Report to:

PRETIVM

PRETIVM RESOURCES INC.

Technical Report and Preliminary Economic Assessment of the Brucejack Project

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Report to:

PRETIVM RESOURCES INC.

TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT OF THE BRUCEJACK PROJECT

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NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Pretivm Resources Inc. (Pretivm) by Wardrop, A Tetra Tech Company (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 Pretivm's contract with Wardrop, P&E, BGC, Rescan, and AMC. This contract permits Pretivm to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, *Standards of Disclosure for Mineral Projects*.

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GLOSSARY

UNITS OF MEASURE

Above mean sea level	amsl
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
Dollar (American)	US$
Dollar (Canadian)	Cdn$
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 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
Cominco Engineering Services Ltd.	CESL
Consensus Economics Inc.	Consensus Economics
Copper-equivalent	CuEq
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	AuEq
Indicator Kriging	IK
Inductively coupled plasma atomic emission spectroscopy	ICP-AES
Inductively coupled plasma	ICP
Internal rate of return	IRR
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
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
Pretivm Resources Inc.	Pretivm
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

Pretivm Resources Inc. (Pretivm) commissioned Wardrop Engineering Inc, A Tetra Tech Company (Wardrop), to complete a preliminary economic assessment (PEA) of the Brucejack Project.

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

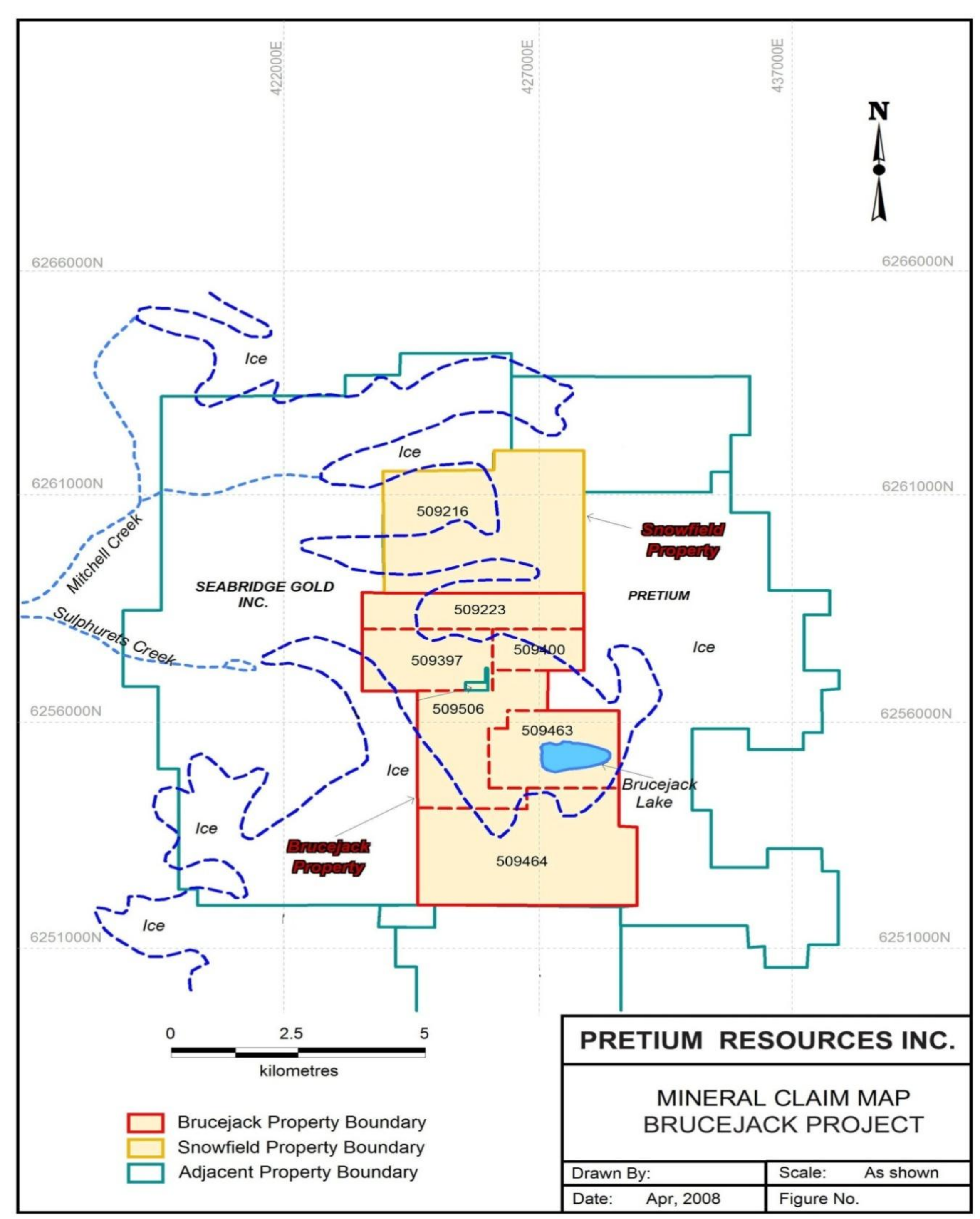
1.2 PROPERTY DESCRIPTION AND LOCATION

The 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, British Columbia (BC) (Figure 1.1).

The Brucejack Project consists of six mineral claims totalling 3,199.28 ha in area; all claims are in good standing until January 31, 2021.

Figure 1.1 Regional Map of BC with Location of Brucejack Project

Figure 1.2 Claim Map of the Brucejack Project



Note: Modified after Blanchflower, 2008.

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, 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, Snowfield and Brucejack deposits.

In 1999, Silver Standard acquired Newhawk, which owned a 60% interest in the Bruceside joint venture (owner of the Brucejack Project) and 100% interest in the Snowfield Project. In 2001, Silver Standard acquired from Blackhawk, the remaining 40% interest in the Bruceside joint venture (and 100% interest in the Brucejack Project). Silver Standard has drilled in excess of 51,200 m in 109 holes on the Brucejack Project (2009 and 2010).

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 June to mid-October.

There are no local resources other than abundant water. The nearest infrastructure is Stewart, BC, which has a minimum of supplies and personnel. Stewart is the most northerly ice-free shipping port in North America and is accessible to store and ship concentrates. Such material is currently being shipped from the Wolverine and Huckleberry mines via this terminal.

1.5 GEOLOGICAL SETTING

The mineralization on the Brucejack Project 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 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.

The deposits on the Brucejack Project are classed as epithermal deposits of gold (±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.

While deposits such as Kerr, and Mitchell are probably best described as gold-enriched copper porphyry systems, most (if not all) of the mineralization on the Brucejack Property has been classified as an epithermal gold-silver-copper, 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) or the mesothermal deposit type.

1.6 RESOURCE ESTIMATE

The Brucejack Mineral Resource estimate encompasses nine distinct modelled mineralization zones, namely the West Zone, West Zone Footwall Zone, Shore Zone, Gossan Hill Zone, Galena Hill Zone, SG Zone, VOK Zone, Bridge Zone and Bridge Zone Halo.

All mineral resources were reported against a 0.30 g/t AuEq cut-off, as constrained within the optimized pit shell (Table 1.1)

Table 1.1 Brucejack Estimated Mineral Resources Based on a Cut-Off Grade of 0.30 g/t AuEq(1)(2)(3)

Category	Tonnes (M)	Gold (g/t)	Silver (g/t)	Contained(3)	
				Gold ('000 oz)	Silver ('000 oz)
Measured	11.7	2.25	75.56	846	28,423
Indicated	285.3	0.80	9.57	7,338	87,782
M+I	297.0	0.86	12.17	8,184	116,205
Inferred	542.5	0.72	8.67	12,558	151,220

(1) Mineral resources for the February 2011 estimate are defined within a Whittle™ optimized pit shell that incorporates project metal recoveries, estimated operating costs and metals price assumptions. Parameters used in the estimate include metals prices (and respective recoveries) of US$1,025/oz Au (71%) and US$16.60/oz Ag (70%). The pit optimization utilized the following cost parameters: mining US$1.75/t, Processing US$6.10/t and G&A US$0.90/t along with pit slopes of 45 degrees. 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, marketing or other relevant issues. The mineral resources in this news release were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM)"CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines" prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.

(2) The quantity and grade of reported Inferred resources in this estimation are uncertain in nature. There has been insufficient exploration to define these Inferred resources as either an Indicated or Measured mineral resource, and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category.

(3) Contained metal may differ due to rounding.

Table 1.2 Brucejack 5.00 g/t AuEq Mineral Resource Grade and Tonnage Estimate(1)(2)(3)(4)

Category	Tonnes (M)	Au (g/t)	Ag (g/t)	Contained(3)	
				Au (oz x '000)	Ag (oz x '000)
Measured	1.947	7.95	241.25	498	15,102
Indicated	1.722	7.33	123.19	406	6,820
M+I	3.669	7.66	185.84	903	21,922
Inferred	4.707	12.54	49.24	1,898	7,452

(1) (2) (3) See footnotes to Table 1.1.

(4) This high-grade resource estimate is a subset of the bulk-tonnage resource estimate and as such is included within the bulk-tonnage resource estimate and is not in addition to the bulk-tonnage resource estimate.

1.7 MINING OPERATIONS

The Brucejack Project is planned to commence as an underground operation to be followed by open pit mining once the underground inventory is depleted. The underground mine will operate for 13 years that includes one year of pre-production development, producing a total of 6.2 Mt of mineralised material. The open pit mine will commence in the final year of the underground operation and operate for five years, producing 2.7 Mt of mineralised material for a total mine production of 8.4 Mt at a nominal rate of 1,500 t/d.

The underground mine design and inventory for both the WZ and VOK lodes are based on a NSR cut-off of $180/t of mineralized material. The open pit mine design and inventory for the SG, GO, GA, and BZ lodes are based on a NSR cut-off of $75/t of mineralized material.

The underground operation is based on conventional rubber tired, diesel powered mobile equipment with loader mucking and truck haulage material handling via a decline ramp system. Mineralized material production will be achieved through implementation of the longhole open stoping (LHOS) method with a combination of rock and paste backfill.

The two lodes targeted for underground mining are the West Zone and VOK Zone.

The open pit operation is based on utilization of a single 4.0 m^3 bucket capacity backhoe excavator loading 45-t trucks with 102 to 115 mm blasthole drills and related support equipment. Benches are assumed to be drilled and blasted in 5 m benches, with the mineralized and waste material mined in two 2.5 m operating benches, adjusted for blast heave as necessary. This configuration allows for selective mining of the mineralized material.

Four lodes have been targeted for open pit mining, SG Zone, Gossan Hill, Galena Hill and Bridge Zone, resulting in four separate pits. The deepest pit is the Bridge Zone with a maximum highwall of 105 m.

1.8 METALLURGICAL TESTWORK REVIEW

Several testing programs were conducted to investigate the metallurgical performance of the mineralized samples. These programs included testwork that were conducted in 2009 and early 2011 by Metallurgical Division at Inspectorate America Corp, as well as historical testwork conducted between 1988 and 1990 for the Feasibility Study on the West Zone that was completed by Cominco Engineering Services Ltd (CESL).

These testing programs included gold/silver bulk flotation, gravity concentration and cyanidation.

The recent preliminary metallurgical testwork investigated the metallurgical responses of the mineral samples from various mineralization zones to bulk flotation, gravity concentration and cyanidation. The testing programs include open circuit process condition optimization and variability tests.

The test results showed that the mineralization was amenable to a combined flowsheet consisting of gravity separation, flotation and cyanidation (including intensive leaching), for the recovery of gold and silver. The variability test results showed that the combined flowsheet could recover approximately 89% to 99% of the gold from the head samples containing approximately 1.79 g/t Au to 73.3 g/t Au.

The testwork and mineralogical study also indicated that there is a significant amount of the gold in the mineralization present as free gold with a wide range of grain sizes. The gravity concentration would recover approximately 30% of the gold from the variability test samples.

The grindability test results showed that the mineralization is moderately hard with an average Bond ball mill work index of 16.0 kWh/t.

Further testwork to optimize the flotation, gravity concentration and cyanidation conditions are recommended.

1.9 MINERAL PROCESSING

According to the test results, the process flowsheet for the Brucejack mineralization will be a combination of conventional bulk sulphide flotation, gravity concentration and cyanidation with gold and silver recovery by the Merrill-Crowe process. The process is developed to produce gold-silver doré.

There will be two process plants, one flotation plant at the mine site to produce bulk gold-silver flotation concentrate/gravity concentrate and one leach plant (cyanidation and recovery) at the leach plant site to produce gold-silver doré. The leach plant will be located east of the proposed mine site, next to the Hwy 37. The proposed process rate is 1,500 t/d with an availability of 92% (365 d/a). The simplified flowsheet is presented in Figure 16.10.

The mine site process plant will consist of two stages of crushing, primary grinding, gravity concentration and flotation processes to produce a gravity concentrate and a bulk flotation concentrate containing gold and silver. The produced bulk rougher/scavenger concentrate and the gravity concentrate will be dewatered and trucked to the leach plant by 20-t trucks.

The leach plant will consist of the bulk concentrate regrinding and gravity concentration, cyanidation and gold and silver recovery by the Merrill-Crowe process. The conventional cyanidation process will leach the reground rougher concentrates (after gravity concentration) to recover gold and silver. An intensive leach process is proposed to recover gold and silver from the tailings of the gravity cleaner concentration. The extracted gold and silver from the leaching circuits together with the high grade gravity concentrate from the tabling process will be refined on site to produce gold-silver doré.

A part of the final flotation tailings will be used for the underground backfilling and the rest will be discharged to the Brucejack Lake. The leach residues will be sent to the tailing storage facility (TSF) after the residual cyanide is destructed.

1.10 Tailings, Waste, and Water Management

Two tailing streams will be produced from two separate process plants: rougher flotation tailings from the mine site at Brucejack Lake and leach tailings from the leach plant located near Bell Irving River and Highway 37.

Approximately 2.2 Mt of the flotation tailings will be paste backfilled to the underground, while approximately 4.5 Mt of the flotation tailings will be deposited in Brucejack Lake.

The tailings distribution line to the lake will be located on the south side of the lake. The pipeline will extend to a depth of approximately 70 m and tailings will be deposited at the bottom of the lake. The flotation tailings are not anticipated to be acid generating.

The concentrate will be trucked from the mine site at Brucejack Lake to the leach plant for secondary processing. Approximately 1.7 Mt of tailings will be deposited as a slurry in a fully double lined side-hill tailings storage facility located adjacent to the leach facility. A starter facility to store two years of production will be constructed initially to a height of 12 m above the downstream toe. The dam will be raised over the mine life to a height of 24 m above the downstream toe. The leach tailings are anticipated to be acid generating and water from the TSF will be treated prior to discharge.

Approximately 8.7 Mt of waste rock, (2.3 Mt from the underground mine and 6.4 Mt from the open pits) will be produced throughout the LOM. This waste rock will be deposited in Brucejack Lake and in the underground mine. Waste rock generated on the south side of the Lake, from the underground mine, the Galena Hill pit and the Bridge Zone pit, will be deposited in the southwest corner of the lake. Waste rock generated on the north side of the lake, from the SG Zone pit and the Gossan Hill pit will be deposited in the northwest corner of the lake. A water cover will be maintained over the waste rock to limit acid generation. It is assumed that the water decanted from the lake will be suitable for discharge and that waste rock will not leach metals at neutral pH.

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 Brucejack Project.

As with other projects in the northern Coast Range of BC, water management is a key issue. Water contained in the waste rock and mill tailings will report to the Brucejack Lake with disposal of these wastes at depth. Brucejack Lake appears to be a fishless lake. A second season of sampling is necessary to confirm this information.

A suitable location with a reasonably small catchment for the leach tailings storage facility, greatly aids in water management. Diversion channels upslope of the TSF will divert most clean run-off flows around the main dam.

Throughout the project, the owner 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 mine site is located west of the leach plant and will be accessible by an upgraded exploration road which will provide a year-round access to the mine site for the transport of the materials and workers.

The Brucejack leach plant site will be accessible by a planned permanent road constructed between a junction with Highway 37 and the leach plant site. Highway 37, a major road access to northern BC, passes approximately 8 km from the Brucejack leach plant site (Figure 1.3).

At the mine site, a crushing facility will be designed to crush the mineralized material from the proposed mine. The mill will produce bulk gold-silver concentrate.

The main facilities at the mine site will consist of the following:

- primary and secondary crushing
- primary grinding and flotation
- concentrate dewatering and handling
- backfill paste plant
- warehouse building
- truck shop
- permanent camp integrated with offices
- utilities and water services.

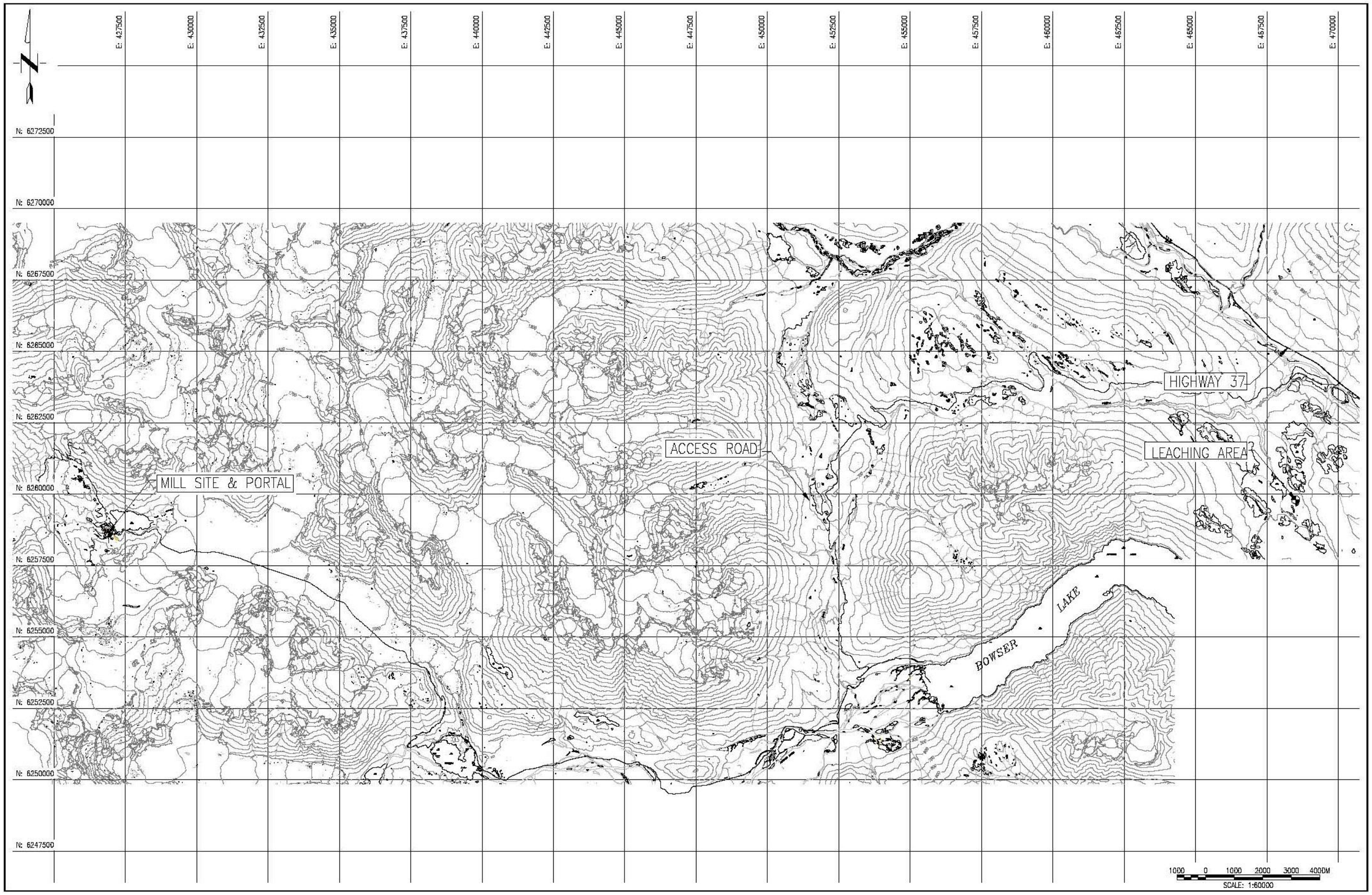
The tailings produced from the process plant at the mine site will be backfilled to underground stopes or deposited in Brucejack Lake, located approximately 1 km north of the process plant.

The main facilities at the leach plant site will consist of the following:

- cyanide leaching and gold recovery process plant
- emergency response and vehicle storage building
- warehouse
- permanent camp integrated with offices
- utilities and water services.

The leach tailings will be deposited in the TSF, located approximately 1 km south of the leach plant.

Figure 1.3 Project General Layout



1.13 Power Supply and Distribution

At the production rate of 1,500 t/d, the operation load is estimated to be approximately 4 MW±10%. The load will be divided between the mine site located near Brucejack Lake and the leach plant site located near Highway 37.

Power for the mine site will be provided by four diesel generators, each rated 1.5 MW. Two diesel generators, each rated 1.5 MW will provide power for the leach plant. This will provide N-1 reliability. Power will be generated and distributed at 4.16 kV. Large loads, above 200 kW, will be fed directly at 4.16 kV. Smaller loads will be fed at 600 V. A heat recovery system will be installed to recover the heat generated from the diesel generators for building heating at both the sites.

1.14 Capital Cost Estimate

The estimated initial capital cost of this project, based on the information available at this time, is US$281.7 million. This includes a contingency amount of US$29.1 million. The capital cost summary is shown in Table 1.2.

Capital Cost Summary

Item	Cost ($)
Direct Works	
Overall Site	9,949,556
Mine Underground	43,669,414
Mine Surface Works	7,855,874
Mine Site Process	33,331,137
Mine Site Utilities	17,104,774
Mine Site Buildings	15,435,659
Tailings	18,001,486
Temporary Facilities	3,917,160
Plant Mobile Equipment (Mine Site)	3,279,180
Leach Area	21,091,405
Leach Plant Site Utilities	15,782,168
Leach Plant Site Buildings	5,984,207
Temporary Facilities	1,374,866
Plant Mobile Equipment (Leach Site)	1,445,778
Direct Works Subtotal	198,222,664
Indirects	
Indirects	46,386,960
Owner's Costs	7,945,899
Contingency	29,148,589
Indirects Subtotal	83,481,448
Total	**281,704,112**

1.15 OPERATING COST ESTIMATE

The total operating cost for the project is estimated at Cdn$158.36/t milled during undergrounding mining and Cdn$68.77/t milled during open pit mining. The estimate includes operating costs for mining, process, general and administration (G&A) and surface services. Tailing/residue disposal operating costs are included in the sustaining capital costs for the project. On average, a total of 265 personnel are projected for the operation, including 130 personnel for mining, 97 personnel for process, and 38 personnel for general management and surface services.

1.16 ECONOMIC EVALUATION

An economic evaluation of the Brucejack Project was prepared by Wardrop based on a pre-tax financial model. For the 16-year LOM and 8.9 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 27.1% internal rate of return (IRR)
- 4.2-year payback on US$281.7 M capital
- US$662 million net present value (NPV) at 5% discount rate.

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

- gold –US$1,100/oz
- silver –US$21.00/oz.

Metal revenues included in the Brucejack cash flow model are based on the average metal production, as presented in Table 1.3.

Table 1.3 Brucejack Project Metal Production

	Average Annual Production		Total Production	
Metal	**Years 1 to 10**	**LOM**	**Years 1 to 10**	**LOM**
Gold (000 oz)	173.2	135	1,731.9	2,157
Silver (000 oz)	1,114.9	918	11,149.2	14,694

Sensitivity analyses were carried out on the following parameters:

- gold price
- silver price
- exchange rate
- gold grade

- silver 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.

1.17 PROJECT DEVELOPMENT PLAN

The project will take approximately two years to complete from the time board approval is received, through construction to introduction of first material into the mill. A further 4 to 6 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 PEA, it is recommended that Pretivm should continue with the next phase of the project in order to identify opportunities and further assess viability of the project.

2.0 INTRODUCTION

Wardrop conducted a preliminary economic assessment of the Brucejack Project, and has prepared this technical report in general accordance with the guidelines provided in National Instrument 43-101 (NI 43-101CP) Standards of Disclosure for Mineral Projects. Wardrop compiled this report based on work provided 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	Peter Mokos, MAusIMM (CP)
18.2: Infrastructure	Wardrop	John Huang, P.Eng.
18.3: Waste and Water Management	BGC	
18.3.1 and **18.3.3**		Hamish Weatherly, P.Geo.
18.3.2 and **18.3.4**		Clint Logue, P.Eng. P.Geo.
18.4: Preliminary Geotechnical Design	BGC	Warren Newcomen, P.Eng.
18.5: Project Execution Plan	Wardrop	Hassan Ghaffari, P.Eng.
18.6: Markets and Contracts	Pretivm	Hassan Ghaffari, P.Eng.
18.7: Environmental	Rescan	Pierre Pelletier, P.Eng.
18.8: Taxes	Pretivm	Hassan Ghaffari, P.Eng
18.9: Capital Cost Estimate	Wardrop	Hassan Ghaffari, P.Eng.
18.10: Operating Cost Estimate	Wardrop	John Huang, P.Eng.
18.10.2: Mining Operating Cost	Wardrop	Peter Mokos, MAusIMM (CP)
18.11: Financial Analysis	Wardrop	Hassan Ghaffari, P.Eng.
19.0 – Conclusions and Recommendations	All	Sign off by section by all contributing QPs
20.0 – References	All	N/A
21.0 – Certificates of QP	All	N/A

3.0 RELIANCE ON OTHER EXPERTS

The Qualified Persons who prepared this report relied upon information provided by the following experts who are not Qualified Persons:

Mr. Joseph Oysenek, Chief Development Officer of Pretivm, has been relied on for advice on matters relating to Taxes

Mr. Ian Chang, Vice President, Project Development of Pretivm, has been relied on for advice on matters relating to Markets and Contracts.

4.0 PROPERTY DESCRIPTION AND TENURE

4.1 DESCRIPTION AND TENURE

In 2010, pursuant to a purchase and sale agreement between 0777666 BC Ltd. (as the seller) and 0890693 BC Ltd. (as the buyer), 0777666 BC Ltd. agreed to sell to 0890693 BC Ltd. the Brucejack and Snowfield Projects. Then, pursuant to an acquisition agreement between Silver Standard (as the seller) and Pretivm (as the buyer), Silver Standard agreed to sell to Pretivm all the issued shares of 0890693 BC Ltd.

The Brucejack Project consists of six mineral claims totalling 3,199.28 ha in area (Table 4.1 and Figure 4.1) and all claims are in good standing until January 31, 2021.

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

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

Table 4.1 Claims List for Brucejack Project

Tenure No.	Tenure Type	Map No.	Owner	Pretivm Interest	Status	In Good Standing To	Area (ha)
509223	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2021	428.62
509397	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2021	375.15
509400	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 1 2021	178.63
509463	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2021	482.57
509464	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2021	1,144.53
509506	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2021	589.78
Total							**3,199.28**

The royalties applicable to the Brucejack Project are as follows:

- "Royalty" means the amount payable by the Owner, calculated as 1.2% of the NSR, with the following exemptions:

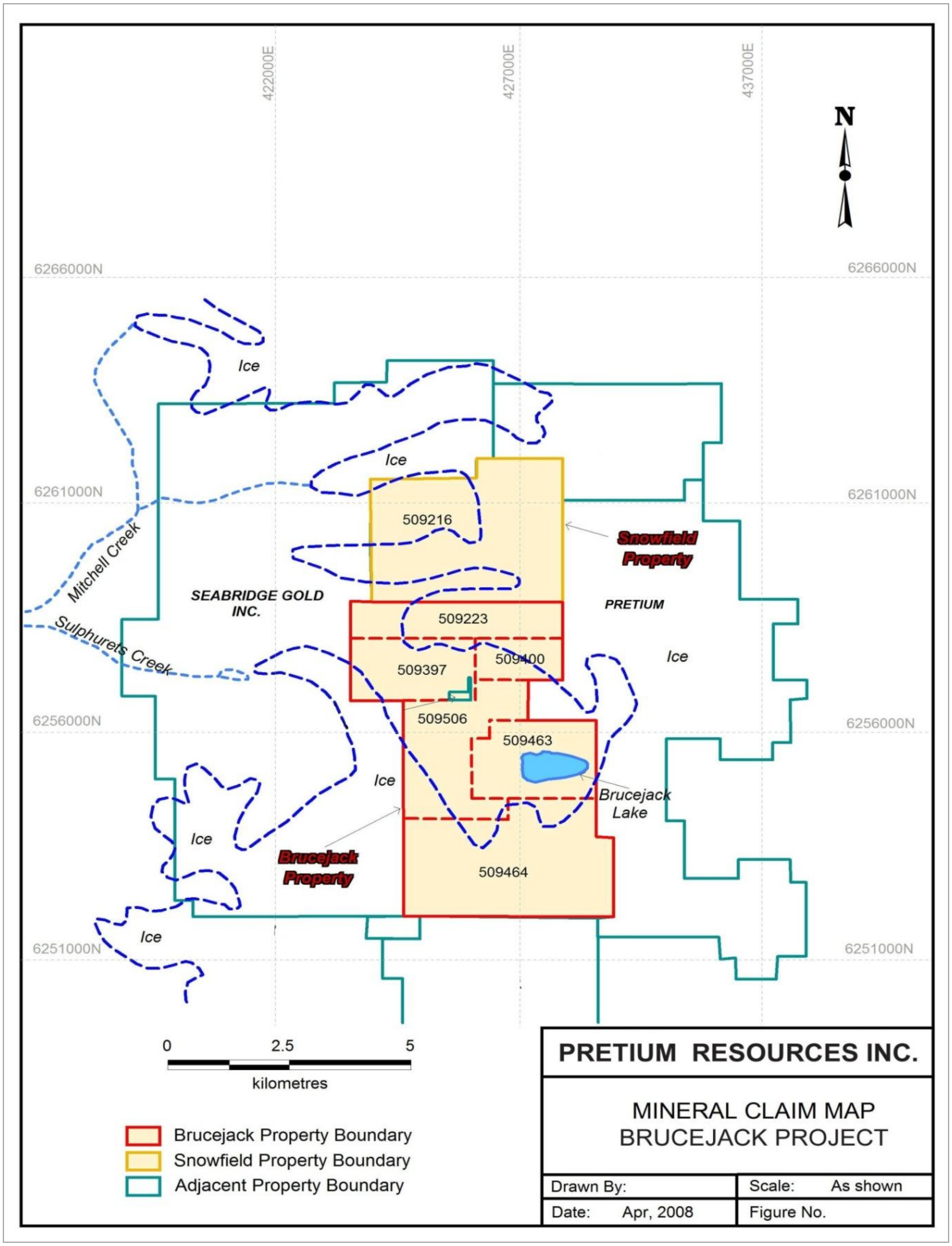
- gold: the first 503,386 oz produced from the Brucejack Project
- silver: the first 17,907,080 oz produced from the Brucejack Project.

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

Figure 4.1 illustrates the six Brucejack Project claims, which adjoin the Snowfield Project to the north. The majority of the Brucejack Project 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 First Nations groups and the Government of BC.

Figure 4.1 Mineral Claim Map of the Brucejack Project



5.0 LOCATION, ACCESS, CLIMATE, PHYSIOGRAPHY AND INFRASTRUCTURE

5.1 LOCATION AND ACCESS

The Brucejack property is located approximately 56°28′20″N latitude by 130°11′31″Wlongitude, which is approximately 950 km northwest of Vancouver, 65 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.

The Brucejack property is located in the Boundary Range of the Coast Mountain Physiographic Belt, along the western margin of the Intermontane Tectonic Belt. The local terrain is generally steep with local reliefs of 1,000 m from valleys occupied by receding glaciers, to ridges at elevations of 1200 masl. Elevations within the Project area range from 1,000 m along the Mitchell Glacier to 1960 masl along the ridge between the Mitchell and Hanging Glaciers.

The Project is easily accessible via helicopter from the town of Stewart or the settlement of Bell II. The flight time from Stewart is approximately 30 minutes; the flight time from Bell II is slightly less. Stewart has an established year-round helicopter base.

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

5.2 CLIMATE AND PHYSIOGRAPHY

The climate is typical of north-western BC, with cool, wet summers, and relatively moderate but wet winters. Annual temperatures range from +20°C to -20°C. Snowfall accumulations range 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.

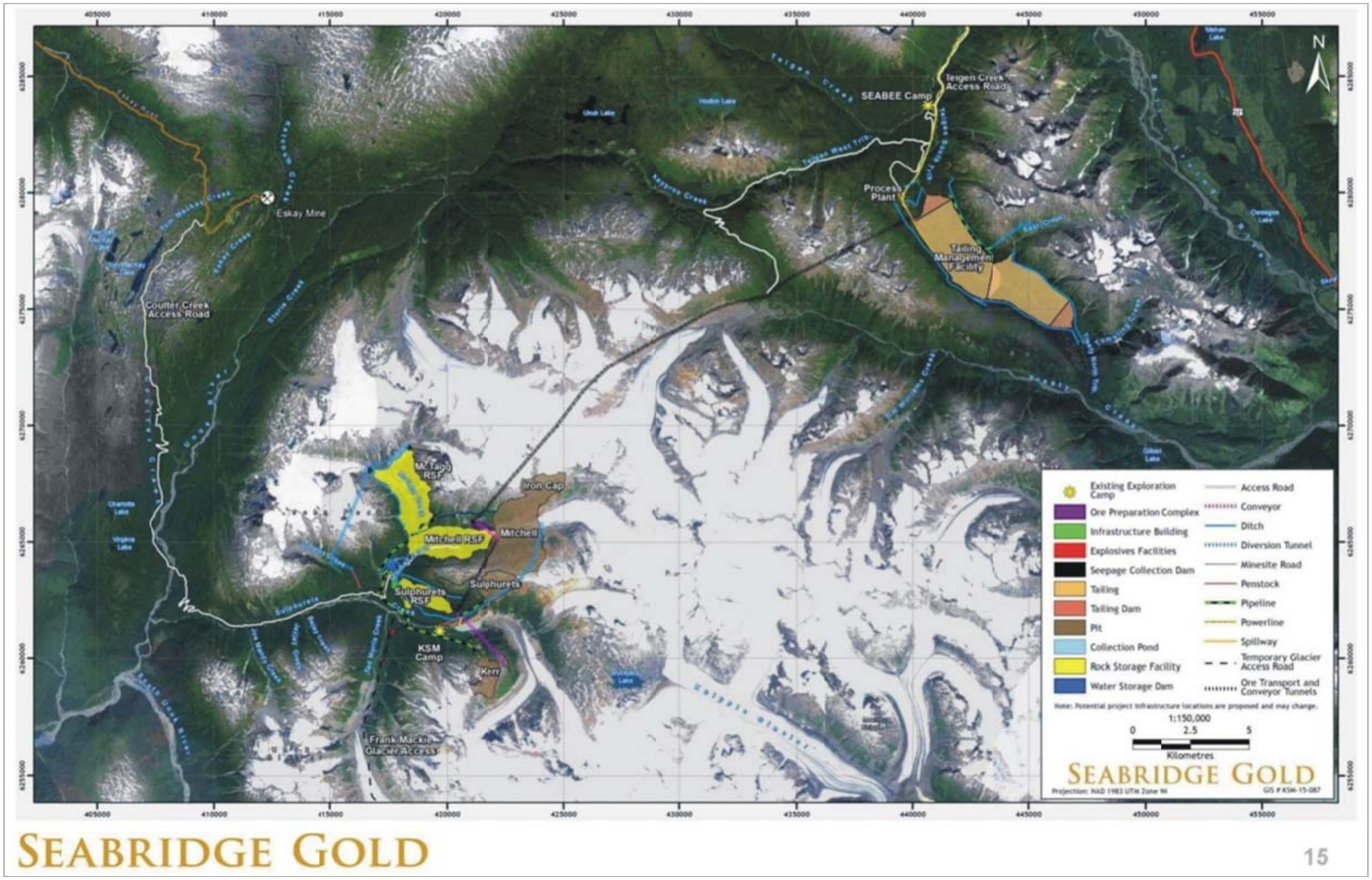
5.3 Infrastructure and Local Resources

The Brucejack property lies immediately east of Seabridge Gold Inc.'s (Seabridge) Kerr-Sulphurets-Mitchell(KSM) Project. The Brucejack Project will likely be influenced by Seabridge's future access plans for that area, as Seabridge discussed in its Updated Preliminary Feasibility Study, (PFS) dated May 2, 2011. The updated PFS was prepared by Wardrop, A Tetra Tech Company (Wardrop). 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. The nearest infrastructure is that found in the town of Stewart, approximately 65 km south of the Project, which has minimal supplies and personnel. Stewart is the most northerly ice-free shipping port in North America and is accessible to store and ship concentrates. Such material is currently being shipped from the Wolverine and Huckleberry mines via this terminal. The towns of Terrace and Smithers are also located in the same general region as the Project. Both towns 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 southeast of the Project. This line runs east-west, and terminates at the deep water port of Prince Rupert on the west coast of BC.

Figure 5.1 KSM Project Planned Road Access



Note: After Seabridge; Wardrop, 2011.

6.0 HISTORY

The Brucejack Project and the surrounding region have a rich precious and base metals exploration history that dates back to the late 1800s. This section describes the mineral exploration of the region, including historical drilling, that was conducted prior to Pretivm's acquisition of the Brucejack Project. In the late 1980s, the Sulphurets project was fully permitted as a 300 t/d underground operation; however, the project was not constructed due to declining metal prices. The historical data have been summarized predominantly from various assessment reports available through the BC Ministry of Energy, Mines and Petroleum Resources.

6.1 HISTORY

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 copper-molybdenum mineralization on the Sulphurets property in the vicinity of the Main Copper Zone, approximately 6 km north-west 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 copper and gold-silver occurrences were discovered in the Sulphurets-Mitchell Creek area.

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

- 1960 to 1979 – Granduc continued exploration, conducting further geological mapping, lithogeochemical sampling, trenching, and diamond drilling on known base and precious metal targets north and north-west of Brucejack Lake resulting in the discovery of gold-silver mineralization in the Hanging Glacier area and molybdenum 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, Shore, West, and Galena Zones. Gold was discovered on the peninsula at Brucejack Lake near the Shore Zone.
- 1982 to 1983 – Exploration was confined to gold and silver-bearing vein systems in the Brucejack Lake area at the southern end of the property from

1982 to 1983. Drilling was concentrated in 12 silver and gold-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).The Newcana JV completed work on the, Mitchell, Golden Marmot, Sulphurets Gold, and Main Copper Zones, along with lesser known targets.
- 1986 to 1991 – Between 1986 and 1991, the Newcana JV spent approximately $21 million eveloping the West Zone and other smaller precious metal veins on what would later become the Bruceside property.
- 1991 to 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 trenching, mapping, airborne surveys, and surface drilling, evaluating various surface targets including the Shore, Gossan Hill, Galena Hill, Maddux, and SG Zones.
- 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 7,352 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 Black Hawk Mining Inc.
- 1997 to 1998 – No exploration or development work was carried out on the Brucejack Project (Budinski et al., 2001).
- 1999 – Silver Standard acquired Newhawk and with it, Newhawk's 60% interest and control of the Brucejack Project and 100% interest in the Snowfield Project (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 Project, resulting in 100% interest in the Project.
- 1999 to 2008 – No exploration or development work was carried out on the Brucejack Project during the period from 1999 to 2008.

The historical interpretation (Pincock Allen and Holt 2001) of mineralized zones on the West Zone, prior to Silver Standard undertaking their exploration work in 2008, is shown in Figure 6.1 (vein location plan map) and Figure 6.2 (cross section map).

Figure 6.1 West Zone Vein Location Plan

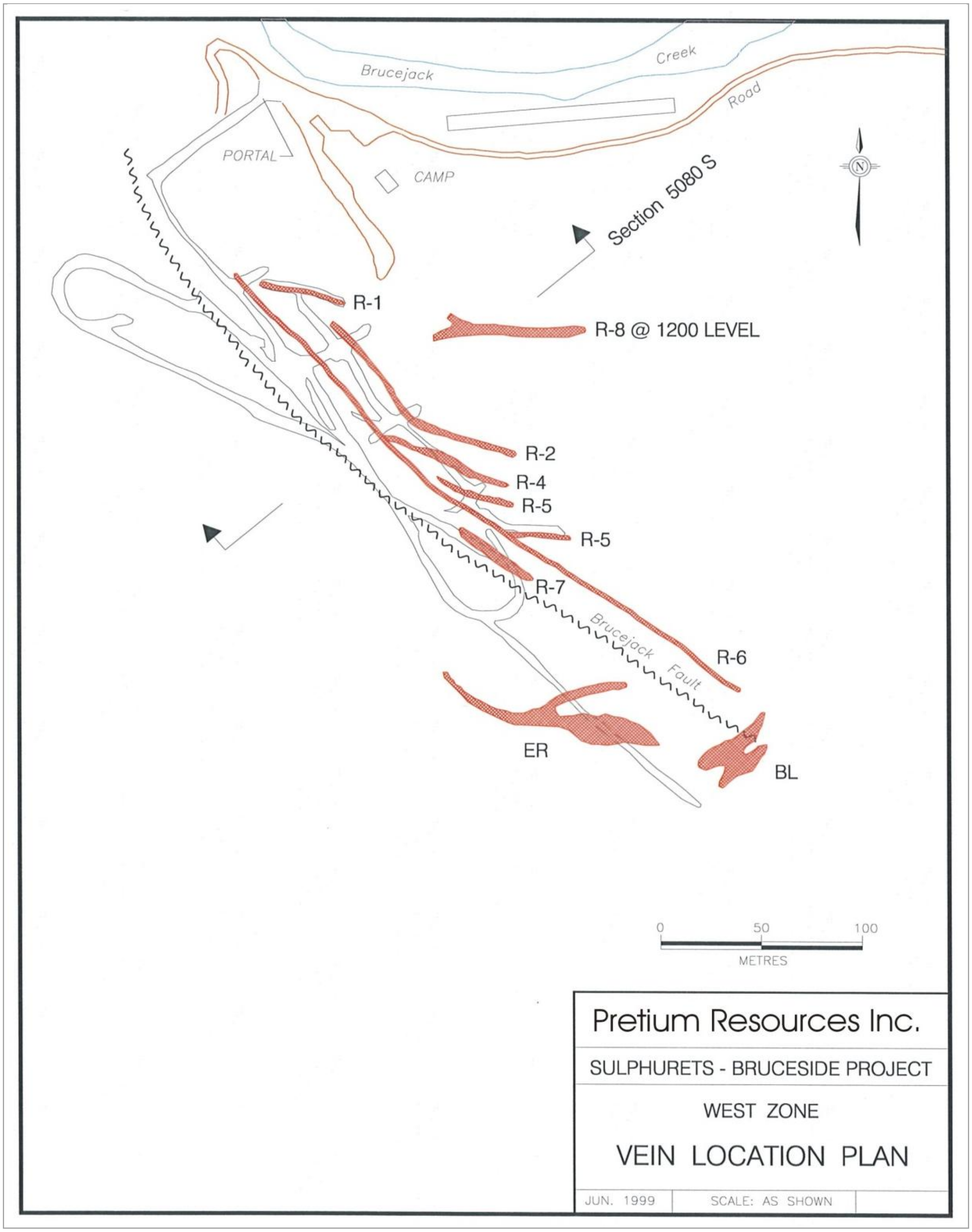
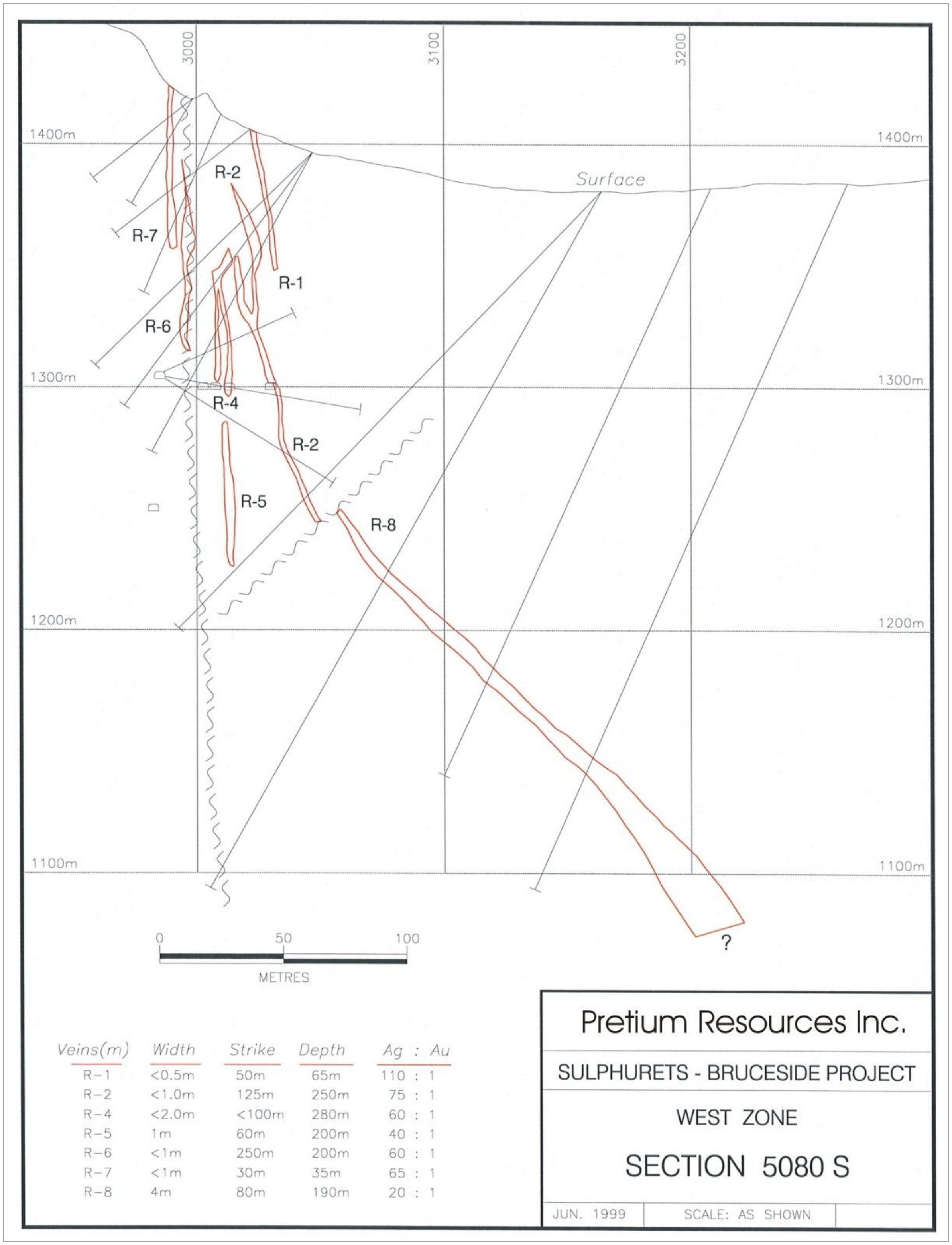


Figure 6.2 West Zone Section 5080 S



6.2 Recent Work Completed By Silver Standard

In 2009, Silver Standard began work on the Brucejack Project, the first since its acquisition. The 2009 program included drilling, rock-chip and channel sampling, and re-sampling of historical drill core.

During the 2009 Brucejack Project 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 ALS 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 ALS Chemex facility in Vancouver. This work updated the QA/QC work for 333 historical surface drill holes and 432 underground drill holes, which are now included in the resource database.

Field work undertaken throughout the 2009 program included the drilling of 37 surface diamond drill holes totalling 17,846 m and 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 were 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.

See Section 11 for details of 2010 drilling program.

6.3 Previous Studies

In 1990, Corona Corp. completed a feasibility study that proposed an underground mine with decline access for the Sulphurets Project (West and R-8 Zones only). Total operating costs of $145/t were estimated based on a 350 t/d mill facility for processing, a capital cost of $42.7 million and a 6.7% pre-tax return at a price of US $400/oz Au and $5/oz Ag. The study concluded that higher metal prices must be realized before a production decision could be made.

This 1990 Corona Sulphurets Project Feasibility Study is no longer relevant, is not NI 43-101 compliant and should not be relied upon.

6.3.1 PRELIMINARY ASSESSMENT

In 2010, Wardrop completed a preliminary assessment (PA) of the Snowfield-Brucejack Project.

The following consultants completed the component studies for the Technical Report:

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

Based on the results of the PA, it was recommended that Pretivm continue with the next phase of the project in order to identify opportunities and further assess viability of the project. The full PA can be consulted at www.sedar.com.

6.4 PREVIOUS PERMITS

The following describes the permits that were obtained by Newhawk for the Sulphurets Project.

A reclamation permit was obtained in 1986 from the BC Ministry of Energy, Mines and Petroleum Resources. The reclamation permit – exploration #MX-1-86 expired in 1999, upon the submission of the Reclamation Report and with the completion of reclamation at site.

Approval in principle was granted in 1989 for the development of a mine at the West Zone in accordance with the mine development review process. The LOM Development Certificate under the BCEAO was first granted in 1993 (certificate 92-06) and was renewed until 1998. In 1998, the Mine Development Certificate was replaced by the Project Approval Certificate and the existing 92-06 certificate became the M98-03 Project Approval Certificate. The last amendment to this certificate was granted in January 2004. The certificate expired in September 2006.

Under the Ministry of Environment, Lands and Parks, a Waste Management Permit to discharge effluent from the camp (PE-7922) was granted in 1987 and cancelled in 1999 as the Sulphurets camp had been dismantled and removed, and the property had been reclaimed.

A Forest Act Special Use Permit (SUP 14912) was granted for the construction of the Bowser Valley road from Brucejack to highway 37 via Wildfire Ridge in 1987 by the BC Ministry of Natural Resource Operations. The permit expired in 2003 and was officially closed in 2010. Numerous Water Act Approvals were granted in 1988 under the SUP 14912 to construct crossings, culverts and other road associated stream crossings.

An Industrial Water Licence for the Sulphurets Gold Project (File 6000626) was applied for in 1992 with the BC Ministry of Environment, Lands and Parks. The application was cancelled in 1999 due to no plans for production.

Under the BC Ministry of Environment, Water Management Division, a Water Rights Rental was obtained (File 6000374); this included conditional water licenses (72124 and 72125) for authorized storage and water diversion from Brucejack Creek for mining and power, and for industrial purposes. A permit to occupy crown-owned land (18067) was also granted.

Approvals to construct underground sump and dams were granted in 1989 under the Mines Act (11220-30 and 14640-02).

A Radioisotope License (9-11402) for a SciTec Inc. Silver Analyzer was submitted to the Canadian Atomic Control Board in 1990; the licence was revoked in 1999 at Newhawk's request.

An Explosives Storage and Use Permit was granted in 1986, and expired in 1993.

7.0 GEOLOGICAL SETTING

This section, which describes the regional and local geology of the Brucejack Project, draws heavily from the Technical Report titled, "Technical Report on the Snowfield Property, Skeena Mining Division, British Columbia, Canada," by Minorex Consulting Ltd., dated April 21, 2008.

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

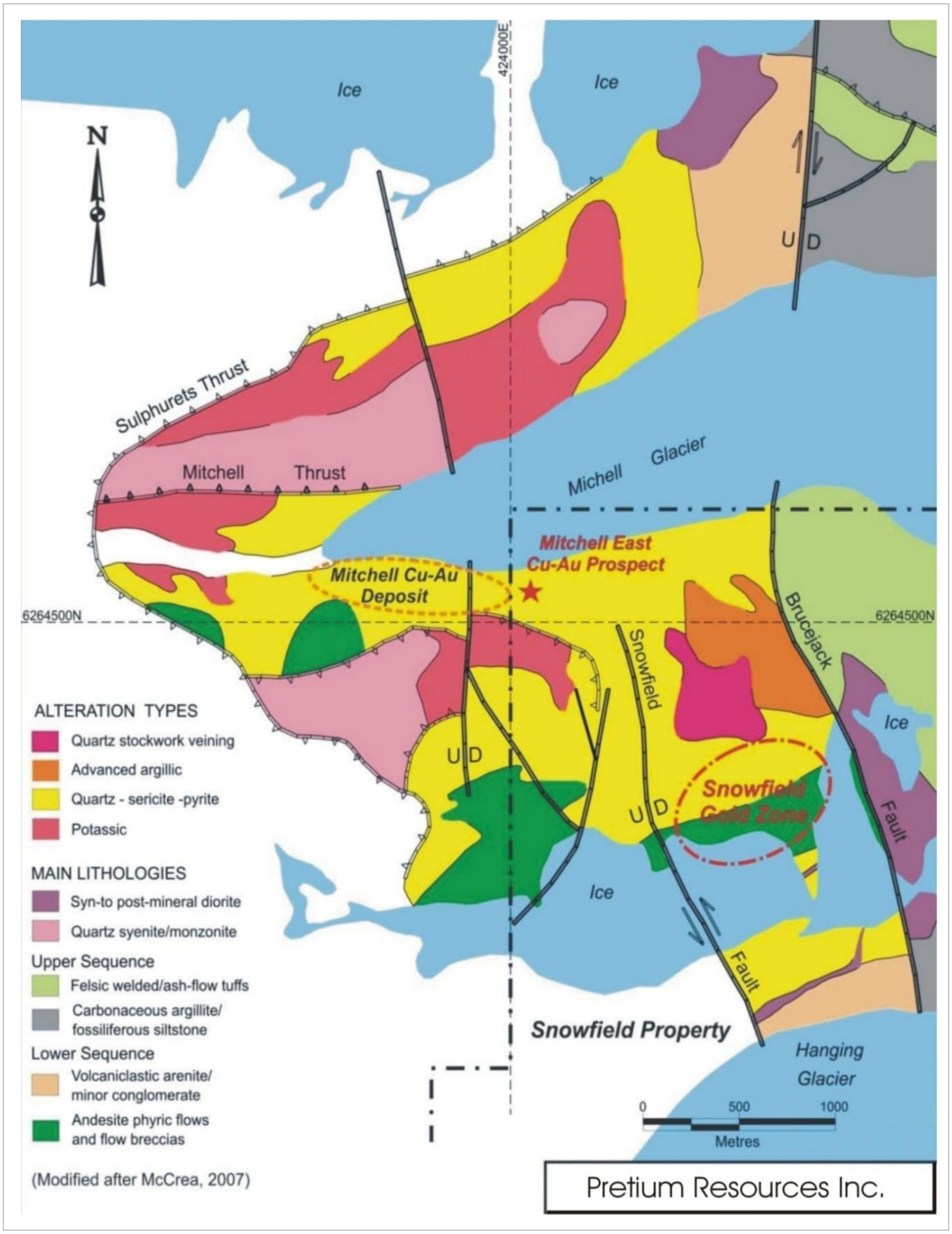
7.1 REGIONAL GEOLOGY

The 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 siltstone and conglomerate
- Upper Unuk River Formation: alternating intermediate volcanic rock and siltstone
- Betty Creek Formation: alternating conglomerate, sandstone, and intermediate to mafic volcanic rock
- Mount Dilworth Formation: felsic pyroclastic tuffaceous rock and flows
- Salmon River and Bowser Formations: alternating siltstone and sandstone.

Britton and Alldrick (1988) 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 was completed by the Geological Survey of Canada, the British Columbia Ministry of Energy, Mines and Resources, and the Mineral Deposits Research Unit (MDRU) at the University of British Columbia. A regional geology map is depicted in Figure 7.3.

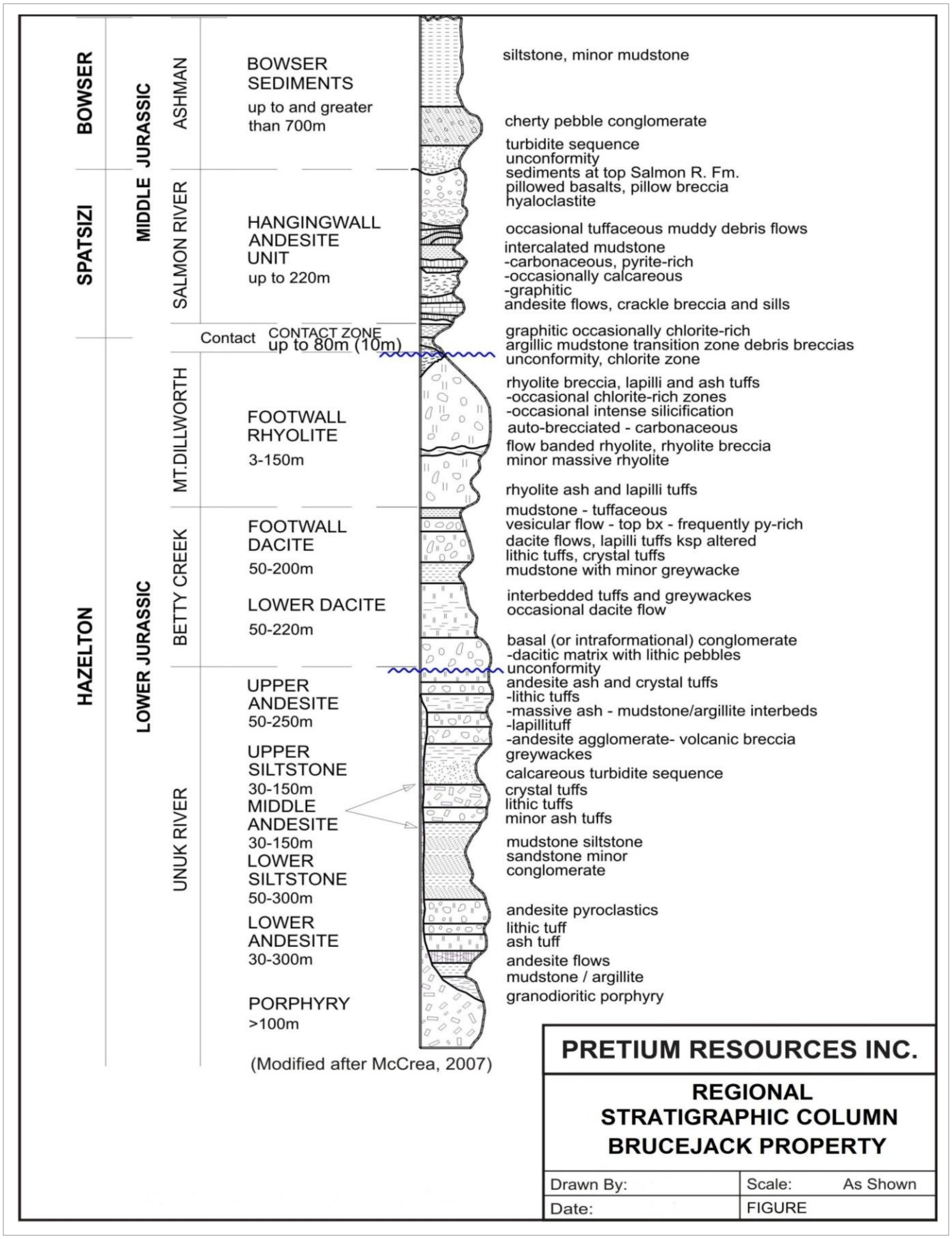
The regional stratigraphic assemblage as originally compiled by Kirkham (1963) and later modified by Britton and Alldrick (1988), Alldrick and Britton (1991), McCrea (2007) and Blanchflower (2008), is illustrated in Figure 7.2 and has been summarized in Table 7.1.

Table 7.1 Summary of Regional Stratigraphy – Oldest to Youngest

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

Source: After Blanchflower, 2008.

Figure 7.2 Regional Stratigraphic Column



Source: After Blanchflower, 2008.

