Report to:

SEABRIDGE GOLD

Kerr-Sulphurets-Mitchell (KSM) Preliminary Economic Assessment Addendum 2009

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SEABRIDGE GOLD

KERR-SULPHURETS-MITCHELL (KSM) PRELIMINARY ECONOMIC ASSESSMENT ADDENDUM 2009

SEPTEMBER 2009

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REVISION HISTORY

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NOTICE

This report was prepared for Seabridge Gold Inc. (Seabridge) by Wardrop Engineering Inc. (Wardrop), Moose Mountain Technical Services (MMTS), BGC Engineering Ltd. (BGC), Rescan Environmental Services Ltd. (Rescan), McElhanney Consulting Services, Ltd. (McElhanney), Klohn Crippen Berger Ltd. (KCBL), Bosche Ventures Ltd. (BVL), W.N. Brazier Associates Inc. (Brazier), and Resource Modeling Inc. (RMI) (collectively the Project Consultants). This document is meant to be read as a whole. This document contains the expression of the professional opinion of Wardrop, MMTS, BGC, Rescan, McElhanney, KCBL, BVL, Brazier, and RMI based on (i) information available at the time of preparation, (ii) data supplied by outside sources, (iii) conclusions of other technical specialists named in this report, and (iv) the assumptions, conditions, and qualifications in this report. The quality of the information, conclusions, and estimates contained herein are based on industry standards for engineering and evaluation of a mineral project, and are consistent with the intended level of accuracy for a preliminary assessment.

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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 tonne	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 (Metric, 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 tonne	wmt
Year (annum)	a

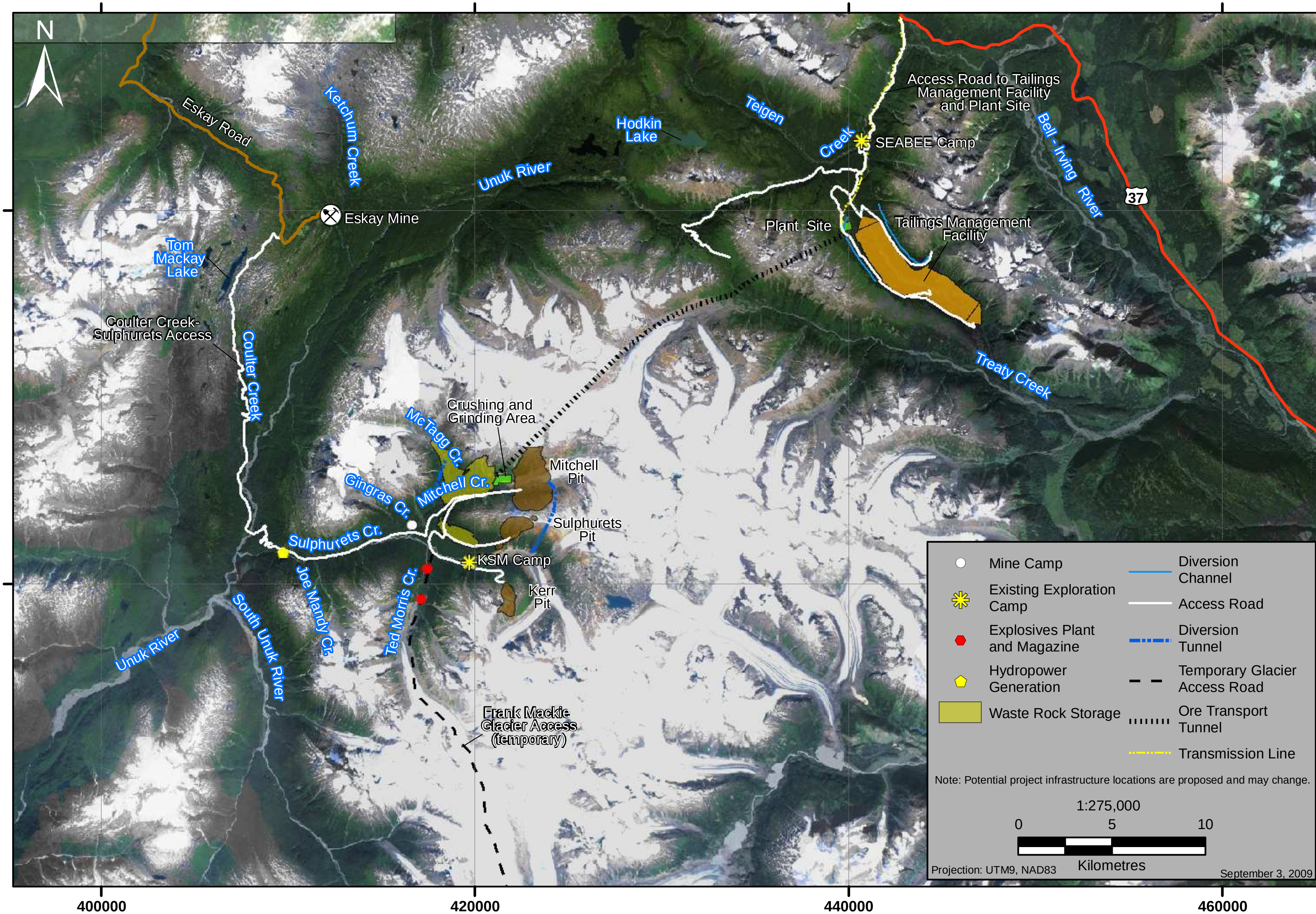
ABBREVIATIONS AND ACRONYMS

3D Block Models	3DBM
Acid Based Accounting	ABA
Acid Rock Drainage	ARD
Acidification-Volatilization-Reneutralization	AVR
Alpine Tundra	AT
ALS Chemex	Chemex
Analytical Solutions Ltd.	ASL
Atomic Absorption Spectrometry	AAS
Atomic Absorption	AA

BC Environmental Assessment Office	BCEAO
BC Public Utilities Commission	BCUC
BGC Engineering Inc.	BGC
Bosche Ventures Ltd.	BVL
British Columbia Environmental Assessment Act	BCEAA
British Columbia Transmission Corporation	BCTC
British Columbia	BC
Canadian Council of Ministers of the Environment	CCME
Canadian Environmental Assessment Act	CEAA
Canadian National Railway	CNR
Carbon-in-Leach	CIL
CDN Resource Laboratories Ltd.	CDN
Chloritic	CL
Closed-circuit Television	CCTV
Coastech Research Inc.	Coastech Research
Coefficient of Variation	CV
Comparative Work Index	CWi
Copper	Cu
Counter-current Decantation	CCD
Customer Base Load	CBL
Cutoff Grade	COG
Cyanidation Circuit	CIL
Distributed Control System	DCS
Eco-Tech Laboratories Ltd.	Eco-Tech
Engelmann Spruce – Subalpine Fir	ESSF
Environmental Assessment	EA
Environmental Management System	EMS
Eskay-Unuk-Mitchell	EUM
Extended Mine Life	EML
Fourth Quarter	Q4
G&T Metallurgical Services Ltd.	G&T
General & Administrative	G&A
General Mine Expense	GME
Global Positioning System	GPS
Gold Equivalent Grade	AuEQ
Gold	Au
High Pressure Grinding Rolls	HPGR
High-density Polyethylene	HDPE
Impact and Benefit Agreements	IBAs
Induced Polarization	IP
Inductively Coupled Plasma	ICP
Interior Cedar – Hemlock	ICH
Internal Rate of Return	IRR
International Plasma Labs Ltd.	IPL
Inverse Distance Cubed	ID^3
Inverse Distance Squared	ID^2

Inverse Distance Weighting	IDW
Iskut Valley Health Services	IVHS
Kerr-Sulphurets-Mitchell	KSM
Klohn Crippen Berger Ltd.	KCBL
Land and Resource Management Plan	LRMP
Lerchs Grossman	LG
Life of Mine	LOM
Locked Cycle Tests	LCT
London Metal Exchange	LME
Maintenance and Repair Contracts	MARP
McElhanney Consulting Services, Ltd	McElhanney
Meta Bisulphite	MBS
Metal Mining Effluent Regulations	MMER
Methyl Isobutyl Carbinol	MIBC
Mineral Titles Online	MTO
MineSight® Economic Planner	MS-EP
MineSight® Strategic Planner	MS-SP
Molybdenum	Mo
Moose Mountain Technical Services	MMTS
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
Nisga'a Lisims Government	NLG
Non-potentially Acid Generating	NAG **or** NPAG
Northwest Transmission Line	NTL
NovaGold Resources Inc.	NovaGold
Official Community Plans	OCP
Operator Interface Stations	OIS
Ordinary Kriging	OK
Ore Research & Exploration Pty Ltd.	OREAS
Original Equipment Manufacturers	OEMs
Other Material	OM
Piteau Associates Engineering Ltd.	Piteau
Potassium Amyl Xanthate	PAX
Potentially Acid Generating	PAG
Predictive Ecosystem Mapping	PEM
Preliminary Economic Assessment	PEA
Primary Cyclone Station	PCS
Probable Maximum Flood	PMF
Probable Maximum Precipitation	PMP
Propylitic	PR
Qualified Person	QP

Quality Assurance/Quality Control	QA/QC
Quantile-Quantile	QQ
Relative Percent Difference	RPD
Rescan Environmental Services Ltd.	Rescan
Resource Modeling Inc.	RMI
Rock Mass Rating	RMR
Rock Quality Designation	RQD
Rotating Biological Contactor	RBC
Run-of-Mine	ROM
Seabridge Gold Inc.	Seabridge
Secondary Cyclone Station	SCS
Semi-autogenous Mill Comminution	SMC
Semi-autogenous/Ball Mill/Crusher	SABC
Sericitic	SE
Silver Standard Resources Inc.	Silver Standard
Silver	Ag
Social and Community Management System	SCMS
Special Direction No. 9	SD9
Specific Gravity	SG
Standard Reference Materials	SRMs
Stantec Consulting Ltd.	Stantec
Static Var Compensators	SVC
Strong PAG	SPAG
Sulphidization-Acidification-Recycling of Precipitate-Thickening	SART
Sulphurets Thrust Fault	STF
Tailing Management Facility	TMF
Terrestrial Ecosystem Mapping	TEM
Thyssen Mining Construction Ltd.	Thyssen
Traditional Knowledge/Traditional Use	TK/TU
Transmission Expansion Policy	TEP
Tunnel Boring Machine	TBM
Unconfined Compressive Strength	UCS
University of Northern British Columbia	UNBC
Valued Ecosystem Components	VECs
Wardrop Engineering Inc.	Wardrop
Weak PAG	WPAG
WN Brazier Associates Inc.	Brazier
Work Index	Wi
Workplace Hazardous Materials Information System	WHMIS



SEABRIDGE GOLD

KSM Project Layout and Road Access

1.0 EXECUTIVE SUMMARY

1.1 INTRODUCTION

The National Instrument 43-101 (NI 43-101) compliant report on the Kerr, Sulphurets, and Mitchell (KSM) property has been prepared by Wardrop Engineering Inc. (Wardrop) for Seabridge Gold Inc. (Seabridge) based on work by the following independent consultants:

- Resource Modeling Inc. (RMI)
- Moose Mountain Technical Services (MMTS)
- WN Brazier Associates Inc. (Brazier)
- Klohn Crippen Berger Ltd. (KCBL)
- Bosche Ventures Ltd. (BVL)
- McElhanney Consulting Services Ltd. (McElhanney)
- BGC Engineering Inc. (BGC)
- Rescan Environmental Services Ltd. (Rescan)

Mr. Michael J. Lechner (P.Geo., RPG, CPG) of RMI visited the property on August 9, 2006, and is the Qualified Person (QP) for all matters relating to the mineral resource estimate.

Mr. Jim Gray (P.Eng.) of MMTS visited the property on September 25, 2008 and is the QP for matters relating to mining, mining capital, and mine operating costs.

Mr. John Huang (Ph.D, P.Eng.) of Wardrop visited the property on September 16, 2008 and is the QP for matters relating to the metallurgical testing review, mineral processing, and process operating costs.

Mr. Frank Grills (P.Eng.) of Wardrop visited the property on September 16, 2008 and is the QP for matters relating to the process capital cost estimate and infrastructure.

Mr. Harold Bosche (P.Eng.) of BVL visited the property on September 16, 2008 and is the QP for matters relating to the infrastructure and tailing delivery and reclaim.

Mr. Neil Brazier (P.Eng.) of Brazier visited the property on September 16, 2008 is the QP for matters relating to power supply.

Mr. Graham Parkinson (P.Geo.) of KCBL visited the property on October 23 to 25, June 9 and 10, as well as June 24 to 29, 2008, and is the QP for matters relating to diversions and seepage collection ponds, tailing dams, tailing access roads, pipeline, haulage, and diversion tunnels, hydro plant, and waste dumps.

Mr. R.W. (Bob) Parolin (P.Eng.) of McElhanney visited the property on June 21, 2008 and is the QP for matters relating to main and temporary access roads.

Mr. Greg McKillop (P.Geo.) of Rescan visited the property on June 9 and 10, and July 29, 2008 and is the QP for matters relating to environmental considerations.

Mr. Warren Newcomen (P.Eng) of BGC visited the property on September 17 to 20, 2008 and is the QP for matters relating to the pit slopes.

This Preliminary Economic Assessment (PEA) addendum has been completed to a +25/-10% level of accuracy.

All dollar figures presented in this section are stated in US dollars, unless otherwise specified. An exchange rate of Cdn$1.00 to US$0.90 has been used.

1.2 GEOLOGY

The KSM property is located in northwest British Columbia (BC) at a latitude and longitude of approximately 56.52°N and 130.25°W, respectively. The property is situated about 950 km northwest of Vancouver, 65 km north-northwest of Stewart, BC and 21 km south-southeast of the Eskay Creek Mine.

The property lies within an area known as “Stikinia”, which is a terrain consisting of Triassic and Jurassic volcanic arcs that were accreted onto the Paleozoic basement. Early Jurassic sub-volcanic intrusive complexes are scattered through the Stikinia terrain and are host to numerous precious and base metal rich hydrothermal systems. These include several well known copper-gold porphyry systems such as Galore Creek, Red Chris, Kemess, and Mt. Milligan.

Seabridge entered into the district to secure the previously identified resources of the Kerr and Sulphurets zones. Between 2002 and 2005, Falconbridge/Noranda conducted target evaluation and testing of several occurrences on the property under an option agreement with Seabridge. That work focused on exploration concepts deemed to be appropriate for Cu-rich porphyry targets. Falconbridge/Noranda withdrew from the KSM Project in 2006 having recognized that the districts potential favoured gold-rich copper porphyry targets. Seabridge followed-up on this previous work delineating the Mitchell Zone, expanding the Sulphurets Zone, and re-evaluating the Kerr Zone.

The Mitchell Zone is underlain by foliated, schistose, intrusive, volcanic, and clastic rocks that are exposed in an erosional window below the shallow north dipping

Mitchell Thrust Fault. These rocks tend to be intensely altered and characterized by abundant sericite and pyrite with numerous quartz stockwork veins and sheeted quartz veins that are often deformed and flattened. Towards the west end of the zone, the extent and intensity of phyllic alteration diminishes and chlorite-magnetite alteration becomes more dominant along with lower contained metal grades. Within the core of the zone, pyrite content ranges between 1 to 20%, averages 5%, and typically occurs as fine disseminations. Gold and copper tends to be relatively low-grade but dispersed over a very large area and appears to be related to hydrothermal activity associated with Early Jurassic hypabyssal porphyritic intrusions. In general, within the currently drilled limits of the Mitchell Zone, gold and copper grades tend to be remarkably consistent between drill holes, which is consistent with a large, stable, and long-lived hydrothermal system.

RMI created a three-dimensional block model for the Mitchell Zone using data from 103 diamond core holes spaced at approximately 100 m intervals totalling 40,416 m of data. Gold and copper grades were estimated with 15-m-long drill hole composites using inverse distance, ordinary kriging, and nearest neighbor methods. RMI validated the estimated block grades using visual and statistical methods. It is RMI's opinion that the Mitchell grade model is globally unbiased and represents a reasonable estimate of insitu resources. Measured, Indicated and Inferred Mineral Resources were classified for a portion of the estimated blocks based on the distance to drilling data coupled with the number of holes that were used in the estimate. A gold equivalent grade (AuEQ) was calculated for the estimated blocks using a gold price of US$650/oz at 70% recovery and a copper price of US$2.00/lb, at 85% recovery. These results are summarized at a 0.50 g/t AuEQ cutoff grade in Table 1.1.

The Sulphurets Zone has been delineated by about 15,207 m of core drilling from 65 drill holes that are spaced at intervals ranging between 50 to 100 m. The majority of the drilling data were collected by Placer Dome and previous operators. The mineralized zone, as currently recognized, consists of two distinct systems referred to as the Raewyn Copper-Gold and Breccia Gold zones which are exposed within the lower plate of the Sulphurets Thrust Fault. The Raewyn Copper Zone hosts porphyry style disseminated chalcopyrite and associated gold in altered sill-like intrusions and volcanic rocks. Hydrothermal alteration in these rocks is characterized by sericite-pyrite-quartz introduction associated with stockwork veins. Gold and copper are concentrated in the stockwork veins and disseminated in the wallrock. The Breccia Gold Zone hosts mostly gold bearing pyrite with minor chalcopyrite and sulfosalts in the matrix to a hydrothermal breccia that cross cut the intrusions of the Raewyn Zone.

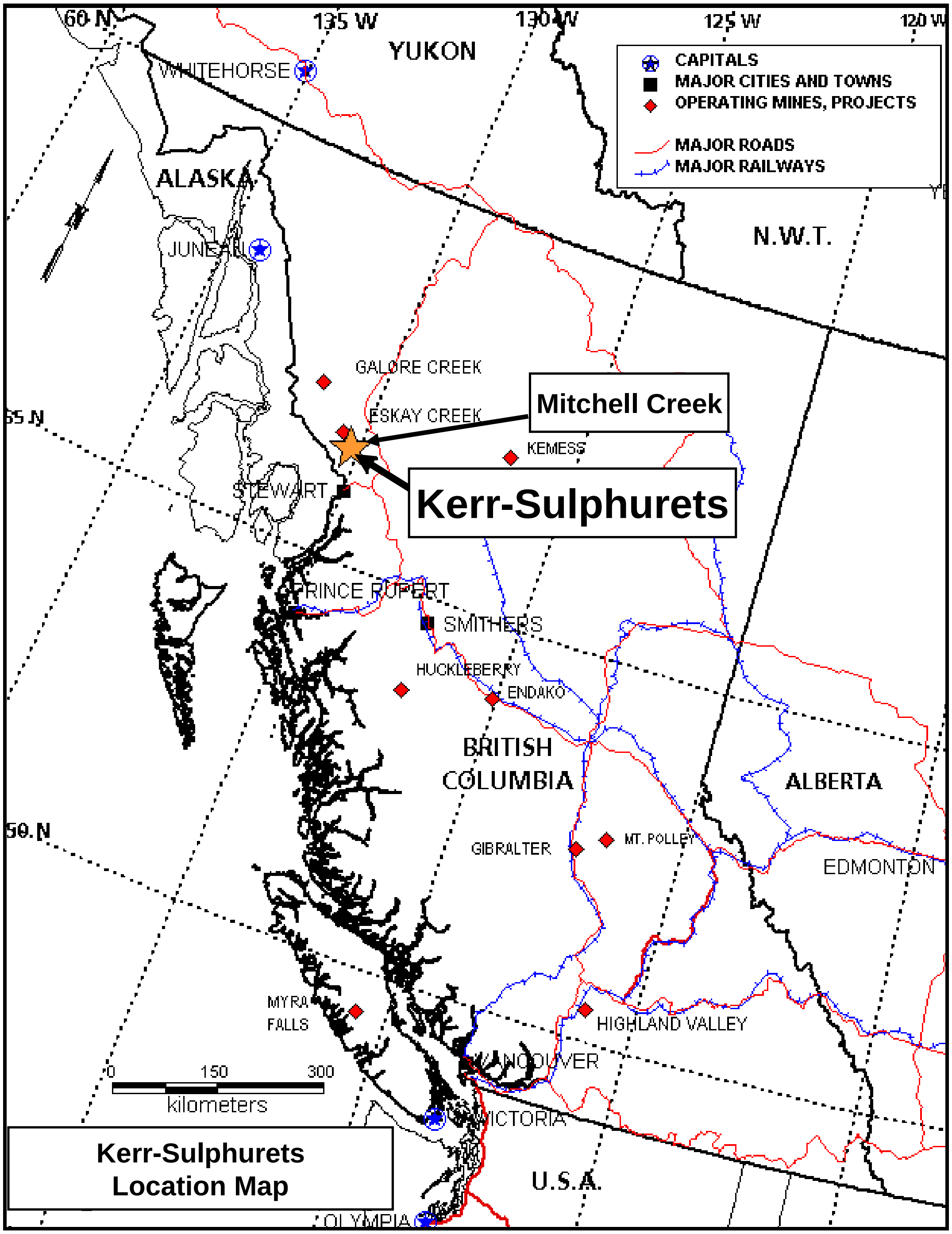
The Kerr Zone has been delineated by about 26,409 m of core drilling from 144 holes that are spaced between 50 to 100 m apart. The majority of the drilling data were collected by Placer Dome and previous operators. The Kerr mineralized zone is characterized by finely disseminated, fracture and veinlet controlled chalcopyrite with minor bornite and tennanite associated with an early Jurassic monzonite porphyry that was intruded into Triassic sedimentary and volcanic rocks. Extensive and intensive hydrothermal alteration of the intrusive rocks and surrounding rocks produced a north-south trending zone of sericite-quartz-pyrite rocks. This hydrothermal alteration trend defines the limits of the copper-gold mineral system.

1.3 PROPERTY DESCRIPTION AND LOCATION

The KSM Project area is located in the coastal mountains of northwestern BC. The proposed pit areas lie within the headwaters of Sulphurets Creek, which is a main tributary of the Unuk River. The proposed Tailing Management Facility (TMF) will be located primarily within a tributary of Teigen Creek. A smaller portion of the proposed TMF, which would not be constructed until well into the operational life of the mine, will be located within a tributary of Treaty Creek. Both Teigen and Treaty Creeks are tributaries of the Bell-Irving River, which is itself a major tributary of the Nass River. Both the Nass and Unuk rivers flow to the Pacific Ocean. Figure 1.1 is a general location map of the project area.

The proposed mining area property consists of 30 contiguous mineral claims and 19 contiguous placer claims. These mineral claims cover an area of approximately 6,726 ha while the placer claims cover about 4,554 ha. It should be noted that most of the placer claims lie “over the top” of the mineral claims. Twenty-six of the mineral claims were converted from 58 legacy claims to BC’s new Mineral Titles Online (MTO) system in 2005; 4 of the claims purchased from a third party remain as legacy claims. Barrick Gold retains a capped 1% net smelter royalty on the property. Seabridge has also acquired 45 contiguous mineral claims (Seabee property) that are located about 19 km northeast of the KSM property. These claims were acquired to secure the mineral rights on the TMF and cover 11,160 ha.

Figure 1.1 General Location Map



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

The property lies in the rugged Coastal Mountains of northwest BC, with elevations ranging from 520 m in Sulphurets Creek Valley to over 2,300 m at the highest peaks. Valley glaciers fill the upper portions of the larger valleys from just below tree line and upwards. The glaciers have been retreating for at least the last several decades. Aerial photos from 1991 indicate the Mitchell Glacier has retreated almost one kilometre laterally and perhaps several hundred metres vertically since then.

The property is drained by Sulphurets and Mitchell watersheds that empty into the Unuk River, which flows westward to the Pacific Ocean through Alaska. The tree line lies at about 1,240 metres above sea level (masl), below which a mature forest of mostly hemlock and balsam fir abruptly develops. Fish are not known to inhabit the Sulphurets and Mitchell watersheds. Large wildlife such as deer, moose, and caribou are rare due to the rugged topography and restricted access; however, bears and mountain goats are relatively common.

The climate is generally that of a temperate or northern coastal rainforest, with sub-arctic conditions at high elevations. Precipitation is high with an annual total precipitation (rainfall and snow equivalents) estimated to be somewhere between the historical averages for the Eskay Creek Mine and Stewart, BC. This range extends from 1,373 to 2,393 mm (data to 2005). The length of the snow-free season varies from about May through November at lower elevations and from July through September at higher elevations. Access to the property is via helicopter.

Deep water loading facilities for shipping bulk mineral concentrates exist in Stewart and are currently used by the Huckleberry mine. The nearest railway is the CNR Yellowhead route, which is located approximately 220 km to the southeast. This line runs east-west and terminates at the deep water port of Prince Rupert on the west coast of BC.

The property lies on Crown land thus all surface and access rights are granted by the Mineral Tenure Act and the Mining Right of Way Act. There are no settlements or privately owned land in this area and no commercial activity, but there is limited recreational activity in the form of helicopter skiing and guided fishing adventures. The closest power transmission lines run along the Highway 37A corridor to Stewart, approximately 50 km to the southeast. The Eskay Creek mine produces its own diesel generated power. There are proposals to develop local hydroelectric power sources and extend the Highway 37A transmission line northward.

1.5 HISTORY

The modern exploration history of the area began in the 1960s, with brief programs conducted by Newmont, Granduc, Phelps Dodge, and the Meridian Syndicate. All of these programs were focused towards gold exploration. Various explorers were attracted to this area due to the numerous large, prominent pyritic gossans that are exposed in alpine areas. There is evidence that prospectors were active in the area prior to 1935. The Sulphurets Zone was first drilled by Esso Minerals in 1969; Kerr was first drilled by Brinco in 1985 and Mitchell by Newhawk Gold in 1991.

In 1989, a 100% interest in the Kerr Zone was acquired by Placer Dome from Western Canadian Mines and in the following year they acquired the adjacent Sulphurets property from Newhawk Gold Mines. The Sulphurets property also hosts the Mitchell Zone and other mineral occurrences. In 2000, Seabridge Resources acquired a 100% interest from Placer Dome in both the Kerr and Sulphurets properties, subject to capped royalties.

There is no recorded mineral production, nor evidence of it, from the property. Immediately west of the property, small-scale placer gold mining has occurred in Sulphurets and Mitchell zones. On the Bruceside property, immediately to the east and currently owned by Silver Standard Resources, limited underground development and test mining was undertaken in the 1990s on narrow, gold-silver bearing quartz veins at the West Zone.

1.6 GEOLOGICAL SETTING

The region lies within “Stikinia”, a terrain of Triassic and Jurassic volcanic arcs that were accreted onto the Paleozoic basement of the North American continental margin in the Middle Jurassic. Stikinia is the largest of several fault bounded, allochthonous terrains within the Intermontane belt, which lies between the post-accretionary, Tertiary intrusives of the Coast belt and continental margin sedimentary prisms of the Foreland (Rocky Mountain) belt. In the Kerr-Sulphurets area, Stikinia is dominated by variably deformed oceanic island arc complexes of the Triassic Stuhini and Jurassic Hazelton groups. An extensive basin formed eastward of the property in the Late Jurassic and Cretaceous that filled with thick accumulations of clastic sedimentary rocks of the Bowser Group. Folding and thrusting due to compressional tectonics in the late Cretaceous generated the area’s current structural features. Remnants of Quaternary basaltic eruptions occur throughout the region.

Early Jurassic sub-volcanic intrusive complexes are common in the Stikinia terrain, and several host well-known precious and base metal rich hydrothermal systems. These include copper-gold porphyry zones such as Galore Creek, Red Chris, Kemess, Mt. Milligan, and Kerr-Sulphurets. In addition, there are a number of related polymetallic zones including skarns at Premier, epithermal veins and subaqueous vein and volcanogenic massive sulfide zones at Eskay Creek, Snip, Bruceside, and Granduc.

1.7 RESOURCES AND MINE PLANNING

RMI constructed three-dimensional block models for the Kerr, Sulphurets, and Mitchell zones. Independent gold and copper grade wireframes were constructed from cross sectional polygons which were then reconciled in bench plan. These wireframes were used by RMI in a multi-pass inverse distance grade interpolation plan. The estimated block grades were validated using visual and statistical methods. Based on those results, it is RMI's opinion that the grade models are globally unbiased and suitable for subsequent pit optimization studies. The estimated block grades were classified into Indicated and Inferred Mineral Resource categories using distance to data in conjunction with the number of drill holes that were used to estimate block grades. The resource information by RMI was reported in a Technical Report filed on SEDAR in March of 2009.

Table 1.1 summarizes the estimated Indicated and Inferred Mineral Resources for each zone. The Mineral Resources tabulated in Table 1.1 were not constrained by conceptual pits although RMI did generate a series of conceptual pits for each zone to test the robustness of the deposits.

Table 1.1 Measured, Indicated and Inferred Mineral Resources for KSM

	Measured Mineral Resources					Indicated Mineral Resources				
Zone	Tonnes (000)	Au (g/t)	Cu (%)	Au oz (000)	Cu lbs (million)	Tonnes (000)	Au (g/t)	Cu (%)	Au oz (000)	Cu lbs (million)
Kerr	No Measured Resources					225,300	0.23	0.41	1,666	2,036
Sulphurets	No Measured Resources					87,300	0.72	0.27	2,021	520
Mitchell	579,300	0.66	0.18	12,292	2,298	930,600	0.62	0.18	18,550	3,692
Total	**579,300**	**0.66**	**0.18**	**12,292**	**2,298**	**1,243,200**	**0.56**	**0.23**	**22,237**	**6,248**

	Measured + Indicated Mineral Resources					Inferred Mineral Resources				
Zone	Tonnes (000)	Au (g/t)	Cu (%)	Au oz (000)	Cu lbs (million)	Tonnes (000)	Au (g/t)	Cu (%)	Au oz (000)	Cu lbs (million)
Kerr	225,300	0.23	0.41	1,666	2,036	69,900	0.18	0.39	405	601
Sulphurets	87,300	0.72	0.27	2,021	520	160,900	0.63	0.17	3,259	603
Mitchell	1,509,900	0.64	0.18	30,842	5,990	514,900	0.51	0.14	8,442	1,589
Total	**1,822,500**	**0.59**	**0.21**	**34,529**	**8,546**	**745,700**	**0.50**	**0.17**	**12,106**	**2,793**

A series of Lerchs Grossman (LG) pit shell optimizations were carried out by MMTS using the resource models provided by RMI.

Mine planning pit optimizations used current projected mining, processing, and general and administrative (G&A) costs and metal recoveries from each of the three separate pit areas: (1) Mitchell, (2) Sulphurets, and (3) Kerr. The 2009 resource definition classifies the mineralization as Indicated and Inferred and both categories were used in the pit optimization. The LG delineated resources are in-situ and use a net smelter return (NSR) cut-off of $6.85 but do not include any mining dilution or mining loss.

MMTS notes that the mine plan incorporates some inferred mineral resources. They are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Therefore MMTS advises that there can be no certainty that the estimates contained in the PEA will be realized.

MMTS identified two sets of potential economic pit limits, a large economic pit limit case called NPV0 and a smaller economic pit limit case called NPV5. Table 1.2 shows that the NPV0 pit has a 51% longer life of mine than the NPV5 LG pit limit, with 44% higher Gold mined and 38% higher Copper mined.

Table 1.2 Comparison of the NPV0 (Larger) – NPV5 (Smaller) LG Economic Pit Limit Resources

	Mineralized Material >Cutoff (kt)	In Situ Grades					Waste (kt)	S/R (t/t)	Copper (M lb)	Au (M oz)
		NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)				
NPV5	1,367,702	21.1	0.210	0.616	2.20	0.0053	2,117,450	1.55	6,346.6	27.1
NPV0	2,062,665	19.8	0.193	0.588	2.60	0.0057	5,294,872	2.57	8,764.9	39.0
Difference	694,963	-1.31	-0.018	-0.028	0.40	0.0004	3,177,422	1.02	2,418.4	11.9
Variance	51%	-6%	-8%	-4%	18%	8%	150%	66%	38%	44%

Note: This table does not include drilling results from the 2009 exploration program.

The starter incremental pit phases will likely be the same for both cases and capital payback should occur in approximately the same time frame.

This PEA Addendum study is based on the smaller NPV5 economic pit limit for the following reasons:

- The next phase of study for the KSM project is a Preliminary Feasibility Study (PFS) which under CIM guidelines requires that only measured and indicated class material be used. There is insufficient time prior to the PFS to meet the drilling required to upgrade the inferred class of mineralized material to measured and indicated inside he NPV0 pit limit. Upgrading the assurance classification inside the smaller NPV5 pit is a more realistic exploration goal and is therefore recommended.
- The design work, economic viability and permitting process to expand the pit to the large NPV0 case can be investigated in future studies without disrupting the current 30 Year mine plan.

Table 1.3 Pit Delineated Resources for KSM from LG Analysis

Pit	Percent of Base Case Pit	Category	Mineralized Material > Cutoff (k/t)	In Situ NSR ($/t)	In Situ Grades Cu (%)	Au (g)	Ag (g)	Mo (%)	Waste (kt)	Strip Ratio
Mitchell	75%	Measured	394,080	$22.5	0.191	0.714	3.07	0.0054	-	-
		Indicated	455,835	$21.4	0.180	0.682	2.88	0.0059		
		Inferred	179,198	$13.4	0.082	0.488	2.44	0.0056		
		Sub-Total	**1,029,113**	**$20.4**	**0.167**	**0.661**	**2.88**	**0.0057**	**1,481,592**	**1.44**
Sulphurets	90%	Indicated	89,820	$25.4	0.254	0.708	0.28	0.0087	-	-
		Inferred	83,768	$21.7	0.188	0.671	0.27	0.0066		
		Sub-Total	**173,588**	**$23.6**	**0.222**	**0.690**	**0.28**	**0.0077**	**483,092**	**2.78**
Kerr	85%	Indicated	146,686	$23.2	0.472	0.259	-	-	-	-
		Inferred	18,315	$21.3	0.398	0.147	-	-		
		Sub-Total	**165,001**	**$23.0**	**0.468**	**0.257**	**-**	**-**	**152,766**	**0.93**
ALL		Measured	394,080	$22.5	0.191	0.714	3.07	0.0054	-	-
		Indicated	692,341	$22.3	0.252	0.596	2.45	0.0064		
		Inferred	281,281	$16.4	0.134	0.521	1.75	0.0059		
		Total	**1,367,702**	**$21.1**	**0.210**	**0.616**	**2.20**	**0.0053**	**2,117,450**	**1.55**

Note: This table does not include drilling results from the 2009 exploration program.

1.8 MINING OPERATIONS

Detailed pit phases are engineered from the results of a LG sensitivity analysis. Pit Delineated Resources, using an NSR cut-off of $6.24, are tabulated in Table 1.4, which includes an estimated 5% mining dilution and 5% mining loss. Dilution grades represent the average grade of material below the incremental cut-off grade for each pit area. Grades used in the mining section of this report have been interpolated by Inverse Distance Weighting (IDW) as described in the resource section of this report. The grade items used are copper (CUIDW), gold (AUIDW), silver (AGIDW), and molybdenum (MOIDW).

Table 1.4 Summary – Indicated and Inferred Pit Delineated Resource

Pit	Mineralized Material (kt)	Diluted Grades					Waste (kt)	Strip Ratio (t:t)
		NSR (Cdn$/t)	CUIDW (%)	AUIDW (g/t)	AGIDW (g/t)	MOIDW (ppm)		
Mitchell								
M621	110,690	26.4	0.215	0.815	2.98	34.6	49,480	0.45
M622i	114,582	20.4	0.151	0.660	2.92	49.3	147,538	1.29
M623i	182,816	20.9	0.165	0.658	2.65	61.7	151,514	0.83
M624i	263,954	19.8	0.156	0.617	2.87	59.0	717,673	2.72
M625i	355,390	18.8	0.159	0.572	2.82	56.8	424,005	1.19
Sub-total	**1,027,432**	**20.4**	**0.164**	**0.634**	**2.83**	**55.0**	**1,490,570**	**1.45**
Kerr								
K612	166,054	22.729	0.465	0.2506	0	0	157,908	0.95
Sulphurets								
S612	174,144	23.402	0.2152	0.6562	0.28	75.7	489,496	2.81
Total	**1,367,630**	**21.1**	**0.207**	**0.591**	**2.162**	**51.0**	**2,137,974**	**1.6**

Notes: NSR is calculated in Cdn$.
* Mill feed produced from ROM Mineralized Material is 1,293 billion tonnes.
This table does not include drilling results from the 2009 exploration program.

The mine production schedule, based on the detailed pit phases above, vary production annually from the three areas to maximize the NPV returns for the project. This work utilized MineSight® schedule optimization. Large-scale shovels, trucks, and mobile equipment were utilized in the mine planning schedules which are then used for the operating cost estimating. A summary of the Production Schedule is given in Table 1.5.

Table 1.5 Summarized Production Schedule

Production Schedule		Y -2	Y -1	Y1	Y2	Y3	Y4	Y5	Y6-10	Y11-20	Y21-30	LOM
Pit to Mill	kt		-	43,201	43,200	39,452	43,200	43,200	208,733	417,803	411,715	1,250,504
Pit to Stockpile	kt	1,675	1,675	2,847	4,063	1,095	2,123	176	7,843	13,306	10,360	45,165
Pit to Sub Grade	kt	643	643	2,467	1,613	5,538	1,524	209	14,014	24,380	21,338	72,268
Stockpile Reclaim	kt		-	-	-	3,748	-	-	7,267	14,197	17,285	42,497
Stockpile Size	kt	1,675	3,350	6,198	10,261	7,608	9,732	9,908	10,484	9,594	2,669	
Total Mineralized Material Mined	kt	2,318	2,318	48,515	48,876	45,985	46,847	43,585	230,590	455,490	443,413	1,367,938
Plant Feed	kt	-	-	43,201	43,200	43,200	43,200	43,200	216,000	432,000	429,000	1,293,001
AUIDW	g/t	-	-	0.826	0.857	0.686	0.628	0.795	0.644	0.582	0.543	0.609
CUIDW	%	-	-	0.214	0.211	0.165	0.129	0.221	0.146	0.164	0.313	0.214
AGIDW	g/t	-	-	2.34	2.90	3.15	2.19	4.19	2.58	2.67	1.18	2.21
MOIDW	ppm	-	-	40.9	41.1	42.9	61.0	25.2	69.0	61.1	39.2	52.0
Total Waste Mined	**kt**	**44,999**	**44,999**	**80,000**	**79,998**	**91,875**	**94,999**	**99,998**	**539,990**	**641,832**	**419,284**	**2,137,974**
Strip Ratio (waste mined/ Mineralized Material mined)	t/t	19.4	19.4	1.6	1.6	2.0	2.0	2.3	2.3	1.4	0.9	1.6
Strip Ratio (waste mined/ Plant Feed	t/t	-	-	1.9	1.9	2.1	2.2	2.3	2.5	1.5	1.0	1.7
Total Material Mined	**-**	**47,318**	**47,318**	**128,515**	**128,874**	**137,860**	**141,846**	**143,583**	**770,579**	**1,097,321**	**862,697**	**3,505,911**
Total Material Moved	**-**	**47,318**	**47,318**	**128,515**	**128,874**	**141,608**	**141,846**	**143,583**	**777,846**	**1,111,518**	**879,982**	**3,548,408**

The mining operations will be typical of open-pit operations in mountainous terrain in western Canada and will employ tried and true bulk mining methods and equipment. There is a wealth of operating and technical expertise, services, and support in western Canada, BC, and in the local area for the proposed operations. A large capacity operation is being designed and large scale equipment is specified for the major operating areas in the mine to generate high productivities, which will reduce unit mining costs and will allow the lowest mining cost to be achieved. Large scale equipment will also reduce the labour requirement on site and will dilute the fixed overhead costs for the mine operations. Much of the general overhead for the mine operations can be minimized if the number of production fleets and the labour requirements are minimized.

Considerable refinement of mine planning and schedules remains to be done during the next study. An improved Resource Model, with 2009 Mitchell and Sulphurets drilling results included, together with expanded geotechnical information on high wall capabilities should improve pit scheduling, optimization results, and overall economics.

1.9 METALLURGICAL TEST REVIEW

Several metallurgical test programs were carried out to assess the metallurgical response of KSM mineralization. The most recent test programs were performed in 2007 and 2008. Laboratory testing programs have developed a conventional grinding and flotation circuit for Mitchell and Sulphurets mineralization producing copper/gold flotation concentrate and additional gold/silver extraction via a leach circuit treating by-product, gold-bearing sulphide concentrates.

According to the metallurgical test results of the 2008 G&T test program, preliminary estimates for copper, gold, silver, and molybdenum metallurgical performances were developed. In the projection, the metal recoveries are based on the combined process of flotation and cyanidation. The flotation process will produce an average 25% copper concentrate grade and a by-product molybdenite flotation concentrate. The cyanidation leach process on gold-bearing pyrite products will produce a gold-silver dore.

1.10 MINERAL PROCESSING

The proposed flotation process is projected to produce a copper/gold concentrate with 25% copper grade containing 60% of the mill feed gold values. Copper flotation recoveries should average 86% with some variability due to copper head changes. A cyanidation circuit (CIL) treating gold-bearing pyrite flotation products will increase the projected overall gold recovery from the Mitchell Zone to around 76%. Silver recovery from the flotation and leaching circuit is expected to be 73% on average. A separate flotation circuit has been included to recover molybdenite from copper concentrate when higher-grade molybdenite mineralization is processed in the mill feed.

The mill feed for the KSM project will be processed at an average rate of 120,000 t/d. The process plant will consist of three separate facilities: an ore crushing/grinding and handling facility at the mine site, a ground ore slurry transportation tunnel facility and a main process facility at the plant site, including secondary grinding, flotation, regrinding, leaching and dewatering.

The primary comminution plant at the Mitchell Valley mine site will reduce the mill feeds from 100% passing 1,200 mm to 80% passing 180 µm by three stages of crushing and one stage of grinding. The crushing will include primary crushing by gyratory crushers, secondary crushing by cone crushers and tertiary crushing by high pressure grinding rolls (HPGR). The primary grinding circuit, consisting of four conventional ball mills, will grind the crushed materials to a particle size of 80% passing 180 microns.

Through a 23-km tunnel, the ground mill feed will be transported by three stages of pumping to the main plant site, which is located north east of the Mitchell pit. The tunnel will also be used for electrical power transmission and providing maintenance services between the main plant site and the Mitchell Valley mine areas.

The main process plant will consist of secondary grinding, flotation, concentrate dewatering, cyanide leaching, gold recovery, and related process facilities. The slurry materials from the primary comminution circuit will be further ground down to 80% passing 125 µm in grinding circuits consisting of ten energy efficient tower mills in closed circuit with hydrocyclones. The ground material will then have copper/gold/molybdenum minerals concentrated by conventional flotation and also produce a gold-bearing pyrite concentrate for gold leaching. Depending on molybdenum content in the copper/gold concentrate, the concentrate will be further processed to produce a copper/gold concentrate and a separate molybdenum concentrate. The gold-bearing pyrite flotation concentrate together with the copper cleaner flotation tailing from the copper/gold cleaner circuit will be leached with cyanide for additional gold and silver recovery. Prior to storing in the tailing facility, the residues from the cyanide leaching circuit will be washed and subjected to cyanide recovery and destruction.

1.11 TAILING AND WASTE MANAGEMENT

The flotation tailing and the cyanide leach residues will be pumped to the tailing management facility (TMF) located near the process plant. This large storage impoundment has capacity for the 30 years of KSM mined resource (1,296,000 kt) with impoundment dam heights of 150 m. Additional storage capacity would be possible by raising the dams or by using another storage area in the Tiegen Creek drainage area. Cyclone sands will be generated from the low-sulphur flotation tailing and used for dam construction to impound the bulk of the tailing products. The high-sulphide gold leach tailing product will also be impounded in the tailing pond and eventually covered by water or low-sulphide flotation tailing product. Water will be managed in the impoundment during operations, by maximizing the return of decanted tailing solutions and minimizing the input of fresh water to the process circuits.

In the Mitchell Valley the waste rock from the operation will be segregated according to its potential to generate acid and soluble metals. A comprehensive testing program using blast-hole cuttings will be established to characterize all rock removed from the pits. This program will be integrated with the ore control program to ensure that mined material is correctly directed to the process plant, the Non-potentially Acid Generating (NPAG) storage area, or the Potentially Acid Generating (PAG) waste storage area.

A PAG waste rock dump will be located adjacent to the Mitchell pit and will be designed to isolate the PAG waste rock from ground water and surface runoff. Leachate resulting from internal moisture and precipitation will flow to the treatment plant where pit seepage and dump waters will be treated prior to release. A conventional high density sludge treatment plant will be employed for the treatment. This plant will also treat haulage tunnel water.

A separate NPAG waste rock dump may also be required. An additional PAG dump may be constructed on the south side of the Sulphurets ridge.

Other overburden will be disposed in the NPAG waste rock dumps. Overburden will be tested for acid generation prior to use.

Some overburden and glacial till will be stored for later use as a cover for the waste rock dumps to create a moisture barrier and a growth medium for eventual revegetation. In addition, the NPAG waste rock will be used as an erosion resistant cover and for basal drains for the PAG waste rock dump and to line runoff channels for non-contact surface water. Much of the current surface area of the zones is barren of vegetation due to the relatively recent glacial ice recession.

1.12 ENVIRONMENTAL CONSIDERATIONS

The KSM project requires certification under both the *British Columbia Environmental Assessment Act* (BCEAA) and *Canadian Environmental Assessment Act (*CEAA) processes. In addition, numerous federal and provincial licences, permits and approvals will be required to use, construct, and operate the project. The BC Environmental Assessment process was initiated in March of 2008 with submittal of a “Project Description” to the BC Environmental Assessment Office (BCEAO). Federal regulatory authorities were also informed of the proposed project at that time

On-site baseline environmental work was initiated by Rescan in the spring of 2008 and continues in 2009, with the second year of a planned two year baseline program.

Rescan is leading this work, the preparation of the environmental assessment and the submissions required to acquire operating permits. Seabridge and its team are involved with consultation meetings with local communities, regulatory agencies, regional and municipal governments, Treaty Nations, and the First Nations to advance the proposed project through the review processes.

1.13 INFRASTRUCTURE

The plant and mine facility layouts are located to take advantage of the natural topography and, to the extent possible, minimize the impact on the environment.

Parallel twin tunnels connected by crosscuts containing the slurry and return water pipelines and services will be constructed to deliver the mill feed for processing and tailing storage. The tunnel will extend from the north side of the Mitchell Zone approximately 23 km to the northeast into the upper reaches of the Tiegen Creek Valley. There is a saddle point approximately 16 km from the Mitchell portal where the tunnel daylights.

Highway 37, a major road access to northern BC passes within 14 km of the KSM Project's proposed tailing site. A preliminary road study by McElhanney proposes a 14 km routing to the plant site and 1km spur road to the Teigen Creek side of the tailing facility. A temporary construction road approximately 15 km long, will be provided from the plant site to the tunnel saddle point to facilitate tunnel construction and PAG rock removal from the tunnel saddle portals. Road access to Mitchell Creek will be provided by a 34 km continuation of the Eskay Creek Mine access road.

Copper concentrates (averaging approximately 1,000 t/d) produced at the process site will be filtered near the plant site and transported 200 km by contract trucking firms on Highway 37 and 37A to a storage site near Stewart, BC. Concentrates will be loaded and shipped via ocean transport to overseas smelters.

1.14 POWER SUPPLY AND DISTRIBUTION

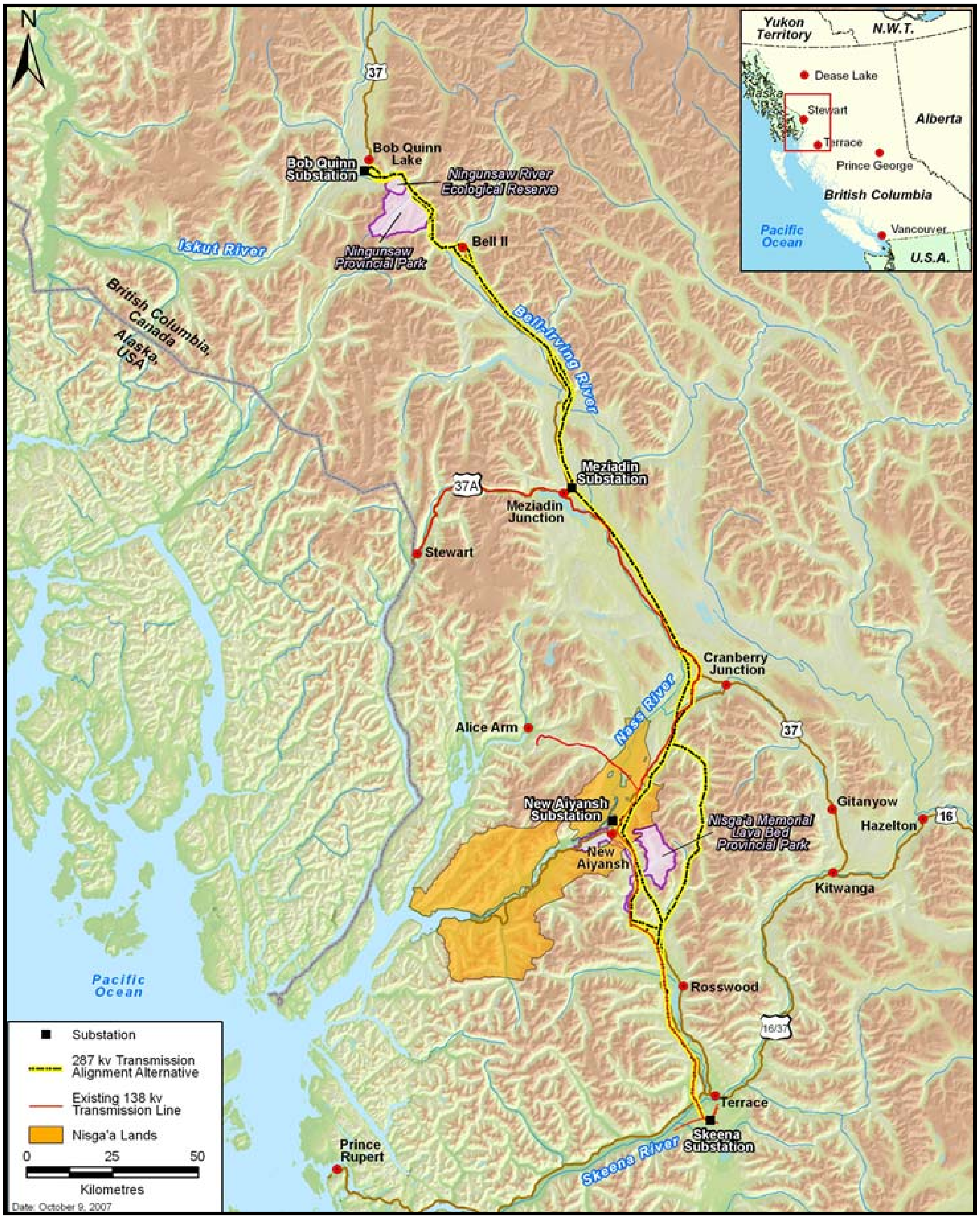
The northern most extension of the current BC Hydro grid in this area of the province is a 220 km long, 138 kV transmission line to Meziadin Junction from the Skeena substation near Terrace, BC. The community of Stewart is provided service by a continuation of the transmission line from Meziadin. The existing 138 kV transmission line does not have adequate capacity to supply an extension to the KSM property. There is a currently proposed new 287 kV "Northwest Transmission Line" (referred to as NTL) from Skeena substation following in proximity to Highway 37 past the KSM property as far north as Bob Quinn Lake. However, due to the uncertainty of this project and the estimated costs, it is proposed to take regular service from BC Hydro at Meziadin Junction under their bulk rate schedule 1823. This will require significant system reinforcement on the part of BC Hydro, including the construction of a new 287 kV transmission line from Skeena to Meziadin (similar to the current NTL plans). As the KSM load is large, in the range of 150 MW, BC Hydro's revenues will be sufficient such that they would under current policy fund this construction, only requiring a bond over a seven year period from KSM. Consequently, the KSM Project would take service at Meziadin and would then be responsible for construction of a 287 kV transmission line from Meziadin to Snowbank Creek, just north of Bell II (102 km in length) and then a further 14 km interconnection to the KSM No. 1 substation, located adjacent to the flotation plant.

Overhead power lines and underground cables will be run from feeder breakers in the 287 kV No. 1 Flotation step-down substation to distribute power around the plant site.

Service to the Mitchell mine and mill site would be provided by a 287 kV cable (23 km in length) through the slurry pipeline tunnel connecting the plant site. This supply would terminate at the 287 to 25 kV step-down Substation No.2 at in the proposed Mitchell plant area. There will be 25 kV cables feeding the mill building and 25 kV overhead power lines extending from the substation to the primary crusher area and around the rim of the open pit mines to service pit equipment.

The PEA capital and operating cost estimates were developed with these electrical service concepts. A map of the proposed KSM and BC Hydro transmission lines, which would be essentially the same as the proposed NTL installation, is shown in Figure 1.2.

Figure 1.2 Map of the Proposed Northwest Transmission Line



The recommended power supply option involves construction of 103 km of 287 kV transmission line from Meziadin Junction, generally parallel to Highway 37, to Snowbank Creek, a point just north of Bell II. The plan is based on use of the same right-of-way and the associated environmental assessment review process, currently underway, for the NTL project and assumes cooperation by BCTC and the BC government. The required environmental studies for the NTL are currently proceeding.

The 287 kV branch line to the mine (also by KSM) includes 14 km of 287 kV transmission line generally following the mine access road.

1.15 Capital Cost Estimate

An initial capital of US$3.083 B is required for the project, based on capital cost estimates developed by the following consultants:

- MMTS – mine capital costs
- KCBL – tailing, water management and mine waste construction costs
- BVL – conveying, and piping costs
- Wardrop assisted by Thyssen Mining Construction Ltd. (Thyssen) – tunnel costs
- Wardrop – process plant and associated infrastructure costs
- Brazier – power supply costs
- McElhanney – access road costs.

All currencies in this section are expressed in United States dollars (US$). Costs in this report have been converted using a fixed currency exchange rate of Cdn$1.00 to US$0.90. The expected accuracy range of the capital cost estimate is +25%, -10%.

Initial capital has been designated as all capital expenditures required to produce concentrate and dore. A summary of the major capital costs is shown in Table 1.6.

Table 1.6 Capital Cost Summary

Description	US$000
Direct Works	
Overall Site	84,000
Mining	320,000
Minesite Crushing and Grinding	381,000
Tunnel Pumping	122,000
Plantsite Grinding and Flotation	248,000
Tailing Dam	118,000
Ore Haulage Tunnel	138,000
Mitchell Diversion Tunnel	36,000
Mitchell Diversion Hydro Plant	3,000
Water Treatment	91,000
Site Services and Utilities	11,000
Ancillary Buildings	65,000
Plant Mobile Fleet	6,000
Temporary Services	121,000
Roads, Power & Infrastructure	258,000
Subtotal	**2,002,000**
Indirects	
Project Indirects	645,000
Owner's Costs	45,000
Contingencies	391,000
Subtotal	1,081,000
Total Capital Cost	**$3,083,000**

1.16 OPERATING COST ESTIMATE

The operating cost for the KSM Project was estimated at US$10.57/t milled. The estimate was based on an average annual process rate of 120,000 t/d milled.

The updated costs in this section are stated in Q2 2009 US dollars, however, the remaining costs are in Q3 2008 US dollars. When it was required, certain costs in this report have been converted using a fixed currency exchange rate of Cdn$1.00 to US$0.90 from Seabridge. The expected accuracy range of the operating cost estimate is +25%, -10%.

Power will be supplied by grid lines at an average cost of US$0.039/kWh. Process power consumption estimates are based on the Bond work index equation for specific grinding energy consumption and estimated equipment load power draws for the rest of the process equipment. The power cost for the mining section is included in the mining operating cost. Power costs for surface service is included in site services.

Table 1.7 Average Operating Cost Summary

	US$/a (000's)	US$/t Milled
Mine		
Mining Costs - Mill Feed	173,744*	4.02*
Mill		
Staff & Supplies	176,544	4.03
Power (Process only)	40,567	0.93
G&A and Site Service		
G&A	32,213	0.75
Site Service	5,913	0.14
Tailing and Water Treatment		
Tailing	6,610	0.15
Water Treatment	23,905	0.55
Total	**459,526**	**10.57**

* including pre-production operating costs of US$168.2 M.

The operating costs are defined as the direct operating costs including mining, processing, tailing storage, water treatment, and G&A. Sustaining capital includes all capital expenditures after the process plant has been put into production.

1.17 ECONOMIC EVALUATION

Metal revenues projected in the KSM cash flow models were based on the average metal values indicated in Table 1.8.

Table 1.8 Metal Production from KSM Project

	Years 1 to 8	Life of Mine
Total Tonnes to Mill (000s)	345,601	1,293,001
Annual Tonnes to Mill (000s)	43,200	43,200
Average Grades		
Gold (g/t)	0.711	0.609
Copper (%)	0.176	0.215
Silver (g/t)	2.74	2.21
Molybdenum (ppm)	52.8	51.9
Total Production		
Gold (000s oz)	6,130	19,278
Copper (000s lb)	1,091,872	5,259,442
Silver (000s oz)	22,249	67,054
Molybdenum (000s lb)	14,859	60,043
Average Annual Production		
Gold (000s oz)	766	644
Copper (000s lb)	136,484	175,721
Silver (000s oz)	2,781	2,240
Molybdenum (000s lb)	1,857	2,006

A full production schedule, which maximizes mine and mill production, was carried forward to a cash flow analysis. In the base case scenario, the three year average prices for gold, copper, silver, and molybdenum were used. The cash flow analysis for this scenario shows that the project has a 30 year mine life and a positive cash flow of US$11.57 billion at a 0% discount rate. The analysis shows that the project has a positive net present value (NPV) of $3.424 billion at 5% discount rate. The project NPV decreases to $1.356 billion in the alternate case but increases to $3.703 billion when using the metal spot prices on July 17, 2009. With the base case three-year metal price average, the cash cost per ounce of gold (net of by-product credits) is negative US$51.00. The corresponding total cost per ounce of gold produced is US$178.00.

The financial analysis shows that the internal rate of return (IRR) will be 12.6% for the base case and will decrease to 8.5% for the alternate case and increase to 13.6% for the spot price case. The payback period is 6.6 years for the three- year base case, 8.8 years for the alternate case, and 5.8 years for the spot price case.

Table 1.9 summarizes the key inputs to the financial model for the base case and the KSM Projects financial results for the alternate cases.

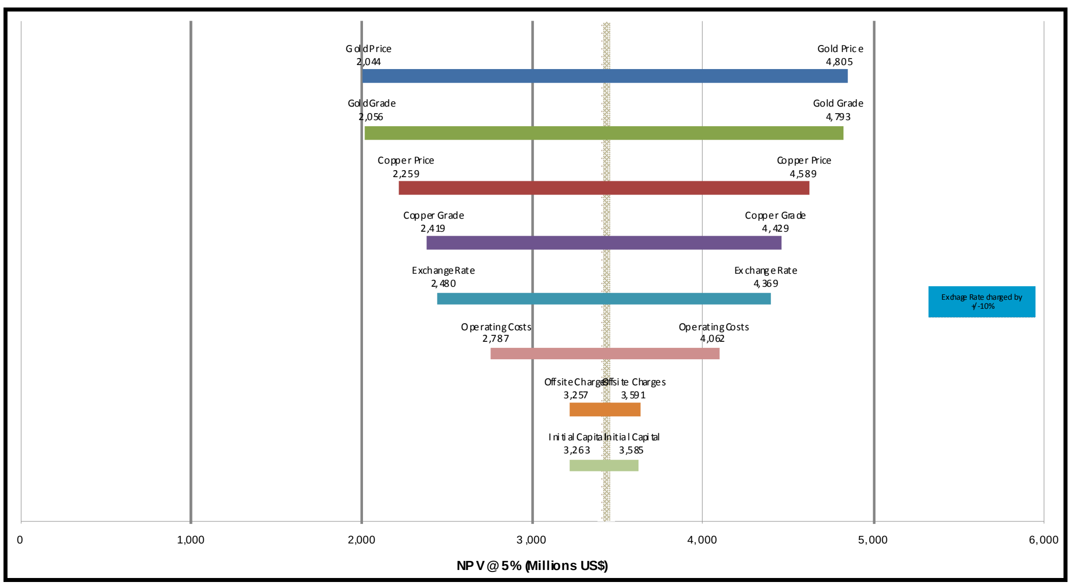
Table 1.9 Summary of the Economic Evaluations

		Base Case 3-year Average	Alternate Case	Spot Price July 27 2009
Gold	US$/oz	778	800	950
Copper	US$/lb	3.00	2.00	2.50
Silver	US$/oz	13.68	12.50	14.00
Molybdenum	US$/lb	26.05	15.00	15.00
Exchange Rate	US:Cdn	0.90	0.90	0.90
NPV (at 0%)	US$B	11.570	6.326	11.707
NPV (at 5%)	US$B	3.424	1.356	3.703
IRR	%	12.6	8.5	13.6
Cash Cost/oz Au	US$/oz	-51	243	114
Payback Period	years	6.6	8.8	5.8
Total Cost/oz	US$/oz	178	472	343

1.17.1 *SENSITIVITY ANALYSIS*

Figure 1.3 displays the sensitivity to NPV, analyzed by variations of metal grades and prices, capital and operating costs, and the exchange rate.

Figure 1.3 Base Case Sensitivity to NPV at 5% Discount Rate



1.18 PROJECT DEVELOPMENT PLAN

The project will take approximately five years to complete after receipt of operating permits. Section 25.0 gives a high-level project schedule.

1.19 OPPORTUNITIES AND RECOMMENDATIONS

The following sections outline areas to investigate for project improvements.

1.19.1 GEOLOGY/RESOURCE RECOMMENDATIONS

- re-survey drill hole collar locations for holes that show an apparent difference in elevation relative to the new topographic base map
- complete drilling programs to upgrade the currently identified inferred resources to indicated resources
- construct an updated geological model for the Kerr deposit
- construct a waste rock classification model for each pit area in order to classify waste material.

1.19.2 MINING RECOMMENDATIONS

- evaluate extended mine life with higher strip ratio ore as presented in the NPV(0) mine case
- additional drilling/resource modeling for improved resource and geotechnical confidence, reducing waste stripping in the early mine schedules
- detailed hydro-geology evaluation of the area to improve the accuracy of pit dewatering design and to assess the diversion and water management for the mining area
- ongoing evaluation of an overall waste rock management plan to reduce haul distances from Kerr and South Mitchell pits
- alternative mining methods and technologies studies to improve efficiencies and reduce fuel consumption
- further climate studies and operability studies to mitigate disruptions and improve safety during extreme mountain weather conditions
- risk assessment and mitigating study for implementing tasks
- a detailed geotechnical study of the potential pit slope angles to refine the project economics.

1.19.3 PROCESS RECOMMENDATIONS

- further evaluation of the use of high pressure grinding rolls (HPGR) to reduce operating costs for energy and grinding media
- further metallurgical test work and mineralization evaluations for each of the pit areas.

1.19.4 OTHER RECOMMENDATIONS

- a geohazard assessment including snow and avalanche loss control programs as the project infrastructure locations become more defined
- optimization of waste dump locations together with appropriate water management during placement and after mine reclamation has been completed
- crushing and conveying of mill feed and waste from Kerr, rather than using mine haul trucks to transport the mill feed and waste long distances over adverse topography; storage of high-PAG Kerr waste adjacent to Mitchell pit for subsequent flooded disposal within the pit upon Mitchell pit closure
- options involving pumping of concentrate to Stewart, rather than concentrate trucking, indicate marginal economical benefit; however, further evaluation work may be warranted in the project's prefeasibility stage
- evaluation of other alternative sites for PAG dumps that allow geological confinement and collection of leachate from the surface of low permeability rock areas.

2.0 INTRODUCTION

This NI 43-101 compliant report has been prepared by Wardrop for Seabridge based on work by the following independent consultants:

- RMI
- MMTS
- Brazier
- KCBL
- BVL
- McElhanney
- BGC
- Rescan.

A summary of the QPs responsible for each section of this report is given in Table 2.1. Certificates are included in Appendix A.

Table 2.1 Summary of Qualified Persons

Section	Description	Qualified Person	
		Company	**Qualified Person**
1.0	Summary	Wardrop	Frank Grills
2.0	Introduction	Wardrop	Frank Grills
3.0	Reliance on Other Experts	Wardrop	Frank Grills
4.0	Property Description & Location	RMI	Michael J. Lechner
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	RMI	Michael J. Lechner
6.0	History	RMI	Michael J. Lechner
7.0	Geological Setting	RMI	Michael J. Lechner
8.0	Deposit Types	RMI	Michael J. Lechner
9.0	Mineralization	RMI	Michael J. Lechner
10.0	Exploration	RMI	Michael J. Lechner
11.0	Drilling	RMI	Michael J. Lechner
12.0	Sampling Methods & Approach	RMI	Michael J. Lechner
13.0	Sample Preparation, Analyses & Security	RMI	Michael J. Lechner
14.0	Data Verification	RMI	Michael J. Lechner
15.0	Adjacent Properties	RMI	Michael J. Lechner

table continues…

Section	Description	Qualified Person	
		Company	Qualified Person
16.0	Mineral Processing & Metallurgical Testing	Wardrop	John Huang
17.0	Mineral Resource & Reserve Estimates	RMI	Michael J. Lechner
18.0	Mining	MMTS	Jim Gray
18.8.3	Mine Power	Brazier	Neil Brazier
18.12	Geotechnical	BGC	Warren Newcomen
19	Waste Management	KCBL	Graham Parkinson
20	Hydrological Survey	Rescan	Greg McKillop
21	Infrastructure and Site Layout	Wardrop	Frank Grills
21.14	Power Supply & Distribution	Brazier	Neil Brazier
22	Access Road	McElhanney	Robert Parolin
23	Logistics	Wardrop	Frank Grills
24	Environmental	Rescan	Greg McKillop
25	Project Execution Plan	Wardrop	Frank Grills
26	Capital & Operating Cost Estimates	Wardrop/KCBL/ BVL/McElhanney/ MMTS/Brazier	Frank Grills, John Huang, Harvey Graham Parkinson Bosche, Bob Parolin, Jim Gray, Neil Brazier
27	Economic Analysis	Wardrop	Frank Grills
28	Interpretation and Conclusions	Wardrop	Frank Grills
29	Opportunities & Recommendations	All	Sign off by section
30	References	Wardrop	Sign off by section

2.1 UNITS OF MEASURE

Units of measure and various conversion factors used in this report are as follows. A complete glossary is available at the beginning of this report, following the Table of Contents.

LINEAR MEASURE

1 inch (")	=	2.54 centimetres (cm)
1 foot (ft)	=	0.3048 m
1 yard	=	0.9144 m
1 mile	=	1.6 km

AREA MEASURE

1 acre (ac)	=	0.4047 ha		
1 square mile	=	640 ac	=	259 ha

WEIGHT

1 short ton (ton)	=	2,000 lb	=	0.907 tonne (t)
1 lb	=	0.454 kilograms (kg)	=	14.5833 troy oz

ASSAY VALUES

1 oz per ton	=	34.2857 g/t
1 troy oz	=	31.1035 grams (g)
1 part per billion (ppb)	=	0.0000292 oz/ton

3.0 RELIANCE ON OTHER EXPERTS

As outlined in Section 2.0, this report has been completed by independent consulting companies. All sections of the report have been provided by experts who are QPs.

Wardrop has followed standard professional procedures in preparing the contents of the KSM PEA Addendum 2009. Data used in this report have been verified where possible and Wardrop has no reason to believe that the data were not collected in a professional manner.

Certificates of QPs are included in Appendix A.

4.0 PROPERTY DESCRIPTION AND LOCATION

This section has been taken from the RMI report entitled "Updated KSM Mineral Resources" dated March 30, 2009, which is available on SEDAR.

The Mitchell property, along with the Kerr and Sulphurets deposits, is located within a package of 30 contiguous mineral claims and 19 contiguous placer claims that are summarized in Table 4.1 (Brassard, 2009). The mineral claims cover an area of approximately 6,726 ha while the placer claims cover about 4,554 ha. It should be noted that most of the placer claims lie "over the top" of the mineral claims. Seabridge has also acquired 45 contiguous mineral claims (Seabee Property) that are located about 19 km northeast of the KSM property. The Seabee claims are not summarized in Table 4.1 or shown on Figure 4.1.

The KSM property is located in northwest BC, at an approximate latitude of 56.5°N and a longitude of 130.3°W. The mineral resources that are subject to this report are located relative to the NAD83 UTM coordinate system. The property is situated approximately 950 km northwest of Vancouver, 65 km north-northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine. Figure 4.1 is a general location map.

The KSM mineral claims were converted from 58 legacy claims to BC's new Mineral Titles Online (MTO) system in 2005. Eleven legacy placer claims were converted in 2005 to nine cell placer claims. Ten cell placer claims have been added to the property and are contiguous with the converted legacy placer claims. In the MTO system, claims are located digitally using a fixed grid on lines of latitude and longitude with cells measuring 15 seconds north-south and 22.5 seconds east-west (approximately 460 m by 380 m at KSM). The legacy claims were located by previous owners by placing tagged posts along the boundaries; however, the survey method employed in locating the legacy claims is not known. With the MTO system, no markings are required on the ground and the potential for gaps and/or overlapping claims inherent in the old system is eliminated.

Figure 4.1 General Location Map

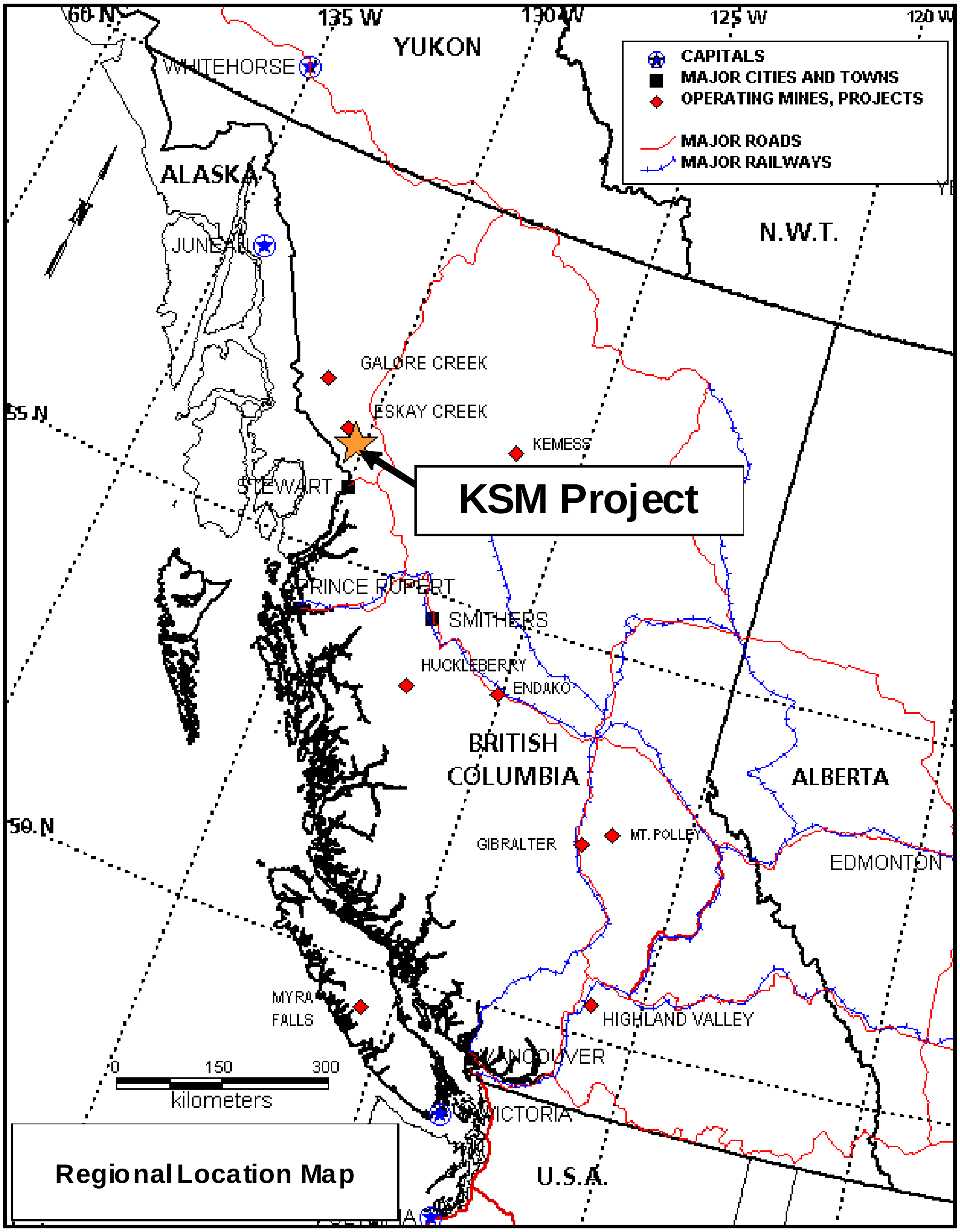


Table 4.1 Seabridge Mining Claims*

Tenure No.	Claim Name	Land Units	ha	Map No.	Expiry Date
516236	ICE 4	17	303.3	104B059	June 30, 2018
516237	ICE 2	4	71.4	104B059	June 30, 2018
516238	OK #1	35	624.5	104B059	December 10, 2018
516239	OK #2	30	535.5	104B059	December 10, 2018
516240	ICE 1	6	107.0	104B059	June 30, 2018
516241	IRON CAP 4	8	142.7	104B059	June 30, 2018
516242	IRON CAP 6	1	71.4	104B059	September 23, 2018
516245	XRAY 1	20	356.9	104B059	October 12, 2018
516248	TEDRAY NO. 1	8	142.7	104B059	August 26, 2018
516251	TEDRAY NO. 6	18	321.3	104B059	August 26, 2018
516252	ED NO. 1	7	125.0	104B059	August 26, 2018
516253	ED NO. 2	10	178.6	104B059	August 26, 2018
516254	TEDRAY NO. 9	16	285.8	104B059	August 26, 2018
516255	TEDRAY 15	12	214.3	104B049	September 23, 2018
516256	TEDRAY NO. 11	3	53.6	104B049	August 26, 2018
516258	TEDRAY 16	6	178.6	104B059	November 3, 2018
516259	TEDRAY 17	10	107.2	104B049	November 3, 2018
516260	TEDRAY 18	6	107.2	104B049	November 3, 2018
516261	KERR 41	26	464.6	104B049	December 20, 2018
516262	KERR 10	19	339.5	104B049	December 17, 2018
516263	KERR 15	36	643.9	104B049	December 17, 2018
516264	KERR 99	22	393.3	104B049	October 30, 2018
516266	KERR 8	10	178.8	104B049	December 17, 2018
516267	KERR 9	1	250.2	104B049	December 17, 2018
516268	KERR 12	18	321.8	104B049	December 17, 2018
516269	TEDRAY 13	6	107.2	104B049	August 26, 2018
254756	ARBEE #35	n/a	25.0	104B059	June 16, 2018
254757	ARBEE #39	n/a	25.0	104B059	June 16, 2018
254758	ARBEE #54	n/a	25.0	104B059	June 16, 2018
254759	ARBEE #55	n/a	25.0	104B059	June 16, 2018
516323	PLACER CLAIM	6	107.2	104B049	September 30, 2009
516325	PLACER CLAIM	7	125.0	104B049	October 1, 2009
516328	PLACER CLAIM	4	71.5	104B049	October 2, 2009
516330	PLACER CLAIM	6	107.2	104B049	October 3, 2009
516332	PLACER CLAIM	6	107.2	104B049	October 4, 2009
516333	PLACER CLAIM	5	89.3	104B049	October 5, 2009
516375	PLACER CLAIM	7	125.0	104B049	October 6, 2009
516676	PLACER CLAIM	1	17.9	104B049	October 7, 2009
516677	PLACER CLAIM	1	17.9	104B049	June 11, 2010
576658	KERR PL1	25	446.9	104B049	February 20, 2010

table continues…

Tenure No.	Claim Name	Land Units	ha	Map No.	Expiry Date
576659	KERR PL2	25	446.6	104B049	February 20, 2010
576660	KERR PL3	25	446.4	104B059	February 20, 2010
576661	KERR PL4	25	446.2	104B059	February 20, 2010
576662	KERR PL5	25	446.0	104B059	February 20, 2010
576663	KERR PL6	25	446.0	104B059	February 20, 2010
576664	KERR PL7	8	142.7	104B059	February 20, 2010
576665	KERR PL8	18	321.4	104B059	February 20, 2010
576666	KERR PL9	16	285.7	104B059	February 20, 2010
576667	KERR PL10	20	357.4	104B049	February 20, 2010

* note: all claims are part of the Skeena Mining Division.

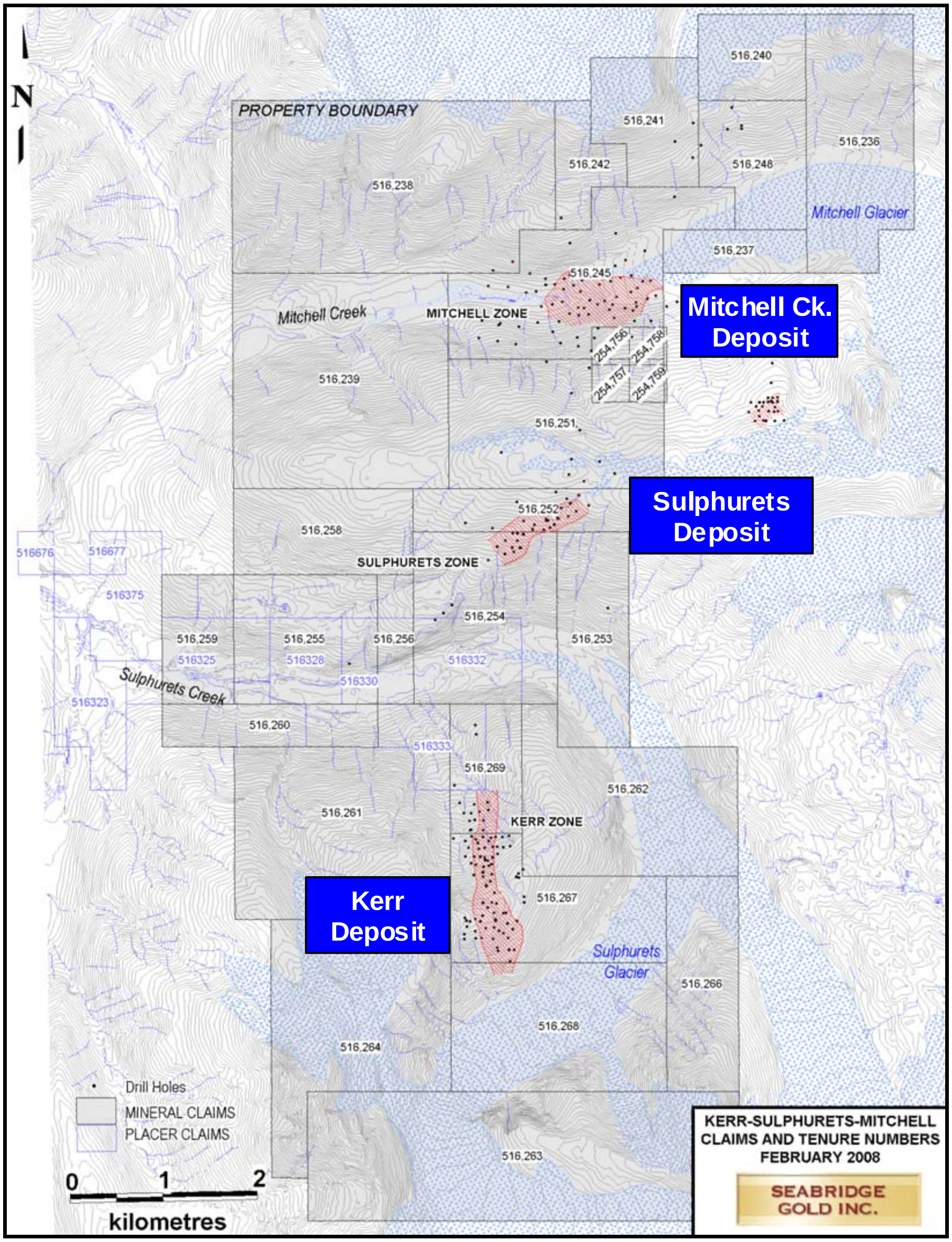
Figure 4.2 is a claim map showing Seabridge's KSM claim block. The claim map shows topographic contours, three mineral deposits (Kerr, Sulphurets, and Mitchell Creek), along with Seabridge's mineral tenure's and drill hole locations.

There is no record or evidence of any historical mining on the property. The BC Mineral Inventory (Minfile) contains 25 mineral occurrences in this area (mostly copper and gold). Also, within the claim group, two non-compliant (pre-NI 43-101) mineral resources were reported by Placer Dome Inc. (Placer Dome) for the Kerr and Sulphurets deposits.

The project consists of two contiguous claim blocks known as the Kerr and Sulphurets (or Sulphside) properties. The claims are 100% owned by Seabridge. Placer Dome (now Barrick Gold Corp. [Barrick]) retains a 1% net smelter royalty (NSR) that is capped at Cdn$4.5 M. Two of the pre-converted claims (Xray 2 and 6) are subject to a contractual royalty obligation in accordance with terms in the underlying Dawson Agreement. The lands covered by these claims are now contained within the converted Xray 1 claim (Tenure No. 516245). There is an additional underlying agreement whereby advance annual royalties payable to Dawson are being paid by Placer Dome.

Annual holding costs for all of the claims (mineral and placer) are approximately Cdn$172,988. In 2007, assessment work was filed to advance the year of expiry to 2018. The Kerr-Sulphurets placer claims have been kept in good standing by paying fees in lieu of completing assessment work. Assessment work was completed on the Seabee claims in 2008 with that work filed in February 2009 which puts those claims in good standing until 2012.

Figure 4.2 Claim Map



The KSM property falls within the Cassiar-Iskut-Stikine Land and Resource Management Plan (LRMP). There are no Protected or Special Management Areas overlapping the KSM property. A Conservation-oriented Protection Area and large River Corridor Special Management Area are currently being considered along the lower two-thirds of the Unuk River, which may impact the approval process of potential development plans and valley access to the project. The government has recognized the significance of historical mining activity in this area, which includes the active Eskay Creek gold and silver mine and the past producing Snip, Granduc, and Premier mines. Based on various anecdotal reports, the provincial government is committed to supporting future mining development in the region.

The KSM Project falls within the traditional lands of several aboriginal groups.

In 2003, an environmental evaluation of the KSM property was undertaken by Stantec Inc. (Stantec) for Falconbridge Ltd. (Falconbridge). A reclamation program addressing surface disturbances resulting from historical exploration work identified by Stantec was undertaken in 2004 by Falconbridge. This reclamation work was deemed to be satisfactory by the Ministry of Energy, Mines, and Petroleum Resources (MEMPR). The Stantec study noted there are extensive areas of naturally occurring sulfide minerals (mostly pyrite) that have been exposed by erosion and glaciation. Natural oxidation of sulfide minerals results in acidic drainage with elevated metal content. This has been occurring over a geological (quaternary) time scale.

The 2006 drilling program proposal (Notice of Work and Reclamation Program) was submitted to the MEMPR on May 25, 2006. Approval was received promptly by Mines Act Permit No. MX-1-571 granted June 5, 2006 by the MEMPR, and Free Use Permit No. 18204 granted June 30, 2006 by the Ministry of Forests. Seabridge's exploration programs are conducted under the permits issued in 2006.

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

This section has been taken from the RMI report entitled "Mitchell Creek Technical Report, Northern British Columbia" dated April 6, 2007, which is available on SEDAR.

The property lies in the rugged coastal mountains of northwest BC, with elevations ranging from 520 m in Sulphurets Creek valley to over 2300 m at the highest peaks. Valley glaciers fill the upper portions of the larger valleys from just below the tree line and upwards. The glaciers have been retreating for at least the last several decades. Aerial photos from 1991 indicate the Mitchell Glacier has retreated almost a kilometre laterally and perhaps several hundred metres vertically since then.

The property is drained by Sulphurets and Mitchell Creek watersheds that empty into the Unuk River, which flows westward to the Pacific Ocean through Alaska. The tree line lies at about 1240 masl, below which a mature forest of mostly hemlock and balsam fir abruptly develops. Fish are not known to inhabit the Sulphurets and Mitchell watersheds. Large wildlife such as deer, moose, and caribou are rare due to the rugged topography and restricted access; however, bears and mountain goats are relatively common.

The climate is generally that of a temperate or northern coastal rainforest, with sub-arctic conditions at high elevations. Precipitation is high with annual rainfall and snowfall totals estimated to be somewhere between the historical averages for the Eskay Creek Mine and Stewart, BC. These range from 801 to 1,295 mm of rain and 572 to 1,098 cm of snow, respectively (data to 2005). The length of the snow-free season varies from about May through November at lower elevations and from July through September at higher elevations.

Access to the property is via helicopter. For the 2006 drilling program, an Astar 350B2 was chartered from Mustang Helicopters Inc. out of Red Deer, Alberta. The following two staging areas were used for mobilizing crews and equipment:

- an area located at kilometre 54 on the private Eskay Creek Mine Road, which is about 25 km to the north-northwest of the property
- along the public Granduc Road, located about 35 km to the south-southeast of the property, which in turn is about 40 km north of the town of Stewart, BC. A section of this road passes through Alaska and the town of Hyder.

Stewart, a town of approximately 500 inhabitants, is the closest population center to the property. It is connected to the provincial highway system via paved, all-weather highway (#37A). The larger population centers of Prince Rupert, Terrace, and Smithers (with a total population of about 32,000) are located approximately 270 km to the southeast.

Deep water loading facilities for shipping bulk mineral concentrates exist in Stewart, and are currently used by both the Eskay Creek and Huckleberry mines. The nearest railway is the CPR Yellowhead route, which is located approximately 220 km to the southeast. This line runs east-west and terminates at the deep water port of Prince Rupert on the west coast of BC.

The property lies on crown land, thus all surface and access rights are granted by the Mineral Tenure Act, the Mining Right of Way Act, and the Mining Rights Amendment Act. There are no settlements or privately owned land in this area and no commercial or recreational activity is known to occur here. The closest power transmission lines run along the highway 37A corridor to Stewart, approximately 50 km to the southeast. The Eskay Creek Mine produces its own diesel generated power. There are proposals to develop local hydroelectric power sources and extend the highway 37A transmission line northward.

AMEC Americas Ltd. (AMEC) of Vancouver, BC, was commissioned by Noranda Inc. in 2004 to complete a scoping study to identify possible technical limitations for a conceptual large open-pit mining operation in the Kerr-Sulphurets area. The study recognized that within the claims, locating large plants, tailings, and waste rock storage sites may be technically challenging; however, ample space and favourable conditions exist in wide valleys approximately 20 km to the east.

6.0 HISTORY

This section has been taken from the RMI report entitled "Mitchell Creek Technical Report, Northern British Columbia" dated April 6, 2007, which is available on SEDAR.

6.1 EXPLORATION HISTORY

The modern exploration history of the area began in the 1960s, with brief programs conducted by Newmont Mining Corp. (Newmont), Granduc Mines Ltd. (Granduc), Phelps Dodge Mining Company (Phelps Dodge), and the Meridian Syndicate. All of these programs were focused towards gold exploration. Various explorers were attracted to this area due to the numerous large, prominent pyritic gossans that are exposed in alpine areas. There is evidence that prospectors were active in the area prior to 1935. The Sulphurets Zone was first drilled by Esso Minerals Ltd. (Esso) in 1969; Kerr was first drilled by Brinco Mining Ltd. (Brinco) in 1985 and Mitchell Creek by Newhawk Gold Mines Ltd. (Newhawk) in 1991.

In 1989, a 100% interest in the Kerr deposit was acquired by Placer Dome from Western Canadian Mining Corp. (Western Canadian) and in the following year they acquired the adjacent Sulphurets property from Newhawk. The Sulphurets property also hosts the Mitchell Creek deposit and other mineral occurrences. In 2000, Seabridge acquired a 100% interest from Placer Dome in both the Kerr and Sulphurets properties, subject to capped royalties.

There is no recorded mineral production, nor evidence of it, from the property. Immediately west of the property, small-scale placer gold mining has occurred in Sulphurets and Mitchell Creeks. On the Bruceside property (immediately to the east and currently owned by Silver Standard Resources Inc.), limited underground development and test mining was undertaken in the 1990s on narrow, gold-silver bearing quartz veins at the West Zone.

Table 6.1 summarizes the more recent exploration history of the Kerr property while Table 6.2 summarizes more recent exploration history of the Sulphurets property.

Table 6.1 Exploration Summary of the Kerr Property

Year	Activity
1982-1883	"Alpha JV" began prospecting and soil geochem surveys of the Kerr gossan focusing on gold.
1984-1985	Brinco optioned the Kerr project, completed some geologic surveys, and drilled 3 holes.
1987-1989	Western Canadian optioned Kerr and completed 59 drill holes and recognized Cu-Au porphyry.
1989	Placer Dome acquires Kerr property.
1990-1992	Placer Dome began delineation drilling of Kerr deposit at 50 m centres by drilling 82 holes.
1992-1996	Placer Dome estimated resources (non-NI 43-101), metallurgical testwork, and scoping studies.
1996-2000	Project was dormant.
2000	Seabridge acquired a 100% interest in Kerr from Placer Dome.
2002	Noranda Inc. acquired an option from Seabridge with the right to earn up to a 65% interest in Kerr.
2003-2004	Noranda Inc. undertook various exploration surveys.
2006	Seabridge purchases Falconbridge (formerly Noranda) option.

Table 6.2 Exploration Summary of the Sulphurets/Mitchell Property

Year	Activity
1880-1933	Limited placer gold exploration and mining.
1935-1959	Placer gold prospecting, prospecting, and staking of mining claims.
1959-1960	Newmont and Granduc conducted surveys including airborne mag. Sulphurets and Iron Cap Au zones discovered. D. Ross, S. Bishop, and W. Dawson prospected and stake claims in area.
1961-1968	Granduc Mines conducted geologic/geochem surveys, drilled 9 holes into Sulphurets Zone. Ross-Bishop-Dawson claims optioned by Phelps Dodge in 1962, Meridian Syndicate in 1965, and Granduc in 1968.
1963	R. Kirkham completed a M.Sc. thesis on the geology of Mitchell and Sulphurets areas.
1981	T. Simpson completed a M.Sc. thesis on the geology of the Sulphurets gold zone.
1971-1977	Granduc conducted additional exploration surveys targeting molybdenum and drilled 6 holes into the Snowfield Zone (Bruceside).
1979-1984	Esso optioned the Sulphurets property and completed early stage exploration including drilling 14 holes (2,275 m).
1985-1991	Granduc optioned Sulphurets to Lacana (later Corona) and Newhawk. Lacana-Newhawk JV spent ~$21 M developing the West Zone and other smaller precious metal veins on the Bruceside property. Drilled 11 holes at Sulphurets. Homestake undertook exploration after acquiring Corona.
1991	Arbee prospect optioned by Newhawk from D. Ross.
1992	Arbee prospect optioned by Placer Dome from Newhawk.

table continues…

Year	Activity
1991-1992	Newhawk commissioned AB geophysical survey over Sulphurets. Newhawk subdivided the Sulphurets property into Sulphside and Bruceside. Placer Dome acquires Sulphside (Sulphurets, Mitchell, Iron Cap, and other prospects).
1992	Placer Dome undertook delineation drilling of Sulphurets deposit at 50 m centres (23 holes).
1993	J. Margolis completed a Ph.D. thesis on the Sulphurets district. Newhawk-Corona drilled 3 holes in the Snowfields and Josephine zones east of Sulphurets.
1992-1996	Placer Dome completed geologic modeling, resource estimation (not NI 43-101 compliant), preliminary metallurgical testwork, and scoping studies.
1999	Silver Standard Resources Inc. acquired Newhawk.
1996-2000	Sulphurets project was dormant.
2000	Seabridge Gold acquired a 100% interest in Sulphurets from Placer Dome.
2002	Noranda Inc. acquired an option to earn up to 65% from Seabridge.
2003-2004	Noranda Inc. undertook various exploration surveys.
2005	Falconbridge (formerly Noranda) completed 4,092 m of diamond drilling in 16 holes.
2006	Seabridge purchased Falconbridge's option and conducted drilling programs on Mitchell and Sulphurets deposits.
2007	Seabridge purchased Arbee prospect from D. Ross.

Technical reports of exploration work (Assessment Reports) applied by previous operators to maintain property standing are kept by the BC MEMPR and are available to the public. Table 6.3 lists various assessment reports that have been filed for the benefit of the KSM Project.

6.2 HISTORICAL RESOURCE ESTIMATES

There are no reported historical resource estimates for the KSM deposits.

6.3 HISTORY OF PRODUCTION

There is no known production from the Kerr, Sulphurets, or Mitchell deposits.

Table 6.3 Kerr-Sulphurets Assessment Reports

AR#	Property Name	Title of Report	Claim(s)	Operator(s)	Owner(s)	Author(s)	Year	Pages	General Work Categories	Work Details
348	Tedray	Report of Geological and Geophysical Surveys	RED, TAY, TED	Granduc	Granduc	Norman, G.W.H.	1960	10	Geological, Geophysical	ground mag, geological mapping
499	Arbee	Mitchell Creek Geological Report	ARBEE, CAN, DAWSON ROSS, JOHN BULL	Ross, D.F.	Ross, D.F.	Malcolm, D.	1962	17	Geological	geological mapping
569	Lyn	Report of Geophysical Survey Lyn 1, Ray 4 Fraction and Ray 19 Claim Groups	LYN, RAY, RAY Y FR., TED	Granduc	Granduc	Norman, G.W.H.	1964	7	Geophysical	35 km AB Mag
836	C.A.	Report on a Geochemical Survey of the CA Claim Group	CA	Silver Ridge Min.	Silver Ridge Min.	Manning, L.	1966	8	Geochemical	77 soils
1006	Arbee Prospect	Report of Geological and Geophysical Surveys	ARBEE, D-R, JB	Kennco Explorations (Canada) Limited	Ross, D.F.	Ney, C.	1966	10	Geological, Geochemical	80 soils, geological mapping
3170	Arbee	Report of Geological Mapping on Parts of the Dawson-Ross Group	ARBEE, DAWSON-ROSS, JOHN BULL, RAN	Granduc	Ross, D.F.	Ostensoe, Erik A.	1971	8	Geological, Physical	geological mapping
5416	Mitch	Geological report on the Mitch claim group, Sulphurite Creek area	MITCH, PATTY, RAY, TED	Granduc	Granduc	Ostensoe, Erik A.; Kruchkowski, Edward R.	1974	54	Geological, Physical, Geochemical	595 rocks, 20 assays, geological mapping
5921	Ted Ray or Big Showing, or Mitchell	Geological and Geochemical Report on the Tedray group	ED, GRACE, TEDRAY	Granduc	Granduc	Ostensoe, Erik A.; Kruchkowski, Edward R.	1975	38	Geological, Geochemical, Physical	500 rocks, 7 trenches totalling 30 m, geological mapping
5958	Tedray	Petrographic Report on the Tedray Claim Group, Mitchell Creek Area	ED, TEDRAY	Granduc	Granduc	Montgomery, J.H.	1976	64	Geological	40 petrographic analyses
6066	Ironcap and Tedray	Geological and Geochemical Report on the Ironcap and Tedray Claim Groups, Sulphurets Creek Area	IRONCAP, TEDRAY	Granduc	Granduc	Ostensoe, Erik A.; Kruchkowski, Edward R.	1976	23	Geological, Geochemical, Physical	70 assays, 7 trenches totalling 30 m, geological mapping
8420	Sulphurets	Diamond Drilling Report; Central 1, ED 1, JCE 2... Claims, NW of Stewart	CENTRAL, ED, ICE, TEDRAY	Esso	Esso	Bridge, Dane A.; Brown, M.G.	1980	101	Drilling, Geochemical	1,073 m DDH in 5 holes, 352 assays
9568	Sulphurets	Diamond Drilling Report of the Tedray 9, 11 Claim, NW of Stewart	TEDRAY	Esso	Esso	Bridge, Dane A.	1981	185	Drilling, Geochemical	3,658 m DDH in 22 holes, 212 assays
12471	Kerr	Geochemical Report on a Silt and Soil Sampling Survey	KERR	Wallster, Dale E.	Wallster, Dale E.	Wallster, Dale E.	1983	14	Geochemical	151 silts and soils
13369	Kerr	Kerr Claims	KERR 12, KERR 15, KERR 41, KERR 7-10	Brinco	Brinco	Graf, Chris	1984	31	Geochemical, Geological	210 soils, geological mapping
14614	Kerr	Geochemical, Geological, Trenching, and Diamond Drilling Report on the Kerr Claims	KERR 99	Brinco	Brinco	Epp, William Robert	1986	14	Drilling, Geochemical, Geological, Physical	189.9 m DDH in 3 holes, 1,448 soils, silts, and talus, 59 rocks, 355 trench channels, 102 core assays, 15 petrographic analyses, 342 m trenching
15493	Kerr	Assessment Report 1986 Geological, Geochemical, Geophysical Surveys	KERR 9, KERR 12, KERR 15, KERR 99	Western Canadian	Western Canadian	Meyers, R.	1986	24	Geochemical, Geological, Geophysical	649 rocks, 593 soils, geological mapping, 6.7 km VLFEM, 5.3 km mag
15688	OK	Assessment Report 1986 Field Season OK Group	OK 2	Newhawk	Newhawk	Tribe, Norman L.	1987	9	Geological	geological mapping

table continues…

AR#	Property Name	Title of Report	Claim(s)	Operator(s)	Owner(s)	Author(s)	Year	Pages	General Work Categories	Work Details
15724	Sulphurets-Brucejack	Progress Report on the Sulphurets Project	ED, ICE, IRON CAP, RED RIVER, SULPHURETS, TEDRAY, XRAY, OK	Lacana Ex.; Newhawk	Granduc	Tribe, Norman L.; Hicks, K.E.; Wells, Ronald C.; Kelly, C.	1986	308	Drilling, Geochemical, Physical	6,645 m DDH in 47 holes, 2,328 assays, 404 m underground development (Brucejack)
16616	Kerr	A Geological, Geochemical, Geophysical and Drilling Report on the Kerr Project	KERR 12, KERR 41, KERR 9, KERR 99-100	Western Canadian	Western Canadian	Kowalchuk, J.M.; Jerema, M.	1987	421	Drilling, Geochemical, Geological, Geophysical, Physical	1,604 m DDH in 14 holes, 548 rocks, 300 assays, 505 soils, 33 trenches totalling 420 m, geological mapping, 12 km VLFEM
18285	Tedray	Claim Geological, Geochemical and Geophysical Report on the Tedray Mineral	TEDRAY 13	Western Canadian	Granduc	Butterworth, Brian P.; Kozak, D.K.	1988	87	Drilling, Geochemical, Geological, Geophysical	115.2 m DDH in 2 holes, 79 rocks, 52 assays, 602 soils, geological mapping, 2.5 km IP, 8.5 km mag
18406	Kerr	Geological and Geochemical Report on the Kerr Property	KERR 7-10, KERR 12, KERR 41, KERR 99-104	Western Canadian	Western Canadian	Casselman, Scott G.	1989	147	Geochemical, Geological, Physical	655 assays, 104 soils, 5 trenches totalling 200 m, geological mapping
19246	Tedray	Induced Polarization Survey on the Tedray 13 Claim	TEDRAY 13	Sulphurets Gold Corporation	Granduc	Le Bel, J.L.	1989	15	Geophysical	8 km IP
19541	Kerr	Diamond Drilling and Environmental Base Studies on the Kerr Project	KERR 7-10, KERR 12, KERR 15, KERR 41	Sulphurets Gold Corporation	Sulphurets Gold Corporation	Payne, John G.; Butterworth, Brian P.; Hewton, R.S.; Casselman, Scott G.	1989	705	Drilling, Geochemical, Geological	4,364.7 m DDH in 20 holes, 7 heavy mineral concentrates, 27 water samples, 17 rocks, 1,641 assays, geological mapping
21552	Kerr	Diamond Drilling Report on the Kerr Property	KERR 100, KERR 104, TEDRAY 13, KERR 101	Sulphurets Gold Corporation	Newhawk; Placer Dome (Vancouver); Sulphurets Gold Corporation	Copland, Hugh; Slauenwhite, M.	1991	922	Drilling, Geochemical, Physical	9,748 m in 73 holes, 3,000 assays, 4 metallurgical samples, 10 km access roads
21821	Sulphurets	Diamond Drill Report on the Sulphurets Property	XRAY 4, XRAY 6	Newhawk	Newhawk	Visagie, David A.	1991	64	Drilling, Geochemical	647.3 m DDH in 4 holes, 255 assays
21828	Sulphurets	Diamond Drill Report on the Tedray 6 and 9 Claims	TEDRAY 6, TEDRAY 9, ED 1	Newhawk	Newhawk	Visagie, David A.	1991	118	Drilling, Geochemical	1,206.3 m in 7 holes, 483 assays
22741	Snowfield	Geological and Geochemical Report on the Dawson-Ross Claims	DAWSON-ROSS 1, DAWSON-ROSS 3	Newhawk	Newhawk	Visagie, David A.	1992	64	Geochemical, Geological	109 rocks, 224 soils, geological mapping
23172	Snowfield	Diamond Drill Report on the Snowfield Property	ICE 5	Newhawk	Newhawk	McPherson, M.D.	1993	39	Drilling, Geochemical, Geological	72.9 m DDH in 1 hole, 39 assays, geological mapping
24734	Arbee	Geological Lithogeochemical Report on the Arbee Claim Group	ARBEE 35, ARBEE 39, ARBEE 54-55	Placer Dome	Placer Dome	Lewis, Thomas D.	1996	21	Geological, Geochemical	38 rocks, geological mapping

7.0 GEOLOGICAL SETTING

The following section was taken directly from RMI's April 2008 NI-43101 report.

The region lies within “Stikinia”, a terrane of Triassic and Jurassic volcanic arcs that were accreted onto the Paleozoic basement of the North American continental margin in the Middle Jurassic. Stikinia is the largest of several fault bounded, allochthonous terranes within the Intermontane belt, which lies between the post-accretionary, Tertiary intrusives of the Coast belt and continental margin sedimentary prisms of the Foreland (Rocky Mountain) belt. In the Kerr-Sulphurets area, Stikinia is dominated by variably deformed, oceanic island arc complexes of the Triassic Stuhini and Jurassic Hazelton groups. Back-arc basins formed eastward of the property in the Late Jurassic and Cretaceous were filled with thick accumulations of fine black clastic sediments of the Bowser Group. Folding and thrusting due to compressional tectonics in the late Cretaceous generated the area’s current structural features. Remnants of Quaternary basaltic eruptions occur throughout the region.

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

At Kerr-Sulphurets, Triassic rocks include marine sediments and intermediate volcanics of the Stuhini Group. The lowermost Stuhini Group is dominated by turbiditic argillite and sandstone, which are overlain by volcanic pillowed flows and breccias. The upper portion consists of turbidites and graded sandstones similar to the base strata. The Stuhini Group is separated by an erosional unconformity from the overlying Jurassic sediments and volcanics of the Jack Formation and Hazelton Group. The Jack Formation is comprised of fossiliferous, limey sediments, mudstones and sandstones. The base is marked by a granodiorite and limestone cobble bearing conglomerate. Overlying the Jack Formation is the Hazelton Group, dominated by andesitic flows and breccias deposited in a volcanic chain with high paleotopographic relief. Distinct felsic welded tuff horizons of the Mount Dilworth Formation are an important stratigraphic marker in the Hazelton Group, as they are closely associated with the Eskay Creek deposit.

A variety of dikes, sills, and plugs of diorite, monzodiorite, syenite, and granite are found in the area. Radiometric dating indicates these are of Early Jurassic age and they are collectively referred to as the “Mitchell Intrusions”. Below the Sulphurets and Mitchell thrust faults, pre- and intra-mineral intrusives have historically been very difficult to differentiate due to intense hydrothermal alteration. Above the faults there are a number of sills and plugs of coarse-grained feldspar porphyritic monzonite to low-silica granite that intruded siliceous hornfelsed sediments and volcanics. Copper and gold mineralization is typically best developed at the margins of these intrusions. There appear to be both pre-, intra-, and post-mineral phases of mineralization.

The alteration zones and associated mineralization have been subjected to low greenschist facies metamorphism along with folding and faulting. In addition, a pre-Jurassic orogenic event imparted at least an equivalent metamorphic grade on the Triassic rocks. The Sulphurets and Mitchell thrust faults are interpreted to have placed Triassic Stuhini Group volcanics and sediments over Jurassic Hazelton Group rocks during a southeast vergent, compressive event during the Cretaceous. The displacement is not considered to be significant, as panels above and below the faults are altered and mineralized by the same intrusive event. Alteration and mineralization styles are distinct, with the upper plates containing a higher proportion of intrusive rock, potassic alteration, and a higher copper to gold ratio, consistent with deeper parts of a copper porphyry system. The lower plate or plates, which host the Kerr, Sulphurets, and Mitchell deposits, have much stronger and pervasive phyllic alteration, with a lower copper to gold ratio, and a geochemical signature that suggests positioning consistent with a shallow setting in a copper-gold porphyry hydrothermal system. The altered areas are often intensely foliated and deformed, leading and previous workers to suggest that regional compressive deformation was focused in these less competent rocks and pre-existing structures.

A large hydrothermal alteration system is associated with the earlier intrusions. It covers an area of over 35 km^2 and is dominated by pyrite rich phyllic-argillic assemblages, resulting in intense gossanous exposures in uncovered and eroded areas. This extensive alteration and mineralization system developed as a result of hydrothermal activity focused on hypabyssal, Late Jurassic “Mitchell” intermediate, porphyritic intrusions. The model is best described as a gold-enriched copper porphyry system controlled by a series of dikes, sills, and plugs rather than a single stock. Mineralization occurs typically associated with quartz veinlet stockworks and sheeted veinlet arrays mainly in altered host rocks adjacent to the intrusions. Less commonly, mineralized intrusive-hydrothermal breccias cut through previously veined and mineralized rocks. Drilling and surface rock chip sample confirms the alteration and mineralization is continuous over distances of hundreds of metres. Post mineralization deformation has modified original geometries and remobilized metals, contributing to the homogeneity and “smoothness” of copper and gold mineralization, especially at the Mitchell deposit.

Principal sulfides are pyrite and chalcopyrite, with minor molybdenite, and trace amounts of tennantite, bornite, sphalerite, and galena. All mineralization is hypogene except for a small remnant of preserved supergene mineralization at the south end of the Kerr deposit, which hosts some chalcocite enrichment at the Main Copper occurrence where a remnant of leached capping with minor native copper and oxide mineralization is preserved at the highest elevations, and intervals of partially leached and oxidized sulfides in weathered rocks near the surface. Leaching of mineralization in frost-shattered rocks at higher elevations, with precipitation at lower elevations where groundwaters surface near the valley floors is an ongoing process.

Figure 7.1 is a generalized geologic map of the KSM region that shows surface geology, structures, drill hole locations, and mineralized zones.

Figure 7.1 Generalized Geologic Map

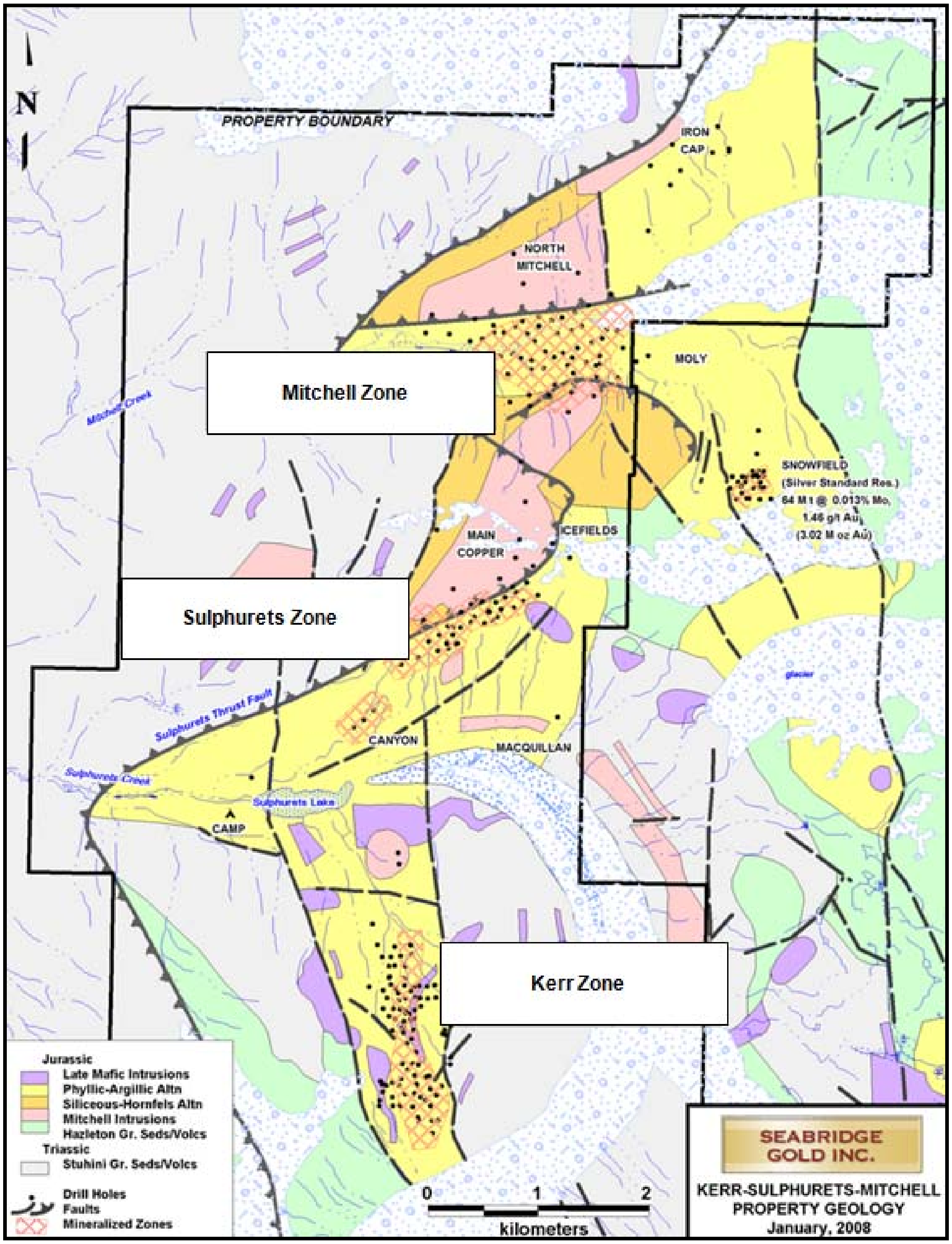


Figure 7.2 is a more detailed geologic map of the Mitchell deposit showing surficial geology, the location/trace of drill holes, and three cross section lines (Section 11, Section 15, and a NW-SE section). Figure 7.3 through Figure 7.5 are cross sections through the Mitchell deposit showing the trace of diamond drill holes, geology, and contoured gold grades. The lines of section for Figure 7.3 through Figure 7.5 are shown on Figure 7.2.

Information regarding the geologic setting of the Kerr and Sulphurets deposit were discussed in the RMI report entitled "Kerr-Sulphurets Technical Report" dated February 29, 2008.

Figure 7.2 Mitchell Zone Geologic Map

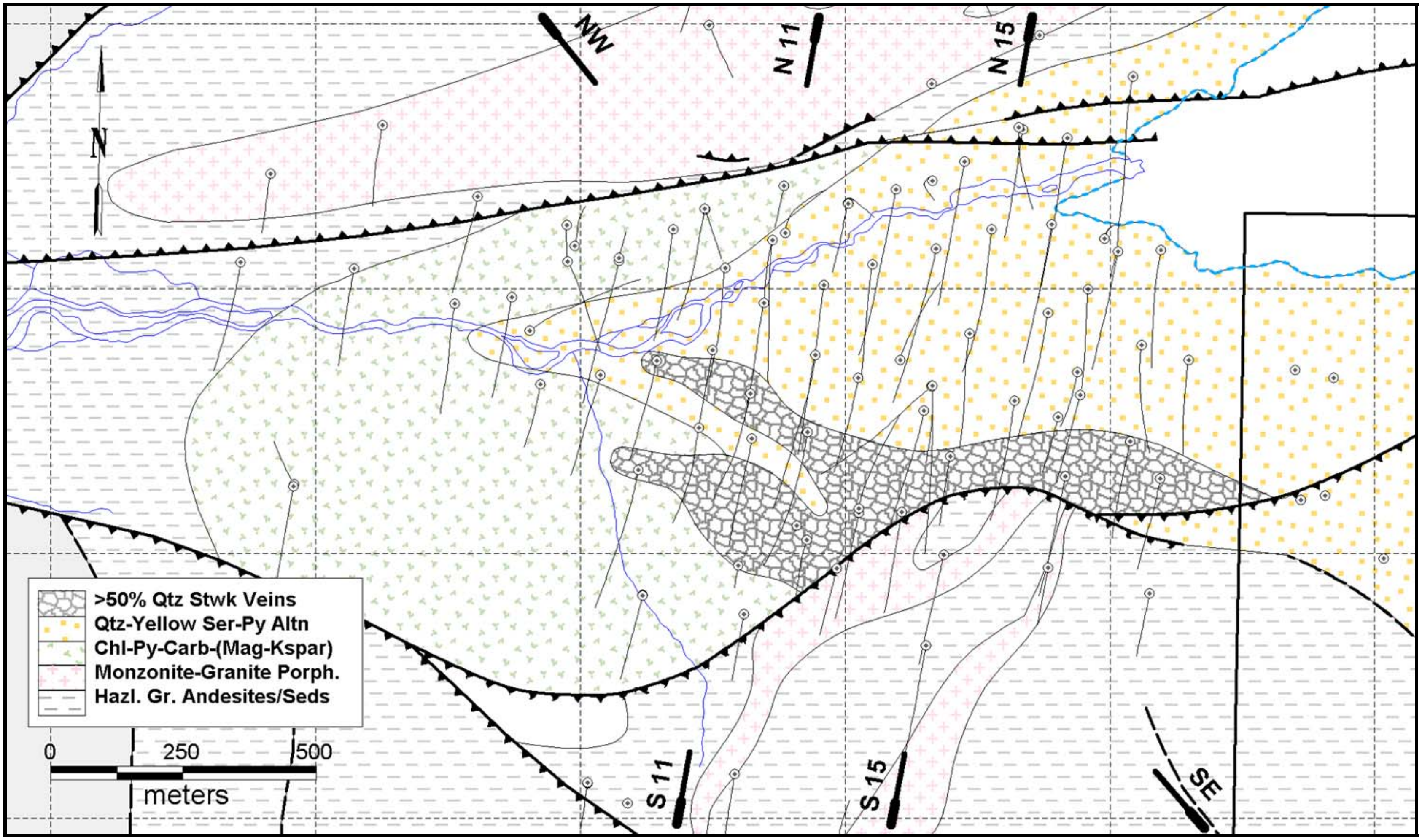


Figure 7.3 Mitchell Cross Section 11

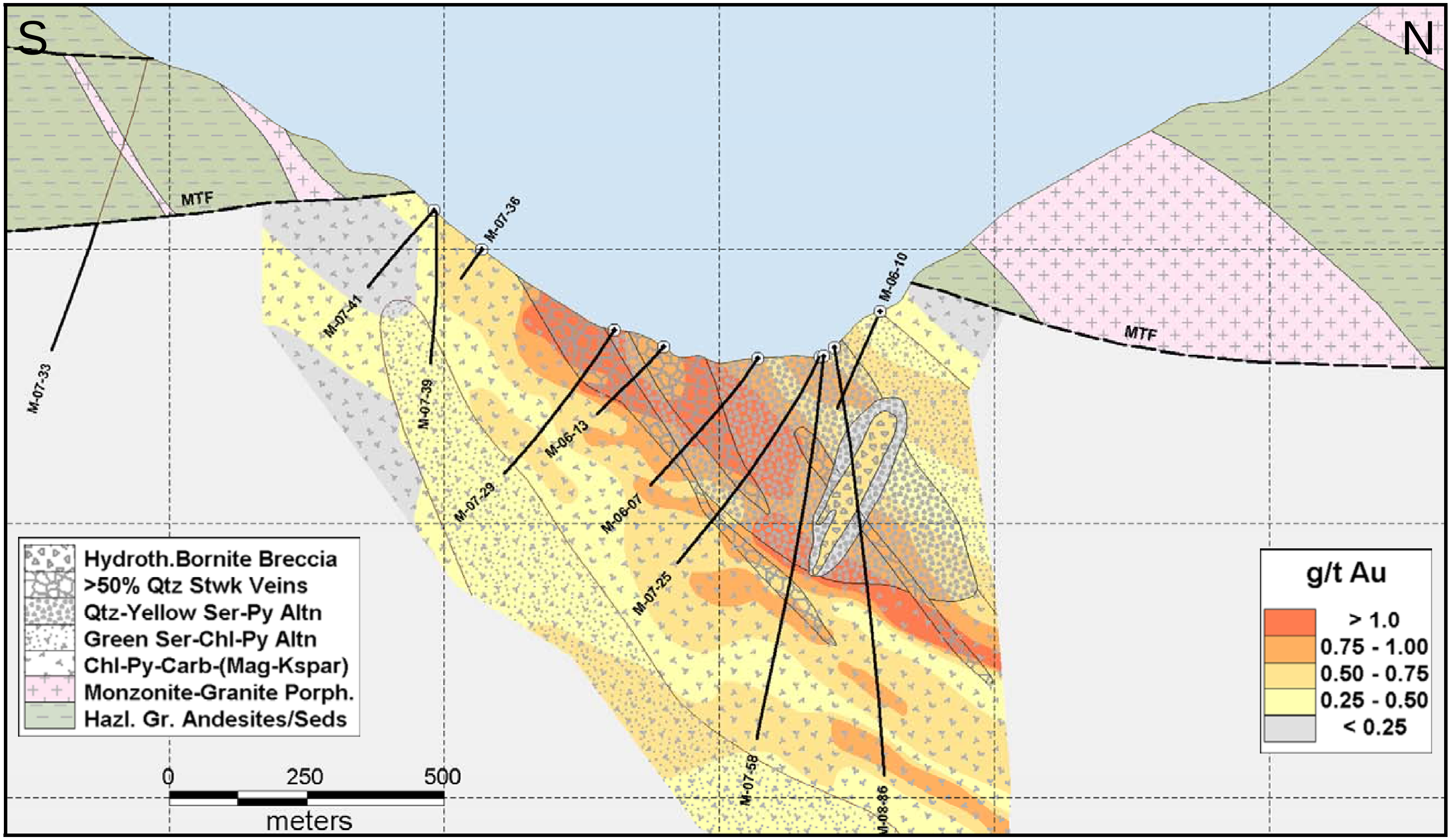


Figure 7.4 Mitchell Cross Section 15

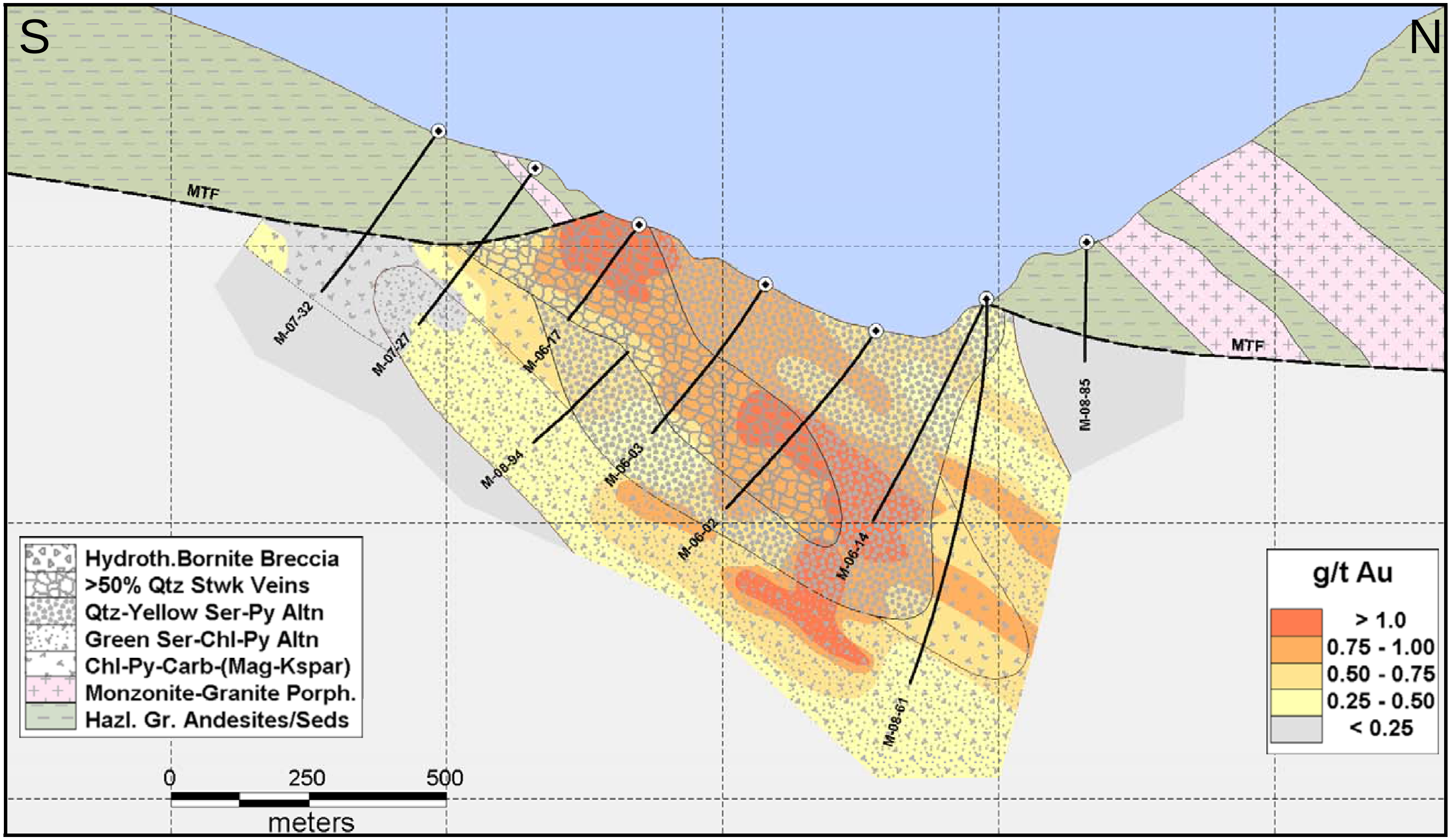
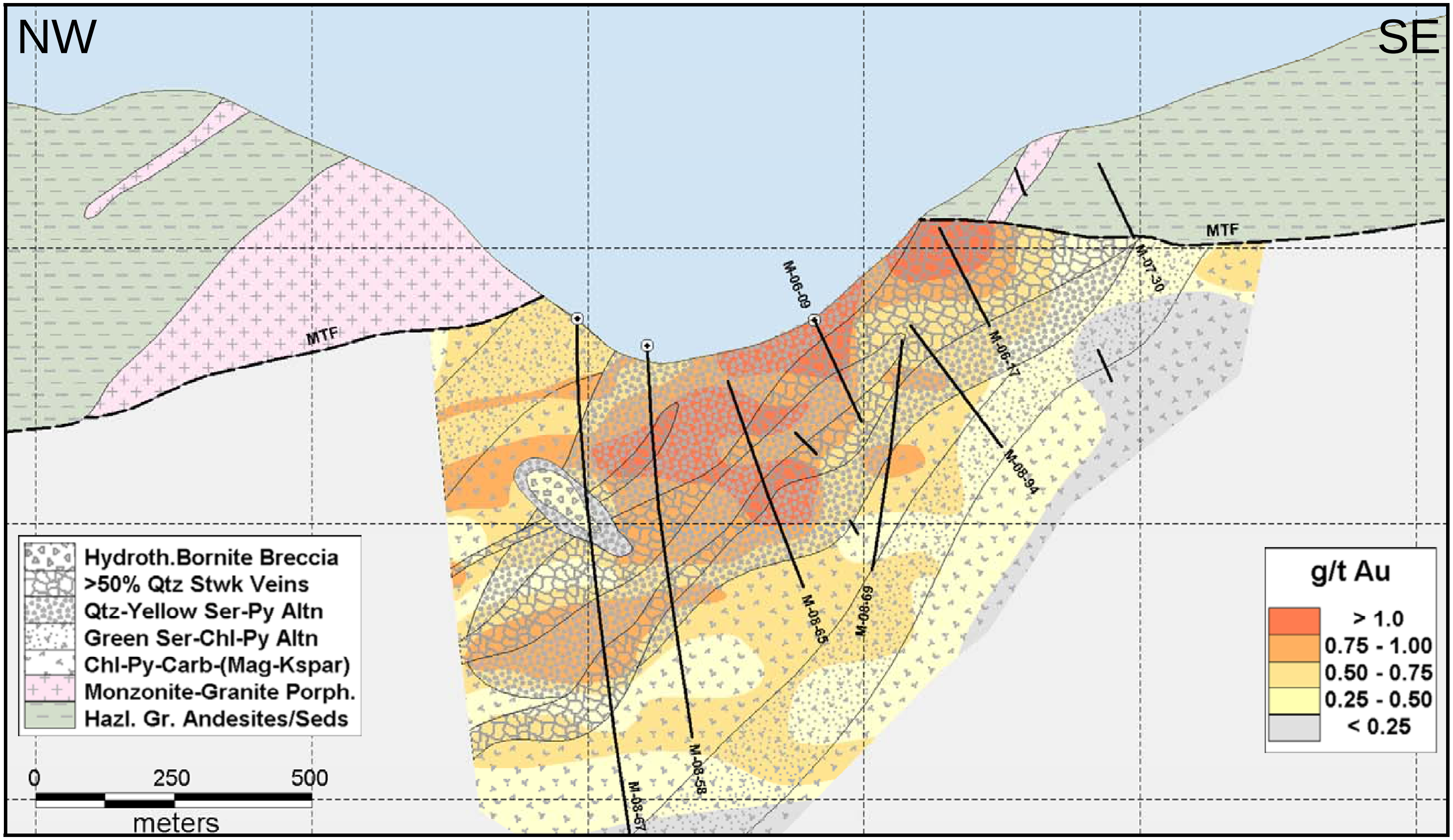


Figure 7.5 Mitchell NW-SE Cross Section



8.0 DEPOSIT TYPES

The following section has been taken from RMI's report entitled "Mitchell Creek Technical Report, Northern British Columbia", dated April 6, 2007, which is available on SEDAR.

