Report to:

PRETIUM RESOURCES INC.

Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project

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TECHNICAL REPORT AND UPDATED 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 Pretium Resources Inc. (Pretium) by Wardrop, a Tetra Tech Company (Tetra Tech), P&E Mining Consultants Inc. (P&E), BGC Engineering Inc. (BGC), Rescan Environmental Services Ltd. (Rescan), and AMC Mining Consultants (Canada) Ltd. (AMC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Tetra Tech'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 Pretium's contracts with Tetra Tech, P&E, BGC, Rescan, and AMC. This contract permits Pretium 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
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d

Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	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 bank cubic metres per annum	Mbm^3/a
Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
Tonne (1,000 kg)	t
Tonnes per day	t/d

Tonnes per hour .... t/h
Tonnes per year .... t/a
Tonnes seconds per hour metre cubed .... ts/hm^3
Volt V
Week .... wk
Weight/weight .... w/w
Wet metric ton .... wmt
Year (annum) .... a

Abbreviations and Acronyms

Absolute Relative Difference .... ABRD
Acid Base Accounting .... ABA
Acid Rock Drainage .... ARD
Aero Geometrics Ltd. .... Aero Geometrics
Alpine Tundra .... AT
AMC Mining Consultants (Canada) 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 Inc.	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
Magnetotelluric	MT
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
New York Stock Exchange	NYSE
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 Person	QP
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
Toronto Stock Exchange	TSX
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
Valley of Kings	VOK
Wardrop, a Tetra Tech Company	Tetra Tech
West Zone	WZ
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

Pretium Resources Inc. (Pretium) commissioned Wardrop, a Tetra Tech Company (Tetra Tech), to complete an updated preliminary economic assessment (PEA) of the Brucejack Project (the Project or the Property).

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

- Tetra Tech: 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, site wide groundwater studies, and geotechnical design for on site facilities
- GeoSpark Consulting Inc. (GeoSpark): quality assurance/quality control (QA/QC) and database management.

1.2 PROPERTY DESCRIPTION AND LOCATION

The Project is located within the Sulphurets District in the Iskut River region, approximately centred at latitude 56°28′20″N by longitude 130°11′31″W, a position approximately 950 km northwest of Vancouver and 65 km north-northwest of the town of Stewart, British Columbia (BC) (Figure 1.1) and 21 km south-southeast of the Eskay Creek Mine.

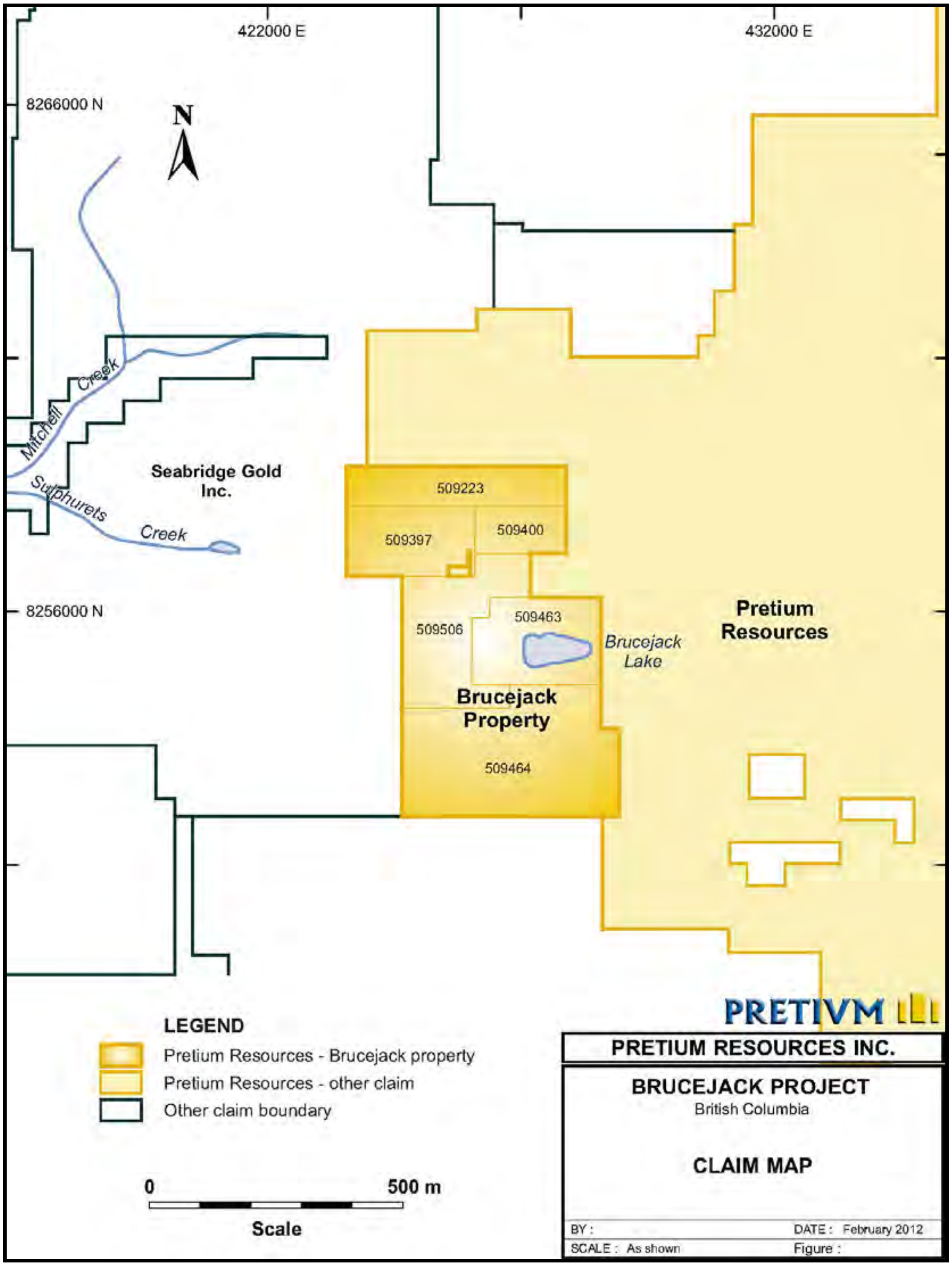
The Project consists of six mineral claims totalling 3,199.28 ha in area; all claims are in good standing until January 31, 2022 (Figure 1.2). The Property basically falls within the boundaries of the Cassiar-Iskut-Stikine Land and Resource Management Plan (LRMP) area with only a minor south-eastern 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 however, none of the claims falls inside any Protected or Special Management Area.

At present the land claims in the area are in review and subject to ongoing discussions between various First Nations and the Government of BC.

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 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 (Cu-Mo) mineralization on the Sulphurets property in the vicinity of the Main Copper Zone, approximately 6 km northwest of Brucejack Lake; however, these claims were not staked until 1960.

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

In 1960, Granduc Mines Ltd. (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. Various operators explored the Sulphurets Property, and an underground program was completed on the West Zone between 1986 and 1991 by the Newcana joint venture (JV).

Between 1986 and 1999 various operators explored the Sulphurets Property, and an underground program was completed on the West Zone (part of the Brucejack Property) between 1986 and 1991 by the Newcana JV.

In 1999, Silver Standard Resources Inc. (Silver Standard) acquired Newhawk Gold Mines Ltd. (Newhawk) and with it, Newhawk's 60% interest and control of the Project JV.

Work on the Property by Silver Standard began in 2009 with a large diamond drilling campaign and resampling program of historical core, followed by a NI 43-101 compliant technical report and resource estimate completed by P&E

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 to 15 m are common at higher elevations while the accumulations range from 2 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 Property is largely underlain by Lower Jurassic rocks of western Stikine terrane, or Stikinia, an oceanic island arc terrane consisting of mid-Paleozoic to Middle Jurassic rocks which underlies much of western BC. Stikinia may have been accreted to the western margin of North America as early as the late Middle Jurassic, and it was likely consolidated with rocks of the North American margin, as well as with rocks of the outboard Insular terrane (Wrangellia and Alexander terranes) in latest Jurassic to mid-Cretaceous time.

The Sulphurets mining camp and the Property lie astride the eastern margin of the core of the McTagg anticlinorium, which is a major north-trending mid-Cretaceous structural culmination in the western Skeena fold belt (SFB). Coincident with the core of the anticlinorium is a prominent and very well mineralized trend which runs for at least 25 km, from at least as far south as the Property to Treaty glacier on the north. On the Property itself, there is a spatial association of alteration and contained mineralized zones with the north-trending Brucejack fault, a late-tectonic brittle structure which probably follows an older structure whose history dates back at least to the Early Jurassic. A number of lines of evidence, including facies changes within the local succession and variations in regional distribution and thickness of the host Hazelton Group rocks, support a long history for the Brucejack lineament and its precursor structure(s). It may, at least in part, have marked the boundary of a volcanic sub-basin, and judging by its general coincidence with mineralized and altered zones across the length of the Property, it may have helped control emplacement of the mineralization and alteration at Brucejack.

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 the Unuk River Formation or the lower members of the Betty White Formation.

Mineralization on the Property has previously been classified as an epithermal gold-silver-copper, low-sulphidation deposit (UBC deposit model No. H04). While there are certainly many features of the mineralized zones at Brucejack with characteristics of low-sulphidation deposits, such as vein mineralogy (e.g., adularia and acanthite) and the common stockwork veining and breccia-veins, which may suggest a lower temperature and shallower level of emplacement, other features, including the general lack of evidence for colliform banding and open-space filling, have been taken to suggest deeper levels of emplacement for the veining. Furthermore, other qualities of the zones, such as the relatively high molybdenum content at the Bridge Zone, and the fact that bulk tonnage style gold in some zones may be more closely correlated with disseminated anhedral pyrite than with veining, have been taken to suggest that at least some of the zones may be more closely allied to porphyry-style systems. Such a suggestion has some credence, particularly when one considers the common association of mineralization at Brucejack with hornblende feldspar phyric flows and fragmental rocks which are rich in groundmass potassium feldspar. Previously, these rocks have been interpreted as intrusive and therefore the mineralized zones were considered by, for example, McPherson et al. (1994), to be broadly "intrusive-related."

Until the property-scale geologic framework is better established, and until Pretium has a better understanding of the controls on the formation of the mineralized zones, Pretium is refraining from rigid adherence to, and acceptance of, a single deposit model. In the meantime, Pretium is taking steps to provide tighter constraints on the possibilities of deposit formation in the form of ongoing geochronological, petrographic, and whole rock geochemical studies, and is planning further similar studies, as well as stable isotopic and fluid inclusion work. One of the goals of such work will be to better understand what the components of the hydrothermal system were, and where the metals, water, and sulphur in the system, or systems, were derived from. The hope is that with this new information, Pretium will be better able to define a deposit model for the Brucejack mineralized zones, and will be better able to place it in a camp-scale geologic framework which will help guide future exploration at both the Brucejack and Snowfield Properties. The currently-held interpretation is that the higher-grade mineralization is structurally-controlled and was remobilized from pre-existing disseminated-style mineralization during later-stage deformation, perhaps related to development of the Skeena fold belt in mid-Cretaceous time.

1.6 RESOURCE ESTIMATE

The current resources as presented in this technical report are comprised of eight different zones on the Property; the West, Bridge, Low Grade Halo, Shore, Galena Hill, Gossan Hill, SG and Valley of Kings (VOK) Zones.

All mineral resources were reported against a 0.30 g/t gold equivalent (AuEq) cut-off, as constrained within the optimized pit shell. Resources for three different pit shells were defined as shown below in Table 1.1 to Table 1.3. In addition, an underground sensitivity to the resource estimate is presented in Table 1.4.

Table 1.1 Brucejack Estimated In-pit Mineral Resources Based on a Cut-Off Grade of 0.30 g/t AuEq

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	12.2	2.50	81.6	0.99	32.1
Indicated	293.0	1.26	10.5	11.91	99.3
M+I	305.3	1.31	13.4	12.89	131.5
Inferred	813.7	0.70	7.7	18.20	201.2

Notes: 1. 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 NI 43-101 technical report 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 and there has been insufficient exploration to define these inferred resources as 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. Mineral resources for the November 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$1,200/oz gold (71%) and US$22.00/oz silver (70%). The pit optimization utilized the following cost parameters: mining US$2.00/t, processing US$7.00/t and G&A US$1.25/t along with pit slopes of 45°.

Table 1.2 Brucejack 5.00 g/t AuEq In-pit Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	2.4	7.93	236.1	0.60	18.0
Indicated	6.9	19.99	60.9	4.46	13.6
M+I	9.3	16.92	105.6	5.06	31.6
Inferred	4.0	25.67	20.6	3.33	2.7

Notes: See notes in Table 1.1 above.

Table 1.3 Brucejack 1.25 g/t AuEq In-pit Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	9.3	3.08	102.20	0.92	30.6
Indicated	64.8	3.62	23.70	7.53	49.4
M+I	74.1	3.55	33.55	8.46	80.0
Inferred	78.5	2.68	16.30	6.76	41.2

Notes: See notes in Table 1.1 above.

Table 1.4 Combined West Zone and VOK 5.00 g/t AuEq Underground Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	2.4	7.29	241.2	0.57	18.9
Indicated	6.1	24.13	53.3	4.76	10.5
M+I	8.6	19.35	106.7	5.33	29.4
Inferred	4.0	25.73	22.0	3.29	2.8

Notes: See notes in Table 1.1 above.

1.7 Mining Operations

The Project is planned to be mined as an underground operation. The two lodes targeted for underground mining are the West Zone (WZ) and Valley of Kings (VOK) Zone.

The underground operation is based on contemporary rubber tired, diesel powered mobile equipment. Truck haulage of rock will be via a decline ramp system. Production will be achieved through implementation of the longhole open stoping (LHOS) method with a combination of rock and paste backfills.

The underground mine will operate for 26 years that includes two years of pre-production development, producing a total of 11.8 Mt of mineralized material from the WZ and VOK lodes. The nominal production rate is 1,500 t/d.

The underground mine design and inventory for both the WZ and VOK lodes are based on a net smelter return (NSR) cut-off of Cdn$180/t of mineralized material.

1.8 Metallurgical Test Work Review

Several testing programs were conducted to investigate the metallurgical performance of the mineralized samples. These programs included test work that were conducted in 2009 and early 2011 by Metallurgical Division at Inspectorate America Corp, as well as historical test work conducted between 1988 and 1990 for the feasibility study on the West Zone that was completed by Cominco Engineering Services Ltd (CESL). Currently, Pretium is conducting a comprehensive metallurgical test program to further assess the metallurgical performance of the mineralization to support the feasibility study. The ongoing test results are not included in this test work review.

The 2009 and 2011 preliminary metallurgical test work 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 test work 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 (BWI) of 16.0 kWh/t.

Further test work to optimize the flotation, gravity concentration and cyanidation conditions is ongoing.

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 Highway 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 17.1.

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-tonne 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 9.4 Mt of flotation tailings will be generated over the mine life. Approximately half of the flotation tailings will be paste backfilled to the underground, while the remaining 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 2.4 Mt of leach tailings will be deposited as a slurry in a fully double lined side-hill TSF located adjacent to the leach plant. A starter dam to store two years of production will be constructed initially to a height of 11 m above the downstream toe. The dam will be raised over the mine life to a height of 26 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 2.4 Mt of waste rock will be produced throughout the LOM. The majority of the waste rock will be deposited underground; however some waste rock will be deposited in Brucejack Lake. Any waste rock put into the lake will have a water cover 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 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 TSF, 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 a rehabilitated old road. Pretium has started construction on reopening the Newhawk access road and is in the process of building a new stretch of road, working up Scott Creek from the Bowser River and up Wildfire Creek from Highway 37. The road is expected to be completed by late 2012. As of the end of 2011, 12 km of new road was completed.

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.

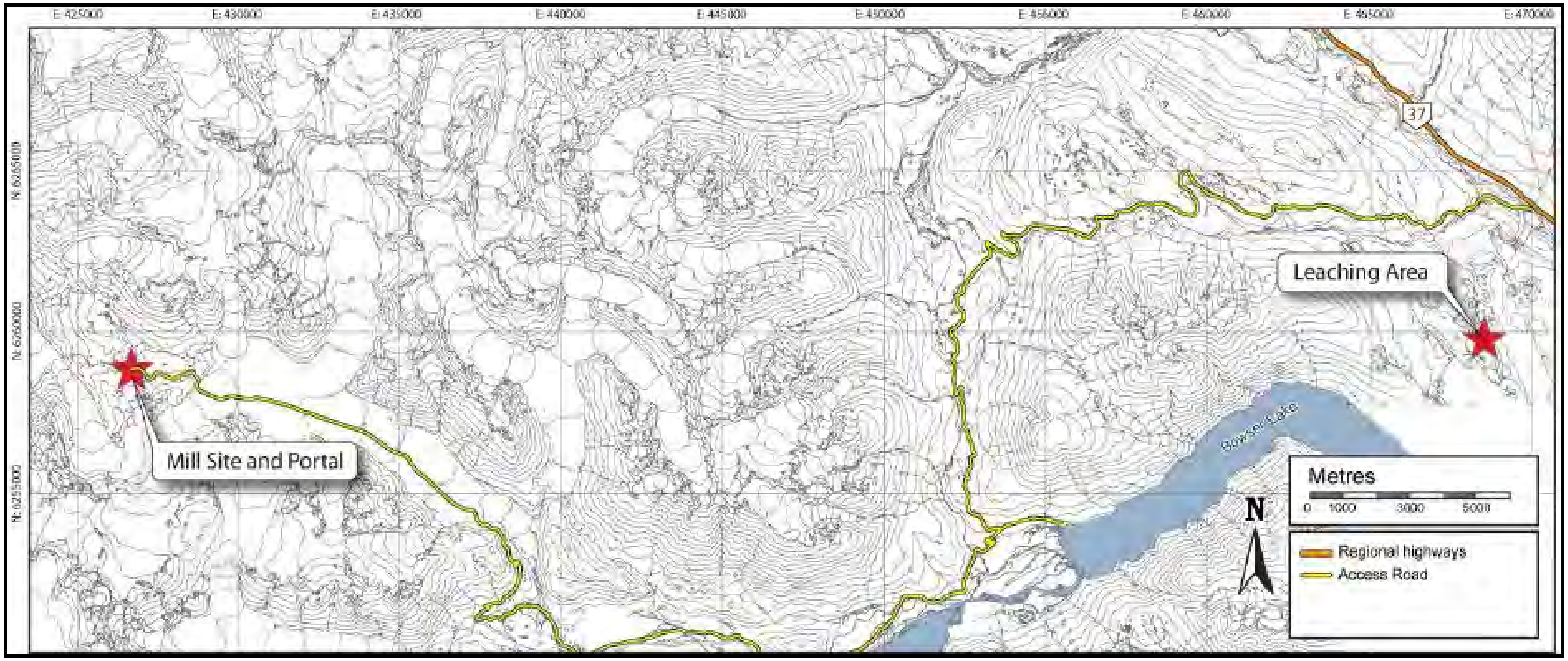
The tailings produced from the process plant at the mine site will be backfilled to underground stopes or deposited in Brucejack Lake, located approximately 1 km north of the process plant.

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 8 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 and mill site will be provided by an overhead power transmission line. Two diesel generators, each rated 1.5 MW will provide power for the leach plant. A heat recovery system will be installed to recover the heat generated from the diesel generators for building heating at the leach plant site.

1.14 CAPITAL COST ESTIMATE

The initial capital cost for the Project was estimated at US$436.26 million with an expected accuracy range of ±35%. The capital cost summary is shown in Table 1.5.

Table 1.5 Capital Cost Summary (Costs in US$)

Description	Total Cost
Direct Works	
Overall Site	11,355,122
Mine Underground (AMC)	114,344,196
Mine Surface Works (AMC)	13,892,222
Mine Site Process	35,411,786
Mine Site Utilities	56,635,194
Mine Site Buildings	16,412,370
Tailings	17,619,560
Temporary Facilities	3,917,160
Plant Mobile Equipment (Mine Site)	3,733,781
Leach Area	28,195,726
Leach Area Utilities	16,608,277
Leach Mine Buildings	6,506,949
Temporary Facilities	1,405,026
Plant Mobile Equipment (Leach Site)	2,278,693
Direct Works Subtotal	**328,316,062**
Indirect Works	
Indirect	58,212,020
Owner's Costs	11,904,000
Contingency	37,827,393
Indirect Works Subtotal	**107,943,413**
Total	**436,259,475**

1.15 OPERATING COST ESTIMATE

The total operating cost for the project is estimated at Cdn$170.90/t milled. The estimate includes operating costs for conventional underground mining, process, material rehandling, 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 268 personnel are projected for the operation, including 114 personnel for mining, 111 personnel for process, and 43 personnel for general management and surface services.

1.16 ECONOMIC EVALUATION

An economic evaluation of the Project was prepared by Tetra Tech based on a pre-tax financial model. For the 24-year LOM and 11.8 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 29.8% internal rate of return (IRR)
- 4.1-year payback on US$436.3 M capital
- US$2,262 million net present value (NPV) at 5% discount rate.

The post-tax economic evaluation of the Project was calculated after the applicable taxes were applied to the estimated annual cash-flows. The following post-tax financial parameters were calculated:

- 25.0% IRR
- 4.2-year payback on US$436.3 million capital
- US$1,454 million NPV at 5% discount rate.

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

- gold – US$1,100/oz
- silver – US$21.00/oz
- exchange rate – 0.93:1.00 (US$:Cdn$).

Metal revenues included in the Project cash flow model are based on the average metal production, as presented in Table 1.6.

Table 1.6 Brucejack Project Metal Production

	Average Annual Production		Total Production	
Metal	**Years 1 to 12**	**LOM**	**Years 1 to 12**	**LOM**
Gold ('000 oz)	325	287	3,899	6,878
Silver ('000 oz)	444	710	5,333	17,030

The PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be categorized as mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

Sensitivity analyses were carried out on the following parameters:

- gold price
- silver price
- exchange rate
- operating cost
- capital cost.

The analyses are presented graphically as financial outcomes in terms of NPV and IRR in Section 22 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 four to six 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 24.

1.18 RECOMMENDATIONS AND CONCLUSIONS

Based on the results of the updated PEA, it is recommended that Pretium should continue with the next phase of the project in order to identify opportunities and further assess viability of the project.

2.0 INTRODUCTION

The following report was prepared to provide a NI 43-101 compliant technical report and PEA update of the Project. Pretium has a 100% outright interest in the Property.

This report was prepared by Tetra Tech at the request of Mr. Ian Chang, Vice President, Project Development, of Pretium.

Pretium is a Vancouver, BC-based company trading on both the Toronto Stock Exchange (TSX) and the New York Stock Exchange (NYSE) under the symbol of "PVG", with its corporate office at:

1600-570 Granville Street
Vancouver, BC V6C 3P1
Telephone: 604-558-1784

This report is considered current as of February 20, 2012.

The purpose of the current report is to provide an independent technical report and PEA update of the Project, in conformance with the standards required by NI 43-101 and Form 43-101F.

Tetra Tech compiled this report based on work provided by the following independent consultants:

- P&E
- AMC
- Rescan
- BGC
- GeoSpark.

A summary of 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	Tetra Tech	John Huang, P.Eng.
3.0	Reliance on Other Experts	Tetra Tech	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 & Mineralization	P&E	Tracy Armstrong, P.Geo.
8.0	Deposit Types	P&E	Tracy Armstrong, P.Geo.
9.0	Exploration	P&E	Tracy Armstrong, P.Geo.
10.0	Drilling	P&E	Tracy Armstrong, P.Geo.
11.0	Sample Preparation, Analyses and Security	P&E	Tracy Armstrong, P.Geo.
12.0	Data Verification	P&E	Fred Brown, M.Sc. (Eng), CPG Pr.Sci.Nat.
	12.1.1 Pretium Quality Control	GeoSpark	Caroline Vallat, P.Geo.
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	John Huang, P.Eng.
14.0	Mineral Resource Estimate	P&E	Fred Brown, M.Sc. (Eng), CPG Pr.Sci.Nat.
15.0	Mineral Reserve Estimate	AMC	Peter Mokos, MAusIMM (CP)
16.0	Mining Methods	AMC	Peter Mokos, MAusIMM (CP)
17.0	Recovery Methods	Tetra Tech	John Huang, P.Eng.
18.0	Project Infrastructure		
	18.1 Infrastructure	Tetra Tech	John Huang, P.Eng.
	18.2 Waste and Water Management	BGC	-
	18.2.1 Brucejack Lake Mine Site Water Management	BGC	Hamish Weatherly, P.Geo.
	18.2.2 Brucejack Lake Waste Management	BGC	Lori-Ann Wilchek, P.Eng.
	18.2.3 Leach Plant Area Water Management	BGC	Hamish Weatherly, P.Geo.
	18.2.4 Leach Plant Tailings Storage Facility Design	BGC	Lori-Ann Wilchek, P.Eng.
	18.3 Underground Geotechnical and Hydrogeology Evaluations	BGC	Warren Newcomen, P.Eng.
19.0	Market Studies & Contracts	Tetra Tech	Hassan Ghaffari, P.Eng.
20.0	Environmental	Rescan	Pierre Pelletier, P.Eng.

table continues…

Report Section		Company	QP
21.0	Capital and Operating Costs		
	21.1 Capital Cost Estimate	Tetra Tech	Hassan Ghaffari, P.Eng.
	21.2 Operating Cost Estimate	Tetra Tech	John Huang, P.Eng.
	21.2.2 Mining Operating Cost	AMC	Peter Mokos, MAusIMM (CP)
22.0	Economic Analysis	Tetra Tech	Sabry Abdel Hafez, Ph.D., P.Eng.
23.0	Adjacent Properties	P&E	Tracy Armstrong, P.Geo.
24.0	Other Relevant Data	Tetra Tech	Hassan Ghaffari, P.Eng.
25.0	Interpretations and Conclusions	All	Sign off by section
26.0	Recommendations	All	Sign off by section
27.0	References	All	Sign off by section

2.1 Sources of Information

This report is based, in part, on internal company technical reports, and maps, published government reports, company letters and memoranda, and public information as listed in Section 27.0 at the conclusion of this report. Several sections from reports authored by other consultants have been directly quoted in this report, and are so indicated in the appropriate sections.

2.2 Units and Currency

Unless otherwise stated all units used in this report are metric. Gold and silver assay values are reported in grams per metric tonne (g/t), unless some other unit is specifically stated.

3.0 RELIANCE ON OTHER EXPERTS

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

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

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

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

Ms. Hannah Chow, CA, a Partner with Sadhra & Chow LLP, has been relied on for computation of the mineral taxes payable in the various years to be used for the post-tax economic evaluation of the project.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 TENURE

In 2010, pursuant to a purchase and sale agreement between Silver Standard (as the seller) and Pretium (as the buyer), Silver Standard sold to Pretium all of the issued shares of 0890693 BC Ltd., the owner of the Brucejack and Snowfield Projects.

4.2 DESCRIPTION AND LOCATION

The Property 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, 2022.

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

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

Table 4.1 Claims List for the Brucejack Property

Tenure No.	Tenure Type	Map No.	Owner	Pretium Interest	Status	In Good Standing To	Area (ha)
509223	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2022	428.62
509397	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2022	375.15
509400	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2022	178.63
509463	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2022	482.57
509464	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2022	1,144.53
509506	Mineral	104B	0890693 BC Ltd.	100%	Good	Jan. 31 2022	589.78
Total							**3,199.28**

There are no annual holding costs for any of the six mineral claims at this time, as the claims are paid up until January 31, 2022.

The royalties applicable to the Project are as follows:

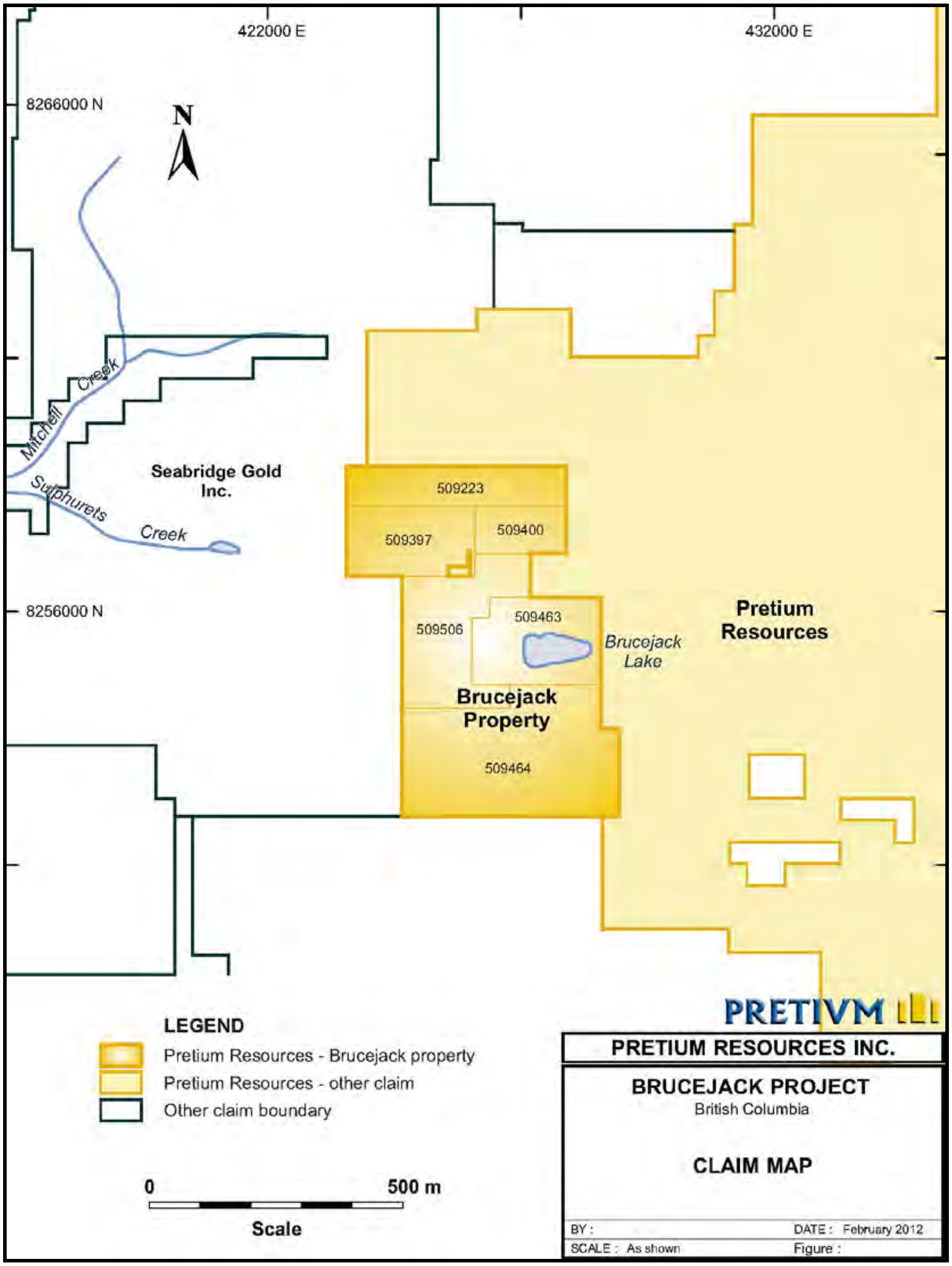
- "Royalty" means the amount payable by the Owner, calculated as 1.2% of the NSR, with the following exemptions:
  - gold: the first 503,386 oz produced from the Project
  - silver: the first 17,907,080 oz produced from the Project.

Figure 4.1 illustrates the six Property claims.

The majority of the Property falls within the boundaries of the Cassiar-Iskut-Stikine 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 Property



5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The Property is situated approximately 56°28′20″N latitude by 130°11′31″W longitude, a position approximately 950 km northwest of Vancouver, 65 km north-northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine. The Property coordinates used in this report are located relative to the NAD83 Universal Transverse Mercator (UTM) coordinate system.

The 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 1,200 masl. Elevations within the Property area range from 1,366 m along Brucejack Lake to 1,650 masl at the Bridge Zone. However, within several areas of the Property, the relief is relatively low to moderate.

The Property area is easily accessible with the use of a chartered helicopter from the town of Stewart, or seasonally from the settlement of Bell II. The flight time from Stewart is approximately 30 min and slightly less from Bell II; however, Stewart has the advantage of having an established year-round helicopter base. Pretium has started construction on re-opening the Newhawk access road. Originally the Newhawk access was by barge over Bowser Lake, then by truck to camp. Pretium is rehabilitating the old road along the Bowser River and up the Knipple Glacier, and is in the process of building a new stretch of road, working up Scott Creek from the Bowser River and up Wildfire Creek from Highway 37. The road is expected to be completed by late 2012. As of the end of 2011, 12 km of new road was completed (see Figure 5.1).

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

5.2 CLIMATE AND PHYSIOGRAPHY

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

Precipitation is high with heavy snowfall accumulations ranging from 10 to 15 m at higher elevations and 2 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 1,200 m elevation. Sparse fir, spruce, and alder grow along the valley bottoms with only scrub alpine spruce, juniper, alpine grass, moss, and heather covering the steep valley walls. The Property, at an elevation above 1,300 m, has only sparse mosses along drainages. Rocky glacial moraine and polished glacial-striated outcrops dominate the terrain above tree line.

5.3 INFRASTRUCTURE AND LOCAL RESOURCES

The Property lies immediately east of Seabridge Gold Inc.'s (Seabridge) Kerr-Sulphurets-Mitchell (KSM) Project. The Snowfield Zone may be influenced by Seabridge's future access plans for that area, as Seabridge discussed in its updated prefeasibility study (PFS) dated May 2, 2011, however the Project will not. The updated PFS was prepared by Tetra Tech. 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.

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

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

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

A high voltage power line running parallel with Highway 37 is planned for construction (www.highway37.com). The plan calls for the new 287 kV line to extend from the community of Terrace to the beginning of the Galore Creek access road at Bob Quinn Lake providing access for the Property to the BC Hydro electric grid (Figure 5.2). The final capacity of this transmission line has yet to be determined and may be increased due to demand.

Figure 5.1 New Access to the Project (In Progress)

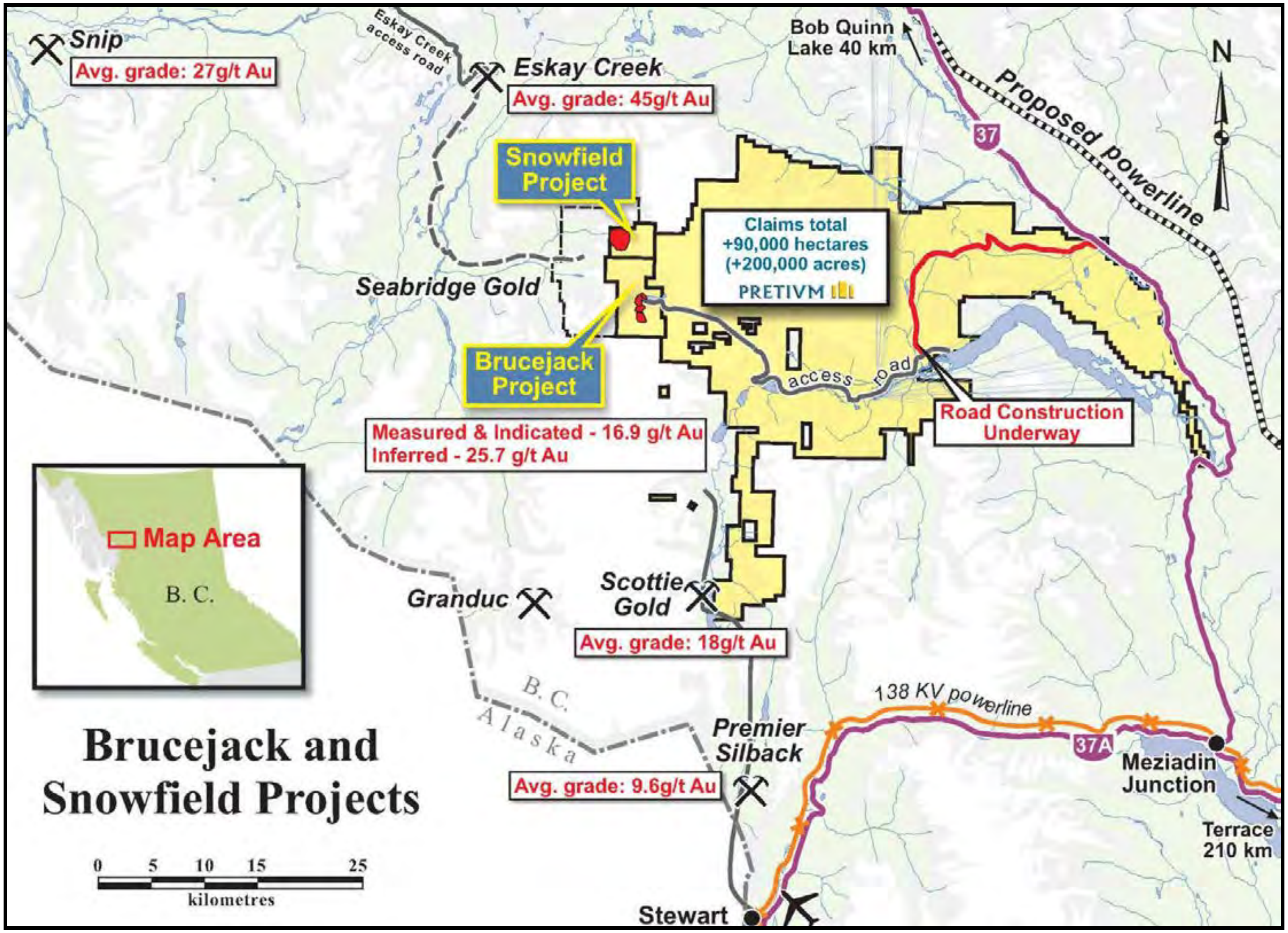
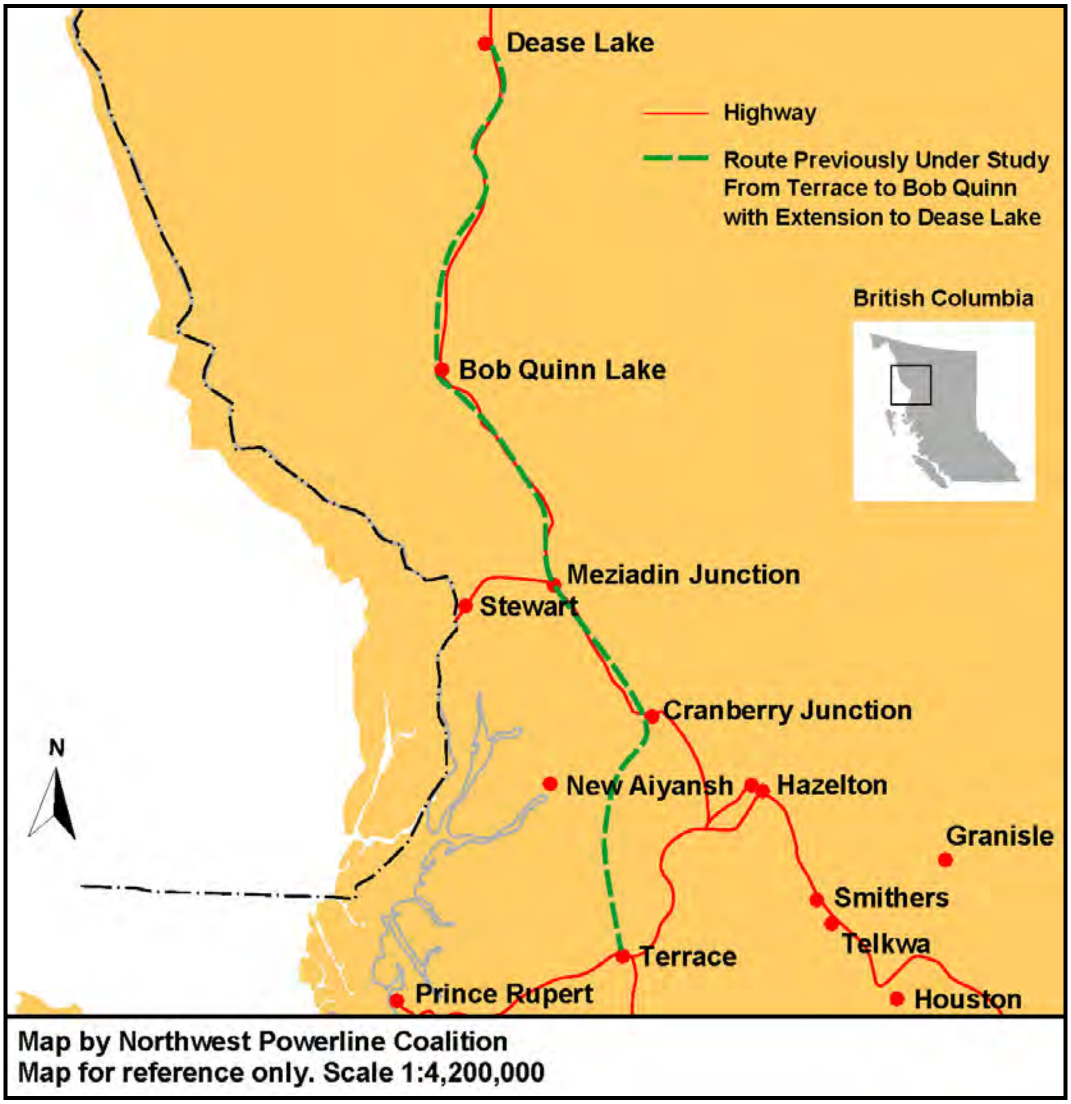


Figure 5.2 Proposed High Voltage Northwest Transmission Line



Source: www.highway37.com

6.0 HISTORY

The Property and the surrounding region have a history rich in exploration for precious and base metals dating back to the late 1800s. This section describes the mineral exploration, including the historical drilling carried out prior to Pretium's acquisition of Brucejack, and post-acquisition. The historical data have been summarized mostly from various assessment reports available through the BC Ministry of Energy, Mines and Petroleum Resources.

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 made 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 Snowfield, 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 JV (the Newcana JV). The Newcana JV completed work on the Snowfield, 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 Cdn$21 million developing 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, JV 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. Newhawk purchased Granduc's interest in the Snowfield Property in early 1992.
- 1991 – Six holes were drilled at the Shore Zone, totalling 1,200 m, to test its continuity and to determine its relationship to the West and R-8 zones. Results varied from 37 g/t Au over 1.5 m to 13 g/t Au over 4.9 m (www.infomine.com).
- 1994 – Exploration in the Brucejack area consisted of detailed mapping and sampling in the vicinity of the Gossan Hill Zone, and 7,352 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 ten of the best deposits (including the West Zone).
- 1996 – Granduc merged with Black Hawk to form Black Hawk Mining Inc.
- 1997-1998 – No exploration or development work was carried out on the Property (Budinski et al., 2001).
- 1999 – Silver Standard acquired Newhawk and with it, Newhawk's 60% interest and control of the Property (www.infomine.com).
- 2001 – Silver Standard entered into an agreement with Black Hawk whereby Silver Standard acquired Black Hawk's 40% direct interest in the Property, resulting in 100% interest in the Property.
- 1999 to 2008 – No exploration or development work was carried out on the Property 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 (underground vein location plan map) and Figure 6.2 (cross section map).

Figure 6.1 West Zone Vein Location Plan

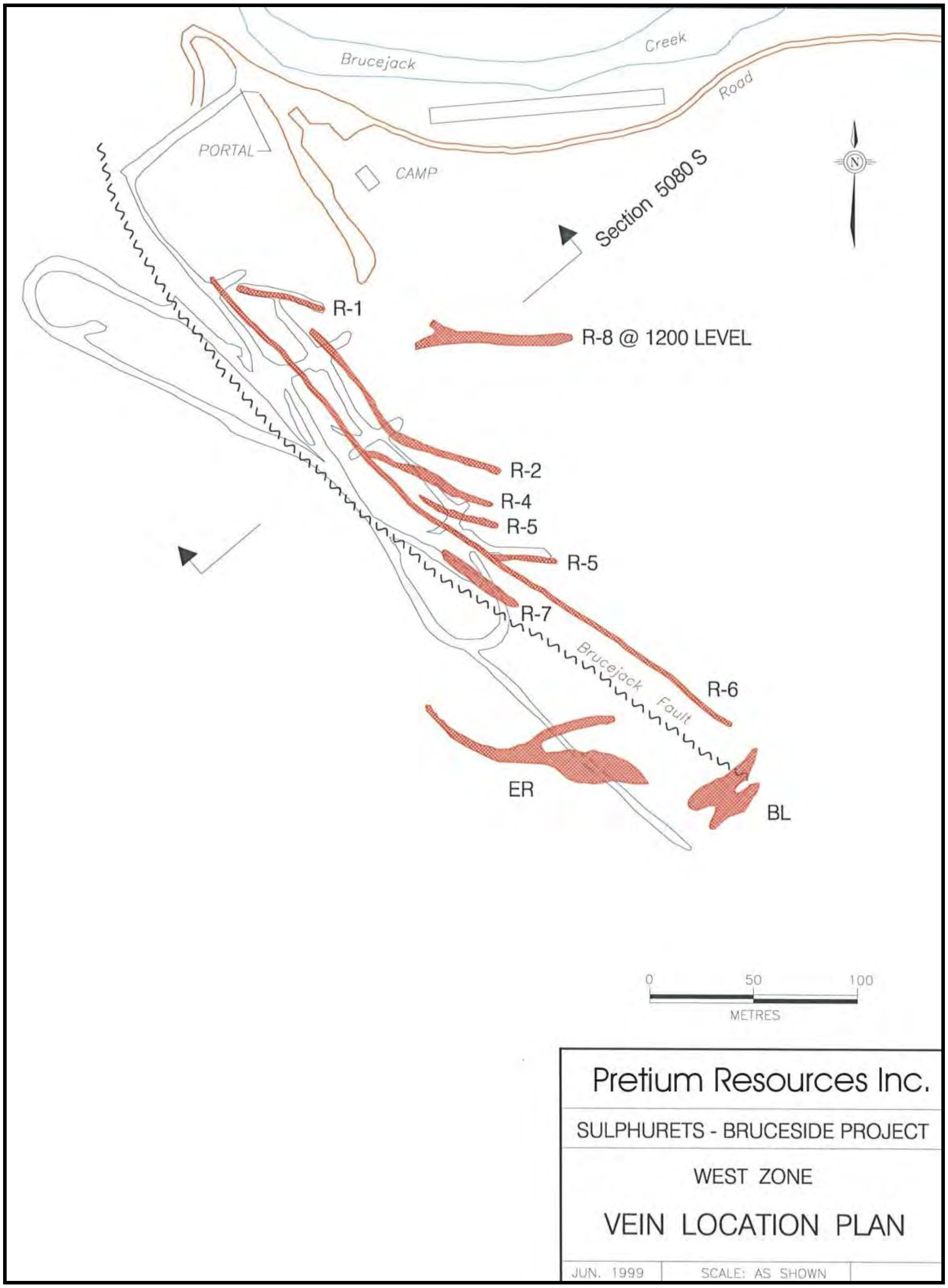
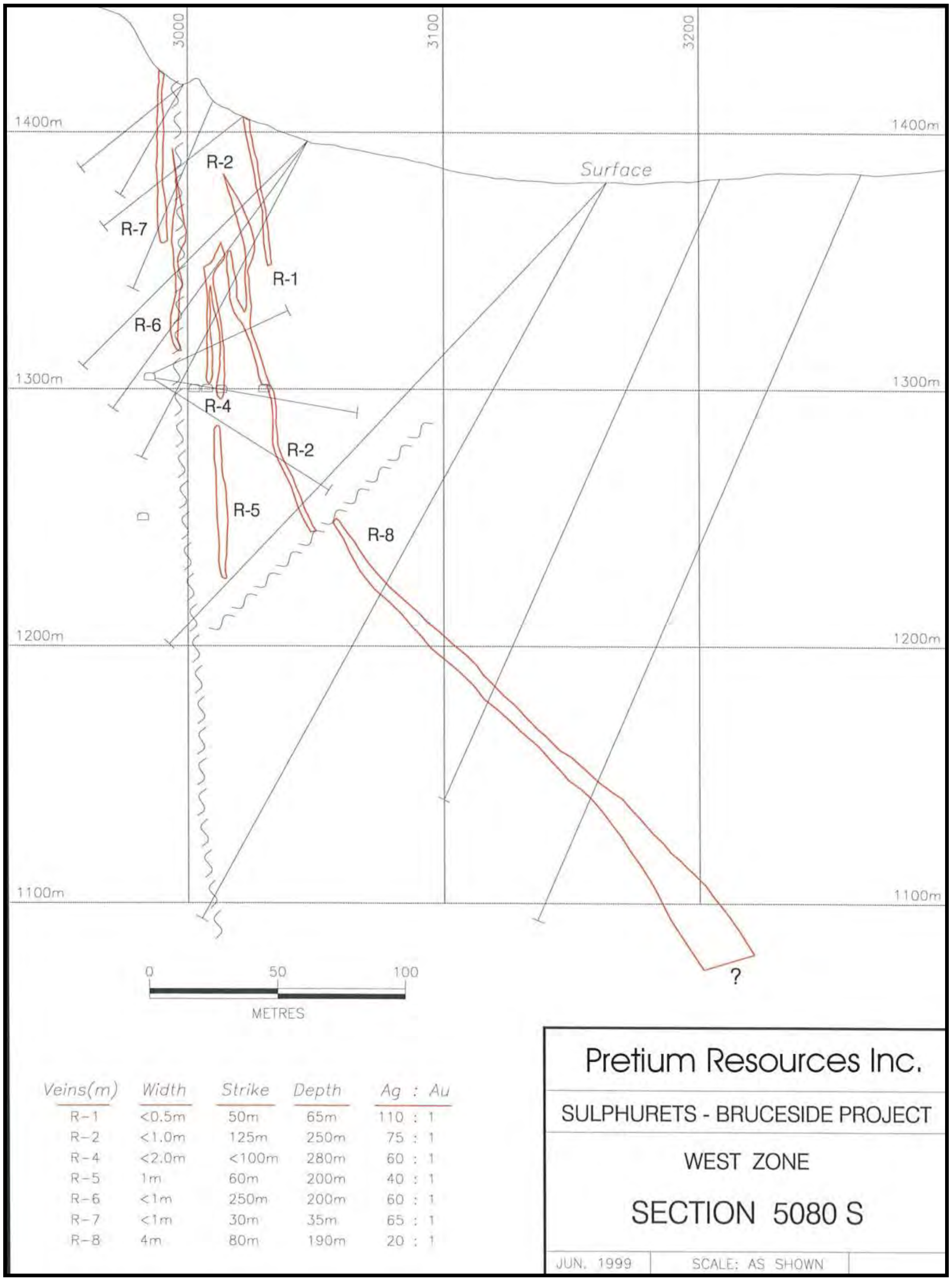


Figure 6.2 West Zone Section 5080 S



6.1 RECENT WORK COMPLETED BY SILVER STANDARD

In 2009, Silver Standard began work on the Property, 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 Property field program, Silver Standard collected a total of 1,940 drill core samples from 25 historical drillholes stored on site 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 (FA) with atomic absorption (AA) finish) as well as 33 other elements by using four acid digest with 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.

Field work undertaken throughout the 2009 program included the drilling of 2,739 rock-chip and channel samples from surface outcrops. This sampling work was mostly done at target areas that were drilled by Silver Standard in 2009, with samples generally collected along north-south oriented lines that corresponded to the surface traces of some of the 2009 drillholes. 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.

A total of 17,846 m of diamond drilling were completed in 37 holes during the 2009 field season.

In 2010, a total of 33,400 m of diamond drilling was completed in 72 holes.

The West Zone ramp was partially dewatered in late 2011 and early 2012. A geotechnical mapping program and updated survey was completed on the dewatered portion of the mine.

6.2 PREVIOUS FEASIBILITY STUDIES AT THE PROPERTY

Corona Corp. (Corona) completed a feasibility study on a proposed underground mine with decline access for the Sulphurets Project (West and R-8 Zones only) in 1990. Total operating costs of Cdn$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 gold and US$5/oz silver. The study concluded that higher metal prices must be realized before a production decision could be made.

The reader is cautioned that the above-mentioned 1990 Corona Sulphurets Project Feasibility Study is no longer relevant, is not NI 43-101 compliant and should not be relied upon.

6.2.1 PRELIMINARY ECONOMIC ASSESSMENT 2010

Silver Standard commissioned Tetra Tech to complete a PEA on the combined resources of the Brucejack and Snowfield Projects in 2010.

The following consultants were commissioned to complete the component studies for the NI 43-101 technical report:

- Tetra Tech: 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 PEA, it was recommended that Silver Standard continue with the next study in order to identify opportunities and further assess viability of the Property. This report was re-issued for Pretium in October 2010, however the report is no longer current.