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

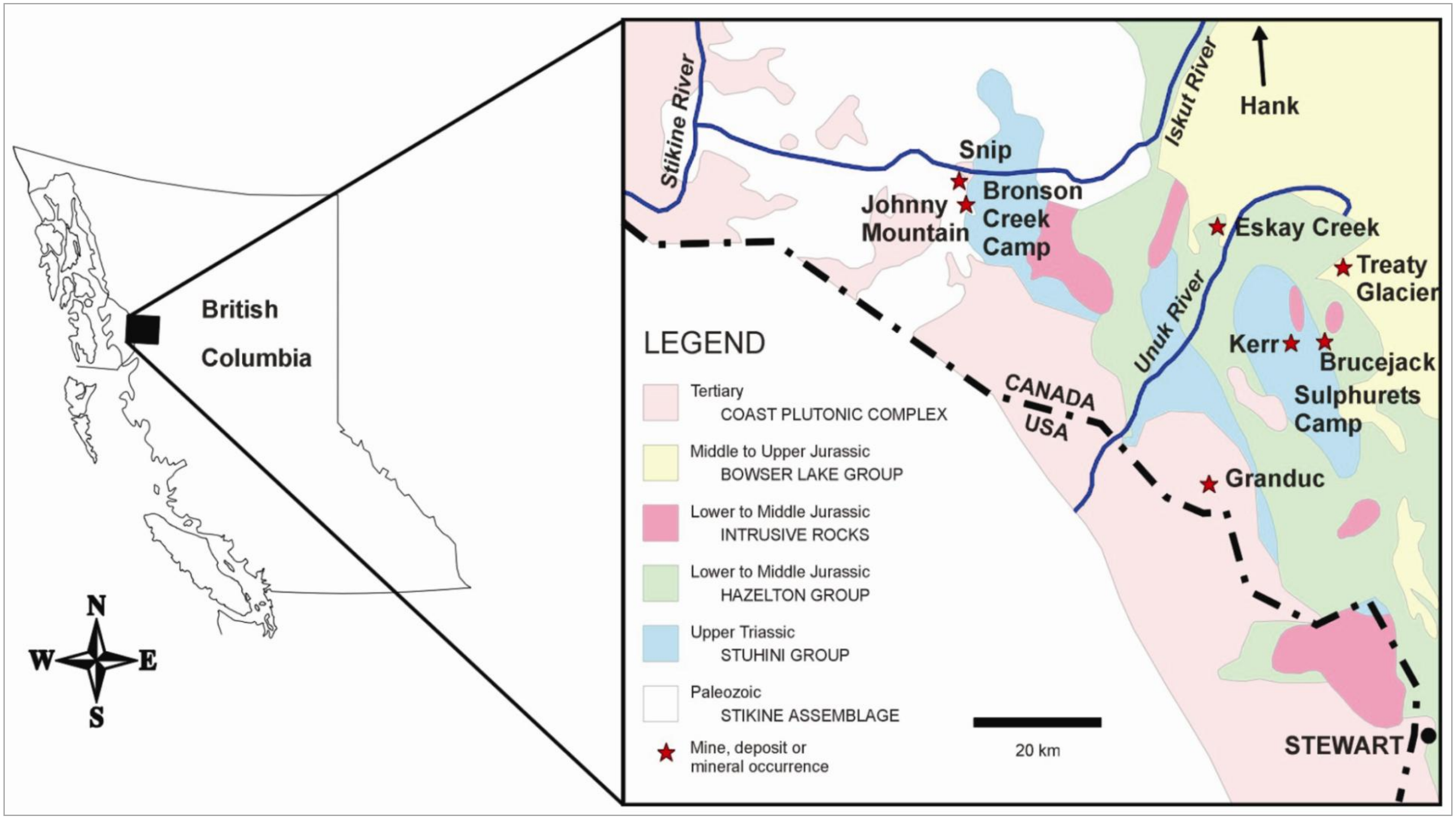
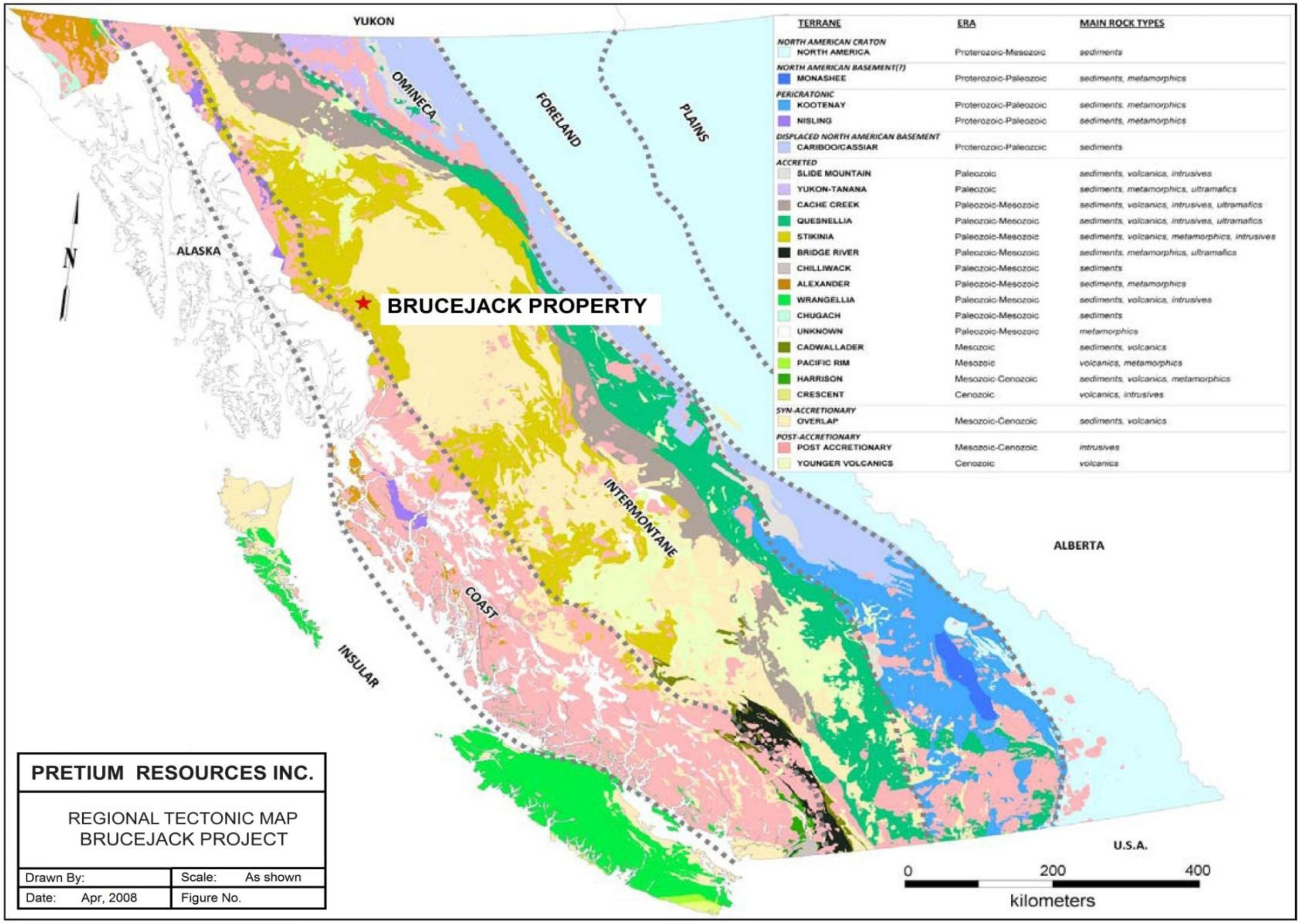


Figure 7.4 Regional Tectonic Map



7.2 Geology, Structure and Alteration 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.

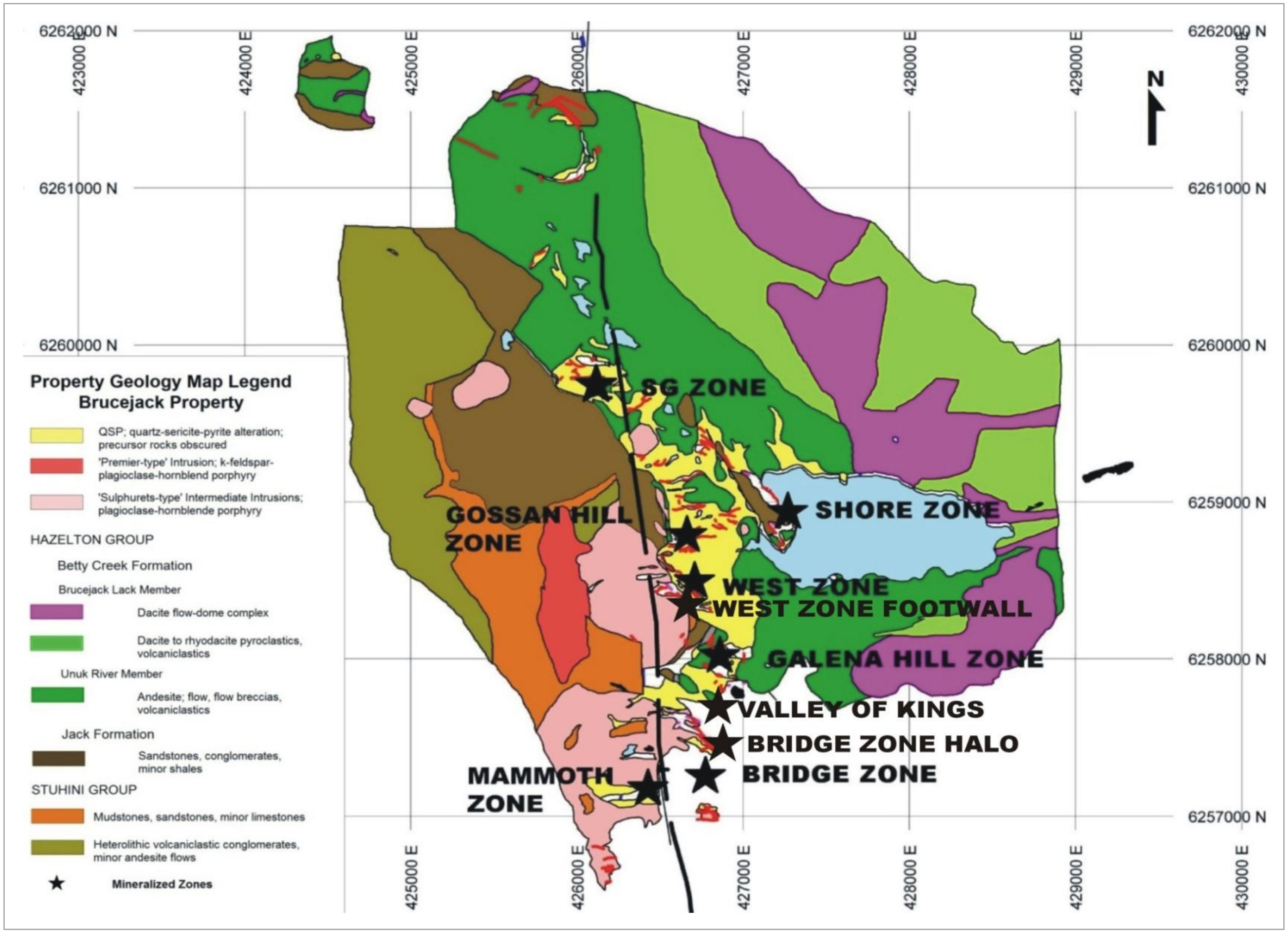
Published descriptions of the regional geology of the Sulphurets Creek-Brucejack Lake area have been presented by the Geological Survey of Canada (Henderson et al., 1992; Kirkham, 1991; Anderson, 1989), geologists working for the British Columbia government (Britten and Alldrick, 1988; Alldrick et al., 1987; Grove, 1986) and by the MDRU at the University of British Columbia (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 Brucejack Property overall become younger to the east (Figure 7.5). 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 roughly the basal formation of the Hazelton Group and is dated at about 196 Ma.

Open folding and probable thrust faulting has also placed a wedge of Jack Formation sandstone and conglomerate 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 breccia, and intermediate tuffaceous rock intercalated with volcaniclastic conglomerate, sandstone and siltstone. 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 7.5 Brucejack Property Geology Map Showing Simplified Geology and Mineralized Zones



The andesite of the Unuk River Member is the most important host rock to Au- and Ag-bearing quartz veins discovered in the Brucejack area and has 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 tuff 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 mudstone 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 south-east of Brucejack Lake, there are three types of intrusions recognized on 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 m to 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 Brucejack property 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 property. 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 TYPE

8.1 INTRODUCTION

While deposits such as Kerr, and Mitchell are probably best described as gold-enriched copper porphyry systems, most (if not all) of the mineralization on the Brucejack Property has been classified as an epithermal gold-silver-copper, 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) or the mesothermal deposit type.

8.2 EPITHERMAL GOLD-SILVER-COPPER, LOW-SULPHIDATION DEPOSITS

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 Property 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.

Among 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, gold-silver model. Other examples in British Columbia include the Blackdome and Silbak-Premier Mines.

8.3 EPITHERMAL GENETIC MODEL

8.3.1 INTRODUCTION

SIMPLIFIED DEFINITION

Epithermal deposits of gold (±silver) 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, sediment, 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 (gold, silver) but some deposits may also contain variable amounts of base metals such as copper, lead, and zinc.

EPITHERMAL SUB-SYSTEMS

Epithermal gold 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-gold, 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 sulphur, carbon, and chlorine.

- 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 carbon and sulphur.

8.3.2 EPITHERMAL MINERALIZATION CHARACTERISTICS

Lindgren (1922, 1933) suggested that degassing magmas are sources of many ore-forming constituents in epithermal gold 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), oxygen and hydrogen 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, carbon and sulphur 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 chlorine, hydrogen sulphide, and carbon dioxide, 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 gold in epithermal deposits are as follows:

- The metals are supplied directly by actively or passively degassing magma (e.g. Taylor, 1987 and 1988) that also provide heat to the paleo-hydrothermal system

- 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 sulphur and carbon 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 gold deposits are listed for each of three deposit subtypes in Table 8.1.

8.3.3 *Diagnostic Characteristics of Epithermal Subtypes*

Grade and Tonnage Characteristics

The size and grade of the principal Canadian epithermal gold 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 gold 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 British Columbia, epithermal mineralization tends to comprise disseminated gold in silicified and/or finely veined rocks. Grades are typically lower, but tonnages larger, than in other more typical vein-type epithermal deposits.

Table 8.1 Summary of Geological Setting, Definitive Characteristics and Examples of Typical Epithermal Gold Deposit Subtypes

	HIGH-SULPHIDATION subtype Hosted in volcanic rocks	LOW-SULPHIDATION subtype Hosted in volcanic and plutonic rocks	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 ($\delta^{13}C_{CO_2} \cong -5\pm2$; $\delta D_{H_2O} \cong -35\pm10$; $\delta^{18}O_{H_2O} \cong +7\pm2$; $\delta^{34}S_{\Sigma S} \cong 0$); magmatic-hydrothermal alunite $\delta^{34}S$>sulphide minerals; $\delta D \cong -35\pm10$; steam-heated alunite $\delta^{34}S \cong$ 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 ($\delta^{13}C \cong -26\pm2$) may be derived from wall rocks
Ore fluids (examples from fluid inclusion studies)	160-240ºC; ≤1 wt.% NaCl (late fluids); possibly to 30 wt.% NaCl in early fluids; boiling common; (Nansatsu district, Japan; Hedenquist et al., 1994)	sulphide-poor: 180-31ºC, ≤1 wt.% NaCl, about 1.0 molal CO_2 (Mt. Skukum: McDonald, 1987) sulphide-rich: ave. 25ºC, <1 to 4 wt.% NaCl (Silbak-Premier: McDonald, 1990)	bimodal: 150-160 (most); 270-280ºC, ≤15 wt.% NaCl; nonboiling: (Cinola: Shen et al., 1982); 230-250ºC, ≤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') AI 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 = enargite; gn = galena; py = pyrite; sp = sphalerite; ss = sulphosalts.

Source: After Taylor, 2007.

Physical Characteristics

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

Mineralogy

Quartz is the predominant gangue mineral in all epithermal gold 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

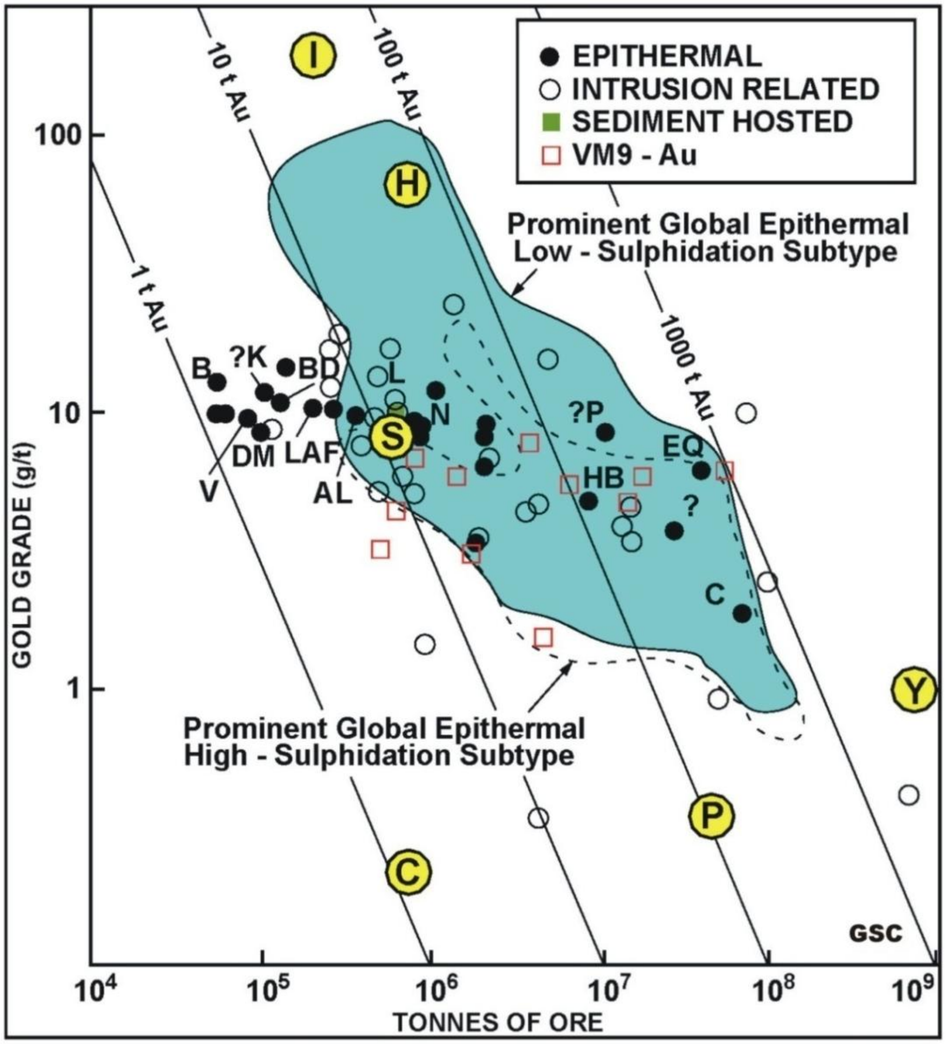
Native gold and electrum occur in low-sulphidation subtype vein deposits that often contain only a few percent or less of sulphides (usually pyrite; e.g. Blackdome, BC). In deposits in which sulphide minerals are abundant (e.g. Venus; Silbak-Premier: sulphide-rich stage), sulphides commonly include chalcopyrite, 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 gold 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 copper abundance can vary significantly (Sillitoe, 1993) and typically dominate that of zinc. Principal Gangue minerals include quartz (vuggy silica), alunite, barite (especially associated with gold). Calcite is not characteristic of high-sulphidation subtype deposits due to the high acidity of the hydrothermal fluids.

Figure 8.1 Plot of Gold Grade (g/t) vs. Tonnage (Economic, or Reserves + Production) for Selected Canadian Epithermal Gold Deposits & Prominent Examples Worldwide



Source: Taylor, 2007

Notes:

(1) 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.

(2) 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 gold-bearing VMS deposits (marine epithermal) by open red squares.

(3) The median grades and tonnages for several comparable types of deposits (yellow-filled circles) from Cox and Singer (1986) include porphyry copper-gold (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].

(4) 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).

(5) Fields for prominent low-sulphidation (blue shading) and high-sulphidation (dashed line) epithermal gold deposits worldwide (global) are based on data in Hedenquist et al. (1996, 2000).

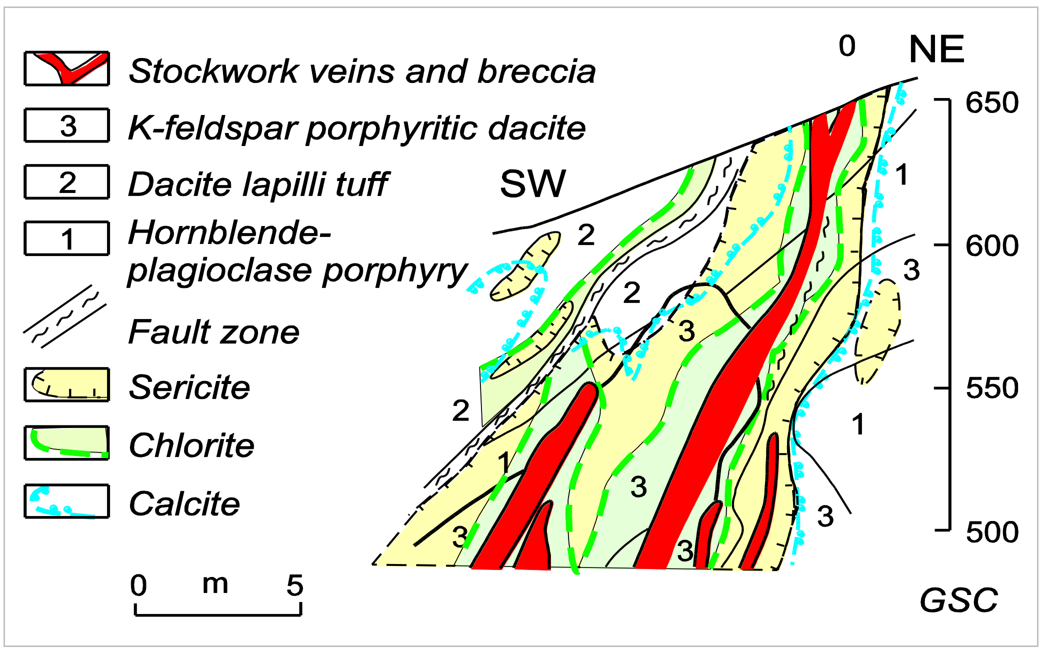
System Dimensions

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 x 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 8.2) (McDonald, 1990).

Figure 8.2 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.

Figure 8.2 Geological Cross-section of a Representative Canadian Epithermal Deposit Illustrating Alteration Mineral Zoning and Selected Features



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 gold 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-to-silver ratios of epithermal gold 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, silver-to-gold 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 silver-to-gold 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 gold 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 sulphur dioxide in high-temperature hydrothermal systems, and also by reaction of host rocks with steam-heated meteoric waters acidified by oxidation of hydrogen sulphide (probably of magmatic origin: e.g. Rye et al., 1992; Bethke et al., 2005), or by dissolution of carbon dioxide. 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 18O/16O 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 modelling.

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 carbon dioxide contents (e.g. Baker, 2002).

8.4 Summary – Epithermal Mineralizing Systems

The geological settings of low, intermediate and high-sulphidation subtype epithermal deposits are illustrated schematically in Figure 8.3.

The locations of epithermal gold 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.

8.4.1 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.

8.4.2 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.

8.4.3 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 8.3, is classified as a transitional to low sulphidation epithermal deposit.

Characteristics shown in Figure 8.3 evolve with time; all features illustrated are not implied to be synchronous.

Local environments and examples of low-sulphidation deposits, as illustrated in Figure 8.3, include:

- (A) basin margin faults; Dusty Mac
- (B) disseminated ore in sedimentary rocks; Cinola
- (C) veins in degassing, carbon dioxide-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.

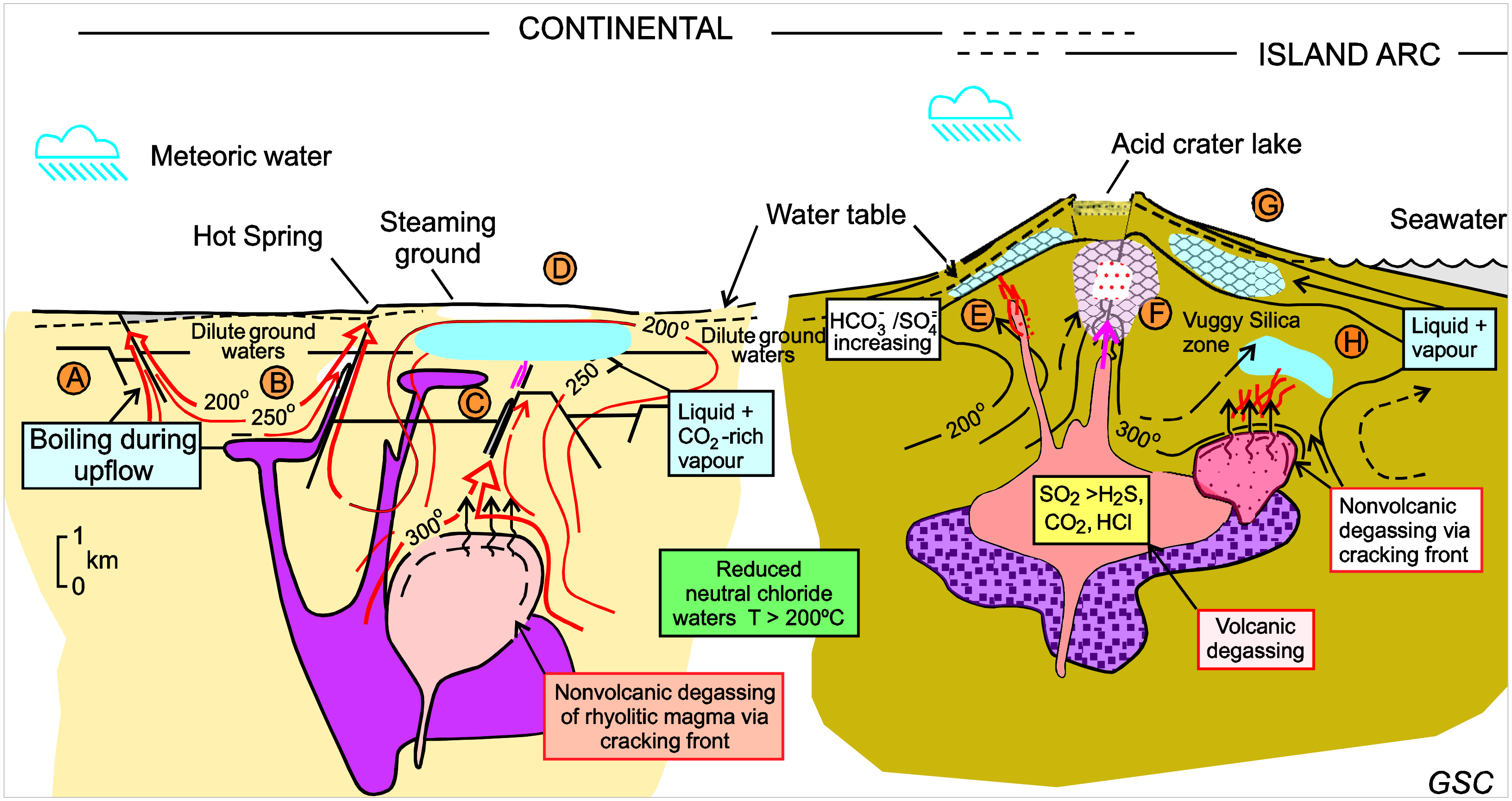
Examples of high-sulphidation environments, as illustrated in Figure 8.3, include:

- (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 8.3:

- 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 sulphur dioxide disproportionates to hydrogen sulphide and sulphuric acid during ascent beneath environment (F).
- Note that free circulation occurs only in crust above about 400°C.
- All temperatures are shown in degrees Celsius.

Figure 8.3 Schematic Cross-section – General Geological and Hydrological Settings of Quartz-(Kaolinite)-Alunite and Adularia-Sericite Deposits



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

9.0 MINERALIZATION

9.1 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:

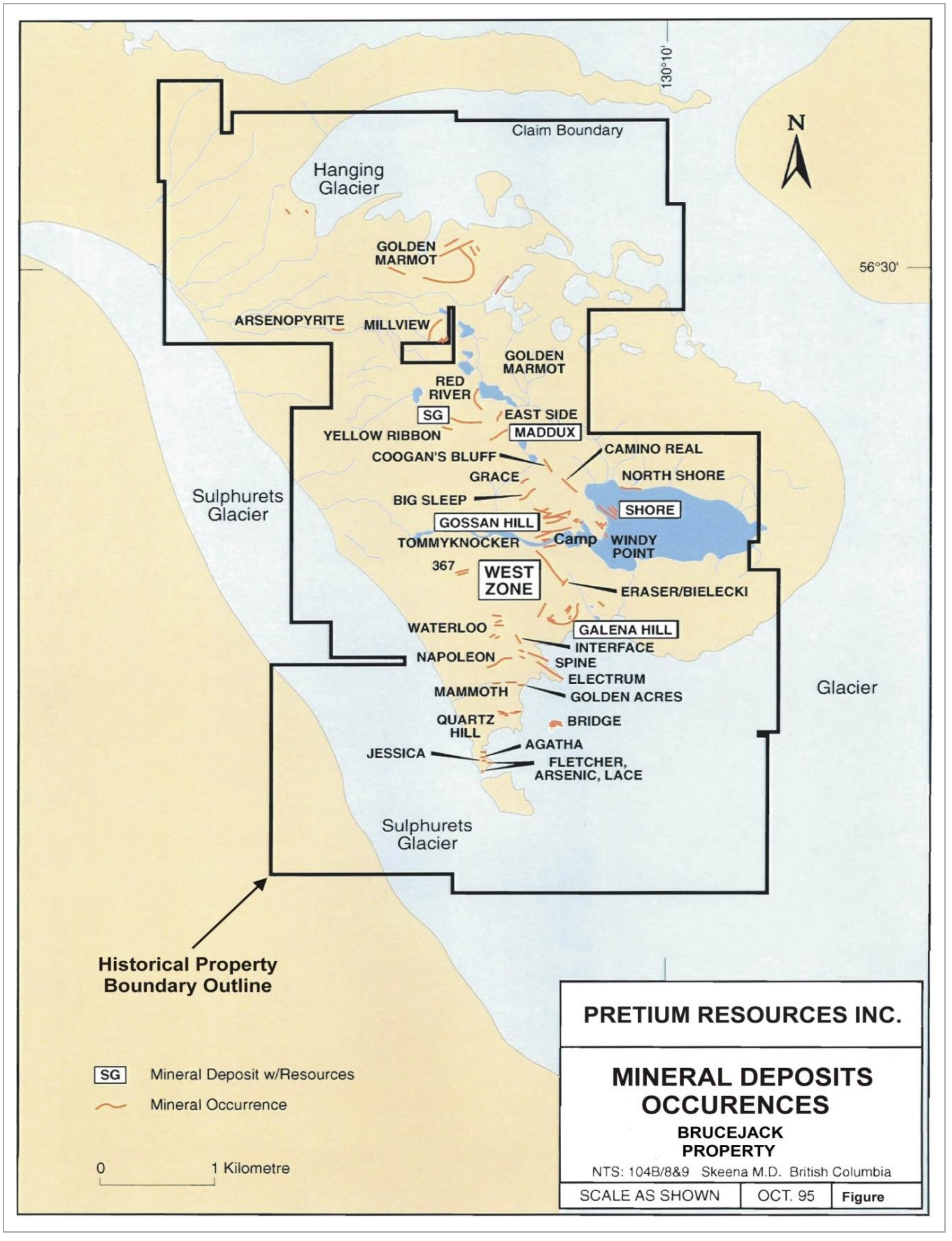
- porphyry-type gold mineralization associated with fine grained syenite to syenodiorite intrusive rocks, intrusive breccias and pyritization
- silver-gold-base metal epithermal veins occurring within or adjacent to fine grained syenodiorite intrusions and associated with large areas of intense sericite, 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 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.

Figure 9.1 Historical Map with Mineral Deposits and Occurrences



Note: Modified after Budinski, 1995.

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 indicate the deposits were formed in a near surface epithermal style environment.

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 VOK 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) North-west trending veins localized along pre-existing structures such as the West, Shore, Bridge and VOK Zones
  - Underground mapping at the West Zone indicated that the north-west trending structures were brecciated, while east-west trending structures were not. This would support the theory of reactivation along pre-existing north-west structures. Reactivation was probably sinistral in movement. The localization of major vein systems within the volcanic rock as opposed to the sedimentary rock is likely the result of preferential ground preparation.
- Stage 3 was marked by the development of north-west trending cleavage and local warping of smaller veins as a result of northeast-southwest shortening.

Pretivm reviewed all of the historical and ongoing exploration results, allowing the company to identify nine zones of potentially near-term economically viable mineralization.

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

- West Zone
- West Zone Footwall Zone

- Bridge Zone
- Bridge Halo Zone
- Galena Hill Zone
- Shore Zone
- SG Zone
- Gossan Hill Zone
- Valley of Kings Zone (VOK).

These nine zones are the focus of Mineral Resource Estimates outlined in Section 16.0 of this report; they are discussed individually in Sections 9.2.1 through 9.2.8.

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 mineralization 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 rock;

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 KSM property 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 Zones) 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.

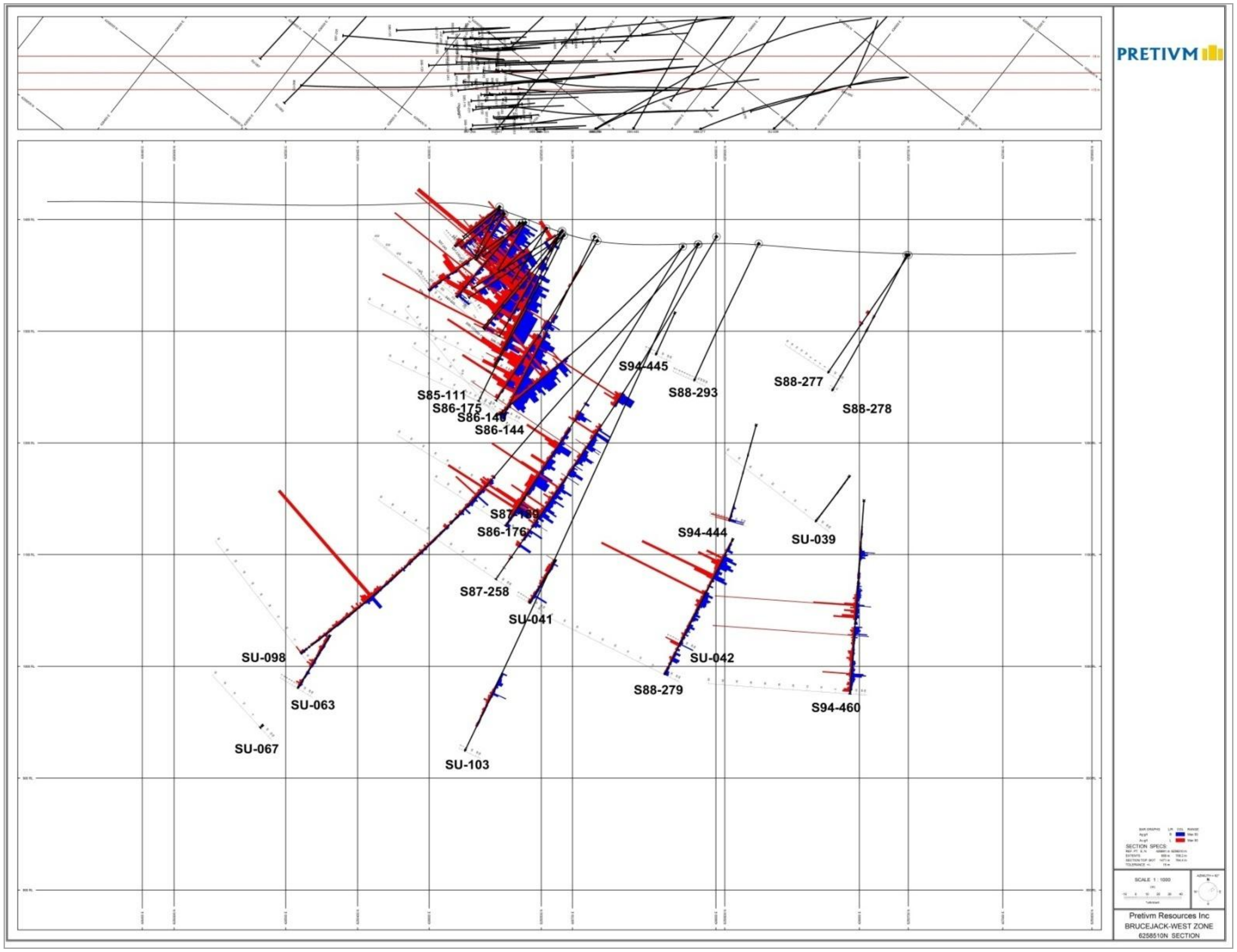
Within the higher grade core area, gold and copper grades correlate closely with one another. The copper-to-gold ratio tends to be slightly higher closer to the phyllic-propylitic transitional areas. In the low-grade peripheral shells, the copper-to-gold grades tend to be the highest. The gold and copper distribution is remarkably smooth and continuous with grades decreasing very gradually outward 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.

9.2.1 *WEST ZONE*

The following descriptions (Sections 8.2.1 through 8.2.9) of the mineralization of the West, West Zone Footwall, Bridge, Bridge Halo, Galena Hill, Shore, SG, Gossan Hill and VOK Zones of the Brucejack Property were provided by Mr. Ron Burk, Chief Geologist at Silver Standard, and Mr. Ken McNaughton, Vice President and Chief Exploration Officer, Pretivm (former Senior VP Exploration of Silver Standard) in the form of an internal company report, dated December 9 2009 and recent verbal communication.

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.

Figure 9.2 Section 6258510N of the West Zone, Brucejack Property



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.

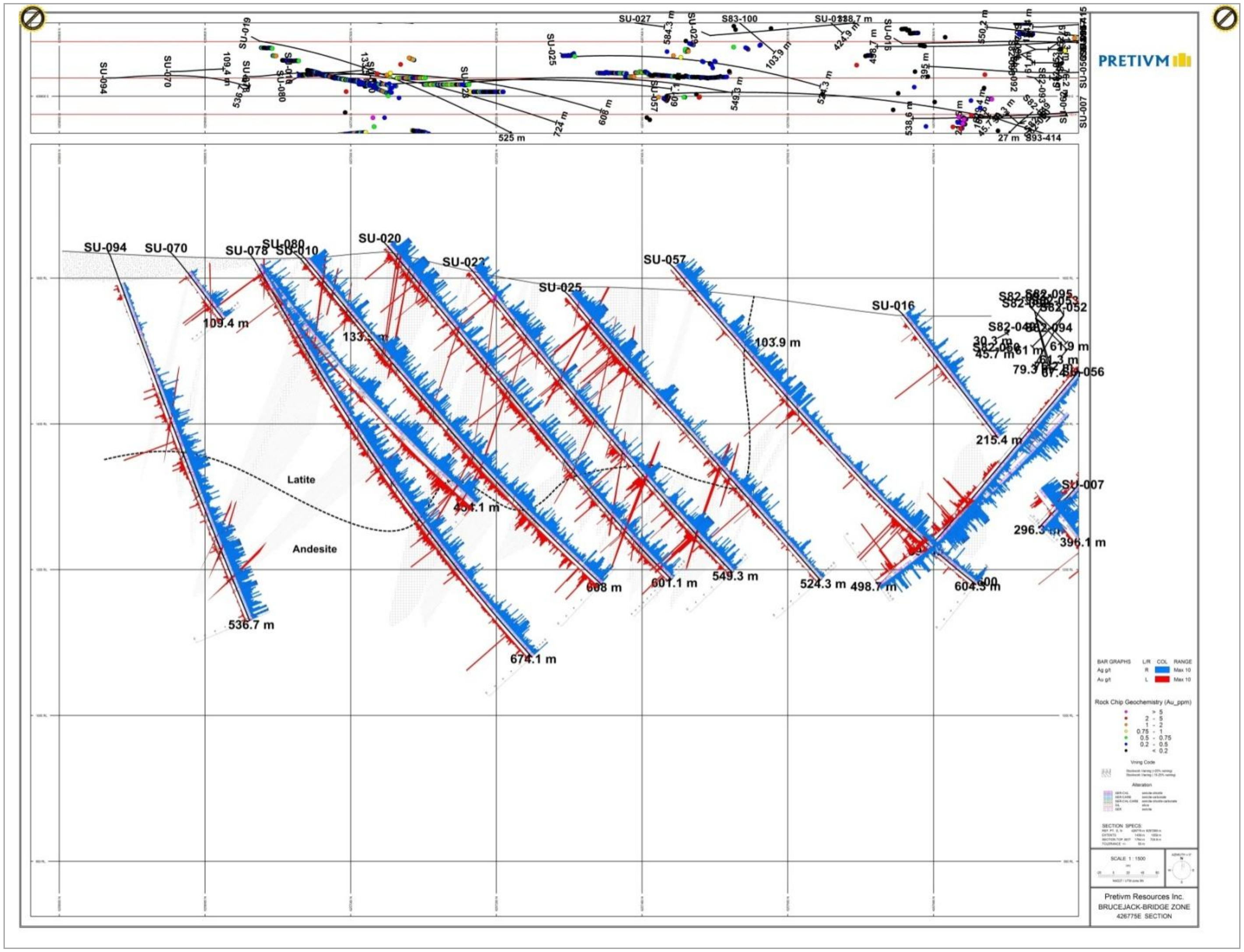
9.2.2 WEST ZONE FOOTWALL ZONE

The West Zone Footwall Zone is located along the entire footwall of the West Zone. It lies approximately 50 m to 200 m south-west of the West zone and was intersected by holes SU-63, SU-98 and SU-100. These holes cover an area 600 m long. Resources are currently classified as Inferred only.

9.2.3 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, decision was made 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.

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



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 drilling of 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 sulphosalt(s).

9.2.4 Bridge Zone Halo Zone

The Bridge Zone Halo Zone was modelled separately from the Bridge Zone as a low grade halo in order to ensure that all potentially economic mineralization was captured for mineral resource estimation. A secondary mineralization halo for the Bridge Zone was subsequently modelled using a 0.20 g/t Au grade shell.

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, indicated that there was a system of E-W and NE-SW-trending quartz veins and quartz stockworks at Galena Hill that, as a whole, defined a zone of hydrothermal alteration and mineralization that is at least 460 m long and 300 m wide.

Rather than target the larger quartz veins, which locally contain high-grade gold + silver mineralization on surface, a decision was made to test for the potential of a low-grade, bulk-mineable deposit. This was done with eight 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 mudstone. A few holes intersected rhyolitic dikes and one hole (SU-05) 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.

9.2.6 VALLEY OF KINGS ZONE

The VOK Zone (Figure 9.5) was discovered by Esso in 1981 and was previously referred to as the Electrum Zone. It lies between the Bridge and Galena Hill Zones. Very little work has been completed on this zone, but what is currently known is that there are multi-kilogram intersections in parallel zones. The best intersection to date was in hole SU-12 which yielded 1.5 m at 16,949 g/t Au and 8,697 g/t Ag. Resources are all currently classified as Inferred in this zone.

9.2.7 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 andesite with a strike length of roughly 530 m and a maximum width of 50 m (Figure 9.6). 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-eastward 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 to 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.

Figure 9.4 Section 426741E of the Galena Hill Zone, Brucejack Property – Looking West

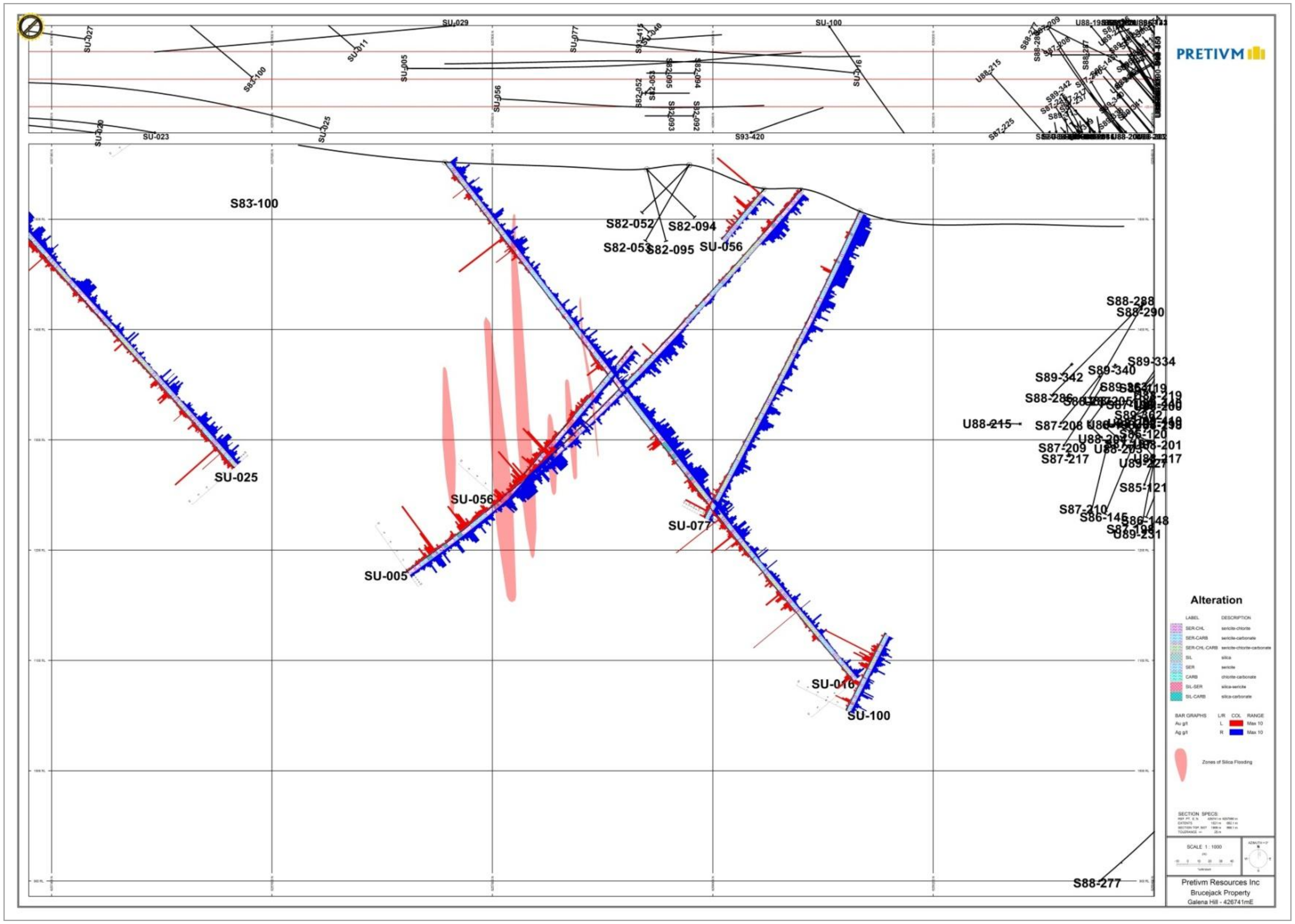


Figure 9.5 Section 426586E Valley of Kings

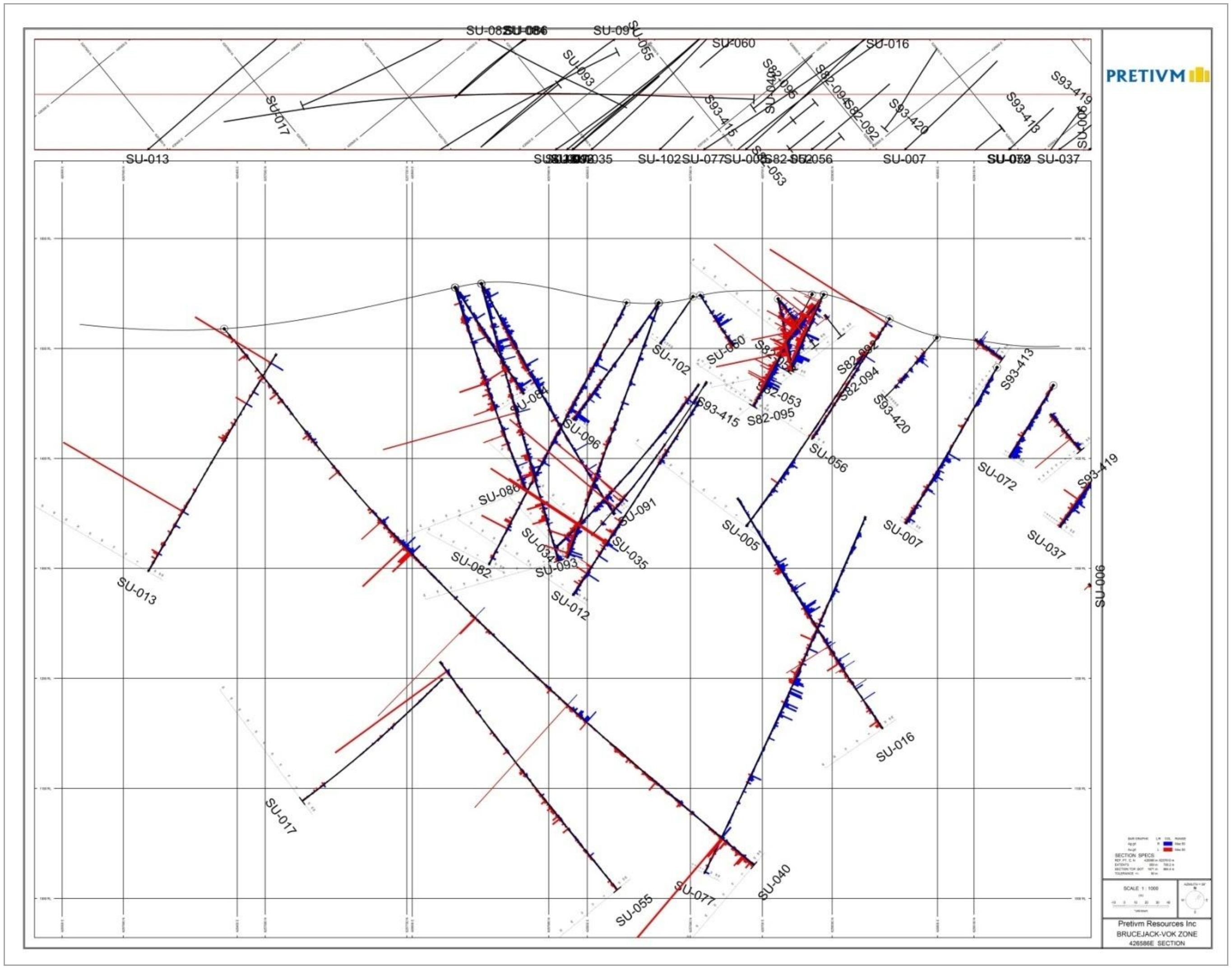


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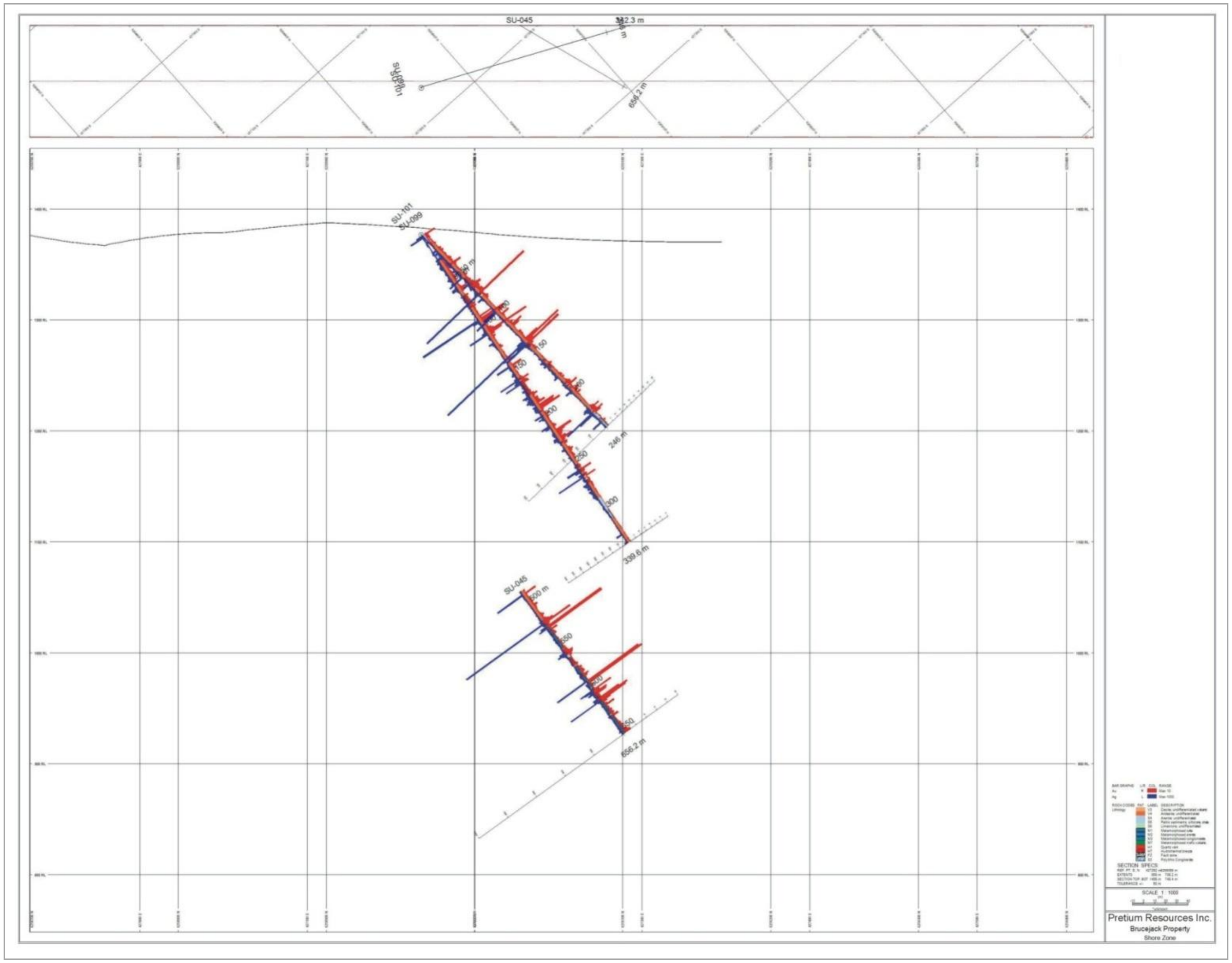
Figure 9.6 Section 427252E of the Shore Zone, Brucejack Property – Looking West-Northwest



9.2.8 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 rock that occurs 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 9.7) – likely re-deposited 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 drill holes; 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.

9.2.9 GOSSAN HILL ZONE

The mineralized zone known as Gossan Hill (Figure 9.8), is a circular area, about 400 m in diameter, of intense quartz-sericite-pyrite alteration developed in Jurassic andesite 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 tetrahedrite, pyragyrite, and polybasite.

Figure 9.7 Section 426146E of the SG Zone, Brucejack Property – Looking West

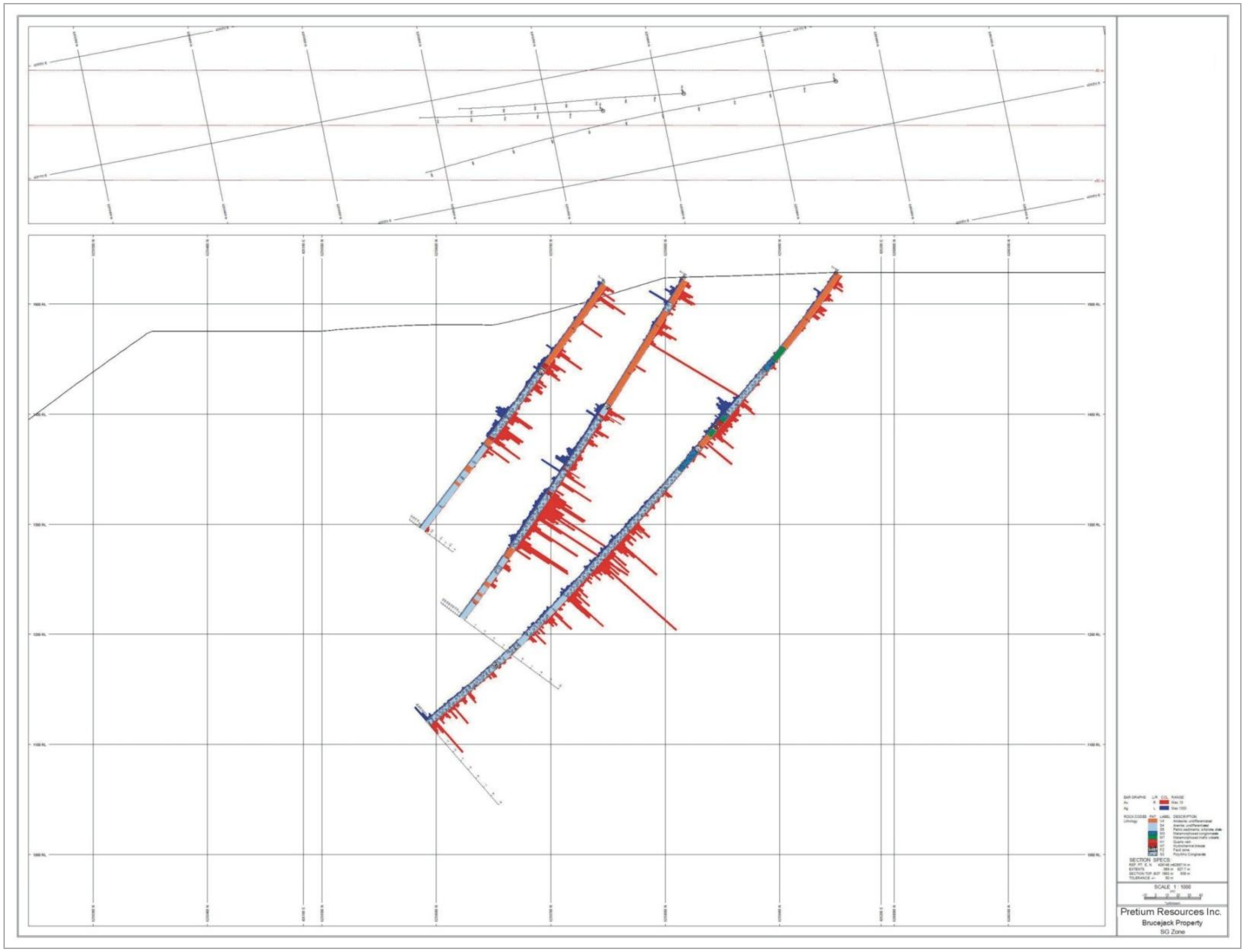
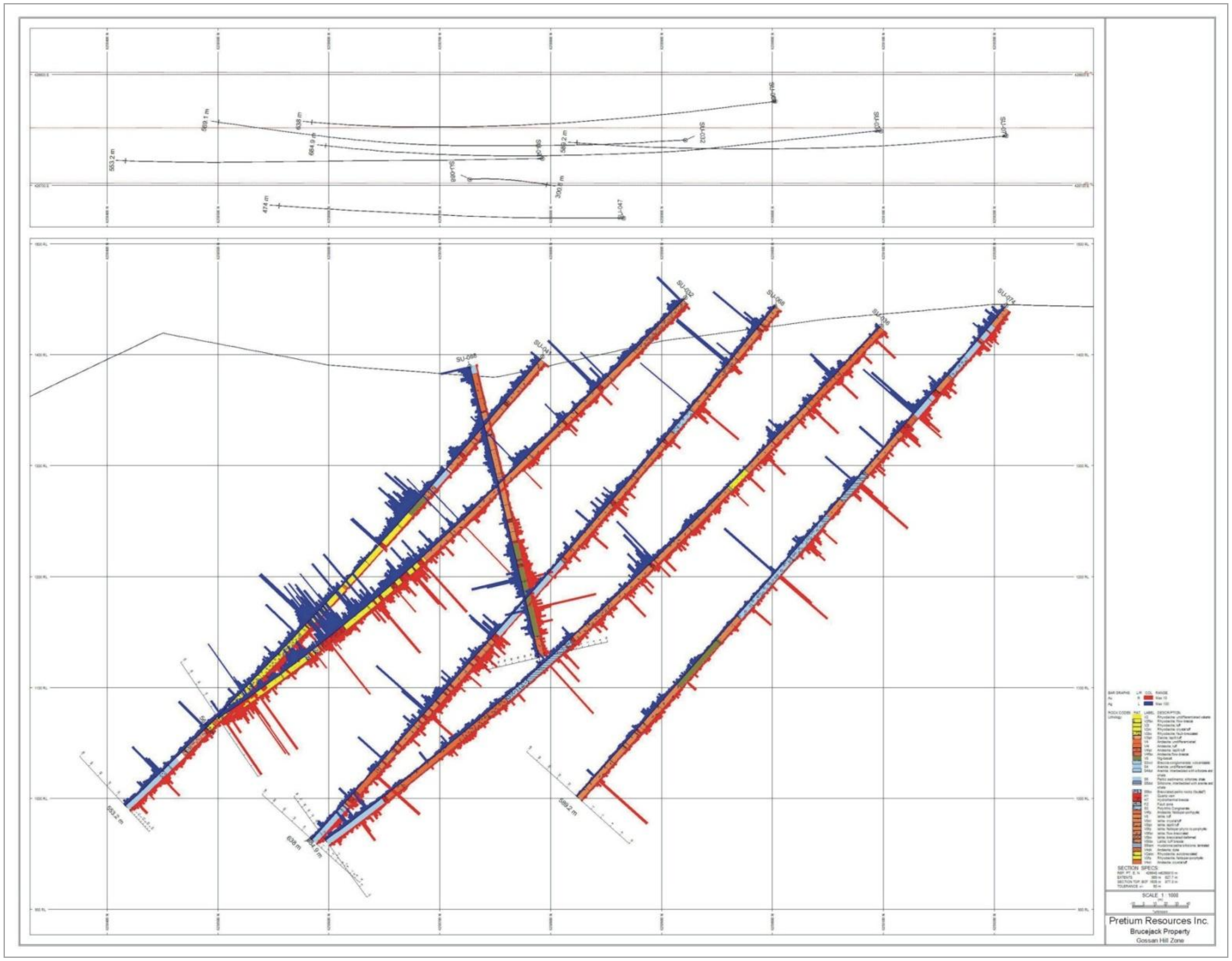


Figure 9.8 Section 426648E Gossan Hill Zone



10.0 EXPLORATION

The 2010 program included mapping on the Brucejack Property, preliminary geotechnical and environmental studies, layout, and permit initiation of the access road to Brucejack Lake, geotechnical drilling as well as PQ metallurgical core drilling at Brucejack.

The preliminary geotechnical and environmental studies are referenced in Section 17 of this report. Metallurgical testing is currently in progress, with no new results to report.

11.0 DRILLING

In 2010, a total of 33,400 m of drilling was completed in 72 drill holes. The 2010 results defined a new area of mineralization in the West Zone, encountered further high-grade gold and silver mineralization in the Galena Hill Zone and expanded the known mineralization in the West, Galena Hill, Bridge and Shore Zones. A drill plan map is presented in Figure 11.1. Table 11.1 presents results of the significant intersections in 2010.

11.1 WEST ZONE

Drilling defined a new area of mineralization in the footwall of the West Zone. Intersections in holes SU-63, SU-66, SU-67, SU-98 and SU-100 define the footwall mineralization measuring approximately 120 m x 500 m. The intersection in SU-98 was less than 50 m from the historical West Zone, which was defined by over 750 surface and underground drill holes and over 5,000 m of underground workings.

11.2 GALENA HILL ZONE/VALLEY OF KINGS ZONE

The Galena Hill Zone, located 500 m south of the West Zone, is host to disseminated gold-silver mineralization together with structurally-controlled high-grade veins. Drilling continued to confirm the location of the high-grade structures intersected in 2009 and 2010. The highlight from the latest hole, SU-106 intersected three bands of mineralization including 0.69 m of 1,710 g/t Au and 1,080 g/t Ag.

This intersection encountered the same zone as defined by the high-grade intercepts in holes SU-12, SU-29, SU-40 and SU-84 previously reported from the 2009 and 2010 programs and included in Table 11.1.

Drilling in 2010 expanded the Galena Hill Zone by 100 m to the northeast and 250 m to the southwest. Galena Hill is open to the east and to depth.

11.3 BRIDGE ZONE

The Bridge Zone, which exhibits porphyry-style gold-silver mineralization, measures approximately 600 m x 900 m, roughly three times the area defined in the 2009 drill program. Holes SU-92, SU-94 and SU-95 expanded the zone 200 m further south than the area defined in 2009 drilling. Holes SU-64, SU-87 and SU-90 show that the zone remains open to the east. The highlight was hole SU-87, which intersected 168 m averaging 1.09 g/t Au and 4.04 g/t Ag and ended in mineralization.