The Kerr-Sulphurets property hosts an extensive alteration and mineralization system that was undoubtedly developed as a result of hydrothermal activity focused on hypabyssal, Early Jurassic "Mitchell" intermediate, porphyritic intrusions. The model is best described as a gold-enriched copper porphyry system controlled by a series of dikes, sills, and plugs rather than a single stock. Mineralization is typically associated with quartz veinlet stockworks and sheeted quartz veinlet arrays mainly in altered host rocks adjacent to the intrusions. Drilling and surface rock chip sampling confirms that the alteration and mineralization is continuous over distances of hundreds of meters. Less commonly, mineralized intrusive-hydrothermal breccias cut through previously veined and mineralized rocks. Principal sulfides are pyrite and chalcopyrite, with minor molybdenite, and trace amounts of tennantite, bornite, sphalerite, and galena. All mineralization is hypogene except for a small remnant of preserved supergene mineralization at the south end of the Kerr deposit, which hosts some chalcocite enrichment, and at the Main Copper (Sulphurets) occurrence where a remnant of leached capping and oxide mineralization is preserved at the highest elevations.

At Mitchell and Sulphurets, copper-gold mineralization is fine grained, pervasive, homogeneous, and continuous for several hundred metres along strike and depth extents. Preliminary work indicates gold is intimately associated with chalcopyrite. The unusually homogeneous nature of the mineralization over large extents may be the result of post-mineral metamorphism and re-distribution of metals during Early Jurassic or Cretaceous deformational events. At Sulphurets, mineralization is somewhat less continuous than Mitchell, where sharp contrasts in grade occur between structurally controlled hydrothermal breccias and alteration zones.

9.0 MINERALIZATION

9.1 KERR ZONE

This section has been taken from the RMI report entitled "Kerr-Sulphurets Technical Report" dated February 29, 2008, which is available on SEDAR.

The Kerr deposit has been delineated by over 26,000 m of core drilling in 144 drill holes spaced at intervals of 50 to 100 m by 6 previous operators between 1987 and 1991. Fine disseminated, fracture and veinlet controlled chalcopyrite mineralization, with minor bornite and tennanite, is associated with intrusion of Early Jurassic monzonite porphyry into Triassic sediments and volcaniclastics, and accompanying hydrothermal alteration. There is a strong phyllic overprint with a high pyrite content, generally 5 to 20%. In many respects, the deposit bears little resemblance to a classic porphyry deposit; however, it has been referred to as a porphyry-type deposit since 1987. Later studies (see Section 30.0 References) indicated that mineralization was localized around one or more previously unrecognized monzonite intrusions and is adequately described as a modified porphyry deposit. Most of the following description has been extracted and modified from the paper by Ditson, et al., 1995. Figure 9.1 is a generalized geological plan showing the surficial geology of the Kerr deposit along with drill holes and the approximated surface trace of 0.30% copper mineralization. Figure 9.2 and Figure 9.3 are two east-west oriented cross sections through the Kerr deposit.

The Kerr deposit is a strongly deformed copper-gold porphyry, where copper and gold grades have been upgraded due to remobilization of metals during later and/or possibly syn-intrusive deformation. Alteration is the result of a relatively shallow, long-lived hydrothermal system generated by intrusion of monzonite. Subsequent regional deformation along the Sulphurets thrust was diverted into Kerr area along pre-existing structures and altered rocks with low competency.

The mineralized area forms a mostly continuous, north-south trending and westerly dipping, irregular body at least 1,700 m long and up to 200 m thick. Higher grades are associated with crackled quartz stockwork, anhydrite veining, and chlorite alteration. It is enveloped by a schistose, pyrite rich phyllic alteration with low to moderate grades. Mineralization is open at depth and along strike.

The surface expression of the deposit is a large, strongly leached schistose, pyritic gossan. Soil geochemistry shows elevated anomalous gold values over the deposit, and a halo of anomalous copper values. Induced polarization detects high chargeability and low resistivity coincident with mineralization.

Figure 9.1 Generalized Geologic Map

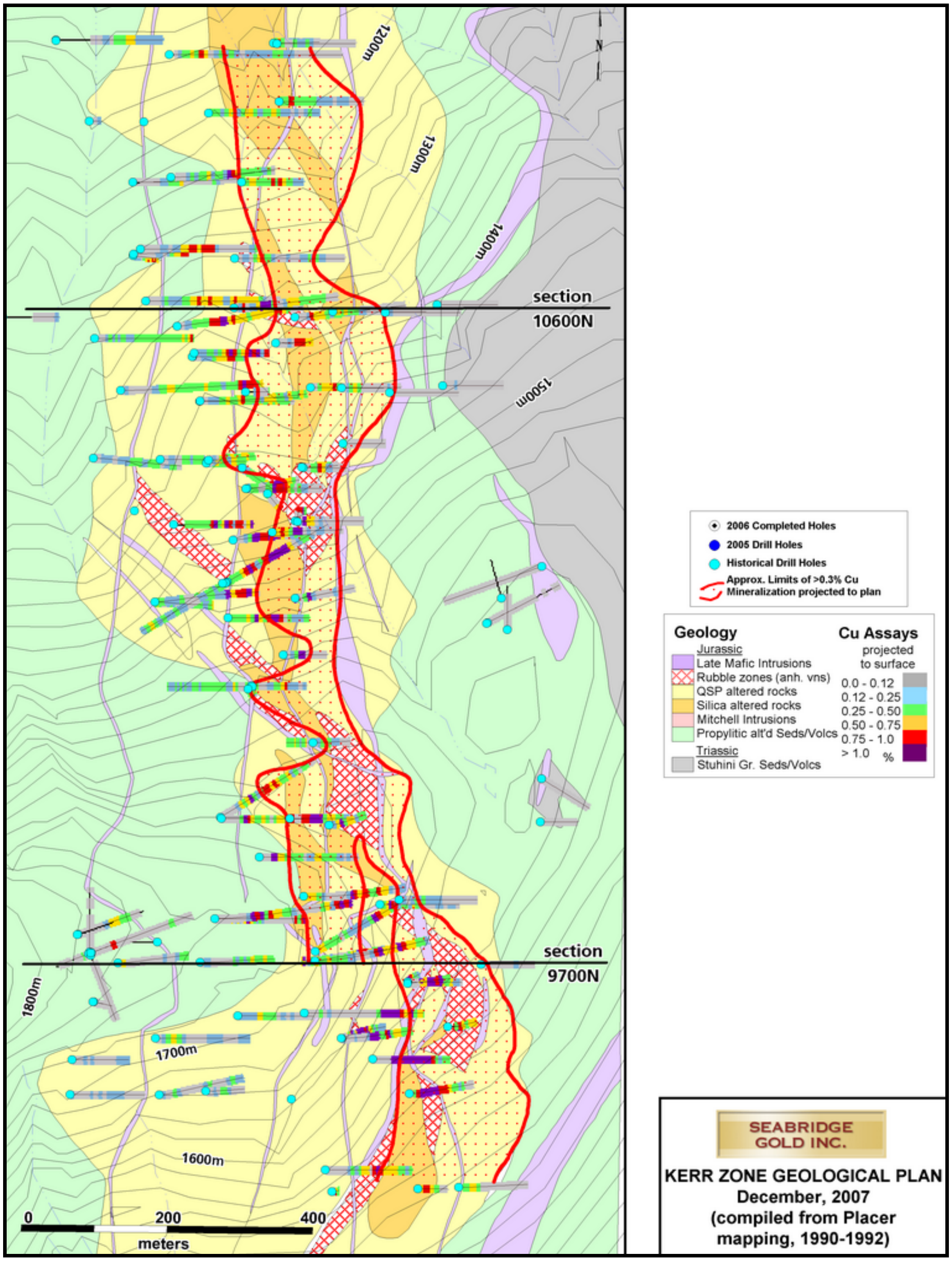


Figure 9.2 Kerr Cross Section 9700 North

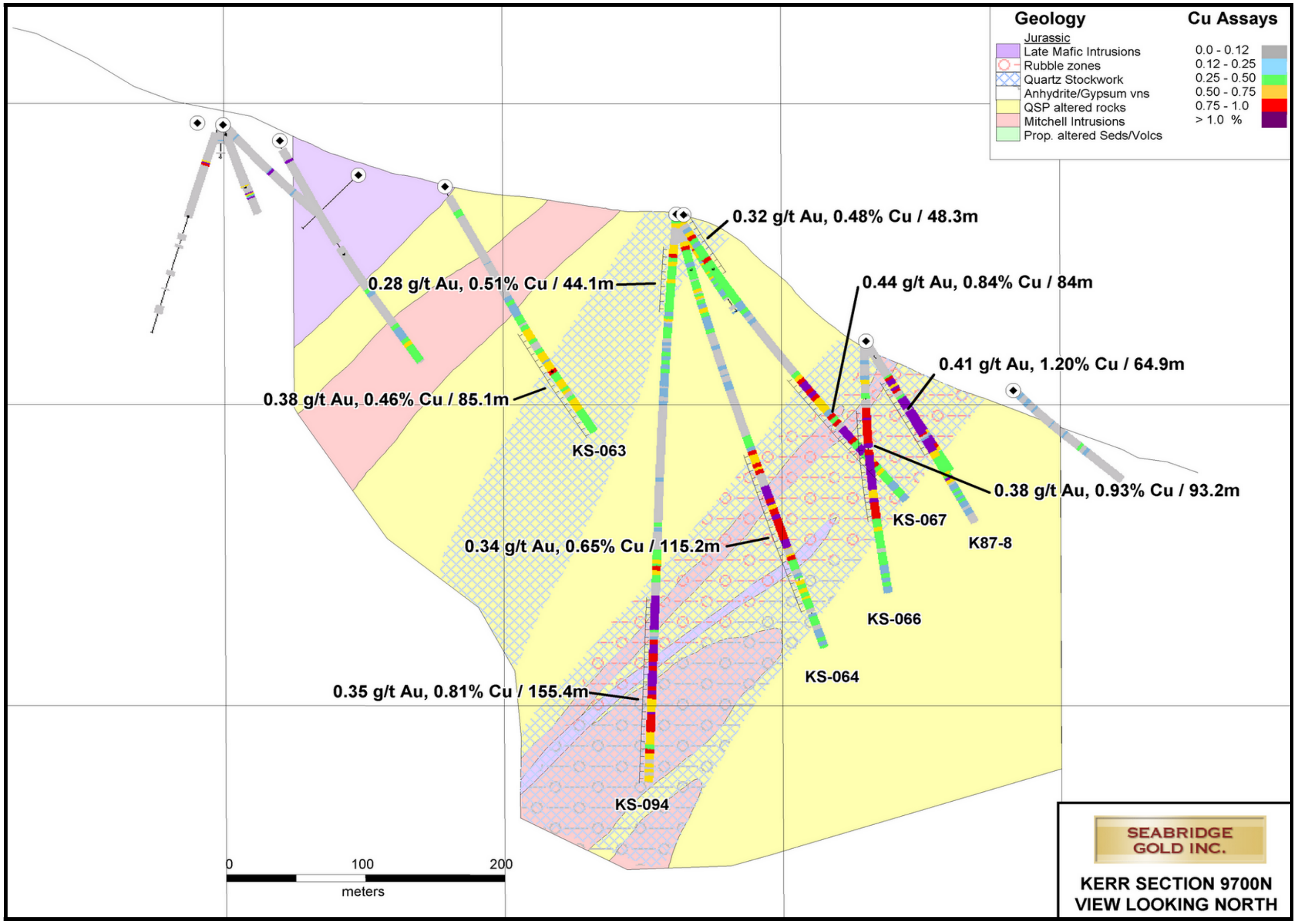
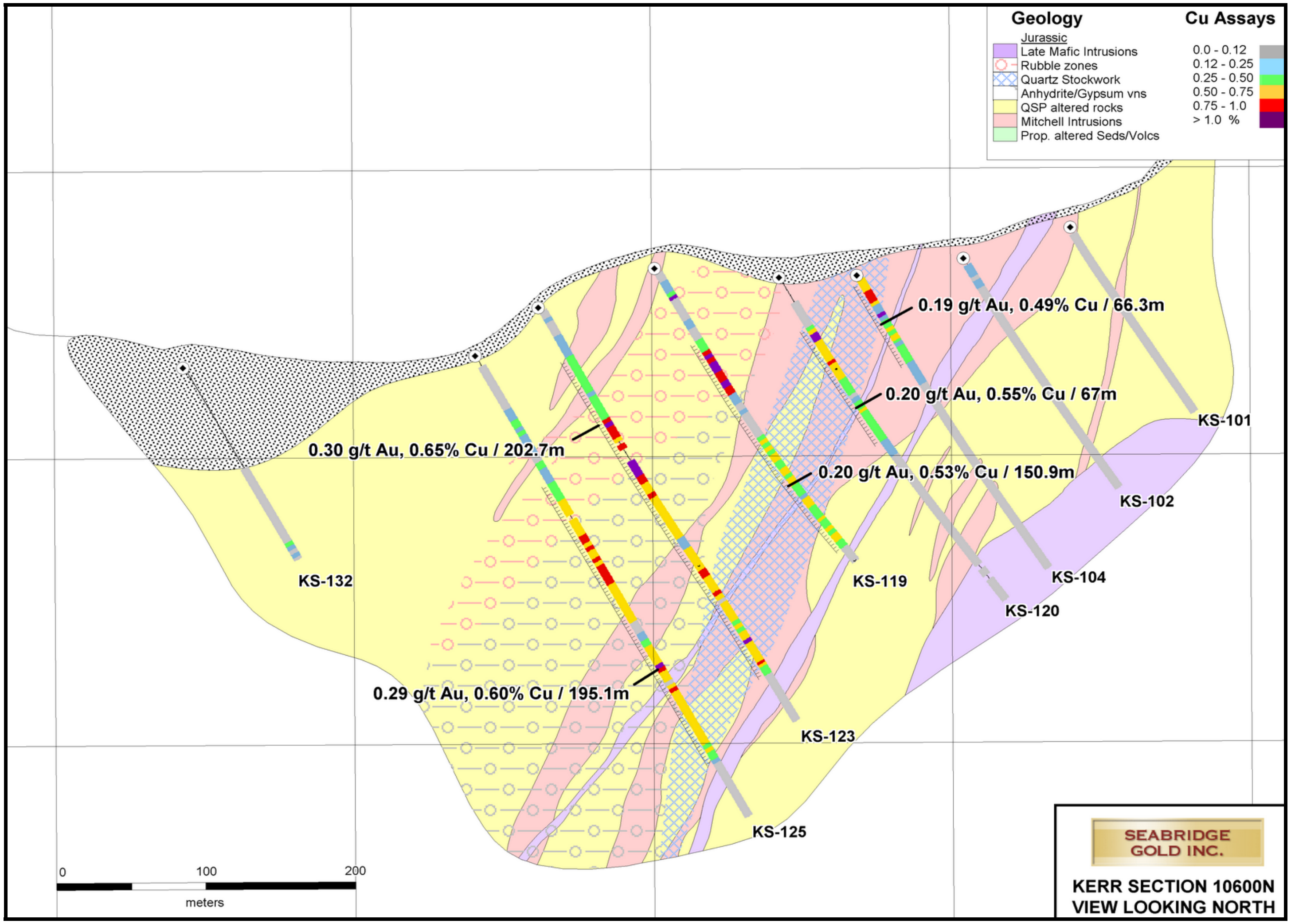


Figure 9.3 Kerr Cross Section 10600 North



9.1.1 LITHOLOGY AND STRUCTURE

The majority of the host volcaniclastic and sedimentary rocks belong to the Stuhini Group which is highly schistose within the deposit. Where they are undeformed, the sedimentary rocks consist primarily of coarse conglomerate, siltstone, mudstone, and minor greywacke. Undeformed volcaniclastic rocks are not present within the deposit but outcrops nearby contain well-bedded, sandy tuffs to coarse volcanic conglomerate. The presence of strongly flattened clasts was used to assign a volcaniclastic origin. Within the core of the deposit, deformation and alteration preclude assignment of protolith, and either “sericite schist” or “chlorite schist” is usually the most appropriate term.

Monzonite intrusions are plagioclase-hornblende-biotite porphyries with common apatite microphenocrysts. Primary hornblende and biotite are not observed but are recognized as hydrothermal chlorite and sericite pseudomorphs. Plagioclase phenocrysts are variably altered to sericite and have diffuse boundaries. Where alteration and deformation are intense, identification of monzonite may hinge on the recognition of plagioclase or hornblende phenocrysts alone. Several intrusive phases appear to be present, including breccias at the margins, but cannot be distinguished clearly by their mineralogy.

Monzonite is probably part of the “Mitchell Intrusions”, which belong to the Early Jurassic Texas Creek plutonic suite. This age is inferred by previous workers from the close relationship between monzonite and porphyritic dikes. Monzonite appears to be most abundant in the lower reaches of the deposit but it is also the suspected protolith for much of the strongly altered material in the upper central portions.

A large area of barren plagioclase porphyry and intrusive breccia occurs in the southeastern corner of the deposit. Alteration includes pervasive chlorite, epidote, sericite and carbonate. K-feldspar is a primary component in the groundmass of some porphyries. The contact between these rocks and mineralizing monzonite is probably a fault.

Plagioclase hornblende porphyry dikes and intrusions similar to the host monzonite are most abundant in the southern half of the deposit. They are generally massive and barren or only weakly mineralized and are inferred to be late phases of the same magma.

Metre-scale, barren albite megacrystic porphyry dikes intrude the deposit along generally north-south trends. Hyalophane megacrystic dikes intrude along east-west trends. These dikes likely correlate with “Premier porphyry” dikes of the Texas Creek plutonic suite commonly associated with copper and gold mineralization throughout the region. Aphanitic andesite dikes are common throughout the deposit, and are highly altered, massive, dark green, and composed of plagioclase, chlorite, ilmenite and sericite. These dikes generally cross-cut schistosity but many folded dikes have been observed on the surface.

Eocene kersantite, andesite and monzonite dikes up to 3 m wide intrude the deposit along the northerly foliation trend. These are composed of highly variable amounts of biotite, fine-grained plagioclase, chlorite, tremolite/actinolite, quartz, and K-feldspar. Coarse white carbonate and possible barite occur as local amygdules, especially along contacts.

The Kerr deposit occurs within a major northerly trending structural zone with strong foliation and widespread shearing. Individual structures within the deposit are masked by pervasive alteration and deformation.

9.1.2 ALTERATION

Abundant pervasive sericite occurs throughout the deposit, which is accompanied by chlorite replacement of mafic minerals in the main monzonite intrusion. Outward from this, strong chlorite-sericite alteration contains more pervasive chlorite than sericite.

Yellow and grey sericite alteration types occur peripheral to these two chlorite-bearing types. Sericite is commonly twice as abundant as chlorite. In drill core, zones of pale green sericite-dominant alteration are common. Patchy quartz is present in amounts varying from 5 to 15%. Pyrite content is generally less than 10%.

Dark green, pervasive chlorite-dominant alteration occurs around the margins of the main monzonite intrusion. It most commonly occurs between sericite-chlorite and intense grey sericite zones and may represent an alteration front. Up to 60% dark chlorite is accompanied by up to 30% sericite. Patchy quartz (5 to 15%) may locally represent dismembered veins. Anhydrite is most visible as white to pink coarsely crystalline veins up to several centimetres wide. Pyrite content is only 1 to 7%. Primary biotite phenocrysts have been replaced by chlorite. Apatite grains up to 15 mm are locally present in some of the most strongly altered zones.

Pervasive grey sericite alteration is characterized by 40 to 60% grey sericite with 5 to 10% quartz and 0 to 7% chlorite. Fine-grained plagioclase is commonly present in amounts varying from 20 to 50%, but much less where quartz is dominant. Intensity of alteration and deformation are such that the rock is best described as sericite or quartz-sericite schist. The pyrite content can be as high as 15%, especially in volcaniclastic rocks.

Pervasive yellow sericite alteration is a peripheral assemblage affecting only the Stuhini Group, primarily in the footwall below the main stockwork zone. This has the lowest average copper grade of all the pervasive alteration types. This style typically contains 5 to 15% original plagioclase, 30 to 60% yellow sericite, 10 to 20% quartz, and 10 to 20% pyrite. Yellow sericite commonly wraps around rounded quartz fragments, giving these rocks an augen-like, granular appearance. Green sericite commonly occurs in minor amounts as a replacement of selected clasts. As alteration and deformation weaken, pervasive sericite changes from yellow to green, and gradually disappears as sedimentary textures become clear.

Anhydrite veining is most commonly associated with chlorite bearing alteration types. It is characteristic of texturally destructive chlorite-sericite alteration and the upper portions of sericite-chlorite altered monzonite. Anhydrite veins locally carry minor chalcopyrite. During deformation, anhydrite was remobilized into irregular, crosscutting networks of veinlets that post-date all other vein types. Anhydrite has hydrated to gypsum to depths of up to 250 m, and leaching by groundwater has produced large areas of voids and broken rock called "rubble". Core recovery in these zones is poor.

9.1.3 MINERALIZATION

The most important mineralization type is quartz stockwork, which drapes over the main monzonite intrusion and extends a considerable distance down the eastern side, along the footwall of the deposit. Deformation of mineralized quartz veins has resulted in segregation of sulphides into interstices between granular recrystallized quartz, resulting in a 'crackled' texture. Chalcopyrite also occurs as fracture fillings in an earlier generation of coarse vein pyrite. Narrow veins and veinlets are commonly highly contorted. The quartz stockwork veins may contain any combination of pyrite, chalcopyrite, bornite, tetrahedrite, tennantite, or rare enargite. Thin films of secondary digenite and chalcocite are also present but are only locally significant near the surface. Small flakes of possibly primary crystalline covellite are locally abundant, especially in rubbled zones and near-surface areas.

In addition to crackled quartz stockwork, mineralization is hosted by several other types of veinlets. Ditson et al. suggest the following vein classification for Kerr:

- pyrite ± quartz, sericite, minor chalcopyrite (pre-deformation)
- quartz ± pyrite, carbonate, anhydrite, sericite, chlorite, chalcopyrite (pre-deformation)
- anhydrite ± chalcopyrite (pre-deformation)
- carbonate ± minor chalcopyrite, bornite (syn-/post-deformation)
- quartz + carbonate, chlorite, chalcopyrite (post-deformation)

Chlorite-bearing alteration types host the greatest variety of vein types. Mineralization grading over 0.4% Cu is generally located within or adjacent to crackled quartz stockwork; however, there are significant tonnages in non-stockwork mineralization grading over 0.4% Cu in the northern sector in monzonite below the stockwork. All mineralization grading over 1% Cu occurs within stockwork. The Au:Cu ratio (g/t:%) for all rocks grading over 0.4% Cu averages 0.4.

Molybdenum values were analyzed are most commonly less than 100 ppm but range up to 423 ppm. Molybdenite is associated with chloritic alteration, and in the northern sector yellow sericite altered rocks below monzonite.

9.1.4 STRUCTURE

The Kerr deposit occurs within a major northerly trending structural zone with strong foliation and widespread shearing. Individual structures within the deposit are masked by pervasive alteration and deformation.

9.2 SULPHURETS ZONE

The Sulphurets deposit has been delineated by over 15,200 m of core drilling in 65 drill holes spaced at intervals of 50 to 100 m. In total, six different operators drilled the project between 1968 and 2006. The deposit is comprised of two distinct zones – Raewyn and Breccia Gold.

The Raewyn Copper-Gold Zone hosts mostly porphyry style disseminated chalcopyrite and associated gold mineralization in moderately quartz stockworked, chlorite-biotite-sericite-magnetite altered volcanics. The alteration and mineralization are centred on a narrow, apparently conformable body of porphyritic quartz monzonite. It has an apparent northeasterly strike and dips about 45° to the north. It may be offset in en echelon style by several north-northeasterly trending vertical structures. The mineralization is open at depth and to the northeast.

The Breccia Gold Zone hosts mostly gold bearing pyritic mineralization with minor chalcopyrite and sulfosalts in a K-feldspar-siliceous hydrothermal breccia that apparently crosscuts the Raewyn porphyry copper-gold deposit. It comprises altered intrusive clasts in a matrix of mainly silica and sulfides. Both zones have an intense phyllic overprint that nearly masks all earlier alteration phases. According to Fowler et al. (1995), the Breccia Zone has an apparent northerly strike and dips to the west, and is open down dip. A late, barren, pyritic monzogabbro cuts off the Breccia zone on the northwest side.

Most of the following description has been extracted and modified from the paper by Fowler and Wells (1995). Figure 9.4 is a generalized geological plan showing the surficial geology of the Sulphurets deposit along with drill holes and the approximated surface trace of 0.30% copper mineralization. Figure 9.5 is a northwest-southeast trending cross section through the Sulphurets deposit.

Figure 9.4 Sulphurets Geologic Map

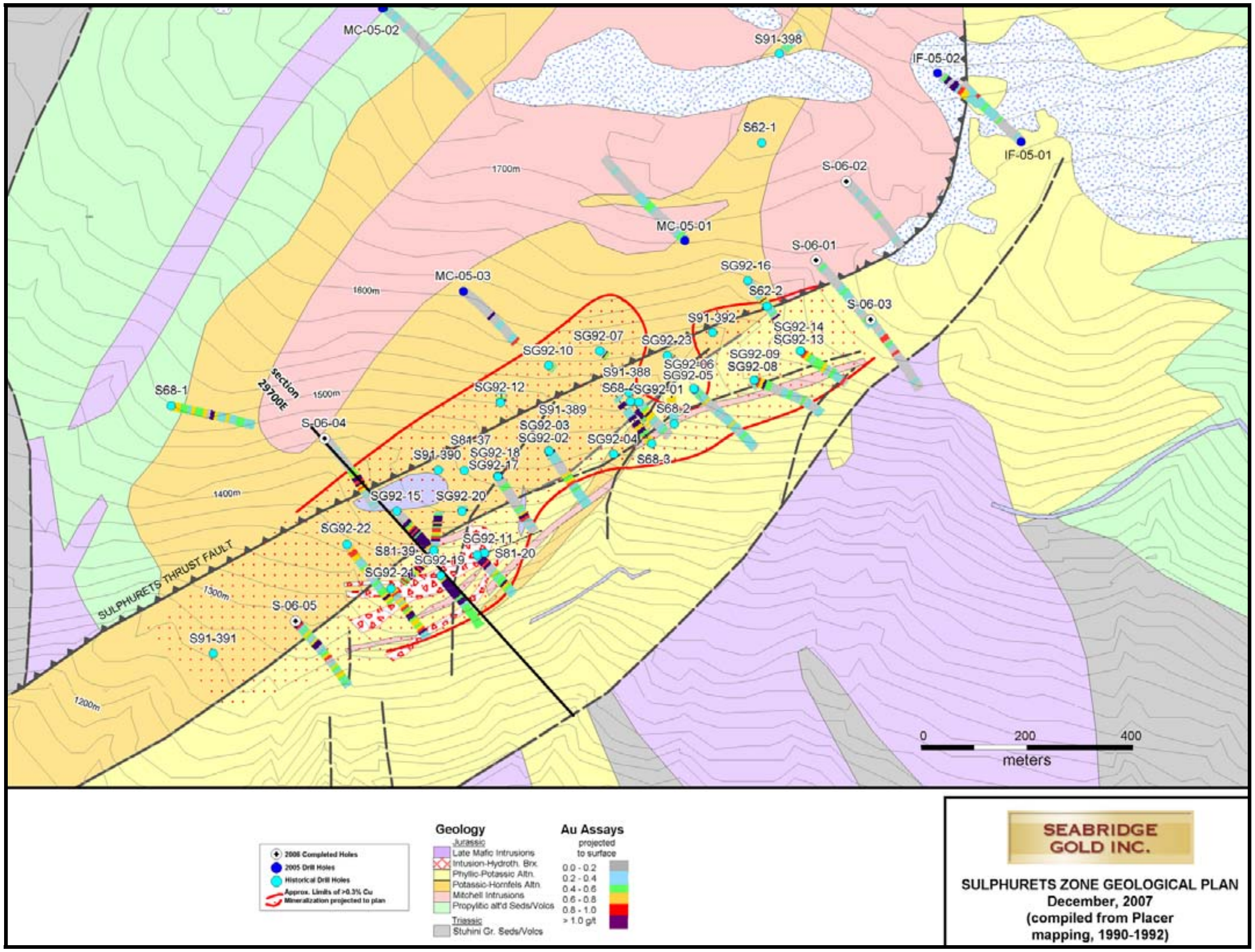
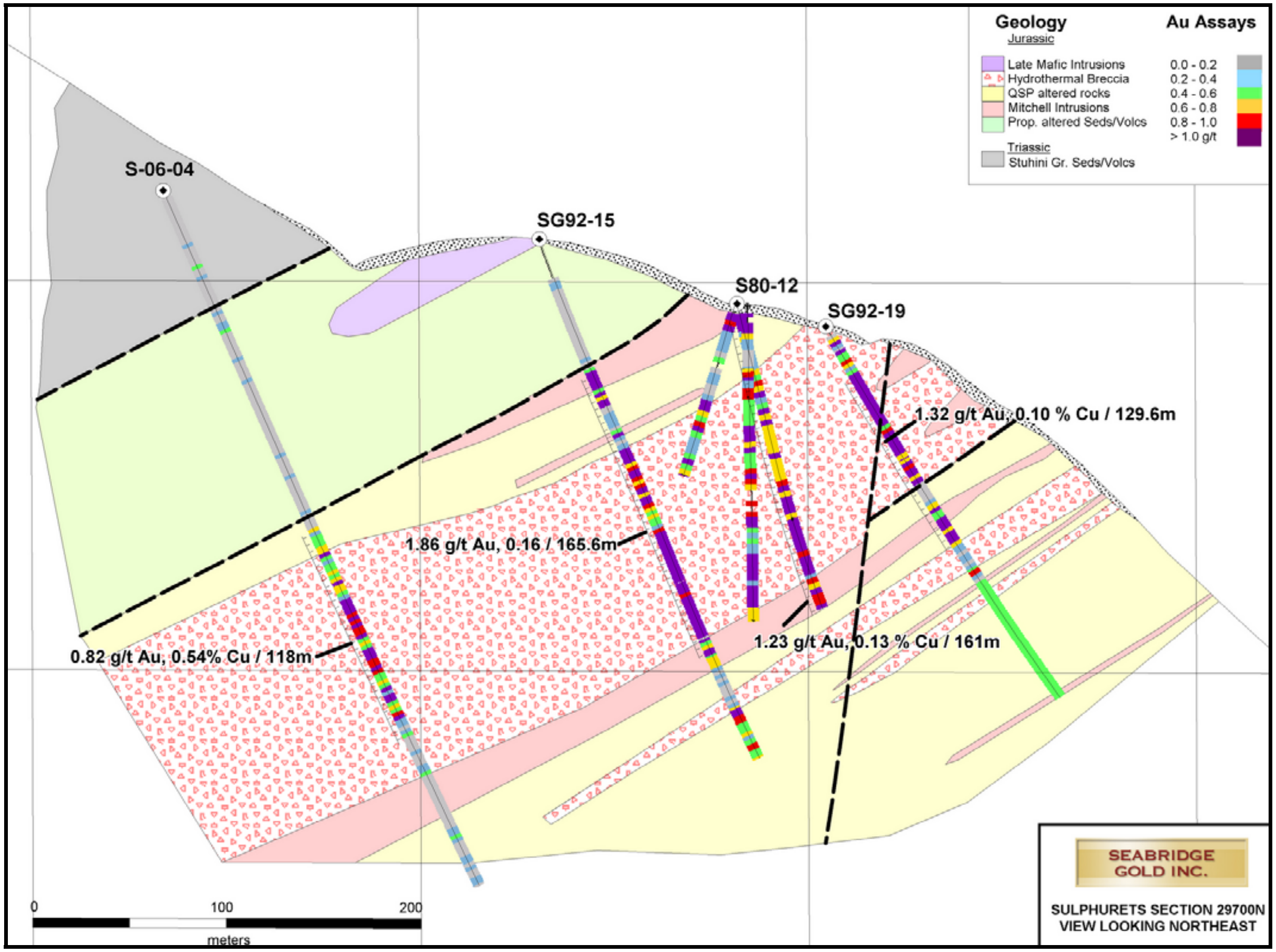


Figure 9.5 Sulphurets Cross Section 29600 East



9.2.1 LITHOLOGY AND STRUCTURE

The Sulphurets deposit (or Sulphurets Gold zone) formed in a high level, transitional porphyry copper-gold system that was thrust over the deeper levels of a syenite-centered porphyry copper-gold deposit (Main Copper zone) along the Sulphurets Thrust Fault (STF). Volcanic sequences on either side of the thrust have been assigned to Hazelton Group. Below the STF, the volcanics consist of propylitic to potassic altered, massive to tuffaceous trachyandesites, with local sediments, intruded by northerly-trending feldspar porphyry dikes. Trachyandesite crystal and ash tuffs, flows, and breccias are interlayered with dark argillites, volcanic derived sandstones, cherts, and cherty tuffs. Generally, in areas of intense alteration and mineralization, the protolith cannot be assigned accurately. Late hornblende phyric monzonite to monzogabbro dikes and sills intrude the area.

The Sulphurets Gold Zone is centred along the Raewyn Fault, a zone of strong faulting and phyllic-quartz-sericite-pyrite, intermediate argillic, and potassium silicate alteration. The Raewyn Fault trends northeasterly, subparallel to the STF, and is well exposed for much of its length along the main cliff, forming a prominent gossan. Copper-gold mineralization is usually coincident with areas of strongest fracturing and potassium silicate alteration. At the southern end of the “Raewyn panel”, auriferous hydrothermal breccias constitute the Breccia Gold Zone.

Above the STF, intermediate volcanics, massive green flows, and tuffs are intruded by feldspar porphyry quartz syenites and potassic monzonite dikes. Rocks in the periphery of the dikes are K-feldspar-altered, and contain disseminated and fracture controlled chalcopyrite. The dikes are grouped with the Mitchell intrusions that correlate with late Jurassic Texas Creek intrusions common throughout the region.

Brittle fracturing, typical of hornfelsed aureoles, is widespread in the upper plate rocks, and numerous northerly to north-northeasterly striking steep-westerly dipping fractures and fracture zones are present. Below the STF, the most prominent feature is the subparallel, northeasterly-dipping Raewyn structural-alteration panel. This panel is separated from the STF by a 100 to 200 m wide section of less deformed and less altered volcanic rocks. It is transected by shallowly and steeply dipping fault sets, some of which are intra-mineral and others post-mineral. Bedding, where visible, dips at fairly steep angles to the north and northwest but it is not as steep as the sub-vertical foliation.

9.2.2 ALTERATION/MINERALIZATION – RAEWYN COPPER-GOLD ZONE

Gold and copper mineralization here is associated with the main Raewyn dike. Average copper and gold values from the mineralized zones below and within the Raewyn panel are fairly consistent. Copper values range from 0.3 to 0.7% and gold values are 0.4 to 1.2 g/t. Strong quartz-sericite-pyrite (phyllic) alteration largely overprints pre-existing assemblages; however, a considerable amount of K-feldspar is present from an early widespread potassic alteration event. Outboard from the quartz-sericite-pyrite alteration, the volcanic rocks are chlorite-altered and locally contain epidote, magnetite, and variable carbonate (propylitic).

Multiphase brecciation, alteration, veining, and widespread recrystallization characterize the zone. Vein assemblages include:

- chalcopyrite, quartz, chlorite, sericite ± albite, and carbonate
- chalcopyrite, quartz, pyrite, biotite, sericite, minor chlorite, and molybdenite
- milky quartz veins with coarse blebby chalcopyrite, minor pyrite, and chlorite.

Below the Raewyn panel, biotite alteration with chalcopyrite may extend for 10 m or more from the intrusion into the wall rocks, and overprints earlier K-silicate assemblages. Locally, siliceous-biotite hydrothermal breccias occur within the panel. Heterolithic, siliceous hydrothermal breccias have significant gold values, little copper, and may have associated dark tourmaline. Late high-angle quartz veins, up to 3 m wide, occur throughout most commonly close to faults and cross-cut all alteration domains. They contain coarse chalcopyrite, elevated gold grades, pyrite, tetrahedrite ± arsenopyrite, and molybdenite.

9.2.3 *Alteration Mineralization – Breccia Gold Zone*

Ditson et al. (1995) suggest the following sequence of events in the Breccia Zone area:

- intrusion of Raewyn monzonite, followed by
- main phase of hydrothermal breccias with K-feldspar alteration, and
- late-stage siliceous hydrothermal activity with local breccia pipes.

The K-feldspar hydrothermal breccias are characterized by numerous, millimetre scale, subangular to rounded, ground mass supported mono- to hetero-lithic fragments in a K-feldspar rich groundmass. Pyrite content ranges from 5 to 20%, and gold content ranges from 0.12 to 5.6 g/t, averaging 1.16 g/t (average copper content of 0.10%). The siliceous breccias are dominated by aphanitic, siliceous, and pyritic groundmass, rare chalcopyrite, and variable gold content ranging from 0.10 to 21.20 g/t, averaging 1.52 g/t. Both breccias locally contain significant amounts of dark coloured tourmaline aggregates and rosettes.

9.3 Mitchell Zone

The Mitchell Zone is exposed in Mitchell Creek Valley through an erosional window exposing the footwall of the Mitchell Thrust Fault. The zone is a moderately dipping, roughly tabular gold-copper deposit measuring approximately 1,600 m along strike, 400 to 900 m down dip, and at least 300 to 600 m thick. It consists of a foliated, schistose zone of intensely altered and sulfide bearing rocks, with a variably distributed stockwork of deformed and flattened quartz veinlets. The schistosity generally follows an east-southeast direction, and dips steeply to moderately to the north. The Mitchell Zone is considered to lie within the spectrum of the gold-enriched copper porphyry environment.

Metals, chiefly gold and copper (in terms of economic value), are generally at low concentrations, finely disseminated, stockwork or sheeted veinlet controlled, and pervasively dispersed over dimensions of hundreds of metres. Grades diminish slowly over large distances; sub-economic grades are encountered at distances of several hundreds of metres beyond the interpreted centre of the system. This is distinct from the Sulphurets and Kerr zones, where there are more abrupt breaks in grade due to higher structural complexity and juxtaposition of weak and moderate grade domains by faulting, both syn-mineral structures controlling breccia contacts and post-mineral faulting and displacements.

9.3.1 *Lithology and Structure*

Due to the intensity of hydrothermal alteration and strong post-mineral shearing, especially at Mitchell Creek, it is difficult to impossible to recognize the original protolith. This is typical of phyllic-argillic or quartz-sericite (illite)-pyrite altered rocks. In chlorite-sericite and propylitic altered rocks, a homogeneous, tuffaceous texture is often observed and the host is believed to be intermediate volcanic tuffs or volcaniclastics. However, these textures may in part be shear related. Diffuse, ghost porphyritic textures may reflect dikes of the Mitchell intrusions. Rare, metre-scale, aphanitic intermediate dykes are post-alteration and unmineralized.

Below the Mitchell Thrust Fault in the drilled area, alteration intensity gradually diminishes westward. Where not obliterated by alteration, fine to coarse, lithic to cystal, tuffaceous, intermediate volcanics are dominant, followed by vaguely bedded, fine grained volcaniclastics and argillites, more common to the west. Government mappers have assigned the stratigraphy under the Mitchell Fault to the Jurassic Hazelton Group; however, in many ways it more closely resembles descriptions of the Triassic Stuhini Group. Within the central and eastern portions of the drilled area, intervals of bleached, vaguely coarse porphyritic textured rocks may be altered dikes of the Mitchell intrusive suite.

Above the Mitchell Fault, alteration is mainly confined to siliceous hornfelsed zones adjacent to porphyritic monzonite and granitic Mitchell intrusions. The host rocks are mostly dark, fine grained volcaniclastics and argillites assigned to the Triassic Stuhini Group. The intrusions appear to have thick, sill-like geometries, with thin, anastomizing dykes in the contact zones. Similar intrusives and surrounding siliceous alteration zones have been mapped above the Mitchell Thrust Fault on both sides of Mitchell Creek Valley.

9.3.2 ALTERATION

The general pattern is intense phyllic-argillic (quartz-sericite (illite)-pyrite) alteration at the east side of the drilled area, which gradually diminishes to the west. This is characterized by complete loss of mafics, introduction of millimetre to centimetre scale, deformed quartz veinlets in stockwork and sheeted arrays, with mostly creamy white to grey sericite and/or illite as the interstitial vein component. There are clearly multiple stages of quartz veining. Later veins are rich with coarse pyrite, often with molybenite, and centimetre scale, near massive coarse pyrite veins are common. The phyllic alteration has a strong foliation best manifested in sericite rich intervals. Sheeted quartz veinlets often follow the foliation and may indicate deformation of pre-existing veins or perhaps contemporaneous formation of quartz veinlets and deformation. In some surface exposures, intensely deformed zones contain coarse clasts of rotated, previously veined material, and strong shear textures are noted in microscopic thin sections. The highest concentrations of pyrite and quartz veinlets are generally strongly coincident with phyllic-argillic alteration. The concentration diminishes gradually westward; however, there are wide intersections at depth of intensely veined to near massive veins and breccias of hydrothermal quartz. These have a strong magnetite and potassic feldspar component and the highest continuous Au-Cu grades, suggesting hotter and deeper mineralizing conditions.

Later veins cut through the clasts and sericite rich matrix. There are also post-deformational, coarse, blotchy, and random quartz-calcite-chlorite decimetre scale veins as well as millimetre scale, random, metamorphic calcite veinlets. Coarse chalcopyrite is often observed in the late calcite veinlets and likely has been remobilized.

The west side of the drilled area is dominated by propylitic alteration, characterized by pervasive chloritization of mafics, and quartz-pyrite alteration of most other silicates. This alteration is also stockwork veinlet controlled. The degree of foliation is much less than the phyllic zone to the east. The abundance of epidote and calcite increase westward. Some phyllic alteration is present but is restricted in extent and probably structurally confined.

There is a widespread transitional alteration facies of pale green sericite and/or bleached chlorite, which may reflect partial or diminishing phyllic-argillic alteration of pre-existing propylitic and local potassic alteration. The highest gold and copper grades are roughly coincident with this transitional area.

9.3.3 STRUCTURE AND METAMORPHISM

Regional mapping by government geologists indicate Jurassic Hazelton Group rocks exhibit overturned folds that are southeast vergent in the region of Kerr-Sulphurets. The thrusts are also southeast vergent. Triassic Stuhini Group rocks above the faults form the east side of a broad north plunging anticlinorium. Post-mineralization deformation resulting in re-mobilization of metals during the Cretaceous has probably contributed to the homogeneity of grades.

10.0 EXPLORATION

This section describes Seabridge's 2008 exploration program at KSM. Prior exploration activities have been described in various Technical Reports prepared by RMI, which are available on SEDAR.

10.1 2008 KSM Exploration Program

Seabridge's 2008 exploration efforts centred primarily around infill and step-out drilling within the Mitchell deposit in order to improve the overall confidence in the estimate of the in situ resources and to try exploring for potentially higher-grade zones within the currently recognized deposit. In 2008, 34 core holes totalling 15,416 m were drilled within the Mitchell Zone. In addition to logging and sampling, representative samples from the 2008 drilling program were selected for metallurgical test work that is currently underway.

Three diamond core holes totalling 1,761 m were also drilled within the Sulphurets zone to follow up on the down-dip extension of previously defined mineralization. Like the 2008 Mitchell drilling, samples from the 2008 Sulphurets drilling program were selected for ongoing metallurgical testing.

The drill core was logged on site by Seabridge geologists who collected a variety of information including lithology, alteration, mineralization, and geotechnical attributes like core recovery, RQD, and fracture frequency. After photographing the core, it was sawn in half with primarily 2-m-long samples collected and sent to Eco-Tech Laboratories Ltd. (Eco-Tech), a commercial laboratory located in Kamloops, BC. The samples were analyzed for gold, copper, and a suite of other elements. A large number of bulk density determinations were completed by Seabridge geologists from all rock types and alteration assemblages. A number of geologic traverses were completed by Seabridge geologists to update the current geologic understanding of the district.

10.2 Results of 2008 Exploration Program

The previous geologic interpretation of the Mitchell deposit was updated using the 2008 core hole data and surface mapping data. RMI notes the updated geologic interpretation remains virtually unchanged from the previous interpretation (refer to Sections 7.0, 8.0, and 9.0). The drilling, sampling, and assay procedures employed for the 2008 exploration program were adopted from the previous year and are discussed in Sections 11.0 and 12.0, respectively.

10.3 INTERPRETATION OF EXPLORATION DATA

RMI combined the 2008 drill hole information with the previously collected data so that an updated geologic model and estimate of mineral resources could be made. The steps involved and results from those activities are discussed in Section 17.0.

10.4 STATEMENT REGARDING NATURE OF INVESTIGATIONS

All of the exploration activities that were conducted at Mitchell in 2008 were either directly carried out by Seabridge's geologic staff (e.g. geologic traverses, geologic mapping, and core logging) or directly supervised by Seabridge personnel (e.g. drilling).

11.0 DRILLING

This section describes Seabridge's 2008 drilling program at KSM. Previous drilling programs have been described in various NI 43-101 Technical Reports prepared by RMI for the Kerr, Sulphurets, and Mitchell deposits, which are available on SEDAR.

11.1 2008 DRILLING CAMPAIGN

Seabridge completed a helicopter supported diamond drilling program at Mitchell Creek in 2008. In the Mitchell Creek area, 34 holes were completed totalling 15,416 m. Hy-Tech Drilling Ltd. from Smithers, BC, drilled all of the holes using a Tech-5000 Fly Rig using NQ tools. Three core holes were also drilled within the Sulphurets deposit totalling 1,761 m.

As previously mentioned, all drilling was helicopter-supported using a Eurocopter A-Star model 350B2 that was contracted from Lakelse Air Ltd. from Terrace, BC. The drilling operations were conducted from the Sulphurets Creek camp which is located southwest of the Mitchell deposit.

Approximately 8,611 diamond core samples were collected from the 2008 Mitchell and Sulphurets drilling program and analyzed by Eco-Tech out of Kamloops, BC, for gold, copper, and a suite of other elements.

11.2 DRILL HOLE SURVEYING

The procedures used for spotting the drill holes, surveying collars, and down-hole surveying methods are basically the same as those described for the 2007 drilling campaign (Lechner, 2008). As mentioned in Section 14.3, a new topographic base map and different datum (NAD83) was adopted by Seabridge. This section briefly describes how the drill hole collar locations were initially acquired and what steps were undertaken to translate those locations into the new coordinate system:

11.2.1 KERR DEPOSIT

All drilling at Kerr predates Seabridge's ownership of the property. Initially the drill hole collars were located in a local mine grid system that was tied to the NAD27 datum by Placer Dome in the early 1990s. Seabridge personnel located nine Placer Dome drill hole collars and surveyed them with their handheld Trimble DGPS instrument. These re-surveyed locations along with the "original" coordinates for all Kerr holes were provided to Aero Geometrics Ltd. (Aero Geometrics). The drill hole collars were adjusted by Aero Geometrics from their original local grid to NAD27 using affine transformation and then further transformed into NAD83 using Canadian National Transformation v2.0. No elevation adjustments were made by Aero Geometrics and, when the transformed drill hole coordinates were compared with the new Light Detection and Ranging (LiDAR) based topographic surface, it was apparent that some adjustment was required. The Kerr drill hole collars were adjusted to match the new NAD83-based topographic surface.

11.2.2 *Sulphurets Deposit*

Holes drilled prior to Seabridge's entry into the district were treated in the same manner as described for the Kerr deposit. Seabridge-era drill holes were located in the field using a Trimble handheld DGPS unit. Depending on terrain, satellite coverage, and other factors, it is possible to achieve sub-metre accuracy. All of the Seabridge drill hole collars were originally located in NAD27 coordinates. These data were sent to Aero Geometrics who converted the drill hole collars to NAD83 coordinates. The translated drill hole collars were compared with the new LiDAR topographic surface. The elevation of drill holes deemed to be too high or low were adjusted to match the new LiDAR surface.

11.2.3 *Mitchell Deposit*

The same procedures were used to locate Seabridge's Mitchell drill holes as was described for the Sulphurets holes. Falconbridge drill holes were located in the field using a standard uncorrected DGPS unit. Like the other two deposits, the elevation for some of the drill holes was adjusted to match the new NAD83 LiDAR topography.

11.3 Drill Core Processing

The following was taken directly from RMI's NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia", dated April 6, 2007, which is available on SEDAR.

Drill core was placed into wooden trays directly upon emptying the core tube at the drill site. A wooden run block, marked with the hole depth in metres, was placed in the core trays upon the completion of each drill run, which (in good conditions) was based on full core runs of 3 m. Core tubes and rods were in metric lengths. The core boxes were covered with a plywood lid that was securely nailed to the core box and then placed in a metal basket manufactured by Longyear for helicopter slinging. The baskets were slung by helicopter to camp, typically after the morning shift change, depending on productivity and weather conditions.

At camp, the core basket was placed near the core logging shack. Each box was laid out in sequence on elevated racks in the core shed. The core was examined for condition, missing core, and depth tag errors. Boxes were labelled with black felt tip pens and embossed steel tags containing the hole number, depth, and box number. The core was then washed with fresh water. Geotechnical data including recovery, RQD, and natural breaks were recorded for each drill run, as marked by the wooden core run blocks. This information was recorded by the geologist or trained logging assistant under direct supervision of a geologist.

The geologist then recorded key geologic information including lithology, alteration, structure, and mineralization using a pre-determined format and coding system that is shown in Table 12.1 through to Table 12.3. The data were either entered directly into the digital database or onto paper logs which was then entered into the digital database at the camp office. The geologist or assistant under the direct supervision of the geologist marked sample intervals on the core at fixed 2-m-long intervals or at geological contacts so that each sample was 2 m or less. Sample lengths of 2 m followed Falconbridge's protocol for copper-gold porphyry prospects, which is in line with accepted industry practices for this style of mineralization.

The core at the beginning of each sample was marked with a wax pencil and a Teflon-coated paper tag with a unique identification number, which was stapled to the core box adjacent to the wax marking. Duplicates of the paper tag with the identification number were also placed at the beginning of each sample and were placed on the sample bag that was sent to the assay laboratory. Another duplicate of the tag, with the identification number, hole number, and depth interval was stored. This information was entered into the digital database assay table. The entire hole (excluding any recovered overburden) was sampled. The core was then digitally photographed. All digital photo files are maintained in the company's digital database. A wax pencil was then used to mark a cut line along the top of the drill core to avoid any sampler induced selection bias and to ensure that the same side of the halved core relative to its placement in the box was put into the sample bag that was sent for assay.

11.4 Relationship Between Drill Hole and Mineralization Orientation

The following was taken directly from RMI's NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia", dated April 6, 2007, which is available on SEDAR.

At Mitchell Creek, most of the holes were drilled at a pre-assigned azimuth and dip of 190° and -60°. Orientation of mineralization has been difficult to determine from surface mapping and sampling as it is finely disseminated and pervasive with no obvious alteration control or relationship to vein density or orientation. It has been assumed that the mineralization at Mitchell Creek is likely orientated similar to the intense foliation and sheeted, deformed quartz stockwork veining, which generally dips at -70° along a N10°E azimuth. The assigned drill hole orientation was chosen to cut this orientation as close to perpendicular as possible. Completion of the program and inspection of the metal distribution based on assay results suggests there is a trend to gold mineralization that, in a general sense, dips moderately on an azimuth of 060 to 080. Thus drilled intervals may be slightly oblique to the mineralization trend and may not accurately reflect true thicknesses.

11.5 DRILL HOLE DATA

Table 11.1 through to Table 11.3 summarize drilling data by the companies that did the drilling for the Kerr, Sulphurets, and Mitchell deposits, respectively.

Table 11.1 Kerr Drill Hole Summary by Company

Company	Year Drilled	Hole Prefix	No. Holes	No. Metres	% of Total
Brinco	1985	85-nnn	3	189.90	0.7
Western Canadian	1987-1988	K87-nnn, K88-nnn, 88-nn	36	5,324.56	20.2
Newhawk Gold	1988	T88-nnn	2	115.21	0.4
Sulphurets Gold	1989	K89-nnn, T89-nnn	20	4,365.35	16.5
Placer Dome	1992	KS-nnn, KS92-nnn	83	16,413.57	62.2
Total	**n/a**	**n/a**	**144**	**26,408.59**	**100.0**

Table 11.2 Sulphurets Drill Hole Summary by Company

Company	Year Drilled	Hole Prefix	No. Holes	No. Metres	% of Total
Granduc	1962, 1968	S62-n, S68-n	6	1,016.02	6.7
Esso	1980, 1981	S80-nn, S81-nn	14	2,275.23	15.0
Newhawk Gold	1991	S91-nn	7	1,306.30	8.6
Placer Dome	1992	SG92-nn	23	5,577.34	36.7
Falconbridge	2005, 2006	MC-05-nn, MQ-05-nn, IF-05-nn	7	1,648.09	10.8
Seabridge	2006, 2008	S-06-nn, S-08-nn	8	3,384.00	22.3
Total	**n/a**	**n/a**	**65**	**15,206.98**	**100.0**

Table 11.3 Mitchell Drill Hole Summary by Company

Company	Year Drilled	Hole Prefix	No. Holes	No. Metres	% of Total
Newhawk	1991	S91-nnn	4	647.30	1.6
Falconbridge	2005	NM-05-nn, WM-05-nn	4	1,197.29	3.0
Seabridge	2006-2008	M-06-nn, M-07-nn, M-08-nn	95	38,571.87	95.4
Total	**n/a**	**n/a**	**103**	**40,416.46**	**100.0**

Figure 11.1 and Figure 11.2 show drill hole collar map locations for the Kerr and Sulphurets/Mitchell deposits, respectively. The purple lines in Figure 11.1 show the

surface trace of 0.4% copper mineralization at Kerr. The blue lines in Figure 11.2 show the 0.35 and 0.25 g/t surface trace of gold mineralization at Sulphurets and Mitchell, respectively. Both collar maps show 15 m topographic contours.

Figure 11.1 Kerr Drill Hole Locations

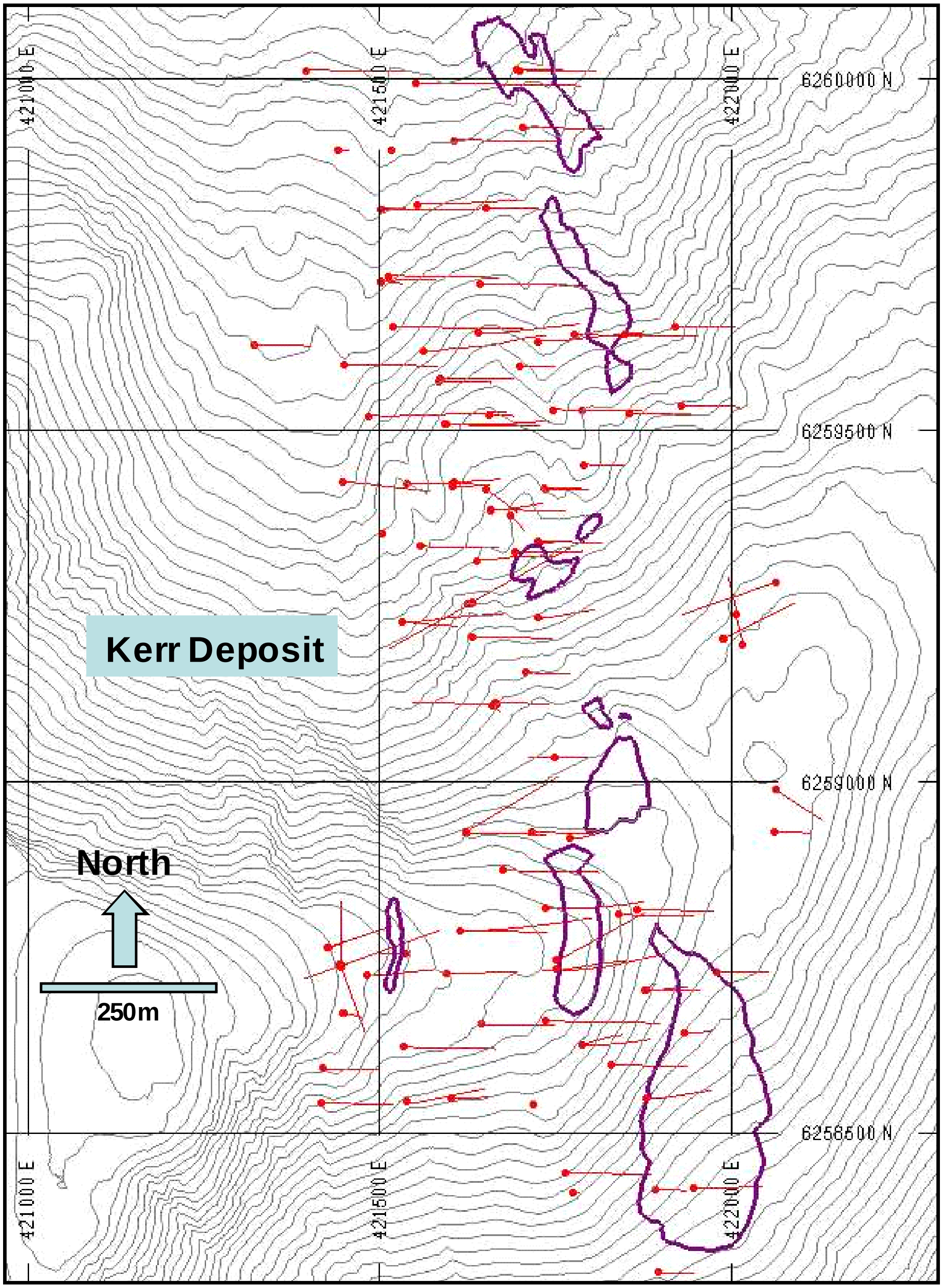
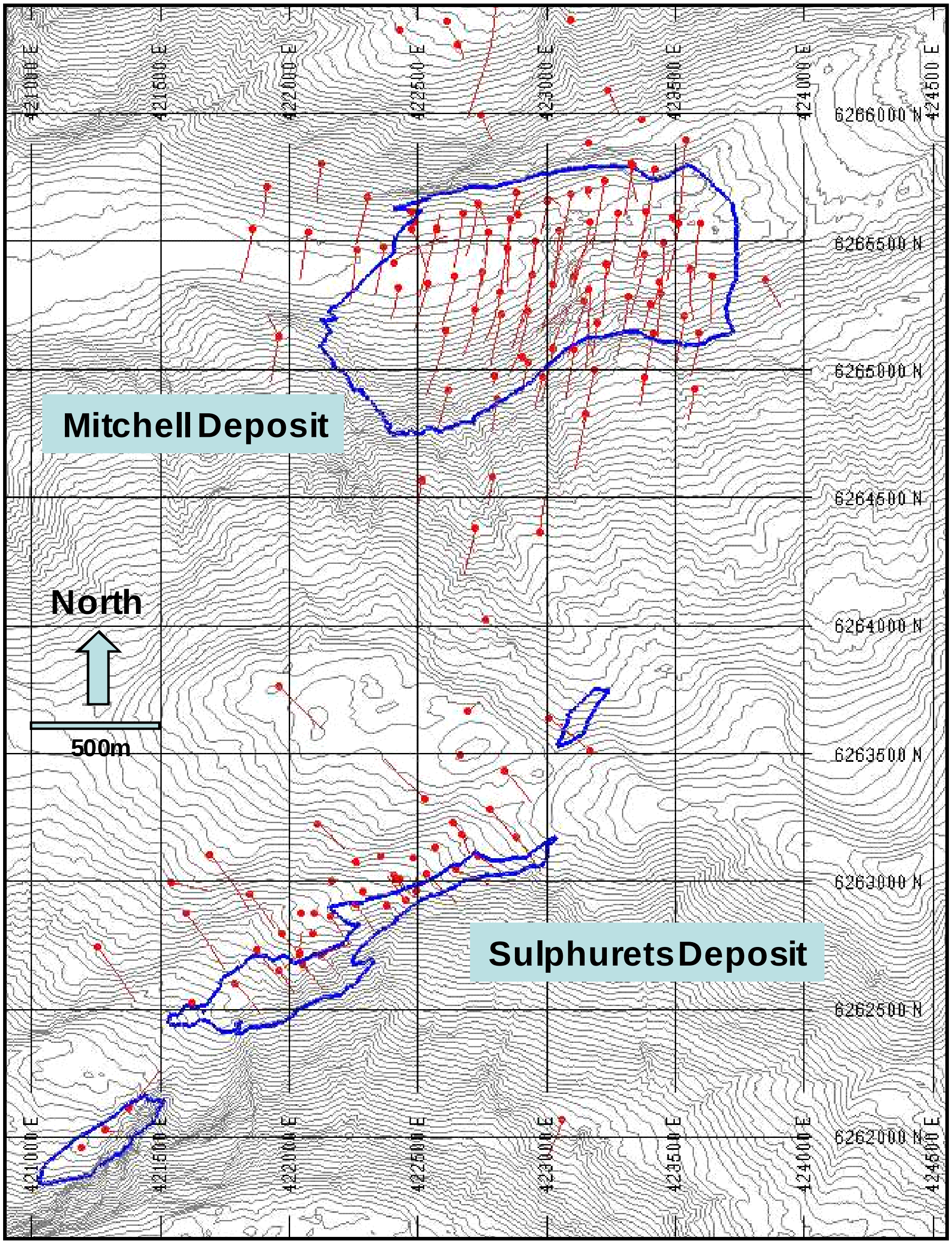


Figure 11.2 Sulphurets/Mitchell Drill Hole Locations



12.0 SAMPLING METHOD AND APPROACH

Seabridge implemented the same sampling methods in 2008 that were developed over the past several years. The following section was taken directly from RMI's NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia", dated April 6, 2008, which is available on SEDAR.

12.1 SAMPLE LENGTH

The 2007 drill core was sawn into primarily 2-m-long samples, which were then shipped off site where they were assayed for gold, copper, and other metals. Of the 7,686 samples that were collected, 7% were less than 2 m long, 90% were exactly 2 m long, and 3% were longer than 2 m. In 2007, approximately 99% of drilled meterage was assayed. The 37 holes that were drilled in 2007 averaged about 413 m in length. Drill hole M07-24E extended a 2006 hole (M-06-24) from a depth of about 356 m to nearly 600 m. After completing the 2007 drilling campaign, the Mitchell deposit has been drilled to roughly 125-m centres. The "core" portion of the deposit has been closed down to roughly 100-m centres over an area that measures approximately 1,400 m (east-west) by 500 m (north-south).

Based on the style of mineralization, it is RMI's opinion that the 2-m-long sample lengths are reasonable and appropriate.

12.2 DRILLING CONDITIONS

Drilling conditions were generally very good. Overburden was not excessive and rock quality was typically very high except in isolated fractured or sheared zones where the rock easily broke along foliation planes. Overall average RQD was about 74% and core recovery averaged about 95%. The frequency of natural breaks averaged about 9.6/m. Poorer recoveries were obtained in fractured and sheared zones especially in brittle, siliceous hornfelsed rocks at shallow depths, primarily in the periphery of the mineralized zone.

12.3 SAMPLE QUALITY

As a result of the strict adherence to the drilling procedures and sampling methodology described above, sample quality representation is considered high. There are no known negative drilling, sampling or recovery factors that could impact results.

12.4 GEOLOGY AND GEOLOGICAL CONTROLS

There has been some discussion regarding geology and controls at Mitchell Creek in previous sections. The deposit is considered to be within the spectrum of the gold-enriched copper porphyry environment and metals (chiefly gold and copper, in terms of economic value) are generally at low concentrations. Mineralization is typically finely disseminated, stockwork or sheeted veinlet controlled and pervasively dispersed over dimensions of hundreds of metres. Grades diminish slowly over large distances; sub-economic grades are encountered at distances of several hundreds of metres beyond the interpreted centre of the system.

Due to the intensity of hydrothermal alteration, especially at Mitchell Creek, it is difficult or impossible to recognize original protoliths. This is most pronounced in phyllic or quartz-sericite-pyrite altered rocks. In chlorite-sericite (logged as IARG or intermediate argillic) and propylitic altered rocks, a homogeneous, tuffaceous texture is often observed; thus, the host is likely intermediate volcanic tuffs or volcaniclastics. Diffuse, ghost-like porphyritic textures may reflect dykes of the Mitchell intrusions. Rare, metre-scale aphanitic intermediate dykes are post-alteration and unmineralized.

At Mitchell Creek, there appears to be a spatial association between the highest continuous copper and gold grades with an area of chlorite-magnetite alteration as recognized by Britton et al. where the rocks appear to be partially overprinted by phyllic alteration, particularly along the western edge of the intensely phyllic altered exposed bluffs, located at the east side of the zone. Roughly coincident with the area of highest Cu and Au mineralization are lower Mg and Na concentrations as determined by ICP analyses. These may be useful in defining domains for the purposes of resource estimation. There is no clear association with other recorded attributes including lithology, quartz vein frequency and intensity, or alteration types.

12.5 LITHOLOGICAL AND ALTERATION CODING

In 2006, Seabridge adopted lithological and alteration descriptions from Fowler and Wells (1995), which distinguished rocks above the STF from those below it. A similar distinction was made with the Mitchell Thrust Fault, where the rocks located between the Sulphurets and Mitchell Faults were seen to be comprised of similar lithologies as those located above the Sulphurets Fault. In 2007, Seabridge simplified the lithologic and alteration coding so that less emphasis was placed on the location of the samples relative to the regional structures and the more emphasis was placed on describing the samples. The lithologic and alteration codes stored in the 2007 drill hole database are summarized in Table 12.1 and Table 12.2, respectively. Other key logged attributes include a numerical alteration intensity from 0 (absent) to 6 (intense), percentage of quartz and pyrite, and quartz veinlet frequency.

Table 12.1 Lithologic Codes

Lithologic Code	Lithology
OVBD	Andesite
ANDS	Intermediate Volcanics, Massive Flows/Tuffs
IVOL	Andesite Lapilli Tuff
VALT	Andesite Tuff
VATF	Overburden
QTVN	Quartz Vein
PHBX	Hydrothermal Breccia
PSBX	Siliceous Hydrothermal Breccia
DDRT	Diorite/Mafic Intrusive
GRAN	Granitic Porphyry
PPFP	Feldspar Porphyry Intrusions
PQMZ	Quartz Monzonite
PMON	Porphyritic Monzonite
VAAT	Andesite Ash Tuff
VAXT	Andesite Crystal Tuff
VU	Volcanic, Unknown Protolith (Intensely Altered)
VUAT	Unknown Ash Tuff
VULT	Unknown Lapilli Tuff
VUTF	Unknown Tuff
VUXT	Unknown Crystal Tuff
SARG	Volcaniclastics/Argillites
SCHT	Schist, Unknown Protolith (Intensely Altered)
SEDS	Undifferentiated Seds
CCSD	Chert/Chemical Seds
SSLT	Siltstone
FLTZ	Fault Zone
NREC	No Recovery

Table 12.2 Alteration Codes

Alteration Code	Alteration Description
CARB	Carbonate Veining, Fault Related
CL	Chlorite Alteration
FEOX	Fe-Oxides Due to Weathering
HEM	Hematization of Intrusives
IARG	Intermediate Argillic – Green Ser, Chl, Py
KP	Potassic – K-Fd, Qt, Py, Cp (Porphyry)
PKBX	Potassic – K-Fd, Qt, Ser, Py, Cp (Hydrothermal Breccia)

table continues…

Alteration Code	Alteration Description
PR	Propylitic – Chl, Ep, Py, Carb, Mag
PSBX	Silica Flooding – Qt, Ser, Py, Tour, Py (Carb) (Hydrothermal Breccia)
QA	Albitic (Core Area) – Ab, Cb, Chl, Py, Cp, Ser (Porphyry)
QB	Potassic – Bio, Qt, Py, Cp (Chl, Ser, Mo) (Porphyry)
QSP	Phyllic – Qt, Ser, Tour, Py, Remnant Ks, Cp, Mo (Hydro. Breccia + Porphyhry)
QSPSTW	Phyllic – Qt, Ser, Py (>60% Quartz Veinlets)
QTVN	Late Quartz Veins
SI	Silica Flooding – Qt, Py, Cp (Tour, Ser) (Porphyry)
SIH	Silicification Due to Hornfelsing - Qt, Py
SIL	Pervasive Silicification

At Mitchell Creek, the IARG (intermediate argillic) alteration unit is more likely a transitional unit between propylitic and phyllic assemblages where chlorite has only been partially sericitized. Seabridge will try to verify by ongoing studies.

Table 12.3 contains a summary list of relevant composited gold and copper grades from Seabridge's 2008 drilling campaign. The samples shown in Table 12.3 were tabulated based on intervals in excess of 50 m in length that were continuously mineralized above a 0.5 g/t gold cutoff grade.

Table 12.3 Relevant 2008 Drill Hole Composite Grades

Drill Hole ID	Total Depth of Hole	From Depth (m)	To Depth (m)	Composite Length (m)	Au (g/t)	Cu (%)
M-08-61	748.00	273.00	599.30	326.30	0.71	0.13
M-08-61	748.00	640.00	690.00	50.00	0.44	0.11
M-08-62	945.00	32.00	100.75	68.75	0.65	0.14
M-08-62	945.00	313.00	571.13	258.13	0.54	0.22
M-08-62	945.00	572.02	745.00	172.98	0.55	0.20
M-08-63	660.00	290.00	569.00	279.00	0.66	0.13
M-08-64	501.00	23.00	345.00	322.00	0.84	0.17
M-08-65	657.00	4.00	380.10	376.10	0.96	0.29
M-08-65	657.00	384.55	482.00	97.45	0.54	0.19
M-08-65	657.00	488.00	592.60	104.60	0.59	0.19
M-08-66	435.00	94.00	435.00	341.00	0.66	0.11
M-08-67	938.40	78.00	372.00	294.00	0.64	0.26
M-08-67	938.40	384.00	469.00	85.00	0.48	0.62
M-08-67	938.40	471.00	714.00	243.00	0.69	0.30
M-08-69	645.00	1.20	79.00	77.80	0.85	0.24
M-08-69	645.00	81.00	586.00	505.00	0.78	0.22
M-08-70	381.00	172.00	225.00	53.00	0.68	0.24

table continues…

Drill Hole ID	Total Depth of Hole	From Depth (m)	To Depth (m)	Composite Length (m)	Au (g/t)	Cu (%)
M-08-70	381.00	251.00	320.00	69.00	0.39	0.17
M-08-71	333.00	99.00	154.00	55.00	0.66	0.13
M-08-72	327.00	80.50	139.00	58.50	0.63	0.11
M-08-73	617.80	91.00	374.00	283.00	0.78	0.20
M-08-73	617.80	390.00	442.00	52.00	0.47	0.16
M-08-76	470.00	9.58	238.78	229.20	0.98	0.25
M-08-76	470.00	320.09	408.00	87.91	0.49	0.14
M-08-77	390.00	15.00	133.40	118.40	0.91	0.26
M-08-77	390.00	135.05	271.00	135.95	0.65	0.19
M-08-79	399.00	313.00	399.00	86.00	0.19	0.45
M-08-86	810.00	43.74	120.00	76.26	0.71	0.17
M-08-86	810.00	200.00	306.00	106.00	0.22	1.22
M-08-86	810.00	336.00	544.14	208.14	0.89	0.28
M-08-86	810.00	545.12	718.00	172.88	0.60	0.23
M-08-90	597.00	2.50	169.90	167.40	0.96	0.28
M-08-90	597.00	173.24	597.00	423.76	0.55	0.23
M-08-91	684.20	124.00	408.00	284.00	0.67	0.16
M-08-92	453.00	53.00	314.00	261.00	0.68	0.18
M-08-92	453.00	316.00	418.00	102.00	0.73	0.20
M-08-93	753.00	119.00	645.00	526.00	0.65	0.19
M-08-94	450.00	2.50	195.00	192.50	0.80	0.23
M-08-94	450.00	197.00	339.00	142.00	0.56	0.17
S-08-08	588.00	274.00	344.20	70.20	0.89	0.46

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

This section describes Seabridge's sample security, sample preparation, and analytical methods that were used in 2008 for the Mitchell Creek deposit. These are essentially the same methods that were described in RMI's NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia", dated April 6, 2007, which is available on SEDAR.

13.1 STATEMENT ON SAMPLE PREPARATION PERSONNEL

All sample preparation was conducted by labourers contracted from Tahltan Native Development Corporation, trained by and under the direct supervision of geologists employed by Falconbridge and Seabridge.

13.2 SAMPLE PREPARATION AND DISPATCH

Upon completion of logging and sample demarcation, the core boxes were moved to the core cutting facilities in camp, usually the following day. The core cutting building is a 14' x 16' plywood platform, covered with a poly tarp on aluminum poles. The walls were left open to facilitate air circulation and prevent dust contamination. Three gasoline engine powered saws with 12" diamond toothed blades designed for rock cutting were utilized on day shifts only. The saws were mounted on secure wooden stands at waist height. The saw blades were cooled, cleaned, and lubricated with fresh, non-recirculated water during cutting. The saw operator placed uncut core boxes on tables adjacent to the saws and cut each piece of core sequentially within each marked sample interval. The assay half of the sample was placed in a heavy duty polythene bag and the other half was returned to the core box. Once a sample interval was completely sawn, the corresponding sample tag number was stapled to the inside at the top of the bag, and the bag was secured with staples. The sample number was also written on the bag with a permanent felt tip marker.

The bags were placed sequentially in rows on pallets or on the floor. Upon completion of a batch of 33 (see Section 13.4), the samples were placed into large polyweave (rice) shipping bags, 6 per bag. The polyweave bag was labelled with the project number, sample numbers, shipment number, and laboratory address, and then secured with plastic tie straps. In addition, for security purposes, the polyweave bag was also secured with a uniquely numbered tie strap and the number recorded on the retained copy of the sample transmittal form. The other copy of the sample transmittal form was placed in the last shipping bag of each batch. The bags were stored adjacent to the core cutting building or helicopter pad until a complete shipment was ready, which usually included several batches. During normal production and good weather, shipments were sent out at least every two days.

The sample shipment was placed inside the project-chartered helicopter or, on rare occasions, put inside a cargo net slung below the helicopter and flown directly to the Granduc Road staging area and unloaded. At the staging area, the shipment was either stored and locked inside a metal bulk shipping container until a truck contracted from Granmac Services Ltd. (Granmac) arrived from Stewart or transferred directly to a waiting truck. The truck was driven to Stewart where the samples were unloaded at the sample preparation facilities by Eco-Tech personnel. Occasionally the samples were taken directly to Stewart via helicopter and transferred to the preparation laboratory by truck contracted by Granmac. The preparation laboratory took an inventory of the shipment and confirmed that the numbered tie strap was not broken or tampered with. Eco-Tech then sent notification of the receipt of shipment with tie strap and sample numbers to Seabridge personnel at camp who confirmed the sample shipment.

13.3 ANALYTICAL PROCEDURES

At the facilities in Stewart, samples were sorted and dried (if necessary). The samples were crushed through a jaw crusher and cone or roll crusher to –10 mesh. The sample was split through a Jones riffle until a –250 g sub sample was achieved. The sub-sample was pulverized in a ring and puck pulverizer so that 95% of the material passed a -140 mesh screen. The sample was then rolled to homogenize it. The resulting pulp sample was then placed in a numbered paper envelope and securely packed in cardboard boxes. These boxes were shipped via Greyhound freight services to the Eco-Tech facilities located in Kamloops, BC.

At Eco-Tech’s laboratory in Kamloops, a 30 g sample size was split out from the pulp envelope and then fire assayed using appropriate fluxes. The resultant doré bead was parted and then digested with aqua regia followed by an atomic absorption (AA) finish using a Perkin Elmer AA instrument. The lower limit of detection for gold is 0.03 g/t or 0.001 oz/t. For other metals, a multi-element inductively coupled plasma (ICP) analysis was completed. For this procedure, a 0.5 g sample was digested with 3 mL of a 3:1:2 ($HCl: HN0_3:H_20$) that contains beryllium, which acts as an internal standard for 90 minutes in a water bath at 95°C. The sample was then diluted with 10 mL of water and analyzed on a Jarrell Ash CP unit. Eco-Tech’s ICP detection limits (lower and upper) are summarized in Table 13.1.

Table 13.1 ICP Detection Limits

Element	Lower	Upper
Ag	0.2 ppm	0.0 ppm
Al	0.01%	10.00%
As	5 ppm	10,000 ppm
Ba	5 ppm	10,000 ppm
Bi	5 ppm	10,000 ppm
Ca	0.01%	10.00%
Cd	1 ppm	10,000 ppm
Co	1 ppm	10,000 ppm
Cr	1 ppm	10,000 ppm
Cu	1 ppm	10,000 ppm
Fe	0.01%	10.00%
La	10 ppm	10,000 ppm
Mg	0.01%	10.00%
Mn	1 ppm	10,000 ppm

Element	Lower	Upper
Mo	1 ppm	10,000 ppm
Na	0.01%	10.00%
Ni	1 ppm	10,000 ppm
P	10 ppm	10,000 ppm
Pb	2 ppm	10,000 ppm
Sb	5 ppm	10,000 ppm
Sn	20 ppm	10,000 ppm
Sr	1 ppm	10,000 ppm
Ti	0.01%	10.00%
U	10 ppm	10,000 ppm
V	1 ppm	10,000 ppm
Y	1 ppm	10,000 ppm
Zn	1 ppm	10,000 ppm

Assay results were then collated by computer and were printed along with accompanying internal quality control data (repeats and standards). Results were printed on a laser printer and were faxed and/or mailed to appropriate Seabridge personnel. Appropriate standards and repeat samples were included on the data sheet.

13.4 QUALITY CONTROL MEASURES

Seabridge essentially implemented the same quality control procedures that they used for their 2006 Mitchell program. Different standard reference material (SRM) sources were used in 2008 versus 2007. They included blank material obtained from landscaping materials (marble) and "barren" river rocks collected near Stewart, BC, along with different commercially certified standards. Assay quality control measures included the insertion of a sample blank and pulp standard within each laboratory batch of approximately 35 samples. Thus a complete batch contained a minimum of one blank and one pulp standard, with the remainder being core samples. The blank and pulp standard were numbered using the same number sequence that was used for the core samples and inserted into each batch shipment at the completion of sawing the core samples for a given batch.

Blanks were obtained from a local tire/hardware store and consisted of unmineralized marble landscaping materials. Later in the drill campaign barren river rocks were collected near Stewart and screened to remove fines and oversize to produce nominal 1" diameter pieces, which were submitted into the sample stream.

The pulp standards that were used by Seabridge for their 2008 drilling/sampling campaign were purchased from CDN Resource Laboratories Ltd. (CDN) out of Delta, BC. Three CDN standards (CDN-GS-12, CDN-GS-13, CDN-GS-18, and CDN-CM-1) were prepared from material that was collected from the Casino copper-gold-molybdenum porphyry property that is located in northern BC. These standards have certified gold and copper values that are definitely relative to the type and tenor of mineralization that has been identified at the Mitchell deposit. In RMI's opinion, these standards are appropriate and reasonable.

In addition to the insertion of control samples with each batch, Seabridge also submitted duplicate core samples by sawing one half of the drill core into two ¼ core splits that were submitted as individual samples to Eco-Tech. Approximately 135 ¼ core duplicates (or about 2% of the total samples) were submitted to Eco-Tech in 2008. About 10% of the 2008 samples (877 samples) that were assayed by Eco-Tech were re-assayed as same pulp "cross-checks" by ALS Chemex Laboratories Ltd. (ALS Chemex) of North Vancouver, BC.

For gold, both Eco-Tech and ALS Chemex employed the same assay preparation and measurement technique. For other metals, the cross-checks compared Eco-Tech ICP analyses with ALS Chemex ore grade/AAS finish analyses. Both methods utilized a triple acid digestion. For finely disseminated, low grade base metal mineralization similar to that which occurs at the Mitchell deposit, the ICP analyses are generally considered to be as reliable as (or more reliable than) ore grade/AAS finish analyses.

Quality assurance/quality control (QA/QC) results for Seabridge's 2008 Mitchell Creek drilling program are discussed in Section 14.0. This includes performance of blanks, standards, duplicates, and a comparison of Eco-Tech assays against ALS Chemex.

13.5 AUTHOR'S OPINION

It is RMI's opinion that the Seabridge's sample preparation, analytical methods, and sample security measures met or exceeded currently accepted industry practices.

14.0 DATA VERIFICATION

Previous NI 43-101 Technical Reports by RMI have discussed various data verification measures that were undertaken by RMI for the Kerr, Sulphurets, and Mitchell properties. This section describes the procedures and results of RMI's database verification procedures used for Seabridge's 2008 data.

14.1 ELECTRONIC DATABASE VERIFICATION

RMI performed an audit of the Mitchell drill hole database by comparing Eco-Tech certified copper and gold assay results with values stored in Seabridge's electronic database. RMI manually checked gold and copper assays from five of Seabridge's 2008 drill holes for verification. The data that were verified are summarized in Table 14.1. RMI discovered that the gold assays for a portion of one drill hole (M-08-94) were incorrectly loaded to the AcQuire database. The gold grade entries for 33 assays were inadvertently imported from an 'ounce per ton' column instead of a 'gram per tonne' column. The data shown in Table 14.1 represent about 15% of the 2008 Seabridge assay data.

Table 14.1 2008 Database Verification

Drill Hole	Number Checked	Metres Checked	Au Errors	Cu Errors
M-08-65	339	653.0	0	0
M-08-72	164	311.8	0	0
M-08-80	229	454.7	0	0
M-08-94	229	447.5	33	0
S-08-07	291	554.2	0	0
Grand Total	**1,252**	**2,421.2**	**33**	**0**

It is RMI's opinion that the Mitchell Creek electronic database that was used to estimate mineral resources that are subject to this report is accurate despite the aforementioned data importation problem. This is based on RMI's own independent comparison of certified assays and the database.

14.2 QUALITY ASSURANCE/QUALITY CONTROL PROTOCOLS

As was discussed in Section 13.0, Seabridge obtained several certified SRMs from CDN. The SRMs were initially prepared and certified by CDN from gold-copper porphyry material obtained from the Casino property located in northern BC.

Approximately 265 SRMs were submitted to Seabridge's primary laboratory (Eco-Tech) as a part of their QA/QC program.

Approximately 262 barren samples or "blanks" were submitted to Eco-Tech, which were obtained from landscaping materials and local stream boulders. About 2% of the samples that were submitted in 2008 had a companion "duplicate" that was derived from ¼ core splits of the drill core and represent field duplicates.

In addition to submitting control samples as a part of their QA/QC program, Seabridge also sent about 877 Eco-Tech pulps to ALS Chemex in Vancouver for check assay purposes. Those pulps were assayed for gold, copper, silver, and several other metals.

Blanks and SRMs were submitted into the sample stream at a frequency of roughly one each per every 35 samples. The commercially prepared SRMs were submitted in their original pouches and therefore were not "blind" to the laboratories.

RMI obtained the raw data that were generated from Seabridge's 2008 QA/QC program and independently reviewed the results. Figure 14.1 and Figure 14.2 plot gold and copper assay results from the landscaping marble (Blank #1) and local stream boulders (Blank #2), respectively, that were submitted to Eco-Tech.

As can be seen in Figure 14.1, there were no "failures" for gold and only some nominal "noise" in copper with the landscaping marble. Similar relationships are seen with Blank #2 (local stream boulders) except the boulders have a higher background value for copper than the marble blanks.

Figure 14.3 through to Figure 14.8 show Eco-Tech's performance for the three commercial gold/copper standards that were submitted by Seabridge in 2008. These graphs plot the value obtained from Eco-Tech as a function of time or job number along the X-axis. Plus and minus 1 and 2 standard deviation lines are also posted on the graphs along with the certified expected value.

Figure 14.3 plots the performance of the CDN standard CGS-13 for gold. This SRM has a certified expected value of 1.01 g/t. As can be seen in Figure 14.3, there are three apparent failures (values less than 2 standard deviations of the expected value). Seabridge personnel investigated these apparent failures and found that a technician had inadvertently provided the wrong standard label (i.e. switched labels).

The performance of the CDN CGS-13 copper standard is shown in Figure 14.4.

Figure 14.1 2008 Blank #1 Performance

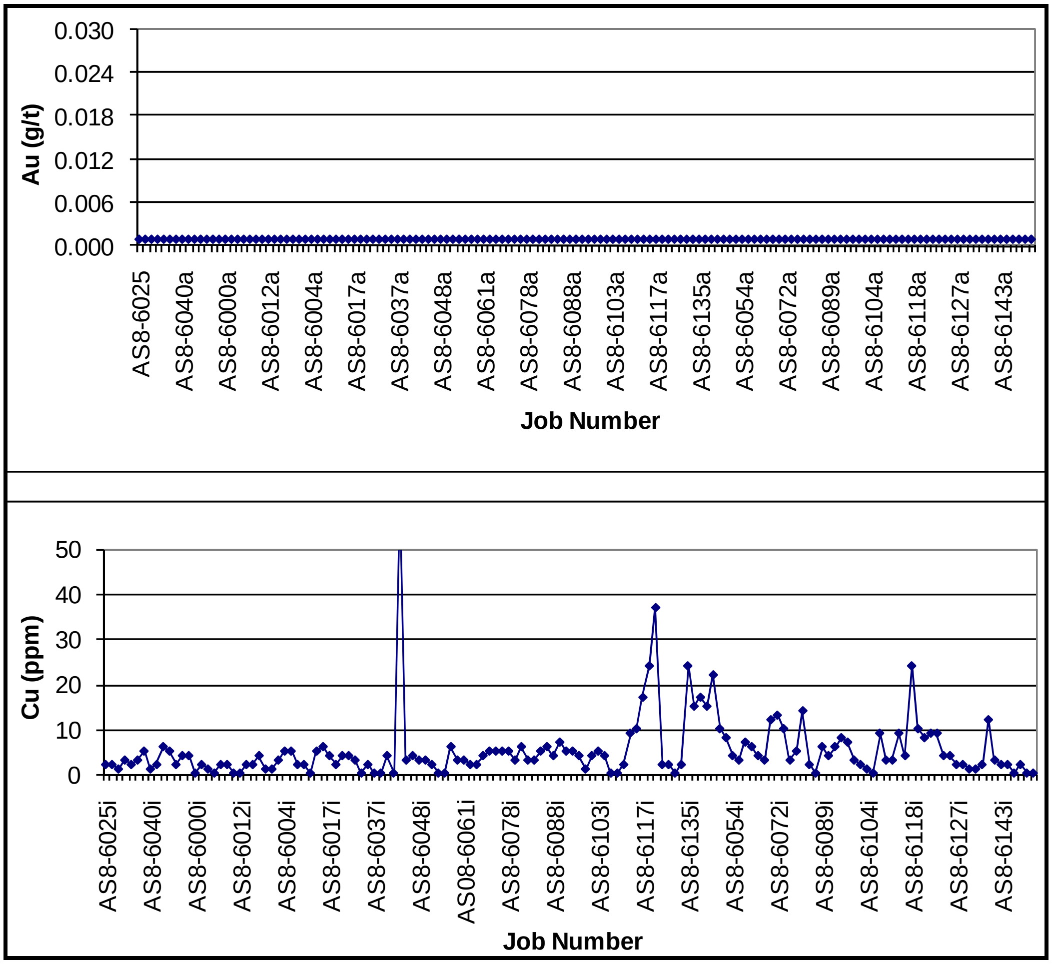
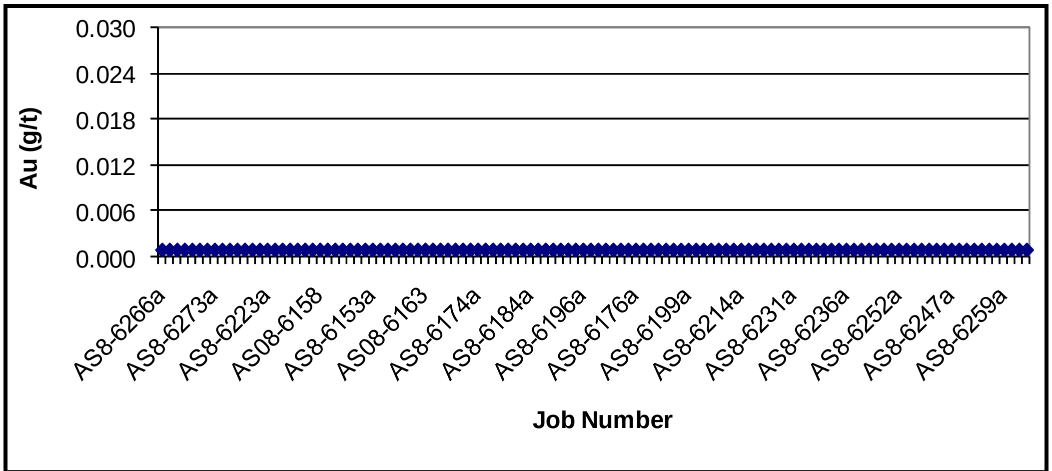


Figure 14.2 2008 Blank #2 Performance



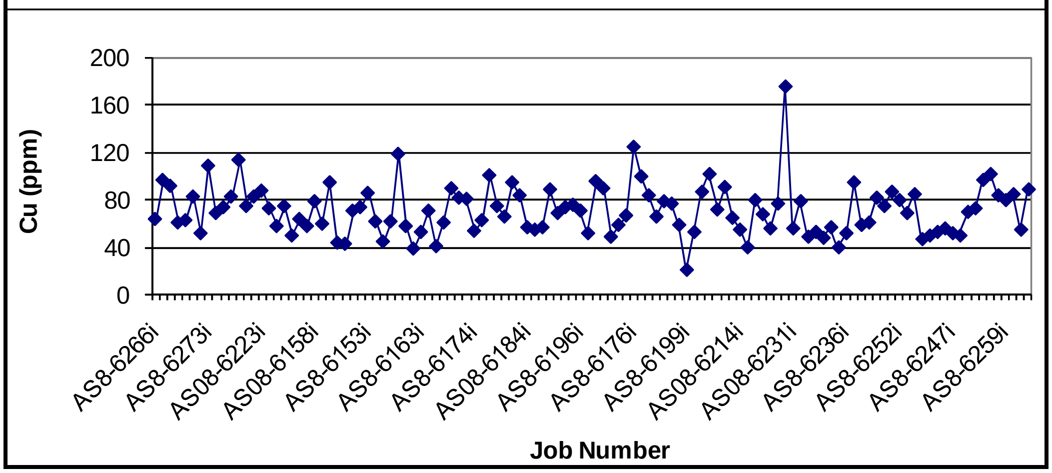
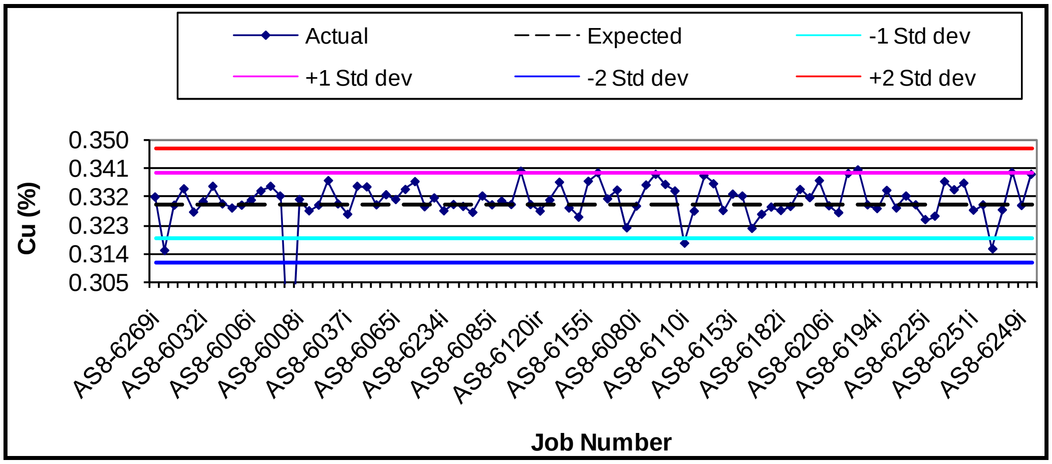


Figure 14.3 2008 Au Standard CGS-13 Results

Figure 14.4 2008 Cu Standard CGS-13 Results



As can be seen in Figure 14.4, the CDN CGS-13 SRM assayed slightly higher than the expected value but generally well within the prescribed tolerances. The apparent failure is attributed to SRM sample switching. The performance of the gold and copper CDN CGS-18 standards is shown in Figure 14.5 and Figure 14.6, respectively.

Figure 14.5 2008 Au Standard CGS-18 Results

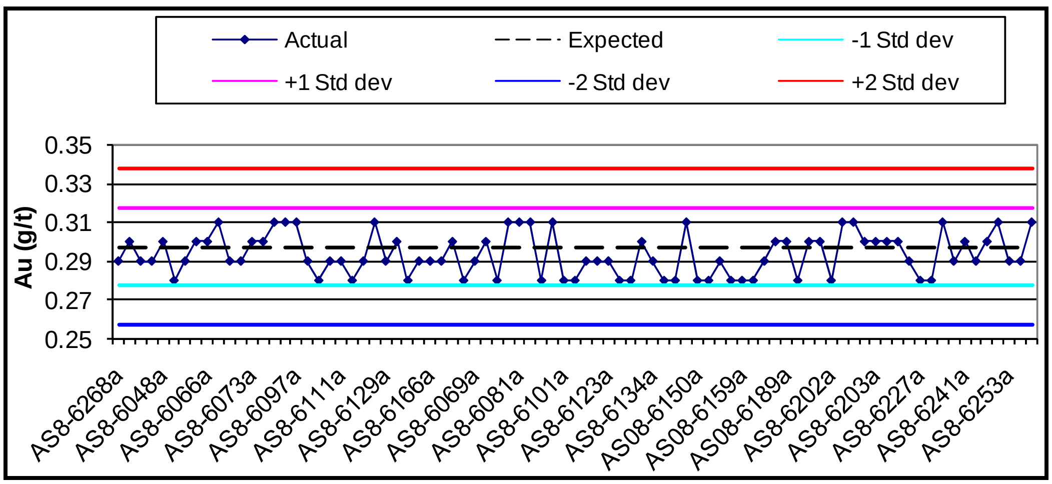
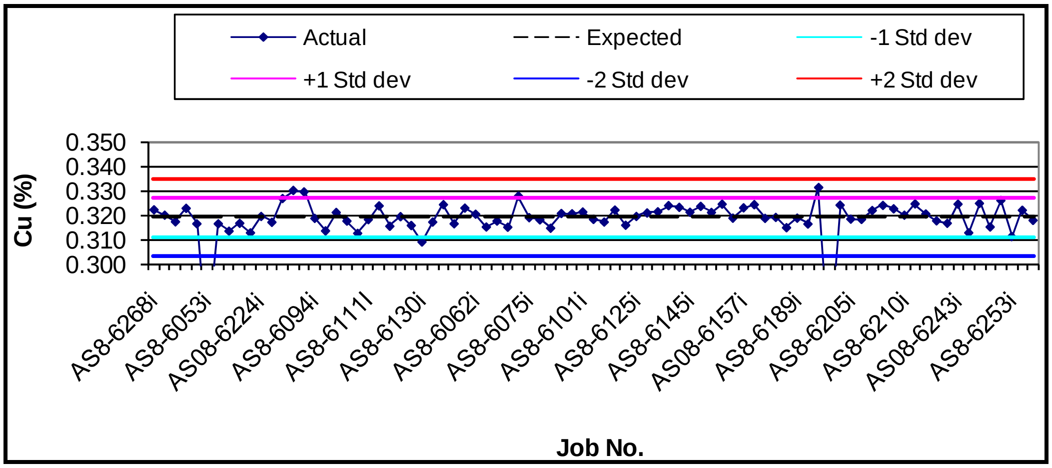


Figure 14.6 2008 Cu Standard CGS-18 Results



The Eco-Tech assay results for CDN CGS-13 gold and copper SRMs show that the laboratory generally returned reasonable values. The “failures” have been interpreted to be switched SRMs. The performance of the gold and copper CDN CM-1 standards is shown in Figure 14.7 and Figure 14.8, respectively.

Figure 14.7 2008 Au Standard CM-1 Results

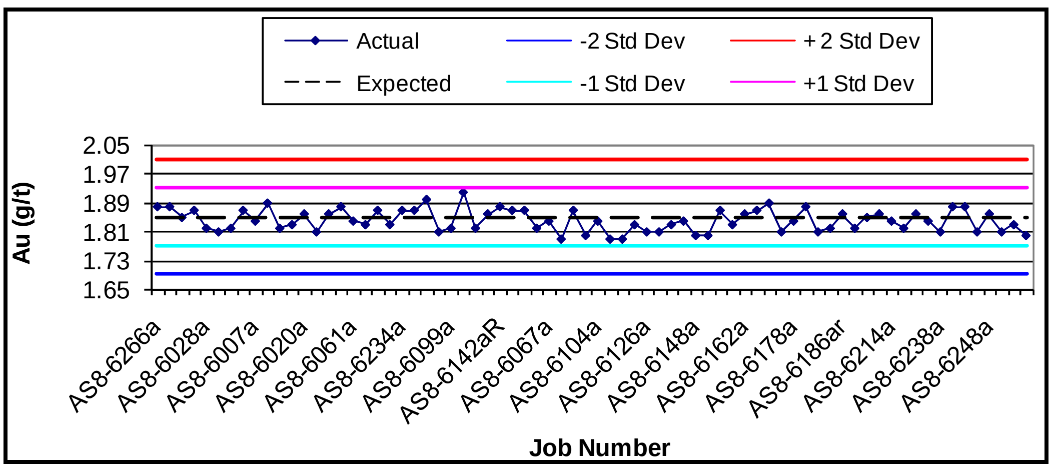
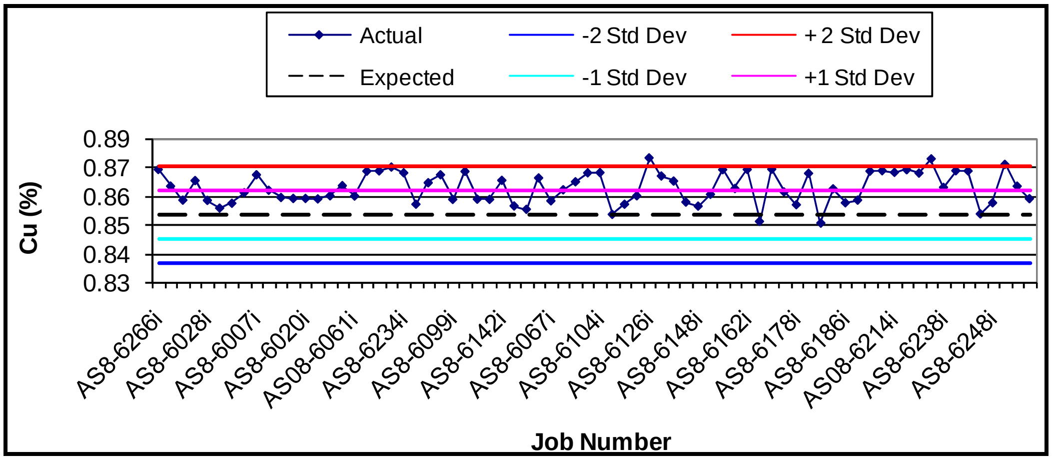


Figure 14.8 2008 Cu Standard CM-1 Results



The Eco-Tech laboratory did an excellent job assaying the CDN CM-1 gold SRM, while the copper SRM values tended to be somewhat higher than the expected value but were usually less than plus 2 standard deviations.

The Eco-Tech assay results for the ¼ split core samples for gold and copper are shown in Figure 14.9 and Figure 14.10, respectively. These figures contain quantile-quantile (QQ) plots that compare the original and field duplicate results against one another. There was a 3% difference in mean grade for the duplicate gold samples and a 0.3% difference in mean copper grade. In RMI's opinion, these results are reasonable.

In 2008, Seabridge submitted about 877 Eco-Tech pulps to ALS Chemex in Vancouver (786 from Mitchell and 91 from Sulphurets). Figure 14.9 and Figure 14.10 are QQ plots that compare the original Eco-Tech assays (X-axis) with the values obtained from ALS Chemex (Y-axis) for the Mitchell deposit only. The 91 pulps from the 2008 drilling at Sulphurets that were re-assayed by ALS Chemex showed similar relationships (i.e. good to excellent correlation).

Figure 14.9 2008 Eco-Tech Duplicate Core Gold Results

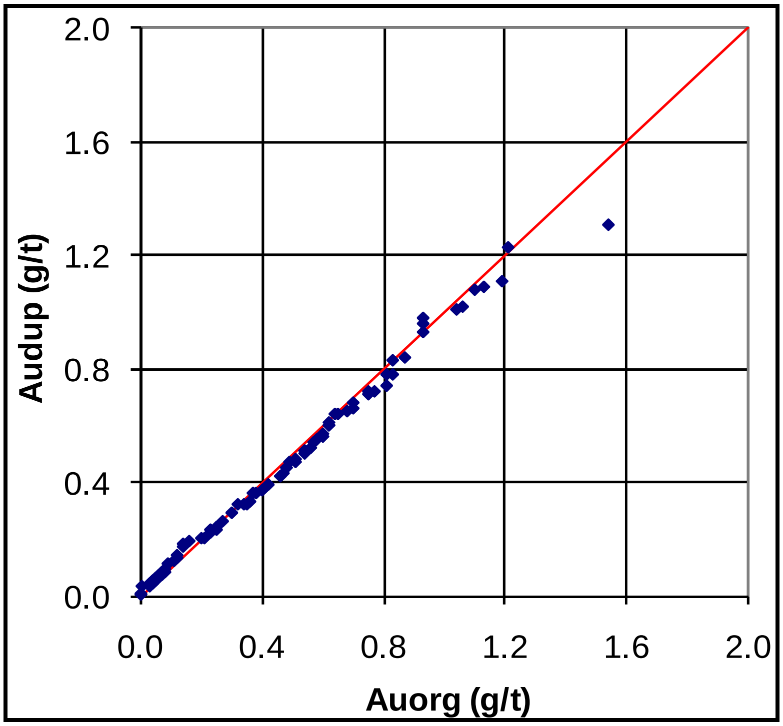


Figure 14.10 2008 Eco-Tech Duplicate Core Copper Results

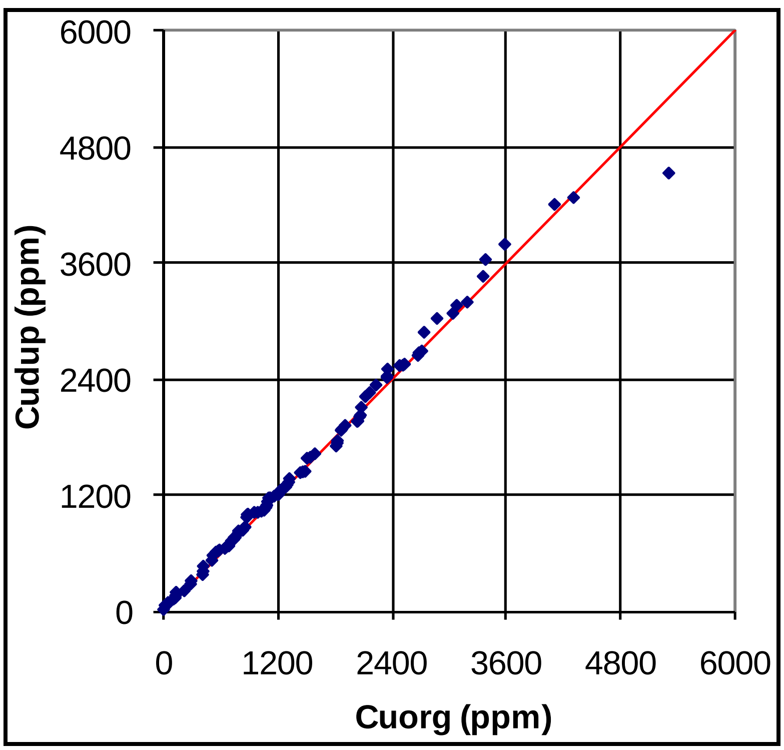


Figure 14.11 2008 Eco-Tech vs. ALS Chemex Au QQ Plot

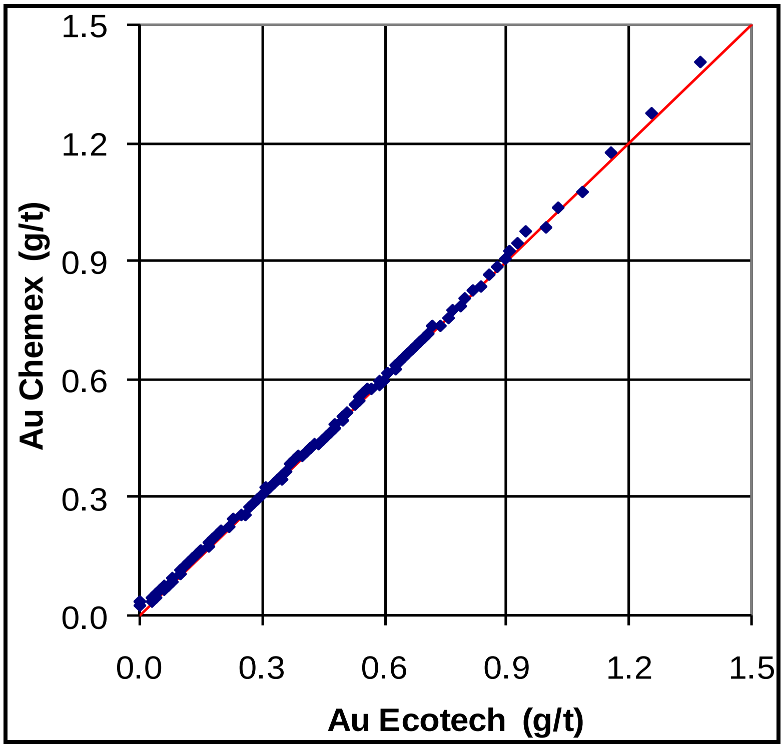
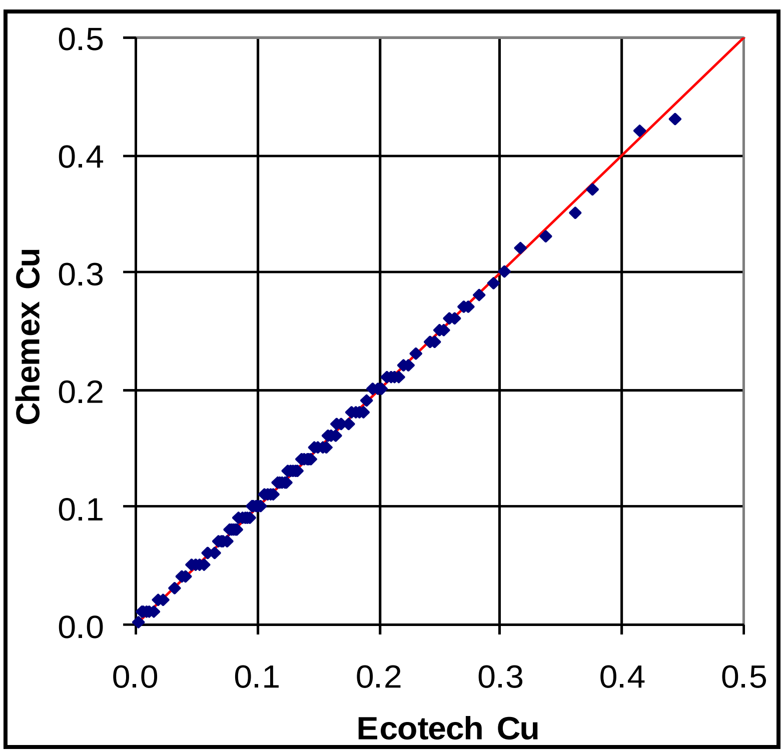


Figure 14.12 2008 Eco-Tech vs. ALS Chemex Cu QQ Plot



As can be seen in Figure 14.11 and Figure 14.12, there is a very close comparison between the original laboratory (Eco-Tech) and the secondary laboratory (ALS Chemex).

It is the opinion of RMI that the 2008 Seabridge gold and copper assays grades are reasonable and supported by satisfactory QA/QC results. In addition, check assay data by a secondary laboratory demonstrates that the original Eco-Tech assays are reasonable.

14.3 TOPOGRAPHIC CONTOUR DATA

Previous topographic base maps available for the project were obtained from BC government TRIM data, which has a contour interval of 20 m. Since the accuracy of this data is not suitable for the level of accuracy required for economic evaluation of mining projects, a new topographic base map was prepared in late 2008.

McElhanney of Vancouver, B.C. was contracted to perform an aerial survey and to provide Seabridge with an updated accurate topographic base map of the three deposits and surrounding area. The data were obtained from a helicopter-borne LiDAR survey undertaken by McElhanney. LiDAR is an optical remote sensing technology that measures properties of scattered light to find range and other information of a distant target. McElhanney's system uses the Leica ALS50-II Airborne Laser Scanner. This uses a Multiple Pulse in Air (MPiA) system, which is a light-based measuring system that emits photons by laser. LiDAR collects topographical data using laser range and return signal intensity data recorded in-flight. The Leica ALS50 system can yield details under tree cover and orthorectify imagery using specialized software. The product provided included gridded bare earth data to 2 m spacing and contours at 1 m intervals in digital formats.

The new topographic map of the district was provided to Seabridge in the UTM NAD83 coordinate system, which is the standard system for all BC government and industry mapping applications. The KSM drill hole database up to this point has been using the UTM NAD27 coordinate system. Seabridge contracted Aero Geometrics of Vancouver to translate the KSM drill hole collar locations from NAD27 to NAD83 datum. Geometrics used MAPS3D software to perform the transformation of all collar coordinates. This software, a product of Seirra Systems, uses the Canadian National Transformation Version 1.1 and 2.0 for the transformation.

The new topographic surface is much more detailed than previous surfaces and will allow for more accurate estimates of tonnages to be made. RMI notes that there are still some local variations between the elevation of the topographic surface and the newly translated drill hole collar locations.

These differences are the result of several factors, such as:

- no transform of the Z-coordinate with Canadian National Transformation software
- the inaccuracy of the initial GPS collar elevation
- holes surveyed below the drill deck and not ground level ("stick-up")
- differences magnified by steep terrain.

RMI is recommending that a detailed DGPS survey be completed by Seabridge of all locatable drill hole locations so that the Z-coordinate differences can be better understood. In addition, Seabridge should ensure that all future drill hole collars are surveyed at ground level to minimize "collar stick-up".

14.4 SPECIFIC GRAVITY DATA

For the Kerr deposit, Placer Dome performed 1,366 bulk density determinations by weighing selected pieces of drill core in air and water using a triple beam balance from which the density calculation was made (i.e. weight in air / weight in air - weight in water). RMI does not know if the samples were completely dried or whether the samples were waxed prior to submersion in water. RMI examined these determinations by lithology, alteration, copper/gold grades, and depth. There was very little difference in the mean density by those attributes so RMI decided that the mean bulk density value of 2.84 g/cm^3 was reasonable. That value was used in calculating resource tonnes.

Seabridge personnel collected 18 determinations from their 2008 Sulphurets drilling program. These values were combined with 337 prior determinations that were collected by Placer Dome in the early 1990s. RMI then performed an analysis of all combined bulk density determinations. Like Kerr, there was not a significant difference in bulk density values by lithology or alteration. An average bulk density value of 2.77 g/cm^3 was chosen by RMI for calculating tonnes for the Sulphurets deposit.

A total of 770 bulk density determinations have been collected by Seabridge from their 2006 through 2008 Mitchell drilling programs. RMI closely compared these determinations by lithology, alteration, grade, depth, and location relative to the Mitchell thrust fault. Based on those analyses, RMI and Seabridge elected to assign bulk density values by several factors including lithology, alteration, and fault domain. Table 14.2 summarizes the bulk density values used in the Mitchell resource area.

Table 14.2 Mitchell Valley Bulk Density Values

Geologic Unit	Bulk Density (g/cm^3)
Overburden	2.00
Glacial ice	0.90
Chlorite-Propylytic Alteration	2.74
Quartz-Sericite-Pyrite Alteration	2.79
Upper Plate Rocks	2.71
Lower Plate Rocks	2.77

RMI recommends that Seabridge continue to collect additional representative samples from the Sulphurets deposit so that a more accurate density measurement can be obtained.

15.0 ADJACENT PROPERTIES

This section has been taken from the RMI report entitled "Updated KSM Mineral Resources" dated March 30, 2009, which is available on SEDAR.

Silver Standard Resources Inc. has recently announced an updated estimate of mineral resources for their Snowfields project, which is located immediately east of Seabridge's Mitchell deposit. Table 15.1 summarizes the publicly disclosed resources of the Snowfield project (P&E Mining Consultants Inc., 2009), which were tabulated using a 0.50 g/t gold equivalent cutoff grade.

Table 15.1 Snowfield Mineral Resources

Resource Category	Mt	Au (g/t)	Ag (g/t)	Cu (%)	Mo (%)	Au (000 oz)	Ag (000 oz)
Measured	31.9	1.49	1.43	0.033	0.014	1,524	1,470
Indicated	102.8	0.86	1.58	0.072	0.011	2,834	5,205
Measured + Indicated	134.7	1.01	1.54	0.063	0.012	4,362	6,675
Inferred	661.8	0.67	1.83	0.137	0.008	14,276	39,000

It appears that the mineralization along the eastern end of the Mitchell deposit is spatially and genetically related to the mineralization within a portion of the adjacent Snowfield deposit.

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

16.1 METALLURGICAL TEST REVIEW

The KSM Project includes three mineralized areas, identified as the Kerr Zone, the Sulphurets Zone, and the Mitchell Zone. The zones contain a significant copper/gold/molybdenum resource.

Several metallurgical test programs were carried out to assess the metallurgical response of these mineralizations. The most recent test programs were performed in 2007 and 2008. The following sections will separately summarize the historical test work and recent KSM test programs.

16.1.1 HISTORICAL TEST WORK

Wardrop received several historical test work reports from Seabridge. The historical test work was conducted by the following laboratories:

- Coastech Research Inc. (Coastech Research), 1989
- Metallurgical Laboratory, Brenda Mines, 1989
- Research Centre, Placer Dome, 1990
- Research Centre, Placer Dome, 1991.

The test work includes the preliminary investigations of mineralogy, mineralized material hardness, and the metallurgical response to flotation process and cyanidation.

TEST SAMPLES

Several different samples have been tested in the historical test programs.

Coastech Research – 1989

Two types of samples from the Kerr mineralized zone were tested in the program — one representing the central high grade zone (High Grade) and the other representing the rest of the mineralized zones (Low Grade). The assayed and calculated grade values are shown in Table 16.1.

Table 16.1 Test Samples – Coastech Research, 1989

Sample	Au (g/t)	Ag (g/t)	Cu (%)
Low Grade			
Assay	0.55	-	0.68
High Grade			
Assay	0.44	2.74	1.05

Metallurgical Laboratory, Brenda Mines – 1989

Sample 106 was tested in this program, along with a sample from Brenda Mines. No sample description was included in the report.

Research Centre, Placer Dome – 1990

Four composites were made up from a total of 560 individual samples of crushed drill core rejects weighing 2.3 t. They were labelled as Composites K-1 to K-4. Two further composites which were received from Coastech Research were also tested. These two composites were labelled as LG-01 and HG-01 for low grade and high grade samples, respectively.

The high grade sample was considered to be representative of the central higher grade zone and the low grade sample representative of the remainder of the copper mineralization. Table 16.2 shows the head grade of the samples.

Table 16.2 Test Samples – Placer Dome, 1990

Composite	Au (g/t)	Ag (g/t)	Cu (%)
K-1	0.26	1.0	0.52
K-2	0.32	1.1	0.59
K-3	0.29	0.9	0.40
K-4	0.44	3.0	1.30
LG-01	0.39	2.2	0.71
HG-01	0.36	2.3	1.03

Research Centre, Placer Dome – 1991

The bulk samples identified as Rubble Zone Trench and Crackle Breccia Zone Trench were employed for the testing program. Exploration personnel from Placer Dome collected the bulk samples. The average gold, silver, and copper values are shown in Table 16.3.

Table 16.3 Test Samples – Placer Dome, 1991

	Au (g/t)	Ag (g/t)	Cu (%)
Rubble Zone Composite	1.21	2.57	0.78
Crackle Breccia	0.34	1.58	0.40

MINERAL SAMPLE CHARACTERISTICS

Mineralogy

In 1990, Placer Dome examined mineralogical characteristics on the K-1 to K-4 composites and the results are summarized in Table 16.4.

Table 16.4 Mineralogical Characteristics – Placer Dome, 1990

Composite	Description
K-1	Sericite/chlorite and silicified tuffaceous rocks
K-2	Rubble zone - quartz/sericite/felsic/volcaniclastic sequence
K-3	Sericite volcaniclastic sequence c/w stockwork and veining
K-4	Quartz-sulphide veins and lenses - high grade

The determination also indicated that the iron and sulphur contents of the 4 samples varied in a narrow range, from 6.7 to 7.2% for iron and from 5.7 to 8% for sulphur.

Grindability

In 1989, using a comparative method, Brenda Mines determined the work index of Sample 106 to be 13.52 kWh/t, which was much lower than the work index of 19.78 kWh/t measured from the Brenda ore.

In 1990, Placer Dome calculated comparative ball mill work indices on Composites K-1 to K-4 and Composites LG-01 and HG-01. The comparative work index increased with product particle size. No significant difference in grindability among the composites was determined. The obtained work indices were low as well, ranging from 7.4 kWh/t at a coarse product of 80% passing 205 µm (Composite K-4) to 12.8 kWh/t at a fine product particle size of 80% passing 45 µm (Composite K-3).

Similar grindability tests were conducted on the 1991 samples by Placer Dome. The comparative grinding work index of the Rubble Zone composite was similar to the data obtained from the 1990s samples. However, the comparative grinding index from the Crackle Breccia composite was much lower, ranging from 6.4 to 8.0 kWh/t, indicating a softer material.

Specific Gravity

Results of bulk and dry specific gravities conducted by Placer Dome in 1990 and 1991 are summarized in the Table 16.5. The average specific gravity (SG) and bulk SG are 2.89 and 2.82, respectively.

Table 16.5 SG Determination Results

Sample	SG	Bulk SG
K-1	2.94	-
K-2	2.90	-
K-3	2.96	-
K-4	2.90	-
HG-01	2.92	-
LG-01	2.88	-
Rubble Zone	2.83	3.00
Crackle Breccia	2.82	2.63
Average	**2.89**	**2.82**

FLOTATION

Metallurgical Laboratory, Brenda Mines – 1989

The test program preliminarily studied the responses of the Sample 106 to conventional copper and gold flotation. Open circuit cleaning tests failed to provide a marketable grade copper concentrate due to the coarse primary grind.

The test work showed that high copper and gold recoveries could be obtained with a grind of 75% minus 200 mesh. However, to obtain the required concentrate grade it was necessary to depress iron sulphides. Depression of the iron sulphides with cyanide and pH control was shown to be possible; however, iron depression was very sensitive to the dosage of sodium cyanide. Small amounts of cyanide improved rougher concentrate grades and avoided precious metal losses in subsequent cleaning steps. The test results suggested using a selective xanthate collector for copper recovery and a dithiophosphate collector for gold recovery.

Research Centre, Placer Dome – 1990

High copper and precious metal recoveries were achieved in all tests. Saleable copper concentrates were produced in four of the six composites tested. Approximately half of the gold and silver reported to the final copper concentrate. A feed particle size of 80% passing 140 μm and a total of 16 minutes of flotation time were required to provide the metallurgical recoveries.

In the tests, lime and cyanide were added to depress iron sulphides. Sodium ethyl xanthate (R325) and Aerofloat 208 were added as copper and gold collectors. MIBC was added as frother. Rougher flotation was performed at pH 10.5.

The rougher concentrate was reground and the reground concentrate was cleaned in three stages of open circuit. The pH of cleaner flotation was adjusted to 11.

Primary grinds were evaluated from a coarse grinding size of 80% passing 140 to 175 µm to a fine grinding size of 80% passing 35 to 45 µm. It was found that the highest metal recoveries were obtained from the finest feed material.

Total copper recoveries for rougher and scavenger flotation ranged from 89 to 96%. Gold recoveries varied from 67 to 94% and silver from 81 to 95%.

The various samples showed differing metallurgical upgrading responses to the test conditions. Although regrinding and cleaning of the rougher concentrate at pH 11 rejected a significant amount of pyrite, Composite K-1 and K-2 produced poor results. The report indicated that the poor response was possibly due to the presence of sericite and mica slimes. It was recommended that sodium silicate or glue be added to the rougher floatation to suppress these minerals.

Gold recovery in the third cleaner concentrate was approximately 50% on average.

Research Centre, Placer Dome – 1991

The test program confirmed the earlier flotation test results that had been conducted in 1990. High final copper concentrate grades were produced from both composites.

Four grind and flotation tests were performed on each of the two mineral samples. The test results are summarized in Table 16.6.

