Report to:

Seabridge Gold Inc.

KSM PROJECT
SEABRIDGE GOLD

Northwestern British Columbia, Canada

2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment

Document No. 735-1552880100-REP-R0002-02

TETRA TECH

Report to:

SEABRIDGE GOLD INC.

KSM PROJECT
SEABRIDGE GOLD

NORTHWESTERN BRITISH COLUMBIA, CANADA

2016 KSM (KERR-SULPHURETS-MITCHELL) PREFEASIBILITY STUDY UPDATE AND PRELIMINARY ECONOMIC ASSESSMENT

EFFECTIVE DATE: OCTOBER 6, 2016

Prepared by Hassan Ghaffari, P.Eng., Tetra Tech, Inc.
Jianhui (John) Huang, Ph.D., P.Eng., Tetra Tech, Inc.
Kevin Jones, P.Eng., Tetra Tech, Inc.
James H. Gray, P.Eng., Moose Mountain Technical Services
Ross Hammett, Ph.D., P.Eng., Golder Associates Ltd.
R.W. Parolin, P.Eng., McElhanney Consulting Services Ltd.
Derrek Kinakin, M.Sc., P.Geo., P.G., BGC Engineering Inc.
Michael J. Lechner, P.Geo., RPG, CPG, Resource Modeling Inc.
J. Graham Parkinson, P.Geo., Khlon Crippen Berger Ltd.
Pierre Pelletier, ERM Consultants Canada Ltd.
Neil Brazier, P.Eng., WN Brazier Associates Inc.
Simon Allard, P.Eng., Amec Foster Wheeler Americas Limited
Tony Lipiec, P.Eng., Amec Foster Wheeler Americas Limited
Mark Ramirez, RM, SME, Amec Foster Wheeler E&C Services Inc.

JS/vc

TETRA TECH
Suite 1000, 10th Fl, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5
Phone: 604.408.3788 Fax: 604.684.6241

IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for Seabridge Gold Inc. (Seabridge) by Tetra Tech, Inc., Moose Mountain Technical Services, Golder Associates Ltd., BGC Engineering Inc., Resource Modeling Inc., McElhanney Consulting Services Ltd., Klohn Crippen Berger Ltd, ERM Consultants Canada Ltd, WN Brazier Associates Inc., and Amec Foster Wheeler Americas Limited, collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Seabridge subject to the respective terms and conditions of its contracts with the individual Report Authors. Those contracts permit Seabridge to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to provincial and territorial securities law. Except for the purposes legislated under Canadian provincial and territorial securities law, any other use of this report by any third party is at that party's sole risk.

TABLE OF CONTENTS

APPENDICES

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

LIST OF TABLES

LIST OF FIGURES

GLOSSARY

UNITS OF MEASURE

ampere	A
annum (year)	a
bank cubic metres	bcm
billion tonnes	Bt
billion	B
centimetre	cm
Coefficients of Variation	CVs
cubic centimetre	cm^3
cubic metre	m^3
day	d
days per week	d/wk
days per year (annum)	d/a
dead weight tonnes	DWT
degree	°
degrees Celsius	°C
diameter	ø
dollar (American)	US$
dollar (Canadian)	Cdn$
dry metric tonne	dmt
foot	ft
gallon	gal
gallons per minute (US)	gpm
gigawatt hours	Gwh
gigawatt	GW
gram	g
grams per litre	g/L
grams per tonne	g/t
gravitational constant	*g*
greater than	>
hectare	ha
horsepower	hp
hour	h
hours per day	h/d
hours per week	h/wk
hours per year	h/a
inch	in
kilo (thousand)	k
kilogram	kg
kilograms per cubic metre	kg/m^3

kilograms per day	kg/d
kilograms per hour	kg/h
kilograms per square metre	kg/m^2
kilometre	km
kilometres per hour	km/h
kilonewton	kN
kilotonne	kt
kilovolt	kV
kilowatt hour	kWh
kilowatt hours per tonne	kWh/t
kilowatt hours per year	kWh/a
kilowatt	kW
less than	<
litre	L
litres per hour	L/h
litres per second	L/s
megavolt-ampere	MVA
megawatt	MW
metre	m
metres above sea level	masl
metres per second	m/s
metres per year	m/a
microns	µm
milligram	mg
milligrams per litre	mg/L
millilitre	mL
millimetre	mm
million tonnes	Mt
million	M
minute (plane angle)	'
minute (time)	min
month	mo
ounce	oz
parts per billion	ppb
parts per million	ppm
percent	%
pound(s)	lb
revolutions per minute	rpm
second (plane angle)	"
second (time)	s
specific gravity	SG
square centimetre	cm^2
square kilometre	km^2
square metre	m^2
square millimetre	mm^2

three-dimensional	3D
tonne (1,000 kg) (metric ton)	t
tonnes per day	t/d
tonnes per hour	t/h
tonnes per year	t/a
troy ounce	troy oz
volt	V
week	wk
weight/weight	w/w
wet metric ton	wmt

ABBREVIATIONS AND ACRONYMS

AACE® International	AACE®
abrasion index	Ai
acid rock drainage	ARD
Acid-base accounting	ABA
acidification, volatilization of hydrogen cyanide gas, and re-neutralization	AVR
Acoustic Televiewer	ATV
ALS Canada Ltd	ALS
alternating current	AC
aluminum oxide	Al_2O_3
Amec Foster Wheeler Americas Limited	Amec Foster Wheeler
American Institute of Professional Geologists	AIPG
American National Standards Institute	ANSI
ammonium nitrate-fuel oil	ANFO
antimony	Sb
Aquatic Effects Monitoring Program	AEMP
armful alteration, disruption, or destruction	HADD
arsenic	As
atomic absorption spectrometry	AAS
atomic absorption	AA
Barrick Gold Corp.	Barrick
Basal Shear Fault	BSF
BC *Environmental Assessment Act*	BCEAA
best applicable practices	BAP
best available technology	BAT
BGC Engineering Inc.	BGC
Biogeoclimatic Ecosystem Classification	BEC
BioteQ Environmental Technologies Inc.	BioteQ
bismuth	Bi
British Columbia Utilities Commission	BCUC
British Columbia	BC
Bulk Mineral Analysis with Liberation	BMAL

bulk mineral analysis	BMA
cadmium	Cd
calcium oxide	CaO
California Air Resources Board	CARB
Canadian Dam Association	CDA
Canadian Development Expense	CDE
Canadian Environmental Assessment Act	CEAA
Canadian Environmental Assessment	CEA
Canadian Exploration Expense	CEE
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian National Railroad	CNR
carbon-in-leach	CIL
carboxymethyl cellulose	CMC
Caro's acid	H_2SO_5
CDN Resource Laboratories Ltd.	CDN Resource
chalcopyrite	Cp
chlorine	Cl
closed-circuit television	CCTV
coarse ore stockpile	COS
cobalt	Co
comparative work index	CWi
Construction Diversion Tunnel	CDT
conventional counter-current decantation	CCD
copper sulphate	$CuSO_4$
copper	Cu
cost breakdown structure	CBS
Cost, Insurance and Freight – Free Out	CIF-FO
Coulter Creek Access Road	CCAR
counter current decantation	CCD
cross-linked polyethylene	XLPE
Cumulative Tax Credit Account	CTCA
Delegation of Authority Guideline	DOAG
Demand Side Management	DSM
direct current	DC
direct cyanide leaching	DCN
discounted cash flow	DCF
Distributed Control System	DCS
east	E
EBC Inc.	EBC
economic, social, and cultural impact assessment	ESCIA
effective grinding length	EGL
Electricity Supply Agreement	ESA
emergency medical technician	EMT
engineering, procurement, construction management	EPCM
environmental assessment	EA

Environmental Design Flood	EDF
Environmental Effects Monitoring	EEM
environmental impact statement	EIS
Environmental Management System	EMS
ERM Consultants Canada Inc.	ERM
Esso Minerals Canada Ltd	Esso Minerals
Factor of Safety	FOS
Fisheries and Oceans Canada	DFO
fluorine	F
Free Carrier	FCA
front-end loader	FEL
G&T Metallurgical Services Ltd.	G&T
galena	Gn
gas insulated	GIS
general and administrative	G&A
general mine expense	GME
Global Climatic Models	GCMs
global positioning system	GPS
gold equivalent	AuEQ
gold	Au
Golder Associates Ltd.	Golder
Goods and Services Tax	GST
Granduc Mines Ltd.	Granduc
Granmac Services Ltd.	Granmac
greenhouse gas	GHG
gross vehicle weight	GVW
Ground Penetrating Radar	GPR
Hazelton Volcanics	HV
Hazen Research Inc.	Hazen
heating, ventilation, and air conditioning	HVAC
height of draw	HOD
hematite	He
high-density polyethylene	HDPE
high-density sludge	HDS
high-pressure grinding roll	HPGR
hydrochloric acid	HCl
hydrogen peroxide	H_2O_2
Independent Geotechnical Review Board	IGRB
Independent Power Producer	IPP
inductively coupled plasma	ICP
Inflow Design Flood	IDF
Input Tax Credit	ITC
internal rate of return	IRR
International Electrotechnical Commission	IEC
International Organization for Standardization	ISO

inter-ramp angle	IRA
inter-ramp height	IRH
inverse distance weighting	IDW
Iron Cap Fault	ICF
iron	Fe
Joint Health and Safety Committee	JHSC
joint venture	JV
Kambert Civil Consulting Ltd.	KCC
Kerr-Sulphurets-Mitchell	KSM
Klohn Crippen Berger Ltd.	KCB
Köeppern Machinery Australia Pty Ltd.	Köeppern
Land Resource Management Plan	LRMP
lead	Pb
Lerchs-Grossmann	LG
life-of-mine	LOM
Light Detection and Ranging	LIDAR
Lilburn & Associates LLC	Lilburn
linear low-density polyethylene	LLDPE
liquefied natural gas	LNG
liquefied petroleum gas	LPG
load factor	LF
load-haul-dump	LHD
local study area	LSA
locked cycle tests	LCT
magnesium oxide	MgO
magnetite	Ma
magneto telluric	MT
maintenance and repair contracts	MARC
manganese oxide	MnO
material take-off	MTO
McElhanney Consulting Services Inc.	McElhanney
McTagg Diversion Tunnels	MTDT
mercury	Hg
metabisulphite	MBS
metal leaching	ML
Metal Mining Effluent Regulations	MMER
Methyl isobutyl carbinol	MIBC
Metso Minerals Industries Inc.	Metso
Mining Rock Mass Rating	MRMR
Mining Rock Mass Rating	MRMR
Ministry of Energy and Mines	MEM
Ministry of Energy, Mines and Natural Gas	MEMNG
Ministry of Energy, Mines and Petroleum Resources	MEMPR
Ministry of Environment	MOE
Ministry of Forests	MOF

Ministry of Forests, Lands and Natural Resource Operations	MFLNRO
Ministry of Transportation and Infrastructure	MOTI
Mitchell Diversion Tunnel	MDT
Mitchell Thrust Fault	MTF
Mitchell Valley Drainage Tunnel	MVDT
Mitchell-Treaty Twinned Tunnels	MTT
molybdenum	Mo
Moose Mountain Technical Services	MMTS
motor control centre	MCC
Multiple Accounts Analysis	MAA
Multiple Pulse in Air	MPiA
Municipal Wastewater Regulation	MWR
Nass Timber Supply Area	Nass TSA
Nass Wildlife Area	NWA
National Ambient Air Quality Objectives	NAAQOs
National Instrument 43-101	NI 43-101
nearest neighbor	NN
Neil S. Seldon & Associates Ltd.	NSA
net cash flow	NCF
net present value	NPV
Net Smelter Price	NSP
net smelter return	NSR
Newhawk Gold Mines Ltd.	Newhawk
Newmont Exploration of Canada Ltd.	Newmont
nickel	Ni
Nisga'a Final Agreement	NFA
Nisga'a Lisims Government	NLG
no net loss	NNL
non-potentially acid generating	NPAG
Nordic Minesteel Technologies Inc.	NMT
North American Datum	NAD
North Pit Wall Dewatering Adit	NPWDA
North Treaty Access Road	NTAR
north	N
Northwest Fault	NWF
Northwest Transmission Line	NTL
Operator Interface Stations	OIS
Ore Control System	OCS
Ore Preparation Complex	OPC
peak ground acceleration	PGA
Phelps Dodge Corp.	Phelps Dodge
phosphorus	P
pit slope angle	PSA
Pocock Industrial Inc.	Pocock
point of delivery	POD

potassium amyl xanthate	PAX
potentially acid generating	PAG
Prefeasibility Study	PFS
Preliminary Economic Assessment	PEA
Pretium Resources Inc.	Pretium
PricewaterhouseCoopers	PwC
prilled ammonium	AN Prill
probable maximum flood	PMF
Probable Maximum Precipitation	PMP
Process Tailing and Management Area	PTMA
programmable logic controller	PLC
Provincial Sales Tax	PST
pyrite	Py
Qualified Person	QP
quality assurance	QA
quality control	QC
quantile-quantile	QQ
Quantitative Evaluation of Minerals by Scanning	QEMSCAN®
Raewyn Copper	RC
Rate Design Application	RDA
real-time kinematic	RTK
regional study area	RSA
request for information	RFI
Resource Management Zone	RMZ
Resource Modelling Inc.	RMI
rhenium	Re
rock quality designation	RQD
Rock Storage Facility	RSF
rotations per minute	RPM
run-of-mine	ROM
Seabridge Gold Inc.	Seabridge
selenium	Se
self-contained breathing apparatus	SCBA
semi-autogenous grinding mill comminution	SMC
semi-autogenous grinding	SAG
SGS Minerals Services	SGS
silica	SiO_2
silver	Ag
Snowfields Slide Dewatering Adit	SSDA
Society for Mining, Metallurgy, and Exploration	SME
sodium cyanide	NaCN
sodium hydrosulphide	NaHS
sodium hydroxide	NaOH
sodium silicate	Na_2SiO_3
sodium sulphide	Na_2S

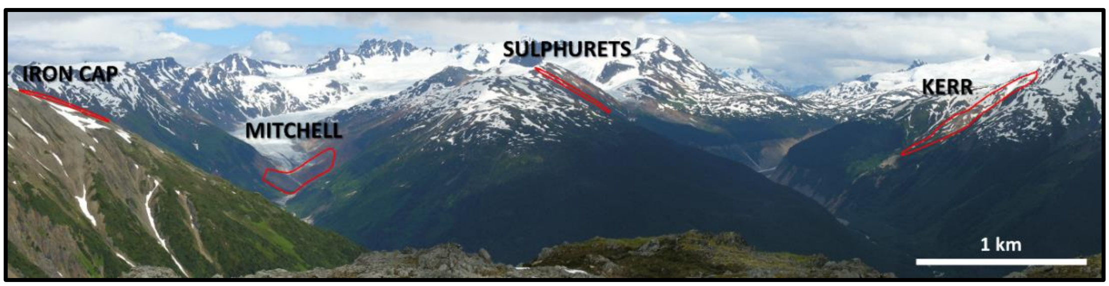
solids liquid separation	SLS
south	S
Special Use Permit	SUP
Species at Risk Act	SARA
Standard Penetration	SPT
standard reference material	SRM
sulphide	S^{-2}
sulphidization, acidification, recycling, and thickening of precipitate	SART
sulphur	S
Sulphurets Thrust Fault	STF
Sulphurets-Mitchell Conveyor Tunnel	SMCT
sulphuric acid	H_2SO_4
Sustainable Resource Management Plan	SRMP
Tailing Management Facility	TMF
Tariff Supplement	TS
temporary water treatment plants	TWTP
tennantite	Tn
Tetra Tech, Inc.	Tetra Tech
tetrahedrite	Tt
The Claim Group Inc.	TCG
the Environmental Assessment Application/Environmental Impact Statement	the Application/EIS
total sulphur	S^T
total suspended sediments	TSS
treatment charge/refining charge	TC/RC
Treaty Creek Access Road	TCAR
tunnel support classes	TSC
ultra-high frequency	UHF
undercut	UC
Uniform Hazard Response Spectra	UHRS
Universal Transverse Mercator	UTM
University of British Columbia	UBC
valued component	VC
Voice over Internet Protocol	VoIP
waste rock facility	WRF
Water Storage Dam	WSD
Water Storage Facility	WSF
Water Treatment Plant	WTP
weak acid dissociable	WAD
west	W
WN Brazier Associates Inc.	Brazier
work breakdown structure	WBS
work index	Wi
x-ray fluorescence	XRF
zinc	Zn

1.0 SUMMARY

1.1 INTRODUCTION

Seabridge Gold Inc.'s (Seabridge) Kerr-Sulphurets-Mitchell (KSM) Project (the Project) involves the development of major gold-copper (Au-Cu) deposits located in northwest British Columbia (BC) off Highway 37, approximately 65 km by air north-northwest of the ice free Port of Stewart, BC. The Project is situated within the coastal mountains of BC, approximately 30 km topographically upgradient of the Alaska-BC border. The Project is one of the few undeveloped projects within the world that has received its environmental approvals, these having been granted by both the Government of Canada and the Government of BC. The Project includes four major mineralized zones, identified as the Mitchell, Kerr, Sulphurets, and Iron Cap deposits. The deposits contain significant gold, copper, silver (Ag), and molybdenum (Mo) mineralization. Figure 1.1 is a panoramic view looking east towards the aforementioned deposits.

Figure 1.1 Panoramic View of KSM Deposits (Looking East)



In conjunction with the environmental approvals, Seabridge also received early-stage construction permits for the Project from the Province of BC in September 2014. The permits issued include:

- authority to construct and use roadways along Coulter Creek and Treaty Creek
- rights-of-way for the proposed Mitchell-Treaty Twinned Tunnel (MTT) alignment connecting Project facilities
- permits for constructing and operating numerous camps required to support constructions activities
- permits authorizing early-stage construction activities at the Mine Site and Tailing Management Facility (TMF).

Seabridge also received permits from the BC Government in October 2016, which allows the construction of an exploration adit to explore mineralization associated with the Deep Kerr deposit.

In this report, the Project has been evaluated with two similar studies that each evaluate different options for mine development. The 2016 Prefeasibility Study (PFS) evaluates mining development mostly by open pit method at a specified processing rate, while the Preliminary Economic Assessment (PEA) is a study that leverages the technical information of the PFS and evaluates mine development dominated by underground mining methods at a processing rate higher than that used in the PFS.

1.2 PRELIMINARY FEASIBILITY STUDY

This National Instrument 43-101 (NI 43-101) PFS has been prepared by Tetra Tech, Inc. (Tetra Tech), for Seabridge, and is based on an update of the 2012 PFS (Tetra Tech 2012) work produced by Tetra Tech and the following independent consultants:

- Moose Mountain Technical Services (MMTS)
- Golder Associates Ltd. (Golder)
- McElhanney Consulting Services Ltd. (McElhanney)
- BGC Engineering Inc. (BGC)
- Resource Modeling Inc. (RMI)
- Klohn Crippen Berger Ltd. (KCB)
- ERM Consultants Canada Ltd. (ERM)
- WN Brazier Associates Inc. (Brazier).

The 2016 PFS envisages a combined open pit/underground block caving mining operation that is scheduled to operate for 53 years. During the initial 33 years of mine life, the majority of ore would be derived from open pit mines, with the tail end of this period supplemented by the initial development of underground block cave mines. Ore delivery to the mill during Year 2 to Year 35 is designed to be maintained at an average of 130,000 t/d. After depletion of the open pits, the mill processing rate would be reduced to just over 95,000 t/d for 10 additional years before ramping down to just over 60,000 t/d for the remaining few years of stockpile reclaim at the end of the mine life. Over the entire 53-year mine life, ore would be fed to a flotation and gold extraction mill. The flotation plant would produce a gold/copper/silver concentrate for transport by truck to the nearby sea port for shipment to Pacific Rim smelters. Extensive metallurgical testing confirms that the Project can produce a clean concentrate with an average copper grade of 25% with a high gold and silver content, making it readily saleable. A separate molybdenum concentrate and gold-silver doré would also be produced at the KSM processing facility.

The capital and operating costs for the 2016 PFS have been estimated to a +25%/-10% level of accuracy. All dollar figures presented in this report are stated in US dollars, unless otherwise specified. This 2016 PFS concludes:

- projected capital costs are down despite substantial enhancements to meet environmental improvements that were committed to in the environmental assessment (EA) review process
- gold and copper reserves are up slightly despite lower metal prices
- the base case estimated total cost, at US$673/oz of gold produced, remains well below the industry average for operating mines
- the base case after-tax payback period is approximately 6.8 years, which is a remarkably low 13% of the 53-year mine life and a key benefit to large producers.

Overall, the 2016 PFS update confirms that KSM is an economic project with an unusually long life in a low-risk jurisdiction.

1.3 PRELIMINARY ECONOMIC ASSESSMENT

Amec Foster Wheeler Americas Limited (Amec Foster Wheeler) and the following independent consultants prepared a Preliminary Economic Assessment study (the PEA):

- Golder
- KCB
- ERM.

The PEA has been prepared as an alternative option to the 2016 PFS for the development of the Project. The PEA was undertaken to evaluate a different approach to the Project by emphasizing low cost block cave mining and reducing the number and size of the open pits, which significantly reduces the surface disturbances in the re-designed Project. The PEA assesses the potential impacts of incorporating higher grade Inferred Mineral Resources delineated at Deep Kerr and Iron Cap Lower Zone into the mine design, and increasing the annual average maximum mill throughput from 130,000 t/d envisioned in the 2016 PFS to 170,000 t/d in the PEA.

The results of the 2016 PFS remain valid and represent a viable option for developing the Project, with the PEA assessing an alternative development option at a conceptual level. The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The results of the PEA were disclosed in Seabridge's press releases entitled "New Study Finds Significant Gains for Seabridge Gold's KSM Project", dated October 6th, 2016. This report will be filed in support of the disclosure of the PEA results.

1.4 KEY STUDY OUTCOMES

The key study outcomes for the projected economic results and average annual metal production are presented in Table 1.1 and Table 1.2, respectively. The PEA shows potential for improvements over the 2016 PFS in unit operating costs, net cash flow, NPV, IRR and project payback by producing a higher percentage of mill feed through underground mining, and processing the mill feed at a higher plant processing rate. Annual gold and copper production in the PEA increase over those shown in the PFS through the benefit of spending greater initial and sustaining capital.

Table 1.1 Projected Economic Results

		Base Case		Recent Spot Case		Alternate Case	
	Unit	2016 PEA*	2016 PFS	2016 PEA*	2016 PFS	2016 PEA*	2016 PFS
Metal Prices							
Gold	US$/oz	1,230		1,350		1,500	
Copper	US$/lb	2.75		2.20		3.00	
Silver	US$/oz	17.75		20.00		25.00	
Exchange Rate	US$/Cdn$	0.80		0.77		0.80	
Cost Summary							
Operating Costs (life-of-mine [LOM])	US$/oz Au	-179	277	32	404	-319	183
Total Cost (Produced)	US$/oz Au	358	673	553	787	218	580
Copper Credits (included in costs)	US$/oz Au	-1,328	-795	-1,104	-636	-1,449	-868
Silver Credits (included in costs)	US$/oz Au	-83	-71	-97	-80	-117	-100
Initial Capital (includes pre-production mining)	US$ billion	5.5	5.0	5.3	4.8	5.5	5.0
Sustaining Capital	US$ billion	10.0	5.5	9.7	5.3	10.0	5.5
Unit Operating Cost Onsite	US$/t	11.61	12.36	11.17	12.09	11.61	12.36
Pre-tax Results							
Net Cash Flow	US$ billion	26.3	15.9	24.1	16.1	38.7	26.3
Net Present Value (NPV) @ 5% Discount Rate	US$ billion	6.1	3.3	5.7	3.5	10.2	6.5
Internal Rate of Return (IRR)	%	12.7	10.4	12.9	11.1	16.9	14.6
Payback Period	Years	5.6	6.0	5.3	5.6	3.9	4.1
Post-tax Results							
Net Cash Flow	US$ billion	16.7	10.0	15.3	10.1	24.7	16.7
NPV @ 5% Discount Rate	US$ billion	3.4	1.5	3.2	1.7	6.0	3.7
IRR	%	10.0	8.0	10.1	8.5	13.4	11.4
Payback Period	Years	6.4	6.8	6.1	6.4	4.7	4.9

Note: Operating and total cost per ounce of gold are after copper and silver credits. Total cost per ounce includes all start-up capital, sustaining capital, and reclamation/closure costs. The post-tax results include the BC Mineral Tax and corporate provincial and federal taxes.

*The results of the economic analysis in the PEA represents forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented. The material factors or assumptions used to develop the forward-looking information, as well as the material risk factors that could cause actual results to differ materially from the forward-looking information in the PEA are more fully described in Section 24 and 25 of the Report. A portion of the Mineral Resources in the PEA mine plans, production schedules, and cash flows include Inferred Mineral Resources, that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized. Due to the conceptual nature of the PEA, none of the Mineral Resources in the PEA have been converted to Mineral Reserves and therefore do not have demonstrated economic viability.

Table 1.2 Average Annual Metal Production (Metal Recovered)

		Years 1 to 7 Average		LOM Average	
	Unit	2016 PEA	2016 PFS	2016 PEA	2016 PFS
Average Grades					
Gold	g/t	0.78	0.82	0.52	0.55
Copper	%	0.26	0.24	0.32	0.21
Silver	g/t	2.7	2.8	2.7	2.6
Molybdenum	ppm	n/a	48	n/a	43
Annual Production					
Gold	'000 oz	1,150	933	592	540
Copper	'000 lb	306,603	204,937	286,217	156,052
Silver	'000 oz	3,290	2,603	2,761	2,164
Molybdenum	'000 lb	n/a	1,593	n/a	1,171

1.5 PROPERTY DESCRIPTION AND LOCATION

The KSM Property (the Property) is located in the coastal mountains of northwest BC at a latitude and longitude of approximately 56.50° north (N) and 130.30° west (W), respectively. The Property is situated approximately 950 km northwest of Vancouver, BC; 65 km by air north-northwest of Stewart, BC; and 21 km south-southeast of the former Eskay Creek Mine. The proposed pit areas lie within the headwaters of Sulphurets Creek, which is a tributary of the Unuk River, which flows into the Pacific through Alaska. The proposed TMF will be located within the tributaries of Teigen and Treaty creeks. The Teigen and Treaty creeks are tributaries of the Bell-Irving River, which is itself a major tributary of the Nass River. The Nass River also flows to the Pacific Ocean through the northwestern portion of British Columbia, entirely within Canadian jurisdiction. Figure 1.2 is a general location map of the Project area.

The KSM Property comprises three discontinuous claim blocks. The claim blocks are referred to as:

- the KSM claims
- the Seabee/Tina claims
- the KSM placer claims.

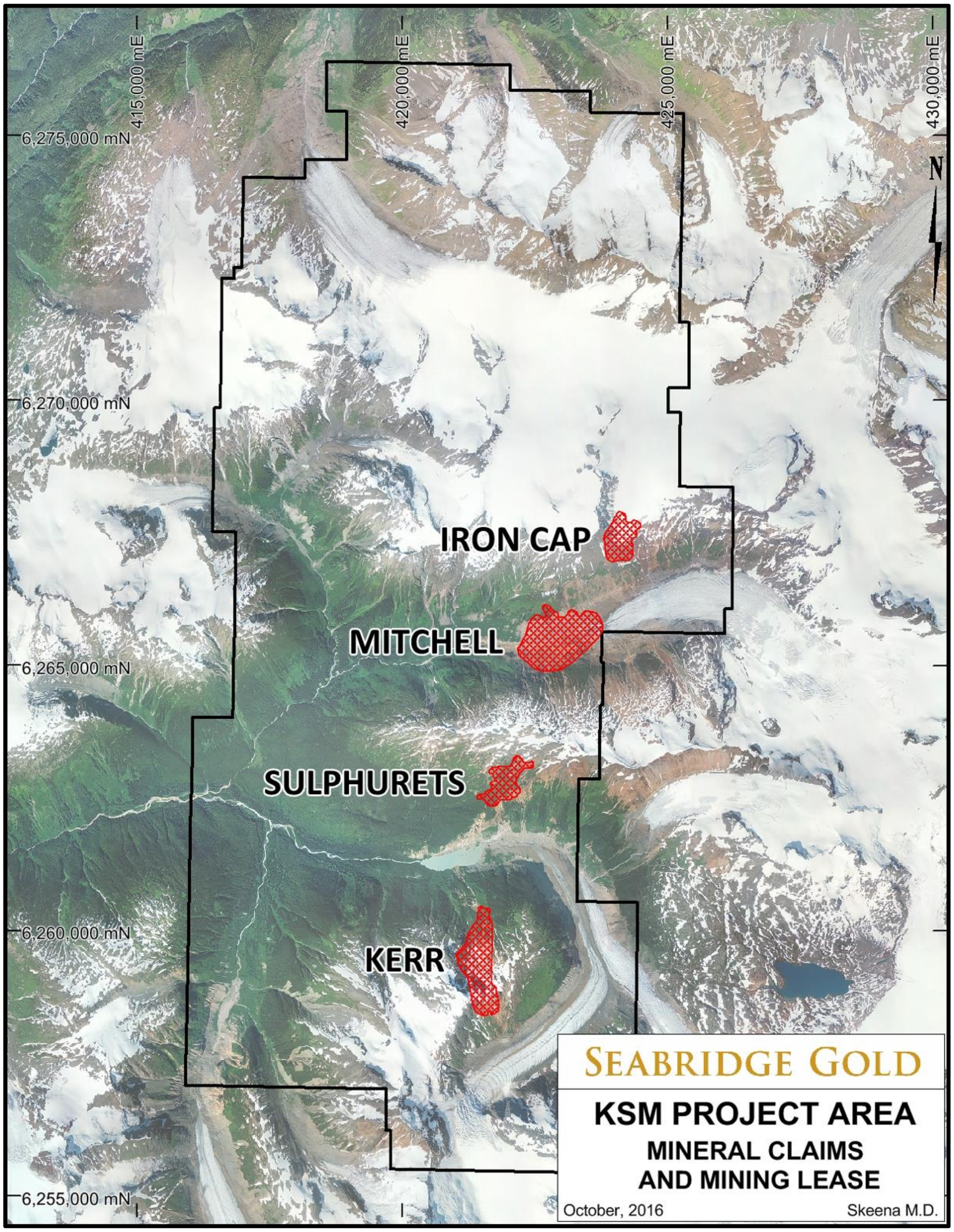
The three claim blocks consist of 71 mineral claims, three placer claims, and two mineral leases. The total area of the mineral claims and leases covers an area of approximately 38,929 ha. The Seabee/Tina claim block is located about 19 km northeast of the Kerr-Sulphurets-Mitchell-Iron Cap mineralized zones. The Seabee/Tina claim block is currently under consideration as the site for the proposed infrastructure. The claims are 100% owned by Seabridge. Placer Dome (now Barrick Gold Corp. [Barrick]) retains a 1% net smelter royalty on the Property that is capped at US$3.6 million. Figure 1.3 is a claim map showing Seabridge's mineral claims and leases.

Annual holding costs for all claims (lode and placer) over the next five years vary from approximately Cdn$450,000 to Cdn$970,000. 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) is the land manager and mineral tenure agent for Seabridge.

Figure 1.2 General Location Map

Source: ERM

Figure 1.3 KSM Mineral Claim Map



Source: Seabridge

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

The KSM Property lies in the rugged coastal mountains of northwestern 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 the 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 Property is drained by the Sulphurets Creek and Mitchell Creek watersheds that empty into the Unuk River, which flows westward to the Pacific Ocean through Alaska. The Treaty Ore Preparation Complex (OPC) 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 at the Mine Site has an estimated average of 1,652 mm (Sulphurets weather station, 2008 to 2011 data) and at the Process Tailing and Management Area (PTMA) has an estimated average of 1,371 mm (Teigan Creek weather station, 2009 to 2015 data). 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 KSM Property can be accessed only via helicopter.

There are multiple deep-water loading facilities for shipping bulk mineral concentrates located in the ice free Port of Stewart, BC. Port facilities are currently used by the Red Chris 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.

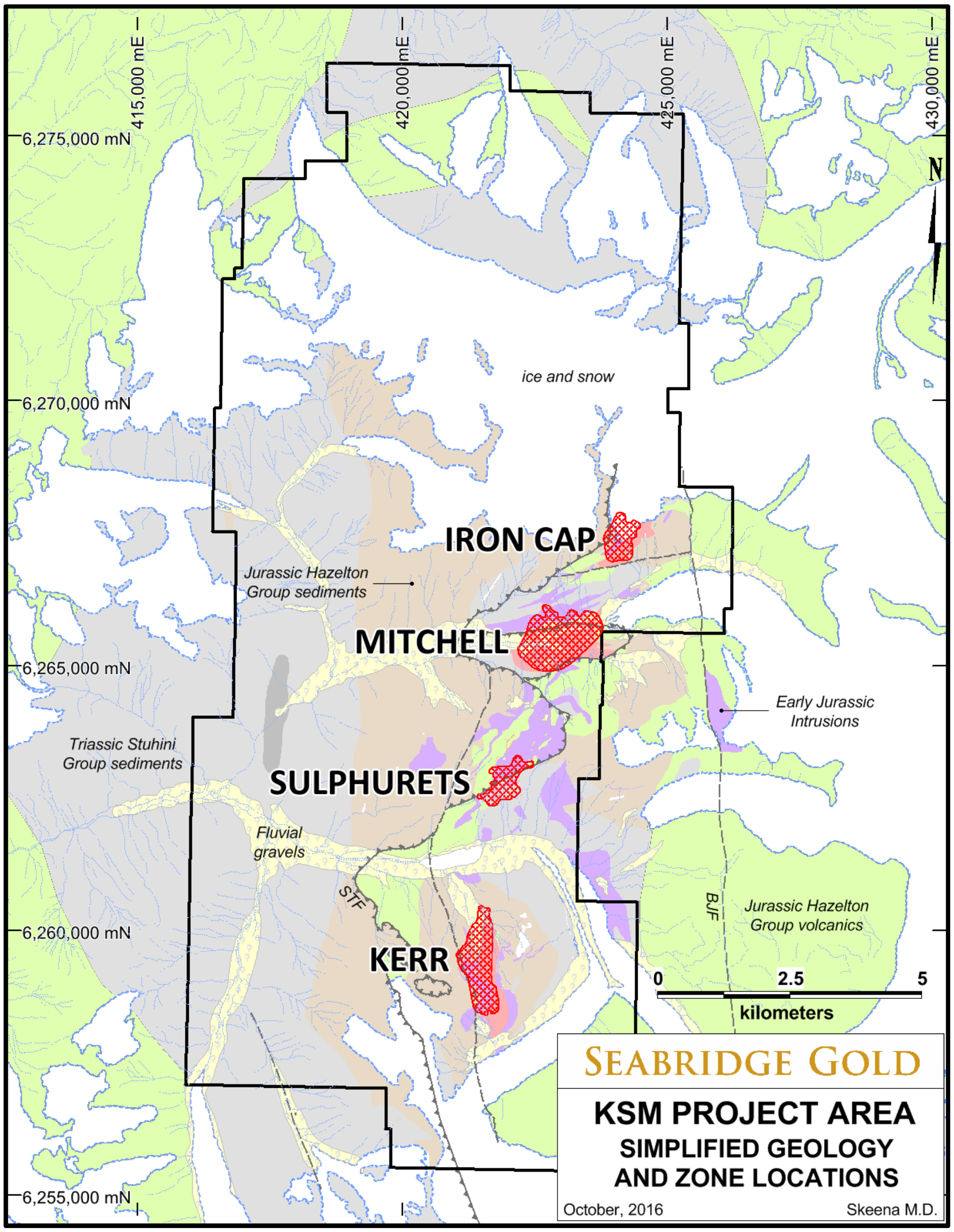
The Property and its access routes are on Crown land; therefore, surface and access rights are granted under, and subject to compliance with, the *Mineral Tenure Act* or the *Land Act* or, at the discretion of the Crown, under the *Mining Right of Way Act*. There are no settlements or privately owned 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, the Northwest Transmission Line (NTL), run along the Highway 37 corridor up to the Red Chris Mine. The Red Chris Mine is approximately 120 km north of the Project site, whereas the NTL is less than 15 km east of the Project or approximately 30 km away from the Treaty OPC by way of the proposed Treaty Creek Access Road (TCAR).

1.7 GEOLOGY

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 (Figure 1.4). 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, Mt. Milligan, and Kerr-Sulphurets.

Figure 1.4 KSM Project Geology



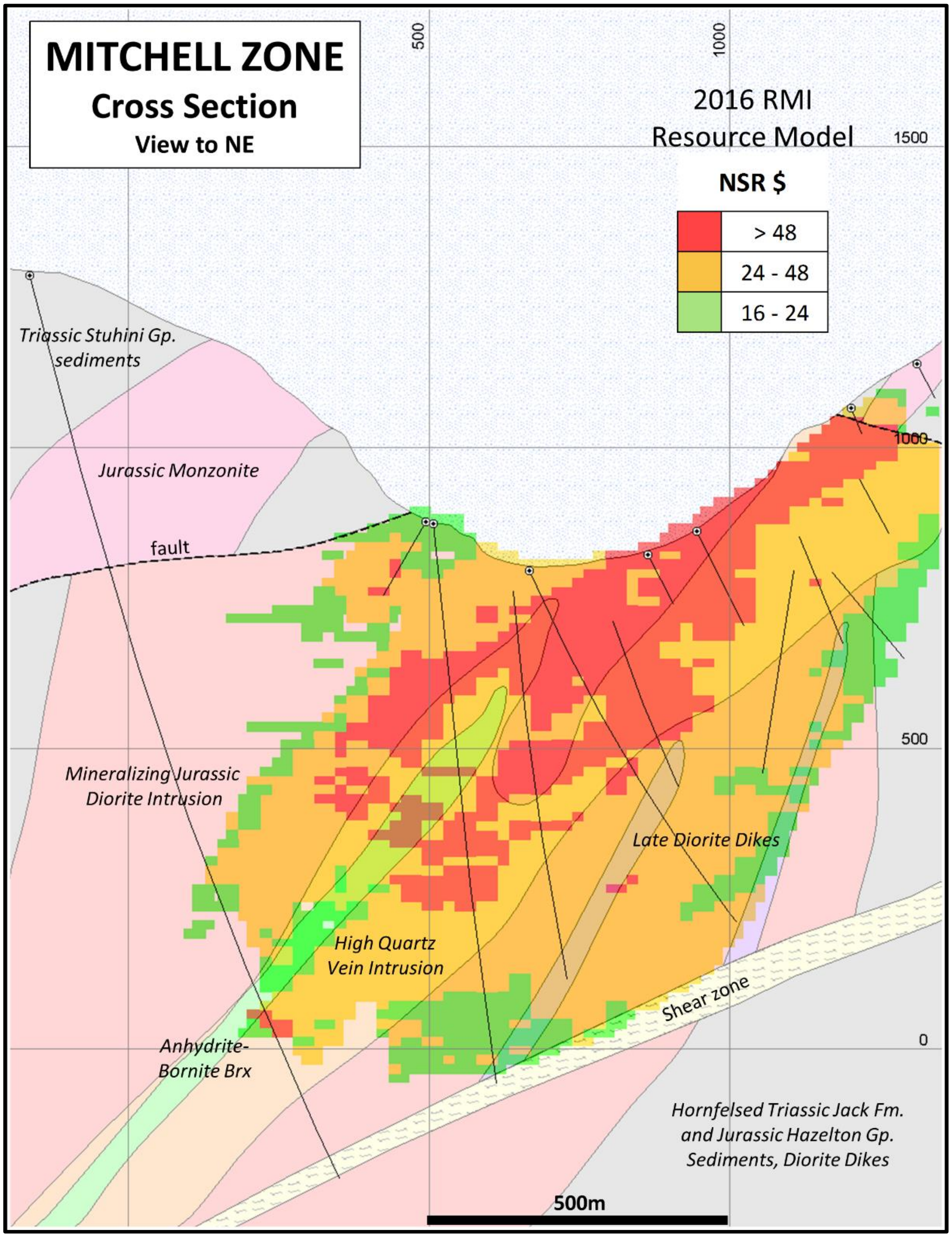
Source: Seabridge

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. Deep drilling since 2012 has identified two sub-parallel, north-south trending, steep west-dipping mineralized zones that appear to coalesce near the topographic surface. After significant deep drilling was completed at the Kerr deposit, an updated geological interpretation and subsequent updated Mineral Resource model were completed. That new model forms the basis for the 2016 Mineral Resources and Mineral Reserves. Approximately 218 diamond core holes totaling about 94,000 m of drilling data were used to construct the Kerr block model used for this PFS.

The Sulphurets deposit comprises 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 north-easterly 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 potassium-feldspar-siliceous hydrothermal breccia that apparently crosscuts the Raewyn Copper-Gold Zone. The Breccia Gold Zone strikes northerly and dips westerly. Approximately 141 core holes totaling about 43,000 m were used to construct the Sulphurets block model used for the 2016 PFS.

The Mitchell Zone (Figure 1.5) 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 common with large, stable, and long-lived hydrothermal systems. Approximately 191 core holes totaling about 68,000 m were used to construct the Mitchell block model used for this 2016 PFS.

Figure 1.5 Mitchell Zone Cross Section



Source: Seabridge
Note: net smelter return (NSR)

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). Significant drilling has been completed at the Iron Cap deposit since the 2012 PFS (Tetra Tech 2012), which resulted in an updated geological interpretation and subsequent updated Mineral Resource model that forms the basis for the 2016 Mineral Resources and Mineral Reserves. Approximately 69 core holes totaling about 35,000 m were used to construct the Iron Cap block model used for the 2016 PFS.

1.8 HISTORY

The modern exploration history of the area began in the 1960s, with brief programs conducted by Newmont Exploration of Canada Ltd. (Newmont), Granduc Mines Ltd. (Granduc), Phelps Dodge Corp. (Phelps Dodge), the Meridian Syndicate, and others. Most of these programs were focused on gold exploration. The 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) 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. 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 KSM Property. Immediately west of the Property, small-scale placer gold mining has occurred downstream in Sulphurets Creek. On the Brucejack 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. The Brucejack Property is currently owned by Pretium Resources Inc. (Pretium) who are building a high grade underground gold mine targeting 2017 commercial production.

1.9 RESOURCES

RMI constructed three-dimensional (3D) block models (3DBM) for the Kerr, Sulphurets, Mitchell, and Iron Cap zones after the 2011 drilling results were finalized. Those models were used for the 2012 PFS (Tetra Tech 2012). No material drilling has been conducted at the Sulphurets or Mitchell deposits since the 2012 PFS (Tetra Tech 2012); therefore, those Mineral Resource grade models were used for the 2016 PFS. Significant drilling was completed at both the Kerr and Iron Cap deposits since the 2012 PFS (Tetra Tech 2012), which necessitated updating the geological interpretation and grade shell wireframes for those deposits. The new drilling and various wireframes were used to develop new block grade models for the Kerr and Iron Cap deposits that were subsequently used for the 2016 PFS.

Inverse distance estimation methods were used for all Mineral Resource models. In the case of the Sulphurets and Mitchell deposits, a multi-pass interpolation strategy was used, using a combination of grade shells or specific geological lithological/alteration assemblages to constrain the estimate. 3D search ellipses oriented with the trend of mineralization were used to find drill hole composites. Similar strategies were used for the more recent models constructed for the Kerr and Iron Cap deposits. Deeper exploration in those two areas has demonstrated that higher-grade mineralization is associated with various structures. Instead of using conventional search ellipses to collect drill hole composites for block grade estimation, a trend plane search was used for the Kerr and Iron Cap models. That search method appears to do a better job of honoring the currently recognized structural controls in those deposits.

The estimated block grades were classified into Measured (Mitchell only), Indicated, and Inferred categories using mineralized continuity, proximity to drilling, and the number of holes used to estimate the blocks. Mineral Resources for the Project were determined by using a combination of conceptual open pit and underground mining methods. Lerchs-Grossmann (LG) conceptual pits were generated for the Kerr, Sulphurets, and Mitchell deposits using metal prices of US$1,300.00/oz of gold, US$3.00/lb of copper, US$20.00/oz of silver, and US$9.70/lb of molybdenum. Mining, processing, general and administrative (G&A), and metal recoveries were used to generate conceptual Mineral Resource pits that demonstrate reasonable prospects for eventual economic extraction. Conceptual block cave shapes were generated by Golder using GEOVIA's PCBC™ Footprint Finder software.

Table 1.3 summarizes the estimated Measured, Indicated, and Inferred Mineral Resources for each zone.