Figure 11.1 2010 Brucejack Property Diamond Drill Plan

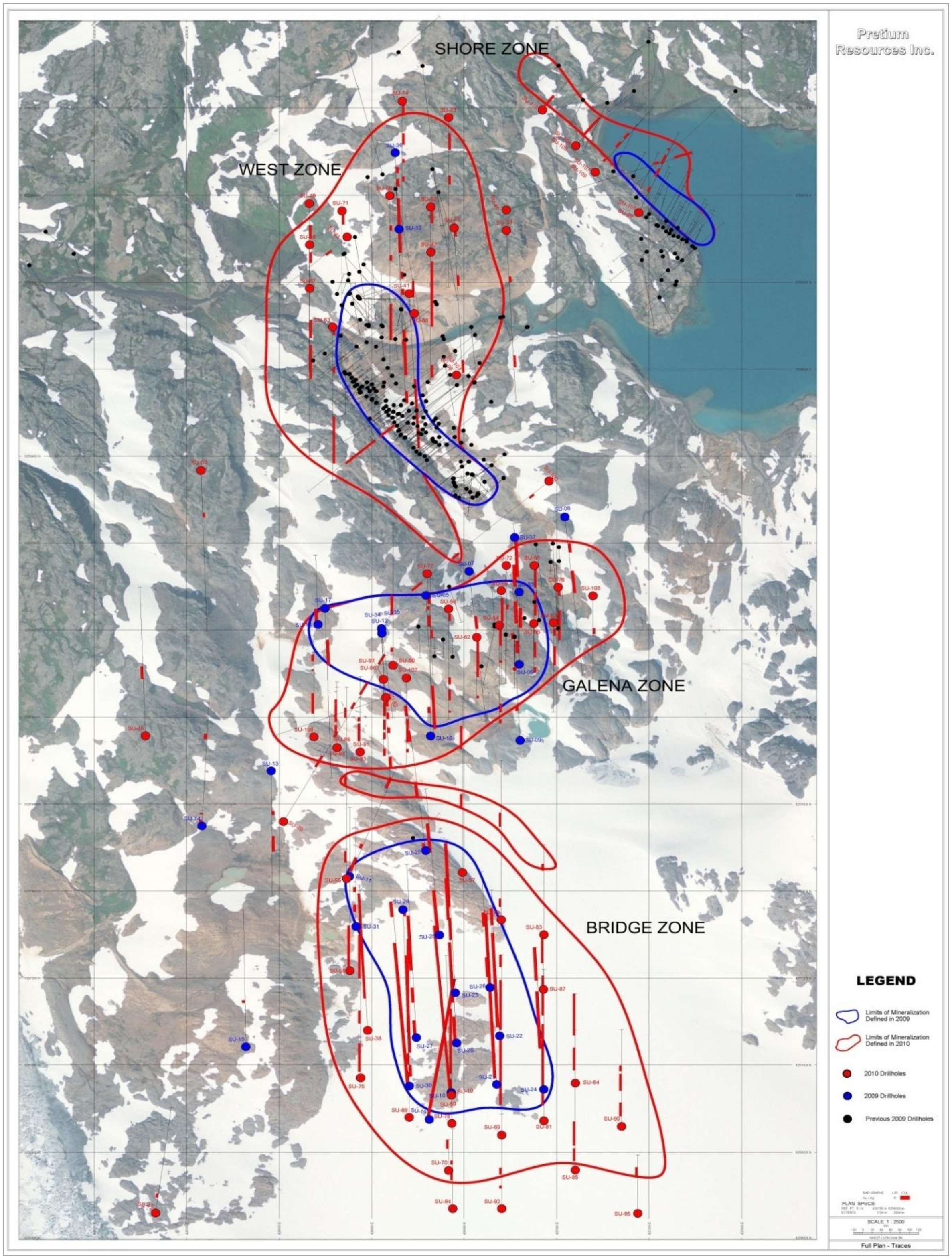


Table 11.1 Significant Drill Intercepts from 2010 Drilling

Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	Zone
SU-38	114.11	489.63	375.52	0.54	3.70	Bridge
SU-39	5.00	17.00	12.00	4.30	12.29	West
	165.00	186.58	21.58	0.58	83.86	
	447.50	490.53	43.03	0.71	21.88	
SU-40	223.50	269.50	46.00	2.16	11.30	Galena Hill/VOK
	346.00	369.50	23.50	1.42	5.22	
	464.45	488.62	24.17	1.40	20.28	
	522.50	572.00	49.50	0.67	1.60	
	647.50	687.93	40.43	2.84	32.75	
SU-41	307.00	553.20	246.20	0.77	8.75	West
SU-42	0.70	32.00	31.30	0.78	44.27	West
	169.50	208.00	38.50	0.55	28.85	
	473.73	520.00	46.27	0.91	40.30	
SU-43	71.50	95.50	24.00	0.63	4.88	West
SU-44	55.50	69.50	14.00	0.57	2.20	Jewel
SU-45	521.77	540.00	18.23	4.35	51.23	Shore
	594.50	656.23	61.73	1.69	42.12	
SU-46	204.00	248.50	44.50	0.68	1.90	Waterloo
SU-47	75.00	109.00	34.00	0.65	7.16	West
	197.50	221.50	24.00	2.07	39.66	
	403.76	448.50	44.74	3.26	67.25	
SU-48	144.50	195.50	51.00	0.62	7.46	SG
	290.50	412.00	121.50	0.87	3.02	
	548.00	555.50	7.50	1.38	11.66	
SU-49	79.50	126.00	46.50	0.66	3.62	West
	550.00	590.50	40.50	1.04	2.81	
SU-50	152.00	166.50	14.50	1.35	2.19	West
SU-51	10.50	79.98	69.48	0.67	3.03	SG
	311.00	330.00	19.00	0.69	1.05	
SU-52	2.06	58.00	55.94	0.45	10.43	West
	194.49	428.31	233.82	2.26	12.54	
SU-53	21.50	45.50	24.00	2.39	49.11	Galena Hill
	83.50	152.50	69.00	0.53	6.55	
SU-54	9.14	133.50	124.36	2.04	27.13	Galena Hill/VOK
	280.50	310.00	29.50	0.58	5.90	
	336.50	349.50	13.00	1.78	14.15	
SU-55	57.50	96.50	39.00	1.61	3.60	Bridge
	331.50	367.00	35.50	0.80	4.33	
	575.00	611.00	36.00	1.20	3.04	
SU-56	16.70	44.00	27.30	0.75	1.68	Galena Hill
	245.50	270.50	25.00	0.59	4.54	

Table continues...

Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	Zone
	289.50	327.80	38.30	0.71	4.77	
	364.70	370.33	5.63	1.63	12.55	
SU-57	233.13	279.50	46.37	1.02	74.03	Bridge
	442.00	524.50	82.50	2.00	9.50	
SU-58	0.11	162.00	161.89	2.09	7.97	Bridge
SU-59	26.50	111.50	85.00	1.37	15.53	Galena Hill
SU-60	48.00	62.71	14.71	0.74	6.84	Bridge
	180.90	250.00	69.10	0.74	13.16	
SU-61	0.00	81.00	81.00	0.65	41.66	Galena Hill
	96.02	180.00	83.98	1.02	4.90	
SU-62	14.00	143.00	129.00	1.48	14.53	Galena Hill
	210.00	218.54	8.54	1.49	14.57	
SU-63	4.20	28.50	24.30	0.72	9.25	West
	143.00	166.50	23.50	0.92	34.02	
	319.00	487.49	168.49	0.80	4.81	
SU-64	170.20	318.80	148.60	0.72	7.50	Bridge
SU-65	65.50	121.00	55.50	1.46	10.59	Galena Hill
	116.00	119.50	3.50	11.87	35.95	
	156.50	175.77	19.27	0.91	6.66	
SU-66	26.89	64.55	37.66	0.88	1.49	West
	358.34	444.00	85.66	1.16	5.39	
SU-67	114.00	177.55	63.55	0.76	4.45	West
	250.00	324.42	74.42	2.17	16.56	
SU-68	20.20	32.50	12.30	3.88	21.65	West
	107.00	125.50	18.50	2.41	8.80	
	113.00	118.24	5.24	5.85	25.61	
	213.50	232.36	18.86	0.91	8.27	
	387.50	516.50	129.00	0.89	10.11	
SU-69	108.50	351.61	243.11	0.85	8.79	Bridge
	377.00	554.00	177.00	1.07	10.40	
	600.94	644.95	44.01	0.79	8.82	
SU-70	77.50	109.42	31.92	0.88	5.45	Bridge
SU-71	5.68	48.74	43.06	0.51	2.28	West
	519.50	523.93	4.43	6.73	5.69	
SU-72	140.70	157.00	16.30	0.65	7.64	Galena Hill
SU-73	57.00	116.50	59.50	2.45	13.17	West
	208.66	240.50	31.84	0.84	22.89	
SU-74	9.00	51.00	42.00	0.76	7.50	West
	115.00	155.00	40.00	1.08	13.68	
	207.00	220.50	13.50	4.85	13.49	
	269.50	286.50	17.00	1.31	5.84	

Table continues...

Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	Zone
SU-75	20.70	123.00	102.30	0.71	2.38	Bridge
	194.00	240.50	46.50	0.55	3.71	
	355.00	510.00	155.00	1.15	3.55	
SU-76	4.50	31.50	27.00	1.05	18.87	Galena Hill
	84.50	189.50	105.00	2.27	20.78	
SU-77	304.50	351.00	46.50	1.18	8.90	Galena Hill
SU-78	118.50	203.50	85.00	0.57	3.78	Bridge
	229.00	527.50	298.50	0.73	7.12	
	621.50	724.50	103.00	1.00	6.81	
SU-79	46.50	75.50	29.00	0.59	17.06	Bridge
	332.50	382.50	50.00	1.31	53.06	
SU-10 (EXT.)	7.00	608.08	601.08	0.76	7.91	Bridge
SU-81	71.00	444.50	373.50	0.64	12.80	Bridge
	485.00	666.50	181.50	0.53	8.54	
SU-82	227.00	239.00	12.00	2.13	4.78	Galena Hill
SU-83	430.50	476.00	45.50	0.66	2.56	Bridge
SU-84	43.00	64.00	21.00	0.58	5.81	Galena Hill
	92.00	155.50	63.50	1.01	12.61	
	198.08	198.52	0.44	5480.00	2140.00	
SU-85	48.90	116.50	67.60	0.89	7.10	Bridge
	132.00	262.50	130.50	0.83	9.29	
	541.89	664.50	122.61	0.55	14.99	
SU-86	82.50	122.00	39.50	1.38	7.09	Galena Hill
	190.50	226.50	36.00	2.01	17.74	
	238.50	260.80	22.30	1.33	7.05	
SU-87	175.50	343.20	167.70	1.09	4.04	Bridge
SU-88	144.00	288.50	144.50	0.95	7.73	West
SU-89	99.50	161.50	62.00	1.17	2.96	Bridge
	188.56	229.00	40.44	0.84	2.45	
	304.00	638.25	334.25	1.02	4.76	
SU-90	69.00	160.00	91.00	0.83	6.82	Bridge
	241.00	353.00	112.00	0.91	22.66	
	371.00	408.50	37.50	0.56	5.20	
SU-91	43.00	102.98	59.98	1.64	11.02	Galena Hill
SU-92	135.76	154.00	18.24	0.72	3.73	Bridge
	230.00	275.00	45.00	0.72	7.07	
SU-93	73.10	88.05	14.95	0.52	27.61	Galena Hill
	105.80	146.10	40.30	1.26	6.50	
	164.00	191.00	27.00	1.17	10.44	
	205.79	232.43	26.64	1.32	6.66	

Table continues...

Hole	From (m)	To (m)	Interval (m)	Au (g/t)	Ag (g/t)	Zone
SU-94	191.50	240.00	48.50	0.61	4.77	Bridge
	269.00	270.50	1.50	34.70	18.60	
	298.50	324.00	25.50	0.84	12.45	
SU-95	316.50	387.00	70.50	0.67	21.76	Bridge
	387.00	410.50	23.50	0.76	25.01	
SU-96	138.00	164.00	26.00	0.57	18.41	Galena Hill
	226.00	249.02	23.02	0.87	10.30	
SU-97	125.41	199.50	74.09	0.49	4.75	Galena Hill
	219.50	266.12	46.62	1.30	8.77	
	289.71	325.00	35.29	1.83	6.18	
SU-98	275.00	339.00	64.00	0.75	5.79	West (met hole not sampled from 0 to 275 m)
	359.00	488.50	129.50	1.40	16.60	
SU-99	91.50	112.00	20.50	1.52	45.76	Shore
	158.50	193.50	35.00	0.73	22.80	
	212.00	240.50	28.50	0.65	10.14	
SU-100	69.50	89.00	19.50	0.49	1.74	West
	398.10	427.71	29.61	1.56	7.03	
	561.00	605.34	44.34	0.85	3.02	
SU-101	135.04	142.50	7.46	3.58	177.68	Shore
	190.00	205.00	15.00	0.57	11.95	
	221.65	229.50	7.85	0.86	46.71	
SU-102	133.00	208.50	75.50	0.64	6.88	Galena Hill
	231.00	277.50	46.50	0.59	6.53	
SU-104	38.70	79.50	40.80	0.63	7.80	Shore
	94.50	117.00	22.50	0.70	10.83	
SU-105	87.50	137.00	49.50	0.71	3.06	Shore
	198.50	212.00	13.50	2.65	63.34	
SU-106	83.00	155.47	72.47	1.37	15.00	Galena Hill
	192.50	241.12	48.62	1.06	25.75	
	269.58	294.00	24.42	0.77	7.31	
SU-107	103.00	125.93	22.93	0.70	24.31	Shore
	147.50	174.12	26.62	1.25	23.56	
SU-108	116.00	137.00	21.00	0.56	7.18	Galena Hill
SU-109	54.00	66.00	12.00	0.89	7.43	
	173.74	200.80	27.06	0.92	20.87	Shore
SU-110	15.00	58.00	43.00	1.34	35.05	Shore

11.4 SHORE ZONE

Eight holes were completed on the Shore Zone, which expanded it to the northwest; the zone remains open and to depth.

In addition to work completed on the West, Bridge, Galena Hill, and Shore Zones, other targets were defined on the property that will require future follow-up sampling and drilling.

The 2010 drilling program was conducted by Radius Drilling Corp. and Matrix Diamond Drilling. There were an average of seven, and a maximum of nine drill rigs on-site during the program.

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%.

Towards the end of the drilling campaign, McElhanney surveyed drill hole collars using a differential GPS.

Crews were de-mobilized from the site for the winter season on September 29, 2010. Most portable equipment was stored in one of several winterized buildings on site. All of the tents were transported by air to Stewart for storage, along with the core from a number of key drill holes.

P&E believes that drilling has been conducted using industry best practice guidelines.

12.0 SAMPLING METHOD AND APPROACH

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

The geologist logged drill core sample intervals**,** based on the assumption that the intervals do not cross geologic contacts. The maximum sample length was two metres. Within any given geologic unit, any 1.5 m sample intervals were extended or reduced to coincide with a given geologic contact. Sample lengths rarely exceeded two metres, or were shorter than 0.5 m. Averaging length was 1.52 m.

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.

In P&E's opinion, 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 2010 program on the Brucejack Project used ALS Chemex as the principal laboratory. 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.

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, (-10 mesh). Samples were riffle split and 500 g were pulverized to 85% passing 75 µm (-200 mesh). The remaining coarse reject material was returned to Pretivm for storage in their Smithers warehouse for possible future use.

Gold was determined using fire assay on a 30 gram aliquot with an atomic absorption (AA) finish. A 33-element package was completed using a four acid digest and ICP-AES analysis, which included the silver analyses.

In P&E's opinion, the sample preparation, security, and analytical procedures are satisfactory.

14.0 DATA VERIFICATION

14.1 SITE VISIT AND INDEPENDENT SAMPLING 2010

The Brucejack property was visited by Mr. Fred Brown, CPG, Pr.Sci.Nat. from September 3 to 5, 2010. Independent verification sampling was done on diamond drill core, with ten samples distributed in ten holes collected for assay. An attempt was made to sample intervals from a variety of low-grade and high-grade material. The chosen sample intervals were then sampled by taking 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 is provided in Figure 14.1 and Figure 14.2.

Figure 14.1 P&E Independent Site Visit Sample Results for Gold

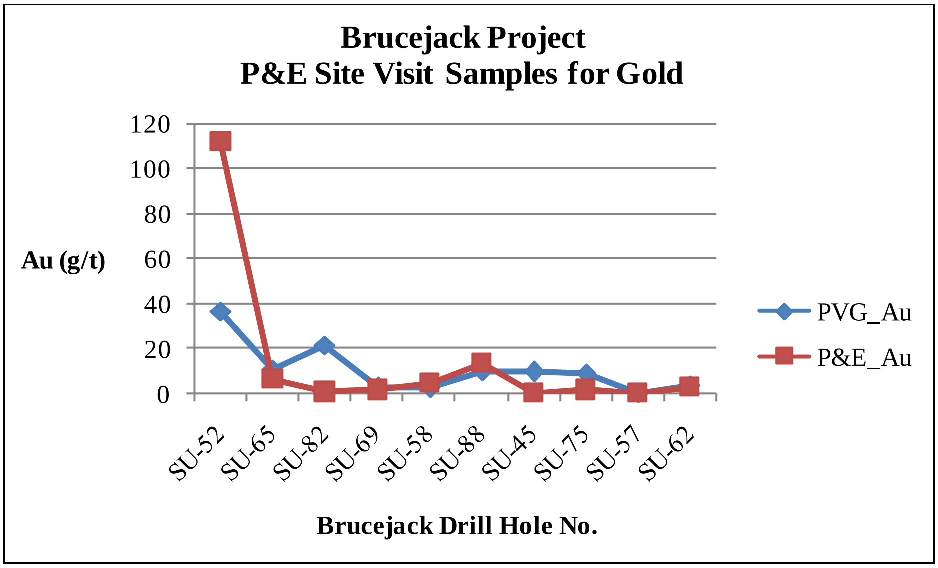
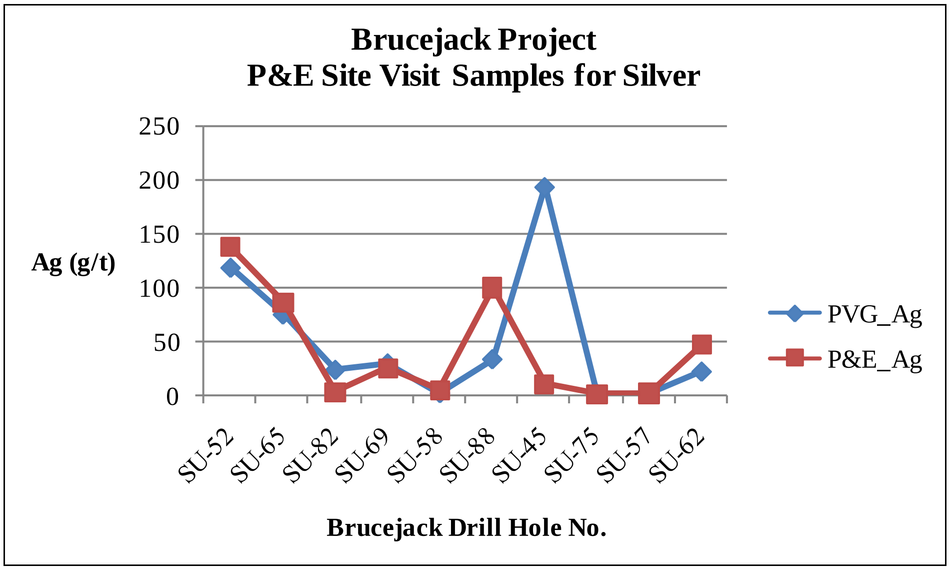


Figure 14.2 P&E Independent Site Visit Sample Results for Silver



14.2 Pretivm Quality Control

The QA/QC program was maintained throughout the 2010 drilling campaign. Certified reference material standards named CDN ME-1 and CDN ME-3 were purchased from CDN Resource Laboratories Ltd. in Langley, BC. Both of these standards were certified for gold, silver and copper. One standard sample, one blank sample and one field duplicate sample (¼ split core) were inserted every 20 samples. In addition, the lab implemented their own internal QC, which included standards, blanks and both coarse reject and pulp duplicates.

14.3 2010 Data Verification Results

Silver Standard monitored the QC program on a real-time basis throughout 2010; any standards failing the QC protocols were re-run with the respective samples. P&E received all the data for the 2010 drilling and verified the performance of the standards, blanks and duplicates.

14.3.1 Performance of Certified Reference Material

Standard ME-1 had 660 data points for gold and silver. None of the data points fell outside three standard deviations from the mean, although several fell between two and three standard deviations. All data points for all elements passed the QC and no action was required.

The ME-3 standard had 643 data points for gold and silver. All data points passed the QC.

14.3.2 Performance of Blank Material

The blank material used for the 2008, 2009 and 2010 drill programs was ¾” crushed granite sold by Imasco Minerals as landscape material.

There were 1,115 blank samples analyzed during the 2010 program. The average gold grade in the blanks was 0.005 g/t Au. Six high values were investigated and deemed to be sampling errors.

For silver, the average grade of the blank material was 0.35 g/t Ag. There was only one high value requiring investigation, and no further action was required

14.3.3 2010 Duplicate Statistics

For the 2010 drill program, there were 1,309 field core duplicate pairs, 843 pulp duplicate pairs analyzed for gold and 24 pulp duplicate pairs analyzed for silver. There were no coarse reject duplicates done.

Data for the gold duplicate types were graphed using simple scatter graphs. At the field duplicate level, the precision for gold was poor. At the pulp level, the correlation was excellent.

The silver duplicates yielded poor precision at the field duplicate level and 1:1 precision at the pulp duplicate level.

14.3.4 Check Samples Assayers Canada

Approximately 524 of the 2010 pulps were sent to Assayers Canada Ltd. (Assayers) in Vancouver, BC as a check on the principal lab. Results were graphed for gold and silver. Precision on the gold pulps was satisfactory. Precision on the silver pulps was excellent.

P&E considers that the data used in this resource estimate are of excellent quality.

15.0 ADJACENT PROPERTIES

Within the adjacent KSM property there are four notable copper-gold mineral deposits, named the Kerr, Mitchell, Sulphurets and Iron Cap Zones. The Iron Cap Zone was discovered in 2010, after the release of the April 2010 Preliminary Feasibility Study. All of these occurrences are situated within the claim holdings currently owned by Seabridge.

Seabridge acquired the property from Placer Dome in June 2000. In 2009, Resource Modelling Inc. completed updated NI 43-101-compliant resource estimates for the Kerr, Sulphurets, and Mitchell Zones. The Mitchell resource estimate was reported in a news release dated March 11, 2009, and the Kerr and Sulphurets resource estimates were reported in a March 25, 2009, news release.

In June 2009, a Preliminary Assessment estimated a 30-year mine life recovering 19.3Moz of gold, 5.3 Blb of copper, 2.8 Moz of silver, and 1.9 Mlb of molybdenum. In April 2010, Seabridge published the results of a Preliminary Feasibility Study.

On February 8, 2011, Seabridge announced an Indicated resource containing 5.1 Moz of gold and 1.7 Blb of copper for the Iron Cap Zone at its 100% owned KSM project. The Iron Cap Zone is immediately adjacent to the Mitchell deposit. The Indicated resource is flanked by a halo of Inferred resources containing an additional 3.4 Moz of gold and 1.3 Blb of copper.

On May 2, 2011, Seabridge announced an update to the April 2010 Preliminary Feasibility Study. The updated Preliminary Feasibility Study includes the Kerr, Sulphurets and Mitchell Zones, as well as the recently modelled and estimated Iron Cap Zone. The Proven and Probable reserves as per the May 2011 updated Preliminary Feasibility Study are presented in Table 15.1.

Table 15.1 May 2011 Proven and Probable Reserves Seabridge KSM Property

		Diluted Grade				Contained Metal			
Pit	Ore (Mt)	**Au (g/t)**	**Cu %**	**Ag (g/t)**	**Mo (ppm)**	**Au (Moz)**	**Cu (Mlb)**	**Ag (Moz)**	**Mo (Mlb)**
Mitchell	1,466.5	0.608	0.165	3.04	61.2	28.7	5,320	143	198
Iron Cap	334.1	0.420	0.202	5.46	48.4	4.5	1,490	59	36
Kerr	212.7	0.254	0.460	1.28	0.0	1.7	2,154	9	0
Sulphurets	179.1	0.621	0.259	0.61	59.8	3.6	1,021	3	24
Total Proven & Probable	**2,192.4**	**0.546**	**0.207**	**3.04**	**53.2**	**38.5**	**9,985**	**214**	**257**

The Proven and Probable reserves of 38.5 Moz of gold (2.192 Bt at 0.546 g/t Au) are derived from total Measured and Indicated Resources of 45.3 Moz of gold (2.549 Bt at 0.55 g/t Au) and include allowances for mining losses and dilution.

The mine life is currently estimated at 52 years at an average mill throughput of 120,000t/d.

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 Brucejack Property.

In October 2010, Wardrop completed a preliminary assessment (PA) of the Snowfield-Brucejack Project.

Based on the results of the PA, it was recommended that Pretivm continue with the next phase of the project in order to identify opportunities and further assess viability of the project. The full PA can be consulted at www.sedar.com.

The effective date of the mineral resource estimate of the adjacent to Brucejack property, Snowfield deposit is July 27, 2010. The resource estimate is presented in Table 15.1.

Table 15.2 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.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 METALLURGICAL TESTWORK REVIEW

16.1.1 INTRODUCTION

The Brucejack project consists of several mineralization zones mainly including the West Zone, VOK Zone, Galena Hill Zone, and the Gossan Hill (R8) Zone. The key valuable metals in the mineralization of the Brucejack project are gold and silver. The testwork conducted on the Brucejack mineralization includes gold/silver bulk flotation, gravity concentration and cyanidation. According to the information available for review, several testing programs have been carried out to investigate the metallurgical performance of the mineralization, including the recent testwork during 2009 and early 2011 and historical testwork conducted between 1988 and 1990 for the Feasibility Study by Cominco Engineering Services Ltd. (CESL), March 1990.

Wardrop has reviewed the testwork and summarizes the results in the following sections.

16.1.2 HISTORICAL TESTWORK

The historical testwork used the composite samples collected from the West Zone and R8 Zone. The Feasibility Study report indicated that the mineralization consists of apparently erratic veins and lenses containing metallic gold (native gold) and metallic silver (native silver), together with a variety of sulphide minerals in a quartz rich environment within a zone of altered volcanic. The gold occurs in a range from relatively coarse grains (40 – 100 µm) to fine grains locked in either pyrite or quartz gangue. The silver occurs in small amounts as metallic form, while most of the silver is intimately associated with or a component of the various sulphide minerals. The major minerals in the samples are listed in Table 16.1.

Table 16.1 Mineralogical Assessment in Samples Tested

Mineral	Pyrite	Sphalerite	Tetrahedrite	Jalpaite	Ruby Silver
Content, %	9.7	0.5	0.1	0.1	0.05
Mineral	Galena	Chalcopyrite	Native Gold	Native Silver	Gangues
Content, %	0.05	Trace	Trace	Trace	89.5

The metallurgical testing included gravity separation, flotation, cyanidation and roasting pre-treatment.

The testwork indicated that gravity separation would recover a significant portion of the contained gold. Cyanide leaching on the gravity tailings produced good overall gold recoveries, but poor silver recoveries (<40%). The report indicated that the poor silver recoveries were attributed to the silver occurrence in the form of relatively insoluble silver sulphides such as tetrahedrite and proustite.

The gold and silver minerals responded well to the flotation concentration. The reagent scheme screening tests showed that 3418A would improve the recovery of the precious metals and reduce concentrate mass pull. Also the testwork indicated that the addition of lime to increase slurry pH from 8.1 to 10.5 could substantially reduce the concentrate weight from 6.8% to 1.5%.

Similar metallurgical performances were produced from the West Zone samples and the R8 Zone sample. It appeared that the R8 mineralization might require a finer primary grinding.

As projected by CESL, the overall gold and silver recoveries by a combined process of gravity separation and flotation would be approximately 89% and 83% for the West Zone mineralization and 88% and 85% for the R8 Zone. The projections for the blend of the two zones are detailed in Table 16.2.

Table 16.2 Metallurgical Performance Projection by CESL – Blend (1990)

Products	Mass Recovery (%)	Grade (g/t)		Recovery (%)	
		Au	Ag	Au	Ag
Gravity Concentrate	0.2	1,139	3,966	22.5	1.1
Flotation Concentrate	4.5	143.4	12,665	66.4	82.4
Tailings	95.3	1.2	119.7	11.1	16.5
Head	100	9.3	777.6	100.0	100.0

16.1.3 2009-2011 Testwork

The Metallurgical Division at Inspectorate America Corp. (Inspectorate), previously Process Research Associates Ltd. (PRA), carried out preliminary metallurgical testwork investigating the metallurgical performance of Brucejack mineralization since 2009 to early 2011. Established in 1992, PRA is an industrial research laboratory 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 test results and procedures, including sample preparation and analysis, are presented in the data reports by PRA released in July 2010 and April 2011.

SAMPLE DESCRIPTION

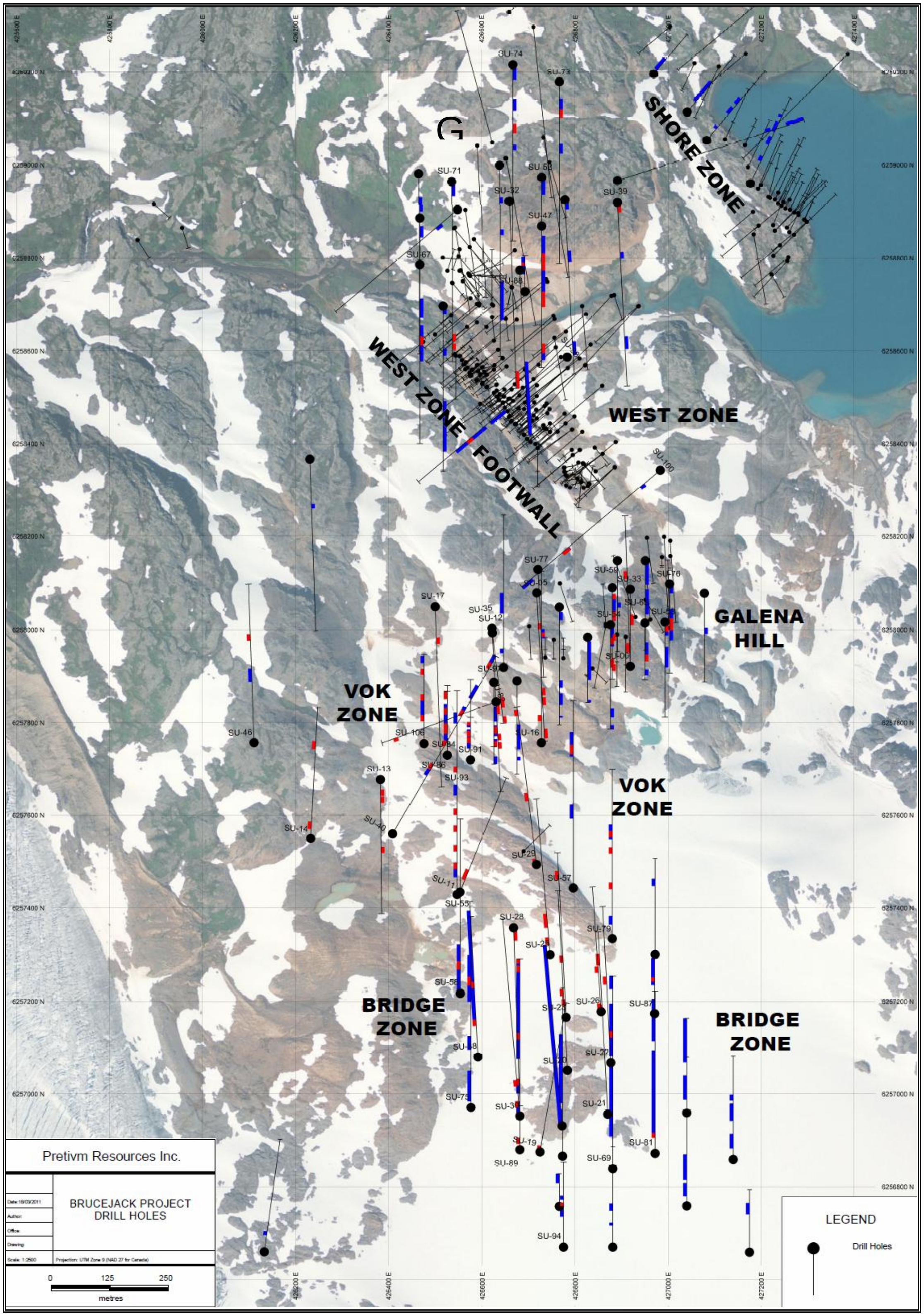
2009-2010 Test Samples

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 holes distribution is presented in Figure 16.1.

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

Figure 16.1 Drill Hole Distribution



2010-2011 Tests Samples

The composite samples prepared in September 2010 were originally from the West Zone, Galena Hill and Bridge Zone. The drill hole interval samples from the Galena Hill and Bridge zones were grouped into high and low grade composite, respectively. However, the testwork was focused on the high grade composite samples only. The other sample as identified as WZ1 composite was composed from separate drill hole intervals from the Gossan Hill, R8 and West zones for the testing. The drill hole, zone and individual sample identifications are presented in Table 16.3.

Table 16.3 Conceptual Master Compositing List (2010/2011)

Composite GH2 (High Grade)			Composite WZ1		
Sample ID	**Zone**	**Hole ID**	**Sample ID**	**Zone**	**Hole ID**
SU-005	Galena Hill	SU-05	SU-032-A	Gossan Hill	SU-32
SU-006- A	Galena Hill	SU-06	SU-036-A	Gossan Hill	SU-36
SU-033	Galena Hill	SU-33	SU-036-B	Gossan Hill	SU-36
SU-54 A	Galena Hill	SU-54	SU-42 A	Gossan Hill	SU-42
SU-76 B	Galena Hill	SU-76	SU-63 A	Gossan Hill	SU-63
			SU-66A	Gossan Hill	SU-66
Composite BZ2 (High Grade)			SU-032-B	R8 Zone	SU-32
SU-021-B	Bridge Zone	SU-21	SU-032-C	R8 Zone	SU-32
SU-025	Bridge Zone	SU-25	SU-42 B	R8 Zone	SU-42
SU-058 A	Bridge Zone	SU-58	SU-63 B	West Zone Footwall	SU-63
SU-58 B	Bridge Zone	SU-58	SU-66 B	West Zone Footwall	SU-66
SU-64 B	Bridge Zone	SU-64	SU-67 A	Gossan Hill	SU-67
SU-69 A	Bridge Zone	SU-69	SU-67 B	Gossan Hill	SU-67
SU-69 B	Bridge Zone	SU-69	SU-74 A	Gossan Hill	SU-74
SU-69 C	Bridge Zone	SU-69	SU-88 A	Gossan Hill	SU-88
SU-75 C	Bridge Zone	SU-75	SU-88 B	Gossan Hill	SU-88
SU-78 C	Bridge Zone	SU-78	SU-98	Main West Zone	SU-098
SU-10 C	Bridge Zone	SU-10	SU-103	Main West Zone	SU-103

The variability tests also used composite samples: Composites SU-98, SU-76B, SU-32A, SU-32C and SU-33.

SAMPLE HEAD ANALYSES

The head assay on the 2009 and 2010 composites is summarized in Table 16.4. The assay data reveals that there was a significant variation between the grades obtained from standard fire assay and metallic analyses procedures. This indicates that the gold in some of the samples occurs in the form of nugget gold.

Table 16.4 Metal and Sulphur Contents of Composite Samples (2009-2010)

Sample ID	Au[1] (g/t)	Au (CN[4]) (g/t)	Au[2] (g/t)	Ag[3] (g/t)	Ag (CN) (g/t)	S(-2) (%)	C(org[5]) (%)	Cu[3] (ppm)	As (ppm)
SU-4	1.86	-	1.75	3.9	-	2.67	0.22	57	0.113
SU-5	0.99	-	1.10	34.8	-	1.58	0.10	235	0.026
SU-6A	1.36	-	1.98	67.3	-	3.63	0.06	101	0.020
SU-6B	1.05	-	5.23	12.9	-	3.79	0.19	90	0.029
SU-10	0.71	-	0.76	8.3	-	1.89	0.13	77	0.011
SU-19	1.35	-	1.57	6.6	-	2.03	0.25	133	0.010
SU-21A	0.62	-	0.64	10.3	-	2.39	0.14	70	0.026
SU-21B	5.23	-	5.05	12.3	-	2.07	0.18	96	0.031
SU-25	1.64	-	2.12	11.4	-	1.86	0.22	34	0.025
SU-27	0.64	-	0.91	4.0	-	1.21	0.15	23	0.033
SU-032A	2.46	1.70	2.24	13.3	11.7	3.50	0.11	66	0.016
SU-032B	0.84	0.78	1.42	71.1	73.8	3.11	0.35	57	0.007
SU-032C	1.90	1.62	3.06	1.9	4.0	2.93	0.29	27	0.024
SU-033	2.17	2.10	3.42	24.5	29.8	3.08	0.21	63	0.018
SU-036A	1.40	0.68	1.30	10.2	8.8	3.23	0.22	104	0.046
SU-036B	0.64	0.41	0.55	3.8	3.0	3.56	0.33	26	0.028
Comp R8	1.14	-	1.44	-	-	-	-	60	0.022
Comp GH	1.65	-	1.73	-	-	-	-	131	0.022
Comp BZ	1.53	-	1.67	-	-	-	-	77	0.020

Notes: [1] Whole Sample Assay. [2] Metallic Analyses. [3] by ICP. [4] CN = cyanide soluble.
[5] org = organic carbon.

The head grade assay on the composite samples for the 2010 and 2011 test is showed in Table 16.5.

As shown in Table 16.4 and Table 16.5, the gold contents in most of the samples tested are lower than the projected mill feed grades.

Table 16.5 Metal Contents of Composite Samples (2010-2011)

Composite	Head Grade	
	Au (g/t)	Ag (g/t)
Composite GH2	4.93	52.9
Composite BZ2	0.91	7.7
Composite WZ1	1.79	25.4
Composite SU-98	73.3	2.5
Composite SU-76B	12.6	130
Composite SU-32C	11.0	10.4
Composite SU-32A	3.8	25.8
Composite SU-33	3.68	22.1

GRINDABILITY TESTWORK

Table 16.6 presents the Bond ball mill work index obtained from the Brucejack mineralization. It appears that, on average, the mineralization is moderately hard.

Table 16.6 Grindability Test Results (2009-2010)

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

SAMPLE SPECIFIC GRAVITY

The specific gravity of the Brucejack mineral samples are shown in Table 16.7. The SG data varied narrowly from 2.71 to 2.84.

Table 16.7 Sample Specific Gravity (2009-2010)

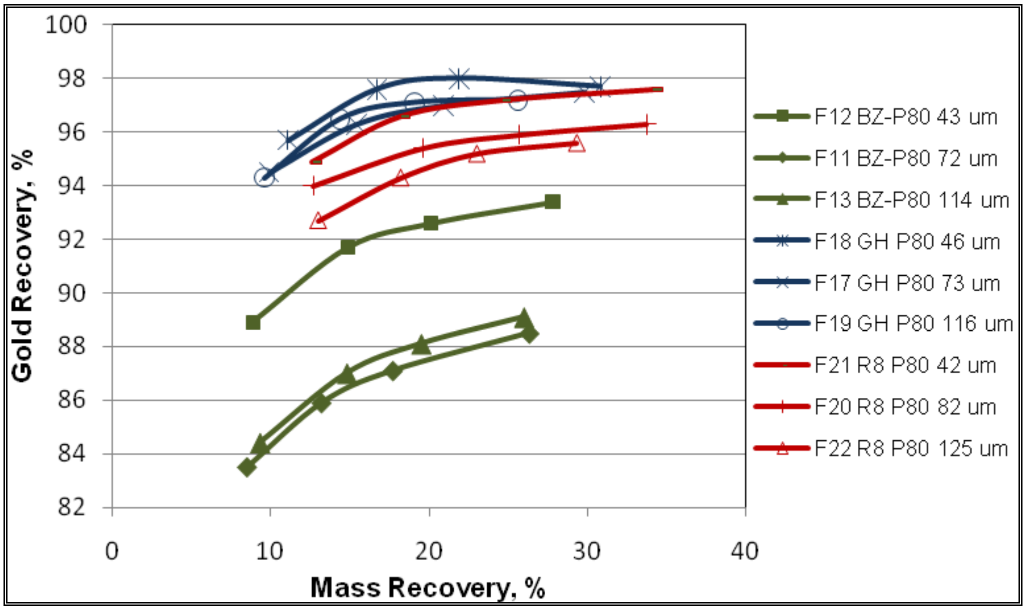
Sample ID	SG		Sample ID	SG
SU-4	2.79		SU-25	2.71
SU-5	2.74		SU-27	2.74
SU-6A	2.82		SU-032A	2.73
SU-6B	2.84		SU-032B	2.73
SU-10	2.76		SU-032C	2.72
SU-19	2.76		SU-033	2.78
SU-21A	2.75		SU-036A	2.82
SU-21B	2.77		SU-036B	2.78

FLOTATION TESTWORK

Primary Grind Size

Three different primary grind sizes were tested on the various composite samples in 2009-2010 and 2010-2011 testing programs. PAX and A208 were used as collectors -- MIBC as the frother, and copper sulphate as the activator (at scavenger flotation only). The 2009-2010 test results are shown in Figure 16.2. The data indicate that gold recovery improves when the primary grind size is finer than 70 µm. The improvement becomes much less significant at a grind size between 80% passing 70 µm and 80% passing 125 µm. The test results also show that gold recovery increases with concentrate mass pull, in particular, when the mass pull is less than 15% to 20%.

Figure 16.2 Effect of Primary Grind Size on Gold Recovery (2009-2010)



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 Bridge Zone 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.

Further investigation was conducted on the samples prepared in the 2010-2011 testing program. The effect of primary grind size on gold and silver recoveries is shown in Figure 16.3 and Figure 16.4, respectively. The test results showed that gold and silver recoveries from the samples of the Galena Hill Zone and the West Zone were higher than the Bridge Zone sample. At the grind size of 80% passing 143 µm, the gold recoveries of the Galena Hill Zone sample and the West Zone sample were approximately 97% and 95% respectively. It appears that a finer grind size produced better metal recovery.

Figure 16.3 Effect of Primary Grind Size on Gold Recovery (2010-2011)

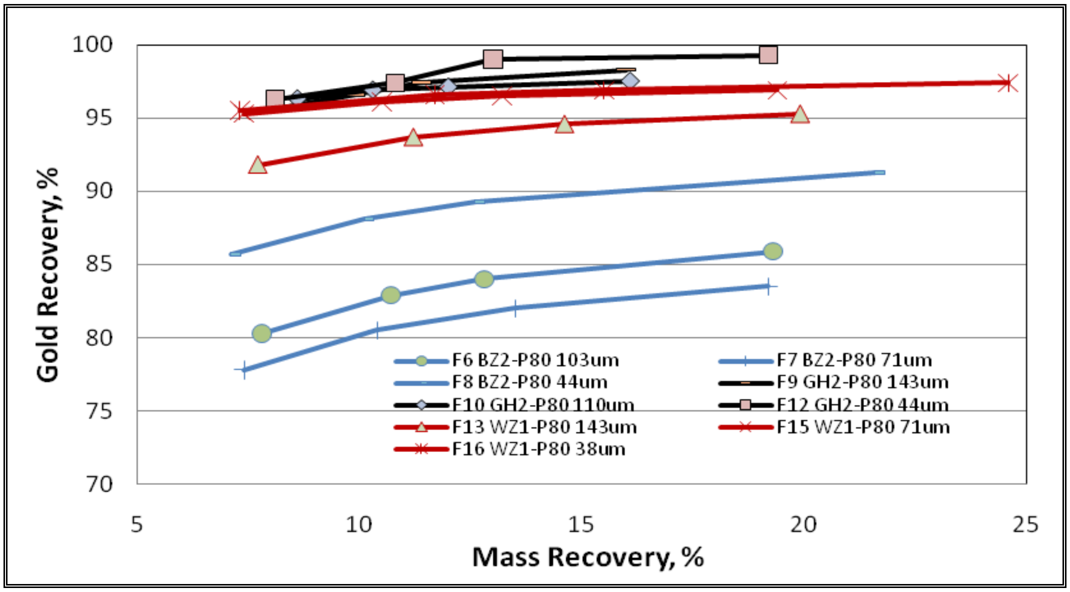
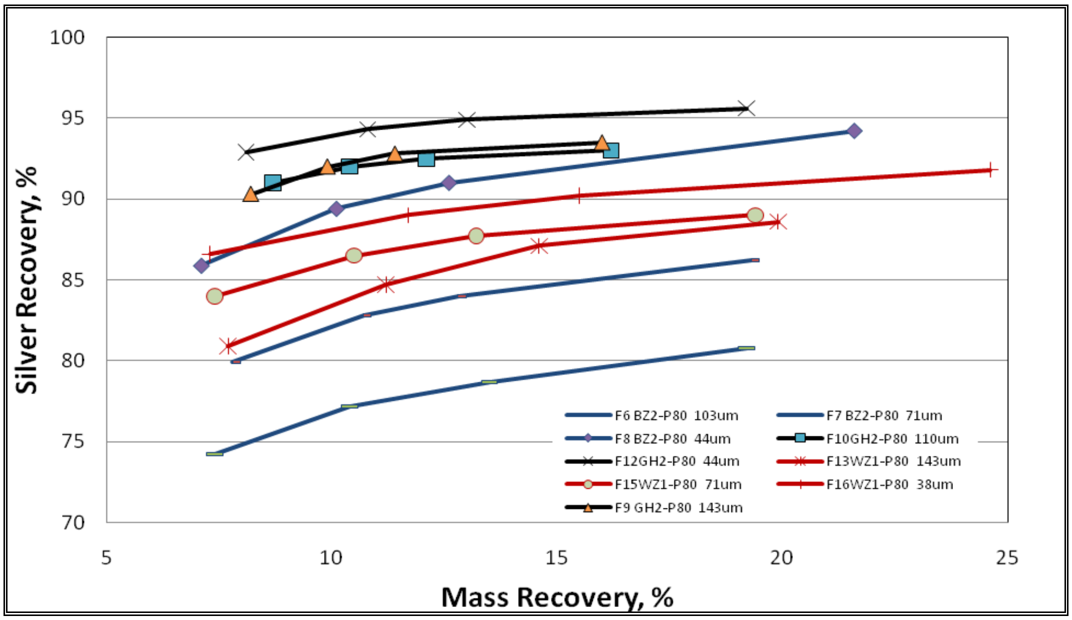


Figure 16.4 Effect of Primary Grind Size on Silver Recovery (2010-2011)

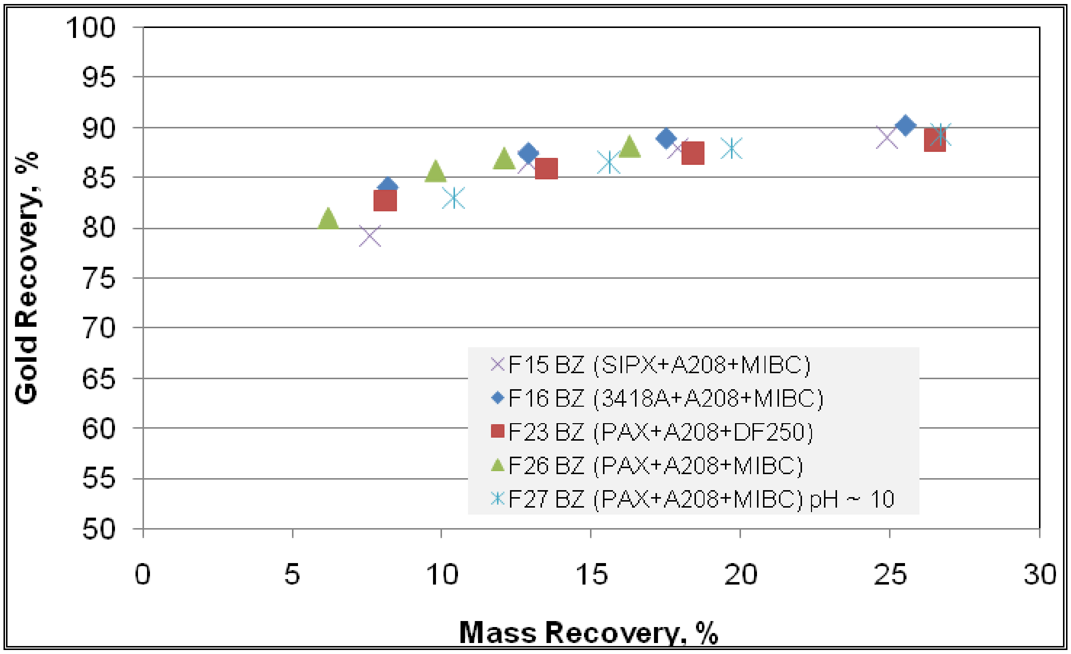


It appears that silver recoveries are lower than gold recoveries for the Galena Hill Zone and the West Zone samples. However, for the Bridge Zone sample, the difference in gold and silver recoveries was much small.

Reagents and Slurry pH

The 2009-2010 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 16.5. It appears that the effect of the reagents and slurry pH on the gold recovery was not significant.

Figure 16.5 Effect of Reagent and Slurry pH on Gold Recovery (2009-2010)

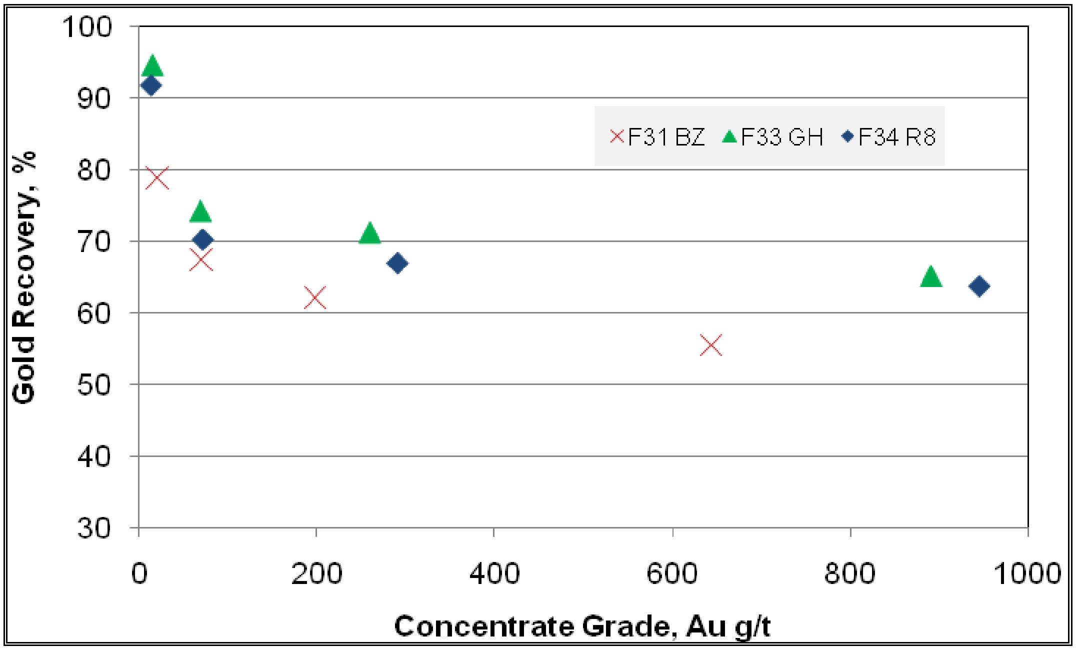


Note: * Test F27 was conducted at a higher pH, the others at natural pH

Cleaner Flotation Testwork

The 2009-2010 testing program also studied the effect of upgrading the rougher flotation concentrates on metal recovery. The tests indicated that the cleaner flotation was able to substantially upgrade the concentrates from the Brucejack mineral samples. However, as shown in Figure 16.6, the gold recovery reduced significantly at the 1st cleaner flotation stage.

Figure 16.6 Effect of Cleaner Flotation on Gold Recovery (2009-2010)



Gravity Concentration Testwork

Metallic gold determination tests and gravity concentration tests showed that the Brucejack mineralization contains a significant amount of fine grain nugget gold. The metallic gold determination test results are shown in Table 16.8 and Table 16.9.

As shows in Table 16.8, 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 amounts of native gold. Compared to Composite BZ in Table 16.9, more gold in Composite R8 and Composite GH may be present in the form of native gold.

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 16.10 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 as gold did. The test results also indicated that some of the samples (such as the SU-36B sample) were less amenable to the gravity concentration.

Table 16.8 Metallic Gold Test Results – Individual Samples (2009-2010)

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**

Note: *Tyler Mesh

Table 16.9 Metallic Gold Test Results – Composite Samples (2009-2011)

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**

Table 16.10 Gravity Concentration Test Results (2009-2010)

Test ID	Sample ID	Primary Grind/ Regrind Size	Grade (g/t)		Recovery (%)	
			Au	Ag	Au	Ag
GF35	BZ	P_{80} 131 µm	685	428	17.0	4.6
GF37	R8	P_{80} 116 µm	70.5	677	2.7	1.8
GF36	GH	P_{80} 116 µm	158	495	11.0	1.8
GF41	GH	P_{80} 116 µm	331	339	25.7	1.4
FG38	R8	P_{80} <25 µm	1,081	1,222	35.6	2.6
FG39	GH	P_{80} <25 µm	1,918	3,103	44.8	4.5
FG40	BZ	P_{80} <25 µm	1,079	984	29.3	5.9
FG42	SU-32B	P_{80} <25 µm	801	4,193	22.6	1.4
FG43	SU-33	P_{80} <25 µm	5,810	8,341	43.9	4.9
FG44	SU-36A	P_{80} <25 µm	3,337	1,653	42.3	4.0
FG45	SU-36B	P_{80} <25 µm	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 concentrate samples, and flotation tailing samples.

The leaching test results on the head samples are summarized in Table 16.11. The tests were conducted at a pH of 10.5 and a sodium cyanide (NaCN) concentration of 3 g/L with three different primary grind sizes.

Table 16.11 Head Sample Cyanidation Test Results (2009-2010)

Test No	Sample ID	Grind Size (P_{80} 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 of the head samples 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 leaching 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 used was high at 5 g/L. The leach retention time was increased to 96 hours. The test results are summarized in Table 16.12.

The test results appear to indicate that approximately between 79% and 86% of the gold can be extracted from the reground concentrates. The addition of potassium permanganate ($KMnO_4$), lead nitrate ($Pb(NO_3)_2$) and oxygen did not improve gold extraction. 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).

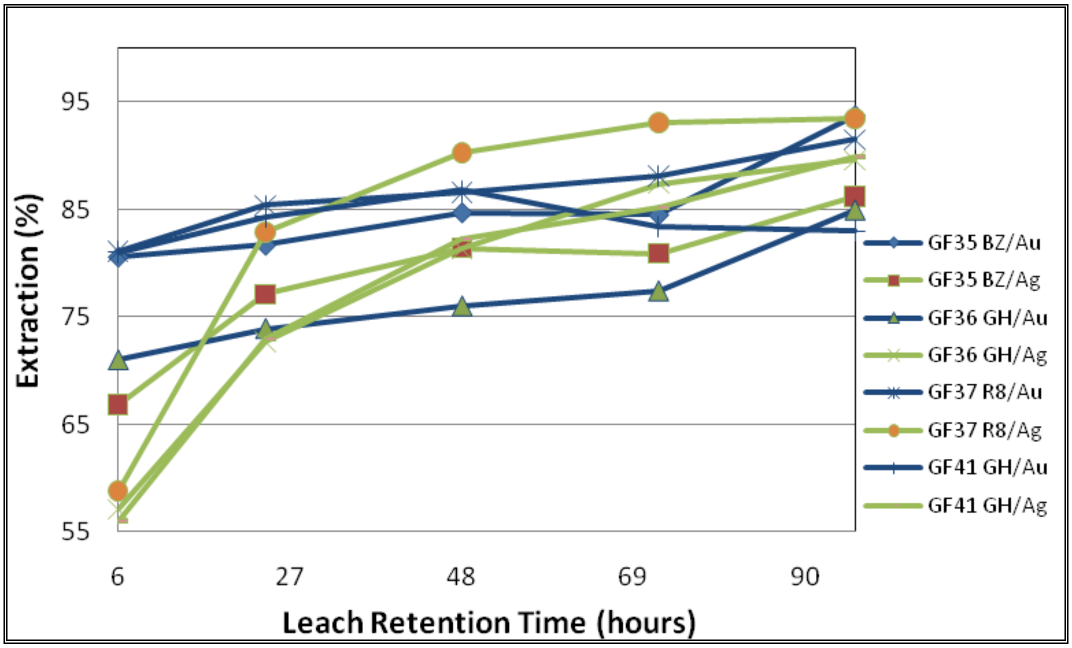
Table 16.12 Concentrate Cyanidation Test Results (2009-2010)

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

Note: * rougher + scavenger concentrate

In the 2010-2011 testing program further tests were conducted on the flotation concentrates produced using a combination of gravity concentration and flotation concentration. The cyanide leach conditions were the same as the previous testing program. Figure 16.7 shows the effect of leach retention time on gold extraction. The results were similar to the previous ones. At the leaching retention time of 27 hours, the leaching extracted between 73% and 86% of the gold and between 73% and 82% of silver. Although the increase in the leaching retention time improved gold and silver recoveries, the improvement of gold extraction was not significant, but the improvement of silver extraction was approximately 10% to 15%.

Figure 16.7 Bulk Concentrate Leaching Retention Time Test Results (2010-2011)



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 three 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 cyanide leaching on gravity tailings (Flowsheet B).

- primary grind, gravity concentration, rougher/scavenger flotation, regrind on the flotation concentrate, gravity concentration on the reground concentrate, followed by cyanide leaching on the gravity tailings, and intensive leaching on the panning tailings (Flowsheet C).

The test results are presented separately in Table 16.13, Table 16.14, and Table 16.15, for the three different combinations.

Table 16.13 Gravity Concentration + Flotation + Cyanide Leach Test Results (Flowsheet A) (2009-2010)

Test ID/Sample ID	Primary Grind/ Regrind Sizes	Grade (g/t)		Recovery/Extraction*	
		Au	Ag	Au (%)	Ag (%)
GF35/Composite BZ					
Gravity Concentrate	P_{80} 131 µm	685	428	17.0	4.4
Flotation Concentrate		18.6	45.2	77.1	77.3
Leach on Flotation Concentrate	P_{90} <25 µm	-	-	93.7	86.2
Head		4.5	10.9	-	-
GF37/Composite R8					
Gravity Concentrate	P_{80} 116 µm	70.5	677	2.7	1.8
Flotation Concentrate		11.5	158	94.4	91.0
Leach on Flotation Concentrate	P_{90} <25 µm	-	-	91.5	93.7
Head		2.8	39.5	-	-
GF36/Composite GH					
Gravity Concentrate	P_{80} 116 µm	158	495	11.0	1.8
Flotation Concentrate		9.6	200	85.5	92.5
Leach on Flotation Concentrate	P_{90} <25 µm			84.9	89.6
Head		1.9	36.3	-	-
GF41/Composite GH					
Gravity Concentrate	P_{80} 116 µm	331	339	25.7	1.4
Flotation Concentrate		7.7	186	71.8	92.8
Leach on Flotation Concentrate	P_{90} <25 µm	-	-	83.0	89.9
Head		1.8	34.5	-	-

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

As shown in Table 16.13, 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 leach 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 16.14 Flotation + Gravity Concentration + Cyanide Leach Test Results (Flowsheet B) (2009-2010)

	Primary Grind/ Regrind Sizes	Concentrate Grade(g/t)		Recovery/Extraction	
		Au	Ag	Au (%)	Ag (%)
GF38/Composite R8[1]					
Flotation Concentrate	P_{80} 128 µm	7.51	106	94.1	88.6
Gravity Concentrate	P_{94} 33 µm	1,081	1,222	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	P_{80} 141 µm	12.9	212.1	97.1	98.7
Gravity Concentrate	P_{90} <25 µm	1,918	3,103	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	-	-
GF40/Composite BZ[1]					
Flotation Concentrate	P_{80} 133 µm	8.60	44.4	85.1	97.3
Gravity Concentrate	P_{90} <25 µm	1,079	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	P_{80} 109 µm	4.71	382	93.1	90.8
Gravity Concentrate	P_{80} <25 µm	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	P_{80} 92 µm	13.5	164	98.5	93.3
Gravity Concentrate	P_{80} <25 µm	5,810	8,341	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	P_{80} 138 µm	8.95	45.7	97.0	94.5
Gravity Concentrate	P_{80} <25 µm	3,337	1,653	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	-	-

Table continues…

	Primary Grind/ Regrind Sizes	Concentrate Grade(g/t)		Recovery/Extraction	
		Au	Ag	Au (%)	Ag (%)
GF45/Composite SU-36B[2]					
Flotation Concentrate	P_{80} 96 µm	2.71	19.3	91.5	95.0
Gravity Concentrate	P_{80} <25 µm	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	-	-

Notes: [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 16.14, 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 leaching 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 leaching extraction rates were lower.

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

As shown in Table 16.15, Flowsheet C produced a higher than 60% gold recovery from the WZ1, GH2, and SU98 samples by two stages of gravity concentrations. This indicates that a significant amount of the gold in the mineralization occurs in form of the nugget gold with a wide range of grain sizes. However, a much smaller amount of silver occurs as native silver. Also, the test results indicated that intensive cyanide leaching produced higher than 93% Au and silver extractions from the high grade gravity cleaner concentration tailings (panning tailings). The gold leaching recoveries from the centrifugal gravity concentration tailings were less than 65% for the WZ1 and GH2 samples. This may imply that a portion of the gold in the mineralization is intimately associated with their bearing minerals.

Further gravity concentration test work was conducted on the blended rougher flotation concentrate produced from the various flotation tests. The centrifugal gravity tailings were subjected to cyanide leaching. As shown in Table 16.16, the gravity concentration on the reground concentrates recovered 37% of the gold from the GH2 concentrate and 29% of the gold from the WZ1 concentrate. The gold leaching extractions from the gravity tailings were 84% for the GH2 sample and 75% for the WZ1 sample.

Table 16.15 Gravity Concentration + Flotation + Secondary Gravity Concentration + Cyanide Leach Test Results (Flowsheet C) (2010-2011)

	Primary Grind/ Regrind Sizes	Concentrate Grade(g/t)		Recovery/Extraction	
		Au	Ag	Au (%)	Ag (%)
GF26/Composite GH2					
Primary Gravity Concentrate	P_{80} 125 µm	1808	183	36.4	0.32
Flotation Concentrate	P_{80} 125 µm	16	302	62.0	98.6
Secondary Gravity Concentrate	P_{80} 7.1µm	1116	2650	51.5	8.1
Gravity Rougher Tailing		5	189	31.3	76.9
Gravity Panning Tailing		70	927	17.2	15
Intensive Leach on Gravity Pan Tailing		-	-	93.6	95.6
Leach on Gravity Rougher Tailing		-	-	61.3	64
Head		5	53	-	-
Overall Recovery		-	-	90.4	71.2
GF27/Composite SU981					
Primary Gravity Concentrate	P_{80} 123 µm	11,959	186	33.2	0.3
Flotation Concentrate	P_{80} 123 µm	214	556	66.2	98.8
Secondary Gravity Concentrate	P_{80} 6.9 µm	13,281	11,323	79.1	27.7
Gravity Rougher Tailing		35	264	19.6	61.0
Gravity Pan Tailing		69	1412	1.4	11.3
Intensive Leach on Gravity Panning Tailing		-	-	95.3	95.8
Leach on Gravity Rougher Tailing		-	-	97.2	66.9
Head		73	205	-	-
Overall Recovery		-	-	99.1	78.9
GF25/Composite WZ1					
Primary Gravity Concentrate	P_{80} 120 µm	1,151	194	26.4	0.4
Flotation Concentrate	P_{80} 120 µm	12	163	70.6	97
Secondary Gravity Concentrate	P_{80} 7 µm	646	1,600	50.4	9.1
Gravity Rougher Tailing		3	107	31.0	77.3
Gravity Pan Tailing		71	716	18.6	13.6
Intensive Leach on Gravity Panning Tailing		-	-	94	96.5
Leach on Gravity Rougher Tailing		-	-	64.3	66.9
Head		2	25	-	-
Overall Recovery		-	-	88.7	72.5

Note: [1.] Composite SU-98 is from the guts of the West Zone and the Galena Hill composite.

