Report to:

SEABRIDGE GOLD

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2012 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study

Document No. 1252880100-REP-R0001-02

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2012 KSM (KERR-SULPHURETS-MITCHELL) PREFEASIBILITY STUDY

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Prepared by: Jiahhui (John) Huang (P.Eng.), Tetra Tech – Wardrop
Sabry Abdel Hafez (P.Eng.), Tetra Tech – Wardrop
Michael J. Lechner (P.Geo., RPG, CPG), Resource Modeling Inc.
J. Graham Parkinson (P.Geo.), Klohn Crippen Berger Ltd.
James H. Gray (P.Eng.), Moose Mountain Technical Services
Neil Brazier (P.Eng.), WN Brazier Associates Inc.
Warren Newcomen (M.S., P.Eng.), BGC Engineering Inc.
Kevin Jones (P.Eng.), EBA Engineering Consultants Ltd.
Harold Bosche (P.Eng.), Bosche Ventures Ltd.
R.W. Parolin (P.Eng.), McElhanney Consulting Services Ltd.
Darby Kreitz (P.Eng.), Allnorth Consultants Ltd.
Tony Wachmann (P.Eng.), Stantec Consulting Ltd.
Pierre Pelletier (P.Eng.), Rescan Environmental Services Ltd.
Ross Hammett (P.Eng.), Golder Associates Ltd.

ALM

TETRA TECH
WARDROP

Suite 800, 555 West Hastings Street, Vancouver, British Columbia V6B 1M1, Canada
Phone: 604.408.3788 Fax: 604.408.3722

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NOTICE

This report was prepared for Seabridge Gold Inc. (Seabridge) by Tetra Tech WEI Inc. (Tetra Tech), Moose Mountain Technical Services (MMTS), BGC Engineering Inc. (BGC), Rescan Environmental Services Ltd. (Rescan), McElhanney Consulting Services, Ltd. (McElhanney), Klohn Crippen Berger Ltd. (KCB), Bosche Ventures Ltd. (BVL), W.N. Brazier Associates Inc. (Brazier), EBA Engineering Consultants Ltd. (EBA), Allnorth Consultants Ltd. (Allnorth), Stantec Consulting Ltd. (Stantec), Golder Associates Ltd. (Golder), 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, EBA, Allnorth, Stantec, Golder, 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 Prefeasibility Study.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF APPENDICES

NOTE: APPENDICES A THROUGH M ARE AVAILABLE ON REQUEST AT THE SEABRIDGE GOLD INC. OFFICES IN TORONTO

GLOSSARY

Units of Measure

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion tonnes	Bt
Billion years ago	Ga
Billion	B
British thermal unit	BTU
Centimetre	cm
Centipoise	mPa·s
Coefficients of Variation	CVs
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
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	Cdn$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L

Grams per tonne	g/t
gravitational constant	*g*
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 hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Kilowatt	kW
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 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 bank cubic metres per annum	Mbm^3/a
Million bank cubic metres	Mbm^3

Million tonnes	Mt
Million	M
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Parts per billion	ppb
Parts per million	ppm
Pascal	Pa
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 circular mils	kCM
Thousand tonnes	kt
Three Dimensional Model	3DM
Three Dimensional	3D
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
HPGR Specific Throughput Rate (throughput (t/h)/(Roll Diameter (m) x Roll Width (m) x Roll Peripheral Speed (m/s))	ts/hm^3
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

Abbreviations and Acronyms

100-year flood level	Q100
3D block model	3DBM
abrasion index	Ai
acid base accounting	ABA
acid rock drainage	ARD
Acidification, Volatilization of HCN gas and Re-neutralization	AVR
Aero Geometrics Ltd.	Aero Geometrics

Allnorth Consultants Ltd.	Allnorth
Alpine Tundra	AT
ALS Chemex Laboratories Ltd.	ALS Chemex
ammonium nitrate-fuel oil	ANFO
antimony	Sb
arsenic	As
atomic absorption	AA
Average Recurrence Interval	ARI
Basic Transmission Extension	BTE
BC Environmental Assessment Office	BCEAO
BGC Engineering Inc.	BGC
block copper grades	CUZON
block gold grades	AUZON
Bosche Ventures Ltd.	BVL
Brenda Mines Ltd. Metallurgical Laboratory	Brenda Mines Met Lab
Brinco Ltd.	Brinco
British Columbia Environmental Assessment Act	BCEAA
British Columbia Utilities Commission	BCUC
British Columbia	BC
Bulk Mineral Analysis with Liberation	BMAL
bulk mineral analysis	BMA
Butterfield Mineral Consultants Ltd.	Butterfield
Canadian Dam Association	CDA
Canadian Environmental Assessment Act	CEAA
Canadian Environmental Assessment Agency	CEA Agency
Canadian National Railroad	CNR
carbon-in-leach	CIL
carboxymethyl cellulose	CMC
Caro's acid	H_2SO_5
CDN Resource Laboratories Ltd.	CDN
closed-circuit television	CCTV
Coastech Research Inc.	Coastech
coefficient of variation	CV
comparative work index	CWi
Concurrent Approval Regulation	CA Regulation
Cone Penetration Test	CPT
Construction Execution Plan	CEP
copper	Cu
cost, insurance, and freight	CIF
Coulter Creek Access Road	CCAR
counter-current decantation	CCD
customer's baseline load	CBL
cut-off grade	COG
Delegation of Authority Guideline	DOAG
Demand Side Management	DSM
Differential Global Positioning System	DGPS

digital elevation model	DEM
direct cyanide leaching	DCN
Distributed Control System	DCS
drill and blast techniques	DBT
EBA Engineering Consultants Ltd.	EBA
Eco Tech Laboratories, Stewart Group	Eco Tech
end-of-period	EoP
Engelmann Spruce – Subalpine Fir	ESSF
engineering, procurement, and construction management	EPCM
engineering, procurement, and construction	EPC
Environmental Discharge Flood	EDF
Environmental Management Plan	EMP
Environmental Management System	EMS
Esso Minerals Canada Ltd.	Esso Minerals
Factor of Safety	FOS
Falconbridge Ltd.	Falconbridge
fisheries sensitive zone	FSZ
fluorine	F
Free Carrier	FCA
G&T Metallurgical Services Ltd.	G&T
gas insulated	GIS
Gemcom Footprint Finder	FF
Gemcom Software International Inc.	Gemcom
general and administrative	G&A
general mine expense	GME
gold equivalent	AuEQ/AuEQV
gold	Au
Golder Associates Ltd.	Golder
Goods and Services Tax	GST
Granduc Mines Ltd.	Granduc
Granmac Services Ltd.	Granmac
gravitational constant	*g*
Ground Penetrating Radar	GPR
Harmonized Sales Tax	HST
Hazelton Volcanics	HV
Hazen Research Inc.	Hazen
health, safety, and environmental	HSE
Health, Safety, Environment, Security, Community Plan	HSESC
heating, ventilation, and air conditioning	HVAC
high pressure grinding rolls	HPGR
high-density polyethylene	HDPE
Human Resources	HR
hydrogen peroxide	H_2O_2
independent power producer	IPP
inductively coupled plasma	ICP
Inflow Design Flood	IDF

Interior Cedar – Hemlock	ICH
internal diameter	ID
internal rate of return	IRR
interramp angles	IRA
inverse distance weighted	IDW
Joint Health and Safety Committee	JHSC
Kerr, Sulphurets, and Mitchell	KSM
Klohn Crippen Berger Ltd.	KCB
Köeppern Machinery Australia Pty Ltd.	Köeppern
Land and Resource Management Plan	LRMP
lead	Pb
Left-hand-right hand-left hand rotation rule	LRL
Lerchs-Grossman	LG
Levelton Consultants Ltd.	Levelton
life-of-mine	LOM
Light Detection and Ranging	LiDAR
lime	CaO
Load-Haul-Dump	LHD
locked cycle tests	LCT
London Metal Exchange	LME
Magneto Telluric	MT
maintenance and repair contracts	MARC
Maximum Credible Earthquake	MCE
McElhanney Consulting Services Ltd.	McElhanney
McGladrey & Associates Professional Land Surveyors	McGladrey
McTagg Diversion Tunnel	MTDT
Measured + Indicated	MI
metabisulphite	MBS
Metal Mining Effluent Regulation	MMER
methyl isobutyl carbinol	MIBC
Metso Minerals Industries Inc.	Metso
Mineral Titles Online	MTO
MineSight Economic Planner	MS-EP
MineSight Strategic Planner	MS-SP
Ministry of Energy Mines and Petroleum Resources	MEMPR
Mintec Inc.'s MineSight®	MineSight
Mitchell Diversion Tunnel	MDT
Mitchell Ore Processing Complex	Mitchell OPC
Mitchell Thrust Fault	MTF
Mitchell-Treaty twin tunnels	MTT
molybdenum	Mo
Moose Mountain Technical Services	MMTS
motor control centres	MCCs
Multiple Pulse in Air	MPiA
Multi-Year Area-Based	MYAB
National Instrument 43-101	NI 43-101

Navigable Waters Protection Act	NWPA
nearest neighbour	NN
net present value	NPV
Net Smelter Prices	NSP
Net Smelter Return	NSR
Newhawk Gold Mines Ltd.	Newhawk Gold
Newmont Mining Corp.	Newmont
non-destructive testing	NDT
Noranda Inc.	Noranda
Northwest Transmission Line	NTL
not potentially acid generating	NPAG
operation and maintenance	O&M
Operator Interface Stations	OIS
Ore Control System	OCS
overall angles	OA
peak ground acceleration	PGA
Phelps Dodge Corp.	Phelps Dodge
pit slope angle	PSA
Placer Dome Inc.	Placer Dome
Pocock Industrial Inc.	Pocock
potassium amyl xanthate	PAX
potentially acid generating	PAG
Predictive Ecosystem Mapping	PEM
Prefeasibility Study	PFS
Pretium Resources Inc.	Pretium
Probable Maximum Flood	PMF
Probable Maximum Precipitation	PMP
programmable logic controller	PLC
Provincial Sales Tax	PST
Qualified Person	QP
quality assurance/quality control	QA/QC
quantile-quantile	QQ
Quartz-sericite-pyrite	QSP
Raewyn Copper	RC
Requests for Information	RFIs
Rescan Environmental Services Ltd.	Rescan
Resource Modeling Inc.	RMI
rhenium	Re
rock quality designation	RQD
Rock Storage Facility	RSF
run-of-mine	ROM
SAG mill comminution	SMC
Seabridge Gold Inc.	Seabridge
semi-autogenous grinding	SAG
semi-autogenous-ball mill-pebble crushing	SABC
SGS Minerals Services	SGS

Shuttle Reconnaissance Topography Mapping	SRTM
silver	Ag
Social and Community Management System	SCMS
Society for Mining, Metallurgy, and Exploration	SME
sodium cyanide	NaCN
sodium hydrosulphide	NaHS
sodium silicate	Na_2SiO_3
sodium sulphide	Na_2S
solids liquid separation	SLS
Special Use Permit	SUP
specific gravity	SG
standard penetration test	SPT
standard reference material	SRM
Stantec Consulting Ltd.	Stantec
static Var compensator	SVC
Sulphidization, Acidification, Recycling of precipitate and Thickening of precipitate	SART
sulphur dioxide	SO_2
Sulphurets Thrust Fault	STF
Sulphurets-Mitchell Conveyor Tunnel	SMCT
sulphuric acid	H_2SO_4
Tailing Management Facility	TMF
Terrestrial Ecosystem Mapping	TEM
Tetra Tech-Wardrop	Tetra Tech
The Claim Group Inc.	TCG
three-dimensional	3D
time and material	T&M
Total Investment Cost	TIC
total suspended sediments/total suspended solids	TSS
Treaty Ore Processing Complex	Treaty OPC
ungulate winter range	UWR
Uniform Hazard Response Spectra	UHRS
University of British Columbia	UBC
Valley of the Kings	VOK
valued component	VC
variable frequency drives	VFD
water storage dam	WSD
Water Storage Facility	WSF
Water Treatment Plant Energy Recovery	WTPER
water treatment plant	WTP
weight/weight	w/w
WN Brazier Associates Inc.	Brazier
work index	Wi
Workplace Hazardous Materials Information System	WHMIS

1.0 SUMMARY

1.1 INTRODUCTION

Seabridge Gold Inc.'s (Seabridge) KSM (Kerr-Sulphurets-Mitchell) Project involves the development of a major gold-copper deposit located in northwest British Columbia (BC) off Highway 37, approximately 68 km by air north-northwest of Stewart, BC. The KSM Project includes four major mineralized zones, identified as the Mitchell, Kerr, Sulphurets, and Iron Cap deposits. The deposits contain significant gold, copper, silver, and molybdenum mineralization.

This National Instrument 43-101 (NI 43-101) compliant report on the KSM property has been prepared by Tetra Tech-Wardrop (Tetra Tech), for Seabridge and has been based on work produced by Tetra Tech and the following independent consultants:

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

Mr. Michael J. Lechner (P.Geo., RPG, CPG) of RMI visited the property most recently from August 30 to September 1, 2011, 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, September 10, 2009, and April 13, 2010. He is the QP for matters relating to mining, mining capital, and mine operating costs for the open pit aspects of the project.

Dr. Jianhui (John) Huang (P.Eng.) of Tetra Tech visited the property on September 16, 2008, and is the QP for matters relating to the metallurgical testing review, mineral processing, process operating costs, TMF area water treatment and process-related infrastructure capital costs, and overall report preparation.

Dr. Sabry Abdel Hafez (P.Eng.) of Tetra Tech is the QP for matters relating to the financial analysis.

Mr. Harold Bosche (P.Eng.) of BVL visited the property on September 16, 2008, and is the QP for matters relating to the site infrastructure layouts, tunnel conveyor, rope conveyor, tailing delivery, reclaim pumping and piping systems, and associated capital costs.

Mr. Neil Brazier (P.Eng.) of Brazier visited the property on September 16, 2008, and from September 12 to 16, 2011, and is the QP for matters relating to power supply, energy recovery plants, and associated costs.

Mr. Graham Parkinson (P.Geo.) of KCBL visited the property from July 26 to August 2, 2010, as well as during the summers of 2008 and 2009, and autumn of 2007. He is the QP for matters relating to the geotechnical design of tunnels, dams, and operating and closure costs for cyclone sand raises of tailing dams, water management, the mine area water treatment plant (WTP), and temporary acid rock drainage (ARD) and sediment control.

Mr. R.W. (Bob) Parolin (P.Eng.) of McElhanney visited the property on June 21, 2008, and during the summers of 2009, 2010, and 2011 and is the QP for matters relating to main and temporary access roads and associated costs.

Mr. Kevin Jones (P.Eng.) of EBA visited the property on September 24 to 26, 2009, and September 12 to 13, 2011, and is the QP for matters relating to winter access road and associated costs.

Mr. Pierre Pelletier (P.Eng.) of Rescan visited the property on April 13, 2010 and May 16, 2012. He is the QP for matters relating to environmental considerations.

Mr. Warren Newcomen (P.Eng) of BGC visited the property on June 1 to 5, 2009 and July 26 to 28, 2010, and is the QP for matters relating to open pit slope stability.

Mr. Darby Kreitz (P.Eng.) of Allnorth visited the site on April 14, 2010, and is the QP for matters relating to construction cost estimates for the water storage dam (WSD) and tailing starter dams, surface diversions, excavations, and earthworks.

Dr. Ross Hammett (P.Eng.) of Golder visited the site on August 8 to 10, 2010, and on October 18 and 19, 2011, and is the QP for matters relating to block caving mining.

Mr. Tony Wachmann (P.Eng.), Director of Mining, Metallurgy, and Infrastructure at Stantec, visited the site on May 16, 2012, and is the QP for matters relating to tunnelling, tunnel construction, and related costing.

The capital and operating costs for the 2012 KSM Prefeasibility Study (PFS) have been estimated to a +25/-10% level of accuracy. All dollar figures presented in this section are stated in US dollars, unless otherwise specified. The Bank of Canada three-year average exchange rate of Cdn$1.00 to US$0.96 has been used, unless otherwise specified.

1.2 GEOLOGY

The KSM property is located in northwest 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, 68 km by air north-northwest of Stewart, BC and 21 km south-southeast of the former 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. After having been granted an option by Seabridge to earn an interest at KSM, between 2002 and 2005, Falconbridge Ltd. (Falconbridge)/Noranda Inc. (Noranda) conducted target evaluation and testing of several occurrences on the property. That work focused on exploration concepts deemed to be appropriate for copper-rich porphyry targets. In 2006, Falconbridge/Noranda agreed to sell its option back to Seabridge having recognized that the district’s 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 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 deformation along the Sulphurets Thrust Fault (STF) was diverted into the Kerr area along pre-existing structures. The mineralized area forms a fairly continuous, north-south trending west dipping irregular body measuring about 1,700 m long and up to 200 m thick.

The Sulphurets deposit is comprised of two distinct zones referred to as the Raewyn Copper-Gold Zone and the Breccia Gold Zone. 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 Raewyn Copper-Gold Zone strikes northeasterly and dips about 45° to the northwest. The Breccia Gold Zone hosts mostly gold-bearing pyritic material mineralization with minor chalcopyrite and sulfosalts in a K-feldspar-siliceous hydrothermal breccia that apparently crosscuts the Raewyn Copper-Gold Zone. The Breccia Gold Zone strikes northerly and dips westerly.

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 (MTF). These rocks tend to be intensely altered and characterized by abundant sericite and pyrite with numerous quartz stockwork veins and sheeted quartz veins (phyllic alteration) 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. In 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 is dispersed over a very large area and 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 are remarkably consistent between drill holes, which is consistent with a large, stable, and long-lived hydrothermal system.

The Iron Cap Zone, which is located about 2,300 m northeast of the Mitchell Zone, is well exposed and consists of intensely altered intrusive, sedimentary, and volcanics. The Iron Cap deposit is a separate, distinct mineralized zone within the KSM district. It is thought to be related to the other mineralized zones but differs in that much of the host rock is hydrothermally altered intrusive (porphyritic monzonite to diorite) rather than altered volcanics and sediments. There is a high degree of silicification that overprints earlier potassic and chloritic alteration. Intense phyllic alteration and high density of stockwork veining, which are pervasive at the nearby Mitchell Zone, are less pervasive at Iron Cap. The surface expression of the Iron Cap Zone measures about 1,500 m (northeast-southwest) by 600 m (northwest-southeast).

Using MineSight® software, RMI created a three-dimensional (3D) computerized block models for the Kerr, Sulphurets, Mitchell, and Iron Cap zones. In general, RMI used the same basic estimation techniques that were developed for these deposits in previous years.

The Kerr Zone has been delineated by 31,359 m of core drilling from 159 holes that are spaced between 50 to 100 m apart. The majority of the drilling data were collected by Placer Dome Inc. (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 porphyritic monzonite 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. Block gold, copper, and silver grades were estimated using inverse distance and nearest neighbour methods.

The Sulphurets Zone has been delineated by about 36,601 m of core drilling from 126 diamond core holes that are spaced at intervals ranging between 50 to 100 m. Seabridge (plus Falconbridge) collected about 76% of the total Sulphurets drilling; most of the remaining data were collected by Placer Dome. 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 STF. The Raewyn Copper-Gold 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 breccia that cross cuts the intrusions of the Raewyn Copper-Gold Zone. Block gold and copper grades were estimated using inverse distance and nearest neighbour methods.

There were 154 diamond core holes used for the Mitchell block model that were spaced at approximately 75 to 100 m intervals, totalling 56,952 m of data. Gold, copper, silver, and molybdenum grades were estimated with 15-m-long drill hole composites using inverse distance and nearest neighbour methods. RMI validated the estimated block grades using visual and statistical methods. It is RMI's opinion that the Mitchell grade models are globally unbiased and represents a reasonable estimate of in-situ resources. Measured, Indicated, and Inferred Mineral Resources were classified for a portion of the estimated blocks based on mineralized continuity and the distance to drilling data coupled with the number of holes that were used in the estimate.

Gold, copper, silver, and molybdenum grades were estimated for the Iron Cap Zone using 52 diamond core holes totalling about 17,790 m of data. The majority of the drilling data (87%) was collected in 2009 and 2010 by Seabridge. The block grades were estimated using inverse distance and nearest-neighbour methods.

A gold equivalent grade (AuEQ) was calculated for the estimated block grades for all four zones 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 cut-off grade (COG) in Table 1.1.

1.3 PROPERTY DESCRIPTION AND LOCATION

The KSM Project area is located in the coastal mountains of north-western BC. The proposed pit areas lie within the headwaters of Sulphurets Creek, which is a tributary of the Unuk River. The proposed Tailing Management Facility (TMF) will be located within the tributaries of Teigen and Treaty creeks. 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 KSM Project comprises three discontinuous claim blocks. The claim blocks are referred to as:

1. the KSM claim block
2. the Seabee/Tina claim block
3. the KSM placer claim block.

The first two claim blocks (KSM and Seabee/Tina) contain 117 mineral claims, consisting of both cell and legacy claims. The total area of the three claim blocks covers an area of 52,133.26 ha. The Seabee/Tina claim block is located about 19 km northeast of the Kerr-Sulphurets-Mitchell-Iron Cap mineralized zones. The Seabee/Tina claim block is currently under consideration as the site for the proposed infrastructure. The claims are 100% owned by Seabridge Gold Inc. Placer Dome (now Barrick Gold Corp.) retains a 1% net smelter royalty on the property that is capped at $4.5 M.

Annual holding costs for all claims (lode and placer) are approximately Cdn$173,000. In 2007, assessment work was filed to advance the year of expiry to 2018. Assessment work was completed on most of the Seabee property claims in 2010 with that work filed in February 2011, which advanced expiry dates to 2017. The placer claims have been kept in good standing by paying fees in lieu of completing assessment work. The Claim Group Inc. (TCG) was contracted to act as a land manager and mineral tenure agent for Seabridge.

Figure 1.1 General Location Map

Source: Rescan.

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 metres above sea level (masl) 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 indicate that the Mitchell Glacier has retreated approximately one kilometre laterally and several hundred metres vertically since 1991.

The deposit portion of the property is drained by the Sulphurets and Mitchell watersheds that empty into the Unuk River, which flows westward to the Pacific Ocean through Alaska. The process plant site and the TMF drain into the Bell-Irving watershed, which is a tributary to the Nass River. The tree line lies at about 1,240 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 and moose, are rare due to the rugged topography and restricted access; however, bears and mountain goats are common.

The climate is generally typical 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 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. The property can be accessed only via helicopter.

There are deep-water loading facilities for shipping bulk mineral concentrates located in Stewart. The facilities are currently used by the Huckleberry Mine. The nearest railway is the Canadian National Railroad (CNR) Yellowhead route, which is located approximately 220 km southeast of the property. This line runs east-west, and can deliver concentrate to deep water ports near Prince Rupert and Vancouver, BC.

The property and its access routes are on Crown land; therefore, surface and access rights are granted under, and subject to compliance with, the *Mineral Tenure Act* or the *Land Act* or, at the discretion of the Crown, under the *Mining Right of Way Act*. There are no settlements or privately owned land in the area; there is limited commercial recreational activity in the form of helicopter skiing, rafting tours, and guided fishing adventures. The closest power transmission lines run along the Highway 37A corridor to Stewart, approximately 50 km southeast of the property. The Eskay Creek Mine produced its own diesel-generated power. Construction of BC Hydro's Northwest Transmission Line (known as the NTL) is underway with completion scheduled for the spring of 2014.

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 on 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 Canada Ltd. (Esso

Minerals) in 1969; Kerr was first drilled by Brinco in 1985, and Mitchell was first drilled by Newhawk Gold Mines Ltd. (Newhawk Gold) in 1991.

In 1989, Placer Dome acquired a 100% interest in the Kerr Zone from Western Canadian Mines; in 1990, Placer Dome acquired the adjacent Sulphurets property from Newhawk Gold. The Sulphurets property also hosts the Mitchell Zone 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 downstream in Sulphurets Creek. On the Bruceside property, immediately to the east, limited underground development and test mining was undertaken in the 1990s on narrow, gold-silver bearing quartz veins at the West Zone. This property is currently owned by Pretium Resources Inc.

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, subaerial to subaqueous epithermal veins and volcanogenic massive sulfide zones at Eskay Creek, Snip, Bruceside, and Granduc.

1.7 Resources

RMI constructed 3D block models for the Kerr, Sulphurets, Mitchell, and Iron Cap zones. Various 3D wireframes were used to constrain the estimate of block grades (e.g. lithology, alteration, structural, and grade envelopes). 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, RMI believes that the grade models are globally unbiased and suitable for subsequent pit optimization studies. The estimated block grades were classified into Measured (Mitchell only), Indicated, and Inferred Mineral Resource categories based on mineralized continuity, along with distance to data in conjunction with the number of drill holes that were used to estimate block grades. Table 1.1 summarizes the estimated Measured, 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

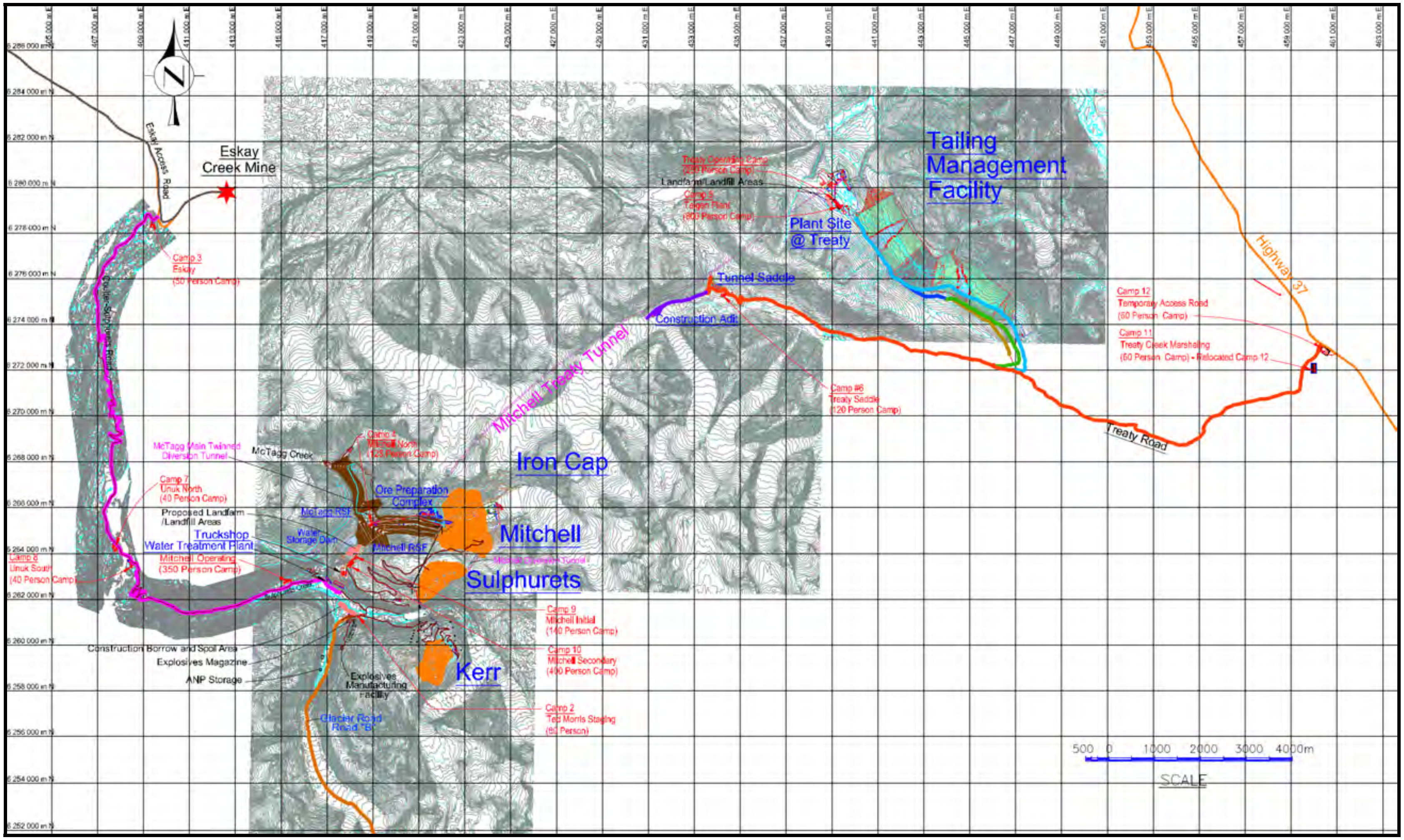
Zone	t (000)	Au (g/t)	Au (000 oz)	Cu (%)	Cu (Mlb)	Ag (g/t)	Ag (000 oz)	Mo (ppm)	Mo (Mlb)
Measured Resources									
Mitchell	724,000	0.65	15,130	0.18	2,872	3.2	74,487	56	89.4
Indicated Resources									
Kerr	270,400	0.24	2,086	0.46	2,741	1.1	9,563	n/a	n/a
Sulphurets	370,900	0.59	7,036	0.21	1,717	0.8	9,540	49	40.1
Mitchell	1,052,900	0.58	19,634	0.16	3,713	3.1	104,940	59	136.9
Iron Cap	361,700	0.44	5,117	0.21	1,674	5.4	62,796	47	37.5
Total	**2,055,900**	**0.51**	**33,873**	**0.22**	**9,845**	**2.8**	**186,838**	**54**	**214.5**
Measured Plus Indicated Resources									
Kerr	270,400	0.24	2,086	0.46	2,741	1.1	9,563	n/a	n/a
Sulphurets	370,900	0.59	7,036	0.21	1,717	0.8	9,540	49	40.1
Mitchell	1,776,900	0.61	34,764	0.17	6,585	3.1	179,426	58	226.3
Iron Cap	361,700	0.44	5,117	0.21	1,674	5.4	62,796	47	37.5
Total	**2,779,900**	**0.55**	**49,003**	**0.21**	**12,717**	**2.92**	**261,325**	**55**	**303.8**
Inferred Resources									
Kerr	85,000	0.24	656	0.28	525	0.9	2,460	n/a	n/a
Sulphurets	177,100	0.50	2,847	0.15	585	1.2	6,833	30	11.7
Mitchell	567,800	0.44	8,032	0.14	1,752	3.4	62,068	51	63.8
Iron Cap	297,300	0.36	3,441	0.20	1,310	3.9	37,278	60	39.3
Total	**1,127,200**	**0.41**	**14,976**	**0.17**	**4,172**	**3.00**	**108,638**	**50**	**114.8**

Note: Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

1.8 Overall General Arrangement

Figure 1.2 is an overview of the general arrangement of the project.

Figure 1.2 KSM Overall Site Plan



1.9 MINE PLANNING

1.9.1 MINING METHODS

Open pit mining and block cave underground mining methods will be used for the KSM Project. Studies conducted by Golder show a viable block cave mine after initial open pit mining at the Mitchell deposit, and a viable standalone block cave mine at Iron Cap. Kerr and Sulphurets are not suitable for block cave mining and are designed as open pits.

The use of block cave mining in this PFS reduces the quantity of mined waste rock by approximately 2.3 Bt, or approximately 40% from the last study.

1.9.2 LG PIT LIMITS

MMTS has produced a series of Lerchs-Grossman (LG) pit shell optimizations using resource models provided by RMI. The pit optimizations use mining, processing, water treatment, tailing, general and administrative (G&A) costs, and process metal recoveries. These are derived for each of three separate pit areas – Mitchell, Sulphurets, and Kerr. The RMI resource models classify the mineralization as Measured, Indicated, and Inferred; only Measured and Indicated categories are used in the pit optimization.

COG is determined using the Net Smelter Return (NSR) in Cdn$/t, which is calculated using Net Smelter Prices (NSP). The NSR (net of offsite concentrate and smelter charges and onsite mill recovery) is used as a cut-off item for break-even ore/waste selection. The NSP includes metal prices, US currency exchange rate, and offsite transportation, smelting, and refining charges. The metal prices from travelling averages, and resultant NSPs used are shown in Table 1.2.

Table 1.2 Metal Prices and NSP

	Metal Price (US$)	NSP (Cdn$)
Cu	3.21/lb	2.93/lb
Au	1,244/oz	39.02/g
Ag	22.98/oz	0.649/g
Mo	14.14/lb	9.70/lb

LG delineated resources are in-situ and use an NSR COG specific to each mining area but do not include any mining dilution or mining loss.

MMTS notes that the economic pit limits are based on mining unit costs derived to meet the local conditions for the project and the specific project arrangements for waste rock management, water management, environmental, and reclamation within this study, as well as certain input parameters, such as pit slope angles, process recoveries, environmental considerations, and reclamation requirements. All of these components affect the mining quantities and activities to release the specified ore and, as such, affect the economic pit limits.

As can be expected during normal progressive mine optimization stages for all open pit mines, some further refinements may result from additional detailed data acquisition. Future operational cost projections or metal price changes could impact the projected pit limits, ore reserves, and waste quantities.

Because of the difficulty in predicting relevant metal prices over such a long project life, the ultimate LG pit limits in this study for Sulphurets and Kerr are selected where an incremental increase in pit size does not significantly increase the pit resource, or an incremental increase in the pit resource results in only marginal economic return. In other words, rather than selecting an economic ultimate pit based on a fixed price case (even if discounted cash flow considerations are included), the ultimate pits for Sulphurets and Kerr are selected where the economic margins drop off.

The ultimate pit for Mitchell is selected where the operating cost per tonne of ore for mining one bench lower by open pit method begins to exceed the unit operating cost of mining incrementally higher with a block cave.

This establishes the limits to the mineable resource base for the mine design work. Price and cash flow sensitivities can then be performed within a more robust mine plan. The LG pit delineated resource for each pit area is summarized in Table 1.3 and Table 1.4.

Table 1.3 Measured and Indicated LG Pit Resources

Pit Area	In Situ Ore (Mt)	In Situ Grades					Waste (Mt)	Strip Ratio (t:t)
		NSR (Cdn$/t)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)		
Mitchell	980	30.3	0.656	0.171	3.05	61	1,342	1.4
Sulphurets	310	27.8	0.599	0.226	0.78	52	859	2.8
Kerr	234	32.0	0.253	0.475	1.23	-	476	2.0
Total	**1,524**	**30.1**	**0.582**	**0.229**	**2.31**	**50**	**2,677**	**1.8**

Note: NSR cut-offs for each area are: Mitchell Cdn$9.57, Sulphurets Cdn$10.17, Kerr Cdn$9.61.

Table 1.4 Measured and Indicated LG Pit Resources – Insitu Metal

Pit	Au (M oz)	Cu (M lb)	Ag (M oz)	Mo (M lb)
Mitchell	20.7	3,697	96.1	130.8
Sulphurets	6.0	1,544	7.8	35.6
Kerr	1.9	2,444	9.2	0.0
Total	**28.5**	**7,685**	**113.1**	**166.4**

1.9.3 FUTURE OPEN PIT OPPORTUNITY

The Inferred Resources within the selected LG pit limits used for this study are included in the waste tonnages in Table 1.3. These Resources, included within the ultimate pit limits, have the potential to be upgraded to Reserves with future exploration drilling or when the pit grade control and blast hole assaying system is in place. Additionally, Inferred material may also have the potential to expand the future economic pit limit if they are upgraded by future exploration drilling.

To analyze this potential, an additional set of pit optimization runs have been completed for the Kerr and Sulphurets deposits, allowing Inferred material to be given a value as well. The Mitchell pit is restricted from further expansion by the block cave designed below the pit. Table 1.5 and Table 1.6 summarize the results of these Inferred LG pit limits incremental to the Measured and Indicated pit resources shown in Table 1.3 and Table 1.4.

Table 1.5 Incremental *Inferred* LG Pit Resources

Pit Area	In Situ Mineralized Material (Mt)	In Situ Grades NSR (Cdn$/t)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Waste (Mt)	Strip Ratio (t:t)
Mitchell	137	15.8	0.294	0.121	3.28	51	(124)	-0.9
Sulphurets	160	20.8	0.509	0.156	1.34	33	176	1.1
Kerr	82	21.6	0.247	0.293	1.12	-	77	0.9
Total	**379**	**19.2**	**0.374**	**0.173**	**1.99**	**32**	**129**	**0.3**

Table 1.6 Incremental *Inferred* LG Pit Resources – Insitu Metal

Pit	Au (M oz)	Cu (M lb)	Ag (M oz)	Mo (M lb)
Mitchell	1.3	365	14.4	15.4
Sulphurets	2.6	550	6.9	11.6
Kerr	0.7	530	2.9	0.0
Total	**4.6**	**1,446**	**24.3**	**26.9**

Note: Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

1.10 GEOTECHNICAL ASPECTS OF THE MINE PLAN

The high topographic relief in the areas of the open pits, block cave mines, and the Rock Storage Facilities (RSFs) requires specific consideration. Conservative designs, alternative/mitigating scenarios, and extra data and analyses have been included in the design.

The mine plan calls for development of three open pits - Kerr, Sulphurets, and Mitchell – and two areas of underground development, beneath the Mitchell pit and at Iron Cap. All the RSFs are confined to the lower Mitchell and McTagg valleys, except the temporary Sulphurets RSF, which is situated on the south side of the Sulphurets ridge.

Potential geohazards are identified in the area of the proposed open pits, block cave mine, RSFs, roads, and other infrastructure; designs include the mitigation of geohazards such as avalanche control, provision of avalanche run-out routes, barriers, and avalanche area and slope hazard avoidance as appropriate. Slope design parameters for the proposed open pits are provided, and the required dewatering/depressurization system for the proposed open pits is evaluated. Geotechnical considerations are provided for the caving mining.

1.10.1 MITCHELL, SULPHURETS, KERR, AND IRON CAP DEPOSITS

A multi-component site investigation and design study focused on the Mitchell pit in 2009, and the Sulphurets, Kerr, and Iron Cap pits in 2010, has been carried out to support the open pit slope angle design. Site investigations included geotechnical drilling, borehole televiewer surveys, photogrammetric mapping, packer testing, installation and reading of piezometers, and field and laboratory testing of core samples. The results of these geotechnical investigations plus additional geotechnical drilling and mapping that was undertaken at Mitchell in 2011 have been applied to the designs of the open pit slopes and block caving mines.

A hydrogeological study has also been undertaken to support the designs. A three-dimensional hydrogeological model has been developed for the area encompassing the three proposed open pits; pit dewatering/pit slope depressurization simulations have been carried out. A depressurization system, including vertical wells and horizontal drains for all pits, as well as an adit for the north wall of the proposed Mitchell pit, has been included in the designs.

The proposed ultimate (final) Mitchell pit slopes will be 1,200 m high. The recommended interramp slope angles for the Mitchell pit vary from 34° to 54°. The proposed ultimate slopes of the Sulphurets pit will be up to approximately 650 m high with the recommended interramp slope angles of the Sulphurets pit varying from 36° to 50°. The proposed ultimate slopes of the Kerr pit will be up to approximately 600 m high. The recommended interramp slope angles of the Kerr pit vary from 34° to 50°. All of the slope design parameters provided for the pit optimization and final pit design are based on wall orientation, overall wall height, and geotechnical domain, with consideration of the dominant controls on slope stability in that domain.

Achieving the proposed open pit slope angles will involve important considerations during mine operations, including the dewatering and depressurization of the proposed pit slopes previously discussed. Deformation monitoring of the interramp and overall pit slopes will also be required to document slope performance and confirm design assumptions.

The quality of the rock mass in the block caving mining areas is generally good and the intact rock is strong to very strong. The fracture spacing is moderately wide to wide and the in-situ rock blocks are large in size. Based on this, the fragmentation is expected to be coarse, particularly during the initial draw of each column; therefore, it is proposed to "pre-condition" the rock masses as a mitigation measure at Mitchell and Iron Cap using hydraulic fracturing. Conservative production rates have also been applied in the design to account for possible restrictions in drawpoint availability due to coarse fragmentation.

The economically viable footprints for caving mining at Mitchell and Iron Cap are very large.

Even using conservative estimates of the size of the undercut required to initiate and propagate the cave up to ground surface, there is no concern about being able to develop a fully functional cave within the viable footprint.

1.10.2 Geotechnical Design Parameters for Mine Rock Storage Facilities

For this study, there are three primary design considerations for the mine RSFs:

- foundation preparation
- maximum lift height
- closure slope criteria.

Allowances have been made to accommodate these design considerations.

Design work assumes that all foundation material is confined and consolidated as part of the dumping sequence. Dumping directions also reduce the operational and downslope risks using typical operating practices from experience at other western Canadian open pit operations on mountainous terrain. A cost allowance has been included to remove tills from the foundations and adjacent areas, on an as required basis and where possible for reclamation covers.

The mining progression is designed to build RSFs in lifts (bottom-up construction) to consolidate the foundations and reduce downslope risks. Top-down dumps have been designed as 100 m maximum lifts, which is significantly less than the 400 m maximum lift height in practice at other mountain mines in western Canada.

Final RSF configurations are designed with terraces at "as dumped" angle of repose, with flat benches between terraces. The overall slope angle is between 26° and 30° to provide the ability for re-sloping to accommodate the end land use and reclamation plan.

1.11 ROCK STORAGE FACILITY OPERATIONS

RSFs for the Kerr, Sulphurets, and Mitchell pits are confined to the lower Mitchell and McTagg areas. This increases the costs of waste haulage from the further mining zones, but reduces the amount of disturbed area, as well as the post-mining reclamation and waste treatment requirements. The Sulphurets RSF will be a temporary staging area used early in the mine life; rock will be hauled to McTagg and Mitchell. The safe operation of high-relief RSFs in mountain terrain has been successfully demonstrated at other operations in western Canada; these approaches are being considered in this planning work. Costs have been included to address reclamation and post-closure requirements.

1.12 OPEN PIT MINING OPERATIONS

Detailed pit phases have been developed from the results of the LG sensitivity analysis integrating detailed pit slope criteria and highwall roads. The ultimate pits have been divided into smaller mining phases, or pushbacks, to allow for more even waste stripping in the optimized scheduling stage of the project design.

The Pit Reserves shown in Table 1.10 include estimated mining loss and dilution, as outlined in Table 1.8. The dilution grades provided in Table 1.9 represent the average grade of material below the incremental COG for each pit area. Waste/ore COGs are based on NSR grades and vary for each pit area as shown in Table 1.7.

Table 1.7 Waste/Ore COG by Pit

Pit	COG NSR (Cdn$/t)
Mitchell	9.57
Sulphurets	10.17
Kerr	9.61

Table 1.8 Open Pit Mining Loss and Dilution

Pit	Mining Loss	Mining Dilution
Mitchell	2.2%	0.8%
Sulphurets	5.3%	3.9%
Kerr	4.5%	3.2%

Table 1.9 Open Pit Dilution Grades

	Mitchell Pit	Kerr Pit	Sulphurets Pit
Cu (%)	0.043	0.106	0.056
Au (g/t)	0.229	0.141	0.333
Ag (g/t)	1.45	0.78	0.59
Mo (ppm)	59.4	-	19.0
NSR ($/t)	7.55	7.60	8.19

Table 1.10 Summary – Pit Phase Reserves

		Diluted Grades						
Area	**Ore (Mt)**	**NSR (Cdn$/t)**	**Au (g/t)**	**Cu (%)**	**Ag (g/t)**	**Mo (ppm)**	**Waste (Mt)**	**Strip Ratio (t:t)**
Mitchell Pit								
M681	88	39.7	0.839	0.227	3.40	24.6	67	0.8
M682i	239	31.4	0.689	0.177	2.58	64.0	423	1.8
M683i	116	28.9	0.643	0.155	3.38	53.0	287	2.5
M684i	209	24.3	0.544	0.136	2.34	86.4	257	1.2
M685i	322	28.2	0.618	0.157	3.24	62.3	486	1.5
Subtotal	**973**	**29.3**	**0.642**	**0.163**	**2.92**	**63.4**	**1,519**	**1.6**
Kerr Pit								
K691	242	30.6	0.244	0.454	1.20	0.0	665	2.7
Sulphurets Pit								
S691	101	31.4	0.654	0.261	0.59	54.9	167	1.7
S692i	217	25.0	0.553	0.200	0.88	48.6	683	3.2
Subtotal	**318**	**27.0**	**0.585**	**0.219**	**0.79**	**50.6**	**850**	**2.7**
TOTAL	**1,533**	**29.0**	**0.567**	**0.221**	**2.20**	**50.7**	**3,035**	**2.0**

1.13 UNDERGROUND MINING OPERATIONS

The underground mine designs for both the Mitchell and Iron Cap deposits are based on modelling using Gemcom Software International Inc.'s (Gemcom) Footprint Finder (FF) and PCBC software. The ramp-up and maximum yearly mine production rates were established based on the rate at which the drawpoints are constructed, and the initial and maximum production rates at which individual drawpoints can be mucked. The values chosen for these inputs were based on industry averages adjusted to suit the anticipated conditions. In particular, the initial and maximum drawpoint production rates were reduced to simulate production environments with expected large fragmentation.

Mitchell is estimated to have a production ramp-up period of 6 years, steady state production at 20 Mt/a for 14 years, and then ramp-down production for another 7 years. Iron Cap is estimated to have a production ramp-up period of 4 years, steady state production at 15 Mt/a for 8 years, and then ramp-down production for another 8 years. The pre-production periods are 7 years at Mitchell and 6 years at Iron Cap. During these periods, the majority of the ventilation and material movement infrastructure is excavated and installed and development of most of the undercut and preconditioning levels occurs.

The underground mining NSR cut-offs vary by operation and are shown in Table 1.11. The underground mining dilution is shown in Table 1.12 and is material with no grade. There is additional dilution within the block cave reserves (material below the NSR cut-offs) and it is mixed within the cave zone as the material is drawn down.

Table 1.11 NSR Cut-off by Underground Mine

Pit	NSR Cut-off (Cdn$/t)
Mitchell	15.41
Iron Cap	15.57

Table 1.12 Underground Mining Dilution (Zero Grade)

Pit	Mining Dilution
Mitchell	9%
Iron Cap	5%

Table 1.13 Summary – Underground Mining Reserves

		Diluted Grades			
UG Mine	Ore (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)
Mitchell	438	0.529	0.165	3.48	33.6
Iron Cap	193	0.450	0.196	5.32	21.5

1.14 MINE PRODUCTION SCHEDULE

Proven and probable ore reserves are summarized in Table 1.14 and Table 1.15; combined totals are provided in Table 1.16.

Table 1.14 Proven Reserves

Pit	Ore (Mt)	Diluted Grades Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Contained Metal Au (Moz)	Cu (Mlb)	Ag (Moz)	Mo (Mlb)
Mitchell Pit	476	0.673	0.171	3.05	60.9	10.3	1,798	47	64
Kerr Pit	-	0.000	0.000	0.00	0.0	0.0	-	0	0
Sulphurets Pit	-	0.000	0.000	0.00	0.0	0.0	-	0	0
Total Proven	**476**	**0.673**	**0.171**	**3.05**	**60.9**	**10.3**	**1,798**	**47**	**64**

Table 1.15 Probable Reserves

Pit	Ore (Mt)	Diluted Grades Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Contained Metal Au (Moz)	Cu (Mlb)	Ag (Moz)	Mo (Mlb)
Mitchell Pit	497	0.613	0.156	2.78	65.8	9.8	1,707	44	72
Kerr Pit	242	0.244	0.454	1.20	0.0	1.9	2,425	9	0
Sulphurets Pit	318	0.585	0.219	0.79	50.6	6.0	1,535	8	35
Mitchell Underground	438	0.529	0.165	3.48	33.6	7.4	1,589	49	32
Iron Cap Underground	193	0.450	0.196	5.32	21.5	2.8	834	33	9
Total Probable	**1,689**	**0.514**	**0.217**	**2.65**	**40.1**	**27.9**	**8,090**	**144**	**149**

Table 1.16 Proven and Probable Reserves

Area	Ore (Mt)	Diluted Grades Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Contained Metal Au (Moz)	Cu (Mlb)	Ag (Moz)	Mo (Mlb)
Mitchell Pit	973	0.642	0.163	2.92	63.4	20.1	3,506	91	136
Kerr Pit	242	0.244	0.454	1.20	0.0	1.9	2,425	9	0
Sulphurets Pit	318	0.585	0.219	0.79	50.6	6.0	1,535	8	35
Mitchell Underground	438	0.529	0.165	3.48	33.6	7.4	1,589	49	32
Iron Cap Underground	193	0.450	0.196	5.32	21.5	2.8	834	33	9
Total Proven & Probable	**2,164**	**0.549**	**0.207**	**2.74**	**44.7**	**38.2**	**9,889**	**191**	**213**

The proven and probable reserves of 38.2 M oz of gold (2.164 Bt at 0.549 g of gold per tonne) are derived from total Measured and Indicated Resources of 49.0 M oz of gold (2.780 Bt at 0.55 g of gold per tonne) and include allowances for mining losses and dilution.

A summary of the production schedule is provided in Table 1.17.

The production schedule is based on the pits being mined in the following years:

- The Mitchell deposit is mined by open pit from Year -2 to Year 23.
- The Sulphurets starter pit is mined from Year -2 to Year 6.
- The Sulphurets final pit is mined from Year 23 to Year 27.
- The Kerr deposit is mined by open pit from Year 27 to Year 50.
- The Mitchell deposit is mined by block cave from Year 26 to 55.
- The Iron Cap deposit is mined by block cave from Year 32 to Year 51.
- All mining of currently defined proven and probable reserves is completed in Year 55.

Open pit mining operations will be typical open pit operations in mountainous terrain in western Canada, with typical open-pit mining methods and equipment. Pit unit operations and activities are planned and costed to meet the local conditions; in particular, weather-related conditions including high snow fall. There is considerable operating and technical expertise, services, and support for the proposed operations both in western Canada and in the local area. The project is a large-capacity operation that utilizes large-scale equipment for the major operating areas in order to generate high productivities, and reduce unit and overall mining costs. Large-scale equipment will also reduce the labour requirement on site and will dilute the fixed overhead costs for 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.

The proposed underground operations will be typical of large underground block cave mines around the world using typical underground mobile and fixed equipment including crushers, conveyors, haulage trains and a fleet of secondary breakers. These mines are large-capacity operations that utilize large equipment to generate high productivity and lower unit costs.

Table 1.17 Summarized Production Schedule

		Year																		
	Unit	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11 to 20	21 to 30	31 to 40	41 to 50	51 to 55	LOM
Ore from Mine To Mill	**Mt**	**-**	**-**	**-**	**18**	**28**	**34**	**43**	**35**	**41**	**30**	**1**	**47**	**25**	**390**	**278**	**98**	**126**	**-**	**1,196**
Au	g/t	-	-	-	0.997	0.846	0.823	0.785	0.795	0.866	0.967	0.771	0.756	0.871	0.627	0.572	0.264	0.225	-	0.596
Cu	%	-	-	-	0.268	0.264	0.280	0.256	0.222	0.238	0.256	0.095	0.183	0.223	0.155	0.213	0.538	0.405	-	0.248
Ag	g/t	-	-	-	2.73	3.12	2.10	1.82	1.71	2.93	4.26	4.88	3.45	3.26	3.06	1.41	1.61	0.78	-	2.25
Mo	ppm	-	-	-	23.1	19.2	31.8	72.1	84.5	50.6	27.0	77.7	40.3	44.2	71.0	44.4	-	-	-	45.3
Ore To Stockpile	**Mt**	**1**	**6.3**	**7.6**	**30**	**0**	**4**	**30.5**	**56**	**53**	**7**	**14**	**35**	**27**	**65**	**-**	**-**	**-**	**-**	**337**
Au	g/t	0.341	0.383	0.344	0.581	0.498	0.333	0.438	0.542	0.635	0.676	0.398	0.412	0.388	0.305	-	-	-	-	0.466
Cu	%	0.288	0.241	0.193	0.201	0.134	0.134	0.121	0.131	0.148	0.168	0.070	0.106	0.107	0.074	-	-	-	-	0.126
Ag	g/t	0.96	1.26	2.53	2.03	2.06	1.87	1.66	1.67	2.22	2.10	4.49	2.66	1.55	1.74	-	-	-	-	2.04
Mo	ppm	93.5	28.2	19.2	30.2	39.4	21.3	53.1	77.9	75.3	65.0	77.1	76.7	88.7	86.1	-	-	-	-	70.1
Stockpile Reclaim	**Mt**	**-**	**-**	**-**	**10.0**	**16**	**13**	**4**	**13**	**6**	**17**	**47**	**-**	**22**	**85**	**104**	**-**	**-**	**-**	**337**
Au	g/t	-	-	-	0.617	0.596	0.640	0.289	0.492	0.630	0.673	0.676	-	0.455	0.404	0.354	-	-	-	0.474
Cu	%	-	-	-	0.176	0.276	0.187	0.150	0.121	0.134	0.156	0.157	-	0.121	0.112	0.089	-	-	-	0.128
Ag	g/t	-	-	-	2.34	1.57	2.45	3.19	0.60	0.52	2.13	2.13	-	2.50	2.36	1.86	-	-	-	2.05
Mo	ppm	-	-	-	35.5	27.0	35.4	30.8	26.3	37.3	78.1	76.3	-	74.7	77.5	77.8	-	-	-	68.7
Stockpile Inventory	Mt	1	6.9	14.5	34	18	9	35	79	126	116	84	119	124	104	0	0	0	0	-
Mitchell Underground	**Mt**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**33**	**199**	**189**	**16**	**438**
Au	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.661	0.518	0.515	0.549	0.529
Cu	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.210	0.166	0.159	0.124	0.165
Ag	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.23	3.43	3.36	1.99	3.48
Mo	ppm	-	-	-	-	-	-	-	-	-	-	-	-	-	-	16.0	22.1	46.8	56.9	33.6
Iron Cap Underground	**Mt**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**96**	**98**	**0.1**	**193**
Au	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.518	0.383	0.287	0.450
Cu	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.192	0.199	0.131	0.196
Ag	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.30	5.33	5.23	5.32
Mo	ppm	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	19.1	26.9	18.6	23.0
Mill Feed	**Mt**	**-**	**-**	**-**	**28**	**45**	**48**	**48**	**48**	**48**	**47**	**47**	**47**	**47**	**475**	**415**	**392**	**413**	**17**	**2,164**
Au	g/t	-	-	-	0.860	0.755	0.772	0.741	0.714	0.836	0.860	0.678	0.756	0.676	0.587	0.524	0.455	0.395	0.547	0.550
Cu	%	-	-	-	0.235	0.268	0.254	0.246	0.195	0.225	0.219	0.156	0.183	0.175	0.148	0.182	0.265	0.244	0.125	0.208
Ag	g/t	-	-	-	2.59	2.55	2.20	1.94	1.41	2.62	3.48	2.19	3.45	2.90	2.93	1.83	3.43	3.04	2.01	2.74
Mo	ppm	-	-	-	27.6	22.1	32.8	68.4	69.0	48.9	45.6	76.4	40.3	58.5	72.2	50.5	15.9	27.8	56.6	44.6
Metal to the Mill																				
Au	M oz	-	-	-	0.8	1.1	1.2	1.1	1.1	1.3	1.3	1.0	1.2	1.0	9.0	7.0	5.7	5.3	0.3	38.3
Cu	M lb	-	-	-	144	264	267	258	204	236	229	163	191	184	1,544	1,664	2,295	2,218	46	9,907
Ag	M oz	-	-	-	2.3	3.7	3.4	3.0	2.2	4.0	5.3	3.3	5.3	4.4	44.8	24.4	43.3	40.4	1.1	190.8
Mo	M lb	-	-	-	1.7	2.2	3.4	7.2	7.2	5.1	4.8	8.0	4.2	6.1	75.5	46.2	13.7	25.3	2.1	212.7
Total Waste Mined	**Mt**	**29**	**46**	**54**	**135**	**142**	**147**	**127**	**66**	**75**	**116**	**128**	**88**	**65**	**523**	**917**	**347**	**207**	**64**	**3,287**
Open Pit Strip Ratio (Waste Mined/ Plant Feed)	t/t	-	-	-	4.8	3.2	3.1	2.7	1.4	1.6	2.5	2.7	1.9	1.4	1.1	2.4	3.5	1.6		2.1

Note: Waste mined in the production schedule in Table 1.17 includes re-handled waste, and waste mined from borrow pit sources for construction purposes.

1.15 METALLURGICAL TEST REVIEW

The KSM Project includes four major mineralized zones, identified as the Mitchell, Kerr, Sulphurets and Iron Cap deposits. The deposits contain significant gold, copper, silver, and molybdenum mineralization.

Several wide-ranging metallurgical test programs were carried out between 2007 and early 2012, to assess the metallurgical responses of the mineral samples from the KSM deposits, especially the samples from the Mitchell deposit. Testwork conducted since early 2011 included: flotation locked cycle tests on the composite samples from Mitchell, Sulphurets, and Kerr deposits, and cyanide leach tests on the samples from the flotation products. The 2011 testing also assessed the resistance of the ore samples to grinding, in particular to semi-autogenous grinding (SAG) milling.

The metallurgical tests to date include:

- mineralogy, flotation, cyanidation and grindability testwork by G&T Metallurgical Services Ltd. (G&T) and SGS Minerals Services (SGS)
- SAG mill comminution (SMC) grindability tests to determine the grinding resistance of the mineralization to SAG/ball milling by Hazen Research Inc. (Hazen) and G&T
- crushing resistance parameters to high pressure grinding rolls (HPGR) crushing of the Mitchell and Sulphurets ore samples by SGS and pilot plant scale HPGR testing on the Mitchell ore sample by Köeppern Machinery Australia Pty Ltd.'s (Köeppern) HPGR pilot plant at the University of British Columbia
- dewatering tests by Pocock Industrial Inc. on the samples of heads, copper concentrates, sulphide leach products, and tailing pulps.

The flotation and cyanidation metallurgical testing established the optimum process-related parameters and investigated metallurgical variability responses and copper-molybdenum separation techniques. Flotation locked cycle tests were performed on the composite samples from the Mitchell, Sulphurets, Kerr, and Iron Cap deposits, particularly on a variety of samples from the Mitchell deposit.

The test results indicate that the mineral samples from the four separate mineralized deposits are amenable to the flotation-cyanidation combined process. The process consists of:

- copper-gold-molybdenum bulk rougher flotation followed by gold-bearing pyrite flotation
- regrinding the bulk rougher concentrate followed by three stages of cleaner flotation to produce a copper-gold-molybdenum bulk cleaner flotation concentrate

- molybdenum separation of the bulk cleaner flotation concentrate to produce a molybdenum concentrate and a copper/gold concentrate containing associated silver
- cyanide leaching of the gold-bearing pyrite flotation concentrate and the scavenger cleaner tailing to further recover gold and silver values as doré bullion.

The reagents used for flotation were 3418A (dithiophosphinates)/A208 (dithiophospate)/fuel oil for copper-gold-molybdenum bulk flotation and A208/potassium amyl xanthate (PAX) for gold-bearing pyrite flotation. The primary grind size used was 80% passing approximately 125 µm, and concentrate regrind size was 80% passing approximately 20 µm.

The samples from the Mitchell deposit produced better metallurgical results with the chosen flotation and cyanide leach extraction circuits when compared to the metallurgical results from the samples taken from the Sulphurets, Iron Cap, and Kerr deposits. The locked cycle tests showed that, on average, approximately 85% of the copper and 60% of the gold in the Mitchell samples, which contain 0.205% Cu and 0.72 g/t Au, were recovered into a concentrate containing 24.8% Cu. The cyanidation further recovered approximately 18% of the gold from the gold bearing products consisting of the cleaner flotation tailing and the gold bearing pyrite flotation concentrate.

1.16 MINERAL PROCESSING

The mill feed from the Mitchell, Sulphurets, Kerr, and Iron Cap deposits will be processed at an average rate of 130,000 t/d. The Mitchell deposit will be the dominant source of mill feed for the process plant, and will be processed through the entire mine life, excluding Years 24 and 25. Ore from the Sulphurets deposit will be processed between Years 2 to 6 and Years 23 and 30. Ore from the Kerr deposit will be processed between Years 27 and 50, while Iron Cap ore will be fed to the process plant between Years 32 and 51.

The proposed flotation process is projected to produce a copper-gold concentrate containing approximately 25% copper. This concentrate should recover between 76% and 88% of the copper, and between 50% and 62% of the gold from the mill feed. Copper and gold flotation recoveries will vary with changes in head grade and mineralogy. For the LOM mill feed containing 0.549 g/t Au and 0.207% Cu, the average copper and gold recoveries to the concentrate are projected to be 81.7% and 53.9%, respectively. As projected from the testwork, the cyanidation circuit (carbon-in-leach) will increase the overall gold recovery to a range of 70% to 79%, averaging 73.2% for the LOM, depending on gold and copper head grades. Silver recovery from the flotation and leaching circuits is expected to be 63% on average. A separate flotation circuit will recover molybdenite from the copper-gold-molybdenum bulk concentrate when higher-grade molybdenite mineralization is processed.

The process plant will consist of three separate facilities: an ore primary crushing and handling facility at the mine site, a 23 km Mitchell-Treaty ore conveyance and transportation tunnel system, and a main process facility at the Treaty plant site, adjacent to the TMF. The processing circuit at the Treaty site will include secondary crushing by cone crushers and tertiary crushing by HPGR, primary grinding by ball mills, flotation, regrinding, leaching, and concentrate dewatering.

Detailed trade-off studies were conducted to economically and technically compare the following comminution options:

- cone/HPGR/ball mills comminution circuit and SAG/ball mills/pebble crushing (SABC) circuit
- location of the comminution circuits – at Mitchell site or Treaty site.

The studies, including comparisons of capital costs and operating costs (energy consumption and metal consumables), indicated that the cone/HPGR/ball mills comminution circuit is economically favourable when compared to the conventional SABC circuit. The study also showed that locating the comminution circuit at Treaty is more feasible than at the Mitchell site.

The comminution plant at the Mitchell mine site will reduce the mill feed from 80% passing 1,200 mm to 80% passing 150 mm by gyratory crushers. The crushed ore will be stockpiled at the Mitchell site prior to being conveyed to the Treaty site, located near the TMF, northeast of the Mitchell mine site.

A 23-km Mitchell-Treaty twin tunnel (MTT) system has been designed to connect the Mitchell and Treaty sites. The crushed ore will be transported through one of the twin tunnels by conveyance. This tunnel will also be used for electrical power transmission and diesel fuel delivery by pipeline. The adjacent tunnel will be used for the transport of personnel and supplies for mine operating and water management activities.

The process plant at the Treaty site will consist of secondary and tertiary crushing, primary grinding, flotation, concentrate regrinding, concentrate dewatering, cyanide leaching, gold recovery, and tailing delivery systems. The crushed ore conveyed from the Mitchell site will be sent to a 60,000-t stockpile adjacent to the tunnel portal. The ore will then be reclaimed and crushed by cone crushers, followed by HPGR.

The ores from the HPGR comminution circuits will be ground to a product size of 80% passing 150 µm by four conventional ball mills in closed circuit with hydrocyclones. The ground ore will then have copper/gold/molybdenum minerals concentrated by conventional flotation to produce a copper-gold-molybdenum concentrate and gold-bearing pyrite products for gold leaching. Depending on molybdenum content in the copper-gold-molybdenum concentrate, the concentrate may be further treated to produce a copper-gold concentrate and a molybdenum concentrate. The molybdenum concentrate will be leached to reduce copper content. The concentrates will be dewatered and shipped to copper and molybdenum smelters.

The gold-bearing pyrite products which consist of the bulk cleaner flotation tailing from the copper-gold-molybdenum cleaner flotation circuit and the gold-bearing pyrite concentrate will be leached with cyanide (CIL) for additional gold and silver recovery. Prior to storage in the lined pond within the TMF, the leach residues from the cyanide leaching circuits will be washed, and subjected to cyanide recovery and destruction. The water from the residue storage pond will be recycled back to the cyanide leach circuit. Any excessive water will be further treated prior to being used as process water for the flotation circuit or sent to the flotation tailing storage pond.

The flotation tailing and the washed leach residues will be sent to the TMF for storage in separate tailing areas. Two water reclaim systems for the flotation tailing pond and the CIL residue pond have been designed to separately reclaim the water from the TMF.

1.17 Tailing, Water Management, and Rock Storage Facilities

1.17.1 Tailing Management Facility

The TMF will be constructed in three cells: the North and South cells for flotation tailing, and a lined cell for CIL tailing. The cells are confined between four dams (North, Splitter, Saddle, and Southeast dams) located within the Teigen-Treaty Creek cross-valley. Design criteria for the dams are based on the Canadian Dam Association Guidelines. The area is moderately seismic and the dams are designed to resist earthquake loads. A site-specific seismic hazard assessment indicates peak ground acceleration at 10,000-year return period of 0.14 *g*. The TMF cells are designed to store the 30-day probable maximum flood (PMF) with snowmelt.

De-pyritized flotation tailing will be stored in the North and South cells. The pyrite bearing CIL (carbon-in-leach) tailing will be stored in a lined central cell. In total, the TMF will have a capacity of 2.3 Bt, which is greater than the 2.19 Bt required for the 55-year mine life.

The North and CIL cells will be constructed and operated first; they will store tailing produced in the first 25 years. The North Cell will then be reclaimed while the CIL and South cells are in operation.

The North, Splitter, and Saddle earth-fill starter dams will be constructed over a two-year period in advance of the start of milling to form the North and CIL cells and will provide start-up tailing storage for two years. Cyclone sand dams will be progressively raised above the starter dams over the operating life of the mine. The North starter dam will be constructed with a low-permeability glacial till core and raised with compacted cyclone sand shells, using the centerline geometry method. The Splitter and Saddle starter dams form the CIL pond. These dams are also subsequently raised with cyclone sand shells but the CIL pond and the Splitter and Saddle dams incorporate HDPE and LLDPE liners in the core and basin floor in order to surround the CIL tailing within a completely lined impoundment.

Cyclone sand dam raises will be constructed from April through October each year, starting with the North Cell. To reach the capacity of 2.3 Bt, an ultimate dam crest elevation of 1068 m will be required for the North Cell dams and 1068 m for the South Cell. This will require a dam height of up to 240 m for the Southeast dam, which is the highest dam of the TMF.

Process water collected in the flotation and CIL tailing cells will be reclaimed by floating pump barges and recycled separately to the plant either for use in the process, for treatment, or to be discharged. Diversions will be constructed to route non-contact runoff from the surrounding valley slopes around the TMF. The diversion channels are sized to allow passage of 200-year peak flows, and are large enough to allow space for passage of snow removal machinery. Buried pipe sections paralleling the channels will be installed in areas of active snow avalanche paths to enhance diversion operability during avalanche periods.

During operation of the North Cell, flood waters will be routed south to Treaty Creek. A short tunnel section in the East Catchment Valley will convey environmental maintenance flows from the East Catchment past an area of potential slope instability in the valley into a pipeline supplying flow to Teigen Creek. As operations switch to the South Cell, the East Catchment Tunnel will be extended to the north to route the entire East Catchment flow around the North Cell towards Teigen Creek and away from the South Cell.

Seepage from the impoundment will be controlled with low permeability zones in the dams and foundation treatment. Residual seepage and runoff water from each tailing dam will be collected at small downstream collection dams provided with grouted foundations and low permeability cores. Seepage collected will be pumped back to the TMF. The seepage dam ponds will also be used to settle solids transported by runoff from the dam and to collect cyclone sand drain-down water.

Based on site data taken between 2007 and 2011, combined with regional long term records, water balance calculations indicate that the TMF North and South flotation cells will have average surpluses of water of 0.14 m^3/s to 0.20 m^3/s during their operating periods. During the five-year transition period between the North and South cells, the total excess flow from the flotation cells is projected to be up to twice this amount as both the North and South cells will be active while the North Cell is being closed. During the life of the mine, excess water from the CIL cell varies from 0.23 m^3/s to 0.10 m^3/s. Management of surplus water during operations will use a combination of storage, discharge to Treaty Creek during freshet if water quality meets standards, or treatment at the Treaty process plant water treatment facility (if required) and discharge.

1.17.2 Mine Area Water Management

Two diversion tunnel routes (each with twinned tunnels, for a total of four tunnels) will be required to route glacial melt water and non-contact valley runoff from the Mitchell and McTagg valleys around the mine area.

The open pit phase of the Mitchell Diversion Tunnels (MDT) and the McTagg Diversion Tunnels (MTDT) are sized to convey 24-h average flows from a 200-year storm. When the Mitchell block caving operation commences, an additional twinned MDT paralleling the first phase tunnels will be driven to protect the underground workings, which are more sensitive to inflows than the open pits. The underground phase of the MDTs will provide increased capacity to convey the 1,000-year storm peak flow.

The second tunnel of each set of twinned tunnels provides redundancy against blockage as each individual tunnel can carry typical freshet flows. The provision of twin tunnels also allows of switching base flows between adjacent tunnels if access for maintenance is required.

Each of the open pit phase MDTs will have a cross-sectional area of 17.7 m^2 and a length of 5.6 km. These tunnels will route water from Mitchell Creek/Mitchell Glacier to the Sulphurets Valley, away from the open pit, primary crushing facility, open pit area, and Mitchell RSF. The Mitchell Diversion will collect melt water from beneath the base and toe of the Mitchell Glacier via separate surface and sub-glacial inlet structures, which improves redundancy. Both surface and subglacial inlets are designed to protect the inlet of the diversion from being blocked by snow avalanches. The Mitchell Diversion will generate hydroelectric power as Sulphurets Valley is lower than Mitchell Valley. In Year 26, the MDTs will be augmented with a second set of more steeply sloping, larger cross section (31 m^2) tunnels to provide protection against the 1,000-year storm flow to the underground workings.

Each MTDT will have a cross sectional area of 13.4 m^2, an initial length of 4 km, and an ultimate length of approximately 7.5 km. The two inlet branches of the ultimate tunnel will collect flows from east and west McTagg valleys and feed into the main diversion tunnel route, around the west side of the McTagg RSF, and discharge into Sulphurets Valley.

The Stage 1 inlet to the MTDT will initially be established in lower McTagg Creek, upstream of the Mitchell RSF. As the mine life progresses, Stage 2 and Stage 3 inlets, with ramped energy dissipating tunnel sections, will be constructed at higher levels further up McTagg Creek and into each branch of the McTagg Valley to divert melt water into the diversion tunnel as the RSF is raised in elevation. The staged inlets will avoid or minimize glacier ice removal. Hydropower will be generated by the McTagg Diversion only in Stage 2 and Stage 3 when the available head increases as the tunnel inlets are raised.

Lined surface diversion channels will be constructed progressively during operations, along the contact of the RSF and the hillside, to divert surface flows. Flood runoff flows of greater than the 200-year event will be routed alongside the lined channels. An in-rock spillway will be constructed at the southwest corner of the McTagg dump to convey surface diversion flows down to diversion pipelines and channels on the west and east sides of the Water Storage Facility (WSF) pond. Flood flows in excess of the 200-year event capacity of the MTDT will be routed into the RSF by spillways at the inlets and then passed over the spillway of the WSF dam.

All contact water from the mine areas (open pits, RSFs, roads, infrastructure) will be directed to the WSF, located in the lower Mitchell Creek area. The facility will be formed with an initially 156 m-high earth/rock fill dam. The facility is sized to store annual freshet flows and volumes resulting from a 200-year wet year. The dam is founded on competent sedimentary rock foundations. Seepage will be controlled with asphalt core zones in the dam and the dam foundation will be grouted. A seepage collection pond will return seepage water to the facility. Snow avalanche hazards have been assessed for the area and the wave modelled from the maximum predicted avalanche in the area will be contained within the design freeboard for the dam. After Year 10, the dam will be raised by 10 m to create sufficient storage for the increased catchment area of the RSFs and open pits.

During operations, secondary diversion ditches and pipelines will be implemented within the mine area to reduce contact water volumes. Open pit contact water and discharge from pit dewatering wells will gravity-flow from the pit rims, via ditches or direct drainage, and via pipelines to the WSF.

Mine area contact water will be treated with a high density sludge lime WTP. Water balance calculations, based on data taken between 2007 and 2011 combined with regional long term records, indicate that during the various stages of mine life the treatment plant will operate year-round at a constant rate of 1.3 m^3/s to 2.2 m^3/s. The WTP will also have additional capacity in the form of a spare clarifier and reactors provided to treat up to 3.3 m^3/s to manage flow increases that may occur due to natural hazards or extreme events. The additional treatment capacity also allows sections of the WTP to be shut down for maintenance when required.

During pre-production operation of the WTP, sludge from the WTP will be filter-pressed and stored in a shed during winter and trucked to a nearby engineered landfill during summer months.

During operations, sludge from the WTP will be filter-pressed and trucked to the MTT. The sludge will be added to the ore conveyor at the tunnel and passed through the ore milling process to add necessary alkalinity to the process and be ultimately disposed of in the tailing pond.

Additional hydropower will be generated in an energy recovery facility from the flow of treatment water from the WSF to the WTP, which is located at a lower elevation in the Sulphurets Valley.

During the initial construction period, to maintain existing water quality, four temporary water treatment facilities located in the mine area will manage water discharge from tunnel portals and from temporary stockpiles of tunnel muck near the portals, as well as treating water from existing seeps and mineralized areas. These facilities will include reactor tanks, agitators, semi-automated lime and polymer flocculent dosing systems, mixing launders, and settling ponds. The treatment will reduce suspended solids and dissolved metals. As well, across the entire KSM site, 16 automated flocculent treatment systems, located below earthworks and at the portals of the tunnels, will be constructed to treat total suspended sediments (TSS) during the construction period. These treatment systems will include engineered sediment ponds. Any potentially acid generating (PAG) tunnel muck will be stored on lined pads located at the temporary water treatment sites and will be hauled to permanent disposal sites within the RSF or TMF once the diversion tunnels and the main WTP are operational.

The main WTP and WSF will be operational before mill start-up to allow pre-production activity in the Mitchell Valley and Mitchell pit area.

1.17.3 *Mine Area Closure Plan*

The RSFs will be placed as close to their final closure configuration as possible; however, some re-contouring will be required at closure. The RSFs will be primarily built bottom up to result in stable dump slopes during operation and closure. Below the treeline, the RSFs will have a soil cover applied to promote plant growth and the area will be re-vegetated where required.

During closure, the diversion channels on the west and northern sides of McTagg RSF surface will be upgraded to closure channels capable of conveying the PMF.

The hydroelectric tunnels will continue to operate after closure, to generate power and act as diversion structures. At the end of operations, the Mitchell and McTagg RSFs will be contoured so that surface closure channels are present to route clean water flows around the RSF in the event of failure of the diversion tunnels or to handle extreme flood events.

An 80 m-high closure dam will be included within the Mitchell RSF at the western side of the Mitchell pit to enhance flooding of PAG rock exposed in the pit and collapsed block cave workings. The dam raises water levels and allows closure storm floods to flow from the pit lake into a closure channel constructed around the Mitchell RSF. The channel routes closure flows that pass over the pit lake around the Mitchell RSF, in the event that the Mitchell diversion hydroelectric tunnel is no longer operational, or if flows occur in excess of the 1,000-year peak capacity of the MDTs.

The MTT, consisting of twinned road and utility tunnels between the Mitchell site and the Treaty site, will remain after mine closure and continue to provide alternate site access on the roadway side. The power cables in the utility tunnel will also remain and convey power generated by the hydroelectric plants to market via the grid.

If any of the tunnels are no longer required at site, the tunnels will be sealed with engineered plugs to flood the tunnels.

1.17.4 TMF Closure Plan

The TMF North and South cells will be closed primarily as dry covered surfaces, each with a small internal pond area and gravel-lined beaches and closure channels. A closure cover of stockpiled soil will be placed on the surfaces of the flotation tailing cells to promote re-vegetation.

At closure, the CIL tailing will remain saturated to mitigate oxidation. The CIL tailing beaches will be dredged into the pond centre, and a layer of flotation tailing will be placed to form beaches and cover the CIL tailing and prevent re-suspension. The final CIL cell surface area will consist of reclaimed and revegetated beaches and a pond area to allow routing of surface runoff.

Surface water flows and flood flows from the reclaimed TMF will be routed from the closed cells via closure channels between cells into a spillway channel excavated in bedrock around the west abutment of the Southeast Dam. An auxiliary closure spillway around the North Dam is being considered as well in order to return post closure drainage patterns to as close as possible to pre-mining conditions.

The cyclone dams and beach surfaces are net non-acid-generating due to the removal of sulphides. The dam faces will be first covered with an erosion protection layer of rock fill quarried from the closure channel cuts. The dam surfaces will then be covered with till and re-vegetated using soil stockpiled during construction of the dams.

1.18 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. In particular, the project will require an amendment to Schedule 2 of the federal *Metal Mining Effluent Regulation* (MMER) in order to construct the TMF in an area occupied by fish.

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. The BCEAO confirmed in April of 2008 that the KSM Project will require an Environmental Assessment. On November 6, 2009, the BCEAO issued a

Section 11 Order to establish the scope, procedures, and methods for the Environmental Assessment. The Canadian Environmental Assessment Agency (CEA Agency) formally advised Seabridge on July 23, 2009, that the KSM Project will undergo a Comprehensive Study as defined by the CEAA.

Four years of on-site baseline environmental work has been completed by Rescan between 2008 and 2011. Baseline work included comprehensive surveys of meteorology, air quality, hydrology, hydrogeology, geochemistry, water quality, fish and aquatic ecology, soils, wetlands, vegetation, wildlife, archaeology, regional social and economic status, land use, and Aboriginal knowledge. The environmental information collected has been considered in the design of the project to avoid, minimize, or mitigate potential adverse environmental effects.

Rescan is also leading the preparation of the Environmental Assessment and the submissions required to acquire operating permits. Seabridge and its team are involved with engagement meetings with local communities, regulatory agencies, regional and municipal governments, the Nisga'a Nation, and relevant First Nations identified by the BCEAO to advance the proposed project through the review processes.

1.19 GEOHAZARDS

A geohazard and risk assessment was completed for the proposed facilities within the KSM project area. As expected for a mountainous, high-relief project site, snow avalanche and landslide hazards exist, with the potential to affect mine construction, operations, and closure.

Geohazard scenarios were identified for the project facilities considered. Using unmitigated geohazard levels as a baseline, these scenarios were assessed in terms of risk to human safety, economic loss, environmental loss, and reputation loss. Geohazard risk levels were assigned to each scenario with ratings ranging from Very Low to Very High.

Mitigation strategies have been identified to reduce the High and Very High Risk scenarios to a target residual risk not exceeding Moderate. Further risk reduction will be achieved where practical and cost-efficient.

1.20 ON-SITE INFRASTRUCTURE

1.20.1 PERMANENT ACCESS ROADS

There will be two primary permanent access roads to the mine and plant site.

The Coulter Creek Access Road will be primarily a single-lane, radio-controlled road constructed for moving large equipment and supplies to the mine site. An existing road leaves Highway 37, south of Bob Quinn, and extends approximately 59 km southwest to the former Eskay Creek Mine. The first 37 km of this road is classified as public road but is subject to controlled and shared access. The remaining 22 km of existing road length is private and subject to a shared access agreement. Upgrades to sections of the existing road will be required.

The new 35 km-long Coulter Creek Access Road will commence near the former Eskay Creek Mine and follow the west side of the valley south for approximately 21 km before crossing the Unuk River. It then turns east through a series of switchbacks and follows the north side of the Sulphurets Creek valley to the Mitchell Creek valley and mine site. Consideration has been given to active and passive snow avalanche control measures through the Sulphurets Creek valley.

The Treaty Creek Access Road will consist of a two-lane road, constructed to provide permanent access from Highway 37 to the plant site and east portal of the MTT. This road will leave Highway 37 approximately 19 km south of Bell II, cross the Bell-Irving River, and follow the north side of the Treaty Creek valley for approximately 18 km. It will then turn north and follow the west side of the North Treaty Creek/Teigen Creek valley for approximately 12 km to the plant site, TMF, and east portal of the MTT. Initially the North Treaty Lower Road will be built low in the valley to facilitate access for construction of the north tailing dam, and provide reduced road grades and access road length during the first half of mine life.

Additional roads will also be required at mine start-up to facilitate maintenance access and construction of the proposed uphill cut-off drainage ditch. Later, once construction of the south tailing dam starts, the remaining 5.7 km of the North Treaty Upper Road will need to be constructed. These roads will be used to transport supplies, equipment, and crew members to and from the plant site, and to transport concentrate to Highway 37 during the life of the mine.

Leaving the Treaty Creek Access Road at approximately kilometre 18 and heading west, there will be a 15 km-long single-lane, radio-controlled road providing access to the MTT saddle construction access portals. It will be used for construction and will be maintained for service access. Near the end of this road, there is a temporary spur road extending approximately 3 km further to the west. This road will provide access to an additional adit entry point required for construction of the MTT.

1.20.2 Winter Access Road

A Winter Access Road will be constructed to access the KSM mine site. The route will begin at the end of an existing all-season road near the abandoned Granduc Mine. The Winter Access Road will start at the toe of the Berendon Glacier, accessing the Frank Mackie Glacier from the Berendon Glacier, and then up and over the glacier into the Ted Morris Creek valley, which is a tributary of Sulphurets Creek. The Winter Access Road will be used to mobilize water treatment supplies and mobile equipment and supplies for construction of access roads and water diversions during the first season. The Winter Access Road will be used during the next two winter seasons as well, until the Coulter Creek Access Road has been completed. The equipment and supplies will enable roads and water diversions to be built for sediment control and water treatment, and to initiate major water diversion and other tunnel construction and pioneering work in the Mitchell valley. It will also allow access for the construction of portions of the Coulter Creek Access Road, near its east end and to the Mitchell mine site area.

1.20.3 Other On-site Infrastructure

The locations of the ore processing facilities at the Mitchell site were selected to take advantage of the natural topography and, to the extent possible, minimize the impact on the environment and geohazard risks.

The ore will be trucked to one of two primary crushers near the Mitchell pit and conveyed to a covered stockpile. Ore will be reclaimed from the stockpile by tunnel feeders and fed on to a tunnel conveyor of approximately 23 km in length, terminating at the Treaty plant site. This tunnel, one of two parallel tunnels (MTT), will extend from the north side of the Mitchell Zone to the northeast into the upper reaches of the Teigen Creek valley, north of the TMF. As described in Section 1.20.1, there is a saddle area, approximately 17 km from the Mitchell portal, where the MTT will be accessed for construction purposes and then enclosed after construction is completed.

One of the MTTs will be used for the conveyor, a water pipeline, a diesel fuel pipeline, and electrical power transmission cables. The other MTT will be a transportation tunnel to provide access for maintenance services to the conveyor tunnel. It will also serve to deliver bulk supplies and move personnel to/from the Mitchell valley mine areas. The proposed tunnel route is through Crown land and approximately 15 km of its length passes through ground subject to mineral claims held by third parties. The overall site plan can be seen on Figure 1.2.

Figure 1.3 shows the primary crushing and crushed ore handling arrangement. Figure 1.4 shows the conveyors, plant access roads, fuel storage, main process buildings, electrical buildings, and haul routes from the Mitchell pit.

The Sulphurets pit will supply ore during Years 2 to 6 and Years 23 to 30. Kerr will then provide ore for blending with Mitchell ore, starting in Year 27 and continuing until Year 50. The ore from the Kerr and Sulphurets pits will be crushed at the respective pits, except for the Sulphurets ore which will be hauled to and crushed at the Mitchell site during Years 2 and 6. During Years 23 to 50, ore will be transported by an overland conveyor and rope conveyor system starting at the Kerr pit.

The ore transport system will include:

- The overland conveyor through the Sulphurets-Mitchell Conveyor Tunnel (SMCT) and a connecting conveyor will transport ore from the Sulphurets pit to an ore stockpile at the Mitchell site. Initial waste from the Sulphurets pit is placed in the Sulphurets RSF and then re-handled at a later time to the Mitchell RSF.

- A separate rope conveyor will be built to connect the Kerr pit to the SMCT conveyor to deliver ore and waste rock material from the Kerr pit to the Mitchell pit site. Both the ore and waste rock that are primarily crushed at the Kerr site will use the same transport system. The waste rock will be diverted to the RSF in the McTagg Valley via conveyors.

The stockpiled ore from the Sulphurets and Kerr pits at the Mitchell pit site will be reclaimed and trucked to the Mitchell crushing/conveying system where ore from the Sulphurets or Kerr pits will be blended with ore from the Mitchell pit.

Figure 1.4 shows the Mitchell mine area and the various other pits. Figure 1.5 shows the lower Mitchell site area including additional infrastructure such as the initial staging, construction and permanent camps, explosive facilities, the WSD, diversion tunnels, and hydro power plants. Access and appropriate haul roads will be provided to all of these areas.

The main processing facilities at the Treaty site consist of stockpiling, secondary and HPGR tertiary crushing, a flotation plant, a cyanide leaching plant, the TMF, a construction and permanent accommodation complex, as well as maintenance and support facilities (Figure 1.6).

Figure 1.3 Processing Facilities at Mitchell

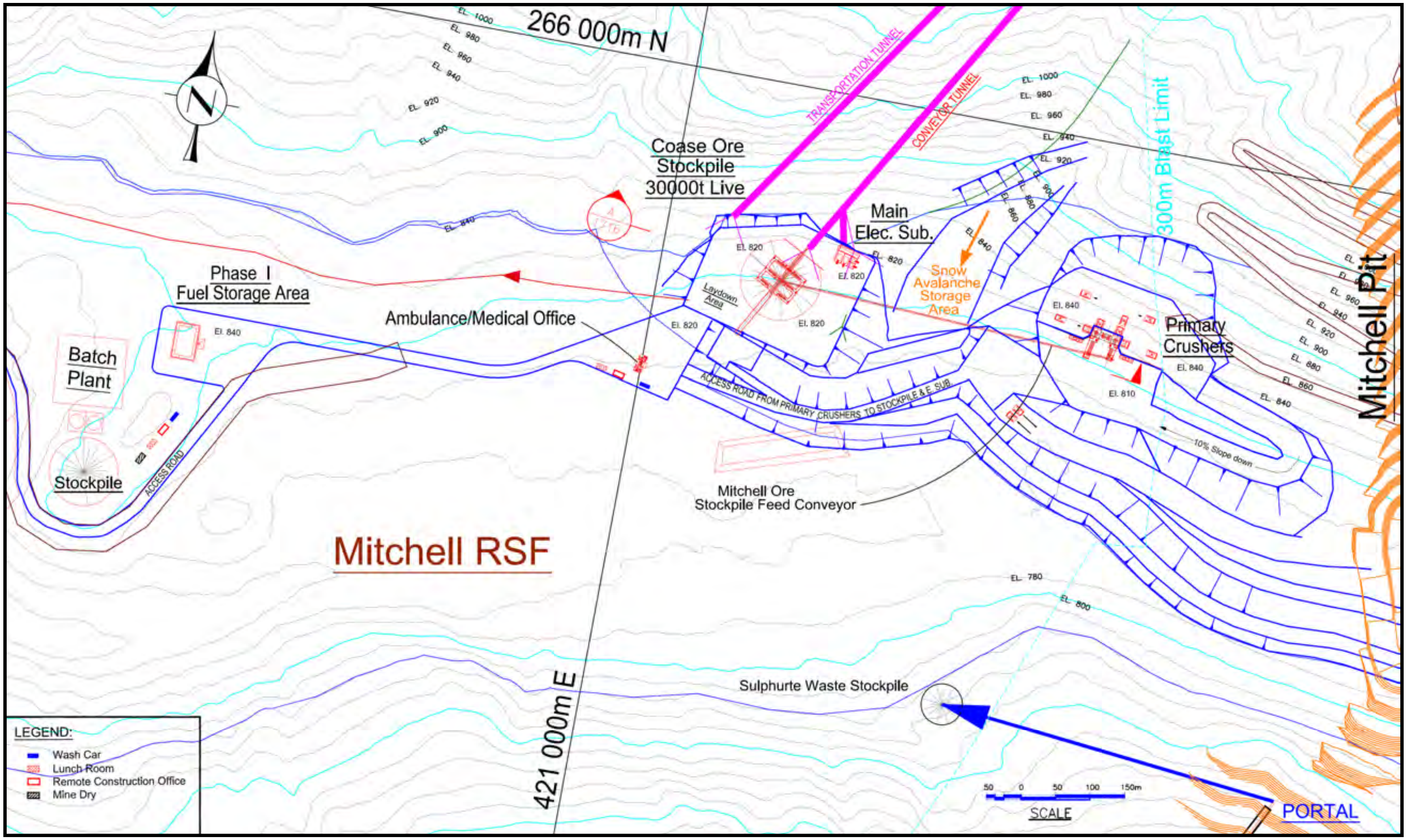


Figure 1.4 Mitchell Area Site Plan

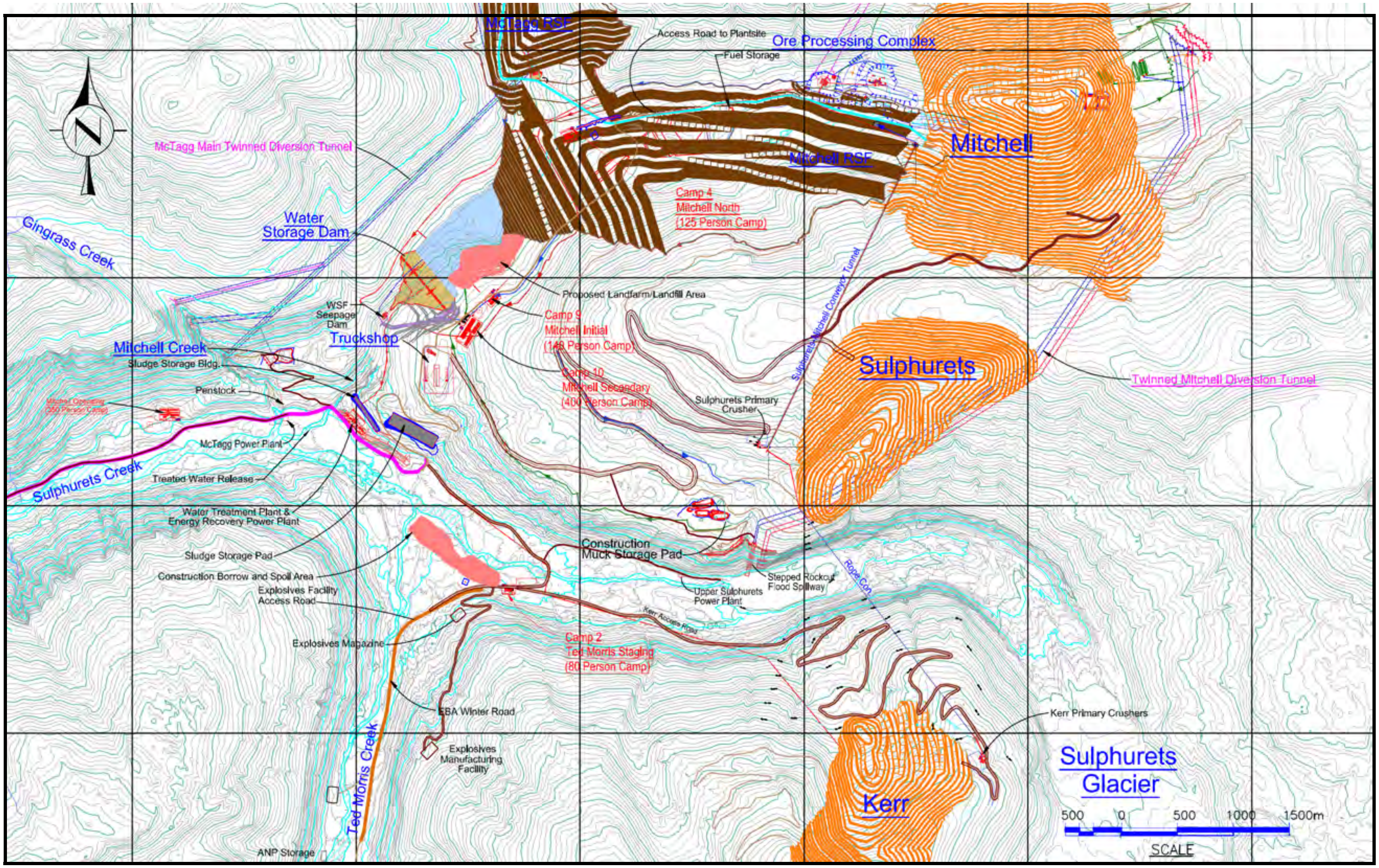


Figure 1.5 Lower Mitchell Area

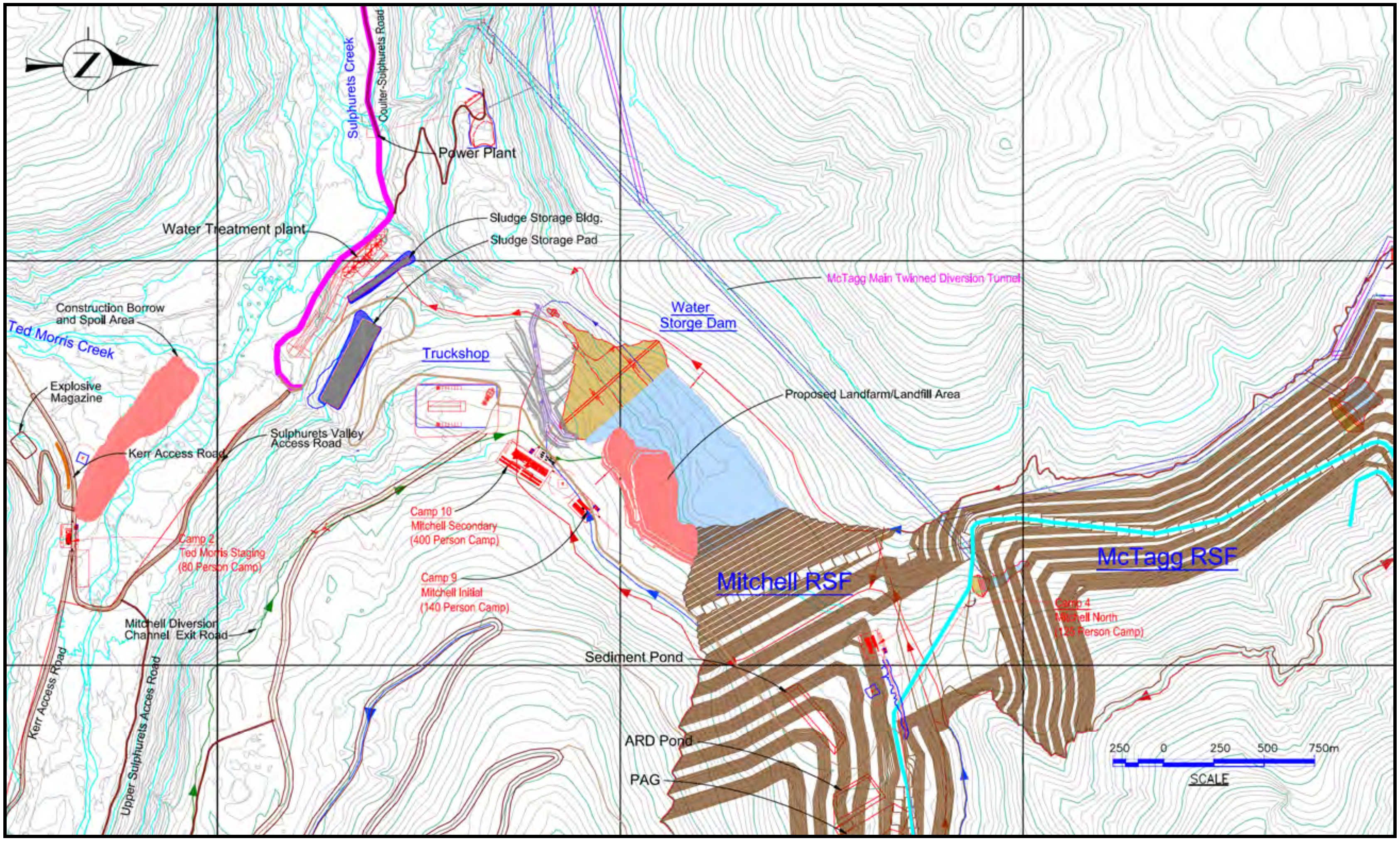
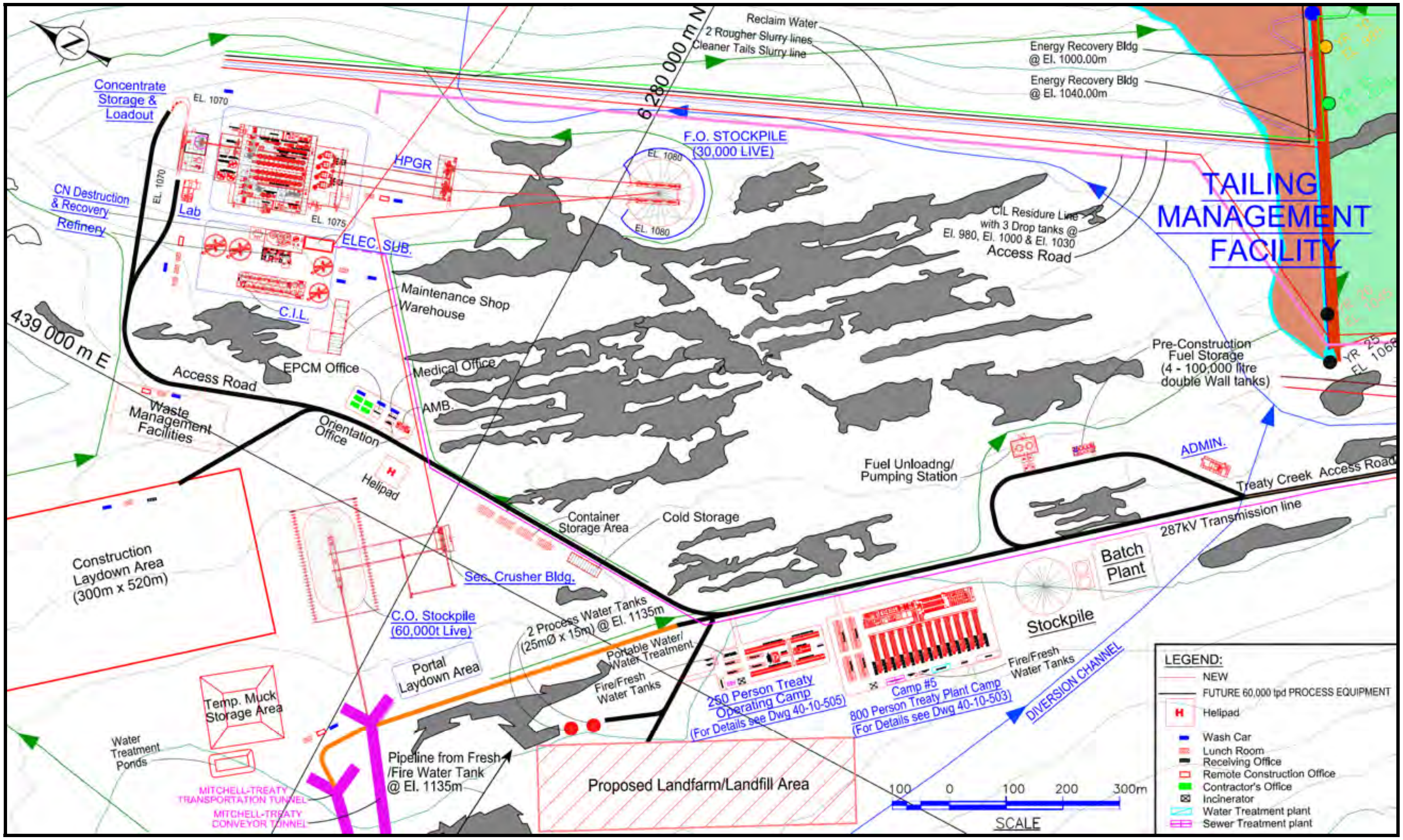


Figure 1.6 Plant Site at the TMF



1.21 OFF-SITE INFRASTRUCTURE

Copper concentrates produced at the process site will be filtered at the plant site and transported by contract trucking firms on Highway 37 and 37A to a storage and concentrate loading facility site in Stewart, BC. Copper concentrates will be loaded and shipped via ocean transport to overseas smelters.

Molybdenum concentrates produced at the process site will be loaded in bags and transported by contract trucking firms on Highways 37 and 16 to Prince Rupert, BC. Molybdenum concentrates will be loaded into containers and shipped via ocean transport to Asia.

The project will utilize a staging area located at Smithers, Terrace, or Stewart to receive and deliver equipment and supplies to the site during construction and operation of the KSM Project.

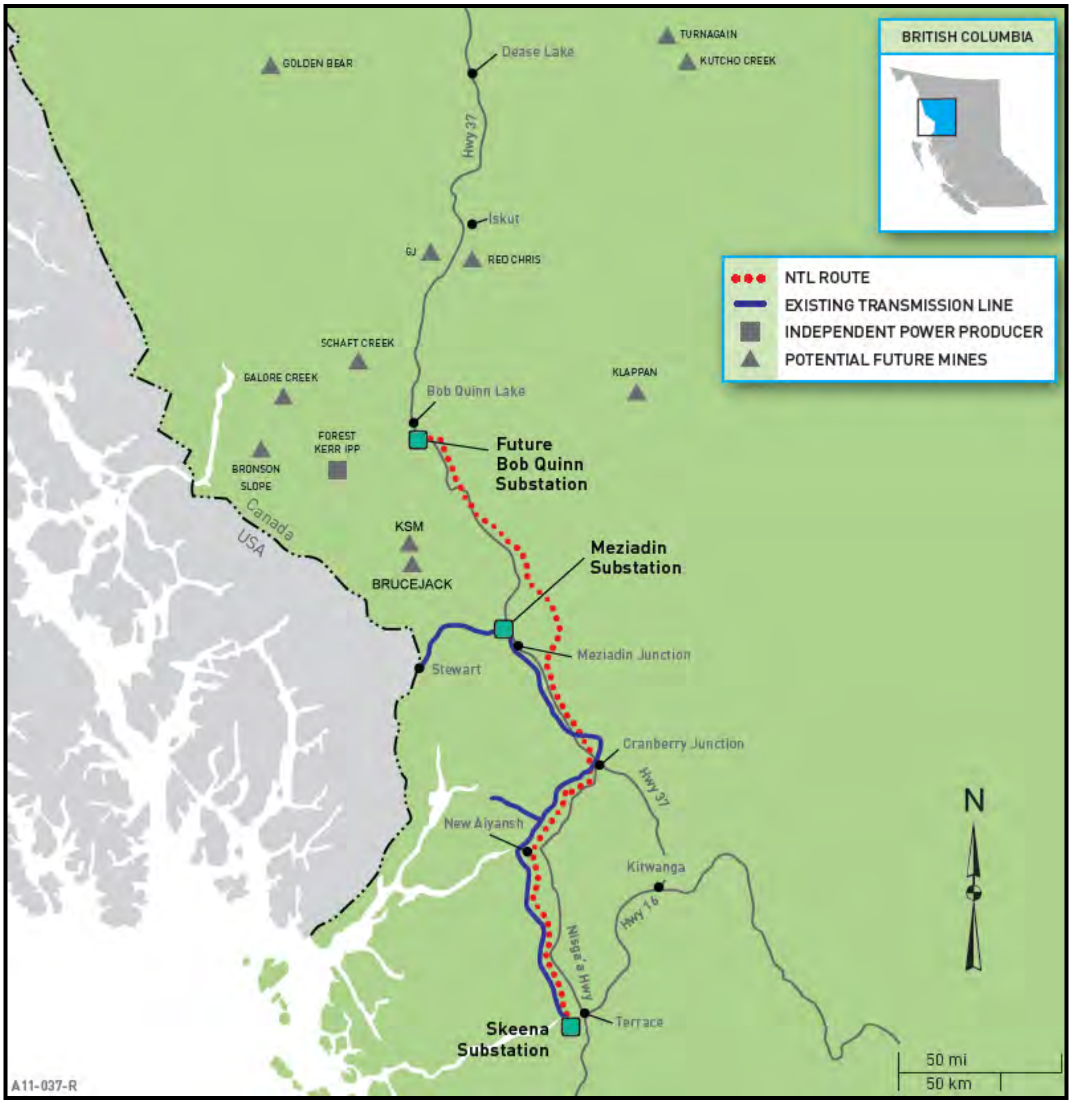
1.22 POWER SUPPLY AND DISTRIBUTION

BC Hydro is the electric utility that serves the project area. Electric service for the KSM Project will be from BC Hydro's Northwest Transmission Line (known as the NTL) that is currently under construction with a scheduled completion in the spring of 2014.

The new 344 km long, 287 kV NTL transmission line will run from the Skeena Substation on the BC Hydro 500 kV grid near Terrace, BC, to Cranberry Junction from which point it will roughly parallel BC Highway 37 to its terminus at Bob Quinn. The KSM Project will construct a 28.5 km long, 287 kV transmission extension from the NTL, originating at the Treaty Creek switching station and terminating at the Treaty Plant Site No.1 Substation. This line will parallel the KSM access road so that a separate right-of-way is not required. The Treaty Creek Switching Station on the NTL will be approximately 20 kilometres south of Bell II. Figure 1.7 is a map from BC Hydro illustrating the routing of the NTL.

The KSM Project will take electrical service from the new NTL as a Transmission Service Customer under Schedule 1823 as published in the BC Hydro tariffs.

Figure 1.7 Map of the Northwest Transmission Line



Source: BC Hydro.

Seabridge has commissioned BC Hydro to carry out a Facilities Study for the KSM Project, following the previously completed System Impact Study. The Facilities Study, which is currently being updated, is the final evaluation required by the utility to define costs and terms of electric service. Upon completion of the Facilities Study, the parties will then be in a position to sign a Facilities Agreement, which forms part of the contract for the supply of electric power for a large bulk Transmission Service Customer such as Seabridge. The updated Facilities Study is expected to be completed in 2012. Seabridge was the first mining company to commission a Facilities Study for power supply from the NTL and the KSM Project therefore has priority for service from this new transmission line.

Service to the Mitchell mine and mill site will be provided from KSM Substation No. 1 via a 138 kV cable (24 km in length, including lead-in to the portals) through the conveyor tunnel connecting the two operation sites. This supply will terminate at the 138 to 25 kV step-down Substation No. 2 at the Mitchell primary crushing and stockpile area. The substation will be a gas insulated substation (GIS), which allows for a very compact indoor installation in a concrete building, built into the mountainside to protect against avalanches and will have protected access by being connected directly to the conveyor tunnel that carries the main power cables.

There will be 25 kV cables from Substation No. 2 feeding the primary crushing and conveying facilities. In addition, 25 kV cables will also feed half of the drives of the main Mitchell to Treaty conveyor. Overhead lines fed from the substation will supply the Mitchell pit and other facilities including the WTP, Mitchell hydro plant, truck shop, camp, explosives plant, and other installations.

1.22.1 MINI HYDRO PLANTS AND ENERGY RECOVERY

Several energy recovery and mini-hydro plants have been included in the project development plan. These plants generate electric power by making use of facilities already included in the project and will result in significant net project energy savings. The plants will all be located within the mining lease area. The total annual energy generation is estimated to be 48,706,000 kWh, excluding the proposed future McTagg installation. All of the plants, similar to small independent power producer (IPP) hydroelectric plants, will operate unattended and automatically controlled by PLC systems. The power will be fed into the local mine distribution power lines. The plants will either displace costly Tier 2 utility power, or will be sold back to BC Hydro under their Standing Offer Program. The per-kilowatt-hour value of the generated electricity will therefore be relatively high.

This section provides a brief summary of the generation plants.

WATER TREATMENT PLANT ENERGY RECOVERY

This generation uses the water running downhill from the water storage pond to the WTP to generate electric power. A small impulse Turgo type turbine will be used. The output will be fed into the plant power distribution system at the WTP. This facility will continue to operate after mine closure.

TAILING ENERGY RECOVERY

The energy recovery pump turbines will be located on only one of the two tailing lines, based on available elevation differences, and will consist of two slurry pumps in series in two stations running in reverse as turbines. Induction generators will feed power back into the local plant electrical distribution system. The two energy recovery stations will be installed in series, located in small buildings at elevations 1030 m and 1000 m.

MITCHELL DIVERSION HYDRO

This plant will make use of the normal (but not flood) stream flows that will be diverted around the mining operations by the MDT. The installation will consist of a Pelton turbine, and will be very similar to IPP run-of-river hydro plants, as it makes use of the flow as it naturally occurs, with no water storage facilities or any other works other than what's required for water diversion around the mine. The equipment will be housed in a small powerhouse building near Sulphurets Creek. Power will be delivered to the open pit electrical distribution system. This plant will continue to operate after mine closure.

MCTAGG DIVERSION HYDRO

This plant will be very similar to the Mitchell diversion scheme. The McTagg plant will be constructed in Year 10 once the diversion tunnel inlets are raised in Phase 2. It will consist of two Pelton turbines, and will feed power into the plant distribution system at the WTP. This facility will continue to operate in Phase 3 and after mine closure.

1.23 PROJECT EXECUTION PLAN

1.23.1 ENGINEERING PROGRAM

The KSM Project will mobilize the engineering program in two phases. The first phase will be to address the early works road access, water management, site roads, diversion and access tunnels, and WSD construction. This will include tendering the construction packages for the early scope and procurement of the long lead items to maintain the schedule. The second phase will include the detailed design, major equipment procurement, and tendering process for the remaining packages.

For the purposes of costing this study, it was assumed that the project would be constructed using the engineering, procurement, and construction management (EPCM) approach with a management team located at both the Mitchell site and the plant site. The Owner will supply all the temporary construction camps and service contractors to manage daily activities on site. It is proposed that marshalling yards would be established early at Stewart, Smithers, Terrace, and at the Treaty Creek Access Road to ensure proper control of the movement of freight during the construction program.

The contracting strategy has been developed to provide opportunities to local communities, contractors, and labourers located in the area.

1.23.2 ACCESS

Site access will be established from three fronts:

- the Winter Access Road from Granduc to the Ted Morris Creek valley
- the Treaty Creek Access Road from Highway 37 to the saddle area and the main plant site
- the Coulter Creek Access Road from Eskay Creek to the Unuk River and on to the Mitchell site.

The Winter Access Road will mobilize in January of 2014, and the Treaty Creek and Coulter Creek access roads will mobilize in April of 2014.

1.23.3 CONSTRUCTION

For this study, it was assumed that the method of construction would be an opened managed site – neither union nor non-union. Rotations will be scheduled to allow sufficient time for personnel to travel and spend time at home with their families.

Mitchell site construction begins with the development of the site access roads to the WTP area, WSD, tunnel entrances, Coulter Creek Access Road, and building locations. Early works material and equipment will be mobilized via the Winter Access Road. Major equipment, general construction materials, and heavy earth moving equipment will be mobilized via the Coulter Creek Access Road. It has been assumed for this study that the main mechanical installations for the crusher, stockpiles, and conveyors will be by one General Contractor. This study assumes that the truck shop and permanent camp facility would be constructed as engineering, procurement, and construction (EPC) projects.

All material and equipment for the plant site will be transported using the Treaty Creek Access Road. This study assumes that the process plant would be constructed by one General Contractor, and the TMF would be constructed by the Earthworks Contractor. The construction schedule for both sites is coordinated around the development of the MTT.

1.23.4 MTT CONSTRUCTION

Plant site tunnel construction will start at the saddle area with two work fronts. As the project progresses, two additional work fronts will be added: from an adit in the Treaty Creek Glacier Valley at kilometre 12, and from the plant site. Mitchell site tunnel construction will start at the MTT south entrance and will connect with the development from the north. In the early stages of the project, helicopter support would be needed to move equipment, personnel, and the supplies needed for early construction work, on-site electrical generation, and manpower support.

1.23.5 MTT Conveyance System

Conveyors, fuel piping, electrical, fire protection, and water lines are planned to be installed in the MTT in three sections, which will be coordinated with tunnel progress and access availability.

1.23.6 Commissioning Program

The commissioning program would need to include at least three teams under the direction of the Commissioning Manager and the Owner's Operation Manager. The Commissioning Manager would report to the project at an early stage to develop the required commissioning plan, procedures, and safety guidelines.

1.24 Capital Cost Estimate

An initial capital of US$5.256 B is estimated for the Project, based on capital cost estimates developed by the following consultants:

- MMTS – mine capital costs, rock RSF and pit area pioneering works
- Allnorth – WSD, tailing starter dams, and surface water management earthworks based on KCB designs and quantities
- BVL – conveying, tailing and reclaim water piping, and pumping
- Stantec and KCB – tunnelling
- Tetra Tech – process plant and associated infrastructure costs including plant site preparation
- Brazier – permanent power supply, MTT conveyor electrical and fire detection, mini hydro plant, and energy recovery systems
- McElhanney – permanent access roads
- EBA – winter access road.

All currencies in this section are expressed in US dollars, unless otherwise stated. Costs have been converted using a fixed currency exchange rate, based on the Bank of Canada three-year average of Cdn$1.00 to US$0.96. The expected accuracy range of the capital cost estimate is +25%/-10%.

This capital cost estimate includes only initial capital, which is defined as all capital expenditures that are required to produce concentrate and doré. A summary of the capital costs is shown in Table 1.18.

This updated PFS estimate is prepared with a base date of Q1/Q2 2012. The estimate does not include any escalation past this date. Budget quotations were obtained for major equipment. The vendors provided equipment prices, delivery lead times, freight costs to a designated marshalling yard, and spares allowances. The quotations used in this estimate were obtained in Q1/Q2 2012, and are budgetary and non-binding. For non-major equipment (i.e. equipment less than $100,000), costing is based on in-house data or quotes from recent similar projects.

All equipment and material costs include Free Carrier (FCA) manufacturer plant Inco terms 2000. Other costs such as spares, taxes, duties, freight, and packaging will be covered separately in the Indirects section of the estimate.

Table 1.18 Capital Cost Summary

		Cost (US$ 000)
Direct Works		
A	Overall Site	199,818
B1	Open Pit Mining	185,826
B3	Underground Mining (Mitchell Block Caving)	0
B5	Underground Mining (Iron Cap Block Caving)	0
C	Crushing, Stockpiles, and Grinding	156,900
D1	Tunnelling	344,213
D2	MTT Transfer System	273,695
D3	Rope Conveyance (Sustaining)	0
E0	Plant Site Crushing	348,699
E1	Plant Site Grinding	458,242
F1	TMF	311,108
F6	Water Treatment	309,462
F8	Environmental	44,225
F9	Avalanche Control	45,845
G	Site Services and Utilities	34,226
J	Ancillary Buildings	96,097
K	Plant Mobile Equipment	10,676
M1	Temporary Services	190,739
M2	Treaty Road Marshalling Yard	10,791
N1	Permanent Electrical Power Supply and BC Hydro Capital Cost Contribution	217,319
N2	Mini Hydro Plants	16,536
N3	Energy Recovery Plants	7,576
P1	Permanent Access Roads	93,433
P2	Temporary Winter Access Roads	18,208
Q	Off-site Infrastructure and Facilities	73,896
Direct Works Subtotal		**3,447,530**
Indirects		
X	Project Indirects	1,056,550
Y	Owner's Costs	106,315
Z	Contingencies	645,743
Indirects Subtotal		**1,808,608**
Total		**US$5,256,138**

1.25 OPERATING COST ESTIMATE

The operating cost for the KSM Project was estimated at US$13.72/t milled. The estimate was based on an average daily process rate of 130,000 t/d milled. The cost estimates in this section are based upon budget prices in Q1/Q2 2012 or based on the data from the database of the consulting firms involved in the cost estimates. When required, costs in this report have been converted using a three-year average currency exchange rate of Cdn$1.00 to US$0.96. The expected accuracy range of the operating cost estimate is +25%/-10%.

Power will be supplied by BC Hydro at an average cost of US$0.047/kWh at the plant 25 kV bus bars, based on the BC Hydro credits for energy conservation by use of HPGR and similar. 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 costs. Power costs for surface service are included in the site services costs.

The estimated electrical power costs are based on the 2012 BC Hydro Tariff 1823 - Transmission Service Stepped Rate and Schedule 1901 - Deferred Account Rate Rider. The electrical power costs also account for local system losses and include 7% PST, which is being re-introduced and is not treated as an input tax credit. The rates take advantage of the implementation of BC Hydro-approved energy conservation measures in the plant design phase, including the HPGR circuit, which will greatly reduce the more costly Tier 2 power in the BC Hydro stepped–rate Schedule 1823.

Table 1.19 Average Operating Cost Summary

	US$/a (000)	US$/t Milled
Mine		
Mining Costs – Mill Feed	251,901	5.31*
Open Pit – Mill Feed		*5.38*
Block Caving – Mill Feed		*5.14*
Mill		
Staff & Supplies	233,012	4.91
Power (Process Only)	53,081	1.12
G&A and Site Service		
G&A	53,556	1.13
Site Service	14,959	0.32
Tailing and Water Treatment		
Tailing	24,440	0.52
Water Treatment	20,238	0.43**
Total Operating Cost	**651,187**	**13.72**

* excluding mine pre-production operating costs.
** LOM average cost calculated by total LOM operating cost divided by LOM process tonnage.

The total operating costs are defined as the direct operating costs including mining, processing, tailing storage, water treatment, site services, and G&A. The hydro-power credit from the recovered hydro-energy during mining operations is not accounted in the operating cost estimate, but included in the financial analysis. Sustaining capital includes all capital expenditures after the process plant has been put into production.

1.26 ECONOMIC EVALUATION

Tetra Tech prepared an economic evaluation of the 2012 KSM PFS based on a pre-tax financial model. For the 55-year mine life and 2,164 Mt reserve, the following pre-tax financial parameters were calculated using the base case metal prices:

- 11.5% internal rate of return (IRR)
- 6.2-year payback on US$5,256 M capital
- US$4,511 M net present value (NPV) at 5% discount rate and US$1,614 M at 8% discount rate.

The base case prices, using the three-year trailing average (as of April 15, 2012) were as follows:

- gold – US$1,330/oz
- copper – US$3.45/lb
- silver – US$25.20/oz
- molybdenum – US$15.00/lb
- exchange rate – Cdn$1.00 to US$0.96.

Metal revenues projected in the KSM cash flow models were based on the average metal values indicated in Table 1.20.

Table 1.20 Metal Production from the KSM Project

	Years 1-7	Years 1-20	LOM
Total Tonnes to Mill (000s)	310,062	926,916	2,164,419
Annual Tonnes to Mill (000s)	44,295	46,346	39,353
Average Grades			
Gold (g/t)	0.79	0.67	0.549
Copper (%)	0.234	0.180	0.207
Silver (g/t)	2.385	2.737	2.740
Molybdenum (ppm)	46.2	61.4	44.8
Total Production			
Gold (000s oz)	5,959	15,003	27,959

table continues…

	Years 1-7	Years 1-20	LOM
Copper (000s lb)	1,364,880	3,024,655	8,075,101
Silver (000s oz)	14,712	50,154	120,826
Molybdenum (000s lb)	9,067	41,477	62,679
Average Annual Production			
Gold (000s oz)	851	750	508
Copper (000s lb)	194,983	151,233	146,820
Silver (000s oz)	2,102	2,508	2,197
Molybdenum (000s lb)	1,295	2,074	1,140

Two additional metal price scenarios were also developed: one using the spot metal prices on April 15, 2012, including the closing exchange rate of that day (Spot Price Case); the other using gold, copper, and silver prices 20% lower than the April 15 prices at the Base Case exchange rate (Alternate Case). The input parameters and results of all scenarios can be found in Table 1.21.

Table 1.21 Summary of the Economic Evaluations

	Unit	Base Case	Spot Price Case	Alternate Case
Metal Price				
Gold	US$/oz	1,330.00	1,650.00	1,320.00
Copper	US$/lb	3.45	3.75	3.00
Silver	US$/oz	25.20	32.00	25.60
Molybdenum	US$/lb	15.00	15.00	15.00
Exchange Rate	US:Cdn	0.96	1.00	0.96
Economic Results				
NPV (at 0%)	US$ M	20,473	31,160	16,776
NPV (at 3%)	US$ M	8,196	13,137	6,612
NPV (at 5%)	US$ M	4,511	7,748	3,503
NPV (at 8%)	US$ M	1,614	3,503	1,031
IRR	%	11.53	14.73	10.35
Payback	Years	6.19	5.16	6.68
Cash Cost/oz Au	US$/oz	141.30	60.04	263.54
Total Cost/oz Au	US$/oz	597.60	535.35	719.84

1.26.1 Sensitivity Analysis

Sensitivity analyses were carried out on the following parameters:

- gold, copper, silver, and molybdenum metal prices
- exchange rate
- capital expenditure

- operating costs.

The analyses are presented graphically as financial outcomes in terms of NPV, IRR, and payback period. The project NPV is most sensitive to gold price and exchange rate followed by operating costs, copper price, capital costs, silver price, and molybdenum price. The IRR is most sensitive to exchange rate and gold price followed by capital costs, operating costs, copper price, silver price, and molybdenum price. The payback period is most sensitive to gold price and exchange rate followed by capital costs, copper price, operating costs, silver price, and molybdenum price. The NPV, IRR, and payback sensitivities can be seen in Figure 1.8, Figure 1.9, and Figure 1.10.

Figure 1.8 Sensitivity Analysis of NPV at 5% Discount Rate

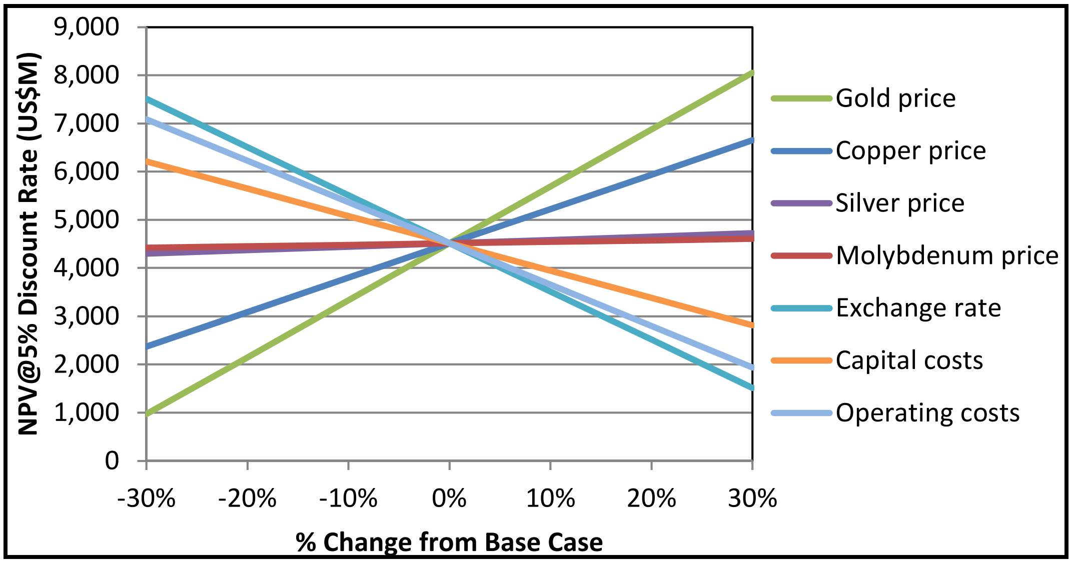


Figure 1.9 Sensitivity Analysis of IRR

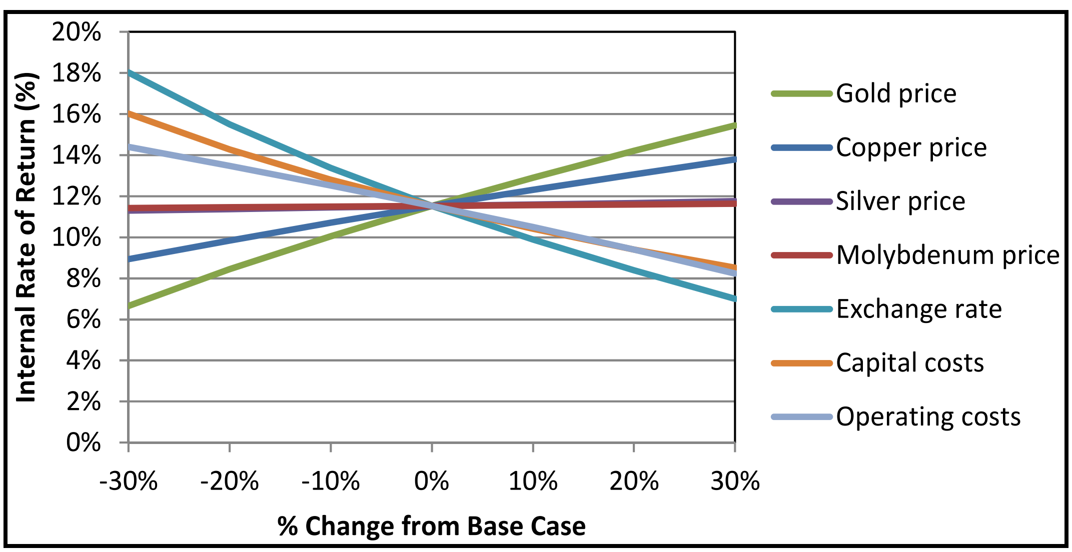
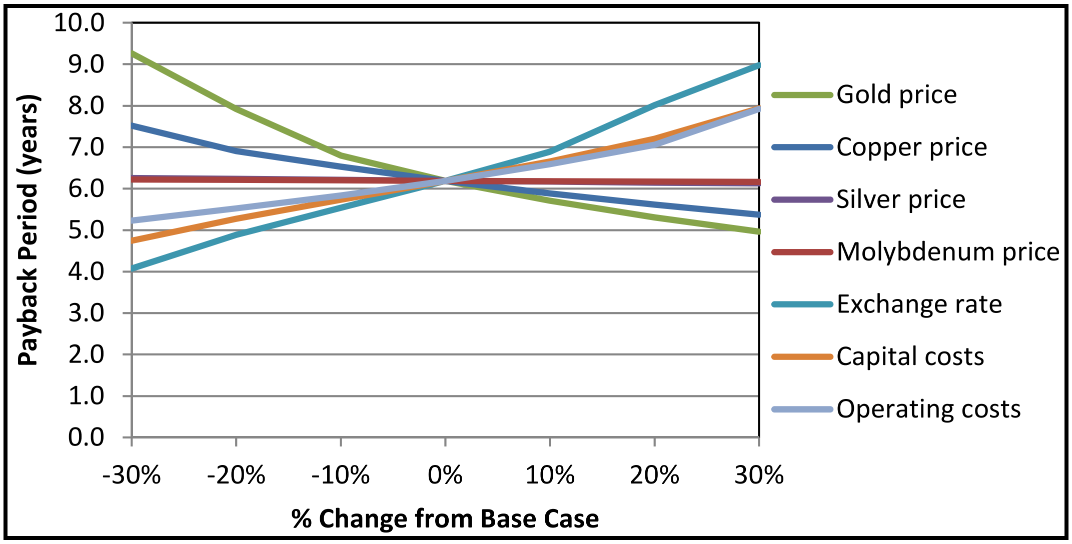


Figure 1.10 Sensitivity Analysis of Payback Period



1.27 RECOMMENDATIONS

Based on the work carried out in this 2012 KSM PFS and the resultant economic evaluation, this study should be followed by a Feasibility Study in order to further assess the economic viability of the Project.

Detailed recommendations are provided in Section 26.0 of this report.

2.0 INTRODUCTION

The KSM copper-gold project is currently owned by Seabridge. The KSM property is located in northwest 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.

This NI 43-101 PFS is intended to be used by Seabridge to present a current assessment of the project's likely economic outcome. This 2012 KSM PFS has been prepared in general accordance with the guidelines provided in NI 43-101 "Standards of Disclosure for Mineral Projects". The intent of the 2012 KSM PFS is to provide a comprehensive review of the economics of the mining operations and related project activities, and to provide recommendations for future work programs. This NI 43-101-compliant report has been prepared for Seabridge based on work performed by the following independent consultants:

- Tetra Tech
- RMI
- MMTS
- Brazier
- KCB
- Allnorth
- BVL
- McElhanney
- EBA
- BGC
- Stantec
- Golder
- Rescan.

A summary of the QPs responsible for each section of this report is provided in Table 2.1. QP certificates are included in Appendix A.

Table 2.1 Summary of Qualified Persons

Section	Description	Qualified Person	
		Company	**Qualified Person**
1.0	Summary	All	Sign off by section
2.0	Introduction	Tetra Tech	Jianhui (John) Huang
3.0	Reliance on Other Experts	Tetra Tech	Jianhui (John) Huang
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 and Mineralization	RMI	Michael J. Lechner
8.0	Deposit Types	RMI	Michael J. Lechner
9.0	Exploration	RMI	Michael J. Lechner
10.0	Drilling	RMI	Michael J. Lechner
11.0	Sample Preparation, Analyses, & Security	RMI	Michael J. Lechner
12.0	Data Verification	RMI	Michael J. Lechner
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Jianhui (John) Huang
14.0	Mineral Resource Estimate	RMI	Michael J. Lechner
15.0	**Mineral Reserve Estimates**		
15.2	Pit Reserves	MMTS	Jim Gray
15.3	Underground Reserves	Golder	Ross Hammett
16.0	**Mining Methods**		
16.1	Open Pit Mining Operations	MMTS	Jim Gray
16.2	Schedule Results	MMTS/ Golder	Jim Gray, Ross Hammett
16.3	Underground Mining	Golder	Ross Hammett
16.4	Pit Slope Design Angles	BGC	Warren Newcomen
17.0	Recovery Methods	Tetra Tech	Jianhui (John) Huang
18.0	**Project Infrastructure**		
18.1	Geotechnical & Water Management	KCBL	Graham Parkinson
18.2	Mitchell and Treaty Plant Site Layout	BVL	Harold Bosche
18.3	Crushed Ore Conveyor System	BVL	Harold Bosche
18.4	Infrastructure Tunnels	BVL	Harold Bosche
18.5	Site Roads	BVL	Harold Bosche
18.6	Process Plant Facilities	BVL	Harold Bosche
18.7	Ancillary Buildings	BVL	Harold Bosche
18.8	Plant Control & Instrumentation	BVL	Harold Bosche
18.9	Sewage	BVL	Harold Bosche
18.10	Communications System	BVL	Harold Bosche
18.11	Potable Water Supply	BVL	Harold Bosche

table continues…

Section	Description	Qualified Person	
		Company	Qualified Person
18.12	Power Supply and Distribution	Brazier	Neil Brazier
18.13	Plant & Mitchell Side Electrical Distribution	Brazier	Neil Brazier
18.14	Permanent & Construction Access Roads	McElhanney	Robert Parolin
18.15	Proposed Winter Access Road	EBA	Kevin Jones
18.16	Logistics	Tetra Tech	Jianhui (John) Huang
18.17	Construction Execution Plan	BVL	Harold Bosche
18.18	Owner's Implementation Plan	BVL	Harold Bosche
19.0	Market Studies and Contracts	Tetra Tech	Jianhui (John) Huang
20.0	Environmental Studies, Permitting, and Social or Community Impact	Rescan	Pierre Pelletier
21.0	Capital & Operating Cost Estimates	Tetra Tech/ KCB/ BVL/ McElhanney/ EBA MMTS/ Brazier/ Allnorth	John Huang, Graham Parkinson, Harold Bosche, Bob Parolin, Kevin Jones, Jim Gray, Neil Brazier Darby Kreitz
22.0	Economic Analysis	Tetra Tech	Sabry Abdel Hafez
23.0	Adjacent Properties	RMI	Michael J. Lechner
24.0	Other Relevant Data and Information	Tetra Tech	Jianhui (John) Huang
25.0	Interpretation and Conclusions	Tetra Tech	Jianhui (John) Huang Sabry Abdel Hafez
26.0	Recommendations	Multiple	Sign off by section

3.0 RELIANCE ON OTHER EXPERTS

Mr. Jack Butterfield of Butterfield Mineral Consultants Ltd. (Butterfield) has been relied on for matters relating to the smelting terms, refining terms, saleability, and sales terms for copper concentrate and molybdenite concentrate. These terms are included in Appendix B and summarized in Section 19.0.

Mr. John Brassard, Owner of TCG, has been relied on for matters relating to mineral and placer claims status and ownership. TCG provided a Title Review of the KSM property dated May 2, 2012, signed by Mr. Brassard, and subsequently updated in respect of placer claims on June 21, 2012 by e-mail report (Appendix M). Mr. Michael J. Lechner, who is responsible for the information in Section 4.0, has relied entirely on the information provided by Mr. Brassard regarding the claims which comprise the KSM property, their ownership and their status in Section 4.0.

Mr. William Threlkeld, Senior Vice President of Seabridge, has been relied on for matters relating to claims acquisition, royalties, and related agreements. Mr. Lechner, who is responsible for the information in Section 4.0, has relied entirely on the information provided by Mr. Threlkeld regarding the information relating to claims acquisition, royalties and related agreements which apply to the KSM property in Section 4.0.

Mr. Jack Meisl of JGM Management Ltd. has been relied on for matters relating to the construction execution plan and schedule.

4.0 PROPERTY DESCRIPTION AND LOCATION

The KSM property is located in northwest BC, at an approximate latitude of 56.50 N and a longitude of 130.30 W. The Mineral Resources that are the subject of this report are located relative to the NAD83 UTM coordinate system. The property is approximately 950 km northwest of Vancouver, 65 km north-northwest of Stewart, and 21 km south-southeast of the Eskay Creek Mine (production ceased in 2009). Figure 4.1 is a general location map.

The KSM property is comprised of three discontinuous claim blocks. These claim blocks are referred to as:

1. the KSM claim group
2. the Seabee/Tina claims
3. the KSM placer claim block.

The first two claim blocks (KSM and Seabee/Tina) contain 117 mineral claims, consisting of both cell and legacy claims. The total area of the three claim blocks covers an area of 52,133.26 ha. The Seabee/Tina claim block is about 19 km northeast of the Kerr-Sulphurets-Mitchell-Iron Cap mineralized zones. The Seabee/Tina claim block is currently being considered for proposed infrastructure siting.

The Kerr-Sulphurets-Mitchell mineral claims were purchased by Seabridge from Placer Dome in 2000. The mineral claims were converted from legacy claims to BC's new Mineral Titles Online (MTO) system in 2005. 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 by 380 m). 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.

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 for the Kerr and Sulphurets deposits.

The original KSM claim group consisted 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.) retains a 1% net smelter royalty that is capped at $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 Seabridge.

Since acquisition of the original KSM claim group, Seabridge has added to the project's property holdings through staking and purchase of several claim groups. These groups include the Seabee group, acquired by staking, the Tina and BJ groups purchased in 2009, and the New BJ group purchased in 2010. The Seabee and Tina groups are together referred to as the Seabee property, and the original KSM group, BJ and New BJ groups are referred to as the KSM property (Figure 4.2). The Kerr-Sulphurets placer claims were part of the original property acquisition from Placer Dome. Additional placer claims were acquired by staking in 2009, 2010, 2011, and 2012 (Figure 4.3).

Annual holding costs for all claims (lode and placer) are approximately Cdn$173,000 per year, which the company has maintained since acquiring the project. In 2007, assessment work was filed to advance the expiry of the KSM property to 2018. Assessment work was completed on most of the Seabee property in 2010 with that work filed in February 2011, which advanced expiry dates to 2017. The BJ group of claims had assessment work from 2010 applied, which advanced expiry dates to 2020, and New BJ Group had assessment work from 2011 applied to advance expiry dates to 2021. The Kerr-Sulphurets placer claims have been kept in good standing by paying fees in lieu of completing assessment work. The Claim Group Inc. (TCG) is the land manager and mineral tenure agent for Seabridge. Seabridge is provided with monthly 90-day forward reports of all land tenures (lode and placer) requiring action within that period. TCG files any work done on the properties, based on details provided by Seabridge, or files cash in lieu of work, for the company. RMI has relied on information with respect to all mining claim matters as provided by TCG in a "Title Review - KSM Property", by John Brassard, dated May 2, 2012.

RMI is unaware of any environmental liabilities associated with the KSM Project. It is RMI's understanding that Seabridge has obtained permits for ongoing exploration work. Seabridge is in the process of obtaining other permits.

Figure 4.1 General Location Map

Source: Rescan.

Table 4.1 KSM Claims – Lease Application EPC461 Approved in 2012

Claim #	Claim Name	Area (ha)	Good to Date	Anniv. Year	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	# of Cells	Map #
254758	ARBEE #54	25	14-Jun-2018	1	125.00	250.00		104B059
254756	ARBEE #35	25	16-Jun-2018	1	125.00	250.00		104B059
254757	ARBEE #39	25	16-Jun-2018	1	125.00	250.00		104B059
254759	ARBEE #55	25	16-Jun-2018	1	125.00	250.00		104B059
516236		303.273	30-Jun-2018	1	1,516.37	3,032.73	17	104B

table continues…

Claim #	Claim Name	Area (ha)	Good to Date	Anniv. Year	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	# of Cells	Map #
516237		71.379	30-Jun-2018	1	356.90	713.79	4	104B
516240		107.016	30-Jun-2018	1	535.08	1,070.16	6	104B
516241		142.709	30-Jun-2018	1	713.55	1,427.09	8	104B
516248		142.725	26-Aug-2018	1	713.63	1,427.25	8	104B
516251		321.344	26-Aug-2018	1	1,606.72	3,213.44	18	104B
516252		124.994	26-Aug-2018	1	624.97	1,249.94	7	104B
516253		178.622	26-Aug-2018	1	893.11	1,786.22	10	104B
516254		285.779	26-Aug-2018	1	1,428.90	2,857.79	16	104B
516256		53.586	26-Aug-2018	1	267.93	535.86	3	104B
516269		107.208	26-Aug-2018	1	536.04	1,072.08	6	104B
516242		71.363	23-Sep-2018	1	356.82	713.63	4	104B
516255		214.346	23-Sep-2018	1	1,071.73	2,143.46	12	104B
516245		356.921	12-Oct-2018	1	1,784.61	3,569.21	20	104B
516264		393.344	30-Oct-2018	1	1,966.72	3,933.44	22	104B
516258		178.573	03-Nov-2018	1	892.87	1,785.73	10	104B
516259		107.173	03-Nov-2018	1	535.87	1,071.73	6	104B
516260		107.197	03-Nov-2018	1	535.99	1,071.97	6	104B
516238		624.456	10-Dec-2018	1	3,122.28	6,244.56	35	104B
516239		535.513	10-Dec-2018	1	2,677.57	5,355.13	30	104B
516262		339.526	17-Dec-2018	1	1,697.63	3,395.26	19	104B
516263		643.881	17-Dec-2018	1	3,219.41	6,438.81	36	104B
516266		178.778	17-Dec-2018	1	893.89	1,787.78	10	104B
516267		250.242	17-Dec-2018	1	1,251.21	2,502.42	14	104B
516268		321.836	17-Dec-2018	1	1,609.18	3,218.36	18	104B
516261		464.635	20-Dec-2018	1	2,323.18	4,646.35	26	104B
30	**Mineral Claims**				**33,632.10**	**67,264.19**		

Note: all claims taken to lease: Lease plan #EPC461.
Source: TGC.

Table 4.2 KSM Claims – Lease Application EPC462 Approved in 2012

Claim #	Claim Name	Area (ha)	Good to Date	Anniv. Year	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	Map #
394782	BJ 7	500	11-Dec-2020	1	2,500.00	5,000.00	104B059
394783	BJ 8	500	11-Dec-2020	1	2,500.00	5,000.00	104B059
394784	BJ 9	400	11-Dec-2020	1	2,000.00	4,000.00	104B059
394792	BJ 16	500	11-Dec-2020	1	2,500.00	5,000.00	104B059
394793	BJ 17	400	11-Dec-2020	1	2,000.00	4,000.00	104B059
394795	BJ 19	500	11-Dec-2020	1	2,500.00	5,000.00	104B059
394796	BJ 20	375	11-Dec-2020	1	1,875.00	3,750.00	104B059
394799	BJ 23	500	11-Dec-2020	1	2,500.00	5,000.00	104B059
394800	BJ 24	300	11-Dec-2020	1	1,500.00	3,000.00	104B059

table continues…

Claim #	Claim Name	Area (ha)	Good to Date	Anniv. Year	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	Map #
394801	BJ 25	500	11-Dec-2020	1	2,500.00	5,000.00	104B059
394802	BJ 26	250	11-Dec-2020	1	1,250.00	2,500.00	104B059
394803	BJ 27	200	11-Dec-2020	1	1,000.00	2,000.00	104B059
394804	BJ 28	100	11-Dec-2020	1	500.00	1,000.00	104B059
394805	BJ 29	300	11-Dec-2020	1	1,500.00	3,000.00	104B049
394806	BJ 30	400	11-Dec-2020	1	2,000.00	4,000.00	104B049
394807	BJ 31	500	11-Dec-2020	1	2,500.00	5,000.00	104B049
16	**Mineral Clams**				**31,125.00**	**62,250.00**	

Note: all claims marked taken to lease: Lease plan #EPC462.
Source: TGC.

Table 4.3 KSM Claims – No Lease Application in Progress

Claim #	Claim Name	Area (ha)	Good to Date	Anniv. Year	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	Map #
705591	BJ GAP1	231.6166	05-Feb-2021	1	1,158.08	2,316.17	104B
705592	BJ GAP2	160.4624	05-Feb-2021	1	802.31	1,604.62	104B
394780	BJ 5	100	30-Nov-2021	1	500.00	1,000.00	104B059
394781	BJ 6	100	30-Nov-2021	1	500.00	1,000.00	104B059
394786	BJ 11	500	30-Nov-2021	1	2,500.00	5,000.00	104B059
394787	BJ 12	500	30-Nov-2021	1	2,500.00	5,000.00	104B059
394788	BJ 13	100	30-Nov-2021	1	500.00	1,000.00	104B059
394789	BJ 13A	25	30-Nov-2021	1	125.00	250.00	104B059
394790	BJ 14	100	30-Nov-2021	1	500.00	1,000.00	104B059
394791	BJ 15	250	30-Nov-2021	1	1,250.00	2,500.00	104B059
394794	BJ 18	300	30-Nov-2021	1	1,500.00	3,000.00	104B059
683463		1246.5185	30-Nov-2021	1	6,232.59	12,465.19	104B
683483		837.5991	30-Nov-2021	1	4,188.00	8,375.99	104B
394808	BJ 31A	375	31-Dec-2021	1	1,875.00	3,750.00	104B049
394809	BJ 32	150	31-Dec-2021	1	750.00	1,500.00	104B049
394810	BJ 33	450	31-Dec-2021	1	2,250.00	4,500.00	104B049
394811	BJ 34	150	31-Dec-2021	1	750.00	1,500.00	104B049
394812	BJ 35	450	31-Dec-2021	1	2,250.00	4,500.00	104B049
18	**Mineral Claims**				**30,130.98**	**60,261.97**	

Source: TGC.

Table 4.4 Seabee/Tina Mineral Claims

Claim #	Claim Name	Area (ha)	Good to Date	Anniv. Year	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	# of Cells	Map #
Seabee Property								
566467	BRIDGE1	445.8258	08-Feb-2017	1	2,229.13	4,458.26	25	104A
566468	BRIDGE2	445.5733	08-Feb-2017	1	2,227.87	4,455.73	25	104A
566469	BRIDGE3	427.7919	08-Feb-2017	1	2,138.96	4,277.92	24	104A
566470	BRIDGE4	427.977	08-Feb-2017	1	2,139.89	4,279.77	24	104A
566471	BRIDGE5	445.7336	08-Feb-2017	1	2,228.67	4,457.34	25	104A
566472	BRIDGE6	445.5766	08-Feb-2017	1	2,227.88	4,455.77	25	104A
566473	BRIDGE7	427.9217	08-Feb-2017	1	2,139.61	4,279.22	24	104A
566474	BRIDGE8	427.7599	08-Feb-2017	1	2,138.80	4,277.60	24	104A
566475	BRIDGE9	427.6131	08-Feb-2017	1	2,138.07	4,276.13	24	104A
566476	BRIDGE10	445.5312	08-Feb-2017	1	2,227.66	4,455.31	25	104A
566477	BRIDGE11	302.8823	08-Feb-2017	1	1,514.41	3,028.82	17	104A
566478	BRIDGE12	427.4311	08-Feb-2017	1	2,137.16	4,274.31	24	104A
566479	BRIDGE13	445.1533	08-Feb-2017	1	2,225.77	4,451.53	25	104A
566481	BRIDGE14	445.0611	08-Feb-2017	1	2,225.31	4,450.61	25	104A
566482	BRIDGE15	444.8427	08-Feb-2017	1	2,224.21	4,448.43	25	104A
566484	BRIDGE16	444.5621	08-Feb-2017	1	2,222.81	4,445.62	25	104A
566485	BRIDGE17	426.7283	08-Feb-2017	1	2,133.64	4,267.28	24	104A
566487	BRIDGE18	444.7114	08-Feb-2017	1	2,223.56	4,447.11	25	104A
566488	BRIDGE19	444.8346	08-Feb-2017	1	2,224.17	4,448.35	25	104A
566489	BRIDGE20	444.969	08-Feb-2017	1	2,224.85	4,449.69	25	104A
566490	BRIDGE21	427.2642	08-Feb-2017	1	2,136.32	4,272.64	24	104A
566491	BRIDGE22	445.1671	08-Feb-2017	1	2,225.84	4,451.67	25	104A
566492	BRIDGE23	427.3078	08-Feb-2017	1	2,136.54	4,273.08	24	104A
566493	BRIDGE24	427.9239	08-Feb-2017	1	2,139.62	4,279.24	24	104A
566494	BRIDGE25	427.9246	08-Feb-2017	1	2,139.62	4,279.25	24	104A
566495	BRIDGE26	444.8785	08-Feb-2017	1	2,224.39	4,448.79	25	104A
566496	BRIDGE27	391.3145	08-Feb-2017	1	1,956.57	3,913.15	22	104B
566497	BRIDGE28	444.4573	08-Feb-2017	1	2,222.29	4,444.57	25	104A
566567	BRIDGE29	427.4572	08-Feb-2017	1	2,137.29	4,274.57	24	104A
571582	SEABEE1	408.8286	08-Feb-2017	1	2,044.14	4,088.29	23	104A
571583	SEABEE2	373.1368	08-Feb-2017	1	1,865.68	3,731.37	21	104A
571584	SEABEE3	444.068	08-Feb-2017	1	2,220.34	4,440.68	25	104A
571585	SEABEE4	426.0832	08-Feb-2017	1	2,130.42	4,260.83	24	104A
571586	SEABEE5	372.6392	08-Feb-2017	1	1,863.20	3,726.39	21	104A
571587	SEABEE6	159.6419	08-Feb-2017	1	798.21	1,596.42	9	104A
573813	SEABEE7	213.2634	08-Feb-2017	1	1,066.32	2,132.63	12	104A
575633	SEA 1	445.1987	08-Feb-2017	1	2,225.99	4,451.99	25	104A
575635	SEA 2	445.3012	08-Feb-2017	1	2,226.51	4,453.01	25	104A
575636	SEA 3	445.4096	08-Feb-2017	1	2,227.05	4,454.10	25	104A

table continues…

Claim #	Claim Name	Area (ha)	Good to Date	Anniv. Year	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	# of Cells	Map #
575638	SEA 4	445.4484	08-Feb-2017	1	2,227.24	4,454.48	25	104A
575639	SEA 5	445.3365	08-Feb-2017	1	2,226.68	4,453.37	25	104A
575642	SEA 6	445.085	08-Feb-2017	1	2,225.43	4,450.85	25	104A
575643	SEA 7	213.4398	08-Feb-2017	1	1,067.20	2,134.40	12	104A
575645	SEA 8	427.0822	08-Feb-2017	1	2,135.41	4,270.82	24	104A
575646	SEA 9	35.598	08-Feb-2017	1	177.99	355.98	2	104A
603133	SEABEE 8	426.5614	08-Feb-2017	1	2,132.81	4,265.61	24	104B
46	**Mineral Claims**				**93,371.49**	**186,742.97**		
Tina Property								
401548	TINA 1	500	28-Feb-2018	1	2,500.00	5,000.00		104B070
401549	TINA 2	500	28-Feb-2018	1	2,500.00	5,000.00		104B070
401550	TINA 3	500	28-Feb-2018	1	2,500.00	5,000.00		104B070
401551	TINA 4	500	28-Feb-2018	1	2,500.00	5,000.00		104B070
401552	TINA 5	500	28-Feb-2018	1	2,500.00	5,000.00		104B070
401553	TINA 6	250	28-Feb-2018	1	1,250.00	2,500.00		104B070
603134	SEABEE 9	53.3796	28-Feb-2018	1	266.90	533.80		104B
7	**Mineral Claims**				**14,016.90**	**28,033.80**		

Source: TGC.

Table 4.5 Seabridge Placer Claims

Claim #	Claim Name	Area (ha)	Good to Date	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	# of Cells	Map #
516677	SUL 11	17.858	11-Jul-2013	357.16	714.32	1	104B
				357.16	**714.32**		
879649	KSM P4	17.8562	3-Aug-2013	357.12	714.25	1	104B
879669	KSM P5	17.8739	3-Aug-2013	357.48	714.96	1	104B
879689	KSM P6	250.1757	3-Aug-2013	5,003.51	10,007.03	14	104B
879709	KSM P7	374.8147	3-Aug-2013	7,496.29	14,992.59	21	104B
879729	KSM P8	338.9714	3-Aug-2013	6,779.43	13,558.86	19	104B
879749	KSM P9	445.8224	3-Aug-2013	8,916.45	17,832.90	25	104B
879769	KSM P10	445.7912	3-Aug-2013	8,915.82	17,831.65	25	104B
879789	KSM P11	445.8481	3-Aug-2013	8,916.96	17,833.92	25	104B
879809	KSM P12	338.8379	3-Aug-2013	6,776.76	13,553.52	19	104B
				53,519.83	**107,039.66**		
896249	KSM PL13	160.5212	8-Sep-2013	3,210.42	6,420.85	25	104B
896250	KSM PL14	339.0053	8-Sep-2013	6,780.11	13,560.21	19	104B
896251	KSM PL15	428.9506	8-Sep-2013	8,579.01	17,158.02	24	104B
896252	KSM PL16	447.0914	8-Sep-2013	8,941.83	17,883.66	25	104B
896253	KSM PL17	447.0909	8-Sep-2013	8,941.82	17,883.64	25	104B
896254	KSM PL18	446.9927	8-Sep-2013	8,939.85	17,879.71	25	104B
896255	KSM PL19	446.993	8-Sep-2013	8,939.86	17,879.72	25	104B

table continues…

Claim #	Claim Name	Area (ha)	Good to Date	Annual Work Due (Cdn$)	Annual CIL (Cdn$)	# of Cells	Map #
896256	KSM PL20	357.4967	8-Sep-2013	7,149.93	14,299.87	20	104B
516328		71.453	28-Sep-2013	1,429.06	2,858.12	4	104B
516330		107.185	28-Sep-2013	2,143.70	4,287.40	6	104B
516332		107.179	28-Sep-2013	2,143.58	4,287.16	6	104B
516333		89.334	28-Sep-2013	1,786.68	3,573.36	6	104B
516323		107.191	28-Sep-2013	2,143.82	4,287.64	6	104B
516325		125.043	28-Sep-2013	2,500.86	5,001.72	7	104B
516375		125.023	28-Sep-2013	2,500.46	5,000.92	7	104B
516676		17.858	28-Sep-2013	357.16	714.32	1	104B
				76,488.16	**152,976.31**		
694483	KSM P1	357.3606	5-Jan-2014	7,147.21	14,294.42	20	104B
694543	KSM P2	410.4906	5-Jan-2014	8,209.81	16,419.62	23	104B
694683	KSM P3	427.8648	5-Jan-2014	8,557.30	17,114.59	24	104B
				23,914.32	**47,828.64**		
576658	KERR PL1	446.861	20-Feb-2014	8,937.22	17,874.44	25	104B
576659	KERR PL2	446.6192	20-Feb-2014	8,932.38	17,864.77	25	104B
576660	KERR PL3	446.3943	20-Feb-2014	8,927.89	17,855.77	25	104B
576661	KERR PL4	446.2295	20-Feb-2014	8,924.59	17,849.18	25	104B
576662	KERR PL5	446.0319	20-Feb-2014	8,920.64	17,841.28	25	104B
576663	KERR PL6	446.0181	20-Feb-2014	8,920.36	17,840.72	25	104B
576664	KERR PL7	142.7335	20-Feb-2014	2,854.67	5,709.34	8	104B
576665	KERR PL8	321.3959	20-Feb-2014	6,427.92	12,855.84	18	104B
576666	KERR PL9	285.6986	20-Feb-2014	5,713.97	11,427.94	16	104B
576667	KERR PL10	357.3994	20-Feb-2014	7,147.99	14,295.98	20	104B
				75,707.63	**151,415.26**		
986922	PL21	35.72	16-May-14	714.40	1,428.80	2	104B
986924	PL22	35.72	16-May-14	714.40	1,428.80	2	104B
986925	PL23	107.17	16-May-14	2,143.40	4,286.80	6	104B
				3,572.20	**7,144.40**		
42	**Placer Claims**						

Source: TGC.

The KSM Project is located on provincial Crown land. The four gold-copper deposits, and the proposed waste rock storage areas, lie within the Unuk River drainage in the area covered by the Cassiar Iskut-Stikine Land and Resource Management Plan, approved by the BC Government in 2000. A part of the proposed ore transport tunnel lies within the boundaries of the South Nass Sustainable Resource Management Plan that is currently in development. The proposed sites for the tailing management and plant facilities lie outside of the boundaries of any land use planning process. Part of the Project, excluding the mineral deposits and their immediately-related infrastructure, lies within the boundaries of the Nass Area, as defined in the Nisga'a Final Agreement, where consultation is required with the Nisga'a Lisims Government under the terms of the Final Agreement. The Tahltan

First Nation has an asserted claim over part or all of the area underlying the Project footprint. Additionally, the Gitanyow and Gitxsan Hereditary Chiefs, including Wilp Skii Km Lax Ha, may have some interests within the broader region, particularly downstream of the plant site and tailing management facility, potentially affected by the Project.

Seabridge has completed an extensive, two-year environmental baseline program, initiated in 2007, in support of the Provincial and Federal Governments' permitting process. Rescan, a Canadian international consulting firm offering a wide range of environmental and engineering services to clients around the world including many of the largest mining companies, is conducting environmental studies under the leadership of its President, Clem Pelletier.

In March 2010, an application was made for a Multi-Year Area-Based (MYAB) permit, covering work at the KSM property for a five-year period. Approval for this work, which covers drilling, geophysical surveys, and base line environmental studies, was granted on June 30, 2010 (Permit # MX-1-571, Approval #10-0100108-0630). Prior to this approval, the Ministry of Energy Mines and Petroleum Resources (MEMPR) granted an extension to the 2009 permit on April 30, 2010, in order to complete the approved program commenced in 2009. Work on the Seabee property is covered by a separate permit, MX-1-763. An application for an MYAB permit for Seabee was approved in June 2012.

Figure 4.2 shows Seabridge's mineral claim blocks including the KSM, Seabee, and Tina groups. The location of the four mineralized zones (Kerr, Sulphurets, Mitchell, and Iron Cap) is depicted in the southwestern portion of the figure. Figure 4.3 shows Seabridge's placer claims.

Figure 4.2 KSM Mineral Claim Map

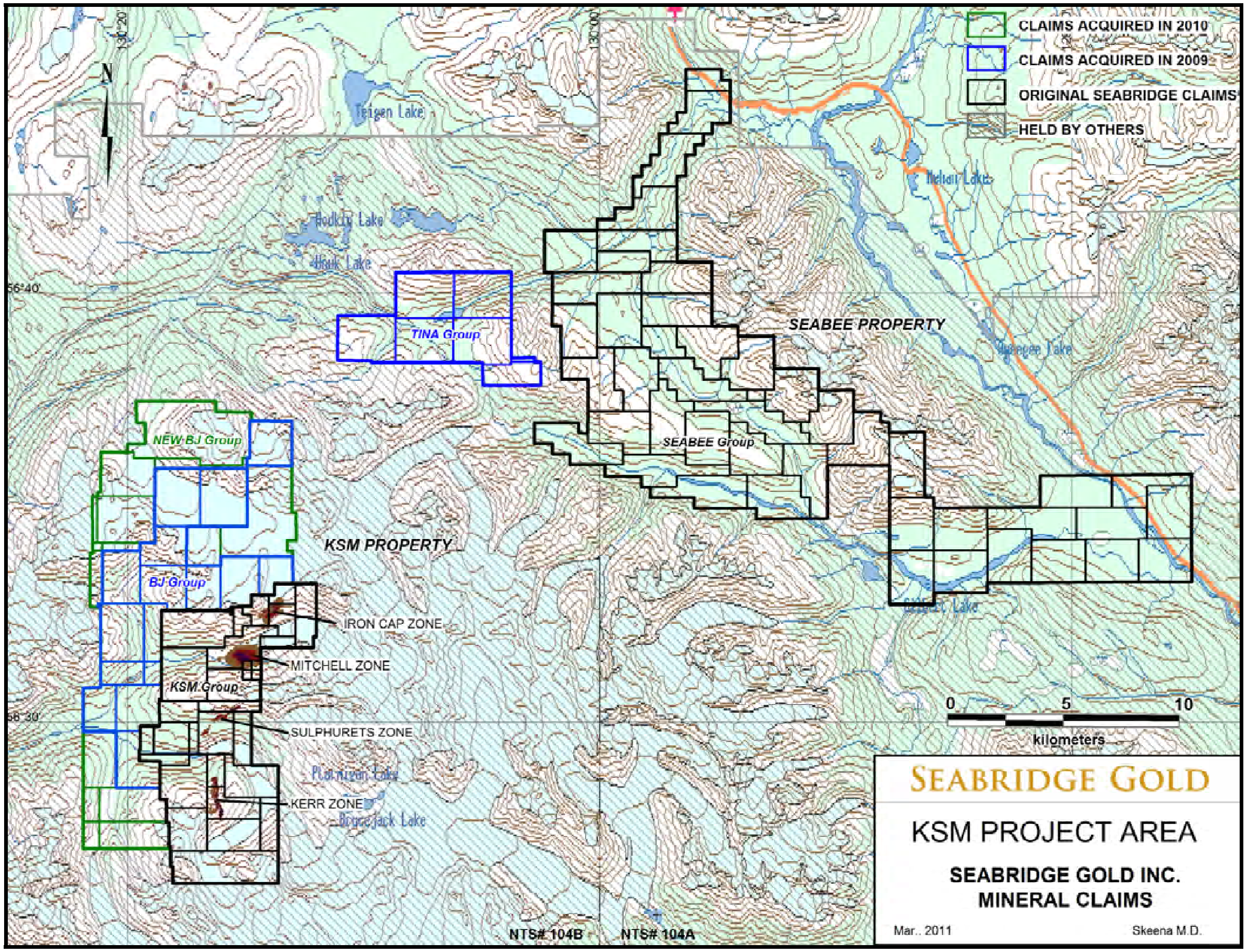
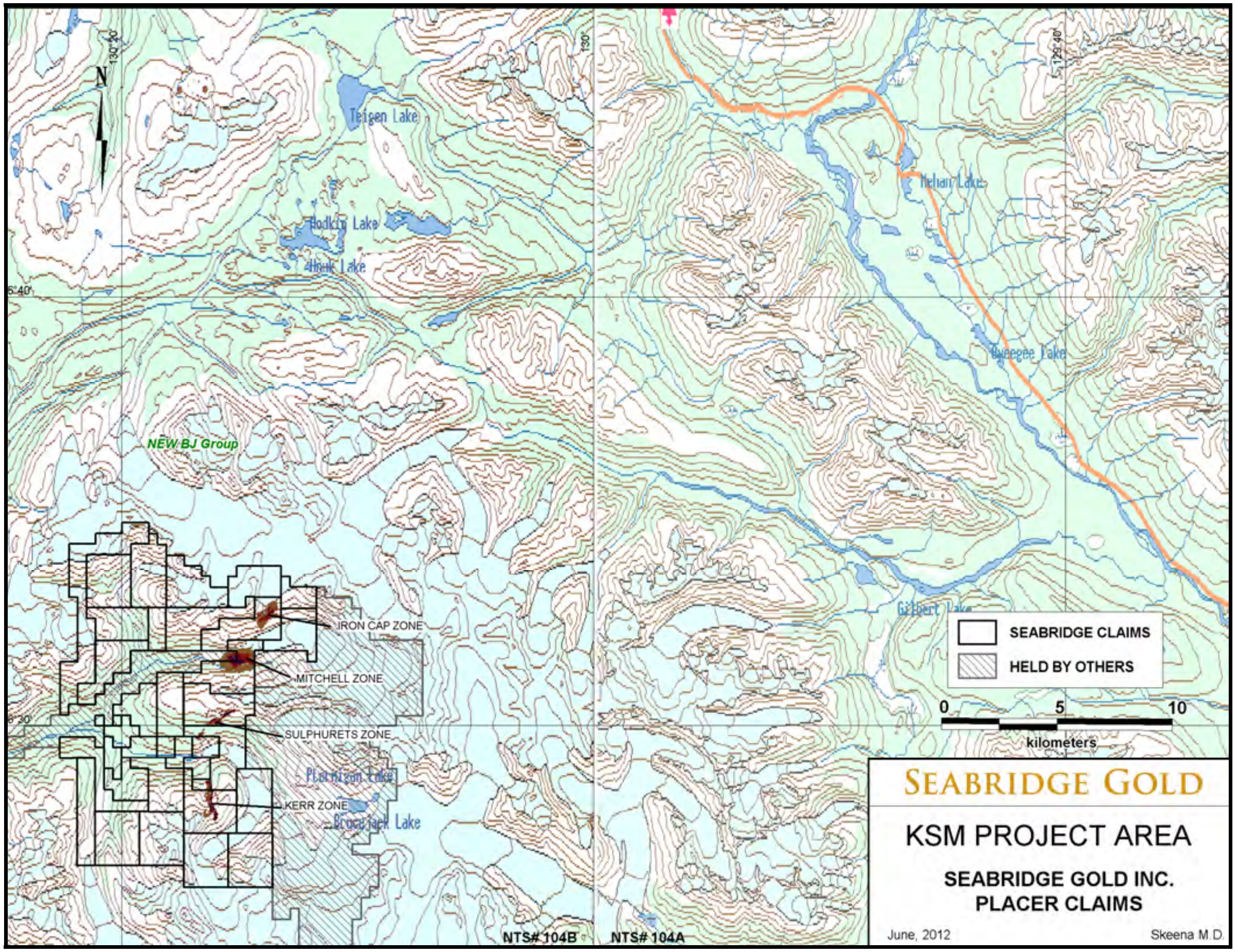


Figure 4.3 KSM Placer Claim Map



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

The following section was taken from RMI's April 6, 2007 NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia" (Lechner, 2007) and remains largely unchanged, and has only been updated for consistency in abbreviations and grammar.

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 indicate the Mitchell Glacier has retreated almost a kilometre laterally and perhaps several hundred metres vertically since 1991.

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. Tree line lies at about 1,240 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 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. Two staging areas for mobilizing crews and equipment were used. These are:

1. 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
2. along the public Granduc Road, which is 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 CNR Yellowhead route, which is located approximately 220 km southeast of the property. This line runs east-west, and can deliver concentrate to deep-water ports near Prince Rupert and Vancouver, 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; there is limited commercial 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 southeast of the property. There are proposals to develop local hydroelectric power sources and extend the Highway 37A transmission line northward.

AMEC plc, of Vancouver, BC, was commissioned by Noranda 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

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 Corp. (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 Ltd. (Brinco) in 1985, and Mitchell Creek by Newhawk Gold in 1991.

In 1989, Placer Dome acquired a 100% interest in the Kerr deposit from Western Canadian Mines; in the following year, they acquired the adjacent Sulphurets property from Newhawk Gold. The Sulphurets property also hosts the Mitchell Creek deposit and other mineral occurrences. In 2000, Seabridge Resources Inc. 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 Pretium Resources Inc. (Pretium), 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 Zone.

Table 6.1 Exploration Summary of the Kerr Zone

Year	Activity
1982-1983	"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 Mines 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 50m centers 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 acquired an option from Seabridge with the right to earn up to a 65% interest in Kerr.
2003-2004	Noranda undertook various exploration surveys.
2006	Seabridge purchases Falconbridge (formerly Noranda) option.
2009	Seabridge drilled 7 holes totalling about 1,159 m; conducted metallurgical testing and permit work.
2010	Seabridge drilled 4 holes totalling about 1,453 m; conducted metallurgical testing and permit work.
2011	Seabridge drilled 4 resource definition holes totalling about 2,338 m, continued with prefeasibility studies.

Table 6.2 summarizes the exploration history of the Sulphurets, Mitchell, and Iron Cap zones.

Table 6.2 Exploration Summary of the Sulphurets, Mitchell, and Iron Cap Zones

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 conducted geologic/geochem surveys and drilled 9 holes into the 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.

table continues…

Year	Activity
1971-1977	Granduc conducted additional exploration surveys targeting molybdenum and drilled 6 holes into the Snowfield Zone (Bruceside).
1979-1984	Esso Minerals optioned 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 Gold. Lacana-Newhawk JV spent approx. $21 M developing 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.
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 Gold.
1996-2000	Sulphurets project was dormant.
2000	Seabridge acquired a 100% interest in the Sulphurets/Mitchell properties from Placer Dome.
2002	Noranda acquired an option to earn up to 65% from Seabridge.
2003-2004	Noranda 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 drilled 29 holes totalling about 9,129 m at the Sulphurets and Mitchell zones.
2007	Seabridge purchased Arbee prospect from D. Ross and drilled 37 holes totalling 15,650 m.
2008	Seabridge drilled 37 holes totalling 17,192 m, started metallurgical testing, obtained new topographic data, and initiated permit related activities.
2009	Seabridge drilled approximately 13,000 m (resource definition, geotechnical and water monitoring), conducted metallurgical testing, and intensified permit data collection.
2010	Seabridge drilled 29 holes totalling about 9,725 m (resource definition and geotechnical), conducted metallurgical testing, and intensified permit data collection.
2011	Seabridge drilled 47 resource definition holes totalling about 20,000 m, continued prefeasibility level work

A total of 63 NQ core holes totalling 20,178 m of drilling data were completed at the KSM Project in 2011. The majority of Seabridge's 2011 drilling campaign was concentrated on the Sulphurets Zone (41 holes). Nine holes were drilled at the Kerr Zone and six holes completed at the Mitchell Zone.

6.2 Historical Resource Estimates

RMI is unaware of any publicly disclosed historical resource estimates for the KSM deposits prior to Seabridge's entry into the district. RMI has prepared NI 43-101 compliant Mineral Resources for the Kerr, Sulphurets, Mitchell, and Iron Cap zones (Lechner, 2007; Lechner, 2008b; Lechner, 2009; Lechner, 2010; Lechner, 2011).

6.3 History of Production

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

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 GEOLOGICAL SETTING

The following section was taken directly from RMI's April 2008 NI 43-101 report (Lechner, 2008b) and remains largely unchanged, and has only been updated for consistency in abbreviations and grammar.

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.

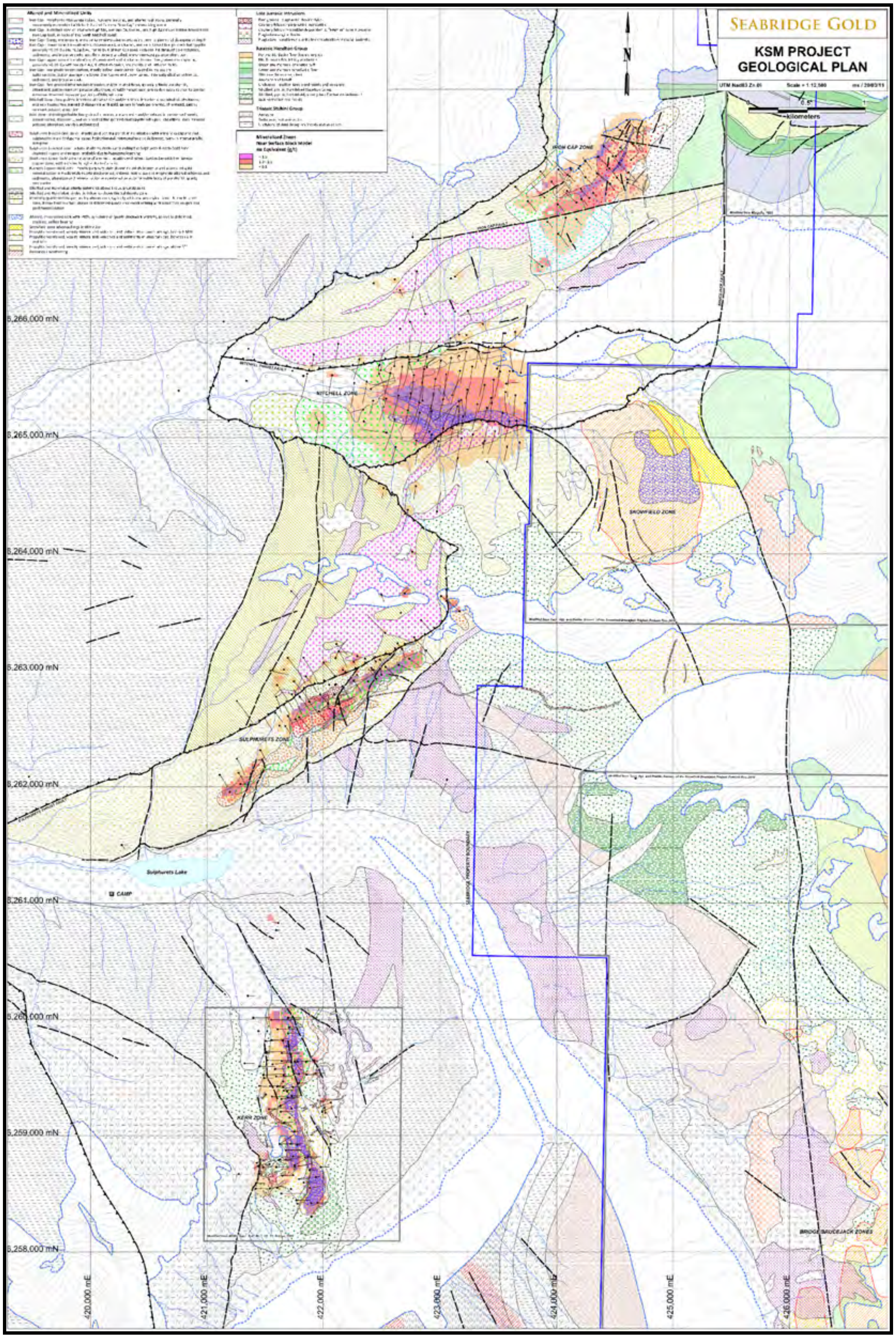
At Kerr-Sulphurets, Triassic rocks include marine sediments and intermediate volcanics of the Stuhini Group. The lowermost Stuhini Group is dominated by turbiditic argillite and sandstone, which are overlain by volcanic pillowed flows and breccias. The upper portion consists of turbidites and graded 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.

Figure 7.1 is a generalized geologic map of the KSM district showing lithology, alteration, major structures, drill hole collar locations, and gold equivalent mineralized zones.

Figure 7.1 KSM Geologic Map



7.2 MINERALIZATION

7.2.1 KERR ZONE

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 the 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.

7.2.2 SULPHURETS ZONE

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 down-dip and along strike to the southwest. 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.

7.2.3 MITCHELL ZONE

The Mitchell Zone is exposed in Mitchell 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 or mylonitic 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 moderately steep to the north. In general, the core area of mineralization has a moderate plunge to the north or northwest, and is lineated in a east-southeast direction.

Recent glacial melt back has provided exceptional surface exposure of a relatively fresh gold-copper porphyry system. A zone of intense quartz and sulfide veining ("High Quartz") forms resistant bluffs in Mitchell Valley. However, the higher grade core area is mostly covered by talus and moraine west of the bluffs. Active oxidation and leaching of sulfides has produced prominent gossans and extensive copper sulfate precipitates at the surface.

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. A property scale geological plan is presented in Figure 7.1.

7.2.4 *Iron Cap Zone*

The Iron Cap deposit is a separate but related mineralized system within the KSM district, and occurs structurally above the Mitchell deposit, in the panel of rocks between the Mitchell and Sulphurets thrust faults. It differs from the Mitchell deposit primarily in that much of the host rock is hydrothermally altered intrusive (porphyritic monzonite to diorite) rather than volcanics and sediments. The volcanics are mostly andesitic porphyry, generally similar to the main host of the Mitchell deposit. There is a high degree of silicification, which overprints earlier potassic and chloritic alteration. Intense phyllic alteration and high density stockwork veining, which are pervasive at Mitchell, are less pervasive at Iron Cap. Copper-bearing zones at Iron Cap demonstrate higher grades than Mitchell, which is consistent with the intrusive setting and potassic alteration, indicating a deeper and hotter environment.

Associated with the silicification are wide zones of hydrothermal brecciation, scattered meter-scale quartz-pyrite-chalcopyrite veins and centimetre-scale quartz-pyrite-chalcopyrite-sphalerite-galena-tetrahedrite veins that are interpreted to be superimposed on earlier stockwork and disseminated mineralization associated with the intrusion. Microscopic examinations of polished thin sections confirm that Iron Cap was also subjected to a post-mineral deformational event evidenced by widespread mylonitic textures. "Mylonite" and "Ultramylonite" are terms used as rock names in petrographic descriptions of several Iron Cap mineralized samples.

Generally intense silicification at the higher, eastern portions gives way to chloritization with some preserved k-spar alteration at depth and towards the west which correlates with increasing proportion of intrusive rock. Relative to Mitchell, stockwork veining is much weaker. There is a distinct overprint of structurally controlled, centimetre-scale quartz-carbonate veins with chalcopyrite, galena, sphalerite, and tetrahedrite but the distribution is not clear. It does not seem to effect the gold and copper distribution on a large scale, but at the vein scale there is often correlation. High silver values are generally associated with presence of galena and sphalerite.

8.0 DEPOSIT TYPES

The following section was taken from RMI's April 6, 2007 NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia" (Lechner, 2007) and remains largely unchanged, and has only been updated for consistency in abbreviations and grammar.

The KSM property hosts an extensive alteration and mineralization system that was 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 metres. 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 EXPLORATION

This section describes Seabridge's 2011 exploration program at KSM. Prior exploration activities have been described in various Technical Reports prepared by RMI (Lechner 2007, 2008a, 2008b, 2009, 2010, and 2011).

9.1 2011 KSM EXPLORATION PROGRAM

Seabridge's 2011 exploration efforts were directed towards the following:

- Infill drilling was conducted within the Sulphurets deposit in order to upgrade resource categories within current pit designs to at least an Indicated level.
- Drilling at Kerr was directed towards upgrading Inferred resources.
- Limited drilling at the Mitchell Zone focused on assessing the potential for developing a cost effective bulk underground mining method.
- Geotechnical core drilling was conducted to provide data for engineering studies at Kerr, Sulphurets, Mitchell, and Iron Cap. Some of these holes intersected the mineralized zones and contributed to the resource database.
- Geotechnical overburden and core drilling was completed in areas of proposed infrastructure well beyond the mineralized zones; this work is documented in a report prepared by KCB.