6.2.2 PRELIMINARY ECONOMIC ASSESSMENT 2011 BRUCEJACK PROJECT

Pretium commissioned Tetra Tech to complete a PEA on the high-grade gold and silver resources at the Project as a "stand-alone" project, and results were made public in June 2011. The following consultants were commissioned to complete the component studies for the NI 43-101 technical report and PEA:

- Tetra Tech: processing, infrastructure, capital cost estimate, processing, operating cost estimate, and financial analysis
- AMC: mining including mine capital and operating cost estimates
- P&E: mineral resource estimate
- Rescan: environmental aspects, waste and water treatment
- BGC: tailings impoundment facility, waste rock and water management, site wide groundwater studies, and geotechnical design for on site facilities.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY AND METALLOGENIC SETTING

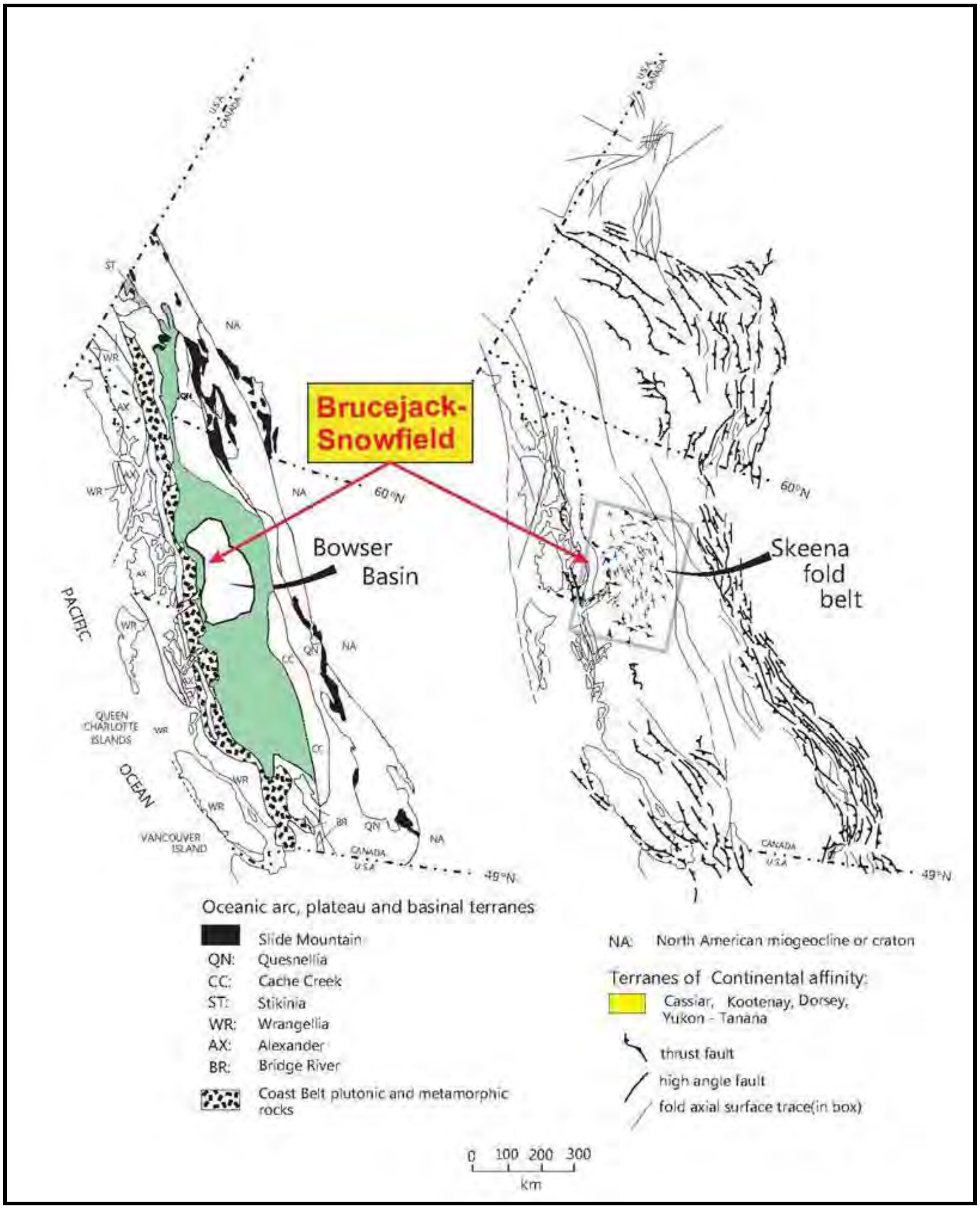
The Property is largely underlain by Lower Jurassic rocks of western Stikine terrane, or Stikinia, an oceanic island arc terrane consisting of mid-Paleozoic to Middle Jurassic rocks which underlies much of western BC (Figure 7.1). Stikinia may have been accreted to the western margin of North America as early as the late Middle Jurassic, and it was likely consolidated with rocks of the North American margin, as well as with rocks of the outboard Insular terrane (Wrangellia and Alexander terranes) in latest Jurassic to mid-Cretaceous time.

The rocks of Stikinia in the northern Cordillera were deformed in two major contractional events. The younger event coincided with the “consolidation” event and resulted in the formation of the SFB in northern BC. The SFB is best displayed immediately to the east of the Property in Middle Jurassic to Lower Cretaceous Bowser Basin/Bowser Lake Group strata which overlie the host rocks to mineralization at Brucejack (Figure 7.1 and Figure 7.2). The formation of the SFB likely post-dated deposition of the host rocks and the formation of the mineralized zones on the Property. The other, older deformation event pre-dated deposition of the host rocks and occurred in latest Triassic to earliest Jurassic time. This earlier event is manifested in the Upper Triassic and older(?) rocks along the western-most margin of the Property. These rocks are very tightly folded and are overlain by some of the immediate Lower Jurassic conglomeratic host rocks to mineralization along a profound regional unconformity which is commonly marked by the presence of polymict conglomerate.

The Sulphurets mining camp and the Property lie astride the eastern margin of the core of the McTagg anticlinorium, which is a major north-trending mid-Cretaceous structural culmination in the western SFB (Figure 7.2 and Figure 7.3). Coincident with the core of the anticlinorium is a prominent and very well mineralized trend which runs for at least 25 km, from at least as far south as the Property to Treaty glacier on the north. On the Property itself, there is a spatial association of alteration and contained mineralized zones with the north-trending Brucejack fault, a late-tectonic brittle structure which probably follows an older structure whose history dates back at least to the Early Jurassic. A number of lines of evidence, including facies changes within the local succession and variations in regional distribution and thickness of the host Hazelton Group rocks, support a long history for the Brucejack lineament and its precursor structure(s). It may, at least in part, have marked the boundary of a volcanic sub-basin, and judging by its general coincidence with mineralized and altered zones across the length of the Property, it may have helped control emplacement of the mineralization and alteration at Brucejack.

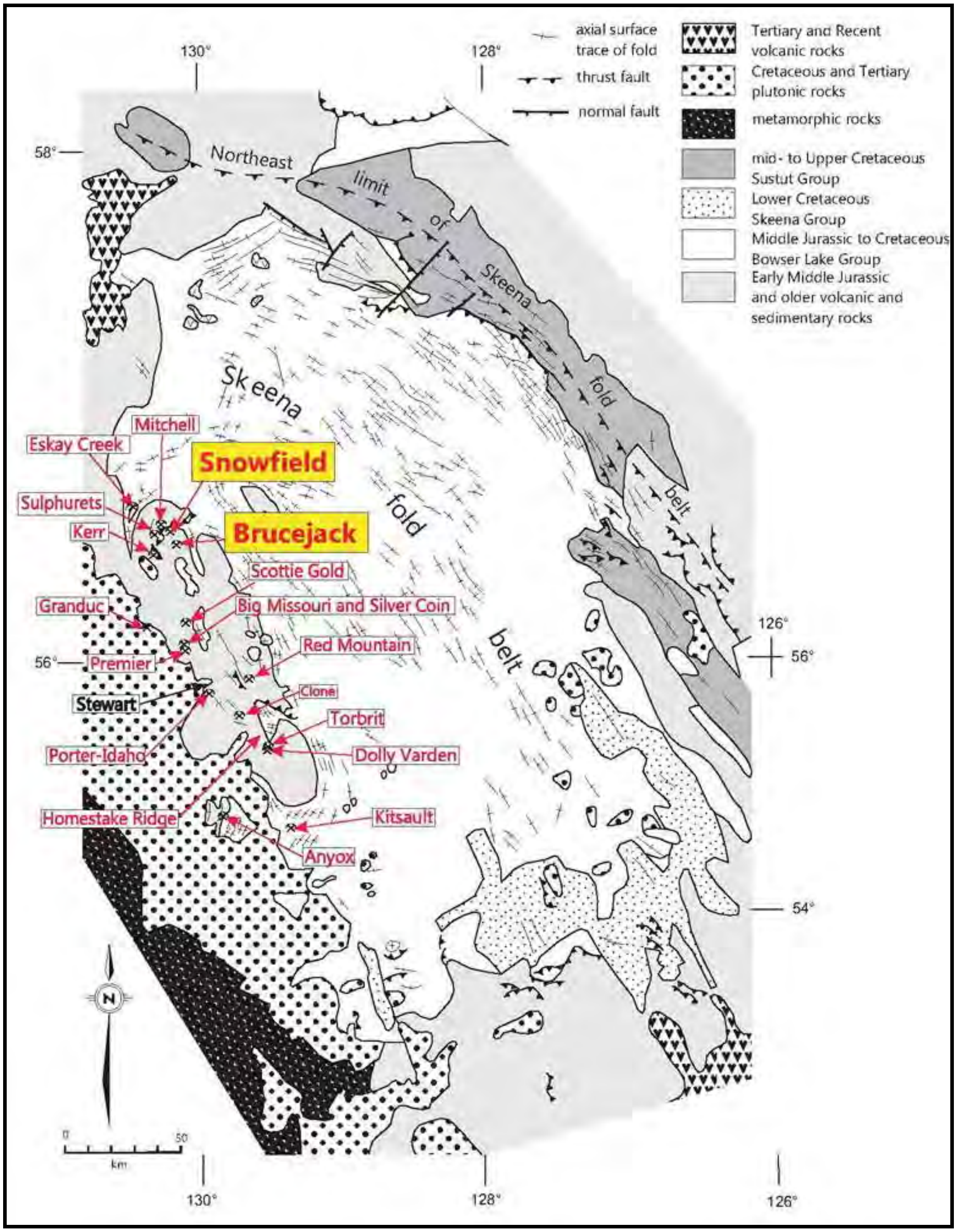
The north-west part of Stikinia, (in particular Lower Jurassic to earliest Middle Jurassic Hazelton Group volcanic rocks and sedimentary rocks), and related mainly Early Jurassic plutons, represent perhaps the most metallogenetically well-endowed assemblage in BC. Not only does it include the Brucejack (31 Moz) and Snowfield (35 Moz) Properties, but it also includes nearby former producers such as Eskay Creek, Silbak-Premier, Big Missouri, Dolly Varden, Torbrit Silver, Granduc, and Anyox (Figure 7.3 and Figure 7.4). In addition, nearby properties host significant precious and base metal resources (e.g., KSM and Red Mountain deposits) as well as a number of high-potential occurrences (e.g., Bravo's Homestake Ridge, Jayden's Silver Coin, Ascot's Big Missouri, and Teton Resources' Clone and Tennyson Properties). The KSM deposits, along with the Snowfield and Brucejack deposits together comprise what is commonly referred to as the Sulphurets mining camp.

Figure 7.1 Tectonic Setting of Brucejack and Snowfield Properties in the North-west Cordillera



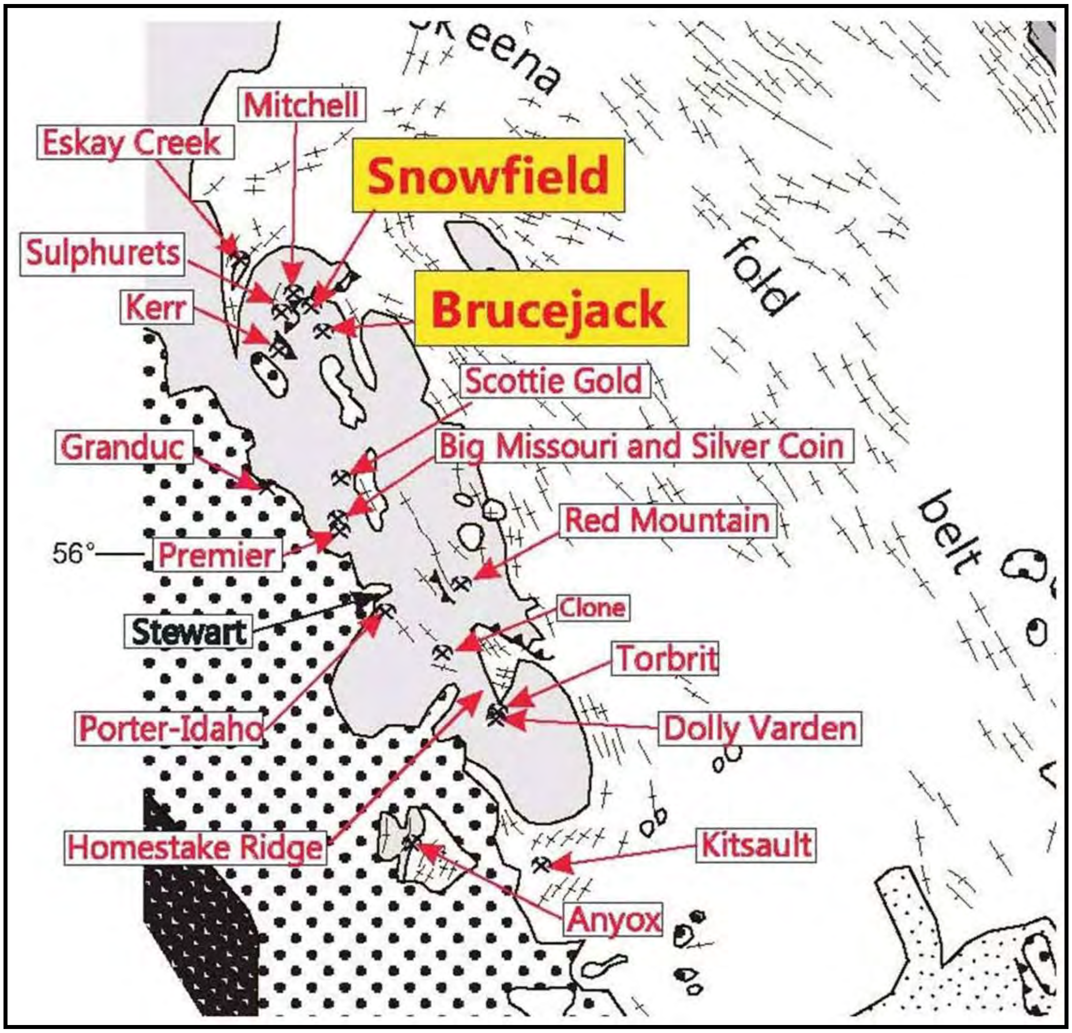
Note: Shows location of project area in north-central Stikine terrane as well as nearby Bowser Basin (left) and latest Jurassic to mid-Cretaceous SFB (right).

Figure 7.2 Regional Structural and Stratigraphic Setting of the Property and Sulphurets Mining Camp in Northwest BC



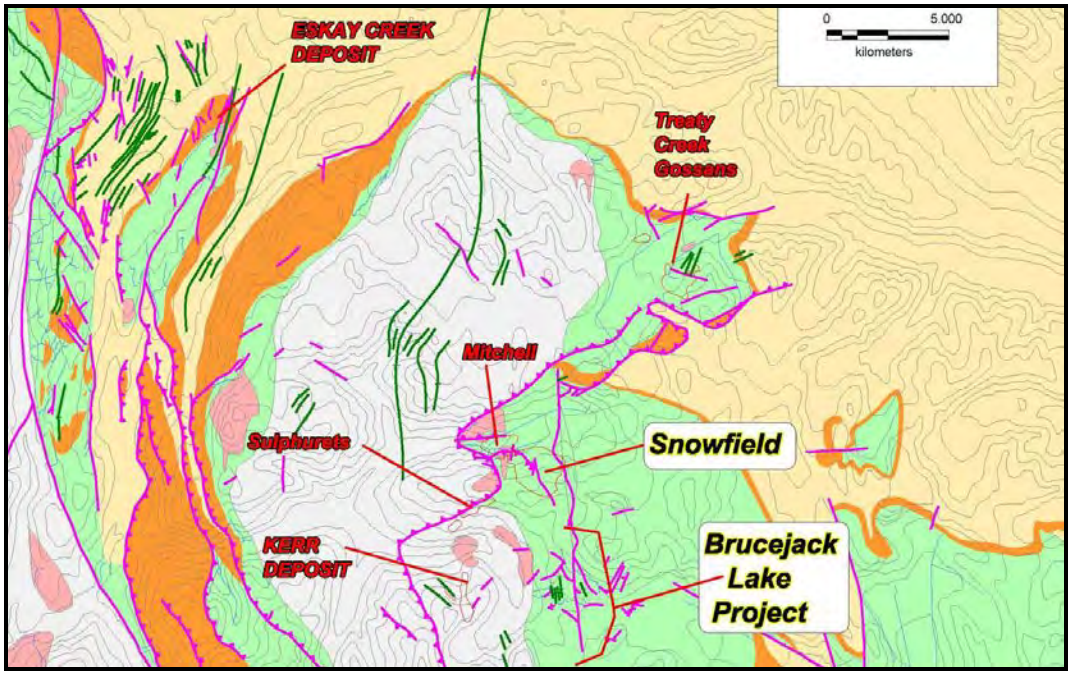
Note: Shows significant past-producing mines as well as selected advanced exploration projects.

Figure 7.3 Detail of Regional Structural and Stratigraphic Setting of the Property and Sulphurets Mining Camp within the Western SFB



Note: Showing earliest Middle Jurassic and older Stikine terrane strata within structural culminations (grey), with overlying earliest Middle Jurassic to mid-Cretaceous clastic rocks of Bowser Basin in white; heavy stipple indicates predominantly granitic rocks of Coast Belt.

Figure 7.4 Northern McTagg Anticlinorium



Note: Shows location of Sulphurets Mining Camp; Upper Triassic Stuhini Group strata in grey, Lower to Middle Jurassic Hazelton Group strata in green (uppermost Lower Jurassic to lowermost Middle Jurassic Salmon River Formation of the Hazelton Group in orange), Middle Jurassic to Lower Cretaceous Bowser Lake Group in salmon, with intrusive rocks in pink.

7.2 LOCAL GEOLOGY

7.2.1 *GEOLOGY OF THE SULPHURETS CAMP*

Following the pioneering 1960s and 1970s thesis and regional mapping work of Kirkham (1963), and Grove (1986), a number of more detailed (1:50,000 to 1:5,000 scale) mapping projects were undertaken in the Sulphurets Camp in the late 1980s and early 1990s. This work accompanied renewed precious metals exploration in the Iskut-Stewart region which was in part spurred-on by the development of the Snip gold mine (approximately 60 km west-northwest) and by the discovery of the Eskay Creek gold-silver deposit (approximately 25 km northwest). The mapping, along with earlier work, has been summarized in papers by Kirkham and Margolis (1995), Henderson et al. (1992), Britton and Alldrick (1988), and Davies et al. (1994), and was also detailed in a number of related maps (e.g., Alldrick and Britton (1989, 1991), Kirkham (1991, 1992), Lewis et al. 2001), from which Figure 7.5 was largely compiled.

7.2.2 STRATIGRAPHIC SETTING AND MAJOR MINERAL DEPOSITS

While the maps and results of these studies differ as they pertain to the Sulphurets Camp, all show that the camp, its regional-scale alteration system, and its significant mineral deposits, are underlain almost exclusively by mid-Mesozoic rocks. They range in age from Late Triassic to Middle Jurassic, and include common volcanic and related dominantly clastic sedimentary rocks, as well as subordinate intrusive rocks. Mapping shows that the western and northern parts of the Sulphurets Camp are largely underlain by older, predominantly Late Triassic fine-grained and well-stratified sedimentary rocks, along with subordinate mafic volcanic rocks, all of which make up the Stuhini Group. Furthermore, this area includes the majority of the plutonic rocks in the camp. Plutonic rocks generally intrude Stuhini Group rocks, were referred to by Kirkham (1963) as the Mitchell intrusions, and where their ages are known, are predominantly Early Jurassic (for summary, see Kirkham and Margolis 1995). The eastern-most part of the area is underlain by Middle Jurassic and younger clastic rocks of the upper Salmon River Formation and Bowser Lake Groups. These rocks help define the eastern limb of the McTagg anticlinorium and where they conformably overlie a medial belt of mainly Lower Jurassic, predominantly volcanic and subordinate sedimentary rocks of the Hazelton Group.

Within the Sulphurets area, Hazelton Group volcanic rocks unconformably overlie the older Stuhini Group and consist mainly of flows and related fragmental rocks (largely pyroclastic) of intermediate composition (generally andesitic), although mafic and felsic rocks have also been mapped. Both Britton and Alldrick (1988) and Kirkham and Margolis (1995) show the Brucejack Property as being underlain almost entirely by volcanic and related sedimentary rocks of Grove's (1986) informally defined Unuk River and Betty Creek Formations, while Davies et al. (1994), who mapped in more detail (1:5,000 versus 1:50,000 scale) and whose mapping Newhawk (e.g., MacPherson et al., 1994) adopted, interpreted most of the more massive rocks common across the Property as intrusive. Lithologies mapped during the present program appear to correlate reasonably well with those of the Unuk River and Betty Creek Formations as described by Britton and Alldrick (1988), however, for the time being Pretium has chosen not to assign these newly mapped units with specific regionally-mapped formations. This is in part because this latest mapping suggests that some of the rocks on the Property appear to be lateral and time-stratigraphic equivalents of one another, which will require detailed study to properly assess.

Hazelton Group rocks are the predominant hosts to mineralization and alteration on Pretium's Brucejack and Snowfield Properties, although to the west and north, Stuhini Group rocks and the Mitchell intrusions are the main hosts to porphyry-style mineralization at the Kerr (copper-gold), Sulphurets (gold), and Mitchell (gold-copper-molybdenum) deposits, all belonging to Seabridge Gold. At Pretium's Snowfield deposit, the protolith of the highly altered hosts to disseminated gold mineralization are interpreted to be intermediate to mafic(?) volcanic and related volcaniclastic rocks of the lower Hazelton Group.

In the northern Sulphurets Camp, in the immediate vicinity of the Sulphurets, Mitchell, and Snowfield deposits, the host rocks to mineralization are intensely altered and there is considerable uncertainty with regard to composition and correlation. Britton and Alldrick (1988) show the area as being underlain by rocks of their lowermost Hazelton Group, including common sedimentary rocks, but show few contacts. Lewis et al. (2001) also show very generalized map units of the lowermost Hazelton Group, which they themselves question. Margolis (1993) has an even more generalized geologic map. One thing which is clear, however, is that a fair proportion of the area in the vicinity of the deposits is considered to be underlain by sedimentary rocks of the lowermost Hazelton Group (Jack Formation), or by the lowermost volcanic units, which they interpret to be mafic in composition. In addition to the lowermost Hazelton Group rocks, a number of intrusions are also shown on most maps of the area, although the details of contacts, composition, and even location do not commonly correspond from map-to-map. Using some of this information, and including more detailed Newhawk mapping in the area surrounding the Snowfield deposit, Pretium produced the compilation map shown in Figure 7.5.

One significant thing this map illustrates is that along the trend of the Brucejack fault, from the south end of the Property north to the Snowfield Property and beyond, one never appears to be very far stratigraphically above or below the Stuhini Group-Hazelton Group unconformity. Therefore, like the Brucejack fault itself, the unconformity is co-spatial with the regional-scale gossanous Sulphurets alteration system, and with most of the camp's more significant mineral resources.

The Brucejack fault is a very obvious topographic lineament. In spite of this, there is considerable discussion about the timing of potential movement on the fault. A much thicker section of Hazelton Group rocks on the east side of the fault suggests considerable east-side-down displacement, which may be interpreted as reflecting post-depositional displacement. Alternately, the presence of the overlying uppermost Salmon River Formation fine clastic units suggests that much of the movement was probably syn-volcanic, and that any later movement on what is the present trace (the lineament itself) was not large.

At Brucejack, the alteration and most of the mineralization and alteration are concentrated in the north-south arcuate band of gossanous rock, which is part of the Sulphurets alteration system. The rocks which host much of the alteration are part of lowermost Hazelton Group, which suggests that the unconformity with the underlying rocks of the Stuhini Group may have helped focus or control the emplacement of alteration and mineralization. It is also possible that some of the more porous and permeable rocks close to the unconformity, such as the conglomerates overlying it, may have helped focus or channel the alteration, and to some degree the mineralization. All of this has clearly been modified by later deformation and/or structures.

The unconformity and the rocks which overlie it are relatively close to the surface the entire distance between the Brucejack and the Snowfield deposits, which is a distance of nearly 10 km. To the east, these rocks are commonly overlain by other, less readily altered lithologies, suggesting that very good potential exists for locating blind deposits in that direction, as well in places along the Brucejack-Snowfield trend where less altered rocks may overlie more favourable stratigraphy.

7.2.3 *Structural Setting and Metamorphism*

Perhaps the most significant map-scale structural features in the Sulphurets region, aside from the McTagg anticlinorium itself, are several south-east- to east-directed thrust faults and related overturned folds which occur to the immediate north and north-west of the Property, and which affect the Sulphurets, Mitchell and Snowfield deposits, (Figure 7.5). Kirkham and Margolis (1995) noted that the faults involve overlying rocks of the Salmon River Formation and Bowser Lake Group and that they are likely part of the latest Jurassic to Middle Cretaceous SFB, as are the McTagg and other major structural culminations (e.g., Oweegee dome) which expose older, pre-Salmon River Formation and Bowser Lake Group rocks in the region. However, as was pointed-out above, it should be noted that the age and nature of the older rocks within the culminations and on either side of these structures are also, in general, quite different. To the west and north-west are Upper Triassic predominantly sedimentary rocks and contained plutons, while to the east and south-east the rocks are almost exclusively volcanic and related sedimentary rocks of the Lower Jurassic Hazelton Group. Because rocks of the Salmon River Formation, particularly in its upper part, represent a relatively quiescent, deeper-water depositional environment and represent an excellent and biochronologically relatively well-controlled datum, this suggests that the thrust faults, and probably also other structures such as the Brucejack fault, may represent basement structures reactivated in part, during formation of the SFB.

As is evident in Figure 7.6, to the south-east and east of the thrust faults, and to a certain extent in their immediate hanging wall(s), foliation in rocks of the Hazelton and Stuhini Groups is generally east-west trending. This is in contrast to the orientation of structural fabrics (particularly foliation) still farther to the east and, in particular, to the west, which have a very pronounced, more northerly (north-north-west to north-east) trend that is much more in accordance with typical Cordilleran structural trends. Given the scale of this "foliation dip domain," the zone of crudely east-west trending pervasive foliation, which persists for at least 10 km from south to north and which is very much in evidence across the length of the Property, is clearly a regional-scale structural attribute. On the outcrop-scale, and as noted in the summary by Kirkham and Margolis (1995), as well as by others, one of the more salient structural features of rocks in the Sulphurets area is this penetrative foliation or cleavage, which is particularly well-developed in the most intensely altered rocks which are now phyllosilicate-rich. Kirkham and Margolis (1995) noted that the rocks have not only been foliated, but they have also been folded, faulted, and overprinted by lower greenschist facies regional metamorphism. They also pointed out that almost all structural features clearly postdate mineralization and alteration, a characteristic which has also been noted clearly by other geologists mapping in the

Sulphurets Camp, irrespective of the scale of mapping (e.g., Roach and MacDonald 1992, Davies et al. 1994).

As mentioned in the Regional Geologic and Metallogenic Setting section above, Upper Triassic rocks of the Stuhini Group, exposed mainly within the eastern half of the northerly plunging McTagg anticlinorium, were deformed prior to deposition of Lower Jurassic clastic and volcanic strata of the Hazelton Group. These relationships are best displayed to the north-west of the Property in the Jack glacier area (type area of the informally defined Jack Formation; Henderson et al., 1992). The contact between tightly folded Upper Triassic rocks and unconformably overlying Lower Jurassic Hazelton Group rocks runs the length of the Property, and appears to be well-exposed along the banks of Brucejack Creek (Henderson et al. 1992). The overlying rocks of the Hazelton Group, which on all maps of that area are characterized by steep dips and the presence of a penetrative foliation, have themselves undergone significant deformation. The nature of that folding on the southern part of the Property is described in Section 7.3.

7.2.4 Brucejack Fault and Related Late Brittle Structures

The well-defined north-striking topographic lineament which marks the trace of the Brucejack fault has long been of interest to regional geologists and explorationists alike. The Brucejack fault also truncates alteration zones, veins, and vein stockwork systems. To the immediate west of the VOK Zone, post-mineral mafic dykes (probably of Tertiary age), which occur within and in the immediate vicinity of the fault, commonly display brittle fracturing. While the surface trace of the fault is obvious, and while it's brittle nature and oblique dextral, east-side-down displacement are generally agreed upon, the magnitude of that displacement remains uncertain.

Kirkham and Margolis (1995) noted that offset contacts and structural fabrics north of the Mitchell glacier suggested an east-side-down dextral displacement of greater than 500 m. They also noted that other authors considered that displacement near Brucejack Lake was also dextral, although with displacement probably less than 100 m in magnitude, and with an uncertain slip direction. Davies et al. (1994) noted that exposures of the fault to the north-west of Brucejack Lake preserved slickenside and clast elongation lineations in a steeply west-dipping surface indicated at least a late period of dip-slip offset. They further postulated that if the linear features represented the net slip direction, then a displacement of 700 to 800 m of reverse (west-side-up) motion was constrained from offset stratigraphic (and faulted?) contacts.

One kilometre north-west of Brucejack Lake, contacts show 200 to 300 m of strike separation, and displacement is likely less than in the north. In contrast, Britton and Alldrick (1988) considered that displacement on the Brucejack fault was on the order of tens of metres.

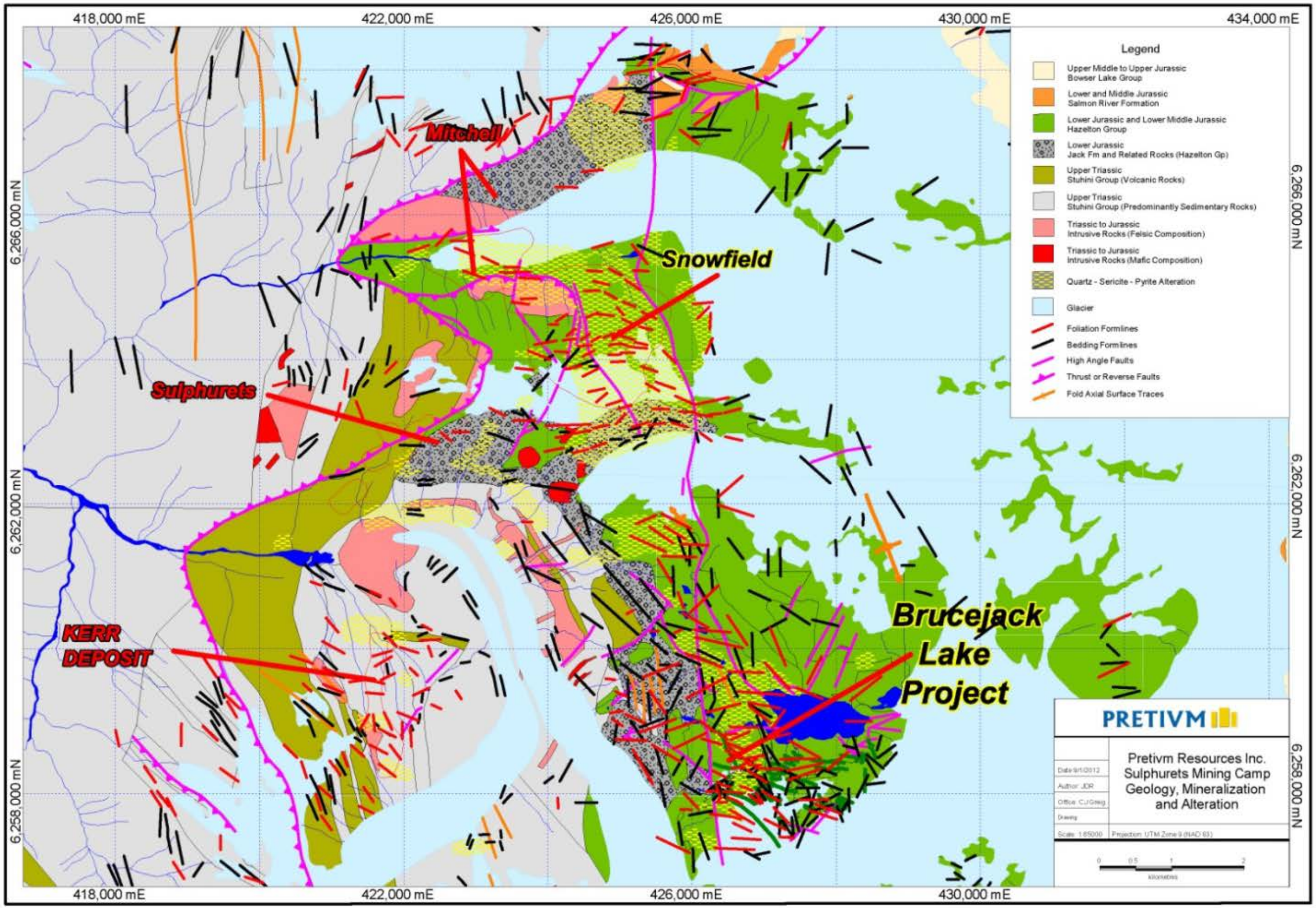
Elsewhere in the camp, Kirkham and Margolis (1995), Davies et al. (1994) and Alldrick and Britton (1991) mapped relatively common northerly, north-easterly and north-westerly striking brittle faults, and rare east-west striking faults, all with typically steep dips, steeply-plunging fault fabrics, and locally, normal-dextral oblique displacements of up to tens of meters.

7.2.5 Radiometric Dating: Ages of Stratigraphic and Intrusive Rocks and Constraints on the Timing of Deformation, Metamorphism and the Mineralizing Events

According to Kirkham and Margolis (1995), a number of internally consistent U-Pb zircon dates from pre-, syn- and post-mineral intrusive phases of the Mitchell intrusions at the KSM deposits suggest that porphyry-style mineralization was emplaced between 192 and 195 Ma. This early Jurassic age is consistent with a number of galena lead (Pb) dates for mineralization from these deposits, as well as for mineralization from Snowfield and from the West Zone. All of these dates fall in the "Jurassic cluster" of galena lead dates defined by Alldrick, Gabites and Godwin (1987) and Alldrick et al. (1990) for the "Stewart Mining Camp." Other Stewart Camp deposits falling in this cluster included the significant Silbak-Premier, and Big Missouri Mines.

Radiometric constraints for an early Jurassic age of mineralization at the KSM deposits, as well as at Snowfield and at Brucejack, are consistent with the structural relationships described above, and with the presently understood ages of host rocks to Brucejack mineralization.

Figure 7.5 Sulphurets Mining Camp Geology and Mineralization



7.3 PROPERTY GEOLOGY

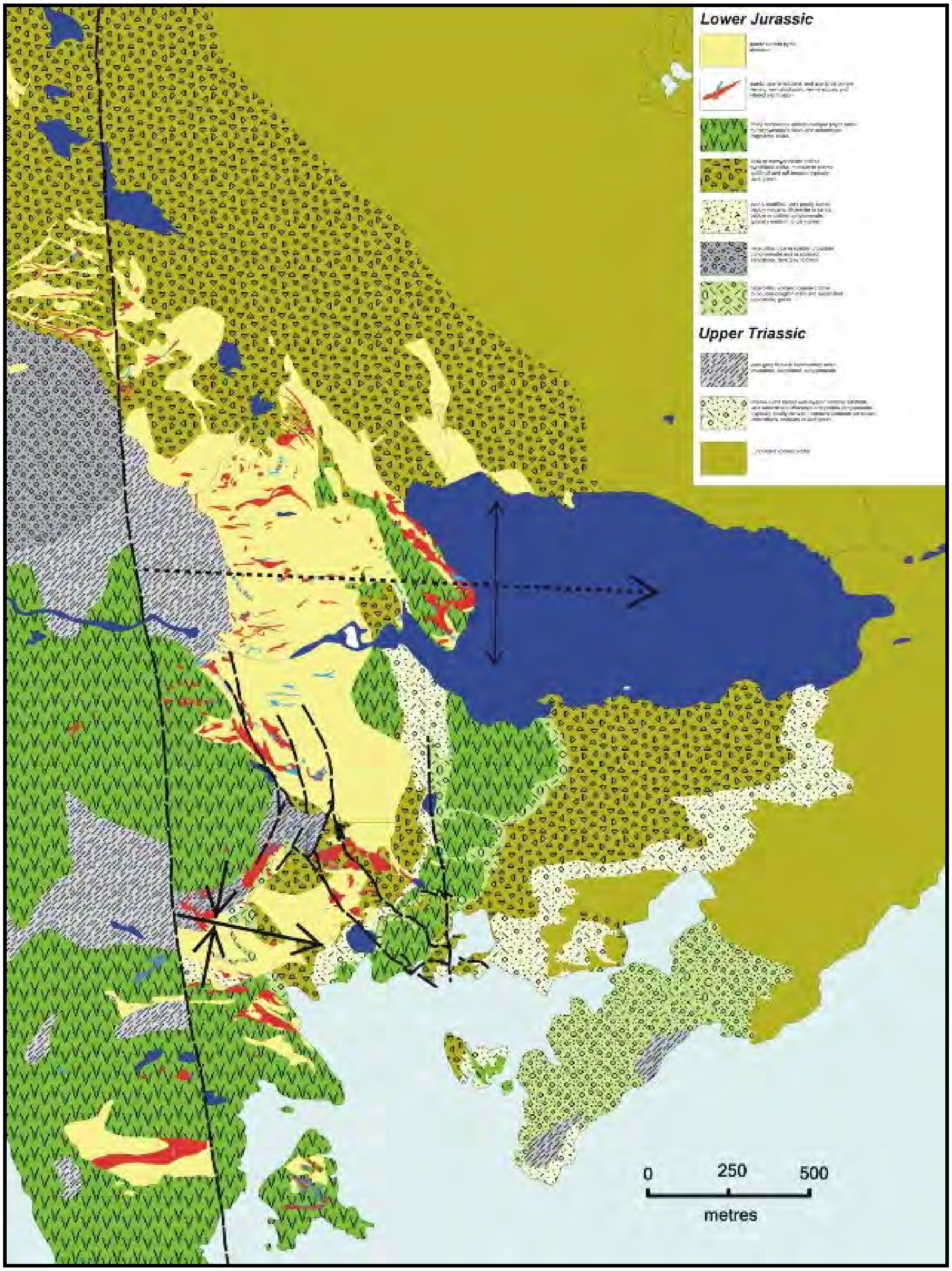
7.3.1 INTRODUCTION

This section on property geology was provided, in large measure, by Mr. Charles Greig, Senior Geologist for Pretium.

Geology on the Property in the vicinity of the mineralized zones can be characterized as a series of northerly-trending, broadly arcuate, concave-westward structural-stratigraphic belts of variably altered rocks. These are bisected on the western side of the Property by a prominent topographic lineament, the Brucejack fault (Figure 7.6 and Figure 7.7). To the south of Brucejack Lake, the arcuate belts of rocks generally trend north-easterly, and to the north of the lake they swing into more north-westerly trends. The belts are not entirely outlined by stratified rocks, but also by rocks of the "Sulphurets alteration system," which, as mentioned above, is a conspicuous belt of intensely quartz-sericite-pyrite (QSP) altered rocks. The alteration belt hosts most of the defined resources on the Property.

The map units are commonly overprinted not only by the very intense QSP and carbonate-chlorite-pyrite alteration, but also by what appears to be at least two phases of folding. As detailed below, these are crudely orthogonal and they yield locally well-developed interference patterns. Bedding typically displays near-vertical dips within what to the east of Brucejack fault are relatively rare well-stratified rocks. All the rocks, and in particular those which are most altered, commonly contain a well-developed steeply-dipping foliation, which generally trends east-northeast to west-southwest.

Figure 7.6 Brucejack Property Geology



With the exception of a limited number of narrow post-mineral and post-tectonic dykes of probable Tertiary age, rocks on the Property appear to be entirely of Late Triassic or Early Jurassic age, with most belonging to the Lower Jurassic part of the Hazelton Group. There are almost no intrusive rocks east of the Brucejack fault, other than meter-scale dykes. This is in contrast to the area west of the fault, where previous mapping (e.g., Kirkham and Margolis 1995) shows them to be relatively common.

Previously, some of the rocks within the belt of intense alteration were mapped as intrusive (e.g., Davies et al. 1994). Drilling over the last three years has provided a property wide database, (including additional detailed mapping), and Pretium now interprets these rocks as extrusive. Most of the bodies of massive fine-grained rocks contain local fragmental layers, which are interpreted to represent interflow block tuff or flow-breccia. In addition, there is little or no evidence in the vicinity of the larger masses for associated dykes, and little evidence for contact aureoles. In a number of outcrops, there is clear evidence for the incorporation of large, angular fragments of these bodies, which are texturally distinctive (they typically contain abundant fine- to medium-grained hornblende and/or feldspar phenocrysts within an aphanitic groundmass) within marginal and/or overlying fragmental units. Furthermore, the relatively massive rocks are commonly interlayered with clastic sedimentary rocks near their basal contacts, and locally they contain fragments of lithologies which are known to be Upper Triassic in age.

East of the Brucejack fault, where most of the mineralized zones have been discovered to date, volcanic or volcanic-derived (volcaniclastic) sedimentary rocks of the Lower Jurassic Hazelton Group predominate. In general order of abundance these include flows, fragmental rocks (pyroclastic rocks) and derived, generally poorly sorted and poorly stratified conglomerate, sandstone and, very locally, their finer-grained equivalents. In a very general sense, the proportion of sedimentary lithologies on the Property increases to the west, toward and west of the Brucejack fault, due to a general progression down-section in that direction, toward the base of the Hazelton Group and its lower contact with rocks of the Upper Triassic Stuhini Group, in which sedimentary lithologies predominate.

Interbedded with and locally capping the sedimentary rocks of the Upper Triassic Stuhini Group is a flow-banded rhyolite. Preliminary U-Pb dating completed by Pretium indicates that the rhyolite is Late Triassic in age. These distinctive rocks are only exposed at surface in the western-most VOK area, immediately east of the Brucejack fault, but very similar rocks have also been intersected in drillholes at West Zone, Gossan Hill, and Golden Marmot. They are interpreted to represent submarine rhyolite flows and associated fragmental rocks, but were formerly mapped as “cherts” by Newhawk (e.g., McPherson et al. 1994) and were previously considered to be dykes or perhaps sills.

Unconformably overlying the Triassic rocks are rocks of the Lower Jurassic Hazelton Group. They comprise four principal intercalated rock types. These include: 1) heterolithic volcanic conglomerate, most common at the base and typically coarse-grained (Jack formation), 2) hornblende and/or feldspar-phyric volcanic rocks, principally flows and related coarse fragmental rocks, 3) weakly stratified fine-grained and poorly-sorted matrix-rich lithic crystal pebble conglomerate or pebbly sandstone which contains scattered but distinctive volcanic pebbles or cobbles, and 4) pyroclastic rocks, including medium- and coarse lapilli tuff and tuff-breccia. The flows include several subtypes which may be distinguished by the grain size and compositions of their phenocryst assemblages generally fine- to medium-grained hornblende and plagioclase feldspar, (+/- medium- and locally coarse-grained potassium feldspar) they are typically rich in groundmass potassium feldspar, and are essentially latites and perhaps trachy-dacites or trachyandesites. These are also the rocks which previous mappers in part mapped as intrusive.

Very little time was spent mapping to the west of the Brucejack fault. Previous work indicates that the most abundant rocks are sedimentary rocks of the Upper Triassic Stuhini Group, typically fine-grained and well-bedded. They are also shown as being intruded by a number of mafic to felsic primarily alkalic intrusions, a number of which have been dated as Early Jurassic in age. The Upper Triassic clastic rocks have been folded across steep northerly-trending folds and related faults and were deformed and eroded prior to deposition of the lowermost rocks of the Hazelton Group, which are known locally as the Jack formation (Henderson et al. 1992). The contact relationships are emblematic of a profound regional-scale latest Triassic to earliest Jurassic deformational event which appears to have affected Triassic and older rocks across the length and breadth of Stikinia (e.g., Greig and Brown 1990, Henderson et al., 1992). The area west of the Brucejack fault was mapped in most detail by Davies et al. (1994), who showed the southern half of the area as being underlain by thin-bedded and fine-grained Upper Triassic Stuhini Group sedimentary rocks. In the northern half, the Stuhini Group is shown by them as being overlain by medium- to coarse-grained sandstone and pebble to cobble conglomerate which they correlated with the lowermost parts of the Hazelton Group (Davies et al. 1994).

7.3.2 STRATIGRAPHY

A number of rock types on the Property bear strong similarities with Upper Triassic and Lower Jurassic rocks which crop out elsewhere in the Iskut-Stewart area (Anderson 1989, Anderson and Thorkelson 1990), however, contact relations among a number of the lithologies suggest that a number of them represent facies equivalents of one another. Others which may bear outward similarities with one another, from Pretium's preliminary dating, appear to have been erupted over a relatively wide range of time. For example, a preliminary U-Pb zircon date from rhyolite in the VOK area strongly suggests the sedimentary rocks immediately down-section of the rhyolite in the southern part of VOK are also Late Triassic or older in age. These fine-grained sedimentary rocks are interbedded with fine-grained, well-bedded dark grey to black rocks near the nose of the "VOK synform".

On the other hand, most of the fine-grained hornblende feldspar-phyric rocks on the Property are outwardly similar in a number of ways, and all appear to be Early Jurassic in age, and yet preliminary U-Pb dating suggests that a number of them may differ significantly in age. Most, when unaltered, are typically rich in groundmass potassium feldspar, and their phenocryst assemblages, which may also contain potassium feldspar, and locally contain both plagioclase and potassium feldspar, contain common fine- to medium- and locally coarse-grained hornblende. This suggests a correlation, at least with the two feldspar-bearing rocks, with flows of the "Premier porphyry" member of Grove's (1986) Unuk River formation. This may indeed be the case for some of the rocks on the Property, and yet compositionally-similar rocks, typically with a groundmass rich in potassium feldspar, interfinger with conglomeratic rocks which most geologists who have worked in the region would reasonably term "maroon and green epiclastic rocks" and quite comfortably assign to the apparently younger Betty Creek Formation of Grove (1986) or Alldrick and Britton (1991).

As a consequence, in Pretium's mapping, which covered only a part of the southern Property, Pretium has refrained from any attempt to formally assign formational names to the map units. Instead, geologists have kept to the more broadly-defined Upper Triassic Stuhini and Lower Jurassic Hazelton groups, although have made notes for some of the units regarding possible regional correlations. The map units below are described in order of age, from oldest to youngest.

7.3.3 SOUTHERN PROPERTY MAP UNITS

UPPER TRIASSIC STUHINI GROUP

Thinly-bedded, well-stratified and well-sorted typically fine-grained sedimentary rocks; dark grey to black.

These rocks were only examined in detail adjacent to the Brucejack fault in the western VOK area, where they are only locally well-exposed. They include common mudstone, siltstone, and thin- to medium-bedded sandstone. They are typically tightly folded and complexly faulted. Relationships between these rocks and the other Upper Triassic rocks described below remain incompletely understood. They may in part represent Jurassic rocks, which in places, such as in the south-eastern most part of the map area, are broadly similar in appearance.

Massive and locally well-layered volcanic siltstone, and subordinate litharenite and pebble conglomerate (typically locally-derived); contains common carbonate concretions; medium to dark green.

The concretion-bearing fine-grained sedimentary sequence occurs in the VOK area and is really only well-defined in drillholes. The green silty to muddy rocks typically occur in the immediate footwall of the Upper Triassic rhyolites in the "VOK syncline," and are commonly separated from the rhyolites by a meter-scale zone of intensely sericite(?)-altered rocks with a distinctive and vivid green colour. Whether or not the overall green colour of this sequence of rocks is alteration-related or a primary feature remains to be established.

In general this sequence of rocks is poorly-stratified, particularly when compared to the dark grey to black rocks described above. Locally however, the sequence contains well-and thinly-bedded tuffaceous or reworked tuffaceous rocks over core intervals of several meters or more. Locally interbedded with the finer-grained rocks are pebble to cobble conglomerate and sandstone which typically contain subangular to subrounded clasts that appear to be locally-derived. The distinctive and characteristic concretions common to these rocks may take many forms, but typically have rounded outlines. They consist largely of calcite, pyrite and chlorite, and while they are typically on the order of several centimeters in scale, they can range in size up to several tens of centimeters in the long dimension. As mentioned above, the relationship between the green fine-grained rocks and their presumably age-equivalent dark coloured fine-grained but well-stratified equivalents to the north-west remains to be firmly established. At this stage they are interpreted to interfinger.

RHYOLITE

Massive to flow-banded, and commonly fragmental; may include associated rhyolite pebble conglomerate and siliceous (and/or silicified sandstone); pale coloured; extremely well indurated. Typically barren of precious metal mineralization. The rhyolite may in part be in place but may also possibly represent locally extensive blocks eroded from the Stuhini Group and incorporated in the overlying rocks of the lowermost Hazelton Group. Only occurs at surface in western VOK area.

SILICEOUS ARGILLITE

Black very well indurated siliceous and very fine-grained sedimentary rocks. Typically intensely veined by white quartz vein stockwork and possibly represent a siliceous exhalite.

LOWER JURASSIC

Heterolithic volcanic coarse cobble to boulder conglomerate and associated sandstone.

Polylithic volcanic cobble to boulder conglomerate is common in the VOK area, where it is commonly a medium green colour (sericitized?) in drill core; in surface outcrops it is typically limonite-stained. The conglomerate is very poorly sorted, and includes common volcanic lithologies, including common fragments of hornblende feldspar phyric latite, as well as moderately common rhyolite clasts/fragments and less common but locally abundant sedimentary rock fragments; it is typically interbedded with pebble conglomerate, pebbly sandstone, and local sandstone; all are typically intensely pyrite- and sericite-altered.

Poorly stratified, very poorly sorted pebbly volcanic litharenite to sandy pebble or cobble conglomerate; typically medium to dark green.

Generally massive to very weakly stratified, these are typically dark green to locally maroon in colour. May be distinguished by the presence of scattered sub-angular to sub-rounded relatively resistant pebbles and cobbles of rhyolite, but also commonly contain scattered flattened latite to trachyandesite volcanic rock fragments, as well as very local rounded concretions. Groundmass material, which predominates, appears to be feldspar crystal-rich (arkosic) volcanic litharenite. Probably represent reworked tuffaceous rocks, and appears to grade laterally and upward into the pyroclastic rocks described below.

Latite to trachyandesite coarse pyroclastic rocks: medium to coarse lapilli tuff and tuff-breccia; typically dark green.

Coarse lapilli tuff and tuff-breccia occur in the core of the VOK syncline as well as to the east and along trend to the north. These rocks are characterized by the presence of distinctive and typically angular chlorite-altered and commonly hornblende- and feldspar-bearing volcanic rock fragments. The volcanic rock fragments may locally range in size up to several meters in long dimension. The pyroclastic rocks are most likely the lateral equivalents of latite to trachyandesite flows described below.

Maroon, mauve and subordinate green, poorly sorted and weakly stratified volcanic cobble to boulder conglomerate.

Generally coarse-grained and sedimentalogically immature cobble to boulder conglomerate rocks are well exposed along the eastern margin of the map area, on the ridge above the Knipple glacier. The rocks are only locally well-stratified and are typically poorly sorted, with angular to sub-angular and locally sub-rounded clasts. They are locally interbedded with maroon to red mudstones, and with rare tuffaceous rocks. They in part represent the lateral equivalents of latite flows described below.

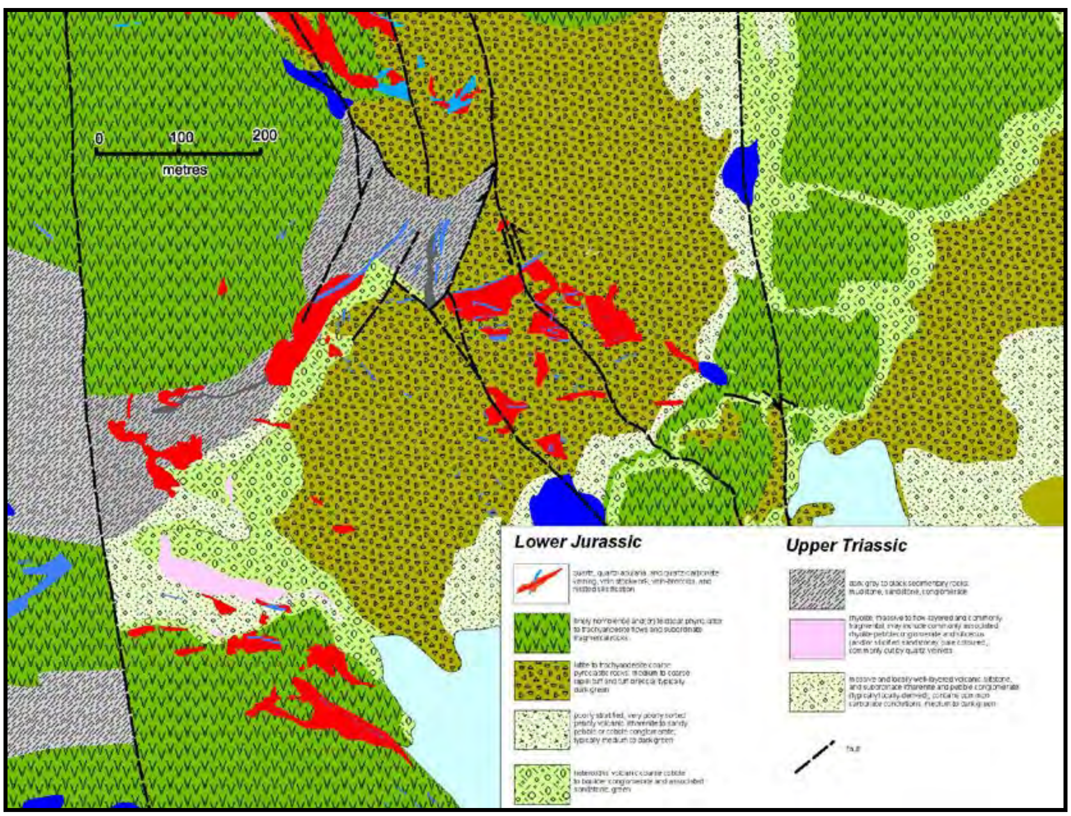
Fine hornblende and (or) feldspar-phyric latite to trachyandesite flows and subordinate fragmental rocks.

This suite of rocks, which are interpreted to represent a series of flows and related fragmental rocks, are characterized by the presence of phenocrysts of hornblende and plagioclase feldspar set in a groundmass rich in fine-grained potassium feldspar. The latites or possible trachyandesites have yielded preliminary U-Pb zircon dates which vary by as much as 15 Ma. This is not surprising given that some flows, such as those near the eastern margin of the area shown in Figure 7.6, are clearly stratigraphically younger than some of the bodies farther west. This uncertainty relates in large part to the nature of what are generally poorly-stratified rocks, to the fact that the rocks have been folded and faulted, and to the fact that they have been intensely altered. The best exposed and least altered rocks lie along the eastern part of the mapped area and are clearly extrusive, with auto-brecciated rocks tailing out into laterally equivalent fragment-supported boulder and coarse cobble conglomerate.

Phenocryst type and grain size in the flows can vary widely in abundance and relative proportion. For example, relatively coarse-grained potassium feldspar phenocrysts may also be common along with plagioclase in some bodies. Gradational variation in phenocryst types in a number of drillholes suggests the possibility that varying phenocryst populations may be in part a function of internal compositional variation, as opposed to variation between flows. Phenocrysts in these rocks are also commonly aligned and may help to outline a well-developed foliation and/or lineation. While foliations measured in outcrop generally conform to those developed regionally, there is a strong likelihood, particularly in the area east near Knipple Glacier, that the foliation does represent an extrusive flow-foliation. There is little doubt that this has been overprinted by later deformation, including folding. In the western part of the mapped area, fine-grained hornblende feldspar-phyric flows and subordinate coarse fragmental rocks directly overlie Upper Triassic rocks, perhaps with a relatively thin intervening sequence of Lower Jurassic black clastic rocks.

While the main focus of Pretium's 2011 geological work was on drilling, a new map was made in the general VOK area, where more detailed work focused on the area between Brucejack Creek at the north end and the Bridge Zone to the south, and between the Brucejack fault on the west and the East Knipple glacier on the east (Figure 7.7).

