1 ESTIMATE BASE DATE AND VALIDITY PERIOD EXCHANGE RATE

Wardrop has prepared this preliminary assessment estimate with a base date of Q3-2010. No escalation beyond Q3-2010 was applied to the estimate. The budget quotes used in this estimate were obtained in Q3-2010.

The estimate was prepared in C$ and then converted into US$ using a currency exchange rate of C$1.00 to US$0.92, based on the average exchange rate as issued by the Bank of Canada for the period of August 18, 2006, to August 18, 2010.

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	Tailings 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 tailings pipelines, barge, pumps, some major mining equipment, 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 details for the water turbidity plant.
- Inputs for the mining components were provided by AMC.

- Power supply and distribution costs were developed based on information for electrical components from recent similar projects completed by Wardrop.
- Instrumentation, piping, and HVAC (heating, ventilating, and air conditioning) 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.
- The project development schedule.

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.

Wardrop has assumed the construction man-hours/workweek to be 10 h/d with a 3 wk on/1 wk off rotation, with a land accessible construction camp.

Silver Standard estimated and provided the Owners' costs, including taxes.

18.9.3 Sustaining Capital

Any work that is scheduled to start after Year 1 is generally included in the sustaining capital costs; therefore, the Brucejack pit, crushers, conveyors, some mine equipment, buildings, water turbidity, and pipelines are included in sustaining capital and are not in the capital cost estimate.

18.9.4 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$51.5 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.
- Five percent 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 (-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
- tailings pump barges

The HPGR grinding equipment costs were estimated from recent similar projects. All other mechanical equipment was 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). The cost of the DCS was based on pricing received for similar recent projects.

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 shops
- administration and mine dry
- assay and metallurgical laboratory
- 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.

18.9.5 Permanent Accommodation and Construction Camps

There is a permanent and construction camp included in the estimate. The 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.

18.9.6 Taxes and Duties

The estimate was prepared with taxes (HST, PST, and GST) and duties on materials excluded.

18.9.7 Freight and Logistics

A freight allowance of 8% was provided for materials and the process equipment. The mining mobile equipment costs include freight.

18.9.8 Owner's Costs and Permit Allowances

Silver Standard has provided an allowance of US$66 M for Owner's costs and US$13 M for Owner's risk.

18.9.9 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
- sustaining capital costs
- sunk costs.

18.9.10 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.

18.9.11 CONTINGENCY

The contingency allowance included in the estimate amounts to US$454,542 M.

The @RISK Monte Carlo simulation program by Palisade Corp. (Palisade) was used to generate this contingency allowance. The various inputs for the @RISK program were based on the accuracy level of the information used in the preparation of the cost estimate for each project area.

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.

18.10 OPERATING COST ESTIMATE

18.10.1 SUMMARY

The operating cost for the project is estimated at C$10.20/t milled. The estimate includes operating costs for mining, process, G&A, surface services, and water treatment costs. A total of 617 personnel is the projected full-time labour requirement for the operation, including 309 for mining operations, 228 personnel for process, and 80 personnel for general management, water treatment, and surface services.

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/a of operation. The currency exchange rate used for the estimate is 1.00:0.92 (C$:US$). The operating cost for the Snowfield-Brucejack Project has been estimated in Canadian dollars with an accuracy range of ±35%. The breakdown of the estimated operating cost is presented in both Canadian and US dollars in Table 18.29.

Table 18.29 Overall Operating Cost[1]

Area	Snowfield			Brucejack			Average	
	Staffing	C$/t Milled	US$/t Milled	Staffing	C$/t Milled	US$/t Milled	C$/t Milled	US$/t Milled
Mining	309	2.63[3]	2.42	309	6.64[4]	6.11	3.35	3.08[2]
Processing	228	5.90	5.43	228	5.62	5.17	5.85	5.38
G&A	48	0.67	0.62	48	0.67	0.62	0.67	0.62
Plant Services	21	0.23	0.21	21	0.23	0.21	0.23	0.21
Water Treatment	11	0.10	0.09	11	0.10	0.09	0.10	0.09
Total	**617**	**9.53**	**8.77**	**617**	**13.26**	**12.20**	**10.20**	**9.38**

[1] Tailings dam construction costs are included in sustaining capital costs.
[2] Mining operating cost is an average of the Snowfield and Brucejack operating costs.
[3] Based on the stripping ratio of 0.57:1 for the Snowfield operation.
[4] Based on the stripping ratio of 2.95:1 for the Brucejack operation.