1.10 OVERALL GENERAL ARRANGEMENT

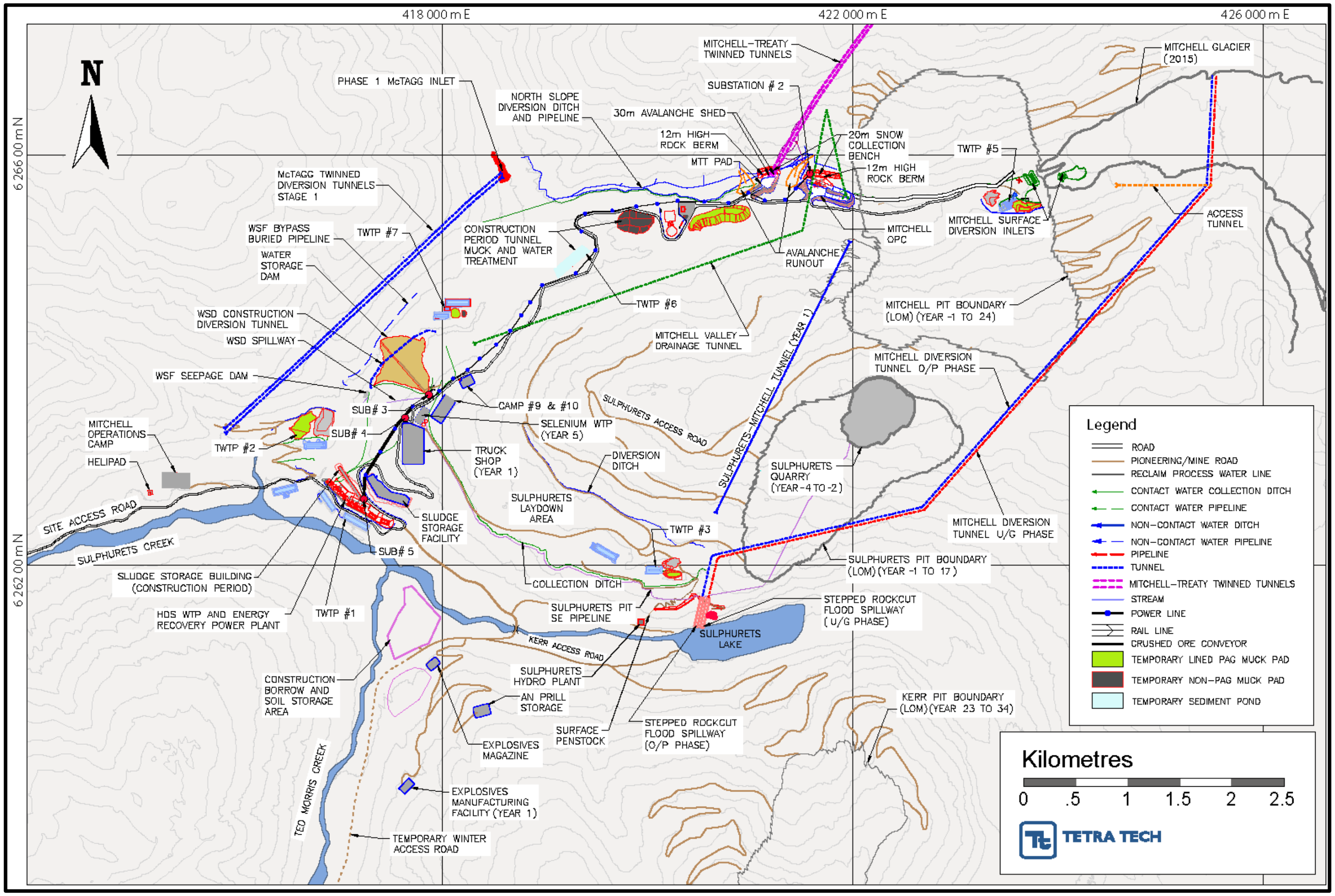
Figure 1.6, Figure 1.7, and Figure 1.8 depict an overview of the Mine Site, Treaty OPC, and TMF area, respectively. This is the general arrangement approved by the Governments of Canada and BC in 2014.

Table 1.3 KSM Mineral Resources as of May 31, 2016

Zone	Type of Constraint	NSR Cut-off (Cdn$/t)	Tonnes (000 t)	Au (g/t)	Au (000 oz)	Cu (%)	Cu (Mlb)	Ag (g/t)	Ag (000 oz)	Mo (ppm)	Mo (Mlb)
Measured Mineral Resources											
Mitchell	Conceptual LG Pit	9	698,800	0.63	14,154	0.17	2,618	3.1	69,647	59	91
	Conceptual Block Cave	16	51,300	0.59	973	0.20	226	4.7	7,752	41	5
	Total Mitchell Measured	n/a	750,100	0.63	15,127	0.17	2,844	3.2	77,399	58	96
Total KSM Measured	**n/a**	**n/a**	**750,100**	**0.63**	**15,127**	**0.17**	**2,844**	**3.2**	**77,399**	**58**	**96**
Indicated Mineral Resources											
Kerr	Conceptual LG Pit	9	355,000	0.22	2,511	0.41	3,208	1.1	12,555	4	3
	Conceptual Block Cave	16	24,400	0.24	188	0.48	258	2.0	1,569	14	1
	Total Kerr Indicated	n/a	379,400	0.22	2,699	0.41	3,466	1.2	14,124	5	4
Sulphurets	Conceptual LG Pit	9	381,600	0.58	7,116	0.21	1,766	0.8	9,815	48	40
Mitchell	Conceptual LG Pit	9	919,900	0.57	16,858	0.16	3,244	2.8	82,811	61	124
	Conceptual Block Cave	16	124,700	0.58	2,325	0.20	550	4.7	18,843	38	10
	Total Mitchell Indicated	n/a	1,044,600	0.57	19,183	0.16	3,794	3.0	101,654	58	134
Iron Cap	Conceptual Block Cave	16	346,800	0.51	5,686	0.23	1,758	4.5	50,174	14	11
Total KSM Indicated	**n/a**	**n/a**	**2,152,400**	**0.50**	**34,684**	**0.23**	**10,784**	**2.5**	**175,767**	**40**	**189**
Measured + Indicated Mineral Resources											
Kerr	Conceptual LG Pit	9	355,000	0.22	2,511	0.41	3,208	1.1	12,555	4	3
	Conceptual Block Cave	16	24,400	0.24	188	0.48	258	2.0	1,569	14	1
	Total Kerr M + I	n/a	379,400	0.22	2,699	0.41	3,466	1.2	14,124	5	4
Sulphurets	Conceptual LG Pit	9	381,600	0.58	7,116	0.21	1,766	0.8	9,815	48	40
Mitchell	Conceptual LG Pit	9	1,618,700	0.60	31,012	0.16	5,862	2.9	152,458	60	215
	Conceptual Block Cave	16	176,000	0.58	3,298	0.20	776	4.7	26,595	39	15
	Total Mitchell M + I	n/a	1,794,700	0.60	34,310	0.16	6,638	3.1	179,053	58	230
Iron Cap	Conceptual Block Cave	16	346,800	0.51	5,686	0.23	1,758	4.5	50,174	14	11
Total KSM Measured + Indicated	**n/a**	**n/a**	**2,902,500**	**0.54**	**49,811**	**0.21**	**13,628**	**2.7**	**253,166**	**44**	**285**
Inferred Mineral Resources											
Kerr	Conceptual LG Pit	9	80,200	0.27	696	0.21	371	1.1	2,836	6	1
	Conceptual Block Cave	16	1,609,000	0.31	16,036	0.43	15,249	1.8	93,115	25	89
	Total Kerr Inferred	n/a	1,689,200	0.31	16,732	0.42	15,620	1.8	95,951	24	90
Sulphurets	Conceptual LG Pit	9	182,300	0.46	2,696	0.14	563	1.3	7,619	28	11
Mitchell	Conceptual LG Pit	9	317,900	0.37	3,782	0.09	631	3.0	30,662	56	39
	Conceptual Block Cave	16	160,500	0.51	2,632	0.17	601	3.5	18,061	44	16
	Total Mitchell Inferred	n/a	478,400	0.38	6,414	0.10	1,232	3.0	48,723	55	55
Iron Cap	Conceptual Block Cave	16	369,300	0.42	4,987	0.22	1,791	2.2	26,121	21	17
Total KSM Inferred	**n/a**	**n/a**	**2,719,200**	**0.35**	**30,829**	**0.32**	**19,206**	**2.0**	**178,414**	**29**	**173**

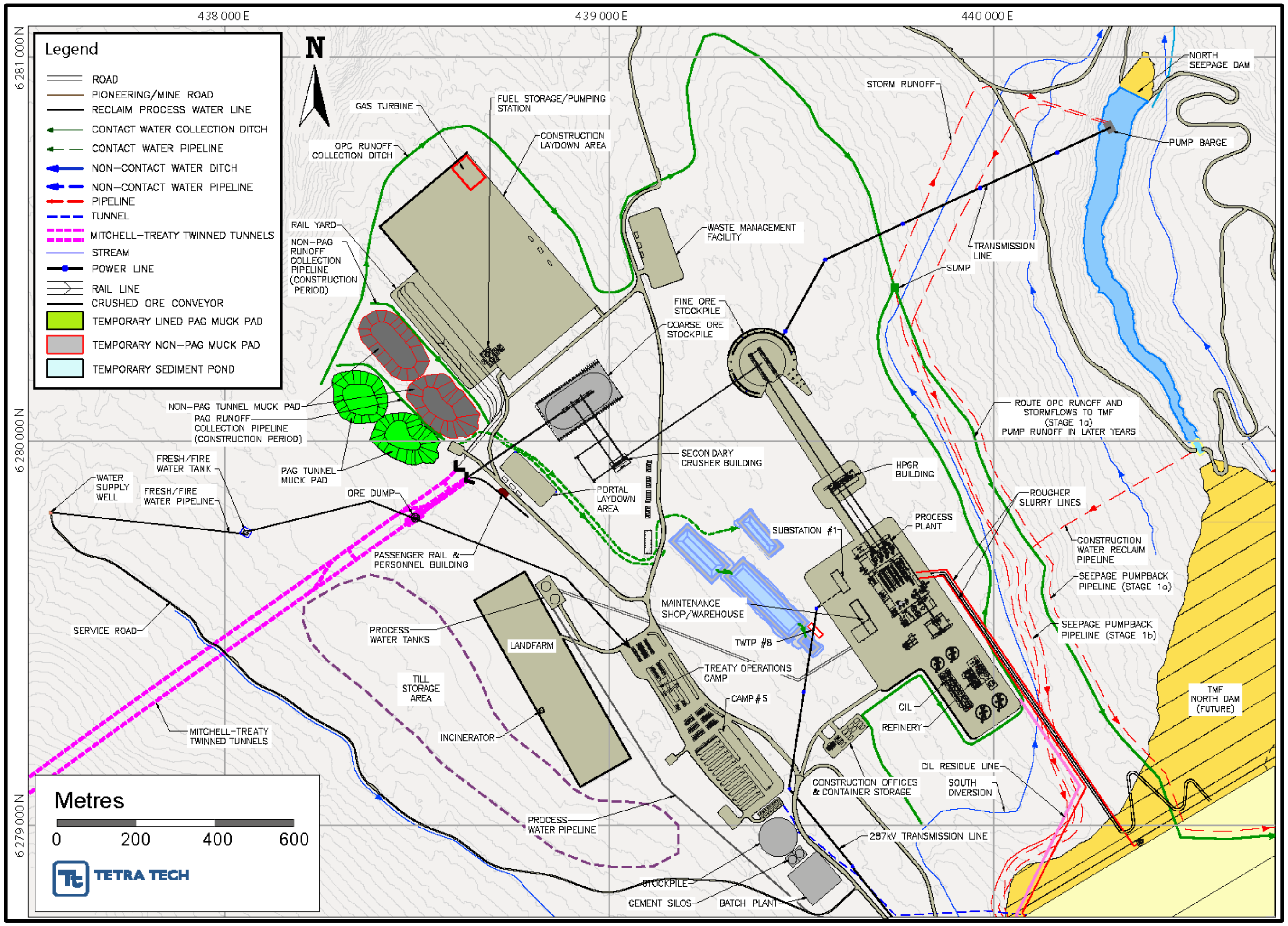
Note: Mineral Resources are reported inclusive of the Mineral Resources that were converted to Mineral Reserves. Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration.

Figure 1.6 2016 PFS Mine Site Layout after Initial Construction



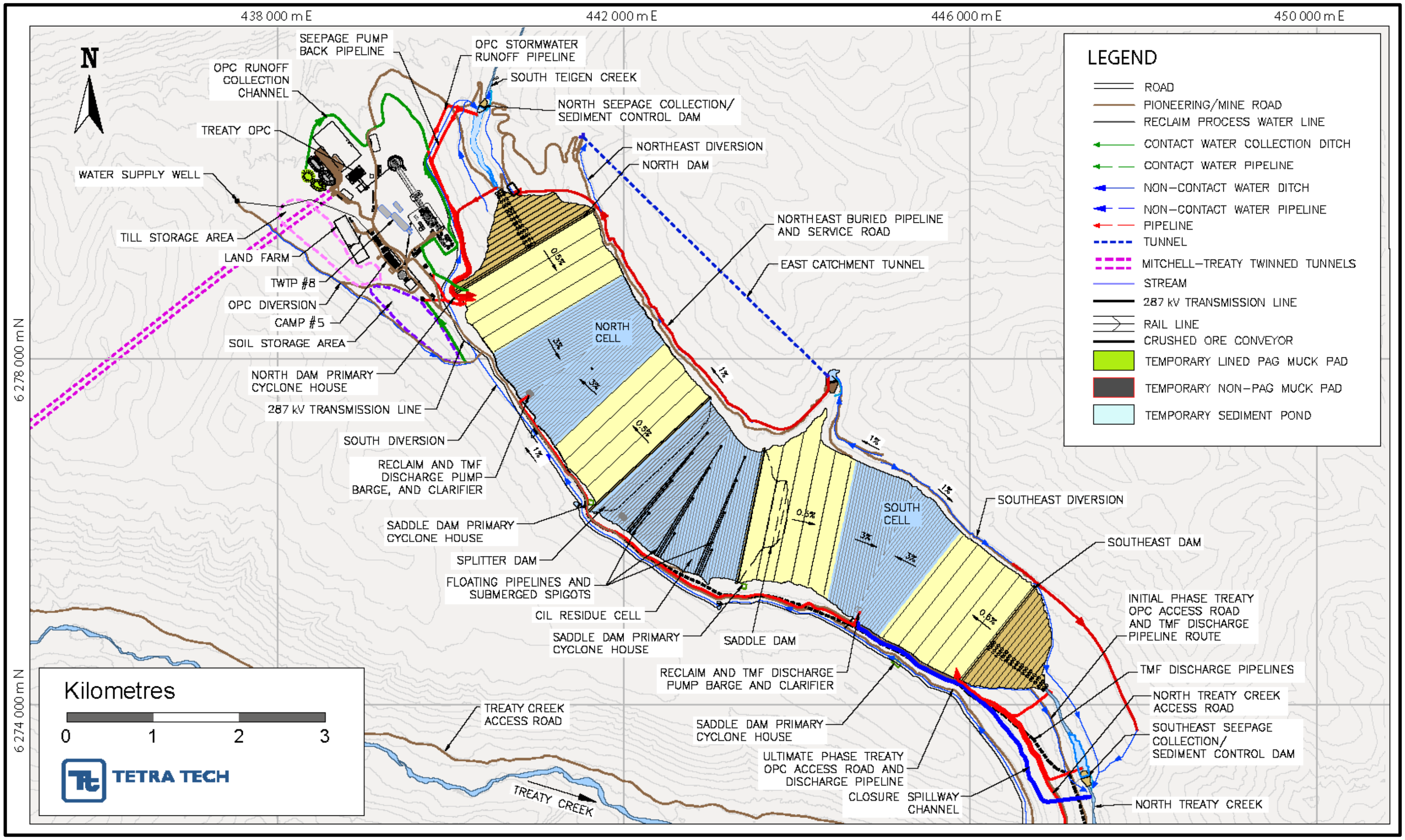
Source: Tetra Tech

Figure 1.7 2016 PFS Treaty Ore Preparation Complex Layout



Source: Tetra Tech

Figure 1.8 2016 PFS Ultimate Tailing Management Facility Area Layout



Source: Tetra Tech

1.11 MINE PLANNING

The proposed mine uses conventional large-scale open pit and block cave underground mining methods. Pit phases at the Mitchell, Kerr, and Sulphurets deposits have been engineered based on the results of an updated economic pit limit analysis. Starter pits have been selected in higher-grade areas. Underground mining has been proposed for the Iron Cap deposit and below the Mitchell open pit to reduce the volume of waste generated from the potential open pits.

1.11.1 MINING LIMITS

LG pit shell optimizations were used to define open pit mine plans in the 2012 PFS (Tetra Tech 2012) and the same limits were confirmed using LG for the 2016 PFS. Ultimate open pits have been modified slightly to implement design changes from the Environmental Assessment (EA) Application/Environmental Impact Statement (EIS) (the Application/EIS) (Rescan 2013) review and updated geotechnical study.

The underground block caving mine designs for both the Mitchell and Iron Cap deposits are based on modeling using GEOVIA's PCBC™ and Footprint Finder 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.

1.11.2 MINERAL RESERVE ESTIMATE

Waste to ore open pit cut-offs and underground shut-offs, including process recovery, were determined using metal prices of US$1,200.00/oz of gold, US$2.70/lb of copper, US$17.50/oz of silver, and US$9.70/lb of molybdenum for NSR calculations.

Open pit Mineral Reserves have been calculated using the updated pit designs and the 2016 Mineral Resource models. These calculations include mining loss and dilution that varies by pit ranging from 2.2 to 5.3% for loss, and 0.8 to 3.9% for dilution. A dynamic cut-off grade strategy has been applied with a minimum NSR of Cdn$9.00/t.

The mining NSR shut-off is Cdn$15.00/t for the Mitchell underground mine and Cdn$16.00/t for the Iron Cap underground mine. The Mitchell Mineral Reserves include 59 Mt of non-mineralized dilution at zero grade (13%) and 7 Mt of mineralized dilution (2%). The Iron Cap Mineral Reserves include 20 Mt of dilution at zero grade (9%) and 25 Mt of mineralized dilution (11%).

Proven and Probable Mineral Reserves for the Project as of July 31, 2016 are stated in Table 1.4.

Table 1.4 KSM Proven and Probable Reserves as of July 31, 2016

Zone	Mining Method	Reserve Category	Tonnes (Mt)	Average Grades Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Contained Metal Au (Moz	Cu (Mlb)	Ag (Moz)	Mo (Mlb)
Mitchell	Open Pit	Proven	460	0.68	0.17	3.1	59.2	10.1	1,767	45	60
		Probable	481	0.63	0.16	2.9	65.8	9.7	1,677	44	70
	Block Cave	Probable	453	0.53	0.17	3.5	33.6	7.7	1,648	51	34
Iron Cap	Block Cave	Probable	224	0.49	0.20	3.6	13.0	3.5	983	26	6
Sulphurets	Open Pit	Probable	304	0.59	0.22	0.8	51.6	5.8	1,495	8	35
Kerr	Open Pit	Probable	276	0.22	0.43	1.0	3.4	2.0	2,586	9	2
Totals		Proven	460	0.68	0.17	3.1	59.2	10.1	1,767	45	60
		Probable	1,738	0.51	0.22	2.5	38.2	28.7	8,388	138	147
		Total	**2,198**	**0.55**	**0.21**	**2.6**	**42.6**	**38.8**	**10,155**	**183**	**207**

Note: The Mineral Reserves tabulated in Table 1.4 are included in the tabulated Mineral Resources in Table 1.3. All Mineral Reserves stated in Table 1.4 account for mining loss and dilution.

1.11.3 MINE PRODUCTION PLAN

During the initial 33 years of mine life, the majority of ore is derived from open pits, with the tail end of this period supplemented by the initial development of underground block cave mines. After Year 1 ramp up, ore delivery to the mill during Year 2 to Year 35 is designed to be maintained at an average of 130,000 t/d. After depletion of the open pits, the mill processing rate will be reduced to about 96,000 t/d for 10 additional years, before ramping down to just over 61,000 t/d. The change in throughput matches the production levels from the block cave with appropriate ramp ups and ramp downs applied. The remaining few years use stockpile reclaim to supplement the declining production from the block caves at the end of the mine life.

LOM production is summarized in Table 1.5 and Figure 1.9.

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

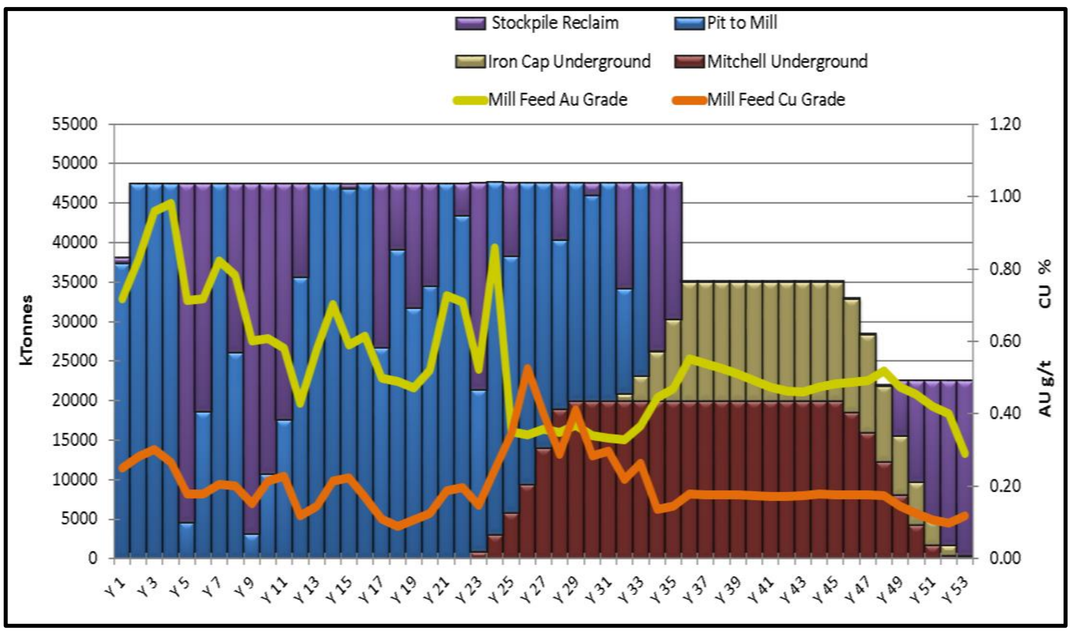
Potential geohazards have been identified in the area of the proposed open pits, block cave mine, RSFs, roads, and other infrastructure; designs include the mitigation of geohazards such as avalanche control, provision of avalanche run-out routes, barriers, and avalanche area and slope hazard avoidance as appropriate.

The mining progression is designed to build RSFs in lifts (bottom-up construction) to consolidate the foundations and reduce downslope risks. 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.

Table 1.5 2016 PFS LOM Production

Description	Unit	LOM
Mine Life	years	53
Open Pit To Mill	Mt	1,066
Open Pit To Stockpile	Mt	455
Stockpile Reclaim	Mt	455
Mitchell Underground To Mill	Mt	453
Iron Cap Underground To Mill	Mt	224
Total Mill Feed	Mt	2,198
Gold	g/t	0.55
Copper	%	0.21
Silver	g/t	2.6
Molybdenum	ppm	42.6
Metal to the Mill		
Gold	Moz	38.8
Copper	Mlb	10,155
Silver	Moz	183
Molybdenum	Mlb	207
Total Waste Mined	Mt	3,003
Pit Strip Ratio	t/t	1.4

Figure 1.9 2016 PFS Mill Feed Production Schedule



Source: MMTS

Ore is mined from Mitchell open pit from Years 1 to 24. Mitchell transitions to block cave mining as the Mitchell pit is mined out. Ore is mined from Sulphurets open pit from Years 1 to 17. Kerr open pit supplements block cave mining from Year 25 to Year 34, and during these years, ore will be transported by an overland conveyor and rope conveyor system starting at the Kerr pit. Mitchell block cave is estimated to have a production ramp-up period of six years, steady state production at 20 Mt/a for 17 years, and then ramp-down production for another 7 years. Iron Cap is estimated to have a production ramp-up period of four years, steady state production at 15 Mt/a for 10 years, and then ramp-down production for another 9 years. The underground pre-production period would be six years, with first underground ore production from Mitchell and Iron Cap in Years 23 and 32, respectively.

All ore will be transported by train from the Mitchell OPC and through the MTT to the Treaty OPC. The ore transport system will also include:

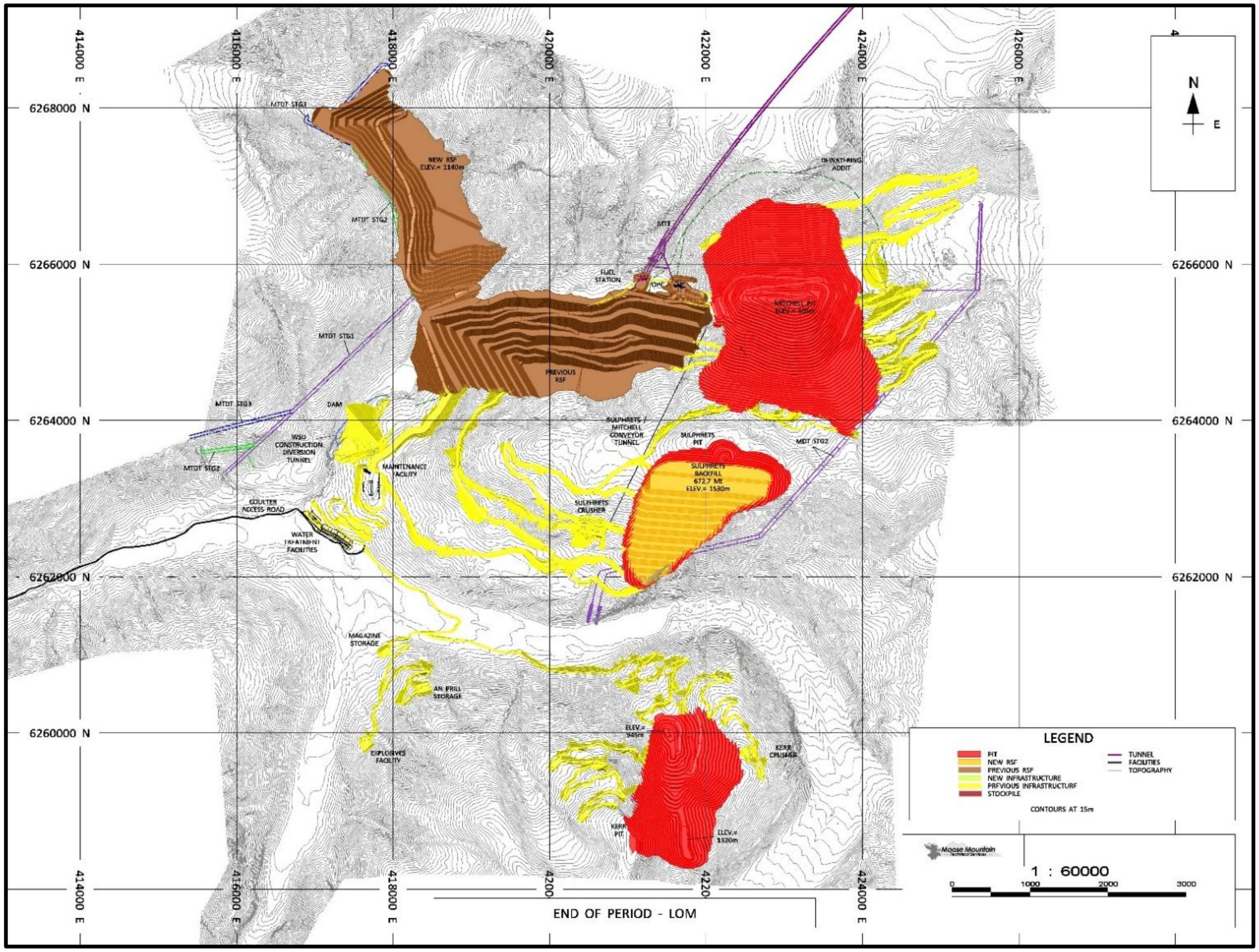
- A conveyor through the Sulphurets-Mitchell Conveyor Tunnel (SMCT) and a connecting conveyor to transport ore from the Sulphurets pit to an ore stockpile at the Mitchell site.
- A separate rope conveyor built to connect the Kerr pit to the SMCT conveyor across the Sulphurets Valley. Waste rock from the Kerr open pit is backfilled into the mined out Sulphurets pit. Ore from the Kerr open pit is transferred to the SMCT to deliver ore to the Mitchell pit site. Both the ore and waste rock that are primary crushed at the Kerr site will use the same rope conveyor transport system.

Figure 1.6 shows the Mine Site area and the various other pits, as well as other on-site infrastructure such as the initial staging, construction and permanent camps, explosive facilities, the Water Storage Facility (WSF), diversion tunnels, and hydro power plants. Access and appropriate haul roads will be provided to all of these areas.

Figure 1.7 shows the main processing facilities at the Treaty OPC, plus other on-site infrastructure such as rail yard and train maintenance building, tunnel muck piles, permanent and construction camps, concrete batch plant, and waste management facility.

Figure 1.10 shows the Mine Site area with the individual pits, RSFs, and major infrastructure including the truck shop, camps, explosives facilities, WSF, water diversion and infrastructure tunnels, the primary crusher and truck dump at Mitchell, and external hauls roads.

Figure 1.10 2016 PFS Open Pit LOM



Source: MMTS

1.12 METALLURGICAL TEST REVIEW

Several wide-ranging metallurgical test programs were carried out between 2007 and 2016 to assess the metallurgical responses of the mineral samples from the KSM deposits, especially the samples from the Mitchell deposit.

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)
- semi-autogenous grinding (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 (UBC)
- dewatering tests by Pocock Industrial Inc. (Pocock) 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. Cyanidation tests were conducted to further recover gold and silver from the gold-bearing sulphide streams (scavenger cleaner tailing from the copper-gold bulk flotation) and pyrite concentrate.

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 (dithiophosphate)/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 (upper zone) deposits. The locked-cycle tests showed that, on average, approximately 85% of copper and 60% of gold in the Mitchell samples, which contain 0.21% copper and 0.72 g/t gold, were recovered into a concentrate containing 24.8% copper. 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.

For the Sulphurets, Iron Cap, and upper Kerr samples, the average head grades of the tested samples fluctuated from 0.25 to 0.62% for copper and 0.23 to 0.60 g/t for gold. The average recoveries reporting to flotation concentrates ranged from 78% to 85% for copper and 41 to 60% for gold. The average copper grades of the concentrates varied from 24 to 28%. The cyanidation further recovered approximately 15 to 29% of the gold from the gold-bearing products.

1.13 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 mill feed will come from open pit mines (upper Mitchell Zone, Sulphurets and upper Kerr Zone) and underground block caving operations (lower Mitchell Zone and Iron Cap deposits). 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. The ore from the Sulphurets deposit will be fed to the plant together with the ore from the Mitchell pit from Years 1 to 17, excluding Years 4, 5, 12, and 13 and with ore from the other deposits during the last four years. Ore from the Kerr deposit, together with ore from the other deposits, will be introduced to the plant during Years 24 to 34, and Year 53, while the Iron Cap ore will be fed to the process plant during Year 32 to the end of mine life.

The proposed flotation process is projected to produce a copper-gold concentrate containing approximately 25% copper. Copper and gold flotation recoveries will vary with changes in head grade and mineralogy. For the LOM mill feed containing 0.55 g/t gold and 0.21% copper, the average copper and gold recoveries to the concentrate are projected to be 81.6% and 55.3%, respectively. As projected from the testwork, the cyanidation circuit (carbon-in-leach [CIL]) will increase the overall gold recovery to a range of 60% to 79%, depending on gold and copper head grades. Silver recovery from the flotation and leaching circuits is expected to be 62.7% 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: ore primary crushing and handling facilities at the Mine Site (Mitchell OPC), a 23 km ore transportation tunnel system between the Mine Site and the PTMA, and a main process facility at the Treaty OPC, adjacent to the TMF.

Gyratory crushers in the comminution plants located at the Mitchell OPC will reduce the mill feed from 80% passing 1,200 mm to 80% passing 150 mm. The crushed ore will be conveyed to a 30,000 t surge bin (two pockets, each 15,000 t) located underground at a train car loading area, prior to being transported by train cars to the Treaty OPC.

A 23 km MTT system has been designed to connect the Mine Site and the PTMA. The crushed ore will be transported through the tunnels by train. This tunnel will also be used for electrical power transmission sourced from the Northwest Transmission Line and for the transport of personnel and supplies for mine operating and water management activities.

The proposed process flow sheet is shown in Figure 1.11.

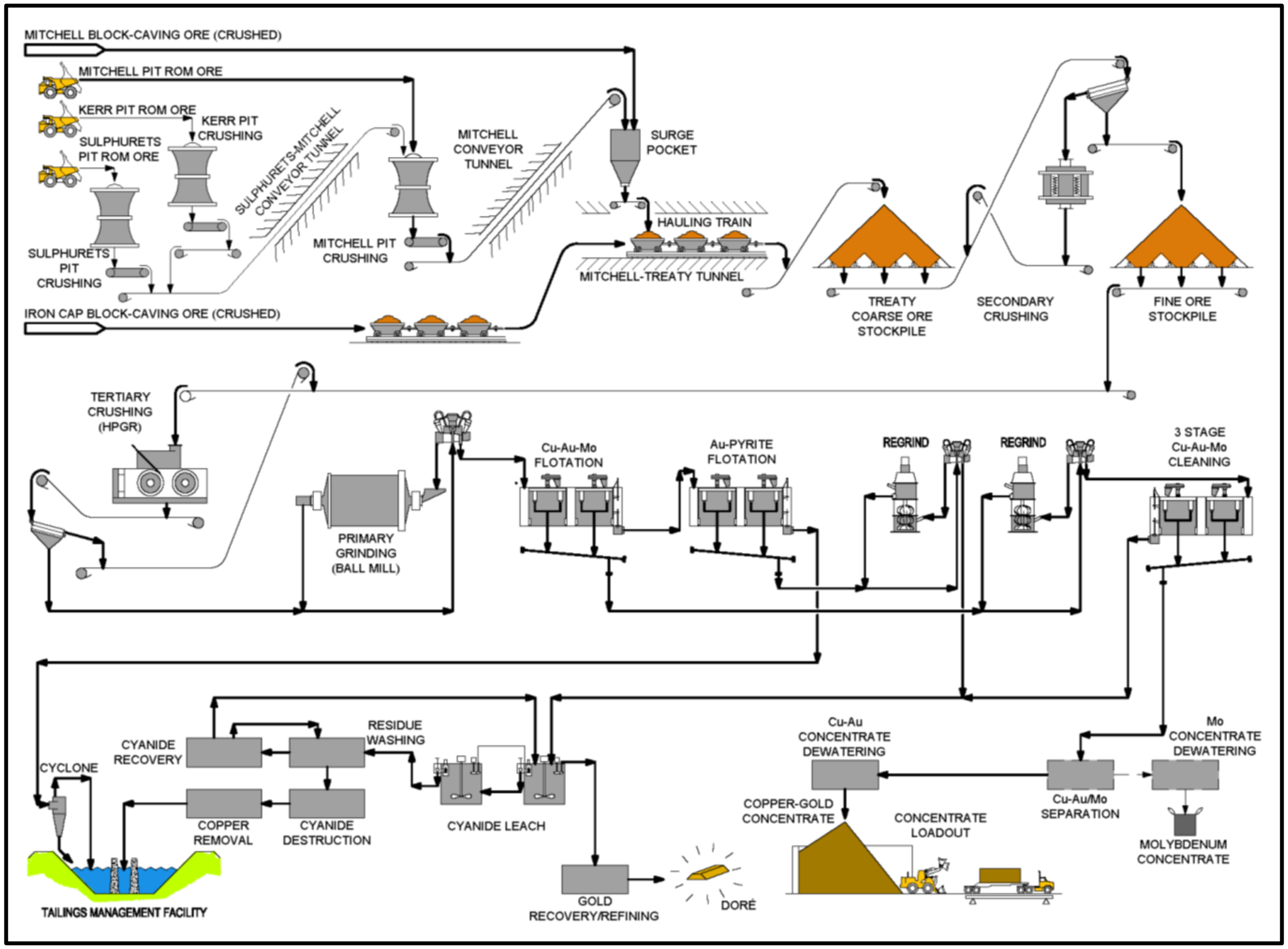
The Process Plant at the Treaty OPC will consist of secondary and tertiary crushing, primary grinding, flotation, concentrate regrinding, concentrate dewatering, cyanide leaching, gold recovery, tailing delivery, and concentrate loadout systems. The crushed ore transported from the Mitchell OPC will be sent to a 60,000 t coarse ore stockpile (COS) adjacent to the tunnel portal. The ore will then be reclaimed and crushed by cone crushers, followed by an HPGR comminution circuit. There is a 30,000 t fine ore stockpile located ahead of the tertiary crushing circuit. The crushing systems will be operated in closed circuits with screens.

The ore 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 levels of copper and other impurities. 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 using CIL treatment 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 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.

Figure 1.11 2016 PFS Simplified Process Flow Sheet



Source: Tetra Tech
Note: ROM – run-of-mine

1.14 TAILING, WATER MANAGEMENT, AND ROCK STORAGE FACILITIES

1.14.1 TAILING MANAGEMENT FACILITY

The TMF (see layout in Figure 1.8) will be constructed in three cells: the North and South cells for flotation tailing, and a lined Central Cell for CIL residue 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 (CDA) 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 (PGA) 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, although an operational phase discharge pipeline and closure spillways are also provided to route the critical duration PMF.

De-pyritized flotation tailing will be stored in the North and South cells. The pyrite bearing CIL residue tailing will be stored in the lined Central Cell. In total, the TMF will have a capacity of 2.3 Bt.

The North and Central 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 Central 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 Central 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 LOM. 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 will form the CIL pond. These dams will also subsequently be raised with cyclone sand shells, but the CIL pond and the Splitter and Saddle dams will incorporate high-density polyethylene (HDPE) and linear low-density polyethylene (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 1,068 m will be required for the North Cell dams and 1,068 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 North and South tailing cells will be reclaimed by floating pump barges and recycled separately to the Process Plant, either for use in the process, for treatment, or to be discharged. Water from the Central Cell will only be directed to the Process Plant for recycling purposes and will not be discharged directly to the receiving environment. 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. As operations switch to the South Cell, the East Catchment Tunnel will be constructed 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 water surpluses 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 LOM, excess water from the Central Cell varies from 0.10 m^3/s to 0.23 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 (if required) and discharge.

Concerns with respect to the construction, operation, and long-term stability of large scale TMFs within the Province of BC have been expressed by the general public, Aboriginal groups, governments, and environmental non-governmental organizations following the Mount Polley Tailings Facility breach that occurred in August 2014. To specifically address these concerns, and to assure the public that the proposed KSM TMF design is robust and safe, Seabridge undertook the following two actions:

- An Independent Geotechnical Review Board (IGRB) was established in January 2015 to independently review and to provide expert oversight, opinion, and advice to Seabridge on the design, construction, operational management, and ultimate closure of the TMF and Water Storage Dam (WSD). The IGRB will review the TMF and WSD on an ongoing basis to ensure that these structures meet internationally accepted standards and practices which effectively minimize risks to employees, lands, and communities.
- Seabridge commissioned KCB to undertake the Best Available Tailing Technology review in August 2015 (KCB 2015) in response to the Independent Expert Engineering Investigation and Review Panel report on the breach of the Mount Polley tailing storage facility. The Review Panel concluded that future projects require not only an improved adoption of best applicable practices (BAP), but also a migration to best available technology (BAT). The KCB report (KCB 2016) also meets the new BC Mining Code requirement that new mines

must provide an alternate assessment of BAT in their provincial permit applications.

The IGRB includes some of the leading world-class experts globally in their field. There are four core members of the IGRB: Dr. Andrew Robertson; Dr. Gabriel Fernandez; Mr. Terry Eldridge, P.Eng. FEC; and Mr. Anthony Rattue, P.Eng., as well as four support members: Dr. Leslie Smith; Dr. Ian Hutchison; Mr. Jim Obermeyer, P.E.; and Dr. Jean Pierre Tournier, whose expertise are called upon as needed. The IGRB provides over 300 years of combined dam design expertise to apply to best design practices for safe construction and operation. Detailed summaries of their experience are included in their first summary report dated April 2016 (IGRB 2016).

The IGRB review of Seabridge's TMF and WSD design was conducted between March 9 and 12, 2015 and was developed to answer five questions:

- Are dams and structures located appropriately?
- Are dam sections, materials, construction methods, and sequencing appropriate for the site?
- What are the greatest design, construction, and operating risks?
- Are the facilities designed to operate effectively?
- Are the facilities designed to be safe?

The IGRB concluded that it was satisfied with the Project's designs and responded favourably to all five questions (IGRB 2016). Additionally, the IGRB presented a series of recommendations for Seabridge to consider during the ongoing engineering design of TMF and WSD as development continues, many of which have been addressed within this PFS.

The BAT study confirmed that the existing TMF design, consisting of centerline dams constructed with double cyclone sand and a till core in association with wet tailings deposition, is the best available technology for tailings deposition, and the most environmentally responsible design to minimize long-term risks associated with the proposed TMF for the Project. This conclusion confirms the findings from KSM's IGRB that the TMF's design is robust and appropriate for KSM's site-specific characteristics.

The BAT study also confirmed that the TMF design that was included in the Project design, which received provincial and federal EA approval, is the best possible design for eliminating risks associated with operation and closure. The study specifically determined that filtered tailing options are impractical and would result in greater environmental impacts and risks, contrary to the assertions of many environmental groups who have advocated that only filtered tailing disposal technologies should be implemented.

As a further step in its review process, Seabridge commissioned an independent review of the BAT report by Dr. Dirk van Zyl. Dr. van Zyl is a world-recognized expert in tailings, mined-earth structures, and sustainability with over 40 years of experience. He is currently a faculty member at UBC's Faculty of Applied Science and was a member of the Independent Expert Engineering Investigation and Review Panel investigating the Mount Polley tailing storage facility breach. In his review of the KCB report (KCB 2015), Dr. van Zyl supported the overall conclusions of the KSM BAT report and that using filtered tailings at KSM is not a feasible option as it will not result in moving to zero failures.

1.14.2 MINE SITE WATER MANAGEMENT

The overall site water management strategy, including the discharge from the WSF via the High-density Sludge (HDS) water treatment plant (WTP) was the strategy that was reviewed and approved during the EA review process.

DIVERSION TUNNELS AND SURFACE DIVERSION

Three diversion tunnel routes totalling approximately 22.4 km will be required to route glacial melt water and non-contact valley runoff from the Mitchell and McTagg valleys around the Mine Site. These tunnels are shown on Figure 1.10.

The open pit phase of the Mitchell Diversion Tunnel (MDT) and the twinned McTagg Diversion Tunnels (MTDTs) are sized to convey flows from an average 200-year storm. When the Mitchell block cave operation commences, an additional MDT paralleling the open pit phase tunnel will be driven to protect the underground workings, which are more sensitive to inflows than the open pits.

The provision of a second MDT during the underground phase provides redundancy against blockage, as each individual tunnel can carry typical freshet flows. The provision of twin tunnels also allows for switching base flows between adjacent tunnels if access for maintenance is required. The single initial tunnel can be maintained during winter low flow periods.

The open pit phase MDT will have a cross-sectional area of 22.8 m^2 and an overall length of 7.0 km. This tunnel 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 MDT 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 23, the MDT will be augmented with a second (twin) tunnel to provide protection against the 1,000-year storm flow to the underground workings.

Each of the twinned MTDTs 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.

To facilitate construction in the Mitchell Valley and the staging of water management as the Mitchell and McTagg RSFs rise and fill the valley areas, an approximately 5.4 km long Mitchell Valley Drainage Tunnel (MVDT) will be constructed under Mitchell Valley to carry the existing flows from Mitchell Creek, which are naturally affected by contact with surface mineralization. When the mine is in operation the tunnel will convey contact water from the Mitchell workings and the mineralized area upstream of the deposit (via the North Pit Wall Dewatering Adit [NPWDA]), around the RSFs into the WSF. If the MVDT was not provided, surface channels would have to be staged in multiple increments and significant energy dissipation structures would have to be constructed in areas of deep overburden. The tunnel is designed to convey the 200-year return period, 24-hour duration storm peak flow (181 m^3/s), which requires a cross section of 6.0 m wide by 6.2 m high.