Table 16.6 Flotation Test Results – Placer Dome, 1991

	Rubble Zone				Crackle Breccia			
Composite Test	**A**	**B**	**C**	**D**	**A**	**B**	**C**	**D**
Primary Grind								
- 80% passing (P_{80}), µm	223	175	149	98	165	110	99	59
Final Concentrate								
Grade	1.5	1.9	1.8	2.5	0.6	1.0	0.6	1.3
- Cu (%)	32.0	30.4	32.3	28.2	30.9	29.9	33.2	26.1
- Au (g/t)	30.5	26.8	27.4	25.5	12.8	9.3	15.0	9.2

table continues…

Composite Test	Rubble Zone A	Rubble Zone B	Rubble Zone C	Rubble Zone D	Crackle Breccia A	Crackle Breccia B	Crackle Breccia C	Crackle Breccia D
Recovery								
- Cu (%)	62.5	76.4	74.2	86.7	50.1	73.0	51.2	82.5
- Au (%)	41.4	44.2	40.4	48.5	23.1	29.6	26.0	35.7
Rougher/Scavenger Concentrate								
Recovery (%) weight	6.8	10.5	7.4	12.5	7.1	10.8	10.2	14.7
Recovery (%) Cu	73.3	86.1	89.3	96.6	73.9	83.6	87.1	93.1
Recovery (%) Au	61.1	74.7	68.5	79.8	51.1	56.8	63.9	66.4

The results indicated that copper and gold recoveries increased with an increase in primary grinding fineness. The finest primary grinds produced the best overall recoveries of copper and gold. The copper grades in the final concentrate grades ranged from 28 to 32% for the Rubble Zone sample and from 26 to 33% for the Crackle Breccia sample.

The gold and silver assay of the solutions from the rougher/scavenger tailing showed that the use of minor quantities of sodium cyanide in the flotation circuit for pyrite depression did not dissolve significant amounts of precious metals.

16.1.2 2007 Test Work

G&T Metallurgical Services Ltd. (G&T) conducted a metallurgical test program in 2007 that included material hardness determinations and flotation and cyanidation testing on three composite samples. These samples were collected from the Mitchell Zone and were of similar chemical and mineralogical composition. Table 16.7 shows the composition of the samples.

Table 16.7 G&T Test Samples Compositions

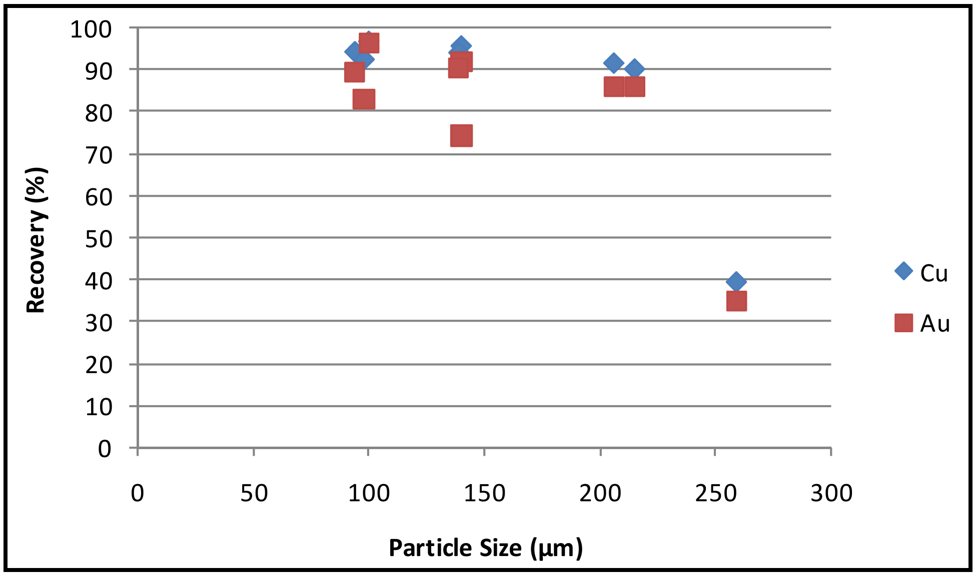
	Units	Composite A	Composite B	Composite C	Composite Average
Element					
Copper	%	0.2	0.2	0.2	0.2
Gold	g/t	0.9	0.9	0.9	0.9
Silver	g/t	3.0	4.0	4.0	4.0
Sulphur	%	4.6	3.6	1.8	3.3
Mineral					
Chalcopyrite	%	0.6	0.6	0.6	0.6
Pyrite	%	10.0	9.4	4.2	7.9
Gangue	%	89.5	90.0	95.2	94.9

Mineralogical determination testing revealed that approximately 53% of chalcopyrite in all three composites liberated at a primary grind size P_{80} of 140 µm. The Bond ball mill work index (Wi) was measured at approximately 14.8 kWh/t as shown in Table 16.8.

Table 16.8 G&T Bond Ball Mill Work Index

Sample	Wi (kWh/t)
A	14.7
B	14.8
C	14.8

Figure 16.1 Effect of Primary Grind Particle Size on Rougher Flotation Recovery



Results of rougher kinetic flotation tests showed that a primary grind P_{80} of 140 µm generated acceptable metallurgical results. A primary grind size P_{80} of 200 µm resulted in a decrease of about 5% in both copper and gold recovery. However, the report indicated that there was potential to improve metal recovery by slightly increasing mass recovery at the primary grind size. The effect of primary grind size on rougher flotation recovery is shown in Figure 16.1.

Open circuit batch cleaner tests revealed that between 80 to 85% of the copper could be recovered to a final concentrate containing approximately 25% copper. Gold recovery to the final copper concentrate ranged from 46 to 54%. About 25 to 40% of the gold in the flotation feed reported to the combination of the copper first cleaner tailing and the pyrite concentrate.

Cyanidation tests on copper first cleaner tailing and pyrite concentrate were conducted to investigate the response of the gold bearing products to cyanidation. The test results showed that between 65 and 70% of the gold contained was extracted after 24 hours of cyanidation leaching. The combined gold recovery from flotation plus cyanidation was about 72%.

Mineralogical examination showed that some of the gold occurred as small inclusions in pyrite. Regrinding the cyanidation feed to P_{80} of 15 μm from 35 μm improved gold extraction from the leach feed by 15% for Sample A.

16.1.3 2008 TEST WORK

In 2008 comprehensive test work was carried out by G&T and Hazen Research Inc., under supervision performed by T.J. Smolik of TJS Mining-Met Services, Inc. The test work included mineralogical characteristic determination, grinding resistance determination and mill sizing simulation, flotation flowsheet development, gold extraction of gold-bearing pyrite concentrate by cyanidation, free gold recovery by gravity separation, and ancillary tests.

TEST SAMPLES

A total of approximately 5,720 kg of individual samples were shipped to the G&T laboratory in two shipments. Most of the samples were collected from the Mitchell Zone. Two samples were generated from the Sulphurets Zone. The samples were constructed into 34 variability test samples (MET samples). The key element assay on the heads is shown in Table 16.9. The drill hole distribution for the Mitchell Zone and section views are shown in Figure 16.2 and Figure 16.3.

Figure 16.2 Mitchell Zone Metallurgical Samples – Plan View

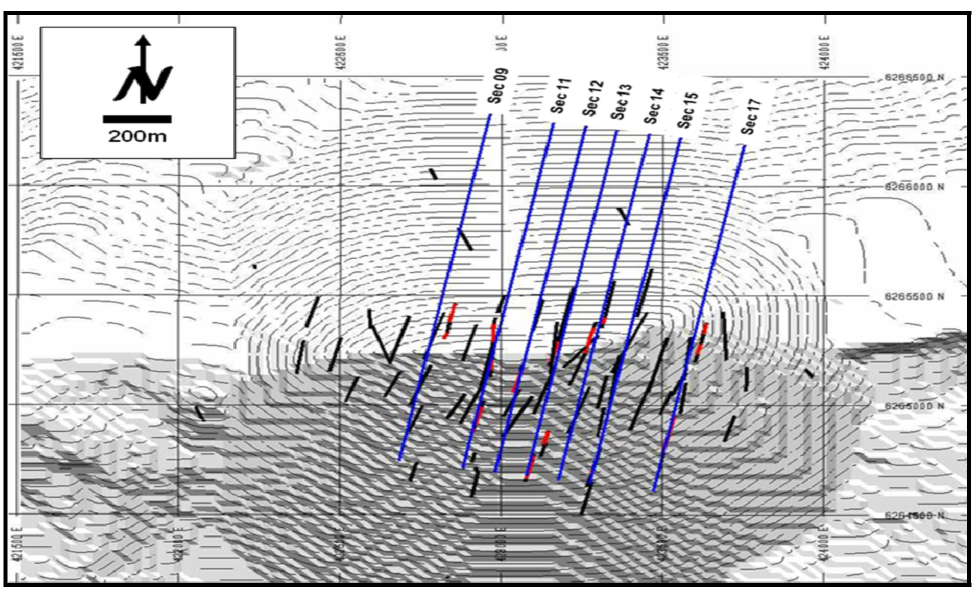


Figure 16.3 Mitchell Zone Metallurgical Samples – Section Views

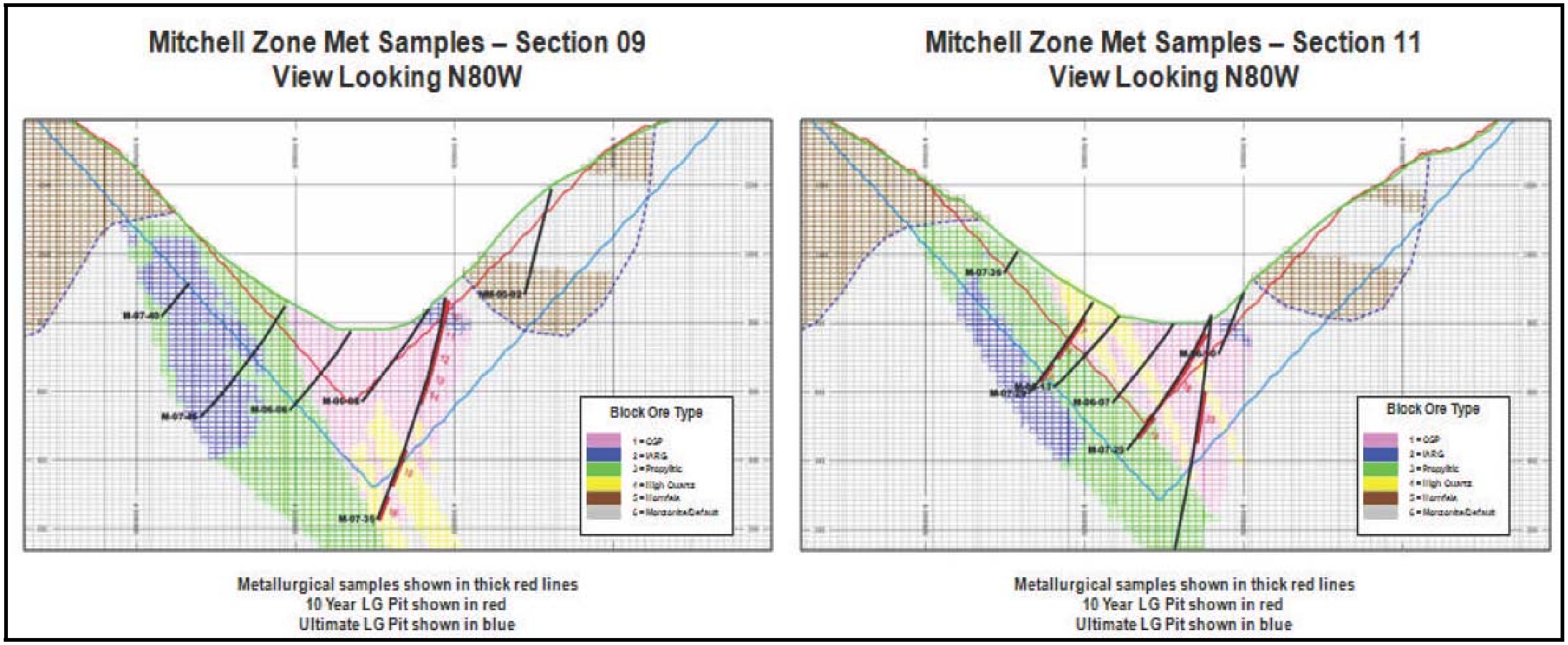


Figure 16.3 (con't) Mitchell Zone Metallurgical Samples – Section Views

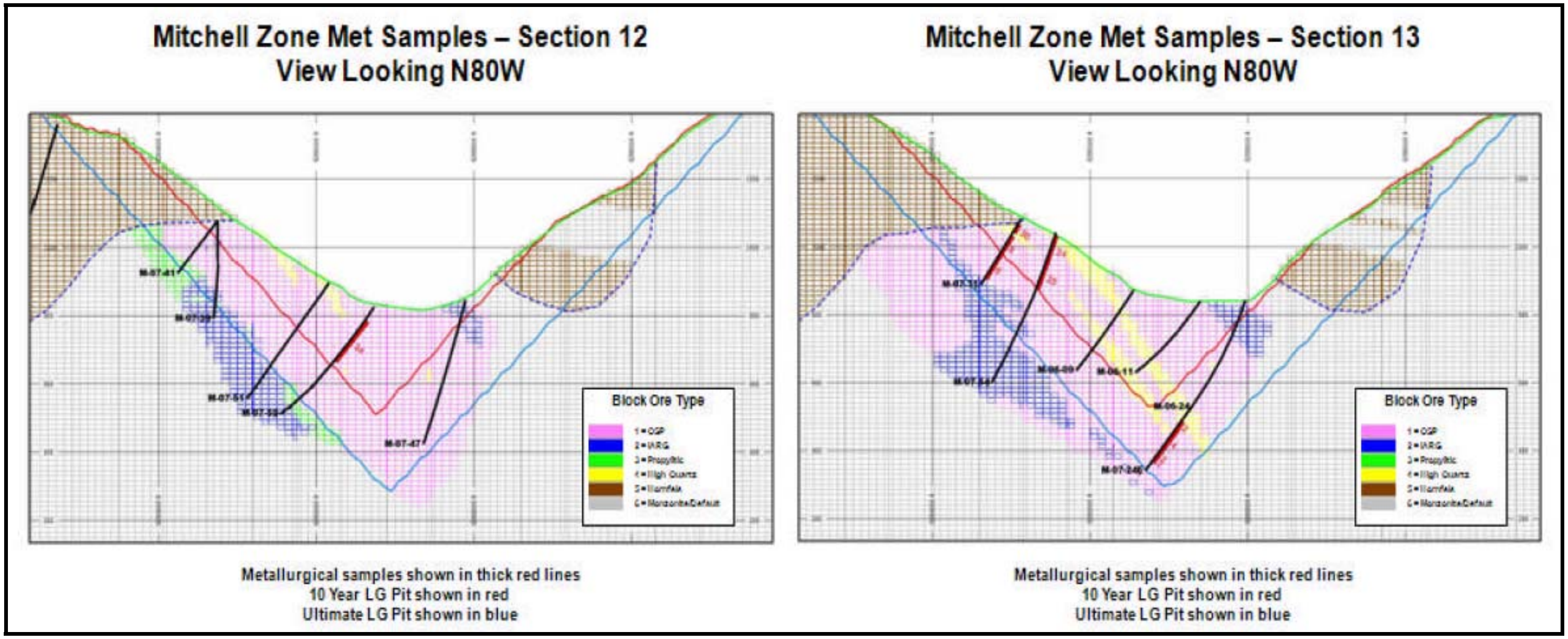


Figure 16.3 (con't) Mitchell Zone Metallurgical Samples – Section Views

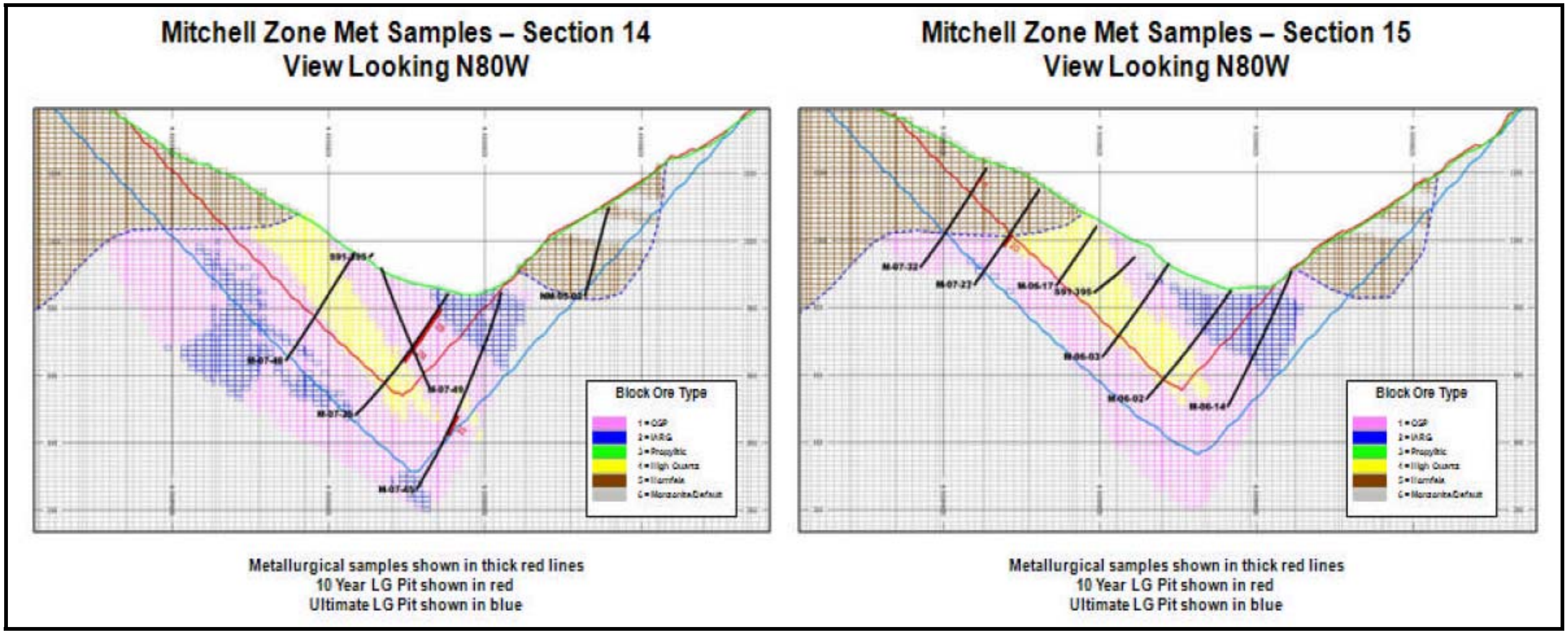
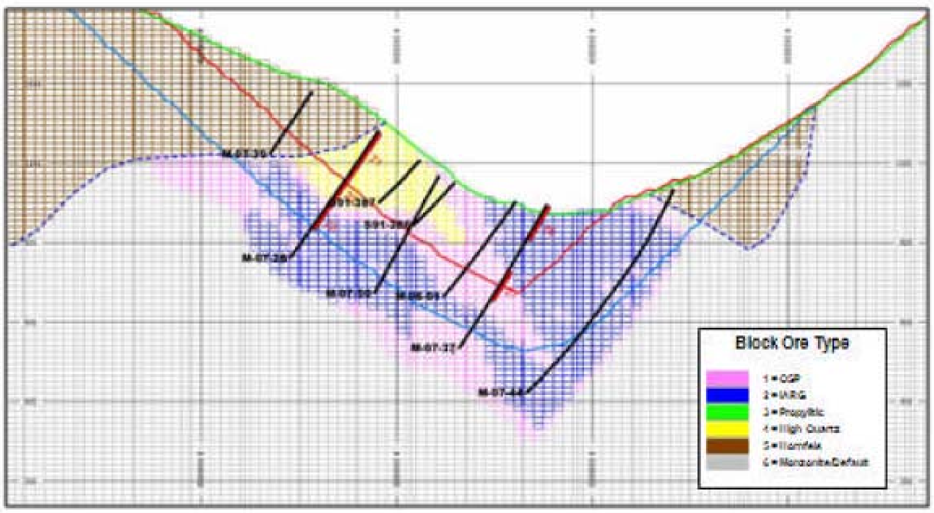


Figure 16.3 (con't) Mitchell Zone Metallurgical Samples – Section Views

Mitchell Zone Met Samples – Section 17
View Looking N80W



Metallurgical samples shown in thick red lines
10 Year LG Pit shown in red
Ultimate LG Pit shown in blue

Table 16.9 Head Assay on Variability Test Samples

Sample ID	Assay (% or g/t)*					Sample ID	Assay (% or g/t)*				
	Cu	Au	Ag	Mo	As		Cu	Au	Ag	Mo	As
MET 2	0.25	0.82	4	0.003	0.003	MET 19	0.30	0.67	4	0.002	0.001
MET 3	0.24	0.65	8	0.004	0.020	MET 20	0.17	0.54	4	0.005	0.004
MET 4	0.26	0.83	3	0.004	0.001	MET 21	0.21	0.83	2	0.004	0.003
MET 5	0.20	0.66	2	0.004	0.001	MET 22	0.20	0.85	3	0.011	0.002
MET 6	0.21	0.74	2	0.010	0.001	MET 23	0.11	0.32	3	0.025	0.010
MET 7	0.28	1.49	3	0.001	0.002	MET 24	0.24	0.86	3	0.001	0.053
MET 8	0.21	0.57	2	0.003	0.002	MET 25	0.14	0.43	2	0.007	0.005
MET 9	0.13	0.48	2	0.002	0.002	MET 26	0.13	0.68	2	0.002	0.004
MET 10	0.07	0.39	3	0.010	0.004	MET 27	0.15	0.82	2	0.003	0.002
MET 11	0.19	0.64	3	0.003	0.003	MET 28	0.16	0.86	3	0.012	0.001
MET 12	0.20	0.79	3	0.002	0.001	MET 29	0.19	0.79	5	0.018	0.006
MET 13	0.30	1.24	4	0.002	0.003	MET 30	0.14	0.22	3	0.003	0.005
MET 14	0.31	1.31	18	0.001	0.004	MET 32	0.22	1.18	2	0.002	0.006
MET 15	0.28	0.87	3	0.003	0.003	MET 33	0.33	0.96	7	0.002	0.008
MET 16	0.44	1.24	5	0.001	0.001	MET 34	0.28	0.85	3	0.004	0.002
MET 17	0.27	0.74	3	0.003	0.003	MET 35	0.12	0.30	1	0.003	0.008
MET 18	0.28	1.34	5	0.001	0.004	MET 36	0.52	0.81	1	0.023	0.005

* g/t for Au and Ag.

A total of 10 composites were generated from the MET samples. Five composites representing the major Mitchell Zone mineralization types projected to be mined in the initial 0-10 years were composed from the various drilling interval samples. The major mineralization types include:

- Composite QSP: quartz, sericite, pyrite
- Composite Hi Qtz: significant quartz veining and pyrite
- Composite IARG: intermediate argillic with sericite, chlorite, and pyrite
- Composite Prop: propylitic rock with sericite, epidote, and pyrite.

Table 16.10 Head Assay on Composites from Main Mineralization Type

Sample ID	Assay (% or g/t)*				
	Cu	Au	Ag	Mo	As
QSP 0-10	0.24	0.94	4	0.001	0.004
QSP 10-30	0.23	1.08	8	<0.001	0.004
QSP 0-30	0.24	0.95	4	0.004	0.002
QSP 0-10 LG	0.17	0.86	4	0.004	0.007
Hi Qtz 0-10	0.21	1.08	4	0.004	0.004
Hi Qtz 10-30	0.27	0.90	4	<0.001	0.004

table continues…

Sample ID	Assay (% or g/t)*				
	Cu	Au	Ag	Mo	As
Hi Qtz 0-30	0.25	1.02	4	0.004	0.001
Prop 10-30	0.26	1.00	3	<0.001	0.001
IARG 0-10	0.10	0.60	4	0.006	0.006
Master Comp 1	0.19	0.84	4	0.003	0.003

* g/t for Au and Ag.

Two samples representing 10-30 years mine production and two samples representing 0-30 years mine production from two of the main rock components (QSP and Hi-Qtz) in the Mitchell Zone were also generated from the drilling interval samples (MET samples). A master composite was also generated from QSP and Hi-Qtz composites for locked cycle tests. The feed grades for the composites are shown in Table 16.10.

The origin of the MET samples and the composite samples are detailed in G&T's test work report submitted to Seabridge.

MINERALOGICAL DETERMINATION

The mineralogical composition study shows that the sulphide mineral content in all three studied samples of QSP 0-30, Hi Qtz 0-30, and Master Composite 1, is dominated by pyrite. About 6 to 8 % of the sample weight is present as pyrite and chalcopyrite. It was indicated that copper was present in the form of chalcopyrite. Detailed analysis data are presented in Table 16.11.

Table 16.11 Mineral Composition Data

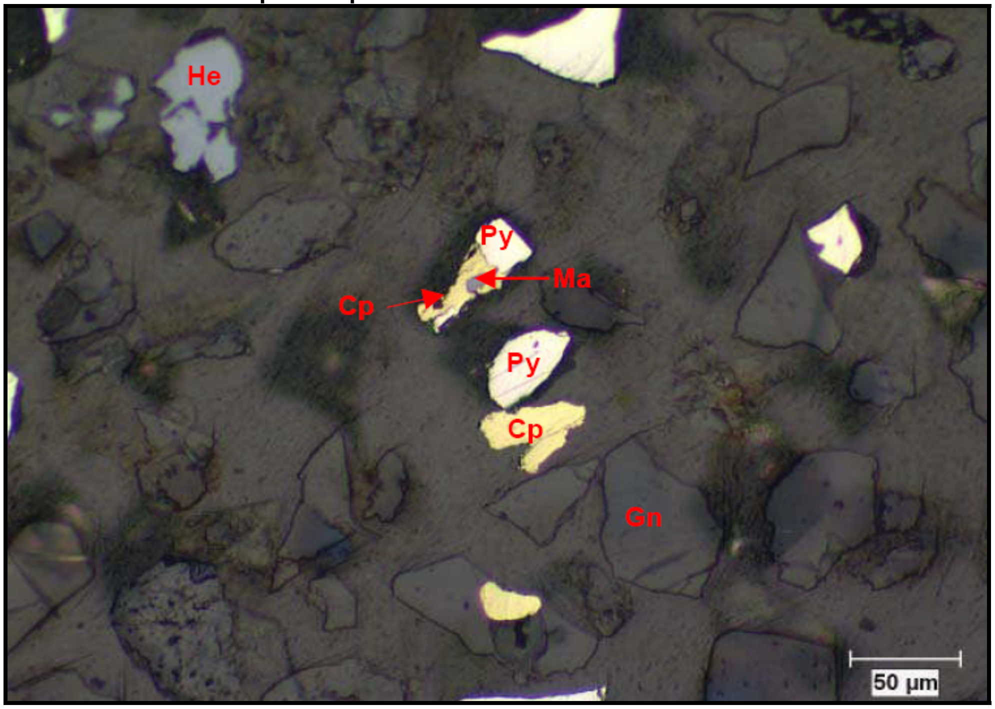
Sample	Mineral Composition (%)		
	Chalcopyrite	Pyrite	Gangues
QSP 0-30	0.66	6.6	92.7
Hi Qtz 0-30	0.67	8.2	91.2
Master Comp	0.54	8.1	91.4

The pyrite to chalcopyrite ratios are relatively high across the three tested samples. The average ratio is 12:1 while the highest ratio reaches 15:1. The two minerals do not show a high degree of interlocking in these samples. Figure 16.4 shows the relationship among main minerals in the samples.

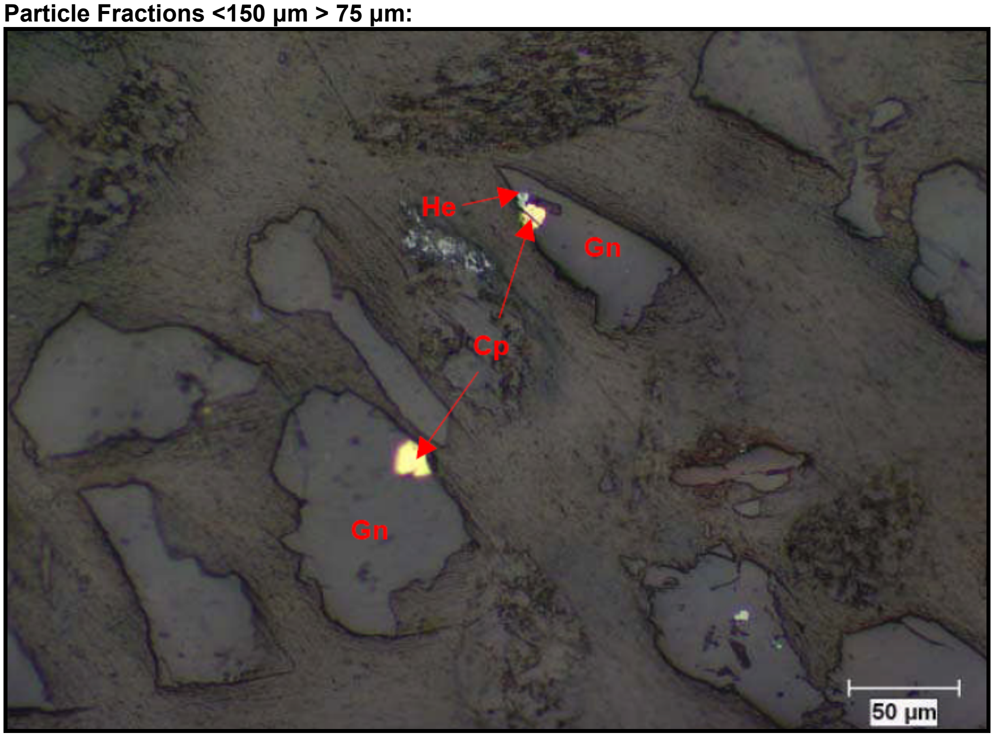
The degree of chalcopyrite liberation ranged from 46 to 56% across the samples tested at a primary grind of 80% passing 116 µm to 136 µm. The Hi Qtz sample showed a higher two-dimensional chalcopyrite liberation than the QSP sample. A primary grind size of 80% passing 125 µm was recommended for the Mitchell Zone.

Figure 16.4 Mineral Relationship – Master Composite

Particle Fractions <75 µm >32 µm:



Particle Fractions <150 µm > 75 µm:



Note: Cp-Chalcopyrite, Py-Pyrite, Ma-Magnetite, He-Hematite, Gn-Gangue

GRINDABILITY AND MILL SIZING SIMULATION

Grindability Determination

The grindability tests included semi-autogenous mill comminution (SMC) testing and standard Bond ball mill Wi. The samples used for the SMC grindability tests were identified as QSP, IARG, CL-RICH, QSP STW/QTVN, and H FELDS. The SMC test results are shown in Table 16.12.

Table 16.12 SMC Test Results

Parameter Sample	Value				
	QSP	IARG	CL-RICH	QSP STW/QTVN	H FELDS
Specific Gravity	2.81	2.42	2.78	2.69	2.71
A (maximum breakage)	70.7	75	68.1	82.6	81.6
B (relation between energy & impact breakage)	0.71	0.40	0.57	0.60	0.44
Axb (overall Au-SAG hardness)	50.2	30.0	38.8	49.6	35.9
DWi	5.5	7.9	7.1	5.4	7.5
Mia kWh/t	16.1	24.8	19.9	16.3	21.2
Ta (estimated abrasion parameter)	0.47	0.33	0.37	0.49	0.35

The DWi and Axb data indicate that on average, the materials were moderately hard in comparison to the JKTech database.

A separate standard Bond ball mill Wi determination testing was carried out by G&T on five composites identified as High Quartz 0-10, High Quartz 10-30, IARG 0-10, QSP 0-10, and QSP 10-30. The test results in Table 16.13 show that the samples have an averaged Bond Wi of 14.8 kWh/t indicating medium hardness to moderate hardness.

Table 16.13 Bond Ball Mill Wi Test Results

Samples	High Quartz 0-10	High Quartz 10-30	IARG 0-10	QSP 0-10	QSP 10-30	Average
Wi (kWh/t)	15.2	15.3	13.9	14.5	15.2	14.8

G&T also compared hardness variation on various variability test sample and main mineralization type composites by the comparative work index (CWi) method. The CWi was calculated from grind calibration data and the standard Bond ball Wi. The data is compared in Figure 16.5 for the various variability test samples and in Figure 16.6 for the composite samples. The average CWi values are 16.7 kWh/t for the individual samples and 15.5 kWh/t for the composite samples. Two of the mineral samples, Met 35 and Met 36, which were from the Sulphurets zone, produced much higher CWi values.

Figure 16.5 Comparative Ball Mill Wi Values – Variability Samples

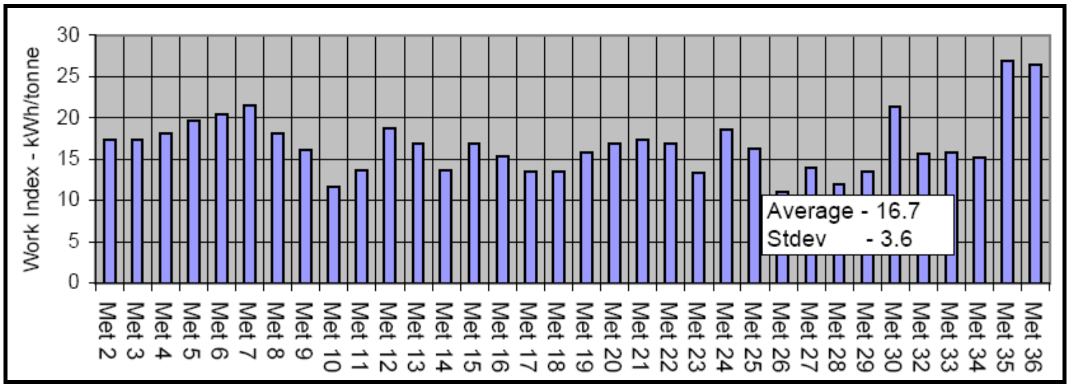
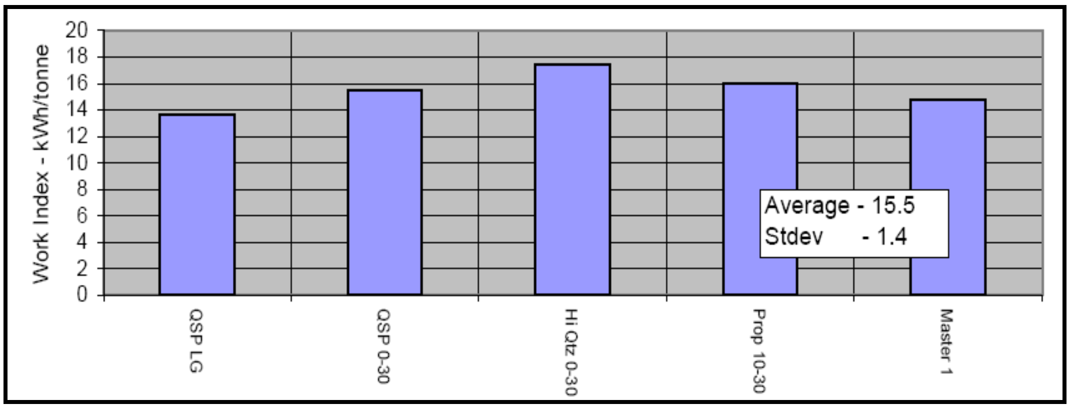


Figure 16.6 Comparative Ball Mill Wi Values – Composite Samples



Mill Sizing Simulation

Three mill sizing simulations were conducted by Contract Support Services Inc. using JK SimMet software. All the simulations were based on the data generated from SMC testing. The simulation input conditions are based on 120,000 t/d (two streams, 60,000 t/d each stream), 92% availability, a feed particle size of 80% passing 150 mm and one of the following conditions:

- Simulation 1: Bond ball mill Wi 14.8 kWh/t, a product particle size of 80% passing 150 µm.
- Simulation 2: Bond ball mill Wi 16 kWh/t, a product particle size of 80% passing 150 µm.
- Simulation 3: Bond ball mill Wi 15 kWh/t, a product particle size of 80% passing 120 µm.

Table 16.14 JK SimMet Simulation Results

Simulation		1a	1b	2a	2b	3a	3b
SAG Mill	Size, D x L (EGL) (ft x ft)	40 x 24	37.7 x 21	40 x 24	37.7 x 21	40 x 24	37.7 x 21
	Circulation Load (% of Feed)	19.5	18.4	19.5	18.4	19.5	18.4
	Gross Power Draw (kW)	18,843	15,570	18,843	15,570	18,843	15,570
Transfer Particle Size, mm		2,500	3,035	2,500	3,035	2,500	3,035
Ball Mills	Size, D x L (EGL) (ft x ft)	22 x 36	22 x 36	22 x 36	22 x 36	22 x 36	24 x 38
	Mill Number	2	2	2	2	2	2
	Gross Power Draw* (kW)	15,644	17,293	16,912	18,695	19,283	21,017
Total Power Draw (kW)		34,487	32,863	35,755	34,265	38,126	36,587
Cyclone Diameter (in)		26	26	26	26	26	26

* with Phantom Cyclones

Simulation results for each primary grinding stream are summarized in Table 16.14. The simulations are based on Phantom cyclone assumption and with primary cyclones for SAG mill discharges. The simulation results appear to show that, when the primary grind size is increased to 80% passing 120 μm, either of the following options will meet the primary grinding requirements:

- one-40 ft dia. x 24 ft L SAG mill and two-22 ft dia. x 36 ft L ball mills, **or**
- one-38 ft dia. x 21 ft L SAG mill and two-24 ft dia. x 38 ft L ball mills.

The simulation also indicated that less energy consumption would be expected if SAG mill discharges are classified by primary cyclones prior to ball mill grinding. The descriptions are detailed in the simulation reports.

PROCESS FLOWSHEET DEVELOPMENT

The flowsheet used for copper and gold recovery from the 2008 metallurgical samples was developed from the 2007 test program. The process used a combination of flotation and cyanidation. Copper and gold were first recovered into a copper-gold rougher concentrate. The copper flotation tailing was refloated to produce a gold-bearing pyrite concentrate. The copper-gold rougher concentrate was reground and cleaned to produce a copper-gold concentrate. The cleaner tailing from the copper cleaning circuit was cyanide leached together with the gold bearing pyrite concentrate after regrinding and aeration.

Flotation Flowsheet Development

Collectors dithiophosphinates (3418A) and dithiophospate A208 were used in the copper circuits and collectors potassium amyl xanthate (PAX) and A208 in the gold-pyrite circuit. The slurry pH at the copper and pyrite rougher flotation was at 10.

VARIABILITY TESTS

A total of 34 samples were used for variability tests, including two samples (Met 35 and Met 36) from Sulphurets Zone. Primary grind sizes ranged from 80% passing 115 to 171 µm, averaging at 149 µm. The rougher concentrate from the copper circuit was reground to approximately 18 µm prior to cleaner flotation.

It appeared that the copper recoveries reporting to the third cleaner concentrates in the open circuit tests increased with copper feed grade. As shown in Figure 16.7, G&T established the relationship between copper recovery and copper feed grade at a fixed concentrate grade of 25% Cu. In general, copper recovery increased with an increase in copper feed grade. The variation in the metallurgical performance of various mineral samples is shown in Figure 16.8.

Figure 16.7 Copper Recovery vs. Copper Feed Grade – Individual Samples

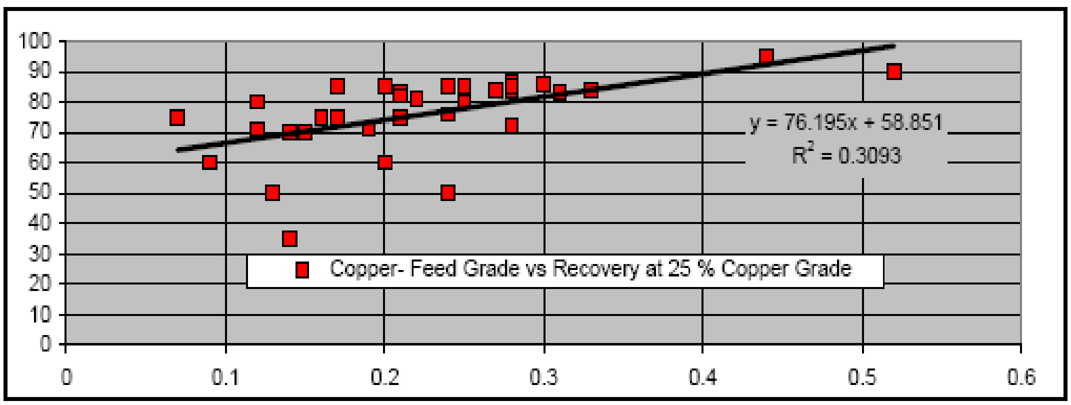
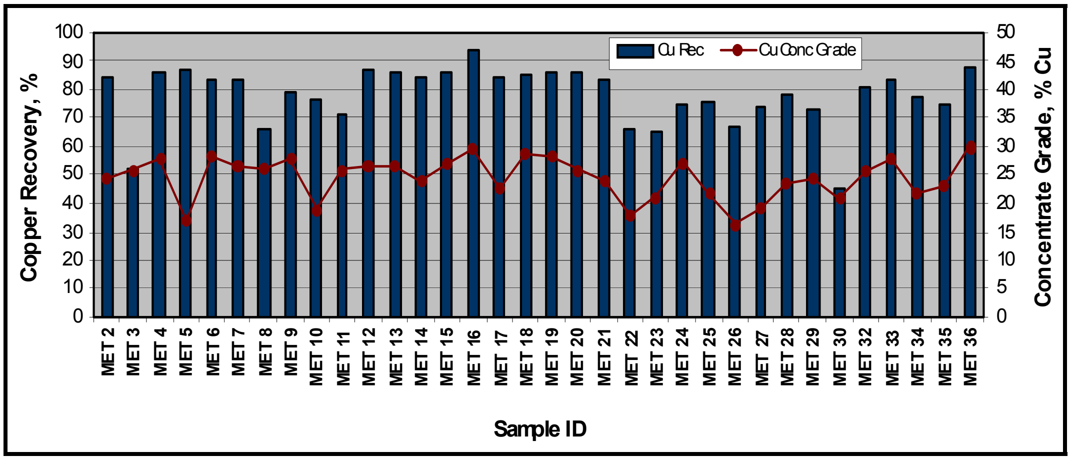
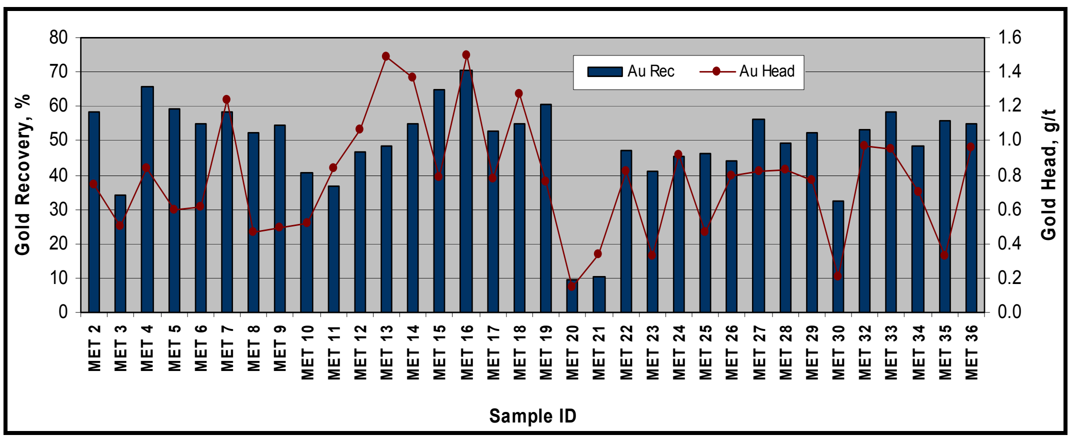


Figure 16.8 Copper Recovery and Concentrate Grade – Individual Samples



The gold recovery to the copper concentrate fluctuated from 30 to 70%, except for the MET 20 and 21 samples, which produced much lower gold recoveries. The tests seemed to shown that gold recovery to copper concentrate increased as a function of head gold content; however, the correlation was not strong. The gold metallurgical performance is plotted in Figure 16.9.

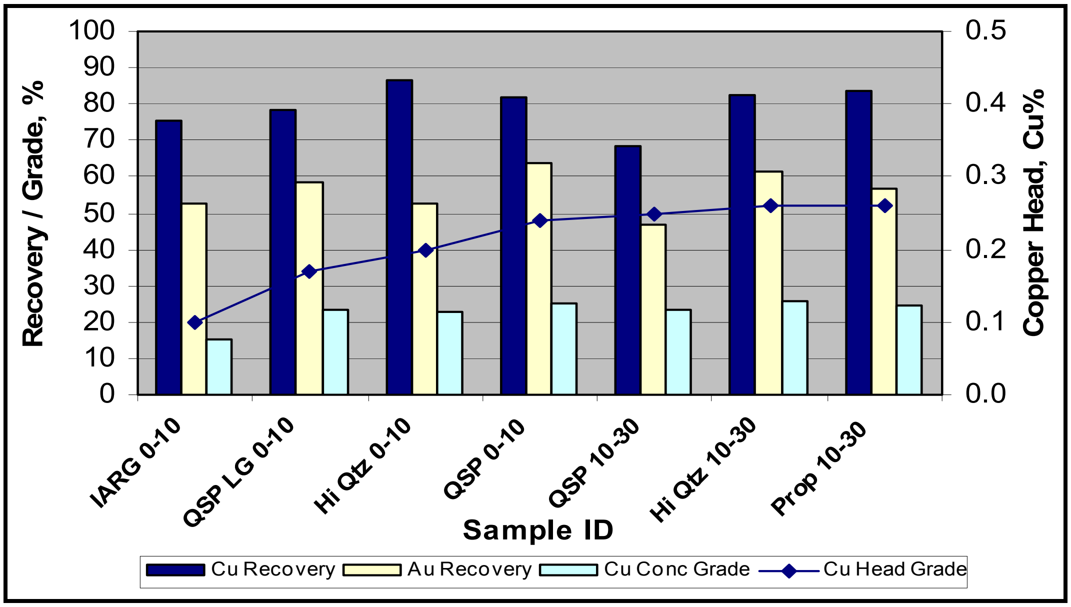
Figure 16.9 Gold Recovery and Feed Grade – Individual Samples



Gold recoveries to the cleaned gold-pyrite concentrate from the gold-pyrite flotation circuit varied from 4 to 29%, averaging at approximately 16%. Combined gold recoveries from both the copper circuit and gold-pyrite circuit ranged from 73 to 96%, averaging at approximately 86%.

Further testing were conducted on seven composites representing the major Mitchell zone mineralization types projected to be mined in the initial 0-10 years and the later years. The test results are shown in Figure 16.10. At primary grind sizes ranging from 130 to 168 µm, the open cycle tests produced third cleaner concentrates with between 69 to 86% copper recovery and between 47 to 64% gold recovery.

Figure 16.10 Metallurgical Performance – Composite Samples



Similar to the variability tests, on average, the combined gold recovery from both the copper circuit and gold-pyrite circuit from the composite samples was approximately 86%.

PRIMARY GRIND SIZE OPTIMIZATION

The effect of primary grind size and regrind size on the metallurgical performance of QSP 0-30 and Hi Qtz 0-30 composites was conducted. The test results, as summarized in Figure 16.11 and Figure 16.12, show that copper and gold metallurgical performance at rougher flotation stage improved with a decrease in primary grind size, although much less significantly when the grind size was finer than 120 µm.

Figure 16.11 Metallurgical Performance vs. Primary Grind Size – QSP 0-30

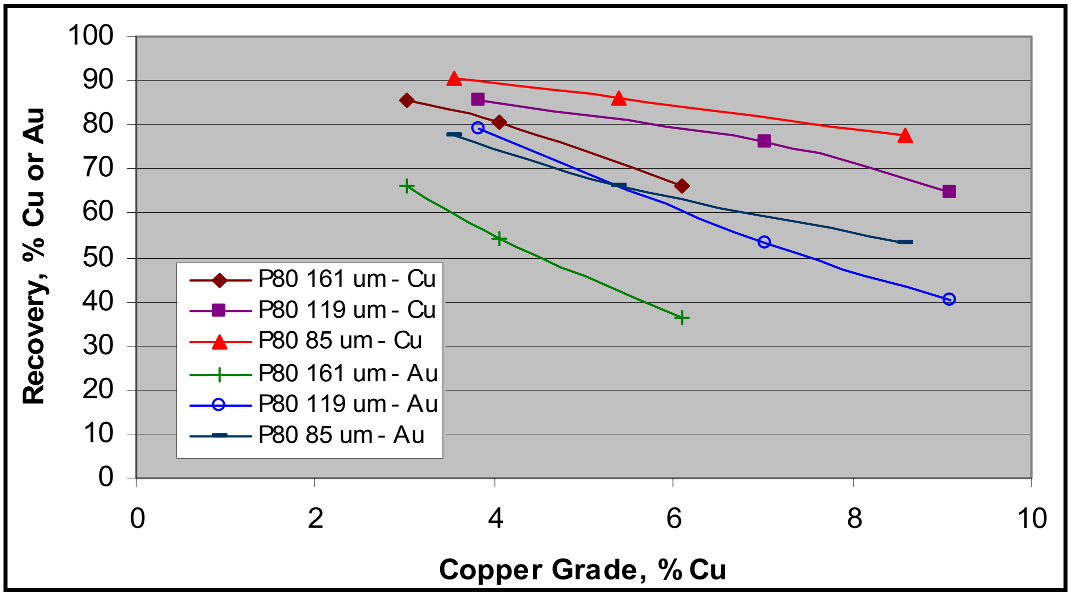
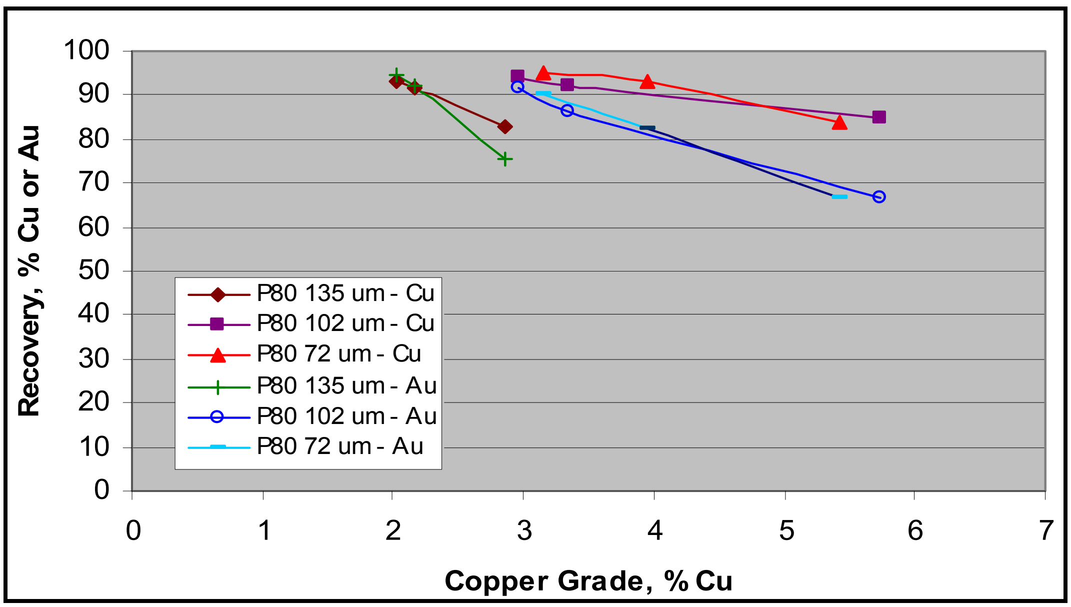


Figure 16.12 Metallurgical Performance vs. Primary Grind Size – Hi Qtz 0-30

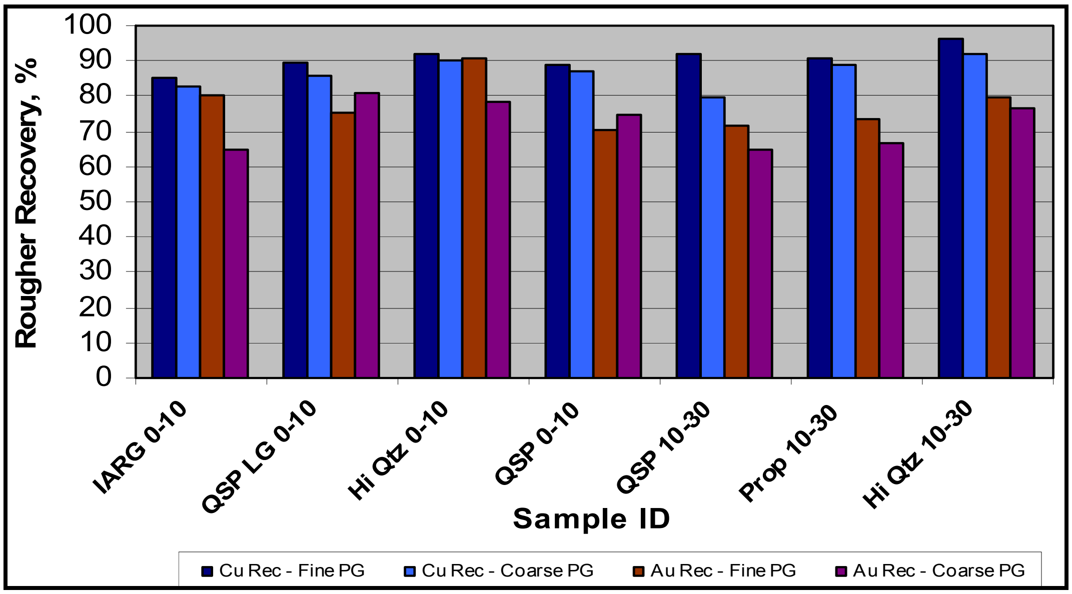


For QSP 0-30 composite, the copper recovery to a rougher concentrate, grading 4% Cu, improved from 81 to 89% when the primary grind size was decreased from 80% passing 161 µm to 80% passing 85 µm. Gold recovery increased significantly with the increase in the grind fineness; however, there was no significant increase when the grind size was finer than 80% passing 120 µm.

Hi Qtz 0-30 composite produced better metal recoveries compared with QSP 0-30 composite. The effect of primary grind size on the metallurgical performance was similar to that observed on QSP 0-30 composite.

Apart from QSP 0-30 and Hi Qtz 0-30 composites, two sets of comparison tests were performed on all the other composite samples to investigate the effect of primary grind size on copper and gold recovery. The average primary grind sizes tested were 80% passing 143 µm and 119 µm. The effect of the grind size on the metal recovery to rougher and third cleaner concentrates are shown in Figure 16.13.

Figure 16.13 Effect of Primary Grind Size on Metallurgical Performance



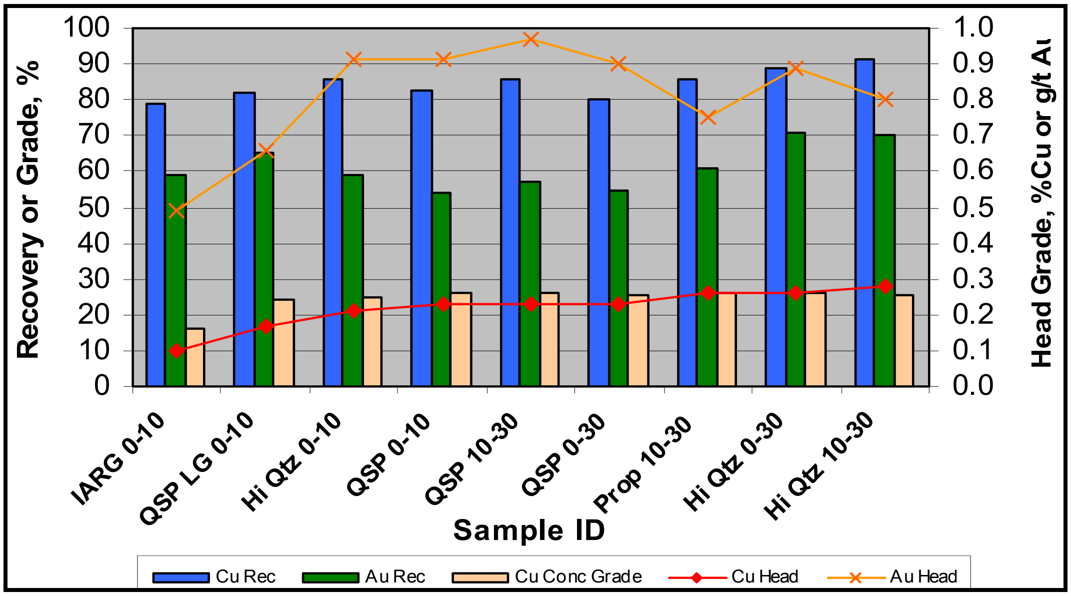
On average, the copper recovery reporting to rougher concentrate was 90.6% at the fine grind size, comparing to 86.6% at the coarse grind size. The average gold recovery to the concentrate increased from 72.3 to 77.3%. However, QSP 0-10 and QSP LG 0-10 composites appeared to show different gold metallurgical responses with a change in primary grind sizes.

At the fine grind size, the total average gold recovery from the copper circuit and pyrite circuit improved by approximately 4 to 89%.

OPEN CIRCUIT TESTS

Open circuit tests with two stages of cleaner flotation at a pH value of 11.5 were performed on the 9 composite samples. Primary grind sizes ranged from 80% passing 87 µm to 137 µm, averaging at 119 µm. Regrind sizes varied from 80% passing 12 µm to 22 µm, averaging at 18 µm. The results are shown in Figure 16.14.

Figure 16.14 Metallurgical Performance – Open Circuit Tests



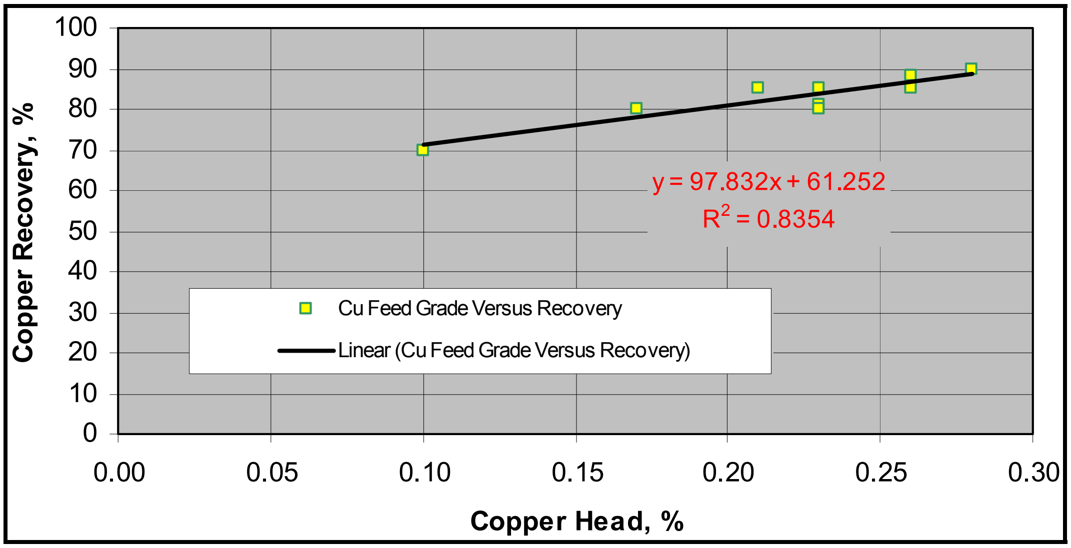
The second cleaner concentrate recovered between 79 to 91% of the copper and 54 to 71% of the gold from the 9 composites. On average, the metal recovery was 84.6% for copper and 61.2% for gold.

The results also appeared to show that copper recovery increased with an increase in copper head grade. The test results also showed that gold recovery did not seem to correlate with gold head grade or copper head grade.

Seven composites produced a concentrate of higher than 25% Cu, excluding 16.2% Cu from the IAGR 0-10 composite and 24.0% Cu from the QSP LG 0-10 composite.

After adjusting the copper recovery to reflect a concentrate grade of 25% Cu, a relationship between the adjusted copper recovery and copper feed grade is plotted in Figure 16.15. It appears that copper recovery is relatively closely related to copper head grade.

Figure 16.15 Copper Recovery vs. Copper Feed – Open Circuit Tests



LOCKED CYCLE TESTS

A master composite generated from Hi Qtz 0-30, QSP 0-30, and QSP 0-10 composites was used for locked cycle flotation testing. A total of three locked cycle tests (LCT) were carried out. The average results from two of the LCTs are shown in Table 16.15.

Table 16.15 Average LCT Results (Tests 141 and 142)

Product	Grade			Recovery			
	Cu (%)	Au (g/t)	Ag (g/t)	Mass (%)	Cu (%)	Au (%)	Ag (%)
Feed	0.21	0.90	4	100	100	100	100
Copper Concentrate	21.1	63.8	2.6	0.9	87	61	56
Copper 1st Cleaner Tailing	0.12	1.87	11	6.9	4	14	19
Gold-Pyrite Concentrate	0.10	2.14	8	5.8	3	14	11
Final Tailing	0.01	0.12	1	86.4	6	11	14

The average results indicate that 87% of the copper was recovered to the copper concentrate containing 21% copper. The concentrate also recovered 61% of the gold and 56% of the silver.

The gold and silver reporting to the gold bearing products, copper cleaner tailing, and gold-pyrite concentrates were approximately 28% and 30%, respectively.

Cyanidation of Gold-Bearing Pyrite Products

Because a portion of the gold is associated with pyrite, the first cleaner tailing and the gold-pyrite concentrate from the flotation circuit were subjected to cyanide leaching to recover the gold. The following discussion presents the results from the bottle roll cyanidation tests on the gold bearing products obtained from the flotation variability tests, open circuit tests, and LCTs.

CYANIDATION TESTS – PRODUCTS FROM FLOTATION VARIABILITY TESTS

A total of 30 cyanide leach tests were carried out on the gold bearing products from the flotation variability tests. Prior to the leach, the combined first cleaner tailing and the gold-pyrite concentrate was reground to a particle size of 80% passing 9 µm to 16 µm and aerated with air for 16 hours.

The test results are summarized in Table 16.16. The average gold extraction was approximately 79%. Increasing leach retention time did not improve gold extraction.

Table 16.16 Cyanidation Test Results – Individual Samples

Leach Time (h)	Sample ID	CN Test No.	Flotation Test No.	Regrind Size (P_{80} µm)	CN Feed (g/t Au)	CN Ext (% Au)
48	MET 2	50	4	11	1.7	60
	MET 5	51	7	9	1.6	79
	MET 8	52	10	9	2.2	74
	MET 11	53	13	10	6.3	94
	MET 14	54	16	15	2.7	81
	MET 17	55	19	13	1.9	87
	MET 20	56	22	11	1.1	58
	MET 23	57	25	15	1.3	82
	MET 26	58	28	13	2.7	85
	MET 29	59	31	10	4.1	83
	MET 33	60	34	16	1.9	88
Average				**12**	**2.5**	**79.2**
24	MET 3	64	5	12	1.4	65
	MET 4	65	6	13	1.6	78
	MET 6	66	8	9	2.4	84
	MET 7	67	9	11	3.4	78
	MET 9	68	11	9	1.3	74
	MET 10	69	12	11	2.7	91
	MET 12	70	14	10	3.3	87
	MET 13	71	15	10	8.9	90
	MET 15	72	17	14	2	85
	MET 16	73	18	13	3.2	82
	MET 18	74	20	11	1.37	63

table continues…

Leach Time (h)	Sample ID	CN Test No.	Flotation Test No.	Regrind Size (P_{80} µm)	CN Feed (g/t Au)	CN Ext (% Au)
24 (con't)	MET 19	75	21	12	1.99	82
	MET 21	76	23	9	2.15	69
	MET 22	77	24	12	2.74	63
	MET 24	78	26	10	4.1	87
	MET 25	79	27	9	1.7	78
	MET 27	80	29	13	2.21	81
	MET 30	82	32	11	1.63	76
	MET 32	83	33	7	3.35	91
Average				**11**	**2.7**	**79.2**
Average All				**11**	**2.6**	**79.2**

Cyanidation Tests – Products from Flotation Open Circuit Tests

Similar to the leach tests of the individual samples, the combined products of the first cleaner tailing and the gold-pyrite concentrate from the open cleaner circuit tests were cyanide leached to confirm the responses of the gold bearing materials to cyanidation. The leach retention time was 24 hours. As shown in Table 16.17, the gold extractions from the leach feed ranged from 65 to 89%. The average extraction was approximately 78% Au.

Table 16.17 Cyanidation Test Results – Composite Samples

Sample ID	CN Test No.	Flotation Test No.	Regrind Size (P_{80} µm)	CN Feed (g/t Au)	CN Extraction (% Au)
QSP 0-10	126	116	10	2.2	82
IARG 0-10	127	117	12	1.3	80
Hi Qtz 0-10	128	118	11	2.3	74
QSP LG 0-10	129	119	12	1.7	74
QSP 10-30	130	120	11	2.3	89
Prop 10-30	131	121	11	1.6	82
Hi Qtz 10-30	133	123	21	2.0	66
QSP 0-30	134	124	12	2.2	78
Hi Qtz 0-30	135	125	12	1.6	65
Average			**12**	**1.9**	**78**

Cyanidation Tests – Products from Flotation Locked Circuit Tests

The mixture of the first cleaner tailing and the gold-pyrite from the LCTs contained approximately 2.0 g/t Au and 9.6 g/t Ag. The leach tests showed that 70% of the gold and 63% of the silver were able to be extracted from the gold bearing products.

Average cyanide and lime consumptions in the two locked cycle flotation/cyanidation tests were 3.2 and 2.3 kg/t. The sodium cyanide concentration was varied between 1,000 and 2,000 ppm in Tests 144 and 145. The cyanide consumption was approximately 10% lower in the test using the lower NaCN concentration. The gold and silver recoveries were equivalent between the two tests. The test results are summarized in Table 16.18.

Table 16.18 Cyanidation Test Results – Master Composite

Sample ID	CN Test No.	Flotation Test No.	Regrind Size (P_{80} µm)	CN Feed		Extraction	
				g/t Au	g/t Ag	% Au	% Ag
Master	143	142	15	2.2	10.1	73.2	64.4
Master	144	141	15	1.8	9.1	67.6	61.5
Average			**15**	**2.0**	**9.6**	**70.4**	**63.0**

By adding the gold recoveries from the copper flotation circuit and the leach circuit, total gold and silver recoveries from the combined flowsheet were 80.7 and 74.9%, respectively.

Free-Gold Recovery

Ten of the drill interval samples were tested for free-gold recovery by gravity separation using centrifugal concentration (Knelson Concentrator) followed by panning. The test results are shown in Table 16.19.

Table 16.19 Gravity Separation Test Results

Sample ID	Pan Concentrate		Knelson Concentrate	
	Grade (g/t Au)	Distribution (%)	Grade (g/t Au)	Distribution (%)
MET 4	231	55	103	61
MET 7	28	9	25	13
MET 10	3	6	4	19
MET 14	27	8	17	11
MET 16	50	17	33	20
MET 18	22	7	13	9
MET 19	15	15	11	20
MET 23	13	12	6	16
MET 29	44	6	11	10
MET 32	20	8	11	11
Average	**45**	**14**	**23**	**19**

On average, approximately 19% of the gold in the samples was recovered to the Knelson concentrate with an average grade of 23 g/t Au.

Most of the pan concentrates contained less than 50 g/t Au with a gold recovery of less than 17%, except for the MET 4 sample. Panning produced a 231 g/t Au concentrate and recovered 55% of the gold from the MET 4 sample.

This data indicates that the MET 4 sample responded well to the gravity separation, although most of them produced poor metallurgical performances.

ANCILLARY TESTS

Settling Tests

Preliminary settling tests were conducted on pyrite flotation tailing. As reported by G&T, the tests on the tailing in slurry form failed to generate normal settling curves. As a result, the tests were carried out on the re-pulped sample from dried tailing from the same test.

The test data reveal that the settling area required was 0.73 m^2/t/d without adding flocculant and 0.30 m^2/t/d with the addition of 10 g/t of flocculant.

Magnetic Separation Tests

In the test program, Davis Tube magnetic separation was used in an effort to recover the metal values lost in the coarser than 200 mesh fraction of the pyrite flotation tailing from Tests 10, 11, and 25. Test results indicated that less than 3% of the coarse tailing weight was recovered into a magnetic fraction assaying approximately 23% iron. No copper or gold assay data was reported.

CONCENTRATE ASSAY

The copper concentrate from the LCT (Test 142) was subjected to multi-element analysis. The assay results are shown in Table 16.20.

Table 16.20 Multi-element Analysis on Concentrate – Master Composite

Element	Unit	Data
Sb	ppm	696
AS	ppm	1184
Co	ppm	48
Cd	ppm	72
Bi	ppm	36
Hg	ppm	0.6
Ni	ppm	120
F	ppm	346
Se	ppm	72

Element	Unit	Data
P	ppm	230
SiO_2	%	9.84
CaO	%	0.54
Al_2O_3	%	3.31
MgO	%	0.48
MnO	%	0.02
Pb	%	0.92
Zn	%	0.42

The concentrates produced from various drill interval samples and composites were assayed for molybdenum, arsenic, and silver contents. The results are presented in Table 16.21 and Table 16.22.

Table 16.21 Multi-Element Analysis on Concentrate – Drill Interval Samples

Sample ID	Content			Sample ID	Content		
	Mo (%)	As (%)	Ag (g/t)		Mo (%)	As (%)	Ag (g/t)
MET 2	0.25	0.012	232	MET 19	0.17	0.011	274
MET 3	0.40	0.376	306	MET 20	0.43	0.017	282
MET 4	0.45	0.008	264	MET 21	0.30	0.082	96
MET 5	0.05	0.002	156	MET 22	0.76	0.093	184
MET 6	0.93	0.001	242	MET 23	1.16	0.172	316
MET 7	0.06	0.013	204	MET 24	0.12	0.016	192
MET 8	0.09	0.010	236	MET 25	0.38	0.003	164
MET 9	0.12	0.008	420	MET 26	0.86	0.017	140
MET 10	1.02	0.253	498	MET 27	0.69	0.201	126
MET 11	0.17	0.077	260	MET 28	0.31	0.032	190
MET 12	0.09	0.013	274	MET 29	0.19	0.069	268
MET 13	0.03	0.056	230	MET 30	0.02	0.374	178
MET 14	0.03	0.193	1118	MET 32	0.51	0.004	158
MET 15	0.09	0.015	228	MET 33	0.06	0.050	402
MET 16	0.05	<0.001	246	MET 34	0.10	0.010	162
MET 17	0.26	0.082	180	MET 35	0.29	0.014	100
MET 18	0.06	0.065	324	MET 36	0.38	0.010	26

Table 16.22 Multi-element Analysis on Concentrate – Composite Samples

Sample ID	Content		
	Mo (%)	As (%)	Ag (g/t)
QSP 0-10	0.10	0.12	158
QSP 10-30	0.30	0.05	478
QSP 0-30	0.27	0.06	382
QSP LG 0-10	0.58	0.16	208
Hi Qtz 0-10	0.18	0.05	154
Hi Qtz 10-30	0.15	0.02	224
Hi Qtz 0-30	0.26	0.04	220
Prop 10-30	0.27	0.06	382
IARG 0-10	0.87	0.12	192
Master	0.58	0.16	316
Average	**0.36**	**0.08**	**271**

As indicated, the element contents in the drill interval samples varied significantly from sample to sample. The composite samples showed much less variation in the element contents. On average, the arsenic content should not attract smelting penalties by most smelters.

Molybdenum contents in some of the samples were high enough to consider producing a molybdenum concentrate, in particular from QSP LG 0-10 and IARG 0-10 composites.

16.1.4 Recommended Test Work

Wardrop recommends further metallurgical test work to optimize process conditions and establish design-related parameters for the next stage of study. The test work should include the samples from Sulphurets and Kerr zones. The recommended testwork should include:

- comminution tests including full JK Drop-Weight tests or SPI tests, Bond Wi tests, as well as abrasion work index and crushability tests (this will depend on the final milling approach)
- copper and gold flotation condition confirmation and optimization tests including optimization of primary grinding particle size, reagents, regrinding particle size, and pulp pH level
- further variability tests on the optimized conditions, in particular for the Sulphurets and Kerr samples
- copper-molybdenum separation tests including flowsheet development, copper mineral suppression reagent optimization, regrinding particle size determination, molybdenum collector screening, and chemical methods (leaching) to reduce impurity levels of the final molybdenum concentrate
- cyanide leach condition optimization tests including cyanide concentration, regrind particle size, pre-treatment by aeration, and leach time
- cyanide recovery tests and detoxification tests
- design related data collecting tests, such as pressure filtration and high rate thickening tests.

The test work summarized here is currently in progress.

16.1.5 Projected Metallurgical Performance

According to the metallurgical test results of the 2008 G&T test program, preliminary estimates for copper, gold, silver, and molybdenum metallurgical performances were developed. In the projection, the metal recoveries are based on the combined process of flotation and cyanidation. The flotation process will produce a copper concentrate with 25% Cu and a molybdenum concentrate with 53% Mo. The gold cyanidation process on gold-bearing pyrite products will produce a gold-silver dore.

The metallurgical projections are based on the following assumptions:

Grade

- copper concentrate:
  - copper grade = 25% Cu, based on bench scale test results.
- molybdenum concentrate:
  - molybdenum grade = 53% Mo, assumed according to similar operations.

Recovery

- copper recovery = 97.832 x (copper feed grade, %) + 61.252, based on bench scale test results.
- total gold recovery =
  - 80% when feed grade is higher than 1 g/t Au
  - 78% when feed grade is between 0.5 g/t Au and 1 g/t Au
  - 73% when feed grade is higher than 0.25 g/t Au but lower than 0.5 g/t Au

  The gold recoveries are projected from bench scale flotation test and cyanidation test results. The gold recovery reporting to flotation concentrate is assumed to be 60% according to bench scale locked cycle flotation test results.
- total silver recovery = 73%, including 55% of the silver reporting to copper concentrate.

  The silver recovery is projected according bench scale locked cycle flotation test results and cyanidation test results.
- molybdenum recovery =
  - 60% when feed grade is higher than 0.01% Mo
  - 40% when feed grade is between 0.005% Mo and 0.01% Mo
  - 0% when feed grade is lower than 0.005% Mo

  The molybdenum recoveries are assumed according to similar operations.

The projected metallurgical performances are shown in Table 16.23.

Table 16.23 Projected Metallurgical Performances

						Product										Recovery					
						Copper Concentrate								Mo Concentrate		Cu Concentrate			Dore		Mo Conc*
		Head Grade				Mass	Grade			Dore Mass				Mass	Grade						
										Au		Ag									
Year	Process Rate t (000s)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (ppm)	t	Cu (%)	Au (g/t)	Ag (g/t)	kg	oz (000s)	kg	oz (000s)	t	Mo (%)	Cu (%)	Au (%)	Ag (%)	Au (%)	Ag (%)	Mo (%)
1	43,201	0.213	0.828	2.26	40.5	301,312	25	71.2	177.9	6,440	207	17,542	564	920.8	53	82.0	60	55	18	18	27.9
2	43,200	0.211	0.717	2.17	45.6	298,582	25	62.2	172.5	5,576	179	16,854	542	1,113.7	53	81.9	60	55	18	18	29.9
3	43,200	0.22	0.711	2.66	40.7	314,715	25	58.6	200.7	5,528	178	20,670	665	884.2	53	82.8	60	55	18	18	26.7
4	43,200	0.314	0.751	2.97	19.2	499,200	25	39	141.5	5,842	188	23,120	743	139.3	53	92.0	60	55	18	18	8.9
5	43,200	0.227	0.734	3.31	39.9	327,297	25	58.1	240.1	5,706	183	25,722	827	855.9	53	83.5	60	55	18	18	26.3
6	43,200	0.324	0.589	1.76	22.6	521,102	25	29.3	80.4	4,579	147	13,716	441	320.5	53	93.0	60	55	18	18	17.4
7	43,200	0.456	0.544	1.19	11.2	740,857	25	19	38.2	4,232	136	9,267	298	99.2	53	94.0	60	55	18	18	10.9
8	43,200	0.413	0.49	1.39	12.6	670,816	25	18.9	49.1	2,753	89	10,787	347	148.5	53	94.0	60	55	13	18	14.5
9	43,200	0.388	0.407	0.96	11.9	629,509	25	16.7	36.2	2,283	73	7,465	240	216.8	53	94.0	60	55	13	18	22.3
10	43,200	0.296	0.678	2.84	15.5	461,358	25	38.1	146.2	5,268	169	22,077	710	68.0	53	90.2	60	55	18	18	5.4
11	43,200	0.316	0.551	0.1	0.5	503,108	25	28.4	4.8	4,284	138	785	25	0.6	53	92.2	60	55	18	18	1.3
12	43,200	0.202	0.348	0.79	16.4	283,456	25	31.8	65.9	1,953	63	6,109	196	4.7	53	81.1	60	55	13	18	0.4
13	43,200	0.185	0.528	0.59	13.9	254,010	25	53.9	55.6	4,109	132	4,624	149	158.1	53	79.4	60	55	18	18	13.9
14	43,200	0.177	0.572	0.79	28	240,368	25	61.6	78.2	4,445	143	6,153	198	818.4	53	78.6	60	55	18	18	35.9
15	43,200	0.171	0.477	1.49	57	230,855	25	53.5	153.2	2,678	86	11,573	372	1,964.5	53	78.0	60	55	13	18	42.3
16	43,200	0.13	0.522	1.72	77.1	165,819	25	81.6	245.9	4,059	130	13,346	429	2,895.1	53	73.9	60	55	18	18	46.1
17	43,200	0.117	0.54	1.86	85.6	147,430	25	94.9	299.4	4,199	135	14,444	464	3,188.7	53	72.7	60	55	18	18	45.7
18	43,200	0.126	0.539	1.94	80.7	160,508	25	87.1	287.6	4,194	135	15,109	486	2,932.7	53	73.6	60	55	18	18	44.6
19	43,200	0.143	0.556	2.28	67	185,461	25	77.7	291.8	4,325	139	17,709	569	2,195.3	53	75.2	60	55	18	18	40.2
20	43,200	0.16	0.595	2.59	58.4	212,071	25	72.8	290.5	4,630	149	20,162	648	1,669.7	53	76.9	60	55	18	18	35.1
21	43,200	0.173	0.607	2.93	49.9	233,187	25	67.5	298.6	4,724	152	22,788	733	1,223.2	53	78.1	60	55	18	18	30.1
22	43,200	0.205	0.539	3.36	38.8	288,209	25	48.5	277.3	4,193	135	26,158	841	874.4	53	81.3	60	55	18	18	27.6
23	43,200	0.185	0.446	1.76	48.7	253,533	25	45.6	164.8	2,506	81	13,674	440	1,450.2	53	79.3	60	55	13	18	36.5
24	43,200	0.144	0.557	2.01	70.6	187,763	25	76.9	254.4	4,329	139	15,631	503	2,526.6	53	75.4	60	55	18	18	43.9
25	43,200	0.227	0.537	1.92	45.8	327,429	25	42.5	139.6	4,176	134	14,956	481	1,490.6	53	83.5	60	55	18	18	39.9
26	43,200	0.173	0.737	2.96	55.3	233,401	25	81.8	301.7	5,728	184	23,045	741	1,592.9	53	78.2	60	55	18	18	35.4
27	43,200	0.198	0.761	3.28	42.5	275,914	25	71.5	282.7	5,915	190	25,525	821	1,009.5	53	80.6	60	55	18	18	29.2
28	43,200	0.212	0.719	3.4	33.5	300,513	25	62.1	269.1	5,595	180	26,464	851	625.0	53	82.0	60	55	18	18	22.9
29	43,200	0.227	0.782	3.91	29.6	328,208	25	61.8	283.3	6,080	195	30,430	978	403.3	53	83.5	60	55	18	18	16.7
30	26,507	0.21	0.617	3.36	30.7	182,196	25	53.9	268.9	2,944	95	16,034	516	109.4	53	81.8	60	55	18	18	7.1
Average	42,644	0.225	0.599	2.136	39.77	325,273	25	47.1	154	4,442	143	16,398	527	1,063.3	53	84.8	60	55	17.4	18	33.2
Total	**1,279,308**	**0.225**	**0.599**	**2.136**	**39.773**	**9,758,187**	**25**	**47.1**	**154**	**133,272**	**4,285**	**491,939**	**15,816**	**31,900.1**	**53**	**84.8**	**60**	**55**	**17.4**	**18**	**33.2**

* Recoveries are from the mining block model along with the input of projected molybdenum recoveries.

16.2 MINERAL PROCESSING

16.2.1 INTRODUCTION

The process plant will consist of three separate facilities:

- an ore crushing/grinding and handling facility at the mine site
- a ground slurry transportation system
- a main process facility at the plant site, including secondary grinding, flotation, regrinding, leaching and concentrate dewatering.

The crushing/grinding plant located at the mine site will reduce the run-of-mine (ROM) particle size to approximately 80% passing 180 µm by three stages of crushing and one stage of grinding. The comminution circuit will include:

- primary crushing – gyratory crushers
- secondary crushing – cone crushers
- tertiary crushing – high pressure grinding rolls (HPGR)
- primary grinding – conventional ball mills.

The ground mineralized material will be transported to the main plant site by 3 stages of pumping through a 23 km tunnel.

The main process plant will be located approximately 23 km northeast of the mine site and will consist of following process facilities:

- secondary grinding
- copper-gold/molybdenum flotation
- copper-gold/molybdenum separation (if required)
- concentrate dewatering
- gold cyanide leach
- gold recovery and related processes.

A tailing management facility (TMF), located southeast of the main process plant, is designed to store flotation tailing and cyanide leach residues.

ROM material will be delivered to the primary crushing facility by trucks. The material will be crushed by two gyratory crushers and followed by four cone crushers. The crushed products will be further crushed in a closed circuit consisting of four HPGRs and screens.

The screen undersized material will feed 4 ball mills in a closed circuit with hydrocyclones and further reduce to 80% passing 180 µm. The hydrocyclone overflows will then be transported to the plant site via three stages of pumping.

At the plant site, the slurry material will be ground further by tower mills to 80% passing 125 µm.

The products from the primary grinding circuits will feed two trains of copper-gold/molybdenum rougher/scavenger flotation circuits. The rougher flotation concentrates from the flotation trains will be further reground to a particle size of 80% passing 20 µm in tower mills.

The reground rougher concentrate will then be upgraded in a cleaner flotation circuit where the concentrate will be subjected to three stages of cleaner flotation to produce copper-gold or copper-gold/molybdenum concentrates with a grade of 25% Cu. Depending on the molybdenum content in the copper-gold/molybdenum concentrate, the bulk concentrate may be further separated into a molybdenum concentrate and a copper-gold concentrate.

The final concentrate(s) will be dewatered by a combination of thickening and pressure filtration to 8% moisture before being transported to smelters, while the molybdenum concentrate will be further dried prior to being shipped.

The copper-gold/molybdenum rougher scavenger flotation tailing will further be floated to produce a rougher gold-bearing pyrite concentrate. The pyrite flotation tailing will be sent to the TMF. The concentrate will be then reground in tower mills to a particle size 80% passing 15 µm. Depending on gangue minerals content and copper content, the gold-pyrite concentrate may be cleaned and subjected to a copper-pyrite separation after regrinding.

The reground gold-pyrite concentrate will be fed to the cyanidation plant together with the first copper cleaner tailing from the copper-gold/molybdenum cleaner flotation circuit. The combined materials will be pre-oxidized by aeration prior to cyanidation. Gold will be extracted by sodium cyanide through carbon-in-leach (CIL) processing.

The loaded carbon will be stripped by the conventional Zadra process, and gold in the pregnant solution will be recovered in the subsequent electrowinning process. The barren solution from the elution circuit will be circulated back to the leach circuit. The gold sludge will be further refined by a conventional pyrometallurgical process to produce gold doré bullion.

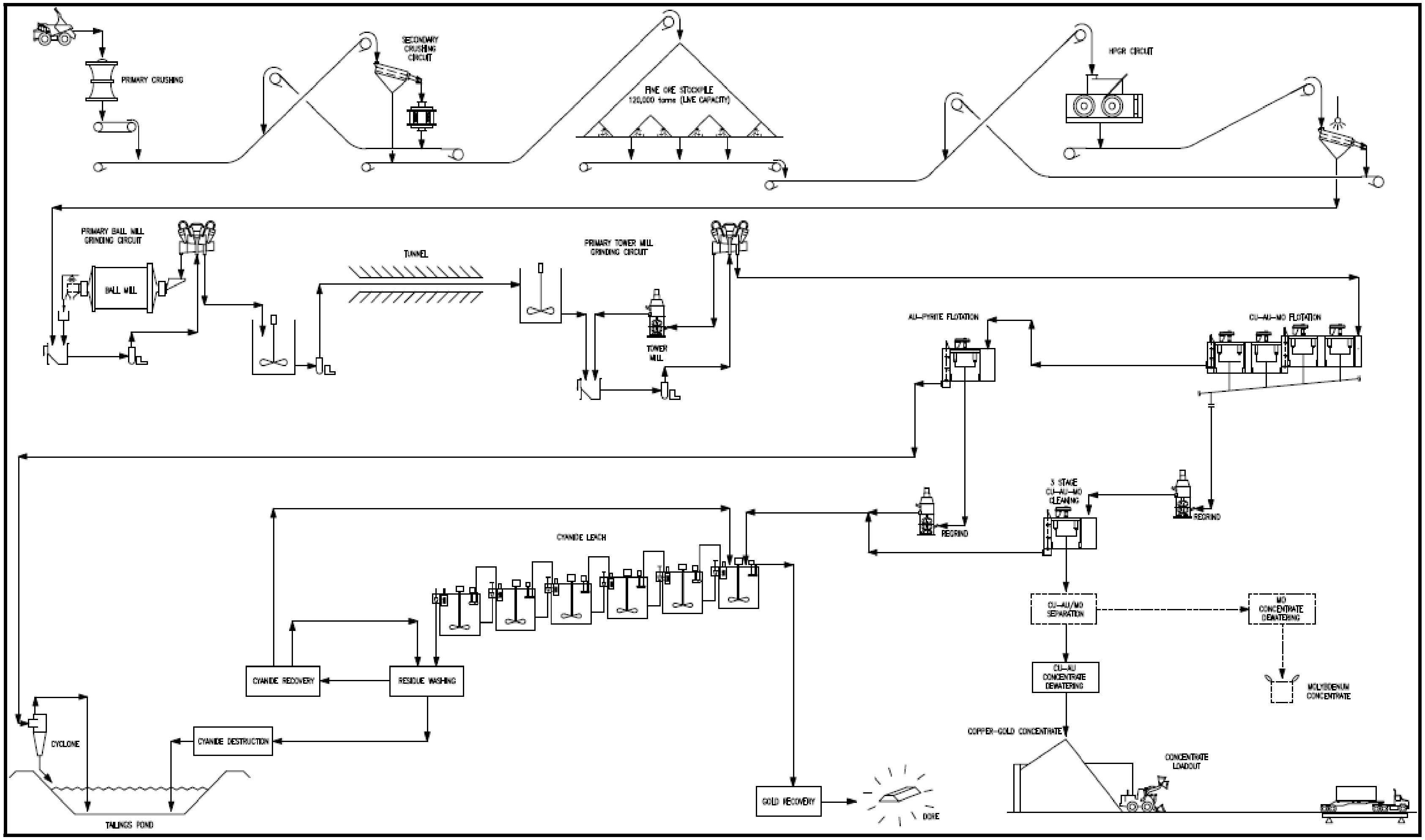
The residues from the leach circuit will be pumped to a conventional counter-current decantation (CCD) washing circuit. The solution from the circuit will be sent to a cyanide recovery circuit using acidification-volatilization-reneutralization (AVR) process or sulphidization-acidification-recycling of precipitate-thickening of precipitate (SART) process.

The washed residues will then be sent to a cyanide destruction circuit. The cyanide destruction treatment will employ a combined process of SO_2 oxidation and peroxide oxidation. The treated residues will then be transported to the centre of the TMF. The residues will be stored under water at all times to prevent the oxidation of sulphides.

The processes are shown in Figure 16.16 in simplified flowsheet format and are detailed in the following sections.

Detailed process flowsheets are available in Appendix B.

Figure 16.16 Simplified Process Flowsheet



16.2.2 PROCESS DESIGN CRITERIA

MAJOR DESIGN CRITERIA

The concentrator is designed to process 120,000 t/d (calendar). The major criteria used in the design are shown in Table 16.24.

Table 16.24 Major Design Criteria

Criteria	Unit	
Daily Process Rate	t/d	120,000
Operating Year	days	365
Primary/Secondary Crushing		
Availability	%	70
Crushing Rate	t/h	7,143
Primary Crushing Particle Size, P_{80}	µm	150,000
Secondary Crushing Particle Size, P_{80}	µm	45,000
HPGR/Grind/Flotation/Leach		
Availability	%	92
Milling and Flotation Process Rate	t/h	5,434
Mine Site Mill Feed Size, P_{80}	µm	3,500
Plant Site Mill Feed Size, P_{80}	µm	180
Primary Grind Size, P_{80}	µm	125
Bond Ball Mill Wi	kWh/t	15
Bond Abrasion Index	g	0.250
Concentrate Regrind Size, 80% Passing		
Cu/Au Rougher/Scavenger Concentrate	µm	20
Au-Pyrite Concentrate	µm	15
Gold-bearing Materials Leach Method		CIL
Feed to CIL Circuit	t/d	12,500

The complete design criteria are detailed in Appendix C.

16.2.3 PROCESS PLANT DESCRIPTION

PRIMARY CRUSHING

Primary crushing will consist of two 60″ by 89″ gyratory crushers, two apron feeders, and two ore stockpile feed conveyors. The top size of the ROM material to the gyratory will be 1,300 mm; oversize materials larger than 1,300 mm will be broken by a rock breaker. The gyratory crushers will reduce the ROM to a particle size of 80% passing 150 mm. The products from each gyratory crusher will be fed to one 1.83 m wide by 79 m long conveyor via one 2.44 m wide by 9.14 m long apron feeder.

SECONDARY CRUSHING

The crushed materials from each primary crushing line will be split into 2 equal portions and further reduced to 80% passing 45 mm in 2 secondary crushing circuits. The circuit will consist of the following key equipment in a closed circuit:

- four MP1000 or equivalent cone crushers each driven by a 750-kW motor
- four 2.7 m wide by 8.0 m long double deck vibrating screens.

Each of the split portions will gravity flow to two vibrating screens. The screen oversize will be further crushed by the cone crushers. The cone crusher products will return to the screen feed conveyor. The screen undersize will be delivered by a conveying system, including inclined/tripper conveyors, to an enclosed surge stockpile with a 120,000-t live capacity.

COARSE ORE HANDLING

The crushed coarse materials will be reclaimed from the 120,000-t stockpile by 6 reclaim apron feeders onto one 1.83 m wide HPGR feed conveyors to two HPGR feed surge bins, each with a live capacity of 300 t.

TERTIARY CRUSHING

The reclaimed materials will be further crushed by four HPGR crushers. Four belt feeders will withdraw the reclaimed materials from the two HPGR feed surge bins and feed to each HPGR crusher separately. Each of HPGR crushers are in closed circuit with a 2.7 m wide by 8.0 m long double deck vibrating screen. The discharges of the HPGR crushers will be wet-screened at a cut size of 6 mm. The screen oversize will return to the feed conveyor of the HPGR feed bin while the screen undersizes will leave the crushing circuit and report to the primary grind circuits. The four HPGR crushing lines will have a process capacity of 5,434 t/h. The key equipment is as follows:

- four HPGR crushers, each equipped with two 2,600 kW motors
- four 2.7 m wide by 8.0 m long vibrating screens
- four 1.5 m wide by 11.0 m long belt feeders.

PRIMARY GRINDING – MINE SITE

There are two primary grinding processes – one at the mine site and the other at the plant site. The grinding circuit at the mine site will employ conventional ball mills to grind the crushed materials while the further grinding circuit (secondary grinding circuit) at the plant site will use energy efficient tower mills. All the primary grinding circuits are designed to have a nominal process rate of 5,434 t/h.

The primary grinding circuit at the mine site will include four grinding circuits, which are made up of the following equipment:

- four 6.7 m diameter by 12.0 m long (22' by 39.5') ball mills, each mill having two 5.25 MW synchronous motors
- four hydrocyclone clusters, each with eight 710 mm diameter hydrocyclones
- two 17 m diameter by 17 high slurry holding tanks.

Each ball mill will be in closed-circuit with a cluster of eight 710 mm diameter hydrocyclones. The hydrocyclone underflow will gravity-flow to the ball mill feed chute, while the overflow with a solid density of 48% w/w will gravity-flow to the hydrocyclone overflow holding tank. There are two 17 m diameter by 17 m high agitated holding tanks at the mine site. Each tank will receive the hydrocyclone overflows from two grinding circuits. Each tank has a holding retention time of 55 minutes.

Two separate automatic ball charging systems will be provided to deliver grinding media to each ball mill feed chute, which will feed the hydrocyclone underflows to the mill.

Lime will be added to each mill as required.

Slurry Transportation

The mineralization slurry prepared from the mine site grinding facility will be delivered to the plant site by 3 stages of pumping through a 26 m-long tunnel, which has a cross section dimension of 4 m wide by 4.3 m high. Two pipelines will be used to transport the slurry. The key equipment is as follows:

- six 600 mm by 550 mm slurry pumps each with an installed power of 2,700 kW
- two 860 mm diameter rubber lined steel pipelines.

Secondary Primary – Grinding

The ground slurry from the mine site will be further ground to 80% passing 125 µm. The secondary grinding process will consist of two grinding lines, each with two closed grinding circuits.

The slurry from the mine site will be separately discharged into two 17 m diameter by 17 m high slurry holding tanks each equipped with a double impeller agitator. Each tank has a surge capacity of 3,800 m^3.

Ten tower mills, each driven by a 1,120 kW motor, will be used for the further grinding process. The mills will be operated together with four hydrocyclone clusters in closed circuits. The key equipment will include:

- 10 tower mills, each driven by an 1,120 kW motor
- six 500 mm by 450 mm centrifugal slurry pumps (4 in operation and 2 on standby), each equipped with 1,100 kW variable speed drive.

Reagents will be added to the primary ball mill cyclone feed pumpboxes or to the cyclone overflow collecting boxes.

GOLD, COPPER, AND MOLYBDENUM FLOTATION

Copper-Gold/Molybdenum Bulk Rougher/Scavenger Flotation

The cyclone overflow of the primary grinding circuit will feed 2 trains of 10 flotation cells, with each cell having a capacity of 200 m^3. The flotation reagents used include lime, A208, 3418A, and MIBC. A bulk copper-gold/molybdenum flotation concentrate of 6% by weight of the flotation feed will be reground and the flotation tailing will be sent to pyrite flotation circuit.

Copper-Gold/Molybdenum Bulk Concentrate Regrinding

The copper-gold/molybdenum bulk concentrate will be reground to a particle size of 80% passing 20 µm in a regrind circuit consisting of six 1,120 kW tower mills and a 250 mm diameter hydrocyclone cluster. The overflow of the hydrocyclones will gravity-flow to the bulk copper-gold/molybdenum cleaner circuit, while the underflow of the hydrocyclones will return to the regrinding mills by gravity.

Copper-Gold/Molybdenum Bulk Concentrate Cleaner Flotation

The hydrocyclone overflow will be cleaned in three stages. In the first stage of cleaner flotation, four 100 m^3 cells will be used; for the second and third stages, two 50 m^3 cells will be in used each stage. First cleaner tailing will be further floated in one cleaner scavenger flotation cell with a 100 m^3 capacity. The concentrate product will be sent to the first cleaner cells and the tailing will be sent to the CIL circuit. The tailing of the second and third flotation stages will be returned to the head of the preceding cleaner flotation circuit. Final copper-gold/molybdenum concentrate will be sent to copper-gold/molybdenum concentrate thickener.

The same reagents used in the primary flotation circuit will be employed in the cleaner circuit.

Copper-Gold and Molybdenum Separation

Depending on molybdenum content, the final copper-gold/molybdenum concentrate may be further processed to produce a copper-gold concentrate and a molybdenum concentrate. The separation will employ the conventional process, which will include copper suppression by sodium sulphide and multi-stages of molybdenum cleaner flotation and regrinding.

CONCENTRATE DEWATERING

The upgraded copper-gold cleaner concentrate will be thickened in a 17 m diameter high rate thickener. The thickener underflow will be directed to the copper-gold concentrate pressure filter to further reduce water content to 8% moisture. The copper-gold product will be stockpiled on site and then transported by road to a port site and by ocean to overseas smelters.

Average copper concentrate produced is estimated to be approximately 885 t/d or 320,000 t/a.

The molybdenum concentrate will be dewatered using a similar process as the copper-gold concentrate. The filtered concentrate will be further dewatered by a dryer to 5% moisture before being bagged and transported to processors.

GOLD RECOVERY FROM GOLD-BEARING PYRITE

Gold-Bearing Pyrite Flotation

The tailing of the copper-gold/molybdenum flotation circuits will be further floated in a pyrite flotation circuit. The pyrite rougher flotation will consist of two lines of seven pyrite rougher flotation cells. The capacity of each cell will be 200 m^3. Depending on gangue minerals content, the pyrite rougher concentrate may be further cleaned prior to subsequent processing.

Tailing from the pyrite rougher flotation will be pumped to the TMF located several kilometres southeast of the main process plant.

Gold Bearing Pyrite Concentrate Regrinding

The pyrite concentrate will be reground to a particle size of 80% passing 15 µm in six 1,120 kW tower mills. A 250 mm diameter hydrocyclone cluster will be incorporated with the mills in closed circuit. The reground materials will report to the gold leach circuit or the copper-pyrite separation circuit.

Depending on copper content, the reground materials may be subjected to a flotation process to separate copper minerals and pyrite. The copper product will be sent to the copper-gold/molybdenum cleaner flotation circuit while the pyrite product will report to the gold leach circuit.

Gold Leach

The reground gold-bearing pyrite product together with the first cleaner scavenger tailing from the copper-gold/molybdenum bulk flotation circuit will be thickened to a solids density of 60% in a 32 m-diameter high rate thickener.

The underflow of the thickener will be pumped to two pre-treatment tanks where the slurry will be diluted with barren solution and oxidized by aeration. Lime will be added to increase the slurry pH to approximately 11.

The pre-treated slurry will be leached by cyanidation to recover gold in a conventional CIL circuit. The leach circuit will consist of 6 agitated tanks, which are 15 m diameter by 15 m high. The tanks will be equipped with in-tank carbon transferring pumps and screens to advance the loaded carbon to the preceding leach tank.

The loaded carbon will be sent to the carbon stripping circuit while the leach residue will be sent to subsequent processes including residue washing, cyanide recovery, and cyanide destruction circuits.

Carbon Stripping and Reactivation

The loaded carbon will be treated by acid washing and the Zadra pressure stripping process for gold desorption. The stripping process will include the circulation of the barren solution through a heat recovery heat exchanger and a solution heater. The solution will then flow up through the bed of carbon and overflow near the top of the stripping vessel. The pregnant solution will be cooled by exchanging heat with the barren solution and will flow through a back pressure control valve to the pregnant solution holding tank for subsequent gold recovery by electrowinning. The barren solution from the electrowinning circuit will then return to the barren solution tank for recycle.

The stripping process will include barren and pregnant solution tanks, two acid wash vessels, two stripping vessels, two heating exchangers, and two solution heaters.

Prior to activation, the stripped carbon will be screened and dewatered. The activation will be carried out in a propane heated rotary reactivation kiln at a temperature of 700°C. The activated carbon will be circulated back into the CIL circuit after abrasion treatment and screen washing.

Gold Electrowinning and Refining

The pregnant solution from the elution system will be pumped from the pregnant solution stock tank through electrowinning cells where the gold will be deposited on stainless steel cathodes. The depleted solution will be subsequently reheated and returned to the stripping vessel. The electrowinning circuit will have a capacity to process 50 kg/d of gold doré bullion and will include 3.5 m^3 electrowinning cells, DC rectifiers, cathodes, anodes, and a pressure filter.

Periodically, the stainless steel cathodes will need to be cleaned to remove precious metal values by pressure washing. Cell muds will fall into the bottom of the electrowinning cells and be pumped through a pressure filter for dewatering on a batch basis. The filter cake will be transferred to the gold room for drying and smelting. An induction furnace will be used for gold refining. The area will be monitored by a security surveillance system.

TREATMENT OF LEACHING RESIDUES

Leach Residue Washing

The residues from the CIL circuit will be pumped to a CCD washing circuit. The CCD circuit will consist of two 32 m-diameter high-rate thickeners. The thickener overflow from the first stage washing will be pumped to cyanide recovery system. The underflow (washed residues) of the second thickener will be sent to cyanide destruction circuit prior to being pumped to the TMF.

Cyanide Recovery

The overflow of the first leach residues washing thickener will be sent to a cyanide recovery circuit where the cyanide will be recovered by an AVR process. The SART process will be an alternative for the cyanide recovery circuit.

The AVR cyanide recovery process will be conducted in a negative pressure system generated by a vacuum system.

The CCD overflow will be acidified by sulphuric acid. The solution will then be pumped to a volatilization tower, which provides a high liquid surface area to promote volatilization.

The gas phase will be directed through an absorption tank in which the caustic solution is circulated counter-current to the gas to re-absorb hydrogen cyanide. The regenerated cyanide solution will be returned to the leach circuit.

The cyanide-depleted solution from the volatilization tower will be alkalized by lime to a pH above 9.5 prior to being pumped to a 10-m clarifier. The metal species will precipitate in the clarifier while the clarified solution will be circulated to the leach residues washing circuit.

Cyanide Destruction

The remaining cyanide in the washed leach residues from the second washing thickener will be decomposed by a combined cyanide destruction process consisting of aeration, sulphur dioxide (SO_2) oxidation, and peroxide (H_2O_2) oxidation. The equipment used will include one 6 m diameter by 6 m high pre-aeration agitation tank, two 11 m diameter by 12 m high SO_2 oxidation tanks, and one 7 m diameter by 7.6 m high peroxide oxidation tank. Compressed air will be provided for the oxidation process.

TAILING MANAGEMENT

The flotation tailing and CIL residues will be pumped/gravity fed to the TMF located several kilometres southeast of the main process plant. The CIL residues will be deposited near the centre of tailing impoundment area and be covered with tailing pond water to prevent sulphide minerals oxidation. The residues will be eventually covered by the flotation tailing, from which most sulphides have been removed.

The supernatant from the tailing impoundment area will be reclaimed to the process water tank by two stages of pumping systems.

REAGENTS HANDLING

The reagents used in the process will include:

- **flotation:** PAX, 3418A, A208, fuel oil, A3302, methyl isobutyl carbinol (MIBC), lime (CaO), sodium sulphide (Na_2S) and sodium silicate (Na_2SiO_3)
- **CIL and gold recovery:** lime, sodium cyanide (NaCN), activated carbon, sodium hydroxide (NaOH), hydrochloric acid (HCl)
- **cyanide recovery and destruction reagents:** metabisulphite (MBS), copper sulphate ($CuSO_4$), hydrogen peroxide (H_2O_2), sulphuric acid (H_2SO_4), CaO, NaOH
- **others:** antiscalant.

All the reagents will be prepared in a separate reagent preparation and storage facility in a containment area. The reagent storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during operation. Appropriate ventilation and fire and safety protection will be provided at the facility.

The liquid reagents (including fuel oil, A208, 3418A, A3302, MIBC, H_2O_2, and antiscalant) will be added in the undiluted form to various process circuits via individual metering pumps.

The acids (including HCl and H_2SO_4) will be diluted with fresh water to 10% solution strength in the respective mixing tanks and stored in separate holding tanks prior to being added to various process circuits by metering pumps.

All the solid type reagents (including PAX, Na_2S, Na_2SiO_3, NaOH, NaCN, $CuSO_4$, and MBS) will be mixed with fresh water to 10% solution strength in the respective mixing tank, and stored in separate holding tanks before being added to various addition points by metering pumps.

The lime will be slaked, diluted into 15% solid milk of lime, and then distributed to various addition points through a closed pressure loop.

Flocculant will be dissolved, diluted to less than 0.5% strength, and then added to various thickener feed wells by metering pumps.

WATER SUPPLY

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

Fresh Water Supply System

Fresh and potable water will be supplied to a 12 m diameter by 9 m high storage tank from nearby wells and local drainage runoff areas. Fresh water will be used primarily for the following:

- fire water for emergency use
- cooling water for SAG mill motors and mill lubrication systems
- potable water supply
- reagent preparation.

By design, the fresh water tank will be full at all times and will provide at least 2 h of firewater in an emergency. The minimum fresh water requirement for process mill cooling and reagent preparation is on average estimated to be approximately 180 m^3/h.

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

Process Water Supply System

Process water will consist primarily of reclaimed water from the TMF, as well as fresh water and copper-gold/molybdenum concentrate thickener overflow. All the water will be directed to a 25 m diameter by 16 m high process water storage tank and then distributed by pumping to various distribution points. On average, total makeup water, including fresh water for process requirements, is estimated to be approximately 1,100 m^3/h. A preliminary estimate of the process requirements is shown in Table 16.25.

Table 16.25 Preliminary Estimate of Process Water Requirements

	Normal (m^3/h)	Average		Notes
		m^3/h	m^3/a	
Total Make-up Water Requirement	1,221	1,123	9,838,000	Including 1% water loss
Including Minimum Fresh Water Requirement	196	180	1,579,000	Reagent preparation and cooling water
Water Discharge To TMF	9,760	8,979	78,656,000	34% solid in flotation tailing and 60% solid in leach residues
Water Retained In TMF	1,345	1,237	10,837,000	Tailing in TMF containing 20% moisture
Total Process Water required from TMF	9,440	8,685	76,078,000	

Air Supply

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

- flotation circuits – low pressure air for flotation cells by air blowers
- leach circuits – high pressure air by dedicated air compressors
- cyanide recovery and destruction circuits – high pressure air by dedicated air compressors
- filtration circuit – high pressure air for filter pressing and drying of concentrate by dedicated air compressors
- crushing circuit – high pressure air for the dust suppression (fogging) system and other services by an air compressor
- plant air – high pressure air for various services by two air compressors
- instrumentation – instrument air will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

Assay and Metallurgical Laboratory

The assay laboratory will be equipped with necessary analytical instruments to provide routine assays for the mine, process, and environmental departments.

The metallurgical laboratory with laboratory equipment and instruments will undertake all necessary test work to monitor metallurgical performance and to improve the process flowsheet and efficiency.

PROCESS CONTROL AND INSTRUMENTATION

The plant control system will consist of a Distributed Control System (DCS) with PC-based Operator Interface Stations (OIS) located in three separate control rooms for crushing, flotation and cyanide leach areas. The plant control rooms will be staffed by trained personnel 24 h/d.

An automatic sampling system will collect samples from various streams for on-line analysis and the daily metallurgical balance.

Closed-circuit television (CCTV) cameras will be installed at various locations throughout the plant including the crushing facility, the stockpile conveyor discharge point, the ground slurry pumping tunnel, tailing piping tunnel, tailing facility, the concentrate handling building, and gold recovery facilities. The cameras will be monitored from the three control rooms.

16.3 RECOMMENDATIONS

Wardrop recommends:

- additional testwork to confirm that the use of HPGR in the grinding circuit will reduce operating costs for energy and grinding media
- an investigation of the optimum solid density of the primary grinding hydrocyclone overflow; a trade-off study is required to clarify the effect of the installation of thickeners to increase slurry solid density, prior to pumping the ground slurry to plant site on the operating costs and capital costs
- additional metallurgical test work and mineralization evaluations for each of the pit zones
- further study on the proposed cyanide recovery and destruction methods.

17.0 MINERAL RESOURCE ESTIMATE

This section has been taken from the RMI report entitled "Updated KSM Mineral Resources" dated March 30, 2009, which is available on SEDAR.

Mineral Resources were estimated for the Mitchell deposit by Mr. Michael J. Lechner, President of RMI. Mr. Lechner is a P.Geo. (British Columbia), a Registered Professional Geologist in the State of Arizona, and is a Certified Professional Geologist with the AIPG. These professional registrations together with Mr. Lechner's professional background and work experience allow him to be the QP for this report as per the requirements as set out by NI 43-101. Neither Mr. Lechner nor RMI have any vested interest in Seabridge securities or the property that is the subject of this technical report. Mr. Lechner and RMI have worked as an independent consultant for Seabridge since 2001.

The Sulphurets and Mitchell resource models were updated by RMI by including newly acquired drill hole data and geologic interpretations. No new data were collected for the Kerr deposit so that model was not updated. However, the February 2008 Kerr model was reblocked and translated into the same NAD83 coordinate system. Various statistical data are presented for the Kerr deposit as a matter of continuity. More information is provided regarding reblocking the Kerr model in Section 17.7.

17.1 GOLD GRADE DISTRIBUTION

The distribution of uncapped and capped raw gold assay grades is summarized at four different cutoff grades by selected lithologic and alteration types in Table 17.1 through to Table 17.6 for the Kerr, Sulphurets, and Mitchell deposits. Grade capping is discussed in Section 0.

As can be seen in Table 17.1 through to Table 17.6, the average gold grade increases going from the Kerr deposit (south part of the district) to the Sulphurets deposit (middle portion of district) and then to the Mitchell deposit. In addition to the gold grade increasing from south to north, the percentage of material above a 0.50 g/t gold cutoff also increases from Kerr (6%) to Sulphurets (17%) to Mitchell (36%). Another important statistical parameter is that the coefficient of variation (CV) decreases from 2.32 for uncapped Kerr assays to 0.95 for uncapped Mitchell gold assays. CVs less than 1.0 indicate that the gold assay population contains few high-grade outliers and that local grade estimation should be feasible.

Gold is seen to be distributed in a number of logged lithologic and alteration types at Kerr and Sulphurets. In the Mitchell deposit, approximately 64% of the contained gold metal is contained in four lithologic units: VATF, VU, VULT, and VUTF, respectively. Contained gold metal is spread throughout a number of logged alteration types at Mitchell.

In general, it has not been possible to identify any particular lithologic unit or alteration type that adequately defines a mineralized gold population for any of the KSM deposits. Quartz-sericite-pyrite alteration tends to be one of the key mineralized units but gold grades are seen to cross cut the various logged alteration types. Given these observations, RMI elected to use grade envelopes to constrain the estimate of block gold grades (see Section 17.5).