Further cyanide leaching tests were carried out on the leaching residue which was further reground to 80% passing 10 µm. The test results showed that the additional leaching further extracted approximately 13% of the gold and 51% of the silver from the leaching residues.

Table 16.16 Gravity/Leaching Test Results on Reground Flotation Concentrate (2010-2011)

		Concentrate Grade		Recovery/Extraction	
	Regrind Size	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)
GR3/C25/C29 GH2 Blended Rougher Concentrate					
Reground Flotation Concentrate	P_{80} 25 µm	23	458	-	-
Gravity Concentrate		2,366	2,849	37	3
Gravity Tailings		18.8	442.8	63	97
Leach on Gravity Tailings		-	-	84	75
Leach Residue Regrinding	P_{80} <10 µm	2.94	117.7	-	-
Leach on Reground Residue		-	-	11	52
Secondary Leach Residue		2.05	56.4	-	-
GR2/C24/C28 WZ1 Blended Rougher Concentrate					
Reground Flotation Concentrate	P_{80} 25 µm	8.6	177.4	-	-
Gravity Concentrate		941	1,543.5	29.3	2.6
Gravity Tailings		6.9	175.3	70.7	97.4
Leach on Gravity Tailings		-	-	74.8	82.6
Leach Residue Regrinding	P_{80} <10 µm	1.96	34.5	-	-
Leach on Reground Residue		-	-	14.3	50.4
Secondary Leach Residue		1.74	18	-	-

Variability Test Work

In 2011 PRA conducted seven variability tests on various samples, three from the West Zone and four from the Galena Hill Zone. The tests studied the metallurgical responses of these samples to the Flowsheet C developed from the composite samples. The test results are summarized in Table 16.17.

Table 16.17 Variability Test Results (2010-2011)

	Grade		Recovery/Extraction		
	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Grind Size
GF26/Comp. GH2 – Head	4.93	52.9	100.0	100.0	Primary Grind Size: P_{80} 125 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	1,808	183	36.4	0.32	
Flotation Concentrate	16.5	302.2	62	98.6	
Secondary Gravity Concentrate	1,116	2,650	51.5	8.1	
Intensive Cyanide Leaching	-	-	93.6	95.6	
Cyanide Leaching	-	-	61.3	64	
Overall Recovery	-	-	91	71	
GF27/Comp. SU98[1] – Head	73.3	205.2	100.0	100.0	Primary Grind Size: P_{80} 123 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	11-958.7	186.1	33.2	0.3	
Flotation Concentrate	213.8	556.1	66	99.1	
Secondary Gravity Concentrate	13-281	11,323	79.1	27.7	
Intensive Cyanide Leaching	-	-	95.3	95.8	
Cyanide Leaching	-	-	97.2	66.9	
Overall Recovery	-	-	99	79	
GF25/Comp. WZ1 – Head	1.79	25.4	100.0	100.0	Primary Grind Size: P_{80} 120 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	1,151	194.4	26.4	0.44	
Flotation Concentrate	9.1	127.9	70.9	97.5	
Secondary Gravity Concentrate	646.4	1,600	50.4	9.1	
Intensive Cyanide Leaching	-	-	94	96.5	
Cyanide Leaching	-	-	64.3	66.9	
Overall Recovery	-	-	89	73	
GF32/Comp. SU 33/GH – Head	3.68	22.1	100.0	100.0	Primary Grind Size: P_{80} 125 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	751.3	201.3	27	1.7	
Flotation Concentrate	9.46	50.4	71.1	91	
Secondary Gravity Concentrate	690.1	818.1	53	10	
Intensive Cyanide Leaching	-	-	87.8	83	
Cyanide Leaching	-	-	74.6	78.4	
Overall Recovery	-	-	92	77	

Table continues...

	Grade		Recovery/Extraction		
	Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Grind Size
GF30/Comp. SU-32C/WZ – Head	11	10.4	100.0	100.0	Primary Grind Size: P_{80} 165 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	6,006	200.8	58	4.2	
Flotation Concentrate	38.0	37.6	41.2	89.3	
Secondary Gravity Concentrate	678.2	1,133	22.2	21.8	
Intensive Cyanide Leaching	-	-	96.8	90.9	
Cyanide Leaching	-	-	91.2	68.8	
Overall Recovery	-	-	97.7	79.1	
GF31/Comp. SU-32A/WZ – Head	3.8	25.8	100.0	100.0	Primary Grind Size: P_{80} 161 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	592.6	203.1	35.9	2.3	
Flotation Concentrate	8.55	64	61.1	84.4	
Secondary Gravity Concentrate	4,142	2,958	83.4	23.9	
Intensive Cyanide Leaching	-	-	89	86	
Cyanide Leaching	-	-	85	71	
Overall Recovery	-	-	96	71	
GF28/Comp. SU-76B/GH – Head	12.61	129.7	100.0	100.0	Primary Grind Size: P_{80} 116 µm; Regrind Size: P_{80} 7 µm
Primary Gravity Concentrate	3617	196	49.9	0.3	
Flotation Concentrate	22.75	373.9	48.8	94.6	
Secondary Gravity Concentrate	1,893	5,301	66.5	11.3	
Intensive Cyanide Leaching	-	-	95	96	
Cyanide Leaching	-	-	81	67	
Overall Recovery	-	-	97	80	

The variability test results indicated:

- there was no significant variation in metallurgical performance between the West Zone and Galena Hill Zone mineralization
- in general, the mineralization tested was amenable to the combined procedure consisting of gravity separation, flotation process and cyanide leaching. Overall average gold recovery was 94.5%, approximately 19% higher than the average silver recovery.

- the overall gold recovery increased with the gold head grade. It appears that the silver overall recovery variation was less significant although silver head grade ranged widely from 10 g/t to 205 g/t.
- the regrind size was finer than 80% passing 10 µm.

The test results from the samples tested by Flowsheet B and Flowsheet C are plotted in Figure 16.8 and Figure 16.9.

Figure 16.8 Variability Test Results – Gold Metallurgical Performance (2010-2011)

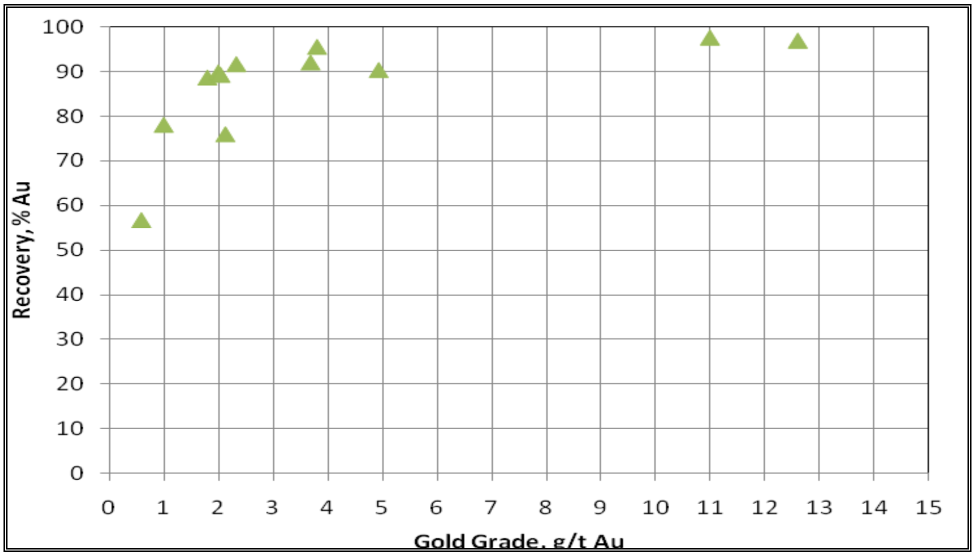
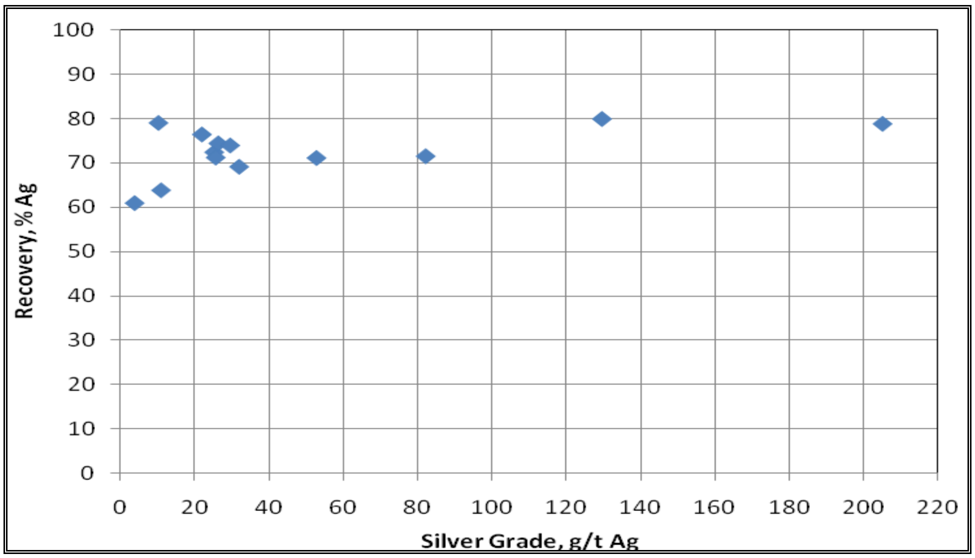


Figure 16.9 Variability Test Results – Silver Metallurgical Performance (2010-2011)



Conclusions

The review of preliminary testwork on the Brucejack mineralization led to the following conclusions:

- Brucejack mineralization is moderately hard.
- The test results suggest that the mineralization be amenable to the combined process. The process should include:
  - gravity concentration to recover coarse free gold and silver
  - flotation to produce rougher and scavenger concentrates
  - regrinding on the rougher and scavenger concentrates
  - gravity concentration to recover fine free gold and silver
  - cyanide leaching on gravity concentration tailings to produce gold/silver doré, including intensive cyanide leaching.
- 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, especially gravity separation process, primary grind size, regrind size, flotation optimization
- further 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 deposits are projected in Table 16.18.

Table 16.18 Projected Metallurgical Performance

Year	Annual Process Rate (t)	Mill Feed Grade		Doré					
				Recovered Metals				Recovery	
				Au		Ag			
		Au (g/t)	Ag (g/t)	kg	oz ('000)	kg	oz ('000)	Au (%)	Ag (%)
1	410,746	8.41	116.9	3,198	103	36,144	1162	92.6	75.3
2	584,332	8.26	112.2	4,464	144	49,241	1583	92.5	75.1
3	586,999	10.85	110.7	5,961	192	48,726	1567	93.6	75.0
4	590,578	9.75	77.5	5,368	173	33,687	1083	93.2	73.6
5	555,312	10.86	130.9	5,645	181	55,163	1774	93.6	75.9
6	540,689	8.88	82.0	4,459	143	32,744	1053	92.8	73.8
7	534,934	10.59	43.8	5,296	170	16,920	544	93.5	72.2
8	548,668	11.98	66.1	6,179	199	26,545	853	94.0	73.2
9	563,108	14.71	42.1	7,849	252	17,094	550	94.8	72.2
10	516,872	11.46	81.8	5,559	179	31,210	1003	93.8	73.8
11	504,443	9.78	90.5	4,597	148	33,857	1089	93.2	74.2
12	533,896	5.62	60.5	2,730	88	23,563	758	91.0	72.9
13	547,482	3.39	45.3	1,655	53	17,939	577	89.1	72.3
14	526,006	3.29	38.7	1,535	49	14,675	472	88.7	72.0
15	547,447	3.34	33.9	1,628	52	13,312	428	89.0	71.8
16	330,284	3.69	30.2	1,101	35	7,145	230	90.3	71.7
Total	**8,421,794**	**8.58**	**73.5**	**67,222**	**2,161**	**457,965**	**14,724**	**93.0**	**74.0**
Average	**-**	**8.58**	**73.5**	**-**	**-**	**-**	**-**	**93.0**	**74.0**

16.2 MINERAL PROCESSING

16.2.1 INTRODUCTION

The proposed concentrator will process the gold/silver mineralization from the Brucejack deposit at a nominal rate of 1,500 t/d with an availability of 92% (365 d/a). The concentrator will produce gold-silver doré.

The Brucejack deposit will be mined by conventional underground mining during the first 13 years, and then by open pit mining during the rest of the mine life. The mill feed grades will reduce significantly when the open pit mining start.

16.2.2 SUMMARY

The process is developed to produce gold-silver doré. The process flowsheets for the Brucejack deposits is a combination of conventional bulk sulphide flotation, gravity concentration and cyanidation with gold and silver recovery by the Merrill-Crowe process. There will be two process plants, one flotation plant at the mine site to produce bulk gold-silver flotation concentrate/gravity concentrate and one leach plant (cyanidation and recovery) at the leach plant site to produce gold-silver doré. The leach plant will be located close to Highway 37.

The mine site process plant will consist of two crushing stages, primary grinding, gravity concentration and flotation processes to produce a gravity concentrate and a bulk flotation concentrate containing gold and silver. The produced bulk rougher/scavenger concentrate and the gravity concentrate will be dewatered and trucked to the leach plant by 20-t trucks.

The leach plant will consist of following processes:

- bulk concentrate regrinding
- gravity concentration
- cyanidation
- gold and silver recovery by the Merrill-Crowe process.

The conventional cyanidation process will leach the reground rougher concentrates (after gravity concentration) to recover gold and silver. An intensive leach process is proposed in order to recover gold and silver from the tailings of the gravity cleaner concentration by tabling. The extracted gold and silver from the leaching circuits, together with the high-grade gravity concentrate from the tabling process, will be refined on-site to produce gold-silver doré.

A part of bulk flotation tailings will be used for the underground backfilling and the rest will be discharged into the Brucejack Lake. The leach residues will be sent to the leaching TSF after the residual cyanide is destructed. Process water from the plants will be recycled as process make-up water. Fresh water will be used for mill cooling, gland seal service, and reagent preparation.

16.2.3 Flowsheet Development

The mill flowsheet is based on PRA's 2009 and 2011 grinding, flotation and leaching testwork, combined with engineering experience.

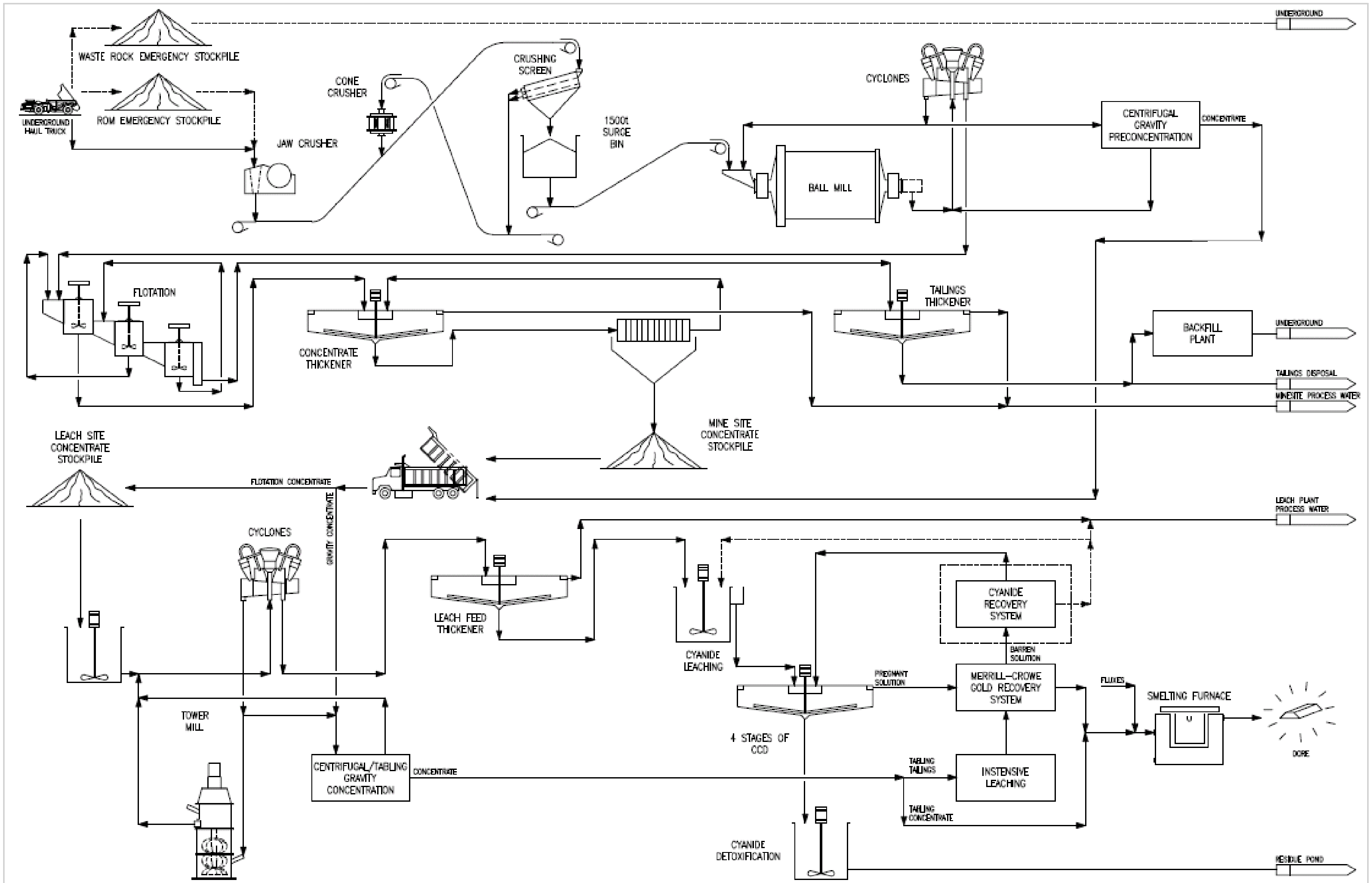
The process plants will consist of the following unit operation:

- Mine site:
  - primary crushing
  - conveying system
  - secondary crushing
  - grinding/ gravity concentration
  - rougher/scavenger flotation
  - concentrate dewatering and loadout

  - tailing disposal to the tailing impoundment or to the underground mine for backfilling.
- Leach plant site:
  - bulk flotation concentrate regrinding and gravity concentration
  - conventional cyanide leaching on the reground rougher/scavenger flotation concentrate
  - intensive cyanide leaching on the gravity cleaner concentration tailings
  - Merrill-Crowe process/refining process to produce doré
  - cyanide recovery, destruction and related processes
  - leach residues disposal to the TSF.

The simplified flowsheet for both the site operations is shown in Figure 16.10.

Figure 16.10 Simplified Process Flowsheet



16.2.4 PLANT DESIGN

MAJOR DESIGN CRITERIA

The concentrator has been designed to process 1,500 t/d, equivalent to 547,500 t/a. The major criteria used in the design are outlined in Table 16.19.

Table 16.19 Major Design Criteria

Criteria	Unit	
Daily Processing Rate	t/d	1,500
Operating Days per Year	d	365
Operating Schedule – Crushing		one shift/day; 10 hours/shift
Operating Schedule – Grinding/Flotation/Leach		two shifts/day; 12 hours/shift
Primary Crushing		
Crushing Availability	%	70
Primary Crushing Product Particle Size, P_{80}	mm	70
Secondary Crushing		
Crushing Availability	%	70
Secondary Crushing Product Particle Size, P_{80}	mm	10
Grinding/Flotation/Leach/Gravity Concentration		
Availability	%	92
Milling & Flotation Process Rate	t/h	68
Free Gold Recovery from Primary Grinding Circuit		Gravity Concentration
Ball Mill Feed Size, F_{80}	mm	10
Ball Mill Grinding Particle Size, P_{80}	µm	125
Ball Mill Circulating Load	%	300
Bond Ball Mill Work Index	kWh/t	16.6
Rougher Concentrate Regrinding Particle Size, P_{80}	µm	~10
Free Gold Recovery from Reground Concentrate		Gravity Concentration
Leach Method – Reground Rougher Concentrates		Conventional Cyanide Leaching
Leach method – Gravity Concentration Cleaner Tailings		Intensive Cyanide leaching
Gold-Silver Recovery from Pregnant Solutions		Merrill- Crowe Process
Gravity Concentration Cleaner Concentrates		Direct Smelting
Feed Rate to Leach Circuit, Design	t/h	14

OPERATING SCHEDULE AND AVAILABILITY

The process plants are designed to operate on the basis of two 12-h shifts per day, 365 d/a; the crushing facility will be operated for the 10-hour day shift only.

The primary crusher and secondary crusher circuits' overall availability will be 70%. The grinding, flotation and primary gravity concentration availability will be 92%. The leach plant availability will be 92%. These availabilities will allow for a potential increase in crushing rate, sufficient downtime for scheduled and unscheduled maintenance of the crushing and process plant equipment, and potential weather interruptions.

16.2.5 PROCESS PLANT DESCRIPTION

FLOTATION/GRAVITY CONCENTRATION PLANT – MINE SITE

Primary Crushing

The crushing facility will have the average process rate of 179 t/h. A jaw crusher is proposed for the primary crushing.

The major equipment and facilities at the site include:

- a hydraulic rock breaker
- a stationary grizzly
- a jaw crusher, 160 kW
- vibrating grizzly feeder
- associated dump pocket and belt conveyor
- belt scales
- a dust collection system.

The primary crusher feed will be trucked from the underground mine or open pits. The mineralization will be reduced to 80% passing 70 mm, using a single jaw crusher. A single rock breaker will be installed to break any oversize rocks.

The crusher product will be conveyed onto a conveying system, and transported to the vibrating screen to classify the final crushing products from the oversize particles, which will be sent to the secondary crushing.

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

Secondary Crushing

The secondary crushing circuit will be operated in closed-circuit with the vibrating screen. The cone crusher product will return to the screen feed conveyor to combine with fresh crushed materials from the jaw crusher as feeds to the vibratory double deck screens. The screened product (finer than 12 mm) will be delivered to the mill feed surge bin by conveyor.

The secondary crushing facility will include:

- a double-deck vibratory screen: 1.6 m wide x 6.1 m long, 20/12 mm apertures
- a cone crusher with 315 kW installed power
- conveyor belts, metal detectors, and self-cleaning magnets
- a dust collection system.

The secondary crushing will be conducted in one train containing a 30 m^3 feed bin and belt feeder, a vibrating dry double-deck screen in closed circuit with one cone crusher.

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

Mill Feed Surge Bin

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

- a mill feed surge bin, having 1,500 t live capacity
- two reclaim belt feeders
- a dust collection system.

The mill feed surge bin will have a live capacity of 1,500 t. The crushed material will be reclaimed from the bin by two belt feeders at a nominal rate of 68 t/h. The belt feeders will reclaim the material to a belt conveyor to feed the ball mill.

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

Primary Grinding, Classification and Primary Gravity Concentration

The primary grinding circuit will consist of a ball mill in a closed circuit with classifying hydrocyclones and gravity concentrator. The grinding will be conducted as a wet process at a nominal rate of 68 t/h of material.

The grinding/gravity concentration circuit will include:

- one 1,100 kW ball mill (3.6 m diameter x 5.4 m long (12 ft x 17.8ft))
- cyclone feed slurry pumps
- three 350 mm hydrocyclones
- one centrifugal concentrator
- one particle size analyzer

- one sampler.

The materials from the surge bin will enter the grinding circuit via the belt conveyor. The ball mill will be operated in closed circuit with hydrocyclones. The product from ball mill will be discharged into the hydrocyclone feed pumpbox, where the slurry will combine with the gravity concentration tailings. The slurry in the hydrocyclone feed pumpbox will be pumped to the hydrocyclones for classification. Sixty-seven percent of the hydrocyclone underflow will return by gravity to the ball mill while, 33% of the hydrocyclone underflow will report to the centrifugal concentrator by gravity. The gravity concentration will recover free gold particles from the hydrocyclone underflow. The tailings from the gravity concentration will return to the hydrocyclone feed pumpbox by gravity. The gravity concentrate will be dewatered and trucked to the leach plant. The cut size for the hydrocyclones will be 80% passing 125 µm, and the circulating load to the ball mill circuits will be 300%.

The new feed to the ball mill circuit will be 68 t/h, and will constitute the feed rate to gold-silver bulk flotation circuit. The pulp density of the hydrocyclone overflow slurry will be approximately 35% solids. Dilution water will be added to the grinding circuit as required.

Flotation

The milled pulp will be subjected to flotation to recover gold, silver and their bearing minerals from the Brucejack mineralization. The flotation circuit will include 13 rougher and scavenger flotation tank cells (16 m^3). The feed to the flotation circuit will be at a rate of 68 t/h. Flotation reagents will be added to the flotation circuits as defined through testing. The flotation reagents added will be copper sulphate ($CuSO_4$) as regulator, potassium amyl xanthate (PAX) and A208 as collectors and methyl isobutyl carbinol (MIBC) as frother. The mass recovery of the rougher concentrate is approximately 15-20% of the flotation feed. The concentrates produced from the rougher flotation circuits will be sampled automatically prior to the dewatering and concentrate stockpiling.

The rougher/scavenger flotation circuit will consist of:

- rougher flotation tank cells (six 16 m^3 cells)
- scavenger flotation tank cells (seven 16 m^3 cells)
- slurry pumps
- sampling system.

Tailings from the flotation circuit will be sampled automatically prior to being sent to the tailings thickener. Most of the flotation tailings will be pumped to the backfill plant and the balance will gravity flow to the Brucejack Lake for storage.

Concentrate Handling

The flotation concentrate will be thickened, filtered and stored prior to being transported to the leach plant by trucks. The concentrate handling circuit will have the following equipment:

- one 7.5 m diameter high-rate thickener
- slurry pumps
- stock tank
- one 70 m^2 pressure filter
- storage and dispatch facility.

The concentrate produced will be pumped to the concentrate thickener. Flocculant will be added to the thickener feed well 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 serves 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 approximately 9%. The filtrate will be returned to the concentrate thickener. The filter press solids will be discharged to the concentrate stockpile which is able to stock the concentrates for seven days. The concentrate will be loaded into trucks to be transported to the leach plant.

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

Tailings Handling

The bulk flotation final tailings will be thickened prior to being pumped to the backfill plant for underground backfilling or for gravity-flowing to the Brucejack Lake for discharge. The backfilling tailings will be further dewatered by the disc filters to the solid percent of 80% to 85%, and then blended with the bonding materials prior to being pumped to the underground stopes.

The tailings handling circuit will include:

- a 15 m diameter high-rate thickener
- slurry pumps
- two disc filters
- a reclaim water barge and pumping system.

Reagent Handling and Storage – Mine Site

Various chemical reagents will be added to the process slurry stream to facilitate the processes. The reagents used in the process will include:

- flotation: PAX, A208, MIBC, and copper sulphate
- flocculant and anti-scalant.

The preparation of the various reagents will require:

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

The chemical reagents will be added to the grinding and flotation to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles into the concentrate products.

Fresh water will be used for the making up or for the dilution of PAX and copper sulphate that will be supplied in powder/solid form. The strength of the reagent solutions will be approximately 20%. These solutions will be stored in separate holding tanks and added to the addition points of the flotation circuit using metering pumps.

The liquid reagents (including A208, MIBC and anti-scalant) will not be diluted and will be pumped directly from the bulk containers to the points of addition using metering pumps.

Flocculant will be prepared in the standard manner as a dilute solution of less than 0.1% solution strength. This will be further diluted in the thickener feed well.

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.

LEACH PLANT – LEACH PLANT SITE

Concentrate Receiving Facility

The flotation concentrate will be transported by 20-tonne trucks from the mine site to the leach plant. The concentrate will report to the concentrate receiving facility which has a capacity for seven days of concentrate storage. This facility will be equipped with a tire washing system.

Regrinding Circuits

The bulk flotation concentrate will be re-pulped to slurry and reground to 80% passing approximately 10 µm by an ultrafine regrinding mill. The mill will be in a closed circuit with hydrocyclones. A portion of the hydrocyclone underflow will be sent to a centrifugal concentrator to recover any fine free gold nuggets. The gravity separation tailings will return to the hydrocyclone feed pumpbox. The centrifugal gravity concentrate, together with the gravity concentrates from the primary grind circuit from the mine site, is upgraded by tabling. The tabling concentrate will be directly sent to the refining smelting while the table tailings will report to the intensive leach circuit.

Dilution water will be added to the regrinding and gravity separation circuits as required from the process water tank. Lime slurry will be added to adjust slurry pH.

Conventional Cyanide Leaching

The reground gold-bearing flotation concentrate from the overflow of the hydrocyclones will gravity flow to the leach feed thickener. The underflow of the thickener will be pumped to the gold leaching circuit.

The key equipment in the leach circuit will include:

- an 8 m diameter leach feed thickener
- an aeration tank
- six leach tanks (6.5 m diameter x 7 m high)
- four 8 m diameter counter-current decantation (CCD) washing thickeners
- slurry pumps.

The leach feed will enter the circuit via the leach feed thickener where the solids will be thickened to a density of 60% solids. The thickener underflow will be pumped to the aeration tank where the slurry will be diluted with the washing thickener overflow 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 leach circuit. The leach circuit will consist of six agitated tanks. The leached slurry will enter four stages of counter current decantation (CCD) washing which consist of four high rate thickeners. The circuit will run in a counter-current arrangement. The pregnant solution will be separated out at the first washing thickener and pumped to the Merrill-Crowe gold-silver recovery system.

Intensive Cyanide Leaching – Table Tailings

The table tailings produced from the gravity tabling separation will be sent to the intensive cyanide leach system. The intensive leach circuit equipped with an automatic control system will be on a batch operation basis. The pregnant solution produced from the intensive leach circuit will join with the pregnant solution form the conventional cyanide leach circuit to form the feed to the Merrill-Crowe gold/silver recovery system. The intensive leach residue will be pumped back to the regrinding circuit for further regrinding.

Merrill-Crowe Gold/Silver Recovery System and Refining

The pregnant gold and silver solution will be pumped from the pregnant solution stock tank to the Merrill-Crowe gold and silver recovery system. The solution will be clarified through two horizontal leaf type clarifiers. Then the clear solution will be pumped to the vacuum system for removing oxygen from the solution in a vacuum tower. The solution exiting from the bottom of the tower will be sent to the cone bottom precipitation tank with agitator. A variable speed feeder will be provided to dose zinc powder to the solution prior to the solution entering the precipitation tank. Lead nitrate will be added to activate the zinc in the precipitation system.

The precipitated precious metals together with the unreacted zinc powder will be pressure filtrated. The barren solution will return to the barren solution tank and then return to the leach circuit. The precious metal sludge will be removed from the precipitate filter on a batch basis. The filter cake will be transferred to the gold room for drying and smelting. An electric induction furnace will be used for the gold-silver refining.

The process will require the following major items of equipment:

- two solution vertical leaf clarifiers
- one Merrill-Crowe tower
- one zinc dust feeder
- one zinc precipitation cone tank with agitator
- one precipitate plate and frame filter
- vacuum pumps
- drying oven
- smelting furnace and refining related devices
- associated pumps.

The precipitation and refinery areas will be in a secure area with a security surveillance system in operation.

Cyanide Recovery and Destruction

Prior to the disposal into the TSF, the leach residue will undergo the sulphur dioxide/air oxidation cyanide destruction procedure. The circuits will include the following equipment:

- two cyanide destruction tanks (two 6.0 m diameter x 6.0 m high)
- slurry pumps.

The excess barren solution from the leaching circuit will be sent to the cyanide destruction system after the residual cyanide is recovered.

Reagent Handling and Storage – Leach Plant

Reagents used in the gold-silver leaching and recovery process will include:

- leach and recovery: lime, sodium cyanide, zinc powder, lead nitrate
- 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
- mixing and holding tanks
- metering pumps
- a flocculant preparation facility
- a lime slaking and distribution facility
- eye-wash and safety showers
- applicable safety equipment, including cyanide alarm systems.

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 leach and recovery circuits and related circuits using metering pumps.

The liquid reagents (including sulphuric acid and anti-scalant) will not be diluted and will be pumped directly from the bulk containers to the points of addition using metering pumps.

Flocculants will be prepared in the standard manner as a dilute solution of less than 0.1% 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 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, flotation plant, leach plant, and the environmental department. The most important of these instruments includes:

- fire assay equipment
- atomic absorption spectrophotometer (AAS)
- 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, leach units, 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 operations for both the mine site and the leach plant site.

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

For the mine site, the overflow solutions from the concentrate thickener and tailing thickener will be re-used in process circuit. The balance of the process water will be supplied from the proposed mine (underground water) or from the Brucejack Lake as required. All process water required will be distributed to the process plant from the process water tank on the mine site.

For the leach plant, the leach feed thickener overflow solution will be re-used in the regrinding and gravity separation circuit. The majority of the process water will be reclaimed water from the leach TSF, the water from the cyanide recovery system and the water from the local wells. All process water required will be distributed to the leach plant from the process water tank on the leach plant site.

AIR SUPPLY

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

- Mine Site
  - crushing circuit – high-pressure air will be provided by dedicated air compressors for dust suppression and equipment services
  - flotation – low-pressure air for flotation cells will be provided by air blowers
  - 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.
- leach plant Site
  - cyanide leach – high-pressure air will be provided by dedicated air compressors
  - cyanide recovery and destruction – high-pressure air will be provided by dedicated air compressors

- filtration – 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

Both the process plants will be provided with plant control system consisting of a Distributed Control System (DCS) with PC- based Operator Interface Stations (OIS) located in both mine site and leach plant site:

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

In addition to the plant control system, a closed-circuit television (CCTV) system will be installed at various locations throughout the plant including the crushing facility, the tailings handling facility, the concentrate handling areas, leaching area and the gold recovery facilities. The cameras will be monitored from the local control rooms.

The plant will be equipped with on-line sampling systems. A sufficient number of samples will be taken and assayed in the assay laboratory for metallurgical accounting.

On-stream particle size analyzers will determine the particle sizes of the primary grinding and 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

This mineral resource estimate has been prepared in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1, and in conformity with generally accepted "CIM Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. Mineral resources have been classified in accordance with the "CIM Standards on Mineral Resources and Reserves: Definition and Guidelines" (2005):

- 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 drill holes."
- 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 drill holes 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 drill holes 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 reserves. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure.

All mineral resource estimation work reported herein was carried out by FH Brown, CPG, Pr.Sci.Nat., and Eugene Puritch, P.Eng., of P&E Mining Consultants Inc., independent Qualified Persons in terms of NI 43-101. This mineral resource estimate is based on information and data supplied by Pretivm. A draft copy of this report was reviewed by Pretivm for factual errors.

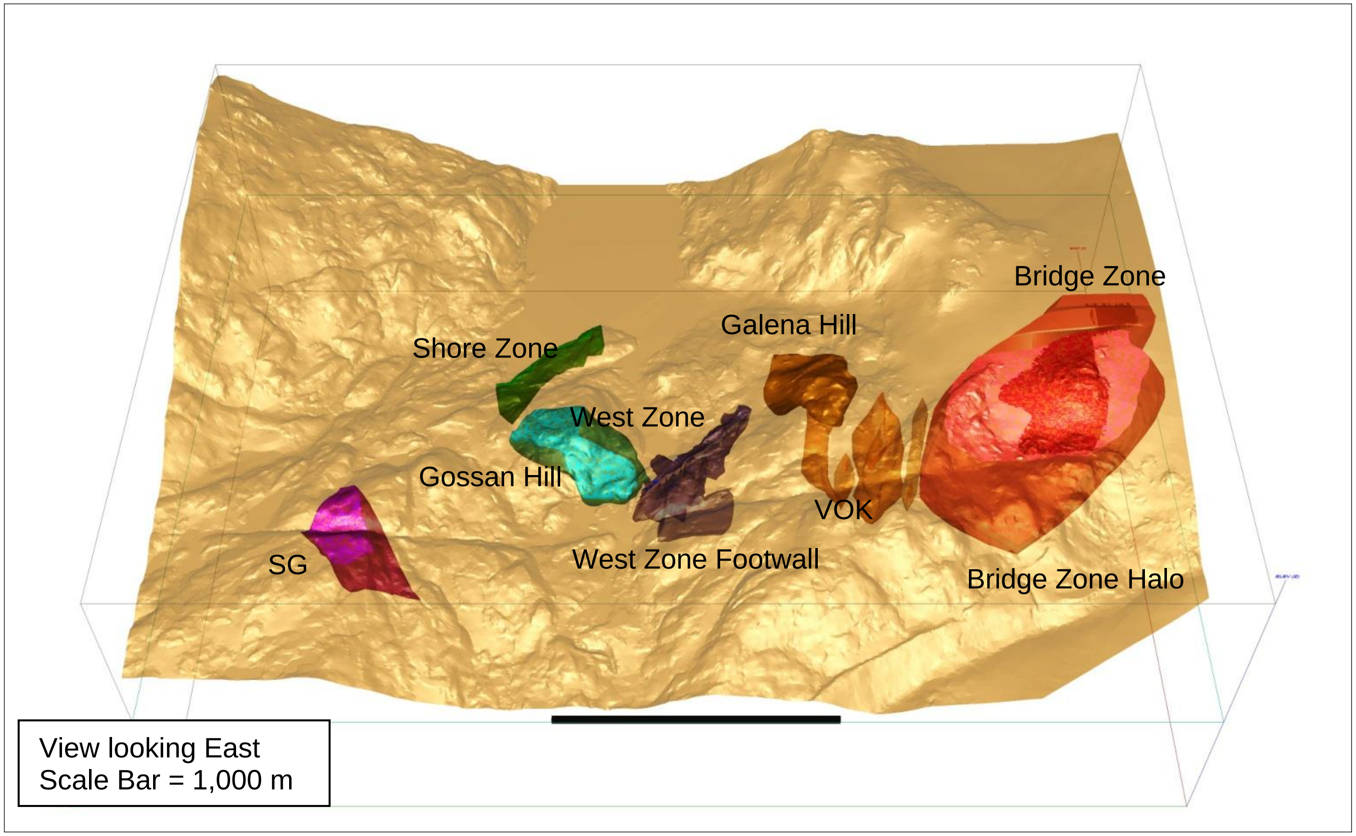
Mineral resource modelling and estimation were carried out using the commercially available Gemcom GEMS™V 5.23 and Snowden Supervisor™ V 7.10.11 software programs. Pit shell optimization was carried out using Whittle Four-X Single Element™ V 1.10.

The Brucejack mineral resource estimate encompasses nine distinct modelled mineralization domains (Figure 17.1). They are:

- West Zone
- West Zone Footwall Zone
- Shore Zone
- Gossan Hill Zone
- Galena Hill Zone
- SG Zone
- VOK Zone
- Bridge Zone
- Bridge Zone Halo.

The VOK Zone has been modelled as four distinct sub-domains. The effective date of this estimate is February 18, 2011.

Figure 17.1 Isometric Projection of the Brucejack Mineralization Zones



17.2 PREVIOUS RESOURCE ESTIMATES

Pincock Allen & Holt Ltd. previously prepared a public mineral resource estimate for the Brucejack deposits, dated April 16, 2001[1]. 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 Au (Table 17.1), based on a gold-equivalent cut-off derived from a Ag:Au equivalency ratio of 66:1.

P&E Mining Consultants Inc. prepared a mineral resource estimate for the Brucejack deposit, dated December 1, 2009[2]. This mineral resource estimate reported a Measured and Indicated mineral resource of 4.04 Moz Au and an Inferred mineral resource of 4.87 Moz Au (Table 17.2) using a cut-off of 0.35 g/t AuEq. The estimate was based on the results of 844 drill holes and constrained within an optimized conceptual pit shell.

Table 17.1 Pincock Allen & Holt Ltd. April 16, 2001, Mineral Resource Estimate

Zone	Class	AuEq Cut-off (oz/t)	Tonnes (x '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**

Table 17.2 Pincock Allen & Holt Ltd., 2009 Mineral Resource Estimate

Class	Tonnes x M	Au g/t	Ag g/t	Au Moz	Ag Moz
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	0.76	11.2	4.87	71.5

[1]Sulphurets-Bruceside Property British Columbia Technical Report, Pincock Allen & Holt Ltd., April 16, 2001.
[2]Technical report and resource estimates on the West, Bridge, Galena Hill, Shore, SG and Gossan Hill gold and silver zones of the Brucejack property, P&E Mining Consultants Inc., dated December 1, 2009.

17.3 SAMPLE DATABASE

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

The supplied databases contain records for 1,002 drill holes. Of these, 94 drill holes were outside the block model limits or had no reported assay data.

The 908 drill hole records (Table 17.3) used for this mineral resource estimate contain collar, survey and assay data. Assay data fields consist of the drill hole ID, downhole interval distances, sample number, gold grades and silver grades. All data are expressed in metric units and all collar coordinates were converted by Silver Standard to the UTM NAD27 system.

Table 17.3 Brucejack Drilling Database Records

Data Type	Record Count
Historical Surface Drilling	362
Historical UG Drilling	439
Pretivm Surface Drilling	107
Total	**908**

The database contains a total of 76,227 Au assays and 75,291 Ag assays. Due to the varying assay protocols in use during different project phases, the following low grade conversions were used:

- For the 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 2009 and 2010 Silver Standard drilling program, Au assay grades less than 0.005 g/t were converted to 0.003 g/t, and Ag assay grades less than 0.5 g/t were converted to 0.25 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 for example in areas of multiple tunnels, junctions, etc.). The digitized data were converted to UTM NAD27 coordinates using the McElhanney conversion factors, imported into GEMS, and a single 3D solid was generated to represent the underground workings.

17.4 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 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. P&E examined downhole survey data for significant deviations.

17.5 Topographic Control

For the Brucejack project, Silver Standard contracted aerial photography specialists Aero Geometrics to produce a topographic map of the property. Using high-resolution aerial photographs taken in 2008, a photo-mosaic was 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 & Resources, Aero Geometrics digitally generated a contoured topographic map with contour lines spaced at 2-m intervals, and presented this map as a digital elevation model (DEM) in AutoCAD file 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 projects, 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, historic mine grid stations) that had been surveyed by McElhanney field crews in 2009.

17.6 Density

A total of 317 bulk density measurements were provided by Silver Standard, with an average bulk density of 2.81 t/m^3 (Table 17.4). Bulk density measurements were measured by ALS Chemex from core samples. For the West Zone, a value of 2.75 t/m^3 has been used historically. A global bulk density of 2.80 t/m^3 was assigned to all lithologies for this mineral resource estimate.

Table 17.4 Brucejack Bulk Density Statistics

	Units	Waste	Mineralization	Total
Count		84	233	**317**
Minimum	t/m^3	2.61	2.50	**2.50**
Maximum	t/m^3	2.97	3.34	**3.34**
Average	t/m^3	2.80	2.81	**2.81**
Standard Deviation		0.08	0.09	**0.08**

17.7 Brucejack Domain Modelling

Silver Standard identified several discrete mineralization domains at Brucejack; Silver Standard considered the West Zone, Shore Zone and VOK Zone to be predominately structurally controlled vein systems related to the north-trending Brucejack Fault and associated Reidel shear structures, and the other domains tentatively defined as mineralized stockwork/breccia/vein systems. (Pers. Comm. Dr. W. Board, 2010).

The overall trend of mineralization in the West Zone, West Zone Footwall Zone and Shore Zone is ~135°, and modelling for these domains was generated 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 drill hole assay intervals, in order to generate a true 3D representation of the extent of the mineralization.

Silver Standard geologists identified the general trend of the VOK mineralization as predominantly east-west, with high-grade zones associated with internal Reidel shear structures. Four distinct sub-domains were modelled for the VOK Zone from successive polylines spaced every 10 m perpendicular to this trend.

For the Gossan Hill Zone, Galena Hill Zone, SG Zone and Bridge Zone domains the 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 drill hole assay intervals, in order to generate a true 3D representation of the extent of the mineralization.

In order to ensure that all potentially economic mineralization was captured for mineral resource estimation, a secondary mineralization halo for the Bridge Zone was subsequently modelled using a0.2 g/t Au grade shell. Three-dimensional models of the low-grade mineralization domain were then created by combining successive polylines into wireframes. In addition, a profile of the glacier margins at Bridge Zone was constructed from ice/bedrock contacts logged during drilling, and the Bridge Zone and Bridge Zone Halo domains were clipped to this surface.

17.8 COMPOSITING

Assay sample lengths for the database range from 0.05 m to 48 m, with an average sample length of 1.50 m. A compositing length of 1.50 m was therefore selected for compositing. Length-weighted composites were calculated for gold and silver 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 drill hole 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 drill hole workspace. Composites were assigned a domain rock code value based on the domain wireframe that the interval midpoint fell within. The composite data was then exported to Gemcom extraction files for grade estimation.

17.9 EXPLORATORY DATA ANALYSIS

Summary assay statistics were calculated separately for surface and underground sample populations (Table 17.5), with fourteen surface drilling assay grades of 1,000 g/t Au or higher, and four underground drilling assay grades reporting assays of 1,000 g/t Au or higher.

The correlation coefficient for the total gold and silver sample populations is 0.19, indicating little correlation. However, for the surface assay sample population considered separately the correlation coefficient is 0.24, rising to 0.32 for the underground population.

Summary composite statistics were calculated by domain for each commodity (Table 17.6). A comparison of the domain averages demonstrates the differences in grade distributions between the defined domains, with the highest average composite grades occurring at the VOK, followed by the West Zone.

Table17.5 Brucejack Summary Assay Statistics

	Au ppm	Ag ppm	Length
Surface			
Mean	2.00	25.31	1.50
CV	47.92	13.07	0.35
Median	0.31	3.90	1.50
Mode	0.09	0.25	1.50
Standard Deviation	95.83	331.02	0.53
Sample Variance	9,184.30	109,574.80	0.28
Kurtosis	19,247.07	7,683.98	1,046.00
Skewness	126.75	72.86	13.27
Range	16,948.50	41,678.73	47.95
Minimum	0.00	0.25	0.05
Maximum	16,948.50	41,678.98	48.00
Count	58,613	57,677	57,677
Underground			
Mean	3.33	123.75	1.40
CV	9.62	5.76	0.33
Median	0.69	20.23	1.50
Mode	0.09	0.86	1.50
Standard Deviation	32.05	712.32	0.46
Sample Variance	1,027.03	507,398.53	0.21
Kurtosis	2,710.76	530.30	65.37
Skewness	43.93	19.42	2.92
Range	2,519.82	27,949.89	14.19
Minimum	0.09	0.86	0.01
Maximum	2,519.90	27,950.74	14.20
Count	17,614	1,7614	17,614
Total			
Mean	2.31	48.34	1.48
CV	36.99	9.35	0.35
Median	0.37	5.83	1.50
Mode	0.09	0.86	1.50
Standard Deviation	85.44	452.08	0.52
Sample Variance	7,299.65	204,375.33	0.27
Kurtosis	23,438.59	2,465.92	895.40
Skewness	138.07	39.91	11.46
Range	16,948.50	41,678.73	47.99
Minimum	0.00	0.25	0.01
Maximum	16,948.50	41,678.98	48.00
Count	76,227	75,291	75,291

Table 17.6 Brucejack Summary Composite Statistics by Domain

	Total	WZ	WZFW	SZ	G0	F4	SG	BZ	BZLG	V0K
Ag Composites										
Mean	46.37	82.89	9.30	24.84	8.52	14.80	4.43	8.34	6.58	12.98
CV	5.64	4.39	2.06	3.28	5.65	2.96	1.48	2.43	3.49	9.72
Median	7.57	18.83	4.55	7.97	3.91	6.44	2.61	3.95	2.57	3.90
Mode	0.00	0.00	0.86	0.00	0.00	0.00	0.25	1.60	0.25	2.00
Standard Deviation	261.55	364.17	19.20	81.49	48.12	43.85	6.55	19.76	22.94	126.23
Sample Variance	68,410.32	132,619.88	368.49	6,641.13	2,315.98	1,922.44	42.96	390.44	526.02	15,932.86
Kurtosis	554.13	289.46	49.32	164.05	3,254.63	272.44	36.24	265.00	653.94	1,089.78
Skewness	20.07	14.58	6.38	11.22	53.18	13.90	4.73	12.87	21.40	31.82
Range	11,517.18	11,517.18	189.41	1,612.38	2,982.86	1185.73	85.23	651.17	822.94	4574.37
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Maximum	11,517.18	11,517.18	189.41	1,612.38	2,982.86	1,185.73	85.23	651.17	822.94	4,574.62
Count	48,990	24,193	333	2322	4491	3,202	1,059	6,592	4,183	2615
Au Composites										
Mean	2.32	2.41	1.28	2.26	1.03	2.17	0.70	0.83	0.43	11.62
CV	27.56	9.04	3.25	8.62	6.85	17.98	1.76	3.17	3.70	22.79
Median	0.51	0.65	0.66	0.47	0.35	0.46	0.31	0.54	0.23	0.41
Mode	0.00	0.00	0.75	0.00	0.00	0.00	0.00	0.54	0.00	0.17
Standard Deviation	64.07	21.75	4.16	19.48	7.02	39.09	1.23	2.63	1.61	264.81
Sample Variance	4,104.50	473.30	17.31	379.48	49.29	1527.66	1.51	6.90	2.58	70,125.53
Kurtosis	14,199.99	2421.59	156.20	810.73	365.29	1923.97	30.37	1,192.18	733.47	904.49
Skewness	112.98	42.57	11.64	25.69	18.17	41.70	4.64	29.36	23.67	29.51
Range	8,909.85	1676.35	62.80	705.80	173.32	1928.79	14.01	134.40	57.02	8.909.85
Minimum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01
Maximum	8,909.85	1676.35	62.80	705.80	173.32	1928.79	14.01	134.40	57.02	8,909.85
Count	48,990	241.93	333	2322	4491	3202	10.59	6,592	4,183	261.5

17.10 Treatment of Extreme Values

The presence of high-grade outliers was evaluated by examining histograms and log-probability graphs of the domain-coded and composited grade data for the defined mineralization domains, and where possible the observed correlations between gold and silver were also taken into account. For the West Zone, West Zone Footwall, Shore Zone and VOK Zone, estimation was done using median indicator kriging, and thresholds for these domains were derived from the high-grade bin. Threshold values were selected that reduce the influence of high-grade outliers during linear estimation while minimizing changes in the composite sample distribution, and composites were capped to this value prior to estimation(Table 17.7).

Table 17.7 Brucejack Capping/Threshold Values

Commodity	Au g/t	Ag g/t
Bridge Zone	8	200
Bridge Zone Halo	8	80
Galena Hill	20	200
Gossan Hill	20	200
SG Zone	10	200
Shore Zone	80	2,100
West Zone	130	4,000
West Zone Footwall	130	4,000
VOK	130	1,000

17.11 Continuity Analysis

For the Bridge Zone, Bridge Zone Halo, Galena Hill, Gossan Hill and SG Zone domains, omnidirectional experimental semi-variograms were modelled from uncapped composite data using a normal-scores transformation (Table 17.8). The downhole variogram was viewed at a 1.5 m lag spacing (equivalent to the composite length) to assess the nugget variance. Nugget and standardized spherical models were used to model the experimental semi-variograms in normal-score transformed space. Semi-variogram model ranges were then checked and iteratively refined for each model relative to the overall nugget variance. Back-transformed variance contributions were calculated for grade interpolation. Continuity ellipses based on the semi-variogram models were then generated for each variable in each domain and used to define the appropriate search ellipses.

Table 17.8 Brucejack Experimental Semi-Variograms

Domain	Element	Experimental Semi-Variogram
Bridge Zone	Ag	0.09 + sph(0.75, 15) + sph(0.15, 260)
	Au	0.30 + sph(0.64, 10) + (0.06, 160)
Bridge Zone Halo	Ag	0.14 + sph(0.63, 15) + sph(0.23, 280)
	Au	0.28 + (sph(0.62, 9) + sph(0.09, 50)
Galena Hill	Ag	0.23 + sph(0.70, 13) + sph(0.07, 100)
	Au	0.40 + sph(0.58, 10) + sph(0.02, 70)
Gossan Hill	Ag	0.23 + sph(0.73, 13) + sph(0.03, 150)
	Au	0.32 + sph(0.61, 12) + sph(0.07, 120)
SG	Ag	0.03 + sph(0.46, 6) + sph(0.51, 20)
	Au	0.15 + sph(0.59, 6) + sph(0.26, 20)

For the West Zone, West Zone Footwall, Shore Zone and VOK domains, median indicator semi-variograms were modelled from uncapped composite data based on observed breaks between high-grade and low-grade sample populations (Table 17.9). Normal-score semi-variograms for each of the three principal directions were calculated, and the horizontal and across-strike directions were, in general, aligned with observed mineralization trends, with the dip-plane direction being variable.

Table 17.9 Brucejack Median Indicator Semi-Variograms

Domain	Element	Units	Indicator	Experimental Semi-Variogram
West Zone	Ag	g/t	66	0.3 + sph (0.4, 5/5/5) + sph (10/10/10)
	Au	g/t	4	0.3 + sph (0.5, 10/10/5) + sph (0.2, 20/50/10)
West Zone Footwall	Ag	g/t	66	0.3 + sph (0.4, 5/5/5) + sph (10/10/10)
	Au	g/t	4	0.3 + sph (0.5, 10/10/5) + sph (0.2, 20/50/10)
Shore Zone	Ag	g/t	80	0.3 + sph (0.40, 30/20/5) + sph (0.30, 50/30/10)
	Au	g/t	4	0.2 + sph (0.55, 15/10/5) + sph (0.25, 50/30/10)
VOK	Ag	g/t	68	0.30 + sph (0.70, 100/100/20)
	Au	g/t	10	0.30 + sph (0.70, 100/100/20)

17.12 BLOCK MODELS

The identified Brucejack mineralization domains extend along a corridor 500 m wide and 3,500 m in length. In order to facilitate mine planning and optimization, a single orthogonal block model was established across the property using a 10 m x 10 m x

10 m block size (Table 17.10). The block model consists of separate models for gold estimated grades, silver estimated grades, indicator kriging probabilities, associated rock codes, percent, density and classification attributes and a calculated gold equivalent 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 are 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 deleted from the model prior to estimation.

Table 17.10 Brucejack Block Model Setup

	Origin	Blocks	Size (m)
X	425,800	200	10
Y	6,256,500	350	10
Z	2,000	140	10
Rotation	0°	-	-

17.13 ESTIMATION AND CLASSIFICATION

The mineral resource estimate was constrained by wireframes that form hard boundaries between the respective composite data files. Individual block grades were used to calculate a gold equivalent block grade model.

For the Bridge Zone, Bridge Zone Halo, Galena Hill, Gossan Hill and SG Zone mineralization domains, block grades were estimated using ordinary kriging of composite values. A two-pass series of expanding search ellipses with varying minimum sample requirements was used for sample selection, estimation and classification.

During the first pass, eight to twelve composite values from three or more drill holes 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.

During the second pass, blocks not populated during the first pass were estimated. Three to twelve composite values from one or more drill holes within a search ellipse corresponding to about 200% of the defined range were required for estimation. All block grades estimated during the second pass were classified as Inferred. All SG Zone and Bridge Zone Halo mineral resources were classified as Inferred.

For the West Zone, West Zone Footwall, Shore Zone and VOK mineralization domains the block estimates were calculated using median indicator kriging. Based on the defined indicator semi-variograms, for each block a high-grade probability, high grade estimate and low-grade estimate were calculated and then combined into a single block estimate. A three-pass series of expanding search ellipses with varying minimum sample requirements was used for sample selection, estimation and classification.

During the first pass, six to twelve composite values from two or more drill holes within a search ellipse corresponding to 50% of the defined range were required for estimation. All block grades estimated during the first pass were classified as Measured. This level of classification was applied only to the West Zone, where extensive surface and underground drilling has defined the continuity of the mineralization.

During the second pass, blocks not populated during the first pass were estimated. Four to twelve composite values from two or more drill holes within a search ellipse corresponding to 100% of the defined range were required for estimation. All block grades estimated during the second pass were classified as Indicated.

During the third pass, blocks not populated during the first or second pass were estimated. Three to twelve composite values from one or more drill holes within a search ellipse corresponding to about 200% of the defined range were required for estimation. All block grades estimated during the third pass were classified as Inferred. All VOK mineral resources were classified as Inferred.

17.14 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. Commodity prices are based on the three-year trailing average as of December 31, 2010. 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 Economic Parameters

Item	Cost/Unit
Mining Cost	US$1.75/t
Processing Cost + G&A	US$7.00/t
Pit Wall Slope Angle	45°
Au Price	US$1,025.00/oz
Ag Price	US$16.60/oz
Au Recovery	71%
Ag Recovery	70%
AuEq Cut-off	0.299 g/t

All mineral resources were reported against a 0.30 g/t AuEq cut-off, as constrained within the optimized pit shell (Table 17.12, Table 17.13, and Table 17.14).

Table 17.12 Brucejack Estimated Mineral Resources Based on a Cut-Off Grade of 0.30 g/t AuEq(1)(2)(3)

Category	Tonnes (M)	Au (g/t)	Ag (g/t)	Contained(3)	
				Au (oz x '000)	Ag (oz x '000)
Measured	11.7	2.25	75.56	846	28,423
Indicated	285.3	0.80	9.57	7,338	87,782
M+I	297.0	0.86	12.17	8,184	116,205
Inferred	542.5	0.72	8.67	12,558	151,220

(1) Mineral resources for the February 2011 estimate are defined within a Whittle™ optimized pit shell that incorporates project metal recoveries, estimated operating costs and metals price assumptions. Parameters used in the estimate include metals prices (and respective recoveries) of US$1,025/oz Au (71%) and US$16.60/ozAg (70%). The pit optimization utilized the following cost parameters: mining US$1.75/t, Processing US$6.10/t and G&A US$0.90/t along with pit slopes of 45 degrees.. 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, marketing or other relevant issues. The mineral resources in this news release were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM)"CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines" prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.

(2) The quantity and grade of reported Inferred resources in this estimation are uncertain in nature.There has been insufficient exploration to define these Inferred resources as either an Indicated or Measured mineral resource, and it is uncertain if further exploration will result in upgrading them to an Indicated or Measured mineral resource category.

(3) Contained metal may differ due to rounding.

Table 17.13 Brucejack 5.00 g/t AuEq Mineral Resource Grade and Tonnage Estimate(1)(2)(3)(4)

Category	Tonnes (M)	Au (g/t)	Ag (g/t)	Contained(3)	
				Au (oz x '000)	Ag (oz x '000)
Measured	1.947	7.95	241.25	498	15,102
Indicated	1.722	7.33	123.19	406	6,820
M+I	3.669	7.66	185.84	903	21,922
Inferred	4.707	12.54	49.24	1,898	7,452

(1) (2) (3) See footnotes to Table 17.12.

(4) This high-grade resource estimate is a subset of the bulk-tonnage resource estimate and as such is included within the bulk-tonnage resource estimate and is not in addition to the bulk-tonnage resource estimate.

Table 17.14 Brucejack 3.00 g/t AuEq Mineral Resource Grade and Tonnage Estimate(1)(2)(3)(4)

Category	Tonnes (M)	Au (g/t)	Ag (g/t)	Contained(3)	
				Au (oz x'000)	Ag (oz x '000)
Measured	3.495	5.43	177.98	610	19,999
Indicated	4.940	4.62	69.33	734	11,011
Mea +Ind	8.435	4.96	114.35	1,344	31,010
Inferred	9.637	7.80	40.74	2,417	12,623

(1) (2)(3) See footnotes to Table 17.12.

(4)This high-grade resource estimate is a subset of the bulk-tonnage resource estimate and as such is included within the bulk-tonnage resource estimate and is not in addition to the bulk-tonnage resource estimate.

17.15 Validation

The block model was validated visually by the inspection of successive section lines in order to confirm that the block model correctly reflects the distribution of high-grade and low-grade samples.

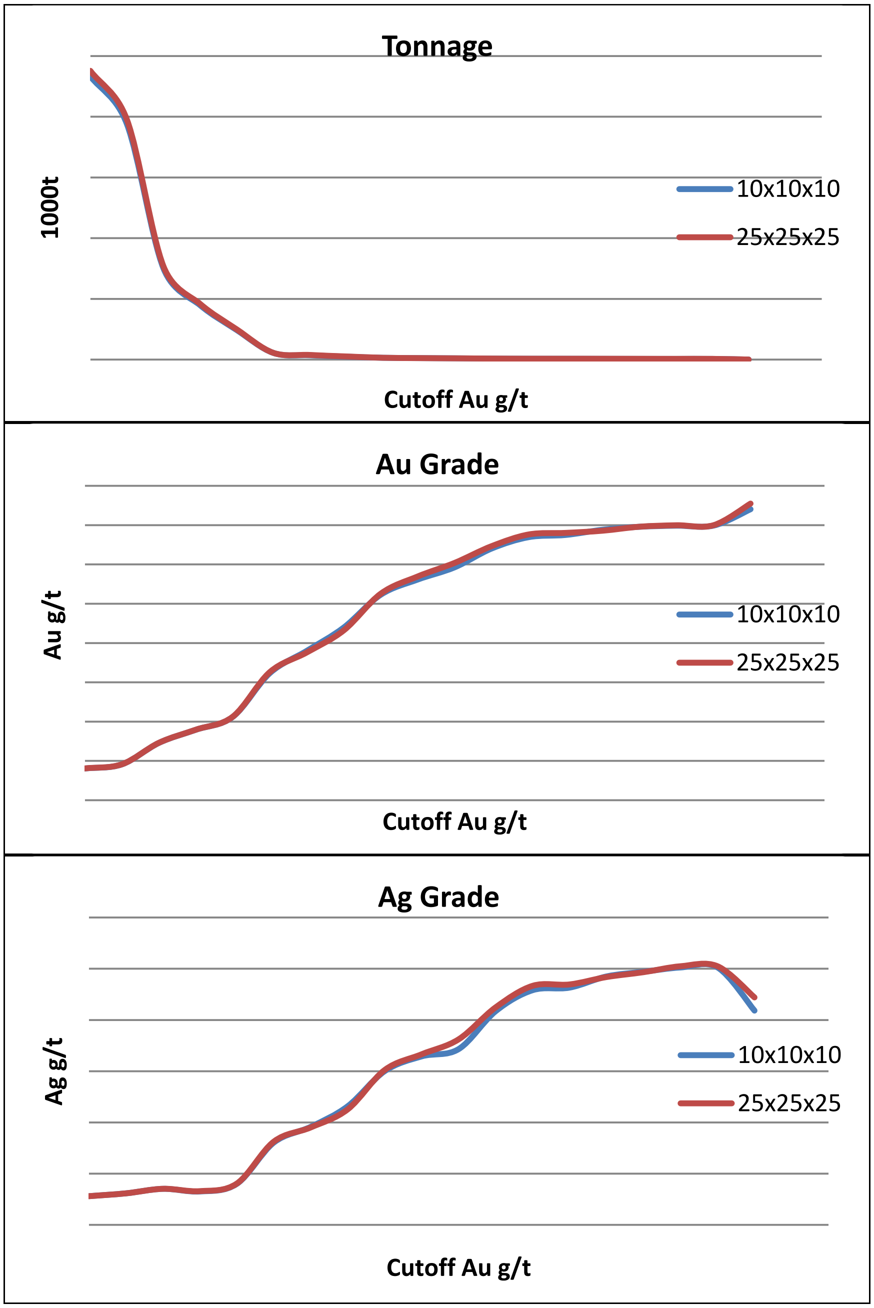
An additional validation check was completed by comparing model block grade estimates to the average grade of capped composites within a corresponding block (Table 17.15). For the domains estimated by ordinary kriging, the observed differences in grades suggest a minimal bias. For the West Zone, West Zone Footwall and Shore Zone a high degree of correlation between the composite average grades and model block estimates was also observed. For the VOK model the presence of multiple very high-grade samples and mixed sample populations has biased the overall comparison of average grades, and indicates that additional work is recommended in order to further differentiate the high and low grade sample populations.