The 2011 Kerr, Sulphurets, and Mitchell drilling programs are tabulated in Table 10.1. Individual mineral zone drilling statistics are summarized in Table 10.3 through Table 10.6.

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, rock quality designation (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, Stewart Group (Eco Tech), a commercial laboratory located in Kamloops, BC. Seabridge has used Eco Tech for their prior drilling campaigns. (Eco Tech was recently purchased by the Stewart Group. RMI will refer to Eco Tech Laboratories, Stewart Group as "Eco Tech" throughout the remainder of this report.) The samples were analyzed for gold, copper, and a suite of other elements. Additional bulk density determinations were completed by Seabridge geologists from all rock types and alteration assemblages.

Geotechnical data collection and studies were contracted to BGC, KCB, and Rescan, all based in Vancouver, BC.

9.2 Results of 2011 Exploration Program

The previous geologic interpretations of the Kerr, Sulphurets, and Mitchell zones were updated using the 2011 core hole data. RMI notes the updated geologic interpretation remains virtually unchanged from the previous interpretation. Approximately 60% of the 2011 drilling program focused on exploration and delineation of Mineral Resources for the Sulphurets Zone.

The drilling, sampling, and assay procedures employed for the 2011 exploration program were adopted from previous years and are discussed in Section 11.0.

9.3 Interpretation of Exploration Data

RMI combined the 2011 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 the results from those activities, are discussed in Section 14.0.

9.4 Statement Regarding Nature of Investigations

All of the exploration activities that were conducted at KSM in 2011 were either directly carried out by Seabridge's geologic staff or directly supervised by Seabridge personnel.

10.0 DRILLING

This section describes Seabridge's 2011 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 (Lechner 2007, 2008a, 2008b, 2009, 2010, and 2011).

10.1 2011 DRILLING CAMPAIGN

Drilling was contracted to Hy-Tech Drilling Ltd., who used a heli-portable drill rig to drill NQ diameter core drill holes through the overburden and bedrock. Accommodations for field personnel were provided by Seabridge at their seasonal exploration camp in Sulphurets Creek Valley on the KSM property. Transportation was by helicopter chartered from Lakelse Air Ltd. of Terrace, BC, under a service contract with Matrix Helicopter Solutions Inc. of Kelowna. Labour for camp support, drill pad construction, and technical assistance was contracted to CJL Enterprises Ltd., Tahltan Northern Exploration Services Ltd., and Tsetaut Ventures Ltd. All core logging and geological interpretations were conducted by Seabridge personnel under the supervision Mr. Mike Savell, Senior Geologist for Seabridge.

Eco Tech (now merged with ALS Chemex Laboratories Ltd. [ALS Chemex]) from Kamloops, BC, analyzed approximately 10,071 diamond core samples that were collected from the 2011 Kerr, Sulphurets, and Mitchell drilling programs. The samples were analyzed for gold, copper, and a suite of other elements.

There were 764 quality control samples (blanks, standards, and duplicates) submitted with the core samples. From these core and control samples, 597 pulps (6%) were selected and analyzed by ALS Chemex in Vancouver, BC, as per the quality assurance/quality control (QA/QC) protocols that were established for prior drilling campaigns. Additional QA/QC samples were submitted with the Eco Tech pulps that were sent to ALS Chemex. The total KSM drill hole database is summarized by zone in Table 10.1.

Table 10.1 KSM Drill Hole Database

Mineral Zone/Area	Total Holes	Total Meterage
Kerr	159	31,358.77
Sulphurets	126	36,601.24
Mitchell	154	56,951.52
Iron Cap	52	17,790.13
Geotechnical Infrastructure	17	1,673.17
Total	**508**	**144,374.83**

10.2 DRILL HOLE SURVEYING

The procedures used for spotting the drill holes, surveying collars and down-hole surveying methods are the basically the same as those described for the 2007 drilling campaign (Lechner, 2008; Lechner, 2011). The following 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.

- **Kerr Deposit** – Previous to Seabridge's ownership of the property, 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 Differential Global Positioning System (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 topo surface.
- **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. This elevation of drill holes did not always conform to the LiDAR survey and were adjusted to topography like was done for the Kerr drilling.

- **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 DGPS unit. Like the other two deposits, the elevation for some of the drill holes was adjusted to match the new NAD83 LiDAR topography.

Table 10.2 summarizes the KSM drill hole database, which was used to estimate resources, organized by company and mineral zone. Table 10.3 through Table 10.6 break down the resource drilling for the Kerr, Sulphurets, Mitchell, and Iron Cap zones by company and year, respectively.

Table 10.2 KSM Drill Hole Summary by Company

Company	Number	Metres	% of Total
Kerr			
Brinco	3	189.90	1
Western Canadian	36	5,324.56	17
Sulphurets Gold	18	4,197.38	13
Placer Dome	82	16,404.43	52
Seabridge	20	5,242.50	17
Total Kerr	**159**	**31,358.77**	**100**
Sulphurets			
Esso	11	1,902.72	5
Newhawk Gold	7	1,306.30	4
Placer Dome	23	5,577.34	15
Falconbridge	7	1,648.09	5
Seabridge	78	26,166.79	71
Total Sulphurets	**126**	**36,601.24**	**100**
Mitchell			
Newhawk Gold	4	647.30	1
Falconbridge	4	1,197.29	2
Seabridge	146	55,106.93	97
Total Mitchell	**154**	**56,951.52**	**100**
Iron Cap			
Esso	5	1,051.26	6
Falconbridge	5	1,246.60	7
Seabridge	42	15,492.27	87
Total Iron Cap	**52**	**17,790.13**	**100**
Total KSM Project			
Esso	16	2,953.98	2
Brinco	3	189.90	0
Western Canadian	36	5,324.56	4
Newhawk Gold	11	1,953.60	1

table continues...

Company	Number	Metres	% of Total
Sulphurets Gold	18	4,197.38	3
Placer Dome	105	21,981.77	15
Falconbridge	16	4,091.98	3
Seabridge	286	102,008.49	71
Grand Total	**491**	**142,701.66**	**100**

Table 10.3 Kerr Zone Drill Hole Summary by Company

Company	Year Drilled	Hole Pre-fix	No. Holes	Metres	% of Total
Brinco	1985	85-nnn	3	189.90	0.6
Western Canadian	1987-1988	K87-nnn, K88-nnn, 88-nn	36	5,324.56	17.0
Newhawk Gold	1988	T88-nnn	2	115.21	0.4
Sulphurets Gold	1989	K89-nnn, T89-nnn	20	4,365.35	13.9
Placer Dome	1992	KS-nnn, KS92-nnn	83	16,413.57	52.3
Seabridge	2009	K-09-nn, MW-09-nna	7	1,158.75	3.7
Seabridge	2010	K-10-nn	4	1,453.00	4.6
Seabridge	2011	K-11-nn	4	2,338.40	7.5
Total	**n/a**	**n/a**	**159**	**31,358.74**	**100.0**

Table 10.4 Sulphurets Zone Drill Hole Summary by Company

Company	Year Drilled	Hole Pre-fix	No. Holes	Metres	% of Total
Esso Resources	1980, 1981	S80-nn, S81-nn	11	1,902.72	5.2
Newhawk Gold	1991	S91-nn	7	1,306.30	3.6
Placer Dome	1992	SG92-nn	23	5,577.34	15.2
Falconbridge	2005, 2006	MC-05-nn, MQ-05-nn, IF-05-nn	7	1,648.09	4.5
Seabridge	2006, 2008, 2009	S-06-nn, S-08-nn, S-09-nn, MW-09-nna	19	6,918.89	18.9
Seabridge	2010	S-10-nn	18	6,538.90	17.9
Seabridge	2011	S-10-nn	41	12,709.00	34.7
Total	**n/a**	**n/a**	**126**	**36,601.24**	**100.0**

Table 10.5 Mitchell Zone Drill Hole Summary by Company

Company	Year Drilled	Hole Pre-fix	No. Holes	No. Meters	% of Total
Newhawk Gold	1991	S91-nnn	4	647.30	1.1
Falconbridge	2005	NM-05-nn, WM-05-nn	4	1,197.29	2.1
Seabridge	2006	M-06-nnn	24	7,505.80	13.2
Seabridge	2007	M-07-nnn	37	15,650.32	27.5
Seabridge	2008	M-08-nnn	34	15,415.75	27.1
Seabridge	2009	M-09-nnn, MW-09-nnA	24	7,720.89	13.6
Seabridge	2010	M-10-nnn, KC10-nn	11	3,186.11	5.6
Seabridge	2011	M-11-nn	16	5,628.06	9.9
Total	**n/a**	**n/a**	**154**	**56,951.52**	**100.0**

Table 10.6 Iron Cap Zone Drill Hole Summary by Company

Company	Year Drilled	Hole Pre-fix	Drill Holes	Drilling (m)	% of Total
Esso Minerals	1980	S80-nn	5	1,051.26	5.9
Falconbridge	2005	IC-05-nn	5	1,246.60	7.0
Seabridge	2009	MW-09-nnA	1	91.62	0.5
Seabridge	2010	IC-10-nnn	41	15,400.65	86.6
Total	**n/a**	**n/a**	**52**	**17,790.13**	**100.0**

Figure 10.1 is a drill hole collar plan map for the entire KSM Project showing the areal distribution of drilling (collars shown as red dots) relative to the four mineralized areas.

Figure 10.1 KSM Drill Hole Locations

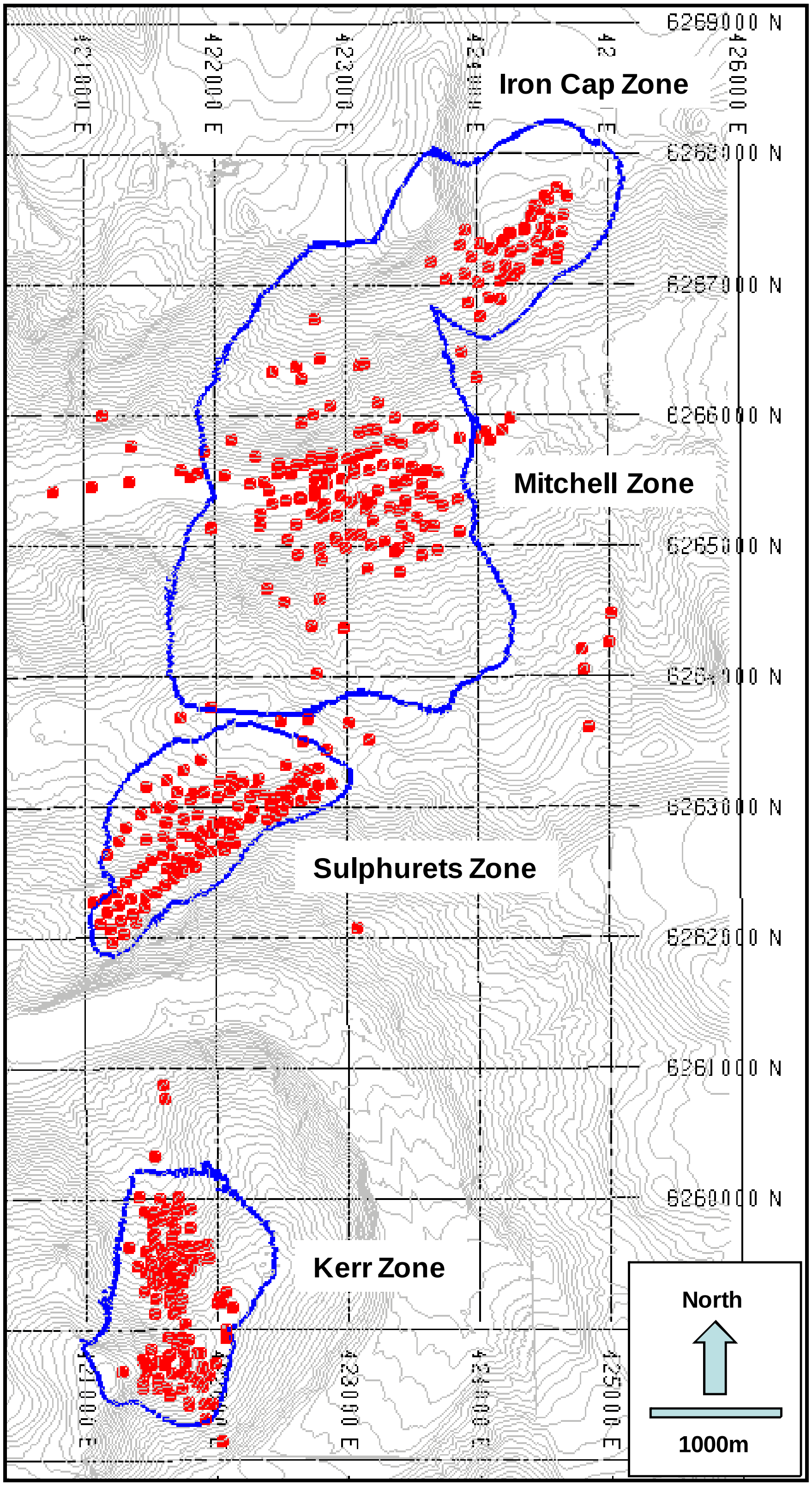


Figure 10.2 through Figure 10.5 are drill hole collar maps for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively, which show the collar location and hole trace in red. The MMTS 2010 PEA pits are shown in blue. For reference purposes, each drill hole collar map contains a reference line of section for drill hole and block model cross sections shown in Section 14.8.

Figure 10.2 Kerr Zone Drill Hole Locations

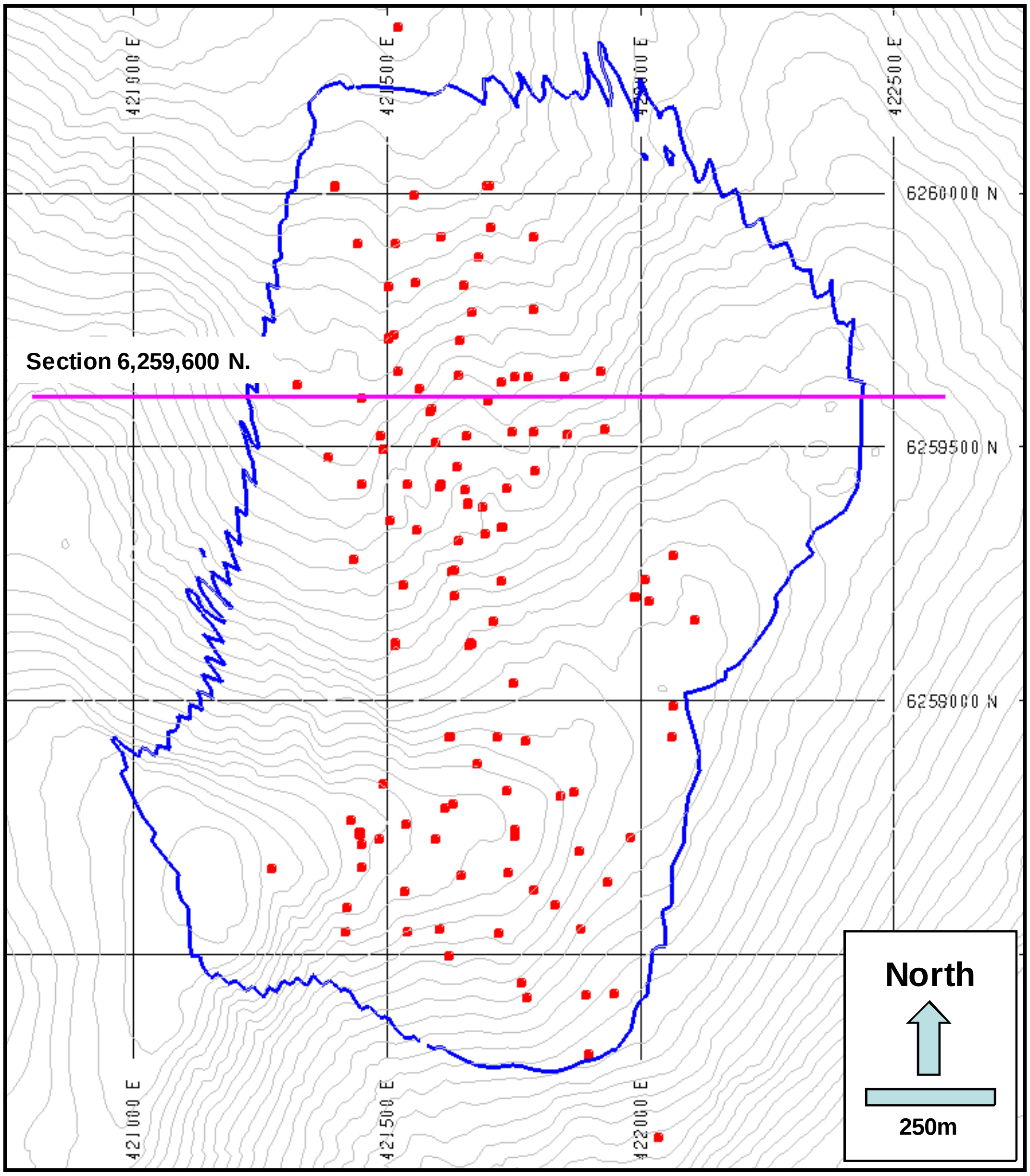


Figure 10.3 Sulphurets Zone Drill Hole Locations

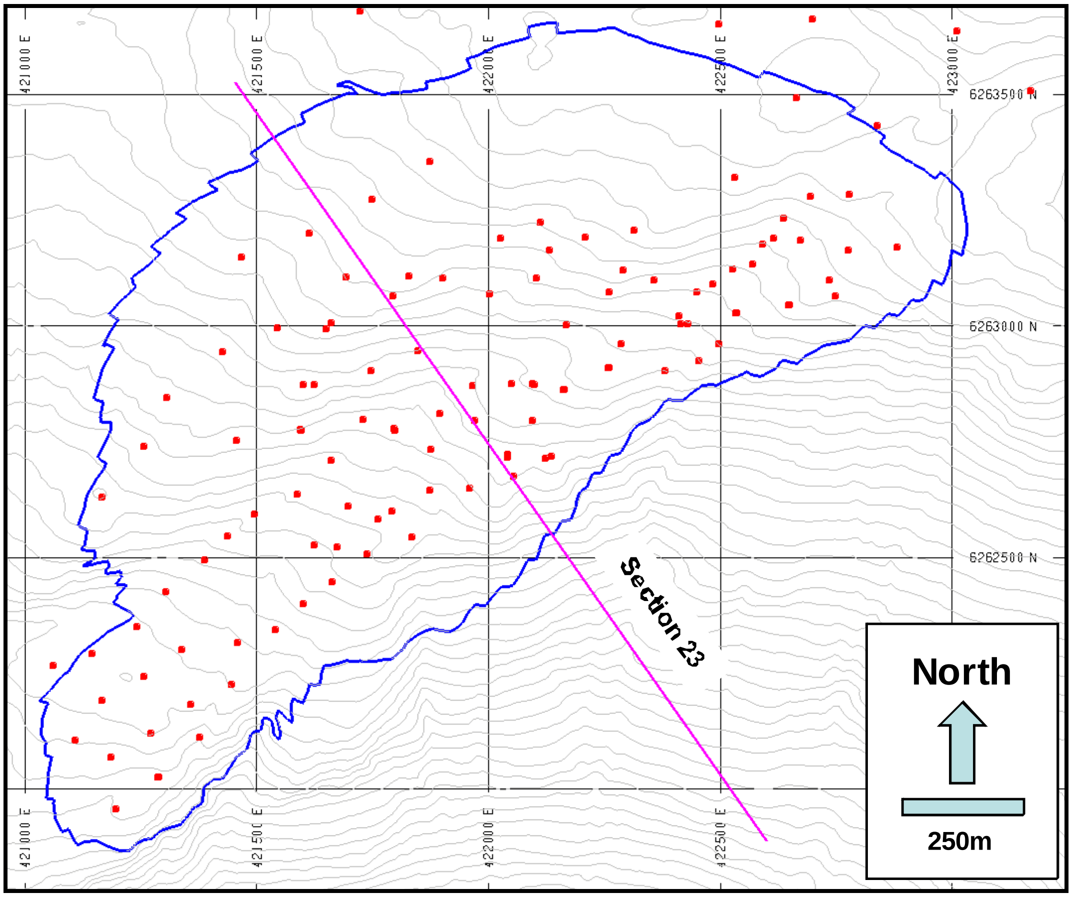


Figure 10.4 Mitchell Zone Drill Hole Locations

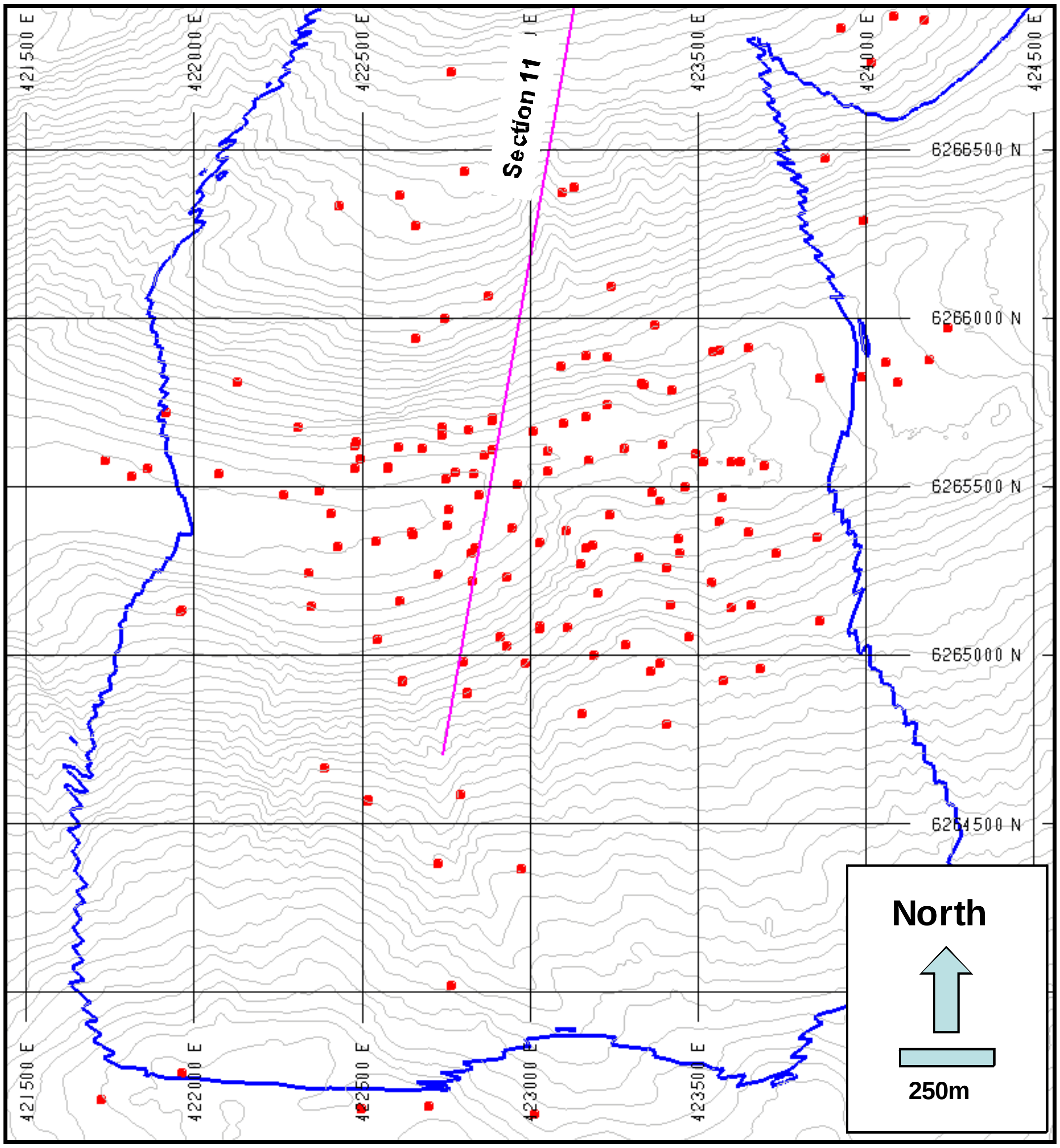
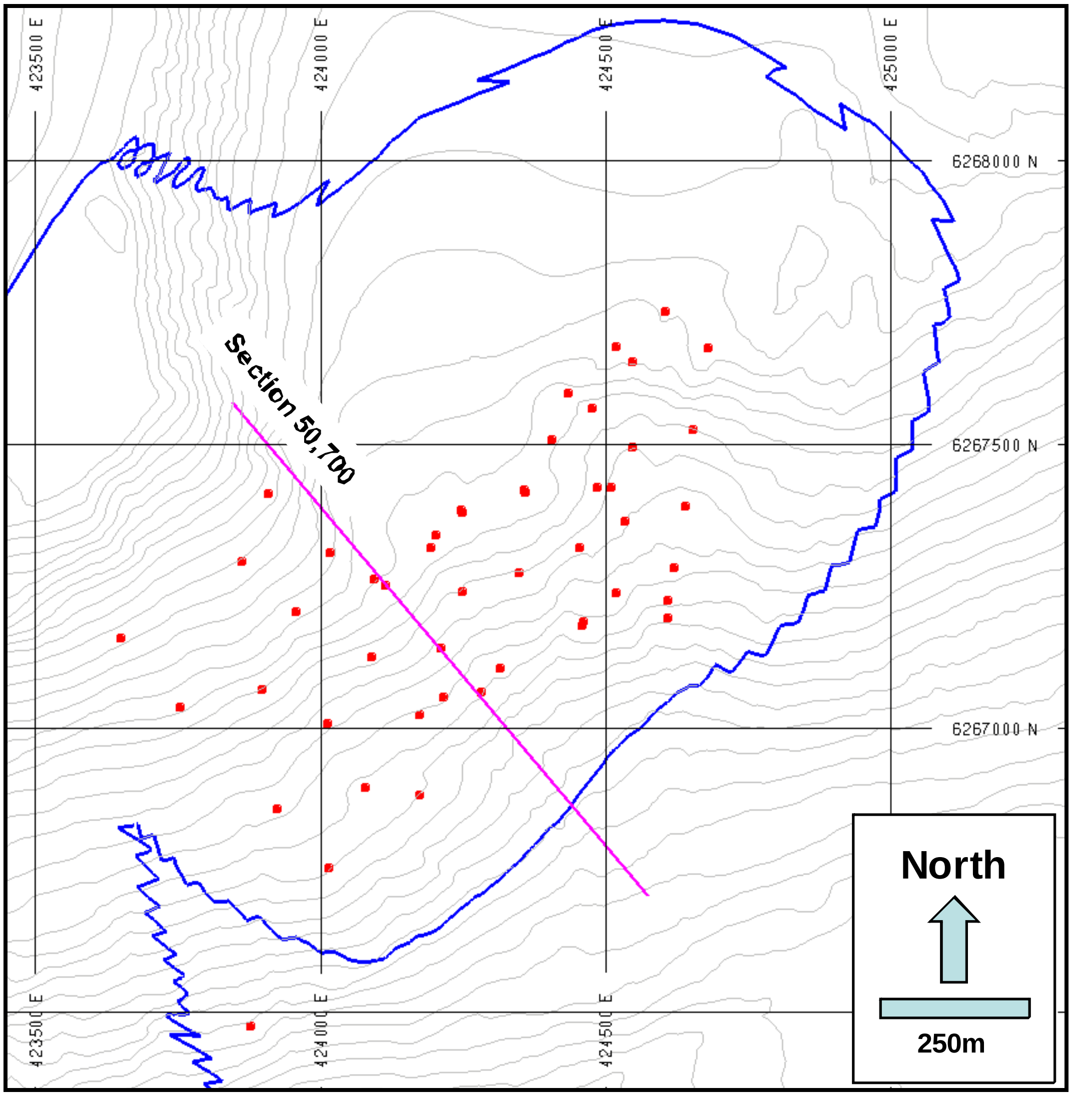


Figure 10.5 Iron Cap Zone Drill Hole Locations



10.3 Drill Core Processing

The following section was taken directly from RMI's April 6, 2007 NI 43-101 report entitled "Mitchell Creek Technical Report, Northern British Columbia" and edited to conform with protocol used in 2011:

> *"Drill core was placed into wooden trays directly upon emptying the core tube at the drill site. A wooden block marked with the hole depth in meters was placed in the core trays upon the completion of each drill run, which in good conditions was three meters. Core tubes and rods were in metric lengths. The core boxes were covered with a plywood lid which was securely nailed to the core box and placed in a metal basket.*

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 12.3. The data were recorded on paper logging sheets which were 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-meter-long intervals or at geological contacts so that each sample was approximately 2 meters maximum length. Sample lengths of 2 meters followed Falconbridge Ltd.'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 was stapled to the core box adjacent to the wax marking. Duplicates of the paper tag with the identification number were also placed were placed on the sample bag that was sent to the assay lab. A third copy 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. Where necessary, 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".

10.4 Relationship between Drill Hole and Mineralization Orientation

At Mitchell, 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 Mitchell mineralization 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 practical. At Mitchell, there is sufficient drilling to conclude that the deposit is aligned along this orientation. However, in a gross sense, the zone has a cylindrical geometry that plunges at about -45° to the northwest. Thus drilled intervals may be slightly oblique to the mineralization trend and may not accurately reflect true thicknesses, although most holes did not completely penetrate the mineralized zone.

At Sulphurets, the historical and current drilling orientation is along an azimuth of 145° inclined at -60°. The general northeasterly strike here appears to reflect a strong stratigraphic control. The strong deformation and schistosity present at Mitchell is not as prevalent at Sulphurets, likely due to the weaker degree and extent of late phyllic alteration, and there is no apparent alienation along the same trend. The plunge direction of -45° to the northwest observed at Mitchell also seems to define the orientation of higher grade zones and breccias within Sulphurets. In general, the drilled intervals of mineralization here are believed to be closer to representing true thicknesses.

Similar to Mitchell, extensive stockwork controlled disseminated mineralization is also found at Kerr along with strong phyllic-argillic alteration. However the associated schistosity dips moderately to the west. The geometry of the deposit is strongly lineated along this trend, which the preferred historical and current drill direction (dipping moderately east) was designed to test. Here the drilled intervals of mineralization are believed to closely indicate the true thickness.

At Iron Cap, mapping, surface sampling, and drilling prior to 2010 had established two dominant structural trends that influenced orientation of mineralization. A regional foliation as observed at other zones is also evident at Iron Cap, which is generally striking from 090° to 120° and dipping moderately to steeply north and north-northwest, and has attenuated pre-existing mineralization. There are several recognized veinlet and fracture orientations. However, the dominant one is a later feature, which controls several centimetres to multi-metre scaled quartz-sulfide veins and trends from 020° to 040° and dips steeply to the west. Stratigraphic bedding is obliterated within the Iron Cap Zone, but beyond the most intense alteration and within the Iron Cap panel, it generally strikes east-west and dips north. Based on these observations, and knowledge of mineralization orientations at Kerr, Sulphurets, and Mitchell, it was determined that drilling inclined holes at an azimuth of 135° would satisfactorily test all of the dominant structural trends with the least bias. In addition, several holes were drilled at a variety of other azimuths and inclinations to test for possible directional bias.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 SAMPLING METHOD AND APPROACH

11.1.1 SAMPLE LENGTH

The 2011 drill core was sawn in half longitudinally into primarily 2 m-long samples, which were then shipped off site where they were assayed for gold, copper, and other metals. Of the 10,071 samples that were collected, 15% were less than 2 m long, 77% were exactly 2 m long, and 8% were longer than 2 m. In 2011, approximately 95% of drilled NQ meterage was assayed. The 63 exploration and geotechnical holes that were drilled in 2011 and used for resource estimation averaged about 329 m in length. After completing the 2011 drilling campaign, the Kerr deposit has been drilled on roughly 50 to 75 m centres over an area which measures about 1,700 m in the north-south direction and 250 m in the east-west direction. The Sulphurets Zone has been drilled to about 50 to 100 m centres over an area measuring about 1,000 m (northeast-southwest) by 250 m (northwest-southeast). The Mitchell Zone has been drilled to roughly 50 to 100 m centres over an area measuring 1,400 m (east-west) by 900 m (north-south). There are areas of wider and closer spaced drilling in each deposit primarily driven by difficulty in constructing drilling platforms in steep terrain. The Iron Cap Zone has been drilled on roughly 50 to 100 m centres covering an area measuring 1,500 m by 600 m.

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

11.1.2 DRILLING CONDITIONS

Drilling conditions were generally good. Overburden was not excessive and rock quality was typically high except in isolated fractured or sheared zones where the rock easily broke along foliation planes. Overall average RQD for the 2011 drilling campaign was about 70% and core recovery averaged about 94%. RQD tended to be poorer for the Kerr and Sulphurets zones, where the average RQDs were 76% and 66%, respectively. Core recovery for the 2011 drilling at Kerr, Sulphurets, and Mitchell zones in 2011 was 91%, 94%, and 95%, respectively.

11.1.3 SAMPLE QUALITY

As a result of strict adherence to the drilling procedures and sampling methods previously described, sample quality and representation are considered good to very good. Core recovery rates improved in 2011 with only 4%, 1%, and 4%of the Kerr, Sulphurets, and Mitchell intervals having recoveries less than 50%, respectively.

11.1.4 GEOLOGY AND GEOLOGICAL CONTROLS

The following sections were taken directly from a prior RMI report (Lechner, 2007) and are still relevant regarding geologic 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 meters. Grades diminish slowly over large distances; sub-economic grades are encountered at distances of several hundreds of meters beyond the interpreted center 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, meter 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".*

11.1.5 Lithological and Alteration Coding

"In 2006, Seabridge adopted lithological and alteration descriptions from Fowler and Wells (1995), which distinguished rocks above the Sulphurets Thrust fault 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 11.1 and Table 11.2, 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".

"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 11.1 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

table continues…

Lithologic Code	Lithology
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 11.2 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)
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% qtz veinlets)
QTVN	Late Quartz Veins
SI	Silica Flooding - Qt,Py,Cp (Tour,Ser) (Porphyry)
SIH	Silicification due to Hornfelsing - Qt, Py
SIL	Pervasive silicification

11.1.6 Relevant Sample Composites

Table 11.3 through Table 11.6 show relevant composited drill hole grades for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. The relevant composites reflect continuous down-hole intersections of material above a 0.50 g/t gold equivalent cut-off grade in excess of 50 m in length. Gold and copper prices of US$650/oz and US$2.00/lb along with gold and copper recoveries of 70% and 85%, respectively, were used to determine the gold equivalent cut-off grade. The composited lengths shown in Table 11.3 through Table 11.6 are not necessarily "true widths" of mineralization although they represent significant zones of mineralization typical of large scale low-grade deposits.

The relevant composited data shown in Table 11.3 through Table 11.6 were sorted by decreasing gold equivalent grade (AuEQV). The average depths, average continuous mineralized lengths, and average gold/copper/gold equivalent grades are shown at the bottom (right hand side) of each table.

Table 11.3 Relevant Kerr Drill Hole Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
K88-011	51.00	163.85	112.85	0.41	1.31	3.68
K88-021	162.10	213.05	50.95	0.54	1.17	3.53
K88-001	176.17	228.25	52.08	0.30	1.22	3.37
KS92-135	28.96	92.05	63.09	0.40	1.15	3.32
KS-073	16.80	125.50	108.70	0.32	1.16	3.28
KS-086	26.30	77.40	51.10	0.53	1.05	3.22
K-11-15	56.20	139.75	83.55	0.50	1.03	3.03
KS-087	138.74	195.95	57.21	0.59	0.97	3.02
KS-075	23.20	149.40	126.20	0.29	0.98	2.79
K89-007	70.30	138.10	67.80	0.37	0.91	2.69
KS-082	27.40	87.90	60.50	0.22	0.95	2.65
KS-091	3.00	72.60	69.60	0.56	0.81	2.62
KS-066	76.20	146.00	69.80	0.37	0.87	2.61
KS-071	111.00	177.00	66.00	0.39	0.87	2.61
KS-094	298.50	382.30	83.80	0.35	0.86	2.56
K89-006	57.20	114.00	56.80	0.32	0.92	2.45
KS92-138	55.78	135.67	79.89	0.38	0.79	2.41
KS-124	259.00	331.00	72.00	0.56	0.71	2.39
K89-005	53.85	127.70	73.85	0.32	0.79	2.34
K89-010	101.00	178.05	77.05	0.20	0.80	2.26
KS-067	135.00	185.30	50.30	0.43	0.71	2.25
K88-018	3.05	75.85	72.80	0.36	0.72	2.20
KS92-141	59.00	110.30	51.30	0.30	0.72	2.16
KS-123	124.05	238.50	114.45	0.35	0.70	2.13
K88-016	3.05	106.07	103.02	0.28	0.53	1.65
KS-127	158.80	209.40	50.60	0.25	0.52	1.57
KS-089	12.10	70.40	58.30	0.30	0.49	1.56
KS92-143	115.50	186.50	71.00	0.34	0.45	1.51
K-10-06	105.50	160.70	55.20	0.31	0.48	1.49
KS-094	21.90	75.80	53.90	0.26	0.48	1.48
KS-127	33.50	151.80	118.30	0.21	0.49	1.47
KS-077	148.90	256.00	107.10	0.18	0.49	1.44
KS-119	136.00	231.66	95.66	0.20	0.48	1.42
KS92-136	95.10	160.00	64.90	0.20	0.48	1.42
K89-004	94.00	239.88	145.88	0.20	0.47	1.42
KS-123	24.00	108.81	84.81	0.24	0.45	1.40
K-11-11	185.00	243.00	58.00	0.30	0.42	1.37
KS-111	3.05	69.00	65.95	0.20	0.45	1.35
KS-127	212.45	268.90	56.45	0.26	0.42	1.33
T89-008	82.00	175.00	93.00	0.21	0.43	1.32
KS-112	5.18	75.80	70.62	0.18	0.44	1.30
K-09-02	93.00	201.00	108.00	0.25	0.41	1.29
K-10-06	29.90	99.00	69.10	0.18	0.43	1.28
KS-081	70.00	143.30	73.30	0.22	0.41	1.27
KS-108	64.00	134.11	70.11	0.19	0.41	1.25
T89-014	140.00	203.00	63.00	0.28	0.38	1.24
KS-124	79.00	253.00	174.00	0.22	0.40	1.24
KS-067	12.30	87.00	74.70	0.27	0.37	1.22

table continues...

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
KS-125	130.15	212.25	82.10	0.33	0.69	2.10
K88-015	130.00	196.00	66.00	0.26	0.71	2.09
KS-067	188.70	256.30	67.60	0.39	0.66	2.08
K89-006	120.00	187.22	67.22	0.30	0.66	2.00
T89-011	213.00	319.40	106.40	0.27	0.66	1.97
KS-076	8.60	90.00	81.40	0.16	0.69	1.92
KS-106	57.30	128.20	70.90	0.19	0.66	1.87
KS-128	149.96	297.40	147.44	0.23	0.63	1.85
K89-003	58.00	136.40	78.40	0.28	0.61	1.84
KS-131	141.10	192.00	50.90	0.25	0.61	1.83
K-10-08	137.30	196.80	59.50	0.44	0.54	1.83
K89-002	20.75	101.19	80.44	0.37	0.56	1.82
KS-123	241.15	299.70	58.55	0.27	0.59	1.79
KS-120	38.40	93.57	55.17	0.24	0.60	1.78
T89-013	15.24	90.00	74.76	0.57	0.47	1.77
KS-125	262.40	324.90	62.50	0.31	0.57	1.77
KS-109	69.00	179.00	110.00	0.28	0.57	1.74
K-09-01	277.50	344.28	66.78	0.18	0.60	1.72
KS-089	173.60	255.00	81.40	0.29	0.56	1.72
K87-005	10.30	62.90	52.60	0.41	0.50	1.71
K-09-01	218.17	276.00	57.83	0.22	0.59	1.69
K-10-08	205.00	263.00	58.00	0.21	0.57	1.68

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
T89-011	105.00	210.00	105.00	0.21	0.38	1.18
KS-116	24.38	87.00	62.62	0.20	0.37	1.16
KS-117	3.70	78.33	74.63	0.21	0.37	1.15
K89-019	159.00	361.49	202.49	0.13	0.40	1.14
KS-116	144.00	218.00	74.00	0.14	0.39	1.13
KS-105	8.15	94.49	86.34	0.19	0.35	1.10
KS-126	80.60	141.80	61.20	0.20	0.35	1.09
T89-008	4.57	76.00	71.43	0.15	0.36	1.06
KS-131	43.00	105.00	62.00	0.19	0.34	1.06
KS-130	28.04	110.64	82.60	0.18	0.34	1.05
KS-115	159.90	215.00	55.10	0.15	0.34	1.02
KS92-139	3.66	54.56	50.90	0.19	0.32	1.01
KS-121	89.70	162.46	72.76	0.18	0.32	1.01
KS-104	36.30	87.50	51.20	0.15	0.33	1.00
K88-022	2.74	55.00	52.26	0.17	0.32	1.00
KS-107	57.91	114.40	56.49	0.16	0.32	0.99
K89-019	105.00	156.00	51.00	0.20	0.29	0.95
KS-121	165.50	218.10	52.60	0.14	0.31	0.94
KS-116	239.30	302.05	62.75	0.15	0.28	0.88
KS-088	102.80	169.60	66.80	0.15	0.28	0.88
KS-122	197.00	251.00	54.00	0.18	0.27	0.86
Average	**95.21**	**171.04**	**75.83**	**0.27**	**0.59**	**1.77**

Table 11.4 Relevant Sulphurets Drill Hole Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
SG92-02	84.00	166.60	82.60	1.32	0.86	3.16
S-11-35	459.00	530.90	71.90	1.00	0.69	2.76
S-11-35	531.60	586.00	54.40	0.79	0.64	2.44
S-09-10	400.42	494.60	94.18	0.58	0.71	2.41
S91-389	71.10	166.70	95.60	0.70	0.64	2.34
S-11-43	355.40	427.58	72.18	0.85	0.58	2.33
S-10-17	290.00	359.00	69.00	1.04	0.49	2.31
S-11-35	370.00	457.00	87.00	1.03	0.47	2.23
S-10-21	287.00	351.70	64.70	0.70	0.59	2.22
S-06-04	188.00	310.00	122.00	0.80	0.52	2.14
S-09-11	183.00	354.00	171.00	0.73	0.55	2.12
MW-09-07A	125.85	182.50	56.65	0.68	0.55	2.10
S-11-45	297.50	359.70	62.20	0.87	0.47	2.08
S-08-08	274.00	344.20	70.20	0.89	0.46	2.06
S-09-10	326.00	399.55	73.55	0.73	0.51	2.03
S-11-50	438.00	489.50	51.50	0.85	0.50	2.03
S-11-43	273.00	354.55	81.55	0.95	0.42	2.02
SG92-07	232.00	291.69	59.69	0.69	0.51	2.00
S-09-14	131.40	263.50	132.10	0.76	0.48	1.98
S-10-22	171.00	249.00	78.00	0.60	0.54	1.98
S-09-13	75.00	140.00	65.00	0.75	0.47	1.94
SG92-15	116.13	190.40	74.27	1.57	0.15	1.94
S-10-23	304.00	406.50	102.50	0.74	0.38	1.72
S81-23	4.80	62.08	57.28	1.37	0.14	1.72
SG92-12	131.00	261.00	130.00	0.58	0.44	1.70
SG92-10	218.00	292.60	74.60	0.57	0.41	1.62
S-09-15	107.00	204.05	97.05	0.55	0.42	1.61
S-11-34	5.00	69.60	64.60	1.28	0.13	1.60
SG92-23	159.70	224.27	64.57	0.54	0.40	1.56
S-10-18	29.20	113.00	83.80	0.64	0.33	1.47
SG92-04	19.00	87.00	68.00	0.51	0.36	1.43
S-11-47	277.00	336.80	59.80	0.64	0.31	1.43
S91-398	12.10	69.00	56.90	0.35	0.45	1.41
S-11-68	10.90	69.00	58.10	0.75	0.25	1.39
S-11-39	361.00	433.00	72.00	0.48	0.35	1.38
SG92-13	25.00	144.82	119.82	0.57	0.30	1.33
S-10-28	90.40	164.00	73.60	0.85	0.18	1.30
S-11-42	149.00	206.00	57.00	1.23	0.02	1.28
S81-24	3.00	60.40	57.40	1.02	0.09	1.25
S-11-51	165.00	222.00	57.00	0.46	0.30	1.24
S-10-17	158.00	218.00	60.00	0.46	0.28	1.15
S81-39	93.00	150.00	57.00	0.98	0.05	1.12
S91-391	97.90	182.40	84.50	0.67	0.11	0.95
MQ-05-01	172.00	222.00	50.00	0.23	0.27	0.91

table continues...

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
S-09-15	263.00	351.00	88.00	0.65	0.48	1.89
SG92-19	14.00	93.80	79.80	1.69	0.05	1.83
S91-388	52.90	104.30	51.40	0.55	0.50	1.77

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
S-10-30	113.00	168.00	55.00	0.30	0.23	0.87
S-10-20	53.00	103.00	50.00	0.47	0.14	0.83
MC-05-02	126.00	180.00	54.00	0.13	0.27	0.82
Average	**177.87**	**253.33**	**75.46**	**0.77**	**0.41**	**1.79**

Table 11.5 Relevant Mitchell Drill Hole Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
M-07-058	297.65	432.00	134.35	0.99	0.35	1.90
M-06-017	17.00	80.60	63.60	1.14	0.29	1.89
MW-09-06A	3.95	87.40	83.45	1.03	0.30	1.81
M-07-048	6.40	58.80	52.40	0.92	0.34	1.79
M-06-009	4.00	296.00	292.00	0.98	0.31	1.78
M-07-029	53.30	163.80	110.50	1.21	0.26	1.76
M-06-007	4.40	287.90	283.50	0.98	0.29	1.72
M-08-065	4.00	380.10	376.10	0.96	0.29	1.69
M-07-051	28.50	146.25	117.75	0.92	0.30	1.68
M-07-035	146.00	484.00	338.00	1.03	0.25	1.68
M-07-059	2.50	152.25	149.75	0.94	0.29	1.67
M-08-090	2.50	169.90	167.40	0.96	0.28	1.67
M-11-127	2.10	591.00	588.90	1.03	0.24	1.64
M-06-013	4.85	105.10	100.25	0.94	0.27	1.63
M-07-058	176.00	288.00	112.00	0.80	0.32	1.63

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
S91-386	0.00	153.70	153.70	0.73	0.17	1.18
M-08-086	545.12	718.00	172.88	0.60	0.23	1.17
M-08-086	200.00	306.00	106.00	0.22	1.22	1.17
M-06-024	110.00	356.80	246.80	0.66	0.20	1.16
M-07-026	377.90	472.72	94.82	0.71	0.18	1.16
M-07-052	98.30	210.31	112.01	0.74	0.16	1.16
M-06-001	5.70	306.00	300.30	0.81	0.13	1.15
M-08-086	43.74	120.00	76.26	0.71	0.17	1.14
M-08-092	53.00	314.00	261.00	0.68	0.18	1.14
M-08-090	173.24	597.00	423.76	0.55	0.23	1.14
M-08-077	135.05	271.00	135.95	0.65	0.19	1.13
S91-395	116.50	190.50	74.00	0.60	0.21	1.13
M-08-093	119.00	645.00	526.00	0.65	0.18	1.13
M-06-017	166.10	223.00	56.90	0.63	0.19	1.12
M-11-125	0.20	157.70	157.50	0.75	0.14	1.12

table continues...

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
M-11-123	0.00	213.73	213.73	1.03	0.25	1.62
M-07-054	3.50	446.00	442.50	0.92	0.27	1.62
M-08-076	9.58	238.78	229.20	0.98	0.25	1.61
M-11-124	3.90	465.15	461.25	0.95	0.26	1.60
M-07-049	0.00	396.85	396.85	1.12	0.22	1.59
M-06-017	99.00	164.60	65.60	0.89	0.27	1.58
M-08-077	15.00	133.40	118.40	0.91	0.26	1.57
M-07-055	121.60	177.65	56.05	0.88	0.26	1.56
M-06-011	3.70	297.00	293.30	0.85	0.27	1.54
M-07-045	300.90	630.00	329.10	1.05	0.21	1.53
M-07-025	9.00	465.00	456.00	0.84	0.27	1.52
M-08-086	336.00	544.14	208.14	0.89	0.28	1.51
M-07-035	516.00	574.30	58.30	0.79	0.28	1.50
M-07-055	6.10	101.57	95.47	0.94	0.22	1.50
M-06-013	107.60	248.00	140.40	0.79	0.27	1.48
M-07-024E	356.80	597.25	240.45	0.83	0.25	1.48
M-07-058	4.50	146.00	141.50	0.81	0.26	1.47
M-08-069	1.20	79.00	77.80	0.85	0.23	1.45
S91-395	0.00	114.10	114.10	0.74	0.28	1.45
M-11-125	158.95	794.00	635.05	0.85	0.23	1.45
M-08-067	471.00	714.00	243.00	0.69	0.30	1.45
M-11-126	3.01	441.00	437.99	0.86	0.23	1.44
S91-387	0.00	60.30	60.30	0.91	0.19	1.41
M-07-050	3.05	123.45	120.40	0.96	0.17	1.40
M-09-095	110.53	205.65	95.12	0.84	0.21	1.39
M-07-027	179.85	231.00	51.15	0.67	0.18	1.12
M-08-062	313.00	571.13	258.13	0.54	0.22	1.10
M-11-127	875.20	926.30	51.10	0.39	0.28	1.10
M-09-109	2.10	201.00	198.90	0.71	0.15	1.08
M-11-122	3.60	67.00	63.40	0.68	0.15	1.08
M-08-065	488.00	592.60	104.60	0.59	0.19	1.07
M-08-091	124.00	408.00	284.00	0.67	0.16	1.06
M-07-054	600.00	670.45	70.45	0.58	0.19	1.06
M-08-062	572.02	745.00	172.98	0.55	0.20	1.06
M-07-058	567.00	720.00	153.00	0.49	0.22	1.05
M-09-099	259.00	337.50	78.50	0.66	0.15	1.05
M-08-061	273.00	599.30	326.30	0.71	0.13	1.04
M-09-096	3.50	191.00	187.50	0.80	0.09	1.04
M-08-065	384.55	482.00	97.45	0.54	0.19	1.04
M-06-010	53.00	198.00	145.00	0.66	0.14	1.03
M-06-002	3.00	100.00	97.00	0.70	0.12	1.02
M-07-057	61.20	171.00	109.80	0.59	0.17	1.02
M-11-124	472.20	620.00	147.80	0.54	0.19	1.02
M-09-107	74.00	148.00	74.00	0.22	0.31	1.01
M-07-047	9.75	89.00	79.25	0.68	0.13	1.01
M-07-037	6.00	143.25	137.25	0.73	0.11	1.01
M-08-062	32.00	100.75	68.75	0.65	0.14	1.00
M-07-054	470.00	544.28	74.28	0.55	0.18	1.00
M-06-003	210.00	310.00	100.00	0.62	0.14	0.99
M-09-099	510.50	599.50	89.00	0.52	0.18	0.99

table continues...

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
M-08-094	2.50	195.00	192.50	0.80	0.23	1.39
M-06-014	269.00	453.00	184.00	0.92	0.18	1.38
M-08-067	384.00	469.00	85.00	0.48	0.62	1.37
M-06-006	6.00	221.70	215.70	0.91	0.20	1.37
M-08-079	313.00	399.00	86.00	0.19	0.45	1.34
M-08-069	81.00	586.00	505.00	0.78	0.22	1.34
M-11-126	516.00	636.00	120.00	0.75	0.23	1.33
M-06-008	34.00	346.00	312.00	0.83	0.20	1.33
M-06-002	102.00	426.00	324.00	0.85	0.19	1.33
M-07-047	198.00	410.15	212.15	0.61	0.27	1.30
M-07-026	24.00	376.20	352.20	0.82	0.19	1.30
M-07-056	4.57	257.50	252.93	0.88	0.16	1.30
M-08-073	91.00	374.00	283.00	0.78	0.20	1.29
M-11-122	298.25	636.00	337.75	0.73	0.22	1.29
M-08-070	172.00	225.00	53.00	0.68	0.23	1.28
M-06-003	5.00	208.00	203.00	0.85	0.16	1.27
WM-05-01	81.50	282.89	201.39	0.80	0.19	1.27
M-07-051	147.40	259.70	112.30	0.66	0.23	1.26
M-07-059	155.60	285.00	129.40	0.61	0.25	1.26
M-07-034	42.00	126.79	84.79	0.70	0.22	1.26
M-08-064	23.00	345.00	322.00	0.84	0.17	1.26
M-07-035	72.00	144.00	72.00	0.80	0.17	1.25
M-08-092	316.00	418.00	102.00	0.73	0.20	1.25
M-07-058	448.00	565.00	117.00	0.63	0.24	1.25
M-07-039	41.00	116.00	75.00	0.73	0.20	1.24
M-08-094	197.00	339.00	142.00	0.56	0.17	0.99
M-08-063	290.00	569.00	279.00	0.66	0.13	0.99
M-08-071	99.00	154.00	55.00	0.66	0.13	0.98
M-06-012	185.00	265.00	80.00	0.60	0.14	0.96
M-07-050	151.25	237.00	85.75	0.62	0.13	0.94
M-06-005	5.60	108.00	102.40	0.54	0.15	0.94
M-08-066	94.00	435.00	341.00	0.66	0.11	0.93
M-07-034	248.00	298.00	50.00	0.52	0.15	0.91
M-08-072	80.50	139.00	58.50	0.63	0.11	0.91
M-07-060	285.00	338.00	53.00	0.46	0.17	0.89
M-06-015	2.90	206.00	203.10	0.63	0.10	0.89
M-07-034	300.00	368.00	68.00	0.49	0.15	0.89
M-09-107	175.00	233.00	58.00	0.53	0.13	0.87
M-08-073	390.00	442.00	52.00	0.47	0.16	0.87
M-07-046	66.00	123.00	57.00	0.48	0.15	0.86
M-06-014	83.00	137.00	54.00	0.64	0.09	0.86
M-10-117	5.20	122.90	117.70	0.54	0.12	0.86
S91-387	61.40	123.90	62.50	0.51	0.14	0.86
M-07-048	343.80	394.39	50.59	0.47	0.15	0.86
M-07-044	466.00	553.00	87.00	0.57	0.11	0.86
M-07-057	257.20	308.00	50.80	0.46	0.15	0.85
M-08-076	320.09	408.00	87.91	0.49	0.14	0.85
M-07-057	173.00	250.75	77.75	0.51	0.13	0.85
M-09-099	177.00	257.00	80.00	0.52	0.13	0.84
M-07-034	190.00	246.00	56.00	0.48	0.14	0.84

table continues...

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
M-07-052	13.70	96.05	82.35	0.82	0.16	1.24
M-07-028	74.37	237.00	162.63	0.79	0.17	1.23
M-08-067	78.00	372.00	294.00	0.64	0.26	1.23
M-07-031	76.00	214.00	138.00	0.69	0.20	1.21
M-07-053	124.00	442.00	318.00	0.75	0.17	1.20
M-07-045	128.00	300.00	172.00	0.78	0.16	1.20
M-11-123	214.50	402.42	187.92	0.68	0.20	1.19
M-07-037	143.85	309.00	165.15	0.85	0.13	1.19
M-11-122	107.00	297.60	190.60	0.62	0.22	1.18
M-07-048	164.00	342.40	178.40	0.72	0.18	1.18
M-09-106	222.00	274.00	52.00	0.50	0.13	0.83
M-08-070	251.00	320.00	69.00	0.39	0.17	0.83
M-10-119	241.00	344.00	103.00	0.63	0.08	0.83
M-10-116	265.00	323.00	58.00	0.66	0.07	0.83
M-07-044	336.00	402.00	66.00	0.67	0.06	0.83
M-07-060	175.00	263.00	88.00	0.48	0.14	0.83
M-07-043	126.00	232.00	106.00	0.50	0.11	0.78
M-08-061	640.00	690.00	50.00	0.44	0.11	0.72
M-09-108	223.00	273.00	50.00	0.41	0.10	0.67
Average	**162.14**	**327.86**	**165.72**	**0.77**	**0.21**	**1.29**

Table 11.6 Relevant Iron Cap Drill Hole Grades

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
IC-10-033	136.00	199.00	63.00	2.64	0.32	2.31
IC-10-011	326.00	377.00	51.00	1.71	0.23	2.30
IC-10-030	2.60	82.00	79.40	0.63	0.59	2.12
IC-10-009	366.00	418.00	52.00	1.39	0.21	1.93
IC-10-029	152.00	251.00	99.00	1.01	0.37	1.91
IC-10-006	2.92	76.20	73.28	0.83	0.44	1.89
IC-10-040	4.20	165.50	161.30	0.89	0.37	1.85
IC-10-028	467.00	584.00	117.00	1.03	0.29	1.77
IC-10-035	105.00	351.00	246.00	0.74	0.25	1.31
IC-10-016	224.40	278.20	53.80	0.74	0.22	1.29
IC-10-045	126.00	184.00	58.00	0.26	0.41	1.27
IC-10-025	1.35	63.00	61.65	0.41	0.33	1.24
IC-10-017	173.00	224.40	51.40	0.23	0.39	1.22
IC-10-039	277.70	450.00	172.30	0.31	0.35	1.19
IC-05-02	146.90	238.00	91.10	0.62	0.21	1.15
IC-05-03	1.50	54.60	53.10	0.40	0.29	1.13

table continues...

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
IC-10-032	18.50	111.00	92.50	0.71	0.43	1.76
IC-10-025	100.00	196.00	96.00	1.04	0.23	1.62
IC-10-010	193.00	300.00	107.00	0.79	0.32	1.60
IC-05-01	3.30	91.30	88.00	0.88	0.26	1.48
IC-10-017	390.70	491.40	100.70	0.79	0.26	1.45
IC-10-008	107.00	215.00	108.00	0.78	0.26	1.43
IC-10-011	2.90	53.00	50.10	0.44	0.39	1.43
IC-10-033	3.40	67.00	63.60	0.31	0.43	1.41
IC-10-029	1.50	90.30	88.80	0.42	0.39	1.39
IC-10-027	189.00	260.50	71.50	0.55	0.33	1.39
IC-10-037	8.50	111.90	103.40	0.90	0.19	1.39
IC-10-035	2.60	103.00	100.40	0.70	0.26	1.36
IC-10-033	75.00	132.00	57.00	0.41	0.37	1.35
IC-10-024	143.00	194.00	51.00	0.35	0.41	1.35
IC-10-034	11.07	75.50	64.43	0.57	0.30	1.35
IC-10-031	117.00	275.00	158.00	0.84	0.20	1.34
IC-10-015	254.00	339.00	85.00	0.31	0.43	1.34
IC-10-007	15.50	66.00	50.50	0.72	0.24	1.33

Drill Hole	From Depth (m)	To Depth (m)	Composited Length (m)	Au (g/t)	Cu (%)	AuEQV (g/t)
IC-10-029	287.00	354.00	67.00	0.67	0.18	1.12
IC-10-032	159.00	224.00	65.00	0.40	0.28	1.11
IC-10-009	203.00	275.00	72.00	0.73	0.15	1.11
IC-10-037	228.50	280.00	51.50	0.53	0.23	1.11
IC-10-008	38.00	97.80	59.80	0.35	0.29	1.09
IC-10-031	345.80	426.00	80.20	0.45	0.24	1.06
IC-05-04	113.00	232.00	119.00	0.41	0.25	1.02
IC-10-013	326.00	378.00	52.00	0.23	0.29	0.96
IC-05-01	157.30	215.30	58.00	0.39	0.22	0.95
IC-05-02	74.90	138.90	64.00	0.40	0.21	0.93
IC-10-023	8.15	72.00	63.85	0.37	0.21	0.91
IC-10-016	50.00	106.00	56.00	0.36	0.21	0.90
S80-14	84.00	138.00	54.00	0.35	0.22	0.89
IC-10-019	2.80	54.00	51.20	0.34	0.21	0.87
IC-10-015	407.00	471.30	64.30	0.08	0.30	0.85
IC-10-026	46.00	106.00	60.00	0.27	0.22	0.82
IC-10-023	188.00	250.00	62.00	0.17	0.25	0.82
IC-05-01	105.30	155.30	50.00	0.33	0.16	0.75
Average	**134.08**	**214.26**	**80.18**	**0.65**	**0.29**	**1.36**

11.2 Sample Preparation, Analyses, and Security

This section describes Seabridge's sample security, sample preparation, and analytical methods that were used in 2011 for the KSM Project. These are essentially the same methods that have been described in previous RMI NI 43-101 reports dealing with the KSM Project (Lechner, 2007, 2008, 2009, and 2010).

11.2.1 Statement on Sample Preparation Personnel

All initial sample preparation (sawing and bagging) was conducted by labourers contracted from Tahltan Native Development Corporation, trained by and under the direct supervision of geologists employed by Seabridge. Drill core and quality control samples were shipped to Eco Tech's preparation facility located in Stewart, BC, and then shipped by Eco Tech to their assay laboratory located in Kamloops, BC, where the prepared samples were analyzed.

11.2.2 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 14″ diamond impregnated 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, the samples were placed into large polyweave (rice) shipping bags, six per bag (three for the larger HQ core). 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 secured with a uniquely numbered tie strap, and the number was 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 once every two days.

The sample shipment was placed inside the project-chartered helicopter, flown directly to the Granduc Road staging area, and unloaded by the pilot. At the staging area, the shipment was either stored and locked inside a metal bulk shipping container or transferred directly to a waiting truck. Trucking was contracted to Granmac Services Ltd. (Granmac) of Stewart, BC. The shipment was transported by truck to Stewart, where Eco Tech personnel unloaded the samples at the sample preparation facilities. The samples were occasionally taken directly to Stewart via helicopter, and then 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 had not been 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.

11.2.3 Analytical Procedures

At the Eco Tech facilities in Stewart, samples were sorted and dried (if necessary), crushed through a jaw crusher and cone or roll crusher to –10 mesh, then 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, then rolled to homogenize. The resulting pulp sample was 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 the 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 mixture of HCl, HNO_3, and, H_2O at a ratio of 3:1:2 that contained 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 ICP unit. Eco Tech's ICP detection limits (lower and upper) are summarized in Table 11.7.

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.

Table 11.7 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

11.2.4 Quality Control Measures

Seabridge implemented the same quality control procedures that they used for their previous KSM programs. Various standard reference material (SRM) sources have been used since 2006. They included blanks of material obtained from commercial landscaping materials (crushed marble and granite) and "barren" river gravels collected near Stewart, BC, along with different commercially certified standards of pre-packaged pulps. 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 randomly by the geologist during the logging process.

Two different blanks were used in 2011. Blank 5 and 6 were purchased in 20 kg bags from a home and garden retailer located in Terrace, BC. Blanks were submitted into the 2011 sample stream at a frequency of about one blank for every 32 samples. Approximately 310 barren samples or "blanks" were submitted to Eco Tech. Figure 11.1 and Figure 11.2 chart the performance of the gold and copper blanks for the 2011 drilling campaign.

Figure 11.1 2011 Au Blank Performance

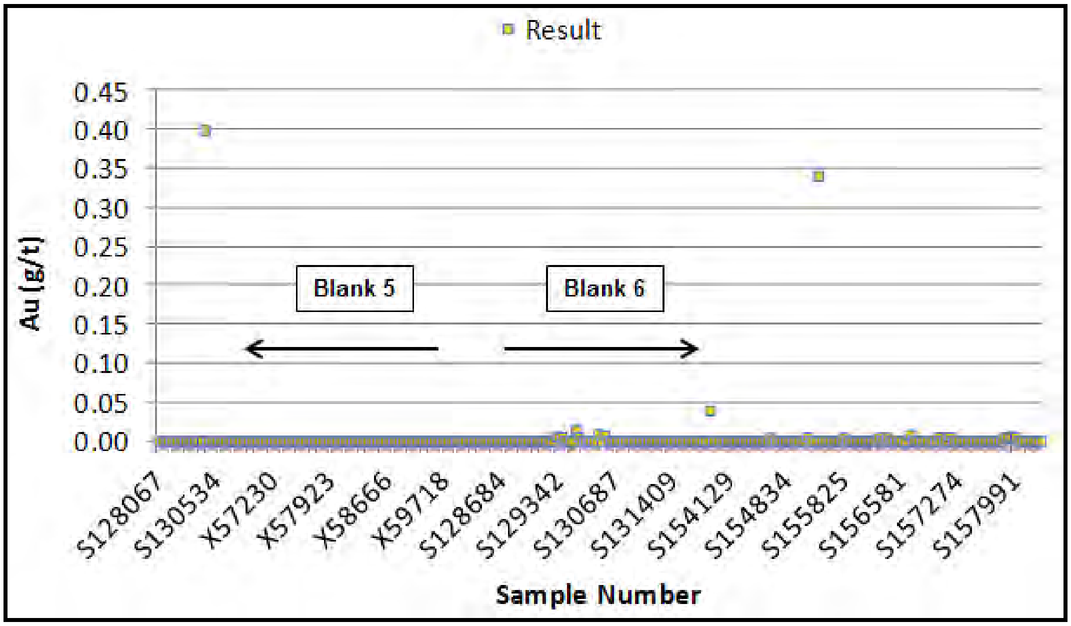
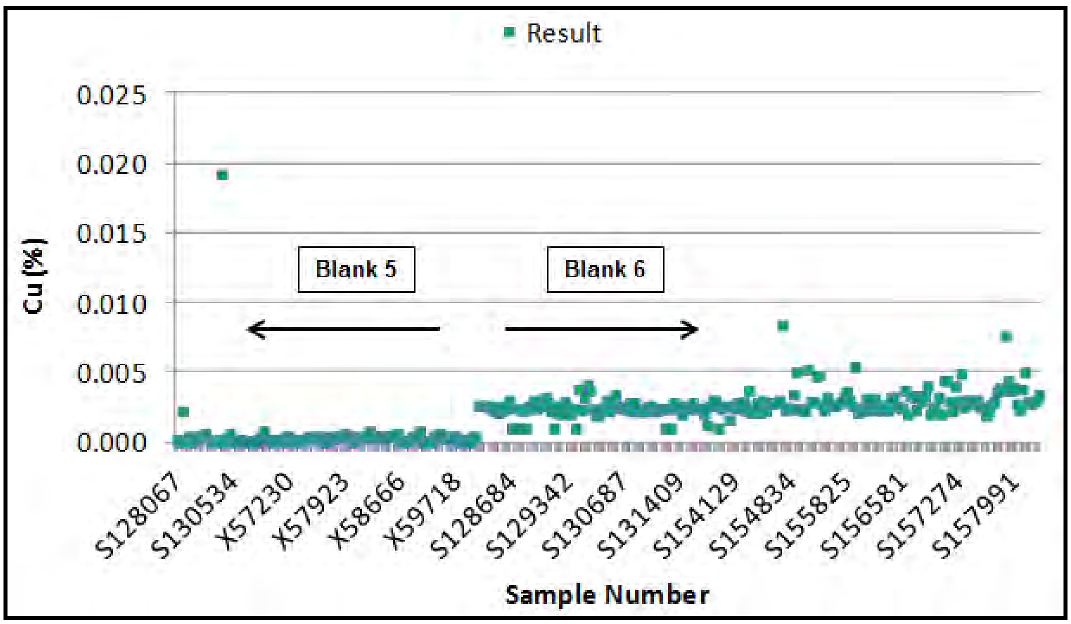


Figure 11.2 2011 Cu Blank Performance



Five of the seven pulp standards that were used by Seabridge for their 2011 drilling/sampling campaign were purchased from CDN Resource Laboratories Ltd. (CDN) out of Delta, BC. The CDN standards (CDN-CM-4, CDN-CM-11A, CGS-19, CGS-22, and CGS-27) were prepared from material that was collected from various granitic intrusives and gold-copper porphyry systems. Two standards (SEA-KSM and SEA-CL2) were prepared from a bulk sample of core collected from the Mitchell

Zone that had been used for crushing tests and felsic material from a Seabridge project located in the Northwest Territories. These last two standards were prepared and certified by Smee & Associates Consulting Ltd. from North Vancouver.

A total of 302 SRMs were inserted into the 2011 sample stream or a frequency of about one SRM for every 33 samples or 3% of the total assay samples. Table 11.8 summarizes the SRMs that were used by Seabridge for their 2011 drilling campaign. Table 11.8 shows the number of SRMs that were submitted, their expected values along with ±2 standard deviation units.

The performance of the various gold, copper, and molybdenum standards are graphed as a function of time (certificate number) in Figure 11.3 through Figure 11.18.

Table 11.8 2011 KSM Standard Reference Materials

Standard	Number Submitted	Gold Values (g/t)			Copper Values (%)			Molybdenum Values (%)		
		Expected	-2 Std Dev	+2 Std Dev	Expected	-2 Std Dev	+2 Std Dev	Expected	-2 Std Dev	+2 Std Dev
CM-4	15	1.18	1.06	1.30	0.508	0.483	0.533	0.032	0.028	0.036
CM-11A	38	1.01	0.91	1.12	0.332	0.332	0.344	0.038	0.034	0.042
CGS-19	35	0.74	0.67	0.81	0.132	0.122	0.142	n/a	n/a	n/a
CGS-22	50	0.64	0.58	0.70	0.725	0.697	0.753	n/a	n/a	n/a
CGS-27	50	0.43	0.39	0.48	0.379	0.364	0.425	n/a	n/a	n/a
SEA-CL2	56	2.07	1.89	2.26	n/a	n/a	n/a	n/a	n/a	n/a
SEA-KSM	58	0.77	0.71	0.84	0.204	0.194	0.214	0.007	0.006	0.008
Total	**302**	**n/a**	**n/a**	**n/a**	**n/a**	**n/a**	**n/a**	**n/a**	**n/a**	**n/a**

Note: Std Dev = standard deviation.

Figure 11.3 2011 Au Standard CM-4 Performance

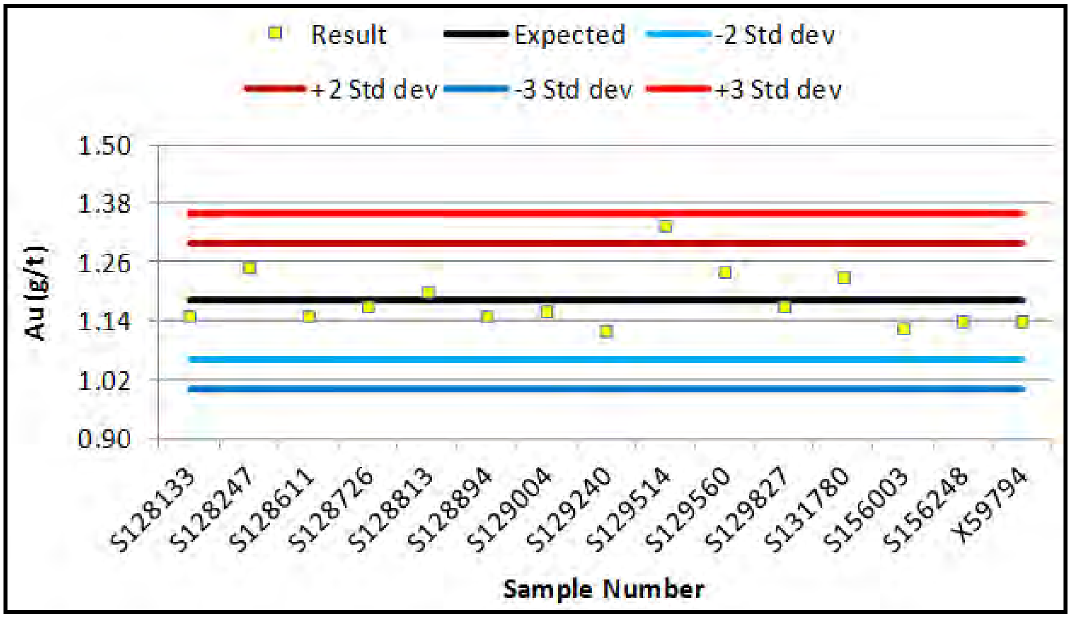


Figure 11.4 2011 Cu Standard CM-4 Performance

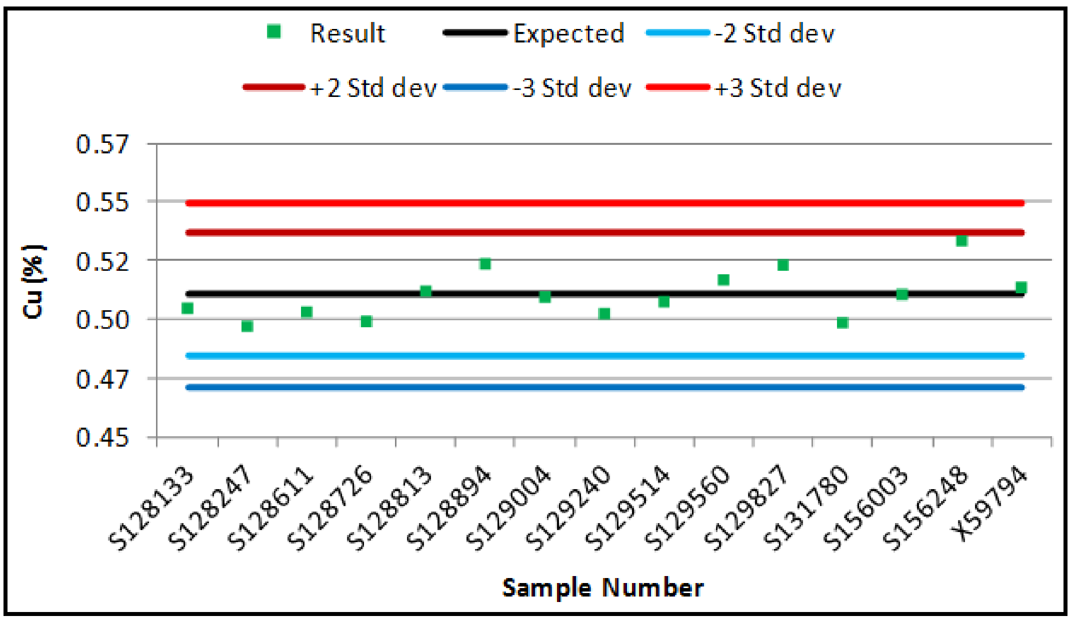


Figure 11.5 2011 Mo Standard CM-4 Performance

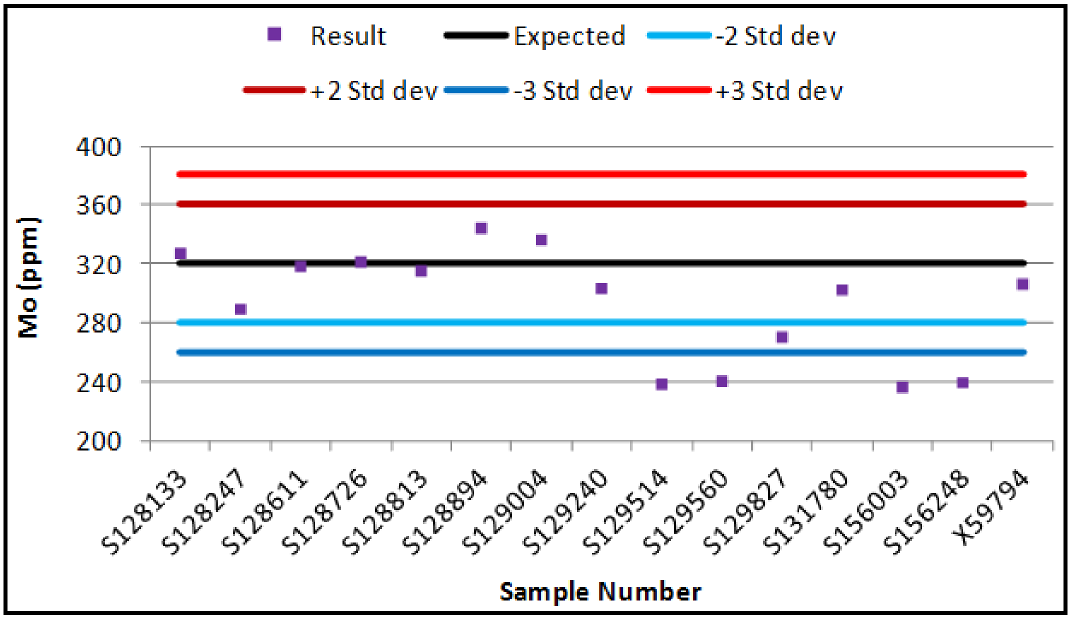


Figure 11.6 2011 Au Standard CM-11A Performance

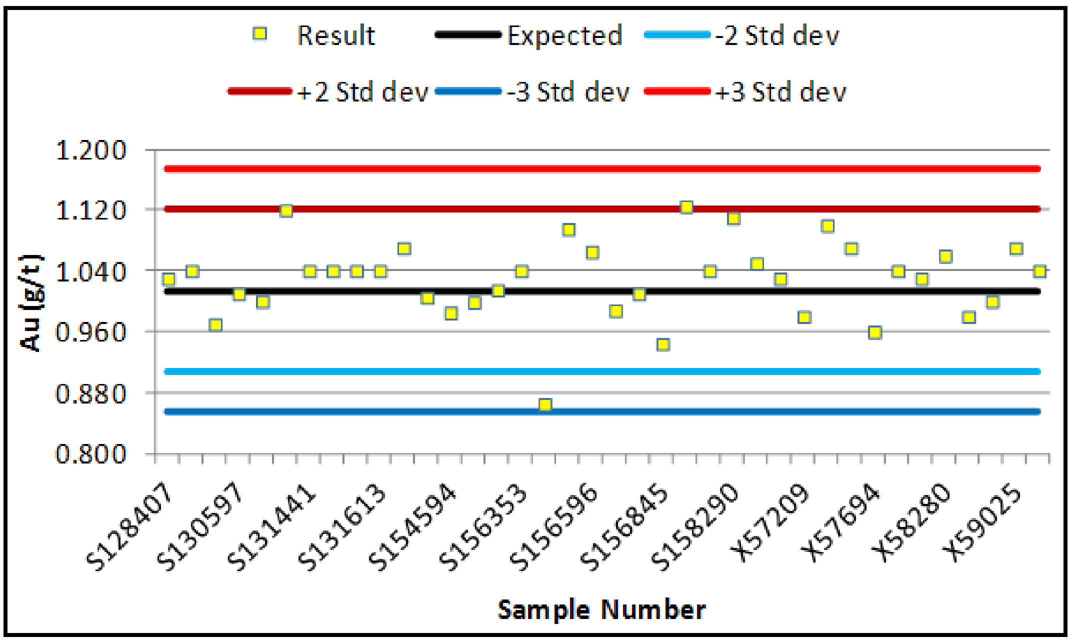


Figure 11.7 2011 Cu Standard CM-11A Performance

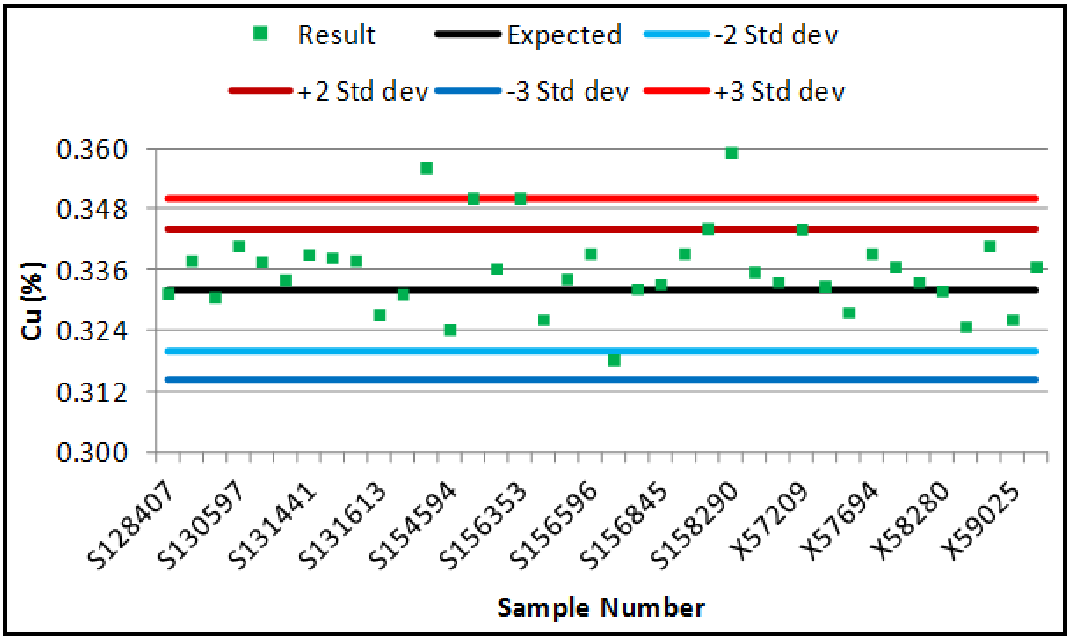


Figure 11.8 2011 Mo Standard CM-11A Performance

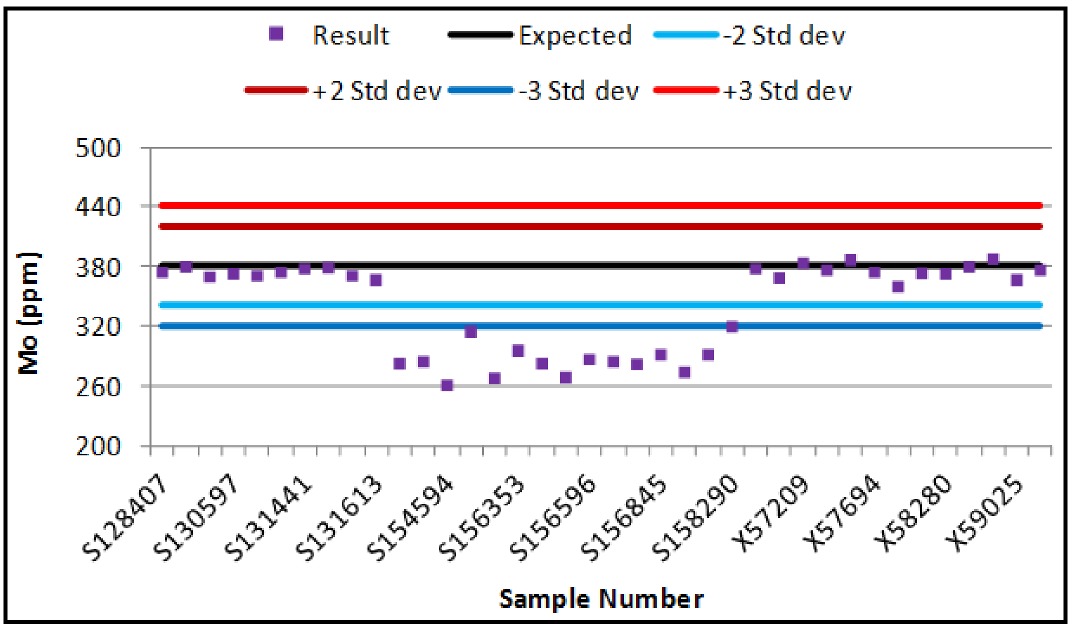


Figure 11.9 2011 Au Standard CGS-19 Performance

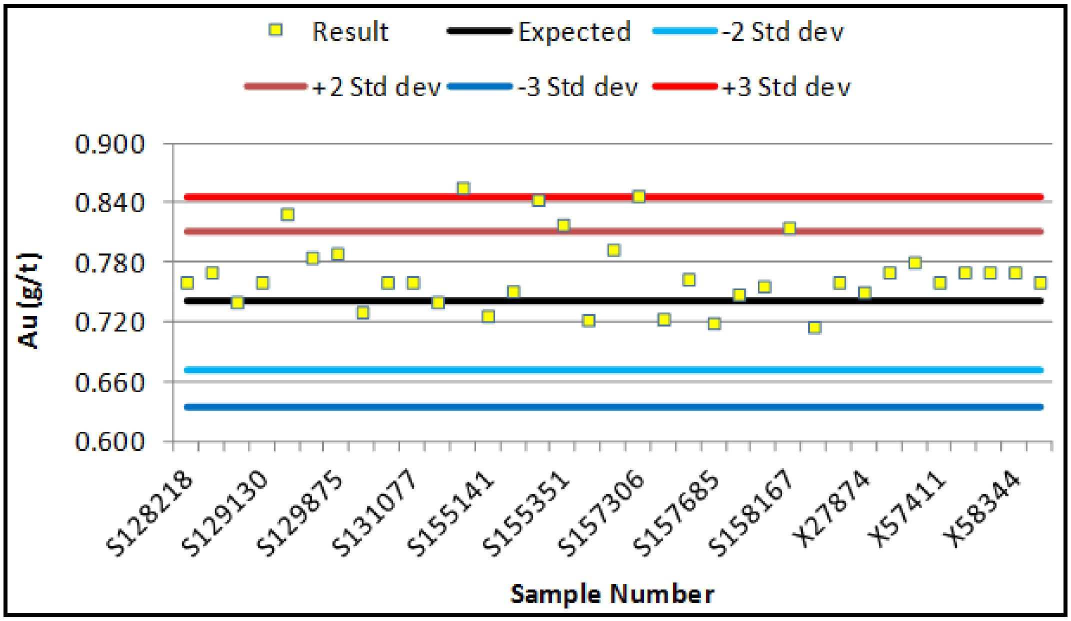


Figure 11.10 2011 Cu Standard CGS-19 Performance

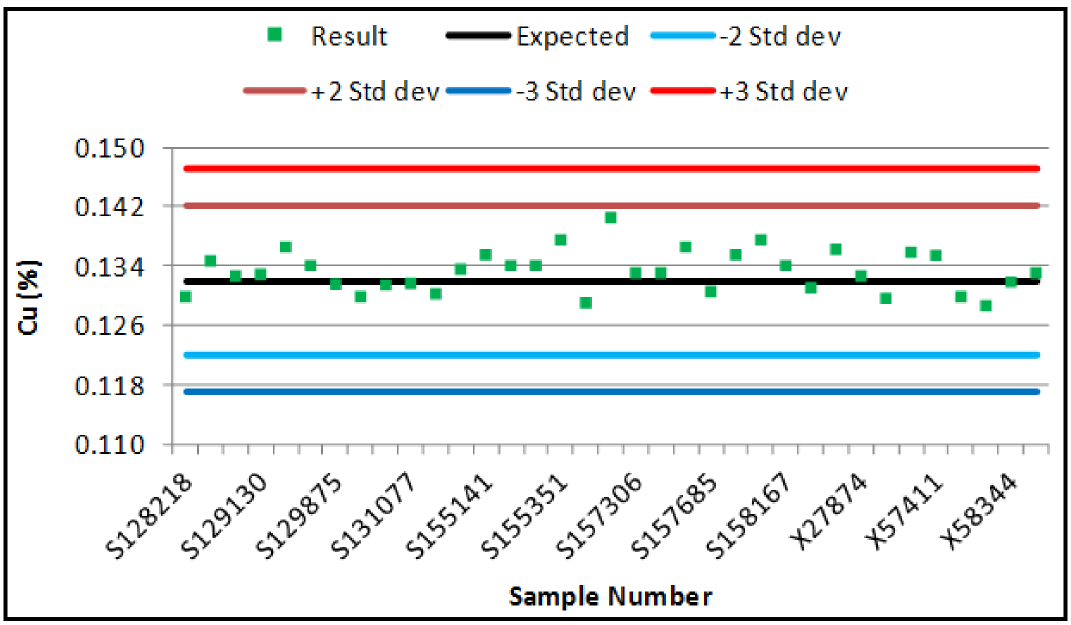


Figure 11.11 2011 Au Standard CGS-22 Performance

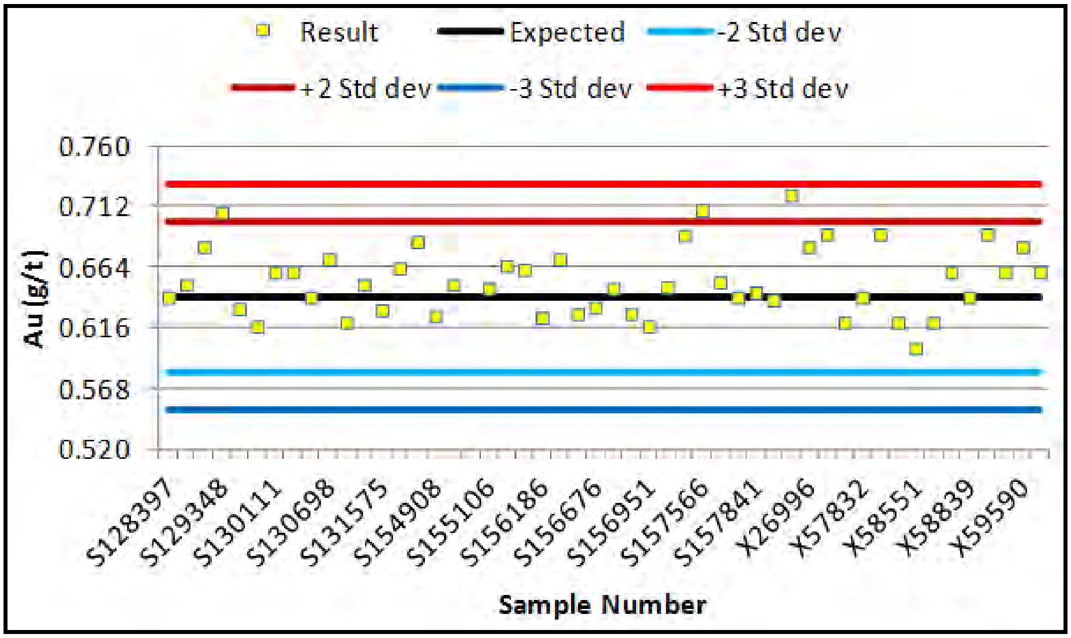


Figure 11.12 2011 Cu Standard CGS-22 Performance

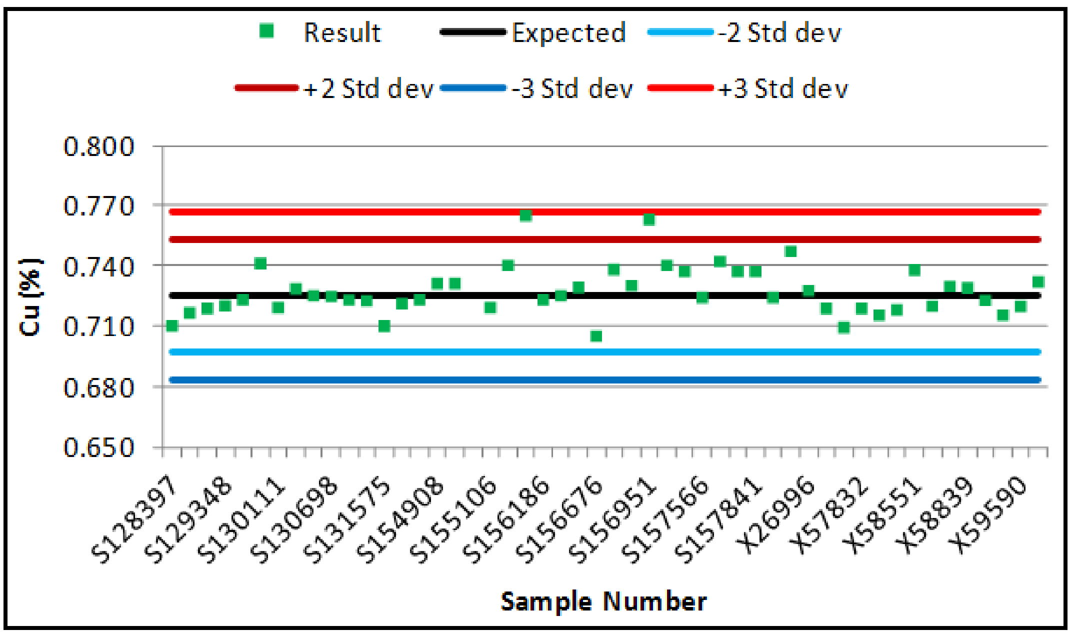


Figure 11.13 2011 Au Standard CGS-27 Performance

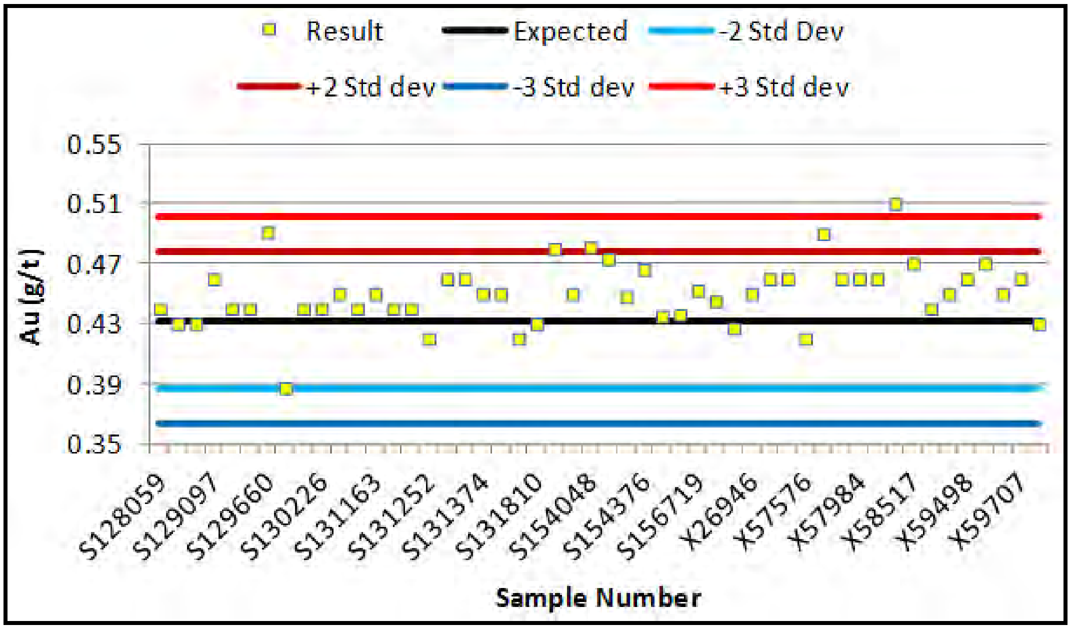


Figure 11.14 2011 Cu Standard CGS-27 Performance

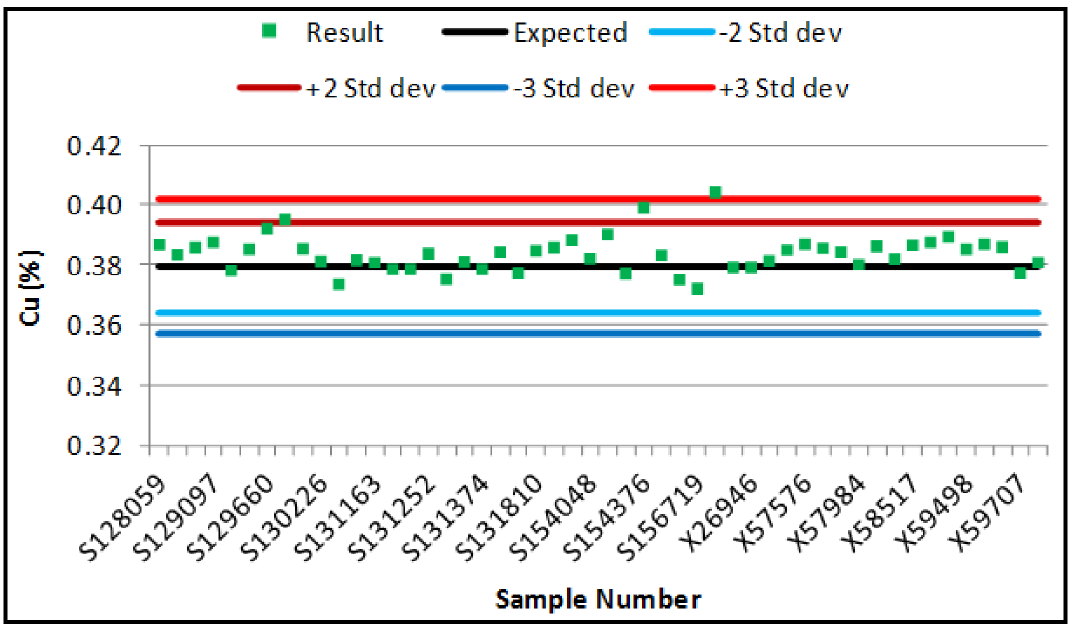


Figure 11.15 2011 Au Standard SEA-CL2 Performance

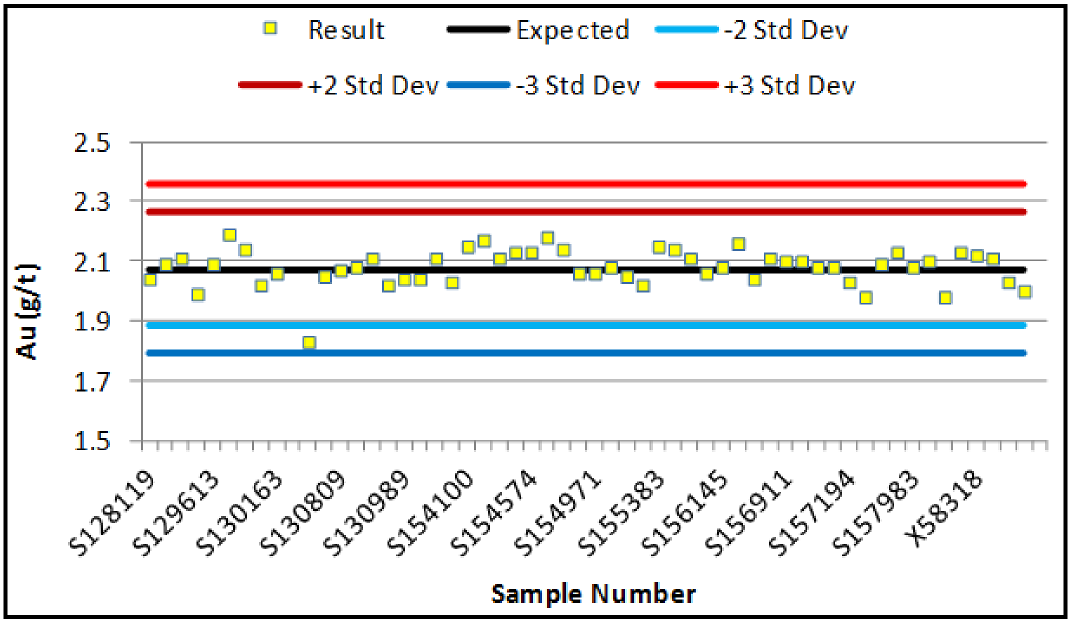


Figure 11.16 2011 Au Standard SEA-KSM Performance

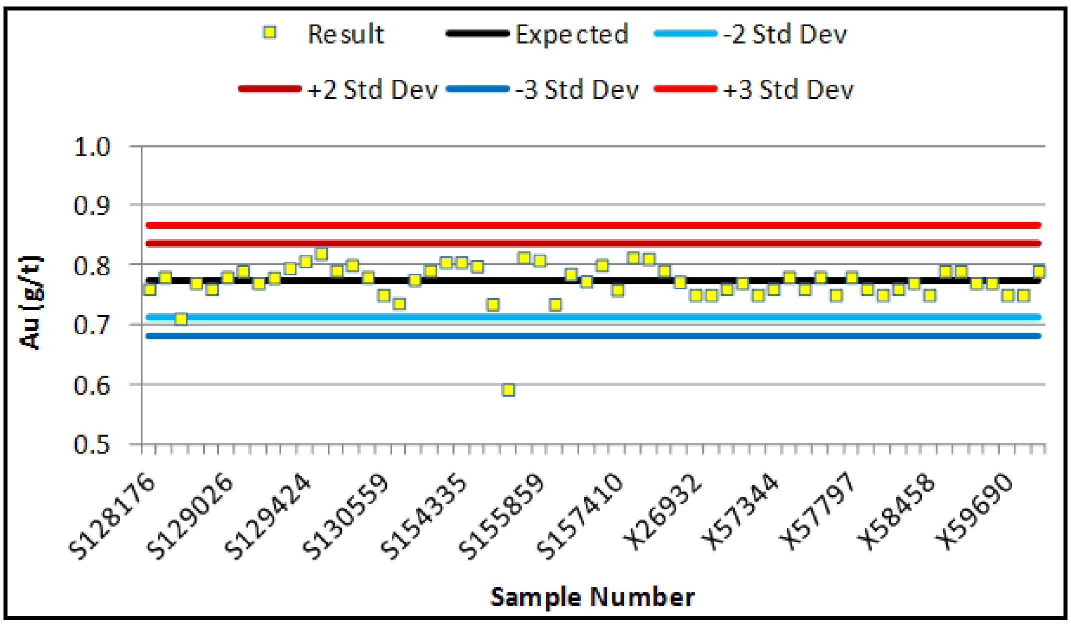


Figure 11.17 2011 Cu Standard SEA-KSM Performance

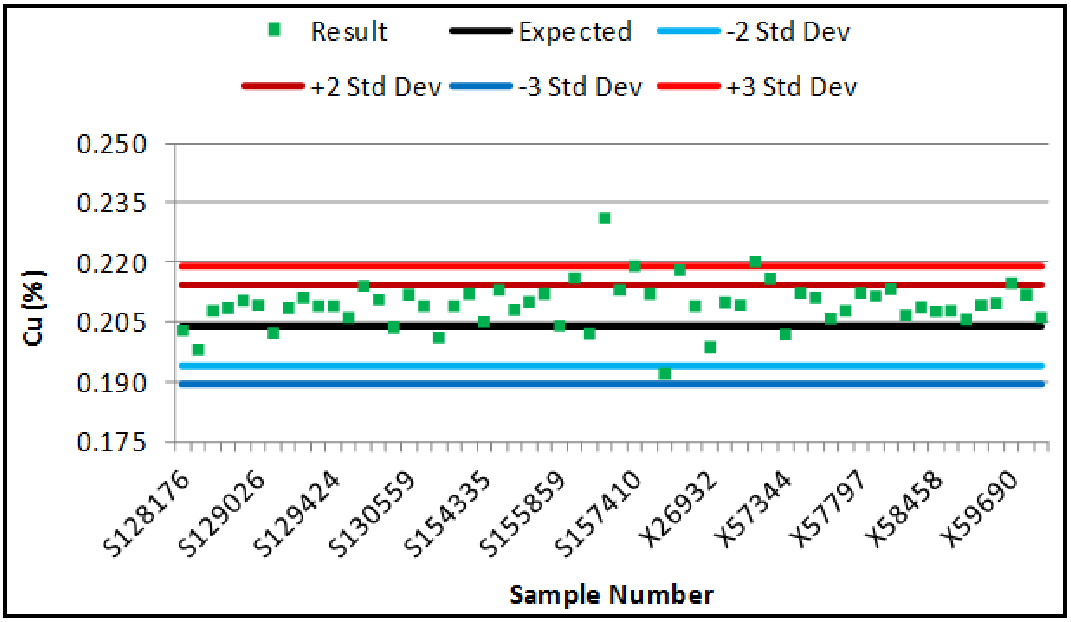
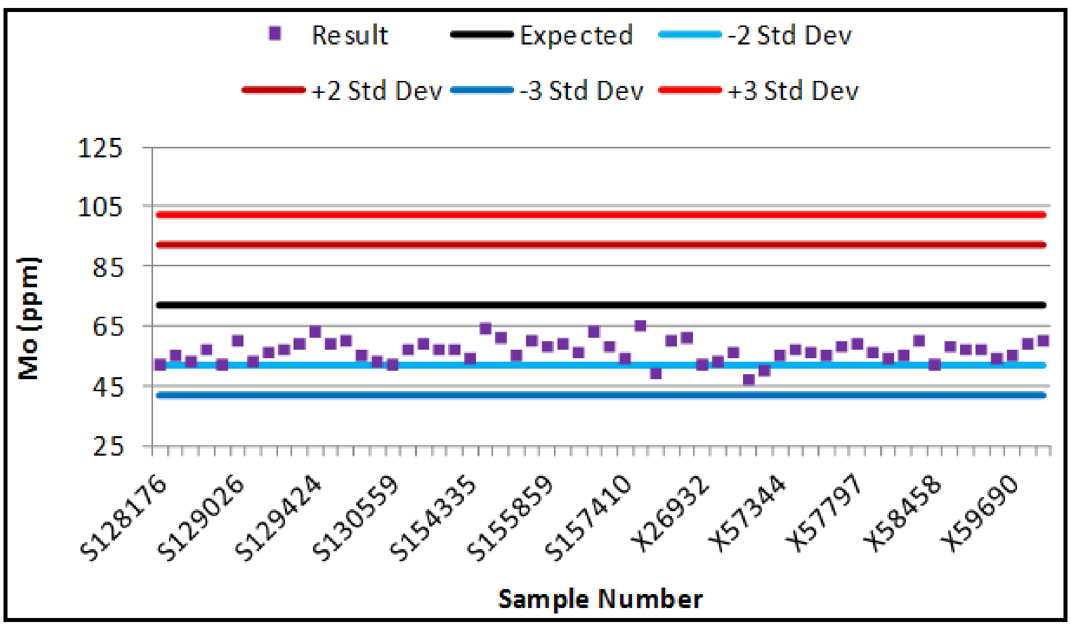


Figure 11.18 2011 Mo Standard SEA-KSM Performance



In general, most of the SRM results track well within ±2 standard deviation of the expected value. One exception is low grade molybdenum standards (Figure 11.8 and Figure 11.18), which routinely came back lower than the expected value. This is particularly evident in Figure 11.18. In RMI's opinion, the poor performance of the lower grade molybdenum standards is not a material issue.

In addition to the insertion of control samples with each batch, Seabridge also submitted duplicate core samples in every second batch by sawing one half of the drill core into two ¼ core splits that were submitted as individual samples to Eco Tech. In 2011, 152 core duplicates, or about 1.5% of the total samples, were submitted to Eco Tech. Table 11.9 summarizes the basic descriptive statistic for the "original" and "duplicate" ¼ core samples for Au, Ag, Cu, and Mo.

Table 11.9 Summary of 2011 ¼ Core Assay Results

Parameter	Au (g/t)		Cu (%)		Ag (g/t)		Mo (ppm)	
	Orig	Dupl	Orig	Dupl	Orig	Dupl	Orig	Dupl
Count	152	152	152	152	152	152	152	152
Min	0.001	0.001	0.0004	0.0003	0.0	0.0	0.1	0.1
Max	3.230	3.510	1.0850	1.0450	18.3	18.6	3,160.0	3,500.0
Mean	0.453	0.431	0.1398	0.1408	1.8	1.8	47.6	47.2
Median	0.246	0.250	0.0890	0.0874	1.0	1.0	12.0	11.0
1st Q	0.111	0.098	0.0380	0.0394	0.6	0.5	4.0	4.0
3rd Q	0.662	0.608	0.1713	0.1872	2.2	2.2	27.5	29.0
Std Dev	0.538	0.531	0.1686	0.1674	2.5	2.4	257.4	283.3
CV	1.19	1.23	1.21	1.19	1.39	1.38	5.41	6.01
% Mean Diff	5%		-1%		0%		1%	

Note: orig = original; dupl = duplicate.

As can be seen in Table 11.9, there is a relatively close comparison in the distribution of original and duplicate ¼ core grades. RMI notes that the duplicate gold sample grades are about 5% higher than the original ¼ core sample. The Cu duplicate is about 1% lower than the original. The ¼ core original (X-axis) and duplicate (Y-axis) sample grades are compared as quantile-quantile (QQ) plots in Figure 11.19 through Figure 11.22 for gold, copper, silver, and molybdenum, respectively.

Figure 11.19 2011 ¼ Core Au QQ Plot

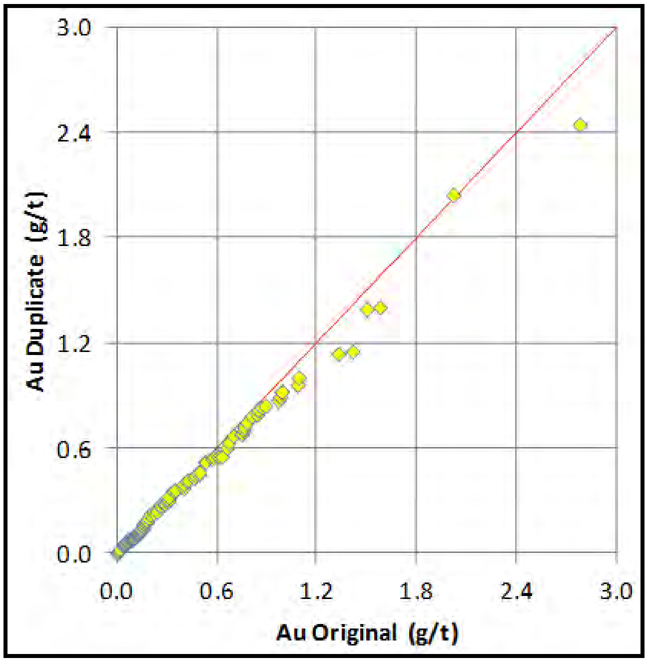


Figure 11.20 2011 ¼ Core Cu QQ Plot

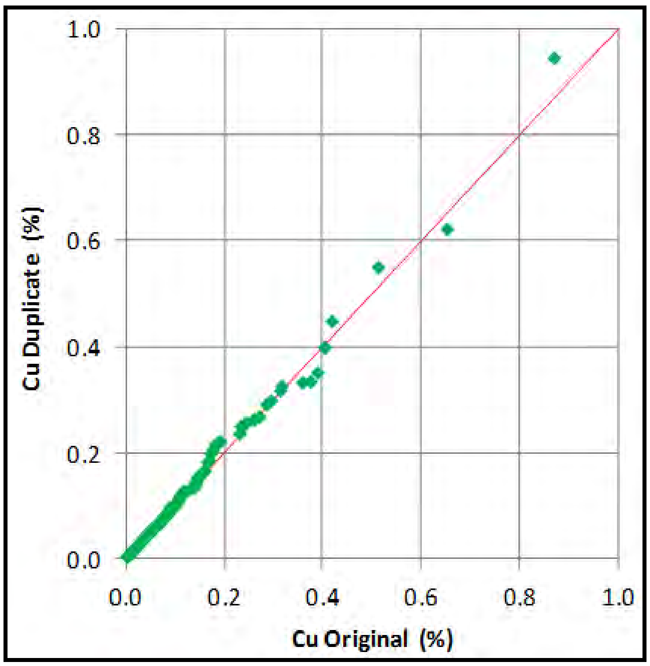


Figure 11.21 2011 ¼ Core Ag QQ Plot

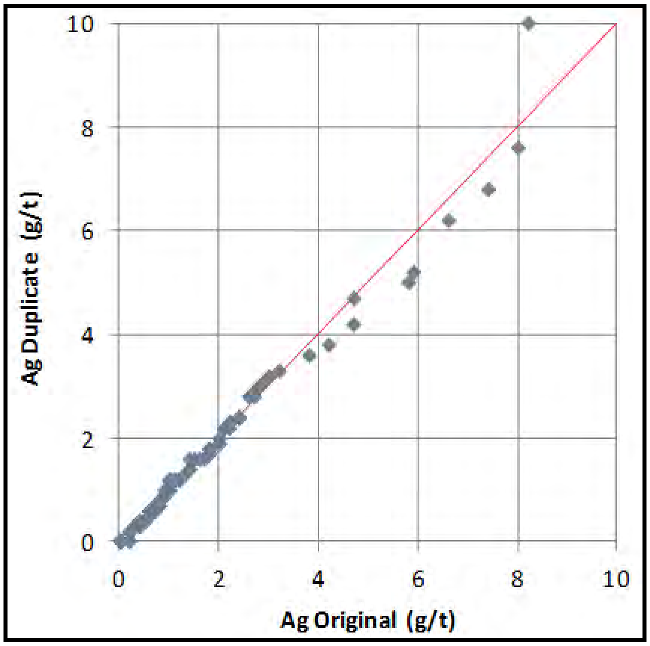
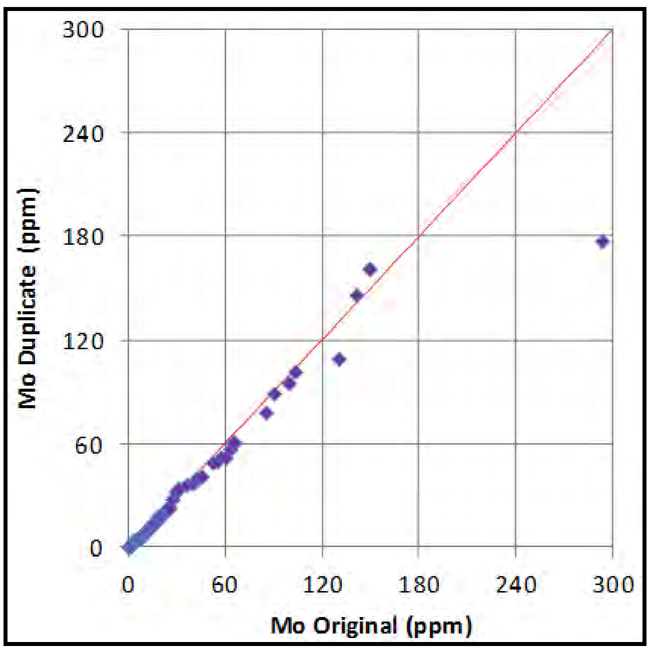


Figure 11.22 2011 ¼ Core Mo QQ Plot



About 6% of the 2011 samples (600 samples) that were prepared and assayed by Eco Tech were re-assayed as same pulp "cross-checks" by ALS Chemex of North Vancouver, BC. Table 11.10 summarizes basic descriptive statistics comparing ALS Chemex ('ALS' in Table 11.10) and Eco Tech results by metal and analytical method. The data in Table 11.10 shows that the mean gold and copper grades as assayed by

ALS Chemex were about 5% lower than Eco Tech. QQ plots compare the same pulp gold, copper, silver, and molybdenum results in Figure 11.23 and Figure 11.26, respectively.

Table 11.10 Summary of 2011 Same Pulp Check Assay Results

Parameter	ALS Au (g/t)	Eco Tech Au (g/t)	ALS Ag (g/t)	ALS ICP Ag (g/t)	ALS ICP Ag (g/t)	Eco Tech ICP Ag (g/t)
Count	597	597	597	597	597	597
Min	0.005	0.015	0.10	0.1	0.1	0.1
Max	4.070	3.950	34.2	35.3	35.3	33.6
Mean	0.486	0.513	1.7	1.8	1.8	1.7
Median	0.380	0.410	0.9	1.0	1.0	1.0
Q1	0.140	0.170	0.4	0.5	0.5	0.4
Q3	0.660	0.680	2.4	2.4	2.4	2.4
Std. Dev.	0.482	0.469	2.6	2.7	2.7	2.5
CV	0.99	0.91	1.49	1.47	1.47	1.49
Mean Diff%	-5%		-4%		8%	

Parameter	ALS Cu (%)	ALS ICP Cu (%)	ALS ICP Cu (ppm)	Eco Tech ICP Cu (ppm)	ALS Cu (%)	Eco Tech ICP Cu (%)
Count	595	593	593	597	595	597
Min	0.001	0.001	8	6	0.001	0.001
Max	1.920	0.984	9,840	19,600	1.920	1.960
Mean	0.161	0.153	1,532	1,609	0.161	0.161
Median	0.125	0.122	1,220	1,232	0.125	0.123
Q1	0.043	0.042	424	422	0.043	0.042
Q3	0.216	0.216	2,160	2,134	0.216	0.213
Std. Dev.	0.176	0.150	1,502	1,795	0.176	0.179
CV	1.10	0.98	0.98	1.12	1.10	1.12
Mean Diff%	5%		-5%		0%	

Parameter	ALS Mo (%)	ALS ICP Mo (%)	ALS ICP Mo (ppm)	Eco Tech ICP Mo (ppm)
Count	596	597	597	597
Min	0.0005	0.0001	0.5	0.5
Max	0.1180	0.0923	923.0	1066.0
Mean	0.0043	0.0032	31.6	32.9
Median	0.0020	0.0013	13.0	14.0
Q1	0.0010	0.0005	5.0	6.0
Q3	0.0050	0.0033	33.0	34.0
Std. Dev.	0.0071	0.0059	59.0	63.1
CV	1.65	1.87	1.87	1.92
Mean Diff%	36%		-4%	

Figure 11.23 2011 Eco Tech vs. Chemex Check Au Assays

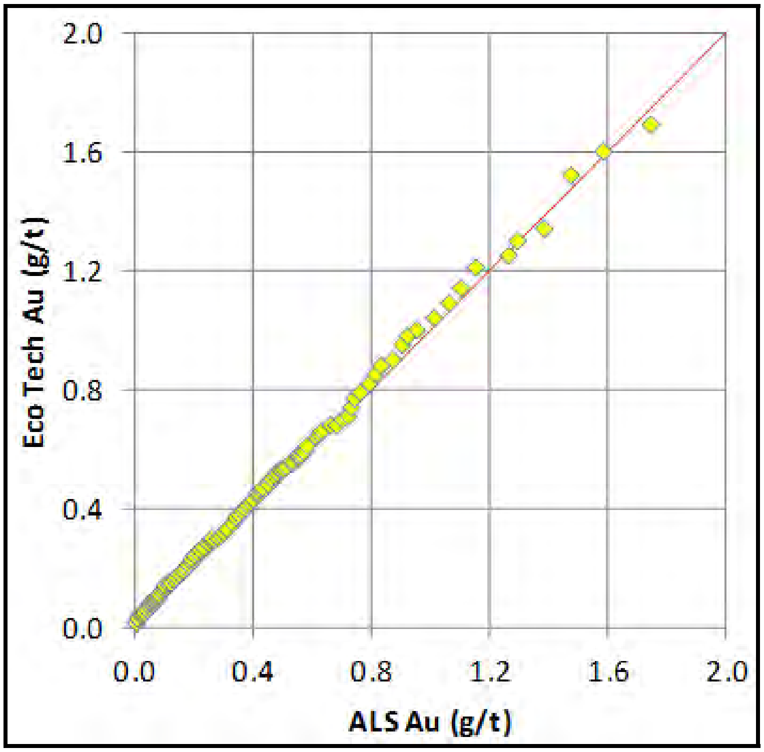


Figure 11.24 2011 Eco Tech vs. Chemex Cu Check Assays

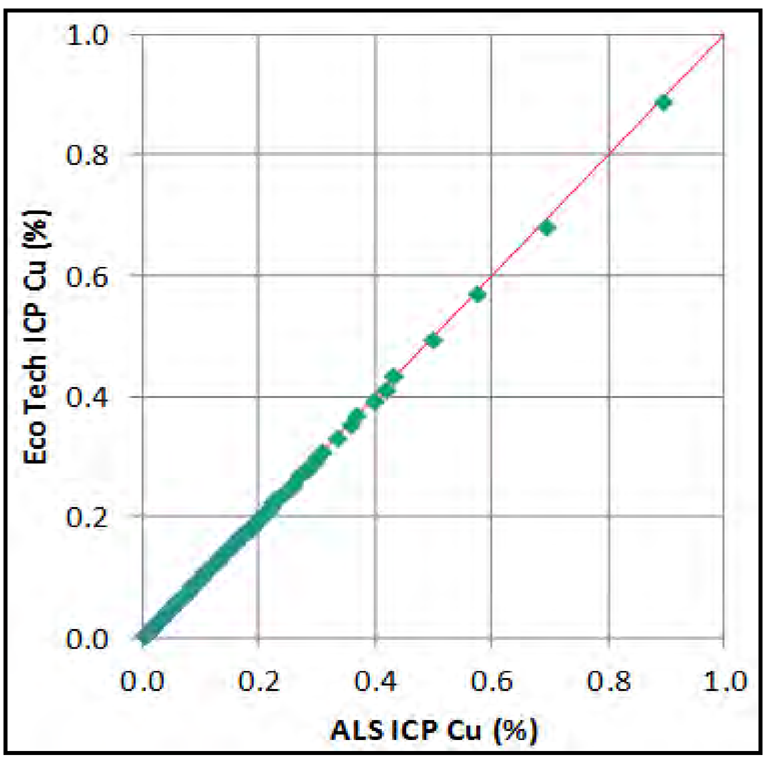


Figure 11.25 2011 Eco Tech vs. Chemex Ag Check Assays

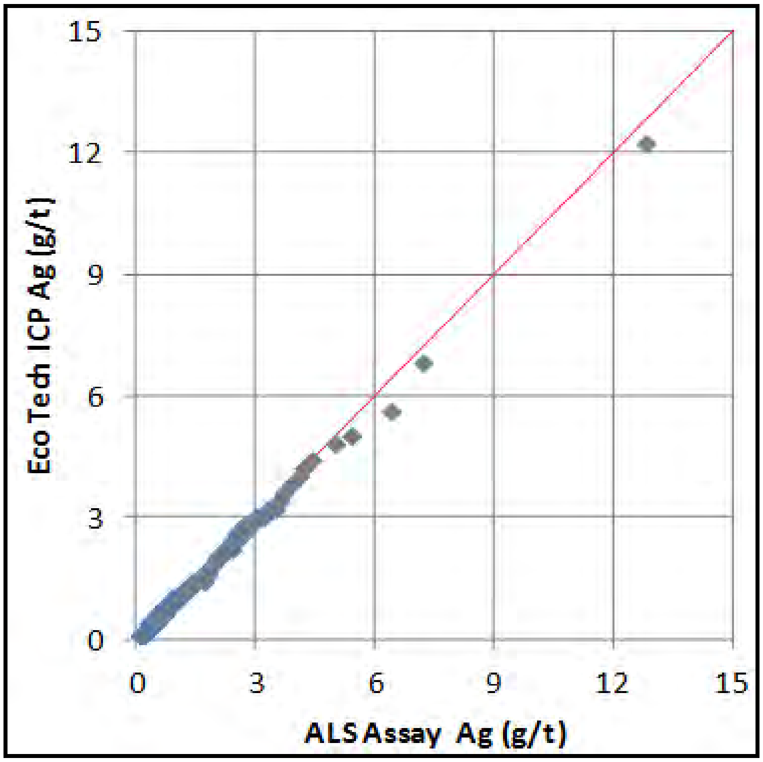
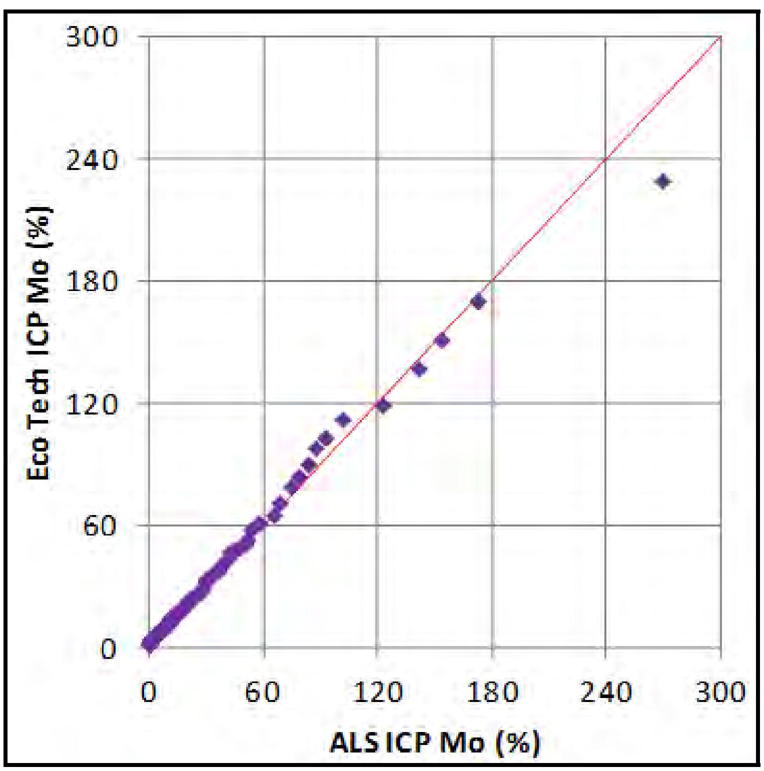


Figure 11.26 2011 Eco Tech vs. Chemex Mo Check Assays



Both Eco Tech and Chemex employed the same assay measurement techniques for gold. 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 (or more reliable than) ore grade, AAS finish analyses.

11.2.5 Corrective Action

During the course of the 2011 assaying program there were several blank and standard reference failures. Most of these were associated with erroneous SRM labelling. The Seabridge QA/QC program properly identified these common errors, and appropriate corrective action was taken.

11.2.6 RMI's Opinion

In RMI's opinion, the sampling methods/approach, security, sample preparation, analytical procedures, and QA/QC protocols/results were adequate and the subsequent assays are suitable to be used to estimate Mineral Resources.

12.0 DATA VERIFICATION

Previous RMI NI 43-101 Technical Reports 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 2011 data.

12.1 ELECTRONIC DATABASE VERIFICATION

RMI performed an audit of the 2011 KSM drill hole database by comparing Eco Tech's certified gold and copper assay results with values stored in Seabridge's electronic database. RMI manually checked gold, silver, copper, and molybdenum assays from four of Seabridge's 2011 drill holes for verification. The data that were verified are summarized in Table 12.1 by drill hole and mineral zone. The data shown in Table 12.1 represent about 10% of the 2011 Seabridge assay data.

Table 12.1 2011 Database Verification

Drill Hole	Zone	Number Checked	Metres Checked	Au Errors	Cu Errors	Ag Errors	Mo Errors
K-11-11	Kerr	279	542	1	1	1	1
S-11-42	Sulphurets	192	376	0	0	0	0
S-11-60	Sulphurets	221	431	0	0	0	0
M-11-126	Mitchell	321	633	0	0	1	0
Grand Total	**n/a**	**1,013**	**1,981**	**1**	**1**	**2**	**1**

RMI notes that the errors that were discovered turned out to be over limit analyses that were re-run and the electronic database was not updated. The five errors discovered out of 4,051 analyses results in an error rate of about 0.1%, which is well within accepted industry standards.

It is RMI's opinion that the KSM electronic database, which was used to estimate Mineral Resources that are the subject of this report, is accurate. This is based on the RMI's own independent comparison of certified assays and the database.

12.2 QA/QC VERIFICATION

Seabridge purchased certified SRMs from CDN and Smee. The SRMs were prepared and certified from various gold-copper porphyry deposits located in BC and the Yukon. Specific information regarding the composition and round-robin assay results for the CDN SRMs that were used by Seabridge can be obtained from CDN's website (www.cdnlabs.com).

Seabridge also asked CDN to prepare a custom standard from KSM drill core that has been collected from various Seabridge drilling campaigns. The core was dried and then mechanically ground in a rod mill and then screened through a 270 mesh sieve. The +270 fraction was retained but not used. The -270 fraction (< 53 µm) was mechanically mixed for three days in a V-Blender rotating at approximately 20 rpm. Seventy 100 g samples where split out and sent for round-robin analysis; 10 samples were sent to 7 commercial laboratories. Those laboratories include ALS Chemex (Vancouver), Acme (Vancouver), Assayers Canada (Vancouver), Actlabs (Ancaster, Ontario), Actlabs (Thunder Bay, Ontario), and TSL Laboratories (Saskatoon). The results from the various laboratories were returned to Smee for tabulation and certification. The standards were packaged in lots of 75 g in tin-tie kraft bags.

Approximately 377 SRMs were submitted to Seabridge's primary laboratory (Eco Tech) as a part of Seabridge's QA/QC program. About 378 blanks were submitted to Eco Tech, along with 179 ¼ core duplicate samples. There were 1,484 Eco Tech pulps shipped to ALS Chemex in Vancouver for check assay purposes. A more thorough discussion of Seabridge's 2011 QA/QC procedures is provided in Section 11.0.

RMI personally reviewed the assay results from the certified standards, blanks, duplicate assays, and same pulp check assays and prepared the charts (Figure 11.1 through Figure 11.26).

Based on a review of the 2011 QA/QC data, RMI believes that the drill hole assay data are representative and suitable to be used to estimate Mineral Resources.

12.3 TOPOGRAPHIC CONTOUR DATA

In 2008, McElhanney of Vancouver, BC, was contracted to perform an aerial survey, and provide Seabridge with an updated accurate topographic base map of the three deposits and surrounding area. McElhanney obtained the data by conducting a helicopter-borne LiDAR survey. 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 scanner 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 Government of BC and industry mapping applications. Seabridge contracted Aero Geometrics of Vancouver to translate the KSM drill hole collar locations from NAD27 to NAD83 datum. Aero Geometrics used Sierra Systems Groups Inc. MAPS 3D software to perform the transformation of all collar coordinates. MAPS 3D uses the Canadian National Transformation Versions 1.1 and 2.0 for the transformation.

RMI and Seabridge noted some discrepancies in the GPS surveyed collar locations and the new LiDAR topographic surface. These differences are believed to be based on:

1. the fact that no transform of the Z-coordinate was considered by the Canadian National Transformation software
2. the inaccuracy of the initial GPS elevation
3. the fact that many of the holes were surveyed immediately below the drill deck and not ground level or "stick-up"
4. differences magnified by steep terrain.

12.4 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 value of 2.84 g/cm^3 by those attributes. Seabridge has since collected 26 bulk density determinations from their 2009/2010 drilling programs using the same methods described for the Placer Dome determinations. RMI notes that the average bulk density for the Seabridge data was 2.84. For the 2011 Kerr model, RMI assigned bulk density based on averages by lithology and/or alteration type (Table 12.2).

A total of 522 bulk density determinations have been collected for the Sulphurets Zone. The majority of those determinations (337) were collected by Placer Dome in 1992. Seabridge collected an additional 185 determinations from their 2009/2011 drilling programs using the weight in air, weight in water method.

A total of 910 bulk density determinations have been performed by Seabridge from their 2006-2011 drilling campaigns using the weight in air, weight in water method.

Seabridge completed 154 bulk density determinations from their 2010 Iron Cap drilling program. Again, the weight in air, weight in water method was used.

Table 12.2 summarizes the bulk density values used to tabulate resource tonnage by mineral zone.

Table 12.2 KSM Bulk Density Values

Zone	Description	Bulk Density Value (g/cm^3)
Kerr	Default	2.84
	CL alt	2.80
	QSP alt	2.88
	Weak CLQSP alt	2.87
	Premier Dike	2.78
	Hornblende Dike	2.86
Sulphurets	Hazelton above Sulphurets Thrust Fault	2.71
	Hazelton below Sulphurets Thrust Fault	2.77
	Main Au Zone	2.79
	Au Leach Zone	2.77
	Raewyn Cu Zone	2.77
	Lower Au Zone	2.77
	Monzonite	2.69
Mitchell	Hazelton above Mitchell Thrust Fault	2.71
	Hazelton below Mitchell Thrust Fault	2.77
	CL-PR Alteration	2.74
	QSP Alteration	2.79
	IARG Alteration	2.78
	Monzonite	2.73
Iron Cap	All Rock Units	2.74
Non Rock Units	Overburden	2.00
	Glacial Ice	0.90

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 METALLURGICAL TEST WORK REVIEW

The KSM Project includes four major mineralized zones, identified as the Mitchell, Kerr, Sulphurets, and Iron Cap deposits. The deposits contain significant gold, copper, silver, and molybdenum mineralization.

Several metallurgical test programs have been carried out to assess the metallurgical response of the mineral materials, especially the samples from the Mitchell deposit. The latest test programs were performed from 2007 through early 2012. The metallurgical testing programs, including historical testing programs, are listed in Table 13.1. The following sections summarize the test work.

Table 13.1 Metallurgical Test Work Programs

Year	Program ID	Laboratory	Mineralogy	Flotation/Cyanide Leach	Grindability	Others
2012	KM3174	G&T		√	√	
2012	KM3080	G&T		√		√
2011	KM3081	G&T				
2011	KM 2897	G&T		√		
2010/2011	KM 2748	G&T	√	√	√	√
2010	KM 2755	G&T	√	√		√
2010	KM 2670	G&T	√	√		
2009/2010	KM 2535	G&T		√	√	
2009/2010		SGS		√	√	√
2009/2010		Köeppern -UBC			√	
2009	KM 2344	G&T	√	√	√	√
2009		Pocock				√
2008	KM 2153	G&T	√	√	√	√
2008		Hazen			√	
2007	KM 1909	G&T	√	√	√	√
1991		Placer Dome		√	√	√
1990		Placer Dome	√	√	√	√
1989		Brenda Mines Met Lab		√	√	
1989		Coastech		√		

Abbreviations:

G&T = G&T Metallurgical Services Ltd.
SGS = SGS Mineral Services
Pocock = Pocock Industrial Inc.
Hazen = Hazen Research Inc.
Placer Dome = Placer Dome Research Centre.
UBC = University of British Columbia
Köeppern = Köeppern Machinery Australia Pty Ltd.
Brenda Mines Met Lab = Brenda Mines Ltd. Metallurgical Laboratory
Coastech = Coastech Research Inc.

Note: *The KM3174, KM3080, and KM3081 test work reports are available in Appendix D.*
The remaining test work reports are included in Appendix E of the "KSM PFS Update 2011" (Wardrop, 2011).

13.1.1 Historical Test Work – Prior to 2007

Tetra Tech received several historical test work reports from Seabridge. The historical test work included preliminary investigations into mineralogy, grindability, and metallurgical responses to flotation. Most of this early test work was conducted on samples from the Kerr Zone.

Historical Test Samples

Coastech Research Inc. – 1989

Two samples from the Kerr mineralized zone were tested in the program: one representing the central high grade copper zone (High Grade) and another representing the remainder of the Kerr Zone (Low Grade). The assay data are shown in Table 13.2.

Table 13.2 Test Samples – Coastech, 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

Brenda Mines Ltd. Metallurgical Laboratory – 1989

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

Placer Dome Research Centre – 1990

Four new Kerr Zone composites, labelled Composites K-1 to K-4, were prepared from 560 individual samples of crushed drill core rejects, weighing a total of 2.3 t.

Two additional Kerr composites, received from the previous Coastech 1989 program, were also included in the test program. These two composites were labelled as LG-01 for low grade and HG-01 for high grade samples, respectively.

Table 13.3 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

Placer Dome Research Centre – 1991

Bulk samples from Kerr Zone, identified as Rubble Zone Trench and Crackle Breccia Zone Trench, were used in the 1991 testing program. Exploration personnel from Placer Dome collected the bulk samples. The average gold, silver, and copper values are shown in Table 13.4.

Table 13.4 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

HISTORICAL 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 13.5.

Table 13.5 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 complete with stockwork and veining
K-4	Quartz-sulphide veins and lenses - high grade

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

Grindability

In 1989, using a comparative method, Brenda Mines determined the work index (Wi) of Sample 106 to be 13.52 kWh/t.

In 1990, Placer Dome determined comparative ball mill work indices on Composites K-1 to K-4 and Composites LG-01 and HG-01. The comparative work index (CWi) increased with finer grinding. The resulting work indices ranged 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

The results of bulk and dry SG measurements conducted by Placer Dome in 1990 and 1991 on the Kerr samples are summarized in Table 13.6. The average SG and the bulk SG are 2.89 and 2.82, respectively.

Table 13.6 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**

HISTORICAL FLOTATION

Brenda Mines Metallurgical Laboratory – 1989

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

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

Placer Dome Research Centre – 1990

Primary open circuit roughing and cleaning tests were conducted on six Kerr composite samples. The test work included the evaluation of primary grind size in the range of 80% passing 175 µm to 80% passing 35 µm. Rougher/scavenger flotation copper recoveries ranged from 89 to 96%, gold recoveries from 67 to 94%, and silver recoveries from 81 to 95%. High rougher copper and precious metal recoveries were achieved from all six composites with the highest metal recoveries obtained at the finer primary grinds.

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

The rougher concentrate was reground and the slurry pH was adjusted to 11 with lime. The rougher concentrate was upgraded using three stages of open circuit cleaning. Saleable copper concentrates were produced from four of the six composites tested. Approximately half of the gold and silver reported to the final copper concentrate.

The 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, composites K-1 and K-2 produced inferior results. The report indicated that the poorer 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 flotation to suppress these minerals.

Placer Dome Research Centre – 1991

The test program confirmed the recoveries achieved in the earlier flotation tests conducted in 1990. High final copper concentrate grades were produced from the two new Kerr composite samples tested.

Four grind and flotation tests were performed on each of the two samples. The test results are summarized in Table 13.7.

Table 13.7 Flotation Test Results – Placer Dome, 1991

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
Primary Grind								
- 80% passing (P_{80}), µm	223	175	149	98	165	110	99	59
Final Concentrate								
Grade								
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
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 improved as primary grind fineness increased. The finest primary grind size produced the best overall copper and gold recoveries. 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.

Gold and silver assays conducted on 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.

13.1.2 Recent Test Work – 2007 to 2012

Since 2007, twelve main testing programs were sequentially carried out to investigate the mineralogical characteristics, ore hardness, metallurgical performance of various mineral samples, and to determine process related parameters, such as unit thickening rates and filtration rates. The metallurgical performance investigations included flotation recoveries of copper, gold, silver, and molybdenum minerals, gravity concentration of gold and silver minerals, and cyanide extraction of gold and silver. The flotation test work included open cycle batch tests, LCTs, and pilot plant tests. Although most test work was conducted primarily on the samples from the Mitchell deposit, the testing programs also investigated the metallurgical performance of the samples from the Sulphurets, Kerr, and Iron Cap deposits.

In general, the mineralization from the four different deposits responded similarly to a flotation concentration and sulphide concentrate cyanidation process with respect to copper, gold, silver, and molybdenum metallurgical performance. The Mitchell samples gave the most consistent results throughout the testing programs.

MITCHELL MINERALIZATION

Test Samples

All the testing samples for the various testing programs were collected from diamond drill cores produced from various drilling programs.

The 2007 testing program used three composite samples. Table 13.8 shows the chemical assays and key mineral distribution of the composite samples.

Table 13.8 Test Samples – Mitchell, 2007 (G&T)

	Units	Composite A	Composite B	Composite C	Average
Element Assay					
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 Distribution					
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

The later test work used the samples collected from 2008 and 2009 drilling programs. The 2008 testing program used a total of approximately 5,720 kg of drill core samples for the testing. Most of the samples were collected from the Mitchell Zone. The 2008 and 2009 drill hole distributions for the Mitchell Zone are shown in Figure 13.1. The variability testing samples are listed in Table 13.9.

Figure 13.1 2008 and 2009 Mitchell Zone Metallurgical Samples – Plan View

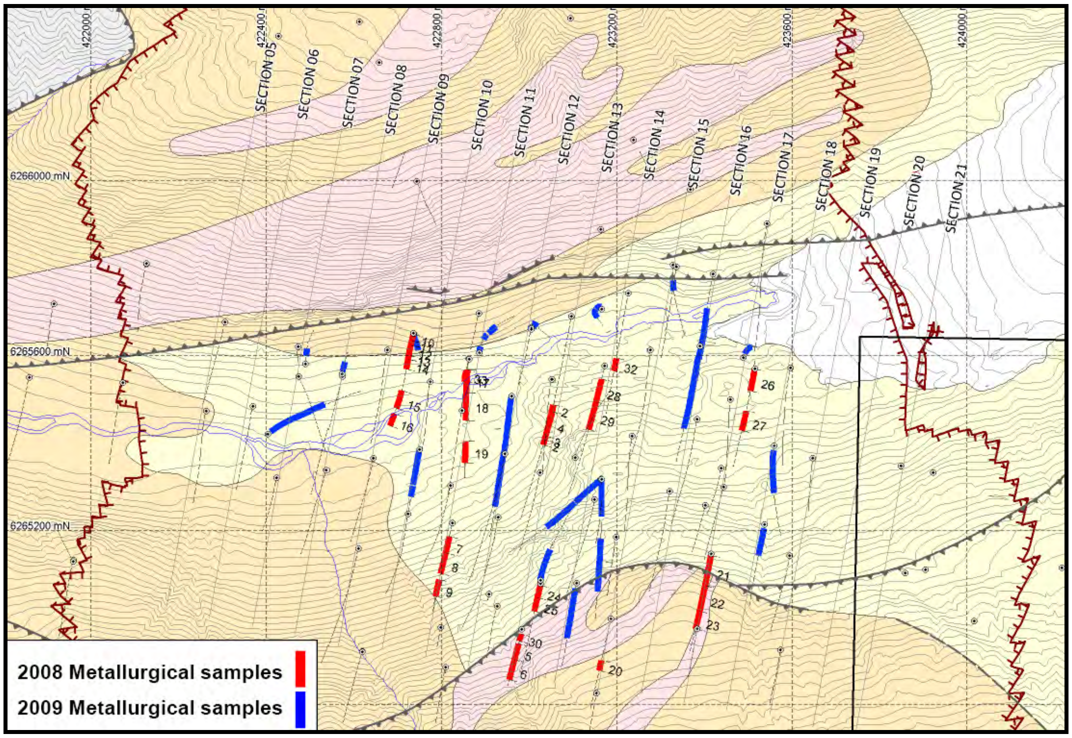


Table 13.9 Head Assay on Variability Test Samples – Mitchell, 2008 (G&T)

Sample ID	Metal Content (% or g/t)*					Sample ID	Metal Content (% 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 additional composites were generated from the "MET" samples, including 9 composite samples representing the major Mitchell Zone mineralization types that were projected to be mined during the different mining periods laid out in the mine plan generated from the 2008 "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment" (Wardrop, 2008). The feed grades for the composites are shown in Table 13.10.

Table 13.10 Head Assay on Composites – Mitchell, 2008 (G&T)

	Metal Content				
Sample ID	**Cu (%)**	**Au (g/t)**	**Ag (g/t)**	**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
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

The 2009/2010 testing programs used a total of 12.1 t of core samples from 3,218 different drill core intervals from the Mitchell and Sulphurets deposits. Eleven composites were generated from the Mitchell deposit according to mineralization types. The metal contents in the composite samples from the Mitchell deposit are shown in Table 13.11.

The assay data indicated that the copper mineral oxidation level was low; only 3% or less of the copper is present in oxide forms.

The Composite PP1 sample was constructed from CL-PR, QSP, and Hi Qtz mineralization, the three dominant mineralization types of the Mitchell deposit. Composite PP2 was selectively prepared with higher molybdenum core intervals.

Table 13.11 Metal Contents of Composites – Mitchell, 2009 (G&T)

Composite	Mineralization Type*	Metal Content Cu(T) (%)	Cu(OX) (%)	Cu(CN) (%)	Au(T) (g/t)	Au(CN) (g/t)	Mo (%)	Ag (g/t)
Comp 40	CL-PR	0.20	0.006	0.008	0.67	0.013	0.004	3.6
Comp 41	BBRX	0.71	0.006	0.008	0.35	0.007	0.010	8.9
Comp 42	QSP	0.28	0.006	0.011	1.02	0.009	0.002	4.1
Comp 43	CL-PR	0.22	0.004	0.011	0.70	0.004	0.004	3.1
Comp 44	Hi Qtz	0.27	0.008	0.019	0.92	0.006	0.010	4.2
Comp 45	IARG	0.13	0.002	0.004	0.57	0.013	0.010	3.5
Comp 46	CL-PR	0.15	0.003	0.004	0.67	0.012	0.011	2.0
Comp 47	QSP	0.16	0.004	0.006	0.73	0.015	0.013	2.3
Comp 48	QSP	0.10	0.003	0.002	0.61	0.013	0.015	2.2
Comp PP1	Blend	0.24			0.76		0.004	
Comp PP2	Blend	0.18			0.64		0.010	

***Notes:**
QSP: Quartz, sericite, pyrite altered rocks
IARG: Intermediate argillic altered rocks (quartz, sericite, chlorite, pyrite, ±clays)
CL-PR: Chlorite-propylitic altered rocks (quartz, chlorite, pyrite, ±magnetite, ±epidote, ±calcite)
Hi Qtz: Altered rocks with >60% quartz veining by volume, higher than average pyrite (7-15%)
BBRX: Bornite breccia (breccia w/bornite, chalcopyrite, pyrite in matrix of quartz, clay, anhydrite)
Blend: Blend from various mineralization types for pilot plant testing
Cu(T): Total copper; **Cu(OX):** oxide copper; **Cu(CN):** cyanide soluble copper
Au(T): Total gold; **Au(CN):** cyanide soluble gold.

In 2010, three additional Mitchell Zone composites were generated using the drill core interval samples from the 2009/2010 drilling program. The sample details are shown below and in Table 13.12:

- PP Composite 3: crushed materials generated from HPGR tests (approximately 5.5 t) for bench tests and pilot plant tests. The HPGR bulk sample was collected from core intervals within the 10-year pit mining model generated in 2009. The cores were selected according to proportion of each ore type above the cut off grade in the 10-year pit. The drill core interval plan is shown in Figure 13.2 and the main element content estimates and percent of mineralization type domain is provided in Table 13.13.
- PP Hi-Mo Composite: halved drill cores (approximately 6.3 t)
- BS Hi-Mo Composite: high molybdenum content drill cores selected from halved drill cores for PP Hi-Mo composite.

Table 13.12 Metal Contents of Composites – Mitchell, 2010 (G&T)

Sample	Cu (%)	Mo (%)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)
PP Composite 3	0.20	0.006	4.29	3.66	0.79	3.2
BS Hi-Mo Composite	0.12	0.013	3.95	3.27	0.57	2.4
PP Hi-Mo Composite	0.16	0.012	4.02	3.67	0.60	-

Figure 13.2 Drill Core Interval Plan for PP Composite 3

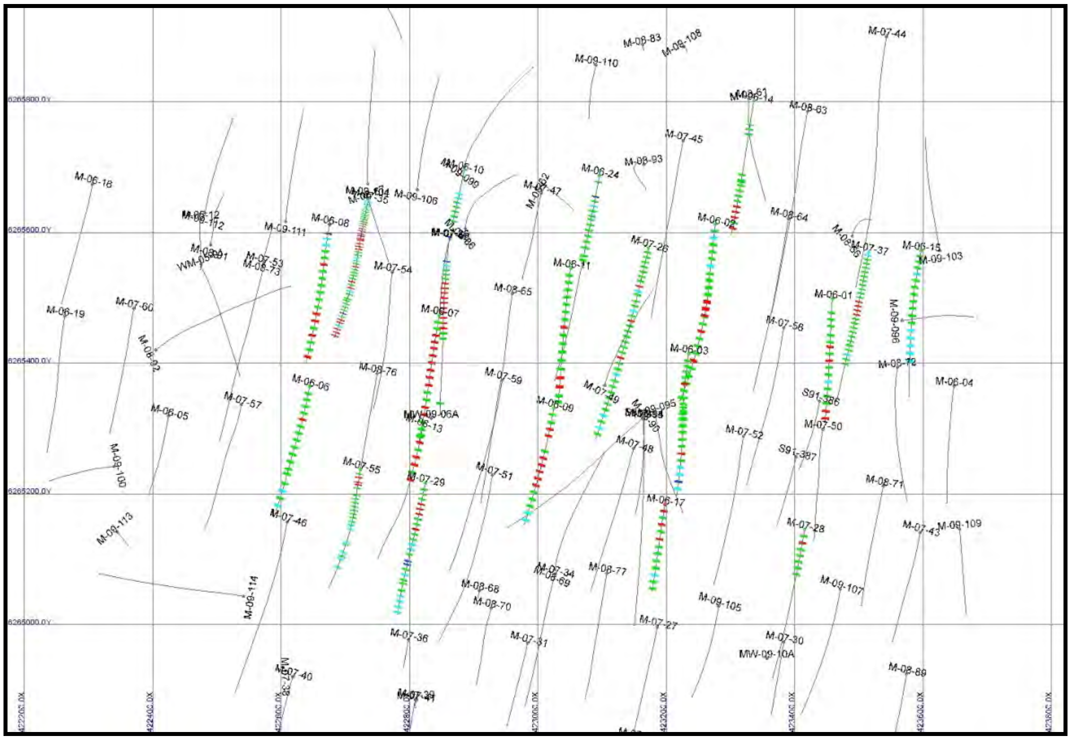


Table 13.13 Element Content Estimate and Percent of Mineralization Type Domain

Sample	Weight (kg)	Content Estimate							% of Mineralization Type Domain			
		Au (ppm)	Cu (%)	Mo (%)	Ag (ppm)	As (ppm)	Qtz (%)	Pyrite (%)	QSP (%)	IARG (%)	CL-PR (%)	High Qtz (%)
1	364	0.706	0.152	0.0149	2.20	41	24.7	5.2	71		7	23
2	359	0.810	0.168	0.0037	3.76	24	11.6	3.9	38	22	40	
3	370	0.812	0.221	0.0037	3.76	23	24.0	2.8	8	11	56	25
4	339	0.695	0.180	0.0046	2.07	24	24.1	5.5	56	9	13	21
5	388	0.878	0.209	0.0056	1.61	34	42.3	5.6	44			64
6	399	0.789	0.169	0.0065	1.79	21	37.2	4.5	55			45
7	346	0.785	0.188	0.0048	2.29	27	32.2	3.8	60		4	36
8	352	0.707	0.211	0.0026	2.91	37	39.1	5.1	17		43	40
9	371	0.937	0.216	0.0036	4.46	16	18.6	5.3	50		28	22
10	398	0.987	0.297	0.0070	3.66	27	40.4	7.1	59		7	34
11	375	0.689	0.216	0.0043	3.67	56	36.9	4.5	65			35
12	353	1.062	0.276	0.0015	5.10	27	25.1	6.0	63		13	25
13	364	0.861	0.202	0.0054	2.73	19	12.4	3.3	4	13	77	6
14	332	0.730	0.117	0.0097	1.65	36	6.7	4.4	100			
15	402	0.583	0.169	0.0021	2.91	21	6.2	2.6	34	6	60	
Total	**5,512**	**0.803**	**0.198**	**0.0053**	**2.95**	**29**	**25.4**	**4.6**	**48**	**4**	**23**	**25**

In the 2011 and 2012 test programs, 10 composites were generated from the Mitchell deposit drill core interval samples.

- Mitchell Year 0 to 5 (KM3080): proposed average mill feed from the Mitchell pit during Years 0 to 5 based on the 2011 mine plan
- Mitchell Year 0 to 10 (KM3080 and KM3081): proposed average mill feed from the Mitchell pit during Years 0 to 10 based on the 2011 mine plan
- Mitchell Year 0 to 20 (KM3080 and KM3081): proposed average mill feed from the Mitchell pit during Years 0 to 20 based on the 2011 mine plan
- Composite 1 (KM3174): proposed average mill feed from the Mitchell pit during Years 0 to 5 based on the 2011 mine plan
- Composite 2 (KM3174): proposed average mill feed from the Mitchell pit during Years 0 to 10 based on the 2011 mine plan
- Composite 3 (KM3174): proposed average mill feed from the Mitchell pit after Year 10 based on the 2011 mine plan
- Composite 4 (KM3174): Mitchell QSP mineralization
- Composite 5 (KM3174): Mitchell QSP mineralization
- Composite 6 (KM3174): Mitchell CL PR mineralization
- Composite 7 (KM3174): Mitchell IAGG mineralization.

Three composite samples were prepared for the test programs of KM3080 and KM3081and the rest were for KM3174. The sample details are shown below and in Table 13.14.

Table 13.14 Metal Contents of Composites – Mitchell, 2011/2012 (G&T)

Sample	Cu (%)	Mo (%)	Fe (%)	S (%)	Au (g/t)	Ag (g/t)
Mitchell Year 0 to 5	0.21	0.008	4.0	3.6	0.60	4
Mitchell Year 0 to 10	0.20	0.005	3.7	3.4	0.67	4
Mitchell Year 0 to 20	0.22	0.006	4.4	3.6	0.64	3
Composite 1	0.20	0.005	3.9	3.17	0.77	4
Composite 2	0.20	0.003	3.8	3.62	0.69	3
Composite 3	0.20	0.004	4.3	4.52	.71	3
Composite 4	0.20	0.006	4.2	4.17	1.10	4
Composite 5	0.23	0.005	4.1	4.89	0.56	3
Composite 6	0.19	0.003	4.2	3.22	0.62	2
Composite 7	0.13	0.009	4.2	3.75	0.65	3

Mineralogy

The mineralogical composition study of the 2008 testing program shows that the sulphide minerals in all three samples (QSP 0-30, Hi Qtz 0-30, and Master Composite 1) are dominated by pyrite, which is present as approximately 6 to 8% of the sample weight. The study also indicated that the copper was present in the form of chalcopyrite. Detailed analysis results are provided in Table 13.15.

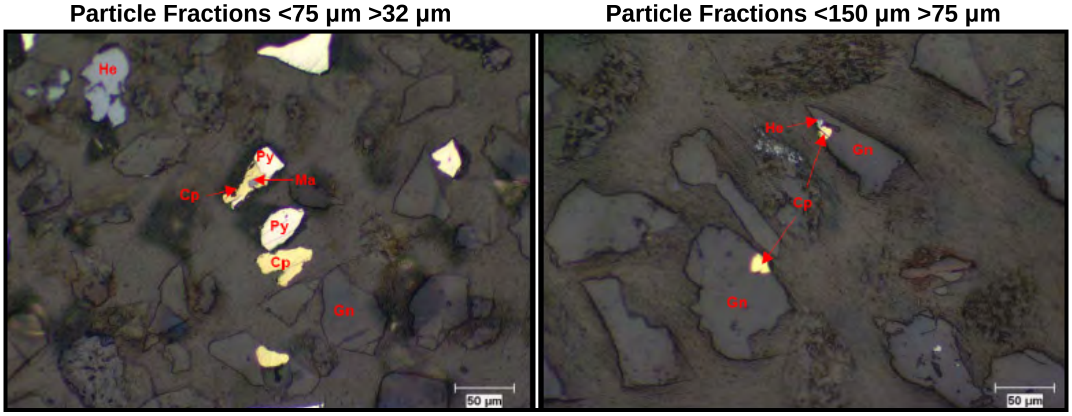
Table 13.15 Mineral Composition Data – Mitchell, 2008 (G&T)

Sample	Mineral Composition (%)		
	Chalcopyrite	**Pyrite**	**Gangue**
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 in the three composite samples. The average ratio is 12:1 while the highest ratio reaches 15:1. There does not appear to be close pyrite-chalcopyrite interlocking. Figure 13.3 illustrates the typical relationship among the main minerals in the samples.

Figure 13.3 Mineral Relationship – Master Composite, Mitchell

Particle Fractions <75 µm >32 µm | **Particle Fractions <150 µm >75 µm**



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

The degree of chalcopyrite liberation ranged from 46% to 56% across the samples tested at a primary grind size 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.

Mineralization Hardness

Various grindability tests have been conducted in a number of test programs including SMC grindability testing, crushing characteristics to HPGR, and standard Bond ball mill work index determination.

Grindability/Crushability Determination – Bond Ball Mill Work Index

Both G&T and SGS carried out standard Bond ball mill work index tests on the Mitchell mineralization. As summarized in Table 13.16, the Bond work indices determined from different testing programs range from 12.5 kWh/t to 15.5 kWh/t, averaging 14.4 kWh/t. The data suggests that the Mitchell samples are of moderate hardness. The Bond abrasion index (Ai) of Composite PP1 was measured at 0.293 g by SGS.

Table 13.16 Bond Ball Mill Work Index Test Results – Mitchell, 2008

Samples	Wi (kWh/t)	Ai (g)
2011/2012 G&T		
Composite 1	14.3	
Composite 2	14.3	
Composite 3	14.9	
Composite 4	14.1	
Composite 5	14.5	
Composite 6	14.4	
Composite 7	15.3	
Sub-average	**14.5**	
2009 G&T		
Composite 40	15.5	
Composite 41	14.8	
Composite 42	15.2	
Composite 43	14.6	
Composite 44	13.4	
Composite 45	14.1	
Composite 46	12.8	
Composite 47	13.3	
Composite 48	12.5	
Sub-average	**14.0**	
2009/2010 SGS		
Composite PP1	13.8	0.293
2008 G&T		
High Quartz 0-10	15.2	
High Quartz 10-30	15.3	
IARG 0-10	13.9	

table continues...

Samples	Wi (kWh/t)	Ai (g)
QSP 0-10	14.5	
QSP 10-30	15.2	
Sub-average	**14.8**	
2007 G&T		
A	14.7	
B	14.8	
C	14.8	
Sub-average	**14.8**	
Total Average	**14.4**	

G&T also compared the hardness variation of various variability test samples and main mineralization type composites by the CWi method in the 2008 testing program. The CWi was calculated from grind calibration data and the standard Bond ball mill work index. The data is compared in Figure 13.4 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 13.4 Comparative Ball Mill Work Index – Variability Samples, 2008

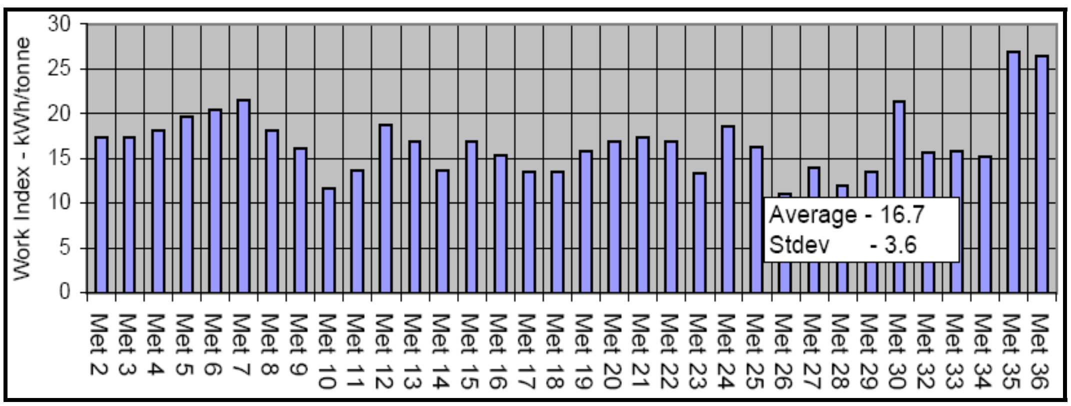
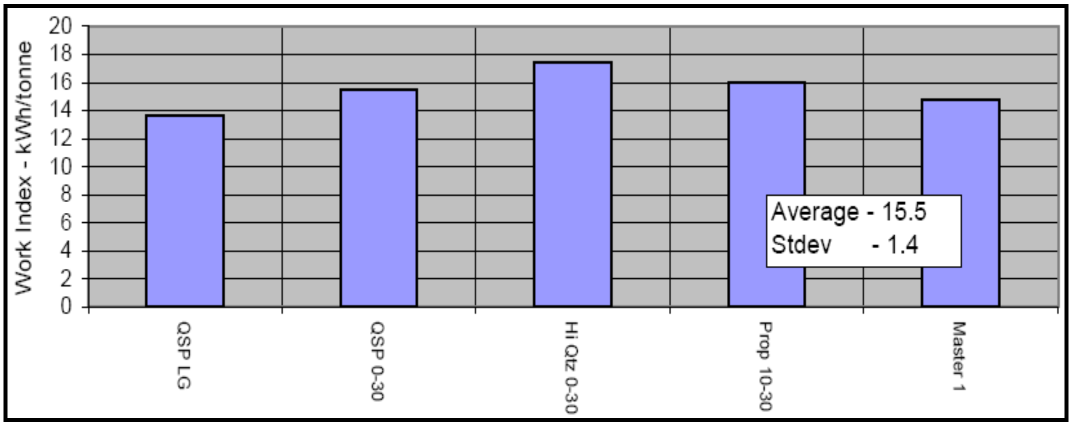


Figure 13.5 Comparative Ball Mill Work Index – Composite Samples (Mitchell, 2008)



Grindability/Crushability Determination – SMC Tests and Simulations

The SMC grindability tests were conducted by Hazen in 2008. The samples used for the grindability tests were identified as QSP, IARG, CL-RICH, QSP STW/QTVN, and H FELDS. The SMC test results are shown in Table 13.17.

Table 13.17 SMC Test Results – Mitchell, 2008

Parameter	Sample				
	QSP	IARG	CL-RICH	QSP STW/QTVN	H FELDS
SG	2.81	2.42	2.78	2.69	2.71
A	70.7	75	68.1	82.6	81.6
b	0.71	0.40	0.57	0.60	0.44
Axb	50.2	30.0	38.8	49.6	35.9
DWi (kWh/m^3)	5.5	7.9	7.1	5.4	7.5
Mia (kWh/t)	16.1	24.8	19.9	16.3	21.2
Ta	0.47	0.33	0.37	0.49	0.35

SG: Specific Gravity
A: Maximum Breakage
B: Relation between Energy & Impact Breakage
Axb: Overall SAG Hardness
DWi: Drop Weight Index (kWh/m^3)
Mia: Coarse Ore Wi (kWh/t)
Ta: Estimated Abrasion Parameter

In 2011, G&T conducted additional SMC tests to investigate the grindability of the Mitchell samples to SAG mills. The test results are summarized in Table 13.18.

Table 13.18 SMC Test Results – Mitchell, 2011/2012

Parameter	Sample						
	Comp 1	**Comp 2**	**Comp 3**	**Comp 4**	**Comp 5**	**Comp 6**	**Comp 7**
SG	2.79	2.81	2.79	2.83	2.75	2.79	2.79
A	59.8	57.3	68.8	60.0	66.2	55.3	53.2
b	0.91	1.01	0.65	0.79	0.90	0.86	1.00
Axb	54.4	57.9	44.7	47.4	59.6	47.6	53.2
DWi (kWh/m^3)	5.12	4.83	6.21	5.96	4.61	5.86	5.23
Mia (kWh/t)	15.2	14.4	17.8	16.9	14.2	16.9	15.4
Ta	0.51	0.54	0.42	0.43	0.56	0.44	0.50

The DWi and Axb data indicate that, on average, the materials are moderately resistant to SAG mill grinding in comparison to the JK Tech database. The 2008 test results showed that Axb ranged from 30.0 to 50.2, while the data of the 2011/2012 tests indicated that the mineral samples are slightly less resistant to SAG milling.

Contract Support Services conducted three SABC circuit simulations to estimate equipment sizing. The simulations used JK SimMet software. All the simulations were based on the data generated from the SMC testing.

The simulation input conditions are based on 120,000 t/d (two streams of 60,000 t/d each), 92% availability, a feed particle size of 80% passing 150 mm and one of the following conditions:

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

Table 13.19 JK SimMet Simulation Results (60,000 t/d SABC Circuit, 2008)

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 (two circuits required) are summarized in Table 13.19. The simulations are based on phantom cyclone assumption and with primary cyclones for SAG mill discharges.

The simulation results also show that, with a primary grind size of 80% passing 120 µm, either of the following options will meet the primary grinding requirements for a 60,000 t/d processing rate:

- 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 indicated that less energy consumption would be expected if SAG mill discharges are classified by primary cyclones prior to ball mill grinding.

In 2012, Contract Support Services conducted a few of the similar SABC simulations on the average data obtained. The simulations produced similar results as produced in 2008.

Grindability/Crushability Determination and Comminution Circuit Simulation – HPGR

In 2009 and 2010, two separate HPGR comminution characteristic testing programs were performed – bench scale testing at SGS and pilot plant scale tests at Köeppern's HPGR pilot plant at UBC.

The bench scale LABWAL tests by SGS were conducted on the Mitchell and Sulphurets composite samples. The tests included batch tests and locked cycle tests (LCT). The test results indicate that the Sulphurets mineralization is harder with respect to HPGR crushing than the Mitchell mineralization. On average, the net specific energy requirement is 2.33 kWh/t for the Mitchell sample and 3.08 kWh/t for the Sulphurets sample. The LCT results, including specific grinding force (N/mm^2) and specific throughput rate (ts/hm^3-(m_c)), are summarized in Table 13.20.

Table 13.20 HPGR Average Test Results – LCT, Mitchell, 2009/2010

Parameter	Unit	Mitchell	Sulphurets
Operation			
Pressure of Operation	bar	65	66
Moisture	% H_2O	1.8	1.7
Dry Net Throughput	t/h	1.9	1.6
Circulating Load	%	34.7	47.1
Net Power	kW	4.4	5.1
Gross Specific Energy Requirement	kWh/t	2.96	3.80
Net Specific Energy Requirement	kWh/t	2.33	3.08
HPGR Product Analysis			
50% Passing	µm	694	1,046
80% Passing	µm	1,988	2,220
Percent Passing 100 Mesh	%	25.3	17.7
Percent Passing 6 Mesh	%	100	100
Flake Thickness	mm	6.0	5.8
Performance Indicators			
Specific Grinding Force	N/mm^2	3.24	3.31
Specific Throughput	ts/hm^3-(m_f)	226	213
Specific Throughput Rate	ts/hm^3-(m_c)	195	187
Ratio mj/mf		0.86	0.88
Specific Power	kWs/m^3	528	657
New minus 100 Mesh Produced	%	19.6	11.9
New minus 6 Mesh Produced	%	73.5	60.6

Based on the test results, SGS also conducted related simulations to size the HPGR.

Köeppern conducted a pilot plant test at its HPGR pilot plant at UBC using approximately 5.5 t drill core samples collected from the Mitchell deposit. The pilot plant HPGR rollers are 0.75 m in diameter and 0.22 m in width. The test report made the following main observations:

- Significant size reduction was achieved in comparison to the other materials tested previously by this laboratory.
- A specific pressing force of 4 N/mm^2 was considered to be optimum on the basis of both size reduction and throughput performance.
- Varying roll speed did not produce a significant impact on HPGR performance.

- An increase in feed moisture had a negative impact on throughput and energy consumption. An increase in feed moisture from 0.4% to 5% resulted in a 56% increase in net specific energy consumption.
- Variation in feed top size did not produce a significant difference in 50% passing particle size of HPGR product.
- Higher HPGR throughputs were achieved with closed circuit tests than with single pass tests at the equivalent machine operating conditions.
- A lower net specific energy consumption (approximately 1.94 kWh/t) was recorded for the closed circuit tests, in comparison with 1.99 kWh/t obtained from the single pass tests.

The typical LCT data are provided in Figure 13.6 and Figure 13.7.

Figure 13.6 HPGR Net Specific Energy Consumption vs. Cycle Number, 2010

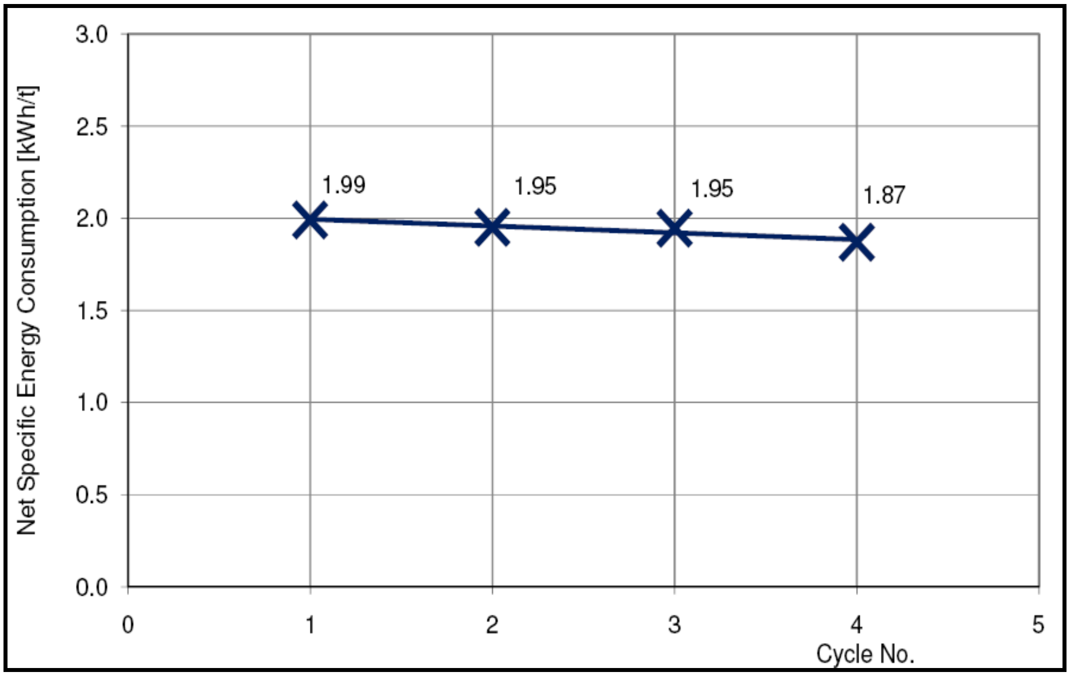
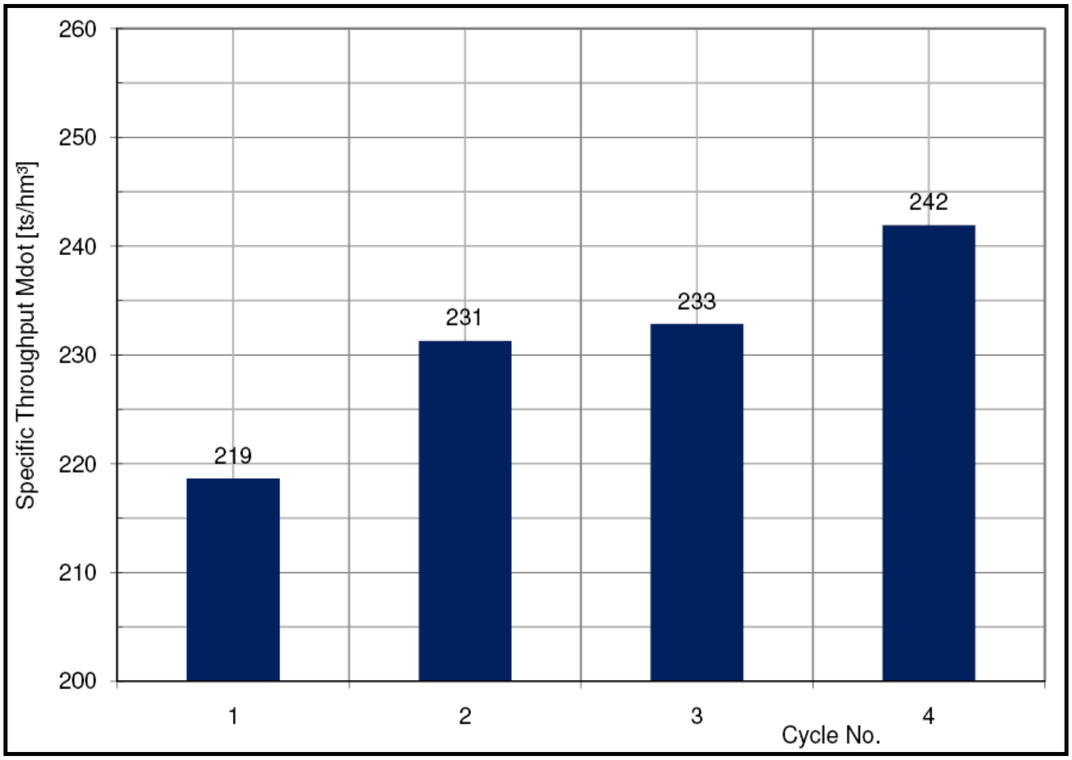


Figure 13.7 Specific Throughput (ts/hm^3) vs. Cycle Number, 2010



The HPGR test work program showed that the Mitchell material is very amenable to the HPGR crushing process. Köeppern's test work report indicates that the results from the program are sufficient for sizing HPGR units and their motors.

SGS performed a preliminary HPGR/ball mill circuit design based on a total production rate of 120,000 t/d and the test results from the bench scale LABWAL test results. The configurations of the crushing and grinding circuit for the Mitchell and Sulphurets ores are summarized in Table 13.20. It appears that processing of Mitchell ore would require four 7.9 ft diameter x 5.4 ft long HPGR crushers, while processing the harder Sulphurets ore on its own, would require five of the same size HPGR crushers.

Table 13.21 Simulation Results – HPGR/Ball Circuit, SGS (2009/2010)

	Mitchell		Sulphurets	
	HPGR	**Ball Mill**	**HPGR**	**Ball Mill**
Crusher/Mill Dimensions				
Train Number	4	4	5	5
Nominal Dimension	7.9' x 5.4'	23.5' x 40'	7.9' x 5.4'	23.5' x 40'
Mill Speed (RPM)	16.9	12.0	15.9	12.0
% of Critical Speed (%)	-	75	-	75
Grinding Steel Balls				
Design Ball Charge (% vol.)	-	29	-	33
Maximum Ball Charge (%)	-	34	-	34
Motor				
Design Power (kW)	15,816	40,759	23,465	57,293
Total Installed Power (kW)	22,400	47,744	28,000	59,680
Classification				
Type	Screens	Hydrocyclones	Screens	Hydrocyclones
Circuit Performance				
Product Particle Size, P_{80} (µm)	1,988	180	2,220	180
Ind. Specific Power Req. (kWh/t)	2.9	7.5	4.3	10.5
Total Specific Power Req.(kWh/t)	10.4		14.8	

Grindability/Crushability Determination – Tower Mill

As a part of the 2009 testing program, Metso Minerals Industries Inc. (Metso) investigated the specific energy consumption for secondary grinding using tower mills. The mill feed particle size was 80% passing 173 µm and the product particle size was 125 µm. The test results indicate that the specific energy requirement for the grinding by a jar mill was 1.36 kWh/t for the Mitchell composite sample. As projected by Metso, the specific energy requirement by a stirred tower mill would be approximately 0.88 kWh/t for a similar particle size reduction.

Grindability/Crushability Determination – Regrinding/IsaMill™

SGS used the IsaMill™ procedure to determine the specific energy requirement for regrinding the gold-bearing pyrite rougher concentrate that was produced from the Mitchell samples. The tests indicated that the specific energy requirement to regrind the concentrate from 80% passing 66 µm to 80% passing 16 µm was 24.2 kWh/t. The grinding media consumption, 2 mm Keramax MT1 grind beads, was 6 g/kWh.

Process Flowsheet and Parameter Development

Substantial test work was conducted to develop the process flowsheet and to optimize the process conditions through various testing programs. A flotation-cyanidation combination process was developed for this mineralization. The process consists of:

- copper-gold-molybdenum bulk rougher flotation followed by gold-bearing pyrite flotation
- regrinding the resulting bulk rougher concentrate followed by three stages of cleaner flotation to produce a copper-gold-molybdenum bulk cleaner flotation concentrate
- molybdenum separation from the bulk cleaner flotation concentrate to produce a rhenium-bearing molybdenum concentrate and a copper/gold concentrate containing associated silver
- cyanide leaching of the gold-bearing pyrite flotation concentrate and the scavenger cleaner tailing product.

The development of the flotation and cyanidation test conditions is summarized in the following sections.

Flotation Tests

Flotation Parameter Development Tests

The tested process parameters for copper-gold-molybdenum bulk concentrate flotation and gold-bearing pyrite concentrate flotation include primary grind size, regrind size, slurry pH, and reagent regimes. After various tests, the following flotation conditions were developed for the LCTs in the most recent testing programs:

- primary grind size: 80% passing approximately 125 µm
- rougher flotation pH: 10
- bulk concentrate regrind size: 80% passing approximately 20 µm
- cleaner flotation pH: 11.5
- flotation reagent:
  - bulk flotation: dithiophosphinates (3418A) + dithiophospate (A208) + fuel oil
  - gold-bearing pyrite flotation: A208 + PAX.