18.10.2 *Mining Operating Cost*

All mining operating costs are shown in Canadian dollars, 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 the Wardrop database and published reports on similar projects in northern BC. Maintenance and major component wear item repair costs were estimated using Western Mining Database and 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 of 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 Owner's 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.

AMC has provided a mine production schedule that shows the ore and waste quantities. The schedule shows tonnages by bench and by mining stage for both the Snowfield and Brucejack mining areas. The material quantities are scheduled in monthly increments from Years 1 to 2 and in yearly increments from Years 3 to 27. AMC has also provided conceptual designs for the pit and waste dump areas.

Mineralized material and waste tonnages were accumulated into yearly increments for the purpose of calculating annual operating costs. The mining sequences and timing of operations for each of the Snowfield and Brucejack areas were also determined based on the schedule provided by AMC.

For the Snowfield area, cycle times for the various haul profiles for each year were obtained from the “Technical Report and Preliminary Assessment on the Snowfield Property” dated June 1, 2010 (Wardrop, 2010). Caterpillar’s® Fleet Production and Cost Analysis (FPC) software (Version 3.05) was used in that report to estimate truck cycle times. 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 Brucejack area consists of several pit zones. To estimate truck cycle times, a simplified approach was taken by assuming a 1,000 m haul profile for ore delivery to the primary crusher and 1,200 m to 2,000 m haul profiles for waste delivery to the waste dump. The average truck speeds for the ore and waste hauls were estimated from the Snowfield cycle times and applied to the simplified haul profiles to obtain the Brucejack truck cycle times.

High material tonnage rates are assigned to Brucejack in Years 3 to 5, Years 8 to 9, and Years 13 to 14. During these years, tonnage rates to be moved from Brucejack range from a low of 516,790 t/d in 150 days to a high of 918,250 t/d in 91 days. In comparison, tonnage rates to be moved from Snowfield range from a low of 130,162 t/d in 232 days, to a high of 158,867 t/d in 274 days. During these years, as many as eight large shovels are theoretically calculated to move the assigned material at Brucejack.

The haul truck productivity and operating hours for each of the Snowfield and Brucejack mining areas were calculated separately in a spreadsheet scheduling program based on the total equipment cycle times. The same procedure was followed in estimating the shovel and drill productivities for each mining area.

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 current production schedule for the Brucejack and Snowfield mining areas promotes significant swings in the equipment and manpower requirements from Years 2 to 16. Stopping and starting operations from one mining area to another adversely affects the practicality of the operations and will increase estimated operating costs. It is assumed that contract labour will be used to maintain hourly labour requirements realized at the peak quantity of mineralized material mined and handled at Brucejack. It is also assumed that the financial impact of the employment of contract labour will be addressed in the next level of project assessment.

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	15
Haul Truck Operator	57
Drill Operator	13
Dozer Operator	31
Grader Operator	18
Water/Sand Truck Operator	-
Dispatch Operator	4
Equipment Trainee	4
Mine Labourer	4
Mine Maintenance	
Mechanic - Heavy Duty	32
Mechanic - Light Duty	16
Electrician	10
Serviceman	16
Welder	16
Tireman	8
Maintenance Labour/Trainee	5
Total Hourly	**259**
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	2
Maintenance Foreman	4
Mechanical Foreman	4
Electrical Foreman	2
Maintenance Planner	2
Maintenance Clerk	1

table continues…

Labour Summary	No.
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	**50**
Total Overall Personnel	**309**

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

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. An average unit operating cost was estimated for the Snowfield-Brucejack Project.