In order to evaluate the conditional bias associated with the use of small blocks across the project area, grade/tonnage curves were calculated against a series of gold cut-offs for both the 10 m x10 m x 10 m estimates, and for a series of 25 m x 25 m x10 m estimates for the Bridge Zone, Bridge Zone Halo, Galena Hill, Gossan Hill and SG domains. The results indicate that, on a global scale, the conditional bias is not significant at this stage (Figure 17.22).

Table 17.15 Validation Statistics and Correlation Coefficients for Composite and Block Estimates

	Ag Composites	Ag Block	Au Composites	Au Block
Galena Hill, Gossan Hill, SG Zone, Bridge Zone				
Mean g/t	8.10	7.93	0.64	0.67
CV	1.72	1.11	1.16	0.84
Median g/t	4.43	5.22	0.46	0.54
Mode	0.00	0.76	0.00	0.26
Standard Deviation	13.90	8.81	0.75	0.57
Sample Variance	193.08	77.67	0.56	0.32
Kurtosis	57.00	22.12	25.69	16.93
Skewness	6.43	3.76	4.22	3.20
Range	195.30	111.54	7.93	6.70
Minimum g/t	0.00	0.04	0.00	0.00
Maximum g/t	195.31	111.58	7.93	6.70
Count	4,035	4,035	4,002	4,002
Correlation Coefficient	0.61		0.75	
West Zone, West Zone Footwall, Shore Zone				
Mean g/t	72.85	69.40	2.11	2.02
CV	3.43	1.67	4.04	1.76
Median g/t	19.69	29.71	0.75	0.96
Mode	0.00	0.00	0.00	0.00
Standard Deviation	249.63	115.71	8.53	3.57
Sample Variance	62,313.05	13,387.88	72.80	12.75
Kurtosis	456.01	23.08	316.94	37.78
Skewness	16.71	4.04	15.64	5.25
Range	9,006.84	1,361.20	217.86	47.92
Minimum g/t	0.00	0.00	0.00	0.00
Maximum g/t	9,006.84	1,361.20	217.86	47.92
Count	4,447	4,447	4,447	4,447
Correlation Coefficient	0.50		0.51	
VOK				
Mean g/t	12.45	10.99	14.09	1.73
CV	5.33	1.80	12.87	3.09
Median g/t	4.69	6.09	0.51	0.57
Mode	4.35	3.23	0.54	0.13
Standard Deviation	66.36	19.75	181.35	5.33
Sample Variance	4,403.69	389.87	32,886.05	28.36
Kurtosis	406.57	35.74	256.39	174.64
Skewness	19.13	5.42	15.83	11.10
Range	1,455.36	184.44	3101.27	95.15
Minimum g/t	0.42	0.72	0.01	0.04
Maximum g/t	1,455.78	185.15	3,101.28	95.20
Count	557	557	557	557
Correlation Coefficient	0.36		0.54	

Figure 17.22 Global Conditional Bias Check



18.0 OTHER RELEVANT DATA AND INFORMATION

18.1 MINING OPERATIONS

18.1.1 INTRODUCTION

For the purpose of this PEA, mining at the Brucejack Project will commence as an underground operation using longhole open stoping methods with rock or paste backfill where applicable.

At the end of the underground mine life, an open pit operation will commence mining near surface mineralization.

The nominal production rate for the project is 1,500 t/d.

Sections 18.1.3 to 18.1.16 details the underground mine plan, and Sections 18.1.17 to 18.1.21 the open pit mine plan. A combined production schedule for the project is presented in Section 18.1.22.

18.1.2 3D BLOCK MODEL

The mine planning work for the PEA on the Brucejack Project is based on a block model provided by P&E. Details of the block model are given in Table 18.1.

AMC conducted a review of the resource model for familiarization purposes and to determine if the resource model provided is appropriate for the PEA assessments.

Table 18.1 Details of the Brucejack Resource Model

Type	Y	X	Z
Minimum Coordinates	6,256,500	425,800	600
Maximum Coordinates	6,260,000	427,800	2,000
User Block Size	10	10	10
Minimum Block Size	10	10	10
Rotation	0	0	0

AMC's resource model review comments are:

- The resource model has been prepared using the NAD 27 British Columbia (BC) coordinate system.
- The resource model uses coarse parent cell dimensions with respect to indicative geology dimensions obtained from the drill hole database.
- The resource model has been depleted for the current understanding of the pre- existing underground excavations (WZ lode only); however, the location requires survey confirmation. AMC does not consider this to be material to the global metal estimates in the resource model.
- The distribution of drilling in the VOK lode is sparse relative to the WZ lode. Assuming similar geological attributes, it is likely that the VOK lode will look more discontinuous following closer spaced drilling, as indicated for the WZ lode.

AMC's comments on the block model specific to the underground study are:

- AMC was required to sub-cell the resource model for planning purposes. The AMC model metal content was within 1% of the resource model metal content after sub-celling.
- AMC considers it is likely that the distribution of the global contained metal for the VOK lode may change significantly for a purpose-built underground resource model with tighter geological controls.

AMC considers the block model acceptable for the purpose of the PEA.

18.1.3 GEOTECHNICAL CONDITIONS

A geotechnical study has been completed by BGC and stoping dimensions have been based on their recommendations.

The geotechnical holes that were drilled in 2010 (SU-77, SU-82 and SU-88) were specifically targeted for determining open pit geotechnical parameters. As such, the data precludes differentiation of hanging wall, footwall, mineralized material and various zone rock mass conditions that would normally be required for a detailed underground mine design.

Based on high-level discussions and review of rock mass data with BGC, AMC has adopted longhole open stoping (LHOS) with backfill as the method for the PEA.

The selected mining method will require further detailed geotechnical assessments, geology, and resource modelling refinements (conducted at later stage studies) to confirm if the LHOS with backfill method is appropriate.

For design purposes, AMC has adopted BGC's median case (50th percentile) for hydraulic radii cut-off. AMC's assessment has assumed stope dimension of 30 m floor to floor, a 30 m strike length and an approximately 20 m transverse width, which falls within the unsupported hydraulic radii limits.

18.1.4 VALUE MODEL

The terms, parameters and metal prices used to estimate the value of the mineralized material at the mine gate (the value after offsite costs are deducted), termed the net smelter return (NSR), are summarized in Table 18.2. Metal prices were provided to AMC at the study outset to allow mine planning to proceed. These prices usually differ from the metal prices used in the financial model which is prepared immediately prior to the completion of the study. The mining plan is typically not recalculated with the financial model prices because the resulting differences normally fall within the level of accuracy of the study. The NSR ($/t of mineralized material) value was incorporated into the Brucejack resource model for cut-off assessment and mine design work.

Table 18.2 NSR Parameters – Underground

Items		Units	US$	Value
Metal Prices	Gold	$/oz	1,050.00	-
	Silver	$/oz	16.60	-
Process Gold Recoveries	West Zone	% Au	-	89.0
	Valley of Kings Zone	% Au	-	92.0
Process Silver Recoveries	West Zone	% Ag	-	87.0
	Valley of Kings Zone	% Ag	-	85.0
Selling Costs (Doré)	Metal Payable – Au	%	-	99.8
	Metal Payable – Ag	%	-	99.8
	Smelting and transport costs	$/oz	3.00	-
	Insurance	% NIV	-	0.15
Net Smelter Price	Gold	$/g	33.544	-
	Silver	$/g	0.436	-

The NSR formula is:

NSR = (Au(g/t) * NSP_Au * Rec_Au(%)) + (Ag(g/t)* NSP_Ag * Rec_Ag(%))

Where:

NSP = Net Smelter Price

Rec_Au = Process recovery of gold

Rec _Ag = Process recovery of silver

18.1.5 CUT-OFF AND PRODUCTION RATE SELECTION

'Stope shape' inventories and likely production rate ranges were estimated for a range of NSR cut-offs: $160/t of mineralized material, $180/t of mineralized material and $200/t of mineralized material.

The NSR cut-off of $180/t of mineralized material has been used for both the VOK and WZ lodes mine design. This cut-off is considered by AMC to be adequate for use in the mine design and includes a nominal $120/t-mineralized material operating cost and a notional profit margin.

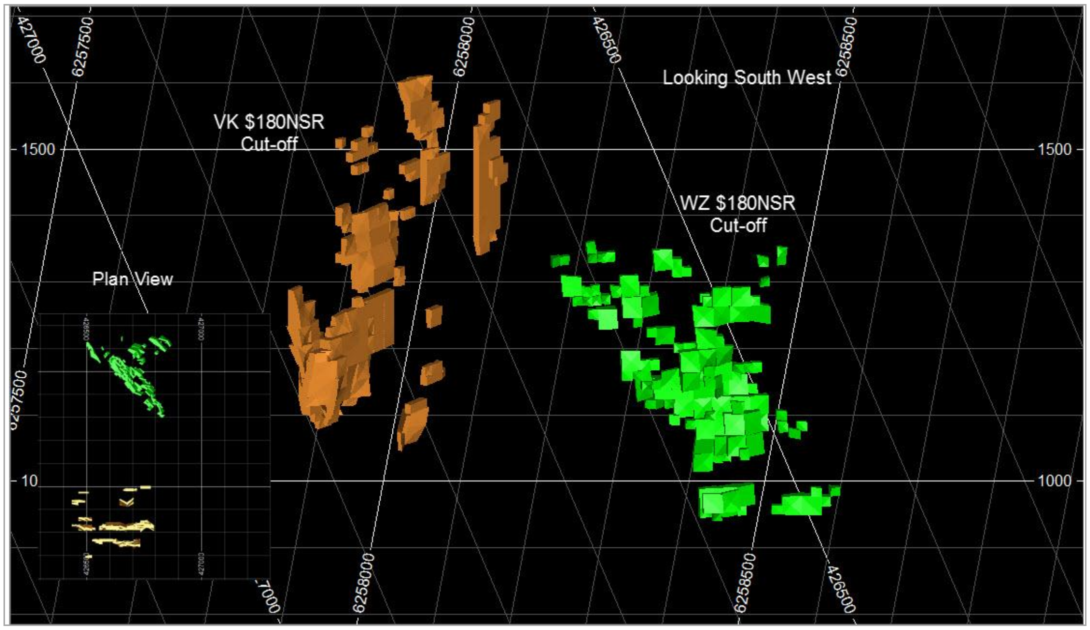
It should be noted that the 'stope shape' inventories include all resource categories (Indicated and Inferred) and are inclusive of anticipated hanging wall and footwall dilutions, but exclude other modifying factors required to convert the 'stope shape' inventory to a mining inventory. For example, the 'stope shape' inventories exclude; dilution from backfill; mineralized material losses from operational and geotechnical effects; removal of individual resource blocks based on incremental economic assessment, and; Inferred resource categorization removal.

Figure 18.1 provides schematics of the 'stope shape' inventory for the VOK and WZ lodes at $180/t of mineralized material NSR cut-off.

The likely preliminary production rate range was based on industry standard empirical rules of thumb: Taylor's Rule (Taylor, 1976), based on five times the reserve tonnes to the power of 0.75; and McCarthy's vertical advance rule (McCarthy, 1993), using a 30 vertical metre advance per annum rate (vmpa).

The notional production rate range at a NSR $180/t of mineralized material cut-off is between 1,100 t/d and 2,200 t/d. The production rate selected for the PEA schedule work is 1,500 t/d for the combined VOK and WZ lodes.

Figure 18.1 Schematics of Stope Shape Inventory above $180/t of Mineralized Material NSR Cut-off



18.1.6 *MINING METHOD*

The mining method adopted for the PEA is LHOS with backfill. The LHOS may be extracted using a primary and secondary transverse sequence, or by using a continuous longitudinal retreat.

For the current level of resource geotechnical information, AMC considers a moderately selective and moderately flexible mining method such as LHOS with backfill is appropriate and the most likely method to deliver an economically viable project with acceptable productivity and operational safety standards.

18.1.7 *MINE DESIGN*

The underground mine design was prepared in Datamine and Mine 2-4D software, and scheduling performed using Earthworks Production Scheduler (EPS)

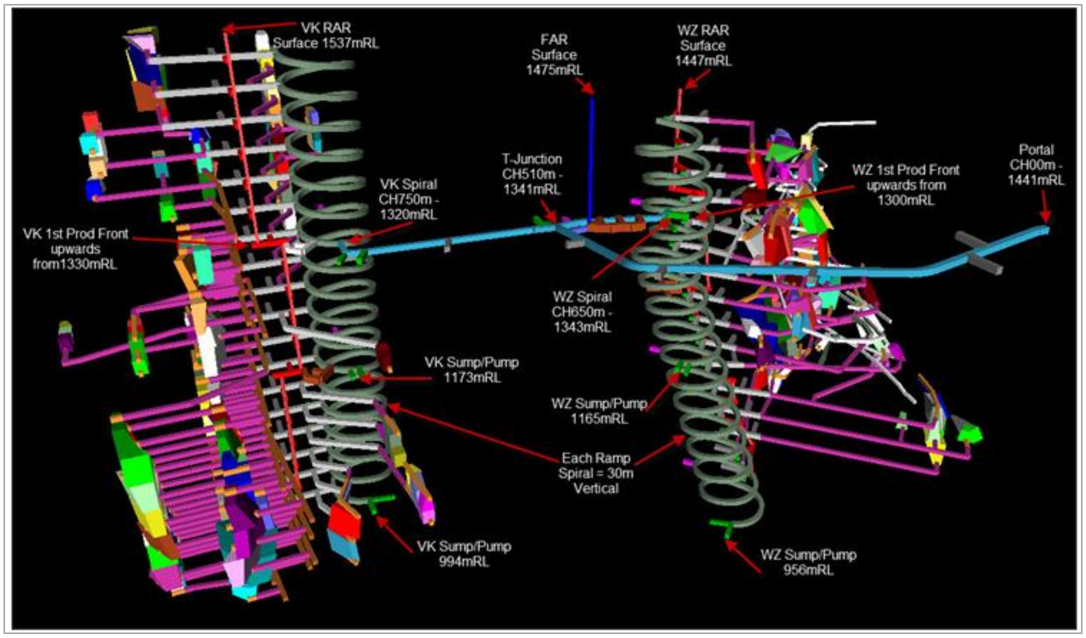
The mine design has been conducted in NAD 27 (BC) coordinates.

For mine design purposes, half height stopes and half-length stopes have been allowed into the design to maximize the mine inventory.

Outlier stopes that were deemed impractical to extract due to excessive development requirements were removed from the mine inventory. However, AMC did not undertake a thorough incremental economic assessment of all stopes (e.g. stope value versus development access cost), and therefore some stopes may prove to be uneconomic to extract. AMC does not consider the potential extent of these to be material for the PEA.

Figure 18.2 shows a general schematic of the mine design highlighting the key mine infrastructure.

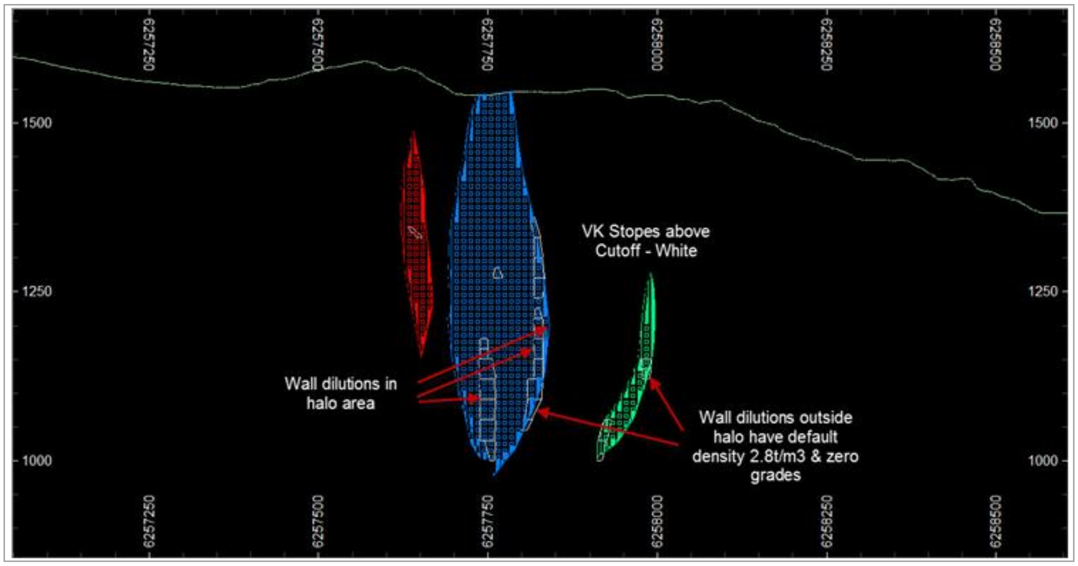
Figure 18.2 Mine Design Schematic – VOK and WZ Lodes



MINING DILUTION AND LOSSES

The designed stope walls are generally within the limits of the defined mineralized zone. The stope wall dilutions (hanging wall and footwall) therefore occur mainly in the mineralization halo area of the resource interpolation wireframes, but there are also situations where they occur along the fringes of the mineralization halo. The fringe areas are allocated a default density of 2.8 t/m^3 and zero grade (Au, Ag, AuEq). Where wall dilutions occur within the halo mineralization, they are assigned the halo grade and density from the resource model. To illustrate the situation, Figure 18.3 provides a typical section through the VOK lode showing stopes within and also along the fringes of the mineralization halo.

Figure 18.3 Typical Resource Section Depicting Stope Wall Dilutions (VOK Lode)



Stope wall and internal dilutions are based on the preliminary stoping inventories, which used the following criteria:

- minimum mining width of 5 m
- minimum waste pillar width separating payable mineralization zones of 10 m
- hanging wall dilution skin of 1 m
- footwall dilution skin of 1 m
- minimum footwall angle of 45 degrees.

Additional modifying factors (backfill dilutions and operational losses) have been applied at the schedule stage to produce the mining inventory. The factors applied were:

- backfill dilutions of 3.2% of the stope and wall dilution tonnage with zero grade (based on 0.5 m for fill wall exposure fall-off, 0.75 m for fill floor over-excavation, using a fill density of 1.8 t/m^3)
- operational losses of 6% after all dilutions.

No losses were applied for geotechnical issues or closure pillar requirements.

PRE-EXISTING DEVELOPMENT

The wireframe dimensions for the pre-existing underground ramp development in the WZ provided by Pretivm are not consistent with previous study reports that state the WZ ramp dimensions are 2.7 m (w) x 3.9 m (h) at a 15% gradient (1 in 6.7). AMC recommends a survey be conducted to confirm the WZ ramp alignment and dimensions.

It is proposed that the pre-existing development will be utilized where practical for ventilation, emergency egress and as a platform for further infill exploration diamond drilling. It may be used to opportunistically start early stope development. The current understanding of the pre-existing ramp alignment is that it traverses across the WZ mineralization. The ramp would ultimately be mined out by stoping if the current alignment interpretation is correct. Also, the pre-existing ramp dimensions are not suited to the proposed fleet size for productive and economical rock handling to the surface (mineralized material, waste, rockfill) and will require stripping to be used as an access way for the mobile fleet.

Dewatering the WZ ramp will be required prior to new mine development approaching within 50 m of the existing workings for inundation protection.

MINE ACCESS

AMC considers that the pre-existing portal location is inappropriate due to the risk of Brucejack Creek inundation (especially during thaw periods), poor access to the crusher location on surface and being within adjacent open pit blast exclusion zones (chiefly the Galena Hill pit) with a notional 300 m exclusion zone.

The location of the new portal access is generally east of the WZ lode at approximately N6258436, E427017 and RL1412 (NAD27 BC co-ordinates). This location is subject to review based on the availability of more detailed geotechnical data.

The portal will require a minor boxcut excavation, which is assumed to be meshed and shotcreted, and would be furnished with tray-up crash beam, tag board, vent status board, inundation protection, etc. The alignment is upgraded for the first 10 m for inundation protection, and is oriented to avoid unfavourable sunrise or sunset glare for exit safety.

The mine design is based on conventional rubber-tired, diesel-powered mobile mining equipment with loader mucking and truck haulage material handling. The relatively shallow depth of the lodes (approximately 400 to 500 m) and the potential for future changes to the mineralization makes the project amenable to access provided by decline development from surface.

18.1.8 MINE DEVELOPMENT

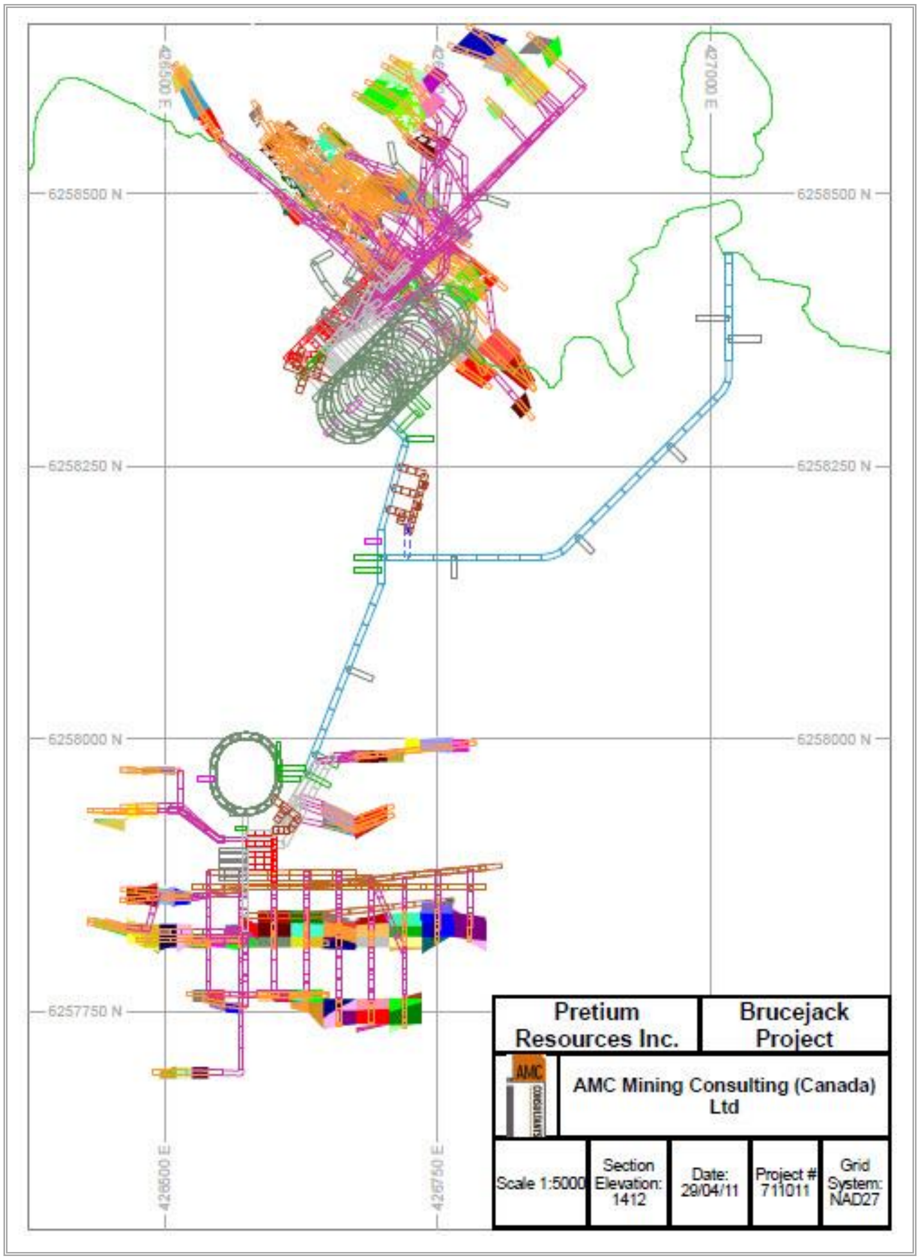
The pre-production and LOM development physicals for the combined WZ and VOK lodes are summarized in Table 18.3. The overall underground development plan is illustrated in Figure 18.4.

The LOM lateral development factor (capital and operating) is 159 tonnes of mineralized material per metre of development advance. The LOM development estimates exclude development that occurs during the production ramp-down period (project Year 14 inclusive and onwards).

Table 18.3 LOM Development Physicals Estimate

	Unit	LOM Qty.	Pre-Production Qty.
Capital Lateral			
Asbuilt Stripping	m^3	74,528	22,209
Access Decline	m	883	873
Spiral Ramp	m	6,915	450
Remuck Bays	m	995	195
Asbuilt Connections	m	296	148
Level Access	m	3,124	134
Return Airway	m	1,099	70
Fresh Airway	m	30	30
Workshop	m	152	152
Service Bay	m	185	0
Explosive Magazine	m	75	75
Electrical Substation	m	120	30
Dewater Pump Station	m	170	75
Dewater Sumps	m	510	95
Capital Vertical			
Fresh Air Raise	m	130	130
Return Air Raise	m	895	126
Return Air Raise Longhole	m	128	0
Operating Lateral			
Hanging wall Drive	m	2,644	22
Crosscut	m	10,108	86
Mineralized Material Drive	m	7,935	14

Figure 18.4 Underground Development Plan



18.1.9 MINE PRODUCTION

AMC has assumed 89 mm stope production drilling using a dedicated production drilling rig. This may be supplemented with smaller 64-76 mm production drilling utilising any spare cablebolting rig capacity (if required).

A stope drill factor of 10 t/m drilled has been assumed, which includes an allowance for slot drilling and re-drills.

The typical design stope dimensions are:

- 30 m stope height from floor to floor
- 30 m strike length
- average 16 m stope width with a range of 7-29 m for the WZ lode and a range of 7-35 m for the VOK lode.

The LOM production is summarized in Table 18.4 for the combined and separate lodes, and excludes development that occurs during the production ramp-down period (project Year 14 inclusive and onwards). The LOM production summary contains Inferred categorized material and therefore cannot be considered, nor referred to, as a Mineral Reserve.

The VOK lode represents approximately 66% of the LOM mineralized material production.

A complete mine schedule is given in Section 18.1.22.

Table 18.4 LOM Production Summary

	Unit	Total LOM Qty.	WZ Lode LOM Qty.	VOK Lode LOM Qty.
Stope Mineralized Material	('000) t	5,871	1,941	3,930
Stope Au Grade	Au g/t	10.5	6.8	12.4
Stope Ag Grade	Ag g/t	86.3	167.6	46.2
Stope AuEq Grade	AuEq g/t	11.9	9.4	13.1
Development Mineralized Material	('000) t	287	124	163
Development Au Grade	Au g/t	10.5	8.2	12.2
Development Ag Grade	Ag g/t	106.1	80.9	38.2
Development AuEq Grade	AuEq g/t	12.2	5.5	12.9
Total Mineralized Material	**('000) t**	**6,158**	**2,065**	**4,093**
Stope Au Grade	Au g/t	10.5	6.8	12.3
Stope Ag Grade	Ag g/t	87.3	162.3	45.9
Stope AuEq Grade	AuEq g/t	11.9	9.2	13.1

18.1.10 ROCK HANDLING

The mine design is based on conventional rubber-tired, diesel-powered mobile equipment with loader mucking and truck haulage material handling. The relatively shallow depth of the lodes (approximately 400-500 m) and the potential for future changes to the mineralization makes the project amenable to access provided by decline development from surface.

Material (rock) handling assumes conventional rubber-tired, diesel-powered loaders (notionally 12-17-tonne or 5-8 m^3 capacities, with remote capability) and trucks (notionally 40-50-tonne capacity).

Decline and spiral ramps are single lane with remuck bays and level accesses that may be stripped for traffic pull-out or pull-in passing. Loaded trucks will have traffic priority. Additional traffic management may be required, such as radio frequency type block lights or radar sensor block lights.

Run-of-mine (ROM) mineralized material is assumed directly truck-tipped into the primary crusher, or it will be stockpiled underground close to the portal entrance when the crusher is unavailable. The underground ROM stockpiles (2 x 30 m length) have a notional capacity of 1,600 tonnes, equivalent to approximately one day of mill feed. The development size of the portal area down to the underground ROM stockpiles will enable a surface type front-end loader (FEL) to rehandle from the underground ROM stockpiles to the primary crusher.

There will also be a surface ROM pad located adjacent to the primary crusher to handle oversize material. Other remuck bays (along the access decline) may also be opportunistically utilized for ROM stockpiling if required.

Surface waste stockpiling until stope voids become available will be required. A 200,000 tonne capacity waste dump pad is required.

Waste will be used for underground and surface roadways. Any excess waste may require subaqueous disposal into Brucejack Lake.

18.1.11 BACKFILL

Backfill consists of either un-cemented rock fill or cemented paste fill.

Paste fill has been assumed for the longitudinal retreats and primary stopes. Rock fill is used at the extremities of the longitudinal retreats and for secondary stopes.

Paste fill is sourced from thickened flotation plant tailings. Rock fill is sourced from underground development muck and other surface rock sources which will require rehandling during times when the rate of development is low. The binder material used in the paste fill is assumed to be Ordinary Portland Cement (OPC).

Voids have been opportunistically left open where they are isolated from other stopes and do not present a hazard to ongoing access and mining.

Paste fill design strengths are in the order of 0.5-1.0 MPa based on the stope dimensions of 30 m high x 30 m long and average transverse width with up to two concurrent paste fill wall exposures. An average cement addition of 5% by weight has been assumed.

The average paste fill demand is 108,000 m^3/a, over a range of 65,000 to 143,000 m^3/a. Cement demand is on average, 11,000 t/a, over a range of 6,000 to 13,000 t/a.

AMC has assumed a modular paste fill plant with a notional peak capacity of 62 m^3/h.

The paste fill plant is notionally located adjacent to the portal and process plant thickeners. The paste would be distributed to the underground workings via pipes in the access decline and in boreholes, and would use positive displacement pumps to distribute to the stope voids as required.

18.1.12 Mine Services

Ventilation

There are two ventilation intakes, the portal and a centrally-located fresh air raise (FAR) located midway between the VOK and WZ lodes. The WZ ramp connections to the new design will also be opportunistically used for ventilation purposes. It should be noted that the FAR may also be required to operate as a return air raise (RAR) at different periods of the mine life, e.g. initially to be used as a RAR during decline development to the two lodes.

There are two ventilation returns, one for each zone, located adjacent to each level access. Main RAR fans will notionally be located underground in vent walls. A suitable evasse will be required at the surface holing point of each surface raise to manage the snow mass, which can be up to 20 m deep.

AMC has made allowance for 160 m^3/sec in airflow rate for the WZ lode (representing approximately 40% of the production rate) and 190 m^3/sec in airflow rate for the VOK, lode based on AMC developed empirical rules.

For mine heating, AMC has made allowance for a direct fired propane unit with 10.5 MBTU/h capacity.

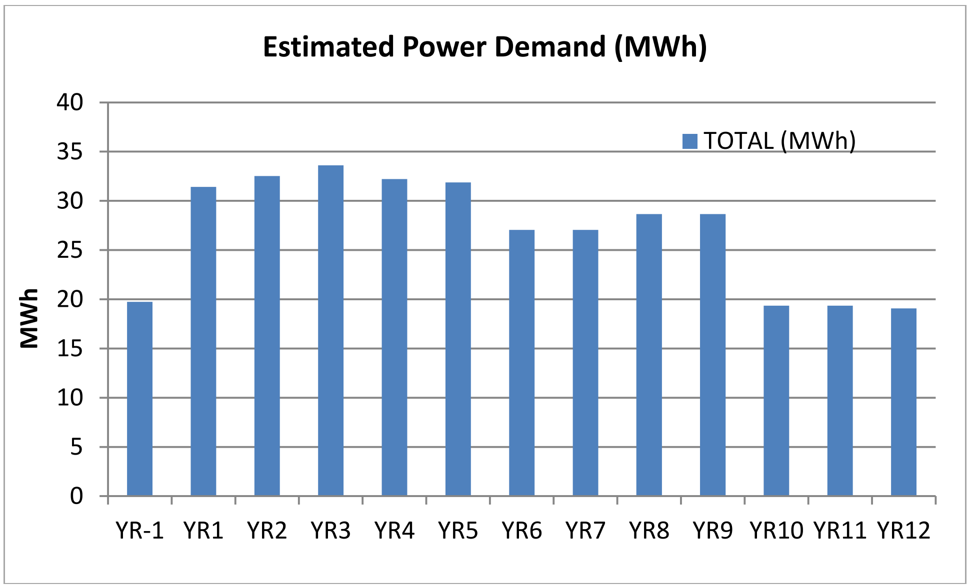
Underground Electrical Power Distribution System

There are eight electrical substations to service the mine:

- three at the WZ lode
- four at the VOK lode
- one central to the FAR, "T" Junction pump station and workshop.

AMC's estimates of power demand is based on the mine schedule and electrical infrastructure and are summarized in Figure 18.5. The pre-production demand is estimated to be 20 MWh/a and estimated operating demand peaks at 34 MWh in project Year 3.

Figure 18.5 Power Demand Estimates



Underground Communications System

Underground communications will consist of leaky feeder transmission using handheld radios for key personnel, fixed radios in light vehicles and maintenance vehicles, and base stations located at the control office, safety office and technical service offices.

The leaky feeder system may also be utilized to control the functions of the main fans, secondary fans (during blasting periods), pump systems, and, if required, tele-remote operation of the loaders.

EXPLOSIVES AND STORAGE HANDLING

An underground explosives magazine is located at the VOK lode adjacent to its RAR system.

Surface explosive magazine facilities have been assumed to be provided by the vendor at no direct upfront cost to the project, with its cost accounted for in the explosives rates.

ANFO (ammonium nitrate and fuel oil) is assumed to be the bulk explosive used for mine development and production. Packaged emulsion will be used as a primer for production and for loading lifter holes in the development headings. Smooth blasting techniques are recommended in development headings.

FUEL STORAGE AND DISTRIBUTION

No underground fuel storage or dispensing has been allowed for in the design. It is assumed that all mobile equipment will refuel at the surface facilities. Minor lube facilities will be provided in the workshop and service bays.

COMPRESSED AIR

It is proposed that all drills and charge-up units will be specified with on-board compressors. Any miscellaneous activities requiring compressed air, such as blast hole cleaning or minor handheld development in Alimak raises, will be serviced by an electric skid-mounted compressor which will be relocated as required. There will be one such compressor per lode.

WATER SUPPLY

As a preliminary water demand estimate for the mine operations, an industry factor of 400 L of water per tonne of mineralized material production has been assumed. This equates to an average demand of 7 L/sec over a 24-hour period.

The water demand estimate covers the following mine activities; production drilling, development drilling, cable bolt drilling, Alimak drilling, diamond drilling, production muckpile dust control, general access dust suppression, geology mapping, workshop activities, backfill line flushes, and fire fighting and potable water.

MINE DEWATERING

Previous study work has indicated normal ground inflows of 20-30 L/sec with peaks of 45 L/sec (Newhawk Gold Mines Feasibility Study, 1990). Diamond drill holes found to be uncapped, will be capped to minimize water transmission to underground workings.

An allowance for withdrawing 90 L/sec of water from the underground mine has been included from the two mineralization zones.

As a general overview of the main dewatering system, each mineralization zone's dewatering system (WZ and VOK) is staged back to a centrally located pump station at the decline access "T" junction (the access point to both mineralization zones). Mine water is discharged to surface from the "T" junction pump station.

Water from various areas of the mine will be collected in local sumps before being gravity fed to the nearest pump station. Boreholes will be utilized where possible between levels. Main pipes are envisaged to be installed in air egress raises as part of a modular ladderway system, and/or as cased boreholes.

EQUIPMENT MAINTENANCE

An underground work shop is centrally located to service the mobile fleet and is adjacent to the "T" junction and central FAR system.

Service bays for slower tramming equipment such as drills are located at each lode spiral ramp system to reduce minimize non-productive tramming time and to maintain appropriate utilization.

MINE SAFETY

Fire Prevention

Fire extinguishers will be provided and maintained in accordance with regulations and best practices at the electrical installations, pump stations, service workshops and wherever a fire hazard is identified to exist.

A suitable number of fire extinguishers will be provided and maintained at each stationary diesel motor and transformer substation.

Every light duty vehicle will carry at least one fire extinguisher of adequate size and proper type.

All heavy duty mobile mine equipment (loaders, trucks, drills, charge-up machines, shotcreter, etc.) will be equipped with automatic fire suppression systems.

A mine-wide stench gas warning system will be installed at the main mine intake airway entries to alert underground workers in the event of an emergency.

Mine Rescue

A mine rescue Emergency Response Plan will be developed, kept up to date, and followed in an emergency. Fully trained and equipped mine rescue teams will be established using the mine work force, and maintained on-site at all times.

A mine rescue room will be provided in the administration building. A trailer with mine rescue equipment will be located on site. The mine rescue teams will be trained for surface and underground emergencies.

Refuge Station

Self-sufficient rescue chambers (independent of a compressed air supply) with medical grade oxygen of appropriate capacity for each chamber will be provided for the active mine areas. These will be located within the average walking pace distance for the duration of a personal self-rescuer device, which will be provided as each person’s personal protective equipment.

Emergency Egress

Egress to surface will be via all raises developed by Alimak (FAR and RAR), the main access decline and the pre-existing WZ ramp.

The raises will be equipped with modular ladderways, incorporating general mine services and stages established in accordance with best practices.

Lateral egresses will have appropriate signage and be maintained for walking access.

Dust Control

All broken material (rock) will be wet down using hoses and sprays after blasting, and prior to and during mucking.

Roadway dust suppression will use a water cart with sprays on an as-needed basis.

18.1.13 Mine Equipment

The typical mine fleet required to extract the mineralized material at 1,500 t/d is summarized in Table 18.5. AMC has assumed new, current technology equipment for the project to achieve appropriate productivity, mechanical availability and operating costs.

Table 18.5 Underground Mobile Equipment List

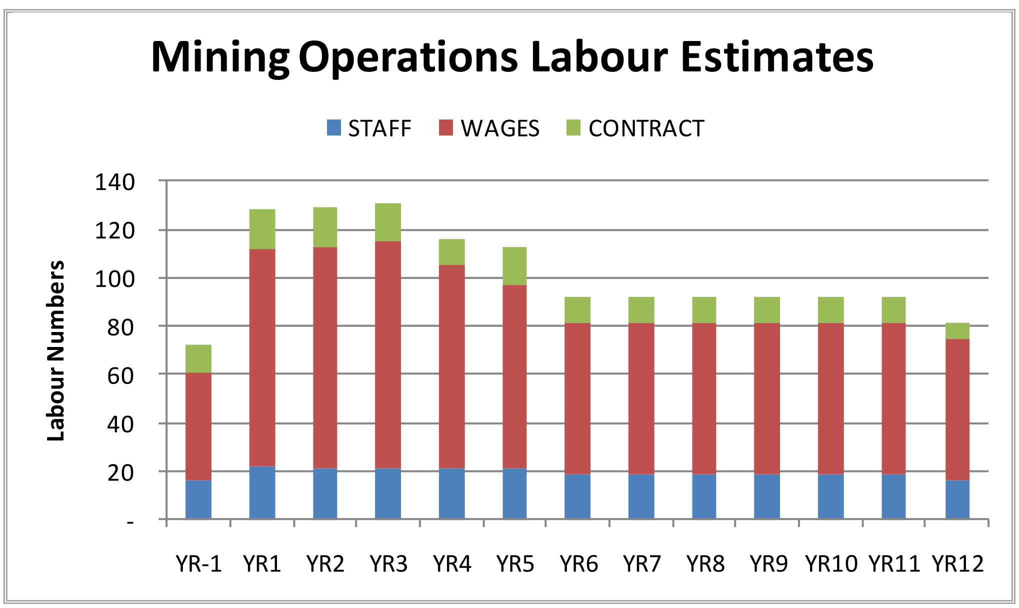
Description	LOM Qty. (ea.)	Pre-Prod Qty. (ea.)	Max Fleet Size (ea.)
Development Jumbo	5	4	5
Truck	10	1	4
Loader – Large	4	0	2
Loader – Small	5	1	3
Production Drill	4	0	1
Cable Drill	3	1	1
Charge-up Rig	7	1	3
Shotcreter and Agitator	2	1	1
Grader	1	0	1
Integrated Tool Carrier	9	1	3
Water Cart	1	0	1
Heavy Vehicle	10	1	2
Light Vehicle	47	6	10

18.1.14 PERSONNEL

The estimate of labour numbers for the mine camp sizing is summarized in Figure 18.6. The labour estimates assume a 7-day week, 2 shifts per day operation on a 7 days on – 7 days off, four panel roster.

The estimate depicts people on-site at any point in time and does not account for the off-site labour, sick leave, absenteeism, annual leave, turnover, etc.

Figure 18.6 Labour Estimate for Underground Operations



18.1.15 Underground Mining Capital Cost

The basis for the cost estimate is owner operation in Q1 2011 cost terms.

The capital cost estimate was prepared at a PEA study level, which is considered a ±30% level of accuracy on the inputs, based on the following:

- basic equipment list
- budget quotes obtained from equipment manufacturers
- in-house database
- preliminary project development plan.

The underground mining capital cost (pre production expansion) is estimated to be $55.4 million as indicated in Table 18.6, and includes capitalized operating activities that occur in the pre-production period (Year -1).

The project LOM capital cost equates to $36.70/t of mineralized material mined and $95.93/AuEq oz mined.

Table 18.6 Underground Mining Capital Cost

Description	Total Pre-Production (CDN$M)	Total Sustaining (CDN$M)
Development	20.7	91.7
Mobile Equipment	14.2	58.5
Fixed Plant	7.0	7.5
Emergency, Egress, and Safety	1.2	1.3
Ventilation	0.9	3.9
Backfill	8.1	7.7
Capitalized Operating	3.4	0.0
Total	55.4	170.6

AMC conducted high-level benchmarking of the capital estimates and considers the estimates to be appropriate for the remote project location, style of mining and production rate.

18.1.16 Underground Mining Operating Costs

The basis for the cost estimate is owner operation in Q1 2011 cost terms.

The underground mine operating costs are shown Table 18.7. The underground mine operating costs exclude processing and G&A. The operating development is notionally the hanging wall drive, cross-cuts and mineralized material drives. The underground production includes direct stoping, materials handling, backfill, mine services, technical engineering and mine supervision.

Table 18.7 Underground Mining Operating Cost

Cost Distribution	Units	Value
Development	Cdn$/t of mineralized material	16.31
Production	Cdn$/t of mineralized material	87.48
Total	**Cdn$/t of mineralized material**	**103.78**

AMC conducted high-level benchmarking of the operating unit rates and considers the estimates to be appropriate for the remote project location, style of mining and production rate.

18.1.17 Open Pit Concept

The open pit mining operation is assumed to commence once the underground operation can no longer sustain the rate of production required to feed the processing plant. Only the SG Zone (SG), Gossan Hill (GO), Galena Hill (GA) and Bridge Zone (BZ) lodes were considered for open pit mining.

18.1.18 Open Pit Optimization Inputs

Mine planning was conducted using Gemcom Software International Inc. (Gemcom) Whittle™, Gemcom Surpac™, and Microsoft Excel software. This includes block model manipulations, pit optimization, conceptual planning, and preliminary assessment level production scheduling.

The following details the inputs and the sources for the inputs that have been used for the open pit optimizations for the Brucejack Project.

Geotechnical Inputs

BGC, provided PEA order of accuracy open pit slope design criteria for the Brucejack Project. BGC recommended 45-degree pit slopes for pits less than 200 m deep. As all the pits identified were less than 200 m deep, this recommendation has been adopted.

Value Model

The NSR terms for open pit mining differ from underground mining due to the lower average grade of material. The NSR terms are summarized in Table 18.8.

Table 18.8 NSR Parameters – Open Pit

Items		Units	Parameters Used	
			US$	Value
Metal Prices	Gold	$/oz	1,050.00	-
	Silver	$/oz	16.60	-
Process Gold Recoveries	Gossan Hill (GO)	% Au	-	80.0
	Galena Hill (GA)	% Au	-	88.0
	Bridge Zone (BZ)	% Au	-	75.0
	SG Zone (SG)	% Au	-	81.0
Process Silver Recoveries	Gossan Hill (GO)	% Ag	-	75.0
	Galena Hill (GA)	% Ag	-	70.0
	Bridge Zone (BZ)	% Ag	-	65.0
	SG Zone (SG)	% Ag	-	70.0
Selling Costs (Doré)	Metal Payable – Au	%	-	99.8
	Metal Payable – Ag	%	-	99.8
	Smelting and transport costs	$/oz	3.00	-
	Insurance	% NIV	-	0.15
NSP	Gold	$/g	33.544	-
	Silver	$/g	0.436	-

The NSR formula is:

NSR = (Au(g/t) * NSP_Au * Rec_Au(%)) + (Ag(g/t) * NSP_Ag * Rec_Ag(%))

Where:

NSP = Net Smelter Price

Rec_Au = Process recovery of gold

Rec _Ag = Process recovery of silver

Operating Costs

Mining costs estimates are based on contractor mining. The estimates have been cross-referenced against other recent projects.

The costs used for the optimization are summarized in Table 18.9. The process and G&A cost shown has been used for the optimization only, and is not the final value used for economic evaluation purposes.

Table 18.9 Open Pit Operating Cost Inputs

Cost Estimate Area	Unit	Open Pit
Mining (Mineralized Material or Waste)	US$/t Mined	4.00
Process, G&A, and Others	US$/t Milled	75.00

Mining Dilution and Recovery

A mining dilution factor of 5% and mining recovery of 100% have been included in the optimization. These figures are based on highly selective mining using a small mining fleet and assume favourable mining conditions.

18.1.19 *Open Pit Optimization and Analysis*

The pit optimizations were conducted using the Lerchs-Grossman (LG) algorithm utilizing Gemcom's Whittle^{TM} software. Each block is assigned a value that 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 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.

When determining the value of a block, the grade of the block material is assessed against the recovery of the material and the sale price, minus processing and selling costs. When an NSR value is used, the recovery is 100%, as metal recoveries have already been accounted for in the NSR calculation. Similarly, the selling costs are already accounted for, and the effective sale value of the NSR is 100 %. Site mining and processing costs are allocated as per normal. This scenario creates an effective NSR break-even cut-off grade equal to the total cost of processing a tonne of mineralized material, with G&A included.

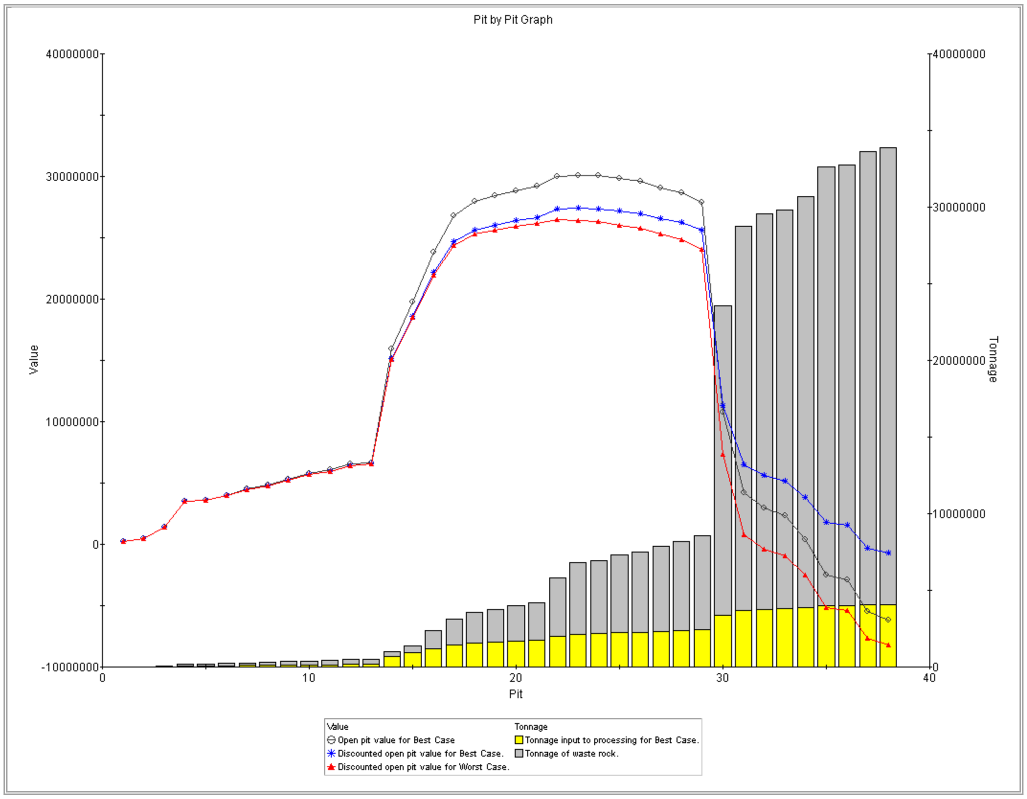
Optimization Results

Figure 18.7 shows the results of the open pit optimization.

The curves in Figure 18.7 are defined as follows:

- Black Curve: 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: 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: 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.

Figure 18.7 Whittle Results



Shell 23, which represents a revenue factor of 1 in the LG optimization, was chosen as the final pit for scheduling and economic evaluation purposes. The range of the peak NPV values for the three curves are relatively close. Due to this closeness and the relative small size of the pits, no interim phases were scheduled.

18.1.20 Open Pit Design and Scheduling

The Brucejack optimization results identified four mining areas, the SG Zone, Gossan Hill, Galena Hill, and Bridge Zone. The site plan layout is illustrated in Figure 18.8, and an oblique view looking east is illustrated in Figure 18.9.

Figure 18.8 Open Pit Layout – Plan View

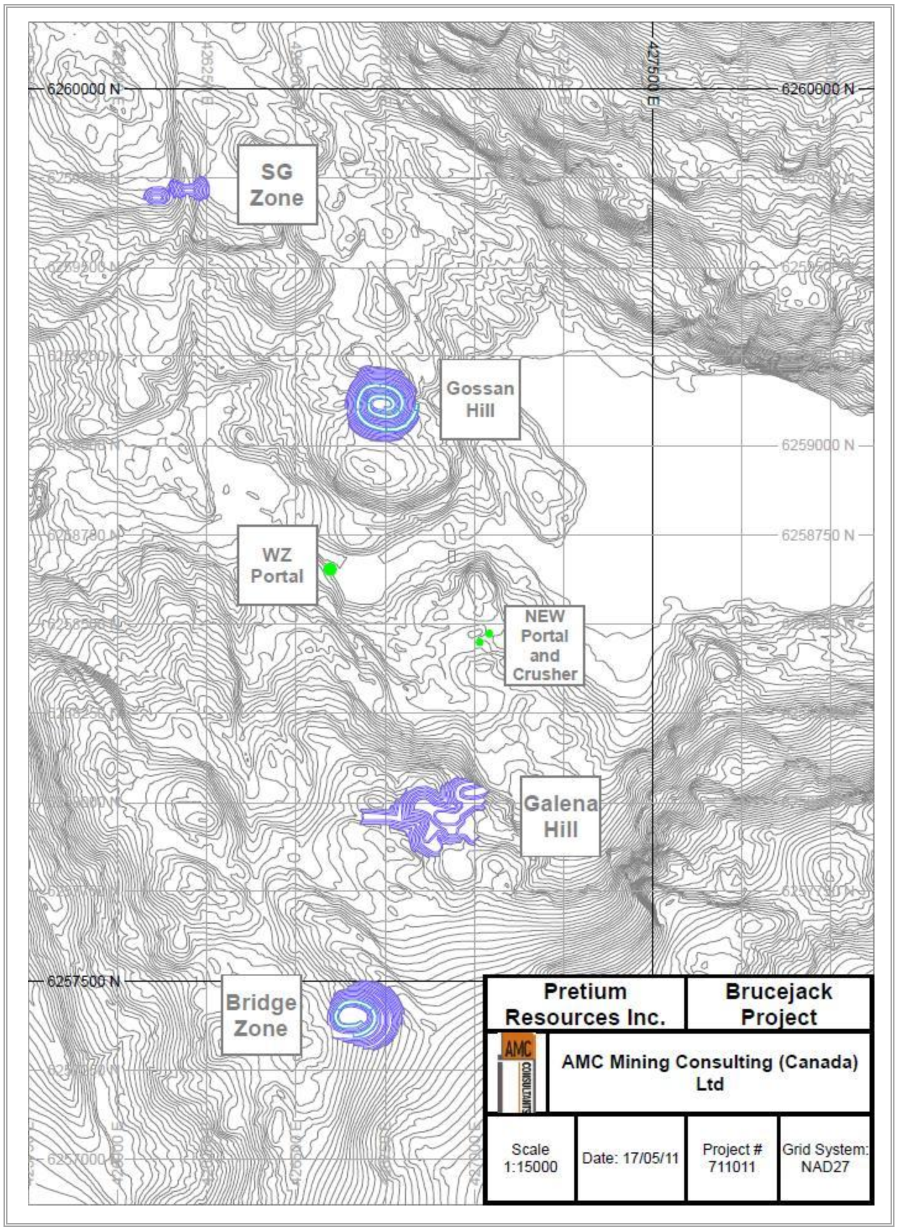
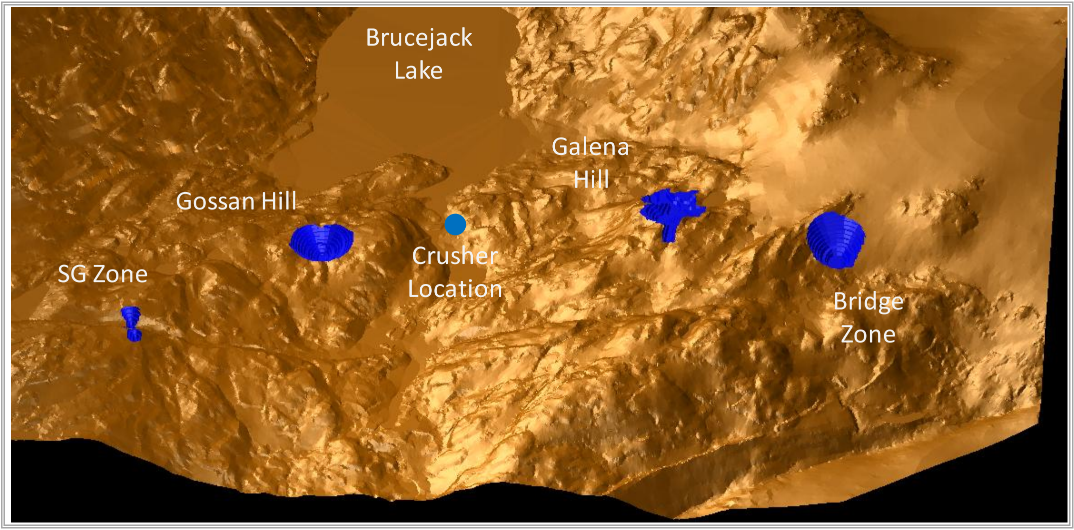


Figure 18.9 Open Pit Layout – Oblique View Looking East



Scheduling for the open pit project was completed using Microsoft Excel with the aim of consistently producing enough mineralized material to feed the processing plant once underground activities have been completed. A complete schedule is shown in Section 18.1.22.

The mineral inventory of each pit is shown in Table 18.10. The mineral inventory is based on a NSR cut-off of $75/t of mineralized material.

Table 18.10 Open Pit Mineral Inventory

Item	Unit	Total	SG	GO	GA	BZ
All Material	('000 t)	8,656	244	3,190	2,821	2,402
Waste	('000 t)	6,392	163	2680	2398	1150
Strip Ratio	Wst:Min	2.8	2.0	5.3	5.7	0.9
Mineralization	('000 t)	2,264	80	509	422	1,252
	NSR ($/t)	99	83	105	107	95
	Au (g/t)	3.3	2.9	3.7	3.3	3.2
	Ag (g/t)	36.0	19.1	15.0	32.9	46.6

The mineral inventory contains material categorized as Inferred and therefore cannot be considered, or referred to, as a mineral reserve.

18.1.21 OPEN PIT MINING OPERATIONS

Open pit mining operations are planned year round on single shift only. Mining is assumed to be undertaken by a contractor using one 75-tonne class backhoe excavator loading 46-tonne capacity trucks. An example mine fleet is shown in Table 3.5.

Table 18.11 Example Open Pit Mine Fleet

Fleet	Class	Example	Number
Backhoe Excavator	75-t Class	Komatsu PC800-8	1
Loader/Tire Handler	200 kW	Cat 966	1
Truck	46-t Capacity	Cat 772	4
Drill	229 mm	Tamrock DP1100	2
Dozer	48-t / 300 kW	D9T	1
Grader	27-t / 220 kW	Cat 16M	1
Watertruck	20-t Highway	Kenworth C500	1
Explosives Truck	20-t Highway	Kenworth T470	1
Lighting Plants	Trailer Mount	Wacker	4
Fuel/Lube Truck	20-t Highway	Kenworth T470	1
Sand truck/Snowplow	20-t Highway	Kenworth C500	1

Mineralized material and waste are planned to be blasted in 5-m benches and blasted material will be selectively mined in two operating benches of 2.5 to 3.5 m depending on the heave from blasting. For the purpose of the study, it has been assumed that grade control will be undertaken by blast hole sampling.

Mineralized material is assumed to be hauled directly to the crusher, with limited stockpiling on an as needed basis.

All waste is assumed to be disposed of into the Brucejack Lake for sub-aqueous storage. The likely method for this would be to tip waste material out into the lake in causeways, then at the end of the mine life use the backhoe excavator to side cast the material into the lake, creating a water depth of approximately three metres to the top of the waste material.

18.1.22 Combined Mine Production Schedule

The combined production schedule for the Brucejack Underground and Open Pit operations is shown in Table 18.12.

Table 18.12: Combined Production Schedule

			Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Underground	Mineralization	('000) t	**6,158**	3	407	584	587	591	555	541	535	549	563	517	504	221	-	-	-	-
		NSR ($/t)	**354.7**	178.0	302.2	292.6	374.2	328.4	382.7	303.1	342.2	393.5	469.4	380.7	331.7	325.3	-	-	-	-
		Au (g/t)	**10.5**	4.0	8.4	8.3	10.8	9.8	10.9	8.9	10.6	12.0	14.7	11.5	9.8	9.3	-	-	-	-
		Ag (g/t)	**87.3**	151.2	116.6	112.2	110.7	77.5	130.9	82.0	43.8	66.1	42.1	81.8	90.5	111.4	-	-	-	-
	Waste	('000) t	**2,327**	281	499	462	482	381	124	0	0	37	61	0	0	0	-	-	-	-
	Pastfill	('000) m^3	**1,241**	0	47	129	142	78	124	141	117	153	84	80	101	45	-	-	-	-
	Rockfill	('000) m^3	**863**	0	62	47	41	108	77	61	83	48	125	109	64	38	-	-	-	-
Open Pit	Total	('000) t	**8,656**	-	-	-	-	-	-	-	-	-	-	-	-	2,194	2,196	2,195	1,466	606
	Waste	('000) t	**6,392**	-	-	-	-	-	-	-	-	-	-	-	-	1,881	1,649	1,669	918	275
	Strip Ratio	Wst:Min	**2.8**	-	-	-	-	-	-	-	-	-	-	-	-	6.0	3.0	3.2	1.7	0.8
	Mineralization	('000) t	**2,264**	-	-	-	-	-	-	-	-	-	-	-	-	313	547	526	547	330
		NSR ($/t)	**98.9**	-	-	-	-	-	-	-	-	-	-	-	-	94.5	103.8	95.1	96.2	105.9
		Au (g/t)	**3.3**	-	-	-	-	-	-	-	-	-	-	-	-	3.0	3.4	3.3	3.3	3.7
		Ag (g/t)	**36.0**	-	-	-	-	-	-	-	-	-	-	-	-	24.6	45.3	38.7	33.9	30.2
Total	Mineralization	('000) t	**8,422**	3	407	584	587	591	555	541	535	549	563	517	504	534	547	526	547	330
		NSR ($/t)	**285.9**	178.0	302.2	292.6	374.2	328.4	382.7	303.1	342.2	393.5	469.4	380.7	331.7	190.0	103.8	95.1	96.2	105.9
		Au (g/t)	**8.6**	4.0	8.4	8.3	10.8	9.8	10.9	8.9	10.6	12.0	14.7	11.5	9.8	5.6	3.4	3.3	3.3	3.7
		Ag (g/t)	**73.5**	151.2	116.6	112.2	110.7	77.5	130.9	82.0	43.8	66.1	42.1	81.8	90.5	60.5	45.3	38.7	33.9	30.2
	Waste	('000) t	**8,719**	281	499	462	482	381	124	0	0	37	61	0	0	1,881	1,649	1,669	918	275

18.2 INFRASTRUCTURE

18.2.1 MINE AND SITE LAYOUT

The general layout of the Project is shown in Figure 18.10.

The general arrangements of the mine site and the leach plant site for the Project are presented in Figure 18.11 and Figure 18.12.

The leach plant site will be accessible by a permanent road to be constructed southwest from Highway 37 to the leach plant site. Highway 37, a major road access route to northern BC, passes approximately 8 km from the leach plant. A 70 km road from the leach plant site to the mine site will be upgraded and used to mobilize equipment and supplies.

The TSF for the leach residue storage will be located approximately 1 km southeast from the leach plant site.