The open circuit batch tests showed that the mineralization responded well to these flotation conditions.

The effect of primary grind size and regrind size on the metallurgical performance was evaluated using the QSP 0-30 and Hi Qtz 0-30 composites generated from the 2008 testing samples. The test results, as summarized in Figure 13.8 and Figure 13.9 show that copper and gold metallurgical performance in the rougher flotation stage improved with an increase in primary grind fineness, although far less significantly when the grind size was finer than 80% passing 120 µm.

Figure 13.8 Metallurgical Performance vs. Primary Grind Size – QSP 0-30, 2008 (G&T)

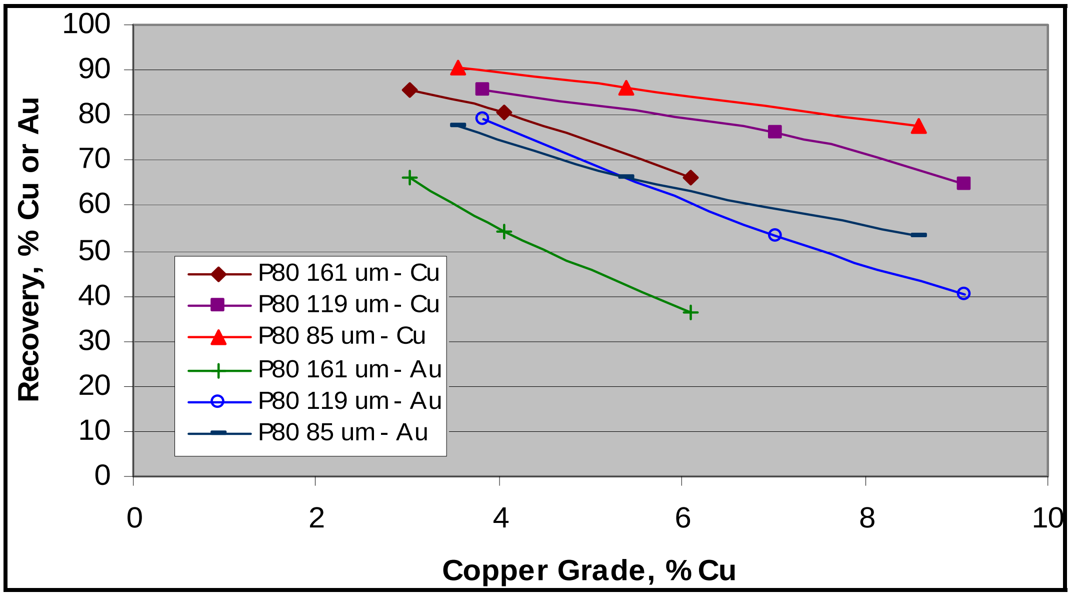
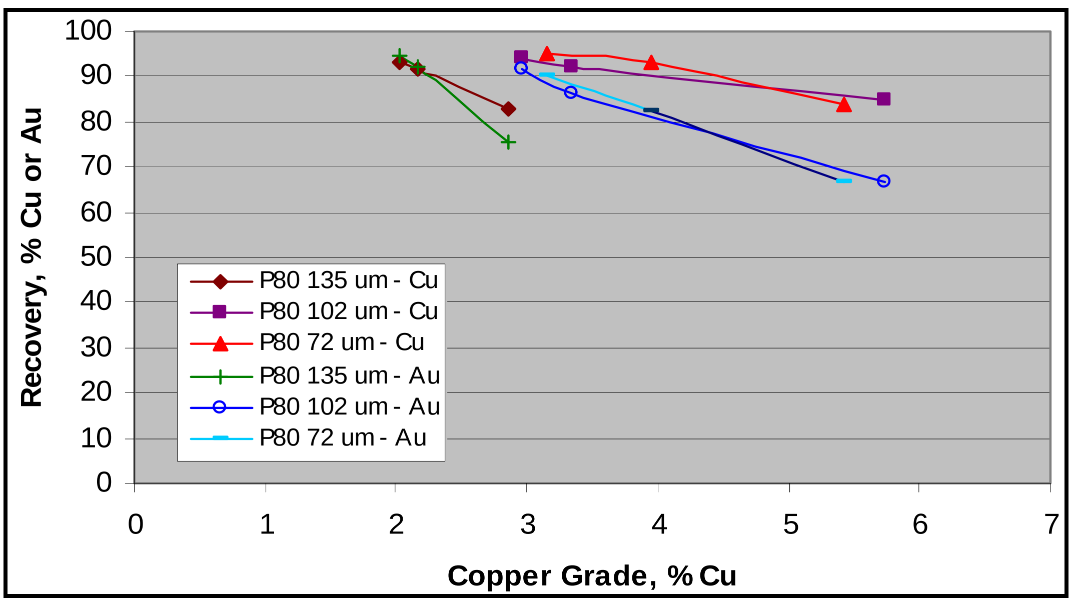


Figure 13.9 Metallurgical Performance vs. Primary Grind Size – Hi Qtz 0-30, 2008 (G&T)

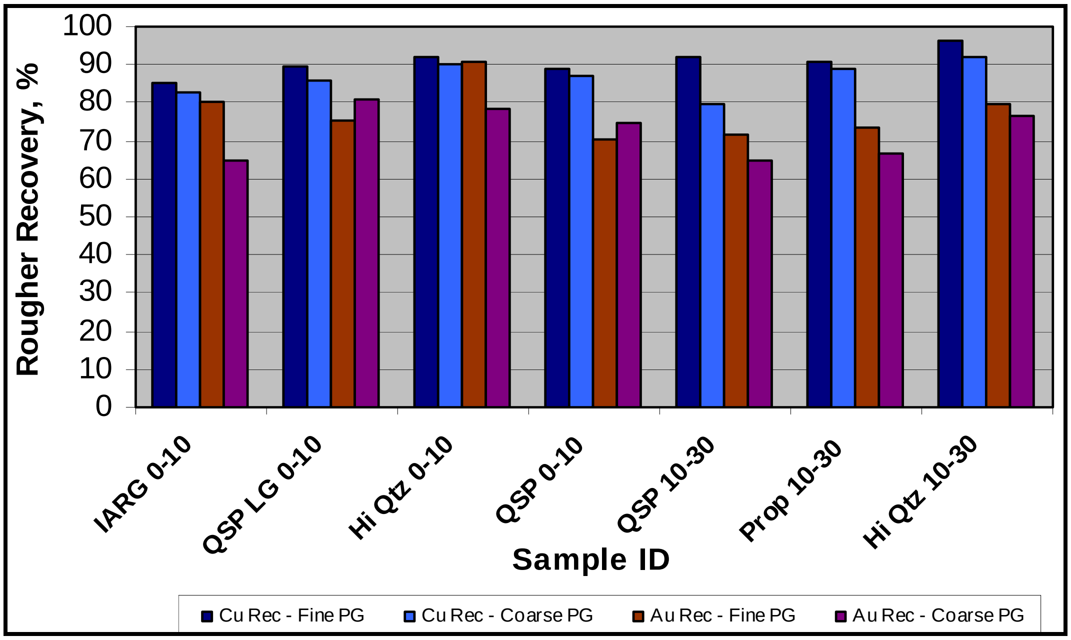


For QSP 0-30 composite, the copper recovery to a rougher concentrate, grading 4% Cu, improved from 81 to 89% when the primary grind size was decreased from 80% passing 161 µm to 80% passing 85 µm. Gold recovery increased significantly with the increase in the grind fineness; however, there was no significant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

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

Apart from QSP 0-30 and Hi Qtz 0-30 composites, G&T performed two sets of comparison tests to investigate the effect of primary grind size on copper and gold recovery from all the other composite samples generated for the 2008 testing program. 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 copper rougher concentrates is shown in Figure 13.10.

Figure 13.10 Effect of Primary Grind Size on Metallurgical Performance, 2008 (G&T)

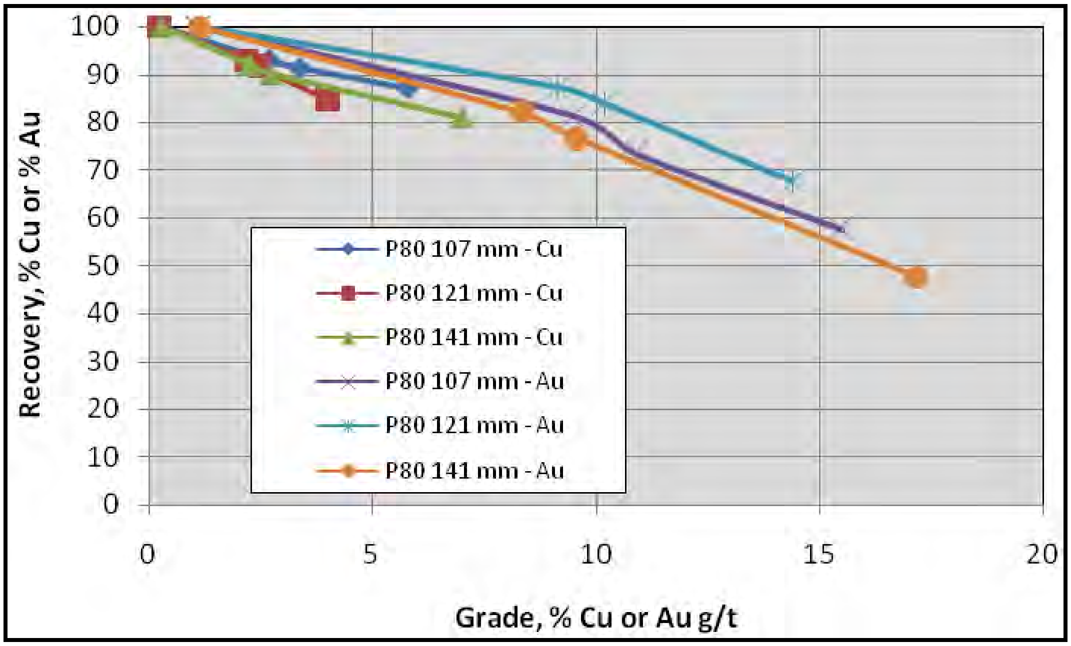


On average, the copper recovery reporting to copper rougher concentrate was 90.6% at the fine grind size, compared to 86.6% at the coarse grind size. The average gold recovery to the concentrate increased from 72.3 to 77.3%. However, QSP 0-10 and QSP LG 0-10 composites appeared to show different gold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

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

In the 2009 testing program, two sets of primary grind size confirmation tests were conducted on Composite 42 (QSP) and Composite 44 (Hi Qtz). The test results appear to indicate that the copper and gold metallurgical response of Composite 42 was not sensitive to primary grinding size changes within the range of 80% passing 100 µm and 141 µm. For Composite 44, the copper and gold recoveries to the rougher/scavenger concentrate at the grind size of 80% passing 100 µm were slightly higher than that at the grind sizes of 80% passing 125 and 165 µm. Test results are provided in Figure 13.11.

Figure 13.11 Effect of Primary Grind Size on Metal Recovery – Mitchell, 2009



Further tests on the pilot plant feed composites showed that the copper and gold recoveries were not very sensitive to the primary grind size between 80% passing 100 and 150 µm. However, metal recoveries reduced at primary grind sizes coarser than the 150 µm.

VARIABILITY TESTS

In the 2008 testing program, a total of 34 samples were used for variability tests, including two samples (Met 35 and Met 36) from the Sulphurets Zone. Primary grind sizes ranged from 80% passing 115 to 171 µm, averaging 149 µm. The rougher concentrate from the copper circuit was reground to approximately 80% passing18 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 13.12, G&T established the relationship between copper recovery and copper feed grade at a fixed concentrate grade of 25% Cu. The variation in the copper metallurgical performance of various mineral samples is shown in Figure 13.13.

Figure 13.12 Copper Recovery vs. Copper Feed Grade – Mitchell, 2008 (G&T)

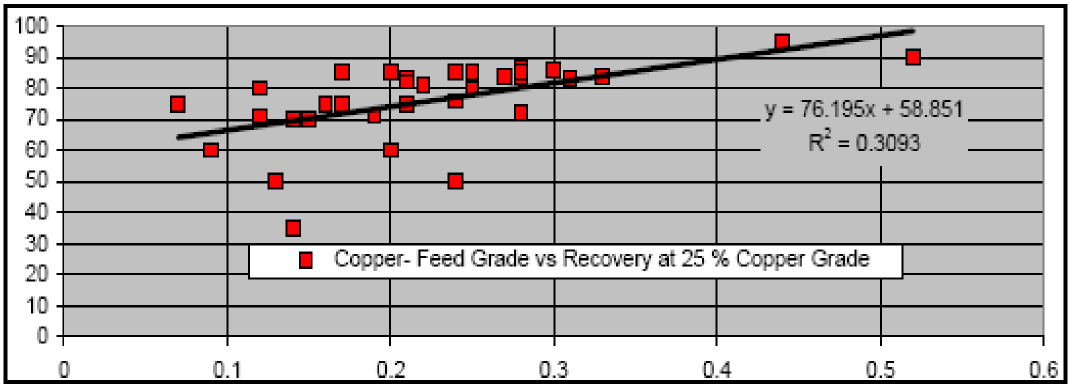
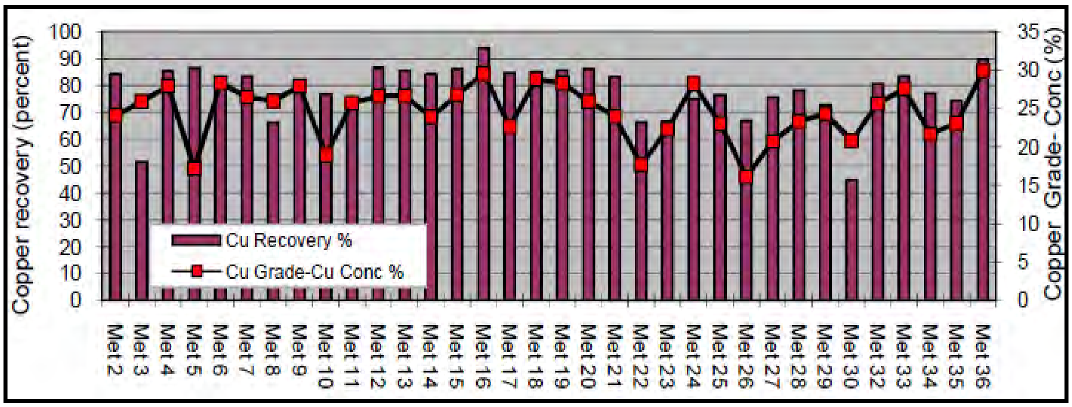
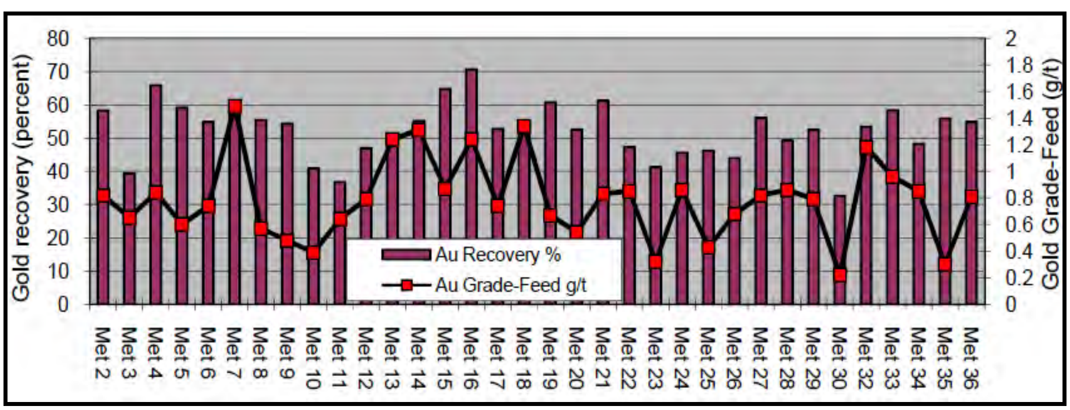


Figure 13.13 Copper Recovery & Concentrate Grade – Individual Samples, Mitchell, 2008 (G&T)



The gold recovery to the copper concentrate fluctuated from 30 to 70%. The tests seemed to show 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 13.14.

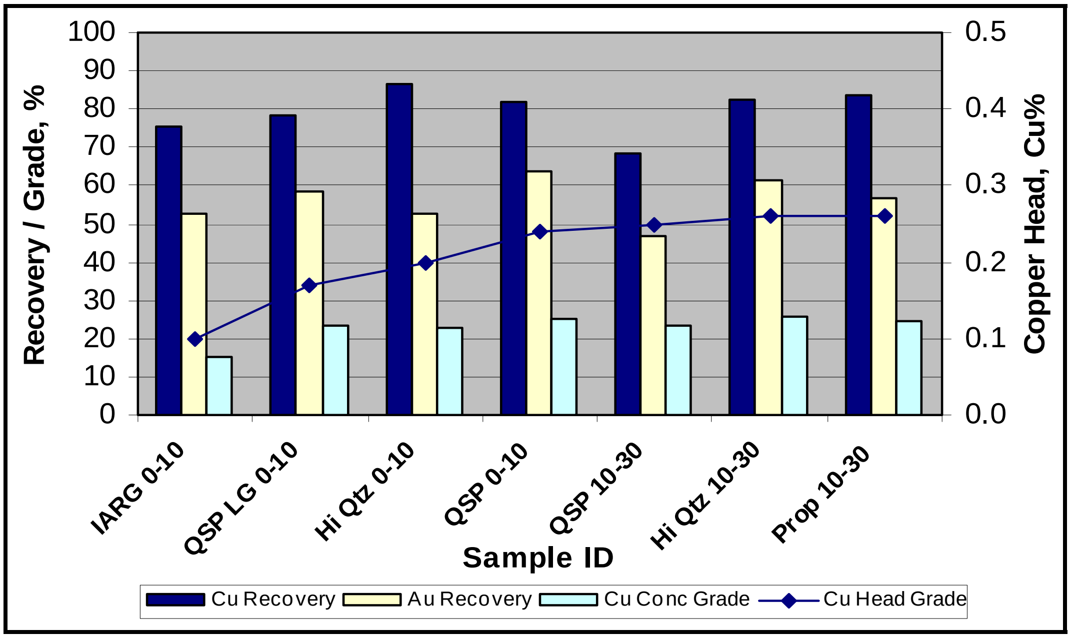
Figure 13.14 Gold Recovery & Feed Grade – Individual Samples, Mitchell, 2008 (G&T)



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

Further tests were conducted on seven composites representing the major Mitchell Zone mineralization types projected to be mined during various operating periods. The test results are shown in Figure 13.15. At primary grind sizes ranging from 130 to 168 µm, the third cleaner concentrates from the open batch flotation tests produced between 69 and 86% copper recovery and between 47 and 64% gold recovery.

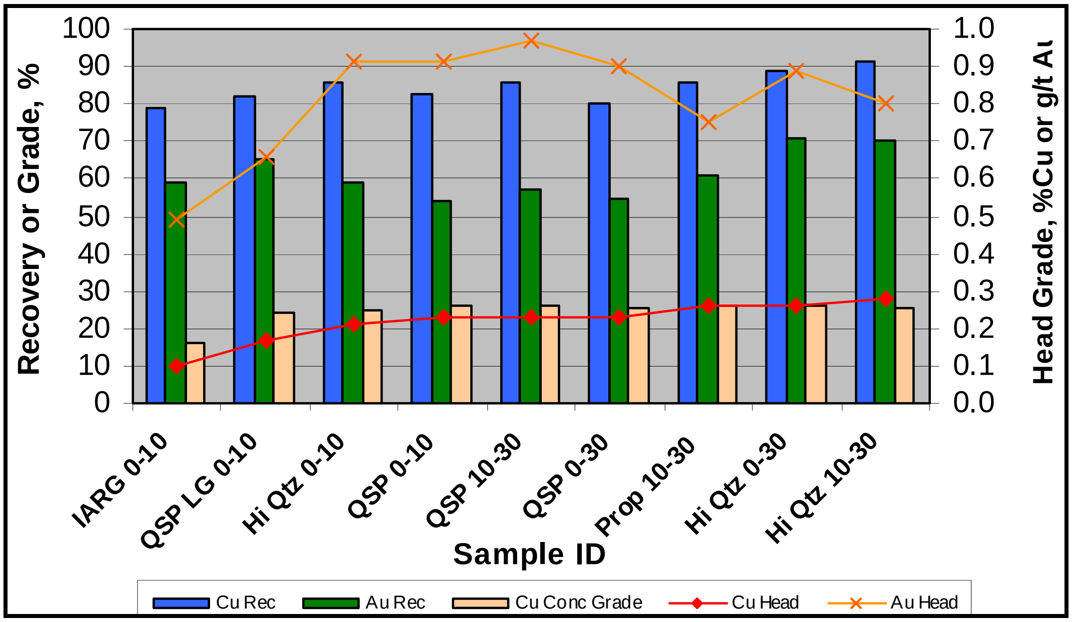
Figure 13.15 Metallurgical Performance – Composites, Mitchell, 2008 (G&T)



Similar to the MET sample variability tests, the total average gold recovery from the copper-gold rougher and scavenger flotation was approximately 86% from the composite samples.

Open circuit tests with two stages of cleaner flotation at a pH of 11.5 were also performed on the nine composite samples. Primary grind sizes ranged from 80% passing 87 µm to 137 µm, averaging 119 µm. Regrind sizes varied from 80% passing 12 µm to 22 µm, averaging 18 µm. The results are shown in Figure 13.16.

Figure 13.16 Metallurgical Performance – Open Circuit Tests, Mitchell, 2008 (G&T)



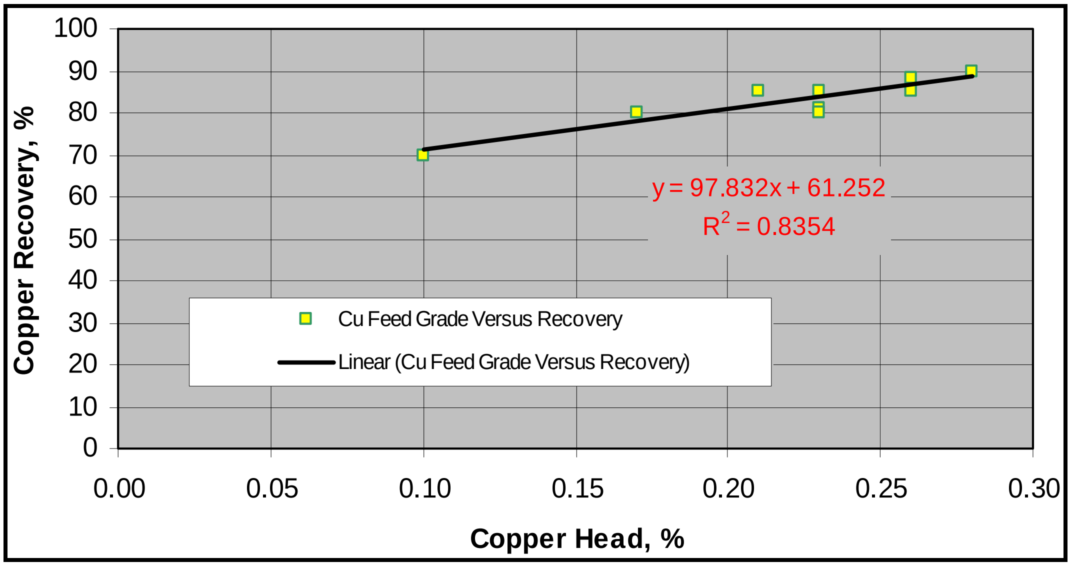
The second cleaner concentrate recovered between 79 to 91% of the copper and 54 to 71% of the gold from all the nine composites. On average, the metal recovery was 84.6% for copper and 61.2% for gold.

The results appeared to indicate that copper recovery increased with an increase in copper head grade. The test results also showed that gold recovery to the copper concentrate did not appear 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 IARG 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 13.17. The graph indicates that increasing copper recovery is related to increasing copper head grade.

Figure 13.17 Copper Recovery vs. Copper Feed – Open Circuit Tests, 2008



The 2009/2010 flotation test work continued with further bench open circuit tests on the composite samples. The reagents used included 3418A and A208 for copper-gold flotation, fuel oil for molybdenum flotation, and the combination of PAX and A208 for gold-bearing pyrite flotation. Lime was used to regulate the slurry pH to approximately 10.0 at the copper-gold rougher flotation stage and pH 11.5 for the copper-gold cleaner flotation. The gold-bearing pyrite flotation was performed at an unadjusted pH value of approximately 9.5.

The results from the testing program are summarized in Figure 13.18 and Figure 13.19. The results indicate some significant variation in the metallurgical performance between the different ore samples. The BBRX mineralization (Composite 41) showed the best metallurgical response to the flowsheet. This was most likely due to the much higher feed grade of this composite. Compared to the 2008 Hi Qtz mineralization test results, the Hi Qtz mineralization (Composite 44) produced a slightly lower level of metallurgical performance.

Figure 13.18 Copper Metallurgical Performance – Mitchell, 2009 (G&T)

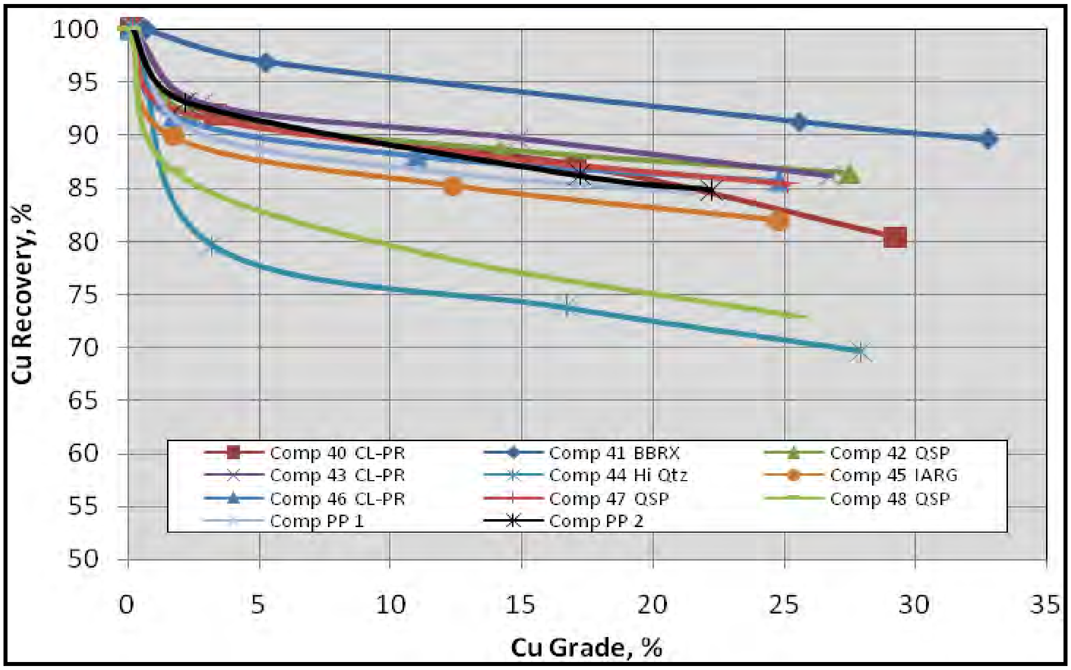
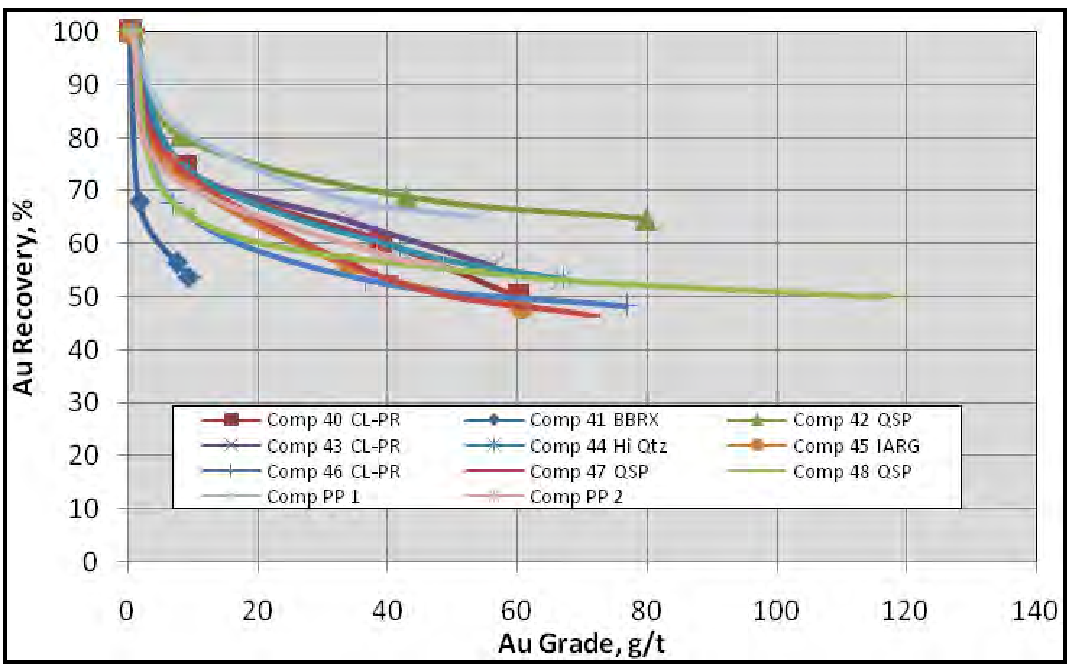


Figure 13.19 Gold Metallurgical Performance – Mitchell, 2009 (G&T)

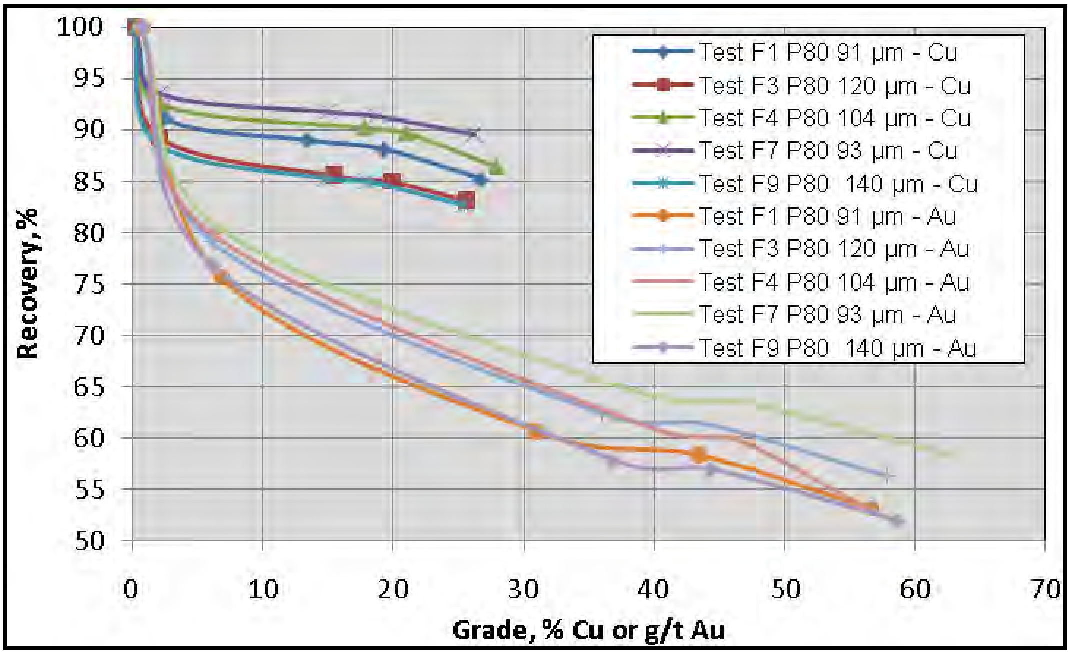


The results also show that most of the cleaner concentrate grades of the individual composites were greater than or close to 25% Cu, averaging 28% Cu. However, the Composites PP 1 and PP 2 produced lower grade concentrates containing 22% Cu.

The average copper recovery was 83%. The average gold recovery to the final copper concentrates was 55%.

In the 2009/2010 testing program, SGS also conducted batch open cycle tests on Composite PP1 and used a flotation flowsheet similar to the one developed by G&T. In the test, SGS added carboxymethyl cellulose (CMC) into cleaner flotation to suppress clay minerals and diesel fuel was added as a molybdenum collector. The SGS data in Figure 13.20 indicates that the effect of primary grind size on the copper and gold metallurgical performance is not very significant.

Figure 13.20 Metallurgical Performance – Mitchell, 2009/2010 (SGS)



The test results from the 2011/2012 testing programs confirmed the findings obtained from the previous metallurgical performance test programs. The test results are summarized in Figure 13.21 to Figure 13.24.

Figure 13.21 Copper Metallurgical Performance – Mitchell, 2011/2012 (G&T, KM 3080)

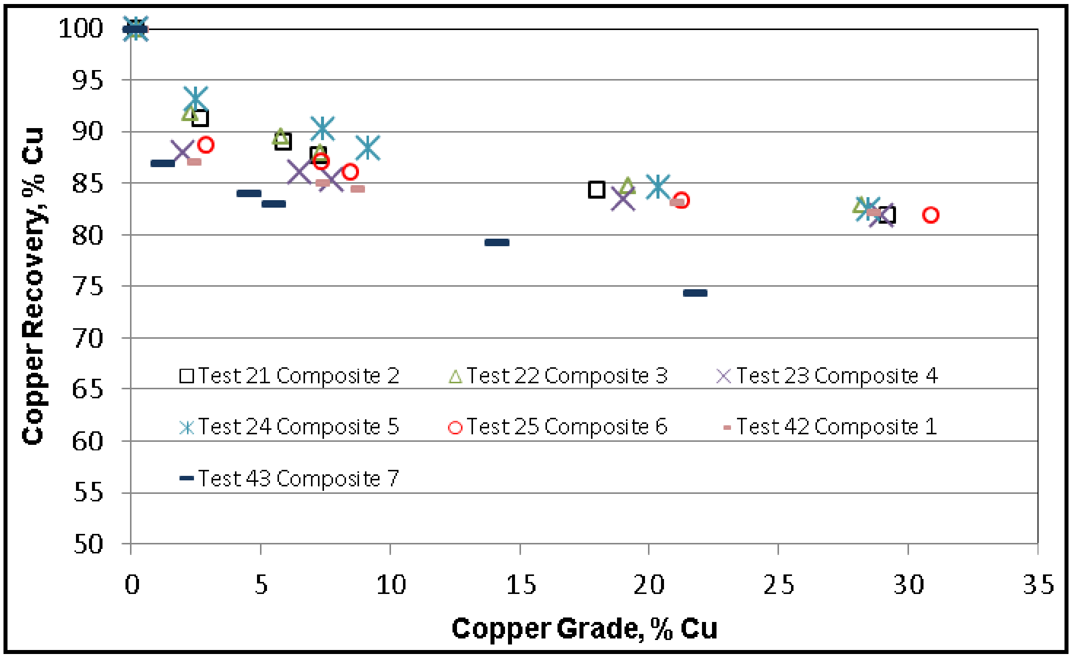


Figure 13.22 Gold Metallurgical Performance – Mitchell, 2011/2012 (G&T, KM 3080)

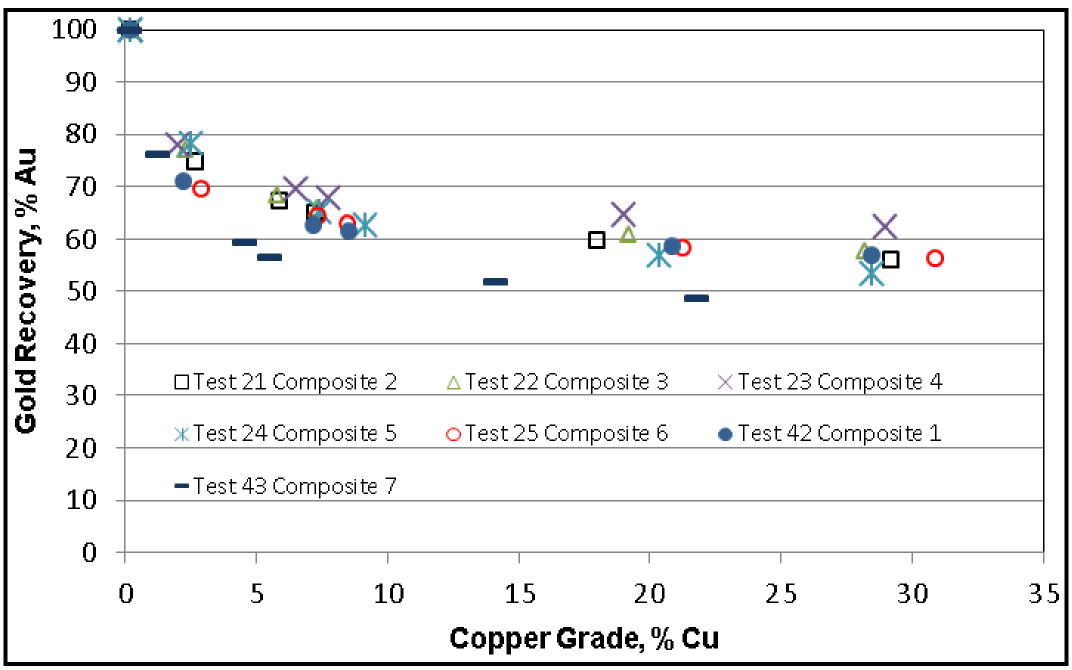


Figure 13.23 Copper Metallurgical Performance – Mitchell, 2011/2012 (G&T, KM 3174)

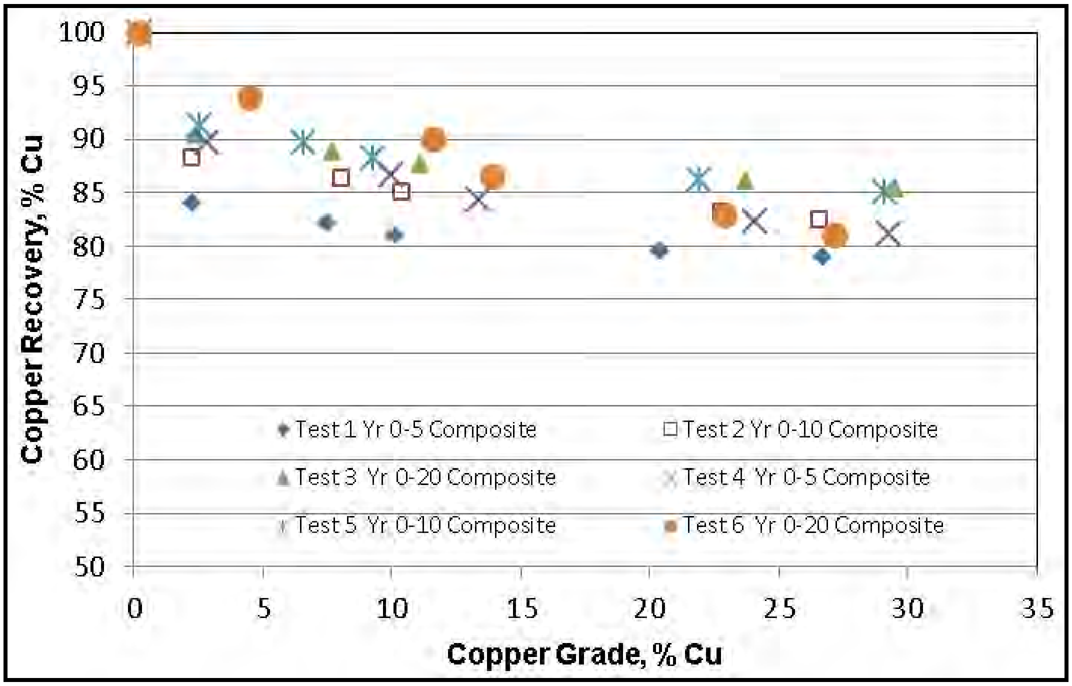
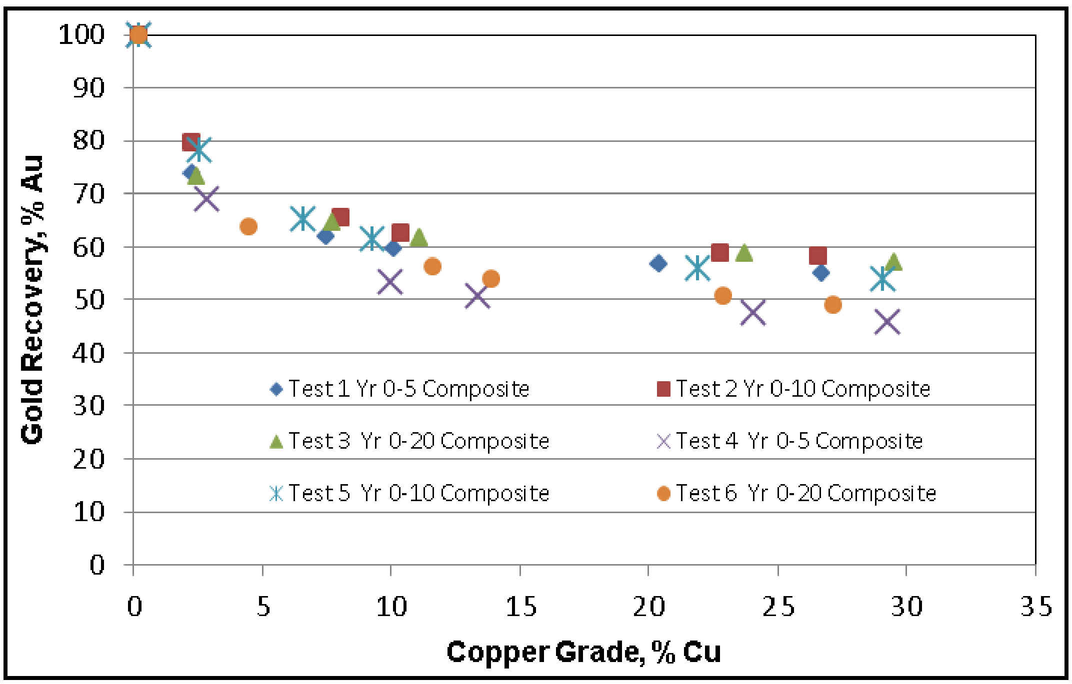


Figure 13.24 Gold Metallurgical Performance – Mitchell, 2011/2012 (G&T, KM 3174)



LOCKED CYCLE TESTS

Fourteen LCTs have been conducted on the various composite samples generated from the various testing programs since 2007. The test results are summarized in Table 13.22 for the Mitchell mineralization and in Table 13.23 for Mitchell mineralization samples blended with the other mineralization.

The test results showed a substantial variation in the concentrate grade, ranging from 20% Cu to 30% Cu. On average, the final copper concentrate contained 24.9% Cu. The average recoveries to the concentrate were 84.9% for copper, 60% for gold, 50% for silver, and 54% for molybdenum. Approximately 26% of the gold and 29% of the silver in the feed reported to other gold-bearing products, which were further extracted by cyanide leaching. The test results showed that better metallurgical performances were achieved in the more recent testing programs.

Table 13.23 shows the effect of blending the Mitchell sample with the samples from the other mineralized zones. Metallurgical performance of the blended samples appears comparable to that produced when treating the Mitchell material on its own.

Table 13.22 Locked Cycle Test Results – Mitchell

Test Program	Comp	Product	Grind Size (P80 µm*)	Mass (%)	Grade				Flotation Recovery (%)			
					Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu	Au	Ag	Mo
G&T 2153/141	Master	Head		100.0	0.21	0.89	4.2	-	100.0	100.0	100.0	-
		Cu/Mo Concentrate	119/16	0.9	20.2	62.8	273	-	87.8	63.0	58.5	-
		Bulk Cleaner Tailing		7.0	0.10	1.66	-	-	3.3	13.0	-	-
		Au-Pyrite Concentrate		5.6	0.10	2.02	-	-	2.6	12.7	-	-
G&T 2153/142	Master	Head		100.0	0.21	0.92	3.7	-	100.0	100.0	100.0	-
		Cu/Mo Concentrate	119/17	0.8	22.0	64.7	242	-	87.0	58.5	52.5	-
		Bulk Cleaner Tailing		6.9	0.14	2.08	-	-	4.5	15.7	-	-
		Au-Pyrite Concentrate		6.0	0.11	2.25	-	-	3.0	14.6	-	-
G&T 2344/73	PP Comp 1	Head		100.0	0.24	0.81	-	-	100.0	100.0	-	-
		Cu/Mo Concentrate	103/14	1.0	22.3	55.7	-	-	89.3	66.2	-	-
		Bulk Cleaner Tailing		6.8	0.13	1.70	-	-	3.7	14.0	-	-
		Au-Pyrite Concentrate		2.5	0.13	1.80	-	-	1.4	5.5	-	-
G&T 2535/18	PP Comp 1	Head		100.0	0.23	0.84	4.0	-	100.0	100.0	100.0	-
		Cu/Mo Concentrate	103/16	0.7	28.0	77.8	260	-	87.2	67.4	47.0	-
		Bulk Cleaner Tailing		7.4	0.19	1.62	17.6	-	6.0	14.2	32.0	-
		Au-Pyrite Concentrate		2.5	0.19	1.37	7.1	-	2.0	4.1	4.4	-
G&T 2535/20	PP Comp 1	Head		100.0	0.24	0.82	3.9	-	100.0	100	100.0	-
		Cu/Mo Concentrate	137/17	0.9	23.8	62.0	248	-	88.1	66.2	55.6	-
		Bulk Cleaner Tailing		7.4	0.10	1.61	11.3	-	2.9	14.4	21.2	-
		Au-Pyrite Concentrate		2.8	0.21	1.69	7.2	-	2.4	5.6	5.1	=
G&T 2670/12	PP Comp 3	Head		100.0	0.20	0.74	3.2	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate	147/15	0.6	30.1	77.7	264	0.386	84.2	58.0	52.6	35.7
		Bulk Cleaner Tailing		6.2	0.19	1.49	-	0.036	6.0	12.5	-	37.9
		Au-Pyrite Concentrate		4.9	0.12	2.04	-	0.014	3.1	13.6	-	11.6

table continues...

Test Program	Comp	Product	Grind Size (P80 µm*)	Mass (%)	Grade				Flotation Recovery (%)			
					Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu	Au	Ag	Mo
G&T 2670/18	PP Comp 3	Head		100.0	0.20	0.79	3.2	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate	147/22	0.6	27.4	70.5	272	0.462	86.1	56.5	53.0	49.7
		Bulk Cleaner Tailing		6.0	0.13	1.98	9.3	0.016	3.9	15.1	17.4	15.8
		Au-Pyrite Concentrate		4.4	0.15	2.26	6.4	0.016	3.4	12.7	8.8	11.7
G&T 2670/22	PP Hi Mo	Head		100.0	0.16	0.60	3.3	0.014	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate	143/21	0.6	22.4	61.7	243	1.200	78.9	56.9	43.8	47.9
		Bulk Cleaner Tailing		6.6	0.17	1.87	10.0	0.042	7.3	20.6	19.8	19.9
		Au-Pyrite Concentrate		5.6	0.16	1.39	6.9	0.026	5.7	12.9	11.6	10.2
G&T 2670/26	BS Hi Mo	Head		100.0	0.12	0.55	2.4	0.010	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate	143/17	0.3	24.9	70.3	185	1.258	71.5	43.2	26.0	42.2
		Bulk Cleaner Tailing		5.8	0.27	1.58	9.7	0.049	13.3	16.6	23.4	28.1
		Au-Pyrite Concentrate		5.7	0.13	1.79	5.5	0.026	6.0	18.3	13.1	14.5
G&T 2897/01	Comp 46 of KM2344	Head		100.0	0.15	0.65	2.3	0.012	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate	120/16	0.6	22.6	80.5	226	1.759	89.1	73.5	58.6	86.3
		Bulk Cleaner Tailing		7.6	0.04	1.01	4.6	0.008	2.1	11.8	15.3	5.1
		Au-Pyrite Concentrate		5.6	0.09	1.16	2.9	0.003	3.3	10.0	7.2	1.4
G&T 3081/93	Mitchell Yr 0-10	Head	137/18	100.0	0.20	0.65	4.7	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.6	27.1	58	427	0.33	83.3	56.3	57.1	55.6
		Bulk Cleaner Tailing		7.4	0.20	2.12	12.0	0.011	7.3	24.2	19.0	22.0
		Au-Pyrite Concentrate		6.4	0.10	1.01	4.0	0.005	3.3	10.1	5.5	8.2
G&T 3081/82	Mitchell Yr 0-20	Head	123/22	100.0	0.21	0.57	3.5	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.8	23.8	44.2	223	0.24	88.2	59.9	49.5	49.2
		Bulk Cleaner Tailing		7.8	0.13	1.54	9.0	0.015	5.0	21.0	20.0	30.6
		Au-Pyrite Concentrate		5.0	0.11	1.0	4.0	0.005	2.6	8.8	5.8	6.2

table continues...

Test Program	Comp	Product	Grind Size (P80 µm*)	Mass (%)	Grade				Flotation Recovery (%)			
					Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu	Au	Ag	Mo
G&T 3081/103	Mitchell Yr 0-20	Head	123/17	100.0	0.22	0.55	4.0	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.6	29.8	56	299	0.267	76.7	57.8	43.0	26.3
		Bulk Cleaner Tailing		8	0.32	1.51	14	0.039	11.6	22.0	28.4	54.0
		Au-Pyrite Concentrate		4.4	0.18	1.02	6	0.006	3.6	8.1	6.7	4.8
SGS	PP Comp 1	Head		100.0	0.21	0.72	-	0.005	100.0	100.0	-	100.0
		Cu/Mo Concentrate	129/28	0.8	23.1	53.7	-	0.410	89.0	59.6	-	65.0
		Bulk Cleaner Tailing		9.2	0.06	1.54	-	0.009	2.62	19.8	-	13.2
		Au-Pyrite Concentrate		5.8	0.09	0.81	-	0.013	2.60	6.6	-	12.0

* primary grind size/regrind size.

Table 13.23 Locked Cycle Test Results – Blended Samples

Test Program	Comp	Product	Grind Size (P80 µm*)	Mass (%)	Grade				Flotation Recovery (%)			
					Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu	Au	Ag	Mo
G&T 2535/19	Mitchell (PP Comp1)/ Kerr (52/53 Blend); 80%:20%	Head		100.0	0.31	0.70	3.5	-	100.0	100.0	100.0	-
		Cu/Mo Concentrate	127/20	1.1	25.3	40.0	168	-	87.4	60.4	51.4	-
		Bulk Cleaner Tailing		8.0	0.12	1.36	8.2	-	3.2	15.5	18.9	-
		Au-Pyrite Concentrate		4.2	0.24	0.94	5.9	-	3.3	5.7	7.1	-
G&T2670/62	Mitchell/Sulphurets Blend; 60%:40%	Head		100.0	0.22	0.67	2.8	0.007	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate	141/22	0.8	24.2	52.0	178	0.664	85.9	59.8	50.9	72.4
		Bulk Cleaner Tailing		8.6	0.09	1.40	5.6	0.008	3.6	18.1	17.2	9.7
		Au-Pyrite Concentrate		3.9	0.19	1.47	4.9	0.010	3.5	8.6	6.8	5.5
G&T 2748/18	Mitchell (PP Comp 1)/ Iron Cap C1/Iron Cap C2; 33%:33%:33%	Head		100.0	0.24	0.79	-	0.004	100.0	100.0	-	100.0
		Cu/Mo Concentrate	135/15	0.8	27.6	59.6	-	0.250	87.8	58.2	-	51.5
		Bulk Cleaner Tailing		8.2	0.09	1.52	-	0.010	2.9	15.7	-	20.7
		Au-Pyrite Concentrate		7.4	0.13	1.85	-	0.003	4.0	17.4	-	5.4

* primary grind size/regrind size.

Pilot Plant Tests

In the 2009 testing program, G&T carried out initial pilot plant tests using approximately 5 t of coarsely crushed drill core. Compared to the bench LCTs, the pilot plant tests produced lower metal recoveries and concentrate grades.

Copper recovery on the PP1 sample averaged 72% into an 18% copper final concentrate. Test P2 produced a 23.9% Cu concentrate. G&T indicated that the low copper recovery might have resulted from pilot plant control or circuit stability issues. This in turn caused copper losses into the pyrite circuit and the first cleaner tailing. These initial pilot plant results are summarized in Table 13.24.

Table 13.24 Pilot Plant Test Results – Mitchell, 2009 (G&T)

Test	Grind Size (P80 µm*)	Grade			Recovery (%)			
		Cu (%)	Au (g/t)	Mo (%)	Mass	Cu	Au	Mo
Composite PP1 (Head Assay: 0.24% Cu, 0.76 g/t Au, 0.004% Mo)								
P1	144	17.1	33.6	0.15	1.0	65.4	46.1	31.5
P2	96	23.9	59.6	0.17	0.7	65.2	51.9	23.6
P3	104	16.3	35.7	0.14	1.3	80.2	58.6	40.8
P4	103	15.5	29.8	0.03	1.2	74.3	50.7	8.8
P5	97	18.4	41.4	0.12	0.9	76.0	52.3	26.4
Average	**109**	**18.2**	**40.0**	**0.12**	**1.0**	**72.2**	**51.9**	**26.2**
Composite PP2 (Head Assay: 0.18% Cu, 0.61 g/t Au, 0.010% Mo)								
P6	84	16.7	33.0	0.70	1.0	79.7	50.3	54.8
P7	91	17.7	42.5	0.95	0.9	81.7	60.5	72.3
P8	88	18.0	36.9	0.81	0.9	79.1	47.4	65.8
Average	**88**	**17.4**	**37.5**	**0.8**	**0.9**	**80.2**	**52.7**	**64.3**

* primary grind size.

In the 2010 testing program, G&T further conducted two pilot plant runs on the PP Composite 3 and the PP Hi-Mo Composite samples. Compared to the 2009 pilot plant tests, the 2010 testing program produced much better metallurgical performances. The flowsheet used for the pilot plant tests is shown in Figure 13.25. The pilot test results are provided in Table 13.25.

Figure 13.25 Pilot Plant Test Flowsheet, 2010 (G&T)

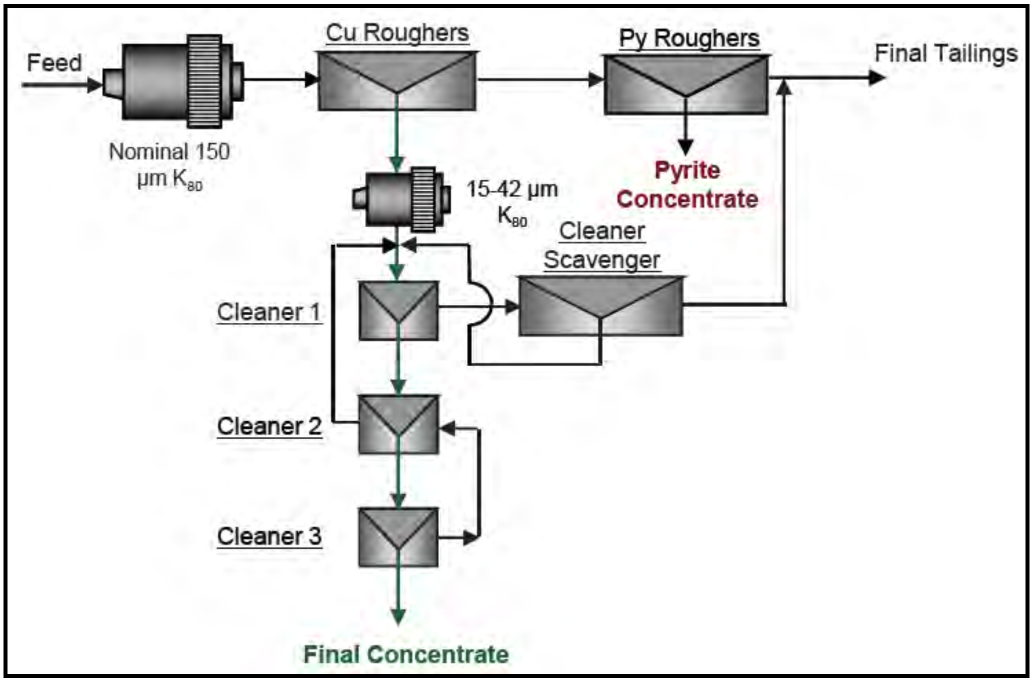


Table 13.25 Pilot Plant Test Results - Mitchell, 2010 (G&T)

Test	Grind Size (P80 µm*)	Grade				Recovery (%)				
		Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Mass	Cu	Au	Ag	Mo
Composite PP3 (Head Assay: 0.20% Cu, 0.79 g/t Au, 3.2 g/t Ag, 0.006% Mo)										
P1	115/16	26.4	62.0	482	0.43	0.7	83.0	50.2	53.1	43.2
		25.2	62.5	382	0.26	0.6	79.2	50.9	54.8	29.0
P2	153/22	25.7	58.7	278	0.32	0.6	74.6	44.6	45.6	27.7
		26.6	69.8	295	0.45	0.5	71.2	45.9	43.9	31.4
		27.2	80.2	316	0.59	0.4	61.2	44.2	39.8	31.5
		26.9	72.3	262	0.26	0.5	69.8	43.5	40.0	22.1
P3	152/23	25.4	64.6	239	0.35	0.6	71.3	54.4	39.1	29.9
		24.3	62.4	240	0.24	0.7	79.2	52.1	49.6	28.5
		25.3	56.2	182	0.27	0.6	81.6	51.4	42.6	29.1
		25.5	58.8	220	0.32	0.6	79.3	52.9	47.2	37.1
P4	143/22	24.8	58.7	268	0.32	0.6	72.6	47.0	45.4	32.3
		26.4	63.8	280	0.33	0.7	80.3	50.8	50.1	32.8
		24.5	64.6	236	0.51	0.8	84.1	65.3	51.7	47.3
		23.6	64.7	215	0.41	0.6	81.8	56.7	44.8	41.4
Average		**25.6**	**64.2**	**278**	**0.36**	**0.6**	**76.4**	**50.7**	**46.3**	**33.1**

table continues...

Test	Grind Size (P80 µm*)	Grade				Recovery (%)				
		Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Mass	Cu	Au	Ag	Mo
PP Hi-Mo Composite (Head Assay: 0.16% Cu, 0.6 g/t Au, 3.2 g/t Ag, 0.012% Mo)										
P5	163/28	22.0	52.1	244	0.31	0.7	77.8	47.3	52.5	33.1
		25.1	67.7	248	0.31	0.4	71.8	45.8	38.5	20.6
		19.3	61.8	276	0.71	0.7	81.5	66.6	59.6	41.6
		20.3	47.2	253	1.20	0.7	78.6	48.5	52.4	63.6
P6	146/21	18.9	56.7	239	0.91	0.6	78.0	54.8	49.9	43.2
		18.2	58.2	247	1.27	0.7	80.5	60.3	54.3	60.9
		20.5	57.8	246	1.21	0.6	80.1	58.3	50.3	60.6
		20.7	57.8	236	1.28	0.6	82.2	58.5	50.6	59.7
P7	143/22	19.7	67.9	259	1.27	0.6	78.9	59.7	51.5	66.8
		20.0	55.4	260	1.38	0.7	80.6	58.5	51.3	70.4
Average		**20.5**	**58.3**	**251**	**0.99**	**0.6**	**79.0**	**55.8**	**51.1**	**52.1**

* primary grind size/regrind size.

For the PP Composite 3, the pilot plant test showed variable results throughout the run period and, on average, did not achieve results as good as from an LCT on the same sample. Copper recoveries were calculated at various intervals during the operating period and ranged from 61 to 84%. The concentrate produced from the pilot plant run averaged 25.6% Cu. It was noted that the metallurgical performance observed from the best pilot plant results was close to the results achieved in the locked cycle testing.

For the PP Hi-Mo Composite, the copper recovery reporting to the final bulk concentrate ranged from 72 to 82% during the test. The copper concentrate produced ranged from 18.2 to 25.1% Cu. The metallurgical performance of the pilot plant was very similar to the performance obtained from a LCT on the same sample.

On average, approximately 50% of the silver in feed was recovered to the copper concentrate for both composites. The average silver concentration in the concentrate was approximately 250 g/t.

The molybdenum recovery into the final bulk concentrate was 52% for Hi-Mo Composite and 33% for Composite PP3.

G&T conducted bulk mineral analysis (BMA) using QEMSCAN on the blended bulk concentrates produced in pilot runs P2, P3, and P5. The results of the BMA analyses are shown in Table 13.26.

Table 13.26 Bulk Concentrate Mineralogy – Mitchell, 2010 (G&T)

Minerals	Mineral Content (%)		
	P2	**P3**	**P5**
Chalcopyrite	77.8	67.3	61.7
Bornite	0.3	0.4	0
Covellite	0.3	0.5	0.5
Enargite	0.2	0.3	0.7
Tennantite	0.2	0.4	0.6
Molybdenite	1.0	0.8	1.2
Galena	0.6	0.3	1.2
Sphalerite	0.7	1.2	1.3
Pyrite	12.0	11.8	18.9
Iron Oxides	0.3	0.4	0.2
Quartz	2.7	7.7	6.3
Micas	2.3	2.8	1.8
Feldspars	0.6	2.4	2.5
Kaolinite	0.1	0.2	0.4
Ti Mineral Group	0.3	1.0	0.7
Apatite	0.1	0.3	0.2
Others	0.5	2.2	1.6
Total	**100**	**100**	**100**

COPPER-GOLD AND MOLYBDENUM SEPARATION TESTS

In the 2009/2010 testing program, preliminary flotation tests were performed in an effort to produce molybdenum concentrate from copper-gold-molybdenum bulk concentrates.

The flotation separation tests were performed on the bulk concentrate produced from pilot plant tests and from bench scale open circuit tests.

The 2009 testing showed that molybdenum concentrates produced from the bulk flotation concentrate from the 2009 pilot plant tests were less than 30% Mo. G&T indicated that aging of the bulk concentrates prior to the molybdenum flotation testing was one of the potential reasons for producing the low grade molybdenum concentrates. A follow-up 20-kg bench scale test on the freshly ground Composite PP2 sample produced a 48% Mo concentrate containing 1.8% Cu.

In 2010, further copper/molybdenum separation tests were conducted on the concentrates produced from the 2010 pilot plant tests. The open circuit test achieving the best overall separation metallurgical performance produced a 51% Mo concentrate with a molybdenum recovery of 72% from the molybdenum-copper concentrate generated from the 2010 pilot plant flotation tests. The test results using sodium sulphide (Na_2S) and PE 26 to suppress copper minerals are shown in

Figure 13.26, while the test results with Na_2S and PE 26 together with NaCN or D910 as depressant are shown in Figure 13.27. It appears that molybdenum concentrate grade improved with adding NaCN. Further tests are recommended to optimize the separation parameters.

Figure 13.26 Cu-Mo Separation Open Circuit Flotation Tests, 2010

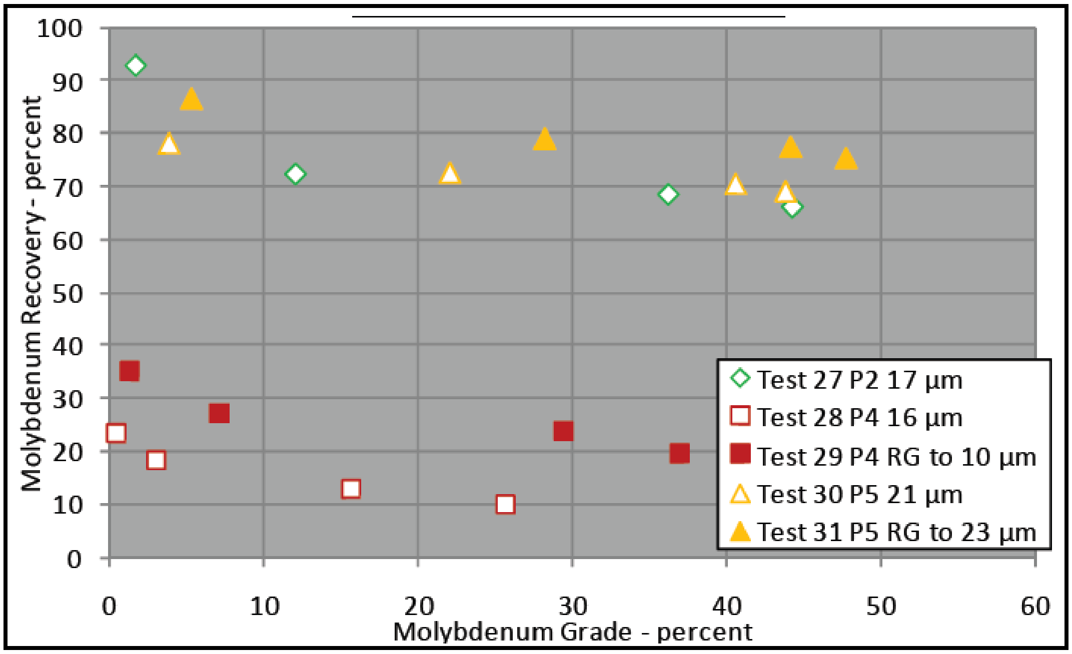
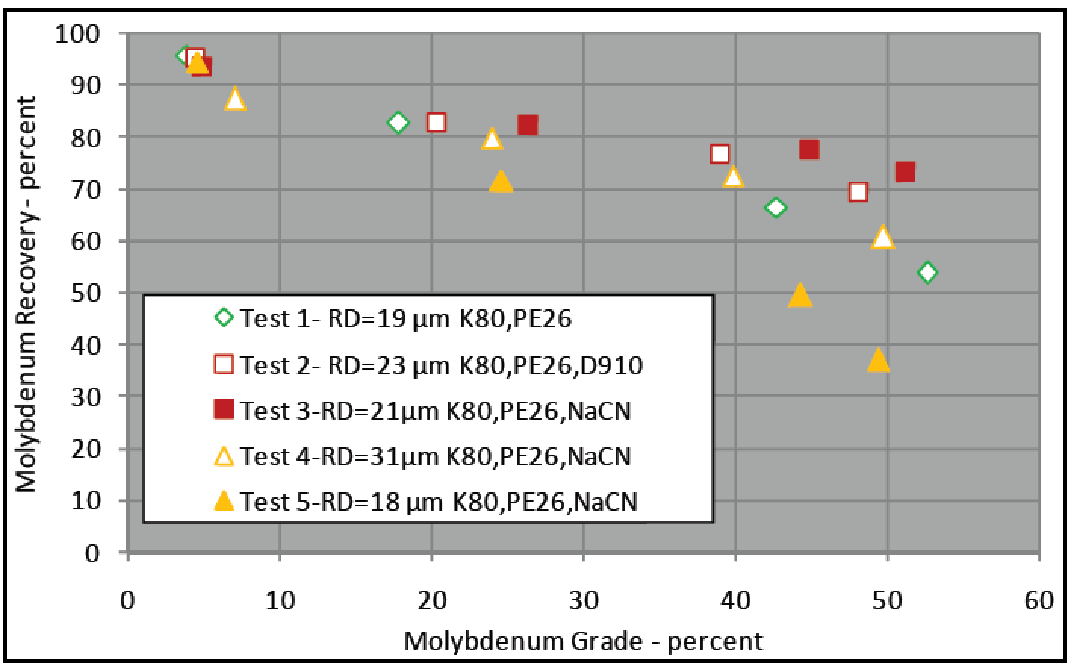


Figure 13.27 Cu-Mo Separation Open Circuit Flotation Tests, 2011



The molybdenum-copper separation LCT recovered 88.5% of the molybdenum from the molybdenum-copper concentrate and produced a 41% Mo concentrate. The test results are provided in Table 13.25.

Table 13.27 Cu-Mo Separation LCT Test Results, 2010

Product	Weight (%)	Grade (%)			Recovery (%)	
		Cu	Mo	C	Cu	Mo
Bulk Concentrate	100.0	19.3	1.28	0.63	100.0	100.0
Mo Concentrate	2.8	2.66	41.2	5.76	0.4	88.5
Cu Concentrate	97.2	19.8	0.15	0.48	99.6	11.5

G&T also conducted preliminary leaching tests on the molybdenum concentrates using both the Brenda-Leach procedure and hydrochloric acid leaching. The test results indicated that the copper and lead contents of the molybdenum concentrate were reduced respectively from 2.06% Cu to 0.26% Cu and from 0.14% Pb to 0.03% Pb. The hydrochloric acid leaching alone on a molybdenum concentrate only reduced copper content from 1.5 to 0.81%.

The assay on the final molybdenum concentrates indicated that the concentrates contained approximately 2,200 g/t rhenium (Re).

Cyanide Leach Tests

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 additional gold and silver. Most of the testing programs conducted cyanide leach tests on the first cleaner tailing and gold-bearing pyrite concentrate respectively or on the blend of the two flotation products.

Cyanidation Tests – Products from Flotation Open Circuit Tests

In the 2008 testing program, a total of 30 cyanide leach tests were carried out on the gold-bearing products generated from the flotation variability tests. Prior to the leaching, 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 13.28. The average gold extraction was approximately 79%. Increasing leach retention time from 24 to 48 hours did not appear to improve gold extraction.

Table 13.28 Cyanidation Test Results – Individual Samples, Mitchell, 2008

48-hour Leach Retention Time				24-hour Leach Retention Time			
Sample ID	**Regrind Size (P80 µm)**	**Feed (g/t Au)**	**Extraction (% Au)**	**Sample ID**	**Regrind Size (P80 µm)**	**Feed (g/t Au)**	**Extraction (% Au)**
MET 2	11	1.7	60	MET 3	12	1.4	65
MET 5	9	1.6	79	MET 4	13	1.6	78
MET 8	9	2.2	74	MET 6	9	2.4	84
MET 11	10	6.3	94	MET 7	11	3.4	78
MET 14	15	2.7	81	MET 9	9	1.3	74
MET 17	13	1.9	87	MET 10	11	2.7	91
MET 20	11	1.1	58	MET 12	10	3.3	87
MET 23	15	1.3	82	MET 13	10	8.9	90
MET 26	13	2.7	85	MET 15	14	2	85
MET 29	10	4.1	83	MET 16	13	3.2	82
MET 33	16	1.9	88	MET 18	11	1.4	63
				MET 19	12	2.0	82
				MET 21	9	2.2	69
				MET 22	12	2.7	63
				MET 24	10	4.1	87
				MET 25	9	1.7	78
				MET 27	13	2.2	81
				MET 30	11	1.6	76
				MET 32	7	3.4	91
Average	**12**	**2.5**	**79**	**Average**	**11**	**2.7**	**79**

Similar tests were conducted on the products generated from the open circuit flotation tests of various composite samples. The leach feeds were subjected to regrinding to 80% passing approximately 20 µm or finer. The leach retention time was 24 hours. As shown in Table 13.29, the gold extractions from the leach feeds ranged from 65 to 89% for the samples from the 2008 testing program and from 69 to 89% for the 2009 testing program. The average gold extraction was approximately 78% from the 2008 test work and 81% from the 2009 test work.

The 2009 test results also indicated that cyanide leaching kinetics was rapid. Approximately 69% of the gold was extracted within a 6-hour leach retention time.

Table 13.29 Cyanidation Test Results – Composites, Mitchell, 2008/2009

2008 Testing Program			2009 Testing Program			
Sample ID	Feed (g/t Au)	Extraction (% Au)	Sample ID	Feed (g/t Au)	Extraction (% Au) 6 h*	Extraction (% Au) 24 h*
QSP 0-10	2.2	82	Comp 40 CL-PR	2.0	80	85
IARG 0-10	1.3	80	Comp 41 BBRX	0.4	54	86
Hi Qtz 0-10	2.3	74	Comp 42 QSP	2.1	69	78
QSP LG 0-10	1.7	74	Comp 43 CL-PR	1.5	81	89
QSP 10-30	2.3	89	Comp 44 Hi Qtz	2.1	65	77
Prop 10-30	1.6	82	Comp 45 IARG	1.7	80	81
Hi Qtz 10-30	2.0	66	Comp 46 CL-PR	1.8	73	81
QSP 0-30	2.2	78	Comp 47 QSP	1.9	48	69
Hi Qtz 0-30	1.6	65	Comp 48 QSP	2.0	71	80
Average	**1.9**	**78**	**Average**	**1.7**	**69**	**81**

* leach retention time.

CYANIDATION TESTS – PRODUCTS FROM FLOTATION LOCKED CYCLE TESTS

The first cleaner tailing and the gold-pyrite concentrate from the various LCTs were cyanide leached to investigate the responses of the gold-bearing products to the leaching process. The test results are summarized in Table 13.30. On average, the leach feed samples contained approximately 1.6 g/t Au and 9.6 g/t Ag. The leaching tests showed that 66% of the gold and 56% of the silver were extracted from the gold-bearing products. Average cyanide consumption was 2.8 kg/t.

Table 13.30 Cyanidation Test Results on LCT Test Products – Mitchell

Testing Program	Sample	Regrind Size (P80 µm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
G&T-2153	Master	15	1.8	67.6	9.1	62.1
G&T-2153	Master	15	2.2	73.2	10.1	64.4
G&T-2344	PP Comp 1	12	1.6	68.0		
G&T-2535	PP Comp 1	15	1.7	69.0	12.6	54.4
G&T-2535	PP Comp 1	15	1.6	81.1	10.9	54.7
G&T-2670	PP Comp 3	21	1.6	61.6		
G&T-2670	PP Comp 3	18	2.0	66.5	8.1	55.5
G&T-2670	PP Hi Mo	19	1.9	68.0	8.6	50.6
G&T-2670	BS Hi Mo	19	1.7	68.9	7.6	48.7
G&T-2897	Comp 46		1.1	63.5		
G&T-3081	Mitchell Yr 0-10	24	1.5	51.2		
G&T-3081	Mitchell Yr 0-20	21	1.2	50.4		
SGS	PP Comp 1	16	1.1	69.8		
Average – Mitchell		**18**	**1.6**	**66.1**	**9.6**	**55.8**

Some of the leaching tests were conducted separately on the first cleaner tailing and the gold-bearing pyrite concentrate produced from the most recent testing programs. The test results indicated that the first cleaner tailing produced lower gold extractions, compared to the gold-bearing pyrite concentrate. On average, the gold extraction from the gold-bearing pyrite concentrate was 77%, which is similar to the results obtained from the products of the open circuit tests. However, it appears that the first cleaner tailing generated lower gold extractions, averaging 58%.

G&T also tested the gold extraction on the first cleaner tailing and the gold-bearing pyrite concentrate produced from the samples blended from the Mitchell Zone and the other zones. The test results are provided in Table 13.31.

Table 13.31 Cyanidation Test Results on LCT Products – Mitchell/Other Zones

Testing Program	Blend Sample	Regrind Size (P80 µm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
2670	Mitchell/Sulphurets[1]	18	1.7	61.0	5.4	51.4
2748	Mitchell/Iron Cap[2]	14	1.4	53.0		
2535	Mitchell/Kerr[3]	16	1.4	68.9	8.5	48.9
Average		**16**	**1.5**	**60.9**	**7.0**	**50.2**

Notes:
[1] 60% PP Comp 3 (Mitchell) + 40% Comp 49/50/51 (Sulphurets)
[2] 1/3 PP Comp 1 (Mitchell) + 1/3 Iron Cap Comp 1+ 1/3 Iron Cap Comp 3
[3] 80% PP Comp 1 (Mitchell) + 10% Comp 52 (Kerr) + 10% Comp 53 (Kerr).

The gold extraction of the blend sample from the Mitchell Zone and the Iron Cap Zone is much lower than the other samples.

CYANIDATION TESTS – PRODUCTS FROM PILOT PLANT TESTS

The first cleaner tailing and gold-bearing pyrite concentrate from the 2009 pilot plant runs (P3 and P5) were carbon-in-leach (CIL) tested for 24 hours. The gold extractions were 72.5% for the Test P3 product and 77.8% for the Test P5 product.

The CIL bottle roll cyanidation tests were also carried out on selected cleaner scavenger tailing and pyrite concentrate streams from the 2010 pilot plant testing. The tests were conducted at variable regrind sizes and sodium cyanide concentrations. The results obtained at 1,000 mg/L NaCN dosage are summarized as follows:

- At an average regrind size of 80% passing 24 µm, the average gold extraction from the 1.6 g/t Au cleaner scavenger tailing was approximately 70%.
- At an average regrind size of 80% passing 20 µm, the average gold extraction from the gold-bearing pyrite concentrate containing 1.9 g/t Au was approximately 77%.

SGS also conducted cyanide leach tests on the gold-bearing products produced by the 2009 G&T pilot plant tests. Two tests (bench bottle-on-roll test and bulk leach test) were conducted on the pilot plant test samples. The bulk leach test by agitation was to prepare leach solutions for cyanide destruction testing and cyanide recovery testing. As shown in Table 13.32, the tests produced lower gold extractions compared to the data obtained by G&T.

Table 13.32 Cyanidation Test Results – LCT Products, 2009 (SGS)

Test Method	Sample Weight	Leach Feed (Au g/t)	Gold Extraction (%)	Cyanide Consumption (kg/t)
Pilot Plant Test Products – First Cleaner Tailing & Pyrite Concentrate				
DCN* (Bottle-on-Roll)	562 g	1.53	59.0	3.26
DCN* (Drum with Agitation)	20 kg	1.90	49.9	2.96

* DCN = direct cyanide leaching.

Gravity Concentration Tests

Gravity Concentration Tests on Head Samples

In the 2008 testing program, 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 13.33.

Table 13.33 Gravity Separation Test Results – Mitchell

	Pan Concentrate		Knelson Concentrate	
Sample ID	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.

GRAVITY CONCENTRATION TESTS ON TAILING SAMPLES

G&T also carried out centrifugal gravity concentration tests to recover gold-bearing minerals from flotation tailing. The test results show that the concentration was able to recover some of the gold from the tailing. Due to a poor match between the calculated gold and measured gold in the feeds, further tests should be conducted to confirm the findings.

SULPHURETS MINERALIZATION

Test Samples

Three composite samples were compiled from crushed drill cores to investigate the metallurgical responses of Sulphurets mineralization. The drill hole locations are shown in Figure 13.28. The chemical assay of these composites is provided in Table 13.34.

Figure 13.28 2008/2009 Sulphurets Zone Metallurgical Samples – Plan View

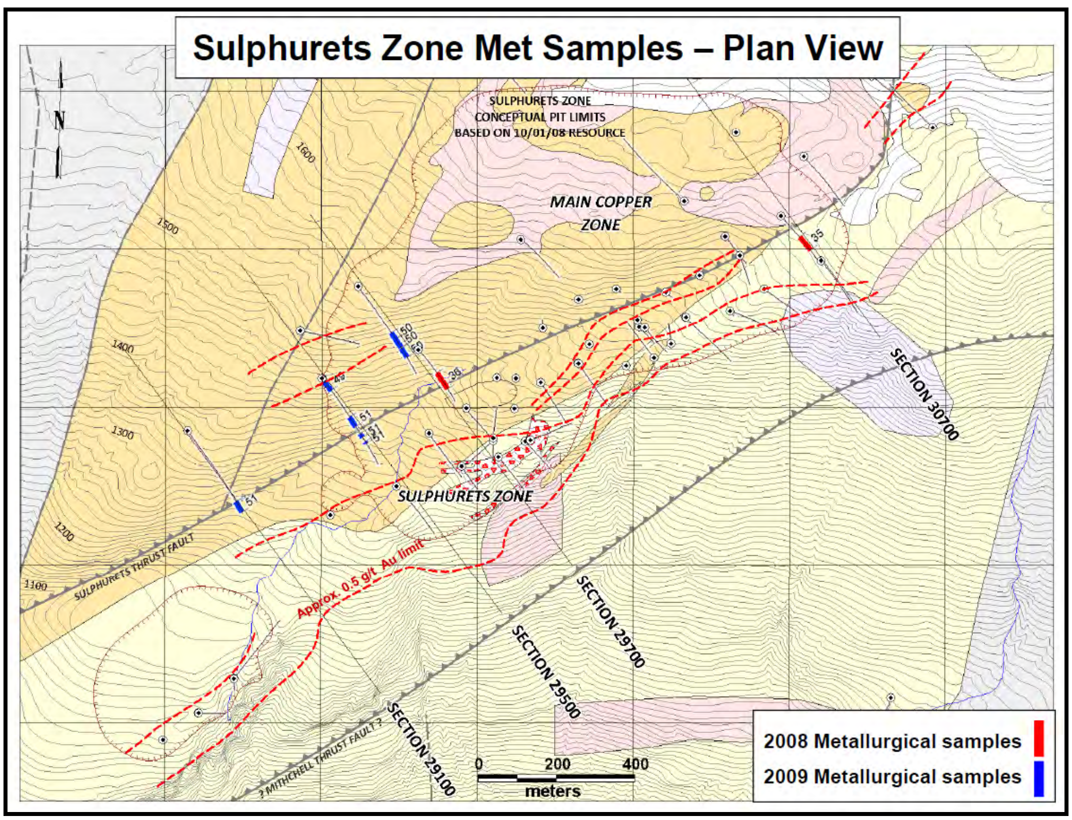


Table 13.34 Metal Contents of Composites – Sulphurets

Composite	Mineralization Type*	Metal Content Cu(T) (%)	Cu(ox) (%)	Cu(CN) (%)	Au(T) (g/t)	Au(CN) (g/t)	Mo (%)	Ag (g/t)
2009 Test Work (G&T)								
Comp 49	Hazelton Volcanics	0.14	0.016	0.016	0.26	0.002	0.003	1.9
Comp 50	Raewyn Copper	0.26	0.007	0.012	0.66	0.006	0.005	1.2
Comp 51	Raewyn Copper	0.37	0.005	0.013	0.81	0.007	0.011	1.4
2011/2012 Test Work (G&T)								
Comp 8	Raewyn Copper	0.46	-	-	0.76	-	0.008	1
Comp 9	Lower Hazelton	0.17	-	-	0.65	-	0.004	2

* **Hazelton Volcanics:** propylitic altered (quartz, chlorite, pyrite) volcanics and sediments of the Main Copper Zone (above Sulphurets Fault).
Raewyn Copper: propylitic altered volcanics and sediments of the Sulphurets Zone (beneath Sulphurets Fault); selected intervals are within crackled, veined, and brecciated transitional zone beneath the Gold Breccia Zone, and have higher than average gold grades.

In 2011/2012 G&T conducted metallurgical tests (G&T, KM3174) on the two samples, representing Raewyn CV mineralization and Lower Hazelton mineralization. The key element assay data are shown in Table 13.34.

Mineralization Hardness

The test results, as provided in Table 13.35, indicate that the Sulphurets samples are more resistant to ball mill grinding compared to the Mitchell samples. The average Bond ball work index is 18.5 kWh/t for the Sulphurets samples; the Bond Ai of the overall Sulphurets composite is 0.233 g.

Table 13.35 Bond Ball Mill Work Index Test Results – Sulphurets

Samples	Wi (kWh/t)	Ai (g)
2011/2012 G&T		
Composite 8	18.7	
Composite 9	16.7	
Sub-average	**17.7**	
2009 G&T		
Composite 49	15.8	
Composite 50	20.8	
Composite 51	19.8	
Sub-average	**18.8**	
2009/2010 SGS		
Composite	19.1	0.233
Total Average	**18.5**	

The 2011/2012 testing program determined the SAG mill grindability parameters for the samples from the Sulphurets deposit. Compared to the samples from the Mitchell deposit, the Sulphurets samples are more resistant to SAG mill grinding. The results are shown in Table 13.36.

Table 13.36 SMC Test Results – Sulphurets, 2011/2012

Parameter	Sample	
	Composite 8	Composite 9
SG	2.73	2.79
A	63.2	57.7
b	0.66	0.67
Axb	41.7	38.7
DWi (kWh/m^3)	62	69
Mia (kWh/t)	19.0	19.9
Ta	0.39	0.36

In 2009, SGS conducted bench scale HPGR tests on the Sulphurets composite samples. The tests included batch open circuit tests and LCTs. The test results indicate that the Sulphurets mineralization is more resistant to HPGR crushing than the Mitchell mineralization. On average, the net specific energy requirement is 3.08 kWh/t for the Sulphurets sample compared to 2.33 kWh/t for the Mitchell sample. The LCT results, including specific grinding force (N/mm^2) and specific throughput rate (ts/hm^3-(m$_c$)) are summarized in Table 13.20. The preliminary HPGR/ball mill circuit simulation results by SGS are provided in Table 13.21. The simulations suggested that the unit power requirement for the HPGR/ball mill circuit would be approximately 14.8 kWh/t for the Sulphurets mineralization, compared to 10.4 kWh/t for the Mitchell mineralization.

Flotation Tests

In the 2009 testing program, G&T performed preliminary flotation tests to investigate the responses of the Sulphurets ores to the flotation conditions established for the Mitchell samples. As indicated in Table 13.37, the Sulphurets ore samples may produce higher grade copper concentrates than the Mitchell samples. Composite 49 (Hazelton Volcanics (HV)) has a lower level copper metallurgical performance compared to the other composites (Raewyn Copper (RC)). This may result from the lower copper head grade in the sample. The test results also showed that Composite 51 produced much lower gold recoveries in the cleaning stage, compared to the other two samples.

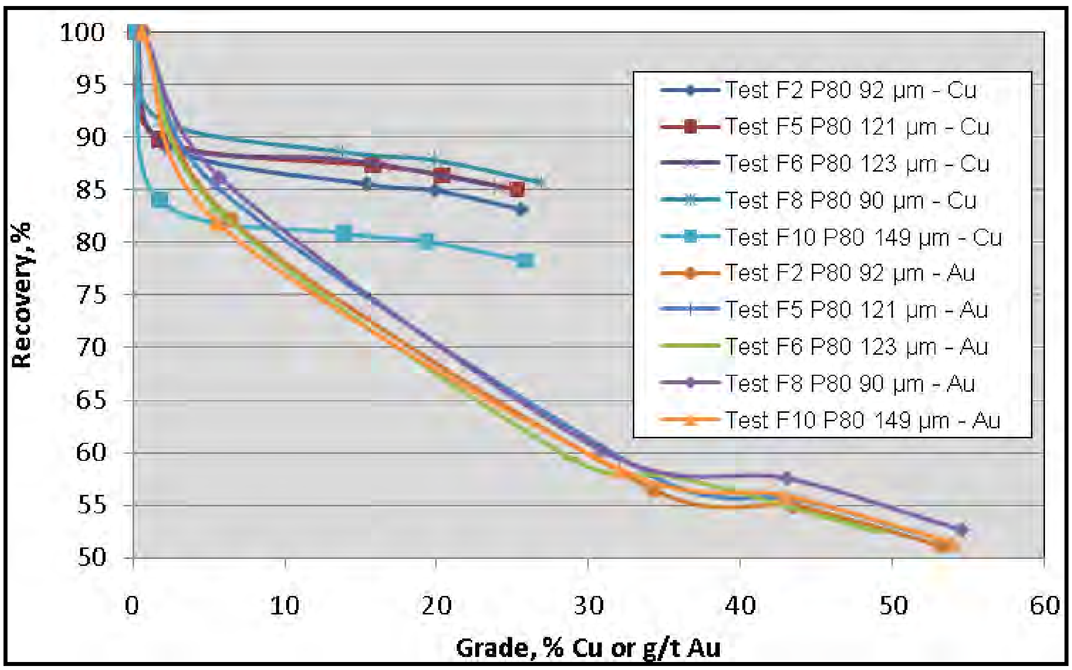
Table 13.37 Batch Flotation Tests –Sulphurets, 2009 (G&T)

Sample ID/ Rock Type	Test ID	Grade Size (µm)		Head		Conc. Grade Cu (%)	Recovery (%)			
							Rougher		Cleaner	
		Primary	Regrind	Cu (%)	Au (g/t)		Cu	Au	Cu	Au
Comp 49/HV	Test 10	132	12	0.14	0.26	27.8	81.3	79.1	75.6	54.3
	Test 34	114	11	0.14	0.26	26.3	75.0	78.6	68.2	50.7
Comp 50/RC	Test 11	102	11	0.26	0.66	29.4	89.5	77.9	86.3	67.4
	Test 35	102	12	0.26	0.66	28.9	88.7	85.6	83.3	68.9
Comp 51/RC	Test 12	127	15	0.37	0.81	28.6	91.1	76.6	87.6	44.2
	Test 36	117	15	0.37	0.81	29.8	92.5	84.5	89.2	47.1

The 2009/2010 SGS testing program involved bench open circuit tests and a LCT on a composite generated from the Sulphurets deposit. The tested flowsheet is similar to that used by G&T except for the addition of CMC, which is used to suppress clay minerals. It appeared that fine primary grind size may improve metal recovery and that the addition of CMC may also improve final concentrate grade.

The batch open circuit tests are summarized in Figure 13.29.

Figure 13.29 Batch Flotation Tests –Sulphurets, 2009 (SGS)



Both G&T and SGS conducted LCTs on the composites generated from the Sulphurets samples. Table 13.38 summarizes the flotation LCT results.

Table 13.38 Locked Cycle Test Results – Sulphurets

Test Program	Composite	Product	Primary/ Regrinding Size (P80 µm)	Mass (%)	Grade				Flotation Recovery (%)			
					Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu	Au	Ag	Mo
SGS	Composite	Head	125/20	100.0	0.20	0.66		0.007	100.0	100.0		100.0
		Cu/Mo Concentrate		0.75	22.7	49.1		0.630	85.7	56.1		66.6
		Bulk Cleaner Tailing										
		+Au-Pyrite Concentrate		17.3	0.08	1.31		0.008	6.73	34.3		20.3
G&T 2670/44	Master Composite (Comp49/50/51)	Head	154/16	100.0	0.24	0.52	1.6	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.7	28.3	41.8	82.0	0.701	80.5	53.9	34.3	72.2
		Bulk Cleaner Tailing		6.3	0.13	1.94		0.016	3.5	23.5		15.1
		Au-Pyrite Concentrate		2.9	0.38	1.41		0.013	4.7	7.9		5.7
G&T 2897/22	Master Composite (Comp49/50/51)	Head	113/-	100.0	0.24	0.50	1.5	0.008	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.70	28.4	41.6	71.4	0.850	79.4	55.6	31.5	68.5
		Bulk Cleaner Tailing		6.3	0.17	1.82	4.2	0.013	4.5	23.0	17.5	9.9
		Au-Pyrite Concentrate		4.0	0.35	1.15	3.5	0.011	6.0	9.3	9.5	5.4
G&T 3174/8	Composite 8	Head	121/19	100.0	0.46	0.70	1	0.008	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.3	29.3	31.4	34	0.227	83.6	58.6	31.1	37.7
		Bulk Cleaner Tailing		9.2	0.37	2.2	1	0.044	7.3	28.9	6.4	51.4
		Au-Pyrite Concentrate		6.7	0.26	0.67	1	0.005	3.8	6.4	4.7	4.4
G&T 3174/9	Composite 9	Head	127/21	100.0	0.16	0.59	2	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.4	26.0	63.7	130	0.170	60.6	40.1	21.3	14.1
		Bulk Cleaner Tailing		5.6	0.82	3.55	11	0.055	28.8	33.7	28.4	68.6
		Au-Pyrite Concentrate		6.7	0.12	1.26	4	0.004	4.8	14.3	11.8	5.4

The SGS tests produced higher copper recoveries at lower concentrate grades than the G&T tests: 85.7% recovery at 22.7% Cu grade, versus 76% recovery at 28% Cu grade. The test on Composite 9 produced a much lower copper recovery compared to the other tests. This may be the result of a low head grade of the Hazelton sample. Further tests should be conducted to investigate the copper metallurgical performance of the mineralization. The average gold recovery for both groups of tests was approximately 56%, excluding a much lower gold recovery from the Hazelton sample. Silver recovery obtained in the tests at G&T averaged 30%. Molybdenum reporting to the bulk concentrate averaged at 69% for the 2009 test samples, but only 26% for the 2011/2012 samples.

As shown in Table 13.23 for the locked cycle flotation test results, the metallurgical performances of the Mitchell-Sulphurets blend sample (60% Mitchell and 40% Sulphurets) were very similar to those achieved with the Mitchell mineralization alone.

Further test work, including variability testing, is recommended to better define the metallurgical performance of the Sulphurets mineralization.

Cyanide Leach Tests

The gold-bearing products, first cleaner tails, and pyrite concentrate from the flotation tests, were subjected to cyanide leaching to recover gold. On average, the gold in the mineralization was more difficult to recover in comparison with the Mitchell mineralization. The Composite 51 sample showed a less favourable metallurgical response to the cyanidation. The results are shown in Table 13.37. Further tests are recommended investigating the sample's mineralogy and the methods to improve the gold leaching.

Table 13.39 Average Cyanidation Test Results – Sulphurets, 2009 (G&T)

Composite ID	Mineralization Type	Leach Head (Au g/t)	Gold Extraction (%)	
			6 h*	24 h*
Comp 49	Hazelton Volcanics	0.80	45.5	55.5
Comp 50	Raewyn Copper	0.97	65.5	70.9
Comp 51	Raewyn Copper	2.20	20.3	21.4
Average		**1.32**	**43.8**	**49.2**

* leach retention time.

Both G&T and SGS conducted cyanidation tests on the products produced from the locked cycle flotation tests.

The test results are provided in Table 13.40. In general, the Sulphurets samples produced lower gold and silver extractions, in comparison with the Mitchell samples. The best gold extraction obtained was 70.5% by SGS using the CIL leach procedure. The direct cyanide leach test produced inferior results.

Table 13.40 Cyanidation Test Results – Flotation LCT Products, Sulphurets, 2009–2011

Test Program	Sample	Regrind Size (P80 µm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
G&T-2670	Master Composite	16	1.7	40.9	3.7	52.4
G&T 2897	Master Composite	-	1.5	34.5	3.3	47.9
Composite 8 - 2011/2012	Raewyn CV	25	1.6	41.7		
Composite 9 - 2011/2012	Lower Hazelton	19	2.5	68.3		
SGS (DCN)*	Composite	-	1.6	51.5	-	-
SGS (CIL)	Composite	-	1.3	70.5	-	-

* direct cyanide leaching.

KERR MINERALIZATION

Test Samples

Four composite samples from the Kerr Zone, identified as Composites 52 and 53 in 2010 and Composite 10 and Composite 11 in 2011/2012, were prepared for the metallurgical testing. The samples were prepared from the drill core intervals. The metal assays of the composites are provided in Table 13.41.

Table 13.41 Metal Contents of Composites – Kerr, 2010 (G&T)

Composite	Mineralization Type*	Metal Content Cu (%)	Au (g/t)	Mo (%)	Ag (g/t)
Comp 52 - 2010	Rubble Zone	0.59	0.22	0.004	2.0
Comp 53 - 2010	Quartz Stockwork	0.61	0.17	0.001	1.5
Composite 10 – 2011/2012	CL Quartz Crackle	0.59	0.26	0.001	1
Composite 11 – 2011/2012	QSP Quartz Crackle	0.68	0.29	0.001	2

* **Rubble Zone:** quartz, sericite, chlorite, pyrite altered rocks with anhydrite ± gypsum veinlets, secondary chalcocite coatings, poor rock quality.
Quartz Stockwork: quartz, sericite, chlorite, pyrite altered rocks with crackled quartz stockwork veinlets, mylonitized, relatively competent.
QSP Quartz Crackle: hosted by strongly deformed to schistose Stuhini Group volcanics, sediments, and minor intrusives; silica and sericitic alteration; higher pyrite content; also with crackled quartz stockwork veining; comprises about half of the resource and generally forms the periphery surrounding the CL Quartz Crackle mineralization,
CL Quartz Crackle: hosted by strongly deformed to schistose Stuhini Group volcanics, sediments, and minor intrusives; finely crackled or fractured quartz stockwork veining with sulfides; comprise just under half of the resource and forms the core of the deposit,

Mineralization Hardness

The samples from the Kerr deposit are more amenable to ball mill grinding when compared to the Mitchell and Sulphurets mineralization. As shown in Table 13.42, the average Bond ball mill work index is 13.9 kWh/t. These results agree with the historical test results.

Table 13.42 Bond Ball Mill Work Index Test Results – Kerr (G&T)

Samples	Wi (kWh/t)
Composite 52 - 2010	13.8
Composite 53 - 2010	13.0
Composite 10 – 2011/2012	14.8
Composite 11 – 2011/2012	14.1
Average	**13.9**

The 2011/2012 testing program determined the grindability of the Kerr samples to SAG mills. The test results revealed that the grindabilty of the Kerr samples to SAG mill grinding is very similar to the samples from the Mitchell deposit. The results are shown in Table 13.43.

Table 13.43 SMC Test Results – Kerr, 2011/2012

	Sample	
Parameter	**Composite 10**	**Composite 11**
SG	2.87	2.86
A	56.9	65.3
b	0.81	0.72
Axb	46.1	47.0
DWi (kWh/m^3)	58	56
Mia (kWh/t)	17.3	17.0
Ta	0.41	0.42

Flotation Tests

The test conditions used for the Mitchell and Sulphurets samples were also used for the composite samples collected from the Kerr deposit. The open circuit batch flotation tests showed that the Kerr samples produced better concentrate grades than the Mitchell or Sulphurets samples. Copper recovery produced was slightly lower than the Mitchell or Sulphurets samples at equivalent copper concentrate tenor. Gold recovery for the Kerr samples was lower because the gold head grades were considerably lower than the samples from the other two ore deposits.

The LCT results, as presented in Table 13.44, indicate that the metallurgical performance of the Kerr samples was not as good as that achieved with the Mitchell and Sulphurets samples despite their lower copper head grades.

On average, the Kerr samples produced a 27.8% Cu concentrate. The copper and gold reporting to the concentrate were 83% and 41%, respectively. Approximately 51% of the gold reported to the gold-bearing pyrite products (first cleaner tailing and gold-bearing pyrite concentrate). The 2011/2012 test program produced better metallurgical performances from the samples tested, than what was achieved previously.

As shown in Table 13.23 for the locked cycle flotation test results, the metallurgical performances of the Mitchell-Kerr blend sample (80% Mitchell and 20% Kerr) were very similar to those achieved with the Mitchell mineralization alone.

Table 13.44 Locked Cycle Test Results – Kerr (G&T)

Test Program	Comp	Product	Primary/ Regrinding Size (P80 µm)	Mass (%)	Grade			Flotation Recovery (%)		
					Cu (%)	Au (g/t)	Ag (g/t)	Cu	Au	Ag
G&T 2535/16	Comp 52	Head	119/15	100.0	0.59	0.22	1.9	100.0	100.0	100.0
		Cu/Mo Concentrate		2.1	22.3	4.05	33.5	81.6	38.8	37.6
		Bulk Cleaner Tailing		7.9	0.43	0.97	6.3	5.7	34.2	26.0
		Au-Pyrite Concentrate		7.7	0.39	0.62	4.2	5.2	21.5	17.0
G&T 2535/17	Comp 53	Head	122/14	100.0	0.62	0.25	1.4	100.0	100.0	100.0
		Cu/Mo Concentrate		1.7	29.3	5.58	31.8	80.6	37.7	37.9
		Bulk Cleaner Tailing		6.8	0.40	0.51	3.6	4.5	13.8	17.5
		Au-Pyrite Concentrate		13.6	0.42	0.66	3.0	9.1	36.0	28.2
G&T 3174/10	Composite 10	Head	124/18	100.0	0.59	0.24	2	100.0	100.0	100.0
		Cu/Mo Concentrate		1.7	30.7	7.2	49	86.3	49.7	39.8
		Bulk Cleaner Tailing		10.3	0.39	0.72	3	6.7	30.8	17.4
		Au-Pyrite Concentrate		10.4	0.16	0.38	1	2.8	16.3	5.1
G&T 3174/11	Composite 11	Head	130/19	100.0	0.69	0.24	3	100.0	100.0	100.0
		Cu/Mo Concentrate		2.0	29.0	5.1	77	83.4	41.1	47.4
		Bulk Cleaner Tailing		9.2	0.34	0.6	5	4.6	22.7	14.4
		Au-Pyrite Concentrate		13.5	0.33	0.54	3	6.5	30.0	14.7

Leach Tests

G&T conducted the cyanidation tests on the first cleaner tailing and the gold-bearing pyrite concentrate produced from the LCTs. The leaching procedure used was the same as that used previously on the Mitchell samples. Test results are provided in Table 13.45.

Table 13.45 Cyanidation Test Results on LCT Test Products – Kerr (G&T)

Test Program	Sample	Regrind Size (P80 µm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
G&T-2535	Comp 52	17	1.1	76.0	5.5	45.8
G&T 2535	Comp 53	15	0.6	59.7	3.2	18.7
G&T 3174	Composite 10	20	0.6	47.2		
G&T 3174	Composite 11	20	0.6	45.6		
Average – Kerr		**18**	**0.7**	**57.1**	**4.4**	**32.3**

On average, the gold extraction from both the gold-bearing products was approximately 57%, slightly lower than the results obtained from the Mitchell samples. The average gold feed grade to the cyanide leach circuit was lower in comparison with the cyanide leach feeds of the Mitchell samples. The test results also indicated that the first cleaner tailing produced slightly lower gold and silver recoveries compared to the gold-bearing pyrite concentrate. The average silver extraction was 32%, which was lower than the average extraction of 56% obtained from the Mitchell samples.

IRON CAP MINERALIZATION

Test Samples

The 2010 test work conducted metallurgical tests on two composite samples generated from a total of 168 samples weighing a total of approximately 689 kg. The assay of the head samples are provided in Table 13.46.

Table 13.46 Metal Contents of Composites – Iron Cap, 2010 (G&T)

Composite	Metal Content						
	Cu (T) (%)	Cu (ox) (%)	Cu (CN) (%)	Au (g/t)	Mo (%)	Ag (g/t)	S (%)
IC 2010 Composite 1	0.14	0.001	0.015	1.06	0.002	6	4.5
IC 2010 Composite 2	0.36	0.004	0.023	0.32	0.003	5	3.6
Iron Cap Blend	0.25			0.75	0.003		3.7

Mineralogy

The mineral content, in each of the two master composites, was determined using the Bulk Mineral Analysis with Liberation (BMAL) function within QEMSCAN. The results of the BMAL analysis indicated that:

- Both composites analyzed contained about 6 to 8% sulphide minerals. The dominant sulphide mineral present was pyrite. The balance of each sample, about 93%, was comprised of non-sulphide gangue minerals consisting of quartz, feldspar, and muscovite.
- Copper is mostly contained in chalcopyrite. Composite 1 also contained chalcocite/covellite and tennantite/tetrahedrite at approximately 4 and 5% of the feed copper respectively.

Mineralization Hardness

The grindability determination tests on the two composite samples from the Iron Cap deposit showed that the mineralization is of moderate hardness to ball mill grinding. The Bond ball mill work indices of both the samples are 14.9 kWh/t.

Flotation Tests

The test conditions used for the Mitchell samples were tested for the two composite samples from the Iron Cap deposit. The open circuit batch flotation tests showed that the Iron Cap mineralization was not sensitive to the primary grind sizes ranging from 80% passing 120 µm to 170 µm.

The flotation LCT results are provided in Table 13.47. On average, the mineralization produced a 25.7% Cu concentrate. The copper and the gold reporting to the concentrate were 85% and 51%, respectively. On average, approximately 39% of the gold reported to the gold-bearing pyrite products (first cleaner tailing and gold-bearing pyrite concentrate).

The results indicate that the copper recoveries from both the Iron Cap samples were comparable to the Mitchell mineralization. It appeared that the gold recoveries to the concentrate were lower than these achieved with the Mitchell mineralization; however, the silver recoveries were higher. Approximately 38% and 55% of the molybdenum from the two samples reported to the final bulk concentrate.

Table 13.47 Locked Cycle Test Results – Iron Cap

Test Program	Comp	Product	Primary/ Regrinding Size (P80 µm)	Mass (%)	Grade				Flotation Recovery (%)			
					Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu	Au	Ag	Mo
G&T 2748/11	Iron Cap 2010 Composite1	Head	150/15	100.0	0.14	1.28	6	0.002	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.5	25.4	147	774	0.180	81.6	55.2	61.0	37.9
		Bulk Cleaner Tailing		10.4	0.06	2.17	11.6	0.004	3.8	17.6	20.1	18.0
		Au-Pyrite Concentrate		7.9	0.11	1.88	6.6	0.002	5.9	11.7	8.7	8.6
G&T 2748/12	Iron Cap 2010 Composite2	Head	147/22	100.0	0.38	0.31	5	0.003	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.3	24.9	10	255	0.115	88.1	45.0	62.0	55.2
		Bulk Cleaner Tailing		10.5	0.06	1.21	7.9	0.003	1.7	40.7	16.6	11.2
		Au-Pyrite Concentrate		6	0.25	0.57	5.2	0.002	4	11.1	6.2	4.3
G&T 2748/17	50%Comp 1: 50%Comp 2	Head	108/19	100.0	0.26	0.82		0.003	100.0	100.0		100.0
		Cu/Mo Concentrate		0.8	26.7	51.9		0.144	85.2	53.3		41.5
		Bulk Cleaner Tailing		10.9	0.06	1.82		0.005	2.4	24.2		17.7
		Au-Pyrite Concentrate		7.3	0.16	1.37		0.003	4.4	12.1		6.2

Cyanide Leach Tests

G&T conducted cyanidation tests on the first cleaner tailing and the gold-bearing pyrite concentrate produced from the LCTs. The leaching procedure used was developed from the Mitchell samples. Test results are provided in Table 13.48.

Table 13.48 Cyanidation Test Results on LCT Test Products – Iron Cap

Testing Program	Sample	Regrind Size (P80 µm)	Feed (Au g/t)	Extraction (Au %)	Feed (Ag g/t)	Extraction (Ag %)
G&T-2748	Iron Cap Comp1	14	1.9	49.7	9.4	62.8
G&T-2748	Iron Cap Comp2	15	1.1	40.4	6.9	56.8
G&T-2748	50% Comp1/50% Comp2	16	1.5	48.6		
Average – Iron Cap		**15**	**1.5**	**46.2**	**8.2**	**59.8**

On average, the gold extraction from both the gold-bearing products was approximately 46%. The test results also indicated that both the first cleaner tailing and the gold-bearing pyrite concentrate produced lower gold recoveries compared to the other mineralization, especially the first cleaner tailing. The average gold feed grade to the cyanide leach circuit was lower in comparison with the cyanide leach feeds of the Mitchell samples. The average silver extraction was high, averaging 60%, which is slightly higher than the average extraction of 56% obtained the Mitchell samples.

The mineralogical study by Surface Science Western on the leaching residues found that the residual gold is present in colloidal type sub-microscopic gold, mainly in pyrite, which occurs in coarse and porous types. Surface Science Western pointed out that the pre-treatment by pressure or biooxidation would be required to release this locked gold.

Flotation Concentrate Assay

The multi-element assay data are provided in Table 13.49 for the concentrates from the Mitchell deposit and Table 13.50 for the concentrates from the other deposits. On average, the impurities in the copper-gold concentrates produced from the Mitchell, Sulphurets, and Kerr deposits should not attract smelting penalties as set out by most smelters.

Table 13.49 Multi-Element Assay – Mitchell Concentrate[1]

Element	Unit	Mitchell									Average
		2153/142 Master Comp	2344/73 Comp PP1	2535/18 Comp PP1	2535/20 Comp PP1	SGS/LCT1 Comp PP1	2670/18(2) Comp PP3	2670/ Pilot Plant Comp PP3	3081/82 Comp Y0-20	3081/93 Comp Y0-10	
Cu	%	22.0	22.3	28	23.8	23.1	27.4	25.7	23.8	27.1	24.8
Au	g/t	64.7	55.7	77.8	62.0	53.7	70.5	65.5	44.2	58.5	61.4
Ag	g/t	257	-	260	248	-	275	304	223	427	285
Mo	%	-	0.23	0.12	0.12	0.41	0.62	0.33	0.24	0.33	0.30
S (T)	%	33.4	34.4	34.7	32.9	38.1	34.5	31.1	35.3	34.2	34.3
S (-2)	%	-	-	32.9	32.1	-	33.3	28.7			31.8
Fe	%	26.8	30.8	29.6	30.7	32.7	30.1	27.6	32.6	30.9	30.2
Sb	ppm	696	698	539	597	-	466	338	210	1,100	581
As	ppm	1,184	934	824	878	-	1174	821	690	2,044	1069
Co	ppm	48	76	52	52	-	68	56	84	62	62.3
Cd	ppm	72	44	60	84	-	88	80	54	112	74
Bi	ppm	36	43	150	127	-	<10	<10	<20	<20	89
Hg	ppm	0.6	<1	<1	<1	-	1	<1	<1	<1	<1
Ni	ppm	120	240	112	156	-	48	80	70	66	112
F	ppm	346	150	100	148	-	89	230	69	129	158
Cl	ppm	-	-	-	-	-	<0.01	<0.01	-	-	
Se	ppm	72	102	82	70	-	73	70	59	75	75
P	ppm	230	215	146	189	-	55	492	52	81	183
Pb	%	0.92	0.19	0.19	0.22	-	0.32	0.23	0.12	0.17	0.30
Zn	%	0.42	0.23	0.25	0.38	-	0.43	0.32	0.26	0.27	0.32
SiO_2	%	9.84	6.67	2.39	7.11	-	3.04	8.23	4.26	2.93	5.56

table continues...

Element	Unit	Mitchell									Average
		2153/142 Master Comp	2344/73 Comp PP1	2535/18 Comp PP1	2535/20 Comp PP1	SGS/LCT1 Comp PP1	2670/18(2) Comp PP3	2670/ Pilot Plant Comp PP3	3081/82 Comp Y0-20	3081/93 Comp Y0-10	
CaO	%	0.54	0.53	0.39	0.54	-	0.27	0.74	0.42	0.52	0.5
Al_2O_3	%	3.31	1.76	0.62	1.37	-	0.57	1.83	0.98	0.85	1.41
MgO	%	0.48	0.36	0.18	0.34	-	0.15	0.47	0.16	0.13	0.28
MnO	%	0.02	0.03	0.011	0.026	-	0.015	0.035	0.012	0.009	0.020
Insol	%	-	8.46	4.02	8.87	-	3.23	10.3	-	-	6.98

Notes:

(1) copper-gold/molybdenum concentrate before molybdenum separation.

(2) testing program and test ID.

Table 13.50 Multi-element Assay – Sulphurets/Kerr/Iron Cap/Blend Concentrate[1]

Element	Unit	Sulphurets			Sulphurets/ Mitchell	Iron Cap		Kerr				Mitchell/ Kerr
		2670/44 Comp	3174/8 Comp 8	3174/9 Comp 9	2670/62 Blend	2748/11 Comp 1	2748/12 Comp 2	2535/16 Comp 52	2535/17 Comp 53	3174/10 Comp 10	3174/11 Comp 11	2535/19(2) Blend
Cu	%	28.3	29.3	26.0	24.2	25.4	24.9	22.3	29.3	30.7	29.0	25.3
Au	g/t	41.8	31.4	63.7	52.0	146.8	10.9	4.05	5.58	7.2	5.1	40
Ag	g/t	82	34	130	178	774	1.3	33.5	31.8	49	77	168
Mo	%	0.70	0.227	0.170	0.66	0.18	0.12	0.013	0.017	0.023	0.038	0.056
S (T)	%	33.6	32.4	34.4	34.9	32.6	33.5	27.1	35.3	34.0	36.1	35.0
S (-2)	%	31.2	-	-	32.2	32.4	32.2	25.9	33.8	-	-	33.4
Fe	%	29.6	27.2	29.3	30.0	26.5	27.8	23.7	27.5	29.4	29.1	29.3
Sb	ppm	445	2,100	370	500	4379	2876	24	121	620	180	492
As	ppm	224	1,768	205	969	3,067	1,107	143	3,276	621	2793	1,369

table continues..

Element	Unit	Sulphurets			Sulphurets/ Mitchell	Iron Cap		Kerr				Mitchell/ Kerr
		2670/44 Comp	3174/8 Comp 8	3174/9 Comp 9	2670/62 Blend	2748/11 Comp 1	2748/12 Comp 2	2535/16 Comp 52	2535/17 Comp 53	3174/10 Comp 10	3174/11 Comp 11	2535/19(2) Blend
Co	ppm	92	-	-	104	50	68	40	52	-	-	68
Cd	ppm	180	68	26	144	320	128	20	8	6	32	80
Bi	ppm	<10	-	-	<10	205	164	95	105	-	-	121
Hg	ppm	2	-	-	1	<1	2	3.4	12	-	-	2.4
Ni	ppm	88	-	-	96	50	88	132	168	-	-	164
F	ppm	155	-	-	174	162	494	320	88	-	-	116
Cl	ppm	<0.01	-	-	<0.01	<0.01	<0.01	-	-	-	-	-
Se	ppm	118	-	-	89	180	108	140	109	-	-	76
P	ppm	92	-	-	113	143	135	1045	233	-	-	224
Pb	%	0.26	0.19	0.72	0.26	1.31	0.43	0.03	0.05	0.04	0.05	0.15
Zn	%	0.54	0.29	0.18	0.92	2.29	1.02	0.30	0.10	0.08	0.75	0.42
SiO_2	%	4.14	-	-	5.82	3.16	5.59	14.0	3.9	-	-	5.12
CaO	%	0.41	-	-	0.38	0.34	0.29	0.83	0.17	-	-	0.43
Al_2O_3	%	0.92	-	-	1.18	0.85	1.28	3.92	0.85	-	-	0.99
MgO	%	0.25	-	-	0.29	0.12	0.18	0.70	0.14	-	-	0.26
MnO	%	0.017	-	-	0.022	0.011	0.017	0.050	0.015	-	-	0.018
Insol	%	4.90	-	-	7.21	5.15	7.66	19.6	5.42	-	-	6.67

Notes:

(1) copper-gold/molybdenum concentrate before molybdenum separation.

(2) testing program and test ID.

However, arsenic (As) and antimony (Sb) contents of the concentrates from the Iron Cap deposit and arsenic content of the concentrate from the Comp 53 of the Kerr sample may attract smelting penalties. Also the lead (Pb) content of the concentrate from the Iron Cap Comp 1 may be higher than the penalty thresholds. Fluorine (F) levels in some of the concentrates may be also higher than the penalty thresholds. It is anticipated that the Iron Cap and Kerr ores will be processed together with ore from the Mitchell deposit. Impurities in the copper concentrates produced from these blended ores should be further reviewed with respect to smelting penalties.

ANCILLARY TESTS

The testing programs also conducted various environment-related tests and determined engineering-related parameters. The key tests are as follows:

- leach residue cyanide destruction, including sulphur dioxide (SO_2)/air, Caro's acid (H_2SO_5), and hydrogen peroxide (H_2O_2)
- cyanide recovery from barren solutions, including the AVR (Acidification, Volatilization of HCN gas and Re-neutralization) process and the SART (Sulphidization, Acidification, Recycling of precipitate and Thickening of precipitate) process.
- static and dynamic thickening tests for conventional thickener sizing and for high rate thickener sizing for primary grinding product, first cleaner tailing + gold-bearing pyrite concentrate, cyanidation residues, and rougher/scavenger flotation tailing.
- filtration testing, including vacuum filtration and pressure filtration for bulk flotation concentrate.

Cyanide Recovery Tests & Cyanide Destruction Tests

Test Material Preparation

A large, agitated bulk cyanide leach test was conducted by SGS on a 20-kg combined sample of first cleaner tailing and pyrite rougher concentrate. The sample was sourced from material generated from the flotation pilot plant testing at G&T.

The leach pulp of the bulk cyanidation test was allowed to settle and 16.7 L of solution were decanted (pregnant solution). The thickened pulp was diluted with 33.3 L of de-ionized water to simulate washing. The diluted pulp was well agitated then allowed to settle. A 26.7-L portion of the supernatant solution was collected (wash solution). The pregnant solution and washed residue pulp were further treated by contacting with cyanide-treated carbon. The resulting barren solution and the washed residue pulp were used for cyanide recovery and destruction testing, respectively.

The cyanide accountability for bulk leaching was close to 100%. The estimated amount of NaCN consumed by the formation of thiocyanate was 1 kg/t feed, while 0.5 kg/t feed equivalent NaCN was oxidized to cyanate. The amount of equivalent NaCN complexed with copper was 2.38 kg/t feed, and the free cyanide determined by a titration with silver nitrate was 0.35 kg/t NaCN.

The cyanide complexed with copper and the free cyanide should be recoverable by the AVR process or the SART process. The AVR process is able to recover the cyanide into a higher cyanide concentration solution than the SART process. A significant drawback of the AVR process, compared with the SART process, is that the cyanide associated with the copper cyanide complex is unrecoverable.

The key chemical analysis of the solution for cyanide recovery and the washed leach pulp for the cyanide destruction are shown in Table 13.51.

Table 13.51 Chemical Analysis of Cyanide Recovery Test Solution and Cyanide Destruction Pulp

Sample	CN_T (mg/L)	CN_{WAD} (mg/L)	CN_F (mg/L)	Cu (mg/L)	Fe (mg/L)	CNS (mg/L)
Leach Solution	853	850	280	562	1.6	700
Washed Pulp	94	90	-	90.4	1.08	220

Note: **CN_T** = total cyanide; **CN_{WAD}** = weak acid dissociable cyanide; **CN_F** = free cyanide.

Cyanide Recovery Tests

Exploratory AVR tests were conducted to investigate the effect of pH on the recovery of cyanide from the barren leach solution. The scrubbing retention time was 4 h; the collected cyanide, acid consumption, and lime consumption are summarized in Table 13.52.

Table 13.52 Cyanide Recovery Test Results – AVR

pH	Recovered CN_{WAD} (%)	Sulphuric Acid Addition (g/L)	Hydrated Lime Addition (g/L)
2	77	3.18	0.78
3	72	2.01	0.24
4	35	1.14	0.16

Exploratory SART tests were also conducted on the barren leach solution to investigate the effects of pH and sodium hydrosulfide (NaHS) dosage on recovering cyanide and copper from CN_{WAD} and copper cyanide complexes. The test results are as follows:

- At an NaHS dosage of 100% stoichiometric requirement, 83 to 94% of the copper was precipitated when reducing the pH level from 5 to 3.

- At pH 3, an increase of NaHS dosage to 120% of the stoichiometric requirement resulted in near complete removal of copper from the solution and regeneration of all the weak acid dissociable cyanide as free cyanide.
- The sulphuric acid addition was approximately 1.9 g/L of feed solution, and the hydrated lime requirement for reneutralization of the SART treated solution was 1.3 g/L of feed solution.

Further optimization of the SART conditions could improve upon these results, should SART be considered for recovery of cyanide into low-concentration cyanide solutions. These SART-generated cyanide solutions might also be considered for feed to further AVR processing to generate higher grade cyanide solutions for recycle to the leaching circuits.

Cyanide Destruction Tests

Three different cyanide destruction methods, including SO_2/air, Caro's acid (H_2SO_5), and hydrogen peroxide (H_2O_2), were tested for oxidation of cyanide and detoxification of the washed pulp. The objective of the test work was to produce treated effluent containing <2 mg/L CN_{WAD}. The results of the cyanide destruction test results are summarized in Table 13.53.

Table 13.53 Cyanide Destruction Test Results – 2009/2010 (SGS)

Test	Method	Oxidant Dosage Stoich (%)	Cumulative Retention Time (~h)	Composition (Solution Phase)			Cumulative Reagent Addition[1] (g/g CN_{WAD})					
				pH	CN_T mg/L	CN_{WAD} mg/L	SO_2 Equivalent	Lime	Cu	H_2SO_5 100%	H_2O_2 100%	Cu mg/L Solution
Cyanidation Washed Pulp				10.7	94	90						
CND 6&7	SO_2/Air	160-200	1	9.6	2-4	<1	4-5	-	0.14	-	-	12
C-1	Caro's Acid	500	1.5[2]	9.0	2.8	1.7	-[3]	37	-	21.9	-	-
H-1	H_2O_2	500	1.5[2]	10.1	12	11	-	-	-	-	6.5	-
SO_2/Air Partially Treated Pulp				10.0	10	10						
C-2	Caro's Acid	500	1.5[2]	9.0	2.8	1.7	-	-	-	21.6		-
H-7	H_2O_2	1,000	0.5	10.0	2.3	0.3	-	-	1.5	-	13	15
SO_2/Air Partially Treated Solution				10.0	10	10						
H-4	H_2O_2	500	1	8.7	1.6	0.4	-	-	-	-	6.5	-

(1) Cu added as $CuSO_4\ 5H_2O$; SO_2 added as $Na_2S_2O_5$
(2) reagent added in three 30-min stages
(3) not used/analyzed.

The results indicated that the residual CN_{WAD} in the washed pulp was reduced to <1 mg/L after the pulp was treated with 4 to 5 g equivalent SO_2 and 0.14 g Cu (added as copper sulphate) per gram of CN_{WAD} in the pulp. The reaction time for this process was one hour at the natural pH. The SO_2/air-treated pulp contained small amounts of CN_T in the form of ferrocyanide complex.

An exploratory test indicated that the residual CN_{WAD} in the solution phase of the washed pulp was reduced to less than 2 mg/L level by using Caro's acid treatment. The reagent consumption was 0.74 g H_2SO_5 (250% of the stoichiometric amount) and 0.6 g/L hydrated lime of the feed to the cyanide destruction.

The tests also indicated that the hydrogen peroxide (H_2O_2) process was not very efficient for cyanide destruction. The residue CN_{WAD} was only reduced from 90 mg/L to 11 mg/L after adding 500% of the stoichiometrically required H_2O_2.

Two-stage cyanide destruction involving SO_2/air treatment followed by a polishing treatment with Caro's acid or hydrogen peroxide was investigated on the pulp and also a tailing filtrate solution. The SO_2/air treated pulp was adjusted with NaCN to 10 mg/L CN_{WAD} for the polishing tests. The results are as follows:

- The polishing test using Caro's acid was unsuccessful. The final product still contained 3.2 mg/CN_{WAD} after the addition of 500% of the stoichiometric Caro's acid.
- The H_2O_2 polishing treatment produced <2 mg/L residual CN_{WAD}. The H_2O_2 dosage was 10 times of the stoichiometric requirement and the copper addition was 0.011 g/L pulp.
- The solution phase (filtrate) of the SO_2/air partially treated pulp responded well to the H_2O_2 polishing treatment. The solution contained less than 1 mg/L residual CN_{WAD} after being treated with five times the stoichiometric H_2O_2 requirement (0.065 g/L solution). Copper sulphate was not used in the treatment of this solution.

Settling Tests

Thickening

Preliminary settling tests were conducted on pyrite rougher flotation tailing in the 2008 testing program. As reported by G&T, the tests on the tailing slurry failed to generate normal settling curves. The tests were subsequently carried out on the re-pulped sample from dried tailing.

The test data reveal that the settling area required for pyrite rougher flotation tailing was 0.73 m^2/t/d without adding flocculent and 0.30 m^2/t/d with the addition of 10 g/t of flocculent.

In 2009, Pocock conducted solids liquid separation (SLS) tests on five flotation products generated by G&T from the bench scale tests and pilot plant tests. The materials tested included flotation feed, copper concentrate, first cleaner tails + gold-bearing pyrite concentrate, cyanidation residues, and rougher/scavenger flotation tailing. The dewatering tests included:

- flocculent screening tests
- static and dynamic thickening tests for conventional thickener sizing and for high rate thickener sizing
- viscosity (rheological properties) tests for rake mechanism and underflow pipeline sizing
- vacuum filtration tests
- pressure filtration tests.

Hychem AF 303 (a medium to high molecular weight, 7% charge density, anionic polyacrylamide) was selected for thickening tests from preliminary screening of a series of flocculents.

The key test results are summarized in Table 13.54 and Table 13.55.

Table 13.54 Recommended Conventional Thickener Operating Parameters – 2009 (Pocock)

Material Tested	Feed (% Solids)	Flocculent (g/t)	Underflow (% Solids)	Unit Area (m^2/t/d)
Flotation Feed Comp	20-25	10-15	60-65	0.125
Coarse Grind Flotation Feed	25-30	10-15	70-74	0.125
Final Copper Concentrate	25-30	5-10	70-72	0.125
Rougher Tailing	15-20	10-15	60-62	0.125
Au-Pyrite Conc. & Cu Cleaner Tailing	15-20	20-25	55-58	0.275-0.307
Cyanide Leach Reside	10-15	20-25	50-53	0.284-0.312

Notes:
- All tests were performed at 20°C and as received pH.
- Hydraulic loading or rise rate (m^3/m^2h) includes a 0.5 scale-up factor.
- Unit area includes a 1.25 scale-up factor; the range of unit areas provided corresponds to the range of underflow densities.
- Coarse grind flotation feed: at a particle size of P_{80} 170 um; simulating stage one primary grind size.

Table 13.55 Recommended High Rate Thickener Operating Parameters – 2009 (Pocock)

Material Tested	Feed (% Solids)	Flocculent (g/t)	Underflow (% Solids)	Net Feed Loading (m^3/m^2h)
Flotation Feed Comp	15-20	15-20	60-65	4.8-6.1
Coarse Grind Flotation Feed	20-25	10-15	70-74	4.8-6.1
Rougher Flotation Tailing	15-20	~20	57-62	3.7-4.8

Filtration

The 2009 Pocock testing program also determined the filtration rates of the copper concentrates produced from G&T pilot plant tests. Both vacuum filtration and pressure filtration methods were tested. The test results are summarized in Table 13.56.

Table 13.56 Filtration Test Results – 2009 (Pocock)

Filtration Method	Bulk Cake Density (dry kg/m^3)	Cake Thickness (mm)	Cake Moisture (%)	Filtration Rate (dry kg/m^2h)	Dry Cake Weight (dry kg/m^2)
Vacuum	1,785	15	19	265(1)	-
Pressure	2,511	51	8	-	117.8(2)

(1) includes scale up factors at vacuum of 67.7 kPa.
(2) feed pressure 552 kPa at 51 mm thickness.

Magnetic Separation Tests

In the 2008 test program, Davis Tube magnetic separation was used in an effort to recover the metal values lost in the coarser than 74 µm 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.

CONCLUSIONS

The substantial test results indicate that the mineral samples from the four separate mineralization deposits are amenable to the flotation-cyanidation combined process. The process consists of:

- copper-gold-molybdenum bulk rougher flotation followed by gold-bearing pyrite flotation
- regrinding the resulting bulk rougher concentrate followed by three stages of cleaner flotation to produce a copper-gold-molybdenum bulk cleaner flotation concentrate

- molybdenum separation of the bulk cleaner flotation concentrate to produce a molybdenum concentrate and a copper/gold concentrate containing associated silver
- cyanide leaching of the gold-bearing pyrite flotation concentrate and the scavenger cleaner tailing to further recover gold and silver values as doré bullion.

The samples from the Mitchell and Sulphurets deposits produced better metallurgical results with the chosen flotation circuit and cyanide leach extraction when compared to the metallurgical results from the samples taken from the Iron Cap and Kerr deposits.

13.2 METALLURGICAL PERFORMANCE PROJECTION

The metallurgical test results obtained from the various test programs were used to predict plant metallurgical performance parameters for copper, gold, silver, and molybdenum. Gold and silver recoveries were based on the combined process of flotation to a saleable concentrate followed by cyanidation of a combined cleaner tailing and pyrite flotation concentrate. The flotation process will produce a copper concentrate containing approximately 25% Cu and a molybdenum concentrate with 50% Mo. The gold cyanidation process on gold-bearing pyrite products will produce a gold-silver doré.

The Mitchell mineralization produced better metallurgical performances, compared to the Sulphurets, Kerr, and Iron Cap mineralization. The metallurgical performance projections of the different KSM ores are summarized in Table 13.57 to Table 13.60.

Table 13.57 Cu-Au Concentrate – Cu Grade

Copper Head	Copper Grade
> 0.80%	27%
0.40 - 0.80%	26%
0.15 - 0.40%	25%
0.10 - 0.15%	23%
0.05-0.10%	17%
< 0.05%	5%

Table 13.58 Cu-Au Concentrate – Metal Recovery Projections

	Head Grade	Recovery
Mitchell		
Copper Recovery	> 1.0% Cu	= 95%
	0.8 - 1.0% Cu	= 92%
	0.234 - 0.8% Cu	= 90.86 x (Cu Head, %) $^{0.027}$
	0.05 - 0.234% Cu	= 18.02 x ln(Cu Head, %) + 113.5
	0.02 - 0.05% Cu	= 20%
	< 0.02%	= 3%
Gold Recovery	n/a	= 0.096 x (Cu Recovery, %) $^{1.446}$
Silver Recovery	n/a	= 1.427 x (Cu Recovery, %) - 70.11
Sulphurets		
Copper Recovery	> 1.0% Cu	= 93%
	0.8 - 1.0% Cu	= 90%
	0.234 - 0.8% Cu	= 90.86 x (Cu Head, %) $^{0.027}$ - 3.5
	0.05 - 0.234% Cu	= 18.02 x ln(Cu Head, %) + 110
	0.02 - 0.05% Cu	= 20%
	< 0.02%	= 3%
Gold Recovery	n/a	= 52.07 x ln(Cu Recovery, %) - 174.1
Silver Recovery	n/a	= 1.065 x (Cu Recovery, %) - 44.80; if copper recovery < 50%, use 5%
Kerr		
Copper Recovery	> 1.0% Cu	= 88%
	0.8 - 1.0% Cu	= 85%
	0.234 - 0.8% Cu	= 90.86 x (Cu Head, %) $^{0.027}$ - 7
	0.05 - 0.234% Cu	= 18.02 x ln(Cu Head, %) + 106.5
	0.02 - 0.05% Cu	= 20%
	< 0.02% Cu	= 3%
Gold Recovery	n/a	= 171.8 x ln(Cu Recovery, %) - 718; if copper recovery < 70%, use 5%
Silver Recovery	n/a	= 132.48 x ln(Cu Recovery, %) - 542.9; if copper recovery < 70%, use 5%
Iron Cap		
Copper Recovery	> 1.0% Cu	= 95%
	0.8 - 1.0% Cu	= 92%
	0.234 - 0.8% Cu	= 90.86 x (Cu Head, %) $^{0.027}$
	0.05 - 0.234% Cu	= 18.02 x ln(Cu Head, %) + 113.5
	0.02 - 0.05% Cu	= 20%
	< 0.02% Cu	= 3%
Gold Recovery	< 8 g/t Au	= 7.457 x ln(Au Head, g/t) + 53.88
	> 8 g/t Au	= 70%
Silver Recovery	n/a	= 61%

Table 13.59 Au-Ag Doré – Metal Recovery Projections

Head Grade	Recovery
Mitchell	
Gold	
<0.1 g/t Au	= 0%
0.1 - 5 g/t Au	= (87.491 x (Au Head, g/t)$^{0.051}$ - (0.096 x (Cu Recovery, %) $^{1.446}$)) x 66% x 98%
5 - 10 g/t Au	= (95 - (0.096 x (Cu Recovery, %) $^{1.446}$)) x 75% x 98%
>10 g/t Au	= (98 - (0.096 x (Cu Recovery, %) $^{1.446}$)) x 80% x 98%
Silver	
< 1 g/t Ag	= 0%
1 - 8 g/t Ag	= (42.74 x (Ag Head, g/t) $^{0.336}$) - (1.427 x (Cu Recovery, %) - 70.11) ; if <0, use 0%
8- 15 g/t Ag	= 86 - (1.427 x (Cu Recovery, %) - 70.11)
>15 g/t Ag	= 88 - (1.427 x (Cu Recovery, %) - 70.11)
Sulphurets	
Gold	
<0.1 g/t Au	= 0%
0.1 - 5 g/t Au	= ((87.491 x (Au Head, g/t)$^{0.051}$ +3)- (52.07 x ln(Cu Recovery, %) - 174.1)) x 49% x 98%
5 - 10 g/t Au	= (95 - (52.07 x ln(Cu Recovery, %) - 174.1)) x 60% x 98%
> 10 g/t Au	= (98 - (52.07 x ln(Cu Recovery, %) - 174.1)) x 70% x 98%
Silver	
< 1 g/t Ag	= 0%
1 - 8 g/t Ag	= (42.74 x (Ag Head, g/t) $^{0.336}$) - (1.065 x (Cu Recovery, %) - 44.80)
8- 15 g/t Ag	= 50.7%
>15 g/t Ag	= 52.7%
Kerr	
Gold	
< 0.1 g/t Au	= 0%
0.1 - 5 g/t Au	= ((87.491 x (Au Head, g/t)$^{0.051}$ + 8)- (171.8 x ln(Cu Recovery, %) - 718))) x 57% x 98%
5 - 10 g/t Au	= (95 - (171.8 x ln(Cu Recovery, %) - 718)) x 65% x 98%
> 10 g/t Au	= (98 - (171.8 x ln(Cu Recovery, %) - 718)) x 75% x 98%
Silver	
< 1 g/t Ag	= 0%
1 - 8 g/t Ag	= (21.59 x ln(Ag Head, g/t) + 40.14) - (132.48 x ln(Cu Recovery, %) - 542.9) ; if <0, use 0
8- 15 g/t Ag	= (86 - (132.48 x ln(Cu Recovery, %) - 542.9))/100; Cap at 86%
>15 g/t Ag	= (88 - (132.48 x ln(Cu Recovery, %) - 542.9))/100; Cap at 88%
Iron Cap	
Gold	
< 0.1 g/t Au	= 0%
0.1 - 8 g/t Au	= (4.278 x ln(Au Head, g/t) + 69.62) - (7.457 x ln(Au Head, g/t) + 53.88)
8 -20 g/t Au	= (4.278 x ln(Au Head, g/t) + 69.62) - 70
> 20 g/t Au	= 20%

table continues...

Head Grade	Recovery
Silver	
< 3g/t Ag	= 0%
1 - 8 g/t Ag	= (21.26 x ln(Ag Head, g/t) + 40.74) - 61
8- 15 g/t Ag	= 25%
>15 g/t Ag	= 27%

Table 13.60 Mo Concentrate Metal Recovery and Grade

Mo Head	Mo Recovery
> 0.010%	47%
0.005-0.010%	35%
0.0025-0.005%	25%
<0.0025%	0%
Molybdenum Grade = 50%	

14.0 MINERAL RESOURCE ESTIMATES

Mineral Resources were estimated for the KSM Project by Mr. Michael J. Lechner, President of RMI. Mr. Lechner is a P.Geo. (BC), a Registered Professional Geologist in the State of Arizona, is a Certified Professional Geologist with the American Institute of Professional Geologists (AIPG), and a registered member of the Society for Mining, Metallurgy, and Exploration (SME). These professional registrations together with Mr. Lechner's professional background and work experience allow him to be the Qualified Person 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.

14.1 GOLD GRADE DISTRIBUTION

Block gold grades were estimated by assay grades that were composited into 15 m long drill hole composites, after high-grade outlier values were capped. Section 14.3 discusses grade capping. Various geologic wireframes were used to constrain the estimate of block grades for each zone. These geologic wireframes represent either distinct alteration types (e.g. Kerr and Iron Cap) or a combination of alteration/lithology and gold grade (Sulphurets and Mitchell).

The distribution of gold based on raw uncomposited data is summarized at four different cut-off grades by the geologic constraint that was used in the estimation process in Table 14.1 to Table 14.4 for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively.

As shown in Table 14.1 through Table 14.4, the average gold grade increases going from the Kerr deposit in the southern part of the district to the Sulphurets (middle portion of district) to the Mitchell deposit in the north. The average gold grade of the Iron Cap Zone is between the mean grade of the Kerr and Sulphurets zones. In addition to the gold grade increasing from south to north the percentage of material above a 0.50 g/t gold cut-off also increases from Kerr (7%) to Sulphurets (24%) to Mitchell (44%). The percentage of Iron Cap gold grades above 0.50 g/t is 22%. Another important statistical parameter is that the coefficient of variation (CV) is relatively low for all for mineralized zones. The CV for uncapped Mitchell gold grade assays is 1.01. That CV is reduced to 0.86 after high-grade outliers are capped (Section 14.3).

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 mineralized zones except for the Kerr deposit. 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 (AUZON). Mineral zones and constraints used to estimate block grades are discussed in Section 14.5.

14.2 COPPER GRADE DISTRIBUTION

The distribution of copper grades based on the original drill hole intervals is summarized at four different cut-off grades by the geologic constraints that were used to estimate block copper grades in Table 14.5 through Table 14.8 for the Kerr, Sulphurets, Mitchell, and Iron Cap deposits, respectively.

As can be seen in Table 14.5 through Table 14.8, in general, the average copper grade decreases in going from the Kerr deposit in the southern part of the district to the Sulphurets (middle portion of district) to the Mitchell deposit. This is an inverse relationship to that of gold. In the Kerr deposit about 41% of the copper assays are above a 0.25% copper cut-off.

Like gold, copper is seen to be distributed in a number of logged lithologic and alteration types in the four mineralized zones. 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 except for Kerr where alteration was used to constrain the estimate of block grades. Copper grades tend to be somewhat lower in chlorite-propylitic alteration than quartz-sericite-pyrite alteration, but this relationship is not well developed. Given these observations, RMI elected to use grade envelopes for Sulphurets, Mitchell, and Iron Cap to constrain the estimate of block copper grades (CUZON) (Section 14.5).

Table 14.1 Distribution of Gold by Alteration – Kerr Zone

	Uncapped Au Statistics Above Cut-off								Capped Au Statistics Above Cut-off				
Alteration	Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0.00	29,327	74%	0.22	6,385	37.1%	0.51	2.34	0.21	6,089	38.9%	0.27	1.30
	0.25	7,719	19%	0.52	4,016	30.0%	0.92	1.77	0.48	3,720	31.4%	0.40	0.83
	0.50	2,055	5%	1.02	2,101	16.5%	1.68	1.64	0.88	1,805	17.5%	0.61	0.70
	1.00	448	2%	2.34	1,046	16.4%	3.27	1.40	1.70	741	12.2%	0.92	0.54
CL-PR	0.00	10,797	61%	0.25	2,684	30.7%	0.22	0.87	0.25	2,677	30.7%	0.21	0.84
	0.25	4,257	29%	0.44	1,861	40.5%	0.23	0.52	0.44	1,854	40.6%	0.21	0.49
	0.50	1,075	9%	0.72	775	22.9%	0.29	0.40	0.71	768	23.0%	0.25	0.34
	1.00	123	1%	1.32	162	6.0%	0.44	0.33	1.27	154	5.8%	0.26	0.20
QSP	0.00	11,037	81%	0.18	2,005	50.6%	0.35	1.94	0.18	1,942	52.3%	0.22	1.23
	0.25	2,058	15%	0.48	990	26.3%	0.73	1.53	0.45	926	27.2%	0.37	0.83
	0.50	452	3%	1.02	462	11.8%	1.44	1.40	0.88	399	12.2%	0.61	0.69
	1.00	91	1%	2.49	226	11.3%	2.74	1.10	1.79	163	8.4%	0.88	0.49
Weak CLQSP	0.00	2,161	75%	0.22	482	33.9%	0.54	2.41	0.20	442	37.0%	0.29	1.40
	0.25	537	18%	0.59	318	28.0%	0.98	1.66	0.52	278	30.5%	0.43	0.82
	0.50	149	5%	1.23	183	14.4%	1.70	1.39	0.96	144	15.7%	0.61	0.63
	1.00	47	2%	2.41	114	23.7%	2.66	1.11	1.57	74	16.8%	0.77	0.49
Premier Dyke	0.00	654	91%	0.09	62	56.7%	0.15	1.58	0.09	59	59.5%	0.13	1.40
	0.25	56	6%	0.48	27	22.5%	0.23	0.49	0.42	24	23.6%	0.12	0.29
	0.50	17	2%	0.75	13	10.3%	0.23	0.31	0.58	10	17.0%	0.02	0.04
	1.00	6	1%	1.07	6	10.5%	0.00	0.00	0.00	0	0.0%	0.00	0.00

table continues...

Alteration	Uncapped Au Statistics Above Cut-off: Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Capped Au Statistics Above Cut-off: Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
Hornblende Dyke	0.00	306	99%	0.06	17	91.3%	0.06	1.06	0.06	17	91.3%	0.06	1.06
	0.25	3	0%	0.50	2	0.0%	0.00	0.00	0.50	2	0.0%	0.00	0.00
	0.50	3	1%	0.50	2	8.7%	0.00	0.00	0.50	2	8.7%	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
Undefined	0.00	4,371	82%	0.26	1,136	27.9%	1.07	4.13	0.22	952	33.2%	0.46	2.10
	0.25	808	10%	1.01	819	13.5%	2.35	2.31	0.79	636	16.1%	0.84	1.07
	0.50	358	4%	1.86	666	11.3%	3.33	1.79	1.35	483	14.0%	1.02	0.76
	1.00	181	4%	2.98	538	47.3%	4.42	1.48	1.99	350	36.7%	1.13	0.57

Table 14.2 Distribution of Gold by AUZON – Sulphurets Zone

AUZON	Uncapped Au Statistics Above Cut-off: Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Capped Au Statistics Above Cut-off: Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0.00	35,450	52%	0.41	14,393	13.9%	0.77	1.89	0.39	13,933	14.4%	0.54	1.37
	0.25	17,023	24%	0.73	12,392	20.7%	1.01	1.38	0.70	11,933	21.4%	0.64	0.92
	0.50	8,663	16%	1.09	9,415	27.9%	1.32	1.21	1.03	8,955	28.8%	0.77	0.74
	1.00	2,877	8%	1.88	5,404	37.5%	2.06	1.09	1.72	4,944	35.5%	1.01	0.59
1	0.00	1,258	8%	1.12	1,410	1.1%	1.34	1.19	1.06	1,335	1.1%	0.92	0.87
	0.25	1,157	19%	1.21	1,395	6.4%	1.36	1.13	1.14	1,320	6.8%	0.92	0.81
	0.50	915	36%	1.42	1,304	23.6%	1.45	1.02	1.34	1,230	24.9%	0.93	0.70
	1.00	465	37%	2.09	971	68.9%	1.80	0.86	1.93	896	67.1%	1.00	0.52

table continues...

Uncapped Au Statistics Above Cut-off									**Capped Au Statistics Above Cut-off**				
AUZON	**Au Cut-off (g/t)**	**Total Metres**	**Inc. Percent**	**Mean Au (g/t)**	**Grd-Thk (g/t-m)**	**Inc. Percent**	**Std. Dev.**	**Coeff. of Variation**	**Mean Au (g/t)**	**Grd-Thk (g/t-m)**	**Inc. Percent**	**Std. Dev.**	**Coeff. of Variation**
2	0.00	1,514	54%	0.30	448	25.0%	0.30	1.01	0.29	433	25.9%	0.23	0.81
	0.25	694	33%	0.48	336	39.3%	0.35	0.73	0.46	321	40.6%	0.23	0.51
	0.50	192	10%	0.83	160	20.5%	0.52	0.63	0.76	145	21.2%	0.26	0.35
	1.00	45	3%	1.53	68	15.2%	0.71	0.46	1.19	53	12.3%	0.08	0.07
3	0.00	7,511	21%	0.59	4,463	5.7%	0.59	0.99	0.59	4,443	5.7%	0.55	0.93
	0.25	5,917	33%	0.71	4,210	20.5%	0.61	0.86	0.71	4,190	20.6%	0.56	0.80
	0.50	3,432	32%	0.96	3,294	37.7%	0.70	0.73	0.95	3,275	37.9%	0.63	0.66
	1.00	1,001	13%	1.61	1,611	36.1%	1.03	0.64	1.59	1,592	35.8%	0.87	0.55
4	0.00	8,075	28%	0.58	4,709	7.2%	1.10	1.88	0.56	4,502	7.5%	0.55	0.99
	0.25	5,830	34%	0.75	4,372	21.0%	1.25	1.67	0.71	4,165	22.0%	0.58	0.81
	0.50	3,091	25%	1.09	3,384	30.5%	1.64	1.50	1.03	3,176	31.9%	0.64	0.63
	1.00	1,037	13%	1.88	1,948	41.4%	2.65	1.41	1.68	1,741	38.7%	0.75	0.44
5	0.00	1,816	57%	0.34	618	23.7%	0.60	1.77	0.30	544	27.0%	0.26	0.88
	0.25	787	29%	0.60	471	28.5%	0.84	1.41	0.50	397	32.4%	0.28	0.56
	0.50	268	11%	1.10	295	22.0%	1.30	1.18	0.82	221	25.0%	0.27	0.33
	1.00	70	4%	2.28	159	25.7%	2.14	0.94	1.22	85	15.6%	0.06	0.05
6	0.00	3,470	72%	0.25	866	32.5%	0.61	2.43	0.24	827	34.0%	0.36	1.50
	0.25	970	19%	0.60	584	26.4%	1.07	1.77	0.56	546	27.6%	0.55	0.97
	0.50	302	6%	1.18	356	15.1%	1.77	1.51	1.05	318	15.8%	0.78	0.74
	1.00	103	3%	2.20	225	26.0%	2.77	1.26	1.82	187	22.6%	0.92	0.50
7	0.00	2,630	92%	0.11	291	56.0%	0.33	2.95	0.10	261	62.5%	0.15	1.49
	0.25	199	5%	0.64	128	16.5%	1.03	1.61	0.49	98	18.4%	0.29	0.58
	0.50	58	1%	1.38	80	9.3%	1.70	1.23	0.86	50	10.4%	0.27	0.32
	1.00	19	1%	2.79	53	18.2%	2.40	0.86	1.21	23	8.8%	0.07	0.06

table continues...

AUZON	Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
		Uncapped Au Statistics Above Cut-off							Capped Au Statistics Above Cut-off				
29	0.00	9,176	84%	0.17	1,589	43.6%	0.45	2.62	0.17	1,589	43.6%	0.45	2.62
	0.25	1,470	12%	0.61	896	22.3%	1.02	1.67	0.61	896	22.3%	1.02	1.67
	0.50	405	3%	1.34	542	11.0%	1.74	1.30	1.34	542	11.0%	1.74	1.30
	1.00	138	2%	2.66	368	23.1%	2.49	0.94	2.66	368	23.1%	2.49	0.94

Table 14.3 Distribution of Gold by AUZON – Mitchell

AUZON	Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
		Uncapped Au Statistics Above Cut-off							Capped Au Statistics Above Cut-off				
All Data	0.00	54,436	31%	0.51	27,596	6.7%	0.51	1.01	0.50	27,388	6.8%	0.43	0.86
	0.25	37,742	26%	0.68	25,738	18.7%	0.53	0.77	0.68	25,529	18.8%	0.41	0.61
	0.50	23,809	34%	0.86	20,589	46.6%	0.59	0.68	0.86	20,380	47.0%	0.42	0.49
	1.00	5,409	10%	1.43	7,719	28.0%	1.02	0.72	1.39	7,510	27.4%	0.59	0.42
Leach Breccia	0.00	1,642	60%	0.28	465	27.0%	0.29	1.03	0.28	465	27.0%	0.29	1.03
	0.25	663	27%	0.51	340	33.1%	0.34	0.66	0.51	340	33.1%	0.34	0.66
	0.50	220	10%	0.85	186	24.1%	0.42	0.49	0.85	186	24.1%	0.42	0.49
	1.00	51	3%	1.43	73	15.8%	0.49	0.34	1.43	73	15.8%	0.49	0.34
Bornite Breccia	0.00	194	38%	0.33	64	19.2%	0.21	0.63	0.33	64	19.2%	0.21	0.63
	0.25	120	47%	0.43	52	48.0%	0.20	0.47	0.43	52	48.0%	0.20	0.47
	0.50	29	14%	0.72	21	28.5%	0.22	0.30	0.72	21	28.5%	0.22	0.30
	1.00	2	1%	1.38	3	4.3%	0.00	0.00	1.38	3	4.3%	0.00	0.00

table continues...

AUZON	Au Cut-off (g/t)	Uncapped Au Statistics Above Cut-off: Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Capped Au Statistics Above Cut-off: Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
1.00 g/t Envelope	0.00	5,540	3%	1.09	6,042	0.2%	0.90	0.82	1.06	5,893	0.2%	0.59	0.55
	0.25	5,391	3%	1.12	6,029	0.9%	0.89	0.80	1.09	5,881	1.0%	0.57	0.52
	0.50	5,249	48%	1.14	5,972	34.7%	0.90	0.79	1.11	5,824	35.6%	0.57	0.51
	1.00	2,576	46%	1.51	3,877	64.2%	1.16	0.77	1.45	3,729	63.3%	0.65	0.45
0.75 g/t Envelope	0.00	10,427	5%	0.75	7,785	0.8%	0.35	0.46	0.75	7,785	0.8%	0.35	0.46
	0.25	9,897	13%	0.78	7,724	7.1%	0.32	0.41	0.78	7,724	7.1%	0.32	0.41
	0.50	8,545	65%	0.84	7,175	63.2%	0.30	0.36	0.84	7,175	63.2%	0.30	0.36
	1.00	1,778	17%	1.27	2,255	29.0%	0.38	0.30	1.27	2,255	29.0%	0.38	0.30
0.50 g/t Envelope	0.00	14,681	9%	0.55	8,037	2.3%	0.29	0.53	0.55	8,036	2.3%	0.29	0.53
	0.25	13,403	37%	0.59	7,853	26.5%	0.28	0.47	0.59	7,851	26.5%	0.27	0.47
	0.50	7,914	49%	0.72	5,720	59.1%	0.28	0.39	0.72	5,718	59.1%	0.28	0.39
	1.00	719	5%	1.35	972	12.1%	0.54	0.40	1.35	971	12.1%	0.52	0.38
0.20 g/t Envelope	0.00	9,724	35%	0.35	3,435	16.7%	0.44	1.25	0.35	3,397	16.9%	0.29	0.82
	0.25	6,347	51%	0.45	2,861	50.3%	0.52	1.15	0.44	2,823	50.8%	0.31	0.71
	0.50	1,418	12%	0.80	1,133	21.7%	1.01	1.27	0.77	1,095	22.0%	0.54	0.69
	1.00	203	2%	1.91	387	11.3%	2.38	1.25	1.72	349	10.3%	0.94	0.55
0.10 g/t Envelope	0.00	7,255	82%	0.17	1,197	55.8%	0.15	0.91	0.17	1,197	55.8%	0.15	0.91
	0.25	1,319	15%	0.40	530	29.9%	0.20	0.49	0.40	530	29.9%	0.20	0.49
	0.50	235	3%	0.73	172	10.8%	0.26	0.36	0.73	172	10.8%	0.26	0.36
	1.00	33	0%	1.27	42	3.5%	0.19	0.15	1.27	42	3.5%	0.19	0.15
Undefined	0.00	4,972	88%	0.11	570	38.6%	0.38	3.33	0.11	549	40.1%	0.26	2.34
	0.25	603	8%	0.58	350	24.4%	0.97	1.67	0.55	329	25.4%	0.56	1.02
	0.50	199	3%	1.06	210	17.8%	1.57	1.49	0.95	189	18.4%	0.83	0.87
	1.00	47	1%	2.33	109	19.2%	2.89	1.24	1.89	88	16.1%	1.31	0.69

Table 14.4 Distribution of Gold by AUZON – Iron Cap Zone

AUZON	Uncapped Au Statistics Above Cut-off								Capped Au Statistics Above Cut-off				
	Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0.00	17,564	44%	0.39	6,886	16.8%	0.42	1.07	0.38	6,676	17.4%	0.36	0.95
	0.25	9,846	34%	0.58	5,728	31.1%	0.48	0.82	0.56	5,513	32.5%	0.40	0.70
	0.50	3,891	16%	0.92	3,588	28.1%	0.61	0.67	0.88	3,346	28.5%	0.48	0.54
	1.00	1,018	6%	1.62	1,654	24.0%	0.85	0.52	1.50	1,444	21.6%	0.59	0.39
Lower Zone	0.00	9,188	22%	0.53	4,847	8.2%	0.49	0.93	0.51	4,725	8.4%	0.41	0.80
	0.25	7,128	43%	0.62	4,451	30.3%	0.52	0.83	0.61	4,330	31.0%	0.43	0.70
	0.50	3,149	25%	0.95	2,984	32.1%	0.64	0.67	0.91	2,863	32.9%	0.49	0.54
	1.00	876	10%	1.63	1,429	29.5%	0.88	0.54	1.49	1,308	27.7%	0.59	0.39
Middle Zone	0.00	1,799	83%	0.18	318	57.8%	0.16	0.92	0.17	301	61.2%	0.12	0.71
	0.25	311	13%	0.43	134	24.6%	0.25	0.58	0.37	117	27.6%	0.12	0.32
	0.50	73	3%	0.77	56	11.3%	0.32	0.42	0.58	34	11.2%	0.09	0.16
	1.00	15	1%	1.34	20	6.3%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Upper Zone	0.00	1,130	45%	0.30	337	28.9%	0.15	0.49	0.29	329	29.7%	0.13	0.44
	0.25	626	47%	0.38	240	51.4%	0.15	0.38	0.37	231	56.4%	0.12	0.33
	0.50	98	9%	0.68	66	19.7%	0.13	0.19	0.67	46	14.0%	0.07	0.10
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
PMON Au-Cu Zone	0.00	350	47%	0.30	105	28.2%	0.15	0.51	0.27	95	36.5%	0.13	0.47
	0.25	185	40%	0.41	75	46.2%	0.13	0.32	0.38	61	46.3%	0.10	0.27
	0.50	44	12%	0.62	27	25.6%	0.08	0.12	0.58	16	17.2%	0.00	0.01
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00

table continues...

AUZON	Uncapped Au Statistics Above Cut-off								Capped Au Statistics Above Cut-off				
	Au Cut-off (g/t)	Total Metres	Inc. Percent	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
FW Weak Min	0.00	3,217	62%	0.29	939	27.8%	0.35	1.19	0.28	890	29.3%	0.29	1.05
	0.25	1,218	24%	0.56	679	28.9%	0.45	0.80	0.52	630	31.6%	0.35	0.69
	0.50	449	10%	0.91	407	21.6%	0.58	0.64	0.82	349	24.0%	0.46	0.56
	1.00	128	4%	1.61	205	21.8%	0.68	0.43	1.55	135	15.2%	0.55	0.36
Mo-Zn Zone	0.00	1,135	74%	0.20	227	48.2%	0.14	0.70	0.20	223	49.1%	0.13	0.67
	0.25	297	21%	0.40	117	35.7%	0.12	0.30	0.38	114	38.8%	0.11	0.28
	0.50	60	5%	0.61	36	16.0%	0.04	0.07	0.60	27	12.1%	0.04	0.07
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Barren PMON	0.00	330	86%	0.15	50	72.1%	0.08	0.54	0.15	50	72.1%	0.08	0.54
	0.25	45	14%	0.31	14	27.9%	0.04	0.13	0.31	14	27.9%	0.04	0.13
	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
Undefined	0.00	414	91%	0.15	62	71.5%	0.13	0.84	0.15	62	71.5%	0.13	0.84
	0.25	36	4%	0.50	18	10.8%	0.14	0.27	0.50	18	10.8%	0.14	0.27
	0.50	19	5%	0.59	11	17.7%	0.09	0.15	0.59	11	17.7%	0.09	0.15
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00

Table 14.5 Distribution of Copper by Alteration – Kerr Zone

Alteration	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
	Cu Cut-off (g/t)	Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0.00	29,185	27%	0.31	8,917	1.9%	0.38	1.25	0.30	8,859	1.9%	0.37	1.22
	0.05	21,448	13%	0.41	8,752	3.0%	0.40	0.97	0.41	8,693	3.0%	0.38	0.95
	0.10	17,732	20%	0.48	8,488	11.0%	0.40	0.84	0.48	8,429	11.1%	0.39	0.82
	0.25	11,916	41%	0.63	7,505	84.2%	0.41	0.65	0.62	7,447	84.1%	0.39	0.63
CL-PR	0.00	10,797	11%	0.50	5,377	0.5%	0.44	0.88	0.50	5,375	0.5%	0.44	0.88
	0.05	9,577	4%	0.56	5,350	0.6%	0.43	0.77	0.56	5,348	0.6%	0.43	0.77
	0.10	9,112	18%	0.58	5,317	6.4%	0.42	0.73	0.58	5,315	6.4%	0.42	0.73
	0.25	7,202	67%	0.69	4,974	92.5%	0.42	0.60	0.69	4,972	92.5%	0.41	0.60
QSP	0.00	11,037	18%	0.25	2,808	1.9%	0.31	1.20	0.25	2,798	1.9%	0.30	1.17
	0.05	9,048	19%	0.30	2,754	5.2%	0.32	1.04	0.30	2,743	5.2%	0.31	1.01
	0.10	6,998	28%	0.37	2,608	18.1%	0.33	0.88	0.37	2,597	18.2%	0.32	0.85
	0.25	3,936	36%	0.53	2,099	74.7%	0.36	0.68	0.53	2,088	74.6%	0.34	0.65
Weak CLQSP	0.00	2,161	34%	0.17	358	4.9%	0.24	1.44	0.16	355	4.9%	0.23	1.40
	0.05	1,436	24%	0.24	340	10.7%	0.27	1.12	0.24	338	10.8%	0.25	1.08
	0.10	908	22%	0.33	302	20.5%	0.29	0.88	0.33	300	20.7%	0.28	0.84
	0.25	435	20%	0.53	229	63.9%	0.33	0.62	0.52	226	63.6%	0.30	0.58
Premier Dyke	0.00	654	59%	0.13	84	4.4%	0.29	2.28	0.08	53	7.0%	0.11	1.37
	0.05	266	13%	0.30	80	7.4%	0.40	1.33	0.19	50	11.6%	0.11	0.58
	0.10	184	17%	0.40	74	21.2%	0.45	1.11	0.24	44	33.2%	0.09	0.40
	0.25	76	12%	0.74	56	67.0%	0.53	0.71	0.34	26	48.2%	0.02	0.07

table continues...

Alteration	Cu Cut-off (g/t)	Uncapped Cu Statistics Above Cut-off: Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Capped Cu Statistics Above Cut-off: Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
Hornblende Dyke	0.00	306	84%	0.03	11	54.2%	0.04	1.11	0.03	11	54.2%	0.04	1.11
	0.05	49	10%	0.10	5	19.4%	0.06	0.60	0.10	5	19.4%	0.06	0.60
	0.10	17	5%	0.16	3	22.5%	0.06	0.35	0.16	3	22.5%	0.06	0.35
	0.25	1	0%	0.30	0	4.0%	0.00	0.00	0.30	0	4.0%	0.00	0.00
Undefined	0.00	4,229	75%	0.07	280	20.7%	0.22	3.29	0.06	267	21.6%	0.14	2.20
	0.05	1,072	13%	0.21	222	13.6%	0.40	1.93	0.20	209	14.2%	0.23	1.17
	0.10	512	6%	0.36	184	13.0%	0.54	1.50	0.33	171	13.6%	0.27	0.80
	0.25	266	6%	0.56	148	52.7%	0.69	1.24	0.51	135	50.5%	0.27	0.54

Table 14.6 Distribution of Copper by CUZON – Sulphurets Zone

CUZON	Cu Cut-off (g/t)	Uncapped Cu Statistics Above Cut-off: Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Capped Cu Statistics Above Cut-off: Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0.00	34,934	40%	0.14	4,719	6.3%	0.19	1.42	0.13	4,673	6.3%	0.19	1.39
	0.05	21,052	22%	0.21	4,422	11.4%	0.22	1.03	0.21	4,377	11.5%	0.21	1.00
	0.10	13,499	23%	0.29	3,884	26.8%	0.24	0.82	0.28	3,839	27.5%	0.23	0.79
	0.25	5,411	15%	0.48	2,620	55.5%	0.27	0.56	0.48	2,554	54.7%	0.25	0.52
Au Zone	0.00	1,033	57%	0.07	71	20.3%	0.09	1.34	0.07	70	20.6%	0.09	1.28
	0.05	439	26%	0.13	56	25.8%	0.11	0.90	0.13	55	26.1%	0.11	0.84
	0.10	170	12%	0.22	38	27.8%	0.14	0.61	0.22	37	28.2%	0.12	0.56
	0.25	44	4%	0.42	18	26.1%	0.12	0.28	0.40	17	25.1%	0.09	0.23

table continues...

CUZON	Cu Cut-off (g/t)	Uncapped Cu Statistics Above Cut-off: Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Capped Cu Statistics Above Cut-off: Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
Leach Au Zone	0.00	1,453	74%	0.04	57	38.8%	0.04	1.09	0.04	55	39.8%	0.04	0.99
	0.05	381	19%	0.09	35	33.6%	0.05	0.58	0.09	33	34.4%	0.04	0.45
	0.10	100	6%	0.16	16	21.9%	0.06	0.41	0.14	14	25.9%	0.03	0.23
	0.25	10	1%	0.32	3	5.6%	0.04	0.13	0.00	0	0.0%	0.00	0.00
Raewyn Cu Zone	0.00	7,411	7%	0.33	2,436	0.6%	0.29	0.88	0.33	2,427	0.6%	0.28	0.85
	0.05	6,905	11%	0.35	2,422	2.6%	0.29	0.82	0.35	2,414	2.6%	0.28	0.79
	0.10	6,070	31%	0.39	2,360	15.8%	0.29	0.74	0.39	2,351	15.8%	0.27	0.71
	0.25	3,779	51%	0.52	1,975	81.1%	0.29	0.56	0.52	1,967	81.0%	0.27	0.52
Lower Au Zone	0.00	8,012	34%	0.09	760	9.9%	0.11	1.16	0.09	756	9.9%	0.10	1.11
	0.05	5,248	34%	0.13	685	25.3%	0.12	0.93	0.13	681	25.4%	0.11	0.88
	0.10	2,494	26%	0.20	493	39.3%	0.15	0.76	0.20	489	39.4%	0.14	0.70
	0.25	450	6%	0.43	194	25.6%	0.22	0.52	0.42	191	25.2%	0.18	0.43
FW Hazelton	0.00	1,816	20%	0.11	191	6.1%	0.09	0.82	0.10	183	6.4%	0.06	0.62
	0.05	1,454	39%	0.12	179	27.4%	0.09	0.71	0.12	171	28.5%	0.06	0.50
	0.10	750	37%	0.17	127	51.4%	0.10	0.59	0.16	119	53.5%	0.05	0.34
	0.25	73	4%	0.39	29	15.1%	0.19	0.48	0.29	21	11.6%	0.01	0.05
Main Copper Zone	0.00	3,470	11%	0.18	642	1.7%	0.15	0.81	0.18	631	1.7%	0.13	0.72
	0.05	3,086	17%	0.20	631	7.2%	0.15	0.72	0.20	620	7.3%	0.13	0.62
	0.10	2,482	48%	0.24	585	42.6%	0.15	0.63	0.23	574	43.4%	0.12	0.53
	0.25	822	24%	0.38	311	48.5%	0.18	0.47	0.37	300	47.6%	0.12	0.33
Main Cu Monzonite	0.00	2,630	64%	0.06	157	22.3%	0.07	1.24	0.05	144	24.3%	0.06	1.03
	0.05	936	16%	0.13	122	19.2%	0.09	0.65	0.12	109	20.9%	0.05	0.45
	0.10	505	16%	0.18	92	38.8%	0.09	0.48	0.16	79	54.8%	0.04	0.24
	0.25	95	4%	0.33	31	19.8%	0.09	0.27	0.00	0	0.0%	0.00	0.00

table continues...

CUZON	Cu Cut-off (g/t)	Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
Undefined	0.00	9,108	71%	0.04	407	28.0%	0.07	1.53	0.04	407	28.0%	0.07	1.53
	0.05	2,603	18%	0.11	293	28.9%	0.10	0.87	0.11	293	28.9%	0.10	0.87
	0.10	929	9%	0.19	175	28.9%	0.13	0.70	0.19	175	28.9%	0.13	0.70
	0.25	137	2%	0.42	58	14.2%	0.21	0.49	0.42	58	14.2%	0.21	0.49

Table 14.7 Distribution of Copper by CUZON – Mitchell Zone

CUZON	Cu Cut-off (g/t)	Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
All Data	0.00	54,433	86%	0.15	8,007	65.7%	0.15	1.01	0.14	7,851	67.0%	0.11	0.75
	0.25	7,561	13%	0.36	2,743	27.8%	0.28	0.77	0.34	2,588	28.8%	0.11	0.31
	0.50	625	1%	0.83	521	4.1%	0.82	0.98	0.65	324	4.0%	0.14	0.21
	1.00	107	0%	1.80	192	2.4%	1.65	0.92	1.22	7	0.1%	0.21	0.18
Leach Breccia	0.00	6	0%	1.11	7	0.0%	0.55	0.50	0.98	6	0.0%	0.38	0.39
	0.25	6	0%	1.11	7	0.0%	0.55	0.50	0.98	6	0.0%	0.38	0.39
	0.50	6	67%	1.11	7	43.7%	0.55	0.50	0.98	6	49.2%	0.38	0.39
	1.00	2	33%	1.87	4	56.3%	0.00	0.00	1.50	3	50.8%	0.00	0.00
Bornite Breccia	0.00	194	10%	0.95	185	1.7%	0.71	0.74	0.33	63	5.0%	0.06	0.19
	0.25	174	25%	1.05	182	10.5%	0.69	0.66	0.35	60	95.0%	0.01	0.04
	0.50	125	24%	1.30	163	17.2%	0.66	0.50	0.00	0	0.0%	0.00	0.00
	1.00	78	40%	1.68	131	70.6%	0.56	0.33	0.00	0	0.0%	0.00	0.00

table continues...

CUZON	Cu Cut-off (g/t)	Uncapped Cu Statistics Above Cut-off: Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Capped Cu Statistics Above Cut-off: Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
0.30% Envelope	0.00	6,006	43%	0.28	1,694	28.7%	0.12	0.42	0.28	1,693	28.7%	0.12	0.41
	0.25	3,443	53%	0.35	1,209	61.8%	0.11	0.30	0.35	1,207	61.8%	0.10	0.30
	0.50	254	4%	0.64	162	9.3%	0.12	0.19	0.63	160	9.5%	0.11	0.17
	1.00	4	0%	1.11	4	0.3%	0.05	0.04	0.00	0	0.0%	0.00	0.00
0.20% Envelope	0.00	13,451	78%	0.20	2,751	64.5%	0.09	0.46	0.20	2,747	64.5%	0.09	0.44
	0.25	2,977	21%	0.33	978	32.0%	0.11	0.32	0.33	974	32.0%	0.09	0.28
	0.50	148	1%	0.66	98	3.3%	0.23	0.34	0.64	94	3.4%	0.11	0.18
	1.00	4	0%	1.80	7	0.3%	0.47	0.26	0.00	0	0.0%	0.00	0.00
0.10% Envelope	0.00	16,818	97%	0.13	2,144	91.7%	0.06	0.49	0.13	2,142	91.7%	0.06	0.48
	0.25	535	3%	0.33	179	7.3%	0.13	0.38	0.33	177	7.3%	0.10	0.30
	0.50	31	0%	0.70	22	0.8%	0.30	0.42	0.64	20	0.9%	0.13	0.20
	1.00	3	0%	1.62	4	0.2%	0.14	0.09	0.00	0	0.0%	0.00	0.00
0.05% Envelope	0.00	6,536	99%	0.07	466	90.2%	0.23	3.28	0.07	439	95.8%	0.05	0.73
	0.25	36	0%	1.28	46	1.2%	2.91	2.28	0.52	19	1.3%	0.29	0.55
	0.50	16	0%	2.55	40	0.3%	4.05	1.59	0.83	13	3.0%	0.14	0.17
	1.00	13	0%	3.02	38	8.2%	4.37	1.45	0.00	0	0.0%	0.00	0.00
Undefined	0.00	11,422	97%	0.07	760	81.1%	0.08	1.23	0.07	760	81.1%	0.08	1.23
	0.25	390	3%	0.37	144	14.9%	0.14	0.39	0.37	144	14.9%	0.14	0.39
	0.50	45	0%	0.69	31	3.5%	0.17	0.24	0.69	31	3.5%	0.17	0.24
	1.00	4	0%	1.06	4	0.5%	0.02	0.02	1.06	4	0.5%	0.02	0.02

Table 14.8 Distribution of Copper by AUZON – Iron Cap Zone

AUZON	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
	Cu Cut-off (g/t)	Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
All Data	0.00	17,138	8%	0.19	3,239	1.3%	0.12	0.63	0.19	3,214	1.3%	0.11	0.61
	0.05	15,713	15%	0.20	3,197	5.9%	0.11	0.55	0.20	3,172	6.0%	0.11	0.54
	0.10	13,214	52%	0.23	3,006	47.4%	0.11	0.47	0.23	2,980	48.1%	0.10	0.45
	0.25	4,220	25%	0.35	1,469	45.4%	0.11	0.30	0.34	1,435	44.6%	0.10	0.28
Lower Zone	0.00	9,164	0%	0.22	2,044	0.0%	0.12	0.55	0.22	2,027	0.0%	0.12	0.53
	0.05	9,164	11%	0.22	2,044	3.9%	0.12	0.55	0.22	2,027	4.0%	0.12	0.53
	0.10	8,176	57%	0.24	1,964	43.8%	0.12	0.49	0.24	1,946	44.5%	0.11	0.47
	0.25	2,964	32%	0.36	1,069	52.3%	0.11	0.32	0.36	1,045	51.6%	0.10	0.29
Middle Zone	0.00	1,799	6%	0.18	316	1.1%	0.09	0.50	0.17	314	1.1%	0.09	0.50
	0.05	1,700	18%	0.18	312	7.8%	0.08	0.46	0.18	311	7.8%	0.08	0.45
	0.10	1,376	56%	0.21	288	56.2%	0.07	0.34	0.21	286	56.4%	0.07	0.34
	0.25	364	20%	0.30	110	34.9%	0.05	0.17	0.30	109	34.7%	0.05	0.17
Upper Zone	0.00	1,130	1%	0.23	265	0.3%	0.11	0.45	0.23	262	0.3%	0.10	0.42
	0.05	1,115	6%	0.24	265	2.1%	0.10	0.44	0.23	261	2.2%	0.09	0.40
	0.10	1,046	52%	0.25	259	40.4%	0.10	0.40	0.24	255	40.9%	0.09	0.36
	0.25	458	41%	0.33	152	57.3%	0.09	0.28	0.32	148	56.7%	0.07	0.22
PMON Au-Cu Zone	0.00	350	8%	0.20	71	1.7%	0.11	0.53	0.20	71	1.7%	0.11	0.53
	0.05	323	13%	0.22	69	3.9%	0.10	0.47	0.21	69	3.9%	0.10	0.47
	0.10	278	43%	0.24	67	36.7%	0.09	0.36	0.24	67	42.0%	0.09	0.36
	0.25	128	36%	0.32	41	57.7%	0.05	0.15	0.33	37	52.3%	0.04	0.13

table continues...

AUZON	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
	Cu Cut-off (g/t)	Total Metres	Inc. Percent	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation	Mean Cu (%)	Grd-Thk (%-m)	Inc. Percent	Std. Dev.	Coeff. of Variation
FW Weak Min	0.00	2,840	36%	0.09	252	11.5%	0.07	0.79	0.09	250	11.6%	0.07	0.78
	0.05	1,811	29%	0.12	223	23.1%	0.07	0.54	0.12	221	23.2%	0.06	0.53
	0.10	1,001	31%	0.16	164	51.8%	0.06	0.38	0.16	163	51.8%	0.06	0.37
	0.25	108	4%	0.32	34	13.6%	0.04	0.12	0.31	34	13.4%	0.03	0.11
Mo-Zn Zone	0.00	1,135	0%	0.20	223	0.0%	0.07	0.36	0.20	221	0.0%	0.06	0.32
	0.05	1,135	3%	0.20	223	1.1%	0.07	0.36	0.20	221	1.1%	0.06	0.32
	0.10	1,105	84%	0.20	221	76.2%	0.07	0.35	0.20	219	76.8%	0.06	0.30
	0.25	157	14%	0.32	51	22.7%	0.10	0.32	0.31	49	22.1%	0.07	0.23
Barren PMON	0.00	330	50%	0.06	21	31.5%	0.03	0.54	0.06	21	31.5%	0.03	0.54
	0.05	165	41%	0.09	14	47.1%	0.03	0.40	0.09	14	47.1%	0.03	0.40
	0.10	30	9%	0.15	4	21.4%	0.03	0.20	0.15	4	21.4%	0.03	0.20
	0.25	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
Undefined	0.00	390	23%	0.12	48	1.0%	0.09	0.77	0.12	48	1.0%	0.09	0.77
	0.05	300	25%	0.16	47	17.0%	0.08	0.50	0.16	47	17.0%	0.08	0.50
	0.10	201	41%	0.19	39	54.9%	0.07	0.36	0.19	39	54.9%	0.07	0.36
	0.25	41	11%	0.31	13	27.2%	0.05	0.15	0.31	13	27.2%	0.05	0.15

14.3 Assay Grade Capping

RMI used cumulative probability plots to identify high-grade outliers for both gold and copper assays. Figure 14.1 through Figure 14.8 show cumulative probability plots using the cumulative normal distribution function for gold and copper by mineral zone.