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	0.09	0.04
Blasting	0.36	0.18
Loading	0.20	0.10
Hauling	1.41	0.71
Support Equipment	0.22	0.11
Dewatering	0.01	0.01
Labour	1.05	0.53
Total Mining Cost	**3.35**	**1.68**

18.10.3 Plant Operating Costs

The estimated process operating cost is approximately C$5.90/t milled for the Snowfield mineralization, and C$5.62/t milled for the Brucejack mineralization. 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 BC
- 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	Snowfield		Brucejack	
		C$/a	C$/t Milled	C$/a*	C$/t Milled
Operating Staff	33	3,981,000	0.091	3,981,000	0.091
Operating Labour	112	10,026,000	0.229	10,026,000	0.229
Maintenance	83	8,301,000	0.190	8,301,000	0.190
Sub-total Labour	**228**	**22,308,000**	**0.509**	**22,308,000**	**0.509**
Metal Consumables		66,444,000	1.517	65,622,000	1.489
Reagent Consumables		94,003,000	2.146	83,264,000	1.901
Maintenance Supplies		26,630,000	0.608	26,630,000	0.608
Operating Supplies		2,530,000	0.058	2,530,000	0.058
Sub-total Consumables and Supplies		**189,606,000**	**4.329**	178,046,000	4.065
Power Supply		46,416,000	1.060	45,668,000	1.043
Sub-total Power		**46,416,000**	**1.060**	**45,668,000**	**1.043**
Total (Process)		**258,331,000**	**5.898**	**246,022,000**	**5.617**

* based on 43.8 Mt/a; annual process operating cost will change depending on whether the mineralized material is coming from Snowfield or Brucejack

The annual process operating cost for the project will change depending on whether the mineralized material being processed is coming from the Snowfield deposit or the Brucejack deposit, as per the production schedule developed by AMC.

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 for the Snowfield mineralization, and C$1.49/t milled for the Brucejack mineralization. The metal consumables include mill and crusher liners and grinding media.

The estimated reagent cost is C$2.15/t milled for the Snowfield mineralization, and C$1.90/t milled for the Brucejack mineralization. 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 a unit electric energy price of C$0.046/kWh and an average power requirement of 125 MW for the Snowfield mineralization, or 123 MW for the Brucejack mineralization.

The operating cost breakdown for the flotation plant, cyanide leach plant, and tailings delivery and reclaim water are detailed below.

CRUSHING, GRINDING, FLOTATION, AND CONCENTRATE DEWATERING

The estimated operating cost for the flotation plant including crushing, grinding, copper pyrite and molybdenum flotation, and flotation concentrate dewatering is shown in Table 18.35. The total cost for the process is estimated at C$4.24/t milled for the Snowfield mineralization, and C$3.88/t milled for the Brucejack mineralization. A total of 161 personnel are required to operate the plant.

Table 18.35 Comminution, Flotation, & Concentrate Dewatering Operating Cost

Description	Staffing	Snowfield		Brucejack	
		C$/a*	**C$/t Milled**	**C$/a***	**C$/t Milled**
Labour					
Operating Staff	22	2,566,000	0.059	2,566,000	0.059
Operating Labour	72	6,473,000	0.148	6,473,000	0.148
Maintenance	67	6,749,000	0.154	6,749,000	0.154
Sub-total Labour	**161**	**15,788,000**	**0.360**	**15,788,000**	**0.360**

table continues…

Description	Staffing	Snowfield		Brucejack	
		C$/a*	C$/t Milled	C$/a*	C$/t Milled
Supplies					
Metal Consumables		66,444,000	1.517	65,622,000	1.498
Reagent Consumables		33,980,000	0.776	19,951,000	0.456
Maintenance Supplies		23,397,000	0.534	23,397,000	0.534
Operating Supplies		2,255,000	0.051	2,255,000	0.051
Power		43,653,000	0.997	42,721,000	0.975
Sub-total Supplies		**169,729,000**	**3.875**	**153,946,000**	**3.515**
Total	**161**	**185,517,000**	**4.236**	**169,735,000**	**3.875**

* based on 43.8 Mt/a; annual process operating cost will change depending on whether the mineralized material is coming from Snowfield or Brucejack.

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 for Snowfield. The cyanide leaching operating costs for Brucejack are estimated at C$1.64/t milled, or C$11.15/t leach feed (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 for both Snowfield and Brucejack. The reagent consumption is the major cost, estimated to be C$1.37/t milled for Snowfield, or C$1.45/t milled for Brucejack. The estimated maintenance supply cost is C$0.03/t milled for both Snowfield and Brucejack.