Figure 18.10 Project General Layout

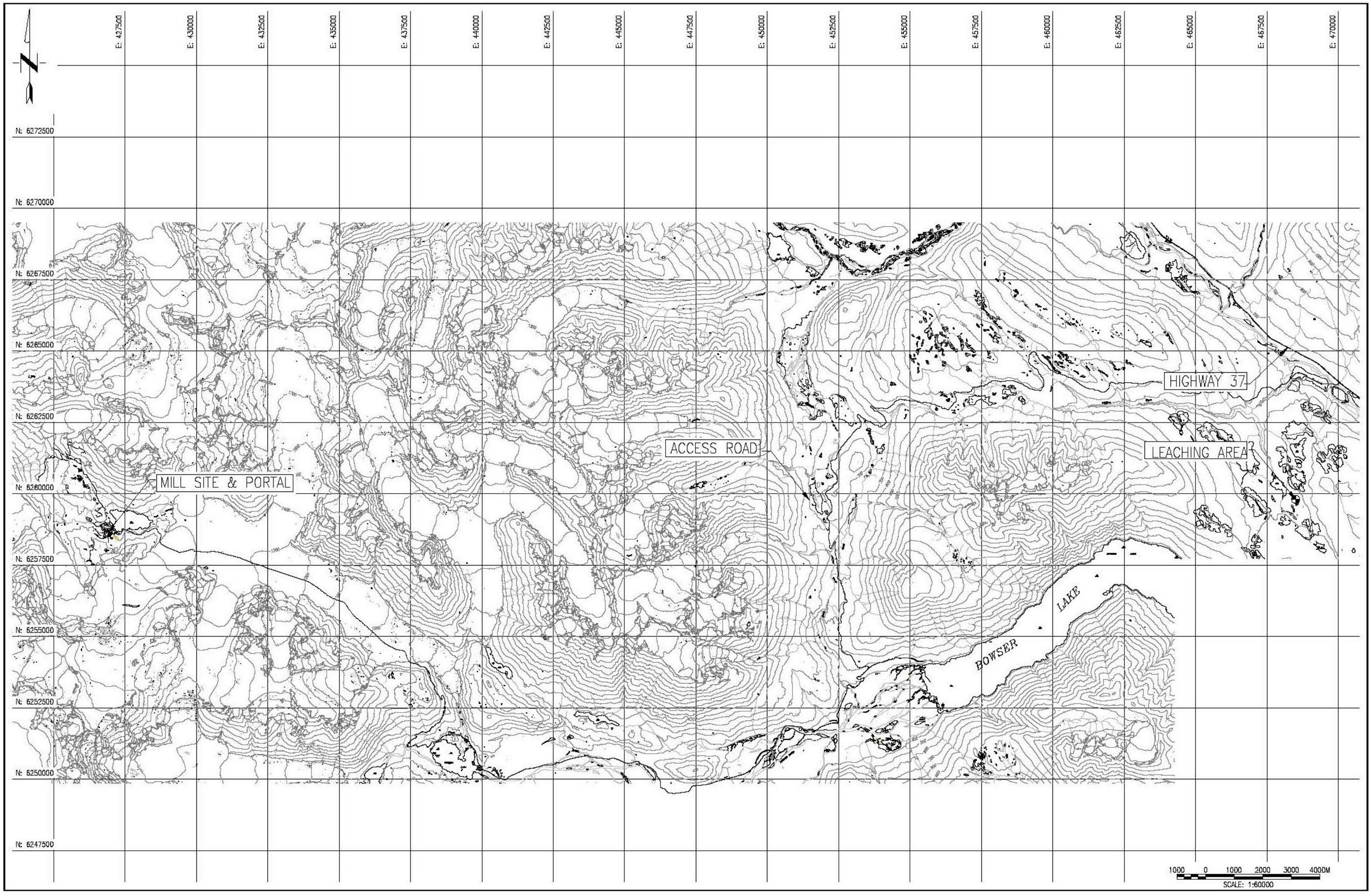


Figure 18.11 Brucejack Overall Site Plan – Mine and Process Plant

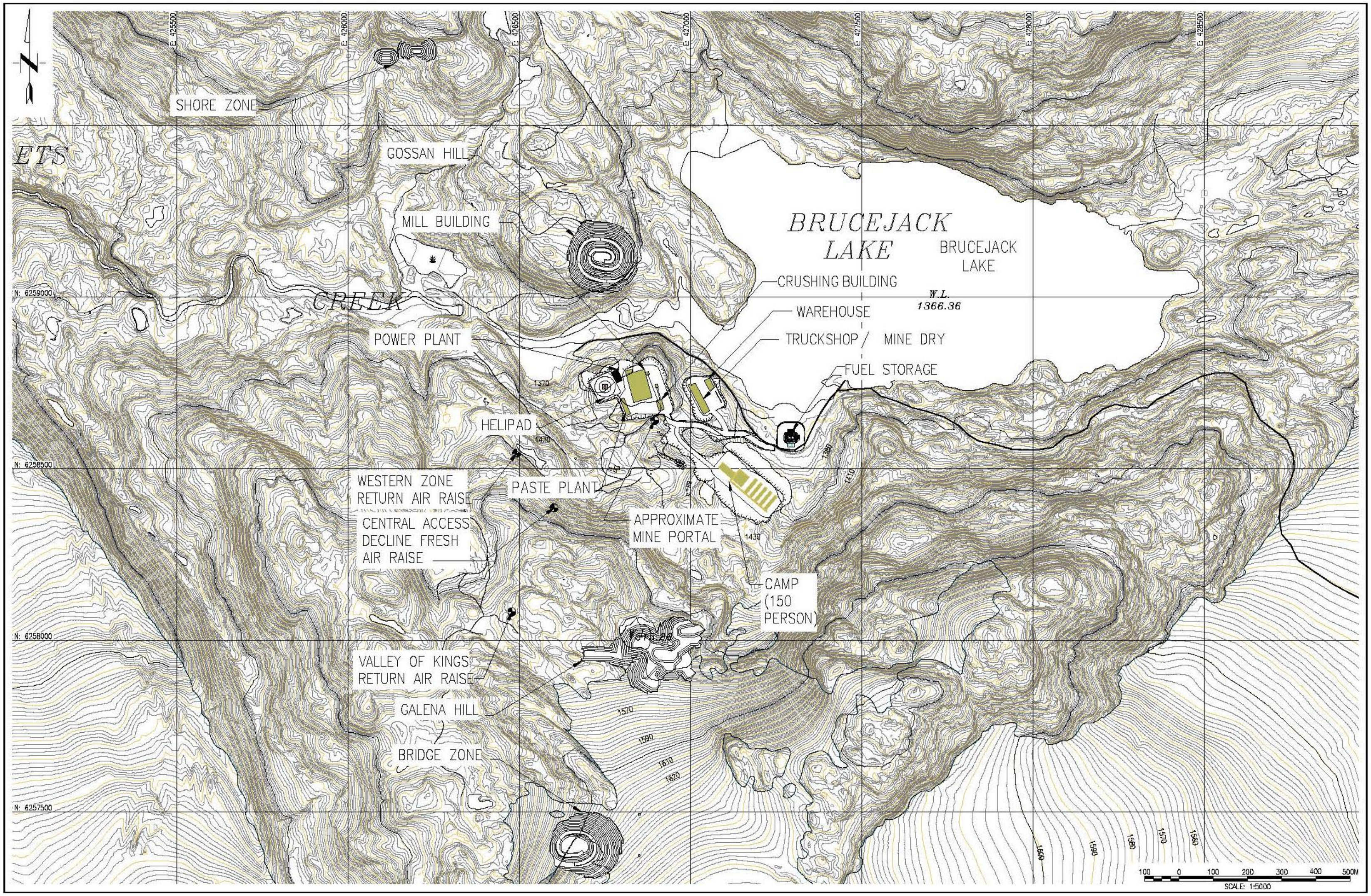
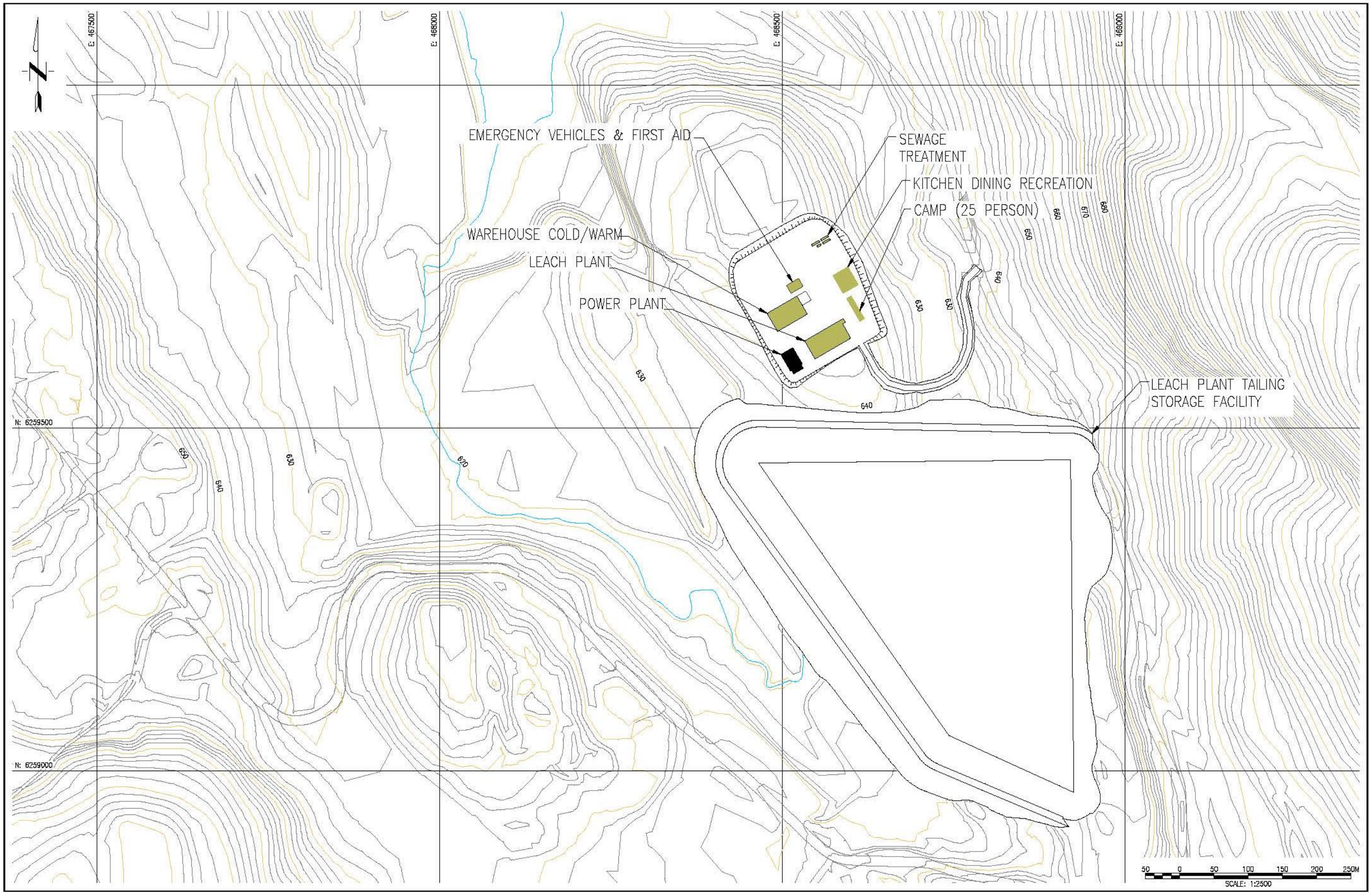


Figure 18.12 Brucejack Overall Site Plan – Leach Plant



18.2.2 ANCILLARY BUILDINGS

Pre-engineered, stick-built, and modular structures will be constructed for the Project.

Mine and process plant site buildings and facilities will include:

- a crusher building
- a mill building
- warehouse
- a truck shop
- a 150-person modular camp integrated with administration offices
- a sewage treatment plant
- a backfill paste plant.

Leach plant site buildings and facilities will include:

- a leach plant
- warehouse with secure doré storage
- maintenance shop and truck wash
- a first aid and emergency vehicle storage building
- a sewage treatment plant
- a 25-person modular camp integrated with administration offices.

A helicopter pad, power plant, and fuel storage facility and fuel station will be located at each site.

18.2.3 TRUCK SHOP

The principal function of the truck shop/warehouse complex is to service mine equipment, and to provide warehousing for the project operations. The facility will be stick built structural steel with metal clad wall and roof systems. The truck shop will include:

- two heavy duty repair bays
- one weld bay
- two light vehicle repair bays
- maintenance workshops
- a truck wash/tire change bay
- an emergency response facility
- a warehouse/mine dry
- 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 mine and the leach plant sites. Each diesel fuel storage tank will have a capacity sufficient for approximately seven days of operation. Diesel storage system will include loading and dispensing equipment. A dedicated service truck will transport diesel to the mining equipment.

18.2.5 Concentrate and Doré Storage

Gold-silver concentrate will be stored in an on-site facility capable of storing a week's worth of production. On-site, the concentrate will be loaded into trucks and transported to the leach plant for processing.

Doré will be stored in a secured vault located at the leach plant and shipped off-site on a regular basis by specialty service provider contracted by the mine.

18.2.6 Roads and Access

The leach plant site will be accessible via an upgraded road from Highway 37. In addition, an approximately 70 km long exploration road from the leach plant site to the mine site will be upgraded. Both roads will be accessible year round and 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:

- an upgrade to the main access road from Highway 37 to the leach plant site
- grading of the mine and leach plant sites
- an upgrade to the exploration road from the leach plant site to the Brucejack mine site
- miscellaneous site roads
- the TSF construction and operation access road
- grading of the pads for on-site buildings and facilities.

The portion of the main access road route roughly follows an access road that was reportedly utilized by Newhawk during their exploration activities. This road will be upgraded for Pretivm exploration purposes and will be further upgraded prior to construction start. 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 mine and leach plant site areas will require a detailed geotechnical investigation to determine the suitability of the proposed locations 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 70-km road extends from the proposed leach plant site. This road will be upgraded for Pretivm exploration purposes and will be further upgraded for construction traffic accessing the Brucejack property. Where possible, the construction access road will serve as access to the tailings pipeline/diversion ditch maintenance areas, a haul road from the rock quarries to the tailings dam, and the permanent mining truck access connecting the two sites. It is believed that the Newhawk track will need up-grading and some 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 and concentrate transportation requirements.

About 10 km of the proposed construction road route passes along and across a glacier. This stretch of the road was used by Newhawk as an exploration access road.

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 in the next phase of the study.

18.2.8 COMMUNICATIONS

The project telecommunications design will incorporate proven and reliable systems to ensure that personnel at the mine 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:

- 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.

18.2.9 *Tailings Storage Facility*

The TSF will be located adjacent to Bell Irving River and Highway 37, approximately 1 km from the leach plant.

The tailings delivery system at the mine and process plant site was designed to transport tailings to the backfill plant or to the tailings deposition area within Brucejack lake.

18.2.10 *Power/Electrical*

Operations Load

The mill throughput will be 1,500t/d. At this production level, the operations load is estimated to be approximately 4 MW±10%. The load will be divided between the mine and process site located near the Brucejack Lake and the leach plant located near Highway 37.

Power Source

Power for the mine site will be provided by four diesel generators, each rated 1.5 MW. This will provide N-1 reliability (i.e. full power can be provided even with one generator out of service). Power will be generated and distributed at 4.16 kV. Large loads, above 200 kW, including those required for the ball mill and the secondary crusher, will be fed directly at 4.16 kV. Smaller loads will be fed at 600 V. A pair of redundant step-down transformers will be installed, from 4.16 kV to 600 V.

A heat recovery system will be installed on the diesel generators to support heating of the site facilities.

Power for the leach plant near the highway will be provided by two diesel generators, each rated 1.5 MW. This will provide N-1 system reliability (i.e. full power can be provided even with one generator out of service). Power will be generated and distributed at 4.16 kV. Large loads, above 200 kW, will be fed directly at 4.16 kV. Smaller loads will be fed at 600 V.

Since smaller loads may be more distributed than those at the mine site, a pair of redundant step-down transformers will be included, from 4.16 kV to 600 V. A heat recovery system will be installed on the diesel generators to help heat the site facilities.

Remote loads required at the TSF, will be serviced via overhead 4.16 kV power lines.

Cabling will be via TECK cables in tray. Electrical heat trace will be installed on all exposed piping systems.

Conceptual one-line diagrams are shown in Figure 18.13 and Figure 18.14.

Figure 18.13 Conceptual One-Line Diagram – Mine Site

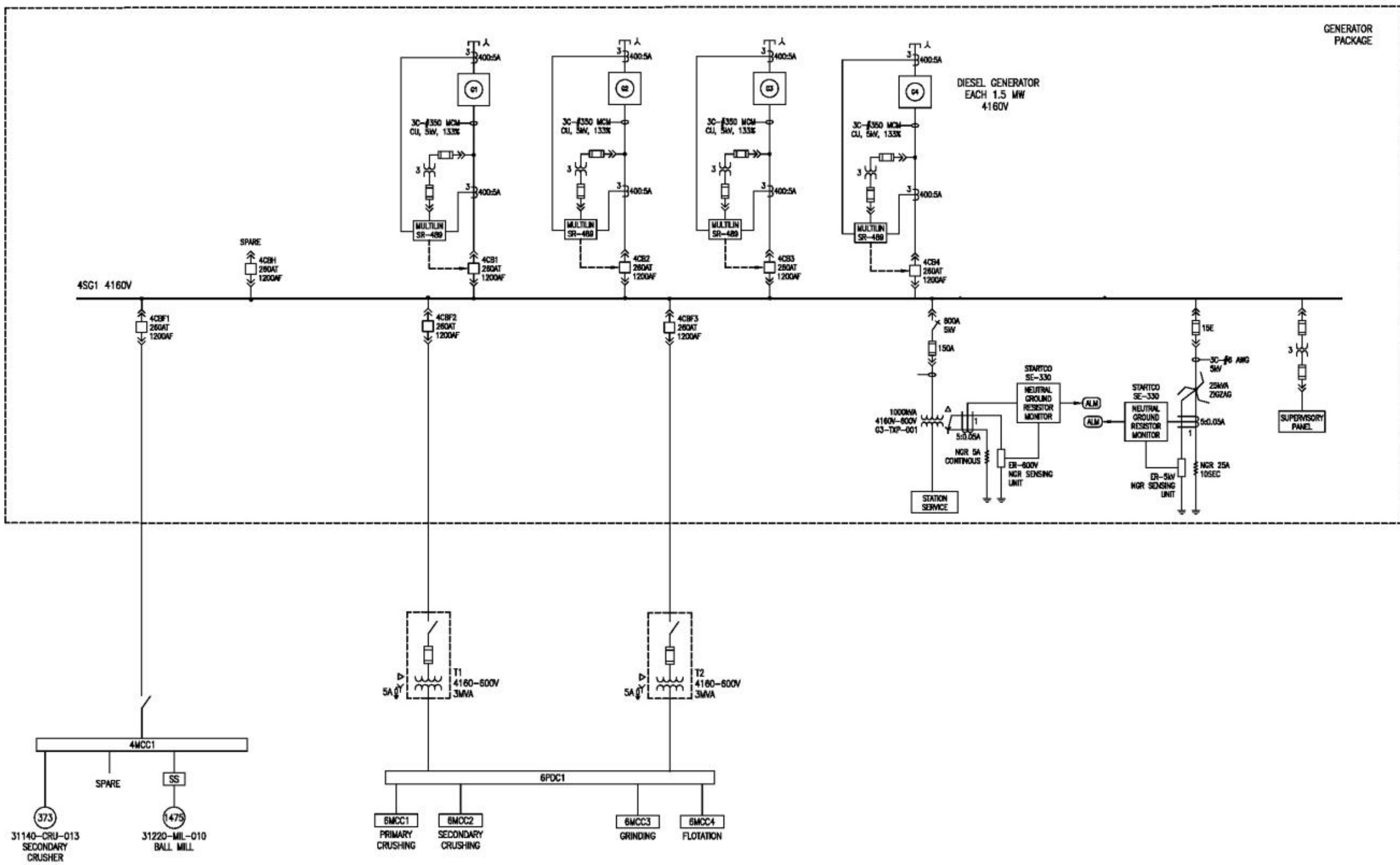
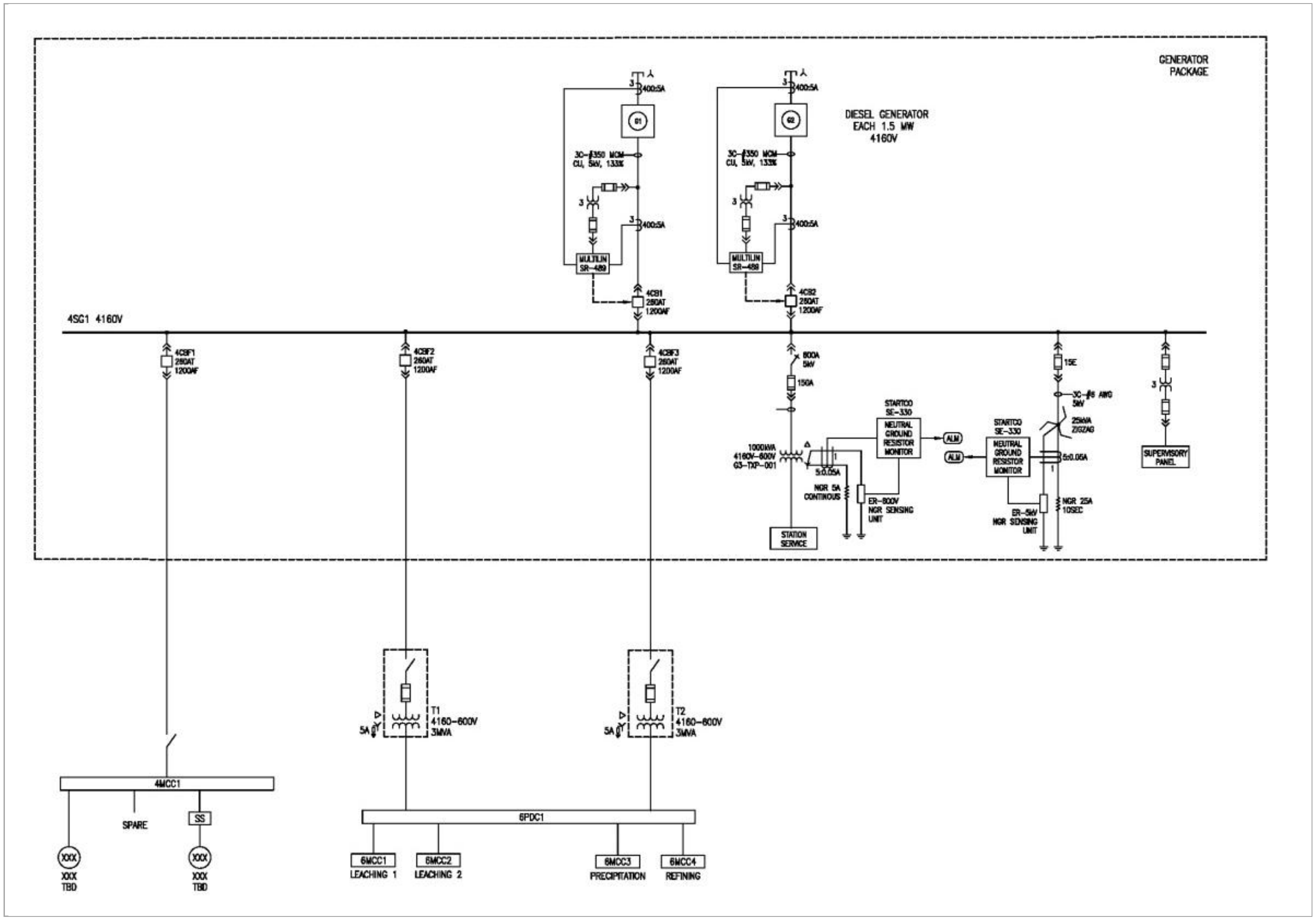


Figure 18.14 Conceptual One-Line Diagram – Leach Plant



18.3 Waste and Water Management

This section outlines the waste and water management strategies for the Brucejack Lake mine site and the leach plant facility, and associated TSF located adjacent to Highway 37 and Bell Irving River near its confluence with Wildfire Creek.

18.3.1 *Brucejack Lake Mine Site Water Management*

General

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 groundwater. As such, through its consultants, Pretivm Resources Inc. will develop a conceptual water management plan that applies to all mining activities undertaken during all phases of the Brucejack Project. 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 are incompliance with the applicable water quality levels and guidelines.

Strategies for water management include:

- 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 through recycling of water wherever possible
- monitoring the composition of release water and treating it to remove or control contaminants as required to meet discharge standards
- construction of diversion channels to direct undisturbed runoff away from mining activities.

WATER MANAGEMENT OVERVIEW

The Brucejack deposit, located west of Brucejack Lake on the east side of the Sulphurets Glacier valley, is proposed to have an underground mine and four open pits (Figure 18.15). The underground facility will be mined first (Years 1 to 12) and when depleted, the open pits will be mined (Years 12 to 16). As mining progresses, 7.2 Mt of waste rock will be deposited into Brucejack Lake along with 4.5 Mt of tailings. An additional 1.5 Mt of rockfill and 2.2 Mt of tailings paste backfill will be deposited in the underground mine. Of the total processed mineralized material, 20% will be trucked to the leach plant as concentrate for secondary processing.

DIVERSION CHANNELS

Fresh water diversion channels will be constructed around the pits and plant site. The channels will discharge to small tributaries of Brucejack Lake and Brucejack Creek. It is desirable to divert as much of this water as practical to Brucejack Lake for cover over the tailings and waste rock deposits.

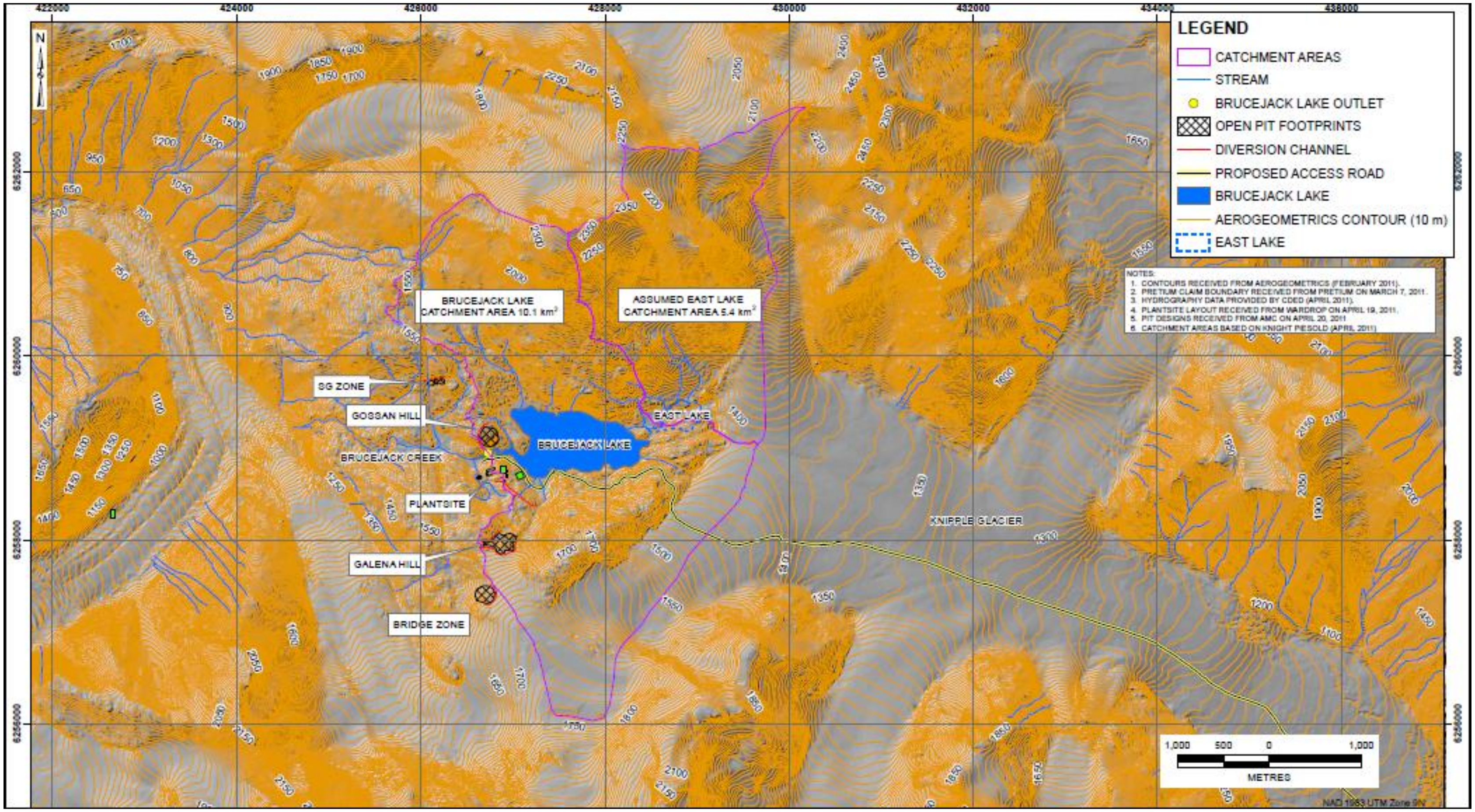
CONTACT WATER

Runoff to the open pits will be managed with a combination of sumps and pumps, and transported via pipeline to the process plant. Excess runoff to the open pits not required for process will be discharged to Brucejack Lake. Seepage to the open pits is expected to be negligible.

Average annual seepage to the 40 km of underground mine tunnels (at full development) is expected to be approximately 1,460 m^3/h. This water will be pumped to the plant for process use. Excess water will be discharged to Brucejack Lake at depth.

The project currently assumes that outflows from Brucejack Lake will be of suitable water quality for discharge to Brucejack Creek. Therefore, water treatment is not being considered for open pit runoff or excess underground seepage water. This assumption needs to be rigorously tested during the next phase of engineering design.

Figure 18.15 Proposed Mine Site Water Management Plan



PROCESS WATER REQUIREMENTS

Water requirements (890 m^3/d) for the Brucejack process plant, including fresh and potable water, will be met from the following sources:

- underground mine seepage water
- runoff from the open pits
- Brucejack Lake.

PRELIMINARY WATER BALANCE

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

- a final tailings settled dry density of 1.2 t/m^3 for the lake deposition and of 1.8 t/m^3 for the underground mine deposition
- a solids specific gravity of 2.65
- tailings production of 1,500 t/d, with 20% sent to the leach facility as concentrate for secondary processing
  - 800 t/d (53% of total production) will be deposited at depth in Brucejack Lake in a slurry of 50% solids by weight (32 m^3/h of slurry water)
  - 400 t/d (27% of total production) will be deposited in the underground mine in a backfill paste of 82% solids by weight (6 m^3/h of slurry water)
- a minimum fresh water requirement of 2 m^3/h for process and 1 m^3/h of potable water
- an average annual precipitation of 2,033 mm and potential lake evaporation and sublimation losses of 215 mm
- annual average runoff of 1,500 mm from undisturbed ground.

Based on a synthetic streamflow dataset (Knight Piésold, 2011), an average annual runoff of 1,624 m^3/h to Brucejack Lake has been estimated for the LOM. This water is likely not required for process during Years 1 – 12 because the volume from the underground mine (1,460 m^3/h) will be in excess of the process requirements. Following shut down of the underground mine, water from Brucejack Lake will be required at an average annual rate of 23 m^3/h (550 m^3/d) from Years 12 to 16 (7.1 m^3/h averaged over the LOM).

18.3.2 BRUCEJACK LAKE WASTE MANAGEMENT

SUMMARY

Two types of waste will be generated at the Brucejack Lake mine site: waste rock, (both from the underground mine and the open pits) and flotation tailings. Flotation tailings are not anticipated to be acid generating; waste rock is anticipated to be acid generating. Flotation tailings will be deposited at depth at the east end of Brucejack Lake and waste rock will be deposited at the northwest and southwest corners of the Lake and in the underground mine. Tailings and waste rock will also be deposited in the underground mine.

CRITERIA

Table 18.13 outlines the tonnages of tailings and waste rock that require disposal at the Brucejack Lake mine site. Tailings should be deposited as deep as possible to limit the potential for suspended solids near the surface of the lake. Additionally, it is assumed that stratification in the lake will prevent tailings deposited at depth from rising in the lake. Waste rock should be covered by at least 1 m of water to limit acid generation.

Table 18.13 Brucejack Lake Waste Deposition Criteria

Criteria	Description
Mineralized Material Tonnage	8.4 Mt
Throughput	1,500 t/d
Mine Life	16 years Underground – Years 1 through 12 Open pits – Years 12 through 16
Waste Rock Tonnage	Total – 8.7 Mt From underground mine – 2.3 Mt From open pits – 6.4 Mt Generated on the south side of Brucejack Lake – 5.9 Mt Generated on the north side of Brucejack Lake – 2.8 Mt Deposited underground – 1.5 Mt Deposited in Brucejack Lake – 7.2 Mt
Tailings Tonnage	Total – 6.7 Mt (20% of mineralized material goes to concentrate) Deposited in underground mine – 2.2 Mt (33% of tailings) Deposited in Brucejack Lake – 4.5 Mt (67% of tailings)
Water Cover	Maximize water cover over tailings Minimum of 1 m water cover over waste rock

DESIGN BASIS

Rescan completed a bathymetric survey of Brucejack Lake in 2010. The approximate volume of Brucejack Lake is 38 Mm^3, corresponding to a lake elevation of 1366 masl. The lake is approximately 150 m deep at its deepest location. At the east end of the lake there is a wide relatively flat area at approximately 90 m to 100 m depth. The outlet of the lake is at the west end where the lake discharges to Brucejack Creek, which drains under the Sulphurets glacier and ultimately into Sulphurets Creek.

A dry density of 1.2 t/m^3 was assumed for the tailings. The anticipated tailings volume to be deposited in Brucejack Lake is expected to be 3.8 Mm^3. A dry density of 2.1 t/m^3 was assumed for the waste rock. The total volume of waste rock to be deposited in Brucejack Lake is expected to be 3.4 Mm^3.

It was assumed that water decanted from the lake will be suitable for discharge. Acid generation from waste rock should be limited by water cover; deposition of the tailings at depth is expected to contain suspended solids at depth. It is assumed that waste rock will not leach metals at neutral pH.

TAILINGS DEPOSITION

Approximately 33% of the tailings will be deposited underground as paste backfill. Approximately 67% of the tailings will be deposited at the east end of Brucejack Lake, at an approximate depth of 70 m. Depositing the tailings at the east end of the lake will maximize the depth of deposition and the distance from the discharge point, which will minimize the potential for suspended solids discharge from the lake. There is sufficient volume in the lake below 70 m depth to store the anticipated tailings volume. Tailings will be deposited sub-aqueously, as a slurry, with an approximate slurry density of 50% (by weight).

The tailings distribution pipeline will be located on the south side of the lake along the proposed access road and extend to a depth of 70 m below the lake surface (Figure 18.16). The tailings pipeline above ground will require protection from rockfall hazards either by burying the pipe or construction of a rockfall collection bench or ditch adjacent to the tailings pipeline. The routing of the tailings pipeline will require a low spot to allow drainage of the tailings line in the event of a plant shut down or to collect tailings in one location in the event of a leak in the tailings line.

At the lake edge, the tailings will enter a de-aeration and mixing tank where air in the tailings can escape and the tailings will mix with lake water. This de-aeration and mixing limits the likelihood of floating of fine particles once the tailings are deposited at depth. The tank will be located 3 m below the low water level in the lake and extend to 3 m above the high water mark. The section of pipe below the lake surface will be ballasted with concrete collars.

WASTE ROCK DEPOSITION

Waste rock produced at the Brucejack Lake mine site will be deposited in Brucejack Lake and in the underground mine. Waste rock generated on the north and south sides of the lake will be deposited in the northwest and southwest corners of the lake, respectively (Figure 18.16). Causeways will be constructed with waste rock so that trucks can dump waste at greater depths and ensure a water cover is maintained over the waste rock. Alternatively, dozers may be required to push the waste rock into the lake. Depending on the acid generating potential of the waste rock, it may be necessary to construct the top few metres of the causeways with non-acid generating rock.

18.3.3 LEACH PLANT AREA WATER MANAGEMENT

GENERAL

The proposed leach plant TSF will be located in the catchment of an unnamed lake, south of Wildfire Creek. The TSF will lie southeast of the lake on the east side of an unnamed creek (Figure 18.17).

The footprint of the tailings impoundment will be about 0.19 km^2, which has an additional 0.28 km^2 of natural watershed area reporting to the facility from the east. The headwaters of the unnamed lake reach a maximum elevation of 1376 masl, and the TSF catchment maximum elevation is 741 masl. Downstream of the TSF, the unnamed creek discharges to the unnamed lake, which in turn reports to Wildfire Creek and ultimately Bell Irving River (Figure 18.17).

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 restrict the amount of total suspended solids to 15 mg/L (Minister of Justice, 2011); in some instances, flocculation may be required to meet discharge standards. The quality of water in streams affected by the project, and of all discharges, will be monitored on a regular basis. TSF water and fresh water from the unnamed lake will be used in process.

Figure 18.16 Brucejack Lake Waste Disposal Plan

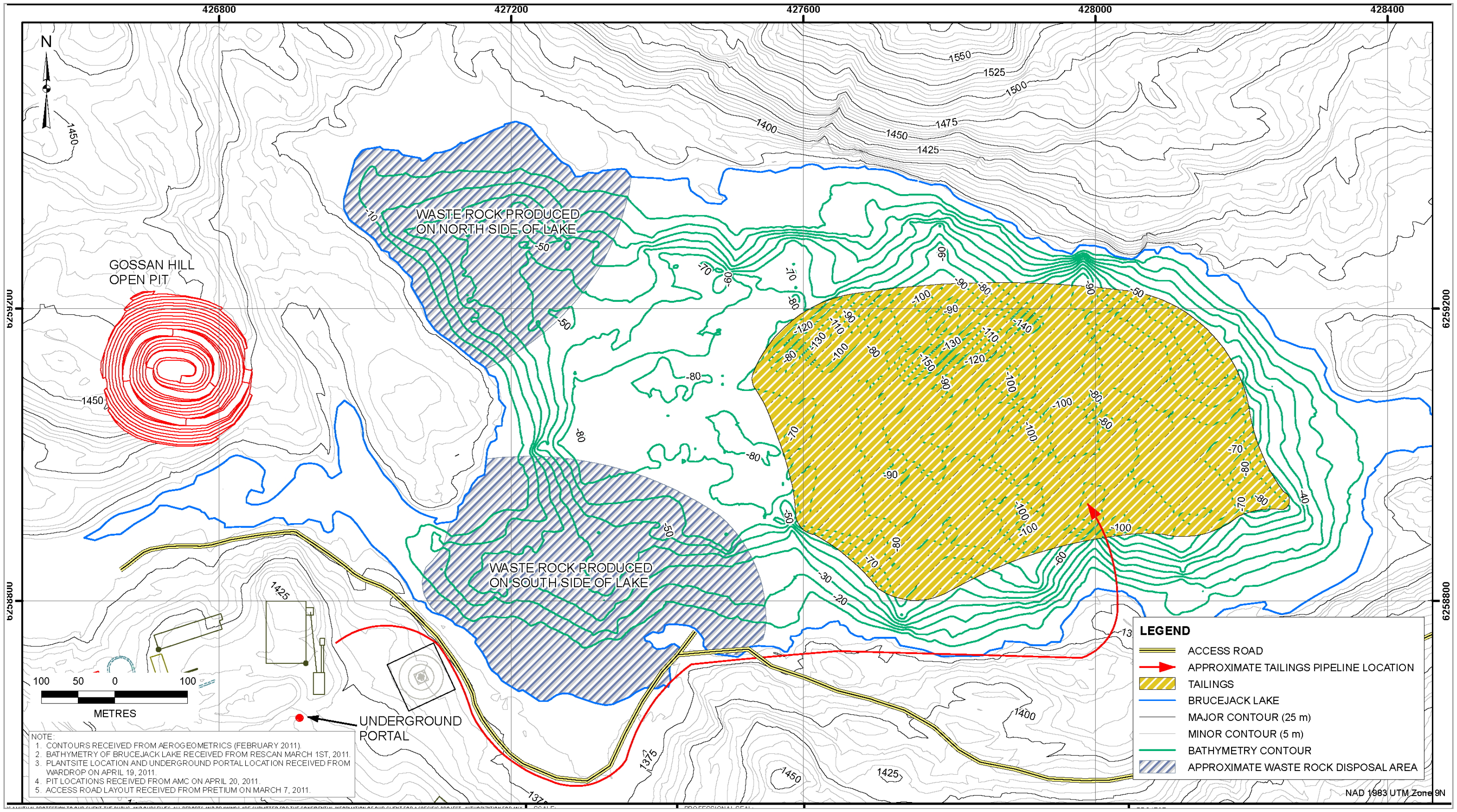
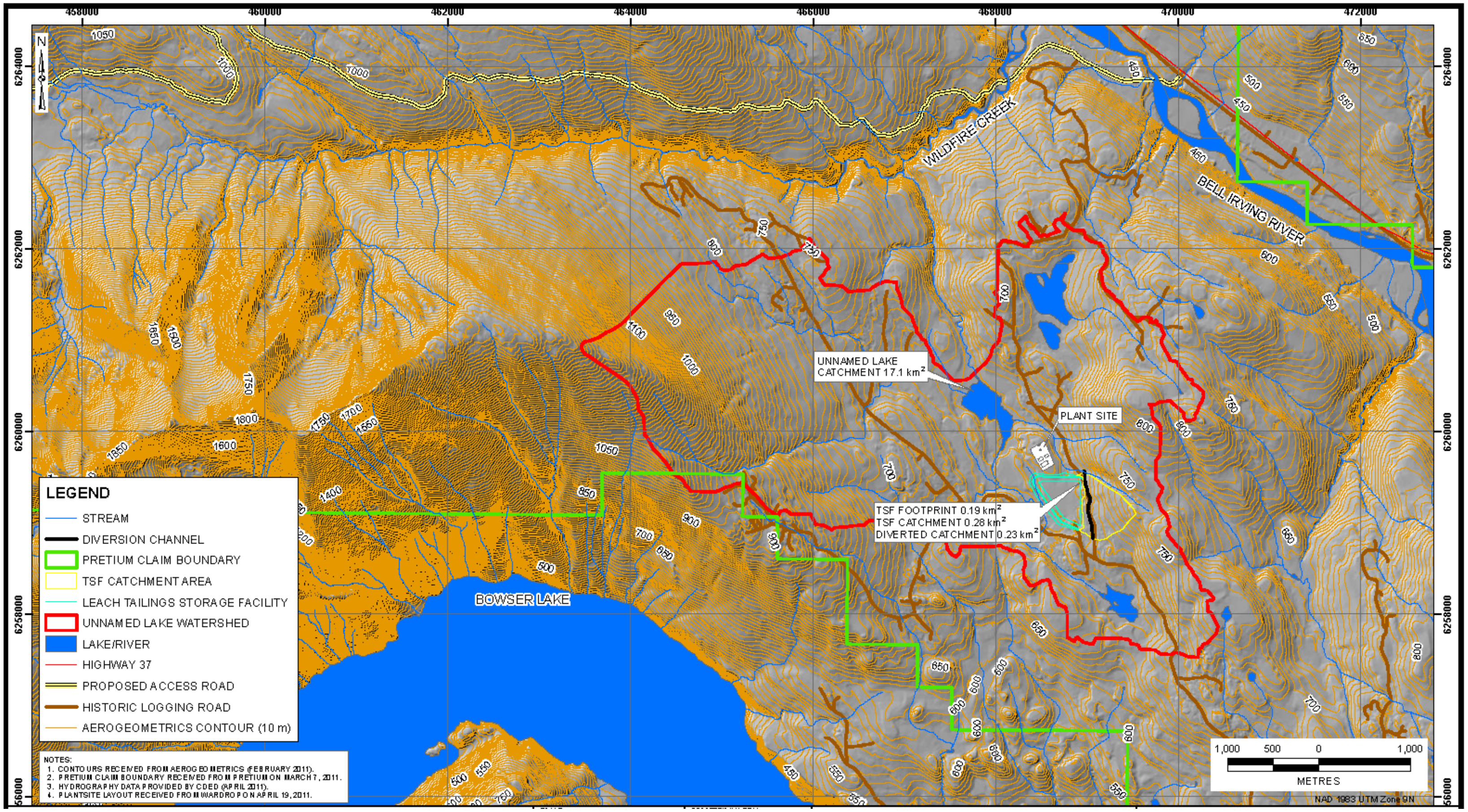


Figure 18.17 Proposed Leach Plant TSF Catchment Areas



DIVERSION CHANNEL

A diversion channel will be constructed upslope of the TSF to divert fresh (or non-contact) water around the impoundment for the duration of the LOM (Figure 18.17). The objective of the channel is to limit inflows, as the TSF is expected to operate with a surplus of water for most years. An area of approximately 0.23 km^2 will be diverted to the north into another unnamed creek. Assuming a diversion efficiency of 80%, the total area reporting to the TSF is estimated to be approximately 0.09 km^2. Approximately 760 m length of channel is proposed and is designed to pass peak flows from a 200-year flood event.

SEEPAGE

The TSF will be double-lined with a high-density polyethylene (HDPE) liner; therefore, seepage through the TSF foundation and dams is not expected. Wells will be located downstream of the dam for groundwater monitoring and will double as pump-back wells, if required.

PROCESS WATER REQUIREMENTS

Water requirements (300 m^3/d) for the leach process plant will be met from two primary water sources:

- reclaim from the TSF pond (232 m^3/d maximum monthly average)
- fresh water supply from the unnamed lake (68 m^3/d) that will be piped to the process plant.

Pond water will be reclaimed from the TSF via a decant tower. The riser will be extended as the level of tailings in the impoundment rises. TSF reclaim rates are expected to be relatively constant throughout the year (9.0 m^3/h or 216 m^3/d annual average).

PRELIMINARY WATER BALANCE

A preliminary water balance model for the leach process plant and TSF 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 3.4
- tailings production of 300 t/d at 50% solids by weight (12.5 m^3/h of slurry water)
- a maximum annual average rate of 9.3 m^3/h and a minimum fresh water requirement of 3 m^3/h

- an average annual precipitation of 813 mm and potential lake evaporation and sublimation losses of 437 mm
- runoff from undisturbed ground (and active tailings beach) is equal to available water minus 85% of potential evaporation (PE). Runoff from inactive beach areas is estimated at available water minus 45% of PE, while runoff from pond areas is equal to available water minus PE.
- process water not supplied from the TSF will be piped from the unnamed lake.

Because average annual precipitation is greater than potential evaporation, the TSF is expected to operate with a net annual surplus of water for most years during the May to October period. Excess water will be pumped to a water treatment plant prior to being discharged to the unnamed lake.

Based on average precipitation conditions, the supernatant pond is estimated to have an average annual surplus volume of 0.11 Mm^3 (12 m^3/h) over the LOM. Assuming that the discharge would be compressed into a six-month (or less) period, the average discharge rate of the pumps will be about 24 m^3/h. Surplus volumes will vary due to natural variations in annual precipitation.

18.3.4 LEACH PLANT TAILINGS STORAGE FACILITY DESIGN

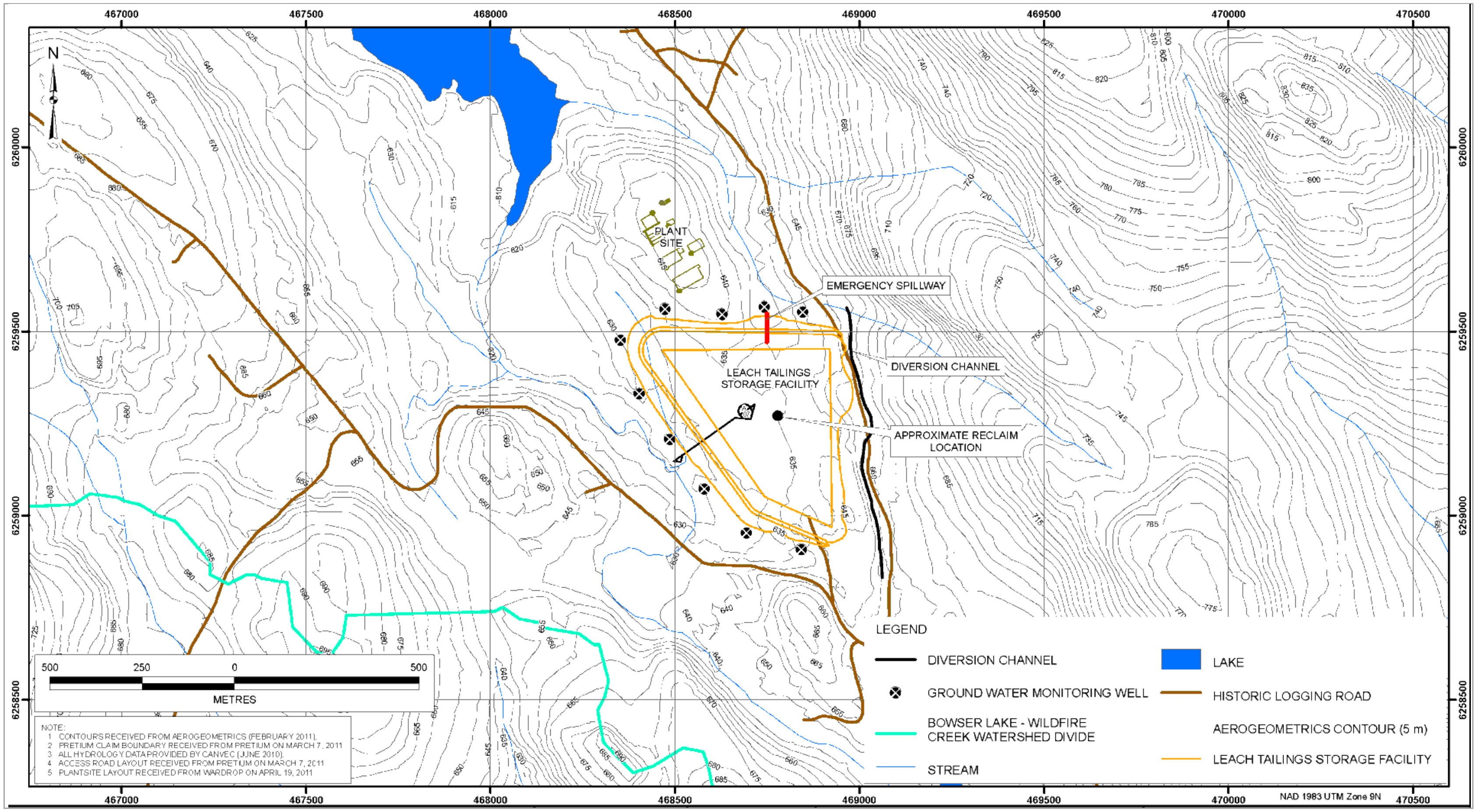
SUMMARY

The leach plant tailings storage facility (TSF) is designed to store 2 Mt of tailings, based on a throughput of 300 t/d. Tailings from the leach plant facility will be deposited in a fully double-lined side hill facility located approximately 300 m south of the leach plant. The facility consists of excavation in the impoundment and dams on three sides to provide the required storage capacity. The TSF is located approximately 4.5 km south of the mouth of Wildfire Creek (Figure 18.6). The ultimate crest elevation of the dam is 646 masl with an ultimate maximum height of approximately 24 m above the downstream toe of the dam.

Tailings will be spigotted from the dam crests. During operations a pond will be created to allow water to be reclaimed from the TSF to the leach facility using a decant structure. At the end of operations, the tailings impoundment will be approximately 550 m long and vary in width between 550 m and 300 m. The facility will be closed by covering and vegetating the tailings impoundment footprint.

The location for the leach tailings storage facility was selected over other sites south of Wildfire Creek, and in the headwaters of Wildfire Creek, because of its anticipated favourable foundation conditions, lack of potential geohazard impacts, and ease of water management, groundwater monitoring, and closure.

Figure 18.18 Leach Plant Area TSF Plan View



DESIGN CRITERIA

Table 18.14 summarizes the design criteria applicable to the leach plant TSF. The design criteria were established in conjunction with Pretivm.

Table 18.14 Preliminary Criteria for Leach TSF Design

Criteria	Description/Comments
Capacity and Throughput	
Total Mineralized Material Mined	10 Mt
Mill Throughput at Leach Plant	300 t/d
Starter Storage Capacity Required at Leach TSF	2 years of production
Storage Capacity Required at Leach TSF	2 Mt
Hydrology	
Operating Pond	1 m over impoundment footprint
Flood Storage Capacity	24 hour probable maximum precipitation (PMP) + 200 year snowmelt
Emergency Spillway Capacity and Freeboard	Probable maximum flood associated with the 24-hour PMP plus 0.5 m freeboard
Diversion Channel Capacity	200-year peak instantaneous flow

TAILINGS STORAGE FACILITY DESIGN BASIS

At the time of preparation of the design, no geotechnical tailings testing results were available. Tailings are expected to be primarily fine-grained (approximately 80% finer than 20 µm to 30 µm). The dry density of the tailings is assumed to be 1.3 t/m^3. Tailings are expected to be acid generating, and have high sulphide content. As previously indicated, water in the TSF will be treated prior to release.

No site-specific investigations have been completed. The foundation conditions and terrain were assessed based on aerial photo interpretation and regional geology maps. The bedrock geology in the area of the proposed leach TSF comprises sandstone and siltstone with rare conglomerate of the Jurassic age Bowser Lake group (GSC 2009). The terrain in the area of the proposed TSF is generally irregular and rock controlled. Surficial till is thin, likely less than 2 m and discontinuous. Impacts from geohazards such as avalanches or rockfall are not anticipated.

Topography used to estimate material quantities is a digital elevation model (DEM) provided by Aero Geometrics Ltd., which was developed from aerial photos taken in August of 2010.

Tailings Storage Facility Design

Facility Design: Impoundment, Dam, Diversions, and Spillways

The TSF will be a side-hill containment facility that will establish storage capacity through dams on three sides, and the use of the hill as the fourth side (Figure 18.18). The dam cross-section is shown in Figure 18.19. Some grading in the impoundment will be completed to increase storage capacity.

The facility is fully lined with two layers of high-density polyethylene (HDPE) liner with a drainage layer in between. Dam faces will be sloped at 3H:1V on the upstream face to facilitate liner placement. The downstream face will be sloped at 2H:1V. The impoundment will be graded so that any seepage through the first layer will collect in a sump between the two layers. Water collected in this sump will be pumped through a pipe in the face of the dam and back into the impoundment.

A 1-m thick layer of low permeability fill under the second HDPE liner layer will act as a bedding layer for the liner. In the dam, two filter layers will be present below the low permeability fill. Rockfill will comprise the main body of the dam (Figure 18.19).

In the event of precipitation greater than the stored flood, an emergency spillway will pass the flood associated with the 24-hour probable maximum precipitation event. The emergency spillway will be located on the north side of the impoundment (Figure 18.18), and will tie into a natural drainage immediately east of the plant site and drain north into an unnamed lake south of Wildfire Creek.

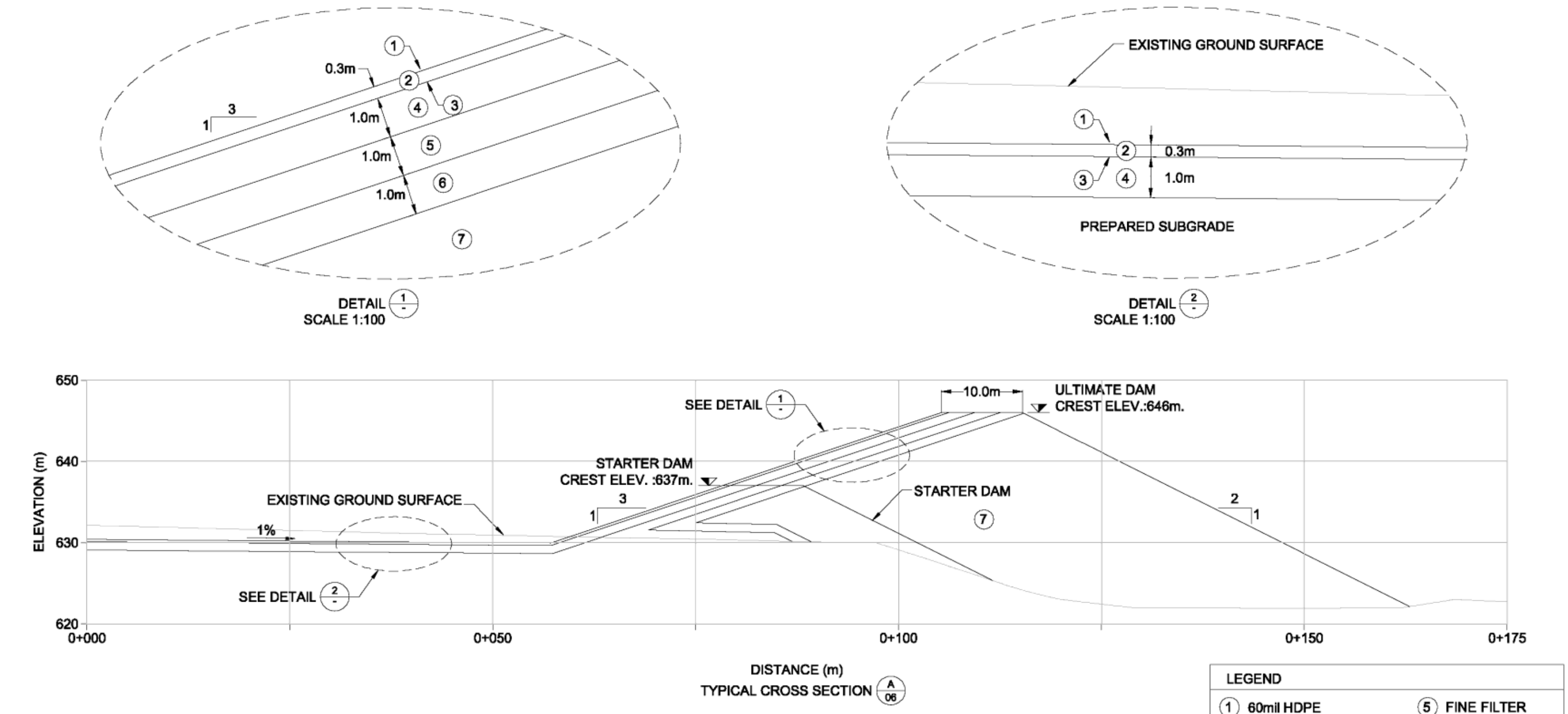
A diversion channel to pass the 200-year peak instantaneous flow will be constructed immediately east of the TSF on the slope above the TSF. If rainfall greater than the design event occurs, it is assumed that the diversion system is overwhelmed and the entire catchment reports to the TSF.

Groundwater monitoring wells will be installed downstream of the facility. A seepage collection and pump-back system, in addition to the impoundment seepage collection, can be installed if required.

Tailings Deposition

Tailings will be deposited as a slurry with approximately 50% solids (by weight). Tailings will be spigotted from the three dam crests, creating a pond near the centre of the facility. Reclaim will be achieved via a rockfill decant with submersible pumps.

Figure 18.19 Leach TSF Dam Cross-Section



NOTE:
1. IN THE IMPOUNDMENT, IN-SITU LOW PERMEABILITY FILLS, IF AVAILABLE, CAN BE COMPACTED TO PREPARE A SMOOTH BEDDING LAYER FOR THE LINER. OTHERWISE, IMPORTED LOW PERMEABILITY FILL CAN BE PLACED AND COMPACTED TO PREPARE A BEDDING LAYER FOR THE LINER.
2. IN THE DAM FOOTPRINT ORGANICS AND SOILS WITH SPT N VALUES LESS THAN 20 WILL BE REMOVED PRIOR TO FILL PLACEMENT.

Tailings Facility Closure

In the final years of operations, tailings will be deposited in the centre of the facility to level the tailings surface. At the end of operations, the pond will be pumped dry and the water treated and released. A geosynthetic clay liner, a layer of granular material, and a layer of growth medium to support vegetation will then cover the TSF. The cover will be graded such that runoff will be directed to two closure spillways, one located on the north side and the other located on the south side of the impoundment. The operational diversion system will be maintained for closure. Ongoing monitoring of groundwater quality surrounding the facility will continue during the closure period.

MATERIAL SOURCES

Three types of locally sourced material are required for dam construction: rockfill, low permeability fill and filter/drainage layers. Rockfill, filter and drainage materials will be sourced from grading within the impoundment, plant site grading and two quarries located approximately 600 m south and 700 m west of the facility. Low permeability fill will be sourced from grading within the impoundment, plant-site grading and the pre-development stripping of the quarries. Additional low permeability fill material, if required, can be sourced from borrow sources approximately 1 km northeast and 2.5 km northwest of the TSF.

Tailings Facility Construction

Construction of the starter dam can likely be completed over the course of two summer seasons. Acceleration of the construction schedule may be possible and will be evaluated at the next stage of design. Pre-development of quarries and grading of the site would be completed in the first summer. Once snow is clear the following spring, placement of low permeability fill bedding and dam rockfill would be begin. Filters would be placed on the dam face once it is constructed. Finally, the liner would be placed in the impoundment on the dam faces.

Subsequent raises will likely be completed in scheduled phases, where several years of storage are constructed at once.

18.4 UNDERGROUND GEOTECHNICAL AND HYDROGEOLOGY EVALUATIONS

Geotechnical evaluations at the PA level for the proposed Brucejack underground developments consisted of:

- a review of existing engineering studies and available data
- development of a preliminary geotechnical model

- provision of recommended stope dimensions
- an estimate of ground support requirements.

In addition, a conceptual hydrogeologic model was developed for the project area, and preliminary estimates of groundwater inflow rates to the mine workings were made.

The majority of data used for this study was obtained from geotechnical drill holes SU-77, SU-82 and SU-88, which BGC drilled, tested and logged in 2010. Historic reports and data from exploration drilling within the study area have also been used to augment information from the geotechnical holes.

At this stage, the Brucejack property is assumed to represent a single structural domain. There is insufficient geotechnical information available to determine statistically significant geomechanical differences between the various lithologies within the Brucejack property, or between the rock masses comprising the footwall and hanging wall. In general, the rock quality of the various lithologies present on the property can be classified as "good to very good", with "strong" intact strengths and "high" "Rock Quality Designation"[1] (RQD) values.

The industry-standard empirical Stability Graph method (Hutchinson and Diederichs, 1996, and Nickson et al., 1992, after Potvin, 1988) was used to estimate acceptable mining dimensions for the stopes. The recommended dimensions, expressed as a "hydraulic radius" (HR) in Table 18.15, assume the stope walls and back will be unsupported, and that major geologic structures are not present in the stope walls or back. Additional stability analyses and support requirement evaluations should be carried out for hanging walls where the hydraulic radius exceeds those recommended in Table 18.15.

Table 18.15 Summary of Stability Graph Method Analysis for West Zone and VOK Zone

Zone	Veins	Hanging wall (HW) or Back (BK)	Maximum HR (m) Unsupported
West	R1-R7	HW	15
	R1-R7	BK	9
	R8	HW	12
	R8	BK	9
VOK	1,3,4	HW	13
	1,3,4	BK	8
	2	HW	11
	2	BK	8

[1] Deere, D.U. and Deere, D.W. 1988. The rock quality designation (RQD) index in practice. *Rock classification systems for engineering purposes,* (ed. L. Kirkaldie), ASTM Special Publication 984, 91-101. Philadelphia: Am. Soc. Test Mat.