Figure 7.7 Brucejack VOK Area Geology



At Brucejack, the most intense alteration appears to be focused in the rocks immediately underlying and overlying the unconformity between Upper Triassic and Lower Jurassic rocks. In the vicinity of VOK, this unconformity is locally marked by the presence of rhyolitic rocks, or rock fragments. They are considered to represent extrusive rocks, but may in part be preserved as erosional relics and in part as large blocks within the basal conglomerates and "reworked" volcanic rocks of the lowermost Hazelton Group. These basal rocks are the interpreted equivalents of the Jack Formation of Henderson et al. (1992) however, the intense alteration which often masks the original rock types in this belt leads to many uncertainties and some confusion in assigning rock types and correlating them with other units. There are rocks at VOK that have been logged previously as volcanic, which upon closer examination may be better described as immature pebble to cobble conglomerates and associated pebbly sandstones, sandstones, and finer-grained lithologies derived from volcanic protoliths. When intensely altered they are readily confused with volcanic rocks, as they are derived entirely from them. Distinguishing these units by mapping is next to impossible within the intense gossan of the Sulphurets alteration zone, however by tracking them out into the less altered periphery, and by utilizing drillhole control, they appear to be equivalents to Jack Formation rocks, and to rocks of Henderson et al.'s (1992) Jack Formation, and Grove's (1986) and Alldrick and Britton's (1991) Unuk River and Betty Creek Formations.

7.4 STRUCTURAL GEOLOGY

7.4.1 INTRODUCTION

The Property is characterized by the presence of steep structural elements, including steeply dipping planar features such as bedding, foliation, or brittle faults, and steeply plunging linear features, such as fold hinges, pencil cleavage, or mineral lineations. The lineament that marks the Brucejack fault is the most obvious to anyone that first arrives on the Property. Perhaps the more notable is the east-west, steeply dipping foliation present in most rocks on the Property, and in particular the more altered rocks. Less obvious, but certainly related are the folds, likely occurring in two phases. These folds are less obvious because of the difficulty of tracing out contacts and recognizing bedding within the generally poorly stratified but highly altered Lower Jurassic rocks east of the Brucejack fault. Other notable structural elements which are of less certain origin include steeply plunging pencil cleavage, and a similarly steeply plunging mineral lineation, typically outlined by fine-grained amphibole.

7.4.2 FOLIATION

Foliation on the Property is pervasive, although it is best displayed in the most altered rocks which are the quartz sericite schists. While fine-grained muscovite defines the foliation in the altered rocks, foliation in the less altered rocks is typically outlined by both sericite and chlorite. In general, the most common foliation orientation is roughly east-west. Locally the orientation appears to be controlled more by the orientation of the nearest competent or resistant rock mass, which, in the case of the northwest-trending West Zone could be an extensive silicified stockwork (which, probably not coincidentally, parallels geologic contacts), or a relatively massive volcanic body, such as the hornblende feldspar-phyric flow underlying much of the Bridge Zone.

Within the broader zones of veining, individual veins, veinlets, and narrower stockwork zones generally have orientations which are sub-parallel to the foliation in the host rocks. The most foliated rocks are those immediately adjacent to the veins.

Locally up to several foliations, all typically steep, may be evident in the rocks. The West Zone Footwall and Golden Marmot Zones host good examples of this type of foliation. This typically occurs in the most intensely foliated lithologies, and/or in adjacent to quartz-vein zones, and where alteration is also the most intense. It is in these areas where the pencil cleavage is generally best-developed. In other places, the coarse pencil cleavage appears to have been developed by the intersection of steep (and commonly curvilinear) joint sets with the steep foliation.

7.4.3 FOLDING

On the southern part of the Property, south of Brucejack Lake and Brucejack Creek, the stratigraphy outlines arcuate south-westerly trending lithologic belts which probably represent sub-parallel curvilinear folds (Figure 7.7). Broad synclines of the north-east trending folds are interpreted to be occupied by massive flows of generally younger hornblende feldspar phyric latite to trachyandesite fragmental rocks. The corresponding anticlines have narrower trends and are outlined by cobble to boulder conglomerates and related immature volcaniclastic rocks. The more coarse-grained members of the conglomerates are commonly intensely QSP altered. The frequency of the folding is on the order of half a kilometre.

The relatively tight anticlinal crests and broader synclines are suggestive of a cuspate-lobate style of folding. The massive and more competent Lower Jurassic, Hazelton Group flows and coarse volcanic fragmental rocks underlie the synformal lobes. The less competent, but relatively well-stratified Upper Triassic Stuhini Group, along with the lowermost Jurassic Jack formation rocks underlying the cusps. As McPherson et al. (1994) noted, this phase of folding likely pre-dated formation of the foliation, as the foliation cuts across the trends of these folds with little evidence of deflection.

A second and probably more important set of folds is oriented normal to the north-easterly trends, and has much tighter wavelengths which are on the order of a hundred meters or so. Like their north-easterly-trending counterparts, they are delineated by the contacts between older, predominantly clastic rocks of the Jack Formation and Stuhini Group, and the younger, predominantly volcanic flows and associated coarse volcanic fragmental rocks of the Hazelton Group. As is the case with the north-east-trending folds, bedding, where it could be measured, is very steeply dipping. As a consequence, plunges of folds also appear to be very steep.

Since the surface traces of these fold axes are parallel with the east-west trending foliation, these folds are interpreted to have been formed synchronously with the foliation.

7.4.4 BRITTLE FAULTING

As noted above, steep post-mineral brittle faults are present across the breadth of the Property. Many form well-defined lineaments, with that of the Brucejack fault being the most prominent. Few have any well-defined offsets, and most offsets appear relatively minor (less than several tens of meters). One exception is in the area near the western margin of Sulphurets glacier, to the north-west of Brucejack Creek, where clean and well-exposed outcrops show apparent dextral offsets of up to 20 or more metres along north-trending faults occupying lineaments which are sub-parallel to the Brucejack fault. Furthermore, many smaller-scale features on the Property show similar apparent displacements (K. Konkin, pers. comm. 2011). Many of the lineaments crosscut vein, vein-stockwork and alteration zones, suggesting that they are relatively young. Among these were several that were outlined by Newhawk in the West Zone drilling and underground work, including the Bruce Fault, a curviplanar west-trending fault dipping 60 or 70° to the north which in part follows Brucejack Creek. McPherson et al. (1994) suggested that displacement on it was on the order of tens of meters.

7.5 MINERALIZATION AND ALTERATION

7.5.1 GENERAL INTRODUCTION

The most prominent geologic features on the Property are the gossanous alteration zones that form a north-south arcuate trend, lying mostly east of the Brucejack fault (Figure 7.5). These form a broad band of variably but generally intensely QSP altered rocks of up to several hundred meters or more across, and approximately five kilometers in strike extent. Within this band, the high grade mineralization is generally associated with stockwork systems. These stockwork systems display good continuity and are characterized by the presence of veins of quartz, quartz-carbonate, quartz-adularia, and pyrite that are typically on the order of millimetres to tens of centimetres in thickness. In rare cases, veins may range in thickness up to nearly 10 m.

To date, the larger mineralized zones such as VOK and West Zone, are up to 500 to 600 m in strike length and down-dip extent, and are open in at least one direction. They may include areas with +25% quartz and/or quartz-carbonate-adularia as veins, vein-stockworks, and vein-breccias, as is shown on the plans and sections in the following section. The high grade mineralization has a structural component, where the bulk tonnage mineralization forms broader halos, not necessarily associated with veining at all. Most high grade zones are either on the margins of or contained within a zone of bulk mineralization. Also, most quartz veins are barren so the presence of veining is not necessarily indicative of mineralization; however, most (but not all) high grade gold intersections are associated with veins. Bonanza grade mineralization has been intersected in veins as narrow as a few centimeters, and up to veins exceeding a meter in width.

The most continuous mineralized features do not appear to be individual veins; instead they are more commonly stockwork veins and vein-breccia systems that often contain decimeter-thick discontinuous individual veins. The thickest veins are rare, do not display good continuity, and are commonly quite massive.

7.5.2 ALTERATION OVERVIEW

The QSP alteration typically contains somewhere between 2 and 20% pyrite. The alteration consists largely of fine-grained white mica (sericite) and abundant calcite that together, with the pervasive foliation, makes it very difficult to recognize protoliths in the alteration belt. Careful examination of drill core has shown that the coarse conglomeratic rocks near the base of the Hazelton Group form a preferential host to alteration and mineralization.

Mineralized zones show abundant evidence for deformation at all scales. This includes brecciation and boudinage of larger veins, and common ptygmatic folding of veinlets. There is some indication that flattening of stockwork, parallel with the foliation has occurred. This can be seen on a broad scale since the majority of the mineralized structural systems are east-west which is believed to have been syn-formational with the foliation; however, locally veining and vein breccias are also seen cross cutting the foliation.

Late, relatively undeformed veins do exist on the Property. For example, there is a very common set of gently-dipping, discontinuous quartz-carbonate chlorite veins. These commonly have associated iron carbonate but little sulphide, and occur property-wide. In addition competent lithologic bodies, in particular more siliceous ones such as the rhyolites, are characterized by the presence of networks of millimetre-scale quartz veinlets, many of which appear to be undeformed, and most of which are free of sulphides.

The alteration hosts forty or more known mineralized zones along its length (Figure 7.8). They vary somewhat in style of veining and in gold and silver grades and ratios, and many have yet to see significant exploration drilling. There are important differences among them but also many similarities. They range from structurally controlled, high-grade gold-rich, silver-poor zones such as VOK, to high-tonnage but relatively low-grade, such as Bridge Zone. Another variety is in the West Zone, which carries very good silver grades along with significant gold mineralization.

While in general the alteration zones on the Property appear to parallel the overall structural-stratigraphic trend north-ward toward Snowfield, it is also important to note that in detail both the alteration and the many closely-related mineralized zones are discordant to the host rocks. That being said, the alteration and mineralization are generally not discordant to the regional foliation.

Previous researchers believed that there was a connection among mineralized zones. For example, McPherson et al. (1994) believed that the north-west trending West, Shore, and Electrum (south margin of present VOK Zone) Zones were linked with the more east-west trending zones, such as Gossan Hill. They envisaged these as being rungs in a somewhat canted ladder to the "ladder legs" of the main zones. Drilling over the last few years has shown that the high grade mineralization in these zones are independent of each other, although the broader bulk tonnage mineralization from one zone may overlap with another.

7.5.3 General Mineralization Overview

Sulphide mineralization within the alteration zone consists of 2 to 20% disseminated pyrite and rare arsenopyrite. Broad zones of bulk tonnage mineralization, locally up to several grams per tonne gold, are generally proportional with the amount of anhedral pyrite. Euhedral pyrite is barren of any mineralization, without exception. Within the veins, sulphides consists of trace to 10% combined disseminated pyrite, tetrahedrite, tennantite, arsenopyrite, chalcopyrite, galena, sphalerite, pyrargyrite, polybasite, and rare native gold and electrum. Locally base metal rich veins can reach several meters in thickness, and contain up to 5% combined lead-zinc. There is a weak association of arsenic with all the gold mineralization, both high grade and bulk tonnage. Locally, coarse clots of black sphalerite are intergrown with the quartz veinlets that host high grade and bonanza grade gold mineralization. Again, the presence of base metals or arsenic does not necessitate corresponding gold mineralization.

Readers are referred to the 1993 Sulphurets Exploration Report (Visagie, 1993b) for detailed descriptions of the mineralized zones on the Property.

Margolis (1993) suggested the bulk tonnage mineralization originated from reaction of the volcanic pile with sea water. In this albitization process, a high temperature end product is magnetite. Later sulphidation of the magnetite created the reducing environment required to precipitate the gold. This mineralization likely formed very shortly after emplacement of the host volcanics.

The high-grade, structurally controlled mineralization was likely a three-stage process (Lewis, 1994), however, this interpretation is based almost exclusively on West Zone, and Shore Zone, the only areas extensively studied at the time. Stage I consisted of fault development and ground preparation. Pre-cursor structures to the West, Shore and Electrum Zones likely formed at this time, as steep north-west trending normal faults with limited displacement. Stage II consisted of syntectonic mineralization and alteration, with massive and stockwork veins emplaced within a differential stress field characterized by east-west compressional stress. The main vein orientations resulting from this stress are, 1) east-west dilational veins such as R8 and Big Sleep; and 2) north-west trending veins localized along pre-existing structures such as the West, Shore and Electrum Zones. Underground mapping on the West Zone indicates the north-west trending structures, particularly R6, have been brecciated while east-west trending structures have not. This would support the theory of reactivation along pre-existing north-west structures. Reactivation was probably sinistral in movement, and may account for the sigmoidal shape of the Big Sleep Zone. The localization of major vein systems generally within the volcanic rocks as opposed to the sedimentary rocks is likely the result of preferential ground preparation within the volcanics; however, there are high grade mineral intercepts at VOK in the Triassic sediments. Stage III was marked by the development of north-west trending cleavage and local warping of smaller veins as a result of northeast-southwest shortening (Lewis, 1994).

The VOK Zone was discovered in 2009, long after this original work. The mineralization is generally east-west, and follows a structural corridor along the axis of the interpreted syncline. Although mineralization can be found in all rock types, the contact of the basal conglomerate with the overlying pyroclastic units, and the contacts along the margins of the older rhyolite, appear to be most favourable. Mineralization appears to be concentrated where these contacts are highly contorted and located with the structural corridor of the synclinal axis. Based on these observations, it appears that the previously developed structural analysis can be applied to the VOK Zone, and that most of the high grade and bonanza grade mineralization is related to east-west compressional stresses generating dilational veins along the synclinal axis.

7.6 Detailed Property Mineralization

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 systems and to vein deposits with both mesothermal and epithermal characteristics. Most of the mineral occurrences are hosted in Upper Triassic rocks or in the lower and middle parts of the Hazelton Group (Britton and Alldrick, 1988).

Early Jurassic and less common Late Triassic sub-volcanic intrusive complexes are common in the Stikine terrane, and several host well-known precious and base metal rich hydrothermal systems. These include copper-gold porphyry deposits such as Schaft Creek, Red-Chris, Kemess, Mt. Milligan, and KSM, as well as copper-gold replacement-style and porphyry(?) deposits such as Galore Creek. In addition, there are a number of related polymetallic deposits, including intrusive-related mineralization at Premier and Red Mountain, epithermal to mesothermal veins at Snip, Middle Jurassic subaqueous replacement and precious metals-enriched VMS mineralization at Eskay Creek, and Upper Triassic VMS mineralization at Granduc.

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

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

The Property is dominated by silver-gold-base metal bearing epithermal veins, as described by Alldrick and Britton (1991).

7.6.1 Brucejack Property Mineralization Overview

The Property has been the focus of periodic exploration over the past several decades resulting in the discovery of many gossanous zones of gold, silver, copper and molybdenum-bearing quartz/carbonate veining, stockwork and breccia- hosted mineralization (Figure 7.8). Typically, the showings occur within gossanous rocks which reflect the weathering of disseminated pyrite in surrounding argillic and phyllic alteration zones. The extent of the gossans, their intensity of alteration and the tenor of the contained mineralization make them attractive exploration targets (Alldrick and Britton, 1991); most have been extensively surface-sampled and many have been drill-tested by at least a few shallow holes.

The mineralization on the Property typically consists of quartz-carbonate-adularia, gold-silver bearing veins, stockwork and breccia zones, along with broad zones of disseminated mineralization. The veins are hosted within variably altered potassium feldspar-bearing volcanic rocks which have been overprinted by sericite-quartz-pyrite±clay alteration. The currently-held interpretation is that the higher-grade mineralization is structurally-controlled and was remobilized from pre-existing disseminated-style mineralization during later-stage deformation, perhaps related to development of the SFB in mid-Cretaceous time (see below).

An alternative interpretation envisages the high-grade mineralization emplaced as a late-stage pulse of mineralization during the later stages in the development of a broad field of disseminated to stockwork-style mineralization. The hydrothermal system was most probably localized along synthetic structures to a “proto-Brucejack” fault zone, and these structures, along with the proto-Brucejack fault itself, were likely syn-volcanic, and may have been mineralized, and re-mineralized, over a considerable period of time during deposition of the potassic volcanic rocks of the Lower to lower Middle Jurassic Hazelton Group. In this scenario, the various styles of vein mineralization, with differing gangue and sulphide mineral assemblages, are seen as being emplaced at relatively widely separated times during the Early to early Middle Jurassic. For the relatively late, distinct, and very gold-rich phase at VOK, this alternative would have it being emplaced along an intensely altered zone which, once weakened, later became a locus for deformation, including folding across the VOK syncline, in mid-Cretaceous time, probably during deformation related to development of the SFB.

Figure 7.8 Historical Map with Mineral Deposits and Occurrences

Note: Modified after Budinski, 1995.

As envisaged by Lewis (1994), veining and deformation was likely a three-stage process:

1. 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 north-west trending normal faults with limited displacement, cutting all rock types.
2. Stage 2 involved development of syntectonic veining 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:
   - east-west dilational veins
   - 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.
3. 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.

Pretium reviewed all of the historical and ongoing exploration results, and has identified ten zones of potentially economic mineralization. Eight of the zones were modelled and are included in the resource estimate herein and are described in the following sections. The ten zones identified on the Property are illustrated in Figure 7.9 and listed below:

1. West Zone
2. West Zone Footwall Zone
3. VOK Zone
4. Gossan Hill Zone
5. Galena Hill Zone
6. Bridge Zone
7. Low grade Halo Zone
8. Shore Zone
9. SG Zone
10. Waterloo Zone

Figure 7.9 Outline of Ten Zones Identified on the Property

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, and are probably related to intrusive activity at depth.

In general, vein mineralization appears to represent a complex system of multiple generations and styles of mineralization and alteration. Veins can generally be classified on the basis of metal content and gangue mineralogy. Typically the exposed veins are thin (less than 1 m) and short (less than 50 m). Individual veins may coalesce into denser vein systems, especially in the more intensely altered areas; locally they may represent in excess of 25% of the rock mass. Such vein systems typically grade gradually into strongly silicified wallrocks. Adularia is common in some veins, and as is seen in West Zone, is often found as pseudomorphs after coarsely bladed calcite.

Base-metal bearing quartz veins consist primarily of thin stringers of quartz ± carbonate which locally contain millimetre- to centimetre-scale zones of disseminated or blebby sulphides, which in order of abundance are pyrite, galena, and sphalerite. Such veins are found locally across the Brucejack Plateau outside the main areas of alteration, and as in the main zones, individual veins are typically gossanous in outcrop.

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 suggest repeated fracturing and filling as the host rocks underwent brittle fracturing. Where visible, gold in the form of electrum typically occurs as coarse aggregates or late stage fracture fillings, as rims on subhedral quartz crystals, or as lace-like networks formed around coarse grains of adularia. Seams of electrum up to a centimeter in thickness have also been observed in sericitized country rocks, with little obvious association to more gangue-rich veins.

Appreciable silver grades are generally present in mineralized zones where gold to silver ratios are less than 1:10, whereas all of the known bonanza grade intersections on the Property have gold to silver ratios of roughly 2:1. In general, and with exception of the West Zone, silver-dominant veins tend to be restricted in extent. High-grade silver mineralization occurs as silver sulphides and sulphosalts that are related to adularia-rich veins in which adularia pseudomorphs bladed calcite; these appear to be relatively late stage.

Precious metal mineralization may be confined to one particular episode of veining, and not necessarily the same vein episode as that in which the base metal mineralization was emplaced. 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.

Detailed Description of Mineralized Zones

The following descriptions of the mineralization of the West, Bridge, Low Grade Halo, Galena Hill, Shore, SG, Gossan Hill and VOK Zones of the Property were provided by Mr. Ken McNaughton, Vice President and Chief Exploration Officer for Pretium.

West Zone

The West Zone gold-silver deposit is hosted by a north-westerly trending band of intensely altered Lower Jurassic latitic to trachyandesitic volcanic and subordinate sedimentary rocks, as much as 400 to 500 m thick, which passes between two more competent bodies of hornblende plagioclase hornblende phyric flows (Figure 7.10 and Figure 7.11). The stratified rocks dip moderately to steeply to the northeast and are intensely altered, particularly in the immediate area of the precious metals mineralization.

The West Zone deposit itself comprises at least ten quartz veins and mineralized quartz stockwork ore 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 what Roach and Macdonald (1991) interpreted as sinistral, mainly ductile deformation. Crack-seal features shown by most of the veins are evidence of brittle deformation overlapping with crystallization of gangue minerals. Thus, at the West Zone, it appears as if localized ductile strain may have generated dilatant structures that served as conduits for the hydrothermal fluids, which deposited silica and precious metals, but hydrostatic overpressures within the conduits may have intermittently induced brittle failure along sub-parallel 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 thickness of the alteration zones across the central part of the deposit is 100 to 150 m.

Gold in the West Zone occurs principally as electrum 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 acanthite. Gangue mineralogy of the veins is dominated by quartz, with accessory adularia, albite, sericite, and minor carbonate and barite.

Figure 7.10 West Zone Assay Cross Section

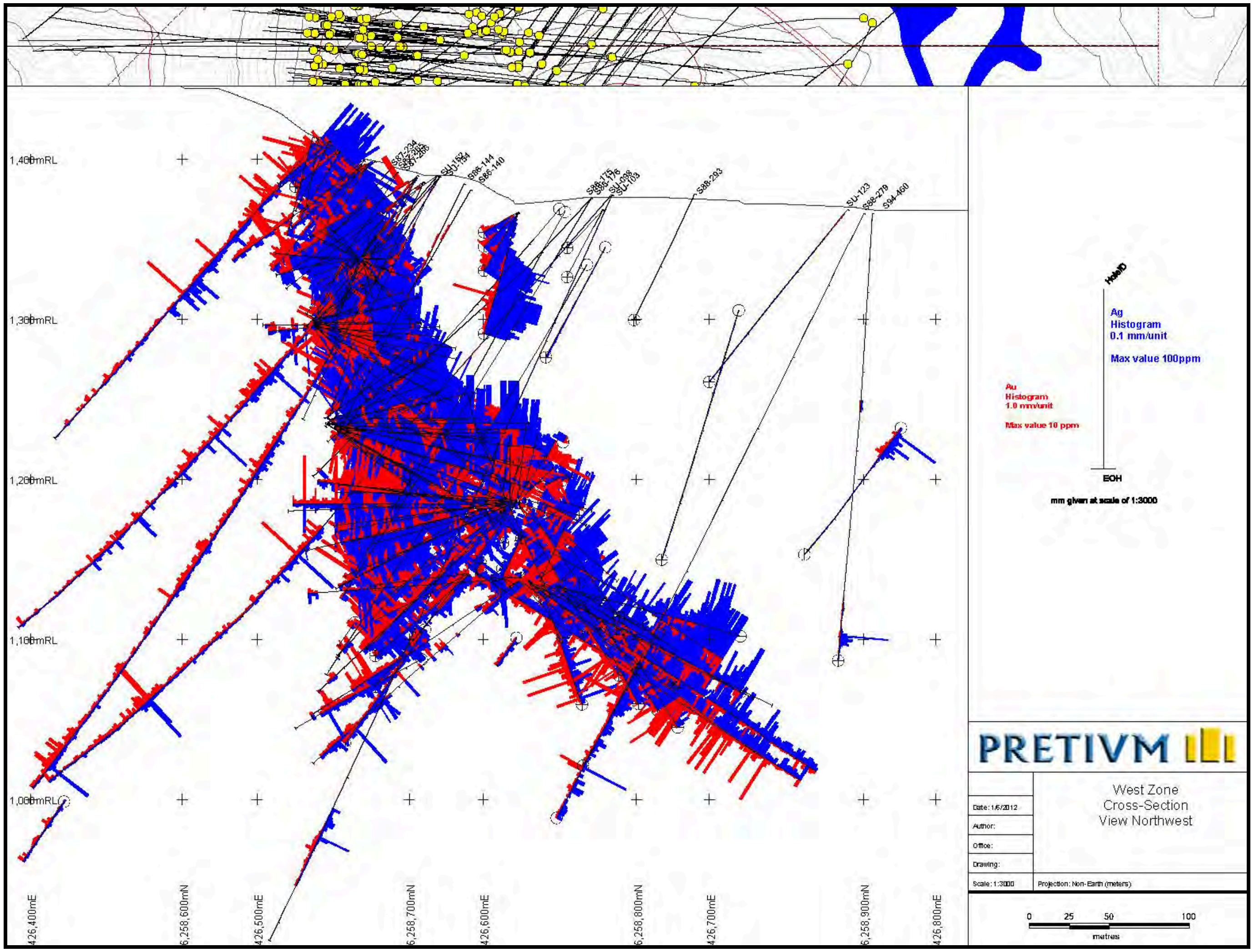
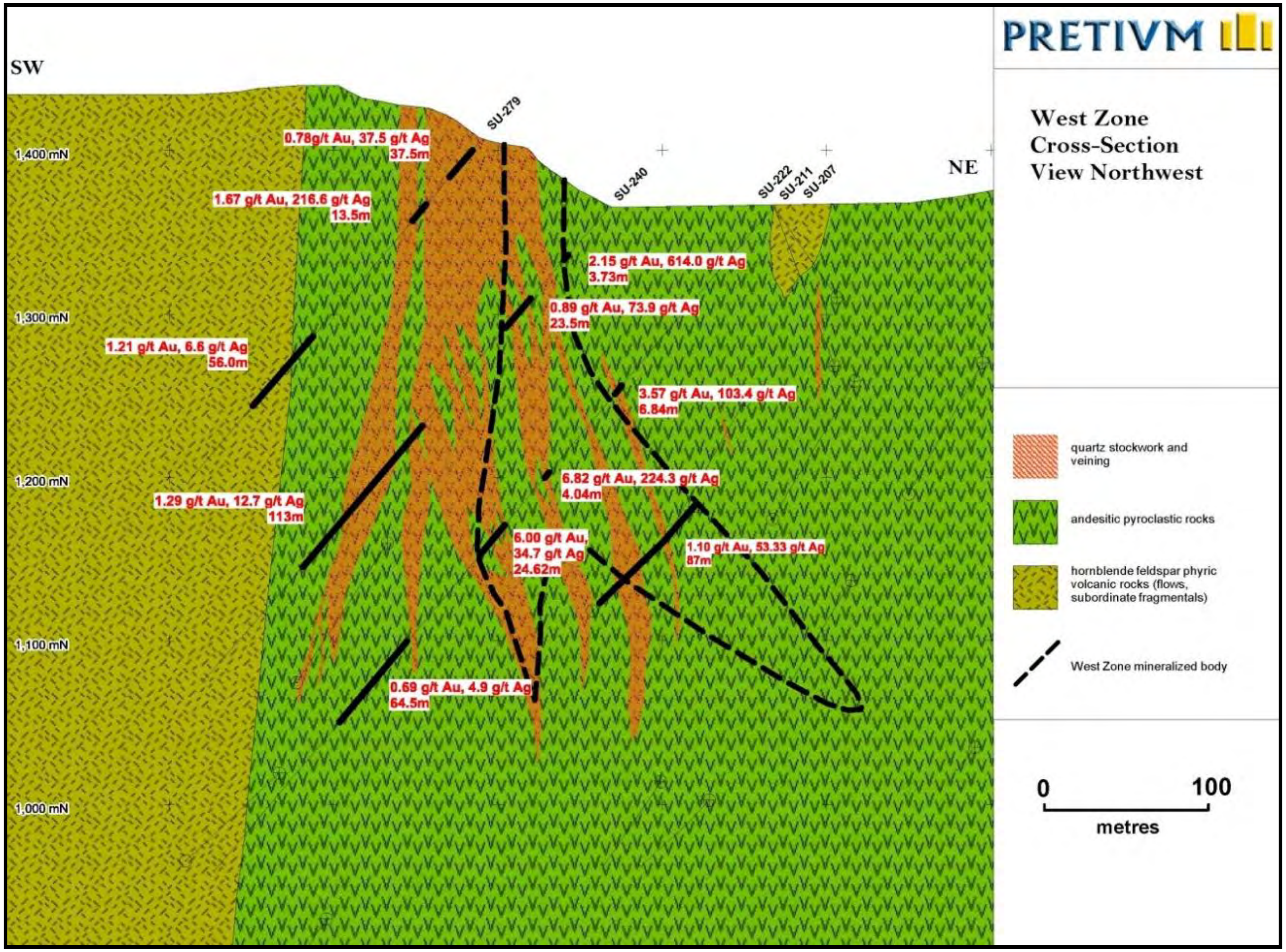


Figure 7.11 West Zone Geological Cross Section



Valley of Kings Zone

The VOK Zone is approximately 500 m south of West Zone, and was initially explored by Esso in 1981. It was later explored by Newhawk and named the Spine Zone, due to the presence of small, silicified and resistant relatively high-standing stockworks that outcrop in several places along its trend. This particular stockwork is topographically prominent but generally quite barren. The early exploration programs included surface grab and trench sampling, with many samples yielding weak to moderately anomalous gold values, but one outstanding sample which returned over 7,000 ppm gold.

The first diamond drill program at VOK was conducted by Silver Standard in 2009. This program intersected a number of intervals of high-grade gold mineralization, including one of 16,948 g/t Au and 8,695 g/t Ag over 1.5 m in hole SU-012 and two additional 1.5 m intersections between 40 and 80 g/t Au in that hole and four more spread between holes SU-034 and 035. Additional diamond drilling in 2010 included 12 drillholes totalling 4,871 m. The results of this program were very encouraging, with five of the holes intersecting high grade gold mineralization, including intersections in holes SU-040 and SU-084 which returned 5,850 g/t Au and 5,480 g/t Au, respectively. Pretium's drilling at the VOK in 2011 totalled 101 holes and more than 43,100 m (Figure 7.12).

The VOK mineralized zone trends approximately west-northwest to east-southeast. Its orientation mirrors that of Electrum Ridge, a pronounced topographic feature near the southern margin of the zone, and drilling to date has extended its strike to over 450 m. The zone is up to 150 m wide and is bound to the west by the Brucejack fault but remains open at depth and to the east. Surface mapping and Pretium's extensive drilling defined a number of lithologic contacts which outline a broad syncline in which fragmental volcanic and clastic sedimentary rocks, along with minor flows of Upper Triassic to Lower Jurrassic age appear to plunge moderately to the east (Figure 7.13). Variably altered hornblende feldspar phyric volcanic rocks of intermediate composition (most probably latite) are interpreted as forming the youngest rocks of the sequence at the VOK, and are seen outcropping to the south, west and to the northwest, and broadly correlative coarse pyroclastic rocks, including common lapilli tuff and tuff breccia may occupy the core of the VOK syncline. Underlying these are interbedded volcanic-derived immature sedimentary rocks, including common pebble and cobble conglomerate and pebbly sandstone. The sedimentary sequence is considered correlative with the basal Jack formation of the Hazelton Group. Generally thin and likely discontinuous rhyolite flows, as well as local siliceous exhalites have been mapped on surface and logged in drill core in the vicinity of this contact. A preliminary Upper Triassic U-Pb date from the rhyolite suggests that it is derived from beneath the unconformable contact, and Pretium's tentative view is that the rhyolites may, in part, be large blocks resting on the unconformity surface. Beneath the rhyolite is a relatively thick and generally poorly stratified sequence of fine-grained concretion-bearing mudstone and siltstone with locally interbedded immature but locally-derived sandstone and pebble conglomerate. In the vicinity of the VOK, contacts and even the unconformity appear to have been folded, commonly tightly. The contacts outline a complex east-plunging syncline with a flanking east-plunging cuspate anticline to the south and what appears to be an even more complex antiformal structure to the north. The northern limb of the fold opens to the northeast, where it may eventually reverse trend and continue to the northwest into the area between the West Zone and Gossan Hill. Much of the complexity may reflect the fact that the east-west trending folds are refolding an earlier northeast-trending set of folds, but some of the complexity may also reflect the fact that there may have been relief on the unconformity surface.

High-grade gold and silver mineralization within the VOK Zone occurs as electrum, and it is generally hosted within quartz-carbonate and quartz-adularia veins and vein stockworks (Figure 7.14 and Figure 7.15). While quartz veining and stockworks are common throughout the zone, the majority of gold intersections are confined to a 75 to 100 m wide zone which closely parallels the axis of the syncline. Within that zone, the mineralization appears to have been concentrated in localized fold noses and along geologic contacts, in particular along the contact between the overlying pyroclastic rocks and the underlying conglomerate, as well as locally along the margins of flow-banded rhyolite. Significant intervals of gold mineralization, including several occurrences of visible gold, have been intersected to the west of the VOK, at the Waterloo Zone, suggesting the possibility that the mineralized trend may extend farther west, across the Brucejack fault.

Gold:silver ratios within the VOK Zone are typically 2:1 or higher, but may vary greatly. This may be due in part to the relative abundance of electrum in the deposit, or to the local presence of silver sulphide minerals. Additional precious metals-bearing minerals found in the VOK, typically in trace quantities, include silver sulphides, acanthite, pyargyrite and tetrahedrite, while base metal-bearing sulphides include sphalerite and galena.

As it is elsewhere on the Property, alteration at the VOK Zone is believed to be Early Jurrassic in age. It consists dominantly of QSP, with lesser sericite-chlorite. The most pervasive of the intense alteration is observed within the sedimentary and fragmental volcanic rocks. Within them, the abundance of phyllosilicate minerals, and the subsequent deformational overprint, has resulted in the development of a pervasive east-west trending and steeply dipping foliation. In the VOK, the foliation appears to be axial planar to the main fold trend.

Figure 7.12 VOK General Geology Map

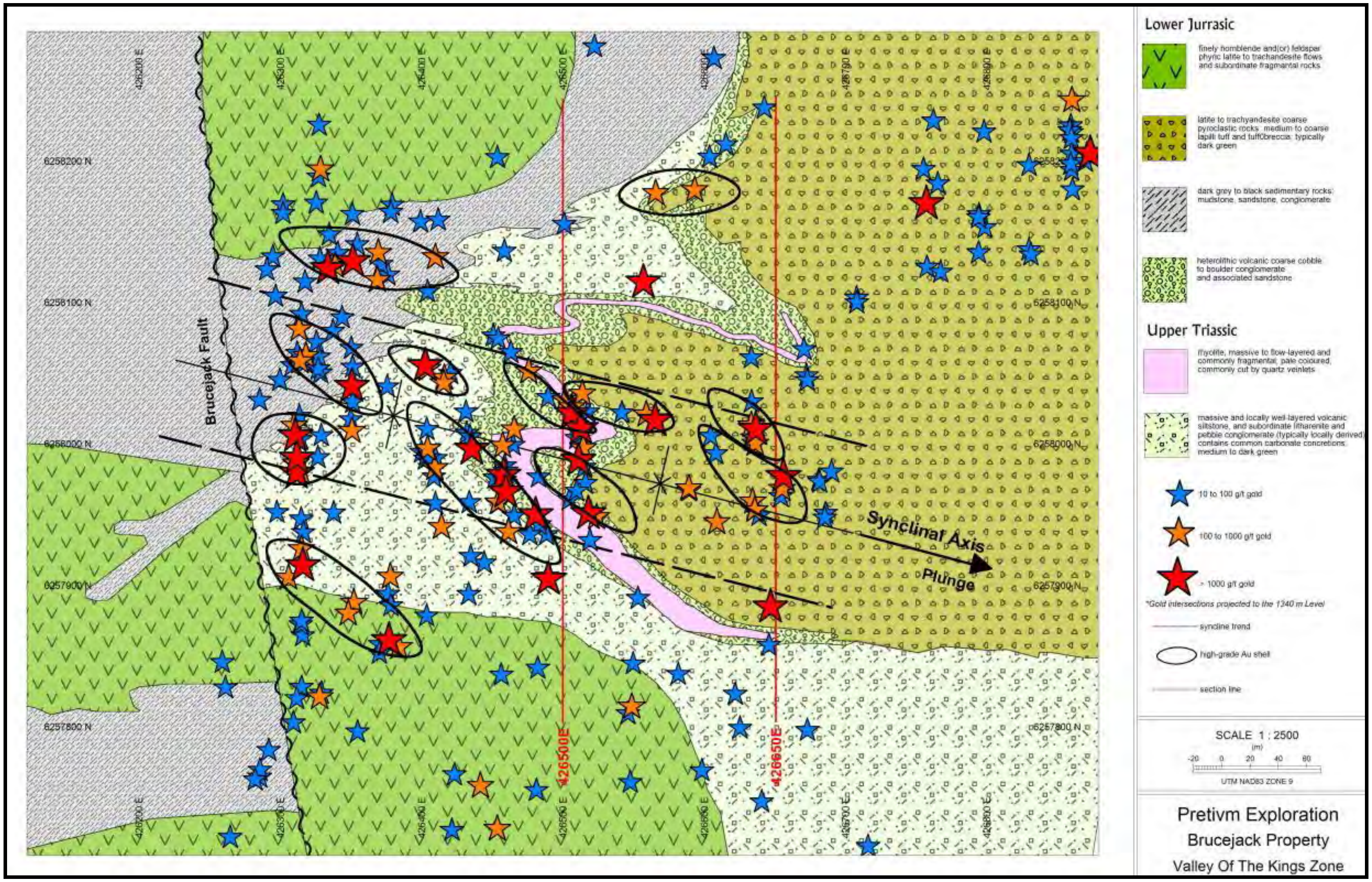


Figure 7.13 VOK Long Section View North

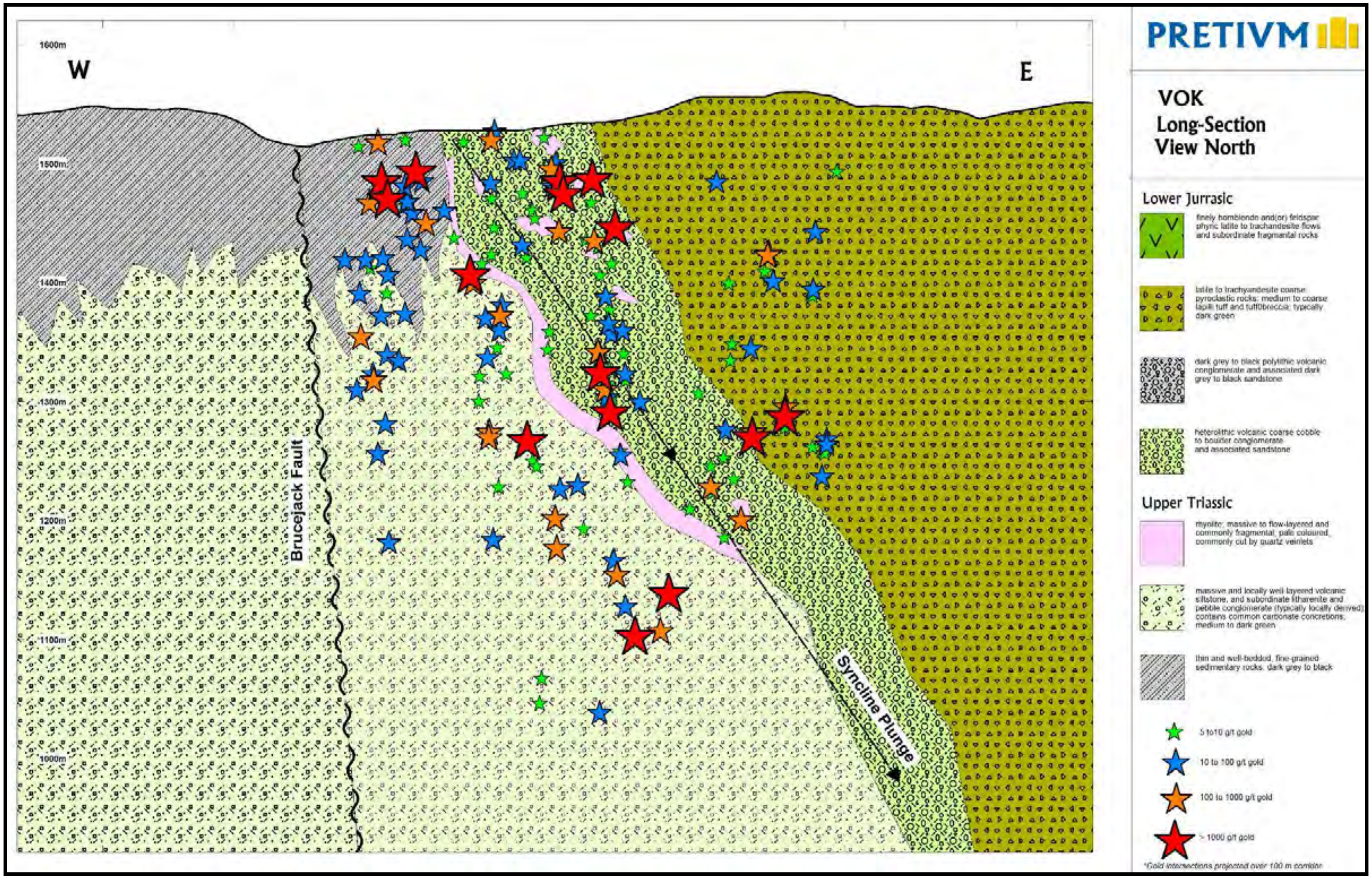


Figure 7.14 VOK Cross Section 6500E

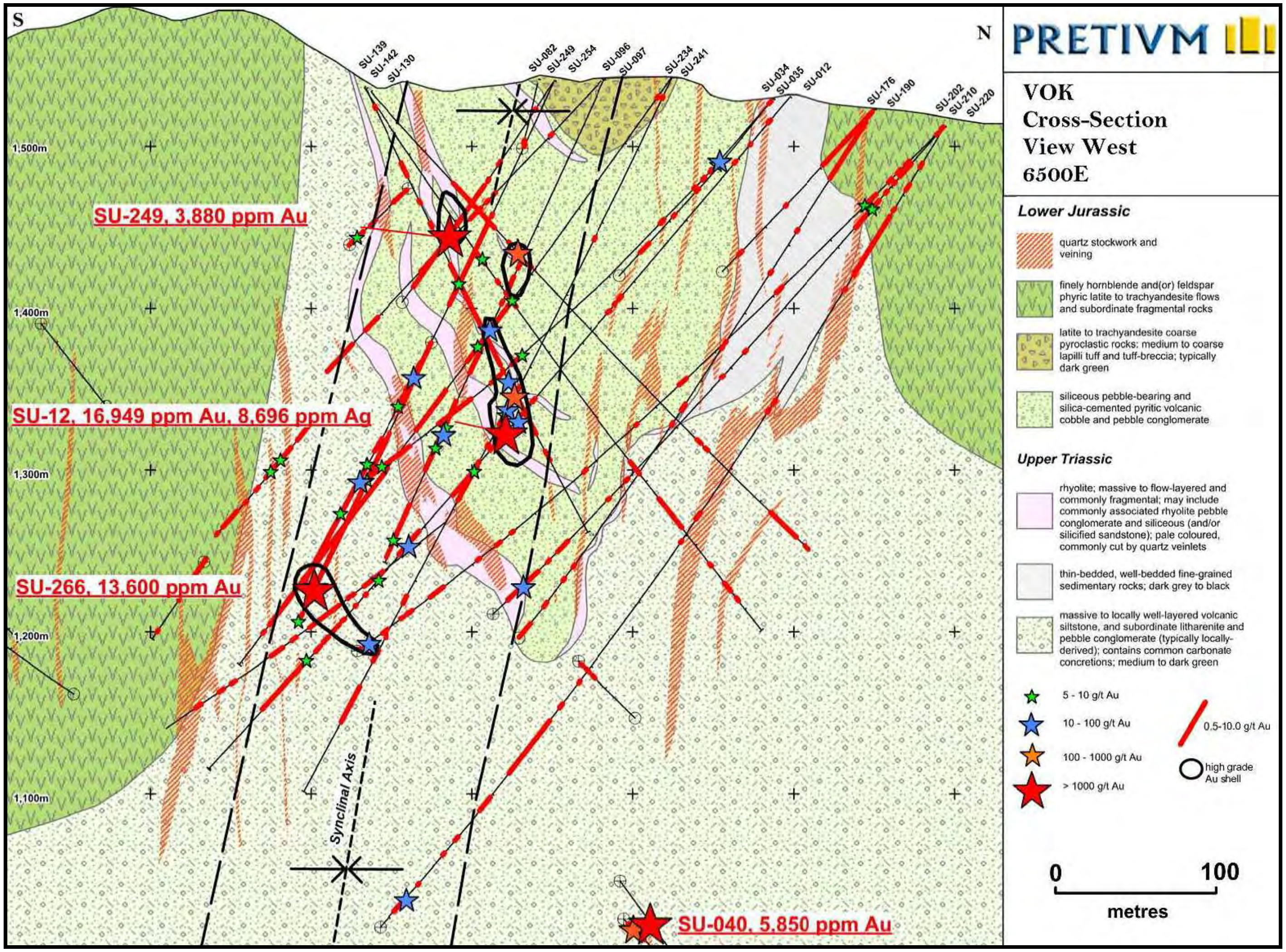
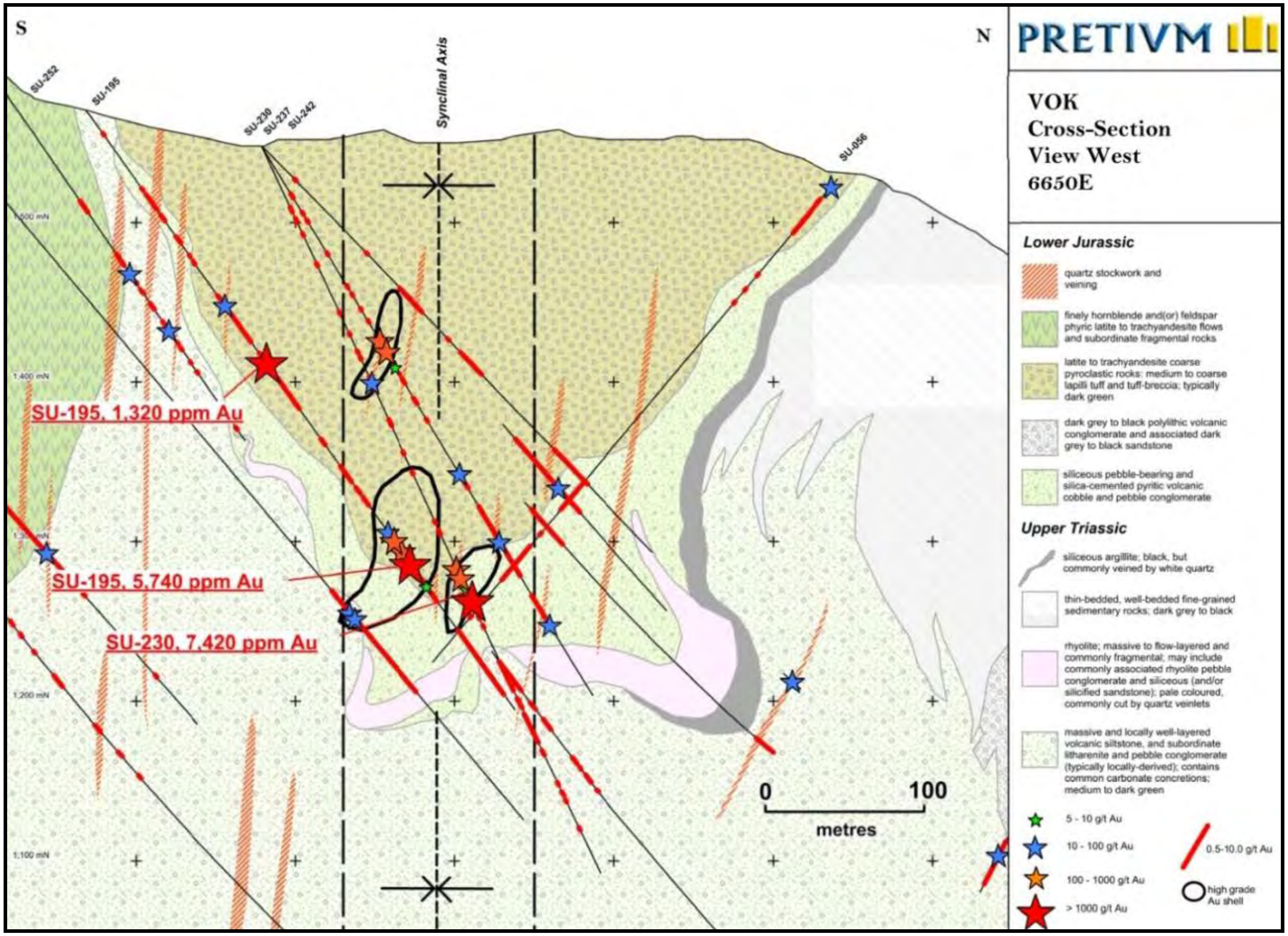


Figure 7.15 VOK Geology Cross Section 6650E



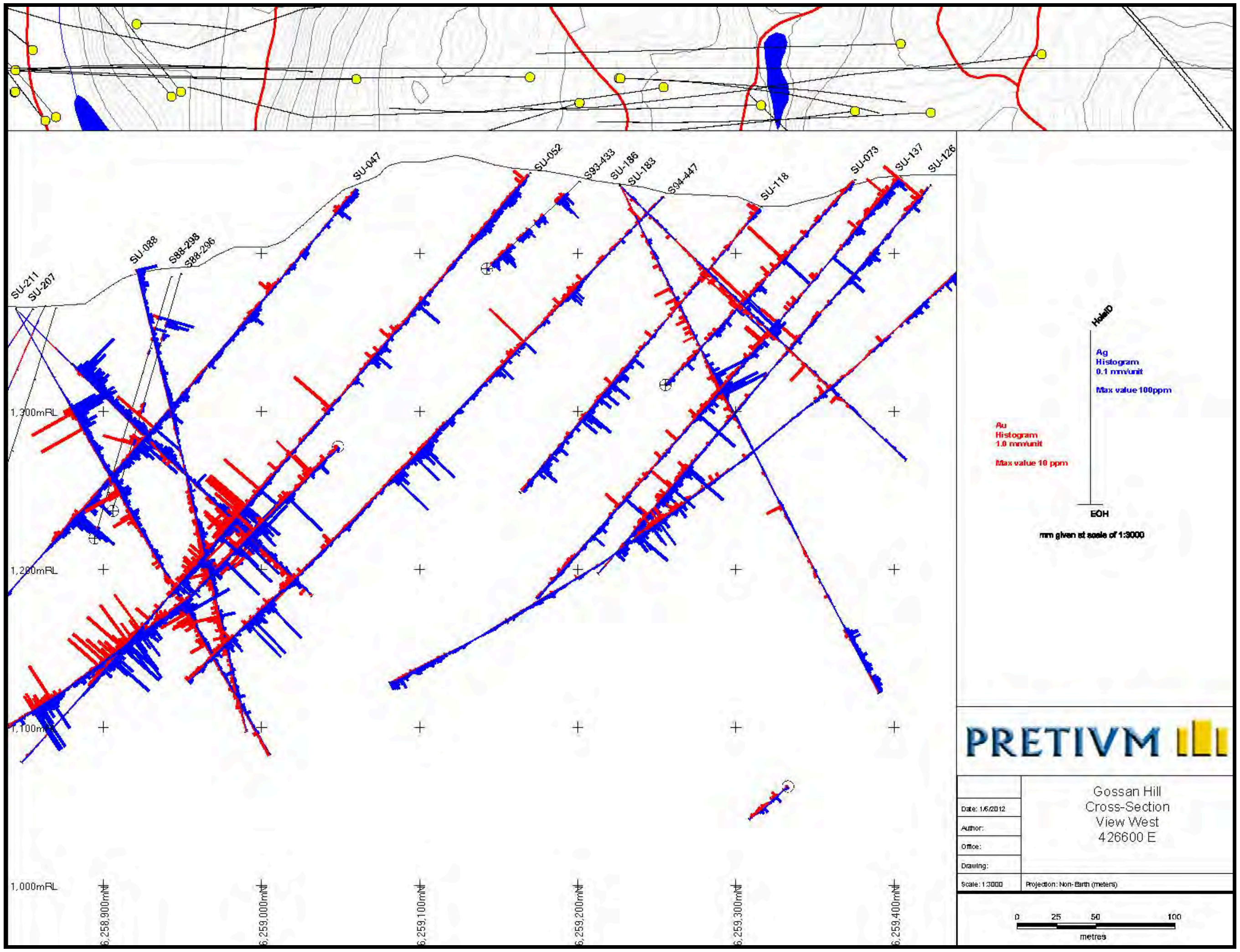
Gossan Hill Zone

The mineralized zone known as Gossan Hill is a circular area, about 400 m in diametre, of intense QSP alteration developed in Lower Jurassic volcanic rocks. The visually impressive alteration zone at Gossan Hill is host to at least 11 quartz vein and quartz vein 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 thick.

Historical work undertaken at Gossan Hill has included rock-chip sampling, hand trenching and limited diamond drilling, with a few +400 m deep drillholes concentrated in the central part of the mineralized area. Precious metal mineralization at the Gossan Hill Zone occurs in two styles. As is the case elsewhere on the Property, a bulk tonnage style of mineralization is associated with anhedral pyrite and a fine quartz stockwork. Higher-grade gold mineralization at Gossan Hill differs somewhat from other zones on the Property in that it is associated with the larger quartz lenses, particularly where they contain local aggregates of pyrite, tetrahedrite, sphalerite, and galena. Electrum is observed in the bonanza grade intersections, while silver also occurs in tetrahedrite, pyragyrite, and polybasite.

Drilling in 2011 was focused on defining structural controls to mineralization within the larger lower grade halo (Figure 7.16). The program was very successful, with numerous holes intersecting very high grade gold mineralization; including 372.3 g/t Au across 7.1 m in hole SU-136, 16.4 g/t Au across 7.9 m in hole SU-147, 168.4 g/t Au over 1.5 m in hole SU-201, and 44.2 g/t Au over 14.0 m in hole SU-207.