Table 18.36 Gold Leach, Gold Recovery, and Cyanide Handling Operating Cost

Description	Staffing	Snowfield			Brucejack		
		C$/a*	C$/t CIL Feed	C$/t Milled	C$/a*	C$/t CIL Feed	C$/t Milled
Operating Staff	11	1,415,000	0.220	0.032	1,415,000	0.220	0.032
Operating Labour	32	2,842,000	0.441	0.065	2,842,000	0.441	0.065
Maintenance	16	1,552,200	0.241	0.035	1,552,200	0.241	0.035
Sub-total Labour	**59**	**5,809,000**	**0.901**	**0.133**	**5,809,000**	**0.901**	**0.133**
Reagent Consumables		60,022,000	9.315	1.370	63,313,000	9.825	1.446
Maintenance Supplies		1,498,000	0.233	0.034	1,498,000	0.233	0.034
Operating Supplies		135,000	0.021	0.003	135,000	0.021	0.003
Power Supply		891,000	0.138	0.020	1,074,000	0.167	0.025
Sub-total Supplies		**62,547,000**	**9.706**	**1.428**	**66,021,000**	**10.246**	**1.507**
Total	**59**	**68,355,000**	**10.608**	**1.561**	**71,830,000**	**11.147**	**1.640**

* based on 43.8 M t/a; annual process operating cost will change depending on whether the mineralized material is coming from Snowfield or Brucejack.

TAILINGS DELIVERY AND WATER RECLAIM

The total operating cost for tailings delivery and water reclaim is estimated at C$0.10/t milled for both Snowfield and Brucejack. 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 – Tailings 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$0.23/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 G&A Operating Cost

G&A	Staffing	Total Cost (C$/a)	Unit Cost (C$/t Milled)
G&A Labour			
General & Administrative	36	3,773,000	0.086
G&A Hourly Personnel	12	1,080,000	0.025
Sub-total Labour	**48**	**4,853,000**	**0.111**
G&A Expenses			
General Office Expense		250,000	0.006
Computer Supplies incl. 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		200,000	0.005
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		100,000	0.002
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	Labour	Total Cost (C$/a)	Unit Cost (C$/t Milled)
Surface Service Personnel	21	1,989,000	0.045
Surface Service Expenses		8,075,000	0.184
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		3,000,000	0.068
Avalanche Control		1,000,000	0.023
Off-site Operation Expenses		200,000	0.005
Total	**21**	**10,064,000**	**0.230**

18.11 Financial Analysis

18.11.1 Introduction

An economic evaluation of the Snowfield-Brucejack Project was prepared by Wardrop based on a pre-tax financial model. For the 27-year LOM and 1,172 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 12.4% IRR
- 5.3-year payback on US$3.5 billion initial capital
- US$2.3 billion NPV at 5% discount rate.

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

- silver – US$14.50/oz
- gold – US$878/oz
- copper – US$2.95/lb
- molybdenum – US$17.00/lb
- rhenium – US$7,809/kg
- exchange rate – 0.92:1.00 (US$: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 Snowfield and Brucejack cash flows are consolidated into one financial model. The production schedules have been incorporated into the pre-tax financial model to develop annual recovered metal production. Market prices for gold, silver, copper, molybdenum, and rhenium 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 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 tailings embankment construction.

Working capital has been calculated based on Q1 Year 1 of the 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 18.29. Metal production quantities are presented in Table 18.40.