Table 17.1 Distribution of Gold by Lithology – Kerr

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	24,346	72	0.23	5,483	35.9	0.52	2.32	0.22	5,418	36.3	0.37	1.68
	0.25	6,865	21	0.51	3,515	31.2	0.92	1.79	0.50	3,450	31.6	0.61	1.22
	0.50	1,803	6	1.00	1,803	17.1	1.69	1.69	0.96	1,739	17.3	1.06	1.10
	1.00	368	2	2.35	867	15.8	3.41	1.45	2.18	802	14.8	1.90	0.87
DDAP	0.00	630	97	0.07	41	59.0	0.35	5.25	0.07	41	59.0	0.35	5.25
	0.25	19	2	0.91	17	10.1	1.80	1.98	0.91	17	10.1	1.80	1.98
	0.50	5	1	2.59	13	6.9	2.89	1.12	2.59	13	6.9	2.89	1.12
	1.00	1	0	7.25	10	24.0	0.00	0.00	7.25	10	24.0	0.00	0.00
DDPL	0.00	1,147	87	0.15	174	50.6	0.43	2.85	0.15	170	51.7	0.31	2.09
	0.25	155	10	0.55	86	20.3	1.08	1.95	0.53	82	20.8	0.71	1.35
	0.50	45	2	1.13	50	9.4	1.89	1.66	1.05	47	9.6	1.18	1.12
	1.00	19	2	1.83	34	19.7	2.76	1.51	1.63	30	17.9	1.64	1.01
DDRK	0.00	650	98	0.04	29	54.0	0.23	5.11	0.04	29	54.0	0.23	5.11
	0.25	16	2	0.82	13	13.4	1.20	1.45	0.82	13	13.4	1.20	1.45
	0.50	5	0	2.10	9	0.0	1.69	0.80	2.10	9	0.0	1.69	0.80
	1.00	5	1	2.10	9	32.6	1.69	0.80	2.10	9	32.6	1.69	0.80
FELS	0.00	1,010	66	0.22	222	40.0	0.16	0.72	0.22	222	40.0	0.16	0.72
	0.25	347	30	0.38	133	46.3	0.16	0.41	0.38	133	46.3	0.16	0.41
	0.50	46	4	0.67	31	11.6	0.25	0.38	0.67	31	11.6	0.25	0.38
	1.00	3	0	1.43	5	2.1	0.44	0.31	1.43	5	2.1	0.44	0.31

table continues…

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
INPP	0.00	1,898	65	0.27	510	28.8	0.53	1.99	0.27	505	29.1	0.47	1.78
	0.25	667	23	0.54	363	30.1	0.83	1.52	0.54	358	30.4	0.72	1.34
	0.50	230	10	0.91	210	23.9	1.33	1.46	0.89	205	24.1	1.14	1.28
	1.00	32	2	2.72	88	17.3	2.96	1.09	2.56	83	16.4	2.44	0.95
SCNG	0.00	1,475	72	0.21	304	44.1	0.21	1.00	0.21	304	44.1	0.21	1.00
	0.25	406	23	0.42	170	36.9	0.28	0.66	0.42	170	36.9	0.28	0.66
	0.50	68	4	0.85	58	12.3	0.47	0.55	0.85	58	12.3	0.47	0.55
	1.00	11	1	1.78	20	6.7	0.42	0.24	1.78	20	6.7	0.42	0.24
SSED	0.00	4,167	80	0.19	786	56.8	0.24	1.27	0.19	786	56.8	0.24	1.27
	0.25	838	17	0.41	339	29.8	0.46	1.13	0.41	339	29.8	0.46	1.13
	0.50	113	2	0.93	105	7.9	1.10	1.18	0.93	105	7.9	1.10	1.18
	1.00	19	0	2.32	44	5.5	2.21	0.95	2.32	44	5.5	2.21	0.95
SSST	0.00	942	84	0.17	156	62.9	0.16	0.96	0.17	156	62.9	0.16	0.96
	0.25	147	13	0.39	58	24.8	0.28	0.72	0.39	58	24.8	0.28	0.72
	0.50	24	2	0.82	19	7.7	0.52	0.63	0.82	19	7.7	0.52	0.63
	1.00	4	0	1.97	7	4.7	0.29	0.15	1.97	7	4.7	0.29	0.15
VHLP	0.00	3,209	78	0.21	674	39.7	0.59	2.79	0.21	660	40.5	0.50	2.43
	0.25	697	16	0.58	407	25.8	1.18	2.03	0.56	393	26.3	0.99	1.75
	0.50	184	5	1.27	233	13.9	2.16	1.70	1.19	219	14.2	1.77	1.48
	1.00	38	1	3.70	139	20.6	3.90	1.05	3.34	125	19.0	3.08	0.92
VLTH	0.00	417	63	0.42	177	15.0	0.96	2.25	0.42	175	15.2	0.90	2.14
	0.25	153	19	0.99	151	16.0	1.41	1.43	0.97	148	16.2	1.31	1.35
	0.50	72	10	1.69	122	16.5	1.80	1.07	1.66	120	16.7	1.65	0.99
	1.00	31	7	2.99	93	52.4	2.14	0.72	2.92	91	51.8	1.87	0.64

table continues…

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
VTLP	0.00	2,160	61	0.25	538	28.2	0.26	1.03	0.25	538	28.2	0.26	1.03
	0.25	842	29	0.46	386	40.2	0.29	0.64	0.46	386	40.2	0.29	0.64
	0.50	216	9	0.79	170	22.2	0.43	0.54	0.79	170	22.2	0.43	0.54
	1.00	32	1	1.58	50	9.4	0.64	0.40	1.58	50	9.4	0.64	0.40
VTUF	0.00	3,707	55	0.27	988	29.3	0.20	0.74	0.27	988	29.3	0.20	0.74
	0.25	1,677	36	0.42	699	45.5	0.20	0.47	0.42	699	45.5	0.20	0.47
	0.50	351	8	0.71	250	20.3	0.24	0.34	0.71	250	20.3	0.24	0.34
	1.00	39	1	1.25	49	5.0	0.26	0.20	1.25	49	5.0	0.26	0.20
VTXL	0.00	1,565	78	0.23	356	35.9	0.65	2.87	0.22	345	37.0	0.46	2.11
	0.25	345	15	0.66	228	21.4	1.29	1.96	0.63	218	22.1	0.86	1.37
	0.50	116	5	1.31	152	14.7	2.08	1.58	1.22	141	15.1	1.30	1.07
	1.00	35	2	2.81	100	28.0	3.30	1.17	2.51	89	25.8	1.76	0.70

Table 17.2 Distribution of Gold by Lithology – Sulphurets

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	13,719	48	0.46	6,340	11.9	0.76	1.64	0.46	6,303	12.0	0.71	1.54
	0.25	7,152	25	0.78	5,586	19.0	0.94	1.20	0.78	5,550	19.1	0.87	1.12
	0.50	3,747	17	1.17	4,380	26.4	1.17	1.00	1.16	4,343	26.5	1.06	0.91
	1.00	1,383	10	1.96	2,708	42.7	1.63	0.83	1.93	2,672	42.4	1.43	0.74
ANDS	0.00	1,890	38	0.48	901	10.7	0.60	1.26	0.48	901	10.7	0.60	1.26
	0.25	1,171	34	0.69	805	24.9	0.68	0.99	0.69	805	24.9	0.68	0.99
	0.50	524	17	1.11	580	25.0	0.84	0.76	1.11	580	25.0	0.84	0.76
	1.00	204	11	1.74	355	39.4	1.06	0.61	1.74	355	39.4	1.06	0.61
BVAT	0.00	338	41	0.73	248	7.0	1.58	2.15	0.70	238	7.3	1.35	1.92
	0.25	200	22	1.15	231	11.2	1.94	1.69	1.10	221	11.6	1.64	1.49
	0.50	126	21	1.61	203	20.8	2.33	1.45	1.53	193	21.6	1.94	1.26
	1.00	54	16	2.82	151	61.0	3.19	1.13	2.63	141	59.4	2.58	0.98
BVTF	0.00	245	33	0.68	167	8.0	0.88	1.29	0.68	167	8.0	0.88	1.29
	0.25	165	21	0.93	154	12.1	0.98	1.05	0.93	154	12.1	0.98	1.05
	0.50	112	27	1.19	134	28.5	1.10	0.92	1.19	134	28.5	1.10	0.92
	1.00	46	19	1.89	86	51.4	1.45	0.77	1.89	86	51.4	1.45	0.77
CCSD	0.00	382	67	0.27	104	32.8	0.40	1.47	0.27	104	32.8	0.40	1.47
	0.25	127	23	0.55	70	27.8	0.60	1.09	0.55	70	27.8	0.60	1.09
	0.50	39	8	1.06	41	19.1	0.89	0.84	1.06	41	19.1	0.89	0.84
	1.00	10	3	2.14	21	20.3	1.22	0.57	2.14	21	20.3	1.22	0.57

table continues…

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
DDRT	0.00	448	83	0.13	59	31.5	0.21	1.63	0.13	59	31.5	0.21	1.63
	0.25	76	9	0.53	40	22.6	0.25	0.47	0.53	40	22.6	0.25	0.47
	0.50	36	7	0.75	27	36.8	0.19	0.26	0.75	27	36.8	0.19	0.26
	1.00	5	1	1.17	5	9.0	0.12	0.10	1.17	5	9.0	0.12	0.10
IVOL	0.00	177	57	0.32	57	21.7	0.35	1.10	0.32	57	21.7	0.35	1.10
	0.25	77	27	0.58	45	29.4	0.40	0.69	0.58	45	29.4	0.40	0.69
	0.50	29	12	0.97	28	26.4	0.43	0.44	0.97	28	26.4	0.43	0.44
	1.00	8	5	1.56	13	22.5	0.33	0.21	1.56	13	22.5	0.33	0.21
PHBX	0.00	838	30	0.87	726	4.1	1.04	1.20	0.87	726	4.1	1.04	1.20
	0.25	587	16	1.19	696	6.8	1.09	0.92	1.19	696	6.8	1.09	0.92
	0.50	452	24	1.43	647	20.4	1.13	0.79	1.43	647	20.4	1.13	0.79
	1.00	254	30	1.97	499	68.8	1.28	0.65	1.97	499	68.8	1.28	0.65
PPFP	0.00	463	95	0.08	38	72.7	0.11	1.28	0.08	38	72.7	0.11	1.28
	0.25	25	4	0.41	10	16.7	0.17	0.41	0.41	10	16.7	0.17	0.41
	0.50	6	1	0.67	4	10.6	0.12	0.18	0.67	4	10.6	0.12	0.18
	1.00	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
PQMZ	0.00	1,030	38	0.58	596	6.6	0.84	1.46	0.58	596	6.6	0.84	1.46
	0.25	640	23	0.87	557	14.6	0.96	1.10	0.87	557	14.6	0.96	1.10
	0.50	404	25	1.16	470	30.8	1.10	0.95	1.16	470	30.8	1.10	0.95
	1.00	149	14	1.92	286	48.0	1.54	0.80	1.92	286	48.0	1.54	0.80
PSBX	0.00	366	24	1.00	365	3.2	1.80	1.81	0.95	347	3.4	1.36	1.43
	0.25	278	23	1.27	353	8.3	1.99	1.57	1.21	336	8.8	1.46	1.21
	0.50	193	27	1.67	323	18.4	2.28	1.36	1.58	305	19.4	1.62	1.02
	1.00	96	26	2.67	256	70.0	2.91	1.09	2.49	238	68.5	1.91	0.77

table continues…

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
SSLT	0.00	279	71	0.24	68	25.1	0.33	1.36	0.24	68	25.1	0.33	1.36
	0.25	80	14	0.63	51	20.7	0.39	0.62	0.63	51	20.7	0.39	0.62
	0.50	40	11	0.91	37	32.9	0.38	0.42	0.91	37	32.9	0.38	0.42
	1.00	10	4	1.46	14	21.2	0.34	0.24	1.46	14	21.2	0.34	0.24
VAAT	0.00	1,420	54	0.33	475	22.6	0.33	0.99	0.33	475	22.6	0.33	0.99
	0.25	653	28	0.56	368	29.8	0.37	0.66	0.56	368	29.8	0.37	0.66
	0.50	260	14	0.87	226	28.9	0.42	0.49	0.87	226	28.9	0.42	0.49
	1.00	59	4	1.51	89	18.7	0.45	0.30	1.51	89	18.7	0.45	0.30
VALT	0.00	233	49	0.54	125	13.2	0.58	1.08	0.54	125	13.2	0.58	1.08
	0.25	120	18	0.91	108	11.4	0.61	0.67	0.91	108	11.4	0.61	0.67
	0.50	79	17	1.19	94	21.7	0.56	0.47	1.19	94	21.7	0.56	0.47
	1.00	40	17	1.67	67	53.7	0.37	0.22	1.67	67	53.7	0.37	0.22
VATF	0.00	409	35	0.58	237	8.1	0.71	1.23	0.58	237	8.1	0.71	1.23
	0.25	267	29	0.81	218	17.4	0.79	0.97	0.81	218	17.4	0.79	0.97
	0.50	148	25	1.19	177	31.4	0.89	0.75	1.19	177	31.4	0.89	0.75
	1.00	48	12	2.12	102	43.1	1.06	0.50	2.12	102	43.1	1.06	0.50
VAXT	0.00	786	49	0.37	294	18.4	0.52	1.39	0.37	294	18.4	0.52	1.39
	0.25	405	32	0.59	240	28.9	0.65	1.09	0.59	240	28.9	0.65	1.09
	0.50	157	15	0.99	155	28.6	0.91	0.92	0.99	155	28.6	0.91	0.92
	1.00	36	5	1.96	71	24.1	1.51	0.77	1.96	71	24.1	1.51	0.77
VU	0.00	603	46	0.39	237	13.2	0.42	1.07	0.39	237	13.2	0.42	1.07
	0.25	323	28	0.64	205	26.2	0.44	0.70	0.64	205	26.2	0.44	0.70
	0.50	154	18	0.93	143	31.8	0.49	0.52	0.93	143	31.8	0.49	0.52
	1.00	43	7	1.58	68	28.8	0.46	0.29	1.58	68	28.8	0.46	0.29

table continues…

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
VUAT	0.00	826	36	0.46	377	11.9	0.38	0.84	0.46	377	11.9	0.38	0.84
	0.25	526	31	0.63	332	24.6	0.38	0.61	0.63	332	24.6	0.38	0.61
	0.50	271	23	0.88	239	35.4	0.38	0.43	0.88	239	35.4	0.38	0.43
	1.00	77	9	1.38	106	28.1	0.34	0.25	1.38	106	28.1	0.34	0.25
VUTF	0.00	827	30	0.59	488	6.7	0.68	1.15	0.59	488	6.7	0.68	1.15
	0.25	578	30	0.79	456	18.6	0.73	0.92	0.79	456	18.6	0.73	0.92
	0.50	329	26	1.11	365	32.0	0.83	0.75	1.11	365	32.0	0.83	0.75
	1.00	112	14	1.87	209	42.7	1.05	0.56	1.87	209	42.7	1.05	0.56
VUXT	0.00	496	48	0.42	209	15.7	0.73	1.73	0.42	209	15.7	0.73	1.73
	0.25	258	28	0.68	176	22.4	0.93	1.37	0.68	176	22.4	0.93	1.37
	0.50	119	18	1.08	129	29.7	1.26	1.16	1.08	129	29.7	1.26	1.16
	1.00	28	6	2.41	67	32.2	2.10	0.87	2.41	67	32.2	2.10	0.87

Table 17.3 Distribution of Gold by Lithology – Mitchell

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	39,424	27	0.53	20,888	5.9	0.50	0.95	0.53	20,736	6.0	0.43	0.81
	0.25	28,704	26	0.68	19,651	18.3	0.51	0.74	0.68	19,499	18.4	0.40	0.59
	0.50	18,391	36	0.86	15,832	47.8	0.56	0.65	0.85	15,680	48.2	0.41	0.48
	1.00	4,149	11	1.41	5,841	28.0	0.97	0.69	1.37	5,690	27.4	0.57	0.41
ANDS	0.00	1,763	59	0.31	540	18.2	0.60	1.96	0.29	517	19.0	0.36	1.23
	0.25	722	20	0.61	441	22.6	0.85	1.38	0.58	419	23.6	0.42	0.72
	0.50	378	19	0.84	319	41.3	1.12	1.32	0.78	297	43.1	0.49	0.62
	1.00	48	3	2.01	97	17.9	2.86	1.42	1.54	74	14.3	1.06	0.69
GRAN	0.00	534	96	0.08	44	68.2	0.18	2.17	0.08	44	68.2	0.18	2.17
	0.25	21	2	0.66	14	10.1	0.63	0.96	0.66	14	10.1	0.63	0.96
	0.50	9	1	1.08	9	7.9	0.80	0.74	1.08	9	7.9	0.80	0.74
	1.00	3	1	2.15	6	13.8	0.56	0.26	2.15	6	13.8	0.56	0.26
IVOL	0.00	888	18	0.69	617	3.1	0.78	1.12	0.67	597	3.2	0.54	0.81
	0.25	731	17	0.82	597	9.5	0.80	0.98	0.79	578	9.8	0.53	0.67
	0.50	577	51	0.93	539	54.0	0.87	0.93	0.90	519	55.7	0.55	0.61
	1.00	120	14	1.71	206	33.4	1.66	0.97	1.55	186	31.2	0.91	0.59
PMON	0.00	1,202	65	0.24	286	12.4	0.30	1.28	0.24	286	12.4	0.30	1.28
	0.25	421	14	0.60	250	22.4	0.25	0.42	0.60	250	22.4	0.25	0.42
	0.50	248	18	0.75	186	51.0	0.21	0.27	0.75	186	51.0	0.21	0.27
	1.00	35	3	1.14	40	14.1	0.17	0.15	1.14	40	14.1	0.17	0.15

table continues…

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
PPFP	0.00	402	73	0.19	75	40.1	0.17	0.91	0.19	75	40.1	0.17	0.91
	0.25	109	21	0.41	45	38.0	0.15	0.37	0.41	45	38.0	0.15	0.37
	0.50	26	6	0.64	16	22.0	0.13	0.21	0.64	16	22.0	0.13	0.21
	1.00	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
QTVN	0.00	457	5	0.90	411	0.9	0.52	0.58	0.90	411	0.9	0.52	0.58
	0.25	434	17	0.94	407	7.2	0.50	0.54	0.94	407	7.2	0.50	0.54
	0.50	355	44	1.07	378	35.5	0.47	0.44	1.07	378	35.5	0.47	0.44
	1.00	156	34	1.49	232	56.5	0.40	0.27	1.49	232	56.5	0.40	0.27
SARG	0.00	834	71	0.21	174	40.6	0.18	0.87	0.21	174	40.6	0.18	0.87
	0.25	243	24	0.43	103	39.9	0.20	0.46	0.43	103	39.9	0.20	0.46
	0.50	45	5	0.76	34	16.0	0.24	0.31	0.76	34	16.0	0.24	0.31
	1.00	5	1	1.28	6	3.5	0.18	0.14	1.28	6	3.5	0.18	0.14
SCHT	0.00	587	13	0.61	361	2.7	0.32	0.52	0.61	361	2.7	0.32	0.52
	0.25	511	24	0.69	351	15.1	0.28	0.41	0.69	351	15.1	0.28	0.41
	0.50	373	56	0.80	297	66.1	0.25	0.31	0.80	297	66.1	0.25	0.31
	1.00	43	7	1.34	58	16.0	0.26	0.19	1.34	58	16.0	0.26	0.19
SEDS	0.00	987	65	0.26	254	28.6	0.38	1.47	0.26	252	28.7	0.36	1.39
	0.25	347	25	0.52	181	33.8	0.54	1.03	0.52	180	34.0	0.50	0.96
	0.50	97	7	0.98	95	19.2	0.85	0.87	0.97	94	19.3	0.77	0.79
	1.00	25	3	1.87	47	18.4	1.32	0.70	1.81	45	17.9	1.14	0.63
UDEF	0.00	3,838	22	0.64	2,449	4.3	0.50	0.79	0.63	2,434	4.3	0.43	0.69
	0.25	2,995	16	0.78	2,344	9.9	0.48	0.61	0.78	2,329	9.9	0.38	0.49
	0.50	2,366	44	0.89	2,102	50.3	0.48	0.54	0.88	2,087	50.6	0.36	0.41
	1.00	659	17	1.32	870	35.5	0.73	0.55	1.30	855	35.1	0.44	0.34

table continues…

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
VAAT	0.00	146	17	0.57	82	3.4	0.30	0.53	0.57	82	3.4	0.30	0.53
	0.25	121	25	0.66	80	18.8	0.24	0.36	0.66	80	18.8	0.24	0.36
	0.50	85	53	0.76	64	65.3	0.22	0.29	0.76	64	65.3	0.22	0.29
	1.00	8	6	1.26	10	12.6	0.18	0.14	1.26	10	12.6	0.18	0.14
VALT	0.00	4,164	21	0.58	2,395	4.5	0.53	0.93	0.57	2,375	4.6	0.45	0.78
	0.25	3,294	29	0.69	2,286	18.8	0.54	0.78	0.69	2,267	19.0	0.43	0.62
	0.50	2,074	37	0.89	1,836	45.5	0.60	0.68	0.88	1,816	45.8	0.44	0.50
	1.00	521	13	1.44	747	31.2	0.99	0.69	1.40	728	30.6	0.60	0.43
VATF	0.00	6,198	18	0.56	3,501	4.6	0.39	0.69	0.56	3,501	4.6	0.39	0.69
	0.25	5,104	32	0.65	3,341	20.8	0.37	0.56	0.65	3,341	20.8	0.37	0.56
	0.50	3,135	40	0.83	2,615	48.6	0.37	0.44	0.83	2,615	48.6	0.37	0.44
	1.00	681	11	1.34	912	26.0	0.48	0.36	1.34	912	26.0	0.48	0.36
VAXT	0.00	698	50	0.39	271	17.8	0.39	1.01	0.39	271	17.8	0.39	1.01
	0.25	348	22	0.64	222	20.2	0.42	0.66	0.64	222	20.2	0.42	0.66
	0.50	193	21	0.87	168	37.4	0.44	0.51	0.87	168	37.4	0.44	0.51
	1.00	45	6	1.49	67	24.7	0.52	0.35	1.49	67	24.7	0.52	0.35
VU	0.00	6,673	17	0.62	4,155	3.9	0.46	0.74	0.62	4,140	3.9	0.43	0.70
	0.25	5,552	26	0.72	3,992	15.2	0.44	0.62	0.72	3,977	15.3	0.41	0.58
	0.50	3,845	44	0.87	3,360	50.3	0.45	0.52	0.87	3,345	50.4	0.41	0.47
	1.00	935	14	1.36	1,272	30.6	0.69	0.50	1.34	1,257	30.4	0.57	0.42
VULT	0.00	2,478	9	0.61	1,517	2.0	0.37	0.60	0.61	1,517	2.0	0.37	0.60
	0.25	2,265	34	0.66	1,487	21.3	0.35	0.54	0.66	1,487	21.3	0.35	0.54
	0.50	1,424	46	0.82	1,164	51.4	0.35	0.43	0.82	1,164	51.4	0.35	0.43
	1.00	284	11	1.35	385	25.4	0.44	0.33	1.35	385	25.4	0.44	0.33

table continues…

Lithology	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
VUTF	0.00	6,237	21	0.54	3,349	5.5	0.57	1.06	0.53	3,310	5.5	0.40	0.75
	0.25	4,942	33	0.64	3,166	22.8	0.60	0.93	0.63	3,128	23.0	0.39	0.61
	0.50	2,887	38	0.83	2,403	47.6	0.72	0.87	0.82	2,365	48.2	0.41	0.51
	1.00	539	9	1.50	808	24.1	1.47	0.98	1.43	769	23.2	0.62	0.44

Table 17.4 Distribution of Gold by Alteration – Kerr

Alteration	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	24,346	72	0.23	5,483	35.9	0.52	2.32	0.22	5,418	36.3	0.37	1.68
	0.25	6,865	21	0.51	3,515	31.2	0.92	1.79	0.50	3,450	31.6	0.61	1.22
	0.50	1,803	6	1.00	1,803	17.1	1.69	1.69	0.96	1,739	17.3	1.06	1.10
	1.00	368	2	2.35	867	15.8	3.41	1.45	2.18	802	14.8	1.90	0.87
CL	0.00	6,888	67	0.22	1,543	32.2	0.35	1.56	0.22	1,536	32.4	0.31	1.41
	0.25	2,246	25	0.47	1,046	37.7	0.53	1.13	0.46	1,038	37.9	0.45	0.98
	0.50	535	6	0.87	464	18.6	0.97	1.11	0.85	457	18.7	0.81	0.95
	1.00	102	1	1.74	177	11.5	1.97	1.13	1.67	170	11.1	1.59	0.95
EP	0.00	111	100	0.05	5	100.0	0.04	0.85	0.05	5	100.0	0.04	0.85
	0.25	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
	0.50	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
	1.00	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
KF	0.00	206	63	0.20	42	19.3	0.23	1.13	0.20	42	19.3	0.23	1.13
	0.25	76	25	0.44	34	43.4	0.21	0.47	0.44	34	43.4	0.21	0.47
	0.50	24	11	0.65	16	30.5	0.25	0.38	0.65	16	30.5	0.25	0.38
	1.00	2	1	1.44	3	6.9	0.00	0.00	1.44	3	6.9	0.00	0.00
MY	0.00	1,964	84	0.18	353	65.0	0.28	1.55	0.18	353	65.0	0.28	1.55
	0.25	314	14	0.39	124	24.7	0.64	1.63	0.39	124	24.7	0.64	1.63
	0.50	35	2	1.04	36	5.3	1.78	1.71	1.04	36	5.3	1.78	1.71
	1.00	5	0	3.90	18	5.0	3.91	1.00	3.90	18	5.0	3.91	1.00

table continues…

Alteration	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
QP	0.00	493	14	0.54	266	4.0	0.65	1.21	0.53	262	4.1	0.47	0.88
	0.25	424	40	0.60	255	27.1	0.68	1.13	0.59	251	27.5	0.48	0.81
	0.50	228	41	0.81	183	51.2	0.88	1.09	0.79	179	51.9	0.58	0.74
	1.00	26	5	1.80	47	17.7	2.34	1.30	1.65	43	16.5	1.40	0.85
QS	0.00	3,666	75	0.22	823	42.6	0.44	1.98	0.22	813	43.1	0.37	1.67
	0.25	903	18	0.52	472	26.9	0.82	1.57	0.51	462	27.2	0.66	1.29
	0.50	237	5	1.06	250	14.6	1.47	1.39	1.02	241	14.7	1.15	1.13
	1.00	55	1	2.40	131	15.9	2.65	1.11	2.22	121	14.9	1.94	0.87
SE	0.00	8,947	73	0.21	1,852	40.3	0.33	1.61	0.21	1,848	40.4	0.32	1.54
	0.25	2,426	21	0.46	1,105	34.7	0.56	1.23	0.45	1,101	34.8	0.53	1.16
	0.50	517	5	0.89	462	14.9	1.10	1.23	0.89	458	14.9	1.03	1.16
	1.00	75	1	2.47	186	10.0	2.31	0.93	2.42	182	9.8	2.10	0.87
SI	0.00	780	74	0.38	294	15.7	1.90	5.03	0.34	265	17.4	0.80	2.35
	0.25	204	10	1.22	248	9.0	3.58	2.94	1.07	219	10.0	1.30	1.21
	0.50	127	9	1.74	222	16.6	4.45	2.55	1.51	192	18.4	1.49	0.98
	1.00	61	8	2.86	173	58.8	6.26	2.19	2.37	144	54.2	1.79	0.75
UDEF	0.00	1,184	78	0.23	275	22.1	0.81	3.49	0.22	264	23.0	0.62	2.76
	0.25	258	14	0.83	214	21.7	1.59	1.92	0.79	203	22.6	1.15	1.45
	0.50	92	5	1.68	155	12.9	2.45	1.45	1.57	144	13.4	1.66	1.06
	1.00	37	3	3.19	119	43.4	3.30	1.03	2.90	108	41.0	1.93	0.67

Table 17.5 Distribution of Gold by Alteration – Sulphurets

Alteration	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	13,719	48	0.46	6,340	11.9	0.76	1.64	0.46	6,303	12.0	0.71	1.54
	0.25	7,152	25	0.78	5,586	19.0	0.94	1.20	0.78	5,550	19.1	0.87	1.12
	0.50	3,747	17	1.17	4,380	26.4	1.17	1.00	1.16	4,343	26.5	1.06	0.91
	1.00	1,383	10	1.96	2,708	42.7	1.63	0.83	1.93	2,672	42.4	1.43	0.74
CL	0.00	202	46	0.39	79	15.7	0.37	0.95	0.39	79	15.7	0.37	0.95
	0.25	108	31	0.61	66	28.9	0.38	0.62	0.61	66	28.9	0.38	0.62
	0.50	46	16	0.95	44	31.8	0.36	0.38	0.95	44	31.8	0.36	0.38
	1.00	13	6	1.47	19	23.6	0.22	0.15	1.47	19	23.6	0.22	0.15
IARG	0.00	155	59	0.22	34	22.4	0.19	0.87	0.22	34	22.4	0.19	0.87
	0.25	63	30	0.42	26	47.0	0.13	0.31	0.42	26	47.0	0.13	0.31
	0.50	18	11	0.59	10	30.6	0.06	0.10	0.59	10	30.6	0.06	0.10
	1.00	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
KP	0.00	260	83	0.16	43	45.9	0.24	1.48	0.16	43	45.9	0.24	1.48
	0.25	44	11	0.53	23	22.8	0.42	0.79	0.53	23	22.8	0.42	0.79
	0.50	14	4	0.94	13	15.5	0.52	0.55	0.94	13	15.5	0.52	0.55
	1.00	4	2	1.64	7	15.8	0.48	0.30	1.64	7	15.8	0.48	0.30
PR	0.00	4,433	64	0.26	1,168	26.5	0.39	1.46	0.26	1,164	26.6	0.36	1.36
	0.25	1,583	23	0.54	858	30.9	0.54	0.99	0.54	854	31.0	0.48	0.89
	0.50	553	10	0.90	498	25.8	0.78	0.87	0.89	494	25.9	0.68	0.76
	1.00	113	3	1.74	196	16.8	1.43	0.83	1.70	192	16.5	1.17	0.69

table continues…

Alteration	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
PSBX	0.00	188	18	0.65	123	4.2	0.64	0.97	0.65	123	4.2	0.64	0.97
	0.25	155	34	0.76	118	19.3	0.65	0.86	0.76	118	19.3	0.65	0.86
	0.50	92	33	1.02	94	34.3	0.74	0.72	1.02	94	34.3	0.74	0.72
	1.00	30	16	1.71	52	42.1	0.95	0.56	1.71	52	42.1	0.95	0.56
QA	0.00	360	31	0.54	193	8.4	0.68	1.27	0.54	193	8.4	0.68	1.27
	0.25	250	28	0.71	177	19.2	0.75	1.07	0.71	177	19.2	0.75	1.07
	0.50	148	35	0.94	140	46.2	0.90	0.96	0.94	140	46.2	0.90	0.96
	1.00	23	6	2.23	51	26.2	1.81	0.81	2.23	51	26.2	1.81	0.81
QB	0.00	241	9	0.83	200	1.8	0.57	0.69	0.83	200	1.8	0.57	0.69
	0.25	219	19	0.90	197	9.2	0.56	0.62	0.90	197	9.2	0.56	0.62
	0.50	173	44	1.03	178	37.5	0.56	0.54	1.03	178	37.5	0.56	0.54
	1.00	68	28	1.53	103	51.5	0.59	0.39	1.53	103	51.5	0.59	0.39
QP	0.00	2,285	32	0.60	1,367	7.9	0.89	1.48	0.59	1,357	8.0	0.83	1.39
	0.25	1,558	30	0.81	1,259	17.9	1.01	1.24	0.80	1,249	18.1	0.93	1.16
	0.50	870	25	1.16	1,014	29.4	1.23	1.06	1.15	1,004	29.6	1.13	0.98
	1.00	302	13	2.02	611	44.7	1.79	0.88	1.99	601	44.3	1.59	0.80
SI	0.00	1,163	13	1.06	1,231	1.9	1.35	1.27	1.04	1,213	1.9	1.17	1.12
	0.25	1,014	22	1.19	1,207	8.0	1.40	1.17	1.17	1,190	8.1	1.20	1.02
	0.50	754	31	1.47	1,109	21.5	1.52	1.03	1.45	1,091	21.8	1.28	0.89
	1.00	393	34	2.15	844	68.6	1.86	0.86	2.11	826	68.1	1.50	0.71
SIH	0.00	389	59	0.35	135	17.3	0.82	2.38	0.34	133	17.4	0.78	2.27
	0.25	161	25	0.69	112	24.8	1.19	1.73	0.68	110	25.1	1.12	1.64
	0.50	63	10	1.24	78	19.6	1.78	1.43	1.22	77	19.8	1.66	1.36
	1.00	25	6	2.10	52	38.4	2.61	1.24	2.05	50	37.7	2.42	1.18

table continues…

Alteration	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
SIL	0.00	781	69	0.29	226	29.8	0.60	2.07	0.29	226	29.8	0.60	2.07
	0.25	239	21	0.66	159	23.8	0.98	1.47	0.66	159	23.8	0.98	1.47
	0.50	76	5	1.37	105	11.8	1.50	1.09	1.37	105	11.8	1.50	1.09
	1.00	36	5	2.19	78	34.7	1.88	0.86	2.19	78	34.7	1.88	0.86
UDEF	0.00	2,984	44	0.49	1,460	9.7	0.73	1.48	0.49	1,456	9.7	0.71	1.45
	0.25	1,659	26	0.79	1,318	18.5	0.86	1.08	0.79	1,315	18.5	0.83	1.05
	0.50	898	18	1.17	1,049	26.5	1.02	0.88	1.16	1,045	26.5	0.99	0.85
	1.00	356	12	1.86	663	45.4	1.35	0.73	1.85	659	45.2	1.28	0.69

Table 17.6 Distribution of Gold by Alteration – Mitchell

Alteration	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	39,424	27	0.53	20,888	5.9	0.50	0.95	0.53	20,736	6.0	0.43	0.81
	0.25	28,704	26	0.68	19,651	18.3	0.51	0.74	0.68	19,499	18.4	0.40	0.59
	0.50	18,391	36	0.86	15,832	47.8	0.56	0.65	0.85	15,680	48.2	0.41	0.48
	1.00	4,149	11	1.41	5,841	28.0	0.97	0.69	1.37	5,690	27.4	0.57	0.41
CL	0.00	6,385	14	0.60	3,838	3.1	0.45	0.75	0.60	3,818	3.1	0.39	0.64
	0.25	5,474	28	0.68	3,721	17.9	0.44	0.65	0.68	3,701	18.0	0.36	0.53
	0.50	3,655	45	0.83	3,034	52.1	0.47	0.57	0.82	3,014	52.3	0.35	0.43
	1.00	766	12	1.35	1,035	27.0	0.80	0.59	1.33	1,016	26.6	0.46	0.35
CL2	0.00	6,120	10	0.59	3,628	2.8	0.51	0.86	0.59	3,586	2.8	0.37	0.64
	0.25	5,503	36	0.64	3,527	22.8	0.52	0.81	0.63	3,485	23.0	0.36	0.58
	0.50	3,277	43	0.82	2,701	50.5	0.60	0.73	0.81	2,659	51.1	0.38	0.46
	1.00	619	10	1.40	867	23.9	1.20	0.86	1.33	825	23.0	0.58	0.43
IARG	0.00	5,080	20	0.55	2,790	5.4	0.37	0.67	0.55	2,790	5.4	0.37	0.67
	0.25	4,062	28	0.65	2,640	18.9	0.34	0.52	0.65	2,640	18.9	0.34	0.52
	0.50	2,661	44	0.79	2,113	56.2	0.34	0.42	0.79	2,113	56.2	0.34	0.42
	1.00	405	8	1.35	546	19.6	0.54	0.40	1.35	546	19.6	0.54	0.40
KP	0.00	698	39	0.42	292	7.9	0.42	1.00	0.42	292	7.9	0.42	1.00
	0.25	423	29	0.64	269	25.7	0.41	0.64	0.64	269	25.7	0.41	0.64
	0.50	221	23	0.88	194	37.9	0.44	0.50	0.88	194	37.9	0.44	0.50
	1.00	57	8	1.45	83	28.5	0.49	0.34	1.45	83	28.5	0.49	0.34

table continues…

Alteration	Au Cutoff (g/t)	Total Metres	Inc. Percent	Uncapped Au Statistics Above Cutoff					Capped Au Statistics Above Cutoff				
				Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	CV
PR	0.00	3,740	34	0.46	1,730	7.9	0.47	1.01	0.46	1,715	7.9	0.39	0.85
	0.25	2,464	27	0.65	1,593	21.5	0.48	0.74	0.64	1,578	21.7	0.36	0.56
	0.50	1,454	30	0.84	1,222	45.1	0.54	0.64	0.83	1,207	45.5	0.36	0.43
	1.00	334	9	1.33	442	25.6	0.95	0.71	1.28	427	24.9	0.48	0.37
QSP	0.00	5,498	21	0.66	3,622	4.4	0.70	1.06	0.65	3,569	4.4	0.52	0.80
	0.25	4,355	23	0.80	3,464	13.0	0.72	0.91	0.78	3,410	13.2	0.50	0.64
	0.50	3,087	38	0.97	2,993	41.4	0.79	0.82	0.95	2,940	42.0	0.50	0.53
	1.00	1,007	18	1.48	1,494	41.2	1.22	0.82	1.43	1,441	40.4	0.63	0.44
QSTW	0.00	2,814	14	0.69	1,931	2.8	0.56	0.81	0.68	1,915	2.8	0.48	0.71
	0.25	2,416	24	0.78	1,877	12.8	0.55	0.71	0.77	1,861	12.9	0.46	0.60
	0.50	1,752	44	0.93	1,630	47.1	0.57	0.62	0.92	1,614	47.4	0.46	0.50
	1.00	499	18	1.44	721	37.3	0.85	0.59	1.41	706	36.8	0.59	0.41
SIH	0.00	2,431	74	0.21	500	33.3	0.36	1.76	0.20	496	33.5	0.34	1.68
	0.25	622	18	0.54	333	29.8	0.60	1.11	0.53	330	30.0	0.55	1.04
	0.50	182	5	1.01	184	17.9	0.94	0.93	1.00	181	18.1	0.85	0.86
	1.00	49	2	1.92	95	18.9	1.45	0.75	1.85	91	18.4	1.28	0.69
SIL	0.00	1,053	66	0.29	302	23.3	0.40	1.41	0.28	299	23.6	0.36	1.27
	0.25	354	12	0.65	232	15.0	0.53	0.80	0.65	228	15.2	0.43	0.67
	0.50	225	18	0.83	186	42.7	0.59	0.71	0.81	183	43.1	0.46	0.56
	1.00	39	4	1.47	57	19.0	1.19	0.81	1.39	54	18.1	0.85	0.61
UDEF	0.00	3,921	46	0.41	1,604	12.5	0.38	0.94	0.41	1,604	12.5	0.38	0.94
	0.25	2,112	20	0.66	1,403	18.0	0.36	0.54	0.66	1,403	18.0	0.36	0.54
	0.50	1,319	27	0.84	1,114	46.5	0.34	0.40	0.84	1,114	46.5	0.34	0.40
	1.00	271	7	1.36	367	22.9	0.40	0.29	1.36	367	22.9	0.40	0.29

17.2 COPPER GRADE DISTRIBUTION

The distribution of uncapped and capped raw copper assay grades is summarized at four different cutoff grades by selected lithologic and alteration types in Table 17.7 through to Table 17.12 for the Kerr, Sulphurets, and Mitchell deposits. Grade capping is discussed in Section 0.

As can be seen in Table 17.7 through to Table 17.12, the average copper grade decreases going from the Kerr deposit (south part of the district) to the Sulphurets deposit (middle portion of district) to the Mitchell deposit. This is the opposite relationship shown by gold. In the Sulphurets deposit, about 24% of the assays are above a 0.25% copper cutoff while only 15% of the Kerr and Mitchell copper assays are above 0.25%. The CV decreases in going from Kerr (1.19) to Mitchell (0.85).

Like gold, copper is seen to be distributed in a number of logged lithologic and alteration types at Kerr and Sulphurets. In general, it has not been possible to identify any particular lithologic unit or alteration type that adequately defines a mineralized copper population for any of the KSM deposits. Copper grades tend to be somewhat lower in chlorite-propylytic alteration than quartz-sericite-pyrite alteration but this relationship is not well developed. Given these observations, RMI elected to use grade envelopes to constrain the estimate of block copper grades (see Section 17.5).

Table 17.7 Distribution of Copper by Lithology – Kerr

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	24,204	36	0.32	7,797	4.2	0.38	1.19	0.32	7,782	4.2	0.38	1.17
	0.10	15,604	20	0.48	7,472	10.5	0.40	0.83	0.48	7,456	10.5	0.39	0.81
	0.25	10,648	15	0.62	6,652	14.8	0.41	0.65	0.62	6,637	14.9	0.39	0.63
	0.40	6,960	29	0.79	5,495	70.5	0.42	0.53	0.79	5,479	70.4	0.40	0.50
DDAP	0.00	630	81	0.06	39	29.7	0.11	1.78	0.06	39	29.7	0.11	1.78
	0.10	121	14	0.23	28	34.0	0.17	0.74	0.23	28	34.0	0.17	0.74
	0.25	32	3	0.44	14	13.4	0.20	0.45	0.44	14	13.4	0.20	0.45
	0.40	14	2	0.64	9	23.0	0.15	0.23	0.64	9	23.0	0.15	0.23
DDPL	0.00	1,147	63	0.15	174	13.0	0.28	1.82	0.15	172	13.1	0.26	1.72
	0.10	422	18	0.36	151	19.1	0.37	1.04	0.35	150	19.3	0.34	0.95
	0.25	221	10	0.53	118	20.4	0.45	0.83	0.53	116	20.6	0.39	0.74
	0.40	105	9	0.78	82	47.5	0.54	0.69	0.77	81	47.1	0.46	0.59
DDRK	0.00	650	92	0.03	18	35.2	0.09	3.13	0.03	18	35.2	0.09	3.13
	0.10	53	7	0.22	12	36.8	0.22	1.00	0.22	12	36.8	0.22	1.00
	0.25	10	1	0.53	5	11.0	0.38	0.72	0.53	5	11.0	0.38	0.72
	0.40	3	1	0.93	3	17.0	0.42	0.45	0.93	3	17.0	0.42	0.45
FELS	0.00	1,010	36	0.31	314	4.5	0.34	1.08	0.31	314	4.5	0.34	1.08
	0.10	647	19	0.46	300	9.6	0.33	0.72	0.46	300	9.6	0.33	0.72
	0.25	459	15	0.59	270	14.6	0.32	0.54	0.59	270	14.6	0.32	0.54
	0.40	313	31	0.72	224	71.3	0.31	0.43	0.72	224	71.3	0.31	0.43

table continues…

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
INPP	0.00	1,898	22	0.54	1,030	1.8	0.52	0.96	0.54	1,029	1.8	0.52	0.95
	0.10	1,480	16	0.68	1,012	5.1	0.51	0.74	0.68	1,011	5.1	0.50	0.73
	0.25	1,171	14	0.82	960	7.9	0.48	0.59	0.82	958	7.9	0.48	0.59
	0.40	914	48	0.96	879	85.3	0.46	0.47	0.96	877	85.3	0.45	0.47
SCNG	0.00	1,475	44	0.19	283	10.5	0.19	0.99	0.19	283	10.5	0.19	0.99
	0.10	826	27	0.31	253	23.1	0.19	0.60	0.31	253	23.1	0.19	0.60
	0.25	432	14	0.43	188	22.8	0.17	0.39	0.43	188	22.8	0.17	0.39
	0.40	222	15	0.55	123	43.6	0.16	0.29	0.55	123	43.6	0.16	0.29
SSED	0.00	4,167	37	0.22	918	8.1	0.23	1.02	0.22	917	8.1	0.22	1.02
	0.10	2,634	31	0.32	843	22.1	0.23	0.72	0.32	843	22.1	0.23	0.71
	0.25	1,361	16	0.47	641	23.2	0.23	0.49	0.47	640	23.2	0.23	0.48
	0.40	683	16	0.63	428	46.6	0.24	0.38	0.62	427	46.6	0.23	0.37
SSST	0.00	942	50	0.15	143	13.3	0.16	1.03	0.15	143	13.3	0.16	1.03
	0.10	472	26	0.26	124	27.3	0.15	0.59	0.26	124	27.3	0.15	0.59
	0.25	228	17	0.37	85	34.1	0.16	0.42	0.37	85	34.1	0.16	0.42
	0.40	69	7	0.52	36	25.4	0.20	0.39	0.52	36	25.4	0.20	0.39
VHLP	0.00	3,209	32	0.29	940	5.8	0.28	0.97	0.29	940	5.8	0.28	0.97
	0.10	2,168	23	0.41	885	13.2	0.28	0.68	0.41	885	13.2	0.28	0.68
	0.25	1,428	16	0.53	761	16.9	0.27	0.50	0.53	761	16.9	0.27	0.50
	0.40	907	28	0.66	602	64.1	0.25	0.38	0.66	602	64.1	0.25	0.38
VLTH	0.00	275	79	0.09	25	21.2	0.17	1.89	0.09	25	21.2	0.17	1.89
	0.10	59	11	0.34	20	19.5	0.25	0.73	0.34	20	19.5	0.25	0.73
	0.25	28	4	0.53	15	12.9	0.23	0.44	0.53	15	12.9	0.23	0.44
	0.40	18	7	0.64	12	46.4	0.22	0.35	0.64	12	46.4	0.22	0.35

table continues…

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
VTLP	0.00	2,160	15	0.48	1,038	1.4	0.45	0.94	0.48	1,032	1.4	0.43	0.91
	0.10	1,831	22	0.56	1,023	8.2	0.45	0.80	0.56	1,017	8.3	0.43	0.77
	0.25	1,346	19	0.70	938	12.1	0.45	0.64	0.69	931	12.2	0.42	0.60
	0.40	945	44	0.86	812	78.2	0.44	0.51	0.85	806	78.1	0.40	0.47
VTUF	0.00	3,707	7	0.53	1,968	0.6	0.38	0.72	0.53	1,967	0.6	0.38	0.72
	0.10	3,459	13	0.57	1,956	4.3	0.37	0.65	0.57	1,956	4.3	0.37	0.65
	0.25	2,975	24	0.63	1,871	14.6	0.36	0.57	0.63	1,871	14.6	0.36	0.57
	0.40	2,089	56	0.76	1,585	80.5	0.36	0.47	0.76	1,585	80.5	0.36	0.47
VTXL	0.00	1,565	43	0.24	368	5.4	0.29	1.21	0.24	368	5.4	0.29	1.21
	0.10	889	22	0.39	348	15.5	0.29	0.75	0.39	348	15.5	0.29	0.75
	0.25	551	15	0.53	291	19.5	0.30	0.56	0.53	291	19.5	0.30	0.56
	0.40	320	20	0.69	219	59.6	0.30	0.44	0.69	219	59.6	0.30	0.44

Table 17.8 Distribution of Copper by Lithology – Sulphurets

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	12,390	37	0.16	1,974	5.0	0.22	1.37	0.16	1,965	5.0	0.21	1.32
	0.05	7,840	19	0.24	1,875	8.7	0.24	1.01	0.24	1,866	8.7	0.23	0.95
	0.10	5,473	24	0.31	1,704	24.0	0.26	0.82	0.31	1,695	24.1	0.24	0.77
	0.25	2,512	20	0.49	1,231	62.3	0.29	0.58	0.49	1,222	62.2	0.25	0.52
ANDS	0.00	1,412	18	0.24	344	1.8	0.24	0.97	0.24	344	1.8	0.24	0.97
	0.05	1,155	14	0.29	338	4.4	0.23	0.80	0.29	338	4.4	0.23	0.80
	0.10	953	31	0.34	323	22.1	0.23	0.69	0.34	323	22.1	0.23	0.69
	0.25	512	36	0.48	247	71.7	0.23	0.48	0.48	247	71.7	0.23	0.48
BVAT	0.00	318	51	0.17	53	6.6	0.24	1.46	0.17	53	6.6	0.24	1.46
	0.05	154	11	0.32	49	4.3	0.28	0.86	0.32	49	4.3	0.28	0.86
	0.10	118	17	0.40	47	17.7	0.27	0.67	0.40	47	17.7	0.27	0.67
	0.25	65	20	0.58	38	71.3	0.23	0.40	0.58	38	71.3	0.23	0.40
BVTF	0.00	172	29	0.23	40	2.9	0.24	1.05	0.23	40	2.9	0.24	1.05
	0.05	123	8	0.32	39	2.8	0.24	0.77	0.32	39	2.8	0.24	0.77
	0.10	109	25	0.35	38	18.3	0.24	0.70	0.35	38	18.3	0.24	0.70
	0.25	66	38	0.46	30	76.0	0.25	0.54	0.46	30	76.0	0.25	0.54
BVXT	0.00	72	2	0.14	10	0.5	0.09	0.64	0.14	10	0.5	0.09	0.64
	0.05	71	42	0.14	10	23.4	0.09	0.63	0.14	10	23.4	0.09	0.63
	0.10	41	43	0.19	8	45.3	0.09	0.50	0.19	8	45.3	0.09	0.50
	0.25	10	13	0.32	3	30.8	0.09	0.27	0.32	3	30.8	0.09	0.27

table continues…

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
CCSD	0.00	382	32	0.09	33	13.0	0.06	0.73	0.09	33	13.0	0.06	0.73
	0.05	258	35	0.11	29	29.0	0.06	0.56	0.11	29	29.0	0.06	0.56
	0.10	123	30	0.16	19	48.2	0.06	0.41	0.16	19	48.2	0.06	0.41
	0.25	10	3	0.34	3	9.8	0.09	0.26	0.34	3	9.8	0.09	0.26
DDRT	0.00	444	74	0.05	21	20.6	0.08	1.83	0.05	21	20.6	0.08	1.83
	0.05	117	15	0.14	16	22.9	0.12	0.88	0.14	16	22.9	0.12	0.88
	0.10	50	8	0.23	12	27.8	0.14	0.60	0.23	12	27.8	0.14	0.60
	0.25	14	3	0.42	6	28.7	0.12	0.30	0.42	6	28.7	0.12	0.30
IVOL	0.00	177	40	0.10	17	12.7	0.08	0.81	0.10	17	12.7	0.08	0.81
	0.05	106	22	0.14	15	16.7	0.07	0.51	0.14	15	16.7	0.07	0.51
	0.10	66	33	0.18	12	56.9	0.06	0.32	0.18	12	56.9	0.06	0.32
	0.25	8	5	0.29	2	13.6	0.02	0.07	0.29	2	13.6	0.02	0.07
PHBX	0.00	721	44	0.10	74	9.3	0.16	1.53	0.10	74	9.3	0.16	1.53
	0.05	404	26	0.17	67	17.2	0.19	1.12	0.17	67	17.2	0.19	1.12
	0.10	215	20	0.25	55	30.4	0.22	0.87	0.25	55	30.4	0.22	0.87
	0.25	68	9	0.47	32	43.1	0.29	0.60	0.47	32	43.1	0.29	0.60
PQMZ	0.00	842	30	0.27	229	2.5	0.27	0.98	0.27	229	2.5	0.27	0.98
	0.05	591	8	0.38	224	1.9	0.25	0.67	0.38	224	1.9	0.25	0.67
	0.10	527	18	0.42	219	11.6	0.24	0.58	0.42	219	11.6	0.24	0.58
	0.25	374	44	0.52	193	84.1	0.22	0.42	0.52	193	84.1	0.22	0.42
PSBX	0.00	366	48	0.16	58	4.7	0.22	1.38	0.16	58	4.7	0.22	1.38
	0.05	190	7	0.29	55	3.2	0.23	0.81	0.29	55	3.2	0.23	0.81
	0.10	164	20	0.33	53	20.5	0.23	0.72	0.33	53	20.5	0.23	0.72
	0.25	89	24	0.47	42	71.6	0.24	0.50	0.47	42	71.6	0.24	0.50

table continues…

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
VAAT	0.00	1,285	18	0.16	205	3.2	0.17	1.05	0.16	205	3.2	0.17	1.05
	0.05	1,051	30	0.19	198	13.8	0.17	0.91	0.19	198	13.8	0.17	0.91
	0.10	664	32	0.26	170	31.9	0.18	0.72	0.26	170	31.9	0.18	0.72
	0.25	247	19	0.43	105	51.2	0.21	0.49	0.43	105	51.2	0.21	0.49
VALT	0.00	168	13	0.23	39	1.4	0.19	0.82	0.23	39	1.4	0.19	0.82
	0.05	146	7	0.26	38	2.4	0.18	0.70	0.26	38	2.4	0.18	0.70
	0.10	134	49	0.28	37	37.7	0.18	0.66	0.28	37	37.7	0.18	0.66
	0.25	51	31	0.44	23	58.5	0.21	0.47	0.44	23	58.5	0.21	0.47
VATF	0.00	271	24	0.12	34	7.7	0.13	1.02	0.12	34	7.7	0.13	1.02
	0.05	206	42	0.15	31	24.9	0.14	0.89	0.15	31	24.9	0.14	0.89
	0.10	93	21	0.24	23	25.4	0.16	0.64	0.24	23	25.4	0.16	0.64
	0.25	35	13	0.40	14	42.1	0.15	0.36	0.40	14	42.1	0.15	0.36
VAXT	0.00	719	29	0.16	117	4.5	0.18	1.14	0.16	117	4.5	0.18	1.14
	0.05	512	23	0.22	112	9.7	0.19	0.88	0.22	112	9.7	0.19	0.88
	0.10	350	28	0.29	100	27.1	0.20	0.69	0.29	100	27.1	0.20	0.69
	0.25	148	21	0.46	69	58.7	0.19	0.41	0.46	69	58.7	0.19	0.41
VU	0.00	603	39	0.16	98	5.0	0.25	1.56	0.16	98	5.0	0.25	1.56
	0.05	368	18	0.25	93	8.7	0.29	1.15	0.25	93	8.7	0.29	1.15
	0.10	257	26	0.33	85	25.3	0.32	0.96	0.33	85	25.3	0.32	0.96
	0.25	98	16	0.61	60	61.1	0.37	0.60	0.61	60	61.1	0.37	0.60
VUAT	0.00	820	36	0.18	147	5.2	0.22	1.22	0.18	147	5.2	0.22	1.22
	0.05	525	17	0.27	139	6.6	0.23	0.87	0.27	139	6.6	0.23	0.87
	0.10	388	23	0.33	129	20.1	0.23	0.70	0.33	129	20.1	0.23	0.70
	0.25	202	25	0.50	100	68.1	0.22	0.44	0.50	100	68.1	0.22	0.44

table continues…

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
VUTF	0.00	789	29	0.19	152	3.4	0.26	1.36	0.19	150	3.4	0.24	1.27
	0.05	557	21	0.26	147	8.3	0.28	1.07	0.26	145	8.4	0.26	0.98
	0.10	395	26	0.34	134	22.1	0.30	0.89	0.33	132	22.3	0.27	0.80
	0.25	193	24	0.52	100	66.3	0.35	0.66	0.51	99	65.9	0.29	0.57
VUXT	0.00	496	42	0.16	79	6.8	0.22	1.37	0.16	79	6.8	0.22	1.37
	0.05	286	19	0.26	73	8.8	0.24	0.95	0.26	73	8.8	0.24	0.95
	0.10	192	17	0.35	66	16.7	0.25	0.73	0.35	66	16.7	0.25	0.73
	0.25	109	22	0.49	53	67.7	0.25	0.51	0.49	53	67.7	0.25	0.51

Table 17.9 Distribution of Copper by Lithology – Mitchell

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	39,421	15	0.15	6,099	2.1	0.13	0.85	0.15	6,054	2.1	0.12	0.78
	0.05	33,467	19	0.18	5,974	9.6	0.13	0.72	0.18	5,929	9.6	0.11	0.65
	0.10	25,871	51	0.21	5,390	52.9	0.13	0.64	0.21	5,345	53.3	0.11	0.56
	0.25	5,962	15	0.36	2,162	35.4	0.20	0.55	0.36	2,117	35.0	0.15	0.43
ANDS	0.00	1,763	8	0.16	276	0.8	0.13	0.81	0.16	275	0.8	0.12	0.79
	0.05	1,621	28	0.17	273	14.0	0.12	0.74	0.17	273	14.1	0.12	0.72
	0.10	1,134	50	0.21	235	49.0	0.13	0.63	0.21	234	49.1	0.13	0.61
	0.25	259	15	0.38	100	36.2	0.17	0.44	0.38	99	36.0	0.15	0.40
GRAN	0.00	534	42	0.08	43	8.9	0.08	1.02	0.08	43	8.9	0.08	1.02
	0.05	311	25	0.13	39	23.7	0.08	0.65	0.13	39	23.7	0.08	0.65
	0.10	178	30	0.16	29	54.2	0.09	0.56	0.16	29	54.2	0.09	0.56
	0.25	16	3	0.37	6	13.2	0.19	0.52	0.37	6	13.2	0.19	0.52
IVOL	0.00	888	2	0.19	172	0.4	0.09	0.48	0.19	172	0.4	0.09	0.48
	0.05	868	12	0.20	172	4.9	0.09	0.47	0.20	172	4.9	0.09	0.47
	0.10	761	63	0.21	163	56.1	0.09	0.40	0.21	163	56.1	0.09	0.40
	0.25	203	23	0.33	66	38.5	0.08	0.23	0.33	66	38.5	0.08	0.23
PMON	0.00	1,202	33	0.11	127	7.5	0.09	0.85	0.11	127	7.5	0.09	0.85
	0.05	810	23	0.15	118	15.7	0.09	0.59	0.15	118	15.7	0.09	0.59
	0.10	536	38	0.18	98	55.6	0.08	0.45	0.18	98	55.6	0.08	0.45
	0.25	78	7	0.34	27	21.2	0.09	0.25	0.34	27	21.2	0.09	0.25

table continues…

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
PPFP	0.00	402	57	0.06	22	12.3	0.07	1.23	0.06	22	12.3	0.07	1.23
	0.05	174	24	0.11	20	30.9	0.07	0.62	0.11	20	30.9	0.07	0.62
	0.10	79	18	0.16	13	46.5	0.08	0.49	0.16	13	46.5	0.08	0.49
	0.25	7	2	0.35	2	10.3	0.13	0.37	0.35	2	10.3	0.13	0.37
QTVN	0.00	457	1	0.27	123	0.1	0.15	0.56	0.27	123	0.1	0.15	0.56
	0.05	452	5	0.27	123	1.6	0.15	0.55	0.27	123	1.6	0.15	0.55
	0.10	428	50	0.28	121	33.5	0.15	0.52	0.28	121	33.5	0.15	0.52
	0.25	199	44	0.40	80	64.8	0.14	0.35	0.40	80	64.8	0.14	0.34
SARG	0.00	834	40	0.07	61	15.9	0.07	0.89	0.07	61	15.9	0.07	0.89
	0.05	498	39	0.10	52	39.0	0.07	0.67	0.10	52	39.0	0.07	0.67
	0.10	173	19	0.16	28	36.0	0.09	0.58	0.16	28	36.0	0.09	0.58
	0.25	13	2	0.41	6	9.1	0.16	0.38	0.41	6	9.1	0.16	0.38
SCHT	0.00	584	15	0.17	99	0.3	0.09	0.55	0.17	99	0.3	0.09	0.55
	0.05	495	2	0.20	98	0.9	0.06	0.33	0.20	98	0.9	0.06	0.33
	0.10	485	68	0.20	97	72.1	0.06	0.31	0.20	97	72.1	0.06	0.31
	0.25	85	15	0.31	26	26.7	0.05	0.17	0.31	26	26.7	0.05	0.17
SEDS	0.00	987	63	0.07	69	15.6	0.10	1.45	0.07	69	15.6	0.10	1.44
	0.05	362	14	0.16	59	13.8	0.12	0.75	0.16	59	13.9	0.12	0.74
	0.10	228	17	0.21	49	38.4	0.12	0.58	0.21	49	38.4	0.12	0.58
	0.25	60	6	0.38	22	32.2	0.14	0.38	0.38	22	32.2	0.14	0.37
UDEF	0.00	3,838	19	0.16	614	2.5	0.11	0.72	0.16	614	2.5	0.11	0.72
	0.05	3,098	13	0.19	599	6.2	0.10	0.53	0.19	599	6.2	0.10	0.53
	0.10	2,604	49	0.22	561	50.9	0.10	0.45	0.22	561	51.0	0.10	0.45
	0.25	739	19	0.34	248	40.4	0.09	0.28	0.34	248	40.4	0.09	0.28

table continues…

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
VAAT	0.00	146	15	0.11	16	3.1	0.05	0.48	0.11	16	3.1	0.05	0.48
	0.05	123	18	0.13	15	14.8	0.04	0.32	0.13	15	14.8	0.04	0.32
	0.10	97	64	0.13	13	75.1	0.04	0.29	0.13	13	75.1	0.04	0.29
	0.25	4	3	0.28	1	7.0	0.03	0.10	0.28	1	7.0	0.03	0.10
VALT	0.00	4,164	4	0.15	645	0.9	0.08	0.54	0.15	645	0.9	0.08	0.54
	0.05	3,984	23	0.16	640	11.8	0.08	0.50	0.16	640	11.8	0.08	0.50
	0.10	3,011	59	0.19	563	60.6	0.07	0.40	0.19	563	60.6	0.07	0.40
	0.25	552	13	0.31	173	26.7	0.06	0.20	0.31	173	26.7	0.06	0.20
VATF	0.00	6,198	9	0.16	971	1.0	0.10	0.64	0.16	971	1.0	0.10	0.64
	0.05	5,663	21	0.17	961	10.5	0.10	0.56	0.17	961	10.5	0.10	0.56
	0.10	4,348	55	0.20	858	55.9	0.09	0.47	0.20	858	55.9	0.09	0.47
	0.25	937	15	0.34	316	32.6	0.09	0.28	0.34	316	32.6	0.09	0.28
VAXT	0.00	698	48	0.08	57	9.6	0.08	1.04	0.08	57	9.6	0.08	1.04
	0.05	363	21	0.14	51	18.4	0.08	0.55	0.14	51	18.4	0.08	0.55
	0.10	218	26	0.19	41	52.0	0.07	0.37	0.19	41	52.0	0.07	0.37
	0.25	37	5	0.31	11	20.0	0.05	0.16	0.31	11	20.0	0.05	0.16
VU	0.00	6,673	9	0.16	1,076	1.4	0.10	0.60	0.16	1,075	1.4	0.09	0.59
	0.05	6,074	17	0.17	1,060	8.2	0.09	0.52	0.17	1,059	8.2	0.09	0.51
	0.10	4,910	59	0.20	973	60.3	0.09	0.43	0.20	971	60.4	0.08	0.42
	0.25	997	15	0.32	323	30.1	0.10	0.29	0.32	322	30.0	0.08	0.26
VULT	0.00	2,478	2	0.19	476	0.4	0.11	0.59	0.19	476	0.4	0.11	0.59
	0.05	2,417	15	0.20	474	6.0	0.11	0.57	0.20	474	6.0	0.11	0.57
	0.10	2,047	62	0.22	446	55.0	0.11	0.50	0.22	446	55.0	0.11	0.50
	0.25	512	21	0.36	184	38.6	0.12	0.34	0.36	184	38.6	0.12	0.34

table continues…

Lithology	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
VUTF	0.00	6,237	15	0.17	1,039	2.3	0.19	1.14	0.16	1,003	2.4	0.15	0.91
	0.05	5,312	21	0.19	1,015	9.7	0.20	1.02	0.18	979	10.0	0.15	0.79
	0.10	4,014	47	0.23	914	44.8	0.21	0.93	0.22	879	46.4	0.15	0.70
	0.25	1,067	17	0.42	449	43.2	0.34	0.80	0.39	413	41.2	0.21	0.55

Table 17.10 Distribution of Copper by Alteration – Kerr

Alteration	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	24,204	36	0.32	7,797	4.2	0.38	1.19	0.32	7,782	4.2	0.38	1.17
	0.10	15,604	20	0.48	7,472	10.5	0.40	0.83	0.48	7,456	10.5	0.39	0.81
	0.25	10,648	15	0.62	6,652	14.8	0.41	0.65	0.62	6,637	14.9	0.39	0.63
	0.40	6,960	29	0.79	5,495	70.5	0.42	0.53	0.79	5,479	70.4	0.40	0.50
CL	0.00	6,888	27	0.41	2,810	1.9	0.41	1.00	0.41	2,810	1.9	0.41	1.00
	0.10	5,028	15	0.55	2,758	6.3	0.39	0.72	0.55	2,757	6.3	0.39	0.72
	0.25	4,006	17	0.64	2,582	13.5	0.38	0.60	0.64	2,581	13.5	0.38	0.60
	0.40	2,821	41	0.78	2,203	78.4	0.38	0.49	0.78	2,203	78.4	0.38	0.49
EP	0.00	111	89	0.04	4	56.4	0.05	1.26	0.04	4	56.4	0.05	1.26
	0.10	12	11	0.16	2	43.6	0.04	0.23	0.16	2	43.6	0.04	0.23
	0.25	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
	0.40	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
KF	0.00	206	62	0.12	25	22.2	0.12	1.04	0.12	25	22.2	0.12	1.04
	0.10	79	23	0.24	19	32.2	0.12	0.50	0.24	19	32.2	0.12	0.50
	0.25	32	11	0.36	11	28.1	0.12	0.32	0.36	11	28.1	0.12	0.32
	0.40	8	4	0.53	4	17.6	0.10	0.19	0.53	4	17.6	0.10	0.19
MY	0.00	1,964	41	0.21	405	10.3	0.24	1.18	0.21	405	10.3	0.24	1.17
	0.10	1,149	32	0.32	364	25.0	0.27	0.85	0.32	363	25.0	0.26	0.83
	0.25	513	12	0.51	262	17.5	0.30	0.59	0.51	262	17.5	0.29	0.57
	0.40	284	14	0.67	191	47.2	0.32	0.48	0.67	191	47.1	0.31	0.46

table continues…

Alteration	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
QP	0.00	493	3	1.09	536	0.1	0.65	0.60	1.07	527	0.1	0.59	0.55
	0.10	480	4	1.11	535	0.7	0.64	0.57	1.10	527	0.7	0.57	0.52
	0.25	459	4	1.16	532	1.0	0.62	0.54	1.14	523	1.0	0.55	0.48
	0.40	441	90	1.19	526	98.3	0.61	0.51	1.17	518	98.2	0.53	0.46
QS	0.00	3,666	35	0.28	1,020	5.4	0.31	1.13	0.28	1,020	5.4	0.31	1.13
	0.10	2,392	25	0.40	965	14.6	0.33	0.81	0.40	965	14.6	0.33	0.81
	0.25	1,490	16	0.55	817	18.5	0.34	0.61	0.55	817	18.5	0.34	0.61
	0.40	889	24	0.71	628	61.6	0.35	0.50	0.71	628	61.6	0.35	0.50
SE	0.00	8,816	32	0.31	2,724	4.6	0.34	1.09	0.31	2,723	4.6	0.34	1.09
	0.10	6,012	24	0.43	2,597	13.0	0.34	0.80	0.43	2,596	13.0	0.34	0.80
	0.25	3,883	18	0.58	2,244	17.8	0.35	0.61	0.58	2,243	17.8	0.35	0.60
	0.40	2,320	26	0.76	1,758	64.6	0.35	0.47	0.76	1,757	64.5	0.35	0.46
SI	0.00	780	82	0.13	104	20.3	0.44	3.29	0.13	99	21.4	0.29	2.27
	0.10	140	6	0.59	83	6.6	0.90	1.52	0.55	78	6.9	0.49	0.88
	0.25	91	3	0.84	76	6.0	1.04	1.24	0.78	71	6.4	0.47	0.60
	0.40	71	9	0.98	70	67.1	1.13	1.16	0.90	65	65.3	0.45	0.50
UDEF	0.00	1,173	79	0.12	139	13.5	0.31	2.59	0.12	139	13.5	0.31	2.59
	0.10	247	11	0.49	120	13.5	0.52	1.07	0.49	120	13.5	0.52	1.07
	0.25	121	3	0.84	101	7.4	0.56	0.67	0.84	101	7.4	0.56	0.67
	0.40	88	7	1.03	91	65.6	0.53	0.52	1.03	91	65.6	0.53	0.52

Table 17.11 Distribution of Copper by Alteration – Sulphurets

Alteration	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	12,390	37	0.16	1,974	5.0	0.22	1.37	0.16	1,965	5.0	0.21	1.32
	0.05	7,840	19	0.24	1,875	8.7	0.24	1.01	0.24	1,866	8.7	0.23	0.95
	0.10	5,473	24	0.31	1,704	24.0	0.26	0.82	0.31	1,695	24.1	0.24	0.77
	0.25	2,512	20	0.49	1,231	62.3	0.29	0.58	0.49	1,222	62.2	0.25	0.52
CL	0.00	202	10	0.21	43	0.9	0.17	0.77	0.21	43	0.9	0.17	0.77
	0.05	182	14	0.24	43	5.0	0.16	0.68	0.24	43	5.0	0.16	0.68
	0.10	154	44	0.26	41	32.1	0.16	0.60	0.26	41	32.1	0.16	0.60
	0.25	66	33	0.41	27	62.0	0.14	0.35	0.41	27	62.0	0.14	0.35
IARG	0.00	155	44	0.09	15	7.0	0.12	1.28	0.09	15	7.0	0.12	1.28
	0.05	86	27	0.16	13	22.6	0.13	0.83	0.16	13	22.6	0.13	0.83
	0.10	44	20	0.23	10	31.3	0.15	0.64	0.23	10	31.3	0.15	0.64
	0.25	14	9	0.41	6	39.2	0.13	0.31	0.41	6	39.2	0.13	0.31
KP	0.00	256	45	0.09	24	10.5	0.09	0.98	0.09	24	10.5	0.09	0.98
	0.05	141	19	0.15	22	15.2	0.09	0.56	0.15	22	15.2	0.09	0.56
	0.10	92	27	0.19	18	43.6	0.08	0.41	0.19	18	43.6	0.08	0.41
	0.25	24	9	0.31	7	30.8	0.04	0.12	0.31	7	30.8	0.04	0.12
PR	0.00	4,366	36	0.12	508	7.2	0.13	1.13	0.12	508	7.2	0.13	1.13
	0.05	2,783	25	0.17	472	15.7	0.14	0.82	0.17	472	15.7	0.14	0.82
	0.10	1,683	26	0.23	392	35.2	0.15	0.64	0.23	392	35.2	0.15	0.64
	0.25	536	12	0.40	213	41.8	0.16	0.40	0.40	213	41.8	0.16	0.40

table continues…

Alteration	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
PSBX	0.00	188	10	0.32	59	0.7	0.26	0.81	0.32	59	0.7	0.26	0.81
	0.05	170	8	0.35	59	1.9	0.25	0.72	0.35	59	1.9	0.25	0.72
	0.10	154	30	0.38	58	15.5	0.25	0.65	0.38	58	15.5	0.25	0.65
	0.25	98	52	0.50	49	82.0	0.23	0.47	0.50	49	82.0	0.23	0.47
QA	0.00	336	15	0.29	96	1.5	0.26	0.91	0.29	96	1.5	0.26	0.91
	0.05	285	17	0.33	95	4.1	0.26	0.77	0.33	95	4.1	0.26	0.77
	0.10	228	24	0.40	91	14.1	0.25	0.62	0.40	91	14.1	0.25	0.62
	0.25	146	43	0.53	77	80.3	0.22	0.41	0.53	77	80.3	0.22	0.41
QB	0.00	241	0	0.60	145	0.0	0.40	0.66	0.59	143	0.0	0.37	0.63
	0.05	241	3	0.60	145	0.4	0.40	0.66	0.59	143	0.4	0.37	0.63
	0.10	233	11	0.62	144	3.4	0.39	0.63	0.61	143	3.4	0.37	0.60
	0.25	206	85	0.67	139	96.2	0.38	0.56	0.67	138	96.1	0.35	0.53
QP	0.00	2,110	39	0.15	317	5.8	0.20	1.30	0.15	316	5.8	0.19	1.27
	0.05	1,283	19	0.23	298	9.3	0.21	0.91	0.23	298	9.3	0.21	0.89
	0.10	874	21	0.31	269	22.8	0.22	0.71	0.31	268	22.9	0.21	0.69
	0.25	421	20	0.47	197	62.1	0.22	0.48	0.47	196	62.0	0.21	0.44
SI	0.00	868	36	0.21	179	4.4	0.25	1.21	0.21	179	4.4	0.25	1.21
	0.05	558	19	0.31	171	6.2	0.26	0.85	0.31	171	6.2	0.26	0.85
	0.10	392	14	0.41	160	10.7	0.25	0.61	0.41	160	10.7	0.25	0.61
	0.25	272	31	0.52	141	78.8	0.23	0.43	0.52	141	78.8	0.23	0.43
SIH	0.00	389	36	0.14	55	5.5	0.20	1.40	0.14	55	5.5	0.20	1.40
	0.05	250	22	0.21	52	11.9	0.22	1.06	0.21	52	11.9	0.22	1.06
	0.10	165	28	0.27	45	32.5	0.24	0.89	0.27	45	32.5	0.24	0.89
	0.25	54	14	0.50	27	50.1	0.31	0.63	0.50	27	50.1	0.31	0.63

table continues…

Alteration	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
SIL	0.00	781	37	0.13	105	6.4	0.13	0.99	0.13	105	6.4	0.13	0.99
	0.05	490	13	0.20	98	7.2	0.13	0.64	0.20	98	7.2	0.13	0.64
	0.10	386	35	0.24	91	46.6	0.12	0.53	0.24	91	46.6	0.12	0.53
	0.25	113	14	0.37	42	39.7	0.15	0.41	0.37	42	39.7	0.15	0.41
UDEF	0.00	2,262	46	0.18	402	4.6	0.29	1.64	0.17	396	4.7	0.26	1.48
	0.05	1,231	12	0.31	384	4.6	0.34	1.10	0.31	377	4.7	0.29	0.95
	0.10	970	19	0.38	365	16.9	0.36	0.96	0.37	358	17.2	0.30	0.81
	0.25	539	24	0.55	297	73.9	0.40	0.73	0.54	290	73.4	0.31	0.57

Table 17.12 Distribution of Copper by Alteration – Mitchell

Alteration	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
All Data	0.00	39,421	15	0.15	6,099	2.1	0.13	0.85	0.15	6,054	2.1	0.12	0.78
	0.05	33,467	19	0.18	5,974	9.6	0.13	0.72	0.18	5,929	9.6	0.11	0.65
	0.10	25,871	51	0.21	5,390	52.9	0.13	0.64	0.21	5,345	53.3	0.11	0.56
	0.25	5,962	15	0.36	2,162	35.4	0.20	0.55	0.36	2,117	35.0	0.15	0.43
CL	0.00	6,385	5	0.17	1,060	0.9	0.10	0.60	0.17	1,060	0.9	0.10	0.60
	0.05	6,085	21	0.17	1,051	9.9	0.10	0.56	0.17	1,051	9.9	0.10	0.56
	0.10	4,713	57	0.20	946	55.5	0.09	0.46	0.20	946	55.5	0.09	0.46
	0.25	1,071	17	0.33	358	33.7	0.09	0.28	0.33	358	33.7	0.09	0.28
CL2	0.00	6,120	1	0.17	1,069	0.2	0.09	0.50	0.17	1,068	0.2	0.09	0.49
	0.05	6,075	15	0.18	1,067	7.2	0.09	0.49	0.18	1,066	7.2	0.09	0.49
	0.10	5,135	69	0.19	990	64.1	0.08	0.43	0.19	990	64.2	0.08	0.42
	0.25	930	15	0.33	305	28.5	0.09	0.28	0.33	305	28.5	0.09	0.27
IARG	0.00	5,080	18	0.12	623	3.8	0.09	0.73	0.12	623	3.8	0.09	0.73
	0.05	4,158	29	0.14	599	17.8	0.08	0.58	0.14	599	17.8	0.08	0.58
	0.10	2,708	45	0.18	488	55.7	0.08	0.46	0.18	488	55.7	0.08	0.46
	0.25	431	8	0.33	141	22.7	0.10	0.30	0.33	141	22.7	0.10	0.30
KP	0.00	698	25	0.13	91	5.0	0.12	0.91	0.13	91	5.0	0.12	0.91
	0.05	524	27	0.17	86	15.0	0.12	0.71	0.17	86	15.0	0.12	0.71
	0.10	338	36	0.22	73	44.2	0.12	0.56	0.22	73	44.2	0.12	0.56
	0.25	88	13	0.37	33	35.7	0.13	0.36	0.37	33	35.7	0.13	0.36

table continues…

Alteration	Cu Cutoff (%)	Total Metres	Inc. Percent	Uncapped Cu Statistics Above Cutoff					Capped Cu Statistics Above Cutoff				
				Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	CV
PR	0.00	3,740	18	0.14	523	2.6	0.10	0.69	0.14	523	2.6	0.10	0.69
	0.05	3,071	17	0.17	510	9.5	0.09	0.53	0.17	509	9.5	0.09	0.53
	0.10	2,438	53	0.19	460	59.9	0.08	0.45	0.19	460	59.9	0.08	0.44
	0.25	448	12	0.33	147	28.0	0.09	0.27	0.33	147	28.0	0.09	0.27
QSP	0.00	5,498	15	0.20	1,112	1.6	0.23	1.12	0.20	1,073	1.7	0.18	0.93
	0.05	4,700	15	0.23	1,094	5.4	0.23	1.00	0.22	1,055	5.6	0.18	0.80
	0.10	3,892	44	0.27	1,033	37.3	0.24	0.91	0.26	994	38.6	0.18	0.72
	0.25	1,466	27	0.42	619	55.7	0.34	0.80	0.40	580	54.1	0.23	0.59
QSTW	0.00	2,814	9	0.21	594	1.0	0.14	0.67	0.21	590	1.0	0.13	0.63
	0.05	2,569	7	0.23	588	2.4	0.14	0.59	0.23	584	2.4	0.12	0.55
	0.10	2,382	58	0.24	574	48.7	0.13	0.56	0.24	570	49.1	0.12	0.51
	0.25	761	27	0.37	285	47.9	0.16	0.44	0.37	280	47.5	0.14	0.37
SIH	0.00	2,431	31	0.10	238	5.7	0.10	0.99	0.10	238	5.7	0.10	0.99
	0.05	1,679	32	0.13	225	25.1	0.10	0.73	0.13	225	25.1	0.10	0.72
	0.10	910	32	0.18	165	46.2	0.11	0.61	0.18	165	46.2	0.11	0.61
	0.25	143	6	0.38	55	23.0	0.15	0.40	0.38	55	23.0	0.15	0.40
SIL	0.00	1,053	29	0.12	128	4.7	0.10	0.84	0.12	128	4.7	0.10	0.79
	0.05	749	16	0.16	122	10.4	0.09	0.58	0.16	122	10.4	0.08	0.51
	0.10	580	46	0.19	109	60.1	0.09	0.50	0.19	109	60.3	0.08	0.42
	0.25	98	9	0.33	32	24.9	0.14	0.44	0.32	31	24.6	0.09	0.27
UDEF	0.00	3,921	37	0.10	401	5.6	0.10	0.99	0.10	400	5.6	0.10	0.98
	0.05	2,467	20	0.15	378	14.2	0.09	0.62	0.15	378	14.2	0.09	0.61
	0.10	1,689	36	0.19	321	55.4	0.09	0.49	0.19	321	55.5	0.09	0.48
	0.25	289	7	0.34	99	24.7	0.12	0.35	0.34	99	24.6	0.11	0.33

17.3 ASSAY GRADE CAPPING

RMI used cumulative probability plots to identify high-grade outliers for both gold and copper assays. Figure 17.1 through to Figure 17.6 show cumulative probability plots using the cumulative normal distribution function.

Figure 17.1 Kerr Au Assay Cumulative Probability Plot

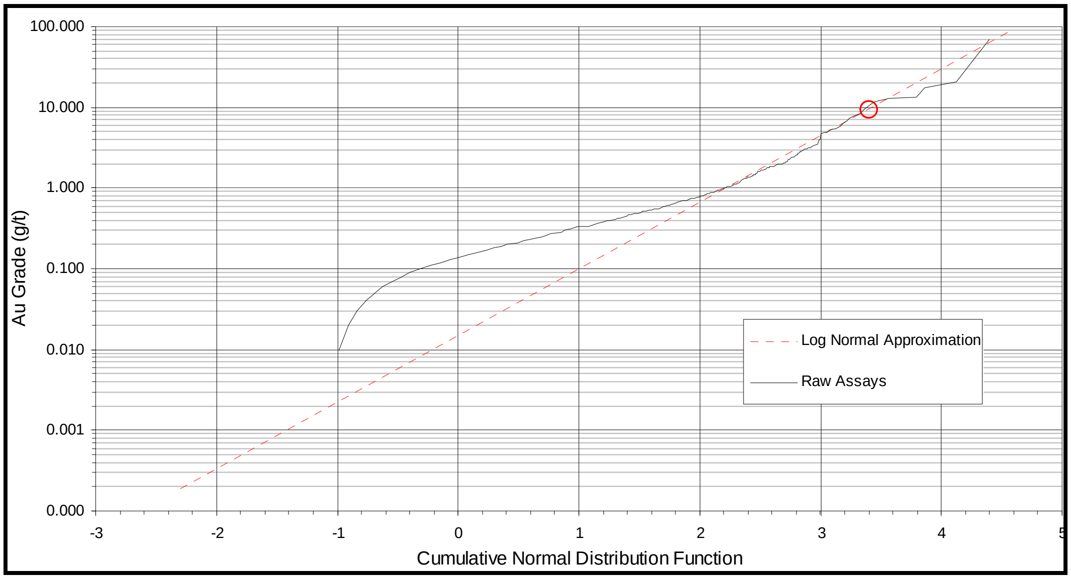


Figure 17.2 Sulphurets Au Assay Cumulative Probability Plot

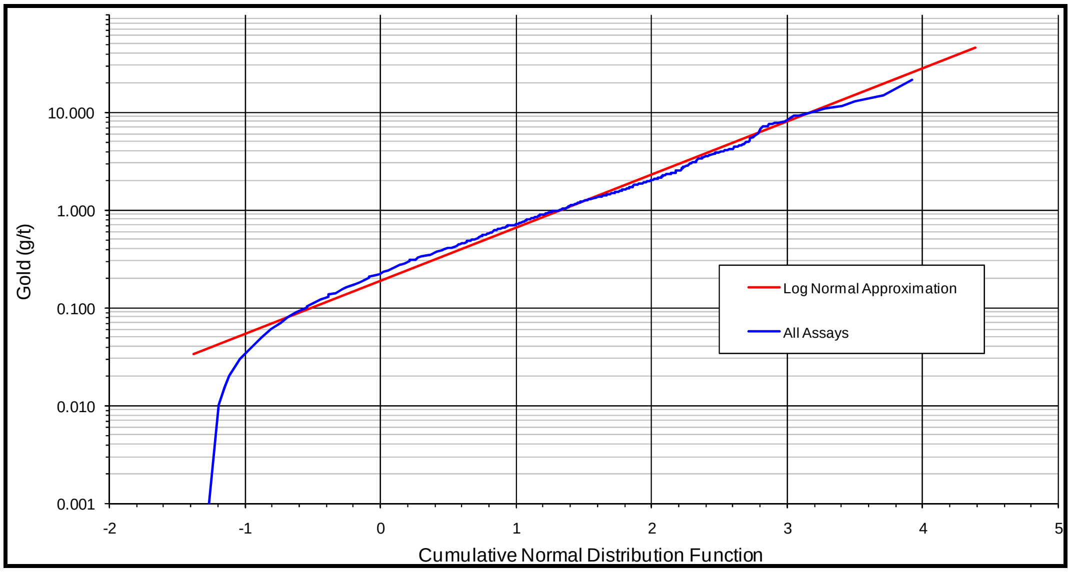


Figure 17.3 Mitchell Au Assay Cumulative Probability Plot

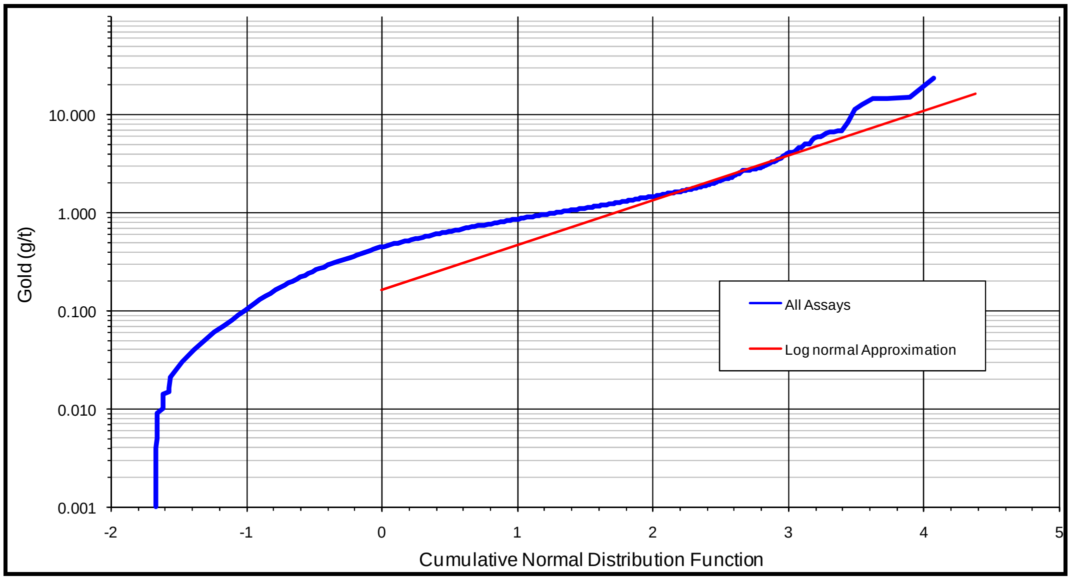


Figure 17.4 Kerr Cu Assay Cumulative Probability Plot

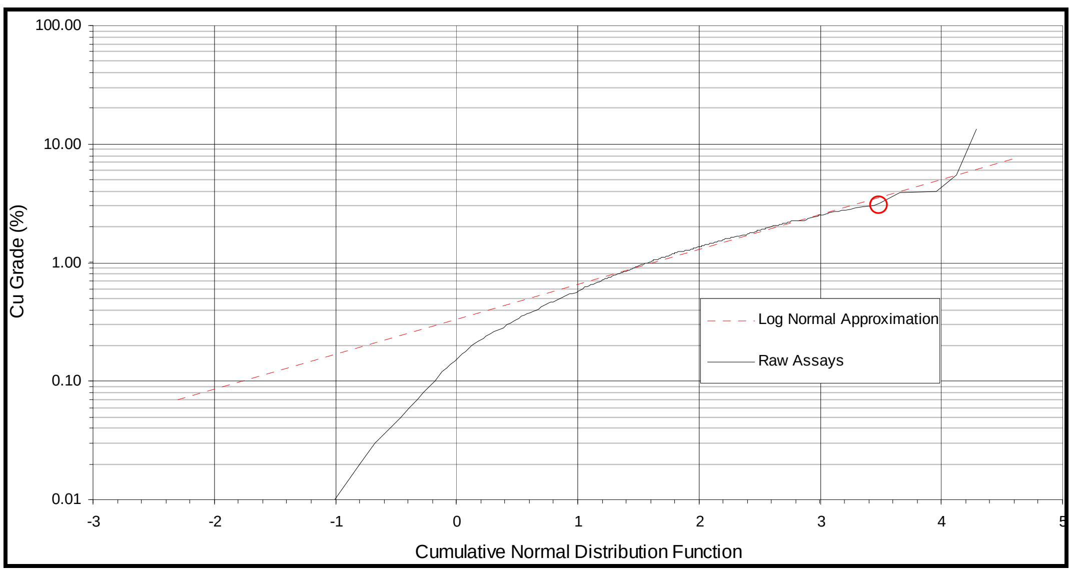


Figure 17.5 Sulphurets Cu Assay Cumulative Probability Plot

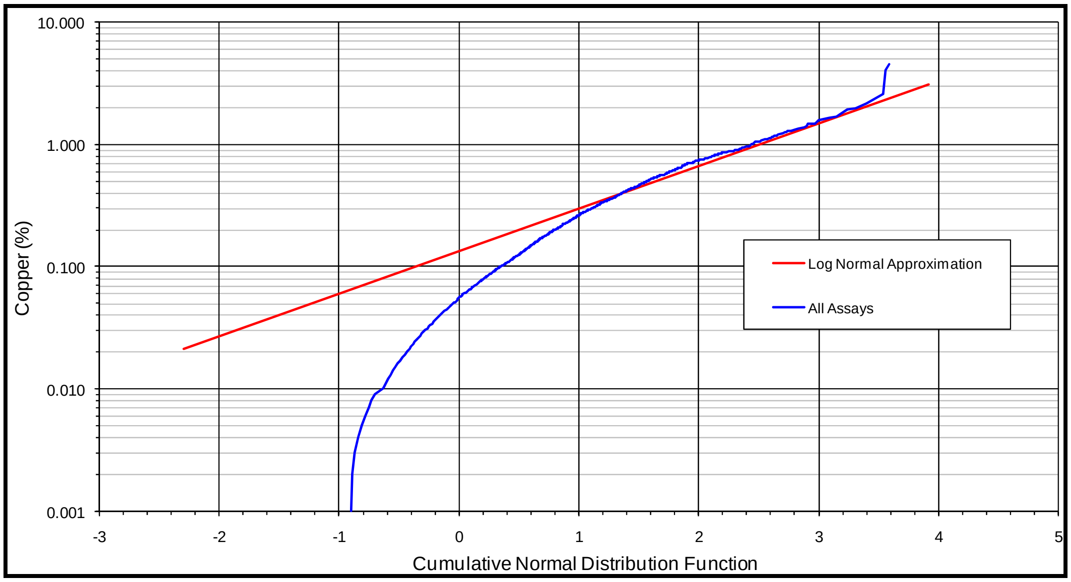
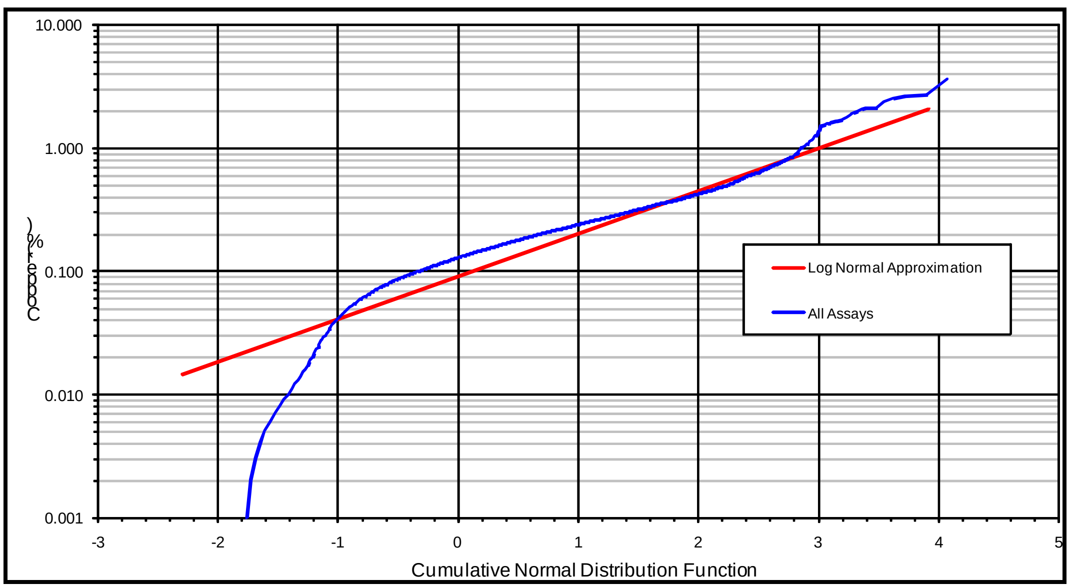


Figure 17.6 Mitchell Cu Assay Cumulative Probability Plot



Based on the information shown in Figure 17.1 through to Figure 17.6, RMI capped raw gold and copper assays at the area highlighted by the red circle where the distribution of grades becomes erratic.

Table 17.13 summarizes the capping limits that were established for gold, silver, copper, and molybdenum by deposit. In addition to the capping limit for each metal, the number of raw assays that were capped prior to creating 15 m-long drill hole composites is also provided.

Table 17.13 Grade Capping Limits

Deposit – Zone	Au (g/t)		Ag (g/t)		Cu (%)		Mo (ppm)	
	Cap Grade	No. Capped	Cap Grade	No. Capped	Cap Grade	No. Capped	Cap Grade	No. Capped
Kerr	10.0	7	n/a	n/a	2.75	10	n/a	n/a
Sulphurets	10.0	7	100	4	2.00	4	600	13
Mitchell (Main Zone)	5.0	19	180	7	0.90	17	1,200	40
Mitchell (Bornite Breccia)	n/a	n/a	n/a	n/a	1.50	22	n/a	n/a
Mitchell (Leach Breccia)	n/a	n/a	n/a	n/a	0.35	15	n/a	n/a

17.4 DRILL HOLE COMPOSITES

The raw drill hole data were composited into 15 m-long composites starting from the drill hole collar. Most of the original assay data were in the range of 1.5 to 3.0 m long, with the majority being 2 m-long. Based on the scale of the deposit, 15 m-long composites were deemed to be an appropriate length for estimating mineral resources.

The assays were composited using MineSight® software. Various geologic data were assigned to the 15 m-long composites using the majority rule method.

17.5 GEOLOGIC CONSTRAINTS

Various litholoIgic, alteration, structural domains, and metal grade envelopes were constructed for each of the deposits by RMI and Seabridge personnel. Most of these three-dimensional wireframes were initially interpreted onto cross sections, which were then reconciled in bench plan prior to building the final wireframe.

As previously mentioned, gold and copper grades within the three deposits are not necessarily confined to distinct geologic units. For this reason, RMI elected to use gold, copper, and molybdenum grade envelopes for constraining the estimate of block grades. The cutoff grades for each metal are summarized in Table 17.14 for each deposit.

Table 17.14 Grade Envelope Cutoffs

Deposit	Au (g/t)	Cu (%)	Mo (ppm)
Kerr	0.20	0.40	n/a
Sulphurets	0.35	0.15	n/a
Mitchell	0.25	0.10	50

17.6 VARIOGRAPHY

RMI generated a number of gold and copper correlograms and variograms using both drill hole assays and 15 m-long drill hole composites. Down-hole correlograms were generated using the original raw assay data in order to establish the nugget effect for gold and copper.

Figure 17.7 through to Figure 17.9 show gold grade correlograms for the Kerr, Sulphurets, and Mitchell deposits, respectively. Figure 17.10 through to Figure 17.12 show copper grade correlograms for the Kerr, Sulphurets, and Mitchell deposits, respectively.

Figure 17.7 Kerr Au Grade Correlogram

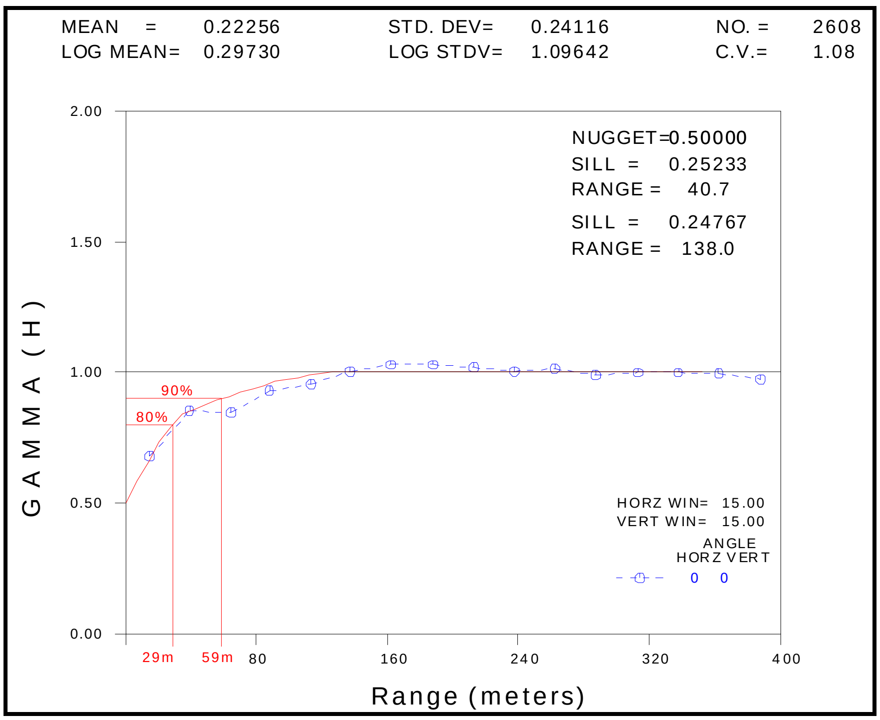


Figure 17.8 Sulphurets Au Grade Correlogram

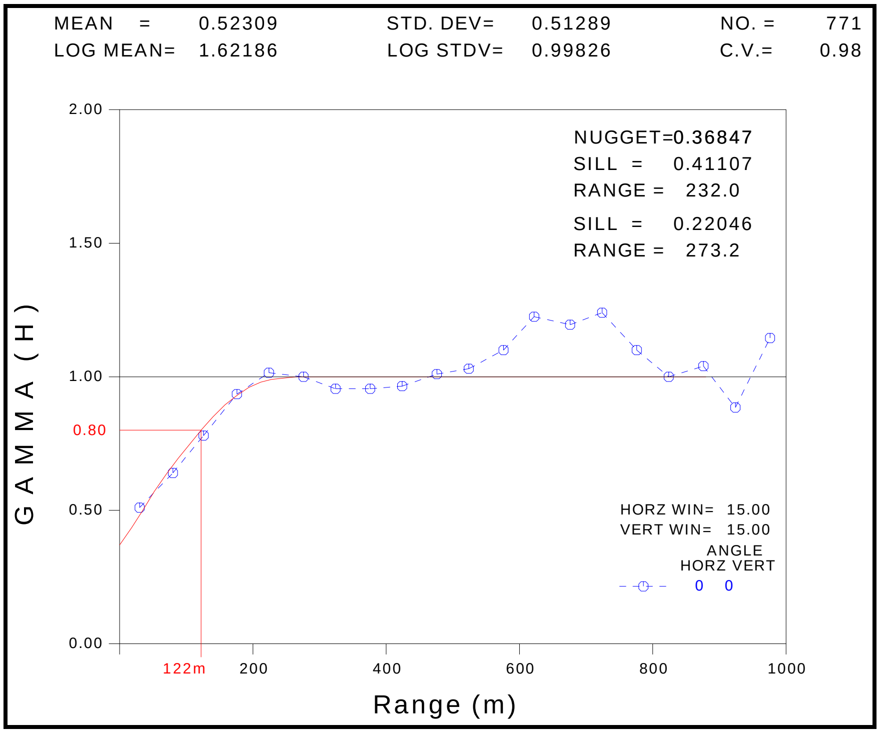


Figure 17.9 Mitchell Au Grade Correlogram

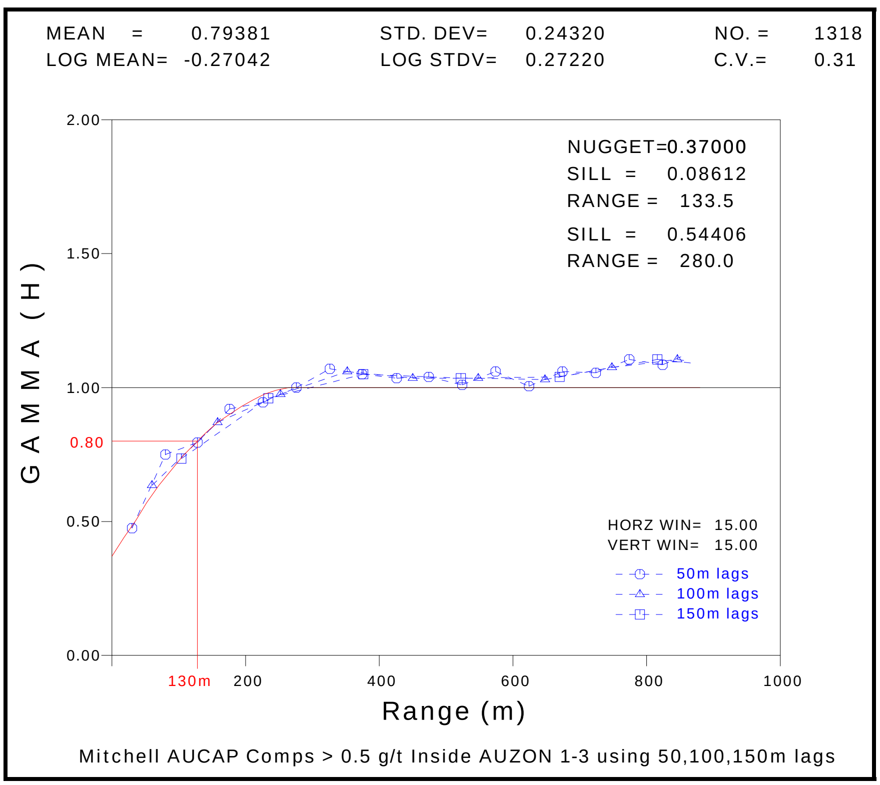


Figure 17.10 Kerr 0.4% Cu Indicator Correlogram

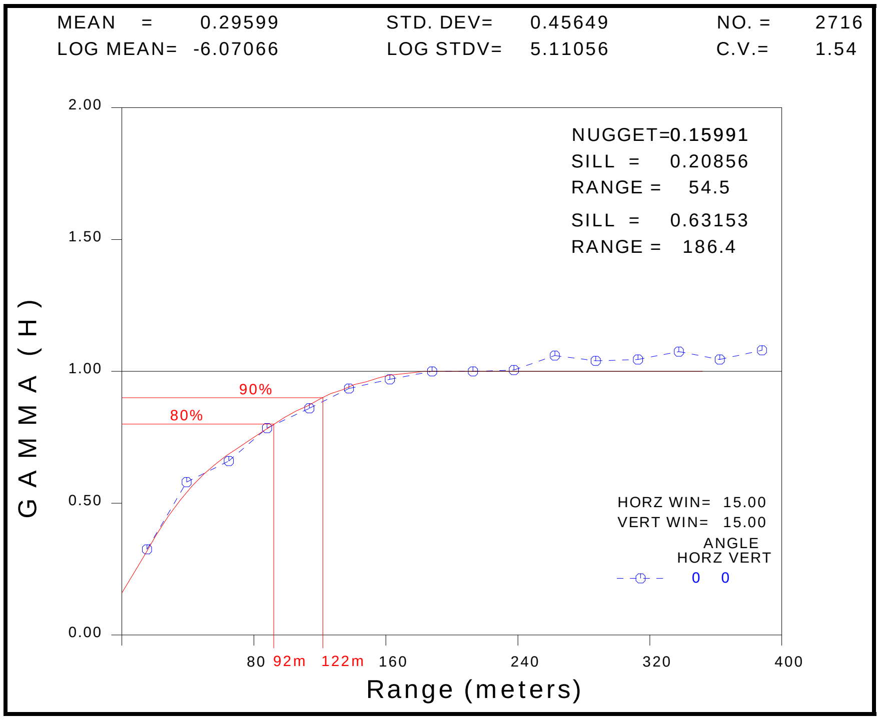


Figure 17.11 Sulphurets Cu Grade Correlogram

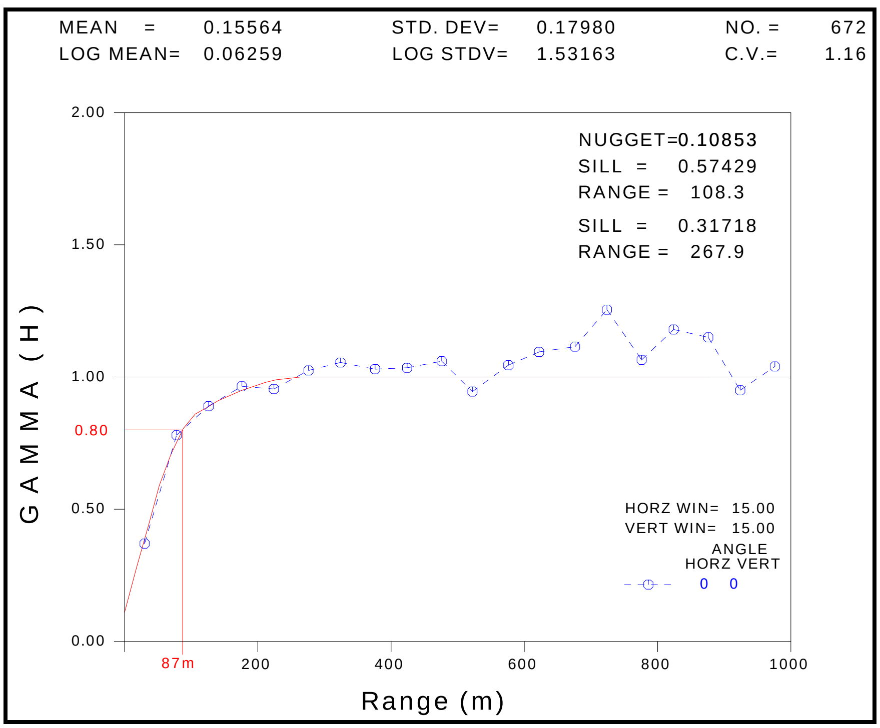
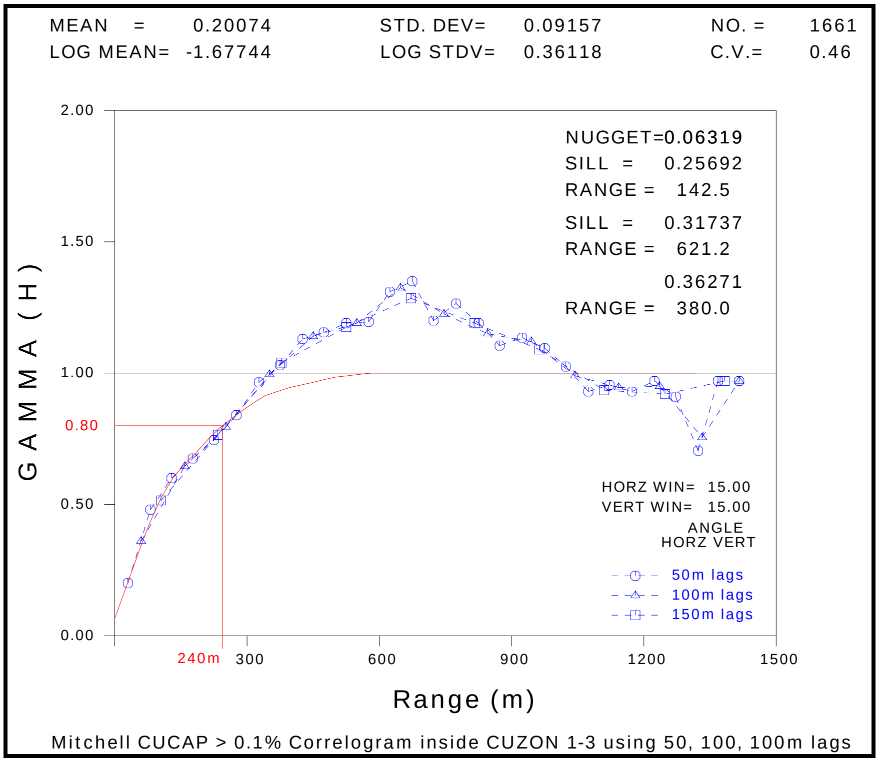


Figure 17.12 Mitchell Cu Grade Correlogram



The correlograms shown in Figure 17.7 through to Figure 17.12 were modelled with nested spherical models. Total ranges for gold are 138 m, 273 m, and 280 m for the Kerr, Sulphurets, and Mitchell deposits, respectively. At 80% of the total sill, gold ranges of 29 m, 122 m, and 130 m were interpreted for the Kerr, Sulphurets, and Mitchell, respectively. Total ranges for copper are 186 m, 268 m, and 380 m for the Kerr, Sulphurets, and Mitchell deposits, respectively. At 80% of the total sill, copper ranges of 92 m, 87 m, and 240 m were interpreted for the Kerr, Sulphurets, and Mitchell, respectively.

The Sulphurets and Mitchell deposits show remarkably long ranges for gold, which is related to the style and intensity of mineralization. The Kerr gold grade range is significantly less than the other two deposits but Kerr has significantly lower gold grades and higher copper grades than the other deposits. The copper range at 80% of the total sill value for the Kerr deposit is 92 m.

17.7 KERR REBLOCKING

In 2008, RMI constructed a block model for the Kerr deposit with block dimensions of 20 m by 20 m by 10 m, which were located in NAD27 coordinate space. The 2009 Mitchell and Sulphurets models, which are also the subject of the report, were constructed with blocks measuring 25 m by 25 m by 15 m that are located in NAD83 coordinate space. There has been no new drilling or other data collected from the

Kerr deposit since the 2008 block model was constructed. In order to more easily integrate the Kerr deposit into the overall mine planning process RMI "reblocked" the 2008 Kerr block model to match the selective mining units (SMUs) of the 2009 Mitchell and Sulphurets models.

The estimated Kerr gold, copper, and gold equivalent grades were volume weight averaged from the original small blocks into the larger SMUs. Gold and copper zone codes along with the original Kerr resource classification code were assigned to the larger SMUs using the "majority" rule method. Most of the reblocked Kerr SMUs are comprised of portions of 8 original smaller blocks (4 from each original 10-m bench).

RMI summarized tonnes and grade by resource category from the original 20 m by 20 m by 10 m blocks at a variety of cutoff grades and then performed the same tabulation for the reblocked Kerr model. As expected, the reblocked model contains more tonnage and lower grades due to the dilution that was incurred by moving to larger SMUs. Indicated resource tonnage increased by 1% while gold and copper grades decreased by 4 and 5%, respectively, using a 0.5 g/t gold equivalent cutoff grade. More dilution was seen with inferred material (+3% more tonnage; 7 and 10% lower copper and gold grades, respectively). RMI believes that the larger dilution hit with inferred material is related to the lower, often scattered and isolated grades of the original inferred blocks, which incurred more external dilution than the indicated material.

In RMI's opinion, the reblocked Kerr model can be used for preliminary mine planning purposes. However, RMI highly recommends that Seabridge drill a series of confirmation holes that will test the current grade model and provide material for metallurgical testwork.

17.8 GRADE ESTIMATION PARAMETERS

Block gold, silver, copper, and molybdenum grades were estimated by three distinct methods:

1. inverse distance weighting
2. ordinary kriging
3. nearest neighbour.

Gold and copper resources summarized in this report are based on inverse distance squared methods.

A multi-pass estimation strategy was used for gold, silver, copper, and molybdenum. The first estimation pass required two or more drill holes to estimate block grades while subsequent passes acted as "cleanup" runs that filled un-estimated blocks by using larger search ellipses and requiring fewer drill holes. The inverse distance and ordinary kriging estimation plans used block/composite zone matching. For example, blocks located inside of the 0.25 g/t gold envelope were estimated by drill hole composites from that same population.

Table 17.15 through to Table 17.17 summarize the key estimation parameters that were used to estimate block gold grades using inverse distance squared methods for the Kerr, Sulphurets, and Mitchell deposits, respectively. Silver grades were estimated for the Sulphurets and Mitchell deposits using the same parameters shown in Table 17.16 and Table 17.17.

The following abbreviations are used in Table 17.15 through to Table 17.22:

- ROTN = major axis rotation in degrees using “left hand” rule about the Z-axis
- DIPN = dip angle of major axis (negative means downward)
- DIPE = semi-major axis rotation using “left hand” rule about the Y-axis.

Table 17.15 Kerr Au Grade Estimation Parameters

		Block Search Distances (m)			No. Composites Used			Search Ellipse Rotations (LRL)		
Pass	Population	X	Y	Z	Min	Max	Max/ Hole	ROTN	DIPN	DIPE
1	Inside 0.20 g/t shape	75	75	15	2	3	1	20	0	60
2	Inside 0.20 g/t shape	125	125	25	2	3	1	20	0	60
3	Inside 0.20 g/t shape	200	200	40	1	6	1	20	0	60
4	Outside 0.20 g/t shape	100	100	20	2	6	2	20	0	60

Table 17.16 Sulphurets Au Grade Estimation Parameters

		Block Search Distances (m)			No. Composites Used			Search Ellipse Rotations (LRL)		
Pass	Population	X	Y	Z	Min	Max	Max/ Hole	ROTN	DIPN	DIPE
1	Inside 0.35 g/t shape	200	200	30	3	6	2	53	0	45
2	Inside 0.35 g/t shape	200	200	30	1	3	1	53	0	45
1	Gold breccia	300	300	60	3	6	2	53	0	45
2	Gold breccia	300	300	60	1	3	1	53	0	45
1	Leach breccia	300	300	60	3	6	2	53	0	45
2	Leach breccia	300	300	60	1	3	1	53	0	45
1	Outside 0.35 g/t shape	100	100	15	1	3	1	53	0	45
1	Upper plate seds	300	300	100	2	6	2	53	0	45

Table 17.17 Mitchell Au Grade Estimation Parameters

Pass	Population	Block Search Distances (m)			No. Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/ Hole	ROTN	DIPN	DIPE
1	Inside 0.25 g/t shape	125	125	30	3	8	2	75	0	40
2	Inside 0.25 g/t shape	250	250	60	3	8	2	75	0	40
3	Inside 0.25 g/t shape	375	375	90	3	8	2	75	0	40
4	Inside 0.25 g/t shape	500	500	120	1	3	1	75	0	40
1	Bornite breccia	250	250	60	3	8	2	275	0	65
2	Bornite breccia	375	375	90	1	3	1	275	0	65
1	Leach breccia	250	250	60	3	8	2	275	0	65
2	Leach breccia	500	500	120	1	3	1	275	0	65
1	Upper plate seds	150	150	45	3	8	2	75	0	40
2	Upper plate seds	75	75	15	1	3	1	75	0	40
3	Upper plate seds	300	300	100	1	3	1	75	0	40
1	Lower plate	150	150	45	3	8	2	75	0	40
2	Lower plate	75	75	15	1	3	1	75	0	40

Table 17.18 through to Table 17.20 summarize the key estimation parameters that were used to estimate block copper grades using inverse distance squared methods for the Kerr, Sulphurets, and Mitchell deposits, respectively.

Table 17.18 Kerr Cu Grade Estimation Parameters

Pass	Population	Block Search Distances (m)			No. Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/ Hole	ROTN	DIPN	DIPE
1	Inside 0.40% shape	75	75	15	2	3	1	20	0	60
2	Inside 0.40% shape	125	125	25	2	3	1	20	0	60
3	Inside 0.40% shape	200	200	40	1	6	1	20	0	60
4	Outside 0.40% shape	100	100	20	2	6	2	20	0	60

Table 17.19 Sulphurets Cu Grade Estimation Parameters

		Block Search Distances (m)			No. Composites Used			Search Ellipse Rotations (LRL)		
Pass	**Population**	**X**	**Y**	**Z**	**Min**	**Max**	**Max/ Hole**	**ROTN**	**DIPN**	**DIPE**
1	Inside 0.15% Cu shape	200	200	30	3	6	2	53	0	45
2	Inside 0.15% Cu shape	200	200	30	1	3	1	53	0	45
1	Gold breccia	300	300	60	3	6	2	53	0	45
2	Gold breccia	300	300	60	1	3	1	53	0	45
1	Leach breccia	300	300	60	3	6	2	53	0	45
2	Leach breccia	300	300	60	1	3	1	53	0	45
1	Outside 0.15% Cu shape	100	100	15	3	6	2	53	0	45
2	Outside 0.15% Cu shape	100	100	15	1	3	1	53	0	45
1	Upper plate seds	300	300	100	2	6	2	53	0	45

Table 17.20 Mitchell Cu Grade Estimation Parameters

		Block Search Distances (m)			No. Composites Used			Search Ellipse Rotations (LRL)		
Pass	**Population**	**X**	**Y**	**Z**	**Min**	**Max**	**Max/ Hole**	**ROTN**	**DIPN**	**DIPE**
1	Inside 0.10% shape - upper plate	125	125	30	3	8	2	75	0	40
2	Inside 0.10% shape - upper plate	250	250	60	3	8	2	75	0	40
3	Inside 0.10% shape - lower plate	375	375	90	3	8	2	75	0	40
4	Inside 0.10% shape - lower plate	500	500	120	1	3	1	75	0	40
1	Bornite breccia	250	250	60	3	8	2	275	0	65
2	Bornite breccia	375	375	90	1	3	1	275	0	65
1	Leach breccia	250	250	60	3	8	2	275	0	65
2	Leach breccia	500	500	120	1	3	1	275	0	65
1	Upper plate seds	150	150	45	3	8	2	75	0	40
2	Upper plate seds	75	75	15	1	3	1	75	0	40
3	Upper plate seds	300	300	100	1	3	1	75	0	40
1	Lower plate	150	150	45	3	8	2	75	0	40
2	Lower plate	75	75	15	1	3	1	75	0	40

Table 17.21 and Table 17.22 summarize the key estimation parameters that were used to estimate block molybdenum grades using inverse distance squared methods for the Sulphurets and Mitchell deposits, respectively.

Table 17.21 Sulphurets Mo Grade Estimation Parameters

Pass	Population	Block Search Distances (m)			No. Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/ Hole	ROTN	DIPN	DIPE
1	No constraint	250	250	60	1	3	1	53	0	45
2	No constraint	250	250	60	3	8	2	53	0	45

Table 17.22 Mitchell Mo Grade Estimation Parameters

Pass	Population	Block Search Distances (m)			No. Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	Inside 50 ppm Mo shape	250	250	60	1	3	1	20	0	45
2	Inside 50 ppm Mo shape	250	250	60	3	8	2	20	0	45

The number of composites and drill holes used to estimate block gold and copper grades were captured during the estimation process along with the distance to the closest composite that was used to estimate each block. These criteria among others were used to classify resources (see Section 17.10). The majority of the mineral resources that are subject to this report are based on blocks that were estimated by initial estimation passes that required two or more drill holes.

17.9 GRADE MODEL VERIFICATION

Estimated block grades were verified by visual and statistical methods. RMI visually compared estimated block gold and copper grades versus drill hole composite grades. In RMI's opinion, there is a reasonable comparison between the drill hole composite grades and the estimated block grades. Grade bias checks were performed for the original Kerr block model (Lechner, 2008). Inverse distance gold and copper grades for the 2008 Kerr model compared very well with the nearest neighbour model.

Figure 17.13 and Figure 17.14 are N10E cross sections showing estimated gold and copper block grades, respectively. Figure 17.15 and Figure 17.16 are block model level maps drawn at the 630 elevation showing estimated block gold and copper grades, respectively.

Figure 17.13 Mitchell Au Block Model Cross Section 11

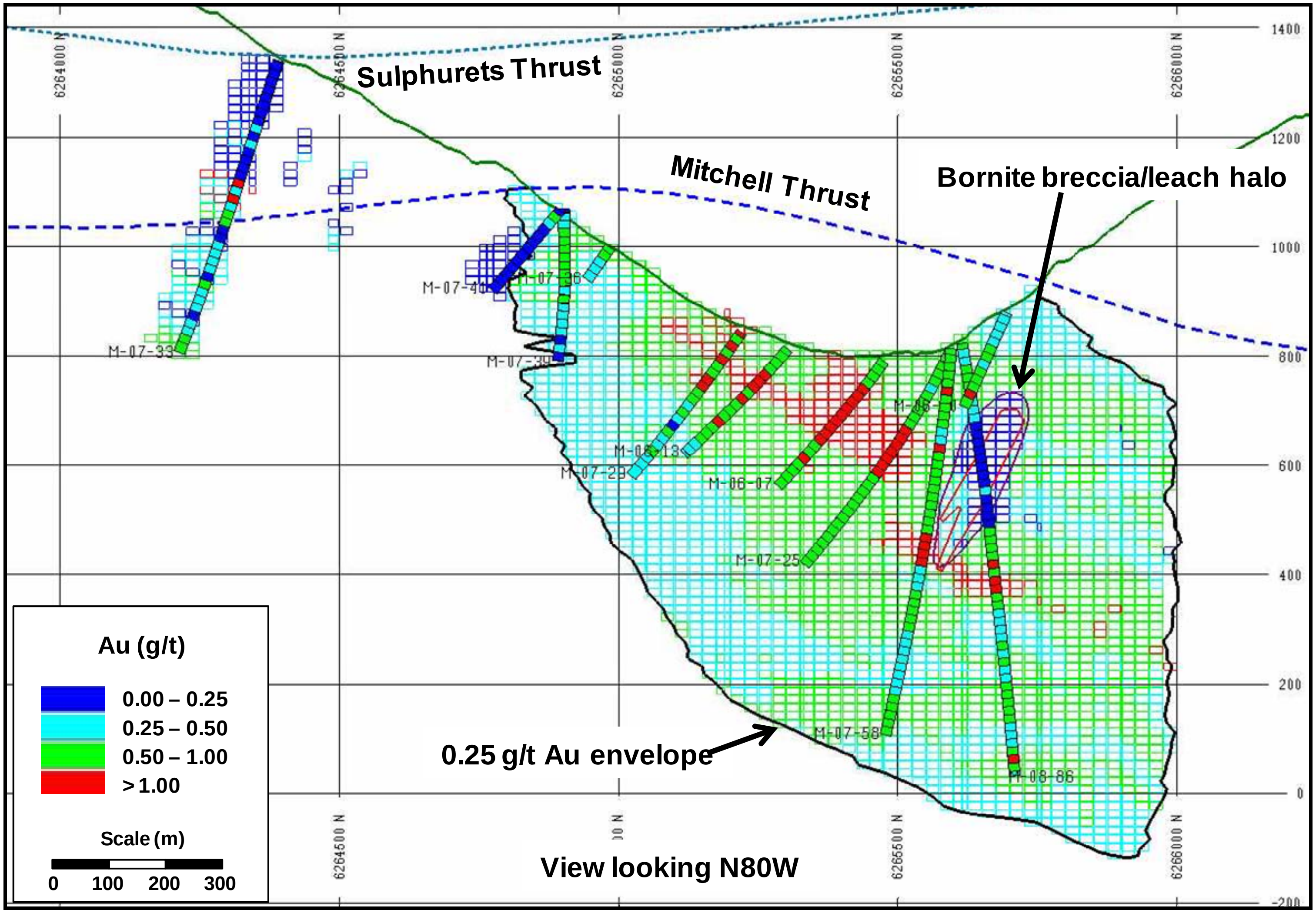


Figure 17.14 Mitchell Cu Block Model Cross Section 11

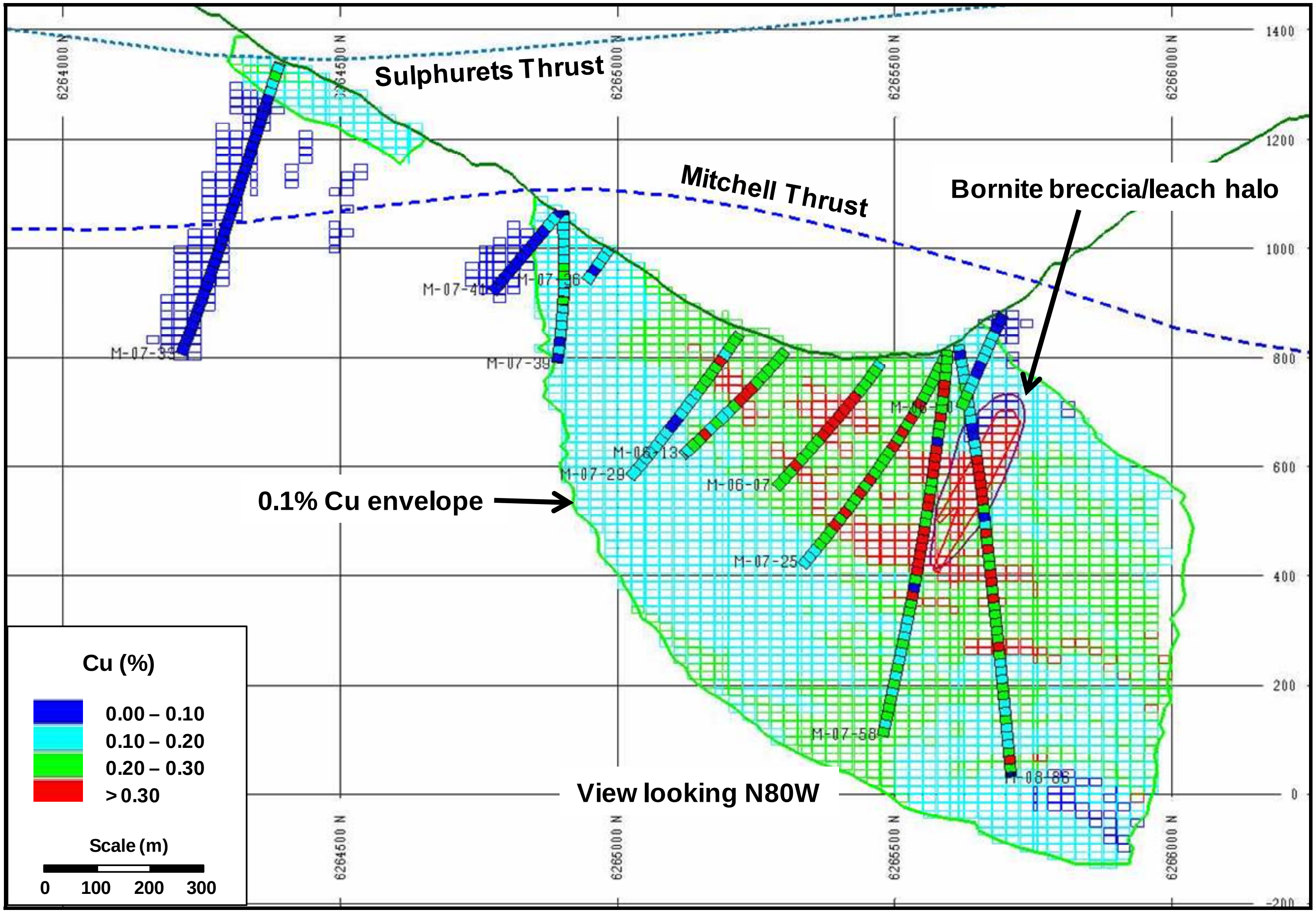


Figure 17.15 Mitchell Au Block Model Level Plan 630

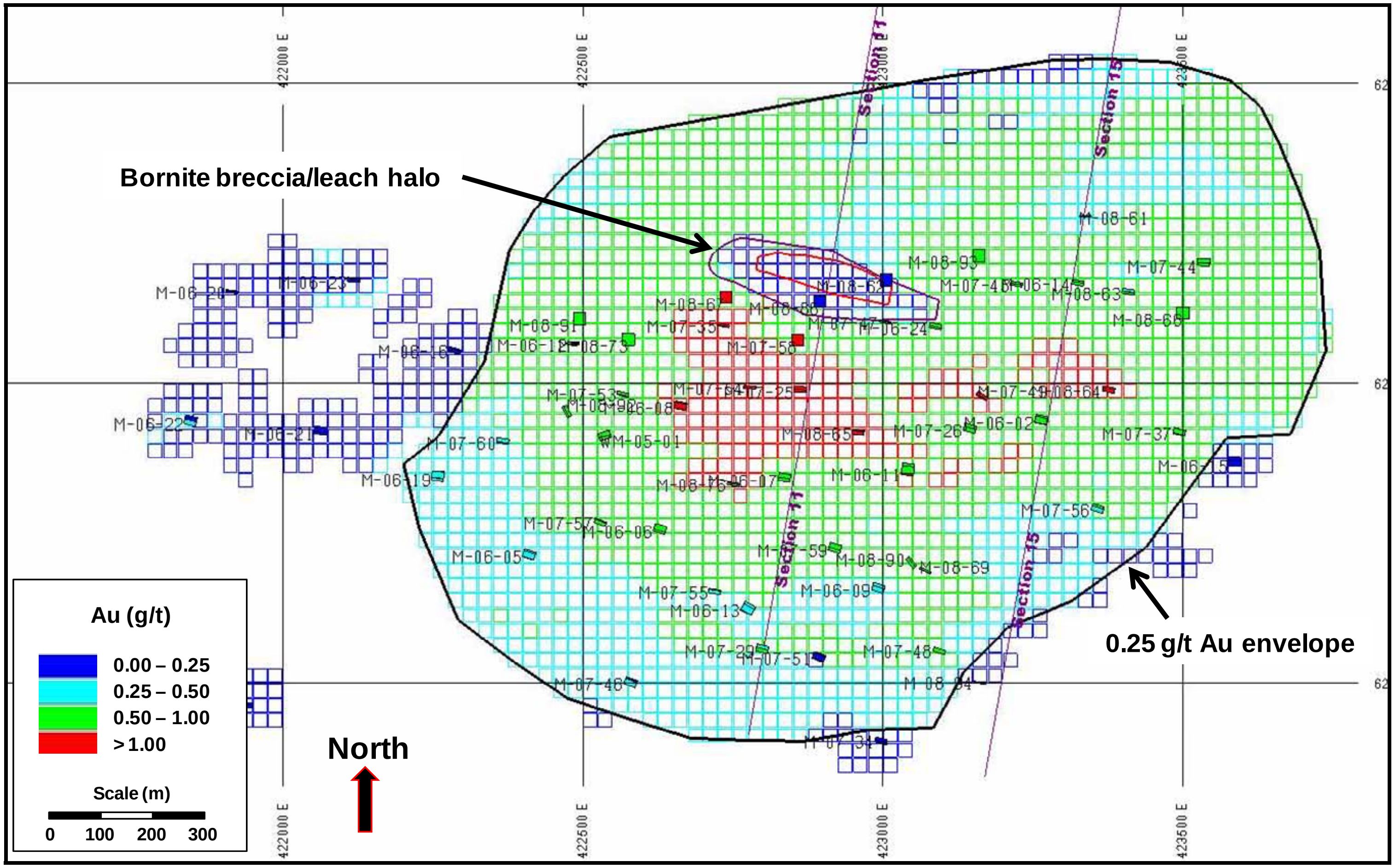
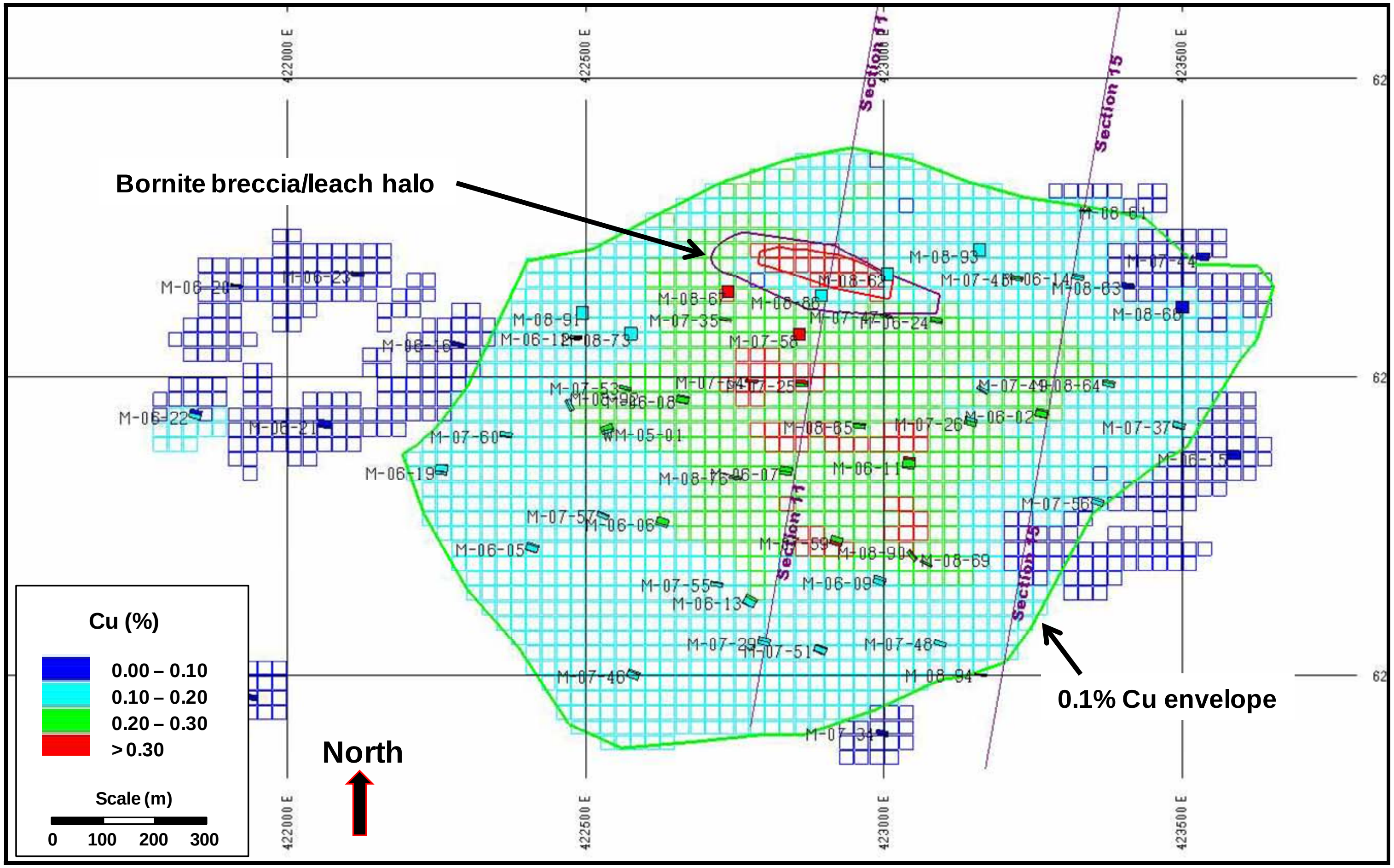


Figure 17.16 Mitchell Cu Block Model Level Plan 630



RMI generated nearest neighbour models for both gold and copper in order to check for potential global biases in the estimated block grades. Table 17.23 compares mean gold and copper grades at a zero cutoff grade for Sulphurets indicated and inferred mineral resource blocks using the three different estimation methods.

Table 17.23 Sulphurets Global Bias Checks

Source of Estimate	Indicated		Inferred	
	Mean Au (g/t)	% Diff vs. NN Model	Mean Au (g/t)	% Diff vs. NN Model
NN	0.6929	0.0	0.6269	0.0
IDW	0.7012	1.2	0.6336	1.1
OK	0.6920	-0.1	0.6451	2.9

Source of Estimate	Indicated		Inferred	
	Mean Cu (%)	% Diff vs. NN Model	Mean Cu (%)	% Diff vs. NN Model
NN	0.2636	0.0	0.2441	0.0
IDW	0.2633	-0.1	0.2577	5.6
OK	0.2602	-1.3	0.2548	4.4

Table 17.24 compares mean gold and copper grades at a zero cutoff grade for Mitchell measured plus indicated and inferred mineral resource blocks using three different estimation methods.

Table 17.24 Mitchell Global Bias Checks

Source of Estimate	Measured + Indicated		Inferred	
	Mean Au (g/t)	% Diff vs. NN Model	Mean Au (g/t)	% Diff vs. NN Model
NN	0.6183	0.0	0.4010	0.0
IDW	0.6192	0.1	0.4123	2.8
OK	0.6166	-0.3	0.4084	1.8

Source of Estimate	Measured + Indicated		Inferred	
	Mean Cu (%)	% Diff vs. NN Model	Mean Cu (%)	% Diff vs. NN Model
NN	0.1772	0.0	0.1307	0.0
IDW	0.1782	0.6	0.1364	4.4
OK	0.1779	0.4	0.1353	3.5

The results provided in Table 17.23 and Table 17.24 show that the inverse distance weighted (IDW) models compare very well with the nearest neighbour grades for the measured plus indicated category. There are wider differences in mean grades for inferred material that is based on less drilling hence lower confidence levels in the estimates.

Possible local biases in the estimate of block grades were examined by preparing a set of “swath plots” for gold and copper. These plots compare mean estimated inverse distance squared (AUIDW and CUIDW) with the nearest neighbour (AUNN and CUNN) estimates by block model columns (eastings), rows (northings), and levels (elevation). Gold and copper swath plots for are shown in Figure 17.13 through to Figure 17.15 for the Sulphurets deposit by easting, northing, and elevation, respectively. Similar plots are shown for the Mitchell deposit as Figure 17.16 through to Figure 17.18. The number of blocks by the rows, columns, and levels are shown by the dashed black line and the units are read from the Y-axis on the right side of the plots.