Figure 7.16 Gossan Hill Assay Cross Section 426600E



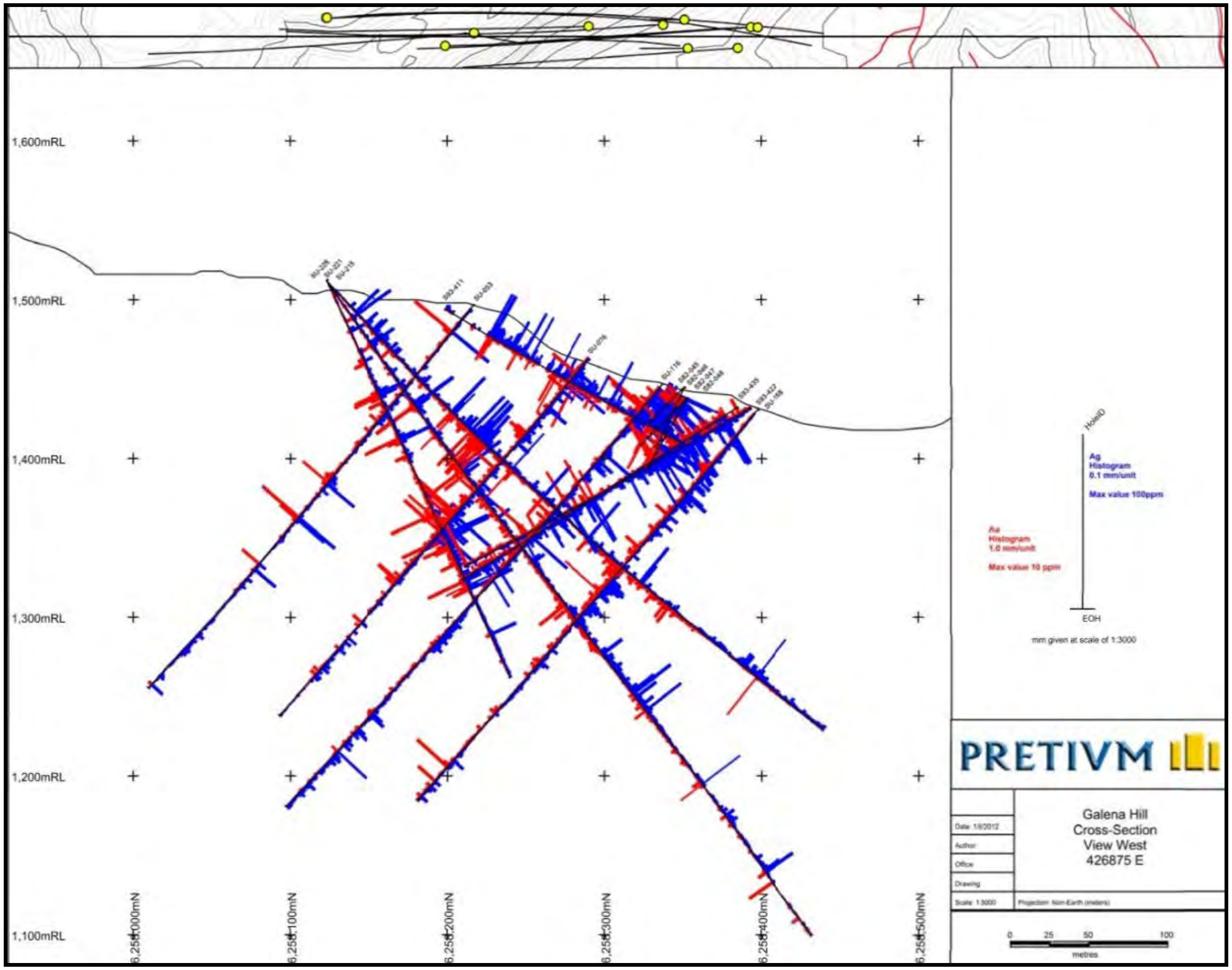
Galena Hill Zone

The prospect known as Galena Hill is situated on a prominent hill southeast of the southern end of the West Zone and east of the VOK north zone. The hill is marked by widespread iron oxide staining of altered volcanic fragmental and reworked volcanic fragmental rocks and the slopes are commonly faced by quartz stockwork. The Galena Hill Zone was tested by Newhawk with 27 bore holes, with half of the holes being less than 100 m in length. The historical work suggested that the Galena Hill system was an underlain by east-west and northeast-southwest trending sets of quartz veins and quartz stockworks within a zone of hydrothermal alteration and mineralization that was at least 460 m long and 300 m wide.

The host volcanic and volcaniclastic rocks are rich in lapilli-sized fragments and host local thin units of carbonaceous and cherty mudstone. A few holes intersected rhyolitic volcanic rocks and one hole (SU-05) yielded a 50 m long intercept of quartz which was enriched in gold and silver along its margins; unfortunately it is likely that the vein was intercepted at a low angle to the core axis and that the hole drilled down the dip of the vein.

As in the West Zone, gold mineralization at the Galena Hill Zone is preferentially associated with quartz veins, although the sericite-altered, intermediate composition host rocks are typically mineralized with disseminated pyrite and do host low- to medium-grade bulk tonnage style mineralization (Figure 7.17). In some veins, trace amounts of native gold and electrum are accompanied by veins which contain common trace to locally massive sphalerite, chalcopyrite, and galena. Galena Hill has produced a number of bonanza grade intersections, including intersections such as 1,025 g/t Au across 1.5 m (hole SU-53), 2,490 g/t Au across 1.59 m (hole SU-54), 5,480 g/t Au across 0.5 m (hole SU-84) and 1,710 g/t Au across 0.69 m (hole SU-106). These intersections may be structurally controlled and represent a significant high grade exploration target.

Figure 7.17 Galena Hill Assay Cross Section 426875E



Bridge Zone

The Bridge Zone is located about 1,500 m south of the West Zone and is centred on a three hectare nunatak 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 that it was transected by a number of discontinuous mineralized quartz veins. The first drillhole to test the zone (SU-10) was drilled in 2009, and was extended in 2010. It intersected 601 m which averaged 0.76 g/t Au and 7.9 g/t Ag, including 118 m which averaged 0.99 g/t Au and 7.6 g/t Ag.

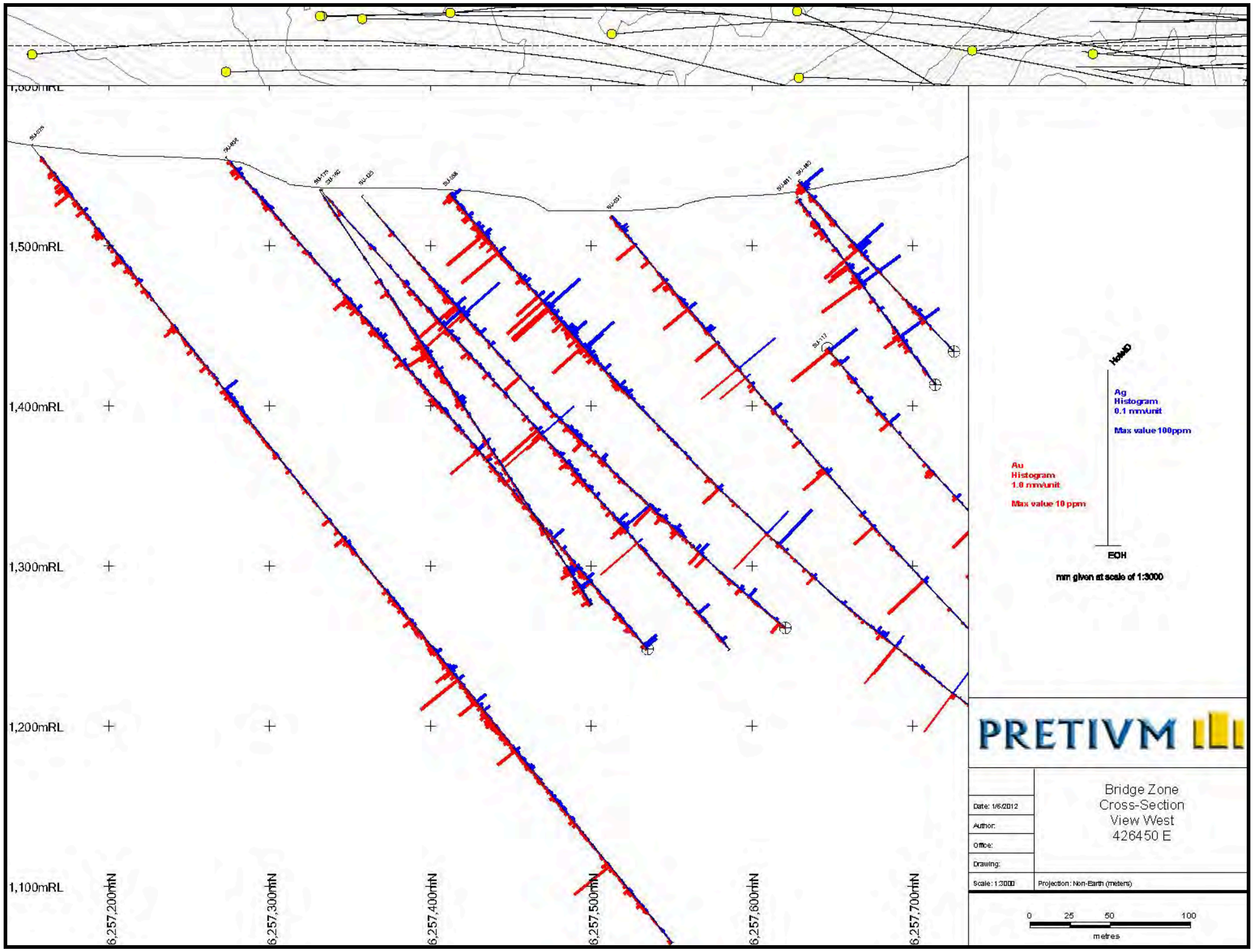
Further and much more extensive drilling in 2010 showed that the gold mineralization at the Bridge Zone is hosted by plagioclase-hornblende phyric volcanic rocks that in general are moderately to strongly sericite-chlorite altered, with disseminated and stringer pyrite making up a few percent of the rock by volume (Figure 7.18). Quartz ± chlorite ± sericite veins, 20 to 200 cm in thickness, were intermittently intersected by the drillholes, and these commonly contain minor to trace amounts of pyrite, sphalerite, galena, molybdenite, and an unknown dark grey, silver-bearing sulphosalt(s). In 2010 it was observed that a number of holes in the Bridge Zone contained appreciable molybdenum. These were then analyzed for the strategic metal rhenium, with the further observation that the molybdenum/rhenium ratio was similar to that in mineralization to the north at Pretium's Snowfield Zone.

A total of 47 drillholes now define the Bridge Zone, which not only includes broad intervals of bulk-tonnage style mineralization, but also numerous high grade intersections. Drilling in 2011 targeted a few of the high-grade intersections and the suggestion is that they are structurally controlled. Intersections from this year's drilling include 458 g/t Au in a one meter intersection in hole SU-125, and a 19.0 g/t Au intersection across 5.85 m in hole SU-166.

Low Grade Halo Zone

Drilling in 2011 filled the gap in drill data which existed between the Bridge Zone and the VOK. This area was shown to host bulk tonnage mineralization, and as a result, the low-grade halos for Bridge Zone, VOK and Galena Hill have been combined into a single zone which encompasses all the peripheral mineralization.

Figure 7.18 Bridge Zone Assay Cross Section

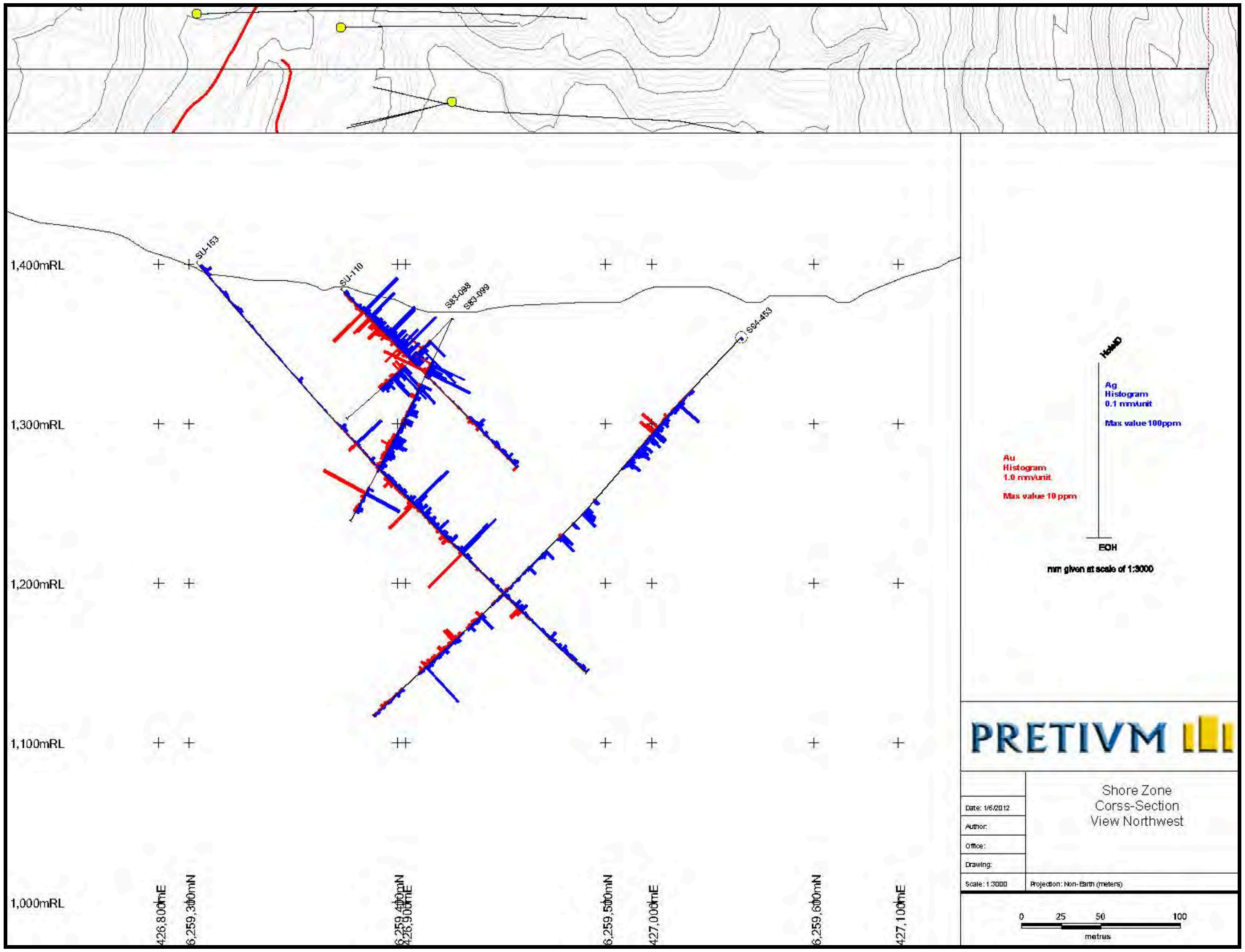


Shore Zone

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 trachyandesite that has a strike length of roughly 530 m and a maximum width of 50 m. The northwest-southeast trend of the zone is coincident with a pronounced lineament (likely a fault) which extends south-eastward from the Brucejack fault 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 of up to 100 m in length. The veins and vein stockworks consist predominantly of quartz with minor carbonate and barite, and they contain patchy sulphide mineralization consisting of variable quantities 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. Drilling in 2011 tested the northwest extension of the zone and returned modest results (Figure 7.19).

Figure 7.19 Shore Zone Assay Cross Section

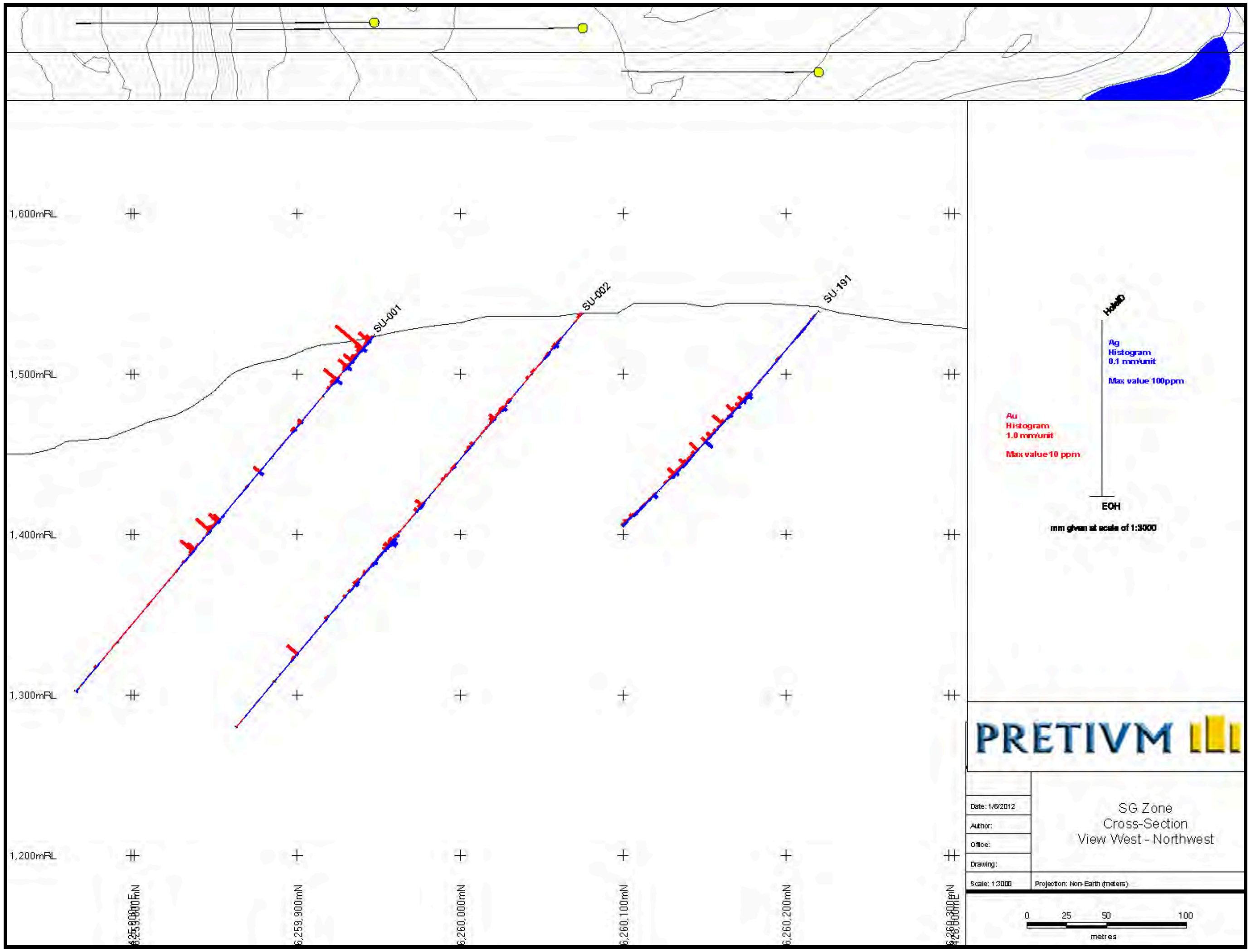


SG/Bonanza Zone

The SG/Bonanza Zone is located in the north-central part of the Property. It is underlain by an area of a gossanous sericite-altered rock on the western side of the Brucejack fault. The zone is hosted in a series of quartz stockwork vein systems close to the fault as well as in an east-striking, 150 m-long quartz stockwork. Host rocks appear mainly to be fragmental latitic to trachyandesitic rocks, likely somewhat re-worked tuff and lapilli tuff, that are intercalated with quartzo-feldspathic sandstone and minor siltstone, (Figure 7.20).

The best intercept at the SG/Bonanza Zone was in hole SU-004, which returned 1.62 g/t Au across 75 m, including 2.57 g/t Au across 27 m. This intersection contained surprisingly little quartz veining; instead, the mineralized lapilli tuff hosts only local quartz-carbonate stockwork veinlets and trace amounts of fine, acicular arsenopyrite, in addition to 1 to 3% disseminated pyrite. Pretium believes that the gold mineralization may be associated with anhedral pyrite, as is the case at the Snowfield deposit, approximately 3 to 4 km to the north. Drilling in 2011 extended the zone 50 m to the west, with hole SU-203 intersecting 24.5 m grading 1.14 g/t Au.

Figure 7.20 SG Zone Assay Cross Section



8.0 DEPOSIT TYPES

Mineralization on the Property has previously been classified as an epithermal gold-silver-copper, low-sulphidation deposit (UBC deposit model No. H04). There are certainly many features of the mineralized zones at Brucejack with characteristics of low-sulphidation deposits, such as vein mineralogy (e.g., adularia and acanthite) and the common stockwork veining and breccia-veins. These may suggest a lower temperature and shallower level of emplacement; however other features, including the general lack of evidence for colliform banding and open-space filling, have been taken to suggest deeper levels of emplacement for the veining. Furthermore, the relatively high molybdenum content at the Bridge Zone, and the fact that bulk tonnage style gold in some zones may be more closely correlated with disseminated anhedral pyrite than with veining, have been taken to suggest that at least some of the zones may be more closely allied to porphyry-style systems. Such a suggestion has some credence, particularly when one considers the common association of mineralization at Brucejack with hornblende feldspar phyric flows and fragmental rocks which are rich in groundmass potassium feldspar. Previously, these rocks have been interpreted as intrusive and therefore the mineralized zones were considered by, for example, McPherson et al. (1994), to be broadly “intrusive-related”.

Until the property-scale geologic framework is better established and there is a better understanding of the controls on the formation of the mineralized zones, Pretium is refraining from rigid adherence to, and acceptance of, a single deposit model. In the meantime, Pretium is taking steps to provide tighter constraints on the possibilities of deposit formation in the form of ongoing geochronological, petrographic, and whole rock geochemical studies, and is planning further similar studies, as well as stable isotopic and fluid inclusion work. One of the goals of such work will be to better understand what the components of the hydrothermal system were, and where the metals, water, and sulphur in the system, or systems, were derived from. The hope is that with this new information, Pretium will be better able to define a deposit model for the Brucejack mineralized zones, and will be better able to place it in a camp-scale geologic framework which will help guide future exploration at both the Brucejack and Snowfield Properties.

9.0 EXPLORATION

In September 2011, Quantec Geoscience was contracted to undertake a Spartan magnetotelluric (MT) survey of the Snowfield and Brucejack Properties. The exploration objectives were to map and detect porphyry mineralization to depth within the Snowfield and Brucejack Projects, and to establish an understanding of the geological system and fluid pathways to great depth within the Snowfield and Brucejack survey area. Results were presented in report number CA00893S, dated September 13, 2011.

There were between 56 and 57 lines surveyed at a spacing of 500 m.

The following definition of MT is taken from Quantec Geoscience's report (2011):

> *"The (MT) method is a passive/inductive method which measures the time-variations in the Earth's natural electric (E) and magnetic (H) fields to image the subsurface resistivity below the sounding site. No source or transmitter is used. The E and H fields are measured over a broad range of frequencies from 10 kHz to 1 Hz (worldwide lightning activity), and from 1 Hz to 0.001 Hz (oscillations of the Earth's ionosphere as it interacts with the solar wind). While the E and H fields are random, (solar wind and lightning activity) the ratio of the fields depends on the subsurface resistivity structure."*

The 16 scattered MT stations over the southern part of the Brucejack area provided a preliminary understanding of the resistivity contrasts. There are clear contrasts between rock types and a significant interpreted northerly structure that appears to correlate with indicated mineralized zones from the Bridge to West Zones. A second interpreted ENE structure is also apparent that may offset the northerly structure. The 16 stations are sparse coverage over 2 km^2 with significant topography, and show differences in resistivity responses between individual stations. While these data may suggest geologic features they should be used with caution. Three dimensional figures from the Quantec Geoscience report are presented in Figure 9.1 and Figure 9.2.

Figure 9.1 3D Geophysical Model According to Quantec MT Survey

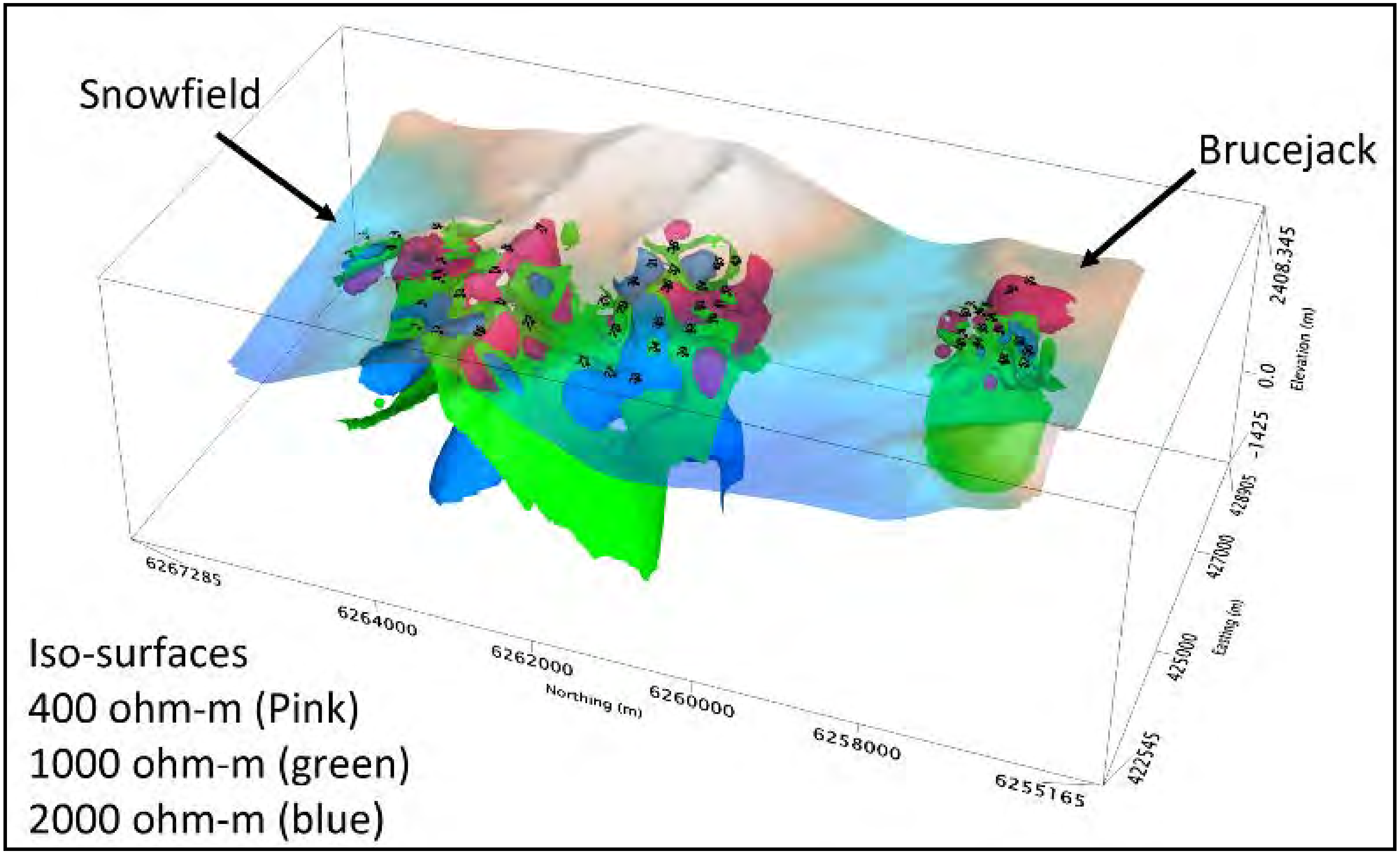
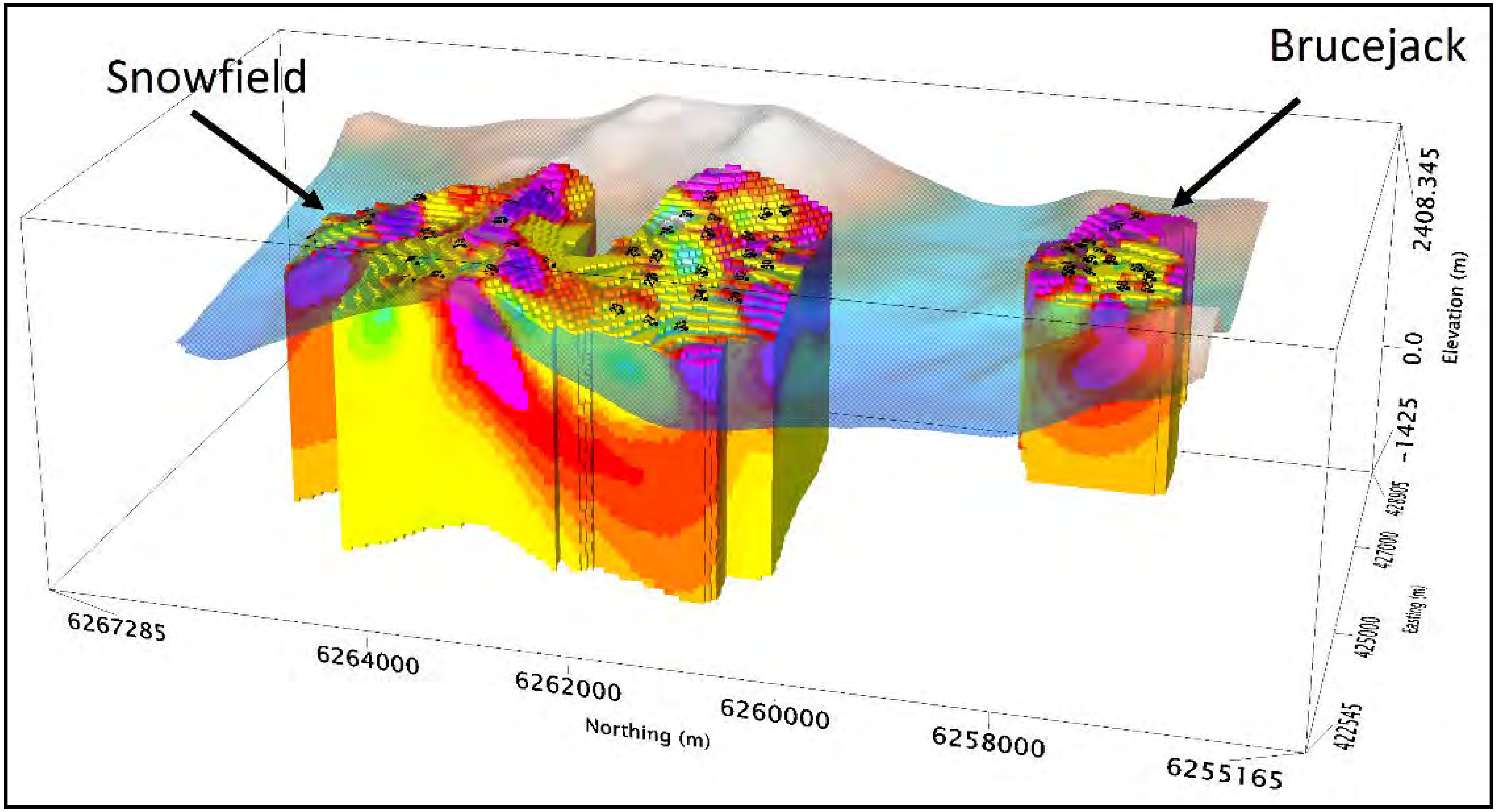


Figure 9.2 3D Geophysical Model According to Quantec MT Survey



10.0 DRILLING

In 2011, a total of 72,805 m of drilling was completed in holes SU-110 to SU-288. A drill plan map is presented in Figure 10.1. Table 10.1 presents the drill collar coordinates, and results of the drilling in 2011 are presented in Table 10.2 to Table 10.8.

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

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

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

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

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

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

Figure 10.1 2010 Brucejack Property Diamond Drill Plan

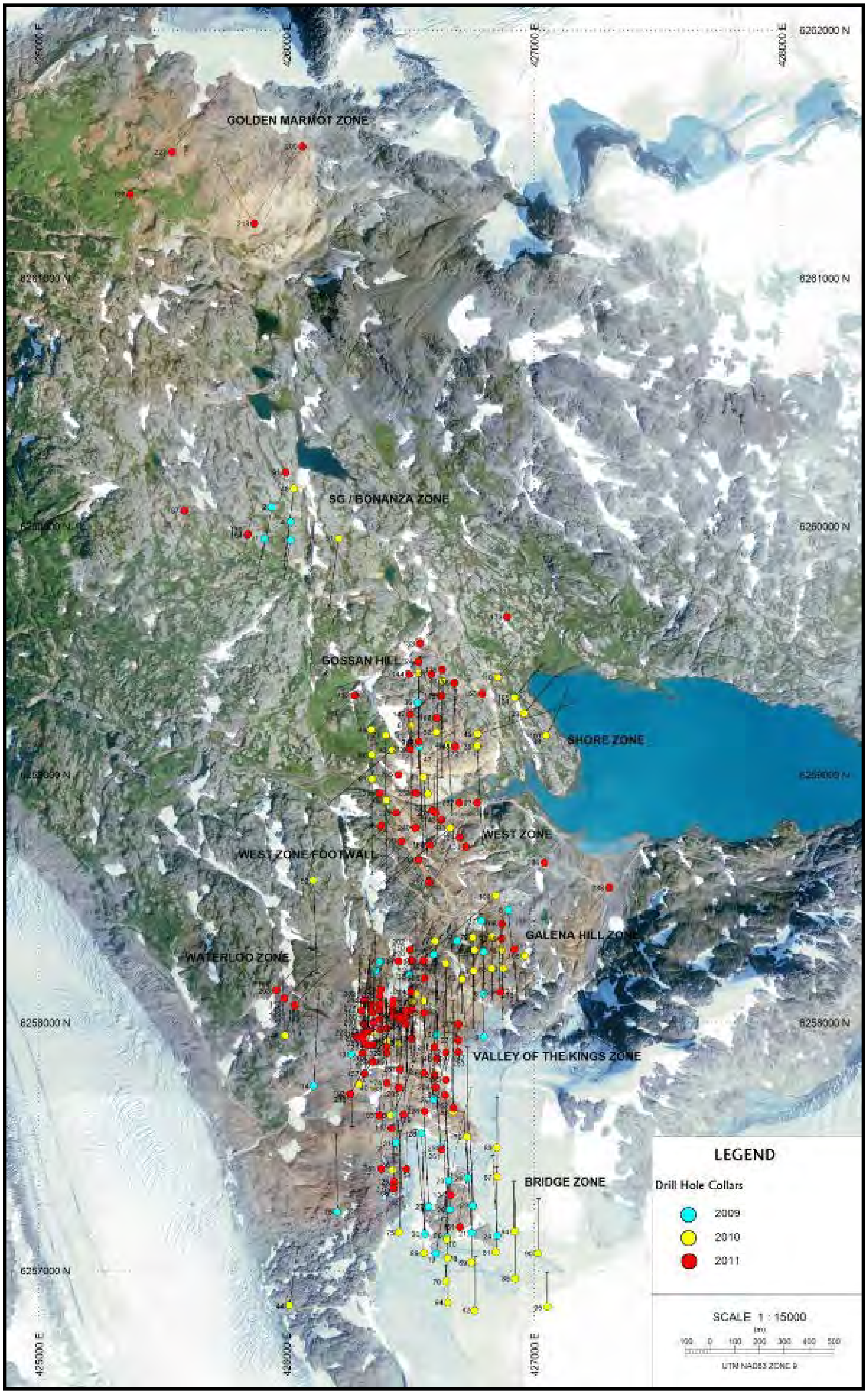


Table 10.1 Drill Collar Coordinates 2011 Drilling

Drillhole	UTM East	UTM North	Elevation (m)	Azimuth (°)	Dip (°)
SU-111	426,919	6,258,296	1,442	180	-50
SU-112	425,990	6,258,097	1,539	45	-50
SU-113	426,350	6,257,909	1,554	360	-47
SU-114	426,403	6,257,877	1,570	360	-50
SU-115	426,456	6,258,015	1,530	180	-50
SU-116	426,866	6,258,338	145	180	-50
SU-117	426,424	6,257,575	1,526	360	-50
SU-118	426,623	6,259,317	1,430	180	-52
SU-119	426,456	6,258,015	1,530	180	-65
SU-120	426,350	6,257,909	1,554	360	-59
SU-121	426,698	6,258,747	1,395	225	-50
SU-122	426,403	6,257,877	1,570	360	-65
SU-123	426,767	6,258,890	1,372	225	-50
SU-124	426,532	6,259,456	1,443	-	-50
SU-125	426,431	6,257,362	1,534	360	-50
SU-126	426,627	6,259,424	1,442	180	-50
SU-127	426,456	6,258,015	1,530	180	-44
SU-128	426,403	6,257,977	1,538	360	-50
SU-129	426,456	6,258,015	1,530	180	-57
SU-130	426,506	6,257,931	1,538	360	-50
SU-131	426,723	6,258,709	1,404	225	-50
SU-132	426,351	6,258,000	1,532	360	-47
SU-133	426,537	6,259,528	1,440	50	-50
SU-134	426,660	6,257,307	1,635	360	-50
SU-135	426,311	6,257,955	1,539	360	-50
SU-136	426,675	6,259,369	1,446	180	-50
SU-137	426,583	6,259,405	1,448	180	-50
SU-138	426,351	6,258,000	1,535	360	-56
SU-139	426,504	6,257,932	1,536	360	-65
SU-140	426,351	6,258,000	1,535	360	-65
SU-141	426,482	6,258,024	1,534	180	-50
SU-142	426,504	6,257,932	1,536	360	-57
SU-143	426,626	6,258,820	1,365	225	-50
SU-144	426,483	6,259,405	1,445	180	-50
SU-145	425,990	6,258,097	1,540	45	-65
SU-146	426,606	6,257,853	1,550	360	-50
SU-147	426,502	6,259,247	1,434	180	-50
SU-148	426,482	6,258,024	1,534	180	-65
SU-149	425,990	6,258,097	1,540	45	-44
SU-150	426,311	6,257,955	1,539	360	-65

table continues…

Drillhole	UTM East	UTM North	Elevation (m)	Azimuth (°)	Dip (°)
SU-151	426,697	6,257,175	1,646	360	-75
SU-152	426,275	6,259,325	1,383	230	-50
SU-153	426,780	6,259,310	1,405	40	-50
SU-154	426,577	6,258,717	1,390	225	-59
SU-155	426,482	6,258,024	1,534	180	-45
SU-156	425,990	6,258,097	1,540	45	-54
SU-157	426,313	6,257,794	1,530	35	-50
SU-158	426,481	6,257,412	1,533	360	-50
SU-159	425,990	6,258,097	1,540	45	-61
SU-160	426,311	6,257,955	1,539	360	-57
SU-161	426,381	6,257,412	1,523	360	-50
SU-162	426,577	6,258,717	1,390	225	-50
SU-163	426,406	6,257,756	1,542	360	-50
SU-164	426,482	6,258,024	1,534	180	-57
SU-165	426,373	6,257,625	1,515	360	-50
SU-166	426,431	6,257,327	1,536	360	-50
SU-167	426,311	6,257,955	1,539	360	-44
SU-168	426,866	6,258,396	1,433	180	-50
SU-169	426,470	6,257,628	1,540	360	-50
SU-170	426,596	6,257,787	1,576	360	-55
SU-171	426,455	6,258,028	1,532	180	-50
SU-172	426,677	6,259,116	1,452	360	-50
SU-173	426,910	6,259,660	1,389	220	-50
SU-174	426,406	6,257,756	1,542	360	-55
SU-175	426,431	6,257,327	1,536	360	-58
SU-176	426,503	6,258,252	1,521	180	-47
SU-177	426,455	6,258,028	1,532	180	-55
SU-178	426,677	6,259,116	1,452	360	-43
SU-179	425,840	6,259,970	1,535	192	-50
SU-180	426,032	6,258,068	1,537	45	-52
SU-181	426,677	6,259,116	1,452	360	-56
SU-182	426,402	6,257,902	1,566	360	-50
SU-183	426,606	6,259,220	1,449	360	-45
SU-184	425,840	6,259,970	1,535	12	-50
SU-185	426,677	6,259,116	1,452	360	-61
SU-186	426,606	6,259,220	1,449	360	-65
SU-187	425,600	6,260,070	1,580	225	-50
SU-188	426,342	6,258,041	1,525	360	-45
SU-189	426,032	6,258,068	1,537	45	-65
SU-190	426,503	6,258,252	1,521	180	-52
SU-191	425,980	6,260,200	1,550	192	-50

table continues…

Drillhole	UTM East	UTM North	Elevation (m)	Azimuth (°)	Dip (°)
SU-192	426,376	6,257,972	1,540	360	-45
SU-193	426,255	6,257,711	1,513	35	-50
SU-194	426,376	6,257,972	1,540	360	-52
SU-195	426,641	6,257,767	1,567	360	-50
SU-196	426,342	6,258,041	1,525	360	-50
SU-197	425,365	6,261,340	1,495	45	-50
SU-198	426,533	6,259,131	1,452	360	-50
SU-199	425,954	6,258,127	1,530	45	-52
SU-200	426,311	6,257,991	1,533	360	-45
SU-201	426,533	6,259,131	1,452	360	-62
SU-202	426,495	6,258,294	1,513	180	-45
SU-203	425,954	6,258,127	1,530	45	-65
SU-204	426,601	6,257,741	1,590	360	-54
SU-205	426,376	6,257,972	1,540	360	-58
SU-206	426,327	6,257,967	1,541	360	-45
SU-207	426,601	6,258,844	1,365	360	-45
SU-208	426,282	6,257,943	1,527	360	-45
SU-209	426,065	6,261,530	1,801	45	-50
SU-210	426,495	6,258,294	1,513	180	-48
SU-211	426,601	6,258,844	1,365	360	-62
SU-212	426,496	6,259,102	1,429	360	-69
SU-213	426,327	6,257,967	1,541	360	-52
SU-214	426,376	6,257,972	1,540	360	-65
SU-215	426,860	6,258,122	1,511	360	-46
SU-216	426,255	6,257,711	1,513	35	-46
SU-217	426,496	6,259,102	1,429	360	-45
SU-218	425,869	6,261,219	1,587	330	-50
SU-219	426,327	6,257,967	1,541	360	-58
SU-220	426,495	6,258,294	1,513	180	-60
SU-221	426,860	6,258,122	1,511	360	-54
SU-222	426,590	6,258,855	1,365	225	-50
SU-223	426,282	6,257,943	1,527	360	-52
SU-224	426,554	6,257,643	1,576	360	-53
SU-225	426,552	6,258,177	1,539	180	-50
SU-226	426,327	6,257,967	1,541	360	-65
SU-227	425,535	6,261,509	1,628	45	-50
SU-228	426,860	6,258,122	1,511	360	-64
SU-229	426,449	6,258,855	1,380	225	-50
SU-230	426,643	6,257,881	1,547	360	-65
SU-231	426,282	6,257,943	1,527	360	-58
SU-232	426,311	6,258,090	1,520	180	-50

table continues…

Drillhole	UTM East	UTM North	Elevation (m)	Azimuth (°)	Dip (°)
SU-233	426,520	6,258,926	1,388	225	-50
SU-234	426,502	6,258,125	1,542	180	-57
SU-235	426,430	6,258,028	1,527	180	-50
SU-236	426,552	6,258,252	1,530	180	-50
SU-237	426,643	6,257,881	1,547	360	-59
SU-238	426,453	6,258,252	1,512	180	-50
SU-239	426,311	6,258,090	1,520	180	-60
SU-240	426,520	6,258,784	1,365	225	-50
SU-241	426,502	6,258,125	1,542	180	-65
SU-242	426,643	6,257,881	1,547	360	-45
SU-243	426,430	6,258,048	1,530	180	-50
SU-244	426,311	6,258,090	1,520	180	-65
SU-245	426,449	6,258,998	1,376	225	-50
SU-246	426,553	6,257,796	1,574	360	-53
SU-247	426,692	6,257,880	1,545	360	-50
SU-248	426,430	6,258,068	1,531	180	-50
SU-249	426,507	6,258,051	1,539	180	-57
SU-250	426,347	6,257,840	1,541	360	-50
SU-251	426,313	6,257,955	1,539	360	-65
SU-252	426,641	6,257,707	1,592	360	-55
SU-253	426,692	6,257,880	1,545	360	-58
SU-254	426,507	6,258,051	1,539	180	-65
SU-255	426,311	6,257,915	1,543	360	-55
SU-256	426,430	6,258,088	1,530	180	-50
SU-257	426,454	6,257,806	1,573	360	-50
SU-258	426,457	6,258,014	1,530	180	-50
SU-259	426,347	6,257,840	1,541	360	-58
SU-260	426,481	6,257,999	1,532	180	-45
SU-261	426,623	6,257,488	1,576	360	-65
SU-262	426,674	6,257,653	1,608	360	-50
SU-263	426,305	6,257,875	1,533	360	-55
SU-264	426,376	6,258,012	1,528	180	-50
SU-265	426,454	6,257,736	1,556	360	-50
SU-266	426,481	6,258,048	1,538	180	-45
SU-267	426,596	6,257,903	1,546	360	-50
SU-268	426,327	6,257,907	1,545	360	-65
SU-269	426,374	6,258,052	1,523	180	-50
SU-270	426,379	6,258,131	1,506	360	-47
SU-271	426,692	6,257,930	1,550	360	-50
SU-272	426,305	6,257,935	1,537	360	-65
SU-273	426,554	6,258,033	1,547	180	-50

table continues…

Drillhole	UTM East	UTM North	Elevation (m)	Azimuth (°)	Dip (°)
SU-274	426,345	6,258,076	1,524	360	-45
SU-275	426,376	6,258,109	1,516	180	-43
SU-276	426,375	6,258,073	1,526	180	-50
SU-277	426,692	6,257,991	1,540	360	-50
SU-278	426,300	6,258,047	1,517	360	-43
SU-279	426,449	6,258,714	1,406	225	-50
SU-280	426,344	6,258,041	1,525	360	-90
SU-281	426,344	6,258,041	1,525	360	-90
SU-282	426,695	6,258,887	1,365	225	-50
SU-283	426,505	6,258,647	1,406	225	-50
SU-284	426,379	6,258,927	1,370	225	-50
SU-285	426,378	6,258,785	1,400	225	-50
SU-286[1]	427,041	6,258,644	-	225	-50
SU-287	426,581	6,258,568	1,410	225	-50
SU-288[2]	427,302	6,258,543	-	225	-50

Notes: 1. Mill 1 condemnation.

2. Mill 2 condemnation.

Table 10.2 SG/Bonanza Zone-Drillhole Intersection 2011

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)
SU-179	109	125.5	16.5	0.76	3.6
SU-184	63	76.5	13.5	3.02	5.0
SU-187	10	14.5	4.5	1.71	2.1
SU-191	114.5	140	25.5	0.62	2.6

Table 10.3 Bridge Zone-Drillhole Intersection 2011

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Comments
SU-117	6.5	26	19.5	1.54	3.4	
SU-125	89	100	11	16.09	31.3	One sample cut to 130 ppm Au
SU-134	114.5	215	100.5	1.21	7.4	
SU-151	3	42	39	0.99	8.0	Deep hole to 1,047 m
SU-158	12	13.5	1.5	26.7	28.6	
SU-161	61	62.5	1.5	20.2	17.1	
SU-165	39.58	96.5	56.92	0.97	4.8	
SU-166	197.5	243.5	46	2.55	5.0	One sample cut to 130 ppm Au
SU-169	51	57	6	3.02	27.7	
SU-175	112.5	144	31.5	1.38	3.7	

Table 10.4 Galena Hill Zone Drillhole Intersection 2011

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Comments
SU-111	20	27.5	7.5	1.53	7.1	
SU-116	7.5	33	25.5	3.91	42.9	
SU-168	46	167	121	1.11	12.7	
SU-215	114.5	169.5	55.0	4.70	47.5	One sample cut to 130 ppm Au
SU-221	144.0	207.0	63.0	1.04	9.7	
SU-228	76.5	85.5	9	3.07	22.3	

Table 10.5 Golden Marmot Zone Drillhole Intersection 2011

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Comments
SU-197	403.7	421.0	17.3	2.78	10.0	Exploration target
SU-209	366.5	392.5	26.0	0.77	11.5	Exploration target
SU-218	107.5	140.5	33.0	0.30	80.4	Exploration target
SU-227	186.21	192.5	6.29	1.0	3.5	Exploration target

Table 10.6 Gossan Hill/Shore Zone Drillhole Intersection 2011

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Zone	Comments
SU-118	2.13	8	5.87	1.32	5.1	Gossan Hill	-
SU-124	70	71.5	1.5	9.39	4.2	Gossan Hill	-
SU-126	9.5	14	4.5	5.19	5.8	Gossan Hill	-
SU-133	120	171	51	1.63	15.3	Gossan Hill/ Shore Zone	-
SU-136	49.5	56.19	6.69	1.79	3.4	Gossan Hill	-
SU-137	2.34	48	45.66	0.85	9.1	Gossan Hill	-
SU-144	No significant values					Gossan Hill	
SU-147	213	264.8	51.8	3.07	7.1	Gossan Hill	-
SU-152	137	161	24	1.58	5.1	Gossan Hill	-
SU-153	202	205	3	4.56	71.2	Shore Zone	-
SU-172	13.5	20	6.5	1.36	139.8	Gossan Hill	-
SU-173	228.5	258.5	30	1.27	30.4	Shore Zone	-
SU-178	10	24	14	0.55	133.2	Gossan Hill	-
SU-181	143.5	215	71.5	2.02	7.5	Gossan Hill	-
SU-183	73	86.5	13.5	1.08	2.5	Gossan Hill	-
SU-185	13.25	27	13.75	0.78	70.4	Gossan Hill	-
SU-186	79.5	107.1	27.6	0.92	2.3	Gossan Hill	-
SU-198	63.0	93.0	30.0	1.47	10.9	Gossan Hill	-

table continues…

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Zone	Comments
SU-201	52.0	53.5	1.5	168.5	94.1	Gossan Hill	Uncut
SU-207	179.0	227.5	48.5	7.7	22.3	Gossan Hill	Two samples cut to 130 ppm Au
SU-211	71.5	88.0	16.5	11.59	48.4	Gossan Hill	-
SU-212	40.0	53.0	13.0	1.68	2.3	Gossan Hill	-
SU-217	235.0	259.0	24.0	0.53	2.8	Gossan Hill	-

Table 10.7 VOK Zone Drillhole Intersection 2011

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Comments
SU-113	143.0	184.0	41.0	6.58	15.8	One sample cut to 130 ppm Au
SU-114	128	156.5	28.5	1.11	7.2	-
SU-115	58.5	79.5	21	17.00	152.1	Three samples cut to 130 ppm Au, one cut to 2,100 Ag
SU-119	40.75	71.25	30.5	3.04	8.7	One sample cut to 130 ppm Au
SU-120	157	163	6	2.28	9.8	-
SU-122	139.5	179.66	40.16	1.14	6.4	-
SU-127	58.5	73.5	15	7.20	12.0	One sample cut to 130 ppm Au
SU-128	15	26.5	11.5	22.17	42.0	One sample cut to 130 ppm Au
SU-129	43.5	72	28.5	2.47	9.9	-
SU-130	91.5	114	22.5	0.94	13.9	-
SU-132	55	98.86	43.86	3.38	27.0	One sample cut to 130 ppm Au
SU-135	75	80.5	5.5	13.49	30.6	One sample cut to 130 ppm Au
SU-138	93	94.5	1.5	14.15	202.0	-
SU-139	147.5	189	41.5	0.90	9.6	-
SU-140	90	108	18	1.15	10.5	-
SU-141	91	92.5	1.5	7.34	45.1	-
SU-142	127.5	129	1.5	5.82	7.6	-
SU-146	181	203.5	22.5	1.41	11.6	-
SU-148	105.5	122	16.5	3.40	17.0	-
SU-150	59.04	78.5	19.46	13.57	76.2	Two samples cut to 130 ppm Au, one sample cut to 2,100 Ag
SU-155	132	133.5	1.5	8.11	6.4	-
SU-157	43.5	49.5	6	37.8	50.4	One sample cut to 130 ppm Au
SU-160	30.5	45.5	15	1.14	5.2	-
SU-163	176.5	201.5	25	2.04	4.4	-
SU-164	204	263	59	1.34	7.1	-
SU-167	20.5	48.5	28	0.79	7.6	-
SU-170	88.5	91.5	3	4.88	354.6	Eastern step out
SU-171	1.61	18	16.39	1.32	15.5	-

table continues…

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Comments
SU-174	192.78	202	9.22	1.89	2.7	-
SU-176	5	46	41	1.08	8.9	-
SU-177	50	77.29	27.29	0.90	5.3	-
SU-182	107.5	139	31.5	1.69	8.9	-
SU-188	51	95	44	0.93	14.5	-
SU-190	10.62	51.5	40.88	1.03	7.0	-
SU-192	Hole caved at 60 m					
SU-194	13.5	108.5	95	1.10	10.3	-
SU-195	149.77	173.5	23.73	1.67	13.7	-
SU-196	7.5	102.9	95.4	0.75	7.7	-
SU-200	14.0	29.5	15.5	13.29	25.1	One sample cut to 130 ppm Au
SU-202	60.0	70.8	10.8	2.09	8.9	-
SU-204	41.0	44.0	3.0	13.6	13.6	-
SU-205	81.0	103.0	22.0	0.82	11.5	-
SU-206	251.5	310.0	58.5	1.02	5.7	-
SU-208	422.0	475.5	53.6	2.20	3.1	One sample cut to 130 ppm Au
SU-210	31.5	79.5	48.0	0.85	7.8	-
SU-213	110.8	185.5	74.7	1.25	8.3	-
SU-214	293.0	387.0	94.0	0.95	3.6	-
SU-216	75.5	87.5	12.0	6.14	24.3	-
SU-219	72.0	170.5	98.5	1.60	13.4	-
SU-220	85.5	106.0	20.5	0.96	15.3	-
SU-223	137.5	139	1.5	16.8	37.7	-
SU-224	5	6.5	1.5	6.52	13.4	-
SU-225	300	330	30	1.4	7.1	-
SU-226	108	222	114	1.07	11.0	-
SU-230	162.5	163.65	1.15	17.6	19.3	-
SU-231	320.88	351.5	30.62	1.74	4.0	-
SU-232	158	177.5	19.5	1.13	4.7	-
SU-234	192	207	15	7.87	9.3	-
SU-235	29.3	53.5	24.2	1.96	9.1	-
SU-236	207.2	235.5	28.3	0.85	5.1	-
SU-237	144.52	147.52	3	87.15	248.4	Two samples cut to 130 ppm Au
SU-238	172	181	9	8.97	62.9	-
SU-239	61.5	63	1.5	165.0	78.3	Uncut
SU-241	225.82	235.5	9.68	18.53	11.1	One sample cut to 130 ppm Au
SU-242	123.5	133	9.5	1.44	4.1	-
SU-243	35.5	78.79	43.29	3.53	16.9	-
SU-244	163.5	165	1.5	20.70	8.8	-
SU-246	274.61	313.5	38.89	0.80	3.8	-
SU-247	138.5	141.5	3	10.49	12.8	-

table continues…

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Comments
SU-248	110	118.32	8.32	1.57	40.7	-
SU-249	115.59	117.5	1.91	37.29	460.0	One sample cut to 130 ppm Au
SU-250	335	336.5	1.5	19.15	11.9	-
SU-251	Metallurgical hole – not sampled					
SU-252	154.86	155.86	1	15.95	11.5	-
SU-253	178.3	184.5	6.2	5.27	9.0	-
SU-254	203	224	21	3.05	8.7	-
SU-255	256	271	15	0.94	2.7	-
SU-256	122.5	148	25.5	1.6	8.7	-
SU-257	179	233	54	1.16	5.5	-
SU-258	Metallurgical hole – not sampled					
SU-259	202.5	215.5	13	11.35	27.5	-
SU-260	63.5	66.34	2.84	71.59	391.4	One sample cut to 130 ppm Au
SU-261	55.5	171	115.5	1.95	20.0	-
SU-262	24.5	26	1.5	14.55	741.0	-
SU-263	337.5	416.5	79	1.14	4.5	-
SU-264	175.5	224.5	49	0.97	8.8	-
SU-265	270	318	48	1.026	3.0	-
SU-266	246	274.5	28.5	1.28	4.6	-
SU-267	72.5	146.5	74	0.8	9.4	-
SU-268	189.5	197	7.5	3.57	15.4	-
SU-269	176	179	3	5.58	8.3	-
SU-270	1.87	15.5	13.63	5.42	8.7	One sample cut to 130 ppm Au
SU-271	98.97	102.18	3.21	4.15	8.9	-
SU-272	253	268	15	3.37	7.2	-
SU-273	106.5	129	22.5	1.10	17.1	-
SU-274	18.5	23.9	5.4	5.67	20.5	-
SU-275	110.86	111.61	0.75	9.21	84.5	-
SU-276	250	311.5	61.5	0.75	8.1	-
SU-277	264.5	285.5	21	0.70	1.4	-
SU-278	81	81.81	0.81	36.4	25.3	-

Table 10.8 West Zone Drillhole Intersection 2011

Hole #	From	To	Interval (m)	Average Au (g/t)	Average Ag (g/t)	Zone	Comments
SU-121	193.05	247.5	54.45	3.26	96.7	West Zone	-
	292	380	88	1.02	8.5	West Zone Footwall	-
SU-123	431.5	460	28.5	0.78	8.3	West Zone	-
	509.15	553	43.85	0.80	11.7		
	675.5	710.5	35	0.79	2.5		
SU-131	172.17	200.5	28.33	1.60	99.3	West Zone	-
	254	286.5	32.5	1.90	11.5	West Zone Footwall	-
SU-143	207	260.5	53.5	3.84	99.7	West Zone	-
	374.5	488.5	114	0.95	4.2	West Zone Footwall	-
SU-154	82	109.5	27.5	7.80	548	West Zone	Two samples cut to 2,100 ppm Ag
SU-162	60.5	102.5	42	3.70	363.0	West Zone	Three samples cut to 2,100 ppm Ag
SU-222	143.16	150	6.84	3.57	103.4	West Zone	-
SU-229	104.5	160	55.5	1.05	13.2	West Zone	-
SU-233	250	273.35	23.35	6.83	23.0	West Zone	-
SU-240	39.6	43.33	3.73	2.15	614.0	West Zone	-
SU-245	141	202.5	61.5	1.03	8.0	West Zone	-
SU-279	8.5	29.39	20.89	0.78	37.5	West Zone	-
SU-282	242.32	451.5	209.18	2.53	29.5	West Zone	One sample cut to 130 ppm Au
SU-283	3.3	128.5	125.2	1.21	13.7	West Zone	-
SU-284	88.5	119.5	31	0.86	2.7	West Zone	-
SU-285	3.3	283.5	280.2	0.92	6.1	West Zone	-
SU-287	36	42	6	1.84	4.3	West Zone	-

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The 2011 program on the Property used ALS Minerals as the principal laboratory. The samples that were originally sent to ALS Minerals in Terrace, BC, for sample preparation were then forwarded to the ALS Minerals facility in Vancouver, BC, for analysis.