Figure 18.29 Pre-tax Cash Flow

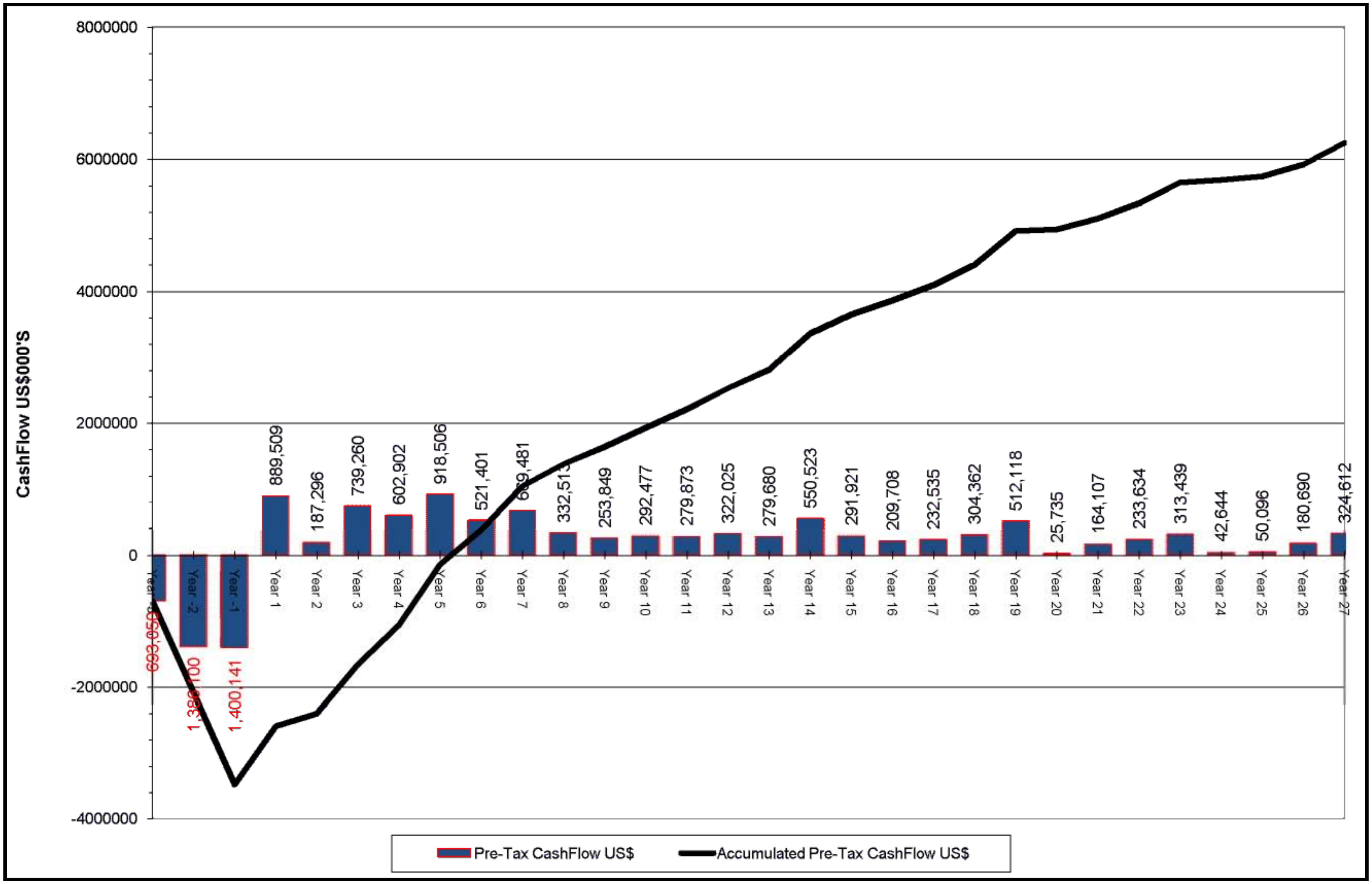


Table 18.40 Metal Production Quantities

	Average Annual Production		Total Production	
Metal	**Years 1 to 8**	**LOM**	**Years 1 to 8**	**LOM**
Gold (000 oz)	960	700	7,679	18,910
Silver (000 oz)	7,855	4,162	62,838	112,364
Copper (000 lb)	39,531	44,582	316,245	1,203,715
Molybdenum (000 lb)	3,514	3,668	28,115	99,042
Rhenium (kg)	9,379	9,011	75,029	243,305

METAL PRICES SCENARIOS

The financial outcome for the three metal price scenarios has been tabulated for NPV, IRR, and payback of capital. A 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 August 27, 2010.

The base case metal prices are based on Wardrop's adopted consensus forecast metal prices from the Energy Metals Consensus Forecast (EMCF). EMCF is published by Consensus Economics Inc. (Consensus Economics) of London. Consensus Economics provide quarterly forecasts (the EMCF) for a variety of metals prices based on an average price from long term projections of 20 analysts representing international banks. The summary of the project economic evaluation is presented in Table 18.41.