Figure 14.1 Kerr Zone Au Assay Cumulative Probability Plot

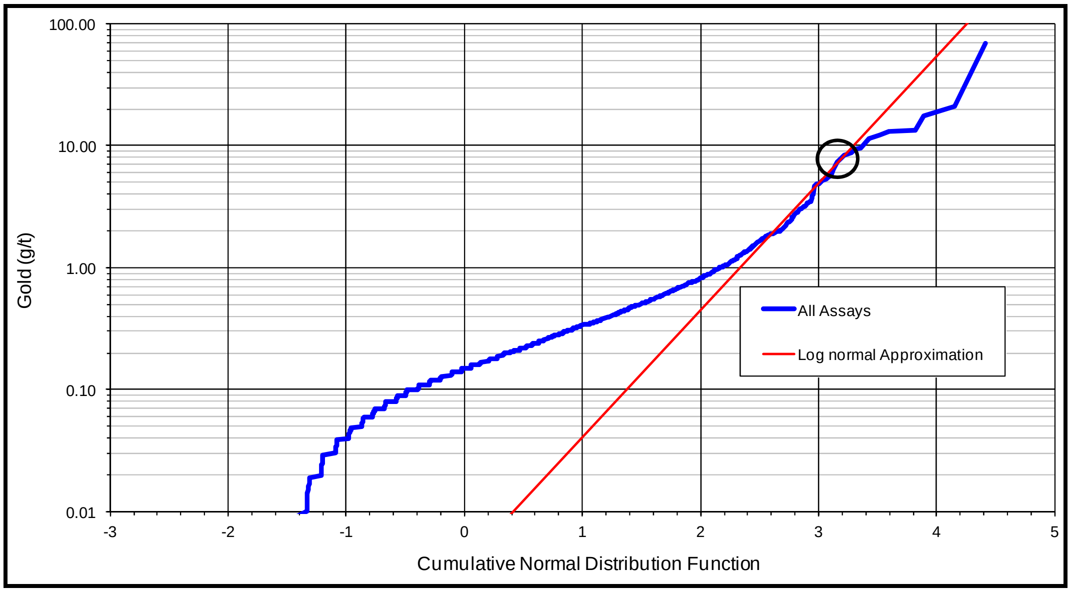


Figure 14.2 Sulphurets Zone Au Assay Cumulative Probability Plot

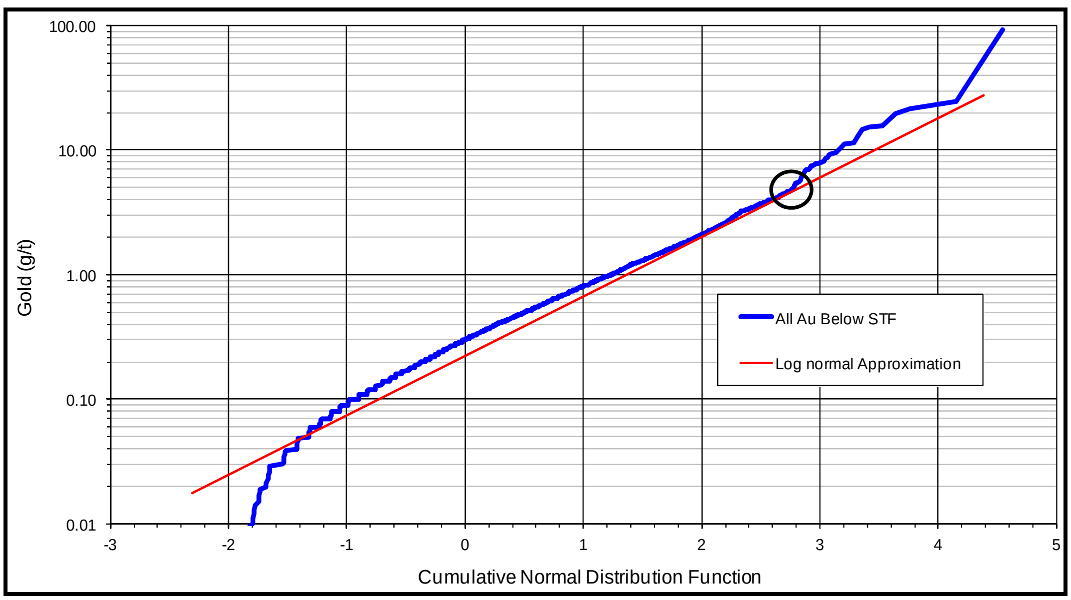


Figure 14.3 Mitchell Zone Au Assay Cumulative Probability Plot

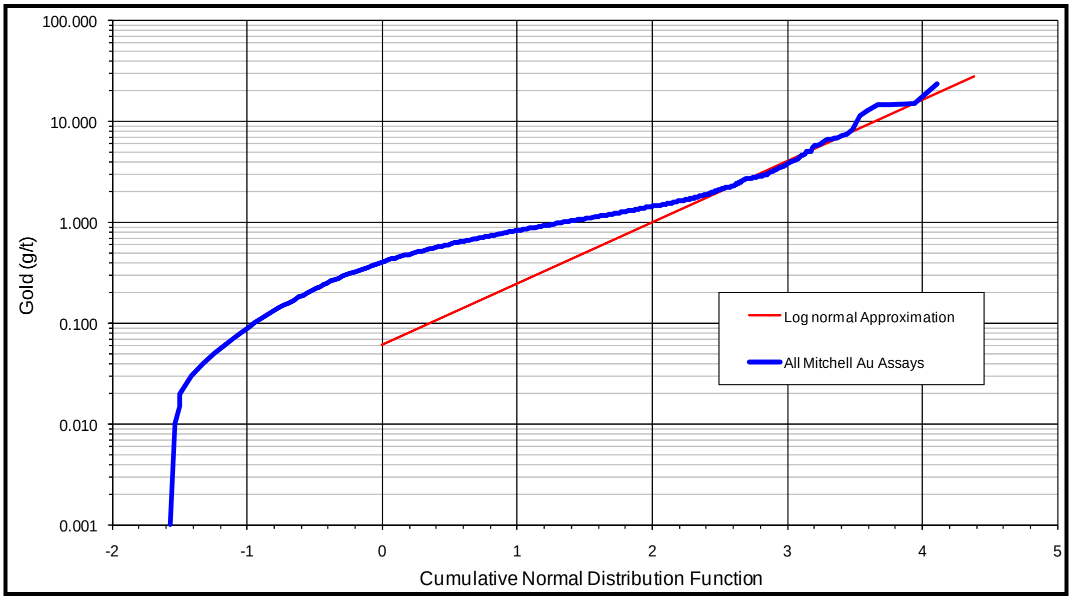


Figure 14.4 Iron Cap Zone Au Assay Cumulative Probability Plot

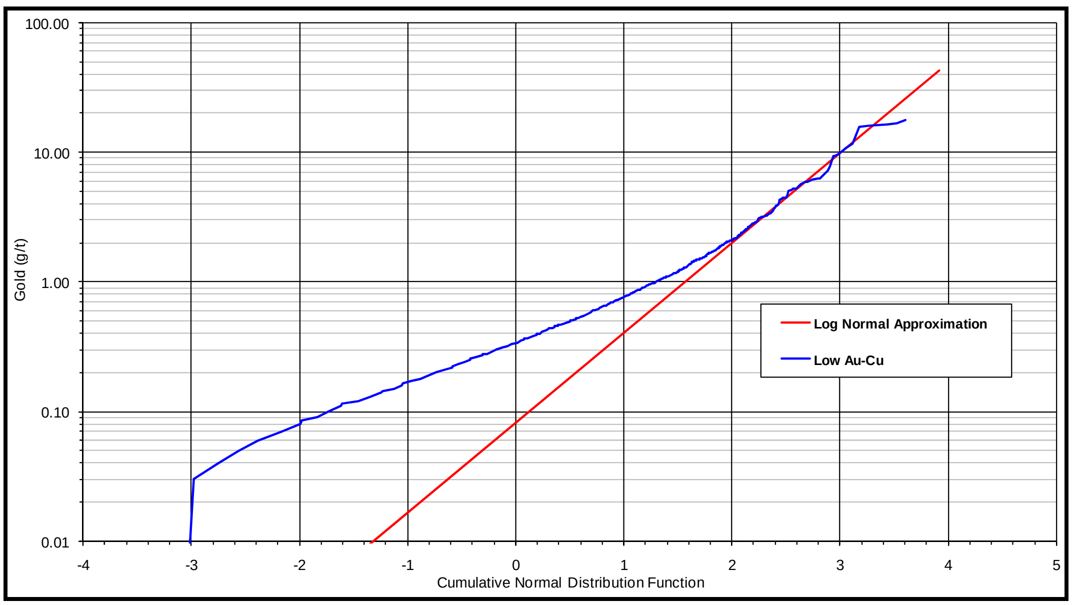


Figure 14.5 Kerr Zone Cu Assay Cumulative Probability Plot

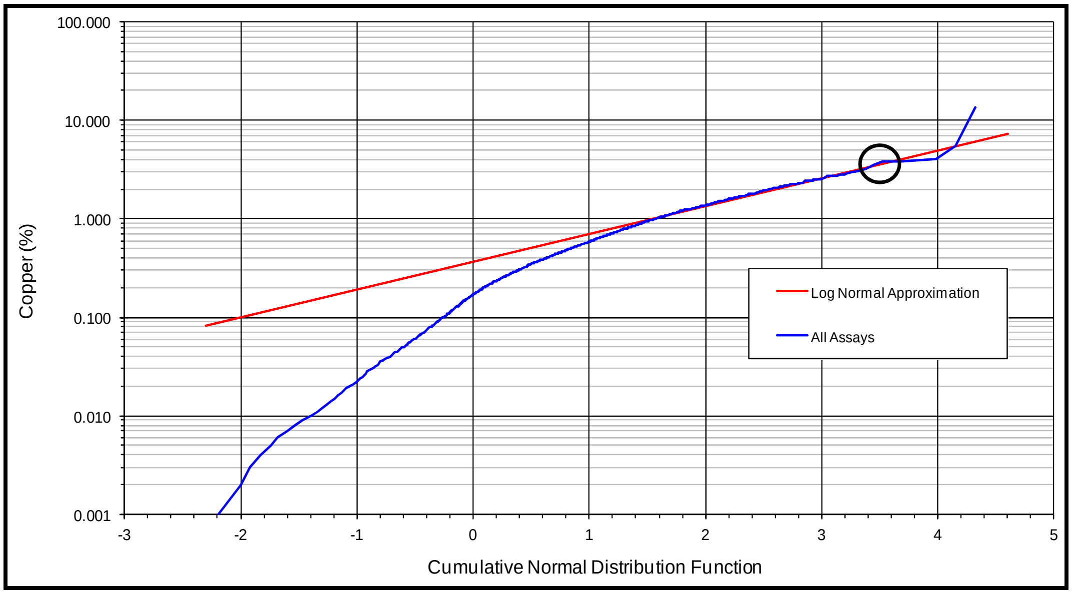


Figure 14.6 Sulphurets Zone Cu Assay Cumulative Probability Plot

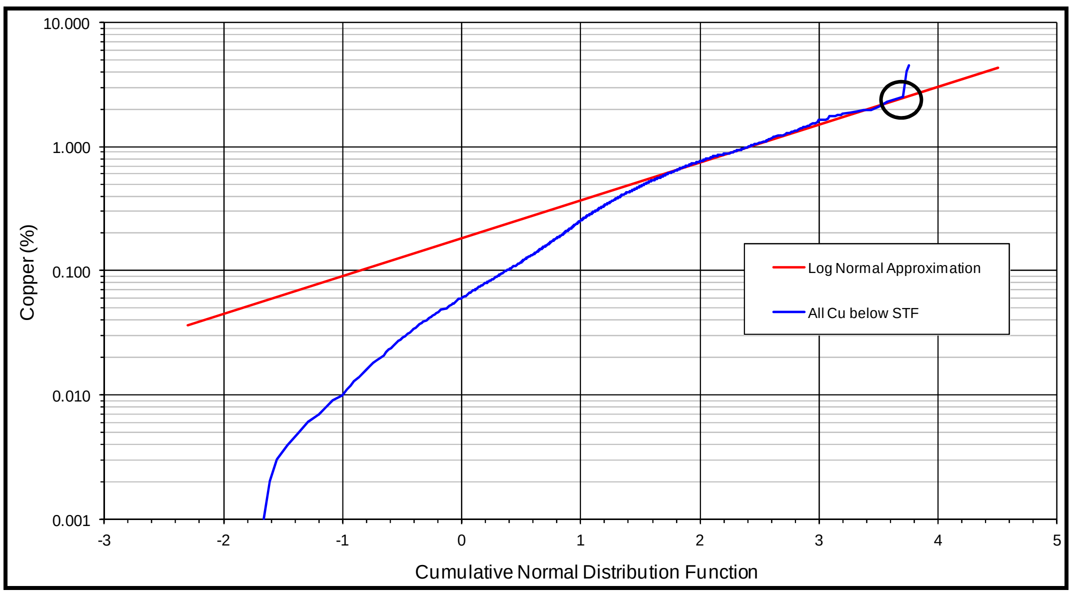


Figure 14.7 Mitchell Zone Cu Assay Cumulative Probability Plot

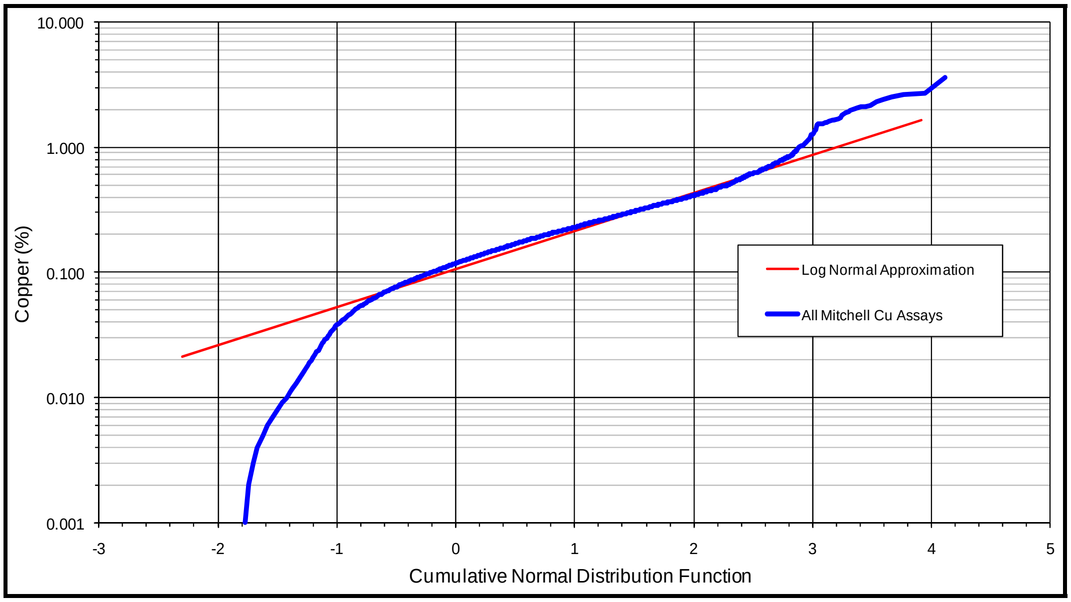
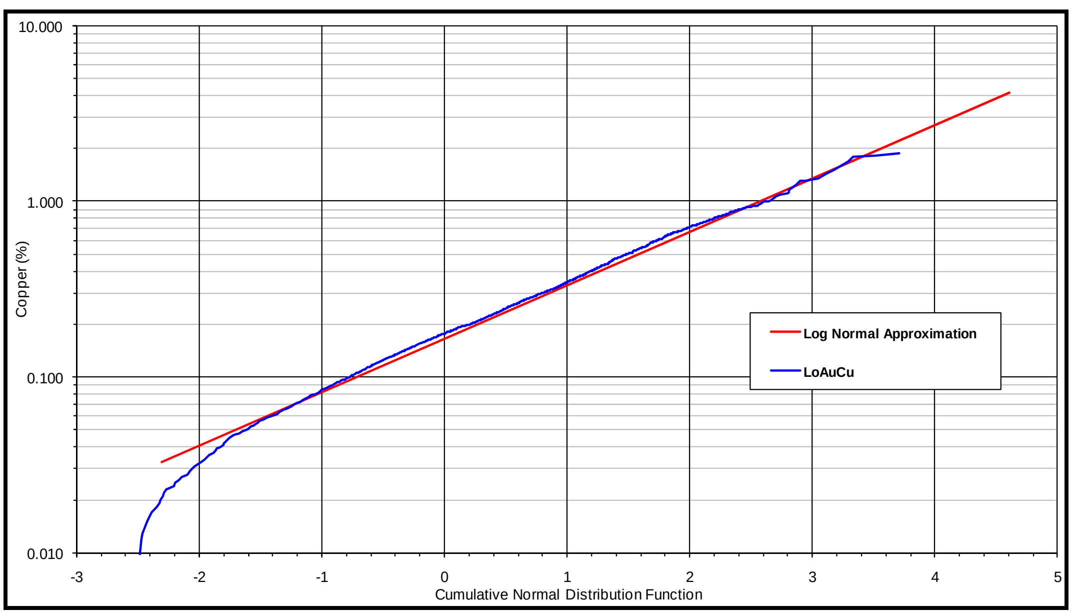


Figure 14.8 Iron Cap Zone Cu Assay Cumulative Probability Plot



Based on the information shown in Figure 14.1 through Figure 14.8 and other cumulative probability plots not shown, RMI capped raw gold and copper assays at the area highlighted by the black circle where the distribution of grades becomes erratic.

Table 14.9 through Table 14.11 summarize the capping limits that were established for gold, copper, and silver/molybdenum by mineral zone.

Table 14.9 Gold Grade Capping Limits

Zone	Attribute	Cap Grade (g/t)
Kerr	HW Intrusive	1.50
	Dykes	0.60
	Mixed HW-FW	2.00
	Uncategorized HW-FW	5.00
	Default	0.90
Sulphurets	Main Cu Hazelton	4.00
	Main Cu Monzonite	1.25
	Main Au Zone	5.00
	Leach Au Zone	1.25
	Raewyn Copper	7.00
	Lower Au Zone	4.00
	FW Hazelton	1.25
Mitchell	All	5.00
Iron Cap	Lower Au Zone	6.50
	Middle Au Zone	1.50
	Upper Au Zone	1.50
	PMON Au-Cu Zone	0.70
	FW Weak Zone	3.00
	Mo-Zn Zone	1.50
	Undefined	1.00

Table 14.10 Copper Grade Capping Limits

Zone	Attribute	Cap Grade (%)
Kerr	HW Intrusive	1.25
	Dykes	0.35
	Mixed HW-FW	3.00
	Uncategorized HW-FW	2.25
	Default	0.60
Sulphurets	Main Cu Hazelton	0.70
	Main Cu Monzonite	0.20
	Main Au Zone	0.50
	Leach Au Zone	0.20
	Raewyn Copper	2.00
	Lower Au Zone	1.00
	FW Hazelton	0.30
Mitchell	Upper Plate	0.90
	Lower Plate	0.90
	Bornite Breccia	1.50
	Bornite Leach Breccia	0.35

table continues…

Zone	Attribute	Cap Grade (%)
Iron Cap	Lower Au Zone	0.90
	Middle Au Zone	0.70
	Upper Au Zone	0.60
	PMON Au-Cu Zone	0.70
	FW Weak Zone	0.60
	Mo-Zn Zone	0.70
	Undefined	0.60

Table 14.11 Silver and Molybdenum Grade Capping Limits

Zone	Attribute	Ag (g/t)	Mo (ppm)
Kerr	All	50	300
Sulphurets	Main Cu Hazelton	20	500
	Main Cu Monzonite	20	500
	All Others	30	1,250
Mitchell	All	180	1,200
Iron Cap	All	n/a	n/a

14.4 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.

14.5 GEOLOGIC CONSTRAINTS

Various lithologic, alteration, structural domains, and metal grade envelopes were constructed for each of the deposits by RMI and Seabridge personnel. Most of these 3D 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 deposits are not necessarily confined to distinct geologic units (e.g. lithology, alteration, etc.). For this reason, alteration zones were used for Kerr while hybrid gold and copper envelopes were used to constrain the estimate of block grades for Sulphurets, Mitchell, and Iron

Cap. Constraints used to estimate gold, silver, copper, and molybdenum are summarized in Table 14.12 for each deposit.

Table 14.12 Constraints Used to Estimate Block Grades

Mineral Zone	Gold	Silver	Copper	Molybdenum
Kerr	Alteration	Alteration	Alteration	n/a
Sulphurets	AUZON	AUZON	CUZON	CUZON
Mitchell	AUZON	AUZON	CUZON	CUZON
Iron Cap	AUZON	AUZON	AUZON	AUZON

Descriptions for alteration types used to constrain the estimate of Kerr gold, silver, and copper grades are summarized in Table 14.13.

Table 14.13 Alteration Code Definitions

Code	Description
1	Chlorite-propylytic
2	Quartz-sericite-pyrite (QSP)
3	Mitchell IARG
4	Kerr weak CLQSP
5	Kerr Premier-style dike
6	Kerr hornblende dike
7	Iron Cap CL-SIL
8	Iron Cap FW SIH
9	Iron Cap KP
10	Iron Cap KP-PMON
11	Iron Cap SIH
12	Iron Cap SIL

The AUZON and CUZON wireframes for the Sulphurets and Mitchell zones are a combination of lithology/alteration and grade. In the case of the Mitchell Zone, the AUZON and CUZON's were more heavily weighted towards grade. A series of gold and copper grade envelopes were designed as 3D wireframes for the Mitchell and Sulphurets zones. In the Sulphurets Zone, the Sulphurets Thrust Fault was used to define upper and lower plates. In the Mitchell Zone, the Mitchell Thrust Fault was used to define upper and lower plates. The AUZON codes used to constrain the estimate of gold, copper, silver, and molybdenum grades for the Iron Cap Zone are a combination of lithology and degree of mineralization. Table 14.14 and Table 14.15 summarize definitions for AUZON and CUZON, respectively.

Table 14.14 AUZON Code Definitions

AUZON	Description
1	Sulphurets Main Gold Zone
2	Sulphurets Leach Gold Zone
3	Sulphurets Raewyn Copper Zone
4	Sulphurets Lower Gold Zone
5	Sulphurets FW Hazelton
6	Sulphurets HW Hazelton
7	Sulphurets Main Copper Monzonite
8	Mitchell Leach Breccia Zone
9	Mitchell Bornite Breccia
10	Mitchell 1.00 g/t Gold Envelope
11	Mitchell 0.75 g/t Gold Envelope
12	Mitchell 0.50 g/t Gold Envelope
13	Mitchell 0.25 g/t Gold Envelope
14	Mitchell 0.10 g/t Gold Envelope
16	Iron Cap Lower Au Zone
17	Iron Cap Middle Au Zone
18	Iron Cap Upper Au Zone
19	Iron Cap PMON Au-Cu Zone
20	Iron Cap Footwall Weak Mineralized Zone
21	Iron Cap Molybdenum-Zinc Zone
22	Iron Cap Barren PMON
29	Default Code

Table 14.15 CUZON Code Definitions

CUZON	Description
1	Sulphurets Main Gold Zone
2	Sulphurets Leach Gold Zone
3	Sulphurets Raewyn Copper Zone
4	Sulphurets Lower Gold Zone
5	Sulphurets FW Hazelton
6	Sulphurets HW Hazelton
7	Sulphurets Main Copper Monzonite
8	Mitchell Leach Breccia Zone
9	Mitchell Bornite Breccia
10	Mitchell 0.30% Copper Envelope
11	Mitchell 0.20% Copper Envelope
12	Mitchell 0.10% Copper Envelope
13	Mitchell 0.05% Copper Envelope
29	Default Code

14.6 VARIOGRAPHY

RMI generated a number of gold and copper correlograms and variograms using both drill hole assays and 15 m long drill hole composites.

Figure 14.9 through Figure 14.12 show gold grade correlograms for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. Figure 14.13 through Figure 14.16 show copper grade correlograms for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. Figure 14.17 through Figure 14.20 show 0.5 g/t AuEQ correlograms for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively.

Figure 14.9 Kerr Zone Au Grade Correlogram

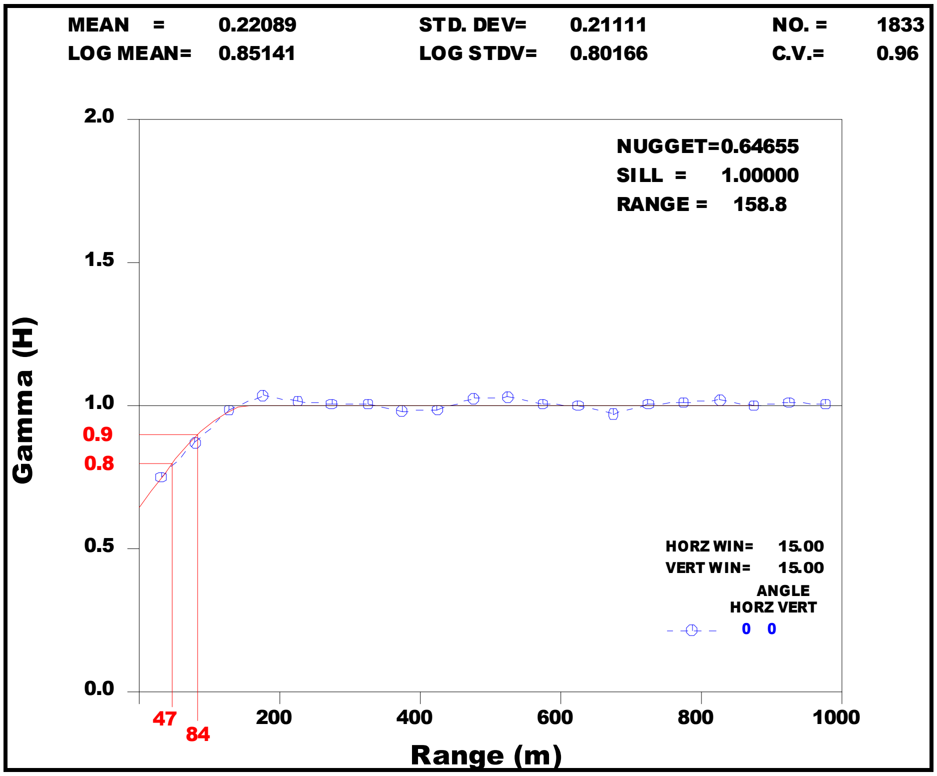


Figure 14.10 Sulphurets Zone Au Grade Correlogram

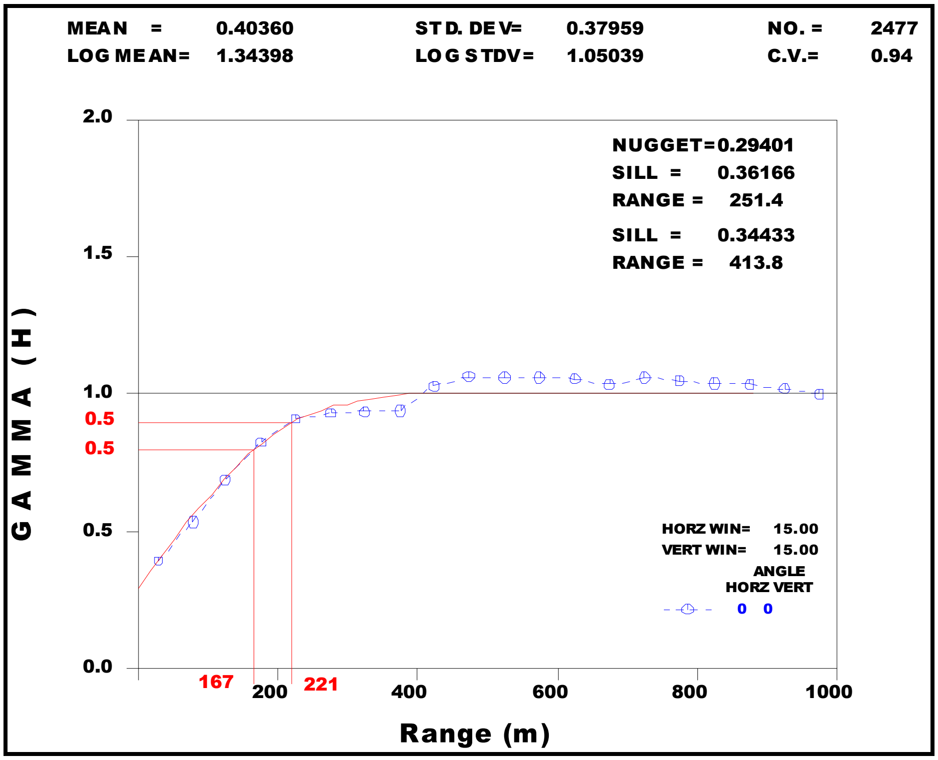


Figure 14.11 Mitchell Zone Au Grade Correlogram

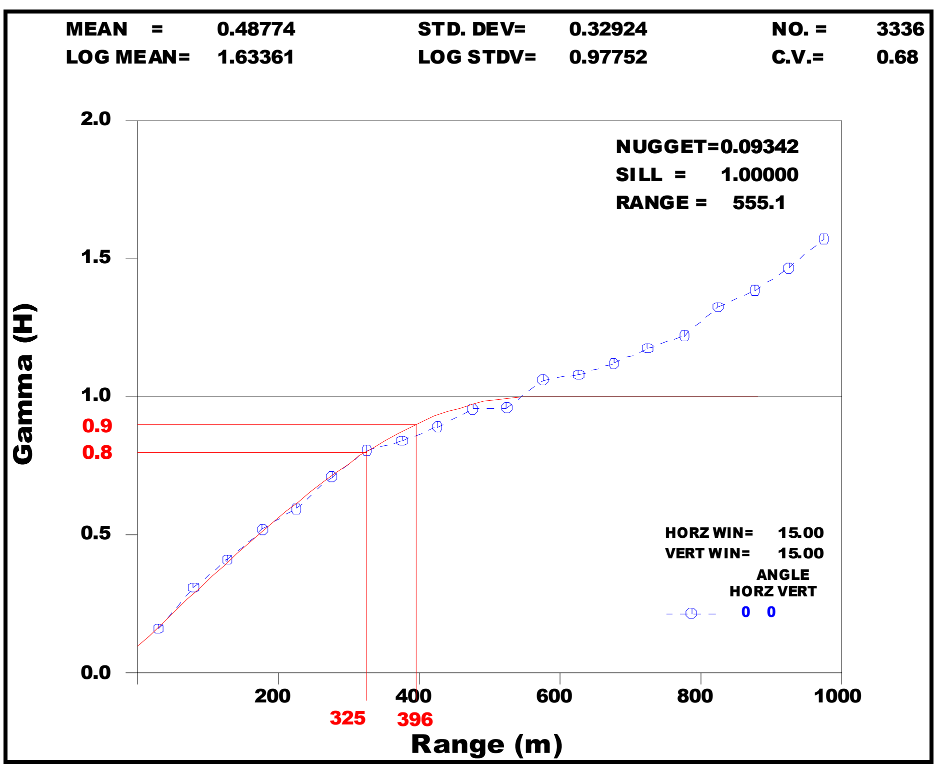


Figure 14.12 Iron Cap Zone Au Grade Correlogram

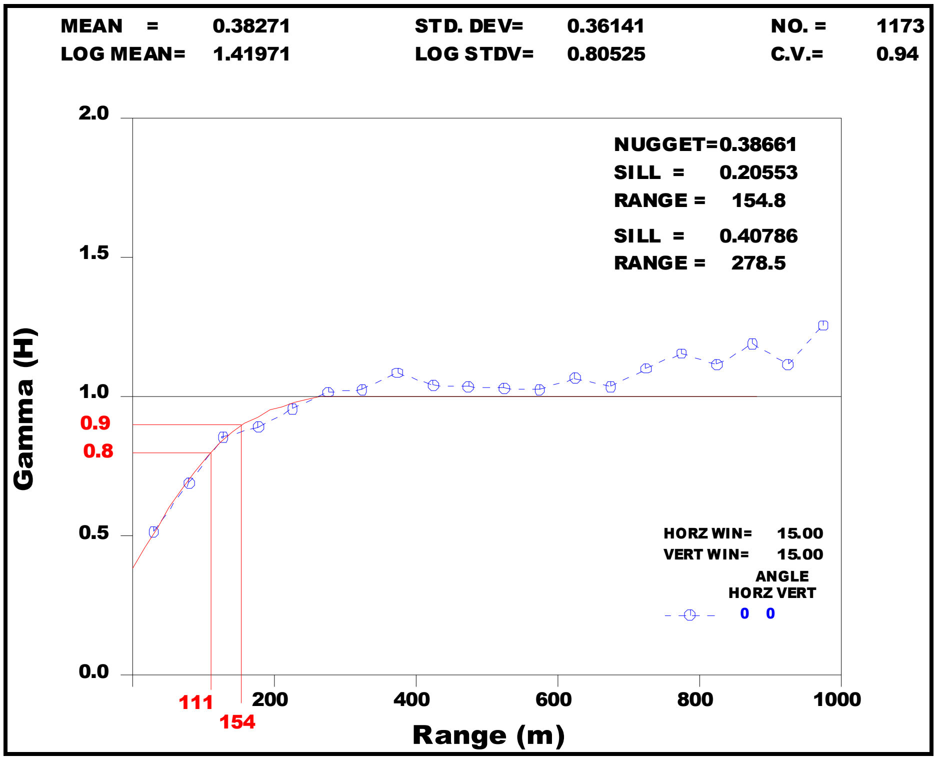


Figure 14.13 Kerr Zone Cu Grade Correlogram

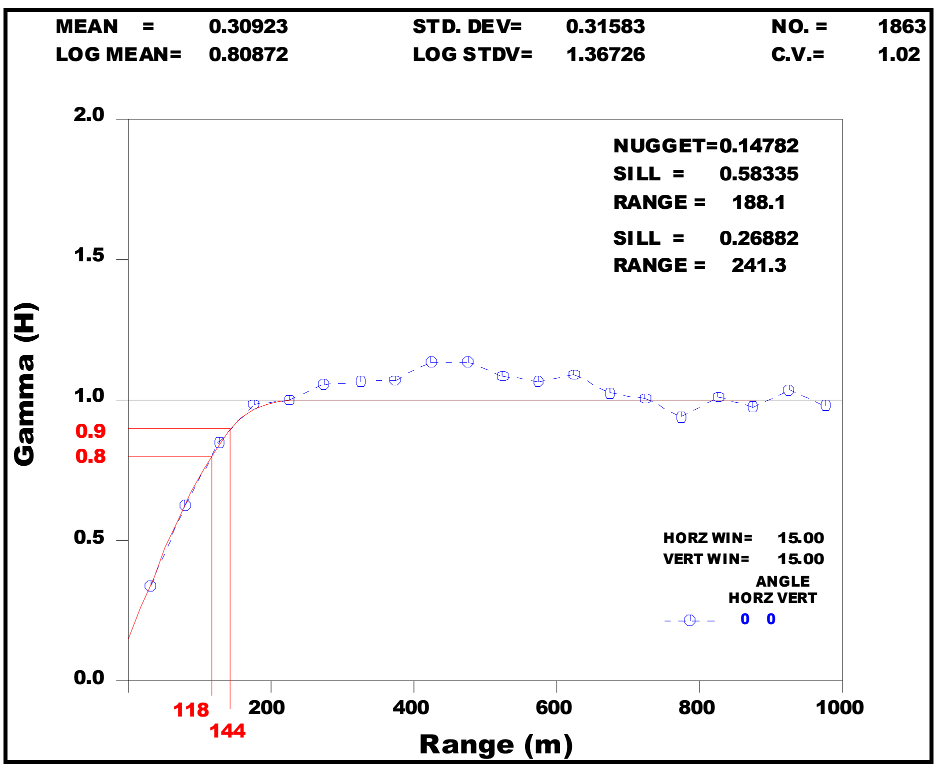


Figure 14.14 Sulphurets Zone Cu Grade Correlogram

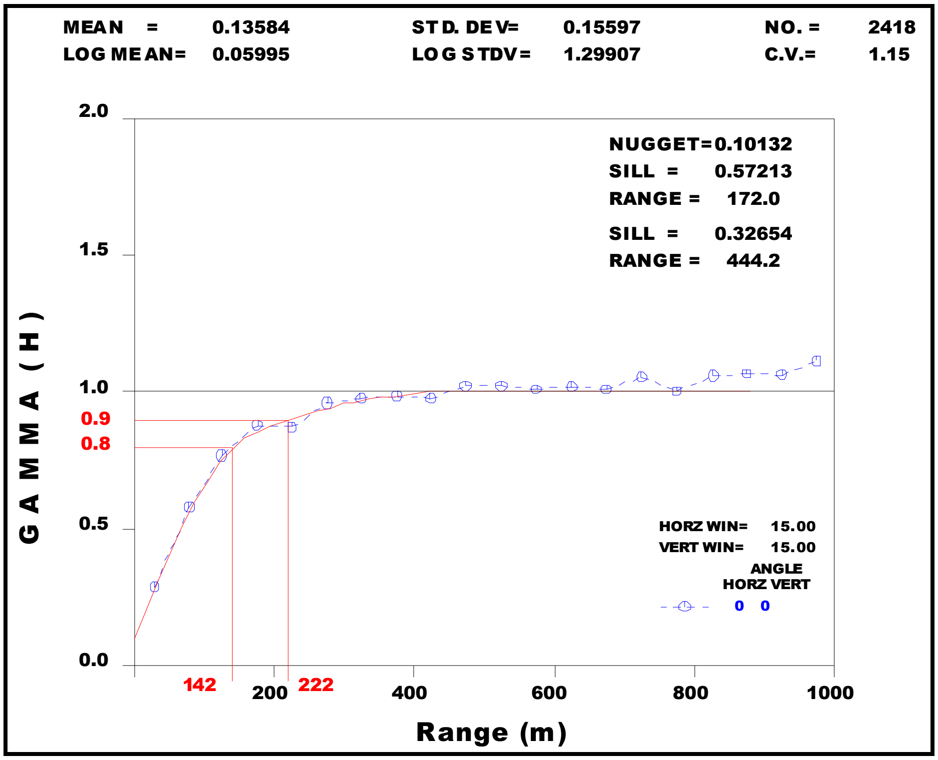


Figure 14.15 Mitchell Zone Cu Grade Correlogram

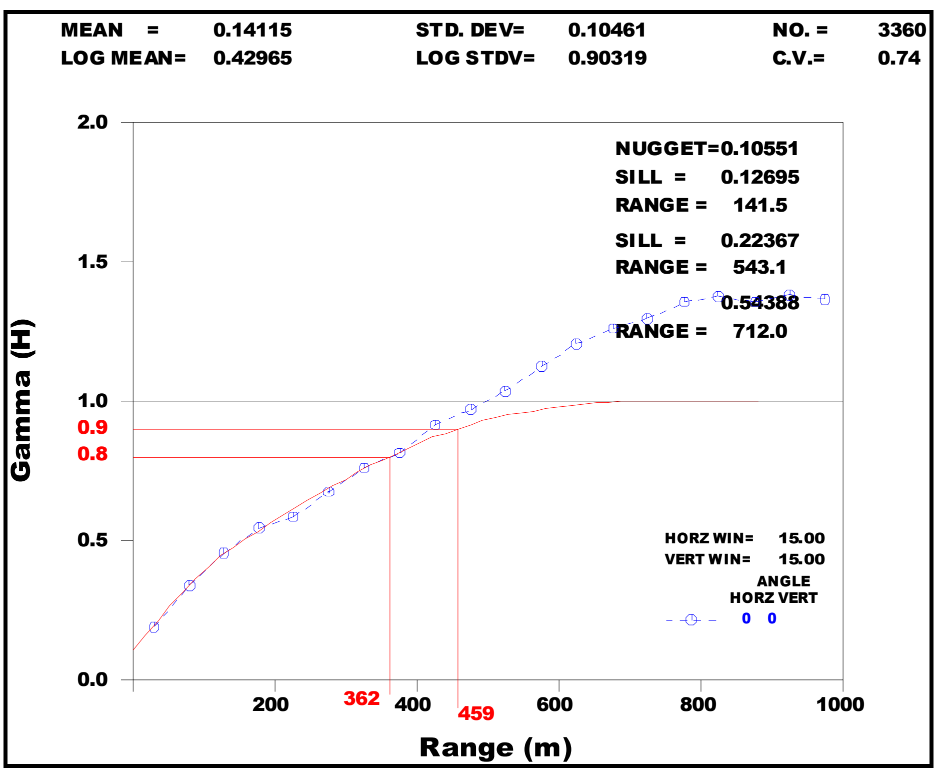


Figure 14.16 Iron Cap Zone Cu Grade Correlogram

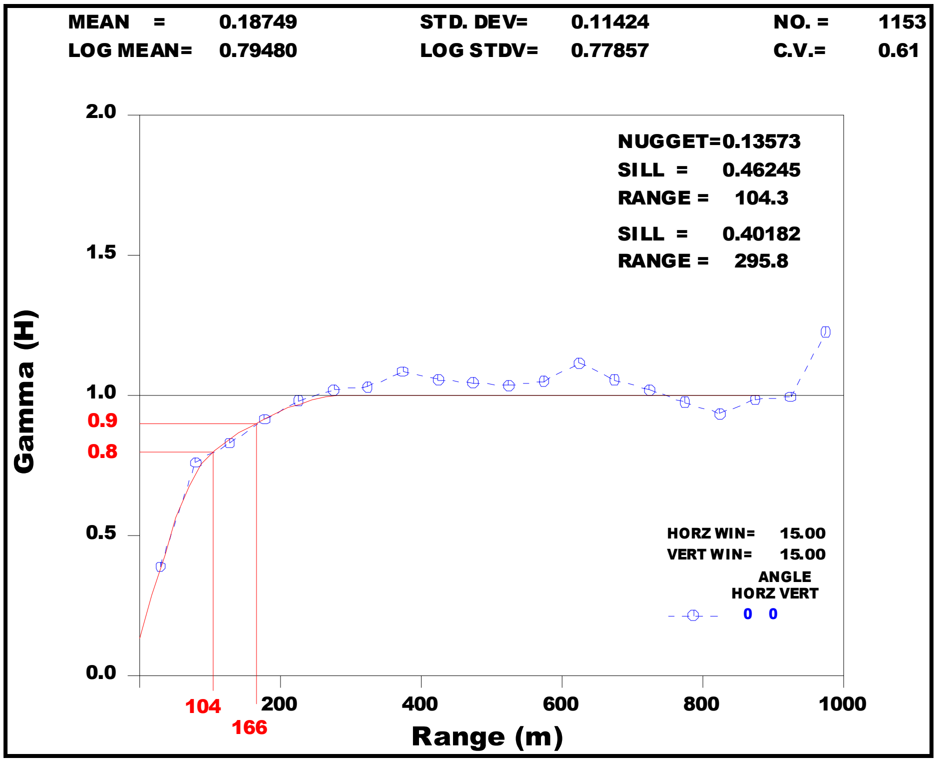


Figure 14.17 Kerr Zone 0.5 g/t AuEQ Correlogram

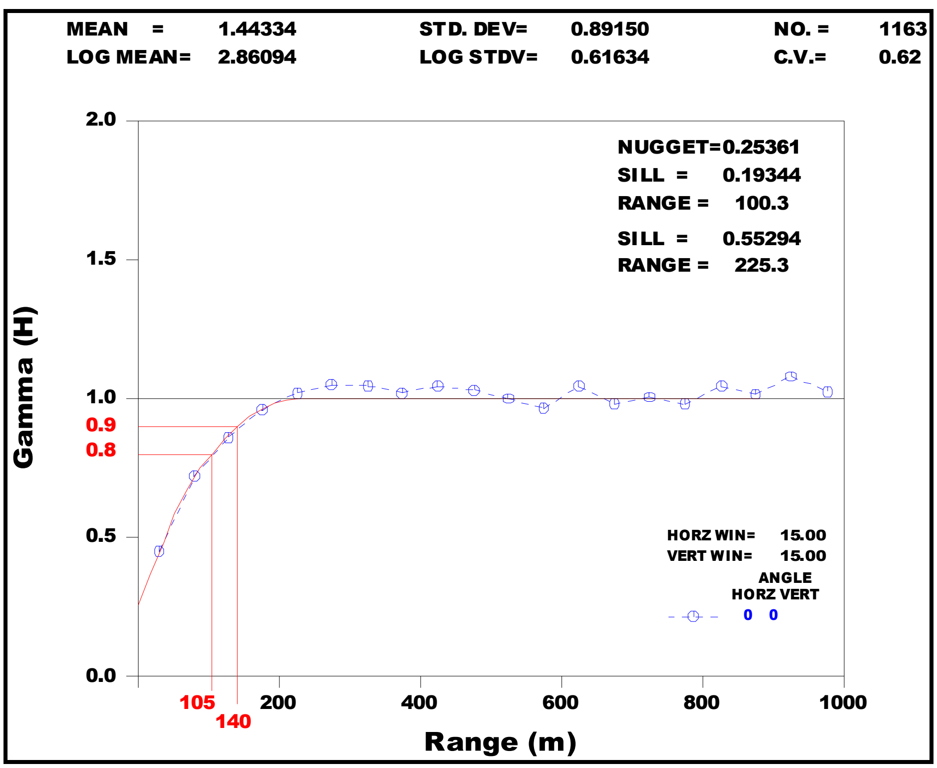


Figure 14.18 Sulphurets Zone 0.5 g/t AuEQ Correlogram

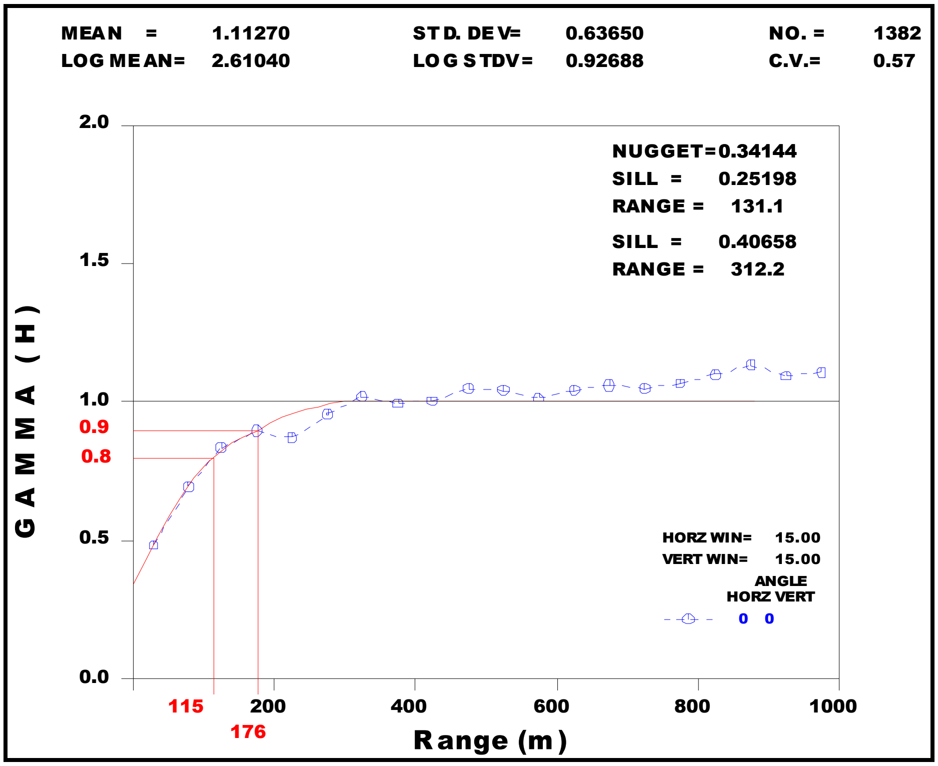


Figure 14.19 Mitchell Zone 0.5 g/t AuEQ Correlogram

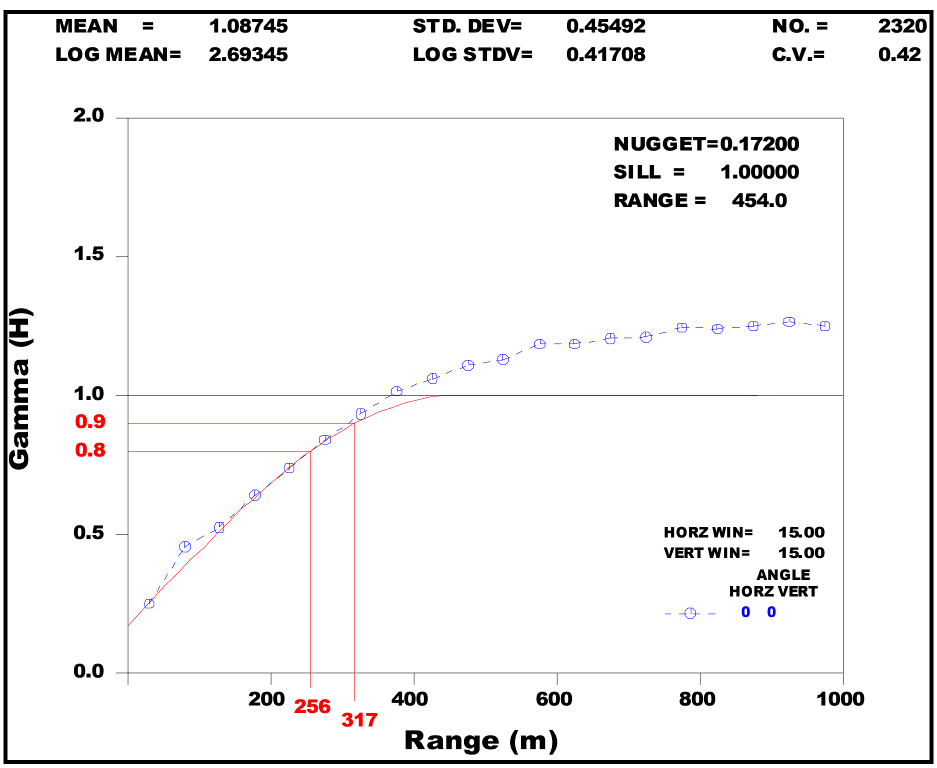
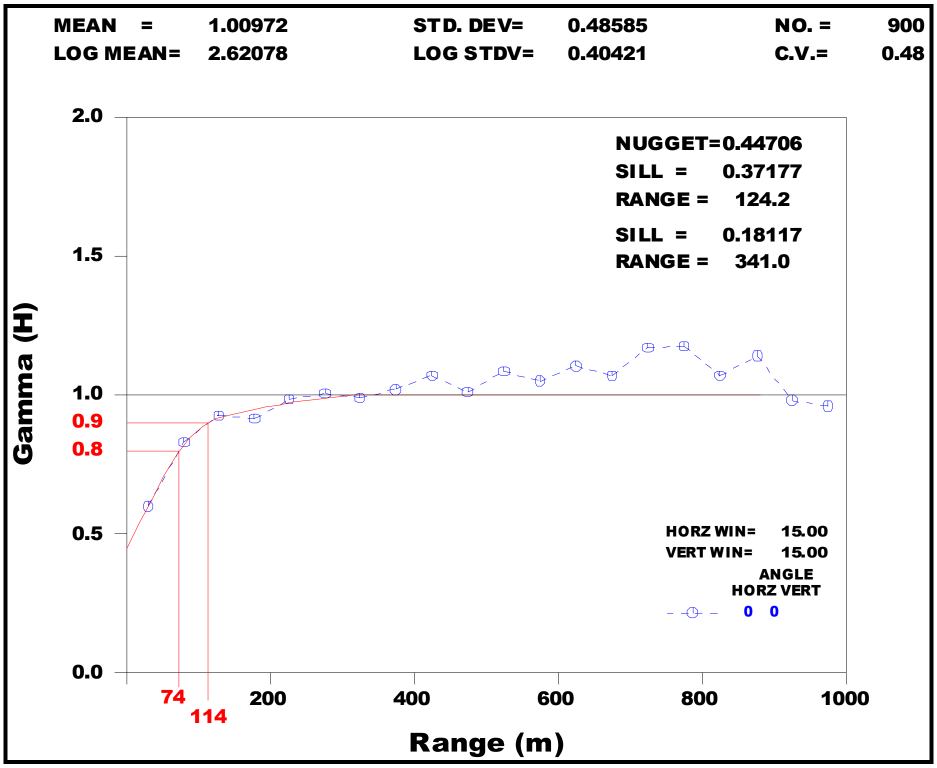


Figure 14.20 Iron Cap Zone 0.5 g/t AuEQ Correlogram



The correlograms shown in Figure 14.9 through Figure 14.20 were modelled as either single structure spherical or nested spherical models.

Total ranges for gold for each zone are as follows:

- Kerr: 159 m
- Sulphurets: 414 m
- Mitchell: 555 m
- Iron Cap: 279 m.

At 80% of the total sill, gold ranges were interpreted for each zone as follows:

- Kerr: 47 m
- Sulphurets: 167 m
- Mitchell: 325 m
- Iron Cap: 111 m.

Total ranges for copper for each zone are as follows:

- Kerr: 241 m

- Sulphurets: 444 m
- Mitchell: 712 m
- Iron Cap: 296 m.

At 80% of the total sill, copper ranges were interpreted for each zone as follows:

- Kerr: 118 m
- Sulphurets: 142 m
- Mitchell: 362 m
- Iron Cap: 104 m.

Total ranges for AuEQ grades for each zone are as follows:

- Kerr: 225 m
- Sulphurets: 312 m
- Mitchell: 454 m
- Iron Cap: 314 m.

At 80% of the total sill, AuEQ ranges were interpreted for each zone as follows:

- Kerr: 105 m
- Sulphurets: 115 m
- Mitchell: 256 m
- Iron Cap: 74 m.

14.7 GRADE ESTIMATION PARAMETERS

RMI constructed a 3D block model using MineSight®, a widely recognized commercial mine engineering software package. Table 14.1 summarizes various block parameters for this non-rotated model which uses NAD83 UTM coordinates.

Table 14.16 KSM Block Model Dimensions

	NAD83 Coordinates				
Parameter	**Minimum**	**Maximum**	**Block Size (m)**	**No. of Blocks**	**Areal Extent (m)**
Easting	420,500	425,900	25	216	5,400
Northing	6,257,800	6,269,000	25	448	11,200
Elevation	-210	2,145	15	157	2,355

Block gold, silver, copper, and molybdenum grades were estimated by two methods: inverse distance weighting, and nearest neighbour. Gold and copper resources summarized in this report are based on inverse distance squared or inverse distance cubed methods.

A multi-pass estimation strategy was used for gold, silver, copper, and molybdenum. The first and second estimation passes required two or more drill holes to estimate block grades while the final pass acted as "cleanup" run that filled un-estimated blocks by using a larger search ellipse and requiring fewer drill holes. The inverse distance estimation plans used strict block/composite matching.

Table 14.17 summarizes the key estimation parameters that were used to estimate block gold, silver, and copper grades for the Kerr Zone. No molybdenum assays were available for a significant portion of the Kerr drill hole data so no estimate was made for that metal. The estimate of Kerr block grades was constrained (controlled) by matching block and drill hole composite alteration codes (Table 14.13 shows the definition of alteration codes). Once a block was estimated, it was flagged so it would not be re-estimated in subsequent runs.

The number of composites used to estimate block gold and copper grades were stored along with the distance to the closest composite and the number of drill holes used to estimate the block.

Table 14.18 summarizes the parameters used to estimate block gold and silver grades for the Sulphurets Zone.

The estimate of Sulphurets gold and silver block grades was constrained (controlled) by matching block and drill hole AUZON composite codes (Table 14.14 shows the definition of AUZON codes). The last two interpolation runs shown in Table 14.18 estimated block grades above the Sulphurets Thrust Fault while all of the prior runs estimated blocks below the Sulphurets Thrust Fault. The number of composites and drill holes used to estimate block gold and silver grades were stored along with the distance to the closet composite.

Table 14.17 Kerr Zone Grade Estimation Parameters *

Estimation Pass	Alteration Codes	ID Power	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
			X	Y	Z	Min	Max	Max/Hole	ROTN	DIPN	DIPE
1	1, 2, 4, 5, 6	3	75	75	15	3	6	2	15	17	42
2	1, 2, 4, 5, 6	3	125	125	25	3	6	2	15	17	42
3	1, 2, 4, 5, 6	3	200	200	40	1	3	1	15	17	42
1	29	3	100	100	40	3	6	2	15	17	42
2	29	3	100	100	40	1	3	1	15	17	42

Table 14.18 Sulphurets Zone Au Estimation Parameters *

Estimation Pass	AUZON Codes	ID Power	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
			X	Y	Z	Min	Max	Max/Hole	ROTN	DIPN	DIPE
1	1, 2, 3, 4, 5	3	75	75	15	3	6	2	50	15	35
2	1, 2, 3, 4, 5	3	125	125	25	3	6	2	50	15	35
3	1, 2, 3, 4, 5	3	200	200	25	1	3	1	50	15	35
1	29	3	75	75	15	3	6	2	50	15	35
2	29	3	125	125	25	1	3	1	50	15	35
1	6, 7, 29	3	75	75	15	3	6	2	50	15	35
2	6, 7, 29	3	125	125	25	1	3	1	50	15	35

*** Notes:** ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule.

Table 14.19 summarizes the parameters used to estimate block gold and silver grades for the Mitchell Zone. Similar to Sulphurets, AUZON codes were used to constrain the estimate of block gold/silver grades for the Mitchell Zone. In addition to AUZON codes, block/composite position relative to the Mitchell Thrust Fault was also used to limit or constrain the estimate of block grades. The field "FLTAR" (fault block) shown in Table 14.19 shows two codes where five means above the Mitchell Thrust Fault and six means below the Mitchell Thrust Fault. Similar to the Kerr and Sulphurets estimation plan, the number of composites and drill holes used to estimate block grades were stored in addition to the distance of the closest composite.

Table 14.20 summarizes the key estimation parameters that were used to estimate block gold, silver, copper, and molybdenum grades using inverse distance squared methods for the Iron Cap Zone.

Table 14.21 summarizes the key estimation parameters that were used to estimate block copper and molybdenum grades using inverse distance methods for the Sulphurets Zone. The plan used CUZON and FLTAR codes to constrain the estimate of block grades. CUZON codes are described in Table 14.15. FLTAR codes 1 and 2 refer to blocks/drill holes below and above the Sulphurets Thrust Fault, respectively. Like the previously described estimation plans, the number of composites and drill holes were stored along with the distance to the closest composite.

Table 14.19 Mitchell Au/Ag Estimation Parameters *

Estimation Pass	AUZON	ID Power	FLTAR	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
				X	Y	Z	Min	Max	Max/Hole	ROTN	DIPN	DIPE
1	8	2	6	250	250	60	3	8	2	320	-55	0
2	8	2	6	375	375	90	1	3	1	320	-55	0
1	9	2	6	250	250	60	3	8	2	320	-55	0
2	9	2	6	500	500	120	1	3	1	320	-55	0
1	10,11,12	2	5	125	125	30	3	8	2	60	0	40
2	10,11,12	2	5	250	250	60	3	8	2	60	0	40
3	10,11,12	2	5	375	375	90	3	8	2	60	0	40
4	10,11,12	2	5	500	500	120	1	3	1	60	0	40
1	10,11,12	2	6	125	125	30	3	8	2	60	0	40
2	10,11,12	2	6	250	250	60	3	8	2	60	0	40
3	10,11,12	2	6	375	375	90	3	8	2	60	0	40
4	10,11,12	2	6	500	500	120	1	3	1	60	0	40
1	13,14	2	5	125	125	30	3	8	2	60	0	40
2	13,14	2	5	250	250	60	3	8	2	60	0	40
3	13,14	2	5	375	375	90	3	8	2	60	0	40
4	13,14	2	5	500	500	120	1	3	1	60	0	40
1	13,14	2	6	125	125	30	3	8	2	60	0	40
2	13,14	2	6	250	250	60	3	8	2	60	0	40
3	13,14	2	6	375	375	90	3	8	2	60	0	40
4	13,14	2	6	500	500	120	1	3	1	60	0	40
1	29	2	5	150	150	45	3	8	2	60	0	40
2	29	2	5	300	300	100	1	3	1	60	0	40
1	29	2	6	150	150	45	3	8	2	60	0	40
2	29	2	6	300	300	100	1	3	1	60	0	40

Table 14.20 Iron Cap Grade Estimation Parameters *

Estimation Pass	AUZON Codes	Inverse Distance Power	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
			X	Y	Z	Min	Max	Max/Hole	ROTN	DIPN	DIPE
1	6-12, 29	2	75	75	25	3	8	2	45	0	45
2	6-12, 30	2	150	150	50	3	8	2	45	0	45
3	6-12, 31	2	150	150	50	1	3	1	45	0	45

Table 14.21 Sulphurets Cu/Mo Estimation Parameters *

Estimation Pass	CUZON	ID Power	FLTAR	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
				X	Y	Z	Min	Max	Max/hole	ROTN	DIPN	DIPE
1	1, 2, 3, 4, 5	3	2	75	75	15	3	6	2	50	15	35
2	1, 2, 3, 4, 5	3	2	125	125	25	3	6	2	50	15	35
3	1, 2, 3, 4, 5	3	2	200	200	25	1	3	1	50	15	35
1	29	3	2	75	75	15	3	6	2	50	15	35
2	29	3	2	125	125	25	1	3	1	50	15	35
1	6,7	3	1	75	75	15	3	6	2	50	15	35
2	6,7	3	1	175	175	25	1	3	1	50	15	35
1	29	3	1	75	75	15	3	6	2	50	15	35
2	29	3	1	125	125	25	1	3	1	50	15	35

*** Notes:** ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new EW axis
LRL = "Left-hand-right hand-left hand" rotation rule.

Table 14.22 summarizes the key estimation parameters that were used to estimate block copper grades using inverse distance methods for the Mitchell Zone. The plan used CUZON and FLTAR codes to constrain the estimate of block grades. CUZON codes are described in Table 14.15. FLTAR codes 5 and 6 refer to blocks/drill holes above and below the Mitchell Thrust Fault, respectively. Like the previously described estimation plans, the number of composites and drill holes were stored along with the distance to the closest composite.

Table 14.23 summarizes the key estimation parameters that were used to estimate block molybdenum grades using inverse distance squared methods for the Mitchell Zone. The estimate of block molybdenum grades were constrained by a 3D molybdenum grade shell wireframe that was constructed using a 50 ppm cut-off grade. Blocks located inside and outside of that wireframe could only be estimated by drill hole composites located inside or outside of the wireframe, respectively.

Table 14.22 Mitchell Cu Estimation Parameters *

Estimation Pass	CUZON	ID Power	FLTAR	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
				X	Y	Z	Min	Max	Max/Hole	ROTN	DIPN	DIPE
1	10, 11, 12	2	5	125	125	30	3	8	2	60	0	40
2	10, 11, 12	2	5	250	250	60	3	8	2	60	0	40
3	10, 11, 12	2	5	500	500	120	1	3	1	60	0	40
1	13	2	5	250	250	60	3	8	2	60	0	40
2	13	2	5	500	500	120	1	3	1	60	0	65
1	29	2	5	150	150	45	3	8	2	60	0	65
2	29	2	5	150	150	45	1	3	1	60	0	65
1	10, 11, 12	2	6	125	125	30	3	8	2	60	0	65
2	10, 11, 12	2	6	250	250	60	3	8	2	60	0	40
3	10, 11, 12	2	6	500	500	120	1	3	1	60	0	40
1	13	2	6	250	250	60	3	8	2	60	0	40
2	13	2	6	500	500	120	1	3	1	60	0	40
1	8	2	6	300	300	75	1	6	2	320	-55	0
1	9	2	6	300	300	75	3	8	2	320	-55	0
2	9	2	6	300	300	75	1	6	2	320	-55	0
1	29	2	6	150	150	45	3	8	2	45	60	0
2	29	2	6	150	150	45	1	3	1	45	60	0

Table 14.23 Mitchell Mo Grade Estimation Parameters

Estimation Pass	Inverse Distance Power	Ellipse Search Ranges (m)			Number of Composites Used			Search Ellipse Rotations (LRL)		
		X	Y	Z	Min	Max	Max/Hole	ROTN	DIPN	DIPE
1	2	300	300	300	1	3	1	20	0	45
2	2	250	250	60	3	8	2	20	0	45

*** Notes:** ROTN = Rotation about Z axis - new north axis
DIPN = Rotation about X axis - dip of new north axis
DIPE = Rotation about Y axis - dip of new east-west axis
LRL = "Left-hand-right hand-left hand" rotation rule.

14.8 Grade Model Verification

Estimated block grades were verified by visual and statistical methods. RMI visually compared estimated block grades (gold, silver, copper, and molybdenum) with drill hole composite grades. In RMI's opinion there is a reasonable comparison between the drill hole composite grades and the estimated block grades. Figure 14.21 and Figure 14.22 are east-west cross sections through the Kerr block model drawn at northing coordinate 6,259,600. These figures show estimated block/composite gold grades (Figure 14.21) and block/composite copper grades (Figure 14.22). Figure 14.23 and Figure 14.24 are block model level maps drawn at the 1,200 m elevation through the Kerr model showing estimated block/composite gold and copper grades, respectively.

Figure 14.25 and Figure 14.26 are northwest-southeast cross sections through the Sulphurets block model drawn at Section 23. These figures show estimated block/composite gold grades (Figure 14.25) and block/composite copper grades (Figure 14.26). Figure 14.27 and Figure 14.28 are block model level maps drawn at the 1,275 m elevation through the Sulphurets model showing estimated block/composite gold and copper grades, respectively. Figure 14.29 and Figure 14.30 are northeast-southwest cross sections through the Mitchell block model drawn at Section 11. These figures show estimated block/composite gold grades (Figure 14.29) and block/composite copper grades (Figure 14.30).

Figure 14.31 and Figure 14.32 are block model level maps drawn at the 660 m elevation through the Mitchell model showing estimated block/composite gold and copper grades, respectively. Figure 14.33 and Figure 14.34 are northwest-southeast cross sections through the Iron Cap block model drawn at Section 50,700. These figures show estimated block/composite gold grades (Figure 14.33) and block/composite copper grades (Figure 14.34). Figure 14.35 and Figure 14.36 are block model level maps drawn at the 1,395 m elevation through the Iron Cap model showing estimated block/composite gold and copper grades, respectively.

The heavy dashed black line shown on the block model cross sections and level plans shown in Figure 14.21 through Figure 14.36 represents a conceptual pit generated by RMI using gold and copper prices of $1, 500/oz and $3.50/lb, respectively.

Figure 14.21 Kerr Zone Au Block Model Section 6,259,600 North

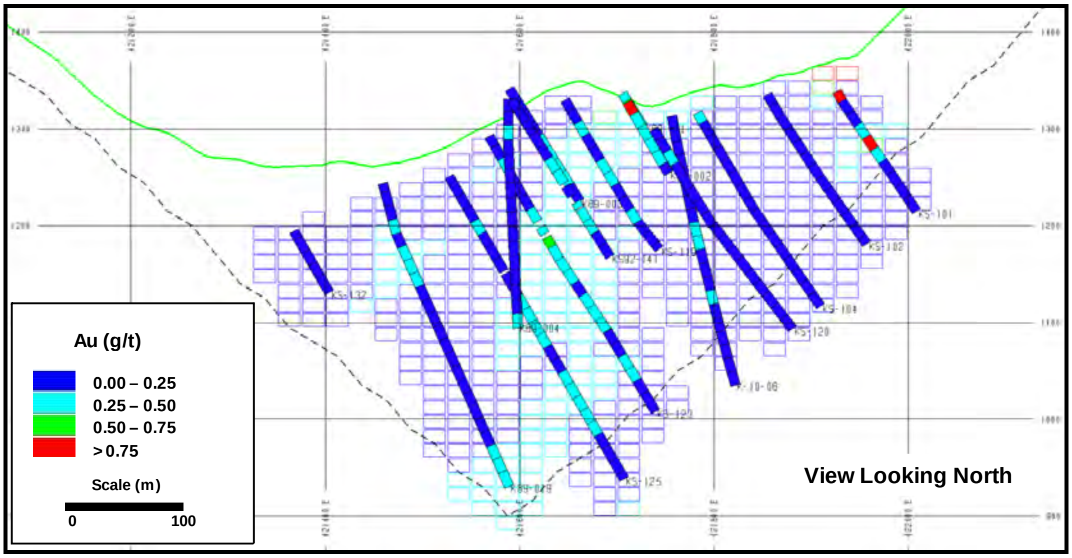


Figure 14.22 Kerr Zone Cu Block Model Section 6,259,600 North

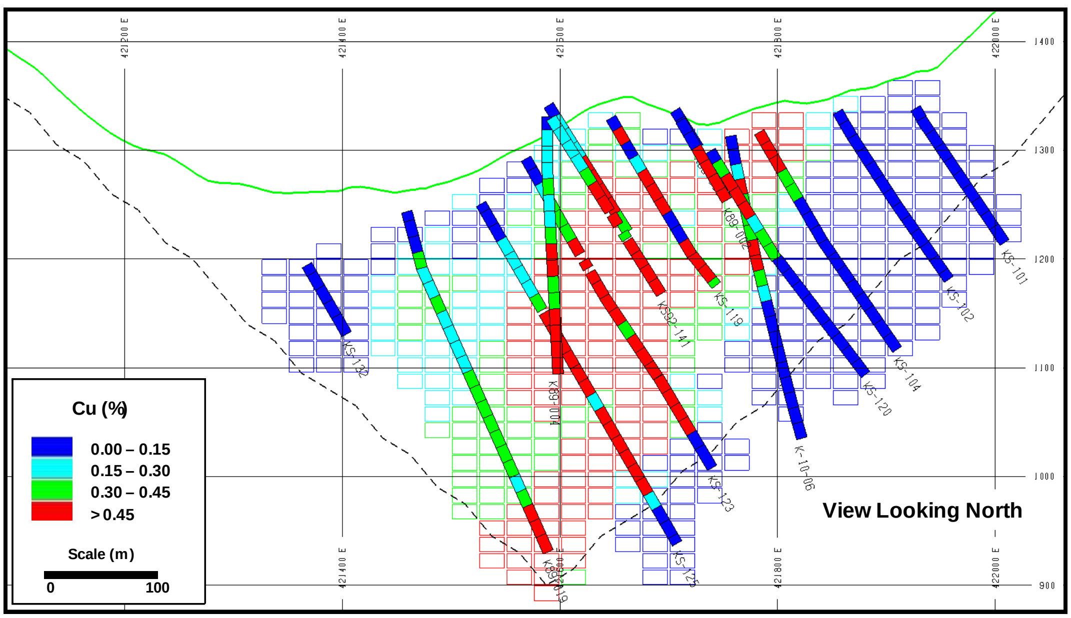


Figure 14.23 Kerr Zone Au Block Model – 1200 Level

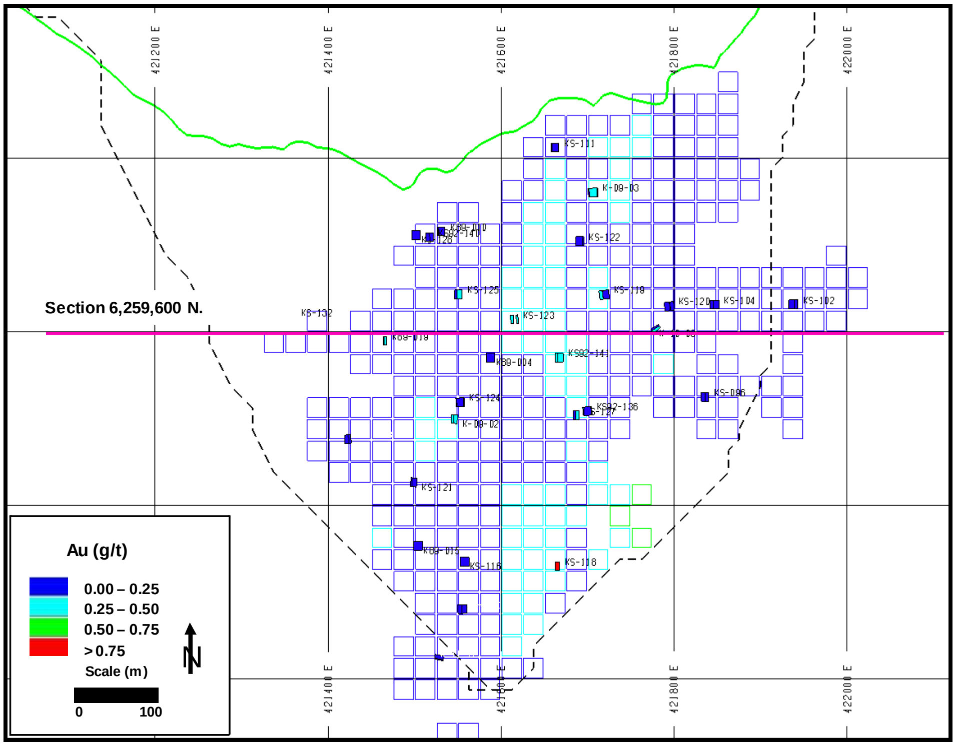


Figure 14.24 Kerr Zone Cu Block Model – 1200 Level

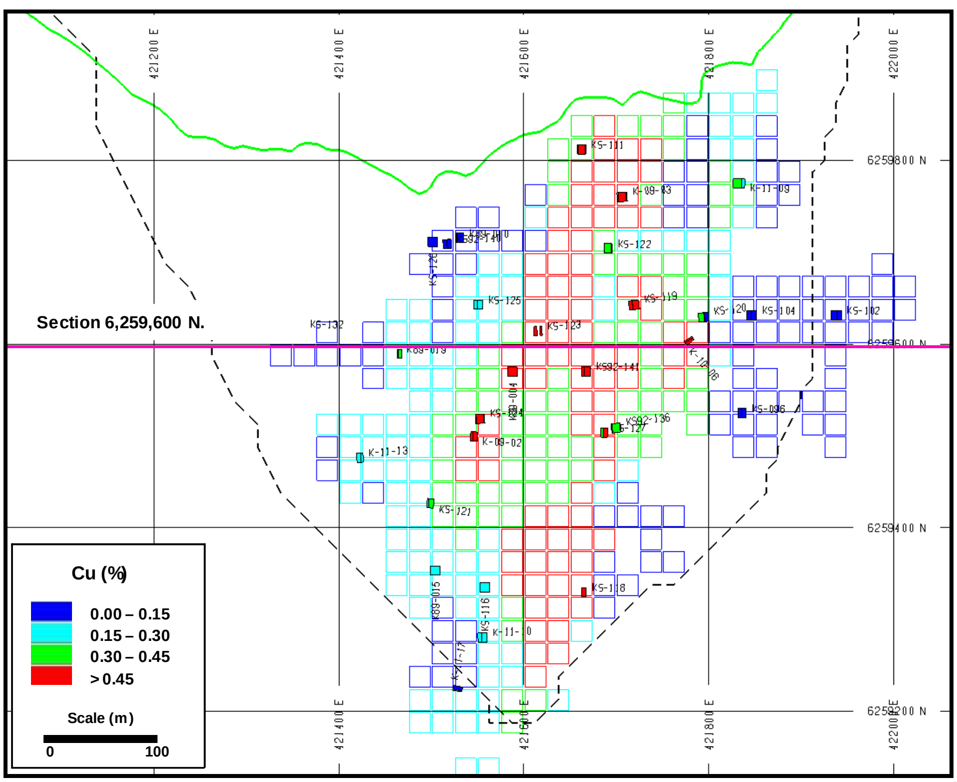


Figure 14.25 Sulphurets Zone Au Block Model Cross Section 23

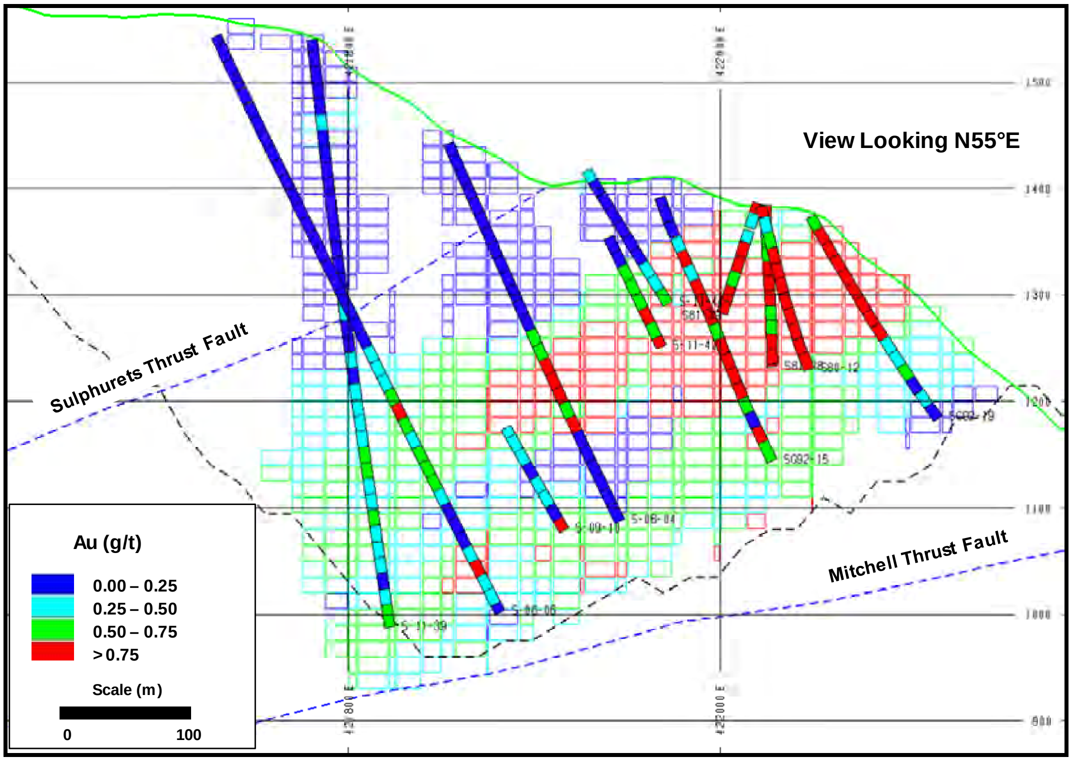


Figure 14.26 Sulphurets Zone Cu Block Model Cross Section 23

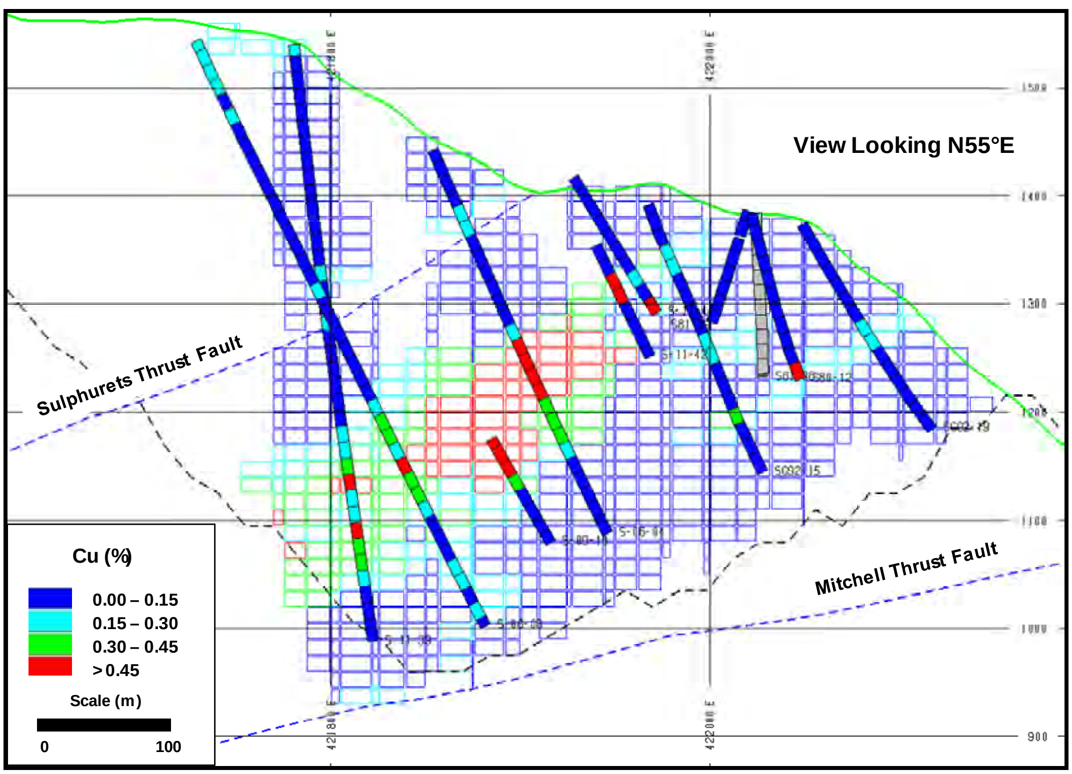


Figure 14.27 Sulphurets Zone Au Block Model – 1275 Level

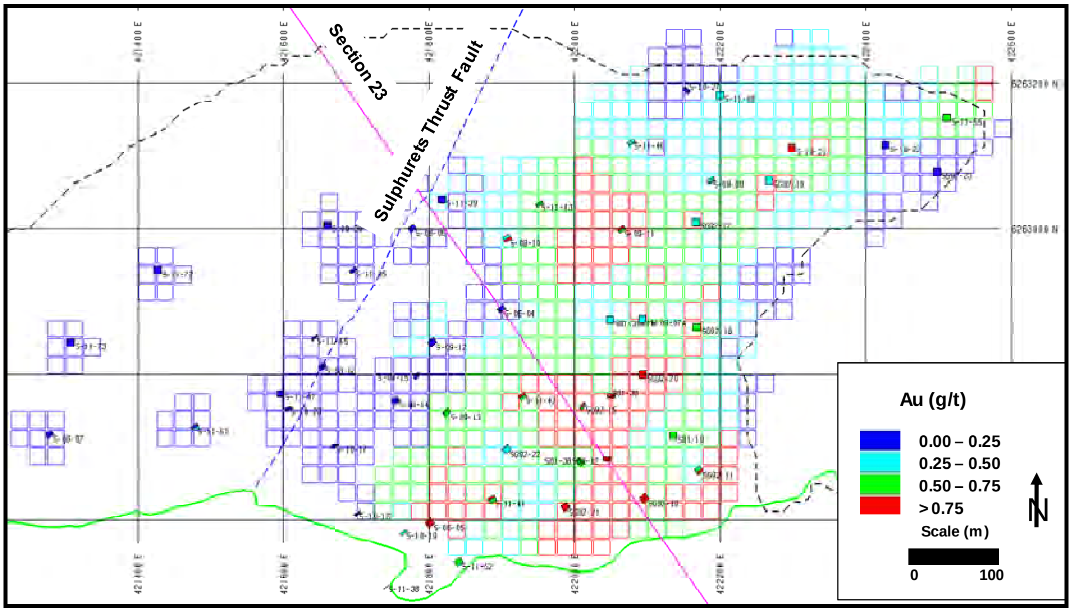


Figure 14.28 Sulphurets Zone Cu Block Model – 1275 Level

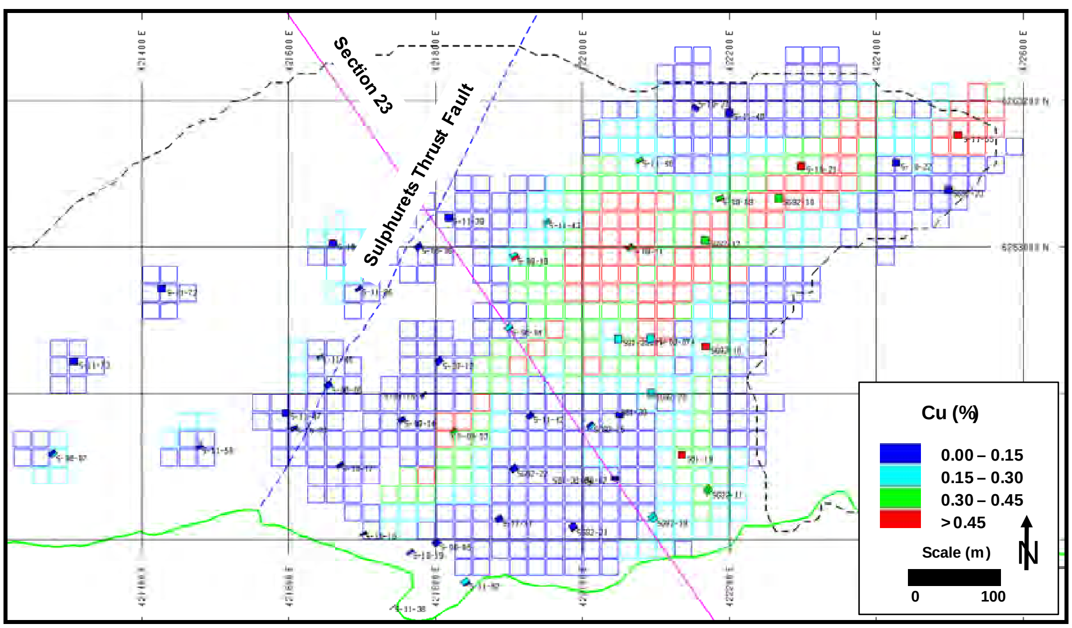


Figure 14.29 Mitchell Zone Au Block Model Cross Section 11

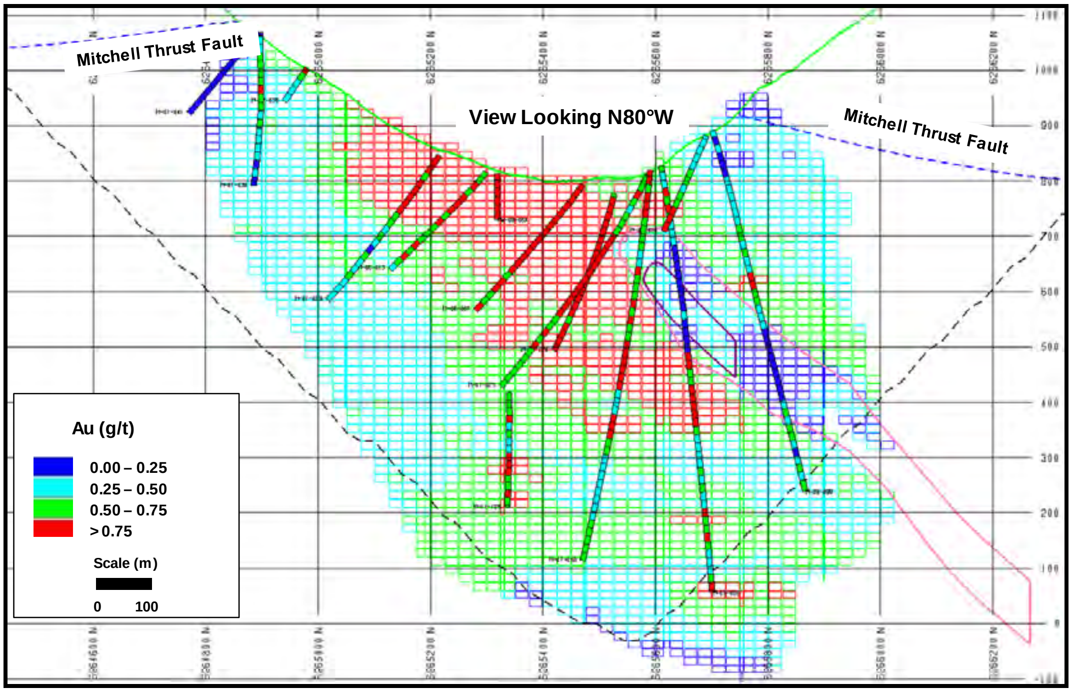


Figure 14.30 Mitchell Zone Cu Block Model Cross Section 11

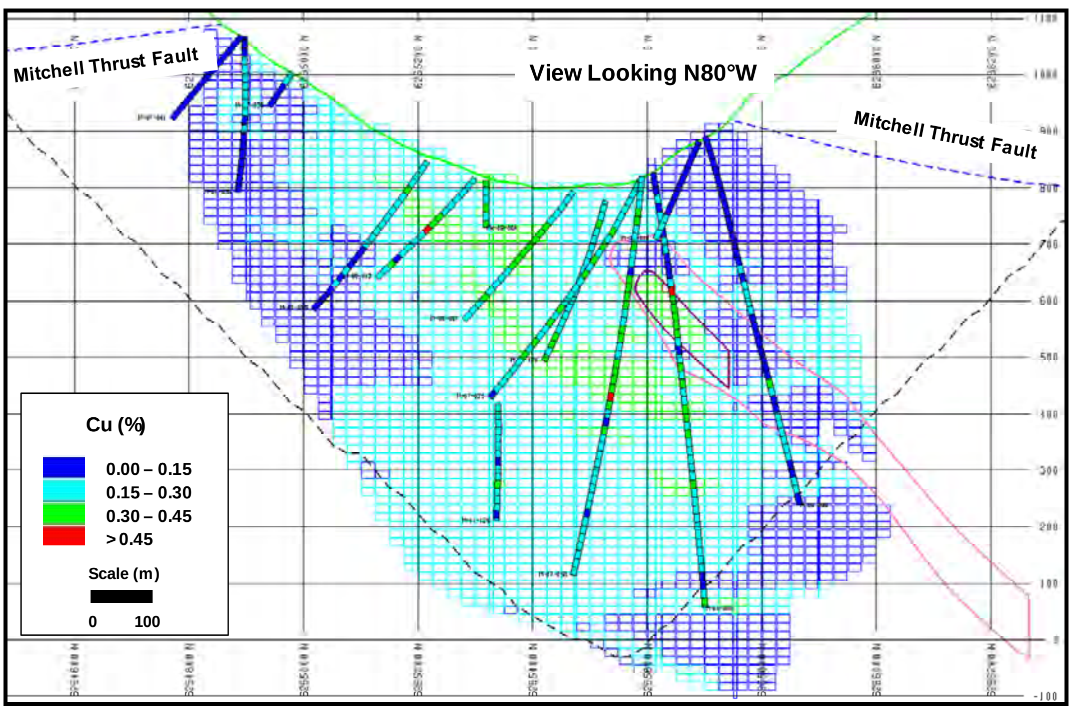


Figure 14.31 Mitchell Zone Au Block Model – 660 Level

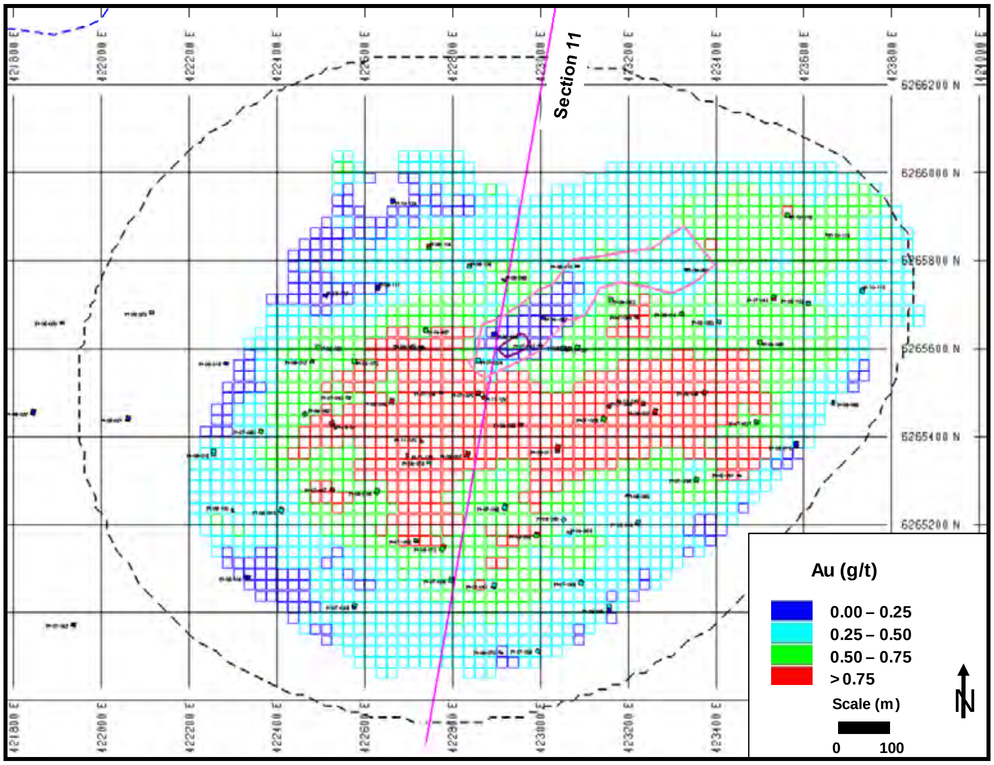


Figure 14.32 Mitchell Zone Cu Block Model – 660 Level

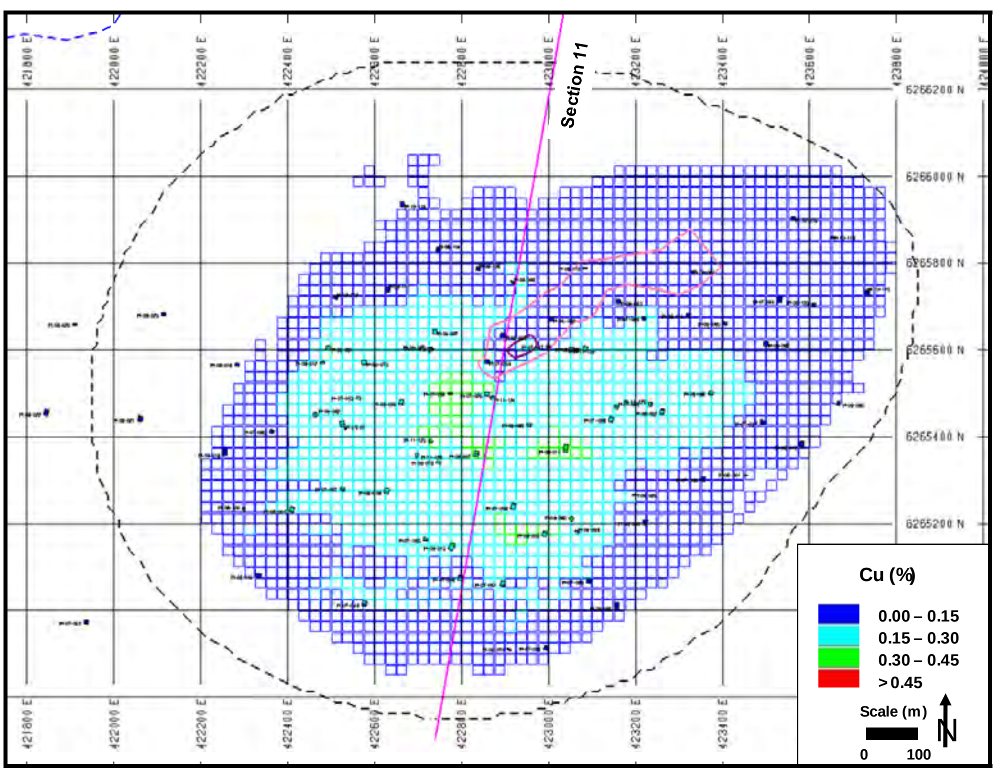


Figure 14.33 Iron Cap Zone Au Block Model – Section 50,700

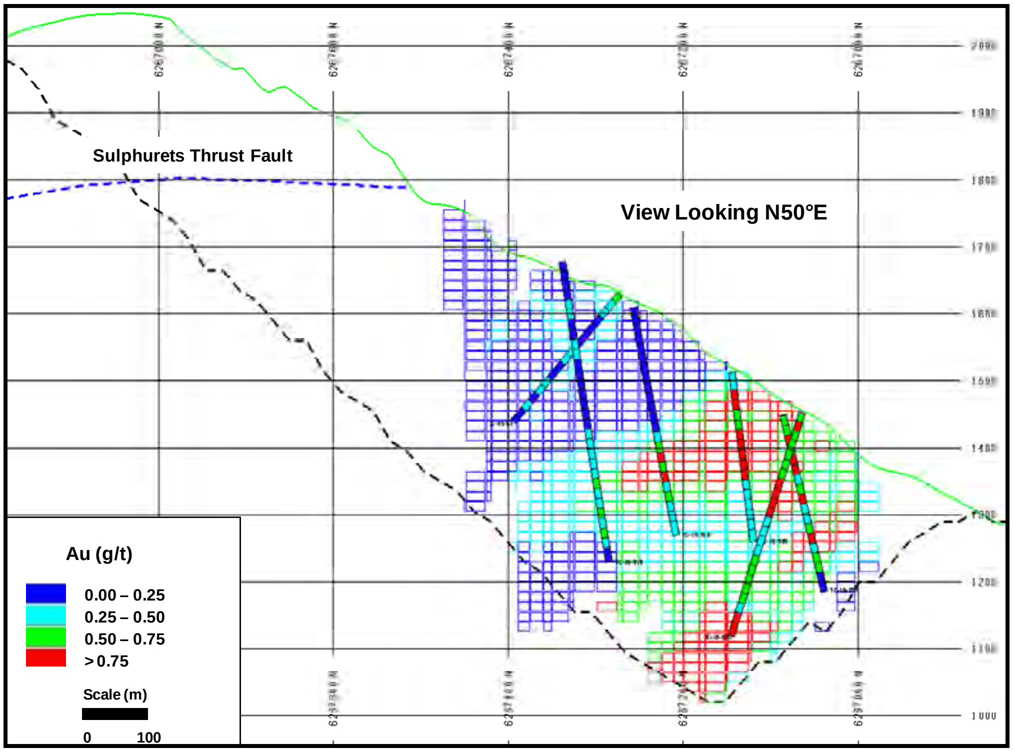


Figure 14.34 Iron Cap Zone Cu Block Model – Section 50,700

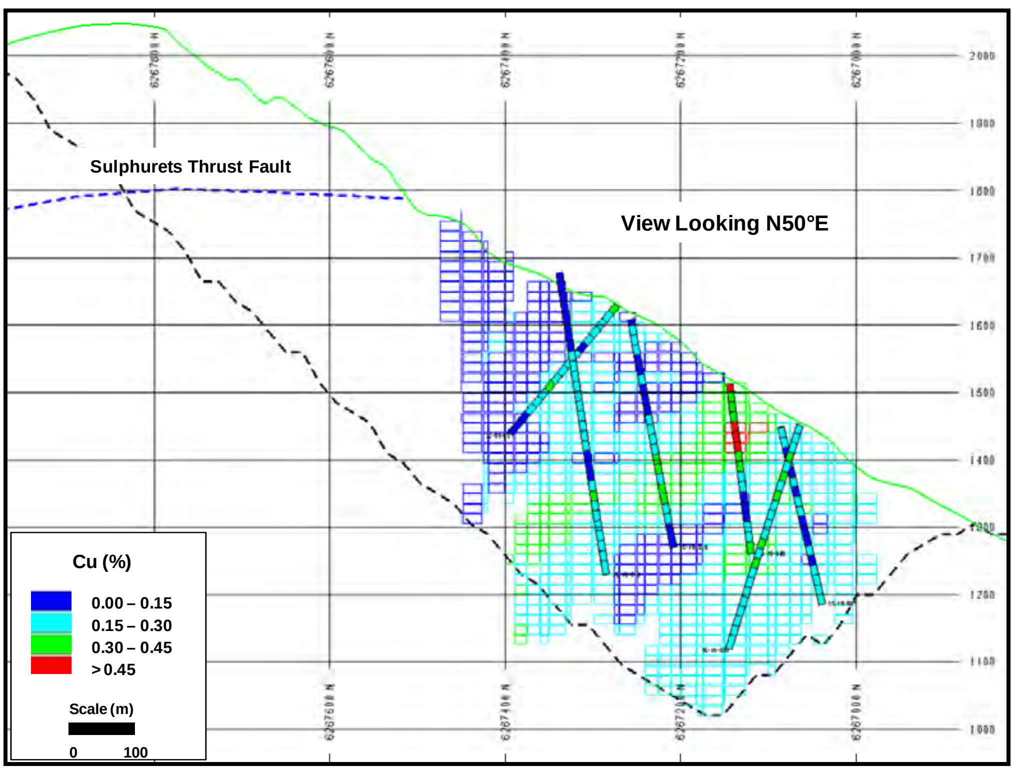


Figure 14.35 Iron Cap Zone Au Block Model – 1395 Level

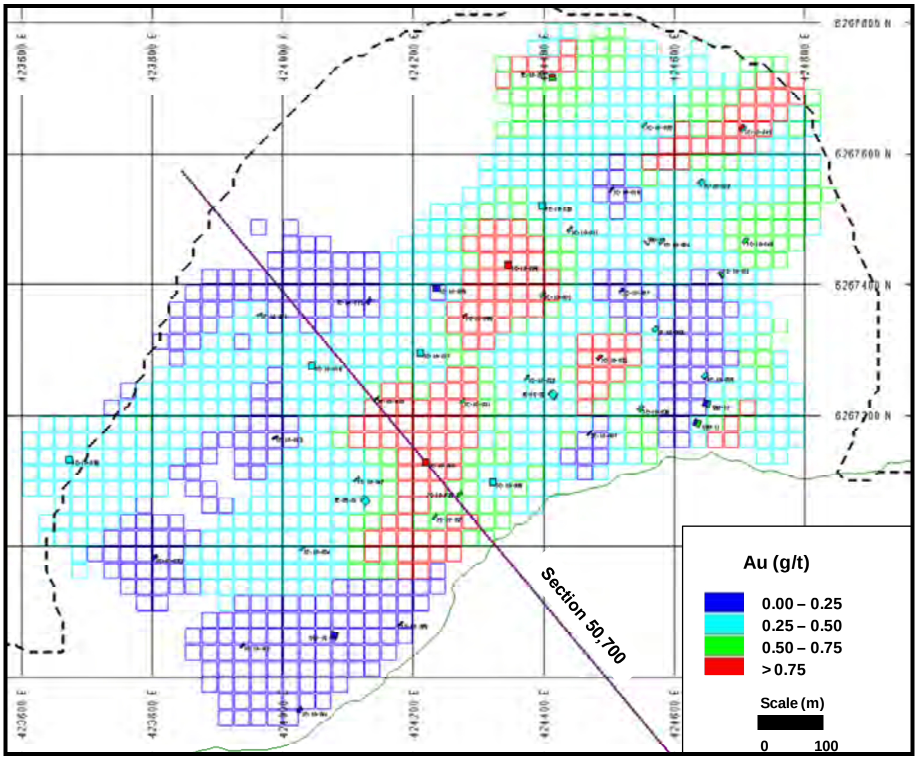
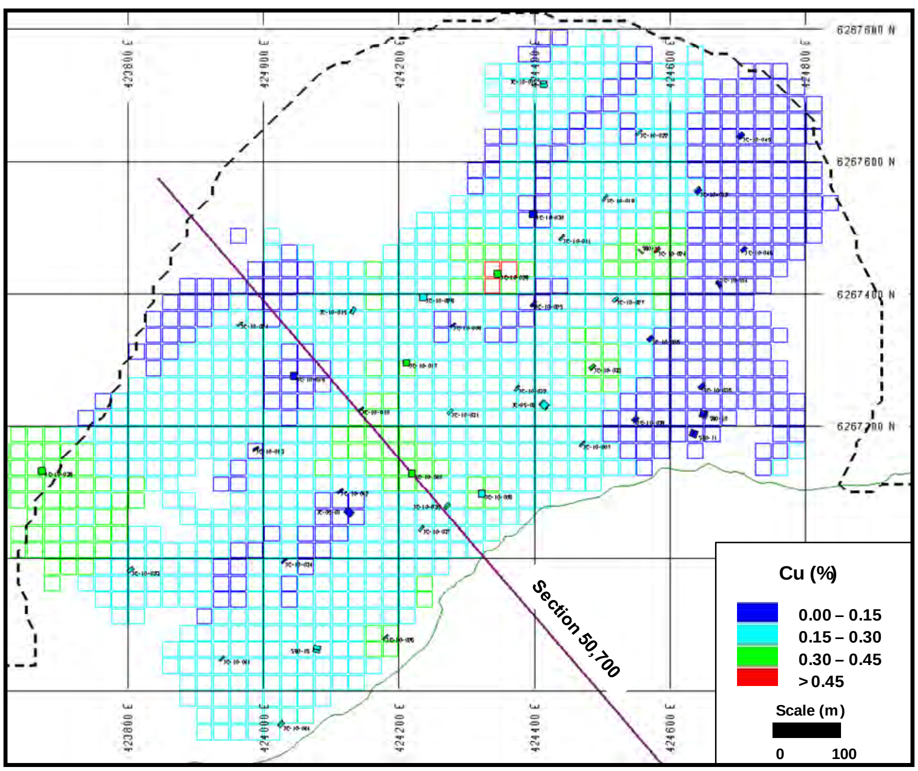


Figure 14.36 Iron Cap Zone Cu Block Model – 1395 Level



RMI generated nearest neighbour models for gold, copper, silver, and molybdenum in order to check for potential global biases in the estimated block grades. Table 14.24 compares mean nearest neighbour (NN) and inverse distance weighted (IDW) grades at a zero cut-off grade for Kerr, Sulphurets, Mitchell, and Iron Cap zones by resource category.

The results shown in Table 14.24 show that the IDW models compare very well with the nearest neighbour grades for the Measured + Indicated (MI) category (only the Mitchell Zone has Measured Resources). There are wider differences in mean grades for Inferred material, which is based on less drilling, hence lower confidence levels in those 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 gold and copper grades (AUIDW and CUIDW) with nearest neighbour gold and copper (AUNN and CUNN) estimates by block model columns (eastings), rows (northings), and levels (elevation). Gold and copper swath plots by elevation are shown in Figure 14.37 through Figure 14.40 for the Kerr, Sulphurets, Mitchell, and Iron Cap zones, respectively. These plots were drawn for Measured (Mitchell only) and Indicated Resources. The number of blocks by elevation is shown by the heavy black line and the units are read from the Y-axis on the right side of the plots.

In RMI's opinion, the swath plots shown in Figure 14.37 through Figure 14.40 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 Kerr, Sulphurets, Mitchell, and Iron Cap models are globally unbiased and represent reasonable estimates of insitu block grades.

Table 14.24 Grade Model Bias Checks

Kerr Zone						
	Indicated			Inferred		
Metal	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.2366	0.2401	-1.5%	0.1622	0.1597	1.6%
Copper (%)	0.4371	0.4411	-0.9%	0.1448	0.1435	0.9%
Silver (g/t)	1.2934	1.2949	-0.1%	1.0414	1.0198	2.1%
Molybdenum (ppm)	n/a	n/a	n/a	n/a	n/a	n/a
Sulphurets Zone						
	Indicated			Inferred		
Metal	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.5562	0.5583	-0.4%	0.3198	0.3182	0.5%
Copper (%)	0.1985	0.1982	0.2%	0.0936	0.0928	0.9%
Silver (g/t)	0.9258	0.9315	-0.6%	1.2817	1.2796	0.2%
Molybdenum (ppm)	53.2	52.9	0.6%	21.4	21.0	1.9%
Mitchell Zone						
	Measured+Indicated			Inferred		
Metal	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.5778	0.5806	-0.5%	0.3877	0.3801	2.0%
Copper (%)	0.1609	0.1606	0.2%	0.1246	0.1216	2.5%
Silver (g/t)	3.0758	3.1265	-1.6%	3.1082	3.0823	0.8%
Molybdenum (ppm)	59.4	60.0	-1.0%	52.9	56.1	-5.7%
Iron Cap Zone						
	Indicated			Inferred		
Metal	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.4027	0.4041	-0.3%	0.3118	0.3195	-2.4%
Copper (%)	0.1874	0.1874	0.0%	0.1673	0.1669	0.2%
Silver (g/t)	4.9669	4.8563	2.3%	3.2451	3.2124	1.0%
Molybdenum (ppm)	43.5	43.4	0.2%	49.4	50.4	-2.0%

Figure 14.37 Kerr Zone Au-Cu Swath Plots by Elevation

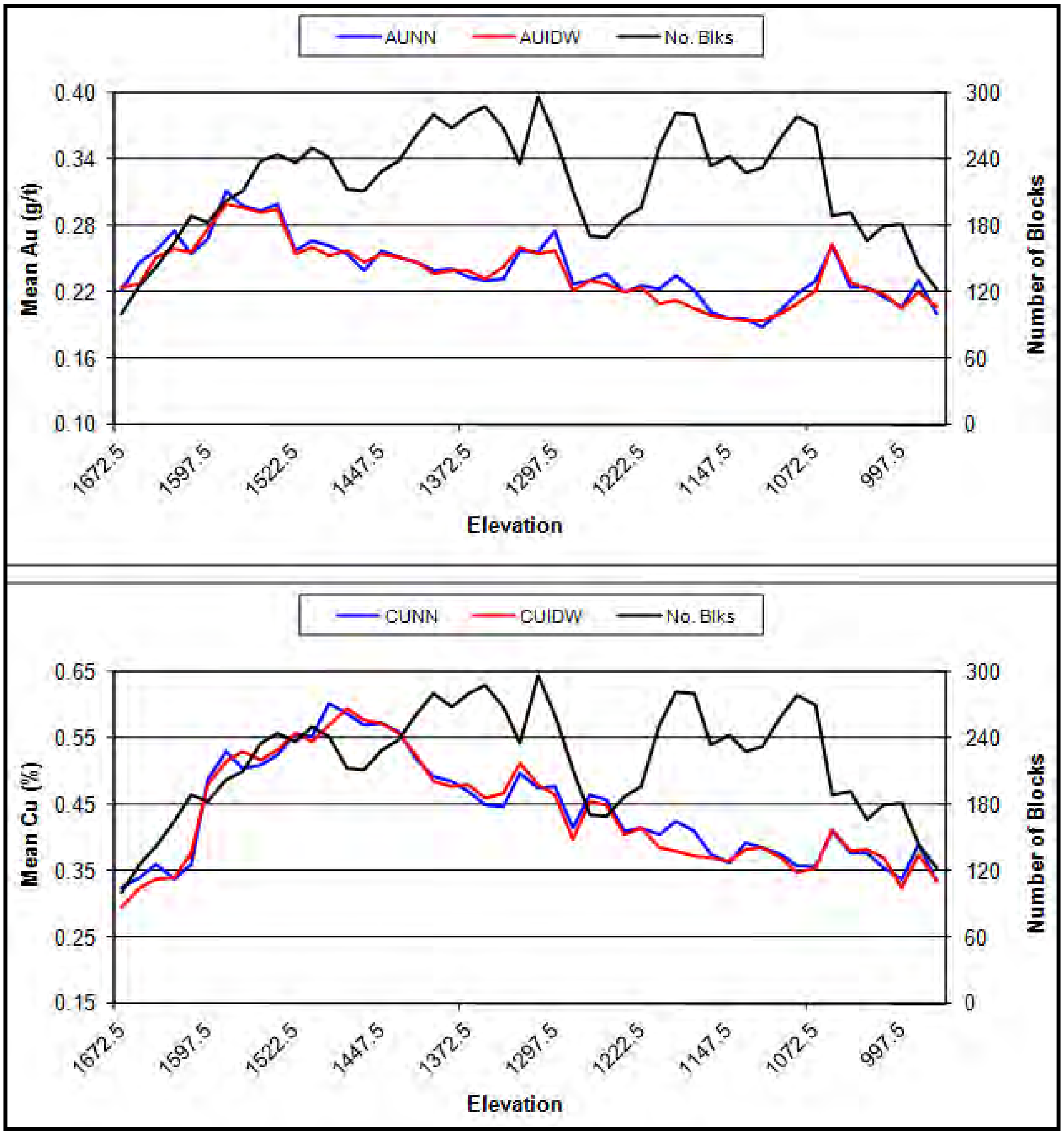


Figure 14.38 Sulphurets Zone Au-Cu Swath Plots by Elevation

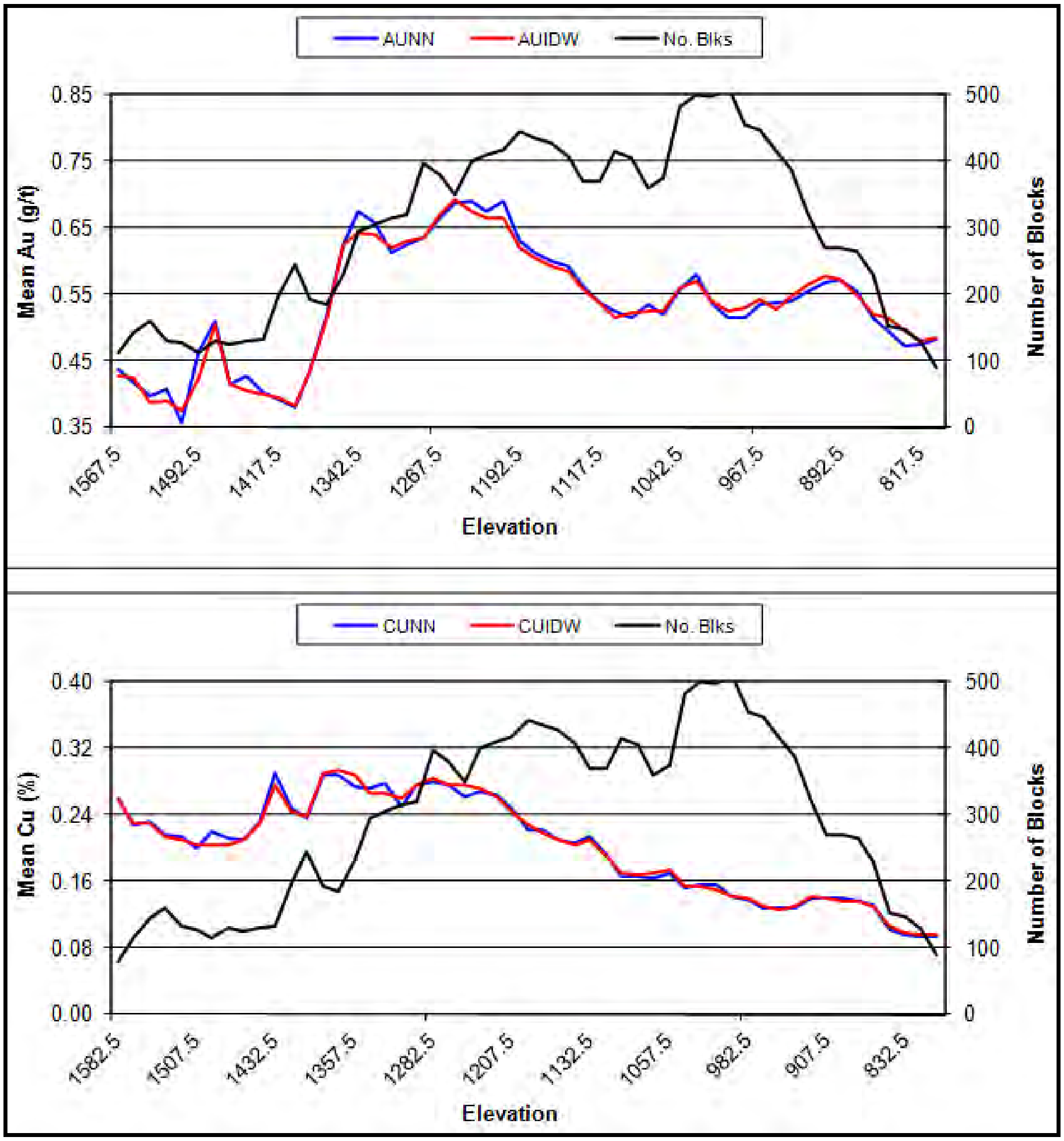


Figure 14.39 Mitchell Zone Au-Cu Swath Plots by Elevation

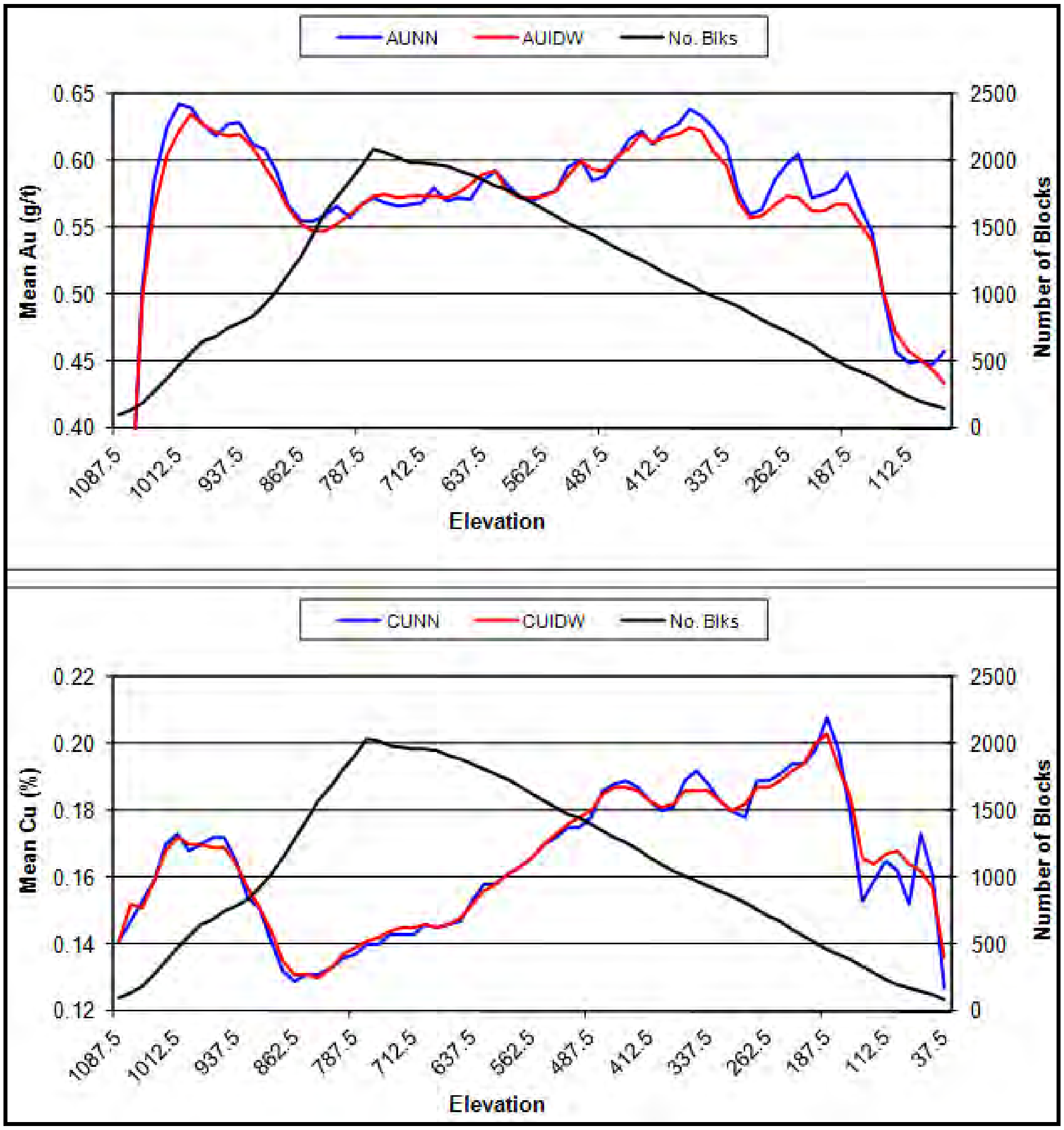
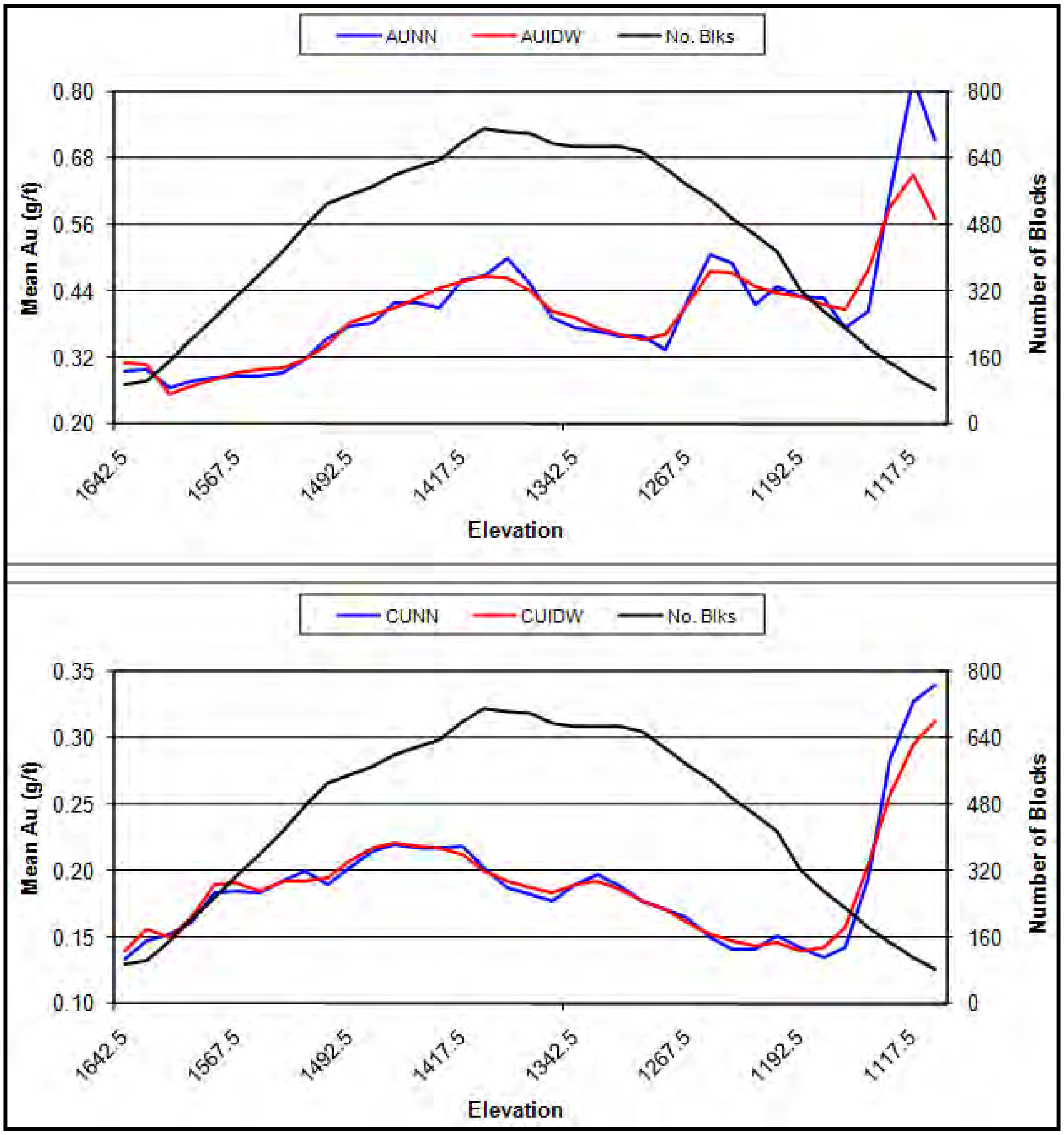


Figure 14.40 Iron Cap Zone Au-Cu Swath Plots by Elevation



14.9 RESOURCE CLASSIFICATION

RMI classified Kerr, Sulphurets, Mitchell, and Iron Cap estimated block grades into Measured (Mitchell only), Indicated, and Inferred Mineral Resources using a combination of distance to data, a required number of drill holes, and manually constructed shapes that represent "mineralized continuity".

To define mineralized continuity, RMI created probabilistic (indicator) AuEQ models for each mineralized zone using a 0.5 g/t AuEQ cut-off. Blocks with an estimated probability in excess of 50% of being above a 0.50 g/t AuEQ cut-off were used as a guide in drawing mid-bench polygons that defined mineralized continuity. The indicator probability model required that at least three drill holes were used to estimate block probabilities using a 150 m spherical search strategy.

Blocks for all four mineralized zones were initially coded with the mineralized continuity polygons and were considered to be Indicated Resources (code = 2). A default code of 5 was assigned to all other blocks. Then criteria such as distance to the closest drill hole and a minimum number of drill holes used to estimate the block grade were tested to see if the block was to remain as an Indicated Resource. If the criteria were not met, the Indicated blocks were re-assigned to Inferred (code = 3). Table 14.25 summarizes the criteria that were used to establish Indicated Resources.

Table 14.25 Indicated Resource Criteria

Mineralized Zone	Block Location	Minimum No. Holes	Distance to Closest Composite (m)
Kerr	Inside mineralized continuity shape	≥2	≤75
Sulphurets	Inside mineralized continuity shape	≥2	≤75
Mitchell	Inside mineralized continuity shape & below Mitchell Thrust Fault	≥2	≤125
Iron Cap	Inside mineralized continuity shape	≥2	≤75

Measured Mineral Resources (code = 1) were only assigned to the Mitchell Zone if:

1. the blocks were located inside of the mineralized continuity shape, and
2. they were estimated by two or more holes with the closest being within 50 m or one hole within 17 m of the block.

Inferred Mineral Resources were assigned to any unclassified blocks (i.e. code = 5) if the distance to drilling data and the minimum number of holes used to estimate block grades were met. Table 14.26 summarizes the criteria used to establish Inferred Resources.

Table 14.26 Inferred Resource Criteria

Mineralized Zone	Block Location	Minimum No. Holes	Distance to Closest Composite (m)
Kerr	Outside mineralized continuity shape	≥ 2	≤ 50
	Inside/outside mineralized continuity shape	≥ 1	≤ 25
Sulphurets	Above Sulphurets Thrust Fault	≥ 2	≤ 37.5
	Above Sulphurets Thrust Fault	≥ 1	≤ 25
	Below Sulphurets Thrust Fault, inside mineralized continuity shape	≥ 1	≤ 50
	Below Sulphurets Thrust Fault, outside mineralized continuity shape	≥ 2	≤ 50
	Below Sulphurets Thrust Fault, outside mineralized continuity shape	≥ 1	≤ 25
Mitchell	Above Mitchell Thrust Fault, inside mineralized continuity shape	≥ 1	≤ 75
	Above Mitchell Thrust Fault, outside mineralized continuity shape	≥ 1	≤ 50
	Below Mitchell Thrust Fault, inside mineralized continuity shape	≥ 2	≤ 175
	Below Mitchell Thrust Fault, outside mineralized continuity shape	≥ 2	≤ 75
	Below Mitchell Thrust Fault, outside mineralized continuity shape	≥ 1	≤ 50
Iron Cap	Inside/outside mineralized continuity shape	≥ 2	≤ 125
	Inside/outside mineralized continuity shape	≥ 1	≤ 75

14.10 SUMMARY OF MINERAL RESOURCES

Mineral Resources were tabulated for the Kerr, Sulphurets, Mitchell, and Iron Cap Zones using a AuEQ cut-off grade. This equivalent grade was calculated based on assumed metal prices and recoveries. A gold price of US$650/oz and a copper price of US$2.00/lb were used to calculate the AuEQ grade. Gold and copper recoveries of 70% and 85%, respectively, were also used to calculate gold equivalency using the following expression:

AuEQ = Au (g/t) + (Cu (%) *
(((Cu price/453.5924)/Au price/31.1035)) * (Cu recovery/Au recovery))*10000

The metal prices and recoveries are the same as those used in past KSM AuEQ calculations; they were selected to enable direct comparisons with previous estimates. RMI notes that some apparent discrepancies in the calculation of contained metal may occur due to the rounding of tonnes and grades.

Mineral Resources are summarized in Table 14.27 at a AuEQ cut-off grade of 0.50 g/t, which has been selected for disclosing Mineral Resources. This cut-off grade is above a "break-even" cut-off grade given today's metal prices and was used for direct comparisons with previous KSM resource estimates. Mineral Resources for the Kerr, Sulphurets, Mitchell, and Iron Cap zones are tabulated in Table 14.28 to Table 14.31, respectively, at a number of AuEQ cut-off grades. Note that the KSM resources shown in Table 14.27 to Table 14.30 are inclusive of Mineral Reserves that were disclosed in 2011 (Wardrop, 2011). No reserves have ever been declared for Iron Cap.

Table 14.27 2011 KSM Mineral Resources at 0.5 g/t AuEq Cut-off Grade

Zone	Tonnes (000)	Au (g/t)	Au (000 oz)	Cu (%)	Cu (Mlb)	Ag (g/t)	Ag (000 oz)	Mo (ppm)	Mo (Mlb)
Measured Resources									
Mitchell	724,000	0.65	15,130	0.18	2,872	3.2	74,487	56	89.4
Indicated Resources									
Kerr	270,400	0.24	2,086	0.46	2,741	1.1	9,563	n/a	n/a
Sulphurets	370,900	0.59	7,036	0.21	1,717	0.8	9,540	49	40.1
Mitchell	1,052,900	0.58	19,634	0.16	3,713	3.1	104,940	59	136.9
Iron Cap	361,700	0.44	5,117	0.21	1,674	5.4	62,796	47	37.5
Total	**2,055,900**	**0.51**	**33,873**	**0.22**	**9,845**	**2.8**	**186,838**	**54**	**214.5**
Measured Plus Indicated Resources									
Kerr	270,400	0.24	2,086	0.46	2,741	1.1	9,563	n/a	n/a
Sulphurets	370,900	0.59	7,036	0.21	1,717	0.8	9,540	49	40.1
Mitchell	1,776,900	0.61	34,764	0.17	6,585	3.1	179,426	58	226.3
Iron Cap	361,700	0.44	5,117	0.21	1,674	5.4	62,796	47	37.5
Total	**2,779,900**	**0.55**	**49,003**	**0.21**	**12,717**	**2.92**	**261,325**	**55**	**303.8**
Inferred Resources									
Kerr	85,000	0.24	656	0.28	525	0.9	2,460	n/a	n/a
Sulphurets	177,100	0.50	2,847	0.15	585	1.2	6,833	30	11.7
Mitchell	567,800	0.44	8,032	0.14	1,752	3.4	62,068	51	63.8
Iron Cap	297,300	0.36	3,441	0.20	1,310	3.9	37,278	60	39.3
Total	**1,127,200**	**0.41**	**14,976**	**0.17**	**4,172**	**3.00**	**108,638**	**50**	**114.8**

Note:
Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.28 Kerr Zone Mineral Resources

AuEQ Cut-off (g/t)	Tonnes (000)	Au (g/t)	Au (000 oz)	Cu (%)	Cu (M lb)	Ag (g/t)	Ag (000 oz)	Mo (ppm)	Mo (M lb)
Indicated Resources									
0.25	288,600	0.24	2,227	0.44	2,799	1.1	10,207	n/a	n/a
0.30	286,600	0.24	2,211	0.44	2,779	1.1	10,136	n/a	n/a
0.35	284,300	0.24	2,194	0.45	2,820	1.1	10,054	n/a	n/a
0.40	280,900	0.24	2,167	0.45	2,786	1.1	9,934	n/a	n/a
0.45	276,600	0.24	2,134	0.46	2,804	1.1	9,782	n/a	n/a
0.50	**270,400**	**0.24**	**2,086**	**0.46**	**2,741**	**1.1**	**9,563**	**n/a**	**n/a**
0.55	263,500	0.25	2,118	0.47	2,730	1.1	9,319	n/a	n/a
0.60	256,400	0.25	2,061	0.48	2,713	1.2	9,892	n/a	n/a
0.65	247,500	0.25	1,989	0.49	2,673	1.2	9,549	n/a	n/a
0.70	237,900	0.25	1,912	0.50	2,622	1.2	9,178	n/a	n/a
0.75	228,400	0.26	1,909	0.51	2,567	1.2	8,812	n/a	n/a
Inferred Resources									
0.25	175,900	0.19	1,075	0.18	698	0.8	4,524	n/a	n/a
0.30	150,200	0.20	966	0.20	662	0.8	3,863	n/a	n/a
0.35	131,300	0.21	886	0.22	637	0.9	3,799	n/a	n/a
0.40	113,800	0.22	805	0.24	602	0.9	3,293	n/a	n/a
0.45	98,500	0.23	728	0.26	564	0.9	2,850	n/a	n/a
0.50	**85,000**	**0.24**	**656**	**0.28**	**525**	**0.9**	**2,460**	**n/a**	**n/a**
0.55	75,000	0.25	603	0.30	496	1.0	2,411	n/a	n/a
0.60	66,200	0.26	553	0.32	467	1.0	2,128	n/a	n/a
0.65	58,800	0.26	492	0.34	441	1.0	1,890	n/a	n/a
0.70	51,800	0.27	450	0.37	422	1.1	1,832	n/a	n/a
0.75	45,100	0.28	406	0.39	388	1.1	1,595	n/a	n/a

Note:

Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.29 Sulphurets Zone Mineral Resources

AuEQ Cut-off (g/t)	Tonnes (000)	Au (g/t)	Au (000 oz)	Cu (%)	Cu (M lb)	Ag (g/t)	Ag (000 oz)	Mo (ppm)	Mo (M lb)
Sulphurets Indicated Resources									
0.25	402,900	0.56	7,254	0.20	1,776	0.8	10,363	47	41.7
0.30	400,800	0.56	7,216	0.20	1,767	0.8	10,309	47	41.5
0.35	397,700	0.57	7,288	0.20	1,753	0.8	10,229	47	41.2
0.40	392,700	0.57	7,197	0.20	1,731	0.8	10,100	47	40.7
0.45	383,500	0.58	7,151	0.21	1,775	0.8	9,864	48	40.6
0.50	**370,900**	**0.59**	**7,036**	**0.21**	**1,717**	**0.8**	**9,540**	**49**	**40.1**
0.55	353,700	0.60	6,823	0.22	1,715	0.8	9,097	50	39.0
0.60	335,000	0.61	6,570	0.23	1,698	0.8	8,616	52	38.4
0.65	313,600	0.63	6,352	0.24	1,659	0.8	8,066	54	37.3
0.70	289,100	0.65	6,042	0.25	1,593	0.8	7,436	57	36.3
0.75	264,900	0.66	5,621	0.26	1,518	0.8	6,813	59	34.4
Sulphurets Inferred Resources									
0.25	292,700	0.39	3,670	0.11	710	1.2	11,293	23	14.8
0.30	271,700	0.40	3,494	0.12	719	1.2	10,482	24	14.4
0.35	246,800	0.43	3,412	0.13	707	1.3	10,315	25	13.6
0.40	226,200	0.44	3,200	0.13	648	1.3	9,454	26	13.0
0.45	202,600	0.47	3,061	0.14	625	1.3	8,468	28	12.5
0.50	**177,100**	**0.50**	**2,847**	**0.15**	**585**	**1.2**	**6,833**	**30**	**11.7**
0.55	154,100	0.53	2,626	0.16	543	1.2	5,945	32	10.9
0.60	134,900	0.55	2,385	0.17	505	1.2	5,205	34	10.1
0.65	115,900	0.58	2,161	0.18	460	1.2	4,472	37	9.5
0.70	98,800	0.61	1,938	0.19	414	1.2	3,812	40	8.7
0.75	83,800	0.64	1,724	0.21	388	1.2	3,233	44	8.1

Sulphurets Indicated Relative to Sulphurets Thrust Fault (STF)									
Location	**Tonnes (000)**	**Au (g/t)**	**Au (000 oz)**	**Cu (%)**	**Cu (M lb)**	**Ag (g/t)**	**Ag (000 oz)**	**Mo (ppm)**	**Mo (M lb)**
Above STF	16,800	0.30	162	0.22	81	1.6	864	38	1.4
Below STF	354,100	0.60	6,831	0.21	1,639	0.8	9,108	50	39.0
Total	**370,900**	**0.59**	**6,993**	**0.21**	**1,720**	**0.8**	**9,972**	**49**	**40.4**

Sulphurets Inferred Relative to Sulphurets Thrust Fault (STF)									
Location	**Tonnes (000)**	**Au (g/t)**	**Au (000 oz)**	**Cu (%)**	**Cu (M lb)**	**Ag (g/t)**	**Ag (000 oz)**	**Mo (ppm)**	**Mo (M lb)**
Above STF	15,200	0.27	132	0.19	64	1.3	635	28	0.9
Below STF	161,900	0.52	2,707	0.15	535	1.2	6,246	30	10.7
Total	**177,100**	**0.50**	**2,839**	**0.15**	**599**	**1.2**	**6,882**	**30**	**11.6**

Note:
Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.30 Mitchell Zone Mineral Resources

AuEQ Cut-off (g/t)	Tonnes (000)	Au (g/t)	Au (000 oz)	Cu (%)	Cu (M lb)	Ag (g/t)	Ag (000 oz)	Mo (ppm)	Mo (M lb)
Mitchell Measured Resources									
0.25	784,900	0.62	15,646	0.17	2,941	3.1	78,229	58	100.3
0.30	781,200	0.62	15,572	0.17	2,927	3.2	80,372	58	99.9
0.35	775,000	0.62	15,448	0.17	2,904	3.2	79,734	58	99.1
0.40	764,000	0.63	15,475	0.17	2,863	3.2	78,602	58	97.7
0.45	744,600	0.64	15,321	0.17	2,790	3.2	76,606	57	93.5
0.50	**724,000**	**0.65**	**15,130**	**0.18**	**2,872**	**3.2**	**74,487**	**56**	**89.4**
0.55	700,600	0.66	14,866	0.18	2,779	3.3	74,332	55	84.9
0.60	670,400	0.67	14,441	0.18	2,660	3.3	71,128	54	79.8
0.65	637,600	0.69	14,145	0.19	2,670	3.4	69,698	52	73.1
0.70	608,400	0.70	13,692	0.19	2,548	3.4	66,506	51	68.4
0.75	575,800	0.72	13,329	0.20	2,538	3.5	64,793	49	62.2
Mitchell Indicated Resources									
0.25	1,145,900	0.55	20,263	0.16	4,041	3.0	110,525	60	151.5
0.30	1,137,600	0.56	20,482	0.16	4,012	3.0	109,724	60	150.4
0.35	1,125,000	0.56	20,255	0.16	3,967	3.0	108,509	60	148.8
0.40	1,110,900	0.56	20,001	0.16	3,917	3.1	110,720	60	146.9
0.45	1,087,000	0.57	19,920	0.16	3,833	3.1	108,338	60	143.7
0.50	**1,052,900**	**0.58**	**19,634**	**0.16**	**3,713**	**3.1**	**104,940**	**59**	**136.9**
0.55	1,010,900	0.59	19,176	0.17	3,788	3.1	100,754	58	129.2
0.60	958,700	0.61	18,802	0.17	3,592	3.2	98,633	57	120.4
0.65	896,100	0.62	17,862	0.18	3,555	3.2	92,193	55	108.6
0.70	830,000	0.64	17,078	0.18	3,293	3.3	88,061	53	97.0
0.75	765,900	0.66	16,252	0.19	3,207	3.4	83,722	51	86.1
Mitchell Inferred Resources									
0.25	762,300	0.39	9,558	0.12	2,016	3.1	75,976	51	85.7
0.30	734,000	0.40	9,439	0.12	1,941	3.2	75,516	51	82.5
0.35	702,300	0.41	9,258	0.12	1,857	3.2	72,254	51	78.9
0.40	666,300	0.42	8,997	0.13	1,909	3.3	70,693	51	74.9
0.45	621,200	0.43	8,588	0.13	1,780	3.3	65,908	51	69.8
0.50	**567,800**	**0.44**	**8,032**	**0.14**	**1,752**	**3.4**	**62,068**	**51**	**63.8**
0.55	509,600	0.46	7,537	0.14	1,572	3.5	57,344	51	57.3
0.60	448,700	0.48	6,924	0.15	1,483	3.7	53,376	50	49.4
0.65	391,500	0.49	6,168	0.16	1,381	3.8	47,831	49	42.3
0.70	339,800	0.51	5,572	0.16	1,198	3.9	42,607	49	36.7
0.75	293,900	0.52	4,914	0.17	1,101	4.0	37,796	48	31.1

Note:
Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.31 Iron Cap Zone Mineral Resources

AuEQ Cut-off (g/t)	Tonnes (000)	Au (g/t)	Au (000 oz)	Cu (%)	Cu (M lb)	Ag (g/t)	Ag (000 oz)	Mo (ppm)	Mo (M lb)
Iron Cap Indicated Resources									
0.25	419,200	0.41	5,526	0.19	1,755	5.0	67,388	44	40.7
0.30	413,000	0.41	5,444	0.19	1,729	5.1	67,719	44	40.1
0.35	403,700	0.42	5,451	0.19	1,691	5.1	66,194	45	40
0.40	391,300	0.42	5,284	0.20	1,725	5.2	65,419	45	38.8
0.45	376,800	0.43	5,209	0.20	1,661	5.4	65,418	46	38.2
0.50	**361,700**	**0.44**	**5,117**	**0.21**	**1,674**	**5.4**	**62,796**	**47**	**37.5**
0.55	344,800	0.45	4,989	0.21	1,596	5.5	60,971	46	35
0.60	325,400	0.46	4,812	0.22	1,578	5.6	58,586	45	32.3
0.65	304,200	0.48	4,695	0.22	1,475	5.7	55,747	43	28.8
0.70	279,800	0.49	4,408	0.23	1,418	5.8	52,175	41	25.3
0.75	250,300	0.52	4,185	0.23	1,269	5.9	47,479	38	21
Iron Cap Inferred Resources									
0.25	373,700	0.33	3,965	0.17	1,400	3.4	40,850	51	42
0.30	365,900	0.33	3,882	0.18	1,452	3.5	41,174	52	41.9
0.35	353,300	0.34	3,862	0.18	1,402	3.6	40,892	53	41.3
0.40	339,100	0.34	3,707	0.18	1,345	3.6	39,248	55	41.1
0.45	318,800	0.35	3,587	0.19	1,335	3.8	38,949	57	40.1
0.50	**297,300**	**0.36**	**3,441**	**0.20**	**1,310**	**3.9**	**37,278**	**60**	**39.3**
0.55	273,800	0.37	3,257	0.20	1,207	3.9	34,331	63	38
0.60	244,800	0.39	3,069	0.21	1,133	4.0	31,482	64	34.5
0.65	224,400	0.41	2,958	0.22	1,088	4.0	28,858	62	30.7
0.70	195,300	0.43	2,700	0.22	947	4.0	25,116	60	25.8
0.75	169,600	0.46	2,508	0.23	860	4.0	21,811	60	22.4

Note:
Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. Inferred Mineral Resources have a high degree of uncertainty as to their existence, and great uncertainty as to their economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

14.11 CONCEPTUAL PIT RESULTS

The Mineral Resources summarized in Table 14.27 were tabulated as “global resources” using a AuEQ cut-off grade that is higher than a conceptual "break-even" cut-off grade using current metal prices. As a test to determine “reasonable expectation of economic viability”, RMI generated a number of conceptual pits for each mineralized zone using the floating cone algorithm. Measured, Indicated, and Inferred Mineral Resources were used in all cases. Five different metal prices and two different constant pit slope angles were used to generate a total of 10 conceptual pits. Mining and processing costs were kept constant for all 10 cases. Table 14.32 summarizes the key parameters that were used to generate the conceptual pits.

Table 14.32 Conceptual Pit Parameters

Conceptual Pit Number	Au Price (US$/oz)	Cu Price (US$/lb)	Ag Price (US$/oz)	Mo Price (US$/lb)	Au Rec. (%)	Cu Rec. (%)	Ag Rec. (%)	Mo Rec. (%)	Mining Cost (US$/t)	Processing Cost (US$/t)	Slope Angle (°)
1	1,000	3.00	20.00	15.00	72	84	69	32	1.75	7.00	45
2	1,250	3.25	25.00	17.50	72	84	69	32	1.75	7.00	45
3	1,500	3.50	30.00	20.00	72	84	69	32	1.75	7.00	45
4	1,750	3.75	35.00	22.50	72	84	69	32	1.75	7.00	45
5	2,000	4.00	40.00	25.00	72	84	69	32	1.75	7.00	45
6	1,000	3.00	20.00	15.00	72	84	69	32	1.75	7.00	40
7	1,250	3.25	25.00	17.50	72	84	69	32	1.75	7.00	40
8	1,500	3.50	30.00	20.00	72	84	69	32	1.75	7.00	40
9	1,750	3.75	35.00	22.50	72	84	69	32	1.75	7.00	40
10	2,000	4.00	40.00	25.00	72	84	69	32	1.75	7.00	40

Mineral Resources for all 10 conceptual pits are tabulated for the Kerr, Sulphurets, Mitchell, and Iron Cap zones using a 0.50 g/t Au equivalent cut-off grade in Table 14.33 to Table 14.36, respectively. For reference, the officially stated Mineral Resources for each zone are shown at the bottom of the tables and are highlighted in yellow. Rounding of tonnes and grade may result in contained metal in the conceptual pits to be greater than the global resource.

RMI notes that conceptual pits based upon a gold price of at least US$1,000/oz and a copper price of US$3.00/lb capture nearly all of the Indicated Mineral Resources and where applicable, Measured Resources, for each zone. Conceptual pits using those same metal prices captured 98% to 100% of the contained Measured and Indicated gold and copper metal relative to the global resource inventory. Those conceptual pits captured 83%, 89%, 63%, and 92% of the Inferred gold ounce resources for the Kerr, Sulphurets, Mitchell, and Iron Cap, respectively. Similar percentages of contained Inferred copper metal were captured by the conceptual pits. RMI notes that Mineral Resources, which are not Mineral Reserves, do not have demonstrated economic viability. Inferred Resources have a high degree of uncertainty as to their existence, and great uncertainty as to economic and legal feasibility. It cannot be assumed that all or any part of an Inferred Resource will ever be upgraded to a higher category.

Table 14.33 Kerr Conceptual Pit Results

Pit - Resource Class	Pit	Au Price	Cu Price	Tonnes (000)	Au (g/t)	Au (oz 000)	Cu (%)	Cu (M lb)	% of Au oz in Pit	% of Cu lb in Pit
Kerr Indicated Resource	**n/a**	**n/a**	**n/a**	**270,400**	**0.24**	**2,086**	**0.46**	**2,741**	**100%**	**100%**
Kerr Indicated inside pit	pit01	$1000 Au	$3.00/lb	260,923	0.25	2,097	0.47	2,703	101%	99%
Kerr Indicated inside pit	pit02	$1250 Au	$3.25/lb	265,359	0.25	2,133	0.47	2,749	102%	100%
Kerr Indicated inside pit	pit03	$1500 Au	$3.50/lb	265,972	0.25	2,138	0.46	2,697	102%	98%
Kerr Indicated inside pit	pit04	$1750 Au	$3.75/lb	266,263	0.24	2,055	0.46	2,699	98%	98%
Kerr Indicated inside pit	pit05	$2000 Au	$4.00/lb	266,980	0.24	2,060	0.46	2,707	99%	99%
Kerr Inferred Resource	**n/a**	**n/a**	**n/a**	**85,000**	**0.24**	**656**	**0.28**	**525**	**100%**	**100%**
Kerr Inferred inside pit	pit01	$1000 Au	$3.00/lb	66,491	0.24	513	0.27	396	78%	75%
Kerr Inferred inside pit	pit02	$1250 Au	$3.25/lb	69,403	0.25	558	0.28	428	85%	82%
Kerr Inferred inside pit	pit03	$1500 Au	$3.50/lb	70,862	0.24	547	0.28	437	83%	83%
Kerr Inferred inside pit	pit04	$1750 Au	$3.75/lb	71,159	0.24	549	0.28	439	84%	84%
Kerr Inferred inside pit	pit05	$2000 Au	$4.00/lb	72,334	0.24	558	0.28	446	85%	85%

Table 14.34 Sulphurets Conceptual Pit Results

Pit – Resource Class	Pit	Au Price	Cu Price	Tonnes (000)	Au (g/t)	Au (oz 000)	Cu (%)	Cu (M lb)	% of Au oz in Pit	% of Cu lb in Pit
Sulphurets Indicated Resource	**n/a**	**n/a**	**n/a**	**370,900**	**0.59**	**7,036**	**0.21**	**1,717**	**100%**	**100%**
Sulphurets Indicated inside pit	pit01	$1000 Au	$3.00/lb	366,514	0.59	6,952	0.21	1,696	99%	99%
Sulphurets Indicated inside pit	pit02	$1250 Au	$3.25/lb	368,592	0.59	6,992	0.21	1,706	99%	99%
Sulphurets Indicated inside pit	pit03	$1500 Au	$3.50/lb	369,838	0.59	7,015	0.21	1,712	100%	100%
Sulphurets Indicated inside pit	pit04	$1750 Au	$3.75/lb	370,020	0.59	7,019	0.21	1,713	100%	100%
Sulphurets Indicated inside pit	pit05	$2000 Au	$4.00/lb	370,228	0.59	7,023	0.21	1,714	100%	100%
Sulphurets Inferred Resource	**n/a**	**n/a**	**n/a**	**177,100**	**0.5**	**2,847**	**0.15**	**585**	**100%**	**100%**
Sulphurets Inferred inside pit	pit01	$1000 Au	$3.00/lb	142,595	0.5	2,292	0.14	440	81%	75%
Sulphurets Inferred inside pit	pit02	$1250 Au	$3.25/lb	151,503	0.5	2,435	0.14	467	86%	80%
Sulphurets Inferred inside pit	pit03	$1500 Au	$3.50/lb	157,164	0.5	2,526	0.14	485	89%	83%
Sulphurets Inferred inside pit	pit04	$1750 Au	$3.75/lb	159,677	0.5	2,567	0.14	493	90%	84%
Sulphurets Inferred inside pit	pit05	$2000 Au	$4.00/lb	162,663	0.5	2,615	0.14	502	92%	86%

Table 14.35 Mitchell Conceptual Pit Results

Pit – Resource Class	Pit	Au Price	Cu Price	Tonnes (000)	Au (g/t)	Au (oz 000)	Cu (%)	Cu (M lb)	% of Au oz in Pit	% of Cu lb in Pit
Mitchell Measured Resource	**n/a**	**n/a**	**n/a**	**724,000**	**0.65**	**15,130**	**0.18**	**2,872**	**100%**	**100%**
Mitchell Measured inside pit	pit01	$1000 Au	$3.00/lb	707,726	0.65	14,790	0.18	2,808	98%	98%
Mitchell Measured inside pit	pit02	$1250 Au	$3.25/lb	717,816	0.65	15,001	0.18	2,848	99%	99%
Mitchell Measured inside pit	pit03	$1500 Au	$3.50/lb	720,891	0.65	15,065	0.18	2,860	100%	100%
Mitchell Measured inside pit	pit04	$1750 Au	$3.75/lb	723,141	0.65	15,112	0.18	2,869	100%	100%
Mitchell Measured inside pit	pit05	$2000 Au	$4.00/lb	723,738	0.65	15,125	0.18	2,871	100%	100%
Mitchell Indicated Resource	**n/a**	**n/a**	**n/a**	**1,052,900**	**0.58**	**19,634**	**0.16**	**3,713**	**100%**	**100%**
Mitchell Indicated inside pit	pit01	$1000 Au	$3.00/lb	1,006,868	0.58	18,775	0.16	3,551	96%	96%
Mitchell Indicated inside pit	pit02	$1250 Au	$3.25/lb	1,035,874	0.58	19,316	0.16	3,653	98%	98%
Mitchell Indicated inside pit	pit03	$1500 Au	$3.50/lb	1,044,872	0.58	19,484	0.16	3,685	99%	99%
Mitchell Indicated inside pit	pit04	$1750 Au	$3.75/lb	1,049,332	0.58	19,567	0.16	3,700	100%	100%
Mitchell Indicated inside pit	pit05	$2000 Au	$4.00/lb	1,051,248	0.58	19,603	0.16	3,707	100%	100%
Mitchell Inferred Resource	**n/a**	**n/a**	**n/a**	**567,800**	**0.44**	**8,032**	**0.14**	**1,752**	**100%**	**100%**
Mitchell Inferred inside pit	pit01	$1000 Au	$3.00/lb	251,923	0.42	3,402	0.12	666	42%	38%
Mitchell Inferred inside pit	pit02	$1250 Au	$3.25/lb	322,393	0.43	4,457	0.12	853	55%	49%
Mitchell Inferred inside pit	pit03	$1500 Au	$3.50/lb	366,660	0.43	5,069	0.13	1,051	63%	60%
Mitchell Inferred inside pit	pit04	$1750 Au	$3.75/lb	412,503	0.44	5,835	0.13	1,182	73%	67%
Mitchell Inferred inside pit	pit05	$2000 Au	$4.00/lb	449,137	0.44	6,354	0.13	1,287	79%	73%

Table 14.36 Iron Cap Conceptual Pit Results

Pit – Resource Class	Pit	Au Price	Cu Price	Tonnes (000)	Au (g/t)	Au (oz 000)	Cu (%)	Cu (M lb)	% of Au oz in Pit	% of Cu lb in Pit
Iron Cap Indicated Resource	**n/a**	**n/a**	**n/a**	**361,700**	**0.44**	**5,117**	**0.21**	**1,674**	**100%**	**100%**
Iron Cap Indicated inside pit	pit01	$1000 Au	$3.00/lb	361,700	0.44	5,117	0.21	1,674	100%	100%
Iron Cap Indicated inside pit	pit02	$1250 Au	$3.25/lb	361,700	0.44	5,117	0.21	1,674	100%	100%
Iron Cap Indicated inside pit	pit03	$1500 Au	$3.50/lb	361,700	0.44	5,117	0.21	1,674	100%	100%
Iron Cap Indicated inside pit	pit04	$1750 Au	$3.75/lb	361,700	0.44	5,117	0.21	1,674	100%	100%
Iron Cap Indicated inside pit	pit05	$2000 Au	$4.00/lb	361,700	0.44	5,117	0.21	1,674	100%	100%
Iron Cap Inferred Resource	**n/a**	**n/a**	**n/a**	**297,300**	**0.36**	**3,441**	**0.20**	**1,310**	**100%**	**100%**
Iron Cap Inferred inside pit	pit01	$1000 Au	$3.00/lb	254,573	0.36	2,946	0.19	1,066	86%	81%
Iron Cap Inferred inside pit	pit02	$1250 Au	$3.25/lb	264,694	0.36	3,064	0.19	1,108	89%	85%
Iron Cap Inferred inside pit	pit03	$1500 Au	$3.50/lb	272,092	0.36	3,149	0.19	1,139	92%	87%
Iron Cap Inferred inside pit	pit04	$1750 Au	$3.75/lb	278,360	0.36	3,222	0.19	1,166	94%	89%
Iron Cap Inferred inside pit	pit05	$2000 Au	$4.00/lb	282,341	0.36	3,268	0.20	1,245	95%	95%

14.12 Risks and Uncertainties

In RMI's opinion, there is little risk associated with the insitu Mineral Resources which are the subject of this report. These estimated resources are based on drilling data that have been verified by RMI and are supported by adequate QA/QC results. Diamond drilling has shown that mineralization tends to be fairly continuous and widespread, especially within the Mitchell Zone. Gold and copper variograms suggest long ranges of mineralized continuity along preferential orientations. The estimated block grades have been demonstrated to be globally unbiased and provide a reasonable estimate of local grades. Back testing previous block models with newly obtained infill drilling results have been favourable.

The resources that are the subject of this report were not confined to a conceptual pit. RMI used the same cut-off grade that has been used for past resource estimates for comparison purposes. That cut-off grade of 0.50 g/t AuEQ is higher than a cut-off grade calculated using current prices or the average price over the past several years. RMI did generate a number of conceptual pits for each mineralized zone and compared resources captured by those pits versus the global inventory using the same cut-off grade. The "base case" conceptual pits captured nearly all the Measured (Mitchell Zone only) and Indicated Mineral Resources for all four zones. The conceptual pits captured less Inferred material than the global inventory, especially for the Mitchell Zone. Inferred material by its very nature is speculative and may never be upgraded into higher categories.

15.0 MINERAL RESERVE ESTIMATES

15.1 INTRODUCTION

Grade items used in mining reserves have been interpolated by Inverse Distance Weighting (IDW), as described in Section 14.0 of this study. The grade items used are copper (CUIDW), gold (AUIDW), silver (AGIDW), and molybdenum (MOIDW).

For the open pit phases included in Table 15.1, the Measured and Indicated (MI) Reserves are based on whole block grades with mining dilution and loss (varying by area) applied. Dilution grades estimated in Table 15.3 represent the average grade of material below the incremental COG for each pit area. Waste/ore COGs for the pit material are based on the NSR values in the blocks and are varied for each pit area as follows:

- Mitchell NSR COG: Cdn$9.57/t
- Sulphurets NSR COG: Cdn$10.17/t
- Kerr NSR COG: Cdn$9.61/t.

The underground reserves listed in Table 15.1 are based on the same resource model as the open pit. The underground mining NSR cut-offs vary by operation as follows:

- Mitchell Underground NSR Cut-off: Cdn$15.41
- Iron Cap Underground NSR Cut-off: Cdn$15.57.

The underground mining dilution is shown in Table 15.5 and is material with no grade. There is additional dilution within the block cave reserves (material below the NSR cut-offs) and it is mixed within the cave zone as the material is drawn down.

Table 15.1 Summarized Measured and Indicated Reserves

Area	Ore (Mt)	Diluted Grades					Waste (Mt)	Strip Ratio (t:t)
		NSR (Cdn$/t)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)		
Mitchell Pit								
M681	88	39.7	0.839	0.227	3.40	24.6	67	0.8
M682i	239	31.4	0.689	0.177	2.58	64.0	423	1.8
M683i	116	28.9	0.643	0.155	3.38	53.0	287	2.5
M684i	209	24.3	0.544	0.136	2.34	86.4	257	1.2
M685i	322	28.2	0.618	0.157	3.24	62.3	486	1.5
Subtotal Mitchell Pit	**973**	**29.3**	**0.642**	**0.163**	**2.92**	**63.4**	**1,519**	**1.6**
Kerr Pit								
K691	242	30.6	0.244	0.454	1.20	0.0	665	2.7
Sulphurets Pit								
S691	101	31.4	0.654	0.261	0.59	54.9	167	1.7
S692i	217	25.0	0.553	0.200	0.88	48.6	683	3.2
Subtotal Sulphurets Pit	**318**	**27.0**	**0.585**	**0.219**	**0.79**	**50.6**	**850**	**2.7**
Subtotal Open Pits	**1,533**	**29.0**	**0.567**	**0.221**	**2.20**	**50.7**	**3,035**	**2.0**
Underground Mining								
Mitchell	438	26.4	0.529	0.165	3.48	33.6	-	-
Iron Cap	193	25.3	0.450	0.196	5.32	21.5	-	-
Subtotal Underground	**631**	**26.1**	**0.505**	**0.174**	**4.05**	**29.9**	**-**	**-**
Grand Total	**2,164**	**28.1**	**0.549**	**0.207**	**2.74**	**44.7**	**3,035**	**1.4**

15.2 PIT RESERVE PARAMETERS

Proven and probable pit reserves are shown in Table 15.1 using block grades from the resource model and with mining dilution and loss applied (varying by area). Dilution grades estimated in Table 15.4 represent the average grade of material below the incremental COG for each pit area.

Waste/ore COGs are based on the incremental cost of processing the material. Process operating costs include plant processing (including crushing/conveying costs where applicable), G&A, surface service, tailing construction, and water treatment costs. The Sulphurets ore is the hardest and will cost approximately Cdn$0.60/t more to process than Mitchell ore. Kerr ore has approximately the same hardness as Mitchell and will cost slightly more (Cdn$0.04/t) to process than Mitchell ore. The total process costs (Cdn$/t ore milled) used to estimate COGs for each mining area are shown in Table 15.2.

Table 15.2 KSM PFS 2012 Unit Process Costs

	Mitchell (Cdn$/t Ore)	Sulphurets (Cdn$/t Ore)	Kerr (Cdn$/t Ore)
Process	7.24	7.84	7.28
G&A	1.15	1.15	1.15
Surface Service	0.31	0.31	0.31
Tailing Construction	0.46	0.46	0.46
Water Treatment Costs	0.41	0.41	0.41
Total Process Costs	9.57	10.17	9.61

Since this is a polymetallic ore, the COG value is therefore represented by the NSR grade, which is treated as a COG item. Because of the different processing costs by area, the NSR COGs are varied for each pit area as follows:

- Mitchell NSR COG: Cdn$9.57/t
- Sulphurets NSR COG: Cdn$10.17/t
- Kerr NSR COG: Cdn$9.61/t.

Table 15.3 Pit Mining Loss and Dilution

Pit	Total Loss (%)	Dilution (%)
Mitchell	2.2	0.8
Sulphurets	5.3	3.9
Kerr	4.5	3.2

Table 15.4 Dilution Grades by Pit Area

	Mitchell	Kerr	Sulphurets
Cu (%)	0.043	0.106	0.056
Au (g/t)	0.229	0.141	0.333
Ag (g/t)	1.45	0.78	0.59
Mo (ppm)	59.4	-	19.0
NSR (Cdn$/t)	7.55	7.60	8.19

Table 15.5 Underground Mining Dilution (Zero Grade)

	Mining Dilution
Mitchell	9%
Iron Cap	5%

Reserves tonnages use specific gravity (SG) as interpolated into the 3D block model (3DBM). Process recoveries are calculated into each block for each metal using a grade/recovery formula, for subsequent use in metal production and cash flow schedules.

16.0 MINING METHODS

A prefeasibility-level production schedule, based on a 130,000 t/d mill feed rate, has been developed for the KSM Project based on a combined open pit and underground mine plan. The pit phases are engineered based on the results of an updated economic pit limit analysis. Underground mining is designed for deeper Mitchell ore and at Iron Cap where block caving shows better economics or where underground mining is more suitable based on non-economic considerations.

16.1 OPEN PIT MINING OPERATIONS

16.1.1 INTRODUCTION

The mine planning work for this study is based on the 3DBM created by RMI for the NI 43-101 published resource model dated February 2012. An ARD 3DBM provided by RMI in March 2012 is used to estimate mine rock volumes by ARD type.

The mine planning for the KSM mineral property is based on work done with Mintec Inc.'s MineSight® (MineSight), a suite of software proven in the industry. The work includes adding engineering items to 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 included the base economic parameters, mining cost data derived from supplier estimates and data from other projects in the local area, recommended prefeasibility pit slope angles (PSAs), projected project metallurgical recoveries, plant costs, and throughput rates.

16.1.2 MINING DATUM

The project design work is based on NAD83 coordinates. 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 survey. Other LiDAR surveys were conducted in 2010 to increase coverage over areas to be used as RSFs. Effort has been made to ensure that all disciplines have used the same topography data.

16.1.3 Production Rate Consideration

A number of factors are considered when establishing an appropriate mining and processing rate. For KSM, key factors include:

- **Resource Size:** Typically, a planned mine life is set at 12.5 to 20 years; beyond this, time-value discounting shows an insignificant contribution to the NPV of the project.
- **Capital Payback:** Capital investment typically is targeted at projects with a payback period of 3 to 5 years.
- **Operational Constraints:** Power, water, or supplies and services for support of operations can limit production.
- **Site Delivery Constraints:** Physical size and weight of equipment and shipping limits can determine the maximum size of units that can be delivered to site. For this evaluation, the largest proven units are reasonably 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 flexibility issues.
  - Generally higher tonnage throughputs require more capital and the size of the project is reflected in the initial investment. Economies of scale can still apply where some access and construction issues have a high fixed component regardless of the size of size of the project.

Higher production rates generally pay back fixed capital earlier and provide a higher rate of return on capital, which improves project NPV.

The throughput has been restricted by the anticipated power availability. There are indications that limitations on power availability in the area may be lifted and a higher KSM mill throughput may therefore be possible.

A throughput of 130,000 t/d sets the open pit mine life at 32 years for the pit reserves. The Project NPV may be improved by increasing the mill throughput above 130,000 t/d early in the mine life. Increasing the mill throughput may improve project economics. Note that the underground mining as described in Section 16.3, extends the project life to 55 years.

16.1.4 Mine Planning 3D Block Model and MineSight Project

A single resource model is used in this study, based on the updated MineSight 3DBM provided by RMI, as noted previously. The resource model is a 3DBM with whole block Cu (%), Au (g/t), Ag (g/t), and Mo (ppm) grades.

The resource model also contains an SG (density) item and a topography (TOPO) item representing the proportion of a block below the topographic surface. Based on the RMI 3DBM (file name: *"ksmp15.012"*), a mine planning 3DBM has been created (file name: *"KSMP15.dat"*), with extra items added for mine planning tasks.

The PFS model dimensions are provided in Figure 16.1. The 15 m block height represents a suitable bench height for large scale mining shovels, and the 25 m x 25 m horizontal dimensions give a block size of 9,375 BCM and approximately 26,000 t per block (assuming average SG). This represents approximately four blocks mined per day for each shovel, a suitable resolution for long range planning. A list of mine planning 3DBM items is given in Table 16.1. The total model area is illustrated for orientation in plan view in Figure 16.2.

Figure 16.1 KSMP Mine Planning Model Limits

	Minimum	Maximum	Size	Number
X	420500	425900	25	216
Y	6257800	6269000	25	448
Z	-210	2145	15	157

Note: X = Easting, Y = Northing, Z = Elevation.

Table 16.1 3DBM Items

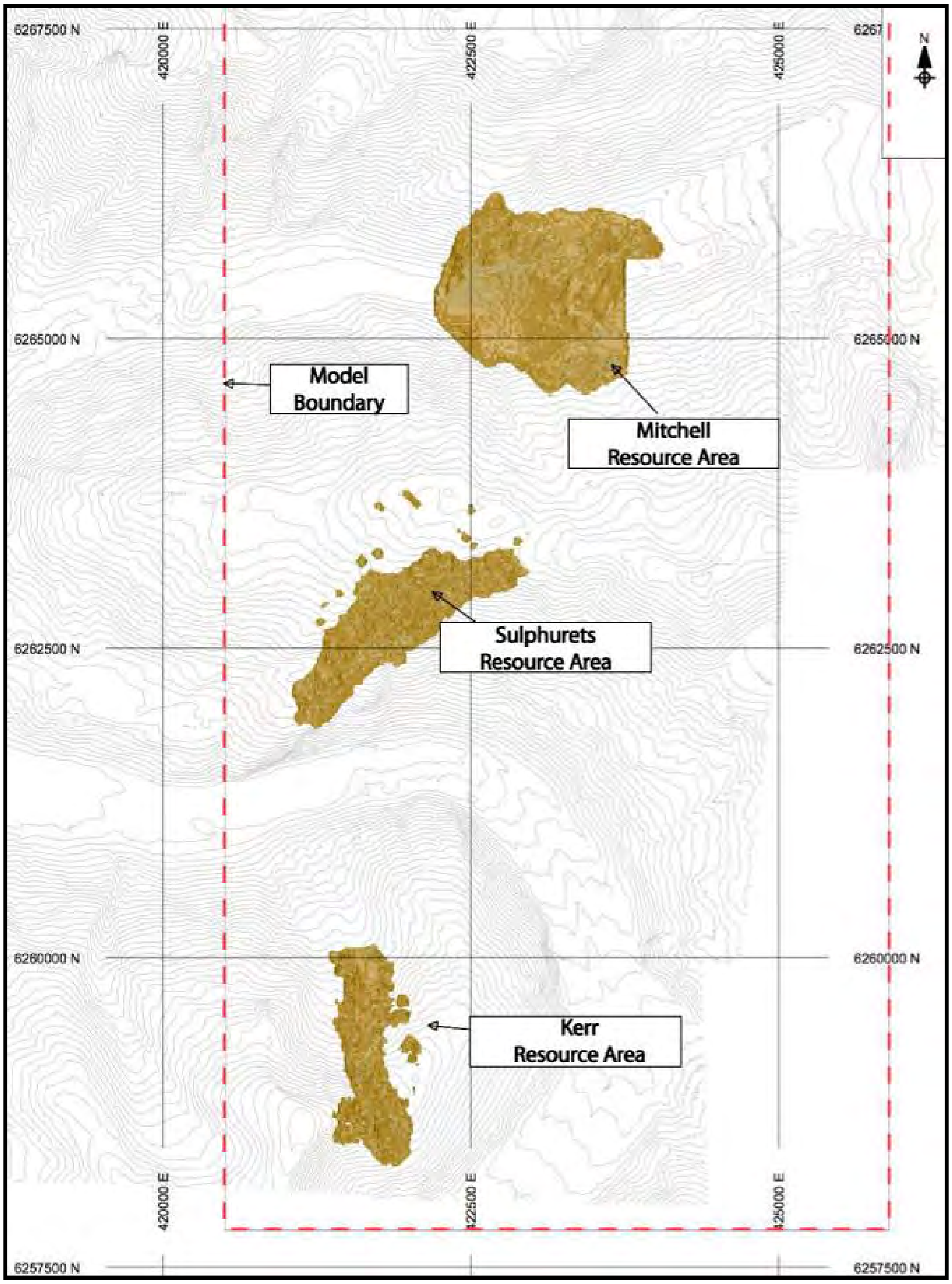
Item	Item By	Description
TOPO	RMI	Percent of Block Below Surface
SG	RMI	Bulk Density (tonnes/BCM)
AUIDW	RMI	Inverse Distance Block Gold Grade (g/t)
CUIDW	RMI	Inverse Distance Block Copper Grade (%)
MOIDW	RMI	Inverse Distance Molybdenum Block Grade (ppm)
AGIDW	RMI	Inverse Distance Block Silver Grade (g/t)
LITH	RMI	Lithologic Code
AREA	RMI	Area Code (1=Kerr; 2=Sulphurets; 3=Mitchell; 4=Iron Cap)
CLASS	RMI	Resource Category (1=Measured; 2=Indicated; 3=Inferred)
ORTYP	RMI	Ore Type Code
ABA*	RMI	Rock Type Classification From ABA Model (1 = PAG, 2 = Uncertain, 3 = not-PAG, 4 = Ice)
SNPRA	RMI	Ratio for Determining PAG, Uncertain, and not-PAG from ABA Model
NSR	MMTS	Net Smelter Return (Cdn$/t)
CUREC	MMTS	Copper Process Recovery (%)
AUREC	MMTS	Gold Process Recovery (%)
MOREC	MMTS	Molybdenum Process Recovery (%)
AGREC	MMTS	Silver Process Recovery Grade (%)

table continues...

Item	Item By	Description
RCU	MMTS	Recovered Copper Grade (%)
RAU	MMTS	Recovered Gold Grade (%)
RAG	MMTS	Recovered Silver Grade (%)
RMO	MMTS	Recovered Molybdenum Grade (%)

* ABA = acid base accounting.

Figure 16.2 KSM Grade Model Resource Areas and NSR >Cdn$7/t Grade Shells – Plan View with 30 m Topography Contours



NET SMELTER RETURN

Ore and waste COGs are based on the NSR in Cdn$/t, which is determined using NSPs (provided in Appendix E). The NSR (net of offsite concentrate and smelter charges and including onsite mill recovery) is used as a cut-off item for break-even ore/waste selection and for the grade bins for cash flow optimization. The NSP is based on base case metal prices, US$ exchange rate, and offsite transportation, smelting, and refining charges, etc. (Appendix E). The metal prices and resultant NSPs used at this early stage of the study, are shown in Table 16.2.

Table 16.2 Metal Prices and NSP

	Metal Price (US$)	NSP (Cdn$)
Cu	3.21/lb	2.93/lb
Au	1244/oz	39.02/g
Ag	22.98/oz	0.649/g
Mo	14.14/lb	9.70/lb

Metallurgical recoveries used for the NSR calculation are based on test work conducted by G&T and evaluated by Tetra Tech, and are listed in detail later in Section 16.1.5.

The NSR calculation is shown in Equation 16.1, shown on the following page.

Equation 16.1 NSR Formula

$$\textbf{NSR} = \frac{Cu}{100} \times \frac{Rec\mathit{Cu}}{100} \times NSP\mathit{Cu} \times 2204.6 + Au \times \frac{Rec\mathit{Au}}{100} \times NSP\mathit{Au} + Ag \times \frac{Rec\mathit{Ag}}{100} \times NSP\mathit{Ag} + \frac{Mo}{1 \times 10^6} \times \frac{Rec\mathit{Mo}}{100} \times NSP\mathit{Mo} \times 2204.6$$

Where:

- Cu = copper grade (%) from the CUIDW 3DBM item
- Au = gold grade (g/t)from the AUIDW 3DBM item
- Mo = molybdenum grade (ppm)from the MOIDW 3DBM item
- Ag = silver grade (g/t)from the AGIDW 3DBM item
- Rec*Cu* = copper recovery (%)
- Rec*Au* = gold recovery (%)
- Rec*Mo* = molybdenum recovery (%)
- Rec*Ag* = silver recovery (%)
- NSP*Cu* = net smelter price for copper (Cdn$/lb)
- NSP*Au* = net smelter price for gold (Cdn$/g)
- NSP*Mo* = net smelter price for molybdenum (Cdn$/lb)
- NSP*Ag* = net smelter price for silver (Cdn$/g).

MINING LOSS AND DILUTION

The KSM Project is a large gold-copper porphyry deposit and the orebody occurs relatively continuously within the COG shells. The pits will be mined with large shovels and trucks at an ore mining rate of 130,000 t/d. As is typical of large porphyries, blasthole assays will be used to determine the waste/ore boundaries for material designations on the pit bench for daily operations.

RMI developed the 3DBM for KSM with 25 m x 25 m x 15 m block sizes. Each block in the model has a volume of 9,375 m3 and weighs approximately 26,000 t. The plant feed will require around 4.5 blocks per day, which is an appropriate selective mining unit for the size of shovel utilized in the mine plan. The interpolation of the metal grades to the 3DBM averages the composites to a single value for each metal. This smoothing is, in effect, a numeric dilution where higher composite values are averaged down; conversely, lower values are averaged up. Because of the continuous/smooth nature of the mineralization, it is assumed this smoothing down and up leads to an average close to the COG within blocks that are on the fringe of being ore or waste.

During operations, an Ore Control System (OCS) from blasthole sampling will be conducted on an approximate 8.5 m spacing to determine cut-off boundaries for shovel dig limits. These smaller ore/waste blocks will be too small to separate with the large shovels, especially after the material has been displaced by blasting. Therefore, the dilution from isolated blasthole blocks will be handled as whole block dilution in the 3DBM. The OCS will define smaller ore/waste zones but these will be smoothed into larger units that the shovels can also selectively mine. These larger units from the OCS are better represented by the 3DBM size blocks and will define contacts between ore and waste. These contact boundaries are approximated by the 3DBM as the smallest sized units the shovels can selectively mine. The 3DBM blocks can therefore be used to define contact dilution factors.

Blasting will create displacement along waste/ore boundaries; as the material is loaded onto the trucks, some ore will be lost to waste (mining loss) and some waste will be added to the ore (dilution). During some seasons, material will stick or freeze to the inside of the truck boxes and create carry-back, which can contribute to mining loss and dilution. Also, misdirected loads can send ore to the waste dump (mining loss), or waste to the crusher or, more likely, to a low grade stockpile (dilution). In order to properly calculate the reserve files for scheduling purposes, mining losses and dilution must be taken into account.

The mining reserves used for scheduling are calculated from grades in the 3DBM using detailed pit designs with the appropriate mining recoveries and dilutions applied. The recoveries and dilutions convert the in-situ ore tonnages into ROM delivered tonnage to the mill. The ROM delivered tonnage (i.e. what the mill will actually “see”) is used to determine the appropriate production schedule.

There are three main parts to recovery and dilution:

- dilution of waste into ore where separate ore and waste blasts are not possible
- loss of ore into waste where separate ore and waste blasts are not possible
- general mining losses and dilution due to handling (haul back in truck boxes, stockpile floor losses, etc.).

In addition to the whole block dilution and the general mining losses and dilution, allowance is made for the contacts between ore and waste on the mining bench as defined by the NSR cut-off. This is affected by the size of the ore areas on the bench and the relative amount of edges. On a block-by-block basis, this is determined by the number of waste neighbours a block has or vice versa for waste. For this project, the Mitchell area has more massive ore zones on a bench than the other areas; therefore, contact dilution for this area is less. For this PFS, MMTS estimated a mining loss and dilution factor that varies by pit area. The full mining recovery (loss) and dilution estimation method including the edge block calculation is provided in Appendix E; the resultant values, shown by pit, are provided in Table 16.3.

Table 16.3 Mining Loss and Dilution

Pit	Total Loss (%)	Dilution (%)
Mitchell	2.2	0.8
Sulphurets	5.3	3.9
Kerr	4.5	3.2

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 ore blocks inside the pit delineated area. These dilution grades are estimated by statistical analysis of grades in blocks with NSR greater than an arbitrary $5/t and less than the cut-off NSR. The dilution grades are estimated in Table 16.4 representing the average grade of material inside this grade envelope.

Table 16.4 Dilution Grades by Pit Area

	Mitchell	Kerr	Sulphurets
Cu (%)	0.043	0.106	0.056
Au (g/t)	0.229	0.141	0.333
Ag (g/t)	1.45	0.78	0.59
Mo (ppm)	59.4	-	19.0
NSR (Cdn$/t)	7.55	7.60	8.19

16.1.5 Economic Pit Limits, Pit Designs

Economic pit limits for the Seabridge Mitchell, Kerr, and Sulphurets deposits have been updated for this study using MineSight's MS-EP program. A sensitivity to slope angles and an analysis of the opportunity to upgrade Inferred resources is also included in this analysis.

Pit Optimization Method

The economic pit limit is selected after evaluating Lerchs-Grossman (LG) pit cases conducted with MS-EP. This PFS is based on the updated 3DBM, created by RMI in January 2011 (block model file: "*ksmp15.012*"), which includes results from 2011 drilling.