Figure 17.17 Sulphurets Au-Cu Swath Plots by Eastings

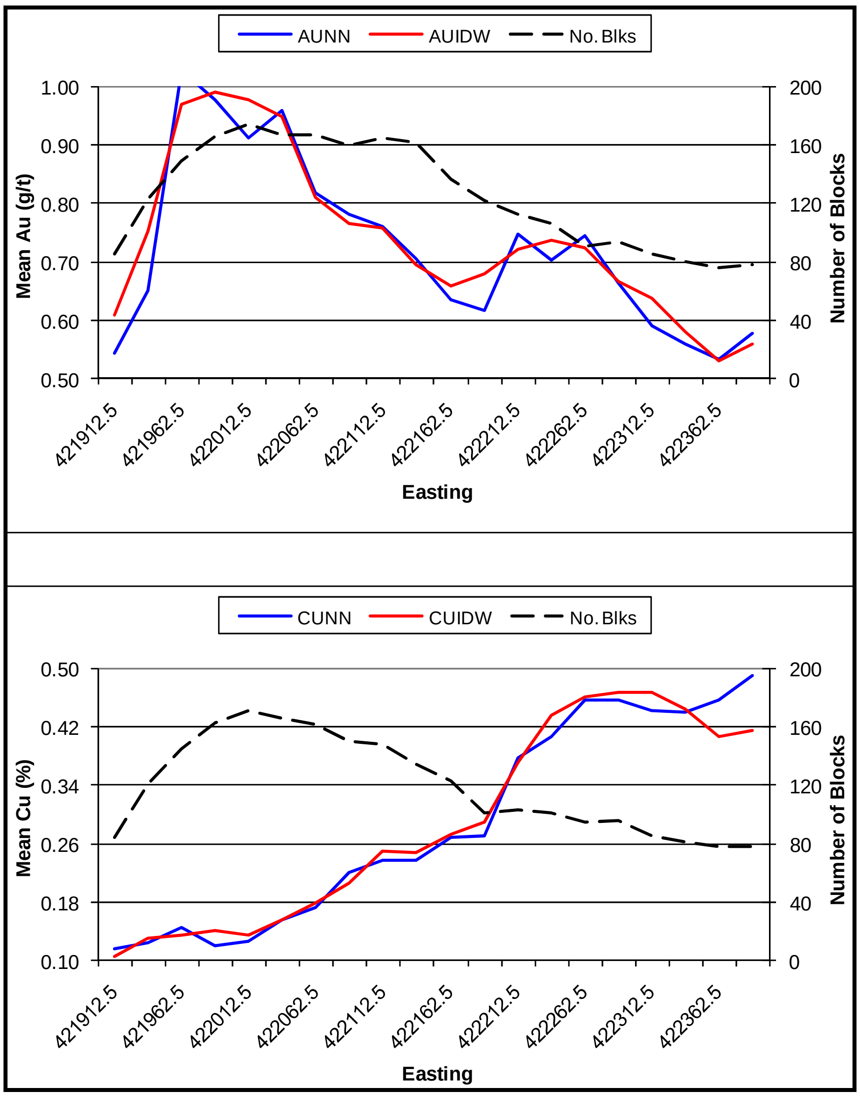
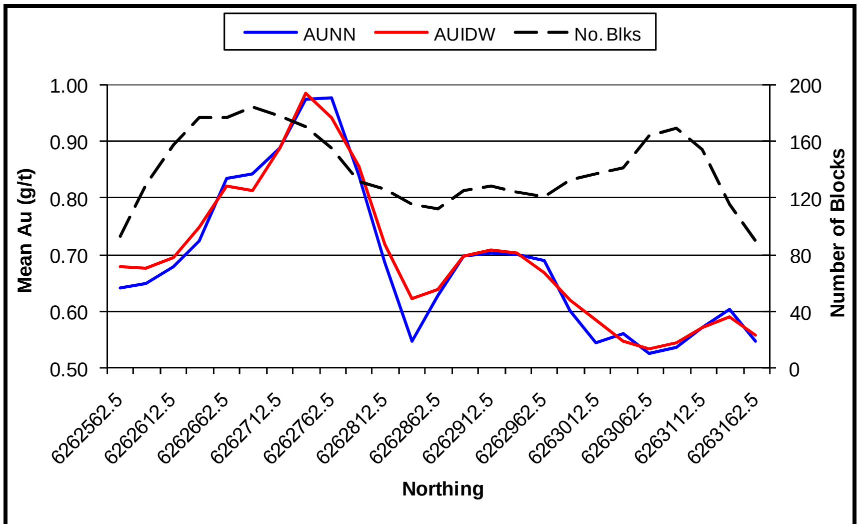


Figure 17.18 Sulphurets Au-Cu Swath Plots by Northings



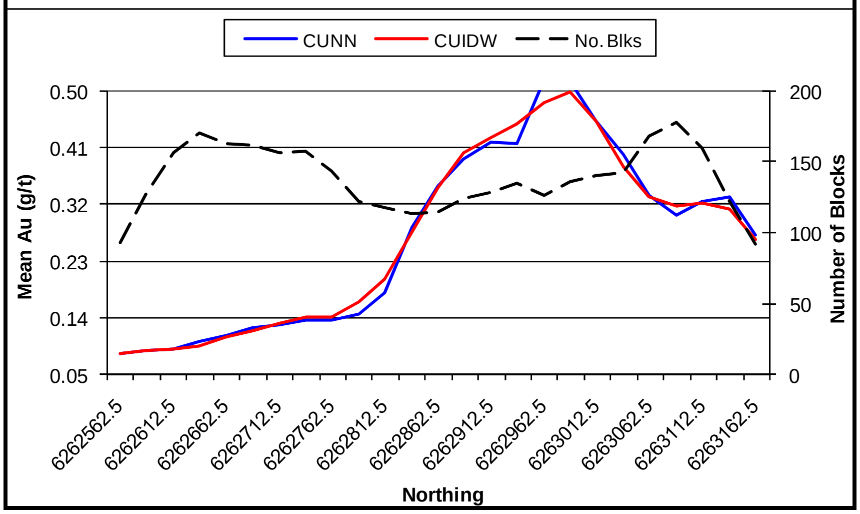


Figure 17.19 Sulphurets Au-Cu Swath Plots by Elevation

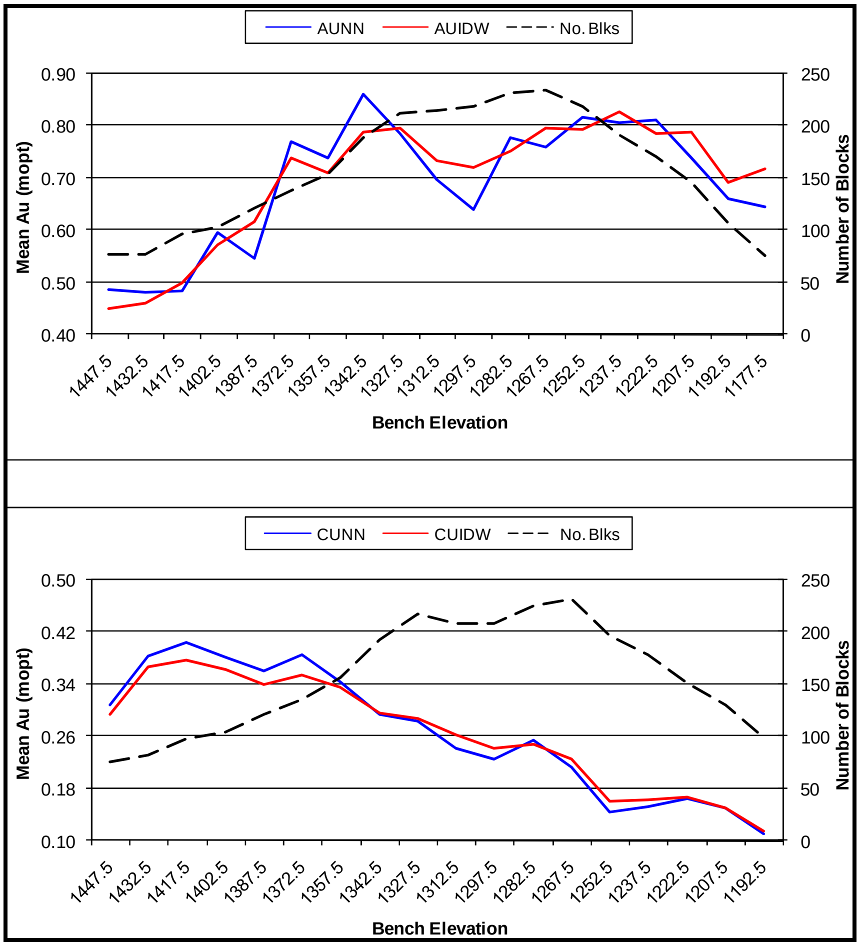


Figure 17.20 Mitchell Au-Cu Swath Plots by Eastings

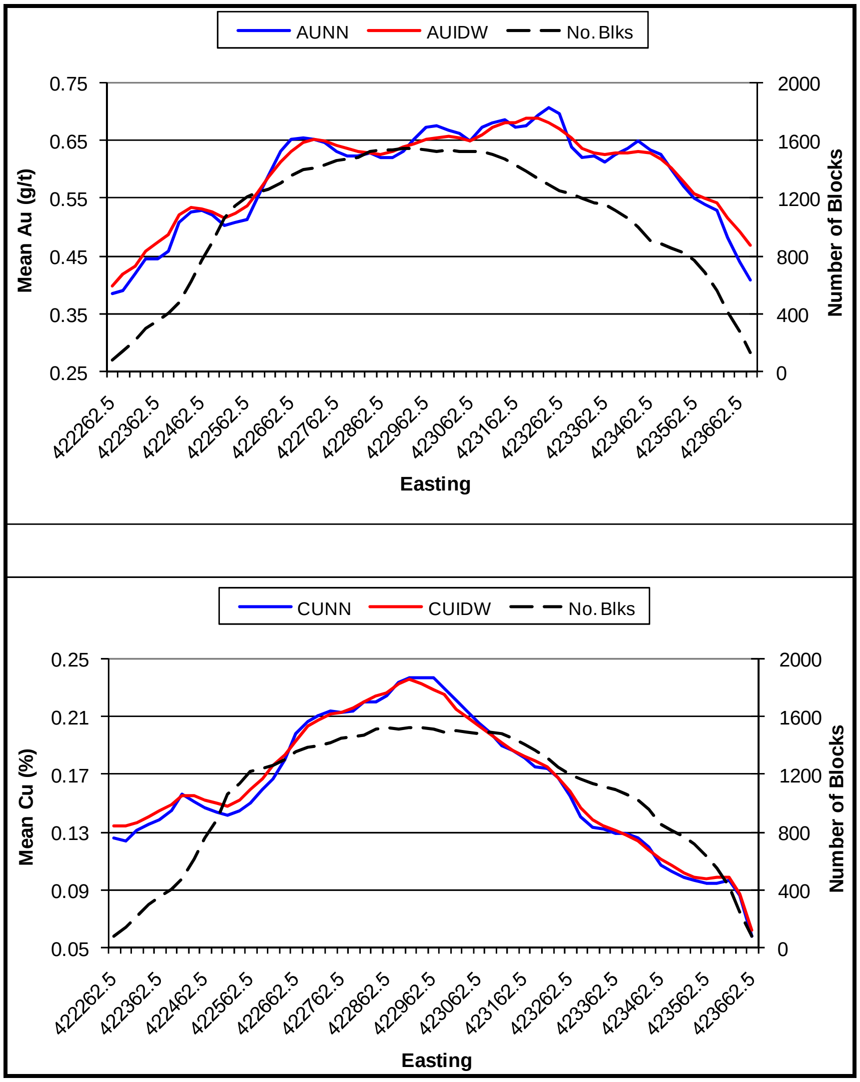


Figure 17.21 Mitchell Au-Cu Swath Plots by Northings

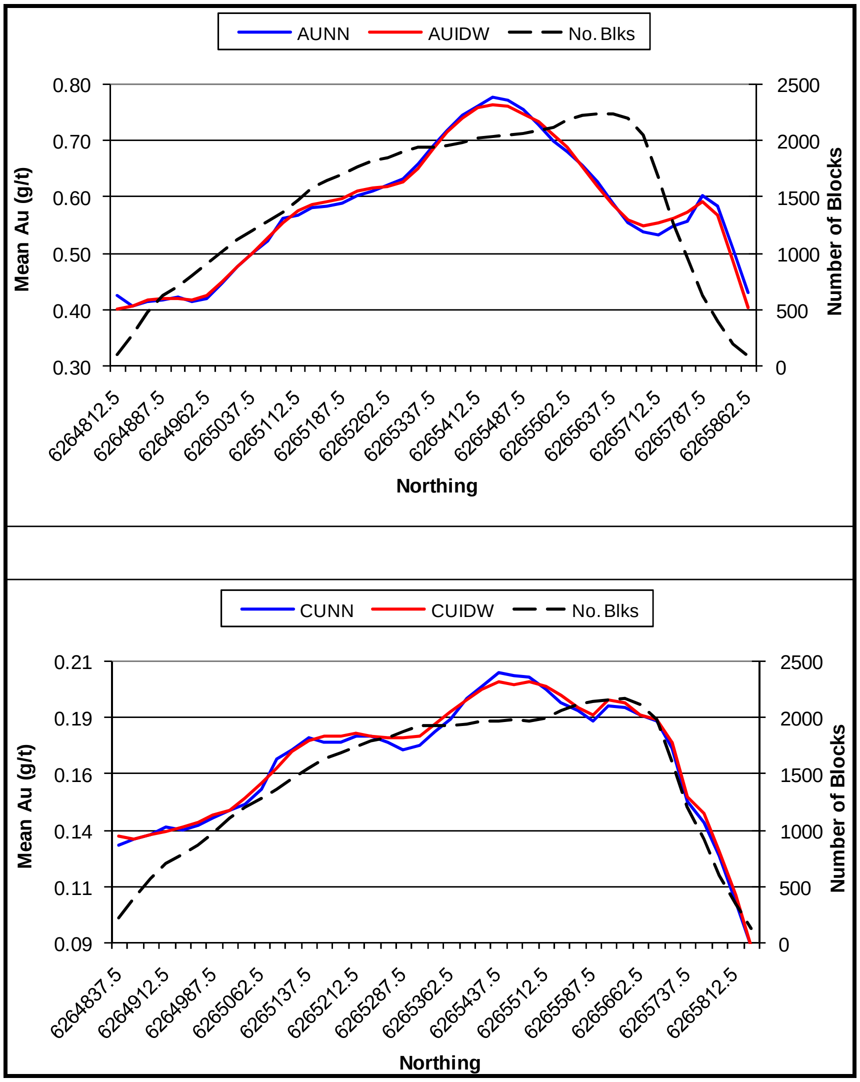
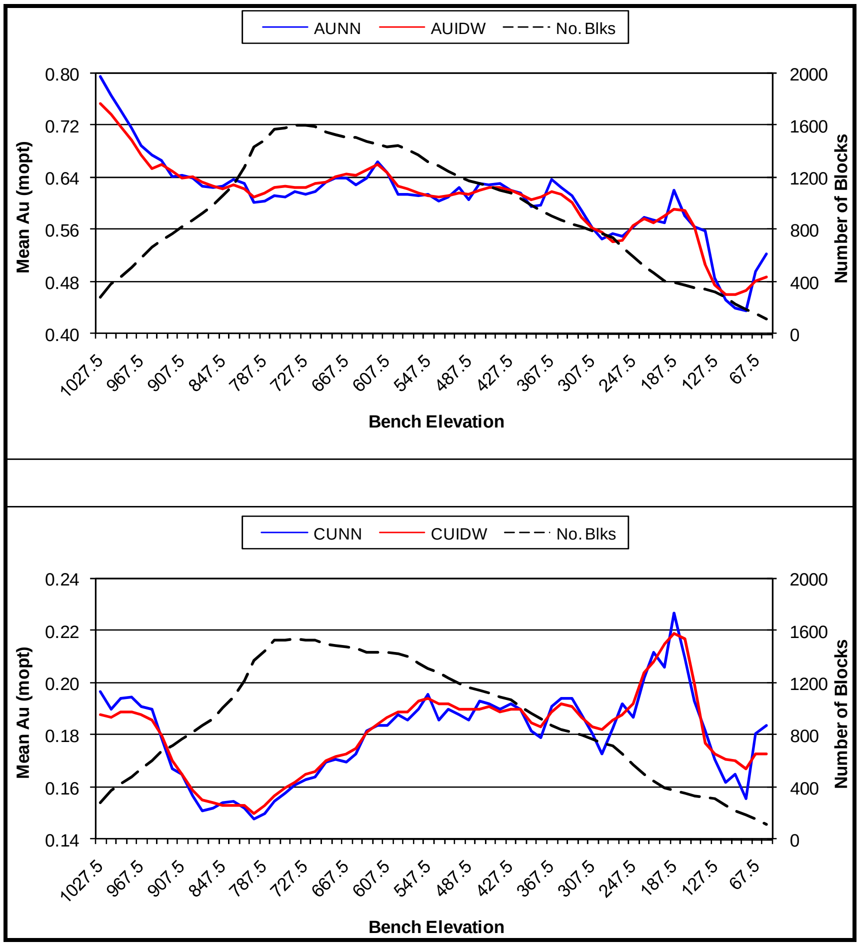


Figure 17.22 Mitchell Au-Cu Swath Plots by Elevation



In RMI's opinion, the swath plots shown in Figure 17.13 through to Figure 17.18 show a close comparison between the inverse distance and nearest neighbour estimates. There do not appear to be any severe local biases in the estimate of gold and copper. Based on visual and statistical checks, it is the opinion of RMI that the Sulphurets and Mitchell models are globally unbiased and represent reasonable estimates of in situ block grades.

17.10 Resource Classification

RMI classified the estimated Sulphurets and Mitchell block grades into measured (Mitchell only), indicated, and inferred mineral resources using a combination of distance to data, proximity to data, a required number of drill holes, and manually constructed shapes that represent "mineralized continuity". The 2008 Kerr grade model was reblocked into larger SMUs (see Section 17.7). The original resource category codes in the Kerr model were assigned to the new blocks using the majority rule function.

RMI digitized shapes around mineralized drill holes for the Sulphurets and Mitchell deposits. These shapes define areas where mineralized continuity has been well established by logging and sample results. The model blocks were coded with these shapes and used to identify possible measured (Mitchell only) and indicated resources.

Indicated mineral resources were assigned to the Sulphurets deposit if the blocks were located inside of the mineralized shape and the blocks had been estimated by three or more drill holes, with the closest one being within 75 m of the block. Inferred mineral resources were assigned to any unclassified (i.e. not previously classified as Indicated) blocks located inside of the mineralized shape. Several other criteria were established for assigning inferred resources. These include:

- blocks located inside of the gold or copper envelopes that were estimated by two or more holes, provided that the closest hole was within 125 m of the block
- blocks located outside of the gold or copper shapes that were estimated by two or more holes (one of which is within 75 m of the block)
- blocks located outside of the gold or copper shapes that were estimated by at least one hole that is within 50 m of the block.

Measured mineral resources were assigned to the Mitchell deposit if the blocks were located inside of the mineralized continuity shape and were estimated by one hole within 17 m or two or more holes, one of which is within 50 m of the block. Any block inside of the mineralized continuity shape that was not classified as measured was assigned indicated status if it was estimated by two or more holes, one of which is within 125 m of the block. All of the bornite breccia and leach breccia blocks located inside of the mineralized continuity shape were classified as indicated.

Inferred mineral resources were assigned to any unclassified blocks located inside of the mineralized continuity shape. Other rules established inferred material based on the following:

- blocks inside of the gold or copper zone estimated by 2 or more holes, one of which is within 175 m of the block
- blocks inside of the gold or copper zone estimated by 1 or more holes, one of which is within 75 m of the block
- lower plate blocks located outside of the gold or copper zone estimated by 2 or more holes, one of which is within 75 m of the block
- lower plate blocks located outside of the gold or copper zone estimated by 1 or more holes, one of which is within 50 m of the block
- upper plate blocks located inside of the gold or copper zone estimated by 1 or more holes, one of which is within 75 m of the block

- upper plate blocks located outside of the gold or copper zone estimated by 1 or more holes, one of which is within 50 m of the block.

17.11 Summary of Mineral Resources

Gold and copper mineral resources were tabulated for the reblocked Kerr model, the updated Sulphurets model, and the updated Mitchell model using a gold equivalent cutoff grade. This equivalent grade was calculated based on assumed metal prices and recoveries. Gold and copper prices of US$650/oz and US$2.00/lb, respectively, were used to calculate the gold equivalent grade along with gold and copper recoveries of 70 and 85%, respectively, using the following formula:

*Gold Equivalent Grade = Au (g/t) + (Cu (%) * 2.562)*

Mineral resources are summarized at a variety of gold equivalent cutoff grades in Table 17.25 through to Table 17.27 for the Kerr, Sulphurets, and Mitchell deposits, respectively. A gold equivalent cutoff grade of 0.50 g/t has been selected for disclosing mineral resources as highlighted in yellow in Table 17.25 through to Table 17.27.

Table 17.25 Summary of Mineral Resources – Kerr

AuEq Cutoff (g/t)	Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
0.30	287,900	0.21	0.34	1,944	2,158	98,600	0.16	0.31	507	673
0.40	252,500	0.22	0.38	1,786	2,114	81,700	0.17	0.35	447	631
0.50	**225,300**	**0.23**	**0.41**	**1,666**	**2,036**	**69,900**	**0.18**	**0.39**	**405**	**601**
0.60	198,600	0.24	0.44	1,533	1,926	59,500	0.19	0.43	364	564
0.70	172,700	0.25	0.48	1,388	1,827	51,600	0.20	0.46	332	523
0.80	150,400	0.26	0.52	1,257	1,724	44,900	0.20	0.50	289	495
0.90	129,900	0.27	0.56	1,127	1,603	39,100	0.21	0.53	264	457
1.00	114,900	0.28	0.60	1,034	1,519	33,800	0.22	0.57	239	425

Table 17.26 Summary of Mineral Resources – Sulphurets

AuEq Cutoff (g/t)	Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
0.30	93,500	0.69	0.25	2,074	515	190,700	0.58	0.15	3,556	631
0.40	92,200	0.70	0.25	2,074	508	180,900	0.60	0.16	3,490	638
0.50	**87,300**	**0.72**	**0.27**	**2,021**	**520**	**160,900**	**0.63**	**0.17**	**3,259**	**603**
0.60	82,100	0.74	0.28	1,953	507	136,300	0.67	0.20	2,937	601
0.70	79,000	0.75	0.29	1,905	505	113,800	0.69	0.23	2,524	577
0.80	75,700	0.76	0.30	1,849	500	91,200	0.72	0.26	2,112	523
0.90	71,500	0.77	0.31	1,769	488	77,000	0.75	0.29	1,857	492
1.00	66,900	0.78	0.32	1,679	472	65,400	0.79	0.32	1,660	461

Table 17.27 Summary of Mineral Resources – Mitchell

AuEq Cutoff (g/t)	Measured Mineral Resources					Indicated Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
0.30	604,600	0.65	0.18	12,634	2,398	977,300	0.60	0.17	18,852	3,662
0.40	596,700	0.65	0.18	12,470	2,367	958,300	0.61	0.17	18,794	3,591
0.50	**579,300**	**0.66**	**0.18**	**12,292**	**2,298**	**930,600**	**0.62**	**0.18**	**18,550**	**3,692**
0.60	551,200	0.68	0.19	12,051	2,308	893,600	0.63	0.18	18,100	3,545
0.70	511,700	0.70	0.20	11,515	2,255	823,900	0.65	0.19	17,217	3,450
0.80	464,900	0.73	0.20	10,912	2,049	727,500	0.67	0.20	15,671	3,207
0.90	405,600	0.76	0.21	9,911	1,877	602,800	0.71	0.21	13,760	2,790
1.00	344,100	0.80	0.22	8,849	1,668	477,500	0.76	0.22	11,669	2,316

AuEq Cutoff (g/t)	Measured + Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
0.30	1,581,900	0.62	0.17	31,486	6,060	765,700	0.43	0.11	10,585	1,856
0.40	1,555,000	0.63	0.17	31,264	5,958	635,800	0.47	0.12	9,608	1,682
0.50	**1,509,900**	**0.64**	**0.18**	**30,842**	**5,990**	**514,900**	**0.51**	**0.14**	**8,442**	**1,589**
0.60	1,444,800	0.65	0.18	30,151	5,853	425,700	0.54	0.16	7,390	1,501
0.70	1,335,500	0.67	0.19	28,732	5,705	358,300	0.56	0.17	6,451	1,343
0.80	1,192,400	0.69	0.20	26,583	5,256	278,600	0.59	0.19	5,285	1,167
0.90	1,008,400	0.73	0.21	23,671	4,667	205,800	0.63	0.20	4,169	907
1.00	821,600	0.78	0.22	20,518	3,984	145,900	0.67	0.21	3,142	675

17.12 CONCEPTUAL PIT RESULTS

The mineral resources summarized in Table 17.25 through to Table 17.27 were tabulated as "global resources" using gold equivalent cutoff grades. As a preliminary test to determine "reasonable expectations for economic viability", RMI generated three conceptual pits for the Kerr, Sulphurets, and Mitchell deposits using the Lerchs-Grossmann (LG) algorithm. In addition to a "base case" pit, "pessimistic" and "optimistic" cases were also run. Measured, indicated, and inferred mineral resources were used for all three cases. For the base case conceptual pits, gold prices of US$800/oz and copper prices of US$2.00/lb were used. Other key parameters that were used to generate the conceptual pits are summarized in Table 17.28.

Table 17.28 Conceptual Pit Parameters

Parameters	Base Case	Pessimistic Case	Optimistic Case
Au Price (US$/troy oz)	$800	$650	$950
Cu Price (US$/lb)	$2.00	$1.00	$2.50
Ag Price (US$/troy oz)	$12	$8	$16
Mo Price (US$/lb)	$25.00	$15.00	$40.00
Au Recovery	78%	70%	80%
Cu Recovery	85%	80%	90%
Ag Recovery	73%	65%	75%
Mo Recovery	50%	40%	60%
Mining Cost (US$/t mined)	$1.50	$1.75	$1.25
Processing Cost (US$/ore tonne)	$5.00	$6.00	$4.00
G&A Cost (US$/ore tonne)	$0.60	$0.75	$0.50
Slope Angle (degrees)	45	40	45

The categorized measured, indicated, and inferred mineral resources inside of the Kerr, Sulphurets, and Mitchell conceptual pits are summarized using a 0.50 g/t Au equivalent cutoff grade in Table 17.29 through to Table 17.31, respectively.

Table 17.29 Kerr Conceptual Pit Results

AuEq Cutoff (g/t)	Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
Base Case	215,900	0.24	0.42	1,666	1,999	55,200	0.20	0.40	355	487
Pessimistic	128,200	0.26	0.48	1,072	1,356	12,000	0.25	0.44	96	116
Optimistic	222,400	0.23	0.41	1,645	2,010	63,200	0.19	0.4	386	557

Table 17.30 Sulphurets Conceptual Pit Results

AuEq Cutoff (g/t)	Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
Base Case	87,300	0.72	0.27	2,021	520	159,900	0.63	0.17	3,239	599
Pessimistic	86,900	0.72	0.27	2,012	517	120,800	0.67	0.2	2,602	532
Optimistic	87,300	0.72	0.27	2,021	520	160,800	0.63	0.17	3,257	602

Table 17.31 Mitchell Conceptual Pit Results

AuEq Cutoff (g/t)	Measured Mineral Resources					Indicated Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
Base Case	579,300	0.66	0.19	12,292	2,426	930,700	0.62	0.18	18,552	3,692
Pessimistic	576,700	0.66	0.19	12,237	2,415	925,000	0.62	0.18	18,438	3,670
Optimistic	579,300	0.66	0.19	12,292	2,426	930,700	0.62	0.18	18,552	3,692

AuEq Cutoff (g/t)	Measured + Indicated Mineral Resources					Inferred Mineral Resources				
	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)	Tonnes (000)	Au (g/t)	Cu (%)	Au (000 oz)	Cu (M lb)
Base Case	1,510,000	0.64	0.18	30,845	6,118	512,010	0.51	0.14	8,395	1,580
Pessimistic	1,501,700	0.64	0.18	30,676	6,085	457,122	0.53	0.14	7,789	1,410
Optimistic	1,510,000	0.64	0.18	30,845	6,118	513,969	0.51	0.14	8,427	1,586

RMI notes that, in general, the base case conceptual pits capture nearly all of the measured and/or indicated mineral resources in each of the models. Ongoing work will refine mining and processing costs for future conceptual pits that will be developed later this year.

17.13 RISKS AND UNCERTAINTIES

At this juncture, RMI is not aware of any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the estimate of mineral resources.

This project is still relatively early in the pre-development stage with numerous studies that are ongoing with significant additional work that will be required in defining the limits of the deposit, identifying possible processing methods, and (if warranted) permitting the project into a developing/producing property. Mining and processing costs, metal recovery, and permitting could materially affect the viability and possible size of this deposit. It is too early to determine the potential impact of these topics on the ultimate size or viability of this project.

18.0 MINING

18.1 INTRODUCTION

A production schedule, based on 120,000 t/d mill feed schedule at a PEA-level, has been developed for the KSM mine as an update to the NI 43-101 report entitled "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008" dated December 19, 2008. Pit phases have been engineered from the results of an updated economic pit limit analysis. Updated pit delineated resources are tabulated in Table 18.1 using:

- whole block grades with 5% mining dilution and 5% mining loss (dilution grades estimated in Table 18.2 represent the average grade of material below the incremental cutoff grade for each pit area)
- waste/mineralized material cutoff grades (COGs) based on a net smelter return (NSR) of Cdn$6.85/t.

Grade items used in this section have been interpolated by Inverse Distance Weighting (IDW), as described in the Section 17.0 of this PEA. The grade items used are copper (CUIDW), gold (AUIDW), silver (AGIDW), and molybdenum (MOIDW).

Table 18.1 Summarized Indicated and Inferred Pit Delineated Resource

Pit	Mineralized Material (kt)	Diluted Grades					Waste (kt)	Strip Ratio (t:t)
		NSR (Cdn$/t)	CUIDW (%)	AUIDW (g/t)	AGIDW (g/t)	MOIDW (ppm)		
Mitchell								
M621	110,690	26.4	0.215	0.815	2.98	34.6	49,840	0.45
M622i	114,582	20.4	0.151	0.660	2.92	49.3	147,538	1.29
M623i	182,816	20.9	0.165	0.658	2.65	61.7	151,514	0.83
M624i	263,954	19.8	0.156	0.617	2.87	59.0	717,673	2.72
M625i	355,390	18.8	0.159	0.572	2.82	56.8	424,005	1.19
Sub-total	**1,027,432**	**20.4**	**0.164**	**0.634**	**2.83**	**55.0**	**1,490,570**	**1.45**
Kerr								
K611	166,054	22.729	0.465	0.2506	0	0	157,908	0.95
Sulphurets								
S611	174,144	23.402	0.2152	0.6562	0.28	75.7	489,496	2.81
Total	**1,367,630**	**21.1**	**0.207**	**0.591**	**2.162**	**51.0**	**2,137,974**	**1.6**

Table 18.2 Dilution Grades

	Mitchell Pit Area	Kerr Pit Area	Sulphurets Pit Area
Cu (%)	0.094	0.130	0.067
Au (g/t)	0.153	0.141	0.194
Ag (g/t)	1.78	-	0.34
Mo (ppm)	31.3	-	45.4
NSR (Cdn$/t)	5.50	5.48	5.52

18.2 INTRODUCTION

The mine planning work for this PEA Addendum 2009 is based on NI 43-101 published resource models dated March 30, 2009 by RMI.

The mine planning for the KSM mineral property is based on work done with MineSight®, a suite of software well proven in the industry. This includes the resource model, pit optimization (MineSight Economic Planner [MS-EP]), detailed pit design, and optimized production scheduling (MineSight Strategic Planner [MS-SP]).

In addition to the geological information used for the block model, other data used for the mine planning includes the base economic parameters, mining cost data derived from supplier estimates and from other projects in the local area, recommended preliminary pit slope angles, and projected project metallurgical recoveries, plant costs and throughput rates.

18.3 MINING DATUM

Project design work is based on NAD83 coordinates. The historical drill hole information is based on various surveys with different sets of control that have been converted to NAD83 and, in particular, a January 2009 topography surface produced from a 2008 LiDar. Effort has been made to ensure that all disciplines are using the same topography data.

18.4 PROJECT PRODUCTION RATE CONSIDERATION

A number of factors are considered when establishing an appropriate mining and processing rate. Key factors in relation to KSM are as follows:

- **Resource Size** – Typically, mine life is set at 12.5 to 20 years. For anything beyond this, time value discounting shows insignificant contribution to the net present value (NPV) of the project, and capital investment typically is targeted at projects with a payback of 3 to 5 years.

- **Operational Constraints** – Power, water, or resources for operations support can limit production but these are not assumed to be limiting at this stage.
- **Construction Constraints** – Physical size and weight of equipment and shipping limits can determine the maximum size of available units. For this evaluation, the largest proven units are assumed.
- **Project Financial Performance** – Generally, economies of scale can be realized at higher production rates and lead to reduced unit operating costs. These are tempered to the above-mentioned physical and operational constraints and generally higher capital requirements for higher tonnage throughputs. Higher production rates generally pay back fixed capital at a faster rate, thereby improving project NPV.

A throughput of 120,000 t/d sets the mine life at 30 years for the pit delineated resources. Project NPV may be improved by increasing the mill throughput above 120,000 t/d, which would require some re-engineering of mining phases to provide sufficient working bench widths for additional mine equipment. Mill throughput is currently constrained by the conveyor capacity. Increasing the mill throughput in further studies may significantly improve project economics, but any NPV benefit from increased throughput is likely to be offset by losses due to increased pre-production costs from larger starter pit phases.

18.5 MINE PLANNING 3D BLOCK MODEL AND MINESIGHT® PROJECT

Two resource models used in this study are based on MineSight® models used for the resource statements included in the resource section of this report.

The Kerr MineSight® resource model contains whole block Cu (%) and Au (g/t) grades, while the Sulphurets/Mitchell/IronCap (SMIC) MineSight® resource model contains whole block Cu (%), Au (g/t), Ag (g/t), and Mo (ppm) grades. It should be noted that the Iron Cap zone in the northeast of the SMIC model was not modelled in the 2008 PEA and all resources in the Iron Cap zone in this 2009 PEA resource model are classed as speculative and are therefore not included in pit delineated resources.

Each of the resource models also contains an SG (density) item and a TOPO item representing the proportion of a block below topography.

Mine planning 3D block models (3DBM) have been created for each model area (Kerr, SMIC).

The 2009 PEA model dimensions are shown in Figure 18.1 and Figure 18.2 for each model. The pit areas are illustrated for orientation in plan view in Figure 18.3.

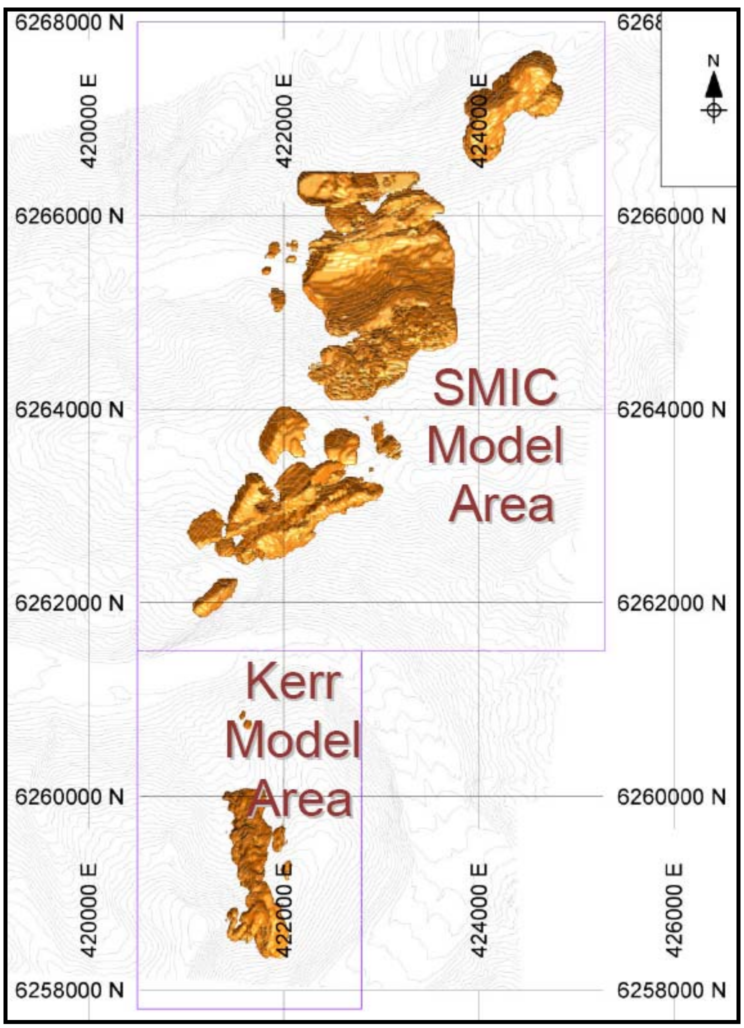
Figure 18.1 SMIC Mine Planning Model Limits

	Minimum	Maximum	Size	Number
X	420500	425300	25	192
Y	6261500	6268000	25	260
Z	-210	2055	15	151

Figure 18.2 Kerr Mine Planning Model Limits

	Minimum	Maximum	Size	Number
X	420500	422800	25	92
Y	6257800	6261500	25	148
Z	450	1950	15	100

Figure 18.3 KSM Model Areas and NSR >$6/t Grade Shells – Plan View



18.5.1 NET SMELTER RETURN

COGs are determined using the NSR in Cdn$/t, which is calculated using Net Smelter Prices (NSP) as calculated in Appendix D. The NSR (net of offsite concentrate and smelter charges and onsite mill recovery) is used as a cutoff item for break-even mineralized material/waste selection and for the grade bins for cash flow optimization. The NSP is based on base case metal prices, US$ exchange rate, and offsite transportation, smelting, and refining charges, etc. (see Appendix D). The metal prices and resultant NSP used are shown in Table 18.3.

Table 18.3 Metal Prices and NSP

	Metal Price (US$)	NSP (Cdn$)
Cu	2.00/lb	2.00/lb
Au	750/oz	25.7/g
Ag	12.50/oz	0.378/g
Mo	12.50/lb	10.30/lb

Metallurgical recoveries used for the NSR calculation are based on 2008 composite test work:

- Cu Recovery (%): RecCu = 197.832 x (Cu) + 61.252
  - *where Cu is Copper head grade in %*
  - *where RecCu is maximum of 95%*
- Ag Recovery (%): RecAg = 74.3%
- Au Recovery (%):

Au Grade (g/t)	0-0.25	0.25-0.5	0.5-1.0	>1.0
RecAu (%)	30	73	78	80

- Mo Recovery (%):

Mo Grade (ppm)	0-50	50-100	>100
RecMo (%)	0	40	60

The NSR formula is:

$$NSR = \frac{Cu}{100} \times \frac{RecCu}{100} \times NSPCu \times 2204.6 + Au \times \frac{RecAu}{100} \times NSPAu + Ag \times \frac{RecAg}{100} \times NSPAg + \frac{Mo}{1 \times 10^6} \times \frac{RecMo}{100} \times NSPMo \times 2204.6$$

Where:

- Cu = copper grade (%)
- Au = gold grade (g/t)
- Mo = molybdenum grade (ppm)

- Ag = silver grade (g/t)
- RecCu = copper recovery (%)
- RecAu = gold recovery (%)
- RecMo = molybdenum recovery (%)
- RecAg = silver recovery (%)
- NSPCu = NSP for copper ($/lb)
- NSPAu = NSP for gold ($/g)
- NSPMo = NSP for molybdenum ($/lb)
- NSPAg = NSP for silver ($/g).

18.5.2 *MINING LOSS AND DILUTION*

The KSM zones are to be mined with large truck/shovel operations at a mill feed mining rate of 120,000 t/d feeding a conventional copper concentrator and gold cyanide leaching circuit. The mining is described as typical hard rock open pit bulk mining method. Large equipment will be used and high mining rates are planned to ensure the lowest possible unit costs for mine operations. The waste and mineralized material will require blasting and typical grade control methods using blast hole sampling. Blast hole kriging will possibly be used to determine COGs and digging control limits for the mining shovels. Blast heave, the lack of loading selectivity, haul back in the trucks, and stockpile reclaim will create some mineralized material loss (mining recovery) and dilution as the material moves from in situ modelled resource to ROM mill feed. Since the ROM mill feed determines the production schedule and revenue stream for the project, proper evaluation of the mining loss and dilution is required. The definition of the mining parameters used in the pit delineated resource calculations is also a NI 43-101 reporting requirement.

The 3DBMs for KSM are based on separate lithological/geostatistical domains. Mineralized material zones have been combined into single whole block grades of copper, gold, silver, and molybdenum grade values for each block for the SMIC model. For the Kerr model, mineralized material zones have been combined into single whole block grades of copper and gold grade values for each block. As such the grade values in each block are “whole block diluted”.

With the planned bulk mining method, a means of determining the mining loss and dilution applicable to the KSM resource model is needed that will reflect the ROM production from the mining operations. Mineralized zones in the 3DBM are made up of relatively large contiguous blocks of ‘ore’ above the COG. There are areas however where isolated blocks of mineralized material are surrounded by waste and also isolated blocks of waste that are surrounded by mineralized material. Higher COGs will result in fewer contiguous blocks and more isolated blocks. Conversely, lower COGs will merge more of the indicated isolated blocks into close-by contiguous blocks.

Mining operations will use blast hole samples on 7 to 10 m spacing to determine the cutoff boundaries for shovel dig limits. "Included" mineralized material and waste blocks on the small blast hole sampling grid will be too small to separate from the shovel face especially after being displaced by blasting. This inclusion of isolated blast hole blocks is handled since the larger blocks in the 3DBM will average-in the isolated blocks from any future blast hole models.

The 3DBM uses 25 m by 25 m by 15 m blocks for all the model areas. Each block represents approximately 5 h of digging for the shovels. With blocks of this magnitude, it can be assumed in the PEA Addendum planning that isolated blocks from the larger 3DBM will be selectively mined on a full block basis and will not be lost or included in the mineralized material. However, bulk mining will cause dilution to the blocks (either mineralized material into waste or waste into mineralized material by neighbouring blocks) where contact is made between mineralized material grade material and waste.

Other mining losses are also noted in mining operations mainly due to misdirected loads, haul back in frozen truck boxes, and stockpile cleanup. These types of losses are small but need to be accounted for.

The economic pit delineated resource is calculated from the resource models, within an economic pit limit using the applicable mining recovery and dilution parameters. The resources in the model are quantified as mineralized material or waste based on a NSR cutoff.

Mining recovery and dilution parameters, in addition to the whole block dilution from the 3DBM, are required to account for the following:

- dilution of waste into mineralized material where blasting "throws" waste into mineralized material at mineralized material/waste boundaries
- loss of mineralized material into waste where blasting "throws" mineralized material into waste diluting the mix below COG
- general mining losses due to haul back from frozen or sticky material in truck boxes, misdirected loads, and repeated handling such as stockpile reclaim.

For this PEA Addendum, an allowance has been made for a mining dilution of 5% and a mining loss of 5%. Since the dilution material on the contact edge of the blocks described above is mineralized, it will have some grade value. The dilution grades are estimated by determining the grades of the envelope of waste in contact with mineralized material blocks inside the pit delineated area. This is estimated by statistical analysis of grades in blocks below the design basis cutoff of Cdn$6.24/t. The dilution grades are estimated in Table 18.4 representing the average grade of material below the incremental COG.

Table 18.4 Dilution Grades

	Mitchell Pit Area	Kerr Pit Area	Sulphurets Pit Area
Cu (%)	0.094	0.130	0.067
Au (g/t)	0.153	0.141	0.194
Ag (g/t)	1.78	-	0.34
Mo (ppm)	31.3	-	45.4
NSR (Cdn$/t)	5.50	5.48	5.52

18.6 ECONOMIC PIT LIMITS, PIT DESIGNS

The economic pit limit is determined using the MS-EP optimization routines in MineSight® which are based on the LG algorithm. The LG algorithm runs against the 3DBM, evaluating the costs and revenues of the blocks within potential pit shells. The routine uses input costs, NSP, plant recoveries, and overall slope angles to expand downwards and outwards from previous interim economic 3D surfaces until the last increment is at break-even economics. Additional cases are included in the analysis to evaluate the smaller high grade pits versus larger lower grade or higher strip ratio pit shells and also different slope angles. Time value block discounting is also evaluated to determine the NPV effect of the delay between earlier stripping costs to the revenue released from deeper mineralized material.

18.6.1 PIT OPTIMIZATION METHOD

Economic pit limit is selected after evaluating LG pit sensitivity cases conducted with MS-EP. The assessment is carried out in two steps:

- Step 1 – Economic Pit Limit Sensitivities
  - Generate sets of LG pit shells using MS-EP-design by varying revenue assumptions and pit slope to test the ‘ore’ body geometric/topographic and pit slope sensitivity.
- Step 2 – Economic Pit Limit Evaluation
  - Generate mineable phases inside selected LG pit limits using MS-EP-design, and then assess the economic (NPV) potential from the mineable phases by scheduling with MS-EP-evaluate.

Two potential economic pit limits are estimated. The first economic pit limit case is a smaller pit determined using the optimum NPV (5%) discounted schedule. The second larger economic pit limit is determined by estimating the pit size where an incremental increase in pit size does not significantly increase the pit resource. Economics of the larger pit limits are tested by a NPV (0%) undiscounted schedule.

18.6.2 ECONOMIC PIT LIMIT SENSITIVITIES

The design basis for the LG pit limit assessment is described in this section.

MINING COSTS

Incremental mining costs are estimated from the 2008 PEA mine cost model. Mining costs for the economic pit limit assessment assumes that all waste from all pit areas is placed in the designed Mitchell Valley waste dump.

The 2008 PEA life-of-mine (LOM) unit mining costs per tonne of material mined is shown in Table 18.5.

Table 18.5 2008 PEA LOM Unit Mining Costs

	LOM Material Mined ($/t)
Drilling	0.07
Blasting	0.25
Loading	0.15
Hauling	1.02
Mine Maintenance	0.02
Mine Operations – Support	0.29
Snow Removal	0.02
Geotechnical	0.02
Unallocated labour cost	0.02
Direct Costs – Subtotals	**1.86**
Mine Operations G&A	0.02
Mine Maintenance G&A	0.02
Mine Engineering G&A	0.02
Technical Services G&A	0.01
Total GME Costs	**0.06**
Total Operating Cost	**1.92**

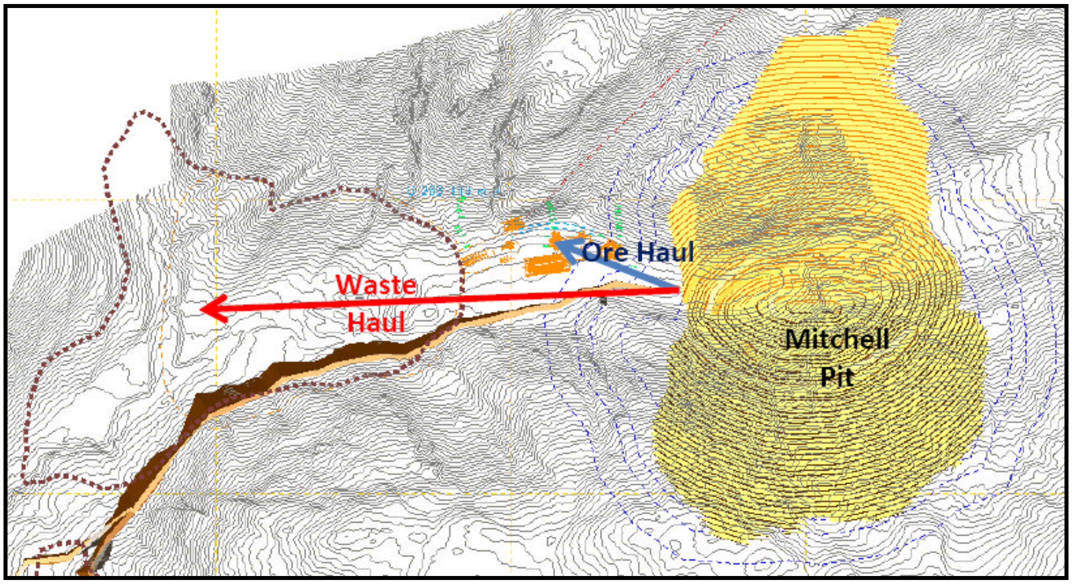
Abbreviations:
G&A = general and administrative.
GME = general mine expense.

The MS-EP routine splits the unit mining costs into variable (hauling) and fixed (all the other mining costs) so that variable costs by area and bench can be applied. This allows the effect of higher cost mining, such as longer hauls or more uphill hauling from lower benches, to be assessed in the LG runs. The costs from the previous study can be split out if the haulage costs for post mining backfill are included in the mining costs, and remaining haulage costs for ore and waste mined are excluded, then the 2008 PEA fixed unit mining cost is an estimated $0.92/t material mined.

MITCHELL HAULAGE

Haulage for the Mitchell pit is estimated in Figure 18.4. It's the shortest haul out of all three pit areas but the top bench material (up to 1900 m) has to be hauled down to the lower elevation dump areas and lower benches need to be hauled up, both at significant cost.

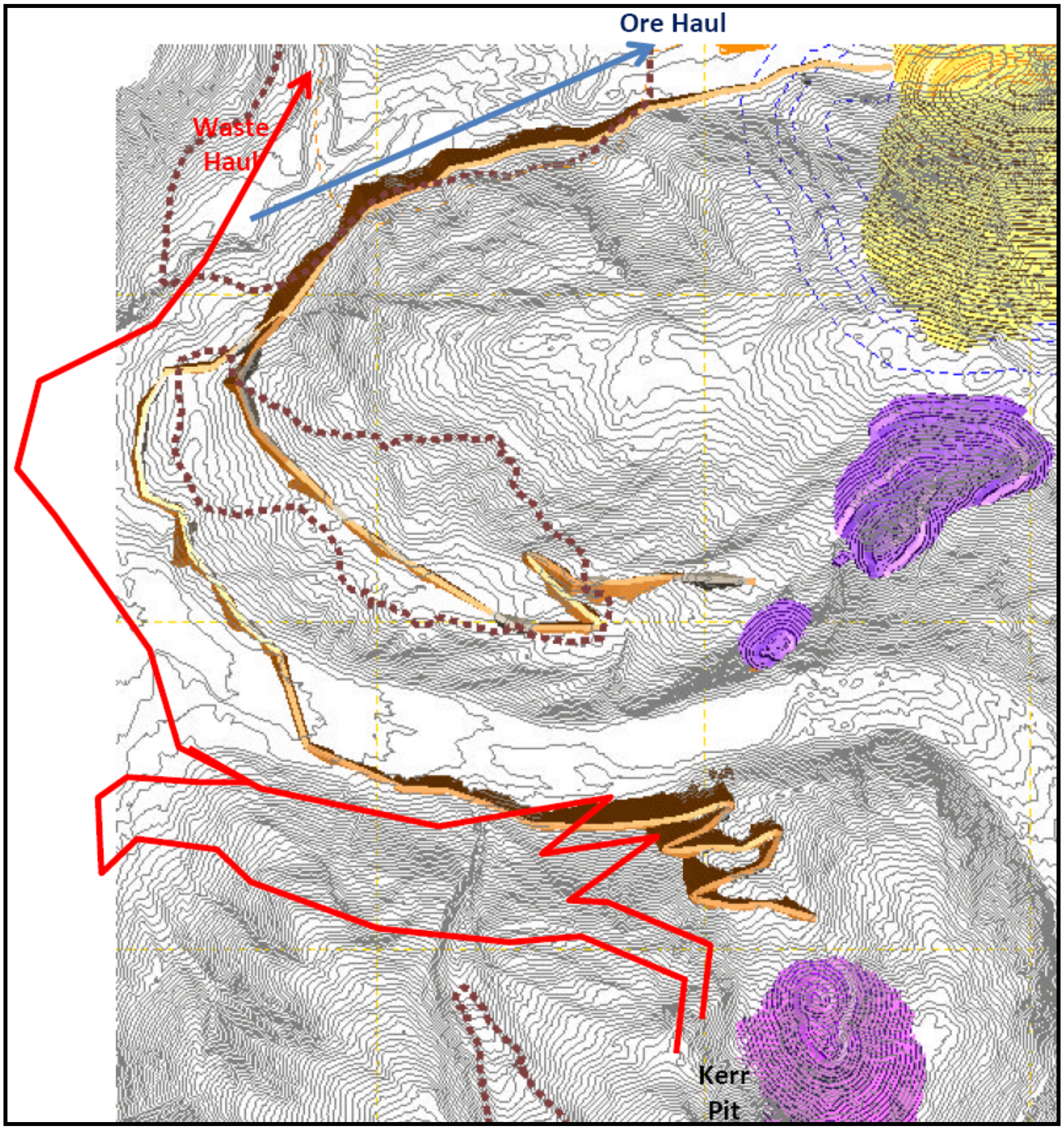
Figure 18.4 Mitchell Pit Summarized Estimated Haulage Route



KERR HAULAGE

Haulage for the Kerr pit is estimated in Figure 18.5. It's the longest haul of all the pits. This LG mining cost assessment will assume that all Kerr waste will be hauled to the Mitchell Valley and not to the potential waste location to the east of Kerr.

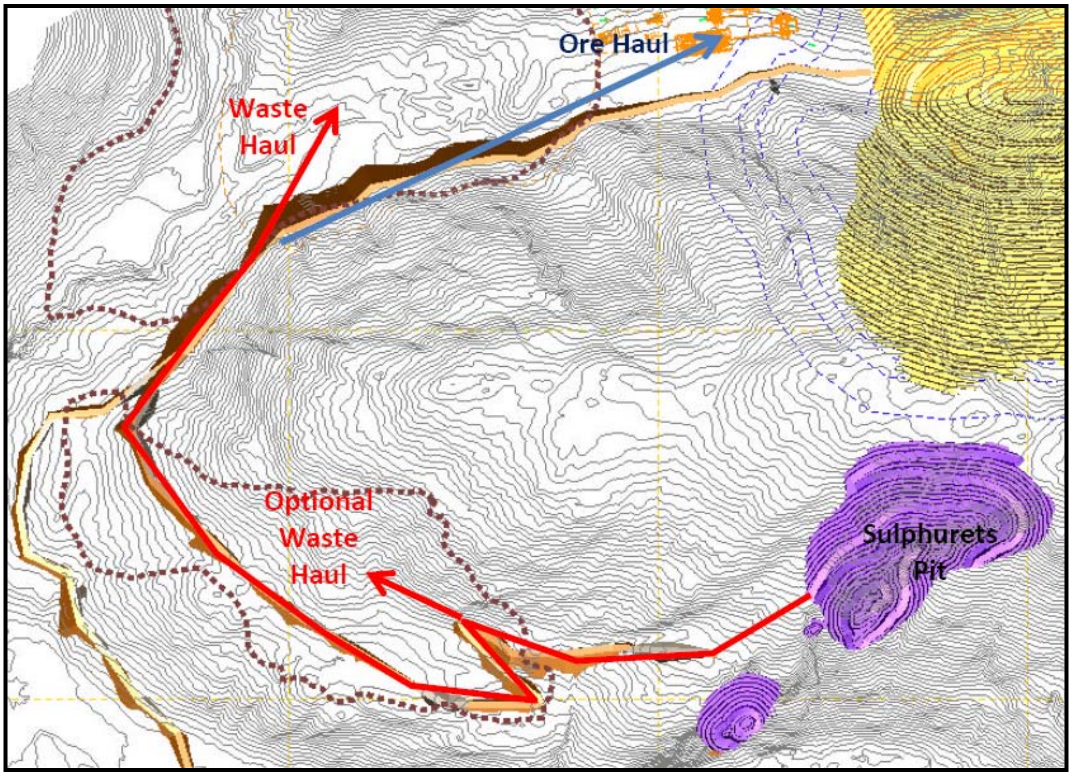
Figure 18.5 Kerr Pit Summarized Estimated Haulage Route



SULPHURETS HAULAGE

Haulage for the Sulphurets pit is estimated in Figure 18.6. This LG mining cost assessment will assume that all Sulphurets waste will be hauled to the Mitchell Valley and not to the potential Sulphurets waste location.

Figure 18.6 Sulphurets Pit Summarized Estimated Haulage Route



INCREMENTAL MINING COSTS

Incremental LG mining costs are estimated in Table 18.6 based on simulations of haul profiles estimated using the destinations illustrated in Figure 18.4 to Figure 18.6.

Table 18.6 Economic Pit Limit Incremental Mining Costs

	Rim Elevation (m)	Haul & Return Cycle Time (min)	Haul Cost (Cdn$/t)	Incremental Mining Costs (Cdn$/t)	
				Mining Cost at Pit Rim	Mining Cost at Top Bench
Mitchell Waste From Pit Rim (780 m)	780	12.8	0.27	1.19	2.24
Mitchell Ore From Pit Rim (780 m)	780	8.7	0.19	1.11	2.15
Kerr Waste From Pit Rim (1,140 m)	1140	53.8	1.15	2.07	2.74
Kerr Ore From Pit Rim (1,140 m)	1140	57.0	1.22	2.14	2.81
Sulphurets Waste From Pit Rim (1,290 m)	1290	35.6	0.76	1.68	2.31
Sulphurets Ore From Pit Rim (1,290 m)	1290	38.8	0.83	1.75	2.38

Assumptions:

Payload for Haul Truck:	350 t
Operating Cost for Haul Truck:	$540/Operating Hour
Mining Costs Excluding Haulage:	$0.92/t

Incremental Cost Below Pit Rim:

Haul and Return per Bench:	0.96 min
Incremental Cost:	$0.02469/t per bench

Incremental Cost Above Pit Rim:

Haul and Return per Bench:	0.48 min
Incremental Cost:	$0.01234/t per bench.

PIT ACCESS AND DEVELOPMENT COSTS

The steep terrain at KSM is a significant consideration for initial mining and requires the assessment of access development costs to each of the pit areas and phases. This assessment is required to ensure that incremental pit phases will be able to pay back the capital required to pioneer and develop access and any preliminary mining, such as cast blasting and dozing the first few benches downslope, etc., on an incremental basis. Under the objective of incremental economic justification, the pre-production development, pre-stripping, and project infrastructure can be assumed to be justified by the whole project in general, but each phase or push-back needs to justify the capital cost of bringing it into production on an incremental basis. These deferred development capital costs are not included in the incremental unit mining costs and are instead compared to the NPV of the incremental phases to make sure that the incremental phases can pay back their individual development costs.

Results from the conceptual pit development assessments are estimated in Table 18.7 and in Appendix D.

Table 18.7 Economic Pit Limit – Incremental Development Capital Estimate

Pit Area	Capital (Million Cdn$)
Mitchell North	82
Mitchell South	0.3
Sulphurets	6.3
Kerr *	10

* Capital estimate is an allowance; Kerr access and development cost studies are ongoing.

PIT SLOPE ANGLE

Maximum overall pit slope angles (PSA) are based on three alternative recommendations by BGC, conservative, base, and optimistic sensitivity cases for this 2009 PEA Addendum assessment.

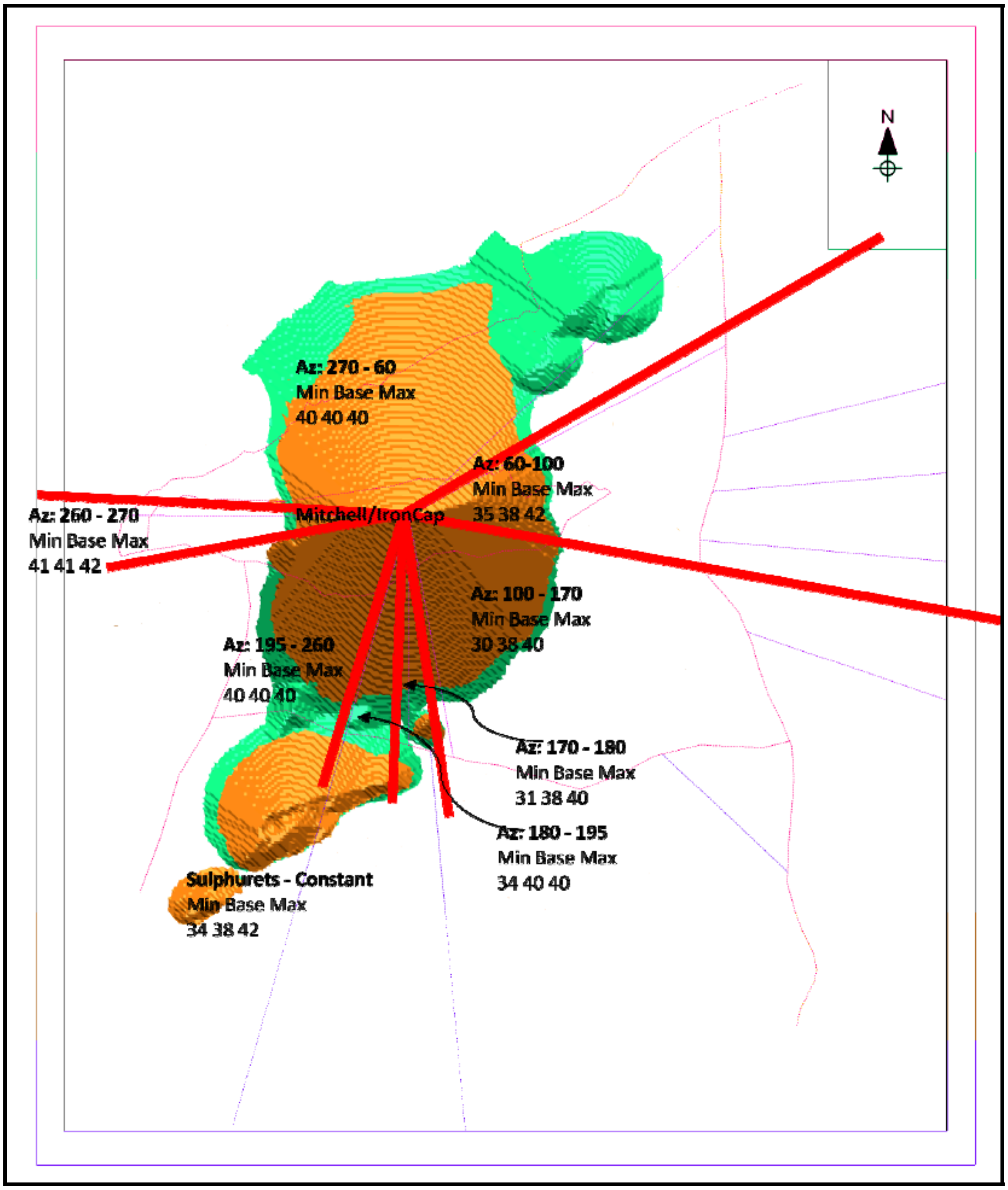
PSA is varied by azimuth for Mitchell; at this PEA stage of the study, PSA is constant for the Kerr and Sulphurets pits. PSA assumptions for the Mitchell pit are shown in Table 18.8.

Table 18.8 Mitchell Iron Cap PSA by Sector

Azimuth (°)	Conservative PSA (°)	Base PSA (°)	Optimistic PSA (°)
0	40	40	40
60	40	40	40
95	37	39	41
100	35	38	42
160	32	38	41
170	31	38	40
180	34	40	40
190	37	40	40
195	40	40	40
260	41	41	42
270	40	40	40

The PSA slope sectors for Mitchell are illustrated in Figure 18.7.

Figure 18.7 Mitchell Pit Slope Angle Sectors



An additional PSA case tests the Mitchell economic pit limit sensitivity by reducing the PSA to 37°.

Sulphurets and Kerr PSA assumptions are shown in Table 18.9 and Table 18.10.

Table 18.9 Sulphurets Constant PSA

Conservative	Base	Optimistic
34°	38°	42°

Table 18.10 Kerr Constant PSA

Conservative	Base	Optimistic
38°	40°	42°

Process Recoveries

Process recovery assumptions are shown in Table 18.11.

Table 18.11 Process Recovery Assumptions

Recovery	Unit				
Copper (maximum of 95%)	%	97.832 x (copper feed grade, %) + 61.252			
Silver	%	73			
Gold	Au g/t	0-0.25	.25 - .50	.50 - 1	> 1
	Au Rec	30%	73%	78%	80%
Molybdenum	Mo ppm		0-50	50-100	>100
	Mo Rec		0%	40%	60%

Metal Prices

Base case metal price assumptions are as follows:

- Cu = US$2.00 /lb
- Au = US$750/oz
- Ag = US$12.5/oz
- Mo = US$12.5/lb.

The NSP is based on base case metal prices, US$ exchange rate, offsite transportation, smelting and refining charges, etc. (the smelter schedule is available in Appendix D).

Base case NSP are as follows:

- Cu = Cdn$2.00/lb
- Au = Cdn$25.7/g

- Ag = Cdn$0.387/g
- Mo = Cdn$10.30/lb.

LG pit shells are generated by varying revenues available for mining by changing the input metal prices in the range from 30 to 150% of the base NSP, as shown in Table 18.12.

Table 18.12 Economic Pit Limit – Pit Number for Each Input Price

Pit #	Price Case (%)
1	30.0
2	35.0
3	40.0
4	45.0
5	50.0
6	55.0
7	60.0
8	65.0
9	70.0
10	75.0
11	80.0
12	85.0
13	90.0
14	95.0
15	**100.0**
16	105.0
17	110.0
18	115.0
19	120.0
20	125.0
21	130.0
22	135.0
23	140.0
24	145.0
25	150.0

Sensitivity cases are run for the various pit slope assumptions described above. Sulphurets and Mitchell LG pits are initially evaluated in a combined search to account for shared mining costs when the large pits from these two deposits merge at the top of the pit slope. Figure 18.8 through to Figure 18.10 summarize the slope sensitivity cases for each series of pit shells for the combined Mitchell and Sulphurets pits, the Sulphurets pit, and the Kerr pit, respectively.

Figure 18.8 Mitchell/Sulphurets – Sensitivity of Pit Size to Pit Slope

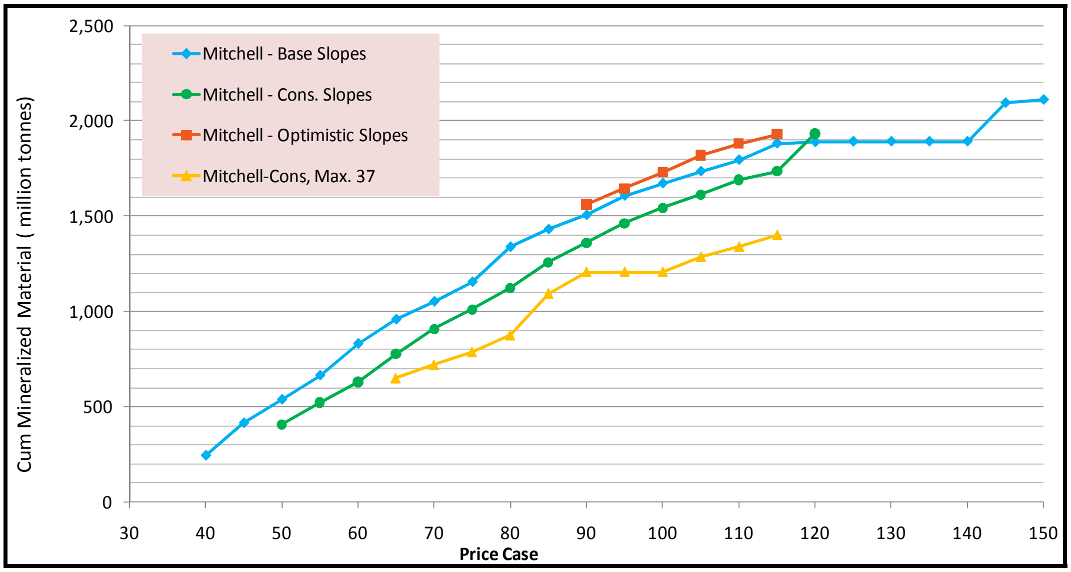


Figure 18.9 Sulphurets – Sensitivity of Pit Size to Pit Slope

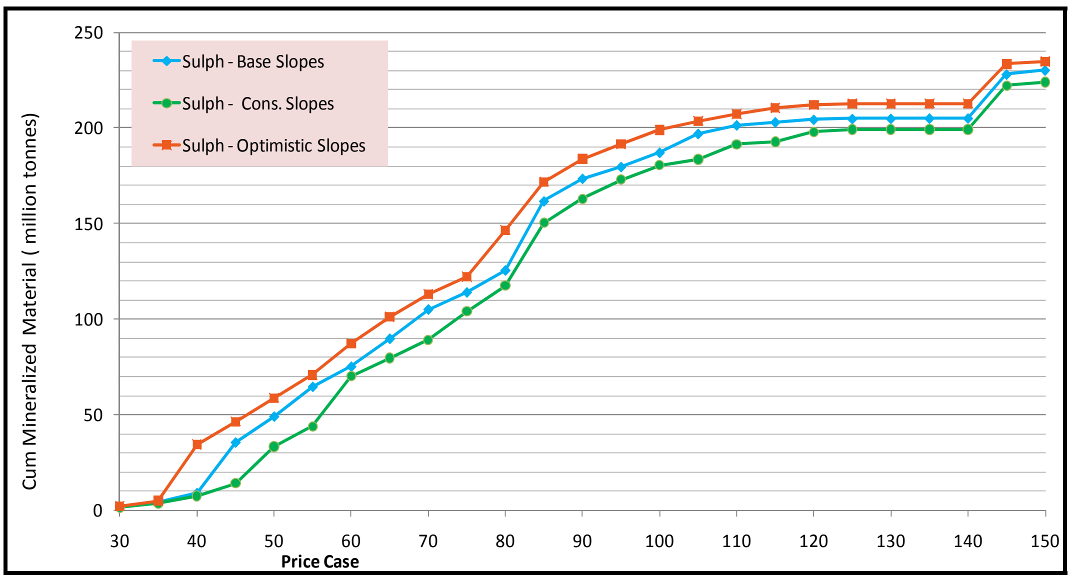
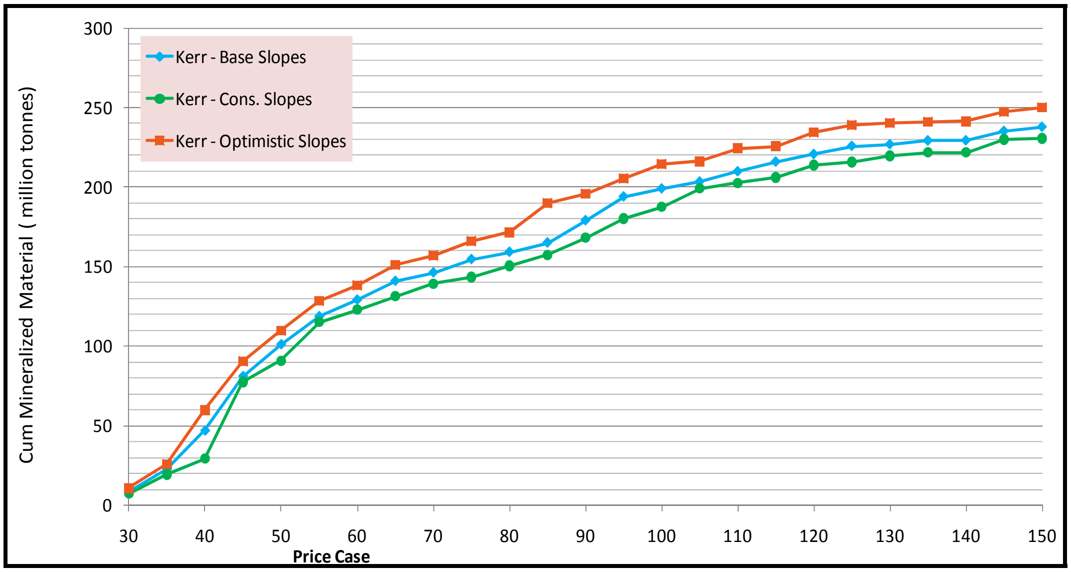


Figure 18.10 Kerr – Sensitivity of Pit Size to Pit Slope



Mitchell LG economic pit limit is significantly sensitive to the PSA assumptions, showing downside potential to the pit resource if the PSA is reduced from the current conservative design basis assumptions to 37°.

The Sulphurets and Kerr economic pit limit sensitivities indicate less sensitivity to the BGC-recommended PSA assumptions.

In Figure 18.8 to Figure 18.10, inflection points occur where an incremental increase in pit size does not significantly increase the pit resource. These inflection points represent potential economic pit limits and are selected for each pit area as follows:

- Mitchell – inflection price case: 115%
- Sulphurets – inflection price case: 110%
- Kerr – inflection price case: 95%.

The pit resources from LG pit limits selected from the inflection points are shown in Table 18.13.

Table 18.13 Summary of the Inflection Point LG Economic Pit Limit Resources

Pit	% of Base Case Pit	Category	Mineralized Material >Cutoff (kt)	In Situ NSR (Cdn $/t)	In Situ Grades Cu (%)	Au (g)	Ag (g)	Mo (%)	Waste (kt)	S/R (t/t)
Mitchell	115	Measured	538,001	21.4	0.185	0.673	3.09	0.0053		
		Indicated	784,433	19.9	0.173	0.629	2.97	0.0057		
		Inferred	344,757	14.0	0.090	0.503	2.33	0.0054		
		Sub-total	1,667,191	19.2	0.160	0.617	2.88	0.0055	4,425,881	2.65
Sulphurets	110	Indicated	91,326	25.3	0.254	0.705	0.28	0.0086		
		Inferred	110,226	21.2	0.190	0.645	0.25	0.0062		
		Sub-total	201,552	23.0	0.219	0.672	0.27	0.0073	634,708	3.15
Kerr	95	Indicated	166,955	22.4	0.455	0.255	-	-		
		Inferred	26,967	20.9	0.426	0.229	-	-		
		Sub-total	193,922	22.2	0.451	0.251	-	-	234283	1.21
All		Measured	538,001	21.4	0.185	0.673	3.09	0.0053		
		Indicated	1,042,714	20.8	0.225	0.576	2.69	0.0060		
		Inferred	481,950	16.0	0.131	0.520	1.83	0.0056		
		Total/Avg.	**2,062,665**	**19.8**	**0.193**	**0.588**	**2.60**	**0.0057**	**5,294,872**	**2.57**

18.6.3 ECONOMIC PIT LIMIT EVALUATION

Each pit area has increasing ore tonnes beyond the base case price assumptions in the cases above. To determine if continued expansion of the pit limit is economically viable from more marginally economic pit shells, the incremental contribution of successive 'push-backs' to project NPV are examined on an NPV basis. Economic pit limits have been chosen after estimating the NPV from selected pit cases from the sensitivity studies above where smaller incremental phases (skins) are combined to approximate reasonably mineable push-back widths. The NPV assessment is carried out with MS-EP-evaluate. NPV is discounted at 5% and then at 0% in each case and is determined by the following:

- Generation of minable phases inside the limiting LG pit. Mineable phases require a minimum mining width of 90 m, with sequential development of the mining phases so that push-backs are only expanded in one direction at a time (to approximate a degree of operability). The resulting LG phases represent approximate mineable phases.
- Production of preliminary optimized mining schedules using these mineable LG phases.
- No capital is included in the NPV calculation.
- Phase design and schedule optimization of each resource area is analyzed separately at a later stage in the engineering process.

Multiple mining push-backs are assumed for the Mitchell pit to even-out strip ratios. Sulphurets and Kerr pits are assumed to be single phase pits since the ultimate pits aren't big enough to allow reasonably sized multiple phases..

The production targets per year in MS-EP-evaluate LG pit assessment for the Mitchell pit are as follows:

- Year 1: Ore = 21,000 kt Waste = 110,000 kt
- Years 2-40: Ore = 43,000 kt Waste = 240,000 kt.

The production targets in MS-EP-evaluate for the Sulphurets and Kerr pits are as follows:

- All Years: Ore = 21,000 kt Waste = 21,000 kt.

Stockpiling for optimized COG is made available for all periods. The stockpile rehandle cost is estimated at Cdn $0.50/t. Graphs of the NPV (excluding capital costs) for a range of pit sizes are shown in Figure 18.11 through to Figure 18.13 for the Mitchell, Sulphurets, and Kerr pits, respectively. It should be noted that Cumulative Present Value (CPV) is synonymous with NPV in these figures.

Figure 18.11 Mitchell – CPV Analysis

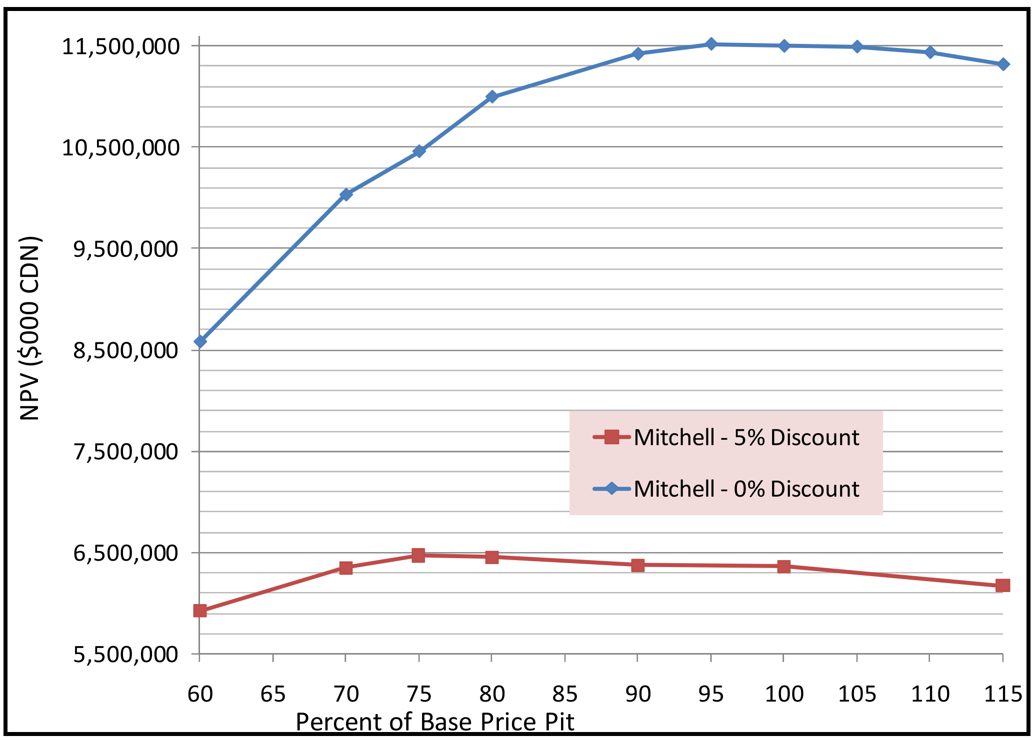


Figure 18.12 Sulphurets – CPV Analysis

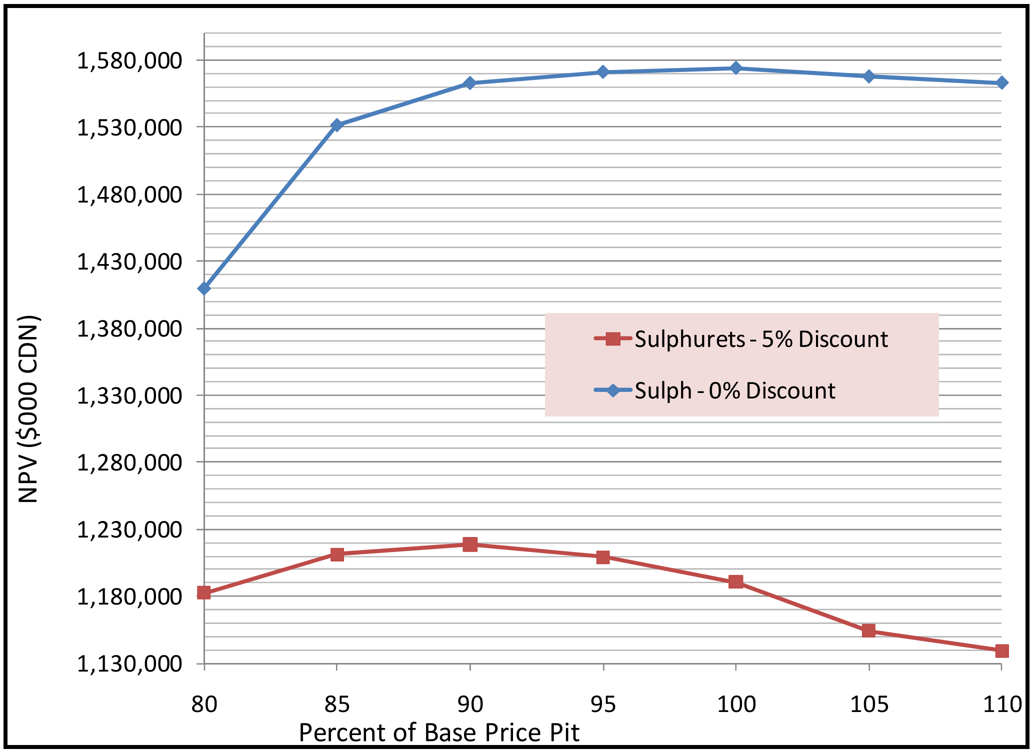
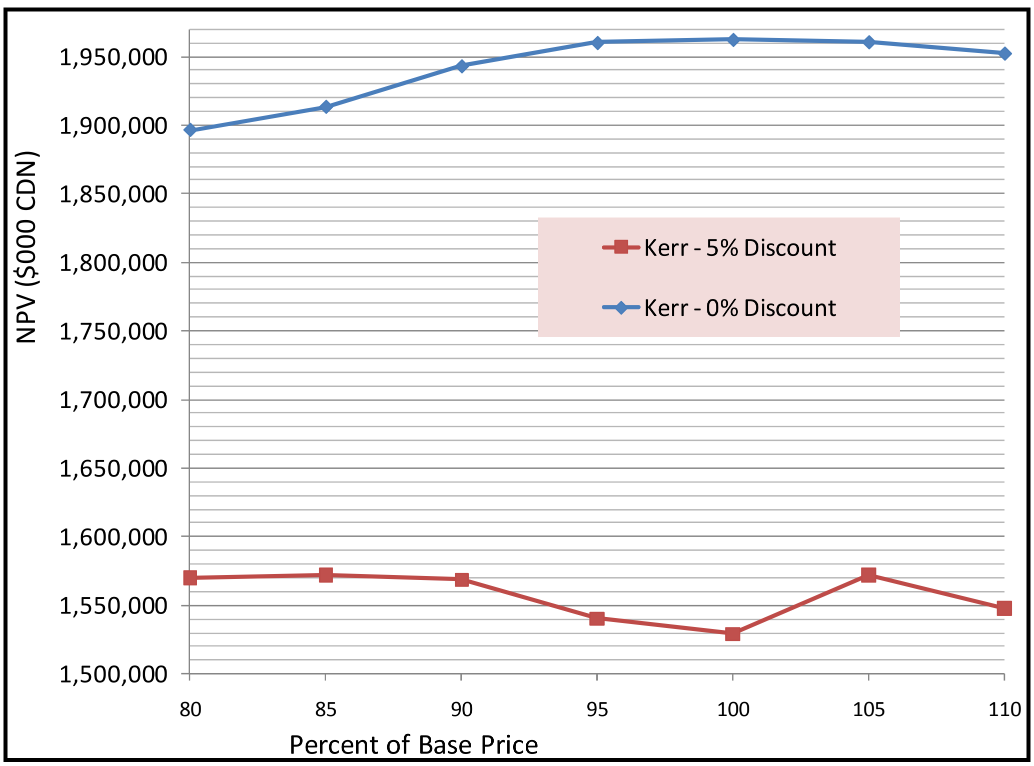


Figure 18.13 Kerr – CPV Analysis



Potential pit limits, selected from the peak discounted NPV (5%) values shown in Figure 18.11 to Figure 18.13, are as follows:

- Mitchell – Inflection Price Case: 75%
- Sulphurets – Inflection Price Case: 90%
- Kerr – Inflection Price Case: 85%.

It should be noted that, at these limits, the Mitchell and Sulphurets walls don't merge.

Resources delineated by the LG pit limits estimated from the NPV (5%) are summarized in Figure 18.14. The MS-EP-evaluate program estimates the NPV at a range of COGs and selects a COG that provides the best NPV. MS-EP-evaluate increases the cutoff NSR value above the processing plus G&A costs in each case tested. It should be noted that no allowance for an increase in capital has been accounted for in the NPV calculation due to an increase in mining rate as a result of COG optimization. The increase in mining rate due to COG optimization is not considered to have an effect on the choice of ultimate pit size.

Table 18.14 Summary of the NPV5 (Smaller) LG Economic Pit Limit Resources

Pit	% of Base Case Pit	Category	Mineralized Material >Cutoff (kt)	In Situ NSR (CDN$/t)	In Situ Grades				Waste (kt)	S/R (t/t)
					Cu (%)	Au (g)	Ag (g)	Mo (%)		
Mitchell	75	Measured	394,080	22.5	0.191	0.714	3.07	0.0054		
		Indicated	455,835	21.4	0.180	0.682	2.88	0.0059		
		Inferred	179,198	13.4	0.082	0.488	2.44	0.0056		
		Sub-total	1,029,113	20.4	0.167	0.661	2.88	0.0057	1,481,592	1.44
Sulphurets	90	Indicated	89,820	25.4	0.254	0.708	0.28	0.0087		
		Inferred	83,768	21.7	0.188	0.671	0.27	0.0066		
		Sub-total	173,588	23.6	0.222	0.690	0.28	0.0077	483,092	2.78
Kerr	85	Indicated	146,686	23.2	0.472	0.259				
		Inferred	18,315	21.3	0.398	0.147				
		Sub-total	165,001	23.0	0.468	0.257			152,766	0.93
ALL		Measured	394,080	22.5	0.191	0.714	3.07	0.0054		
		Indicated	692,341	22.3	0.252	0.596	2.45	0.0064		
		Inferred	281,281	16.4	0.134	0.521	1.75	0.0059		
		Total/Avg.	**1,367,702**	**21.1**	**0.210**	**0.616**	**2.20**	**0.0053**	**2,117,450**	**1.55**

Totals difference between the NPV5 and inflection point selected (referred to as EML) economic pit limits are summarized in Table 18.15.

Table 18.15 Comparison of the EML (45 year case) – NPV5 LG Economic Pit Limit Resources

	Mineralized Material >Cutoff (kt)	In Situ Grades					Waste (kt)	S/R (t/t)	Copper (M lb)	Au (M oz)
		NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)				
NPV5	1,367,702	21.1	0.210	0.616	2.20	0.0053	2,117,450	1.55	6,346.6	27.1
EML	2,062,665	19.8	0.193	0.588	2.60	0.0057	5,294,872	2.57	8,764.9	39.0
Difference	694,963	-1.31	-0.018	-0.028	0.40	0.0004	3,177,422	1.02	2,418.4	11.9
Variance	51%	-6%	-8%	-4%	18%	8%	150%	66%	38%	44%

Note: This table does not include drilling results from the 2009 exploration program.

Table 18.15 shows that the EML pit has a 51% larger LOM resource than the NPV5 LG pit limit, with 44% higher gold mined and 38% higher copper mined.

Although the discounted NPV for the larger EML economic pit limit is less than the discounted NPV for the NPV5 economic pit limit, starter incremental pit phases will likely be the same for both cases and capital payback should occur in the same time frame.

It is recommended that the focus of this PEA Addendum study be the NPV5 pit.Change between the current PEA Addendum pit and the earlier 2008 PEA economic pit limits is summarized in Table 18.16.

Table 18.16 Comparison of the NPV5 – PEA 2008 LG Economic Pit Limit Resources

	Mineralized Material >Cutoff (kt)	In Situ Grades NSR ($/t)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Waste (kt)	S/R (t/t)	Cu (M lb)	Au (M oz)
2008 PEA	1,412,285	17.4	0.225	0.599	2.10	0.0040	2,497,325	1.77	6,996.1	27.2
2009 NPV5	1,367,702	21.1	0.210	0.616	2.20	0.0053	2,117,450	1.55	6,346.6	27.1
Difference	-44,583	3.72	-0.014	0.017	0.10	0.0013	-379,875	-0.22	-649.5	-0.1
Variance	-3%	21%	-6%	3%	5%	33%	-15%	-13%	-9%	0%

Orthographic and NS section views of the Mitchell scheduled LG phases inside the chosen ultimate pit limit are shown in Figure 18.14 and Figure 18.15, followed by a plan view of the phases and ultimate pits for all three mining areas in Figure 18.16.

Figure 18.14 Mitchell Scheduled LG Phases inside the NPV5 Pit Limit – Orthographic View from the West

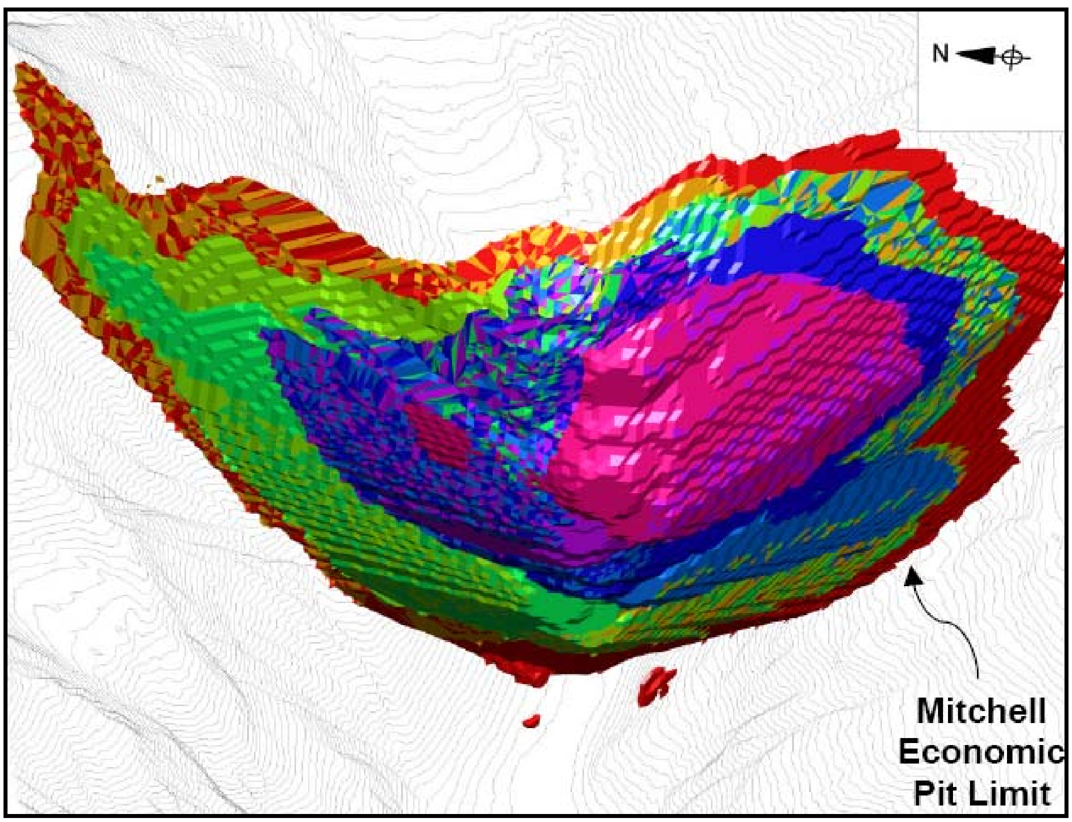


Figure 18.15 Mitchell Scheduled NPV5 LG Phases – NS Section at East 422950

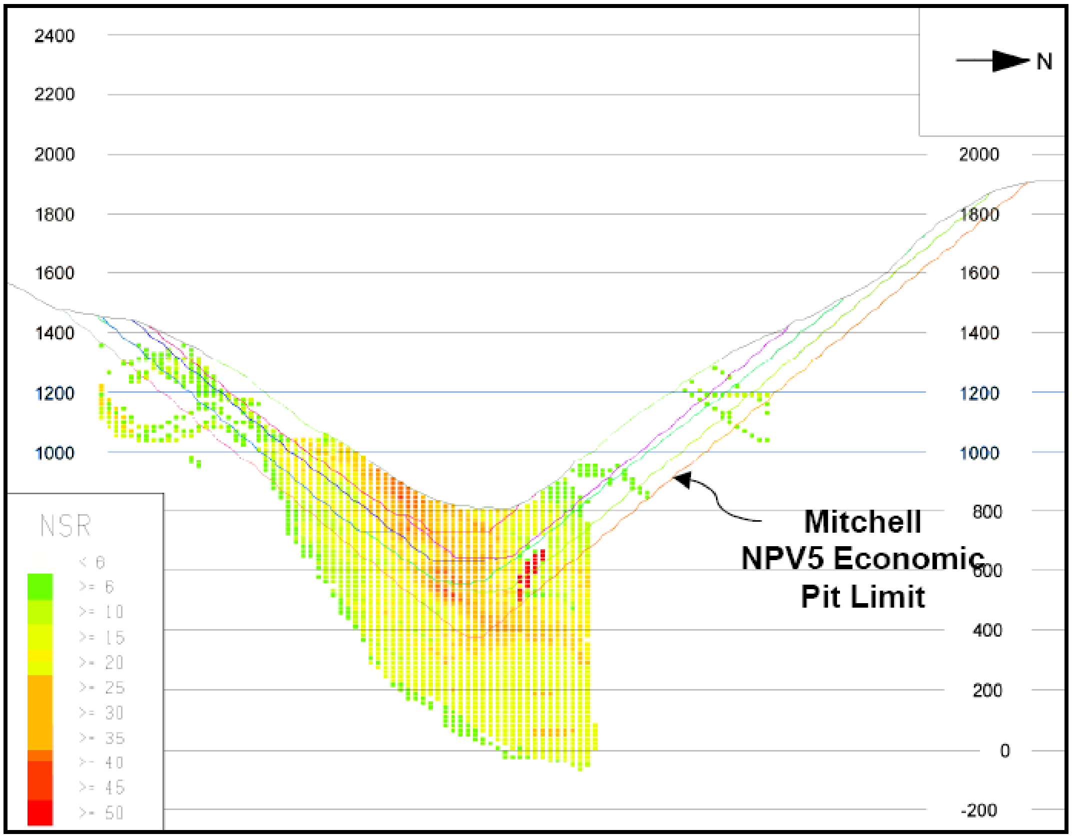
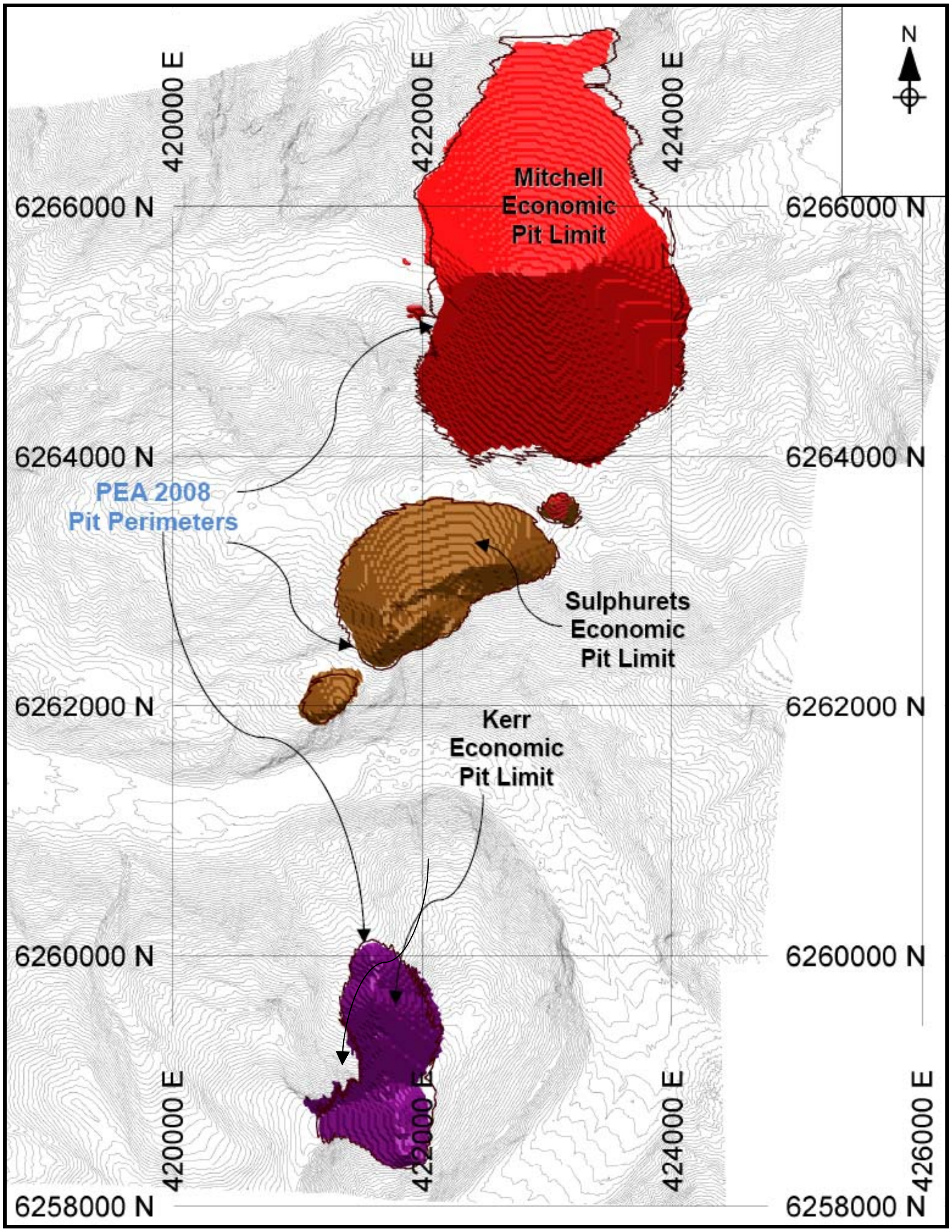


Figure 18.16 NPV5 LG Pit Limits with 2008 PEA Pit Perimeters – Plan View



18.6.4 *PIT SLOPE ANGLE PROJECT RISK*

BGC's conservative sensitivity case recommends a PSA of 40° for the north high wall and the southwest sector of the south high wall. Due to the lack of geotechnical data and lack of precedence of open pits with high wall heights comparable to the indicated Mitchell pit, MMTS has chosen to test the effect of reducing the Mitchell maximum PSA to 37°.

Ongoing evaluations show NPV from pit limits including a 37° PSA Mitchell pit is sufficient to payback a PEA total capital estimate of $4.6 billion.

18.6.5 *Waste Capacity*

Potential waste placement locations have been identified in the 2008 PEA in the following areas:

- Mitchell Valley
- west of the Sulphurets pit
- west of the Kerr pit.

Further investigation is required in this study to identify a waste placement strategy for the updated estimated waste production.

18.7 Detailed Pit Designs

MMTS has completed PEA-level pit designs demonstrating the viability of accessing and mining economical resources at the KSM site. The designs are developed using MineSight® software, estimated geotechnical parameters, suitable road widths for the equipment size, and minimum mining widths based on efficient operation for the size of mining equipment chosen for the project.

18.7.1 *Haul Road Widths*

Haul road widths are designed to provide safe and efficient haulage and to comply with the following BC Mines Regulations for minimum width:

- For dual lane traffic, a travel width of not less than three times the width of the widest haulage vehicle used on the road.
- Where single lane traffic exists, a travel width of not less than two times the width of the widest haulage vehicle used on the road.
- Shoulder barriers are at least 3/4 of the height of the largest tire on any vehicle hauling on the road along the edge of the haulage road wherever a drop-off greater than 3 m exists. The shoulder barriers are designed at 1.5:1 (H:V) side slope. The width of the barrier is excluded from the travel width.

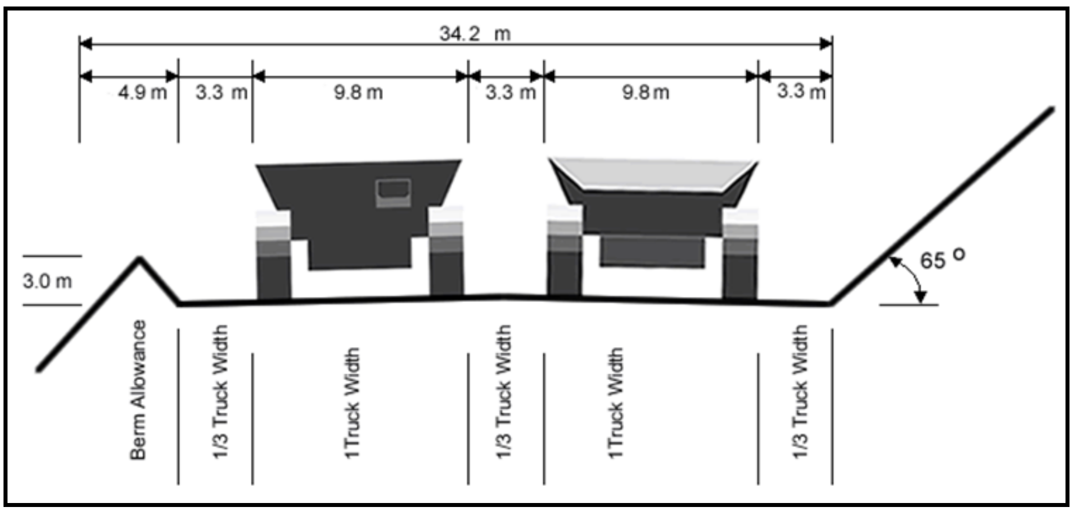
Figure 18.17 through to Figure 18.20 show typical road cross sections for haul roads.

Ditches are included within the travel width allowance. For crowned haul roads, the width of this ditch allowance is 4.5 m. Ditches are not added to the in-pit high wall roads as there is adequate water drainage at the edge of the road between the crowned surface and lateral embankments, such as high walls or lateral impact berms. During run off, when water is flowing, this ditch allowance is still part of the running surface and can be used as lateral clearance for haul trucks and driven on if required to avoid obstructions. In practice, specifically designed excavated ditches in haul roads quickly get filled in by road grading and, when maintained as open ditches, can create a hazard if haul trucks or light vehicles catch a wheel in them. Avoiding the addition of ditch width to the 3-truck travel width on the in-pit high wall roads can significantly reduce the pit waste stripping.

Based on a 345-tonne truck, the haul road design basis is as follows:

- largest vehicle overall width: 9.8 m
- maximum tire height (59/80R63): 4.0 m
- minimum haul road outside berm height: 3.0 m
- berm width: 4.9 m
- ditch width: 4.5 m
- double lane high wall haul road allowance: 34.3 m
- double lane external haul road allowance: 39.2 m
- single lane high wall haul road allowance: 24.5 m
- single lane external haul road allowance: 29.4 m.

Figure 18.17 Dual Lane Highwall Haul Road Cross Section



Note: high wall face slopes will vary by geotechnical design criteria.

Figure18.18 Dual Lane External Haul Road Cross Section

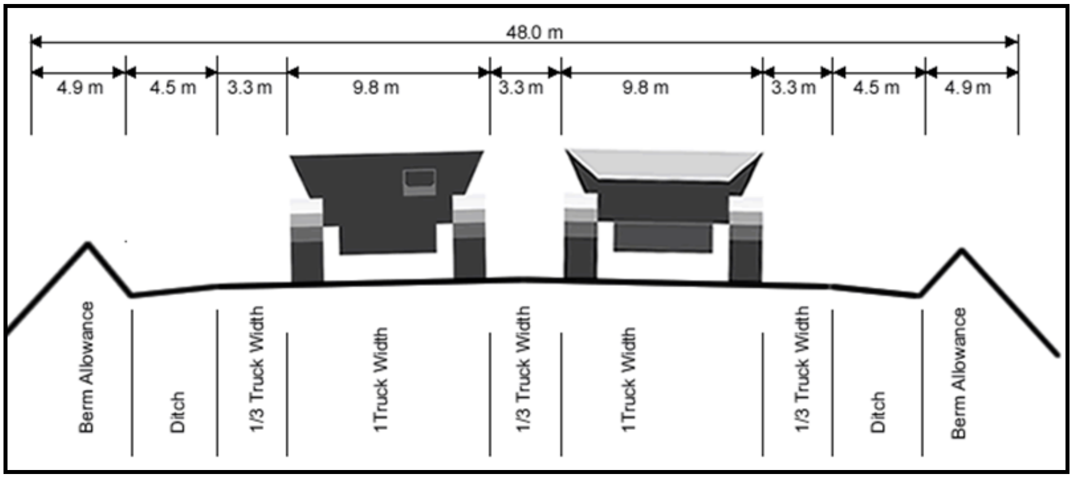
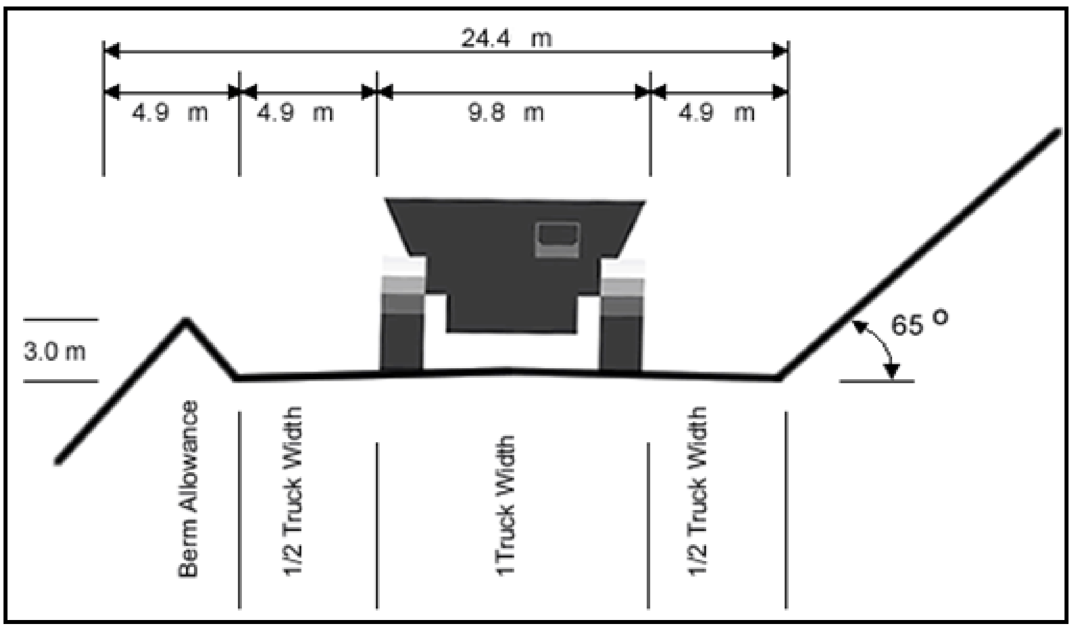
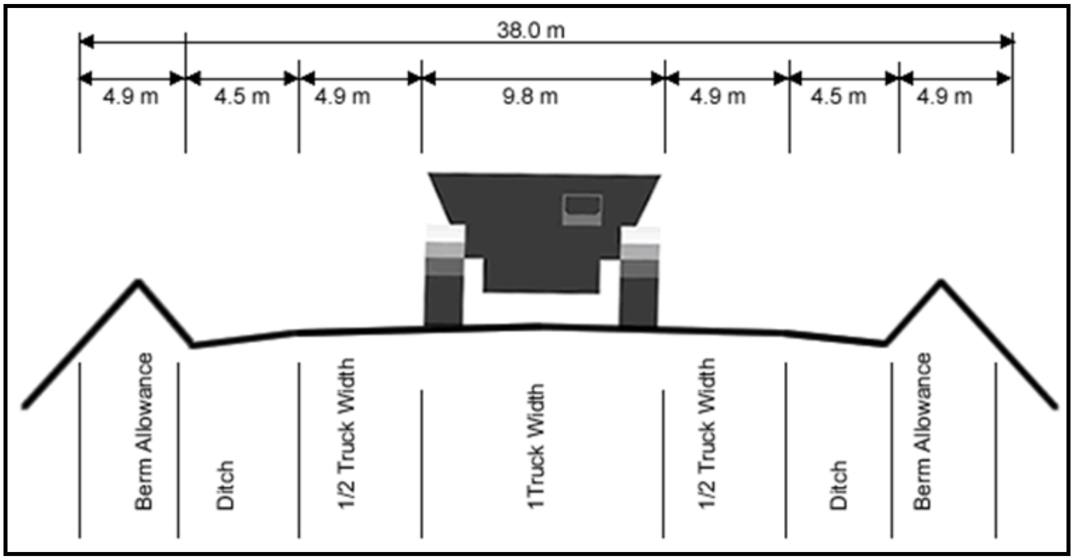


Figure 18.19 Single Lane High Wall Haul Road Cross Section



Note: high wall face slopes will vary by geotechnical design criteria

Figure 18.20 Single Lane External Haul Road Cross Section



18.7.2 DESIGN STANDARDS

MINIMUM MINING WIDTH

The design standards applied in the current pit designs are summarized in Appendix D. A minimum mining width between pit phases is reserved to maintain a suitable mining platform for efficient mining operations. This width is established based on equipment size and operating characteristics. For the KSM PEA, minimum mining width generally conforms to 50 m, which provides sufficient room for 2-sided truck loading but, due to the configuration of merging pits, it is sometimes less.

In areas where minimum shovel mining width is not achieved, such as initial outcrop benches, drill and blast ramps will be cut on original side slopes. Crawler-dozers, shovel casting, or loader tramming will be utilized to move material over the crest to ravel down slope. Truck/shovel excavation of this material will be done as rehandle from lower benches where sufficient bench width has been achieved. This technique has been used at other mountaintop mines and allows for higher efficiencies with large mine equipment, to keep costs down in the capitalization period. The rehandle on the slope helps with the development of the outside edge of lower benches and the impact of the extra cost of the rehandle is time deferred.

ACCESS CONSIDERATIONS

As stated in the design criteria summary, haul road widths are dictated by equipment size. One-way haul roads must have a travel surface more than twice the width of the widest haul vehicle. Two-way roads require a running surface more than three times the width of the widest vehicle planned to use the road. One-way roads are not normally employed for main long term haul routes as they limit the safe by-passing of trucks and consequently lead to reduced productivity, if pull-outs are used. They are, however, an appropriate option for low volume traffic flow or shorter-term operations where the construction of a two-way road is not warranted. In the current study, the use of one-way haul roads is limited to the bottom two or three benches of some pits. An access ramp is not designed for the very last bench of each pit bottom, on the assumption that the ramp is mill feed grade and will be removed upon retreat.

Road grades are designed at a maximum grade of 8%. A decision to design steeper roads can be considered after more weather data has been accumulated. Switchbacks are designed flat, with ramps entering and exiting at design grade. In practice however, grades will be transitioned such that visibility and haul speeds are optimized going around the switchback. Where possible, switchbacks are located such that they tie into future phase access development.

Ramp optimization in deeper pits has not been done for this PEA. With no geotechnical details, ramps in the high walls are assumed as necessary to meet the conservative low overall PSAs. If future geotechnical studies indicated steeper walls are possible, then a skin analysis will be made whether to place ramps outside the LG shell to maximize resource recovery or inside to reduce waste stripping. The grade of the material being lost or gained due to the ramp, and the strip ratio carried to the top of the wall will be considered in future design stages of the project.

In the final pit wall, access up from the lowest pit benches requires a spiral ramp designed to exit at the lowest point on the pit rim or joining with infrastructure features (such as the crusher location or previously designed haul road junctions). In the mountainous terrain at KSM, benches above the lowest point of the pit rims can be accessed by external ramps built on the original hill side slopes, reducing the need for internal ramps in the final wall. Switchbacks and flat grade segments should be minimized. Whether the decline ramp is built inside or outside the LG ultimate pit shell, the amount of mineralized material lost under the ramp, or extra waste mined above the ramp, is minimized if the ramp is not located on the higher strip ratio wall.

In some phases, it may be necessary to leave a high wall ramp in the upper benches of the phase in order to gain access to subsequent pit phases. These intermediate high wall ramps may not be needed when the final pit phase is completed.

VARIABLE BERM WIDTH

Pit designs for KSM are designed honouring overall PSAs, a nominal bench face angle (75°), and variable safety berm widths with a minimum 8 m width. Due to the low overall pit slope angles, berm widths are generally greater than 15 m. Where haul roads intersect designed safety benches, the haul road width is counted towards the safety berm width for the purpose of calculating the maximum overall pit slope angle.

BENCH HEIGHT

The KSM pit designs are based on the digging reach of the large shovels (15 m operating bench) with double benching between high wall berms; therefore, the berms are separated vertically by 30 m. Single benching will be employed, if required, to maximize mineralized material recovery and maintain the safety berm sequence as warranted. The berm width is varied to meet the maximum interslope pit slope angle with a minimum of 8 m.

18.7.3 LG PHASE SELECTION

The LG pits discussed above are used to evaluate alternatives for determining the economic pit limit and the best push-backs or phases on which to begin detailed design work. LG pits provide a geometrical guide to detailed pit designs. Among the details will be the addition of roads and bench access, removal of impractical mining areas with a width less than the minimum, and insuring the pit slopes meet the detailed geotechnical recommendations.

The LG pit cases selected as the economic pit limits for the KSM mine areas discussed above are

- Mitchell economic pit limit: 75% price case LG pit
- Sulphurets economic pit limit: 90% price case LG pit
- Kerr economic pit limit: 85% price case LG pit.

Smaller pit shells exist within the economic pit limits that have higher economic margins due to lower strip ratios or better grades than the full economic pit limit. Mining these pits as phases, from higher margins to lower, maximizes revenue and minimizes mining costs at the start of mining operations thereby shortening the project capital payback and improving the project cash flow. Where a higher number of smaller push-backs increases this effect, it needs to be balanced with the higher efficiencies and resultant lower unit mining costs of big mining areas from bigger push-backs. The first phases (starter pits) have the greatest effect on capital pre-stripping requirements.

The selection of LG pit cases to guide the design of starter pits requires the consideration of some practical mining constraints. The starter pits must:

- be large enough to accommodate the multiple unit mining operations of drilling, blasting, loading, and hauling
- have bench sizes large enough so the number of benches mined per year is reasonable (sinking rate)
- be wide enough so the shovels can load the trucks efficiently.

The pit areas are examined to find the lowest LG price case that can sustain mining operations. A starter phase (at the start of the production schedule) should also be able to supply two years of mill feed.

Waste from the starter pits is pre-stripped to expose mineralized material for plant start-up and can be used for some construction fills (it may be more cost effective to do some borrow for construction from other areas to reduce costs if hauls are too long from the starter pit area). A second cost effective alternative for construction material is to borrow from upper benches of future pit phases.

MITCHELL PITS

Where possible, phase sequencing should start at one side of the ultimate pit and expand in one direction. This is more efficient in operations where blasts from subsequent phases only bury access to lower benches on one side at a time. However, the Mitchell pit phases are designed to alternate from the north and south sides of the Mitchell Valley (a two-sided expansion) for two reasons:

- Initial access is required to both sides of the valley to reach the top of the ultimate pit on the north and south sides.
- The upper benches of the Mitchell pit are mostly waste on both the north and south walls. Breaking into north- and south-side phases enables a smoother waste mining schedule and reduces the maximum truck fleet size.

Each phase maintains sufficient bench width to promote efficient shovel operation.

Ramps are left in the high walls to enable access to the upper benches of subsequent phases and to help achieve the design basis overall PSAs.

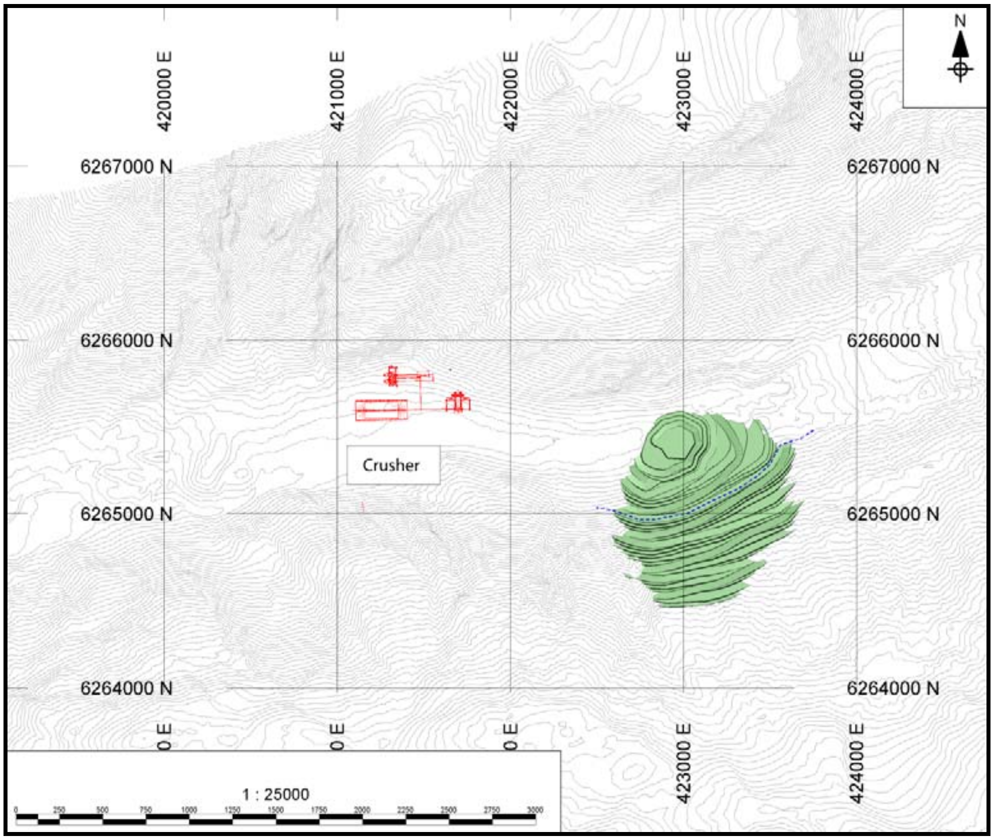
An emergency Mitchell Glacier water bypass ditch is designed along the south side of the pit and is included in all south intermediate phases. Since this bypass will be used in emergency situations only, it is included as part of a mild sloping double-lane haul road that provides access to the high wall and the Mitchell Glacier diversion dam on the east side of the Mitchell pit. In Figure 18.21 through to Figure 18.25, the bypass is indicated by a blue dotted line.

High wall waste is brought out of the pit using external side hill roads directly off the north and south benches.

MITCHELL PHASE M621

Mitchell phase M621 begins on the south side of the valley at a bench elevation of 1365 m and is mined down to a bottom pit elevation of 735 m. The haul road from the pit bottom reaches the surface at an elevation of 793 m. An emergency water diversion ditch is left in the high wall at the 945 m elevation. Waste is hauled from the high wall to external haul roads west of the pit. An illustration of the pit is provided in Figure 18.21.

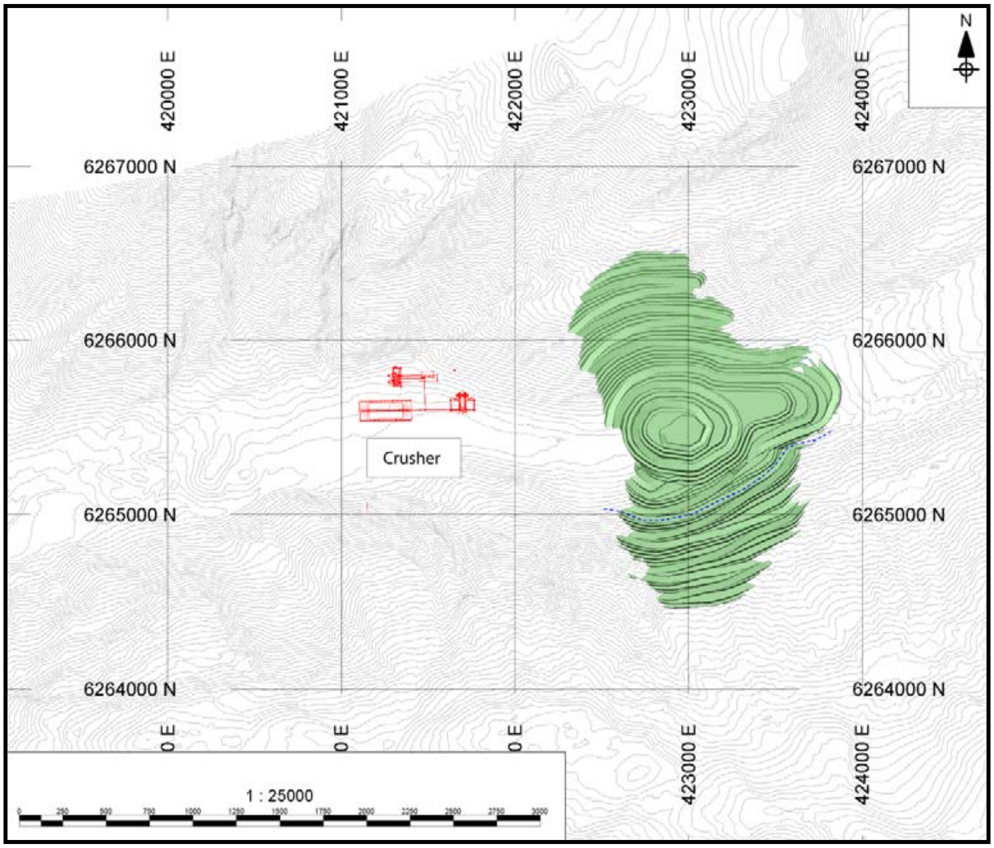
Figure 18.21 Plan View of Mitchell Starter Pit M621



MITCHELL M622I

Mitchell phase M622i begins on the north side of the valley at a bench elevation of 1425 m and is mined down to a bottom pit elevation of 645 m. The haul road from the pit bottom reaches the surface at an elevation of 793 m. M622i can be mined independently down to the 800 m elevation. M622i is incremental to M621 below the 800 m elevation. An illustration of the pit is provided in Figure 18.22.

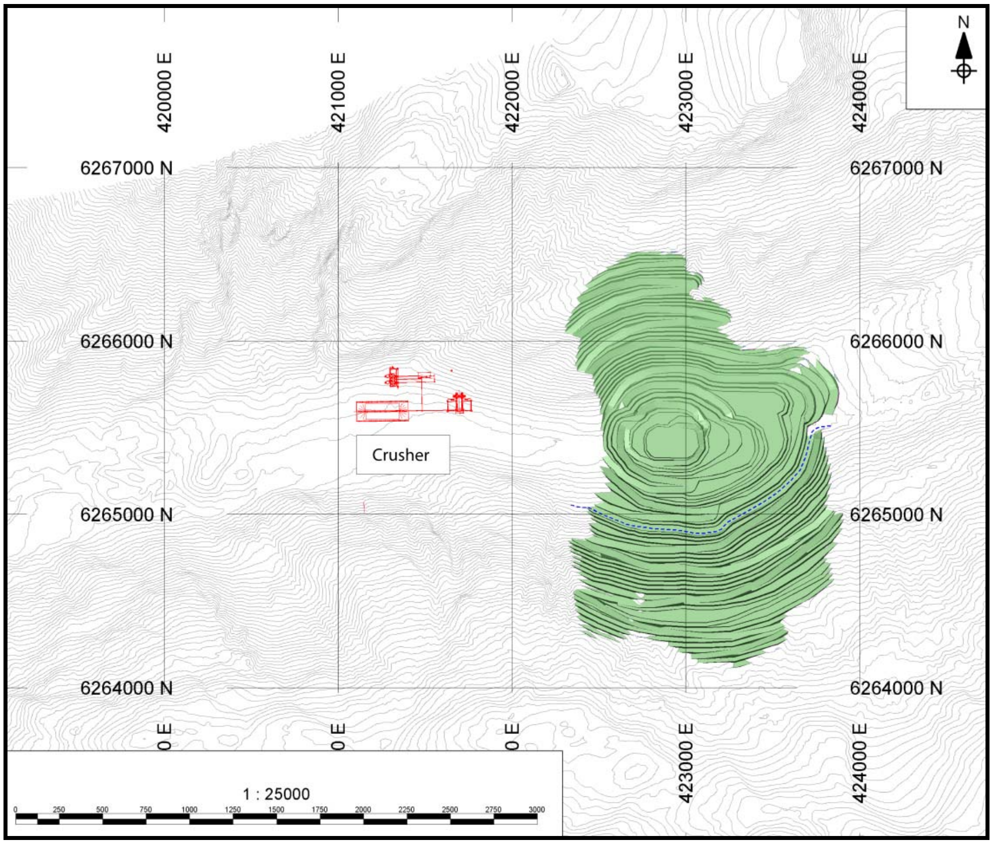
Figure 18.22 Plan View of Mitchell Pit M622i



MITCHELL PHASE M623I

Mitchell phase M623i begins on the south side of the valley at a bench elevation of 1515 m and is mined down to a bottom pit elevation of 630 m. M623i is incremental to M622i. The haul road from the pit bottom reaches the surface at an elevation of 775 m. In this phase, the emergency water diversion ditch is re-established at 915 m as the emergency ditch established in phase M621 is mined out. An illustration of the pit is provided in Figure 18.23.