ALS Minerals is an internationally recognized minerals testing laboratory operating in 16 countries and has an International Organization for Standardization (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 Minerals 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 g aliquot with an atomic absorption (AA) finish. In addition, a 33-element package was completed using a four acid digest and inductively coupled plasma atomic emission spectroscopy (ICP-AES) analysis, which included the silver.

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

12.0 DATA VERIFICATION

12.1 SITE VISIT AND INDEPENDENT SAMPLING 2010

The Property was visited by Mr. Fred Brown, CPG, Pr.Sci.Nat. from September 13 to 15, 2011. Independent verification sampling was done on diamond drill core, with ten samples distributed in nine 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 Minerals in Terrace, BC for analysis.

At no time, prior to the time of sampling, were any employees or other associates of Pretium 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 12.1 and Figure 12.2.

Figure 12.1 P&E Independent Site Visit Sample Results for Gold

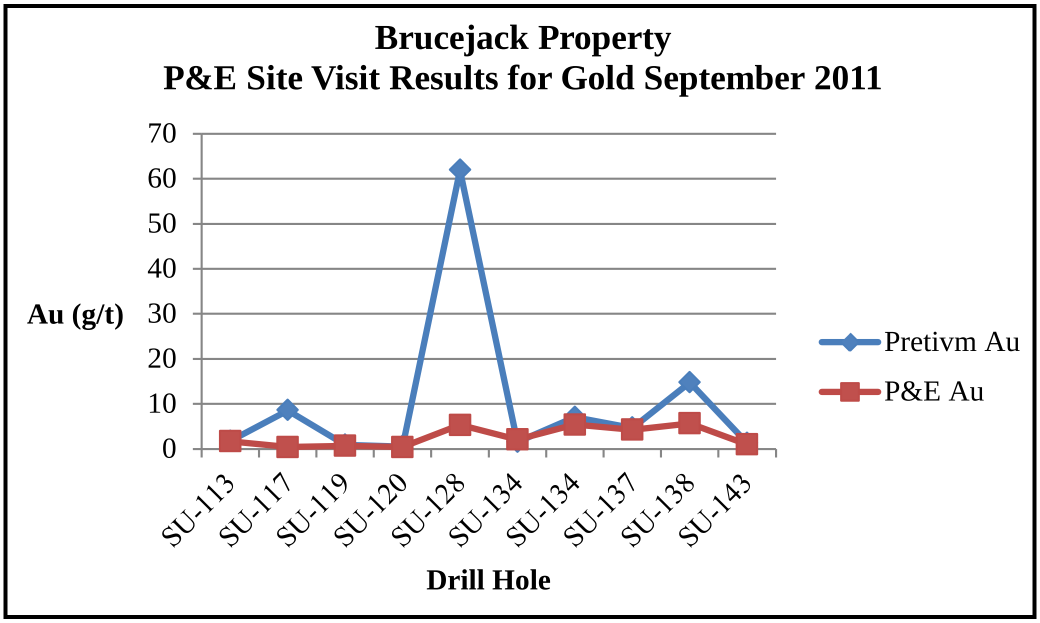
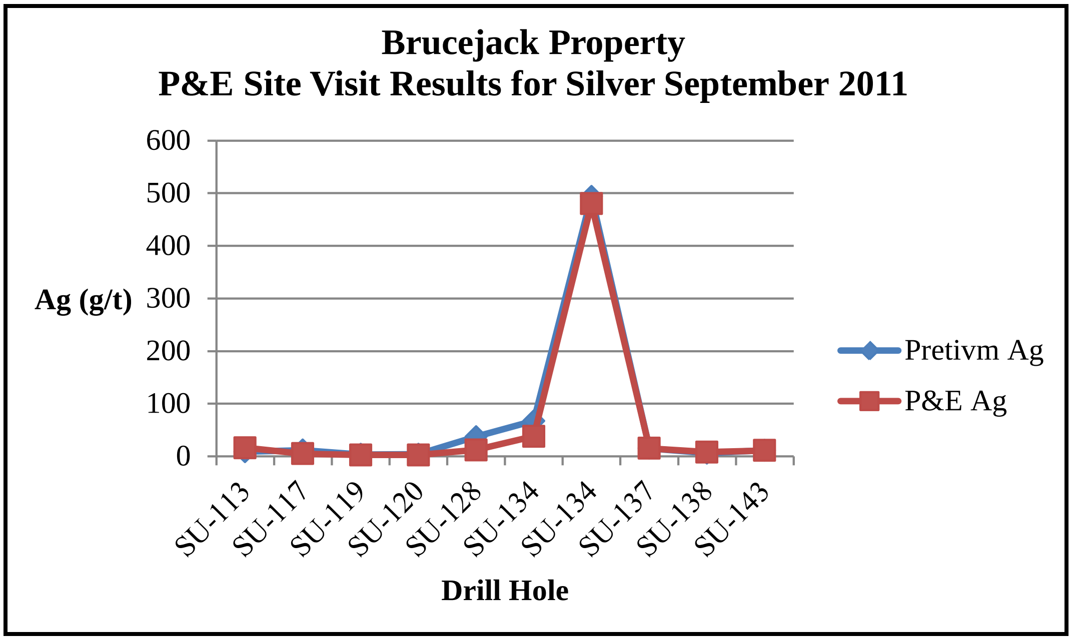


Figure 12.2 P&E Independent Site Visit Sample Results for Silver



12.1.1 PRETIUM QUALITY CONTROL

During 2011, data were entered into a data capture tool suited for import to a database. Data were then sent digitally to Caroline Vallat at GeoSpark Consulting Inc. (GeoSpark).

Updated sampling data were provided regularly, including down hole primary sample details as well as reference to duplicate samples and standards and blanks inserted throughout the sampling. The drillhole header, survey, and down hole attribute data were also provided on a regular basis.

To the best of GeoSpark's knowledge the sample data were provided in their original digital state, and there is no reason to suspect any data tampering.

Data provided were loaded to a relational database suited for QA/QC on the analytical results reported by ALS Minerals, in Vancouver. The database was reviewed regularly to ensure that the data remained functional for use by eliminating down hole interval overlaps, records beyond end of hole depths, addressing any data entry issues related to the down hole attribute codes, etc. Updates were also made to the sampling table wherever quality assurance and quality control measures revealed data entry issues related to the sample identities or details. For example, there were some issues where the wrong standard identity was entered and the returned analytical results clearly revealed the correct standard identity. All updates to sample identities were tracked within the database.

All analytical results were entered directly from ALS Minerals and SGS Canada (SGS) assay certificates to a database managed by GeoSpark. The analytical results were then provided to Pretium personnel for use internally.

The analytical records in the 2011 Brucejack database exist as they were provided from the labs.

The data were provided as they were originally produced and there was no need to perform manual verification.

Ongoing review of the analytical results took place in order to remedy any suspect analytical results. Re-analyses were requested from the lab whenever there was a question of the accuracy of results reported. In addition internal lab repeat and field duplicates were reviewed in order to monitor the repeatability or precision of results reported.

Any re-analyses were further reviewed and the results were assigned to the primary samples which were denoted with a suffix of “R” meaning re-run.

Duplicates, standards, and blanks were inserted approximately every twentieth sample and amount to 5.86%, 5.76%, and 5.98% relative to the total number of primary samples submitted to ALS Minerals. In addition a representative set of check samples was submitted to SGS for analysis using similar analytical methods and techniques. The check samples serve to define any bias in the primary results.

This amount of QA/QC data is sufficient to represent the quality of the sample analytical results reported by ALS Minerals.

Ongoing documentation of the analytical result QA/QC was provided to Pretium.

A summary QA/QC report was generated relating to the 2011 Project analytical results was written by Caroline Vallat and is dated December 22, 2011. An excerpt from the Vallat report concludes that:

> *"With consideration of inhomogeneity within the Project mineralization as a function of narrow high-grade vein mineralization and thorough review of the analytical results reported on field duplicates, a satisfactory level of precision has been inferred for the primary sample results. Additionally, initial review of the internal lab repeat results at the time of analysis reporting has increased the confidence in results reported by ALS.*
>
> *Mineral concentrations reported on standard and blank materials have been consistently monitored in order to remove any concern of local contamination or instrumentation issues. The detailed review of the standard and blank results has inferred that there is strong accuracy in the primary sample results reported by ALS.*

> *Check sampling has shown that there is no need for concern of bias in the primary sample results reported.*
>
> *The quality assurance and quality control measures taken and addressed herein have allowed for overall confidence in the analytical results reported for the 2011 Project data."*

Upon finalization of the 2011 Project database maintained by GeoSpark, the data were provided directly to P&E.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 METALLURGICAL TEST WORK REVIEW

13.1.1 INTRODUCTION

The 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 Project are gold and silver. The test work 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 test work during 2009 and early 2011 and historical test work conducted between 1988 and 1990 for the feasibility study by CESL in March 1990. Currently, Pretium is conducting a comprehensive metallurgical test program to further assess the metallurgical performance of the mineralization to support the feasibility study. The ongoing test results are not included in this test work review.

Tetra Tech has reviewed the test work and summarizes the results in the following sections.

13.1.2 HISTORICAL TEST WORK

The historical test work 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 to 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 13.1.

Table 13.1 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 test work 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 (less than 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 test work 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 13.2.

Table 13.2 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

13.1.3 2009-2011 Test Work

The Metallurgical Division at Inspectorate America Corp. (Inspectorate), previously Process Research Associates Ltd. (PRA), carried out preliminary metallurgical test work 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 test work 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

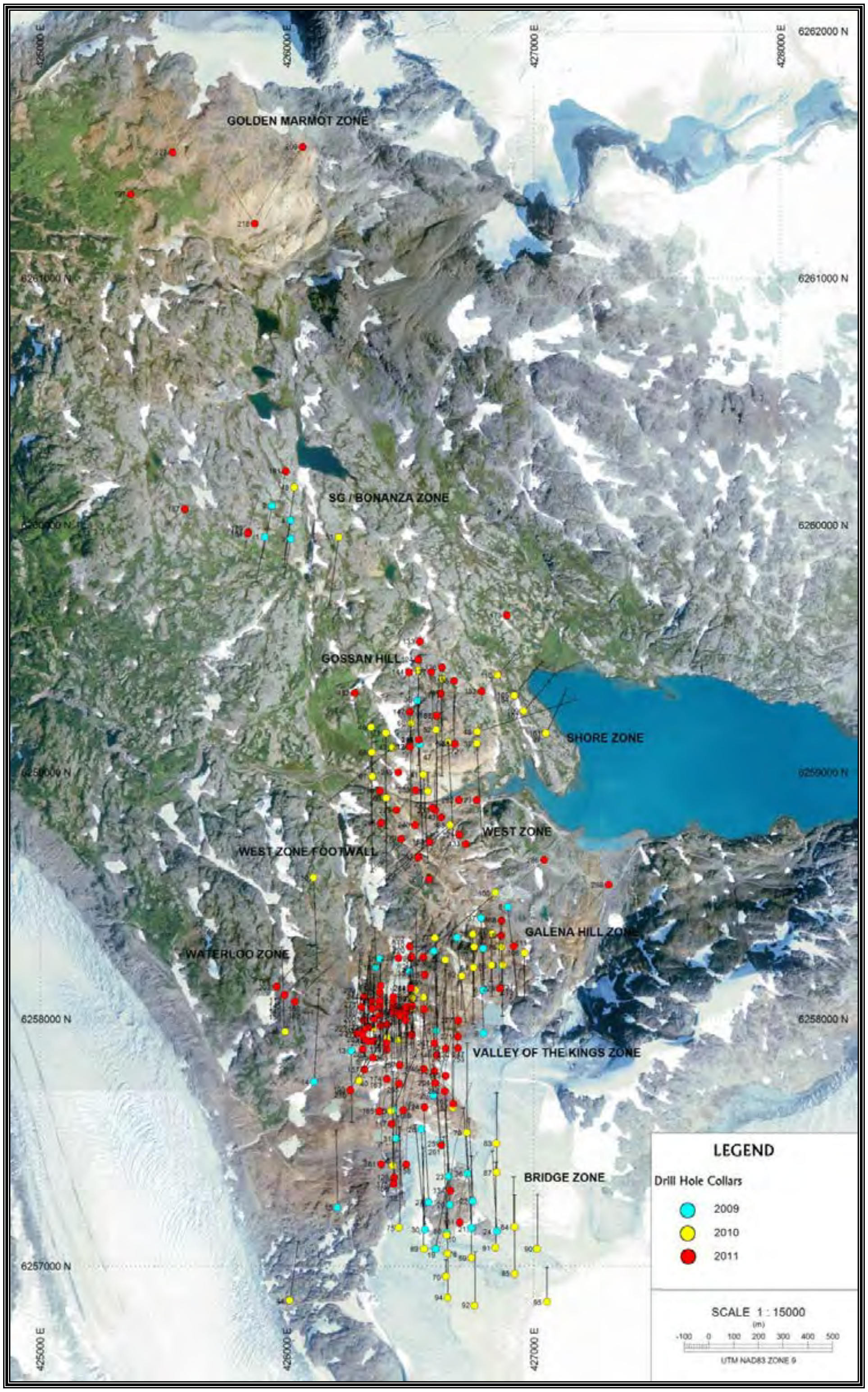
2009-2010 Test Samples

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 drillholes distribution is presented in Figure 13.1.

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 13.1 Drillhole Distribution



2010-2011 Tests Samples

The composite samples prepared in September 2010 were originally from the West Zone, Galena Hill and Bridge Zone. The drillhole interval samples from the Galena Hill and Bridge Zones were grouped into high and low grade composite, respectively. However, the test work 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 drillhole, zone and individual sample identifications are presented in Table 13.3.

Table 13.3 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 13.4. 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 13.4 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 (%)
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 13.5.

As shown in Table 13.4 and Table 13.5, the gold contents in most of the samples tested are lower than the projected mill feed grades.

Table 13.5 Metal Contents of Composite Samples (2010-2011)

Composite	Head Grade Au (g/t)	Head Grade Ag (g/t)
Composite GH2	4.93	52.9
Composite BZ2	0.91	7.7
Composite WZ1	1.79	25.4
Composite SU-98	73.30	2.5
Composite SU-76B	12.60	13.0
Composite SU-32C	11.00	10.4
Composite SU-32A	3.80	25.8
Composite SU-33	3.68	22.1

GRINDABILITY TEST WORK

Table 13.6 presents the BWI obtained from the Brucejack mineralization. It appears that, on average, the mineralization is moderately hard.

Table 13.6 Grindability Test Results (2009-2010)

Sample ID	BWI (kWh/t)
BZ	16.4
GH	15.6
R8	16.2

SAMPLE SPECIFIC GRAVITY

The specific gravity (SG) of the Brucejack mineral samples are shown in Table 13.7. The SG data varied narrowly from 2.71 to 2.84.

Table 13.7 Sample Specific Gravity (2009-2010)

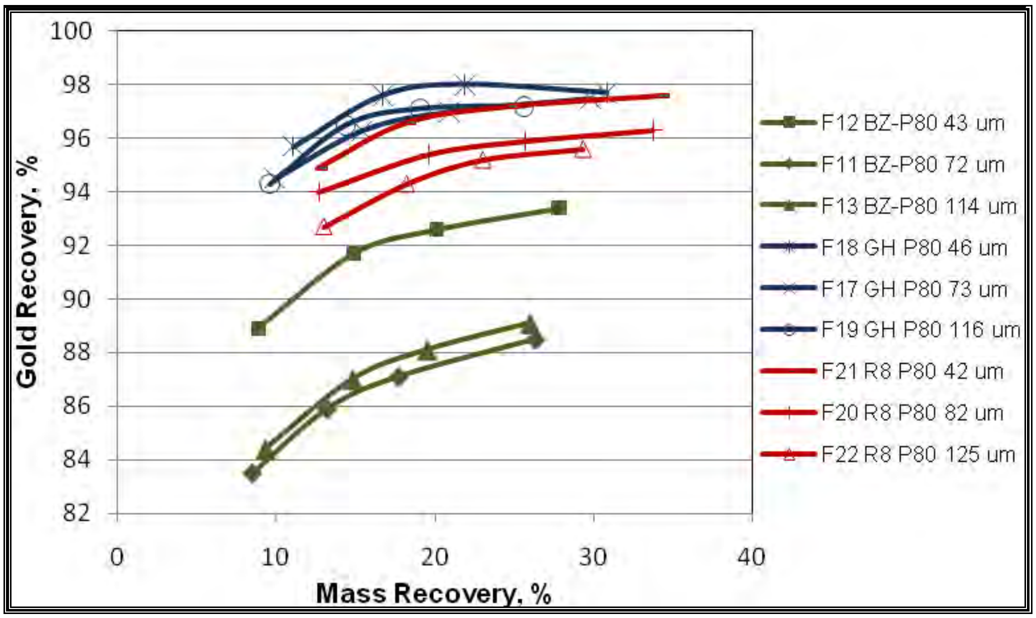
Sample ID	SG		Sample ID	SG
SU-4	2.79		SU-25	2.71
SU-5	2.74		SU-27	2.74
SU-6A	2.82		SU-032A	2.73
SU-6B	2.84		SU-032B	2.73
SU-10	2.76		SU-032C	2.72
SU-19	2.76		SU-033	2.78
SU-21A	2.75		SU-036A	2.82
SU-21B	2.77		SU-036B	2.78

FLOTATION TEST WORK

Primary Grind Size

Three different primary grind sizes were tested on the various composite samples in 2009 to 2010 and 2010 to 2011 testing programs. Potassium amyl xanthate (PAX) and A208 were used as collectors -- methyl isobutyl carbinol (MIBC) as the frother, and copper sulphate as the activator (at scavenger flotation only). The 2009 to 2010 test results are shown in Figure 13.2. 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 13.2 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 to 2011 testing program. The effect of primary grind size on gold and silver recoveries is shown in Figure 13.3 and Figure 13.4, 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 13.3 Effect of Primary Grind Size on Gold Recovery (2010-2011)

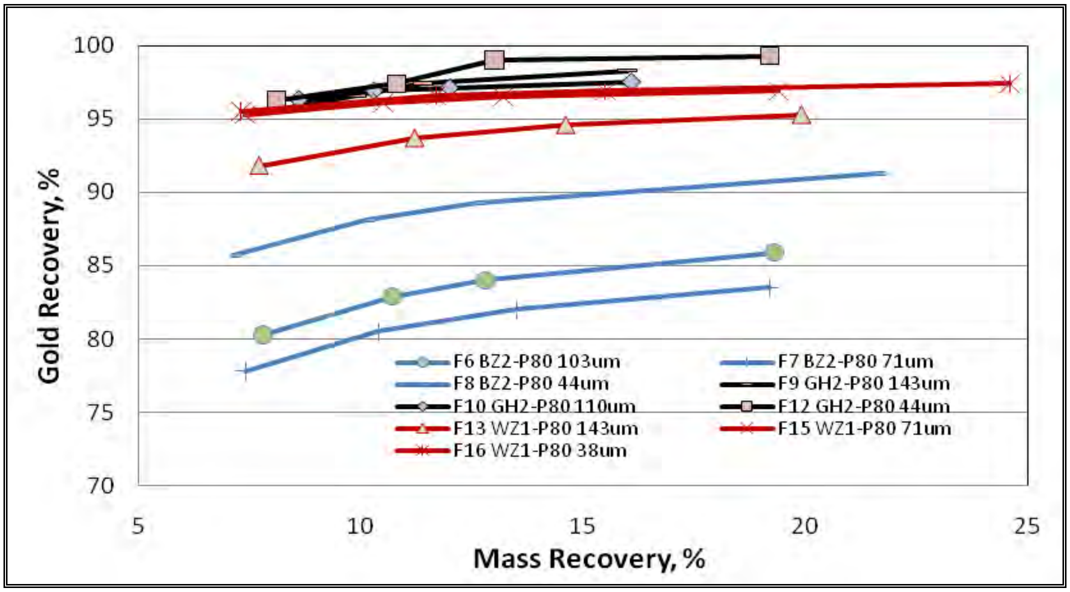
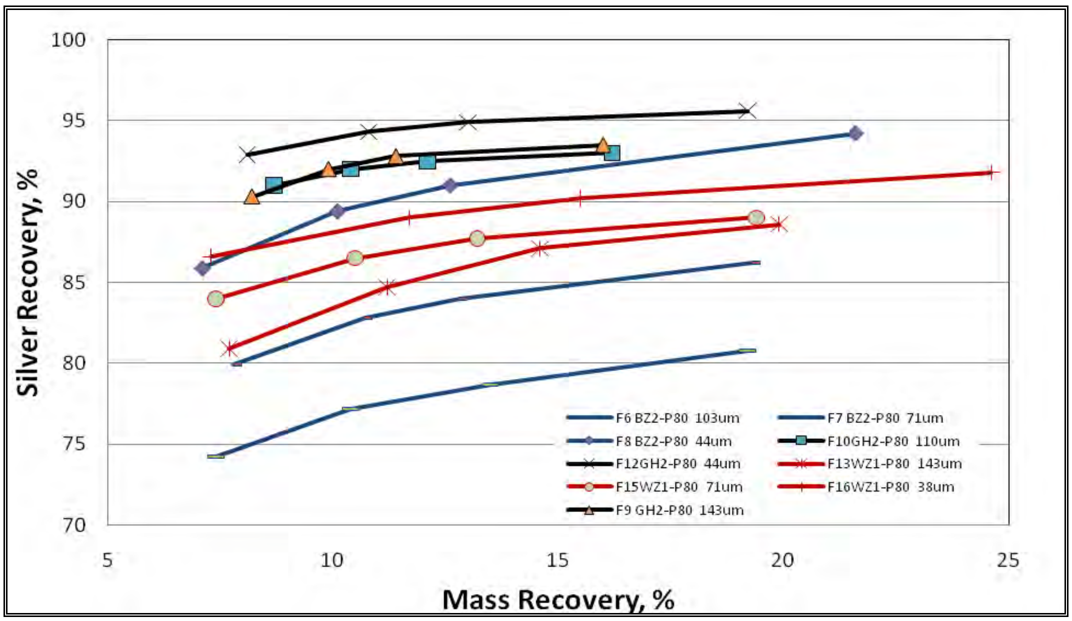


Figure 13.4 Effect of Primary Grind Size on Silver Recovery (2010-2011)

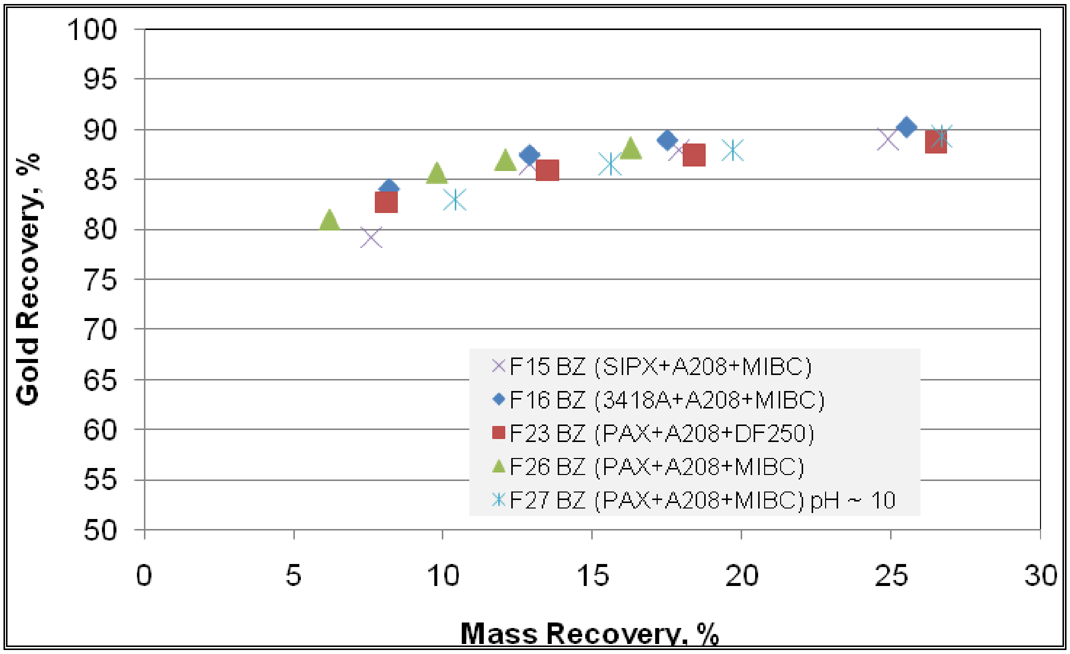


It appears that silver recoveries are lower than gold recoveries for the Galena Hill Zone and the West Zone samples. However, for the Bridge Zone sample, the difference in gold and silver recoveries was much smaller.

Reagents and Slurry pH

The 2009 to 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 13.5. It appears that the effect of the reagents and slurry pH on the gold recovery was not significant.

Figure 13.5 Effect of Reagent and Slurry pH on Gold Recovery (2009-2010)

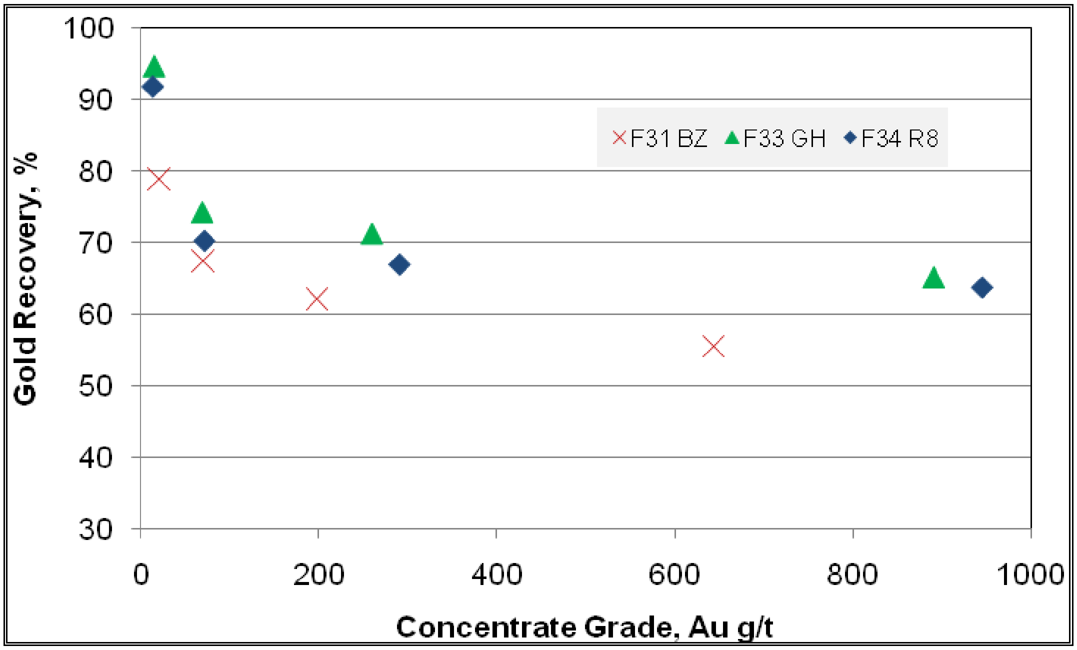


Note: * Test F27 was conducted at a higher pH, the others at natural pH.

Cleaner Flotation Test Work

The 2009 to 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 13.6, the gold recovery reduced significantly at the first cleaner flotation stage.

Figure 13.6 Effect of Cleaner Flotation on Gold Recovery (2009-2010)



Gravity Concentration Test Work

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 13.8 and Table 13.9.

As shown in Table 13.8, 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 13.9, 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 13.10 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 13.8 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 13.9 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 13.10 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 Test Work

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 13.11. 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 13.11 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 13.12.

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).

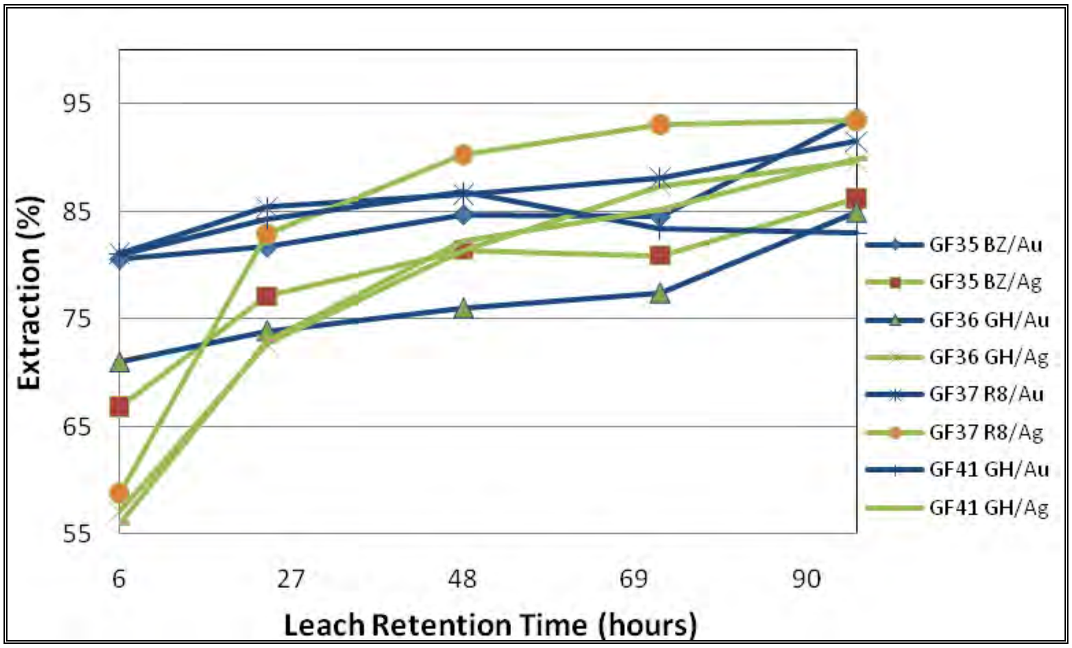
Table 13.12 Concentrate Cyanidation Test Results (2009-2010)

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 to 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 13.7 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 13.7 Bulk Concentrate Leaching Retention Time Test Results (2010 to 2011)



Gravity + Flotation + Cyanidation Test Work

According to the finding of the preliminary test work, 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:

1. primary grind, gravity concentration, rougher/scavenger flotation, and regrind on the flotation concentrate, followed by cyanidation on the reground concentrate (Flowsheet A)
2. 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)

3. 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 13.13, Table 13.14, and Table 13.15, for the three different combinations.

Table 13.13 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	-	-

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

As shown in Table 13.13, the flotation and gravity concentration recovered approximately 94% 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 13.14 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 13.14, 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 13.15, 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 13.16, 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 13.15 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 SU98[1]					
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 13.16 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 13.17.

Table 13.17 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		Grind Size
	Au (g/t)	**Ag (g/t)**	**Au (%)**	**Ag (%)**	
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 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 13.8 and Figure 13.9.

Figure 13.8 Variability Test Results – Gold Metallurgical Performance (2010-2011)

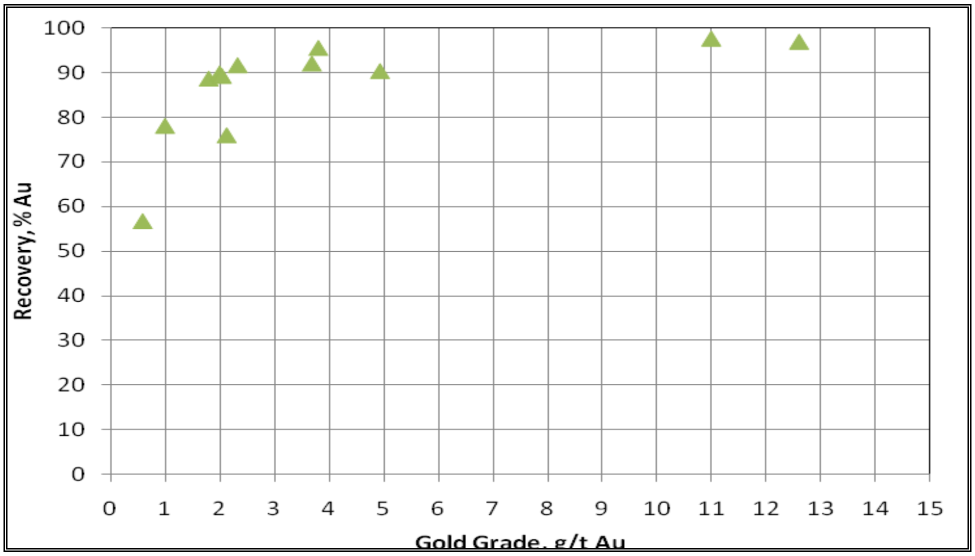
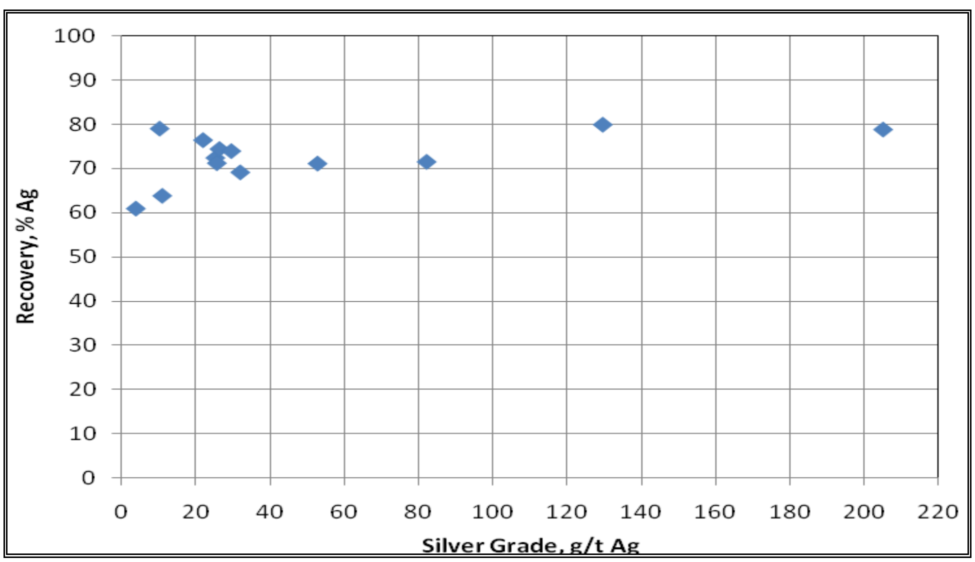


Figure 13.9 Variability Test Results – Silver Metallurgical Performance (2010-2011)



Conclusions

The review of preliminary test work 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 test work have not yet been optimized. Excess cyanide was used to ensure cyanide dosage was not a limiting factor in evaluation of previous metal recovery.

RECOMMENDATIONS

Further test work is recommended to:

- confirm the findings of the test work 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 26.0 provides more detailed recommendations.

13.1.4 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 13.18. There are no test results available for review for the mineralization from the VOK Zone. The metallurgical performance of the VOK mineralization is assumed to be similar to the mineralization from the Galena Hill and Bridge Zones because the VOK Zone is between the Galena Hill and Bridge Zones. Further test results are required to better project the metallurgical performance of these mineralization.

Table 13.18 Projected Metallurgical Performance

Year	Annual Process Rate (t)	Mill Feed Grade Au (g/t)	Mill Feed Grade Ag (g/t)	Doré Recovered Metals Au kg	Doré Recovered Metals Au ('000 oz)	Doré Recovered Metals Ag kg	Doré Recovered Metals Ag ('000 oz)	Recovery Au (%)	Recovery Ag (%)
1	200,000	8.2	22.7	1,513	49	3,242	104	92.5	71.3
2	402,554	9.9	88.5	3,719	120	26,402	849	93.2	74.1
3	458,386	14.6	78.3	6,363	205	26,439	850	94.8	73.7
4	482,642	25.6	23.4	11,904	383	8,076	260	96.5	71.4
5	506,906	32.2	47.5	15,764	507	17,436	561	96.5	72.4
6	508,127	16.4	87.9	7,911	254	33,075	1,063	95.2	74.1
7	522,737	13.8	34.2	6,825	219	12,842	413	94.5	71.8
8	524,101	28.3	23.6	14,328	461	8,826	284	96.5	71.4
9	540,000	22.0	18.5	11,458	368	7,120	229	96.5	71.2
10	540,000	22.2	18.5	11,556	372	7,092	228	96.5	71.2
11	540,000	25.7	21.0	13,408	431	8,065	259	96.5	71.3
12	533,176	32.1	19.1	16,527	531	7,254	233	96.5	71.2
13	488,210	35.6	17.0	16,784	540	5,890	189	96.5	71.1
14	520,165	13.5	12.0	6,623	213	4,407	142	94.4	70.9
15	540,000	19.0	24.6	9,847	317	9,487	305	95.8	71.4
16	540,000	24.4	38.2	12,703	408	14,862	478	96.5	72.0
17	540,000	16.0	50.7	8,229	265	19,838	638	95.1	72.5
18	540,000	14.1	21.0	7,230	232	8,100	260	94.6	71.3
19	540,000	13.3	36.1	6,803	219	14,004	450	94.4	71.9
20	540,000	14.8	85.2	7,557	243	34,022	1094	94.8	74.0
21	540,000	15.3	122.3	7,849	252	49,843	1602	94.9	75.5
22	540,000	8.2	164.2	4,098	132	68,514	2203	92.5	77.3
23	540,000	7.3	193.5	3,641	117	82,015	2637	92.1	78.5
24	200,564	7.0	310.0	1,281	41	52,842	1699	91.9	85.0
Total	**11,827,566**	**18.9**	**59.3**	**213,921**	**6,878**	**529,693**	**17,030**	**95.7**	**75.5**
Average	**-**	**18.9**	**59.3**	**-**	**-**	**-**	**-**	**95.7**	**75.5**

14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

This mineral resource estimate has been prepared in accordance with the guidelines of the Canadian Securities Administrators' NI 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 drillholes."
- Indicated Mineral Resource: "An 'Indicated Mineral Resource' is that part of a mineral resource for which quantity, grade or quality, densities, shape and physical characteristics, can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed."
- Measured Mineral Resource: "A 'Measured Mineral Resource' is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity."

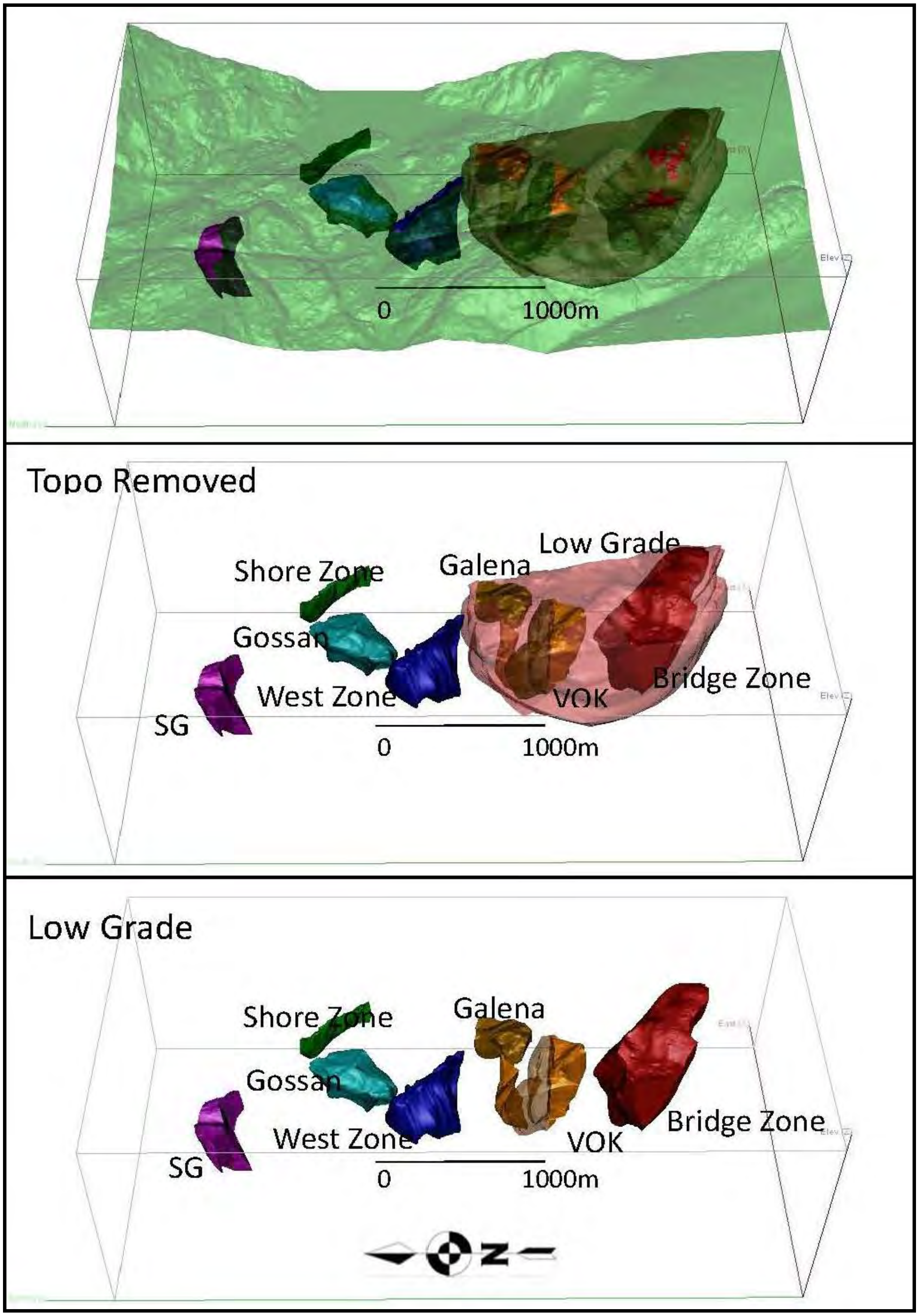
Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no guarantee that all or any part of the mineral resource will be converted into mineral reserves. Confidence in the estimate of Inferred Mineral Resources is insufficient to allow the meaningful application 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 Mr. Fred Brown, CPG, Pr.Sci.Nat., and Mr. Eugene Puritch, P.Eng., of P&E, independent QPs in terms of NI 43-101. This mineral resource estimate is based on information and data supplied by Pretium. A draft copy of this report was reviewed by Pretium 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 eight distinct modelled mineralization domains, namely the West Zone, Shore Zone, Gossan Hill Zone, Galena Hill Zone, SG Zone, VOK Zone, Bridge Zone and a Low Grade Halo (Figure 14.1). The VOK Zone has been further divided into three sub-domains. The effective date of this estimate is November 28, 2011.

Figure 14.1 Isometric Projection of the Brucejack Mineralization Zones



14.2 PREVIOUS RESOURCE ESTIMATES

A previous public mineral resource estimate for the Brucejack deposits dated April 16, 2001 was prepared by Pincock Allen & Holt[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 14.1), based on a gold-equivalent (AuEq) cut-off derived from a silver:gold equivalency ratio of 66:1.

Table 14.1 Pincock Allen & Holt April 16, 2001 Mineral Resource Estimate

Zone	Class	AuEq Cut-off (oz/t)	Tonnes ('000)	Au g/t	Ag g/t	Au ('000 oz)	Ag ('000 oz)
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**

Note: M = Measured, I = Indicated

A mineral resource estimate dated December 1, 2009 for the Brucejack deposit was prepared by P&E[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 14.2) using a gold equivalent cut-off of 0.35 g/t. The estimate was based on the results of 844 drillholes and constrained within an optimized conceptual pit shell.

Table 14.2 Combined Mineral Resource Estimate at a 0.35 g/t AuEq Cut-off[1, 2, 3]

Class	Tonnes (Mt)	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., effective date December 1, 2009.

A mineral resource estimate dated February 18, 2011 for the Brucejack deposit was prepared by P&E[3]. This mineral resource estimate reported a Measured and Indicated mineral resource of 8.18 Moz Au and an Inferred mineral resource of 12.56 Moz Au (Table 14.3) using a gold equivalent cut-off of 0.30 g/t. The estimate was based on the results of 908 drillholes and constrained within an optimized conceptual pit shell.

Table 14.3 Combined Mineral Resource Estimate at a 0.30 g/t AuEq Cut-off[1, 2, 3]

Class	Tonnes (Mt)	Au g/t	Ag g/t	Au (Moz)	Ag (Moz)
Measured	11.7	2.25	75.56	0.846	28.423
Indicated	285.3	0.80	9.57	7.338	87.782
Measured + Indicated	297.0	0.86	12.17	8.184	116.205
Inferred	542.5	0.72	8.67	12.558	151.220

14.3 SAMPLE DATABASE

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

The supplied databases contain records for 1,182 drillholes (Table 14.4). Of these, 108 drillholes fall outside the block model limits or had no reported assay data. Drillhole records consist of collar, survey and assay data. Assay data fields consist of the drillhole ID, downhole interval distances, sample number, gold grade and silver grade. All data are in metric units and all collar coordinates were converted by Pretium to the UTM NAD83 system.

Table 14.4 Brucejack Drilling Database Records

Data Type	Record Count	Total (m)
Historical Surface Drilling	452	60,853.83
Historical UG Drilling	442	33,749.90
Surface Drilling	288	124,333.90
Total	**1,182**	**218,937.63**

[3]Technical report and updated resource estimate on the Brucejack property, P&E Mining Consultants Inc., dated February 18, 2011.

The database contains a total of 124,765 non-zero gold assays and 123,829 non-zero silver assays. Due to the varying assay protocols in use during different project phases, all gold assays identified as below detection limits or less than 0.003 g/t were converted to a grade of 0.0025 g/t, and silver assays identified as below detection limits or less than 0.50 g/t were converted to 0.25 g/t.

Pretium also provided an AutoCAD format wireframe of the historical underground mining development at the West Zone. Pretium used historic mine plans to digitize the underground development. Underground workings were digitized on 44 east-west sections in the mine grid coordinate system using AutoCAD software. Section lines were generally spaced every 10 m, with a reduction to 5 m spacing in areas of more complex development (i.e., in areas of multiple tunnels, junctions etc.). The digitized data were then used to generate a single 3D solid to represent the underground workings.

14.4 DATABASE VALIDATION

Industry standard validation checks were completed on the supplied database, and minor corrections made where necessary. 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 drillhole 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 Pretium with a Reflex EZ-Shot® magnetic instrument. For the Pretium drilling, survey measurements were taken on average every 46 m. Downhole survey data were examined by P&E for significant deviations and minor corrections made where necessary.

P&E believes that the databases are suitable for mineral resource estimation.

14.4.1 TOPOGRAPHIC CONTROL

Aerial photography specialists Aero Geometrics were contracted by Pretium to produce a topographic map of the Property. Using high-resolution a photo-mosaic was first made of the Brucejack and adjoining Snowfield properties. Using this photo-mosaic and elevation data obtained from 1:50,000 scale national topographic maps published in 1979 by the Surveys and Mapping Branch of the Department of Energy, Mines & Resources, Aero Geometrics digitally generated a contoured topographic map with contour lines spaced at two-meter intervals and presented this map as digital elevation model (DEM) in AutoCAD format. In order for this topographic map to be consistent with the NAD27, Zone 9 UTM grid system being used by Pretium for the project, it was necessary to make minor 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 drillhole collars, the western shoreline of Brucejack Lake, historic mine grid stations) that had been surveyed by McElhanney field crews in 2009. The DEM was subsequently converted to NAD83.

Sub-glacial bedrock topography was prepared by BGC based on ice radar surveys, drillhole data and glacial geomorphology.

14.4.2 Bulk Density

A total of 1,038 bulk density measurements were provided by Pretium, with an average bulk density of 2.78 t/m^3 (Table 14.5). Bulk density measurements were measured from core samples by ALS Minerals, and were used to create a bulk density model for the mineral resource estimate.

Table 14.5 Brucejack Bulk Density Statistics

	Units	Total
Count		**1,038**
Minimum	t/m^3	**2.49**
Maximum	t/m^3	**3.71**
Average	t/m^3	**2.78**
Median	t/m^3	**2.79**
Standard Deviation	-	**0.10**

14.5 Brucejack Domain Modelling

Several discrete mineralization domains at Brucejack have been identified by Pretium, with the West Zone, Shore Zone and VOK Zone considered by Pretium 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.

The overall trend of mineralization in the West Zone and Shore Zone is ~135°, and modelling for these domains was generated from successive polylines oriented perpendicular to the trend of the mineralization. The outlines of the polylines were defined by the selection of mineralized material at or above 0.50 g/t Au with apparent continuity along strike and down dip. In some cases mineralization below 0.50 g/t Au was included for the purpose of maintaining continuity. All polyline vertices were snapped directly to drillhole assay intervals. A true 3D model was then created by combining successive polylines into a triangulation wireframe.

The general trend of the VOK mineralization has been identified by Pretium geologists as predominantly 90°, with internal higher-grade zones associated with Reidel shear structures. Two distinct domains were modelled for the VOK from successive polylines spaced every 20 m perpendicular to the VOK trend. The outlines of the polylines were defined by the selection of mineralized material at or above 0.50 g/t Au with demonstrated continuity along strike and down dip. In some cases mineralization below 0.50 g/t Au was included for the purpose of maintaining continuity. The southernmost VOK Zone was then visually partitioned into a separate high-grade sub-domain and a low-grade sub-domain to better reflect the large number of high-grade intersections within this domain. All polyline vertices were snapped directly to drillhole assay intervals. True 3D models of the domains were then created by combining successive polylines into triangulation wireframes.

For the Gossan Hill Zone, Galena Hill Zone, SG Zone and Bridge Zone domains the mineralization models were generated from successive polylines oriented north-south. The outlines of the polylines were defined by the selection of mineralized material at or above 0.50 g/t Au with demonstrated continuity along strike and down dip. In some cases, mineralization below 0.50 g/t Au was included for the purpose of maintaining continuity. All polyline vertices were snapped directly to drillhole assay intervals. A true 3D model was then created by combining successive polylines into a triangulation wireframe.

In order to ensure that all potentially economic mineralization was captured for mineral resource estimation, a secondary low-grade mineralization halo was subsequently modelled using a gold 0.2 g/t grade shell around the Bridge Zone, VOK and Galena domains. A 3D model of the low-grade mineralization domain was then created by combining successive polylines into a triangulation wireframe.

14.5.1 COMPOSITING

Assay sample lengths within the defined mineralization domains vary from 0.01 to 12.87 m, with an average sample length of 1.46 m. A compositing length of 2.00 m was selected in order to ensure equal sample support for estimation. Length-weighted composites were calculated for gold and silver within the defined mineralization domains. Missing sample intervals in the historical data were treated as null values. The compositing process started at the first point of intersection between the drillhole and the domain intersected, and halted upon exit from the domain wireframe. End-of-run composites that were less than 1.00 m in length were discarded so as not to introduce a short sample bias into the estimation process. The wireframes that represent the interpreted mineralization domains were also used to back-tag a rock code field into the drillhole workspace, and assays and composites were assigned a rock code value that was based on the domain wireframe within which the interval midpoint fell. The composite data were visually validated against domain wireframes and exported to extraction files for data analysis and grade estimation.

14.5.2 SUMMARY STATISTICS

A total of 94,518 assay samples fall within the defined mineralization domains, including thirty-two assay samples of 1,000 g/t Au or higher. Summary assay statistics were calculated separately by domain (Table 14.16).

Summary uncapped composite statistics were also calculated by domain for each commodity (Table 14.17).