Support guidelines for the main access ramp, main level access, and mineralized material access excavations have been estimated using the Q-Support Chart by Grimstad and Barton (1993) and Empirical Rules of Thumb (US Army Corps of Engineers, 1980). The recommended standard support is summarized in Table 18.16. A uniform bolting pattern for the roof and the sidewalls has been provided for operational simplicity and efficiency. The ground support recommendations provided are for guidance and cost estimating purposes only, and do not account for geological complexities or stress-induced behaviour.

As part of the geotechnical drilling program, BGC completed eight packer tests in three drill holes to estimate the hydraulic conductivity (K) of the intersected rock mass. Results of packer testing predict K to range from 2×10^{-8} m/sec to 4×10^{6} m/sec. Tests were completed at depths ranging from approximately 68 to 323 m below ground surface. Based on available testing data, there is no discernable relationship between bedrock hydraulic conductivity and depth.

Table 18.16 Ground Support Estimates for West Zone and VOK Zone

Drift Type (w x h (m))	ESR1	Roof	Sidewall (to 1.5 m above Floor)
Main Access Ramp 5 x 5.5	1.6	2.4 m #7 Rebar 1.2 x 1.2 m 100%	2.4 m #7 Rebar 1.2 x 1.2 m 100%
		Mesh 50%	Mesh 50%
		Shotcrete Nil	Shotcrete Nil
		5 m Cable (intersections) 2.5 m^2/bolt	-
Main Level Access 5 x 5	2	2.4 m #7 Rebar 1.2 x 1.2 m 100%	2.4 m #7 Rebar 1.2 x 1.2 m 100%
		Mesh 25%	Mesh 25%
		Shotcrete Nil	Shotcrete Nil
		5 m Cable (intersections) 2.5 m^2/bolt	-
Mineralized Material Access 4 x 4.5	3	2.4 m #7 Rebar 1.2 x 1.2 m 100%	2.4 m #7 Rebar 1.2 x 1.2 m 100%
		Mesh Nil	Mesh Nil
		Shotcrete 50 mm Thick 100%	Shotcrete 50 mm Thick 100%

Previous estimates of groundwater inflows range between 45 and 164 m^3/h, based on water balance calculations and pump design (CESL, 1990; Corona, 1990) for the historical underground workings, which are approximately 2,300 m long and have an average elevation of 1261 masl. This corresponds to a range of 0.02 m^3/h to 0.06 m^3/h per metre of tunnel. Calibration of a simplified analytical model of flow to an underground tunnel (Freeze and Cherry, 1976) guided by hydraulic conductivity estimates from the 2010 site investigation predicts a range of inflows of 0.01 m^3/h to 0.13 m^3/h per metre of tunnel. Extending the analysis to a planned elevation of 1000 masl for the proposed workings of the West Zone, inflows are estimated at

0.06 m^3/h (0.02 L/sec) per metre of tunnel and predicted to range between 0.02 and 0.27 m^3/h (0.01 and 0.07 L/sec) per metre of tunnel. Additional investigation, analysis and detailed information on the staging of the underground developments will be necessary to confirm these estimates. Design of an appropriate dewatering system to mitigate and manage tunnel inflows will require more detailed analyses.

It will be necessary to dewater the existing dormant underground workings before mining operations begin. Groundwater inflows to the new workings during development and operations will also have to be managed. A pumping system will be required to extract water from the flooded workings. This system will also pump out inflows from new workings during mining operations. To facilitate design of the dewatering system, three pilot wells approximately two hundred metres deep and six inches in diameter will be drilled into the historical workings. A pilot pumping test program of these wells will be conducted to confirm whether the workings can be intercepted and dewatered without significant flow blockage (i.e. from potential cave-ins) and that optimal pumping rates can be achieved.

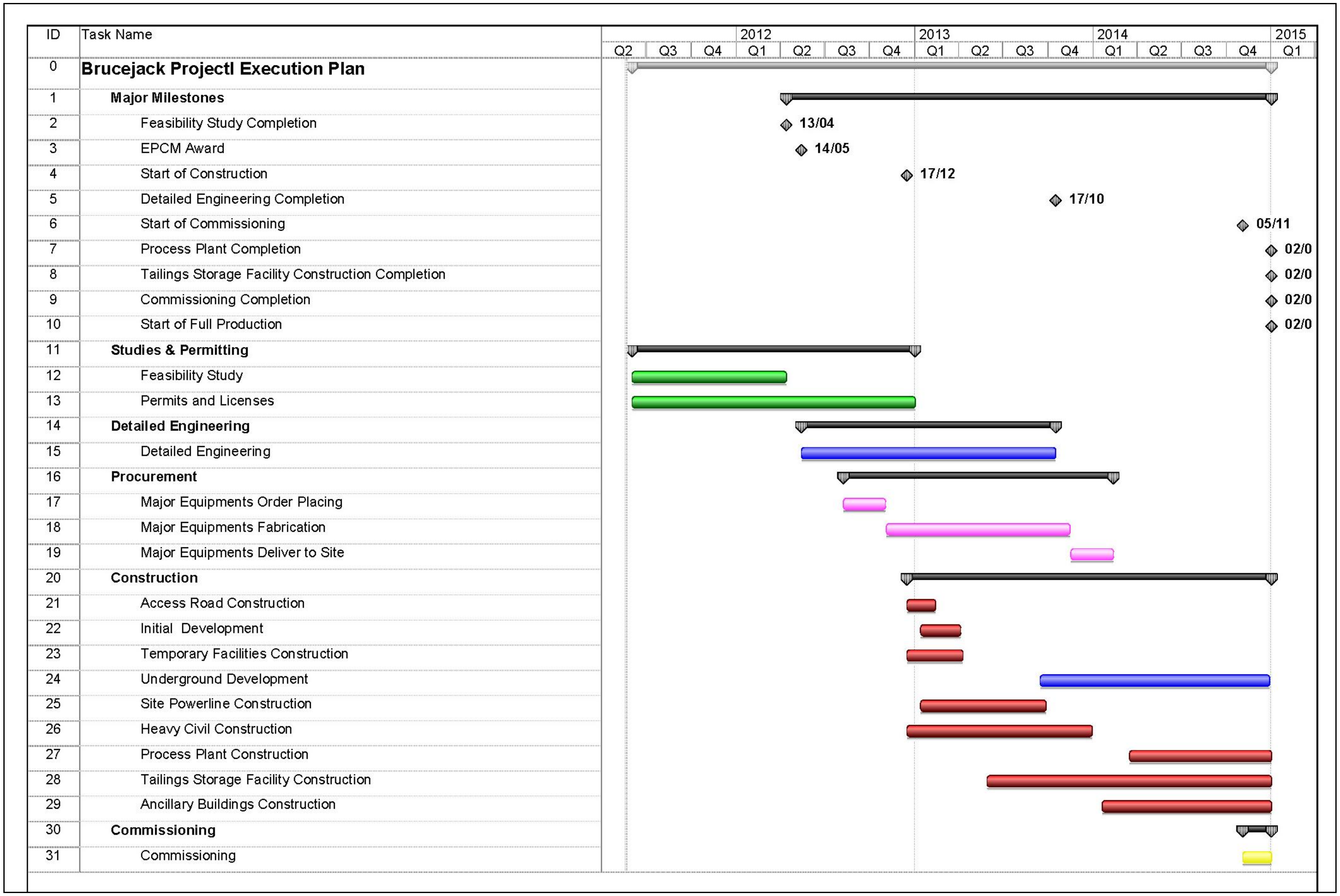
Following this assessment, the diameter of existing pilot wells will be increased, and the wells will be equipped with larger pumps to expedite dewatering of the historic workings. It is anticipated that approximately four weeks will be required for drilling and test pump installation. It is also expected that a minimum of three wells (8 to 10-inch diameter) pumping at the maximum anticipated yield of 175 to 350 US gpm (11 to 22 L/sec, Driscoll, 1986) will be required to dewater the underground workings over three months.

18.5 PROJECT EXECUTION PLAN

The preliminary project execution schedule (the Schedule) was developed to provide a high-level overview of all activities required to complete the project. The Schedule is summarized in Figure 18.11.

Upon receipt of construction and operating permits, the project will take approximately two years to complete, from the time board approval is received, through construction to introduction of first mineralized material in the mill and commissioning.

Figure 18.20 Brucejack High Level Execution Plan



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
- underground development
- commissioning.

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

18.6 MARKETS AND CONTRACTS

The final products to be produced by the Brucejack Project are a gold and silver doré. The gold and silver doré will likely be transported to a North American-based precious metals refinery and sold to precious 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 Brucejack property is situated within the Sulphurets District in the Iskut River region. The property is located in the Boundary Range of the Coast Mountain physiographic belt along the western margin of the Intermontane tectonic belt.

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

Tree line is at approximately 1200 masl. The West Zone deposit on which the Brucejack project is centred is located west of Brucejack Lake at 1400 masl.

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 Brucejack project is located in a remote area for which little baseline environmental data are publically available. Pretivm has engaged Rescan, a Vancouver-based consulting firm with extensive BC mining-related environmental assessment experience, to undertake the baseline studies required for an environmental assessment of the project. Baseline studies for the Brucejack project have been initiated at this writing.

TERRAIN, SOILS, AND GEOLOGY

The Brucejack project is located in a rugged area with elevations ranging from about 700 m at the planned leach plant and tailings storage facility to 1,400 m at the deposit. Surrounding peaks are up to 2,200 m in elevation. Glaciers and ice fields surround the mineral deposits to the north, south, and east.

The West and Shore Zone deposits are high grade, porphyry-style gold deposits. The gold mineralization at these deposits 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 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 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 values in the 2.5 to 3.0 range. In water with near-neutral pH, evidence of precipitation, such as white aluminum oxyhydroxide and iron staining, are likely to be found from processes which have been occurring naturally over a geological time scale. Baseline ABA and metals analyses for various rock types will be undertaken to evaluate potential ARD concerns. Pending more detailed assessment, it is difficult to predict the ratio of net acid neutralizing to net acid generating rock. The net acid generating rock will also be evaluated for kinetic rate of reaction, which will give an indication of the type of management strategy required.

Climate, Air Quality, and Noise

The climate of the region is relatively extreme and daily weather patterns in the Iskut region are unpredictable. Prolonged clear sunny days can prevail during the summers. Precipitation in the region is about 1,600 mm to 2,000 mm annually. The majority of precipitation is received in the fall and winter, from September through to February. Annually, Stewart receives 70% of its yearly precipitation during this time. October tends to have the highest or second highest precipitation levels for the year. Stewart regularly receives 30% of its precipitation as snow that falls from November to March. In October, when Stewart typically has its heaviest precipitation, 97% of it falls as rain. Late spring or early summer months typically receive the least amount of rainfall on an annual basis. Snow pack ranges from 1 m to 2 m, but high winds can create snowdrifts up to 10 m deep. Silver Standard established a full meteorological station to collect site-specific weather data near the Brucejack Lake camp in mid-October 2009. The station measures wind speed and direction, air temperature and pressure, rainfall, snowfall, relative humidity, solar radiation, net radiation and snow depth.

Assumed average monthly climate data for the Brucejack Lake mine site and Bell Irving TSF are shown in Tables 18.17 and 18.18 respectively (BGC, 2011). Average precipitation data reported at the Unuk River Eskay Creek (#1078L3D) Meteorological Service of Canada (MSC) climate station were used to characterize both the Brucejack mine site and the Bell Irving TSF site. Data from this station are available for the period of September 1989 to June 2009 (BGC, 2011). The station

(56° 39' N, 130° 27' W) is located at an elevation of 887 m approximately 26 km northwest of Brucejack Lake and 60 km west-northwest of the Bell Irving TSF.

Table 18.17 Average Monthly Climate Data for the Brucejack Lake Mine Site (BGC, 2011)

Month	Average Temperature (°C)	Average Precipitation (mm)	Average Evaporation/ Sublimation (mm)
January	-11.3	249	5
February	-9.1	214	5
March	-7.2	181	7
April	-2.6	97	12
May	1.1	88	24
June	5.1	67	39
July	7.3	83	43
August	7.3	139	37
September	2.7	207	25
October	-2.4	247	7
November	-7.9	215	6
December	-9.8	248	5
Average/Total	-2.2	2,034	215

Table 18.18 Average Monthly Climate Data for the Bell Irving TSF (BGC, 2011)

Month	Average Temperature (°C)	Average Precipitation (mm)	Average Evaporation/ Sublimation (mm)
January	-8.8	100	5
February	-6.4	86	4
March	-0.3	72	6
April	3.8	39	12
May	8.2	35	40
June	11.9	27	97
July	14.1	33	106
August	13.4	55	89
September	9.3	83	52
October	3.9	99	15
November	-3.7	86	6
December	-8.8	99	5
Average/Total	3.1	814	437

Precipitation at the mine site is currently assumed to be similar to that observed at the Unuk River Eskay Creek station given their close proximity (19 km) and similar basin physiography. However, the Bell Irving TSF is approximately 41 km east of the mine site and located behind a range of glaciated mountains with peak elevations of up to 2,300 m. This range is expected to have a rain shadow effect with reduced precipitation in its lee. Therefore, average annual precipitation at the Bell Irving TSF is expected to be about 60% (813 mm) of that recorded at Unuk River Eskay Creek. This adjustment factor is based on comparison of the Unuk River Eskay Creek data to the Bob Quinn AGS climate station records (#1200R0J) and to output results from the ClimateBC climate data generation model (Wang, 2006). The Bob Quinn AGS station is located approximately 70 km northwest of the Bell Irving TSF, in a narrow valley at a similar elevation (610 m) as the TSF, and has data available for the period of December 1977 to April 1994 (BGC, 2011).

Average monthly temperature data used at the mine site are based on scaling the Unuk River Eskay Creek data to an elevation of 1400 m assuming an adiabatic lapse rate of -0.6°C per 100 m. Temperature data used at the Bell Irving TSR are based on unadjusted Bob Quinn AGS records (BGC, 2011).

Annual evaporation was estimated using the temperature-based method of Zhan and Shelp (2009), which is a modification of the Blaney-Criddle method. The annual evaporation total included sublimation for the winter months; these values are based on BGC's experience elsewhere and represent a minor component of the water balance (BGC, 2011).

WATER RESOURCES

Flow Volumes

The Brucejack Lake catchment, where the adit and mill will be located, drains into Sulphurets Creek, which flows into the Unuk River toward Alaska. The leach plant catchment drains into Wildfire Creek and 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 then flows through Misty Fjords National Monument in Alaska, and finally into Behm Canal on the Pacific coast.

Total precipitation reduces as one moves from west to east across the region (and to the north) which is consistent with runoff depths recorded at nearby Water Survey of Canada (WSC) hydrometric stations (BGC, 2011). Table 18.19 summarizes annual runoff depths for three hydrometric stations in the region and a long-term synthetic streamflow dataset based on streamflow monitoring station records (Knight Piésold, 2011).

Table 18.19 Runoff Depths at Regional Hydrometric Stations, BGC 2011

Station	ID	Area (km^2)	Lat/Long	Direction from Project	Period of Record	Annual Runoff Depth (mm)
Iskut River below Johnson River	08CG001	9,350	56° 44’ 20” N 131° 40’ 25” W	WNW	1959-2005	1,535
Potential Future Brucejack Lake Hydropower Dam	-	10.1	56°28'10.03"N 130°11'8.98"W	-	1980-2009 (synthetic)	1,499
Iskut River at Outlet of Kinaskan Lake	08CG003	1,250	57° 31’ 50” N 130° 10’ 45” W	ESE	1964-1996	433
Driftwood River above Katsberg Creek	08JD006	403	55° 58’ 34” N 126° 40’ 34” W	N	1979-2005	643

Water Quality

Little historical baseline water quality information is available for the Brucejack area. Pretivm has initiated an assessment of water and sediment quality and related aquatic ecology. The sparse water quality data collected to date at Brucejack Lake indicates 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 potentially affected rivers and their tributaries are being assessed as part of the baseline program. Results from one sampling season indicate that fish may not occur in Brucejack Lake. Mitigation measures and any compensation that may be due as a result of project-related fisheries impacts will be discussed and developed in consultation with the appropriate agencies and relevant Aboriginal groups.

ECOSYSTEMS AND VEGETATION

The 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. Pretivm intends to map plant communities and plant species of conservation concern to aid environmental impact assessment.

WETLANDS

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

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 and the fieldwork-intensive Terrestrial Ecosystem Mapping work. Pretivm 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 will be documented, as they are important from both ecosystem and socioeconomic (i.e. hunting) perspectives, and are especially sensitive to development. Aerial surveys following government protocols will be used to assess mountain goat populations to aid in the development of appropriate mitigation techniques. Breeding birds and raptors will be documented in the project areas, and will be given special attention due to statutory protection and conservation concerns.

TRADITIONAL KNOWLEDGE AND TRADITIONAL LAND USE

The Brucejack Project site is located on Crown land in an area historically used by several First Nations groups. 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 Brucejack Project area is included in the LRMP, 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 outcomes 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 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 an exploration road from Bowser Lake 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 traplines exist in the project area, as do commercial recreational and fishing guide territories. 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, in the vicinity of the leach plant site. Further timber harvesting in the project area is possible subject to a viable market for the timber.

VISUAL AND AESTHETIC RESOURCES

The Brucejack Project is located in a relatively remote and undisturbed area characterized by rugged mountains, glaciers, forests, and rivers. The nearest road is Highway 37, about 8 km east of the proposed leach plant and tailings facilities, which are therefore unlikely to be visible from the highway. The controlled-access Eskay Mine road terminates about 20 km north of the proposed adit and mill. The mine will be located in an isolated area that is not visible from either the Eskay Mine road or Highway 37.

18.7.3 SOCIOECONOMIC SETTING

Northwestern BC is a sparsely populated area with a number of small, predominantly Aboriginal communities, and larger centres of Smithers, Terrace and Stewart, which provide services and supplies to much of the region. It is a remote area; communities within the region are generally dispersed and isolated from one another. Transportation and communication options are limited, with the region intersected by Highways 37 (north to south) and 16 (east to west).

The region has suffered from declining population and weakening economic prospects, particularly among the communities near Highway 37. The regional population declined by 5.9% between 2001 and 2006, while the province had a 5.3% population increase over the same period.

The region has a strong dependence on primary resource industries, principally mining and forestry. Mineral exploration activity has grown in recent years, 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 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. Pretivm is working and intends to working 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 on Highway 16 and 37 corridors are compiled from the Northwest BC Mining Projects Socioeconomic 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 Canadian National Railway (CNR) also follows this corridor. Most of the communities along this corridor are discussed in this section. The Highway 16 corridor is recovering from the economic downturn of the 1990s, and has excess capacity with respect to social service infrastructure. The respective communities are incorporated providing a framework and capacity to:

- plan for, finance, and deliver services that might be required
- meet incremental growth from new mine developments.

HIGHWAY 37 CORRIDOR

Highway 37 connects with Highway 16 at Kitwanga and runs northwards to the Yukon border. At Meziadin Highway 37A branches off from 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.

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, and monitoring the composition of released water and treating it to remove or control contaminants as required to meet discharge standards. Diversion channels will be constructed to direct runoff away from disturbed areas.

Water management is described in detail in Sections 18.3.

Water Supply

Refer to Section 18.3.3.

Potable water will likely be sourced from wells near the adit and the leach plant.

Internal Recycling Strategies

Process water will be recycled where feasible to reduce the volumes of water released to the environment. Tailings supernatant water will be recovered from the leach tailings and returned to the plant.

Storm Water Management

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

Water Treatment

It is anticipated that water discharged from the leach tailings facility will require lime treatment to adjust pH and to reduce metals concentrations. Tailings will be placed at the bottom of Brucejack Lake to ensure that suspended solids concentrations do not exceed the 15 mg/L TSS discharge criterion on surface.

Construction Water Management

Pretivm 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

Refer to section 18.3 for more information.

The mill tailings will be contained in a subaqueous environment at the bottom of Brucejack Lake in perpetuity.

The leach tailings will be contained in a lined surface facility. Pretivm will develop and implement a leach tailings management plan. The goals of this management plan will be to:

- manage the structure in a safe and environmentally responsible manner
- manage the discharge to ensure that all effluent meets or exceeds the permitted water quality levels and guidelines
- provide a framework for continual improvement in the environmental safety and operational performance of the structure
- define environmental and performance monitoring and reporting.

Tests will be undertaken to characterize the tailings and supernatant water to allow estimation of the rate of oxidation and resulting water quality. This information will guide planning for tailings water management.

At closure, the leach tailings facility will be 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.

Waste Rock and Overburden Management

Refer to Section 18.3 for more information.

The Brucejack Project will potentially generate waste rock over the anticipated LOM. Some of the waste rock will be disposed underground with the reminder deposited subaqueously in Brucejack Lake to reduce ML/ARD.

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. Pretivm 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

Pretivm 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 Pretivm 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

Since most of the mining will be underground and most of the tailings will be stored subaqueously, air emissions will not represent a significant component of contaminant dispersion for the project. Baseline studies, utilizing on-site meteorological stations and wind monitoring stations, will collect atmospheric data in the Brucejack Project area to allow air dispersion modelling. 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 need to adjust the current approach.

Pretivm 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 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.

18.7.7 Design Guidance

Pretivm will develop and implement a comprehensive Environmental Management System (EMS) for the construction, operation, and closure phases of the Brucejack Project. The EMS will comprise a series of written plans outlining the scope of environmental management to ensure compliance with both regulatory requirements and Pretivm environmental policy.

Environmental management and mitigation measures will be provided for each of the following areas:

- air emissions and fugitive dust
- water management
- tailings and waste rock
- diesel and tailings pipelines
- concentrate load-out
- metal leaching/ARD containment
- materials management
- erosion control and sediment
- spill contingency and emergency response
- fish and fish habitat
- wildlife management
- waste management
- archaeological and heritage site protection.

18.7.8 Social and Community Management Systems

Pretivm will develop and implement broad Social and Community Management Systems (SCMS) for the construction, operation, and closure phases of the Brucejack Project. The SCMS will comprise an ongoing engagement plan and impact benefit agreements (IBAs) to be developed through a series of written agreements and relationship-building initiatives with First Nations. Monitoring and oversight of the SCMS will require a team of staff responsible for coordinating community development initiatives, training, communications and commitment tracking, and fund management.

Social, community management, and relationship-building measures will be provided for each of the following areas:

- impact benefit agreements
- community engagement meetings
- training
- participation in community events
- reporting and feedback mechanisms.

Two employees, on a rotational basis, will be required for environmental monitoring including:

- federal MMER monitoring requirements
- permit and license compliance monitoring
- environmental effects monitoring
- reclamation research and monitoring.

Pretivm environmental staff, supported by specialist consultants, will also research and advise the Mine Manager on alternative mitigation strategies as part of the mine's process of continual improvement. Outside laboratories will be required for some analyses, while more routine analyses, such as conventional water sample analysis, will be done in-house. Resources will be required for ongoing equipment upgrades and replacement, specialized equipment procurement, helicopter support, and mitigation and reclamation research.

18.7.9 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 follows: “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.” Pretivm 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

Pretivm respects the traditional knowledge of the Aboriginal peoples who have historically occupied or used the project area. Pretivm 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. Pretivm 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. Pretivm 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 Brucejack Project. Pretivm 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

Pretivm 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

Pretivm 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 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. Pretivm 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 volcanic activity (within the last 10,000 years), along with frequent landslides, debris flows, and snow avalanches, continue to modify the landscape. Pretivm'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

Pretivm is committed to making every reasonable effort toward maintaining biodiversity in the project area. Biodiversity is defined by the BC Ministry of Forests, Lands and Natural Resource Operations 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

Pretivm will undertake a systematic mapping of the project area using both Predictive Ecosystem Mapping (PEM) and Terrestrial Ecosystem Mapping (TEM) methods. The PEM method will be used over the whole of the project area, while 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, Pretivm will map plant communities and plant species of conservation concern to aid environmental impact assessment.

Environmental Standards

Pretivm will design, construct, operate, and decommission the 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 as follows:

- 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
- Transportation of Dangerous Goods Act
- Workplace Hazardous Materials Information System (WHMIS) 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 processes by which compliance will be met and demonstrated, and will include ongoing monitoring and reporting to relevant parties.

DESIGN FOR SOCIAL AND COMMUNITY REQUIREMENTS

Pretivm will strive to establish collaborative and cooperative relationships with relevant Treaty and First Nations people, other communities, and interested stakeholders. Pretivm recognizes that its social licence to operate is dependent on being a good corporate citizen and neighbour to all groups with regional interests.

Pretivm is committed to a process that ensures that communities benefit from employment, training, and contracting opportunities; potential negative impacts are mitigated; and any commitments and benefit agreements are respected. Pretivm 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 ongoing commitments to communities and stakeholders.

18.7.10 CONSULTATION ACTIVITIES

Pretivm will initiate a consultation program relevant and useful to each consultation group. The proposed Brucejack Project consultation program will include government agencies, First Nations and Treaty Nations participation in the BCEAO technical working group meetings, leadership meetings, community meetings, information distribution, focus groups and workshops.

Consultation activities will reflect the BCEAO and CEAA consultation requirements, as well as Pretivm's goals for meaningful and sustainable relationships with the leaders and community members affected by and involved in the Brucejack Project.

Community engagement and consultation are fundamental to the success of the proposed Brucejack project and will take place during the project's planning and regulatory review, construction, and operations phases. Prior to beginning the BCEAA process, Pretivm will initiate project and company introductions with the potentially affected Treaty and First Nation groups. Subsequent consultation activities in the form of information sharing will occur during the planning and regulatory review, construction, and operations phases. These consultations will include BCEAO technical working group meetings (with government agency, Treaty, and First Nations participation), leadership meetings, community meetings, project information distribution, focus groups and workshops, communication tracking, and issue identification and resolution.

CONSULTATION POLICY REQUIREMENTS

The BCEAA and the CEAA contain provisions for consultation with Treaty Nations, First Nations, and the public as a component of the environmental assessment process. Public consultation measures will comply with the *Public Consultation Policy Regulation*, *BC Reg. 373/2002*.

CONSULTATION GROUPS

Treaty and First Nations

Pretivm may be delegated the responsibility of information sharing with potentially affected Treaty and First Nations. If so, the process will be initiated with the potentially affected Treaty and First Nations, as identified by the Government of BC, and will continue.

Government

Pretivm will engage and collaborate with the federal, provincial, Treaty Nations, Regional, and Municipal government agencies as required with respect to topics such as: land and resource management; protected areas official community plans (OCPs); environmental and social baseline studies; and effects assessment mitigation, management, monitoring and reclamation plans.

Public and Stakeholders

Pretivm will consult with the public and relevant stakeholder groups[2], including: land tenure holders; trappers, guides, outfitters, recreation and tourism businesses; economic development organizations; businesses and contractors (e.g. suppliers and service providers); and special interest groups (e.g. environmental, labour, social, health, and recreation groups).

18.7.11 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 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.

2The 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 Brucejack Project.

Some key milestones for Pretivm are:

- Preliminary Economic Assessment
- Project Description to BCEAO: on hold, typically one month after PEA
- Prefeasibility: on hold typically two years after submission of project description
- Submission of EA: to be determined, typically four months after completion of Prefeasibility Study
- Complete Feasibility Study: to be determined, typically 14 months after completion of Prefeasibility Study.

British Columbia Environmental Assessment Act Process

The British Columbia Environmental Assessment Act (BCEAA) requires that certain large-scale project proposals undergo an environmental assessment and obtain an Environmental Assessment Certificate before they can proceed. Proposed mining developments that exceed a threshold criterion of 75,000 t/a, as specified in the Reviewable Project Regulations, are required under the Act to obtain an Environmental Assessment Certificate from the Ministers of Environment and Energy, Mines and Petroleum Resources before the issuance of any permits to construct or operate. The Brucejack Project will thus require an Environmental Assessment Certificate, because its proposed production rate exceeds the specified threshold.

Authorizations Required

Lists of the major federal and provincial licences, permits, and approvals required to constructing, operating, decommissioning, and close the 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 the large number of minor permits, licences, approvals, consents, and authorizations, and potential amendments that will be required throughout the life of the mine.

British Columbia Authorizations, Licences and Permits

Provincial permitting, licensing, and approval processes (statutory permit processes) may proceed concurrently with the BCEAA review or may, at the proponent's option, follow the Environmental Assessment Certificate. At this time, it is too early to ascertain whether Pretivm will seek concurrent approvals under the BCEAA process. However, no statutory permit approvals may be issued before an Environmental Assessment Certificate is obtained. Statutory permit approval processes are normally more specific than the environmental assessment level of review, and, will require detailed and possibly final engineering design information for certain permits, as, for example, with the TSF structures and others. Table 18.20 presents a list of provincial authorizations, licences, and permits required to develop the Brucejack Project. The list includes only the major permits and is not intended to be comprehensive.

Table 18.20 List of British Columbia Authorizations, Licences and Permits Required to Develop the Brucejack Project

BC Government Permits and Licences	Enabling Legislation
Environmental Assessment Certificate	BCEAA
Permit Approving Work System and Reclamation Program (mine site – initial development)	Mines Act
Amendment to Permit Approving Work System and Reclamation Program (preproduction)	Mines Act
Reclamation Program (bonding)	Mines Act
Amendment to Permit Approving Work System and Reclamation Program (mine plan production)	Mines Act
Approvals to Construct and Operate TSF Dam	Mines Act
Permit Approving Work System and Reclamation Program (gravel pit/wash plant/rock borrow pit)	Mines Act
Water Licence – Notice of Intention (application)	Water Act
Water Licence – Storage and 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 of Occupation – Borrow/Gravel Pits	Land Act
Surface Lease – Mine Site Facilities	Land Act
Waste Management Permit – Effluent (tailings and 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)	Health Act/Environmental
Special Waste Generator Permit (waste oil)	Environmental Management Act (Special Waste Regulations)

Federal Approvals and Authorizations

Federal approvals include an authorization from the Federal Minister of Environment approving the combined Application/Comprehensive Study Report for the Brucejack Project. Major stream-crossing authorizations will be required from Fisheries and Oceans under the Fisheries Act. Approvals for water crossings will also be required under the Navigable Waters Protection Act. An explosive factory licence will be required under the Explosives Act. The MMER under the Fisheries Act, administered by Environment Canada, may require a Schedule 2 authorization because the area proposed for the TSF contains fish habitat. Other activities under federal jurisdiction, such as radio communication and aviation, will require licensing.

Table 18.21 lists some of the federal approvals required.

Table 18.21 List of Federal Approvals and Licences that May be Required to Develop the Brucejack Project

Federal Government Approvals and Licences	Enabling Legislation
CEAA Approval	Canadian Environmental Assessment Act
MMER	Fisheries Act/Environment Canada
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

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%.

18.9 Capital Cost Estimate

The initial capital cost for the Brucejack Project was estimated at US$281.7million with an expected accuracy range of ±35%.

The estimate was developed by Wardrop, with inputs from the following consultants:

- BGC – material take-offs for tailings and water management
- Rescan – water turbidity control and environmental costs
- AMC – mine development
- Pretivm.

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 18.22.

Table 18.22 Capital Cost Summary

Description	US$				
	Labour Cost	Material Cost	Construction Equipment Cost	Process Equipment Cost	Total Cost
Direct Works					
Overall Site	2,619,179	3,317,021	4,013,356	-	9,949,556
Mine Underground (AMC)	-	43,669,414	-	-	43,669,414
Mine Surface Works (AMC)	-	7,855,874	-	-	7,855,874
Mine Site Process	10,893,341	9,754,343	606,753	12,076,700	33,331,137
Mine Site Utilities	5,270,992	1,377,924	1,175,181	9,280,677	17,104,774
Mine Site Buildings	6,657,254	6,793,490	608,351	1,376,564	15,435,659
Tailings	5,448,540	6,401,974	6,150,972	-	18,001,486
Temporary Facilities	3,917,160	-	-	-	3,917,160
Plant Mobile Equipment (Mine Site)	150,660	-	-	3,128,520	3,279,180
Leach Area	6,290,524	6,253,119	813,057	7,734,705	21,091,405
Leach Area Utilities	4,925,820	1,167,234	1,204,445	8,484,669	15,782,168
Leach Mine Buildings	2,740,539	2,838,307	255,517	149,844	5,984,207
Temporary Facilities	779,666	592,875	2,325	-	1,374,866
Plant Mobile Equipment (Leach Site)	70,308	-	-	1,375,470	1,445,778
Direct Works Subtotal	49,763,983	90,021,575	14,829,957	43,607,149	198,222,664
Indirects					
Indirects	2,554,896	43,832,064	-	-	46,386,960
Owner's Costs	4,786,447	3,159,452	-	-	7,945,899
Contingency	-	29,148,589	-	-	29,148,589
Indirects Subtotal	7,341,343	76,140,105	-	-	83,481,448
Total	**57,105,326**	**166,161,680**	**14,829,957**	**43,607,149**	**281,704,112**

18.9.1 Estimate Base Date and Validity Period

Wardrop has prepared this preliminary assessment estimate with a base date of Q2-2011. No escalation beyond Q2-2011 was applied to the estimate. The budget quotes used in this estimate were obtained in Q2-2011.

18.9.2 Estimate Approach

The capital cost estimate was structured as per the project work breakdown structure (WBS) consisting of the main areas as shown in Table 18.23.

Table 18.23 Project WBS

Major	
11	Overall Site
21	Mine Underground
22	Mine Surface Works (AMC)
31	Mine Site Process
32	Mine Site Utilities
33	Mine Site Buildings
34	Tailings
35	Temporary Facilities
36	Plant Mobile Equipment
51	Leach Area
52	Leach Area Utilities
53	Leach Area Buildings
54	Leach Temporary Facilities
55	Leach Mobile equipment
91	Indirects
98	Owner's Costs
99	Contingency

The capital cost estimate was developed based on the following:

- Budget quotations were obtained for the supply of the crushers. An in-house database was used for the balance of the equipment.
- 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 by WEI.

- 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.

Owner provided the owners' costs, including taxes.

18.9.3 Sustaining Capital

Any costs associated with work that is scheduled to start after Year 1 are included in the sustaining capital costs 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 Cdn$90/h was used throughout the estimate. This labour rate was based on guidelines and requirements of the Construction Labour Relation Agreement BC 2011. 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:

- 50% locally
- 25% regionally
- 25% out of town.

The source and availability of labour should be verified in the next phase of the study.

A productivity factor of 1.2 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

Excavation of top soil and allowance for rock excavation was based on the geotechnical information available at the time of the estimate preparation. Structural fill pricing are 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 unit rates. Earthwork quantities do not include any allowance for bulking or compaction of materials; these allowances are included in the unit prices.

Geotechnical information is not available, Wardrop made the following assumptions:

- an average of 50% of the excavated material deemed to be excavation in rock, of which 50% is rippable rock and the balance is a drill and blast type
- surplus excavated material is stockpiled at a location within 5 km from the site.

Concrete

Concrete quantities are included in the building rates, and are based on estimated quantities with an allowance included for over-pour and wastage.

Structural Steel

The structural steel is included in the building rate.

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:

- jaw crusher
- secondary crusher.

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 cost in each area and is based on experience with similar recent projects.

Piping and Valves

Piping and valve costs were estimated as a percentage of cost in each area and is based on experience with recent similar projects.

Electrical

Electrical allowances in the buildings were included as a percentage of equipment in each area, based on experience with recent similar projects.

The power distribution and major equipment was estimated based on recent project experience with similar projects

Instrumentation

Instrumentation was estimated as a percentage of the of cost in each area and 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.

18.9.5 Temporary Works

The estimate has provided for Catering and Housekeeping for the construction staff.

18.9.6 Mobile Equipment

The estimate has provided for mobile equipment at both the mine site and leach site areas.

18.9.7 Permanent Accommodation and Construction Camps

There are permanent and construction camps 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.

18.9.8 Taxes and Duties

The estimate was prepared with taxes (HST, PST, and GST) and duties on materials excluded.

18.9.9 Freight and Logistics

A freight allowance of 8% was provided for materials and the process equipment. The mining mobile equipment costs include freight.

18.9.10 Spares

A spares allowance of 5% was provided for materials and the process equipment, to cover commissioning and strategic spares.

18.9.11 Owner's Costs and Permit Allowances

Owner has provided an allowance of US$6.42 m for Owner's costs and US$1.53 m for Owner's risk.

18.9.12 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.

18.9.13 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.

18.9.14 CONTINGENCY

The contingency allowance included in the estimate amounts to US$29.1 million.

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

Total operating cost for the project is estimated at Cdn$158.35/t milled for the duration of underground mining and Cdn$68.77/t milled for the duration of open pit mining. The estimate includes mining, process, G&A and surface service costs, excluding residue dam construction costs which are included in sustaining capital costs. A total of 265 personnel are projected to be required for the operation, including 130 for mining operation, 97 personnel for process and 38 personnel for surface service and general management.

The unit costs are based on an annual mineralized material production rate of 547,500 t/a or 1,500 t/d and 365 d/a. The currency exchange rate used for the estimate is 1:0.93 (Cdn$:US$). The operation cost for the Brucejack Project has been estimated in Canadian dollars within an accuracy range of ±35%. The breakdown operating cost estimates are presented in Table 18.24.

Table 18.24: Overall Operating Cost*

Area	Personnel	Unit Operating Cost (Cdn$/t milled)	
		Underground Mining	Open Pit
Mining**	130	103.78	15.29
Processing	97	34.56	34.56
G&A	25	14.87	13.78
Plant Services	13	5.14	5.14
Total	**265**	**158.35**	**68.77**

Notes: *tailing/residue management cost are included in sustaining capital cost
**including backfill cost

18.10.2 Mining Operating Cost

The underground mine operating cost is shown in Table 18.25. The underground mine operating costs exclude processing, and G&A. The operating development is notionally the hanging wall drive, crosscuts, and mineralization drives. The underground production includes direct stoping, materials handling, backfill, mine services, technical engineering and mine supervision.

Table 18.25 Underground Mining Operating Cost

Cost Distribution	Unit	Value ($)
Development	Cdn$/t mined	16.30
Production	Cdn$/t mined	87.48
Total	**Cdn$/t mined**	**103.78**

The open pit mine operating cost has been estimated at $15.29/t of mineralized material. This excludes processing and G&A. It is assumed to include load and haul; drill and blast; mine technical services, and mine management.

18.10.3 Process Operating Cost

The estimated process operating cost is Cdn$34.56/t milled or Cdn$19 million per year. The cost includes Cdn$0.94/t milled for concentrate transportation from the mine site to the leach plant site by trucks. The estimate is based on an annual process rate of 547,500 tonnes at operation availability of 92%.

A summary of the plant operation costs is shown in Table 18.26.

Table 18.26 Summary of Process Operating Cost

Description	Labour Force	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Labour Force			
Operating Staff	14	1,439,571	2.63
Operating Labour	51	3,818,827	6.98
Maintenance	32	2,647,483	4.84
Subtotal Labour Force	97	7,905,882	14.44
Major Consumables			
Metal Consumables		615,191	1.12
Reagent Consumables		1,558,583	2.85
Supplies			
Maintenance Supplies		1,871,785	3.42
Operating Supplies		753,800	1.38
Subtotal Consumables and Supplies		4,799,359	8.77
Power Supply		6,216,608	11.35
Subtotal Power		6,216,608	11.35
Total (Process)		**18,921,849**	**34.56**

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:

- Labour force requirement including supervision, operation, and maintenance; salary/wage levels based on current labour rates in comparable operations in British Columbia. Benefit burden of 35% including holiday and vacation payment, pension plan, various other benefits, northern allowance and tool allowance costs
- Power supply from on-site fuel power plant

- Crusher/mill liner and mill grinding media consumptions estimated from the Bond ball mill work index and the Wardrop database
- Maintenance supply costs, including building maintenance costs, 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.

The estimated labour force cost is $14.44/t milled. A total of 97 persons are estimated for the process operation, including14 staff for management and professional services, 51 operators for operating and assaying, and 32 personnel for maintenance. The estimate is based on 12 hours per shift, 24 hours per day and 365 days per year, excluding the crushing operation, which is proposed to operate at 10 hours per day.

The operating cost for the major metal consumables is estimated to be Cdn$1.12/t milled. The metal consumables include mill and crusher liners and mill grinding media.

The estimated reagent cost is Cdn$2.85/t milled. Reagent consumptions are estimated from laboratory test results and comparable operations. The reagent costs are from the current budget prices from potential suppliers or Wardrop database.

The maintenance supplies are estimated at $3.42/t milled. The power cost is estimated based on the average power requirement of 2.3 MW and a unit electric energy price of Cdn$0.33/kWh.

18.10.4 OPERATING COST – PRE-CONCENTRATION (MINE SITE)

The estimate operating cost for the flotation plant including crushing, grinding, bulk flotation, gravity concentration, concentrate dewatering, flotation tailing delivery and water reclaim is shown in Table 18.27. Total cost for the process is estimated at Cdn$21.40/t milled. A total of 62 personnel are required to operate the plant. The management and the technical support including assay will service all the processes.

Table 18.27 Operating Cost – Pre-concentration (Mine Site)

Description	Labour Force	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Labour Force			
Operating Staff	6	986,997	1.80
Operating Labour	30	2,390,778	4.37
Maintenance Labour	26	1,653,321	3.02
Subtotal Labour Force	62	5,031,097	9.19

Description	Labour Force	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Supplies			
Metal Consumables		615,191	1.12
			Table continues...
Reagent Consumables		223,052	0.41
Maintenance Supplies		913,960	1.67
Operating Supplies		408,800	0.75
Power		4,520,928	8.26
Subtotal Supplies		6,681,931	12.21
Total	**62**	**11,713,028**	**21.40**

18.10.5 OPERATING COST - GOLD LEACH AND RECOVERY (LEACH PLANT)

The operating costs for cyanide leaching plant including gold leach, gold recovery, cyanide recovery, leach residue cyanide destruction and disposal, and water reclaim are estimated at Cdn$12.23/t milled. The costs are shown in Table 18.28. The leaching and cyanide handling operations will be operated by their designated personnel. Labour force cost is estimated at Cdn$4.69/t milled. The reagent consumption is the major cost which is estimated to be Cdn$2.44/t milled. The estimated maintenance supply cost is Cdn$1.37/t milled.

Table 18.28 Operating Cost – Gold Leach and Gold Recovery

Description	Human Power	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Labour Force			
Operating Staff	4	452,574	0.827
Operating Labour	15	1,123,184	2.05
Maintenance	12	994,162	1.82
Subtotal Labour Force	31	2,569,920	4.69
Supplies			
Reagent Consumables		1,335,531	2.44
Maintenance Supplies		750,000	1.37
Operating Supplies		345,000	0.63
Power Supply		1,695,680	3.10
Subtotal Supplies		4,126,212	7.54
Total	**31**	**6,696,132**	**12.23**

18.10.6 General and Administration and Surface Services

G&A costs are estimated to be Cdn$14.87/t milled during underground mining, or Cdn$13.78/t milled during open pit mining. The costs are developed by Wardrop and Pretivm.

The G&A costs include:

- labour cost for administrative personnel
- expense and services related to general administration, travelling, human resources, safety and security
- allowances for insurance, regional taxes and licenses allowance
- sustainability, including environment, community liaison and engineering consulting
- transportation of personnel, including air and road transportations
- camp accommodation costs.

A summary of the G&A costs for the underground mining period are provided in Table 18.29.

Table 18.29 G&A Operating Cost – Underground Mining

	Labour Force	Total Cost (Cdn$/a)	Unit Cost (Cdn$/t milled)
G&A Labour Force			
G&A	17	1,603,058	2.93
G&A Hourly Personnel	8	612,000	1.12
Subtotal Labour Force	25	2,215,058	4.05
G&A Expense			
General Office Expense		150,000	0.27
Computer Supplies including Software		50,000	0.09
Communications		100,000	0.18
Travel		100,000	0.18
Audit		50,000	0.09
Consulting/External Assays		100,000	0.18
Head Office Allowance: Marketing		200,000	0.37
Environmental		500,000	0.91
Insurance		238,427	0.44
Regional Taxes and Licenses Allowance		300,000	0.55
Legal Services		100,000	0.18
Warehouse		200,000	0.37
Recruiting		100,000	0.18
Entertainment/Memberships		50,000	0.09
Medicals and First Aid		100,000	0.18
Relocation Expense		50,000	0.09
Training/Safety		200,000	0.37
Accommodation/Camp Costs		2,433,638	4.45

	Labour Force	Total Cost (Cdn$/a)	Unit Cost (Cdn$/t milled)
Crew Transportation (Flight+Bus)		632,580	1.16
Liaison Committee/Sustainability		100,000	0.18
Small Vehicles		70,000	0.13
Others		100,000	0.18
Subtotal Expense		5,924,645	10.82
Total	**25**	**8,139,702**	**14.87**

The surface service cost estimates are shown in Table 18.30 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 18.30 Plant Services Operating Cost

Surface Service	Labour Force	Total Cost (Cdn$/a)	Unit Cost (Cdn$/t milled)
Surface Service Personnel	13	1,021,853	1.87
Surface Service Expense	-	1,790,000	3.27
Small Vehicles/Equipment	-	100,000	0.18
Potable Water and Waste Management	-	240,000	0.44
Supplies	-	100,000	0.18
Building Maintenance	-	450,000	0.82
Building Heating	-	200,000	0.37
Road Maintenance	-	500,000	0.91
Avalanche Control	-	100,000	0.18
Off-Site Operation Expense	-	100,000	0.18
Total	**13**	**2,811,853**	**5.14**

18.11 FINANCIAL ANALYSIS

18.11.1 INTRODUCTION

An economic evaluation of the Brucejack Project was prepared by Wardrop based on a pre-tax financial model. For the 16-year LOM and 8.4 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 27.1% IRR
- 4.2-year payback on US$281.7 million initial capital
- US$662 million NPV at 5% discount rate.

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

- gold – US$1,100/oz
- silver – US$21.00/oz
- exchange rate – 0.93:1.00 (US$:Cdn$).

Sensitivity analyses were carried out to evaluate the project economics for several metal prices scenarios.

18.11.2 PRE-TAX MODEL

FINANCIAL EVALUATIONS

The Brucejack cash flow from underground and open pit operation was consolidated into one financial model. The production schedules have been incorporated into the pre-tax financial model to develop annual recovered metal production. The annual NSR contribution of each metal has been determined by deducting the refining and transportation charges (from mine site to market) from gross revenue.

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.

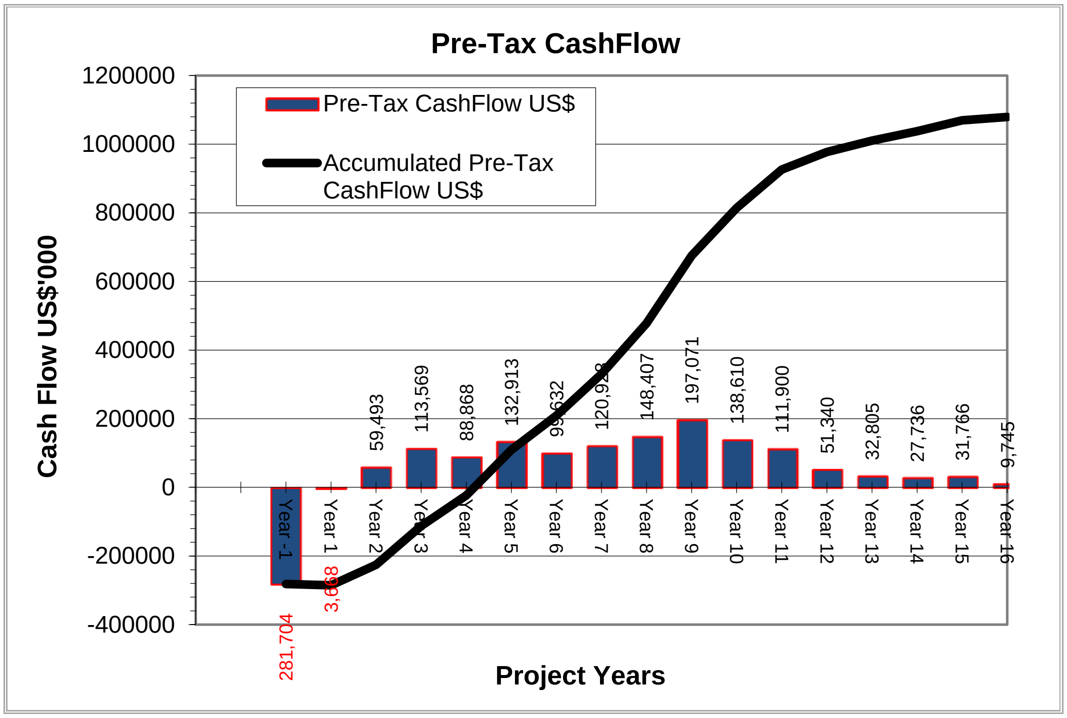
Working capital has been calculated based on a three-month operating cost in Year 1 of the mine operation. It will be recovered at the end of the mine life.

Metal production quantities are presented in Table 18.31. The annual pre-tax cash flow is presented in Figure 18.21.

Table 18.31 Metal Production Quantities

Metal	Average Annual Production		Total Production	
	Years 1 to 10	LOM	Years 1 to 10	LOM
Gold (000 oz)	173.2	135	1731.9	2,157
Silver (000 oz)	1,114.9	918	11,149.2	14,694

Figure 18.21 Pre-tax Cash Flow



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 May 27, 2011.

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 18.32.

Table 18.32 Summary of Pre-tax NPV, IRR, and Payback by Metal Price

Economic Returns	Unit	Base Case	Alternate Case	Spot Prices*
Net Cash Flow	M US$	1,079	512	2,133
NPV at 5.0% Discount Rate	M US$	662	255	1,416
Project IRR	%	27.1	14.3	48.3
Payback	years	4.2	6.6	2.5
Exchange Rate	US$:C$	0.93	0.93	1.02
Mine Life	years	16	16	16
Au Price	US$/oz	1,100	878	1,536.3
Ag Price	US$/oz	21.00	14.50	37.86

Note: *Spot prices as at May 27, 2011

ROYALTIES

The royalties applicable to the Brucejack Project are as follows:

- "Royalty" means the amount payable by the Owner, 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 smelter or buyers of doré, in-house database numbers were used to benchmark the terms supplied by the owner.

Contracts will generally include payment terms as follows:

- Doré:
  - Gold and Silver – pay 99.8% of gold content; a refining and transport charge of $2.00/troy oz will be deducted from the metal price.

18.11.4 Markets and Contracts

Markets

The project will produce gold and silver doré that will be transported by truck from the mine site to the smelter.

Contracts

There are no established contracts for the sale of doré currently in place for this project.

Transport Insurance

An insurance rate of 0.5% will be applied to the provisional invoice value of doré to cover transport from the mine site to the smelter.

18.11.5 Sensitivity Analysis

Sensitivity analyses were carried out on the following parameters:

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

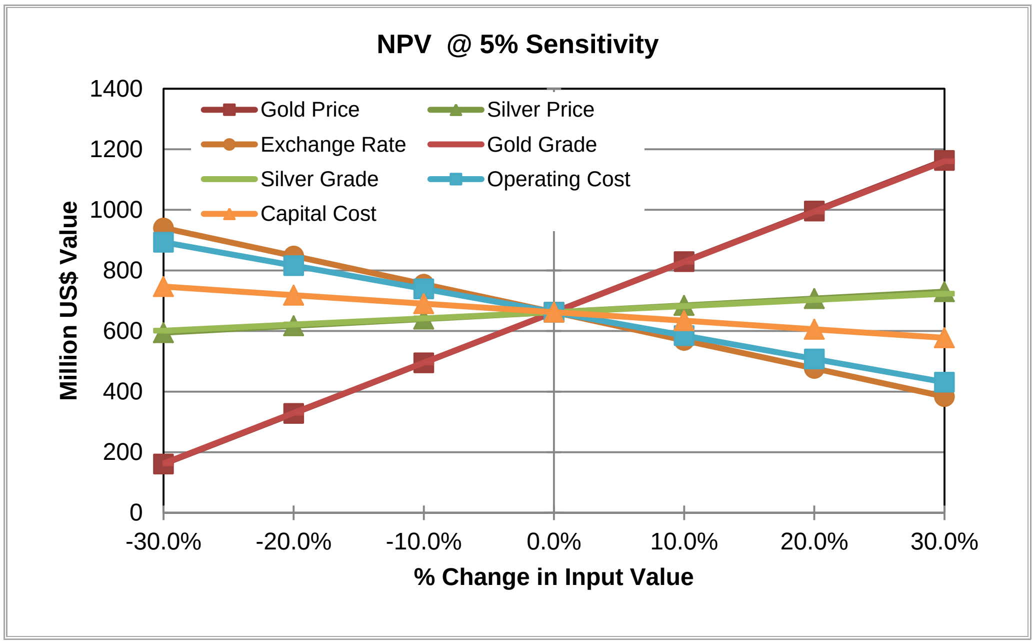
The analyses are presented as financial outcomes in terms of NPV in Table 18.33 and Figure 18.22 and IRR in Table 18.34 and Figure 18.23. The project NPV (at 5% discount rate) is most sensitive to the gold price, gold grade, and exchange rate.

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

Table 18.33 Output Variable Values for NPV

		NPV Sensitivity				
	Units	-20.0%	-10.0%	0.0%	+10.0%	+20.0%
Au Price	US$M	328	495	662	829	996
Ag Price	US$M	617	640	662	685	707
Exchange Rate	US$M	848	755	662	569	477
Au Grade	US$M	330	496	662	828	994
Ag Grade	US$M	621	642	662	683	703
Operating Cost	US$M	816	739	662	585	508
Capital Cost	US$M	718	690	662	634	606

Figure 18.22 NPV 5% Sensitivity Analysis

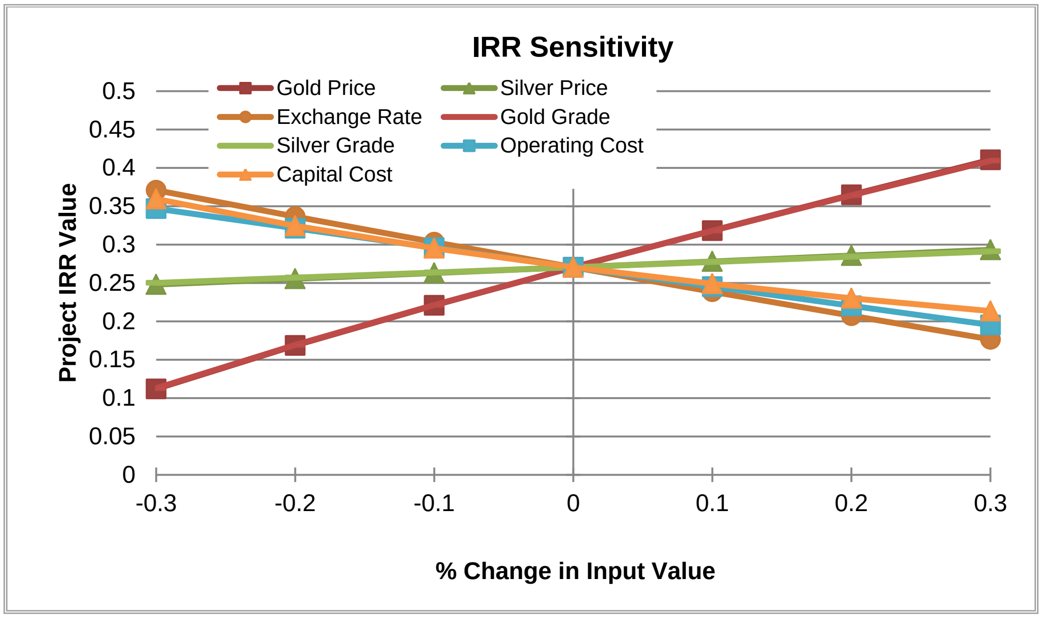


Note: The lines representing the grade and the price of gold overlay

Table 18.34 Output Variable Values for Project IRR

	IRR Sensitivity (%)				
	-20.0%	-10.0%	0.0	+10.0%	+20.0%
AU Price	16.9	22.1	27.1	31.8	36.5
Ag Price	25.5	26.3	27.1	27.8	28.6
Exchange Rate	33.7	30.3	27.1	23.9	20.7
Au Grade	16.9	22.1	27.1	31.8	36.4
Ag Grade	25.7	26.4	27.1	27.7	28.4
Operating Cost	32.1	29.6	27.1	24.5	22.0
Capital Cost	32.5	29.5	27.1	24.9	23.0

Figure 18.23 IRR Sensitivity Analysis



Note: The lines representing the grade and the price of gold overlay

19.0 CONCLUSIONS AND RECOMMENDATIONS

19.1 CONCLUSIONS

Based on the results of this PEA, Wardrop recommends that Pretivm proceed with the next phase of the project in order to identify opportunities and further assess viability of the project.

Based on these conclusions and recommendations, the next 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$5 M and production of the subsequent pre-feasibility report is projected to cost approximately US$4 M, for a total of US$9 M.

19.2 RECOMMENDATIONS

19.2.1 GEOLOGY

P&E is of the opinion that Pretivm should undertake a comprehensive exploration program in 2011, with the focus being on:

- attempting to convert inferred resources to measured and indicated
- testing for extensions of the known mineralization
- prospecting, mapping, and trenching numerous other showings, which were located as part of historical programs.

A 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/HYDROLOGICAL

The following is recommended for the leach TSF at the next level of design:

- Geotechnical and hydrogeological site investigations (i.e. mapping, drilling, geophysics, test pits and associated geotechnical and geochemical laboratory testing) will be completed to confirm the assumptions used to develop the preliminary designs presented in this report. Collection of baseline surface water, groundwater quantity and quality data in the creeks and lake downstream of the TSF should be completed.
- Borrow studies to identify specific locations and characterize potential areas for rockfill, granular filters, and low permeability soils will be completed. This will include surface mapping, drilling, testing pitting and geotechnical and geochemical laboratory testing.
- A seismic hazard assessment will be completed for the proposed TSF site.
- Geotechnical and geochemical laboratory testing will be completed on representative samples of tailings.
- Dam slope stability and seepage analyses will be completed once geotechnical site investigations and laboratory testing are complete.
- A detailed geohazard assessment must be completed to identify and characterize potential geohazards impacting the TSF and auxiliary facilities.
- The closure plan will be re-evaluated.

The following is recommended for the Brucejack Lake and leach TSF water management and water balances at the next stage of design:

- Existing climate and hydrometric stations (i.e. Brucejack Lake and Brucejack Creek) must continue to be monitored and maintained with an appropriate level of quality control.
- Climate stations, including rain gauges, and flow monitoring sites will be installed in the unnamed lake watershed to confirm assumptions on the water balance.
- Acceptable risk tolerance criteria must be established for water management and confirmation of the flood design criteria for pipelines and pumping.
- It is currently assumed that surplus water in Brucejack Lake will be suitable for discharge to the environment and the water in the TSF supernatant pond will not be. This assumption needs to be verified in the next stage of engineering design.

The following is recommended for the underground developments at the next level of design:

- A three-dimensional geologic model should be developed to identify the distributions of rock types, alteration types, weathering grades, and major geologic structures.
- Detailed structural mapping of exposed rock outcrop along drill roads or other access roads, including discontinuity orientation, character, and continuity
- Four to five dedicated geotechnical core holes between 400 m and 500 m deep, targeting the proposed PEA level mining excavation in each of the West Zone and Valley of the Kings Zone areas, for a total of eight to ten geotechnical core holes. These holes should target the hangingwall, footwall, and waste rock adjacent to the ore zone to determine stope and infrastructure excavation stability.
- A point load testing program to evaluate the potential for further division of the geotechnical units according to alteration
- Additional laboratory testing of rock samples collected from the geotechnical core holes
- Additional packer testing of geotechnical core holes and installation of additional vibrating wire piezometers.
- Additional stability analyses of proposed stopes should be completed at the next stage of study after the geotechnical model has been updated. Particular consideration should be given to major structural features and the extent of the weathered zone, as both have the potential to significantly affect stope stability. In addition, crown pillar stability analysis should be completed.
- A pilot dewatering / pumping test should be conducted to evaluate the feasibility of dewatering the historical underground workings before mining begins and provide a large scale estimate of the hydraulic conductivity of the rock mass of the Brucejack study area. An updated assessment of groundwater inflows to the new workings during development and operations should be carried out following the field-testing.
- A suitable pumping system should be designed to extract water from the currently flooded workings and to handle inflows as mining progresses.

19.2.3 Environmental

It is recommended that Pretivm 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.

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 AMC's recommendations regarding mining for the next phase of study:

- Prepare an underground specific resource grade block model that is sufficiently detailed and focused on refined geology controls and mineralisation distribution. This work may identify tighter higher grade zones or more sporadic distribution than is currently interpreted. The model cell size resolution should be commensurate with the proposed mining unit size.
- Conduct a full survey of the existing West Zone underground workings.
- Incorporate glacial surfaces into topography surfaces and block model.
- Conduct a backfill options study that examines all practical backfill types and distribution systems.
- Conduct testwork on the backfill materials (identified from the options study) to determine engineered backfill specifications.
- Conduct mining method options study based on the results of the above recommendations.
- Conduct materials handling study based on the results of the above recommendations to confirm the choice of material handling option.
- Examine the option of using contractor mining versus owner mining for all mining functions.
- Conduct a strategic production rate versus cut-off trade-off study that takes into account various inputs such as process plant throughput, recovery, power demand and capital cost, the different lode recoveries, mine life, rate of return, open pit stockpiling (inclusion or exclusion of low grade stockpiles), ahead of undertaking the detailed study phase. This should be conducted using the underground resource model, to clearly define the cut-off and production rate combination that optimizes Pretivm's corporate goals.
- Produce a new stope inventory based on the underground resource model and cut-offs determined from the detailed costs and results from the strategic trade-off study above.
- Confirm the overall project time-line with respect to start dates for the portal excavations and decline access development, WZ pre-existing ramp dewatering and WZ ramp stripping.
- Complete detailed mine design and schedules incorporating both underground and open pit aspects.

- Undertake adequate engineering and planning to determine the mine ventilation and infrastructure requirements.

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 and the effect of raw water from the underground mine and open pits on flotation
  - gold and silver cyanidation, including cyanide solution handling
  - gravity concentration should be further optimized on the Brucejack mineralization
  - ancillary tests
  - pilot plant scale tests.
- Optimization of primary grinding circuit should be conducted.
- The mill throughput should be optimized further.
- The potential energy saving opportunities should be investigated.

20.0 REFERENCES

_____1972: Porphyry deposits; & Report of Activities, Part B: November 1971 to March 1972; Geological Survey of Canada, Paper 72-1, Part B, p. 62-64.

_____1973: Tectonism, volcanism and copper deposits; & Volcanism and Volcanic Rock; Geological Survey of Canada, Open File 164, p. 129-151.

_____1973: The tops and bottoms of porphyry copper deposits; Economic Geology. v. 68. D. 700-815.

_____1978: Porphyry model; International Molybdenum Encyclopedia 1778-1978, Volume I - Resources and Production, (ed.) A. Sutulov; Intermet Publications, Santiago, Chile, p. 261-270.

_____1979: Magma and hydrothermal fluids; Geochemistry of Hydrothermal Ore Deposits, 2nd edition, (ed.) H.L. Barnes; Wiley Interscience, New York, p. 71-136.