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 RSF to convey surface diversion flows down to diversion pipelines and channels on the west and east sides of the 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.

WATER STORAGE FACILITY

All contact water from the Mine Site areas (open pits, RSFs, roads, infrastructure) will be directed to the WSF, located in the lower Mitchell Creek area. The WSF will be formed with a 165 m high rock fill asphalt core dam built to full height by Year −1 and is sized to store annual freshet flows and volumes resulting from a 200-year wet year. The WSF dam is founded on competent sedimentary rock foundations. Seepage will be controlled by the asphalt core in the dam and the dam foundation will be grouted. A seepage collection pond will return seepage water to the WSF. 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.

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

WATER TREATMENT

Mine area contact water will be treated with a HDS lime water treatment, of which the discharge from the plant was approved in principle as a component of the environmental assessment review process. The HDS WTP will have initially required capacity of 4.4 m^3/s (four operating circuits), with a fifth circuit provided to treat up to 5.4 m^3/s. The additional circuit manages flow increases that may occur due to natural hazards or extreme events. The additional treatment capacity also allows sections of the HDS WTP to be shut down for maintenance when required. In Year 5, two additional circuits will be installed bringing total ultimate treatment capacity to 7.5 m^3/s.

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 HDS WTP will operate year-round at a variable rate, averaging 2.2 m^3/s annually, with lower rates during winter and reaching 7.5 m^3/s during high streamflow periods.

During pre-production operation of the HDS WTP, sludge from the HDS 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 HDS WTP will be filter-pressed and trucked to the MTT. The sludge will be added to the MTT ore trains and passed through the ore milling process to add necessary alkalinity to the process and ultimately disposed of in the tailing pond.

Additional hydropower will be generated in the Energy Recovery Power Plant from the flow of treatment water from the WSF to the HDS WTP, which is located at a lower elevation in the Sulphurets Valley.

During the initial construction period, to maintain existing water quality, temporary water treatment plants (TWTPs) located in the Mine Site 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 TWTPs and will be hauled to permanent disposal sites within the RSF or TMF once the diversion tunnels and the HDS WTP are operational. The temporary water treatment plants at the Mine Site were part of the early stage construction permits approved in September 2014.

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

1.15 ENVIRONMENTAL CONSIDERATIONS

The Project is subject to the BC *Environmental Assessment Act*, the Canadian *Environmental Assessment Act*, and Chapter 10 of the Nisga'a Final Agreement (NFA).

As of June 2016, the Project has successfully gone through the provincial and federal processes, and the appropriate certificates/approvals have been obtained. Additionally, permits for early stage constructions activities for the first two and half years of site activity were obtained. These permits covered the following mine components:

- KSM Project *Mines Act* and *Environmental Management Act* Permit Application for Limited Site Construction – May 2013
- Special Use Permits for the Coulter Creek Access Road (CCAR) and TCAR
- KSM construction camps
- KSM Project Treaty Transmission Line
- MTT Permit Application.

Seabridge is currently in the process of obtaining numerous provincial and federal permits to allow for the construction of parts of the Project, as well as expanding exploration activities, including but not limited to the following:

- Fisheries Authorization application, including draft Compensation Plans
- Metal Mining Effluent Regulations (MMER) Schedule 2 Amendment Application
- *International Rivers Improvements Act* Licence Application.

The Project underwent a harmonized EA process with the provincial and federal governments. The governments conducted the EA cooperatively in accordance with the principles of the Canada-BC Agreement on Environmental Assessment Cooperation (Cooperation Agreement 2004). The process included a working group comprised of federal and provincial officials, the Nisga'a Lisims Government (NLG) and Aboriginal groups, and local government agencies. Representatives of the US federal and Alaska state agencies were extensively involved in the EA process, as a matter of courtesy, given that the mineral deposits are located on a tributary of the Unuk River, a transboundary river, 30 km upstream of the US/Canada border. Authorizations are not required from any US federal or state regulatory agency for the Project to proceed into construction and operation.

1.15.1 COMMUNITY AND ABORIGINAL ENGAGEMENT

A cornerstone of Seabridge's successful strategy of securing public support for the Project, to assist in the environmental approval and permitting processes and to promote the future development of the Project, was the undertaking of respectful, thorough, and exhaustive public and Aboriginal engagement. This strategy was initiated in the fall of 2007, prior to the development of the initial project description that initiated the EA process and continues today, more than two years after receipt of the federal and provincial EA approvals. A key contributing factor in this strategy was the opening of the local Smithers office in March 2011 and the hiring of employees resident to the area to staff the office.

During the EA process, Seabridge received letters of support from the following organizations:

- Town of Smithers
- Town of Terrance
- Gitxsan Nation.

Seabridge has also secured a Benefits Agreement with the Nisga'a Nation in June 2014. This Benefits Agreement focused on economic benefits; opportunities for jobs; and contracting, ongoing engagement, and project certainty. Additionally, a "Sustainability Agreement" was reached with the Gitanyow Wilps also in June 2014. In this agreement Seabridge agreed to provide funding for certain programs relating to wildlife, fish, and water quality monitoring to address some of the concerns raised by the Gitanyow Huwilp, as well as for a committee to establish a means of maintaining communications about project-related issues throughout the life of the Project. Discussions are underway with the remaining Aboriginal peoples whom have an interest in the ensuring the Project is developed responsibly.

Seabridge has also been active in the communities prior to and after receipt of the EA approvals, focusing on community education and employment initiatives. To date, Seabridge has donated an excess of Cdn$702,000.

Seabridge's ongoing success in its successful community and Aboriginal engagement programs was the recent receipt of the permits required to develop the Deep Kerr exploration adit. A 10-month process culminated in receipt of the permits in October 2016.

1.15.2 Closure and Reclamation

The Project will be closed in accordance with the closure plan outlined in Section 20.7, and in further detail in the Application/EIS (Rescan 2013).

Closure and reclamation planning for the Project will contribute to the success of closure and reclamation during mining and at the end of mine life, which will reduce the need to restructure Project components, limit the amount of material re-handling, and reduce the environmental effects of the Project. Mine development and operation will incorporate techniques to minimize surficial disturbance and, where possible, progressively reclaim areas affected during construction and operation. Stabilizing and rehabilitating surfaces will reduce the potential for degradation of the resources due to extended exposure to climatic factors, reducing closure-related capital costs at the cessation of mining activities.

Closure activities will be conducted within the following regulatory framework:

- *British Columbia Mines Act* (1996A) and Health, Safety and Reclamation Code (BC Ministry of Energy, Mines and Petroleum Resources [MEMPR] 2008)
- Federal *Fisheries Act* (1985)
- BC *Environmental Management Act* (2003)
- BC *Water Act* (1996B)
- Riparian Management Area Guidebook (BC Ministry of Forests [MOF] and BC Ministry of Environment [MOE] 1995).

Part 10 of the BC Health, Safety and Reclamation Code (BC MEMPR 2008) focuses on reclamation and closure. Section 10.7 of the code identifies reclamation standards. Section 10.7.4 (Land Use) of the code indicates "The land surface shall be reclaimed to an end land use approved by the chief inspector that considers previous and potential uses." Section 10.7.5 (Capability) of the code indicates "Excluding lands that are not to be reclaimed, the average land capability to be achieved on the remaining lands shall not be less than the average that existed prior to mining, unless the land capability is not consistent with the approved end land use." Section 10.7.6 (Long Term Stability) of the code states "Land, watercourses and access roads shall be left in a manner that ensures long-term stability."

The Project closure and reclamation plan has three objectives that provide assurance to the province that the site will be left in a condition that will limit any future liability to the people of BC:

- to provide stable landforms
- to re-establish productive land use
- to protect terrestrial and aquatic resources.

1.16 GEOHAZARDS

Geohazard and risk assessments were completed for the proposed facilities within the 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.

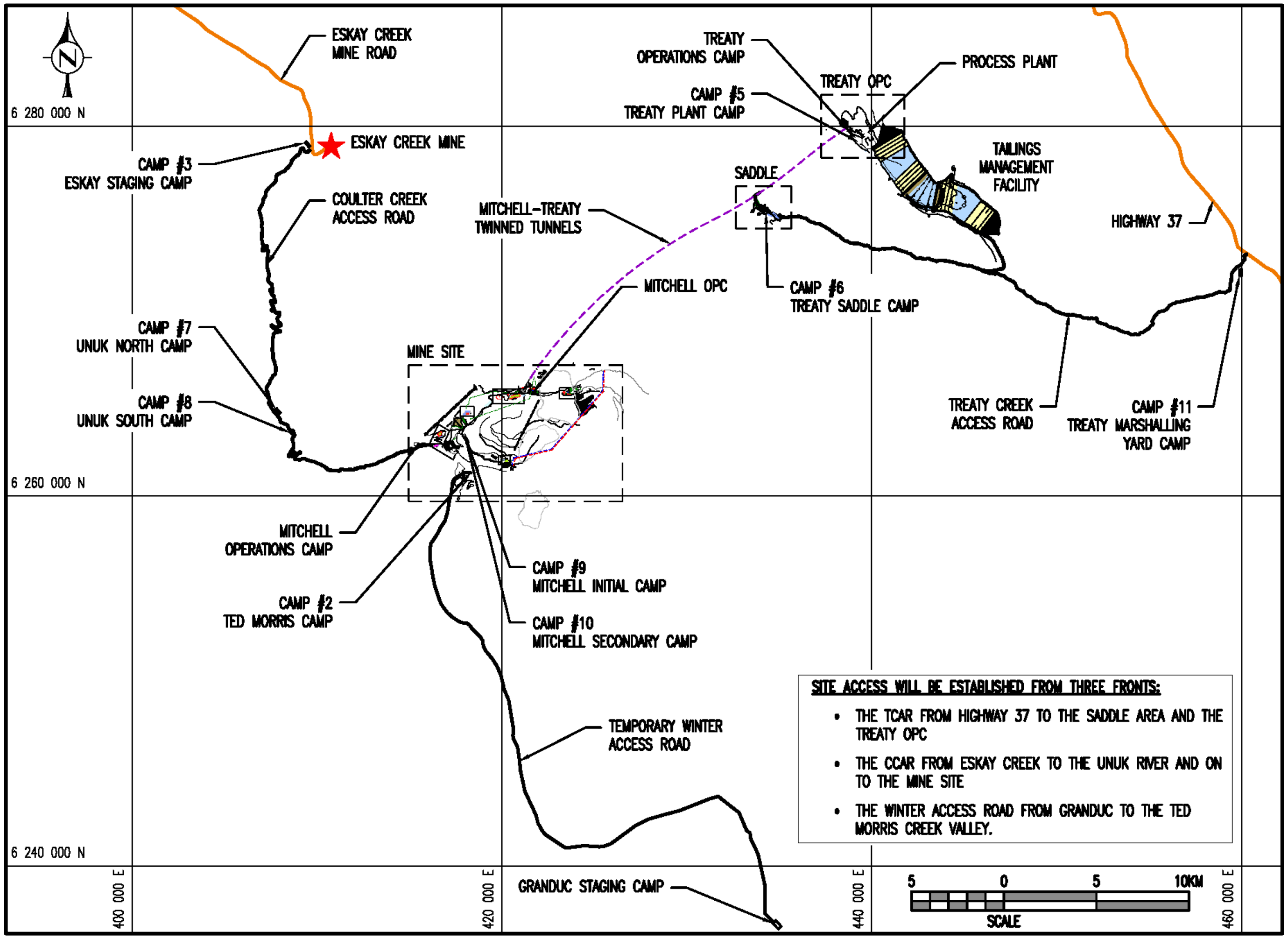
1.17 SITE ACCESS AND INFRASTRUCTURE

Site access will be established from three fronts (Figure 1.12):

- the TCAR from Highway 37 to the Saddle Area and the Treaty OPC
- the CCAR from Eskay Creek to the Unuk River and on to the Mine Site
- the Winter Access Road from Granduc to the Ted Morris Creek Valley.

On-site infrastructure consists of tunnels, ore transportation system, and electrical power supply and distribution infrastructure. Off-site infrastructure comprises new concentrate storage in the Port of Stewart.

Figure 1.12 2016 PFS Site Access and Camp Locations



Source: Tetra Tech

1.17.1 PERMANENT ACCESS ROADS

There will be two primary permanent access roads to the Mine Site and PTMA.

The CCAR will be primarily a single-lane, radio-controlled road constructed for moving large equipment and supplies to the Mine Site. An existing road starts at 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 and select bridges will be required.

The new 35 km long CCAR starts near the former Eskay Creek Mine and follows 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 passive snow avalanche control during alignment layout, but no active measures are planned (e.g. snow sheds) since the road is intended to close during winter due to high maintenance cost. During winter, access to the Mine Site will exclusively be through the MTT.

The TCAR 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. 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. At this juncture, TCAR continues toward the Saddle area as a single lane road, while the North Treaty Access Road (NTAR) will follow the west side of the North Treaty Creek/Teigen Creek Valley for approximately 12 km to the Treaty OPC, TMF, and east portal of the MTT. Initially the lower NTAR will be built low in the valley to facilitate access for construction of the North 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 Dam starts, the remaining 5.7 km of the upper NTAR will need to be constructed. These roads will be used to transport supplies, equipment, and crew members to and from the Treaty OPC, and to transport concentrate to Highway 37 during the life of the mine.

1.17.2 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 construction equipment, materials and supplies during the winter season of the first construction year, prior to CCAR pioneering road completion. 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 Mine Site. It will also allow access for the construction of portions of the CCAR, near its east end and to the Mine Site.

1.17.3 OFF-SITE INFRASTRUCTURE

Copper concentrates produced at the Treaty OPC will be filtered at the Process Plant and transported by contract trucking firms via Highway 37 and 37A to one of the port vendors in Stewart, BC. A concentrate storage building (approximately 100 m by 66 m) will be required. Copper concentrates will be loaded via ship loader and shipped via ocean transport to overseas smelters.

Multiple staging areas will be used in the Project, with the majority of equipment and materials anticipated to be delivered to the Port of Stewart, supplemented by overland freight delivered to Terrace or Stewart. As freight is received by the Project it will be staged along transport routes at staging areas in the Stewart Port, Terrace, Smithers, and ultimately at the marshalling/staging area at the Highway 37/TCAR intersection, prior to shipping material and equipment to the point of installation by the Project Team.

1.17.4 MTT TRANSPORT SYSTEM

The MTT have been revised from the designs used in the Application/EIS (Rescan 2013) to accommodate the change from an ore conveyor system, truck delivery system for personnel and freight, and a tunnel based fuel pipeline, to a more efficient train-based system for ore, personnel, and supplies. The trains will travel on a conventional ballasted track structure, be electrically driven by an overhead catenary system, and be controlled by an automated train control system managed from a remote control room without an on-board operator. Similar systems are commonly used in other large tonnage mining operations.

MTT Ore Transport

At the Mitchell OPC, ore will be crushed then conveyed through a tunnel and dumped into two live underground ore bins within the MTT. Loading chutes under the ore bins will feed ore into awaiting trains that will transport the ore to an unloading station at the Treaty end of the MTT. The train cars will dump ore into a live underground unloading bin. Apron feeders will unload the bin onto a conveyor to transport the ore to the top of the Treaty COS.

Each ore train will consist of one, 140 t electric locomotive and 16, 42 m^3 belly dump ore cars that has the capacity to deliver 800 t/h from Mitchell to Treaty based on 90-minute cycle times. On average, eight trains will deliver approximately 130,000 t/d (approximately 5,400 t/h) of ore to meet the process plant requirements. An additional four trains will be purchased to provide available train operating time for mechanical availability and to handle an increase in plant feed of up to 10,000 t/h in order to replenish coarse ore inventories as required.

MTT FREIGHT TRANSPORT

Transportation of personnel, freight and fuel through the MTT will be handled by rail with sufficient mechanical availability from the train fleet discussed above. On surface, staging areas near the Treaty portal will be used to load personnel, freight, and fuel onto specialty train cars. These staging areas will be road accessible. Surface cranes will load the flatbed train cars with freight and personnel will be loaded from a dedicated passenger siding at Treaty. Empty fuel train cars, which will carry a removable fuel tank, will be loaded via fuel lines from the main Treaty fuel storage tank.

The specialty personnel, freight, and fuel train cars will be shunted by 20 t battery locomotives into the tunnel on the Treaty end, then picked up by the 140 t electric locomotives and transported from Treaty to Mitchell.

Three separate, enclosed, underground staging areas near the Mitchell portal will be used to offload passengers, freight, and fuel respectively. Personnel will exit the Mitchell portal by bus. Freight and fuel staging areas will include gantry cranes to offload the train payloads onto awaiting flatbed tractor-trailer units. Fuel train cars will be re-loaded with an empty fuel tank for return to Treaty.

All freight hauls will be scheduled during stoppages in ore delivery. Train scheduling will also occlude personnel transfer when freight or ore hauls are in the tunnels to increase traffic safety and to ensure people will not be exposed to explosives, fuel, or hazardous substances. Estimated freight, fuel, and personnel movement requirements through the MTT will call for a daily average of five return train trips.

1.17.5 TUNNELING

A total of nine major tunnel projects will be excavated throughout the Project life, during the pre-production and during operations. These tunnels will be classified as either infrastructure tunnels or water tunnels. This does not include development work for the block caves.

The infrastructure tunnels will provide for the transportation of ore, personnel, and supplies between the Mine Site and the Treaty OPC. The principal infrastructure tunnel is the MTT, which transports all the ore from the Mitchell OPC to the Treaty OPC and personnel and supplies between the Treaty OPC and Mine Site, via the train system. During operations a conveyor tunnel for the transfer of ore from Sulphurets pit, and later, ore and waste from Kerr pit to the Mitchell Valley will be constructed.

The water tunnels include diversion tunnels named the MDT, the MTDT, and the MVDT; and the slope drainage tunnels for the Mitchell high wall (NPWDA) and the Snowfields landslide (Snowfields Slide Dewatering Adit [SSDA]).

Engineering for the major components of the MTT have been developed by two experienced tunnel contactors and have been adapted to form the estimates for the infrastructure and water tunnels.

Table 1.6 Summary of Infrastructure and Water Tunnels

Tunnel	Length (km)
Pre-production Tunnels	
Infrastructure Tunnels	
MTT	51.4
Water Tunnels	
MDT Open Pit Phase	9.1
MTDT Stage I	8.0
MVDT and OPC Decline	5.4
Construction Diversion Tunnel (CDT) at WSD	0.9
Seepage Collection Tunnel (SCT) at WSD	1.8
Pre-production Tunnels Total	**76.6**
Operating Phase Tunnels	
Infrastructure Tunnels	
Iron Cap Connection	1.0
SMCT	3.0
Water Tunnels	
MDT Underground Phase	8.9
MTDT Stages II and III	14.7
NPWDA and SSDA	5.3
East Catchment Tunnel	4.0
Mitchell Block Cave Dewatering Tunnels	12.0
Operating Phase Tunnels Total	**48.9**

MTT Tunneling

Crushed ore from the open pits and underground operations will be transported through the MTT to the COS at the Treaty OPC. The MTT have been revised from the designs for the Application/EIS (Rescan 2013) from a conveyor to a train-based system; however, the tunnel location and alignment have not changed. The MTT will comprise approximately 51.4 km of excavation, including the main tunnels, cross-cuts for ventilation/safety and track cross-overs for maintenance during operations, as well as sidings for loading, unloading, and freight, as well as excavations for the underground loading and unloading infrastructure.

It is anticipated that four tunnel support classes (TSC) will be required for ground control along the length of the MTT as described in Table 1.7.

Installation of these four types of tunnel support affects the daily advance rate of the tunnel. Table 1.7 uses the distribution of TSC, along with associated advance rates as determined by the contractors. These have been used to choose an appropriate rate to use in the scheduling of the MTT as well as the other tunnels.

Table 1.7 Tunnel Support Class and Advance Rates for the MTT

Tunnel Support	Percentage of Tunnel (%)	Advance Rates (m/d)
TSC I	29	7.1
TSC II	21	6.8
TSC III	42	6.6
TSC IV	8	5.1

The tunnels will be driven in accordance with the BC *Mines Act* and Regulations using mechanized drill and blast techniques and will follow the conditions contained within the License of Occupation.

The MTT are on the critical path of the construction schedule and have therefore been broken into two segments to allow for concurrent development workplaces resulting in a shorter total tunnel construction period. This will be accomplished with headings at the Treaty Valley, an adit as the saddle of a transecting valley, located 6.1 km from the Treaty portals, and by headings in the Mitchell Valley thus creating six active headings. During construction, rail will be installed in both the North and South tunnels for the future operations rail haulage system; however, only the North Tunnel will be used for hauling tunnel muck during construction.

During tunnel construction, installation of the infrastructure required for the operation of the train haulage system will be scheduled where it doesn't disrupt the tunnel advance rate. This will include the ore train electrical system, the mine area electrical feed line, loading and unloading facilities, and the tunnel ventilation system required for MTT operations. Time is provided in the MTT construction schedule for mobilization at the start and after final breakthrough, for completion of the tunnel infrastructure, and system testing and commissioning.

Each pair of headings will rely upon a primary and a secondary ventilation circuit. The primary circuit will provide fresh air under positive pressure through the South Tunnel. Cross cuts near the advancing face will exhaust out the North Tunnel. Two secondary circuits, made up of fans on flexible air ducts, will be established to intercept fresh air from the primary circuits and ventilate the pair of advancing faces.

During the operations phase, air will be moved through the tunnel by the piston effect of the trains. To allow for segments of the MTT to be isolated for maintenance, sets of ventilation doors with axial vane fans will be installed at the portals and at the track cross-overs for each tunnel.

In addition to the twin haulage tunnels, the MTT will include sidings at the Mine Site end for freight, personnel, and fuel transportation, as well as twin underground loading pockets with train ore loading systems connected to the Mitchell OPC primary crushers via a tunnel and conveyor. Also at the Mine Site end of the MTT is a future tunnel access to the Mitchell and Iron Cap block caves. At the Treaty end of the MTT an underground ore bin is connected to the surface COS via a tunnel and conveyor.

During operations, the crosscuts between the twin tunnels will be bratticed off to provide independent airways for egress for personnel. The brattices will have air sealed man-doors and additionally, thirty-six, 12-person refuge stations will be set up in the cross-cuts.

COSTING AND SCHEDULING OF OTHER TUNNELS

The detailed contractor tunneling estimate for the MTT has been adapted to estimate the excavation costs and advance rates for the other associated excavations in the MTT, as mentioned above. They have also been used for the other infrastructure and water tunnels. The elements used in estimating all tunnels are:

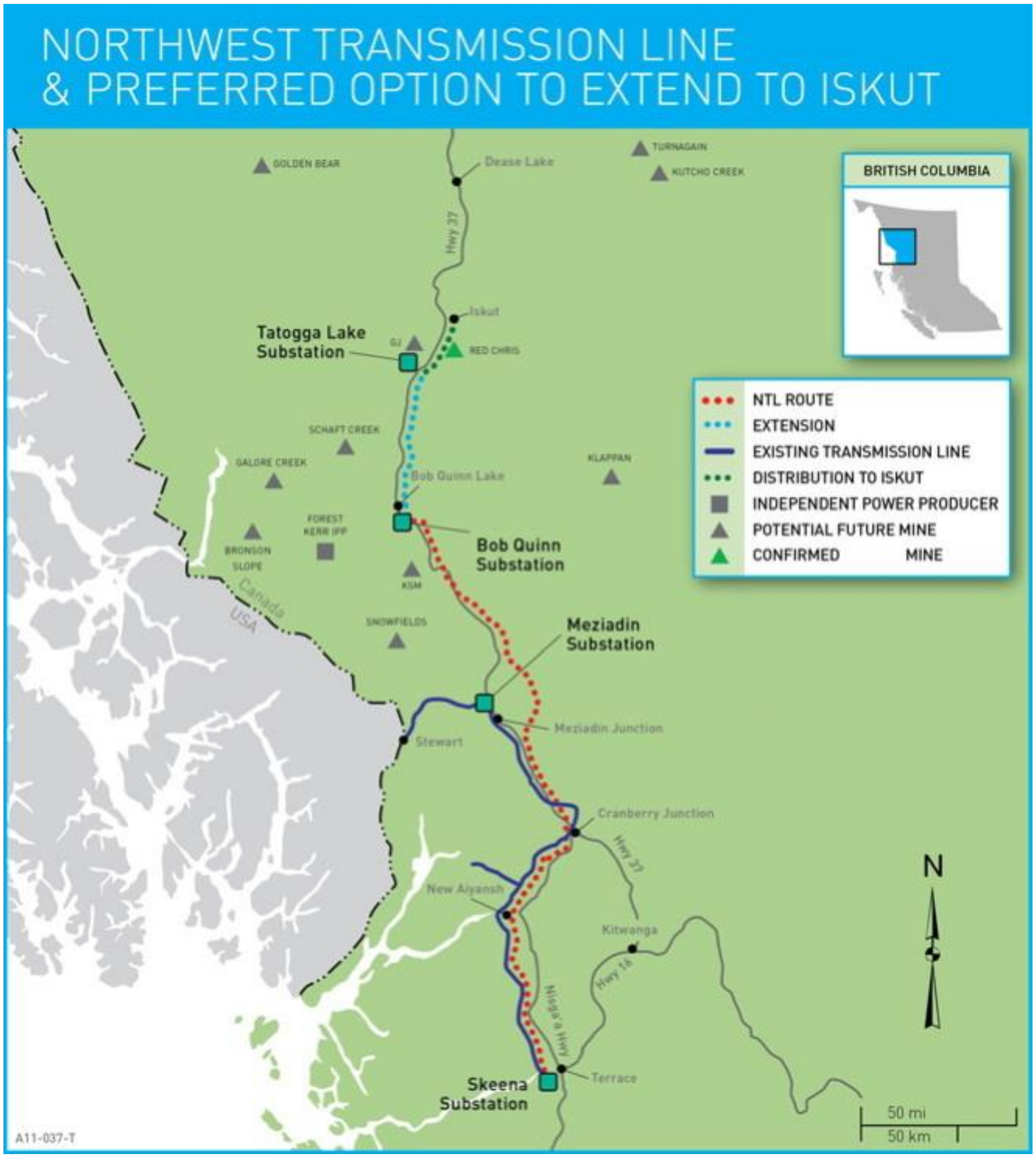
- portals – preparation, excavation and setup
- mobilization and equipment use
- excavation of tunnels according to tunnel support type
- bulk excavation
- tunnel cleaning
- demobilization.

1.17.6 POWER SUPPLY AND DISTRIBUTION

BC Hydro is the electric utility that serves the Project area. Electric service for the Project will be from BC Hydro's NTL that was completed in 2014 and parallels Highway 37. The NTL provides an economic and reliable source of power at a cost of US$0.05/kWh

The new 344 km long, 287 kV, NTL runs from the Skeena Substation on the BC Hydro 500 kV grid near Terrace, BC, to Cranberry Junction, from which point it roughly parallels BC Highway 37 to its terminus at Bob Quinn. The Project will construct a 30 km long, 287 kV transmission extension from the NTL, originating at the Treaty Creek Switching Station (BC Hydro designation TCT) and terminating at the Treaty OPC No.1 Substation (designation FLT1) that will be part of the Project infrastructure. This spur line will parallel the TCAR in a common corridor. Land tenure for the right-of-way has been obtained. The Treaty Creek Switching Station on the NTL will be approximately 20 km south of Bell II. Figure 1.13 is a map from BC Hydro illustrating the routing of the NTL.

Figure 1.13 Map of the Northwest Transmission Line



Source: BC Hydro

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

Seabridge commissioned BC Hydro to carry out a Facilities Study for the Project, following the previously completed BC Hydro System Impact Study. The Facilities Study is the final evaluation required by the utility to define connection costs and terms of electric service. A draft version of the Facilities Study has been issued. Upon the final issue of the study, the parties will be in a position to sign a Facilities Agreement that, in conjunction with the Electricity Supply Agreement (ESA), forms the standard contract for the supply of electric power for a large bulk Transmission Service Customer such as Seabridge. The Project, on the basis of the current application, has priority for service from the new NTL. Currently there is a reservation of 150 MVA for the Project, but an application has been made to increase this to 200 MVA.

Service to the Mine Site and PTMA will be provided from KSM Substation No. 1 via a 138 kV cable (24 km approximate length, including lead-in to the portals) through one of the twin MTT connecting the Treaty OPC to the Mine Site. This supply will terminate at the 138 kV to 69 kV to 25 kV step-down Substation No. 2 at the Mitchell OPC. This substation will be of the gas insulated (GIS) type, which is very compact and allows for an 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 MTT tunnel that carries the main power cables.

There will be 25 kV cables from Substation No. 2 feeding the primary crushing plant and train loading facilities. In addition, 25 kV cables will also feed half of the rectifier stations as located in the MTT tunnels for the main Mitchell to Treaty rail transport system. Twenty-five kilovolt and 69 kV cables will also connect to overhead lines fed from the substation as required to supply the Mitchell and future pits and other facilities including the HDS WTP, WSF dam pumping installation, the Mitchell hydro plant, truck shop, camp, explosives plant, and other installations in the Mine Site.

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 49,205,060 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 programmable logic controller (PLC) systems. The generated power will be metered and fed into the local mine distribution power lines. The generating plants will either displace costly Tier 2 utility power (as per BC Hydro Tariff 1823), or will be sold back to BC Hydro under their Standing Offer Program for small generating schemes. Thus, the per-kilowatt-hour dollar value of the generated electricity will be relatively high, significantly more than the average project overall purchase price of electricity.

This section provides a brief summary of the energy recovery generation plants.

Water Treatment Plant Energy Recovery

This energy recovery facility will use the water running downhill from the water storage pond to the HDS WTP to generate electric power. Two twin jet Pelton type impulse turbines will be used as required to match the flow. The output will be fed into the plant power distribution system at the HDS WTP. This facility will continue to operate after mine closure.

Tailing Energy Recovery

The tailing energy recovery pump turbines will be located in the tailing discharge system running from the Process Plant to the TMF. The energy recovery system will be based on the available elevation differences, and will consist of two series stations each utilizing a slurry pump running in reverse as a turbine. Induction generators will feed power back into the local plant electrical distribution system. The two series energy recovery stations will be located in small buildings at elevations 1,030 m and 1,000 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 MDTs. 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 local mine 25 kV 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 energy recovery will be constructed in two stages corresponding to water diversion Stage 2 (Year 10) and diversion Stage 3 (Year 15). McTagg diversion Stage 1 does not warrant an energy recovery plant.

The plant will consist of two Pelton turbines, and will feed power into the plant distribution system at the HDS WTP. This facility will continue to operate in Phase 3 and after mine closure.

1.18 PROJECT EXECUTION PLAN

1.18.1 ENGINEERING PROGRAM

The Project will complete 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 WSF construction. This will include tendering the construction packages for the early works scope and procurement of the long-lead items to maintain the Project schedule. The second phase will include the detailed design of the remaining facilities, major equipment procurement, and tendering process for the remaining construction packages.

It was assumed for the 2016 PFS that the Project will be constructed using the engineering, procurement, and construction management (EPCM) approach with a management team located at both the Mine Site and the Treaty OPC. The Owner will supply all the temporary construction camps and service contractors to manage daily activities on site.

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

1.18.2 SITE CONSTRUCTION

It is assumed for the 2016 PFS that the method of construction will be an open 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.

Mine Site construction begins with the development of the site access roads to the HDS WTP area, WSF, tunnel entrances, CCAR, and building locations. Early works material and equipment will be mobilized via the Winter Access Road and the pioneering road along the CCAR alignment. Major equipment, general construction materials, and heavy earth moving equipment will be mobilized via the CCAR.

Construction material and equipment for the PTMA will be transported using the TCAR. Helicopter support is planned to be used prior to TCAR pioneering road completion. The construction schedule for both sites is coordinated around the development of the MTT.

1.18.3 COMMISSIONING PROGRAM

The commissioning program will 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.

1.19 CAPITAL COST ESTIMATE

An initial capital of US$5.005 billion is estimated for the Project. All currencies in are expressed in US dollars, unless otherwise stated. Costs have been converted using a fixed currency exchange rate of US$0.80 to Cdn$1.00. 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 1.8.

This 2016 PFS estimate is prepared with a base date of Q2 2016. 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 2016, and are budgetary and non-binding. Pricing for all major equipment is based on budgetary quotations provided by vendors obtained in Q1/Q2 2016. For non-major equipment, pricing is based on in-house data or recent quotes from similar projects.

All equipment and material costs are based on FCA (Free Carrier) Ex-works (Incoterms 2010). Other costs such as spares, taxes, duties, freight, and packaging are covered separately in the indirect costs section of the capital cost estimate.

Table 1.8 2016 PFS Initial Capital Cost Summary

Area No.	Area Description	Item Cost (US$)
1 – Direct Costs		
1.1	Mine Site	1,218,098,000
1.2	Process	1,336,423,000
1.3	TMF	440,697,000
1.4	Environmental	14,592,000
1.5	On-site Infrastructure	22,851,000
1.6	Off-site Infrastructure	119,580,000
1.7	Permanent Electrical Power Supply and Energy Recovery	158,861,000
	Total Direct Costs	**3,311,102,000**
2 – Indirect Costs		
2.91	Construction Indirect Costs	449,092,000
2.92	Spares	34,314,000
2.93	Initial Fills	19,664,000
2.94	Freight and Logistics	99,015,000
2.95	Commissioning and Start-up	6,120,000
2.96	EPCM	230,957,000
2.97	Vendor's Assistance	23,075,000
	Total Indirect Costs	**862,237,000**
3 – Owner's Costs		
3.98	Owner's Costs	160,232,000
4 – Contingency		
4.99	Contingency	670,995,000
	2016 PFS Initial Capital Cost Total	**5,004,566,000**

Notes: Costs have been rounded to the nearest thousands of dollars.

Sustaining capital costs were also estimated leveraging the same basis of information applied to the initial capital estimate with respect to vendor quotations, labour, and material costs. The sustaining capital costs total US$5.503 billion and consist of:

- open pit mine development, principally mobile fleet replacement
- underground mine development at Mitchell and Iron Cap block cave mines
- process improvements, principally at Mitchell and Treaty OPC, MTT, and SMCT
- TMF expansions, mainly comprising dam raises and CIL basin expansions
- permanent electrical power supply and energy recovery systems
- Project indirect costs, including construction indirects, spares, freight and logistics, EPCM, vendor assistance, and contingency.

1.20 OPERATING COST ESTIMATE

The average operating cost for the Project is estimated at US$12.03/t milled at the nominal process rate of 130,000 t/d or US$12.33/t for the LOM average as shown in Table 1.9. The cost estimates in this section are based upon budget prices in Q1/Q2 2016 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$0.80. The expected accuracy range of the operating cost estimate is +25%/-10%.

The estimates do not include energy recovery credit (approximately US$0.12/t milled LOM) from mini hydropower stations and the cost (approximately US$0.15/t milled LOM) related to Provincial Sales Tax (PST).

Table 1.9 2016 PFS Average Operating Cost Summary

	At the Nominal Feed Rate of 130,000 t/d		LOM Average (US$/t milled)
	(US$/a)	(US$/t milled)*	
Mine			
Mining Costs – Mill Feed	190,223,000**	4.59**	4.59
Open Pit – Mill Feed	-	*4.40***	*4.40*
Block Caving – Mill Feed	-	*4.99***	*4.99*
Mill			
Process	251,066,000	5.29	5.34
G&A and Site Service			
G&A	43,272,000	0.91	1.03
Site Service	18,914,000	0.40	0.44
Tailing and Site Water Management			
Tailing Dam Management	6,065,000	0.13	0.13
Selenium Water Treatment	9,469,000	0.20	0.21
HDS Water Treatment	22,033,000	0.46	0.53
Mine Site Water Pumping	2,453,000	0.05	0.06
Total Operating Cost	**543,495,000**	**12.03**	**12.33**

Notes: *The estimates, excluding mining operating costs, are based on a mill feed rate of 130,000 t/d; the costs do not reflect higher unit costs late in the mine life when the mill feed rates are lower.
**Mining operating costs are LOM average unit costs calculated by total LOM operating costs divided by LOM process tonnages; mining operating costs exclude mine pre-production costs. The annual cost is the LOM average cost
Costs have been rounded to the nearest thousands of dollars.

Power will be supplied by BC Hydro at an average cost of US$0.050/kWh at the plant 25 kV bus bars, based on the BC Hydro credits for energy conservation by use of HPGR and similar, and the cost of "peaking" power to avoid a BC Hydro contract demand of over 150 MVA. 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 site services, water treatment plants, TMF seepage water pumping, and the Mine Site water pumping are included in their area costs separately.

The estimated electrical power costs are based on the 2016 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 not treated as an Input Tax Credit (ITC). 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 costlier Tier 2 power in the BC Hydro stepped-rate Schedule 1823. The 5% Goods and Services Tax (GST) is not included in the power rates as it is an ITC.

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

1.21 ECONOMIC EVALUATION

Tetra Tech prepared an economic evaluation for the 2016 PFS based on a pre-tax financial model. For the 53-year mine life and 2,198 Mt Mineral Reserve, the following pre-tax financial parameters were calculated using the base case metal prices:

- 10.4% IRR
- 6.0-year payback on US$5.005 billion capital
- US$3.263 billion NPV at a 5% discount rate.

The tax component of the financial model used for the post-tax economic evaluation was prepared and reviewed by other consultants (see Section 22.0 for further details).

Based on the tax analysis and review, the following post-tax financial results were calculated:

- 8.0% IRR
- 6.8-year payback on US$5.005 billion capital
- US$1.539 billion NPV at a 5% discount rate.

The base case results incorporate historical three-year trailing averages for metal prices as of July 31, 2016 as follows:

- gold – US$1,230/oz
- copper – US$2.75/lb
- silver – US$17.75/oz
- molybdenum – US$8.49/lb
- exchange rate – Cdn$1.00 to US$0.80.

Metal revenues projected in the KSM cash flow models were based on the average metal values indicated in Table 1.10.

Table 1.10 2016 PFS Metal Production from the Project

	Years 1 to 7	LOM
Total Tonnes to Mill (000s)	322,750	2,198,559
Annual Tonnes to Mill (000s)	46,107	41,484
Average Grades		
Gold (g/t)	0.82	0.55
Copper (%)	0.24	0.21
Silver (g/t)	2.8	2.6
Molybdenum (ppm)	48.3	42.6
Total Production		
Gold ('000 oz)	6,529	28,597
Copper ('000 lb)	1,434,560	8,270,423
Silver ('000 oz)	18,224	114,671
Molybdenum ('000 lb)	11,154	62,080
Average Annual Production		
Gold ('000 oz)	933	540
Copper ('000 lb)	204,937	156,052
Silver ('000 oz)	2,603	2,164
Molybdenum ('000 lb)	1,593	1,171

In addition to the base case, three metal price/exchange rate scenarios were also developed: the first uses the metal prices and exchange rate used in mine optimization and design (2016 Design Case); the second uses the spot metal prices and closing exchange rate on July 1, 2016 (Recent Spot Case); the third uses higher metal prices to indicate upside potential (Alternate Case). The input parameters and pre- and post-tax results of all scenarios can be found in Table 1.11.

Table 1.11 2016 PFS Summary of the Pre- and Post-tax Economic Evaluations

	Unit	Base Case	2016 Design Case	Recent Spot Case	Alternate Case
Metal Price					
Gold	US$/oz	1,230.00	1,200.00	1,350.00	1,500.00
Copper	US$/lb	2.75	2.70	2.20	3.00
Silver	US$/oz	17.75	17.50	20.00	25.00
Molybdenum	US$/lb	8.49	9.70	7.00	10.00
Exchange Rate	US:Cdn	0.80	0.83	0.77	0.80
Pre-tax Economic Results					
NPV (at 0%)	US$ million	15,933	13,727	16,101	26,319
NPV (at 3%)	US$ million	6,217	5,128	6,461	11,138
NPV (at 5%)	US$ million	3,263	2,510	3,507	6,541
NPV (at 8%)	US$ million	960	475	1,175	2,928
IRR	%	10.4	9.2	11.1	14.6
Payback	years	6.0	6.5	5.6	4.1
Cash Cost/oz Au	US$/oz	277	311	404	183
Total Cost/oz Au	US$/oz	673	720	787	580
Post-tax Economic Results					
NPV (at 0%)	US$ million	9,983	8,537	10,109	16,721
NPV (at 3%)	US$ million	3,513	2,789	3,691	6,696
NPV (at 5%)	US$ million	1,539	1,028	1,718	3,663
NPV (at 8%)	US$ million	-2	-343	161	1,282
IRR	%	8.0	7.0	8.5	11.4
Payback	years	6.8	7.4	6.4	4.9

1.21.1 SENSITIVITY ANALYSIS

Sensitivity analyses were carried out on the pre-tax base case for 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 pre-tax NPV, IRR, and payback period. The Project NPV is most sensitive to gold price and exchange rate followed by operating costs, copper price and capital costs. The IRR is most sensitive to exchange rate, capital costs and gold price followed by operating costs and copper price. The payback period is most sensitive to gold price and exchange rate followed by capital costs, copper price and operating costs. Since the majority of costs are in Canadian dollars and the economic analysis is developed in American dollars, a significant increase in the exchange rate by 30% will result in a significant increase in the costs when converted to American dollars and this leads to sharp increase in the payback period. Also, when gold price decreases by 30%, the revenue side decreases significantly and this results in sharp increase in the payback period. Financial outcomes are relatively insensitive to silver and molybdenum prices. The NPV, IRR, and payback sensitivities can be seen in Figure 1.14, Figure 1.15, and Figure 1.16.