Table 14.6 Brucejack Summary Assay Statistics

	Total	West Zone	Shore Zone	Gossan	Galena	SG	Bridge Zone	Low Grade	VOK 211	VOK 212	VOK 220
Gold											
Minimum (g/t)	0.0025	0.0025	0.0100	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025	0.0025
Maximum (g/t)	18,754.5	2,341.4	9,490.9	1,550.0	2,490.0	18.5	458.0	208.0	18,754.5	10,000.0	5,850.0
Mean (g/t)	3.00	2.63	9.36	1.37	2.01	0.76	0.89	0.35	7.68	3.87	7.82
Standard Deviation	119.24	24.52	238.81	22.47	41.29	1.49	6.83	2.94	249.63	148.82	192.85
Coefficient of Variation	39.80	9.31	25.50	16.37	20.57	1.97	7.68	8.53	32.51	38.46	24.67
Total Length (m)	138,292	39,339	3,824	9,186	6,307	1,617	13,078	29,889	23,698	10,006	1,348
Sample Count 1,000 g/t +	32	4	2	1	2	0	0	0	19	3	1
Sample Count 100 to 999.99 g/t	157	78	8	10	2	0	4	5	35	11	4
Sample Count 20 to 99.99 g/t	529	345	29	20	24	0	18	21	54	16	2
Sample Count 0 to 19.99 g/t	93,800	26,640	2,263	6,481	4,384	1,133	8,998	20,302	15,941	6,742	916
Total Sample Count	**94,518**	**27,067**	**2,302**	**6,512**	**4,412**	**1,133**	**9,020**	**20,328**	**16,049**	**6,772**	**923**
Silver											
Minimum (g/t)	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Maximum (g/t)	41,679.0	41,679.0	15,340.5	5,330.0	1,490.0	282.2	776.0	2,700.0	9,312.0	4,400.0	720.0
Mean (g/t)	38.28	109.88	38.75	10.91	15.94	4.89	8.06	5.45	10.21	7.66	9.70
Standard Deviation	400.21	725.48	347.91	81.34	54.53	11.03	24.64	31.86	135.99	74.13	38.00
Coefficient of Variation	10.45	6.60	8.98	7.45	3.42	2.26	3.06	5.84	13.32	9.68	3.92
Total Length (m)	138,292	39,339	3,824	9,186	6,307	1,617	13,078	29,889	23,698	10,006	1,348
Sample Count 1,000 g/t +	513	471	12	2	3	0	0	4	15	6	0
Sample Count 100 to 999.99 g/t	3,607	3,060	103	70	103	2	65	68	86	37	13
Sample Count 20 to 99.99 g/t	11,722	8,372	358	395	479	37	491	592	731	221	46
Sample Count 0 to 19.99 g/t	78,676	15,164	1,829	6,045	3,827	1,094	8,464	19,664	15,217	6,508	864
Total Sample Count	**94,518**	**27,067**	**2,302**	**6,512**	**4,412**	**1,133**	**9,020**	**20,328**	**16,049**	**6,772**	**923**

Table 14.7 Brucejack Summary Composite Statistics by Domain

	Total	West Zone	Shore Zone	Gossan	Galena	SG	Bridge Zone	Low Grade	VOK 211	VOK 212	VOK 220
Gold											
Minimum (g/t)	0.0025	0.0040	0.0250	0.0020	0.0020	0.0020	0.0020	0.0020	0.0020	0.0020	0.0040
Maximum (g/t)	10,873.4	846.4	531.1	509.3	1,482.7	12.3	229.4	143.0	10,873.	2,860.0	2,749.8
Mean (g/t)	1.95	2.14	2.20	1.12	1.92	0.70	0.83	0.32	4.27	2.14	8.29
Standard Deviation	55.90	12.87	16.85	9.91	29.80	1.21	3.62	1.64	123.89	54.01	132.20
Coefficient of Variation	28.63	6.03	7.67	8.88	15.53	1.73	4.37	5.18	29.00	25.23	15.94
Total Length (m)	137,966	39,160	3,756	9,142	6,312	1,606	13,078	29,866	23,708	9,992	1,346
Sample Count 1,000 g/t +	15	0	0	0	1	0	0	0	10	2	2
Sample Count 100 to 999.99 g/t	96	34	5	8	4	0	2	1	36	6	0
Sample Count 20 to 99.99 g/t	392	240	20	22	10	0	9	11	57	15	8
Sample Count 0 to 19.99 g/t	68,480	19,306	1,853	4,541	3,141	803	6,528	14,921	11,751	4,973	663
Total Sample Count	**68,983**	**19,580**	**1,878**	**4,571**	**3,156**	**803**	**6,539**	**14,933**	**11,854**	**4,996**	**673**
Silver											
Minimum (g/t)	0.25	0.25	0.59	0.25	0.25	0.25	0.25	0.25	0.25	0.25	0.25
Maximum (g/t)	35,889.4	35,889.4	1,213.7	1,886.5	748.4	146.8	606.9	688.4	5,581.2	1,304.8	341.5
Mean (g/t)	29.03	81.74	24.87	9.66	14.42	4.51	7.55	5.11	8.25	6.53	9.31
Standard Deviation	229.87	421.89	67.33	43.22	35.38	7.84	17.12	16.65	65.64	30.93	25.73
Coefficient of Variation	7.92	5.16	2.71	4.47	2.45	1.74	2.27	3.26	7.96	4.74	2.77
Total Length (m)	137,966	39,160	3,756	9,142	6,312	1,606	13,078	29,866	23,708	9,992	1,346
Sample Count 1,000 g/t +	254	245	1	2	0	0	0	0	5	1	0
Sample Count 100 to 999.99 g/t	2,712	2,364	86	36	54	1	41	43	55	23	9
Sample Count 20 to 99.99 g/t	9,073	6,414	352	295	405	22	345	443	571	186	40
Sample Count 0 to 19.99 g/t	56,944	10,557	1,439	4,238	2,697	780	6,153	14,447	11,223	4,786	624
Total Sample Count	**68,983**	**19,580**	**1,878**	**4,571**	**3,156**	**803**	**6,539**	**14,933**	**11,854**	**4,996**	**673**

The correlation coefficients between silver and gold for domain coded composite values were also calculated, with the VOK domains displaying a high degree of correlation (Table 14.8).

Table 14.8 Correlation Coefficients by Domain

Domain	Correlation Coefficient
Gossan	0.14
Low Grade	0.18
SG	0.31
West Zone	0.37
Galena	0.41
Shore Zone	0.41
VOK 220	0.63
VOK 212	0.75
VOK 211	0.98

14.5.3 *Treatment of Extreme Values*

Grade capping analysis was conducted on the domain-coded and composited grade variable data in order to evaluate the potential influence of extreme values during estimation. The presence of high-grade outliers was identified by decile analysis and examination of histograms and log-probability plots, and where possible the observed correlations between gold and silver were also taken into account.

For the Gossan, Galena, SG, Bridge Zone and Low Grade domains, a Parish capping level for gold was selected in order to reduce the total contained metal in the 99th percentile to less than 10% of the total metal. The silver percentile corresponding to the gold capping level was then selected as the capping level for silver.

For the West Zone, Shore Zone and VOK domains, estimation was done using indicator kriging (IK), and capping levels for these domains were derived from the high-grade gold composite sample distribution. The corresponding silver percentile was then selected as the capping level for silver.

Composites were capped to the selected value prior to estimation (Table 14.9).

Table 14.9 Brucejack Composite Capping Levels

Domain	Element	Cap (g/t)	Uncapped Average (g/t)	Capped Average (g/t)	Number Capped	Capped (%)
Bridge Zone	Au	10	0.83	0.73	27	0.41
	Ag	120	7.55	7.22	26	0.40
Low Grade Halo	Au	10	0.32	0.29	32	0.21
	Ag	120	5.11	4.78	32	0.21
Galena Hill	Au	30	1.92	1.03	9	0.29
	Ag	320	14.42	13.96	10	0.32
Gossan Hill	Au	10	1.12	0.65	45	0.98
	Ag	90	9.66	8.16	44	0.96
SG Zone	Au	10	0.70	0.70	1	0.12
	Ag	140	4.51	4.50	1	0.12
Shore Zone	Au	220	2.20	1.98	4	0.21
	Ag	600	24.87	24.03	6	0.32
West Zone	Au	220	2.14	2.03	13	0.07
	Ag	6,000	81.74	79.10	14	0.07
VOK 211	Au	900	4.27	2.50	10	0.08
	Ag	500	8.25	7.30	12	0.10
VOK 212	Au	900	2.14	2.14	2	0.04
	Ag	800	6.53	6.42	2	0.04
VOK 220	Au	900	8.29	3.83	2	0.30
	Ag	250	9.31	9.13	3	0.45

14.5.4 *Continuity Analysis*

For the Bridge Zone, Low Grade Halo, Galena Hill, Gossan Hill and SG domains, isotropic experimental semi-variograms were modelled from uncapped composite samples using a normal-scores transformation (Table 14.10). The downhole variogram was viewed at a 2.00 m lag spacing (equivalent to the composite length) to assess the nugget variance contribution. 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 estimation, and continuity ranges based on the semi-variogram models were then generated for each variable by domain and used to define the appropriate search strategy.

Table 14.10 Brucejack Experimental Semi-Variograms

Domain	Element	Experimental Semi-Variogram
Bridge Zone	Ag	0.13+spherical(0.50, 50 m)+ spherical(0.22, 90 m)
	Au	0.34+spherical(0.60, 10 m)+ spherical(0.07, 70 m)
Low Grade Halo	Ag	0.16+spherical(0.65, 20 m)+ spherical(0.19, 300 m)
	Au	0.32+spherical(0.58, 10 m)+ spherical(0.09, 100 m)
Galena Hill	Ag	0.17+spherical(0.73, 20 m)+ spherical(0.10, 100 m)
	Au	0.40+spherical(0.57, 10 m)+ spherical(0.03, 70 m)
Gossan Hill	Ag	0.32+spherical(0.56, 10 m)+ spherical(0.12, 80 m)
	Au	0.33+spherical(0.59, 10 m)+ spherical (0.08, 60 m)
SG	Ag	0.19+spherical(0.81, 30 m)
	Au	0.15+spherical(0.85, 30 m)

For the West Zone, Shore Zone and VOK domains, median indicator semi-variograms modeled from uncapped composite data for each of the three principle directions were calculated and iteratively refined (Table 14.11). In general the horizontal and across-strike directions were aligned with observed mineralization trends, with the dip-plane direction normal to both.

Table 14.11 Brucejack Median Indicator Semi-Variograms

Domain	Element	Median	Experimental Semi-Variogram
West Zone	Ag	17.66	0.20+spherical(0.20, 8/8/6 m)+spherical(0.60, 120/90/40 m)
	Au	0.70	0.20+spherical(0.40, 8/10/5 m)+spherical(0.40, 40/40/30 m)
Shore Zone	Ag	9.07	0.10+ spherical (0.60, 7/9/10 m)+ spherical (0.30, 50/40/20 m)
	Au	0.49	0.10+spherical (0.90, 30/20/20 m)
VOK 211	Ag	3.58	0.20+spherical(0.50, 20/20/10 m)+spherical(0.30, 110/110/25 m)
	Au	0.31	0.20+spherical(0.40, 10/20/10 m)+spherical(0.40, 50/50/20m)
VOK 212	Ag	2.79	0.20+spherical(0.50, 30/30/20 m)+spherical(0.30, 110/70/40 m)
	Au	0.31	0.20+spherical(0.30, 10/30/10 m)+spherical(0.50, 70/50/40 m)
VOK 220	Ag	3.67	0.30+spherical(0.40, 12/12/12 m)+spherical(0.30, 65/65/65 m)
	Au	0.45	0.30+spherical(0.50, 10/10/10 m)+spherical(0.30, 30/30/30 m)

The modelled semi-variograms were then used to define appropriate ranges for mineral resource estimation and classification (Table 14.12).

Table 14.12 Estimation Search Ranges

Domain	Pass	Ranges (m)		
Bridge Zone	1	70		
	2	230		
Low Grade Halo	1	380		
Galena Hill	1	40		
	2	280		
Gossan Hill	1	40		
	2	160		
SG	1	300		
Domain	**Pass**	**Major Direction (m)**	**Sub-major (m)**	**Minor m)**
West Zone	1	20	20	10
	2	40	40	20
	3	120	120	20
Shore Zone	1	30	30	20
	2	240	240	40
VOK 211	1	40	40	20
	2	120	120	60
VOK 212	1	40	40	30
	2	120	120	90
VOK 220	1	30	30	30
	2	150	150	150

14.5.5 Block Models

The modelled Brucejack mineralization domains extend along a corridor 500 m wide and 3,500 m in length. In order to facilitate mine planning and optimization, and to limit conditional bias, a single orthogonal block model was established across the property using a 25 by 25 by 10 m block size (Table 14.13).

Table 14.13 Brucejack Block Model Setup

	Minimum (°)	Maximum (°)	Blocks (°)	Size (m)
X	425,500	427,500	80	25
Y	6,256,500	6,260,500	106	25
Z	400	1,900	150	10
Rotation	0°			

The block model consists of separate models for gold estimated grades, silver estimated grades, IK 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. The volume of the known historical workings was also calculated for the West Zone and depleted from the model prior to estimation.

14.5.6 ESTIMATION AND CLASSIFICATION

The mineral resource estimate was constrained by wireframes that form hard boundaries between the respective composite data files, and classification levels were implemented algorithmically based on the estimation pass and sample selection criteria. Individual block grades were used to calculate a gold-equivalent block grade model.

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

1. During the first pass, six to fifteen composite samples from two or more drillholes within the defined search range were required for estimation. All block grades estimated during the first pass were classified as Indicated.
2. During the second pass, blocks not populated during the first pass were estimated. Three to fifteen composite samples from one or more drillholes within the defined search range were required for estimation. All block grades estimated during the second pass were classified as Inferred. SG Zone and Low Grade Halo mineral resources were all classified as Inferred.

For the West Zone, Shore Zone and VOK mineralization domains all block grades were estimated using IK of capped composite values. A gold threshold value of 4.00 g/t was selected based on observed breaks across all domains (Figure 14.2), and used to differentiate the high-grade and low-grade populations. The corresponding silver percentile was selected as the threshold value for silver (Table 14.14). Based on the defined threshold value, for each block a high-grade probability, high-grade estimate and low-grade estimate were calculated and then combined into a single block estimate.

Figure 14.2 Log-Probability Graph of Gold Composite Data

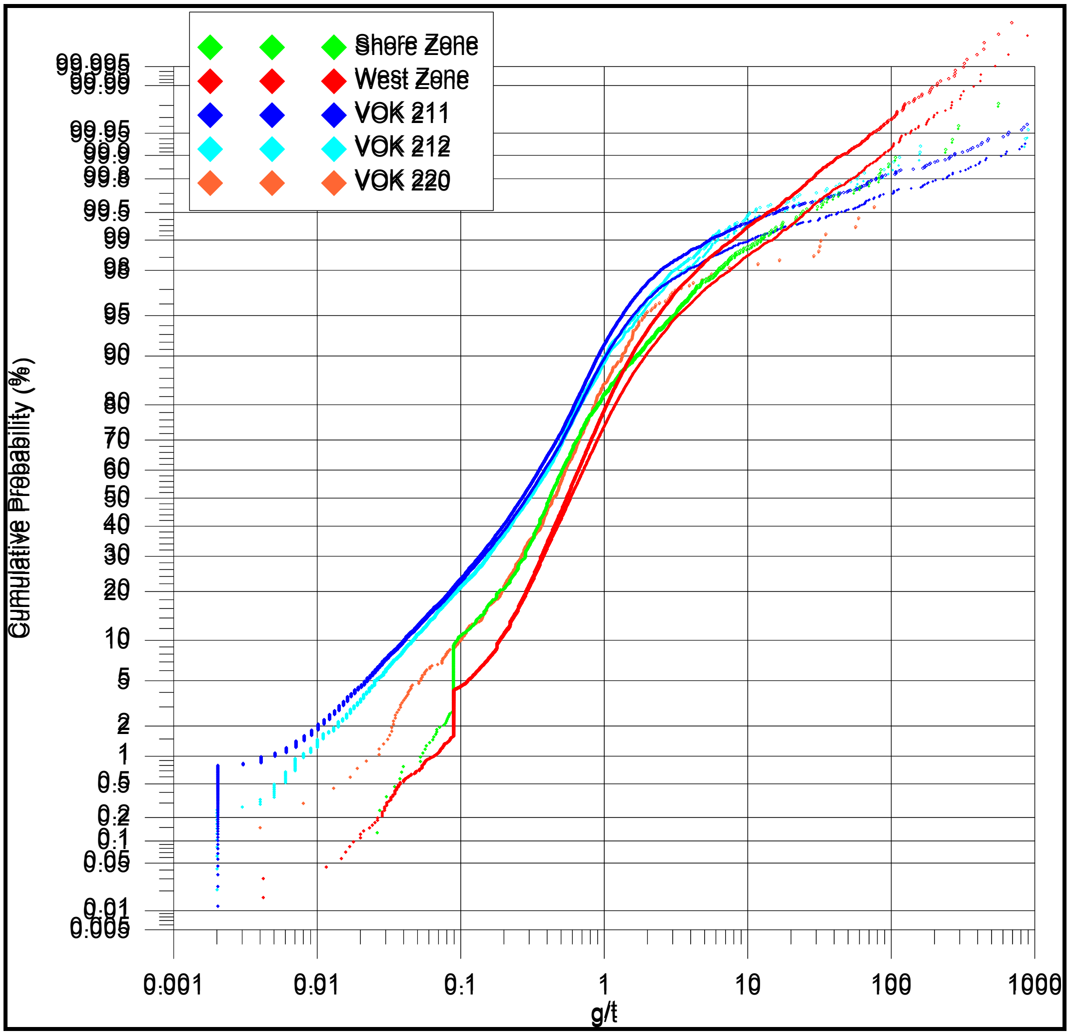


Table 14.14 IK Threshold Values

Domain	Gold Threshold (g/t)	Silver Threshold (g/t)
West Zone	4.00	200
Shore Zone	4.00	70
VOK 211	4.00	40
VOK 212	4.00	40
VOK 220	4.00	60

For the West Zone, Shore Zone and VOK Zones, a three-pass series of expanding search ellipses with varying minimum sample requirements was used for sample selection, estimation and classification:

1. During the first pass, nine to 15 composite samples from 3 or more drillholes within the defined search 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 better defined the continuity of the mineralization.
2. During the second pass, blocks not populated during the first pass were estimated. Six to 15 composite samples from 2 or more drillholes within the defined search range were required for estimation. All block grades estimated during the second pass were classified as Indicated.
3. During the third pass, blocks not populated during the first or second pass were estimated. Three to 15 composite samples from 1 or more drillholes within the defined search range were required for estimation. All block grades estimated during the third pass were classified as Inferred.

A single-pass isotropic bulk density model was implemented using inverse distance squared linear estimation, with each block estimate requiring between three and 15 samples. Glacial ice was assigned a bulk density of 0.90 t/m^3.

14.6 BRUCEJACK MINERAL RESOURCE ESTIMATE

In order to ensure that the reported mineral resources meet "reasonable prospects for economic extraction," conceptual Lerchs-Grossman (LG) optimized pit shells were developed based on all available mineral resources (Measured, Indicated and Inferred), using the economic parameters listed in Table 14.15. Commodity prices are based on the three-year trailing average as of November 2011. 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 14.15 Economic Parameters

Item	Cost/Unit
Mining Cost	US$2.00/t
Processing Cost + G&A	US$8.25/t
Pit Wall Slope Angle	45°
Gold Price	US$1,200.00/oz
Silver Price	US$22.00/oz
Gold Recovery	71%
Silver Recovery	70%
Gold Equivalent Cut-off	0.3 g/t

All mineral resources were reported against a 0.30 g/t AuEq cut-off, as constrained within the optimized pit shell (Table 14.16).

Table 14.16 Brucejack Estimated In-pit Mineral Resources Based on a Cut-Off Grade of 0.30 g/t AuEq

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	12.2	2.50	81.6	0.99	32.1
Indicated	293.0	1.26	10.5	11.91	99.3
M+I	305.3	1.31	13.4	12.89	131.5
Inferred	813.7	0.70	7.7	18.20	201.2

Notes:
1. Mineral resources are defined within an 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$1,200/oz Au (71%) and US$22.00/oz Ag (70%). The pit optimization utilized the following cost parameters: mining US$2.00/t, processing US$7.00/t and G&A US$1.25/t along with pit slopes of 45°. 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 NI 43-101 technical report were estimated using the 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 and there has been insufficient exploration to define these inferred resources as 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.

14.7 Brucejack Mineral Resource Sensitivity

In order to demonstrate the sensitivity of the mineral resource estimate to the cut-off grade, mineral resources were also tabulated against a 5.00 g/t AuEq cut-off and a 1.25 g/t AuEq cut-off accumulated within the 0.30 g/t optimized pit shells (Table 14.17 and Table 14.18).

Table 14.17 Brucejack 5.00 g/t AuEq In-pit Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	2.4	7.93	236.1	0.60	18.0
Indicated	6.9	19.99	60.9	4.46	13.6
M+I	9.3	16.92	105.6	5.06	31.6
Inferred	4.0	25.67	20.6	3.33	2.7

Notes: See notes 1 to 3 in Table 14.16 above.

4. The high-grade resource sensitivity estimate is a subset of the resource estimate and as such is included within the resource estimate and is not in addition to the resource estimate.

Table 14.18 Brucejack 1.25 g/t AuEq In-pit Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	9.3	3.08	102.20	0.92	30.6
Indicated	64.8	3.62	23.70	7.53	49.4
M+I	74.1	3.55	33.55	8.46	80.0
Inferred	78.5	2.68	16.30	6.76	41.2

Notes: See notes 1 to 3 in Table 14.16 above.

4. The high-grade resource sensitivity estimate is a subset of the resource estimate and as such is included within the resource estimate and is not in addition to the resource estimate.

In addition, an underground sensitivity analysis was done simultaneously for the high-grade VOK and West Zone domains, based on a 5 by 5 by 5 m block size. For this analysis the reduced block size model was estimated using the identical parameters and criteria as defined for the parent 25 by 25 by 10 m mineral resource estimate, except that the results of the underground analysis were not restricted to a conceptual pit shell. The inclusion of the underground sensitivity analysis is not meant to supersede or replace the bulk-tonnage mineral resource estimate (Table 14.19 through Table 14.23).

Table 14.19 Combined West Zone and VOK 5.00 g/t AuEq Underground Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	2.4	7.29	241.2	0.57	18.9
Indicated	6.1	24.13	53.3	4.76	10.5
M+I	8.6	19.35	106.7	5.33	29.4
Inferred	4.0	25.73	22.0	3.29	2.8

Notes: See notes 1 to 3 in Table 14.16 above.

4. The underground resource sensitivity is an analysis of the underground potential at Brucejack and is not meant to supersede or replace the mineral resource estimate.

Table 14.20 West Zone 5.00 g/t AuEq Underground Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	2.4	7.29	241.2	0.57	18.9
Indicated	1.5	7.04	149.4	0.34	7.1
M+I	3.9	7.70	206.5	0.91	26.0
Inferred	0.2	7.00	94.79	0.05	0.71

Notes: See notes 1 to 3 in Table 14.16 above.

4. The underground resource sensitivity is an analysis of the underground potential at Brucejack and is not meant to supersede or replace the mineral resource estimate.

Table 14.21 VOK 211 5.00 g/t AuEq Underground Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained	
				Au (Moz)	Ag (Moz)
Measured	0	0	0	0	0
Indicated	3.8	30.26	22.8	3.7	2.8
M+I	3.8	30.26	22.8	3.7	2.8
Inferred	2.1	27.53	17.1	1.9	1.2

Notes: See notes 1 to 3 in Table 14.16 above.

4. The underground resource sensitivity is an analysis of the underground potential at Brucejack and is not meant to supersede or replace the mineral resource estimate.

Table 14.22 VOK 212 5.00 g/t AuEq Underground Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained Au (Moz)	Contained Ag (Moz)
Measured	0	0	0	0	0
Indicated	0.7	29.88	22.76	0.67	0.51
M+I	0.7	29.88	22.76	0.67	0.51
Inferred	0.3	24.56	22.32	0.23	0.21

Notes: See notes 1 to 3 in Table 14.16 above.

4. The underground resource sensitivity is an analysis of the underground potential at Brucejack and is not meant to supersede or replace the mineral resource estimate.

Table 14.23 VOK 220 5.00 g/t AuEq Underground Mineral Resource Grade and Tonnage Sensitivity

Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Contained Au (Moz)	Contained Ag (Moz)
Measured	0	0	0	0	0
Indicated	0.1	8.2	14.1	0.03	0.06
M+I	0.1	8.2	14.1	0.03	0.06
Inferred	1.3	26.39	16.9	1.1	0.7

Notes: See notes 1 to 3 in Table 14.16 above.

4. The underground resource sensitivity is an analysis of the underground potential at Brucejack and is not meant to supersede or replace the mineral resource estimate.

14.8 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 the average grade of the composites to the model block grade estimates at zero cut-off. Composite grades were also compared to a nearest neighbour (NN) model, generated using the same search criteria as that used for the mineral resource estimate (Table 14.24).

Table 14.24 Validation Statistics for Block Estimates

Domain	Composite Average (g/t)	Capped Composite Average (g/t)	Block Average (g/t)	NN Average (g/t)
Gold				
Bridge Zone	0.83	0.73	0.70	0.68
Galena	1.92	1.03	1.01	0.94
Gossan	1.12	0.65	0.61	0.58
Low Grade	0.32	0.29	0.28	0.22
SG	0.70	0.70	0.70	0.67
Shore Zone	2.20	1.98	1.27	1.01
VOK 211	4.27	2.50	2.84	1.91
VOK 212	2.14	1.49	1.07	1.05
VOK 220	8.29	3.83	2.24	1.94
West Zone	2.14	2.03	1.07	0.94
Silver				
Bridge Zone	7.55	7.22	7.5	7.5
Galena	14.42	13.96	12.3	11.4
Gossan	9.66	8.16	7.9	8.3
Low Grade	5.11	4.78	4.6	3.6
SG	4.51	4.50	4.7	5.0
Shore Zone	24.87	24.03	19.4	18.3
VOK 211	8.25	7.30	6.9	5.9
VOK 212	6.53	6.42	6.8	6.5
VOK 220	9.31	9.13	8.9	8.5
West Zone	81.74	79.10	20.5	18.5

A total of 21 block estimates reported values greater than 100 g/t Au (Table 14.25). These high-grade blocks were queried individually in order to validate the IK estimate. Within the constraints of the information currently available all estimates appear reasonable.

Table 14.25 High Grade Gold Block Estimates

Domain	Estimated Au Grade (g/t)	Class	Grade Above Threshold (g/t)	Grade Below Threshold (g/t)	Probability Above Threshold	Number of Samples	Contained Au ('000 oz)
VOK 220	100.51	Inferred	603.86	0.57	0.17	15	56
VOK 220	100.51	Inferred	605.71	0.59	0.17	15	56
VOK 211	101.10	Indicated	366.36	0.31	0.28	12	57
VOK 220	101.68	Inferred	581.90	1.32	0.17	15	56

table continues…

Domain	Estimated Au Grade (g/t)	Class	Grade Above Threshold (g/t)	Grade Below Threshold (g/t)	Probability Above Threshold	Number of Samples	Contained Au ('000 oz)
VOK 211	102.14	Indicated	418.79	0.36	0.24	11	57
VOK 211	103.35	Indicated	517.32	0.59	0.20	15	57
VOK 211	104.08	Inferred	570.14	0.73	0.18	15	56
VOK 220	105.16	Inferred	626.72	1.22	0.17	15	59
VOK 220	112.14	Inferred	617.41	1.22	0.18	15	63
VOK 220	114.38	Inferred	615.41	1.21	0.18	15	64
VOK 211	120.64	Indicated	900.00	0.37	0.13	15	68
VOK 211	121.04	Inferred	511.69	0.84	0.24	15	65
VOK 211	121.32	Inferred	512.03	0.84	0.24	15	65
VOK 211	124.31	Inferred	513.14	0.91	0.24	15	66
VOK 220	126.64	Inferred	610.06	0.61	0.21	15	71
VOK 211	131.20	Indicated	383.94	0.59	0.34	12	73
VOK 211	136.84	Indicated	386.16	1.11	0.35	15	75
VOK 220	150.87	Inferred	900.00	0.69	0.17	15	84
VOK 211	168.96	Inferred	678.14	0.13	0.25	15	91
VOK 212	186.21	Indicated	709.92	0.72	0.26	15	102
VOK 211	219.75	Indicated	900.00	0.52	0.24	15	122

As a further check of the mineral resource model, the total volume reported at zero cut-off was compared by domain with the calculated volume of the defining mineralization wireframe (Table 14.26). All reported volumes fall within acceptable tolerances.

Table 14.26 Volume Comparison

Domain	Resource Volume ('000 m^3)	Wireframe Volume ('000 m^3)
Bridge Zone	136,272	136,040
Galena	13,541	13,529
Gossan	19,084	19,084
Low Grade	514,600	527,909
SG	10,209	10,210
Shore Zone	3,863	3,856
VOK 211	26,976	27,148
VOK 212	15,287	15,414
VOK 220	8,756	8,969
West Zone	42,910	43,257

15.0 MINERAL RESERVE ESTIMATES

A mineral reserve has not been estimated for the Project as part of this PEA.

A mineral reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a preliminary feasibility study.

16.0 MINING METHODS

16.1 MINING OPERATIONS

16.1.1 INTRODUCTION

For the purpose of this PEA, mining at the Project will use longhole open stoping methods with rock or paste backfill where applicable.

The nominal production rate for the Project is 1,500 t/d at steady-state operation.

Sections 16.1.3 to 16.1.16 details the underground mine plan.

16.1.2 3D RESOURCE MODEL

The mine planning work for the Project PEA is based on a resource model estimated by P&E specifically for the underground portion of the deposit. The resource model is depleted for pre-existing development and is in NAD 83 Zone 9 BC coordinate system. Details of the resource model framework are summarized in Table 16.1.

AMC conducted a high-level review of the P&E resource model for familiarization purposes and to determine suitability for the PEA assessment.

Table 16.1 Details of the Brucejack Underground Resource Model Framework

Type	Y	X	Z
Minimum Coordinates	6,257,725	426,195	810
Maximum Coordinates	6,258,960	426,790	1,595
User Block Size	5	5	5
Minimum Block Size	5	5	5
Rotation	0	0	0

AMC's comments concerning the P&E underground resource model are as follows:

- The resource model has been depleted for the current estimate of the pre-existing underground excavations (WZ lode only); however, the location of the pre-existing development 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 at depth is sparse relative to the WZ lode and the upper portions of the VOK lode. Assuming similar geological attributes, it is likely that the VOK lode at depth will reveal increased discontinuity following closer spaced drilling, as indicated for the WZ lode and the upper portions of the VOK lode.
- The Mineral Resource has been categorized into Measured, Indicated and Inferred. There is some smoothing of boundaries required so as not to create isolated clusters of a Mineral Resource category around a drillhole.
- The individual grade shells used for domaining are based on a 0.5 g/t Au cut-off. There appears to be inconsistencies in the grade shells used for domaining as some high-grade assays appear excluded and some very low-grade assays included.
- Histograms of all the gold and all the silver grades show two populations (bi-modal) – indicative of two types of mineralization. This is supported by the mineralogy of the deposit as stated in the P&E technical report. Within the individual mineralized domains there is a considerable difference between the mean grades for the two populations on comparison of the gold grade values for the WZ and VOK (combined) domains.
- There are 1,190 collars in the drillhole database of which 60 holes do not have assays. Seven holes are wells that have not been sampled and should be excluded from the resource database. The database contained 124,776 assay intervals which had no overlaps but 113 gaps greater than 20 m. A total of 5% of the core was un-assayed, excluding the well holes.

AMC's comments on the resource model specific to the underground study are as follows:

- AMC was required to sub-cell the resource model for planning purposes. The AMC resource model's overall metal content was within 2% and 4% of the P&E resource model silver and gold metal content report respectively 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's high-level review considers the P&E underground resource model to be acceptable for the purpose of the PEA.

16.1.3 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 hangingwall, footwall, mineralized material and various zone rock mass conditions that would normally be required for a detailed underground mine design.

Following discussion and review of rock mass data with BGC, AMC has adopted 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.

16.1.4 Value Model

The NSR 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), are summarized in Table 16.2.

The NSR ($/t of mineralized material) value was incorporated into the Brucejack resource model for cut-off assessment and mine design work.

Table 16.2 NSR Parameters – Underground

Items		Units	US$	Value
Metal Prices	Gold	$/oz	1,100.00	-
	Silver	$/oz	21.00	-
Process Gold Recoveries	Grade <0.20 g/t	%	-	10.0
	Grade >0.20 & <0.50 g/t	%		50.0
	Grade >0.50 & <3.35 g/t	%		13.751*LN(Au g/t)+72.357
	Grade >3.35 & <20.0 g/t	$		3.8797*LN(Au g/t)+84.349
	Grade >20.0 g/t	%	-	97.0
Process Silver Recoveries	Grade <2.0 g/t	%	-	10.0
	Grade >2.0 & <10.0 g/t	%		50.0
	Grade >10.0 & <200.0 g/t	%		0.0418*(Ag g/t)+70.392
	Grade >200.0 g/t	%	-	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

table continues…

Items		Units	US$	Value
Net Smelter Price	Gold	$/g	35.1457	-
	Silver	$/g	0.5764	-

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

It should be noted that weighted-averaging cannot be applied to NSR grade values ($/t of mineralized material) due to its non-linear relationship with the input grades (gold and silver).

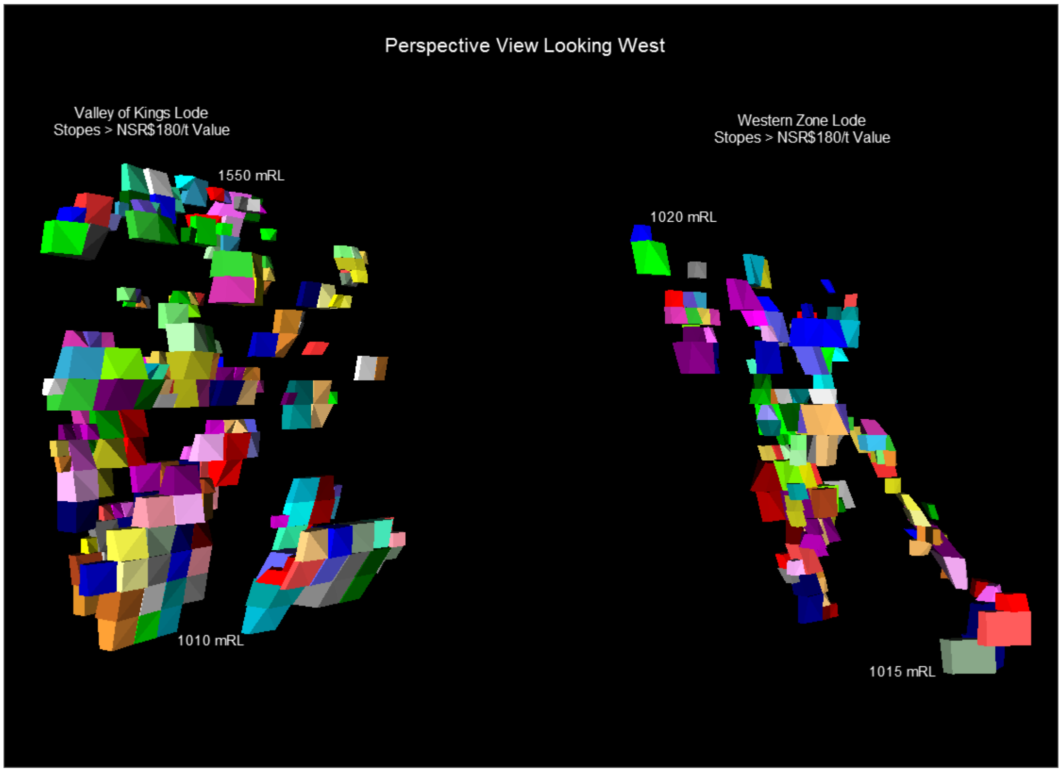
16.1.5 Cut-off and Production Rate Selection

The PEA design has used a US$180/t of mineralized material NSR cut-off, being the same as for the previous June 2011 PEA.

The NSR cut-off of US$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 US$160/t-mineralized material operating cost (mine, process and G&A estimate) and a notional profit margin.

Figure 16.1 shows a perspective view of the stoping inventory for the VOK and WZ lodes above a US$180/t of mineralized material NSR cut-off.

Figure 16.1 Perspective View of Stope Inventory above NSR Cut-off US$180/t of Mineralized Material



This PEA has used a nominal production rate of 1,500 t/d for the combined VOK and WZ lodes, being the same as for the previous June 2011 PEA.

AMC considers the production rate to be significantly conservative. 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 likely optimal production rate range at a NSR US$180/t of mineralized material cut-off is between 2,060 t/d and 2,870 t/d.

16.1.6 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.

AMC recommends a comprehensive underground geotechnical assessment to confirm the suitability of the mining method and also the stope dimensions adopted in this PEA.

Based on the current underground geotechnical knowledge, 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.

16.1.7 MINE DESIGN

The underground mine design was prepared using CAE's Mineable Shape Optimiser (MSO), Studio 3 and Mine 2-4D software, and scheduled using Earthwork's Enhanced Production Scheduler (EPS).

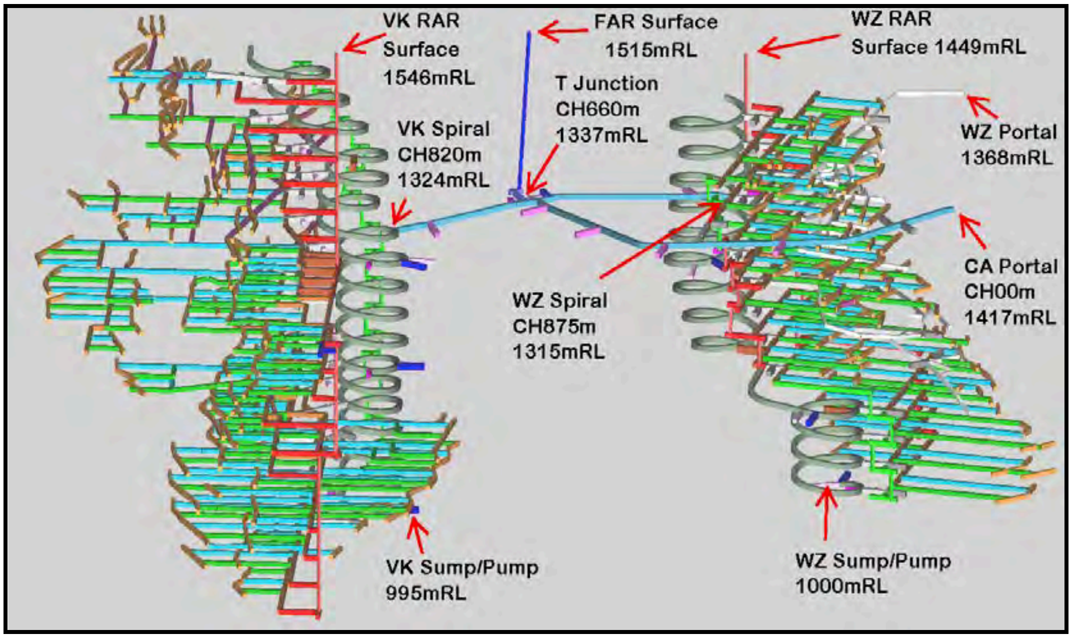
The mine design has been conducted in the NAD 83 Zone 9 (BC) coordinate system.

The mine design extracts all Mineral Resource categories (Measured, Indicated and Inferred) above the NSR cut-off. For mine design purposes, half-height stopes and half-length stopes have been allowed into the design to maximize the mine inventory. Stope shapes are restrained to 5 to 15 m below the surface topography. Extraction of these stopes may result in surface subsidence, but these stopes are clear of surface infrastructure.

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

Figure 16.2 shows an annotated perspective view highlighting the key mine design features.

Figure 16.2 Mine Design Perspective View – VOK and WZ Lodes



MINING DILUTION AND LOSSES

The mine design stope walls are generally within the limits of the defined mineralized zone. The stope wall dilutions (hangingwall 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 2.78 t/m^3 and zero grade (gold, silver). Where wall dilutions occur within the halo mineralization, they are assigned the halo grade and density from the resource model.

Stope wall and internal dilutions are based on the stoping shapes generated from the MSO mine design process which used the following mining criteria:

- minimum mining width of 5 m (essentially limited by the resource model resolution)
- minimum 10 m waste pillar width separating mineralization zones above cut-off
- hangingwall dilution skin of 1 m
- footwall dilution skin of 1 m
- minimum footwall angle of 50° (assumed rill angle for gravity flow of broken rock).

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

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

No additional losses were applied for geotechnical issues such as surface pillar requirements or under-cutting production front closures.

AMC recommend a comprehensive underground geotechnical assessment to confirm the suitability of the stope dilutions and recoveries adopted in this PEA.

Pre-existing Development

According to the June 2011 PEA, the wireframe dimensions for the pre-existing underground ramp development in the WZ provided by Pretium are not consistent with previous study reports that state the WZ ramp dimensions are 2.7 m wide by 3.9 m high at a 15% gradient (1 in 6.7).

AMC recommends a comprehensive survey be conducted to confirm the WZ ramp extents (missing raise development), 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 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.

The current estimate of the pre-existing WZ ramp alignment is that it traverses the WZ mineralization which will ultimately be mined out by stoping if the current alignment interpretation is correct.

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

Mine Access

AMC considers that the pre-existing portal location is not suitable for long-term access due to the risk of Brucejack Creek inundation (especially during thaw periods), significant surface access distance to the crusher location (approximately 0.6 km), location within a nominal 300 m blast exclusion zone for potential Galena

Hill open pit operation and the WZ stoping activity mining out the existing ramp alignment between approximately 1,345 to 1,285 masl reduced length (RL). However, this portal could provide important access as an additional heading during the initial development of the mine.

The location of the new Central Access (CA) portal is generally east of the WZ lode at approximately N6,258,640, E426,895 and RL1,417. The portal location is subject to review based on the availability of more detailed geotechnical data.

The portal will require a minor boxcut excavation, incorporate inundation protection and oriented to avoid unfavourable sunrise or sunset glare for exit safety.

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

AMC recommend considering additional mine access points to the VOK lode (e.g. extension of the VOK spiral ramp to surface breakthrough) to reduce production lead-time and ventilation resistance.

16.1.8 *Mine Development*

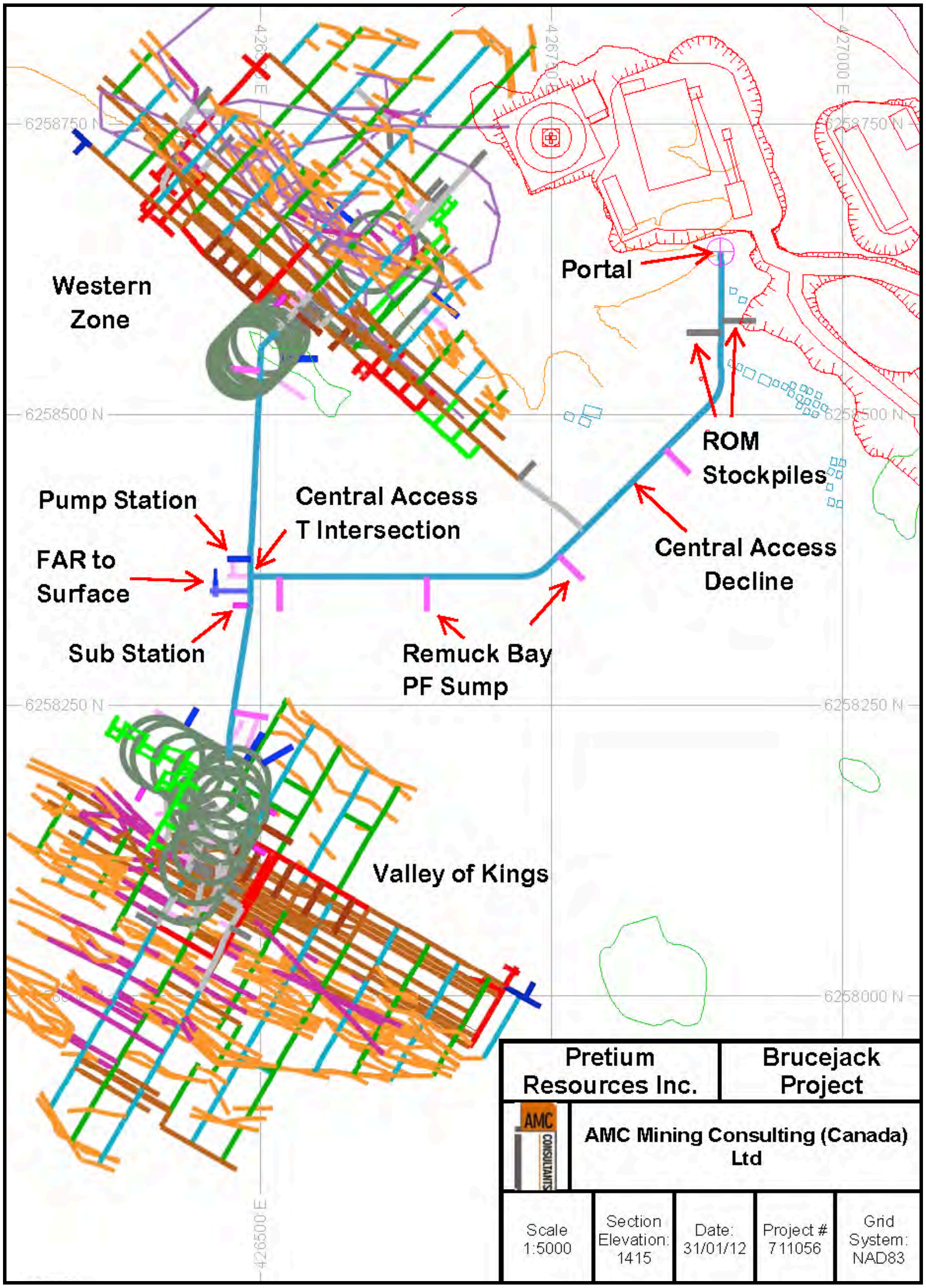
The pre-production and LOM development physicals for the combined WZ and VOK lodes are summarized in Table 16.3. The overall underground development plan is illustrated in Figure 16.4.

The LOM lateral development factor (capital and operating) is 196 t of mineralized material per metre of development advance.

Table 16.3 LOM Development Physicals Estimate

	Unit	LOM Quantity	Pre-Production Quantity
Capital Lateral			
WZ Ramp Stripping	m^3	69,959	41,395
Central Access Decline	m	970	970
Lode Spiral Ramps	m	6,543	1,790
Remuck Bays	m	880	350
Pastefill Sumps / Remuck Bays	m	180	180
Level Access	m	1900	767
Return Airway	m	2,175	449
Fresh Airway	m	50	50
Egress Access	m	1,045	121
Workshop	m	314	314
Service Bay	m	150	0
Explosive Magazine	m	84	50
Electrical Substation	m	135	45
Dewater Pump Station	m	190	50
Dewater Sumps	m	535	130
Capital Vertical			
Fresh Air Raise Alimak	m	177	177
Return Air Raise Alimak	m	687	383
Return Air Raise Longhole	m	452	61
Egress Raise Longhole	m	712	59
Operating Lateral			
Hangingwall Drive	m	8,763	1,961
Crosscut Longitudinal Access	m	2,107	332
Crosscut Primary Waste	m	6,715	852
Crosscut Primary Mineralized Material	m	2,480	152
Crosscut Secondary Waste	m	6,597	648
Crosscut Secondary Mineralized Material	m	2,664	108
Mineralized Material Drive	m	12,174	733

Figure 16.3 Underground Development Plan



16.1.9 Mine Production

AMC has assumed 89 to 102 mm stope production drilling using a dedicated production-drilling rig. This could be supplemented using any excess capacity from the 76 to 89 mm longhole or 56 to 64 mm cable bolting rigs (if required).

A stope drill factor of 8 t/m drilled has been assumed, which includes slot drilling and re-drills.

The global extraction sequence is bottom-up with three production fronts for each lode. Production assumes that up to four stopes can be active at any one time. Sequencing assumes ventilation and egress requirements are established prior to stopes producing. The macro stope mining sequence is; establish top and bottom drill-muck access, stope drilling, blast delays (one day), stope muck, survey and backfill preparation delays (five days), backfill and sequence delay for backfill exposure where applicable (21 days).

AMC has assumed single heading development of 160 m/mo and multi-heading development of 200 m/mo.

The typical design stope dimensions are:

- 30 m stope height from floor to floor
- 30 m strike length
- an average 17 m stope width with a range of 7 to 41 m for the WZ lode
- an average 27 m stope width with a range of 7 to 80 m for the VOK lode.

The LOM production is summarized in Table 16.4 for the combined and separate lodes. Mine scheduling assumes an April 2014 start date.

The LOM production summary contains Inferred categorized Mineral Resource and therefore cannot be considered, nor referred to, as a Mineral Reserve.

The VOK lode represents approximately 75% of the LOM mineralized material production.

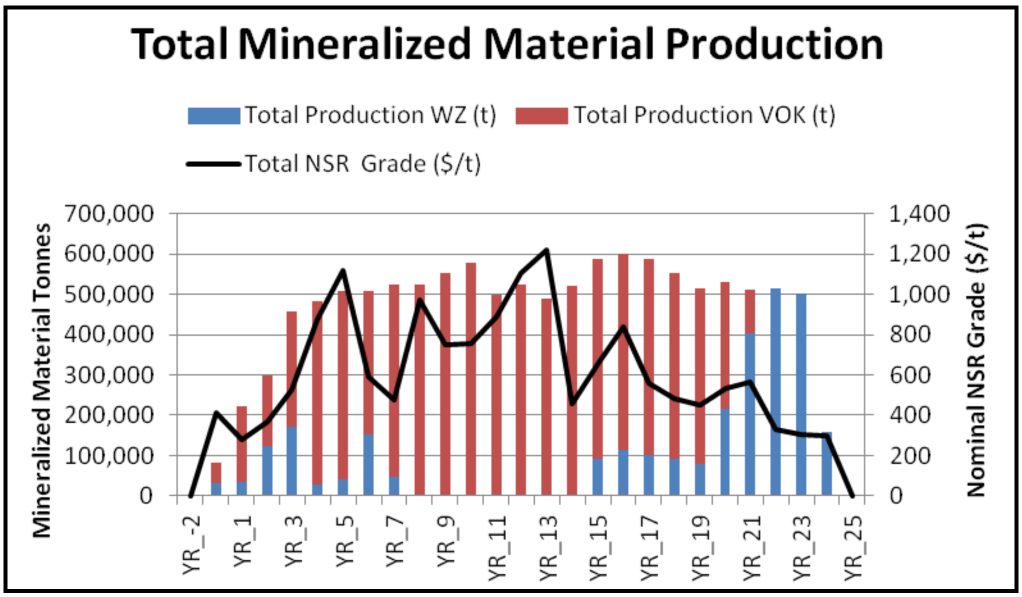
AMC recommends comprehensive scheduling be conducted to optimize the mine production schedule in the next phase of study. There may be opportunities to reduce the lead-time to production (via additional development accesses) and to minimise the production volatility (tonnes and grade) with iterative design and schedule refinements.

Figure 16.4 charts the mine production schedule.

Table 16.4 LOM Production Summary

	Unit	Total LOM Quantity	WZ Lode LOM Quantity	VOK Lode LOM Quantity
Stope Mineralized Material	'000 t	11,214	2,714	8,500
Stope Au Grade	g/t Au	19.0	6.9	22.9
Stope Ag Grade	g/t Ag	59.3	190.4	22.9
Development Mineralized Material	'000 t	613	189	424
Development Au Grade	g/t Au	17.3	6.5	22.1
Development Ag Grade	g/t Ag	67.4	180.5	17.1
Total Mineralized Material	**'000 t**	**11,828**	**2,903**	**8,924**
Total Au Grade	g/t Au	18.9	6.9	22.8
Total Ag Grade	g/t Ag	59.8	189.8	16.9

Figure 16.4 LOM Production Schedule



16.1.10 Rock Handling

The mine design and production schedule is based on contemporary rubber-tired, diesel-powered mobile equipment with mucking loaders notionally 12 to 17-tonne capacity, and haulage trucks notionally 40 to 55-tonne capacity.

The central access decline and lode 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 alternatively stockpiled underground close to the CA portal entrance when the crusher is not operational. The underground ROM stockpiles (2 x 30 m length) have a notional capacity of 1,600 t, equivalent to approximately one day of mill feed. Other remuck bays (along the access decline) may also be opportunistically utilized for ROM stockpiling if required. The development size of the portal area down to the underground ROM stockpiles will enable a surface type front-end loader (FEL) to rehandle mineralized material from the underground ROM stockpiles to the primary crusher.

There will be a surface ROM pad located adjacent to the primary crusher to handle oversize stope material and mineralized material in excess of the milling capacity. This will nominally be 200,000t capacity.

A 600,000 m^3 capacity surface waste dump pad is required for waste stockpiling until stope voids become available for underground disposal. All development waste generated will be consumed as backfill for secondary stopes and/or underground roadway construction.

16.1.11 BACKFILL

Stope backfill will use un-cemented rock fill (RF) or cemented paste fill (PF).

PF (standard strength) is used for the continuous longitudinal retreats, and primary transverse stopes with PF (high strength) used for the undercut closure stopes. RF is used at the retreat extremities of the continuous longitudinal stopes and secondary transverse stopes. As there is insufficient development waste to meet RF demand, PF (low strength) will be also used in the secondary transverse stopes to make up the deficit.

PF is produced from the thickened flotation plant tailings (total solids discharge). Three PF categories will be produced with baseline strength requirements. Rock fill is sourced from the underground waste development muck which will require rehandling. The binder material used in the PF is nominally Ordinary Portland Cement.

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

PF design strengths are in the order of 0.2 to 1.0 MPa based on the PF duty and stope dimensions of 30 m high by 30 m long by 16 m transverse width (with up to two concurrent PF wall exposures). AMC has assumed cement addition rates of 2%, 5% and 10% by weight for low, standard and high strength PF respectively.

AMC recommends comprehensive PF test work be conducted to confirm the suitability of the flotation plant tailings to be used as PF and to determine the PF strength, cement addition and cure-time relationships.

The average PF demand is 161,000 m^3/a with a maximum demand of 211,000 m^3/a. Cement demand is on average, 9,400 t/a with a maximum demand of 16,400 t/a.

AMC has assumed a modular PF plant with a 62 m^3/h instantaneous rate.

The PF plant is located adjacent to the CA portal and process plant thickeners. The paste is distributed to the underground workings via two pipes in the CA decline and then via boreholes interconnecting the levels. The VOK lode requires a positive displacement pump to distribute to the stope voids approximately above the CA portal elevation (1,415 masl). The WZ lode can be gravity fed from the CA portal elevation.

16.1.12 *Mine Services*

Ventilation

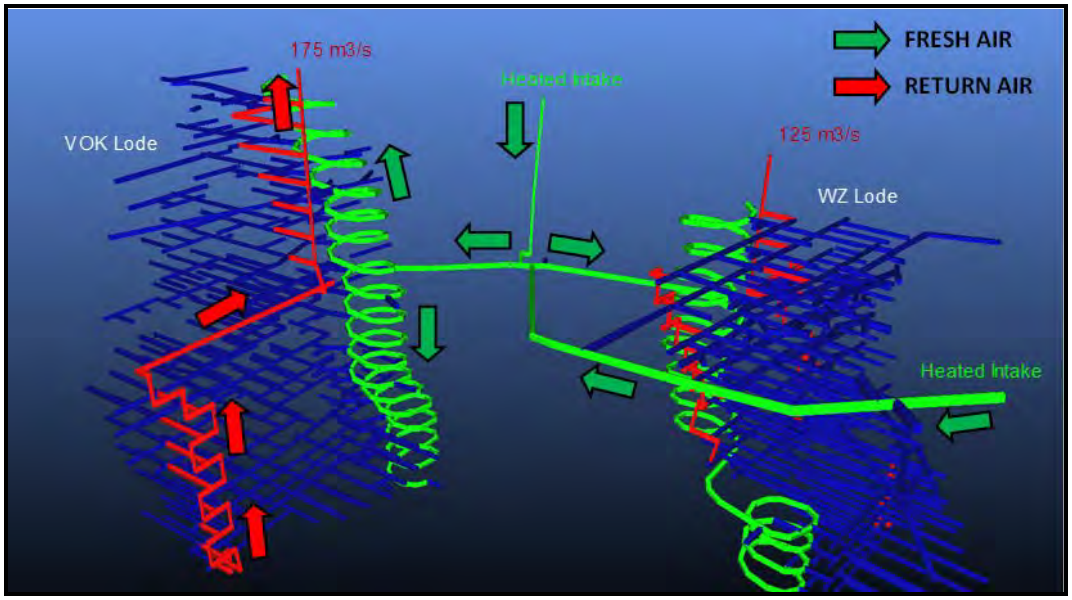
Ventilation estimates have been undertaken using Ventsim software.

There are two ventilation intakes, the Central Access (CA) portal and CA 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. The CA FAR will function as a return air raise (RAR) during the CA decline development to the two lodes.