_____1979: The occurrence and significance of phyllic overprinting at porphyry copper-molybdenum deposits (abstract); The Canadian Institute of Mining and Metallurgy, v. 72, no. 803, p. 78.

_____1980: Types of porphyry molybdenum deposits; Mining Magazine, June 1980, p. 550-551.

_____1981: Gradients in silicic magma chambers: implications for lithospheric magmatism; Journal of Geophysical Research, v. 86, p. 10153-10192.

_____1984: Volatiles in magmatic systems; in Fluid-mineral Equilibria in Hydrothermal Systems; Reviews in Economic Geology, v. 1. p.-155-175.

_____1985: Tectonic and petrochemical control on distribution and metal contents of granite-related molybdenum deposits (extended abstract); Granite-related Mineral Deposits, (ed.) R.P Taylor and D.F. Strong; The Canadian Institute of Mining and Metallurgy Conference, Halifax, September, 1984, p. 165-168.

_____1986: Molybdenum, tungsten and tin deposits and associated granitoid intrusions in the northern Canadian Cordillera and adjacent parts of Alaska; & Mineral Deposits of Northern Cordillera, (ed.) J.A. Morin, The Canadian Institute of Mining and Metallurgy, Special Volume 37, p. 216-233.

_____1986: Space-time distribution, crustal setting and Cd:Mo ratios of Central Andean porphyry copper deposits: metallogenic implications; Geology and

Metallogeny of Copper Deposits, Proceedings of the Copper Symposium, 27th International Geological Congress, Moscow, 1984, p. 235-250.

1988: Comb quartz layers in felsic intrusions and their relationship to the origin of porphyry deposits; & Recent Advances in the Geology of Granite-related Mineral Deposits, (ed.) R.P. Taylor and D.F. Strong; The Canadian Institute of Mining and Metallurgy, Special Volume 39, p. 50-71.

_____1988: Evolution of silicic magma in the upper crust: the mid-Tertiary Latir volcanic field and its cogenetic granitic batholith, northern New Mexico, U.S.A.; Transactions of the Royal Society of Edinburgh; Earth Sciences, v. 79, p. 265-288.

_____1990: The Mount Pleasant caldera: geological setting of associated tungsten-molybdenum and tin deposits; in Mineral Deposits of New Brunswick and Nova Scotia, (ed.) D.R. Boyle; 8th International Association on the Genesis of Ore Deposits Symposium, Field Trip Guidebook, Geological Survey of Canada, Open File 2157, p. 73-77.

_____1991: Gold in the Canadian Cordillera - a focus on epithermal and deeper environments; in Ore Deposits, Tectonics and Metallogeny in the Canadian Cordillera; British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 1991-4, p. 163-212.

_____1994: Geological and geochemical zoning of the Grasberg Igneous Complex, Irian Jaya, Indonesia; Journal of Geochemical Exploration, v. 50, p. 143-178.

Alldrick, D. J. (1989): Volcanic Centers in the Stewart Complex; British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 1989-1.

Alldrick, D. J., and Britton, J. M. (1988): Geology and Mineral Deposits of the Sulphurets Area; British Columbia Ministry of Energy, Mines and Petroleum Resources Open Map File 1988-4.

Alldrick, D. J., and Britton, J. M. (1991): Sulphurets Area Geology; British Columbia Ministry of Energy, Mines and Petroleum Resources, Open Map File 1991-21.

Ambrus, J. 1978: Chile; & International Molybdenum Encyclopaedia 1778-1978, Volume I, Resources and Production, (ed.) A. Sutulov; Intermet, Santiago, Chile, p. 54-85.

Anderson, J.A. 1982: Characteristics of leached capping and appraisal; &Advances in Geology of the Porphyry Copper Deposits, Southwestern North America, (ed.) S.R. Titley; University of Arizona Press, Tucson, Arizona, p. 275-296.

Anderson, R. G. (1989): A Stratigraphic, Plutonic, and Structural Framework for the Iskut River Map Area, Northwestern British Columbia; in Current Research, Part E, Geological Survey of Canada, Paper 89-1E, p. 145-154.

Anderson, R. G., and Thorkelson, D. J. (1990): Mesozoic stratigraphy and setting for some mineral deposits in Iskut River map area, northwestern British Columbia; Geological Survey of Canada, Paper 90-1F.

Anderson, R.G. and Bevier, M. L. (1990): A note on Mesozoic and Tertiary K-Ar geochronometry of pluton sites, Iskut River map area, northwestern British Columbia; Geological Survey of Canada, Paper 90-1E.

Anstett, T.F., Bleiwas, D.I., and Hurdelbrink, R.J. 1985: Tungsten availability - market economy countries; United States Bureau of Mines, Information Circular 9025,51 p.

Argall, G.O. 1981: Takeovers shake U.S.A. mining companies; World Mining, May, p. 56-59.

Ayres, L.D., Averill, S.A., and Wolfe, W.J. 1982: An Archean molybdenite occurrence of possible porphyry type at Setting Net Lake, northwestern Ontario, Canada; Economic Geology, v. 77, p. 1105-1119.

Baker, R.C. and Guilbert, J.M. 1987: Regional structural control of porphyry copper deposits in northern Chile (abstract); Geological Society of America, Abstracts with Programs, v. 19, no. 7, p. 578.

Barr, D.A., Fox, P.E., Northcote, K. E. and Preto, V.A. (1976): The Alkaline Suite Porphyry Deposits - A Summary; in Porphyry Deposits of the Canadian Cordillera, Sutherland Brown, A. Editor, Canadian Institute of Mining and Metallurgy, Special Volume 15, pages 359-367.

BGC, 2011. Brucejack Underground Preliminary Assessment – Water Management Plan. Report prepared for Pretivm Resources Inc. June 1, 2011.

Blanchard, R. 1966: Interpretation of leached outcrops; Nevada Bureau of Mines, Bulletin 66, 196 p.

Blanchflower, J.D., 2008: Technical Report on the Snowfield Property, Skeena Mining Division, British Columbia, Canada. Report for Silver Standard Resources.

Bookstrom, A.A. 1981: Tectonic setting and generation of Rocky Mountain porphyry molybdenum deposits; & Relations of Tectonics to Ore Deposits in the Southern Cordillera, (ed.) W.R. Dickinson and W.D. Payne; Arizona Geological Society Digest, v. 14, p. 251-226.

British Columbia Geological Survey, Minfile, 2008: MINFILE Detail reports on the Mitchell (104B 182), Iron Cap (104B 173), and Brucejack (104B 345).

Britton, J. M. and Alldrick, D. J. (1988): Sulphurets Map Area; in Geological Fieldwork 1987, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 1988-1, pp. 199-209.

Budinski D., McKnight R and Wallis C. (2001) Sulphurets-Bruceside Property British Columbia technical report. Pincock Allen & Holt Ltd. report for Silver Standard Resources.

Budinski, David, R. (1995): Summary Report on the Snowfield Project Sulphurets Property, Skeena Mining Division; private report prepared for Orcan Consultants.

Burk, R. (2007): Presentation on the Snowfield Gold Project; presented at the Exploration Roundup Conference in Vancouver, B.C., January, 2007; PowerPoint slides and text.

Burnham, C.W. 1967: Hydrothermal fluids at the magmatic stage; Geochemistry of Hydrothermal Ore Deposits, (ed.) H.L. Barnes; Holt, Rinehart and Winston Inc., New York, p. 34-76.

Bushnell, S.E. 198s: Mineralization at Cananea, Sonora, Mexico, and the paragenesis and zoning of breccia pipes in quartzofeldspathic rock; Economic Geology, v. 83, p. 1760-1781.

Candela, PA. 1989: Calculation of magmatic fluid contributions to porphyry-type ore system: predicting fluid inclusion chemistries; Geochemical Journal, v. 23, p. 295-305.

Candela, PA. and Holland, H.D. 1986: A mass transfer model for copper and molybdenum in magmatic hydrothermal systems: the origin of porphyry-type ore deposits; Economic Geology, v. 81, no. 1, p. 1-19.

Carlile, J.C. and Mitchell, A.H.G. 1994: Magmatic arcs and associated gold and copper mineralization in Indonesia; Journal of Geochemical Exploration, v. 50, p. 91-142.

Carson, D.J.T. and Jambor, J.L. 1974: Mineralogy, zonal relationships and economic significance of hydrothermal alteration at porphyry copper deposits, Babine Lake area, British Columbia; The Canadian Institute of Mining and Metallurgy, Bulletin, v. 76, no. 742, p. 110-133.

Carten, R.B. 1986: Sodium-calcium metasomatism: chemical, temporal, and spatial relationships at the Yerington, Nevada, porphyry copper deposit; Economic Geology, v. 81, p. 1495-1519.

Carten, R.B., Geraghty, E.P., and Walker, B.M. 1988a: Cyclic development of igneous features and their relationship to high-temperature hydrothermal features in the Henderson porphyry molybdenum deposit, Colorado; Economic Geology, v. 83, p. 266-296.

Carten, R.B., Walker, B.M., Geraghty, E.P., and Gunow, A.J. 1988b: Comparison of field-based studies of the Henderson porphyry molybdenum deposit, Colorado with experimental and theoretical models of porphyry systems; in Recent Advances in the Geology of Granite-related Mineral Deposits, (ed.) R.P. Taylor and D.F. Strong; The Canadian Institute of Mining and Metallurgy, Special Volume 39, p. 351-366.

Carten, R.B., White, W.H., and Stein, H.J. 1993: High-grade, granite-related molybdenum systems: classification and origin: in Mineral Deposit Modeling, (ed.) R.V. Kirkham, W.D. Sinclair, R.I. Thorpe, and J.M. Duke; Geological Association of Canada, Special Paper 40, p. 521-554.

Carter, N.C. 1981: Porphyry copper and molybdenum deposits, west-central British Columbia; British Columbia Ministry of Energy, Mines and Petroleum Resources, Bulletin 64,150 p.

CESL (1990), Sulphurets Property, Newhawk Gold Mines Ltd., Feasibility Study, dated March 1990.

Chen Chucai and Li Guansheng 1990: Dexing copper mine; Mining Magazine, v. 162, p. 287-288.

Christiansen, E.H., Burt, D.M., Sheridan, M.F., and Wilson, R.T. 1983: The petrogenesis of topaz rhyolites from the western United States; Contributions to Mineralogy and Petrology, v. 83, p. 16-30.

Christopher, PA. and Pinsent, R. 1982: Geology of the Ruby Creek and Boulder Creek area near Atlin (104N/llW); British Columbia Ministry of Energy, Mines and Petroleum Resources, notes to accompany Preliminary Map 52,10 p.

Cline, J.S. and Bodnar, R.J. 1991: Can economic porphyry copper mineralization be generated by a typical calc-alkaline melt?: Journal of Geophysical Research, v. 96, p.8113-8126.

Columba C., M. and Cunningham, C.G. 1993: Geologic model for the mineral deposits of the La Joya district, Oruro, Bolivia; Economic Geology, v. 88, p. 701-708.

Cominco Engineering Services Ltd , March 1990: Feasibility Study Sulphurets Property Newhawk Gold Mines Ltd., Volume II,Draft

Conception, R.A. and Cinco, J.C., Jr. 1989: Geology of Lepanto Far Southeast gold-rich porphyry copper deposit, Mankayan, Benguet, Philippines (abstract); 28th International Geological Congress, Washington, D.C., Abstracts, v. 1, p. 319-320.

Corona Corporation (1990), Newhawk Gold Mines Ltd., Sulphurets Project, Feasibility Study, dated October, 1990.

Cox, D. 1985: Geology of the Tanama and Helecho porphyry copper deposits and vicinity, Puerto Rico; United States Geological Survey, Professional Paper 1327,57 p.

Cox, D.P. and Singer, DA. 1988: Distribution of gold in porphyry copper deposits; United States Geological Survey, Open File Report 88-46,22 p.

Deere, D.U. and Deere, D.W. 1988. The rock quality designation (RQD) index in practice. Rock classification systems for engineering purposes, (ed L. Kirkaldie), ASTM Special Publication 984, 91-101. Philadelphia: Am. Soc. Test Mat.

Dilles, J.H. and Einaudi, M.T. 1992: Wall-rock alteration and hydrothermal flow paths about the Ann-Mason porphyry copper deposit, Nevada - a 6-km vertical reconstruction; Economic Geology, v. 87, p. 1963-2001.

Driscoll, F. G. (1986), Groundwater and Wells, 8 ed., Johnson Screens, St. Paul. Minnesota, 1089 p.

Eaton, PC. and Setterfield, T.N. 1993: The relationship between epithermal and porphyry hydrothermal systems within the Tavua Caldera, Fiji; Economic Geology, v. 88, p. 1053-1083.

Einaudi, M.T. 1982: Description of skarns associated with porphyry copper plutons: Advances in Geology of the Porphyry Copper Deposits, (ed.) S.R. Titley; The University of Arizona, Press, Tucson, Arizona, p. 139-183.

Ewert W., Armstrong, T., Brown F., and Puritch E. (2009): Technical report and resource estimates on the West, Bridge, Galena Hill, Shore, SG and Gossan Hill gold and silver zones of the Brucejack property. P&E Mining Consultants Inc. report for Silver Standard Resources.

Flores V., R. 1994: Precious metal deposits of the Refugio area, northern Chile; Society for Mining, Metallurgy and Exploration, Inc., for Society of Mining Engineers Annual Meeting, Albuquerque, New Mexico, February 14-17,1994, Pre-print 94-109,9 p.

Fraser, R.J. 1993: The Lac Troilus gold-copper deposit, northwestern Quebec: a possible Archean porphyry system; Economic Geology, v. 88, p. 1685-1699.

Freeze, R. A., and J. A. Cherry (1979), Groundwater, 2nd ed., 604 pp., Prentice Hall, New Jersey.

Geological Survey of Canada., 2009. Geological Compilation of Bowser and Sustut Basins Draped on Shaded Relief Map, North-Central British Columbia. Geological Survey of Canada Open File 5794. 2009.

Geyti, A. and Thomassen, B. 1984: Molybdenum and precious metal mineralization at Flannefjeld, southeast Greenland; Economic Geology, v. 79, p. 1921-1929.

Goldie, R. 1990: Ok Tedi: a copper-gold porphyry emplaced in a compressional environment; The Gangue (Newsletter of the Mineral Deposits Division of the Geological Association of Canada), issue no. 33, October, 1990, p. 10-14.

Grant, J.N., Halls, C., Sheppard, S.M.F., and Avila, W. 1980: Evolution of the porphyry tin deposits of Bolivia; Granitic Magmatism and Related Mineralization, (ed.) S. Ishihara and S. Takenouchi; Mining Geology Special Issue, no. 8, The Society of Mining Geologists of Japan, p. 151-173.

Grant, N., Halls, C., Avila, W., and Avila, G. 1977: Igneous geology and the evolution of hydrothermal systems in some sub-volcanic tin deposits of Bolivia; Geological Society of London, Special Volume 7, p. 117-126.

Grimstad and Barton. 1993. Updating the Q-system for NMT. Proc. Int. Symp. On Sprayed Concrete, Fagernes, (eds. Kompen, Opsahl and Berg), Oslo: Norwegian Concrete Association.

Guan Xunfan, Shou Yongqin, Xiao Jinghua, Lian Shuzhao, and Li Jimao 1988: A new type of tin deposit - the Yinyan porphyry tin deposit in China; -in Geology of Tin Deposits in Asia and the Pacific, (ed.) C.S. Hutchison; Springer-Verlag, Berlin, New York, p. 487-494.

Guilbert, J.M. 1986: Recent advances in porphyry base metal deposit research; in Geology and Metallogeny of Copper Deposits, Proceedings of the Copper Symposium, 27th International Geological Congress, Moscow, 1984, p. 196-208.

Guilbert, J.M. and Park, C.F., Jr. 1986: The Geology of Ore Deposits; W.H. Freeman, New York, 985 p.

Gustafson, L.B. 1978: Some major factors of porphyry copper genesis; Economic Geology, v. 73, p. 600-607.

Gustafson, L.B. and Hunt, J.P. 1975: The porphyry copper deposit at El Salvador, Chile; Economic Geology, v. 70, p. 857-912.

Heidrick, T.L. and Titley, S.R. 1982: Fracture and dike patterns in Laramide plutons and their structural and tectonic implication: American Southwest; Advances in Geology of the Porphyry Copper Deposits, Southwestern North America, (ed.) S.R. Titley; University of Arizona Press, Tucson, Arizona, p. 73-91.

Heinrich, C.A., Ryan, C.G., Mernach, T.P., and Eadington, P.J. 1992: Segregation of ore metals between magmatic brine and vapor: a fluid inclusion study using PIXE microanalysis; Economic Geology, v. 87, p. 1566-1583.

Heithersay, P.S., O'Neill, W.J., van der Helder, P., Moore, C.R., and Harbon, P.G. 1990: Goonumbla porphyry copper district - Endeavour 26 North, Endeavour 22 and Endeavour 27 copper gold deposits; in Geology of the Mineral Deposits of Australia and Papua New Guinea, (ed.) F.E. Hughes; The Australasian Institute of Mining and Metallurgy, Melbourne, Australia, p. 1385-1398.

Hildreth, W. 1979: The Bishop Tu E evidence for the origin of compositional zonation in silicic magma chambers; Ash-flow Tuffs; Geological Society of American, Special Paper 180, p. 43-75.

Hollister, V.F. 1974: Regional characteristics of porphyry copper deposits of South America; Society of Mining Engineers of American Institute of Mining, Metallurgical and Petroleum Engineers, Transactions, v. 255, p. 45-53.

Huang Dianhao, Wu Chengyu, and Nie Fengjun 1988: Geological features and genesis of the Jinduicheng porphyry molybdenum deposit, Shaanxi Province, China; Chinese Journal of Geochemistry, v. 7, p. 136-147.

Huang Xunde and Zhang Dingyuan 1989: Geochemical zoning pattern of the Yinyan tin deposit; Journal of Geochemical Exploration, v. 33, p. 109-119.

Hudson, T., Arth, J.G., and Muth, K.G. 1981: Geochemistry of intrusive rocks associated with molybdenite deposits, Ketchikan Quadrangle, southeastern Alaska; Economic Geology, v. 76, p. 1225-1232.

Hunt, J. P. (1991): Porphyry Copper Deposits, Economic Geology Monograph 8, 1991, pp 192-206.

Hutchinson, J., and Diederichs, M. 1996. Cable bolting in underground mines. Bitech Publishers, Richmond, B.C., Canada. p. 406.

Imai, A, Muyco, J.D., Domingo, E.G., Almeda, RL.,Vilones, RI., Jr., Yumul, G.P., Jr., Damasco, F.V., Baluda, R.P., Malihan, T.D., Datuin, R.T., Punongbayan, R.S., Listanco, E.L., and Santos, R.A. 1992: Porphyry copper and gold mineralization in the Philippines; in Mineral Deposits of Japan and the Philippines; 29th International Geological Congress Field Trip Guide Book, Volume 61, Field Trip C36, p. 1-30.

Ishihara, S. 1981: The granitoid series and mineralization; Economic Geology Seventy-fifth Anniversary Volume, 1905-1980, (ed.) B.J. Skinner; Economic Geology Publishing Co., p. 458-484.

James, A.H. 1971: Hypothetical diagrams of several porphyry copper deposits; Economic Geology, v. 66, p. 43-47.

James, L.P. 1978: The Bingham copper deposits, Utah, as an exploration target: history and pre-excavation geology; Economic Geology, v. 73, p. 1218-1227.

Johnson, C.M., Czamanske, G.K., and Lipman, P.W. 1989: Geochemistry of intrusive rocks associated with the Latir volcanic field, New Mexico, and contrasts between evolution of plutonic and volcanic rocks; Contributions to Mineralogy and Petrology, v. 103, p. 90-109.

Jones, B.K. 1992: Application of metal zoning to gold exploration in porphyry copper systems; Journal of Geochemical Exploration, v. 43, p. 127-155.

Jones, G.J. 1985: The Goonumbala porphyry copper deposits, New South Wales; Economic Geology, v. 80, p. 591-613.

Keith, J.D. and Shanks, W.C., III 1988: Chemical evolution and volatile fugacities of the Pine Grove porphyry molybdenum and ash-flow tuff system, southwestern Utah; & Recent Advances in the Geology of Granite-related Mineral Deposits, (ed.) R.P. Taylor and D.F. Strong; The Canadian Institute of Mining and Metallurgy, Special Volume 39, p. 402-423.

Keith, J.D., Shanks, W.C., 111, Archibald, DA., and Farrar, E. 1986: Volcanic and intrusive history of the Pine Grove porphyry molybdenum system, southwestern Utah; Economic Geology, v. 81, p. 553-577.

Kesler, S.E. 1973: Copper, molybdenum and gold abundances in porphyry copper deposits; Economic Geology, v. 68, p. 106-112.

Kesler, S.E., Jones, L.M., and Walker, R.L. 1975: Intrusive rocks associated with porphyry copper mineralization in island arc areas; Economic Geology , v. 70, p. 515-526.

Kirkham, R. V. (1991): Provisional Geology of the Mitchell-Sulphurets Region, Northwestern British Columbia (104B/8, 9); Geological Survey of Canada, Open File 2416.

Kirkham, R.V. and Sinclair, W.D. 1984: Porphyry copper, molybdenum, tungsten Canadian Mineral Deposit Types: a Geological Synopsis, (ed.) O.R. Eckstrand; Geological Survey of Canada, Economic Geology Report 36, p. 50-52.

Kirkham, R.V., McCann, C., Prasad, N., Soregaroli, A.E., Vokes, F.M., and Wine, G. 1982: Molybdenum in Canada, part 2: MOLYFILE - an index-level computer file of molybdenum deposits and occurrences in Canada; Geological Survey of Canada, Economic Geology Report 33,208 p.

Kirkham, R.V., Sinclair, W.D. :1995 Geology of Canadian Mineral Deposit Types. Geology of Canada No. 8, edited by O.R. Eckstrand, O. R., Sinclair, W.D., Thorpe, R.I.

Kirkham. R.V. 1971: Intermineral intrusions and their bearing on the origin of porphyry copper and molybdenum deposits; Economic Geology, v. 66, p. 1244-1250.

Knight Piésold Ltd., 2011. Brucejack Creek Hydroelectric Project – Project Description for Preliminary Assessment – Draft Report. Prepared for Pretivm Resources Inc. April 8, 2011.

Kontak, D.J. and Clark, AH. 1988: Exploration criteria for tin and tungsten mineralization in the Cordillera Oriental of southeastern Peru; & Recent Advances in the Geology of Granite-related Mineral Deposits, (ed.) R.P. Taylor and D.F. Strong; The Canadian Institute of Mining and Metallurgy, Special Volume 39, p. 157-169.

Kooiman, GJA., McLeod, M.J., and Sinclair , W.D. 1986: Porphyry tungsten-molybdenum orebodies, polymetallic veins and replacement bodies, and tin-bearing greisen zones in the Fire Tower zone, Mount Pleasant, New Brunswick; Economic Geology, v. 81, p. 1356-1373.

Lameyre, J. and Bowden, P. 1982: Plutonic rock types series: discrimination of various granitoid series and related rocks; Journal of Volcanology and Geothermal Research, v. 14, p. 169-186.

Lang, J.R., Stanley, C.R. and Thompson, H.F.H. (1993): A Subdivision of Alkalic Porphyry Cu-Au Deposits into Silica-saturated and Silica-undersaturated Subtypes; in Porphyry Copper-Gold Systems of British Columbia, Mineral Deposit Research Unit, University of British Columbia, Annual Technical Report - Year 2, pages 3.2-3.14.

Lehmann, B. 1990: Metallogeny of tin; Lecture Notes in Earth Sciences, v. 32, Springer-Verlag, Berlin, 211 p.

Lin Guiqing 1988: Geological characteristics of the ignimbrite-related Xiling tin deposit in Guangdong Province; in Geology of Tin Deposits in Asia and the Pacific, (ed.) C.S. Hutchison; Springer-Verlag, Berlin, New York, p. 495-506.

Linn, K.O., Wieselmann, Ed., GaIay, I., Harvey, J.J.T., Tufiio, G.F., and Winfield, W.D.B. 1981: Geology of Panama's Cerro Colorado porphyry copper deposit; Mineral and Energy Resources, v. 24, no. 6, p. 1-14.

Lipman, P.W. 1984: The roots of ash flow calderas in western North America: windows into the tops of granitic batholiths; Journal of Geophysical Research, v. 89, p. 8801-8841.

Lipman, P.W. and Sawyer, DA. 1985: Mesozoic ash-flow caldera fragments in southeastern Arizona and their relation to porphyry copper deposits; Geology, v. 13, p. 652-656.

Liu Wengzhang 1981: Geological features of mineralization of the Xingluokeng tungsten (molybdenum) deposit, Fujian Province; & Tungsten Geology, China, (ed.) J.V. Hepworth and Yu Hong Zhang, UN Economic and Social Commission for Asia and the Pacific Regional Mineral Resources Development Centre, Bandung, Indonesia, p. 339-348.

Lowell, J.D. and Guilbert, J.M. 1970: Lateral and vertical alteration-mineralization zoning in porphyry ore deposits; Economic Geology, v. 65, p. 373-408.

Lowell, J.U. 1974: Three new porphyry copper mines for Chile?; Mining Engineering, v. 26, no. 11, p. 22-28.

MacDonald, G.D. and Arnold, L.C. 1993: Intrusive and mineralization history of the Grasberg deposit Irian Jaya, Indonesia; for presentation at the Society of Mining Engineers Annual Meeting Reno, Nevada, February 15-18,1993, Society for Mining, Metallurgy, and Exploration, Inc., Preprint number 93-92, p. 1-10.

Manning, DAC. and Pichavant, M. 1988: Volatiles and their bearing on the behaviour of metals in granitic systems; Recent Advances in the Geology of Granite-related Mineral Deposits, (ed.) R.P. Taylor and D.F. Strong; The Canadian Institute of Mining and Metallurgy, Special Volume 39, p. 13-24. McCutcheon, S.R.

Margolis, J. (1993): Geology and Intrusion Related Copper-Gold Mineralization, Sulphurets, British Columbia; Ph. D. thesis prepared for University of Oregon.

McCarthy, P.L., 1993, “Economics of narrow vein mining,” seminar, Narrow Vein Mining, Australian Institute of Mining and Metallurgy, Bendigo, Victoria, Australia, pp. 89-97.

McCrea, J. A. (2007): 2006 QA/QC Review, Snowfield Gold Zone; private report prepared for Silver Standard Resources Inc., pp. 17 with illustrations.

McCrea, J. A. (2007): Technical Report on the Snowfields Project, Skeena Mining Division, British Columbia, Canada; private report prepared for Silver Standard Resources Inc., pp. 33 plus appendices.

McCrea, J. A. (2008): 2007 QA/QC Review of Drill Hole Sample Data, Snowfield Project; private report prepared for Silver Standard Resources Inc., pp. 18 with illustrations.

McCutcheon, S.R., Anderson, H.E., and Robinson, P.T. in press: Stratigraphy and eruptive history of the Late Devonian Mount Pleasant caldera complex, Canadian Appalachians; Geological Magazine.

McInnes, B.I.A. and Cameron, E.M. 1994: Carbonated alkaline hybridizing melts from a sub-arc environment: mantle wedge samples from the Tabar-Lihir-Targa-Feni arc, Papua New Guinea; Earth and Planetary Science Letters, v. 122, p. 125-144.

McKinnon, A and Seidel, H. 1988: Tin; Register of Australian Mining, 1988189, (ed.) R. Louthean; Resource Information Unit Ltd., Subiaco, Western Australia, p. 197-204.

McMillan, W. J. (1991): Porphyry Deposits in the Canadian Cordillera; in Ore Deposits, Tectonics and Metallogeny in the Canadian Cordillera, B. C. Ministry of Energy, Mines and Petroleum Resources, Paper 1991-4, pages 253-276.

McMillan, W. J. and Panteleyev, A. (1988): Porphyry Copper Deposits; in Ore Deposit Models, Roberts, R.G. and Sheahan, P.A, Editors, Geoscience Canada, Reprint Series 3, pages 45-58.

McMillan, W.J. 1991: Porphyry deposits in the Canadian Cordillera; in Ore Deposits, Tectonics and Metallogeny in the Canadian Cordillera, British Columbia Geological Survey Branch, Paper 1991-4, p. 253-276.

McMillan, W.J. and Panteleyev, A. 1980: Ore deposit models - 1. Porphyry copper deposits; Geoscience Canada, v. 7, p. 52-63.

McMillan, W.J., Newman, K., Tsang, L., and Sanford, G. 1985: Geology and ore deposits of the Highland Valley camp; Geological Association of Canada, Field Guide and Reference Manual Series, no. 1,121 p.

McPhearson, M. D. (1993): 1993 Summary Report on the Sulphurets Property, Snowfield Property, Skeena Mining Division; private report prepared for Newhawk Gold Mines Ltd.

Meldrum, S.J., Aquino, R.S., Gonzales, R.I., Burke, R.J., Suyadi, A, Irianto, B., and Clarke, D.S. 1994: The Batu Hijau porphyry copper-gold deposit, Sumbawa Island, Indonesia; Journal of Geochemical Exploration, v. 50, p. 203-220.

Metallurgical Division at Inspectorate America Corp., December 2009 to July 2010: Data Reports

Metallurgical Division at Inspectorate America Corp., September 2010 to April 2011: Data Reports

Meyer, J. and Foland, Kd. 1991: Magmatic-tectonic interaction during early Rio Grande rift extension at Questa, New Mexico; Geological Society of America Bulletin, v. 103, p. 993-1006.

Meyer, J.W., Osborne, L.W., Atkin, SA., Molling, PA., Moore, R.F., and Olmore, S.D. 1982: Preliminary geology and molybdenum deposits at Questa, New Mexico; The Genesis of Rocky Mountain Ore Deposits: Changes with Time and

Tectonics; Proceedings of Denver Region Exploration Geologists Society Symposium, November 1982, p. 151-155.

Miller, R.N. 1973: Production history of the Butte district and geological function, past and present; & Guidebook for the Butte Field Meeting of Society of Economic Geologists, (ed.) R.N. Miller; August 18-21, 1973, p. F1-F10.

Minister of Justice, 2011. Fisheries Act - Metal Mining Effluent Regulations. May 1, 2011.

Mitchell, A.H. and Garson, M.S. 1972: Relationship of porphyry copper and circum-Pacific tin deposits to paleo-Benioff zones; Institution of Mining and Metallurgy, Transaction, v. 81, p. B10-25.

Moyle, A.J., Doyle, B.J., Hoogvliet, H., and Ware, A.R. 1990: Ladolam gold deposit, Lihir Island; & Geology of the Mineral Deposits of Australia and Papua New Guinea, (ed.) F.E. Hughes; The Australasian Institute of Mining and Metallurgy, Melbourne, Australia, p. 1793-1805.

Moyle, J.E. 1984: Development and construction begins at East Kemptville, North America's only primary tin mine; Mining Engineering, April 1984, p. 335-336.

Muller, D. and Groves, D.I. 1993: Direct and indirect associations between potassic igneous rocks, shoshonites and gold-copper deposits; Ore Geology Reviews, v. 8, p. 383-406.

Muller, D., Heithersay, P.S., and Groves, D.I. 1994: The shoshonite porphyry Cu-Au association in the Goonumbala district, N.S.W., Australia; Mineralogy and Petrology, v. 50, p. 299-321.

Mutschler, F. E. and Mooney, T. C. (1993): Precious Metal Deposits Related to Alkaline Igneous Rocks - Provisional Classification, Grade-Tonnage Data, and Exploration Frontiers; IUGS/UNESCO Conference on Deposit Modeling, Ottawa, 1990, Proceedings Volume, Geological Association of Canada, Special Paper 40, pages 479-520.

Mutschler, F.E., Griffin, M.E., Stevens, D.S., and Shannon, S.S., Jr. 1985: Precious metal deposits related to alkaline rocks in the North American Cordillera - an interpretive review; Transactions of the Geological Society of South Africa, v. 88, p. 355-377.

Mutschler, F.E., Wright, E.G., Ludington, S., and Abbott, J.T. 1981: Granite molybdenite systems; Economic Geology, v. 76, p. 874-897.

Nickson, S.D., 1992. Cable support guidelines for underground hard rock mine operations. M.Sc. Thesis, University of British Columbia, Vancouver, B.C .

Noble, S.R., Spooner, E.T.C., and Harris, F.R. 1984: The Logtung large tonnage, low-grade W scheelite)-Mo porphyry deposit, south-central Yukon Territory; Economic Geology, v. 79, p. 848-868.

Nokleberg, W.J., Bundtzen, T.K., Berg, H.C., Brew, D.A., Grybeck, D., Robinson, M.S., Smith, T.E., and Yeend, W. 1987: Significant metalliferous lode deposits and placer districts of Alaska; United States Geological Survey, Bulletin 1786,104 p.

Norman, D.I. and Sawkins, F.J. 1985: The Tribag breccia pipes: Precambrian Cu-Mo deposits, Batchawana Bay, Ontario; Economic Geology, v. 80, p. 1593-1621.

Ojeda F., J.M. 1986: Escondida porphyry copper deposit, II Region, Chile: exploration drilling and current geological interpretation; & Papers Presented at the Mining Latin/Mineria Latinoamerican Conference; Institute of Mining and Metallurgy, Meeting, November 17-19, Santiago, Chile, p. 299-318.

P&E Mining Consultants Inc., February 2009: Technical Report and Resource Estimate on the Snowfield Deposit, Skeena Mining Division, British Columbia, Canada.

Panteleyev, A. (1995): Porphyry Cu-Au: Alkalic, in Selected British Columbia Mineral Deposit Profiles, Volume 1 – Metallics and Coal, Lefebure, D. V. and Ray, G. E., Editors, British Columbia Ministry of Energy of Employment and Investment, Open File 1995-20, pages 83-86.

Panteleyev, A. 1981: Berg porphyry copper-molybdenum deposit; British Columbia Ministry of Energy, Mines and Petroleum Resources, Bulletin 66, 158 p.

Parrish, I.S. and Tully, J.V. 1978: Porphyry tungsten zones at Mt. Pleasant, N.B.; The Canadian Institute of Mining and Metallurgy Bulletin, v. 71, no. 794, p. 93-100.

Pearson. M.F.. Clark.. K.F.. and Porter. E.W. 1988: Mineralogy, fluid characteristics, and silver distribution at Real de Angeles, Zacatecas, Mexico; Economic Geology, v. 83, p. 1737-1759.

Perello, J.A. 1994: Geology, porphyry Cu-Au, and epithermal Cu-Au-Ag mineralization of the Tombulialato district, North Sulawesi, Indonesia; Journal of Geochemical Exploration, v. 50, p. 221-256.

Perello, J.A., Fleming, J.A., O'Kane, K.P., Burt, P.D., Clarke, G.A., Himes, M.D., and Reeves, AT. 1995: Porphyry Copper-gold-molybdenum Mineralization in the Island Copper Cluster, Vancouver Island; The Canadian Institute of Mining and Metallurgy, Special Volume 46, p. 214-238.

Phillips, W.J. 1973: Mechanical effects of retrograde boiling and its probable importance in the formation of some porphyry ore deposits; Institution of Mining and Metallurgy Transactions, v. B82, p. 90-98.

Potvin, Y. 1988. Empirical open stope design in Canada. Ph.D. thesis, Dept. Mining and Mineral Processing, University of British Columbia, 343 p.

Preto, V. 1972: Geology of Copper Mountain, British Columbia Department of Mines and Petroleum Resources, Bulletin 59, 87 p.

Process Research Associates Ltd., 2007: Various spreadsheets documenting the preliminary metallurgical studies and results carried out on behalf of Silver Standard Resources Inc., November 2006 to March, 2007.

Rehrig, W.A. and Heidrick, T.L. 1972: Regional fracturing in Laramide stocks of Arizona and its relationship to porphyry copper mineralization; Economic Geology, v. 67, p. 198-213.

Richards, J.P. and Kerrich, R. 1993: The Porgera gold mine, Papua New Guinea: magmatic hydrothermal to epithermal evolution of an alkalic-type precious metal deposit; Economic Geology, v. 88, p. 1017-1052.

Richardson, J.M. 1988: Field and textural relationships of alteration and greisen-hosted mineralization at the East Kemptville tin deposit, Davis Lake complex, southwest Nova Scotia; in Recent Advances in the Geology of Granite-related Mineral Deposits, (ed.) R.P. Taylor and D.F. Strong; The Canadian Institute of Mining and Metallurgy, Special Volume 39, p. 265-279.

Roach, S. and MacDonald, A. J. (1992): Silver-Gold Mineralization, West Zone, Brucejack Lake, Northwestern British Columbia (104B/8E); in Geological Fieldwork 1991, British Columbia Ministry of Energy, Mines and Petroleum Resources, Paper 1992-1, p. 503-511.

Ross, K.V., Godwin, C.I., Bond, L., and Dawson, K.M. 1995: Geology, alteration and mineralization of the Ajax East and Ajax West deposits, southern Iron Mask Batholith, Kamloops, British Columbia; & The Canadian Institute of Mining and Metallurgy, Special Volume 46, p. 565-580.

Saegart, W.E. and Lewis, D.E. 1977: Characteristics of Philippine porphyry copper deposits and summary of current production and reserves; American Institute of Mining and Metallurgy, Transactions, v. 262, p. 199-208.

Scott, K.M. 1981: Wall-rock alteration in disseminated tin deposits, southeastern Australia; Proceedings of the Australasian Institute of Mining and Metallurgy, no. 280, December, p. 17-28.

Seabridge Gold Inc. (2008): Geology and Mineral Resource Information on the Kerr, Sulphurets and Mitchell deposits; http://www.seabridgegold.net/PKerrSulGold.htm.

Setterfield, T.N., Eaton, P.C., Rose W.J., and Sparks, R.S.J. 1991: The Tavua Caldera, Fiji: a complex shoshonitic caldera formed by concurrent faulting and downsagging; Journal of the Geological Society, v. 148, p. 115-127.

Shannon, J.R., Walker, B.M., Carter, R.B., and Geraghty, E.P. 1982: Unidirectional solidification textures and their significance in determining relative ages of intrusions at the Henderson mine, Colorado; Geology, v. 19, p. 293-297.

Siems, P.L. 1989: Lithium silicate alteration of tin granites: an analog of potassium silicate alteration in porphyry copper and molybdenite deposits (abstract); The Geological Society of America, Abstracts with Programs, v. 21, no. 5, p. 143.

Sikka, D.G., Petruk, W., Nehru, C.E., and Zhang, Z. 1991: Geochemistry of secondary copper minerals from Proterozoic porphyry copper deposit, Malanjkhand, India; Ore Geology Reviews, v. 6, p. 257-290.

Sillitoe, R.H. 1972: A plate tectonic model for the origin of porphyry copper deposits; Economic Geology, v. 67, p. 184-197.

Sillitoe, R.H. and Bonham, H.F., Jr. 1984: Volcanic landforms and ore deposits; Economic Geology, v. 79, p. 1286-1298.

Sillitoe, R.H. and Gappe, I.M., Jr. 1984: Philippine porphyry copper deposits: geologic setting and characteristics; Committee for Co-ordination of Joint Prospecting of Mineral Resources in Asian Offshore Areas (CCOP), Technical Report 14,89 p.

Sillitoe, R.H., Grauberger, G.L., and Elliott, J.E. 1985: A diatreme-hosted gold deposit at Montana Tunnels, Montana; Economic Geology, v. 80, p. 1707-1721.

Sillitoe, R.H., Halls, C., and Grant, J.N. 1975: Porphyry tin deposits in Bolivia; Economic Geology, v. 70, p. 913-927.

Sinclair, A.J., Drummond, A.D., Carter, N.C., and Dawson, K.M. 1982: A preliminary analysis of gold and silver grades of porphyry-type deposits in western Canada; in Precious Metals in the Northern Cordillera, (ed.) A.A. Levinson; The Association of Exploration Geochemists, p. 157-172.

Sinclair, W.D. 1982: Gold deposits of the Matachewan area, Ontario; in Canadian Gold Deposits, (ed.) R.W. Hodder and W. Petmk; The Canadian Institute of Mining and Metallurgy, Special Volume 24, p. 83-93.

Smith, R.L. 1979: Ash-flow magmatism; in Ash-flow Tuffs; Geological Society of America, Special Paper 180, p. 5-27.

Soregaroli, A.E. and Sutherland Brown, A. 1976: Characteristics of Canadian Cordilleran molybdenum deposits; in Porphyry Deposits of the Canadian

Cordillera, (ed.) A. Sutherland Brown; The Canadian Institute of Mining and Metallurgy, Special Volume 15, p. 417-431.

Strishkov, V.V. 1984: The copper industry of the U.S.S.R.: problems, issues and outlook; United States Bureau of Mines, Mineral Issues, 80 p.

Sutherland Brown, A. 1969: Mineralization in British Columbia and the copper and molybdenum deposits; The Canadian Institute of Mining and Metallurgy, v. 72, p. 1-15.

Sutherland Brown, A., Editor, (1976): Porphyry Deposits of the Canadian Cordillera; Canadian Institute of Mining and Metallurgy, Special Volume 15, 510 pages.

Suttill, K.R. 1988: Cerro Rico de Potosi; Engineering and Mining Journal, March, 1988, p. 50-53.

Sutulov, A. 1977: Chilean copper resources said to be world's 1argest American Metal Market, August 4, p. 18-19.

Taylor, H.K., 1996, “Rates of working of mines — a simple rule of thumb,” Transactions, Institute of Mining and Metallurgy, London, Vol. 95, pp. A203-A204.

Taylor, R.G. and Pollard, P.J. 1986: Recent advances in exploration modelling for tin deposits and their application to the Southeast Asian environment; Regional Conference on the Geology and Mineral Resources of Southeast Asia V Proceedings, v. 1, Geological Society of Malaysia, Bulletin 19, p. 327-347.

Taylor, R.P. and Strong, D.F. (ed.) 1988: Recent Advances in the Geology of Granite-related Mineral Deposits; The Canadian Institute of Mining and Metallurgy, Special Volume 39, 445 p.

Theodore, T.G. and Menzie, W.D. 1984: Fluorine-deficient porphyry molybdenum deposits in the western North American Cordillera; Proceedings of the Sixth Quadrennial International Association on the Genesis of Ore Deposits Symposium, E. Schweizerbart'sche Verlagsbuchhandlung, Stuttgart, p. 463-470.

Theodore, T.G., Blake, D.W., and Kretschmer, E.L. 1982: Geology of the Copper Canyon porphyry copper deposits, Lander County, Nevada; in Advances in the Geology of the Porphyry Copper Deposits, (ed.) S.R. Titley; The University of Arizona Press, Tucson, Arizona, p. 543-550.

Theodore, T.G., Blake, D.W., Loucks, T.A., and Johnson, C.A. 1992: Geology of the Buckingham Stockwork molybdenum deposit and surrounding area, Lander County, Nevada; United States Geological Survey, Professional Paper 798-D, p. Dl-D307.

Thompson, T.B., Trippel, A.D., and Dwelley, P.C. 1985: Mineralized veins and breccias of the Cripple Creek district, Colorado; Economic Geology, v. 80, p. 1669-1688.

Tindall, M. (1991): Report on the 1991 Sulphside Exploration Program, Sulphurets Property, Skeena Mining Division, British Columbia; private report prepared for International Corona Corp, 1992.

Titley, S.R. (ed.) 1982: Advances in Geology of the Porphyry Copper Deposits - Southwestern North America; The University of Arizona Press, Tucson, Arizona, 560 p.

Titley, S.R. 1993: Characteristics of porphyry copper occurrence in the American southwest; in Mineral Deposit Modeling, (ed.) R.V. Kirkham, W.D. Sinclair, R.I. Thorpe, and J.M. Duke; Geological Association of Canada, Special Paper 40, p. 433-464.

Titley, S.R. and Beane, R.E. 1981: Porphyry copper deposits; in Economic Geology Seventy-fifth Anniversary Volume, 1905-1980, (ed.) B.J. Skinner; Economic Geology Publishing Co., p. 214-269.

Titley, S.R. and Hicks, C.L. (ed.) 1966: Geology of the Porphyry Copper Deposits, Southwestern North America; The University of Arizona Press, Tucson, Arizona, 287 p.

Titley, S.R., Thompson, R.C., Haynes, F.M., Manske, S.L., Robison, L.C., and White, J.L. 1986: Evolution of fractures and alteration in the Sierrita-Esperanza hydrothermal system, Pima County, Arizona; Economic Geology, v. 81, p. 343-370.

Tosdal, R. M. and Richards, J. P. (2001): Magmatic and structural controls on the development of porphyry Cu ± Mo ± Au deposits; Reviews in Economic Geology Vol.14, p. 157-181.

U.S. Army Corps of Engineers. 1980. Engineering and design: Rock reinforcement. Engineering Manual EM 1110-1-20907. Available from the Office of the Chief of Engineers, Washington, D.C.

van Leeuwen, T.M. 1994: 25 years of mineral exploration and discovery in Indonesia; Journal of Geochemical Exploration, v. 50, p. 13-90.

van Leeuwen, T.M., Taylor, R., Coote, A, and Longstaffe, F.J. 1994: Porphyry molybdenum mineralization in a continental collision setting at Malala, northwest Sulawesi, Indonesia; Journal of Geochemical Exploration, v. 50, p. 279-315.

Vila, T. and Sillitoe, R.H. 1991: Gold-rich porphyry systems in the Maricunga gold-silver belt, northern Chile; Economic Geology, v. 86, p. 1238-1260.

Villalpando, BA. 1988: The tin ore deposits of Bolivia; & Geology of Tin Deposits in Asia and the Pacific, Selected Papers from the International Symposium on the Geology of Tin Deposits held in Nanning, China, October 26-30,1984, p. 201-215.

Visagie, D. and Roach, S. (1991): Evaluation Snowfield Project, Skeena Mining Division; private report prepared for Newhawk Gold Mines Ltd., 1992.

Wallace, S.R., Muncaster, N.K., Jonson, D.C., Mackenzie, W.B., Bookstrom, A.A., and Surface, V.A. 1968: Multiple intrusion and mineralization at Climax, Colorado; Ore Deposits of the United States, 1933-1967 (Graton-Sales volume), (ed.) J.D. Ridge; American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., New York, p. 605-640.

Wampler, P. 1993: Geology, hydrothermal alteration, and geographical information system analysis of the Zortman gold mine, Montana (extended abstract); & Integrated Methods in Exploration and Discovery, (ed.) S.B. Romberger and D.I. Fletcher; Conference Program and Extended Abstracts, Golden, Colorado, April, 1993, p. A33 124-125.

Wardrop Engineering Inc. 2010: Technical Report and Preliminary Assessment of the Snowfield-Brucejack Project Document No. 1053750400-REP-R0001-04.

Westra, G. 1978: Porphyry copper genesis at Ely, Nevada: Papers on Mineral Deposits of Western North America, (ed.) J.D. Ridge; 5th International Association on the Genesis of Ore Deposits Quadrennial Symposium Proceedings, Volume 11, Nevada Bureau Mines and Geology, Report 33, p. 127-140.

Westra, G. and Keith, S.B. 1981: Classification and genesis of stockwork molybdenum deposits; Economic Geology, v. 76, p. 844-873.

White, W.H., Bookstrom, AA., Kamilli, R.J., Ganster, M.W., Smith, R.P., Ranta, D.E., and Steininger, R.C. 1981: Character and origin of Climax-type molybdenum deposits; Economic Geology Seventy-fifth Anniversary Volume, 1905-1980, (ed.) B.J. Skinner, Economic Geology Publishing Co., p. 270-316.

Whitney, J.A. 1975: Vapour generation in a quartz monzonite magma: a synthetic model with application to porphyry copper deposits; Economic Geology, v. 70, p. 346-358.

Wilson, J.C. 1978: Ore fluid-magma relationships in a vesicular quartz latite porphyry dike at Bingham, Utah; Economic Geology, v. 73, p. 1287-1307.

Wojdak, P. (2007: Snowfield gold project; in Exploration and Mining in British Columbia 2006, Northwest Region; British Columbia Ministry of Energy, Mines and Petroleum Resources, p. 40.

Woodcock, J.R. and Hollister, V.F. 1978: Porphyry molybdenite deposits of the North American Cordillera; Mineral Science and Engineering, v. 10, p. 3-18.

Wright, F., and Tse, P., 2008: Snowfield Project, Preliminary Cyanidation and Flotation Studies; private report prepared for Silver Standard Resources Inc., March 28, 2008, pp. 233.

Wright, F., September 8, 2010: Preliminary Metallurgical Study on the Brucejack Lake Gold Project; prepared for Silver Standard Resources Inc., Draft – For Review.

21.0 CERTIFICATES OF QUALIFIED PERSONS

21.1 HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, of Vancouver, BC, do hereby certify:

- I am a Manager of Metallurgy with Wardrop Engineering Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, BC.
- This certificate applies to the technical report entitled “Technical Report and Preliminary Economic Assessment of the Brucejack Project” dated June 3, 2011, (the “Technical Report”).
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1988) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia (#30408).
- My relevant experience with respect to mineral process engineering includes 22 years experience in mining and plant operation, project studies, management, and engineering.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- I am responsible for Sections 1.14, 1.16, 1.17, 18.5, 18.6, 18.8, 18.9, and 18.11 of the Technical Report.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have had no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of June 2011, at Vancouver, BC.

"Original document signed and sealed by
Hassan Ghaffari, P.Eng. "

Hassan Ghaffari, P.Eng.
Manager of Metallurgy
Wardrop Engineering Inc.

21.2 JOHN HUANG, P.ENG.

I, Jianhui (John) Huang, of Burnaby, BC, do hereby certify:

- I am a Senior Metallurgist with Wardrop Engineering Inc. with a business address at #800 - 555 West Hastings Street, Vancouver, BC.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment of the Brucejack Project" dated 27 May, 2011 (the "Technical Report").
- I am a graduate of North-East University (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals (M.Eng., 1988), and Birmingham University (Ph.D., 2000).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #30898).
- My relevant experience with respect to mineral engineering includes more than 28 years of involvement in mineral process for base metal ores, gold and silver ores, and rare metal ores.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on April 12, 2010.
- I am responsible for Sections 1.1, 1.8, 1.9, 1.12, 1.13, 1.15, 1.18, 2.0, 3.0, 16.0, 18.2, 18.10 (excluding 18.10.2), 19.1, and 19.2.5 of the Technical Report.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have had prior involvement with the Property that is the subject of the Technical Report, by acting as a Qualified Person for the "Technical Report and Preliminary Assessment on the Snowfield Property", dated June 1, 2010.
- I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of June 2011, at Vancouver, BC.

"Original document signed and sealed by John (Jianhui) Huang, P.Eng. "

John (Jianhui) Huang, P.Eng.
Senior Metallurgist
Wardrop Engineering Inc.

21.3 PIERRE PELLETIER, P.ENG.

I, Pierre Pelletier, P.Eng., of Vancouver, BC, do hereby certify:

- I am the President of Rescan Environmental Services Ltd. with a business address at 600 - 1111 West Hastings St., Vancouver, BC.
- This certificate applies to the technical report entitled Technical Report and Preliminary Economic Assessment of the Brucejack Project, dated June 3, 2011 (the "Technical Report").
- I am a graduate of the University of Montana, Montana College of Mineral Science and Technology, (Environmental Engineering, 1993).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, #27928.
- My relevant experience is an environmental engineer with 20 years of experience in mining and the environment. Over the last 15 years I have managed several Environmental and Social Impact Assessments. I have also permitted treatment plants and mine closure plans, lead due diligences and environmental audits and have been the "Qualified Person" for environmental and social aspects of several Preliminary Economic Assessments, Pre-Feasibility and Feasibility Studies.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was April 13, 2010, for the day.
- I am responsible for Sections 1.11 and 18.7 of the Technical Report.
- I am independent of Pretivm Resources Inc. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 3rd day of June 2011, at Vancouver, BC.

"Original document signed and sealed by Pierre Pelletier, P.Eng."

Pierre Pelletier, P.Eng.
President
Rescan Environmental Services Ltd

21.4 TRACY ARMSTRONG, P.GEO.

I, Tracy Armstrong, of Magog, PQ, do hereby certify:

- I am a Senior Associate Geologist with P&E Mining Consultants Inc. with a business address at 2, County Court Blvd., Suite 202, Brampton, ON, L6W 3W8.
- This certificate applies to the technical report entitled "Technical Report and Preliminary Economic Assessment of the Brucejack Project" dated June 3, 2011, (the "Technical Report").
- I am a graduate of Queen's University (B.Sc. Honours in Geological Sciences, 1982).
- I am a member in good standing of the Order of Geologists of Quebec (Licence #566), the Association of Professional Geoscientists of Ontario (Licence #1204), and the Association of Professional Geoscientists of British Columbia (Licence # 34720).
- My relevant experience is as follows: Underground Production Geologist, Agnico-Eagle Laronde Mine (1988-1993); Exploration Geologist, Laronde Mine (1993-1995); Exploration Coordinator, Placer Dome (1995-1997); Senior Exploration Geologist, Barrick Exploration (1997-1998); Exploration Manager, McWatters Mining (1998-2003); Chief Geologist Sigma Mine (2003); Consulting Geologist (2003-present).
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I am responsible for Sections 1.2, 1.3, 1.4, 1.5, and 4.0 through 15.0 of the Technical Report.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have had prior involvement with the Property that is the subject of the Technical Report, by co-authoring previous Technical Reports on the Snowfield and Brucejack Properties, the most recent of which is titled "Technical Report and Updated Resource Estimates on the Snowfield-Brucejack Project, Skeena Mining Division, British Columbia Canada," dated April 4, 2011.
- I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of June 2011, at Magog, PQ.

"Original document signed and sealed by Tracy Armstrong, P.Geo."

Tracy Armstrong, P.Geo.
Senior Associate Geologist
P&E Mining Consultants Inc.

21.5 FRED H. BROWN, P.ENG.

I, Fred H. Brown, of Lynden, WA, USA, do hereby certify:

- I am an independent geological consultant with a business address at SuiteB-10 16010 Grover St., Lynden, WA, USA.
- This certificate applies to the technical report entitled “Technical Report and Preliminary Economic Assessment of the Brucejack Project” dated June 3, 2011, (the “Technical Report”).
- I am a graduate of New Mexico State University (B.Sc., 1987), and the University of the Witwatersrand (M.Sc., 2005).
- I am a member in good standing of the American Institute of Professional Geologists (CPG #11015).
- My relevant experience is as follows: Underground Mine Geologist, Freegold Mine, AAC (1987-1995); Mineral Resource Manager, Vaal Reefs Mine, Anglogold (1995-1997); Resident Geologist, Venetia Mine, De Beers (1997-2000); Chief Geologist, De Beers Consolidated Mines (2000-2004); Consulting Geologist (2004-2010).
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was September 2, 2010, for 5 days.
- I am responsible for Sections 1.6, 17.0, and 19.2.1 of the Technical Report.
- I am independent of the Issuer as defined by Section 1.4 of the Instrument.
- I have had prior involvement with the Property that is the subject of this Technical Report, including acting as co-author of the following reports: “Technical Report and Resource Estimate on the Snowfield Property, Skeena Mining Division, British Columbia, Canada,” dated February 13, 2009, “Technical Report and Updated Resource Estimate on the Snowfield Property, Skeena Mining Division, British Columbia, Canada,” dated December 1, 2009, and “Technical Report and Preliminary Assessment on the Snowfield Property”, dated June 1, 2010.
- I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3rd day of June 2011, at Lynden, WA, USA.

"Original document signed and sealed by
Fred Brown, M.Sc. (Eng), CPG, Pr.Sci.Nat."

Fred Brown, M.Sc. (Eng), CPG, Pr.Sci.Nat.
Consulting Geologist
P&E Mining Consultants Inc.

21.6 WARREN NEWCOMEN, P.ENG.

I, H. Warren Newcomen, of Kamloops, BC, do hereby certify that:

- I am a Geotechnical Engineer with BGC Engineering Inc. (BGC) with a business address at 503 - 1315 Summit Drive, Kamloops, BC, V2C 5R9.
- This certificate applies to the technical report titled Technical Report and Preliminary Economic Assessment of the Brucejack Project, dated June 3, 2011, (the “Technical Report”).
- I am a graduate of the University of British Columbia (B.A.Sc., 1985) and the University of California at Berkeley, (M.S., 1990), and I have practiced my profession continuously since 1990.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Registration No. 16123.
- My relevant experience with respect to pit slope designs and slope stability includes design work for the following projects: KSM Project, BC; Cortez Hills, Nevada; Donlin Creek Project, Alaska; Galore Creek Project, BC; Golden Bear Project, BC; Goldstrike Mine, Nevada; Palabora Mine, South Africa; New Afton Project, BC; Ajax Project, BC.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was August 25 and 26, 2010 for two days.
- I am responsible for Section 18.4 of the Technical Report.
- I am independent of Pretivm Resources Inc. as defined by Section 1.4 of the Instrument.
- I have had prior involvement with the Property that is the subject of the Technical Report, by acting as a Qualified Person for the “Technical Report and Preliminary Assessment on the Snowfield Property”, dated June 1, 2010.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 3rd day of June 2011, at Kamloops, BC.

"Original document signed and sealed by H. Warren Newcomen, P.Eng."

H. Warren Newcomen, P. Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

21.7 Hamish Weatherly, P.Geo.

I, Hamish Weatherly, of North Vancouver, BC, do hereby certify that:

- I am a Geoscientist with BGC Engineering Inc. (BGC) with a business address at Suite 500 - 1045 Howe Street, Vancouver, BC, V6Z 2A9.
- This certificate applies to the technical report titled Technical Report and Preliminary Economic Assessment of the Brucejack Project, dated June 3, 2011, (the “Technical Report”).
- I am a graduate of the University of Waterloo (B.Sc., 1992) and the University of British Columbia, (M.Sc., 1995), and I have practiced my profession continuously since 1996.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Registration No. 25567.
- My relevant experience with respect to mine site hydrology and water management planning includes design work for the following projects: Pueblo Viejo Mine, Dominican Republic; Donlin Creek Project, Alaska; Cerro Casale Project, Chile; Cochenour Mine, Ontario; Detour Lake Project, Ontario.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- Due to seasonal/weather restrictions, I have not yet visited the site.
- I am responsible for Sections 18.3.1 and 18.3.3 of the Technical Report.
- I am independent of Pretivm Resources Inc. as defined by Section 1.4 of the Instrument.
- I have had prior involvement with the Property that is the subject of the Technical Report, by providing hydrologic input for the “Technical Report and Preliminary Assessment on the Snowfield Property,” dated June 1, 2010.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 3rd day of June 2011, at Vancouver, BC.

"Original document signed and sealed by Hamish Weatherly, P.Geo."

Hamish Weatherly, P. Geo.
Senior Hydrologist
BGC Engineering Inc.

21.8 CLINT LOGUE, P.ENG. P.GEO.

I, Clint Logue, of Vancouver, BC, do hereby certify that:

- I am an Engineer with BGC Engineering Inc. (BGC) with a business address at 500-1045 Howe Street, Vancouver, BC, V6Z 2A9.
- This certificate applies to the technical report titled Technical Report and Preliminary Economic Assessment of the Brucejack Project, dated June 3, 2011, (the “Technical Report”).
- I have a bachelor of applied science in geological engineering from the University of British Columbia and I have practiced my profession continuously since 1995.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (professional engineer and geoscientist, registration number 28162); Association of Professional Engineers, Geologists and Geophysicists of Alberta (professional engineer and geologist, registration number 85484); Professional Engineers Ontario (professional engineer, registration number 100169541); and the Department of Licensing, the State of Washington (licensed geologist and engineering geologist, registration number 1914).
- My relevant experience includes technical lead for scoping through feasibility level design for tailings facilities, plant site and water handling facilities for the Donlin Creek Project in Alaska, USA; project manager and technical lead for scoping through feasibility level design for tailings and waste rock facilities, plant site, and water handling facilities for the Agua Rica Project in Argentina; technical lead for scoping through final design and construction of tailings facilities and water handling facilities for the Pueblo Viejo Project in the Dominican Republic; and project manager and technical lead for feasibility level design of plant site, slag storage, open pits, and water handling facilities for the Onça Puma Project in Brazil.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- I have not visited the project site due to schedule and seasonal constraints.
- I am responsible for Sections 1.10, 18.3.2, and 18.3.4 of the Technical Report.
- I am independent of Pretivm Resources Inc. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 3rd day of June 2011, at Vancouver, BC.

"Original document signed and sealed by
Clint Logue, P.Eng., P.Geo."

Clint Logue, P.Eng., P.Geo.
Senior Geotechnical Engineer
BGC Engineering Inc.

21.9 PETER MOKOS, MAUSIMM (CP)

I, Peter P Mokos, of Vancouver, BC, do hereby certify:

- I am a Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd. with a business address at Suite 1330, 200 Granville Street, Vancouver, British Columbia Canada V6C 1S4.
- This certificate applies to the technical report entitled Technical Report and Preliminary Economic Assessment of the Brucejack Project, dated June 3, 2011 (the "Technical Report").
- I am a graduate of Ballarat College of Advanced Education, located in the state of Victoria of Australia with the qualification of Bachelor of Engineering (Mining), 1985.
- I am a member in good standing of The Australasian Institute of Mining and Metallurgy (AusIMM) of Australia, 1991, Certificate No. 95.91. My membership status is Chartered Professional (AusIMM) #109937.
- My relevant experience is operational and planning roles for projects in Australia, Papua New Guinea and Ghana and 14 years of mining consultancy work having undertaken numerous mine studies for projects similar to that at the Brucejack Project.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was May 18, 2011 for 2 days of surface only visits.
- I am responsible for Section(s) 1.7, 18.1, 18.10.2, and 19.2.4 of the Technical Report.
- I am independent of Pretivm Resources Inc. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.

- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 3rd day of June 2011, at Vancouver, BC.

"Original document signed and sealed by Peter P Mokos, MAusIMM (CP)"

Peter P Mokos, MAusIMM (CP)
Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd.

22.0 DATE AND SIGNATURE PAGE

The effective date of this Technical Report, titled "Technical Report and Preliminary Economic Assessment of the Brucejack Project", is June 03, 2011.

Signed,

"signed and sealed"

Hassan Ghaffari, P.Eng.
Manager of Metallurgy
Wardrop Engineering Inc.

"signed and sealed"

John Huang, P.Eng.
Senior Metallurgist
Wardrop Engineering Inc.

"signed and sealed"

Pierre Pelletier, P.Eng.
President
Rescan Environmental Services Ltd

"signed and sealed"

Tracy Armstrong, P.Geo.
Senior Associate Geologist
P&E Mining Consultants Inc.

"signed and sealed"

Fred Brown, M.Sc. (Eng), CPG,
Pr.Sci.Nat.
Consulting Geologist
P&E Mining Consultants Inc.

"signed and sealed"

H. Warren Newcomen, P. Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

"signed and sealed"

Hamish Weatherly, P. Geo.
Senior Hydrologist
BGC Engineering Inc.

"signed and sealed"

Clint Logue, P.Geo. P.Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

"signed and sealed"

Peter Mokos, MAusIMM (CP)
Principal Mining Engineer
BGC Engineering Inc.