Figure 1.14 2016 PFS Sensitivity Analysis of Pre-tax NPV at a 5% Discount Rate

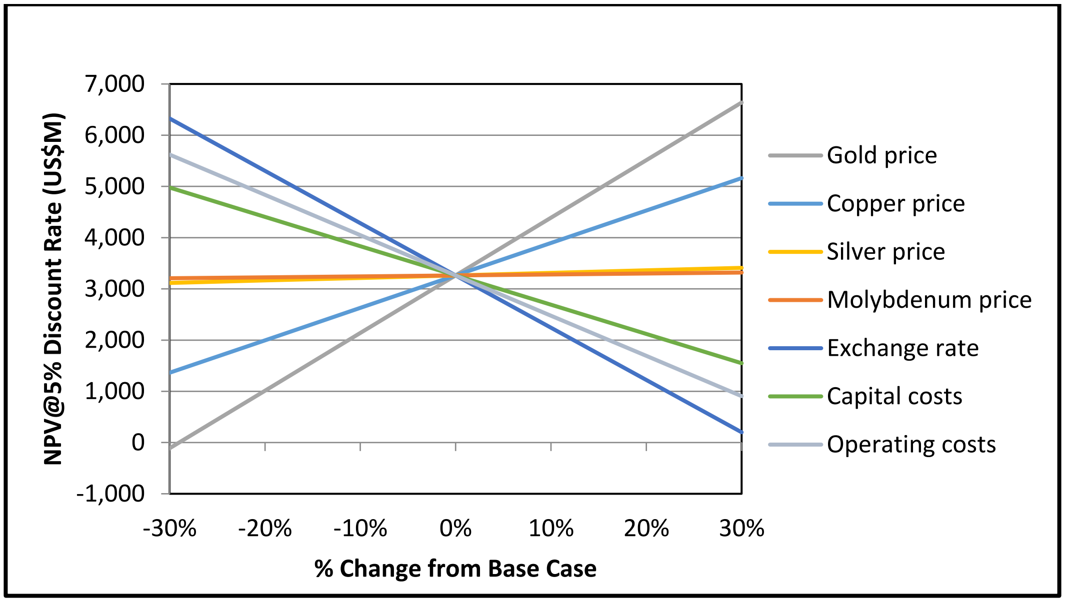


Figure 1.15 2016 PFS Sensitivity Analysis of Pre-tax IRR

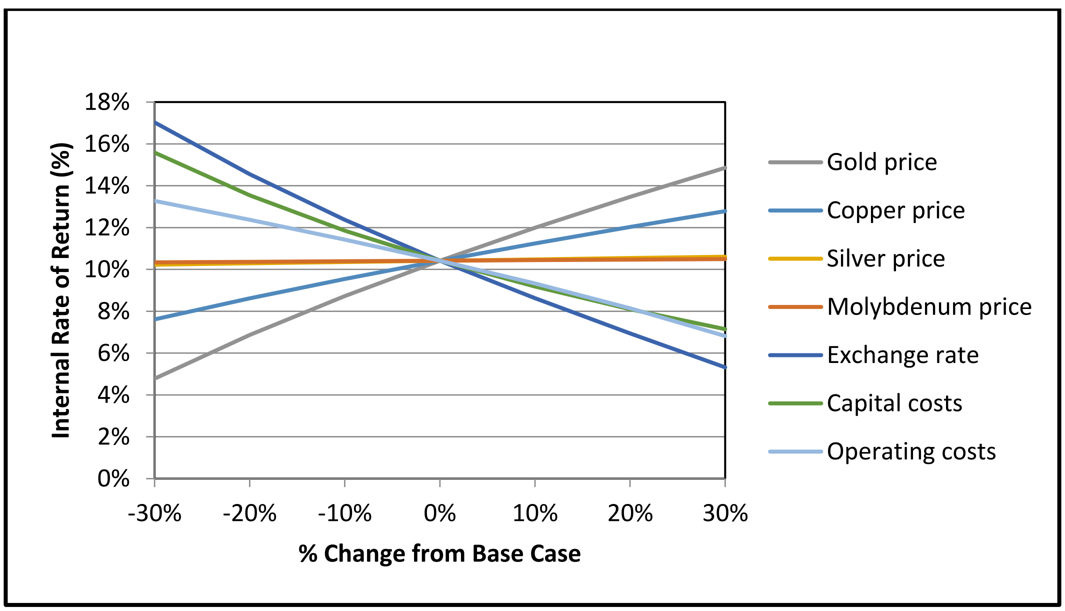
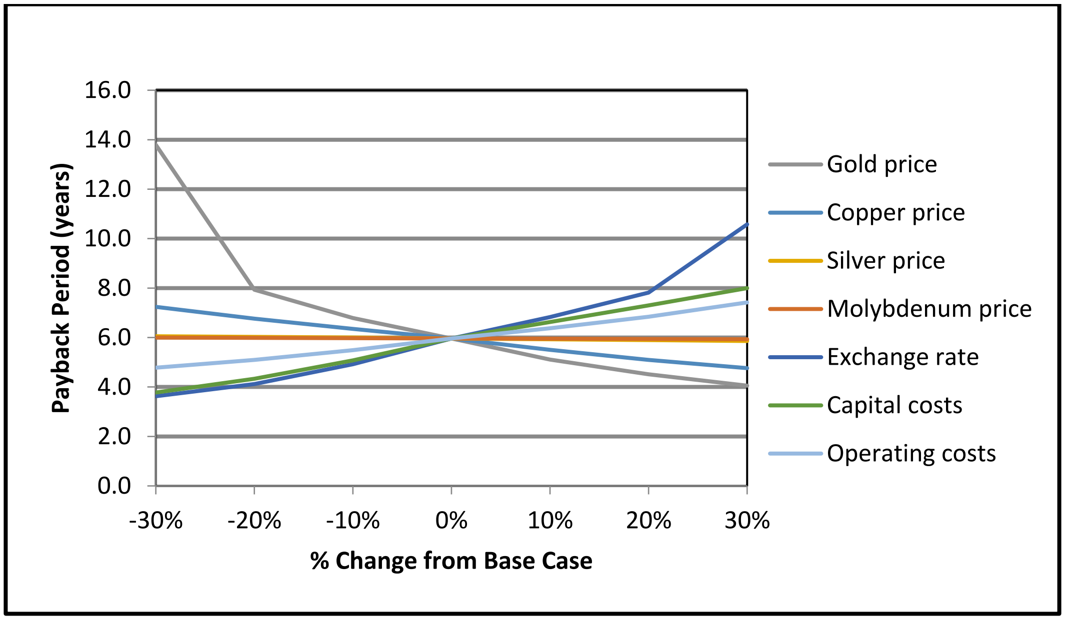


Figure 1.16 2016 PFS Sensitivity Analysis of Pre-tax Payback Period



1.22 PRELIMINARY ECONOMIC ASSESSMENT OF THE KSM PROJECT

1.22.1 INTRODUCTION

The PEA was undertaken to evaluate a different approach to developing the Project by emphasizing low-cost block cave mining and reducing the number and size of the open pits, which significantly reduces the surface disturbances in the re-designed Project. The PEA is based on the same Mineral Resource estimates that were used in the 2016 PFS, except the Inferred Mineral Resources are included in the PEA design, capital and operating cost estimates, and projected economics.

The PEA envisages a combined open pit/underground block cave mining operation that is planned to operate for 51 years. Over the entire 51-year mine life, mineralized material would be fed to a copper and gold extraction mill. The proposed plant for the PEA mine design will have an average process rate of 170,000 t/d. The Mitchell open pit and Deep Kerr underground mines will be the main source of mill feed, contributing approximately 83% of the total plant feed over the LOM, supplemented by the Sulphurets open pit and Iron Cap underground mine production.

The flotation plant would produce a gold/copper/silver concentrate for transport by truck to a nearby sea port at Stewart, BC for shipment to Pacific Rim smelters. Metallurgical testing indicates that KSM can produce a clean concentrate with an average copper grade of 25% with a high gold and silver content, making it readily saleable. Separate gold-silver doré would be produced at the KSM processing facility.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The results of the 2016 PFS remain valid and represent a viable option for developing the Project, with the PEA assessing an alternative development option at a conceptual level of design.

1.22.2 MINING METHODS

The PEA utilizes Measured, Indicated, and Inferred Mineral Resources in mine planning. Mill feed in the PEA is drawn from open pit mining and underground block cave mining of the Mitchell deposit, open pit mining of the Sulphurets deposit and underground block cave mining of the Kerr and Iron Cap deposits. Approximately 22% of the mill feed would come from open pit operations and 78% from underground block cave operations. Waste to mill feed cut-offs were determined using a NSR for each block in the model for the open pit mines, and a NSR shut-off for the block cave underground mines. NSR is calculated using prices and process recoveries for each metal accounting for all off-site losses, transportation, smelting and refining charges. Metal prices of US$1,200/oz of gold, US$2.70/lb of copper, and US$17.50/oz of silver are used in the NSR calculations.

LG pit shell optimizations were used to define open pit mine plans in the PEA. The pit limits of the PEA are contained inside the pit limits of the 2016 PFS. The mine design for the PEA focuses on reducing waste and selecting blocks with higher values. As a result, the PEA mine plan contains 2.4 Bt less waste in the open pit mine plan.

The underground block caving mine designs for Mitchell, Iron Cap, and Kerr are based on modeling using GEOVIA's PCBC™ Footprint Finder 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.

Mitchell is estimated to have a production ramp-up period of five years, steady state production at 21.9 Mt/a for 28 years, and then ramp-down production for another 3 years. Iron Cap is estimated to have a production ramp-up period of three years, steady state production at 14.6 Mt/a for 11 years, and then ramp-down production for another 4 years. Kerr is estimated to have a production ramp-up period of six years, steady state production at 25.5 Mt/a for 38 years with some variations during years where the operation transitions from first to second lift and second to third lift. Ramp down lasts 4 years.

The underground pre-production period is five years for Mitchell and Iron Cap and three years for Kerr. The first underground mill feed production from Mitchell, Iron Cap and Kerr comes in Years 9, 10, and 4, respectively. An elevated shut-off is used in the PEA underground mine designs compared to what was used in the 2016 PFS. In the PEA, the mining NSR shut-off is Cdn$20/t for the Mitchell underground mine, Cdn$23/t for the Iron Cap underground mine and Cdn$22/t for Kerr.

Mineral Resources contained in the mine plans for the PEA are shown in Table 1.12.

Table 1.12 Mineral Resources Included in the PEA Mine Plan

Zone	Mining Method	Classification	Tonnes (Mt)	Average Grades			Contained Metal		
				Au (g/t)	Cu (%)	Ag (g/t)	Au (Moz	Cu (Mlb)	Ag (Moz)
Mitchell	Open Pit	Measured	223.7	0.79	0.20	3.0	5.7	966	21.9
		Indicated	194.6	0.75	0.19	2.8	4.7	817	17.7
		Inferred	11.6	0.47	0.20	5.2	0.2	50	1.9
	Block Cave	Measured	244.9	0.68	0.21	4.2	5.4	1,134	33.1
		Indicated	361.0	0.65	0.20	4.1	7.5	1,592	47.6
		Inferred	87.5	0.40	0.13	3.1	1.1	259	8.7
Iron Cap	Block Cave	Indicated	121.5	0.64	0.24	4.1	2.5	643	15.8
		Inferred	77.4	0.46	0.22	3.5	1.1	384	8.7
Sulphurets	Open Pit	Indicated	91.8	0.70	0.29	0.6	2.1	584	1.7
		Inferred	11.1	0.59	0.25	0.8	0.2	60	0.3
Kerr	Block Cave	Indicated	24.4	0.26	0.54	1.1	0.2	290	0.8
		Inferred	931.5	0.31	0.49	1.7	9.3	9,962	52.0
Total Open Pit		Measured + Indicated	510.1	0.76	0.21	2.5	12.4	2,367	41.2
		Inferred	22.7	0.53	0.22	3.1	0.4	111	2.2
Total Block Cave		Measured + Indicated	751.8	0.64	0.22	4.0	15.6	3,659	97.3
		Inferred	1,096.4	0.33	0.44	2.0	11.6	10,605	69.3
Total Material Mined		Measured + Indicated	1,261.8	0.69	0.22	3.4	28.0	6,026	138.6
		Inferred	1,119.1	0.33	0.43	2.0	12.0	10,716	71.6

The PEA mining study took a different approach to the 2016 PFS. The PEA mine plan was carried out with the aim of reducing the amount of waste rock produced in the open pits with the mill feed drawing more on the underground resources. The mine production plan starts in lower-cost open pit areas using conventional large scale equipment before transitioning into block cave underground bulk mining later in the mine life. Starter pits have been selected in higher-grade areas and cut-off grade strategy optimizes revenues to minimize the payback duration. Smaller pits were designed for the Mitchell and Sulphurets deposits as well as mining the Kerr deposit solely by underground mining methods. This approach substantially shrinks the Project's footprint.

1.22.3 Open Pit Mining

For the Mitchell pit, the Mineral Resource model was subjected to an optimization analysis to define the mining limits by analysing a series of 46 nested pit shells. In the case of the Sulphurets pit, a much smaller pit than the optimal pit limit was chosen in order to constrain the amount of waste rock produced during the LOM.

The Project is designed as a conventional truck-shovel operation. The pit design for the Mitchell area includes three nested phases to balance stripping requirements while satisfying the Process Plant requirements. In the case of the Sulphurets area the material is mined in one phase. The Mineral Resources included in the Mitchell and Sulphurets PEA open pit mine plans are shown in Table 1.12.

The PEA production schedule results in an open pit LOM of eight years with stockpile reclaim extending into Year 14. The mine will require three years of pre-production before the start of the Process Plant operations. Five, 363 t haul trucks will be required in Year −3, increasing to 19 by Year −1, and peaking at 39 in production Years 1 to 3.

1.22.4 UNDERGROUND MINING – DEEP KERR, IRON CAP, AND MITCHELL

The underground block caving mine designs for Deep Kerr, Iron Cap, and Mitchell are based on modeling using GEOVIA's PCBC™ Footprint Finder 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.

Mitchell is estimated to have a production ramp-up period of five years, steady state production at 21.9 Mt/a for 28 years, and then ramp-down production for another 3 years. Iron Cap is estimated to have a production ramp-up period of three years, steady state production at 14.6 Mt/a for 11 years, and then ramp-down production for another 4 years. Deep Kerr is estimated to have a production ramp-up period of six years, steady state production at 25.5 Mt/a for 38 years with some variations during years where the operation transitions from first to second lift and second to third lift. Ramp down lasts four years. The underground pre-production period is five years for Mitchell and Iron Cap and three years for Deep Kerr.

The first underground mill feed production from Deep Kerr, Iron Cap, and Mitchell comes in Years 9, 10, and 4, respectively.

1.22.5 RECOVERY METHODS

The proposed KSM Process Plant will have an average process rate of 170,000 t/d. The Mitchell open pit and Deep Kerr underground mines will be the main source of mill feed, contributing about 83% of the total plant feed over the LOM, supplemented by the Sulphurets open pit along with the Mitchell and Iron Cap underground mine production.

The overall process flow diagram developed for the 2016 PFS has been carried through to the PEA, except for the molybdenum recovery circuit, which has been eliminated. For the purpose of this PEA, the process circuit will incorporate three-stage crushing, milling, conventional flotation and cyanidation processes for the recovery of copper, gold and silver. With the use of the same flowsheet in the PEA as with the PFS, it was assumed that the PFS testwork is representative and is used to support the PEA process design. Characterization and metallurgical testwork on Mitchell, Sulphurets, Iron Cap, and Deep Kerr samples is presented in Section 13.0. A small number of the minor element assays of the final bulk concentrates from Deep Kerr indicate that concentrations of arsenic, antimony and mercury may be near or above typical smelter penalty limits. The concentrate minor element data should be reviewed by a concentrate marketing specialist to identify potential concentrate marketing issues.

The major design consideration in the Process Plant equipment sizing and layout for the PEA was the use of the largest equipment sizing available in order to minimize pumping and piping requirements, process building footprint, and capital costs. Redesign of the facilities was limited to optimizing the layout provided by the use of the larger equipment in the PEA relative to the 2016 PFS.

Recovery equations for each ore type produced by Tetra Tech were reviewed and considered adequate for the purposes of the PEA by Amec Foster Wheeler. The metallurgical performance projections of the four KSM ore types are summarized in Section 13.2.

1.22.6 PRODUCTION

The mine production plan starts in lower-cost open pit areas using conventional large scale equipment before transitioning into block cave underground bulk mining later in the mine life. Starter pits have been selected in higher-grade areas and cut-off grade strategy optimizes revenues to minimize the payback duration.

After initial ramp-up the throughput averages of 170,000 t/d for the first 20 years, after the rate is reduced to 130,000 t/d for the following 15 years, and then is further reduced to around 77,000 t/d for 12 years; during the remaining 3 years of production, throughput averages 28,000 t/d. In the PEA, KSM's mine life is estimated at approximately 51 years. Production starts from open pits at Mitchell and Sulphurets and lasts until Years 8 and 5 of production, respectively. During that period the Kerr block cave is developed and first mill feed is produced in Year 4 of production. In Year 9 and 10 Mitchell and Iron Cap caves enter into production. Underground production ends first at Iron Cap in Year 27, then at Mitchell in Year 44, and finally at Kerr in Year 51 of production.

At Mitchell, a near-surface higher-grade gold zone outcrops allowing for gold production in the first seven years that is substantially above the mine life average grade. The mine plan is specifically designed for mining highest gold grade first to facilitate an early capital investment payback.

1.22.7 CAPITAL AND OPERATING COSTS

CAPITAL COSTS

Initial capital costs (including contingency of US$927 million and pre-production mining costs) are estimated at US$5.5 billion, approximately 9.7% higher than the initial capital estimate in the 2016 PFS. This increased initial capital is related to the higher throughput that required a bigger mining fleet at the start of production, larger size of equipment at the mill and changes in the tailing management facility due to a higher mill rate. Also, contingency is higher to reflect the lower level of cost accuracy of the PEA compared to the 2016 PFS.

Sustaining capital over the 51-year mine life is estimated at US$10.0 billion and is dominated by capitalizing the underground mine expansions at Kerr, Mitchell, and Iron Cap block caves. In addition to sustaining capital, a further US$540 million has been charged against the Project for closure and post-closure costs.

Initial capital and sustaining capital estimates for the PEA are summarized in Table 1.13.

Table 1.13 PEA Capital Cost Estimate

Area	Cost (US$ millions)
Direct Costs	
Mine Site	1,272
Process	1,447
Tailing Management Facility	509
Environmental	15
On-site Infrastructure	23
Off-site Infrastructure	120
Permanent Electrical Power Supply and Energy Recovery	167
Total Direct Costs	**3,553**
Indirect Costs	848
Owner's Cost	161
Contingency	927
Total Initial Capital	**5,489**
Total Life of Mine Sustaining Capital	**10,018**

OPERATING COSTS

Average mine, process, and G&A operating costs over the PEA calculated mine life (including waste mining and on-site power credits, excluding off-site shipping and smelting costs) are estimated at US$11.61/t milled (before base metal credits). Estimated unit operating costs are 6% lower than the 2016 PFS primarily due to reduction in process and G&A cost associated with higher throughput. A breakdown of estimated unit operating costs is shown in Table 1.14.

Table 1.14 PEA LOM Average Unit Operating Costs

Area	Cost (US$/t milled)
Mining*	4.47
Process	5.19
G&A	0.86
Others	1.09
Total Operating Costs	**11.61**

Note: *excluding pre-production cost of both open pit and underground mining

1.22.8 ECONOMIC ANALYSIS

The PEA has been evaluated using a discounted cash flow (DCF) analysis. Cash inflows consist of annual revenue projections for the mine. Cash outflows such as capital, including the six years of pre-production costs, operating costs, taxes, and royalties are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are taken to occur at the end of each period.

Under the assumptions presented in this report, the PEA demonstrates positive economics. The after-tax NPV at a 5% discount rate over the estimated mine life is US$ 3.366 billion. The after-tax IRR is 10%. Payback of the initial capital investment is estimated to occur in 6.4 years after the start of production or less than 12% of mine life. A payback period representing less than 20% of mine life is considered highly favorable.

The Project is most sensitive to changes in metal prices and foreign exchange, less sensitive to changes in capital costs, and least sensitive to operating cost and labour costs changes.

1.22.9 CONCLUSIONS

The PEA offers a viable option for development of the Project and reduces a number of the project risks. By including Deep Kerr, annual average maximum throughput of 130,000 t/d envisioned in the 2016 PFS could be increased to 170,000 t/d in the PEA without significant redesign of facilities. Increased throughput increases the metal production, reduces payback periods and improves estimated projected IRRs and NPVs. The PEA mine plans in total would reduce the amount of waste rock by 81% (approximately 2.4 Bt) compared to the PFS, substantially shrinking the Project's foot print and its environmental impact, and reducing water treatment costs.

1.23 RECOMMENDATIONS

Based on the work carried out in the 2016 PFS and the PEA, and the resultant economic evaluation, the 2016 PFS should be followed up by more advanced studies in order to further assess the technical and economic viability of the Project. Specific recommendations made by Qualified Persons (QPs) are detailed in Section 26.0 and are briefly summarized in the following subsections.

1.23.1 2016 PFS RECOMMENDATIONS

The key recommendations for advance studies emanating from the 2016 PFS focus on improving both open pit and underground mine design through additional drilling and testing; water related topics to further refine the inputs and results of site wide water balance analyses from the construction period through closure; and tunnels to develop more design-specific information to assist in reducing Project and operational risk and associated construction and operating costs. Other recommendations address data collection needs for RSFs, ore transportation, TMF, metallurgical testing, and process engineering.

1.23.2 PEA RECOMMENDATIONS

The exploration drilling program on Deep Kerr should be augmented with geotechnical activities to provide a better understanding of the rock structure characteristics important for the mine design. Amec Foster Wheeler also recommends that Seabridge perform further testwork to confirm process parameters and test domain composites and point samples to address geometallurgical variability that will support more advanced studies on the Deep Kerr deposit.

2.0 INTRODUCTION

2.1 PROJECT OVERVIEW

The Project is currently 100%-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 approximately 950 km northwest of Vancouver, 65 km north-northwest of Stewart, BC, and 21 km south-southeast of the Eskay Creek Mine.

The KSM 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.

2.2 TERMS OF REFERENCE

This report was prepared for Seabridge to summarize the results of the 2016 PFS, as well as a PEA of an alternative development option, performed at a conceptual level, of the Project. The report includes an updated Mineral Resource estimate on the Property. The QPs that authored the report are independent of Seabridge Gold Inc. and the Project.

2.2.1 2016 PREFEASIBILITY STUDY

The intent of the 2016 KSM PFS update 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 2016 KSM PFS update has been prepared for Seabridge based on work performed by the following independent consultants:

- Tetra Tech
- MMTS
- Golder
- McElhaney
- BGC
- RMI
- KCB

- Brazier
- ERM.

2.2.2 Preliminary Economic Assessment

The results of the PEA were disclosed in Seabridge's press release entitled "New Study Finds Significant Gains for Seabridge Gold's KSM Project", dated 6 October 2016. This Report will be filed in support of the disclosure of the PEA results.

2.3 Sources of Information

The key information sources for the 2016 PFS and PEA were:

- previously filed technical report titled 2012 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study effective date June 22, 2012, amended date November 11, 2014 (Tetra Tech 2012)
- documents referenced in Section 3.0 (Reliance on Other Experts)
- documents referenced in Section 27.0 (References) of this Report
- additional information provided by Seabridge personnel where required.

2.4 Effective Dates

The Report has a number of effective dates as follows:

- KSM Mineral Resource estimate: May 31st, 2016
- KSM Mineral Reserve estimate: July 31st, 2016
- Latest information on mineral tenure, surface rights and Project ownership: October 6th, 2016

The overall effective date of the 2016 PFS and PEA is October 6th, 2016.

2.5 Qualified Persons

The name of the QPs and a summary of the sections of the report that they take responsibility is provided in Table 2.1. QP certificates are included as the Signature Pages at the end of the report.

The following QPs conducted a site visit of the Property:

- Derek Kinakin (M.Sc., P.Geo., P.G.) of BGC visited the Property from August 19th to 21st, 2013.
- Hassan Ghaffari (P.Eng.) of Tetra Tech visited the Property on September 20th, 2014.

- James H. Gray (P.Eng.) of MMTS visited the Property on September 25th, 2008; September 10th, 2009; and April 13th, 2010.
- Michael J. Lechner (P.Geo., RPG, CPG) of RMI visited the Property from September 11th to September 14th, 2015.
- Jianhui (John) Huang (Ph.D., P.Eng.) of Tetra Tech visited the Property from on September 16, 2008.
- Neil Brazier (P.Eng.) of Brazier visited the Property from September 1 to 4, 2013, and from September 12 to 16, 2011.
- J. Graham Parkinson (P.Geo.) of KCB visited the Property from September 18 to 22, 2014, as well as during the summers of 2008 and 2009, and autumn of 2007.
- Pierre Pelletier (P.Eng.) of ERM visited the Property on April 13, 2010 and May 16, 2012.
- Ross D. Hammett (Ph.D., P.Eng.) of Golder visited the Property from August 8 to 10, 2010 and from October 18 and 19, 2011.
- R.W. Parolin (P.Eng.) of McElhanney visited the Property on June 21, 2008, and during the summers of 2009, 2010, and 2011.

Table 2.1 Summary of Qualified Persons

Section	Description	Qualified Person	
		Company	Qualified Person
1.0	Summary	Multiple	Sign off by Section
2.0	Introduction	Multiple	Sign off by Section
3.0	Reliance on Other Experts	Multiple	Sign off by Section
4.0	Property Description and Location	RMI	Michael J. Lechner, P.Geo., RPG, CPG
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	RMI	Michael J. Lechner, P.Geo., RPG, CPG
6.0	History	RMI	Michael J. Lechner, P.Geo., RPG, CPG
7.0	Geological Setting and Mineralization	RMI	Michael J. Lechner, P.Geo., RPG, CPG
8.0	Deposit Types	RMI	Michael J. Lechner, P.Geo., RPG, CPG
9.0	Exploration	RMI	Michael J. Lechner, P.Geo., RPG, CPG
10.0	Drilling	RMI	Michael J. Lechner, P.Geo., RPG, CPG
11.0	Sample Preparation, Analyses, and Security	RMI	Michael J. Lechner, P.Geo., RPG, CPG

table continues...

Section	Description	Qualified Person	
		Company	Qualified Person
12.0	Data Verification	RMI	Michael J. Lechner, P.Geo., RPG, CPG
13.0	Mineral Processing and Metallurgical Testing	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
14.0	Mineral Resource Estimate	RMI	Michael J. Lechner, P.Geo., RPG, CPG
15.0	Mineral Reserve Estimates	-	-
15.1	Introduction	MMTS/ Golder	James H. Gray, P.Eng./ Ross Hammett, Ph.D., P.Eng.
15.2	Pit Reserves	MMTS	James H. Gray, P.Eng.
15.3	Underground Reserves	Golder	Ross Hammett, Ph.D., P.Eng.
16.0	Mining Methods	-	-
16.1	Open Pit Mining Operations	MMTS	James H. Gray, P.Eng.
16.1.5	Pit Slope Design Angles	BGC	Derrek Kinakin, M.Sc., P.Geo., P.G.
16.2	Schedule Results	MMTS/ Golder	James H. Gray, P.Eng./ Ross Hammett, Ph.D., P.Eng.
16.3	Underground Mining	Golder	Ross Hammett, Ph.D., P.Eng.
17.0	Recovery Methods	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
18.0	Project Infrastructure	-	-
18.1	Site Layout	Tetra Tech	Hassan Ghaffari, P.Eng.
18.2	Tailing, Mine Rock and Water Management	KCB	J. Graham Parkinson, P.Geo.
18.3	MTT Transport System	MMTS	James H. Gray, P.Eng.
18.4	Infrastructure Tunnels	MMTS	James H. Gray, P.Eng.
18.5	Site Roads	Tetra Tech	Hassan Ghaffari, P.Eng.
18.6	Process Plant Facilities	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
18.7	Ancillary Buildings	Tetra Tech	Hassan Ghaffari, P.Eng.
18.8	Sewage	Tetra Tech	Hassan Ghaffari, P.Eng.
18.9	Communications System	Tetra Tech	Hassan Ghaffari, P.Eng.
18.10	Potable Water Supply	Tetra Tech	Hassan Ghaffari, P.Eng.
18.11	Power Supply and Distribution	Brazier	Neil Brazier, P.Eng.
18.12	Plant and Mitchell Site Electrical Distribution	Brazier	Neil Brazier, P.Eng.
18.13	Permanent and Construction Access Roads	McElhanney	R.W. Parolin, P.Eng.
18.14	Proposed Winter Access Road	Tetra Tech	Kevin Jones, P.Eng.
18.15	Logistics	Tetra Tech	Hassan Ghaffari, P.Eng.
18.16	Construction Execution Plan	Tetra Tech	Hassan Ghaffari, P.Eng.
18.17	Owner's Implementation Plan	Tetra Tech	Hassan Ghaffari, P.Eng.

table continues...

Section	Description	Qualified Person	
		Company	Qualified Person
19.0	Market Studies and Contracts	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
20.0	Environmental Studies, Permitting, and Social or Community Impact	ERM	Pierre Pelletier, P.Eng.
21.0	Capital and Operating Cost Estimates	-	-
21.1	Initial Capital Cost Estimate	Tetra Tech/MMTS/ Golder/Brazier	Sign-off by Section
21.2	Sustaining Capital Cost Estimate	Tetra Tech/MMTS/ Golder/Brazier	Sign-off by Section
21.3	Operating Cost Estimate	Tetra Tech/MMTS/ Golder	Sign-off by Section
22.0	Economic Analysis	Tetra Tech	Hassan Ghaffari, P.Eng.
23.0	Adjacent Properties	RMI	Michael J. Lechner, P.Geo., RPG, CPG
24.0	Other Relevant Data and Information	Amec Foster Wheeler	Simon Allard, P.Eng.
24.16	Mining Methods	Amec Foster Wheeler/Golder	Simon Allard, P.Eng., Mark Ramirez, RM SME, Ross Hammett, Ph.D., P.Eng.
24.16.1	Open Pit Mining Method	Amec Foster Wheeler	Simon Allard, P.Eng.
24.16.2	Deep Kerr Mining Method	Amec Foster Wheeler	Mark Ramirez, RM SME
24.16.3	Iron Cap Mining Method	Golder	Ross Hammett, Ph.D., P.Eng.
24.16.4	Mitchell Mining Method	Golder	Ross Hammett, Ph.D., P.Eng.
24.17	Recovery Methods	Amec Foster Wheeler	Tony Lipiec, P.Eng.
24.18	Project Infrastructure	Amec Foster Wheeler /Golder/KCB	Simon Allard, P.Eng., Mark Ramirez, RM SME, Ross Hammett, Ph.D., P.Eng. J. Graham Parkinson, P.Geo.
24.18.1	Onsite Infrastructure	Amec Foster Wheeler /KCB	Simon Allard, P.Eng. J. Graham Parkinson, P.Geo.
24.18.2	Offsite Infrastructure	Amec Foster Wheeler	Simon Allard, P.Eng.
24.18.3	Deep Kerr Project Infrastructure	Amec Foster Wheeler	Mark Ramirez, RM SME,
24.18.4	Iron Cap Project Infrastructure	Golder	Ross Hammett, Ph.D., P.Eng.
24.18.5	Mitchell Project Infrastructure	Golder	Ross Hammett, Ph.D., P.Eng.
24.19	Market Studies and Contracts	Tetra Tech	Jianhui (John) Huang, Ph.D., P.Eng.
24.20	Environmental Studies, Permitting, and Social or Community Impact	ERM	Pierre Pelletier, P.Eng.
24.21	Capital and Operating Costs	Amec Foster Wheeler	Simon Allard, P.Eng., Mark Ramirez, RM SME, Tony Lipiec, P.Eng.
24.22	Economic Analysis	Amec Foster Wheeler	Simon Allard, P.Eng.,
25.0	Interpretation and Conclusions	All	Sign off by Section
26.0	Recommendations	All	Sign off by Section

3.0 RELIANCE ON OTHER EXPERTS

3.1 PREFEASIBILITY STUDY

Mr. Hassan Ghaffari, P.Eng. relied on:

- Seabridge management concerning private royalties applicable to the Project and applied in Section 22.0.
- Lilburn & Associates LLC (Lilburn) concerning tax matters relevant to this PFS and detailed in Section 22.0. The reliance is based on a letter from Lilburn to Seabridge titled "Assistance with the tax portion of the economic analysis prepared by Tetra Tech WEI Inc. ("Tetra Tech") in connection with the NI 43-101 report of the 2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study (the "Report") on the KSM project (the "Project")" and dated September 13, 2016
- PricewaterhouseCoopers (PwC) concerning tax matters relevant to the PFS and detailed in Section 22.0. This reliance is based on a letter from PwC to Seabridge titled "Assistance with the review of income and mining tax portions of the economic analysis prepared by Seabridge Gold Inc. ("SGI") in connection with the NI 43-101 Technical Report (the "Report") on the Kerr-Sulphurets-Mitchell ("KSM") mining project (the "Project")" and dated September 14, 2016.

Dr. John Huang, Ph.D., P.Eng. relied on:

- Mr. Neil Seldon of Neil S. Seldon & Associates Ltd. (NSA) 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 and Section 24.19.

Mr. Michael Lechner, P.Geo, RPG, CPG relied on:

- Mr. John Brassard, Owner of TCG, for matters relating to mineral and placer claims status and ownership. TCG provided a Title Review of the KSM Property dated Augusts 16, 2016, signed by Mr. Brassard. 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, for matters relating to claims acquisition, royalties, and related agreements detailed in Section 4.0.

3.2 PRELIMINARY ECONOMIC ASSESSMENT

3.2.1 OWNERSHIP, MINERAL TENURE AND SURFACE RIGHTS

The QPs have not independently reviewed ownership of the Project area, including mineral and surface rights, and the underlying property agreements, including royalty agreements. The Amec Foster Wheeler QPs have fully relied upon, and disclaim responsibility for information provided by experts for this information through the following document:

- A letter prepared by Mr. Rudi Fronk, Chairman and CEO of Seabridge Gold, Inc. with the title "KSM Project – Surface Access and Property Payment Obligations and dated October 6th, 2016.
- A letter prepared by Mr. John L. Brassard, President of The Claim Group Inc., with the title "Seabridge Gold Inc., Title Review-KSM Property, Province of British Columbia" and dated October 6th, 2016.

This information is used in Section 24.22 of the Report.

3.2.2 TAXES

The Amec Foster Wheeler QPs have not independently reviewed the taxation information. The Amec Foster Wheeler QPs have fully relied upon, and disclaim responsibility for, taxation information derived from experts retained by Seabridge contained in the following document:

A letter authored by PricewaterhouseCoopers LLP ("PwC") with the title:

"NI 43-101 Technical Report Prepared for Seabridge Gold Inc. – Taxation Narrative" dated November 1, 2016.

PwC is a limited liability partnership, which is a member firm of PriceWaterhouseCoopers International Limited, each member firm of which is a separate legal entity.

This information is used in Section 24.22 of the Report.

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 North American Datum (NAD)83 Universal Transverse Mercator (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:

- the KSM claims
- the Seabee/Tina claims
- the KSM placer claims.

The three KSM claim blocks include 71 mineral claims (cell and legacy), 2 mining leases and 3 placer claims with a combined area of approximately 38,929 ha. The Claim Group Inc. (TCG) acts as Agent of behalf of Seabridge with respect to maintaining all pertinent records associated with the KSM claims. All claims and leases are in good standing under the *Mining Tenure Act* of BC, and are recorded as owned 100% by Seabridge.

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 being considered for proposed infrastructure siting.

The KSM 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 million. 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 three KSM placer claims are shown in Figure 4.3.

Annual holding costs for all leases and claims vary by year depending on whether the fees are paid in cash or whether the value of work completed on developing the claims is used in lieu of a cash payment. Over the next five years, the annual cash holding costs to keep the claims and leases valid range between Cdn$450,000 to Cdn$970,000. Those estimated costs can be reduced significantly if work expenditures are applied in lieu of cash fees.

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 new BJ Group had assessment work from 2011 applied to advance expiry dates to 2021. 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 letter titled "Title Review – KSM Property", by John Brassard, dated October 6, 2016.

Table 4.1 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. Table 4.2 shows Seabridge's placer claims.

Figure 4.1 General Location Map

Source: ERM

Figure 4.2 KSM Mineral Claim Map

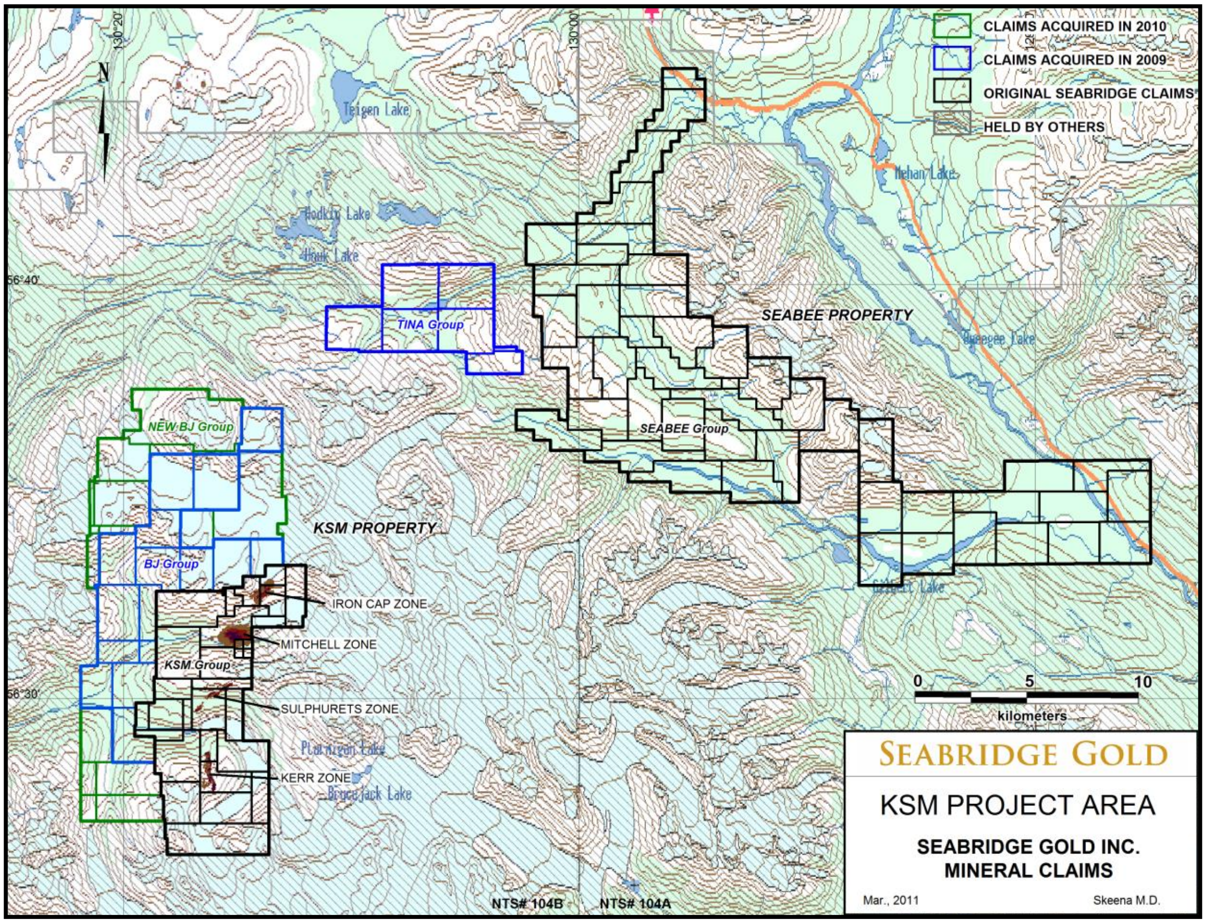


Figure 4.3 KSM Placer Claim Map

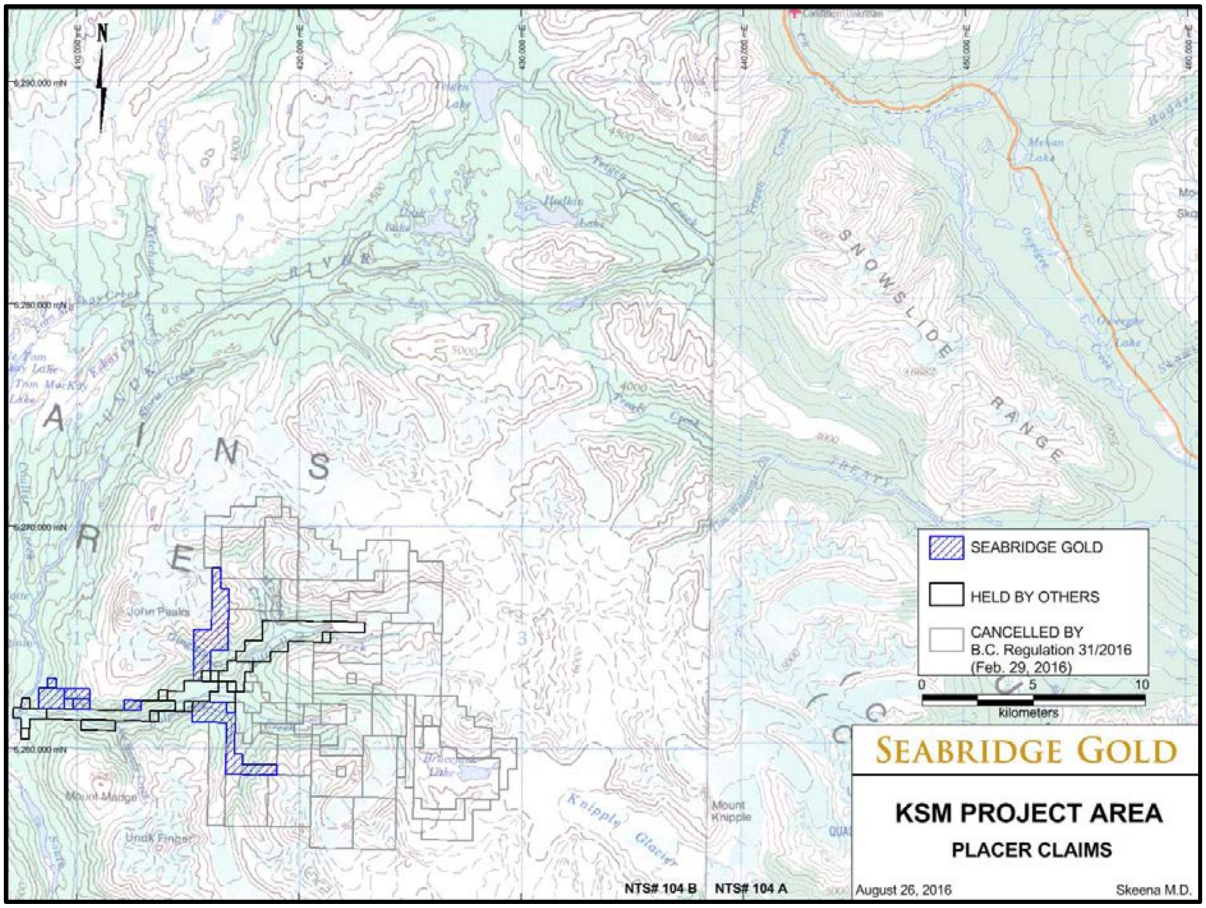


Table 4.1 KSM Mineral Claims and Leases

Claim No.	Claim Name	Area (ha)	Good To Date	Anniversary Date	Anniv. Year	Work Due (Cdn$/ha)	Annual Work Due (Cdn$)	Annual Cash-in-Lieu (Cdn$)
Claims (18)								
394780	BJ 5	100.0000	30-Nov-2021	30-Nov-2021	1	5.00	500.00	1,000.00
394781	BJ 6	100.0000	30-Nov-2021	30-Nov-2021	1	5.00	500.00	1,000.00
394786	BJ 11	500.0000	30-Nov-2021	30-Nov-2021	1	5.00	2,500.00	5,000.00
394787	BJ 12	500.0000	30-Nov-2021	30-Nov-2021	1	5.00	2,500.00	5,000.00
394788	BJ 13	100.0000	30-Nov-2021	30-Nov-2021	1	5.00	500.00	1,000.00
394789	BJ 13A	25.0000	30-Nov-2021	30-Nov-2021	1	5.00	125.00	250.00
394790	BJ 14	100.0000	30-Nov-2021	30-Nov-2021	1	5.00	500.00	1,000.00
394791	BJ 15	250.0000	30-Nov-2021	30-Nov-2021	1	5.00	1,250.00	2,500.00
394794	BJ 18	300.0000	30-Nov-2021	30-Nov-2021	1	5.00	1,500.00	3,000.00
394808	BJ 31A	375.0000	31-Dec-2021	31-Dec-2021	1	5.00	1,875.00	3,750.00
394809	BJ 32	150.0000	31-Dec-2021	31-Dec-2021	1	5.00	750.00	1,500.00
394810	BJ 33	450.0000	31-Dec-2021	31-Dec-2021	1	5.00	2,250.00	4,500.00
394811	BJ 34	150.0000	31-Dec-2021	31-Dec-2021	1	5.00	750.00	1,500.00
394812	BJ 35	450.0000	31-Dec-2021	31-Dec-2021	1	5.00	2,250.00	4,500.00
683463	-	1,246.5185	30-Nov-2021	30-Nov-2021	1	5.00	6,232.59	12,465.19
683483	-	837.5991	30-Nov-2021	30-Nov-2021	1	5.00	4,188.00	8,375.99
705591	BJ GAP1	231.6166	05-Feb-2021	05-Feb-2021	1	5.00	1,158.08	2,316.17
705592	BJ GAP2	160.4624	05-Feb-2021	05-Feb-2021	1	5.00	802.31	1,604.62
Totals	-	**6,026.1966**	-	-	-	-	**30,130.98**	**60,261.97**
Mineral Leases (2)								
1031440	-	6,085	06-Oct-2016	-	-	20.00	121,700.00	-
1031441	-	5,162	06-Oct-2016	-	-	20.00	103,240.00	-
Totals	-	**11,247**	-	-	-	-	**224,940.00**	-

Table 4.2 KSM Placer Claims

Claim No.	Claim Name	Area (ha)	Good to Date	Next Annual Work Due ($/ha)	Annual Work Due (Cdn$)	Annual Cash-in-Lieu (Cdn$)	No. of Cells	Map Number
Placer Claims (3)								
986922	PL21	35.7243	16-May-2017	20.00	714.49	1,428.97	2	104B
986924	PL22	35.7244	16-May-2017	20.00	714.49	1,428.98	2	104B
986925	PL23	107.1670	16-May-2017	20.00	2,143.34	4,286.68	6	104B
					3,572.32	**7,144.63**	-	
Totals	-	**178.6200**	-	-	**3,572.32**	**7,144.63**	-	

Table 4.3 Seabee/Tina KSM Claims

Claim No.	Claim Name	Area (ha)	Good To Date	Anniv. Date	Anniv. Year	Work Due (Cdn$/ha)	Annual Work Due (Cdn$)	Annual Cash-in-Lieu Due (Cdn$)	No. of Cells	Map Number
Seabee Property Mineral Claims (46)										
566467	BRIDGE1	445.8258	08-Feb-2017	08-Feb-2017	1	5.00	2,229.13	4,458.26	25	104A
566468	BRIDGE2	445.5733	08-Feb-2017	08-Feb-2017	1	5.00	2,227.87	4,455.73	25	104A
566469	BRIDGE3	427.7919	08-Feb-2017	08-Feb-2017	1	5.00	2,138.96	4,277.92	24	104A
566470	BRIDGE4	427.9770	08-Feb-2017	08-Feb-2017	1	5.00	2,139.89	4,279.77	24	104A
566471	BRIDGE5	445.7336	08-Feb-2017	08-Feb-2017	1	5.00	2,228.67	4,457.34	25	104A
566472	BRIDGE6	445.5766	08-Feb-2017	08-Feb-2017	1	5.00	2,227.88	4,455.77	25	104A
566473	BRIDGE7	427.9217	08-Feb-2017	08-Feb-2017	1	5.00	2,139.61	4,279.22	24	104A
566474	BRIDGE8	427.7599	08-Feb-2017	08-Feb-2017	1	5.00	2,138.80	4,277.60	24	104A
566475	BRIDGE9	427.6131	08-Feb-2017	08-Feb-2017	1	5.00	2,138.07	4,276.13	24	104A
566476	BRIDGE10	445.5312	08-Feb-2017	08-Feb-2017	1	5.00	2,227.66	4,455.31	25	104A
566477	BRIDGE11	302.8823	08-Feb-2017	08-Feb-2017	1	5.00	1,514.41	3,028.82	17	104A
566478	BRIDGE12	427.4311	08-Feb-2017	08-Feb-2017	1	5.00	2,137.16	4,274.31	24	104A
566479	BRIDGE13	445.1533	08-Feb-2017	08-Feb-2017	1	5.00	2,225.77	4,451.53	25	104A
566481	BRIDGE14	445.0611	08-Feb-2017	08-Feb-2017	1	5.00	2,225.31	4,450.61	25	104A
566482	BRIDGE15	444.8427	08-Feb-2017	08-Feb-2017	1	5.00	2,224.21	4,448.43	25	104A
566484	BRIDGE16	444.5621	08-Feb-2017	08-Feb-2017	1	5.00	2,222.81	4,445.62	25	104A
566485	BRIDGE17	426.7283	08-Feb-2017	08-Feb-2017	1	5.00	2,133.64	4,267.28	24	104A
566487	BRIDGE18	444.7114	08-Feb-2017	08-Feb-2017	1	5.00	2,223.56	4,447.11	25	104A
566488	BRIDGE19	444.8346	08-Feb-2017	08-Feb-2017	1	5.00	2,224.17	4,448.35	25	104A
566489	BRIDGE20	444.9690	08-Feb-2017	08-Feb-2017	1	5.00	2,224.85	4,449.69	25	104A
566490	BRIDGE21	427.2642	08-Feb-2017	08-Feb-2017	1	5.00	2,136.32	4,272.64	24	104A
566491	BRIDGE22	445.1671	08-Feb-2017	08-Feb-2017	1	5.00	2,225.84	4,451.67	25	104A
566492	BRIDGE23	427.3078	08-Feb-2017	08-Feb-2017	1	5.00	2,136.54	4,273.08	24	104A

table continues…

Claim No.	Claim Name	Area (ha)	Good To Date	Anniv. Date	Anniv. Year	Work Due (Cdn$/ha)	Annual Work Due (Cdn$)	Annual Cash-in-Lieu Due (Cdn$)	No. of Cells	Map Number
566493	BRIDGE24	427.9239	08-Feb-2017	08-Feb-2017	1	5.00	2,139.62	4,279.24	24	104A
566494	BRIDGE25	427.9246	08-Feb-2017	08-Feb-2017	1	5.00	2,139.62	4,279.25	24	104A
566495	BRIDGE26	444.8785	08-Feb-2017	08-Feb-2017	1	5.00	2,224.39	4,448.79	25	104A
566496	BRIDGE27	391.3145	08-Feb-2017	08-Feb-2017	1	5.00	1,956.57	3,913.15	22	104B
566497	BRIDGE28	444.4573	08-Feb-2017	08-Feb-2017	1	5.00	2,222.29	4,444.57	25	104A
566567	BRIDGE29	427.4572	08-Feb-2017	08-Feb-2017	1	5.00	2,137.29	4,274.57	24	104A
571582	SEABEE1	408.8286	08-Feb-2017	08-Feb-2017	1	5.00	2,044.14	4,088.29	23	104A
571583	SEABEE2	373.1368	08-Feb-2017	08-Feb-2017	1	5.00	1,865.68	3,731.37	21	104A
571584	SEABEE3	444.0680	08-Feb-2017	08-Feb-2017	1	5.00	2,220.34	4,440.68	25	104A
571585	SEABEE4	426.0832	08-Feb-2017	08-Feb-2017	1	5.00	2,130.42	4,260.83	24	104A
571586	SEABEE5	372.6392	08-Feb-2017	08-Feb-2017	1	5.00	1,863.20	3,726.39	21	104A
571587	SEABEE6	159.6419	08-Feb-2017	08-Feb-2017	1	5.00	798.21	1,596.42	9	104A
573813	SEABEE7	213.2634	08-Feb-2017	08-Feb-2017	1	5.00	1,066.32	2,132.63	12	104A
575633	SEA 1	445.1987	08-Feb-2017	08-Feb-2017	1	5.00	2,225.99	4,451.99	25	104A
575635	SEA 2	445.3012	08-Feb-2017	08-Feb-2017	1	5.00	2,226.51	4,453.01	25	104A
575636	SEA 3	445.4096	08-Feb-2017	08-Feb-2017	1	5.00	2,227.05	4,454.10	25	104A
575638	SEA 4	445.4484	08-Feb-2017	08-Feb-2017	1	5.00	2,227.24	4,454.48	25	104A
575639	SEA 5	445.3365	08-Feb-2017	08-Feb-2017	1	5.00	2,226.68	4,453.37	25	104A
575642	SEA 6	445.0850	08-Feb-2017	08-Feb-2017	1	5.00	2,225.43	4,450.85	25	104A
575643	SEA 7	213.4398	08-Feb-2017	08-Feb-2017	1	5.00	1,067.20	2,134.40	12	104A
575645	SEA 8	427.0822	08-Feb-2017	08-Feb-2017	1	5.00	2,135.41	4,270.82	24	104A
575646	SEA 9	35.5980	08-Feb-2017	08-Feb-2017	1	5.00	177.99	355.98	2	104A
603133	SEABEE 8	426.5614	08-Feb-2017	08-Feb-2017	1	5.00	2,132.81	4,265.61	24	104B
Totals	-	**18,674.2970**	-	-	-	-	**93,371.49**	**186,742.97**	-	-

table continues...