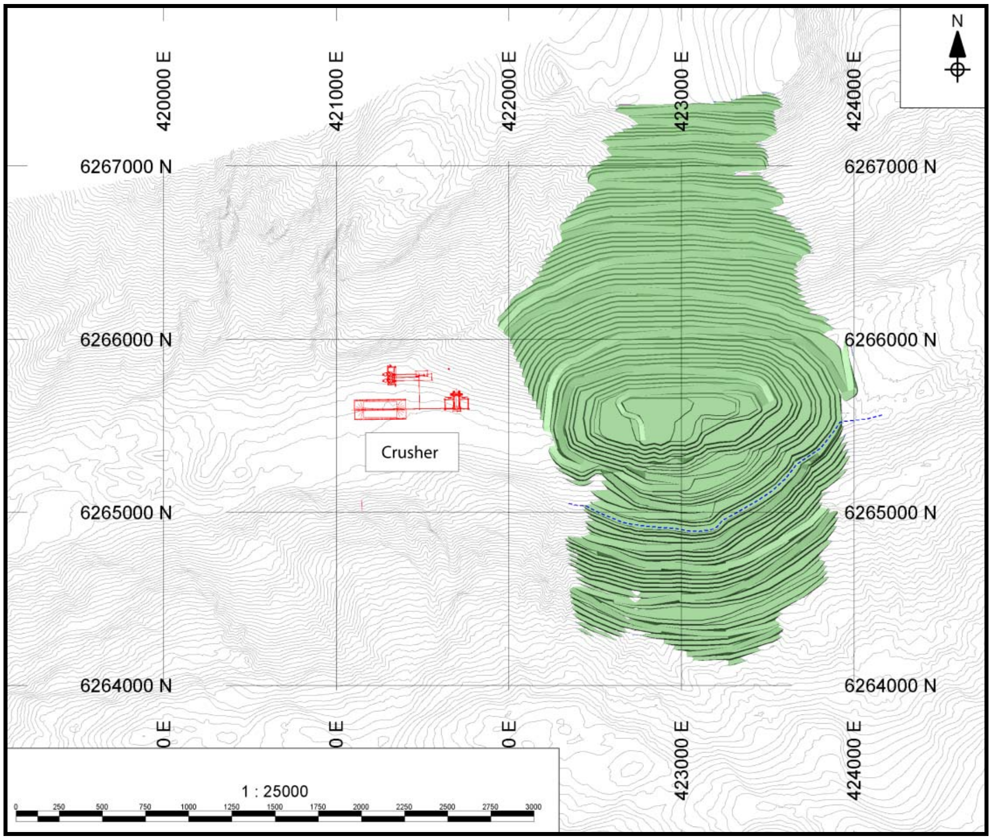
Figure 18.23 Plan View of Mitchell Pit M623i



MITCHELL PHASE M624I

Mitchell phase M624i begins on the north side of the valley at a bench elevation of 2025 m and is mined down to a bottom pit elevation of 525 m. M624i is incremental to M623i. The haul road from the pit bottom reaches the surface at an elevation of 775 m. An illustration of the pit is provided in Figure 18.24.

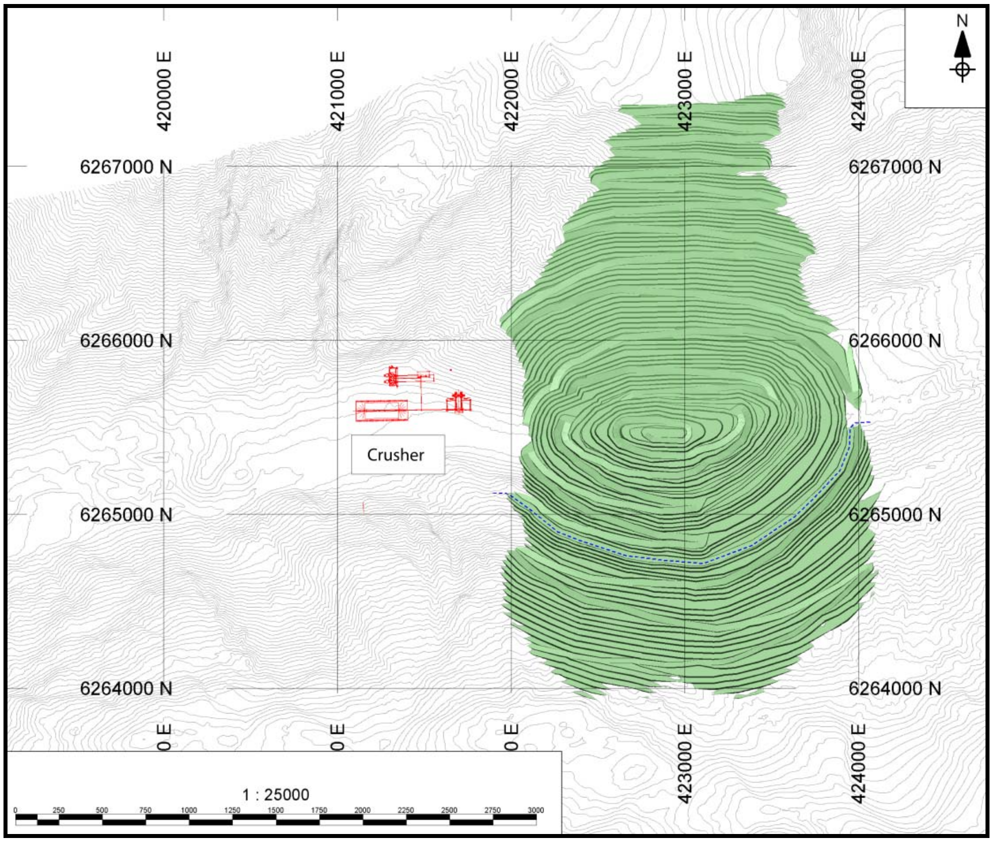
Figure 18.24 Plan View of Mitchell Pit M624i



MITCHELL ULTIMATE PHASE M625I

Mitchell phase M625i begins on the south side of the valley at a bench elevation of 1665 m and is mined down to a bottom pit elevation of 375 m. M625i is incremental to M624i. The haul road from the pit bottom reaches the surface at an elevation of 775 m. In this phase, the emergency water diversion ditch is re-established at 945 m elevation as the previous ditch is mined out. An illustration of the pit is provided in Figure 18.25.

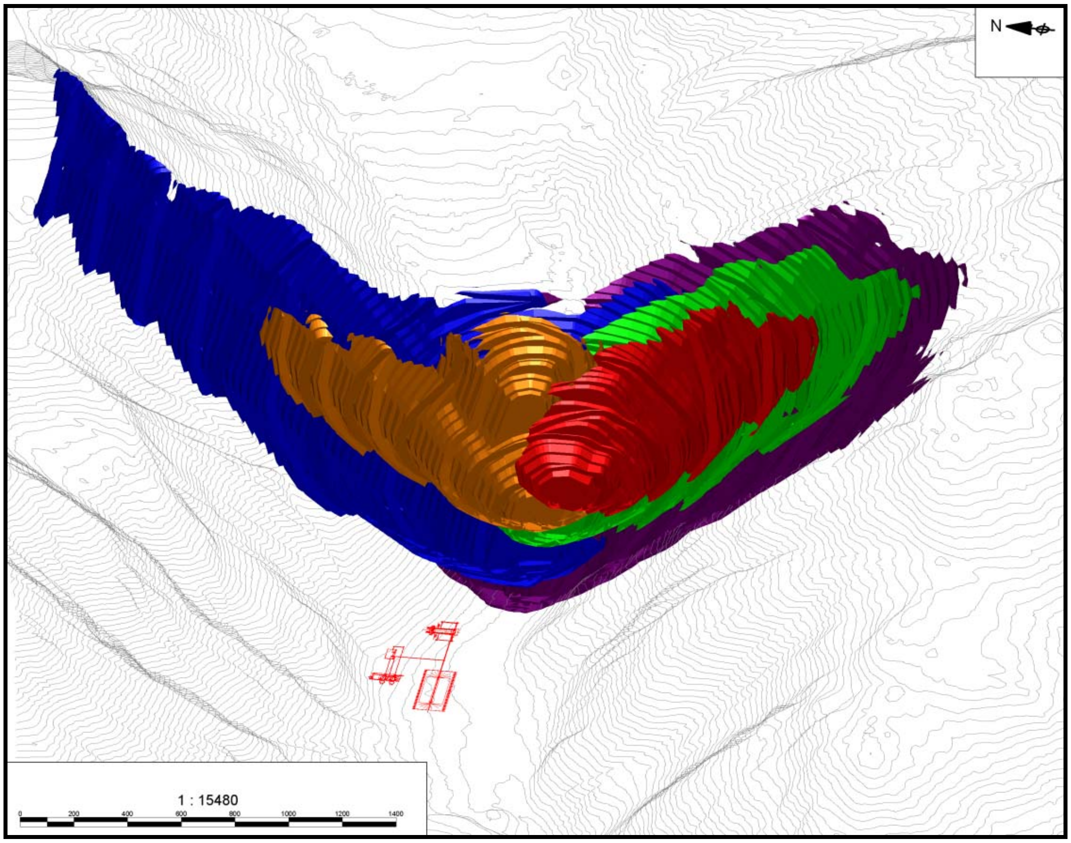
Figure 18.25 Plan View of Mitchell Pit M625i



The future design work (PFS) will attempt to reduce the number of switchbacks in the Mitchell pit high wall using external haul roads.

An orthographic view from the west of all the Mitchell pit phases is shown in Figure 18.26.

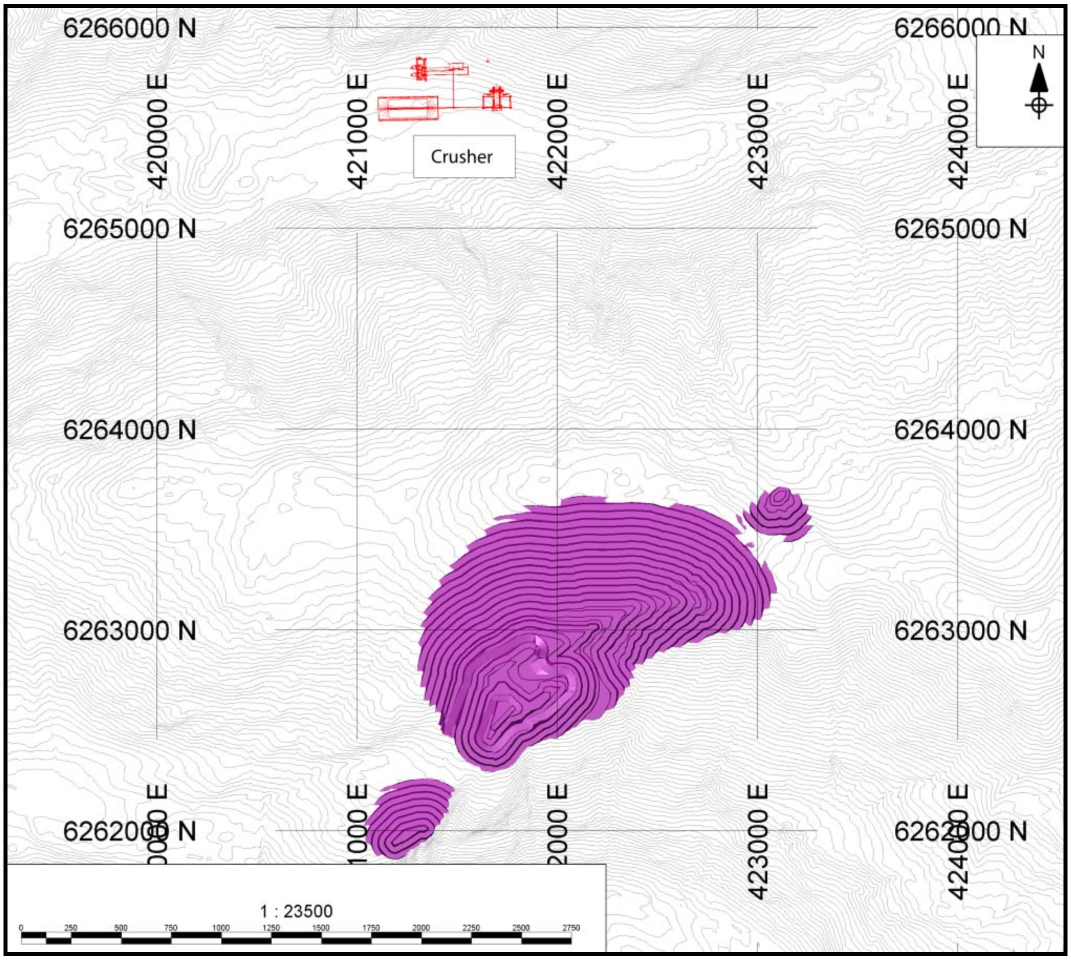
Figure 18.26 Orthographic View of All Mitchell Pits from the West



SULPHURETS PIT

The Sulphurets LG economic pit limit is the only LG pit phase used to guide pit design in the Sulphurets pit area. The Sulphurets ultimate pit is accessed from the west using external haul roads. Waste is hauled to waste dumps west of the pit and in the Mitchell/McTagg Valley. Ore is hauled downhill and westward, and then north into McTagg Valley, and east into the Mitchell Valley where the crusher is located. The crest of the Sulphurets pit is at 1710 m with pit bottoms at 1050 m and 900 m. A plan view of the Sulphurets S612 pit is shown in Figure 18.27.

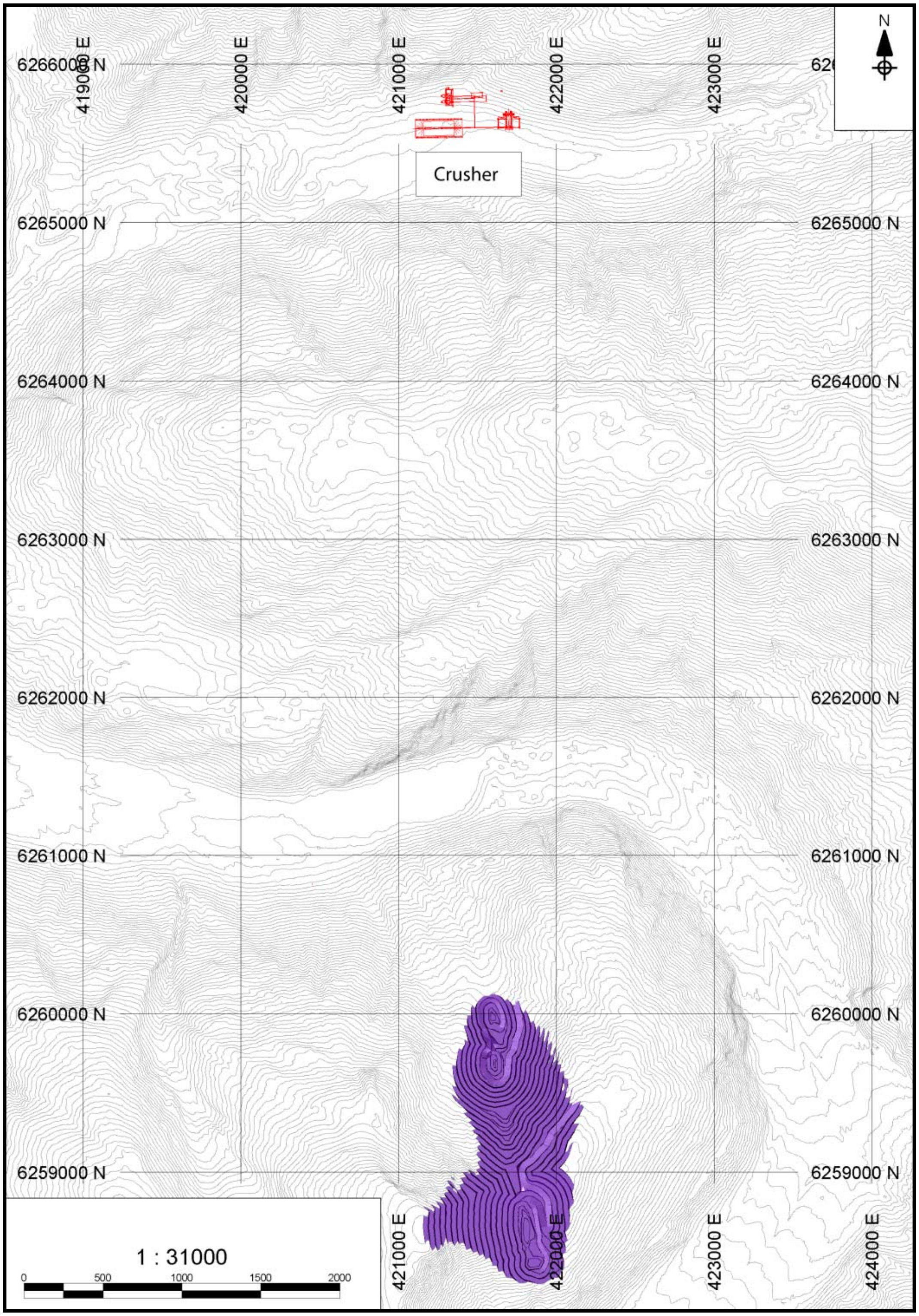
Figure 18.27 Plan View of Sulphurets Ultimate Pit S612



KERR PITS

The Kerr LG economic pit limit is the only LG pit phase used to guide pit design in the Kerr pit area. The crest of the Kerr ultimate pit is at 1911 m with pit a bottom at 1005 m. All ore at Kerr is hauled to the crusher in the Mitchell Valley. All waste at Kerr is hauled to the McTagg and Mitchell valleys. A plan view of the Kerr K612 pit is shown in Figure 18.28.

Figure 18.28 Plan View of Kerr Ultimate Pit K612



COMBINED PIT AREAS

All the ultimate pit phases are shown in Figure 18.29 and Figure 18.30.

Figure 18.29 Plan View of all Ultimate Pit Phases

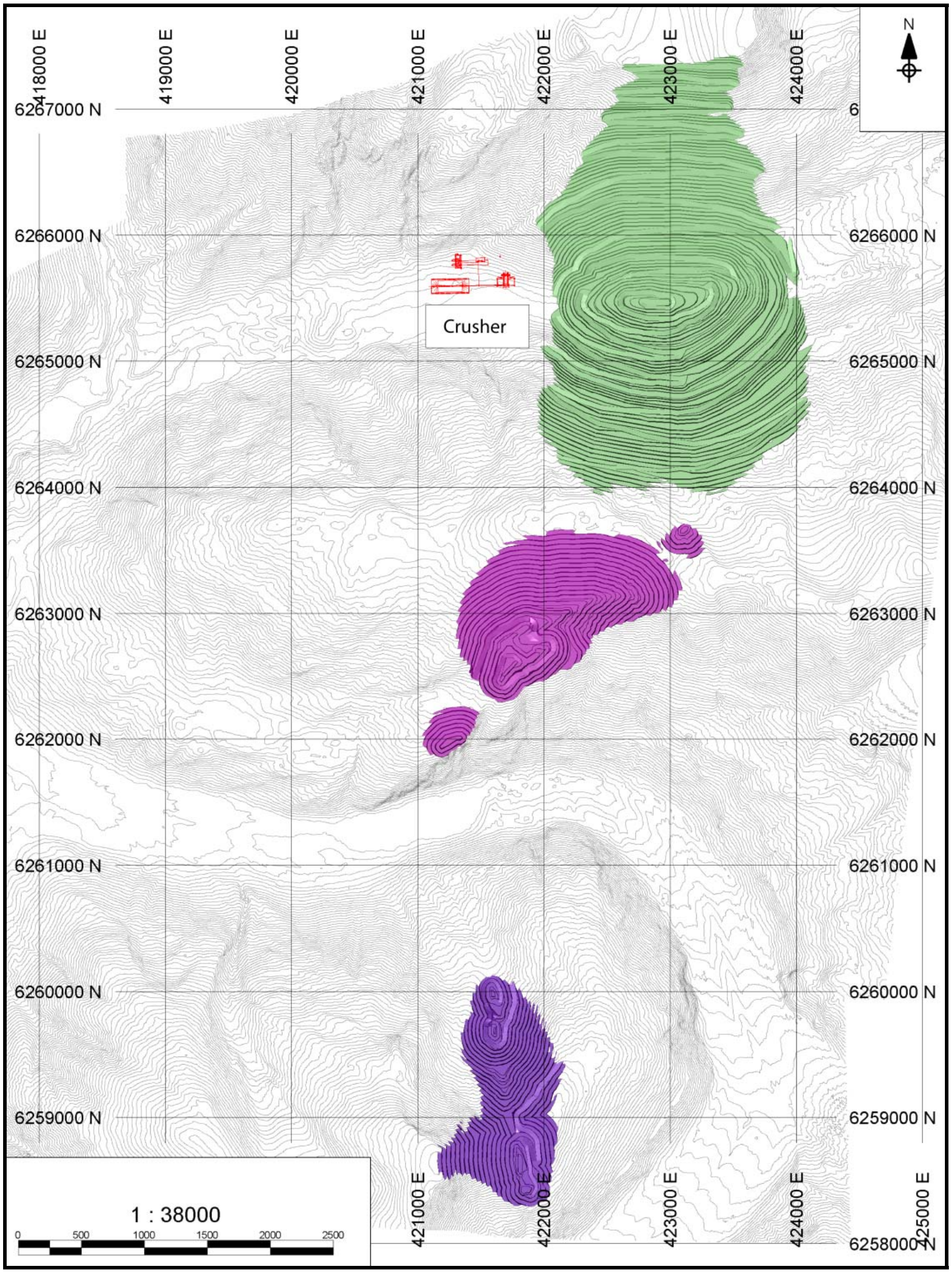
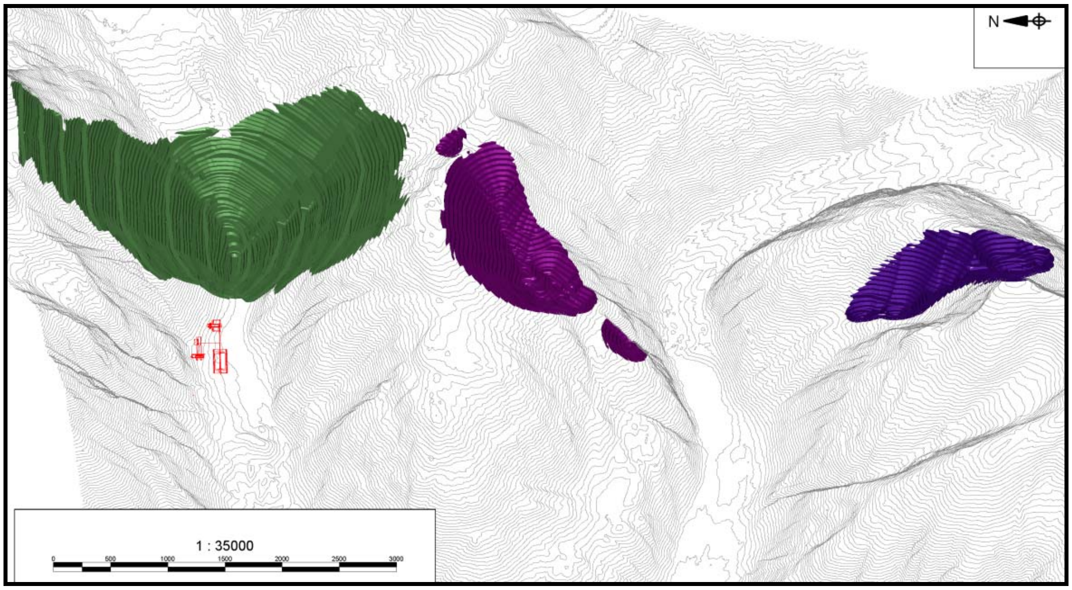


Figure 18.30 Orthographic View from the West of all Designed Pit Phases



18.7.4 PIT RESOURCES

Table 18.18 lists the waste and pit delineated resource for the material within the ultimate pit limits and for each incremental pit phase. Pit delineated resources are estimated using the MineSight® PITRES routine with the following parameters:

- Whole block grades with 5% mining dilution and 5% mining loss. (Dilution grades estimated in Table 18.17 represent the average grade of material below the incremental COG for each pit area.)
- Waste/mineralized material COGs based on NSR is Cdn$6.85/t.

Table 18.17 Dilution Grades

	Mitchell Pit Area	Kerr Pit Area	Sulphurets Pit Area
Cu (%)	0.094	0.130	0.067
Au (g/t)	0.153	0.141	0.194
Ag (g/t)	1.78	-	0.34
Mo (ppm)	31.3	-	45.4
NSR (Cdn$/t)	5.50	5.48	5.52

Table 18.18 Summarized Measured, Indicated, and Inferred Pit Delineated Resource for KSM

Pit	Mineralized Material (kt)	Diluted Grades					Waste (kt)	Strip Ratio (t:t)
		NSR (Cdn$/t)	CUIDW (%)	AUIDW (g/t)	AGIDW (g/t)	MOIDW (ppm)		
Mitchell								
M621	110,690	26.4	0.215	0.815	2.98	34.6	49,840	0.45
M622i	114,582	20.4	0.151	0.660	2.92	49.3	147,538	1.29
M623i	182,816	20.9	0.165	0.658	2.65	61.7	151,514	0.83
M624i	263,954	19.8	0.156	0.617	2.87	59.0	717,673	2.72
M625i	355,390	18.8	0.159	0.572	2.82	56.8	424,005	1.19
Sub-total	**1,027,432**	**20.4**	**0.164**	**0.634**	**2.83**	**55.0**	**1,490,570**	**1.45**
Kerr								
K612	166,054	22.729	0.465	0.2506	0	0	157,908	0.95
Sulphurets								
S612	174,144	23.402	0.2152	0.6562	0.28	75.7	489,496	2.81
Total	**1,367,630**	**21.1**	**0.207**	**0.591**	**2.162**	**51.0**	**2,137,974**	**1.6**

The cost of processing Sulphurets mineralized material is expected to be $7.50/t, which is $0.65/t higher than the cost of processing Mitchell and Sulphurets mineralized material. Waste/mineralized material COGs for Sulphurets should be increased to reflect the increased cost of processing Sulphurets material in future studies. The quantity of ROM mineralized material inside the Sulphurets pit in the $6.85/t to $7.50/t grade bin is 4 Mt which is 1% of the Sulphurets resource.

18.8 MINE PLAN

18.8.1 LOM PRODUCTION SCHEDULE

The mine production schedule after pioneering is developed with MS-SP, a comprehensive long range scheduling tool for open pit mines. It is typically used to produce a LOM schedule that will maximize the NPV of a property, subject to user specified conditions and constraints. Annual production requirements, mine operating considerations, product prices, recoveries, destination capacities, equipment performance, and operating costs are used to determine the optimal production schedule. Scheduling results are presented by period as well as cumulatively and include:

- tonnes and grade mined by period broken down by material type, bench, and mining phase
- truck and shovel requirements by period in number of units and number of operating hours

- tonnes transported by period to different destinations (mill, stockpiles and waste dumps).

The mine schedule considers "Time 0" to be the time that the mill starts; the full capacity production of mill feed is expected in Year 1. The production schedule specifies pioneering as Year -3, and pre-production as Year -2 and Year -1.

Mine Load and Haul Fleet Selection

The mine load and haul fleet is selected prior to production scheduling. Similar projects in the area have shown that the lowest cost per tonne fleet of cable shovels and haul trucks for large hard rock open pit mines that are currently being used are the 100-t bucket class shovel matched with the 345-t class truck. Suitable drills to match this size of truck/shovel fleet are indicated in this section. The performance and costs of 100-t cable shovels and 85-t diesel hydraulic shovels, matched with 345-t haul trucks, 311 mm electric drills and 311 mm diesel hydraulic drills, are used in the following work. Productivities of the selected equipment have been benchmarked with industry experience.

Schedule Criteria

The KSM schedule setup includes truck efficiencies based on the equipment operating efficiency from the design basis (Appendix D). Truck availability assumptions for MS-SP are listed in Table 18.19. Shovel availability assumptions for MS-SP are listed in Table 18.20.

Table 18.19 MS-SP Truck Fleet Availability Assumptions

Up to Hours	% Availability
7,000	87.5
14,000	86.3
21,000	83.4
28,000	82.5
42,000	80.5
4E+07	79.5

Table 18.20 MS-SP Electric Shovel Availability Assumptions

Up to Hours	% Availability
7,000	85.4
14,000	83.5
21,000	82.5
28,000	81.5
35,000	77.6
4E+07	81.5

Load times for the shovels include operator efficiency. Details for the load times are included in Appendix D.

At this time, only mill feed, material types are defined. Waste types will be added in future studies. In order to optimize the project, NPV grade bins have been specified (based on NSR block values). Typical of bulk mining in this kind of deposit, it is assumed that blast hole assays will be used for mill feed COGs and for a COG strategy. The material types specified in Table 18.21 are used for selectivity within the MS-SP optimized scheduler. Mining operations will not use this many grade bins in actual operations.

Table 18.21 Material Types Defined for MS-SP

	NSR Grade Bins (Cdn$/t)
COG (covers milling and G&A)	6.85
Subgrade (covers milling plus S/P R/H + 20% rec loss)	8.82
Low Grade	9.65*
Mid-grade	11.00*
High Grade 1	12.00*
High Grade 2	14.00*
High Grade 3	16.00*
High Grade 4	>16.00

* bins for COG optimization.

Mining precedence is required to specify the mining order of the pit phases based on relative location of the phases. For example, if the phases represent progressive expansions in a single direction, then the first expansion must stay ahead (vertically below) of the next expansion and so on. Even though some of the Mitchell phases alternate from the south to north sides of the valley, the KSM precedences are simplified as shown in Table 18.22.

Table 18.22 Pit Precedence for Scheduling

Phase A ID	Constraint	Phase B ID
M622i	After	M621
M623i	After	M622i
M624i	After	M623i
M625i	After	M624i

In addition to pit precedence, MS-SP also tracks the haul cycle time and resultant variable unit cost from each pit and bench to the primary crusher, stockpiles, or designated waste dumps, in order to determine appropriate costs for optimization.

The primary program objective in each period is to maximize the NPV. The MS-SP NPV calculation is guided by the following inputs:

- 5% discount rate.
- Cdn$0.50/t for ore fixed mining costs.
- Cdn $0.80/t for waste fixed mining costs
- Cdn $6.85/t for processing and G&A costs
- net smelter metal prices of Cdn $2/lb copper, Cdn $25/g gold, Cdn $0.38/g silver, $10/lb molybdenum
- variable mining costs for loading and hauling (calculated from the hourly operating cost and unit productivity for each haul route used).

There are 360 mine operating days scheduled per year and 21 h/d (see the design basis in Appendix D). A default cycle time of 20 minutes is assumed, if required. The default cycle time is only used if a valid calculated cycle time is not available. Annual mill feed of 43,200 kt/a is targeted based on 120,000 t/d of mineralized material milling.

Haul and return times are estimated using computerized simulations. Haul productivity calculations use the following criteria:

- For all benches in all pits, the haul and return times are linearly interpolated based on the haul and return times calculated in Appendix D.
- Appendix D shows haul and return for all pits from the bottom of the pit, a mid-point in the pit, the pit rim, and a point in the upper benches of the pit.
- The haul and return times are de-rated by a 90% operator efficiency and an 83% operating and utilization efficiency.
- A dump and manoeuvre time of 1.5 minutes is also used.

The de-rated haul, return, dump, and manoeuvre times are added and used as the cycle time in MS-SP. The linear interpolation of truck cycle times is carried out for all phases from all benches to all estimated destinations. Resulting average haul truck productivity over LOM is 691 t per operating hour for mineralized material, and 688 t per operating hour for waste.

Shovel productivity includes:

- a 30-second cycle time per pass, and a 55-second spot and wait time per load for the diesel hydraulic shovel
- a 28-second cycle time per pass and a 55-second spot and wait time per load for the electric shovel
- 84% job efficiency

- 83% operating efficiency
- 2% de-rate for high vertical advance rates

The resulting average diesel hydraulic shovel productivity over the LOM is 5,319 t per operating hour for mineralized material and 4,893 t per operating hour for waste. The resulting average electric shovel productivity over the LOM is 5,982 t per operating hour for mineralized material and 5,142 t per operating hour for waste.

COG Optimization

Typically the mill feed grade can be increased by sending low and mid-grade classes to stockpiles. The mill feed rate is maximized and this effectively increases the revenue per tonne milled. However, stockpiling also results in increased total mined rock, and the mine cost per tonne milled in the relevant time period also increases. At some point, the cost of mining more material will exceed the incremental revenue from the higher grade milled.

In this study, subgrade is material that has sufficient revenue (NSP) to pay for processing only. The fixed overhead costs and mining costs are considered sunk, in pursuit of deeper more valuable material. Although subgrade incrementally contributes to positive cash flow, it is not included at this level of study since it is assumed that it is preferred to delay it from milling until late in the schedule and it therefore contributes little to the project's economic returns.

To test project payback and IRR sensitivity to the variable mill feed COG, a set of long range production schedules are assessed with MS-SP. The best case is a variable ROM mill feed COG shown in Table 18.23 where the relative NPV peaks.

Table 18.23 Schedule Mill Feed COG Grades

Mill Feed Year	ROM Mill Feed COG (Cdn $/t)
Year 1 to Year 2	14.00
Year 3 to Year 28	8.82
Year 29 to Year 30	6.85

Any incremental increase in COG reduces the LOM IRR. At this stage of study, there is little value to add by increasing the COG beyond the above case. Table 18.23 is the basis for the 2009 PEA mine production schedule.

Schedule Results

The summarized production schedule results are shown in Table 18.24. Full results are in Appendix D.

Table 18.24 Summarized Production Schedule

	Unit	Year										LOM
		-2	-1	1	2	3	4	5	6 to 10	11 to 20	21 to 30	
Pit to Mill												
Mineralized Material	kt		-	43,201	43,200	39,452	43,200	43,200	208,733	417,803	411,715	1,250,504
AUIDW	g/t		-	0.826	0.857	0.714	0.628	0.795	0.654	0.590	0.552	0.618
CUIDW	%		-	0.214	0.211	0.169	0.129	0.221	0.150	0.167	0.322	0.219
AGIDW	g/t		-	2.34	2.90	3.14	2.19	4.19	2.61	2.68	1.19	2.22
MOIDW	ppm		-	40.9	41.1	43.0	61.0	25.2	68.7	61.4	39.4	52.1
Pit to Stockpile												
Mineralized Material	kt	1,675	1,675	2,847	4,063	1,095	2,123	176	7,843	13,306	10,360	45,165
AUIDW	g/t	0.296	0.296	0.377	0.481	0.370	0.345	0.326	0.336	0.348	0.292	0.343
CUIDW	%	0.187	0.187	0.128	0.064	0.032	0.049	0.073	0.091	0.054	0.120	0.090
AGIDW	g/t	5.69	5.69	2.15	2.03	2.12	1.92	1.65	2.17	1.71	0.12	1.80
MOIDW	ppm	33.6	33.6	34.6	54.0	60.9	69.2	37.0	51.0	55.8	18.1	43.8
Pit to Subgrade (Wasted)												
Mineralized Material	kt	643	643	2,467	1,613	5,438	1,524	209	14,014	24,380	21,338	72,268
AUIDW	g/t	0.148	0.148	0.365	0.313	0.269	0.324	0.262	0.288	0.299	0.228	0.274
CUIDW	%	0.152	0.152	0.122	0.037	0.065	0.036	0.093	0.110	0.052	0.127	0.090
AGIDW	g/t	1.49	1.49	2.04	1.87	2.07	1.65	2.57	1.90	1.75	0.09	1.32
MOIDW	ppm	45.9	45.9	33.7	40.9	37.5	51.6	29.7	31.0	49.7	12.1	33.2

table continues…

	Unit	Year										LOM
		-2	-1	1	2	3	4	5	6 to 10	11 to 20	21 to 30	
Stockpile Reclaim												
Mineralized Material	kt		-	-	-	3,748	-	-	7,267	14,197	17,285	42,497
AUIDW	g/t		-	-	-	0.392	-	-	0.370	0.352	0.318	0.345
CUIDW	%		-	-	-	0.122	-	-	0.016	0.066	0.089	0.072
AGIDW	g/t		-	-	-	3.26	-	-	0.46	2.33	0.87	1.50
MOIDW	ppm		-	-	-	42.0	-	-	8.6	53.8	36.2	37.9
Stockpile Size	kt	1,675	3,350	6,198	10,261	7,608	9,732	9,908	10,484	9,594	2,669	2,669
Total Pit Mineralized Material Mined												
Mineralized Material	kt	2,318	2,318	48,515	48,876	45,985	46,847	43,585	230,590	455,490	443,413	1,367,938
AUIDW	g/t	0.255	0.255	0.776	0.807	0.653	0.606	0.790	0.621	0.567	0.531	0.591
CUIDW	%	0.177	0.177	0.204	0.193	0.153	0.122	0.220	0.145	0.158	0.308	0.208
AGIDW	g/t	4.53	4.53	2.32	2.79	2.99	2.16	4.18	2.58	2.60	1.11	2.17
MOIDW	ppm	37.0	37.0	40.1	42.1	42.8	61.1	25.2	66.0	60.6	37.5	50.9
Plant Feed												
Mineralized Material	kt	-	-	43,201	43,200	43,200	43,200	43,200	216,000	432,000	429,000	1,293,001
AUIDW	g/t	-	-	0.826	0.857	0.686	0.628	0.795	0.644	0.582	0.543	0.609
CUIDW	%	-	-	0.214	0.211	0.165	0.129	0.221	0.146	0.164	0.313	0.214
AGIDW	g/t	-	-	2.34	2.90	3.15	2.19	4.19	2.58	2.67	1.18	2.21
MOIDW	ppm	-	-	40.9	41.1	42.9	61.0	25.2	69.0	61.1	39.2	52.0
Metal to the Mill												
AUIDW	M oz	-	-	1.15	1.19	0.95	0.87	1.10	4.47	8.08	7.49	25.31
CUIDW	M lb	-	-	204	201	157	123	210	704	1,559	2,960	6,119
AGIDW	M oz	-	-	3.25	4.03	4.37	3.05	5.82	18.11	37.02	16.24	91.90
MOIDW	M lb	-	-	3.89	3.91	4.09	5.81	2.40	32.42	58.20	37.10	147.82

table continues…

	Unit	Year										LOM
		-2	-1	1	2	3	4	5	6 to 10	11 to 20	21 to 30	
Average Process Recoveries												
Au	%			78.1	78.3	77.2	77.1	78.1	77.2	76.4	74.2	76.2
Cu	%			83.6	83.8	82.8	77.3	84.8	79.6	81.3	90.8	86.0
Ag	%			72.9	72.9	72.9	73.0	73.0	73.0	73.0	73.0	73.0
Mo	%			26.8	33.2	34.9	39.0	12.4	42.3	39.8	45.3	40.6
Waste Mined												
Total Waste Mined	kt	44,999	44,999	80,000	79,998	91,875	94,999	99,998	539,990	641,832	419,284	2,137,974
Strip Ratio (waste mined/mineralized material mined)	t/t	19.4	19.4	1.6	1.6	2.0	2.0	2.3	2.3	1.4	0.9	1.6
Strip Ratio (waste mined/plant feed)	t/t	-	-	1.9	1.9	2.1	2.2	2.3	2.5	1.5	1.0	1.7
Total Material *Mined*		**47,318**	**47,318**	**128,515**	**128,874**	**137,860**	**141,846**	**143,583**	**770,579**	**1,097,321**	**862,697**	**3,505,911**
Total Material *Moved*		**47,318**	**47,318**	**128,515**	**128,874**	**141,608**	**141,846**	**143,583**	**777,846**	**1,111,518**	**879,982**	**3,548,408**

A schedule of mineralized material and waste mined from each phase is illustrated in Figure 18.31 and Figure 18.32. The different colours show the different pit phases as indicated in the legend.

Figure 18.31 Schedule of Mineralized Material Mined by Phase

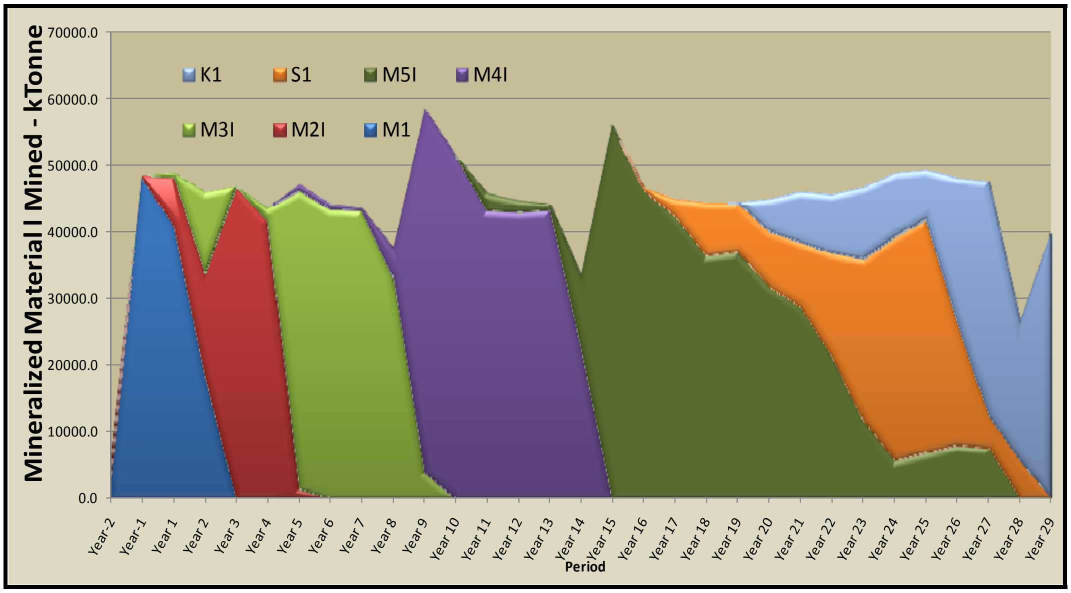


Figure 18.32 Schedule of Waste Mined by Phase

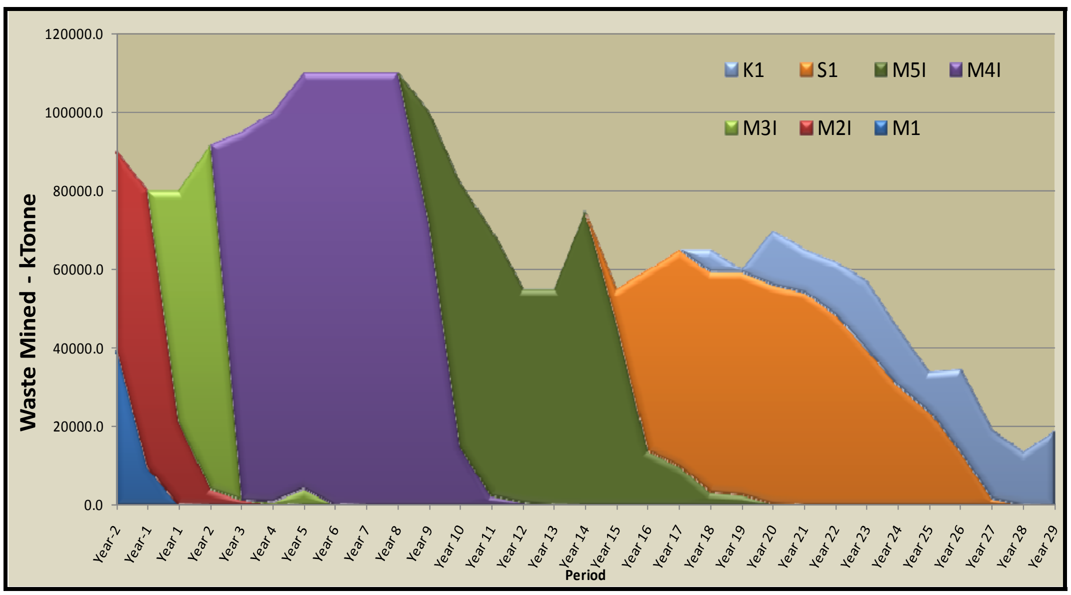
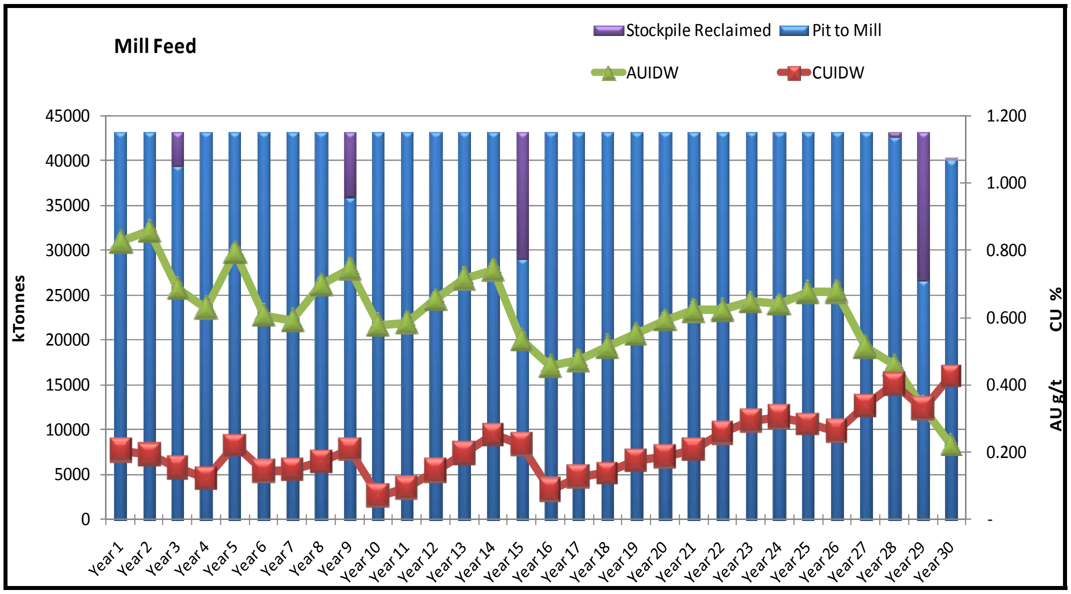


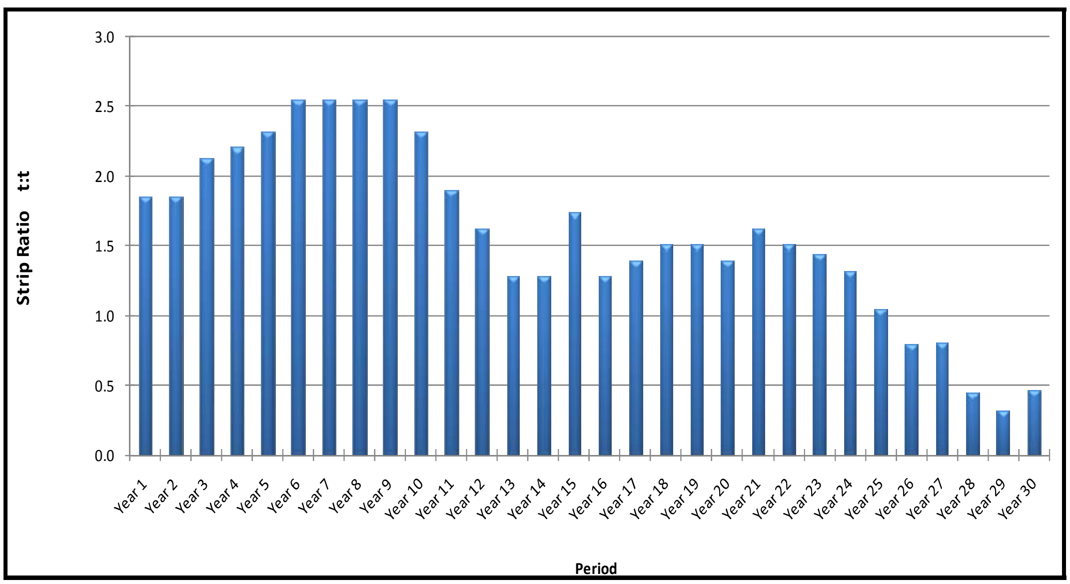
Figure 18.33 illustrates that significant stockpile reclaim is required in Years 3, 9, and 15 to even out the strip ratio during pre-stripping of the subsequent Mitchell phases.

Figure 18.33 ROM Mineralized Material Source and Mill Feed Cu Grade



The strip ratio by period is shown in Figure 18.34. In periods where long hauls are required, the scheduled strip ratio is decreased. In periods where short hauls are required, the scheduled strip ratio is increased. This is done to smooth out the truck fleet size over the LOM. In more detailed stages of engineering, more smoothing of the schedule will be done to even out the strip ratio, which will better utilize the shovels than the approach taken in this study.

Figure 18.34 Strip Ratio (Waste Mined/Plant Feed)



18.8.2 WASTE ROCK STORAGE

DESIGN PARAMETERS

MS-SP is capable of reporting tonnages of material by waste type to detail and cost a waste rock management plan. However, at this stage of study, waste rock characterization is still to be developed in the 3D block model and only one type of waste material is scheduled with MS-SP at this time. Allowance for general waste types has been included in the waste placement in the dumps.

At this stage of planning, all dumps are designed with a natural angle of repose of 37°. A 30% swell factor is then applied to these in situ volumes to calculate the loose dumped volumes that need to be placed. When the waste rock classification is better defined, more detailed dump designs can be done to segregate different materials, step back dump lifts to reduce the overall final overall dump face angle, and facilitate final reclamation.

PAG WASTE ROCK

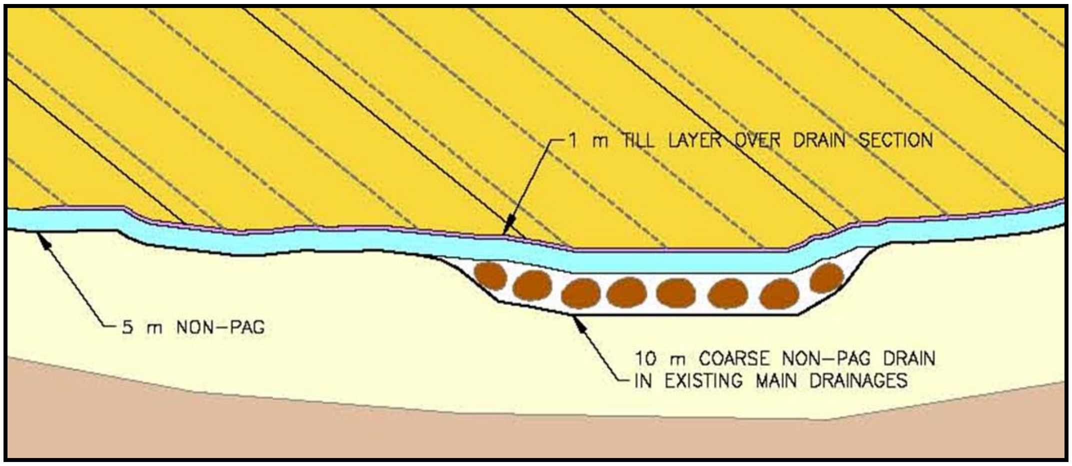
Potentially acid generating (PAG) waste rock requires special handling to minimize the environmental impact from acid rock drainage (ARD) and metal leaching. Spatial distribution of the PAG rock has not yet been modelled and results from metal leaching test work are not yet available.

The following general design parameters are from KCBL guidelines:

- Non-potentially acid generating (NPAG) rock drains and waste dump base:
  - 5 m NPAG base allows groundwater to flow under the dump without becoming contaminated.
  - 10 m NPAG underdrains collect flows and channel water out of the dump.
  - Some PAG leachate will enter underdrains; some will be collected on the overlying till layer and transported separately to treatment.
  - If additional suitable till is available for basal sealing layers, explore opportunities to separate leachate from groundwater flows under the dump.

The NPAG drain and base is illustrated in Figure 18.35.

Figure 18.35 Waste Dump NPAG Drain and Base



Source: KCBL.

All the PAG waste rock from mining operations is dumped on top of the NPAG base. In the Mitchell Valley, waste rock mined from the north side of the valley is dumped from various elevations using a top down and wrap around placement method. Waste rock arising from the south side of the Mitchell Valley will be placed in smaller lifts in a bottom-up sequence. The dump method concepts are illustrated in Figure 18.36.

At reclamation, after re-sloping the dump surfaces to 23°, a low permeability 1 m till layer is placed on top of the PAG waste material to encapsulate it. A 3 m NPAG rock layer is placed on top of the till to protect it from frost and erosion.

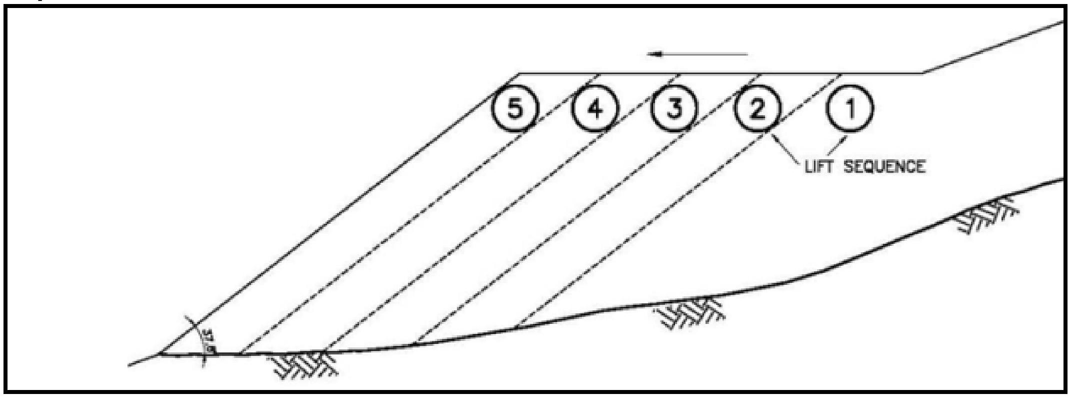
The above design guidelines minimize the ground and surface water contact with PAG rock, and reduce acid generation by minimizing air flow through the dump.

Sources of till and NPAG rock suitable for both the dump base and the underdrains will need to be sourced in future studies.

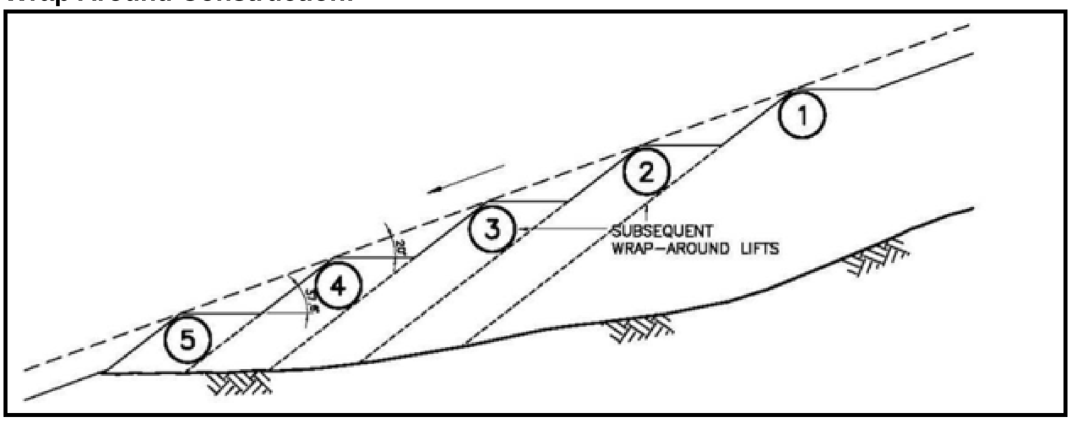
The planned sequencing of waste placement is illustrated in the mine end of period maps in Appendix D.

Figure 18.36 Dump Method Concepts

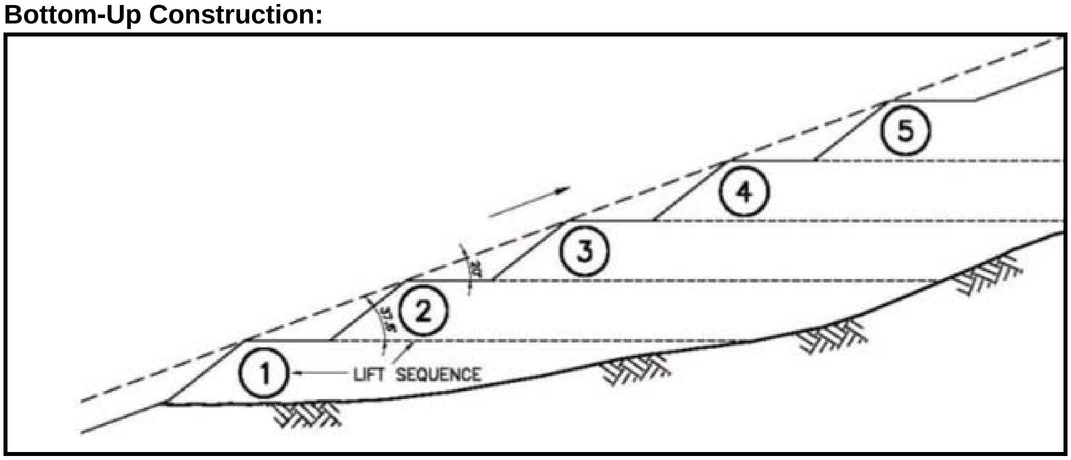
Top Down Construction:



Wrap Around Construction:



Bottom-Up Construction:



Source: KCBL.

WASTE DUMP WATER MANAGEMENT

All water runoff from the PAG waste will be treated during the mine life and post mining as required to neutralize the pH and reduce metal content before discharging to the environment. The location of the water diversion ditches and tunnels are shown in the Mine General Arrangement drawing in Appendix D.

A water treatment dam will be built downstream of the conjunction of the Mitchell and McTagg valleys. This water treatment dam will collect water from mining disturbed areas including the waste dump underdrains and ground water runoff from the surfaces of the dumps, pit areas, and areas that are lower than the diversion ditches.

Prior to mining, fresh water diversion ditches will be built above and around the contact of the final waste dumps and pit areas. This will reduce the amount of water that will come in contact with PAG material.

A diversion tunnel will be built to divert Mitchell glacial runoff away from the pit and dump areas. This diversion tunnel will be in place before waste placement crosses the Mitchell Valley. If delayed, glacier runoff will need to be diverted around the Mitchell Valley dumps with ditches.

The Mitchell Glacier runoff tunnel goes underneath the Sulphurets ridge and discharges into a penstock in the Sulphurets Valley. In the case of a large runoff event that exceeds the capacity of the tunnel, excess water will be routed along an emergency berm on the south side of Mitchell pit and then along the emergency spillway along the south side of the Mitchell Valley.

Other groundwater from the downstream side of Mitchell pit and the upstream side of the Mitchell Valley waste dumps will flow underneath the dump in the NPAG underdrain. This will allow all the water to get to the treatment pond without having to pump it over the dump as it progresses in height. An emergency pump will be put in place just upstream of the waste dump, in case the underdrain blinds off or is crushed from the weight of the waste material above and water flow through the pipe is cut off.

At closure, a dam will be built on the upstream side of the Mitchell Valley dump and the south dump haul road will become a spillway (see the Mine General Arrangement drawing in Appendix D).

Dump Monitoring and Planning

The long term operation of the waste dumps will be similar to the large steep terrain dumps being operated for many years in the Elk Valley mines. This requires some foundation preparation and development of monitoring and operating guidelines to ensure safe and continuous operations. Dumping during the initial stages of mining will be done with low lifts in areas that are non-critical. As experience is gained and stable foundations are established, dumping can proceed with higher lifts as required.

ANNUAL WASTE VOLUMES AND PLACEMENT

Annual waste volumes produced from the 120,000 t/d schedule are shown for selected periods in Table 18.24.

WASTE DUMP ACCESS ROADS

Pioneering access to each pit and subsequent phases use roads with a maximum 15% grade and are constructed with a balanced cut and fill. Pioneering roads are 10 m wide and enable major mining equipment to reach the top of each pit phase to start mining. Waste from the upper portions of each pit phase is hauled down and used to build out full haul roads at a maximum gradient of 8% at the full 38 m double lane width.

Dump access roads on the north side of the Mitchell Valley from phases M622i and M624i are built in cut and fill. Due to the steep topography, the roads above the crusher area are built completely in cut to avoid any fill material rolling down to the valley bottom (Figure 18.37). Three dump access roads are built on the north side of the Mitchell Valley:

- an upper dump access road that ends at the 1605 m elevation
- a mid-dump access road that ends at 1230 m elevation
- a lower dump access road that ends at the 990 m elevation.

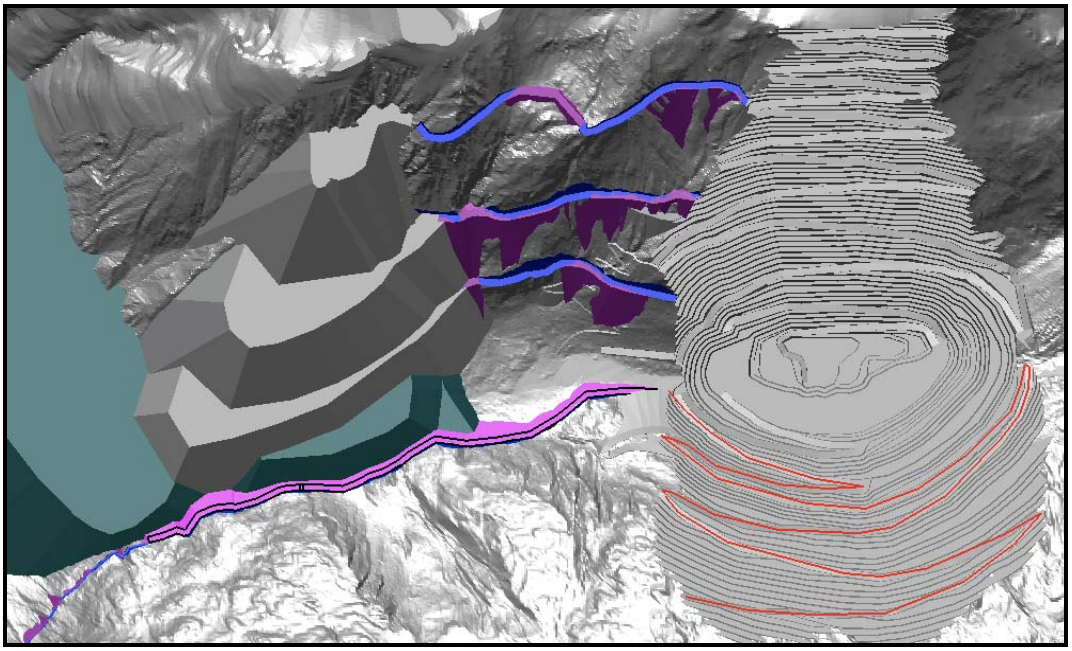
Using these roads, dumps can then be built from waste on the north slope of Mitchell Valley using a top-down dumping procedure with wrap arounds for the lower lifts. Then the dumps can expand westwards and to the south until they toe out on the thin lift bottom-up dumps on the south side of the Mitchell Valley.

On the south side of Mitchell Valley, the waste must be dumped from the bottom-up in thin lifts to provide a compacted base. This compacted base will contribute to consolidating the foundations for dumps from the north side of Mitchell Valley. In order to build the dumps in this manner, all the material from M621, M623i, and M625i of the Mitchell pit upper benches must be hauled downhill to the lower elevation dump platform. Therefore no dump access roads are designed along the south side of Mitchell Valley above the main access road.

The waste will be hauled using the ore haul road down to the crusher and then hauled westward along the main access road into the Mitchell Valley. To build any dump access haul roads higher up on the south side of Mitchell Valley will be difficult and expensive due to the steep topography in many areas. High dump access roads on the south valley slope carry the additional risk of fill material running down and covering up the main access road below.

Figure 18.37 illustrates general waste dump access on the north and south sides of the Mitchell pit. The south side is shown as the red line over which the waste will be hauled down to the valley floor and then hauled westward along the main access road (shown in pink, outlined in black) to the bottom-up designed dumps (shown in green). The waste from the North side of Mitchell pit is hauled along the cut/fill dump access roads (shown in blue and purple) to the three different top down and wrap around designed dumps.

Figure 18.37 General Waste Dump Access for Mitchell Pit



18.8.3 *MINE PRE-PRODUCTION DETAIL*

PRE-PRODUCTION DESCRIPTION

The primary objectives of mine pre-production development are to:

- Expose sufficient mill feed for start-up.
- Establish mining areas that will support the equipment required to achieve mineralized material production that satisfies the annual mill feed requirements on a sustainable basis.
- Act as a source of material required for construction of the mine, mill, and site infrastructure.

Mine pre-production site development activities are currently scheduled to start in Year -2 in order to meet the timeline for overall site development.

Site development for the mine area will consist of the following activities:

- tree clearing and grubbing
- drainage control and water management facilities
- topsoil salvage
- pioneering access
- initial pit development
- haul road construction
- infrastructure construction
- pit power.

Mine Tree Clearing and Grubbing

Much of the mine area is devoid of trees due to the recent retreat of the local glaciers. Clearing and grubbing is required mainly in the lower elevation site works and waste dump areas. An area estimated at 950 ha will need to be cleared and grubbed of trees and brush. This includes the following areas:

- pit area
- waste dumps
- ore stockpile
- mine haul roads
- explosives manufacturing plant and explosives magazine
- truck shop.

Mine Drainage

Prior to mining, a surface water management plan needs to be implemented. This will consist of a network of diversion ditches/settling ponds and collection ditches/ponds.

The primary purpose of the diversion ditch network is to prevent surface water from entering areas where it will become contaminated. Surface water that is diverted through this network must be retained in a settling pond before being released to the environment. These diversion ditches are primarily located around the perimeter of the pit, the waste dumps, and the mineralized material stockpile and need to collect water from haul roads and pit bench runoff.

The primary purpose of the collection ditch network is to collect all water that comes into contact with the mining operation. This water is part of a closed-circuit and must be conducted to the treatment pond where it will be settled and, if necessary, treated.

The collection ditches will primarily be located within the pit area, at the toes of the waste dumps, at the toe of the mineralized material stockpile, and near the toes of all mine haul roads.

At the current stage of this project, little data has been collected to determine the water inflow quantities (both surface and groundwater) that can be expected for the KSM area.

At this time, vertical dewatering wells have not been included as part of the required activities. Results from the ongoing field programs may show this to be a future requirement to lower the water table or piezometric pressure within the pit prior to mining.

Mine Topsoil Salvage

Much of the mine area is devoid of soil or subsoil trees due to the recent retreat of the local glaciers. All topsoil that is suitable for reclamation purposes will be salvaged to temporary stockpiles.

Ore Haul Road Construction

A Mitchell pre-production ore haul road to the primary crusher is constructed from run-of-mine waste.

Mine Power

During the pre-production stage of development, temporary electric power will be provided in the open pit mine area from containerized diesel generator sets as required for construction activities.

For permanent mine power, refer to Section 21.12.6. There will be 25 kV overhead distribution lines constructed from the pit substation as required to serve these loads. The overhead distribution system in the pits will, as normal, evolve with pit development. Initially, overhead power lines will be installed around the perimeter of pit phase M623i. This will allow mining to proceed until Year 3, at which time the lines will be relocated to facilitate mining of the next Mitchell pit phases. Power is established at the Sulphurets pit in Year 17 and the Kerr pit in Year 20. Short stub lines will be constructed to supply power from the pit perimeter power lines into the operating benches of the pits. At the end of these line extensions, 25 kV to 7,200 V portable substations and switch-houses will supply the large electric shovels and drills which will be rated at 7,200 V.

Each electric shovel and each electric drill will have an allowance of up to 1,000 m of trailing cable. Power line locations are illustrated in the Mine General Arrangement drawing in Appendix D.

Mine Infrastructure Construction

Site preparation is also included for the following areas:

- mine equipment erection site
- explosives manufacturing plant
- explosives magazines.

Facilities such as the offices, maintenance shops, and fuel tanks will be available before mining commences at the mine site.

Pioneer Access

Pioneering roads will be required for initial access to the upper start benches of each pit (and subsequent phases). These roads will be cut into the topography both within the pit limits and outside of the pit limits. The primary equipment used for this stage of development is track dozers and small diameter percussive diesel drills. Service equipment and explosives supplies will also need to use these early roads. These roads are built at a 15% grade in a balanced cut and fill method. A pioneering description is available in Appendix D.

Initial Pit Development

Once the pioneering roads are in place, the larger mine equipment will have access and be able to start mining. The upper benches are typically small in area and do not offer enough room for the shovel-truck fleet to operate. These small upper benches will be drilled with the smaller-size diesel drill. Track dozers will push the waste material down or a shovel will sidecast down the hillside to a lower bench elevation where the larger drill fleet and shovel-truck fleet can operate. The pioneering fleet will create minimum width haul roads for the first production fleet to begin pre-stripping operations (drill, trucks, and shovels).

Pit phase M622i is mined down to the 1185 bench elevation during the pre-production period, pit phase M612 is mined down to the 1095 bench elevation during the pre-production period.

Mineralized material will be hauled to the mineralized material stockpile. Waste material will be hauled to the Mitchell dump, the haul roads, or other areas where fill is required for construction purposes.

PIONEERING AND PRE-PRODUCTION SCHEDULE OF ACTIVITIES

A preliminary schedule of the pioneering and pre-production development activities is shown in Figure 18.38. Pioneering roadwork starts in Year -3 (2013) when the Frank Mackie winter road is available; this roadwork must be completed in the same year. Other pioneering tasks, including assembly pad preparation, continue into Year -2

(2014). Pre-production mining starts six months after the access road from Eskay Creek is completed and lasts two years from Year -2 (2014) to Year -1 (2015). Plant start-up is scheduled for Year 1 (2016).

Due to the high demand for mining equipment in the current commodities cycle, purchasing commitments for large mining equipment are required well in advance of mining activities. A three-year lead time is required for the electric cable shovel, meaning a commitment must be made by mid-Year -4 (2012). A two-year lead time is required for the haul trucks and the large drills, meaning a commitment must be made by mid-Year -3 (2013). Tree-clearing and grubbing activities must be started in Year -2 (2014) in order to prepare the site for mining activities.

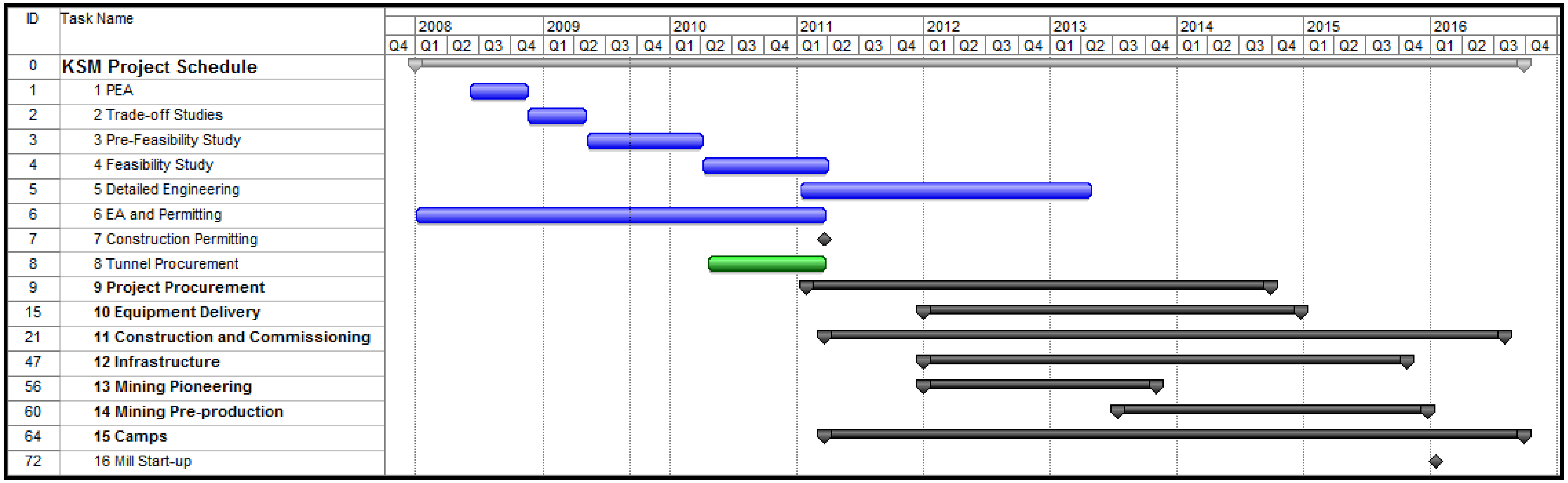
The site for mine equipment erection must be constructed during pioneering and completes before the Eskay Creek access road is completed. Equipment delivery and assembly for large mining equipment (shovels, trucks, and drills) begins as soon as the Eskay Creek access road is completed.

Preparation of the sites for explosives facilities must begin in the summer of Year -2 and completed prior to the start of mining with large mining equipment. Temporary explosives storage will be required for the pioneering stage of mine development and may be required for initial pre-production.

The mine power distribution network must be completed before Year 1 (2016). The entire pre-production fleet is diesel powered; electric equipment will only begin operation after the ore tunnel is completed.

During pre-production, the Mitchell pit phase M621 is mined to 1095 m and M622i is mined to 1185 m, exposing the necessary mineralized material required to achieve the full mill production rate of 120,000 t/d of mill feed. This development must be completed by the end of Year -1 when the mill is scheduled to receive the first mineralized material. A schedule of mine pioneering and pre-production activities is shown in Figure 18.38. A more detailed Gantt chart is available in Appendix D.

Figure 18.38 Schedule of Mine Pre-production Activities



18.8.4 *MINE PRODUCTION DETAIL*

End-of-period mine status maps have been developed and are shown in Appendix D. Each period is described in this section.

PIONEERING

During the pioneering phase, the main access road to the Mitchell pit area laydown area is built with dozers and excavators. This road is designed 10 m wide using a balanced cut and fill method. The Mitchell M621 and M622i accesses to the initial benches are also built using dozers, 10 m wide at 15% grade with balanced cut/fill.

PRE-PRODUCTION

During the pre-production period, M621 is mined to 1095 m and M622i is mined to 1170 m elevation. M261 material is used to build the M621 haul roads (8% grade, 38 m width) and the remaining material is used to widen the main access road and build a dump on the south side of the Mitchell Valley at 810 m elevation. This dump will be built from the bottom up in lifts. M622i material will build the M622i access (8% grade, 38 m width); the remaining material will fill-in the dump access road to the 1200 dump on the north side of the valley and start to build that dump using a top-down method. During this period, both the north and south dumps in the valley will remain out of the Mitchell Creek until the Mitchell diversion tunnel is built and the glacier water can be routed through the tunnel.

YEAR 1

By the end of Year 1, M612 is mined to 930 m and M622i is mined to 945 m elevation. The M612 material is used to widen the main access road further to the west and continue the 810 south dump to the west. M622i material will be used to fill in the 990 dump access road to proper width and the remaining material will be dumped to the 990 dump on the north side of the valley in a top-down method.

YEAR 5

At the end of Year 5, M621 is mined out and M622i is mined down to 645 m; M623i is mined to 1005 m and M624i is mined to 1335 m. All M621 and M623i waste is dumped on the south side of the Mitchell valley in lifts ending at the 880 m elevation. M624i waste is first used to build a 1605 dump access road to operating width and the remaining material is dumped into the 1605 dump on the north side in a top-down method. Dumping all the waste production from the M624 pit may be too high an intensity to maintain stability at the dump crest. If operating experience dictates, some of the material shown to the 1605 dump may have to be hauled to the 1200 dump and placed as a wrap around.

YEAR 10

At the end of Year 10, M622i has been mined to completion, and M623i is strictly in ore and down to 630 m. M624i accounts for most of the material during this period and it is mined to 750 m with the upper waste going to the 1605 north dump and the lower waste going to the 1230 north dump. M625i is mined to 1455 m and the material is used to help build the lower 990 dump using a bottom-up method and building in lifts. At the end of Year 10, there is still no waste dumped into the McTagg Valley. This allows the maximum time to build the McTagg diversion tunnel.

YEAR 20

At the end of Year 20, M623i and M624i have been mined to completion and M625i is only in ore and mined down to 600 m. The waste from M625i is hauled to the south dump (which now extends up the McTagg Valley), building lifts from the bottom up and ending at the 855 m elevation. The Sulphurets pit has been started and is mined down to 1485 m. The initial waste from Sulphurets goes into building an 8% access down to the dump area in the McTagg Valley. Once this access is completed, waste is hauled to the Sulphurets dump using the existing access road and building the dump to a final closure slope of 23°. The remaining waste is hauled to the south dump in the McTagg Valley. A small 15% access has also been built to the top of the Kerr pit to allow mining to start there. The Kerr pit has just been started and mined down to 1740 m. Waste from Kerr pit during this period will go into building the 8% access to the Mitchell/McTagg valleys. Reclamation has been started on the Mitchell Valley dumps, pushing the NPAG material from the 1605 north dump down over the rest of the lower material, finishing at the final 23° slope.

LIFE-OF-MINE

At the LOM, all the Mitchell, Sulphurets, and Kerr pits have been mined to completion. Waste from the Kerr and Sulphurets pits is hauled to the south dump in the McTagg Valley, building from the bottom up in lifts and to the final closure slope of 23°, filling it up to the 1025 m elevation.

18.9 MINE OPERATIONS

The mining operations will be typical of open-pit operations in mountainous terrain in western Canada and will employ tried and true bulk mining methods and equipment. There is a wealth of operating and technical expertise, services, and support in western Canada, BC, and in the local area for the proposed operations. A large capacity operation is being designed; therefore, large scale equipment is specified for the major operating functions in the mine to generate high productivities, which will reduce unit mining costs and allow the lowest mining cost to be achieved. Large scale equipment will also reduce the labour requirement on site and will dilute the fixed overhead costs for the mine operations. Much of the general overhead for the mine operations can be minimized if the number of production fleet units and the labour requirements are minimized.

18.9.1 ORGANIZATION

GENERAL ORGANIZATION

The KSM operations will be organized in the general manner as illustrated in Figure 18.39. Most of the direct operating functions will be performed on site; however, financial and administrative support that does not need to be onsite should be done offsite. Consideration should be given to outsourcing activities such as payroll, accounting, etc. to reduce employee transportation and onsite accommodation requirements. Mine operations will deal solely with the organization areas as highlighted in Figure 18.39. Other areas of the organization are dealt with elsewhere in the report.

Mine operations are organized into three areas: direct mining, mine maintenance, and general mine expense (GME).

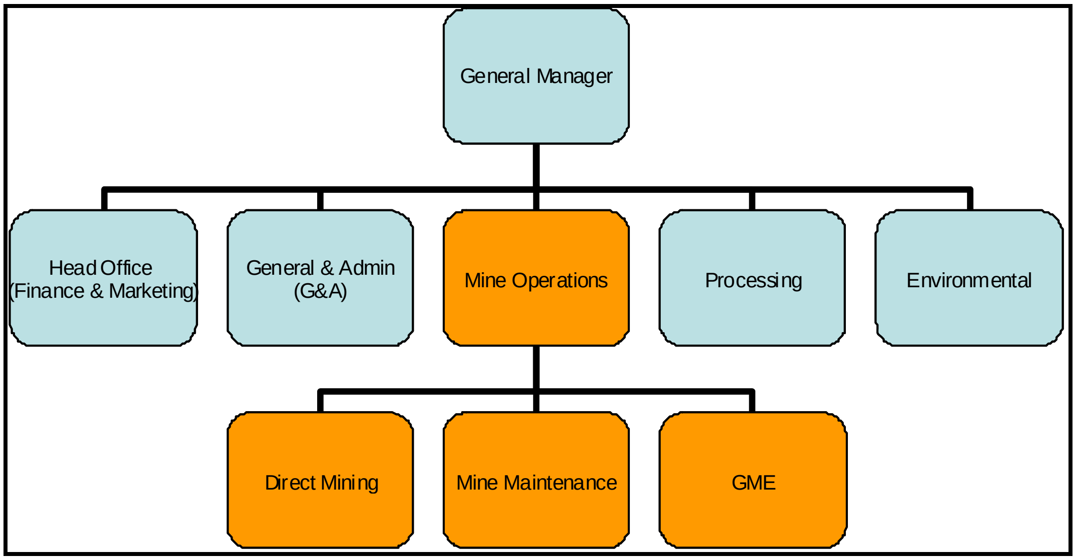
The direct mining area accounts for the drilling, blasting, loading, hauling, and pit maintenance activities in the mine. Costs collected for this area include the mine operating labour, mine operating supplies, equipment operating hours and supplies, and distributed mine maintenance costs. The distributed mine maintenance costs include items such as maintenance labour, repair parts, and energy (fuel or electricity), which contribute to the hourly operating cost of the equipment and are distributed as an hourly operating cost that is applied to the scheduled equipment operating hours.

The mine maintenance area accounts for the overhead of supervision, planning, and implementation of all activities within the mine maintenance function. Costs collected for this area include salaried personnel (supervisors, technical planners, and clerical), operating supplies for the various services provided by this area, and general shop costs The cost in these items are not included in the distributed mine maintenance costs.

The GME area accounts for the supervision, safety, and training of all personnel required for the direct mining activities as well as technical support from mine engineering and geology functions. Costs collected for this area include the salaries of personnel and operating supplies for the various services provided by this function.

In this study, the direct mining and mine maintenance are planned as an owner-operated fleet with the equipment ownership and labour being directly under operations. It may be possible to contract out some of the direct mining activities under typical mine stripping contracts and maintenance and repair contracts (MARC) as has been done at other operations. The viability and cost effectiveness of contracting can be determined in future detailed planning and commercial negotiations. The exception for this study involves blasting where (similar to other western Canadian mining operations) the mine will employ the blasting crew but, due to the specialty expertise required, the supply and onsite manufacturing of blasting materials is assumed to be contracted out. All infrastructure required for the blasting supply contractor will be provided by the operations.

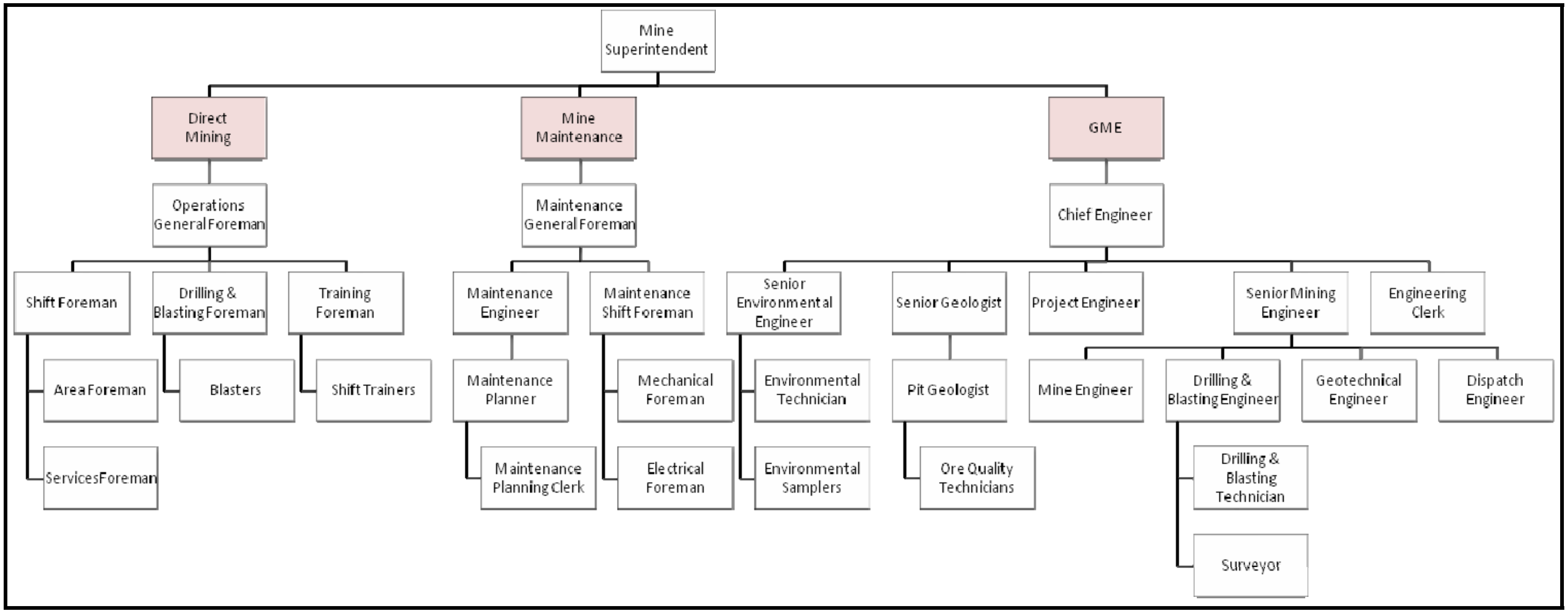
Figure 18.39 General Organization Chart



Mine Operations Organization

Details of the mine operations organization are illustrated in Figure 18.40, showing the breakdown of the direct mining, mine maintenance, and GME functions.

Figure 18.40 Mine Operations Organization Chart



18.9.2 DIRECT MINING AREA

The direct mining area accounts for the drilling, blasting, loading, hauling, and pit maintenance activities in the mine.

In situ rock will require drilling and blasting to create suitable fragmentation for efficient loading and hauling of both mineralized and waste material. Mineralized material and waste limits will be defined in the blasted muck pile through blast hole assays and grade control technicians. A fleet management system will assist in optimizing deployment and utilization of the loading and haulage fleet to meet the production plan. Support personnel and equipment will be required to maintain the mining area, ensuring the operation runs safely and efficiently. Descriptions of the direct mining unit operations are outlined in this section.

DRILLING

Areas will be prepared on the bench floor blast patterns in the in situ rock. The spacing and burden between blast holes will be varied as required to meet the specified powder factor for the various rock types. Dozers will be used to establish initial benches for the upper hill side benches. Drill ramps will be cut between benches where the outside holes on established benches do not meet the burden and spacing requirement of the pattern for the next bench below.

The blast hole drills will be fitted with GPS navigation and drill control systems to optimize drilling. The GPS navigation will enable stakeless drilling and is recommended for efficiency in detecting hole locations and accuracy of set-up, particularly since this is a high snowfall area. Accuracy of set-up also optimizes the blast performance and it is a proven technology utilized at most mines in Western Canada.

The drills will also be fitted with automatic samplers to provide grade control samples from the drill cutting in the mineralized material zones. The drillers will take the cuttings samples (2 to 3 samples per hole may be required) and bag and tag the samples for the mineralized material control technician to collect each day. These samples will be used for blast hole kriging to define the mineralized material/waste boundaries on the bench as well as stockpile grade bins for the grade control system to the mill.

Diesel hydraulic and electric rotary drills (311 mm bit size) will be used for production drilling, both in mineralized material and waste.

A 150 mm diesel hydraulic percussive drill is also specified to operate in all pit phases for controlled blasting techniques on high wall rows and development of initial upper benches. Development drilling requirements have not been detailed in this study but an allowance has been made for costing purposes. Drilling for controlled blasting requirements have been estimated based on an estimate of the length of the pit wall exposed on a bench in any given year multiplied by the typical cost per metre of wall.

A detailed drill study is recommended for more advanced project studies. This will help determine the penetration rate that can be expected for the selected drills and the specific rock types that exist within the pit area.

BLASTING

Powder Factor

The drilling and blasting design is required to provide a particle size distribution (fragmentation) within the broken rock (muck pile) and ease of digging within the muck pile that is suitable for high productivity from the shovel and truck fleet. This is a function of various aspects of the rock strength and rock fabric and how it is affected by the applied explosives. A "hard digging" muck pile will cause poor productivity in the loading fleet, excessive bucket and tooth wear, increased maintenance costs, poor load profiles, longer loading times for the trucks, and will adversely affect areas as diverse as mining plan conformance and mill throughput. The most common approach to the provision of a muck pile that is "easy" to dig is the combination of long inter-row delays and an increased powder factor.

Similar large open pit projects in the KSM area use a powder factor of 0.32 kg/t for competent rock, which will achieve a fragmentation adequate for the size of shovels to be used at KSM. Informal discussions with other mines and explosive suppliers in BC confirm that a power factor of 0.32 kg/t is suitable in this area.

A detailed blasting study is recommended for more advanced project studies. This will assist in determining the most applicable powder factor and explosives type for the rock types present at KSM. Some operations also increase the blasting energy in mineralized material to enhance mineralized material comminution (crushing and grinding). Blasting for improved mine-to-mill performance can be optimized in future studies.

Explosives

A contract explosives supplier will provide the blasting materials and technology for the mine. Because of the remote nature of the operation, an explosives plant will be built on site. The nature of the business relationship between the explosives supplier and the mining operator will determine who is responsible for obtaining the various manufacture, storage, and transportation permits as well as any necessary licences for blasting operations. This will be established during commercial negotiations.

Until the extent of ground water and surface water in the blast holes is determined, it is assumed that all of the holes will use a 75/25 emulsion/ammonium nitrate and fuel oil (ANFO) mix explosive. Higher use of ANFO and possible borehole liners to keep the ANFO dry (to prevent incomplete detonations) can be investigated in future studies to reduce blasting costs.

Blasting accessories for initiation and detonation of the explosives in the blast holes will be stored in magazines.

Specifications for the blasting plant, explosives storage magazines, and the locations of these facilities must adhere to the Explosives Act of Canada regulations as published by the Explosives Regulatory Division of Natural Resources Canada as well as regulations as published by the MEMPR in BC (in particular, the Health, Safety and Reclamation Codes for Mines in BC). The location of the blasting plant and the explosives magazines are determined by the table of distances that govern the manufacturing and storage of explosives and blasting agents.

Explosives Loading

Loading of the explosives will be done with bulk explosives loading trucks provided by the explosives supplier. The trucks should be equipped with GPS guidance and be able to receive automatic loading instructions for each hole from the engineering office. This practice is common now in Western Canada and the explosives supplier's trucks have this capability already installed. GPS guidance will be a necessity to be compatible with stakeless drilling.

The explosives product that is being used is a mix of ANFO and emulsion; therefore, the container on the truck will have two separate compartments. The separation will be set at the proper ratio so that both compartments will be emptied at the same time. This will minimize trips back and forth from the blast pattern to the explosives storage site.

The holes will also have to be stemmed to avoid fly-rock and excessive air blasts. Crushed rock will be provided for stemming material and will be dumped adjacent to the blast pattern. A loader with a side dump bucket is included in the mine fleet to tram and dump the crush into the hole. The crushed rock is provided by the onsite rock crusher specified for mine roads.

From time to time during the high snowfall period, it is expected that some of the mining areas will be shut down and may lose regular road access. If a pattern is partially loaded, it will be necessary to tie-in the loaded holes and blast before snow accumulation gets too high to find the surface lines for tie-in. To blast a partial blast, it will be necessary to 'square-off' the pattern by loading some holes to complete some rows in the pattern.

Blasting Operations

The blasting crew will be mine employees and will be on day shift only. Based on existing mines of similar size and previous experience, the estimated crew size will be six people. The main duties of the blasting crew will include setting up guard fences around the loading area, guiding and directing the explosives loading truck, preparing the boosters and primers ahead of the actual loading of the holes, stemming the blast holes after they are loaded, tie-in of the blast patterns, and detonation of the blasts.

The blasting crew will coordinate the drilling and blasting activities to ensure a minimum of two weeks of broken material inventory is maintained for each shovel. The drilling areas and ramps for the hillside holes will be prepared in suitable time for the next pattern and ramps will be surveyed if required or dozers will be equipped with GPS capabilities. In winter, the pattern preparation will also include snow removal.

Due to the snow, the drilled holes will need to be covered. Also, the blast patterns will not be staked; therefore, the blasting activities will also need to have GPS controls. The blasters will require handheld GPS units to identify the holes for the pattern tie-in. The pattern size may be limited by the rate of snowfall in some months. As the snow depth gets too high, it will be problematic to find the holes and the down line, making it difficult to tie-in the blast. This may require smaller, more frequent blasts to complete smaller patterns before the snow gets too deep. A detonation system will be used consisting of electric cap initiation, a detonating cord, surface delay connectors, non-electric single-delay caps, and boosters.

The explosives contractor will supply and manufacture bulk explosives on site. The explosives contractor’s employees will deliver explosives to the blast hole using a digitally controlled ‘Smart’ truck, as is common in Western Canadian surface mines.

A 1.8 m subgrade is assumed to ensure that there are minimum high spots between holes on the resultant bench floor. The height of the explosives column is calculated from the explosives density and hole diameter to give the required powder factor. The remainder of the hole is backfilled with drill cuttings or crushed rock.

Based on the desired powder factor, the blasting specifications for the KSM operations have been evaluated for the large diameter hole size. The blasting assumptions are summarized in Table 18.25.

It has been assumed that all rock will require drilling and blasting. These parameters are typical for other mines in the western Cordillera and will be re-evaluated in the future with a detailed blasting study using site specific rock strength parameters.

Table 18.25 Blasting Assumptions

Blasting Pattern – Mineralized Material & Waste	Specifications
Spacing	8.9 m
Burden	8.9 m
Hole Size	12¼ in / 311mm
Explosive In-Hole Density	1.12 g/cc
Explosive Avg. Downhole Loading	85.1 kg/m
Bench Height	15 m
Collar	4.5 m
Loaded Column	12.3 m
Sub-drill	1.8 m
Charge per hole	1,049 kg/hole
Rock SG	2.66 t/m^3
Yield per hole	3,160 t/hole
Powder factor	0.32 kg/t

LOADING

Ore and waste will be defined in the blasted muck pile. A fleet management system will assist in optimizing deployment and utilization of the loading and haulage fleet to meet the production plan, to track each load to ensure material is hauled to the correct destination, as well as to provide production statistics for management and reconciliation of the mine operations with respect to the mine plan.

The design basis assumes minimizing the supplier and model of shovels to simplify the maintenance function and reduce capital equipment and maintenance spares. Two 85-t payload diesel hydraulic shovels and three 100-t dipper electric cable shovels have been selected as the primary digging units. The diesel hydraulic shovels are selected for flexibility and mobility in accessing the thin top pit benches.

The loading units will be fitted with a GPS-based digging monitor that will enable digital dig boundaries from the 'ore' control system to define the ore types and waste on the shovel operator's graphics screen in the cab. It also provides elevation control, improving the bench floors, which effects shovel and truck efficiencies and maintenance.

There are years where there is a large component of ore being reclaimed from the stockpile to feed the mill. In these years, it is intended to relocate the necessary shovels to the stockpile area for the required length of time.

Bench widths are designed to ensure maximum operating widths to allow double-sided loading of trucks at the shovels. However, there are areas where single-sided loading will be necessary and reduced productivity for the shovel will be encountered, such as the upper benches of the pit phases where the end of the bench meets topography. For this study, this effect on shovel productivity has been accounted for but it is assumed that it is a relatively small percentage of the total material mined, or that ancillary equipment will be deployed to prepare the digging areas for higher shovel productivity. This can entail dozing small benches down slope to the next bench, trap dozing, etc.

It is recommended that further optimization of the shovel fleet be completed in more advanced studies. Specifically, there are many years where a significant portion of the large shovel's production capability is not being fully used due to increased haul distances, limiting the trucks available to the shovels. Evening out the haul distances would even out the annual waste requirement, giving a more even shovel usage year to year. However, this may conflict with revenue maximization (targeting higher grades), which the scheduling algorithm is optimizing to improve project NPV. Also, optimization studies should evaluate whether the use of a rubber-tired front end loader would provide economical benefits to the operation, providing flexibility and mobility that the electric cable shovel cannot provide.

HAULING

Ore and waste haulage will be handled by large off-highway haul trucks with a 345-t payload. Haulage profiles have been estimated from pit centroids at each bench to designated dumping points for each time period. These haul profiles are inputs to the truck simulation program and the resulting cycle times are used in the MineSight® schedule optimization routine (MS-SP), which is set to maximize project NPV by using the shortest haul to a feasible destination. The payload, loading time, and haul cycle then determine the truck productivity.

A GPS-based fleet management/dispatch system is specified for the trucks, shovels, and ancillary equipment fleets to ensure coordination and proper management of the fleet over multiple pit phases in a large mining area. State-of-the-art wireless communication and location systems for management and potential navigation assistance should be considered during the detailed planning and specifications for the project. Other operations are applying these equipment operating aids to increase the efficiencies of the large mining equipment and managing the ancillary support fleet and thereby reducing operating costs. The capacities and capabilities of these systems have improved greatly in the last few years and the costs are decreasing.

It is recommended to complete further optimization of the haulage fleet in more advanced studies. In this study, it is assumed that the large off-highway haul trucks are used for all mining requirements. However, there is the potential to use a smaller sized shovel-truck fleet for such specific activities as the opening up of upper benches where the initial mining room is limited and for the completion of small benches on the pit bottoms.

In recent years, the availability of large truck tires has been a major issue for operating mines and securing supply could be a project risk. To mitigate this, extra effort has been specified in road maintenance to enhance tire life. This includes a rock crusher for road grading material.

PIT MAINTENANCE

Pit maintenance services include haul road maintenance, mine dewatering, transporting operating supplies, relocating equipment, and snow removal.

The snow fleet will be manned by mine operations staff in normal winter conditions with operators taken from reduced activities such as dust control and summer field programs. During severe storms, additional crew to operate the snow fleet will be drawn from truck and shovel operations as the fleets shut down. This will ensure priority fleets remain operating.

18.9.3 *MINE MAINTENANCE AREA*

The mine maintenance area accounts for the supervision and planning of the mine maintenance activities.

Mine maintenance activities will be directed under the Mine General Foreman who will assume overall responsibility for mine maintenance and will report to the Mine Superintendent (in an alternate organization, this position may be filled at a Superintendent level reporting to the General Manager). Maintenance planners will coordinate planned maintenance schedules. The daily maintenance shift coordination will be carried out by Mechanical and Electrical Foremen.

The mine maintenance department will perform break-down and field maintenance repairs, regular preventive maintenance, component change-outs, in-field fuel and lube servicing, and tire change-outs. Major component rebuilds are done by specialty shops off site and are costed as sustainable capital repairs.

18.9.4 *GENERAL MINE EXPENSE AREA*

This section describes the mine GME as costed in the mine cost model, available in Appendix D.

The GME area accounts for the supervision, safety, and training for the direct mining activities as well as technical support from mine engineering and geology functions. Mine operation supervision will extend down to the Shift Foreman level.

A mine General Foreman will assume responsibility for overall supervision for the mining operation. A General Mine Foreman will be responsible for overall open pit supervision and equipment coordination. Supervision will also be required for drilling and blasting, training, and dewatering. A Mine Shift Foreman is required on each

12-hour shift, with overall responsibility for the shift operation. Security/first-aid staff and mine clerks will also report to the Mine Superintendent.

Initial training and equipment operation will be provided by experienced operators. As performance reaches adequate levels, the number of trainers can be decreased to a sustaining level.

A Chief Mine Engineer will direct the mine engineering department. The Senior Mining Engineer will coordinate the mining engineers, drilling and blasting engineers, the mine planning group, surveyors, and geotechnical monitoring. A Senior Surveyor will assume responsibility for surveying for the entire property and will supervise the surveyors. Surveying will use GPS-based systems.

The geology department will include a Senior Geologist, Pit Geologists, and Ore Grade Technicians. This department will be responsible for local step out and infill drill programs for onsite exploration activities and updating the long range mine orebody models. The geology department will also provide grade control support to mine operations, managing and executing the blast hole sampling and blast hole kriging of the short range blast hole models for operations planning and ore grade definition.

The Geotechnical Engineer will assume responsibility for all mine geotechnical issues including pit slope stability and hydrogeological studies. The Geotechnical Engineers will also have oversight for the whole property for any geohazard monitoring and assessment programs being carried out by safety personnel or third party consultants. This includes avalanche monitoring and control.

18.10 Mine Closure and Reclamation

At the end of the mine life, a mine closure and reclamation plan will be implemented that will meet the end land use objectives and satisfy the regulatory commitments.

Ultimately, the goal is to re-establish the land to a productive environment that will be compatible to its natural surroundings. Restoration of terrestrial and aquatic life will be the primary objectives. Stable, re-shaped landforms will be created to ensure self maintenance capability in perpetuity.

Progressive reclamation, in conjunction with on-going mining activities, will be practiced where applicable to minimize overall mine closure costs. This approach will also allow early monitoring of reclamation activities and advance closure to certain mine areas.

Although there is little surficial soil in the pre-mining topography, any suitable soils excavated during mining will be stockpiled and used to cap recontoured landscapes at decommissioning. Where possible, the direct placement of suitable topsoil material will be carried out as part of a progressive reclamation effort. This will reduce the amount of disturbed land during operations, and will minimize topsoil stockpile losses and re-handling costs.

Post closure landform, reclamation, and ARD/heavy metal impacts of the project will be the subject of extensive work in future studies. The following general design aspects for post-mining matters are based on typical considerations for other projects in this area and specific early evaluation of the rock. Detailed design criteria will be adjusted based on these future studies and requirements.

18.10.1 MINE WASTE DUMP RECLAMATION

Mine dumping will be comprised of a mix of top-down end dumping, wrap arounds, and bottom-up lift dumping. The top-down dumps will form at the natural angle of repose of 37° and will later be reclaimed with dozers to the closure slope of 23°. The volume of material that is dozed to meet the 23° reclaim slope is reduced with proper design of the wrap arounds. The bottom-up dumps will be built at the final closure slope of 23°; at reclamation, the entire dump will be capped with a layer of low permeability till and NPAG material.

18.10.2 MINE ROADS AND DYKES

Decommissioned mine roads will be scarified and capped with available surficial soils. Dykes and dams that are exposed above the water line will also be scarified and capped with suitable soils. The surfaces will then be seeded to establish vegetation.

18.10.3 PIT AREAS

Generally, mined out pits will naturally fill with water from surface runoff and groundwater, forming lakes. Spillways will be constructed to manage the overflow and directed to the watercourses as established in the mine closure water management plan. Ditches will be constructed on the pit berms to manage the runoff from the pit walls. Typically the pit walls are not planned to be re-sloped.

18.11 MINE EQUIPMENT

The mining equipment descriptions in this section provide general specifications so that dimensions and capacities can be determined from the manufactures specification documents.

18.11.1 MAJOR MINE EQUIPMENT

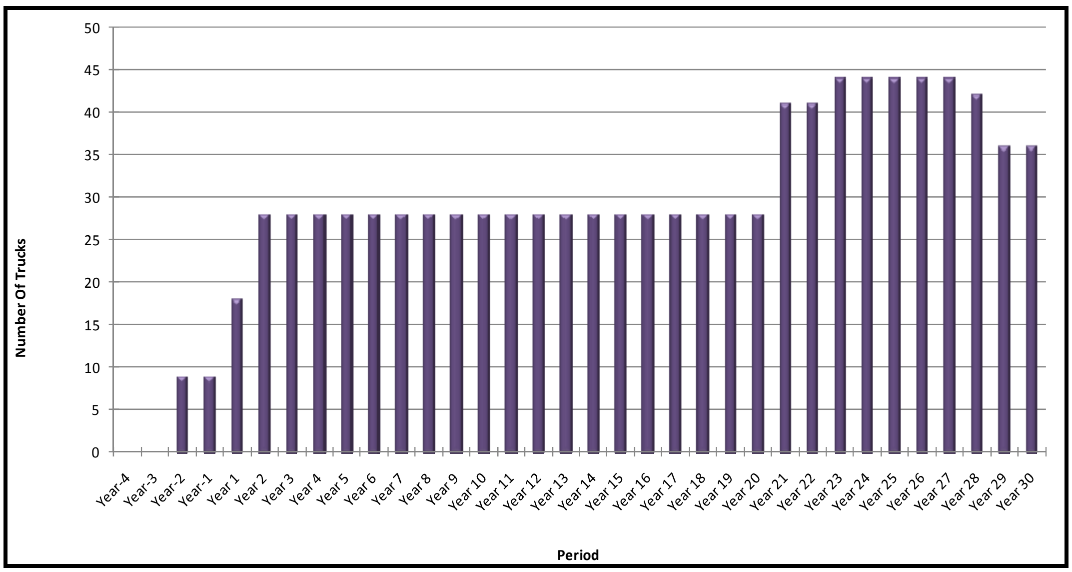
The production requirements for major mining equipment over the LOM are summarized in Table 18.26. The full fleet schedule requirements are shown in Appendix D. According to the current production schedule and the haulage assumptions, the maximum number of trucks required until Year 20 is 28 trucks; for the remaining LOM, the maximum number of trucks required is 44. All other equipment in Table 18.26 show the maximum number of units required.

Table 18.26 Major Mine Equipment Requirements

	PP	Y10	Y20	Y30	Max
Drilling					
Primary Drill – 311 mm Diesel Hydraulic Drill	2	2	2	2	2
Primary Drill – 311 mm Electric Drill	0	2	2	1	2
High Wall Drill - 150 mm Diesel Hydraulic Drill	1	1	1	1	1
Loading					
Primary Shovel - 85 t diesel hydraulic shovel	2	2	2	2	2
Primary Shovel - 100 t electric cable shovel	0	3	2	0	3
Hauling					
Haul Truck – 345 t	9	28	28	36	44

The haul truck fleet size schedule is shown in Figure 18.41.

Figure 18.41 Haul Truck Fleet Size



18.11.2 DRILLING EQUIPMENT

The primary production drilling will be carried out in mineralized material and waste with electric rotary drills with a 311 mm hole size. The production drills will be fitted with GPS navigation and drill control systems to optimize drilling. Production drilling assumptions are listed in Table 18.27.

Table 18.27 Production Drilling Assumptions

Production Drill – Mineralized Material & Waste	Electric Rotary	Diesel Rotary
Bench Height (m)	15	15
Subgrade (m)	1.8	1.8
Hole Size (mm)	311	311
Penetration Rate (m/h)	33.0	33.0
Hole Depth (m)	16.8	16.8
Overdrill (m)	1.0	1.0
Setup Time (min)	2.0	2.0
Drill Time (min)	32.3	32.3
Move Time (min)	2.0	2.0
Total Cycle Time (min)	36.3	36.3
Holes per Hour	1.65	1.65
Re-drills (%)	6	6

A 150 mm diesel percussive drill is also specified to operate in all pit phases for controlled blasting techniques on high wall rows and for development of initial upper benches. Development drilling requirements have not been detailed in this study but an allowance has been made for costing purposes.

A detailed drill study is recommended for more advanced project studies. This will help determine the penetration rate that can be expected for the selected drills and the specific rock types that exist within the pit area.

18.11.3 Blasting Equipment and Facilities

The blasting activities will require an explosives manufacturing plant as well as management and maintenance facilities for the explosives contractor. The KSM operation will own and build the facilities for use by the contractor, including room and board facilities. This will also include serviced buildings for offices, warehousing, manufacturing, light maintenance, and power and communication links.

The contractor will provide specialty equipment such as the computer-controlled bulk loading trucks and any other site-specific equipment required. The operating costs of these facilities as well as the ownership and operating costs of the explosives equipment is part of the contractor's unit rate for the supply of explosives delivered to the hole.

A blast hole stemming unit will be required to load cuttings into the hole and stem the unloaded portion of the hole. This unit will be provided by the KSM operation.

The blasting activities will also require onsite storage magazines for explosives and blasting accessories. These facilities will also be provided by the KSM operation.

The current assumption is that blasting explosives and supplies will be delivered on a weekly basis to the mine. In order to ensure that there is no disruption of supply to the mine operations, a two-week supply of explosives and blasting accessories will be maintained in the onsite storage magazines. A larger onsite inventory will be considered in future studies as allowances are made for weather/access issues.

Specifications for blasting plant and explosives storage magazines and the locations of these facilities must adhere to the Explosives Act of Canada, regulations as published by the Explosives Regulatory Division of Natural Resources Canada, and regulations as published by the MEMPR in BC (in particular, the Health, Safety, and Reclamation Codes for Mines in BC). The location of the blasting plant and the explosives magazines is determined by the table of distances that govern the manufacturing and storage of explosives and blasting agents.

A geohazard assessment for selected explosive plant locations has not been completed at this time. It is recommended that this be completed at the next level of study to determine if the chosen sites are suitable or if potential hazards exist from slope instability, glacier activity, or avalanche potential.

18.11.4 *Loading and Hauling Equipment*

The shovel-truck fleet selected for KSM is the 100-t dipper class of electric shovel, and the 345-t payload class of truck. An 85-t class diesel-hydraulic shovel is also required to access difficult development benches and enable pre-production mining before power is established to the mine site. The 85-t units loading the 345-t trucks are suitable as full production shovels as well.

A GPS-based fleet management/dispatch system is specified for the trucks, shovels, and ancillary equipment fleets to ensure coordination and proper management of the fleet over multiple pit phases in a large mining area. State-of-the-art wireless communication and location systems for management and potential navigation assistance should be considered during the detailed planning and specifications for the project. The capacities and capabilities of these systems have improved greatly in the last few years and the costs are decreasing.

Diesel generator sets are included to facilitate relocating the electric drills and electric shovels as required.

Future studies will evaluate tire supply issues which have been a problem in the past.

18.11.5 *Dewatering Equipment*

It is important to control the water that is in the active mining areas. In-pit water generally increases the cost of mining especially in blasting where explosives loading, explosives costs (ANFO vs. emulsions), and blast performance are affected by water. Flooded box cuts need to be drained. Rock cuts to tires increase in wet conditions and the presence of water in the shovel digging area can greatly decrease the average tire life of the trucks. Rocks can easily be hidden in puddles that the haul trucks have to drive through, which can lead to instantaneous tire failure. Wet muck that the shovels are digging will freeze to the sides of the truck boxes in the winter and this “carry back” results in less material being hauled per truck load (i.e. lower productivities). Water also affects the stability of walls and dumps. All of these effects need to be addressed by an effective pit dewatering program.

If required, horizontal drain holes must be established in the final walls as they are exposed. The design and amount required should be determined by geotechnical consultants. On the active bench floor, the water that is collected from the horizontal drain holes will be directed to the sump where it can be removed from the pit.

Sloped bench floors will also aid in keeping the digging face dry. A gradient of 1% is usually sufficient to collect the water to one area where sump pumps can then be used to pump the water out of the pit. The direction of the slope will have to be determined individually for each pit but, generally, the floors should slope downwards to the initial starting point of each bench. That way, as the shovels dig outwards and away from this starting point, the water will drain back away from the shovel digging area. A sump can be dug into the bench floor to collect this water and a pump put in to remove the water from the pit. The sloping of the floors will also cause the berms to be sloped and the ditches that are established in the berms will naturally drain to one side of the pit. This side of the pit is where the berm sumps should be established.

In situ water also reduces drilling productivity and creates many problems in blasting operations. Large amounts of water can lead to holes caving and under-blasting due to incomplete detonation. It can also lead to hard digging, as some holes are unable to be loaded because they don’t stay open. This leads to higher maintenance costs on the shovel and lower productivity because of poor digging conditions.

It may be necessary to install vertical dewatering in advance of pit development and pump water from these to remove the in situ water. The design and amount of vertical wells required should be determined by geotechnical consultants. At this stage of the project, it is assumed that vertical well dewatering is not required.

All surface water and precipitation in the pit will be handled by submersible sump pumps installed in each active pit bottom as part of the flexible and moveable bench scale pumping system. The sump pump will be connected to semi-permanent and permanent piping systems to convey the sump water out of the pits. The sump will be installed with each box cut as the benching is advanced. With the high amount of precipitation, it is assumed that the box cuts will have to be made wide enough to facilitate the sump pump and piping (as required) as the face advances and until a bench sump can be established on each new bench. The excavation of the sumps is therefore included in the direct mining costs but the pump handling and piping is included in mining support costs.

Pit water will be collected and treated for ARD and metal leaching before discharging.

18.11.6 MINE SUPPORT EQUIPMENT

The mine support equipment fleet requirements over the LOM are summarized in Table 18.28.

Table 18.28 Mine Support Equipment Fleet

Fleet	Function	Year5
Hole Stemmer – 3 t	Blast Hole Stemmer	2
Track Dozer – 630 kW	Shovel Support	5
Rubber Tired Dozer – 250 kW	Pit Clean Up	4
Fuel/Lube Truck	Shovel and Drill Fuelling & Lube	2
Wheel Loader Multipurpose – 14 t	Pit Clean Up	2
Water Truck – 20,000 gal	Haul Roads Water Truck	4
Track Dozer – 430 kW	Dump Maintenance	4
Motor Grader – 400 kW	Road Grading	5
Motor Grader – 220 kW	Road Grading	2
Tire Manipulator	Tire Changes	2

BLAST HOLE STEMMERS

Two blast hole stemmers with the ability to lift approximately 2 t of material are included in the fleet. After the blast holes have been loaded with blast material and charged, the hole stemmer takes drill cuttings and fills the top portion of the blast holes. The capping contains the blast to within the hole and outward into the ground, rather than having the blasted energy escape out of the top of the hole.

TRACK DOZERS

A fleet of five 630 kW track dozers will support the shovels and four 430 kW track dozers will support the dump operations. The dozer fleet will also be used to prepare drill ramps, maintain pit floors, and slope inactive dumps as they become available.

The dump dozers maintain the operating surface of the dump and assist in ‘spotting’ the trucks as they back into the dump crest to dump their load by ensuring a lateral impact berm is always in place. Dozing is required to level any material free-dumped on the top surface of the dump to fill slumped or settled areas and to maintain good driving conditions for the trucks.

The dozers are used on some occasions to trap-load to the shovels. From a bench above the shovel’s working bench, the dozers push material to the shovels. Under certain circumstances, this facilitates a higher productivity for the shovels and, where there are frost overhangs, it provides a safer working condition.

The dozers are used to push out haul routes for the haul trucks and other equipment within the mining pits, and to assist the graders to maintain those haul routes.

The dozers rip material that requires ripping before digging or dozing. The productivity and required hours of ripping are not estimated and it is assumed that the fleet chosen has the ripping hours available for what is required on site.

Intermittently, the dozers assist in other odd jobs in the mining pits and at the waste dumps, such as towing vehicles, cutting ditches, cleaning shovel work areas, etc.

RUBBER-TIRED DOZERS

Four 463 kW rubber-tired dozers are included in the mine operations support fleet.

The rubber-tired dozers are used to clean up spill rock around the shovel faces, assist in clearing haul routes of large debris, and assist in cable handling and placement for shovel and drill moves. It is important when considering tire life for the haulers that the possibility of tire cuts is minimized. Most tire cuts occur at the shovel face from spill rock from the face itself or from material spilled out of overloaded trucks and off of the shovel buckets. Keeping a dozer at each shovel face minimizes the occurrence of this issue. Tire cuts are also prevalent for the haulers travelling at high speeds along the haul routes and running over large debris that has fallen off of other haulers. Where required, the rubber-tired dozers assist the motor graders to sweep the haul roads of this debris.

MOTOR GRADERS

Five 400 kW motor graders and two 220 kW motor graders are included in the mine operations support fleet.

The graders are used to maintain haul roads within the mining pits and on all routes to the ore crusher, ore stockpile, and waste destinations. The graders ensure the routes are free of debris and that they conform to the design parameters of the routes for cross-section and grade.

The graders are also used occasionally to level benches and waste dumps that have been excavated or dumped off design targets. With the advanced mining electronic control systems that are being recommended for the operation, the occurrence of these situations is minimized.

TIRE MANIPULATOR

Two wheel loaders fitted with tire manipulators will be used to change tires on the rubber tired fleet. The tire manipulators are dedicated to moving tires wherever they are needed on site as well as changing tires on the large rigid frame haul trucks.

MULTIPURPOSE WHEEL LOADER

Two wheel loaders, equipped with a variety of quick-coupling attachments and with a capability of approximately 14 t per bucket, are included in the fleet. The wheel loaders are outfitted with a bucket to assist in earth works (where required) and a cable-reeler to move shovel cables in long distance shovel moves. The shovels are outfitted with on-board cable-reelers but require support for long distance moves. The wheel loaders are outfitted with fork tines to support the movement of supplies and small components required for both operations and maintenance. They are also fitted with a brush for cleaning work areas at the truck shop and the offices. The wheel loaders will also feed blasted rock to the crusher.

FUEL LUBE TRUCKS

Two articulated trucks outfitted with fuel/lube arrangements are included in the mine operations support fleet. These fuel/lube trucks are used to provide lubrication maintenance to mining equipment while in the mining pit and in other working areas on site. The articulated trucks are chosen for navigation into working areas that may not be possible with standard flatbed semi-trucks. The size of the trucks is dictated by the fuel/lube arrangement that is included to support the large hydraulic and cable shovels, large haul trucks, large track and wheel dozers, and large motor graders.

WATER TRUCKS

Four rigid-frame trucks outfitted with water bodies are included in the fleet. The water trucks spray the width of the haul roads with a sheet of water in order to minimize the airborne dust that is created by the equipment on the gravel roads. The airborne dust may create both visibility (productivity) and environmental issues that are mitigated by the use of the water trucks. The size of the water bodies are chosen to correspond to the width of the roads and the distance of the road to the waste dump.

18.11.7 MINE ANCILLARY EQUIPMENT

The mine ancillary equipment fleet is listed in Table 18.29.

Table 18.29 Mine Ancillary Equipment Fleet

Fleet	Function	Year5
Track Dozer – 430 kW	Pit Support	1
Float Tractor/Trailer – 189 t	Float Tractor & Trailer	1
Hydraulic Excavator – 6 t	Utility Excavator	3
Sump Pump - 1,400 gal/min	Pit Sump Dewatering	4
Light Plant	Lighting Plant	6
250 t Crane	Utility Crane	2
Crew Cab	Supervision and Crew transportation	20
Ambulance	Ambulance	1
Hydraulic Excavator – 4 t	Utility Excavator	2
Mine Rescue Truck	Rescue Truck	1
Crew Bus	Crew Bus	5
Maintenance Truck – 1 t	Service Truck	6
Fire Truck	Fire Truck	1
Screening Plant - 12" max.	Road Crush & Stemmings	1
Picker Truck	Maintenance & Overhauls	2
Scraper - 37 t	Crush Haul for Winter Roads, Drill Steels, etc.	6
Crane – 40 t Hydraulic Extendable	Utility Crane	3
Wheel Loader – 14 t	Crusher (Road Crush) Loader	1
Snow Cat	Winter Off Road Crew Transport	1
100 t Crane	Utility Crane	2
Forklift – 30 t	Forklift	1
Forklift – 10 t	Forklift	2
Service Truck	Service Truck	5
Welding Truck	Welding Truck	4
Powerline Truck	Powerline Maintenance	2
Screening Plant - 12" max.	Road Crush	2

This section is a description of the equipment chosen and the tasks that the equipment performs in support of the mining operations.

The 433 kW dozer will be used for pit utility work including road maintenance, road construction, towing vehicles, dozing snow, and cutting ditches.

Three hydraulic excavators with the ability to pass approximately 7 to 9 t per bucket are included in the fleet. This equipment digs ditches along the haul routes for dewatering of the routes (as described in the section on haul route design), helps to construct small earth structures and ramps within the pits and mine operations areas, and assists with the excavation of sumps and other small excavations where required.

Three hydraulic excavators with the capability to pass approximately 3 to 5 t per bucket are included in the fleet. This equipment is outfitted with a vibratory hammer and is used to break up oversized material that is handled by the shovel and haul trucks but cannot be handled by the primary crusher or waste screener. The excavator is also outfitted with a bucket to dig material for the waste crusher (for road traction and hole stemming material) as well as to maintain the water diversion channels and structures.

There are 20 crew cabs are included in the fleet. These vehicles are used for transportation of mine maintenance, technical, and managerial personnel around the mine site. Seven units are used for maintenance, five units for management, one unit for surveying, one unit for environmental, three units for geotechnical, and three units for engineering.

Four 1-ton pickup trucks are included in the fleet. These pickup trucks are used to transport small good around the mine site, primarily for maintenance items but also for other miscellaneous goods.

Five maintenance service trucks outfitted with service units are included in the fleet. These units include the tooling required for maintenance personnel to perform service of other equipment in the field and will be used by maintenance personnel to perform service of other equipment in the field.

One forklift with a 30 t capacity and two forklifts with 10 t capacities are included in the fleet. The forklifts are used in the warehouse and around the maintenance shop to assist in the transportation of machine components and other goods. Many machine components exceed the 10 t weight limit of the smaller forklift, thus the need for the larger one. Most stocked items are under the 10 t limit of the smaller forklift.

Two trucks outfitted with picker arms are included in the fleet. The picker trucks are used by maintenance to lift components into equipment in the field. They are also used on occasion to lay small pipelines and transport heavier goods into the field that require lifting to larger heights than can be achieved by the wheel loader.

Five crew busses are included in the fleet. The busses are used to transport personnel coming on shift to the working areas, and personnel coming off shift out of the working areas. The busses transport most operators to and from the working areas, and also meet the requirements of maintenance and staff for transporting crews into the field.

One ambulance, one fire truck, and one mine rescue truck are included in the fleet. These three units are used to maintain the safety of personnel and equipment working on site.

One tractor with a flatbed trailer is included in the fleet. This unit is utilized for transporting tracked equipment throughout the various mining pits and working areas. Whenever possible, all long distance movement of tracked equipment is accomplished with the flatbed.

One screening plant and crusher with the ability to produce ¾" and 3" material is included in the fleet (Fintec 570 screening plant or equivalent). The screening plant will be fed by a hydraulic excavator.

Three mobile cranes are included in the fleet – two with the ability to lift 250 t components at least 20 m in height, and one with the ability to lift 100 t components at least 20 m in height. The cranes are required to lift equipment components for the initial field erection of the equipment and for major component change-out, especially for the shovels. The cranes are also used to lift the equipment itself in order to block it off for maintenance work on ground-based components such as tires and tracks. One mobile extension crane with the ability to lift up to 40 t will be used for field maintenance.

Four welding trucks are included in the fleet. The welding truck is used in support of maintenance personnel's needs for welding equipment and equipment items such as truck bodies, dozer blades, shovel buckets, etc.

Two powerline trucks with a man-lift basket are included in the fleet. The powerline trucks are required for the safe movement of all powerlines on site and, in particular, to relocate the pit supply power as the mining phases are advanced.

Six 110 kW water sump pumps are included in the fleet to pump water out of the pits.

Light plants will be required to provide lighting on the dumps and intermittently in the pits for road construction and field maintenance.

Snow Fleet

The following equipment is chosen specifically for support of the duties of the snow fleet. All equipment is chosen to start operation during pre-production and continue to the end of mine life, unless otherwise noted. The equipment is replaced as required and the costs for this equipment are applied according to the details included in the cost model.