There are two ventilation returns, one for each lode, that are located adjacent to each level's access. The RAR fans will be located at surface with surface holing points at ridge lines and fitted with evases.

The overall primary ventilation circuit plan is illustrated in Figure 16.5.

Figure 16.5 Perspective View of Primary Ventilation Circuit



AMC has made allowance for 125 m^3/s in airflow rate for the WZ lode (representing approximately 25% of the production rate) and 175 m^3/s in airflow rate for the VOK lode based on ventilation system simulation.

Mine heating is assumed to be a push-pull system for practical control. Mine heaters will be located at the CA portal and CA FAR. For mine heating, AMC has made allowance for direct-fired propane units with a combined 44 MBTU/h capacity.

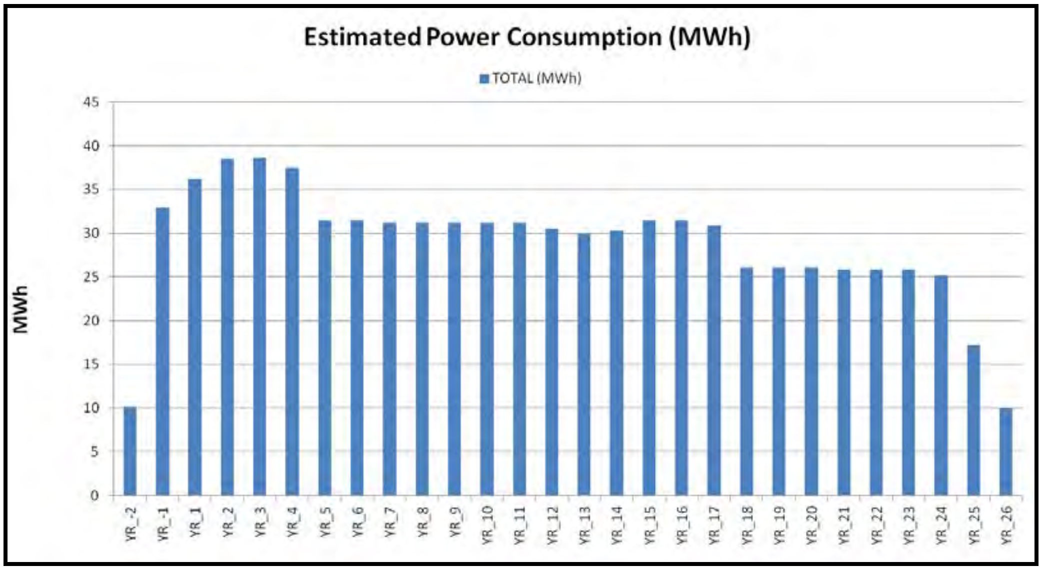
Underground Electrical Power Distribution System

There are nine electrical substations to service the mine:

- four at the WZ lode
- four at the VOK lode
- one in the CA decline adjacent to the "T" junction.

AMC's estimate of power consumption is based on the mine schedule and electrical infrastructure and is summarized in Figure 16.6. The pre-production consumption is estimated to peak at 38.5 MWh/a with a steady-state operating demand of 31.2 MWh/a. The peak power demand is 4.5 MW.

Figure 16.6 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 in the control office, safety office and technical service offices.

The leaky feeder system will be utilized to control the functions of the main fans, secondary fans (during blasting periods) and pump systems. If required, it will have the capability to be utilized for tele-remote operation of the loaders.

Explosives and Storage Handling

Underground explosive magazines are located at the VOK and WZ lode adjacent to its respective 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.

Ammonium nitrate and fuel oil (ANFO) is assumed to be 80% of the bulk explosive used for mine development and production. Packaged emulsion will be used as a primer and wet holes for production and for loading lifter holes in the development headings. Smooth blasting techniques are assumed to be employed in development headings.

Fuel Storage and Distribution

No underground fuel storage or dispensing has been allowed for in the PEA design. The PEA assumes that all mobile equipment will refuel at the surface facilities and a dedicated underground fuel-lube truck is used for fuelling of the flow tramming drill equipment. AMC recommends this system of operation be reviewed at later stage studies.

Compressed Air

It is proposed that all drills and charge-up units will be specified with on-board compressors to eliminate installation of a permanent compressed air reticulation system. 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.

Water Supply

As a preliminary water demand estimate for the mine operations, an industry empirical rule factor of 400 L of water per tonne of mineralized material production has been assumed. This equates to an average demand of 7 L/s 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 firefighting and potable water.

Mine Dewatering

Previous study work has indicated normal ground inflows of 20 to 30 L/s with peaks of 45 L/s (Newhawk Gold Mines Feasibility Study, 1990). An allowance for withdrawing 50 L/s of water from the underground mine has been included from the two lodes.

As a general overview of the main dewatering system, each lode's dewatering (WZ and VOK) is staged back to a centrally located pump station at the CA decline "T" junction (the common access point to both lodes). Mine water is discharged to surface from the "T" junction pump station. AMC has assumed a "dirty-water" type system (e.g. discharge of solids with water) with settling of solids conducted at surface facilities.

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. Rising main pipes are envisaged to be installed in cased boreholes.

Equipment Maintenance Workshop and Service Bay

The PEA has assumed one workshop for the underground operation located at the VOK lode 1,310 masl. Allowance has been made for an overhead hoist and service pit along with underground cleaning, servicing and repair facilities.

Service bays for slower tramming equipment such as drills are located at each lode's spiral ramp system to assist minimizing non-productive tramming time.

Daily maintenance lube facilities will be provided in the workshop and service bays.

Mine Safety

Fire Prevention

Every light duty vehicle will have a fire extinguisher and 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.

Allowance has been made for a leaky feeder system that can provide communications in an emergency.

Mine Rescue

A mine rescue room will be provided in the administration building. A trailer with mine rescue equipment will be located on site.

Refuge Station

Mobile 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 distance for the duration of a personal self-rescuer device, which will be provided as each person's personal protective equipment.

A static refuge station may be established adjacent to the CA "T" junction FAR servicing both WZ and VOK lodes.

Emergency Egress

Egress to surface will be via the FAR developed by Alimak, the egress raises developed by longhole method, the main central access decline and the pre-existing WZ ramp. These are all fresh air intakes.

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

Dust Control

Roadway dust suppression will use a water cart.

16.1.13 MINE EQUIPMENT

The typical mine fleet required to extract the mineralized material at 1,500 t/d is summarized in Table 16.5. AMC has assumed contemporary equipment for the project to achieve appropriate productivity, mechanical availability and operating costs.

Fleet requirements have been estimated based on average industry productivities and are inclusive of material rehandle requirements where applicable.

Table 16.5 Underground Mobile Equipment List

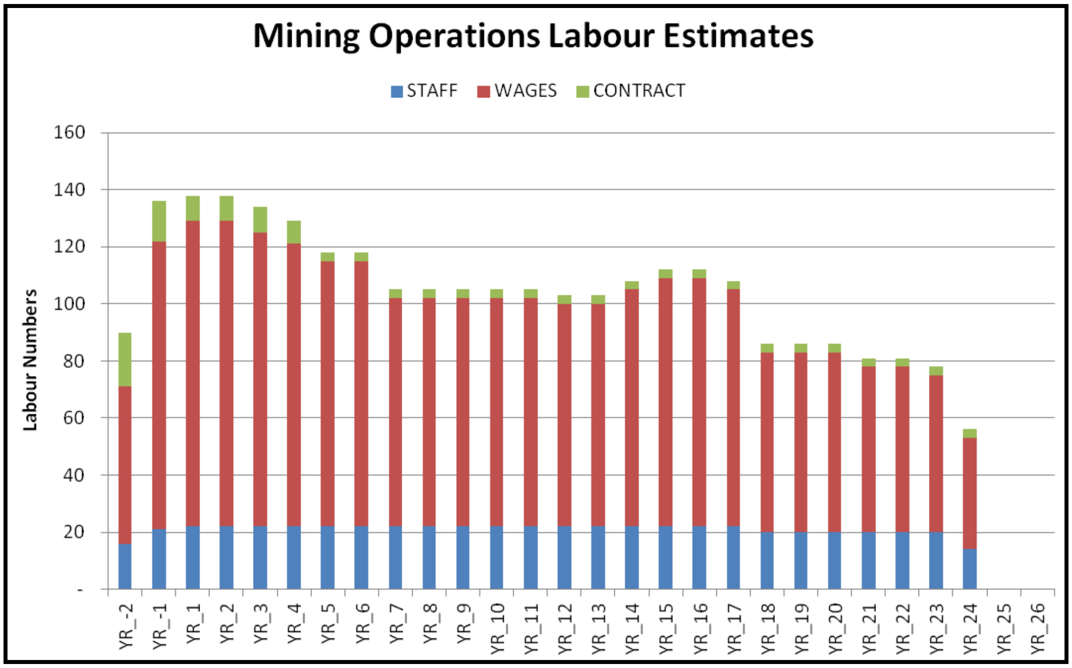
Description	LOM Quantity (each)	Pre-production Quantity (each)	Maximum Fleet Size (each)
Development Jumbo	8	5	6
Truck	18	2	3
Loader – Large	13	2	2
Loader – Small	10	2	2
Production Drill – Large	15	1	2
Longhole Drill – Small	3	1	1
Cable Drill	7	1	1
Charge-up Rig	12	3	3
Shotcreter and Agitator	6	1	1
Grader	5	1	1
Integrated Tool Carrier	16	3	3
Water Cart	3	1	1
Heavy Duty Vehicle	14	2	2
Light Vehicle	138	25	25

16.1.14 PERSONNEL

The estimate of labour numbers for the mine camp sizing is summarized in Figure 16.7. The labour estimates assume a seven-day week, two shifts per day operation on seven days on, seven days off roster.

The estimate reflects instantaneous site presence and does not account for the off-site labour, sick leave, absenteeism, annual leave, turnover, etc.

Figure 16.7 Labour Estimate for Underground Operations



16.1.15 UNDERGROUND MINING CAPITAL COST

The basis for the cost estimate is owner operation in Q1 2012 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 capital) is estimated to be Cdn$137.8 million as summarized in Table 16.6, and includes capitalized operating activities that occur in the pre-production period (Year -2 and Year -1).

The project LOM capital cost equates to Cdn$29.53/t of mineralized material mined.

Table 16.6 Underground Mining Capital Cost

Description	Total Pre-production (Cdn$ million)	Total Sustaining (Cdn$ million)
Development	44.7	68.4
Mobile Equipment	31.8	120.2
Fixed Plant	8.6	9.7
Emergency, Egress, and Safety	1.5	4.2
Ventilation	5.7	5.1
Backfill	14.4	3.9
Capitalized Operating	31.0	0.0
Total	**137.8**	**211.4**

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

16.1.16 Underground Mining Operating Costs

The basis for the cost estimate is owner operation in Q1 2012 cost terms.

The underground mine operating costs are summarized in Table 16.7. The underground mine operating costs exclude processing and G&A. The operating development includes the hangingwall drive, cross-cuts and mineralized material drives. The underground production includes direct stoping, materials handling, backfill, mine services, technical engineering and mine supervision.

Table 16.7 Underground Mining Operating Cost

Cost Distribution	Units	Value
Development	Cdn$/t of mineralized material	15.84
Production	Cdn$/t of mineralized material	88.51
Total	**Cdn$/t of mineralized material**	**104.35**

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.

17.0 RECOVERY METHODS

17.1 MINERAL PROCESSING

17.1.1 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 mill feed will supplied from the underground mine using the conventional mining methods.

17.1.2 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-tonne 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.

17.1.3 FLOWSHEET DEVELOPMENT

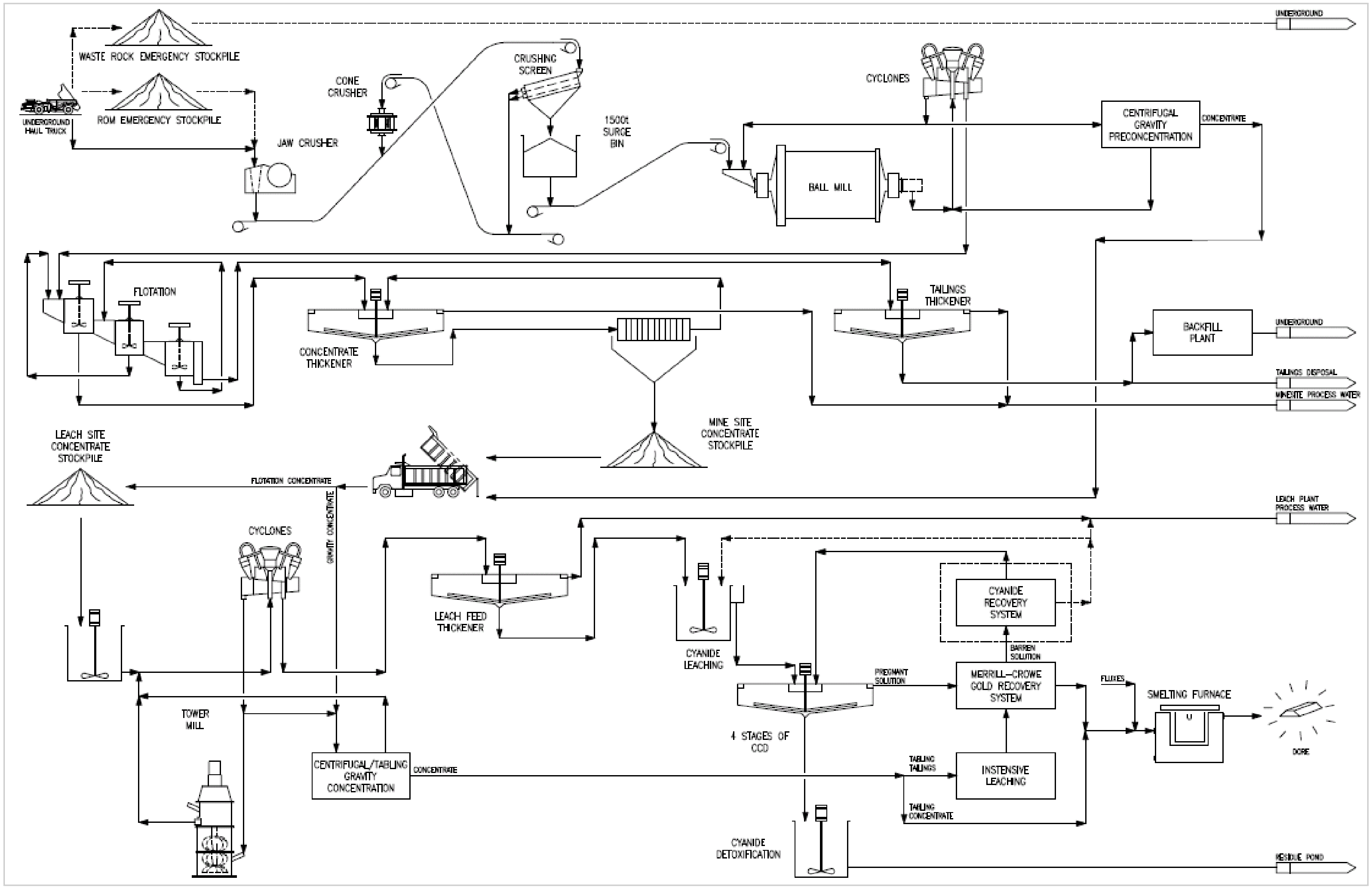
The mill flowsheet is based on PRA's 2009 and 2011 grinding, flotation and leaching test work, combined with engineering experience.

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 17.1.

Figure 17.1 Simplified Process Flowsheet



17.1.4 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 17.1.

Table 17.1 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	18

OPERATING SCHEDULE AND AVAILABILITY

The process plants are designed to operate on the basis of two 12-hour 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.

17.1.5 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 by 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 by 5.4 m long (12 ft by 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 14 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 20 to 25% 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 (seven 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 9 m diameter high-rate thickener
- slurry pumps
- stock tank
- one 90 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:

- a 9 m diameter leach feed thickener
- an aeration tank
- seven leach tanks (6.5 m diameter by 7 m high)
- four 9 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 50% 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 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 by 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 test work 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-hour 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. A sulphur dioxide monitor/alarm system will also be used to monitor the cyanide destruction area.

18.0 PROJECT INFRASTRUCTURE

18.1 INFRASTRUCTURE

18.1.1 MINE AND SITE LAYOUT

The general layout of the Project is shown in Figure 18.1.

The general arrangements of the mine site and the leach plant site for the Project are presented in Figure 18.2 and Figure 18.3.

The leach plant site will be accessible by a permanent road originating 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 to the mine site will be upgraded and used to mobilize equipment and supplies. Pretium has started construction on reopening the Newhawk access road. The road is expected to be completed by late 2012. As of the end of 2011, 12 km of new road was completed.

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

Figure 18.1 Project General Layout

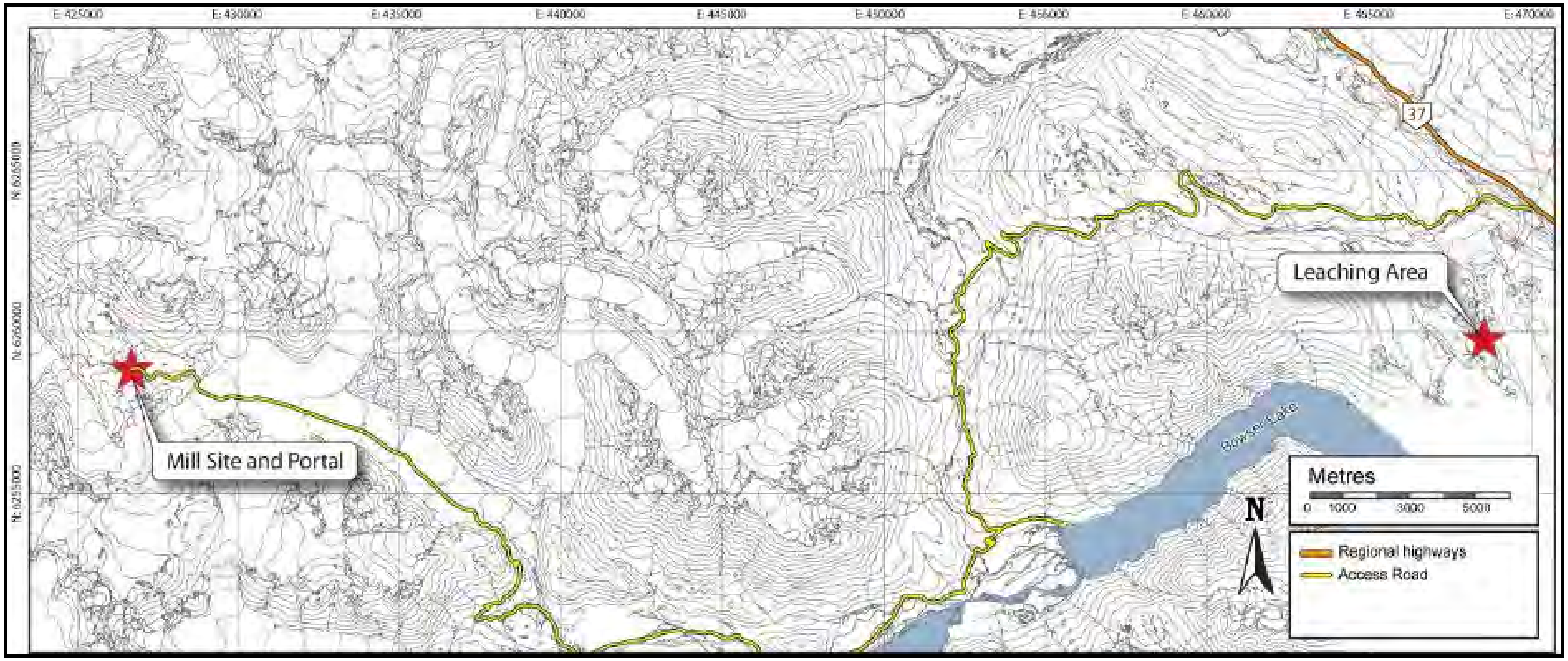


Figure 18.2 Brucejack Overall Site Plan – Mine and Process Plant

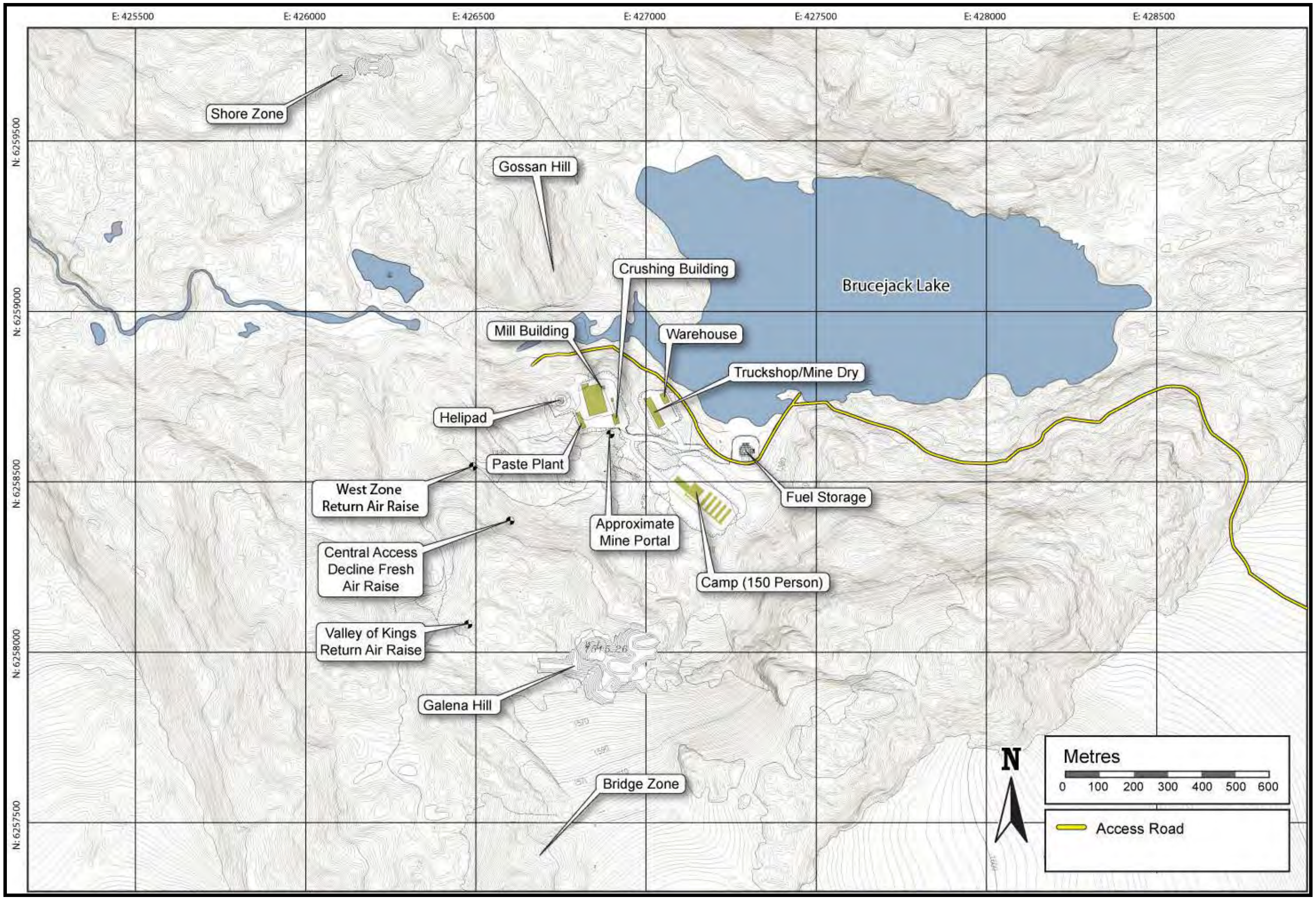
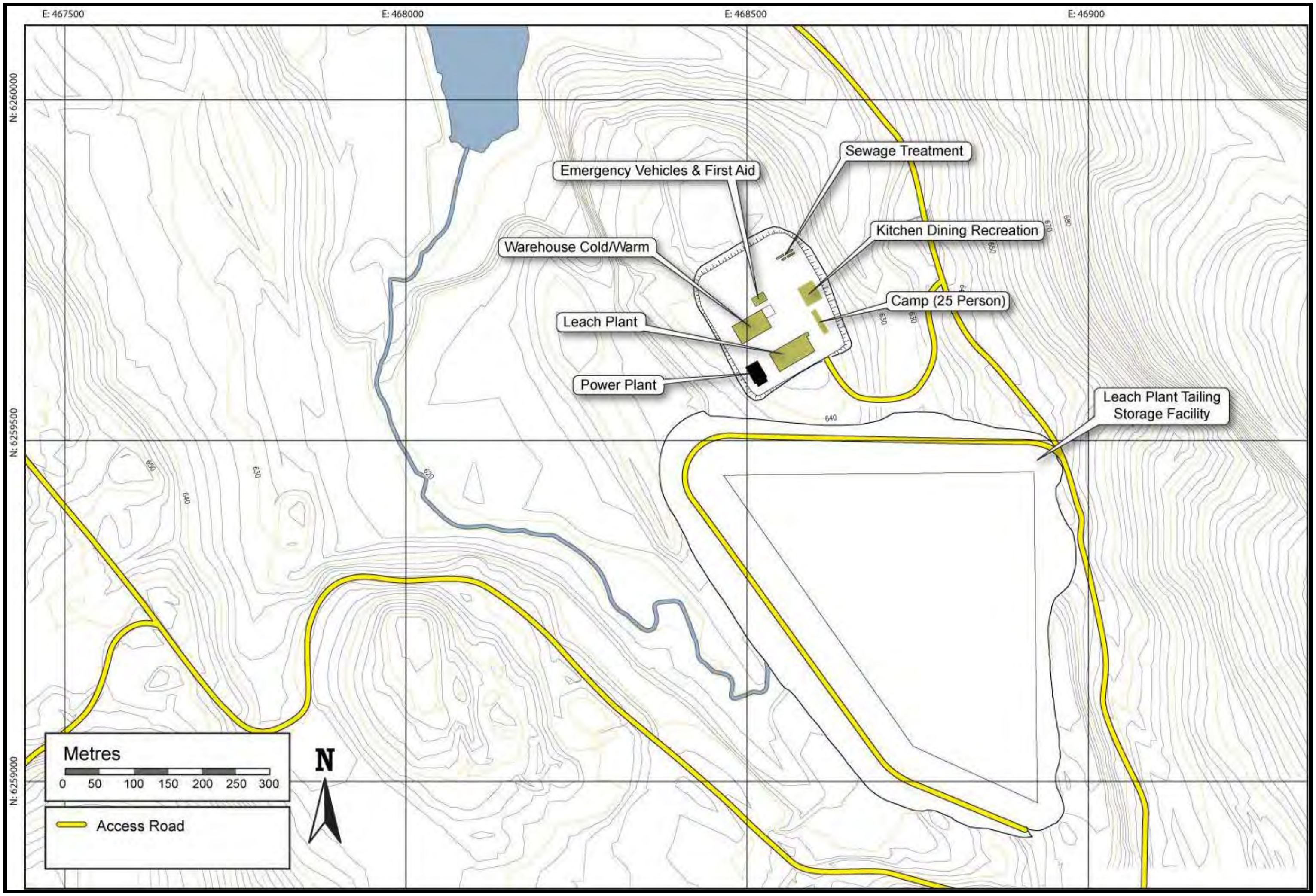


Figure 18.3 Brucejack Overall Site Plan – Leach Plant



18.1.2 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 and fuel storage facility and fuel station will be located at each site.

18.1.3 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.

18.1.4 *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.

18.1.5 *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.

18.1.6 *Roads and Access*

The leach plant site will be accessible via an upgraded road from Highway 37. Pretium has started construction on reopening the Newhawk access road to the mine site. Originally the Newhawk access was by barge over Bowser Lake, then by truck to camp. Pretium is rehabilitating the old road along the Bowser River and up the Knipple Glacier, and is in the process of building a new stretch of road, working up Scott Creek from the Bowser River and up Wildfire Creek from Highway 37. The road is expected to be completed by late 2012. As of the end of 2011, 12 km of new road was completed. Both roads will be accessible year round.

18.1.7 *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
- miscellaneous site roads
- the TSF construction and operation access road
- grading of the pads for on-site buildings and facilities.

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 to the mine site. This road will be 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.

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.

An 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.

18.1.8 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.

18.1.9 TAILINGS STORAGE FACILITY

Two tailings streams will be generated from the separate process plants: flotation tailings and leach tailings. At the mine, approximately half of the flotation tailings will be used as backfill in the underground mine, with the remainder deposited sub-aqueously in Brucejack Lake. The leach tailings, generated from the leach plant, will be deposited in a double lined side-hill TSF. The leach TSF is located adjacent to Bell Irving River and Highway 37, less than 1 km southeast from the leach plant.

18.1.10 Power/Electrical

Operations Load

The mill throughput will be 1,500t/d. At this production level, the operations load is estimated to be approximately 8 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

The power to the mine and process site will be provided by power transmission line with a 138 kV tie at the planned BC Hydro Long Lake Hydro Project substation along the Brucejack access road. The power line would cross westwards to meet the existing access roads for the Premier Mine, and follow the road northwards to the Granduc Mine. The transmission line would follow a north- trending valley to join the Brucejack Access Road, and then to the Mine Site. Note that under this option, the Leach Plant would be powered by diesel generation rather than a transmission line. Alternatively, the 138 kV substation for the Brucejack mine could be located at the mine site.

Two diesel generators, each rated 1.5 MW, will provide power for the leach plant near the highway. 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 leach site facilities.

18.2 Waste and Water Management

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

18.2.1 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, Pretium will develop a conceptual water management plan that applies to all mining activities undertaken during all phases of the 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
- constructing diversion channels to direct undisturbed runoff away from mining activities.

WATER MANAGEMENT OVERVIEW

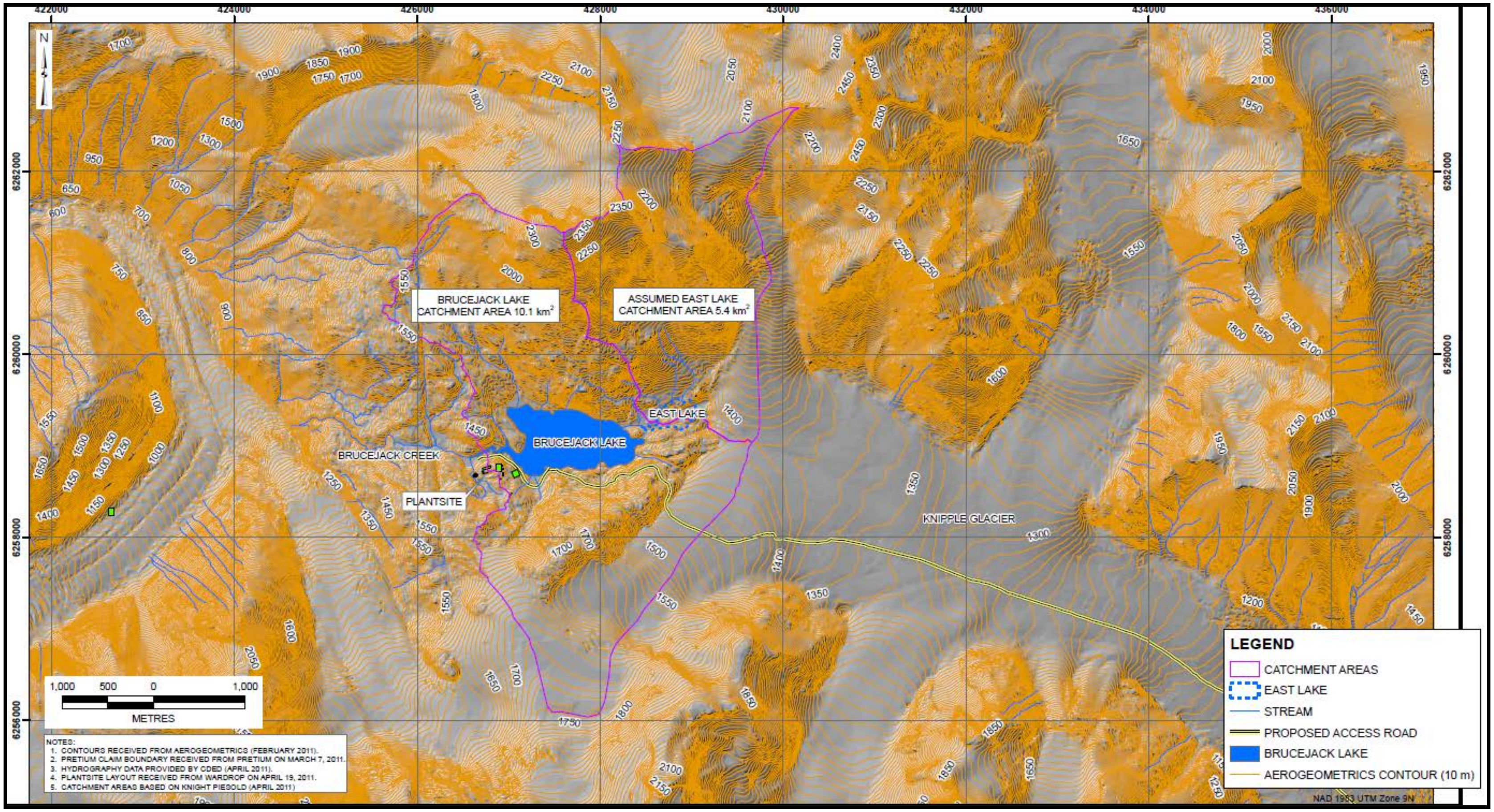
The Brucejack deposit, located west of Brucejack Lake on the east side of the Sulphurets Glacier valley, will be mined as an underground operation with a 24-year mine life (Figure 18.4). As mining progresses, waste rock and tailings (flotation and leach tailings) will be generated. The majority of the waste rock and approximately half of the flotation tailings will be used as backfill in the underground mine. The remainder of the flotation tailings and possibly some waste rock will be deposited in Brucejack Lake. Of the total processed mineralized material, 20% will be trucked to the leach plant as concentrate for secondary processing.

CONTACT WATER

Average annual seepage to the 63 km of underground mine tunnels (at full development) is expected to be approximately 2,400 m^3/h. This estimate comes from the overall average inflow estimate per metre of tunnel (0.038 m^3/h) times the total length of tunnels (63 km). 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 excess underground seepage water or runoff to Brucejack Lake that comes into contact with the submerged tailings and waste rock. This assumption needs to be rigorously tested during the next phase of engineering design.

Figure 18.4 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 two sources:

1. underground mine seepage water
2. 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 SG 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. Only a small portion of this water is required for process (as a potential fresh water requirement) because the volumes from the underground mine (2,400 m^3/h) will be in excess of the process requirements. At the end of operations, the estimated seepage rate into the underground workings is estimated at 2,400 m^3/h, which is well in excess of the approximately 40 m^3/h required for process.

18.2.2 Brucejack Lake Waste Management

Summary

Two types of waste will be generated at the Brucejack Lake mine site: waste rock 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 eastern end of Brucejack Lake. If any waste rock is deposited in Brucejack Lake, it will be deposited in the southwest corner of the lake. However, the current plan is to use the waste rock and some flotation tailings as backfill for the underground mine.

Criteria

Table 18.1 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 18.1 Brucejack Lake Waste Deposition Criteria

Criteria	Description
Mineralized Material Tonnage	11.8 Mt
Mill Throughput at Mine Plant	1,500 t/d
Mill Throughput at Leach Plant	300 t/d
Mine Life	24 years
Waste Rock Tonnage	Total – 2.4 Mt Deposited underground – 2.4 Mt Deposited in Brucejack Lake – 0 Mt
Flotation Tailings Tonnage (generated from mine plant)	Total – 9.4 Mt (20% of mineralized material goes to concentrate) Deposited in underground mine – 50% Deposited in Brucejack Lake – 50%
Leach Tailings Tonnage (generated from leach plant)	Total – 2.4 Mt
Water Cover in Brucejack Lake	Maximize water cover over flotation 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 1,366 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 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 flotation tailings. The total anticipated volume of flotation tailings to be deposited in Brucejack Lake is expected to be 3.8 Mm^3. Although all waste rock is assumed to be deposited underground, up to 1.4 Mt or 0.74 Mm^3 may be deposited in Brucejack Lake, assuming a dry density of 1.9 t/m^3 for the waste rock.

It was assumed that water decanted from the lake would 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 half of the flotation tailings will be deposited underground as paste backfill and the remaining tailings will be deposited at the east end of Brucejack Lake where it is deepest. Depositing these 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 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 approximately 70 m below the lake surface (Figure 18.5). 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.

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 mine site will be deposited in the underground mine and possibly up to 0.74 Mm^3 in Brucejack Lake. Any waste rock deposited in the lake will be put into the southwest corner (Figure 18.5). Causeways will be constructed with waste rock so that trucks can dump waste at greater depths to maintain a water cover 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.

18.2.3 LEACH PLANT AREA WATER MANAGEMENT

GENERAL

The proposed leach plant TSF will be located southeast of Unnamed Lake and south of Wildfire Creek (Figure 18.6).

The footprint of the leach 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 1,376 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 18.6).

Disturbed areas, such as overburden storage sites, will be vegetated or otherwise protected from erosion. Runoff from these areas will be directed to settling ponds with sufficient capacity to provide the retention time required to achieve discharge standards. The Metal Mining Effluent Regulations (MMER) restrict the amount of total suspended solids (TSS) 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 Unnamed Lake will be used in process.

Figure 18.5 Brucejack Lake Waste Disposal Plan

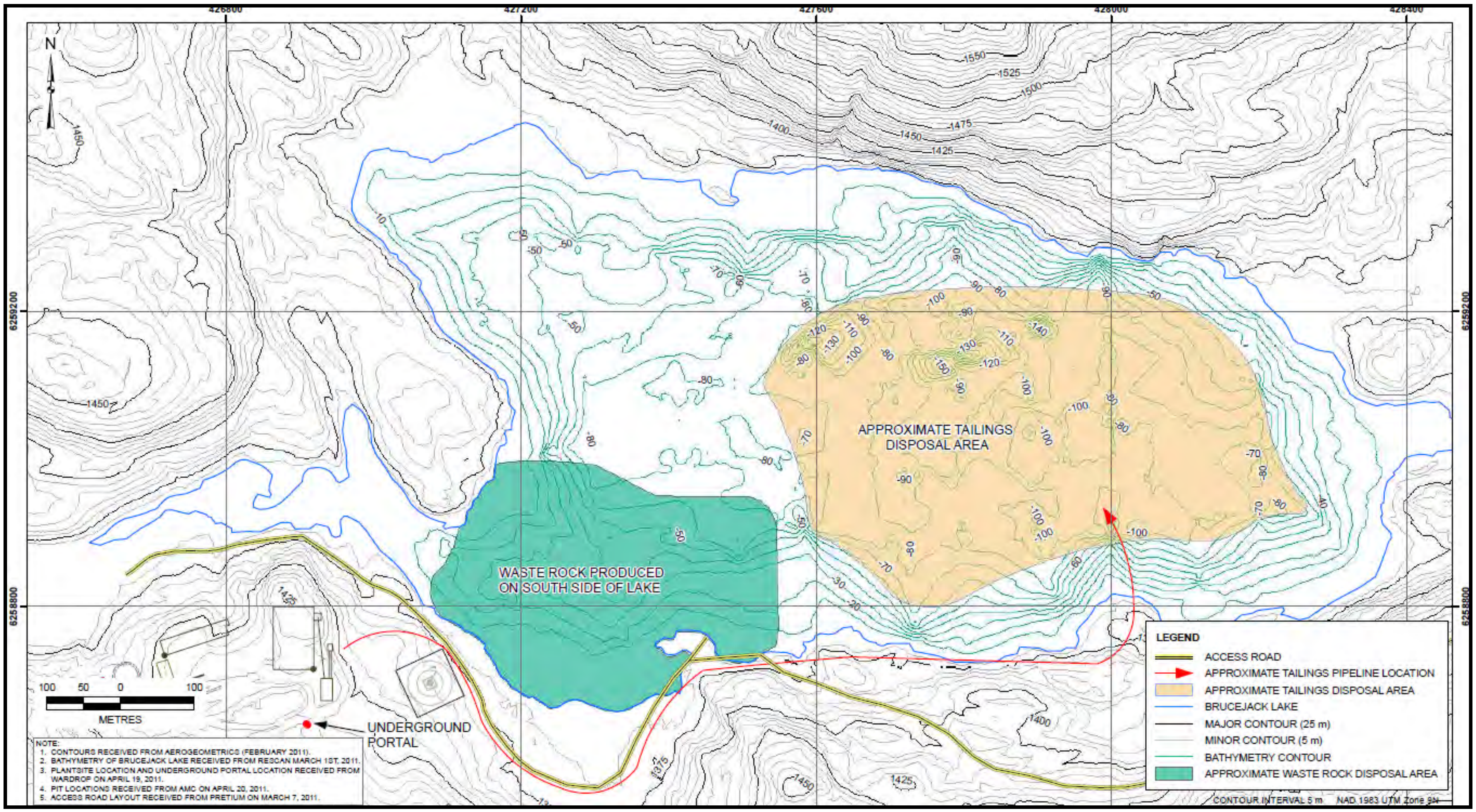
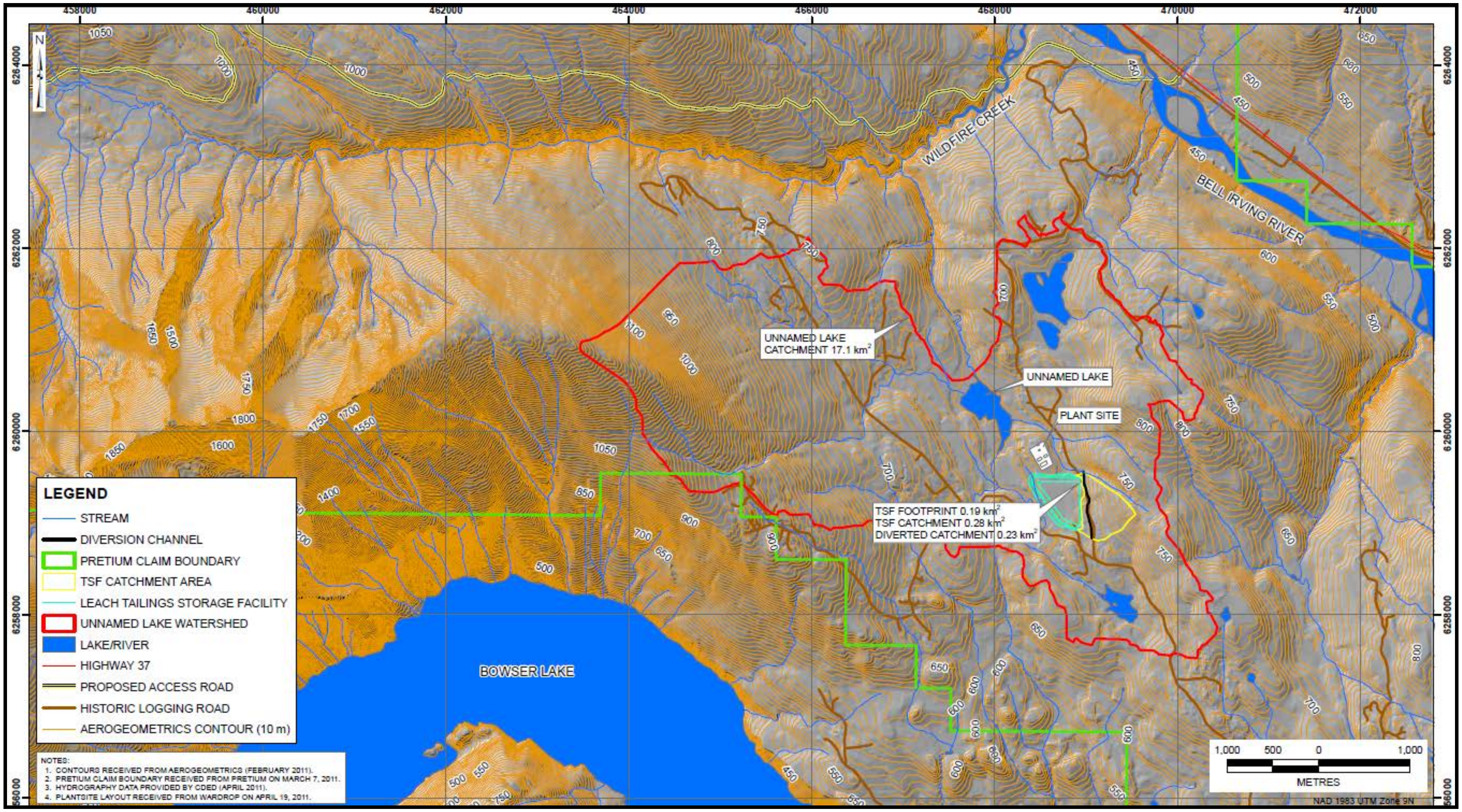


Figure 18.6 Proposed Leach Plant TSF Catchment Areas



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 18.6). 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 SG 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 PE, 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.

18.2.4 Leach Plant Tailings Storage Facility Design

Summary

The leach plant TSF is designed to store 2.4 Mt of tailings, based on a leach plant 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 some excavation within 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 18.7). The ultimate crest elevation of the dam is 647 masl with an ultimate maximum height of approximately 26 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 TSF was selected over other sites within the same general area (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.

TETRA TECH WARDROP

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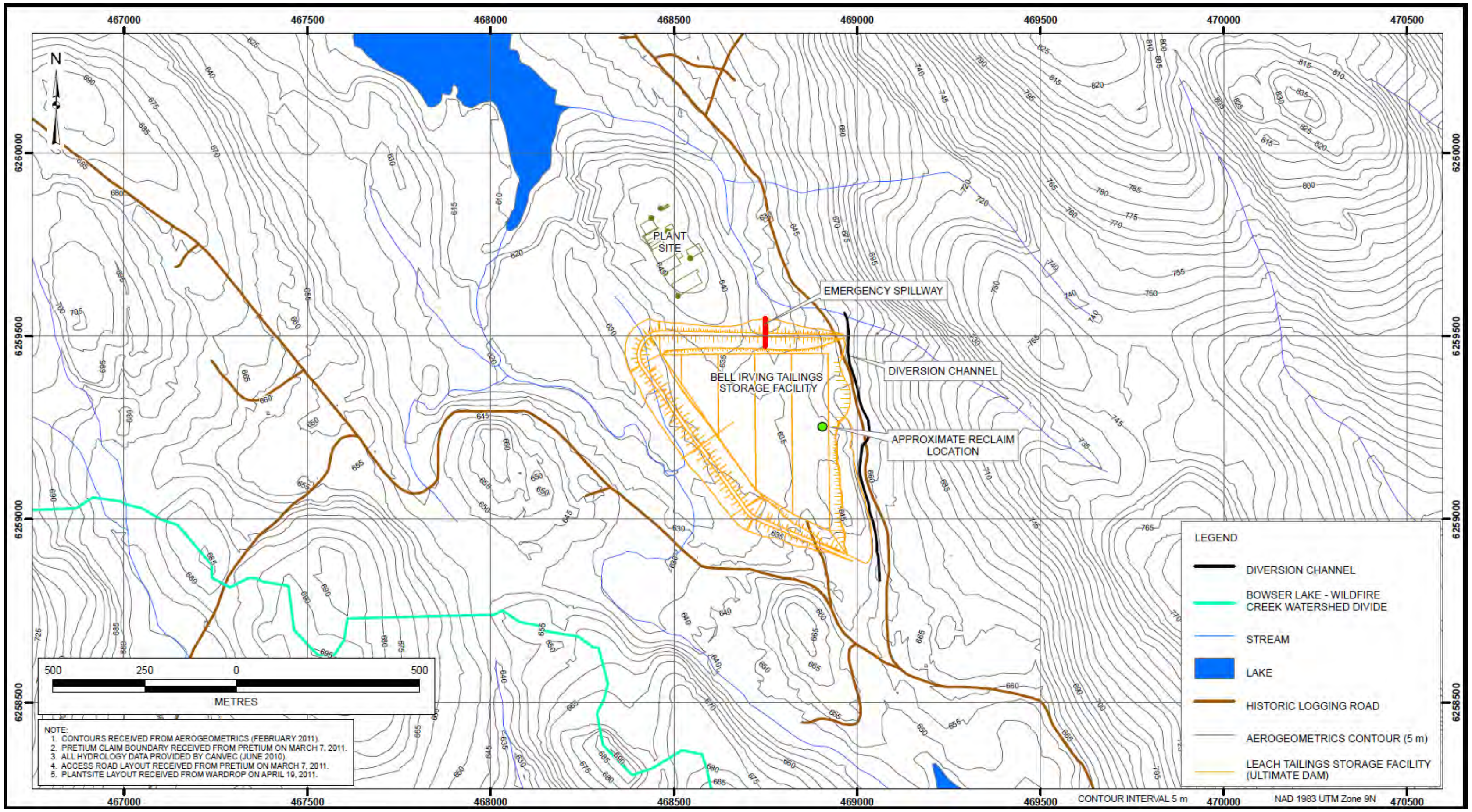
Figure 18.7 Leach Plant Area TSF Plan View



DESIGN CRITERIA

Table 18.2 summarizes the design criteria applicable to the leach plant TSF. The design criteria were established in conjunction with Pretium.

Table 18.2 Preliminary Criteria for Leach TSF Design

Criteria	Description/Comments
Capacity and Throughput	
Total Mineralized Material Mined	11.8 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	2.4 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 to 30 µm). The dry density of the leach tailings are 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 2010.

Tailings Storage Facility Design

Facility Design: Impoundment, Dam, Diversions, and Spillways

The leach 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 18.7). The dam cross-section is shown in Figure 18.8. Some grading in the impoundment will be completed to increase storage capacity.

The facility is fully lined with two layers of 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-metre 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 18.8).

In the event of precipitation greater than the design flood storage, an emergency spillway will pass the peak flow associated with the 24-hour PMP event. The emergency spillway will be located on the north side of the impoundment (Figure 18.7), tying into a natural drainage immediately east of the plant site and draining north into Unnamed Lake.

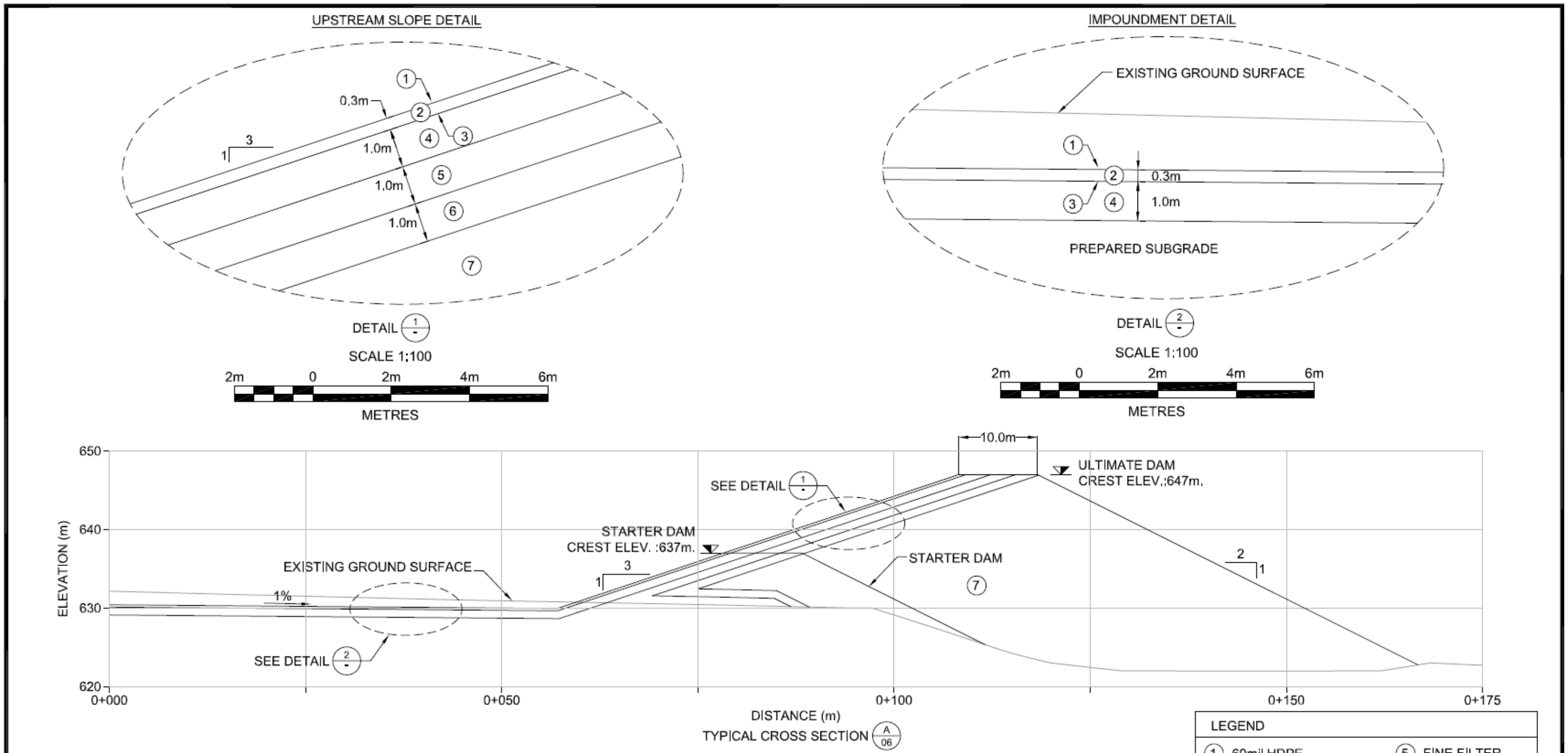
A diversion channel to pass the 200-year peak instantaneous flow will be constructed immediately east of the TSF on the slope above the TSF. If rainfall greater than the design event occurs, it is assumed that the diversion system is overwhelmed and the entire catchment reports to the TSF.

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 discharged from spigots located along 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 18.8 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 mining operations, tailings will be deposited such that there is a relative level to the tailings surface at closure. 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 rock quarries located within less than 1 km 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 to 3 km from the facility.

Tailings Facility Construction

Construction of the 11-metre high 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.

18.3 UNDERGROUND GEOTECHNICAL AND HYDROGEOLOGY EVALUATIONS

Geotechnical evaluations at the PEA 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 Property is assumed to represent a single geotechnical domain. There is insufficient geotechnical information available to determine statistically significant geomechanical differences between the various lithologies within the 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 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 18.3, assume the stope walls and back will be unsupported, and that major geologic structures (i.e. faults) 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 HR exceeds those recommended in Table 18.3.

Table 18.3 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 18.4. 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 of the rock mass.

As part of the geotechnical drilling program, BGC completed eight packer tests in three drillholes to estimate the hydraulic conductivity (K) of the intersected rock mass. Results of packer testing predict K to range from $2x10^{-8}$ m/s to $4x10^{6}$ m/s. Tests were completed at depths ranging from approximately 68 to 323 m below ground surface. Based on available testing data, there is no discernible relationship between bedrock hydraulic conductivity and depth.