Claim No.	Claim Name	Area (ha)	Good To Date	Anniv. Date	Anniv. Year	Work Due (Cdn$/ha)	Annual Work Due (Cdn$)	Annual Cash-in-Lieu Due (Cdn$)	No. of Cells	Map Number
Tina Property Mineral Claims (7)										
401548	TINA 1	500.0000	28-Feb-2018	28-Feb-2018	1	5.00	2,500.00	5,000.00	-	104B070
401549	TINA 2	500.0000	28-Feb-2018	28-Feb-2018	1	5.00	2,500.00	5,000.00	-	104B070
401550	TINA 3	500.0000	28-Feb-2018	28-Feb-2018	1	5.00	2,500.00	5,000.00	-	104B070
401551	TINA 4	500.0000	28-Feb-2018	28-Feb-2018	1	5.00	2,500.00	5,000.00	-	104B070
401552	TINA 5	500.0000	28-Feb-2018	28-Feb-2018	1	5.00	2,500.00	5,000.00	-	104B070
401553	TINA 6	250.0000	28-Feb-2018	28-Feb-2018	1	5.00	1,250.00	2,500.00	-	104B070
603134	SEABEE 9	53.3796	28-Feb-2018	28-Feb-2018	1	5.00	266.90	533.80	-	104B
Totals	-	**2,803.3796**	-	-	-	-	**14,016.90**	**28,033.80**	-	-

5.0 ACCESSIBILITY, CLIMATE, INFRASTRUCTURE, LOCAL RESOURCES 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), remains largely unchanged, and has only been updated for consistency in abbreviations and grammar.

The Property lies in the rugged Coastal Mountains of northwestern 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 the tree line and upwards. The glaciers have been retreating for at least the last several decades. Aerial photos indicate the Mitchell Glacier has retreated more than one 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. 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, 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. Exploration activities have typically been carried out from late May into November. It is envisioned that operations would be conducted throughout the year with assets required for snow removal.

Access to the Property is via helicopter. Three 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. This area has not been utilized since 2011.

3. The Bell II Lodge, on Highway 37, 40 km east-northeast.

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, Kitimat, and Smithers, with a total population of about 36,000, are located approximately 270 km to the southeast.

Deep-water loading facilities for shipping bulk mineral concentrates exist in Stewart, and are currently utilized by the Red Chris Mine. Historically they have been used by several other mines in northern, BC. 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; therefore 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 37, 40 km east of the Project, and along the 37A corridor to Stewart, approximately 50 km southeast.

Pretium is currently developing their high-grade underground Brucejack gold project, adjacent to the east side of the Property. The Brucejack project is scheduled to come on stream in 2017 with access to the Brucejack mine site via a road branching from Highway 37, and includes several kilometers on glacial ice.

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, 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. Several short hole, reconnaissance level drilling programs were undertaken between 1969 and 1991. The Sulphurets Zone was first drilled by Esso Minerals in 1969, Kerr was first drilled by Brinco Ltd. in 1985, Mitchell Creek by Newhawk in 1991, and Iron Cap by Esso Minerals in 1980.

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. The Sulphurets Property also hosts the Mitchell Creek deposit and other mineral occurrences. In 2000, Seabridge acquired a 100% interest from Placer Dome in both the Kerr and Sulphurets properties, subject to capped royalties.

There is no recorded mineral production, nor evidence of it, from the Property. Immediately west of the Property, small-scale placer gold mining has occurred in the Sulphurets and Mitchell creeks. On the Brucejack Property immediately to the east and currently owned by 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 Project.

Table 6.1 Exploration Summary of the Kerr and Deep Kerr Zones

Year	Activity
1982-1983	"Alpha Joint Venture (JV)" began prospecting and soil geochemical surveys of the Kerr gossan focusing on gold.
1984-1985	Brinco optioned the Kerr project, completed some geological 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 50 m 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 1,159 m; conducted metallurgical testing and permit work.
2010	Seabridge drilled 4 holes totalling 1,453 m; conducted metallurgical testing and permit work.
2011	Seabridge drilled 4 resource definition holes totalling 2,338 m, completed magneto-telluric geophysical surveys
2012	Seabridge drilled 5 exploration holes totalling 3,730.6 m
2013	Seabridge drilled 29 resource definition holes totalling 23,822.4 m, completed induced polarization and down-hole geophysical surveys
2014	Seabridge drilled 16 resource definition holes totalling 15,909.6 m, completed magneto-telluric and gravity geophysical surveys
2015	Seabridge drilled 5 resource definition holes totalling 6,437.2 m, completed airborne magnetic geophysical survey

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

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

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/geochemical 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.
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. 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.
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.

table continues...

Year	Activity
2008	Seabridge drilled 37 holes totalling 17,192 m, started metallurgical testing, obtained new topographic data, and initiated permit related activities.
2009	Seabridge drilled 11,843.8 m (resource definition, geotechnical and water monitoring), conducted metallurgical testing, and intensified permit data collection.
2010	Seabridge drilled 86 holes totalling 26,755.7 m (resource definition and geotechnical), conducted metallurgical testing, and intensified permit data collection.
2011	Seabridge drilled 54 resource definition holes totalling 18,087.2 m, continued prefeasibility-level work, completed magneto-telluric geophysical surveys.
2012	Seabridge drilled 25 holes totalling 15,452.85 m.
2013	Seabridge drilled 11 holes totalling 8,857.4 m, completed induced polarization and downhole geophysical surveys.
2014	Seabridge drilled 13 holes totalling 13,601.25 m, completed magneto-telluric and gravity geophysical surveys.
2015	Seabridge drilled 8 holes totalling 4,581.1 m, completed airborne magnetic geophysical survey.

6.2 HISTORICAL RESOURCE ESTIMATES

RMI is unaware of any publicly disclosed historical Mineral 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 2008a; Lechner, 2008b; Lechner, 2009; Lechner, 2010; Lechner, 2011; Lechner, 2014). Those Mineral Resources have been replaced and updated with this Technical Report as discussed in Section 14.0.

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

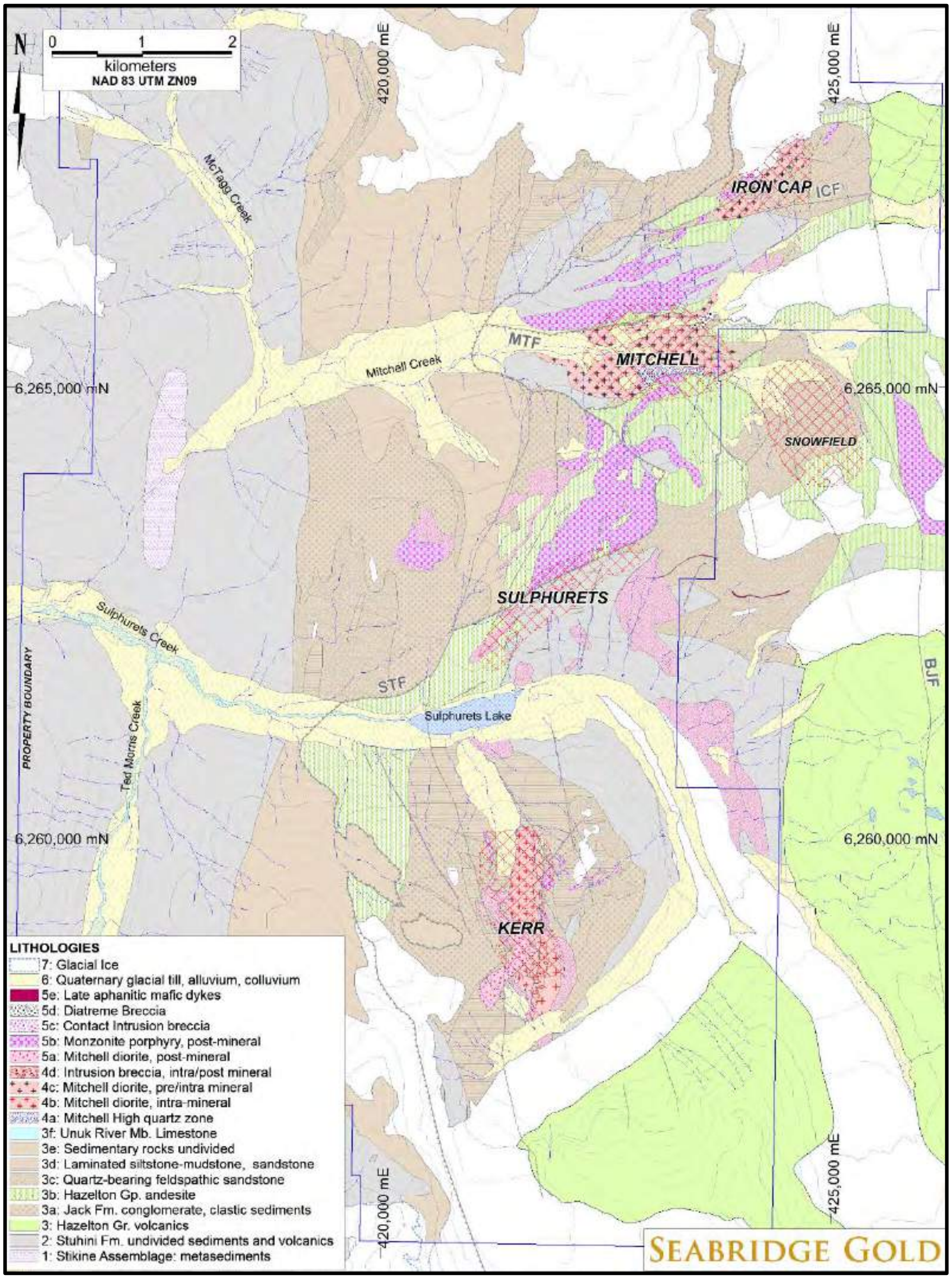
The region lies within "Stikinia", a terrane of Triassic and Jurassic volcanic arcs that were accreted onto the Paleozoic basement of the North American continental margin in the Middle Jurassic. Stikinia is the largest of several fault bounded, allochthonous terranes within the Intermontane belt, which 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 KSM 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 sinistral transpression tectonics in the mid-Cretaceous followed by extensional conditions generated the area's current structural features. The most important structure is the north-northeast striking, moderately west-northwest dipping STF, which transects the Property and is spatially and genetically related to mineralization at KSM. 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 zones such as Galore Creek, Red Chris, Kemess, Mt. Milligan, and Kerr-Sulphurets. In addition, there are a number of related polymetallic zones including skarns at Premier, epithermal veins and subaqueous vein and replacement sulphide zones at Eskay Creek, Snip, Brucejack, 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 zoned 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 Zone.

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 STF and MTF, this suite of intrusions include mineralized and non-mineralized stocks. The mineralized stocks include the Kerr, Sulphurets, Mitchell and Iron Cap zones, all positioned below and within a few hundred meters of its current track. There are a number of sills and plugs of coarse-grained feldspar porphyritic monzonite to low-silica granite that intruded mineralized porphyries and siliceous hornfelsed sediments and volcanics. These are more abundant above the STF, and are similar to "Premier" type porphyritic intrusions common in the Stewart district. Figure 7.1 is a generalized geologic map of the KSM district showing lithology, major structures, and mineralized zones. Drill hole locations are shown for all KSM deposits in Section 10.0 (Figure 10.1 to Figure 10.5).

Figure 7.1 Geology of the KSM District



7.2 MINERALIZATION

7.2.1 KERR ZONE

The Kerr Zone is centered on a north-south trending, steep westerly dipping, tabular intrusive complex that drilling demonstrates has a horizontal extent of 2,400 m and a vertical extent of at least 2,200 m. The complex includes an east and west limb that may coalesce near the current surface. The west limb is up to 500 m thick, and the east limb is up to 300 m thick. There are several distinct intrusive phases, the earliest of which are fine grained diorites with 5% to 60% quartz-sulphide vein stockworks, and these appear to contribute the majority of metals. Later phases envelope and sometimes invade the earlier phase, and are characterized by coarser textures, less veining, and lower metal contents. The intrusions are hosted by an Early Jurassic sequence of rhythmically bedded siltstones, sandstones, conglomerates, and debris flows that have been altered adjacent to the intrusions, but generally contain marginal metal grades.

At Kerr, dating of the intrusions indicates an age of approximately 190 million years, much older than most known porphyry systems. Subsequent tectonic events have modified the original geologic positioning so that mineral assemblages characteristic of deeper parts of a porphyry system are now at higher levels. For example, at the southern end of Kerr, potassic altered diorite with magnetite veins and high gold-copper grades, typical of the roots of porphyry systems, are found near the surface at an elevation of about 1,600 m. At the northern end, phyllic altered diorite cut by veins with advanced argillic assemblages, which are characteristic of much shallower depths, have been intersected at elevations less than 0 m. The modified geometry of the Kerr porphyry system has significant modeling and exploration implications, as higher grade deep core zones may occur at exploitable depths.

Various lithology and alteration types were modeled as 3D wireframes for the Kerr deposit and are summarized in Table 7.1. Mineralization occurs mainly in the PAND1 intrusion and IBX breccia and wall rock complex; however it may extend into the adjacent late PAND2 intrusion and wall rock sediments. The dominant copper mineral is chalcopyrite, which typically occurs as isolated grains about 0.2 to 2 mm across, disseminated and clustered in quartz veins, fractures, and surrounding haloes. Bornite is present almost exclusively in the north half of the east leg, within a phase containing over 50% crackled quartz veins, accompanied by coarse grained chalcopyrite and minor tennantite. Tennantite is rare, but widely distributed in late quartz-carbonate veins, mostly in wall rocks, along with minor sphalerite, rare galena, and arsenopyrite. Dark, arseniferous pyrite is associated with these minerals. Molybdenum is a minor constituent, mostly contained within the PAND1 and IBX units and closely associated with copper. This is distinct from the Mitchell zone, where molybdenum is distributed mainly in a shell in wall rocks peripheral to the copper zone. Visible gold has not been observed except under microscopic examinations, where it is observed as less than 100 µm inclusions within sulphides, mainly chalcopyrite, and sulphide grain boundaries.

Table 7.1 Kerr Zone Wireframe Models

Type	Code	Description
Lithology	OVBN	overburden; includes Kerr slide material
Lithology	PMFP	late porphyritic dykes, non-mineralized, with distinct centimetre-scale, white potassium-spar phenocrysts
Lithology	IBX	mixed complex of altered and sheared host rocks, dykes, and breccias usually altered and mineralized
Lithology	PAND1	feldspar and hornblende porphyritic diorite, part of the Mitchell Intrusive Suite, early mineralizing phase; at Kerr, stockwork veining is usually weak, except for relatively narrow zones of high stockworked, early phases with above average metal grades
Lithology	PAND2	similar to PAND1, but a later intrusive phase, generally with only marginal metal grades
Lithology	HAZELTON_FW	Hazelton Group layered sedimentary rocks, dominantly grey-green marine siltstones, sandstones, conglomerates, debris flows, volcanic derived, also black and dark grey, carbonaceous argillite and siltstone; when strongly bleached and altered, they are often foliated and logged as tuffs; includes scattered intervals of dioritic dykes, geometry uncertain
Lithology	PMFP_HW	same as PMFP, but in panel above F2 fault
Lithology	HAZELTON_HW	same as HAZELTON_FW, but in panel above F2 fault
Alteration	IARG_HG	dominantly phyllic alteration—intense, pervasive quartz-sericite-clay-chlorite-pyrite replacement of most minerals, generally less competent, foliated to schistose; pyrite 5% to 15%, disseminated, fine veinlets, rare sub-meter veins; variable but minor fine disseminated chalcopyrite, tennantite (tn)-tetrahedrite (tt), enstatite, sphalerite; may have associated anomalous mercury, in part ore grade copper and gold values; generally this is an outer phyllic zone that effects mostly wall rocks
Alteration	IARG	similar to IARG_HG, but less intense, more competent, less sulphide, remnant chlorite, generally anomalous or marginal metal grades
Alteration	QSP	dominantly phyllic alteration—intense, pervasive quartz-sericite-clay-pyrite replacement of most minerals, similar to and transitional with IARG; pyrite 5% to 15%, pyrite>chalcopyrite, typically ore grade copper and gold values; rare tn-tt, enstatite, bornite associated with some structures
Alteration	CL	dominantly potassic alteration—quartz-chlorite-pyrite, pervasive and vein controlled, almost all secondary potassium-spar and biotite has been replaced by chlorite; competent rock, generally effects intrusive rocks, usually mineralized; pyrite 1% to 5%, pyrite and chalcopyrite abundances usually similar, generally ore grade copper and gold values in PAND1 unit
Alteration	PR	dominantly propylitic alteration—generally weak, fracture and vein controlled quartz-pyrite-epidote-carbonate-hematite mineralization in wall rocks; late, millimetre to centimetre scale, tension gash and dilational space filling carbonate veins and lenses are very common; peripheral to Kerr Zone, includes porphyry-related "hornfels" alteration
Structure	F2	normal fault, displacement on order of 10s of meters, post-mineral and post-thrusting

Photographs of polished drill core samples representative of rock types from the Kerr deposit are shown in Figure 7.2. The width of the core samples (vertical axis) in the photographs shown in is approximately 2.5 cm. Representative lithologic and alteration cross sections through the Kerr deposit are shown in Figure 7.3 and Figure 7.4, respectively. Lithologic and alteration level plan maps are shown in Figure 7.5 and Figure 7.6, respectively.

Figure 7.2 Kerr Polished Drill Core Photographs

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
K-15-49	1476.5	0.43	0.84	0.0006	3	4	2	49

Kerr – PAND1 diorite intrusion: early, fine grained, potassic altered diorite with mostly intact stockwork qtz-cp-py veins, coarse cp and high cp/py ratio

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
K-15-49	1616.9	0.85	1.16	0.0026	8	21	51	44

Kerr – PAND1 diorite intrusion: intense phyllic overprint, all mafics destroyed, with abundant py, high grade due to addition of Au and Cu by overprint

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
K-15-50	1538.9	0.17	0.22	0.0019	1.1	9	6	56

Kerr – PAND2 diorite intrusion: later, slightly coarser grained, potassic altered diorite with no stockwork qtz-cp-py veins, a few orphaned vein fragments assimilated from PAND1

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
K-15-49	1706.3	0.22	0.58	0.0069	2	16	103	682

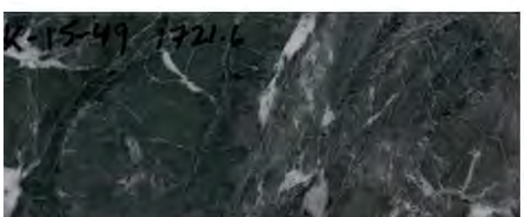
Kerr – PAND2 diorite intrusion: later, slightly coarser grained, potassic altered diorite with no stockwork qtz-cp-py veins, phyllic overprint with Cu, Pb, Zn

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
K-15-49	1630.4	4.06	0.43	0.0001	37	46	2910	5910

Kerr – late qtz-carb-sulfide vein in structure penetrated by late AA (advanced argillic) fluids, rich in Au, Ag, As, Pb, Zn, Fe, +/- Hg, sub-meter scale, occur in intrusions and wall rocks

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
K-15-49	1317.8	0.33	0.50	0.0006	1	46	5	20

Kerr – IBX intrusion breccia: phyllic altered with abundant py, disseminated cp, veined clasts of sediment, py veinlets terminated at clast



HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
K-15-49	1721.5	0.18	0.42	0.0033	1	5	4	46

Kerr – Sediments: chloritic-hornfels altered, brecciated, wall rocks, mostly fracture controlled and disseminated low-grade mineralization

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
K-12-20	587.7	0.05	0.26	0.0010	0.8	35	76	396

Kerr – Sediments: intensely phyllic altered, abundant fine disseminated and veinlet py, this interval occurs within the IBX domain, ~ 100m from intrusion

Figure 7.3 Kerr Lithologic Cross Section

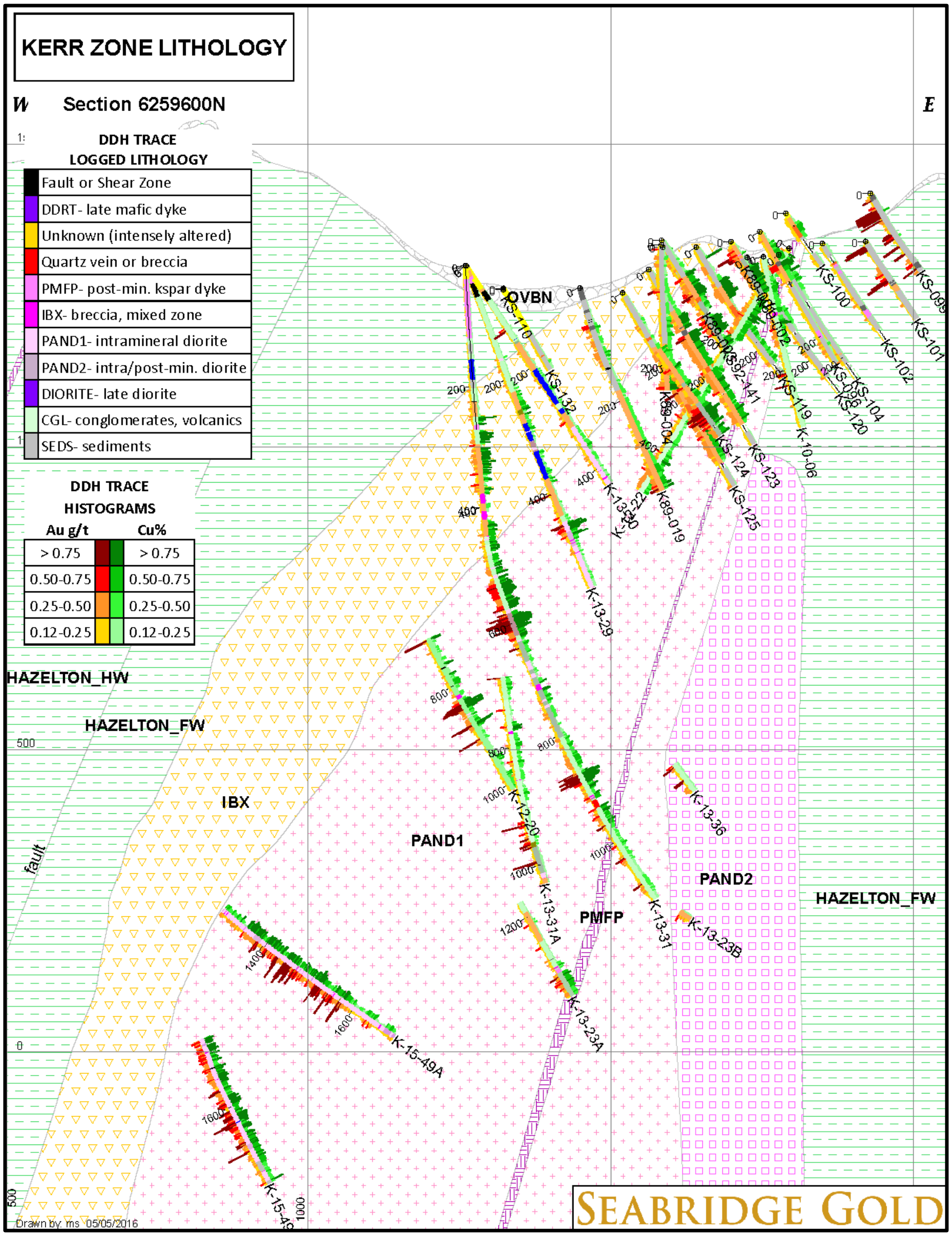


Figure 7.4 Kerr Alteration Cross Section

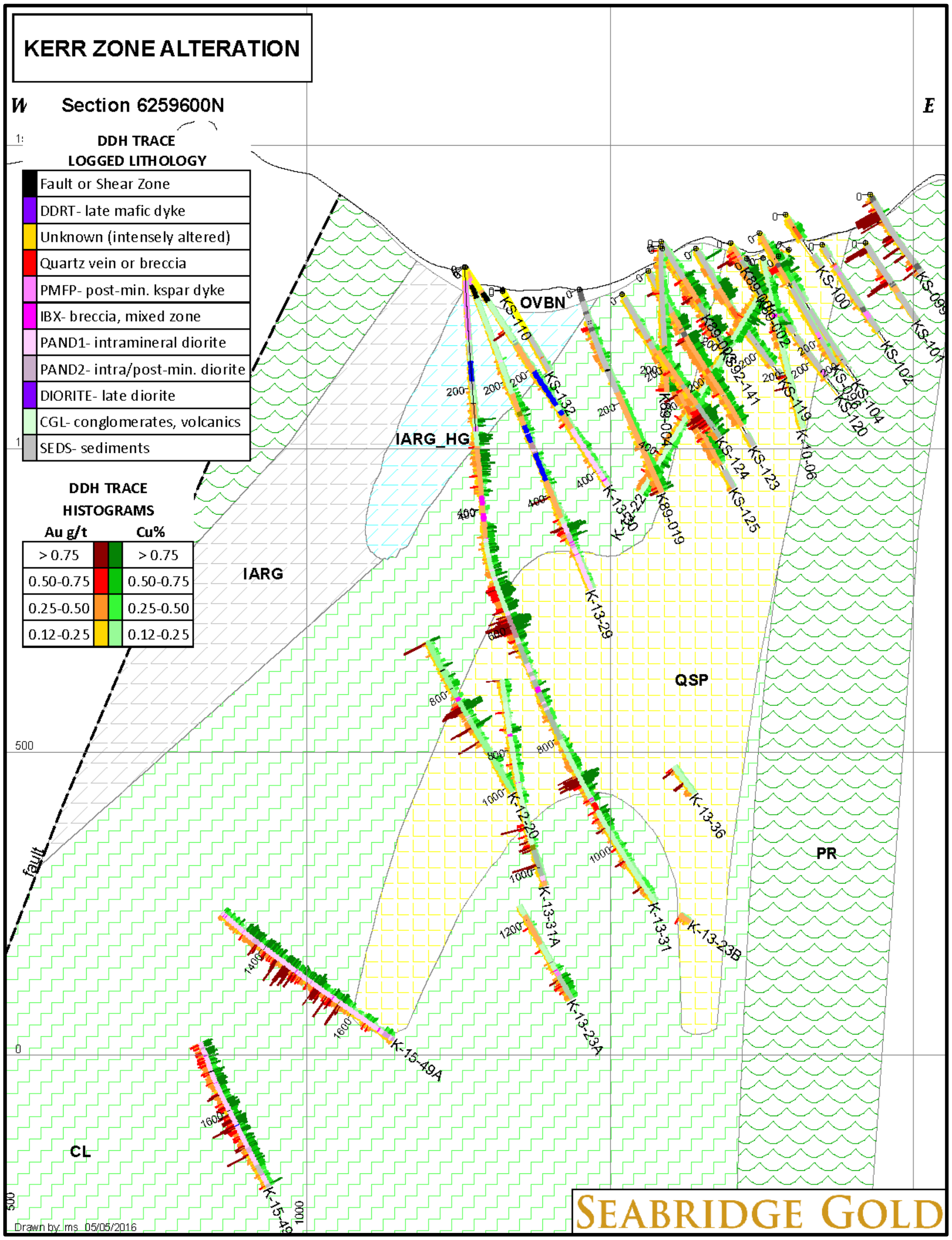


Figure 7.5 Kerr Lithologic Level Plan

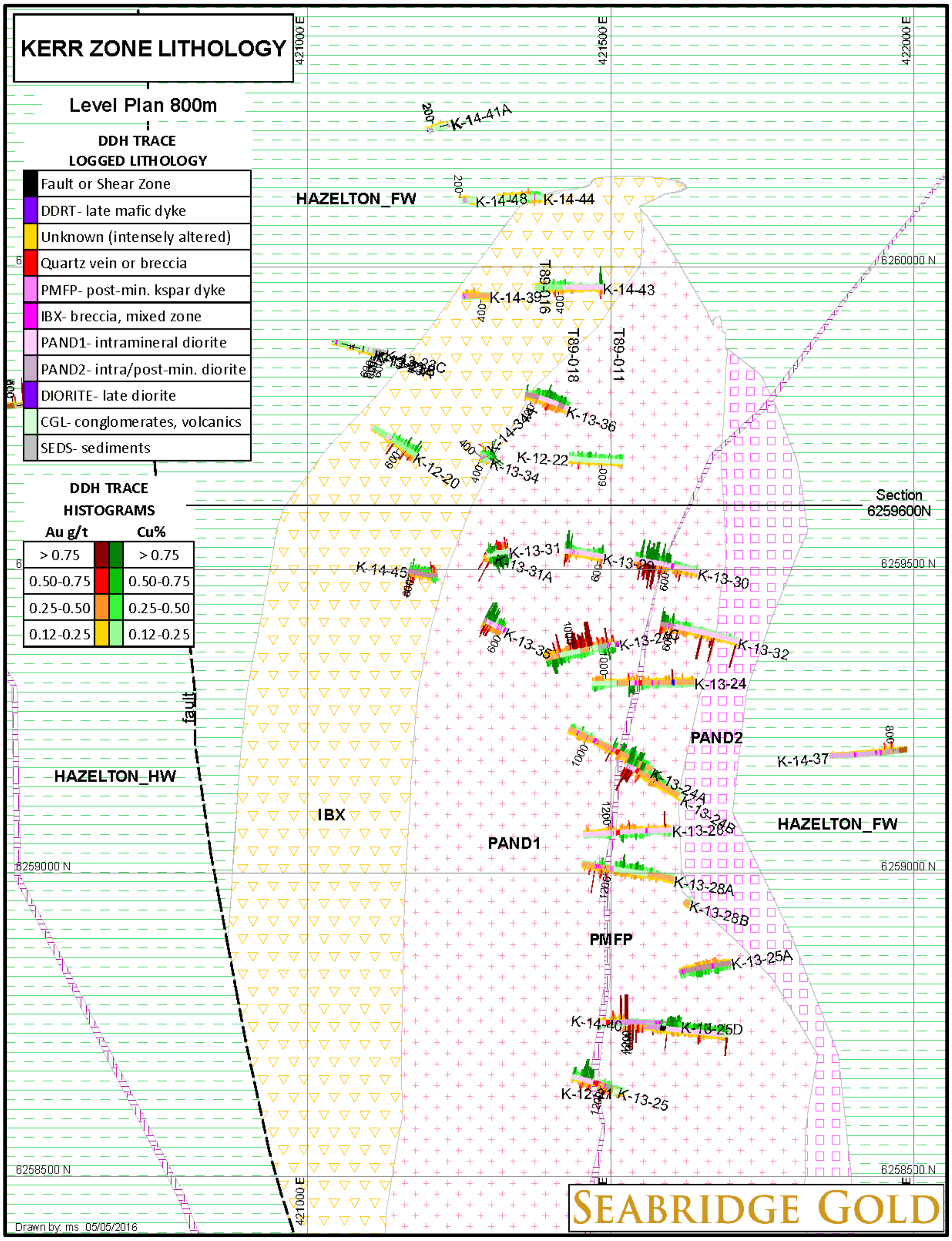
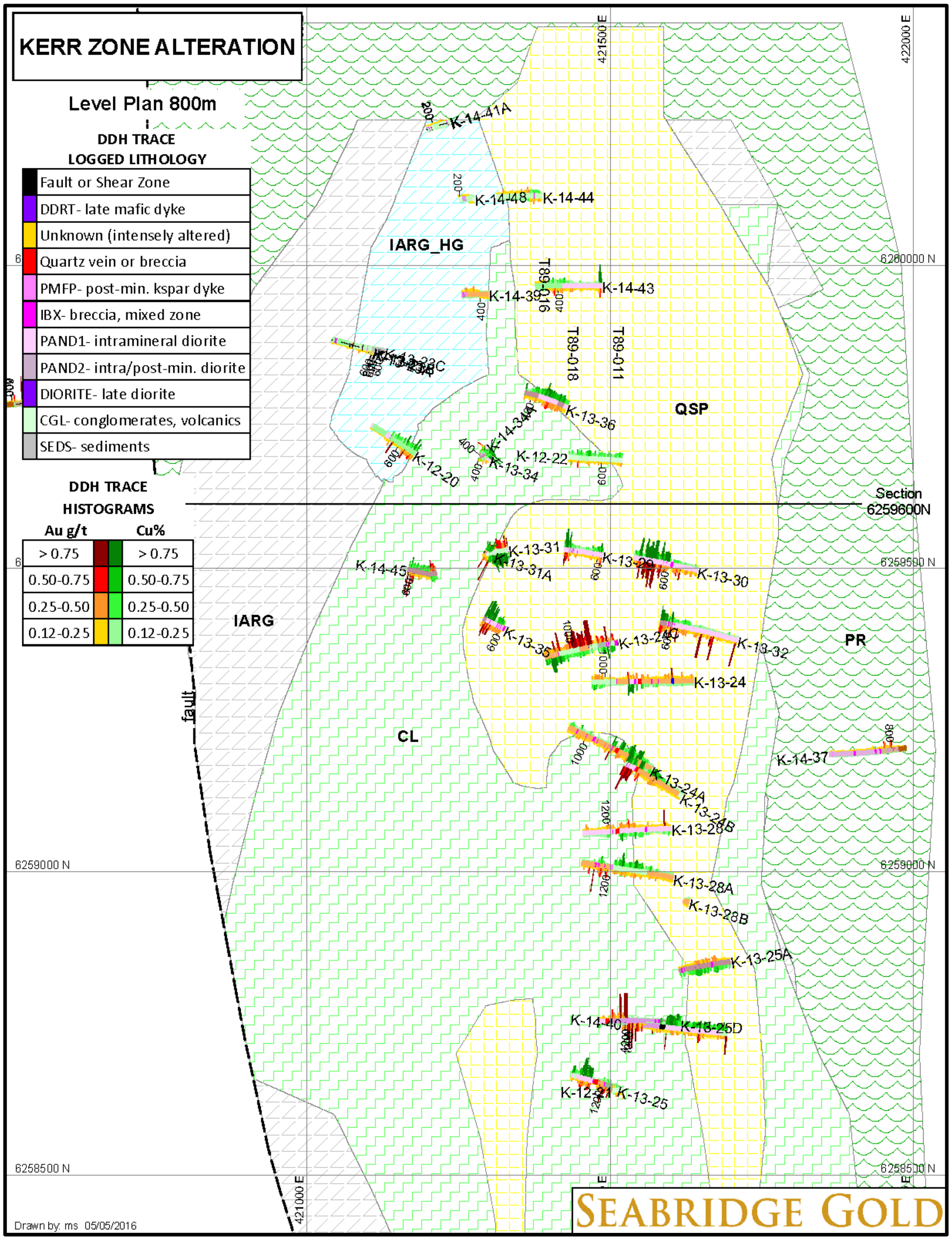


Figure 7.6 Kerr Alteration Level Plan



7.2.2 SULPHURETS ZONE

The Sulphurets Zone 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 sediments and volcanics. The alteration and mineralization are centred on meter scale; sills and dykes of altered porphyritic diorite are believed to tap a stock at depth. It has an apparent north-northeast strike and dips about 45° to the northwest, with approximate true dimensions of 1,000 m in strike, 550 m down dip, and up to 250 m in thickness. It remains open down dip and along strike to the northeast at depth. It may be offset in an echelon style by several north-northeast trending vertical structures. The mineralization is open down-dip. It crops out at surface on the cliff face above Sulphurets Lake and its upper surface follows or is clipped by the STF. The Breccia Gold Zone hosts mostly gold-bearing pyritic mineralization with minor chalcopyrite and sulfosalts in a potassium-feldspar-siliceous hydrothermal breccia that apparently crosscuts the Raewyn Zone. It comprises altered intrusive clasts in a matrix of mainly silica, potassium-feldspar and sulphides. An intense structurally controlled phyllic overprint nearly masks all earlier alteration phases over parts of the zone. Above the STF, low-grade, but widespread disseminated copper-gold mineralization, occurs in hornfels and skarned sediments and volcanics that have been intruded by an non-mineralized porphyritic monzonite. This is referred to as the Main Copper Zone, and it makes up less than 5% of the Sulphurets Resource. Table 7.2 lists various lithologic, alteration, and mineralized wireframes that were constructed by Seabridge personnel for the Sulphurets block model.

Table 7.2 Sulphurets Zone Wireframe Models

Type	Code	Description
Lithology & Alteration	S_HAZ_HW	chlorite-silica-pyrite-magnetite altered, grey-green, volcanics and sediments above the STF in the Main Copper Zone
Lithology & Alteration	S_HAZ_FW	chlorite-silica-pyrite altered, grey-green, volcanics and sediments below the STF in the Sulphurets Zone
Lithology & Alteration	S_RAE_CU	chlorite-silica-biotite-magnetite altered, dark green-grey to black, mottled textures, volcanics and sediments, Raewyn Copper-Gold Zone, bulk of Sulphurets Zone
Lithology & Alteration	S_SUL_AU	silica-pyrite-sericite altered, pale grey, aphanitic rocks and hydrothermal breccia, high gold/copper ratio, Breccia Gold Zone
Lithology & Alteration	S_LCH_AU	low-grade, silica-pyrite-chlorite altered rocks surrounding S_SUL_AU
Lithology & Alteration	S_LWR_AU	chlorite-silica-pyrite altered volcanics and sediments, low-grade ore

Photographs of polished drill core samples that are representative of several key rock types from the Sulphurets Zone are shown in Figure 7.7. The width of the core samples (vertical axis) is approximately 2.5 cm. Figure 7.8 and Figure 7.9show a representative lithologic level plan map and cross section through the Sulphurets deposit, respectively.