- Six scrapers with the ability to haul 37 t are included in fleet. The scrapers are required to haul crushed rock material along the roads after heavy snowfall activities. The scrapers also remove large amounts of snow from the haul roads and mine working areas as necessary. The scrapers are also used on occasion (less than 5% of the time) for small earthmoving jobs. They may also be used for reclamation projects.
- One wheel loader with the capability to pass approximately 14 t per bucket is included in the fleet. The wheel loader is utilized during snowfall periods to clear snow from the plant area and truck shop, as well as ancillary routes within the mine. The wheel loader is also used to load the cone crusher described below.

- Two cone crushers with the ability to produce minus 6" rock are included in the fleet (Fintec 1080 or equivalent). The cone crushers will produce crushed material from mine waste rock, to spread onto haul roads by the scrapers during periods of heavy snowfall. The crushers will be fed by the wheel loader described above (the crushed product is also used for blast hole stemming).
- One Snowcat with the ability to transport five passengers are included in the fleet. The snowcat is used to transport operators to equipment that is in a location that is inaccessible to the crew bus or vans because of heavy snowfall.

The snow fleet has a low utilization as it is only required in wintertime. Other than the use of the crusher to produce road gravel, operating this equipment outside of wintertime is optional and not necessary.

18.11.8 MINE ANCILLARY FACILITIES

SHOPS AND OFFICES

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

- a welding bay
- an electrical shop
- an ambulance
- a first aid room
- a first aid office
- a machine shop area
- a mine dry
- a warehouse
- offices for administration, mine supervision, and engineering/geology staff, a lunch room, and a foreman's office.

There will be 8 truck maintenance bays that will be suitable for the 345 t haul trucks. The workshop/warehouse/office complex will be located on one end of the building, together with two small truck service bays as well as the welding bay, electrical shop, machine shop, ambulance, and first aid room. A heated pad outside will be used for washing equipment prior to maintenance and another pad for tire changes.

18.12 MINING GEOTECHNICAL

18.12.1 INTRODUCTION

OVERVIEW

BGC Engineering Inc. (BGC) has developed open pit slope design criteria to support a preliminary economic assessment (PEA) of the Kerr-Sulphurets-Mitchell (KSM) project in north central British Columbia, with a focus on the Mitchell Zone. Mine layout, environmental assessments, and engineering services for the project are being provided by Moose Mountain Technical Services (MMTS), Rescan Environmental Services Ltd. (Rescan), and Klohn Crippen Berger Ltd. (KCBL), respectively.

A draft memorandum outlining the open pit design criteria was issued to Seabridge on April 30, 2009. This section provides a summary of the geotechnical findings to date as outlined in that memorandum.

PREVIOUS WORK

"Provisional geotechnical assessments" of the rock mass of the Mitchell zone were carried out by Piteau Associates Engineering Ltd. (Piteau) between November 2007 and April 2008 (Piteau, 2008). This work resulted in recommendations for pit slope design criteria, including maximum interramp heights and overall slope angles. The recommendations were based on an inspection of rock core from the project area and a review of various geologic reports, plans, and cross-sections made available by Seabridge. The slope design criteria recommendations were provided to MMTS for mine design purposes. The results of that work were summarized in the original 2008 PEA.

CURRENT WORK

To support PEA Addendum 2009 work being undertaken by Seabridge, BGC developed slope design criteria for the proposed Mitchell pit based on a site visit completed in 2008 and additional analyses of rock mass quality and structural geology of the project area based on data provided by Seabridge in early 2009. Using this information, three design criteria cases were developed to assist MMTS in developing pits for the PEA Addendum 2009; each one based on separate design assumptions. The multiple design criteria cases are intended to assist mine planners in evaluating the sensitivity of the project economics to the pit slope design and geotechnical assumptions. The design options are based on:

- The geotechnical domains and geological discontinuity data used to develop the design criteria.
- The slope design methods, assumptions, and constraints.

- The recommended design criteria and operational factors required to achieve the criteria for each of the three cases.

The results of rock mass characterization work previously reported by BGC (BGC, 2008) have been superseded by this work. Recommendations for ongoing work to support pre-feasibility level evaluations have been provided to Seabridge.

18.12.2 ENGINEERING GEOLOGY

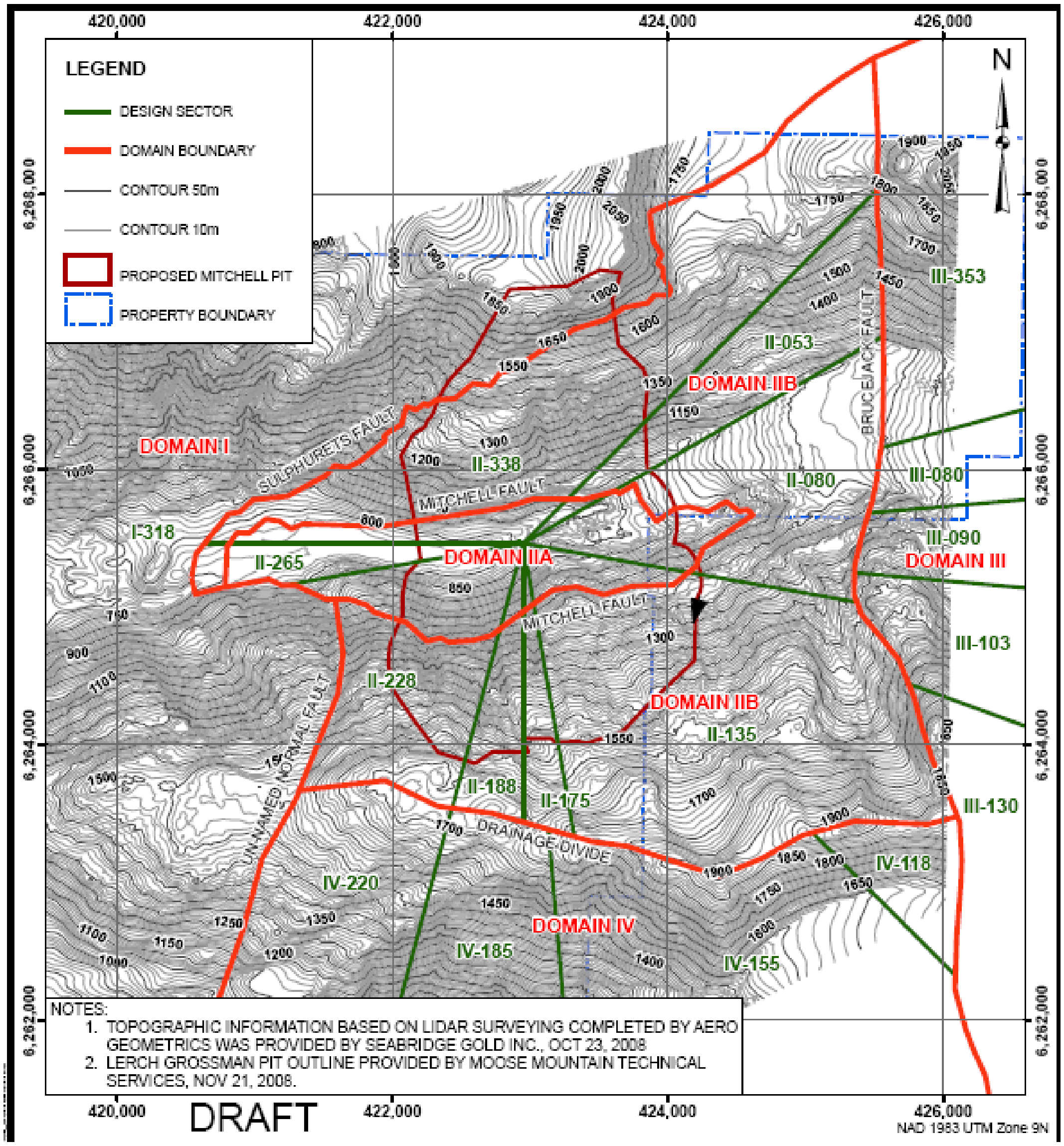
The KSM Project is located within the Triassic Stuhini Group and Jurassic Hazelton Group rocks of the Stikinia Terrrane (Seabridge, 2007). The Stuhini and Hazelton Group rocks include volcanic and sedimentary assemblages that have been folded, faulted, and intruded by Jurassic aged igneous rocks of varying composition. A zone of intense phyllic alteration covers an area of over 35 km^2 and forms intense gossanous exposures.

In the area of the Mitchell Zone, rocks of Stuhini Group have been thrust over rocks of the Hazelton Group by the Sulphurets and Mitchell faults. A variety of diorite, monzodiorite, syenite and granite dykes, sills and plugs (the "Mitchell Intrusions") have resulted in distinct porphyry style alteration of the rock mass. The "Upper Plate" (hanging wall) Stuhini Group rocks of the Mitchell Zone contain a higher percentage of intrusive rock, potassic alteration, and a higher copper to gold ratio. The "Lower Plate" (footwall) Hazelton Group rocks, which host the main mineralization of the Mitchell zone, have much stronger and more pervasive phyllic (quartz-sericite-pyrite) alteration. At depth, increased silicification and quartz veining is present. The Mitchell Fault is a distinct feature in the area of the proposed pit and can be traced visually for hundreds of metres along the north side of the Mitchell Creek Valley. The Sulphurets Fault is equally distinct and forms the boundary between the "Upper Plate" and "Lower Plate" rocks on the north side of Mitchell Creek.

18.12.3 GEOTECHNICAL DOMAINS

For the purpose of PEA level slope designs, BGC has divided the Mitchell deposit area into five geotechnical domains based on the structural geologic fabric and alteration of the rock mass. Design discontinuity sets (Appendix D) have been determined for each domain. The domain boundaries and locations are shown on Figure 18.42. Boundaries are based on regional faults or have been inferred based on natural topographic features which divide the study area.

Figure 18.42 The Geotechnical Domain Boundaries and Locations



Structural geology data for the KSM project site used for amended PEA level design has come from multiple sources:

- data collected by BGC during our 2008 site visit,
- data collected by the Mineral Deposits Research Unit (MDRU) of the University of British Columbia and Seabridge provided by Mike Savell, P.Geo., and
- Ph.D. level research by J. Margolis (1993).

Geological discontinuity observations include: joints, faults, bedding, and foliation. BGC sorted these data by area and developed equal-area stereographic projections to determine domains of homogeneous geological structural fabric which are appropriate for use at a PEA level of study

Domain I includes the hanging wall of the Sulphurets fault and covers the area west of an unnamed normal fault SW of the Mitchell deposit (Drawing 1). The rocks of this domain are assumed to be unaltered for the most part; the rocks have been correlated with the Stuhini Group. This domain may represent rocks west of the McTagg Antiform. The mapped foliation dips steeply (>60^o) to the east; bedding dips steeply to the west, as shown in Table 1.

Domains IIA and IIB include the rocks in the footwalls of the Mitchell and Sulphurets faults, respectively. The domains are limited to the south by the drainage divide of the Mitchell-Sulphurets ridge and to the east by the Brucejack fault. This domain may represent the hinge zone of the McTagg Antiform as the bedding data for the domain appears to be representative of regional folding. The rocks of Domain IIA and IIB are most likely part of the Hazleton Group. Domain IIA includes the footwall of the Mitchell fault and extends below the valley floor to the limit of the proposed mining. The rock mass of Domain IIA is intensely hydrothermally altered over most of its extent; corresponding to the previously defined “Group A” alteration (BGC, 2008). The rock mass of this domain is expected to be weak. Domain IIB includes the “Group B” (BGC, 2008) altered rock mass of the hanging wall of the Mitchell fault. This rock mass is expected to be more competent and stronger than the rock mass associated with “Group A” alteration.

The sub-division of the structurally homogeneous Domain II into sub-domains based on alteration type is supported by previous work. Work completed by Margolis (1991) notes that the protoliths of rocks in the intensely altered zones of the deposit area cannot be determined; this ambiguity is less apparent above the Mitchell fault. Resource Modelling Inc. (2008) notes:

“Above the Mitchell fault, alteration is mainly confined to siliceous hornfelsed zones adjacent to porphyritic Monzonite and granitic Mitchell intrusions.”

The foliation in these domains dips steeply (>60^o) to the north; bedding is folded, dipping from the northeast to the northwest. As the discontinuity data for these domains are identical, the kinematic stability analysis results are the same. However, the achievable angles and slope heights will differ between the sub-domains due to the differing rock mass strength of the Group A and Group B rocks.

Domain III includes rocks east of the Brucejack fault; it is not expected to occur in the Mitchell pit. The foliation in Domain III is typically steep (>60°) and dips to the NNW. Bedding in this domain may be affected by regional folding, dipping moderately (between 30^o and 60°) from the northeast to the northwest. The character of the rock mass of this domain is currently unknown. Discontinuity data (Table 4) is based on the faults of the study area and mapping along the eastern part of the Mitchell-Sulphurets ridge.

Domain IV includes the area south of the Mitchell-Sulphurets ridge drainage divide and east of the un-named normal fault. The currently proposed Mitchell pit does not intersect this domain. However, the proposed Sulphurets pit, which is not included in BGC's current scope of work, is in this domain. The mapped fault sets of the study area are assumed to occur in this domain. The foliation noted in this domain is also typically steep (>60°) and dips to the north-northwest; bedding in this domain appears to be affected by regional folding, dipping steeply from the east-northeast to north-northwest.

18.12.4 *Open Pit Design Criteria*

Open pit slopes can be divided into three scales: bench, interramp, and overall. Each scale requires a design and these designs are combined to develop the final open pit slope profile. Slope angles, catch benches, ramps and allowances for dewatering berms (if required) must all be considered in determining the final slope profile. The slope design criteria are selected to limit the potential for failures which would lead to lost production or lost resources. Potential bench and interramp scale instabilities may be determined by combinations of minor and intermediate scale geological discontinuities. Overall slope stability is generally dictated by major geological discontinuities, persistent rock mass fabrics and rock mass quality.

Designs are completed for each geotechnical domain; the domains are divided into design sectors, where appropriate. Blending of the designs between sectors should always be completed in the sector with the steepest overall slope angles. Geotechnical and regulatory design constraints (Table 5) have been considered in determining the PEA design criteria. The design process for each slope scale is discussed in the following sections, with a summary of the PEA level design criteria for three cases.

Table 18.30 PEA Level Pit Slope Design Constraints

Constraint	Value	Source
Design Factor of Safety (FOS) – Discontinuity Controlled Stability	1.2	BGC
Design Factor of Safety (FOS) – Rock Mass Controlled Stability	1.2	BGC
Single Bench Height	15 m	Seabridge
Minimum Catch Bench Width	8 m	B.C. Mines Act 6.23.2
Ramp Width	32 m	Seabridge

18.12.5 *Design Criteria Cases*

For the PEA Addendum 2009, BGC has developed open pit slope design criteria for three cases based on the available geotechnical data and engineering assumptions.

These assumptions will require varying level of efforts to validate at future stages of study.

Case A (designated “Conservative” case) assumes that the structures compiled from surface mapping in each of the domains will extend to depth and are fully continuous, with no large scale roughness or cohesion along gouge infilled faults. Bedding and foliation planes are assumed to be at residual shear strength; with assumed friction angles of 26^o and 30^o, respectively. A slope geometry based on single benches is assumed for Case A. Double benching may require site specific experience with the rock and site conditions before it could be attempted. The criteria for this case are based on the available, but limited, geotechnical data and assumed shear strengths that represent the low to middle range of strengths expected in porphyry type deposits. Slope depressurization through vertical wells and horizontal drains has been assumed to be effective. It is also assumed that controlled blasting will be carried out. The pre-feasibility level geotechnical investigations are expected to further support these preliminary assumptions.

Case B (designated “Base” case) assumes favourable mining conditions and geological structure allowing a double bench slope configuration (Bh = 30 m) to be used for all walls of the proposed pit. This case assumes that fault and bedding/foliation structures are continuous with shear strengths represented by a friction angle of 35^o. The assumed friction angle is plausible considering the possible effects of:

- large scale roughness or waviness on the discontinuity planes,
- partially healed bedding or foliation planes,
- fault infillings which are thin or do not have a high clay content.

To validate these preliminary shear strength assumptions at the pre-feasibility level of study, geotechnical drilling, photogrammetric mapping, and laboratory testing will be required. Slope depressurization through vertical wells and horizontal drains is assumed to be effective in this case. Controlled blasting is also assumed.

Case C (designated “Optimistic” case) assumes that mining conditions are favourable for a double bench slope configuration. Discontinuity sets interpreted for each of the preliminary geotechnical domains are assumed to be limited in continuity (persistence) and / or density. Achievable interramp and overall slope angles in Case C are therefore determined by the rock mass strength of the final wall rock of the proposed pit. This case is based on optimistic assumptions with a low likelihood of validation for all domains or design sectors at the pre-feasibility level of study. Slope depressurization through vertical wells and horizontal drains is assumed to be effective in this case. Controlled blasting is also assumed.

18.12.6 Bench Scale Design

The required bench scale design criteria include:

- Bench height (Bh)
- Bench face angle (Ba)
- Catch bench width (Bw).

Seabridge has selected a 15 m bench height (Bh) for mining the Mitchell deposit; a double bench configuration (Bh = 30 m) has been utilized by MMTS in designs completed to date. A single bench is assumed in Case A of the current work; double benching is assumed for Cases B and C. The practicality of double benching has to be confirmed with more detailed bench designs conducted at the pre-feasibility level of study.

From industry experience a 65^{o} bench face angle (Ba) can typically be achieved in porphyry deposits using traditional production drill and blast methods. This is assumed to be applicable to the Mitchell deposit at this preliminary stage of design.

The minimum design catch bench width must satisfy both regulatory and geotechnical requirements. Geotechnical requirements include catchment of discrete rocks (Ryan and Pryor, 2000) and retention of bench scale failures. Additional width may be needed to allow for long term crest break-back over the life of the bench. Based on geotechnical criteria, the minium recommended bench width for a single bench is 9.5 m; for a double bench slope height, the minimum recommended bench width is 12 m. The recommended bench widths include a 1.5 m allowance for back-break of the bench crest. Wider benches may be required due to the geometry of the interramp scale slopes or at the discretion of the mining engineer.

18.12.7 INTERRAMP SCALE DESIGN

The interramp slope scale represents an intermediate scale of slope between an individual bench and the overall pit slopes. Design criteria required for this scale of slope are:

- Interramp angle (Ia)
- Interramp height (Ih).

The interramp design criteria are determined by a combination of geometric factors related to the bench configuration and the geotechnical slope stability criteria determined from slope stability analyses. Maximum possible interramp angles are controlled by the bench geometry; where adverse geological structures (faults, foliation, bedding, etc.) exist the achievable interramp angles will be reduced by these potential failures.

To determine the interramp angle which will result in a low likelihood for instability, BGC has analyzed potential wedge and plane shear failures based on the mean orientations of fault, bedding, and foliation sets for each domain for the “conservative” and “base” cases. Shear strengths are varied according to the assumptions of the analysis case. The results of the analyses are presented in Appendix D.

The maximum interramp height determines the number of benches which can be "stacked" before a wide berm or ramp is required, to de-couple segments of the overall pit slope and limit the size of any potential interramp scale failures. Controls on the interramp height for the Mitchell deposit are based on the stability of slopes due to rockmass strength of the units which make up the final open pit walls.

18.12.8 *Generic Rock Mass Slope Stability*

Slope instability controlled by rock mass quality and strength has been considered through generic analyses based on models consisting of one, two, and three layers. The unit divisions within the generic models are based on the intersection of the domain boundaries with the current topography and the PEA pit shell provided by MMTS. The one layer model generally applies to the proposed east and west walls of the Mitchell pit. The two layer model applies to the south wall of the Mitchell pit and the three layer model applies to the proposed north wall of the Mitchell pit. The boundaries between the layers have been assumed to be horizontal, to simplify the analyses.

Analyses of potential non-linear failure surfaces were carried out for slopes ranging from 500 m to 1600 m in height at angles from 30^o to 50^o to develop slope height vs. slope angle relationships for various factors of safety in each geotechnical domain and design sector. Slope heights vs. slope angle graphs have been developed for use in the PEA level slope design (Appendix D). The results of these analyses are used to determine achievable angles for interramp scale slopes in Case C and overall scale slopes for all analysis cases.

Hydrogeologic conditions for the Mitchell pit are not currently known. At this stage of design, the rock mass stability analyses have been conducted assuming some residual pore pressure, equivalent to a pore pressure co-efficient (Ru) of 0.09 and representing partially (25%) saturated conditions. These pore pressure estimates need to be confirmed before a high degree of confidence can be gained in the results of the overall and interramp stability analyses.

18.12.9 *Overall Slope Scale Design*

The three design scales have been integrated to develop an overall slope design profile. The overall angles presented in the PEA level design criteria represent that maximum angle for our current understanding of the rock mass strengths, structural fabric, interramp geometry, and bench geometry of the proposed Mitchell pit slopes.

18.12.10 PEA ADDENDUM SLOPE DESIGN CRITERIA

PEA Addendum open pit design criteria developed for the Mitchell pit of the KSM project are presented below by design sector and domain for the "base case" (Table 18.31). This case, as well as the other two cases (i.e. "conservative" and "optimistic" have assumed that depressurization of the slopes will be carried out and that partial pore pressures will remain.

Design sectors are defined by ranges of slope azimuths and have been kept constant between analysis cases. The slope azimuth of the pit wall refers to the compass direction one would face if standing at the bottom of the pit, looking toward the wall in question.

Overall slope, interramp, and bench scale criteria are included for each design sector. The maximum overall angles (Oa) presented only account for those ramps required by the interramp geometry, not any that may be needed for access. Further reduction in the Oa may be required to account for access. The overall slope heights are based on pit shells provided to BGC by Seabridge and MMTS; these heights are provided for reference only. The interramp height limit (Ih) should not be exceeded by a continuous stack of benches. Once the Ih is reached a wide bench or ramp is required to break the wall into segments. The presented bench widths (Bw) are minimum widths required to meet the PEA open pit geotechnical requirements.

Table 18.31 PEA Addendum Slope Design Criteria – "Base" Case (Case B)

Domain	Design Sector	Slope Azimuth Start (°)	Slope Azimuth End (°)	Bh[1] (m)	Ba[1] (°)	Bw[1] (m)	Ia limit from kinematic analyses[2] (°)	Ia limit from rock mass analyses[3] (°)	Ia Control	Ia[1] (°)	Ih[3] limit (m)	Oh[4] (m)	Oa Limit from generic stability analyses[3] (°)	Oa[1] (°)
I	I-186	180	192	30	65	12	50.0	50.0	rockmass	49	1200	1200	40.0	40
	I-197	192	201	30	65	12	50.0	50.0	rockmass	49	1200	1200	40.0	40
	I-318	201	075	30	65	12	50.0	50.0	rockmass	49	1200	1600	40.0	40
	I-080	075	085	30	65	12	50.0	50.0	rockmass	49	1200	600	50.0	49
	I-090	085	095	30	65	12	50.0	50.0	rockmass	49	1200	600	50.0	49
	I-103	095	110	30	65	12	50.0	50.0	rockmass	49	1200	600	50.0	49
	I-140	110	170	30	65	12	50.0	50.0	rockmass	49	1200	1200	40.0	40
	I-175	170	180	30	65	12	50.0	50.0	rockmass	49	1200	1200	40.0	40
IIA	IIA-188	180	195	30	65	22	40.0	45.0	BDc - N-FT	39	840	1200	40.0	39
	IIA-228	195	260	30	65	22	40.0	45.0	BDc - 226-FT	39	840	1200	40.0	39
	IIA-265	260	270	30	65	17	46.0	45.0	BDc - 226-FT	44	540	600	42.5	42
	IIA-338	270	045	30	65	17	50.0	45.0	rockmass	44	540	1600	40.0	40
	IIA-053	045	060	30	65	17	50.0	45.0	rockmass	44	540	1600	40.0	40
	IIA-080	060	100	30	65	24	39.0	45.0	FO - Bda	38	1000	600	42.5	38
	IIA-135	100	170	30	65	25	38.0	45.0	FO - Bda	37	1100	1200	40.0	37
	IIA-175	170	180	30	65	24	39.0	45.0	BDc - N-FT	38	1000	1200	40.0	38
IIB	IIB-188	180	195	30	65	22	40.0	50.0	BDc - N-FT	39	1200	1200	40.0	40
	IIB-228	195	260	30	65	22	40.0	50.0	BDc - 226-FT	39	1200	700	40.0	40
	IIB-265	260	270	30	65	15	46.0	50.0	BDc - 226-FT	45	1200	600	50.0	46
	IIB-338	270	045	30	65	12	50.0	50.0	rockmass	49	1200	1600	40.0	40
	IIB-053	045	060	30	65	12	50.0	50.0	rockmass	49	1200	1600	40.0	40
	IIB-080	060	100	30	65	24	39.0	50.0	FO - Bda	38	1200	600	50.0	38
	IIB-135	100	170	30	65	25	38.0	50.0	FO - Bda	37	1200	1200	40.0	38
	IIB-175	170	180	30	65	24	39.0	50.0	BDc - N-FT	38	1200	1200	40.0	38
III	III-190	150	230	30	65	17	50.0	45.0	rockmass	44	540	600	42.5	42
	III-243	230	255	30	65	17	50.0	45.0	rockmass	44	540	600	42.5	42
	III-260	255	265	30	65	17	50.0	45.0	rockmass	44	540	600	42.5	42

	III-350	265	075	30	65	17	50.0	45.0	rockmass	44	540	1200	36.5	36
	III-080	075	085	30	65	17	50.0	45.0	rockmass	44	540	600	42.5	42
	III-090	085	095	30	65	17	50.0	45.0	rockmass	44	540	600	42.5	42
	III-103	095	110	30	65	17	50.0	45.0	rockmass	44	540	1200	36.5	36
	III-130	110	150	30	65	17	50.0	45.0	rockmass	44	540	1200	36.5	36
IV	IV-185	175	195	30	65	26	37.0	45.0	BJS-FT - Bda	36	1300	600	42.5	37
	IV-220	195	245	30	65	25	38.0	45.0	BJS-FT - Bda	37	1170	600	42.5	38
	IV-250	245	255	30	65	17	50.0	45.0	rockmass	44	540	600	42.5	42
	IV-342	255	068	30	65	17	50.0	45.0	rockmass	44	540	1200	36.5	36
	IV-074	068	080	30	65	17	50.0	45.0	rockmass	44	540	600	42.5	42
	IV-090	080	100	30	65	17	50.0	45.0	rockmass	44	540	600	42.5	42
	IV-118	100	135	30	65	18	44.0	45.0	N-FT - BDa	43	570	600	42.5	42
	IV-155	135	175	30	65	25	38.0	45.0	N-FT - BDa	37	1170	1200	36.5	36

Notes:

1. See text for determination of bench height (Bh), bench face angle (Ba), catch bench width (Bw), interramp angle (Ia), and overall slope angle (Oa).
2. Results of kinematic analyses can be found in Appendix B.
3. The interramp height (Ih) may be limited in some sectors by rock mass quality. The results of both generic and two layer rock mass stability analyses can be found in Appendix C.
4. The maximum overall height (Oh) for each sector has been estimated based on the economic pit shell provided by Seabridge and MMTS.

18.12.11 Discussion

Open pit slope design criteria presented at the current stage of study should be considered to be preliminary estimates as they are based on relatively limited data. A wider range of rock mass properties is reflected in the current design criteria compared to those previously utilized by Seabridge. In addition, the updated criteria consider the potential for discontinuity controlled instability in the proposed pit walls of the Mitchell pit. The reliability of all criteria recommended to date, including those of the current work, for the proposed Mitchell pit are limited by a lack of sub-surface discontinuity data and hydrogeological characterization of the pit area.

The cases developed indicate a range of possible slope angles for various assumptions given the available data. The overall slope angles recommended by the current work are in some sectors and cases less than those previously used by Seabridge; however, the recommended angles also reflect additional data and higher proposed slopes than previously considered by BGC.

18.12.12 Review of MMTS Pit Design

BGC has not reviewed the final open pit designs provided by MMTS as part of this PEA Addendum report and therefore cannot validate that the proposed "base case" pit meets the criteria outlined by BGC. BGC did review and utilize the PEA Addendum pit to assist in optimizing the pre-feasibility study geotechnical drilling program; however, it should be pointed out that the preliminary economic pits provided did not include ramps or dewatering benches which were recommended to limit interramp slope heights.

18.13 Conclusions and Recommendations

A PEA Addendum mining design and mine production schedule is presented in this document for KSM. The economic mining limits and capital and operating cost estimates that result are based on the criteria as specified. In certain instances, reasonable assumptions have been made for typical conditions for the area.

To improve the resource classification, exploration and drilling information for 2009 should be added to the drill hole database and the resource model rebuilt for subsequent studies.

Additional geotechnical studies, consisting of the following, should be carried out for future studies:

- refinement of geotechnical units
- compilation of structural discontinuity information and sorting of discontinuity information into preliminary structural domains

- kinematic stability analyses of potential structurally controlled failures
- assessment of the potential for rock mass failure
- definition of a comprehensive geotechnical program to collect strategic geomechanical and structural geologic information from key areas within the proposed open pit.

The results of these studies should be utilized to conduct further mine planning exercises and the results used to update a Pre-Feasibility Study following development of a new mine plan. For future studies, it is recommended to:

- quantify the distribution and magnitude of pore water pressures of the proposed slopes and develop operations design parameters for pit dewatering
- review mine planning considerations that could improve the stability of the proposed open pit slopes (controlled blasting) or optimize the slope geometry (compound slope angles).

A detailed hydrogeology evaluation of the area is needed to improve the accuracy of pit dewatering design. Vertical dewatering wells have not been included as part of the required PEA activities. Results from field programs may show this to be a future requirement in order to lower the water table within the pit prior to mining.

Combined with the hydrogeology evaluation, a hydrology assessment is needed so that the diversion and water management plan can be developed for the mining area considering the combined surface and groundwater quantities.

19.0 GEOTECHNICAL

19.1 TAILING MANAGEMENT FACILITY

Sections 19.1, 19.2, and 19.6 were prepared by Graham Parkinson (P.Geo.) and Harvey McLeod (P.Eng) of KCBL.

19.1.1 SUMMARY

The proposed TMF is located in a valley between Treaty Creek and Teigen Creek, approximately 24 km east of the mine site.

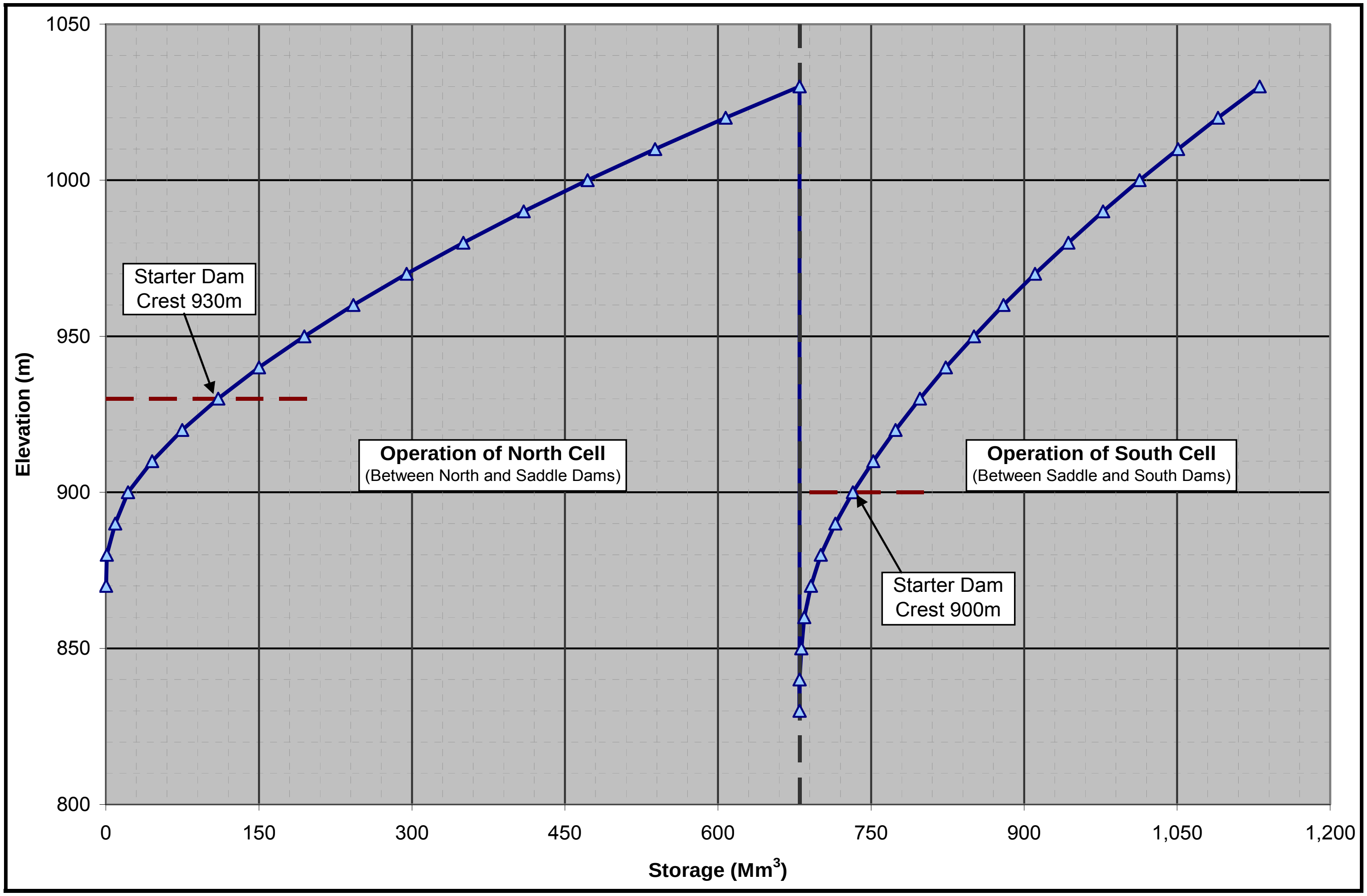
The TMF is divided into two cells that will be constructed sequentially. The North Cell between the North Dam and the Saddle Dam will be built first and will operate for approximately 20 years, at which point it will be closed and operations will shift to the South Cell with the construction of the South Dam. (An alternative has also been developed, which would allow storage of all tailings in the North Cell with dams up to elevation 1065 m, which would then provide the South Cell for future storage of tailings for an expanded mine case. The pond filling curve for the alternative is included in Appendix F).

The total storage capacity of the TMF at elevation 1030 m is 1.1 billion cubic metres, which is sufficient to store all tailing from the project at dam heights of approximately 160 m at the North Dam and 200 m at the South Dam. The storage elevation curve for the TMF is shown in Figure 19.1. In the event of expansion of the mineral resource, additional storage potentially be obtained by increasing the height of the dams.

Tailing will be pumped from the mill at a rate of approximately 120,000 t/d for the 30-year mine life. The tailing will be pumped at 34% solids and is estimated to have an average settled density of 1.3 t/m^3 in the TMF.

The TMF will be operated, as far as practical, as a closed system with excess water recycled back to the processing circuits. Diversions channels will direct uphill runoff around the TMF.

Figure 19.1 Storage-Elevation Curve for the KSM TMF



19.1.2 SITE CONDITIONS

CLIMATE

Assumed climate data for the KSM Project are shown in Table 19.1. These data are based on regional climate models, climate and hydrology data available from site climate stations at the mine and tailing areas, and correlations with long-term regional climate stations. The TMF is 24 km inland from the mine site, located behind a large glaciated range of mountains with peaks up to 2200 m elevation, and is therefore expected to have less annual precipitation than at the mine site.

Table 19.1 Climate Data for the KSM Project

Month	Average Monthly Temp (°C)	Assumed Mine Site Avg. Monthly Precipitation (mm) [1]	Mine Site Percent of Annual Runoff [3]	Assumed TMF Avg Monthly Precipitation (mm) [2]	TMF Percent of Annual Runoff	Avg Monthly Evaporation (mm)
Jan	-8.4	205	2%	143	2%	0
Feb	-5.8	169	2%	118	2%	0
Mar	-4.0	135	2%	94	2%	0
Apr	0.6	75	2%	52	2%	0
May	4.2	74	8%	52	20%	70
Jun	8.1	54	12%	38	24%	70
Jul	10.4	66	20%	46	21%	70
Aug.	10.5	114	25%	79	12%	60
Sep	5.8	179	12%	125	6%	30
Oct	0.8	194	9%	135	5%	0
Nov	-4.5	174	3%	121	4%	0
Dec	-6.9	208	2%	145	2%	0
Total		**1,650**	**100%**	**1,150**	**100%**	**300**

[1] data based on limited observed site data at the Sulphurets station, 16.5 km northeast of the mine site, and distribution interpolated from Eskay Creek station.
[2] annual precipitation assumed to be 1,150 mm based on data from a limited period of overlap between the station at Teigen Creek and the station at Sulphurets.
[3] runoff distribution from Rescan 2008 Baseline Studies Report, March 2009, Chapter 5.

FOUNDATION CONDITIONS

Based on preliminary surficial and drill hole site investigations, foundation conditions at the North and South dam sites appear to be favourable for dam construction with generally shallow (0 to 10 m below surface) bedrock on the dams' west abutments and moderately thick (20 m to 50 m) overburden (till with some colluvial and alluvial debris) on the east abutments. Site investigations to date suggest that the local bedrock under the entire TMF area is low permeability, hard metamorphic sandstones and siltstones of the Bowser Lake Group with steeply dipping beds.

Overburden foundation conditions in the Saddle Dam area are potentially less favourable as the wide, flat valley bottom is covered with an alluvial debris fan and localized swampy areas. Conditions at the Saddle Dam site may require some foundation improvements that are not yet costed. Geotechnical drilling at each of the tailing dam sites was completed in September 2008. Foundation analysis and PFS-level design engineering is underway.

19.1.3 TAILING DAMS

DAM DESIGN

All three dams will be built with starter dams of local borrow materials. . Dams will be raised with a central low permeability core zone and a downstream cycloned sand zone using the centreline construction method. The dams will reach a maximum height of up to 220 m. .

Drawings showing the layout of the TMF are included in Appendix F as Drawings D-1002 and D-1003. The starter crest elevation for the North and Saddle dams will be 930 m, while the South Dam starter crest will be 900 m. All three dams can be built to an ultimate crest elevation of 1065 m, with annual raises expected to range between approximately 5 m and 15 m.

Starter dams will be homogenous earthfill dams, constructed with local borrow or tunnel waste rock (North Dam) with 2.5H:1V upstream and downstream slopes. The glacial till upstream facing (core) will be keyed-in to a trench excavated in bedrock or other impermeable substrate. A 1 m-thick blanket drain will be laid beneath the downstream slopes of the dam. Fill is locally available in the form of colluvial deposits (debris cones) and till deposits. Filters and drain material are assumed to be primarily sourced from local sand and gravel deposits but may require processing

The low permeability dam core will be constructed in the centreline raise above the starter dam with compacted glacial till placed with 1.5H:1V slopes on both upstream and downstream. The main dam fills for the raises will use mechanically compacted, non acid generating cyclone sand with a downstream slope of 3H:1V. Hydraulically compacted cyclone sand will be placed on the upstream slopes of the dam, overlying the tailing beach. This will allow subsequent core placement to occur on compact cyclone sand from the previous raise rather than loose tailing. This reduces the risk of damage to the core due to consolidation of the tailing immediately upstream of the dam.

DAM CONSTRUCTION

Table 19.2 summarizes the material requirements for the dams. For construction of the starter dams, general fill and core material will be excavated from local borrow sources (<2 km haul distance) that have been identified at each dam site. Starter dam construction and subsequent dam raises of the glacial till core have been costed assuming rental equipment rates.

Table 19.2 Material Requirements for Dam Construction

Dam	Starter Dam (M m^3)			Ultimate Dam (M m^3)		
	General Fill	Core	Total	Cyclone Sand	Core	Total
North Dam	3.0	1.0	4.0	23.3	2.1	28.4
Saddle Dam	0.9	0.4	1.3	26.3	1.7	29.0
South Dam	4.3	1.2	5.5	40.0	2.8	47.0
Total	**8.3**	**2.6**	**10.8**	**89.6**	**6.6**	**104.4**

Cyclone sand placement on downstream slopes for annual dam raises will occur over a six month period from mid-April to mid-October. During this time, tailing will be pumped from the mill and pass through a primary cyclone station (PCS) located above the west abutment of the North Dam. The fine PCS cyclone overflow will be spigotted into the TMF, and the coarse cyclone underflow will be piped to skid-mounted secondary cyclone stations (SCS) on the dam crests. The SCS underflow will be <15% fines (<75 µm) and will be pumped to cells on the downstream face of the dam and compacted with bulldozers, while the overflow will be spigotted into the TMF to form a beach against the dam. During the remaining six months of the year, whole tailing will bypass the cyclones and be spigotted directly into the TMF.

19.1.4 WATER MANAGEMENT

The TMF will be operated, as far as practical, as a closed system with excess water recycled back to the plantsite. The TMF will provide storage for a minimum of one-month mill water supply, seasonal water storage and flood storage.

Diversion channels will be constructed to divert, as far as practical, runoff away from the impoundment. The diversion channels will be constructed as wide channels to permit cleaning of snow avalanche materials.

Flows in the diversions will be directed seasonally either away from or towards the impoundments to keep the water balance close to that of a zero discharge facility. A pump barge will recycle water back to the processing circuits, and, if required, pump any excess water to the crest of the North Dam where it will be piped down past the North Seepage Collection Dam and released into the Teigen Creek tributary. TMF discharge would only be to the Canadian receiving waters of the Teigen Creek drainage. Any water releases required to balance annual storage will be timed with precipitation and creek flows and is assumed (based on work by others) to meet Metal Mining Effluent Regulations (MMER) as well as other BC and Canadian water quality standards.

A diversion dam will be constructed on the major creek located east of the Saddle Dam to route water from this 11.9 km^2 catchment around the North Cell of the TMF. Diverted water will be transported along the northeast side of the TMF in a 6.25 km long, -0.5% grade ditch with a base width of 4 m, sufficient to allow passage of heavy snow clearing equipment. The diversion ditch itself diverts an additional 3.17 km^2 catchment area of the slope above the ditch. Two 500 m sections of this diversion route have high avalanche risk and will consist of a buried covered channel with a 14.5 m^2 cross sectional area. The East Diversion Dam is designed to divert the maximum expected monthly base flow, with any storm events being routed through a spillway into the TMF. An additional diversion along the access road on the west side of the impoundment will collect water from the 11.2 km^2 catchment on the western side of the North Cell and route it around the TMF to the north.

When the South Cell is in operation, a diversion channel for the south-western slope area (2.93 km^2 catchment) will be constructed along the access road to the South Dam. The South Cell has an annual net water loss during average climatic conditions but this will be compensated for by routing a portion of the water from the diverted North Cell catchments into the South Cell. All diversion channels will be large enough to allow access to snow clearing equipment for cleanout before the freshet commences.

Seepage collection dams will be built 200 m downstream of the ultimate toe of each dam. These dams will collect seepage and flows from the drainage of the cyclone sand deposition. Pumps will return this water back into the TMF.

The TMF is designed to store the probable maximum 30-day flood (PMF) [1], with 1 m of additional freeboard to account for wind and wave action as well as other variations in water surface elevation.

19.1.5 CLOSURE

The TMF will be closed as a “dry” structure with minimal pond/wetland area and re-vegetated with grasses and trees. Any acid generating tailings will have been stored at depth, or covered with non-acid generating tailing and will remain saturated. Surface drainage within the impoundment will be directed towards a closure spillway, excavated in rock at the west abutment of the North Dam. The closure spillway is designed to route the TMF. The final landform will be configured to reduce the risk of concentrated surface water flows and erosion.

[1] The PMF has no associated Annual Exceedance Probability (AEP) and extrapolation of flood statistics beyond 1/1000 year is not recommended.

19.2 ROCK DUMPS

The 2.5 billion tonnes of rock ultimately expected to be produced from the project by stripping overburden from the ore will be disposed of in dumps located in the Mitchell, McTagg, and Sulphurets valleys.

The initial rock dump will be located within Mitchell Valley, adjacent to Mitchell Pit. Additional expansion of this dump is possible in McTagg Valley.

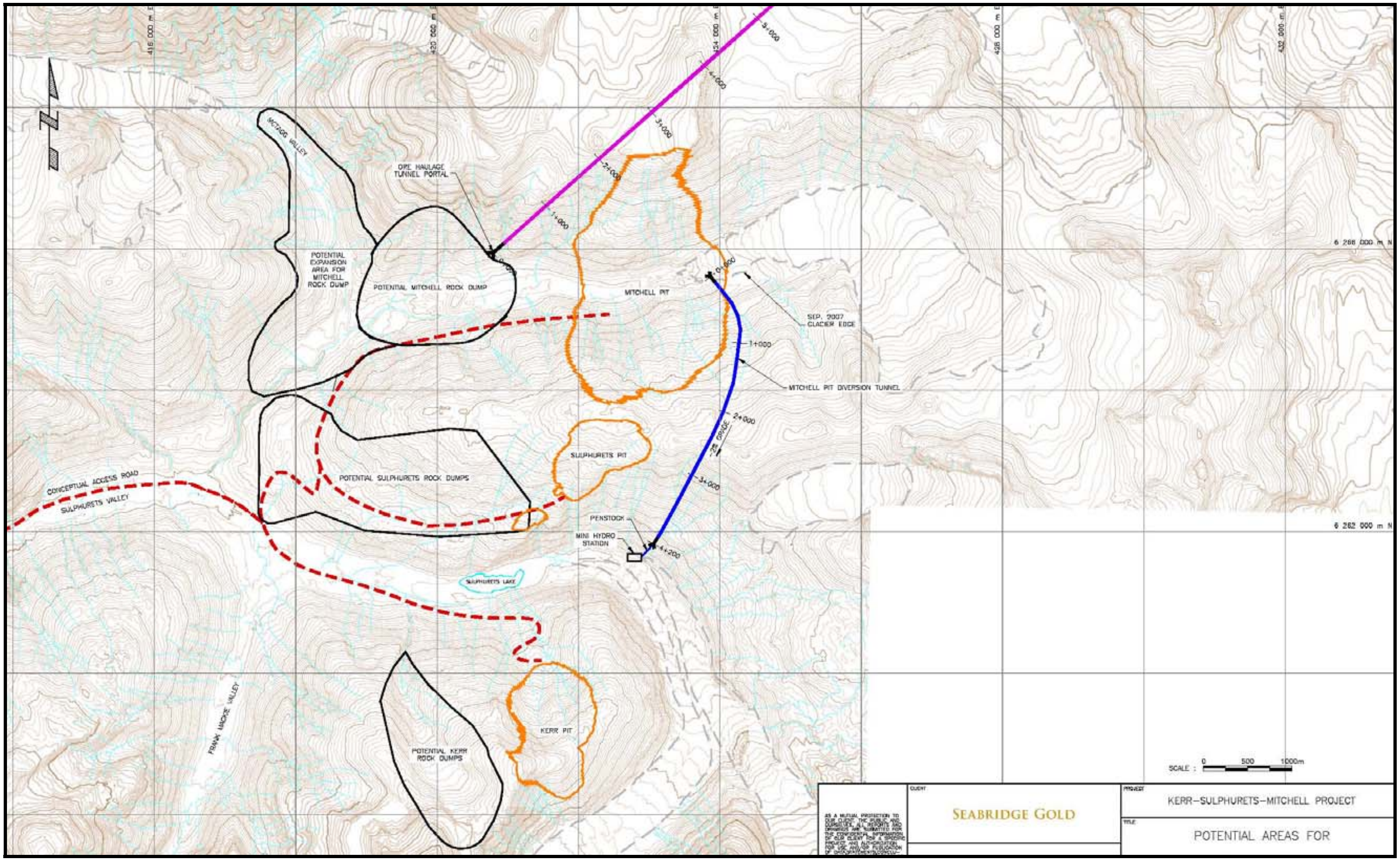
Rock stripped from the Sulphurets pit can be accommodated in an adjacent sidehill dump located on the ridge to the west of the pit. A high level dump site is available adjacent to the west side of Kerr pit but crushing and transporting Kerr pit waste rock to other disposal sites is also being considered.

Sites for the rock dumps have been selected to allow for the collection of dump drainage for treatment, if required, to meet water discharge criteria. During September 2008, KCBL completed three drill holes in the Mitchell rock dump area and conducted packer and falling head testing. These drill holes suggest that low permeability bedrock underlies a thick sequence of glacial till in the Mitchell rock dump foundation area. Based on this and the fact that bedrock topography confines flow to several locations where collection is feasible, investigations are currently focussing on the suitability of collecting dump leachate directly on bedrock.

Water management for the rock dumps will include measures to reduce water contact with PAG rock. Options being examined include basal drains of NPAG rock, low permeability covers on top of the dumps, and tunnels to divert creek flows around the dumps during operations. After closure, surface diversions and spillways will route water around the dumps. All diversion channels have been designed to be large enough to allow access to snow clearing equipment.

The current reclamation cover concept is a 1 m-thick low permeability till layer to reduce infiltration, overlain with a 3 m layer of NPAG rock to protect the till from erosion and freeze-thaw damage. The dump designs include placement strategies to reduce infiltration. These combine bottom up constructed shells of more compact trafficked rock layers (stripped from lower elevations in the pit) that will be placed to encapsulate looser, high dumped rock stripped and dumped from higher elevations above the valley. Designs for the rock dumps allow for progressive covering and reclamation during mining operations.

Figure 19.2 Potential Areas Investigated for Rock Dump Locations



19.3 MITCHELL VALLEY WATER TREATMENT

Geochemical testing is currently underway on rock samples to quantify the estimates of ARD leachate water quality. Based on preliminary assumptions and data from similar projects a conceptual assessments of the range of water quality was used to examine possible water treatment costs.

To manage seasonal and storm event flows resulting from peak melt periods during freshet, a water treatment storage pond will be constructed downstream of the Mitchell rock dump. A water balance for the mine area incorporating the separation of clean and impacted flows from the rock dumps, pit dewatering, ore haul tunnel seepage, and upstream catchments estimated that the average flow requiring treatment could be up to 1.2 m^3/s. To allow this average treatment rate, a water treatment pond with an ultimate capacity of 11 M m^3 is required to attenuate the runoff from the annual freshet and to handle a coincident 3-day, 25-year storm event.

At this stage of the project, insufficient data is available to predict actual water quality of impacted flows and thus treatment requirements. However, it is expected that water treatment will be a significant cost so it is recommended that an allowance of Cdn$50 M in capital costs and Cdn$20 M in annual operation costs be included in the project economic analysis. The water treatment system will be designed to meet applicable water quality standards.

19.4 MITCHELL DIVERSION HYDROELECTRIC PLANT

Section 19.4 was prepared by Robin Fitzgerald (P.Eng.) of KCBL with data from Neil Brazier (October 29, 2008 Report “Upper Sulphurets Creek Water Development Plan for Mini-Hydroelectric Project”). Power potential has been scaled to reflect heads available now that inlet locations have been surveyed.

Diversion of water from Mitchell Valley will create potential for the generation of hydroelectric power by installation of a penstock at the outlet of the Mitchell diversion tunnel. Approximately 100 m head will be available to a powerhouse located in Sulphurets Valley below the tunnel outlet, and will allow the generation of up to several megawatts of power. A study based on an assumed power requirement of 1 MW to power the water treatment plant showed that sufficient flow is available to produce power for the treatment plant for 5 months of the year, and offset diesel or other power sources during the remainder of the year.

Table 19.3 illustrates the monthly power production estimate for an installed plant of one 500 kW unit and one 2,500 kW unit.

Table 19.3 Monthly Flows and Power Produced by Two Turbine Installation (0.5 MW and 2.5 MW)

	Month											
	J	F	M	A	M	J	J	A	S	O	N	D
Flow (cms)	0.30	0.26	0.23	0.24	1.09	1.60	2.68	3.34	1.67	1.16	0.44	0.30
Hours/month	744	672	744	720	744	720	744	744	720	744	720	744
Avg. kW Available	248	213	193	199	909	1,337	2,231	2,781	1,394	964	370	248
kW Generated	248	213	193	199	909	1,337	2,231	2,500	1,394	964	370	248
Monthly MW.h	184	143	143	143	676	963	1,660	1,860	1,004	717	266	184
TOTAL ANNUAL MW.h OF GENERATION = 7,945							**CAPACITY FACTOR = 0.49**					

NOTE: Based on: **Net Head** = 100 m **Eff.** = 0.85
Max. Turbine Generator Output = 3,000 kW (sum large synchronous machine and small induction generator)

19.5 TUNNEL GEOTECHNICAL

Geotechnical aspects of Section 19.5 were prepared by Garry Stevenson (P.Eng.) of KCBL. Cross-sectional designs of the ore haulage tunnel and tunnel costs were prepared by Thyssen Mining Construction Ltd. (Thyssen). Tunnel infrastructure layouts were designed by Harold Bosche of BVL. Geological mapping of the tunnel route, assessment of route alternatives, and geotechnical assessment were conducted by Shane Warner (E.I.T.) and Graham Parkinson (P.Geo.), both of KCBL. Hydrology of the diversion tunnel was prepared by KCBL under the direction of Graham Parkinson (P.Geo.) and Harvey McLeod (P.Eng.) of KCBL.

19.5.1 SUMMARY

The KSM Project involves the construction of three major tunnels:

- The Mitchell-Teigen tunnels, consisting of a 16.3 km twin tunnel from Mitchell Pit to the West Saddle Portal and a 7.2 km twin tunnel from the East Saddle Portal to the mill
- a 4.2 km Mitchell diversion tunnel through Sulphurets ridge between the Mitchell and Sulphurets valleys
- at later mine stages (>10 years), a 3.4 km McTagg diversion tunnel or high level diversion channel.

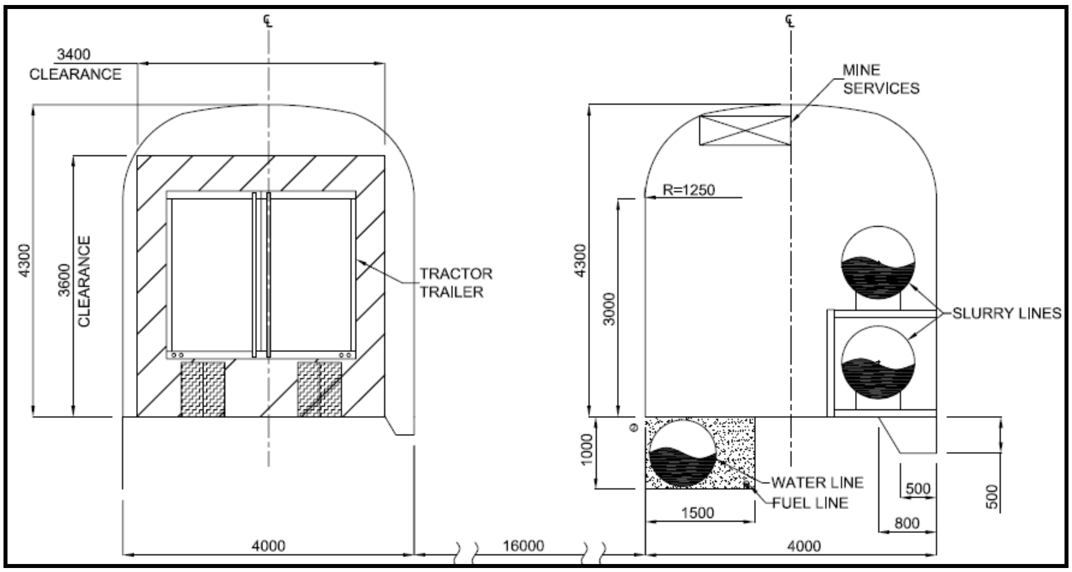
19.5.2 TUNNEL DESIGN

MITCHELL-TEIGEN TUNNEL

This tunnel connects the mill and the mine area, and consists of cross-connected twin tunnels constructed using drill and blast techniques (DBT). One tunnel will house pipelines used to transport slurried ore from the mine to the mill, as well as slurry reclaim water, power, and fuel from the mill area to the mine area. The other tunnel will allow light vehicles and low-boy trucks to travel between the mill area and the mine site. This will provide an important, reliable supply route to the mine area during the winter when avalanche risks are high on the surface access road.

A cross section of the ore haulage tunnel as developed by Thyssen is shown in Figure 19.3.

Figure 19.3 Mitchell- Teigen Tunnel Cross Section – Twin 4.3 m x 4.0 m DBT Tunnels for Separate Ore and Low-boy Truck Transport



Source: Thyssen, 2009.

The cross-sectional area of the twin 4.3 m by 4.0 m DBT tunnels (33.5 m^2) is nearly identical to the previously considered single large tunnel; however, the twin tunnel configuration provides several advantages for both construction and operation. The twin tunnels improve construction efficiency by providing two working faces at each heading, which allows for drilling on one face while the other is being mucked or supported. The narrower span tunnels also allow for shorter ground support bolts, and reducing installation time and costs. Ventilation will be improved as air can be forced in one tunnel and drawn out the other, rather than having to use ducting. Having a dedicated fresh-air tunnel removes the need to completely evacuate the tunnel when blasting, which saves significant travel time, particularly once the faces advance several kilometres from the portal. Thyssen suggests that these factors combined could increase the advance rate by approximately 30%.

In terms of safety, in the event of an incident in one tunnel, the other can be accessed through the regularly spaced cross-links and used for refuge/escape/ emergency access. During construction, the two tunnels can be setup with one-way travel in a loop, which will reduce the risk of traffic accidents and eliminate the need for passing bays. Having traffic always go with the air-flow should also reduce ventilation resistance. During operations, the truck access and ore transport systems are separated; thus, unlike a large single tunnel, any incident with one system should not disrupt the other.

The western (mine side) portals of the ore haulage tunnel are located just west of Mitchell pit at an elevation of 900 m. The tunnel route slopes upward to the mill at a constant grade of approximately 0.6% for 16.1 km before briefly daylighting at the Teigen/Treaty saddle at an elevation of approximately 1000 m. From here, the tunnel route continues at 0.6% grade for an additional 7.2 km to the eastern portals located next to the mill above the TMF area at an elevation of 1140 m.

Approximately 1.5 km north of the base case location, two alternative saddle portal locations have been examined, which place these saddle portals outside of the Treaty Creek drainage. The total tunnel length would increase up to a total of 23.8 km, depending on the location of the saddle portals. A future trade-off study will select the final location of the saddle portals.

The geology of the route is presented in plan as Figure 19.4 and in profile as Figure 19.5.

The thickness of the Johnstone Icefield along the ore haulage tunnel route was investigated in early 2009 using ice radar and was found to be up to approximately 200 m. Given that the tunnel alignment is nearly 1000 m below surface at this point, ice thickness is not expected to be a significant issue to tunnelling in the area due to the depth of rock cover over the tunnel.

Figure 19.4 Geological Map of Tunnel Routes

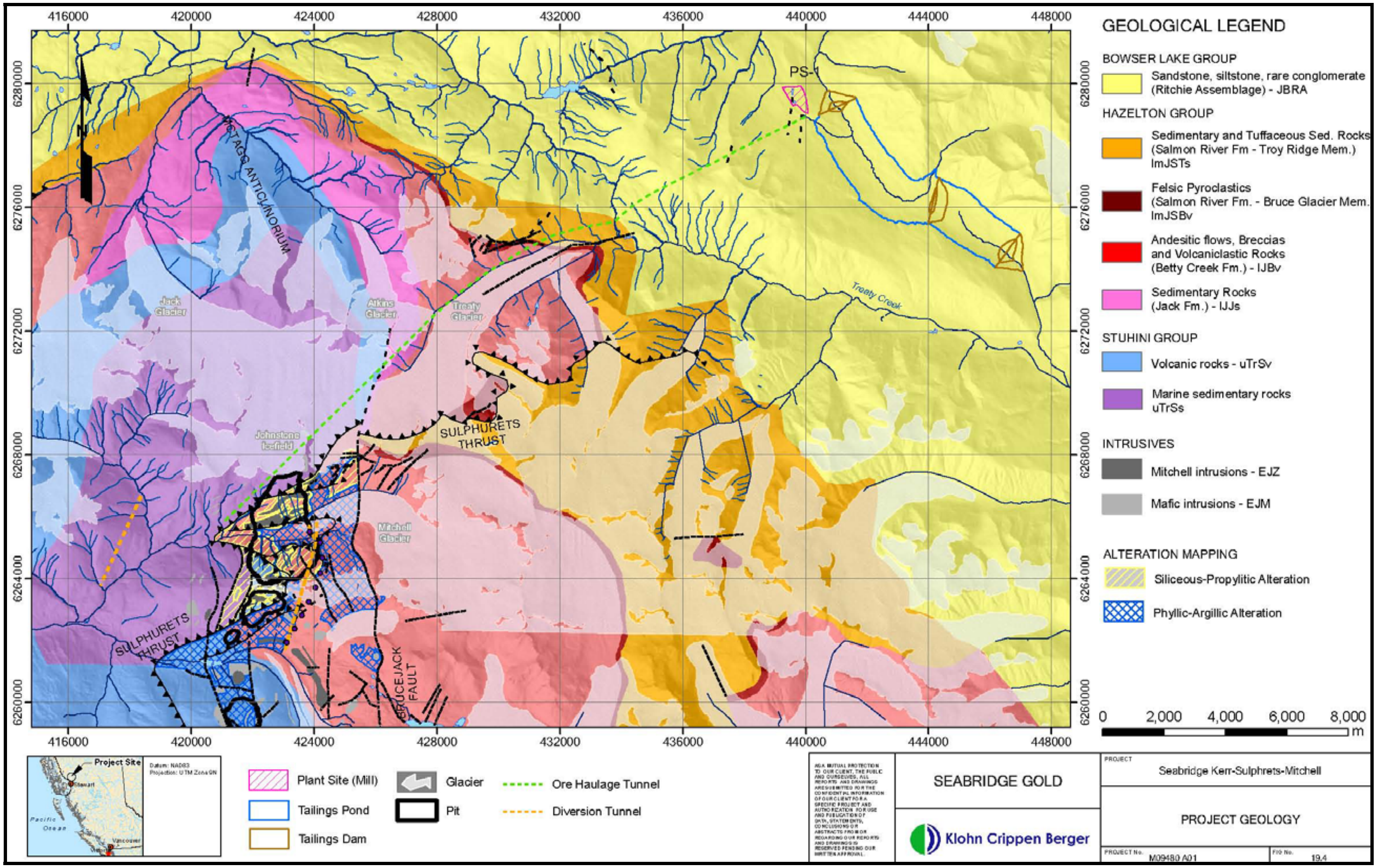
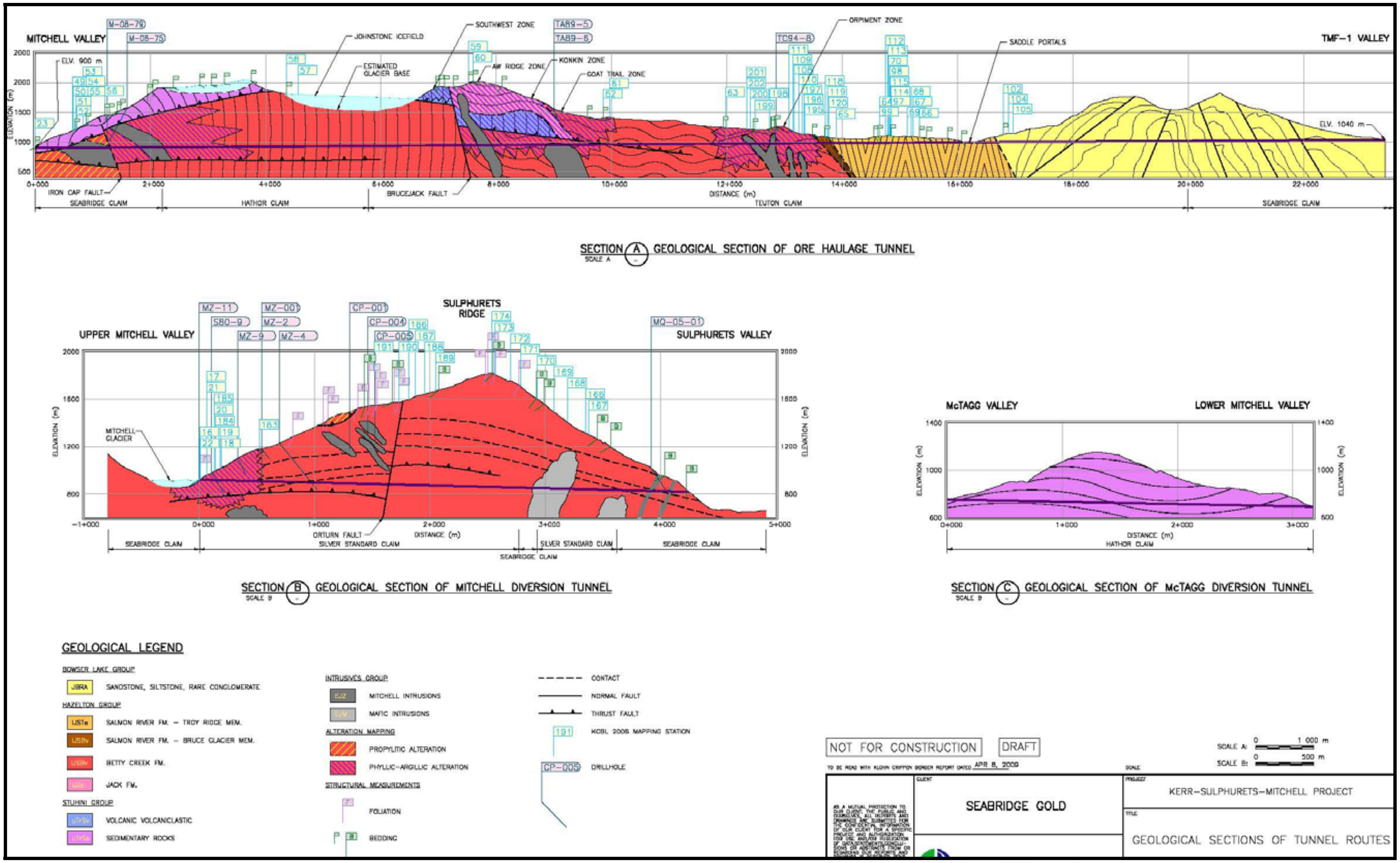


Figure 19.5 Geological Profile of Tunnel Routes



MITCHELL DIVERSION TUNNEL

The Mitchell diversion tunnel will be completed during pre-production and will link the Mitchell and Sulphurets valleys to divert flows from Mitchell Valley upstream of Mitchell pit into Sulphurets Valley. The tunnel is 4.2 km long. The Stage I initial Mitchell inlet will be located at the base of Mitchell Valley, 150 m upstream of the upstream rim of the 5-year Mitchell pit, and is estimated to be at an elevation of 840 m based on recent ice radar investigations on the lower portion of the Mitchell Glacier. Based on historical aerial photos, the glacier is receding at roughly 35 m/a; as it recedes and the pit expands, the inlet will be relocated further upslope, outside the ultimate pit boundary.

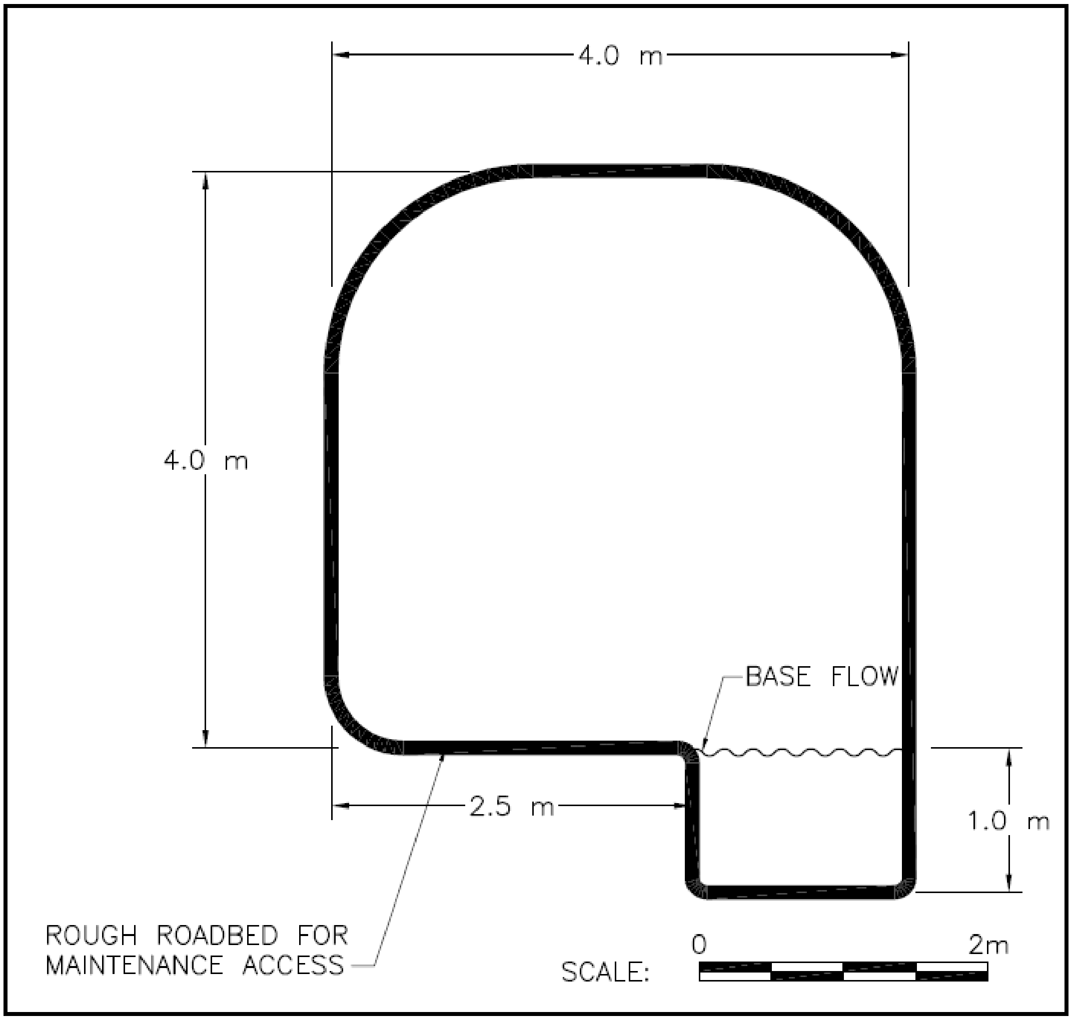
The Mitchell diversion tunnel is designed to have a cross section of 4.0 m by 4.0 m (Figure 19.6) to route the 1:100 year peak rainfall event. The tunnel has a catchment of 33 km^2 upstream of the Mitchell pit, and base flows of up to 3.6 m^3/s will be handled in the drainage ditch at the base of the tunnel. The tunnels are designed to allow flood flows to fill the entire tunnel thus maximizing the amount of water that can be discharged.

The tunnel will slope downhill at a 2% grade, emerging at the north wall of Sulphurets Valley at an elevation near 755 m. Here, flows will be channelled by a weir into a penstock pipe for the Sulphurets micro-hydro plant. An adjacent spillway will route flood flows that exceed the capacity of the penstock around the hydro plant.

Based on hydraulic modelling of the 1:100 year flood event, a 20 m high inlet dam has been selected to provide sufficient inlet immersion to drive peak design flows into the tunnel and maintain 5 m of freeboard. Multiple levels of inlets to the tunnel allow redundancy of operation. A series of debris catch basins are designed to allow regular cleanout of sediment by an excavator.

In the case of overtopping of the dam, emergency spillways will route water around Mitchell pit on berms. If these fail to contain the flood, mobile equipment in the pit can retreat and continue to work at higher elevations until the pit can be completely pumped out after the event.

Figure 19.6 Cross Section of the 4.0 m by 4.0 m Mitchell Water Diversion Tunnel



WATER TREATMENT DAM CONSTRUCTION DIVERSION TUNNEL

A temporary 2.5 m by 2.5 m by 800 m-long diversion tunnel is planned to facilitate construction of the mine area water treatment pond dam, located downstream of the Mitchell rock dump. Once the dam is complete, this diversion tunnel will be plugged.

MCTAGG DIVERSION TUNNEL

After Year 10 of operations, either the McTagg diversion tunnel or a high level surface diversion channel will be needed to route McTagg Creek around the Mitchell rock dump once it expands into McTagg Valley. The catchment upstream of the inlet is 22 km^2, and base flows are expected to be on the order of 2.2 m^3/s. The McTagg diversion tunnel would have the same 4.0 m by 4.0 m cross-section as the Mitchell diversion tunnel.

19.5.3 *Estimate of Tunnelling Costs*

Estimated costs for the Mitchell-Teigen tunnel and Mitchell diversion tunnel excavations were calculated by Thyssen based on geotechnical information and geological mapping gathered by KCBL. These costs are presented in Appendices I and K.

19.6 Geotechnical Opportunities and Recommendations

A more detailed geotechnical and hydrogeological evaluation based on additional drilling is required for the proposed tailing area to provide a basis for feasibility-level determinations of stability and seepage. This program will be built on the preliminary drilling completed in 2008 and is planned for 2010. The detailed site investigations will include additional hydrogeological and geotechnical drill testing of the tailing dam areas as well as drill testing of borrow areas for material characterization and volume determination. Further optimization of rock dump layouts and a more detailed evaluation of the impacts of diversion, cover, and leachate collection system efficiencies on the required water treatment rate and storage pond volume are required. A comprehensive water management plan needs to be developed to create an operating plan for the mine area. As there is significant precipitation and limited opportunities for further surface diversions, the cost to manage or treat water may be high.

The mine rock dump areas and, to a lesser extent, the TMF, are in areas of high snow avalanche hazard and a comprehensive snow avalanche management plan is required to ensure security of the operating facilities and human safety.

20.0 HYDROLOGICAL SURVEY

20.1 GENERAL HYDROLOGICAL SETTING

The KSM Project area is located in the coastal mountains of northwestern BC. The proposed pit areas lie within the headwaters of Sulphurets Creek, which is a main tributary of the Unuk River. The proposed TMF will be located primarily within a tributary of Teigen Creek. A smaller portion of the proposed TMF, which would not be constructed until well into the operational life of the mine, will also be located within a tributary of Treaty Creek. Both Teigen and Treaty creeks are tributaries of the Bell-Irving River, which is itself a major tributary of the Nass River. Both the Nass and Unuk rivers flow to the Pacific Ocean.

The project area lies within a transition zone from a wetter coastal climate to a drier interior climate. This longitudinal gradient results from storms, which are formed over the Pacific Ocean, losing moisture as they pass over successive mountain ranges travelling inland from the coast. In addition to the longitudinal precipitation gradient, there also exists a gradient that delivers greater precipitation to higher elevations due to the rugged topography and orographic nature of most storms in the area. Therefore, on average, the proposed pit areas will likely receive greater precipitation due to their western position within the project area and the high elevation of the surrounding topography in relation to the proposed TMF. Mean annual precipitation in the pit area is expected to be approximately 1,600 to 2,000 mm, which will vary depending on elevation. Annual precipitation is expected to be less at the proposed TMF than that experienced in the proposed pit areas.

The proposed pit areas are located within the headwaters of Sulphurets Creek. The Sulphurets Creek watershed is characterized by steep, narrow valleys and is highly glaciated. Both characteristics tend to result in a high percentage of precipitation resulting in surface runoff. Steep hill slopes tend to promote surface runoff of precipitation in the form of rainfall or snowmelt, while glaciers can produce high runoff volumes during the summer months regardless of precipitation. Consequently, annual runoff coefficients (percent of precipitation resulting in surface runoff) for the proposed pit area drainages are expected to be high, on the order of 80 to 100%.

The area of the proposed TMF is characterized by relatively low gradient hill slopes and a relatively wide valley bottom with a substantial wetland complex. In addition, the proposed TMF lacks the upstream glaciers found in the Sulphurets Creek watershed. These characteristics tend to promote precipitation losses in the form of infiltration and evapotranspiration, thereby reducing the production of surface runoff. Consequently, annual runoff coefficients for the proposed TSF area are expected to be on the order of 60 to 80%.

A typical hydrological year for watersheds in the project area can be divided into four main flow periods – winter, spring/freshet, summer, and fall. Winter (approximately November to April) is characterized by ice-covered streams with low to negligible stream flow, depending on the elevation of the stream and watershed area. The spring/freshet period (late April or May to July) is characterized by high flows rates due to snowmelt and may contain the annual peak flow for any given year. For watersheds in the area of the proposed TMF, summer (approximately July or August to mid-September) is characterized by steadily decreasing high to moderate flows that are augmented by rainfall and melt water from residual snow patches. Flows can continue to rise through the summer in Sulphurets Creek and its tributaries due its glacierized headwaters, which can provide substantial melt-water late into the summer. Fall (mid-September to November) is characterized by generally moderate to low flows but interrupted by rain-fed storm events, which can generate peak flows in excess of freshet flows and may contain the annual peak flow for any given year.

21.0 INFRASTRUCTURE AND SITE LAYOUT

21.1 MINE AND SITE LAYOUT

The overall plant and mine layout is shown in Drawing No. 10-10-300 (Appendix G), with individual facilities located to take advantage of the natural topography and, to the extent possible, minimize the impact on the environment.

Due to the limited usable topography, the plant site is located approximately 23 km northeast of the Mitchell Zone. Parallel twin tunnels connected by crosscuts containing the slurry and return water pipelines and services will be constructed to deliver the mill feed for processing and tailing storage. The tunnel will extend from the north side of the Mitchell Zone approximately 23 km to the northeast into the upper reaches of the Teigen Creek Valley. There is a saddle point approximately 16 km from the Mitchell portal where the tunnel daylights.

Highway 37, a major road access to northern BC, passes within 14 km of the KSM Project's proposed tailing site. A preliminary road study by McElhanney proposes a 14 km routing to the plant site and a 1 km spur road to the Teigen Creek side of the tailing facility. A temporary construction road approximately 15 km long will be provided from the plant site to the tunnel saddle point to facilitate tunnel construction and PAG rock removal from the tunnel saddle portals. Road access to Mitchell Creek will be provided by a 34 km continuation of the Eskay Creek Mine access road. There is one major bridge crossing over the Unuk River. The mine areas will be accessed via a road connection from the Eskay Creek Mine access road, paralleling first Coulter Creek, before crossing the Unuk River and progressing up Sulphurets Creek. Temporary winter access will be provided by a road over the Frank Mackie Glacier complete with provision of a temporary bridge over the Bowser River. The total distance is 41 km with 28 km of glacier travel. This road will be utilized to transport equipment to the mine site during the initial stages of the project. Site access is currently by helicopter only.

Avalanche sheds will be constructed in the upper side of Treaty Creek near the tunnel portals. Waste rock landforms are located adjacent to the Mitchell zone pit.

The process plant will consist of three separate facilities:

- an ore crushing/grinding and handling facility at the mine site (Mitchell side)
- ground slurry transportation

- a main process facility at the plant site including secondary grinding, flotation, regrinding, leaching, cyanide recovery/destruction and concentrate dewatering.

The TMF is located in a valley between Treaty Creek and Teigen Creek, approximately 24 km east of the mine site. The TMF catchment will collect a sufficient annual surplus of water during operation for plant water supply operations and will store a one month plant water supply.

The overall plant layout (shown in Sketch No. 10-20-SK02, Appendix G) is compact and, to the degree possible, takes advantage of the natural ground contours. Due to ground conditions at the plant site area, a significant volume of cut-to-fill material is required to locate structures and equipment on solid foundations. Equipment foundations are assumed to be conventional foundations, where heavy equipment will be located on bedrock with spread footings and raft foundations over the remainder of the area. Concrete slabs will either rest on piles going to bedrock, or all fill to sound bedrock will be removed and replaced with non-frost susceptible engineered backfill. Excavation and fill to an average depth of 1.5 m below grade has been used for estimating purposes.

The primary water treatment plant will be situated in the lower Mitchell Valley to treat potential ARD, tunnel drainage, and surface drainage waters from the rock dumps at the Mitchell site.

Major avalanche run-out hazards have not been observed in the plant site plateau area. Plant domestic and process water supply will be provided from water diversions constructed around the perimeter of the tailings dam.

The plant site will be terraced with plant site roads to establish construction grade. The construction grade assumed has been established from the ore haulage tunnel portal exit elevation. All terracing has been based on nominal geotechnical information provided for the plant area.

Construction laydown areas have also been assigned and these areas will be cleared and levelled at the same time.

21.2 TUNNEL

21.2.1 BACKGROUND

The ore crushing/grinding and handling facility was located to suit the limited topography in the vicinity of the mineralized zone. The main process plant and the tailings storage facilities are located approximately 24 km east of the open pits. Numerous locations within the property boundaries were evaluated for the process and tailings facilities, taking into consideration such factors as environmental, social, ease of access, and constructability. The most logical way of transporting the mineralized material from the pit to the processing facility was through a twin tunnel, the Mitchell Teigen Tunnel (referenced as the Ore Haulage Tunnel in the 2008 PEA). The locations of the open pits plant sites, tailings storage, and slurry transportation tunnel can be seen on Drawing No. 10-10-300 (Appendix G).

21.2.2 *Tunnel Requirements*

There are essentially three tunnels associated with the study, as discussed below.

Mitchell Teigen Tunnel

Ground slurry mineralized material will be delivered from an ore crushing/grinding and handling facility at the mine Mitchell Zone side through a 23 km long tunnel connecting to the processing facility at Plant Site #1. The tunnel is constructed using DBT.

The mineralization slurry prepared from the mine site grinding facility will be delivered to the plant site by 3 stages of pumping through a 23 m long tunnel. Each tunnel has a cross section dimension of 3.4 m wide by 4 m high and is connected by cross cuts at approximately 300 m centres. Two pipelines will be used to deliver the mill feed slurry. There will be one return water pipeline from the TMF.

To supply power to the mine site, electrical cables will be installed in the roof of the tunnel and a buried diesel fuel pipeline will run through the tunnel. A gradual slope up from the pit side to the plant site will allow seepage water to drain back to the mine side where it will be collected and treated as necessary before release to the environment. Fire protection and a ventilation system will be installed to ensure air movement.

The parallel twin tunnel approach (shown in Figure 19.3) overcomes the significant ventilation issues that could arise during the driving of a single larger tunnel. The twin tunnel application reduces the ventilation costs to bring air to the construction face compared to a single large tunnel. It was assumed that the cost for the two smaller tunnels would be no more than a single large drill/blast tunnel and it will also reduce the construction schedule period.

In terms of safety, in the event of an incident in one tunnel, the other can be accessed through the regularly spaced cross-links and used for refuge/escape/ emergency access. During construction, the two tunnels can be setup with one-way travel in a loop, which will reduce the risk of traffic accidents and eliminate the need for passing bays. Having traffic always go with the air-flow should also reduce ventilation resistance. During operations, the truck access and ore transport systems are separated; thus, unlike a large single tunnel, any incident with one system should not disrupt the other.

Approximately 1.5 km north of the base case location, two alternative saddle portal locations have been examined, which place these saddle portals outside of the

Treaty Creek drainage. The total tunnel length would increase up to a total of 23.8 km, depending on the location of the saddle portals. A future trade-off study will select the final location of the saddle portals.

During initial construction of the tunnel at the saddle point (east and west portals), infrastructure requirements, fuel, labour, etc. will be serviced by helicopter support until an access road is constructed to the saddle area from the plant site.

MITCHELL DIVERSION TUNNEL

The Mitchell diversion tunnel is approximately 3.5 m wide by 3 m high by 4.2 km long through Sulphurets Ridge between the Mitchell and Sulphurets valleys. The Mitchell diversion tunnel will link the Mitchell and Sulphurets valleys and divert flows from the Mitchell Valley upstream of the Mitchell Zone into the Sulphurets Valley. The tunnel is constructed using DBT.

The initial Mitchell Zone inlet will be located at the base of Mitchell Valley, 150 m upstream of the rim of the 5-year Mitchell Zone pit, and is estimated to be at an elevation of 925 m. The tunnel will slope downhill at a 2% grade, emerging at the north wall of Sulphurets Valley at an elevation of 845 m. Here, flows will be channelled by a weir into the penstock pipe for a 1.78 MW capacity Sulphurets micro-hydro plant, with an adjacent spillway to route flood flows around the hydro plant that exceed the capacity of the penstock.

MCTAGG DIVERSION TUNNEL

After Year 10 of operations, either the McTagg diversion tunnel or a high level surface diversion channel will be needed to route McTagg Creek around the Mitchell rock dump once it expands into McTagg Valley.

21.3 MITCHELL SIDE CONVEYORS

Crushing of the mineralized material will be carried out using two gyratory crushers located near the Mitchell pit ramp exit. The discharge from each of the crushers will be conveyed to single 2.1 m (84”) wide stockpile feed conveyor that has the capability to handle up to 10,000 t/h to handle the surges coming from the two crushers. The stockpile feed conveyor is over 550 m in length and transports the crushed product to the surge stockpile located near to the tunnel connecting the open pit side of the project to the mill feed processing side. The crushed coarse materials will be reclaimed from the 120,000-t stockpile by 6 reclaim apron feeders onto one 1.83 m (72”) wide HPGR feed conveyor to two HPGR feed surge bins, each with a live capacity of 300 t.

21.4 SITE ROADS

The main ring road around the plant is estimated at a length of 5,000 m and a width of 6 m. Site roads will be supported on crushed rock, excavation, and fill to an average depth of 1.5 m below grade, which has been used for estimating purposes.

21.5 SADDLE POINT ACCESS ROAD

An access road will be constructed during the initial phases of construction from the plant side to the Mitchell Teigen Tunnel saddle area in order to facilitate the driving of an east and west head of the Mitchell Teigen Tunnel.

21.6 PROCESS PLANT

The process plant facilities will consist of a primary crushing station located at the mine side. There will be one 120,000 t live storage capacity housed in a steel clad storage facility, a pebble crushing section, an HPGR, and a ball mill grinding section.

On the plant side, there will be secondary grinding, a flotation section, a refinery for the dore, a CIP plant, a cyanide destruction plant, a reagent preparation area, a filtration area, a concentrate handling area, and a tailings handling area. The grinding and flotation building will be a stick-built structural steel building complete with overhead crane, electrical rooms, HVAC, and offices.

21.7 ANCILLARY BUILDINGS

Ancillary building construction for the study will be pre-engineered structures or stick-built structures as applicable. The following buildings are included in the study:

- plant site:
  - fuel storage facility
  - fuel station
  - administration building
  - assay and metallurgical laboratory
  - warehouse and maintenance building
  - concentrate storage building
  - first-aid building
  - sewage treatment plant
  - 250-person camp modular camp
  - 1,200-person construction camps (construction camps will be set up at the plant, saddle, and mine sites)

- mine site:
  - truck shop
  - 300-person camp modular camp
  - fuel station
  - diesel fuel storage and dispensing
  - sewage treatment plant
- off site:
  - concentrate storage building (Stewart, BC).

21.8 Assay Office

An assay laboratory will be located in a separate building at the southern side of the mill building. It will be equipped to perform daily analysis of mine and process samples. The laboratory will be a 755 m^2 single-storey pre-engineered structure.

21.9 Concentrate Storage

The on-site concentrate storage facility, approximately 7,500 m^2 in area, will have a 5 day storage capacity equating to 1,200 t/d of concentrate.

Concentrate will be loaded at the KSM site into trucks and hauled to a concentrate storage facility at Stewart, BC.

21.10 Warehouse/Truck Shop/Mine Dry

The warehouse/truck shop/mine dry building will be a pre-engineered building, approximately 200 m long by 40 m wide by 19 m high. The building will be designed to provide facilities for maintenance and repair, warehouse storage, minor office space, clean and dry areas, and general storage. It will be located west of the Sulphurets Zone, adjacent to the Sulphurets Canyon Road and the haul road between the Sulphurets and Kerr zones.

The truck shop/mine dry will comprise eight maintenance bays, two light vehicle repair bays, a truck and lube bay, a truck wash bay, a welding and machine shop, an electrical and instrument shop, a 1,200 m^2 storage warehouse with an upper level mezzanine area, and a dry area including lockers, offices, restrooms, first aid, and emergency vehicle storage. Waste oil will be disposed of in the refuse incinerator with any remaining oil removed and discarded at an approved facility.

21.11 FUEL STORAGE

The diesel fuel tanks will be stored in a high-density polyethylene (HDPE)-lined containment area. Two 15 m (diameter) by 9.75 m (high) fuel tanks and one 3.2 m (diameter) by 3 m (high) gasoline tank will be provided. Fuel dispensing facilities will be provided, including light vehicle and fast fill for mining equipment.

21.12 POWER SUPPLY AND DISTRIBUTION

21.12.1 AREA TRANSMISSION FACILITIES

Power generation and transmission in the province of BC are governed by the regulations of the BC public Utilities Commission (BCUC). The majority of the power in BC is generated by BC Hydro, although there are an increasing number of private (IPP) generators. The major transmission system in BC is operated by the British Columbia Transmission Corporation (BCTC). Both BC Hydro and BCTC are Crown Corporations.

In general, new bulk transmission customers in BC are responsible for constructing the required transmission line and other facilities to connect their plant to the nearest suitable point in the BC Hydro/BCTC transmission system. In addition, the customer is responsible for what is termed “Basic Transmission Extension”, which covers the utilities’ cost to serve the customer such as metering, circuit breakers, and line protection. Capital costs as may be required to upgrade the BCTC transmission system to serve the new customer are designated as “System Reinforcement” costs. These costs are also to the customer’s account, unless BC Hydro’s projected sales revenues over a seven-year period are greater than the system reinforcement capital cost.

The BC Hydro power grid in the northwest of the province currently only extends to Meziadin Junction, a point south of the KSM Project. There is an existing 138 kV transmission line from the 500 kV Skeena Substation, located near Terrace, BC to Meziadin Junction and then extending west to the town of Stewart. This existing 138 kV interconnection does not have adequate capacity to supply the KSM Project and also terminates approximately 100 km south of the mine site. The nearest source of adequate capacity is the 500 kV Skeena Substation which is located along Highway 16, east of Terrace, BC, approximately 220 km south of Meziadin Junction (as per the proposed new 287 kV line route).

There are two viable options for power supply to the KSM Project from the provincial power grid. These are:

- **Option 1** – power supply to the KSM mine via the proposed Northwest Transmission Line (NTL) project, if this project is constructed. This proposed line would run within 14 km of the proposed KSM Substation No. 1.

- **Option 2** – service under BC Hydro's current bulk service tariff 1823, from Meziadin junction, the nearest point on the transmission system.

Option 1 is based on the fact that BCTC is in the planning stage for an approximately 335 km long, 287 kV high-voltage transmission line from the Skeena Substation to a new substation to be located near Bob Quinn Lake, north of the KSM Project. This undertaking is known as the NTL project and is one of the alternatives that would ensure an adequate power supply for the KSM Project. The environmental assessment for this project is well advanced, although the transmission line construction project itself has not been committed to.

The NTL project was originally announced in partnership with NovaGold Resources Inc. (NovaGold) to serve their Galore Creek Project. When Galore Creek was put on hold, the NTL was also put on hold. However, the BC provincial government has publically stated that the project could be re-initiated on similar terms to the Galore Creek proposed arrangement, if another large mining project in the area goes forward.

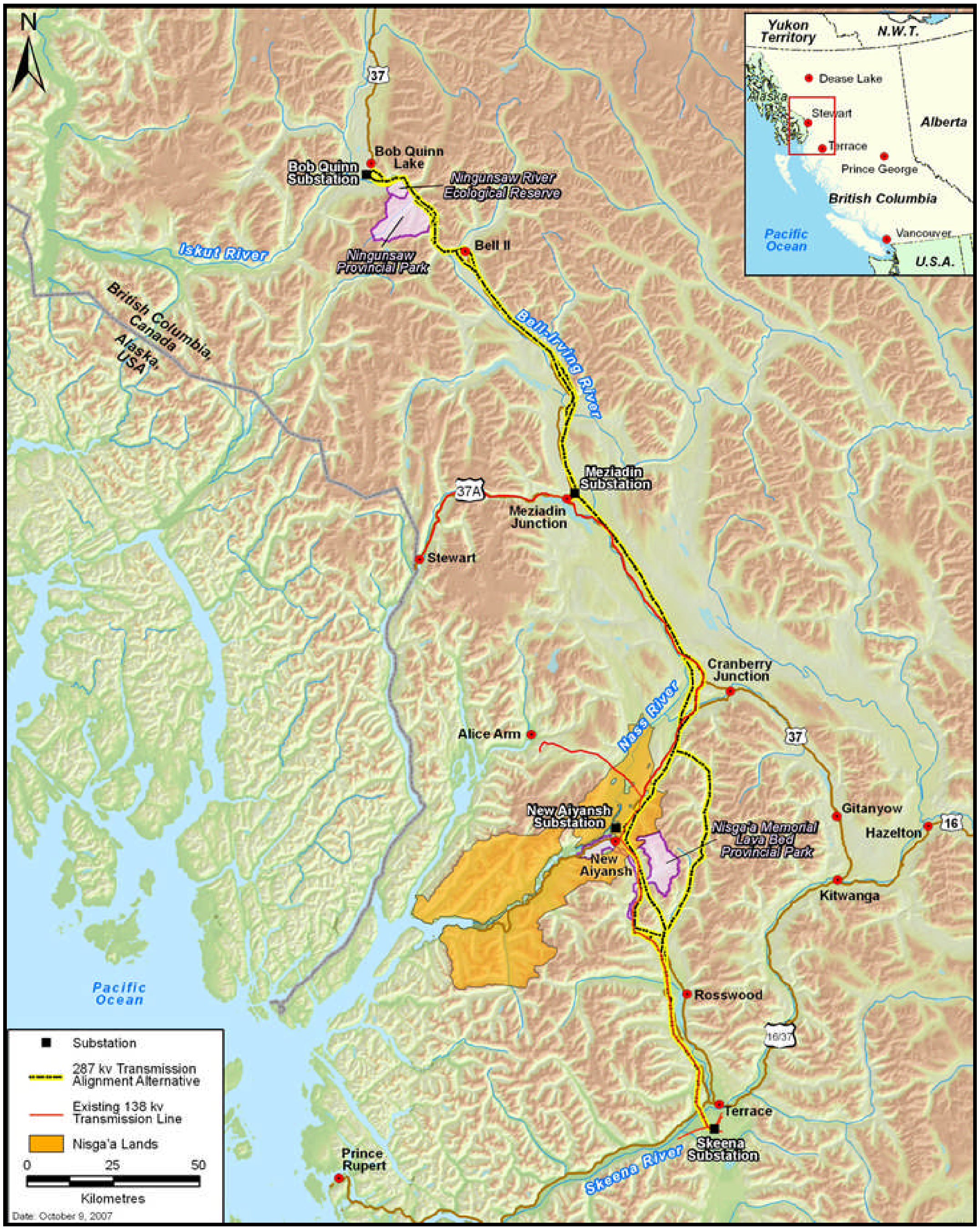
In September 2008, the BC Provincial Government announced a Cdn$10 M contribution to re-start the environmental assessment (EA) for the NTL line, which had been put on hold when the Galore Creek project was delayed. It is understood that the environmental studies are proceeding satisfactorily.

With regard to the regulatory framework covering the construction of the proposed new NTL transmission facilities by BCTC, the following quotation has been taken from BCTC's website:

> "The Transmission Expansion Policy (TEP) enables BCTC to advance proposals for review by the BC Utilities Commission that considers expanding the transmission system in anticipation of future transmission needs. BCTC developed the TEP with stakeholder input in 2005, in response to the provincial government's Special Direction No.9 (SD9). The TEP sets out a framework that allows BCTC to plan and expand the transmission system in the absence of firm customer contracts for transmission service when it is in the best interest of ratepayers to do so."

For electrical power supply to the KSM mine, the alternative designated as Option 1 is supply from the NTL project with interconnection via a proposed switching station along the NTL line at Snowbank Creek near Bell II, approximately 103 km north of Meziadin Junction. This tap would feed an approximately 14 km long transmission interconnection to the KSM No. 1 Substation, located at the flotation plant.

Figure 21.1 NTL Project Map



The viability and the cost of Option 1 is based on a number of factors including the NTL project going forward with at least one other major customer to share the costs and/or with an arrangement such as was negotiated by NovaGold for their Galore Creek Project. In this arrangement, NovaGold was to contribute a substantial sum towards the construction of the line, with the remaining costs to be funded by BCTC, until other customers came on-line who would then make proportionate contributions towards the original capital cost of the NTL.

The second option that has been studied for power supply to the KSM Project, Option 2, is to simply request service from BC Hydro under their current bulk service tariff 1823 “Transmission Service – Stepped Rate” from their nearest transmission interconnection point, that being Meziadin Junction. Supply under this schedule is available in, to quote from the tariff, “Rate Zone 1 excluding the Districts of Kingsgate-Yahk and Lardeau-Shutty Bench.” The proposed point of service (Meziadin) falls within this area.

Under Option 2, the construction of the transmission line and associated facilities from the service point at Meziadin Junction to the mine would be the responsibility of the customer, as per BC Hydro policy. As the 138 kV system at Meziadin Junction has limited capacity, system reinforcement to that point would be required. In this case, the transmission system reinforcement would require a new 287 kV transmission line to Meziadin, similar to that currently planned under the NTL project. Under Option 2, it is assumed that BC Hydro/BCTC would construct the line similar to that proposed under the NTL project, for which an EA is already under way.

When a BC Hydro customer requests service that requires system reinforcement as described here, the associated capital costs are to the customer’s account unless BC Hydro’s projected sales revenues over a seven-year period are greater than the line capital cost. As the KSM project has a very large power load (in the vicinity of 150 MW), the project would meet these criteria. Hence, the capital cost of the required new transmission line from Skeena to Meziadin would be funded by BC Hydro/BCTC, although a bond would be required over the seven-year period.

The cost per kilowatt hour of electricity purchased from BC Hydro under either Option 1 or 2 would essentially be the same. The only difference being that, under Option 2, the mine would be responsible for line losses as the metering point would be at Meziadin. However, with a line voltage of 287 kV the line losses over the 103 km interconnection will not be large.

The plan for actual energy purchase for both connection options is to contract supply from BC Hydro. Wheeling of the required power from other generators over the provincial grid is also possible and tariffs exist covering this option. However, as the BC Hydro rates are based on a mix of a large amount of older low cost hydroelectric generation, but smaller quantities of newer more expensive generation, their current (2008) bulk rate for power for projects such as KSM is in the range of US$0.039/kWh for firm power, including PST but excluding GST (assumed to be recovered). These electricity rates cannot be matched by other generators, even without considering wheeling charges.

21.12.2 Transmission System Capacity

For Option 1, the NTL transmission line from Skeena to Bob Quinn (as per the BCTC report by Chesterman Consulting Inc., dated June 22, 2005 titled "A Project Review of the North West Transmission Extension in Northwest British Columbia") would support a total load of 260 MW. This would allow for one or two other major mines in addition to the KSM Project. With an increase in the NTL line conductor size and with appropriate additional active compensation, the NTL capacity may be significantly increased.

For Option 2, service as per the standard BC Hydro tariff, the 287 kV line from Skeena Substation to Meziadin and the proposed KSM line from Meziadin to the mine would have adequate capacity for any probable KSM load.

21.12.3 Transmission System Facilities and Costs

For power supply Option 1, the project costs are (to a large extent) dependent on the NTL capital costs, which have been estimated by BCTC. With regard to the sharing of these costs by customers, the BC Minister of Energy and Mines has stated that new customers would be responsible for their share of the construction costs of the NTL project. This is in agreement with long standing BC Hydro policy (and current BCTC policy) where new customers pay all costs for their transmission interconnection past the service (metering) point in the existing transmission system. BCTC does not, and is not allowed to under BCUC regulation, construct branch lines to mines or other major loads, or to IPP generators for that matter.

BCTC preliminary cost estimates for the NTL project were carried out in 2006. As per BCTC comments on escalation:

> "Construction cost escalation rates were assessed by two independent consulting firms. One consultant recommended applying 5%, 5%, 4%, 4%, 4%, and 3% (the lower rate) from 2006 to 2011, respectively. The other consultant suggested 8%, 8%, 7%, 6%, 5%, and 4% (the higher rate) from 2006 to 2011, respectively, for this sector of construction."

The KSM Project Option 1 power supply capital costs were previously estimated in 2008 dollars, with 50% of the cost of the common NTL facilities included as KSM costs and 100% of the cost of KSM specific items (such as the Bell II switching station, branch line to the mine, mine substation, etc.) included in the total mining project power supply costs.

The previous Option 1 costs have been compared to power supply costs under Option 2. As the Option 2 costs were significantly lower, and as Option 2 is not dependent on a project with an uncertain future (the NTL project), this study is based solely on Option 2.

The necessary system reinforcement by BC Hydro/BCTC to cater to Option 2 includes:

- modifications to the Skeena Substation
- a new 287 kV transmission line from the Skeena Substation approximately 220 km to Meziadin Junction
- a new switching station at Meziadin
- a series capacitor compensation station.

The costs attributable to the KSM Project for Option 2, supplied as per standard BC Hydro tariffs, include the cost of construction by KSM of the entire transmission line and other facilities from Meziadin Junction to the mine. These have been estimated and found to be significantly less than the estimated Option 1 (NTL service) costs. This is primarily due to the fact that under the current tariffs, although service from Meziadin (the closest point in the transmission grid) would require a new 287 kV line in lieu of the existing 138 kV line, this system reinforcement would be funded by BCTC as the projected power sales revenues over seven years would exceed the estimated line capital cost. It should be noted that a bond would be required over the seven-year period to ensure cost recovery, should the mine cease operations prematurely. In addition to the foregoing factors, service to the KSM mine requires a transmission line approximately 20 km shorter than the NTL project and does not require a terminal station at Bob Quinn.

Advantages of Option 2 (service under regular BC Hydro tariffs) include a lower capital cost and the elimination of the uncertainty of the NTL project. Construction for Option 2 could proceed as soon as the KSM and current NTL environmental permitting is complete. Early work such as engineering and equipment ordering could proceed as soon as funding is available.

Supply Option 2, as discussed, involves construction of a 103 km of 287 kV transmission line from Meziadin Junction to Snowbank Creek, a point just north of Bell II. This study is based on the use of the same right-of-way and the associated EA (now underway) as per the NTL project. This assumes cooperation by BCTC and the government.

The 287 kV branch line to the mine (by KSM as required for both Options 1 and 2) includes 14 km of 287 kV transmission line following the mine access road to KSM Substation No. 1 at the flotation plant. The final connection to the KSM mine and mill Substation No. 2 would be accomplished with a 23.5 km long section of solid dielectric 287 kV cable through the 23 km long mine access tunnel. This installation would be the same for both power supply Options 1 and 2.

A detailed estimate of the costs for Option 2 has been carried out to a PEA-level of accuracy and is included in the project budget.

21.12.4 TRANSMISSION LINE SCHEDULE

The 287 kV line extension by BCTC from their Skeena Substation to Meziadin Junction and the required new transformers in the Skeena Substation is a considerable undertaking. It is estimated that First Nations issues, line design, surveying, construction, and commissioning will require a further four-year period after the completion of the current environmental studies. Environmental studies that had previously commenced were put on hold when Galore Creek was postponed but, as of late September 2008, these studies were reinstated based on a Cdn$10 M provincial government contribution and have been progressing satisfactorily.

For Option 2, service under existing BC Hydro tariffs, it is assumed the data collected for the EA for the last section of the NTL project could be used for KSM transmission line from the Meziadin to Bell II.

21.12.5 KSM MAIN 287 KV SUBSTATION

The KSM 287 kV step-down Substation No. 1 at the flotation plant would be constructed and owned by KSM, in accordance with BCTC and BC Hydro policy. This is also most economical.

The substation equipment has been sized based on the latest Project Load List that shows peak loads of just under 150 MW. A number of energy conservation measures have been incorporated into the project in order to keep the load below 150 MW. If the project power requirements surpass this level, considerable cost would be incurred since the KSM Project would then be responsible to BC Hydro for “generation reinforcement”. In other words, large capital contributions to BC Hydro would be required to compensate for the large amount of new generation that would be necessary to meet the KSM load.

To supply the KSM project load, two parallel 287 – 25 kV, 35/47 MVA, step-down power transformers have been selected for installation at the No. 1 Substation at the flotation plant and two 287 – 25 kV, 90/120 MVA step-down transformers are included for the No. 2 Substation located at the mine and mill. With two parallel transformers in both substations, system redundancy is provided to allow for failure of any single transformer. All substation transformers will be provided with full oil containment facilities, or will use environmentally acceptable vegetable oils or silicone fluids for cooling. The secondary plant site distribution voltage is 25 kV at both substations.

Two switched reactors have been included in the No. 1 Substation for compensation of the incoming 287 kV line to limit Ferranti effect over-voltages. Two switched reactors are included at each end of the 23 km-long 287 kV cable to compensate for cable capacitance. Preliminary studies have indicated that these 6 switched reactors plus automatic excitation control of the 42 MW of 0.85 power factor synchronous ball mill motors will provide good system voltage control, without the use of transformer automatic tap changers or power electronic equipment such as static var compensators. The final requirement for line compensation equipment would be identified in technical studies carried out by BCTC, which are currently being initiated.

The two substations are interconnected by 3 single core 500 mm^2 XLPE solid dielectric power cables suspended from the back (roof) in the pipeline tunnel. The 500 mm^2 (1,000 kCM) conductor size quoted is the minimum physical size the vendor makes at 287 kV (due to the field gradient at the conductor) and has more than adequate capacity to carry any anticipated load, including allowance for the cable charging current.

The substations do not include harmonic filters so, if these are required, they would be best located at the process plant near the harmonic sources and will be in the process plant budget.

21.12.6 MINE POWER

Power to the mine itself will be provided by local 25 kV distribution lines. The required pit 25–7.2 kV portable substations, pit switch-houses, and trailing cables are included elsewhere in the project budget.

21.12.7 CONSTRUCTION AND STANDBY POWER

Modular diesel generator sets will be provided to supply construction power for tunnel driving and other initial construction activities. The permanent power supply facilities (perhaps with limited initial capacity) would be scheduled to be available at as early at date as possible to eliminate the need for local diesel generation at the flotation plant and saddle areas, but not at the mine and mill site where construction power will be required until several months after completion of the 23 km long tunnel. The construction gensets will, however, be retained and reconfigured to serve as future standby/emergency generation for the mine, process plant, and accommodation centre. The cost of this equipment is included in the appropriate areas of the project budget and is not included in the primary power supply budget.

As noted, the transmission projects could possibly be scheduled so that KSM could complete construction of the 287 kV line from Meziadin to the mine at an early date and then temporarily energize this 287 kV line at 138 KV from the service now available at Meziadin Junction. This would limit the length of time a portion of the construction power at the mine would have to be provided by costly diesel generation. BCTC has previously studied this option in conjunction with planning for the Galore Creek project.

21.12.8 ENERGY CONSERVATION AND SELF GENERATION

Under BC Hydro Rate Schedule 1823, energy charges are based on a two-tier system with the last (nominally 10%) Tier 2 power being much more costly. How much energy consumption actually falls within the second high-cost tier depends on the customer base load (CBL). Project design factors that favour energy conservation will be considered by BC Hydro in their determination of the CBL, and thus will have significant economic impact. More detailed project design work should be scheduled in the next phase of study to address these issues.

Opportunities exist to beneficially generate electric power from water that has to otherwise be diverted around the mining operations. An allowance has been included in another area of the project estimate for these facilities.

21.12.9 FUTURE POWER SUPPLY RELATED ENGINEERING STUDIES

The next stage of project studies should include:

- completion of a preliminary service study by BC Hydro that will confirm utility related costs and requirements
- more detailed cost estimates for items to be constructed by KSM, such as the transmission lines, substations, line compensation, and similar
- confirmation of energy conservation measures including process design, equipment selection, energy recovery schemes, etc. since these will have significant economic impact due to the high cost of Tier 2 power.

21.13 SEWAGE

The treatment plants (plant side and pit side) will be a rotating biological contactor (RBC) system. The solid and liquid material will be separated in the treatment plant with the liquid stream discharging to the tailing pond and the solid material pumped out and trucked away twice per year by a specialized licensed contractor. The treatment plant will be constructed in modules with all modules used for the construction camp. Modules will be removed after construction so that the remaining system is optimized to service the operations facilities.

21.14 COMMUNICATIONS SYSTEM

A fibre optic cable has been included in the 23-km long tunnel to the mine and mill site to provide communications from the flotation plant area. Telecommunications to the flotation plant site has been allowed for by an allowance in the capital cost estimate.

Radio transceivers will be used for remote monitoring and control. A fibre optic backbone will be installed throughout the plant site to facilitate the control systems communication. A UHF radio system will be used for mobile communication.

21.15 Potable Water Supply

Water storage reservoirs for the process plant can be combined with diversion intake structures. These will be designed as project water requirements become better established. During the winter months, well water from a field of wells near the plant site may be needed to supply fresh water for process make up and domestic use at the plant and camp facility.

With two plant sites, two potable water sources are now required.

21.16 Explosives Storage and Handling

For information on explosives storage and handling, refer to Section 18.0 (Mining).

21.17 Geotechnical Conditions

Additional surface geotechnical investigation will be required to confirm the foundation ground conditions for the plant and building foundations for the next phase of the project.

22.0 ACCESS ROADS

22.1 INTRODUCTION

22.1.1 BACKGROUND

McElhanney was retained by Seabridge to complete a scoping-level study of the road access options to the KSM zones. Seabridge's KSM Project involves the development of a major gold-copper deposit located in remote northwest BC, approximately 40 km southwest of Bell II on Highway 37.

Proposed permanent road access routes to the mine and plant site include a 35 km road south from the Eskay Creek Mine and a 14 km road southwest from Highway 37 respectively. Both these routes and other alternatives will be evaluated in this report. The location, size, and cost of tunnels are not included in the scope of work.

22.1.2 PROJECT OBJECTIVES

Based on the available 1:20,000 TRIM data and 1:50,000 topographic maps, the proposed access routes to the KSM mine site and plant site were identified, marked on the base maps, and evaluated in terms of degree of difficulty of construction. The main objectives of this section are to estimate order of magnitude capital costs for road construction, and to compare route options and construction schedule alternatives.

22.2 ROUTE SELECTION

22.2.1 ROUTE DESCRIPTION

A number of feasible access options are considered including three permanent access roads, and two temporary winter roads to the mine site with two access roads to the plant site (see route maps in Appendix H).

ESKAY-UNUK-MITCHELL (EUM) ROUTE

This route commences at the Eskay Creek Mine (915 m elevation) and loses elevation quickly going south down to the Unuk River and a major bridge crossing at kilometre 8. The proposed road parallels the Unuk River to the confluence with Sulphurets Creek where it traverses across the north side of the Suphurets Canyon up to Mitchell Creek and then climbs up to the Mitchell Zone. The total distance is approximately 33 km.

ESKAY-COULTER-MITCHELL ROUTE

Heading southwest from the Eskay Creek Mine road, this route climbs gently towards Tom Mackay Lake then follows the height of land on the east side of Coulter Creek down to a proposed bridge crossing on the Unuk River (kilometre 20) just upstream of Sulphurets Creek. The remaining road follows the same route as the EUM route. The total distance is similar at 32 km.

TEIGEN-UNUK-MITCHELL ROUTE

Leaving Highway 37 north of Bell II, this route bridges Teigen Creek at kilometre 1 then continues up the south side of the Teigen Creek valley and along a major tributary of Teigen Creek to the headwaters of the Unuk River (kilometre 20). The first 19 km of this route are required to be built for access to the plant site and tunnel construction headings. This route parallels the Unuk River first on the north side then the south after a bridge crossing at kilometre 33. The last 27 km are in common with the EUM route described above. The total length is 73 km.

TREATY-UNUK ROUTE

This alternate route to the proposed plant sites starts at Highway 37, approximately 20 km south of Bell II. The route crosses the Bell Irving River then transects two existing forestry cutblocks before paralleling Treaty Creek on the north side. The length of the access road is 28 km to the Teigan Plant Site #1 and 32 km to the Teigan-Unuk Plant Site #2.

FRANK MACKIE GLACIER ROUTE

This proposed temporary winter access road starts at the Grandduc airstrip and follows the Bowser River to the toe of the Frank Mackie Glacier. From here it climbs up over the ice field and down to the Sulphurets Creek exploration camp. The total distance is 41 km with 28 km of glacier travel.

KNIPPLE GLACIER ROUTE

Commencing on Highway 37 near Bell I, this route involves barging equipment 20 km down Bowser Lake then following an old mining access trail up the north side of the Bowser River to the toe of the Knipple Glacier. Beyond this point, the road follows the ice field up to Brucejack Lake then down the glacier to Sulphurets Creek camp. Including the barge, the total distance is 63 km with 22 km over the glacier.

22.2.2 Road Design Requirements

The KSM Project access road is classified as a resource development road. The design criteria proposed for the permanent access would be a single lane (6 m surface), radio-controlled road capable of carrying the legal axle loading for trucks on BC highways on a year-round basis. The road is required to provide vehicle access for development of the mine site as well as year-round access for supplies, equipment, and crew transport.

Alignment controls, such as maximum 10% sustained adverse grades and 50 m minimum radius horizontal curves, are recommended to allow transport of the largest pieces of equipment to the mine site. Typical road cross sections (shown in Dwg. 1439-0-601, Appendix H) depict the approximate range of construction for varying terrain conditions. They also correspond to the construction categories defined below.

Table 22.1 Construction Categories

Category	Description
1	Existing Road/Upgrade
2	Other Material (OM) or Fluvial Fan/0-30% Sideslope/South Aspect
3	OM & Rippable Rock/30-50% Sideslope/North Aspect/Sidecast
4	OM & Some Solid Rock or Talus Slope/>50% Sideslope/Short End Haul
5	Solid Rock/Drill & Blast/End Haul
6	Wetlands/Overland Construction/End Haul Rock Ballast/Geotextiles

All bridges will be designed for 100-t loading (L100), 1.5 m clearance above the Q100 and shall meet the requirements of the *Navigable Waters Protection Act*.

22.2.3 Cost Estimate

This section uses order of magnitude costs to derive capital cost estimates for both the permanent and temporary road access construction. Historical costs for resource road construction in the northwest of BC have been adjusted to 2008.

Order of magnitude unit rates (Cdn$/km) are intended to be inclusive of roadwork costs: site preparation, subgrade construction, gravelling, culvert and minor bridge installation, and administrative costs (tendering, engineering survey and design, geotechnical, archaeological and environmental mitigation, and construction management). Major bridge crossings and avalanche protective structures are shown separately. Advanced planning, permitting, and the cost of financing are not included in the unit rates.

Capital costs are based on terrain conditions as defined by the construction categories and are based solely on a review of the TRIM mapping (refer to route maps in Appendix H). For this level of cost estimating, the like sections cover several kilometres in length and a wide range of terrain. The order of magnitude costs assigned to the construction categories are as follows.

Table 22.2 Construction Category Cost Estimates

Category	Description	Estimate (Cdn$)
1	Existing Road/Upgrade (0-5,000 m^3/km)	$300,000/km
2	OM or Fluvial Fan/0-30% Sideslope (5,000-10,000 m^3/km)	$800,000/km
3	Rippable Rock/30-50% Sideslope (10,000-20,000 m^3/km)	$1,000,000/km
4	Rock or Talus Slope/>50% Sideslope (20,000-30,000 m^3/km)	$1,200,000/km
5	Solid Rock End Haul (30,000-50,000 m^3/km)	$1,500,000/km
	Major Bridge Installations	$15,000/lineal m installed
	Avalanche Protective Structures	$25,000/lineal m installed
Temporary Winter Roads		
	Overland Construction	$200,000/km
	Ice Field Construction	$100,000/km
	Temporary Bridges	$10,000/m installed

Table 22.3 Route Cost Estimates

Construction Category	Unit Rate ($/km)	Eskay-Unuk-Mitchell Route		Eskay-Coulter-Mitchell Route		Teigen-Unuk-Mitchell Route		Teigen Plant Site & Teigen Intermediate Tunnel Portal Access		Treaty-Unuk Route Plant Site #2	
		Length (km)	Cost	Length (km)	Cost	Length (km)	Cost	Length (km)	Cost	Length	Cost
1											
2	$800,000	5	$4,000,000	3	$2,400,000			9	$7,200,000	15	$12,000,000
3	$1,000,000	11	$11,000,000	14	$14,000,000	10	$10,000,000	6	$6,000,000	12	$12,000,000
4	$1,200,000	10	$12,000,000			17	$20,400,000	14	$16,800,000	5	$6,000,000
5	$1,500,000	7	$10,500,000								
6											
Major Bridges											
Unuk River			$1,500,000		$1,500,000		$1,200,000				
Storey Creek			$500,000								
Gingras Creek			$500,000								
Mitchell Creek (2 crossings)			$1,600,000								
Coulter Creek					$500,000						
Teigen Creeks (2 crossings)									$1,300,000		
Bell Irving River											$1,500,000
Avalanche Sheds											
Sulphurets Canyon & Teigen Creek			$12,500,000				$17,500,000				
Upper Treaty Creek										100m	$2,500,000
S/A Eskay-Unuk-Mitchell Route				km58 to km73	$33,100,000	km46 to km73	$44,700,000				
TOTALS		**33**	**$54,100,000**	**32**	**$51,500,000**	**73**	**$93,800,000**	**29**	**$31,300,000**	**32**	**$34,000,000**
Temporary Winter Access											
Frank Mackie Glacier Route											
13 km overland sections at $200,000/km			$2,600,000								
28 km Ice Field Construction at $100,000/km			$2,800,000								
Bowser River temporary bridge			$200,000								
			$5,600,000								
Knipple Glacier Route											
21 km overland sections at $200,000/km			$4,200,000								
22 km Ice Field Construction at 100,000/km			$2,200,000								
Bowser River Bridge			$400,000								
Scott Creek Temporary Bridge			$200,000								
			$7,000,000								

22.2.4 Construction Schedule

Although a detailed construction schedule is not known, it is necessary to consider feasible implementation scenarios to arrive at a representative time line.

Scenario #1

The following assumptions have been made regarding Scenario #1:

- one camp and construction heading at Eskay Creek
- one camp and construction heading at Highway 37
- 20 to 25 km substantially complete road per heading per 5-month construction season.

Scenario #1 will result in the following:

- road access to the plant site and intermediate tunnel portals after Year 1
- road access to the Mitchell Zone after Year 2 constructing either route from Eskay Creek.

Scenario #2

The following assumptions have been made regarding Scenario #2:

- one camp and construction heading at Highway 37
- 20 to 25 km substantially complete road per 5-month construction season.

Scenario #2 will result in the following:

- road access to the plant site and intermediate tunnel portals after Year 1
- road access to the Mitchell Zone after Year 3 following the Teigen-Unuk-Mitchell route.

Scenario #3

The following assumptions have been made regarding Scenario #3:

- establish a remote camp and construction heading at Mitchell Creek by moving in equipment over one of the glacier routes or by air lifting.

Scenario #3 will result in the following:

- reduces the timeline for road access to the Mitchell Zone by one year for either of the above scenarios

- expect an increase in road construction costs by 25 to 30% for additional helicopter support
- may be an advantage for tunnel construction.

22.3 Conclusions

There are many combinations of access routes and construction scenarios, and the order of magnitude costs can be debated up or down but, for the purposes of this scoping level report, they provide a starting point and a consistent method of comparison.

Based on the preliminary mapping exercise and many assumptions, the least cost scenario for road access to the plant site and KSM zones is achieved by constructing the Teigen plant site and Teigen intermediate tunnel portals road and the Eskay-Coulter-Mitchell route. The result would be a substantially complete road after 2 years at a total cost of approximately Cdn$82.8 M.

22.4 Closure

This section has been prepared to assist Seabridge in evaluating road access alternatives to the KSM Project. The recommendations and cost estimates contained herein represent McElhanney's best professional judgment in light of the knowledge and information available at the time of preparation. McElhanney trusts that this section meets Seabridge's requirements and provides an understanding of the order of magnitude costs associated with developing road access.

23.0 LOGISTICS

Seabridge's KSM Project involves the development of a major gold-copper deposit located in remote northwest BC, approximately 40 km southwest of Bell II on Highway 37.

There are two transportation route possibilities for bringing equipment and supplies to the KSM property. One is using Highway 37 from the south and the other is barging equipment and supplies to Stewart and then transporting via Highway 37A to Highway 37 to a point where the junction of the mine access road is proposed. The proposed access road to the TMF and process plant from the Highway 37 junction is about 14 km in length. A second proposed route necessary to access the mine area involves extending the Eskay Creek Mine road to the south for a distance of about 35 km. The existing highways leading to the project area will require some upgrading of bridges and other crossings so that they are able to handle the equipment that is proposed to be transported over them. More evaluation of the upgrades will be identified during the next phase of the project study.

A temporary winter access road has also been proposed in this study. It starts at the Grandduc airstrip and follows the Bowser River to the toe of the Frank Mackie Glacier. From here, it climbs up over the ice field and down to the Sulphurets Creek exploration camp. This temporary winter access route is about 41 km in total length including 28 km of glacier travel. The purpose of this route is to get early access to the mine area to start construction prior to completion of the road from Eskay Creek or the tunnel. The logistics of this route will be further investigated in the next phase of the project.

For the purpose of this PEA, concentrates will be moved by truck to Stewart, BC. Stewart is BC's most northerly ice-free port and is capable of accommodating ocean going vessels. The KSM Project will require sufficient concentrate storage facilities for loading a Handymax vessel, which is about 50,000 t. Other operating materials, consumables, and supplies may also be stored at Stewart. If the barging option is not feasible, these items will be stored in another nearby location, such as Smithers. Although there are other projects in BC that are potential shippers through Stewart, there is currently enough room to construct additional warehouse facilities capable of handling the volume needed for the KSM Project.

Concentrate transport to Stewart by conventional trucking or pipeline has not been fully evaluated and will form part of the PFS phase of the project.

24.0 ENVIRONMENTAL

24.1 INTRODUCTION

The KSM Project is located in the mountainous terrain of north western BC, approximately 940 km northwest of Vancouver and approximately 65 km northwest of Stewart, as shown in Figure 24.1. The proposed project area lies approximately 20 km southeast of the Eskay Creek Mine and within 30 km of the BC-Alaska border. At the present time, access to the property is via helicopter.

The area is rugged, 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.