Table 18.4 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 m^3/h 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 1,261 masl. This corresponds to a range of 0.02 to 0.06 m^3/h per metre of tunnel with a bulk rock mass hydraulic conductivity of $4x10^{-8}$ m/s. 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 to 0.13 m^3/h per metre of tunnel.

In January 2012 the inflow analysis to the planned underground workings was updated using the new development design provided by AMC for the VOK Zone and the West Zone. The updated inflow estimates range from 0.02 to 0.4 m^3/h for the VOK Zone and from to 0.03 to 0.3 m^3/h for the West Zone. A breakdown of inflow estimates per metre of tunnel for each proposed "level" of the underground workings is shown in Table 18.5. For each elevation level, the estimated average inflow and a range of maximum and minimum inflow rates is presented.

For the VOK Zone, water levels from MW-BGC11-BJ-5A/B, SU-82S/D and SU-77 were used to estimate the maximum (1,527 masl) and average (1,498 masl) water level above the underground workings. For WZ, water levels for MW-BGC11-BJ-1A/B, MW-BGC11-BJ-2A, MW-BGC11-BJ-3A/B, MW-BGC11-BJ-4A, MW-BGC11-BJ-6A and SU-88S/D were used to estimate the maximum (1,385 masl) and average (1,373 masl) water level above the underground workings. The minimum ($1.5x10^{-8}$ m/s) and geometric mean ($1.7x10^{-7}$ m/s) hydraulic conductivities from the 2010 site investigation were used as well as the hydraulic conductivity calibrated to previous inflows ($4x10^{-8}$ m/s). While calculating inflow estimates the following values were used with the analytical model:

- For average inflow per metre tunnel the average tunnel radius (2.9 m), average water level above the proposed workings and the hydraulic conductivity calibrated to previous inflows were used.
- For minimum inflow per metre tunnel the average tunnel radius, average water level above the proposed workings and minimum hydraulic conductivity were used.
- For maximum inflow per metre the maximum tunnel radius (4.5 m), maximum water level above the proposed workings and geometric mean hydraulic conductivity were used.

Additional investigation, analysis and detailed information on the staging of the underground developments will be necessary to refine the inflow estimates. Design of an appropriate dewatering system to mitigate and manage tunnel inflows will require more detailed analyses. This dewatering system should be sized to dewater the existing dormant underground workings before mine operations begin as well as to manage inflows.

Table 18.5 Inflow Estimates Per Metre of Tunnel for Each Level of Proposed Underground Workings at VOK Zone and West Zone

Level #	Elevation (masl)	Average Inflow* (m^3/hr)	Minimum Inflow* (m^3/h)	Maximum Inflow* (m^3/h)
VOK Zone				
1	1,520	n/a	n/a	0.02
2	1,490	0.0040	0.002	0.05
3	1,460	0.0103	0.004	0.08
4	1,430	0.0200	0.006	0.10
5	1,400	0.0200	0.008	0.10
6	1,370	0.0300	0.010	0.10
7	1,340	0.0300	0.010	0.20
8	1,310	0.0300	0.010	0.20
9	1,280	0.0400	0.010	0.20
10	1,250	0.0400	0.020	0.20
11	1,220	0.0500	0.020	0.20
12	1,190	0.0500	0.020	0.30
13	1,160	0.0500	0.020	0.30
14	1,130	0.0600	0.020	0.30
15	1,100	0.0600	0.020	0.30
16	1,070	0.0700	0.030	0.30
17	1,040	0.0700	0.030	0.30
18	1,010	0.0700	0.030	0.40
West Zone				
1	1,405	N/A	N/A	N/A
2	1,375	N/A	N/A	0.03
3	1,345	0.0080	0.003	0.05
4	1,315	0.0100	0.005	0.08
5	1,285	0.0200	0.007	0.10
6	1,255	0.0200	0.009	0.10
7	1,225	0.0300	0.010	0.10
8	1,195	0.0300	0.010	0.20
9	1,165	0.0400	0.010	0.20
10	1,135	0.0400	0.020	0.20
11	1,105	0.0500	0.020	0.20
12	1,075	0.0500	0.020	0.20
13	1,045	0.0500	0.020	0.30
14	1,015	0.0600	0.020	0.30

Note: * Inflow per metre of tunnel

19.0 MARKET STUDIES AND CONTRACTS

The final products to be produced by the 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 feasibility phase of this project.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The 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 northwestern BC, with cool, wet summers and relatively moderate but wet winters. The optimum field season is from late June to mid-October.

Tree line is at approximately 1,200 masl. The West Zone deposit on which the Project is centred is located west of Brucejack Lake at 1,400 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.

20.2 ENVIRONMENTAL SETTING

The Project is located in a remote area for which little baseline environmental data are publically available. Pretium has engaged Rescan, a Vancouver-based consulting firm with extensive BC mining-related environmental assessment (EA) experience, to undertake the baseline studies required for an environmental assessment of the project. At time of writing, baseline studies for the Project was initiated.

TERRAIN, SOILS, AND GEOLOGY

The Project is located in a rugged area with elevations ranging from about 700 m at the planned leach plant and TSF 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 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.

20.2.1 ACID ROCK DRAINAGE

Baseline sampling has just begun for the project, but some prognostications can be based on general knowledge of the region. Although the exploration adit and Brucejack Lake water show no discernable acid rock drainage (ARD) signature, it is probable that elsewhere there will be a reasonably strong chemical signature characteristic of acidic drainage resulting from the oxidation of naturally occurring sulphide minerals. The drainage would likely include elevated concentrations of sulphate, iron, and copper. Elsewhere in the region, seeps around natural gossans indicate natural acid conditions with pH 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 acid base accounting (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.

20.2.2 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. A full meteorological station was established near the Brucejack Lake camp in mid-October 2009 to collect site-specific weather data. 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 Table 20.1 and Table 20.2 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 20.1 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

table continues…

Month	Average Temperature (°C)	Average Precipitation (mm)	Average Evaporation/ Sublimation (mm)
November	-7.9	215	6
December	-9.8	248	5
Average/Total	-2.2	2,034	215

Table 20.2 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).

20.2.3 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 20.3 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 20.3 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. Pretium 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.

20.2.4 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 seasons of sampling have shown that fish do not occur in Brucejack Lake. Fish are also absent from the Sulphurets Creek, into which Brucejack Lake flows, all the way to a barrier just above its confluence with the Unuk River. One season of sampling indicates that fish do not occur in waterbodies in the vicinity of the leach TSF and appear to be prevented from migrating into this subcatchment by a barrier on Wildfire Creek.

20.2.5 ECOSYSTEMS AND VEGETATION

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

20.2.6 WETLANDS

The project encompasses areas of wetland. Wetlands in Canada are valued ecosystem components (VECs) 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.

20.2.7 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. Pretium 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 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.

20.2.8 TRADITIONAL KNOWLEDGE AND TRADITIONAL LAND USE

The 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.

20.2.9 NON-ABORIGINAL LAND USE

The western part of the 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 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.

20.2.10 *Visual and Aesthetic Resources*

The 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.

20.3 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. Pretium 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.

20.3.1 HIGHWAY 16 CORRIDOR

Highway 16 extends from the Prince Rupert port eastwards to Terrace, Hazelton, Smithers, and Prince George. The CNR also follows this corridor. 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.

20.3.2 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.

20.4 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 Section 18.2.

20.4.1 *Water Supply*

Refer to Section 18.2.3.

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

20.4.2 *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.

20.4.3 *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.

20.4.4 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.

20.4.5 CONSTRUCTION WATER MANAGEMENT

Pretium 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.

20.5 WASTE MANAGEMENT

20.5.1 TAILINGS MANAGEMENT

Refer to section 18.2 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. Pretium 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.

20.5.2 Waste Rock Management

Refer to Section 18.2 for more information.

The 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 metal leaching (ML)/ARD.

20.5.3 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. Pretium 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.

20.5.4 Non-hazardous Waste Management

Pretium 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 Pretium's environmental policy. Waste management will involve segregation of wastes into appropriate management channels. Project waste collection/disposal facilities will include one or more incinerators, a permitted landfill, waste collection areas for recyclable and hazardous waste, and sewage effluent/sludge disposal. Most facilities will be duplicated at the mine and plant sites. Waste collection areas will have provisions to segregate waste according to disposal methods, and facilities to address spillage and fire.

20.6 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 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.

Pretium 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 carbon dioxide emissions
- the segregation of waste prior to incineration to minimize toxic air emissions.

20.7 Design Guidance

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

20.8 Social and Community Management Systems

Pretium will develop and implement broad Social and Community Management Systems (SCMS) for the construction, operation, and closure phases of the 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:

- IBAs
- 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.

Pretium 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.

20.9 Design Guidance

20.9.1 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.

20.9.2 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." Pretium 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.

20.9.3 Integration of Traditional Knowledge

Pretium respects the TK of the Aboriginal peoples who have historically occupied or used the project area. Pretium 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. TK will guide aspects of the project, including any future changes once the mine is approved. Pretium 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. Pretium is committed to a process that invites and considers input from people with TK of the project area towards the EA and design of the Project. Pretium is striving to establish a cooperative working relationship with all relevant Treaty and First Nations people to ensure opportunities to gather TK.

20.9.4 Baseline Research

Pretium 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, TK, 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.

20.9.5 VALUED ECOSYSTEM COMPONENTS

Pretium 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 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.

20.9.6 ARCHAEOLOGY AND HERITAGE RESOURCES

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

20.9.7 ENVIRONMENTAL ASSESSMENT STRATEGY AND SCOPE

The EA of the 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. Pretium will use the EA process as an opportunity to refine project design to minimize long-term environmental impacts and to identify appropriate mitigation and management procedures.

20.9.8 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. Pretium'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.

20.9.9 BIODIVERSITY AND PROTECTED SPECIES

Pretium 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 EA process.

20.9.10 Ecosystems and Vegetation

Pretium 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, Pretium will map plant communities and plant species of conservation concern to aid environmental impact assessment.

20.9.11 Environmental Standards

Pretium will design, construct, operate, and decommission the 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 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.

20.9.12 Design for Social and Community Requirements

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

Pretium 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. Pretium will meet its requirements through the development and implementation of the SCMS. The SCMS will define the process by which the company will maintain its involvement and ongoing commitments to communities and stakeholders.

20.10 Consultation Activities

Pretium will initiate a consultation program relevant and useful to each consultation group. The proposed 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 British Columbia Environmental Assessment Office (BCEAO) and CEAA consultation requirements, as well as Pretium's goals for meaningful and sustainable relationships with the leaders and community members affected by and involved in the Project.

Community engagement and consultation are fundamental to the success of the proposed Project and will take place during the project's planning and regulatory review, construction, and operations phases. Prior to beginning the British Columbia Environmental Assessment Act (BCEAA) process, Pretium 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.

20.10.1 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 EA process. Public consultation measures will comply with the *Public Consultation Policy Regulation*, *BC Reg. 373/2002*.

20.10.2 CONSULTATION GROUPS

TREATY AND FIRST NATIONS

Pretium 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

Pretium 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

Pretium will consult with the public and relevant stakeholder groups[1], including: land tenure holders; trappers, guides, outfitters, recreation and tourism businesses; economic development organizations; businesses and contractors (e.g. suppliers and service providers); and special interest groups (e.g. environmental, labour, social, health, and recreation groups).

[1]The public, in this context, pertains to the communities of Smithers, Terrace, Stewart, and Dease Lake. Stakeholders are individuals or groups of people with potential interests or issues with the Project.

20.11 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 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 2014.

Some key milestones for Pretium are:

- the PEA
- the Project Description to BCEAO: to be submitted after PEA update
- prefeasibility: on hold typically two years after submission of project description
- submission of EA: to be determined, typically four months after completion of the FS
- complete feasibility study: to be determined.

20.11.1 BRITISH COLUMBIA ENVIRONMENTAL ASSESSMENT ACT PROCESS

The BCEAA requires that certain large-scale project proposals undergo an EA 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 Project will thus require an Environmental Assessment Certificate, because its proposed production rate exceeds the specified threshold.

20.11.2 AUTHORIZATIONS REQUIRED

Lists of the major federal and provincial licences, permits, and approvals required to constructing, operating, decommissioning, and close the 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.

20.11.3 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 Pretium 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 EA 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 20.4 presents a list of provincial authorizations, licences, and permits required to develop the Project. The list includes only the major permits and is not intended to be comprehensive.

Table 20.4 List of BC 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)

20.11.4 FEDERAL APPROVALS AND AUTHORIZATIONS

Federal approvals that may be required include an authorization from the Federal Minister of Environment approving the combined Application/Comprehensive Study Report for the Project. It is possible that a federal review of the Project may not be triggered by either fisheries or explosives triggers.

Table 20.5 lists some of the federal approvals that may be required.

Table 20.5 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

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

The initial capital cost for the Project was estimated at US$436.26 million with an expected accuracy range of ±35%.

Tetra Tech developed the estimate 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
- Pretium.

The capital cost estimate consists of four main parts:

1. direct costs
2. indirect costs
3. contingency
4. owner's costs.

The capital cost summary and its distribution by area is shown in Table 21.1.

Table 21.1 Capital Cost Summary (Costs in US$)

Description	Labour Cost	Material Cost	Construction Equipment Cost	Process Equipment Cost	Total Cost
Direct Works					
Overall Site	3,074,010	3,685,024	4,596,088	0	11,355,122
Mine Underground (AMC)	66,960	113,775,036	0	502,200	114,344,196
Mine Surface Works (AMC)	0	13,892,222	0	0	13,892,222
Mine Site Process	12,080,834	9,946,889	621,215	12,762,848	35,411,786
Mine Site Utilities	5,128,202	43,195,474	1,168,037	7,143,481	56,635,194
Mine Site Buildings	7,396,948	6,976,915	624,776	1,413,731	16,412,370
Tailings	5,526,144	6,212,492	5,880,924	0	17,619,560
Temporary Facilities	3,917,160	0	0	0	3,917,160

table continues…

Description	Labour Cost	Material Cost	Construction Equipment Cost	Process Equipment Cost	Total Cost
Plant Mobile Equipment (Mine Site)	167,400	0	0	3,566,381	3,733,781
Leach Area	8,909,828	8,286,390	949,719	10,049,789	28,195,726
Leach Area Utilities	5,473,134	1,184,423	1,236,965	8,713,755	16,608,277
Leach Mine Buildings	3,045,043	3,045,600	262,416	153,890	6,506,949
Temporary Facilities	793,755	608,883	2,388	0	1,405,026
Plant Mobile Equipment (Leach Site)	78,120	0	0	2,200,573	2,278,693
Direct Works Subtotal	**55,657,538**	**210,809,348**	**15,342,528**	**46,506,648**	**328,316,062**
Indirect Works					
Indirect	2,648,640	55,563,380	0	0	58,212,020
Owner's Costs	0	11,904,000	0	0	11,904,000
Contingency	0	37,827,393	0	0	37,827,393
Indirect Works Subtotal	**2,648,640**	**105,294,773**	**-**	**-**	**107,943,413**
Total	**58,306,178**	**316,104,121**	**15,342,528**	**46,506,648**	**436,259,475**

21.1.1 *Estimate Base Date and Validity Period*

Tetra Tech has prepared this preliminary assessment estimate with a base date of Q1 2012. No escalation beyond Q1 2012 was applied to the estimate. 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 21.2.

Table 21.2 Project WBS

Major	Areas
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

table continues…

Major	Areas
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.
- 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.

Tetra Tech has assumed the construction man-hours/workweek to be 10 h/d with a 3-week-on/1-week-off rotation.

Owner provided the owners' costs, including taxes.

21.1.2 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.

21.1.3 ELEMENTS OF COSTS

DIRECT COSTS

Labour Rates, Productivity, and Travel Allowances

A blended labour rate of Cdn$100/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.

Tetra Tech 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-week-on/1-week-off 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, Tetra Tech 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 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 database 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.

21.1.4 TEMPORARY WORKS

The estimate is based on catering and housekeeping for the construction staff.

21.1.5 MOBILE EQUIPMENT

The estimate is based on mobile equipment at both the mine site and leach site areas.

21.1.6 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.

21.1.7 Taxes and Duties

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

21.1.8 Freight and Logistics

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

21.1.9 Spares

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

21.1.10 Owner's Costs and Permit Allowances

Owner has provided an allowance of US$11.1 million for Owner's costs and US$1.7 million for Owner's risk.

21.1.11 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 engineering, procurement and construction management (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.

21.1.12 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.

21.1.13 CONTINGENCY

The contingency allowance included in the estimate amounts to US$37.83 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.

21.2 OPERATING COST ESTIMATE

21.2.1 SUMMARY

The total operating cost for the project is estimated at Cdn$170.90/t milled. The estimate includes mining, process, material rehandling at the portal surge stockpile, G&A and surface service costs, excluding residue dam construction costs, which are included in sustaining capital costs. A range from 56 personnel to 138 personnel is required for the mining operation according to the mining schedule. A total of 268 personnel are projected to be required for the operation, including average 114 personnel for mining operation, 111 personnel for process and 43 personnel for surface service and general management.

The unit cost estimates 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 Project has been estimated in Canadian dollars within an accuracy range of ±35%. The breakdown operating cost estimates are presented in Table 21.3.

Table 21.3 Overall Operating Cost

Area	Personnel	Unit Operating Cost (Cdn$/t milled)
Mining*	114	103.63
Processing	111	37.70
Material Rehandling	Included in process	0.02
G&A	30	19.26
Plant Services	13	10.29
Total	**268**	**170.90**

Notes: * Including backfill cost.
Average personnel for the duration of mining.
Unit mining cost is Cdn$103.63/t milled, or Cdn$104.35/t mined (Table 21.4).

21.2.2 MINING OPERATING COST

The underground mine operating cost is shown in Table 21.4. 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 21.4 Mining Operating Cost

Cost Distribution	Unit	Value
Development	Cdn$/t mined	15.84
Production	Cdn$/t mined	88.51
Total	**Cdn$/t mined**	**104.35**

21.2.3 PROCESS OPERATING COST

The estimated process operating cost is Cdn$37.70/t milled or Cdn$20.6 million per year. The cost includes Cdn$17.69/t milled for pre-concentration, Cdn$18.04/t milled for gold leach and recovery and Cdn$1.97/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 t at an operation availability of 92%.

A summary of the plant operation costs is shown in Table 21.5.

Table 21.5 Summary of Process Operating Cost

Description	Labour Force	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Labour Force			
Operating Staff	20	2,444,400	4.47
Operating Labour	57	4,994,659	9.12
Maintenance Labour	34	3,056,619	5.58
Subtotal Labour Force	111	10,495,678	19.17
Major Consumables			
Metal Consumables	-	619,443	1.13
Reagent Consumables	-	3,289,119	6.01
Subtotal Major Consumables	-	3,908,562	7.14
Supplies			
Maintenance Supplies	-	2,228,356	4.07
Operating Supplies	-	1,444,563	2.64
Power Supply	-	2,561,418	4.68
Subtotal Supplies	-	6,234,337	11.39
Total (Process)	**-**	**20,638,576**	**37.70**

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 BC. Benefit burden of 30% for the staff and of 38% for the labour including holiday and vacation payment, pension plan, various other benefits, northern allowance and tool allowance costs.
- Power supply for the mine and process plant at the mine site from electricity transmission line.
- Power supply for the leach plant from on-site fuel power generators.
- Crusher/mill liner and mill grinding media consumptions estimated from the bond grinding media/liner consumption estimate equations and the Tetra Tech database.
- Maintenance supply costs, including building maintenance costs, based on approximately 10% of major equipment capital costs
- Laboratory supplies, service vehicles consumables and other costs based on Tetra Tech's in-house database and industry experience
- Reagent costs based on the consumption rates from test results and quoted budget prices or Tetra Tech database.

The estimated labour force cost is Cdn$19.17/t milled. A total of 111 persons are estimated for the process operation, including 20 staff for management and professional services, 57 operators for operating and assaying, and 34 personnel for maintenance. The estimate is based on 12 hours per shift, 24 h/d and 365 d/a, excluding the crushing operation, which is proposed to operate at 10 h/d.

The operating cost for the major metal consumables is estimated to be Cdn$1.13/t milled. The metal consumables include mill and crusher liners and mill grinding media.

The estimated reagent cost is Cdn$6.01/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 Tetra Tech database.

The maintenance supplies are estimated at Cdn$4.07/t milled while the operating supply cost is Cdn$2.64/t milled.

The power cost is estimated at Cdn$4.68/t milled. The power cost is estimated based on the power requirement of 2.4 MW and a unit electric energy price of Cdn$0.33/kWh for onsite fuel generator on leach plant site and a unit electric energy price of Cdn$0.058/KWh for transmission line on mine site.

21.2.4 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 21.6. Total cost for the process is estimated at Cdn$17.69/t milled. A total of 69 personnel are required to operate the plant. The management and the technical support including assay will service all the processes.

Table 21.6 Operating Cost – Pre-concentration (Mine Site)

Description	Labour Force	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Labour Force			
Operating Staff	8	1,540,900	2.81
Operating Labour	30	2,914,060	5.32
Maintenance Labour	31	1,993,515	3.64
Subtotal Labour Force	69	6,448,475	11.78
Supplies			
Metal Consumables	-	619,443	1.13
Reagent Consumables	-	250,158	0.46
Maintenance Supplies	-	953,356	1.74
Operating Supplies	-	596,050	1.09
Power	-	817,290	1.49
Subtotal Supplies	-	3,233,559	5.91
Total	**69**	**9,684,771**	**17.69**

21.2.5 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$18.04/t milled. The costs are shown in Table 21.7. Their designated personnel will operate the leaching, recovery/refining and cyanide handling circuits. Labour force cost is estimated at Cdn$6.03/t milled. The reagent consumption is the major cost, which is estimated to be Cdn$5.55/t milled. The estimated maintenance supply cost is Cdn$2.33/t milled.

Table 21.7 Operating Cost – Gold Leach and Gold Recovery

Description	Human Power	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Labour Force			
Operating Staff	7	903,500	1.65
Operating Labour	15	1,335,649	2.44
Maintenance	12	1,063,104	1.94
Subtotal Labour Force	34	3,302,253	6.03
Supplies			
Reagent Consumables		3,038,961	5.55
Maintenance Supplies		1,275,000	2.33
Operating Supplies		517,500	0.95
Power Supply		1,744,129	3.19
Subtotal Supplies		6,575,590	12.01
Total	**34**	**9,877,843**	**18.04**

21.2.6 General and Administration and Surface Services

Tetra Tech and Pretium developed the G&A costs, which are estimated to be Cdn$19.26/t milled.

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 is provided in Table 21.8.

Table 21.8 G&A Operating Cost

	Labour Force	Total Cost (Cdn$/a)	Unit Cost (Cdn$/t milled)
G&A Labour Force			
G&A	18	1,982,440	3.62
G&A Hourly Personnel	12	1,020,000	1.86
Subtotal Labour Force	30	3,002,440	5.48
G&A Expense			
General Office Expense	-	150,000	0.27
Computer Supplies including Software	-	52,000	0.10
Communications	-	100,000	0.18
Travel	-	103,000	0.19
Audit	-	52,000	0.09
Consulting/External Assays	-	103,000	0.19
Head Office Allowance: Marketing	-	200,000	0.37
Environmental	-	550,000	1.00
Insurance	-	238,428	0.44
Regional Taxes and Licenses Allowance	-	350,000	0.64
Legal Services	-	103,000	0.19
Warehouse	-	200,000	0.37
Recruiting	-	150,000	0.27
Entertainment/Memberships	-	50,000	0.09
Medicals and First Aid	-	103,000	0.19
Relocation Expense	-	52,000	0.09
Training/Safety	-	200,000	0.37
Accommodation/Camp Costs	-	3,621,165	6.61
Crew Transportation (flight and bus)	-	885,429	1.62
Liaison Committee/Sustainability	-	110,000	0.20
Small Vehicles	-	70,000	0.13
Others	-	100,000	0.18
Subtotal Expense	-	7,543,022	13.78
Total	**30**	**10,545,462**	**19.26**

The surface service cost estimates are shown in Table 21.9 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

- access road maintenance
- avalanche control.

Table 21.9 Surface Services Operating Cost

Surface Service	Labour Force	Total Cost (Cdn$/a)	Unit Cost (Cdn$/t milled)
Surface Service Personnel	13	1,161,510	2.12
Surface Service Expense	-	4,469,700	8.17
Small Vehicles/Equipment	-	103,000	0.19
Potable Water and Waste Management	-	247,200	0.45
Supplies	-	103,000	0.19
Building Maintenance	-	463,500	0.85
Building Heating	-	950,000	1.74
Road Maintenance	-	2,000,000	3.65
Avalanche Control	-	500,000	0.91
Off-Site Operation Expense	-	103,000	0.19
Total	**13**	**5,631,210**	**10.29**

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

An economic evaluation of the Project was prepared by Tetra Tech based on a pre-tax financial model. For the 24-year LOM and 11.8 Mt of mine plan tonnage, the following pre-tax financial parameters were calculated:

- 29.8% IRR
- 4.1-year payback on US$436.3 million initial capital
- US$2,262 million NPV at 5% discount rate.

A post-tax economic evaluation of the project was prepared with the inclusion of applicable taxes (Section 22.6).

The following post-tax financial parameters were calculated:

- 25.0% IRR
- 4.2-year payback on US$436.3 million capital
- US$1,454 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$).

The PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be categorized as mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

Sensitivity analyses were carried out to evaluate sensitivity of the project economics to the key parameters.

22.2 PRE-TAX MODEL

22.2.1 FINANCIAL EVALUATIONS

The production schedule has been incorporated into the pre-tax financial model to develop annual recovered metal production. The annual at-mine revenue 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 the at-the-mine-revenues to derive annual operating cash flows.

Initial and sustaining capital costs have been incorporated on a year-by-year basis over the mine life and deducted from the operating cash flows to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of doré; sustaining capital includes expenditures for mining and milling additions, replacement of equipment, and tailings embankment construction.

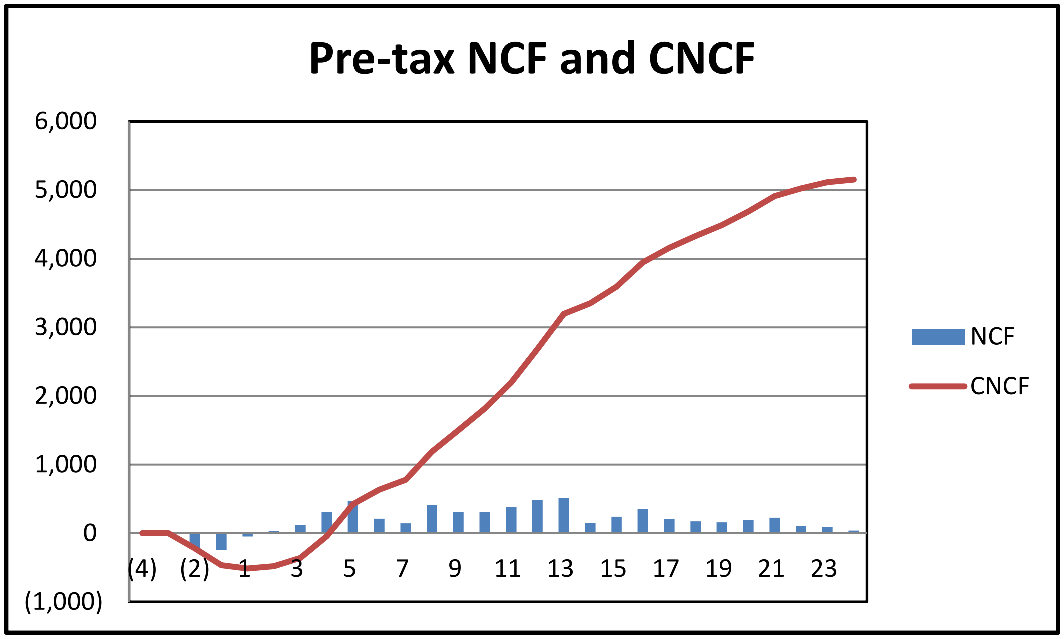
Working capital has been calculated based on a three-month operating cost in Year 1 of the mine operation. It will be recovered at the end of the mine life.

Metal production quantities are presented in Table 22.1. The annual pre-tax net cash flows (NCFs) and cumulative net cash flows (CNCF) are presented in Figure 22.1.

Table 22.1 Metal Production Quantities

Metal	Average Annual Production		Total Production	
	Years 1 to 12	LOM	Years 1 to 12	LOM
Gold ('000 oz)	325	287	3,899	6,878
Silver ('000 oz)	444	710	5,333	17,030

Figure 22.1 Pre-tax Cash Flow



22.2.2 *METAL PRICES SCENARIOS*

The financial outcome for the two metal price scenarios has been tabulated for NPV, IRR, and payback of capital. A discount rate of 5% was applied to both cases identified by the following metal price scenarios:

- base case
- spot metal prices as of February 17, 2012.

The summary of pre-tax project economic evaluation is presented in Table 22.2.

Table 22.2 Summary of Pre-tax NPV, IRR, and Payback by Metal Price

Economic Returns	Unit	Base Case	Spot Prices*
NCF	US$ million	5,133	9,467
NPV at 5.0% Discount Rate	US$ million	2,262	4,330
Project IRR	%	29.8	43.4
Payback	Years	4.1	3.2
Exchange Rate	US$:C$	0.93	0.997
Gold Price	US$/oz	1,100	1,733.6
Silver Price	US$/oz	21.00	33.46

Note: *Spot prices as at February 17, 2012.

The summary of post-tax project economic evaluation is presented in Table 22.3.

Table 22.3 Summary of Post-tax NPV, IRR, and Payback by Metal Price

Economic Returns	Unit	Base Case	Spot Prices*
NCF	US$ million	3,357	6,185
NPV at 5.0% Discount Rate	US$ million	1,454	2,808
Project IRR	%	25.0	36.5
Payback	Years	4.2	3.3
Exchange Rate	US$:C$	0.93	0.997
Gold Price	US$/oz	1,100	1,733.6
Silver Price	US$/oz	21.00	33.46

Note: *Spot prices as at February 17, 2012.

The PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be categorized as mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. There is no certainty that the PEA will be realized.

22.2.3 ROYALTIES

The royalties applicable to the Project are as follows:

- "Royalty" means the amount payable by the Owner, calculated as 1.2% of the at-mine-revenue, with the following exemptions:
  - gold: the first 503,386 oz produced from the Property
  - silver: the first 17,907,080 oz produced from the Property.

22.3 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.

22.4 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.

22.5 Sensitivity Analysis

Sensitivity analyses were carried out on the following parameters by changing one parameter at a time between -30 to +30% at 10% intervals while holding the rest of parameters constant:

- gold price (AuP)
- silver price (AgP)
- exchange rate (FOREX)
- operating cost (OPEX)
- capital cost (CAPEX).

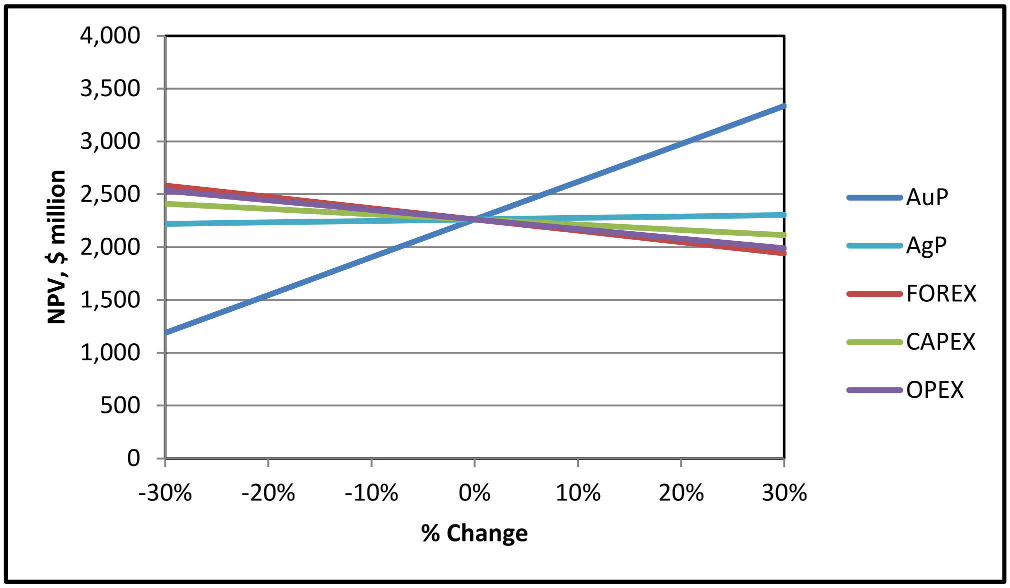
The analyses are presented as financial outcomes in terms of NPV in Figure 22.2, IRR in Figure 22.3 and payback period in Figure 22.4. The project NPV (at 5% discount rate) is most sensitive to the gold price and less sensitive to the rest of parameters.

Similarly, the project IRR is most sensitive to gold price followed by CAPEX, FOREX, OPEX and silver price. Payback period is most sensitive to the gold price and less sensitive to the rest of parameters.

The PEA is preliminary in nature and includes Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied that would enable them to be categorized as mineral reserves. Mineral resources that are not mineral reserves do not have

demonstrated economic viability. There is no certainty that the PEA will be realized.

Figure 22.2 NPV (5%) Sensitivity Analysis



Note: The lines representing the grade and the price of gold overlay

Figure 22.3 IRR Sensitivity Analysis

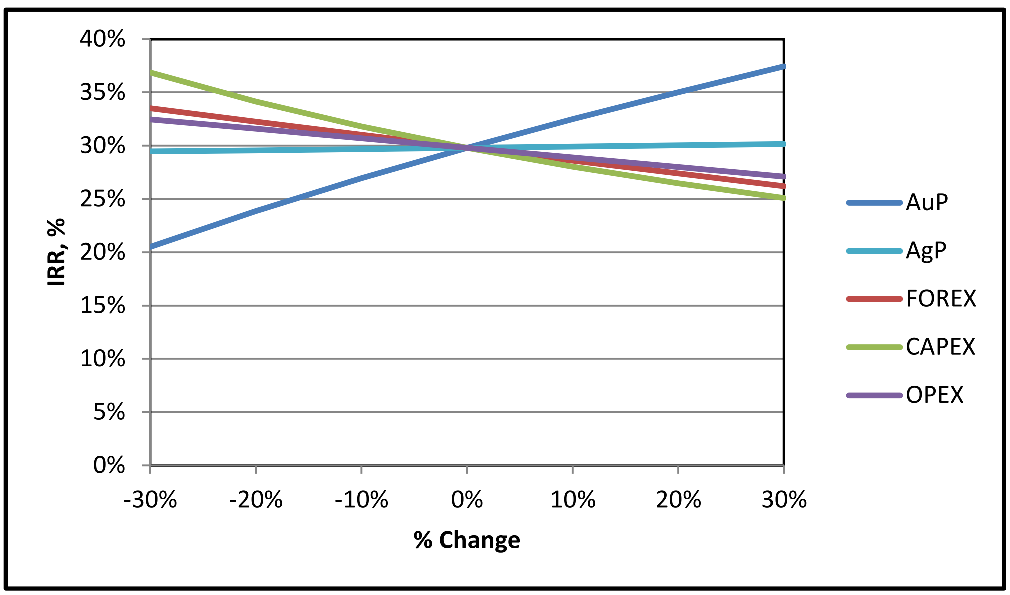
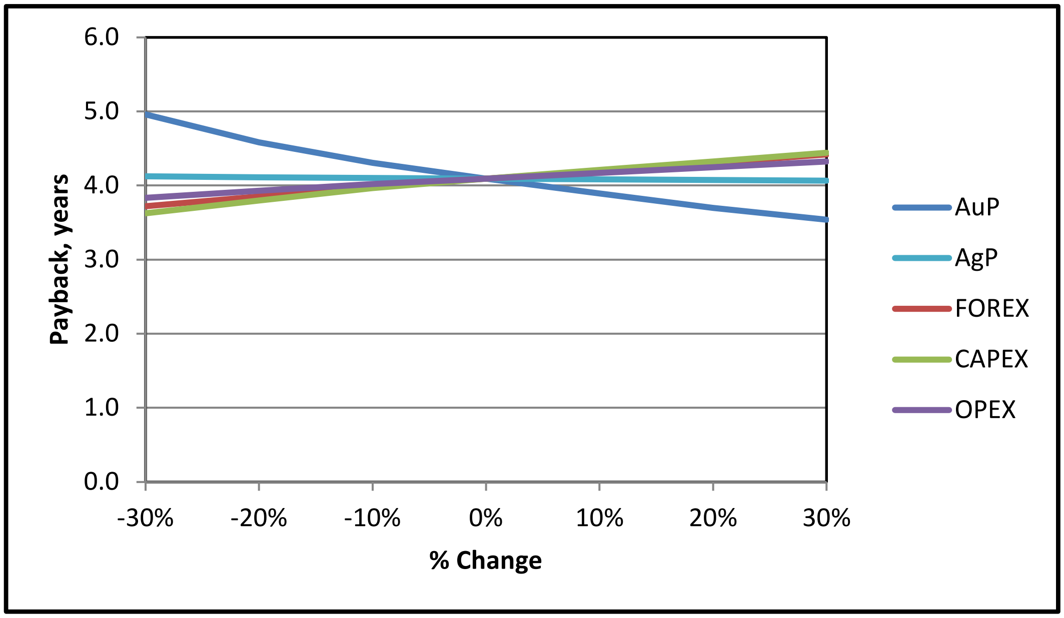


Figure 22.4 Payback Period Sensitivity Analysis



22.6 TAXES

22.6.1 CORPORATION TAXES – FEDERAL

A rate of 15% will be assessed on taxable income. Accelerated provisions apply in determining taxable income. These include deductions for the following:

- exploration and pre-production development expenditures at 100% to the extent not financed with flow-through shares
- 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 22.2.2).

22.6.2 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.

22.6.3 MINING TAXES – PROVINCIAL

Two taxes apply:

- provincial net current proceeds – at 2% on net revenue less operating cost
- net provincial revenue tax – at 13%.

23.0 ADJACENT PROPERTIES

Within the adjacent KSM property there are four notable copper-gold mineral deposits, named the Kerr, Mitchell, Sulphurets and Iron Cap Zones. All of these occurrences are situated within the claim holdings currently owned by Seabridge.

Seabridge acquired the property from Placer Dome in June 2000.

On May 2011, an updated PFS increased mineral reserves, extended mine life, estimated base case operating costs as US$105/oz during the first seven years and improved base case total net cash flow by US$4.5 billion. Engineering work is now focused on evaluating two different throughput expansion scenarios and potential underground mining in the later years. These results indicate an estimated Reserve statement as shown in Table 23.1. All information for this section has been taken from the Seabridge website at www.seabridgegold.net.

Table 23.1 KSM Reserves as of May 2011

Zone	Reserve	Mt	In Situ Average Grades				Contained Metal			
			Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Au (Moz)	Cu (Mlb)	Ag (Moz)	Mo (Mlb)
Mitchell	Proven	617.9	0.64	0.17	3.06	60.2	12.6	2,279	61	82
	Probable	848.6	0.59	0.16	3.02	61.8	16.0	3,040	82	116
	Total	1,466.5	0.61	0.16	3.04	61.2	28.7	5,320	143	198
Iron Cap	Probable	334.1	0.42	0.20	5.46	48.4	4.5	1,490	59	36
Sulphurets	Probable	179.1	0.62	0.26	0.61	59.8	3.6	1,021	4	24
Kerr	Probable	212.7	0.25	0.46	1.28	Nil	1.7	2,155	9	Nil
Totals	**Proven**	**617.9**	**0.64**	**0.17**	**3.06**	**60.2**	**12.6**	**2,279**	**61**	**82**
	Probable	**1,574.5**	**0.51**	**0.22**	**3.03**	**50.4**	**25.8**	**7,706**	**153**	**175**
	Total	**2,192.4**	**0.55**	**0.21**	**3.04**	**53.2**	**38.5**	**9,985**	**214**	**257**

The QPs for this report have not verified the information concerning Seabridge, and the information is not necessarily indicative of the mineralization on the Property.

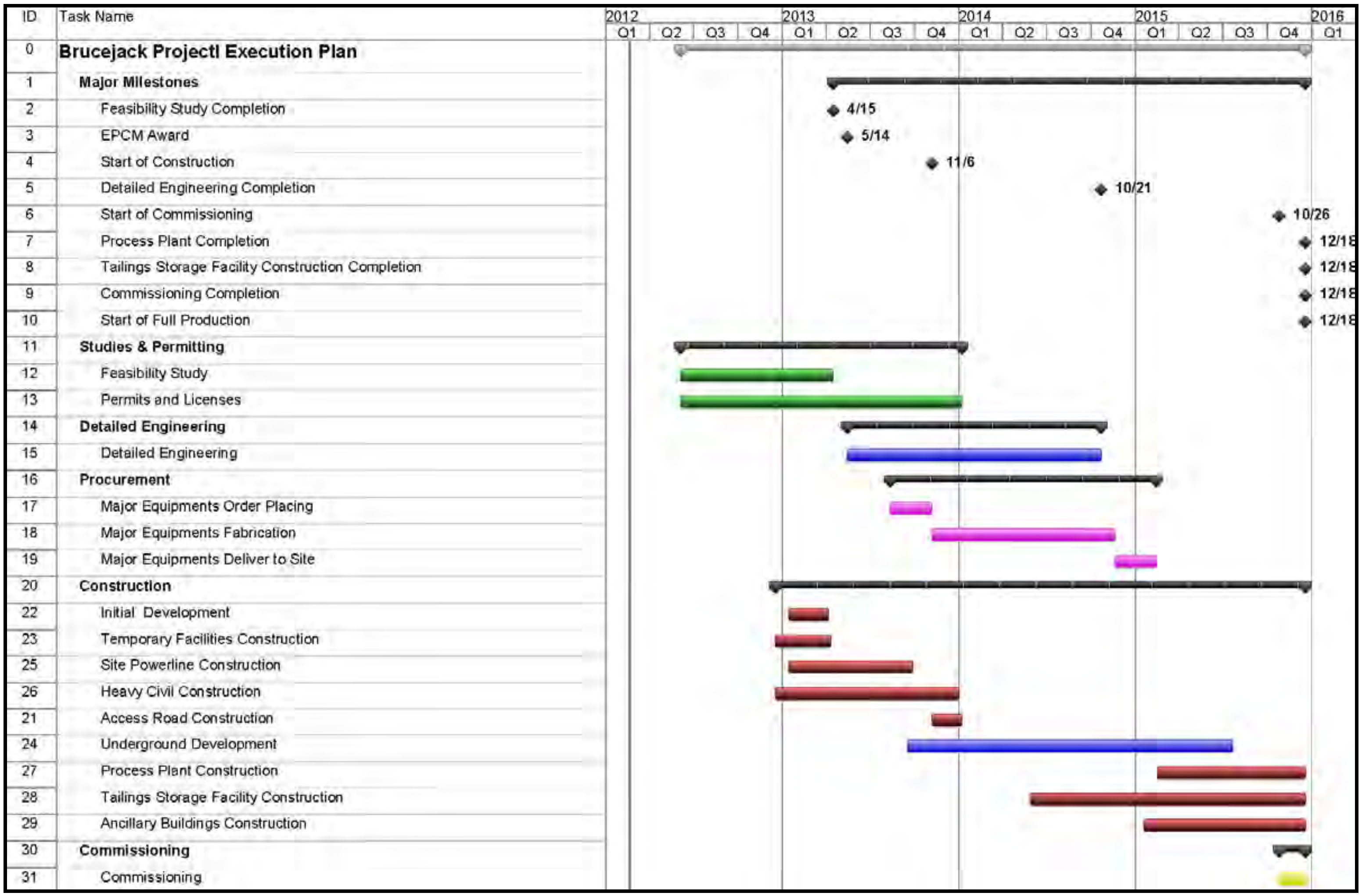
24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 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 24.1.

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 24.1 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.

25.0 INTERPRETATION AND CONCLUSIONS

Based on the results of this PEA, Tetra Tech recommends that Pretium 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 Cdn$20 million and production of the subsequent report is projected to cost approximately Cdn$6 million, for a total of Cdn$26 million.

25.1 GEOLOGY

The current, updated resources on the Project were derived from modelling eight zones and subsequently defining resources in optimized pits at 0.30 g/t AuEq cut-off, 1.25 g/t AuEq cut-off and 5.0 g/t AuEq cut-off. In order to ensure that the reported mineral resources met “reasonable prospects for economic extraction”, conceptual LG optimized pit shells were developed based on all available mineral resources (Measured, Indicated and Inferred).

Commodity prices were based on the three-year tailing average as of November 2011. The results from the optimized pit-shells were used solely for the purpose of reporting mineral resources that have reasonable prospects for economic extraction.

In addition, an underground sensitivity analysis was done simultaneously for the high-grade VOK and West Zone domains, based on a 5 by 5 by 5 m block size. For this analysis the reduced block size model was estimated using the identical parameters and criteria as defined for the parent 25 by 25 by 10 m mineral resource estimate, except that the results of the underground analysis were not restricted to a conceptual pit shell. The inclusion of the underground sensitivity analysis is not meant to supersede or replace the bulk-tonnage mineral resource estimate.

This Project is clearly a superior project with good high grade resources and excellent exploration potential to expand the VOK Zone and develop higher grade resources on numerous targets in other zones, particularly Galena Hill, Gossan Hill, and Bridge Zones.

26.0 RECOMMENDATIONS

26.1 GEOLOGY

It is recommended, based on the current updated resource estimate, to undertake the following at Brucejack:

- Proceed to a full feasibility study on the high grade underground resources for the VOK and West Zones, and all accompanying metallurgical, engineering, and environmental studies
- Develop a ramp to the VOK from the 1330 level drift off the West Zone Ramp
- Complete 24,000 m of infill drilling to expand zones currently left open and to upgrade Inferred resources to Indicated
- Complete non-linear estimation on the high grade mineralization at the Gossan Hill, Galena Hill, and Bridge Zones.

The estimated cost for the recommended work and studies is Cdn$20 million.

26.2 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 VOK 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 suitable pumping system should be designed to extract water from the currently flooded workings and to handle inflows as mining progresses.

26.3 ENVIRONMENTAL

It is recommended that Pretium 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.

26.4 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 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, variable recoveries, mine life, rate-of-return, stockpiling, etc ahead of undertaking the detailed study phase. This should be conducted using the underground specific resource model, to clearly define the cut-off and production rate combination that optimizes Pretium's corporate goals.
- Conduct a backfill options study that examines all practical backfill types, distribution systems and backfill material balances.
- Conduct test work on the backfill materials (identified from the options study) to determine engineered backfill specifications.

- Conduct a full survey of the existing West Zone underground workings.
- Confirm mining methods and stope parameters based on the results of the above recommendations.
- Produce a new stope inventory based on the above underground resource model and cut-off determined from the strategic trade-off study above.
- Complete detailed mine design and schedules incorporating both underground and potential open pit aspects. Schedules and design to consider additional mine access points to the VOK lode (e.g. extension of the VOK spiral ramp to surface breakthrough) to reduce production lead-time and ventilation resistance.
- Undertake adequate engineering and planning to determine the mine ventilation, mine heating and mine infrastructure requirements.
- Undertake detailed mine cost estimates using first principals methodology and mine specific vendor quotes.

26.5 PROCESS AND METALLURGY

The following are recommendations for the next phase of study:

- Further test work is required to confirm the previous testwork findings, optimize the process flowsheet, and investigate metallurgical performances. The test work should be conducted on representative samples and fresh drill core samples. The test work 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.

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28.0 CERTIFICATES OF QUALIFIED PERSONS

28.1 HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, of Vancouver, BC, do hereby certify:

- I am a Manager of Metallurgy with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, BC, V6B 1M1.
- This certificate applies to the technical report entitled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project dated February 20, 2012 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) 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, License #30408. My relevant experience with respect to mineral process engineering includes 22 years of 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, 1.17, 19.0, 21.1 and 24.0 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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.

Signed and dated this 21st day of February, 2012 at Vancouver, BC

"Original document signed and sealed by Hassan Ghaffari, P.Eng."

Hassan Ghaffari, P.Eng.
Manager of Metallurgy
Tetra Tech WEI Inc.

28.2 JOHN HUANG, P.ENG.

I, Jianhui (John) Huang, of Burnaby, BC, do hereby certify:

- I am a Senior Metallurgist with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, BC, V6B 1M1.
- This certificate applies to the technical report entitled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project dated February 20, 2012 (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, 13.0, 17.0, 18.1, 21.2 and 26.5 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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.

Signed and dated this 21st day of February, 2012 at Vancouver, BC

"Original document signed and sealed by John (Jianhui) Huang, P.Eng."

John (Jianhui) Huang, P.Eng.
Senior Metallurgist
Tetra Tech WEI Inc.

28.3 Sabry Abdel Hafez, Ph.D., P.Eng.

I, Sabry Abdel Hafez, Ph.D., P.Eng., of Vancouver, BC, do hereby certify:

- I am a Senior Minerals Economist with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, BC, V6B 1M1.
- This certificate applies to the technical report entitled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project dated February 20, 2012 (the "Technical Report").
- I am a graduate of Assiut University, (B.Sc Mining Engineering, 1991; M.Sc. in Mining Engineering, 1996; Ph.D. in Mineral Economics, 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #34975. My relevant experience is in mine evaluation. I have more than 19 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from the conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties. I have been involved in the technical studies of several base metals, gold, coal, and aggregate mining projects in Canada and abroad. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.16 and 22.0 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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 parts of the Technical Report that I am responsible for 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 parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 21st day of February, 2012 at Vancouver, BC

"Original document signed and sealed by Sabry Abdel Hafez, Ph.D., P.Eng."

Sabry Abdel Hafez, Ph.D., P.Eng.
Senior Minerals Economist
Tetra Tech WEI Inc.

28.4 Pierre Pelletier, P.Eng.

I, Pierre Pelletier, of Vancouver, BC, do hereby certify:

- I am the President of Rescan Environmental Services Ltd. with a business address at 600 – 1111 West Hastings Street, Vancouver, BC, V6E 2J3.
- This certificate applies to the technical report entitled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project dated February 20, 2012 (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, License #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, led 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, 20.0 and 26.3 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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 21st day of February, 2012 at Vancouver, BC

“Original document signed and sealed by Pierre Pelletier, P.Eng.”

Pierre Pelletier, P.Eng.
President
Rescan Environmental Services Ltd.

28.5 TRACY ARMSTRONG, P.GEO.

I, Tracy Armstrong, of Magog, PQ, do hereby certify:

- I am an independent geological consultant contracted by P&E Mining Consultants Inc. whose business address is at 2, County Court Blvd., Suite 202, Brampton, ON, L6W 3W8.
- This certificate applies to the technical report entitled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project dated February 20, 2012 (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 to 1.5, 4.0 to 11.0, 23.0, 25.1 and 26.1 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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.

Signed and dated this 21st day of February, 2012 at Magog, PQ

"Original document signed and sealed by Tracy Armstrong, P.Geo."

Tracy Armstrong, P.Geo.
Independent Geological Consultant
P&E Mining Consultants Inc.

28.6 FRED H. BROWN, M.SC. (ENG), CPG, PR.SCI.NAT.

I, Fred H. Brown, of Lynden, WA, USA, do hereby certify:

- I am an independent geological consultant with a business address at Suite B-10 16010 Grover Street, Lynden, WA, USA.
- This certificate applies to the technical report entitled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project dated February 20, 2012 (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 13 to 15, 2011.
- I am responsible for Section 1.6 and co-authoring Sections 12.0, 14.0 and 26.1 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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.

Signed and dated this 21st day of February, 2012 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.

28.7 CAROLINE J. VALLAT, P.GEO.

I, Caroline J. Vallat, of Nanaimo, BC, do hereby certify:

- I am an independent geological consultant with GeoSpark Consulting Inc. with a business address at 2441 Sun Valley Drive, Nanaimo, BC, V9T 6E8.
- This certificate applies to the technical report entitled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project dated February 20, 2012 (the "Technical Report").
- I am a graduate of the University of Victoria (B.Sc. Honours in Geological Sciences, 25004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, Licence #35680. My relevant experience is as follows: Geological consultant and business owner/operator specializing in QA/QC and database management, GeoSpark Consulting Inc., 2007 (current); Senior Geological Database Manager/Project Manager/Supervisor, Maxwell GeoServices Canada Inc. (2005-2007) and Independent Geological Consultant (2004-2005). I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I am responsible for Section 12.1.1 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 of the Instrument.
- I have had prior involvement with the Property that is the subject of the Technical Report, as database manager and analytical result quality monitoring and reporting.
- 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 21st day of February, 2012 at Nanaimo, BC

"Original document signed and sealed by Caroline J. Vallat, P.Geo."

Caroline J. Vallat, P.Geo.
Independent Geological Consultant
GeoSpark Consulting Inc.

28.8 WARREN NEWCOMEN, P.ENG.

I, H. Warren Newcomen, of Kamloops, BC, do hereby certify that:

- I am a Senior Geotechnical Engineer with BGC Engineering Inc. with a business address at 234 St. Paul Street, Kamloops, BC, V2C 6G4.
- This certificate applies to the technical report titled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project, dated February 20, 2012 (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 #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 Sections 18.3 and 26.2 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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.

Signed and dated this 21st day of February, 2012 at Kamloops, BC

"Original document and sealed by
H. Warren Newcomen, P.Eng."

H. Warren Newcomen, P.Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

28.9 HAMISH WEATHERLY, P.GEO.

I, Hamish Weatherly, of North Vancouver, BC, do hereby certify that:

- I am a Geoscientist with BGC Engineering Inc. 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 Updated Preliminary Economic Assessment of the Brucejack Project, dated February 20, 2012 (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 #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 1.10, 18.2.1, 18.2.3 and 26.2 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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.

Signed and dated this 21st day of February, 2012 at Vancouver, BC

"Original document signed and sealed by Hamish Weatherly, P.Geo."

Hamish Weatherly, P.Geo.
Senior Hydrologist
BGC Engineering Inc.

28.10 LORI-ANN WILCHEK, P.ENG.

I, Lori-Ann Wilchek, of Vancouver, BC, do hereby certify that:

- I am a Senior Geotechnical Engineer with BGC Engineering Inc. 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 Updated Preliminary Economic Assessment of the Brucejack Project, dated February 20, 2012 (the "Technical Report").
- I am a graduate of the University of British Columbia (Bachelor of Applied Science, May 1996) and the University of Alberta (Master of Science, November 2000). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #28802. My relevant experience with respect to tailings storage facility designs includes work for the following projects: Galore Creek Project, BC; Pueblo Viejo Project, Dominican Republic; Lagunas Norte Mine, Peru. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I did not complete a personal inspection of the Property.
- I am responsible for Sections 1.10, 18.2.2, 18.2.4 and 26.2 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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 21st day of February, 2012 at Vancouver, BC

"Original document signed and sealed by Lori-Ann Wilchek, P.Eng."

Lori-Ann Wilchek, P.Eng.
Senior Geotechnical Engineer
BGC Engineering Inc.

28.11 PETER P. 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, BC, V6C 1S4.
- This certificate applies to the technical report entitled Technical Report and Updated Preliminary Economic Assessment of the Brucejack Project, dated February 20, 2012 (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 two days of surface only visits.
- I am responsible for Sections 1.7, 15.0, 16.0, 21.2.2 and 26.4 of the Technical Report.
- I am independent of Pretium Resources Inc. as defined by Section 1.5 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 21st day of February, 2012 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.