Figure 7.7 Sulphurets Polished Core Photographs

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
S-09-14	177	0.92	0.70	0.0200	0.3	30	48	158

Sulphurets – Raewyn Cu-Au zone: potassic altered, hornfels sediments, with fine disseminations and clusters of cp and py, mostley fracture controlled, rare veining

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
SG92-15	187.3	2.83	0.08	0.0020	0.6	308	8	70

Sulphurets – Breccia Gold zone: intensely silicified, pyritized rock fragments in matrix of silica, k-spar and pyrite, minor cp, molybdenite, tennantite

Figure 7.8 Sulphurets Lithologic Level Plan

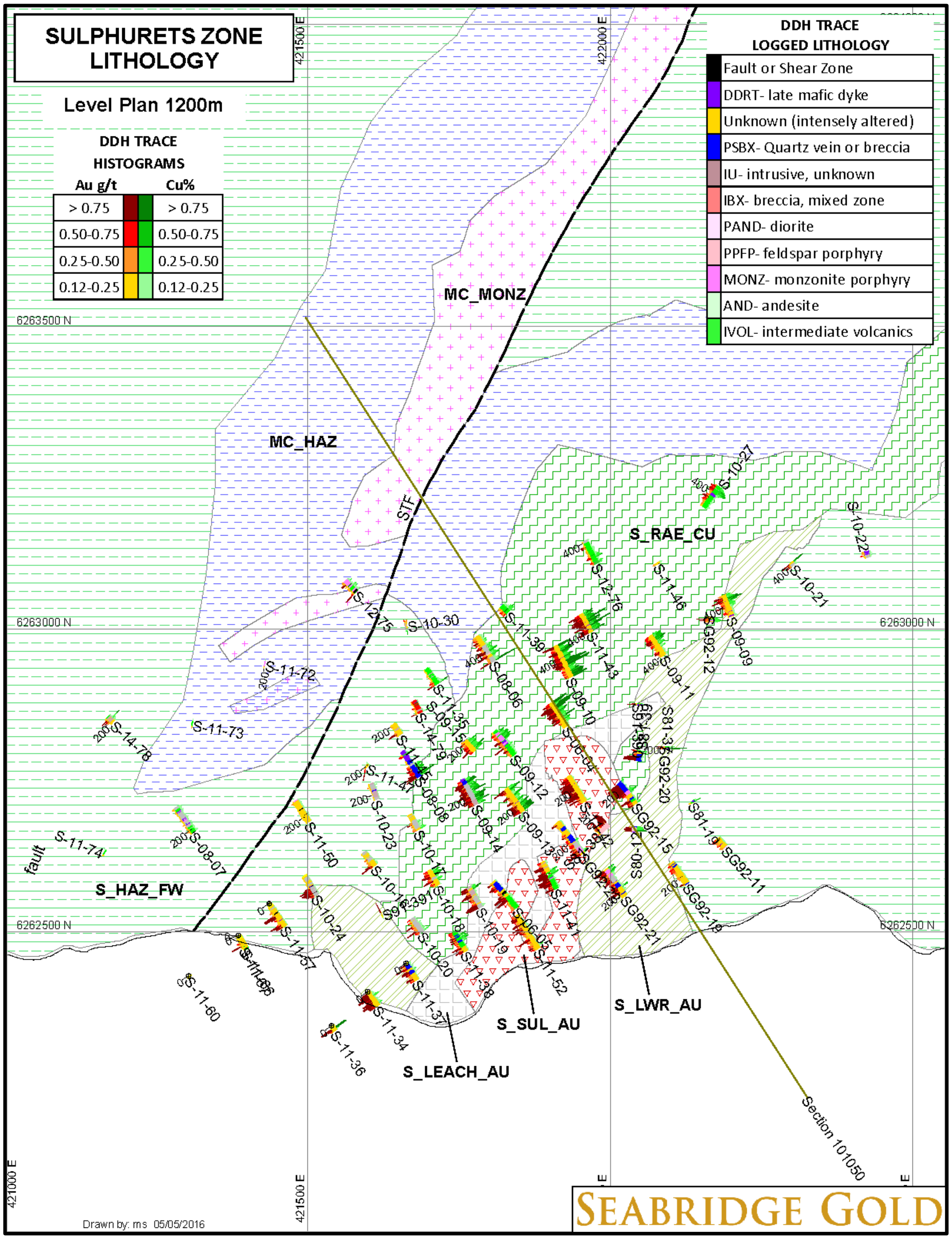
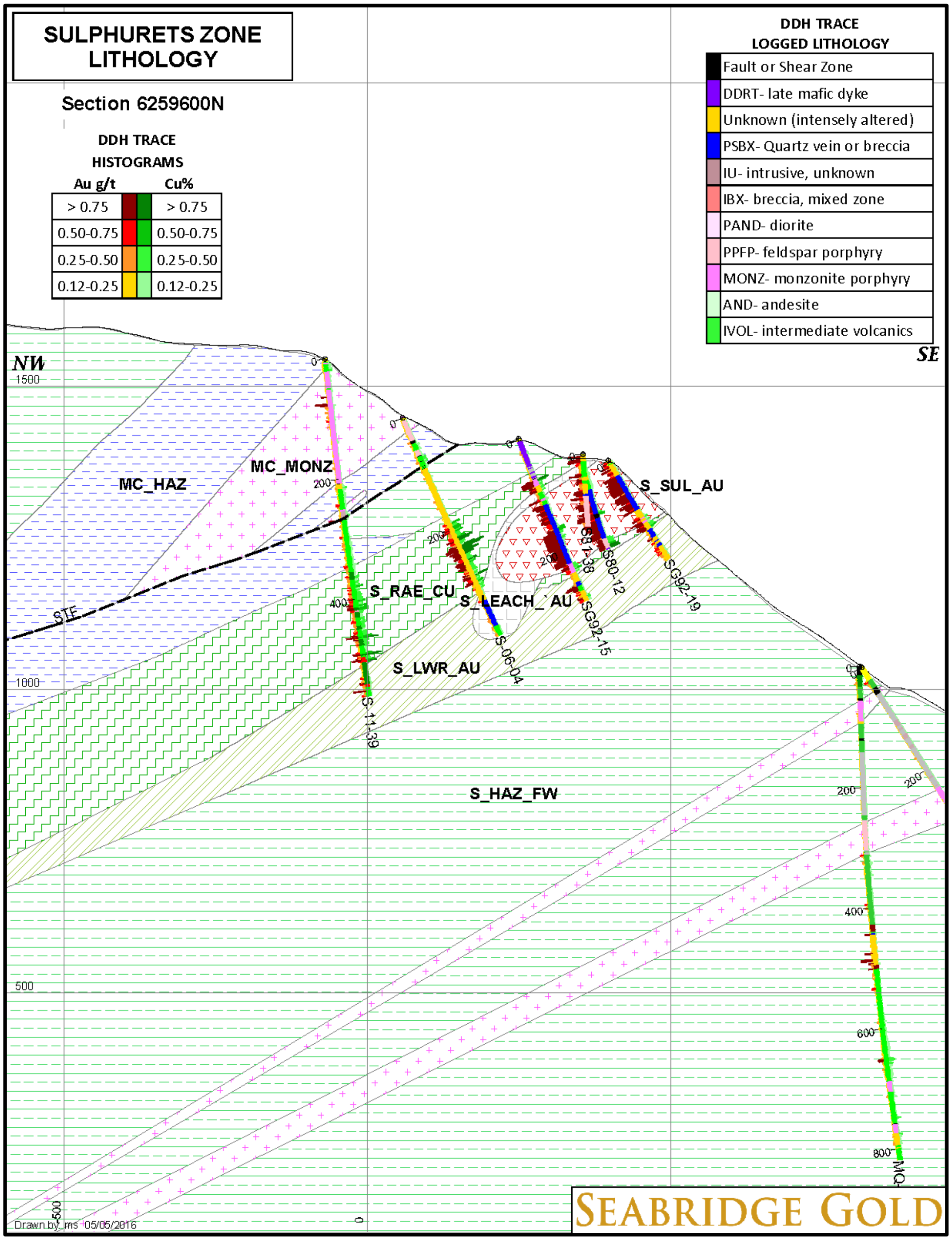


Figure 7.9 Sulphurets Lithologic Cross Section



7.2.3 MITCHELL ZONE

The Mitchell Zone crops out in Mitchell Valley, through an erosional window exposing the footwall of the MTF. The zone is a moderately dipping, roughly tabular gold-copper deposit with approximate true dimensions of 1,600 m in strike, 1,500 m down dip, and up to 850 m in thickness. It remains open down dip and along strike to the northeast at depth. It consists of a foliated, schistose or mylonitic zone of intensely altered and sulphide 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 an 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 sulphide 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 sulphides has produced prominent gossans and extensive copper sulphate precipitates at the surface.

The deposit is genetically related to multiple diorite intrusions that cut sedimentary and volcanic rocks of the Stuhini Group (Upper Triassic) and sandstones, conglomerates, and andesitic rocks of the Jack Formation (basal Hazelton Group; Lower Jurassic). Mineralization and accompanying alteration and stockworks proceeded in four stages. Hosted by Phase 1 plutons (196 ±2.9 Ma and 192.2 ±2.8 Ma), Stage 1 sheeted veins and stockworks contain most of the copper-gold mineralization and potassic and propylitic alteration. A Stage 2 disseminated and stockwork-hosted molybdenum halo (190.3 ±0.8 Ma; rhenium-osmium) is peripheral and contiguous with the core copper-gold system. It is associated with phyllic alteration and is temporally related to a Phase 2 pluton (189.9 ±2.8 Ma) that outcrops central to the halo. Stage 3 consists of poorly mineralized massive pyrite veins associated with advanced argillic alteration and is related to Phase 3 diorite, diatreme breccia emplacement and intrusion breccia dikes. Stage 4 consists of high-level, gold-rich veins that are lateral to, and overprint, the main deposit. The geochemistry of the intrusions, nature and extent of alteration assemblages, high silica content of the ore zone and molybdenum mineralization, indicate that the Mitchell porphyry is calc-alkalic. The deposit was deformed during development of the Skeena fold and thrust belt (mid-Cretaceous), during which it was severed along the MTF. This offset portion now lies approximately 1,200 to the east-southeast in the hanging wall of the MTF (Snowfield Zone) (Febbo, et al., 2015).

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.

The “Bornite Breccia” is found in the center of the Mitchell Zone towards the hanging wall side. It was only intersected in three holes (including one interval of 86 m with 1.42% copper and 0.23 g/t gold), and the interpreted dimensions are about 400 m long down dip, 60 m thick, and 250 m along strike. Its geometry roughly aligns with the northwest plunging trend of the Mitchell deposit. The breccia is composed of a chaotic, swirly mix of crackled and milled light grey quartz, anhydrite and clay, with disseminated and interstitial pyrite, chalcopyrite, bornite, and minor tennantite and molybdenite. In deeper intersects the breccia transitions to a mostly quartz, anhydrite, pyrite, and chalcopyrite hosting structure with only traces of bornite. The breccia body is interpreted to be related to structurally controlled, late advance argillic fluids.

A small portion of the Mitchell Resource (less than 2%) is found in the hanging wall of the MTF, where disseminated and veinlet chalcopyrite occur in magnetite skarn and hornfels altered sediments and volcanics adjacent to a non-mineralized porphyritic monzonite. This style is identical to the Main Copper Zone above the Sulphurets Zone.

Deep drilling in 2015 intersected a distinct medium- to coarse-grained, sub-porphyritic monzodiorite beneath the Mitchell Zone, with grades below the Mitchell average. This intrusion is interpreted to be a later phase, with primary potassium-feldspar phenocrysts, an alteration mineral assemblage dominated by secondary potassium-feldspar, magnetite, epidote and traces of actinolite, and a poor development of stockwork quartz veins and sulphides. This mineral assemblage is characteristic of the deep peripheral zones of porphyry deposits. Also confirmed was the presence of a roughly 50 m thick, banded, mylonitic shear zone that may offset the base of the Mitchell deposit Basal Shear Zone (BSF). The zone dips to the northwest and appears to parallel the MTF.

As the Bornite Breccia and BSF may have structurally offset portions of the Mitchell Zone, potential remains for additional mineralization to be discovered. Various geologic wireframes that were constructed by Seabridge personnel are tabulated in Table 7.3.

Table 7.3 Mitchell Zone Wireframe Models

Type	Code	Description
Lithology	MONZ	mostly barren, late stage, porphyritic monzonite, above MTF
Lithology	PAND1	sub-porphyritic diorite, comprises several similar phases of Mitchell intrusions, main host for mineralization
Lithology	PKFP	potassium feldspar porphyritic monzodiorite, weakly mineralized
Lithology	SEDIMENTS	wall rocks, mostly Hazelton Group sediments
Lithology & Alteration	BRNBRX	Late structure, chaotic, swirly mix of crackled and milled light grey quartz, anhydrite and clay, with dpyrite, chalcopyrite, bornite, and minor tennantite and molybdenite; only traces of bornite in deeper portions
Lithology & Alteration	HI-QTZ	CL and/or QSP altered rocks with greater than 40% by volume quartz stockwork veinlets, may reflect early quartz- and sulphide-rich intrusive phase
Alteration	CL	chlorite-silica altered rocks, often magnetic, ± anhy, quartz stockwork veinlets, moderately to well mineralized; chlorite is metamorphosed secondary biotite; this zone reflects extent of potassic alteration in addition to KP.
Alteration	IARG	sericite-clay-silica altered rocks, often schistose, ± anhy, quartz stockwork veinlets, associated with phyllic overprint, hosts low-grade copper-gold mineralization but most of molybdenum mineralization
Alteration	KP	potassium-spar-magnetite-chlorite-biotite altered rocks, with variable to intense quartz stockwork veining, moderate to well mineralized
Alteration	LCHBRX	zone of copper and gold depletion, forms halo around upper part of BRNBRX
Alteration	QSP	quartz-sericite-clay-pyrite altered rocks, reflects variably phyllic overprint, ± anhy, pale grey to white, quartz stockwork veinlets, moderately well mineralized, pyrite>chalcopyrite
Alteration	PR	chlorite-silica-pyrite-epidote-carbonate altered rocks, low density of quartz stockwork veinlets, weakly mineralized peripheral zone, includes hornfelsed wall rocks

Photographs of polished drill core samples representative of rock types from the Mitchell Zone are shown in Figure 7.10. The width of the core samples (vertical axis) is approximately 2.5 cm. A representative cross section and level plan are also shown in Figure 7.11 to Figure 7.14.

Figure 7.10 Mitchell Polished Drill Core Photographs

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
M-06-009	245.45	0.76	0.26	0.0047	3.3	45	26	46

Mitchell – PAND, CL altered with B veins, fine disseminated cp and py, in veins and haloes, moderate density of quartz stockwork veins

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
M-07-053	243.95	1.28	0.39	0.0022	5	0.1	30	55

Mitchell – PAND, KP altered (k-spar, magnetite), with A and B veins, fine disseminated cp and py, high density of quartz stockwork veins

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
M-08-086	251	0.28	3.64	0.0086	68.1	35	56	37

Mitchell – BRNBRX: Breccia structure, with anhy-py-cp-bn-mo, originally early, high qtz intrusion, traces of tennantite, bornite oxidized to indigo blue after cutting

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
M-15-131A	1583.5	0.28	0.10	0.0007	1.3	6	5	67

Mitchell – PKFP, monzodiorite porphyry, late phase, coarser grained, calcic-potassic altered, A+B veins, low grade, may indicate lower or lateral edge of mineralization

Representative lithologic and alteration cross sections are shown in Figure 7.11 and Figure 7.12, respectively.

Figure 7.11 Mitchell Lithologic Cross Section

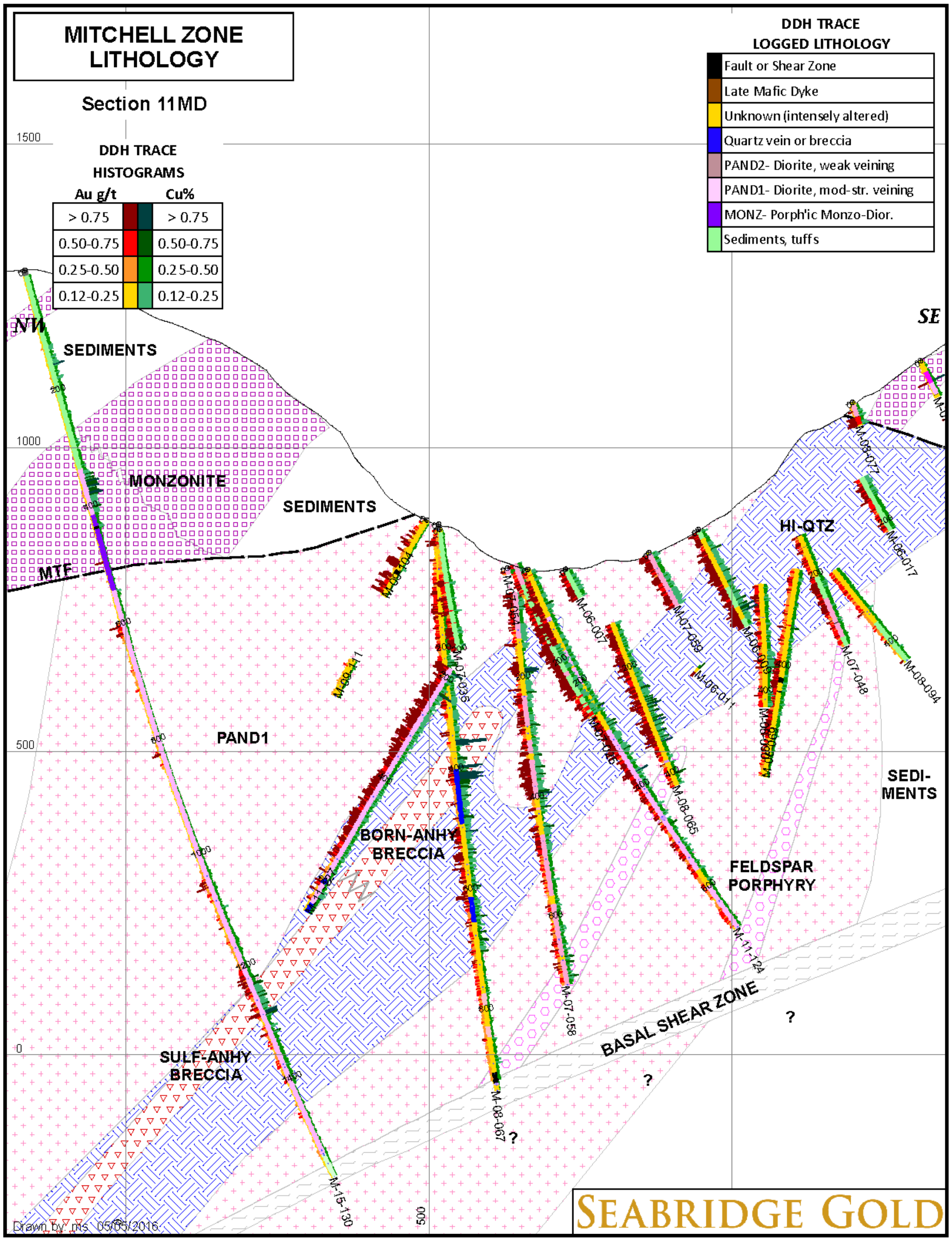


Figure 7.12 Mitchell Alteration Cross Section

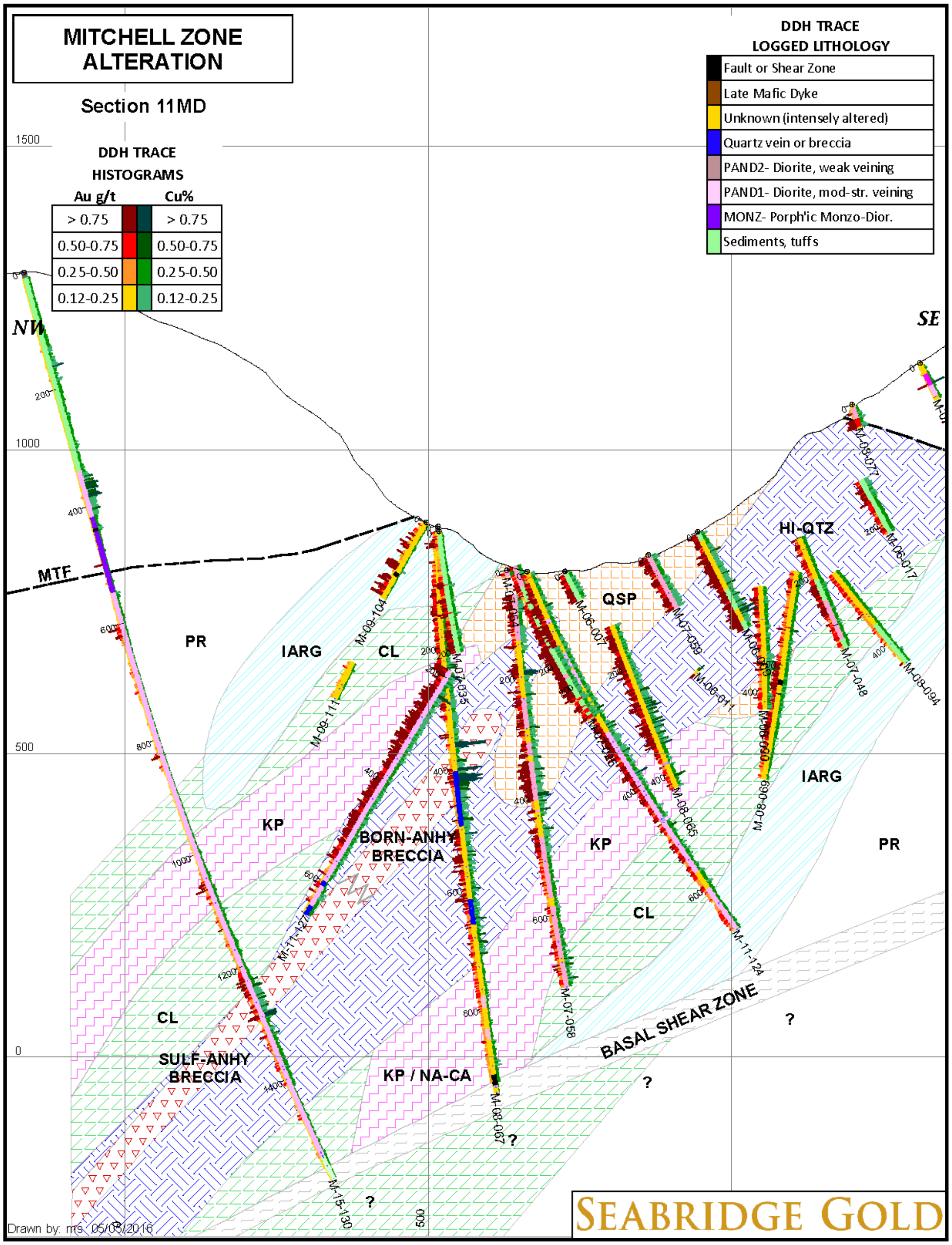


Figure 7.13 Mitchell Lithologic Level Plan

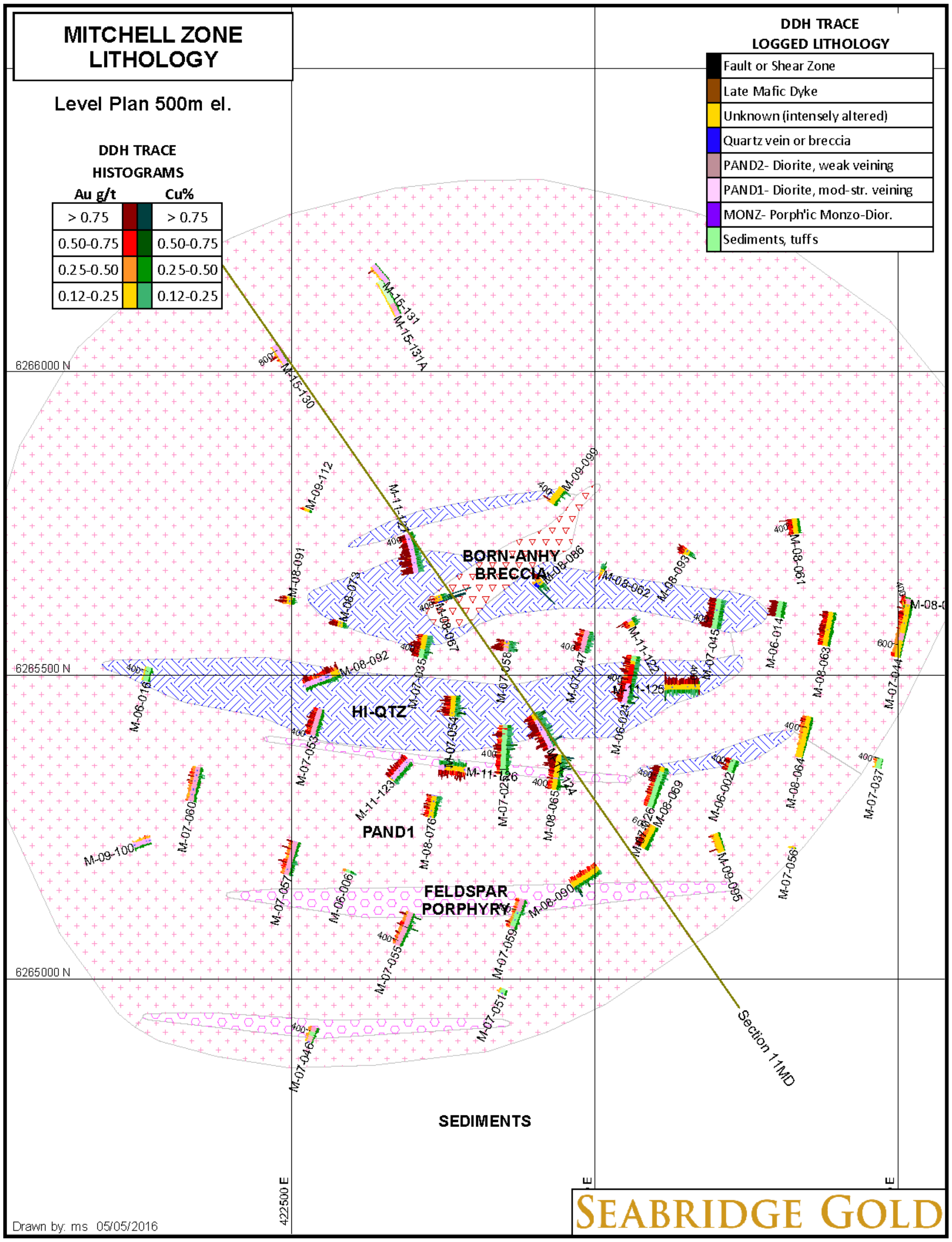
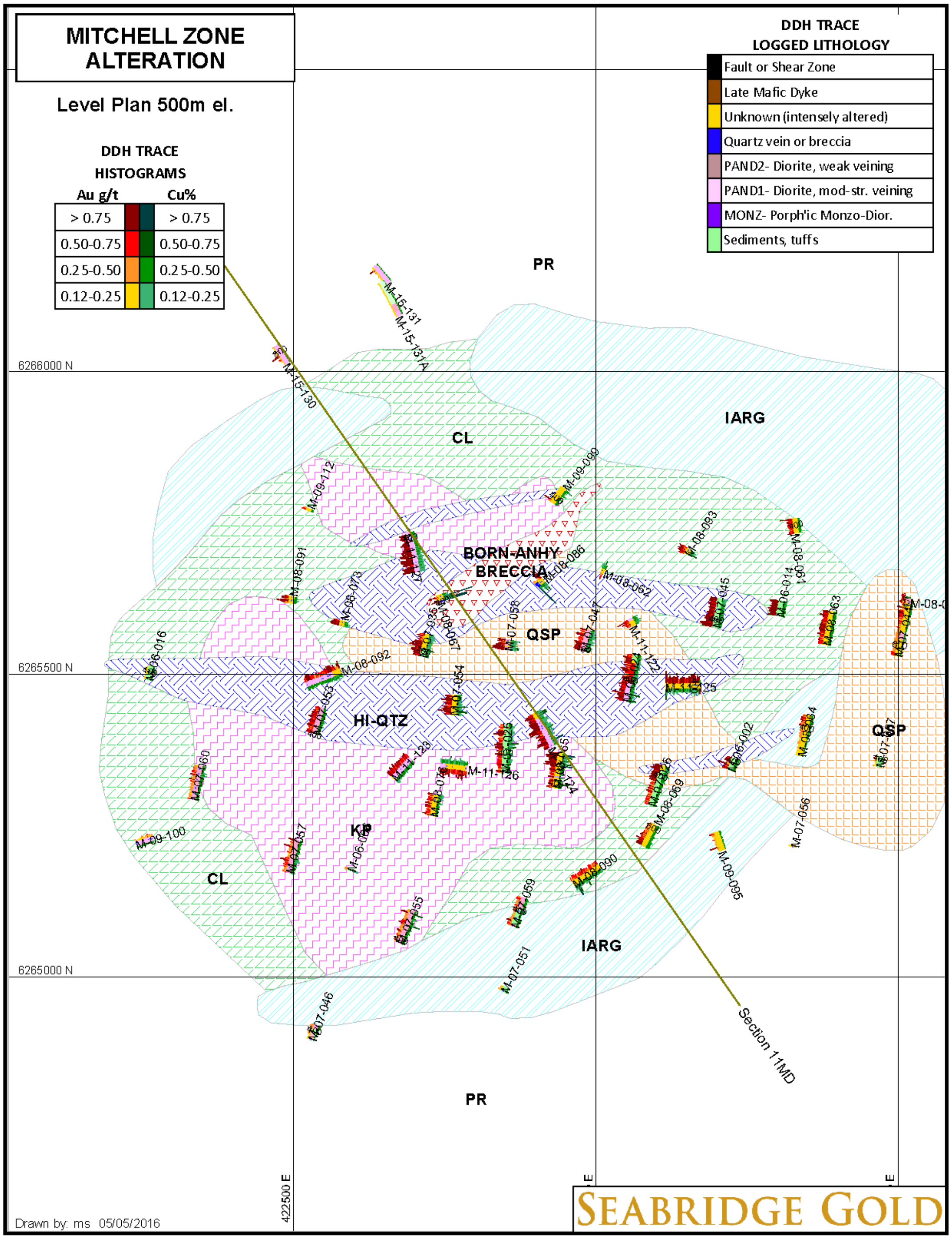


Figure 7.14 Mitchell Alteration Level Plan



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 MTF and STF. It is now considered to be hosted by a fine-grained porphyritic diorite similar to the Mitchell and Kerr zones. It has similar geometry; a northwest dipping tabular body with approximate true dimensions of 850 m in strike, 1,200 m down dip, and up to 500 m in thickness. It remains open down dip and along strike to the north-northeast. Intense alteration has obliterated most original textures. Stockwork veinlet density is generally low, and no high quartz volumes phases have been identified as at Kerr and Mitchell. Intrusive breccia phases are more common, but are not necessarily well mineralized. Host rocks are mostly sandstones, siltstones, and conglomerates of the Hazelton Group. There is a high degree of silicification in the wall rocks and peripheral phases, which overprints earlier potassic and chloritic alteration. Copper-bearing zones at Iron Cap demonstrate higher grades than Mitchell, which is consistent with the intrusive setting and potassic alteration, suggestive of a deeper and hotter environment. However, minerals usually indicative of higher levels, such as galena, sphalerite, and silver or arsenic sulphosalts, although minor in abundance, are distributed throughout the upper half of the zone, and may be a consequence of telescoping or downward migration of the hydrothermal system.

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, similar to the Mitchell deposit.

Generally intense silicification at the higher, eastern portions gives way to chloritization with some preserved potassium-spar alteration at depth and towards the west, which correlates with an 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.

The Iron Cap Zone terminates at the south along the north-dipping Iron Cap Fault (ICF). South of the fault, hornfelsed sediments are mineralized with marginal gold and copper grades similar to intervals above the MTF at Mitchell. A few holes through this area contain higher than average molybdeum grades, including in interval of 133 m with 0.10% molybdenum; however, there are no Resources defined south of the ICF due to insufficient drilling. Various lithologic and alteration wireframes used in developing geologic models for the Iron Cap deposit are summarized in Table 7.4.

Table 7.4 Iron Cap Wireframe Models

Type	Code	Description
Lithology	DIOR	sub-porphyritic, intermediate intrusive rock (dioritic), 1 to 5 mm average grain size, less than 10% aphanitic groundmass
Lithology	HAZL SEDS	Hazelton Group sandstones, mudstones, conglomerates (Jurassic) between ICF and Northwest Fault (NWF) (Iron Cap Zone panel)
Lithology	HAZL SEDS FW	Hazelton Group sandstones, mudstones, conglomerates (Jurassic) below ICF
Lithology	HAZL SEDS HW	Hazelton Group sandstones, mudstones, conglomerates (Jurassic) above NWF
Lithology	IU	highly altered rock, lithic textures almost all destroyed, probably mostly intrusive origin, in part intrusive breccia esp. near DIOR contact
Lithology	PAND1	porphyritic intrusive rock, phenocrysts generally elongate green sericite pseudomorphs, 3 to 5 mm long and 1 to 2 mm thick, may be aligned
Lithology	PMON	porphyritic monzonite, generally weak or non-mineralized
Alteration	HFLS FW	quartz-chlorite-sericite hornfels altered HAZL SEDS, below ICF
Alteration	HFLS FW	quartz-chlorite-sericite hornfels altered HAZL SEDS, above NWF
Alteration	KP	potassic altered DIOR, pervasive potassium-feldspar, shreddy chlorite, mt at depth, rare biotite, commonly with centimetre-scale quartz-chlorite-chalcopyrite-pyrite stockwork veinlets
Alteration	PMON KP	potassic altered PMON, pink rimmed (hematitic) orthoclase phenocrysts, chloritized mafics
Alteration	QSP-CL	strong pervasive phyllic alteration, quartz-sericite-phengite-muscovite-chlorite-pyrite
Alteration	SIL-QSP	strong pervasive silica alteration, with sericite-muscovite-illite-phengite-pyrite

Photographs of polished drill core samples representative of rock types from the Iron Cap deposit are shown in Figure 7.15. The width of the core samples (vertical axis) is approximately 2.5 cm.

Representative lithologic and alteration cross sections are shown in Figure 7.16 and Figure 7.17, respectively. Lithologic and alteration level plan maps are shown in Figure 7.18 and Figure 7.19, respectively.

Figure 7.15 Iron Cap Polished Drill Core Photographs

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
IC-13-051	831.6	0.59	0.37	0.0010	3.6	32	43	84

Iron Cap – DIOR: diorite porphyry, more intense phyllic overprint, abundant fine disseminated py, py>cp

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
IC-13-049	632.9	1.98	0.59	0.0020	5	27	37	130

Iron Cap – DIOR: diorite porphyry, phyllic altered but with some preserved potassic alteration, kinked cp rich, B type quartz veins

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
IC-14-060	735.4	0.12	0.03	0.0001	0.5	3	14	50

Iron Cap – HFLS: intensely hornfelsed sediment, brown tones due to remnant secondary biotite dusting

HOLE-ID	DEPTH	Au g/t	Cu %	Mo %	Ag g/t	As ppm	Pb ppm	Zn ppm
IC-10-017	31.9	0.19	0.07	0.0012	0.6	15	33	22

Iron Cap – IU: intensely phyllic altered rock, intrusive breccia or conglomerate?

Figure 7.16 Iron Cap Lithologic Cross Section

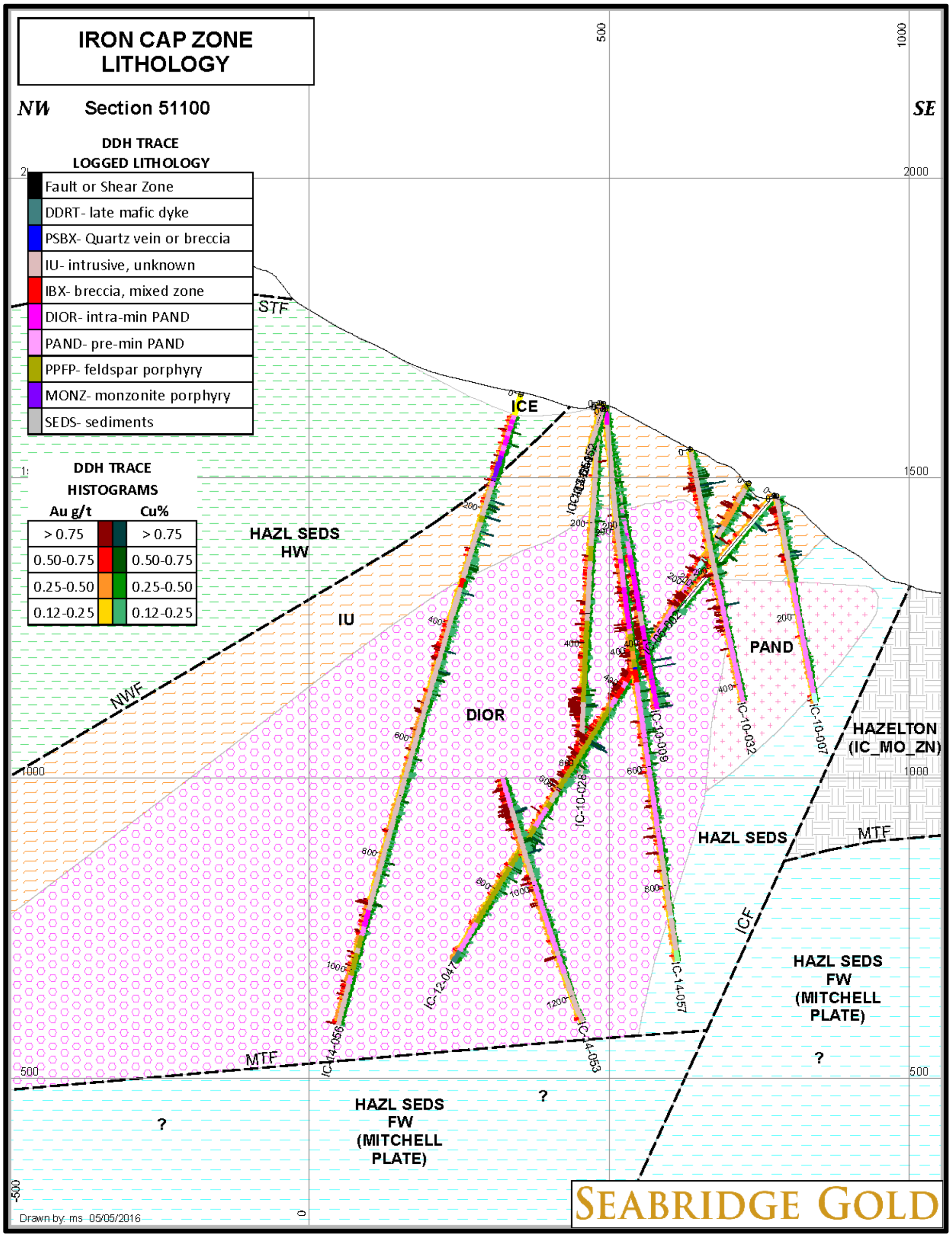


Figure 7.17 Iron Cap Alteration Cross Section

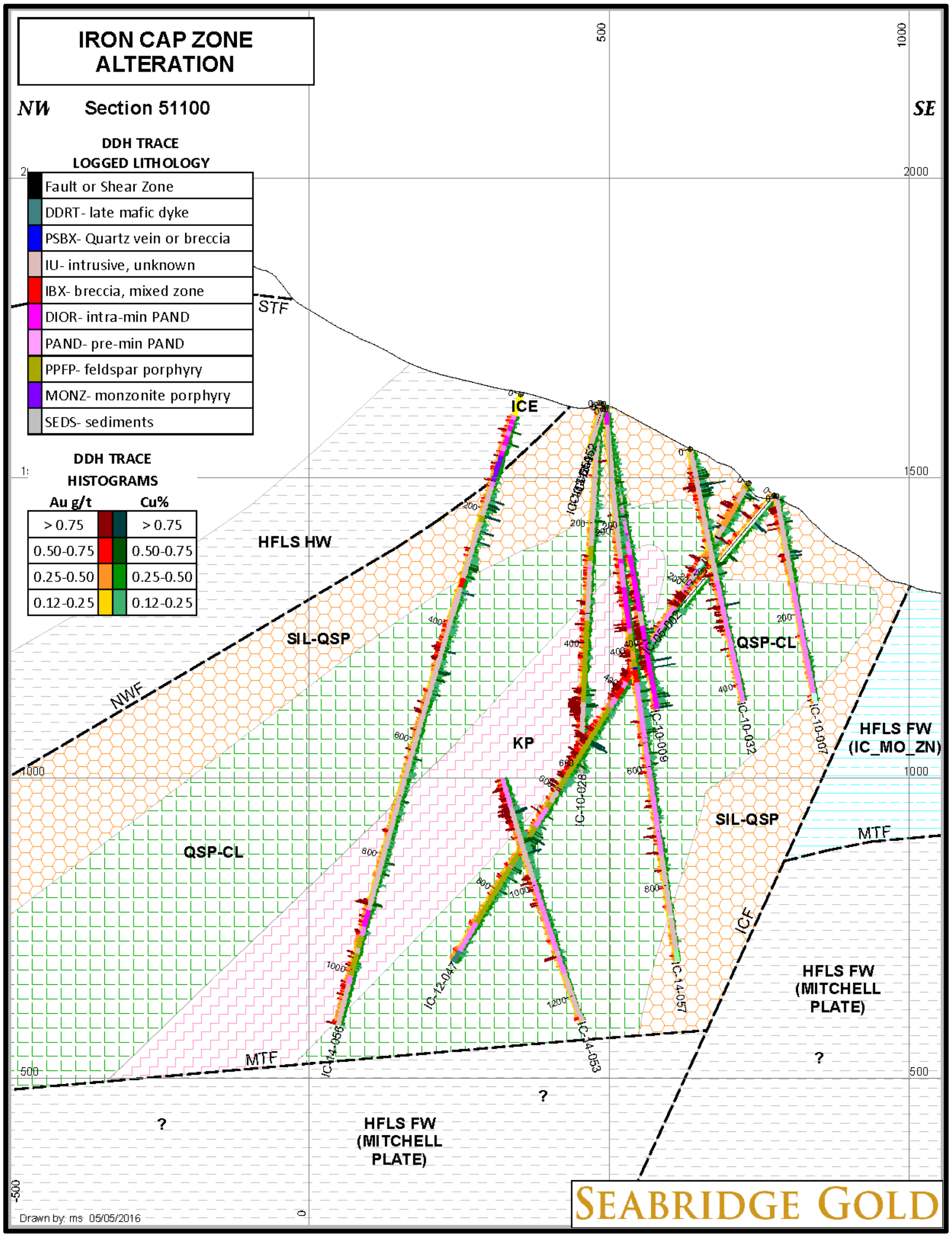


Figure 7.18 Iron Cap Lithologic Level Plan

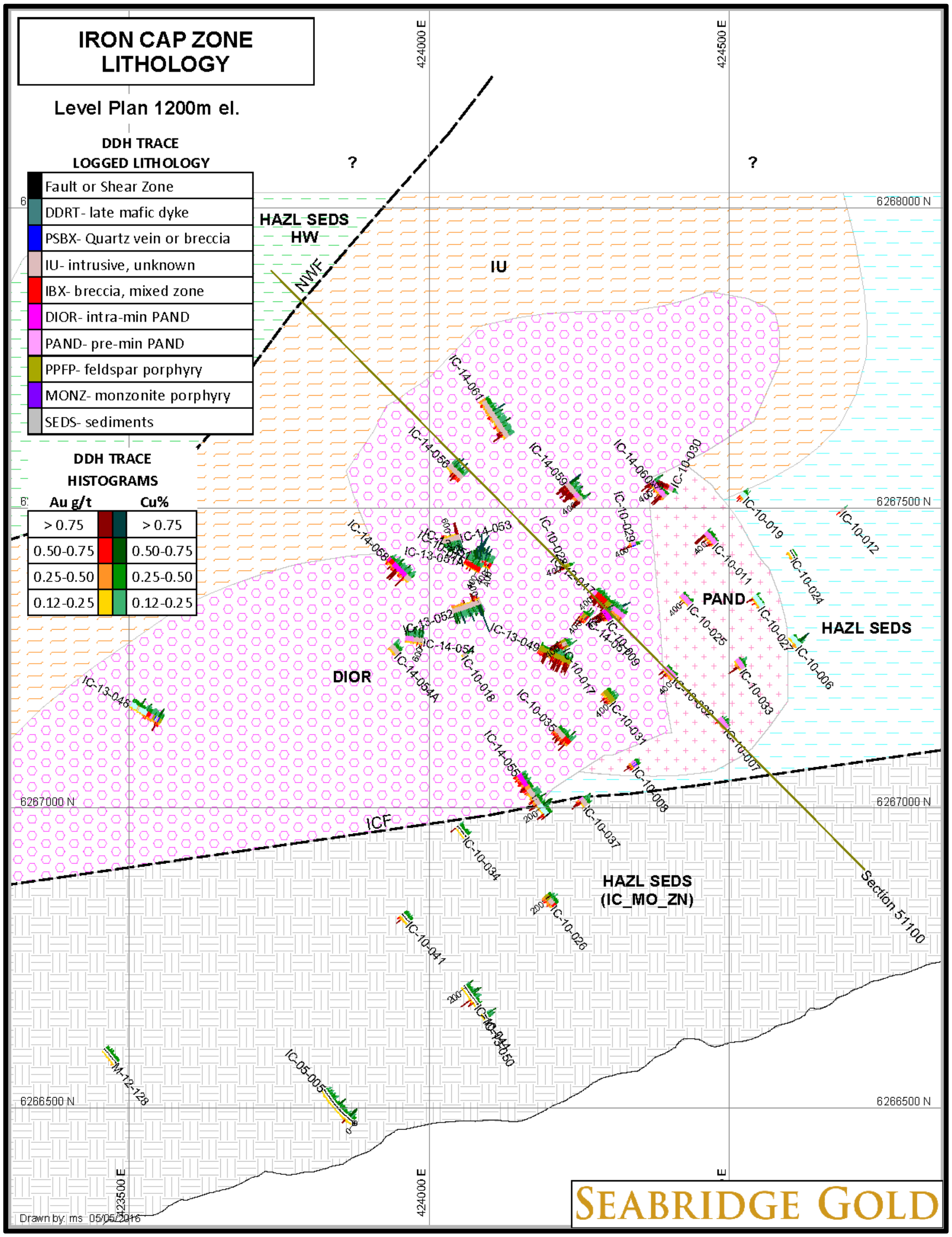
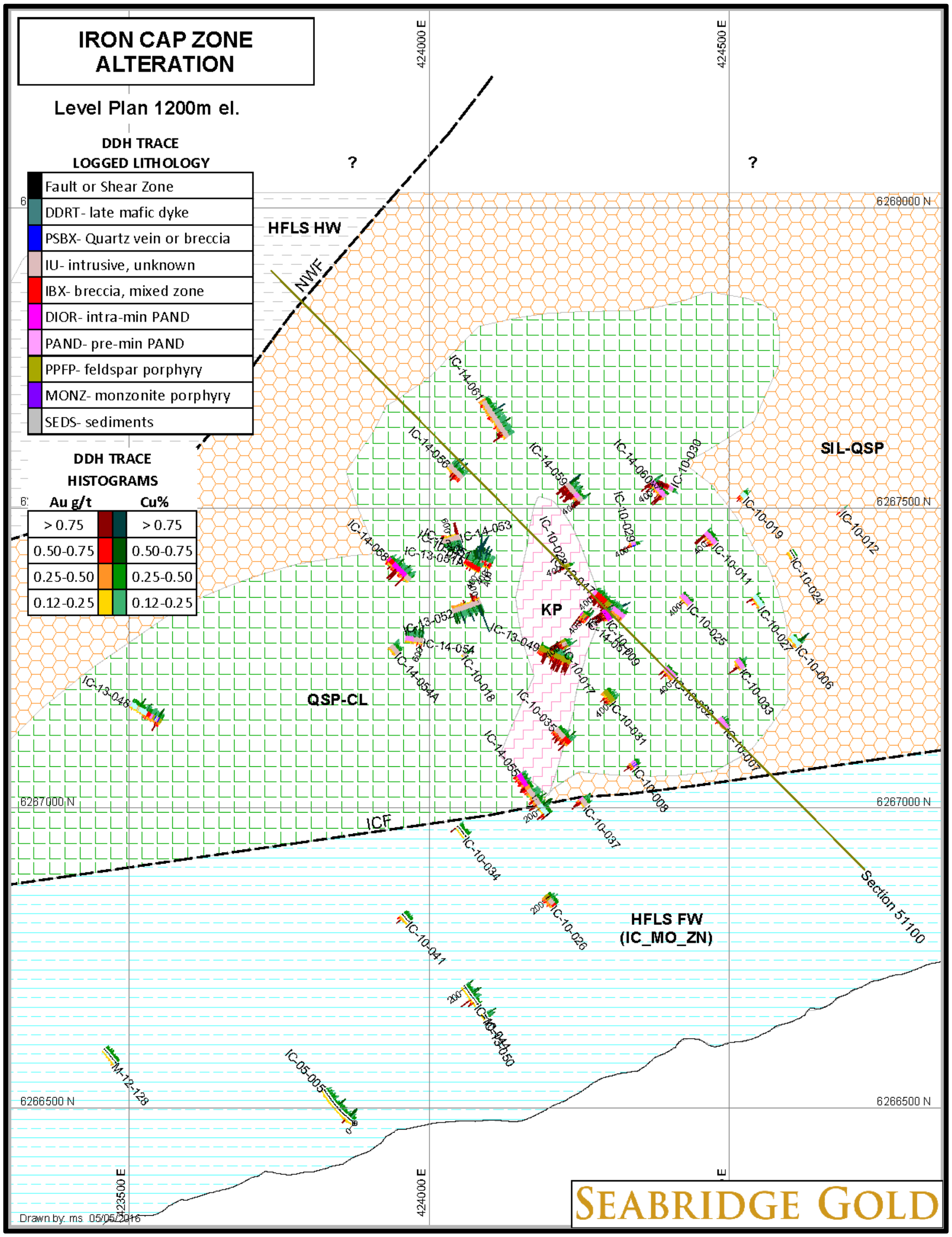


Figure 7.19 Iron Cap Alteration Level Plan



8.0 DEPOSIT TYPES

The KSM intrusive complex demonstrates many features characteristic of giant diorite or monzonite hosted gold-copper porphyry systems, such as Grasberg, Oyu Tolgoi, Bingham, and Pebble. Porphyry deposits are the product of magma genesis at convergent plate margins. Melting of lower crust by upwelling lithospheric and asthenospheric mantle is the source of primitive and oxidized, metal- and volatile-rich magmas. These buoyant, hydrated magmas are forced to shallow depths up deep penetrating faults. As they rise, they become progressively enriched in soluble metals and other elements. Near the surface, they experience drastic temperature, pressure, and chemical changes that force the precipitation of metals and unique mineral assemblages in an upward and outwardly zoned pattern that characterizes a porphyry system.