24.2 LICENSING AND PERMITTING

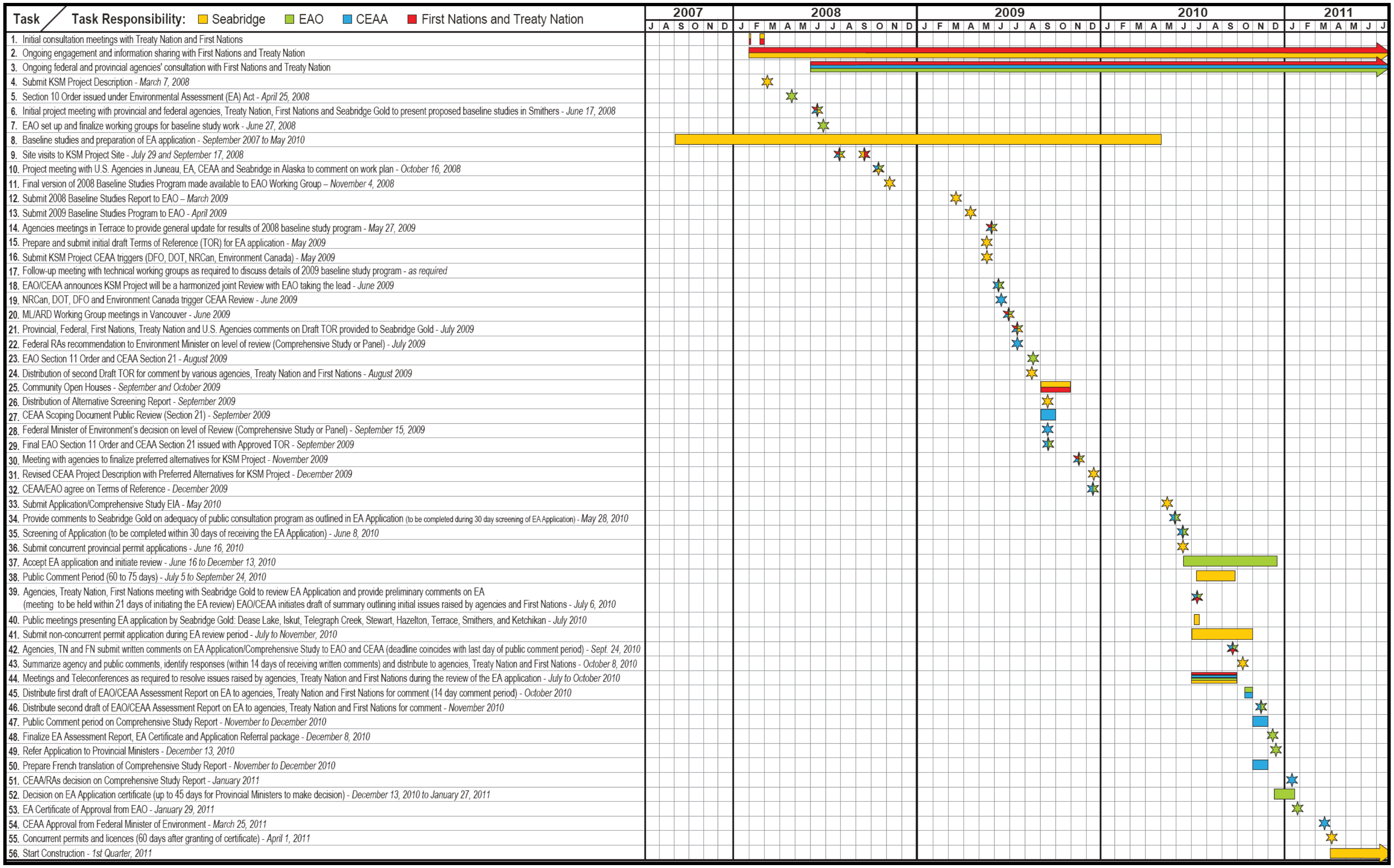
Mining projects in BC are subject to regulation under federal and provincial legislation to protect workers and the environment. This section discusses the principal licences and permits required for the KSM Project. Figure 24.2 outlines the approval schedule for the project up to the issuance of high level federal and provincial approvals in principle.

24.2.1 BRITISH COLUMBIA ENVIRONMENTAL ASSESSMENT ACT PROCESS

The *British Columbia Environmental Assessment Act* (BCEAA) requires that certain large-scale project proposals undergo an environmental assessment and obtain an Environmental Assessment Certificate before they can proceed. Proposed mining developments that exceed the threshold criteria laid out in the *Reviewable Project Regulations* are required under the Act to obtain an Environmental Assessment Certificate from the Ministers of Environmental and Energy, Mines and Petroleum Resources before the issuance of any permits to construct or operate. Seabridge has been advised by the BC Environmental Assessment Office (BCEAO) that the KSM Project will require an Environmental Assessment Certificate because its proposed production rate is greater than the *Reviewable Project Regulations* threshold of 75,000 t/a.

Figure 24.1 KSM Project Location

Figure 24.2 Regulatory Review and Approval Schedule



24.2.2 CANADIAN ENVIRONMENTAL ASSESSMENT ACT PROCESS

The Canadian Environmental Assessment Agency (CEAA) has advised Seabridge that the KSM Project will require an environmental assessment under the *Canadian Environmental Assessment Act*. The assessment is required because:

- Fisheries and Oceans Canada may issue a permit or licence under paragraphs 36(5)(a) to (e), where the regulation made pursuant to those paragraphs contains a provision that limits the application of the regulation to a named site of the *Fisheries Act* and may issue a permit or licence under subsection 35(2) of the *Fisheries Act*
- Environment Canada may issue a permit or licence under subsection 10(1) of the *International River Improvements Regulations*
- Natural Resources Canada may issue a licence under paragraph 7(1)(a) of the *Explosives Act*
- Transport Canada may issue an approval under section 5(2) and/or 5(3) of the *Navigable Waters Protection Act*.

The Canadian government has determined that the project will be reviewed as a comprehensive study, owing to the proposed construction of a structure for the diversion of 10,000 m^3/a or more of water from a natural water body into another natural water body (Part III - section 9, *Comprehensive Study List Regulations* of the Act).

AUTHORIZATIONS REQUIRED

Lists of the major federal and provincial licences, permits, and approvals required to construct, operate, decommission, and close the KSM 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.

24.2.3 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 Seabridge will seek concurrent approvals under the BCEAA process. However, no statutory permit approvals may be issued before an Environmental Assessment Certificate is obtained. Statutory permit approval processes are normally more specific than the environmental assessment level of review, and for example, will require detailed and possibly final engineering design information for certain permits such as the TMF structures and others.

Table 24.1 presents a list of provincial authorizations, licences, and permits required to develop the KSM Project. The list includes the major permits and is not intended to be comprehensive.

Table 24.1 List of British Columbia Authorizations, Licences, and Permits Required to Develop the KSM Project

BC Government Permits and Licences	Enabling Legislation
Environmental Assessment Certificate	*BCEAA*
Permit Approving Work System & Reclamation Program (Minesite – Initial Development)	*Mines Act*
Amendment to Permit Approving Work System & Reclamation Program (Pre-production)	*Mines Act*
Amendment to Permit Approving Work System & Reclamation Program (Bonding)	*Mines Act*
Amendment to Permit Approving Work System & Reclamation Program (Mine Plan-Production)	*Mines Act*
Approvals to Construct & Operate TMF Dam	*Mines Act*
Permit Approving Work System & Reclamation Program (Gravel Pit/Wash Plant/Rock Borrow Pit)	*Mines Act*
Water Licence – Notice of Intention (Application)	*Water Act*
Water Licence – Storage & Diversion	*Water Act*
Water Licence – Use	*Water Act*
Licence to Cut – Mine Site/TMF	*Forest Act*
Licence to Cut – Gravel Pits and Borrow Areas	*Forest Act*
Licence to Cut – Access Road	*Forest Act*
Licence to Cut – Transmission Line	*Forest Act*
Special Use Permit – Plant Access Road, Extension of Eskay Road	*Forest Act*
Road Use Permit – Eskay Road	*Forest Act*
Licence of Occupation – Borrow/Gravel Pits	*Land Act*
Licence of Occupation/Statutory Right of Way - Transmission Line	*Land Act*
Pipeline Permit – Diesel Pipeline	*Pipeline Act*
Surface Lease – Mine Site Facilities	*Land Act*
Waste Management Permit – Effluent (Tailing & Sewage)	*Environmental Management Act*
Waste Management Permit – Air (Crushers, concentrator)	*Environmental Management Act*
Waste Management Permit – Refuse	*Environmental Management Act*
Camp Operation Permits (Drinking Water, Sewage, Disposal, Sanitation and Food Handling)	*Health Act/Environmental Management Act*
Special Waste Generator Permit (Waste Oil)	*Environmental Management Act (Special Waste Regulations)*

24.2.4 Federal Approvals and Authorizations

Federal approvals include an authorization from the federal Minister of Environment approving the combined Application/Comprehensive Study Report for the KSM Project. Major stream crossing authorizations will be required from Fisheries and Oceans under the *Fisheries Act*. Approvals for water crossings will also be required under the *Navigable Waters Protection Act*. An explosive factory licence will be required under the *Explosives Act*. The *Metal Mining Effluent Regulations* (MMER) under the *Fisheries Act*, administered by Environment Canada, may require a Schedule 2 authorization because the area proposed for the TMF contains a fish habitat. A permit or licence may be required under the *International River Improvements Regulations* if the waste rock dump seepage storage impoundment alters seasonal flows to the Unuk River. Other federal requirements, such as those with respect to radio communication and aviation matters, will need licences. Table 24.2 lists some of the federal approvals required.

Table 24.2 List of Federal Approvals and Licences Required to Develop the KSM Project

Federal Government Approvals & Licences	Enabling Legislation
CEAA Approval	*Canadian Environmental Assessment Act*
MMER	*Fisheries Act/*Environment Canada
Fish Habitat Compensation Agreement	*Fisheries Act*
Section 35(2) Authorization	*Fisheries Act*
Navigable Water: Stream Crossings Authorization	*Navigable Waters Protection Act*
Explosives Factory Licence	*Explosives Act*
Ammonium Nitrate Storage Facilities	*Canada Transportation Act*
Radio Licences	*Radio Communication Act*
Radioisotope Licence (Nuclear Density Gauges/ X-ray analyzer)	*Atomic Energy Control Act*
Dam Licence	*International River Improvements Act*

24.3 Environmental Setting

The KSM Project is located in a remote area for which little baseline environmental data are publically available. Seabridge has engaged Rescan, a Vancouver-based consulting firm with extensive mining-related environmental assessment experience in BC, to undertake the baseline studies required for an environmental assessment of the project.

Baseline studies for the KSM Project were initiated in April 2008 following issuance of the Section 10 order from the BCEAO. Some preliminary water quality, meteorology, and hydrology data were collected in 2007.

24.3.1 TERRAIN, SOILS, AND GEOLOGY

The KSM Project is located in a very rugged area with elevations ranging from about 220 m at the Sulphurets-Unuk confluence to over 1900 m at the top of the ridge above the Kerr deposit. Surrounding peaks, such as Unuk Finger, are in the range of 2200 m in elevation. Glaciers and ice fields surround the mineral deposits to the north, south, and east.

Recent and rapid deglaciation has resulted in over-steepened and unstable slopes in many areas. Recently deglaciated areas typically have limited soil development, consisting of glacial till and colluvium. Lower elevation areas with mature vegetation may have a well developed organic soil layer.

Avalanche chutes are common throughout the area and management of snow avalanches will be a concern for the development and operation of the project. Similarly, project design may have to consider the potential for debris flows in some areas.

The baseline study program has assessed and mapped terrain and soil, including baseline metal content of soils, for key components of the project area in 2008 and 2009.

GEOLOGICAL SETTING

Refer to Section 7.0 for details on the geological setting of the KSM property.

24.3.2 ACID ROCK DRAINAGE

Baseline sampling indicates a strong chemical signature of acidic drainage resulting from the oxidation of naturally occurring sulphide minerals and includes elevated sulphate, iron, and copper. Seeps around natural gossans indicate natural acid conditions with pH in the 2.5 to 3.0 range. Neutral pH water exhibits characteristics of acidic water precipitation such as white aluminum oxyhydroxide and iron staining. The acidic drainage in the area has occurred naturally and has been present over a geological time scale.

Historical evaluations by Placer Dome looked at the potential for ARD on the Kerr deposit and, to a lesser extent, the Sulphurets deposit. The assessments included ABA and metals analysis for various rock types. The ABA results indicate that there is high sulphur content in the mineralized rock at the site and that the mineralized rock appears to have a high probability for acid generation.

The strong correlation of neutralization potential (NP) and carbon content in the rock indicated that most of the NP in the rock will be useable. There is some variation in NP between rock types, with sericite having the most NP while chlorite/anhydrite altered intrusions have the least. However, although the mineralized rock contains NPAG sulphate/sulphur, the net NP is less than zero, indicating that it is likely that it will be acid-generating. Drill core samples exhibit iron staining that suggests that oxidation has occurred to depths of several tens of metres below ground surface.

Baseline ABA and metals analyses for various rock types are currently underway, addressing all three deposits (Kerr, Sulphurets, and Mitchell) in order to refine characterization of potential ARD concerns. The Mitchell Zone ARD potential will be assessed in detail. 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.

24.3.3 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 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. Snowfalls and strong winds can be expected from early-October until mid-April with temperatures varying widely between 0° and -40°C. Snow pack ranges from 1 to 2 m but high winds can create snowdrifts up to 10 m.

Seabridge has established two full meteorological stations to collect site specific weather data – one near the proposed open pits and another near the proposed TMF. In addition, two wind monitoring stations have been established to facilitate modelling of wind dispersion of dust; precipitation gradient monitoring has been established to forecast runoff for facility design. Snow courses have been established at several sites in the project area.

Air quality is being monitored through the use of nine dust fall monitoring stations located strategically throughout the project area. It is anticipated that baseline levels of air contaminants will be very low, consistent with undisturbed natural areas.

Ambient noise levels are also very low as would be expected from a remote site.

24.3.4 WATER RESOURCES

FLOW VOLUMES

The project area drains to two major river systems, the Unuk and the Bell-Irving. The Unuk River flows into Alaska within 30 km of the project area and the Bell-Irving River flows eventually into the Nass River, which in turn flows into the Skeena River before reaching the Pacific Ocean. Proximity to the coast, relatively high precipitation rates, mountainous terrain, and the presence of glaciers result in high amounts of runoff within the project area.

Some historical hydrometric data is available for this region from the Water Survey of Canada, including flow data from the Unuk River and the Bell-Irving River. However, most of the regional data is historical (both the Unuk River and Bell-Irving River data collection sites were decommissioned in 1996) and from relatively large watersheds; therefore, the data may not represent current hydrological conditions of the sites of interest.

The area of the open pits is drained by Sulphurets Creek and its tributary Mitchell Creek, which flows to the Unuk River. Both creeks originate from glaciers. These glaciers are rapidly receding, leading to very high summer flows. It will be necessary to divert water from Mitchell Creek to enable excavation of the Mitchell pit. This water will be returned to the Mitchell-Sulphurets system.

The proposed location for the TMF and two associated dam structures will have potential impacts on the drainages of Teigen and Treaty creeks, both of which are tributaries of the Bell-Irving River. Water would be diverted from Treaty Creek to Teigen Creek to simplify water management and centralize any required treatment.

Sixteen hydrometric stations have been installed on key drainages within and adjacent to the project site to collect local information that (combined with a regional hydrological assessment) will provide the data required for the baseline studies and the development of the PFS. In addition, a glacier monitoring program will provide data to predict the potential influence of glaciers on project water balance.

WATER QUALITY

Little historical baseline water quality information is available for the KSM area. Seabridge has initiated a comprehensive assessment of water and sediment quality and related aquatic ecology. Sampling was conducted on a regular basis throughout 2008 and has continued during the 2009 field season. Information available to date indicates that the drainages of Mitchell and Sulphurets creeks are naturally affected by the concentration of metals occurring in the mineralized zones.

Naturally-occurring seeps in the mineralized zones have pH values in the range of 2.5 to 3.0 and exhibit elevated levels of sulphate, iron, and copper. The geochemistry of these seeps is characteristic of metal leaching/ARD caused by the oxidation of naturally occurring sulphide minerals. Previous work on the Kerr and Sulphurets deposits by Placer Dome indicated that there is high sulphur content in the mineralized rock at the site and that the mineralized rock appears to have a high probability for acid generation. Mitchell Creek is strongly discoloured by iron staining of the substrate and suspended sediments for several kilometres downstream of the Mitchell deposit.

Both Sulphurets and Mitchell creeks have high suspended solids levels, resulting from sediment released by upstream glaciers. Retention time in Sulphurets Lake is not sufficient to clarify the water of Sulphurets Creek and the plume of sediment from Sulphurets Creek can be seen for a considerable distance below its confluence with the Unuk River. It is expected that the high sediment loads, high metal content, lack of stream side vegetation and low temperatures relating to their glacier sources will result in low aquatic productivity for these creeks.

24.3.5 FISHERIES

The Unuk and Bell-Irving rivers are large river systems with high fisheries and cultural values. They provide important spawning routes for Pacific salmon (all five species) and anadromous steelhead trout, as well as habitat for resident trout (cutthroat, rainbow), resident char (e.g. Dolly Varden and/or bull trout), and whitefish.

The fisheries resources and fish habitat of the potentially affected tributaries of the Unuk and Bell-Irving rivers are currently being assessed as part of a two-year baseline program. The mainstem of Treaty and Teigen creeks hosts high quality spawning, rearing, and overwintering habitat. The tributaries of these two creeks that drain the valley currently being considered as a TMF location are known to be occupied by Dolly Varden and/or bull trout. A short distance upstream of the confluence with the Unuk River, a cascade on Sulphurets Creek likely inhibits the passage of migratory fish. Sampling in both 2008 and 2009 was not successful in locating fish anywhere upstream of this barrier.

Mitigation measures and any compensation that may be due as a result of fisheries impacts related to the project will be discussed and developed in consultation with the appropriate agencies and relevant Aboriginal groups.

24.3.6 ECOSYSTEMS AND VEGETATION

The KSM 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.

Seabridge has commenced a systematic mapping of the project area using both Predictive Ecosystem Mapping (PEM) and Terrestrial Ecosystem Mapping (TEM) methods. The PEM method is being used over the whole of the project area; whereas, the more intensive TEM method will be restricted to areas of disturbance such as access roads, pits, plant site, and the TMF.

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, Seabridge is mapping plant communities and plant species of conservation concern to guide the environmental assessment and project design. Plant tissue is being collected and analyzed to establish baseline metal content.

24.3.7 WETLANDS

The project encompasses several areas of wetland along the proposed access routes and in the proposed TMF location. Wetlands in Canada are valued ecosystem components under the *Canadian Environmental Assessment Act*. They are conserved and managed through federal initiatives such as the Federal Policy on Wetland Conservation; the objective of which is to "promote the conservation of Canada's wetlands to sustain their ecological and socioeconomic functions, now and in the future" (Government of Canada, 1991).

Baseline studies currently underway include the 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 are also being collected from potentially affected wetland sites.

24.3.8 WILDLIFE

The region encompassing the proposed project is home to many terrestrial wildlife species including black and grizzly bears, mountain goats, moose, avian species (e.g. bird of prey, and migratory songbirds and waterfowl), amphibian species (e.g. western toad), small mammals, and marmots. Comprehensive baseline surveys have been initiated to more fully characterize the wildlife populations and distribution, and to understand their significance to the area. Habitat suitability mapping for several species is being conducted in parallel with the PEM and TEM work.

Seabridge is applying appropriate due diligence to consider the potential impacts on species, especially listed species, which could occur in the area. A number of listed species are known or expected to occur in the proposed project area: wolverine and fisher, tailed frogs, western toad, and rusty blackbird (identified through past work on other mining projects in the region). Species of concern include those that may not be of conservation concern but are of regional importance for other reasons identified in the Cassiar Iskut-Stikine Land and Resource Management Plan (LRMP) as well as moose, mountain goat, marmot/arctic ground squirrel, and grizzly bear, amongst others.

Grizzly bears have been observed in the project study area. These bears feed on salmon during the salmon spawning period and eat vegetation and small mammals such as marmots during the rest of the year. Black bears are ubiquitous throughout the area. An ongoing grizzly bear survey will indicate grizzly bear population and distribution.

Moose are important in the region from both ecosystem and socioeconomic (e.g. hunting) perspectives. Low elevation and wetland areas are important moose habitat in the study area. Moose populations are being assessed using standard winter aerial survey methods.

Mountain goat usage of the project area is well documented. They are important from both ecosystem and socioeconomic (e.g. hunting) perspectives and are especially sensitive to development. Part of the project area was officially designated as ungulate winter range for mountain goats in late 2008. The use of helicopters within specific goat sensitive areas is being managed to minimize potential adverse effects on this population. Aerial surveys following government protocols are being used assess mountain goat populations to aid in the development of appropriate mitigation.

Breeding birds and raptors have been documented in the project areas and are being given special attention due to statutory protection and conservation concerns. They are included in the ongoing baseline surveys using standard methods approved by the BC government.

24.3.9 TRADITIONAL KNOWLEDGE AND TRADITIONAL LAND USE

The KSM Project site is located on Crown land in an area historically used by several First Nations groups. The Project lies within the boundaries of the Nass Area, as defined in the Nisga'a Final Agreement.

Traditional Knowledge/Traditional Use (TK/TU) studies are currently underway on behalf of Seabridge and will involve the potentially affected First Nations and Treaty Nations. It is anticipated that these studies will identify areas and seasons where a 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.

24.3.10 NON-ABORIGINAL LAND USE

The western part of the KSM Project area is included in the Cassiar Iskut-Stikine Land and Resource Management Area. The Cassiar Iskut-Stikine LRMP was approved by the province in 2000. The LRMP is a sub-regional integrated resource plan that establishes the framework for land use and resource management objectives and strategies, and provides a basis for more detailed management planning. The LRMP outlines the management direction, research and inventory priorities, and economic strategies for the Cassiar Iskut-Stikine area, and presents an implementation and monitoring plan to reach the established objectives. Detailed planning initiatives and resulting products are expected to be guided by, and be consistent with, the LRMP management direction.

A small part of the project area – a section of the proposed ore transport tunnel alignment – lies within the boundaries of the South Nass Sustainable Resource Management Plan area, currently in the planning process.

The KSM 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 the nearby Brucejack Lake, south of the Sulphurets deposit, in the 1990s was supported by a temporary road from Bowser Lake and over Knipple Glacier.

The nearby Bell II Lodge on Highway 37 has a successful heli-ski operation that covers a very broad area, including several runs within the area of the project. Guide outfitter territories and trap-lines exist in the project area and commercial recreational and fishing guide territories also exist there. The relative remoteness of the site suggests that recreational hunting and fishing is fairly limited in the immediate project area.

Commercial timber harvesting has occurred near Highway 37 about 10 km to the east of the project site. Further timber harvesting in the project area is possible subject to a viable market for the timber.

24.3.11 VISUAL AND AESTHETIC RESOURCES

The KSM Project is located in a relatively remote and undisturbed area characterized by rugged mountains, glaciers, untouched forest, and wild rivers. The nearest road is Highway 37, about 10 km to the east of the proposed TMF. The TMF will not be visible from the highway, although parts of the access road and transmission line may be seen.

The controlled-access Eskay Mine road terminates about 20 km to the north of the proposed pits. The mine will be located in an isolated area that is not visible from the Eskay Mine road. Potential travellers on the Unuk River would be able to observe the bridge that will be required to travel from the Eskay Mine road to the KSM Project but such travellers are rare. It is reported that perhaps one commercial raft trip per year occurs on the river.

24.3.12 ARCHAEOLOGY AND HERITAGE RESOURCES

Preliminary archaeological assessments have found evidence of short term historic hunting camps near the proposed midpoint tunnel portals. Ongoing archaeological assessments will determine the presence of artifacts or sites, and conduct any required mitigation prior to any major disturbance being created.

Several small log buildings have been reported in the general project area. It is believed that most of these buildings were constructed in relatively recent times by trappers or placer miners, although one building reportedly dates from the operation of the Yukon telegraph line.

24.4 Consultation Activities

Community engagement and consultation is fundamental to the success of the proposed KSM Project and will take place during the project's planning and regulatory review, construction, and operations phases. Prior to beginning the BCEAA process, Seabridge (at the direction of the BCEAO) initiated project and company introductions with the potentially affected Treaty and First Nation groups. Subsequent consultation activities in the form of information sharing are occurring and will continue to take place 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.

The following section lists identified consultation groups and describes the project's consultation program activities.

24.4.1 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 environmental assessment process. Public consultation measures proposed for the project are compliant with the *Public Consultation Policy Regulation*, *BC Reg. 373/2002*. The consultation process is structured to address the needs and interest of all required consultation groups; it provides opportunities for community interests and concerns to be brought forth regarding the proposed project. The process is also a means through which issues with respect to the project can be resolved. The BCEAA and CEAA processes jointly provide a mechanism to ensure that the issues and concerns of all consultation groups are considered and incorporated in project planning and development.

24.4.2 Consultation Groups

Treaty and First Nations

Seabridge has been delegated the responsibility of information sharing with potentially affected Treaty and First Nations. This process has been initiated with the potentially affected Treaty and First Nations, as identified by the provincial Crown, and will continue.

Government

Seabridge has initiated and will continue engagement and collaboration 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

United States and State of Alaska regulators, as well as US federally recognized tribes will be engaged through the BCEAA process.

Public and Stakeholders

Seabridge will consult with the public and relevant stakeholder groups[1], including:

- land tenure holders
- trappers
- guide outfitters
- recreation and tourism businesses
- economic development organizations
- businesses and contractors (e.g. suppliers and service providers)
- special interest groups (e.g. environmental, labour, social, health, and recreation).

[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 KSM Project.

24.4.3 CONSULTATION ACTIVITIES AND APPROACH

ACTIVITIES

Seabridge has initiated a consultation program that is intended to be relevant and useful to each consultation group. The proposed KSM Project consultation program will include:

- government agency, Treaty Nation, and First Nations participation in the BCEAO technical working group meetings
- leadership meetings
- community meetings
- information distribution (project information and updates; events and regulatory notices)
- focus groups and workshops
- communication tracking
- issue identification and resolution.

APPROACH

The consultation activities will occur through an approach that reflects the BCEAO and CEAA consultation requirements, as well as Seabridge's goals for meaningful and sustainable relationships with the leaders and community members implicated in the KSM Project.

This approach will include:

- early engagement and consultation
- opportunity and support for participation
- provision of accessible information
- intercultural sensitivity
- transparency
- accountability
- trust, respect, and long-term relationship building.

24.5 SOCIOECONOMIC SETTING

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

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

The region has a large dependence on primary resource industries; mining and forestry are the predominant industries. Mineral exploration activity has shown significant growth and the mining industry represents a significant source of employment. Due to the large 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 KSM 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. Seabridge is working with and intends to continue to work with Treaty Nation and First Nations groups, and members of local communities to maximize benefits through employment and business opportunities, training and skills development programs.

Sudden and rapid economic growth such as that which may occur during the initial construction and operation of the KSM Project can put pressure on local community resources such as housing, roads, schools, as well as social and medical services. Seabridge believes that coordinated planning among local communities and governments, coupled with ongoing consultation and communication, provides a strong framework for early identification of adverse effects, finding accepted solutions, and enhancing benefits.

The following northwestern BC socioeconomic setting is compiled from the *Northwest BC Mining Projects Socio Economic Impact Assessment*, prepared in 2005 for the Ministry of Small Business and Economic Development, updated using data from the 2006 Census of Canada. Seabridge is conducting further socioeconomic baseline studies to provide current information for the environmental assessment required by the province.

24.5.1 Highway 16 Corridor

Highway 16 extends from the Prince Rupert port eastwards to Terrace, Hazelton, Smithers, and Prince George. The Canadian National Railway (CNR) also follows this corridor. Most of the communities along this corridor are discussed in this section.

The Highway 16 corridor is recovering from the economic downturn of the 1990s and has excess capacity with respect to social service infrastructure. The respective communities are incorporated providing a framework and capacity to:

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

Terrace

Terrace lies in a significant location along the freight corridor, near the junction of Highways 16 and 37. Neighbouring communities include Kitimat (64 km to the south), and Prince Rupert (140 km west).

With a population of 11,320 in 2006, Terrace is an important regional service centre for trade and public administration. Terrace is also the service centre for Kitimat, a community of 8,987 people. It exhibits a resource-based economy, relying heavily on forestry employment and the provision of mining and other services for the region. The highest proportion of jobs is now provided by the service sector (department stores, grocery stores, hotels, etc). The unemployment rate was 9.3% in 2006.

Terrace's role as a service centre is supported by infrastructure such as the Northwest Regional Airport, CNR, Northwest Community College, a University of Northern British Columbia campus, and Mills Memorial Hospital.

Smithers

Smithers is considered the regional service centre for the Bulkley Valley given its strategic location along the routes of Highway 16 and the CNR, approximately halfway between the cities of Prince Rupert and Prince George. The town is the divisional point for the CNR and, with its regional airport runway expansion, has become a hub for a range of activities across northwest BC.

The town has strong forestry and public service sectors, and (indicative of its service-based role) provides a range of commercial, business, administrative, recreational, and cultural services for much of the Bulkley Valley. There has been increased economic diversification over recent years, although the town continues to be dominated by the forestry and public service sectors. Smithers acts as a staging site for mineral exploration, access to various existing mines, as well as for visitors interested in outdoor activities.

The 2006 population was estimated to be 5,145, down from 5,900 in 1996. The unemployment rate of 8.5% is slightly higher than the provincial average of 6.0.

THE HAZELTONS

Located to the northwest of Smithers are the Hazeltons: Old Hazelton, New Hazelton, South Hazelton, and Kispiox. The Hazelton area is centrally located within First Nations land claims area. The District of New Hazelton is the principal commercial, administrative, and retail centre. There is heavy reliance on the public sector and logging for employment. The 2006 unemployment rate for New Hazelton was 19.4% compared to 6.0% for the province.

The population of New Hazelton declined 16.4% from 2001 to 2006 to 627 people. The other communities, which are mostly First Nations, declined less or remained about the same.

24.5.2 *HIGHWAY 37 CORRIDOR*

Highway 37 connects with Highway 16 at Kitwanga and runs northwards to the Yukon border. Highway 37A to the Port of Stewart connects to Highway 37 at Meziadin. 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 are heavily reliant on the public sector and mining for employment. Since 1996, Highway 37 communities have experienced an overall decline in population.

STEWART

Stewart is located 60 km west of Meziadin junction on the west coast of BC, at the terminus of the 145 km Portland Canal and 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.

While largely resource-based, Stewart's economy is more diverse than those of the other Highway 37 communities. Main recent employers of community residents included the Eskay Creek and Huckleberry mines, although the Eskay Creek mine closed early in 2008. With a historically strong base in the forest sector, Stewart acts as an export centre for raw logs. It is also known as one of the gateways to Alaska and receives a reasonable amount of tourism. The town experiences strong economic variations between the seasons. Many hotels, eateries, and small businesses shut down for the winter.

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. A number of services have also been lost in the past few years including the bank, pharmacy, and the downgrading of the hospital to a Health Centre. Stewart has ample capacity in physical infrastructure, housing stock and developable land to support new northwest developments.

The population of Stewart has declined in tandem with the general decrease in mining activity in the region. The population reached a high of 1,837 in 1982, and declined to 496 in 2006. The unemployment rate is 8.2%. While most residents are employed, current employment opportunities are rare and many residents have been driven away in search of work.

DEASE LAKE

Dease Lake is located 83 km north of Iskut, and 236 km south of the Alaska Highway. As the largest settlement along Highway 37, it acts as the primary service centre for the region. The town of Dease Lake was established as a Hudson's Bay trading post in 1838. It is a regional centre for some government services including health, police, and Northern Lights College. The First Nations and non-Aboriginal communities are well integrated and co-dependant, often relying on each other to fulfill needs and opportunities. The First Nations reserve is located near the north end of the community.

The population declined significantly since the 1990s as government services were centralized in Smithers. In 2006, the Dease Lake area had a population of approximately 452, including 68 living on the nearby First Nations reserve. Due to a high number of people employed in seasonal work, the population fluctuates during the year.

The Dease Lake community is heavily dependent on the resource base of the Stikine region. Mining is the primary industry in the area, although guide-outfitting, hunting, fishing, and wilderness tourism are also significant industries. A First Nations development corporation is headquartered in Dease Lake and has employees in Dease Lake, Telegraph Creek, and Iskut. The unemployment rate is currently 22.3%.

ISKUT

Iskut is a small, rural, primarily-Aboriginal community located along Highway 37 approximately 252 km north of Meziadin Junction, and 83 km south of Dease Lake. The majority of the approximately 335 residents are members of a First Nations community. Most of the community resides on Iskut IR 6.

The unemployment rate was 24.2% in 2006. Major employers include Band administration, Iskut Valley Health Services (IVHS), and the Klappan Independent Day School. Iskut community members have also been employed through various mineral exploration activities. A number of Iskut residents worked until recently at the nearby Eskay Creek mine, on a two-week rotating shift basis. The Eskay Creek mine has now closed. Iskut has a post office, gas station, grocery store, Band office, and serves as a staging area for nearby wilderness parks. Eddontennajon is a small non-native community adjacent to Iskut.

24.5.3 NORTHWEST TRANSMISSION LINE

In 2007, the province of BC announced that a new 287 kV transmission line would be constructed from near Terrace to Bob Quinn Lake following the Highway 37 corridor. This line would replace the existing 128 kV transmission line between Terrace and Meziadin and extend the electricity grid northwards into a previously unserviced area. The transmission line will provide high voltage electricity to within 10 to 15 km of the KSM Project site.

The environmental assessment for the proposed extension of the provincial electricity grid to Bob Quinn Lake is ongoing, with the BC government acting as the proponent.

24.6 DESIGN GUIDANCE

24.6.1 PROJECT DEVELOPMENT PHILOSOPHY

Seabridge intends the KSM Project to be a showcase of sustainable mining practices. 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.

24.6.2 PRECAUTIONARY PRINCIPLE

The 1992 Rio Declaration on Environment and Development defined the precautionary principle as: “Where there are threats of serious or irreversible damage, lack of full scientific certainty shall not be used as a reason for postponing cost-effective measures to prevent environmental degradation.”

Seabridge 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.

24.6.3 INTEGRATION OF TRADITIONAL KNOWLEDGE

Seabridge respects the Traditional Knowledge of the Aboriginal peoples who have historically occupied or used the project area. Seabridge recognizes that it has significant opportunity to learn from people who may have generations of accumulated experience regarding the character of the plants and animals and the spiritual significance of the area.

Traditional Knowledge will guide aspects of the project, including any future changes once the mine is approved. Seabridge 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.

Seabridge is committed to a process that invites and considers input from people with Traditional Knowledge of the project area towards the environmental assessment and design of the KSM Project. Seabridge is striving to establish a cooperative working relationship with all relevant Treaty and First Nations people to ensure opportunities to gather Traditional Knowledge.

24.6.4 BASELINE RESEARCH

Seabridge has initiated comprehensive baseline studies of the regional project area's atmosphere/climate, surface hydrology, aquatics, water and sediment, limnology, fish habitat and community, rock geochemistry, soils, vegetation, and wildlife to characterize the local and regional ecosystem prior to major disturbances. Archaeology, heritage, land use, cultural, Traditional Knowledge, and socioeconomic baseline studies are also being carried out to characterize the regional human environment. The methodologies for the baseline studies are being developed in consultation with regulatory agencies and Treaty and First Nations peoples of the area.

24.6.5 VALUED ECOSYSTEM COMPONENTS

Seabridge recognizes that different components of the natural and socioeconomic environments will be of special importance to local communities and other stakeholders, based upon scientific concern or cultural values. These components are widely termed valued ecosystem components (VECs) and will be given particular consideration during project assessment, planning, and design.

VECs applicable to the project will be identified through a comprehensive issues scoping exercise, which will include consultation with federal and provincial regulatory bodies, local Treaty and First Nations, and other stakeholders.

24.6.6 ENVIRONMENTAL ASSESSMENT STRATEGY AND SCOPE

The environmental assessment of the KSM Project that is required under federal and provincial legislation will focus on the identified VECs to ensure the primary concerns of all stakeholders are addressed. The methodology to be applied has been developed to ensure a comprehensive, logical, and transparent assessment and involves examination of the potential effects of each mine component through all project stages.

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

24.6.7 ECOSYSTEM INTEGRITY

The project area ecosystem is relatively undisturbed by human activities, although it is not static. Glacier retreat and relatively recent (within the last 10,000 years) volcanoes, along with frequent landslides, debris flows, and snow avalanches, continue to modify the landscape.

Seabridge'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 unmitigatable adverse impacts. 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.

24.6.8 BIODIVERSITY AND PROTECTED SPECIES

Seabridge is committed to making every reasonable effort toward maintaining biodiversity in the project area. Biodiversity is defined by the BC Ministry of Forests and Range as "the diversity of plants, animals and other living organisms in all their forms and levels of organization, and includes the diversity of genes, species and ecosystems, as well as the evolutionary and functional processes that link them".

Species diversity refers to the variety and abundance of different types of organisms within a region. Ecosystem diversity refers to the variety of ecosystems or habitats within a region. For the purpose of the environmental assessment of the KSM Project, biodiversity will be considered at the species and ecosystem (habitat) levels.

Increasing human population worldwide and the demand for resources to feed, clothe, house, and entertain that population has imposed huge pressure on the natural environment. In many cases biodiversity has suffered dramatically, and many species and their habitats have become threatened or extinct due to habitat loss.

The Canadian *Species at Risk Act* was created to protect wildlife species from becoming extinct in two ways: by providing for the recovery of species at risk due to human activity, and by ensuring through sound management that species of special concern do not become endangered or threatened. It includes prohibitions against killing, harming, harassing, capturing or taking species at risk, and against destroying their critical habitats.

The objectives of the "Convention on Biological Diversity" (the Convention) signed by Canada in 1992 are to conserve biodiversity, to use biological resources in a sustainable manner, and to share benefits resulting from the use of genetic resources. The Convention recognizes environmental assessment as an important decision-making tool for the protection of biodiversity. Canada issued the Canadian

Biodiversity Strategy in 1995 in response to the Convention. Although the Biodiversity Strategy does not explicitly recommend a strategic plan or program for the mining sector, it does address associated issues such as ecosystem rehabilitation, reduction or elimination of harmful substance release to the environment, improving methods for monitoring ecosystems, and identifying mechanisms to use Traditional Knowledge.

Biodiversity is not an isolated concept but a part of project planning (mitigations and monitoring), environmental effects analysis, and consideration of sustainability. Seabridge is applying this concept in carrying out scoping, effects analysis, project design and mitigation, determination of effects significance, and monitoring. This concept will be integrated both implicitly and explicitly throughout the environmental assessment.

The impact of mining activities, access road development, and related mitigation practices on many species is unknown and certain practices that benefit some species are often detrimental to others. Seabridge will use an ecosystem management approach that provides suitable habitat conditions for all native species. In this way, habitat diversity is used as a surrogate to maintain biodiversity; however, at the same time, special efforts may be needed to protect the habitat of species known to be at risk such as threatened, endangered, or regionally important species.

Seabridge will engage specific strategies for addressing these species in the development of the project.

24.6.9 ENVIRONMENTAL STANDARDS

Seabridge will design, construct, operate, and decommission the KSM Project to meet all applicable BC and Canadian environmental and safety standards and practices. Some of the pertinent federal and provincial legislation that establish or enable these standards and practices are outlined below:

- *Environment and Land Use Act (BC)*
- *Environmental Management Act (BC)*
- *Health Act (BC)*
- *Forest Act (BC)*
- *Forest and Range Practices Act (BC)*
- *Fisheries Act (BC)*
- *Land Act (BC)*
- *Mines Act (BC)*
- *Soil Conservation Act (BC)*
- *Water Act (BC)*

- *Wildlife Act (BC)*
- *Canadian Environmental Protection Act*
- *Canada Transportation Act*
- *Fisheries Act*
- *Transportation of Dangerous Goods Act*
- *Workplace Hazardous Materials Information System (WHIMIS) Safety Act*.

A key commitment in meeting these standards will be the development and implementation of an Environmental Management System (EMS). The EMS will define the process by which compliance will consistently be met and demonstrated, and will include ongoing monitoring and reporting to relevant parties.

24.6.10 Design for Social and Community Requirements

Seabridge is striving to establish strong collaborative and cooperative relationships with relevant Treaty and First Nations people (as identified by the Crown), other communities, and interested stakeholders. Seabridge recognizes that its social licence to operate is dependent on being a good corporate citizen and neighbours to all groups with interests in the region.

Following best practices in the industry, the community is committed to a process to ensure that communities benefit from employment, training, and contracting opportunities, that potential negative impacts are mitigated, and that any commitments and benefit agreements are respected. Seabridge will meet its requirements through the development and implementation of a Social and Community Management System (SCMS). The SCMS will define the process by which the company will maintain its involvement and on-going commitments to communities and stakeholders.

24.7 Water Management

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

As such, Seabridge will develop a comprehensive water management plan that applies to all mining activities undertaken during all phases of the KSM Project. The main objective of this water management plan will be to regulate the movement of water in and around the mine site to ensure long term environmental protection.

The goals of this management plan will be to:

- provide a basis for management of the freshwater on the site, especially with the changes to flow pathways and drainage areas

- protect ecologically sensitive sites and resources, and avoid harmful impacts on fish and wildlife habitat
- provide and retain water for mine operations
- define required environmental control structures
- manage water to ensure that any discharges meet and/or exceed the permitted water quality levels and guidelines.

The strategies for water management include diverting surface water from disturbed areas, protecting disturbed areas from water erosion, collecting surface water from disturbed areas and treating to meet discharge standards prior to release, minimizing the use of fresh water, recycling water wherever possible to minimize the amount of water released, and monitoring the composition of release water and treating it to remove or control contaminants as required to meet discharge standards.

Diversion channels or tunnels will be constructed to direct runoff away from disturbed areas. Seabridge is proposing a diversion tunnel to collect water discharging from the base of the Mitchell Glacier and divert it away from entering the Mitchell Pit. This tunnel will eventually discharge the diverted water back into the Mitchell-Sulphurets drainage basin. Channels will likely be constructed to collect surface runoff above all pit high walls, waste rock dumps, the plant site, and the TMF, where permitted by terrain characteristics. These diversions will isolate surface water from exposed metal rich rock and tailing and allow the runoff to be released with little or no treatment.

Diversion structures will be designed to manage freshet flows and 1-in-100-year storm events. Greater capacity will be provided if required based on an assessment of the consequences of failure. Lesser capacity may be provided where overflows can be stored and managed by other downstream structures, such as the TMF.

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 MMER limits total suspended solids to 15 mg/L. Flocculation may be required to meet discharge standards in some instances.

Where possible, reclaim water will be used in preference to fresh water for makeup purposes in order to minimize the withdrawal of fresh water from natural systems and reduce the volume of contact water discharged to the environment. Contact water may require treatment.

The quality of water in streams affected by the project, and of all discharges, will be monitored on a regular basis.

24.7.1 Water Supply

Process water will be obtained from the TMF.

Potable water for use in office and accommodation facilities and kitchens will likely be sourced from water diversions constructed around the perimeter of the plant site, waste rock dump, TMF and other infrastructure. Makeup water for gland water and other selected applications in the process plant may also be derived from water diversions, depending upon the quality and seasonal availability of water available from other sources. During the winter months, well water from a field of wells near the plant site may be needed to supply fresh water for process make up and domestic use at the plant and camp facility.

24.7.2 Internal Recycle Strategies

Process water will be recycled where feasible to reduce the volumes of water released to the environment. Water will be recycled to the process plant from the tailing thickener (if used) and concentrate thickeners and filters. Tailing supernatant will be recovered from the TMF using barge mounted pumps and returned to the plant. It is anticipated that the TMF should provide adequate water for most processing requirements.

24.7.3 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-100-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. Lesser capacity may be provided where overflows can be stored and managed by other downstream structures, such as the TMF.

24.7.4 Discharge Strategy and Quality

Discharges will be controlled where feasible to mimic natural flows in order to minimize adverse effects on local hydrological regimes. Some modification of natural flows will be required from time to time to avoid disturbed areas and to optimize dilution in order to consistently meet discharge standards.

Discharges from the mine will be managed to meet the federal government MMERs and negotiated provincial water quality objectives.

24.7.5 Construction Water Management

Water management risks are often highest during construction when facilities for diversion, collection, and control of runoff are least reliable. Seabridge 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.

24.8 Waste Management

24.8.1 *Tailing Management*

It is assumed that the high sulphide content of the pyrite tailing from the process plant will cause this material to quickly oxidize and generate acid. The proposed solution to this acid generation, and potential subsequent metal leaching, is to store the tailing permanently under water where oxidation is vastly reduced or eliminated. As described in Section 19.1, the TMF is designed to isolate the pyrite tailing in a stable subaqueous environment in perpetuity.

In order to ensure that the TMF continuously meets its objectives, Seabridge will develop and implement a tailing management plan. The goals of this management plan are to:

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

Tests are currently underway to characterize the tailing and supernatant in order to estimate the rate of oxidation and resulting water quality. This information will guide planning for tailing water management.

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

Ditches will be constructed on both sides of the TMF where feasible to divert surface flows. Flows in the diversions will be directed seasonally either away from or towards the impoundments to keep the water balance close to that of a zero discharge facility. A pump barge will pump any excess water to the crest of the North

Dam where it will be piped down past the North Seepage Collection Dam and released into the Teigen Creek tributary.

At closure, the TMF will be configured as a "dry" structure with minimal pond/wetland area, and revegetated with grasses and trees. Surface drainage within the impoundment will be directed towards a closure spillway excavated in rock at the west abutment of the North Dam. 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 of Teigen Creek meet permit requirements.

24.8.2 WASTE ROCK AND OVERBURDEN MANAGEMENT

The KSM Project will potentially generate 2.5 billion tonnes of waste rock over the anticipated life of the mine. Waste rock will be segregated according to its potential to generate acid and leach metals. A comprehensive testing program using blast hole cuttings will be established to characterize all rock removed from the pits. This program will be integrated with the ore control program to ensure that rock is correctly directed to the process plant, the NPAG dump, or the PAG storage area.

The PAG waste rock dump will be located downstream of the Mitchell pit and will be designed to isolate the PAG waste rock from ground water and surface runoff. Leachate resulting from internal moisture and precipitation will flow to the Mitchell pit where it will be treated if necessary prior to release. A conventional high density sludge treatment plant will be employed for the treatment.

Water management for the rock dumps will include measures to reduce water contact with PAG rock. Options being examined include basal drains of NPAG rock, and low permeability covers on top of the dumps and tunnels to divert creek flows around the dumps during operations. After closure, surface diversions and spillways will route water around the dumps.

The current reclamation cover concept is a 1 m-thick low permeability till layer to reduce infiltration overlain with a 3 m-layer of NPAG rock to protect the till from erosion and freeze-thaw damage. The dump designs include placement strategies to reduce infiltration. These strategies include bottom-up constructed shells of more compact trafficked rock layers stripped from lower elevations in the pit that are placed to encapsulate looser, high dumped rock stripped and dumped from higher elevations above the valley. Designs for the rock dumps allow for progressive covering and reclamation during mining operations.

Some overburden and glacial till will be stored for later use as a cover for the waste rock dumps to create a moisture barrier and a growth medium for eventual revegetation. Much of the current surface area of the deposits is barren of vegetation due to the relatively recent ice recession.

24.8.3 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. Seabridge 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.

24.8.4 NON-HAZARDOUS WASTE MANAGEMENT

Seabridge will initiate a comprehensive waste management program prior to the inception of construction of the project to minimize any potential adverse effects to the environment, including wildlife and wildlife habitat, while at the same time ensuring compliance with regulatory requirements, permit and licence obligations, and Seabridge Environmental Policy. It is important to establish a waste management culture from the outset of the project. The program will extend from the procurement process, where excess packaging will be avoided, through to decommissioning of the project. The mantra of “Reduce, Reuse, Recycle, and Recover” will be followed to address waste management. 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.

24.9 AIR EMISSION CONTROL

Air emissions can represent a substantial component of contaminant dispersion for a site. Baseline studies, utilizing two on-site meteorological stations and two separate wind monitoring stations, will characterize the atmospheric environment of the KSM Project area to allow air dispersion modelling. Mitigation will then be developed to minimize adverse impacts from emissions. Regular monitoring of emissions will assess the success of the mitigation methods and warn of any requirement to adjust the current approach.

24.9.1 EMISSIONS

Seabridge will implement an air emissions and fugitive dust management plan to ensure that the levels of air emissions and fugitive dust generated by project activities are at or lower than the regulatory requirements of the Canada and BC Ambient Air Quality Objectives to ensure the protection of biological receptors such as vegetation, fish, wildlife, and human health.

Potentially 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)
- the use of preventative maintenance to ensure optimum performance of light-duty vehicles, the diesel mining equipment, and the incinerators, thereby reducing air emissions
- the use of large haul trucks for ore and waste transport to minimize the number of trips required between the source and destination
- the use of appropriate control methods such as road watering and vehicle speed regulations to minimize the generation of fugitive dust
- the use of monitoring programs to ensure healthy work environments and protection of other biological receptors
- the use of slurry pipelines for moving crushed and ground ore and a pipeline for diesel fuel to reduce the number of haul truck trips and the consequent amount of diesel emissions and fugitive dust
- the implementation of a recycling program to reduce the amount of incinerated wastes and hence CO_2 emissions
- the segregation of waste prior to incineration to minimize toxic air emissions.

24.9.2 DUST CONTROL

Dust is generated at mining sites by many common activities including blasting, rock excavation, haulage and stockpiling, crushing and screening operations, ore and waste conveying, and vehicle travel on gravel roads. Seabridge will use a range of control and mitigation measures to reduce dust creation and dispersion. Some of these measures include the following:

- Blasting will be designed with appropriate delays and blast hole stemming to direct energy into rock breaking rather than dust creation.

- Loader and shovel operators will be instructed to minimize drop distances when moving rock in order to reduce dust creation.
- Crushing and screening operations will be enclosed and equipped with bag houses to collect dust.
- Conveyor transfer points will be enclosed and equipped with dust control systems such as water sprays or bag houses.
- Conveyors will incorporate wind covers where required.
- Haul roads and access roads will be treated for dust control. The selection of dust control methods will consider the need to avoid the use of products that may attract wildlife to roads.

24.10 Operating Plan and Costs

24.10.1 Environmental Management Systems

Seabridge will develop and implement a comprehensive EMS for the construction, operation, and closure phases of the KSM Project. The EMS will comprise a series of written plans that outline the scope of environmental management pertaining to compliance with both regulatory requirements as well as Seabridge environmental policy.

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

- air emissions and fugitive dust
- water management
- tailing and waste rock
- diesel and tailing pipelines
- concentrate loadout
- metal leaching/ARD prediction and prevention
- materials management
- erosion control and sediment
- spill contingency and emergency response
- fish and fish habitat
- wildlife management
- waste management
- access road
- archaeological and heritage site protection.

24.10.2 Social and Community Management Systems

Seabridge will develop and implement broad SCMS for the construction, operation, and closure phase of the KSM Project. The SCMS will comprise an ongoing consultation plan and community development plans to be developed through a series of written agreements and relationship building initiatives with First Nations communities. Monitoring and oversight of the SCMS will require a team of staff responsible for coordinating community development initiatives, training, communications and commitment tracking, and fund management.

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

- impact benefit agreements
- community development plan (support for selected local education, health, and social infrastructure, etc.)
- community engagement meetings
- training
- participation in community events
- reporting and feedback mechanisms.

24.10.3 Environmental and Social Management Capital Costs

Capital costs for environmental and social management functions will total about US$3 M plus the cost of the water collection systems and treatment plants.

Environmental management cost items include laboratory equipment for environmental-specific analyses, office and field equipment for eight employees, a vehicle for each of the mine site and plant/tailing management site, and water collection systems and treatment facilities.

Social management capital costs will include any required up-front payments under the yet to be negotiated impact and benefits agreements with area First Nations.

24.10.4 Environmental and Social Management Operating Costs

Environmental and social management operating costs are estimated at US$5.4 M/a. Eight employees at several different levels of seniority working on a rotational basis with at least three employees on site at all times will be required for environmental monitoring including:

- federal MMER monitoring requirements
- fisheries compensation monitoring

- BC permit compliance monitoring and environmental effects monitoring
- reclamation research and monitoring.

Seabridge environmental staff will also research and advise the Mine Manager on alternative mitigation strategies as part of the mine's process of continual improvement. They will be supported by specialist consultants. Outside laboratories will be required for some analyses while other more routine analyses, such as of conventional water samples, 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.

The bulk of the social management operating costs will be incurred through the payment of any annual disbursements required under yet to be negotiated Impact and Benefit Agreements (IBAs) with the relevant, local Treaty and First Nations (as identified by the Crown).

A team of employees, including a First Nations and Community Coordinator, will be required to develop and implement the SCMS. Resources will be required for ongoing community engagement meetings, community development plans (potentially including grants and scholarships), staff participation in community events, and communication feedback vehicles, such as community reports and/or newsletters. Additional resources will be needed for developing and staffing training programs for local community members. Importantly, operating costs do not include any potential separate service or joint venture contract that may be negotiated.

25.0 PROJECT EXECUTION PLAN

25.1 PROJECT SCHEDULE

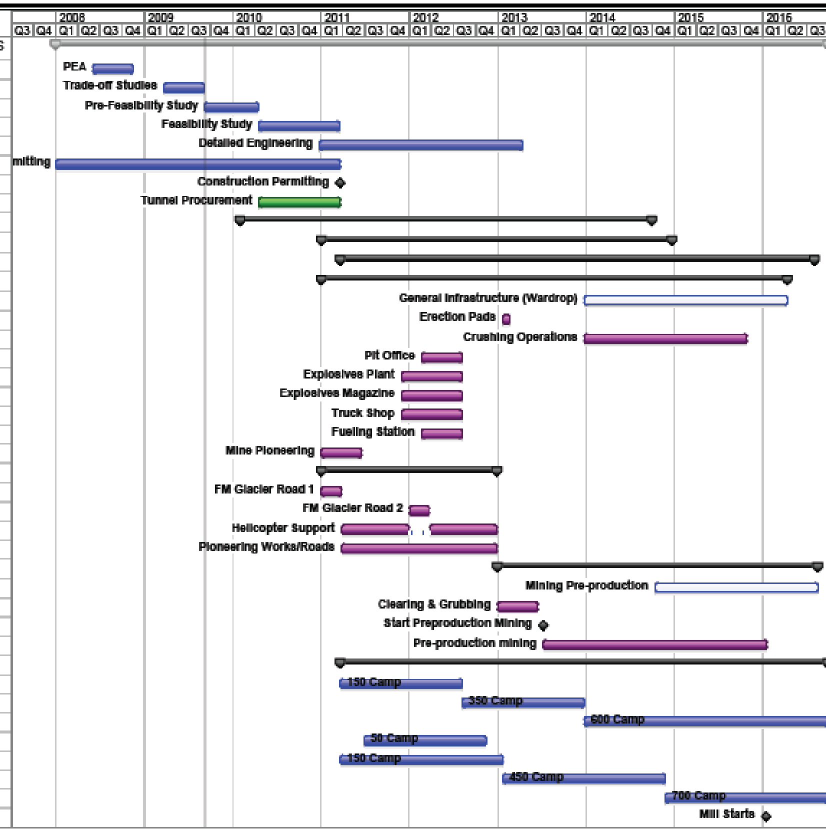
After receipt of operating permits the project will take approximately five years to complete. The environmental assessment process has already been partially completed. The longest lead items, which determine the procurement time length, are the HPGR crushers and the grinding mills.

The project schedule summary is shown in Figure 25.1. A complete detailed schedule is available in Appendix D.

As the environmental and permitting timeline is critical, other work such as bulk sampling, pilot plant testing, and detailed engineering can proceed as shown in Figure 25.1, and have some flexibility in their completion times.

Figure 25.1 Project Schedule Summary

ID		Task Name	Duration	Start	Finish
0		KSM Addendum Project Schedule	2280 days	Tue 01/01/08	Mon 26/09/16
1		1 PEA	6 mons	Mon 02/06/08	Fri 14/11/08
2		2 Trade-off Studies	6 mons	Mon 23/03/09	Fri 04/09/09
3		3 Pre-Feasibility Study	8 mons	Mon 07/09/09	Fri 16/04/10
4		4 Feasibility Study	12 mons	Mon 19/04/10	Fri 18/03/11
5		5 Detailed Engineering	30 mons	Mon 27/12/10	Fri 12/04/13
6		6 EA and Permitting	42 mons	Tue 01/01/08	Mon 21/03/11
7		7 Construction Permitting	0 days	Mon 21/03/11	Mon 21/03/11
8		8 Tunnel Procurement	12 mons	Tue 20/04/10	Mon 21/03/11
9		**9 Project Procurement**	**1216 days**	**Mon 01/02/10**	**Tue 30/09/14**
16		**10 Equipment Delivery**	**1036 days**	**Mon 03/01/11**	**Tue 23/12/14**
23		**11 Construction and Commissioning**	**1400 days**	**Tue 22/03/11**	**Mon 01/08/16**
49		**12 Infrastructure**	**1376 days**	**Mon 03/01/11**	**Mon 11/04/16**
50		*12.1 General Infrastructure (Wardrop)*	30 mons	Tue 24/12/13	Mon 11/04/16
51		12.2 Erection Pads	1 mon	Tue 22/01/13	Mon 18/02/13
52		12.3 Crushing Operations	24 mons	Tue 24/12/13	Mon 26/10/15
53		12.4 Pit Office	6 mons	Tue 21/02/12	Tue 07/08/12
54		12.5 Explosives Plant	9 mons	Tue 29/11/11	Tue 07/08/12
55		12.6 Explosives Magazine	9 mons	Tue 29/11/11	Tue 07/08/12
56		12.7 Truck Shop	9 mons	Tue 29/11/11	Tue 07/08/12
57		12.8 Fueling Station	6 mons	Tue 21/02/12	Tue 07/08/12
58		12.9 Mine Pioneering (temp office, fuel/explosive stge, maint facility)	6 mons	Mon 03/01/11	Fri 17/06/11
59		**13 Mining Pioneering**	**520 days**	**Mon 03/01/11**	**Fri 28/12/12**
60		13.1 Frank Mackie Glacier Road Open Window 1	3 mons	Mon 03/01/11	Fri 25/03/11
61		13.2 Frank Mackie Glacier Road Open Window 2	3 mons	Mon 02/01/12	Fri 23/03/12
62		13.3 Helicopter Support Period	20 mons	Mon 28/03/11	Fri 28/12/12
63		13.4 Pioneering Works/Roads	23 mons	Mon 28/03/11	Fri 28/12/12
64		**14 Mining Pre-production**	**946 days**	**Mon 31/12/12**	**Mon 15/08/16**
65		*14.1 mining pre-production*	480 days	Tue 14/10/14	Mon 15/08/16
66		14.2 Clearing & Grubbing	6 mons	Mon 31/12/12	Fri 14/06/13
67		14.3 Start Pre-Production Mining	0 days	Tue 09/07/13	Tue 09/07/13
68		14.4 Pre-production mining	33 mons	Tue 09/07/13	Mon 18/01/16
69		**15 Camps**	**1440 days**	**Tue 22/03/11**	**Mon 26/09/16**
70		15.1 Plant Camp	18 mons	Tue 22/03/11	Mon 06/08/12
71		15.2 Plant Camp 1st Expansion	18 mons	Tue 07/08/12	Mon 23/12/13
72		15.3 Plant Camp 2nd Expansion	36 mons	Tue 24/12/13	Mon 26/09/16
73		15.4 Saddle Camp	18 mons	Tue 28/06/11	Mon 12/11/12
74		15.5 Mitchell Camp	24 mons	Tue 22/03/11	Mon 21/01/13
75		15.6 Mitchell Camp 1st Expansion	24 mons	Tue 22/01/13	Mon 24/11/14
76		15.7 Mitchell Camp 2nd Expansion	24 mons	Tue 25/11/14	Mon 26/09/16
77		16 Mill Start-up	0 days	Fri 15/01/16	Fri 15/01/16



Project: KSM Addendum Project Sche
Date: Tue 08/09/09

Task | Progress | Summary | External Tasks | Deadline
Split | Milestone | Project Summary | External Milestone

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26.0 CAPITAL AND OPERATING COST ESTIMATES

26.1 CAPITAL COST ESTIMATE

An initial capital requirement of US$3.426 B is projected for the KSM Project, based on capital cost estimates developed by the following consultants:

- MMTS – mine capital costs
- KCBL – tailing and water management costs
- Thyssen/Wardrop – tunnel costs
- BVL – conveying, pumping and piping costs
- Wardrop – process plant and associated infrastructure costs
- Brazier – power supply costs
- McElhanney – access road costs.

Currencies are expressed in both Canadian and United States dollars. All costs in this section are stated in fourth quarter (Q4) 2008 prices, excluding the costs updated in second quarter (Q2) of 2009.

When it was required, certain costs in this report have been converted using a fixed currency exchange rate of Cdn$1.00 to US$0.90 (based on the 3-year average). The expected accuracy range of the capital cost estimate is +25%, -10%.

Initial capital has been designated as all capital expenditures required to produce concentrate and dore. A summary of the major capital costs is shown in Table 26.1.

Table 26.1 Capital Cost Summary

Description	Cdn$ (000)	US$ (000)
Direct Works		
Overall Site	94,000	84,000
Mining	355,000	320,000
Mine Site Crushing and Grinding	423,000	381,000
Tunnel Pumping	135,000	122,000
Plant Site Grinding and Flotation	278,000	248,000
Tailing Dam	131,000	118,000
Mitchell Teigen Tunnel*	154,000	138,000
Mitchell Diversion Tunnel	40,000	36,000
Mitchell Diversion Hydro Plant	4,000	3,000
Water Treatment	101,000	91,000
Site Services And Utilities	12,000	11,000
Ancillary Buildings	72,000	65,000
Plant Mobile Fleet	7,000	6,000
Temporary Services	134,000	121,000
Power Supply	157,000	141,000
Roads, Infrastructure & Off-Site Facilities	130,000	117,000
Subtotal	**2,633,000**	2,002,000
Indirects		
Project Indirects	717,000	645,000
Owner's Costs	50,000	45,000
Contingencies	434,000	391,000
Subtotal	1,091,000	1,081,000
TOTAL CAPITAL COST	**Cdn$3,426,000**	**US$3,083,000**

*The Mitchell Teigen Tunnel is synonymous with the Ore Haulage Tunnel

The detailed breakdown of this capital cost estimate is included in Appendix I.

26.1.1 *MINE CAPITAL COST*

Mine capital costs are derived from a combination of supplier quotes and historical data collected by MMTS. This includes the labour, maintenance, major component repairs, fuel, and consumables costs.

MMTS has provided the mining capital cost estimate. The equipment mine capital costs include delivery to site and assembly but do not include taxes or duties. Mine capital costs are shown in Table 26.2.

Table 26.2 Mine Capital Costs

	Cdn$ (000)
Pre-production (in Operating Capital)	0
Pioneering Work	3,000
Mobile Equipment	314,000
Surface Equipment	3,900
Explosive Storage	19,000
Fuel Storage and Distribution	3,600
Dewatering	1,900
Electrical	10,600
Communication	250
Safety	150
Engineering Equipment	100
Dispatch Offices	650
Other Mining Costs	1,860
TOTAL MINE CAPITAL	**Cdn$355,300**

26.1.2 *MINING BASIS OF ESTIMATE*

Unit costs for consumable and labour rates are estimated from sources listed below while the magnitude of consumables and labour required are determined for each specific activity from experience and first principles.

The unit costs are based on the following data and are detailed in Appendix D:

- Salaries for the supervisory and administrative job category are based on MMTS's experience of similar functions in BC mines. An average burden rate of 50% has been applied to base salaries to include all statutory Canadian and BC, social insurance, medical and insurance costs, pension, and vacation costs.
- For hourly employees, general labour rates expected in BC mines were used. An average burden rate of 80% has been applied to base wages to include all statutory Canadian and BC, social insurance, medical and insurance costs, pension, and vacation costs.
- Mine designs to determine the size and makeup of the mine fleet as well as fuel requirements which is affected by distance from the pit to the various destinations over the existing and future topography.
- Budgetary quotations, including freight for all consumables, tires, and fuel. The long term fuel price is estimated at a delivered cost to site of Cdn$0.88/L.
- Power costs were estimated as the sum of energy charges, demand charges and estimated at an overall Cdn$0.04/kWh (power costs vary

between Cdn$0.04 and $0.045 between mining process and electrical costs in this PEA Addendum 2009).

- Mining equipment consumables, major equipment replacements, sustaining capital, labour loading factors, equipment life, and costs are based on vendor information and MMTS's data base from similar mining operations.

As per a directive from Seabridge, pre-stripping was transferred from the initial capital cost and placed in operating capital.

The MMTS estimate of start-up capital costs includes the following:

- mine equipment
- services and infrastructure
- pre-production tasks
- support and auxiliary equipment.

Mine capital costs (new and sustaining) are shown to milling Year 1 in Table 26.3. This schedule shows capital by year that the respective equipment is required to be working. Actual expenditure will be required sufficiently before that to allow for manufacturing, delivery, and erection.

Table 26.3 Mine Capital Schedule – New and Replacement

Fleet Capital Cost		PP (Cdn$ M)	Year 1 (Cdn$ M)
Drilling			
Diesel Drill – 311 m	Primary Drill	10.3	-
Electric Drill – 311 m	Primary Drill	-	10.4
Diesel Drill – High Wall – 150 mm	High Wall Drill	1.0	-
Blasting			
Hole Stemmer – 3 t	Blast Hole Stemmer	0.5	-
Loading			
Major			
Diesel Hydraulic Shovel - 85 t	Loading Mineralized Material & Waste	39.3	-
Electric Cable Shovel – 104 t	Loading Mineralized Material & Waste	-	65.5
Support			
Track Dozer – 630 kW	Shovel Support	5.0	5.0
Rubber Tired Dozer – 250 kW	Pit Clean Up	2.4	2.4
Fuel/Lube Truck	Shovel Fuelling & Lube	1.9	-
Wheel Loader Multipurpose – 14 t	Pit Clean Up	2.1	-
Hauling			
Major			
Haul Truck – 345 t	Hauling Ore/Waste	53.6	53.6
Support			

Fleet Capital Cost		PP (Cdn$ M)	Year 1 (Cdn$ M)
Water Truck – 20,000 gal	Haul Roads Water Truck	6.6	-
Track Dozer – 430 kW	Dump Maintenance	4.4	1.5
Motor Grader – 400 kW	Road Grading	5.4	1.8
Motor Grader – 220 kW	Road Grading	2.3	-
Tire Manipulator	Tires	1.1	1.1
Pit Maintenance			
Track Dozer – 430 kW	Pit Support	1.5	-
Float Tractor/Trailer – 189 t	Float Tractor & Trailer	2.8	-
Hydraulic Excavator – 6 t	Utility Excavator	3.4	-
Sump Pump - 1,400 gal/min	Pit Sump Dewatering	0.1	0.1
Light Plant	Lighting Plant	0.1	0.1
250-t Crane	Utility Crane	7.0	-
Crew Cab	Supervision and Crew Transportation	0.4	0.4
Ambulance	Ambulance	0.1	-
Hydraulic Excavator – 6 t	Utility Excavator	1.1	-
Mine Rescue Truck	Rescue Truck	1.3	-
GMC Guide XL Crew Bus	Crew Bus	0.4	0.1
Maintenance Truck – 1 t	Service Truck	0.2	0.1
Fire Truck	Fire Truck	0.3	-
Screening Plant – 12" max.	Road Crush & Stemmings	0.3	-
Picker Truck	Maintenance & Overhauls	0.3	0.3
Scraper – 37 t	Crush Haul for Winter Roads, Drill Steels	7.8	-
Crane 40 t Hydraulic Extendable	Utility Crane	1.1	0.5
Wheel Loader	Crusher (Road Crush) Loader	1.0	-
Snow Cat	-	0.2	-
100-t crane	Utility Crane	4.0	-
Forklift – 30 t	Forklift	0.4	-
Forklift – 10 t	Forklift	0.3	-
Service Truck	Service Truck	0.3	0.1
Welding Truck	Welding Truck	0.5	-
Powerline Truck	Powerline Maintenance	0.1	0
TOTAL CAPITAL COST		**$170 M**	**$143 M**

table continues…

26.1.3 Process Capital Cost

Basis of Estimate

This section describes the estimating guidelines for the preparation of the capital cost estimate for the KSM PEA. The estimate has been produced in Microsoft Excel

2007. The estimate breakdown structure is user-defined by area and commodity code. The contingency has been determined by historical data from similar projects.

Currencies are expressed in both Canadian and United States dollars. All costs in this section are stated in Q4 2008 prices; where available, Q2 2009 prices were applied. No allowance is included for escalation beyond this quarter.

The capital cost estimate for the process plant has been completed by Wardrop and is based on the information shown in Table 26.4.

Table 26.4 Basis of Estimate

Commodity	Estimate Basis
Plant and Equipment	
Major Equipment (>$1,000,000)	Single budget price quotations based on duty specifications & data sheets
Major Equipment (>$500,000)	Telephone and e-mail budget price quotations based on duty specifications
Minor Equipment (<$500,000)	In-house database and/or factored equipment costs from similar projects
Bulk Materials & Site Works	
Site Preparation & Roads	Estimated on a cost/unit area based on a preliminary earthworks volumes calculated from a 3D model (LAN desktop)
Concrete – Building Foundations	Estimated on a cost/unit area based on historical data for similar buildings
Concrete – Equipment Foundations	
Structural – Equipment Supports	
Structural – Building Steel	
Architectural (including Ancillary Buildings)	
Building Services	
Service Piping & Valves	Percentages of direct equipment costs, by area, based on the study equipment list and historical data from similar projects
Process Piping & Valves	
Electrical	
Instrumentation & Controls	
Installation	
Installation Labour	Hours calculated or based on historical data and in-house experience
Productivity	1.15 productivity factor has been assumed for the estimate
Vendor Representatives/ Supervision	An allowance based on complexity
Contractor Distributables/ Preambles	Included in the unit labour rate
Freight	
Main Bulks & Major Equipment	An allowance based on specific equipment and complexity. Freight costs to site have been included in the material section.

Commodity	Estimate Basis
	Unless specifically quoted, freight has been factored as 8% on equipment and materials, and 6% on mobile equipment.
Air Freight (for equipment and personnel)	Minimum allowance included plus helicopter support for initial ore slurry tunnel construction and mining pioneering work
Commissioning	
Commissioning Start up	Assessments based on in-house data
Construction & Commissioning Spares	Based on 7% of process equipment costs
Mining Spares	Based on 5% of mine rolling stock

table continues…

The following backup documentation and information is included in:

- process design criteria – Appendix C
- preliminary flowsheets – Appendix B
- general arrangement drawings – Appendices F and G
- plant mobile equipment list – Appendix I.
- equipment load list – Appendix G.

26.1.4 PERMANENT ACCOMMODATION AND CONSTRUCTION CAMPS

There are three construction camps included in the estimate. The camps are estimated on a modular basis and will be expanded to accommodate increasing labour force during construction.

Two permanent camps have been allowed for on the plant and pit side.

26.1.5 LABOUR RATES

Different labour rates were applied to various areas of the project. In general, a labour rate of $80/h has been used.

26.1.6 TAXES

Taxes have been excluded.

26.1.7 LOGISTICS

No logistics study has been performed for this project.

26.1.8 OWNERS' COSTS (INCLUDING OWNERS COMMISSIONING ALLOWANCE)

An allowance has been included for the Owners' costs. This cost has been provided by the Owner.

26.1.9 EXCLUSIONS

The following are not included in the capital cost estimate:

- force majeure
- schedule delays such as those caused by:
  - major scope changes
  - unidentified ground conditions
  - labour disputes
  - environmental permitting activities
  - abnormally adverse weather conditions
- receipt of information beyond the control of the EPCM contractors
- cost of financing (including interests incurred during construction)
- PST and GST
- royalties or permitting costs
- schedule acceleration costs
- working capital
- cost of this study
- sunk costs.

26.1.10 ASSUMPTIONS

The following assumptions have been made in the preparation of this estimate:

- All material and installation subcontracts are competitively tendered on an open shop, lump sum basis.
- Site work is continuous and is not constrained by the Owner.
- There is a 70-hour work week with a rotation of 2-weeks in/2 weeks out for the construction phase of the project.
- Skilled tradespersons, supervisors, and contractors are readily available.
- The geotechnical nature of the destination site is expected 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.

26.1.11 CONTINGENCY

A contingency allowance was built up to cover additional costs, which will be incurred as a result of more detailed design and investigations. It is considered that this estimate will adequately cover minor changes to the current scope to be expected during the next phase of the project.

Several major costs (and allowances) are assumed to contain a certain amount of contingency; therefore, a lower contingency was applied across the board. The average contingency for the project is calculated to be 15%. Refer to Section Z of the detailed capital cost estimate in Appendix I for a detailed breakdown.

26.1.12 ELECTRICAL POWER DISTRIBUTION COSTS

Table 26.5 summarizes the capital costs for the recommended BC Hydro power supply option.

Table 26.5 KSM Power Supply Construction Cost Summary

Description	Capital Cost (Cdn$000)
BC Hydro/BCTC Service Costs System	4,500
Series Capacitor Station 75 MV AR Capacitor Bank from Meziadin Substation	11,600
Transmission Line 287 kV – 103 km 287 kV Line from Meziadin Substation to Snowbank Creek	61,300
Transmission Line 287 kV – 287 kV Line from Snowbank Creek to Plant Site including Indirects	10,100
Flotation Plant 287 kV Substation #1 Area H1; 287 kV Line from Snowbank Creek to Plant Site including Indirects	20,400
Tunnel – 287 kV Cable; 287 kV Cable on Messenger	19,800
Tunnel – 287 kV Cable; 287kV Cable – Splices & Terminations	4,200
Tunnel – 287 kV Cable; 287 kV Cable – Cable Installation including Support	1,000
Tunnel – 287 kV Cable; Fibre Optic Cable – 20,000 m	800
Tunnel – 287 kV Cable; Indirects	1,100
Pit 25 kV Overhead Lines – 25 kV; Area Power Distribution – Power Lines including Indirects	2,800
Pit & Mill 287 Substation # 2; Area Power Distribution - Power Lines including Indirects	19,300
Total Construction Cost (Not Including Contingency and Owner's Costs)	**156,900**

For this PEA, it has been assumed that the power distribution to the KSM mine will be provided by BC Hydro under their current tariffs from their nearest transmission interconnection point, Meziadin Junction. The transmission line and associated facilities from that point on to the mine would be the responsibility of KSM as per BC Hydro policy. As the 138 kV system at Meziadin Junction has limited capacity, system reinforcement would be required. At this point, the transmission system reinforcement would require a new 287 kV transmission line to Meziadin, similar to that currently planned under the NTL project.

When a BC Hydro customer requests service that requires system reinforcement as described above, the associated capital costs are to the customer's account unless BC Hydro's projected sales revenues over a seven-year period are greater than the line capital cost. Since the KSM Project has a very large load (in the vicinity of 150 MW) the project would meet these criteria. Hence the capital cost of the required new transmission line from Skeena to Meziadin would be funded by BC Hydro/BCTC.

Service as per the standard BC Hydro tariff, the 287 kV line from Skeena Substation to Meziadin, and the proposed KSM line from Meziadin to the mine would have adequate capacity for any probable KSM load.

The single line diagram for the study is attached in Appendix G.

26.2 OPERATING COST ESTIMATE

The operating cost for the KSM Project was estimated at US$10.57/t milled. The estimate was based on an average daily process rate of 120,000 t milled.

Currencies are expressed in both Canadian and United States dollars. The partial costs in this section were updated according to budget prices in Q2 2009; however, the remaining costs were estimated according to the budget prices in Q3 2008.

When it was required, certain costs in this report have been converted using a fixed currency exchange rate of Cdn$1.00 to US$0.90 (as requested by Seabridge). The expected accuracy range of the operating cost estimate is +30%, -15%.

Power will be supplied by grid lines at an average cost of Cdn$0.042/kWh. Process power consumption estimates are based on the Bond work index equation for specific grinding energy consumption and equipment load power draws for the rest of the process equipment. The power cost for the mining section is included in the mining operating cost. Power costs for surface service are included in site services.

Table 26.6 Operating Cost Summary

	Cdn$/a (000)	Cdn$/t Milled	US$/a (000s)	US$/t Milled
Mine				
Mining Costs – Mill Feed*	200,300	4.66	180,163	4.19
Mill				
Staff & Supplies	196,160	4.48	176,544	4.03
Power (Process only)	45,075	1.03	40,567	0.93
G&A and Site Services				
G&A	28,591	0.65	25,732	0.59
Site Service	6,570	0.15	5,913	0.14
Tailing and Water Treatment				
Tailing	7,446	0.17	6,701	0.15
Water Treatment	26,762	0.61	23,905	0.55
TOTAL	**510,904**	**11.75**	**459,526**	**10.57**

* including pre-production operating costs of $174.2 M and mining GME.

The operating costs are defined as the direct operating costs including mining, processing, tailing storage, water treatment, and G&A. Sustaining capital includes all capital expenditures after the process plant has been put into production.

26.2.1 *MINE OPERATING COSTS*

All mining operating costs are shown in Canadian dollars, unless otherwise specified. Mine operating costs are derived from a combination of supplier quotes and historical data collected by MMTS. This includes the labour, maintenance, major component repairs, fuel, and consumables costs. The current fleet hourly operating costs are used as a constant basis over the schedule periods and estimates are input for sustaining and replacement capital.

From the basic operating capacities of the equipment, the travel speed characteristics of the trucks, and the haul road profiles, the equipment productivities for the shovels and trucks are calculated from the MineSight® production scheduling program. The truck speeds and cycle times for the various haul cycles are calculated by using a computerized simulation program. The equipment productivity and the scheduled production are used in the scheduling program to calculate the required equipment operating hours. These are multiplied by the hourly consumables consumption rates and unit operating costs to calculate the total equipment operating costs for each time period. The cost of minor parts and running repairs are included in the distributed operating costs for the major mining equipment.

Major part replacement for the major equipment fleets are calculated separately from the expected life of the major part, the cost of the part, and the fleet size for that equipment. This puts the large cost item repairs into future years giving a more representative the cash flow. The same type of life expectance parameters are used for equipment replacement cost calculations.

Blasting costs are based on studies from similar projects and historical blasting costs. Geotechnical costs for high wall control blasting, horizontal drains, etc. are based on other study data collected by MMTS.

Labour factors in manhours/equipment operating hour are assigned to each of the equipment types. Labour costs are calculated by multiplying the labour factor by the equipment operating hours, and labour costs are allocated to the equipment where labour has been assigned. The total hours required for each job type on all the equipment are summated and any additional labour required to complete a crew is assigned to unallocated labour. Some trades in mine operations (Grader Operator, Track Dozer Operator, Scraper Operator, Crusher Operator, Water Truck Operator, and Fuel Truck Operator) and mine maintenance (Crane Operator, Welder, Tireman, Labourer, and Serviceman) are treated as shared labour during the unallocated labour assignment and labour contingents of these are therefore not rounded off in Table 26.7 and Table 26.8. The mine hourly and salaried labour schedules are summarized in Table 26.7 and Table 26.8 and listed in detail in Appendix D.

Table 26.7 Mine Hourly Labour Schedule Manning Levels

Hourly Labour Summary	Year 5
Mine Operations	
Drill Operator	15
Blasters	8
Shovel Operator	20
Haul Truck Driver	87
Grader Operator	12
Excavator Operator	11
Loader Operator	1
Track Dozer Operator	34
Scraper Operator	6
Crusher Operator	4
Water Truck Operator	7
Fuel Truck Operator	3
Mine Maintenance	
Electrician	8
HD Mechanic	40
LD Mechanic	2
Machinist	2
Crane Operator	7
Welder	10
Tireman	2
Labourer Serviceman	1
TOTAL HOURLY	**280**

The mine salaried labour schedule is shown in Table 26.8.

Table 26.8 Mine Salaried Labour Schedule Manning Levels

Salaried Labour Summary	Year 5
Mine Operations	
Operations General Foreman	1
Shift Foreman	2
Area Foreman	8
Training General Foreman	1
Shift Trainers	4
Drilling & Blasting Foreman	1
Blasters	2
Maintenance General Foreman	1
Maintenance Planner	2
Maintenance Planning Clerk	2
Maintenance Shift Foreman	2
Mechanical Foreman	6
Electrical Foreman	2
Services Foreman	4
Administration Assistant	1
Technical Services	
Senior Geologist	1
Pit Geologist	2
Ore Grade Technicians	4
Project Engineer	1
Senior Environmental Engineer	1
Environmental Technician	2
Senior Mining Engineer	1
Mine Engineer	2
Drilling & Blasting Engineer	1
Drilling & Blasting Technician	2
Surveyor	2
Engineering Clerk	1
Dispatch Engineer	1
Senior Geotechnical Engineer	1
Environmental Samplers	2
Chief Engineer	1
Total Salaried	**64**

The labour rates are based on current salaries for G&A employees and hourly rates for mine operations and maintenance personnel in the area and are shown in Table 26.9 and Table 26.10.

Table 26.9 Mine Operating and Maintenance Hourly Labour Rates

Position	Base Rate (Cdn$/h)	Cdn$/ Manhour*
Mine Operations		
Drill Operator	30	41.88
Blasters	30	41.88
Shovel Operator	31	43.26
Haul Truck Driver	28	39.12
Grader Operator	29	40.50
Excavator Operator	29	40.50
Loader Operator	30	41.88
Track Dozer Operator	29	40.50
Scraper Operator	28	39.12
Crusher Operator	28	39.12
Water Truck Operator	28	39.12
Fuel Truck Operator	28	39.12
Mine Maintenance		
Electrician	36	50.58
HD Mechanic	36	50.58
LD Mechanic	31	43.69
Machinist	36	50.58
Crane Operator	28	39.56
Welder	36	50.58
Tireman	31	43.69
Labourer Serviceman	31	43.69

* includes loading.

Table 26.10 Mine G&A Salaries

Position	Base Salary (Cdn$)	Payroll Burden (Cdn$)	Salary With Burden (Cdn$)
Mine Operations			
Operations General Foreman	115,000	20.00	138,000
Shift Foreman	105,000	20.00	126,000
Area Foreman	90,000	20.00	108,000
Training General Foreman	90,000	20.00	108,000
Shift Trainers	80,000	20.00	96,000
Drilling & Blasting Foreman	105,000	20.00	126,000
Blasters	90,000	20.00	108,000
Maintenance General Foreman	115,000	20.00	138,000
Maintenance Planner	80,000	20.00	96,000
Maintenance Planning Clerk	55,000	20.00	66,000
Maintenance Shift Foreman	105,000	20.00	126,000
Mechanical Foreman	90,000	20.00	108,000
Electrical Foreman	90,000	20.00	108,000
Services Foreman	90,000	20.00	108,000
Administration Assistant	55,000	20.00	66,000
Technical Services			
Senior Geologist	105,000	20.00	126,000
Pit Geologist	70,000	20.00	84,000
Ore Grade Technicians	60,000	20.00	72,000
Project Engineer	90,000	20.00	108,000
Senior Environmental Engineer	105,000	20.00	126,000
Environmental Technician	80,000	20.00	96,000
Senior Mining Engineer	115,000	20.00	138,000
Mine Engineer	105,000	20.00	126,000
Drilling & Blasting Engineer	105,000	20.00	126,000
Drilling & Blasting Technician	60,000	20.00	72,000
Surveyor	60,000	20.00	72,000
Engineering Clerk	70,000	20.00	84,000
Dispatch Engineer	55,000	20.00	66,000
Senior Geotechnical Engineer	110,000	20.00	132,000
Environmental Samplers	110,000	20.00	132,000
Chief Engineer	50,000	20.00	60,000

LOM unit operating costs are listed in Table 26.11 and Table 26.12. Complete mine cost tables including mine capital and operating cost schedules are in the Appendices D and I.

Table 26.11 Mining Costs per Tonne Mill Feed

	LOM Cost Cdn$/t Mill Feed
Drilling	0.18
Blasting	0.74
Loading	0.65
Hauling	2.49
Pit Maintenance	0.36
Geotechnical	0.04
Unallocated Labour	0.01
GME	0.19
Total Mining Cost	**4.66**

Table 26.12 Mining Costs per Tonne Material Mined

	LOM Cost Cdn$/t Material Mined
Drilling	0.07
Blasting	0.27
Loading	0.24
Hauling	0.92
Pit Maintenance	0.13
Geotechnical	0.02
Unallocated Labour	0.00
GME	0.07
Total Mining Cost	**1.72**

Graphs of unit operating cost are shown as Cdn$/t material mined (waste and mineralized material) and Cdn$/t milled (Figure 26.1 and Figure 26.2). The distribution of unit cost by mining area is shown in Figure 26.3.

Figure 26.1 Unit Operating Cost for Mining in $/t Material Mined

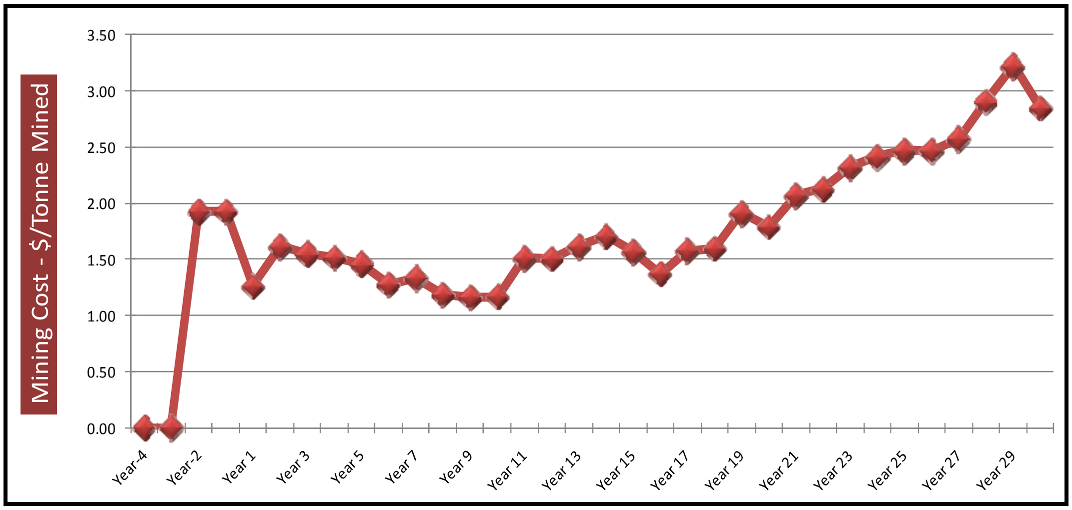


Figure 26.2 Unit Operating Cost for Mining in $/t Milled

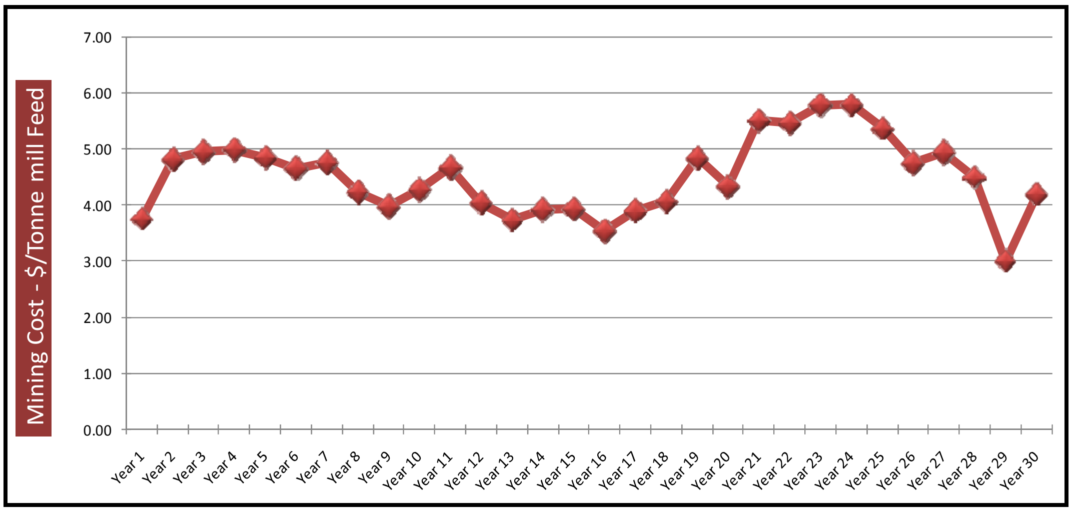
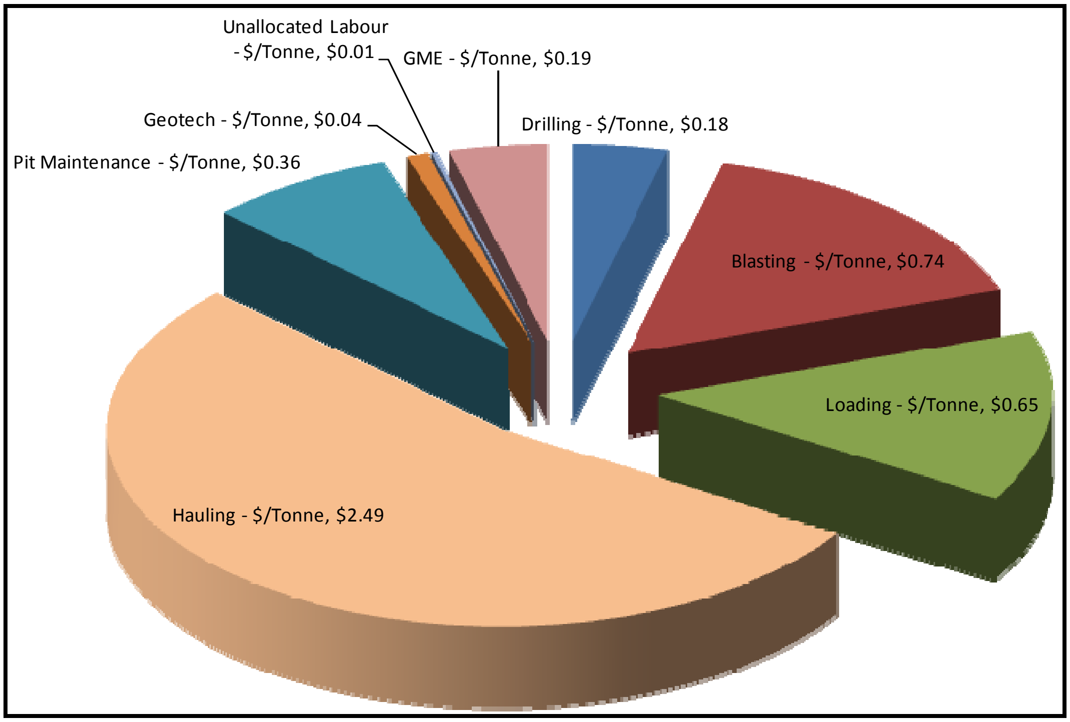


Figure 26.3 Unit Operating Cost Distribution in $/t Milled



MINE FUEL CONSUMPTION

Fuel consumption rates are estimated in the mine schedule for each equipment type. These consumption rates are applied to the operating hours of the equipment to estimate the total fuel consumption. Fuel costs have been included in the unit operating costs estimated above.

Explosive factory fuel consumption is estimated based on the quantity of explosives used, and an estimated 40 L diesel fuel consumed per tonne of explosives.

Fuel quantities scheduled for the first 5 years of milling are shown in Table 26.13.

Table 26.13 Mine Fuel Consumption Schedule

Fuel Consumption		Y-1	Y-2	Y1	Y3	Y4	Y5
Drilling	m^3	1,548	1,548	1,603	1,603	1,603	1,565
Blasting (Explosives Factory)	m^3	795	795	1,306	1,306	1,480	1,525
Loading	m^3	6,400	6,400	8,914	9,198	8,783	9,044
Hauling	m^3	18,149	18,149	33,249	52,280	51,594	51,956
Pit Maintenance	m^3	4,367	4,437	5,357	5,441	5,351	5,518
Total	m^3	**31,259**	**31,329**	**50,429**	**69,829**	**68,811**	**69,608**

26.2.2 PROCESS OPERATING COSTS

SUMMARY

All process operating costs are shown in Canadian dollars, unless otherwise specified.

The average annual process operating cost is estimated to be approximately Cdn$240 M or Cdn$5.51/t milled. The process operating costs are based on a process rate of 120,000 t/d milled and 92% plant availability.

The estimated process operating costs are summarized in Table 26.14 and include the following:

- personnel requirements including supervision, operation, and maintenance; salary/wage levels based on current labour rates in comparable operations in BC
- liner and grinding media consumption estimated from the Bond ball mill work index and abrasion index equations and quoted budget prices or Wardrop's database
- maintenance supplies based on approximately 5% of major equipment capital costs
- reagents based on test results and quoted budget prices or Wardrop's database
- other operation consumables including laboratory, filtering cloth, service vehicles consumables
- power consumption for the process plant at the power unit cost of $0.042/kWh.

Table 26.14 Summary of Process Operating Costs

Description	Personnel	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)	Annual Cost (US$)	Unit Cost (US$/t Milled)
Human Power					
Operating Staff	31	4,208,000	0.096	3,787,000	0.086
Operating Labour	116	11,252,000	0.257	10,127,000	0.231
Maintenance	81	8,934,000	0.204	8,040,000	0.184
Sub-total Human Power	**228**	**24,394,000**	**0.557**	**18,997,729**	**0.501**
Major Consumables					
Metal Consumables		63,546,000	1.451	57,192,000	1.306
Reagent Consumables		80,497,000	1.838	72,447,000	1.654
Supplies					
Maintenance Supplies		25,504,000	0.582	22,953,000	0.524
Operating Supplies		2,219,000	0.051	1,997,000	0.046
Sub-total Consumables & Supplies		**171,766,000**	**3.922**	**171,766,000**	**3.529**
Power Supply		45,075,000	1.029	40,567,000	0.926
Sub-total Power		**45,075,000**	**1.029**	**40,567,000**	**0.926**
PROCESS OPERATING COST TOTAL		**241,235,000**	**5.508**	**217,111,000**	**4.957**

PERSONNEL

The projected personnel requirements are 228 persons including:

- 31 staff for management and professional services
- 116 operators including laboratories for quality control, process optimization and assaying
- 81 personnel for maintenance.

Salary/wage rates are based on current rates in northern BC including base salary, holiday and vacation pay, pension plan, various benefits, and tool allowance costs.

Total estimated personnel cost is Cdn$0.56/t milled. The detailed personnel description and costs are shown in Appendix I for each processing plant area.

OPERATING AND MAINTENANCE SUPPLIES

Major consumables and operating suppliers are estimated at Cdn$3.92/t milled. The major consumables include metal and reagents consumables. The liner and grinding media consumption were estimated from the Bond abrasion index equation and the prices from the latest supplier budget prices or Wardrop database.

Reagent consumptions were estimated from laboratory test results and comparable operations. The reagent costs were from the current budget prices from potential suppliers or Wardrop's database.

The maintenance supplies are estimated at Cdn$0.58/t milled. Maintenance supplies are estimated based on comparable operations or approximately 5% of major equipment capital costs.

Operating Costs per Area of Operation

Table 26.15 shows the operating cost of each processing area. The mill operating cost is estimated at about Cdn$241 M/a, or Cdn$5.51/t milled. The details of operating costs for each processing area are further discussed in this section.

Table 26.15 Operating Costs per Area of Operation

Description	Personnel	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Crushing/Grinding/Copper Flotation Plant	149	163,798,000	3.740
Molybdenum Flotation Plant	4	10,451,000	0.239
Leach Plant	51	28,287,000	0.646
Cyanide Solution Handling	8	20,123,000	0.459
Tunnel Pumping	8	12,172,000	0.278
Tailing Management/Reclaimed Water	8	6,404,095	0.146
Total	**228**	**241,235,000**	**5.508**

Crushing, Grinding, Copper, and Pyrite Flotation

The operating cost for crushing, grinding, copper, and pyrite flotation is estimated to be approximately Cdn$3.74/t milled and is shown in Table 26.16. The cost estimate includes 149 personnel to operate the circuits as well as the metallurgy and assay laboratories. Management will oversee the process plant. Metallurgical and assay laboratories will service other areas of the mine, including mining and geological exploration.

Major consumables include liners, grinding media, and flotation reagents. The annual power consumption for crushing, primary grinding, concentrate regrinding, and other processes is estimated at 851 GWh. Details of the estimate are shown Appendix I.

Table 26.16 Grinding, Copper and Pyrite Flotation Operating Costs

Description	Personnel	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Personnel			
Operating Staff	20	2,692,000	0.061
Operating Labour	64	6,167,000	0.141
Maintenance	65	7,270,000	0.166
Sub-total Personnel	**149**	**16,128,000**	**0.368**
Supplies			
Major Consumables			
Metal Consumables		63,221,000	1.443
Reagent Consumables		30,903,000	0.706
Supplies			
Maintenance Supplies		16,020,000	0.366
Operating Supplies		1,802,000	0.041
Power Supply		35,725,000	0.816
Sub-total Supplies		147,670,000	**3.371**
TOTAL	**149**	**163,798,000**	**3.740**

Molybdenum Flotation

Table 26.17 shows that the estimated operating cost for molybdenum flotation is approximately $0.24/t milled. Four operators will be required for this circuit. Major consumables include regrind wear materials and molybdenum flotation reagents. The annual power consumption for this circuit is estimated to be approximately 853 MWh. Details of the costs are shown in Appendix I.

Table 26.17 Molybdenum Flotation Operation Costs

Description	Personnel	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Personnel			
Operating Labour	4	399,000	0.009
Sub-total Personnel	**4**	**399,000**	**0.009**
Supplies			
Major Consumables			
Metal Consumables		326,000	0.007
Reagent Consumables		9,601,000	0.219
Supplies			
Maintenance Supplies		80,000	0.002
Operating Supplies		10,000	0.000
Power Supply		36,000	0.001
Sub-total Supplies		**10,052,000**	**0.230**
Total	**4**	**10,451,000**	**0.239**

Gold Leach and Recovery Circuit

The gold leach and recovery circuit will be operated by designated personnel including staff, and operation and maintenance labour. The total operating cost is estimated to be Cdn$0.65/t milled. The personnel cost is estimated to be Cdn$0.13/t milled (Table 26.18). The cost for major consumables and supplies is estimated at Cdn$0.52/t milled. The power consumption for this circuit is estimated at 12 GWh/a.

Table 26.18 Gold Leach and Recovery Circuit Operating Costs

Description	Personnel	Annual Cost (Cdn$)	Unit Cost (Cdn$/CIP)	Unit Cost (Cdn$/t Milled)
Personnel				
Operating Staff	11	1,516,000	0.353	0.035
Operating Labour	24	2,351,000	0.548	0.054
Maintenance	16	1,664,000	0.388	0.038
Sub-total Personnel	**51**	**5,531,000**	**1.288**	**0.126**
Supplies				
Major Consumables				
Major Consumables		20,810,000	4.848	0.475
Supplies				
Maintenance Supplies		1,317,000	0.307	0.030
Operating Supplies		130,000	0.030	0.003
Power Supply		498,000	0.116	0.011
Sub-total Supplies		**22,756,000**	**5.301**	**0.520**
Total	**51**	**28,287,000**	**6.589**	**0.646**

Cyanide Recovery and Destruction Circuit

The cyanide recovery and destruction circuits will require eight operators. The total unit cost for the circuits is estimated at Cdn$0.46/t milled. This cost includes a labour cost of Cdn$0.02/t milled and a total processing supplies cost of Cdn$0.44/t milled. The annual power consumption will be approximately 1.9 GWh. Details are shown in Table 26.19. A more detailed cost estimate is shown in Appendix I.

Table 26.19 Cyanide Recovery and Destruction Operating Costs

Description	Personnel	Annual Cost (Cdn$)	Unit Cost (Cdn$/CIP)	Unit Cost (Cdn$/t Milled)
Personnel				
Operating Staff	8	748,000	0.174	0.017
Sub-total Personnel	**8**	**748,000**	**0.174**	**0.017**
Consumables and Supplies				
Reagent Consumables		19,183,000	4.469	0.438
Maintenance Supplies		87,000	0.020	0.002
Operating Supplies		25,000	0.006	0.001
Power Supply		80,000	0.019	0.002
Sub-total Supplies		**19,375,000**	**4.513**	**0.442**
TOTAL	**8**	**20,123,000**	**4.687**	**0.459**

Tunnel Pumping Operation

The operating cost estimate for the tunnel conveyors is shown in Table 26.20. The major operating cost components are maintenance and power supply. The total unit cost is estimated to be Cdn$0.28/t milled including power supply, which is estimated at 114 GWh/a.

Table 26.20 Tunnel Conveyor Operating Costs

Description	Personnel	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Personnel			
Operating Labour	8	748,000	0.017
Sub-total Personnel	**8**	**748,000**	**0.017**
Supplies			
Maintenance Supplies		6,500,000	0.148
Operating Supplies		117,000	0.003
Power Supply		4,807,000	0.110
Sub-total Supplies		**11,424,000**	**0.261**
Total	**8**	**12,172,000**	**0.278**

Tailing and Reclaimed Water Operation

Tailing operation cost estimates for tailing delivery to the TMF and water reclamation costs are shown in Table 26.21, which details the unit costs for labour, maintenance supplies, operating suppliers, and power supply.

The major cost contributing factor of tailing operations is power consumption for reclaiming water from the tailing storage pond. The annual power requirement is estimated to be approximately 94 GWh, which accounts for Cdn$0.09/t milled. A more detailed breakdown is shown in Appendix I.

Table 26.21 Tailing and Reclaimed Water Operating Costs

Description	Personnel	Annual Cost (Cdn$)	Unit Cost (Cdn$/t Milled)
Personnel			
Operating Labour	8	840,000	0.019
Sub-total Personnel	**8**	**840,000**	**0.019**
Supplies			
Maintenance Supplies		1,5000,000	0.034
Operating Supplies		136,000	0.003
Power Supply		3,928,000	0.090
Sub-total Supplies		**5,564,000**	**0.127**
Total	**8**	**6,404,000**	**0.146**

26.2.3 TMF OPERATING COSTS AND WATER TREATMENT COSTS

The tailing dam ongoing construction and operation costs are estimated to be approximately Cdn$7.3 M/a, or Cdn$0.167/t milled.

The cost for water management, including diversions and collection dam operations, is estimated to be approximately Cdn$26.8 M/a, or Cdn$0.611/t milled.

26.2.4 GENERAL AND ADMINISTRATIVE

G&A are the costs that do not relate directly to the mining or processing operating costs. The costs include:

- personnel – general manager and staffing in accounting, purchasing, and environmental departments, and G&A
- various employees in surface services
- G&A expenses including insurance, administrative supplies, medical services, legal services, human resources related expenses, travelling,

accommodation/camp costs, air crew transportation, road maintenance, and external assay/testing.

The G&A expenses are estimated at approximately Cdn$28.6 M/a, or Cdn$0.65/t milled, including approximately Cdn$0.10/t for personnel and Cdn$0.55/t for general expenses. The major costs are accommodation, crew transportation, and road maintenance estimated at about Cdn$13.7 M/a. A summary of the G&A estimate for personnel and general expenses are shown in Table 26.22 and Table 26.23, respectively.

The site service cost is estimated at Cdn$0.15/t milled or about Cdn$6.6 M/a. The estimate is based on similar projects in North America.

Table 26.22 G&A Personnel Costs

G&A	Personnel	Base Rate ($/a)	Loaded Salary ($/a)	Total Cost ($/a)	Unit Cost ($/t Milled)
Staff					
General Manager	1	135,000	203,000	203,000	0.005
Human Resources Manager	1	96,000	144,000	144,000	0.003
Controller/Accountant	1	96,000	144,000	144,000	0.003
Chief Purchaser	1	96,000	144,000	144,000	0.003
Environmentalist	1	96,000	144,000	144,000	0.003
Environmental Technician	2	80,400	121,000	241,000	0.006
Safety and Training Officer	1	76,000	114,000	114,000	0.003
Public Relation Officer	1	80,400	121,000	121,000	0.003
Warehouse Supervisor	1	86,700	130,000	130,000	0.003
Surface Foreman	2	86,700	130,000	260,000	0.006
Secretary	2	96,000	144,000	288,000	0.007
Clerks (General/Accounting)	4	76,000	114,000	456,000	0.010
Sub-total Staff	**18**			**2,389,000**	**0.055**
Labour					
Yard Foreman	2	86,700	130,000	260,227	0.006
Warehouse/First Aid	4	76,000	114,000	456,000	0.010
Labourers – Yard/Surface Shops	4	58,300	87,000	350,000	0.008
Electrician – Surface Shops	2	80,400	121,000	241,000	0.006
Mechanic – Surface Shops	2	80,400	121,000	241,000	0.006
Carpenter – Surface Shops	1	80,400	121,000	121,000	0.003
Security	4	76,000	114,000	456,000	0.010
Sub-total Labour	**19**			**2,125,000**	**0.049**
TOTAL PERSONNEL	**37**			**4,513,000**	**0.103**

Table 26.23 G&A Expenses

G&A Expenses	Total Cost ($/a)	Unit Cost ($/t Milled)
Insurances	1,500,000	0.034
External Assays/Testings	250,000	0.006
Safety & Training Supplies	1,500,000	0.034
Medical Service/First Aid	200,000	0.005
Security Supplies	100,000	0.002
Legal Services – Allowance	200,000	0.005
Regulatory Compliance – Allowance	300,000	0.007
Consulting – Allowance	350,000	0.008
Small Vehicles	250,000	0.006
Head Office Expenses	200,000	0.005
Recruitment	150,000	0.003
Communications	300,000	0.007
Computer Services	100,000	0.002
Travel & Expenses	150,000	0.003
Professional Associations	100,000	0.002
Accommodation/Comp Costs	6,987,000	0.160
Road Maintenance	2,500,000	0.057
Avalanche Control	500,000	0.011
Regional Taxes & Licenses Allowance	1,500,000	0.034
Environmental Expenses	500,000	0.011
Crew Air Transportation	4,240,000	0.097
Warehouse	2,000,000	0.046
Miscellaneous	200,000	0.005
TOTAL	**24,078,000**	**0.550**

27.0 ECONOMIC ANALYSIS

27.1 INTRODUCTION

Wardrop completed a cash flow analysis for Seabridge's KSM Project in BC. The analysis is based upon providing a plant feed of 43 Mt/a over a 30-year open pit mine life. The study completed is an economic assessment to understand the viability and upside potential of the KSM property. Cash flow and sensitivity analyses were carried out and are described in this section.

27.1.1 CASH FLOW ANALYSIS

The cash flow analysis was carried out using the full production schedule at three-year average metal prices. Additional cash flows were generated using different average metal prices (one-year and two-year averages).

27.1.2 SENSITIVITY ANALYSIS

The project was analyzed to determine the sensitivity to changes according to a number of different parameters, including:

- gold and copper prices
- gold and copper grade
- operating costs
- initial capital expenditures
- offsite charges
- exchange rate.

27.2 ASSUMPTIONS

The following assumptions were used in the cash flow analyses:

- All dollars are stated in US currency.
- All analyses are pre-tax.
- Mine schedule, mine capital, infrastructure, and operating costs are as provided by MMTS.

- Tailings barge, piping, conveying, and off-site storage facilities/infrastructure costs are as provided by BVL.
- Mill capital and operating costs are as provided by Wardrop.
- Tunnel costs are provided by Wardrop with the assistance of Thyssen.
- Waste management, tailings dam, hydro generation, and water management capital and operating costs are as provided by KCBL.
- Access and temporary road capital costs are as provided by McElhanney.
- Power supply and distribution are as provided by Brazier.
- An exchange rate of US$0.90:Cdn$1.00 has been used.
- A 5% discount rate has been used for NPV determination.

Assumptions for treatment and handling charges, based on similar projects, are as follows:

- copper concentrate:
  - smelting charge of US$85/dry metric tonne (dmt)
  - refining charges of US$0.085/lb of copper, US$8.000/oz of accountable gold and US$0.450/oz of accountable silver
  - price participation of 1.5% above a base price of US$1.50/lb capped at US$0.04/lb
  - metal losses of 1.0 unit for copper, 2.5% for gold concentrate, and 10% for silver
  - transportation costs including ocean at US$65/wet metric tonne (wmt) plus Cdn$25/wmt for trucking to the Stewart port
  - concentrate loss of 0.50% (during transport and re-handling)
  - Insurance costs of 0.15% of net invoice value (NIV)
  - representation of US$0.50/wmt
  - no known metal penalties.
- molybdenum concentrate:
  - roasting charge of US$1.500/lb
  - metal loss of 2% of molybdenum
  - transportation, concentrate losses, insurance, and representation costs the same as for copper concentrate.
- gold dore:
  - a combined smelting and transportation cost of US$2/oz
  - metal loss of 0.2% gold
  - insurance of 0.15% of NIV
  - representation of 0.02% of NIV (estimated based on copper concentrate ratio).

27.3 ANALYSIS

27.3.1 CASH FLOW ANALYSIS

BASE CASE

The base case cash flow used a three-year average metal price taken from the London Metal Exchange (LME) as of June 30, 2009. While gold was priced at US$777.90/oz and copper priced at US$3.00/lb, the KSM Project generated a pre-tax NPV (discounted at 5%) of US$3,424 M, an IRR of 12.6%, and a payback of 6.6 years.

Gold revenues were approximately US$15.0 B which accounted for 45% of all revenue. Other metal credits total approximately US$18.3 B, of which US$15.8 B was from copper. These metal credits effectively cancel the operating costs which totalled US$17.3 B over the project life. The average revenue per tonne milled was US$25.7.

The cash cost per ounce of gold net of by-product metals was calculated to be US$-51 with a capital cost of US$229/oz for a combined cost of US$178/oz. The total capital costs were US$4.4 B and the unit operating cost totalled US$13.36/t milled, which includes offsite charges.

Table 27.1 summarizes the key results of the cash flow while the details can be found in Appendix J.

Table 27.1 Base Case Cash Flow Key Values

Economic Returns	Unit	Pre-Tax
Project NPV		
8.0% Discount Rate	million US$	1,470
5.0% Discount Rate	million US$	3,424
3.0% Discount Rate	million US$	5,621
0.0% Discount Rate	million US$	11,570
Project IRR	**%**	12.6
Payback	years	6.6
MineLife	years	29.9
Operating Cash Flow		
Years 1-8		
Total	000 US$	4,310,844
Average	000 US$	538,856
LOM		
Total	000 US$	15,986,999
Average	000 US$	534,136

table continues…

Economic Returns	Unit	Pre-Tax
Capital Costs		
Pre-production (pre-strip)	000 US$	151,414
Initial Capital	000 US$	3,083,080
Working Capital	000 US$	103,059
Sustaining Capital	000 US$	1,079,297
Total Capital Costs	**000 US$**	**4,416,851**

Production Summary	Unit	Years 1-8	LOM
Mill Feed Grade			
Gold	g/t	0.711	0.609
Copper	%	0.176	0.215
Silver	g/t	2.74	2.21
Molybdenum	ppm	52.8	51.9
Material Mined			
Mill Feed	kt	345,601	1,293,001
Waste	kt	791,455	2,210,242
Total	kt	1,137,056	3,503,243
Average	kt	142,132	115,623
Strip Ratio		**2.29**	**1.71**
Total Production			
Gold	koz	6,130	19,278
Copper	klbs	1,091,872	5,259,442
Silver	koz	22,249	67,054
Molybdenum	klbs	14,859	60,043
Average Production			
Gold	koz	766	644
Copper	klbs	136,484	175,721
Silver	koz	2,781	2,240
Molybdenum	klbs	1,857	2,006

Unit Cost Summary	Unit	Years 1-8	LOM
Operating Costs			
Mining	US$/t milled	4.00	3.91
Processing	US$/t milled	4.96	4.96
G&A + Site Services	US$/t milled	0.88	0.88
Treatment/Handling	US$/t milled	2.47	2.90
Tailings	US$/t milled	0.15	0.15
Water Treatment	US$/t milled	0.34	0.55
Total Operating Cost	**US$/t milled**	**12.80**	**13.36**

table continues…

Unit Cost Summary	Unit	Years 1-8	LOM
By-product Credits			
Copper	US$/t milled	9.48	12.20
Silver	US$/t milled	0.88	0.71
Molybdenum	US$/t milled	1.12	1.21
Total	**US$/t milled**	**11.48**	**14.12**
Net Cost After Credits			
Total	**US$/t milled**	**1.32**	**-0.77**
Cost/oz Au			
Cash Cost	US$/oz Au	75	-51
Capital Cost	US$/oz Au		229
Total Cost	**US$/oz Au**		**178**

ADDITIONAL CASH FLOWS

Historically, metal prices have been increasing over the past three years, aside from the latest market fluctuations. Two additional cash flow scenarios were created using the metal price inputs shown in Table 27.2 to give insight on project returns if metal prices continue at these trends. A fixed exchange rate of US$0.90:Cdn$1.00 was used in all of the scenarios.

The payback period is 6.6 years for the base case, 8.8 years for the alternate case, and 5.8 years for the current prices (June 17, 2009) average metal prices.

Table 27.2 summarizes the results of the four cash flows that were created. The detailed cash flows for all cases can be found in Appendix J.

Table 27.2 Summary of the Economical Evaluations

		Base Case 3-year Average	Alternate Case	Current Price July 17 2009
Gold	US$/oz	778	800	950
Copper	US$/lb	3.00	2.00	2.50
Silver	US$/oz	13.68	12.50	14.00
Molybdenum	US$/lb	26.05	15.00	15.00
Exchange Rate	US:Cdn	0.90	0.90	0.90
NPV (at 0%)	US$B	11.570	6.326	11.707
NPV (at 5%)	US$B	3.424	1.356	3.703
IRR	%	12.6	8.5	13.6
Cash Cost/oz Au	US$/oz	-51	243	114
Payback Period	years	6.6	8.8	5.8
Total Cost/oz	**US$/oz**	**178**	**472**	**343**

27.3.2 Sensitivity Analysis

Parameter Sensitivity

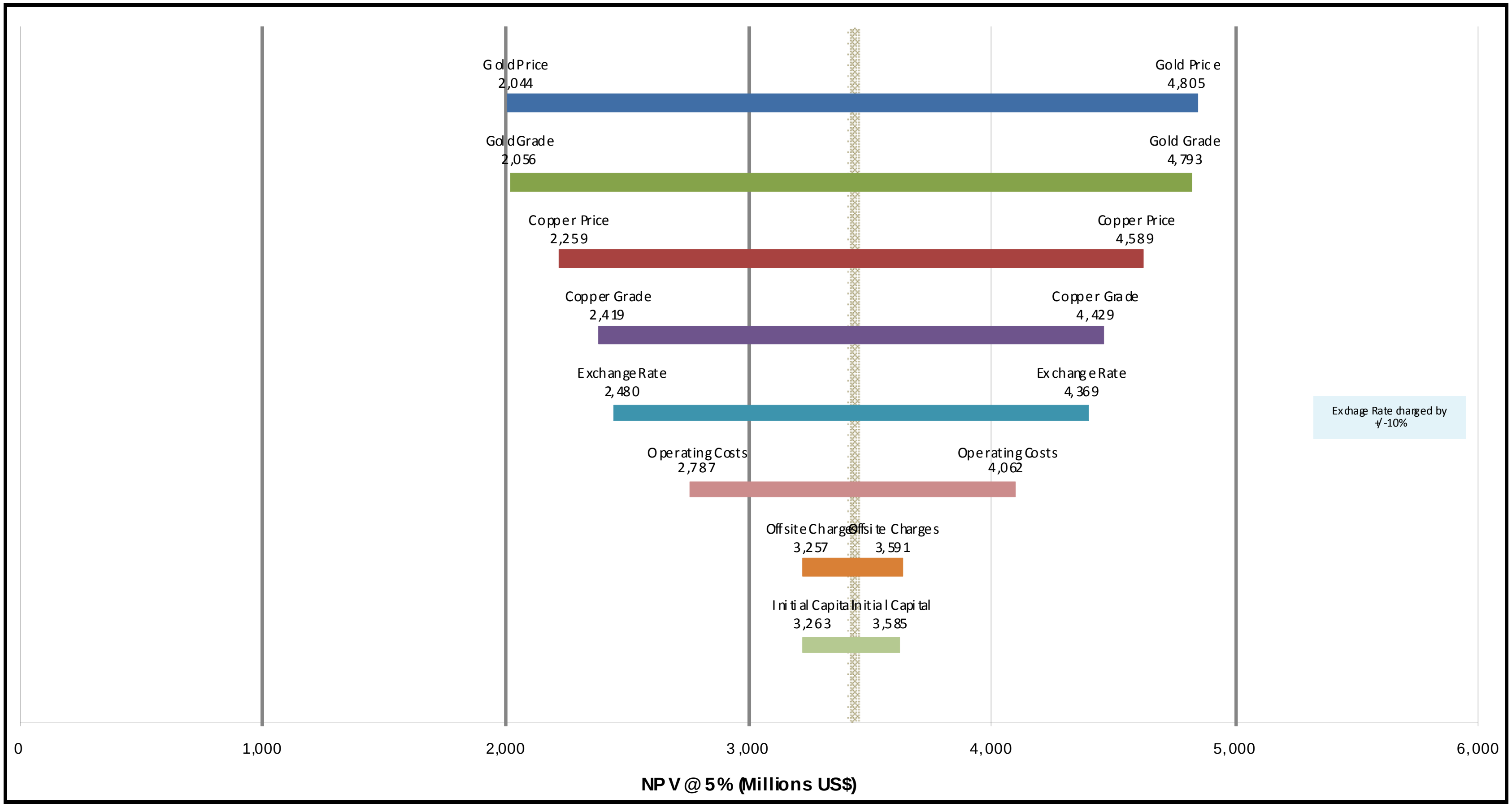
Various parameters were adjusted to understand the impact they had on the project's discounted NPV. The following parameters were analyzed:

- gold price
- copper price
- gold grade
- copper grade
- offsite charges (treatment terms and transportation)
- operating costs
- initial capital costs
- exchange rate.

Each parameter was adjusted by ±20% except for the exchange rate, which was only adjusted by ±10%. The results were arranged by highest to lowest impact as seen in Figure 27.1.

The KSM Project is most sensitive to gold price and grade. The discounted NPV varies from the base case by up to US$1.4 B (change of ±41%). Copper price and grade, as well as exchange rate, have a strong impact on the project (±32%). The initial capital cost and offsite charges had the least influence of all the parameters examined (change of <20%).

Figure 27.1 Sensitivity to NPV by Changing Parameters (-20% to +20%)



28.0 INTERPRETATION AND CONCLUSIONS

Based on the work carried out in this PEA and the resultant economic evaluation, this scoping-level study should be followed by the trade-off studies referred to in this document as well as a PFS in order to further assess the economic viability of the project.

29.0 OPPORTUNITIES & RECOMMENDATIONS

29.1 CONCEPTUAL 45-YEAR ECONOMIC PIT OPPORTUNITY

The Mitchell deposit has significant tonnage of deep mineralization that is not captured in the 30 year mine plan used as the basis for this 2009 PEA. The 30 year mine plan resulted from pit optimizations designed to maximize a 5% net present value discounted mining schedule. Potential cash flows resulting from operating years beyond 30 years have little impact on net present values as their discounted values approach zero. To examine the potential of production life beyond the 30 year mine plan, pit optimizations were also carried out designed to maximize total undiscounted net cash flow for the project. This pit schedule indicated that another 15 years of mill feed (600 to 700 Mt) might possibly be developed from deeper mineralization not captured in the 30 year pits. This extended mine life schedule was evaluated to assess possible economic returns. This conceptual extended mine plan opportunity has been developed for the KSM EML 45 year case based on the economic pit limit assessment described in this 2009 PEA. The EML 45 year scenario would follow a similar development path to the 30 year scenario and capital payback would occur in approximately the same time frame as the 30 year scenario. The mine schedule and financial model results for the EML 45 year scenario are summarized and compared to the 30 year base case mine scenario in Table 29.1 below.

Table 29.1 Comparison of 30 Year Base Case and Conceptual Extended Mine Life (45 Year Case)

	30 Year Mine Plan	45 Year Mine Plan
Total Tonnes to Mill	**1.29 billion**	**1.93 billion**
Average Grades		
Gold (g/t)	0.61	0.60
Copper (%)	0.22	0.20
Silver (g/t)	2.21	2.42
Molybdenum (ppm)	51.9	52.2
Total Production		
Gold (oz)	19.3 million	28.2 million
Copper (lb)	5.23 billion	7.2 billion
Silver (oz)	67.1 million	111.6 million
Molybdenum (lb)	60.0 million	88.7 million

table continues...

	30 Year Mine Plan	45 Year Mine Plan
Total Tonnes to Mill	**1.29 billion**	**1.93 billion**
Material Mined		
Mill Feed (kt)	1.29 million	1.93 million
Waste Material Mined (kt)	2.2 million	5.1 million
Strip Ratio	1.71	2.66
Life of Mine Annual Production		
Gold (oz)	644,000	633,000
Copper (lb)	176 million	162 million
Silver (oz)	2.2 million	2.5 million
Molybdenum (lb)	2.0 million	2.0 million
Net Cash Flow	$11.6 billion	$16.0 billion
NPV @ 5%	$3.4 billion	$3.5 billion
IRR (%)	12.6	12.1
Operating Costs Per Ounce of Gold Produced (LOM)	-51	21
Total Costs Per Ounce of Gold Produced (includes all capital)	178	210

The development of resources beyond 30 years of production into the future will require more exploration drilling and considerable engineering work to demonstrate viability. One concept level opportunity mine plan has been developed for the KSM EML case with a 45-year mine life based on the economic pit limit assessment described in this 2009 PEA Addendum. Future metal prices, operating cost experienced at KSM and permitting requirements will play a key role in determining whether these additional resources will be developed. However, this analysis provides some optimism that an extension of KSM operating life beyond 30 years may be possible.

29.2 GEOLOGY/RESOURCE RECOMMENDATIONS

- The Sulphurets deposit remains open along strike from the "Canyon Zone" at the southwest end of the deposit northeasterly towards the main zone of mineralization. Drilling should target permissive geometry along strike or down-dip from existing gold intercepts. This program should be carefully designed with contingencies for dropping or adding holes based on the drilling results.
- If possible, test the continuity of mineralization between the Mitchell and Iron Cap deposits by drilling methods. Little is understood about the Iron Cap Zone other than quartz-sericite-pyrite alteration is more intense than at Mitchell and there appears to be more base metal mineralization, particularly in narrow veins. The 2005 Falconbridge holes intersected low-grade gold mineralization near the surface. Offset holes from existing known mineralization should be designed to

aid in determining the possible geometry of mineralization and its possible relationship to the nearby Mitchell deposit.

- Three-dimensional lithologic models are required for the Kerr and Sulphurets Zones. These models may aid in the metallurgical characterization of the various mineralized units, provide details for possible pit slope angles, and help to characterize waste rock into appropriate classes.
- The Mitchell resource model includes grades for copper, gold, silver, and molybdenum. The Kerr and Sulphurets resource model include grades for copper and gold but not silver or molybdenum. Modelling the molybdenum and silver in these other areas may significantly enhance the project economics.

29.3 MINING RECOMMENDATIONS

- Additional inferred resources are present in the Mitchell mineralized zone. These resources will require considerably more drilling to develop economics and future potential.
- Conceptual studies of waste dump locations and water management during placement and after mine reclamation have been completed. Optimization of dump layouts and more detailed evaluation of efficiency of diversions, covers, and PAG leachate will be presented in the Preliminary Feasibility Study.
- Foundation testing and hydrogeological investigation of the Mitchell waste dump site was completed in 2009. Additional testing of the McTagg, Sulphurets, and Kerr sites is required to generate foundation preparation requirements for the waste dumps.
- The Mitchell-Teigen tunnel route investigation has established characteristics of major rock types along the route. A more detailed study of faults and structure is necessary to evaluate tunnelling risk and to more accurately examine construction options to determine schedule requirements and more accurate site specific tunnelling costs. Depth of glaciers needs to be determined in key areas with geophysical methods.
- The high mining rate requires a large mining fleet and the high level of operating activity within the limited space in the constrained valleys could create congestion and other operating in-efficiencies. A detailed simulation of the mining operation at distinct periods in the operating schedule is required to determine the operability of the mine plan.

29.4 GEOTECHNICAL RECOMMENDATIONS

- Geotechnical designs are required to confirm that the proposed slope angles for the Sulphurets and Kerr deposits are viable. These studies would be similar to those carried out for the Mitchell deposit, but at a lesser level of detail due to the size of the deposits and anticipated shallower pit depths.
- Review mine planning considerations that could improve the stability of the proposed open pit slopes (controlled blasting) or optimize the slope geometry (compound slope angles) and develop a design criteria for pit wall excavation.

29.5 PROCESS RECOMMENDATIONS

- Using larger capacity equipment for grinding and flotation should be investigated in the next stage study to reduce capital costs.

29.6 OTHER RECOMMENDATIONS

- A geohazard assessment is needed including snow and avalanche loss control programs as the project infrastructure locations become more defined.
- The Mitchell Valley has significant quantities of glacial moraines that have not been modelled in sufficient detail. More detailed overburden mapping should be carried out in the subsequent study. This will be needed for infrastructure planning and for construction and closure cover materials. Additional evaluation of potential borrow sources needs to be completed to assist with the overall project planning.

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