Table 18.41 Summary of Pre-tax NPV, IRR, and Payback by Metal Price Scenario

Economic Returns	Unit	Base Case	Alternate Case	Spot Prices*
Net Cash Flow	M US$	6,246	3,503	12,949
NPV at 5.0% Discount Rate	M US$	2,302	881	5,951
Project IRR	%	12.4	8.2	21.7
Payback	years	5.3	6.8	3.5
Exchange Rate	US$:C$	0.92	0.92	0.948
Mine Life	years	27	27	27
Au Price	US$/oz	878	800	1,235
Ag Price	US$/oz	14.50	12.55	19.03
Mo	US$/lb	17.00	13.91	15.88
Re	US$/kg	7,811	6,613	5,311
Cu	US$/lb	2.95	2.35	3.26

* spot prices as at August 27, 2010.

ROYALTIES

There are no royalties on the Snowfield property; however, there are royalties for gold and silver produced from the Brucejack deposit. "Royalty" means the amount payable by Silver Standard, calculated as 1.2% of the NSR, with the following exemptions:

- gold: the first 503,386 oz produced from the Brucejack property
- silver: the first 17,907,080 oz produced from the Brucejack property.

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 and Silver– pay 99.8% of gold 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.

- Rhenium:
  - There will be a 60% deduction from the recovered rhenium by the smelter; therefore, the mine will receive 40% of the recovered rhenium revenue.

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 molybdenum concentrate that 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 will be 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
- rhenium 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 18.42 and Figure 18.30 and IRR in Table 18.43 and Figure 18.31. The project NPV (at 5% discount rate) is most sensitive to the gold price, gold grade, and exchange rate.

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

Table 18.42 Output Variable Values for NPV

	Units	NPV Sensitivity				
		-20.0%	-10.0%	0.0%	+10.0%	+20.0%
Cu Price	US$ M	1,996	2,149	2,302	2,455	2,607
Au Price	US$ M	548	1,425	2,302	3,178	4,055
Ag Price	US$ M	2,113	2,207	2,302	2,396	2,490
Mo Price	US$ M	2,142	2,222	2,302	2,381	2,461
Re Price	US$ M	2,225	2,263	2,302	2,340	2,378
Exchange Rate	US$ M	3,582	2,942	2,302	1,661	1,021
Cu Grade	US$ M	1,750	2,025	2,302	2,580	2,860
Au Grade	US$ M	461	1,380	2,302	3,226	4,166
Ag Grade	US$ M	2,136	2,219	2,302	2,384	2,467
Mo Grade	US$ M	2,157	2,229	2,302	2,374	2,446
Operating Cost	US$ M	3,366	2,834	2,302	1,769	1,237
Capital Cost	US$ M	2,956	2,629	2,302	1,975	1,648