The intrusions display characteristics of both calc-alkalic than alkalic types. Although they have relatively high magnetite and gold contents, the chemical composition straddles the alkalic/calc-alkalic boundary. The high quartz vein, pyrite and molybdenum contents, strong phyllic to advanced argillic overprinting, and large scale of the deposits are considered characteristic of calc-alkaline magmatism.

The KSM complex is a cluster of deposits located in the Stikine arc terrane within the Intermontane Belt of the Canadian Cordillera, geographically inboard of the Coast Plutonic Complex, and accreted to the North American plate. Long-lived arc magmatism during the Late Triassic to Early Jurassic generated paired belts of alkalic and calc-alkalic porphyry deposits that extend along the axis of the Canadian Cordillera. The Stikine terrane comprises three unconformity-bounded island arc volcanosedimentary successions that span 200 Ma of geologic evolution.

The composite intrusive complex has demonstrated vertical continuity down to near-magmatic bornite-bearing core zones and upward through voluminous mineralized stock works into near surface epithermal vein deposits. This vertical zonation is typical of many of the world's largest mining districts. The original architecture has been rearranged by three phases of progressive deformation related to the mid-Cretaceous Skeena fold and thrust. Phyllosilicate alteration assemblages and stockwork vein networks are commonly mylonitized adjacent to thrust faults. Alteration and metal zoning confirms the adjacent Snowfield deposit is the truncated cap of the Mitchell deposit, with an offset of approximately one kilometer.

The KSM complex hosts an extensive alteration and mineralization system centered on a cluster of hypabyssal, Early Jurassic “Mitchell” sub-porphyritic diorite to monzodiorite, island arc tholeiite series intrusions. The Kerr, Mitchell and Iron Cap deposits are hosted by multiphase stocks with dimensions of approximately 600 to 1,200 m in diameter and up to 2,200 m vertical. The Kerr stock is elongated and bifurcated in a north-south trend extending 2,400 m, whereas Mitchell and Iron Cap stocks trend north-northeast. All tend to plunge towards the northwest. The intrusions at Sulphurets occur as much smaller dykes but the trend is similar. Mineralization may be associated with quartz veinlet stockworks and sheeted quartz veinlet arrays, with vein density decreasing in later phases. Host rocks may be mineralized for up to several hundred meters from the intrusions. Less commonly, mineralized intrusive-hydrothermal breccias cut through previously veined and mineralized rocks.

Principal sulphides are pyrite and chalcopyrite, with minor molybdenite, and trace amounts of tennantite, bornite, sphalerite, and galena. Magnetite and hematitized magnetite are common, especially in deeper parts of the deposits, and anhydrite is common in certain phases though unevenly distributed. Native gold is rarely observed, and most occurs as microscopic clusters at sulphide grain boundaries or inclusions. All mineralization is hypogene, except for a small remnant of preserved supergene mineralization at the upper limits of the Kerr deposit where chalcocite coatings on pyrite and chalcopyrite have been observed, and at the Main Copper (Sulphurets) occurrence where a remnant of leached capping and partial oxide mineralization is preserved at the highest elevations.

All of the KSM deposits are open at depth.

9.0 EXPLORATION

This section describes Seabridge's 2012 through 2015 exploration programs at KSM. Prior exploration activities have been described in various Technical Reports prepared by RMI (Lechner 2007; 2008a; 2008b; 2009; 2010; 2011; and 2014). Since 2012, most of the exploration activity at KSM has centered on drilling programs designed to test for potential high-grade feeder zones that may be associated with the currently recognized near surface mineralized areas. Much of the discussion in this section describes various geological observations and results obtained by the focused deep drilling programs conducted from 2012 through 2015.

9.1 2012 KSM EXPLORATION PROGRAM

Updated geological models support a conceptual undiscovered central core zone to the porphyry gold/copper deposits in the KSM mining district, expected to contain significantly higher copper and gold grades in a deposit of similar scale to that which has been delineated. A central core zone to this style of porphyry system is an essential component of the fluid processes that formed the KSM mineral system defined to date. The program in 2012 was designed to utilize accumulated knowledge to test the concept of a preserved central bornite-core zone on the KSM land holdings.

The erosional level and structural displacement in the KSM district preserved the entire vertical mineralized column of a porphyry gold/copper system. Analysis of translation along thrust faults strongly supports the idea that a high-grade core zone is preserved on Seabridge claims. Initially, four separate targets were identified as potential deep core zones that were supported by the results of the magneto telluric (MT) geophysical survey including:

- dip projection of the Sulphurets deposit
- lateral projections to the Mitchell/Iron Cap deposits
- lateral projections to the Sulphurets deposit
- dip projection of the Kerr deposit.

9.1.1 RESULTS OF 2012 EXPLORATION PROGRAM

KERR

The dip projection of the Kerr deposit returned a very discrete and high-amplitude, low-resistivity anomaly in the MT survey. The Kerr Resource is open at depth and this anomaly provides good support for the dip continuity of the deposit. Initially, the drill core showed the intense hydrothermal alteration characteristic of the Kerr deposit, and abundant sulphide minerals with better than average copper grades as illustrated in Table 9.1.

Table 9.1 Select 2012 Kerr Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
K-12-20	1,011.0	559.8	1,011.0	451.1	0.27	0.45
		829.0	852.0	23.0	0.90	1.17
K-12-21	855.3	20.0	493.0	473.0	0.31	0.90
		503.0	519.0	16.0	2.20	0.04
		537.0	553.3	16.3.0	6.90	0.03
		672.0	738.0	66.0	0.38	0.37
		767.0	781.0	14.0	1.80	0.15
K-12-22	862.5	21.0	177.0	156.0	0.24	0.65
		227.0	552.0	325.0	0.27	0.48
		754.5	776.5	22.0	0.33	0.98

Source: Seabridge

SULPHURETS

A low-resistivity MT anomaly along the dip projection of the Sulphurets deposit was tested with three drill holes, totaling 2,306 m. Significant drill hole intercepts are summarized in Table 9.2. These holes encountered extensive thermally metamorphosed or hornfelsed rock.

Table 9.2 Select 2012 Sulphurets Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
S-12-75	717	44.0	77.0	33.0	0.25	0.18
		100.7	153.7	52.9	0.25	0.16
		192.0	294.0	102.0	0.32	0.22
		366.5	388.0	21.5	1.25	0.07
		394.0	555.0	161.0	0.41	0.22
S-12-76	699	468.0	533.1	65.1	0.36	0.29
		683.0	695.0	12.0	0.92	0.02
S-12-77	900	646.0	661.0	15.0	0.72	0.02

Source: Seabridge

The lateral projection to the northeast of Sulphurets is known as the Ice Field target. Several previous drill holes designed to extend the Sulphurets deposit encountered intensive alteration with gold grades. A low-resistivity MT anomaly corresponded to this target area. Two drill holes, totaling 1,410 m were completed to test this target with select results summarized in Table 9.3.

Table 9.3 Select 2012 Ice Field Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
IF-12-03	783	64.6	78	13.4	2.31	0.01
		180.0	215	35.0	0.47	0.07
IF-12-04	627	153.4	228	74.5	0.54	0.16
		250.0	336	85.9	0.48	0.02
		358.0	401	43.0	0.48	0.02

Source: Seabridge

The lateral projection to the southwest of the Sulphurets deposit is now known as the Camp Zone. Drilling in the Camp Zone identified what is believe to be a preserved portion of an epithermal gold-silver occurrence associated with the upper parts of the KSM mineral system. Argillic alteration is dominant in this zone, which also contains high gold, silver, lead, and zinc concentrations, particularly within veins and structures. This newly discovered zone has similarities to Pretium Gold's nearby high-grade Brucejack deposit, representing the epithermal or upper portion of a very large gold-copper porphyry system. However, the new Camp Zone appears to be part of the epithermal system preserved in the bottom of Sulphurets Valley. Select Camp Zone drill hole assay results are summarized in Table 9.4.

Table 9.4 Select 2012 Camp Zone Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
C-12-01	405.0	200.0	274.00	74.0	0.850	0.01
C-12-02	363.0	114.0	136.00	22.0	8.940	0.01
C-12-03	310.0	37.2	39.00	1.8	7.820	0.15
		63.0	145.40	82.4	0.493	0.02
		151.2	250.00	98.7	2.110	0.04
		262.0	280.00	18.0	2.020	0.03
C-12-04	546.0	73.9	79.05	5.15	0.890	0.01
		167.0	168.50	1.5	10.200	0.01
		322.5	375.50	53.0	2.470	0.02
		357.5	360.50	3.0	16.300	0.01
C-12-05	144.0	26.5	28.50	2.0	2.480	0.02
C-12-06	600.3	263.5	264.60	1.1	10.250	0.13
		375.0	376.50	1.5	28.700	0.04
		458.0	458.60	0.6	16.850	0.18
		494.3	522.80	28.5	1.930	0.09

table continues…

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
C-12-07	497.3	86.7	129.00	42.3	0.800	0.01
		191.0	272.00	81.0	1.420	0.02
		383.2	398.00	14.8	0.670	0.01
		445.0	481.00	36.0	1.330	0.02
C-12-08	447.0	39.5	52.00	12.5	0.790	0.01
		51.5	52.00	0.5	10.650	0.06
		108.6	129.80	21.2	0.840	0.01
		154.0	184.90	30.9	1.770	0.01
		268.5	284.50	16.0	1.180	0.00
		367.2	381.50	14.3	1.040	0.01
C-12-09	494.7	190.8	193.10	2.3	3.450	0.07
		192.6	193.10	0.5	13.400	0.28
C-12-10	600.3	138.0	144.70	6.7	1.250	0.06
		249.0	257.00	8.0	1.030	0.03
		435.1	452.00	16.9	0.880	0.03
		532.6	556.00	23.4	1.270	0.05
		596.8	600.30	3.5	8.830	0.02
C-12-11	396.0	102.1	102.70	0.6	8.500	0.36
		200.0	206.50	6.5	6.610	0.04
		200.0	201.50	1.5	27.500	0.02
		223.5	241.30	17.8	0.700	0.06
		361.0	370.50	9.5	0.910	0.03
C-12-12	310.0	69.0	108.20	39.2	2.230	0.02
		115.5	121.50	6.0	0.720	0.01

Source: Seabridge

MITCHELL

Two deep drill holes were completed to test a MT low-resistivity anomaly between Mitchell and Iron Cap. The holes were collared above the STF and penetrated through the MTF. In the panel between the STF and MTF intense hydrothermal alteration was encountered indicative of the margins to a porphyry system. The results from the two Mitchell holes are summarized in Table 9.5.

Table 9.5 Select 2012 Mitchell Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
M-12-128	828	9.0	661	652.0	0.16	0.11
M-12-129	903	614.6	903	288.4	0.06	0.10

Source: Seabridge

MCQUILLAN ZONE

Four drill holes were completed at the McQuillan Zone in 2012. These holes targeted a discrete magnetic anomaly down dip of surface alteration and mineralization in the

Sulphurets and McQuillan prospect area. Below the MTF, alteration intensity increases significantly and is characterized by potassium feldspar, biotite, and magnetite with chalcopyrite and pyrite. This alteration mineral assemblage indicates that the McQuillan target zone is within a high-temperature and high-pressure environment. Select assay results for the 2012 McQuillan holes are summarized in Table 9.6.

Table 9.6 Select 2012 McQuillan Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
MQ-12-02	690.00	525.0	654.8	129.8	0.78	0.24
MQ-12-03	921.00	474.0	501.0	27.0	1.68	0.01
MQ-12-04	778.35	6.0	24.0	18.0	2.98	0.07
		364.0	372.0	8.0	0.92	0.21
		656.7	778.4	121.7	0.56	0.02
MQ-12-05	924.30	552.4	835.6	283.2	0.48	0.18
		643.0	679.0	36.0	1.59	0.24
		901.8	905.7	3.9.0	0.56	0.53

Source: Seabridge

9.1.2 INTERPRETATION OF 2012 EXPLORATION DATA

KERR

A high-grade, deep core zone was confirmed and additional drilling is warranted.

SULPHURETS

Low-resistivity anomalies reflect hornfelsed and mineralized sediments, but no clear vectors to a core zone were determined. At the peripheral Ice Field and Camp zones, epithermal styles of gold mineralization, including high-grade veins and lower-grade disseminated, were confirmed and additional drilling is warranted.

MITCHELL

Low-resistivity anomalies reflect hornfelsed and mineralized sediments or altered intrusions, and as the intervals are above the MTF, and well beyond either Iron Cap or Mitchell, they are interpreted to be associated with an undiscovered, blind porphyry system, and additional exploration is warranted.

MCQUILLAN

Results are consistent with those expected in a deep core zone environment, and additional exploration is warranted.

9.2 2013 KSM EXPLORATION PROGRAM

Exploration in 2013 was designed to accomplish three goals. First, a program was completed to confirm and define a porphyry core zone that was envisioned in the deep parts of the Kerr deposit. That program began with wide-spaced drilling on the target, named Deep Kerr, which established the presence of higher-grade copper and gold intervals. Once that concept had been confirmed, drilling was stepped up to complete sufficient holes so that an initial Mineral Resource estimation could be completed.

Second, an assessment was conducted on additional deep core targets within the Property. The potential of this effort became obvious with the success at Kerr. The targets were evaluated principally by drilling and detailed geology. Those preliminary results provided a prioritization for subsequent geophysical surveys and more aggressive drill testing.

The final goal for 2013 was an appraisal of the Camp Zone. Definition of the mineral controls, geological limits of the system, and size potential of the target was required to understand how this target will fit into the Project.

9.2.1 RESULTS OF 2013 EXPLORATION PROGRAM

KERR

Three drill holes completed in 2012 provided a strong indication that the dip projection of the Kerr deposit could provide a large and higher-grade ore zone, as conceived in the exploration model. Geophysical surveys showed a discrete high-amplitude, low-resistivity anomaly below the surface exposure of the Kerr deposit. Part of this anomaly corresponded with the Deep Kerr Zone and was interpreted to indicate abundant sulphide minerals in altered rocks producing a resistivity contrast.

The first drill holes completed in 2013 immediately confirmed the concept and showed that, sitting down dip and in part continuous from the Kerr deposit, the Deep Kerr reported significant mineralized intervals containing total metal values per tonne that are approximately two times KSM's Proven and Probable Reserve average, with some intervals exceeding 1.0% copper, and gold grades as high as 1.7 g/t. Five large core drilling rigs were quickly moved on to the Project in order to expedite the drilling.

The drilling concentrated on about 1,000 m of strike of the projected Deep Kerr target. During the 2013 program, 29 diamond drill holes were attempted, 25 holes were completed through the Deep Kerr target, 2 were lost due to significant hole deviation, and 2 were terminated due to weather and made ready for re-entry. Of the 25 holes completed, 23 encountered significant gold and copper grades over extensive widths. The weighted average of the drill intercepts from the Deep Kerr zone yields a grade of 0.46 g/t gold and 0.71% copper over a width of 220 m. A summary of significant drill hole assay intersections from the 2013 Kerr drilling program are summarized in Table 9.7.

Table 9.7 Select 2013 Kerr Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
K-13-23	1,470.40	643.5	671.50	28.0	0.39	0.57
		880.4	932.40	52.0	0.26	0.42
		1,030.2	1,386.30	356.1	0.39	0.68
K-13-23A	1,368.40	639.1	666.10	27.0	0.35	0.69
		779.3	1,023.50	244.2	0.17	0.42
		932.4	1,007.70	75.3	0.26	0.59
		1,319.3	1,368.40	49.1	0.38	0.58
K-13-23B	1,359.40	643.7	670.30	26.6	0.29	0.50
		725.0	775.00	50.0	0.04	0.40
		803.4	839.00	35.5	0.27	0.64
		953.0	1,249.40	296.4	0.59	0.65
K-13-23C	1,278.40	908.9	1,224.40	315.5	0.45	0.65
K-13-24	1,221.40	610.3	625.00	14.6	0.51	0.65
		703.7	737.00	33.3	0.16	0.45
		807.0	929.70	122.7	0.86	0.85
K-13-24A	-	791.0	1,014.30	223.3	0.31	0.42
		1,080.4	1,139.60	59.2	0.26	0.62
		1,061.8	1,205.00	143.2	0.19	0.40
K-13-24B	1,155.00	534.0	584.80	50.8	0.28	0.41
		691.0	730.10	39.1	0.10	0.40
		762.0	780.10	18.1	1.59	0.86
		789.0	851.30	62.3	0.43	0.69
		881.7	931.00	49.2	0.36	0.70
		1,027.0	1,102.00	75.0	0.41	0.57
K-13-24C	1,284.00	532.9	594.00	62.5	0.33	0.51
		825.0	1,053.00	228.0	0.96	0.72
		1,064.5	1,107.60	43.1	0.37	0.41
K-13-25	1,256.00	928.8	1,022.10	93.3	0.27	0.54
		1,029.3	1,171.00	141.7	0.27	0.69
K-13-25A	1,336.85	883.6	959.20	75.6	0.99	0.42
		967.4	993.00	25.6	0.35	0.53
		1,010.0	1,085.40	75.4	0.23	0.36
		1,158.4	1,334.40	176.0	0.28	0.62
K-13-25B	1,191.30	878.8	938.80	60.0	0.36	0.60
		880.9	1,106.80	225.9	0.25	0.45
K-13-25C	1,299.70	1,103.0	1,230.00	127.0	0.47	0.75
K-13-28	1,339.80	684.6	749.40	64.8	1.05	0.42
		904.0	1,012.40	108.4	0.60	0.75
K-13-28A	1,407.40	886.4	1,043.40	157.0	0.56	0.50
		1,217.4	1,285.40	68.0	0.24	0.56

table continues…

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
K-13-28B	1,284.40	695.0	740.60	45.6	0.85	0.37
		883.6	1,022.40	138.8	0.43	0.68
K-13-28C	894.40	728.3	756.10	27.8	0.84	0.55
		889.3	892.90	3.5	0.72	3.04
K-13-29	992.50	223.4	257.00	33.5	0.08	0.50
		455.9	486.30	30.4	0.44	1.02
		572.4	810.40	238.0	0.55	0.89
K-13-30	771.00	326.0	645.60	319.6	0.33	0.53
K-13-31	1,113.40	250.7	378.80	128.0	0.24	0.69
		421.9	670.40	248.5	0.39	0.77
		817.4	975.40	158.0	0.37	0.61
K-13-31A	1,200.40	450.4	684.40	229.1	0.55	0.80
		684.4	704.40	20.0	0.23	0.50
		758.4	833.40	75.0	0.19	0.48
		1,105.1	1,143.40	38.2	0.43	0.68
K-13-32	777.40	389.0	464.20	75.2	0.24	0.62
		535.0	654.00	119.0	0.40	0.71
K-13-32A	672.40	215.6	279.00	63.4	0.55	0.42
		449.0	616.00	167.0	0.37	0.63
K-13-34	1,188.40	498.0	1,136.50	636.7	0.43	0.85
K-13-35	1,023.40	449.8	775.30	325.4	0.30	0.70
K-13-36	906.40	406.0	795.75	389.7	0.43	0.69

Source: Seabridge

IRON CAP

Iron Cap was the first new target zone in 2013 to be evaluated for deep higher-grade potential. Iron Cap had been explored since 1991 by previous owners, focusing on surface showings and shallow drilling, they concluded it is the expression of a small epithermal vein system. Ongoing exploration since 2010 by Seabridge determined that the epithermal system was superimposed on the upper portion of a much larger gold-copper porphyry deposit. With success at Kerr and a district-scale deposit theory, the highest priority target for a second magmatic core discovery at KSM became Iron Cap.

During 2013, drill holes confirmed the existing resource model at Iron Cap, down to about 200 m. Below that point, the holes entered volcanic and intrusive rocks as well as chaotic breccia zones with variable intensity of veining and alteration. Discrete intervals containing orthoclase and magnetite alteration, intense stockwork veining, and concentrations of chalcopyrite with minor bornite characteristics of a core zone were encountered. Significant 2013 Iron Cap drill hole assay results are summarized in Table 9.8.

Table 9.8 Select 2013 Iron Cap Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
IC-13-048	1,011.3	187.5	199.3	11.7	0.47	0.24
		243.3	267.3	24.0	0.42	0.34
		346.5	839.8	493.3	0.30	0.30
IC-13-049	1,035.4	9.0	1,032.4	1,023.4	0.77	0.24
IC-13-050	431.5	64.7	110.4	45.7	0.13	0.26
		122.6	202.8	80.2	0.25	0.26
		216.0	230.0	14.0	0.70	0.31
		286.0	324.9	38.9	1.27	0.36
		396.9	425.4	28.5	0.38	0.24
IC-13-051	1,088.3	225.0	490.4	265.4	0.21	0.41
		502.4	750.4	248.0	0.39	0.24
		750.4	1,003.4	253.0	0.90	0.38
IC-13-051A	1,169.3	401.1	503.4	102.3	0.14	0.46
IC-13-052	1,070.6	363.4	506.4	141.9	0.41	0.48
		558.4	598.4	40.0	0.66	0.41

Source: Seabridge

MCQUILLAN ZONE

The McQuillan Zone targets the down dip projection of surface alteration and mineralization in the Sulphurets and McQuillan prospect areas. Alteration intensity generally increases down hole at McQuillan and is characterized by a progression from chlorite and sericite to potassium feldspar, biotite, and magnetite with localized chalcopyrite. Modeling of previous drill holes and downhole geophysical surveys was employed to refine the target.

CAMP ZONE

Four additional drill holes were completed in the Camp Zone in 2013. These holes were designed to evaluate alternative orientations to the structural controls on this target area. Results indicated that structures are oriented northwest-southeast and control the distribution of argillic alteration and gold concentrations. Significant 2013 Camp Zone drill hole assay results are summarized in Table 9.9.

Table 9.9 Select 2013 Camp Zone Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
C-13-14	402.0	7.8	402.0	394.1	0.84	0.05
C-13-15	533.7	180.0	207.0	27.0	0.82	0.01
		241.0	276.0	35.0	0.93	0.01
		369.0	378.0	9.0	0.79	0.02
C-13-16	444.0	47.0	48.0	1.0	18.10	0.12
		95.0	112.0	17.0	0.96	0.02
		156.0	186.0	30.0	2.06	0.03
		377.0	392.0	14.9	0.93	0.01
C-13-17	441.0	82.8	118.6	35.8	0.73	0.02
		278.0	321.0	43.0	1.36	0.04
		333.0	344.9	11.9	0.74	0.02
		404.0	414.0	10.0	0.76	0.01

Source: Seabridge

9.2.2 INTERPRETATION OF 2013 EXPLORATION DATA

KERR

Within Deep Kerr, several intervals of bornite-bearing (Cu_5FeS_4) rocks were intersected, indicating higher-temperature ore forming processes were being encountered in the drilling. As the drilling progressed, the Deep Kerr Zone became recognizable as:

- a wide continuous alteration zone characterized by anhydrite, potassium feldspar, and magnetite as minerals
- abundant chalcopyrite and locally bornite, with an observable decrease in pyrite content
- an increase in the abundance of quartz veins with copper minerals, both internal to and at the margins of the veins.

IRON CAP

Evidence strongly suggests that the Iron Cap deposit sits above and is displaced to the south-southeast of a near-magmatic high-grade core zone. Additional work was undertaken during the winter season to refine the target, with the plan to aggressively drill the Iron Cap core zone in 2014.

MCQUILLAN ZONE

The hole completed in 2013 shows a homogenous alteration pattern down hole, indicating it could be oblique to the copper-gold target zone.

CAMP ZONE

Although results indicate potential to expand the Camp Zone, the target does not appear to match the potential for discovery of additional deep core zones, and it was therefore relegated to a lower priority.

9.3 2014 KSM EXPLORATION PROGRAM

The 2014 exploration program was designed to accomplish three goals. First, the initial focus was to expand on the strike and dip potential of the Deep Kerr deposit. Drilling in 2013 did not define the limits of the Deep Kerr deposit. In an effort to identify those limits, the program intended to step out to the north and at depth to define the scale of the deposit.

Second, working off results in 2013, exploration drilling was expanded on the Lower Iron Cap Zone to test continuity and extent of another potential core zone target. Historical drilling on the Iron Cap deposit indicates plunge continuity to the northwest. Using the understanding developed from Deep Kerr, a program was developed to extend the Iron Cap deposit down plunge to the northwest where a core zone target was postulated.

The final goal for 2014 was to complete an appraisal of two additional core zone targets (which subsequently became three). Integrating geophysical surveys across the Property with the understanding from Deep Kerr, suggested additional untested higher-grade target zones remained to be discovered. The aim in this program was to determine which of these targets held the greatest potential for additional discoveries.

9.3.1 RESULTS OF 2014 EXPLORATION PROGRAM

DEEP KERR ZONE

Work on the Deep Kerr Zone in 2014 began by focusing on confirmation of the Mineral Resource block model and then the extension of the deposit to the north and down dip. Results from the 2014 drilling campaign are being integrated into the Mineral Resource model, which will permit expansions of the 2013 Inferred Mineral Resource estimate.

Two holes (K-14-25D and 28C) were drilled into the existing resource to evaluate the performance of the model by determining how well the new data matched up against the model's predicted block grades. In order to expedite this work, daughter holes were completed from two widely-spaced parent drill holes that were started in 2013. In each case the results showed mineralized intervals consistent with those predicted by the model; there was little difference in the bulk grade of the mineralized interval, with individual copper grades ranging from -12% to +30%.

The north strike projection of the Deep Kerr deposit was a primary target at the end of 2013. The northernmost drill holes in the 2013 program intersected well mineralized intrusive rocks. Three additional sections were drilled in 2014 at 140 m intervals, stepping north from the 2013 data. Mineralized zones consistent with the Deep Kerr deposit model were encountered in the first two cross sectional step-outs (holes K-14-39, 43, 44 and 48), 280 m north of previous drilling. On the northern most section (holes K-

14-41 and 41A), a large interval of post mineral intrusive rock was intersected. The projection of the high-grade Deep Kerr intervals was coincident with fault structures.

Three drill holes (K-14-34A, 40 and 45) were targeted to provide mineralogical zoning indicators and extend the depth projection of the Deep Kerr Zone. Holes K-14-34A and 45 were set up to drill down the interpreted Deep Kerr Zone, and encountered long sections of the mineralized zone; however, this orientation was difficult to maintain and technical limitations terminated the holes before reaching the limits of the deposit. These two holes therefore bottomed in mineralization. Hole K-14-40 was drilled perpendicular to the zone.

In 2013, the south limit of the Deep Kerr deposit was provisionally established at the southernmost drill hole (K-13-26) in the zone at that time. As the 2014 program progressed, it became clear the southern boundary was arbitrary. Two drill holes were completed to confirm a southern extension—one hole (K-14-42) at the southern limit of the 2013 Mineral Resource model and one hole (K-14-46) 550 m beyond the 2013 model. Significant 2014 drill hole assay results are summarized in Table 9.10.

Table 9.10 Select 2014 Kerr Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
K-14-25D	1,515.4	910.4	1,011.4	101.0	0.29	0.37
		1,025.3	1,133.4	108.1	0.21	0.35
		1,300.8	1,486.4	185.6	0.18	0.47
K-14-28C	1,304.5	900.0	1,257.4	357.4	0.50	0.63
K-14-34A	1,611.4	450.0	806.4	356.4	0.19	0.62
		871.4	1,608.4	736.5	0.36	0.59
K-14-39	1,272.4	508.0	694.4	186.4	0.19	0.43
		781.4	945.4	164.0	0.34	0.33
		945.4	1,197.4	252.0	0.55	0.69
K-14-40	1,011.4	704.4	926.3	221.9	0.24	0.45
K-14-41	1,080.0	636.2	682.3	46.1	2.35	0.19
		821.4	965.5	144.1	0.58	0.27
K-14-41A	1,098.4	618.0	847.0	229.0	1.12	0.07
K-14-42	951.4	486.8	536.0	49.1	0.28	0.86
		678.4	738.8	60.4	0.28	0.67
K-14-43	1,044.5	512.5	659.5	147.0	0.53	0.71
		689.5	757.5	68.0	0.31	0.31
		879.5	881.5	2.0	63.40	0.23
K-14-44	995.0	529.0	565.9	36.9	0.26	0.60
		580.1	676.8	96.7	0.28	0.39
K-14-45	1,131.3	271.4	368.4	97.0	0.26	0.48
		400.4	1,123.0	722.6	0.36	0.59
K-14-46	789.5	193.0	241.4	47.0	0.27	0.45
K-14-48	1,212.3	971.4	1,161.3	189.9	0.35	0.36

Source: Seabridge

IRON CAP LOWER ZONE

The Iron Cap Lower Zone is interpreted as a northwest plunging, northeast-southwest striking tabular body below the existing reserves. Following the zone down plunge intercepted higher-grade copper-gold consistent with a core zone. A total of 10,429 m in 10 drill holes tested this Lower Iron Cap target in 2014. Work is ongoing to estimate an initial Mineral Resource for the Lower Iron Cap Zone.

The Lower Iron Cap Zone is a series of related, intermediate-composition intrusions, each with extensive and intensive hydrothermal alteration including potassic, phyllic, and silicic alteration, all of which contain copper, gold and silver. Drill holes that targeted the southwestern strike projections of the target zone penetrated numerous intrusive events with variable grade distribution enhanced in the contact zones between these intrusions. The holes drilled along the northern strike projection encountered more consistent intrusive rock with much less grade variability, like hole IC-14-59 with 592.7 m of 1.14 g/t gold and 0.37% copper. Hydrothermal alteration in these holes to the north exhibit vertical continuity over 1,000 m tested so far, indicating significant potential at depth, particularly down the apparent north-northwest plunge.

Drill hole IC-14-61 approaches to within 1,000 m of the proposed trace of the MTT alignment, potentially making the Iron Cap Lower Zone an attractive early development option with lower capital and operating costs than other deposits at KSM that are further from key infrastructure. Significant 2014 Iron Cap drill hole assay results are summarized in Table 9.11.

Table 9.11 Select 2014 Iron Cap Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
IC-14-53	1,329.4	488.4	1,002.4	514.0	0.68	0.30
IC-14-054	1,107.0	322.4	832.5	510.1	0.41	0.28
IC-14-054A	1,050.0	320.0	542.4	222.2	0.33	0.19
		604.4	872.0	267.6	0.39	0.23
		917.4	1,021.0	103.5	0.24	0.25
IC-14-055	624.3	4.2	140.5	136.3	0.33	0.19
		193.6	253.2	58.6	0.37	0.29
		257.5	624.3	366.8	0.59	0.17
IC-14-056	1,095.8	39.0	163.1	124.1	0.38	0.17
		163.1	324.0	160.9	0.21	0.35
		396.4	556.4	160.0	0.45	0.30
		582.4	853.4	271.0	0.25	0.24
		879.4	1,095.8	216.4	0.46	0.16
IC-14-057	927.4	186.0	459.4	273.4	0.46	0.15
		459.4	589.4	130.0	0.31	0.35

table continues…

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
IC-14-058	1,143.3	316.3	500.3	184.0	0.51	0.27
		542.3	807.8	264.8	0.48	0.20
		810.0	1,001.3	191.3	0.30	0.16
		1,001.3	1,143.3	142.0	0.49	0.31
IC-14-059	1,032.0	1.6	145.0	143.4	0.45	0.37
		178.7	771.4	592.7	1.14	0.37
		773.4	1,032.0	258.6	0.39	0.17
IC-14-060	967.1	2.4	359.0	356.6	0.39	0.21
		429.0	525.3	96.3	0.53	0.17
		772.4	836.4	64.0	0.65	0.08
IC-14-061	1,152.4	27.0	55.0	28.0	0.16	0.43
		431.4	794.4	362.5	0.38	0.28
		876.2	1,152.4	276.2	0.46	0.31

Source: Seabridge

OTHER TARGETS

Three additional deep target concepts were drill tested during 2014. Initial results were inconsistent. Along the east side of the Kerr deposit, drilling revealed a thick package of thermally metamorphosed sedimentary rocks with numerous pyrite veins. This package of rocks contains alteration and chemical characteristics interpreted to represent the margin of the intrusive mineral system at Kerr. Drill holes collared to the northwest of the Sulphurets deposit, designed to test for an intrusive source, encountered intensely metamorphosed sedimentary and volcanic rocks with extensive fracture and vein controlled phyllic and potassic alteration. Long intervals of highly anomalous gold concentrations are reported in these drill holes, indicating proximity to the dip projection of the Sulphurets deposit. The interval from 312.5 m to 526.9 m passed through the Raewyn copper-gold zone of the Sulphurets deposit, with grades as expected. One drill hole was also completed into the east side of the McQuillan target. These holes encountered intrusive rocks similar to other mineralized intrusion at KSM; however, the alteration style and intensity is indicative of a post- or inter-mineral rock. Select drill hole assay results from 2014 drilling campaign are summarized in Table 9.12.

Table 9.12 Select 2014 Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
K-14-38	1,058.50	63.0	73.3	10.3	0.87	0.01
S-14-78	1,305.30	6.6	33.0	26.4	0.78	0.08
		464.5	480.8	16.3	0.93	0.01
		569.0	717.0	148.0	1.56	0.04
		725.0	747.2	22.2	0.51	0.03
S-14-79	1,433.05	312.5	526.9	214.4	0.70	0.40
		1,287.4	1,312.3	24.9	1.13	0.03
MQ-14-07	747.40	28.0	38.0	10.0	0.21	0.42

Source: Seabridge

9.3.2 INTERPRETATION OF 2014 EXPLORATION DATA

DEPP KERR ZONE

Results of the model evaluation drilling are within the acceptable ranges for an Inferred Mineral Resource classification, and they are consistent with the expectation for a predictive geological model. North limit testing was inconclusive. It is not known at this time if the Deep Kerr mineralization continues farther north along faults and beyond the post-mineral intrusion. Depth extension tests confirm that the Deep Kerr Zone plunges west-northwest and continues to at least 1,350 m below surface. South limit testing confirmed significant strike potential, but additional drilling is required to extend the Mineral Resource model and establish the grade distribution.

IRON CAP LOWER ZONE

Continuity of the Iron Cap Zone to depth was confirmed. There is a higher grade central zone, but it is has structural control and is confined to a roughly 150 m diameter column. The potential for a bornite bearing core zone related to higher temperatures and pressures deeper within the intrusive has not been ruled out.

OTHER TARGETS

These tests did not confirm any new significant zones. The intervals of anomalous gold mineralization down dip of and beneath Sulphurets suggest possible continuity between here and similar mineralization in the Camp Zone.

9.4 2015 KSM EXPLORATION PROGRAM

The exploration program plan at KSM for 2015 was designed to improve the understanding of block cave targets and support engineering/environmental aspects of development scenarios. The overall objective is to enhance project economics by finding the best 2.3 Bt - the mineable material with the highest margins - which is the optimum Mineral Resource size as determined by scoping studies. The objectives for 2015 were:

- Drilling at Deep Kerr to expand both the length and width of block cave shapes that confine the current Mineral Resource estimate; geological projections of the mineralized zone indicate that the block cave shapes are limited by drill data. Extending the footprint of the block cave shapes could increase the potential mining rate for this higher grade material, thereby generating a significant economic benefit to the Project. This work will evaluate the performance of the Inferred Mineral Resource block model and permit projections of drilling required to advance the Deep Kerr Zone to Mineral Reserves.
- Drilling the plunge projection of the Mitchell high-grade zone to test development scenarios that include exploitation of a larger part of Mitchell as a block cave mine; these holes will also provide additional information for the Lower Mitchell block cave shape included in the 2012 PFS (Tetra Tech 2012).
- Complete the surface evaluation of sources for deleterious elements that may impact infrastructure planning, and define additional potential quarry sites for construction; a high-resolution airborne magnetic survey will also enhance the sub-surface geological model and contribute to exploration targeting and infrastructure area condemnation.

9.4.1 RESULTS OF 2015 EXPLORATION PROGRAM

DEPP KERR

The Kerr deposit is centered on a north-south trending, steep westerly-dipping, tabular intrusive complex that drilling demonstrates has a horizontal extent of 2,400 m and vertical extent of at least 2,200 m. The complex includes an east and west limb that may coalesce near the current surface. The west limb is up to 500 m thick, and the east limb up to 300 m thick. There are several distinct intrusive phases, the earliest of which are fine grained diorites with 5% to 60% quartz-sulphide vein stockworks, and these appear to contribute the majority of metals. Later phases envelope and sometimes invade the earlier phase, and are characterized by coarser textures, less veining, and lower metal contents. The intrusions are hosted by an Early Jurassic sequence of rhythmically bedded siltstones, sandstones, conglomerates, and debris flows that have been altered adjacent to the intrusions but generally contain marginal metal grades.

The holes in the 2015 Deep Kerr program were collared well outside the mineral deposit in order to achieve the deep intersections that test the dip extension. Drill holes were designed to intercept the mineralized target at right angles to the strike of the zone and downhole directional drilling tools were used to steer the holes to target areas. These locations better defined the western limits of the mineralized system, and demonstrate that a north-south trending normal fault places unaltered fine-grained sedimentary rock against the outer weakly mineralized parts of the mineral system. As the drill holes advance to the east, alteration and mineralization increase as a series of potassically-altered intrusions are encountered. Drill hole K-15-49 passed out of the intensely altered and mineralized zone into younger intrusions with lower concentrations of gold and copper. The drill hole was not extended into the eastern high-grade zone encountered in previous shallower drilling because projected depths would have been prohibitive. This eastern zone remains a high-potential target.

Holes K-15-49 and K-15-49A confirmed down-dip extension of the strong mineralization in the west limb intersected by hole K-14-45, an oblique hole that cut 503 m grading 0.40 g/t gold and 0.67% copper. This mineralization occurs mainly in the PAND1 diorite, the early mineralizing phase of the Kerr intrusive complex. It is finer grained, exhibits a high-chalcopyrite to pyrite ratio, has remnants of secondary potassic alteration now mostly overprinted by retrograde chloritic alteration, and a mostly intact stockwork of quartz-sulphide veinlets. Portions of the PAND1 intrusion are overprinted by phyllic alteration characterized by sericitization of mafic minerals, higher pyrite content, and higher gold and copper grades. The later PAND2 diorite is coarser grained, with very few intact quartz veinlets, and a lower metal content. Wall rock sediments are mineralized adjacent to the intrusions, but generally lower grade. Currently, the lithology model lumps moderately mineralized PAND2 dykes, breccias, and sediment intervals in the hanging wall of the PAND1 body into the IBX domain; however, definition drilling is expected to enable resolution of this into finer components. A few meter-scale, late quartz-carbonate-sulphide veins with strong gold and elevated copper, lead, zinc, and aresenic levels cut the intrusions and wall rocks, and indicate penetration of late advanced argillic fluids along steep fractures.