The LG assessment is carried out by generating sets of LG pit shells using "MS-EP-design" by varying revenue assumptions and pit slope to test the deposit's geometric/topographic and pit slope sensitivity.

The ultimate pit limit is determined by estimating the pit size where an incremental increase in pit size does not significantly increase the pit resource. The selected pit limit is chosen where the economic return starts to significantly drop off. Economics of the selected pit limits are also tested to determine that they are economically viable.

Economic Pit Limit Assessment

This section outlines the design basis for the LG pit limit assessment.

LG Pit – Unit Mining Costs

Mining unit costs to generate the LG pits are based on the 2011 PFS mine cost model. The LG runs use the estimated cost for the last incremental pit shell from previous studies since this is where the incremental cut-off conditions will be applied. Earlier studies used variable bench mining costs but, for this study, the pit limiting shells can be determined using the anticipated costs for the lower pit benches.

The average unit mining costs per tonne mined used in the LG pit program are shown in Table 16.5.

Table 16.5 LG Pit – Unit Mining Costs

	Ore Mining Cost (Cdn$/t)	Waste Mining Cost (Cdn$/t)
Mitchell	1.55	1.83
Sulphurets	1.89	2.00
Kerr	1.66	1.66

LG Pit – Process Operating Costs

Total process costs, estimated by Tetra Tech in February 2012, for the MS-EP runs include all plant operations (process), G&A, surface service, tailing construction, and water treatment costs.

The average unit mining costs per tonne mined are shown in Table 16.6.

Table 16.6 2012 KSM PFS Unit Process Costs

	Mitchell (Cdn$/t Ore)	Sulphurets (Cdn$/t Ore)	Kerr (Cdn$/t Ore)
Process	7.24	7.84	7.28
G&A	1.15	1.15	1.15
Surface Service	0.31	0.31	0.31
Tailing Construction	0.46	0.46	0.46
Water Treatment Costs	0.41	0.41	0.41
Total Process Costs	**9.57**	**10.17**	**9.61**

PIT SLOPE ANGLES

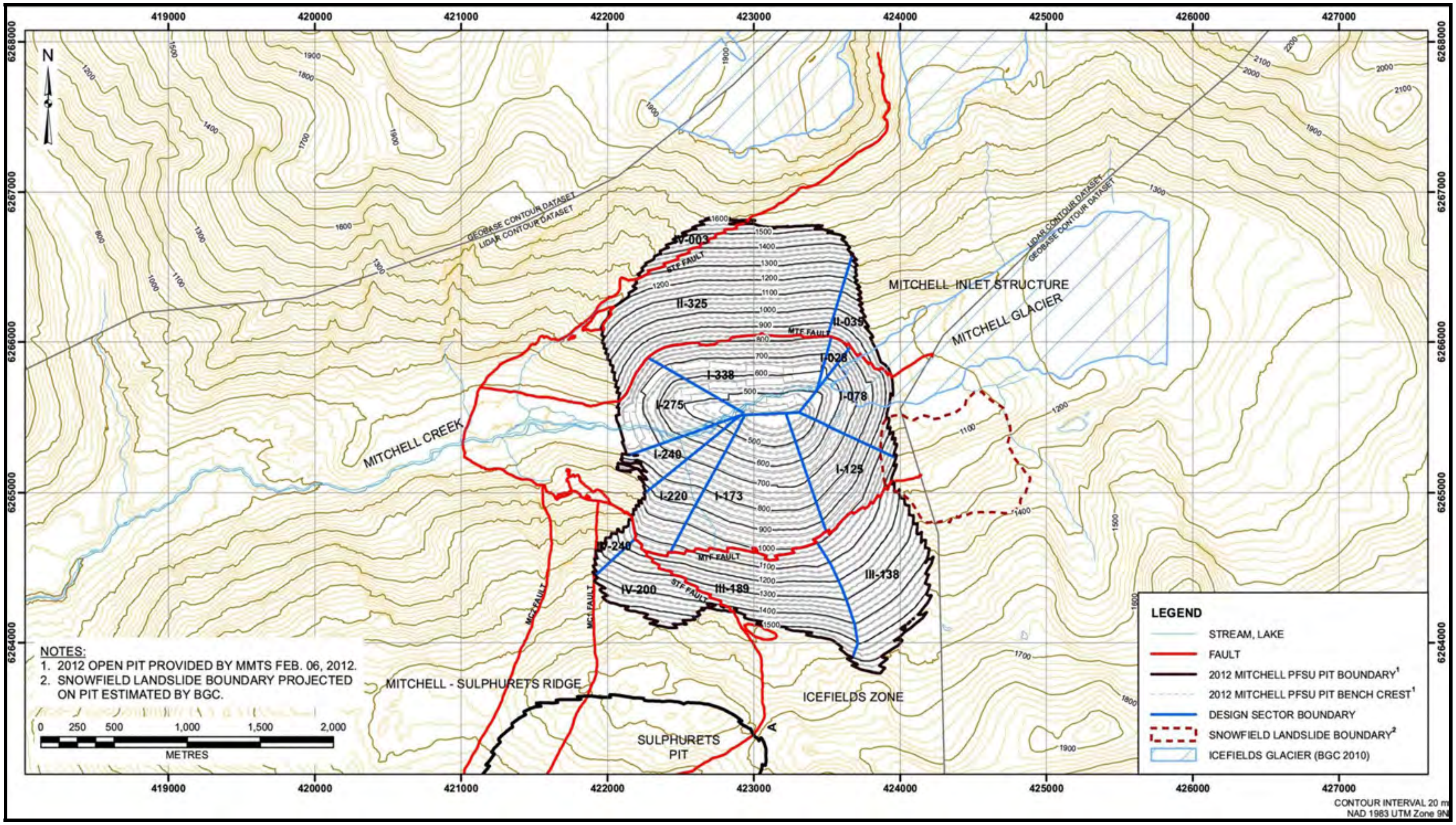
Maximum PSAs are based on recommendations by BGC, as presented in Section 16.4.

BGC recommends varying the maximum interramp angles (IRA) and overall angles (OA) by both pit face azimuth and by specific material zones delineated in pit area. The PSA value used in the MS-EP runs, considers the maximum IRA, the maximum OA, and includes an allowance for estimated haul ramps.

Figure 16.3 to Figure 16.5 and Table 16.7 to Table 16.9 summarize pit slope assumptions for the Mitchell, Sulphurets, and Kerr pit areas.

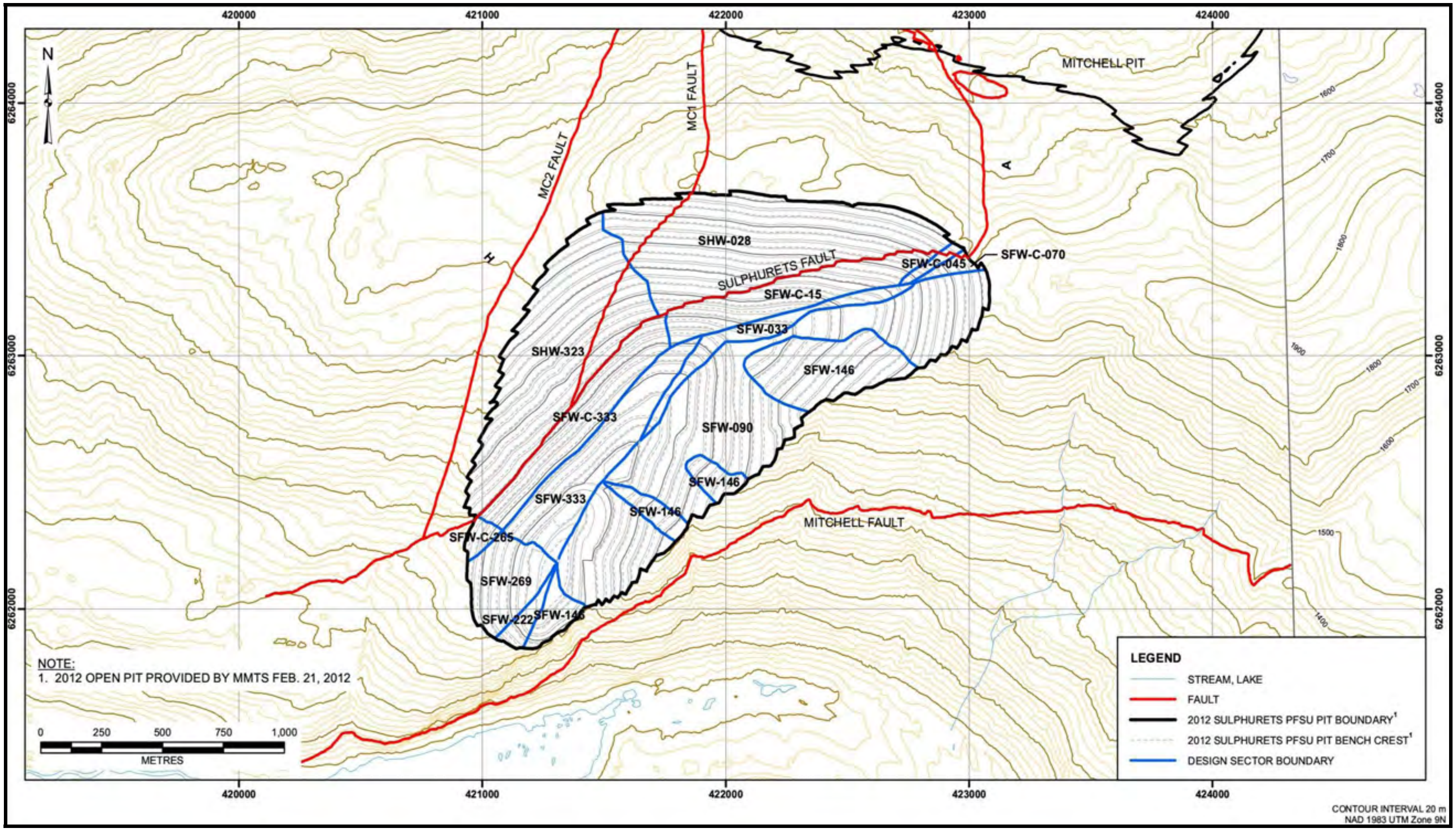
SEABRIDGE GOLD

Figure 16.3 Mitchell Pit Slope Design Sectors



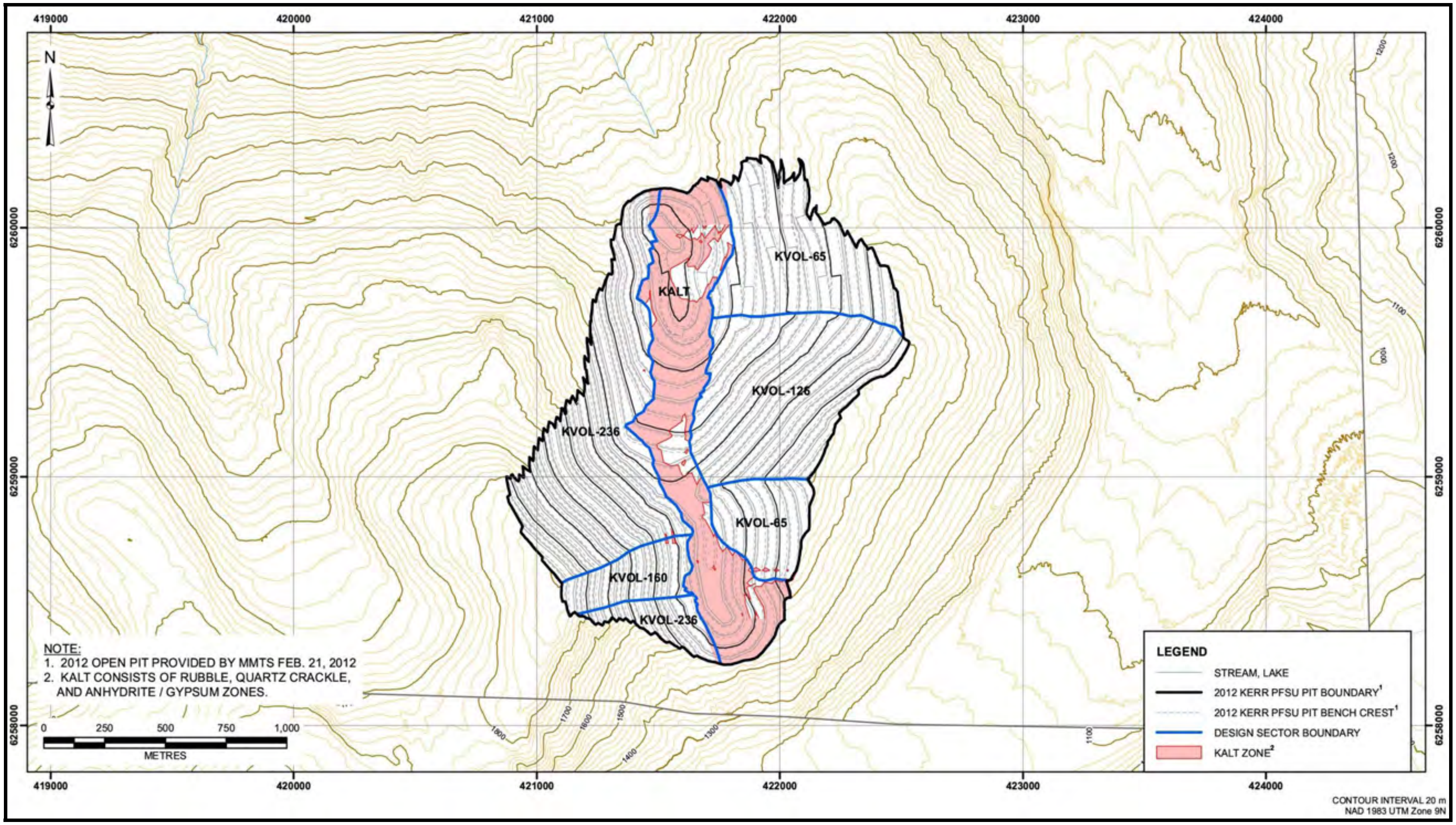
Source: BGC.

Figure 16.4 Sulphurets Pit Slope Design Sectors



Source: BGC.

Figure 16.5 Kerr Pit Slope Design Sectors



Source: BGC.

Table 16.7 Mitchell Pit Slope Domains and Pit Slope Angles for LG Pits

Domain & Sector Azimuth	Azimuth Start (°)	Azimuth End (°)	Maximum IRA (°)	Maximum OA (°)	LG PSA (°)
I-173	135	210	36	36	34.5
I-220	210	230	40	40	38
I-240	230	250	48	48	45
I-275	250	300	53	52	50
I-338	300	015	53	46	44
I-028	015	040	53	46	44
I-078	040	115	48	48	46
I-125	115	135	46	45	43
II-325	270	020	53	44	44
II-035	020	050	46	47	46
II-058	050	065	40	42	40
II-078	065	090	36	41	36
III-099	090	108	54	55	54
III-138	108	168	34	35	34
III-189	168	210	46	47	46
IV-168	145	190	46	47	46
IV-200	190	210	39	40	39
IV-240	210	270	34	36	34
IV-003	325	040	46	47	46

Table 16.8 Sulphurets Pit Slope Domains and Pit Slope Angles for LG Pits

Domain & Sector Azimuth	Azimuth Start (°)	Azimuth End (°)	Maximum IRA (°)	Maximum OA (°)	LG PSA (°)
SHW-V-280	270	290	49	50	46
SHW-V-323	290	355	40	42	37
SHW-V-028	355	060	45	45	42
SHW-V-075	060	090	36	41	36
SFW-C-265	220	310	45	49	45
SFW-C-333	310	355	49	53	49
SFW-C-015	355	035	50	53	50
SFW-C-045	035	055	45	49	45
SFW-C-070	055	085	40	45	40
SFW-V-190	172	207	40	44	37
SFW-V-222	207	237	47	50	44
SFW-V-269	237	300	37	41	35
SFW-V-333	300	005	40	44	37
SFW-V-033	005	060	36	37	34
SFW-V-090	060	120	40	41	38
SFW-V-146	120	172	36	40	34

Table 16.9 Kerr Pit Slope Domains and Pit Slope Angles for LG Pits

Domain & Sector Azimuth	Azimuth Start (°)	Azimuth End (°)	Maximum IRA (°)	Maximum OA (°)	LG PSA (°)
KVOL-236	180	292	50	49	46
KVOL-335	292	017	36	37	34
KVOL-065	017	112	34	36	34
KVOL-126	112	140	40	41	37
KVOL-160	140	180	45	45	41
KALT-180	135	225	36	25	25
KALT-000	225	135	36	40	34

Economic pit limit sensitivity to the LG PSA has been tested by producing additional sets of LG pits using 5° flatter PSAs and 5° steeper PSAs than those shown in Table 16.7 to Table 16.9.

PROCESS RECOVERIES

Process recovery assumptions used to generate the LG pits are provided by Tetra Tech and are shown in Table 16.10 to Table 16.12.

Table 16.10 Mitchell Process Recovery Assumptions

Metal	Head Grade	Recovery (%)
Cu	>1.0% Cu	95
	0.8 - 1.0% Cu	92
	0.227 - 0.8% Cu	= 90.68 x (Cu Head, %)^0.027
	0.05 - 0.227% Cu	= 18.02 x ln(Cu Head, %) + 113.5
	0.02 - 0.05% Cu	20
	<0.02%	3
Au	(to Dore) >10 g/t Au	= (98 - (0.054 x (Cu recovery, %)^1.575)) x 80% x 99%
	(to Dore) 5 - 10 g/t Au	= (95 - (0.054 x (Cu recovery, %)^1.575)) x 75% x 99%
	(to Dore) 0.1 - 5 g/t	=(87.491 x (Au Head, g/t)^0.051 - (0.054 x (Cu recovery, %) ^1.575)) x 68% x 99%
	(to Dore)<0.1 g/t	= 0
	(to Cu concentrate) n/a	= 0.054 x (Cu recovery, %)^1.575
Mo	>0.010%	47
	0.005-0.010% Mo	35
	0.0025-0.005% Mo	25
	<0.0025% Mo	0

table continues...

Metal	Head Grade	Recovery (%)
Ag	(to Dore) <1 g/t	0
	(to Dore) 1–8 g/t	= (43.16 x (Ag head, g/t)^0.329) – (1.496 x (Cu recovery, %) – 76.58; if<0 then use 0%
	(to Dore) 8–15 g/t	= 86 – (1.496 x (Cu recovery, %) – 76.58)
	(to Dore) >15 g/t	=88 – (1.496 x (Cu recovery, %) – 76.58)
	(to Cu concentrate) n/a	= 1.496 x (Cu recovery, %) – 76.58

Table 16.11 Sulphurets Process Recovery Assumptions

Metal	Head Grade	Recovery (%)
Cu	>1.0% Cu	93
	0.8 - 1.0% Cu	90
	0.227 - 0.8% Cu	= 90.68 x (Cu Head, %)^0.027 – 3.5
	0.05 - 0.227% Cu	= 18.02 x ln(Cu Head, %) + 110
	0.02 - 0.05% Cu	20
	<0.02%	3
Au	(to Dore) >10 g/t Au	= (98 - (0.054 x (Cu recovery, %)^1.575)) x 70% x 99%
	(to Dore) 5 - 10 g/t Au	= (95 - (0.054 x (Cu recovery, %)^1.575)) x 60% x 99%
	(to Dore) 0.1 - 5 g/t	=((87.491 x (Au Head, g/t)^0.051 + 3) - (0.054 x (Cu recovery, %)^1.575 – 2)) x 49% x 99%
	(to Dore)<0.1 g/t	= 0
	(to Cu concentrate) n/a	= 0.054 x (Cu recovery, %)^1.575 - 2
Mo	>0.010%	47
	0.005-0.010% Mo	35
	0.0025-0.005% Mo	25
	<0.0025% Mo	0
Ag	(to Dore) <1 g/t	0
	(to Dore) 1–8 g/t	= (43.16 x (Ag head, g/t)^0.329) – 35.3
	(to Dore) 8–15 g/t	50.7
	(to Dore) >15 g/t	52.7
	(to Cu concentrate) n/a	35

Table 16.12 Kerr Process Recovery Assumptions

Metal	Head Grade	Recovery (%)
Cu	>1.0% Cu	84
	0.8 - 1.0% Cu	81
	0.227 - 0.8% Cu	= 90.68 x (Cu Head, %)^0.027 - 9
	0.05 - 0.227% Cu	= 18.02 x ln(Cu Head, %) + –104.5
	0.02 - 0.05% Cu	20
	<0.02%	3
Au	(to Dore) >10 g/t Au	= (98 - (0.054 x (Cu recovery, %)^1.575 - 18)) x 80% x 99%
	(to Dore) 5 - 10 g/t Au	= (95 - (0.054 x (Cu recovery, %)^1.575 - 18)) x 75% x 99%
	(to Dore) 0.1 - 5 g/t	= ((87.491 x (Au Head, g/t)^0.051 + 8) - (0.054 x (Cu recovery, %)^1.575 – 18)) x 68% x 99%
	(to Dore) <0.1 g/t	0
	(to Cu concentrate) n/a	= 0.054 x (Cu recovery, %)^1.575 - 18
Mo	>0.010%	47
	0.005-0.010% Mo	35
	0.0025-0.005% Mo	25
	<0.0025% Mo	0
Ag	(to Dore) <1 g/t	0
	(to Dore) 1-8 g/t	= (21.26 x ln(Au Head, g/t) + 40.74) – 37
	(to Dore) 8-15 g/t	49
	(to Dore) >15 g/t	51
	(to Cu concentrate) n/a	37

METAL PRICES

The base case metal prices and NSP are given in Table 16.13. The NSP is based on base case metal prices, an exchange rate of Cdn$1.00 to US$0.95, and off site transportation, smelting and refining charges, etc. (the smelter schedule is provided in the Design Basis Memorandum, available in Appendix E).

Table 16.13 Metal Prices and NSP Base Case

Metal	Price (US$)	NSP (Cdn$)
Cu	3.21/lb	2.93/lb
Au	1244/oz	39.02/g
Mo	14.14/lb	9.70/lb
Ag	22.98/oz	0.649/g

LG pits are generated by varying prices in the range from 30% to 150% of the base NSP.

LG ECONOMIC PIT LIMITS

Pit shell cases are created by varying the input LG prices. Sensitivity cases have been run for the various pit slope assumptions described in Section 16.1.5. Figure 16.6 to Figure 16.8 summarize the revenue and slope sensitivity cases for the Mitchell, Sulphurets, and Kerr pits, respectively.

Figure 16.6 Mitchell Sensitivity of Ore Tonnes to Pit Slope and Pit Size

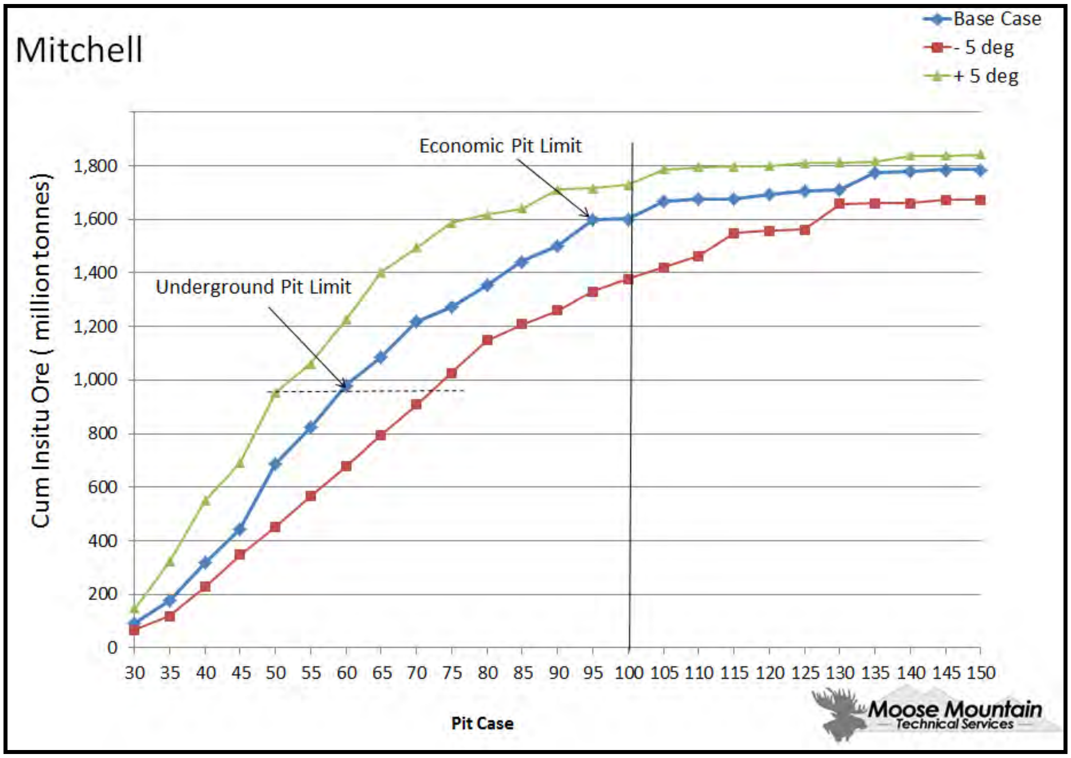


Figure 16.7 Sulphurets Sensitivity of Ore Tonnes to Pit Slope and Pit Size

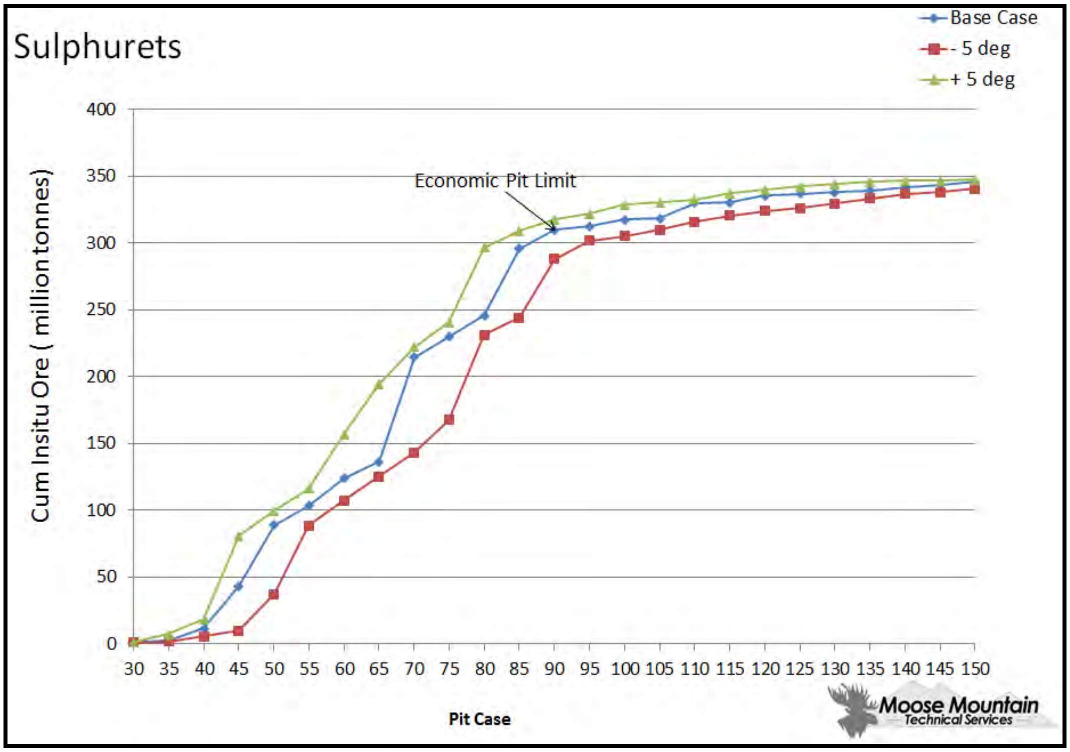
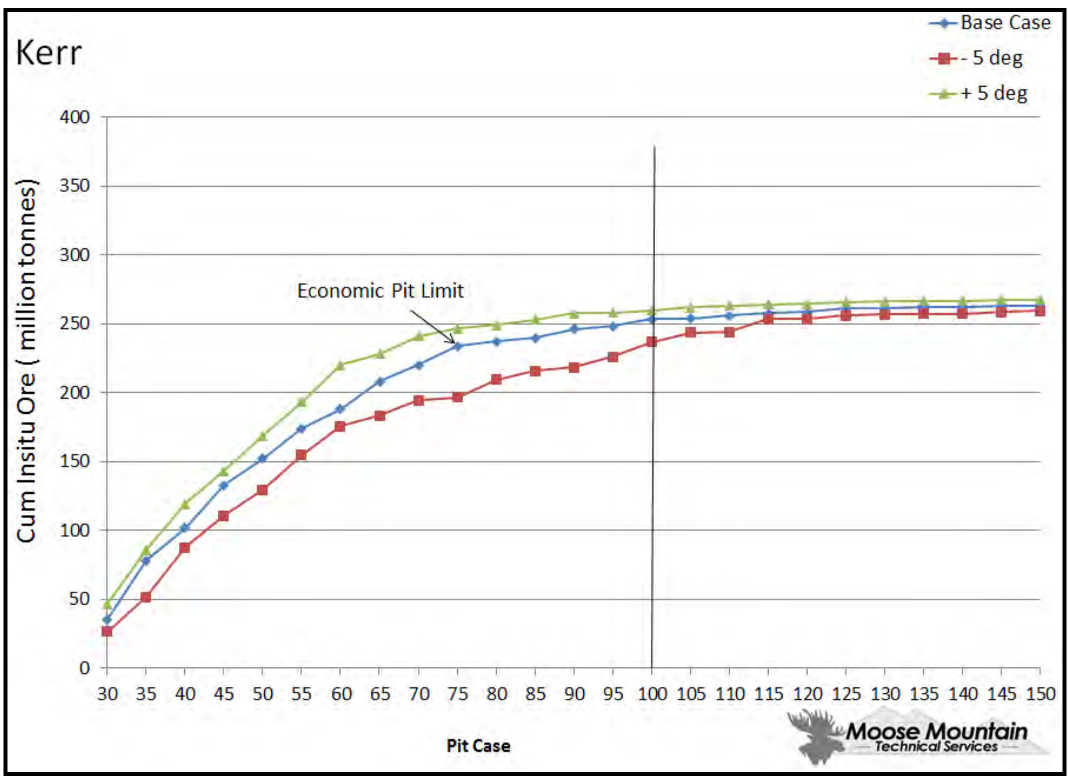


Figure 16.8 Kerr Sensitivity of Ore Tonnes to Pit Slope and Pit Size



Inflection points in Figure 16.6 to Figure 16.8 occur where an incremental increase in pit size does not significantly increase the pit resource, or an incremental increase in the pit resource results in only marginal economic return. In the Sulphurets and Kerr areas, these inflection points represent potential economic pit limits and are selected for each pit area. The ultimate pit for Mitchell is selected where the operating cost per tonne of ore for mining one bench lower by open pit method begins to exceed the unit operating cost of mining incrementally higher with a block cave:

- Mitchell – open pit/underground pit case: 60%
- Sulphurets – inflection pit case: 90 %
- Kerr – inflection pit case: 75 %

The pit resources from LG pit limits selected are shown in Table 16.14.

Figure 16.6 to Figure 16.8 show the sensitivity of the LG pit limit to PSAs with the following results:

- The Mitchell economic pit limit is sensitive to PSA assumptions.
- The Sulphurets economic pit limit is not sensitive to PSA assumptions.
- The Kerr economic pit limit is not sensitive to PSA assumptions.

Table 16.14 Summary of the LG Pit Limit Measured and Indicated Resources

Pit Area	LG Pit Case	In Situ 'Ore' (Mt)	In Situ Grades					Waste (Mt)	Strip Ratio (t:t)
			NSR (Cdn$/t)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)		
Mitchell	60	980	30.3	0.656	0.171	3.05	61	1,342	1.4
Sulphurets	90	310	27.8	0.599	0.226	0.78	52	859	2.8
Kerr	75	234	32.0	0.253	0.475	1.23	-	476	2.0
Total		**1,524**	**30.1**	**0.582**	**0.229**	**2.31**	**50**	**2,677**	**1.8**

COMPARISON OF LG PIT LIMITS

The differences between the 2012 PFS and the 2011 PFS economic pit limits are summarized in Table 16.15.

Table 16.15 Comparison of the 2012 PFS and the 2011 PFS Inflection Point LG Pit Resources for Ultimate Economic Pit limits

Pit	In Situ 'Ore' (Mt)	Insitu Grades						Waste (Mt)	Strip Ratio
		NSR (Cdn$/t)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)			
PFS 2011 Measured & Indicated LG Pit Resource									
Mitchell	1,442	24	0.618	0.165	3.05	62		3,105	2.2
Sulphurets	166	30	0.646	0.277	0.59	63		547	3.3
Kerr	212	30	0.259	0.473	1.32	-		462	2.2
Total	**1,820**	**25**	**0.555**	**0.211**	**3.09**	**54**		**4,632**	**2.2**
PFS 2012 Measured & Indicated LG Pit Resource									
Mitchell	1,599	28.4	0.610	0.165	3.06	59		4,106	2.6
Sulphurets	310	27.8	0.599	0.226	0.78	52		859	2.8
Kerr	234	32.0	0.253	0.475	1.23	-		476	2.0
Total	**2,142**	**28.7**	**0.569**	**0.207**	**2.53**	**52**		**5,441**	**2.5**
Difference (PFS 2012 minus PFS 2011)									
Mitchell	157	28.4	0.534	0.164	3.18	36		1,001	6.4
Sulphurets	144	25.3	0.544	0.167	1.01	39		312	2.2
Kerr	22	51.5	0.191	0.490	0.33	-		14	0.7
Total	**322**	**28.6**	**0.516**	**0.187**	**2.02**	**35**		**1,327**	**4.1**

Table 16.15 indicates the following significant changes:

- The updated Mitchell and Kerr pit limits are larger as a result of the higher metal prices used.
- The updated Sulphurets economic pit has an increased resource, primarily due to the upgrading of Inferred material to Indicated, and an increase in average gold grade.

Plan view, north-south section views, and orthographic views of the LG pits for all four mining areas are shown in Figure 16.9 to Figure 16.12.

Figure 16.9 Plan View of the KSM LG Pit Limits

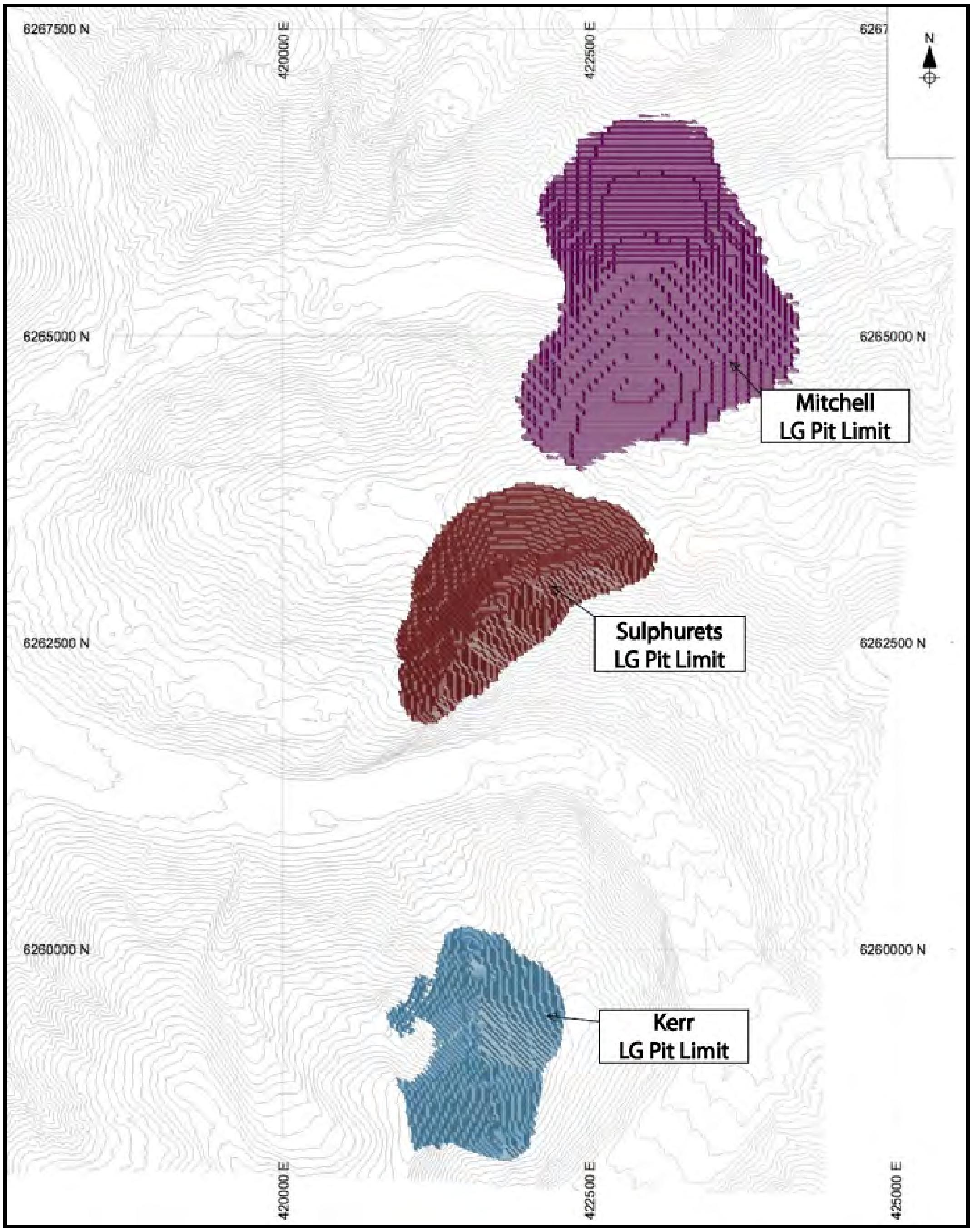


Figure 16.10 Mitchell Open/Underground Pit and Economic Pit Limit – N-S Section at East 422950, Viewed from the East

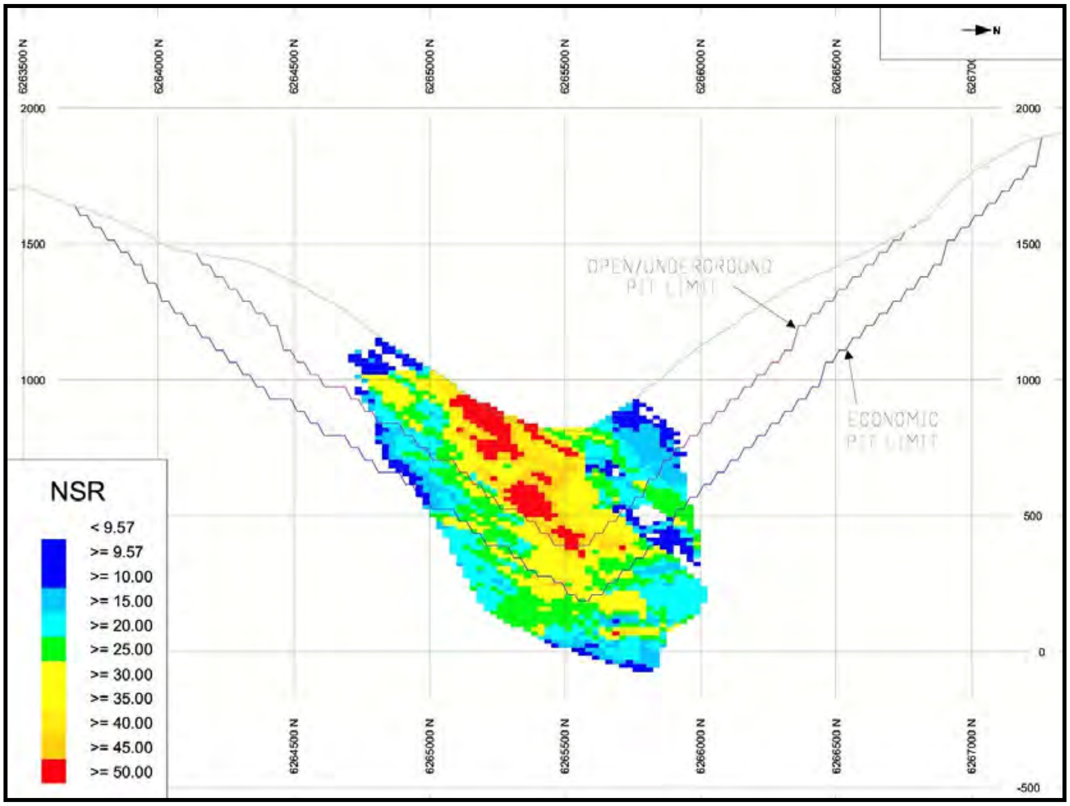


Figure 16.11 Sulphurets Economic Pit Limit – NS Section at East 421725 Viewed from the East

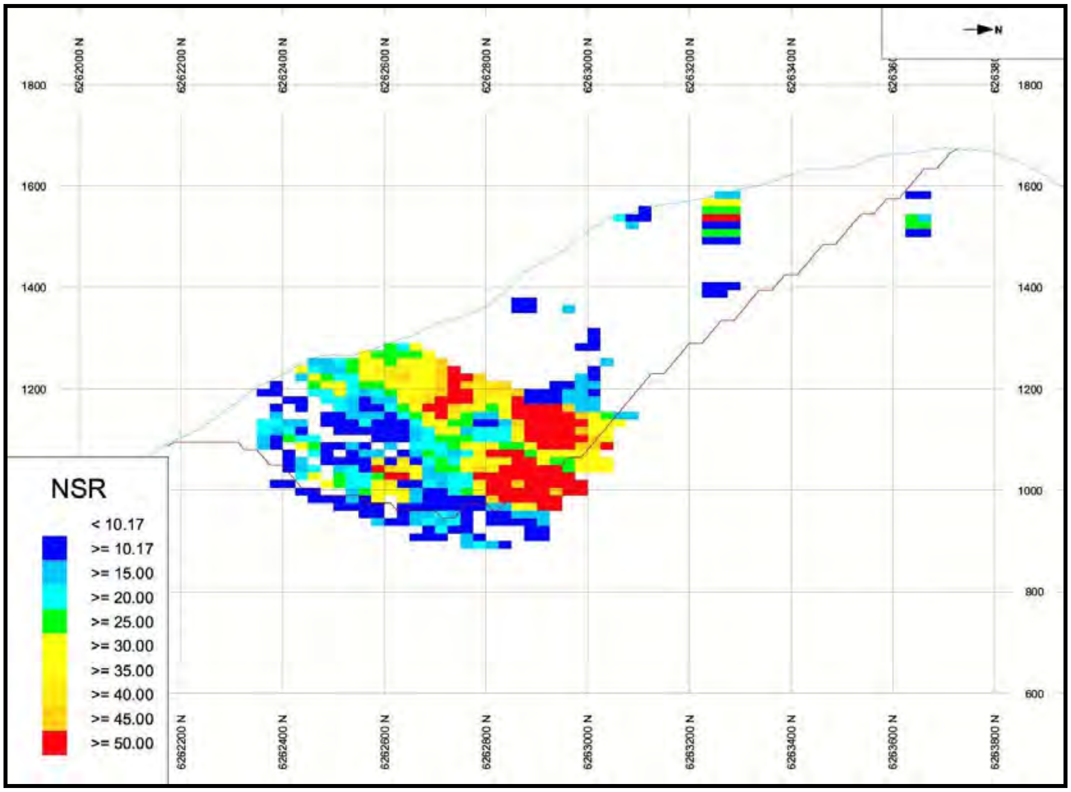
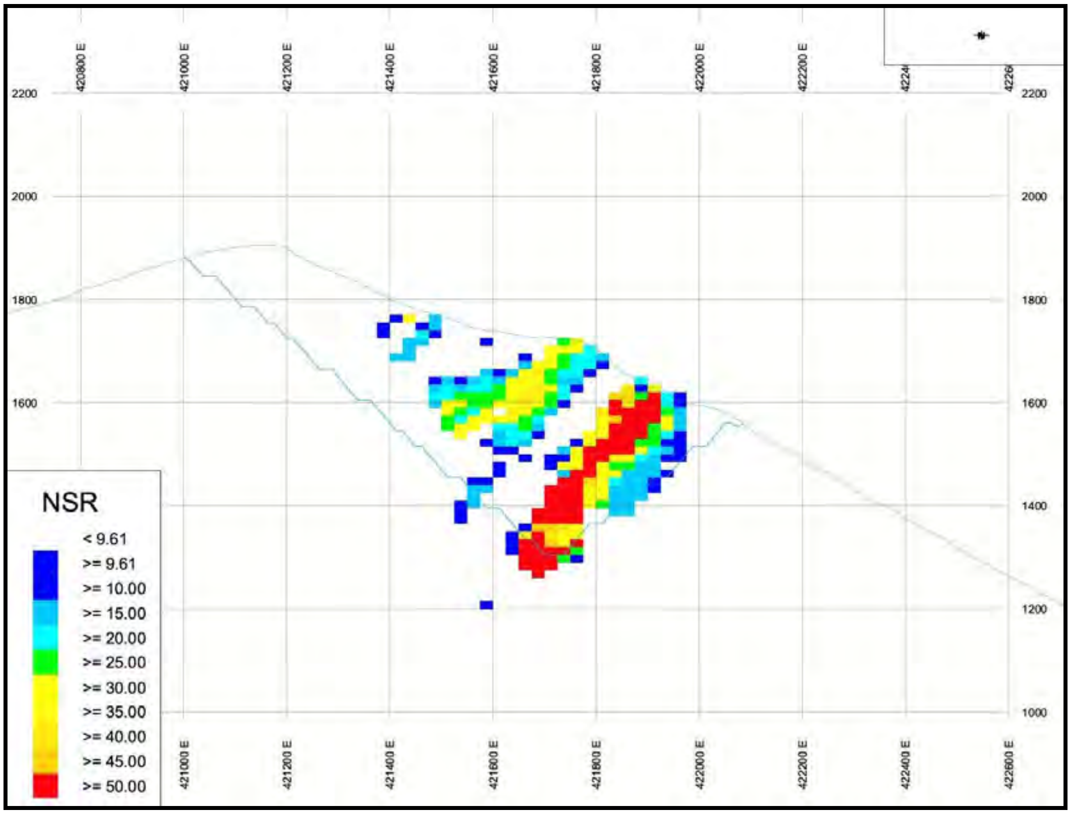


Figure 16.12 Kerr Economic Pit Limit – E-W Section at North 6258725, Viewed from the South



Opportunity – Inferred Pits and Resources

There is an opportunity to upgrade current Inferred material by infill drilling. After investing the required exploration expense and upgrading, this Inferred material would become part of the future reserve.

To analyze this potential, an additional set of pit optimization runs have been completed, allowing Inferred material to be given a value as well. Table 16.16 provides the results of these runs, and compares them to the original pits created with only Measured and Indicated material.

Table 16.16 Potential Increase due to Inferred Material

Pit	Au (M oz)	Cu (M lb)	Ag (M oz)	Mo (M lb)
Measured and Indicated LG Pit Limits – Insitu Metal				
Mitchell	20.7	3,697	96.1	130.8
Sulphurets	6.0	1,544	7.8	35.6
Kerr	1.9	2,444	9.2	0.0
Total	**28.5**	**7,685**	**113.1**	**166.4**
Measured, Indicated, and Inferred LG Pit Limits – Insitu Metal				
Mitchell	22.0	4,062	110.5	146.1
Sulphurets	8.6	2,094	14.7	47.2
Kerr	2.5	2,974	12.2	0.0
Total	**33.1**	**9,130**	**137.4**	**193.3**

Note that the Mitchell increase is due to Inferred material within the 60% shell, which is the Mitchell open pit limit truncated by the underground plan. The Sulphurets and Kerr increases are due to expanded pit limits and including Inferred within the pit limit used for this PFS.

FURTHER WORK TOWARD OPEN PIT DESIGN

The following items will need to be addressed in the ongoing engineering of the Project:

- optimization of the RSF waste management placement, updated pit sequencing and timing with revised detailed pit designs, and revised production schedules to reduce waste transport costs and potential environmental impact
- continued evaluation of trade-off studies comparing larger pit phases, creating more efficient and lower cost working areas, versus smaller pit phases, creating improved (smoother) production schedules.

16.1.6 DETAILED PIT DESIGNS

MMTS has completed PFS-level pit designs that demonstrate 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.

HAUL ROAD WIDTHS

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

- For dual lane traffic, a travel width of not less than three times the width of the widest haulage vehicle used on the road is required.
- 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 is required.
- Shoulder barriers should be at least three-quarters 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 34° slope, which is slightly less than the angle of repose. The width of the barrier must be added to the travel width to get the total road width.

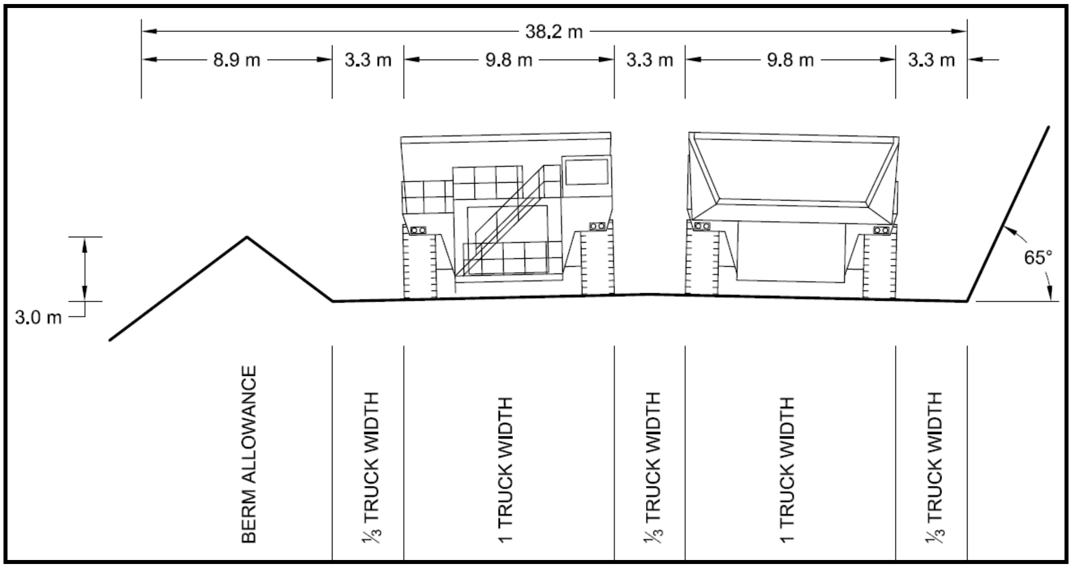
Ditches are included within the travel width allowance. For crowned haul roads, the width of this ditch allowance is 4.5 m. Ditches are not added to the in-pit highwall roads; there is adequate water drainage at the edge of the road between the crowned surface and lateral embankments, such as highwalls 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; it can also be driven on, if required, to avoid obstructions. In practice, specifically-designed excavated ditches in haul roads tend to be filled in by road grading and, when maintained as open ditches, can create a hazard if the wheel of a haul truck or light vehicle should happen to get caught in them. Avoiding the addition of ditch width to the three-truck travel width on the in-pit highwall roads can significantly reduce the pit waste stripping.

Based on a 363-t truck, the haul road design basis is as follows:

- largest vehicle overall width: 9.8 m
- double lane highwall haul road allowance: 38.2 m
- double lane external haul road allowance: 47.2 m
- single lane highwall haul road allowance: 28.5 m
- single lane external haul road allowance: 37.4 m.

Figure 16.13 to Figure 16.16 show typical road cross sections for haul roads.

Figure 16.13 Dual Lane Highwall Haul Road Cross Section



Note: Highwall face slopes will vary by geotechnical design criteria.

Figure16.14 Dual Lane External Haul Road Cross Section

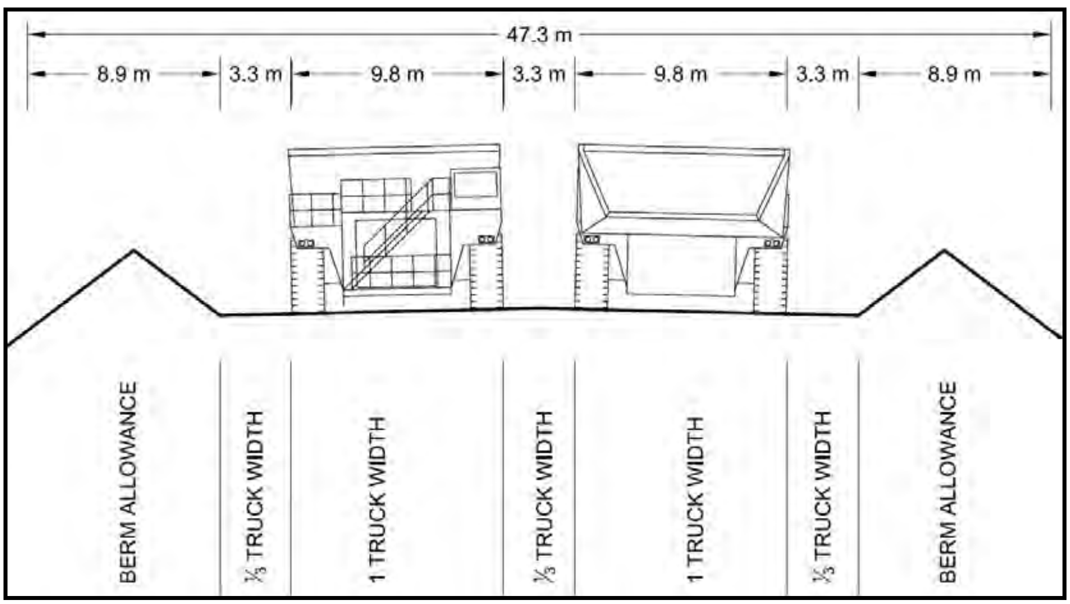
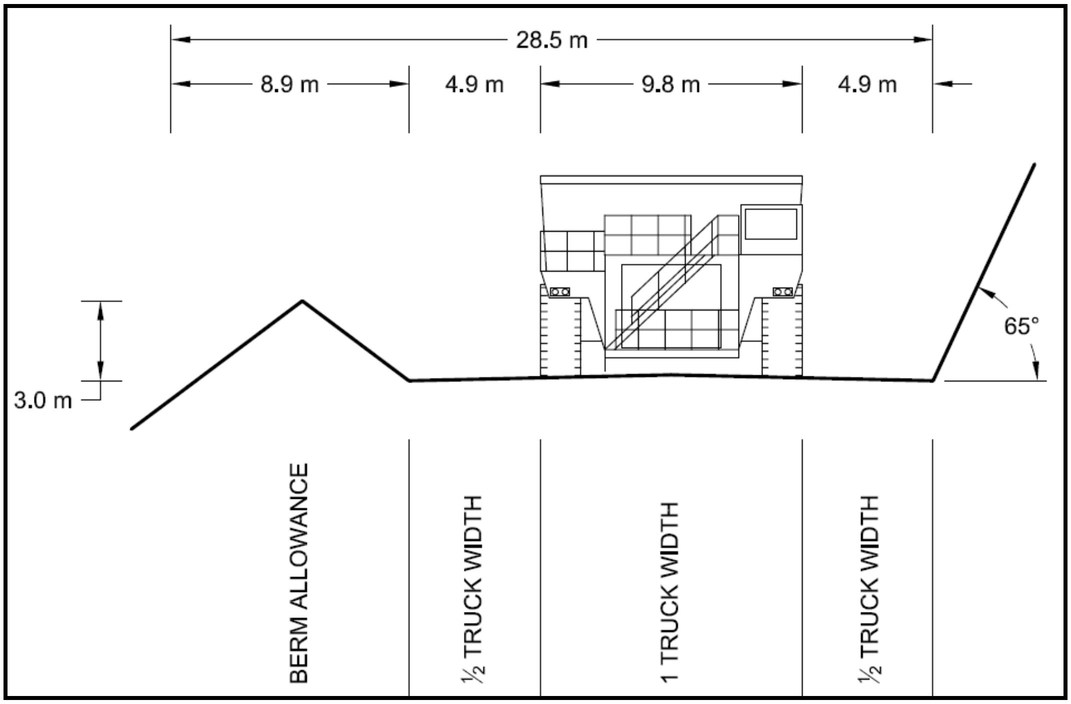
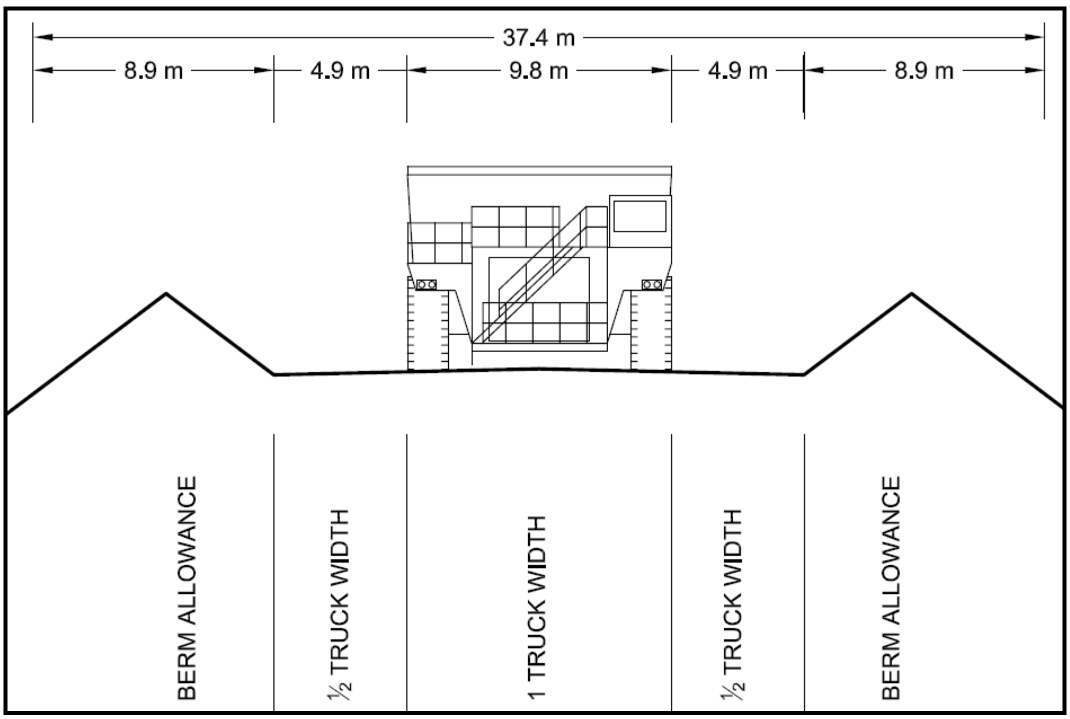


Figure 16.15 Single Lane Highwall Haul Road Cross Section



Note: Highwall face slopes will vary by geotechnical design criteria.

Figure 16.16 Single Lane External Haul Road Cross Section



The haul road widths and cut slope geometries outlined above may or may not be suitable for external haul roads placed on steep side slopes where significant rock cuts and fills are required. These need to be evaluated on a site-by-site basis to accommodate the geology and geometry of the road alignment. The design parameters outlined above (road width, cut slope angles, heights between benches, etc.) will need to be assessed with regard to the quality of the rock and the geologic structure in the area where the road exits the pit. This could vary depending on the location of the haul roads with respect to the pit design sectors, and may be influenced by the bench geometries in that area. Topographic constraints may result in steeper cut slopes being required to minimize the height of the cut due to steep topography in the area. Trade-offs between artificial support costs and excavation costs may be required to optimize the external haul road designs.

DESIGN STANDARDS

Detailed design parameters for pits and RSFs are provided by BGC and KCBL, respectively, according to their geotechnical testing and evaluations (Sections 16.4 and 18.1.6).

Minimum Mining Width

The design standards applied in the current pit designs are summarized in the design basis provided in Appendix E. 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 this study, the 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 the 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. Where bench width is sufficient, this material will be truck/shovel excavated as rehandle from lower benches. This technique has been used at other mountaintop mines; it allows for higher efficiencies with large mine equipment, and reduces costs 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 because they limit the safe by-passing of trucks and consequently lead to reduced productivity. One-way roads are, however, an appropriate option for low volume traffic flow or shorter-term operations. For this updated PFS, 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 last two benches of each pit bottom, assuming that the ramp is ore and will be removed upon retreat.

Road grades are designed at a maximum grade of 8%. Steeper roads (10%) 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 so 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.

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 roads 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 ore 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.

Variable Berm Width

Pit designs for KSM are designed honouring overall PSAs, a nominal bench face angle (60° to 70°) and variable safety berm widths with a minimum 8 m width. Due to the low overall PSAs and double benching between berms, 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 PSA.

Bench Height

The KSM pit designs are based on the digging reach of the large shovels (15 m operating bench) with double benching between highwall berms; therefore, the berms are separated vertically by 30 m. Single benching will be employed, if required, to maximize ore recovery and maintain the safety berm sequence as warranted.

LG Phase Selection

The LG pits discussed previously are used to evaluate alternatives for determining the economic pit limit and the optimal push-backs or phases before commencing detailed design work. LG pits provide a geometrical guide to detailed pit designs.

Among the details are the addition of roads and bench access, the removal of impractical mining areas with a width less than the minimum, and to ensure the pit slopes meet the detailed geotechnical recommendations.

The LG pit cases selected as the pit limits for the KSM mine areas discussed above are:

- Mitchell open-pit/underground pit limit: 60% price case LG pit
- Sulphurets economic pit limit: 90% price case LG pit
- Kerr economic pit limit: 75% price case LG pit.

There are smaller pit shells within the economic pit limits that have higher economic margins, due to their lower strip ratios or better grades than the full economic pit limit. Mining these pits as phases from higher to lower margins maximizes revenue and minimizes mining cost at the start of mining operations, which therefore shortens the project capital payback and improves the project cash flow. To increase this effect, a higher number of smaller push-backs have been balanced with the higher efficiencies and resultant lower unit mining costs of bigger mining areas from larger push-backs. The first phases (starter pits) have the greatest effect on capital pre-stripping requirements.

The selection of LG pit cases identified 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
- must have bench sizes large enough so the number of benches mined per year is reasonable (sinking rate)
- must be wide enough so the shovels can load the trucks efficiently
- must be able to provide at least 2 years of ore.

The LG pits generated for each area are examined to find the lowest LG price case that can sustain mining operations.

Waste from the starter pits is pre-stripped to expose ore grade material for plant start-up. This material can be used for some construction fills; however, it may be more cost effective to use borrow material from other areas, which will reduce costs if hauls are too long from the starter pit area. A second cost effective alternative for construction material is to borrow the material from the upper benches of future pit phases.

The description of the detailed pit designs and phases in this section uses the following naming conventions:

- The letters M, S, and K signify Mitchell, Sulphurets, and Kerr, respectively.

- The middle digit signifies the design series. In this report, the fifth pit design series is used.
- The final digit signifies the pit phase number (Kerr has only one phase).
- A suffix of 'i' indicates that the drawing reserve tonnage for the phase is incremental from the previous phase. If there is no 'i' specified, it is cumulative up to the phase indicated.

Mitchell Pits

Where possible, phase sequencing should start at one side of the ultimate pit and expand in one direction. This sequencing is more efficient for operations where blasts from the subsequent phase only bury access to lower benches on one side at a time. It also allows the final ramp to be established on one side of the ultimate pit. However, the Mitchell pit phases are designed to alternate from the north and south sides of the Mitchell Valley (a two-sided expansion) because the upper benches of the Mitchell pit are mostly waste on the north and south walls. Breaking the push-back designs 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 highwalls in some of the intermediate phases to enable access to the upper benches of subsequent phases.

Where possible, in order to balance the waste hauls and keep upper elevation waste going to upper elevation RSF platforms, the highwall waste is brought out of the pit using external side hill roads directly off the south benches.

The Mitchell pit phases have been designed to mine vertically through the Snowfield Landslide on the southeast side of the pit and not undermine it, as directed by BGC.

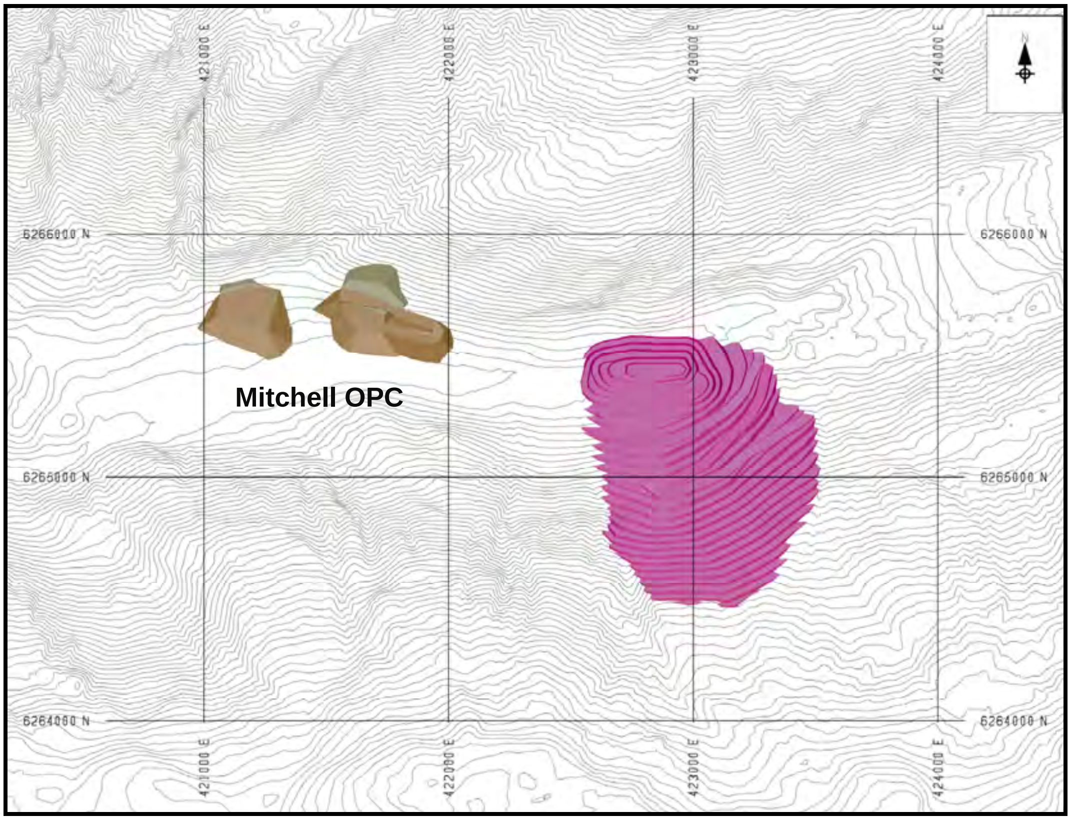
Mitchell Phase M681

Mitchell phase M681 is a starter phase that mines the south side of the Mitchell Valley. Mining begins at a bench elevation of 1370 m on the south valley wall. The south side is initially accessed from the Sulphurets Ridge Crusher Access Route. Initial narrow benches are established with dozers to an elevation of 1285 m by dumping the waste material northwards.

Subsequent to the dozer and cast mining of the narrow benches, M681 pre-stripped waste is then used to fill out a road to the Mitchell Ore Processing Complex (Mitchell OPC). A detailed description of the M681 pre-strip is in Appendix E.

M681 is mined down to a bottom pit elevation of 690 m. The haul road from the pit bottom reaches the surface at an elevation of 783 m. Waste is hauled to the valley bottom to build part of the base for the ultimate Mitchell RSF. A plan view of the M681 pit with the Mitchell OPC pad for orientation is provided in Figure 16.17.

Figure 16.17 Plan View of Mitchell Starter Pit M681 with Mitchell OPC Fills

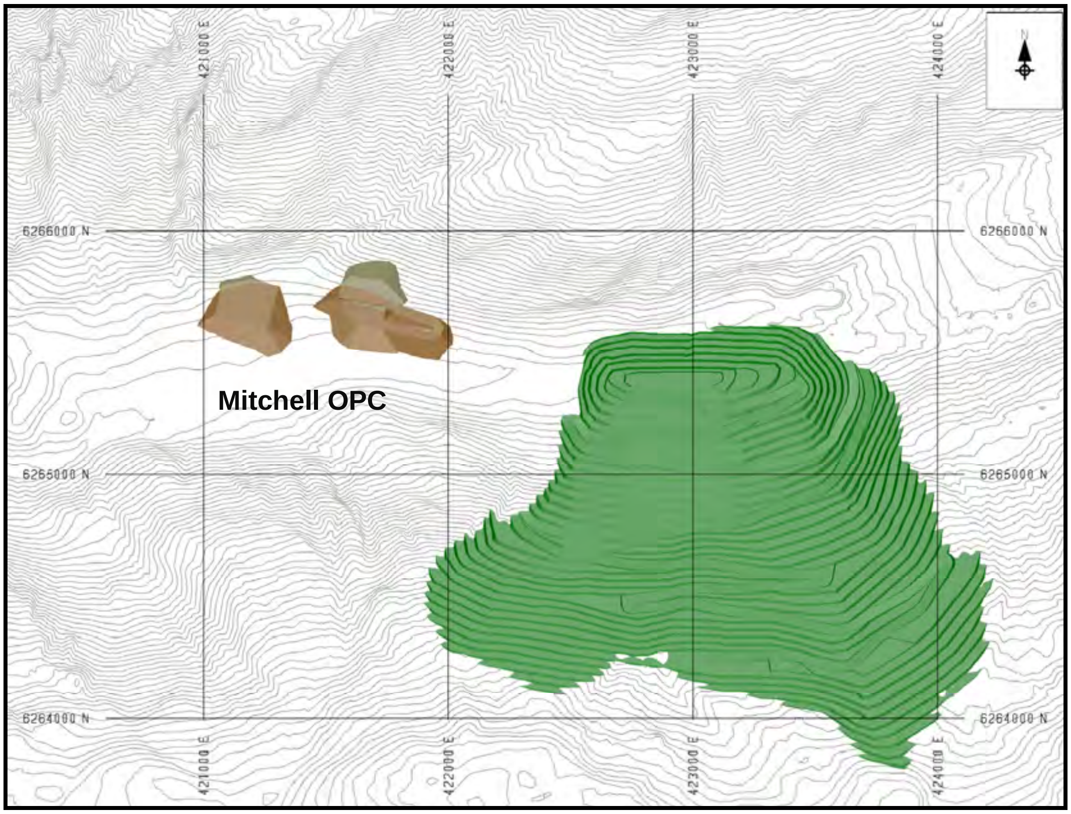


Mitchell M682

Mitchell phase M682 is an expansion of M681on the south side of the valley, starting at a bench elevation of 1680 m with a pit bottom elevation of 630 m. The haul road from the pit bottom exits the pit at an elevation of 795 m. M682 can be mined independently of M681 down to the 1370 m elevation. M682 is incremental to M681 below the 1370 m elevation. A plan view of M682 is provided in Figure 16.18. Below the 1545 m elevation, M682 is accessed by an external road built with M681 waste. Above 1425 m elevation, the M682 pit is mined to the ultimate pit shape.

Production waste is hauled to the Mitchell RSF.

Figure 16.18 Plan View of Mitchell Pit M682 with Mitchell OPC Fills

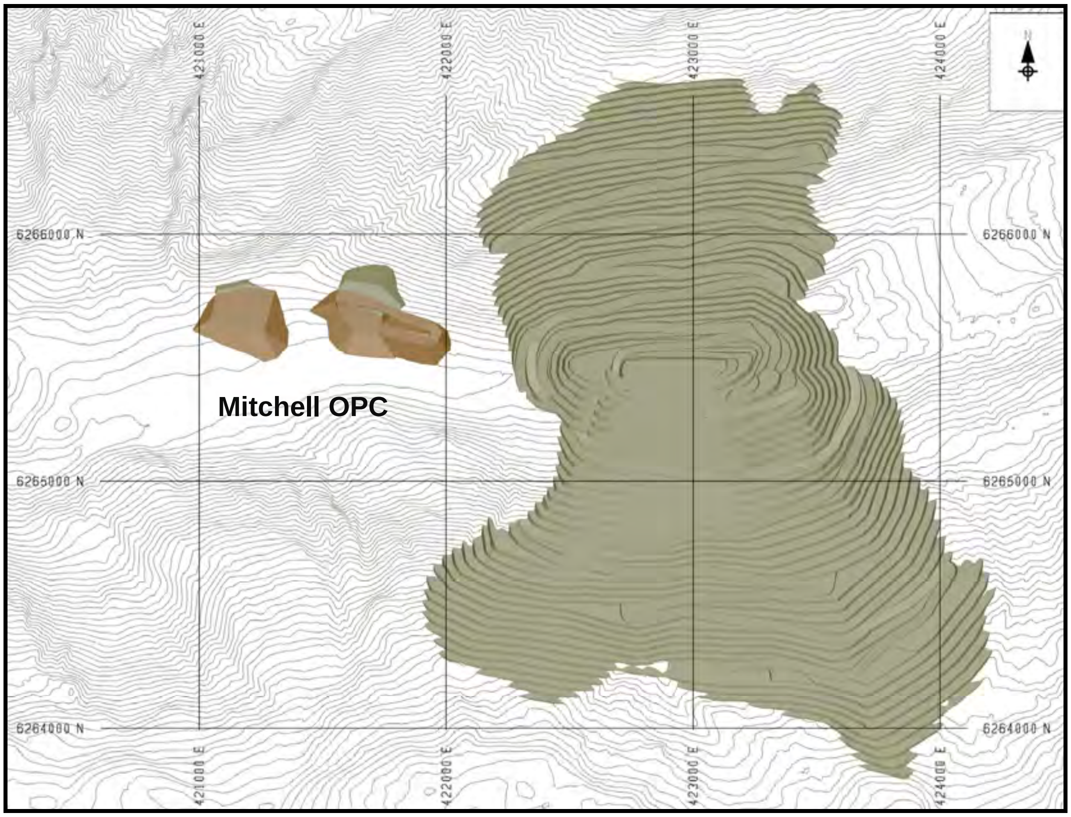


Mitchell Phase M683

Mitchell phase M683 is the first expansion on the north side of the valley starting at a bench elevation of 1490 m with a pit bottom elevation of 585 m. M683 is incremental to M682 below 890 m elevation. The haul road from the pit bottom reaches the surface at an elevation of 770 m.

M683 is accessed by an external road and waste is hauled to the Mitchell RSF using an external haul road below elevation 1420 m. A plan view of M683 is provided in Figure 16.19.

Figure 16.19 Plan View of Mitchell Pit M683 with Mitchell OPC Fills

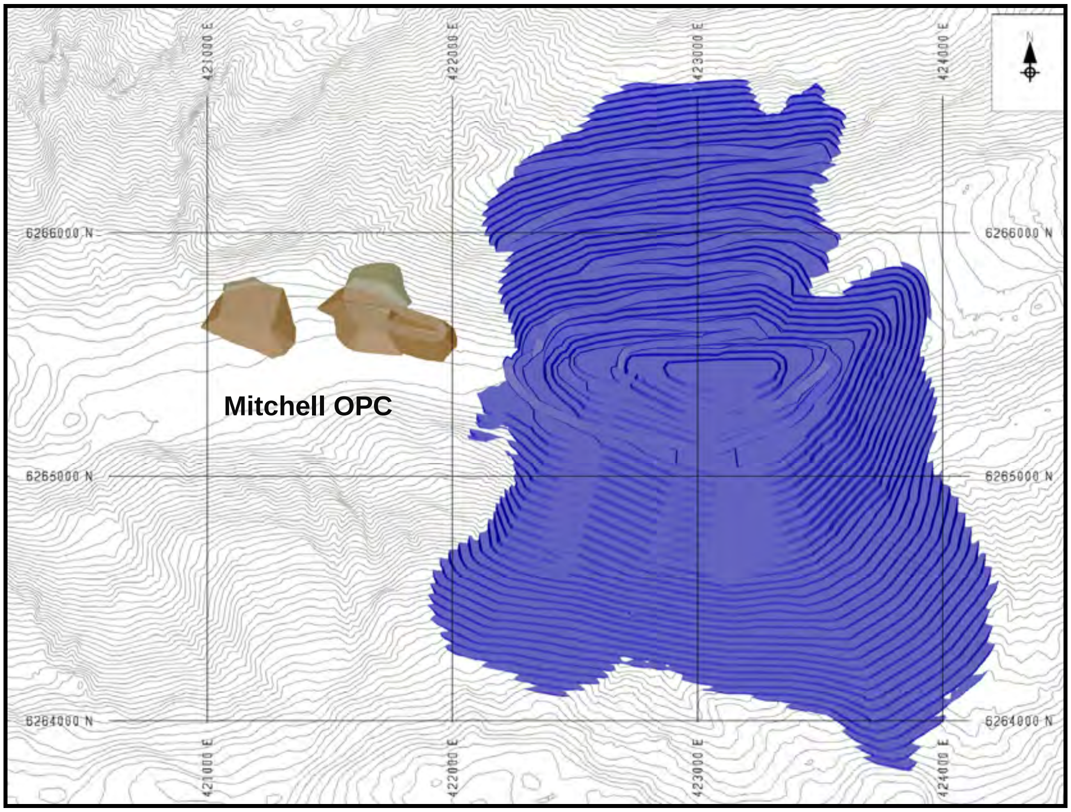


Mitchell Phase M684

Mitchell phase M684 is an expansion on the south side of the valley. M684 is mined from an elevation of 1425 m and the pit bottom is at 555 m elevation. M684 is incremental to M683. The haul road from the pit bottom reaches the surface at an elevation of 775 m.

The M684 pit is accessed by the haul road left in the M682 highwall. Waste is hauled along the terraces and then to the Mitchell or McTagg RSF. A plan view of M684 is provided in Figure 16.20.

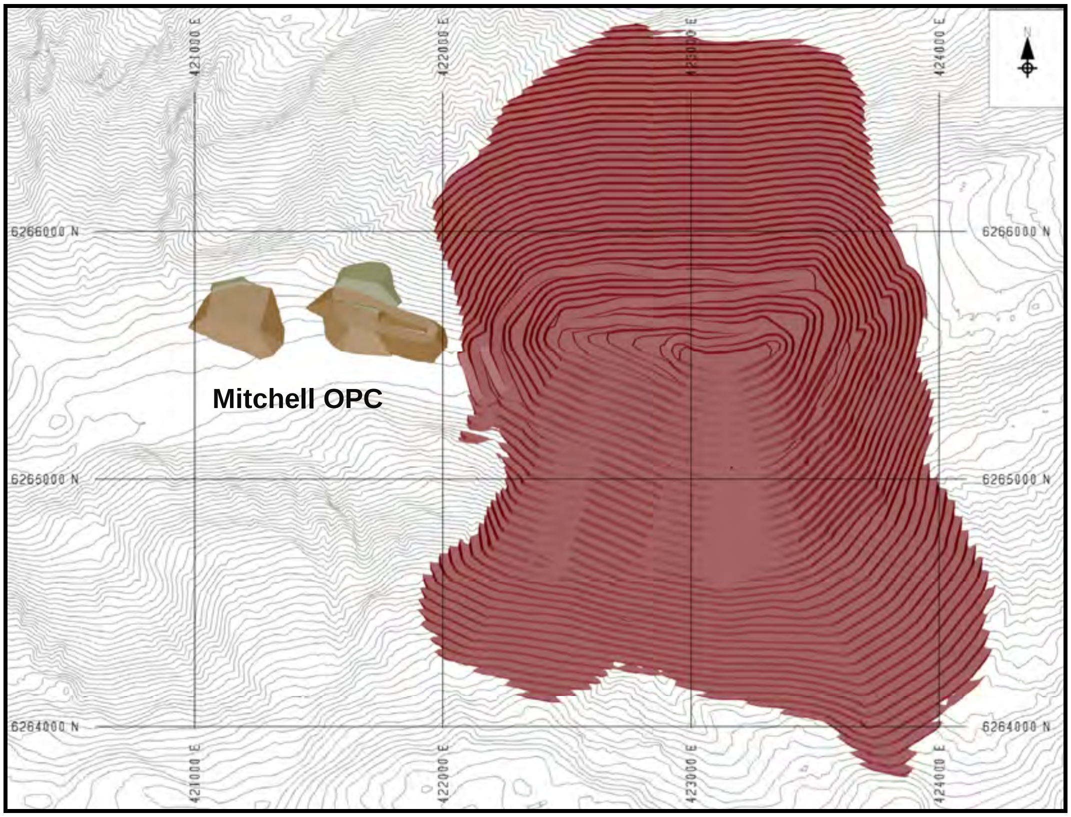
Figure 16.20 Plan View of Mitchell Pit M684 with Mitchell OPC Fills



Mitchell Ultimate Phase M685

Mitchell phase M685 is incremental to M684 and mines the north pit wall to the ultimate pit limit from the 1635 m elevation down to the pit bottom at 405 m elevation. The haul road from the pit bottom reaches the surface at an elevation of 780 m. Upper waste is dozed to 1515 m elevation. Below1515 m elevation, highwall waste is hauled via an external road to a 1270 m elevation ramp left in the M683 highwall. The waste is hauled to the Mitchell or McTagg RSF. A plan view of M655i is provided in Figure 16.21.

Figure 16.21 Plan View of Ultimate Mitchell Pit M685 with Mitchell OPC Fills



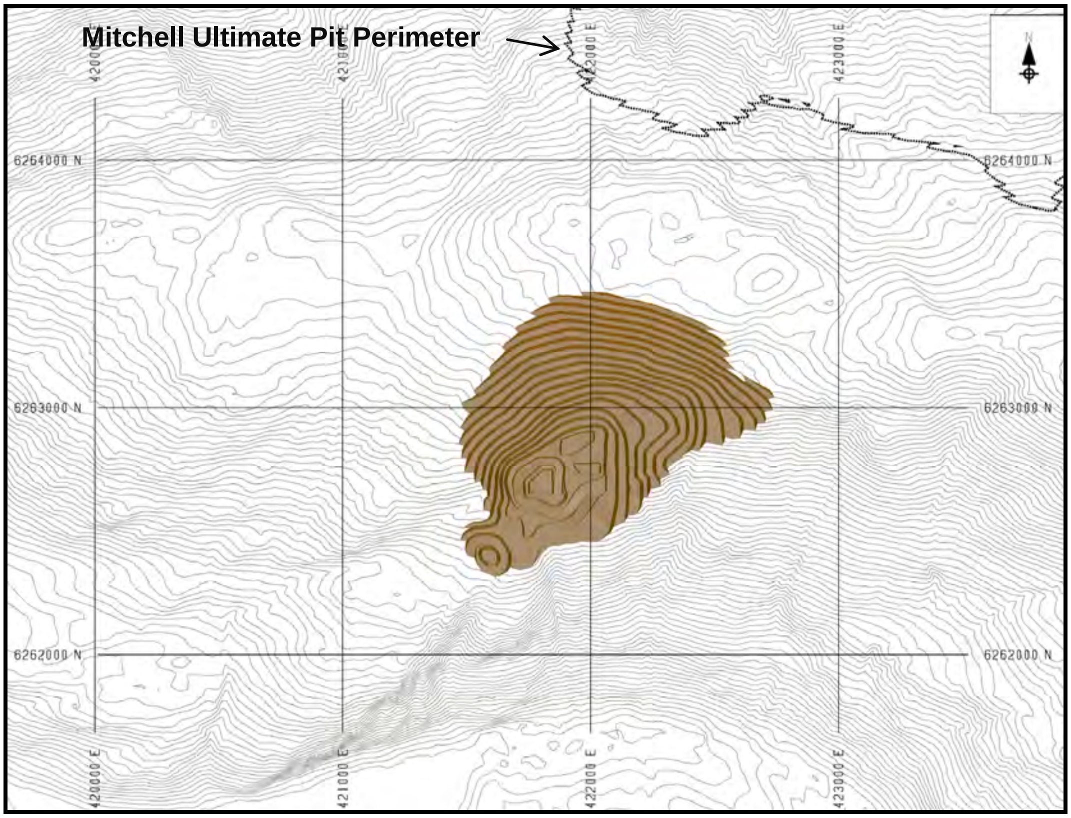
Sulphurets Pits

The mine plan for the Sulphurets area involves two mining phases, which are designed using the LG economic pit limit as the ultimate pit limit guide.

Sulphurets Phase S691

The first Sulphurets phase is designed as a starter phase, and will produce ore that is blended with Mitchell ore at the beginning of the mine life. It is accessed via the Sulphurets access road, which is built during the pre-production period. NAG waste from S691 will be placed on the Sulphurets RSF and PAG waste is hauled to the Mitchell RSF. The S691 pit crest is at an elevation of 1710 m and the pit bottom is at an elevation of 1125 m. A plan view of the S691 pit is shown in Figure 16.22 with the Mitchell ultimate pit perimeter shown for orientation.

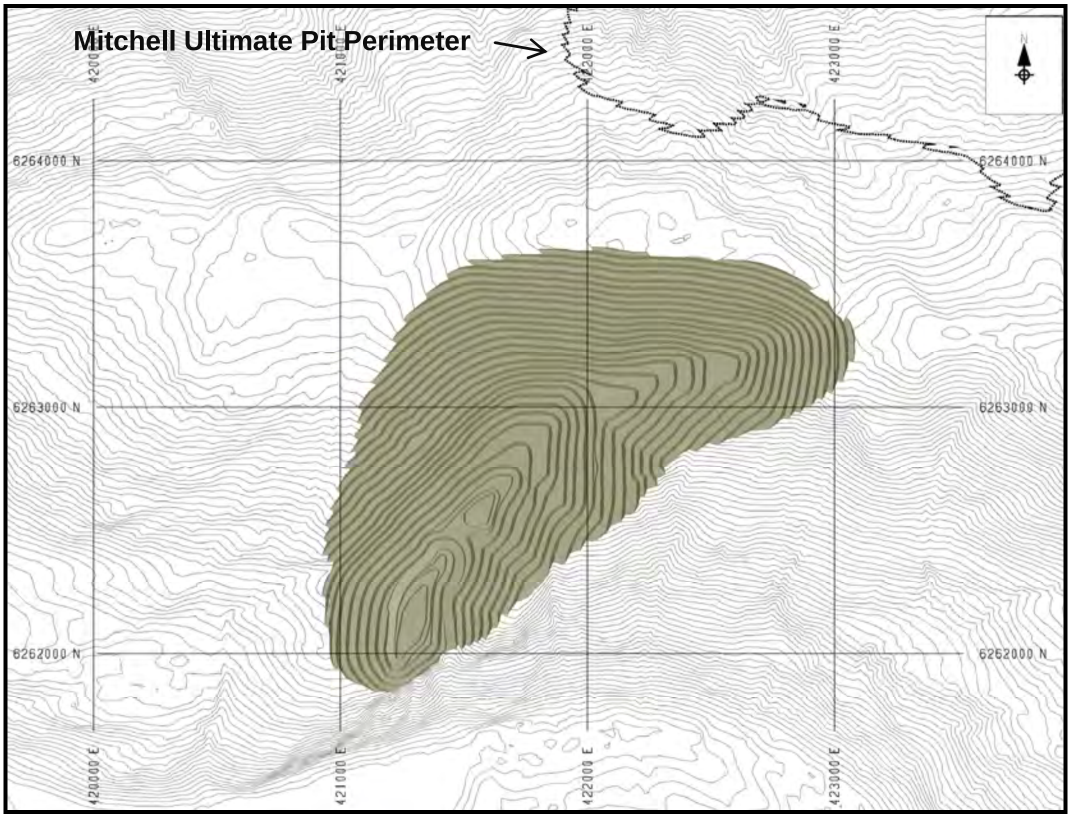
Figure 16.22 Plan View of Sulphurets Pit S691



Sulphurets Ultimate Phase S692

Sulphurets phase S692 is incremental to S691. The pit crest is at 1725 m with a pit bottom at 825 m. Waste exits the pit via external roads connected to the Sulphurets access road. Waste will be placed on the Mitchell RSF and McTagg RSF. Ore is hauled downhill to a pit rim primary crusher on the south slope of Sulphurets Ridge, and then conveyed north through the SMCT into the Mitchell Valley where the coarse ore stockpile is located. A plan view of the S692 pit is shown in Figure 16.23 with the Mitchell ultimate pit perimeter shown for orientation.

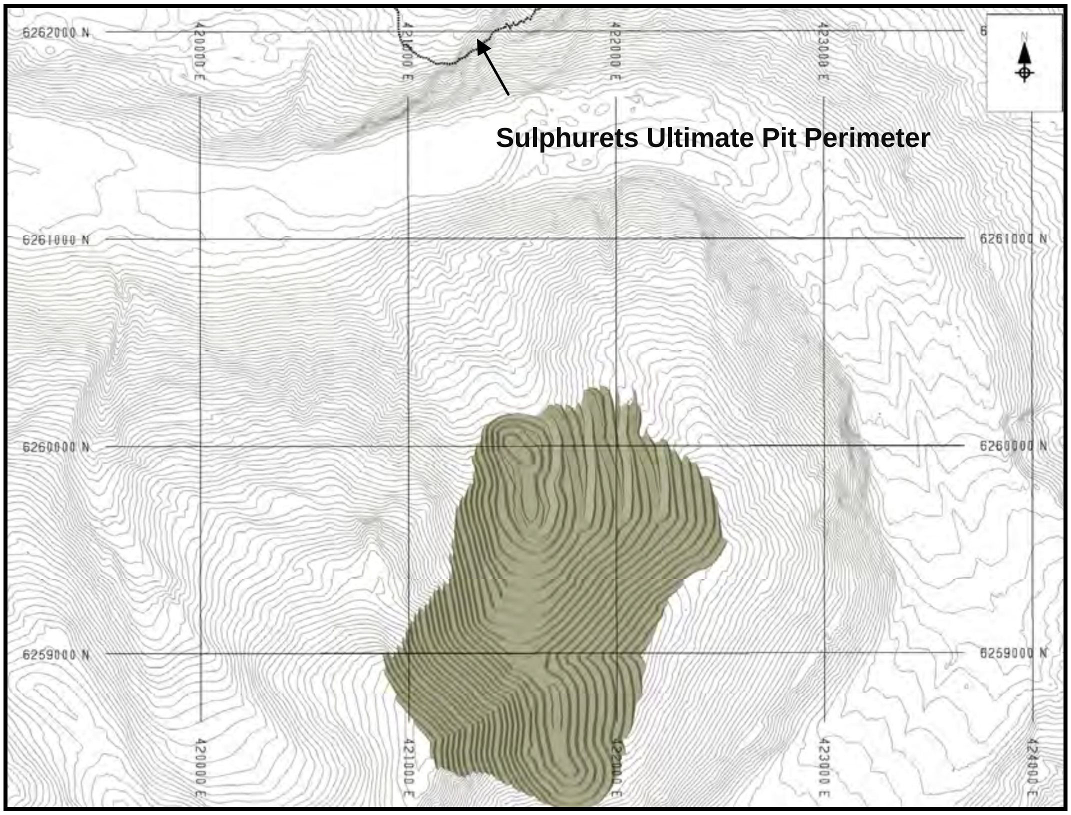
Figure 16.23 Plan View of Ultimate Sulphurets Pit S692



Kerr Pit K691

The Kerr pit is a single phase pit designed using the LG economic pit limit as a guide. The crest of the Kerr ultimate pit is at 1905 m with pit bottoms at 1305 m and 960 m. All ore and waste is hauled to a primary crusher on the east side of the pit. It is then conveyed to the Mitchell Valley using a rope conveyor, a tunnel conveyor (through the SMCT), and a second rope conveyor. Initial access to the Kerr pit is established with a service road built from the bottom of Sulphurets Valley to the east side of the Kerr pit (where the crusher will be located) at the 1460 m elevation. Access to the highest benches of Kerr will be established with a small service road, and the upper benches will be dozed down to approximately 1800 m where haul truck access can be established to the crusher. A plan view of the Kerr K691 pit is shown in Figure 16.24 with the Sulphurets ultimate pit perimeter included for orientation.

Figure 16.24 Plan View of Kerr Ultimate Pit K691



Combined Pit Areas

All the KSM ultimate pit phases are combined in Figure 16.25 and Figure 16.26.

The detailed pit designs are considered adequate for this prefeasibility design stage. Future feasibility design work will attempt to optimize the strip ratio change between phases for smoother equipment fleet size requirements.

BGC has checked the pit designs and verified that they conform to the design parameters outlined in Section 16.2.

Figure 16.25 Plan View of All Ultimate Pit Phases with Mitchell OPC Fills

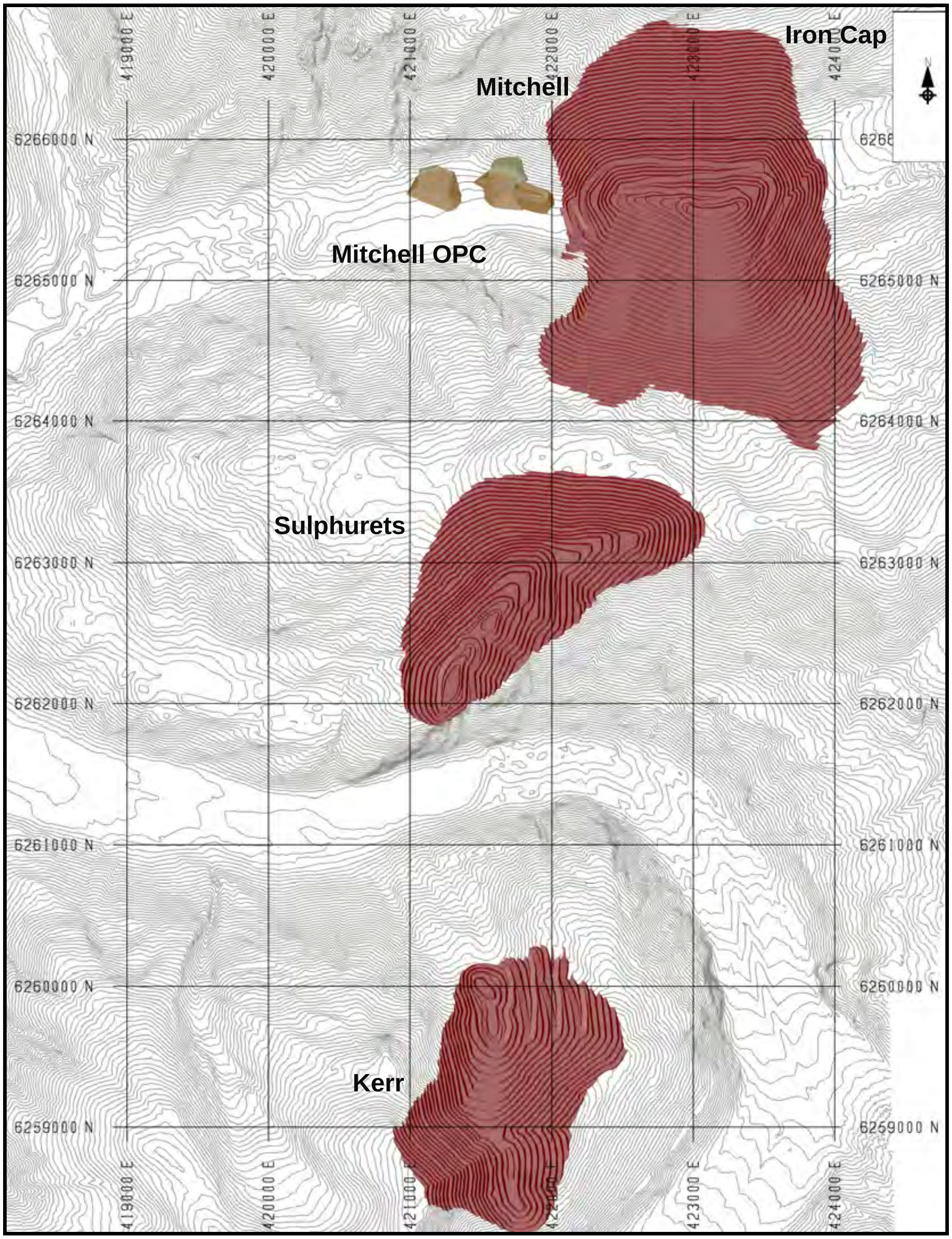
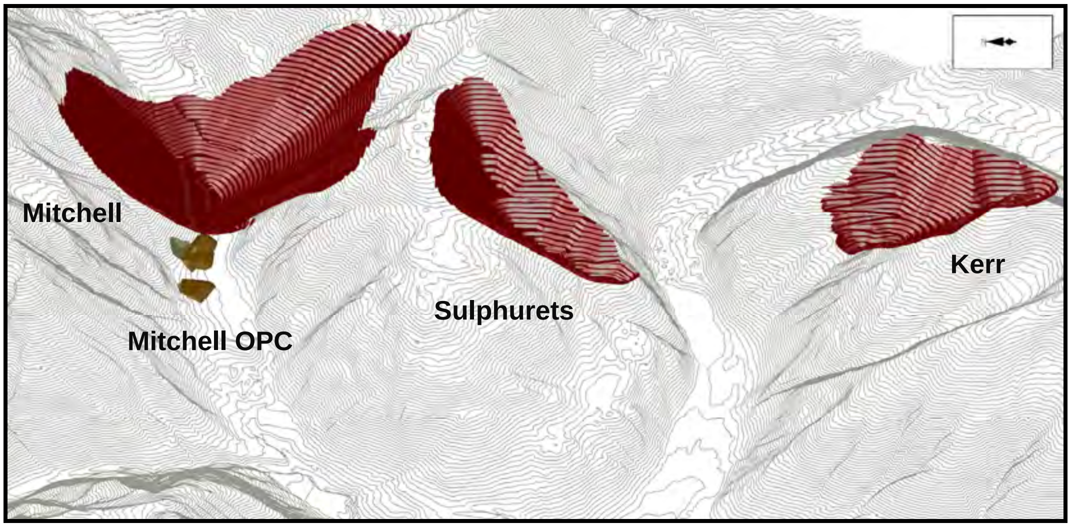


Figure 16.26 Orthographic View from the East of all Ultimate Pit Phases



16.1.7 *Mine Plan*

LOM Production Schedule

The open pit mine production schedule is developed with MS-SP, a comprehensive long-range schedule optimization 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. The production schedule includes:

- tonnes and grade mined by period, broken down by ore and waste material type, bench, and mining phase
- truck and shovel requirements by period in number of units and operating hours
- tonnes transported by period to different destinations (mill, stockpiles, and waste dumps).

The underground mining production schedule is generated based on the development requirements, the size, and capacity of the individual Mitchell and Iron Cap block caves and then integrated into the total property production schedule. The ore production from each is inserted where it provides the best contribution to the project economics. The open pit ore targets are then adjusted to meet the mill capacity. In the later years of the schedule, the open pit reserves are depleted and mill production is limited to the capacity of production from the underground only.

In the mine schedule, "Time 0" refers to the mill start date; full mill feed production capacity is expected in Year 2. The production schedule specifies:

- pioneering: Years -6, -5, -4
- pre-production: Year -3,-2,-1
- LOM operations: Year 1 onward.

Open Pit 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 that are currently being used for large hard rock open pit mines are the 100-t bucket class shovel matched with the 363-t truck. These sizes of units are proven in operating mines around the world. Diesel hydraulic shovels (85 t bucket class) are added to the fleet when a more mobile loading unit is needed. Suitable drill sizes (311 mm hole size) are chosen to match this size of truck/shovel fleet. The following performance and costs are estimated based on the use of this large-scale mining equipment.

Productivities of the selected equipment include shovel loading times and truck haul cycle estimates for multiple pit-to-destination combinations.

Schedule Criteria

The production schedule setup includes a number of parameters. Haul times include an allowance for operator efficiency. Haul truck speed is limited to a maximum of 50 km/h. Haul truck operating efficiencies are included in the design basis (Appendix E). Truck availability assumptions for MS-SP are shown in Figure 16.27.

Figure 16.27 MS-SP Haul Truck Fleet Availability Assumptions

Up to hours	%Availability
10000	89
20000	88
30000	87
40000	86
50000	85
60000	84
70000	83

Shovel availability assumptions for MS-SP are listed in Figure 16.28.

Figure 16.28 MS-SP Shovel Availability Assumptions

Up to hours	%Availability
10000	89
20000	88
30000	87
40000	86
50000	85
60000	84
70000	83
80000	82
90000	81

Load times for the shovels include an allowance for operator efficiency. Details for the load times are included in Appendix E.

In order to optimize the project NPV, grade bins have been specified (based on NSR block values); the MS-SP optimizer develops a COG strategy to increase the project NPV by stockpiling lower grade material for processing later in the LOM schedule. This increases mill head grades and therefore revenues early in the production schedule. Seven grade bins were used for the schedule optimization software to optimize head grades. It is assumed that blast hole assays will be used for ore COGs and for a COG strategy, which is typical of bulk mining for this kind of deposit.

Mining precedence is required to specify the mining order of the pit phases in the production schedule based on the 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, they are dependant at the pit bottoms. The KSM precedence's are simplified as shown in Table 16.17.

Table 16.17 Pit Precedence for Scheduling

Phase A ID	Constraint	Phase B ID
M652i	After	M651
M653i	After	M652i
M654i	After	M653i
M655i	After	M654i
S652i	After	S651

Each pit area is scheduled in MS-SP independently.

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 estimated operating and capital costs, process recoveries, and metal prices.

MS-SP uses 355 mine operating days scheduled per year and 21 operating hours per day (the design basis is provided in Appendix E). The mine operating days are based on a weather study included in Appendix E. Allowance has been made for days where the cumulative effect of severe snow storms or poor visibility requires the mine to completely shut down.

An annual mill feed of 47,450 kt/a is targeted based on an average throughput of 130,000 t/d.

Haul and return times are estimated for waste and ore to estimated destinations. 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 E.
- A dump and manoeuvre time of 1.5 minutes.

Haul truck utilization has been adjusted to account for recovery from severe storm conditions as simulated in the weather study included in Appendix E. The haul, return, dump, and manoeuvre times are added and included in the cycle times in MS-SP. The linear interpolation of truck cycle times is carried out for all phases from all benches to all material destinations.

Shovel productivity includes:

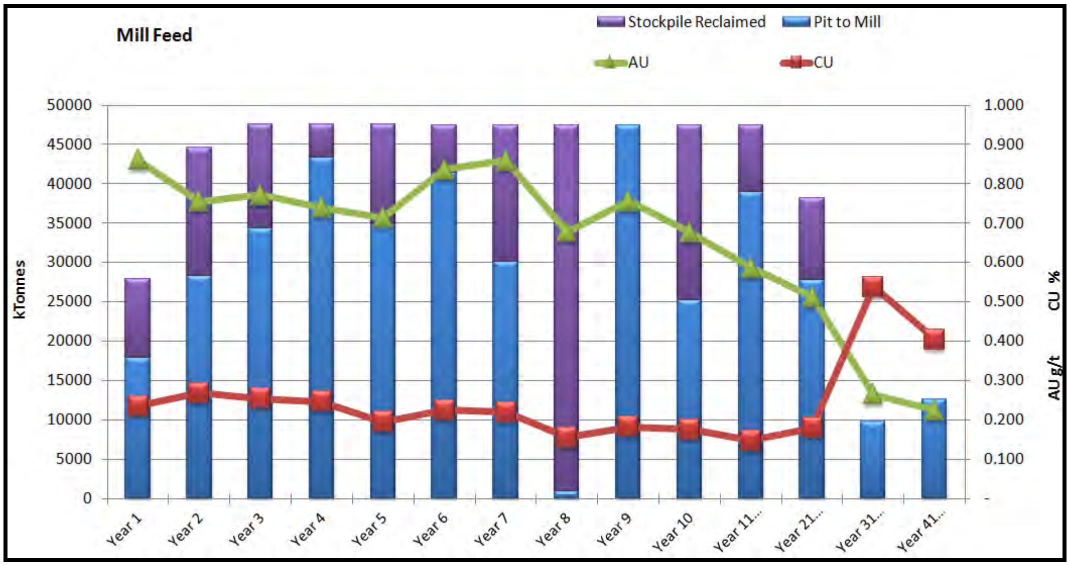
- 35 second cycle time per pass, and 10 second spot-and-wait time per load for the shovels (hydraulic and cable)
- 84% job efficiency
- 80% operating efficiency
- 95% utilization efficiency.

Cut-off Grade Optimization

Typically, the ore grade can be increased by hauling low and mid-grade classes to stockpiles. The ore grade is maximized and this effectively increases the revenue per tonne milled early in the schedule. The lower grade stockpiled material is then milled at the end of the production schedule. However, stockpiling also results in increased total material mined and the mining cost per tonne milled in the relevant time period also increases. Additionally, oxidation can cause significant metallurgical recovery loss in the stockpile. At some point, the cost of mining more material as a result of increased stockpiling and with the metallurgical recovery loss will exceed the incremental revenue from the higher grade milled. A variable COG strategy has been applied for the KSM production schedule to maximize NPV, minimize stockpiling, and assist in haul fleet smoothing.

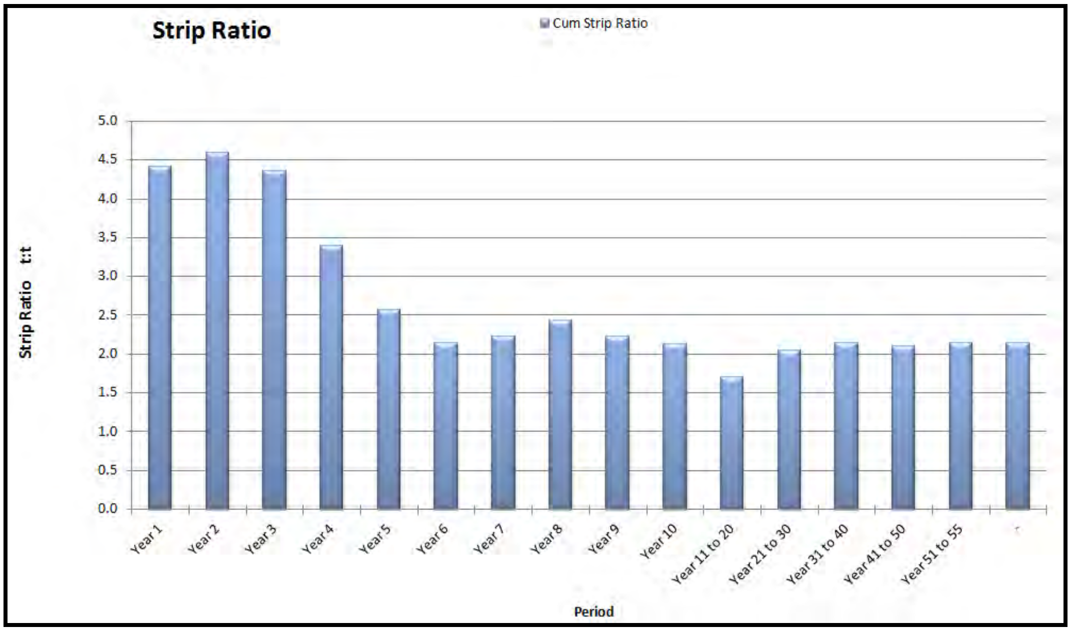
Figure 16.29 shows the LOM open pit ore production schedule and illustrates that significant stockpile reclaim is required throughout the mine life to even out strip ratio during the pre-stripping of the Mitchell phases.

Figure 16.29 Open Pit ROM Mill Feed Sources and Mill Head Grades for Feed Cu and Au



The cumulative strip ratio by period, shown in Figure 16.30, is calculated from total waste mined, which includes rehandle and construction borrows. To smooth out the truck fleet size over the LOM, the scheduled strip ratio is decreased in periods where long hauls are required and, in periods where short hauls are required, the scheduled strip ratio is increased. In future detailed stages of engineering, further smoothing of the schedule will be completed by changing the pit phase sizes and by phasing the RSF destinations to even out the strip ratio, which will result in more efficient utilization of the shovels.

Figure 16.30 Cumulative Strip Ratio



Rock Storage Facilities

Design Parameters

Mined waste rock in the KSM mine plan is placed in RSFs in as close proximity to the mining areas as possible. The mine production schedule tracks rock types based on acid base accounting (ABA) quality for water modeling purposes. All RSF designs assume a natural angle of repose of 37°. A 20% swell factor is applied to in situ volumes to calculate the loose volume requiring placement.

Further details on the RSF design are available in Section 18.1.6 and the design basis memorandum in Appendix E.

Construction Methods

Several different construction methods will be used for waste placement: top-down, bottom-up, and wraparounds. Top-down placement involves truck-dumping the material from the crest down to a platform or the topography below. With this method, the platform height is restricted to approximately 300 m. Bottom-up placement involves driving the truck to the bottom of the platform and placing the material in lifts (approximately 30-50 m high, or less if geotechnically required), and constructing the RSF to final limits from the bottom working upwards. Wraparounds are smaller top-down-type RSFs that are built onto the face of an existing RSF, creating a series of terraces. These are used to facilitate intermediate haul roads and lower the overall slope angle of high dumps, which may be required for final closure and, if re-sloping is necessary, it will reduce the re-sloping costs.

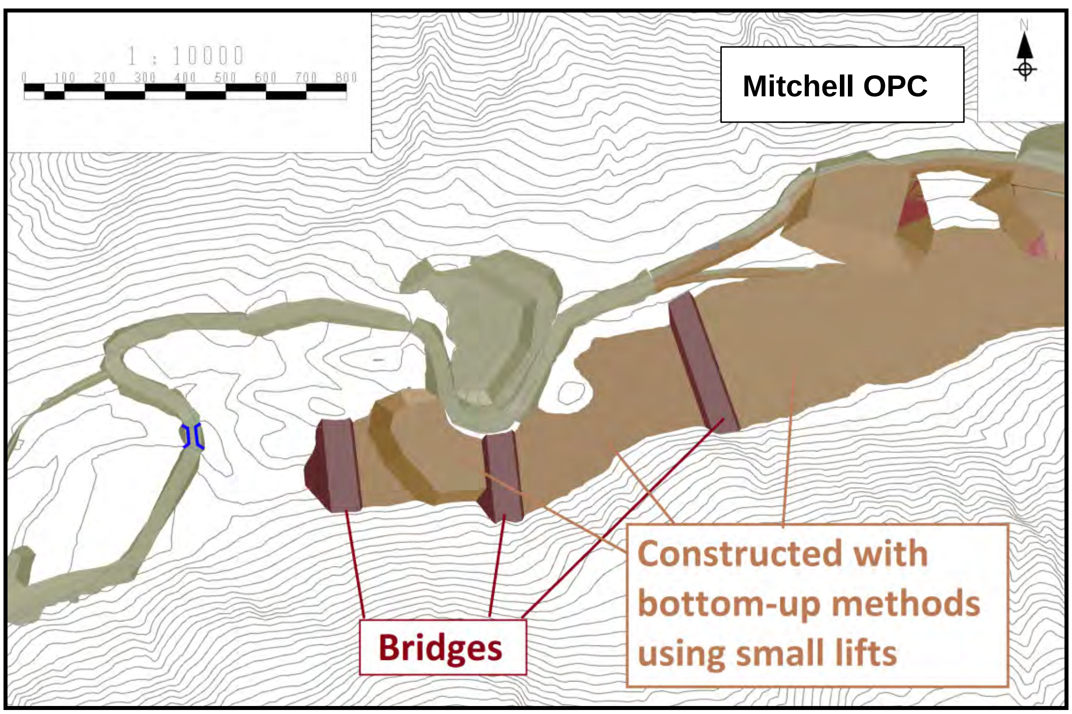
Foundation Preparation

If required, design work for RSF foundation preparation will be performed at the feasibility-level design stage. At this PFS design stage, a cost allowance for removal of merchantable timber, where required, is included. The recovery of till from the foundation area as reclamation material for capping the final dumps from the McTagg and Mitchell RSF areas is not practical due to safety concerns and storage limitations.

The waste in the valley bottoms is planned to initially be placed in low height lifts across the narrow valley floors to confine and consolidate weaker foundation material before higher lifts are placed. To establish these lifts, suitable valley crossings will be located in narrow and suitable rock foundations, and a bridge of rock fill will be placed progressing from one side of the valley to the other. If required, loose tills and clays at the toe of the bridge are removed with a backhoe and placed on the upstream side of the bridge. Once the bridge is keyed in all the way across the valley, lifts of mine rock can be placed on the upstream side and the loose tills and clays under the small lift will be constrained on the downstream side by the bridge. These small lifts can be gradually built on top of each other in successive layers, until the total thickness of the platform is enough to create stability across the entire surface. As the lift height increases, the foundations are consolidated.

This concept is illustrated in Figure 16.31.

Figure 16.31 Plan View of Early RSF Construction Methods

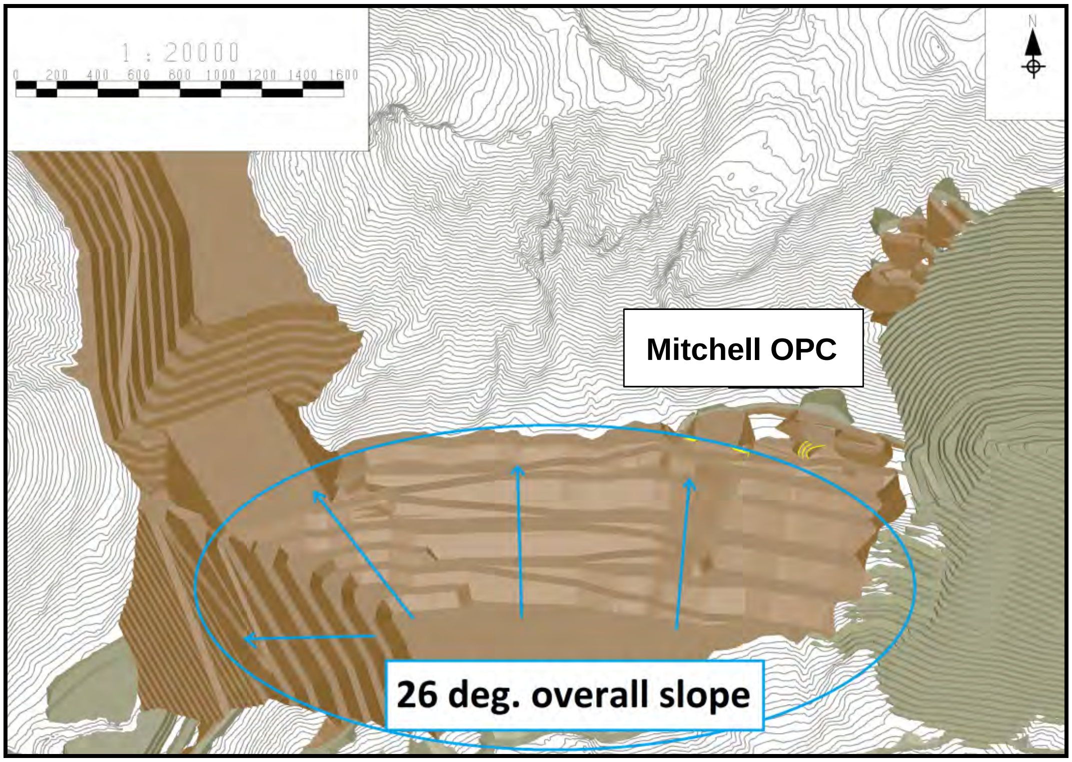


RSF Construction – Mitchell

All large platform faces on the south slope of the Mitchell Valley are constructed with wraparound methodology and a 37° face angle. A terrace is created approximately every 100 m vertically using wraparounds to allow for a 26° overall slope face angle. The resultant 26° overall face slope allows for re-sloping, if necessary. MMTS has also made allowance for re-sloping costs. The terraced RSFs (90-105 m lift height) in the Mitchell Valley are shown in Figure 16.32.

The terraces also provide haul road access out to the Mitchell valley RSFs. These terraces provide haulage corridors, allowing the higher material from the south side of Mitchell pit to be taken out to the western end of the RSF at a flatter haul profile, thus reducing the haulage costs. Finally, the terraces also provide ore access from the pit to the Mitchell OPC as required.

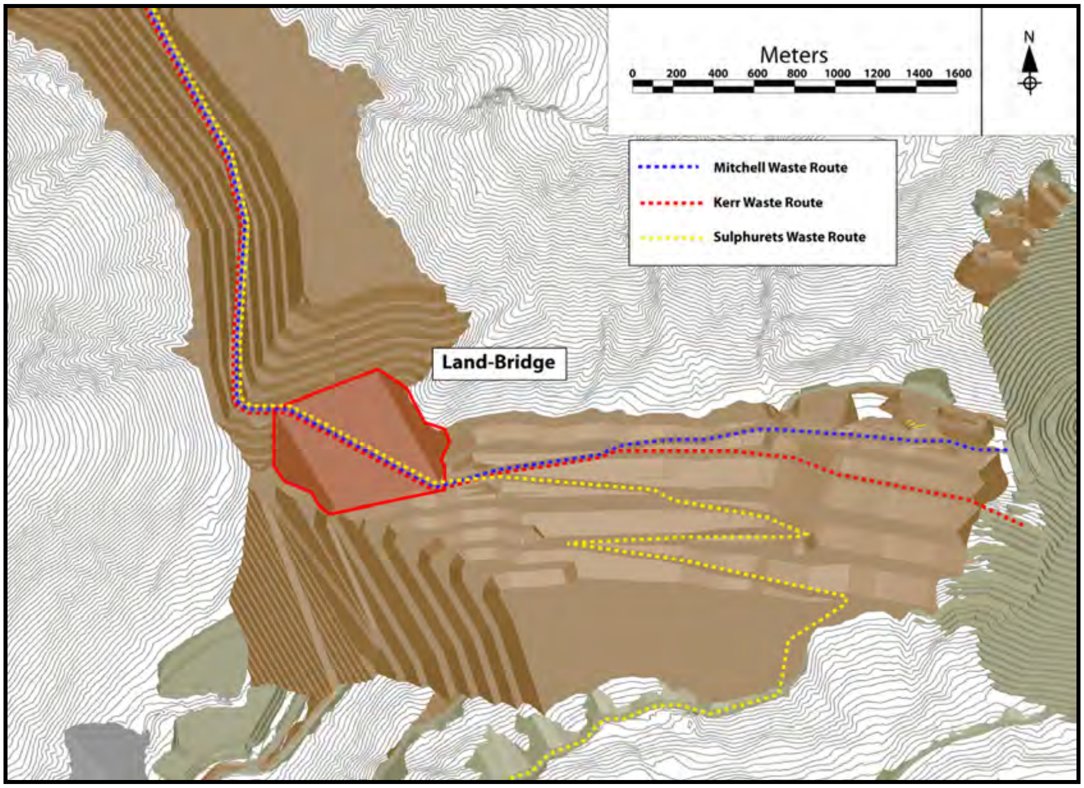
Figure 16.32 Plan View Showing Overall Slopes for South Mitchell Valley RSF



RSF Construction – McTagg

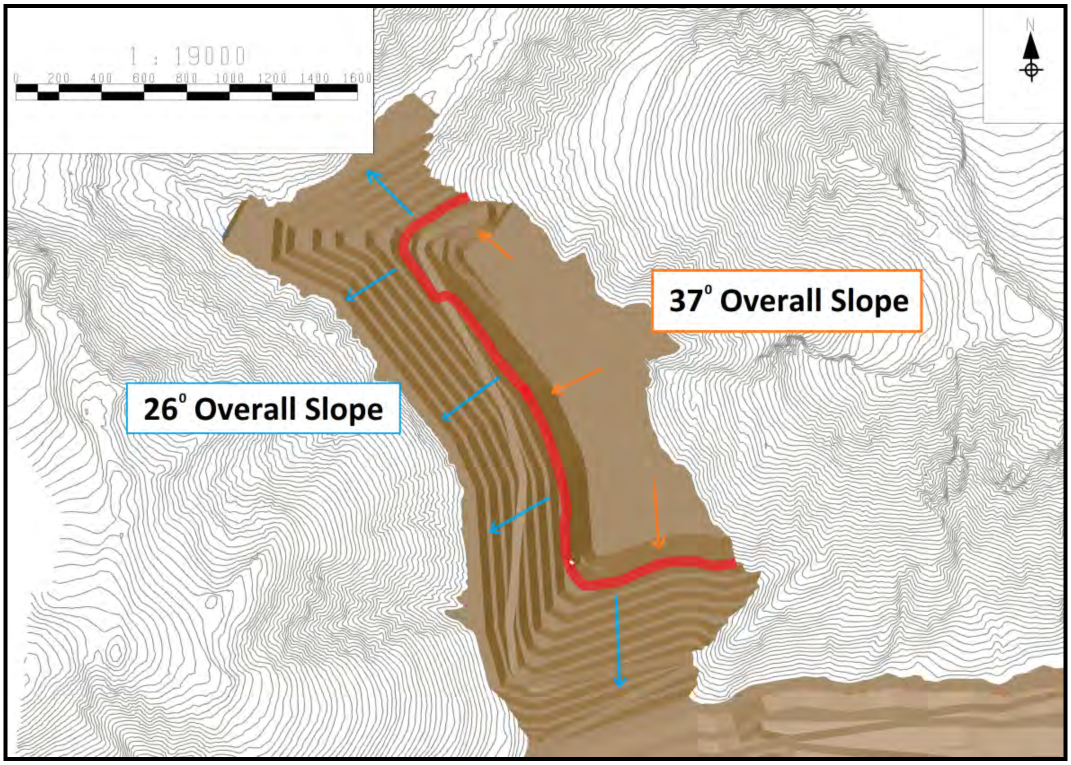
The McTagg RSF is built in lifts from the bottom up. Waste from the late Mitchell and Sulphurets phases is hauled along the Mitchell RSF, over a land-bridge, and onto the McTagg RSF. Kerr waste is conveyed from the Sulphurets portal exit, across the land-bridge to the McTagg RSF. The land-bridge between the McTagg RSF and Mitchell RSF is rehandled at the end of the mine life and hauled to the top of the McTagg RSF. Waste routes and the land-bridge are shown in Figure 16.33.

Figure 16.33 Plan View Showing Waste Routes to McTagg RSF



Below 1100 m elevation, the overall slope angle of the McTagg RSF is 26°. Above 1100 m elevation, the overall slope angle of the McTagg RSF is 37°. The final overall slope angles of the McTagg RSF are shown in Figure 16.34.

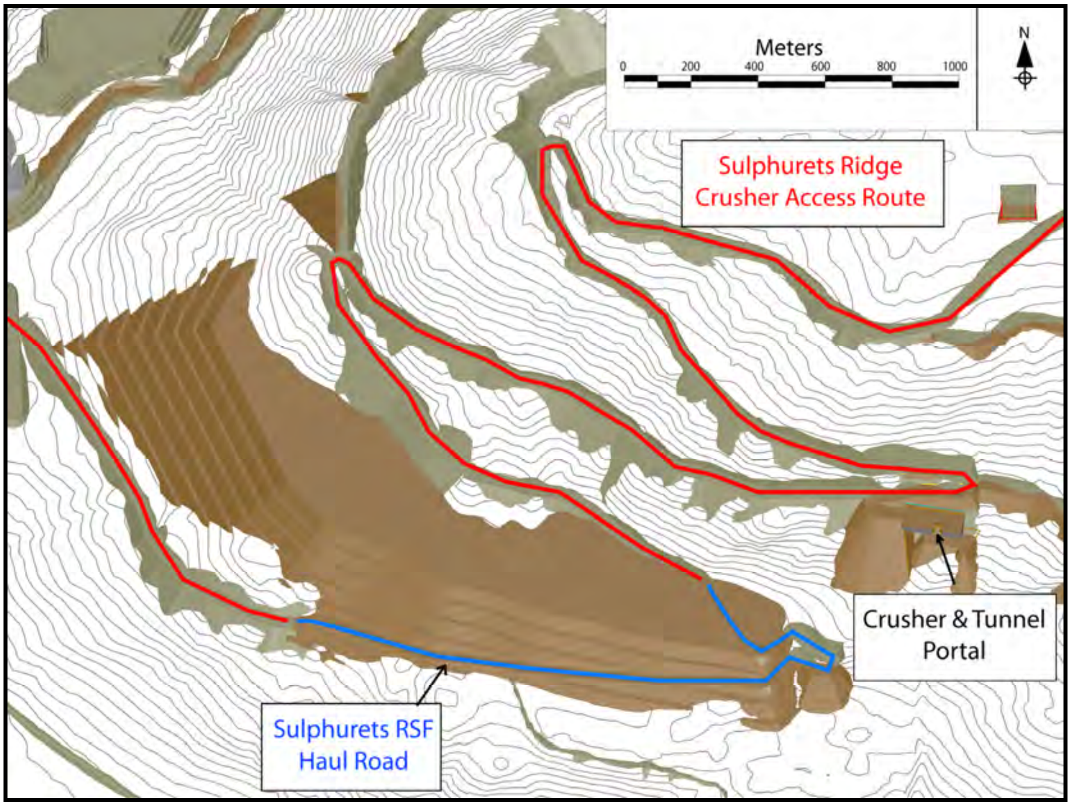
Figure 16.34 Plan View Showing Overall Slopes for the McTagg RSF



RSF Construction – Sulphurets RSF

The Sulphurets RSF is used as short-term storage for non-acid-generating waste. Between Years 21 and 30, the waste will be rehandled and placed on the Mitchell RSF. Using the bottom-up construction method, material is placed in the Sulphurets RSF in lifts, maintaining an overall slope angle of 26°. The initial Sulphurets Ridge access (built during pre-production) will be relocated onto the dump face as the Sulphurets RSF lifts are built progressively upward. The dump face haul road will tie into the Sulphurets Ridge Crusher Access Route (as shown by the blue line in Figure 16.35).

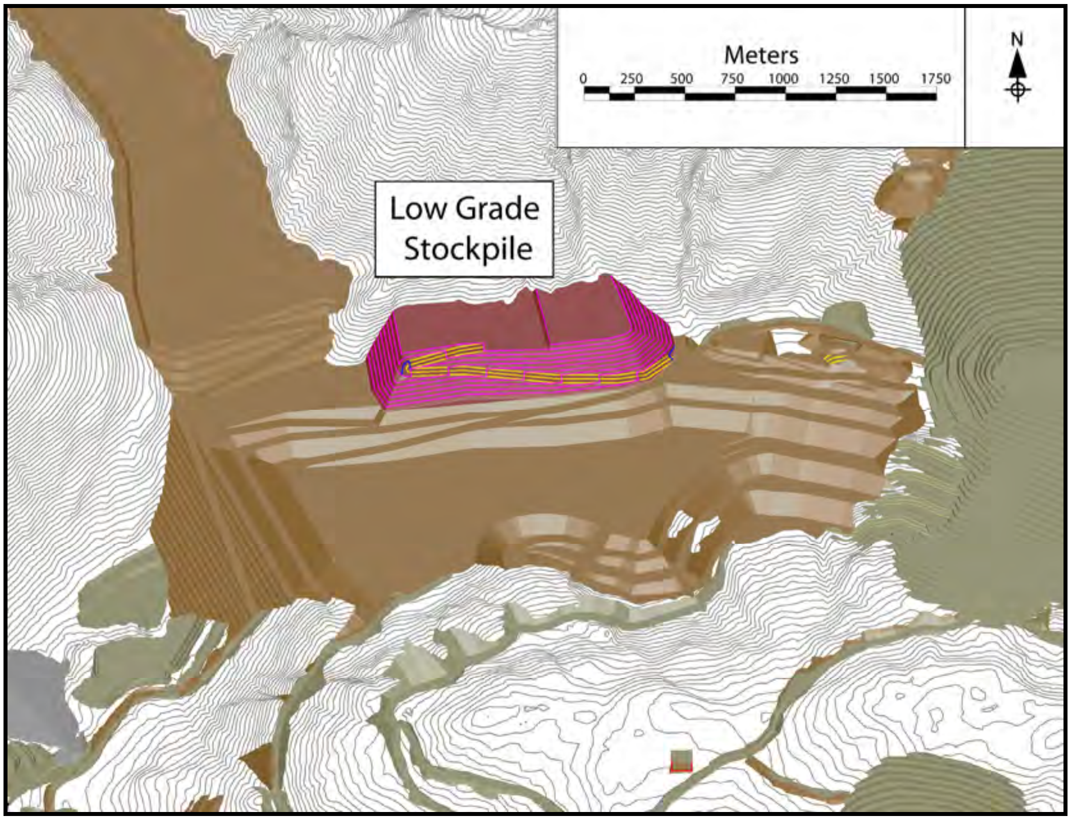
Figure 16.35 Sulphurets RSF Bottom-up Construction Method



Low Grade ROM Stockpile

Low grade ore is stockpiled throughout the mining schedule. The stockpile is built up to follow the cut-off grade strategy, and then is reclaimed in later years. This will maximize the grade of the ore feed to the plant in the early years of the schedule and is reclaimed to even out the mining fleet requirements as required. These aspects improve the project's cash flow. The low grade ore stockpile is placed to the west of the Mitchell OPC and reaches a maximum size of 140 Mt. Provision has been made for an HDPE pipeline diversion around the surface of the stockpile, which can be moved as required. The maximum stockpile size is shown in Figure 16.36.

Figure 16.36 Maximum Low Grade Ore Stockpile



RSF Monitoring and Planning

The long-term operation of the RSFs will be similar to that of the large, steep-terrain RSFs that have been in operation for many years in southeast BC Rocky Mountain coal mines. These operations involve high-relief RSF phases with clear dumping in single lifts of up to 400 m. Clear dumping is a technique whereby truck loads are dumped directly over the crest of the dump face; the load is not dumped short and then pushed over the edge. The clear dumping technique maintains a stable dump platform but requires well-established monitoring and operating practices. Foundation preparation also needs to be assured.

As indicated previously, rock placement during the initial mining stages will be achieved with low lifts and using the bottom-up construction method in areas that are non-critical, in order to establish consolidated foundations for future high relief dumps. As experience is gained and stable foundations are established, placement can proceed with higher lifts, as required, and utilize the top-down construction method.

The monitoring and safe operating practices referred to above, require all RSFs to be fitted with wireline extensometers and automated radar or other scanning equipment in areas where a significant downslope risk exists (i.e. above the Mitchell OPC, WSF, etc.). These measurements and techniques establish the safe operating limits for each dump face on the active RSF platforms and warn of any unsafe conditions that may arise. By moving dumping operations to alternative dump sites, any unstable conditions can be given time to consolidate and return to safe operating limits.

Annual Waste Volumes

Annual waste volumes produced from the 130,000 t/d schedule, shown by pit phase and by year, are provided in Table 16.18.

Table 16.18 Waste Tonnages by Area and Year (Mt)

	Mitchell Pit (MT)					Sulphurets (MT)		Kerr	Borrow +	TOTAL
Year	M1	M2	M3	M4	M5	S1	S2	(MT)	RH (MT)	(MT)
-4	-	-	-	-	-	-	-	-	10	**10**
-3	-	-	-	-	-	12	-	-	17	**29**
-2	22	-	-	-	-	25	-	-	-	**46**
-1	29	-	-	-	-	25	-	-	-	**54**
1	15	84	-	-	-	30	-	-	5	**135**
2	1	118	-	-	-	20	-	-	4	**142**
3	0.2	117	-	-	-	30	-	-	0.5	**147**
4	0.1	98	6	-	-	17	-	-	5	**127**
5	-	3	49	-	-	6	-	-	9	**66**
6	-	2	70	-	-	3	-	-	-	**75**
7	-	1	87	29	-	-	-	-	-	**116**
8	-	-	65	62	-	-	-	-	1	**128**
9	-	-	9	77	-	-	-	-	3	**88**
10	-	-	0.3	52	-	-	-	-	13	**65**
11-20	-	-	-	38	485	-	-	-	-	**523**
21-30	-	-	-	-	-	0.2	683	110	122	**916**
31-40	-	-	-	-	-	-	-	347	-	**347**
41-50	-	-	-	-	-	-	-	207	-	**207**
51-52	-	-	-	-	-	-	-	-	64	**64**
TOTAL	67	423	287	257	485	167	683	665	253	**3,287**

RSF Access Roads

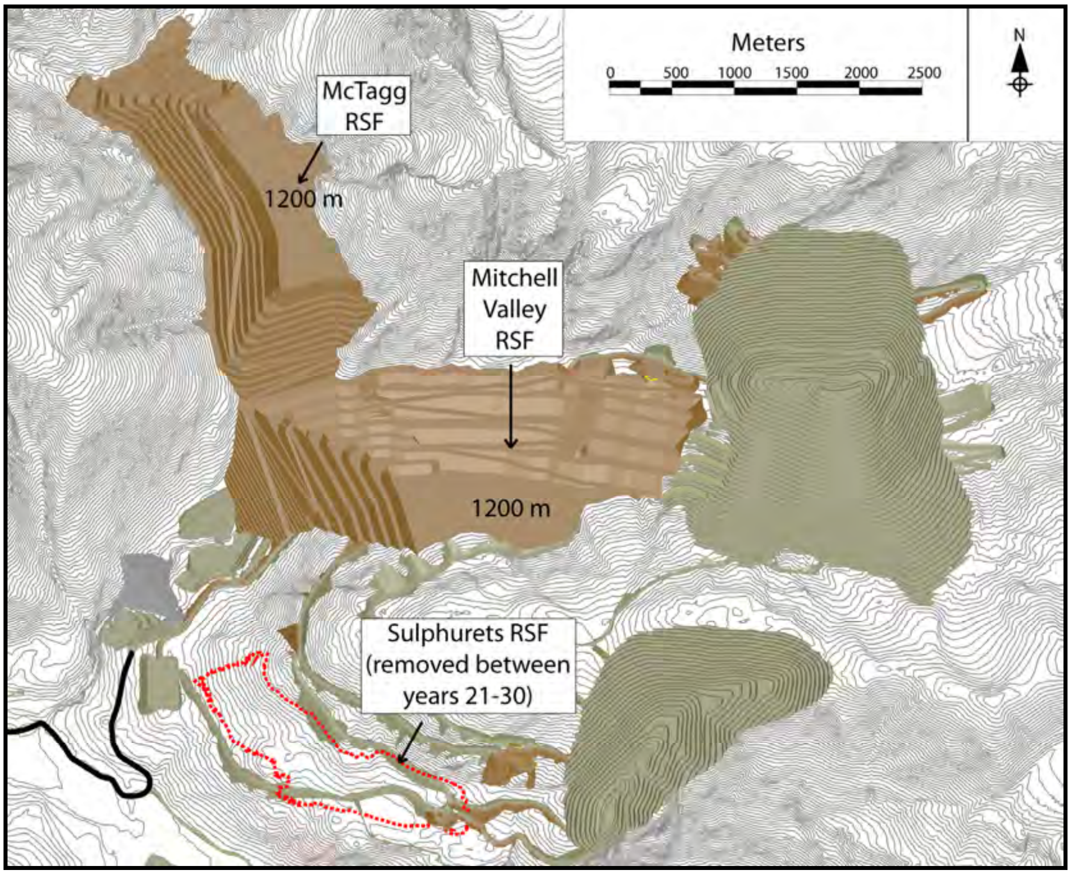
Pioneering access to each pit and subsequent phases use roads with a maximum 15% grade; these are constructed using balanced cut and fill wherever possible. Pioneering roads are 10 m wide and enable major mining equipment to reach the top of each pit phase and start mining. After the pioneering road is established to the top benches of each pit phase, bench waste from the upper portions of each pit phase is used to fill full-width haul roads at a maximum gradient of 8% at the 38 m double lane width, to connect with permanent surface roads and highwall roads in the long term road network. This road network connects the mining areas with the primary crusher and stockpile areas for ore, and the RSF areas for waste.

As described earlier, the terraced RSFs on the south side of the Mitchell Valley provide level access to the south Mitchell Valley RSF platforms.

Final RSF Configuration

The final RSFs for the KSM Project (Figure 16.37) will have overall slope angles of 26° to 30°. The final post closure configuration will be adapted in accordance with the closure plan. Costs for this work are included during the later years of the operation, when the waste strip ratio drops to low levels and ancillary equipment then becomes available for other duties. This will result in the reclamation occurring in the latter part of the LOM schedule using the mining equipment.

Figure 16.37 Final RSF Configuration



MINE PRE-PRODUCTION DETAIL

Pre-production Description

The mine pre-production development phase has three primary objectives:

- expose sufficient ore for start-up

- establish mining areas that will support the equipment required to achieve ore production, and annual mill feed requirements on a sustainable basis
- provide material required for construction of the mine, mill, and site infrastructure.

This section describes the development and pre-production activities that will be accomplished by the mine personnel and mine fleet equipment, and are included as capitalized mining costs in the cost model. Other development and construction activities are covered by other disciplines.

Mine pre-production site development activities are currently scheduled to start in Year -6, in order to meet the timeline for overall site development. Site development for the mine area will consist of:

- tree clearing and grubbing
- drainage control and water management facilities
- topsoil salvage
- pioneering access to construction and initial mining areas
- initial pit bench development
- haul road construction
- infrastructure construction
- pit power distribution construction.

Mine Area 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 of trees and brush is required, mainly in the lower elevation site works and waste dump areas, over an estimated area of 825 ha, and includes:

- pit area
- waste dumps
- ore stockpile
- mine haul roads
- crushing and slurry facilities area
- portal area
- explosives manufacturing plant and explosives magazine
- truck shop.

Mine Drainage

Mine drainage is broken into two separate ditch networks: the diversion network and collection network. The primary purpose of the diversion ditch network is to prevent non-impacted surface water (clean water system) from entering areas where it could become impacted. These diversion ditches are primarily located around the perimeter of the pit, the waste dumps, and the ore stockpiles.

The primary purpose of the collection ditch network is to route water that comes into contact with the mining operation. This water is part of a closed-circuit and is transported to the water storage dam, if necessary, where it will then be treated in the WTP. The collection ditches are primarily located within the pit area, at the toes of the waste dumps, at the toe of the ore stockpiles, and within the footprint of all mine haul roads.

Details on mine drainage are available in Section 18.1.7 (Mine Area Water Management).

Ore Haul Road Construction

A haul road is constructed from the first mining phase in the Mitchell pit to the primary crusher during pre-production from mine waste.

Mine Power

Mine power is required for electric drills, shovels, and pit pumping. Some lighting and electrical service is also required to the mine ancillary facilities including mine offices, mine maintenance facilities, and explosive manufacturing and storage facilities. Details on power supply and distribution, including the initial capital requirements for start-up and ongoing electrification of the mining operations, are provided in Sections 18.12 and 21.1. These details will form the basis for future procurement activities. The mine operating costs include the labour required for ongoing pit electrical service and maintenance work, as well as the expenses for a field line truck and service vehicles.

Mine Infrastructure Construction

Site preparation is also included for:

- the mine equipment erection site
- the explosives manufacturing plant and explosives magazines.

Facilities for the offices, maintenance shops, and fuel tanks will be available at the mine site before mining commences (as listed in the project schedule). These facilities are described further in Section 18.0 (Project Infrastructure).

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, which are built at a 15% grade in a balanced cut and fill method wherever possible. Pioneering is described in Appendix E.

Initial Pit Development

Once the pioneering roads are in place, the larger mine equipment will have access to the working areas and will commence 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 hill side to a lower bench elevation where the larger drill fleet and shovel-truck fleet can operate. The pioneering operations will create minimum width haul roads for the first production fleet to begin pre-stripping operations (drills, trucks, and shovels).

Pioneering and Pre-production Schedule of Activities

Pioneering roadwork starts in Year -6 when the Frank Mackie Winter Access Road is available. Other pioneering tasks continue into Year -3, including assembly pad preparations. After initial pioneering equipment is assembled, access is developed to laydown areas, camps, and tunnel portals. Tunnel portal access roads are critical path tasks and should receive the highest priority.

Production mining in Mitchell pit starts approximately one year after the Coulter Creek Access Road (CCAR) from Eskay Creek to Mitchell Creek is completed. Pre-production mining lasts two and a half years, and will conclude just prior to the start of Year 1. Plant start-up is scheduled for the beginning of Year 1.

Due to the high demand for mining equipment in the current commodities cycle, purchasing commitments for the large mining equipment are required well in advance of the mining activities. A two-year lead time is required for the electric cable shovel, the haul trucks, and the large drills; a procurement commitment for these items must be made by the start of Year -5.

Initial tree-clearing and grubbing activities for pioneering road development must be started in Year -6 in order to prepare the sites for mining activities. The major clearing and grubbing work for pre-stripping will take place in Year -5.

The site for mine equipment erection must be constructed during the pioneering phase and be completed before the CCAR is completed. Equipment delivery and assembly for the large mining equipment (shovels, trucks, and drills) begins as soon as the CCAR is completed.

Preparation of the sites for the explosives facilities must be started in Year -4 and will be completed prior to the start of mining with the large mining equipment. Temporary explosives storage will be required for the pioneering stage of mine development, and may be required for the initial pre-production.

The mine power distribution network must be completed before Year 1. The entire pre-production fleet is diesel-powered; electric equipment will only begin operation after the ore tunnel is completed.

Before pre-production begins the large mining fleet will mine colluvium from a borrow source in the Mitchell Valley to provide construction fill for the Mitchell OPC. During pre-production, Mitchell pit phase M681 is mined to 1080 m and M682 is mined to 1545 m in the Mitchell Valley; Sulphurets pit phase S691 is mined to an elevation of 1455 m. This will expose the necessary ore required to achieve the full mill production rate of 130,000 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 ore.

Mine pioneering and pre-production activities are described further in Appendix E.

A schedule of mine pioneering and pre-production activities is provided in Appendix G.

Mine Production Detail

Detailed end-of-period (EoP) mine status maps are shown in Appendix E. The mine plan for each period is described in this section with illustrative EoP figures.

Pioneering (Year -6 to Year -4)

During the first two years of the pioneering period, the KSM mining area is accessible via a winter road constructed from a staging area at Granduc across the Frank Mackie Glacier and down the Ted Morris valley. Pioneering equipment and consumables required during the pioneering period are delivered using the winter road.

Access roads are established to all mine site tunnel portals and the Mitchell and Sulphurets pits. The explosives facilities, water treatment facilities, and Mitchell OPC pad preparations take place during the pioneering period.

Large mining equipment delivered after the CCAR is completed is used to mine borrow sources to begin the construction of the WSF.

The mine layout at the end of pioneering is shown in Figure 16.38.

Pre-production (Year -3 to Year -1)

At the start of Year -2, a colluvium pit west of the Mitchell OPC is mined with the large mining fleet to complete fills required for the Mitchell OPC construction. Pre-stripping the Mitchell and Sulphurets pits begins when the MDT is completed.

During the pre-production period, M681 is mined to an elevation of 1080 m and M682 is mined to an elevation of1545 m. M681 waste is dozed to an elevation of 1285 m to create a mining platform. Waste below these elevations is dumped to fill a double-lane haul road to the Mitchell OPC and Mitchell RSF. Once the road is built, waste is hauled west into the Mitchell Valley and used to build a rock base pad for the Mitchell RSF at an elevation of 780 m. This pad is built from the bottom-up in lifts.

During the pre-production period, S691 is mined to an elevation 1455 m. All PAG waste is hauled to the Mitchell Valley and dumped in the Mitchell RSF. Sulphurets NPAG waste is hauled to the Sulphurets RSF, which is built to an elevation of 930 m.

A small ore stockpile is built in the area to east of the Mitchell OPC, within the final Mitchell pit limit.

The mine layout at the end of pre-production is shown in Figure 16.39.

Year 1

By the end of Year 1, M681 is mined down to an elevation of 915 m and M682 is mined down to an elevation of 1395 m; S691 is mined down to an elevation of 1395 m. All Mitchell waste and Sulphurets PAG waste material is used to build up a 780 m base pad for the Mitchell RSF and terrace roads. Sulphurets NPAG waste is hauled to the Sulphurets RSF, which is built to an elevation of 960 m. Mitchell and Sulphurets ore is hauled directly to the Mitchell primary crushers.

An ore stockpile is built in the area to west of the Mitchell OPC.

The east and west edges of the Mitchell RSF are built in lifts at an overall slope of 2:1 with an access road in the final face. Other portions of the Mitchell RSF are built in 100 m lifts using top-down techniques.

The mine layout at the end of Year 1 is shown in Figure 16.40.

Year 2

By the end of Year 2, M681 is mined to 810 m, M682 is mined to 1275 m, and S691 is mined to 1350 m elevation. Mitchell waste and Sulphurets PAG waste material is hauled to the 840 m Mitchell RSF base pad. Sulphurets NPAG waste is hauled to the Sulphurets RSF, which is built to an elevation of 975 m. Ore is hauled to the Mitchell primary crushers or stockpiled adjacent to the Mitchell OPC.

The mine layout at the end of Year 2 is shown in Figure 16.41.

Year 3

By the end of Year 3, M681 is mined to 750 m, M682 is mined to 1155 m, and S691 is mined to 1290 m elevation. Mitchell waste and Sulphurets PAG waste is used to build the 900 m terrace road or is used to extend the 840 m elevation base pad for the Mitchell RSF. Sulphurets NPAG waste is hauled to the Sulphurets RSF, which is built to an elevation of 990 m. Ore is hauled to the Mitchell primary crushers or stockpiled adjacent to the Mitchell OPC.

The mine layout at the end of Year 3 is shown in Figure 16.42.

Year 4

By the end of Year 4, M681 is mined to completion (690 m elevation), M682 is mined to 975 m, and S691 is mined to 1245 m elevation. Mining begins on the north phase of Mitchell and M683 is mined to an elevation of 1380 m. Mitchell and Sulphurets waste is used to build the terrace road to an elevation of 1020 m or is used to extend the 840 m elevation base pad for the Mitchell RSF. Ore is hauled to the Mitchell primary crushers or stockpiled adjacent to the Mitchell OPC.

The mine layout at the end of Year 4 is shown in Figure 16.43.

Year 5

By the end of Year 5, M682 is mined to 885 m, M683 is mined to 1245 m, and S691 is mined to 1200 m elevation. Mitchell and Sulphurets waste is used to build the terrace road to an elevation of 1065 m or is used to extend the 840 m elevation base pad for the Mitchell RSF. Ore is hauled to the Mitchell primary crushers or stockpiled adjacent to the Mitchell OPC.

The mine layout at the end of Year 5 is shown in Figure 16.44.

Year 10

By the end of Year 10, M682 and M683 are mined to completion (585 m elevation), M684 is mined to 885 m, and S691 is mined to completion 1125 m elevation. Mitchell and Sulphurets waste rock is used to build the terrace road to an elevation of 1200 m or hauled to the Mitchell RSF. Ore is hauled to the Mitchell primary crushers or stockpiled adjacent to the Mitchell OPC.

The mine layout at the end of Year 10 is shown in Figure 16.45.

Year 20

At the end of Year 20, mining of Mitchell pit phase M684 is completed (495 m) and M685 is mined to 525 m.

Waste rock from the Mitchell pit is dumped on the Mitchell RSF (990 m elevation) or the McTagg RSF (945 m elevation). Ore from the Mitchell pit is hauled to the Mitchell primary crushers or stockpiled.

The mine layout at the end of Year 20 is shown in Figure 16.46.

Year 30

At the end of Year 30, mining of the final Mitchell pit phase M685 is completed (405 m elevation). The Sulphurets crusher, SMCT, Kerr crusher, and rope conveyor are complete and operational. Sulphurets ultimate pit S692 is mined to completion at an elevation of 825 m. Access to the Kerr pit is established. All cut material from the Kerr access road is hauled to the Mitchell RSF and a NPAG borrow pit is used to create road fills. Kerr pit phase K691 is mined to 1920 m.

All waste from the Sulphurets RSF is rehandled and hauled to build a land-bridge between the Mitchell and McTagg RSF. The land-bridge enables shorter hauls for upper waste benches on the south side of the Mitchell pit going to the McTagg RSF. Waste from the Sulphurets pit is hauled to the Mitchell RSF until it reaches maximum capacity. Once the Mitchell RSF is filled, Sulphurets waste is hauled to the McTagg RSF. Waste from Kerr is crushed and conveyed to the Mitchell Valley and over the land-bridge to the McTagg RSF.

Ore from the Kerr pit is crushed and conveyed along a rope conveyor across Sulphurets Valley and through the SMCT to the Mitchell OPC. Ore from the Sulphurets pit is crushed and conveyed through the SMCT to the Mitchell OPC.

The mine layout at the end of Year 30 is shown in Figure 16.47.

Year 40

At the end of Year 40, the Kerr pit K691 is mined to an elevation of 1350 m. Waste from Kerr is crushed and conveyed to the Mitchell Valley and over the land-bridge to the McTagg RSF. Ore from the Kerr pit is crushed and conveyed along a rope conveyor across Sulphurets Valley and through the SMCT to the Mitchell OPC.

The mine layout at the end of Year 40 is shown in Figure 16.48.

Year 50

At the end of Year 50, Kerr pit K691 is mined to completion at an elevation of 960 m. Waste from Kerr is crushed and conveyed to the Mitchell Valley and over the land-bridge to the McTagg RSF. Ore from the Kerr pit is crushed and conveyed along a rope conveyor across Sulphurets Valley and through the SMCT to the Mitchell OPC.

The mine layout at the end of Year 50 is shown in Figure 16.49.

End of Open Pit Mining

At the end of the open pit mining phase, waste from the landbridge is rehandled and placed on the top of the McTagg RSF. Once the landbridge is removed, a closure channel is established around the Mitchell RSF by placing moraine material and NPAG riprap on berms along the north and west toes of the Mitchell RSF.

The mine layout at LOM is shown in Figure 16.50.

Figure 16.38 End of Pioneering (Year -4)

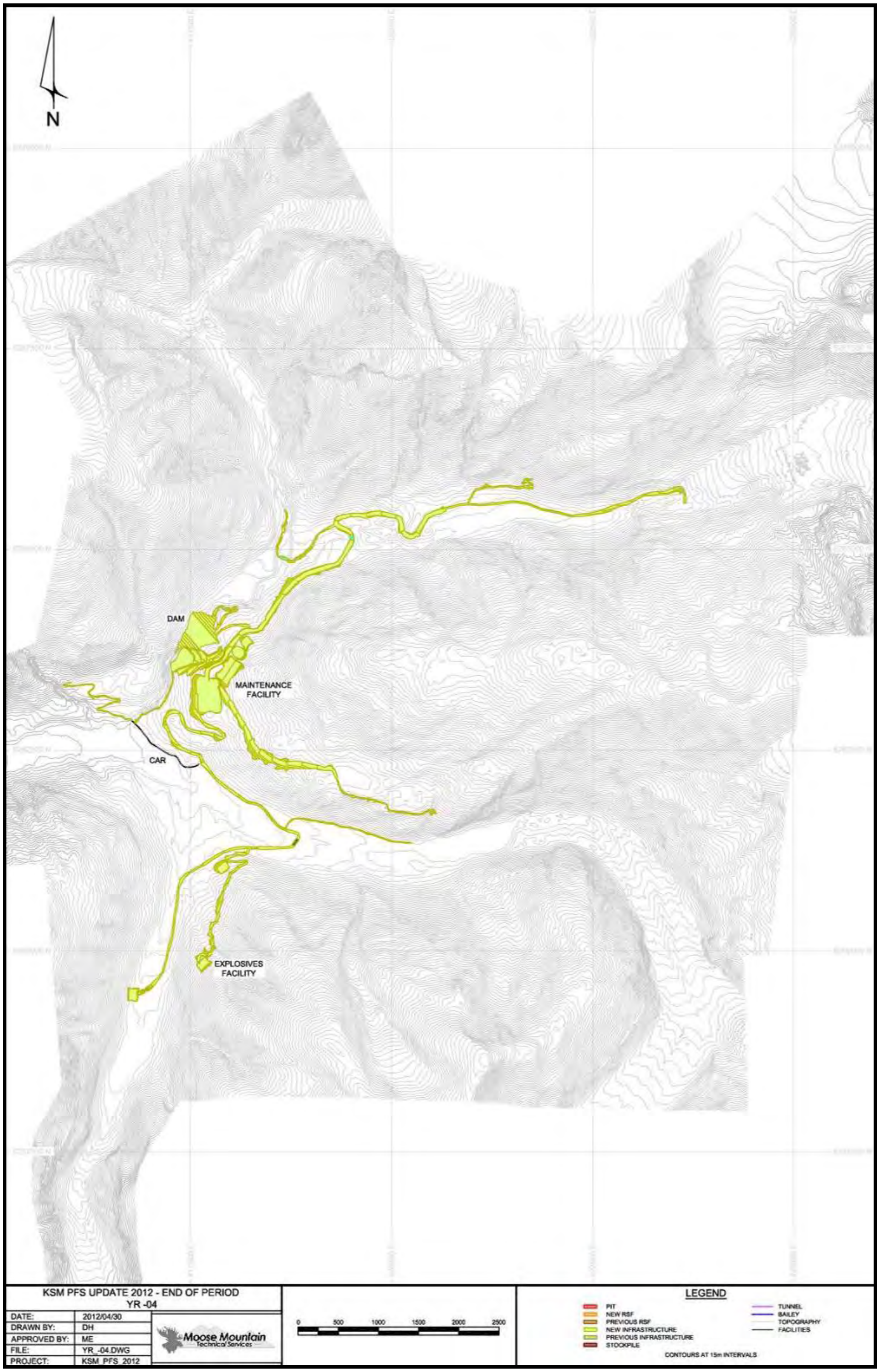


Figure 16.39 End of Pre-production (Year -1)

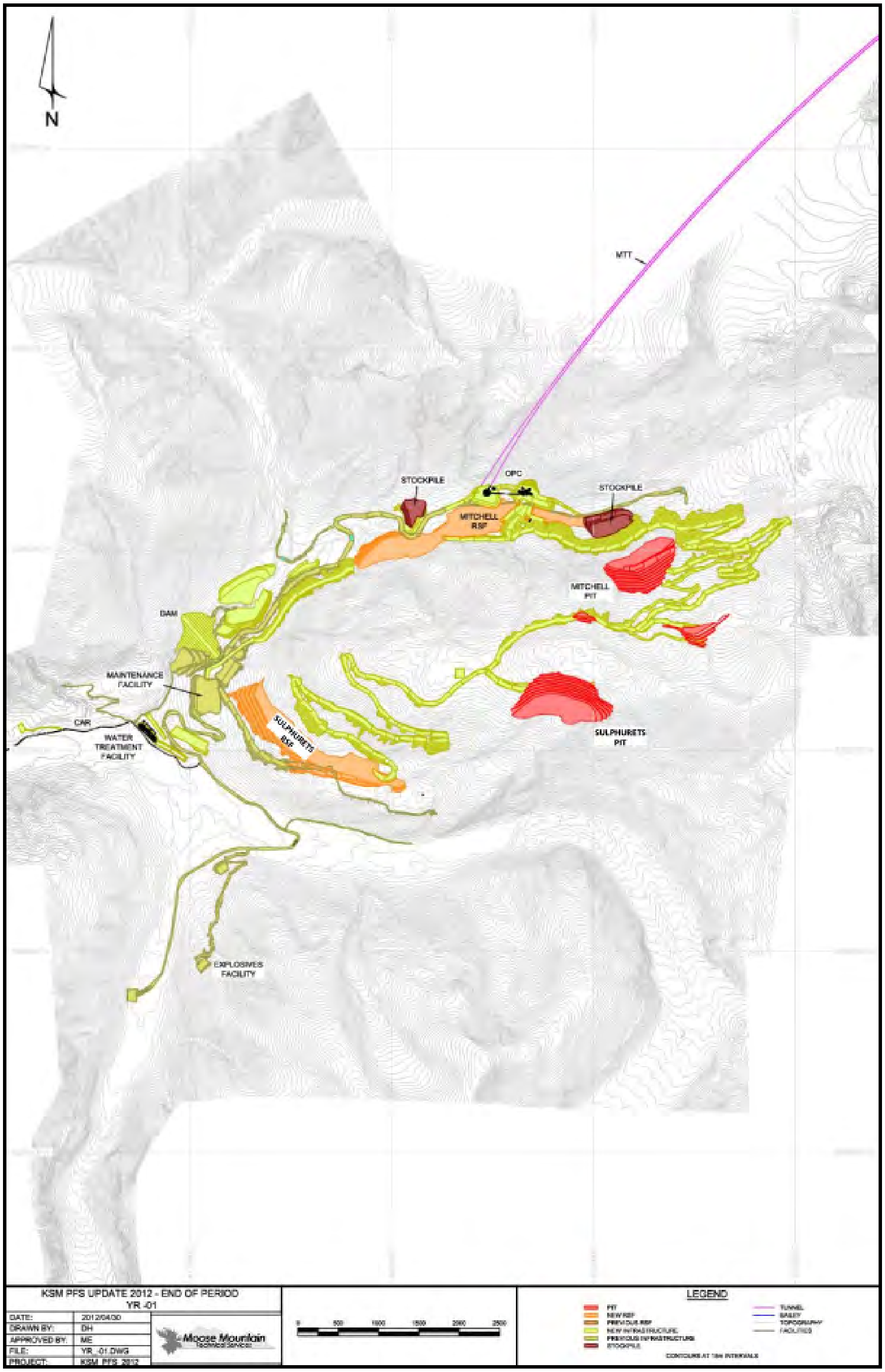


Figure 16.40 End of Year 1

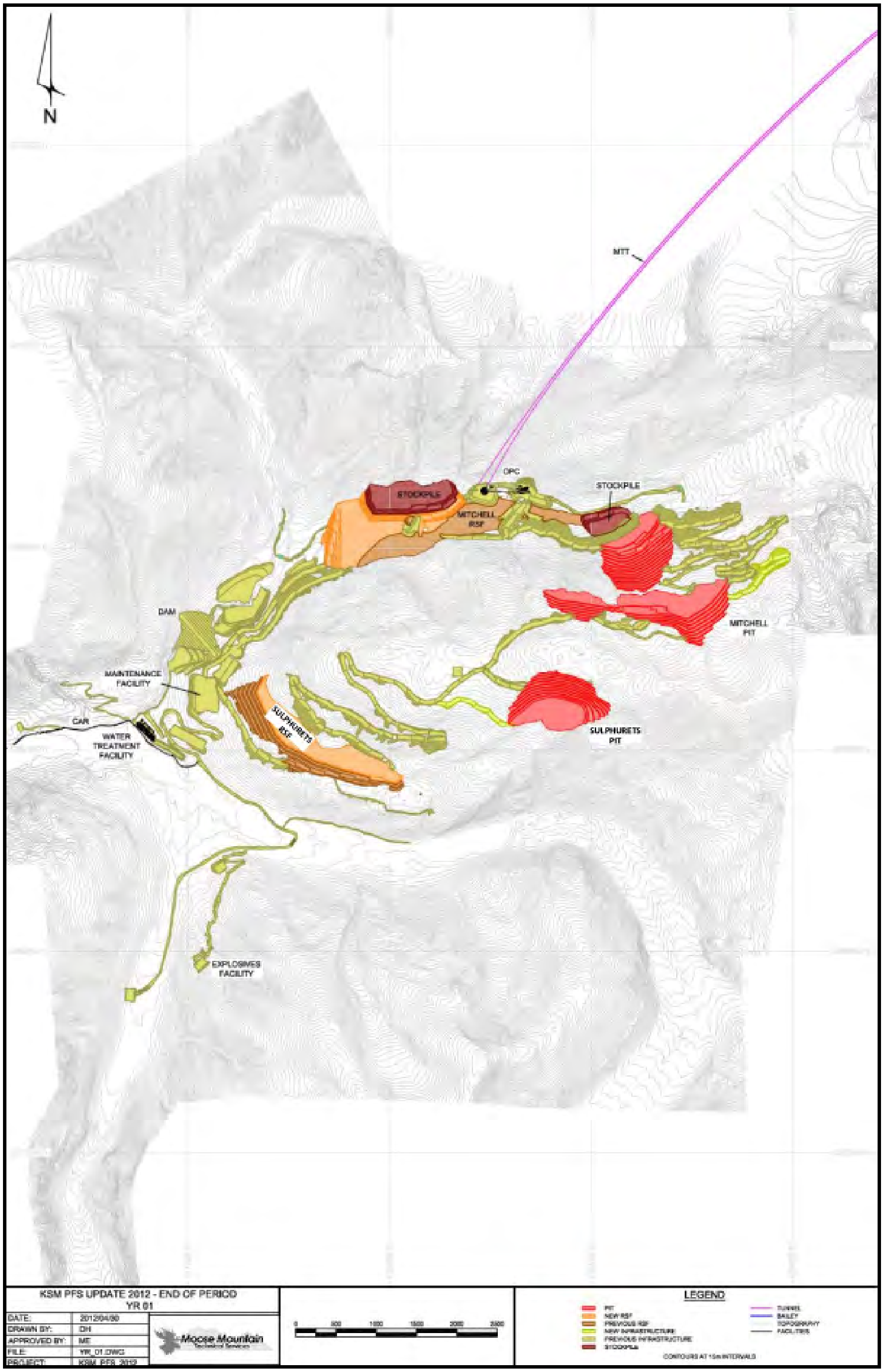


Figure 16.41 End of Year 2

Figure 16.42 End of Year 3

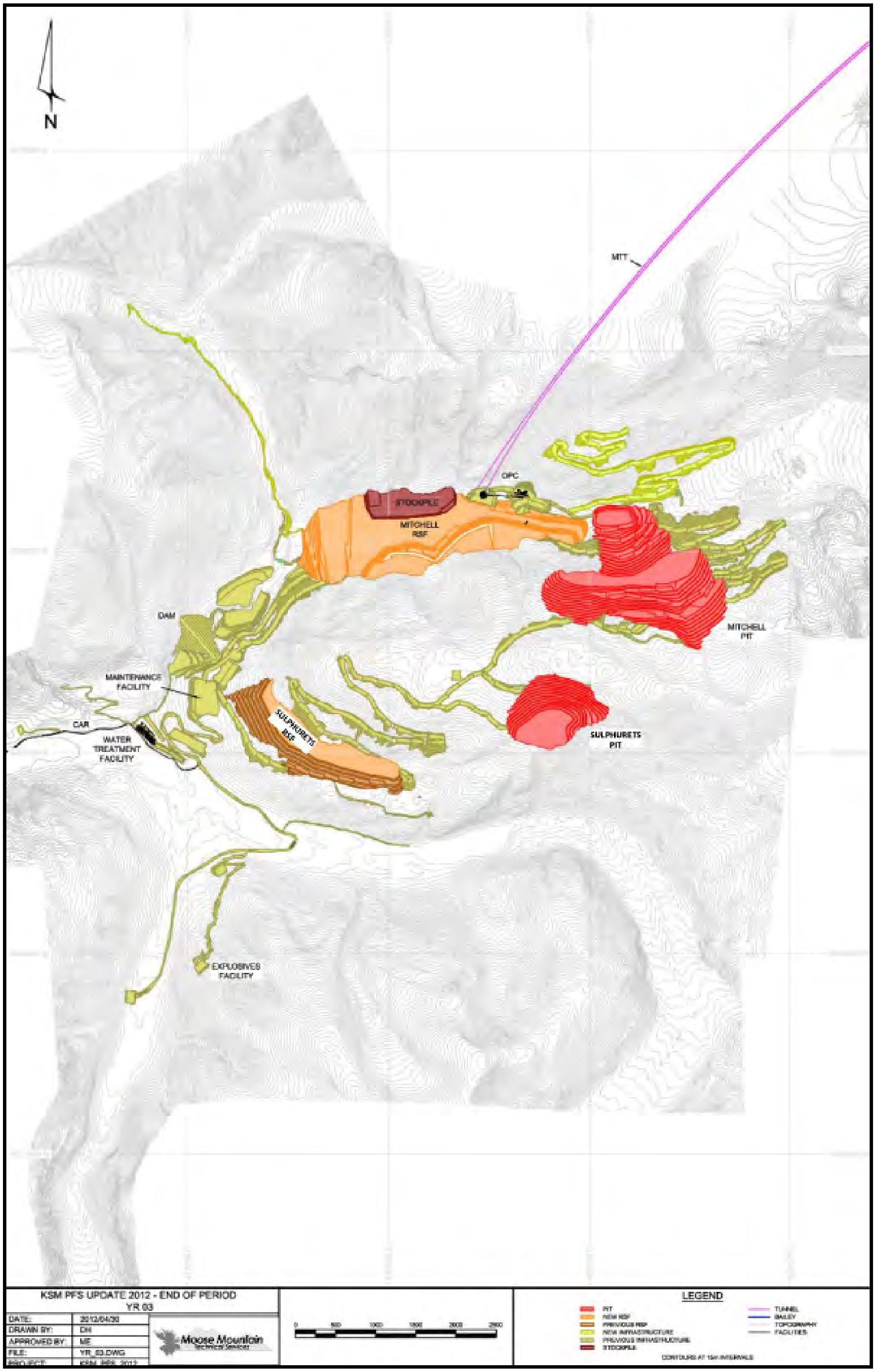


Figure 16.43 End of Year 4

Figure 16.44 End of Year 5

Figure 16.45 End of Year 10

Figure 16.46 End of Year 20

Figure 16.47 End of Year 30

Figure 16.48 End of Year 40

Figure 16.49 End of Year 50

Figure 16.50 Open Pit Life of Mine

MITCHELL GLACIER

The Mitchell Glacier toe has receded eastward at an average of 31 m/a between 1982 and 2008, and a mean of 56 m/a between 2004 and 2008.

The 2012 KSM PFS mine plan has been developed with the assumption that the Mitchell Glacier toe will continue to recede at least an average of 31 m/a, which is significantly less than the recent rate. With this assumption, the Mitchell Glacier will remain out of the active mining areas throughout the duration of this mine plan. Therefore, mining of the Mitchell Glacier is assumed to not be required in this mine plan.

16.1.8 OPEN PIT MINE OPERATIONS

The open pit and underground operations are considered as separate operations in this study, with their own facilities and management, technical, and operating personnel. Future detailed planning may be able to reduce costs by integrating some of the support services and facilities. The following description is for the open pit operations. Underground operations details are provided in Section 16.3 and Appendix E.

KSM mining operations will be typical of open pit operations in mountainous terrain in western Canada, and will employ accepted bulk mining methods and equipment. There is considerable operating experience and technical expertise for the proposed operation in western Canada. Services and support in BC and in the local area are well-established as well.

A large capacity operation is being designed; therefore, large-scale equipment is required for the major operating functions in the mine. This will generate high productivities and therefore minimize unit mining and overall mining costs. Large-scale equipment will also reduce the on-site labour requirement, 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 labour requirements are minimized.

ORGANIZATION

General Organization

The KSM operations will be generally organized in the manner illustrated in Figure 16.51. Mine operations will deal solely with the organization areas as highlighted in Figure 16.51. Other areas of the organization are dealt with elsewhere in the report. Mine operations is organized into three areas: direct mining, mine maintenance, and general mine expense (GME).

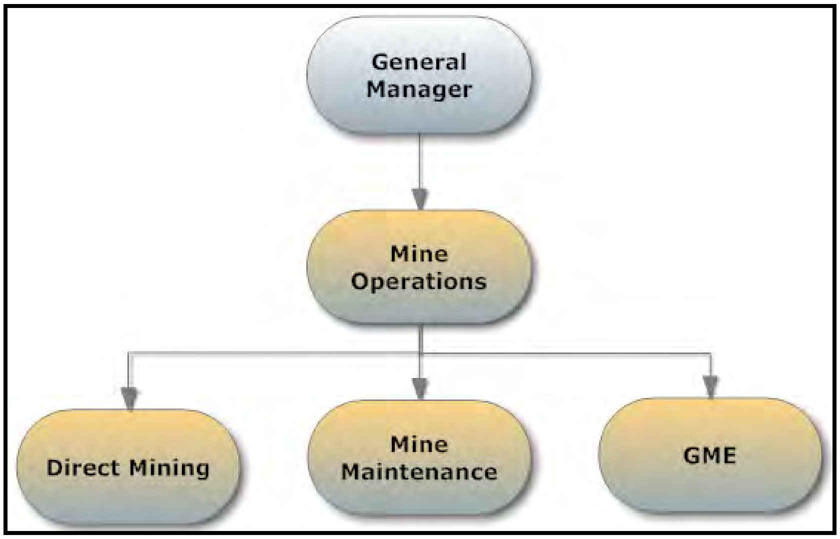
The direct mining area accounts for the drilling, blasting, loading, hauling, and pit maintenance activities in the mine. Costs collected for this area include the mine operating labour, mine operating supplies, equipment operating hours and supplies, and distributed mine maintenance costs. The distributed 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. These are in turn 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, 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.

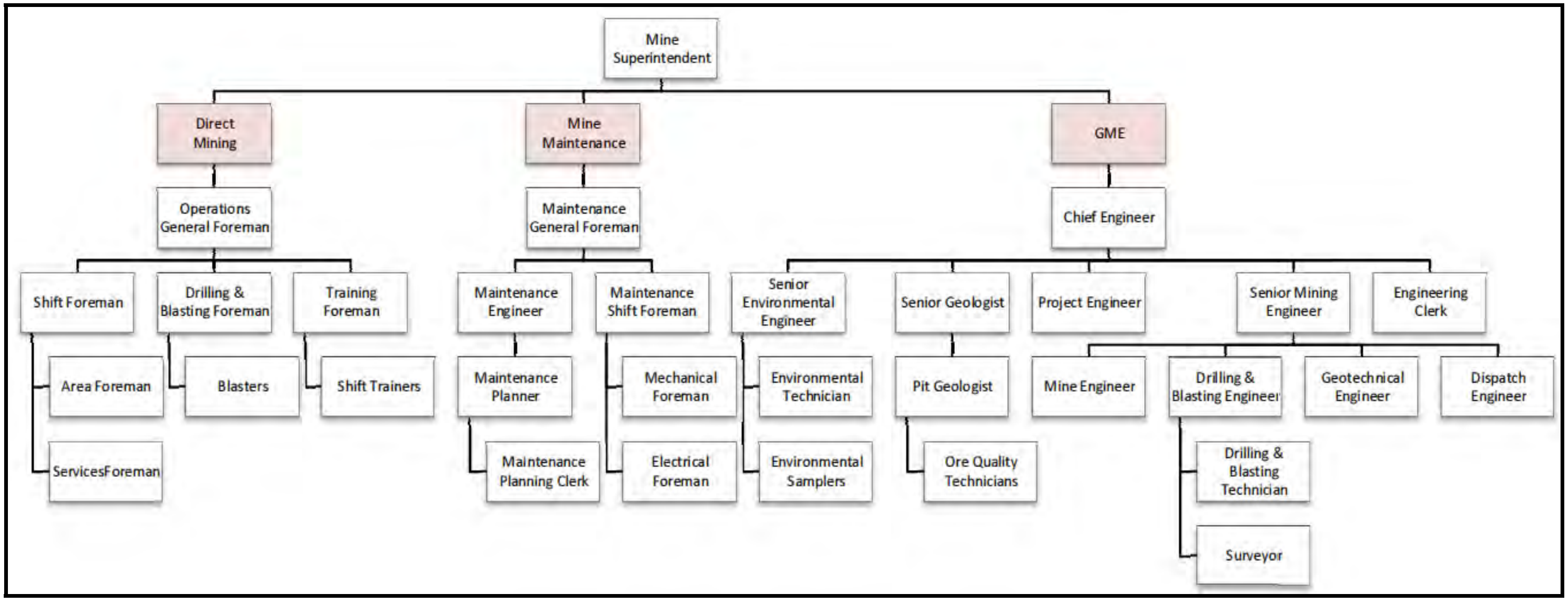
Figure 16.51 General Organization Chart – Open Pit Operations



Open Pit Mine Operations Organization

Details of the open pit mine operations organization are illustrated in Figure 16.52, showing the breakdown of the direct mining, mine maintenance, and GME functions.

Figure 16.52 Open Pit Mine Operations Organization Chart



DIRECT MINING ACTIVITIES – OPEN PIT

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. Ore 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. General descriptions of the direct mining unit operations are included 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 portions of each phase. 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, which is recommended for efficiency in locating hole locations and accuracy of set-up, particularly since this is a high snow fall area. Drills will be fitted with automatic samplers to provide ore grade control samples from drill cuttings in the ore zones. These samples will be used in the OCS for blast hole kriging to define the ore/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 ore and waste.

Diesel hydraulic percussive drills measuring 6½″ (165 mm) will operate in all pit phases for controlled blasting techniques on high wall rows, pioneering drilling during pre-production, and development of initial upper benches. Drilling for controlled blasting requirements have been estimated based on an estimate of the length of pit wall exposed on a bench in any given year.

Blasting

Powder Factor

An appropriate powder factor has been used to provide adequate fragmentation and digging conditions for the shovels, balanced with reducing blasting costs. 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. A blasting study carried out by Orica (Appendix E) suggests that a power factor of 0.35 kg/t is suitable in this area. Future optimization and Feasibility Study planning can investigate further mine to mill performance with respect to blasting. These optimizations are being implemented in operating mines in Western Canada. This level of refinement has not been included in the PFS.

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. For this study, the explosives contractor delivers the prescribed explosives to the blast holes and supplies all blasting accessories. These are costed on a per kilogram basis for explosives and on an itemized basis for accessories.

Until the extent of ground water and surface water in the blast holes is determined, it is assumed that half of the holes will use a 70/30 emulsion/ammonium nitrate-fuel oil (ANFO) mix explosive ("wet" product) and half of the holes will use a 35/65 emulsion/ANFO mix ("dry" product). Higher use of ANFO, and possible use of borehole liners to keep the ANFO dry to prevent incomplete detonations, can be investigated in future studies to reduce blasting costs.

Blasting accessories will be stored in magazines.

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 are located in the PFS as 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 should 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. The 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.

A smaller "goat" truck is needed for development areas with small access roads and narrow bench working conditions, as well as for squaring-off blast patterns when the mine roads have been closed due to excessive snow fall. "Goat" trucks are similar to a logging skidder and are named because of their high manoeuvrability. The "goat" truck will be used at the start of each incremental phase in Mitchell pit and the first few benches of the Kerr and Sulphurets pits.

The blast holes will 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.

Occasionally during the high snow fall period, access to a partially loaded pattern may be cut off for the explosives loading trucks. In these instances, it will be necessary to square-off and shoot the pattern using the "goat" truck. This will include tying in the loaded holes before the snow accumulation gets too high to locate unloaded holes or to find the downlines for primed holes. In these instances, it is necessary to square-off the pattern by loading some holes to complete already loaded sections of a blast pattern.

Blasting Operations

The blasting crew will be comprised of mine employees and will be on day shift only. Based on existing mines of similar size and previous experience, the crew size will be eight people. 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. 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 control. The blasters will require a hand-held GPS to identify the holes for the pattern tie-in. The pattern size may be limited by the rate of snowfall in some months. A detonation system will be used that consists of electric cap initiation, 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.