Figure 18.30 NPV 5% Sensitivity Analysis

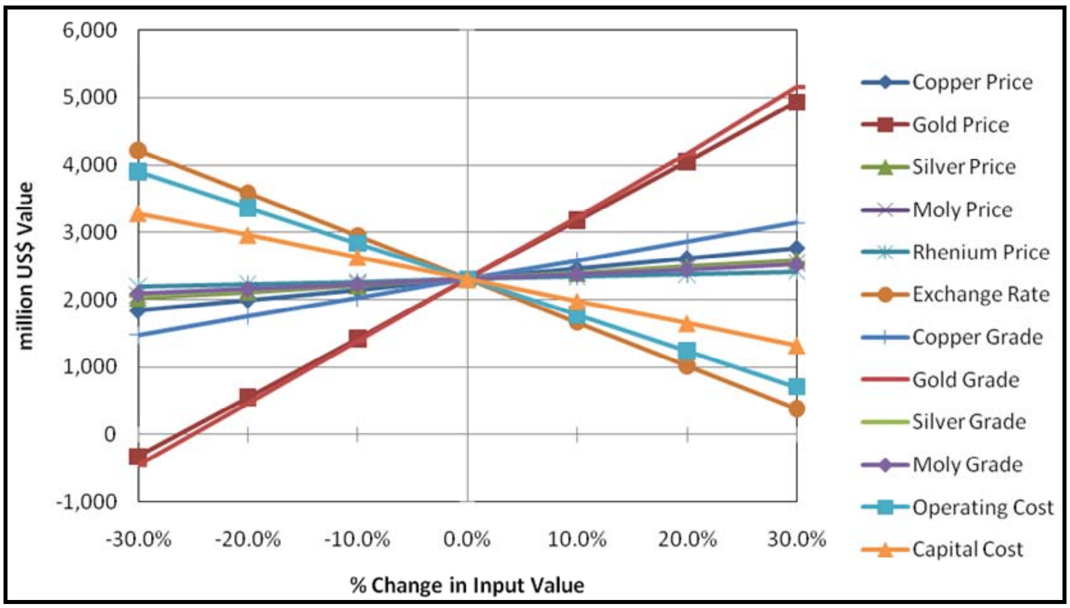
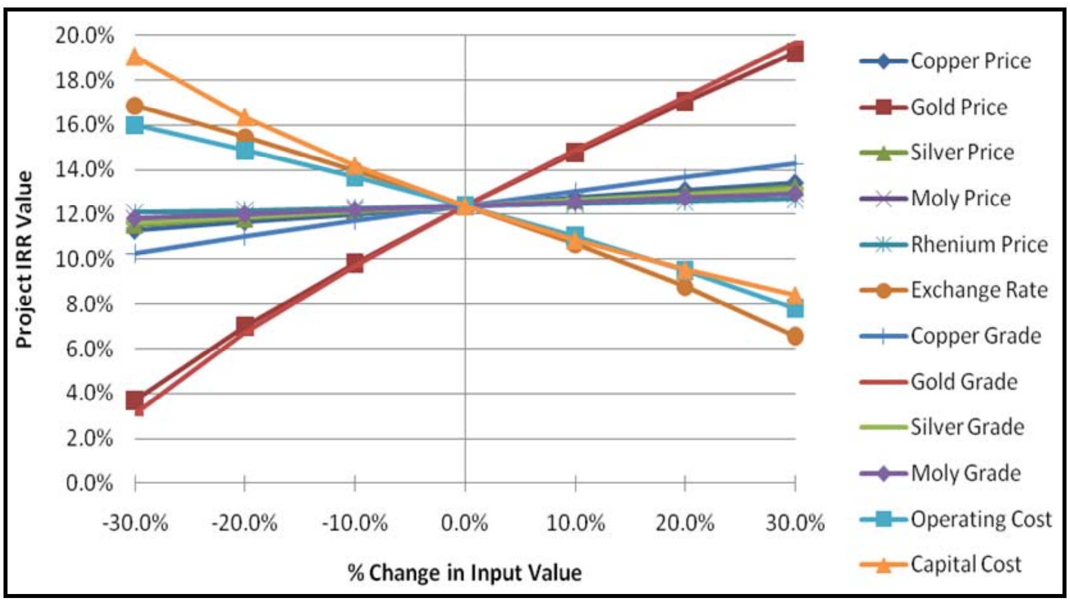


Table 18.43 Output Variable Values for Project IRR

	IRR Sensitivity (%)				
	-20.0%	-10.0%	0.0	+10.0%	+20.0%
Cu Price	11.7	12.0	12.4	12.7	13.1
AU Price	7.0	9.8	12.4	14.8	17.0
Ag Price	11.8	12.1	12.4	12.7	12.9
Mo Price	12.0	12.2	12.4	12.6	12.8
Re Price	12.2	12.3	12.4	12.5	12.6
Exchange Rate	15.5	14.0	12.4	10.7	8.8
Cu Grade	11.0	11.7	12.4	13.0	13.7
Au Grade	6.7	9.7	12.4	14.9	17.2
Ag Grade	11.9	12.1	12.4	12.6	12.9
Mo Grade	12.0	12.2	12.4	12.6	12.7
Operating Cost	14.9	13.7	12.4	11.0	9.5
Capital Cost	16.4	14.2	12.4	10.9	9.6

Figure 18.31 IRR Sensitivity Analysis



19.0 CONCLUSIONS AND RECOMMENDATIONS

19.1 CONCLUSIONS

Based on the results of the PA, Wardrop recommends that Silver Standard proceed with the next phase of the project, a Pre-feasibility Study, 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 are provided in this section for the next phase of design (pre-feasibility level) to confirm the geotechnical assumptions for the TSF, waste dumps, and water balances.

The following is a list of recommendations for further open pit design studies:

- Five to eight geotechnical core holes should be drilled at each property, 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 in each property.
- 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 estimates of hydraulic conductivity of the Snowfield and Brucejack rock masses.
- 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.
- 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.
- Utilizing the above data, the PEA design of the open pit slopes should be reassessed.

- A preliminary review and identification of geohazards in and around the pit area via aerial photographs, satellite imagery, and field mapping should be conducted.

The following is a list of recommendations for the waste dump design:

- Geotechnical and hydrogeological site investigations will need to be completed (i.e. mapping, drilling, geophysics, and/or test pit excavations).
- The extent and thickness of all glaciers and icefields in the vicinity of the proposed East and Southwest dumps must be determined.
- Geotechnical stability analyses of the dumps should be conducted once the site investigations have been completed.

The following is a list of recommendations for the 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 the Snowfield and Brucejack open pit and waste dump areas 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 be needed to 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 four 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 and Brucejack deposits.
- 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.

The following is a list of recommendations for the TSF, pit, and waste dump water balances:

- Existing climate and hydrometric stations (i.e. Brucejack Lake, Scott Creek) must continue to be monitored and maintained with an appropriate level of quality control.
- A precipitation gauge is also recommended for the Scott Creek valley to confirm water balance assumptions and peak flow estimates for the TSF.
- Automated hydrometric gauges should be installed on Mitchell Creek and Sulphurets Creek.
- Snowpack surveys should be conducted throughout the Scott Creek and Sulphurets Creek watersheds prior to snowmelt to quantify snowfall distribution and confirm precipitation measurements at the Brucejack Lake climate station.
- Pit dewatering groundwater should be of sufficient volume to provide a freshwater source for the process plant during the winter months (300 m^3/h). However, this assumption needs to be evaluated further during the next level of engineering design.
- 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 scheduling and cut-off grade optimization is required to optimize the ore blending strategy between the Snowfield and Brucejack projects, in order to maximize the NPV. This should include the effects of the timing of the capital expenditure required to bring Brucejack into production.
- Optimization of the size of the shovel and truck fleet should be conducted during the next phase of study. This should include a dilution and mining recovery evaluation comparing reductions in operating cost using large equipment against the downstream effects on processing and revenue. This would include a trade-off between the cost of under-utilizing the excavator fleet against smaller equipment that could be fully utilized and transported between the two operations. Finally, given the high electricity costs, an economic evaluation should also be undertaken on the use of electric-driven shovels and drills versus the hydraulic equivalents.
- Detailed studies on the location of the crushers should be undertaken in an attempt to reduce operating costs.
- A detailed hydrogeological evaluation of the pit areas should be conducted in order to determine the design of overall dewatering systems in and around the open pits.
- 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 should be conducted to compare the efficiency and cost-effectiveness of owner-run blasting versus full-contractor blasting.
- An economic evaluation of using a mining contractor versus the owner mining scenario should be completed.
- An investigation should be conducted of all possibilities for expansion to the Snowfield waste dumps.
- The Brucejack waste dump should be evaluated for possible reductions to haulage distances and redesigned to create a single water collection point to the west of the dump.

- For the Bridge Zone pit and Snowfield waste dumps, research needs to be completed on the safest and most effective method for removing the ice cover, including a load bearing analysis to determine if truck haulage across the ice is viable as has been currently assumed for the Bridge Zone pit ice removal.
- Complete an evaluation of potential acid generation from the waste dumps and prepare a plan to minimize acid runoff.

19.2.5 PROCESS AND METALLURGY

The following are recommendations for the next phase of study:

- Further testwork is required to confirm the previous testwork findings, optimize the process flowsheet, and investigate metallurgical performances. The testwork should be conducted on representative samples and fresh drill core samples. 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
  - gold and silver cyanidation, including cyanide solution handling
  - gravity concentration should be further optimized, especially on the Brucejack mineralization
  - ancillary tests, including settling and filtration tests
  - copper recovery by hydrometallurgical processes
  - pilot plant scale tests.
- Optimization of primary grinding circuit should be conducted, including a SAG mill/ball mill/pebble crushing (SABC) circuit.
- 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 on the Snowfield-Brucejack Project", is September 10, 2010.

Signed,

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