Based on the desired powder factor, the blasting specifications for the KSM operations have been evaluated for a large diameter blast hole size of 311 mm. Blasting assumptions are summarized in Table 16.19. 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 16.19 Blasting Assumptions

Blasting Pattern – Ore & Waste	Specifications
Spacing	8.5 m
Burden	8.5 m
Hole Size	12¼″
	311mm
Explosive In-Hole Density	1.25g/cc
Explosive Average Downhole Loading	95.0 kg/m
Bench Height	15 m
Collar	6 m
Loaded Column	11 m
Sub-drill	2 m
Charge per Hole	1,046 kg/hole
Rock SG	2.77 t/m^3
Yield per Hole	3,002 t/hole
Powder Factor	0.35 kg/t

Loading

Ore and waste will be defined in the blasted muck pile as defined by the OCS, and 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. Three 85-t dipper 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 an electronic navigation-based digging monitor, which will enable digital dig boundaries from the OCS 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 affects 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 operating room is suitable for efficient double-sided loading of trucks at the shovels, but 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, and other dozing activities.

Optimization of the shovel fleet is required in future studies. Specifically, there are many years in the PFS where a significant portion of the large shovel's production capability is not being fully used due to increased haul distances limiting the available trucks to the shovels. Evening out the haul distances and modifying the pit phase designs will assist evening out the annual waste requirement giving a more constant shovel usage year to year.

Hauling

Ore and waste haulage will be handled by large off-highway haul trucks with a 363-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 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.

Further optimization of the haulage fleet is required in future studies to even out the truck fleet requirements. Also in this PFS, 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.

Pit Maintenance

Pit maintenance services include haul road maintenance, mine dewatering, transporting operating supplies, relocating equipment, and snow removal.

Haul road maintenance is paramount to low haulage costs; dozer and grader hours are allocated and adjusted to maintain the haul road network throughout the LOM production schedule.

A fleet of ancillary service vehicles are allocated to install and service the in-pit sump pumps and the highwall horizontal drains. This includes connecting these pumps to the pit dewatering pipeline system. This crew will also service and supply mobile light plants.

A fleet of service equipment is allocated for summer season construction and will be used in winter for snow clearing. This includes scrapers and loaders. Extra equipment is also included for the snow fleet to handle regular seasonal snow clearing as well as standby equipment for snow storms to ensure there are minimum snow delays for the production fleet. 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 standby snow fleet will be drawn from truck and shovel operations as the fleets shut down. This will ensure priority fleets remain operating.

A rock crusher for road grading material is included to improve truck travel speeds, reduce mechanical fatigue to the haul trucks, and to enhance tire life, which is a major mine operating cost.

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 breakdown and field maintenance and 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.

GENERAL MINE EXPENSE AREA

This section describes the mine GME as costed in the mine cost model in Appendix E.

The GME area accounts for the supervision, safety, environment, 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 and 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, and will manage and execute 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 hydro-geological 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.

GME costs also include engineering consulting on an ongoing basis for specialty items such as geotechnical, environmental, and geo-hydrology expertise and third-party reviews.

A fleet management system is specified for the trucks, shovels, and the 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 manage 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.

16.1.9 MINE CLOSURE AND RECLAMATION

At the end of the mine life, an approved closure and reclamation plan will be implemented that will meet the end land use objectives and satisfy the regulatory commitments. The mining costs provide an allowance for the following general reclamation activities. Details on mine closure and reclamation are available in Section 18.1.

MINE RSF RECLAMATION

Mine RSFs have been designed as a mix of top-down end dumping, wraparounds, and bottom-up lift dumping. The RSFs will be placed during the mine life, as close to their final closure configuration as possible; however, some re-contouring will be required at closure. The top-down dumps will form at the natural angle of repose of 37°. Salvaged reclamation and soil material will be placed on the RSF surfaces after resloping, where required. The design criteria for RSF reclamation and closure in each area are included in Appendix E.

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 planted/seeded as required.

16.1.10 MINE EQUIPMENT

The mining equipment descriptions in this section provide general specifications so that dimensions and capacities can be determined from vendor specification documents.

MAJOR MINE EQUIPMENT

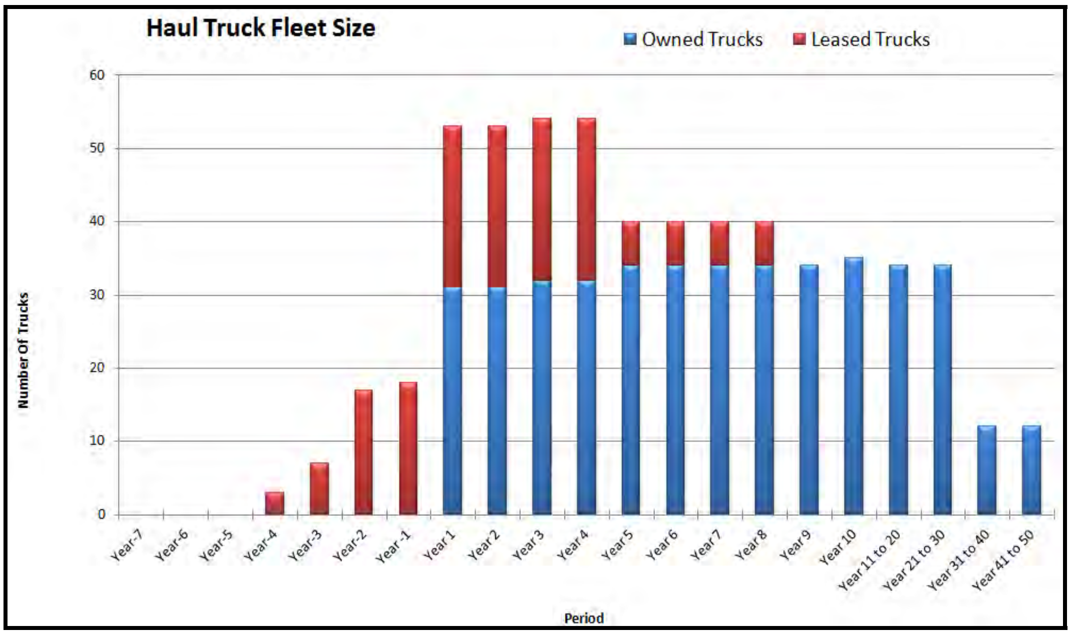
The production requirements for the major mining equipment over the LOM are summarized in Table 16.20. The full fleet schedule requirement is shown in Appendix E. According to the current production schedule and the haulage assumptions, a maximum haulage fleet of 58 trucks is required over the LOM. The haulage fleet will be comprised mostly of owned equipment, but will contain some leased trucks during pre-production and the first 8 years of operation.

Table 16.20 Major Mine Equipment Requirements

	PP	Y5	Y10	Y20	Y30	Max
Drilling						
Primary Drill – 311 mm Electric Drill	1	3	3	3	3	3
Primary Drill – 311 mm Diesel Hydraulic Drill	2	3	1	1	1	3
High Wall Drill – 150 mm Diesel Hydraulic Drill	4	4	4	4	4	4
Loading						
Primary Shovel – 85 t (40 m^3) Diesel Hydraulic Shovel	2	2	2	1	1	2
Primary Shovel – 100 t (56 m^3) Electric Cable Shovel	0	3	3	3	3	3
Hauling						
Haul Truck – 363 t	0	34	35	34	34	35
Leased Haul Truck – 363 t	18	26	4	0	0	22

The haul truck fleet size schedule is shown in Figure 16.53.

Figure 16.53 Haul Truck Fleet Size



DRILLING EQUIPMENT

The primary production drilling will be carried out in ore 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 16.21.

Table 16.21 Production Drilling Assumptions

Production Drill – Mineralized Material & Waste	Electric Rotary	Diesel Rotary
Bench Height	15m	15m
Subgrade	2.0m	2.0m
Hole Size	311mm	311mm
Penetration Rate	40.0m/h	40.0m/h
Hole Depth	18m	18m
Over Drill	1.0m	1.0m
Setup Time	2.0 min	2.0 min
Drill Time	27.0 min	27.0 min
Move Time	2.0 min	2.0 min
Total Cycle Time	31.0 min	31.0 min
Holes per Hour	1.94	1.94
Re-drills	6%	6%

A 150 mm diesel percussive drill is also specified for drilling, which is required to operate in all pit phases for controlled blasting techniques on high wall rows, pioneering drilling during pre-production, and development of initial upper benches.

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 Equipment and Facilities

Blasting activities are detailed in Section 16.1.8.

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 selection of explosives plant locations has avoided geohazards identified in a study conducted by BGC in 2010-2011.

Loading and Hauling Equipment

The shovel-truck fleet selected for KSM is the 100-t (56 m^3) dipper class of electric shovel, and the 363-t payload class of truck. An 85-t (40 m^3) dipper class diesel-hydraulic shovel is also required for difficult to access development benches, and enables pre-production mining before power is established to the mine site. The 85-t units loading the 363-t trucks are suitable as production shovels as well. Loading and hauling is discussed in Section 16.1.8.

Dewatering Equipment

It is important to control the water present in active mining areas. In-pit water generally increases the cost of mining especially in blasting, where explosives loading, explosives costs (ANFO versus emulsions), and blast performance is affected by water. Wet conditions and standing water increase the occurrence of rock cuts to tires. Rocks can easily be hidden in puddles that the haul trucks have to drive through and this can lead to instantaneous tire failure. Wet muck that the shovels are digging will freeze to the sides of the truck boxes in the wintertime, 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, and flooded box cuts must be drained. All of these conditions increase operating costs and need to be addressed by an effective pit dewatering program.

The dewatering activities will include the following:

- horizontal drain holes in bench faces
- sloped pit floors as required
- in-pit sumps
- vertical dewatering wells
- a dewatering tunnel behind the north highwall
- water collection system.

Pit water will be collected and treated at the WSF before discharging.

MINE SUPPORT EQUIPMENT

The mine support equipment fleet requirements are summarized in Table 16.22. The fleet size in Year 5 and Year 10 is shown as representative of the LOM requirement. A description of the equipment chosen and the tasks that the equipment performs in support of the mining operations are included in Appendix E.

Table 16.22 Mine Support Equipment Fleet

Fleet	Function	Year 5	Year 10
Hole Stemmer – 3 t	Blast Hole Stemmer	2	3
Track Dozer – 430 kW	Shovel Support	6	5
Rubber Tired Dozer – 350 kW	Pit Clean Up	3	4
Fuel/Lube Truck	Shovel and Drill Fuelling & Lube	2	2
Wheel Loader Multipurpose – 14 t	Pit Clean Up	3	3
Water Truck – 20,000 gal	Haul Roads Water Truck	2	2
Track Dozer – 430 kW	Dump Maintenance	4	4
Motor Grader – 400 kW	Road Grading	4	4
Tire Manipulator	Tire Changes	3	3

MINE ANCILLARY EQUIPMENT

The mine ancillary equipment fleet is listed in Table 16.23. The fleet sizes in Year 5 and Year 10 are shown as representative of the LOM requirement.

Table 16.23 Mine Ancillary Equipment Fleet

Fleet	Function	Year 5	Year 10
Track Dozer – 430 kW	Pit Support	3	3
Float Tractor/Trailer – 189 t	Float Tractor & Trailer	1	1
Hydraulic Excavator – 6 t	Utility Excavator	2	3
Sump Pump - 1,400 gal/min	Pit Sump Dewatering	6	6
Light Plant	Lighting Plant	6	10
250 t Crane	Utility Crane	2	2
Crew Cab	Supervision and Crew Transportation	18	18
Ambulance	Ambulance	1	1
Hydraulic Excavator – 4 t	Utility Excavator	4	3
Mine Rescue Truck	Rescue Truck	1	1
Crew Bus	Crew Bus	4	4
Maintenance Truck – 1 t	Maintenance Truck	5	5
Fire Truck	Fire Truck	1	1
Screening & Crushing Plant - 12" max.	Road Crush & Stemmings	1	1
Picker Truck	Maintenance + Overhauls	2	2
Scraper – 37 t	Crush Haul for Winter Roads etc.	5	5
Crane 40 t Hydraulic Extendable	Utility Crane	2	2
Wheel Loader – 14 t	Crusher (Road Crush) Loader	1	1
Snowcat	Winter Off Road Crew Transport	6	6
40 t Crane	Utility Crane	2	3
Forklift – 30 t	Forklift	1	1
Forklift – 10 t	Forklift	2	2
Service Truck	Service Truck	5	3
Welding Truck	Welding Truck	4	4
Powerline Truck	Powerline Maintenance	2	2

A description of the equipment chosen and the tasks that the equipment performs in support of the mining operations is included in Appendix E.

Snow Fleet

All of the following snow fleet equipment is chosen to start operating 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.

The following equipment is chosen specifically for support duties of the snow fleet:

- Scrapers with the ability to haul 37 t are included in fleet. The scrapers are required to haul and spread crushed rock for traction control and remove snow from the haul roads and mine working areas as necessary. The scrapers are also used on occasion for small earthmoving jobs and reclamation projects.
- One wheel loader with an approximately 14 t bucket 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 at the crushing and screening plant.
- Six snowcats to transport operators to equipment 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 scraper for summer construction projects and stockpiling road crush, operating the snow fleet equipment outside of wintertime is not currently scheduled.

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:

- 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 foreman's office.

The recommended shop sizing for the open pit operations includes eight service bays, one welding bay, and three wash bays. This will accommodate the fleet for the LOM PFS production plan. The mine maintenance facility will also include a machine shop area, tool storage area, mine muster area, warehouse, and office complex. A separate tire bay facility will be required with an exterior heated pad to accommodate at least two trucks and a tire manipulator; the pad area should be 30 m x 30 m. A layout of the truck shop is shown in Appendix C.

16.2 Schedule Results

The summarized production schedule results are shown in Table 16.24 including both open pit and underground mining. Full results for the open pit and underground scheduling are included in Appendix E.

Table 16.24 Summarized Production Schedule – Open Pit and Underground

		Year																		
	Unit	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11 to 20	21 to 30	31 to 40	41 to 50	51 to 55	LOM
Ore from Mine To Mill	**Mt**	**-**	**-**	**-**	**18**	**28**	**34**	**43**	**35**	**41**	**30**	**1**	**47**	**25**	**390**	**278**	**98**	**126**	**-**	**1,196**
Au	g/t	-	-	-	0.997	0.846	0.823	0.785	0.795	0.866	0.967	0.771	0.756	0.871	0.627	0.572	0.264	0.225	-	0.596
Cu	%	-	-	-	0.268	0.264	0.280	0.256	0.222	0.238	0.256	0.095	0.183	0.223	0.155	0.213	0.538	0.405	-	0.248
Ag	g/t	-	-	-	2.73	3.12	2.10	1.82	1.71	2.93	4.26	4.88	3.45	3.26	3.06	1.41	1.61	0.78	-	2.25
Mo	ppm	-	-	-	23.1	19.2	31.8	72.1	84.5	50.6	27.0	77.7	40.3	44.2	71.0	44.4	-	-	-	45.3
Ore To Stockpile	**Mt**	**1**	**6.3**	**7.6**	**30**	**0**	**4**	**30.5**	**56**	**53**	**7**	**14**	**35**	**27**	**65**	**-**	**-**	**-**	**-**	**337**
Au	g/t	0.341	0.383	0.344	0.581	0.498	0.333	0.438	0.542	0.635	0.676	0.398	0.412	0.388	0.305	-	-	-	-	0.466
Cu	%	0.288	0.241	0.193	0.201	0.134	0.134	0.121	0.131	0.148	0.168	0.070	0.106	0.107	0.074	-	-	-	-	0.126
Ag	g/t	0.96	1.26	2.53	2.03	2.06	1.87	1.66	1.67	2.22	2.10	4.49	2.66	1.55	1.74	-	-	-	-	2.04
Mo	ppm	93.5	28.2	19.2	30.2	39.4	21.3	53.1	77.9	75.3	65.0	77.1	76.7	88.7	86.1	-	-	-	-	70.1
Stockpile Reclaim	**Mt**	**-**	**-**	**-**	**10.0**	**16**	**13**	**4**	**13**	**6**	**17**	**47**	**-**	**22**	**85**	**104**	**-**	**-**	**-**	**337**
Au	g/t	-	-	-	0.617	0.596	0.640	0.289	0.492	0.630	0.673	0.676	-	0.455	0.404	0.354	-	-	-	0.474
Cu	%	-	-	-	0.176	0.276	0.187	0.150	0.121	0.134	0.156	0.157	-	0.121	0.112	0.089	-	-	-	0.128
Ag	g/t	-	-	-	2.34	1.57	2.45	3.19	0.60	0.52	2.13	2.13	-	2.50	2.36	1.86	-	-	-	2.05
Mo	ppm	-	-	-	35.5	27.0	35.4	30.8	26.3	37.3	78.1	76.3	-	74.7	77.5	77.8	-	-	-	68.7
Stockpile Inventory	Mt	1	6.9	14.5	34	18	9	35	79	126	116	84	119	124	104	0	0	0	0	-
Mitchell Underground	**Mt**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**33**	**199**	**189**	**16**	**438**
Au	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.661	0.518	0.515	0.549	0.529
Cu	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.210	0.166	0.159	0.124	0.165
Ag	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.23	3.43	3.36	1.99	3.48
Mo	ppm	-	-	-	-	-	-	-	-	-	-	-	-	-	-	16.0	22.1	46.8	56.9	33.6
Iron Cap Underground	**Mt**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**96**	**98**	**0.1**	**193**
Au	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.518	0.383	0.287	0.450
Cu	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.192	0.199	0.131	0.196
Ag	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	5.30	5.33	5.23	5.32
Mo	ppm	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	19.1	26.9	18.6	23.0
Mill Feed	**Mt**	**-**	**-**	**-**	**28**	**45**	**48**	**48**	**48**	**48**	**47**	**47**	**47**	**47**	**475**	**415**	**392**	**413**	**17**	**2,164**
Au	g/t	-	-	-	0.860	0.755	0.772	0.741	0.714	0.836	0.860	0.678	0.756	0.676	0.587	0.524	0.455	0.395	0.547	0.550
Cu	%	-	-	-	0.235	0.268	0.254	0.246	0.195	0.225	0.219	0.156	0.183	0.175	0.148	0.182	0.265	0.244	0.125	0.208
Ag	g/t	-	-	-	2.59	2.55	2.20	1.94	1.41	2.62	3.48	2.19	3.45	2.90	2.93	1.83	3.43	3.04	2.01	2.74
Mo	ppm	-	-	-	27.6	22.1	32.8	68.4	69.0	48.9	45.6	76.4	40.3	58.5	72.2	50.5	15.9	27.8	56.6	44.6
Metal to the Mill																				
Au	M oz	-	-	-	0.8	1.1	1.2	1.1	1.1	1.3	1.3	1.0	1.2	1.0	9.0	7.0	5.7	5.3	0.3	38.3
Cu	M lb	-	-	-	144	264	267	258	204	236	229	163	191	184	1,544	1,664	2,295	2,218	46	9,907
Ag	M oz	-	-	-	2.3	3.7	3.4	3.0	2.2	4.0	5.3	3.3	5.3	4.4	44.8	24.4	43.3	40.4	1.1	190.8
Mo	M lb	-	-	-	1.7	2.2	3.4	7.2	7.2	5.1	4.8	8.0	4.2	6.1	75.5	46.2	13.7	25.3	2.1	212.7
Total Waste Mined	**Mt**	**29**	**46**	**54**	**135**	**142**	**147**	**127**	**66**	**75**	**116**	**128**	**88**	**65**	**523**	**917**	**347**	**207**	**64**	**3,287**
Open Pit Strip Ratio (Waste Mined/ Plant Feed)	t/t	-	-	-	4.8	3.2	3.1	2.7	1.4	1.6	2.5	2.7	1.9	1.4	1.1	2.4	3.5	1.6		2.1

Note: Waste mined in the production schedule in Table 16.24 includes re-handled waste and waste mined from borrow pit sources for construction purposes.

16.3 Underground Mining

16.3.1 Introduction

This section presents the results of the pre-feasibility assessment of the proposed block caving mines for the Mitchell and Iron Cap deposits. The complete reports by Golder, titled "Pre-feasibility Block Cave Mine Design - Mitchell Deposit" and "Pre-feasibility Block Cave Mine Design - Iron Cap Deposit", dated May 31, 2012, are included in Appendix E.

The potential of mining the Mitchell deposit by a combination of open pit and underground methods was investigated in a report titled "Block Cave Mining Study" (Golder 2011 a), which concluded that it was possible to mine the upper portions of the Mitchell deposit by open pit methods and the deeper portions by block caving. Golder was subsequently engaged to evaluate the potential to mine the Iron Cap deposit using block caving methods. The location, dimensions, and dip of the mineralized material at Iron Cap indicated that it was a suitable candidate for block caving.

The mineral resource block model used for this study contained gold, silver, copper, and molybdenum grades as well as NSR value based on the NSR formula in the "KSM PFS Update 2011" dated June 15, 2011 (Wardrop, 2011). The model also contained Measured, Indicated, and Inferred grades but the Inferred grades were set to zero and are not included in this PFS.

16.3.2 Mitchell Deposit

The Mitchell deposit extends approximately 1,500 m east-west (along strike) and 400 m to 1400 m north-south and is between approximately 300 m and 900 m in the vertical dimension. The deposit is massive, reasonably continuous, and in general, geometrically suitable to mine by block caving.

The geological resource contains 1,747 Mt of mineralized material grading 3.2 g/t Ag, 0.61 g/t Au, 0.17% Cu, and 59 ppm Mo. This resource was evaluated using Gemcom's FF software to evaluate the economic potential for a block cave mine. A footprint at elevation 235 m produced the most value and resulted in 438 Mt of block cave resources with 9% unplanned waste dilution at zero grade as shown in Table 16.25.

Table 16.25 Geological and Block Cave Resources for Mitchell

Category	Tonnes (million)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (ppm)
Geological resources[1]	1,747	3.20	0.61	0.17	59
Mineral inventory	757	3.54	0.56	0.17	50
Block cave resources from PCBC[2,3]	**438**	**3.48**	**0.53**	**0.16**	**34**
Dilution	39	0	0	0	0
Recovery	58%				
Dilution	9%				

[1] Geological resources presented in Table 1.1 of the Pre-feasibility Update report (Seabridge 2011).

[2] PCBC includes column mixing with dilution and shutting of columns (drawpoints) when NSR < $15.41 so a portion of the diluted mineral inventory is not recovered.

[3] Block cave resources can be considered as Probable Mineral Reserves within the complete pre-feasibility study report.

The quality of the rock mass at the Mitchell deposit is rated as good. No major structural features have been identified that might influence the caving mechanism and the progression of the cave in any significant manner. Cavability assessments were made using Laubscher's and Mathews' methods which involve assessing cavability based on experience at other mining operations with rock of similar quality. Both methods indicate that the size (area) of the footprint required to initiate and propagate caving is between approximately 110 m and 220 m. These dimensions are significantly smaller than the size of the footprint of the deposit that can potentially be mined economically by caving. This fact, together with the general large-sized three-dimensional shape of the deposit, suggests that the Mitchell deposit is amenable to cave mining. In situ stresses have been estimated from hydraulic fracturing tests and based on high induced stresses in the cave back, as predicted by numerical modelling, it is expected that stress-induced fracturing of the rock mass may contribute to caving. More sophisticated numerical analyses are recommended to confirm and quantify stress-related impacts as part of future studies.

A significant proportion of the rock at Mitchell is predicted to have block sizes greater than 2 m^3. Without some remediation measure being adopted, such large sized blocks will require significant secondary blasting, and there will likely be a significant adverse impact on production and significant damage to the drawpoints that will require ongoing rehabilitation. As a result of this, it is proposed to precondition the rock by hydrofracturing. The cost and schedule to do this have been incorporated into this study. There are a number of uncertainties associated with preconditioning due to the limited number of caving mines where it has been applied and tested. It is also difficult to obtain definitive field data that demonstrates the degree of improvement obtained. However, the results from these mines are encouraging, and there is sufficient experience to indicate that such fragmentation concerns do not represent a fatal flaw at Mitchell. It is recognized that uncertainty in fragmentation and the effectiveness of preconditioning to enhance fragmentation needs to be addressed via production and cost risks. It is also very difficult to quantify the effect of attrition as the rock is brought down within the cave except that experience has indicated that, in caving mines operating under similar rock conditions to those at Mitchell, fragmentation of rock drawn down more than approximately 100 m is generally good. For this study, it was assumed that fragmentation of the initial 100 m of draw height is approximately equal to the estimated in situ block size and, above this, only limited secondary blasting would be required.

The expected coarse fragmentation at Mitchell will result in relatively large isolated drawcone diameters of 13 m or more for a loading width of 5 m. The present experience in other operating mines is that a 15 m by 15 m drawpoint spacing performs well under these coarse fragmentation conditions. Some caving mines operating in good quality rock have successfully expanded the layout to approximately 17 m by 17 m but it was considered prudent for this study to adopt the slightly more conservative 15 m by 15 m spacing.

The underground mine design was based on modelling using Gemcom's FF and PCBC software. FF modelling indicated that the optimum footprint for the Mitchell deposit is approximately 728 m wide in the north-south direction, 1,022 m wide in the east-west direction, and 860 m vertically with the footprint elevation at 235 m. PCBC modelling indicated that the block cave could produce 55,000 t/d, requiring the development of 120 new drawpoints per year. The final mine design includes approximately 145 km of drifts and raises, including a 5% contingency to account for the excavations of design items such as service bays, sumps, and electrical substations. The design is composed of six main types of levels including preconditioning, undercut, extraction, secondary breakage, haulage, and conveying. In addition, there are two tunnels (access ramp and conveyor) from the footprint to surface to provide for mine access and material handling. The floors of the extraction drifts and drawpoints are designed to be concreted, which will increase the speed and productivity of the Load-Haul-Dump (LHD) vehicles as well as reduce equipment maintenance. The six levels of the mine design will be accessed through internal ramps beginning on the extraction level. These ramps are strategically positioned to maintain access to the levels during caving and for ventilation purposes. There are 34 extraction drifts on the extraction level and each drift is designed with 3 ore passes. This reduces the average LHD haul distance to approximately 100 m and improves productivity.

Production material will be hauled from drawpoints to one of three ore passes situated within the same extraction drift. The ore passes from neighbouring extraction drifts will feed a stationary rockbreaker on the secondary breaking level, which will reduce the size of the material further and feed it to the haulage level via passes with chutes. A train on the haulage level will haul the material to one of two gyratory crushers, where it will be crushed and conveyed to the surface.

The proposed mobile equipment is typical of that used in underground mines and is comprised of those pieces directly related to moving ore to the crushers (8.6 m^3 LHDs, secondary rockbreakers, and the train), the development equipment (4.6 m^3 LHDs and 18 m^3 trucks) as well as the ANFO loaders and ground support machines. In addition, service equipment is included for construction and mine maintenance activities. At peak operation, Mitchell will require a fleet of approximately 60 pieces of mobile underground equipment. The mine workforce includes both staff and labour positions and the size varies according to the stage of the mine life with a peak quantity of 489 personnel in Year 7.

The majority of the main ventilation infrastructure is also located on the extraction level. It consists of two fresh air raises, two fresh air drifts, a fresh air ring drift, multiple internal ventilation raises, a return air drift, and two exhaust raises. The conveying level starts beneath the cave and finishes on surface near to the main conveyor transporting material to the plant site. It is designed to accommodate both production ore and development waste material. The required airflow for the Mitchell mine to achieve a production rate of 55,000 t/d is 860 m^3/s based upon the diesel equipment utilized, air velocity considerations, and a contingency of 20% per level. Heating of the mine air in the winter months is included in the design and cost estimates. It is estimated that the Mitchell mine will require approximately 17,400 kWh of electricity at peak operation. The main contributors to this total are the crushers, conveyor belts, and ventilation fans.

The maximum estimated groundwater inflow for the Mitchell block cave mine is 13,200 m^3/d. At the time of completing this prefeasibility assessment, estimates by others of the surface inflows into the crater at Mitchell were not available. These surface inflows will report to the drawpoints and will be managed in a similar manner to the groundwater inflows. In future studies, the water management system will need to be enhanced to cater for this additional inflow.

The mine development schedule was separated into three phases:

- an initial pre-production phase which involves developing the primary access ramp and conveyor drifts
- an ore production phase that involves creating enough openings to start and ramp-up production from the cave
- the final phase, once the mine has reached steady-state production and the development fleet is only required to create enough openings to maintain production.

The average annual development quantity is about 4,000 m, with peak development occurring during the second phase, when about 15,000 m is required per year.

The mine production schedule was developed using Gemcom's PCBC software. It was assumed that sloughing of peripheral waste rock would occur into the crater and cover the upper surface of the material being drawn down. This was modelled in PCBC by adding an infinite supply of waste material on top of the mineralized material. As material is drawn from the drawpoints, the waste mixes with mineralized material as dilution with zero grade (unplanned dilution) and the combined material reports to the drawpoint. The PCBC analyses account for this unplanned dilution. Due to the large fragmentation that is estimated to report to the drawpoints at

Mitchell, particularly during the early stages of mining, a draw rate of 200 mm/d was chosen as a maximum cap in the PCBC analysis but an average draw rate of 108 mm/d is required to reach production targets (the maximum draw rate modelled never exceeds 165 mm/d so there are roughly twice as many drawpoints available as are required to meet production targets). Initially, it is assumed that a drawpoint can produce at 60 mm/d and that this will steadily increase until 50% of a column is mined. Then, the drawpoint will produce up to the set maximum of 200 mm/d. Mitchell is estimated to have a production ramp-up period of 6 years, steady state production at 20 Mt/a for 14 years, and then ramp-down production for another 7 years.

16.3.3 IRON CAP

This section presents the results of the pre-feasibility assessment of the proposed block caving mine for the Iron Cap deposit. The deposit extends approximately 1,200 m SW-NE (along strike), 700 m NW-SE, and 700 m vertically. It is understood that the deposit remains open at depth. Open pit mining methods were used to evaluate the mining potential of this deposit as part of the KSM PFS Update 2011 (Wardrop, 2011). However, due to various environmental concerns such as pre-stripping the overlying ice cap and creating additional PAG waste, Seabridge decided to assess other mining options. Golder was engaged to evaluate the potential to mine the Iron Cap deposit using block caving methods to the pre-feasibility level of engineering study. The location, dimensions, and dip of the mineralized material at Iron Cap indicated that it was a suitable candidate for block caving.

The geological resource contains 362 Mt of mineralized material grading 5.4 g/t Ag, 0.44 g/t Au, 0.21% Cu, and 37 ppm Mo. This resource was evaluated using Gemcom's FF software to evaluate the economic potential for a block cave mine. A footprint at elevation 1210 m produced the most value and resulted in approximately 193 Mt of block cave resources, including 5% unplanned waste dilution at zero grade as shown in Table 16.26.

Table 16.26 Geological and Block Cave Resources for Iron Cap

Category	Tonnes (millions)	Ag (g/t)	Au (g/t)	Cu (%)	Mo (ppm)
Geological Resources[1]	362	5.4	0.44	0.21	37
Mineral Inventory	321	6.3	0.52	0.23	23
Block Cave Resources[2,3]	**193**	**5.3**	**0.45**	**0.20**	**21**
Dilution	16	0	0	0	0
Recovery	60%				
Total Dilution	5%				

[1] Geological Resources presented in Table 1.1 of the Pre-feasibility Update report (Seabridge 2011).

2 PCBC includes column mixing with dilution and shutting of columns (drawpoints) when NSR < $15.41 so a portion of the diluted mineral inventory is not recovered.

3 Block Cave Resources can be considered as Probable Mineral Reserves within the complete pre-feasibility study report.

The Iron Cap deposit appears to be composed of strong, moderately fractured rock. Rock quality variations are most commonly attributed to variations in fracture frequency as the strength of the rock mass does not vary significantly within the deposit. The fracture frequency is higher for Iron Cap than the nearby Mitchell deposit, resulting in a corresponding lower predicted median in situ block size of 2.5 m^3 compared to approximately 6 m^3 for the Mitchell deposit. No major structural features have been identified that might influence the caving mechanism and the progression of the cave in any significant manner. There are several gaps in data that have been identified in the geotechnical and hydrogeological studies, which will need to be addressed as part of future studies.

The cavability assessments made using Laubscher's and Mathews' methods indicate that the size (diameter) of the footprint required to initiate and propagate caving is between approximately 100 m and 220 m. This footprint size is significantly smaller than the size of the footprint of the deposit that can potentially be mined economically by caving. This fact, together with the general large-sized, continuous nature of the deposit, suggests that the Iron Cap deposit is amenable to cave mining. There have been no fracture propagation assessments applicable to preconditioning designs or in situ stress interpretations developed at the Iron Cap deposit. Measurements carried out in the Mitchell deposit may not accurately reflect the fracture propagation and stress environment at Iron Cap because of the effects of surface topography. Future drilling programs should include hydraulic fracturing tests.

A significant proportion of the rock at Iron Cap is predicted to have block sizes greater than 2 m^3. Without some remediation measure being adopted, such large sized blocks will require significant secondary blasting, and there will likely be a significant adverse impact on production and significant damage to the drawpoints that will require ongoing rehabilitation. As a result of this, it is proposed to precondition the rock by hydrofracturing. The cost and schedule to do this have been incorporated into this study. There are a number of uncertainties associated with preconditioning due to the limited number of caving mines where it has been applied and tested. It is also difficult to obtain definitive field data that demonstrates the degree of improvement obtained. However the results from these mines are encouraging, and there is sufficient experience to indicate that such fragmentation concerns do not represent a fatal flaw at Iron Cap. It is recognized that uncertainty in fragmentation and the effectiveness of preconditioning to enhance fragmentation needs to be addressed via production and cost risks. It is also very difficult to quantify the effect of attrition as the rock is brought down within the cave except that experience has indicated that in caving mines operating under similar rock conditions to those at Iron Cap, fragmentation of rock, drawn down more than approximately 100 m is generally good. For this study, it was assumed that fragmentation of the initial 100 m of draw height is approximately equal to the estimated in situ block size and, above this, only limited secondary blasting would be required.

The expected coarse fragmentation at Iron Cap will result in relatively large isolated drawcone diameters of 13 m or more for a loading width of 5 m. The present experience in other operating mines is that a 15 m by 15 m drawpoint spacing performs well under these coarse fragmentation conditions. Some caving mines operating in good quality rock have successfully expanded the layout to approximately 17 m by 17 m, but it was considered prudent for this study to adopt the slightly more conservative 15 m by 15 m spacing.

The Iron Cap block cave design was based on modelling from FF and PCBC software. FF modelling indicated that the optimum footprint for the Iron Cap deposit is at an elevation of 1,210 m. It is approximately 545 m wide in the north-south direction, 570 m wide in the east-west direction, and has an average depth of 400 m. PCBC modelling indicated that the block cave could produce 15 Mt/a, requiring development of 120 new drawpoints per year. The mine design requires approximately 64 km of drifts and raises, including a 5% contingency to account for the excavations of detailed design items such as service bays, sumps, and electrical substations. Four main levels are required to cave the Iron Cap deposit and include the preconditioning level, undercut level, extraction level, and conveying level. The design also includes a return air drift located below the conveying level. The floors of the extraction drifts and drawpoints are designed to be concreted, which will increase the speed and productivity of the LHD vehicles as well as reduce equipment maintenance. Personnel, material, and supplies will access the Iron Cap mine through a drift driven from the MTT. A conveyor drift will be driven parallel to the access ramp, and the two will be connected every 300 m to provide emergency egress and a ventilation loop during construction. The total length of the access ramp is 3.4 km. Two fresh air portals and one exhaust portal are planned on the north slope of the Mitchell Valley. These tunnels may act as an alternative access to the underground from the surface in case of emergency. The fresh air tunnels will connect to surface, and a perimeter drift will be constructed around the entire mine footprint to provide fresh air to the mine workings.

Production material will be hauled directly from the drawpoints to one of four gyratory crushers installed on the extraction level perimeter drift. The crushed material will be transported by one of two conveyor belts which both feed a conveyor that will transport the production material to a 2,000 t surge bin located above the MTT conveyor.

The proposed mobile equipment is typical of that used in underground mines and is comprised of those pieces directly related to moving ore to the crushers (8.6 m^3 LHDs and secondary rockbreakers), the development equipment (4.6 m^3 LHDs and 18 m^3 trucks) as well as the ANFO loaders and ground support machines. In addition, service equipment is included for construction and mine maintenance activities. At peak operation, Iron Cap will require a fleet of approximately 67 pieces of mobile underground equipment. The Iron Cap mine workforce includes both staff and labour positions and the size varies according to the stage of the mine life with a peak quantity of 548 personnel in Year 7.

The required airflow for the Iron Cap mine is 526 m^3/s, based upon the total diesel equipment used on each mining level including a minimum 20% contingency. The Iron Cap ventilation model is designed to operate as a positive pressure or forced air system to facilitate mine air heating during the winter months and to prevent any air being drawn into the mine through the caved material. Heating of mine air in the winter months is included in the design and cost estimates for Iron Cap. It is estimated that the Iron Cap mine will require approximately 9,200 kWh of electricity at peak operation. The main contributors to this total are the crushers, conveyor belts, and ventilation fans.

The underground water management system at Iron Cap is currently designed to handle 7,640 m^3/d. This caters for the groundwater inflow and the ice melt. At the time of completing this pre-feasibility assessment, estimates by others of the surface inflows into the crater at Iron Cap were not available. These surface inflows will report to the drawpoints and will be managed in a similar manner to the groundwater inflows. In future studies, the water management system will need to be enhanced to cater for this additional inflow. To provide for good drainage, the underground drifts have been graded so that water will run towards the MTT, or towards the Mitchell Valley. Water exiting the mine will be collected and processed in existing contact water facilities.

The mine development schedule was separated into three phases:

- an initial pre-production phase, which involves developing the primary access ramp and conveyor drifts
- an ore production phase that involves creating enough openings to start and ramp-up production from the cave
- the final phase, once the mine has reached steady-state production and the development fleet is only required to create enough openings to maintain production.

The average annual development quantity during the peak development period is about 10,000 m/a.

The mine production schedule was developed using Gemcom's PCBC software. It was assumed that sloughing of peripheral waste rock would occur into the crater and cover the upper surface of the material being drawn down. This was modelled in PCBC by adding an infinite supply of waste material on top of the mineralized material. As material is drawn from the drawpoints, the waste mixes with mineralized material as dilution with zero grade (unplanned dilution) and the combined material reports to the drawpoint. The PCBC analyses account for this unplanned dilution. Due to the large fragmentation that is estimated to report to the drawpoints at Iron Cap, particularly during the early stages of mining, a draw rate of 200 mm/d was chosen as a maximum cap in the PCBC analysis but an average draw rate of 108 mm/d is required to reach production targets (the maximum draw rate modelled never exceeds 165 mm/d, and averages about 110 mm/d, so there are roughly twice as many drawpoints available as are required to meet production targets). Initially, it is assumed that a drawpoint can produce at 60 mm/d and that this will steadily increase until 50% of a column is mined. Then, the drawpoint will produce up to the set maximum of 200 mm/d. Iron Cap is estimated to have a production ramp-up period of 4 years, steady state production at 15 Mt/a for 9 years, and then ramp-down production for another 6 years.

16.4 PIT SLOPE DESIGN ANGLES

16.4.1 *OVERVIEW*

BGC has provided open pit slope design parameters for the three proposed open pits of the KSM gold-copper porphyry project: Kerr, Sulphurets, and Mitchell. BGC has undertaken geotechnical site investigations, utilized available local and regional geological data, and utilized well-established geotechnical design methods to estimate the design pit slope angles for the proposed open pits.

BGC has identified geotechnical rock mass units associated with the primary rock and alteration types, based on the results of the site investigation and geological interpretations by Seabridge. Major geological structures (faults and foliation) have been included in the geotechnical slope stability analyses for each pit. Slope stability analyses were conducted using industry standard limit-equilibrium software, finite element analysis software, and in-house proprietary BGC tools.

BGC completed hydrogeological studies for each of the proposed pits, and numerical simulations of pit dewatering/depressurization have been carried out. BGC interpreted hydrostratigraphic units, estimated hydraulic conductivity and storage parameter values, and formulated a conceptual hydrogeologic model for the project area. The conceptual model was used as the basis for developing a numerical hydrogeologic model. The calibrated numerical model was used to evaluate the effort required to depressurize the open pit slopes to satisfy geotechnical constraints identified in the open pit slope designs. Preliminary dewatering/depressurization plans, including the number of vertical wells, horizontal drains, and the extraction rates required to achieve sufficient depressurization of the rock mass were developed to support the costing study. In addition, the need for a dewatering adit and associated drainage gallery was identified and simulated to achieve the depressurization targets of the upper north slope of the Mitchell pit.

BGC reviewed the proposed pit areas and surrounding terrain for potential geohazards, including the identification of snow avalanche paths and potential landslides, utilizing aerial photographs and satellite imagery. BGC completed ground-truthing of potential geohazards; the preliminary design of mitigation structures were completed by those responsible for the various project facilities at risk from the identified geohazards.

16.4.2 MITCHELL PIT DESIGN

The proposed Mitchell pit will be located within a glacially modified valley and targets a mineral deposit located in the valley floor, resulting in 1,200 m high ultimate slopes. This scale of the Mitchell pit north and south slope heights is greater than any previously achieved in the industry; a small number of currently operating mines have slopes approaching 1,000 m high.

A multi-component site investigation program was completed to provide data for the Mitchell pit design work. Approximately 4,100 m of geotechnical drilling was completed, distributed over 10 core holes. BGC geotechnically logged all holes. Optical and acoustic televiewer surveys were completed in each hole to provide geological discontinuity orientations for rock slope design. Packer testing was undertaken in each hole, and vibrating wire piezometers were installed. Photogrammetric mapping of sections of the north and south valley walls was completed to provide additional data on the rock mass fabric of the study area.

A laboratory testing program was completed, consisting of the following tests:

- uniaxial compressive strength (16 tests)
- Brazilian tensile strength (31 tests)
- small scale direct shear testing (8 tests)
- grain size and index testing (4 tests)
- specific gravity (44 tests).

An appropriate quantity of good quality data was collected to characterize the geological units of the study area and support slope designs.

The structural geology of the Mitchell study area is defined by major faults, foliation, and rock mass fabric (joints, etc.). The Sulphurets and Mitchell Thrust faults dip approximately 30° toward the west. Sets of west and east dipping normal faults, dipping approximately 60°, are observed in the study area. The east dipping normal faults are interpreted to be associated with the Brucejack Fault, which is mapped on a regional scale but does not occur in the pit area. Foliation is best developed in the phyllic altered rock mass in the footwall of the MTF. The foliation dips moderately to steeply (45° to 80°) north. Additional discontinuity sets have also been identified from the site investigation results. The proposed Mitchell pit has been divided into four geotechnical domains, based on the different structural geology fabrics in the area; discontinuity sets and geotechnical units for each domain are identified for use in the slope designs. Design sectors are based on the anticipated main orientations of the proposed pit walls, as determined from previous pit optimization studies.

Recommended inter-ramp slope angles vary from 34° to 54° based on wall orientation, overall wall height, geotechnical domain, and controls on slope stability. Inter-ramp slope heights are limited to 150 m, after which a geotechnical berm (or ramp) with a minimum width of 20 m is required. All final pit slopes are assumed to be excavated using controlled blasting. Depressurization of the proposed pit slopes requires a combination of vertical wells, horizontal drains, and a dewatering adit with drainage galleries. The east and west overall slopes of the proposed Mitchell pit are within the range of slope heights that have been achieved in other porphyry metal mines in the world. The heights of the north and south slopes, when existing topography is included in the height estimates, are beyond the current experience of the open pit mining industry.

The Mitchell open pit slope designs are outlined in Table 16.27.

Table 16.27 Mitchell Zone Pit Slope Design Parameters

Domain	Design Sector	Slope Azimuth		Catch Bench Geometry			Inter-ramp Geometry		Slope Design Control
		Start (°)	End (°)	Height Bh (m)	Angle Ba (°)	Width Bw (m)	Height Irh (m)	Angle IRA (°)	
I	I-173	135	210	30	60	24.7	150	36	Benchstack (B1 - P)
	I-220	210	230	30	70	25.2	150	40	Benchstack (B1 - B3)
	I-240	230	250	30	70	15.6	150	48	Benchstack (B1 - B3)
	I-275	250	300	30	70	11.6	150	53	Benchstack (B1 - B3)
	I-338	300	015	30	70	11.6	150	53	Rockmass stability
	I-028	015	040	30	70	11.6	150	53	Rockmass stability
	I-078	040	115	30	70	15.6	150	48	Benchstack (A1 - B3)
	I-125	115	135	30	60	11.5	150	46	Benchstack (Bench geometry)
II	II-325	270	020	30	70	11.5	150	53	Rockmass stability
	II-035	020	050	30	70	17.8	150	46	Benchstack (A3-E1)
	II-058	050	065	30	70	25.2	150	40	Benchstack (A3-E1)
	II-078	065	090	30	70	31.0		36	Benchstack (A3-E1)
III	III-099	090	108	30	70	10.5	150	54	Benchstack (Bench geometry)
	III-138	108	168	30	70	34.3	150	34	Benchstack (B2-P)
	III-189	168	210	30	70	17.8	150	46	Rockmass stability
IV	IV-168	145	190	30	70	17.8	150	46	Benchstack (A1-B1)
	IV-200	190	210	30	70	26.6	150	39	Benchstack (B1-D1)
	IV-240	210	270	30	70	34.3	150	34	Benchstack (B1-D1)
	IV-003	325	040	30	70	17.8	150	46	Benchstack (F1-D1 / E1-A1)

Notes:

1. Geotechnical berms (minimum 20 m wide) must be added to the slopes every 150 m.
2. No ramp allowances have been included in these slope designs; their addition will reduce the achievable overall angles.

16.4.3 SULPHURETS PIT DESIGN

The proposed Sulphurets pit will be located on a glacially modified ridge between the Mitchell and Sulphurets valleys. The proposed mine plan would result in ultimate pit slopes with maximum heights of approximately 650 m, and a footprint of approximately 2 km x 1 km, with the long axis of the pit trending parallel to the strike of the STF.

A site investigation program including geotechnical drilling and hydrogeological testing was completed in 2010. Data from five geotechnical drill holes (consisting of approximately 1,950 m of drilling) was used to divide the Sulphurets Zone into three geotechnical domains: the hanging wall of the STF, the footwall of the STF, and an altered (crackled) zone associated with and defined by the STF. The STF dips approximately 30° toward the west. Sets of west and east dipping normal faults dipping approximately 60° are also dominant in this zone. Foliation in the Sulphurets Zone is well developed in the altered rock mass of the STF footwall, and dips moderately to steeply (45° to 80°) north. Additional joint and bedding sets have also been identified.

Laboratory testing of core samples from the completed geotechnical drilling included:

- uniaxial compressive strength (13 tests)
- Brazilian tensile strength (20 tests)
- small scale direct shear tests of natural discontinuities (5 tests)
- index testing of discontinuity infilling material (3 tests).

The rocks of the Sulphurets Zone are typically moderately strong when weathered, and strong when fresh. The RQD of the rocks of the Sulphurets Zone varies from fair to good, generally increasing in quality with depth below surface or distance from the STF.

The slope designs estimated by BGC assume final walls will be excavated using controlled blasting, consistent with the approach proposed for the Mitchell pit. The recommended inter-ramp slope angles vary from 36° to 50° based on wall orientation, overall wall height, rock mass quality, and structural controls on slope stability. Inter-ramp slope heights are limited to 150 m after which a geotechnical berm (or ramp) with a minimum width of 20 m is required. Depressurization of the pit slopes is required and should be achievable with a combination of vertical wells and horizontal drains.

Table 16.28 outlines the Sulphurets open pit slope designs.

Table 16.28 Sulphurets Zone Pit Slope Design Parameters

Domain	Design Sector	Slope Azimuth Start (°)	Slope Azimuth End (°)	Catch Bench Geometry Height Bh (m)	Catch Bench Geometry Angle Ba (°)	Catch Bench Geometry Width Bw (m)	Inter-ramp Geometry Height Irh (m)	Inter-ramp Geometry Angle Ira (°)	Slope Design Control
SHW-V	SHW-V-280	270	290	30	65	11.8	150	49	Benchstack (MC1-T)
	SHW-V-323	290	355	30	65	21.3	150	40	Benchstack (F1-T)
	SHW-V-028	355	060	30	65	16.3	150	45	Benchstack (FO-T)
	SHW-V-075	060	090	30	65	27.2		36	Benchstack (STF - P)
SFW-C	SFW-C-265	220	310	30	65	16.3		45	Benchstack (MC1,MC2 - T)
	SFW-C-333	310	355	30	65	11.8		49	Benchstack (B1,B2-T)
	SFW-C-015	355	035	30	65	11.5		50	Benchstack (Bench geometry)
	SFW-C-045	035	055	30	65	16.3		45	Benchstack (A1-STF)
	SFW-C-070	055	085	30	65	21.3		40	Benchstack (A1-STF)
SFW-V	SFW-V-190	172	207	30	65	21.3	150	40	Benchstack (B1-P)
	SFW-V-222	207	237	30	65	14.0	150	47	Benchstack (A1-T)
	SFW-V-269	237	300	30	65	25.7	150	37	Benchstack (MC-T)
	SFW-V-333	300	005	30	65	21.3	150	40	Benchstack (FO-T)
	SFW-V-033	005	060	30	65	27.2	150	36	Benchstack (A4-D1)
	SFW-V-090	060	120	30	65	21.3	150	40	Benchstack (FO-A3)
	SFW-V-146	120	172	30	65	27.2	150	36	Benchstack (B1-A4)

Notes:

1. Geotechnical berms (minimum 20 m wide) must be added to the slopes every 150 m.
2. No ramp allowances have been included in these slope designs; their addition will reduce the achievable overall angles.

16.4.4 KERR PIT DESIGN

The proposed Kerr open pit is located on the south side of the Sulphurets Valley near the height of land and above the Sulphurets Glacier. The proposed mine plan will result in ultimate pit slopes approximately 600 m high, with a proposed pit footprint of approximately 2 km x 0.5 km.

A site investigation program including four geotechnical drill holes (consisting of approximately 1,500 m of drilling) and hydrogeological testing was completed in 2010. Data from the site investigation was used to divide the Kerr Zone into two geotechnical domains: a central altered zone and a surrounding unaltered zone; both are composed primarily of volcanic rocks. The structural geology of the Kerr Zone includes sets of west and east dipping normal faults (dipping greater than 60°) as well as bedding and joints.

Laboratory testing of core samples from the geotechnical drilling included:

- uniaxial compressive strength (10 tests)
- Brazilian tensile strength (14 tests)
- small scale direct shear tests of natural discontinuities (4 tests)
- index testing of discontinuity infilling material (3 tests).

The rocks of the altered zone are typically medium-strong, but are highly fractured with poor RQD values. The rocks of the unaltered zone are strong to very strong, with good to excellent RQD values.

The slope designs estimated by BGC assume that final walls will be excavated using controlled blasting. The recommended inter-ramp slope angles vary from 34° to 50°; based on overall wall height, wall azimuth, rock mass quality, and geological structures. Inter-ramp slope heights are limited to 150 m after which a geotechnical berm (or ramp) with a minimum width of 20 m is required. Depressurization of the pit slopes is required and should be achievable with a combination of vertical wells and horizontal drains.

Kerr open pit slope designs are presented in Table 16.29.

Table 16.29 Kerr Zone Pit Slope Design Parameters

Domain	Design Sector	Slope Azimuth		Catch Bench Geometry			Inter-ramp Geometry		Slope Design Control
		Start (°)	End (°)	Height Bh (m)	Angle Ba (°)	Width Bw (m)	Height Irh (m)	Angle Ira (°)	
KVOL	KVOL-236	180	292	30	65	11.5	150	50	Benchstack (Bench geometry)
	KVOL-335	292	017	30	65	27.2	150	36	Benchstack (F2 - T)
	KVOL-065	017	112	30	65	30.5	150	34	Benchstack (Bed3,4 - T)
	KVOL-126	112	140	30	65	21.3	150	40	Benchstack (H1 - T)
	KVOL-160	140	180	30	65	16.3	150	45	Benchstack (B1 - Bed4)
KALT	KALT-180	135	225	30	60	24.7	150	36	Rockmass stability
	KALT-000	225	135	30	60	24.7		36	Benchstack (Rockmass stability)

Notes:

1. Geotechnical berms (minimum 20 m wide) must be added to the slopes every 150 m.
2. No ramp allowances have been included in these slope designs; their addition will reduce the achievable overall angles.

16.4.5 SLOPE DESIGN IMPLEMENTATION

Achieving the proposed design criteria will require important considerations during mine operations. Depressurization of the pit walls will be required through the use of vertical wells and horizontal drains. Geological structures may affect bench and inter-ramp scale slope stability and therefore depressurization of these structures will be required.

Based on groundwater modelling results, approximately 76 in-pit wells will be required over the life of mine for the Mitchell pit. The total drilling length for the vertical wells is estimated to be approximately 15,200 m. In addition, it is estimated that approximately 628 km of horizontal drains will be required to aid in depressurization of the pit slopes over the mine life. The average annual groundwater extraction rate for Mitchell pit is predicted to be approximately 11,980 m^3/d throughout the life of the pit: 6,580 m^3/d will be captured by vertical wells, with 4,460 m^3/d captured by the 3.5 km-long adit and drainage gallery, and the remaining 940 m^3/d captured by horizontal drains.

The average annual groundwater extraction rate for the Kerr pit is estimated to be approximately 1,200 m^3/d: 740 m^3/d will be captured by vertical in-pit wells, while the remaining 460 m^3/d will be captured by horizontal drains. Approximately 36 vertical wells with a total drilling length of 7,200 m will be required throughout the life of the pit. In addition, it is estimated that approximately 108 km of horizontal drains will be required to aid in depressurization of the pit slopes over the life of the pit.

The average annual flow to the Sulphurets pit is estimated to be 1,010 m^3/d; 890 m^3/d will be captured by vertical in-pit wells, and the remaining 120 m^3/d will be captured by horizontal drains. Approximately 30 vertical wells with a total drilling length of 6,000 m will be required throughout the life of the pit. In addition, it is estimated that approximately 166 km of horizontal drains will be required to aid in depressurization of the pit slopes over the life of the pit.

The efficiency of the proposed pit dewatering system is sensitive to the hydraulic properties of the bedrock. It is important to continue to characterize the hydraulic properties of the bedrock as the Project advances. Current rock mass hydraulic conductivity estimates in the vicinity of the open pits are limited to point-scale measurements (e.g. slug tests and constant rate packer injection tests during drilling). Larger-scale estimates of rock mass hydraulic conductivity and storage properties (i.e. airlifting tests and pumping tests) to confirm the feasibility of the proposed depressurization system, should be obtained at the Feasibility Study stage of the Project. Dewatering and depressurization response must be monitored throughout mining operations to determine if targets are being met. An extensive monitoring network of piezometers (standpipe and vibrating wire) should be in place and integrated with the open pit slope monitoring system.

Monitoring of pit slope displacements at various scales will be required. Inter-ramp and overall scale slopes should be monitored for deformations. The slope deformation monitoring system designed for the Mitchell pit would meet or exceed the size and complexity of those currently in operation at other large open pits such as Goldstrike, Las Pelambres, Palabora, Bougainville, and Bingham Canyon. The monitoring system should include multiple robotic-theodolites and survey prisms, mobile slope stability radar units, slope inclinometers, piezometers, and extensometers. The system would be computerized and use radio telemetry or a similar technology to provide real-time data to on-site geotechnical and mining staff. Similar monitoring systems would also be required for the Sulphurets and Kerr pits; the requirements of those systems would be scaled according to the proposed wall heights for those pits.

It will be important to manage geological hazards during mining operations. Additional engineered structures adjacent to the pit, or modifications to the pit slope geometry, may be required to mitigate the risk of snow avalanches. In addition, the KSM project area has been recently de-glaciated and large scale slope deformation features have been identified in the Mitchell and Sulphurets valleys.

Finally, there are two large landslides in the vicinity of the Project that will require appropriate mitigation plans at future study stages. The Snowfield landslide is situated on the south slope of the Mitchell Valley and east of the proposed Mitchell pit, and the Kerr landslide is situated on the south slope of the Sulphurets Valley and below the elevation of the proposed Kerr pit.

17.0 RECOVERY METHODS

17.1 INTRODUCTION

The proposed KSM plant will have an average process rate of 130,000 t/d. The process plant will receive ore from the Mitchell, Kerr, Sulphurets, and Iron Cap deposits. The Mitchell deposit will be the dominant resource of mill feed for the process plant and will supply mill feed throughout the projected LOM, except for Years 24 and 25. The ore from the Sulphurets deposit will be fed to the plant together with the ore from the Mitchell pit from Years 2 to 6 and Years 23 to 30. The ore from the Kerr deposit, together with the Mitchell mineralization, will be introduced to the plant from Years 27 to 50, while Iron Cap ore will be fed to the process plant from Years 32 to 51.

A combination of conventional flotation and cyanidation processes are proposed for the Project. The process plant will consist of three separate facilities:

- an ore primary crushing and handling facility at the Mitchell mine site
- an ore conveyance transportation system through the MTT
- a main process facility at the plant site at the Treaty OPC area, including secondary/tertiary crushing, primary grinding, flotation, regrinding, leaching, and concentrate dewatering.

The primary crushing facility located at the Mitchell mine site will reduce the run-of-mine (ROM) particle size to approximately 80% passing 150 mm by gyratory crushers. Ore from the Sulphurets and Kerr deposits will be crushed at their respective sites, excluding the Sulphurets ore produced during Years 2 to 6, which will be crushed at the Mitchell site. The Iron Cap mineralization will be mined by block caving and be crushed in the underground mine prior to being conveyed and loaded onto the MTT conveyor. The crushing circuit at the Mitchell site will include:

- primary crushing by two 60” by 89” gyratory crushers
- crushed ore transport conveyors.

The crushed ore will be transported by a conveyance system through the MTT to the main plant site located at the Treaty OPC site, approximately 23 km northeast of the mine site. The main process plant will consist of the following process facilities:

- secondary crushing by cone crushers
- tertiary crushing by HPGR

- primary grinding by ball mills
- copper-gold/molybdenum bulk flotation
- copper-gold/molybdenum separation depending on molybdenum grade of mill feed
- copper-gold concentrate and molybdenum concentrate dewatering
- gold CIL cyanide leaching of scavenger cleaner tailing and pyrite rougher concentrate
- gold recovery
- cyanide recovery, and then cyanide destruction of washed CIL residue prior to disposal of the residue in the lined pond within the TMF.

The TMF, located southeast of the main process plant, is designed to store flotation tailing and cyanide leach tailing, which will be stored in a lined pond within the TMF.

The mill feed produced from the Mitchell crushing facility or from the Iron Cap block caving site will be conveyed by the MTT overland conveyer to the coarse ore stockpile at the Treaty OPC site. A stockpile will be located at the exit portal of the MTT tunnel and will have a live capacity of 60,000 t. The coarse ore will be reclaimed and be further crushed by four cone crushers and then four HPGRs in closed circuit with vibrating screens.

The screen undersized material will be fed to four ball mills in closed circuit with hydrocyclones. Ore solids will be reduced to a particle size of 80% passing 125 to 150 µm.

The products from the primary grinding circuits will feed four trains of copper-gold/molybdenum rougher/scavenger flotation circuits. The copper rougher flotation concentrates from the flotation circuits will be 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 with three stages of copper cleaner flotation producing a copper-gold or copper-gold/molybdenum concentrate with an average grade of 25% Cu. Depending on the molybdenum content in the copper-gold/molybdenum concentrate, the bulk concentrate may be treated by flotation to produce a molybdenum concentrate and a copper-gold concentrate. The molybdenum concentrate will be leached using the Brenda Mines procedure to reduce copper and lead contents.

The final copper concentrate(s) will be dewatered by a combination of thickening and pressure filtration to 9% moisture before being transported the Stewart port site for ship loading and delivery to copper smelters, while the molybdenum concentrate will be further dried prior to being shipped in bags to the port at Prince Rupert for delivery to molybdenum smelters.

The copper-gold/molybdenum rougher scavenger flotation tailing will be subjected to further flotation producing a gold-bearing pyrite concentrate. The final pyrite flotation tailing will be sent to the TMF for storage. The pyrite concentrate will be reground in tower mills to a particle size of 80% passing 20 µm.

The reground gold-pyrite concentrate and the first copper cleaner tailing from the copper-gold/molybdenum cleaner flotation circuit will be separately leached in a CIL cyanidation plant to recover the contained gold. The sulphide pulp will be pre-oxidized by aeration prior to cyanidation. Dissolved gold will be adsorbed onto activated carbon in the CIL circuit.

The loaded carbon from the two streams will be combined and gold stripped from the carbon by a conventional Zadra pressure stripping process, and the 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 produced from the electrowinning circuit will be smelted using a conventional pyrometallurgical technique to produce gold-silver doré bullion.

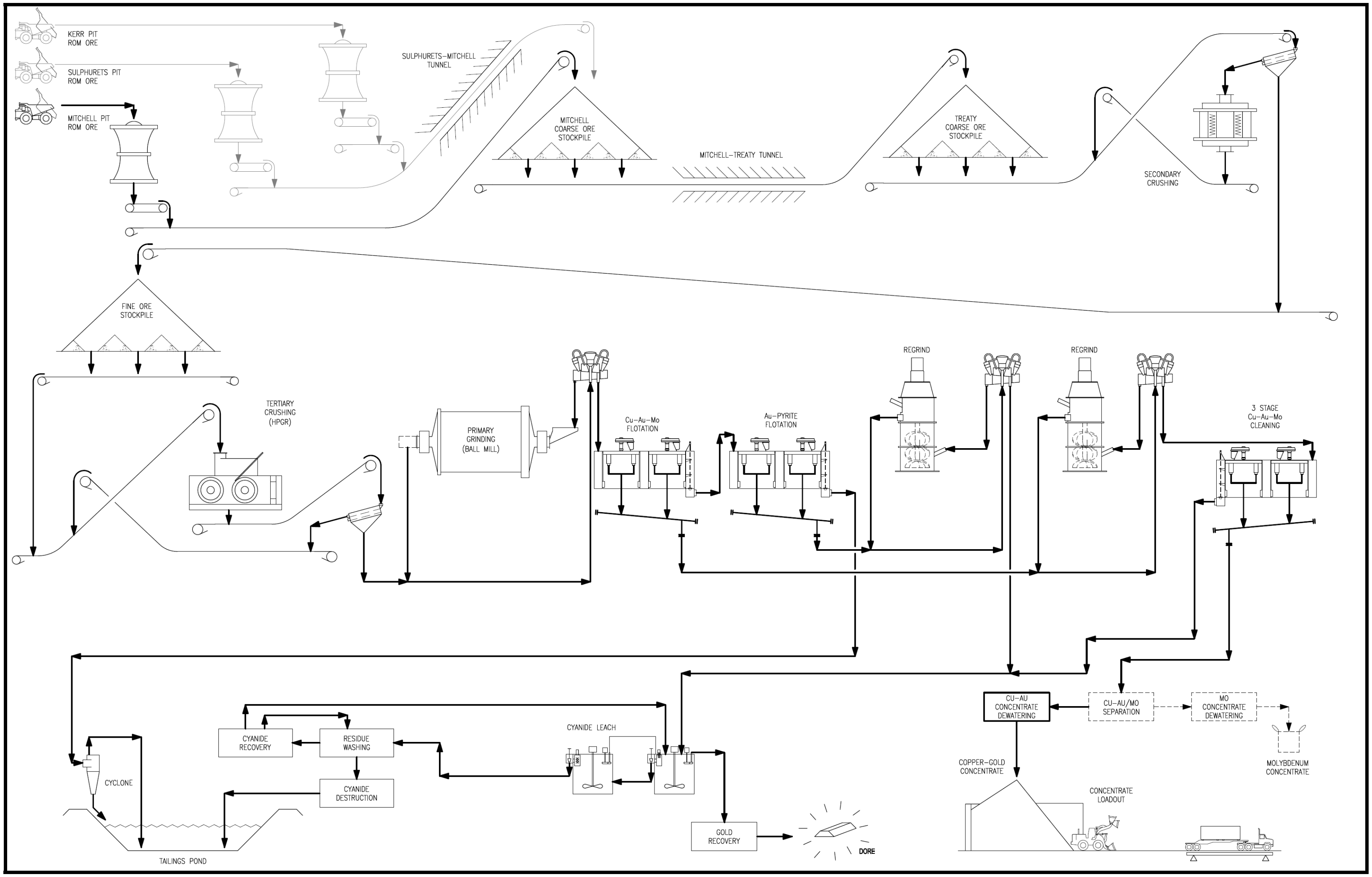
The residues from the leach circuit will be pumped to a conventional counter-current decantation (CCD) washing circuit. The solution from the circuit will be sent to a cyanide recovery circuit using a SART process, followed by an AVR process.

The washed residues will be treated by an SO_2/air cyanide destruction process to destroy the remaining WAD cyanide. The treated residues will then be transported by pipeline to a lined pond within the TMF. The sulphide leach residues will be stored under water at all times to prevent the oxidation of sulphides.

These processes are shown in the simplified flowsheet in Figure 17.1 and are detailed in the following sections.

Detailed process flowsheets, and general site and plant layouts are available in Appendix C.

Figure 17.1 Simplified Process Flowsheet



17.2 MAJOR PROCESS DESIGN CRITERIA

The concentrator is designed to process an average of 130,000 t/d. The major criteria used in the design are shown in Table 17.1.

Table 17.1 Major Design Criteria

Criteria	Unit	Value
Daily Process Rate	t/d	130,000
Operating Year	day	365
Primary/Secondary Crushing		
Availability – Primary Crushing	%	70
Availability – Secondary Crushing	%	85
Primary Crushing Product Particle Size, P_{80}	µm	150,000
Secondary Crushing Product Particle Size, P_{80}	µm	45,000
HPGR/Grind/Flotation/Leach		
Availability	%	94
Milling and Flotation Process Rate	t/h	5,762
Mill Feed Size, P_{80}	µm	2,000
Primary Grind Size, P_{80}	µm	125-150
Bond Ball Mill Wi - Design	kWh/t	16
Bond Abrasion Index	g	0.293
Concentrate Regrind Size, 80% Passing		
Cu/Au Rougher/Scavenger Concentrate	µm	20
Au-Pyrite Concentrate	µm	20
Gold-bearing Materials Leach Method		CIL
Feed Mass to CIL Circuit	t/d	14,600

The complete design criteria are detailed in Appendix D.

17.3 PROCESS PLANT DESCRIPTION

17.3.1 PRIMARY CRUSHING

There will be four primary crushing sites throughout the project life, including at the Mitchell, Kerr, and Sulphurets sites, as well as in the underground block caving mine. At the Mitchell site, primary crushing will consist of two 60″ by 89″ gyratory crushers, two apron feeders, and one stockpile feed conveyor. The ROM material feed to the gyratory crushers will be from the Mitchell pit and the initial 5-year Sulphurets pit, and will be approximately 80% passing 1,200 mm. The oversize materials will be broken by a rock breaker. The gyratory crushers will reduce the ROM to a particle size of 80% passing 150 mm or less. The products from each gyratory crusher will be fed to one 1.83 m wide by 37 m long conveyor via one 2.13 m wide by 10 m long apron feeder. The crushed ore from the two conveyors will be fed to a 2.13 m wide by 495 m long coarse ore stockpile feed conveyor, which will carry the ore to a 30,000 t live capacity covered stockpile.

The Sulphurets ore will supplement the mill feed between Years 2 to 6 and Years 23 through 30. The ROM ore from the Sulphurets pit will be trucked to the Mitchell site during Years 2 to 6 and crushed at the Mitchell crushing facility. The ore produced after Year 23 will be crushed by a 60″ by 89″ gyratory crusher at the Sulphurets mine site. The crushed ore will be conveyed to the Mitchell site via the 3.0 km SMCT to a 10,000-t Sulphurets/Kerr coarse ore stockpile.

The ore from the Kerr deposit will be processed with the ore from the Mitchell deposit or from the Iron Cap underground mine between Years 27 and 50. The ROM ore and waste rock from the Kerr pit will be crushed by two 60″ by 89″ gyratory crushers at the Kerr mine site. The crushed ore will then be conveyed to Mitchell through a 2,480 m cross valley rope conveyor to the Sulphurets site, followed by the 3.0 km overland conveyor through the SMCT to the Sulphurets/Kerr coarse ore stockpile at the Mitchell site. The ore from stockpile will be trucked to the Mitchell crushing facility or to the 10-Mt surge stockpile for later reclaiming and delivery to the Mitchell crushing facility.

Waste rock from the Kerr site will be conveyed to the SMCT portal at the Mitchell site in a similar manner as the ore; the waste rock will then be transported to the RSF at the Mitchell site via an additional conveying system.

The Iron Cap ore will be mined by block caving and crushed on site to 80% passing 150 mm or finer. The crushed ore will be conveyed to the MTT and loaded onto the MTT overland conveyer, which will bring the crushed Iron Cap ore to the coarse ore stockpile at the Treaty OPC plant site. The ore will supplement the mill feed during Years 32 to 51.

17.3.2 Coarse Ore Transport From Mitchell Site to Treaty Site

The crushed ore will be reclaimed by six 1.8 m wide by 8.5 m long apron feeders from the coarse ore stockpile at the Mitchell site and transported to the plant site by an overland conveyor through the larger of the twin MTT tunnels. The ore conveyance tunnel is approximately 6 m wide by 4.5 m high. One 1.83 m wide by 23.5 km long overland conveyor will be used to transport the crushed ore to the Treaty OPC site. The conveyor will be equipped with 6 drive stations with a total installed power of 29,800 kW. Dust collecting systems will be installed at each of transfer points to collect fugitive dust. The conveyor drive stations will be equipped with fire water spray system for any emergency fire suppression.

17.3.3 Coarse Material Handling

The crushed ore from the MTT overland conveyor will be discharged onto a tripper conveyor. The 2.1 m wide by 30 m long tripper conveyor will transfer the ore to a 60,000-t live capacity covered coarse ore stockpile at the Treaty OPC site. The ore then will be reclaimed by six 1.8 m wide by 8.5 m long apron feeders and conveyed in two lines to the secondary crushing circuit. A dust collecting system will be installed at each of the transfer points to collect fugitive dust.

17.3.4 Secondary Crushing

The reclaimed coarse ore will be conveyed to the secondary crushing facility and fed to four vibrating screens. Each screen oversize will feed a secondary cone crusher. Each secondary crusher is in closed circuit with a screen. The cone crusher product will return to the screen feed conveyor. A spare cone crusher is provided in the circuit when any of the other four cone crushers require maintenance.

Screen undersize product that is finer than 50 mm will be delivered by conveying to an enclosed surge stockpile with a 60,000-t live capacity. The circuit will consist of the following key equipment:

- five MP1000 or equivalent cone crushers, each driven by a 750-kW motor
- five 3.7 m wide by 7.3 m long double deck vibrating screens (one on standby).

17.3.5 Tertiary Crushing Material Conveyance/Storage

The crushed ore from secondary crushing will be reclaimed from the 60,000-t stockpile by six 1.5 m by 7.6 m reclaim apron feeders onto two 1.37 m-wide HPGR feed conveyors. These conveyors will deliver the ore to two tertiary crusher HPGR feed surge bins, each with a live capacity of 400 t.

17.3.6 Tertiary Crushing

The reclaimed ore will be further crushed by four HPGR crushers. Four belt feeders will withdraw the reclaimed ore from the two HPGR feed surge bins and feed each of the four HPGR crushers separately. Each HPGR crusher is in closed circuit with a 4.0 m wide by 8.0 m long double deck vibrating screen. Discharge from 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 undersize will leave the crushing circuit and report to the ball mill grinding circuits. The four HPGR crushing lines will have a total process capacity of 5,762 t/h. The key equipment is as follows:

- four HPGR crushers, each equipped with two 2,900 kW motors
- four 4.0 m wide by 8.0 m long vibrating screens

- four 1.5 m wide by 10.0 m long belt feeders.

17.3.7 PRIMARY GRINDING

The grinding circuit will employ conventional ball mills to grind the HPGR product to a particle size of 80% passing 125 to 150 µm. All the primary grinding circuits are designed to have a nominal processing rate of 5,762 t/h.

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

- four 7.6 m diameter by 11.9 m long (25′ by 39′) ball mills, each mill driven by two 7.0 MW synchronous motors
- six 700 mm by 650 mm centrifugal slurry pumps (4 in operation and 2 on standby), each equipped with 1,650 kW variable speed drive
- four hydrocyclone clusters, each with twelve 710 mm diameter hydrocyclones

Each ball mill will be in closed-circuit with a cluster of twelve 710 mm diameter hydrocyclones. The hydrocyclone underflow will gravity-flow to the ball mill feed chute, while the overflow of each hydrocyclone cluster with a solid density of 37% weight/weight (w/w) will gravity-flow to one of four copper-gold-molybdenum rougher flotation trains.

Lime will be added to each mill as required. Flotation collectors will be added to the hydrocyclone feed sumps or to the hydrocyclone overflow collecting sumps.

17.3.8 COPPER, GOLD AND MOLYBDENUM FLOTATION

COPPER-GOLD/MOLYBDENUM BULK ROUGHER/SCAVENGER FLOTATION

There will be four copper-gold-molybdenum bulk rougher flotation trains. The overflow of the four hydrocyclone clusters from the primary grinding circuits will separately feed the four flotation trains, each consisting of five 200-m^3 flotation cells. The flotation reagents used will include lime, A208, 3418A, fuel oil, and MIBC. A bulk copper-gold/molybdenum rougher flotation concentrate, approximately 6% by weight of the flotation feed, will be reground. The flotation tailing will be sent to the 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 three tower mills, each with an installed power of 2,240 kW, 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 flow.

Copper-Gold/Molybdenum Bulk Concentrate Cleaner Flotation

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

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

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 a conventional process, which will include copper suppression by sodium sulphide and four-stages of molybdenum cleaner flotation and regrinding. The circuit will include the following key equipment:

- one 15 m diameter high rate thickener
- six 30-m^3 conventional mechanical flotation cells
- one 1.5 m diameter by 4.5 m high column cell
- one 1.1 m diameter by 4 m high column cell
- two 1.0 m diameter by 4 m high column cells
- one nitrogen gas generator
- one regrinding stirred mill.

The copper-gold/molybdenum bulk concentrate will be thickened prior to the copper-gold/molybdenum separation. The thickener underflow will be diluted and conditioned with sodium sulphide and gravity flow into the molybdenum rougher flotation cells. The rougher flotation tailing will be scavenged by flotation and the scavenger concentrate will return to the rougher flotation head while the tailing will be the final copper-gold concentrate reporting to the copper-gold concentrate dewatering circuit.

The resulting rougher molybdenum concentrate will be classified by a hydrocyclone. The hydrocyclone underflow will be reground by a stirred mill and join with the hydrocyclone overflow reporting to the molybdenum cleaner flotation circuit. Four stages of cleaner flotation are designed to upgrade the molybdenum rougher flotation concentrate to marketable grade. The tailing of each cleaner flotation will be returned to the head of the preceding molybdenum cleaner flotation circuit while the first cleaner tailing will be sent to the molybdenum rougher flotation conditioning tank. To reduce sodium hydrosulfide consumption, the molybdenum flotation cells will be aerated by nitrogen gas, which will be generated on site by a nitrogen generator.

The final cleaner flotation concentrate will be leached to reduce copper content if copper content is higher than 0.2%. The leached product will be dewatered in a molybdenum concentrate dewatering facility.

17.3.9 CONCENTRATE DEWATERING

The upgraded copper-gold concentrate will be thickened in a 15-m diameter high rate thickener. The thickener underflow will be directed to the copper-gold concentrate pressure filter to further reduce water content to 9% moisture. The copper-gold concentrate will be stockpiled on site and then transported by trucks to a port site at Stewart where the concentrate will be stored and loaded into ships for ocean transport to overseas smelters.

The average copper concentrate produced is estimated to be approximately 890 t/d or 325,000 t/a.

The molybdenum concentrate will be dewatered using a similar process to the copper-gold concentrate. The filtered concentrate will be further dewatered by a dryer to 5% moisture before being bagged and transported to processors. The key equipment used in the dewatering processes will include:

- copper-gold concentrate dewatering:
  - one 15 m diameter high rate thickener
  - one 8 m diameter by 7 m high concentrate stock tank
  - two 160-m^2 pressure filters
- molybdenum concentrate dewatering:
  - one 2 m diameter high rate thickener
  - one molybdenum concentrate leaching system
  - one 4-m^2 pressure filter
  - one 2.5 t/h dryer.

17.3.10 Gold Recovery From Gold-bearing Pyrite Products

Gold-bearing Pyrite Flotation

The tailing of the copper-gold/molybdenum rougher flotation circuits will be further floated in a pyrite flotation circuit. The pyrite rougher flotation will consist of four parallel lines of five pyrite rougher flotation cells. The capacity of each cell will be 200 m^3.

Tailing from the pyrite rougher flotation will gravity flow, or be pumped to the TMF located southeast of the main process plant.

Gold-bearing Pyrite Concentrate Regrinding

The pyrite concentrate will be reground to a particle size of 80% passing 20 µm in three 2,240 kW tower mills. A hydrocyclone cluster consisting of twenty-six 250-mm diameter hydrocyclones will be incorporated with the mills in closed circuit. The hydrocyclone overflow 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 from the other minerals. The copper concentrate will be sent to the copper-gold/molybdenum cleaner flotation circuit while the flotation tailing will report to the gold leach circuit.

Gold Leach

The reground gold-bearing pyrite product and the first cleaner scavenger tailing from the copper-gold/molybdenum bulk flotation circuit will be separately thickened to a solids density of 65% in two 35 m-diameter high rate thickeners.

The underflow of each thickener will be pumped to two separate cyanide leaching lines. Each line will consist of two pre-treatment tanks and five cyanide leaching tanks. In the pre-treatment tanks, the thickener underflow will be diluted with barren solution and aerated. Lime will be added to increase the slurry pH to approximately 11.

The pre-treated slurry will be leached by sodium cyanide to recover gold in a conventional CIL circuit. The leach circuit will consist of 5 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 leaving the first CIL tanks of the two leaching lines will be transferred to the carbon stripping circuit while the leach residue will be blended and sent to subsequent processes including residue washing, cyanide recovery, and cyanide destruction circuits.

The key equipment in the leach circuit will include:

- two 35 m high rate thickeners
- four 9 m diameter by 10 m high aeration tanks
- ten 15 m diameter by 15 m high CIL leach tanks equipped with in-tank carbon transferring pumps and screens
- one 3 m wide by 4 m long carbon safety screen.

Compressed air will be provided for the leaching process from four dedicated air compressors.

Carbon Stripping and Reactivation

The loaded carbon will be treated by acid washing and the Zadra pressure stripping process for gold desorption.

The loaded carbon will be acid washed prior to being transferred to two elution vessels. The stripping process will include the circulation of the barren solution through a heat recovery heat exchanger and a solution heater. The heated solution will then flow up through the bed of the loaded carbon and overflow near the top of the stripping vessels. 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 3-t acid wash vessels, two 3-t stripping vessels, four heat exchangers, and two solution heaters and associated pumps.

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

The carbon reactivation process will include one reactivation kiln, one carbon quench tank, and a carbon abrasion tank equipped with an attrition agitator, reactivated carbon sizing screen, carbon storage bin, and fine carbon handling associated equipment.

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 80 kg/d of gold-silver doré bullion and will include two 3.5 m^3 electrowinning cells, direct current 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. The cell mud will fall into the bottom of the electrowinning cells and 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. A 125-kW induction furnace will be used for gold refining. The area will be monitored by a security surveillance system.

17.3.11 Treatment of Leach Residues

Leach Residue Washing

The residues from the CIL circuit will be pumped to a two-stage conventional CCD washing circuit. The CCD circuit will consist of two 40 m diameter high-rate thickeners. The thickener overflow from the first stage washing will be pumped to the cyanide recovery system. The underflow (washed residues) of the second thickener will be sent to the 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 copper will be removed and the cyanide will be recovered from the solution by a SART/AVR process.

The SART/AVR cyanide recovery process will be carried out in a negative pressure system generated by a vacuum system.

The CCD overflow will be acidified by sulphuric acid. Sodium hydrosulfide will be added to precipitate the heavy metals in the solution, especially the copper. The solution will then be pumped to two volatilization towers in series. The solution together with pressurized air will be sprayed in the towers to provide a high liquid surface area to promote volatilization.

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

The cyanide-depleted solution from the volatilization tower will be settled in a 10 m diameter clarifier. The metal species will precipitate in the clarifier while the clarified solution will be circulated to the leach residues washing circuit and the leach circuit after the solution is treated with lime to a pH above 9.5.

CYANIDE DESTRUCTION

The remaining cyanide in the washed leach residues from the second washing thickener will be decomposed by a SO_2/air oxidation cyanide destruction process. Sodium metabisulphite will be used as SO_2 source. The equipment used will include one 6 m diameter by 6 m high pre-aeration agitation tank, three 11 m diameter by 12 m high SO_2 oxidation tanks, and a wet alkaline scrubbing system. Compressed air will be provided for the oxidation process.

17.3.12 TAILING MANAGEMENT

The flotation tailing and the treated CIL residues will separately gravity flow or be pumped to the TMF located southeast of the main process plant. The flotation tailing and CIL residue will be stored in separate areas within the TMF.

The CIL residue will be deposited in a lined CIL residue storage pond. The residue will be covered with the supernatant to prevent sulphide minerals oxidation. The residue will be eventually covered by the flotation tailing, from which most sulphides have been removed. The supernatant from the CIL residue pond will be reclaimed by pumping to the CIL circuit for reuse. The excess water will be sent to the WTP to further remove impurities before it is disposed to the environment or reused in flotation circuit as process water.

There will be two flotation tailing pipelines directing the flotation tailing to the TMF. The flotation tailing from one of the tailing pipelines will be classified to produce coarse tailing sands by two stages of hydrocyclone classification. The coarse fraction will be used to construct the tailing dam and the fines will directly report to the TMF together with the tailing from the other line. The supernatant from the tailing impoundment area will be reclaimed to the process water tank by two stages of pumping. The water will be used as process water for flotation circuits.

One energy recovery system will be installed on one of the rougher flotation tailing lines, which will deliver the tailing to the north dam, to generate electrical energy.

17.3.13 REAGENTS HANDLING

The reagents used in the process will include:

- Flotation: PAX, 3418A, A208, fuel oil, MIBC, lime (CaO), 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 ($CuSO_4$), sulphuric acid (H_2SO_4), lime, sodium hydroxide (NaOH)
- Others: flocculant, 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, MIBC, HCl, H_2SO_4, and antiscalant) will be added in the undiluted form to various process circuits via individual metering pumps.

All the solid type reagents (including PAX, Na_2S, Na_2SiO_3 if required, NaOH, NaCN, $CuSO_4$, and MBS) will be mixed with fresh water to 10-25% solution strength in the respective mixing tank, and stored in separate holding tanks before being added to various addition points by metering pumps.

There will be two lime preparation systems – one at the mine site and the other at the plant site. Lime will be slaked, diluted into 15% solid milk of lime, and then distributed to various addition points through a closed pressure loop.

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

17.3.14 WATER SUPPLY

Three separate water supply systems will be provided to support the operation – a fresh water system, a process water system for grinding/flotation circuits and a process water system for CIL/gold recovery circuits.

FRESH WATER SUPPLY SYSTEM

Fresh and potable water will be supplied to two 12 m diameter by 9 m high storage tanks from nearby wells and local drainage runoff areas. One tank will be located at the plant site and the other at mine site. Fresh water will be used primarily for the following:

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

By design, the fresh water tanks 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 250 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 delivery to various service points.

PROCESS WATER SUPPLY SYSTEM

Two process water systems will supply the process water for the process plant. The water for each circuit will be from different sources, as follows:

- **Water for Grinding/Flotation Circuits:** reclaimed water from the flotation tailing pond, copper-gold/molybdenum concentrate thickener overflow and the CIL feed thickener overflow, as well as fresh water. The dominant process water will be the supernatant fluid from the flotation tailing impoundment area.
- **Water for CIL Leaching/Gold Recovery Circuits:** reclaimed water from the CIL storage pond, barren solution and fresh water. As required, the water reclaimed from the flotation tailing pond may also be used in these circuits.

The water reclaimed from the flotation tailing impoundment area will be sent to a 25 m diameter by 15 m high process water surge tank by two stages of pumping systems, while the bulk concentrate thickener overflow will be directed to the primary grinding circuits. The process water tank will be located approximately 25 m higher than the process plant base elevation. The water will flow to the various service points by gravity. A booster pump station is provided at the plant site to pump water to the various distribution points where high pressure water is required.

The water from the CIL residue storage pond will be pumped to an 8 m diameter by 8 m high process water surge tank located at the plant site. The water will service for the CIL leach/gold recovery circuits. Any excessive water from the CIL residue storage pond will be treated at the WTP located at the plant site. The treated water will be used for the grinding/flotation circuits or, if it meets discharge criteria, can be released to the appropriate Treaty or Teigen drainage system. The overall site water management is detailed in Section 18.1.

17.3.15 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 service air – high pressure air for various services by two dedicate air compressors
- instrumentation – instrument air at mine site and plant site will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

17.3.16 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 plant production and metallurgical results.

17.3.17 PROCESS CONTROL AND INSTRUMENTATION

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

- Mitchell site primary crusher control room
- Treaty plant site control room.

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

A crushing control room at the Sulphurets pit will be added before Year 23. The Sulphurets pit crushing control room will be relocated to the Kerr pit crushing plant in Year 27.

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

Process control will be enhanced with the installation of an automatic sampling system. The system will collect samples from various streams for on-line analysis and the daily metallurgical balance.

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

17.4 YEARLY METALLURGICAL PERFORMANCE PROJECTION

According to the metallurgical projections described in Section 13.2 and the current mine production schedule, metal recovery and concentrate grades for the project are projected on a yearly basis, as indicated in Table 17.2. For more accurate metallurgical performance projections, further test work is recommended, especially locked cycle flotation tests and cyanidation tests on various ore composite samples from the Sulphurets, Kerr, and Iron Cap deposits.

Table 17.2 Projected Metallurgical Performance

	Unit	Year															Total	Average
		1	2	3	4	5	6	7	8	9	10	11 to 20	21 to 30	31 to 40	41 to 50	51 to 55		
MILL FEED																		
Mill Feed Tonnage	kt	27,851	44,610	47,550	47,550	47,550	47,500	47,451	47,451	47,451	47,451	474,501	415,187	392,457	413,244	16,615	2,164,419	47,450
MILL FEED GRADE																		
Au	g/t	0.860	0.755	0.772	0.741	0.714	0.836	0.860	0.678	0.756	0.676	0.587	0.518	0.455	0.395	0.547	0.549	0.549
Cu	%	0.235	0.268	0.254	0.246	0.195	0.225	0.219	0.156	0.183	0.175	0.148	0.180	0.265	0.244	0.125	0.207	0.207
Ag	g/t	2.587	2.553	2.200	1.938	1.413	2.621	3.483	2.186	3.449	2.904	2.934	1.821	3.431	3.042	2.014	2.740	2.740
Mo	g/t	27.56	22.07	32.79	68.37	68.97	48.87	45.65	76.36	40.33	58.49	72.15	51.62	15.85	27.80	56.57	44.80	44.80
METAL RECOVERY																		
Copper-Gold Concentrate																		
Au	%	61.6	60.4	58.9	57.2	56.7	59.7	60.4	54.2	57.0	56.3	53.3	53.3	51.3	50.0	50.3	53.9	53.9
Cu	%	87.4	86.1	85.3	84.0	82.6	85.5	86.2	80.0	82.9	82.1	79.0	79.9	82.2	81.8	76.0	81.7	81.7
Ag	%	54.6	53.3	52.0	48.2	47.7	53.7	52.8	44.0	48.1	47.1	42.7	41.3	50.9	51.0	38.8	47.6	47.6
Molybdenum Concentrate																		
Mo	%	25.0	-	25.0	35.0	35.0	30.3	25.0	35.0	25.0	35.0	35.0	29.9	-	25.0	34.9	29.3	29.3
Dore																		
Au	%	16.3	16.0	16.4	17.2	17.7	16.4	17.1	20.4	18.9	19.0	20.6	18.2	19.9	20.8	22.3	19.2	19.2
Ag	%	4.3	10.0	11.1	11.2	3.8	10.8	12.2	11.5	16.7	14.1	18.7	15.1	16.4	16.3	15.5	15.8	15.8
PRODUCTION																		
Copper Concentrate																		
Tonnage - Concentrate	kt	229	412	412	394	306	365	359	237	287	297	2,407	2,449	3,357	3,229	68	14,807	325
Grade - Au	g/t	64.6	49.4	52.4	51.2	62.8	64.9	68.7	73.7	71.3	60.8	61.7	46.8	27.3	25.3	66.8	43.3	43.3
Grade - Cu	%	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	23.0	23.0	24.3	25.5	25.5	23.0	24.7	24.7
Grade - Ag	g/t	172.1	147.5	131.9	112.9	104.6	182.8	243.6	193.1	274.3	218.3	246.8	127.5	204.3	198.7	189.7	190.5	190.5
Molybdenum Concentrate																		
Tonnage - Concentrate	t	384	-	780	2,276	2,296	1,408	1,083	2,536	957	1,943	23,966	12,835	-	5,743	656	56,862	1,247
Grade - Mo	%	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0
Doré																		
Tonnage - Au	kg	3,911	5,399	6,021	6,062	6,017	6,533	6,983	6,565	6,783	6,112	57,449	39,076	35,591	34,002	2,030	228,534	5,010
- Au	koz	125.8	173.6	193.6	194.9	193.5	210.0	224.5	211.1	218.1	196.5	1,847.0	1,256.3	1,144.3	1,093.2	65.3	7,347.5	161.1
Tonnage - Ag	kg	3,063	11,424	11,639	10,289	2,577	13,430	20,124	11,965	27,269	19,367	260,435	114,227	220,964	205,389	5,199	937,363	20,550
- Ag	koz	98.5	367.3	374.2	330.8	82.9	431.8	647.0	384.7	876.7	622.6	8,373.2	3,672.5	7,104.1	6,603.4	167.2	30,136.9	660.7

18.0 PROJECT INFRASTRUCTURE

18.1 TAILING, GEOTECHNICAL, AND WATER MANAGEMENT

18.1.1 INTRODUCTION

This section addresses updates made in 2012 to geotechnical designs for tailing and mine rock management as well as site-wide water management. The design incorporates changes to the mine design, mining production schedules, and site investigations conducted in both the TMF and mine area during 2011.

Figure 18.1 shows general arrangements of the updated mine site facilities. Figure 18.2 shows the arrangements of the TMF facilities.

One of the major design changes to the TMF was the inclusion of the cyclone sand Splitter Dam to form a lined impoundment for the CIL Residue Cell between the Splitter Dam and the cyclone sand Saddle Dam. The CIL Residue Cell is located between the North and South cells (in previous designs, a dam referred to as the “South Dam” separated the North and South cells).

Other TMF design revisions included changes to the East Catchment Diversion Tunnel routes based on results of the site investigations and the inclusion of the CIL Cell. Designs of surface diversion channels were also updated and a rock cut closure spillway was designed for the Southeast Dam to route closure flows into Treaty Creek.

In the mine area, water management structures for the Mitchell Diversion were upgraded to have the capacity to divert higher storm flood flows to better protect the block caving workings, now included in the mine plan. Revisions were made to the locations of diversion tunnel inlets and outlets, the Mitchell Pit Closure Dam, and surface closure channels to accommodate the subsidence zone from block caving of the Mitchell deposit under the pit.

KCB re-assessed climate and hydrology analyses based on new baseline site data recorded by Rescan, as well as historical data for the region. These analyses determined similar values to those adopted in 2011 for an average year, and various return period wet year, dry year, and storm events for the mine site area and the TMF area. These values were used by KCB for design updates to tailing and water management structures.

Figure 18.1 Mine Site General Arrangement

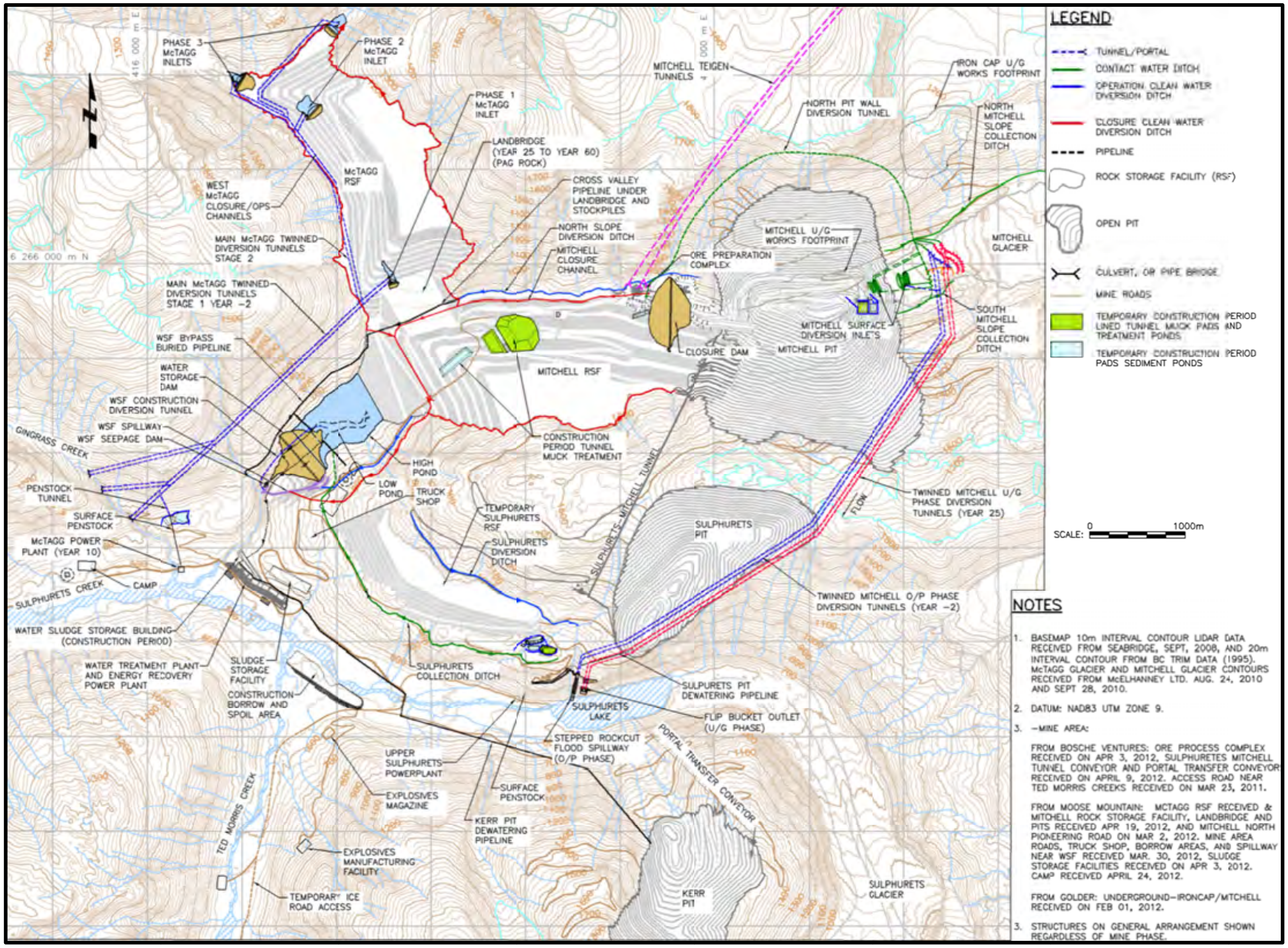
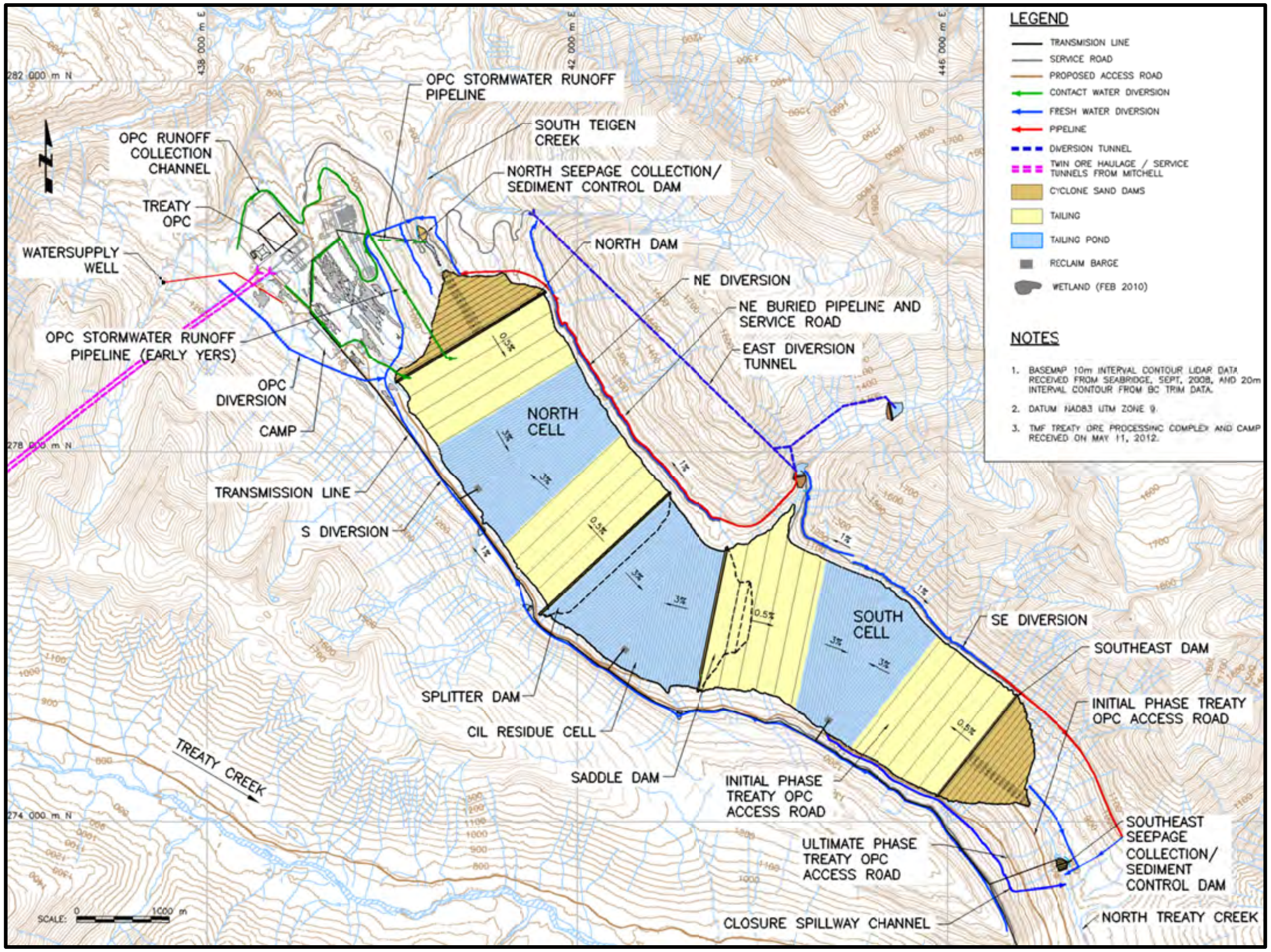


Figure 18.2 TMF Site General Arrangement



Complete details of prefeasibility design updates for the TMF and mine area are provided in the following 2012 KCB reports:

- "TMF Engineering Design Update" (2012)
- "Mine Area Water Management and RSF Geotechnical Design Update" (2012).

The following sections summarize recent site investigations and present 2012 updates to the geotechnical and water management designs.

18.1.2 MINE AREA SITE CHARACTERIZATION

In support of 2012 updates to the designs, KCB completed the following program of site investigations in 2011.

MINE AREA SITE INVESTIGATIONS

The primary purposes of mine area site investigations were to confirm data used in designs for the WSD grouting program, and to guide modifications to inlet structures of the Mitchell Diversion Tunnels for the increased diversion capacity necessary to protect block caving workings.

Table 18.1 lists objectives and methods for investigations completed at each of the mine area sites.

Table 18.1 Objectives and Methods for Mine Area Site Investigations

KSM Mine Area Site	Objectives	Methods
Mitchell Glacier	Thickness of glacial ice, glacier water balance, diversion intake conditions	Drill and sample, ice radar survey, hydrogeologic tests, geologic mapping, laboratory tests
West McTagg Glacier	Thickness of glacial ice, diversion intake conditions	Drill and sample, hydrogeologic tests, geologic mapping, laboratory tests
WSD	Spillway and grout curtain rock and hydrogeological conditions	Drill and sample, seismic survey, hydrogeologic tests
WTP	Foundation conditions	Seismic survey
Mitchell OPC	Foundation conditions	Seismic survey
Sulphurets Creek and Ted Morris Creek	Foundation conditions	Seismic survey, mapping and terrain analysis

Specifically, the 2011 investigations included:

- 1 bedrock core hole at the WSD site (total length = 201 m)

- 1 open hole piezometer in the WSD (total length = 201 m)
- 6 Mitchell glacier core holes (total length ice = 545, rock = 252 m)
- 31 borehole packer tests in bedrock
- measured water levels in 15 open-hole piezometers
- 7 seismic refraction surveys (total length = 4,665 m)
- 9 Mitchell Glacier ice radar survey lines (total length = 3,275 m)
- 4 engineering geologic mapping and terrain analysis studies including margins of Mitchell Glacier, McTagg power plant area, WSD Spillway area as well as Sulphurets and Ted Morris Valley.

Figure 18.3 illustrates the locations of all KCB drill holes completed to date in the mine area. Figure 18.4 shows mapped geology of the mine area. Figure 18.5 provides a profile view that summarizes geology intersected by the Mitchell-McTagg RSF foundation geotechnical drill holes. The chart is annotated with test results, including standard penetration test (SPT) values, permeability results from packer testing, and geotechnical laboratory test results from drill hole samples including moisture content, liquid limits, plastic limits, friction angle, and cohesion.

Overburden ranges from 2.4 m in the area of the Mitchell pit western rim to approximately 80 m to 90 m in the central Mitchell Valley. Mitchell RSF foundations consist of moraine/colluvial deposits and small areas of lacustrine soils.

The proposed Mitchell OPC, where primary crushing is to be conducted, will be situated on bedrock above the valley bottom sediments. Pads and access ramps will be constructed from a nearby colluvial borrow source.

Drill holes in the Mitchell and McTagg valleys and in the area of the WSD foundations intersected Stuhini group sedimentary rock (typically argillites, meta-conglomerates or mudstones and Mitchell intrusives and Stuhini limestones). Additional WSD drilling in 2011 confirmed observations in 2010 that grouting of the WSD foundation will be required, but showed low permeability rock exists at depth away from the canyon wall area.

WSD borrow area drill holes in 2010 indicated that sufficient soil materials with suitable characteristics are available for earthfill zones of the dam.

Seismic surveys, mapping, and drilling in Mitchell Valley during 2009 and 2011 confirmed design parameters used for the RSF, WSD, WTP, power plants and Mitchell OPC designs. Overburden thickness contours are presented in Figure 18.6.

Seismic lines completed along the WSD spillway in 2011 confirmed that bedrock rock was located near the surface and that the area was suitable for construction of the spillway. In 2010, seismic surveying conducted in the McTagg power plant penstock and the WTP foundation areas confirmed that ground conditions are composed of terraced moraine soils with depths to bedrock of 5 m to 25 m.

Ice Penetrating Radar was used during 2011 to investigate the thickness of the Mitchell Glacier ice for locating diversion inlets. Ice radar conducted in 2010 along the route of the MTT showed that the proposed MTT route is well below the base of the ice.

Deep penetrating Magneto Telluric (MT) electromagnetic surveying was completed in 2011 along ice covered portions of the MTT route to image resistivity of geological structures along the route. This information is being assessed for use in delineating faulting and locations of rock unit contacts along the tunnel route. Additional drilling is planned for 2012 along the MTT route and at inlet and outlet areas of the Mitchell Glacier Diversion to provide information for detailed design.

Figure 18.3 Mine Area Site Investigation Plan

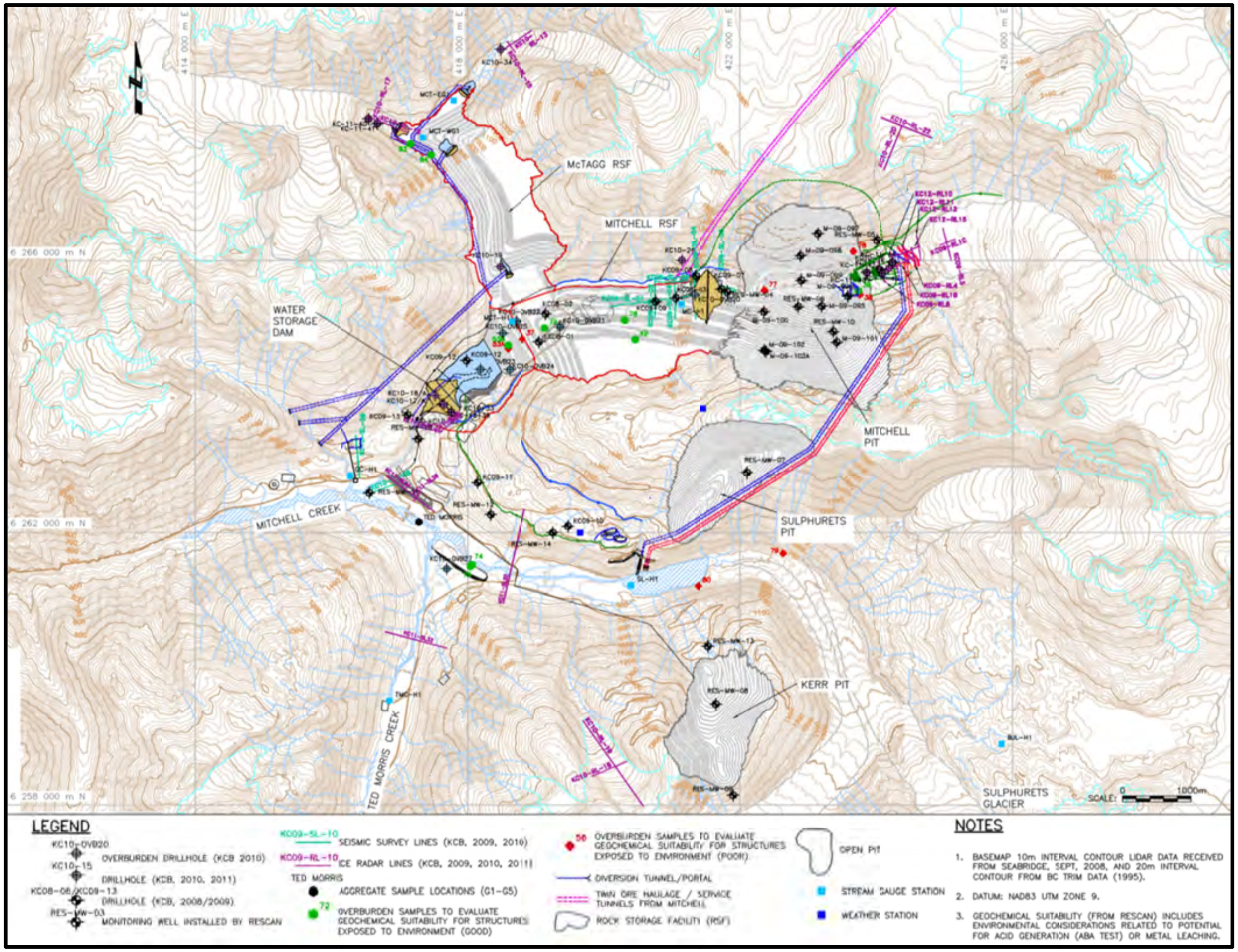


Figure 18.4 Mine Area Mapped Geology

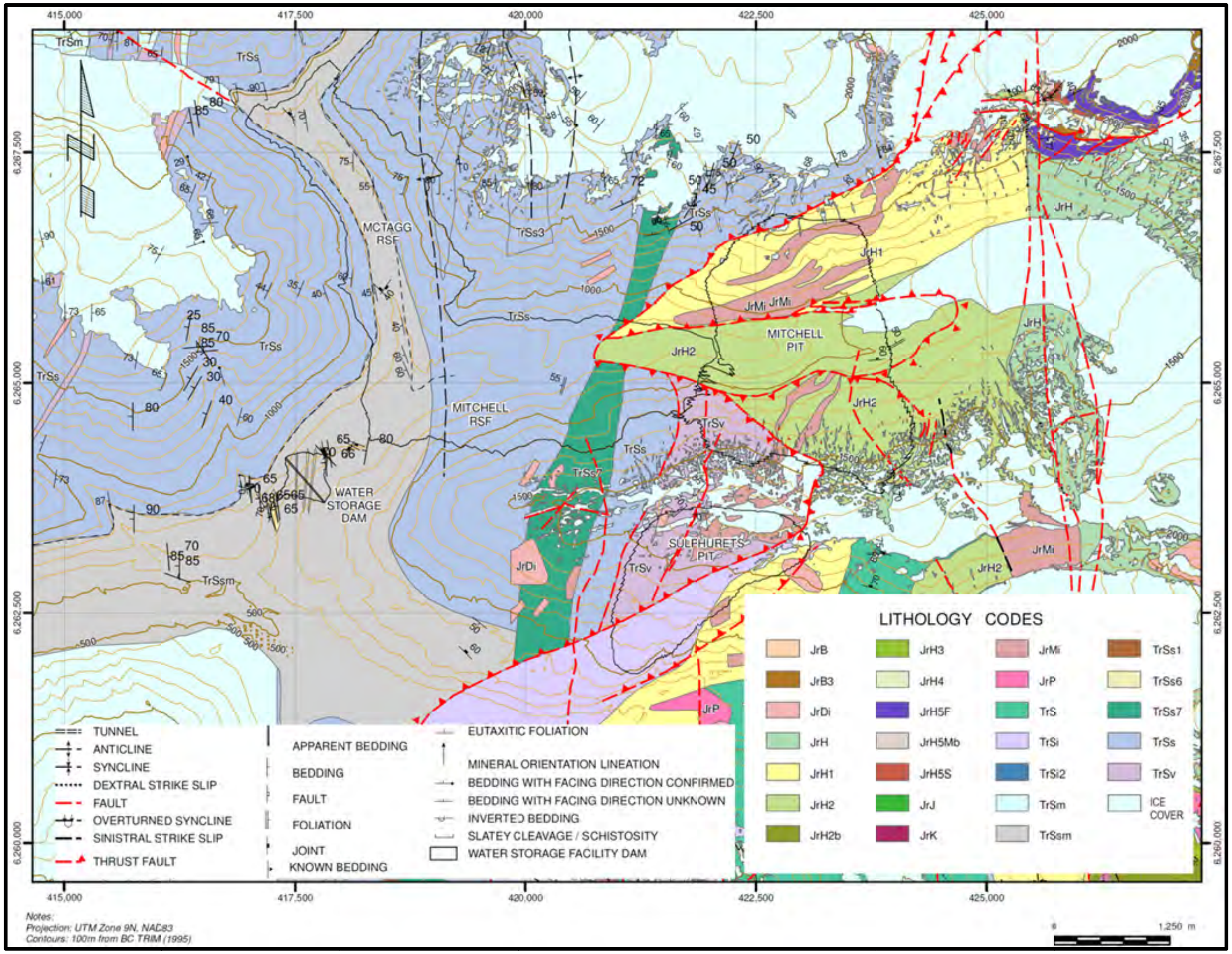


Figure 18.5 Mitchell Valley Drill Hole Geology Section (RSF and Mitchell OPC Area Foundations)

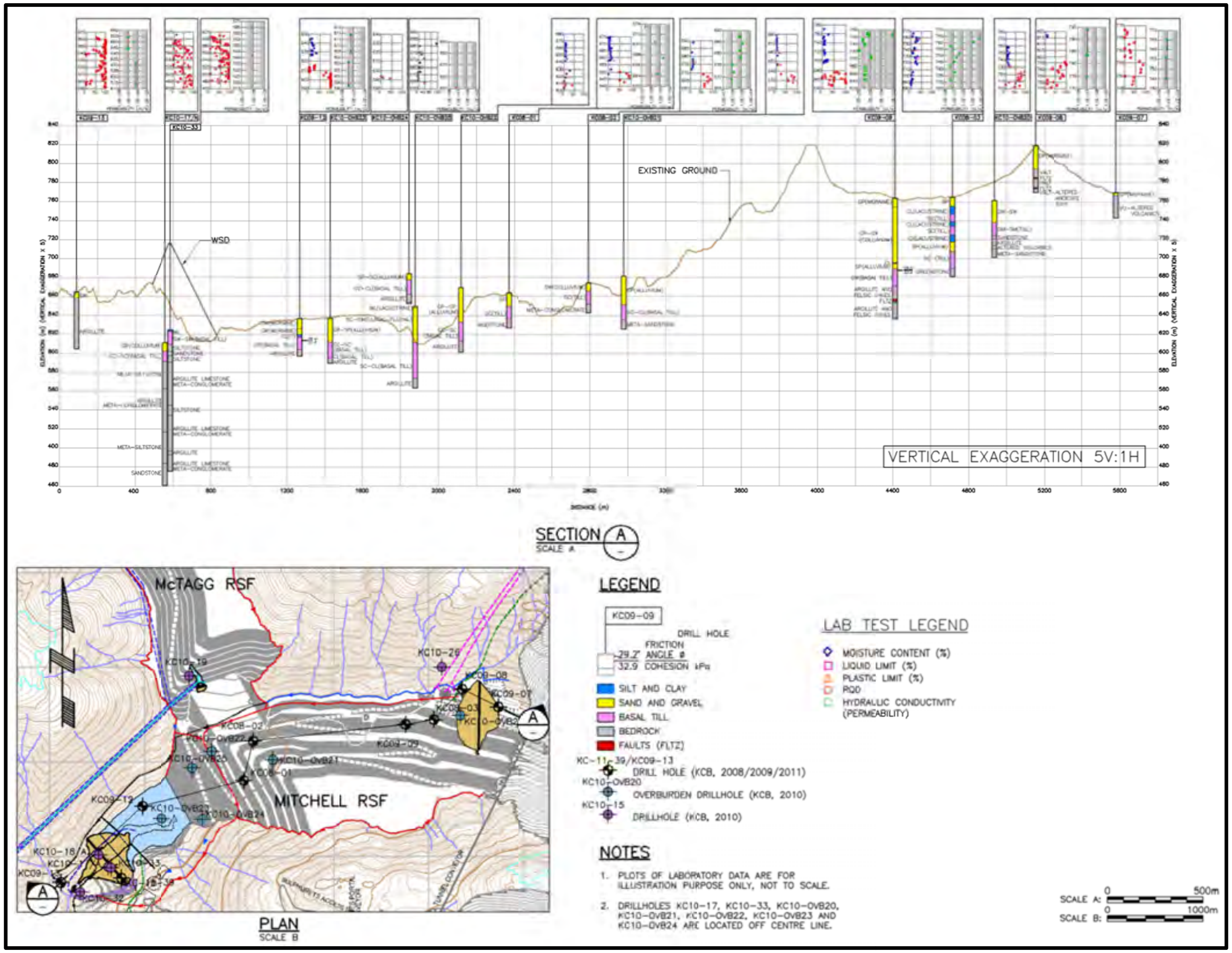
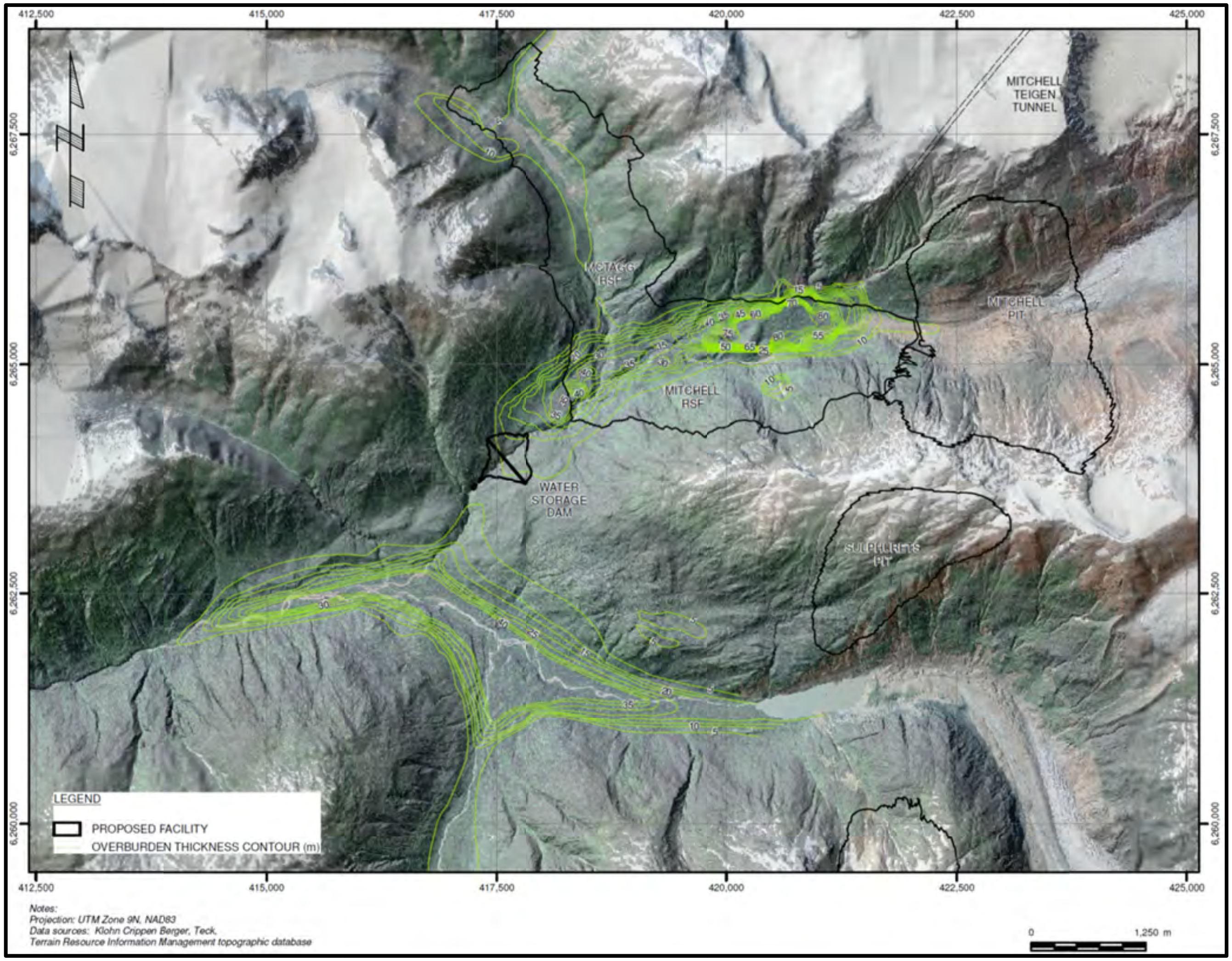


Figure 18.6 Mitchell Valley/Mitchell OPC Area Overburden Thickness



Mine Area Site Investigation Findings – RSF Foundations

Information from the site investigations was used in geotechnical stability analyses of the RSFs. These analyses show that the conditions for RSF foundation stability can be addressed with controlled placement of mine rock, as required.

In the central area of Mitchell Valley, a region of lacustrine clay and sand deposits, located at depth, requires specific measures to address geotechnical stability. The lower strength of these clay layers can be managed by restricting adjacent dump heights until preloading of the toe area can be completed. This can be accomplished by placing a toe berm in Year 1 after the valley floor area has been used for construction laydown purposes. This will confine the toe and assist pre-consolidation of the clay layers to improve stability of subsequent lifts on the south slope in the area of the Mitchell RSF facing the ore preparation complex.

Another area requiring specific measures is the toe region of the Mitchell RSF within the water storage pond. Water levels of the pond fluctuate seasonally, requiring a provision in the base RSFs of either free draining rock in the area flooded by the pond, or the provision of engineered rock drains under the RSF in this area flooded by the pond.

With these measures, geotechnical stability of the RSF configurations proposed by MMTS for the Mitchell, McTagg, and Sulphurets RSFs will meet the design criteria for RSF stability. Monitoring during RSF construction will be carried out to confirm performance.

Mine Area Site Investigation Findings – WSD Foundation Conditions

Site investigations have been carried out for the water retaining dam located in Lower Mitchell Canyon. Strong and relatively unweathered competent sedimentary rocks (argillites, phyllites and, to a lesser extent, fine grained sandstones/siltstones/ limestones, and intrusives) are exposed in the canyon area. Where present, weathering depths are on the order of 5 m, and little or no overburden is present under most of the dam footprint. The canyon is narrow, resulting in a high dam elevation relative to storage obtained.

The orientations of the rock bedding and the topography of the site are favourable for stable dam construction, as is the rock topography, which will result in wedging of the dam fill. The presence of flexural fracturing in the bedrock on the left (east) bank bedding planes will require grouting of the dam foundation. Most intervals of the rock mass are primarily tight, as shown by packer testing and geological logs; however, a localized area of the east abutment exhibits increased permeability due to flexing of bedding planes. The grout curtain has thus been designed to be 25 m deep on the western abutment and up to 150 m deep on the eastern abutment.

An injection test (inverse pumping test) monitored from adjacent drill holes was conducted beneath the left (east) abutment to confirm the packer and slug tested permeability values for the bulk rock mass, assess interconnectivity between fractures, and define groundwater storage properties. The hydrogeological testing and subsequent modelling showed that the designed grout curtain and natural upward movement of groundwater will limit seepage of contact water from the impoundment.

Results of the 2011 drill hole KC11-39, drilled farther to the east of the previous WSD drilling, encountered mixed volcanic and sediments. The drill hole showed that rock quality improves farther from the canyon wall, as expected; however, fractures were still present that would require grouting. This drill hole confirmed the depth requirements assumed for grouting of the WSD foundation in the 2010 PFS.

Mine Area Site Investigation Findings – Borrow Materials for Dam Fill and Closure Covers

Site investigations show that borrow areas within 1 to 2 km upstream of the water storage pond, will provide suitable earth materials required for the dam's central zone, filter, and drain materials. Drilling and sampling has shown that sufficient quantities are present to construct the dam.

The mining fleet will quarry rock fill for the dam, using rock exposed upstream of the dam along benches developed for the access road. Additional rock fill for the upstream shell of the dam will come from pre-production stripping in the Sulphurets area.

Borrow materials in the dam and impoundment area vary in reactivity to the contact water stored in the WSF. Based on the testing to date, it is expected that sufficient borrow materials of low reactivity are available. The WSD has been designed to include acid-resistant asphalt cores and other features that reduce the sensitivity of dam performance to potential acid degradation. By design, the dam will not be sensitive to settlement of the dam fill.

Significant resources of alluvial and glaciofluvial sands, gravels, and cobbles are present around the mine area in the Sulphurets and Ted Morris valleys as well as at Teigen Creek near the TMF North Dam. The abundant sandy/silty moraines available in these valleys, adjacent to the mine area, are suitable for a reclamation cover but are limited for use as low permeability layers for closure covers.

Mine Area Geotechnical Laboratory Test Programs

As part of an evaluation of foundation conditions and suitability of materials for construction borrow, laboratory testing was carried out on samples of overburden materials in the mine area.

Tests included:

- index tests:
  - grain size distribution
  - moisture content
  - Atterberg limits
- direct shear and consolidated undrained triaxial test
- standard proctor testing to evaluate moisture-density relationships
- permeameter permeability test
- aggregate suitability for use as fine and coarse concrete aggregate and road base and surfacing material
- field permeability testing on overburden soils and bedrock (falling head, slug tests and packer tests)
- SPTs on overburden soils.

The site investigations indicate that most soils in the proposed mine area are comprised of coarse grained soils (sandy gravels and cobbles), such as the abundant moraine units and alluvial deposits associated with Sulphurets Creek.

Limited quantities of fine grained soils consisting of lacustrine silts, clays, sands, and some silty or clayey basal till units at depth occur in the mine area.

Geotechnical properties of overburden soils from laboratory testing and field investigations are presented in Table 18.2.

Table 18.2 Summary of Typical Geotechnical Properties from Laboratory Testing of Overburden Soils

Items	Fine Grained Soils	Coarse Grained Soils
Water Content (%)	15% to 30% 22% average	5% to 22% 9.3% average
Grain Size Distribution		
Sand Content	15% to 44% 28% average	20% to 60% 41% average
Fines Content (<75 µm)	53% to 80% 62% average	2% to 40% 17% average
Clay Content (<2 µm)	20% to 35% 26 average	6% to 13% 9.5 average
Atterberg Limits		
Plasticity Index	7% to 17%	8% to 11%
	CL to CI	CL
Standard Proctor	OWC[1] = 13% MDW[2] = 1996 kg/m^3	n/a

table continues…

Items		Fine Grained Soils	Coarse Grained Soils
Bulk Unit Weight (kN/m^3)		n/a	25, 24.7
Shear Strength	ϕ'_{peak}	n/a	29.2°, 38.4° (38.5° from one triaxial test)
	c'_{peak} (kPa)	n/a	33, 37 (198 from one triaxial test)[3]
	ϕ'_{res}	n/a	25.9°, 30.3°
	c'_{res} (kPa)	n/a	0, 27
Permeability (m/s)			
Laboratory		1.2×10^{-9} (one test)	2.2×10^{-7} ~7.2×10^{-7} (three tests)
Field		4×10^{-8} to 8×10^{-8} (three tests)	10^{-5} to 10^{-7} (26 tests)
SPT ($(N_1)_{60}$ values)		21-64, 35 avg.	13-82, 47 avg.

Notes: [1] OWC = optimum water content; [2] MDW = maximum dry weight; [3] cemented sample from KC09-12 at 23.2 m.

Geotechnical Properties of Aggregate Materials

KCB contracted Levelton Consultants Ltd. (Levelton) to test aggregate materials to Canadian Standards Association and BC Ministry of Transportation standards for concrete and road material uses. Results of grain size determinations found that the alluvial-fluvio glacial deposits of Ted Morris Creek and the alluvial deposits of the Teigen Creek area typically have an excess of fines, which will require washing of the aggregate for either use; otherwise, the gradations are classified as good.

Specialized testing of petrographic and durability properties and accelerated alkali reactivity expansion revealed that the aggregate had some surface encrustment of cemented materials, slightly high levels of organics and expansion properties over the threshold for requiring further long term silica reactivity testing. Overall, Levelton ranked the aggregate "fair" to "good" for use as coarse and fine concrete aggregate and "good" for road bed use, if the excess of fines are washed before use. The Levelton report is included within KCB's "2010 Site Investigation Report".

MINE ROCK CHARACTERIZATION

Over the 55-year mine life, a total of 3.03 Bt of mine rock and overburden (resulting from the mining of 2.16 Bt of ore with strip ratio 1.5) will be stored in the RSFs. The inclusion of the underground phases of mining has resulted in a significant reduction in mine rock from the 5.04 Bt in the 2011 open pit only designs.

Based on current RSF layouts, elevation differences between the crest and toe within the RSFs can be as great as 600 m. Slope height of the RSF benches can reach 105 m locally.

For the purposes of open pit design, BGC assessed geotechnical units for open pit rocks. Based on these units and on experience with similar mine rock piles, KCB grouped mine rock into four classes of friction angles, which range from 35° to 48° for peak friction angles and 31° to 40° for residual angles.

Results of point load tests by BGC ranged from 1.3 MPa to 6.3 MPa with median values from 2.5 MPa to 4.7 MPa ("Report on Open Pit Design", BGC, 2009). This indicates mine rock strength is generally high to very high.

With ongoing exposure to moisture and oxygen within the RSFs, mine rock may undergo chemical/physical weathering, and potentially lose shear strength with time. Due to the cumulative effect that these factors have on the degradation of rock strength, KCB recommended a closure rock slope angle of less than 37° based on friction angles of 40° for overall mine rock as placed during operation and 30° for closure. These values were used by KCB for RSF stability calculations, and adopted by MMTS as the RSF design criteria.

Mine Area Climate

The KSM Project site receives significant annual precipitation with much of it falling as snow between October and May, while peak rainfall is associated with storms coming in from the Pacific between August and October. Major elevation variations and numerous glaciers help create diverse climatic conditions across the site.

The project area is subdivided into two climatic regions: the western Sulphurets watershed (mine site) and the eastern Teigen watershed (TMF site). The two regions are 24 km apart and have significantly different climates. The two areas are separated by the Johnstone Icefield (ranging from 1,800 m to 2,200 m elevation), Treaty Glacier, and North Treaty Ridge.

Significant orographic and rain shadow effects have been recorded in the area. Subsequently, KCB performed extensive analysis of the climate variations in the KSM area. Algorithms were developed based on the UBC watershed model to estimate effects of the variations in precipitation with attitude and to adjust glacier and snow melt rates in response to climatic variations.

Mine Area Temperature

Weather data recorded at the Sulphurets weather station between 2007 and 2011 indicated the following:

- The mean annual temperature is approximately 0.3°C.
- Mean monthly temperatures ranged from -13.5°C in December to 13.6°C in July.
- Temperature extremes ranged from -31.1°C to 30.2°C.

- Mean daily temperatures are above freezing from May to October.
- Freezing temperatures could occur from October to May.

Canadian metrological service data indicates that frost penetration for the area is typically 1.5 m or more.

Mine Area Precipitation and Hydrology

The estimated mean annual precipitation is 1,652 mm at the elevation of the Sulphurets weather station (880 masl). Annual lake evaporation is estimated at 400 mm. Runoff at the mine site is influenced by the effects of both seasonal snowmelt and glacial melt. Both the Mitchell and McTagg glaciers are losing significant ice mass on an annual basis. Runoff from glacier-influenced catchments is therefore larger than the annual precipitation over these catchments. Effects of glacial meltwaters are included in the analysis of flows and extreme events. Monthly precipitation, evaporation, and runoff distribution for the mine site are provided in Table 18.3, as well as the runoff distributions for the glacier catchments. Locations of climate and flow measurement stations around the mine site are shown on Figure 18.3.

Precipitation listed in Table 18.3 is representative of the Sulphurets weather station at 880 masl. Precipitation increases at higher elevations within the Mitchell and McTagg valleys at a nominal rate of +5% per 100 m.

The 2012 KCB design reports present detailed analyses of climate and hydrology data for the mine and TMF areas.

Table 18.3 Climate Data for the KSM Mine Site

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Precipitation (mm)[1]	215	50	50	66	99	83	115	149	264	297	132	132	**1,652**
Lake Evaporation (mm)	0	0	0	0	86	93	99	80	43	0	0	0	**400**
Site Runoff Distribution (%)	0	0	1	1	4	14	35	17	17	7	3	1	**100**
Mitchell Glacier Runoff (mm)	37	37	73	73	110	404	1,248	917	514	183	37	37	**3,670**
McTagg Glacier Runoff (mm)	66	33	33	33	197	625	954	526	461	197	99	66	**3,290**

[1] Weather station at 880 m elevation.

MINE AREA HYDROGEOLOGY

In the mine area, groundwater flow occurs primarily within fractures and faults in the bedrock matrix. Groundwater levels mimic topography, with the high terrain in the area surrounding the WSF resulting in the impoundment area being a zone of active groundwater discharge. In the vicinity of the WSD, groundwater levels within the localized zone of flexural fracturing in bedrock beneath the left (east) bank of the WSD are low, and similar to the level of Mitchell Creek due to locally high permeability (1E-06 m/s). Permeability beneath the right (west) bank is lower (<1E-07 m/s), and groundwater levels there are near surface.

To guide seepage mitigation design for the WSF, a 3D groundwater model covering the mine area was constructed using the FEFLOW® software package. The model focussed on the WSF and extended to catchment divides for Mitchell and McTagg creeks to the north and Sulphurets Creek to the south. The model was benchmarked to groundwater levels in wells and to stream flow data recorded by Rescan. Seepage through the WSD structure itself was modelled separately using Seep/W© software.

The results of both models indicate seepage from the base of the water storage pond to groundwater is approximately 2 L/s with the addition of the foundation grout curtain. The pre-existing and relatively steep groundwater gradients resulting from recharge of the surrounding high topography contain seepage of contact water from the water storage pond within the Mitchell Canyon area, and seepage from the Mitchell and McTagg RSFs within respective valleys. As a result, there is no cross valley seepage of water from the WSF. Contact seepage from the WSF is predicted to discharge into the seepage collection dam at the dam toe where it will be recovered for treatment. All seepage from the RSFs is predicted to discharge into the WSF.

18.1.3 TMF SITE CHARACTERIZATION

TMF SITE INVESTIGATIONS

The purpose of the 2011 TMF area site investigations was to provide additional information on geological, hydrogeologic, and geotechnical properties of the site and borrow materials to support engineering design updates.

KCB conducted investigations at the sites of the North Dam, Saddle Dam, and Southeast Dam in 2010, 2009, and 2008. In 2011, additional investigations were conducted in the area of the Saddle, Southeast, and Splitter dams in support of designs for the CIL Residue Cell. A plan map of all investigations to date is provided in Figure 18.7.

Specific objectives and methods of the 2011 TMF area field investigation program are listed in Table 18.4.

Table 18.4 Objectives and Methods for TMF Area Site Investigations

TMF Area Site	Objectives	Methods
North Dam	Dam foundation conditions, borrow availability	Drill and sample, seismic survey, hydrogeologic tests, terrain analysis, laboratory tests
Saddle Dam Area	Liquefaction potential, dam foundation conditions, borrow availability	CPT, drill and sample, seismic survey, hydrogeologic tests, terrain analysis, laboratory tests
Splitter Dam	Dam foundation conditions, borrow availability	Drill and sample, seismic survey, hydrogeologic tests, terrain analysis, laboratory tests
Southeast Dam	Liquefaction potential, dam foundation conditions, borrow availability	CPT, drill and sample, seismic survey, hydrogeologic tests, terrain analysis, laboratory tests
Treaty Plant Site	Plant site foundation conditions, tunnel conditions	Drill and sample, hydrogeologic tests, terrain analysis, laboratory tests
East Catchment Valley	Landslide hazard to diversion tunnel and intake; tunnel conditions, intake dam foundation conditions	Drill and sample, hydrogeologic tests, geologic mapping, terrain analysis, laboratory tests
Treaty Creek Valley	Potential alternative TMF; dam foundation conditions	Seismic survey, geologic mapping, geohazard assessment, terrain analysis
Saddle Portal Area	Cut-and-cover tunnel and portal conditions, landslide hazard to portal area	Geologic mapping, geohazard assessment

Specifically, the geotechnical investigations completed in 2011 comprised:

- 13 Odex and/or core holes (total length = 586 m)
- 9 Cone Penetration Test (CPT) soundings (total length = 146 m)
- Installation of 22 open-hole piezometers (total length = 568 m)
- 18 borehole packer tests
- 10 borehole air-lift yield tests
- 30 soil permeability tests
- measurement of water levels in 51 open-hole piezometers
- 12 seismic refraction surveys (total length = 7,885 m)
- 4 engineering geologic mapping and terrain analysis studies (TMF plant site area, East Catchment area, Treaty Valley and Saddle Portals).

Procedures and results of the investigation programs are described in the respective Site Investigation Reports (KCB, 2009, 2010a, 2011, 2012).

The updated surficial geology of the TMF site is shown on Figure 18.8.

Figure 18.7 TMF Site Investigation Plan

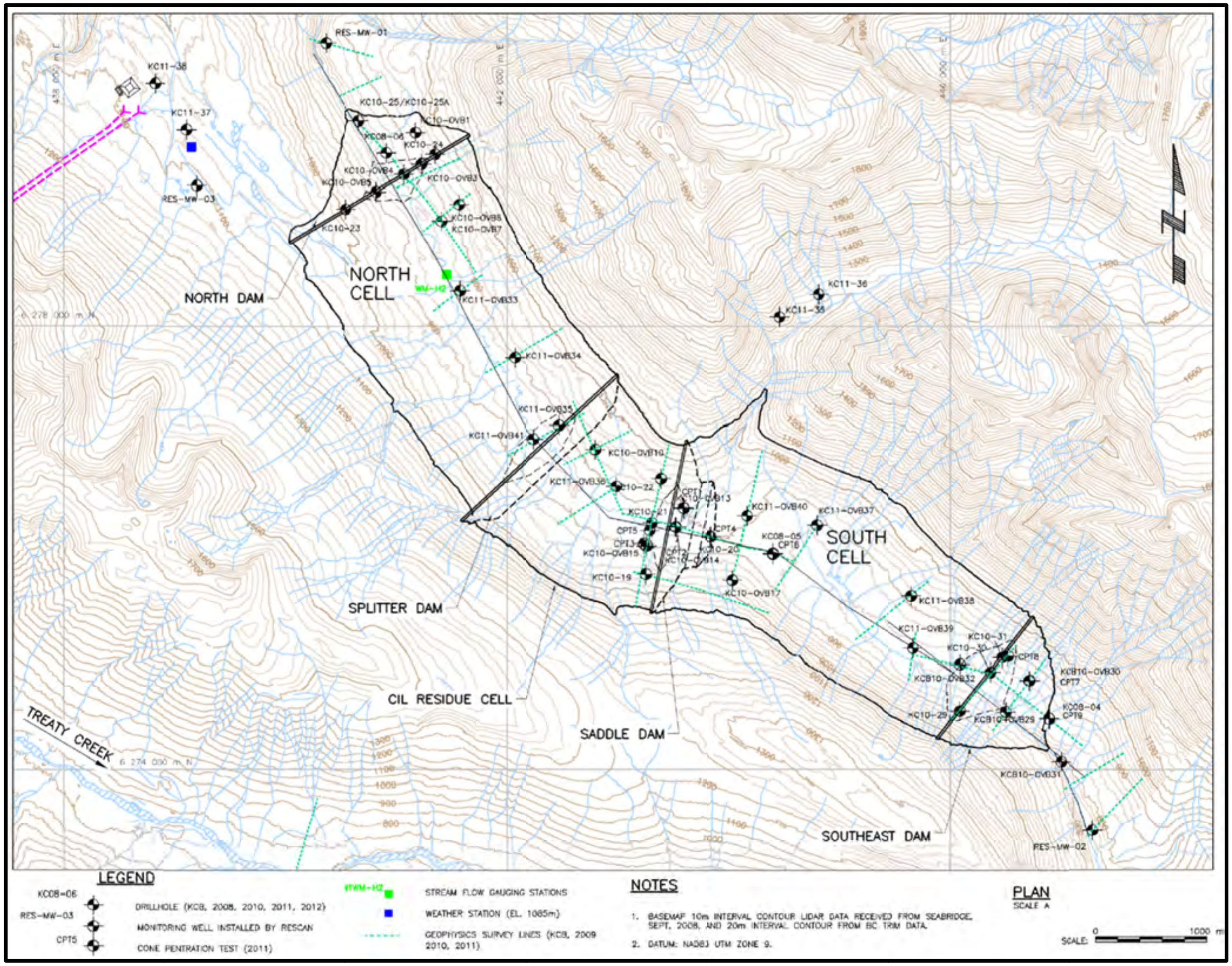
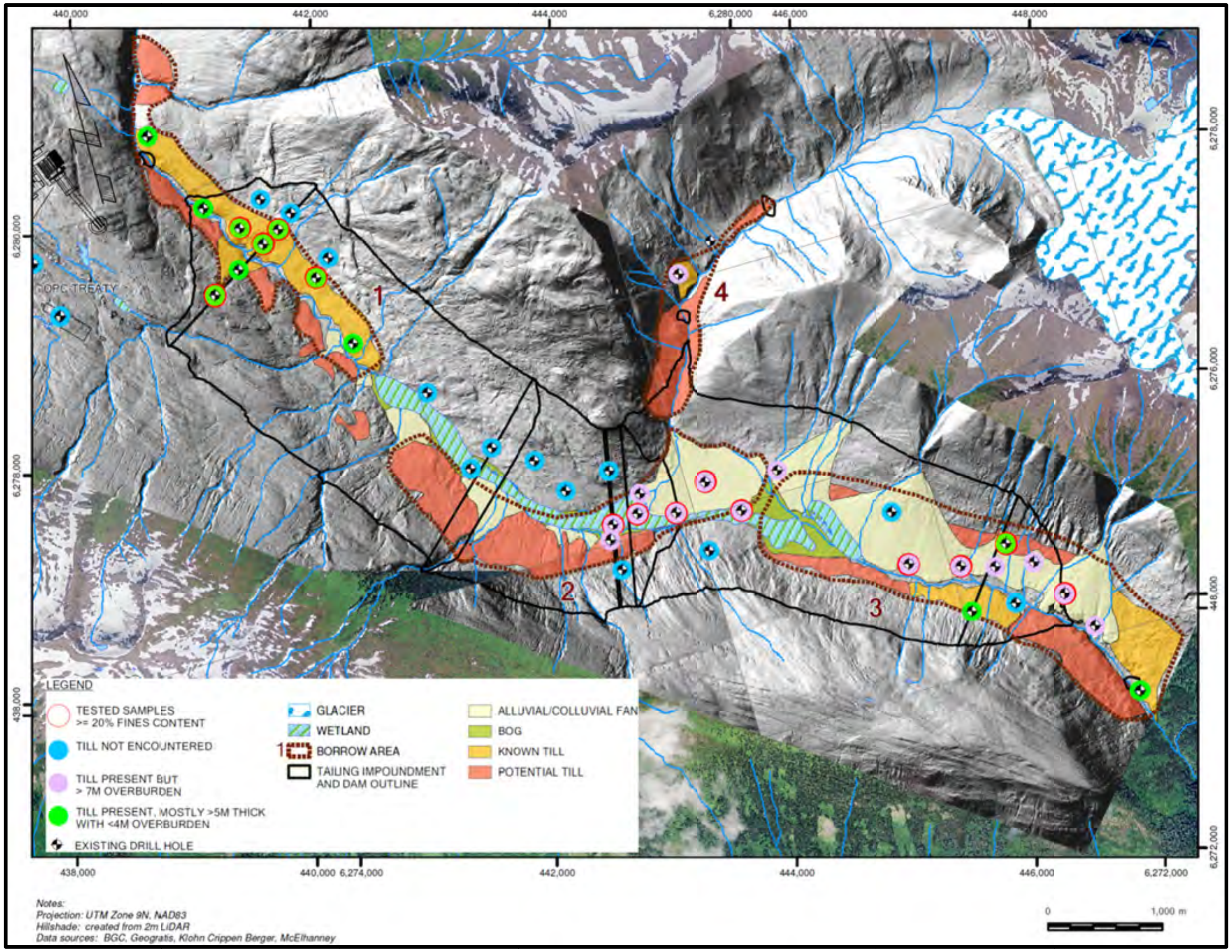


Figure 18.8 TMF Surficial Geology and Borrow Area Plan



Findings from site investigations pertinent to the design of each of the four dams in the TMF are outlined in the following sections.

North Dam

The North Dam spans the valley of the South Tributary of Teigen Creek. The dam is underlain by bedrock on both the left (southwest) and right (northeast) abutments to roughly El. 900 m; below this elevation the dam rests on till in the bottom of the valley. Colluvium and organic soils overlying bedrock are typically less than about 2 m thick on the left abutment and roughly 6 m thick on the right abutment.

Till near the bottom of the valley is up to about 30 m thick but thins to less than 5 m on the lower valley slopes; till is mostly absent above El. 900 m. The thickness of the till overlying bedrock is typically thinner on the left abutment than on the right abutment. The South Tributary of Teigen Creek has incised the till deposits so that only a few meters of alluvium and till is present along the stream channel.

The matrix of the till is composed of sand and a mix of low plasticity silt and clay, with fines content ranging from about 5% to 40%. Water content in till samples is typically less than 10%, reflecting the high density. SPT blowcounts in the till are generally greater than 20 blows per 300 mm [(N1)60] and indicate the till is dense to very dense.

Falling head tests in till in borehole KCB08-06 at 5 m and 10 m depth yielded a hydraulic conductivity (k) of 3 x 10^{-6} m/s. Visual examination of till core samples shows finer till at depth, suggesting that the hydraulic conductivity of the till at depth is at least one order of magnitude lower than was measured in the falling head tests.

Interbedded siltstone, mudstone, and fine-grained sandstone bedrock was encountered below the till; these rocks are typically hard and medium-strong to strong. RQD values typically range from 20% to 80%.

Bedrock is folded in a syncline whose axis is roughly coincident with the valley bottom. A fault is inferred to strike parallel to the syncline axis based on abrupt changes in the dip of bedding observed along the fold axis, and a coincident northwest-trending lineament visible in overhead imagery. Four north-trending lineaments intersect the North Dam and are also inferred to represent faults.

Piezometers installed in the bedrock typically show the water table at or slightly below the ground surface, with local areas on the right abutment displaying artesian conditions. Artesian conditions are interpreted to result from confined flow in west-dipping strata on the east limb of the syncline reaching the surface along faults and associated fracture zones.

Hydraulic conductivity of bedrock at the TMF typically ranges from about 2 x 10^{-6} m/s to 1 x 10^{-7} m/s.

Splitter Dam

The Splitter Dam spans the valley of the South Tributary of Teigen Creek southeast (upstream) of the North Dam. The left (southwest) abutment is underlain by surficial deposits except for a small area of bedrock above roughly El. 1000 m. The right (northeast) abutment of the dam is underlain by bedrock topped by a thin (<4 m) layer of colluvium and organic soils.

A narrow strip of bog deposits (compressible peat and silt) overlying saturated alluvium (<10 m total) is present along the valley bottom. To restrict seepage through these pervious soils, a slurry trench cut-off wall will be constructed through the alluvium and keyed at least 3 m into the lower permeability till or bedrock which underlies the alluvium.

Seismic and surface mapping shows that the left abutment is typically underlain by up to 10 m of alluvium over up to 35 m of till; however, till thins to less than 5 m above about El. 950 m. Till was not encountered in boreholes near the Splitter Dam but is inferred to have properties similar to the till at the North Dam.

Interbedded siltstone and fine-grained sandstone bedrock was encountered near the Splitter Dam site. These rocks are similar to those encountered at the North Dam (hard and medium-strong to strong; RQD 20% to 80%).

Bedrock structure is similar to the North Dam (syncline); however, the valley is wider suggesting that the syncline is less tightly folded. The northwest-striking fault, inferred to be present beneath the North Dam, projects to the middle of the Splitter Dam but it is not known whether the fault extends that far south.

Piezometers installed in the bedrock show the water table slightly below the ground surface, except for two wells near the valley bottom which have small artesian flows.

Saddle Dam

The Saddle Dam is located southeast of the Splitter Dam, near the midpoint of the valley. The dam axis is slightly west of the drainage divide between the South Tributary of Teigen Creek and the North Tributary of Treaty Creek. The left (south) abutment is underlain by till up to about El. 950 m; bedrock covered by roughly 3 m of colluvium and organic soils is present above this elevation. The right (north) abutment of the dam is underlain by bedrock topped by a thin (<1 m) layer of colluvium and organic soils.

The valley bottom beneath the dam contains roughly 20 m of alluvial deposits underlain by till. The till is irregularly distributed but has a maximum thickness of over 60 m in the deepest part of a buried bedrock channel beneath the centre of the dam. The till thins rapidly as it climbs the west side of the valley and is less than 5 m thick at El. 950 m. Bog deposits (compressible peat and silt) are locally present atop the alluvium along the valley bottom.

Alluvial deposits consist of pervious sand and gravels topped by a thin (<3 m) layer of peat and silt. Water content of alluvial samples is as high as 36%, reflecting the lower density of the saturated alluvial soils. STP blowcounts in alluvium are as low as 5 to 10 blows per 300 mm [(N1)60], indicating that some of the alluvial deposits are loose to compact.

Stratigraphy derived from CPT indicate localized, discontinuous layers of potentially liquefiable alluvium are present beneath the Saddle Dam. Removal or remediation of loose, shallow deposits will be necessary. Dam designs were developed to mitigate liquefaction risks from deeper alluvial deposits.

Till deposits are similar to those encountered at the North Dam (fines content of 5% to 40%; water content typically less than 10%; high density). Due to the presence of cobbles and boulders and the compact, dense nature of the matrix, till was typically cored in the vicinity of the Saddle Dam.

Interbedded siltstone and fine-grained sandstone bedrock was encountered near the Splitter Dam site; these rocks are similar to those encountered at the North Dam (hard and medium-strong to strong; RQD typically 40% to 80%).

Bedrock structure is similar to the north dam (syncline); however, the valley is wider suggesting that the syncline is less tightly folded. RQD in borehole KC10-OVB14 located near the center of the crest of the dam is unusually low (mostly <25%, or very poor rock), suggesting the presence of a fault at this location. Piezometers installed in the bedrock show the water table slightly below the ground surface.

Southeast Dam

The Southeast Dam spans the valley of the North Tributary of Treaty Creek and is entirely underlain by surficial deposits. Both the left (southwest) and right (east) abutments are underlain by lateral moraine debris above roughly El. 900 m. The moraine debris is expected to be loose, pervious, and typically about 10 m thick. The moraines are underlain by bedrock.

Below El. 900 m, the left abutment is underlain by a wedge of till that thickens toward the valley bottom from about 5 m to 15 m; the till is underlain by bedrock. Below El. 900 m, the right abutment is underlain by a wedge of alluvium that thickens toward the valley bottom up to a maximum of about 25 m; the alluvium is underlain, in turn, by up to about 20 m of till. The North Tributary of Treaty Creek has incised the till deposits so that only a few metres of alluvium and till are present along the stream channel.

The alluvium consists of fine to coarse sand and gravel. SPT blowcounts are greater than 15 blows per 300 m [(N1)60] indicating the alluvium is compact to dense.

Till deposits are similar to those encountered at the North Dam (fines content of 5% to 40%; water content typically less than 10%; high density). Due to the presence of cobbles and boulders and the compact, dense nature of the matrix, till was typically cored in the vicinity of the Saddle Dam.

Interbedded siltstone and fine-grained sandstone bedrock was encountered near the Southeast Dam site. These rocks are similar to those encountered at the North Dam (hard and medium-strong to strong; RQD typically 40% to 80%).

Bedrock structure is similar to the north dam (syncline); however, the valley is wider suggesting that the syncline is less tightly folded. Piezometers installed in the bedrock show the water table slightly below the ground surface.

LABORATORY TESTING OF TMF AREA SAMPLES

The following laboratory test programs were conducted on samples of overburden taken at mapping sites and recovered in the drill holes:

- index testing (grain size, Atterberg limits and moisture content) of overburden drill and surface grab samples; and
- triaxial testing for strength and hydraulic conductivity of a till sample.

TAILING CHARACTERIZATION AND LABORATORY TESTING

Two tailing streams will be produced by the plant: the bulk rougher flotation tailing[1] representing about 90% of the ore (by dry weight) and a fine, sulphide-rich cleaner tailing comprising 10% of the ore. The sulphide stream will be cyanide leached by the CIL method and then processed for gold recovery. A two-stage cyanide destruction circuit is proposed, using SO_2 followed by a second-stage hydrogen peroxide treatment[2].

KCB conducted laboratory tests in 2009, 2010, and 2012 on samples of flotation tailing and CIL tailing from pilot plant tests by G+T Metallurgical Services submitted to KCB.

During 2011/2012 KCB completed an extensive program of tailing testing with the primary objective of examining performance (i.e. strength and permeability) of cyclone sand and dam drain materials at stress levels corresponding to ultimate dam heights. Results of the testing program showed that cyclone sand produced from the KSM tailing is suitable for construction of the cyclone sand dams as designed.

[1] Referred to as “Flotation Tailing” in this report.

[2] Hence this stream is called “CIL Tailing” in this report.

Tailing and cyclone sand testing was performed on samples provided by a 2011 pilot plant run by G&T. The tailing samples tested consisted of:

- **Flotation Tailing:** a sample of flotation tailing with a P_{80} of 110 µm
- **CIL Residue Tailing:** a sample of CIL tailing with P_{80} = 20 µm
- **Cyclone Underflow Sand:** a sample of cyclone underflow sands for dam construction was prepared by screening flotation tailing to reduce the fines content to 17%. This sample was subjected to stress levels comparable to the base of the dams and tested for permeability before and after these tests.

Flotation tailing is classified as not-potentially acid generating (NPAG) and will be cycloned to produce sand fill for construction of the tailing dams during the summer months. The fine cyclone overflow tailing will be discharged along the upstream crest of the tailing dams. The entire flotation tailing stream will be discharged along the dam crests during the winter months.

The CIL tailing is a high sulphide concentration material and is classified PAG. This material will be deposited underwater in the CIL Residue Cell in the centre of the TMF and kept saturated to mitigate against the onset of acid-generation.

The 2011 KCB laboratory testing program also tested samples of Bowser Group Sedimentary rock found at the TMF site (lightly metamorphosed sandstones and siltstones). This material is proposed to be quarried or borrowed from alluvial deposits and processed for use as drain rock. Rock strength was found to be suitable for use under the loads of the designed heights for the dams.

TMF Area Climate

TMF Area Temperature

Weather data recorded at the Teigen Creek weather station between 2009 and 2011 showed the following:

- The mean annual temperature is approximately 0°C.
- The mean monthly temperatures ranged from -8°C in December-February, to 11°C in July.
- Temperature extremes ranged from -27°C to 29°C.
- Mean daily temperatures are above freezing from May to October.

Freezing temperatures could be encountered from October to May.

TMF Area Precipitation and Hydrology

Based on correlations to longer-term weather data at the Snowbank Road and Eskay Creek Mine and stream flow records in Teigen Creek, estimated mean annual precipitation is 1,371 mm at the TMF (elevation 1,085 masl). The monthly precipitation and runoff distribution is provided in Table 18.5. Precipitation increases at higher elevations within the Teigen valley at a nominal rate of 5% per 100 m.

Figure 18.7 shows the location of the climate and flow measurement stations around the TMF site.

Table 18.5 Average Monthly Precipitation at the TMF Weather Station

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Precipitation (mm)[1]	151	110	123	82	55	69	82	82	165	206	164	82	**1,371**
Pond Evaporation (mm)	0	0	0	0	75	81	86	70	38	0	0	0	**350**
Site Runoff Distribution (%)	1	1	1	3	16	32	19	8	9	7	2	1	**100**

[1] Weather station at El. 1,085 m.

TMF Area Hydrogeology

The TMF is underlain by discontinuous glacial and fluvial overburden and underlain by variably metamorphosed sedimentary bedrock of the Bowser Lake Group. The TMF valley exhibits significant topographic variations. Elevations range between 2,000 m at the peaks and 750 m at the locations of the seepage collection dams. Groundwater levels and flow patterns mimic the topography. This results in generally artesian conditions in the impoundment area with artesian level varying with location in the valley bottom. The artesian conditions provide hydraulic containment of seepage within the TMF valley. Structural features (occasional faults and fractures) control local groundwater flow within the bedrock, which is otherwise competent, intact, and generally uniform over the TMF footprint. As these features have only limited and localized intersections with the TMF structures, grouting can be used to limit their effect on seepage if required.

As a basis for assessment of seepage management, a geologic model was created using overburden and borehole data from the site investigations. The topographic data and the geologic model were combined to create a 3D groundwater model using the FEFLOW® software package. The model included low permeability cut-offs beneath the south and southeast TMF dams to assess the effectiveness of these structures in reducing seepage under the dams. The groundwater model was benchmarked to existing steady-state conditions by inputting observed water levels from wells and stream flow data recorded by Rescan. Once benchmarked, the groundwater model was used to assess seepage from the TMF at the full pond height of elevation 1,060 masl.

Model results completed without the CIL liner indicate total seepage of contact water from the TMF into the regional groundwater system is less than 16 L/s (with proposed cut-offs). Modelling of TMF seepage with the CIL liner present is currently underway and the flux of groundwater in contact with the CIL tailing is expected to be greatly reduced. Seepage from the TMF is controlled by vertical groundwater gradients in the valley and low permeability till materials, which variably underlie the impoundment. The pre-existing and relatively steep groundwater gradients from the surrounding high terrain toward the valley bottom cause any seepage to report to surface within the valley bottom close to the toe of the tailing dam. As a result, contact seepage will report to the seepage collection dams and be returned to the impoundment. This effect reduces the potential for migration of contact water beyond the TMF.

Details of the hydrogeology, FEFLOW® and Seep/W© models of the TMF area and the dams are reported by KCB in the Hydrogeology appendix to the "2011 TMF Engineering Design" report.

18.1.4 KSM Project Area Seismicity

A site-specific seismic hazard assessment was carried out in 2010 and updated with data to 2012 to establish seismic ground motion parameters for the TMF and RSF sites. The seismic hazard assessment was conducted in accordance with the Canadian Dam Association (CDA) (2007) recommendations for seismic hazard assessment.

Based on the KCB seismic hazard assessment, the peak ground accelerations (PGAs) listed in Table 18.6 are recommended for both the TMF and RSF sites. The 10,000-year return period PGA of 0.14 *g* for the TMF site should be associated with an earthquake magnitude of M7.0 in seismic deformation and liquefaction assessments. For the TMF site, spectral accelerations corresponding to the 5% damped Uniform Hazard Response Spectra (UHRS) are recommended as listed in Table 18.7.

Table 18.6 Recommended Design PGAs for TMF and RSF Sites

Return Period (Years)	PGA (*g*)
475	0.04
975	0.05
2,475	0.08
10,000	0.14

Table 18.7 Recommended 10,000-year Return Period UHRS for the TMF Site

Period (sec)	Spectral Acceleration (*g*) (5% Damped)
PGA	0.14
0.1	0.28
0.2	0.32
0.5	0.27
1.0	0.20
2.0	0.10
3.0	0.07
4.0	0.05

The values in Table 18.6 and Table 18.7 show that the seismic hazard in the KSM area is relatively low for sites in BC. For comparison, the site of the Highland Valley Copper Mine (where similar tailing dams have been constructed) has a PGA with a 10,000-year return period of 0.28 *g*.

18.1.5 Design Criteria

RSF and Mine Site Water Management Structure Design Criteria

Design criteria adopted for the prefeasibility design of the mine area facilities are summarized as follows:

- Over a 55-year mine life, the production of 3.03 Bt of mine rock from three open pits requires storage.
- The criteria for dam safety and flood management in Table 18.8 are assessed for the WSD and the Seepage Dam based on the 2007 Dam Safety Guidelines (CDA, 2007).

Table 18.8 WSF and WSF Seepage Dam Safety and Flood Management Criteria

	WSF Dam	WSF Seepage Dam
CDA Consequence Category	Very High	Significant
Seismic	Maximum Credible Earthquake (MCE) ground acceleration of 0.14 *g*	MCE ground acceleration of 0.14 *g*
Diversions	200-year 24-h average daily flow	200-year 24-h average daily flow
Environmental Discharge Flood (EDF) Storage	200-year Average Recurrence Interval (ARI) wet year with diversions operational	Operating surge storage: equivalent to 14 days of WSF seepage and catchment runoff assuming failure of WTP system and 200-year 24-h flood with snowmelt, with diversions operational.
Inflow Design Flood (IDF) Routing	2/3 between the 1,000 year ARI and PMF events, with snowmelt, with diversions failed	500-year 24-h flood with diversions failed
Static Factor of Safety (FOS)	Long term steady state: FOS >1.2	FOS >1.5
	End of construction: FOS >1.5	
	Rapid drawdown: FOS >1.3	
Pseudo-static FOS	FOS >1.0 for a ground acceleration of 50% of the PGA from the 10,000 year seismic event	FOS >1.0
Post-earthquake FOS	FOS >1.2	FOS >1.2
Sediment Control	Minimum water volume of 1 Mm^3	Not applicable

As a result of the reduction in total mine rock with the inclusion of underground development, the footprint of the RSFs will be reduced, with no need to place mine rock in the presently glaciated areas of upper McTagg Valley. In addition to the slightly reduced RSF footprint in the 2012 mine plan, as a result of the smaller

Mitchell open pit and subsequent removal of the North Haul Road, high face dumping will not be required and all RSFs will be placed bottom up.

The criteria shown in Table 18.9 were used for the design of the RSFs, in conjunction with the guidelines developed by the BC Waste Dump Research Committee (1991-1995) and the Mines Health, Safety, and Reclamation Code of BC.

Table 18.9 Geotechnical Stability Design Criteria for Key Areas of RSFs

	Region of the RSF		
	Above the WSF	**Above the Mitchell OPC & Infrastructure**	**McTagg Valley & Areas of Mitchell West**
Construction	Bottom-up	Bottom-up	Bottom-up *
Static FOS	FOS >1.4, considering degradation of the mine rock due to geochemical weathering	FOS >1.3, considering the lower strength clay layer in some portions of the foundation	FOS > 1.1 to 1.3 (Construction period) Long Term >1.3 Toe stabilized by initial bottom up preload from platform construction prior to placement
Rapid Drawdown FOS	FOS >1.25	Not applicable (no reservoir)	Not applicable (no reservoir)
Pseudo-static Seismic FOS	FOS >1.0, for a ground acceleration of 50% of the PGA from the 10,000-year seismic event	FOS >1.0, for a ground acceleration of 50% of the PGA from the 500-year seismic event	FOS >1.0, for a ground acceleration of 50% of the PGA from the 500-year seismic event

TMF Design Criteria

Basic design criteria adopted for the prefeasibility design of the TMF are summarized as follows:

- Production schedule:
  - The mine life is 55 years with a maximum milling rate of 130,000 t/d of ore production for a total of 2.16 Bt of tailing. The average dry tailing density in the impoundment will be 1.5 t/m^3. Although total ore milled is 2.16 Bt, a TMF capacity of 2.30 Bt was selected to provide contingency storage.
- Tailing production:
  - Flotation tailing production is equal to 89.5% of ore production. Sufficient sulphide flotation is achieved in the mill process to achieve NPAG behaviour.
  - CIL Residue tailing production is equal to 10% of ore production. High sulphide concentration results in PAG behaviour. Design of the CIL

Residue Cell allows for storage consistent with 13% of ore production to provide excess CIL storage in case of variations.

- Cyclone underflow sands for dam construction will have a fines content of less than 17% and a minimum percolation rate of 5 x 10^{-6} m/s.

The criteria presented in Table 18.10 for tailing dam safety and flood management are assessed based on the 2007 Dam Safety Guidelines (CDA, 2007).

Table 18.10 TMF Dam Safety and Flood Management

	North Dam, Saddle Dam (Stage 1), Southeast Dam	**Splitter Dam, Saddle Dam (after Stage 1)**	**Seepage Dams (North, Saddle, Southeast)**
CDA Consequence Category	Extreme	Significant	Significant
Seismic	MCE ground acceleration of 0.14 *g* (1:10,000 y event)	MCE ground acceleration of 0.07 *g* (1:2,475 y event)	MCE ground acceleration of 0.07 *g* (1:2,475 y event)
Impoundment Diversions	200-year 24-h peak daily flow, (and re-direct up to 2 m^3/s from the East Catchment to Teigen Creek for base flow)	200-year 24-h peak daily flow	200-year 24-h peak daily flow
EDF Flood Storage	30-day PMF with 100-year snowmelt will be stored in the TMF without discharge, with diversions failed. Seasonal surplus water will be discharged directly into Treaty Creek or treated	30-day PMF with 100-year snowmelt will be stored in the TMF without discharge, with diversions failed. Seasonal surplus water will be discharged directly into Treaty Creek or treated	Operating surge storage: equivalent to 14 days of tailing dam seepage assuming failure of the reclaim pumping system and 200-year 24-h flood with snowmelt with diversions working
IDF Flood Routing			500-year 24-h flood with diversions failed
Static FOS	FOS >1.5	FOS >1.5	FOS >1.5
Pseudo-static FOS	FOS >1.0	FOS >1.0	FOS >1.0
Post-earthquake FOS	FOS >1.2 assuming all uncompacted tailing are liquefied	FOS >1.2 assuming all uncompacted tailing are liquefied	FOS >1.2

The TMF reclaim pond is designed to:

- provide a start-up water volume for one month of mill water
- provide a minimum water depth for floating a reclaim barge and achieving water clarification.

18.1.6 Design of Mine Site Facilities

Rock Storage Facilities

At the PFS-level, there are three primary RSF design considerations:

- foundation conditions
- maximum lift height
- closure slope criteria.

Conservative RSF designs were developed in collaboration with MMTS to address the aforementioned design considerations using existing data. RSF layouts were designed by MMTS, with geotechnical guidance on slope stability and geotechnical recommendations from KCB.

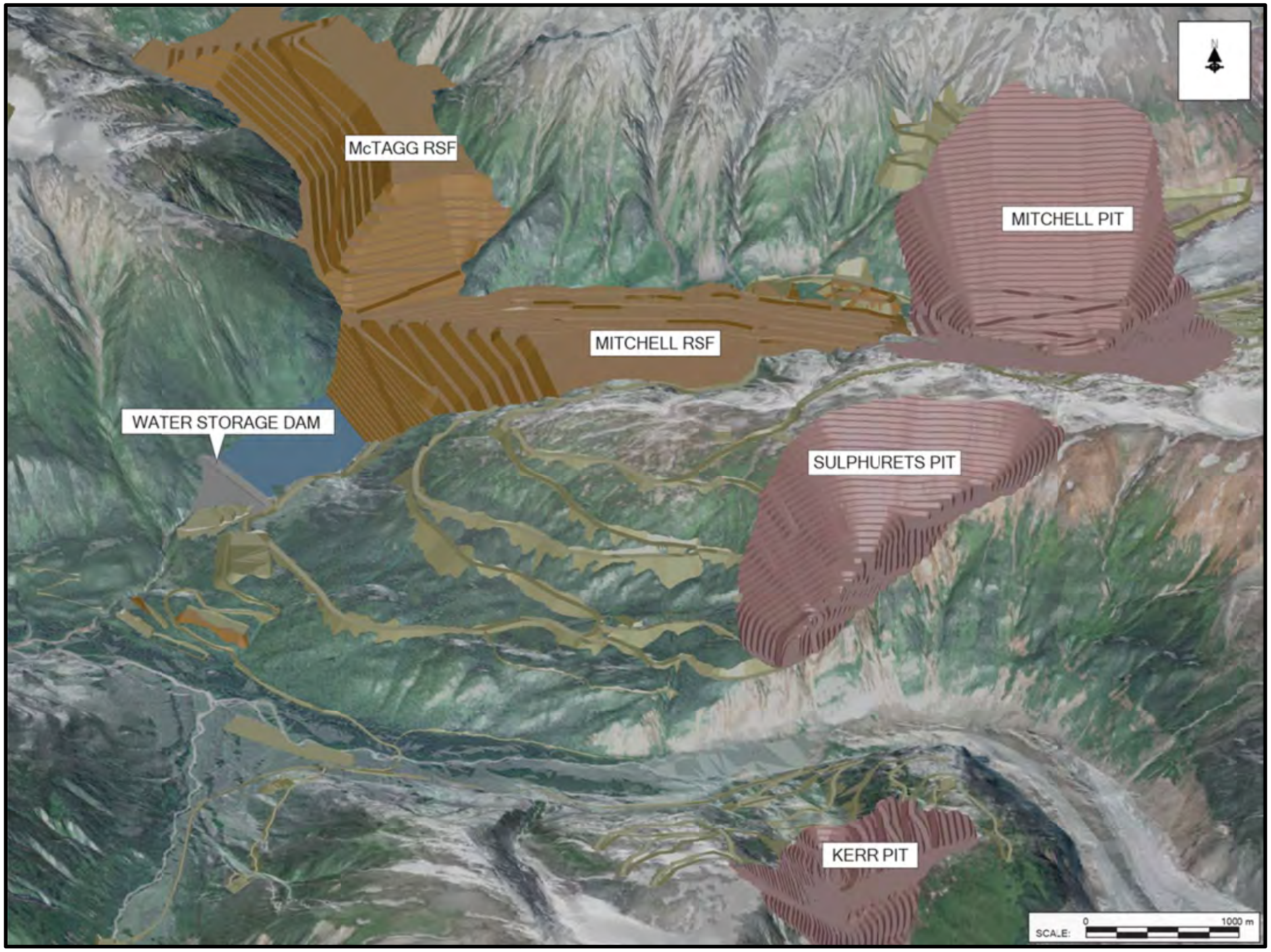
The mine rock placement progression was designed to build RSFs in progressive lifts (bottom-up construction) to initially confine toe areas and consolidate foundations to improve stability and reduce downslope risks. Renderings of the mine site layouts, including the RSFs, at Year 5, Year 10, and end of mine life are provided in Section 16.0.

Prior to closure, final mine rock placement configurations are designed to have maximum 105 m terraces at “as dumped” angle of repose, with flat benches between terraces. The overall resulting final slope angle at the end of operations is 26° to facilitate re-sloping for final closure.

As a result of the reduced footprint, the McTagg RSF layout no longer involves placement of mine rock (over the 55-year mine life) in the upper areas of the east and west forks of the McTagg Valley where significant glacier ice is present.

Current estimates of glacier recession rates suggest that glacier ice will not be present in this area by the time the RSF elevation reaches the toe of the glaciers, and that only minimal ice removal will be required where relic ice is present under rocky debris to establish diversion inlets upstream of the RSFs. Figure 18.9 illustrates the ultimate configuration of the McTagg and Mitchell RSFs at the end of production.

Figure 18.9 Ultimate Mitchell and McTagg RSF Layouts



RSF Site Preparation

Site preparation for the RSFs will include removal of merchantable timber, stripping of organic, weak, and soft soils where required, and proof-rolling where applicable. To prevent blocking of voids in the drain rock, placement of a graded filter blanket may be required where rock drains are constructed over regions of soft overburden (e.g. areas of the Mitchell RSF within Mitchell Valley). Localized basal drains will be created in key natural drainages of the RSF foundations and in areas where drainage through the RSFs is required by procedures to place coarse competent rock such that self-segregation will occur to form permeable drain layers.

RSF Stability Analyses

RSF geotechnical stability during construction and closure was analyzed by KCB using SLOPE/W© 2007 software. A wide range of potential RSF configurations were analyzed. The locations of the three sections presented here are representative of worst-case scenarios for common foundation conditions and down-slope hazards.

The results of selected stability analysis for sections cut across the final RSF layouts are summarized in Table 18.11.

Table 18.11 Results of RSF Stability Analyses – Factors of Safety

Section	Conditions	Factors of Safety	
		Target	Calculated
"A" — Mitchell RSF Facing Mitchell OPC	Static - Year 1 raise	1.3	1.3
	Static - following raises (with toe berm in place)	1.4	1.4
	Pseudo-static (earthquake loads)	>1.0	1.1
"B" — Mitchell RSF Facing	Static - short term	1.4	1.6
	Static - long term	1.4	1.4
	Rapid draw-down of WSF	1.2	1.2
	Pseudo-static (earthquake loads)	>1.0	1.4
"C" — McTagg RSF*	Static - short term	1.4	1.6
	Static - long term	1.4	1.4

* Pseudo-static case not calculated for McTagg RSF as the facility is confined by the valley.

RSF Safety and Monitoring

Mine staff will perform the following inspections of the RSF daily, looking for indications that would provide advanced warning of a possible RSF failure:

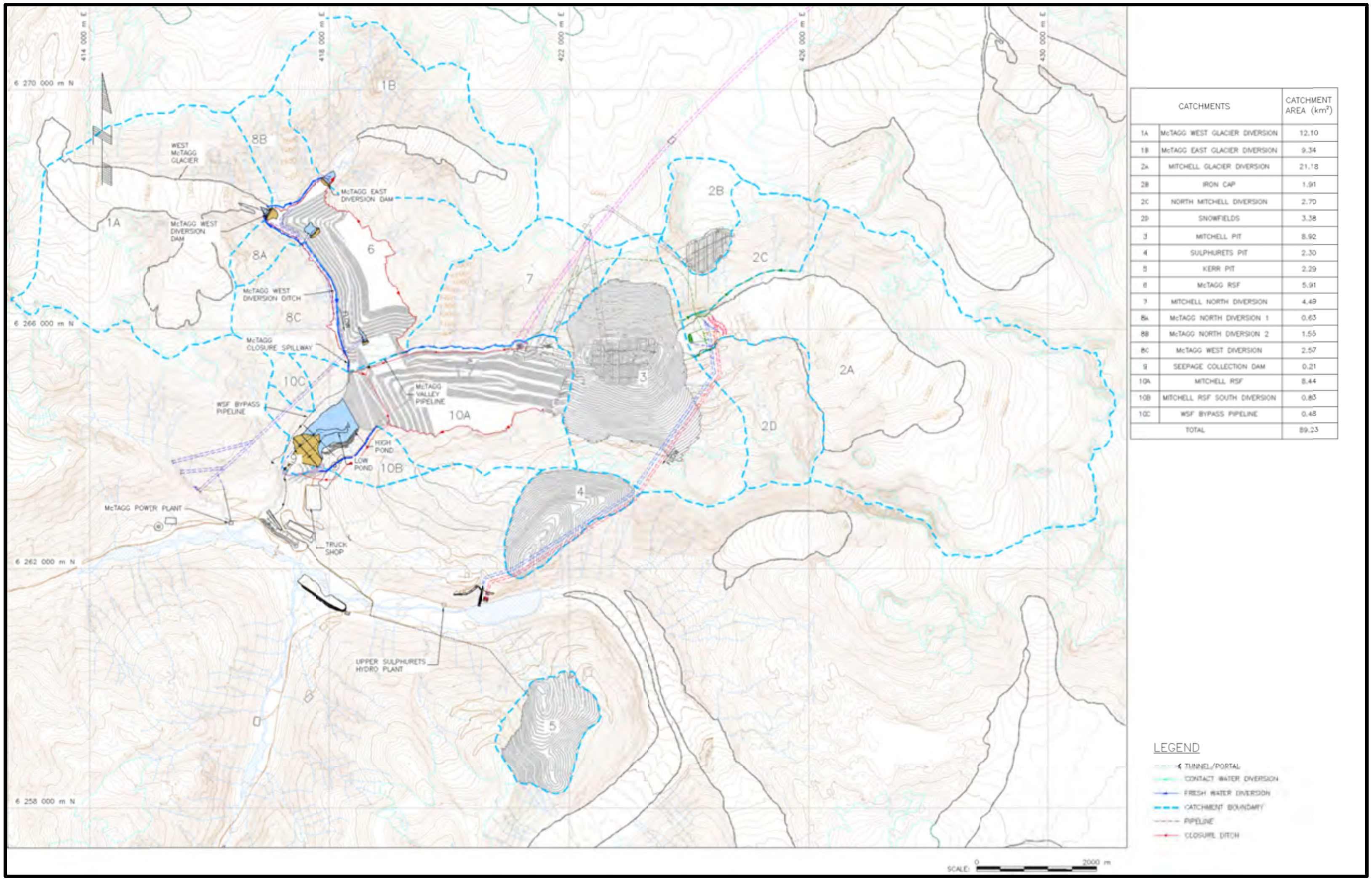
- readings of electric piezometers installed in foundation silt and clay layers
- readings of inclinometers installed in areas close to the mill/plant facilities and the area facing the water storage pond

- monitoring of surface survey monuments installed during RSF construction; the use of automated radar or optical scanning instruments will be considered for areas with higher downslope hazards.

18.1.7 MINE AREA WATER MANAGEMENT

Figure 18.10 illustrates ultimate water management structures as existing at the end of mine life, showing diversion tunnel routes and operational phase surface diversions. Catchment boundaries are indicated in light blue.

Figure 18.10 Mine Site Ultimate Water Management Plan



CATCHMENTS		CATCHMENT AREA (km^2)
1A	McTAGG WEST GLACIER DIVERSION	12.10
1B	McTAGG EAST GLACIER DIVERSION	9.34
2A	MITCHELL GLACIER DIVERSION	21.18
2B	IRON CAP	1.91
2C	NORTH MITCHELL DIVERSION	2.70
2D	SNOWFIELDS	3.38
3	MITCHELL PIT	8.92
4	SULPHURETS PIT	2.30
5	KERR PIT	2.29
6	McTAGG RSF	5.91
7	MITCHELL NORTH DIVERSION	4.49
8A	McTAGG NORTH DIVERSION 1	0.63
8B	McTAGG NORTH DIVERSION 2	1.55
8C	McTAGG WEST DIVERSION	2.57
9	SEEPAGE COLLECTION DAM	0.21
10A	MITCHELL RSF	8.44
10B	MITCHELL RSF SOUTH DIVERSION	0.83
10C	WSF BYPASS PIPELINE	0.48
TOTAL		89.23

DIVERSION TUNNELS

The construction of two diversion tunnel systems is required before mine start-up. The MDT (Mitchell Diversion Tunnels) and the MTDT (McTagg Diversion Tunnels) are designed to route glacial melt water and non-contact valley runoff around the proposed mine area.

Each diversion tunnel has been twinned to provide redundancy against blockage and to allow switching of base flows between the adjacent twin tunnels if access for maintenance is required.

The tunnels required at mine start-up are sized to convey design storm flows with each of the twin tunnels sharing the design flow of the diversion. Each side of the twin tunnels has been designed to have capacity to carry peak annual freshet flows in the event the other tunnel becomes blocked.

Hydraulic design of the diversion tunnels allows the tunnel flows to either run partially full as a free flow tunnel at design capacity, or in the case of the later stages of the MTDT, to run completely full as pressure tunnels. Inlet controls and drain-down systems have been provided to allow shutdown of the tunnels for maintenance as required.

MITCHELL DIVERSION TUNNELS

The MDT is designed to be constructed in two phases. The open pit phase is constructed at start-up and is the same system as was designed to protect the open pit in the 2011 PFS Update. Design capacity of the open pit phase MDT is the 200-year, 24 hour average event.

The twin MDTs for the open pit phase each have 17.7 m^2 cross sections and are 5.6 km long. The tunnels route water from Mitchell Creek/Mitchell Glacier to the Sulphurets Valley, away from the open pit, Mitchell OPC area and the Mitchell rock storage area. The Mitchell Diversion inlets will include two separate types of inlet structures to improve redundancy in the collection of melt water. Inlets will be located both underground (beneath the base of the ice) as well as on surface at the toe of the Mitchell Glacier. Both of these inlets are designed to continue functioning during snow avalanches. The open pit phase diversion is provided with a stepped spillway outlet into Sulphurets Lake capable of dissipating continuous flows of up to the flow of the 200-year event.

In Year 26, underground mining will commence and an additional set of tunnels (underground, or underground phase MDT) will be constructed parallel to the open pit phase tunnels in order to add the capacity required to protect the underground workings. Design criteria for the underground phase consist of diverting the 1,000 year instantaneous daily peak event. This is achieved by the addition of higher capacity 5.1 m by 6.6 m twin tunnels, with steeper grade at 1.5% and thus higher velocity in parallel to the initial phase Mitchell Diversion. The higher velocity of the underground phase storm tunnels is acceptable against erosion as they only see occasional flows that exceed the 200-year, 24-hour average capacity of the open pit phase diversion. The underground phase storm diversion tunnels utilize a flip bucket outlet that discharges to Sulphurets Lake from portals adjacent to the stepped spillway of the open pit phase outlet.

Flows in the open pit phase MDT will be available from mine start-up throughout closure to generate hydroelectric power, as Sulphurets Valley is lower than Mitchell Valley.

Figure 18.11 illustrates the MDT and intake structures.

McTagg Diversion Tunnels

Alignments and inlet locations of the MTDT were revised for the 2012 KSM PFS, and diversion design capacity has been increased to the 200-year, 24-hour peak flow. As the staging plan of the RSFs has changed, the route of the initial phase tunnel has been shortened to divert water from an inlet just above the confluence of McTagg and Mitchell creeks. This change shortened construction time required and reduces capital costs.

The Stage 1 twin MTDTs each have a 27 m^2 cross section and are 3.75 km long. Initially, the Stage 1 section of the tunnels will be a free flow, unpressurized tunnel and will not be used to generate power as head available is relatively low due to the low initial elevation at the Stage 1 inlet. As mine life progresses, the tunnels will be extended further up McTagg Valley, and the Stage 2 and Stage 3 inlets will be constructed at progressively higher elevations in the valley. In each stage, the flood outlets of the tunnels will be raised higher in Gingrass Creek, resulting in the pressurization of the initial tunnel section and higher head available for power generation. The flows diverted in Stage 2 and 3 begin to be used to generate hydroelectric power, which continues to be produced on into closure. At Stage 3, two inlet branches of the tunnel will be established to collect flows from the East and West McTagg valleys to feed flow into the diversion tunnel route. Staging plans for the MTDT are shown in Figure 18.12.

Figure 18.11 Mitchell Diversion Tunnels

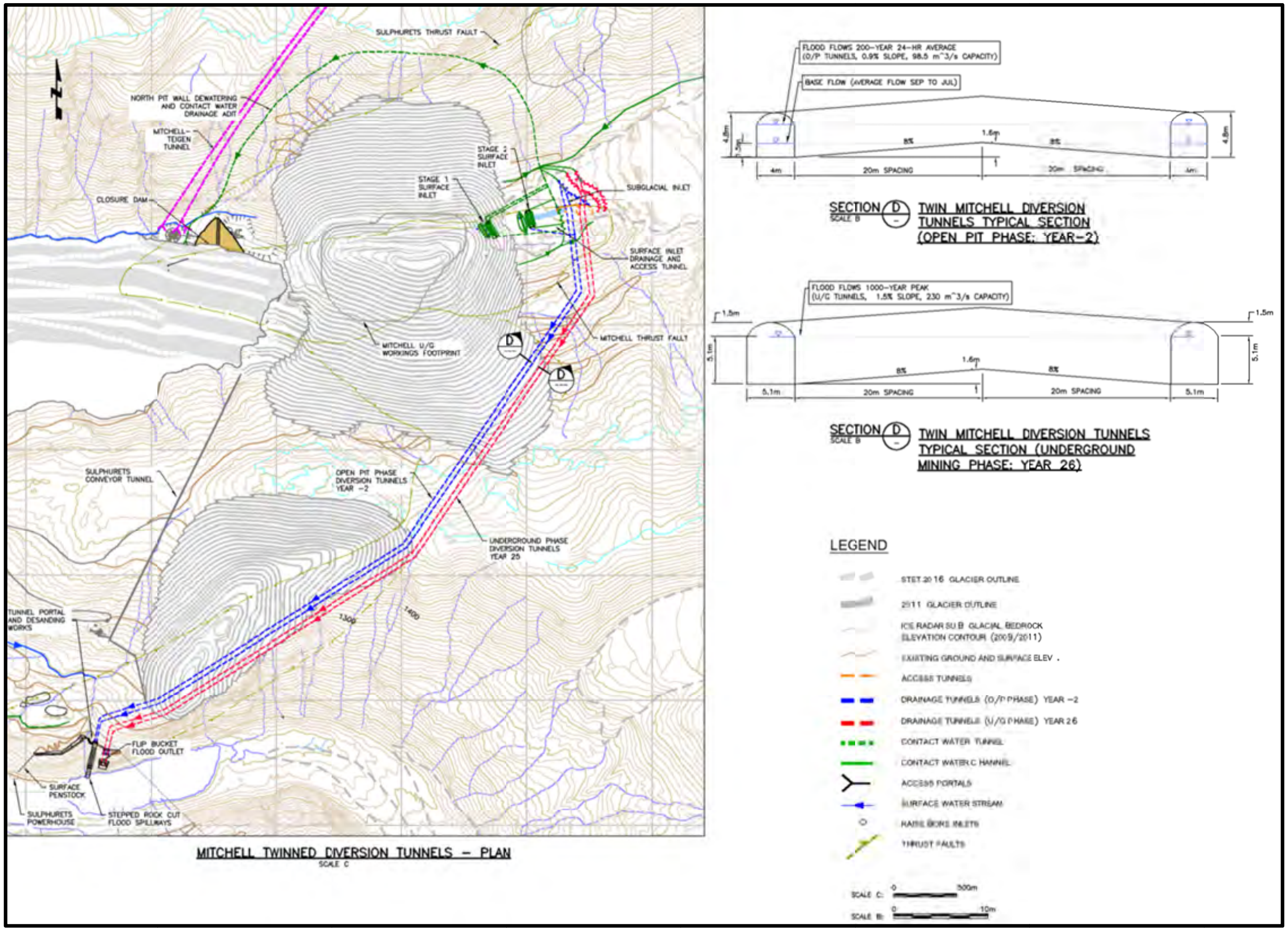
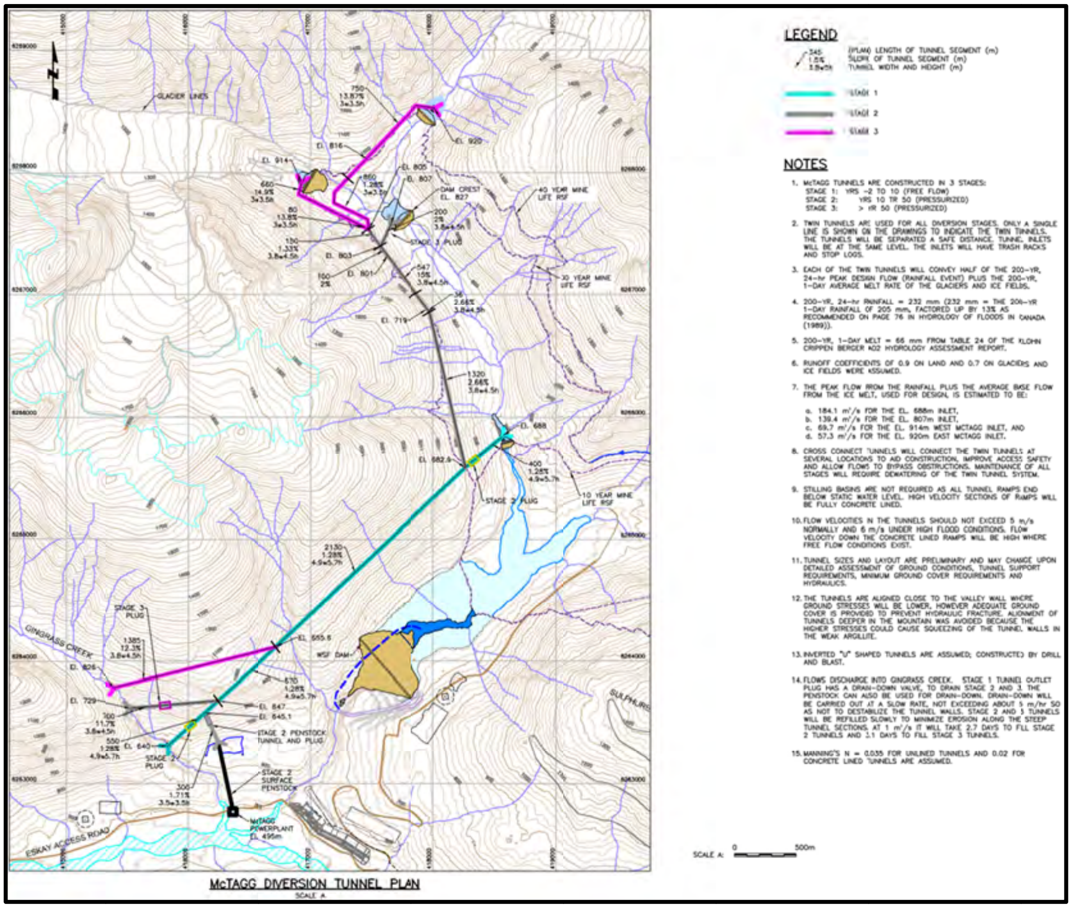


Figure 18.12 Staging Plan for McTagg Diversion Tunnels



Design of Mine Area Diversion Channels

Surface fresh water diversion channels will be constructed progressively during operations along the toes of the Mitchell and McTagg RSFs and above the temporary Sulphurets RSF. Lined channels will be created at the margin of the RSFs against the natural ground of the hillsides in order to divert surface flows.

At the junction of the McTagg and Mitchell RSFs, a spillway will be constructed in rock at the southwest corner of the McTagg dump to convey flow down to the WSF bypass pipeline, located along the west side of the WSF pond. Peak storm flood flows in excess of the 200-year storm event capacity for the MTDT and WSF diversions will be routed through the RSF into the WSF.

Contact Water Collection Systems

Contact water pumped out of the Mitchell pit will be allowed to drain through the Mitchell RSF into the WSF via engineered rock drains incorporated in the RSF. Contact water from the temporary Sulphurets RSF and Sulphurets pit will be collected in a lined channel and routed into the WSF. Contact water from Kerr pit and the seepage dam catchment will be routed directly to the WTP in a gravity pipeline. Contact water from the Sulphurets RSF will be collected in a channel and routed to the WSF. Water pumped from the Mitchell block cave developments will be routed to the WSF. Water flowing from the Iron Cap block cave will be routed by gravity through the North Wall dewatering adit and into the WSF. Natural contact water from the mineralized slopes above south and north sides of the Mitchell Glacier will be routed through the North Wall dewatering adit into the WSF and treated before discharge.

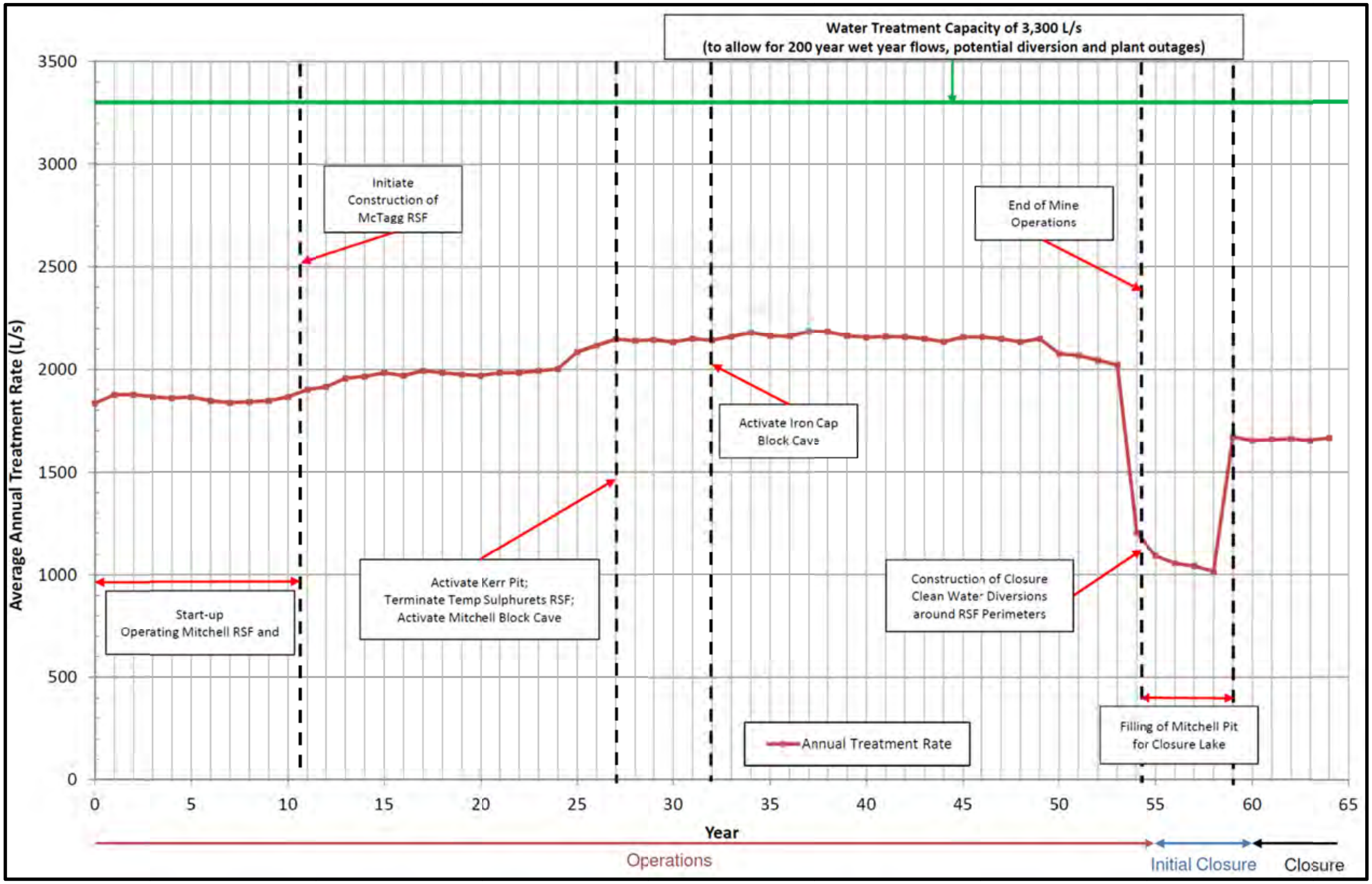
WATER STORAGE FACILITY

Seepage requiring treatment from the Mitchell, McTagg, and Sulphurets RSFs will be collected in the lower Mitchell area by the WSD. This rockfill earth core dam will create the WSF pond, which will be large enough to handle seasonal freshet flows as well as volume accumulated from a 200-year wet year.

Figure 18.13 shows required water treatment rates in the WSF over the life of mine. The plot shows three cases and indicates the effect of mine operations milestones.

The base case shown corresponds to treating all contact water in Mitchell Valley, including natural flows from mineralized areas upstream of the Mitchell deposit. Seabridge has elected to treat this naturally occurring runoff instead of diverting it in order to improve water quality of the water diverted by Mitchell Diversion. The two other cases show the effect on the treatment rate of not treating the natural flows of North and South Mitchell Glacier slope waters.

Figure 18.13 Mine Area Water Balance Curve for WSF



Design of the WSD is shown in section view in Figure 18.14. The dam will be located in the lower Mitchell area where competent rock foundation conditions are present. The dam will be raised in two stages. Due to lower volumes of contact water that will be stored in the WSF during the first 10 years of operation, the WSF will be initially built to a starter crest elevation of 708 masl (149 m height). The ultimate final crest elevation will be established at 716 masl (165 m height) in Year 10. Emergency spillways designed to route the IDF will be cut into rock on the southeast side of the dam for each stage. There will be appropriate freeboard for avalanche wave mitigation and flood routing.

Water in the WSF is predicted to be acidic, similar to existing water in Mitchell Creek. Two asphalt cores are included in the earth core to control seepage and provide redundancy against leakages. The two cores also allow for the control of fresh water saturation levels within the dam and the adjacent foundation independently from that of the reservoir level if required. Asphalt is inert with respect to acidic water. The low permeability central earth zone of the dam will be wider than normal to provide additional resistance against potential acid degradation. WSD and seepage dam foundations will be grouted at 2.5 m spacing. Based on drilling results, the depth of the grout curtain has been designed to vary from 25 m at the west abutment to 150 m at the east abutment to control seepage.

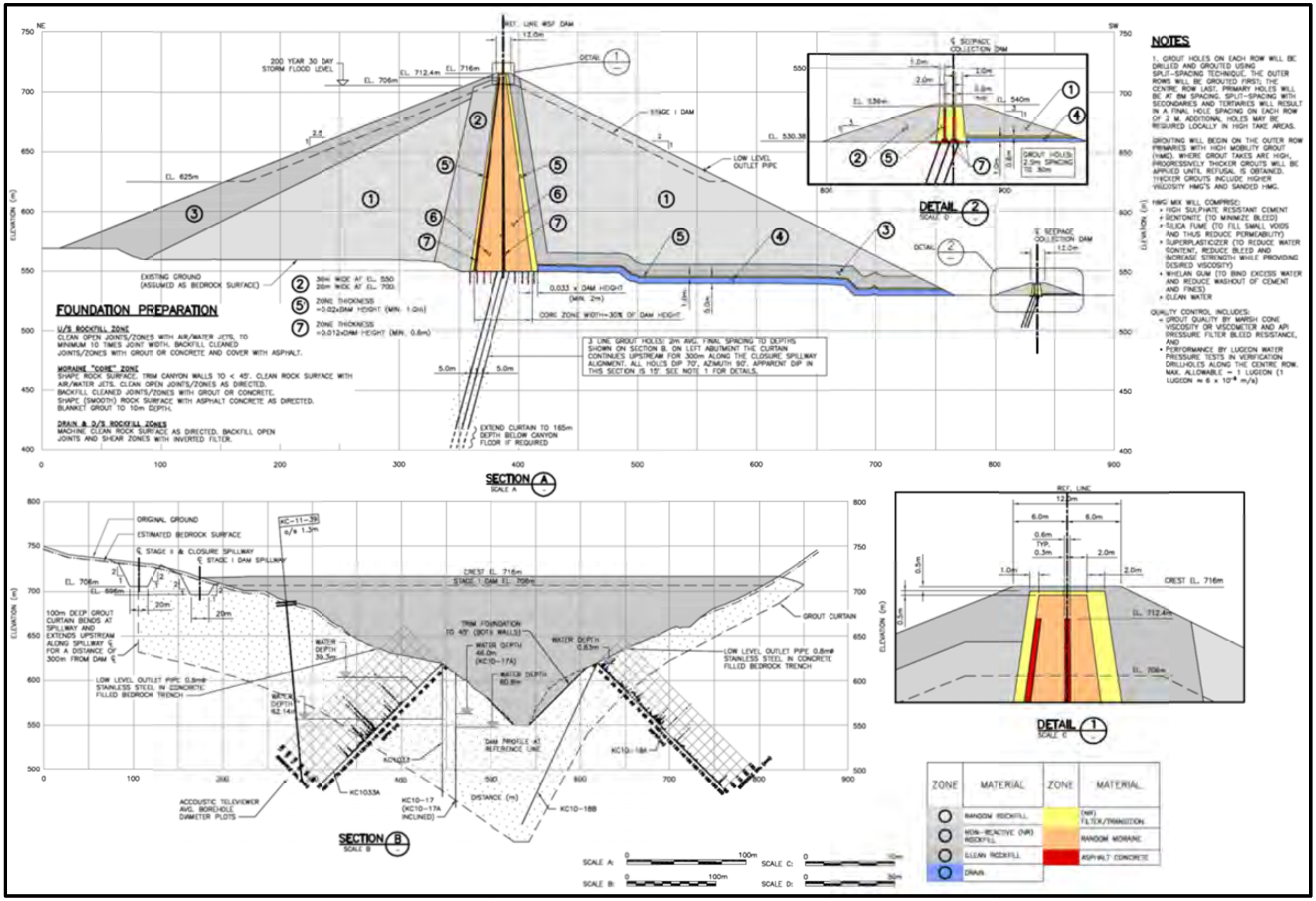
Fill for dam zones is specified such that critical zones of the dam (sections in contact with the core) and drain zones are constructed with materials that have low potential to react with acidic water.

Low-level outlet pipelines are provided under the WSD. These stainless steel pipes are equipped with control valves and are set in asphalt-covered, concrete-filled bedrock trenches that pass under the dams. The pipeline routes curve up onto the walls of the valley just upstream of the dam. Within the impoundment, the outlet pipelines are equipped with trashrack inlets at 20 m elevation intervals. The pipelines terminate above impoundment elevation at cleanout ports, which allow the entry of pipe cleaning equipment if required. A suction dredge and floating pipeline are provided to clean out sediment from the impoundment, which can be placed in cells on the RSF surfaces to dewater.

A high-density polyethylene (HDPE) lined steel penstock leads from the outlet of the low-level outlet pipelines to the energy recovery plant and WTP situated below the WSD.

There will be a 20 m high seepage dam located downstream of the WSF. The seepage dam will have 3H:1V side slopes with an impervious asphalt core. Water collected in this dam will be sent directly to the WTP by an HDPE gravity pipeline. During construction, the HDPE pipeline will route runoff and sediment from the construction of the WSD and construction diversion tunnel to the total suspended solids (TSS) polishing ponds at the WTP site for sediment removal.

Figure 18.14 Water Storage Facility Water Storage Dam Section



A 900 m-long, 4.3 m by 4 m diversion tunnel and cofferdam were included in cost estimates to route the creek around the dam footprint during construction.

HYDROELECTRIC POTENTIAL OF DIVERSIONS

Diverting Mitchell and McTagg creeks into tunnels creates an opportunity for hydroelectric power generation. Generated power can be used during mine operations or sold to the grid via the power lines through the MTT. During operations, the hydroelectric plants will reduce the power requirements of the mine. Upon mine closure, the hydroelectric plants will continue in operation to generate income and offset water treatment costs.

Brazier has assessed the hydroelectric capacities and revenue for the diversion hydroelectric facilities. The Mitchell Diversion has a designed hydroelectric generating capacity of 7.5 MW. The McTagg Diversion comes into operation in Phase 2 (Year 10) of the diversion and has an installed capacity of 15.5 MW.

Characteristics of hydroelectric plants installed on the diversions are similar to run of the river installations, in that they provide peak power during freshet flows.

18.1.8 WATER TREATMENT

TEMPORARY MINE AREA CONSTRUCTION PERIOD WATER TREATMENT

During construction, four temporary water treatment facilities will operate during the construction period to manage potential metals, TSS, and ammonia in drainage from tunnel portals and from temporary stockpiles of tunnel muck near the portals and other flows of contact water. These installations will include semi-automated lime and polymer flocculent dosing systems, reactor tanks, agitators, launders, and engineered systems of settling ponds. Aeration systems will be provided to reduce ammonia loadings generated from explosives use.

For areas of the site where only TSS treatment has been identified as required (such as soil borrow area and soil cuts), a total of 16 automated flocculent treatment systems and sediment ponds will operate to deal with suspended sediments generated during the construction period. These treatment sites will be situated below earthworks and at the portals of the tunnels identified as requiring only TSS and ammonia control.

PAG tunnel muck will be stored on lined pads located at the temporary water treatment sites adjacent to the tunnel portals and Mitchell OPC cuts. Where the temporary pads are located outside the WSF catchment, this material will be hauled to permanent disposal sites within the WSF catchment once the diversion tunnels and the main WTP are operational.

During the construction start-up period, temporary treatment plants for TSS removal and water treatment are provided in the proposed mine area. These are intended to deal with drainage from existing mineralized zones, PAG cuts, tunnel portals, and PAG tunnel muck piles during the construction period before the WTP is in operation.

The WSF and WTP are planned to be in operation during the pre-production period to capture sediment and runoff from stripping and fill placement during Mitchell OPC and haul road construction.

MINE AREA WATER TREATMENT PLANT

During operations, contact water from mining operations and natural seeps in the area will be treated with a high density sludge lime WTP. Data taken between 2007 and 2011 combined with regional long term records and water balance calculations indicate that during the various stages of mine life, the WTP will operate year round at a constant rate of 1.7 m^3/s to 2.2 m^3/s. Annual variations in base case water treatment rates are shown in Figure 18.13; this includes additional treatment flows associated with treating natural contact water flows from upstream of the Mitchell Deposit. The WTP also has additional capacity in the form of a spare clarifier and reactors provided to treat up to 3.3 m^3/s to manage system “upsets” that may occur due to natural hazards or extreme events such as the 200-year wet year. The additional treatment capacity also allows sections of the WTP to be shut down for maintenance when required.

Sludge from the WTP will be dewatered by filter presses at the WTP and trucked to the Mitchell OPC and added to the ore conveyor and passed through the ore milling process. The sludge will contribute alkalinity necessary to the process; it will then be disposed of within the tailing pond. Additional hydropower will be generated from the flow of treatment water from the WSF to the WTP as the treatment plant is located at a lower elevation in the Sulphurets Valley.

Brazier estimates that between 1.7 and 1.9 MW of hydropower will be generated by the WTP energy recovery turbine over the first 30 years of mine life at a relatively constant rate throughout each year, from contact water flowing from the WSF reservoir to the WTP. The annual generation of power will average 15,451 MWh/a over the first 30 years. The turbine will be sized large enough to handle the maximum treatment flow rate of 3.3 m^3/s.

18.1.9 MINE AREA CLOSURE PLAN

During operations, mine rock will be placed close to the final closure configuration in the RSFs and re-contoured locally as necessary at closure. RSFs will be primarily built bottom-up, resulting in stable mine rock slopes during operation and closure. RSFs will have a 0.5 m till cover applied to promote plant growth in areas below the treeline and the RSF surface will be re-vegetated where appropriate.

Upon closure, the diversion tunnels, hydroelectric plants, and the WTP energy recovery facility will remain in operation, generating more power than required to operate the WTP, and also generating income to offset treatment and maintenance costs.

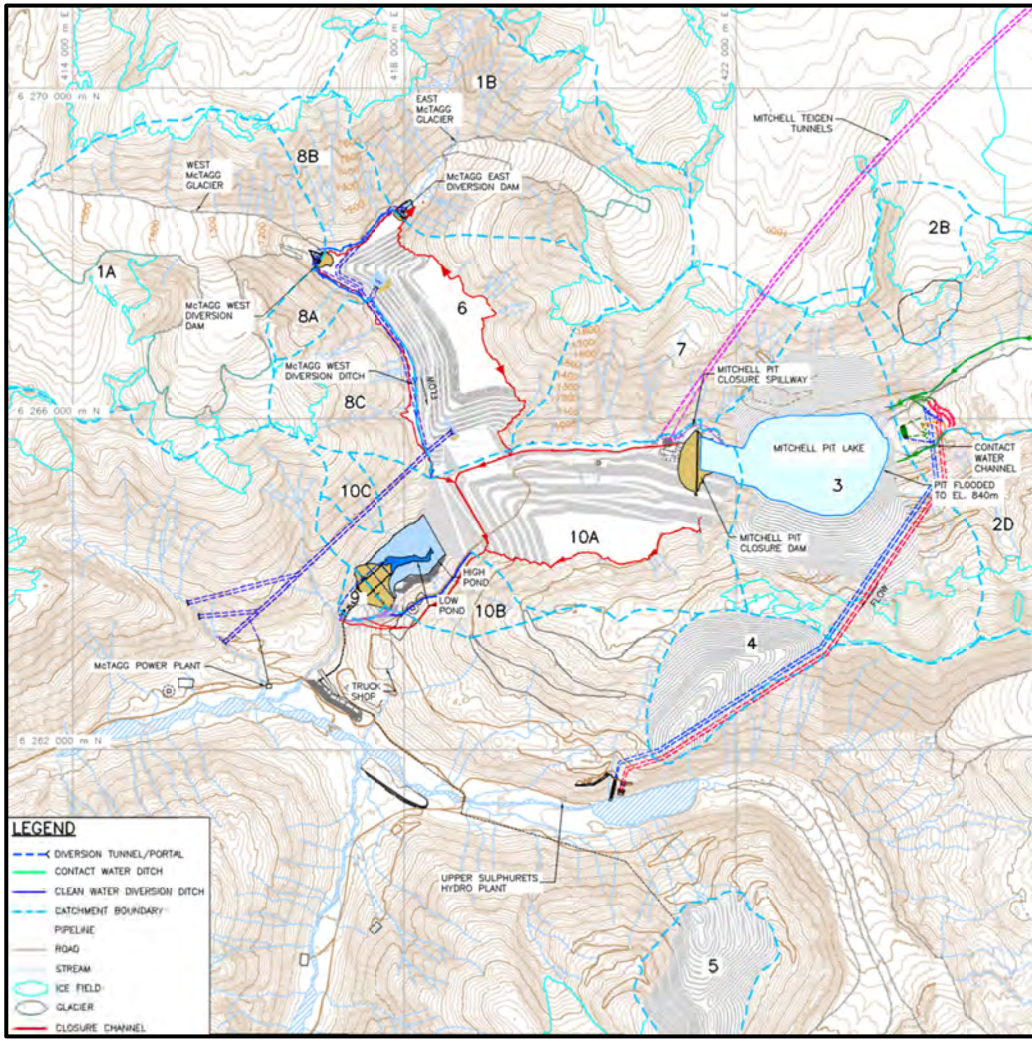
Upon closure, the diversion channels maintained on the McTagg and Mitchell RSF surfaces during operations will be upgraded to closure channels capable of conveying the predicted maximum flood. Upon decommissioning of the mine, the RSFs will be contoured such that lined surface closure channels are present to route clean water flows around the RSFs, in the event of failure of the diversion tunnels or to handle extreme flood events.

The Mitchell closure dam will be constructed to 870 masl across the valley on the west side of the Mitchell pit to allow partial flooding of the pit walls and to flood the collapsed underground workings located beneath the base of the pit. The Mitchell closure dam is largely constructed of mine rock placed as part of the RSF development, but will have an acid resistant asphalt core keyed into the shallow bedrock rock present near the pit rim. The closure dam has been moved down the Mitchell Valley to be outside the zone of instability resulting from subsidence of the block cave workings. Closure dam freeboard of 30 m above the pit lake elevation of 840 m has been provided to mitigate against potential waves from avalanches and landslides into the pit lake.

A spillway around the closure dam will be constructed upon closure into rock on the north side of the valley. Normal flows from runoff and precipitation into the pit spill into the Mitchell RSF basal drain through a spill pipe around the dam and report to the WSF for treatment. Larger flows such as storm flows beyond the 1,000-year peak flow capacity of the tunnels will pass into the pit lake where they are attenuated by storage provided in the pit lake behind the dam, and are slowly passed through the spill pipe to the WSF for treatment. Extreme events such as the PMF overtop a 10 m high weir into the closure dam spillway, and spill into the closure channel constructed along the north side of the Mitchell Valley. These flows will be routed around the WSF.

The closure water management plan is shown in Figure 18.15.

Figure 18.15 Mine Site Closure Water Management Plan



CATCHMENTS		CATCHMENT AREA (km²)
1A	McTAGG WEST GLACIER DIVERSION	12.10
1B	McTAGG EAST GLACIER DIVERSION	9.34
2A	MITCHELL GLACIER DIVERSION	21.18
2B	IRON CAP	1.91
2C	NORTH MITCHELL DIVERSION	2.70
2D	SNOWFIELDS	3.38
3	MITCHELL PIT	8.92
4	SULPHURETS PIT	2.30
5	KERR PIT	2.29
6	McTAGG RSF	5.91
7	MITCHELL NORTH DIVERSION	4.49
8A	McTAGG NORTH DIVERSION 1	0.63
8B	McTAGG NORTH DIVERSION 2	1.55
8C	McTAGG WEST DIVERSION	2.57
9	SEEPAGE COLLECTION DAM	0.21
10A	MITCHELL RSF	8.44
10B	MITCHELL RSF SOUTH DIVERSION	0.83
10C	WSF BYPASS PIPELINE	0.48
11	SULPHURETS RSF	0
12	SULPHURETS RSF (UPHILL)	0
TOTAL		89.2

18.1.10 TAILING MANAGEMENT FACILITY DESIGN

Mine life for the 2012 KSM PFS is 55 years. The design capacity for the TMF is 2.3 Bt, based on 2.16 Bt of ore plus contingency for expansion. Based on tailing consolidation test work and modelling done in 2010 and 2011, an average in situ tailing density of 1.5 t/m^3 was used.

The TMF (Figure 18.2) will be located within a cross-valley between Teigen and Treaty creeks. Three cells will be constructed: the North Cell and South Cell will store flotation tailing, and the CIL Residue Cell (fully lined with a geomembrane) will contain PAG CIL residue tailing.

Dams will be constructed using the centerline construction method with compacted cyclone sand shells and low permeability glacial till cores (amended with bentonite where necessary and incorporating geomembranes for dams retaining the CIL tailing). The dams will be progressively raised over their operating life to an ultimate elevation of 1068 m.

Process water in the flotation and CIL tailing cells will be reclaimed by floating pump barges and recycled to the plant. Diversions will route non-contact runoff from surrounding valley slopes around the TMF. Diversion channels are sized to allow passage of design flows and are have large enough base widths to allow space for snow removal machinery. Buried pipe sections are used in active snow avalanche paths.

The Staging Plan (Figure 18.16) shows the sequence of operation of the TMF. The North Cell will be filled first; simultaneously, the CIL cell will be operated. During operation of the North Cell, floods will be routed south. A pipeline and surface channel will divert environmental maintenance flows (of up to 2 m^3/s) from the east catchment around the TMF into Teigen Creek. As the operation switches to the South Cell, the East Catchment Tunnel will be used to route east catchment flood flows away from the South Cell.

Seepage from the impoundment will be controlled with low permeability zones in the tailing dams and dam foundation treatment. Seepage and runoff water from the tailing dams will be collected downstream at seepage collection dams and pumped back to the TMF. The ponds behind the collection dams will also be used to settle solids eroded by runoff from the dam and fines from cyclone sand construction drain-down water.

Water balance calculations, based on site data taken between 2007 and 2011 combined with regional long term records, indicate that the TMF will have an average surplus of water of 0.53 m^3/s during North Cell operation, 0.82 m^3/s during the transition from North to South Cell, and 0.41 m^3/s during South Cell operation.

Management of surplus water may use a combination of storage, discharge during freshet, or treatment and discharge.

After initial closure, channels constructed between the cells will allow water levels in the North, CIL, and South Cells to equilibrate to the same elevation. Runoff will be routed through a closure spillway to Treaty Creek.

TAILING STAGING PLAN

The layout of the TMF is shown on Figure 18.2; sequences of cell stagings are shown in Figure 18.16. Tailing flows will be routed by gravity in slurry pipelines from the plant to the North Cell. Energy will be recovered during early years of operation of each cell from discharge of the tailing into the impoundment. Tailing will be pumped to the CIL and South Cell when required during later stages of the project.

Tailing flows will be retained by four cyclone sand tailing dams (North Dam, Splitter Dam, Saddle Dam and Southeast Dam) with designs similar to those used at the currently operating Highland Valley tailing facility.

During operation, elevations of annual dam crest raises will be set to provide 12 months of tailing storage and to store the PMF[3] of 53 Mm^3 with 1 m of freeboard.

The North Cell will be constructed first (Figure 18.16); it will store flotation tailing production for 25 years, and then be closed and reclaimed over a five year period. The CIL Residue Cell will be constructed and operated in parallel with the North Cell, and will be filled to about half its capacity with PAG CIL Residue Tailing. At Year 25, the South Cell goes into operation, providing flotation tailing storage for the remaining mine life. At the end of this period, the CIL Residue Cell will be filled to ultimate capacity. Subsequently, the South and CIL Residue Cells will be closed and reclaimed over a five year period.

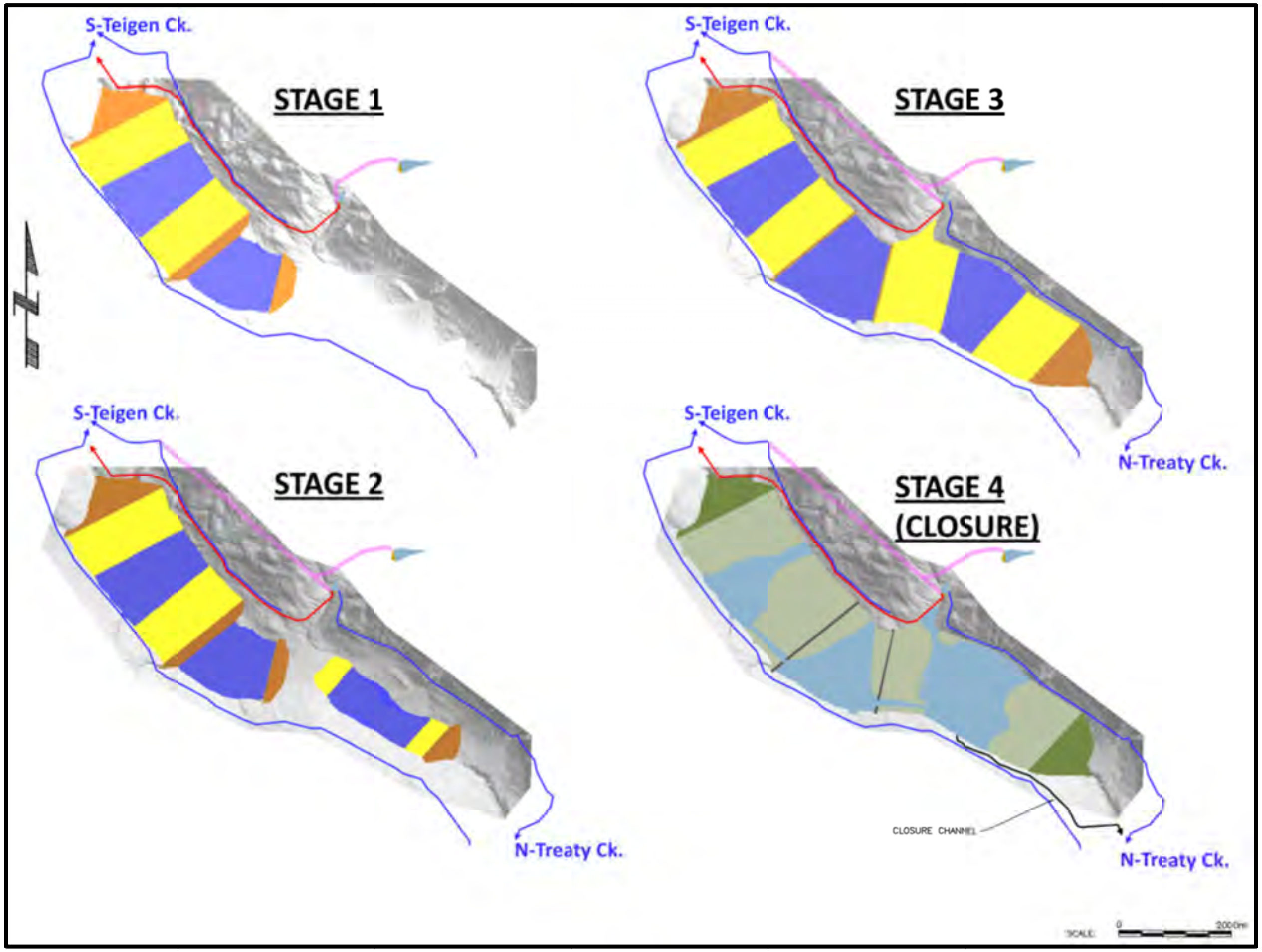
Earth fill starter dams at the North, Splitter and Saddle Dam sites will be initially constructed to store a minimum of 8.4 Mm^3 of water for mill start-up and to contain the first 12 months of tailing production[4]. The dams will then be annually raised by cycloning tailing sand. Cyclone sand raises will continue each year to 1,068 masl. A 700 m-wide beach in the North Cell will create a region of separation between the reclaim pond and the dams by the end of the first year of filling. This separation will increase to 1,200 m at the ultimate elevation. The separation between the tailing beach and pond increases the margin of safety against overtopping of the tailing dam, and mitigates seepage through the tailing dam and underlying foundation stratum.

A starter dam at elevation 920 masl will be completed by Year 25 to allow deposition to begin in the South Cell. Between Year 25 and closure at Year 55, the Southeast Dam will be raised to its ultimate elevation.

[3] The design operating PMF ranges from 65 Mm^3 at start-up to 53 Mm^3 at the ultimate dam elevation.

[4] Final height of starter dams to be re-evaluated during feasibility design based on initial tailing production ramp-up and seasonal timing of expected mill start-up.

Figure 18.16 TMF Staging Plan



Note: Raising of cyclone dams within each stage not shown on these diagrams.

Legend: ***brown*** *= operating dam* ***yellow*** *= operation tailing surface* ***dark blue*** *= process water*
light blue *= runoff water* ***light green*** *= reclaimed tailing surface* ***dark green****= decommissioned dam surfaces.*

TMF DAM STRUCTURES

Over an initial two-year construction period, three earth fill starter dams will be constructed at the North Cell (North, Splitter and Saddle) to provide start-up flotation and CIL tailing storage for two years. These dams will be progressively raised over their operating life to an ultimate elevation of 1068 m, providing storage capacity of 2.3 Bt. A summary of the tailing dams is provided in Table 18.12.

TMF Starter Dams

Starter dams will be earthfill embankments, with shells of compacted random fill supporting the central till cores. Cores will be keyed into the underlying foundations to cut off seepage through weathered near-surface soils and any pervious strata. A drain blanket is provided to depress the phreatic levels in the downstream half of the dam. Riprap erosion protection will not be placed on the upstream slope due to the temporary exposure of the dam to the pond water. Figure 18.17 and Figure 18.18 also show typical sections of the starter dams.

Main Tailing Dams

The North, Splitter, Saddle, South, and Southeast dams will be compacted cyclone sand dams with vertical till cores constructed by the centreline method. Dimensions of the dams are summarized in Table 18.12. Details of the North, South, and Southeast dam designs are shown in Figure 18.17, Figure 18.18 and Figure 18.19.

A system of finger drains will be installed in downstream areas of the foundations at the base of the cyclone sand to keep water levels in the dam depressed. Main drains in the centre of the valley floor will collect and convey seepage to the toe of the dam. Smaller secondary drains will convey water laterally into the mains drains.

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 located above the west abutment of the North Dam. Fine cyclone overflow will be spigotted into the TMF, and coarse cyclone underflow will be piped to skid-mounted secondary cyclone stations on the dam crests where the coarse fraction will be used for dam raise material.

Table 18.12 Summary of Tailing Dams

Dam	Starter Dam					Ultimate Dam				
	Crest Elevation (masl)	Maximum Height[(a)] (m)	Crest Length (m)	Random Fill Volume (Mm^3)	Core Volume (Mm^3)	Crest Elevation (masl)	Maximum Height[(a)] (m)	Ultimate Crest Length (m)	Cyclone Sand Volume (Mm^3)	Core Volume above Starter (Mm^3)
North Dam	930	80	680	3.59	0.95	1068	218	1,900	47.16	3.42
Splitter Dam	935	61	890	3.74	1.08	1068	194	1,930	31.31	3.75
Saddle Dam	935	35	780	2.09	0.75	1068	168	1,600	22.99	3.39
Southeast Dam	930	101	890	12.32	1.72	1068	239	1,400	60.45	3.20
Totals			**3,240**	**21.73**	**4.50**			**6,830**	**161.91**	**13.77**

(a) = maximum height measured at dam crestline.

Figure 18.17 North Tailing Dam

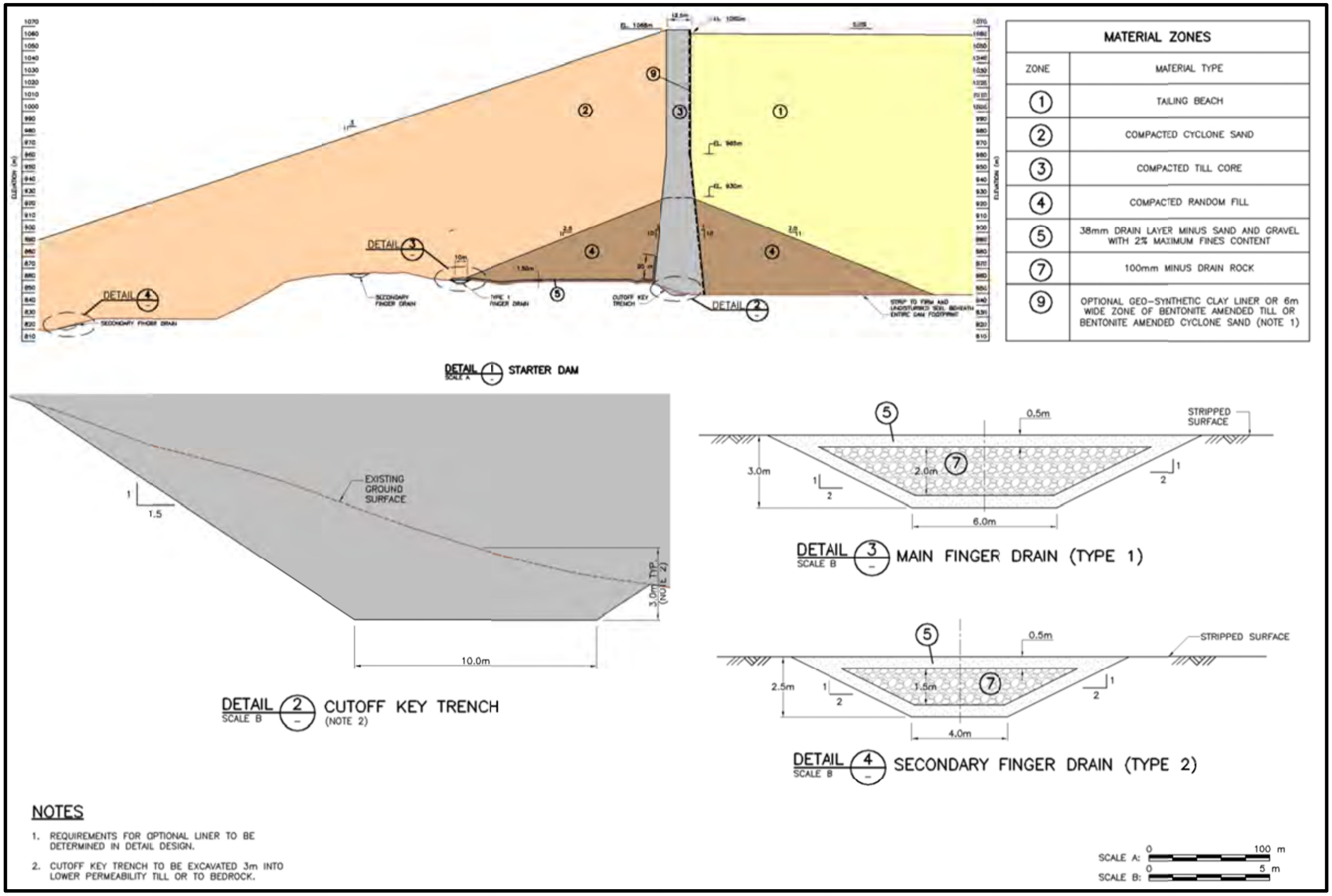


Figure 18.18 Saddle and Splitter Tailing Dams

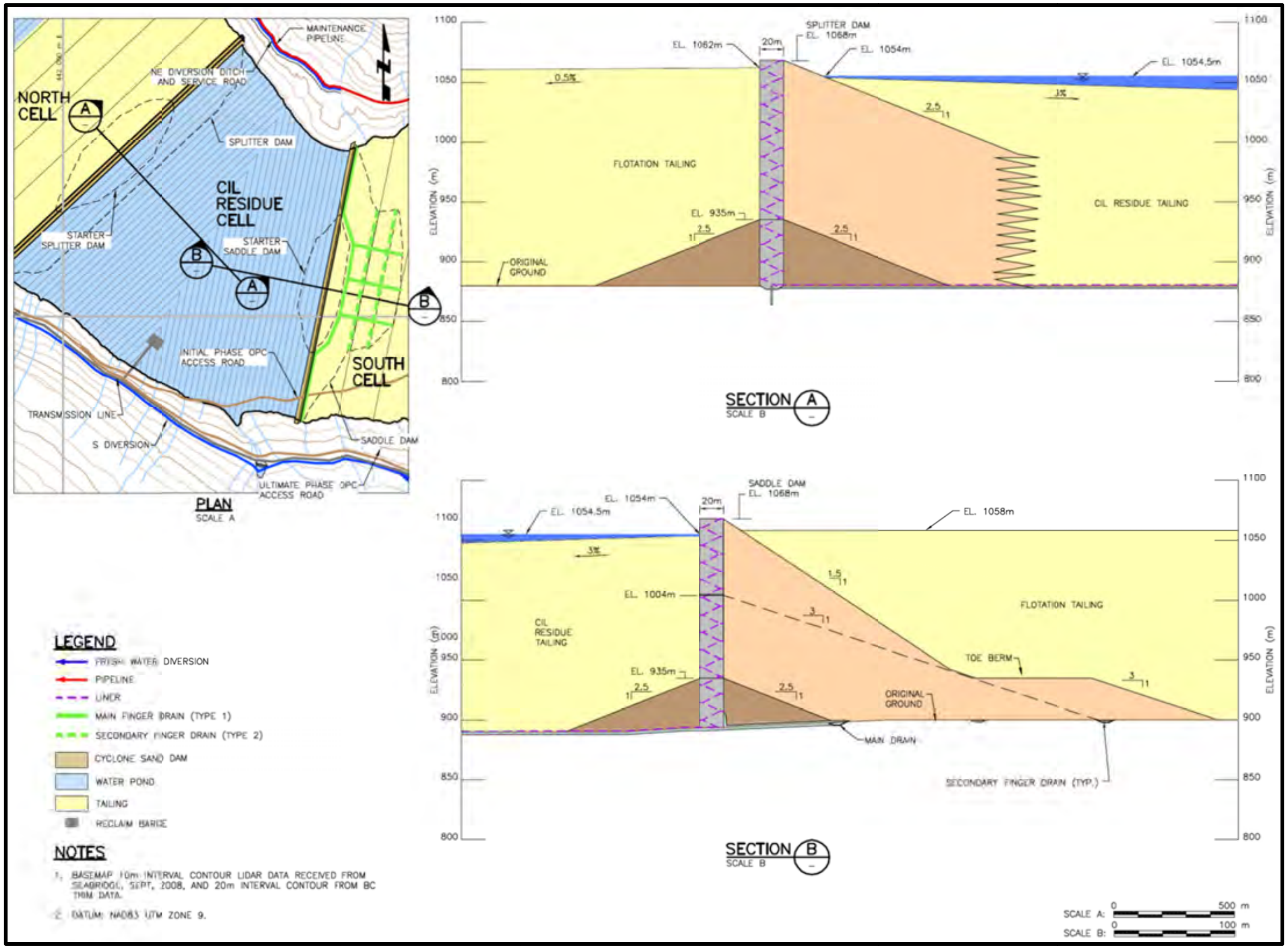
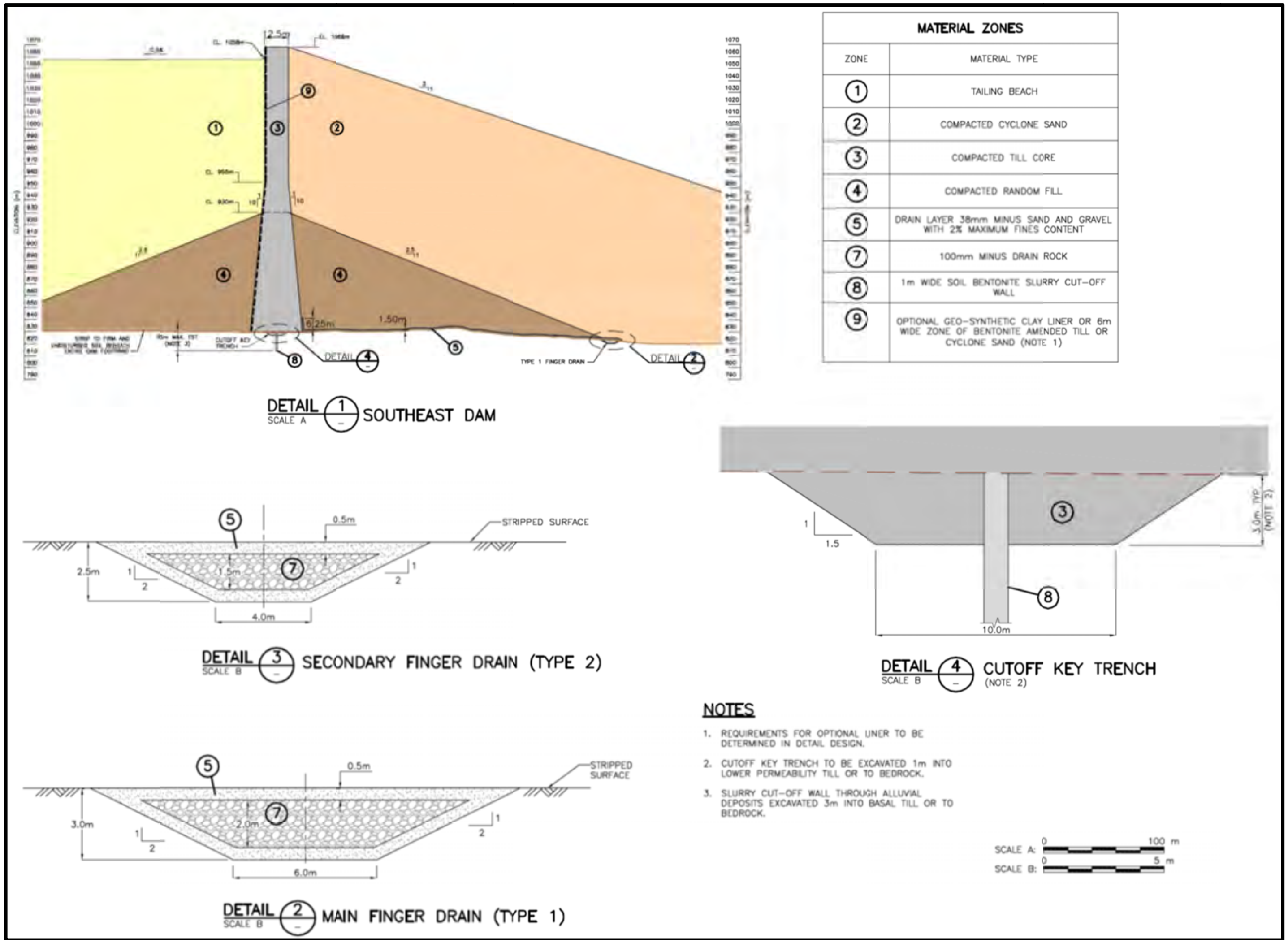


Figure 18.19 Southeast Tailing Dam



MATERIAL ZONES	
ZONE	MATERIAL TYPE
1	TAILING BEACH
2	COMPACTED CYCLONE SAND
3	COMPACTED TILL CORE
4	COMPACTED RANDOM FILL
5	DRAIN LAYER 38mm MINUS SAND AND GRAVEL WITH 2% MAXIMUM FINES CONTENT
7	100mm MINUS DRAIN ROCK
8	1m WIDE SOIL BENTONITE SLURRY CUT-OFF WALL
9	OPTIONAL GEO-SYNTHETIC CLAY LINER OR 6m WIDE ZONE OF BENTONITE AMENDED TILL OR CYCLONE SAND (NOTE 1)

NOTES

1. REQUIREMENTS FOR OPTIONAL LINER TO BE DETERMINED IN DETAIL DESIGN.
2. CUTOFF KEY TRENCH TO BE EXCAVATED 1m INTO LOWER PERMEABILITY TILL OR TO BEDROCK.
3. SLURRY CUT-OFF WALL THROUGH ALLUVIAL DEPOSITS EXCAVATED 3m INTO BASAL TILL OR TO BEDROCK.

TMF Dam Construction

Table 18.12 summarizes material requirements for the dams. For construction of the starter dams, general fill and core material will be excavated by a contractor fleet from local borrow sources (<2 km haul distance) that have been identified at each dam site. Raising of the cyclone sand TMF dams will be conducted by a fleet of dedicated equipment.

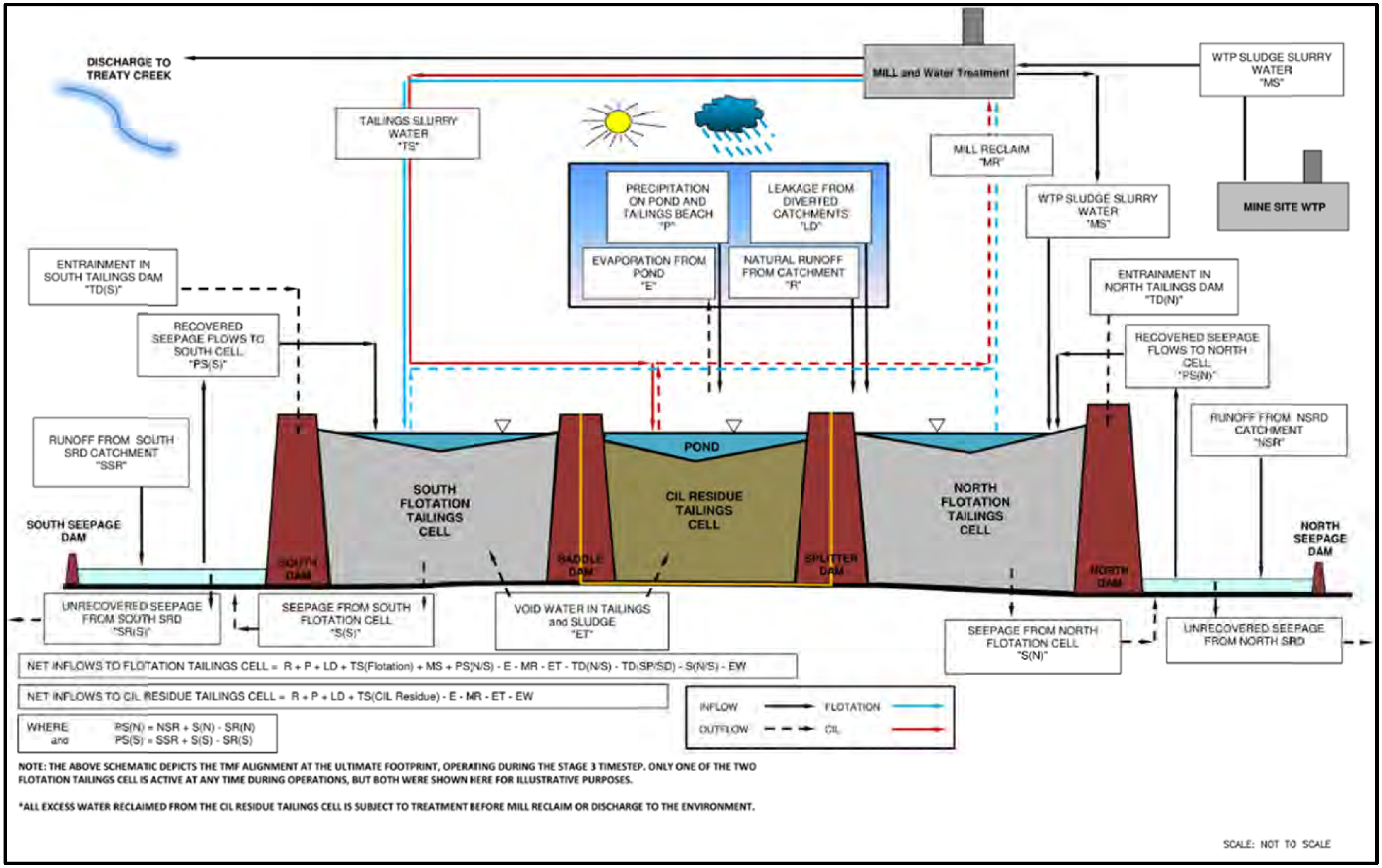
TMF Seepage Recovery Dams

Seepage recovery dams will be constructed of compacted till in a similar manner as for the tailing starter dams, but with flatter 3H:1V upstream and downstream slopes. An inclined till core is provided on the upstream face cutoff trenches into overburden and grout curtains in bedrock where required have been specified to restrict seepage. The dams will also serve to settle solids transported by runoff or produced by dam construction activities. .

TMF Area Water Management

Figure 18.20 shows the schematic water balance with water inputs and outputs from the impoundment.

Figure 18.20 Schematic TMF Water Cycle



TMF Diversion Channels

Two main diversions – the Northeast Diversion and the South Diversion – will be constructed around the TMF North Cell with additional diversions around the plant site to divert non-contact runoff water into a tributary of Teigen Creek at the north end of the TMF.

At start-up, in order to maintain flows into Teigen Creek, the south diversion is extended south to the south end of the TMF valley to capture local flows from the south valley slope.

Once in operation, the South Cell area is diverted by the Southeast Diversion channel, which routes non-contact flows to Treaty Creek around the east side of the South Cell. Diversions in the TMF area are designed to route 200-year peak flows. Diversion channels are shown on Figure 18.21.

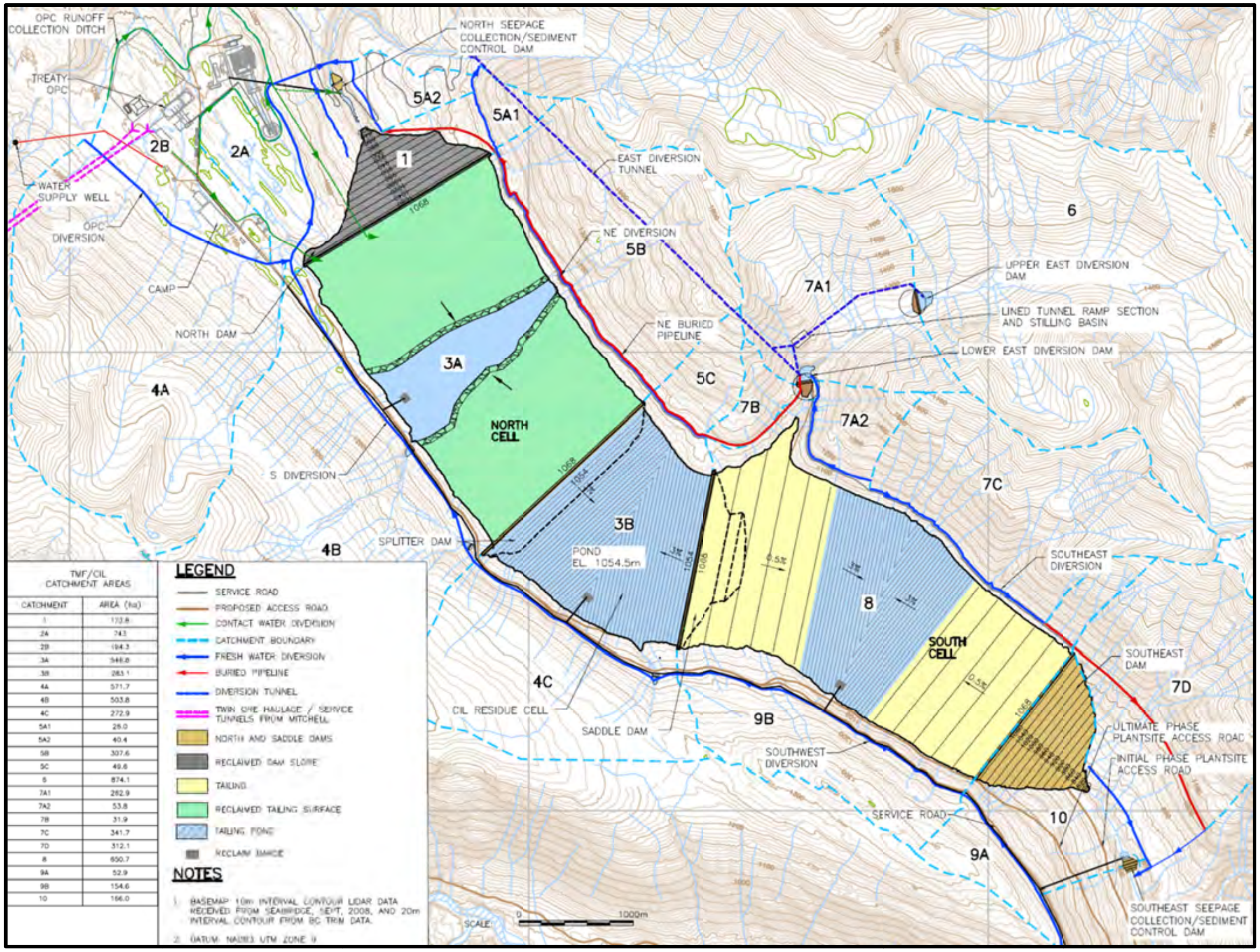
To increase maintenance flows towards Teigen Creek, two diversion dams will be installed in the East Valley catchment. The upper dam diverts flows into a tunnel around a slide zone. The lower dam will divert up to 2 m^3/s into a buried pipeline. The pipeline bypasses the TMF along the east side of the North Cell and releases water into Teigen Creek. Initially, any higher flows from the East Valley will be passed over the lower dam spillway and into Treaty Creek tributary. As the South Cell is developed, flows from the East Catchment above 2 m^3/s will then be routed north through an extended tunnel and into Teigen Creek.

TMF AREA EXTREME FLOOD ROUTING AND STORAGE

The TMF cells are designed to not discharge during extreme flood events. Instead, the TMF cells are designed with sufficient freeboard to store the PMF, resulting from a 30-day Probable Maximum Precipitation (PMP) storm combined with a 100-year 30-day snow melt. It is assumed that all of the perimeter diversions will be inoperative during this extreme flood event. The PMF inflow volume is estimated to range from 65 Mm^3 at facility start-up to 53 Mm^3 at the ultimate dam crest elevation.

Several options are available for dealing with surplus water from floods or annual surpluses. If water quality meets guidelines, surplus water will be discharged to Teigen Creek during spring runoff when the creek flows are highest. Alternatively, water will be reclaimed from the pond and routed back to the Treaty Ore Processing Complex (Treaty OPC) WTP, where it can be treated and discharged on a continuous basis throughout the year. Partial storage of water in the TMF may also be considered in combination with these options.

Figure 18.21 TMF Diversion Channels



CATCHMENT	AREA (ha)
1	172.8
2A	74.3
2B	194.3
3A	546.8
3B	283.1
4A	571.7
4B	503.8
4C	272.9
5A1	28.0
5A2	40.4
5B	307.6
5C	49.6
6	874.1
7A1	262.9
7A2	53.8
7B	31.9
7C	341.7
7D	312.1
8	650.7
9A	52.9
9B	154.6
10	166.0

TMF Closure Plan

Figure 18.22 shows the TMF closure plan. Diversion channels will be decommissioned to re-direct flow back into the tailing facility. The seepage recovery dams will be retained in operation until long term groundwater quality is confirmed. Once pond water quality is achieved, riprap-lined channels will be built to connect the water ponds of each cell. A 6 m-wide approach channel will be excavated in rock along the right abutment of the Southeast Dam, routing floods from the TMF to the stepped spillway, releasing water into North Treaty Creek. Invert elevation of this approach channel is set to maintain separation between the tailing pond and dam and the channel and spillway are designed to safely pass the PMF from the entire upstream catchment.

Flotation tailing beaches exposed above the closure ponds are NPAG. The beaches will be capped by up to 0.5 m of till and re-vegetated. The steeper dam surfaces will be suitably reclaimed to prevent erosion and meet post-closure land use targets. Reclamation will include:

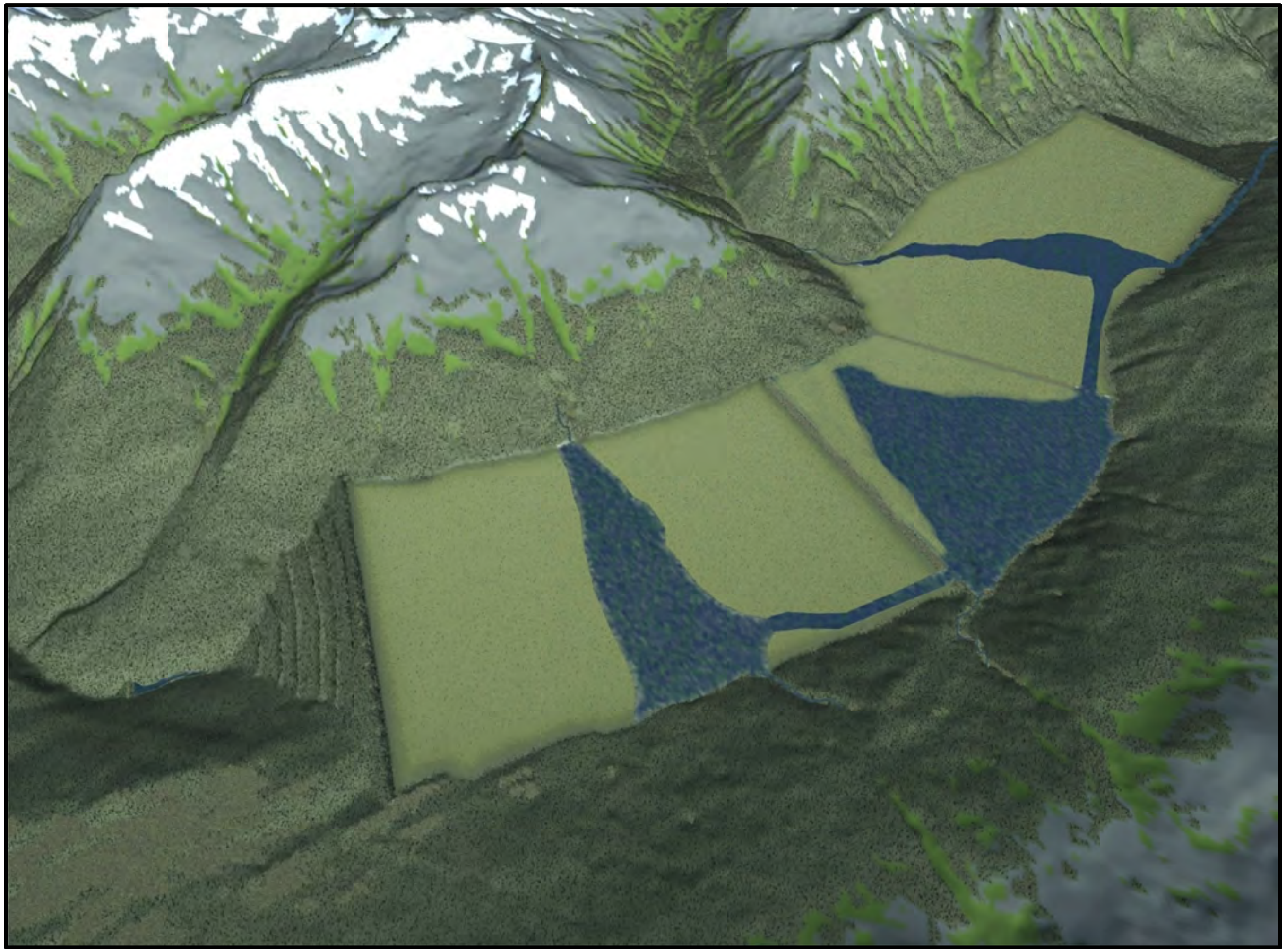
- a vegetated cover composed of 0.5 m of compacted till overlain by 0.5 m of loosely placed till or soil
- a 1.0 m thick graded rockfill cover; the 3H:1V exterior slope of the dam is flatter than the 2.5H:1V maximum slope adopted for reclamation of cyclone sand dams at other mine sites, which is a design feature intended to reduce susceptibility of the dam face to erosion.

At closure, CIL tailing will be dredged into the CIL pond from any exposed beaches and covered by at least 1.0 m of flotation tailing at the base of the pond. Closure water levels will be set to provide 3.0 m or more of water cover, or saturated flotation tailing over the CIL tailing. Closure configuration of the CIL pond will consist of surrounding flotation tailing beaches with gravel and riprap covers to prevent wave or ice erosion, and provision of a closure pond with flotation tailing at its base, over the saturated CIL tailing at depth.

Figure 18.22 shows the TMF closure plan for the current operations scenario of the TMF.

After closure and reclamation is complete, construction of a closure spillway in the North Cell may be considered in order to return drainage patterns to pre-mining conditions.

Figure 18.22 TMF Closure Plan



Note: 3D Image based on rendering provided by Rescan.

18.2 Mitchell and Treaty Plant Site Layout

The overall plant and mine layout is shown in Figure 1.2. Individual facilities will be located to take advantage of the natural topography to minimize the impact on the environment as much as possible. Due to the limited amount of usable terrain in the Mitchell area, most process facilities will be located at the Treaty plant site (referred to as 'Treaty OPC' in this section), approximately 24 km northeast of the Mitchell Zone. Two parallel tunnels, referred to as the MTT, will be constructed to connect the Mitchell and Treaty areas. One tunnel will contain a conveyor, water and diesel fuel pipelines, as well as electrical power transmission cables. The other tunnel will be a transportation tunnel to provide access for maintenance services to the conveyor tunnel. Crushed ore for processing will be conveyed through the conveyor tunnel from the Mitchell OPC to the Treaty OPC. The tunnels will extend from the north side of the Mitchell Zone, approximately 23 km to the northeast into the upper reaches of the Treaty Creek Valley. There is a saddle area in the topography, approximately 17 km from the Mitchell portal, where the two tunnels will be accessed for construction purposes. This access is only for construction and it will be enclosed after construction is complete.

The process plant will consist of three separate facilities:

- ore crushing, conveying, and stockpiling at the mine site (known as the Mitchell OPC), shown in Figure 1.3
- crushed ore transportation system
- process facility at the Treaty OPC, as indicated in Figure 1.6, including secondary and HPGR crushing, grinding, flotation, regrinding, leaching, cyanide recovery/destruction, concentrate dewatering, and ancillary buildings.

At the Mitchell pit (Figure 1.3), the primary crushers will discharge onto conveyors feeding a stockpile. Ore will be reclaimed from under the stockpile by apron feeders and fed on to a tunnel conveyor of approximately 23 km in length, terminating at the Treaty OPC.

Figure 1.4 shows the lower Mitchell site area including additional infrastructure such as the initial staging, construction, and operations camps, explosive facilities, the water storage dam, diversion tunnels, and power plants. Access and appropriate haul roads will be provided to all of these areas.

The WTP and the Water Treatment Plant Energy Recovery (WTPER) facilities will be situated in the lower Mitchell Valley. The WTPER will use the water running downhill from the WSD to the WTP to generate electric power. The WTP will treat:

- potential contact water
- open pit mine drainage from the Mitchell, Kerr, and Sulphurets pits

- tunnel drainage
- surface drainage waters from the WSD and RSFs.

The Upper Sulphurets diversion hydro plant, at the terminus of the MDT, is also located in the Mitchell area. The facilities will make use of the normal stream flows that will be diverted around the Mitchell pit mining operations by the MDT.

The Treaty OPC layout is shown in Figure 1.6, including process facilities, a construction and operations accommodation complex, administration, maintenance and support facilities, and related ancillary buildings.

The Treaty OPC will be slightly terraced and the plant site roads will be constructed from the plant to the MTT portal exit elevation and TMF. All terracing quantities have been based on geotechnical information collected for the plant area.

The TMF will be located in a valley between Treaty Creek and Teigen Creek, approximately 24 km east of the mine site. The TMF catchment will provide a start-up water volume for two months of mill water make-up. It will also provide a minimum water depth for floating of a reclaim barge and achieve adequate water clarification for process purposes.

Two tailing energy recovery buildings will be located at separate locations straddling the north dam tailing line between the process plant and the TMF. Each plant will consist of one slurry pump running in reverse as a turbine, with an induction generator to supply power back into the local plant electrical distribution system.

Major avalanche run-out hazards have not been observed in the Treaty OPC site plateau area. Plant, domestic, and process water supply will be provided from water diversions constructed around the perimeter of the tailing dam, and from wells.

Construction laydown areas, offices, lunchrooms, a concrete batch plant, and material sorting areas have also been designated, and these areas will be cleared and levelled at the same time.

18.3 CRUSHED ORE CONVEYOR SYSTEM

Primary crushers near the Mitchell open pit will crush the ore and convey it to a 30,000 t live covered stockpile near the MTT portal. Reclaim feeders under the stockpile will feed ore onto the tunnel conveyor, which extends underneath the stockpile. The tunnel conveyor is a continuous length and is driven by four booster drive stations plus a drive at the head and tail ends. The conveyor is 72” wide and designed to handle the present process plant requirements; it is also capable of handling an increased plant feed requirement. Crushed ore will be discharged onto a tripper conveyor at the Treaty OPC, where it will be stockpiled on a 60,000 t live capacity covered pile; the Treaty OPC will be fed from this stockpile.

18.4 INFRASTRUCTURE TUNNELS

18.4.1 BACKGROUND

The Mitchell area, including the Mitchell OPC, will be located in the vicinity of the Mitchell mineralized zone. The Treaty OPC and the TMF will be located approximately 24 km east of the open pits. Numerous locations for the process and tailing facilities were evaluated for environmental, social, ease of access, and constructability factors, which led to the location selection discussed in this report. The most practical way to transport mineralized material from the Mitchell site to the Treaty OPC is by primary crushing at the Mitchell area and conveying the crushed ore through a tunnel to the Treaty OPC approximately 24 km away. The locations of the open pits, ore processing facilities, tailing storage, and tunnel conveying system are shown in Figure 1.2.

18.4.2 TUNNEL REQUIREMENTS

The following are the infrastructure tunnels associated with this study:

- a dual conveyor/transportation tunnel (the MTT)
- a conveyor tunnel between the Kerr/Sulphurets side and the Mitchell side (the SMCT), which will be constructed after 20 years of plant operation.

The Project will also require tunnels for the diversion of water and dewatering of the Mitchell pit walls. The following water management tunnels are discussed in Section 18.1:

- twin MTDTs
- twin MDTs
- the WSD construction diversion tunnel
- the Mitchell pit dewatering adit (a tunnel around the north side of the Mitchell pit to drain the north pit wall)
- the TMF area east catchment diversion tunnel.

In order to maintain the pre-production construction schedule and allow start-up of the WSF and WTP, road access to each of these areas, including access to tunnel headings, will be a scheduled construction priority. This will enable heavy tunnel equipment to be transported to each heading to enable tunnelling to start as soon as possible. Access to these headings is indicated in the project schedule provided in Section 18.17.

MITCHELL-TREATY TUNNEL

The conveyor tunnel portion of the MTT will be 4.3 m high x 6.0 m wide; the transportation tunnel will be 4.3 m high x 4.5 m wide, and the two tunnels will be connected by cross-cuts every 300 m. The primary crushed ore from the Mitchell site will be conveyed to the Treaty OPC through the 22.7 km-long tunnel. The tunnel will extend from the north side of the Mitchell Zone to the northeast, into the upper reaches of a tributary of Treaty Creek Valley, near the northeast end of the TMF. There is a saddle area between valleys, located approximately 17 km from the Mitchell portal, where construction of the tunnels will take place. After construction is completed, this area will be enclosed. The MTT will be driven from five headings using drill and blast techniques (DBT):

- the Mitchell OPC portal
- the saddle area portal, from two directions
- the construction adit
- the Treaty OPC portal.

Tunnel construction activities at the various locations will begin as soon as access is achieved to these portals.

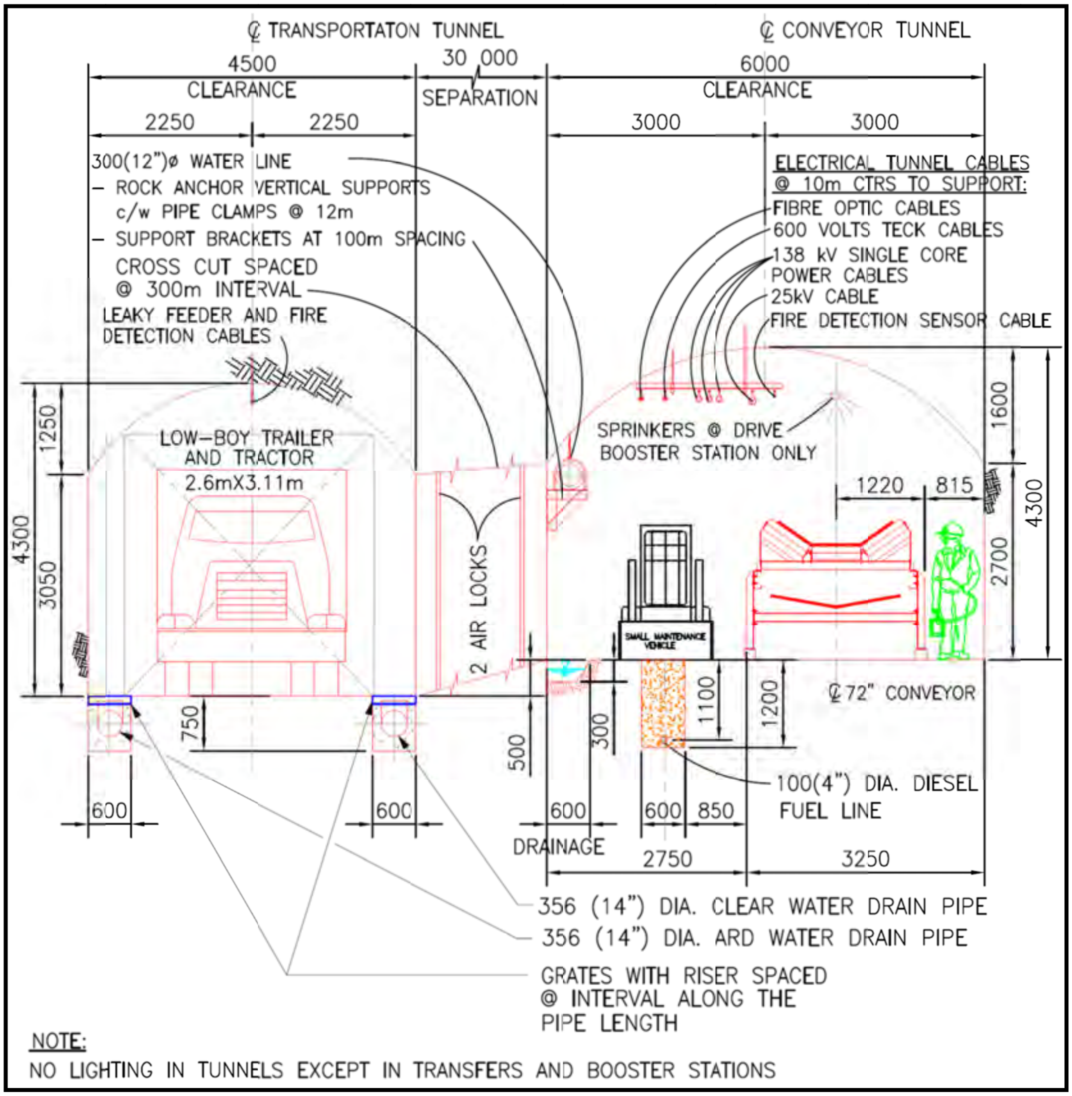
The MTT conveyor tunnel will house the 72”-wide conveyor and conveyor drives, a water line, a diesel fuel pipeline, and electrical power transmission cables. The other tunnel is a transportation tunnel that will provide a route to allow for maintenance services between the main plant sites, delivery of bulk supplies by truck, and movement of personnel to/from the Mitchell Valley mine areas. A gradual tunnel slope up from the Mitchell side towards the Treaty 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 detection will be installed in both tunnels and the conveyor drive stations will have sprinklers; each tunnel will have its own ventilation system to ensure air movement. Firefighting equipment will be installed at various locations throughout the tunnels.

The dual tunnel approach (shown in Figure 18.23) will overcome the significant ventilation issues during the driving of a single larger tunnel. The dual tunnel approach will facilitate the ventilation in bringing air to the construction face as compared to a single large tunnel. Roof support requirements and tunnel stability are also improved with the twin tunnel design because of the narrower tunnel. The construction advance rates and total costs are expected to be similar for the dual versus single tunnel.

In the event of a safety-related incident in one tunnel, the other tunnel can be accessed through the regularly-spaced cross-cuts and used for refuge/escape/ emergency access. During construction, the two tunnels can be set up for one-way looped travel, which will reduce the risk of traffic accidents and eliminate the need for passing bays. During operations, the truck access and ore transport systems will be separated, which will prevent any incident in one system disrupting the other.

During initial construction of the tunnel at the saddle portal, infrastructure requirements, fuel, labour, etc. will be serviced by helicopter support until an access road is constructed to the saddle area from the Treaty Creek Access Road.

Figure 18.23 Mitchell-Treaty Tunnel Cross Section with 72″ Conveyor



SULPHURETS MITCHELL CONVEYOR TUNNEL

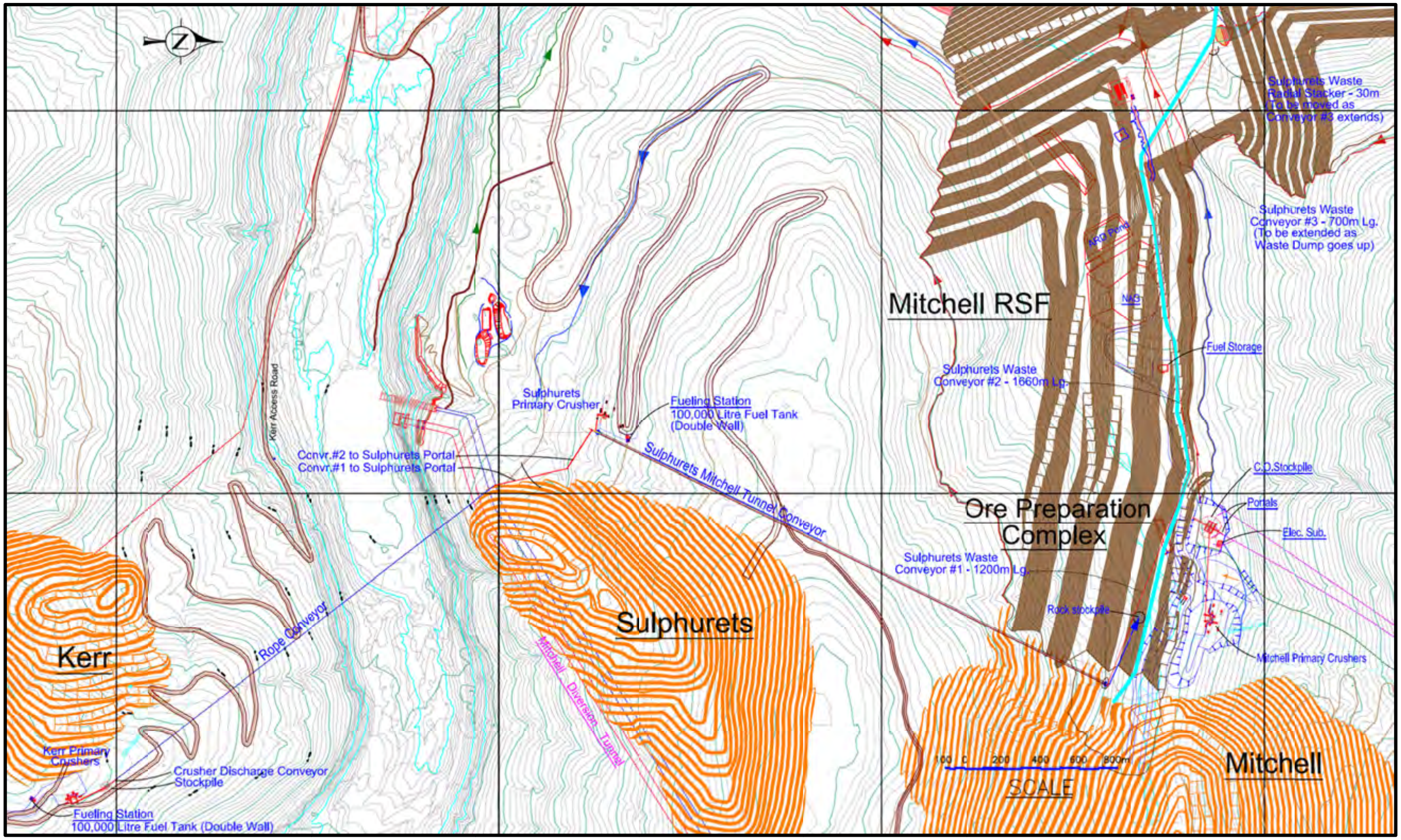
The SMCT will be 6.5 m wide x 5.0 m high and approximately 2.96 km long. Sulphurets ore will be mined and blended with Mitchell mineralization between Years 23 and 30. After Year 27, ore from the Kerr deposit will be processed together with Mitchell ore. A conveyor material handling system and associated services will deliver the Kerr and Sulphurets ores to the Mitchell area for delivery into the crushed ore stockpile. This tunnel will be constructed using DBT so it is ready for operation in Year 23.

Ore from the Kerr and Sulphurets pits will be crushed at their respective pits. The Kerr ore and waste will be mined on a campaign basis and transported by means of a rope conveyor system starting at the Kerr pit (Figure 18.24). At a transfer point, near the Sulphurets crusher, the material will be transported on the SMCT conveyor and stockpiled at the Mitchell side for blending with the Mitchell ore. Similarly, the waste from Mitchell will be transported to the Mitchell side but then conveyed from Mitchell to the McTagg RSF by a system of overland conveyors. Ore from the Sulphurets pit will be trucked to the crusher and then conveyed directly onto the SMCT conveyor to the Mitchell side, where it will be rehandled, and fed through one of the primary crushers and stockpiled.

The SMCT will slope down at 8.45% from the Sulphurets side when travelling towards the Mitchell side. It will primarily be used to house the 72″-wide conveyor; however, electrical power cables will be installed. The electrical cables will be suspended from the tunnel roof. Water seepage into the tunnel will be collected in a ditch along one side of the tunnel and routed to the WSF to allow for treatment, if required. Passenger-type vehicles will travel the tunnel both for inspection and for moving personnel. Refuge bays will be installed at regular intervals to meet code requirements. It is not expected that this tunnel will be used to transport major supplies or materials.

The tunnel will be simultaneously driven from the inlet and outlet portals using the DBT method.

Figure 18.24 SMCT – Plan View



18.5 Site Roads

There will be two primary permanent access roads to the Mitchell and Treaty mine and plant sites. The Coulter Creek Access Road will be primarily a single-lane, radio-controlled road constructed for moving large equipment and supplies to the mine site. An existing road leaves Highway 37, south of Bob Quinn, and extends approximately 59 km southwest to the former Eskay Creek Mine. The first 37 km of this road is classified as public road but is subject to controlled and shared access. The remaining 22 km of existing road length is private and subject to a shared access agreement. Upgrades to sections of the existing road will be required.

The new 35 km-long Coulter Creek Access Road will commence near the former Eskay Creek Mine and follow the west side of the valley south for approximately 21 km before crossing the Unuk River. It then turns east through a series of switchbacks and follows the north side of the Sulphurets Creek Valley to the Mitchell Creek Valley and mine site. Consideration has been given to active and passive snow avalanche control measures through the Sulphurets Creek Valley. The Treaty Creek Access Road will consist of a two-lane road, constructed to provide permanent access from Highway 37 to the Treaty OPC and east portal of the MTT.

The Treaty Creek Access Road will leave Highway 37 approximately 19 km south of Bell II, cross the Bell-Irving River, and follow the north side of the Treaty Creek Valley for approximately 18 km. It will then turn north and follow the west side of the North Treaty Creek/Teigen Creek Valley for approximately 12 km to the Treaty OPC site, TMF, and east portal of the MTT. Initially the North Treaty Lower Road will be built low in the valley to facilitate access for construction of the north tailing dam, and provide reduced road grades and access road length during the first half of mine life.

Additional roads will also be required at mine start-up to facilitate maintenance access and construction of the proposed uphill cut-off drainage ditch. Later, once construction of the south tailing dam starts, the remaining 5.7 km of the North Treaty Upper Road will need to be constructed. These roads will be used to transport supplies, equipment, and crew members to and from the Treaty OPC site, and to transport concentrate to Highway 37 during the LOM.

At about kilometre 18, there will be a 15 km-long access road leading to the MTT saddle, and another 3 km extension from the saddle to reach the MTT construction adit.

A Winter Access Road will be constructed to access the KSM mine site in order to mobilize water treatment supplies and mobile equipment, along with supplies for construction. This access is intended to be used for the first three seasons, until the Coulter Creek Access Road has been completed. The Winter Access Road will cross the Frank Mackie Glacier from the Granduc area and end up at the Mitchell Valley side at Ted Morris Creek.

The site can currently only be accessed by helicopter. Helicopter support will augment the road pioneering work and set up of construction camps.

Avalanche protection will be constructed so that work can be safely carried out at the tunnel portals. Rock storage landforms will be developed adjacent to the Mitchell Zone pit.

18.6 Process Plant Facilities

The Mitchell site process facilities (Mitchell OPC) will include: primary crushing conveying, and stockpiling. The process facilities at the Treaty OPC will include stockpiling, conveying, secondary and tertiary crushing, primary grinding, flotation, concentrate regrinding, concentrate dewatering, cyanide leaching, gold recovery, tailing delivery systems, and a water service system.

The main process equipment will be housed in structural steel buildings, complete with overhead cranes, electrical rooms, heating, ventilation, and air conditioning (HVAC), and offices.

18.7 Ancillary Buildings – Permanent and Construction

Ancillary building construction considered for the study will be pre-engineered structures or stick-built structures, as applicable. The HVAC for these buildings will be designed to industrial standards. The following buildings are included in the study:

- Treaty OPC:
  - fuel storage facility
  - fuel distribution station
  - administration building
  - assay and metallurgical laboratory
  - warehouse and maintenance building
  - concentrate storage building
  - cold storage/reagent storage building
  - first-aid building
  - potable water treatment plant
  - sewage treatment plant
  - incinerator
  - 250-person modular operations camp
  - 800-person construction camp
  - pre-construction fuel storage

- EPCM and Contractors' offices, concrete batch plant, and numerous other construction related facilities
- Mitchell OPC and lower Mitchell areas:
  - truck shop including first aid facilities
  - 350-person modular operations camp
  - potable water treatment plants
  - 140-person initial and 400-person secondary construction camps
  - fuel stations
  - diesel fuel storage and dispensing
  - sewage treatment plants
  - incinerator
- off site:
  - new concentrate storage and loadout at the existing port facility (Stewart, BC)
  - marshalling/staging areas (at Smithers, BC, and off Highway 37 on the Treaty Creek Access Road, in addition to other nearby communities to receive and deliver equipment and supplies to the site during construction and operation of the KSM mine).

18.7.1 Treaty OPC Plant Site

Fuel Storage and Distribution (Permanent and Construction)

The main fuel storage tanks at the Treaty OPC area are sized to store 1,500,000 L, which is enough for 10 days of fuel requirements. All fuel storage areas will either be bunded or will be approved double-wall type tanks. Other fuel stations will be located near the Mitchell OPC, truck shop, at Sulphurets, and at Kerr. Gasoline will also be similarly stored where required.

A fuel dispensing facility will be provided for light vehicles, and a fast-fill facility for mining equipment.

All locations of fuel unloading, loading, and dispensing will be designed to have containment collection facilities and provisions for fuel/water separators.

Administration Building

The pre-engineered administration building will be approximately 1,000 m^2 in plan area.

Offices and open plan work areas will be provided for senior management and administration. There will also be a small lunch room, mud and storage room, meeting rooms, and an electrical/mechanical room.

ASSAY AND METALLURGICAL LABORATORY

The pre-engineered assay laboratory will be located in a separate building near the mill building at the Treaty OPC. It will be equipped to perform daily analysis of mine and process samples. The laboratory will be a 755 m^2 single-storey structure.

FIRST AID BUILDINGS

The first aid buildings will be pre-engineered structures, located at both the Mitchell and Treaty OPC areas, equipped with first aid facilities and emergency vehicles. The Mitchell truck shop will have its own first aid facilities

CONCENTRATE STORAGE

The on-site concentrate storage facility will be a pre-engineered structure, approximately 7,500 m^2 in area. It will have a 5-day storage capacity equating to 820 t/d of concentrate. Concentrate will be loaded into trucks at the Treaty OPC and hauled to a concentrate storage and load out facility at a deep water port facility at Stewart, BC.

COLD STORAGE/REAGENT STORAGE BUILDING

The cold storage/reagent storage building will be located at the Treaty OPC area and will be approximately 20 m wide x 60 m long x 8 m high.

WAREHOUSE AND MAINTENANCE BUILDING

A 22 m wide x 36 m long x 8 m high warehouse and maintenance pre-engineered building will be constructed at the Treaty OPC area. It will be located adjacent to the cold storage facility.

Some warehousing facilities will also be constructed at the Mitchell site.

18.7.2 MITCHELL MINE SITE

TRUCK SHOP

The Mitchell site truck shop will be a pre-engineered building, approximately 190 m long x 50 m wide x 13 m high. This facility 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 in the lower Mitchell area (Figure 1.4), near the 140-person and 400-person construction camps.

The truck shop/mine dry will comprise 8 maintenance bays, 2 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 disposed of at an approved facility.

18.8 Plant Control and Instrumentation

The plant control system will consist of a DCS with PC-based OIS located in the following two separate control rooms:

1. Mitchell pit primary/secondary crusher control room
2. Treaty plant site control room.

Details of the control philosophy are provided in Section 13.0.

18.9 Sewage

The treatment plants (Treaty and Mitchell sites) will use the latest technology treatment process. The solid and liquid material will be separated in the treatment plant with the liquid stream discharging to an approved area such as the tailing pond and the solid material pumped out and trucked away 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.

18.10 Communications System

A fibre optic cable has been included in the 22.7 km-long tunnel to the mine and mill site to provide communications to and from the process plant area. Telecommunications to the plant site has been allowed for 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.

Plant area wired telephone service will be provided by a VoIP system. A local cell phone system is also planned, as is satellite television for the camps.

18.11 Potable Water Supply

Potable water will come from well and 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 facilities. Several potable water sources will be required at the Mitchell and Treaty areas.

18.12 Power Supply and Distribution

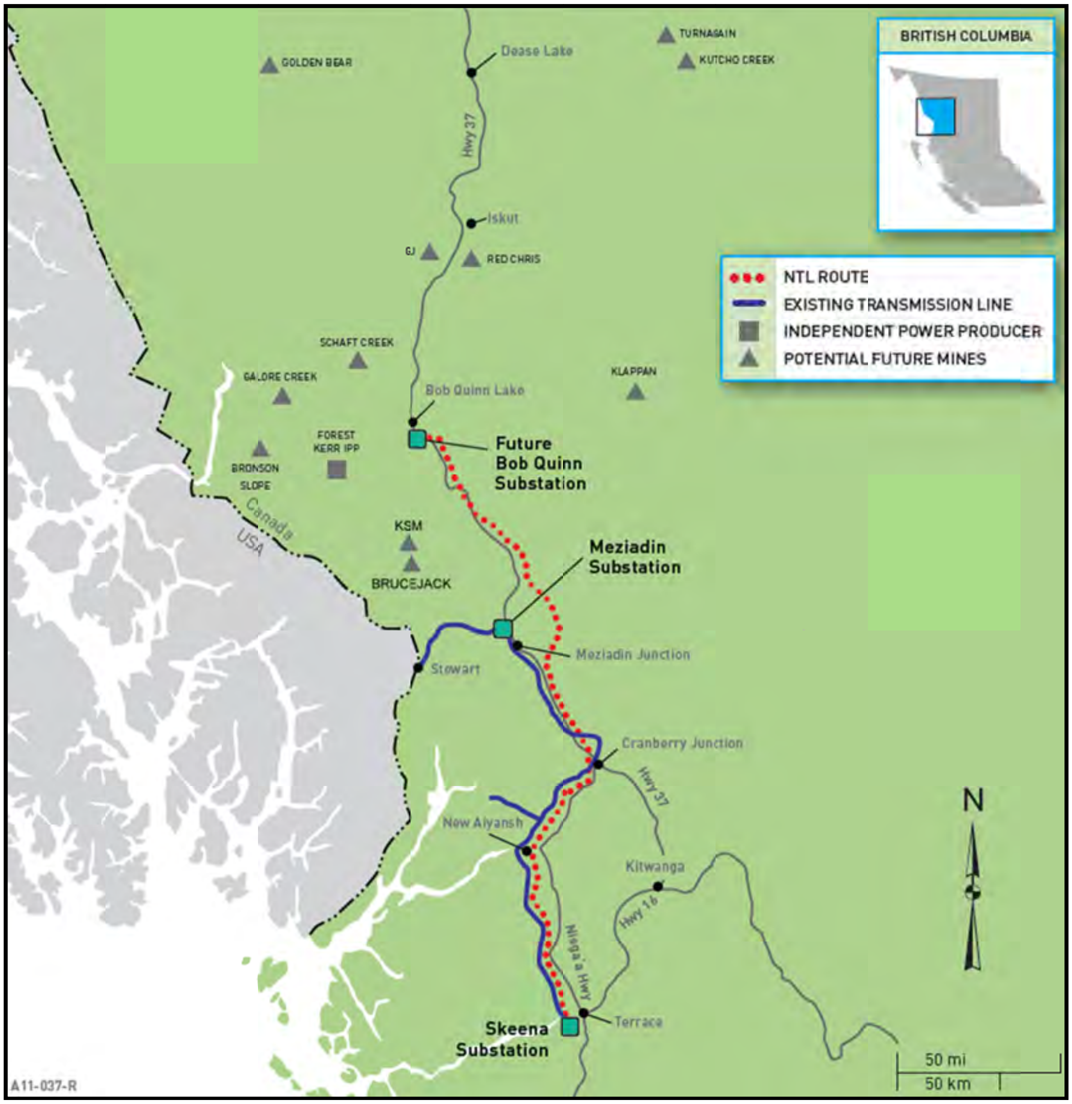
Power generation and transmission utilities in the province of BC are regulated by the British Columbia Utilities Commission (BCUC), acting under the *Utilities Commission Act*. The majority of the power in BC is generated by BC Hydro, although there are an increasing number of private (IPP) generators. The major generating and transmission system in BC is owned and operated by BC Hydro, which is the electric utility that would serve the KSM Project.

18.12.1 Northwest Transmission Line

The northern-most currently operating extension of the existing BC Hydro grid in the vicinity of the project is a 220 km long, 138 kV transmission line to Meziadin Junction from the Skeena substation, near Terrace, BC. This existing transmission line does not have enough capacity to supply an extension to the KSM property. A new, 344 km long, 287 kV transmission line (the NTL) will run north from the Skeena substation to Cranberry Junction, from which point it will run in proximity to Highway 37 to Bob Quinn Lake. The NTL is currently under construction with completion scheduled for spring 2014.

The KSM Project will take electrical service from the new NTL (Figure 18.25) via an approximately 28.5 km long, 287 kV extension from a switching station on the NTL located in the vicinity of the Treaty Creek Access Road junction with Highway 37. This extension will be constructed and operated by the KSM Project, in accordance with the established tariff requirements. Line construction will utilize steel monopoles or similar, such that the line can be generally run in the Treaty Creek Access Road right-of-way, thus eliminating the requirement for a separate, cleared access route.

Figure 18.25 NTL Project Map



Source: BC Hydro.

18.12.2 KSM Interconnection Point (Point of Power Delivery)

The KSM Project will receive power from BC Hydro at 287 kV from the proposed Treaty Creek switching station. This station will be located adjacent to Highway 37, approximately 20 km south of Bell II, near the proposed Treaty Creek Access Road terminus. This will be the utility point of connection (and the power metering point).

18.12.3 Extension From The NTL to KSM And Site Distribution

The NTL will be extended to the KSM Project via a 287 kV spur line from the proposed Treaty Creek switching station. This 28.5 km-long line, to be constructed and operated by KSM in accordance with the BC Hydro tariffs, will generally follow the mine access road to the KSM Substation No. 1 at the Treaty process plant near the TMF. Power to the facilities in this area will be distributed at 25 kV from the substation.

The 287 kV voltage for the proposed transmission line extension to the KSM Project is based on the 287 kV transmission voltage selected for the NTL. A review of the technical specifications of the NTL confirmed that a voltage lower than 230 kV would not be feasible (since 230 kV is not used in this part of the grid), and the installation of a step-down substation at Highway 37 would be uneconomic.

A switching station will be installed where the spur line taps off the NTL to the mine, along Highway 37 near Treaty Creek, close to where the mine access road will intersect with Highway 37. This station will be known as the Treaty Creek Switching Station. This switching station will form a part of the BC Hydro system and will be constructed and owned by BC Hydro. The KSM Project capital cost estimate includes a contribution to BC Hydro to cover the KSM share of the cost of this installation. The estimate also includes a separate amount to cover what BC Hydro terms “the basic line extension”, which is the KSM supply breaker bay.

The 287 kV transmission line from the BC Hydro Treaty Creek Switching Station will cross Highway 37 and the Bell-Irving River then closely follow the mine access road along the north side of Treaty Creek for approximately 12 km to a deviation point where the line transitions from following the access road to following the water diversion ditch up to the No. 1 Substation at the Treaty plant site. To facilitate construction along the road right-of-way and beside the ditch, the transmission line will be constructed of steel monopoles; in general, guy wires will not be used. To protect against avalanche damage, some structures will be mounted on tall concrete piers of about 65 m^3 of concrete each, to raise the pole bases above the avalanche flow.

KSM Substation No. 1 will be a gas insulated (GIS) facility installed in a building that will be integrated into the process plant site to avoid undue alienation of wetlands. It will provide service to the process plant as well as to the remote Mitchell mine and crusher site via a 24-km long 138 kV cable that will run through the conveyor tunnel connecting the two plant sites. This cable will terminate at the 138-to-25 kV step-down Substation No. 2 at the Mitchell plant area. This substation will also be of the GIS type and will be housed in a concrete structure built into the hillside to protect against avalanches. There will be 25 kV cables feeding the primary crushing, conveying, and stockpile areas. There will be 25 kV overhead power lines extending from the substation to the Mitchell pit, service complexes, water treatment plant, truck shop, camp, hydro plants energy recovery plants, etc.

18.12.4 SYSTEM STUDIES

Seabridge first commissioned a BC Hydro System Impact Study in 2009, and subsequently commissioned a Facilities Study, which is the next step in the process. The System Impact Study was completed and subsequently updated in November

2011. Due to changes in the plant site with the addition of the Mitchell to Treaty conveyor and the new Treaty Creek Access Road, the BC Hydro Facilities Study is in the process of being finalized based on the latest data. Seabridge was the first mining company to commission these studies and thus, in accordance with long-standing BC Hydro rules, the KSM Project load will have priority over other customers taking power from the NTL.

The updated Facilities Study, once complete, will lead to a Facilities Agreement that will define all capital costs and special conditions for service; an Electricity Supply Agreement will follow. Both agreements will have set terms that can only varied by the results of the Facilities Study, which are standard agreements for "transmission customers" in BC and are not subject to further negotiation or revision.

LOAD FLOW STUDIES

System load flow studies have been performed by KSM project consultants using system analysis software to confirm process plant and mine power system voltage control from no load to full load. System voltage stabilization is based on switched reactors to control light load over voltages due to 287 kV transmission line and 138 kV cable capacitance and also assumes automatic control of the process plant synchronous ball mill drive motor excitation systems for instantaneous voltage control. KSM Substation No. 1 also includes a ±20 MVA static Var compensator as identified by the BC Hydro System Impact Study as being required to ensure system transient stability.

SYSTEM IMPACT AND FACILITIES STUDIES

The technical viability of the KSM interconnection from the Skeena Substation was confirmed by the System Impact Study, which was commissioned by Seabridge in 2009 and completed by BC Hydro in 2010. BC Hydro will also complete a Facilities Study in 2012 based on the updated project details.

The 2011 updated System Impact Study confirmed the previous study and concluded that the utility regional transmission system requirements are compatible with the proposed interconnection of the KSM 150 MVA mine load into their north Skeena area transmission system. The BC Hydro study states:

This study has confirmed that the KSM load can be interconnected to the BCH transmission system with the following facilities:

- *KSM Built Facilities:*
  - *A 287kV Floatation Substation 1(FLT1) at the customer's Mine site 13 km east of SBT;*
  - *A 287kV Floatation Substation 2 (FLT2) inside the customer's site with a +/-20 MVAr STATCOM and connected by cables to FLT1;*

  - *A 287kV transmission line from the new Snowbank Terminal Substation (SBT) to FLT1;*
  - *Circuit breakers with Point-on-wave switching for transformer energization at FLT1 and FLT2 to control voltage sag at the POI;*
  - *Fibre optic cable from SBT to FLT1 and FLT2 with communication channel for transfer tripping and Remedial Action Schemes (RAS)*
- *Method of Connection:*
  - *Construct new 287 kV substation at Snowbank Creek and loop the NTL (2L102) in to the new substation (now changed to Treaty Creek).*
  - *Terminate the customer owned 287 kV line at the new substation*
- *System Reinforcement 287 kV Upgrades:*
  - *Relocate the 35 MVAr 287kV switchable NTL line reactor from Bob Quinn (BQN) to SBT;*
  - *Make Forrest Kerr IPP (FKR) generation selectable for shedding within 11 cycles if either line SBT-FLT1 or line FLT1-FLT2 are tripped by a protection system;*
  - *Make 42 MW Synchronous Motor at FLT2 selectable for shedding within 15 cycles for tripping of any 287 kV line north of SBT;*
  - *A direct transfer trip to FKR and all load customers to achieve rapid separations and avoid temporary overvoltages: Trip SBT-FLT1 287 kV line breakers as well as FKR generation and load circuits at BQN for multi-phase faults or unsuccessful single pole reclosing on SKA-SBT line;*
  - *Protection to trip FKR and all BQN load customers but not KSM load for multiphase faults or unsuccessful single-pole operation on the BQN-SBT circuit.*

It is to be noted that the cost of any required system upgrades will be to the account of BC Hydro under the tariffs, but bonding may be required for 7 years to guarantee the operation of the mine. Capital contributions towards the NTL will be required. These will be formalized under a new BC Hydro tariff, which has not been issued as of the date of this report.

The Facilities Study, when completed in mid-2012, will provide definitive cost estimates and detailed requirements for:

- system reinforcement
- the Basic Transmission Extension (BTE)
- the transmission lines and customer facilities required to be installed by each party to facilitate the supply of electricity to the KSM Project
- the contribution required towards NTL construction (this PFS includes an estimate of these costs based on the latest advice from BC Hydro.)

The signed Facilities Agreement that follows the completion of the Facilities Study will become part of the Electricity Supply Agreement, which will form the contract for power supply between BC Hydro and the Owner. This is a standard agreement, approved by BCUC, the general conditions of which are not subject to legal review or negotiation by the customer.

SHORT CIRCUIT STUDIES

The KSM Project consultants have carried out a preliminary short-circuit study for the KSM plants, based on the proposed line extension from Skeena, as required for prefeasibility design and cost estimates.

The design short circuit levels were determined to be:

- Plant Substation No. 1 (25 kV Bus Bars)
  - momentary: apply ANSI factors (Appendix A of short circuit report)
  - interrupting: 23 kA minimum.
- Mitchell Substation No. 2 (25 kV Bus Bars)
  - momentary: apply ANSI factors (Appendix A of short circuit report)
  - interrupting: 18 kA minimum

Service from the Skeena Substation to the KSM Project via the NTL will be delivered over a single-circuit line. BC Hydro service studies indicate very high reliability for single-circuit high voltage transmission lines, with few outage-hours in a year. Occasional service interruptions and planned maintenance outages can be expected and are considered normal for mining projects.

18.12.5 ELECTRIC UTILITY REQUIREMENTS AND TARIFFS

The electric service to the KSM Project (including all terms and conditions such as rates and metering requirements, connection charges, and many aspects of the KSM connecting transmission line) would be in accordance with the latest edition of BC Hydro Electric Tariff, in particular Rate Schedule 1823 “Transmission Service – Stepped Rates”, including:

- BC Hydro Electric Tariff Supplement 5, Agreement For Customers Taking Electricity under 1821 (now 1823 it is noted)
- BC Hydro Electric Tariff Supplement 6, Agreement For New transmission Service Customers.

BC Hydro is planning a new tariff for customers that draw service from the NTL. This new tariff is not expected to change the cost of power itself; however, it will address how the capital cost of the NTL will be split between new mining customers taking service from this line.

Although not currently expressly covered in the tariffs, it is possible that power for the Project could be purchased from an IPP and wheeled over the BC Hydro transmission system to the mine, although there are no current wheeling tariffs to cover this scenario. However, the cost per kilowatt hour for electric power supplied by an IPP would not be competitive with electric power purchased under BC Hydro Rate Schedule 1823, since electricity purchases from an IPP would be for costly “new clean” energy from recently constructed hydro or wind projects. In addition, transmission wheeling charges would apply.

As per BC Hydro Schedule 1823 "Transmission Service – Stepped Rates" issued April 1, 2012, for bulk electricity customers drawing service at transmission voltage, the published rates are:

- Industrial Transmission Service:
  - mines, chemical plants, large sawmills, pulp and paper mills, large manufacturing; Schedule 1823 of the Electric Tariff
- Demand Charge:
  - Cdn$6.263/kVA of billing demand
- Energy Charge:
  - Cdn$0.03261/kWh applied to all kilowatt hours up to and including 90% of the Customer's Baseline Load (CBL) in each billing year
  - Cdn$0.07360/kWh applied to all kilowatt hours above 90% of the CBL in each billing year
- Minimum Charge:
  - Cdn$6.263 /kVA of billing demand per billing period
- Rate Rider (as per Schedule 1901):
  - 5% rate rider applied to all charges before taxes and levies.

As shown above, BC Hydro Schedule 1823 is a two-tier schedule, nominally with 90% of the CBL being charged at economical Tier 1 power rates and the last 10% plus all power above the CBL being at costly Tier 2 rates. This system is designed to encourage energy conservation, as consumption reductions due to energy conservation measures are applied against costly Tier 2 power. The rate rider is a surcharge applied to the final bill.

The actual power cost per kilowatt hour for KSM, in accordance with the rate schedule calculation, depends on:

- the plant load factor being the ratio of the average load to the maximum demand
- what percentage of the load is economical Tier 1 power versus costly Tier 2 power
- to a lesser extent, the plant power factor and load factor.

Under their Power Smart program, BC Hydro has procedures for certification of energy conservation measures designed into new plants in order to credit these measures against costly Tier 2 power. The cost of electric power used for the purposes of this study assumes that these measures will be fully implemented and that Tier 2 power will essentially be eliminated. These energy conservation design features include using HPGR in lieu of SAG milling and other energy conservation design features as may be certified by BC Hydro.

Seabridge's consultants completed a study in early 2012 that estimated the cost to supply power to the Project. The overall cost of electricity was estimated to be Cdn$0.049/kWh at the mine 25 kV bus bars. This cost estimate is based on the April 2012 rate schedule with BC Hydro "PowerSmart" energy savings due to the use of efficient HPGR grinding, as confirmed by a separate study to be submitted to BC Hydro for approval. The estimated power cost was applied to all KSM Project operating cost calculations. The power cost estimate also includes the upcoming re-imposition of the 7% BC Provincial Sales Tax (PST). This will result in 7% PST being charged on the entire power bill, which will not be a deductible input tax credit as is the case with the Goods and Services Tax (GST) and Harmonized Sales Tax (HST). Consequently, power costs will rise an additional 7% for industrial and commercial customers over the current level. This factor is included in the above-listed cost of power.

As indicated in the April 2012 BC Hydro Tariff Supplements Nos. 5 and 6, customers that have a maximum contract power demand greater than 150 MVA must provide a large, non-refundable capital contribution towards the utility system transmission and generation reinforcement. This contribution would apply to the entire load, not just the load that exceeds 150 MVA. The load for the KSM Project is currently shown to be above 150 MVA, even assuming that energy conservation measures are implemented. Previous informal discussions with the BC Hydro indicated that the 150 MVA limit may be eliminated or at least relaxed in the future. However, an announcement on this subject has not been forthcoming since it was first discussed two years ago and it appears quite probable that the requirement will not be eliminated.

To eliminate very substantial capital costs associated with generation reinforcement, the project budget includes a combustion (gas) turbine to be installed in or near Terrace to feed peaking power into the BC Hydro system and thus eliminate the peaks in demand above 150 MVA. The turbine would utilize natural gas from the existing PNG gas line that runs through Terrace. The capital cost of the installation has been included in the project budget and the cost of power for the Project has been adjusted to account for fuel and operation and maintenance (O&M). The turbine could run unattended under automatic control, with generation being dispatched from the mine. Rather than the turbine being operated by the KSM mine, this facility could be contracted to a third party.

18.12.6 KSM Transmission Line Implementation And Costs

The branch line from Treaty Creek to the mine (by KSM) includes the construction of approximately 28.5 km of 287 kV transmission line following the mine access road to KSM Substation No. 1 at the process plant. The final connection to the KSM mine and mill Substation No. 2 would be accomplished with an approximately 24 km-long section of solid dielectric 138 kV cable through the approximately 22.83 km long conveyor tunnel (MTT).

The transmission line prefeasibility cost estimates for the 28.5 km spur line are based on the current cost of monopole steel line construction for 230 to 345 kV transmission lines, with costs added for extensive concrete pier foundations to provide protection from avalanches.

18.12.7 Transmission Line Construction Schedule

The 287 kV line extension from Treaty Creek to the Treaty plant site would commence in the second year of the construction schedule, after the access road is complete. Construction of the line could easily be completed in one summer and fall construction period.

The environmental assessment for the 28.5 km section of transmission line from Treaty Creek to Substation No. 1 is included in the KSM Project environmental assessment report.

18.12.8 Treaty Creek Switching Station

The project costs include a 287 kV switching station, as required by BC Hydro, to be located on the 287 kV transmission line near Highway 37 near Treaty Creek, the point at which the mine access road joints the highway. This facility will include three 287 kV circuit breakers, six air switches, and would have line protection relaying, communication equipment, and the KSM utility metering equipment. This facility would be owned and operated by BC Hydro. The project budget includes a capital cost contribution to BC Hydro to cover construction costs. The mine power point of delivery and metering would be at this station.

18.12.9 KSM Substation No. 1

The KSM 287 kV step-down Substation No. 1 at the Treaty process plant will be constructed and owned by Seabridge, in accordance with BC Hydro policy, which is also the most economical solution. This installation is shown on Tetra Tech's single line drawings in Appendix C; the substation location is shown in Figure 1.6.

The substation equipment has been sized based on the latest project load list. Redundant transformer capacity was included in the design. To supply the KSM process plant load (three 287 – 138 – 25 kV), three winding 75/100/125 MVA step-down power transformers were selected for installation at Substation No. 1 at the Treaty process plant site. The three transformers provide redundancy that allows one transformer to be out of service. The 138 kV tertiary windings are connected to the 24 km long tunnel cables feeding the Mitchell (open pit mine) area Substation No. 2.

The substation is a GIS design utilizing 138 and 287 kV gas insulated circuit breakers and bus bars allowing a compact design all contained in a building adjacent to the process plant. Connections to transformers are by high voltage solid dielectric cables.

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 will be 25 kV. One switched reactor is included in Substation No. 1 for compensation of the incoming 287 kV line to limit Ferranti effect over-voltages. In addition, two switched reactors are also included in Substation No. 1 at the end of the 24 km long 138 kV cable to compensate for cable capacitance and help thus control bus voltage. Preliminary studies have indicated that the switched reactors, plus the two switched reactors at Substation No. 2 and 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, although automatic transformer tap changers have been included. The System Impact Study identified the requirement for a ±20 MVAR static Var compensator (SVC) for dynamic reasons. The cost of this unit has been included in the budget and will also serve to further stabilize steady state voltages. It is possible that the SVC will be located in the Treaty Creek switching station.

Substation No. 1 will also include a line-up of 25 kV metalclad or GIS switchgear, as required for power distribution around the plant site.

The substation does not include harmonic filters. If these are required by harmonic generating plant loads, they would be best located at the process plant near the harmonic sources and would be included in the process plant budget.

18.12.10 138 kV Cable

The two substations will be interconnected by three 138 kV, single-core, 300 mm^2, XLPE solid dielectric power cables suspended from the back (roof) in the pipeline tunnel that will run between the two plant sites. The 300 mm^2 (600 kCM) conductor size quoted is the minimum physical size that vendors typically manufacture at 138 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. In order to limit induced sheath currents, the cable sheaths will be “cross bonded”, which is the normal design for high voltage, high current, single-core cable installations.

The approximately 24.5 km-long cables will be installed in the 23.83 km conveyor tunnel, supported by galvanized steel messenger cables suspended by rock bolts just below the tunnel centre arch. The extra power cable length will be required to allow for the cable duct installation between the substations and the tunnel portals. The cables will have short circuit protection by high speed 138 kV circuit breakers at either end, with differential protection coordinated by a fibre optic link.

The Mitchell access tunnels (the MTT) will have redundant, single mode, fibre optic links that will be used by the cable protective relay system and will serve as the primary project communication link between the two plant sites.

18.12.11 Mitchell Substation No. 2

The 138 to 25 kV Mitchell Substation No. 2 is critical infrastructure for the KSM Project. As an alternative to a standard 138 kV air-insulated outdoor substation, the Mitchell Substation No.2 is planned as a GIS. This is a very compact design, requiring only a fraction of the space of a conventional air insulated high voltage substation. The compact design allows for the total installation to be included in a reinforced concrete building that provides a high degree of protection against geohazards such as avalanches. It also eliminates hazardous high voltage overhead lines in the vicinity of the Mitchell plant site and requires much less plant area. A separate detailed study was completed covering the GIS option. A geohazards overview report is included in Appendix F.

The substation includes:

- two 138 to 25 kV, 25/33 MVA power transformers (two units provided for redundant capacity)
- five 138 kV GIS circuit breakers
- two switched 138 kV reactors to compensate 138 kV cable capacitance
- a line-up of 25 kV metalclad or GIS switchgear for site power distribution.

The Mitchell Substation is shown on the single line drawings included in Appendix C.

18.12.12 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 (also serving as pit switch-houses), and trailing cables for the 7,200 V pit mobile electric shovels and drills, are included the electrical distribution project budget.

18.12.13 Construction and Standby Power

Modular diesel generator sets will be provided to supply construction power for tunnel driving, camps, plant construction sites, and other initial construction-related facilities. A total of eight modular high-speed diesel power plants have been included to supply power for tunnel operations and plant site construction plus numerous smaller installations. The capital and operating costs of these facilities plus local distribution have been included in project indirect costs (not in the power supply budget). The power distribution costs within the various tunnels are included in tunnelling costs.

The construction generating stations are modular, complete with switchgear, and designed for programmable logic controller (PLC) automatic unattended operation. Environmentally approved double-walled fuel storage tanks and associated piping are included for each power station. However, bulk long term fuel storage for power generation during the construction phase at the Mitchell facilities was included elsewhere in the project budget. Additionally, the electrical construction power supply estimates include fuel storage and fuel consumption only for tunnel and process plant construction electrical power generation, but not fuel for tunnel or process plant diesel powered construction equipment. These estimates have been included elsewhere in the project budget.

The utility power supply, in lieu of diesel generation, will be scheduled to be available at as early a date as possible to eliminate the need for local diesel generation at the Treaty process plant and saddle areas, but not at the mine site where construction power will be required until after completion of the 22.83 km-long access tunnel (MTT).

Some of the construction gensets will be retained and reconfigured to serve as future standby/emergency generation for the mine, process plant, and accommodation centres. The cost to refurbish construction gensets and reconnect this equipment for standby service in the permanent plant has been included in the process plant electrical budget.

The estimates include the purchase rather than rental of construction gensets. The relatively long KSM construction period will make construction genset rental uneconomic.

18.12.14 ENERGY CONSERVATION

Under BC Hydro Schedule 1823, the power purchase price is based on a two-tier system with the last (nominally 10%) Tier 2 energy being much more costly. The actual amount of energy consumption that falls within the second high-cost tier depends on the CBL. Typically, this is set for a new project after a year of operation has established the load characteristics. Project design factors that favour energy conservation will be considered by BC Hydro in their determination of the CBL before a new project is actually built. This will have a significant positive economic impact by reducing the average cost of power. Detailed studies have been initiated for submission to BC Hydro to address these issues and thus realize significant power purchase price reductions based on the reduction or elimination of costly Tier 2 energy.

18.12.15 Energy Recovery and Self Generation

The KSM Project presents several opportunities for energy recovery from process plant flows, as well as power generation from mini hydro projects taking advantage of water flows that must otherwise be diverted around the mining operations. The total projected average annual energy production from these schemes is 48,706 MWh/a.

As these energy recovery and mini hydro schemes, to a large extent, make use of facilities otherwise required for the mining project, they are generally economically attractive and will also reduce the total energy consumption of the project. The sale price of power sold back to the utility, with the mine acting as an IPP generator, is the price for "new" power. This is valued considerably higher than the KSM Project purchased power, the price of which is lower since it is largely "heritage" power generated by older BC Hydro facilities that are already amortized. Likewise, any generation that serves as "load displacement" would be displacing high cost (under Rate Schedule 1823) Tier 2 power. In the economic evaluation, a sale price of Cdn$0.0736/kWh, which is the current BC Hydro Tier 2 energy price, has been used for both energy recovery generation and for valuing the output of the mini hydro plants.

All of the listed energy recovery plants will be located within the KSM mining lease. The energy recovery plants recover energy from process plant flows and do not require water licences. All environmental matters are covered by the process plant environmental review. The Mitchell diversion scheme utilizes diverted water flows to generate power. The environmental assessment for these plants is covered in the overall KSM mine environmental assessment.

All of the generating plants, similar to small IPP hydroelectric plants, will operate unattended and will be automatically controlled by PLC systems. The locally generated power will be fed into the 25 kV mine distribution power lines. Each facility will have revenue class metering equipment as required by BC Hydro in order to determine the output of the plants for payment purposes. BC Hydro regulations for power purchase programs such as their Standing Offer Program allow generators to be "behind" customer loads, which applies to the KSM Project.

The generation projects included in this study are summarized in Table 18.13.

Table 18.13 Power Generation

Project Name	Type	Installed Capacity (kW)	Net Annual Generation (MWh)	Notes
WTP	Energy Recovery	2,300	14,067	1 Turgo impulse or 1 Francis reaction turbine, variable head
Plant Tailing	Energy Recovery	885	7,132	Total of 2 plants and 2 pumps as turbines on one line
Mitchell Diversion	Mini Hydro	7,500	27,507	1 Turgo impulse turbine
McTagg Diversion	Mini Hydro			Deferred
Total		**10,685**	**48,706**	**Mean Annual Output**

18.12.16 Power Sales

If the KSM mine and process plant project energy conservation measures do not eliminate all Schedule 1823 costly Tier 2 energy, the small energy recovery projects will be used to displace the remaining Tier 2 mine and process plant energy requirements.

BC Hydro has a simplified Standing Offer Program to encourage the development of small clean energy projects throughout BC. The program is a process to purchase energy from small projects with a capacity greater than 0.05 MW (50 kW) but not more than 15 MW. The process is much more practical for small projects than a BC Hydro competitive “Call For Power”. The Standing Offer Program has been updated in 2011.

The payment price offered by BC Hydro for energy delivered under the Standing Offer Program is determined by the location of the project and the year, month, and time of day the energy is delivered. However, the actual price used for the KSM PFS for power sales is the somewhat lower number of Cdn$0.0736/kWh, due to the fact that energy production is currently considered as subtracting from mine Tier 2 power demand, in order to ensure the maximum demand is less than 150 MW (which is the trigger point under the BC Hydro tariffs for large capital contributions towards system transmission and generation costs).

In addition to the Standing Offer Program and energy displacement previously discussed, there are other options for power sales, including participating in a BC Hydro ”Clean Power Call” for IPPs. There are also funding programs under “Load Displacement” or “Demand Side Management” (DSM) programs provided by BC Hydro. But construction funded under these programs is not eligible to participate in the Standing Offer Program (i.e. BC Hydro would make capital cost contributions towards construction, but the power output would not be sold to BC Hydro; rather, it would just displace mine power purchases).

18.12.17 Water Treatment Plant Energy Recovery

This energy recovery plant will consist of a turbine at the discharge end of the 1,360 m-long pipeline from the runoff water treatment storage facility to the WTP, as indicated in Figure 1.5.

The energy recovery facility uses the water running downhill from the water treatment storage pond to the WTP in order to generate electric power. The flow will be through a 0.75 m internal diameter (ID) 1,360 m-long HDPE-lined pipeline from the storage facility to the treatment plant. The flow during the life of the project is in the range of 2 m^3/s. Due to seasonal variations, the gross head will vary from significantly during the year and will rise about 7 m during the life of the project. A 2,500 kW machine has been selected such that it has adequate capacity to pass the maximum flows without using a turbine bypass system that would be reserved for emergency operation.

A relatively small Turgo impulse type turbine or Francis reaction turbine will be used. The water turbine would be of all stainless steel construction. The turbine will be selected and designed taking into account the variable project generating head (pressure) due to the varying levels of the treatment water storage pond.

The generator will be housed in a small pre-engineered building adjacent to the WTP. The power output will be fed into the 25 kV mine power distribution system at the WTP.

18.12.18 Plant Tailing Energy Recovery

This plant will consist of two slurry pumps in one tailing line running in reverse as turbines, driving induction generators to supply power back into the local plant electrical distribution system. The plant normal tailing flow is 1.476 m^3/s in each of two tailing lines at an SG of 1.28. The upper station will operate at 26 m net (slurry) head while the lower station will operate at 39.2 m of net (slurry) head. Only one of the two tailing lines will have energy recovery implemented, because the second line has less available hydraulic head (pressure) for generation.

Tailing pumps operating in reverse as turbines (referred to as PATs) are selected as the drivers, as this equipment can withstand the abrasive nature of the tailing. The generators are of the induction type (basically induction motors running at slight over speed). The use of induction generators reduces costs and greatly simplifies the equipment and plant operation. As the tailing flow is available whenever the process plant is operational, the power generation plant capacity factor is high.

The equipment is located in two buildings at two separate locations straddling the north tailing line between the process plant and the TMF; the location is shown on plan in Figure 1.6. The installation is shown on the single line diagrams in Appendix I. A project report has been prepared, titled “Plant Tailing System Energy Recovery Evaluation”, which covers all details of this installation. The selection of appropriate tailing pumps for use as turbines requires a detailed technical study that is extensively covered in this report.

18.12.19 Mitchell Diversion Mini Hydro

This plant will make use of the normal stream flows that the MDT will divert around the mining operations. The installation will consist of a Pelton turbine; it will be very similar to IPP run-of-river hydro plants, in that it makes use of the flow as it naturally occurs, with no water storage facilities or other works other than that required for water diversion around the mine. The penstock from the diversion tunnel exit will run down to the power house located just above Sulphurets Creek.

The diversion flow will vary widely from summer to winter. The generation design flow will be 4.5 m^3/s and the gross head will be 205 m. The design output will be 7,500 kW. The plant capacity factor is 0.41 and the estimated average net energy production will be 27 GWh/a.

The turbine and switchgear will be housed in a small reinforced concrete powerhouse building designed to resist avalanches, located near Sulphurets Creek. Power will be delivered to the open pit 25 kV electrical distribution system.

The MDT is shown in Figure 18.11. Drawings of the actual power plant are provided in Appendix I. This plant will continue to operate after the mine is closed. The report covering all details of this installation is included in Appendix I.

18.12.20 McTagg Diversion Mini Hydro

The McTagg installation is deferred and is included in sustaining capital, as the current revised plans show the initial available hydraulic head to be quite low.

18.13 Plant and Mitchell Side Electrical Distribution

18.13.1 Main Power Feed

The total mine and process plant annual energy consumption is estimated to be 1,305 GWh and the average annual load is estimated to be 149 MW, in accordance with the current (HPGR) project load list for a 130,000 t/d operation. With a typical load factor in the range of 0.87 for a project such as KSM, the peak load is estimated as minimum 171 MW. The plant running (normal every day) load is estimated to be 159 MW. The required utility supply will be reduced 10.6 MW below this value in the summer and fall by self-generation from energy recovery and mini-hydro projects. During the winter low stream flow conditions, the average self-generation will be 4.1 MW. To prevent the project demand from exceeding the 150 MW trigger point for generation reinforcement, the proposed gas turbine located in or near Terrace will be operated. This has been included in project costs.

18.13.2 Main Substations

There will be two main substations; the first substation, located at the Treaty process plant, will be sub-fed via underground cable to the second substation located near the Mitchell pit. Both substations will contain transformers to step-down to 25 kV for distribution around the plants.

18.13.3 Power Distribution – Treaty Substation No. 1

The Treaty Substation No. 1 will contain three main 287-138-25 kV, 75/100/125 MVA transformers. Each transformer will feed a line-up of 25 kV switchgear. The 25 kV line-ups will be connected by a normally-open bus tie circuit breaker. There will be a total of 30 circuit breakers connected to distribution feeders. These feeders will distribute power throughout the site. Appendix I provides a more detailed description of the main substations.

Ball Mills

Each of the four ball mills (rated 11,000 kW) will be fed via dedicated 25 kV feeders and step-down transformers to 13.8 kV.

Step-down to 4.16 kV

The ball mills will be fed at 13.8 kV. Other large fixed speed motors (generally those rated 250 HP and greater) and large variable speed drives (generally those rated over 400 HP) will be fed at 4,160 V. The 4,160 V supply will be derived from 25 kV to 4,160 V outdoor liquid filled step-down transformers. Redundancy will be provided by utilizing sets of two transformers, each feeding a 4,160 V metal clad switchgear line-up with the two line-ups connected by a tire breaker that may be closed if one of the transformers fails or must be taken out of service. Typical motors in this group include:

- cone crushers
- large conveyors
- thickener underflow pumps
- cyclone cluster pumps complete with variable frequency drives (VFD)
- HPGR complete with VFD.

Some motors will be connected to downhill conveyors and capable of some power regeneration (e.g. the Mitchell pit rock pile conveyor). In this case, the VFD will be a four quadrant type; hence, it will be able to capture mechanical energy and feed back into the electrical system.

CONVEYOR SYSTEM

One-half of the MTT conveyor drive motors will be fed by 25 kV feeder cables from Substation No. 1 (the feed end of the conveyor will be fed from Mitchell area Substation No. 2).

STEP-DOWN TO 600 V

Motors and other loads below 200 kW will be fed from one of several 600 V systems. Generally these systems will consist of a pair of dry type 25 kV to 600 V step-down transformers, feeding two line-ups of 600 V power distribution centres (with tie breaker), which in turn feed a series of 600 V motor control centres (MCCs). General power and lighting will also be fed from the 600 V system.

REMOTE LOADS

Remote Treaty area loads will be served by 25 kV overhead lines. Examples of remote loads include:

- fresh water pumping
- TMF return water pumps
- ancillary buildings.

18.13.4 MITCHELL SUBSTATION NO. 2

The Mitchell Substation No. 2 will be fed by 138 KV cables running through the conveyor tunnel from the Treaty No. 1 substation. The Mitchell Substation will have two 138 to 25 kV, 25/33 MVA, ONAN/ONAF stepdown transformers. Each transformer will feed a line-up of 25 kV switchgear. The 25 kV line-ups will be connected by a normally-open bus tie circuit breaker. There will be a total of 11 circuit breakers connected to distribution feeders. These feeders will distribute power throughout the Mitchell site. Appendix I provides a more detailed description of the main substations.

POWER FEED TO PITS AND PRIMARY CRUSHER

The Mitchell primary crusher will be fed from the substation by a 25 kV cable. The Mitchell pit overhead power line will be fed from a section cable leading into the substation. Another cable will feed an overhead pole line feeding the truck shop, WTP, explosives facility, and also connecting to the mini hydro and energy recovery power plants.

The mining electric shovels and drills will be served at 7.2 kV via portable 25 to 7.2 kV step-down unit substations fed from the perimeter pit pole line. The estimates include appropriate lengths of trailing cable and couplers.

There will also be 25 kV cables to feed one-half of the MTT conveyor drives.

REMOTE LOADS

Remote loads will be served by 25 kV overhead lines. Examples of remote loads include:

- truck shop
- permanent camp
- WTP
- explosives magazine.

18.13.5 POWER DISTRIBUTION – FLOTATION SUBSTATION

The Treaty Substation No. 1 will include three main 287-138-25 kV, 75/100/125 MVA step-down transformers. Each transformer will feed a line-up of 25 kV switchgear interconnected by tie-breakers to provide redundancy in case one of the main transformers is out of service. There will be approximately 11 distribution feeders connected to each line-up. These feeders will distribute power throughout the site. Details are provided in the single line diagrams in Appendix I.

STEP-DOWN TO 4.16 KV AND 600 V

The distribution system at the process plant will be similar to the system used at the Mitchell plant.

18.13.6 ANCILLARY SYSTEMS

Ancillary systems to be provided under electrical include:

- emergency power
- general power and lighting (indoor and outdoor)
- electrical heating
- heat trace
- fire alarm
- communications
- closed circuit TV.

18.14 Permanent and Construction Access Roads

18.14.1 Background

Seabridge retained McElhanney to complete a study of permanent and construction road access options to the KSM zones, and mine facilities. EBA was retained to study the proposed construction period Frank Mackie Winter Access Road from Granduc.

Various alignments for proposed access road networks to the mine facilities have been considered. McElhanney's field work commenced in 2009 and continued through the summer of 2011 in assessing the various options.

Current proposed permanent access roads include the existing 59 km resource access road from Highway 37 to the former Eskay Creek Mine. This will connect to the proposed 35 km-long Coulter Creek Access Road south from the Eskay Creek Mine site to the proposed KSM mine site.

The Treaty Creek Valley road network will include a 30 km two-lane access road from Highway 37 to the Treaty plant site, TMF, and east portal of the MTT.

From km 17.9 on the Treaty Creek road, a single-lane road will extend further up the Treaty Creek Valley to provide access to the MTT mid-tunnel access portals, and additional construction access adit further to the west. The total road length from Highway 37 to the MTT mid-tunnel access portals will be 33 km.

Roads known as the North Treaty Lower and North Treaty Upper will be built to the north, off of the Treaty Creek road at km 16.9 and km 17.9, respectively, at different times during the life of the mine. These will be two-lane roads and each will be approx. 12 km in length. Select sections of each road will parallel the drainage cut-off ditch in the North Treaty/Teigen Creek valley. Another single-lane (4 m wide) road, approx. 4 km in length, will provide additional access for construction and maintenance to the south end of the above-mentioned drainage cut-off ditch.

Alternate TMF sites and corresponding access routes to them, and to the plant site, were analyzed by Rescan using an Alternatives Assessment Process and associated reporting. Results of that study, and preferred access route findings were presented to the Working Group Committee in Smithers on March 29 and 30, 2012. No significant objections were raised.

The routes currently proposed will be described in this report. McElhanney's full report relating to current access road options is contained in Appendix J.

18.14.2 Objectives and Methodology

Utilizing the August 2008 LiDAR survey data and digital elevation model (DEM) provided by McElhanney's mapping division, the initial preferred access routes identified by Seabridge and McElhanney were defined on the base mapping.

Based on the mapped information, in 2009 the preliminary alignment files for each proposed road centerline were uploaded to GPS units. The GPS units were used to locate and flag the initial routes on the ground. The objective was to locate and map the most appropriate road alignment for each route based on design standards established by the project team.

The routes were assessed in the field and adjusted as deemed appropriate. Often several preliminary lines were investigated in order to achieve the preferred road location. Selecting the ultimate road locations is an iterative process involving both field and office design. Based on the preliminary layout, terrain information was gathered, along with bridge and major culvert crossing information. The originally flagged centerline provided a base for follow-up environmental and geotechnical assessments.

Based on the field reconnaissance, design standards, and associated surveys and preliminary assessments/input by other sub-consultants; preliminary road design plans and profiles, conceptual bridge and stream crossing structure designs, and a construction cost estimate were prepared. Engineering assessments were conducted in conjunction with available geotechnical and environmental studies of all then proposed routes. The field reconnaissance and bridge site surveys confirmed the accuracy of the LiDAR data.

In 2009 and 2010, BGC and Rescan conducted further geotechnical and environmental assessments, respectively, on the proposed routes. Suggested changes were made to the alignments at select locations.

In the summer of 2010, Mr. Bob Parolin of McElhanney assessed these proposed changes, and implemented the recommendations where appropriate. Some changes to alignments were adopted, with or without revision, while other changes were not made. The review involved ground-truthing the revised routes, and updating mapping and survey information as required. Road and bridge design updates were prepared for inclusion in the previous "KSM PFS Update" dated June 15, 2011 (Wardrop, 2011).

During the 2011 season, McElhanney crews completed additional field assessments on the potential alignments. Work included RTK-GPS layout of then current design alignments, with some field modifications as deemed appropriate. All proposed road locations were marked with survey flagging. Flagging was marked with survey crew and date information (black felt marker), and locations identified by RTK-GPS survey methods. Select field station references are now indicated on the road plan/profile design drawings for cross reference.

Work included gathering of detailed information to be utilized in refining the design(s), including drainage culvert requirements, soils, vegetation, potential borrow/waste areas, drainage culvert requirements, and other relevant information.

Work also included realigning sections of the Coulter Creek route, both west of the Unuk River, and through the climb out of the Unuk River valley, east and along the north bank of the Sulphurets Creek Valley. This work was conducted to reduce the extent of long and steep road gradients, which were deemed undesirable.

Additional stream crossing surveys were completed on “smaller” tributaries not surveyed earlier. Generally this includes any stream with an estimated 100-year peak flow of 6.0 m^3/s or greater. Details of all such structure requirements will ultimately be needed to satisfy the requirements of the Ministry of Forests for the Special Use Permit (SUP) application. Preliminary stream crossing structure designs have been completed for all sites surveyed. Only a few additional sites indicated as requiring similar work have been identified by subsequent office evaluation, based on mapped drainage areas, and calculated peak flow values.

During 2011, attention shifted to consider the Treaty Creek Valley as one of the primary access routes. Based on available Shuttle Reconnaissance Topography Mapping (SRTM), McElhanney considered route options for access development from Highway 37, up the Treaty Creek Valley to the MTT mid-tunnel access point, and also up the North Treaty Creek/South Teigen Creek Valley to the proposed Treaty processing plant site, TMF, and east portal of the MTT.

In September/October 2011, these routes were assessed using methods similar to those employed on the other access routes, as described above. In autumn 2011, McElhanney’s mapping division acquired additional LiDAR data in the Treaty Creek Valley, and the subsequent DEM produced forms the basis for the current preliminary road design work, and other facilities, in those areas.

In late summer 2011, BGC and Rescan conducted preliminary geotechnical and environmental assessments, respectively, on the Treaty Valley routes considered at that time, and as described above. Due to time constraints before the onset of winter, more detailed assessment work will be required in these areas during the 2012 summer season.

Subsequent to design work completed over the winter of 2011/2012, realignment of a few select sections should be considered, pending field re-assessment, as a means to improve the road geometry and reduce construction costs. Additional field assessment work will be conducted during the 2012 summer season by McElhanney, BGC, and Rescan.

Also, during prefeasibility design in winter 2011/2012, the originally planned alignments had to be shifted to accommodate other mine support infrastructure and take into account construction and long term haul considerations. Therefore, McElhanney and other sub-consultants will need to conduct additional field work during the 2012 summer season to verify the suitability of the office-adjusted alignments.

Current updated preliminary road design plans and profiles, and conceptual bridge and stream crossing structure designs are provided in Appendix J.

18.14.3 Route Descriptions

The current proposed access roads include the:

- Eskay Creek access
- Coulter Creek access
- Treaty Creek access
- North Treaty Lower access
- North Treaty Upper access (Phases 1 and 2)
- Cut-off Ditch access
- MTT Tunnel Adit access
- Frank Mackie Winter Access Road.

Currently proposed road locations are shown in Figure 18.26.

The current updated route descriptions, including relocations of the road alignments, are provided in the following sections.

Eskay Creek Mine Access Road

This road was constructed in the early 1990s to provide access to the Eskay Creek Mine, which commenced gold production in 1994 and closed in 2007. The road commences at Highway 37, south of Bob Quinn, and follows the Iskut River Valley west for approx. 37 km to Volcano Creek. The road was built as a single-lane, 5 m-wide gravel road, with a nominal design speed of 60 km/h. The road is currently being used to service construction of the Forrest Kerr hydro-electric development by AltaGas. This development is located at approximately km 30.

Beyond Volcano Creek, the 22 km-long Eskay Creek Spur Road was used solely for access to the Eskay Creek Mine, but will potentially be used for access to a second hydro-electric development proposed by AltaGas. The road was constructed as a single-lane, 5 m-wide gravel road, with a nominal design speed of 50 km/h. There are four single span bridges along this section of road. Information suggests that the access is still passable, and has been used by low-bed trucks hauling equipment in recent years.

AltaGas holds the SUP (Special Use Permit from the Ministry of Forests) over the first 43 km of the road, with Barrick Gold Corp. holding the SUP over the section from approximately km 43 to km 59. Jurisdiction over the latter section of road might change. Both sections of road will be subject to shared access/maintenance agreements.

A detailed evaluation of the road condition and suitability for Seabridge requirements is planned for the summer of 2012. An allowance has been made in the budget for upgrades to the existing access road between Highway 37 and the start of the Coulter Creek Access Road.

COULTER CREEK ACCESS ROAD

Heading southwest from the Eskay Creek Mine access road, at a location approx. 59 km off of Highway 37, this road will follow an existing mine access road for approx. 3 km towards Tom MacKay Lake. It will then descend out of the alpine meadows along the height of land on the east side of Coulter Creek.

During 2011, the road location was significantly realigned between approx. km 7.0 and km 13.8, with more minor alignment optimization from there to the Unuk River crossing at km 20.9. The overall road length increased by approx. 1.7 km. The previous alignment resulted in significant through-cuts and sections of road with sustained grades of up to 15%. The proposed changes in road alignment has shortened the sustained grades, and reduced them to a more acceptable maximum 12%. Earthwork volumes have also been reduced in the process. Efforts were made to avoid potential geohazards mapped previously by BGC.

There are five bridge structures proposed along this section, between the start of the road and the Unuk river crossing. Most stream crossings through this section, with the exception of Coulter Creek, have been deemed non-fish bearing.

Rescan identified and mapped a number of blue-listed ecosystems along the road as it descends along Coulter Creek to lower elevations. To avoid some of these sensitive ecosystems, and to reduce the impact on fisheries sensitive zones, in 2010 the Unuk River bridge crossing was re-located upstream of the originally proposed crossing. The proposed three-span bridge crossing will be 88 m in length. The Unuk River is a major crossing, and will need to meet the requirements of the Navigable Waters Protection Act (NWPA).

Beyond the Unuk River, the route will climb steeply through a series of “switchbacks” from km 23 to km 25, into the Sulphurets Valley and canyon. This section of road was realigned in the field during 2011, and further refined in the office to optimize the road geometry, and attempt to minimize rock excavation volumes. This is a difficult section of road, with very limited options for improvement. Maximum road grades are 12%, reduced through the switchbacks. Through-cuts have been minimized but significant sections will still require full bench cut and end haul to waste. Steep, unstable rock areas to the south have been avoided.

The road crests at km 25.2 then descends for approx. 1 km, following the steep north side of the Upper Sulphurets Creek Valley. Along this section, it will traverse significant sections with steep rock, crosses numerous avalanche paths, and is exposed to rock fall hazards (km 26 to 31). Again, much of this area requires full bench cut and end haul to waste. Waste opportunities are very limited along this section.

Engineered structures, and avalanche monitoring and control will be required to mitigate the hazards along this section. Consideration has been given for the construction of six snow sheds between km 26.5 and km 30.6, with lengths ranging from 50 m to 80 m (total 370 m allowance). These would be placed only in the most channelized avalanche/drainage paths. Avalanche monitoring and active control would be used initially during project construction and mine start-up. Snow shed protection is optional, and would be constructed only after the mine has commenced normal operations. Other passive means of avalanche control including deflecting berms and retarding mounds could be employed, but have not been considered in the design at this time.

Most snow avalanche mitigation would be handled by active measures including monitoring (Canadian Avalanche Centre or contractors), road closures, no-stopping zones, helicopter bombing, Howitzers, GasEx exploders, etc. Despite the presence of mountain goats in this valley, it is our understanding that with appropriate cautions, the latter more pro-active measures could be used along this section of road.

Previously Rescan had recommended that the road be moved to the south side of the valley, to avoid winter goat habitat and some of the avalanche-prone terrain. Due to unstable terrain adjacent to Sulphurets Creek, and the length of span required to bridge the creek, this option was rejected.

Beyond approximately km 31.5, the Sulphurets Creek Valley widens considerably and the road location will continue on the north side of the valley to the bridge crossings of Gingras and Mitchell creeks.

The access road beyond approximately km 33.9 (Mitchell Creek crossing) was realigned to accommodate the location of the WTP on the bench lands, as proposed by KCB. The new alignment extends approximately 800 m to the southeast, at which point it was realigned to avoid potentially unstable ground to the south. It turns northeast, and climbs to km 35, where McElhanney's road design now ends. Beyond this point, the road design is determined by the mine development, and is the responsibility of MMTS.

TREATY CREEK ACCESS ROAD (TO KM 17.9)

The Treaty Creek Access route will leave Highway 37 approximately 19 km south of Bell II, and head west. It will be constructed as a two lane (8 m surface) all-season road to the junction of the Treaty Creek and North Treaty Upper road at km 17.9.

A traffic study has been completed, and warrants show that minimal updates should be required at the proposed intersection with the highway.

Previously meetings were held between McElhanney, Seabridge, and the provincial Ministry of Transportation and Infrastructure to discuss and establish a set of design criteria for the originally proposed intersection at the Highway 37/Teigen Creek access road location. That location required full intersection reconstruction, with a left turn lane for north bound traffic.

The results of the current Highway 37/Treaty Creek intersection assessment have been delivered to the Ministry of Transportation and Infrastructure. A decision on intersection requirements acceptable to the Ministry has not been made. Allowance has been made in the current cost estimate for extensive intersection upgrades, but this might not be necessary. In either case, additional detailed design drawings will be required to meet the Ministry's final requirements, prior to construction.

Initially the road will follow a former forestry access road. At approximately km 0.6, a three-span 119 m long bridge is proposed for the crossing of the Bell-Irving River. There is evidence of a previous bridge installation at this site. This is a major river and will need to meet the requirements of the NWPA.

The proposed road essentially follows an existing forestry access trail for approx. 4 km. Significant upgrades will be required. At km 4.7 potential debris flows will be handled by the installation of twin bridges, which also satisfy fish passage requirements.

Between approx. km 3.7 and km 3.8, the road bisects a polygon identified by Rescan as an environmentally sensitive ecosystem. The road location will be realigned in the field during the 2012 season to avoid this area.

The proposed road follows the north side of the Treaty Creek Valley. It will generally be located between the flatter riparian zone below and the steeper avalanche-prone terrain on the north slope. The proposed road location was kept low on the slope to avoid the steeper side hill terrain which would require full bench cuts and end haul.

Beyond approx. km 8.6, the road will start to climb; it will traverse moderately sloping side hill terrain to approx. km 17. A number of streams are crossed, which are assumed to be fish-bearing. A total of 7 bridges (at 5 sites) are designed between the Bell-Irving major crossing and the future intersection at km 17.9. Bridge lengths vary from 15 m to 24 m. There are also two culvert crossings that require detailed site plans and general arrangements (>6.0 m^3/s, 100-year peak flows).

The North Treaty Creek crossing at approx. km 16.3 has the potential for flows to be shared, or shift between the two channels present. The twin bridge installations (one on each channel), are each designed to pass the full estimated Q100 flows. The natural peak flows will be reduced once the planned cut-off ditch along the west slope of the North Treaty/Teigen Creek Valley is in place to re-direct flows north to Teigen Creek.

Several avalanche chutes will be crossed along this route. No snow shed structures are being considered along any sections of the Treaty Creek access roads for snow avalanche control. Deflection berms and retarding mounds may be considered where appropriate, but have not been detailed at this time. The primary consideration will be the application of active avalanche control measures along this route, similar to those described for the Coulter Creek road.

At approx. km 16.9, there will be an intersection with the North Treaty Lower Road. The double-lane road will turn north through a switchback and follow a path low on the west bank of the North Treaty Creek Valley, eventually climbing to the Treaty plant site. This road is described later in this section.

The main Treaty Creek double-lane road will continue further west to approx. km 17.9. At this point, there will be a future intersection. Heading west, it will transition into a single-lane road leading to the MTT saddle access portal and tunnel adit access point. This route is described later in this section.

At the proposed intersection, another double-lane road will be built in future. It will be known as the North Treaty Upper road, and will be built many years in the future, once it is necessary to construct the south tailing dam, and close the northern section of the North Treaty Lower road, which will be used during the earlier part of the mine life.

Treaty Creek Access road – West To Tunnel Saddle Access Portal

Beyond km 17.9 and heading west up the Treaty Creek Valley, the single-lane road will provide construction period access, and longer term maintenance access, to the MMT saddle access point and beyond. From km 17.9 to km 33, the road standard is reduced to a nominal 6 m finished road width.

Most of this road will traverse moderate to steep side hill conditions. Much of this section of road will be subject to snow avalanches. Passive structures have not been considered. It is anticipated that active snow avalanche mitigation measures will be utilized, similar to those described earlier for the Coulter Creek road.

Only one bridge is required along this section, at approx. km 20.2, and one open-bottom engineered arch structure at km 29.9 for fish passage. Initially, many of the stream crossings have been assumed to be non-fish bearing due to excessive stream gradient. Ultimate structure selections could change, subject to future fisheries evaluations by Rescan.

Based on mapped drainage areas, it appears that one additional site plan survey/design will be required at approx. km 29.0.

Approaching the tunnel saddle access portal, efforts were made to coordinate the road location design with required tunnel muck spoil pads, sediment ponds, and drainage requirements as defined by KCB. In order to achieve this, the road location had to be shifted substantially downhill from the location established during the 2011 field program. To avoid the other facilities and still maintain reasonable road grades, it was necessary to re-route the road further to the west, adding a switchback, before climbing up to the proposed tunnel saddle access portal location.

A creek crossing site plan surveyed earlier in this area is no longer relevant. A new crossing location will need to be verified and surveyed. The current road design was terminated at a work location approx. 100 m from the actual tunnel portals. The proposed relocation will need to be field-truthed during the 2012 work program.

Treaty Creek – Tunnel Adit Access Road(s)

At approx. km 32.2, an additional Tunnel Adit Access Road will head west. The purpose of this road is to access an intermediate tunnel construction access point, to speed MTT construction, and allow hauling of tunnel excavation back to the tunnel access portal spoil area. The main access road will be approx. 2.9 km long, extending to the tunnel adit location, above the proposed temporary muck storage pads. There will also be a 0.6 km-long road providing access to the base of the temporary tunnel muck storage pads.

This route to the intermediate tunnel adit was not previously considered in time to permit field assessment during the 2011 season. The road designs are based on available mapping only, and will need to be field assessed during the 2012 work program. Again, efforts were made to coordinate the location of these roads with the temporary tunnel muck storage and drainage requirements as defined by KCB.

North Treaty Creek Access Roads

There are currently three access road alignments proposed within the North Treaty/Teigen Creek valley. They shall be referred to as the North Treaty Lower Access Road, the North Treaty Upper Access Road, and the Cut-off Ditch Access Road. The North Treaty Upper Access Road is further split into Phase 1 (2.2 km) and Phase 2 (5.7 km).

Initially only the North Treaty Upper Access Road was considered, to extend from the Treaty Creek Access Road, and travelling approx. 12 km north to the Treaty plant site. The initial proposed alignment was traversed during the summer 2011 season. This route leaves the Treaty Creek Access Road at approx. km 17.9. There will be a switchback leading to a “sustained” climb (nominal 10%). This road must climb to attain an elevation sufficient to clear the future south tailing dam elevation (nom. 1070 m). The road would then parallel the proposed drainage cut-off ditch, which will divert drainage off the west slope of the valley, north to the Teigen Creek Valley. In order to satisfy the geometric slope requirements of the revised cut-off ditch location as proposed by KCB, it was necessary to move the road alignment significantly off of the field traversed line. Therefore, the proposed new alignment will need to be set out and field-truthed again during the 2012 season.

The road location and terrain dictates that significant portions will need to be built using full bench/end haul to waste construction. Construction of this access road would be critical to gaining early access for construction of the MTT tunnel, commencing at the east portal.

Therefore consideration is now given to constructing the North Treaty Lower Access Road. This would leave the Treaty Creek Access Road at approx. km 16.9. Again, this route has not been assessed in the field, and the current design is based only on available mapping. This route would cross a number of steep gullies. The terrain has some steep sections, but is generally flatter than the North Treaty Upper Access Road location. Though some full bench and end haul construction will be required, gaining access to the MTT east portal location will be quicker.

BGC prepared a Project Memorandum titled “North Treaty Road Realignment (dated March 15, 2012). The summary suggests no geohazards were identified that are likely to prevent the construction of the North Treaty Lower Access Road. Rescan has not completed any field evaluation along the North Treaty Lower Access Road alignment at this time.

The North Treaty Lower Access Road would result in a slightly shorter haul distance between Highway 37 and the plant site, and generally flatter grades. This road would be used for approximately the first 25+ years of the mine life, until such time as it is necessary to construct the south tailing dam. The North Treaty Lower Access Road would match to the North Treaty Upper Access Road approx. 8.1 km north of the Treaty Main turnoff. The North Treaty Lower Access Road would have the added benefit of allowing access to the lower valley to initiate construction of the tailing dam(s). Eventually the north section of this road would be buried by the south tailing dam and settling pond.

Early in the mine construction it would still be necessary to build a section of the Upper road that parallels the cut-off drainage ditch (Phase 1: 2.2 km). This would be built to the ultimate double-lane standard (8 m width). Construction of the Cut-off Ditch Access Road would also be required. This would be for construction access and maintenance only, and would be built to a lower standard (4 m road width). The power transmission line is proposed to follow this route.

Except for approximately 4 km of the future North Treaty Upper Access Road, all other proposed roads in the North Treaty Valley will need additional assessment work by McElhanney, Rescan, and BGC during the 2012 field season, to confirm the suitability of their proposed locations, which at this time are based only on available air photo and LiDAR mapping information.

18.14.4 ROAD DESIGN REQUIREMENTS

The KSM Project access roads are classified as resource development roads. The design criteria proposed for each of the roads is included, along with typical cross sections, in Appendix J.

The Eskay Creek and Coulter Creek access roads will be maintained for the life of the mine to support the mine development, transport of oversize loads, and to provide alternate emergency access.

The Coulter Creek roads will be single-lane (6 m surface) radio-controlled roads with turnouts and widenings to allow the largest vehicles and loads access to the mine site. The Coulter Creek road would have some sections with sustained maximum grades of 12%. Design speeds vary greatly, in large part controlled by the terrain.

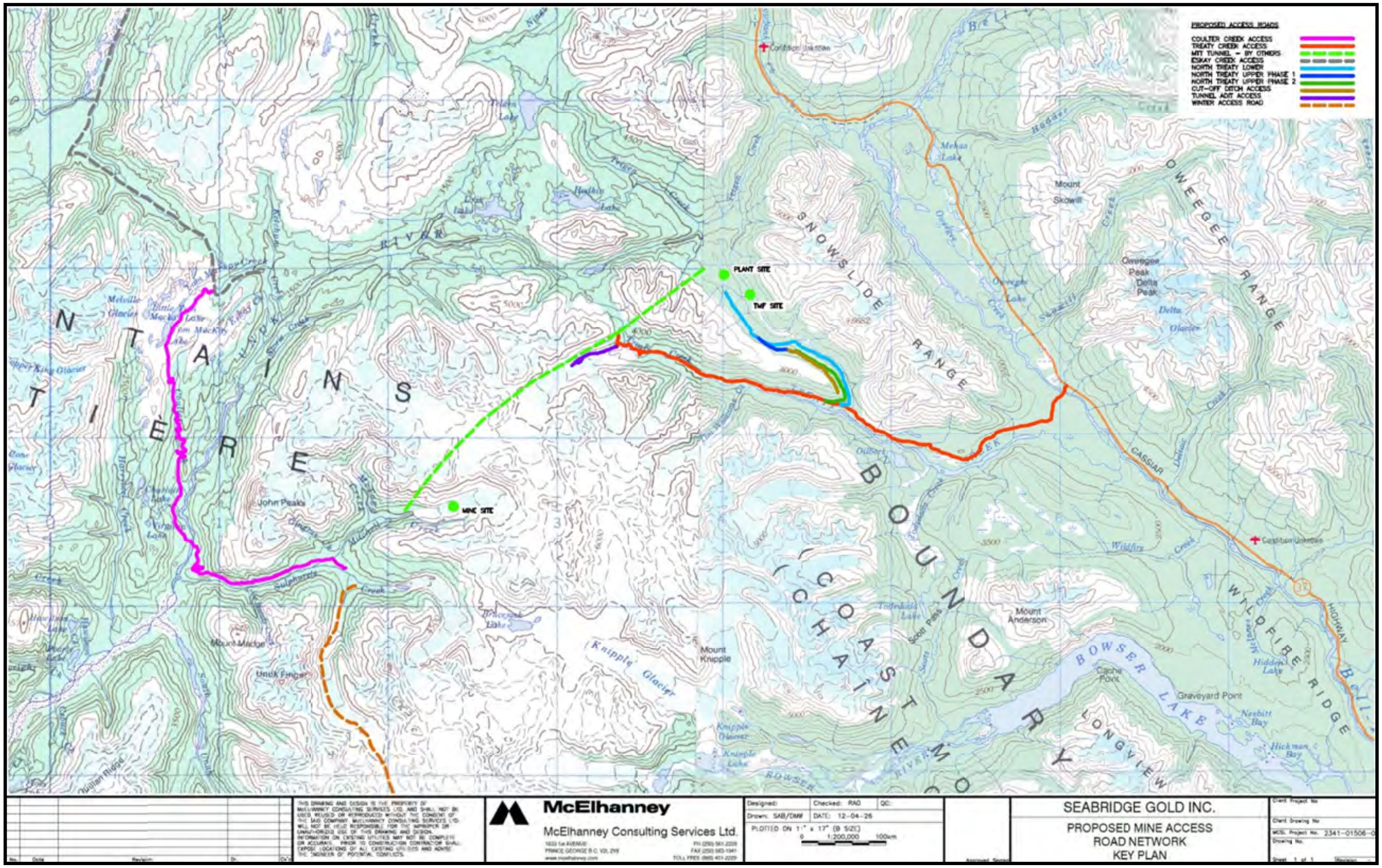
The proposed Treaty Creek Access Road to km 17.9, and the connecting North Treaty access road(s), will be required for permanent access to the Treaty plant site, the TMF, and eventually to the mine site via the MTT. This will be a two-lane road (8 m finished surface), capable of carrying the legal axle loading for trucks on BC highways on a year-round basis. The roads will provide access for supplies, equipment, and crew transport, and be used for hauling concentrate to Highway 37.

Alignment controls such as maximum 12% short pitch grades (10% sustained), and minimum 100+ m radius horizontal curves are recommended for the higher-traffic volumes anticipated on this route. Except for a few control sections, the nominal minimum design speeds for these sections of road is 50 km/h, and maximum 60 km/h where feasible.

West of km 17.9, the Treaty Creek Access Road and adjoining Tunnel Adit Access Road will be required to support tunnel construction. These will be single-lane (6 m surface) radio-controlled roads with vehicle turnouts and widenings. Nominal minimal design speeds of 40 km/h have been achieved. The Tunnel Adit Access Road can be deactivated upon completion of the tunnel, while the Treaty Creek Access Road will be maintained for long term maintenance access.

All bridges will be designed to BC Forest Service L100 loading (90,680 kg GVW) and minimum 1.5 m clearance above the estimated 100-year flood level (Q100). Select structures must meet additional requirements, as prescribed by the *Navigable Waters Protection Act*. All bridges, including those on the Treaty Creek Access Road, will be single-lane.

Figure 18.26 Proposed Access Roads Network



18.15 PROPOSED WINTER ACCESS ROAD

Figure 18.27 shows an approximate alignment for the proposed winter access road from a laydown area near the Granduc Mine to the KSM Project site. Additional topographic information is still required to refine the alignment of the sections of the route that are particularly challenging from a grade perspective and those areas where the micro topography is particularly rough. The full EBA report is provided in Appendix J.

Two reconnaissance visits, coupled with a small haul of equipment into Pretium's Brucejack Lake property, have pointed to the feasibility of gaining access onto the Berendon Glacier near the Granduc Mine and then onto the Frank Mackie Glacier.

The quantity of snowfall throughout the winter, the relatively warm climate, and the topography will not allow the construction of a typical "ice road", where conventional highway vehicles could be used. The road must instead be a "snow road" where tracked equipment pulls skid-mounted sleds to haul the various loads to site. Maximum weight and sizes of the loads that could be hauled over the route by the tracked equipment pulling sleighs are anticipated to be limited to the size that could be hauled on normal highway transport trucks (33 t maximum weight and 2.3 m maximum width). Most of the surface of the glaciers is relatively smooth with little crevassing. However, some locations near the terminus of the glaciers in both the Ted Morris Creek and Bowser River valleys have prevalent crevasses. Snowfall appears to be sufficient to fill in most of the crevasses during the winter, at least on the Berendon Glacier. There are still some questions about the amount of snowfall in the Ted Morris Creek Valley being sufficient to cover the extreme micro topography from rock fall debris in this area.

The suggested route will be approximately 38.4 km long. It appears that as much as 32.8 km of the road will be constructed on the glaciers. Although the topographic data is not very precise, the bulk of the route appears to have grades of 4-6%. Steeper grades upwards of 30% exist at the toe of the Berendon Glacier and on the small side glacier that allows access onto the Frank Mackie Glacier from the Berendon Glacier. There are also steep sections with grades of up to 15% near the crest of the Frank Mackie Glacier. The total vertical variation is roughly 1,020 m (3,350 ft) between the Granduc Mine area and the crest of the Frank Mackie Glacier.

There is increased risk of avalanches and rock fall hazards at some areas along the route, such as the narrow portion of the Ted Morris Creek Valley at the terminus of the glacier (Figure 18.28 and Figure 18.29). There is considerable evidence of recent rock falls in this area; piles of rock debris have been observed on top of the glacier and in the valley bottom. This is also the area where there is limited concern about there being enough snow in the valley to pad over the rock fall debris. This may entail pushing or hauling snow from other locations on the glacier to allow the road to be constructed.

Figure 18.27 Proposed Winter Glacier Access Route

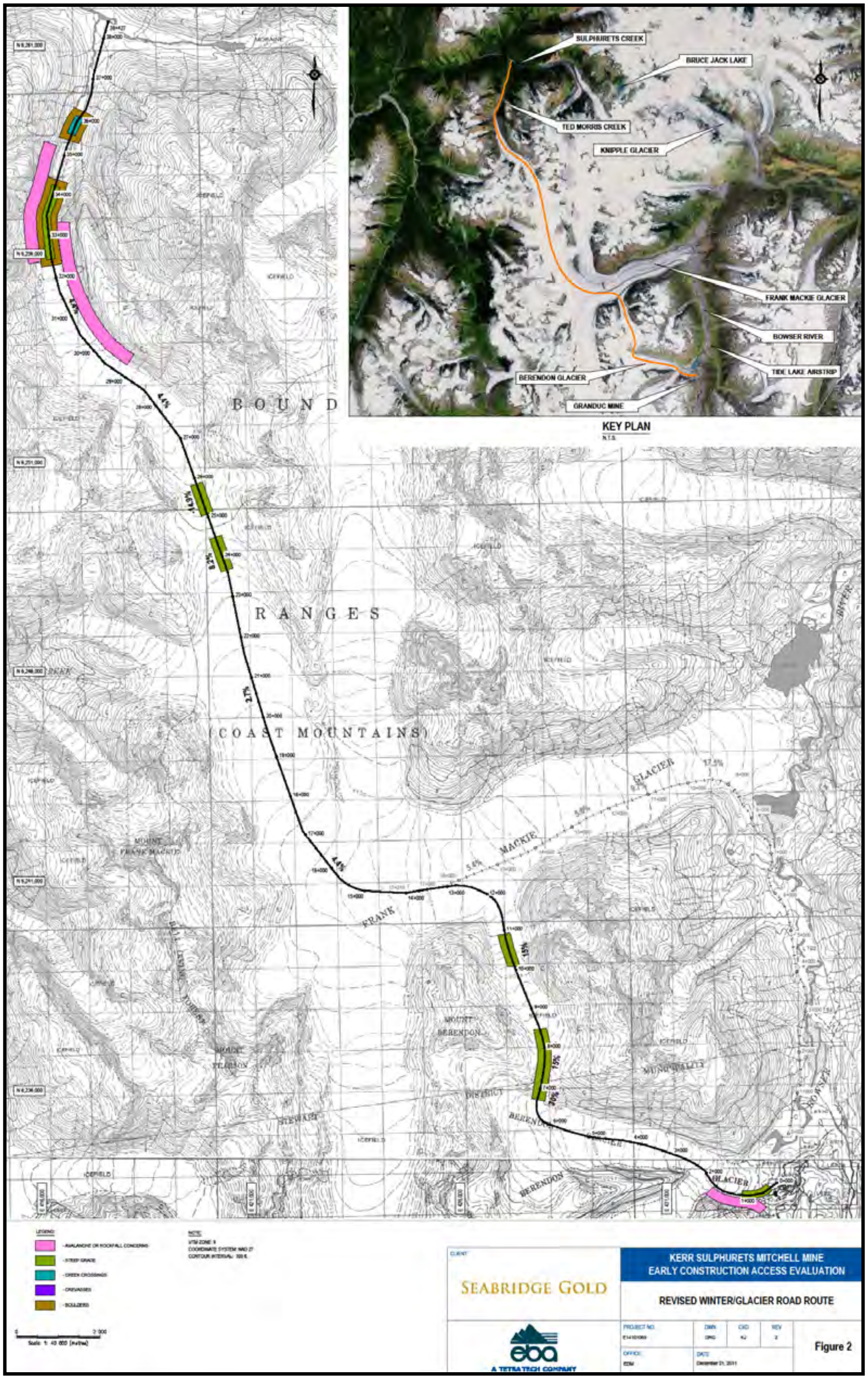


Figure 18.28 Looking South up the Frank Mackie Glacier*

* at approximately Sta.32+500 (Figure 18.27). Note the rock fall debris on the glacier surface.

Figure 18.29 Looking South up the Ted Morris Creek Valley*

* at approximately Sta.33+500 (Figure 18.27). Note the creek emerging from a melt water tunnel in the glacier, and the rock fall debris on the glacier surface and in the bottom of the valley.

Figure 18.27 identifies areas along the route that are expected to present a challenge or areas where risks will be greater during construction and operation of the winter access road. Selected photographs taken during the route reconnaissance, some of which identify these areas of concern, are provided in Appendix J.

Safety and environmental issues must be addressed with the proposed winter road. These include natural hazards such as avalanches, rock falls, and road failure into underlying caverns or crevasses. Any of these issues present real hazards to the safety of the personnel working on the road, and bring about the possibility that various types of substances could spill into the environment.

A road similar to the proposed winter access road was constructed across the Knipple Glacier into the Brucejack Lake area is a precedent for the approach with the various provincial government permitting and regulatory groups. Also, Pretium's use of the Berendon and Frank Mackie glaciers to gain access to their site in the winter of 2010 also provides some precedent. The support camps will require a specific water licence for water supply and sewage. Waste will likely be incinerated or trucked off-site.

A comprehensive evaluation of the landslide and rock fall hazards, as well as avalanche evaluations, will have to be conducted as part of the required detailed planning. During construction and operation, an avalanche team will have to continually monitor snow conditions and undertake avalanche control measures.

Ground Penetrating Radar (GPR) surveys will need to be conducted to look for cavities and snow bridges over crevasses in the glacier.

Detailed safety and environmental spill plans will have to be developed to support this operation. Measures that should be employed to prevent spills include using double-walled "Envirotanks" for storage and transport of hydrocarbons. All equipment must be in good working order with appropriate spill collection and cleanup equipment.

The vehicles hauling on the glacier will travel in convoys in case of breakdowns. Several "survival shacks" will be located on the glacier in case rapid changes in weather force the convoy to stop before reaching the end of the road. Haul equipment will have GPS navigation devices and will remain in radio communication with other haul equipment and camps. The winter road will be marked with regular highly visible stakes for visual guidance in case of white-out conditions.

Some baseline environmental data has been collected in the Ted Morris Creek and Bowser River valleys. It is understood that there are fish in the Bowser River; therefore, the road must be located to avoid the low flow channel.

18.16 Logistics

A preliminary logistics study was performed to determine the preferred means of transporting mining and construction equipment to the KSM site, and concentrate from the KSM site to storage and concentrate off-loading facility port sites. The logistics study is provided in Appendix G.

As shown in Figure 1.2, there are several transportation route possibilities for bringing equipment and supplies to the KSM property:

- The first route involves road access via Highway 37 from the south. Heavy equipment may also be transported through Smithers, BC, by transport truck or rail to the closest viable rail siding at Smithers, and then loaded onto transport trucks and taken to the KSM site along Highway 16 and Highway 37 north to the junction of the Treaty Creek Access Road.
- Access to the KSM mine site involves extending the Eskay Creek Mine road to the south for a distance of about 35 km. This road parallels Coulter Creek, before crossing the Unuk River and progressing up Sulphurets Creek.
- A third route involves bringing equipment and supplies by barge to Stewart, BC, and then transporting the equipment via Highway 37A to Highway 37, to the junction of Treaty Creek Access Road and Highway 37.

The existing highways leading to the project area may require some upgrading of bridges and other crossings in order to accommodate the equipment loads. Further evaluation of the upgrades will be identified during the next phase of the project study.

The project will also utilize a marshalling/staging area at Smithers, to receive and deliver equipment and supplies to the site during construction and operation of the KSM mine. Equipment will be transported via truck or rail to the closest viable rail siding at Smithers, and then loaded onto transport trucks and taken to the KSM site along Highway 16 and Highway 37 north to the KSM access road at the junction of the Treaty Creek Access Road and Highway 37.

A proposed Winter Access Road will be constructed that leads to the KSM mine, as detailed in Section 18.15. The Winter Access Road will be used to mobilize water treatment supplies and mobile equipment, as well as supplies for construction of access roads and water diversions during the first season. The Winter Access Road will be used during the next two winter seasons as well, until the Coulter Creek Access Road has been completed. It will also provide access for the construction of portions of the Coulter Creek Access Road, near its east end and to the Mitchell mine site area.

Copper concentrate will be transported from the KSM site by trucks to a deep water port facility in Stewart, BC, and then loaded onto oceangoing vessels. Stewart Bulk

Terminals is a deep-water port capable of handling concentrates for export via ocean vessels. Other operating materials, consumables, and supplies may also be stored at Stewart.

The terminal is at the head of the Portland Canal, which is a 150 km fjord that is ice-free throughout the year. The terminal is accessible via truck on Highway 37A; however, there is no direct rail service. The terminal currently handles concentrates for Imperial Metal Corp.'s Huckleberry Mine and Yukon Zinc Corp.'s Wolverine Mine. In addition, there is strong interest from other Projects in the region for handling services at the terminal.

Because existing facilities will be used for other concentrate business, and given the large volume of concentrates anticipated from the KSM Project, a substantial capital investment and contractual obligation may be required in order to secure handling services.

Potential port options may also be available in Prince Rupert; however, brownfield or greenfield investment may be required.

For purposes of this study, Tetra Tech assumed that the copper concentrates will be shipped in bulk, and that the annual output will be approximately 322,000 t.

In the case of molybdenum concentrate, it will be transported in bags from the KSM site via trucks to the port of Prince Rupert. The bags will be transferred from the trucks to containers and then delivered to Fairview Terminal for ultimate loading onto an oceangoing vessel.

It was assumed that the processed molybdenum will be loaded in 1-t bags for transport purposes, and that the annual output will be approximately 1,800 t.

Mr. Jack Butterfield of Butterfield Mineral Consultants Ltd. was relied on for matters relating to the smelting terms, refining terms, saleability, and sales terms for copper concentrate and molybdenum concentrate. This report assumes that both the copper and molybdenum concentrates will be shipped via standard ocean transport to overseas smelters in Asia. Ocean freight and other related costs will be contingent on the final destination and sales arrangements.

18.17 Construction Execution Plan

The Construction Execution Plan (CEP) describes how the KSM Project could be constructed. It defines the Project Management methods and construction elements required to execute the Construction Management for the Project. It also establishes an execution philosophy and defines the organization, work processes, and required systems.

The processes outlined in the CEP would ensure that the Project is completed in a timely, efficient, and safe manner and that the project deliverables and facilities will satisfy Owner expectations.

This section generally outlines the Construction Program and the level of planning that has been achieved to date.

18.17.1 Introduction

A detailed Early Works Plan has been developed to ensure an efficient project start-up that is safe, under control, and follows the project objectives and guidelines. "Site Capture" is defined as the readiness of the Owner and Engineering, Procurement, and Construction Management (EPCM) Contractor to support the main Construction Program. Infrastructure and support services must be in place and functioning efficiently for a successful Construction Program.

The following are included in this Early Works Plan:

- site access plan: winter access road, pioneer roads, bridges, followed by completed roads
- site preparation plan: detailed cut/fill program
- solid drill/blast program
- explosive supply: storage and controls
- sourcing road materials: setting up crushing and screening facilities
- fuel supply and storage plan on site immediately upon achievement of road access
  - temporary double lined portable fuel storage tanks
- accommodations:
  - exploration specifications
  - prefabricated permanent accommodations to follow as soon as the completed road access and site preparation is ready; fabricate and install the permanent facilities as early as possible
  - health and hygiene program in place; washrooms and lunchrooms will be supplied by the Owner to ensure a high standard is continuous throughout the construction program
- temporary construction power:
  - standalone power supply systems (gensets) in containers with fuel systems
  - power distribution to begin on the ground then relocation to power poles
- detailed safety program:
  - medical facilities

  - emergency response plan, facility, with helicopter support
  - informative site orientation program
- early works environmental plan to manage waste, spills, and fuelling
- employee transportation plan for the early construction program; air and ground planning required
- logistics supply plan for the early material requirements:
  - material storage plan including lay-down areas
- batch plant installation and supply of cement and aggregates
- Infrastructure buildings and services to support the site and staff.

KEY PROJECT OBJECTIVES

The key project objectives are as follows:

- Deliver an optimized, safe, and environmentally compliant facility in accordance to the project systems and procedures.
- Achieve project completion within the agreed project schedule.
- Construct a quality facility, which meets the defined project objectives, using practical and industry standard methods.
- Perform all activities in a safe and effective manner, with zero recordable incidents.
- Limit the number of design and construction changes to less than 5% of Total Investment Cost (TIC).
- Ensure that all regulations, license agreements, applicable specifications, and standards are met.
- Meet the established and agreed budget.
- Provide a positive working environment for all personnel resulting in a high level of motivation.

18.17.2 SCOPE

The scope of the project includes early works installation of access roads, bridges, avalanche controls, setting up of temporary construction camps, marshalling yards, and helicopter support for safety transportation of crews and moving of equipment, fuel and supplies to remote locations. The MTT tunnel program and water treatment facilities will start at the Mitchell site and at the Treaty Creek-Teigen Creek saddle locations. The MTT tunneling program is the critical path for the project and will require helicopter support for the first year of construction while access roads are under construction.

The Mitchell mine site consists of temporary and permanent water treatment facilities and the construction of a major water storage dam. Infrastructure buildings include a truck shop, permanent camp facility, water and sewage treatment plants and power plants. The site has two major twinned diversion tunnels with portal entrances and exits to construct.

The Treaty OPC site consists of a major mill building complete with mechanical systems for the processing of materials from Mitchell. Electrical supply to the project will be gained by a transmission line built from Highway 37 along the Treaty Creek access road to the Treaty OPC site. Infrastructure buildings include a 350-person permanent camp, an administration building, and a laboratory building.

The Treaty OPC site includes the construction of a major TMF complete with starter dams, seepage dams, tailing fill lines, and reclaim water pumping and piping systems complete with recalim barges and access roads.

As tunnel development progresses, the conveyers, fire and water lines, and high voltage and communications cabling will be installed in sections.

The commissioning program will start as soon as power is available, and the mill building and first section of conveyers are ready to check-out and pre-commission with no load on the system.

18.17.3 PROJECT SCHEDULE

A preliminary project schedule has been developed with a start date for the construction program planned for January 1, 2014 with the Frank Mackie Winter Access Road program. The Winter Access Road will continue for 2015 and 2016. Access road construction will mobilize on April 1, 2014, at the Treaty Creek and Coulter Creek locations.

Mitchell site construction begins with the development of the site access roads to the WTP area, WSD, tunnel entrances, Coulter Creek Access Road, and building locations. Early works material and equipment will mobilize on the Winter Access Road and the major equipment, general construction materials, and heavy earth moving equipment will mobilize by the Coulter Creek Access Road. The main mechanical areas for the crusher, stockpiles, and conveyers will be constructed by one General Contractor. The truck shop and permanent camp facility will be constructed as Engineering, Procurement, and Construction (EPC) projects.

The Treaty OPC site will utilize the Treaty Creek Access Road to transport all material and equipment. The process plant will be constructed by one General Contractor and the TMF will be constructed by the Earthworks Contractor.

Plant site tunnel construction will start at the Treaty-Teigen Saddle junction from two work fronts. As the project progresses, two additional work fronts will be added at an adit at km 12 and at the Treaty OPC site. Mitchell site tunnel construction will start at the MTT south entrance and meet up with the development from the north. In the early stages of the project, helicopter support will be provided to move in equipment and supply the fuel for daily use.

Conveyors, fuel piping, electrical, fire protection, and water lines will be installed in three sections which will be coordinated with the tunnel progress and access availability.

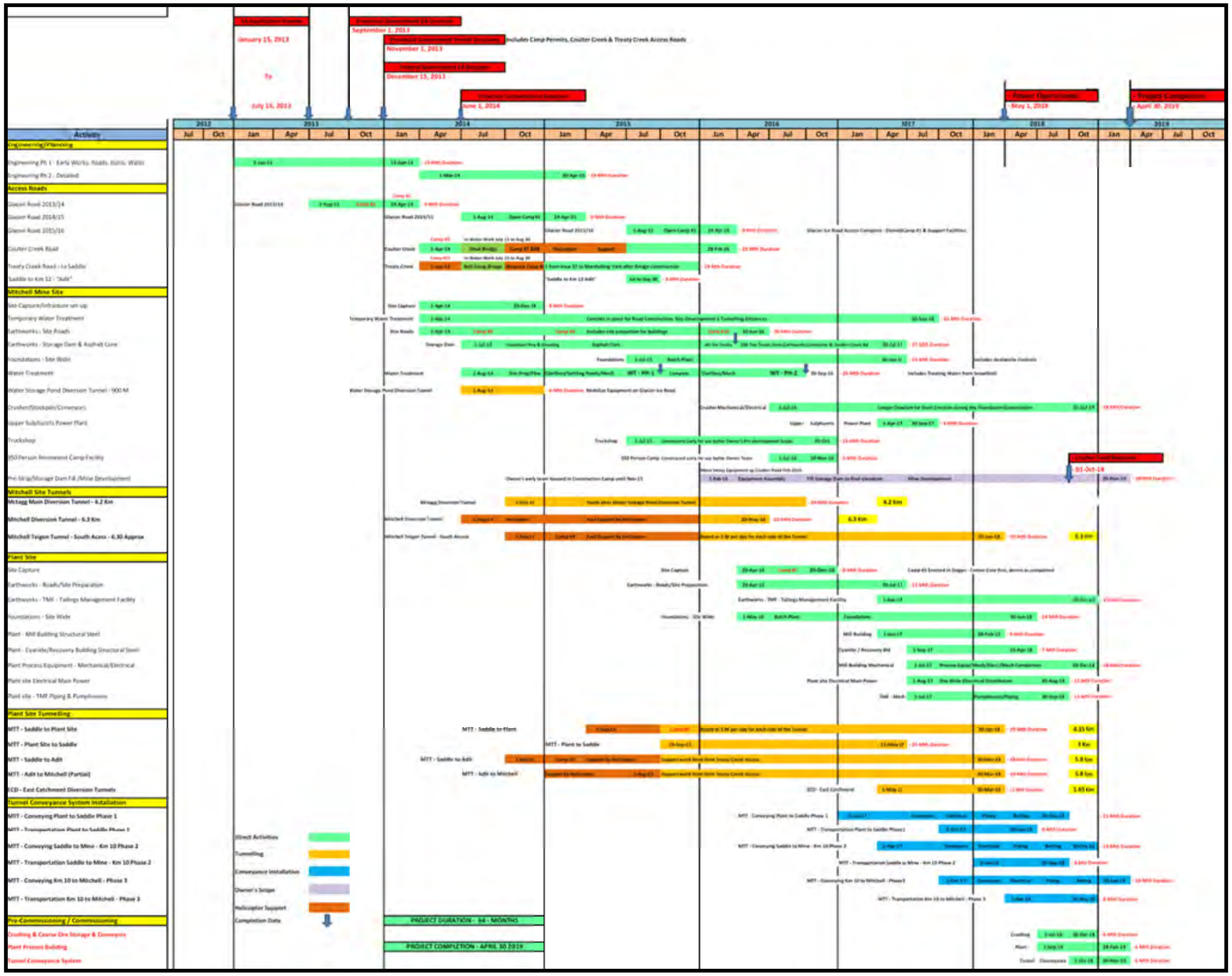
The project duration is 64 months with a planned completion date of April 30, 2019.

The high level schedule is shown in Figure 18.30. The complete schedule is available in Appendix G.

TETRA TECH WARDROP

SEABRIDGE GOLD

Figure 18.30 Construction Schedule Summary



18.17.4 Construction Management

The Construction Management team would be responsible for the management of all construction activities at the construction sites. Next to the provision of safety leadership and controls, the administration and coordination of contractors is the primary Construction Management activity. This includes monitoring and correcting, as necessary, compliance with contractual requirements for quality of work, scheduled progress, documentation, imposing high safety and environmental standards, and providing good industrial and community relations.

The Construction Management team will:

- establish site safety culture and a "safety first" environment
- provide safety orientation and training with adherence to the Health, Safety, Environment, Security, Community Plan (HSESC)
- provide safety and risk mitigation including holding workshops with contractors to review conditions and to reinforce work instructions and procedures
- monitor compliance with the Project Environmental Management Plan
- provide the leadership to accomplish compliance with the Project QA/QC Management Plan
- manage contractors at site
- attest contractors' progress payment certificates
- monitor and status construction schedules and cost data for reporting progress
- manage office services and administration for field activities
- provide project controls for construction activities and prepare regular reports
- hold kick-off and weekly meetings with each contractor and coordination meetings with all contractors
- solicit engineering support as required
- deliver the Project following the point of Pre-commissioning and Mechanical Completion but not including wet commissioning.

18.17.5 Construction Supervision and Contractor Management

The objective of all site construction activities is the timely and cost-effective completion of the Project facilities to project design and required standards. Construction management staff, while ensuring that standards are maintained, will provide all appropriate assistance to contractors in achieving this end.

The contracts management group, which falls under the Site Procurement Manager, will use the integrated project data management system to track contractor invoicing, changes, and Requests for Information (RFIs). The overall management of the construction site and includes the following construction strategy.

CONSTRUCTION STRATEGY

The project will be construction driven, with the construction management team actively engaged with planning and engineering to influence the makeup and timing of engineering deliverables, and to match fabrication and implementation schedules.

The area construction managers/leads will be mobilized early in the design phase to work closely with the engineering, procurement and construction groups to develop a detailed early works and construction plan.

The project will create and maintain a zero incident culture by providing a safe, well managed work site for all employees. Training programs and continuous management support is the main focus throughout the implementation period, resulting in no down time or lost time created by incidents.

One of the project key objectives is to maximize the offsite construction direct labour man hours, and develop detailed workface planning early for project implementation. To achieve this, the project team will set-up alignment sessions with the main general contractors to maximize the collective experience of the teams.

The project team will maximize modularization, pre-dressing of vessels, preassembly, and pipe prefabrication. Module yard and fabrication shop space will be sourced and booked early in the implementation period to confirm availability.

In order to meet the project requirements for development of the mine and plant sites, EPCM activities must commence early to support the development of safety, schedule, cost management, environmental management, emergency response, security, and community interface plans. There is also a need to mobilize the site construction management team in advance to support the early site work, set up site facilities consisting of first aid and gate entry, and meet the planned construction start date.

18.17.6 CONTRACTING PACKAGING AND STRATEGY OVERVIEW

GENERAL

The preliminary packaging strategy for the KSM mine site and Treaty plant site locations is divided into contract packages. During the contractor expression of interest, pre-qualifications phase of the project and the advancement of detailed engineering, the packages will be combined to reduce the number of final contracts and form a contracting strategy for the Project. The KSM contract packaging approach and strategy are included in Appendix G.

A portion of the packages listed in the strategy document have been combined early in the process to reduce contracts and provide various contractors with a natural flow and control of their work areas.

The project will be constructed as a managed open site, neither union nor non-union.

To maximize the available resources in the community, the project team will source and qualify local suppliers and contractors to promote business opportunities in the local area. Consideration has been provided in the strategy to assist the Owner with local involvement by dividing the packages into reasonable scopes for the local contractors to tender.

The contracting strategy document defines the type of package, contract, method of payment, engineering responsibility, purchasing of major equipment, purchasing for minor and miscellaneous equipment, installation contractor and the management of the contract package.

The contract packaging approach outlines the scope of work for each package and the strategy for managing both the Contractor and Owner supplied equipment, facilities and services. The above items will be considered carefully and included in the contract scope document write-ups during the tendering phase of the project.

The type and quantity of contracts are listed for the following areas:

- Transportation Contracts (6)
- Service Contracts (Site Support) (8)
- Temporary Infrastructure Buildings EPCM (4)
- Construction Installation Contracts Mine Site (11)
- Permanent Buildings Mine Site (2)
- Construction Installation Contracts Plant Site (5)
- Permanent Buildings Plant Site (3)
- BOO – Build, Own, and Operate (1)
- Total Packages – 40.

The methods of payment for contracts are as follows:

- lump sum – construction installation contracts
- lump sum and unit rates – earthworks and foundations.
- time and material (T&M) cost reimbursable – service contracts.

The majority of the contracts on site will be tendered on a lump sum basis. The level of engineering completion for the tendering process is anticipated at 85% to 90% complete and should be close to 100% complete before the final negotiations and award is complete. This approach will provide the best possible pricing for the Project. Areas that are questionable in a lump sum contract will be adjusted by pre-approved unit rates established during the tendering period.

This approach will minimize the package risk and keep the Project on target for lump sum pricing.

18.17.7 SITE ORGANIZATION

The EPCM site organization has been developed to provide a balanced combination of senior managers, area managers, engineers, superintendents and discipline specialists to provide the Owner and contractors continuous support during the installation period. Organization charts are available in Appendix G.

The site organization and staffing plan addresses the following construction management area:

- site wide health, safety, and environmental (HSE) program
- staff turnaround coverage
- fast track construction program
- office administration and project controls
- field engineering support
- commissioning and start-up planning
- close coordination with all stakeholders.

The organization chart provides for all site activities that are to be managed from the site construction office. The two major work sites will have a dedicated safety lead and safety specialist to assist the contractors with the daily issues, training requirements. They will also provide the required reporting and continuous planning

Site specific orientations presented by the safety team will be available at both construction sites. A short visitor/truck driver orientation will also be provided.

The mine site and plant site will have a senior construction manager directly responsible for the management of cost control, scheduling, engineering, material coordination, quality assurance support, and pre-commissioning planning. He will be directly supported by area managers and area leads.

Each construction and service contract will have a package leader assigned as a single point contact during the construction period.

A materials management team will coordinate with the contractors on receiving and moving materials to the contractor's lay down areas and the worksites.

Roles and Responsibilities

To ensure the Project Team have a clear understanding of the management structure and the roles and responsibilities for the major positions on the team, job descriptions will be developed and presented as a general guide for the position.

Levels of Authority

The Delegation of Authority Guideline (DOAG) must be established by the Owner as early as possible, which will lay out the authority level for each senior position on the project for both the Owner and the EPCM group. It will be implemented and managed by EPCM personnel. It will facilitate the efficient and expeditious execution of the project scope of work.

The DOAG applies to all transactions executed on the project, initiated by the Owner or the EPCM group. The Managers with authority to execute transactions may only approve those within their area and level of responsibility.

A basic premise of approval delegation is that the delegated authority bears with it the obligation to exercise sound judgment. Consequently, approval indicates that goods and services have been received, prices are correct, tax, legal and withholding requirements have been satisfied, the project's interests are protected, and proper documentation exists to justify transactions.

The DOAG can only be amended by the Owner.

18.17.8 Construction Infrastructure

The construction infrastructure includes the setup of all facilities and equipment required to support the construction effort in an efficient manner to achieve the project's goals and timelines in the project schedule. The establishment of the construction infrastructure, for the most part, is set up during the Early Works Scope and managed by the EPCM Contractor. The operation of the construction facilities is provided by the site services contractor. The packaging strategy in Appendix G includes the scope of work for service providers.

A detailed plan will be developed for all camp locations, major work sites, access road construction and the Tunnelling execution program. Construction drawings and plans are available in Appendix G.

The EPCM contractor will provide a Site Services Superintendent to coordinate the project requirements with contractors and service providers to ensure that there are no delays to the KMS construction program.

Contracts and amendments required for service providers for operating and maintaining the site will be prepared and issued by the EPCM contractor. The EPCM construction management staff will be responsible for each contract to ensure that the contractor meets the performance standards specified. The service contracts that will be implemented are shown below.

Construction site infrastructure plot plan has been developed to define the location of EPCM, Owner and Contractor supplied facilities and services.

18.17.9 *Field Engineering*

Field Engineering Scope

The field engineering group consists of a field engineering manager, field area engineers, site document control staff, and Q&A staff. The group will be responsible to the Project Engineering Manager on technical and procedural issues, and to the Site Construction Manager regarding work flow.

Project engineering will be handled by the home office engineering department. Field engineering will be directly involved with the contractors in issuing drawings and technical documents, processing RFIs, site technical document control, site surveys, and Q& A. More specifically, field engineering will be responsible for:

- all decisions regarding field engineering
- coordinating all engineering requirements between site and home office
- coordinating site surveys
- site document control including the issuing of drawings and documents to contractors
- managing the RFI process
- early reconciliation of defects
- assist in determining progress
- for record documentation
- planning and supervising all QA/QC programs and non-destructive testing (NDT).

18.17.10 *Quality Assurance/Quality Control Systems*

The Discipline Field Engineer, in conjunction with the QA Manager, will review the Contractor's Quality Management System prior to start of works. The subsequent monitoring of the Contractor's implementation of their QA/QC systems on Site will be through the Construction Management Team.

The Discipline Field Engineer will establish and implement, in coordination with the QA/QC Manager Independent Surveys, Field Inspections, and QC laboratory sampling and testing as necessary through the Survey Contractor and QC Laboratory Contractor.

18.17.11 Health, Safety, Environment, Security, and Community

The following is a brief overview of the HSESC construction management plan. A detailed project specific HSESC plan, HSESC manual, and standard operating procedures well be finalized during the project front end planning.

Health, safety, environment and community are of high importance in the engineering, design, construction and commissioning of the project. In order to achieve zero incidents, total commitment is required from all project personnel (Owner, EPCM, Contractor's and Vendors) to remove all conditions that could lead to injury.

This project will conform to national and industry standards regarding health and safety, as well as to the Owner's and EPCM Manager's policies and procedures. A copy of these procedures will be issued to each contractor before mobilization to site.

The Project Manager and the HSESC Manager will lead in the development and implementation of the site-specific HSESC Plan. All workers (Owner's, EPCM personnel and contractor employees) are responsible for performing their work in a manner consistent with legislation, industry standards, and company policies, practices and procedures.

Health

Facilities and Infrastructure

The KSM Project has planned for two medical facilities: one at the mine site, and the second at the Treaty plant site. Facilities will have helicopter support, emergency response teams available, and a 'Doctor on Call' program to support emergencies.

The personnel and medical facilities required on site during construction are:

- A manager appointed to take charge of first aid/ medical arrangements.
- A qualified medical practitioner to act as a medical adviser.
- Ready access to a suitably qualified person to provide medical treatment.
- One trained and certified first aid person per 50 employees on every shift, or the provision of full-time emergency services or paramedic support on site. More first aid personnel may be required in underground or remote locations.

- Suitably stocked first-aid boxes (or equivalent) in readily accessible locations.
- For permanent facilities, a first aid or 'sick' room that provides privacy for injured or sick employees during their wait for medical treatment or recuperation.
- An emergency vehicle, suitable for conveying injured or sick persons to a local treatment centre or 'pick-up point', where a local ambulance service is deemed adequate.
- Basic diagnostic capabilities for local diseases, where the site is 'established' and remote.

The site medical and first aid treatment system must be integrated into the site emergency procedures and safety reporting system and must conform to Occupational Health and Safety regulations for construction.

All site personnel will be informed of the first aid/medical arrangements and the protocol for activating the emergency procedure. Notices indicating contact details for first aid personnel (or appointed persons), the emergency contact number/radio frequency, and the location of the first aid boxes must be posted around the site. Special arrangements may be required to give first aid information to employees with reading or language difficulties.

SAFETY

Orientations

All new employees to the construction site will attend the Site Safety Orientation immediately upon arrival at site. Visitors are required to be escorted by a Safety-Oriented Worker at all times until they have completed the Site Safety Orientation.

In addition to the project orientation, Contractors are required to provide job specific safety orientation to all new employees prior to the start of work.

Documented ongoing pre-task job hazard analyses with crews for specific tasks will be conducted at the beginning of each shift.

Contractors must ensure that their workers are suitably trained and competent in the safe work procedures and health and safety regulations pertaining to their duties. Likewise, any equipment operators are required to have equipment-specific training and certification.

Safety Meetings

A weekly safety meeting will be required by every contractor (including the EPCM contractor and Owner) on site. It will provide an opportunity for all personnel to contribute timely information on safety items that relate to project activities. Weekly safety meetings are conducted by contractor management and provide an important communication link between all their respective crews.

Minutes of these meetings are recorded on the weekly Safety Meeting Form. Contractors are required to immediately address as many issues as possible. Issues from the weekly safety meetings that cannot be resolved immediately are to be submitted to the project safety department and transferred to the safety meeting action log for the Superintendents review and action.

Audits and Inspections

The Joint Health and Safety Committee (JHSC) will inspect a minimum of one area of the project each week. The JHSC will develop the inspection schedule and use an action plan format. When completed, the inspection will be submitted to the EPCM Safety Manager for assignment of responsibility and completion date.

Emergency Response

An emergency response team will be assembled from the site personnel. It will be organized and led by the EPCM HSESC team. Personnel will receive formal training in:

- first aid
- fire fighting
- rescue techniques
- hazardous material handling and clean up.

The team will be provided with the following emergency equipment:

- protective gear for firefighting and hazardous material handling
- fully equipped rescue vehicle
- ambulance
- fire truck
- dedicated communication devices (hand-held and vehicle mounted)
- tools (e.g.: axes, shovels, cutters, saws, etc.).

SECURITY

Site security will be handled by a security contractor reporting directly to the Owner. The contractor will develop a site-specific Security Plan.

The main components of the Site Security Plan should contain the following topics:

- site access control and surveillance plans
- identification tag control
- traffic control and enforcement
- criminal activities (liaison with police authorities).

ENVIRONMENTAL

Scope

A site-specific Environmental Management Plan (EMP) will be produced to guide the mitigation and management of environmental impacts arising from project activities. The purpose of the EMP will be to:

- document environmental concerns
- outline appropriate protection measures
- provide specific instructions for protecting the environment and minimizing environmental effects for achievement of zero incidents.

Spill Prevention

Site personnel will be educated in Spill Prevention Controls. Environmental site inspection activities will be documented. In addition to written reports, photographs will be used to document environmental compliance.

In the case where an environmental incident occurs, an Incident Report will be completed and actioned.

Archaeological Finds

Historical resources include archaeological, historical, and paleontological artefacts. Archaeological awareness will be part of the site orientation program. If archaeological artifacts are discovered during construction activities, the following steps that must be taken:

- All construction activity in the area is to be immediately stopped.
- The Owner's site Construction Manager is to be notified.
- The applicable government agency is to be notified.
- Construction activity in the area will not resume until the area has been investigated and cleared by the government agency involved.

COMMUNITY

According to the current plan, relations with the local communities will be the responsibility of the Owner. It will still be the responsibility of the EPCM Contractor to make the Owner aware of any situation that is happening or arising that may affect the communities.

This may include, but is not limited to, items such as:

- project schedule (current and upcoming activities)
- project influence on local community (negative and positive)
- possible employment opportunities
- traffic and/or road conditions
- personnel safety and site security
- environmental performance
- complaints arising from construction activities.

18.17.12 PRE-COMMISSIONING/COMMISSIONING

OVERVIEW

The commissioning period starts with the introduction of fresh feed and ends when the plant has reached full or a pre-determined percentage of design production. The Owner is responsible for both the plant and the necessary operations and maintenance activities during the commissioning period. Support will be available from the EPCM contractor and other contractors to assist in situations or conditions that may be atypical of normal operation.

GENERAL

The Commissioning Program is subdivided into three phases of mechanical completion by the Contractor.

This first phase of mechanical completion is without ore or principal product. Project discipline lead engineers and coordinators are responsible for the work conducted up to and including energization of equipment including:

- Equipment inspection and sign off to verify that all installation work is in accordance with the drawings, specifications and manufacturers operations and maintenance manuals.
- Green tags are complete and signed off by the Construction Manager and Owner's representative.

No-load Testing

This second phase of mechanical completion proves that the facility can operate in a controllable and stable manner without feed (i.e. water only test). It commences when the:

- first phase mechanical completion is complete
- piping installations are complete
- electrical/instrumentation drives are energized and field device signals tested to and from the PLC systems
- mechanical, piping and electrical areas are all complete and signed off by the construction manager and Owner's representative.

Process Commissioning – Ore Feed Commissioning

The third phase of mechanical completion (ore feed commissioning) commences when the water-commissioning activities are completed.

The facility will be considered complete and available to hand over to operations when the crushing and conveying system can deliver and operate according to design performance criteria.

The process commissioning stage is the responsibility of the Owner and shall be performed by the Owner's commissioning team aided by the construction manager, commissioning manager and contractor support personnel.

OBJECTIVE

The objective of the Commissioning Program is to take the project facilities and equipment from the completion of construction (mechanical completion) through to a fully operational facility:

- in a safe manner
- in the minimum time
- in a cost effective manner.

This is achieved by the following activities:

- Inspection and testing that the equipment and facilities as supplied, erected and installed, meet the design and performance criteria for the intended duty in a safe and environmentally acceptable manner.
- Testing and adjusting the operation and control of all components of the project facilities to ensure that the equipment operates as part of an integrated and fully coordinated system.

- Integrate operations and maintenance personnel into the commissioning process as early as possible to assist in the final check out and start up of the facilities.
- The commissioning phase will include the demobilization of temporary construction infrastructure.

Tagging Systems will be defined for turnover of individual equipment, systems, buildings and the complete facility. A detailed deficiency list system will be developed to ensure the turnovers to the operations team are complete.

18.18 OWNER'S IMPLEMENTATION PLAN

The KSM project will be constructed as outlined in Section 18.17 and in the time frames indicated in the Project Schedule in Appendix G. It will be the responsibility of the Owner to attain the environmental and operating permits allowing mine site access road development, all project construction, and all plant operations/ancillaries needed for the KSM Project.

The Owner will manage the engineering work required to prepare design documents for reviews by the Province of British Columbia and the Government of Canada. The Owner will also manage any on site drilling, hydrology, and geotechnical work as well as the engineering work needed to prepare a final Feasibility Study for the KSM Project.

During site construction, the Owner will be responsible for:

- community relations
- governmental affairs
- special environmental work/reporting
- site security
- recruitment
- training of operating personnel for the Mitchell pre-mining phases
- all personnel needed for ultimate mine and plant operations.

The Owner will recruit and train early administrative staff for the above areas to work in the following locations:

- Vancouver – initial engineering support, Human Resources (HR), product sales, legal, accounting and enterprise IT systems development
- Smithers office – governmental affairs, environmental management, KSM security management, HR for employee recruitment, and First Nations and local community relations

- Terrace office – employee task and safety training programs, purchasing, accounting and transportation/logistical support
- Stewart Port Site – management of deliveries and security for incoming construction equipment/materials and outgoing concentrate shipments
- Treaty OPC Operations – operations, maintenance, security, and administrative personnel
- Mitchell Mine Site and Water Management – operations, maintenance, security and warehousing personnel.

Some of the early hires for the Owner's staffing can be used within the Construction Organization, especially in the areas of Security, Environmental Reporting, Safety, Purchasing and Logistics.

The time sequences for these Owner activities are outlined in Table 18.14.

Table 18.14 Owner's Activities by Year

Year	Activities
2012	• Complete Prefeasibility Study Update by July 1, 2012. • Prepare all supporting documents for EA submission by November 1, 2012. • Complete reviews by Technical Specialists and complete a KSM Risk Assessment session and report for the EA submission during September. • Complete 2012 planned KSM drilling programs and other fieldwork for exploration evaluations, access roads, tunnel geotechnical, Mitchell Glacier, WSD and TMF.
2013	• Initiate drilling program for exploration and metallurgical samples. • Initiate metallurgical program on annual ore composites representing the first 10 years of mined production from Mitchell and Sulphurets pits. • Initiate Detailed Engineering work for Phase 1 of construction, roads, camps, etc. • Initiate the required Feasibility level drilling and hydrological Field Program for further definition of the Mitchell and Sulphurets open pit geotechnical and hydro-geological programs. • Initiate Work on final Feasibility Study for KSM Project. • Initiate concentrate smelter contract negotiations . • Recruit and fill manager level positions for overall operations, mining, processing and geology. • Add KSM staff for environmental, permitting and community affairs.
2014	• Attain Provincial Environmental Permits by first quarter of 2014. • Complete metallurgical program on annual ore composites representing the first 10 years of mined production from the Mitchell and Sulphurets pits. • Finalize concentrate smelting contract terms. • Complete final Feasibility Study for KSM Project. • Finalize Detailed Engineering work for Phase 1 construction activities. • Initiate Detailed Design of Phase II engineering work and commit to large equipment purchases.

table continues...

Year	Activities
2014 (con't)	• Obtain concurrent permitting for use of Winter Access Road during Winter 2014. • Obtain Federal Permits so the in-water bridge piers on the Bell-Irving and Unuk rivers can be installed during July of 2014. • Continue drilling program for Mitchell and Sulphurets open pit geotechnical and hydro-geological definition and for delineation drilling of "Inferred Resources" within the early pit stages. • Add KSM staffing to direct the Construction Safety, Environmental and Security areas for the access road construction and the helicopter supported remote camps, water treatment, and tunnel construction efforts. • Recruit Senior KSM Staff to participate in the detailed design and equipment procurement/deliveries/logistics for the mine, TMF, and process facilities. • Expand the recruitment and training capabilities for future KSM employees at Smithers and Terrace Offices.
2015	• Continue with on-site exploration drilling for better delineation of "Inferred" category resources in later pit stages at Mitchell and development of any other required hydro-geological wells. • Complete final detailed design for Phase II engineering work and finalize all equipment purchases. • Initiate enterprise computer systems work in Vancouver. • Recruit three Senior Corporate Managers for Vancouver KSM Office to work on product sales, governmental/community relations, environmental permitting, legal assistance, and accounting. • Add staff at Terrace office for construction logistics and KSM employee recruiting/training.
2016	• Add Managers and some staff for site activities in mining, administration, and processing areas in new offices constructed at Mitchell mine and Treaty OPC. • Get enterprise computer systems work started at Terrace and at the on-site Treaty OPC offices. • Continue recruitment and training of future KSM employees and use them in construction logistics areas, security and for on-site, construction and equipment/maintenance warehousing. • Add four personnel at the Stewart Port Facilities location for security and management of incoming KSM shipments and, ultimately, for the outgoing concentrate shipments.
2017	• Recruit 5-10 laboratory staff and Manager for newly completed on-site laboratory at Treaty OPC. • Recruit and train 75-100 operating and maintenance staff for the first phase of mine equipment start-up for construction at WSD, and pre-stripping activities at Mitchell and Sulphurets. • Recruit and train eight personnel for operation of the WTP. • Recruit superintendent-level employees for the Mitchell mine, Treaty OPC, and site administration offices. • Continue on-site development of enterprise computer systems. • Recruit and train senior warehousing and purchasing personnel for Mitchell maintenance facility and Treaty OPC. • Initiate water storage operations and control in the TMF.

table continues...

Year	Activities
2018	• Continue recruitment and training of 50-100 employees for administration, warehousing, and Treaty OPC operations. • Recruit and train approximately 150 employees for the Mitchell and Sulphurets mine pre-stripping and ore mining/stockpiling work. • Recruit and employ the engineering staff for the Mitchell mine office and Treaty OPC. • Recruit and train the staff for all site services and tunnel operations. • Complete the on-site development of enterprise computer systems. • Conduct off-site and on-site training sessions of employees for plant operations and TMF operations. • Continue water storage and train KSM employees in the TMF water pumping and management systems.
2019	• Complete the recruitment and training of all KSM operations personnel (total of 650 employees). • Initiate full scale mining and primary crusher operations at Mitchell from first phase mine development stages at Mitchell and Sulphurets. • Start ore processing from the primary crushers at Mitchell through the MTT and fill the Treaty OPC crushed ore stockpiles. • Treaty OPC plant equipment will undergo commissioning and start-up in early 2019. • Produce first copper concentrate and doré at the Treaty OPC in early 2019. • Initiate sand tailing production at the TMF and train employees on procedures for this long term dam construction effort. • Shipment of copper concentrate from the Port of Stewart will be initiated by mid-2019.

19.0 MARKET STUDIES AND CONTRACTS

Butterfield was engaged by Seabridge to conduct studies on saleability and smelting terms for the copper and molybdenum concentrates expected to be produced at KSM. The first study was conducted in March 2010 and included the saleability studies and smelter terms for the two concentrates. The second study, in May 2012, updated the smelter terms for the copper-gold concentrate. This section was taken directly from the May 2012 study, with only minor revisions made for consistency. The complete May 2012 study is included in Appendix B.

Dollar amounts in this section are all expressed in 2012 US dollars.

19.1 COPPER CONCENTRATE

19.1.1 *MARKETABILITY*

East Asian custom smelters will be more attractive to Seabridge than the domestic or US smelters because of the proximity of the KSM property to the Pacific Ocean and because some of the Asian smelters pay for a higher percentage of gold than do the Canadian smelters. For these reasons, the copper mines in BC that are close to the Pacific Ocean almost always receive a higher FOB mine return when shipping their concentrates to Asian markets rather than to domestic or US smelters. Accordingly, the reports in March 2010 and May 2012 examine only the Asian custom smelting markets.

There are five East Asian countries that have custom copper smelters and to which Seabridge might wish to consider selling their concentrates. These countries are Japan, South Korea, India, China, and the Philippines. All of them operate efficient smelters and all or most of them would probably be willing to conclude a term contract with Seabridge shaped to meet the needs of a large new mine.

Of the above five countries, probably the best targets for Seabridge are Japan and Korea. They are closer to Canada west coast ports than the smelters in India, China, and the Philippines. Since the mine pays the freight to the smelter port, this would be a direct saving. Also the receiving ports in India and China are sometimes congested. In addition, a greater number of bulk ore carriers move to Japanese and Korean ports than to the Indian, Chinese, or Philippine ports so that they offer better flexibility.

Moreover, since 2006 the mines have been unable to produce as much concentrate as the custom smelters have wanted to receive. The underlying reasons are that many mines are ageing so their remaining reserves are lower in grade than before, strikes, environmental restrictions, governmental regulations and delays in start-ups. These difficulties seem unlikely to go away as time proceeds. It suggests that concentrate may remain in short supply.

19.1.2 Smelter Terms

The smelting terms to apply are generally negotiated half-yearly and can vary substantially from one period to the next. Sometimes a smelter buyer and a concentrate seller will conclude a contract of duration two to five years with the understanding that the terms to apply for each half-year period will be the Asian Benchmark terms.

Asian Benchmark terms for the past three years have been very sharp with the numbers averaging $57.57/dmt for the treatment charge and 5.63 cents per pound of copper paid for the refining charge. These low terms may not have been possible had the price for sulphuric acid not been at record levels for part of the period. Accordingly the future terms may be a little higher and using $75/dmt of concentrate for the treatment charge and 7.5 cents per pound of copper for the refining charge is recommended.

Over the past 40 years, copper smelters have become significantly more efficient. It has led to smelters almost everywhere reducing their treatment and refining charges.

For copper concentrate delivered CIF (cost, insurance, and freight) to smelter ports in Japan or Korea the terms may be:

- Receive payment for:
  - The agreed copper content less 1.0 unit at the London Metal Exchange price for Grade A copper and less a refining charge of 7.5 cents per pound.
  - The agreed gold content to be paid for under the schedule shown in Table 19.1. The price is to be the London Fixing price for gold less a refining charge of $8 per troy oz gold.
  - Silver, if over 30 g/t is present, receive payment for 90% of the agreed content at the London Settlement price less a refining charge of 50 cents per troy oz silver.

Table 19.1 Gold Payment Schedule

Gold Content (g/t)	Payment Schedule (%)
<1	No Payment
1 to 3	90
3 to 5	93
5to 7	95
7 to 10	96.5
10 to 20	97
20 to 30	97.5
Over 30	97.75

From these three payments deduct a treatment charge of $75 per dmt. Also, deduct penalties, if applicable under the following schedule:

- arsenic: $3.00 per 0.1% over 0.2%
- antimony $3.00 per 0.1% over 0.1%
- bismuth $1.00 for each 0.1% over 0.05%
- zinc $1.50 per 1.0% over 3.0%
- lead $1.50 per 1% over 1%
- nickel plus cobalt $0.30 per 0.1% over 0.5% combined
- chlorine $0.50 per 0.01% over 0.03%
- fluorine $0.125 per 10 ppm over 0.002%
- mercury $0.20 per 10 ppm over 5 ppm
- A1203 $5.00 per 1.0% over 3%.

In addition:

- The price of sulphuric acid can have a profound effect on smelting economics since smelting one tonne of copper concentrate generates approximately one tonne of sulphuric acid. The acid price varied substantially in recent years. The largest demand is for the production of fertilizers, for which demand is expected to grow as world population increases and the demand for food grows. Accordingly, the acid production may be more rewarding for the smelters as time proceeds, helping to smelter charges low.
- Certain costs that a smelter must pay are rising, particularly the cost of oil and electric power and neither of these trends seem likely to reverse. Also, there is general inflation which, in the US, was 0.96% in 2010 and 1.1% in 2011 for a total of 2.06% over two years. Because of their dependence on

oil and electric power the direct costs of smelting copper concentrate will increase more than the GDP deflator index suggests and 5% annually for the next 5 years may be a reasonable assumption.

These factors are expected to offset each other.

A concern for the receiving smelters may be relatively low copper content of the KSM concentrate. They generally like to see the copper content at higher than 25%.

19.2 MOLYBDENITE CONCENTRATE

19.2.1 MARKETABILITY

There are several large and medium-sized roasters which might be candidates to toll or purchase the KSM molybdenum concentrate. The largest single roasting installation appears to be owned by Molymet in Chile, which is jointly owned by several of the copper producers there. The Jinduicheng roaster in China is known to be large and may have to import concentrate to satisfy the rapidly growing local demand. In the US, the Langeloth roasters near Pittsburgh have a capacity of 40 million pounds molybdenum content, and the Phelps Dodge roasters at Fort Madison, Iowa, have a reported annual capacity of 38 million pounds. The capacity of the roasters near Rotterdam, Holland (originally built by Climax Molybdenum) is some 20 million pounds molybdenum annually. In Belgium, the SADACI roaster, now owned by the Chilean Molymet group, has been expanded to a capacity of more than 20 million pounds annually. Locally, the roaster at the Endako Mine continues to operate. Several of these roasters may be interested in the KSM material. Butterfield recommended looking for toll roasting space, or a roaster that is ready to buy its molybdenite concentrate with only an expected annual production of only 2,000 tonnes (dry).

The significant level of rhenium expected in the KSM concentrate complicates the marketing. The price of refined rhenium changed significantly. The contained rhenium may receive significant credits if the rhenium can be recovered. Clearly, attempting to find a roaster that can recover rhenium and is prepared to pay for it is important.

19.2.2 Smelter Terms

The expected terms for KSM molybdenite concentrate may be:

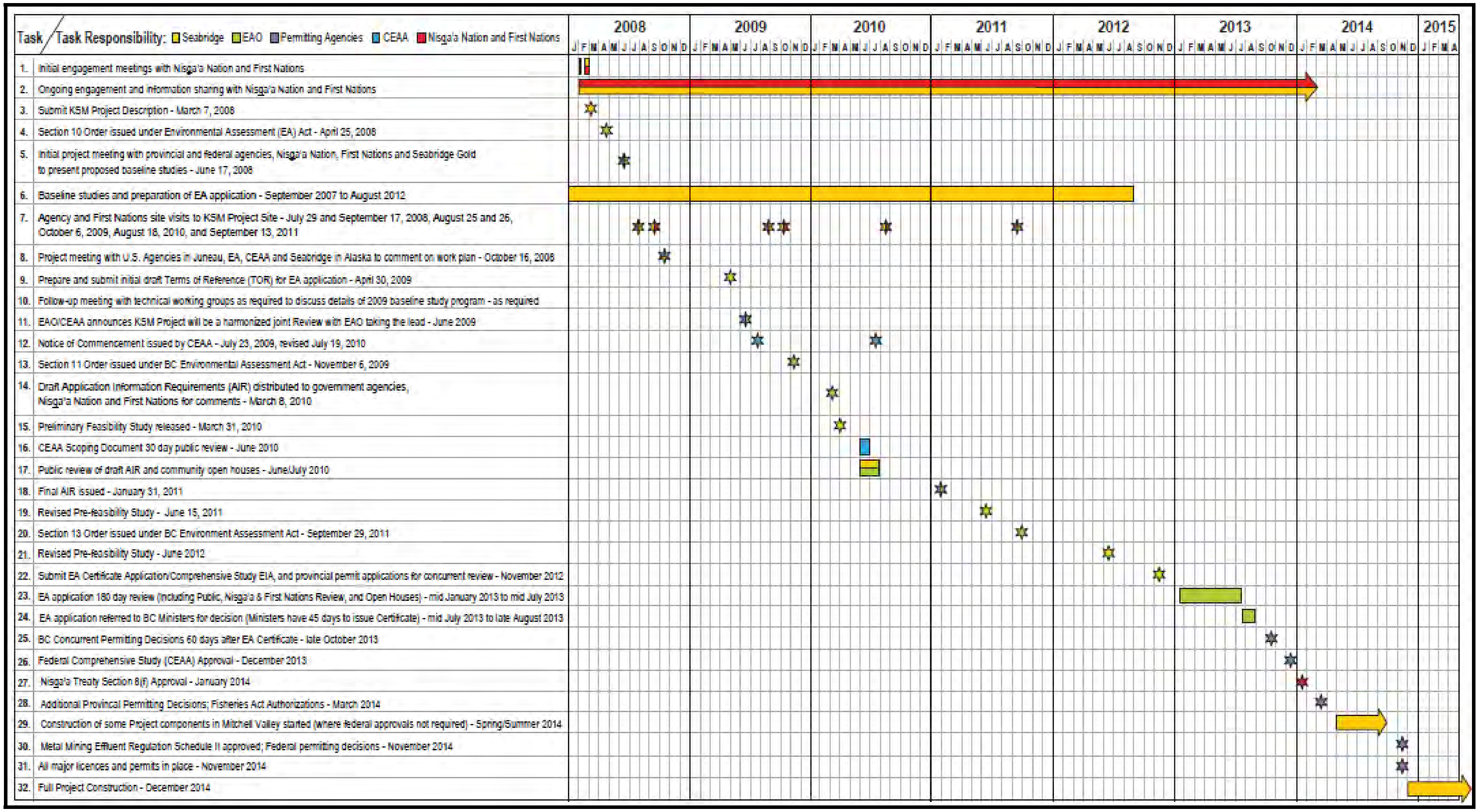
- Receive payment for 99.0% of the agreed molybdenum content at the London Metal Exchange price less a treatment charge of $2.00 per pound.
- From the sum of these payments deduct a penalty if the copper content exceeds 0.1%. The penalty may be 30 cents per pound of the agreed molybdenum content for each 0.1% excess.
- Payment terms are negotiable, but would be on average 20 days after delivery of the concentrate to a major roaster or to a port close to a major roaster.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 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 20.1 outlines the best-case scenario for an approval schedule for the Project up to the start of construction.

Figure 20.1 Regulatory Review and Approval Schedule



20.1.1 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 BCEAA to obtain an Environmental Assessment Certificate from the Ministers of Environment and Forests, Lands and Natural Resource Operations before construction or operation permits are issued. Seabridge was advised by the BCEAO in April 2008 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.

On November 6, 2009, the BCEAO issued a Section 11 Order to establish the scope, procedures, and methods for the environmental assessment. On January 31, 2011, the BCEAO issued the Application Information Requirements that detail the data and interpretation that are required for the Environmental Assessment Certificate Application. . On September 29, 2011, the BCEAO issued a Section 13 Order amending the Section 11 Order to clarify the use of access roads and Highway 37 for the purpose of the mine, and to clarify consultation requirements with Aboriginal groups.

20.1.2 Canadian Environmental Assessment Act Process

The Canadian Environmental Assessment Agency (CEA Agency) formally advised Seabridge on July 23, 2009, that the KSM Project will require an environmental assessment under the *Canadian Environmental Assessment Act* (CEAA). 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 Government of Canada 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 CEAA).

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 to be comprehensive due to the complexity of government regulatory processes, which evolve over time, and due to the large number of minor permits, licences, approvals, consents and authorizations, and potential amendments that will be required throughout the life of the mine. However, the major and significant permitting processes understood to be required for the Project are identified.

20.1.3 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, Seabridge is proceeding to seek concurrent approvals under the Environmental Assessment Act Concurrent Approval Regulation (CA Regulation). The CA Regulation allows eligible provincial permit applications to be reviewed concurrently with the EA Application. Section 15.4 of the Section 11 Order for the KSM Project identifies the deadline for submitting concurrent permit applications. Seabridge must apply and submit concurrent permit application at the same time as the EA Application. However, no statutory permit approvals may be issued before an Environmental Assessment Certificate is obtained. Section 8(3) of the CA Regulation requires permit decisions be made within 60 days of the issuance of the EA Certificate (i.e. issue the eligible approval or provide reasons for the refusal, or specify a later date, on which Seabridge will be given a decision on the application and provide reasons for the delay. Statutory permit approval processes are normally more specific than those required for 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.

Seabridge is working with the Ministry of Forests, Lands and Natural Resource Operations and statutory agencies to define the list of permits that will comprise the concurrent permit application package.

Table 20.1 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 20.1 List of BC 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 (Mine Site – 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 and Operate Tunnels	*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*
Mining Lease	*Mineral Tenure 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 (Coulter Creek Access Road); Construction of Treaty Creek Access 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*
Surface Lease – Mine Site Facilities	*Land Act*
Pipeline Permit – Diesel Pipeline	*Pipeline 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)*

20.1.4 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 Canada under the *Fisheries Act*. Approvals for some 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, will require a Schedule 2 amendment because the area proposed for the TMF contains a fish habitat. A licence will likely be required under the *International River Improvements Regulations* because the water storage facility is expected to alter seasonal flows to the Unuk River above threshold criteria of 0.3 m^3/s. Other federal requirements, such as those with respect to radio communication and aviation matters, will need licences. Table 20.2 lists some of the federal approvals required.

Table 20.2 List of Federal Approvals and Licences Required to Develop the KSM Project

Federal Government Approvals & Licences	Enabling Legislation
CEA Agency Approval	CEAA
Metal Mining Effluent Regulations (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*

20.2 Environmental Setting

The KSM Project is located in a relatively undeveloped 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.

20.2.1 Baseline Studies

Baseline studies for the KSM Project have been conducted annually since 2008 following issuance of the Section 10 Order from the BCEAO. Some preliminary water quality, meteorology, and hydrology data were collected in 2007 and limited additional information was collected by previous operators.

Baseline studies have included:

- **Climate** – 12 tipping bucket rain gauges, 10 snow courses, 5 automated weather stations monitored over periods up to 4.5 years, and evaporation pans to collect::
  - temperature
  - relative humidity
  - barometric pressure
  - wind speed and direction
  - solar radiation
  - precipitation at various elevations to estimate precipitation gradient with elevation
  - snow depth and snow water equivalent
  - visibility.
- **Air quality** – 5 dustfall stations operating for 4 years, and 9 operating for 3 years at various project area locations.
- **Hydrology** – monthly stream flow monitoring at 20 sites over periods of 3 months up to 4.5 years. A glacial monitoring program has been operational for over 4 years and includes glacier delineation, mass balance determination, and glacier dynamics monitoring.
- **Groundwater** – from 2009 to 2010, 28 monitoring wells were installed and monitored for seasonal and long term water levels, tested for overburden and bedrock hydraulic conductivities with packer and slug tests, and sampled for groundwater chemistry. In addition, 14 geotechnical holes drilled by KCB and BGC were also monitored and slug tested, 11 of which are vibrating wire piezometers.
- **Geochemistry** – 1,821 static tests (ABA and elemental analysis) and 44 kinetic tests on potential ore and waste rock, 18 field kinetic barrels containing potential ore and waste rock, 223 ABA tests of non-deposit material, and comprehensive static (ABA, elemental analysis, particle size distribution) and 17 kinetic (aging, humidity cells, subaqueous column) testing of a range of tailing blends.
- **Aquatics** – testing of 28 stream and river sites and four lakes to assess sediment quality (physical, organics, metals, nutrients) and primary and secondary producers over two years.
- **Surface Water Quality** - testing of water from 48 stream sites on a weekly, monthly, quarterly or annual basis over periods of up to five years for general parameters, major anions, nutrients, cyanides, total organic carbon and total and dissolved metals. Four lakes were sampled annually over two years for the same parameters. Natural surface water from four to six stream sites was sampled during freshet and low flow periods and tested for toxicity to algae, plants and invertebrates.

- **Fisheries** – comprehensive assessment of fish (species, population, spawning, size, condition, age, growth, fecundity, diet, genetics, tissue metal content, etc.) and fish habitat within the footprint of proposed facilities and access roads and downstream environments over four years. Potential habitat compensation sites were assessed, including aspects associated with ecosystems, wildlife and wetlands.
- **Soils and terrain** – terrain mapping, slope analysis, soil mapping and analysis for pH, total organic carbon and metal content.
- **Vegetation** – fieldwork over three summers mapped ecosystems and vegetation in and adjacent to the project footprint and predictive and terrestrial ecosystem mapping was completed. One hundred plant tissue samples were analyzed for metal content.
- **Wildlife** – specific surveys for moose (winter), mountain goats (summer and winter), grizzly bear (spring/summer and fall DNA studies), furbearers, hoary marmot colonies, ground squirrel colonies, bats, birds (raptors, terrestrial breeding, water dependent) and amphibians; as well as habitat suitability mapping for moose, mountain goats, grizzly bears, American marten, and hoary marmot.
- **Wetlands** – ground and aerial surveys over two years to classify wetland extent and function in and near the footprint of the proposed facilities, and assessment of wetland hydrology.
- **Archaeology** – archaeological impact assessment conducted over four field seasons in areas of proposed facility footprints, access roads, transmission lines and infrastructure, fish compensation areas, and drill and camp sites.

20.2.2 Terrain and Soils

The KSM Project is located in a very rugged area with elevations ranging from about 220 m at the Sulphurets Creek - Unuk River confluence to over 1900 m at the top of the ridge above the Kerr deposit. The valleys are typical of glaciated, or formally glaciated, valleys of the BC Cordillera, where gentle upper slopes drop into steeper valley walls that grade into broad and gently sloping valley floors. Glaciers and ice fields surround the mineral deposits to the north, south, and east.

20.2.3 Geohazards

A geohazard and risk assessment was completed for proposed facilities within the KSM project area. As expected for a mountainous, high-relief project site, snow avalanche and landslide hazards exist, with the potential to affect mine construction, operations and closure.

Geohazard scenarios were identified for the facilities considered. Using unmitigated geohazard levels as a baseline, these scenarios were assessed in terms of risk to human safety, economic loss, environmental loss, and reputation loss. Mitigation strategies have been identified to reduce the High and Very High Risk scenarios to a target residual risk not exceeding Moderate. Where practical and cost-efficient, further risk reduction will be achieved.

20.2.4 ROCK GEOCHEMISTRY

Most of the bedrock in and adjacent to the deposits is frequently sulphide-bearing and is currently PAG on exposure to air and water. Seeps around natural gossans overlying the deposits indicate natural acid conditions with pH in the 2.5 to 3.0 range. The acidic drainage in the area has occurred naturally and has been present over a geological time scale.

The baseline geochemistry data have been used to develop correlations for the block models for the deposits to estimate the volumes of PAG and NPAG rock to be placed in the RSFs and exposed in the final pit walls. Runoff that comes in contact with PAG rock will require treatment. A system of diversions will be required to direct clean surface water away from disturbed areas and to direct contact water to the WTP.

Static and kinetic tests indicate that acid generation will not result from the cycloned tailing material proposed to be used for construction of the TMF embankments. Sulphide tailing will be stored in a lined central cell between the North and South cells. At closure the sulphide tailing cell will be covered with a layer of rougher tailing dredged from the rougher cells. This rougher tailing material will be kept in a saturated state isolating the sulphide tailing from oxygen and maintaining its stability. Surplus water from the TMF will have to be managed to ensure that discharges meet appropriate standards.

20.2.5 CLIMATE

The climate of the KSM Project area is typical of the northern Coast Mountains of BC, with distinct differences between the western (more coastal influenced) mine area and the eastern (more interior influenced) process plant and TMF area. Precipitation is estimated to range from about 1,614 to 1,652 mm and 1,083 to 1,371 mm per year in the mine and process plant areas, respectively, depending upon elevation. The majority of precipitation is received in the fall and winter from September through to February. October tends to have the highest or second highest precipitation levels for the year. 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.

20.2.6 HYDROLOGY

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 eventually flows into the Nass 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.

The area of mining (open pits and underground) 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 water storage dam will trigger the *International River Improvements Act Regulation* due to a reduction in freshet flows below the dam. Under the Regulation, a permit will be required for the dam.

The proposed location for the TMF and associated dam structures and diversions will need to be managed to minimize and avoid adverse effects on fish habitat in the drainages of Teigen and Treaty creeks, both of which are tributaries of the Bell-Irving River.

20.2.7 WATER QUALITY

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. 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. The high sediment loads, high metal content, lack of stream side vegetation, and low temperatures relating to their glacier sources has resulted in low aquatic productivity for these creeks.

20.2.8 FISHERIES

The Unuk and Bell-Irving rivers have high fisheries 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.

A cascade on Sulphurets Creek a short distance upstream of the confluence with the Unuk River likely inhibits the passage of migratory fish. Extensive sampling has not been successful in locating fish anywhere upstream of this barrier. Therefore the watercourses flowing through the mining area are assumed to be devoid of fish and development of the mining area should not have significant effects on fish.

The mainstems of Treaty and Teigen creeks host spawning, rearing, and overwintering habitat. The tributaries of these two creeks that drain the proposed TMF location are known to be occupied by Dolly Varden. Compensation for the Dolly Varden habitat adversely affected by construction and operation of the TMF will be required under federal legislation.

Mitigation measures to protect fisheries resources and compensation plans to replace affected fish habitat are being discussed and developed.

20.2.9 VEGETATION AND ECOSYSTEMS

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. Almost half of the immediate project area is not vegetated (water, ice) or is sparsely vegetated. No plants of conservation concern have been observed, although nine vulnerable ecosystems have been identified or predicted to occur in the area. Mitigation may be required to avoid or minimize adverse effects to these ecosystems.

Seabridge mapped plant communities and plant species of conservation concern concurrent with the field work for Predictive Ecosystem Mapping (PEM) and Terrestrial Ecosystem Mapping (TEM) to guide the environmental assessment and project design. Plant tissue was collected and analyzed to establish baseline metal content.

20.2.10 WETLANDS

The Project encompasses several areas of wetland, such as along the proposed access routes, in the vicinity of the proposed process plant and in the proposed TMF location. Wetlands in Canada 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). Location of infrastructure was designed to minimize the effects on wetlands where practicable. However, other focused mitigation or compensation will be required to address potential adverse project effects on wetlands.

20.2.11 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.

A number of federally and provincially listed species are known or expected to occur in the proposed project area: wolverine and fisher, western toad, and rusty blackbird. Species of concern also 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) and by Aboriginal groups, such as moose, mountain goat, marmot, and grizzly bear.

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. Grizzly bears may be displaced or deterred from using traditional travel corridors by some aspects of the Project. Mitigation measures will be required to address these effects.

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. Some areas of this habitat, and traditional travel corridors between them, may be affected by the Project. Mitigation will be required to address these effects.

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 (UWR) 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. In addition, a mountain goat winter range management plan is being developed to compensate for the loss of the designated UWR.

Mitigation may also be required to address potential effects of project facilities, such as access roads and the TMF, on species such as birds and amphibians.

20.2.12 TRADITIONAL KNOWLEDGE AND TRADITIONAL LAND USE

The KSM Project site is located on Crown land in an area historically used or claimed by several Aboriginal peoples. The Project lies within the boundaries of the Nass Area, as defined in the Nisga'a Final Agreement. Discussions are ongoing with these Aboriginal groups regarding potential effects on their rights and relevant mitigation.

20.2.13 Non-aboriginal Land Use

The western part of the KSM Project area is included in the Cassiar Iskut-Stikine LRMP 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.

A small part of the project area – a section of the proposed ore transport tunnel alignment – lies within the boundaries of the Nass South Sustainable Resource Management Plan area. A draft version of this plan was published in February 2012 following a public review period.

The whole region surrounding the KSM Project is heavily staked and several other mining companies have active exploration programs nearby. The adjacent Snowfield-Brucejack property is being explored by Pretium. On February 23, 2011, Pretium announced that Measured and Indicated Resources for Snowfield-Brucejack were 34.1 M oz of gold and 191.9 M oz of silver; Inferred Resources were 21.7 M oz of gold and 202.2 M oz of silver (at a cut-off grade of 0.30g/t AuEQ). The proximity of the Snowfield and Mitchell deposits suggests that the eventual development of these two deposits will require close coordination.

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 KSM 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.

20.2.14 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. Visual effects of the Project will be apparent to heli-skiers who may be accustomed to undisturbed terrain in the Teigen, Treaty, and Mitchell Creek valleys.

The controlled-access Eskay Creek 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 Creek Mine road. Potential travellers (reportedly about one commercial raft trip per year) on the Unuk River would be able to observe the bridge that will be required to travel from the Eskay Creek Mine road to the KSM Project.

20.2.15 Archaeology and Heritage Resources

Preliminary archaeological assessments have found evidence of short term historic hunting camps near the proposed midpoint tunnel portals and on the north side of the Mitchell Creek Valley in the vicinity of the proposed Mitchell pit. Ongoing archaeological assessments will determine the presence of artefacts 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 outside the project footprint reportedly dates from the operation of the Yukon telegraph line.

20.3 Design Guidance

20.3.1 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 effects and to ensure that the Project provides lasting benefits to local communities while generating substantial economic and social advantages for shareholders, employees, and the broader community.

20.3.2 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. Seabridge is striving to establish a cooperative working relationship with the Nisga'a Nation and all relevant First Nations people to ensure appropriate opportunities to gather information regarding relevant historical and contemporary use.

20.3.3 Valued Ecosystem and Socioeconomic 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 components (VCs) and will be given particular consideration during project assessment, planning, and final design.

VCs applicable to the Project have been identified through a comprehensive issues scoping exercise, which included consultation with federal and provincial regulatory bodies, local Aboriginal groups, and other stakeholders. The preliminary VCs are listed in the Application Information Requirements issued by the BCEAO.

20.3.4 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 VCs to ensure the primary concerns of 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 the project stages.

Seabridge is using the environmental assessment process as an opportunity to refine the project design in order to minimize potential environmental effects, and identify appropriate mitigation and management procedures.

20.3.5 Environmental Standards

Seabridge will design, construct, operate, and decommission the KSM Project to meet 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)
- *Canada Transportation Act* (Canada)
- *Fisheries Act* (Canada)
- *Transportation of Dangerous Goods Act* (Canada)

- *Workplace Hazardous Materials Information System* (WHMIS) *Safety Act* (Canada)
- *Nisga'a Final Agreement* (Canada).

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.

20.3.6 DESIGN FOR SOCIAL AND COMMUNITY REQUIREMENTS

Seabridge is striving to establish strong collaborative and cooperative relationships with relevant Aboriginal peoples (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 neighbour to all groups with interests in the region.

Following best practices in the industry, Seabridge is committed to a process to ensure that:

- communities benefit from employment, training, and contracting opportunities
- potential negative effects are mitigated
- 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 Seabridge will maintain its involvement and on-going commitments to communities and stakeholders.

20.4 WATER MANAGEMENT

Water management is 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 water.

A comprehensive water management plan will apply to all activities undertaken during the construction, operation, and decommissioning/closure 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 fresh water on the site, especially with the changes to flow pathways and drainage areas

- protect ecologically sensitive sites and resources, and avoid harmful effects 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 it to meet discharge standards prior to release
- minimizing the use of fresh water
- recycling water wherever possible to minimize the amount of water released
- monitoring the composition of release water and treating it to remove or control contaminants as required to meet discharge standards.

Diversion structures will be constructed to direct runoff away from disturbed areas where practicable. Approximately 25% of flows from Mitchell Glacier are from water flowing over the PAG rock at the Mitchell Glacier toe and from Snowfield slope areas, and will be treated after being diverted to the WSF. This water consists of contact water from the south slope of Mitchell Valley (Snowfield area) and flows of subglacial contact water running over PAG rock exposed under the glacier in the area below the MTF. These contact water flows will be collected both in surface trenches and in subglacial inlets under the toe area of the glacier, and routed via the Mitchell pit north wall dewatering adit and RSF drains to the WSF.. Additional subglacial inlets farther up the glacier will collect clean, non-contact water (from above the MTF) and route it through the MDT for discharge without treatment to Sulphurets Creek.

A similar tunnel to the MDT is proposed to divert McTagg Creek away from the RSFs proposed for the valley of that creek, discharging the water to the lower reaches of Gingras Creek, which flows into Sulphurets Creek immediately downstream of the mouth of Mitchell Creek. The McTagg diversion inlets will be staged in elevation and location, with an initially lower level tunnel inlet in Lower McTagg. Subsequent inlets will be established at higher elevations in McTagg once the valley is filled. The shorter initial leg reduces completion time and contributes to advancing the schedule to commence water treatment.

Channels will be constructed to collect surface runoff above pit high walls, RSFs, the process plant, and the TMF, where permitted by terrain characteristics. These diversions will isolate clean surface water from exposed metal-rich rock and tailing and allow the runoff to be released with little or no treatment. The MTT tunnel will have a pipeline that can route TMF water to the WSF for treatment in the event of an upset at the TMF side.

Diversion structures will be designed to manage freshet flows and 1-in-200 year storm events. Greater capacity will be provided, if required, based on an assessment of the consequences of failure. Lesser capacity may be provided where overflows can be stored and managed by other downstream structures, such as the TMF. All mine area surface diversion ditches are designed for the 200-year 24-hour average capacity with the consequence of overtopping being additional water treatment

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 will be treated as required prior to release.

The quality of water in streams affected by the Project, and discharges, will be monitored on a regular basis.

20.4.1 WATER MANAGEMENT STRUCTURES AS ENERGY SOURCES

Several of the proposed water management structures will result in significant potential hydraulic head that can be harnessed for electricity generation. Seabridge proposes to construct small hydroelectric facilities to recover the energy generated by the Mitchell and McTagg Diversion tunnels. A hydro turbine for energy recovery will also be installed on the discharge pipeline from the water storage facility at the point where it flows into the WTP. These facilities could continue to operate after mine closure, providing electricity to power the post-closure WTP and for sale to the provincial electrical grid. It is estimated that these three facilities will produce almost 48,706 MWh of electricity per year.

Additional electric energy will be recovered from the tailing discharge pipeline.

20.4.2 WATER SUPPLY

Process water will be obtained from the TMF whenever practicable from excess flows at the CIL Cell and supplemented pumping from a barge in the North Cell (Years 1 to 25) or South Cell (Years 25 to 55).

Potable water for use in office and accommodation facilities and kitchens will be sourced from water diversions constructed around the perimeter of the process plant site, waste rock dump, TMF and other infrastructure, or from well fields. 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 from other sources. During the winter months, well water from a field of wells near the process plant may be needed to supply fresh water for process make up and domestic use at the process plant and camp facility.

20.4.3 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 thickeners and filters. It is anticipated that excess water from the TMF should provide adequate water for most processing requirements.

20.4.4 Storm Water Management

Storm water will be managed throughout the construction, operation, and closure of the Project to minimize erosion and transport of contaminants. Diversion structures and collection and treatment facilities will be designed to handle 1-in-200-year storm events, as projected using available historic hydrological and meteorological data. Greater capacity will be provided if required based on an assessment of the consequences of failure. Lesser capacity may be provided where overflows can be stored and managed by other downstream structures, such as the TMF.

20.4.5 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 flow volumes in order to consistently meet discharge standards.

Discharges from the mine will be managed to meet the federal government MMER and negotiated provincial water quality objectives.

20.4.6 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.

20.5 Waste Management

20.5.1 Tailing Management

Tailing will be piped to the TMF. Energy will be recovered from the hydraulic head between the process plant and the TMF and used to generate electricity.

Laboratory tests confirm that the high sulphide content of the sulphide (cleaner) tailing from the process plant will cause this material to quickly oxidize and generate acid if exposed to the air. The proposed solution to this acid generation, and potential subsequent metal leaching, is to store the tailing in a permanently saturated state where oxidation is vastly reduced or eliminated. The TMF is designed to isolate the sulphide (cleaner) tailing in a stable subaqueous environment in perpetuity. The lined storage of CIL residue tailing will minimize the potential risks of seepage of contact water and reduce the risk of discharge of sulphides to the environment due to a dam breach (as CIL residue tailing are contained in centre of TMF, well away from perimeter dams).

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 the construction, operation and closure of the KSM Project
- provide a means to manage the TMF itself (managing substances going into and out of the facility)
- manage any discharge from the TMF to ensure that all effluent meets and/or exceeds the permitted water quality standards
- 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.

Seepage from the TMF will be collected at the North Seepage Dam and operate indefinitely. The Saddle Seepage Dam will operate for Stage 1 (Years 1 to 25), transition to the Southeast Seepage Dam during the Interim Stage (Years 25 to 30), and will also pump back to the TMF indefinitely.

Ditches will be constructed on both sides of the TMF to compensate for the loss of catchment area contributions to Teigen and Treaty Creeks surface water flows. Base flows up to 2 m^3/s in the largest catchment east of the TMF will be diverted by a weir to a tunnel that will bypass a potential geohazard area and then be transported by pipeline to the natural watercourse downstream of the North Seepage Collection Dam. .

At closure, the TMF will be configured as a "dry" structure with one or more small pond/wetland areas, and revegetated with grasses and trees. Surface drainage within the impoundments (Splitter Dam breached between North and CIL cells, Saddle Dam breached between CIL and South cells) will be directed towards a closure spillway excavated in rock around the right abutment of the Southeast Dam. This channel will lead to a rock cut spillway discharging closure runoff towards North Treaty . No discharge will be permitted until water quality meets discharge standards. The water will be treated at the TWTP 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 Treaty Creek meets permit requirements.

20.5.2 WASTE ROCK AND OVERBURDEN MANAGEMENT

The KSM Project will potentially generate 3.03 Bt of waste rock over the anticipated life of the mine. Extensive static and kinetic testing suggests that most of the waste rock will be PAG with no delay to the onset of ARD. Current, ongoing kinetic testing indicates that some NPAG rock types will have a lag time to the onset of ARD on the order of 50 to 200 years. Based on static testing, a small proportion of waste rock can be classified as NPAG. NPAG rock will be segregated wherever practical on the basis of a comprehensive testing program using blast hole cuttings to characterize rock removed from the pits. This program will be integrated with the ore control program. NPAG rock will be used for construction and reclamation purposes.

Waste rock from the operation will be deposited in the adjacent Mitchell Creek and McTagg Creek valleys. During Years -2 to 5, a portion of waste rock will be temporarily stored in a depression on Sulphurets Ridge. At Year 30, this material at the Sulphurets RSF will be rehandled to the landbridge. Surface runoff will be diverted away from these RSFs. Seepage and precipitation runoff from the facilities will be collected and treated.

A conventional high density sludge treatment plant is proposed to be employed for the treatment. This plant will also treat ore transport tunnel drainage water, pit dewatering water, and other water that does not meet standards for direct discharge.

Overburden and glacial till will be salvaged for later use to cover selected RSF surfaces at closure. Space limitations will restrict the volumes of overburden that can be stockpiled for later use. Overburden will be tested for acid generation prior to use.

20.5.3 HAZARDOUS WASTE MANAGEMENT

Hazardous waste materials, such as spoiled reagents and used batteries, will be generated throughout the life of the Project, from construction to decommissioning. 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.

20.5.4 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. 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 process plant sites. Waste collection areas will have provisions to segregate waste according to disposal methods and facilities to address potential spillage and fire.

20.6 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, have characterized the atmospheric environment of the KSM Project area to allow air dispersion modelling. Mitigation is being developed to minimize adverse effects from emissions. Regular monitoring of emissions will assess the success of the mitigation methods.

20.6.1 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 meet 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 preventative maintenance to ensure optimum performance of light-duty vehicles, the diesel mining equipment, and the incinerators, thereby reducing air emissions
- use of conveyors to transport ore and waste from the Kerr pit to the OPC and RSFs
- the use of large haul trucks for ore and waste transport from the Mitchell and Sulphurets pits 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 conveyor for moving crushed ore from the mine area to the Treaty plant site, 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.

20.6.2 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.

20.7 OPERATING PLAN

20.7.1 *ENVIRONMENTAL MANAGEMENT SYSTEMS*

Seabridge will develop and implement an 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
- traffic and access road management
- archaeological and heritage site protection
- vegetation and wetland management
- explosives management
- geohazard management.

20.7.2 Social and Community Management Systems

Seabridge will develop and implement a broad SCMS for the construction, operation, and closure phases of the KSM Project. The SCMS will comprise an ongoing consultation plan and community development plans to be developed through relationship building initiatives with Aboriginal and non-Aboriginal communities. Monitoring and oversight of the SCMS will require a designated person responsible for coordinating community development initiatives, training, communications and commitment tracking.

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

- impact benefit agreements, if any are negotiated
- 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
- procurement strategy.

20.8 Hydrological Survey

The KSM Project area is located in the coastal mountains of northwestern BC. The proposed pit areas lie within the upper areas of Sulphurets Creek watershed, which is a main tributary of the Unuk River. The proposed TMF will be located within the tributaries of Teigen and Treaty creeks. The North Dam of the TMF will be located about 5 km upstream of the confluence of South Teigen Creek with Teigen Creek. The South Dam will be located near the divide between Teigen and Treaty creeks. The Southeast Dam will be located about 4 km upstream of the confluence of North Treaty Creek and 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 form over the Pacific Ocean and lose moisture as they travel inland and pass over successive mountain ranges. In addition to the longitudinal precipitation gradient, there is also 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 TMF.

Mean annual precipitation in the pit area is expected to be approximately 1,600 mm, which will vary depending on elevation. Annual precipitation is expected to be less at the proposed TMF than at the proposed pit areas, and is estimated to be approximately 1,100 to 1,400 mm.

The proposed pit areas will be located the upper areas of Sulphurets Creek watershed. The Sulphurets Creek watershed is characterized by steep, narrow valleys, and is highly glacierized. 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, ranging from 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. These characteristics tend to promote precipitation losses in the form of infiltration and evapotranspiration, thereby reducing the production of surface runoff. In addition, the proposed TMF area is located down gradient from the Sulphurets Creek watershed along a longitudinal precipitation gradient in the region that delivers less precipitation to areas further inland from the Pacific Ocean. Consequently, surface runoff from watersheds in the proposed TMF area is expected to be substantially less than for Sulphurets Creek.

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 to the contribution of glaciers, which can provide substantial melt-water late into the summer. Fall (mid-September to November) is characterized by generally moderate-to-low flows, but is 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 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COSTS

An initial capital of US$5.256 B is estimated for the Project, based on capital cost estimates developed by the following consultants:

- MMTS – mine capital costs, rock RSF and pit area pioneering works
- Allnorth – WSD, tailing starter dams, and surface water management earthworks based on KCB designs and quantities
- BVL – conveying, tailing and reclaim water piping, and pumping
- Stantec and KCB – tunnelling
- Tetra Tech – process plant and associated infrastructure costs including plant site preparation
- Brazier – permanent power supply, MTT conveyor electrical and fire detection, mini hydro plant, and energy recovery systems
- McElhanney – permanent access roads
- EBA – winter access road.

All currencies in this section are expressed in US dollars, unless otherwise stated. Costs have been converted using a fixed currency exchange rate, based on the Bank of Canada three-year average of Cdn$1.00 to US$0.96.

The expected accuracy range of the capital cost estimate is +25/-10%.

This capital cost estimate includes only initial capital, which is defined as all capital expenditures that are required to produce concentrate and doré. A summary of the capital costs is shown in Table 21.1.

This estimate is prepared with a base date of Q1/Q2 2012. The estimate does not include any escalation past this date. Budget quotations were obtained for major equipment. The vendors provided equipment prices, delivery lead times, freight costs to a designated marshalling yard, and spares allowances. The quotations used in this estimate were obtained in Q1/Q2 2012, and are budgetary and non-binding.

For non-major equipment (i.e. equipment less than $100,000), costing is based on in-house data or quotes from recent similar projects.

All equipment and material costs include Free Carrier (FCA) manufacturer plant Inco terms 2000. Other costs such as spares, taxes, duties, freight, and packaging are covered separately in the Indirects section of the estimate.

Table 21.1 Capital Cost Summary (US$)

		Cost (US$ 000)
Direct Works		
A	Overall Site	199,818
B1	Open Pit Mining	185,826
B3	Underground Mining (Mitchell Block Caving)	0
B5	Underground Mining (Iron Cap Block Caving)	0
C	Crushing, Stockpiles, and Grinding	156,900
D1	Tunnelling	344,213
D2	MTT Transfer System	273,695
D3	Rope Conveyance (Sustaining)	0
E0	Plant Site Crushing	348,699
E1	Plant Site Grinding	458,242
F1	TMF	311,108
F6	Water Treatment	309,462
F8	Environmental	44,225
F9	Avalanche Control	45,845
G	Site Services and Utilities	34,226
J	Ancillary Buildings	96,097
K	Plant Mobile Equipment	10,676
M1	Temporary Services	190,739
M2	Treaty Road Marshalling Yard	10,791
N1	Permanent Electrical Power Supply and BC Hydro Capital Cost Contribution	217,319
N2	Mini Hydro Plants	16,536
N3	Energy Recovery Plants	7,576
P1	Permanent Access Roads	93,433
P2	Temporary Winter Access Roads	18,208
Q	Off-site Infrastructure and Facilities	73,896
Direct Works Subtotal		**3,447,530**
Indirects		
X	Project Indirects	1,056,550
Y	Owner's Costs	106,315
Z	Contingencies	645,743
Indirects Subtotal		**1,808,608**
Total		**5,256,138**

The detailed breakdown of this capital cost estimate is included in Appendix K.

21.1.1 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)
- HST
- royalties or permitting costs
- schedule acceleration costs
- working capital
- cost of this study
- sunk costs.

21.1.2 MINE CAPITAL COST

Mine capital costs are derived from a combination of supplier quotes and historical data collected by MMTS. This includes labour, maintenance, major component repairs, fuel, and consumables costs.

The equipment mine capital costs include delivery to the site and assembly but do not include taxes or duties. Mine capital costs are shown in Table 21.2.

Table 21.2 Mine Capital Costs

	Cdn$ (000)
Pre-production (in Operating Costs)	0
Mobile Equipment	144,444
Explosive Manufacturing and Storage	10,196
Dewatering	12,600
Electrical	6,987
Communication	400
Safety	150
Engineering Equipment	950
Dispatch Offices	672
Other Mining Costs	2,670
Haul Roads	14,500
Total Mine Capital	**193,569**

21.1.3 MINING BASIS OF ESTIMATE

Unit costs for consumables and labour rates are estimated from the 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 E:

- Salaries for the supervisory and administrative job category are based on MMTS's experience with similar functions in BC mines. A burden has been applied to base salaries to account for all statutory Canadian and BC holidays, social insurance, medical and insurance costs, pension, and vacation costs. (Note: the labour unit costs used in the open pit costs are from an earlier study; however, the net change of rate and burden is within 1% of the revised 2012 study costs.)
- For hourly employees, general labour rates expected in BC mines are applied. A burden is applied to employee base wages to include all statutory Canadian and BC holidays, social insurance, medical and insurance costs, pension, and vacation costs. The size and makeup of the mine fleet are based on the mine design; fuel requirements, which are affected by distance from the pit to the various destinations over the existing and future topography, are included in the estimate. (Note: the labour unit costs used in the open pit costs are from an earlier study; however, the net change of rate and burden is within 1% of the revised 2012 study costs.)
- Freight costs for all consumables, tires, and fuel, are included in the estimate as part of the budgetary quotations. The long term fuel price is estimated at a delivered cost to site of Cdn$0.937/L.

- Power costs are estimated as the sum of energy charges, demand charges and estimated at an overall Cdn$0.0502/kWh.
- Mining equipment consumables, major equipment replacements, sustaining capital, labour loading factors, equipment life, and costs are based on vendor information and MMTS's database from similar mining operations.

As directed by Seabridge, the operating costs for the pioneering work (excluding the fuel) are placed in the capital cost. This includes all pioneering work done to the end of Year -4 in the schedule. Pioneering and pre-production equipment capital is included in the capital costs.

Mine mobile equipment capital costs are shown in Table 26.3. The mobile equipment capital schedule includes all equipment purchases to end of Year -1, plus an assumed 10% deposit required for equipment required in Year 1.

Table 21.3 Mine Mobile Equipment Capital Schedule

Fleet Capital Cost	Purpose/Use	Capital (Cdn$ M)
Drilling		
Diesel Drill – 311 mm	Primary Drill	11.9
Electric Drill – 311 mm	Primary Drill	
Diesel Drill – High Wall – 150 mm	High Wall Drill	4.4
Blasting		
Hole Stemmer	Blast Hole Stemmer	0.5
Loading		
Major		
Diesel HydraulicShovel – 85 t (40 m^3)	Loading Ore & Waste	30.5
Electric Cable Shovel – 104 t (56 m^3)	Loading Ore & Waste	0
Electric Cable Shovel – 49 t (26 m^3)	Loading Ore & Waste	0
Support		
Track Dozer – 433 kW	Shovel Support	9.8
Rubber Tired Dozer – 372 kW	Pit Clean Up	1.3
Fuel/Lube Truck	Shovel Fuelling & Lube	1.1
Wheel Loader Multipurpose – 14 t	Pit Clean Up	1.1
Hauling		
Major		
Haul Truck – 363 t	Hauling Ore/Waste	0*
Support		
Water Truck – 20,000 gal	Haul Roads Water Truck	2.2
Track Dozer – 433 kW	Dump Maintenance	4.9
Motor Grader – 397 kW	Road Grading	4.0
Tire Manipulator	Tires	1.1

table continues…

Fleet Capital Cost	Purpose/Use	Capital (Cdn$ M)
Pit Maintenance		
Track Dozer – 433 kW	Pit Support	3.3
Float Tractor/Trailer – 170 t	Float Tractor & Trailer	3.9
Hydraulic Excavator – 6 t	Utility Excavator	2.4
Sump Pump - 1,400 gal/min	Pit Sump Dewatering	0.2
Light Plant	Lighting Plant	0.1
250-t Crane	Utility Crane	2.8
Crew Cab	Supervision and Crew Transportation	0.6
Ambulance	Ambulance	0.1
Hydraulic Excavator – 6 t	Utility Excavator	4.1
Mine Rescue Truck	Rescue Truck	0.1
GMC Guide XL Crew Bus	Crew Bus	0.3
Maintenance Truck – 1 t	Service Truck	0.2
Fire Truck	Fire Truck	0.3
Screening Plant – 12" max.	Road Crush & Stemmings	0.4
Picker Truck	Maintenance & Overhauls	0.3
Scraper – 37 t	Crush Haul for Winter Roads, Drill Steels	4.3
Crane 100 t Hydraulic Extendable	Utility Crane	2.1
Wheel Loader	Crusher (Road Crush) Loader	1.1
Snow Cat	Crew Transport (snow fleet)	0.6
40-t crane	Utility Crane	1.9
Forklift – 30 t	Forklift	0.6
Forklift – 10 t	Forklift	0.3
Service Truck	Service Truck	0.3
Welding Truck	Welding Truck	0.3
Powerline Truck	Powerline Maintenance	0.3
Preproduction Replacement Capital		0.7
Estimated Mobile Equipment Capital Cost		**$103.8 M**

Note: trucks are leased and costs are included in pre-production operating costs.

21.1.4 PROCESS EQUIPMENT AND INFRASTRUCTURE BASIS OF ESTIMATE

The work breakdown structure for the estimate is user-defined by area and section code. The contingency has been analyzed by the disciplines and the resultant contingency for this project is 14%.

The capital cost estimate for the process plant has been completed by Tetra Tech and is based on the information shown in Table 21.4.

Table 21.4 Process Capital Cost Basis of Estimate

Commodity	Estimate Basis
Plant and Equipment	
Major Equipment (>$1,000,000)	Single budget price quotations based on 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	
Plant Site Preparation & Roads	Estimated on a cost/unit area based on a preliminary earthworks volume 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 (incl. 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	1.30 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. 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 5% of process equipment costs
Mining Spares	Based on 5% of major mine equipment stock

The following supporting documentation is provided in the indicated appendices:

- process design criteria (Appendix D)
- preliminary flowsheets (Appendix C)
- general arrangement drawings (Appendix C).

21.1.5 PERMANENT ACCOMMODATION AND CONSTRUCTION CAMPS

There will be a total of 11 construction camps set up at construction and marshalling yard areas in order to facilitate the construction of this project. The camp sizes range from 40- to 800-person capacities. Some of the early camps, where access is by either the Winter Access Road or helicopter, will be the foldaway type or similar for ease of transport. The camp names, number and, capacity are shown in Table 21.5.

Table 21.5 Construction Camps

Construction Camps	Capacity
Camp 1 – Granduc	80-person
Camp 2 – Ted Morris Staging	80-person
Camp 3 – Eskay	50-person
Camp 4 – Mitchell North	125-person
Camp 5 – Treaty Plant	800-person
Camp 6 – Treaty Saddle	120-person
Camp 7 – Unuk North	40-person
Camp 8 – Unuk South	40-person
Camp 9 – Mitchell Initial	140-person
Camp 10 – Mitchell Secondary	400-person
Camp 11 – Treaty Creek Marshalling Camp (relocated Camp 12)	60-person
Camp 12 – Temporary Access Road	60-person

Note: Camp12 will initially be set up adjacent to the Highway 37 Treaty Creek Access Road, and moved to the Camp 11 location after the Bell-Irving River bridge has been installed.

Camp 1 will be set up near Granduc for the coordination of receiving equipment and supplies prior to the winter season. It is from this location that transportation will take place over the Winter Access Road to the Ted Morris area in the Mitchell Valley. Camp 2 will be set up in this area as soon as it arrives over the Winter Access Road. This will become the headquarters for receiving the equipment and materials arriving by Winter Access Road over the next 8 to 10 weeks.

During the first season of construction in the Mitchell Valley, Camps 4, 8, and 9 will be built to service the various work fronts. Additionally, Camps 3 and 7 will be built to service the Coulter Creek Access Road, along with Camp 12 at the Treaty Creek Access Road, and Camp 6 at the MTT saddle.

In addition to the 11 construction camps, the estimate also includes two operations camps, located as follows:

- a 250-person operations camp, constructed at the Treaty OPC area
- a 350-person operations camp, constructed west of the truck shop in the Mitchell Valley.

21.1.6 Labour Rates

Different labour rates were applied to various areas of the project. In general, a labour rate of US$103.68/h (Cdn$108/h) has been used for construction labour.

21.1.7 Taxes

HST has been excluded from the estimate.

21.1.8 Freight and Logistics

Freight and logistics for main bulk material and equipment are calculated at 8%. If available, supplier quotations are used in the estimate for freight, export packing, etc.

A nominal allowance is allowed for air freight.

21.1.9 Owners' Costs (Including Owners Commissioning Allowance)

An allowance has been included for the Owners' costs. This cost has been provided by the Owner.

21.1.10 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 77-hour work week with a rotation of 3-weeks in/1-week out for the construction phase of the project. The exception to this turnaround rotation for construction personnel is the tunnelling crews, who are scheduled for 10 h/d with two -weeks in/2-weeks out.
- A productivity factor of 1.30 was applied to the labour portion of the construction estimate to allow for the inefficiency of long work hours, climatic conditions, and the 3-week in/1-week out rotation. This was based on in-house data supplied by contractors on previous similar projects for northern BC construction.

- 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 stockpiles for soil densification have not been allowed in this estimate.

21.1.11 Contingency

A contingency allowance was included to cover additional costs, which could occur as a result of more detailed design, unexpected site conditions, or unusual cost escalation. This estimate adequately covers minor changes to the current scope expected during the next phase of the project.

Several areas that have major costs (and allowances) are assumed to contain a certain amount of contingency; therefore, a lower contingency was applied for these areas. The contingency has been analyzed by the disciplines and the resultant contingency for this project is 14%.

21.2 Operating Costs

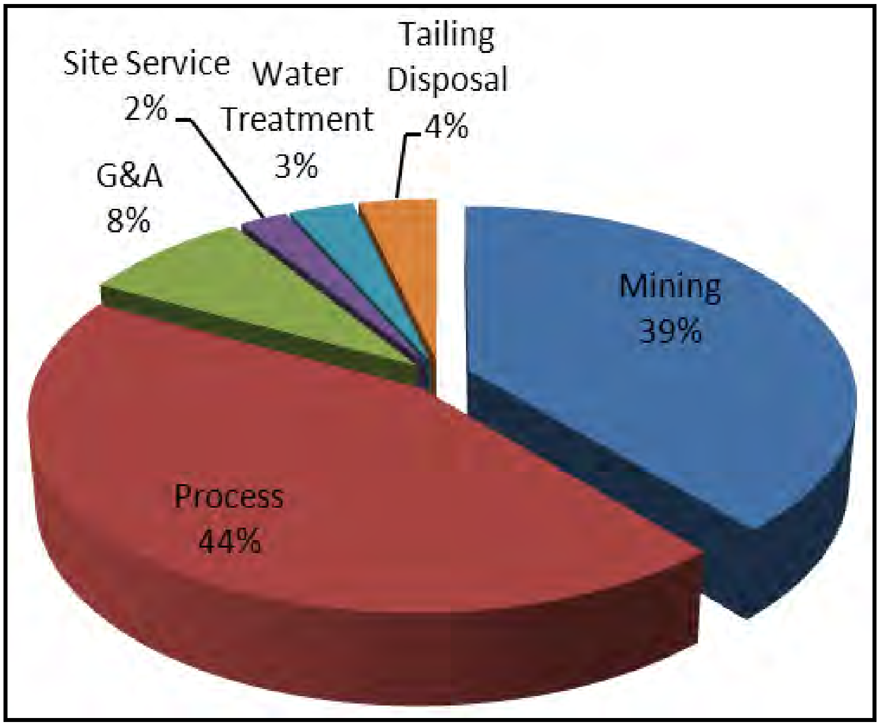
The operating cost for the KSM Project was estimated at US$13.72/t milled (Table 21.6). The cost distribution for various areas is shown in Figure 21.1. The estimate was based on an average daily process rate of 130,000 t milled.

Table 21.6 Operating Cost Summary

	US$/a (000)	US$/t Milled
Mine		
Mining Costs – Mill Feed	251,901	5.31*
Open Pit – Mill Feed		*5.38*
Block Caving – Mill Feed		*5.14*
Mill		
Staff & Supplies	233,012	4.91
Power (Process Only)	53,081	1.12
G&A and Site Service		
G&A	53,556	1.13
Site Service	14,959	0.32
Tailing and Water Treatment		
Tailing	24,440	0.52
Water Treatment	20,238	0.43**
Total Operating Cost	**651,187**	**13.72**

* excluding mine pre-production operating costs.
** LOM average cost calculated by total LOM operating cost divided by LOM process tonnage.

Figure 21.1 Operating Cost Distribution



Currencies are expressed in US dollars, unless otherwise specified. The cost estimates in this section are according to budget prices in Q1/Q2 2012 or from databases of the consulting firms involved in preparing the cost estimates.

When required, certain costs in this report have been converted using a fixed currency exchange rate of Cdn$1.00 to US$0.96. The expected accuracy range of the operating cost estimate is +25/-10%.

Power will be supplied by BC Hydro at an average cost of US$0.047/kWh at the plant 25 kV bus bars, based on the BC Hydro credits for energy conservation by use of HPGR and similar. 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 costs. Power costs for surface services are included in the site services costs.

The estimated electrical power costs are based on the 2012 BC Hydro Tariff 1823 - Transmission Service Stepped Rate and Schedule 1901 - Deferred Account Rate Rider. The electrical power costs also account for local system losses and include 7% PST, which is being re-introduced and is not treated as an input tax credit. The rates take advantage of the implementation of BC Hydro-approved energy conservation measures in the plant design phase, including the HPGR circuit, which will greatly reduce the more costly Tier 2 power in the BC Hydro stepped–rate Schedule 1823.

The operating costs are defined as the direct operating costs including mining, processing, tailing storage, water treatment, and G&A. The hydropower credit from the recovered hydro-energy during mining operations is not accounted for in the operating cost estimate, but is included in the economic analysis. Sustaining capital includes all capital expenditures after the process plant has been put into production.

21.2.1 Open Pit Mine Operating Costs

All open pit mining operating costs are shown in Canadian dollars, unless otherwise specified. Mine operating costs, including labour, maintenance, major component repairs, fuel, and consumables costs, are derived from a combination of supplier quotes and historical data collected by MMTS. 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 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 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, historical blasting costs and a blasting study conducted for the KSM Project. Geotechnical costs for high wall control blasting, horizontal drains, etc. are based on recommendations from BGC and from 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 added, 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 are therefore not rounded off in Table 21.7 and Table 21.8.

The mine hourly and salaried labour schedules are summarized in Table 21.7 and Table 21.8, and listed in detail in Appendix E.

Table 21.7 Mine Hourly Labour Schedule Manning Levels

Hourly Labour Summary	Year 5
Mine Operations	
Drill Operator	16
Blasters	8
Shovel Operator	21
Haul Truck Driver	158
Grader Operator	11
Excavator Operator	17
Loader Operator	15
Track Dozer Operator	39
Scraper Operator	6
Mine Maintenance	
Electrician	17
HD Mechanic	82
LD Mechanic	4
Machinist	8
Crane Operator	7
Welder	15
Tireman	3
Labourer Serviceman	7
Total Hourly	**434**

Note: Water truck and fuel truck operators are included in haul truck driver quantity.

Table 21.8 Mine Salaried Labour Schedule Manning Levels

Salaried Labour Summary	Year 5
Mine Operations	
Operations General Foreman	1
Shift Foreman	2
Area Foreman	4
Training General Foreman	1
Shift Trainers	4
Drilling & Blasting Foreman	1
Maintenance General Foreman	1
Maintenance Planner	2
Maintenance Planning Clerk	2

table continues…

Salaried Labour Summary	Year 5
Maintenance Shift Foreman	2
Mechanical Foreman	6
Electrical Foreman	2
Services Foreman	4
Administration Assistant	1
Technical Services	
Chief Engineer	1
Senior Geologist	1
Pit Geologist	2
Ore Grade Technicians	4
Project Engineer	1
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
Total Salaried	**53**

Mine labour rates are based on current salaries for G&A employees, and hourly rates for mine operations and maintenance personnel in the area (Table 21.9 and Table 21.10), plus a payroll burden.

Table 21.9 Mine Operating and Maintenance Hourly Labour Rates

Position	Cdn$/ Manhour*
Mine Operations	
Drill Operator	45.24
Blasters	45.24
Shovel Operator	46.75
Haul Truck Driver	42.22
Grader Operator	43.73
Excavator Operator	43.73
Loader Operator	45.24
Track Dozer Operator	43.73
Scraper Operator	42.22
Crusher Operator	42.22

table continues…

Position	Cdn$/ Manhour*
Water Truck Operator	42.22
Fuel Truck Operator	42.22
Mine Maintenance	
Electrician	52.93
HD Mechanic	52.93
LD Mechanic	47.85
Machinist	52.93
Crane Operator	42.22
Welder	52.93
Tireman	46.75
Labourer Serviceman	36.25

* includes burden.

Table 21.10 Mine G&A Salaries

Position	Salary With Burden (Cdn$)
Mine Operations	
Operations General Foreman	166,750
Shift Foreman	152,250
Area Foreman	130,500
Training General Foreman	130,500
Shift Trainers	116,000
Drilling & Blasting Foreman	152,250
Maintenance General Foreman	166,750
Maintenance Planner	116,000
Maintenance Planning Clerk	73,950
Maintenance Shift Foreman	152,250
Mechanical Foreman	130,500
Electrical Foreman	130,500
Services Foreman	130,500
Administration Assistant	79,750
Technical Services	
Chief Engineer	181,250
Senior Geologist	159,500
Pit Geologist	101,500
Ore Grade Technicians	87,000
Project Engineer	130,500
Senior Mining Engineer	159,500
Mine Engineer	123,250
Drilling & Blasting Engineer	123,250

table continues…

Position	Salary With Burden (Cdn$)
Drilling & Blasting Technician	87,000
Surveyor	87,000
Engineering Clerk	87,000
Dispatch Engineer	109,548
Senior Geotechnical Engineer	159,500

Mining LOM unit operating costs, including pre-production operating costs, are listed in Table 21.11 and Table 21.12. Complete mine cost tables, including mine capital and operating cost schedules, are available in Appendix E.

Table 21.11 Open Pit Mining Costs per Tonne Mill Feed

	LOM Cost (Cdn$/t Mill Feed)
Drilling	0.17
Blasting	0.79
Loading	0.76
Hauling	3.19
Pit Maintenance	0.61
Geotechnical	0.09
Unallocated Labour	0.01
GME	0.25
Total Mining Cost	**5.87**

* Including pre-production operating cost.

Table 21.12 Open Pit Mining Costs per Tonne Material Mined

	LOM Cost (Cdn$/t Material Mined)
Drilling	0.05
Blasting	0.25
Loading	0.24
Hauling	1.01
Pit Maintenance	0.20
Geotechnical	0.03
Unallocated Labour	0.00
GME	0.08
Total Mining Cost	**1.87**

Note: Material mined includes re-handled waste and borrow sources for construction material.

A graph of mine unit operating cost is shown as Cdn$/t material mined (waste and mineralized material) in Figure 21.2. The distribution of unit cost by mining area is shown in Figure 21.3.

Figure 21.2 Unit Operating Cost for Mining ($/t Material Mined)

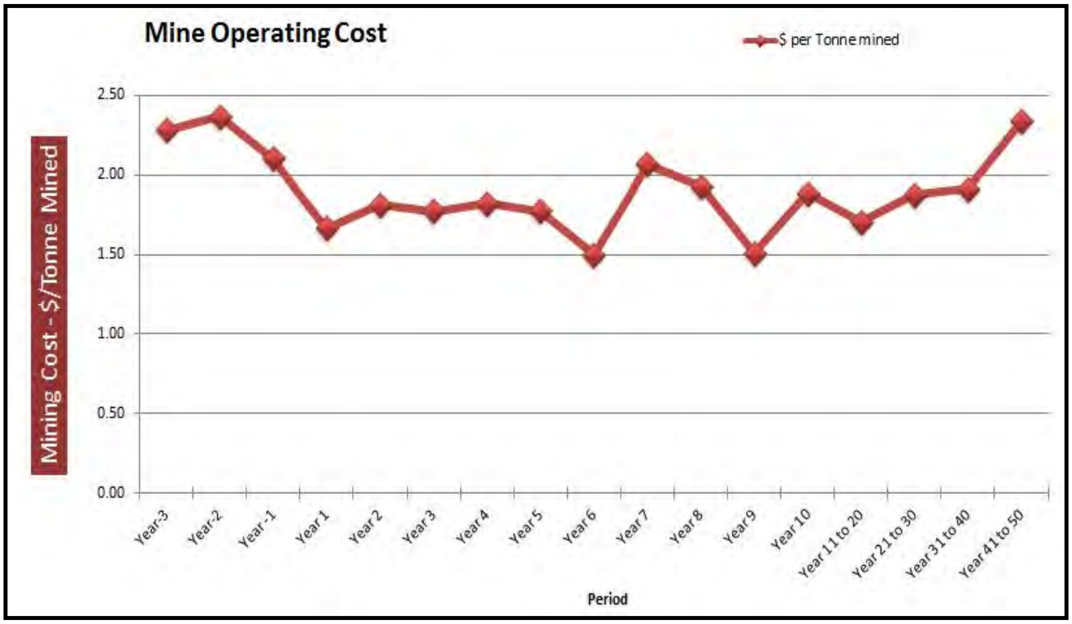
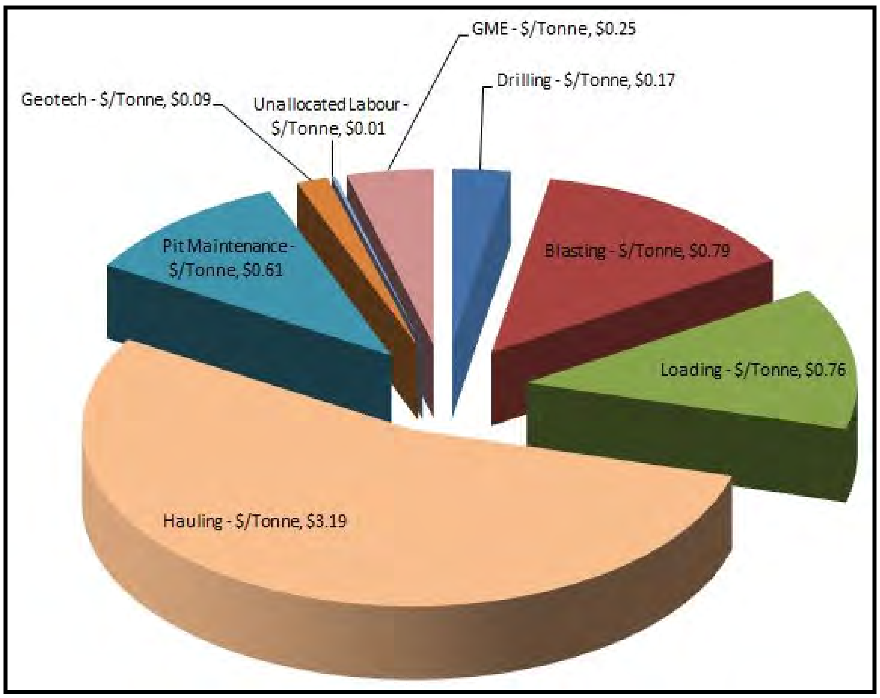


Figure 21.3 LOM Mine Plant Feed Operating Cost Distribution ($/t Mill Feed)



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.

Estimated fuel quantities scheduled for the first five years of milling are shown in Table 21.13.

Table 21.13 Mine Fuel Consumption Schedule

Fuel Consumption		Y1	Y2	Y3	Y4	Y5
Drilling	m^3	1,759	1,561	1,838	1,867	1,241
Blasting (Explosives Factory)	m^3	2,779	2,625	2,910	3,060	2,346
Loading	m^3	9,984	9,238	9,184	9,523	9,511
Hauling	m^3	56,413	54,624	56,013	55,818	42,008
Pit Maintenance	m^3	6,360	6,348	6,334	6,325	6,343
Total	m^3	**77,295**	**74,394**	**76,279**	**76,593**	**61,449**

21.2.2 UNDERGROUND MINING OPERATING COSTS

MITCHELL UNDERGROUND

The average block caving mining operating cost for the Mitchell underground mine is estimated at Cdn$5.00/t and consists of the equipment and labour that are required to move material from the draw point to the surface conveyor portal and the fixed costs to operate the mine (Table 21.14). This includes the use of the LHDs, secondary breakers, crushers and conveyors, and the labour required to plan and execute the mining plan. Mine labour comprises approximately 52% of the total Mitchell underground mining cost while crushing and conveying is 15%, secondary breaking is 13% and production mucking and haulage is 12%.

Table 21.14 Mitchell Underground Mine Operating Cost Breakdown

Activity	OPEX ($/tonne)	(%)
Labour	$2.60	52%
Crusher and conveying	$0.80	15%
Stationary and mobile rockbreaking	$0.61	13%
Production LHD and haulage	$0.58	12%
Fixed costs	$0.36	7%
Rehabilitation	$0.04	1%
Total	**$ 5.00**	

The initial and sustaining block caving capital cost estimate includes the purchase and installation of all equipment and the excavation of all the underground workings. The pre-production capital expenses, over the first 6 years of the underground mine life, are estimated at Cdn$800 M with an average sustaining capital cost of Cdn$74 M over the remaining 31 years. The total initial and sustaining capital cost for Mitchell underground mining are estimated to be $3.1 B.

IRON CAP UNDERGROUND

The average block caving mining operating cost for Iron Cap underground mine is estimated at Cdn$6.15/t and consists of the equipment and labour that are required to move material from the drawpoint to the MTT conveyor tunnel and the fixed costs to run the mine (Table 21.15). This includes the use of the LHDs, crushers, conveyors, mine services, and the labour required to plan and execute the mining plan. Mine labour comprises approximately 56% of the total Iron Cap underground mining cost while crushing and conveying is 17%, production mucking is 13%, 9% accounts for fixed costs, and secondary breaking is 5%.

Table 21.15 Iron Cap Underground Mine Operating Cost Breakdown

Activity	OPEX ($/tonne mined)	(%)
Labour	$ 3.45	56%
Crushing and Conveying	$ 1.04	17%
Production Mucking (LHD)	$ 0.82	13%
Fixed	$ 0.53	9%
Mobile Rockbreaking	$ 0.28	5%
Rehabilitation	$ 0.04	0%
Total	**$ 6.15**	

The initial and sustaining block caving capital cost estimate includes the purchase and installation of all equipment and the excavation of all the underground workings. The pre-production capital expenses, over the first 5 years of the underground mine life, are estimated at Cdn$509 M with an average sustaining capital cost of Cdn$46 M over the remaining 21 years. The total initial and sustaining capital costs for Iron Cap underground mining are estimated to be Cdn$1.5 B.

Details on the underground mining operating costs are provided in Appendix E.

21.2.3 PROCESS OPERATING COSTS

SUMMARY

All process operating costs are shown in US dollars, unless otherwise specified. The average annual process operating costs for different mineralization are estimated to be approximately:

- Mitchell and Iron Cap mineralization: US$281 M (US$5.92/t milled)
- Kerr mineralization: US$285 M (US$6.00/t milled)
- Sulphurets mineralization: US$312 M (US$6.58/t milled).

The process operating costs for these mineral materials are based on a process rate of 130,000 t/d and 94% plant availability. The estimated operating costs for the Sulphurets and Kerr ore are different from the Mitchell and Iron Cap ore. Therefore, when the mill processes the Mitchell ore together with the ore from the other deposits, the operating cost has been estimated based on the ratio of the different ore tonnages processed and their individual operating costs.

The estimated process operating costs are summarized in Table 21.16, and include:

- 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 Tetra Tech's database
- maintenance supplies based on approximately 5% of major equipment capital costs
- reagents based on test results and quoted budget prices or Tetra Tech's database
- other operation consumables including laboratory, filtering cloth, and service vehicles consumables
- No taxes have been included in the estimate, unless specified.
- power consumption for the process plant at the power unit cost of US$0.047/kWh.

Table 21.16 Summary of Process Operating Costs

Description	Mitchell & Iron Cap			Sulphurets			Kerr		
	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)
Human Power									
Operating Staff	34	4,338,000	0.091	34	4,338,000	0.091	34	4,338,000	0.091
Operating Labour	112	9,929,000	0.209	120	10,617,000	0.224	120	10,617,000	0.224
Maintenance	80	8,216,000	0.173	80	8,216,000	0.173	80	8,216,000	0.173
Sub-total Human Power	**226**	**22,483,000**	**0.474**	**234**	**23,171,000**	**0.488**	**234**	**23,171,000**	**0.488**
Major Consumables & Supplies									
Major Consumables									
Metal Consumables		72,570,000	1.529		93,220,000	1.965		70,605,000	1.488
Reagent Consumables		103,991,000	2.192		103,991,000	2.192		103,991,000	2.192
Supplies									
Maintenance Supplies		27,386,000	0.577		28,768,000	0.606		31,819,000	0.671
Operating Supplies		2,892,000	0.061		2,892,000	0.061		2,892,000	0.061
Sub-total Consumables & Supplies		**206,839,000**	**4.359**		**228,871,000**	**4.823**		**209,307,000**	**4.411**
Power Supply		51,784,000	1.091		60,411,000	1.273		52,054,000	1.097
Sub-total Power		**51,784,000**	**1.091**		**60,411,000**	**1.273**		**52,054,000**	**1.097**
Process Operating Cost Total		**281,106,000**	**5.924**		**312,453,000**	**6.585**		**284,533,000**	**5.996**

PERSONNEL

The projected personnel requirements are between 226 and 234 persons, including:

- 34 staff for management and professional services
- between 112 and 120 operators, including personnel at laboratories for quality control, process optimization, and assaying
- 80 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.

The total estimated personnel cost ranges from US$0.47/t milled to US$0.49/t milled. The detailed personnel description and costs are shown in Appendix K for each processing plant area.

OPERATING AND MAINTENANCE SUPPLIES

Major consumables and operating suppliers are estimated at US$4.36/t milled for Mitchell and Iron Cap mineralization, US$4.41/t milled for Kerr mineralization, and US$4.82/t milled for Sulphurets mineralization. The major consumables include metal and reagents consumables. The liner and grinding media consumption were estimated from the Bond abrasion index equation and the budget prices from the potential suppliers or Tetra Tech database.

Reagent consumptions were estimated from laboratory test results and comparable operations. The reagent costs were from the Q1/Q2 2012 budget prices from potential suppliers or Tetra Tech's database.

The maintenance supplies are estimated at US$0.58/t milled for Mitchell and Iron Cap mineralization, US$0.67/t milled for Kerr mineralization, and US$0.61/t milled for Sulphurets mineralization. Maintenance supplies are estimated based on comparable operations or approximately 5% of major equipment capital costs.

OPERATING COSTS PER AREA OF OPERATION

Table 21.17 shows the operating cost of each processing area. The details of operating costs for each processing area are further discussed in following sections and shown in Appendix K.

Table 21.17 Operating Costs per Area of Operation

Description	Mitchell & Iron Cap			Sulphurets			Kerr		
	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)
Crushing/Grinding/Copper Flotation Plant	147	186,750,000	3.936	155	217,905,000	4.592	155	190,204,000	4.009
Tunnel Transport	8	13,272,000	0.280	8	13,272,000	0.280	8	13,272,000	0.280
Molybdenum Flotation Plant	4	6,494,000	0.137	4	6,686,000	0.141	4	6,467,000	0.136
Leach Plant	51	36,873,000	0.777	51	36,873,000	0.777	51	36,873,000	0.777
Cyanide Solution/Residue Handling	8	32,841,000	0.692	8	32,841,000	0.692	8	32,841,000	0.692
Tailing Management/ Reclaim Water	8	4,491,000	0.095	8	4,491,000	0.095	8	4,491,000	0.095
Water Treatment		385,000	0.008		384,000	0.008		385,000	0.008
Total	**226**	**281,106,000**	**5.924**	**234**	**312,453,000**	**6.585**	**234**	**284,533,000**	**5.996**

Crushing, Grinding, Copper, and Pyrite Flotation

The operating cost for crushing, grinding, copper, and pyrite flotation is estimated to be approximately:

- US$3.94/t milled for Mitchell and Iron Cap mineralization
- US$4.01/t milled for Kerr mineralization
- US$4.59/t milled for Sulphurets mineralization.

The breakdown costs are shown in Table 21.18. The cost estimate includes personnel to operate the circuits as well as the metallurgy and assay laboratories. 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 copper-gold flotation process is estimated at:

- 878 GWh for Mitchell and Iron Cap ores
- 1,062 GWh for Sulphurets ore
- 884 GWh for Kerr ore.

Details of the estimate are shown Appendix K.

Table 21.18 Grinding, and Copper/Pyrite Flotation Operating Costs

Description	Mitchell and Iron Cap			Sulphurets			Kerr		
	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)
Personnel									
Operating Staff	23	2,754,000	0.058	23	2,754,000	0.058	23	2,754,000	0.058
Operating Labour	60	5,310,000	0.112	68	5,999,000	0.126	68	5,999,000	0.126
Maintenance	64	6,710,000	0.141	64	6,710,000	0.141	64	6,710,000	0.141
Sub-total Personnel	**147**	**14,774,000**	**0.311**	**155**	**15,462,000**	**0.326**	**155**	**15,462,000**	**0.326**
Supplies									
Major Consumables									
Metal Consumables		71,893,000	1.515		92,350,000	1.946		69,954,000	1.474
Reagent Consumables		39,482,000	0.832		39,482,000	0.832		39,482,000	0.832
Supplies									
Maintenance Supplies		18,106,000	0.382		19,488,000	0.411		22,540,000	0.475
Operating Supplies		1,187,000	0.025		1,187,000	0.025		1,187,000	0.025
Power Supply		41,309,000	0.871		49,935,000	1.052		41,578,000	0.876
Sub-total Supplies		**171,977,000**	**3.624**		**202,442,000**	**4.266**		**174,741,000**	**3.683**
Total	**147**	**186,750,000**	**3.936**	**155**	**217,905,000**	**4.592**	**155**	**190,204,000**	**4.009**

Molybdenum Flotation

Table 21.19 shows that the estimated operating cost for molybdenum flotation is approximately US$0.14/t milled for the mineralization. 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 3.1 GWh. Details of the costs are shown in Appendix K.

Table 21.19 Molybdenum Flotation Operation Costs – Mitchell and Iron Cap

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)
Personnel			
Operating Labour	4	398,000	0.008
Sub-total Personnel	**4**	**398,000**	**0.008**
Supplies			
Major Consumables			
Metal Consumables		678,000	0.014
Reagent Consumables		3,766,000	0.079
Supplies			
Maintenance Supplies		182,000	0.004
Operating Supplies		24,000	0.001
Concentrate Leach		1,299,000	0.027
Power Supply		147,000	0.003
Sub-total Supplies		**6,096,000**	**0.128**
Total	**4**	**6,494,000**	**0.137**

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 US$0.78/t milled for all areas of mineralization. The personnel cost is estimated to be US$0.11/t milled (Table 21.20). The cost for major consumables is estimated at US$0.62/t milled. The power consumption for this circuit is estimated at 19.4 GWh/a. A detailed cost estimate is shown in Appendix K.

Table 21.20 Gold Leach and Recovery Circuit Operating Costs

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t CIL*)	Unit Cost (US$/t Milled)
Personnel				
Operating Staff	11	1,584,000	0.288	0.033
Operating Labour	24	2,156,000	0.391	0.045
Maintenance	16	1,506,000	0.273	0.032
Sub-total Personnel	**51**	**5,246,000**	**0.952**	**0.111**
Supplies				
Major Consumables				
Major Consumables		29,267,000	5.314	0.617
Supplies				
Maintenance Supplies		1,351,000	0.245	0.028
Operating Supplies		96,000	0.017	0.002
Power Supply		913,000	0.166	0.019
Sub-total Supplies		**31,627,000**	**5.743**	**0.667**
Total	**51**	**36,873,000**	**6.695**	**0.777**

* CIL = carbon in leach.

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 US$0.69/t milled for all the mineralization. This cost includes a labour cost of US$0.02/t milled and a total processing reagent supplies cost of US$0.66/t milled. The annual power consumption will be approximately 10.3 GWh. The estimates are shown in Table 21.21. A detailed cost estimate is shown in Appendix K.

Table 21.21 Cyanide Recovery and Destruction Operating Costs

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/CIL*)	Unit Cost (US$/t Milled)
Personnel				
Operating Staff	8	689,000	0.125	0.015
Sub-total Personnel	**8**	**689,000**	**0.125**	**0.015**
Consumables and Supplies				
Reagent Consumables		31,260,000	5.676	0.659
Maintenance Supplies		387,000	0.070	0.008
Operating Supplies		19,000	0.003	0.001
Power Supply		486,000	0.088	0.010
Sub-total Supplies		**32,153,000**	**5.838**	**0.678**
Total	**8**	**32,841,000**	**5.963**	**0.692**

* CIL = carbon in leach.

Tunnel Conveyance Operation

The operating cost estimate for the tunnel conveyance is shown in Table 21.22. The major operating cost components are maintenance and power supply. The total unit cost is estimated to be US$0.28/t milled, including power supply, which is estimated at 135.8 GWh/a.

Table 21.22 Tunnel Conveyance Operating Costs

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)
Personnel			
Operating Labour	8	689,000	0.015
Sub-total Personnel	**8**	**689,000**	**0.015**
Supplies			
Maintenance Supplies		6,083,000	0.128
Operating Supplies		114,000	0.002
Power Supply		6,387,000	0.135
Sub-total Supplies		**12,583,000**	**0.265**
Total	**8**	**13,272,000**	**0.280**

Tailing and Reclaimed Water Operation

The cost estimates for tailing delivery to the TMF and water reclamation for all the mineralization are shown in Table 21.23, which details the unit costs for labour, maintenance supplies, operating suppliers, and power supply. The total cost is expected to be approximately US$0.10/t milled. The major cost contribution of tailing and reclaim water operations is power consumption for reclaiming water from the tailing storage pond. The annual power requirement is estimated to be approximately 53.5 GWh, which accounts for US$0.05/t milled. A more detailed breakdown is shown in Appendix K.

Table 21.23 Tailing Delivery and Reclaimed Water Operating Costs

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Milled)
Personnel			
Operating Labour	8	689,000	0.015
Sub-total Personnel	**8**	**689,000**	**0.015**
Supplies			
Maintenance Supplies		1,150,00	0.024
Operating Supplies		133,000	0.003
Power Supply		2,519,000	0.053
Sub-total Supplies		**3,802,00**	**0.080**
Total	**8**	**4,491,000**	**0.095**

21.2.4 TMF Operating Costs and Mitchell Water Treatment Costs

The tailing dam ongoing construction and operation costs are estimated to be approximately US$24.4 M/a, or US$0.52/t milled.

The cost for Mitchell water management, including diversions and collection dam operations, is estimated to be approximately US$20.2 M/a, or US$0.43/t milled.

21.2.5 General and Administrative

G&A costs are the costs that do not relate directly to the mining or processing operating costs. These costs include:

- personnel – general manager and staffing in accounting, purchasing, environmental, and other G&A departments
- expenses – including insurance, administrative supplies, medical services, legal services, human resources related expenses, travelling, accommodation/camp costs, air/bus crew transportation, regional and property taxes, and external assay/testing.

The G&A costs are estimated at approximately US$53.6 M/a, or US$1.13/t milled, including approximately US$0.24/t for personnel and US$0.89/t for general expenses. The major costs are accommodation and crew air transportation, estimated at about US$15.4 M/a. A summary of the G&A estimate for personnel and general expenses are shown in Table 21.24 and Table 21.25, respectively.

Table 21.24 G&A Personnel Costs

G&A	Personnel	Total Cost with Burden (US$/a)	Unit Cost (US$/t Milled)
Mine Site			
General Manager	1	360,000	0.008
Assistant General Manager	1	240,000	0.005
Operations Manager	1	240,000	0.005
Logistics Manager - Transportation	1	216,000	0.005
Camp Logistics Manager	1	168,000	0.004
Human Resources Superintendent	1	156,000	0.003
Payroll Supervisor	1	120,000	0.003
Admin Manager	1	180,000	0.004
Administrative Assistants	2	180,000	0.004
Environmental Superintendent	1	168,000	0.004
Waste Management Technologists	4	432,000	0.009
Environmental Sampling Technicians	4	336,000	0.007
Wildlife Technologists	4	432,000	0.009

table continues...

G&A	Personnel	Total Cost with Burden (US$/a)	Unit Cost (US$/t Milled)
Aquatics Coordinator	1	144,000	0.003
Aquatics Technologists	2	216,000	0.005
Records/GIS Coordinators	1	108,000	0.002
Safety and Training Superintendent	1	144,000	0.003
Training Supervisors	4	432,000	0.009
Warehouse Supervisor	1	120,000	0.003
IT Technicians	4	408,000	0.009
Shipping & Transport Supervisor	1	132,000	0.003
Shipping & Transport Coordinator	1	96,000	0.002
General Clerks	8	576,000	0.012
Warehouse Assistants/First Aid	8	622,080	0.013
First Aid /Safety Coordinators	8	725,760	0.015
Security Manager	1	120,000	0.003
Security Officers	8	663,552	0.014
Stewart			
Stewart Port Supervisor	1	108,000	0.002
Stewart Port Assistants	3	270,000	0.006
Security Officers	2	165,888	0.003
Smithers/Terrace			
Office Manager	1	168,000	0.004
Community Coordinator	1	144,000	0.003
Assistant Community Coordinator	1	96,000	0.002
Human Resources -Recruiting Manager	1	168,000	0.004
Chief Purchasing/Logistics Manager	1	132,000	0.003
Purchasing Agent	1	96,000	0.002
Employee Travel - Site Transportation Supt	2	264,000	0.006
Chief Accountant	1	192,000	0.004
Controller/ Accountant	1	138,000	0.003
Environmental Manager	1	192,000	0.004
Assistant Environmental Manager	1	144,000	0.003
Permitting Coordinator	1	144,000	0.003
Compliance Specialists	4	432,000	0.009
Permitting- Government Relations Coordinator	1	144,000	0.003
Public Relations Officer	1	144,000	0.003
Employee Training/Safety Manager	1	132,000	0.003
Staff Employees (Accounting, etc...)	8	576,000	0.012
TOTAL	**106**	**11,585,000**	**0.244**

Table 21.25 G&A Expenses

G&A Expense	Total Cost with Burden (US$/a)	Unit Cost (US$/t Milled)
Insurances	1,977,600	0.042
External Assays/Testing	1,186,600	0.025
Safety & Training	3,850,000	0.081
Medical Service/First Aid	197,800	0.004
Security Supplies	197,800	0.004
Legal Services - Allowance	200,000	0.004
Regulatory Compliance/Permits - Allowance	380,000	0.008
Consulting - Allowance	346,080	0.007
Small Vehicles	896,000	0.019
Site & Off-site Offices Expenses/General Administration	340,000	0.007
Corp Head Office Expense, incl. Manpower	4,280,400	0.090
Recruitment	300,000	0.006
Communications	200,000	0.004
Travel	381,600	0.008
Accounting Services, incl. Auditing	500,000	0.011
System Management/Computer Services/Rents/Furniture	1,330,000	0.028
Professional Associations	98,900	0.002
Accommodation/Camp Costs	9,357,900	0.197
Regional Taxes & Licences Allowance	1,483,200	0.031
Environmental Expenses	988,800	0.021
Crew Air Transportation	6,005,000	0.127
Bus Transportation	1,342,300	0.028
Warehouse	1,977,600	0.042
Miscellaneous	197,700	0.004
Total	**41,970,300**	**0.885**

21.2.6 Surface Services

The site service cost is estimated at US$0.32/t milled or about US$15.0 M/a. The estimate is based on similar projects in North America or in-house experience. The estimate, as shown in Table 21.26, includes:

- personnel – general surface services human power
- surface mobile equipment and light vehicle operations
- portable water and waste management
- general maintenance including yards, roads, fences, and building maintenance

- off-site operation expense
- building heating
- power supply
- avalanche control.

Table 21.26 Surface Service Cost Estimate

Surface Service	Personnel	Total Cost with Burden (US$/a)	Unit Cost (US$/t Milled)
Personnel			
Surface Foreman	2	240,000	0.005
Electrician-Surface Shops	2	230,000	0.005
Mechanic-Surface Shops	2	221,000	0.005
Carpenter-Surface Shops	1	98,000	0.002
Labourers-Yard/Surface Shops	4	302,000	0.006
Tunnel Maintenance	16	1,568,000	0.033
Surface Foreman-Offsite	2	240,000	0.005
Operators-Offsite	4	386,000	0.008
Helpers-Offsite	4	302,000	0.006
Sub-total Personnel	**37**	**3,587,000**	**0.076**
Expenses			
Potable Water/Waste Management			
Potable Water		49,400	0.001
Domestic Waste		197,800	0.004
Hazardous Waste		197,800	0.004
Sewage		29,700	0.001
Building Maintenance			
Supplies Operating		296,600	0.006
Supplies Repairs		296,600	0.006
Tool Allowance		197,800	0.004
Services Purchased		148,300	0.003
Small Vehicles/Equipment		383,700	0.008
Supplies		197,800	0.004
Building Heating		2,966,400	0.063
Tunnel Ventilation		257,500	0.005
Road Maintenance		2,472,000	0.052
Power Line Maintenance		257,500	0.005
Power Supply		1,841,500	0.039
Avalanche Control		1,375,300	0.029
Off-site Operation Expenses		206,000	0.004
Sub-total Expenses		**11,371,700**	**0.240**
Total	**37**	**14,958,900**	**0.315**

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

Tetra Tech prepared an economic evaluation of the KSM PFS based on a pre-tax financial model. For the 55-year mine life and 2,164 Mt reserve, the following pre-tax financial parameters were calculated using the base case metal prices:

- 11.5% IRR
- 6.2-year payback on US$5,256 M capital
- US$4,511 M NPV at 5% discount rate and US$1,614 M at 8% discount rate.

The base case prices, using the three-year trailing average (as of April 15, 2012) were as follows:

- gold – US$1,330/oz
- copper – US$3.45/lb
- silver – US$25.20/oz
- molybdenum – US$15.00/lb
- exchange rate – Cdn$1.00 to US$0.96.

Sensitivity analyses, along with two alternate metal price scenarios, were developed to evaluate the project economics.

The detailed financial model is provided in Appendix L.

22.2 PRE-TAX MODEL

Metal revenues projected in the KSM cash flow models were based on the average metal values indicated in Table 22.1.

Table 22.1 Metal Production from the KSM Project

	Years 1-7	Years 1-20	LOM
Total Tonnes to Mill (000s)	310,062	926,916	2,164,419
Annual Tonnes to Mill (000s)	44,295	46,346	39,353
Average Grades			
Gold (g/t)	0.79	0.67	0.549
Copper (%)	0.234	0.180	0.207
Silver (g/t)	2.385	2.737	2.740
Molybdenum (ppm)	46.2	61.4	44.8
Total Production			
Gold (000s oz)	5,959	15,003	27,959
Copper (000s lb)	1,364,880	3,024,655	8,075,101
Silver (000s oz)	14,712	50,154	120,826
Molybdenum (000s lb)	9,067	41,477	62,679
Average Annual Production			
Gold (000s oz)	851	750	508
Copper (000s lb)	194,983	151,233	146,820
Silver (000s oz)	2,102	2,508	2,197
Molybdenum (000s lb)	1,295	2,074	1,140

22.2.1 *Financial Evaluations – NPV and IRR*

The production schedule has been incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tonnage processed, head grades, and recoveries.

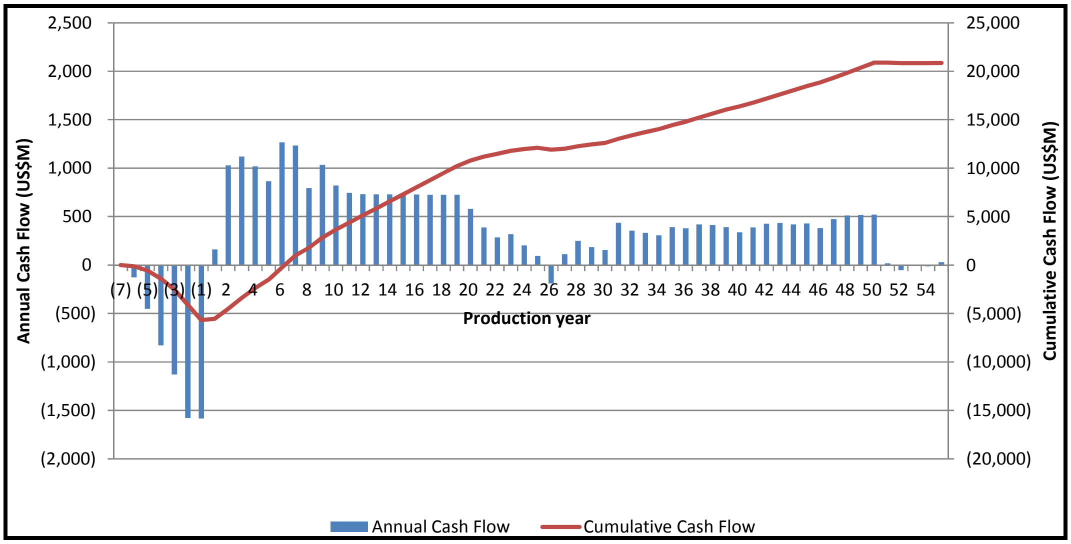
Gold revenues and additional metal credits were calculated based on market prices. Unit operating costs for mining, processing, site services, G&A, off-site charges (smelting, refining, transportation, and royalties), tailings storage and handling, water treatment, and energy recovery areas were applied to annual milled tonnages to determine the overall operating cost which was deducted from the revenues to derive annual operating cash-flow (Net Revenue).

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

Working capital is estimated at three months of the first year on-site operating cost and applied to the first year of expenditures. The working capital is recovered at the end of the mine life and aggregated with the salvage value contribution and applied towards reclamation during closure.

The undiscounted annual cash flows are illustrated in Figure 22.1.

Figure 22.1 Undiscounted Annual and Cumulative Cash-Flow



22.3 METAL PRICE SCENARIOS

In addition to the base case, two additional metal price scenarios were also developed: one using the spot metal prices on April 15, 2012, including the closing exchange rate of that day (Spot Price Case); the other using gold, copper, and silver prices 20% lower than the April 15 prices at the Base Case exchange rate (Alternate Case). The input parameters and results of all scenarios can be found in Table 22.2.

Table 22.2 Summary of the Economic Evaluations

	Unit	Base Case	Spot Price Case	Alternate Case
Metal Price				
Gold	US$/oz	1,330.00	1,650.00	1,320.00
Copper	US$/lb	3.45	3.75	3.00
Silver	US$/oz	25.20	32.00	25.60
Molybdenum	US$/lb	15.00	15.00	15.00
Exchange Rate	US:Cdn	0.96	1.00	0.96
Economic Results				
NPV (at 0%)	US$ M	20,473	31,160	16,776
NPV (at 3%)	US$ M	8,196	13,137	6,612
NPV (at 5%)	US$ M	4,511	7,748	3,503
NPV (at 8%)	US$ M	1,614	3,503	1,031
IRR	%	11.53	14.73	10.35
Payback	Years	6.19	5.16	6.68
Cash Cost/oz Au	US$/oz	141.30	60.04	263.54
Total Cost/oz Au	US$/oz	597.60	535.35	719.84

22.4 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the following parameters:

- gold, copper, silver and molybdenum metal prices
- exchange rate
- capital expenditure
- operating costs.

The analyses are presented graphically as financial outcomes in terms of NPV, IRR, and payback period. The project NPV is most sensitive to gold price and exchange rate followed by operating costs, copper price, capital costs, silver price, and molybdenum price. The IRR is most sensitive to exchange rate and gold price followed by capital costs, operating costs, copper price, silver price, and molybdenum price. The payback period is most sensitive to gold price and exchange rate followed by capital costs, copper price, operating costs, silver price, and molybdenum price. The NPV, IRR, and payback sensitivities can be seen in Figure 22.2, Figure 22.3, and Figure 22.4.

Figure 22.2 Sensitivity Analysis of NPV at 5% Discount Rate

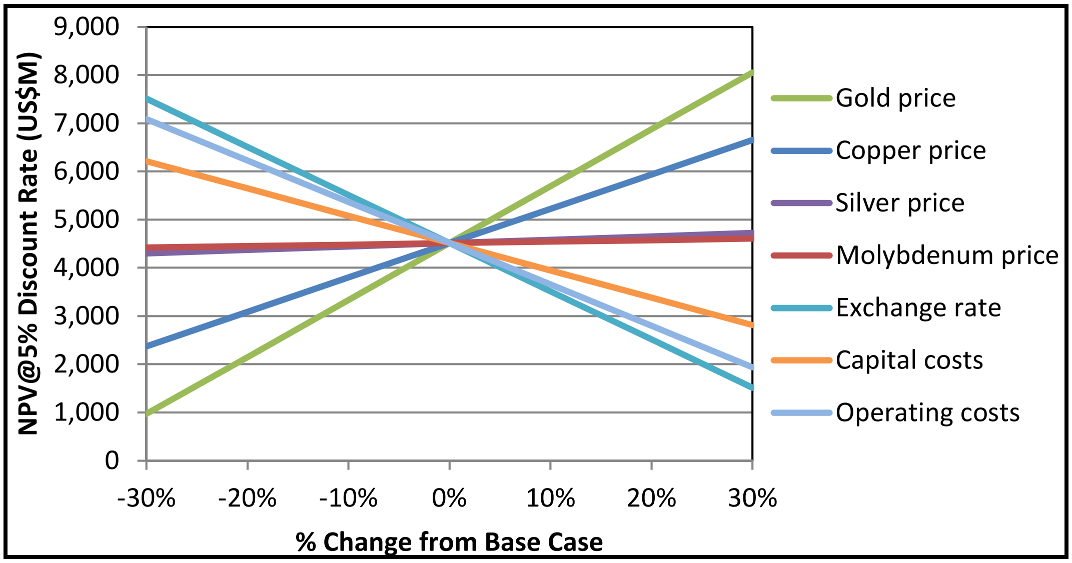


Figure 22.3 Sensitivity Analysis of IRR

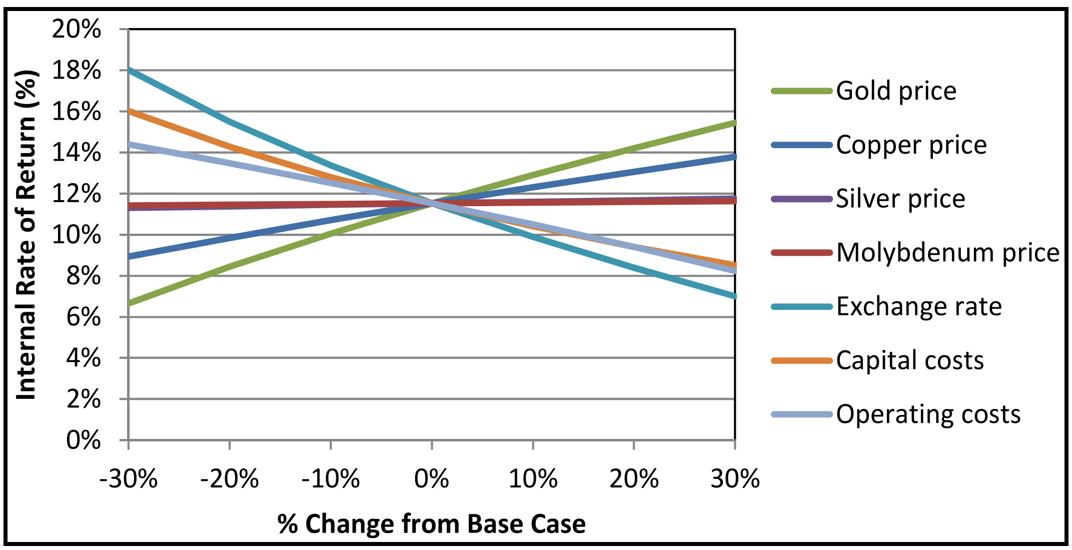
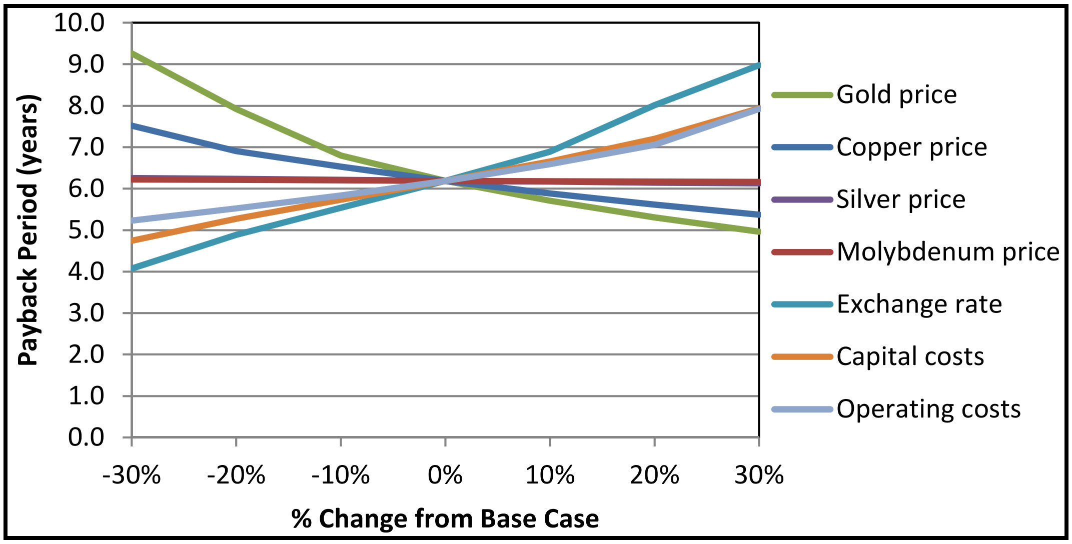


Figure 22.4 Sensitivity Analysis of Payback Period



22.5 ROYALTIES

KSM is subject to a royalty of 1% of the NSR payable to Barrick Gold Corp., capped at Cdn$4.5 M. The full amount is paid in Year 1 in the financial model.

22.6 SMELTER TERMS

In the absence of letters of interest or letters of intent from potential smelters or buyers of concentrate, Butterfield has provided smelter terms for delivery of copper concentrate to an East Asian smelter and molybdenum concentrate to a range of roasters.

Copper concentrate contracts will generally include payment terms as follows:

- **Copper** – pay 100% of content less 1.0 unit at the London Metal Exchange (LME) price for Grade A copper less a refining charge of US$0.075/accountable lb. The refining charge is not subject to price participation.
- **Gold** –gold payment varies according to gold content in concentrate; pay 97.75% on the gold content in excess of 30 g/dmt less a refining charge of $8/accountable troy oz from the LME price; lower gold contents are payable on a sliding scale to 90% payment at 1 g/dmt less a refining charge of $8/accountable troy oz from the LME price.
- **Silver** – pay 90% on the silver content in excess of 30 g/dmt less a refining charge of $0.50/accountable troy oz from the LME price.
- **Treatment Charge** – US$75/dmt of concentrate delivered.

- **Penalty Charge** – No penalty is applied according to concentrate assay data.

Molybdenum concentrate contracts will generally include payment terms as follows:

- **Molybdenum** – pay 99% of content at the LME price less a treatment charge of US$2.00/accountable lb.

22.7 Concentrate Transport Logistics

Transportation charges prepared by Tetra Tech for truck, rail, port handling, and ocean freight have been based on an average copper concentrate tonnage of 321,840 t/a (wet basis) and a molybdenum concentrate tonnage of 1,812 t/a.

Copper concentrate from the mine site will be transported by truck to Stewart Bulk Terminals. Initial capital will be required to upgrade the facility to meet KSM's requirements; these costs were estimated by BVL and Tetra Tech to be Cdn$79 M. Despite the initial capital requirements, this option was determined to be least expensive option due to the low transportation costs.

Transportation costs for the copper concentrate are listed below:

- trucking – Cdn$32.94/wmt
- port storage and handling – Cdn$18.00/wmt
- ocean transport to Asian port – US$67.99/wmt
- allowance – Cdn$2.35/wmt
- moisture content – 9%.

Molybdenum concentrate from the mine site was assumed to be loaded into 2-t bags and then transported by truck to Prince Rupert. The bags will then be loaded into containers and transferred to Fairview Terminal. Transportation costs for the molybdenum concentrate are listed below:

- trucking – Cdn$66.96/wmt
- port storage and handling – Cdn$15.63/wmt
- ocean transport to Asian port – US$88.93/wmt
- moisture content – 5%.

22.7.1 *Concentrate Transport Insurance*

An insurance rate of 0.15% was applied to the provisional invoice value of the concentrate to cover land-based and ocean transport from the mine site to the smelter.

22.7.2 OWNERS REPRESENTATION

For a 10,000 wmt shipment lot, a charge of US$5,000 was applied for services provided by the Owner's representative. Duties would include attendance during vessel unloading at the smelter port, supervising the taking of samples for assaying, and determining moisture content.

22.7.3 CONCENTRATE LOSSES

Concentrate losses are normally estimated at 0.06% per handling during shipment from the mine to smelter. For deliveries to Asia, an overall loss of 0.42% was applied to the provisional invoice value for the following seven handlings:

1. loading truck at mine
2. off-loading truck at railhead
3. reloading rail cars at railhead
4. off-loading railcars at port storage shed
5. loading vessel
6. off-loading vessel into truck transport to smelter
7. off-loading truck into smelter storage bins.

23.0 ADJACENT PROPERTIES

In 2010, Pretium purchased the Snowfield and Brucejack mineral resource properties from Silver Standard. In February 2011, Pretium announced an updated estimate of Mineral Resources for their Snowfield Project, which is located immediately east of Seabridge's Mitchell deposit. Table 23.1 summarizes the publicly-disclosed resources of the Snowfield Project, which were tabulated using a 0.30 g/t gold equivalent cut-off grade (Pretium, 2011b).

In April 2012, Pretium disclosed a new resource estimate for their Brucejack property, which is located east of Seabridge's KSM property. The Brucejack deposit was recently consolidated into two recognized mineralized zones: Valley of the Kings (VOK) and West Zone. Table 23.2 summarizes the publicly disclosed resources for the Valley of the Kings zone using a 5 g/t gold equivalent cut-off grade (Pretium, 2012). Table 23.3 summarizes resources for the West zone (Pretium, 2012).

Pretium recently completed (February 2012) a Preliminary Economic Assessment study for their Brucejack Project. Pretium announced that they plan on completing a 24,000 m drilling program in 2012 and have started a Feasibility Study, which is anticipated to be complete in the first quarter of 2013.

RMI has not verified the resources disclosed by Pretium for their Snowfield and Brucejack deposits. While there appear to be similarities between the Mitchell and Snowfield deposits, the Brucejack mineralization reported by Pretium is not necessarily indicative of mineralization found at the nearby Kerr, Sulphurets, Mitchell, and Iron Cap zones.

RMI has not verified the information shown in Table 23.1, Table 23.2, and Table 23.3. It is RMI's opinion that a portion of the mineralization shown in Table 23.1 is similar to mineralization associated with the Mitchell Zone because the Mitchell and Snowfield zones are located immediately adjacent to each other. However, RMI notes that there are distinct differences between the upper portion of the Snowfield mineralized system and the Mitchell Zone.

The mineralization shown in Table 23.2 and Table 23.3 is unlikely to be indicative of the mineralization currently recognized at the KSM property.

Table 23.1 Pretium Snowfield Mineral Resources Using a 0.30 g/t Cut-off

Resource Category	Mt	Au (g/t)	Ag (g/t)	Cu (%)	Mo (ppm)	Re (ppm)	Au oz (000)	Ag oz (000)	Cu lb (billions)	Mo lb (millions)	Re oz (millions)
Measured	189.8	0.82	1.69	0.09	97.4	0.57	4,983	10,332	0.38	40.8	3.5
Indicated	1,180.3	0.55	1.73	0.1	83.6	0.50	20,934	65,444	2.60	217.5	19.0
Measured + Indicated	1,370.1	0.59	1.72	0.1	85.5	0.51	25,917	75,776	2.98	258.3	22.5
Inferred	833.2	0.34	1.90	0.06	69.5	0.43	9,029	50,964	1.10	127.7	11.5

Table 23.2 Pretium Valley of the Kings Mineral Resources Using a 5 g/t Cut-off

Resource Category	Mt	Au (g/t)	Ag (g/t)	Au oz (millions)	Ag oz (millions)
Indicated	8.9	17.30	14.50	4.9	4.1
Inferred	12.7	25.50	11.60	10.4	4.7

Table 23.3 Pretium West Valley of Kings Mineral Resources Using a 5 g/t Cut-off

Resource Category	Mt	Au (g/t)	Ag (g/t)	Au oz (millions)	Ag oz (millions)
Measured	2.4	5.85	347.00	0.5	26.8
Indicated	2.5	5.86	190.00	0.5	15.1
Measured + Indicated	4.9	5.85	267.00	0.9	41.9
Inferred	4.0	6.44	82.00	0.8	10.6

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional information or explanation necessary to make the technical report understandable and not misleading.

25.0 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

This report was completed to assess the economic viability of the current KSM Project to a PFS level. Based on the work carried out in the 2012 KSM PFS for the Project, economic viability has been demonstrated to this study level.

The current study should be followed by the recommended studies referred to in Section 26.0 of this report, as well as a Feasibility Study, in order to assess the technical and economic viability of the Project.

The 2012 KSM PFS envisages a large tonnage open pit and underground block caving mining operation at a nominal rate of 130,000 t/d of ore fed to a flotation mill, which would produce a copper/gold/silver concentrate for transport by truck to the nearby deep-water sea port at Stewart, BC. A separate molybdenum concentrate and gold-silver doré will also be produced at the processing facility.

25.2 PROJECT ECONOMICS

Tetra Tech prepared an economic evaluation of the KSM PFS based on a pre-tax financial model. The model was evaluated for the 55-year mine life and 2,164 Mt reserve.

A base case economic evaluation was undertaken incorporating historical three-year trailing averages for metal prices as of April 15, 2012. Two additional metal price scenarios were also developed:

- Spot Price Case – uses the spot metal prices on April 15, 2012, including the closing exchange rate of that day
- Alternate Case – with metal prices 20% lower than the April 15 prices at the Base Case exchange rate.

The pre-tax financial model was established on a 100% equity basis, excluding debt financing and loan interest charges. The financial outcomes have been tabulated for NPV, IRR, payback of capital, and cost per ounce of gold. Discount rates of 8%, 5%, 3%, and non-discounted were applied to all scenarios. The results are presented in Table 25.1.

Table 25.1 Summary of Pre-Tax NPV, IRR and Payback by Metal Price Scenario

	Unit	Base Case	Spot Price Case	Alternate Case
Metal Price				
Gold	US$/oz	1,330.00	1,650.00	1,320.00
Copper	US$/lb	3.45	3.75	3.00
Silver	US$/oz	25.20	32.00	25.60
Molybdenum	US$/lb	15.00	15.00	15.00
Exchange Rate	US:Cdn	0.96	1.00	0.96
Economic Results				
NPV (at 0%)	US$ M	20,473	31,160	16,776
NPV (at 3%)	US$ M	8,196	13,137	6,612
NPV (at 5%)	US$ M	4,511	7,748	3,503
NPV (at 8%)	US$ M	1,614	3,503	1,031
IRR	%	11.53	14.73	10.35
Payback	Years	6.19	5.16	6.68
Cash Cost/oz Au	US$/oz	141.30	60.04	263.54
Total Cost/oz Au	US$/oz	597.60	535.35	719.84

Notes:
Operating and total costs per ounce of gold are after base metal credits.
Total costs per ounce include all start-up capital, sustaining capital and reclamation/closure costs.

25.3 PROJECT RISKS

The Project viability is subject to numerous risks. A full risk assessment is planned for Q4 2012 for the KSM Project. This assessment will include risk reviews of safety, environmental, technical, business, commercial, and project logistic issues. These reviews will identify and assess relevant risks and determine mitigation strategies. A formal risk management program will commence during the Feasibility Study phase and will continue through commissioning. The project team will periodically review risks and opportunities and take appropriate action to minimize the impact on overall costs and scheduling.

26.0 RECOMMENDATIONS

26.1 INTRODUCTION

This section outlines the areas to investigate for project improvements. A high-level budgetary estimate for the completion of each recommended item is provided in Canadian dollars.

26.2 MINING

26.2.1 OPEN PIT

MMTS recommends the project proceed to further mine planning for the open pit with the following specific studies. These recommendations are not necessarily contingent on positive results from previous phases but reflect the ongoing level of detail required to advance the project, leading to eventual construction and operation level designs. Mine planning work discussed in this section will cost between $500,000 and $750,000 depending on the results of future exploration and geology modeling, pit geotechnical studies, bench marking studies, mine waste management studies, mine reclamation planning, and closure planning.

Specific mine plan recommendations for the ongoing studies are as follows:

- The Mitchell pit wall is a 1250 m wall height; the design has been reviewed by an expert panel and is deemed technically feasible. For management and investment assurance, the resultant pit design should be benchmarked with other operations and projects around the world to gain some comparative experience from current industry practice. Information gained from the benchmarking exercise can be used in improving the operability of the designs for the ongoing mine studies.
- The following work items represent opportunities to reduce mine operating and capital costs based on the fact that some aspects of this PFS are conservative.
    - Review the details of the ABA modelling and the conservative assumption for all inclusive water treatment. More detailed evaluations of the Waste and Water management plans may show that the same performance can be achieved with less cost.

- Review, in detail, the capital cost estimates of the mine access and haul roads with an experienced contractor to reduce the conservative construction and cost contingencies in the current design.
- The 2012 KSM PFS uses conservative truck cycle times for waste hauls. When the waste management plan has been optimized, the truck haul cycles should be simulated for optimization of the mining equipment fleet. This should include cycle time details, shovel and fleet interaction analysis, and fuel consumption details especially with the high proportion of downhill loaded hauling. Some of this level of detail may be provided by the equipment vendors during commercial negotiations.
- Investigate the increased use of higher lift dumps using top down methods to significantly reduce mining costs, as foundation investigation allows.
- The complete impact of phase sizes in Mitchell on the production schedule has not been investigated thoroughly. A full optimization study using updated haul cycles for each scenario should be done on different phase size scenarios in Mitchell. This work would not affect the size of the ultimate pit but may change the size of the intermediate phases.
- Truck manufacturers are considering the cost impacts of offering LNG power haul trucks. Alternate energy sources including LNG and catenary assist can reduce the mine haul costs significantly. Analysis of alternative power from the manufacturers needs to be evaluated at the next level of study.

- The 2012 KSM PFS has some aspects that require more detailed work to meet the assurance required for higher level studies. The following specific issues have been identified:
  - The KSM Project is located in the west coast snow belt, and in confined terrain that will have significant impact on open pit operating conditions, productivities, and available operating days. Weather and avalanche study information has been updated and needs to be evaluated and incorporated to refine the design parameters for future construction and operations level design work.
  - The blast hole drill productivity for the 2012 KSM PFS is based on typical values for the type of rocks in the project area. Samples of specific rock types from KSM need to be evaluated for advanced levels of planning. Rock samples for the major rock types are sent for evaluation to prospective drill manufacturers along with local operating conditions such as elevation, precipitation, and climate.
  - The 2012 KSM PFS LOM production schedule shows significant fluctuation in shovel utilization and truck fleet size resulting in excess equipment capacity. This has not been optimized since the pit design and mine waste management plan may change significantly with revised pit phase designs, RSF designs, and production schedules. Smoothing

of these fleets needs to be addressed (this will be included in future mine planning work).

The open pit mine planning design will need to be reworked to construction and operations level of detail in future studies. This will be based on updated production targets, and CAPEX/OPEX inputs, with updated Underground designs, and the incorporation of results from detailed production targets and requirements and benchmarking studies as the project approaches operations level of detail.

26.2.2 UNDERGROUND

Based on the results of the combined open pit/underground consolidated plan, the resultant underground potential will need to be upgraded to feasibility level.

In situ stresses have been estimated from hydraulic fracturing tests and, based on high induced stresses in the cave back as predicted by numerical modelling. It is expected that stress-induced fracturing of the Mitchell rock mass may contribute to caving. More sophisticated numerical analyses are recommended to confirm and quantify stress-related impacts as part of future studies.

This underground work is projected to cost around $1.5 to $2.0 M.

26.3 PIT SLOPES

Some additional field work, including geotechnical drilling, hydrogeological testing and laboratory testing are recommended by BGC for the proposed pits in the Mitchell, Sulphurets, and Kerr zones at the next stage of study. It is expected that additional work will be completed to further refine the geological interpretation in each zone by Seabridge, including the location of faults and alteration zones. Future work for the open pit slope design and hydrogeological evaluations will be focused on increasing the confidence level of the slope designs and slope depressurization plans. It is estimated that the next stage of combined geotechnical and hydrogeology work for the KSM Project will require a budget of approximately $5 M.

In addition, BGC recommends the following specific work, expected to cost an additional $5 to $6 M:

- Long term pumping tests of the rock mass at each of the three proposed open pits using 6″ to 10″ diameter wells to provide bulk estimates of rock mass hydraulic conductivity and provide data for the optimization of the slope depressurization plan.
- In-situ stress testing in the Mitchell valley to provide an estimate of the pre-mining stress state for use in numerical modelling.
- Numerical stress/deformation modelling of the proposed Mitchell pit, specifically the North and South slopes.

- Refinement of the design for the proposed Mitchell north slope dewatering adit. The proposed adit is to be integrated with the water management plan for the Mitchell Glacier outflows.
- A risk assessment for potential water into the pit over the Mitchell pit east wall from the Mitchell Glacier diversion tunnel. The results of this risk assessment should be used to guide the design of ramps on the final east wall of the Mitchell pit. This assessment will also assist with evaluating integration of the Mitchell pit north wall dewatering adit into the water management plan.
- Deformation monitoring of the Snowfield and Kerr landslides and studies to evaluate the effects on mining on these slope deformation features. Monitoring may be accomplished via INSAR or ground based techniques. A monitoring program should be initiated for these landslides as soon as practical.

26.4 METALLURGICAL TESTING

Tetra Tech recommends further metallurgical test work to optimize process conditions and to establish design-related parameters for the next stage of study. Tetra Tech makes the following recommendations:

- Additional metallurgical test work and mineralogical evaluations should be conducted to optimize process conditions and to establish design-related parameters for the next stage of study. The test work should include variability testing of samples from Sulphurets, Kerr, and Iron Cap zones. The cost of the test work is estimated at $600,000.
- The potential additional value of rhenium in the molybdenum concentrate should be evaluated ($100,000).
- Further study should be conducted to optimize the proposed cyanide recovery and destruction methods ($100,000).
- A metallurgical laboratory test program should be performed on ore composites representing each year of the initial 10 years of open pit mine production from Mitchell and Sulphurets ($200,000) according to the updated mining plan.
- Further study into economical water treatment methods is recommended for water from the CIL pond ($200,000).

26.5 ACCESS ROADS

26.5.1 MAIN ACCESS ROAD NETWORK

The revisions to the Coulter Creek Access Road that were recommended as a result of the environmental and geotechnical assessments conducted in 2009 and 2010 were incorporated, where feasible, in the current designs. Subsequently, significant sections of the road were realigned by McElhanney during 2011 to improve the road geometry.

Reassessment of the realigned sections will be required by BGC and Rescan, towards ensuring the final road designs meet environmental, geotechnical, and archaeological requirements. Additional work will be required on the road and structure design packages to meet the requirements for Ministry of Forests SUP application.

The Treaty Creek and North Treaty/Teigen Creek valley access road(s) preliminary designs are based on available information. Designs are based on geohazard and risk assessment reporting by BGC and preliminary environmental reporting by Rescan. More detailed environmental investigation is anticipated during the 2012 season, including fisheries evaluations of all stream crossings. Alignment and crossing structure selection might need to be revised.

McElhanney has identified some potential opportunities for improvements to the Treaty Creek road alignment at select locations. These should be looked at early in the 2012 field season, prior to significant work being undertaken by other consultants along the currently proposed routes.

The majority of the bridge and major stream crossing structure general arrangement design drawings have been prepared based on site surveys completed previously. Most are sufficiently detailed for inclusion in SUP applications, and to the Department of Fisheries and Oceans Navigable Waters Protection Branch where required.

Hydrological issues have been taken into consideration for each drainage structure. Currently only a few sites are identified as needing re-survey/design, as a result of proposed road alignment changes made since the 2011 field work. Additional detailed assessments by Rescan in the Treaty Creek Valley, North Treaty/Teigen Creek valley, and at the realigned Coulter Creek Access Road sections might also prompt additional work requirements by McElhanney.

Current preliminary road designs are made without the benefit of any sub-surface geotechnical investigation. During construction activities conditions shall be monitored by suitably qualified geo-technical professionals, and prescriptions made with respect to foundations, slope stability, cut/fill slopes, and other construction related concerns.

26.5.2 TEMPORARY WINTER ACCESS ROAD

Preliminary findings indicate that it is feasible to construct and use a winter haul road into the site. Numerous risks will also have to be considered to ensure that road construction and equipment haul can be carried out in a safe and environmentally-appropriate manner.

Trained avalanche specialists should examine the route and develop a plan for avalanche control. This study is expected to cost around $50,000.

Rock fall hazards should be examined by a rock mechanics specialist along the worrisome portions of the route, especially where evidence of relatively recent rock fall exists. This study is expected to cost around $25,000.

A GPR survey could be carried out on the glacier during the summer to determine if any significant voids, caverns, or meltwater channels are present that could be a hazard to the road construction, hauling equipment, and operators. An initial GPR survey should be conducted immediately before the road is constructed and, as a safety precaution, at regular intervals during its operation ($100,000).

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APPENDIX A

CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF QUALIFIED PERSON

I, R.W. Parolin, of Prince George, BC, do hereby certify:

- I am a Senior Project Engineer with McElhanney Consulting Services Ltd. with a business address at 1633 First Ave, Prince George, BC, V2L 2Y8.
- This certificate applies to the technical report entitled "KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012", with a date of June 22, 2012 (the "Technical Report").
- I am a graduate of the University of New Brunswick, (B.Sc., Forest Engineering, 1974).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#11134).
- My relevant experience is 31 years of location, survey, design, and construction of roads in the Forestry, Mining, and Oil & Gas sectors.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on June 21, 2008, and during the summers of 2009, 2010, and 2011.
- I am responsible for matters relating to permanent access roads in Sections 1.20.1, 18.14, and their associated costs included in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008", dated December 22, 2008, the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009" dated September 8, 2009, the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study" dated March 31, 2010, and the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 18th day of June, 2012 at Prince George, BC

"Original document signed and sealed by R.W. Parolin, P.Eng."

R.W. Parolin, P.Eng.
Senior Project Engineer
McElhanney Consulting Services Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Darby Kreitz, of Prince George, BC, do hereby certify:

- I am President of Allnorth Consultants Limited with a head office business address at 2011 PG Pulpmill Road, Prince George, BC, V2L 4V1.
- This certificate applies to the technical report entitled “KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012”, with a date of June 22, 2012 (the “Technical Report”).
- I am a graduate of the University of Saskatchewan (B.Sc. Civil Engineering, 1992).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#22775).
- My relevant experience is with the construction costs analysis for civil and earth embankment structures.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was on April 14, 2010.
- I am responsible for matters relating to the costs of the water storage dam, tailing starter dam, and tailing management facility in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008”, dated December 22, 2008, the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009” dated September 8, 2009, the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study” dated March 31, 2010, and the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update” dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 25th day of June, 2012 at Prince George, BC

“Original document signed and sealed by Darby Kreitz, P.Eng.”

Darby Kreitz, P.Eng.
President
Allnorth Consultants Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, J. Graham Parkinson, of Vancouver, BC, do hereby certify:

- I am a Geoscientist with Klohn Crippen Berger Ltd. with a business address at #500-2955 Virtual Way, Vancouver, BC, V7M 4X6.
- This certificate applies to the technical report entitled “KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012”, with a date of June 22, 2012 (the “Technical Report”).
- I am a graduate of the University of British Columbia, (Bachelors Degree – Physics, 1978), and the University of Alberta (Special Certificate – Geophysics, 1984).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#19008), and the Association of Professional Engineers, Geologists, and Geophysicists of Alberta (Member 36393).
- My relevant experience includes 15 years with Klohn Crippen and Klohn Crippen Berger engaged in the evaluation and development of mine waste facilities; involvement in over 20 mine waste facilities; involvement in a number of Environmental Impact Assessments, Baseline Studies, Mine Waste Facility Site Investigations, and the design of several major mine tailings dams; and 8 years experience in engineering and exploration geophysics for the mining, engineering, and petroleum industries.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was from July 26 to August 2, 2010, as well as during the summers of 2008 and 2009, and autumn of 2007.
- I am responsible for Sections 1.11, 1.17, and 18.1, as well as matters relating to geotechnical parameters for the RSFs, water diversions, seepage collection ponds, TMF, water treatment facility, and associated capital, operating, sustaining, reclamation and closure costs in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008”, dated December 22, 2008, the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009” dated September 8, 2009, the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study” dated March 31, 2010, and the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update” dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June 2012 at Vancouver, BC

“Original document signed and sealed by J. Graham Parkinson, P.Geo.”

J. Graham Parkinson, P.Geo.
Geoscientist
Klohn Crippen Berger Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Harold Bosche, of Richmond, BC, do hereby certify:

- I am President of Bosche Ventures Ltd. with a business address at 10111 Craig Court, Richmond, BC, V6X 3J8.
- This certificate applies to the technical report entitled “KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012”, with a date of June 22, 2012 (the “Technical Report”).
- I am a graduate of the University of Saskatchewan, (B.Sc. Mechanical Engineering, 1965).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#14934).
- My relevant experience is with respect to conveyors, piping systems and infrastructure includes 41+ years of experience in feasibility, design, and construction of facilities for mineral processing.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was on September 16, 2008.
- I am responsible for Sections 1.8, 1.20.3, 1.23, 18.2 to 18.11, 18.17, 18.18, and for matters relating to the infrastructure and process layouts, rope conveyor, overland tunnel and process conveyors, tailing delivery, reclaim pumping and piping systems, construction schedule and execution, and associated capital costs in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008”, dated December 22, 2008, the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009” dated September 8, 2009, the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study” dated March 31, 2010, and the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update” dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June 2012 at Vancouver, BC

“Original document signed and sealed by Harold Bosche, P.Eng.“

Harold Bosche, P.Eng.
President
Bosche Ventures Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, James H. Gray, of Calgary, Alberta, do hereby certify:

- I am a Mining Engineer with Moose Mountain Technical Services with a business address at 1975 1st Avenue South, Cranbrook, BC, V1C 6Y3.
- This certificate applies to the technical report entitled "KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012", with a date of June 22, 2012 (the "Technical Report").
- I am a graduate of the University of British Columbia, (Bachelor of Applied Science – Mineral Engineering, 1975).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#11919), and the Association of Professional Engineers, Geologists, and Geophysicists of Alberta (Member #M47177).
- My relevant experience includes operation, supervision, and engineering in North America, South America, Australia, Eastern Europe, and Greenland.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on April 13, 2010.
- I am responsible for Sections 1.9 and 1.14 (relating to open pit), 1.12, 15.1 and 15.2 (relating to open pit), 16.1 and 16.2 (relating to open pit), 26.2.1, and for costs relating to open pit mine operating, open pit mine capital (initial and sustaining), and open pit development (initial and sustaining) in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008", dated December 22, 2008, the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009" dated September 8, 2009, the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study" dated March 31, 2010, and the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information, and belief, the technical report, within my sections of responsibility referred to above, contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June 2012 at Calgary, AB

"Original document signed and sealed by James H. Gray, P.Eng."

James H. Gray, P.Eng.
Principal Mining Engineer
Moose Mountain Technical Services

CERTIFICATE OF QUALIFIED PERSON

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

- I am a Senior Metallurgist with Tetra Tech-Wardrop with a business address at #555-800 W. Hastings St., Vancouver, BC, V6B 1M1.
- This certificate applies to the technical report entitled "KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012", with a date of June 22, 2012 (the "Technical Report").
- I am a graduate of North-East University, China (B.Eng., 1982), Beijing General Research Institute for Non-ferrous Metals, China (M.Eng., 1988), and Birmingham University, United Kingdom (Ph.D., 2000).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30898).
- My relevant experience includes over 29 years involvement in mineral processing for base metal ores, gold and silver ores, and rare metal ores.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was September 16, 2008.
- I am responsible for Sections 1.15, 1.16, 1.21, 2.0, 3.0, 13.0, 17.0, 18.16, 19.0, 24.0, 25.1, 25.3, and 26.4, and for matters pertaining to process capital costs in Sections 1.24 and 21.1, and process, G&A, and site services operating costs in Sections 1.25 and 21.2 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008", dated December 22, 2008, the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009" dated September 8, 2009, the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study" dated March 31, 2010, and the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June, 2012 at Vancouver, BC

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

Jianhui (John) Huang, P.Eng.
Senior Metallurgist
Tetra Tech-Wardrop

CERTIFICATE OF QUALIFIED PERSON

I, Kevin Jones, of St. Albert, Alberta, do hereby certify:

- I am the Vice President – Arctic Development with EBA Engineering Consultants Ltd. with a business address at 14940, 123 Ave NW, Edmonton, Alberta, T5V 1B4.
- This certificate applies to the technical report entitled "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update 2011", with a date of (the "Technical Report").
- I am a graduate of Lakehead University, (B.Eng. Civil, 1981).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of Alberta (#M34713) and the Northwest Territories (#L341).
- My relevant experience includes over 29 years of geotechnical engineering on a variety of arctic resource based projects. The bulk of the work has focused on the design of infrastructure for these projects, including all-weather and seasonal access roads. Specific involvement on the world's longest and most advanced winter road, the Tibbett to Contwoyto Winter Road (TCWR) in Northwest Territory, is applicable to this project. I have also selected routes and evaluated temporary winter access roads for a mining project in Nunavut (100 km) and an oil field development project in northwestern Siberia, Russia (200 km). I have also been involved in overseeing the evaluation of drilling pads for support of exploration drilling rigs on floating ice covers and the use of offshore ice roads in the Alaskan and Canadian Beaufort Sea.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was from September 12 to 13, 2011.
- I am responsible for Sections 1.20.2, 18.15, and 26.5.2, and for matters relating to temporary winter access roads and associated costs in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008", dated December 22, 2008, the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009" dated September 8, 2009, the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study" dated March 31, 2010, and the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June 2012 at Edmonton, Alberta

"Original document signed and sealed by Kevin Jones, P.Eng."

Kevin Jones, P.Eng.
Vice President, Arctic Development
EBA Engineering Consultants Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Michael J. Lechner, of Stites, ID, USA, do hereby certify:

- I am an independent consultant and owner of Resource Modeling Inc., an Arizona Corporation, with a business address at 124 Lazy J Drive, PO Box 295, Stites, ID, 83552.
- This certificate applies to the technical report entitled "KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012", with a date of June 22, 2012 (the "Technical Report").
- I am a graduate of the University of Montana, (B.A. Geology, 1979).
- I am a registered professional geologist in the State of Arizona (#37753), a Certified Professional Geologist with the AIPG (#10690), and a P.Geo. with the Province of British Columbia (#155344).
- My relevant experience includes over 33 years of work as an exploration geologist, mine geologist, engineering superintendent, and resource estimator for a variety of precious metal deposits located around the world.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was from August 30 to September 1, 2011.
- I am responsible for Sections 1.2 to 1.7, 5.0 to 12.0, 14.0, and 23.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the following reports:
  - "Mitchell Creek Technical Report, Northern British Columbia, NI 43-101 Technical Report", April 6, 2007
  - "Kerr-Sulphurets Technical Report, Northern British Columbia, NI 43-101 Technical Report" February 29, 2008
  - "Updated Mitchell Creek Technical Report, Northern British Columbia" dated March 27, 2008.
  - "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008", dated December 22, 2008.
  - "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009" dated September 8, 2009
  - "January 2010 Updated KSM Mineral Resources" dated January 25, 2010
  - "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study" dated March 31, 2010
  - "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011.

- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22 day of June 2012 at Stites, Idaho

"Original document signed and sealed by Michael J. Lechner, P.Geo."

Michael J. Lechner, P.Geo.
President
Resource Modeling Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Neil Brazier, of Richmond, BC, do hereby certify:

- I am a Principal with WN Brazier Associates Inc. with a business address at #8–3471 Regina Ave., Richmond, BC.
- This certificate applies to the technical report entitled “KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012”, with a date of June 22, 2012 (the “Technical Report”).
- I am a graduate of the University of Saskatchewan (B.Sc. Electrical Engineering, 1969) and I have practiced my profession continuously since graduation.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#8337).
- My relevant experience includes engineering, construction supervision, and commissioning of a large number of diesel and combustion turbine power plants, high voltage transmission lines and substations for mining applications.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was September 12 to 16, 2011.
- I am responsible for Sections 1.22, 18.12, 18.13, and for costs relating to permanent electrical power and energy recovery plants in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008”, dated December 22, 2008, the “Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009” dated September 8, 2009, the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study” dated March 31, 2010, and the “Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update” dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June 2012 at Vancouver, BC

“Original document signed and sealed by Neil Brazier, P.Eng.“

Neil Brazier, P.Eng.
Principal
WN Brazier Associates Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Pierre Pelletier, of Vancouver, British Columbia, do hereby certify:

- I am an Environmental Engineer with Rescan Environmental Services Ltd. with a business address at 600 – 1111 West Hastings St., Vancouver, British Columbia, V6E-2J3.
- This certificate applies to the technical report entitled "KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012", with a date of June 22, 2012 (the "Technical Report").
- I am a graduate of University of Montana, Montana College of Mineral Science and Technology, (Environmental Engineering, 1992).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #27928.
- My relevant experience was gained over 20 years working in mining and the environment. I have experience managing Environmental and Social Impact Assessments, permitting treatment plants and mine closure plans, leading due diligences and environmental audits and the environmental and social aspects of several Preliminary Economic Assessments, Pre-Feasibility and Feasibility Studies.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on May 16, 2012
- I am responsible for Sections 1.18 and 20.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June 2012 at Vancouver, British Columbia.

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

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

CERTIFICATE OF QUALIFIED PERSON

I, Ross David Hammett, of Burnaby, BC, do hereby certify:

- I am a Senior Engineer and Principal with Golder Associates Ltd. with a business address at: 500 - 4260 Still Creek Drive, Burnaby BC, Canada V5C 6C6.
- This certificate applies to the technical report entitled “KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012”, with a date of June 22, 2012 (the “Technical Report”).
- I am a graduate of James Cook University of North Queensland (Ph.D., 1976; M.Eng.Sc, 1972.; B.E Civil, 1970).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License # 11020.
- I have 40 years of experience in mining and civil engineering. I have provide consulting services for more than 150 underground mining projects and has provide services related to mine planning, mining method selection, mine design, geotechnical studies, support designs, blasting, backfill, caving mechanics, rock stress control, geohydrology, mine dewatering, mining systems, mining automation, and environmental aspects of mining.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was on October 18 and 19, 2011.
- I am responsible for Sections 1.13, 16.3, 26.2.2, and those portions of Sections 1.9, 1.14, 15.1, and 16.2 related to block caving, and for costs relating to block caving operating and capital costs (initial and sustaining) in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day June 2012 at Burnaby BC.

“Original document signed and sealed by Ross D. Hammett, Ph.D., P.Eng.

Ross D. Hammett PhD, P.Eng
Senior Engineer & Principal
Golder Associates Ltd.

CERTIFICATE OF QUALIFIED PERSON

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

- I am a Senior Mining Engineer with Tetra Tech-Wardrop with a business address at 800-555 West Hastings St., Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled “KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012”, with a date of June 22, 2012 (the “Technical Report”).
- I am a graduate of Assiut University (B.Sc. Mining Engineering, 1991; M.Sc. Mining Engineering, 1996; Ph.D. in Mineral Economics, 2000).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License # 34975.
- My relevant experience is mine evaluation, with more than 19 years of experience in the evaluation of mining projects, advanced financial analysis, and mine planning and optimization. My capabilities range from conventional mine planning and evaluation to the advanced simulation-based techniques that incorporate both market and geological uncertainties. I have been involved in technical studies of several base metals, gold, coal, and aggregate mining projects in Canada and abroad.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- I have not conducted a personal inspection of the Property.
- I am responsible for Section(s) 1.26, 22.0, and 25.2 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June 2012 at Vancouver, British Columbia

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

Sabry Abdel Hafez, Ph.D., P.Eng.
Senior Mining Engineer
Tetra Tech-Wardrop

CERTIFICATE OF QUALIFIED PERSON

I, Tony Wachmann, of West Vancouver, British Columbia, do hereby certify:

- I am a Director of Mining, Metallurgy, and Infrastructure, with Stantec Consulting Ltd. with a business address at 1100 – 111 Dunsmuir St., Vancouver, British Columbia.
- This certificate applies to the technical report entitled “KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012”, with a date of June 22, 2012 (the “Technical Report”).
- I am a graduate of Queens University, (BSc., Civil Engineering, 1975).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (License #13023), APEGGA, APEY, NAPEG, and APEGS.
- My relevant work experience over 35 years' has been in engineering for mining, mineral processing, infrastructure, chemical, and industrial facilities. The work includes prefeasibility and feasibility studies, trade-offs, technical and financial audits, and project evaluations, as well as detailed design, procurement, logistics and project/construction management.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was on May 16, 2012.
- I am responsible for matters relating to tunnelling in Section 1.23 and for all costs related to tunnelling in Sections 1.0 and 21.0 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day June 2012 at West Vancouver, British Columbia.

“Original document signed and sealed by Tony Wachmann, P.Eng.”

Tony Wachmann, P.Eng
Director Mining, Metallurgy and Infrastructure
Stantec Consulting Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Warren Newcomen, of Kamloops, BC, do hereby certify:

- I am a Senior Geological Engineer with BGC Engineering Inc. with a business address at #234 St Paul Street, Kamloops, BC, V2C 6G4.
- This certificate applies to the technical report entitled "KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update 2012", with a date of June 22, 2012 (the "Technical Report").
- I am a graduate of the University of British Columbia (B.A.Sc., 1985) and the University of California at Berkeley (1990).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#16123).
- My relevant experience with respect to pit slope designs and slope stability includes design work for the following projects: Cortez Hills Project, Nevada; Donlin Creek Project, Alaska; Galore Creek Project, BC; Golden Bear Project, BC; Goldstrike Mine, Nevada; Palabora Mine, South Africa; New Afton Project, BC; Ajax Project, BC.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was from July 26 to 28, 2010.
- I am responsible for Sections 1.19, 16.4, and 26.3 of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had involvement with the property that is the subject of the Technical Report, in acting as a Qualified Person for the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008", dated December 22, 2008, the "Kerr-Sulphurets-Mitchell Preliminary Economic Assessment Addendum – 2009" dated September 8, 2009, the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study" dated March 31, 2010, and the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 22nd day of June 2012 at Kamloops, BC

"Original document signed and sealed by Warren Newcomen, M.S., P.Eng."

Warren Newcomen, M.S., P.Eng.
Senior Geological Engineer
BGC Engineering Inc.