Holes K-15-50 and K-15-50A tested the west limb 200 m on strike to the south of K-15-49 and K-15-49A. Continuity of the mineralized PAND1 intrusion was confirmed, however on this section gold and copper grades are lower. This is attributed to a higher proportion of lower grade PAND2 intrusion, as well as weaker overprinting phyllic alteration. Hole K-15-49B tested the west limb 200 m north of K-15-49, and lower grades were also found to be due to a lower proportion of the PAND1 intrusion. Significant drill hole assay results are summarized in Table 9.13.

Table 9.13 Select 2015 Kerr Drill Hole Assay Results

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
K-15-49	1,755.4	1,272.0	1,755.4	483.4	0.43	0.56
	Incl.	1,466.4	1,716.4	250.0	0.49	0.70
K-15-49A	1,710.4	1,178.3	1,244.1	65.8	0.41	0.36
		1,304.4	1,644.2	339.8	0.53	0.60
	Incl.	1,358.2	1,555.0	196.8	0.69	0.72
K-15-49B	1,731.4	963.5	1,020.1	56.6	0.67	0.12
		1,379.0	1,461.6	82.6	0.43	0.55
		1,534.5	1,668.8	134.4	0.20	0.45
	Incl.	1,574.0	1,627.2	53.2	0.31	0.56
K-15-50	1,764.4	1,430.4	1,764.4	334.0	0.41	0.30
	Incl.	1,433.0	1,598.4	165.4	0.56	0.27
	Incl.	1,659.4	1,713.8	54.4	0.29	0.41
K-15-50A	1,718.5	1,246.5	1,369.5	123.0	0.44	0.30
		1,452.5	1,704.5	252.0	0.38	0.31
	Incl.	1,559.3	1,620.5	61.2	0.63	0.42

Source: Seabridge

Mitchell Zone

In order to drill test the deep projection of the central zone and maintain orientations as close as possible at right angles to the interpreted mineralization trend, the holes were started well above and outside of the Mitchell deposit reserve. Directional drilling techniques were used down the hole to steer the holes to the target areas. The first two holes in the 2015 program confirmed continuity of mineralization in the panel above the MTF, which hosts disseminated and veinlet chalcopyrite in magnetite skarn-style altered sediments and volcanics, a distal component of the Mitchell porphyry system. Intersects up to 192 m wide grading 0.34% copper and 0.14 g/t gold support revisions of models that will enable conversion of waste to ore in Mitchell open pit scenarios that are planned to precede underground block caving.

Below the MTF, where the Mitchell Mineral Reserves and Resources are located, the holes encountered identical sections of altered intrusive rocks that are recognized as host to parts of the Mitchell deposit. The intrusion is pervasively hydrothermally altered and contains abundant stock work quartz veins. Alteration increases systematically down hole, progressing through intense quartz-sericite-pyrite and into chlorite-magnetite-orthoclase alteration. The intervals encountered in holes M-15-130 and 131 pass through several phases of the Mitchell intrusive system, some of which contain gold and copper grades above the Mitchell deposit average. Variable but mostly lower grades were encountered in a brecciated zone with abundant anhydrite, similar to the "Bornite Breccia" intersected several hundred meters higher, but without bornite. This was intersected from 1,232.3 to 1,510 m in M-15-130, 1,357.5 to 1,453.4 m in M-15-131, and 1,214.5 to 1,353.6 m in M-15-131A. The geometry is consistent with the moderately northwest dipping orientation of the bornite breccia, however copper and gold grades tend to be higher along the up-dip contact of the zone. This structure is interpreted as a late feature that controlled flow of hydrothermal fluids with an advanced argillic chemistry characteristic of the late stages of porphyry evolution. Bornite was confined mostly to shallower portions of the structure where cooler conditions favoured precipitation.

Hole M-15-131A intersected a distinct medium to coarse grained, sub-porphyritic monzodiorite from 1,376.8 to 1,655 m with grades below the Mitchell average. This intrusion is interpreted to be a later phase, with primary potassium-feldspar phenocrysts, an alteration mineral assemblage dominated by secondary potassium-feldspar, magnetite, epidote and traces of actinolite, and a poor development of stockwork quartz veins and sulfides. This low-grade intrusion has been intersected in several other holes, but over much narrower widths suggesting the thickness in this hole reflects a local thickening or flexure and does not reflect the true volume of displaced higher grade.

All three holes confirmed the presence of a roughly 50 m thick, banded, mylonitic shear zone referred to as the BSF that may offset the base of the Mitchell deposit. Prior to the 2015 drilling program, only one drill hole (M-08-962) intersected the BSF. The zone dips to the northwest and appears to be oriented parallel with the MTF.

Table 9.14 summarizes the composited assay results for significant drill hole intersections from Mitchell Zone drilling in 2015. In drill hole M-15-130, the MTF is located at 601 m and in M-15-131 it is at 691 m. M-15-131A was wedged off of M-15-131 at 622.6 m.

Table 9.14 Select 2015 Mitchell Composited Drill Hole Assays

Hole ID	Total Depth (m)	From (m)	To (m)	Thickness (m)	Au Grade (g/t)	Cu Grade (%)
M-15-130	1,581.0	334.0	441.5	107.5	0.11	0.39
		601.7	638.4	36.6	0.65	0.05
		1,034.4	1,076.4	42.0	0.59	0.12
		1,207.4	1,381.8	174.4	0.55	0.28
	Incl.	1,217.4	1,296.3	78.9	0.73	0.40
M-15-131	1,674.0	253.0	444.5	191.5	0.14	0.34
		1,190.5	1,357.5	167.0	0.81	0.25
	Incl.	1,248.5	1,357.5	109.0	0.96	0.32
		1,449.5	1,505.0	55.5	0.42	0.24
M-15-131A	1,760.5	1,043.5	1,303.5	260.0	0.53	0.18
	Incl.	1,043.5	1,070.5	27.0	0.80	0.17
	Incl.	1,108.5	1,214.5	106.0	0.66	0.18
	Incl.	1,250.5	1,299.9	49.4	0.48	0.29
		1,379.5	1,579.5	200.0	0.43	0.16
	Incl.	1,433.5	1,503.5	70.0	0.51	0.19

Source: Seabridge

OTHER EXPLORATION

A high-resolution aeromagnetic survey was contracted to Precision Geosurveys Inc. East-west oriented lines spaced at 100 m were flown to cover the majority of the Project area. The objective was to provide guidance for geological interpretation of the sub-surface for exploration modeling and targeting, as well as geological modeling under infrastructure areas. This was supported by selective geological mapping.

9.4.2 INTERPRETATION OF 2015 EXPLORATION DATA

KERR ZONE

The holes drilled in 2015 confirmed a high degree of continuity of mineralization over very considerable distances. The west leg was sufficiently drilled and extended down dip to enable an expansion of Mineral Resources in the Inferred category, and this is discussed in Section 14.0.

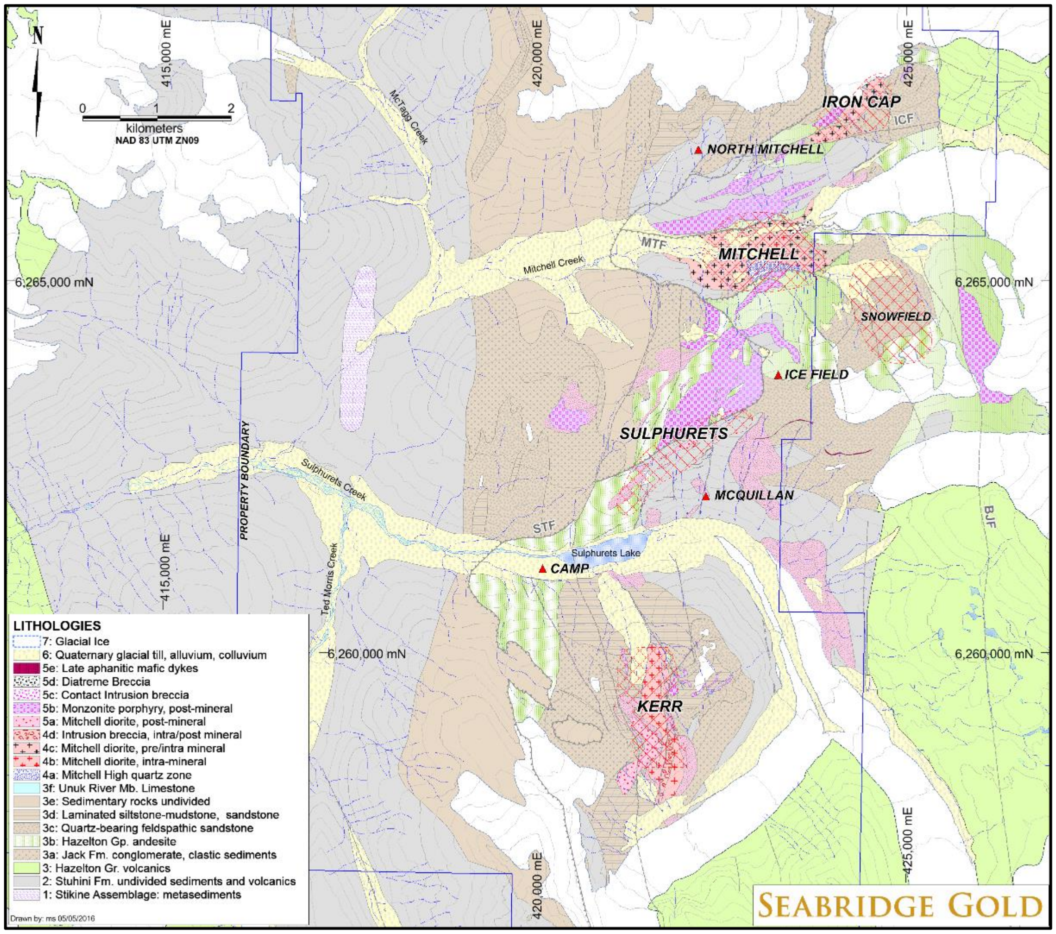
MITCHELL ZONE

Extension of the Mitchell deposit down dip by an additional 400 m was confirmed on two sections, 200 m apart; however, insufficient drilling was completed to define significant additional resources in this area. The system may be narrowing and grades slowly diminishing, at least in the area tested. Displacement of portions of the deposit has likely occurred along structures such as the bornite breccia and Basal Shear Zone.

OTHER EXPLORATION

An interim revision of the KSM Property geology map was completed and is shown in Figure 9.1. Further geological mapping is planned for infrastructure areas in 2016.

Figure 9.1 KSM Property Geology Map



9.5 STATEMENT REGARDING NATURE OF INVESTIGATIONS

All of the exploration activities that were conducted at KSM between 2012 and 2015 as described above were either directly carried out by Seabridge's geology staff or directly supervised by Seabridge personnel.

10.0 DRILLING

10.1 INTRODUCTION

Drilling methods, procedures, extent of drilling, and relevant results for the Project have been described in various NI 43-101 Technical Reports prepared by the Qualified Person responsible for this section of this Technical Report (Lechner 2007; 2008a; 2008b; 2009; 2010; 2011; 2012; and 2014).

The majority of KSM drilling information stored in the end-of-year 2015 KSM database was collected by Seabridge (83%). Seabridge has conducted annual drilling campaigns at KSM beginning in 2006. The remaining 17% of the drilling data were collected by Placer Dome (about 9%) and Falconbridge/Noranda (about 2%), with the remainder collected by six other companies (6%). The 2005 Falconbridge drill campaign was conducted as a joint venture with Seabridge. A summary of KSM drill hole data in the end-of-year 2015 database, organized by company, is shown in Table 10.1. The majority of the 647 core holes shown in Table 10.1 were used to estimate Mineral Resources disclosed in this Technical Report, but some of the data tested several non-resource targets in the KSM Property. The companies listed in Table 10.1 have been arranged in approximate chronological order starting with Esso Minerals in the 1960s. Minor core drilling was completed at KSM by several companies in the early 1980s, but ramped up significantly in the late 1980s and early 1990s by Placer Dome. Seabridge systematically added to the KSM drill hole data after their entry into the district in 2000, with annual drill campaigns beginning in 2006.

Table 10.1 End-of-year 2015 KSM Drill Hole Summary

	Core Holes Drilled			Assayed Meters			
Company	No. Holes	Drilled Meters	Percentage of Total (%)	Au	Cu	Ag	Mo
Esso	20	3,536	1.4	3,331	2,286	1,408	1,363
Granduc	6	1,016	0.4	319	563	0	437
Brinco	3	190	0.1	182	40	182	0
Western Canadian	36	5,325	2.1	4,739	4,739	4,739	0
Newhawk	13	2,069	0.8	1,913	1,789	119	0
Sulphurets Gold	18	4,197	1.6	3,811	3,811	1,566	0
Placer Dome	105	21,982	8.6	20,930	20,930	1,337	0
Falconbridge	16	4,092	1.6	4,015	4,015	4,015	4,015
Seabridge	430	212,009	83.3	190,360	190,310	190,512	190,360
Grand Total	**647**	**254,416**	**100.0**	**229,600**	**228,481**	**203,879**	**196,175**

Seabridge contracted Boart Longyear to complete the initial 2006 core drilling program. Since then, Hy-Tech Drilling Ltd. from Smithers BC has completed all of Seabridge's Mineral Resource definition core drilling at KSM using their own manufactured Tech-5000 Fly Rigs. Drilling was completed using either HQ or NQ tools. Helicopter support for Seabridge's KSM exploration programs has been provided by Lakelse Air Ltd. from Terrace, BC since 2007. This long standing relationship with local drilling and air support contractors has allowed for a continually growing understanding about local drilling conditions.

A description of drilling methods completed at KSM since the last Technical Report (Lechner 2014) will be discussed in this section, along with various summaries of the current drill hole data associated with each mineralized area.

10.2 2013 TO 2015 DRILLING CAMPAIGNS: TYPE AND EXTENT

Seabridge completed helicopter supported diamond drilling programs at the Project from 2013 through 2015. Hy-Tech Drilling Ltd. drilled all of the diamond core holes using Tech-5000 Fly Rigs using NQ and HQ tools. The drilling operations were conducted from the Sulphurets Creek camp which is located northwest of the Kerr deposit.

The 2013 program primarily focused on intersecting deep mineralization at the Kerr deposit (29 holes) and the Iron Cap deposit (6 holes). Additional drilling was completed at the Camp and McQuillan targets.

The 2014 program focused on exploring the down-dip extension of mineralization below the Kerr Zone (16 holes) and Iron Cap Zone (10 holes). Additional drilling was completed at the Sulphurets Zone (9 holes) and McQuillan Zone (2 holes).

In 2015, Seabridge drilled nine core holes totaling nearly 14,000 m. The majority of that meterage was designed to extend the known limits of mineralization beneath the Kerr deposit (6 holes). Three holes (about 5,000 m) were drilled to test the down-dip projection of mineralization at the Mitchell deposit. Table 10.2 summarizes the core drilling programs completed at KSM in 2014 and 2015. Table 10.3 summarizes the drill hole database by Mineral Resource area and by company collecting data for Kerr, Sulphurets, Mitchell, and Iron Cap. The data in Table 10.3 show that Seabridge has collected 72%, 74%, 97%, and 93% of the drilled meterage for the Kerr, Sulphurets, Mitchell, and Iron Cap deposits, respectively.

Table 10.2 2013 to 2015 KSM Drill Hole Summary

Deposit	No. Holes	Drilled Meters	Assayed Meters	
			Au	Cu
2013 Program				
Kerr	29	31,739	23,238	23,236
Iron Cap	6	5,806	5,382	5,382
2013 Subtotal	**35**	**37,545**	**28,620**	**28,618**
2014 Program				
Kerr	16	18,027	15,518	15,518
Sulphurets	2	2,738	2,724	2,724
Iron Cap	10	10,429	10,045	10,045
McQuillan	1	747	743	743
2014 Subtotal	**29**	**31,941**	**29,030**	**29,030**
2015 Program				
Kerr	6	8,741	6,201	6,201
Mitchell	3	5,016	4,349	4,349
2015 Subtotal	**9**	**13,756**	**10,550**	**10,550**
2013 to 2015 Total	**73**	**83,243**	**68,200**	**68,198**

Table 10.3 KSM Drill Hole Summary by Area and Company

Company	Core Holes Drilled			Assayed Meters			
	No. Holes	Drilled Meters	Percentage of Total (%)	Au	Cu	Ag	Mo
Kerr Deposit							
Brinco	3	190	0	182	40	182	0
Western Canadian	36	5,325	6	4,739	4,739	4,739	0
Newhawk	2	115	0	110	110	110	0
Sulphurets Gold	18	4,197	4	3,811	3,811	1,566	0
Placer Dome	82	16,404	17	15,504	15,504	1,337	0
Seabridge	77	67,580	72	53,790	53,788	53,790	53,790
Kerr Deposit Total	**218**	**93,811**	**100**	**78,136**	**77,992**	**61,725**	**53,790**
Sulphurets Deposit							
Esso	6	1,016	2	319	563	0	437
Brinco	14	2,275	5	2,100	1,509	177	705
Newhawk	7	1,306	3	1,157	1,036	6	0
Placer Dome	23	5,577	13	5,426	5,426	0	0
Falconbridge	4	984	2	965	965	965	965
Seabridge	87	31,996	74	31,208	31,206	31,208	31,208
Sulphurets Deposit Total	**141**	**43,155**	**100**	**41,174**	**40,704**	**32,355**	**33,314**
Mitchell Deposit							
Esso	1	210	0	204	204	204	204
Newhawk	4	647	1	646	643	3	0
Falconbridge	4	1,197	2	1,180	1,180	1,180	1,180
Seabridge	182	65,502	97	59,870	59,823	60,022	59,870
Mitchell Deposit Total	**191**	**67,556**	**100**	**61,900**	**61,850**	**61,409**	**61,254**
Iron Cap Deposit							
Esso	5	1,051	3	1,028	573	1,028	454
Falconbridge	5	1,247	4	1,229	1,229	1,229	1,229
Seabridge	59	32,677	93	31,680	31,680	31,680	31,680
Iron Cap Total	**69**	**34,975**	**100**	**33,937**	**33,482**	**33,937**	**33,363**

Figure 10.1 is a drill hole location map for the entire KSM district, showing pre-2013 and post-2013 drill holes for the four recognized Mineral Resource areas. Detailed drill hole location maps are presented in Figure 10.2 through Figure 10.5 for the Kerr, Sulphurets, Mitchell, and Iron Cap deposits, respectively. Figure 10.2 to Figure 10.5 also show the outline of conceptual resource pits and resource block cave that define the Mineral Resources for the Project.

Figure 10.1 KSM Drill Hole Locations

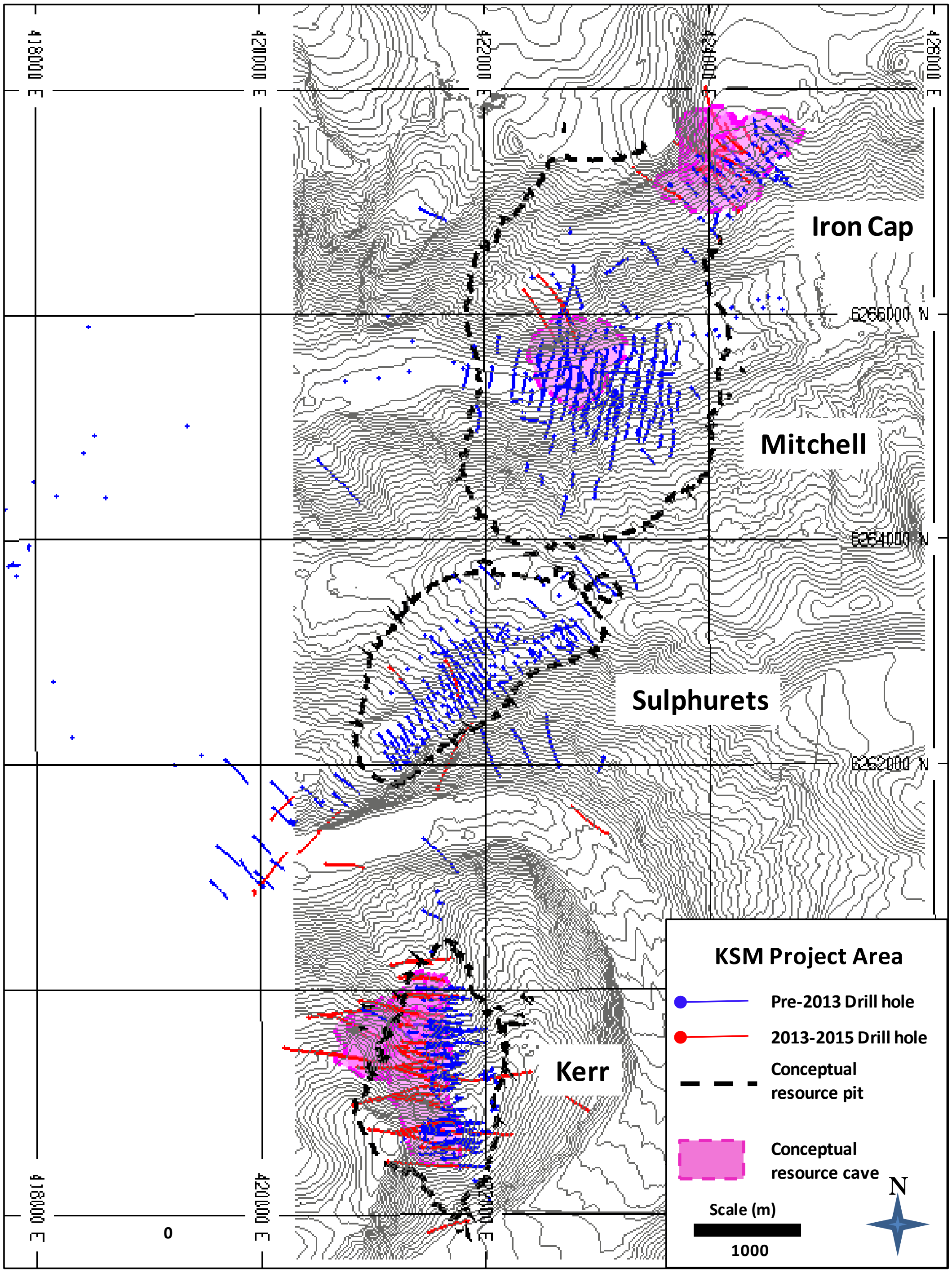
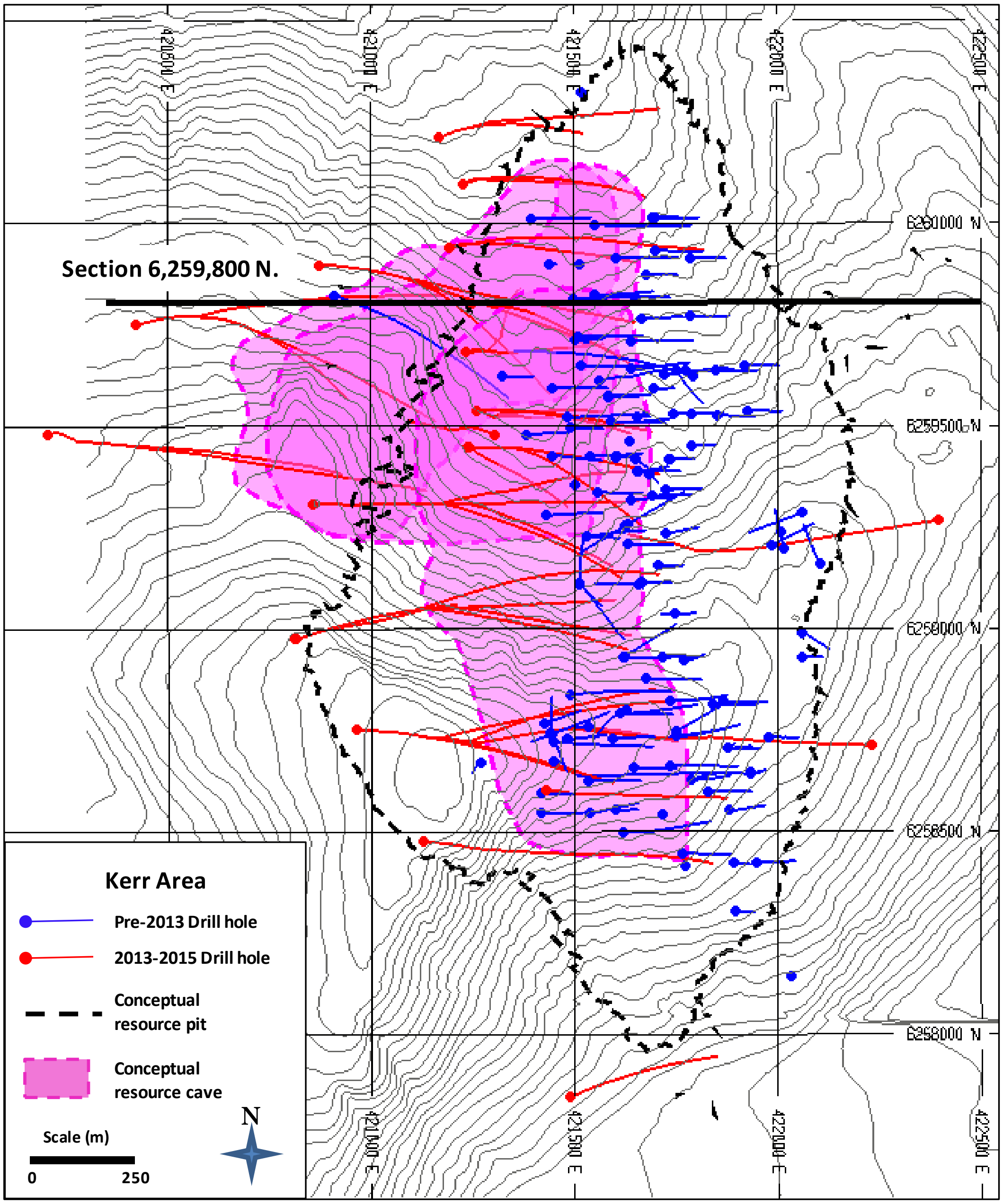
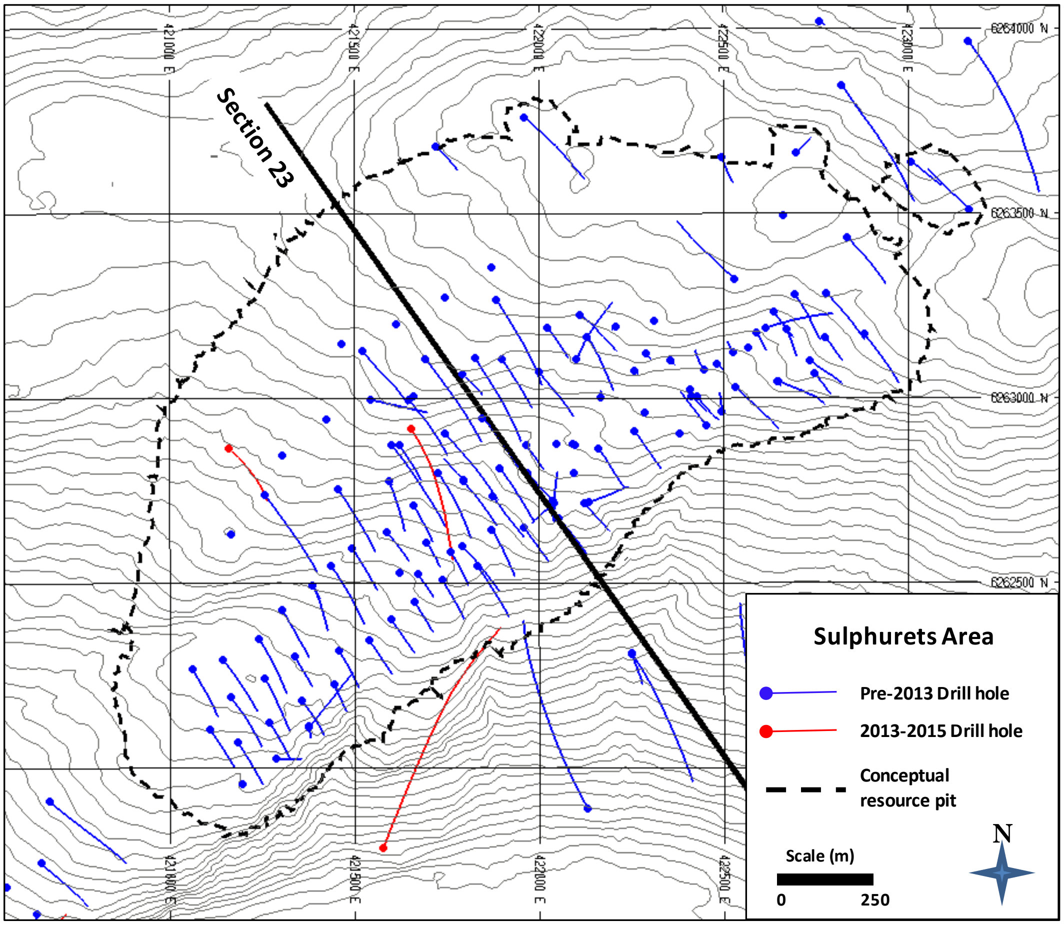


Figure 10.2 Drill Hole Locations – Kerr Deposit



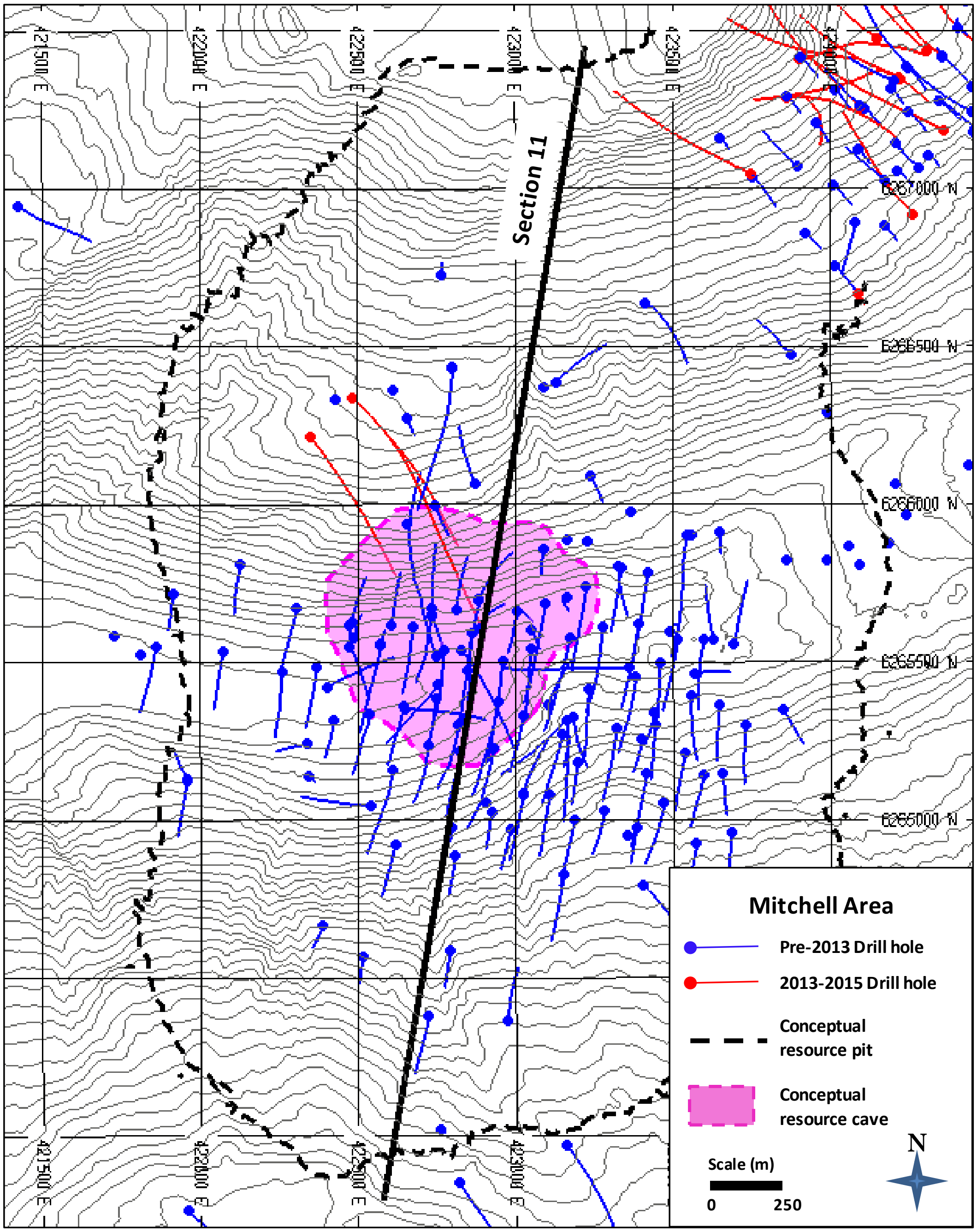
Drilling at the Kerr deposit has identified a mineralized area measuring roughly 1,800 m north-south by 800 m east-west, and 2,000 m vertically. The drill hole spacing in the upper open pit resource area is approximately 50 to 75 m. Drill hole spacing through the block cave resource, which has been classified as nearly all Inferred material, ranges between 100 to 200 m.

Figure 10.3 Drill Hole Locations – Sulphurets Deposit



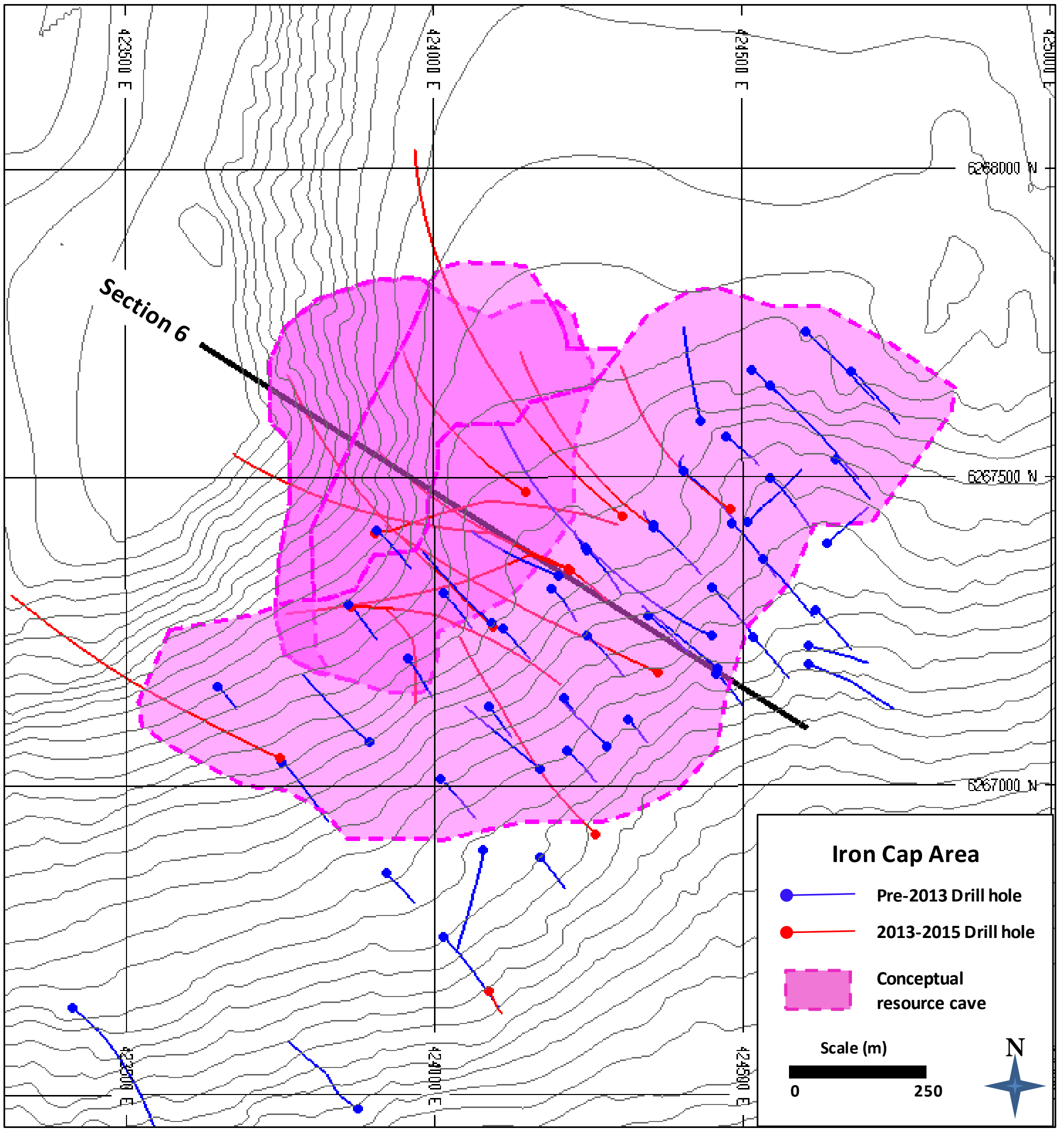
Drilling at the Sulphurets deposit has identified a mineralized area measuring roughly 2,000 m northeast-southwest by 600 m northwest-southeast, and 500 m vertically. The drill hole spacing in the open pit resource area ranges between 50 to 75 m.

Figure 10.4 Drill Hole Locations – Mitchell Deposit



Drilling at the Mitchell deposit has identified a mineralized area measuring roughly 1,600 m east-west by 1,000 m north-south, and 1,000 m vertically. The drill hole spacing in the upper open pit resource area is approximately 75 to 100 m. Drill hole spacing through the block cave resource, which has been classified predominantly as Inferred material, ranges between 100 to 200 m.

Figure 10.5 Drill Hole Locations – Iron Cap Deposit



Drilling at the Iron Cap deposit has identified a mineralized area measuring roughly 700 m northeast-southwest, by 600 m northwest-southeast, and 1,000 m vertically. The drill hole spacing in the upper block cave resource shapes ranges between 70 to 75 m. Drill hole spacing through the lower block cave resource, which has been classified predominantly as Inferred material, ranges between 100 to 200 m.

10.3 Drilling Procedures: 2013 to 2015 Campaigns

Hy-Tech Drilling Ltd. drilled all of the diamond core holes using Tech-5000 Fly Rigs utilizing HQ, NQ, and AQ tools. Helicopter support was provided by Lakelse Air Ltd. using Eurocopter A-Star machines. Similar to previous campaigns completed at KSM, the drilling operations were conducted from the Sulphurets Creek camp, which is located northwest of the Deep Kerr deposit.

Seabridge used directional drilling methods for a portion of their program. Tech Directional Services Inc. from Ontario, Canada were contracted to provide directional drilling services using DeviDrill equipment. DeviDrill is a steerable wireline core barrel that allows a “daughter” hole to be wedged off of a “mother” hole and vectored towards a target zone with reasonable accuracy. Small diameter core (AQ) was retrieved during the crucial turn away from the mother hole so minimal data was lost. Bearing and inclination data were collected using a miniature electronic single-shot survey tool (DeviTool PeeWee) that is designed to pass through the DeviDrill bit. Information regarding this drilling method can be found at http://www.techdirectional.com/. Drill holes with a letter designation after the hole number represent wedged drill holes that utilized the directional drilling method. Of the 29 drill holes completed in 2013, 15 were wedged off from mother holes at depths ranging from 180 to 750 m. Total “saved” drilling was 7,907 m had every wedged hole been started from surface. Total metres drilled in wedged holes was 10,598.

Drill core was placed into wooden core boxes by the drill contractor at the rig and delivered twice daily by helicopter from the rigs to Seabridge's Sulphurets Creek camp. An inventory of the core was completed by Seabridge geologists, which included a review of core condition, a check of run block depths, and generation of a quick down-hole lithologic log.

The drill core was typically scanned for various base metal quantities using a Niton hand held x-ray fluorescence (XRF) analyzer prior to cleaning the core. Seabridge has determined that a factor of 2.0 to 2.2 times the Niton copper reading closely approximates the assayed copper content percentage. The Niton readings are primarily used to alert/train the logging geologist about apparent mineralized intersections. That data was written on the core with wax markers and are visible on core photos. Magnetic susceptibility was also recorded for each drill hole using a hand held device. The mag readings were exported out of the device as .csv files, but the data is not currently being used.

After cleaning, the core was logged for lithology, alteration, structure, and oxidation state onto paper logs by Seabridge geologists. That information was later entered into Microsoft Excel® spreadsheets by each logger. Separate paper logs were used to capture geotechnical information like core recovery, rock quality designation (RQD), and fracture count. The geotechnical logs are based on data between core run blocks.

Assay samples were laid out by the logging geologist. Samples were primarily laid out in 2-m lengths, but were broken at distinct lithologic, alteration, or mineralization contacts. Likewise, samples were broken at core diameter changes. The sample data were hand recorded onto paper logs with hole name, from depth, to depth, and various sulpide mineral estimates.

Pieces of drill core (14 to 20 cm long) were marked for bulk density determination about every 100 m down-the-hole by the logging geologist by labeling the wooden core box with

"SG". Those small pieces were not cut for assay sample. Periodically a contract employee weighed the core pieces in air and water so that a bulk density could be calculated.

Prior to sawing the drill core it was photographed. Two close-up photographs were taken for each core box and the two photos were "stitched" together to create a detailed photograph. After all logging procedures were complete the core boxes were moved to the core cutting facilities located adjacent to the core logging tents.

10.4 Summary and Interpretation of 2013 to 2015 Drilling Campaigns

10.4.1 Kerr Deposit

Based on encouraging results from three drill holes completed in 2012, Seabridge designed a drilling program for 2013 that concentrated on wide-spaced testing of the down dip projection of mineralization below the previously outlined Kerr deposit. Geophysical surveys showed a discrete high-amplitude, low-resistivity anomaly below the surface exposure of the Kerr deposit. Part of this anomaly corresponded with the Deep Kerr Zone and was interpreted to indicate abundant sulphide minerals in altered rocks producing a resistivity contrast.

The first drill holes in 2013 immediately confirmed that there was a strong possibility of having continuously mineralized material from the surface to a depth of more than 1,000 m. Assays from the 2013 drill holes showed significant mineralized intervals containing total metal values per tonne that were approximately two times higher than KSM's Proven and Probable Mineral Reserve grade, with some intervals exceeding 1.0% copper, and gold grades as high as 1.7 g/t.

The 2013 drilling concentrated on about 1,000 m of strike of the projected Deep Kerr target. During the 2013 program, 29 diamond drill holes were attempted, 25 holes were completed through the Deep Kerr target, 2 were lost due to significant hole deviation, and 2 were terminated due to weather and made ready for re-entry. Of the 25 holes completed, 23 encountered significant gold and copper grades over extensive widths. The weighted average of the drill intercepts from the Deep Kerr Zone yields a grade of 0.46 g/t gold and 0.71% copper, over a width of 220 m.

Several intervals from the 2013 drilling program contained bornite-bearing (Cu_5FeS_4) material indicating higher temperature ore forming processes were present. The 2013 drilling results demonstrated several characteristics about the Deep Kerr mineralization:

- a wide continuous alteration zone characterized by anhydrite, potassium feldspar, and magnetite as minerals
- abundant chalcopyrite and locally bornite, with an observable decrease in pyrite content
- an increase in the abundance of quartz veins with copper minerals both internal to and at the margins of the veins.

The 2014 Kerr drilling campaign was designed to expand on the known strike and dip extents of the Deep Kerr deposit by completing some step-out holes. Sixteen core holes totaling approximately 18,000 m were completed (about 15,500 m of assayed core hole data).

Depth extension tests associated with the 2014 drilling program confirmed that the Deep Kerr Zone plunges west-northwest and continues to at least 1,350 m below the topographic surface. South limit testing confirmed significant strike potential but additional drilling would be required to expand the Mineral Resource.

The 2015 Deep Kerr drilling program was designed to improve the understanding of potential block cave targets and to begin addressing various engineering and environmental aspects associated with possible development scenarios. Six drill holes were completed in 2015 in an effort to expand the recognized strike length and width of mineralization that might be exploited by block cave mining methods.

After integrating results from the 2015 drilling program with all prior data, the following observations have been made:

- The Kerr deposit is centered on a north-south trending, steep westerly dipping tabular intrusive complex.
- Drilling has demonstrated that the mineralized system has horizontal and vertical extents of 2,400 m and 2,200 m, respectively.
- Deep mineralization is characterized by two north-south west dipping limbs that appear to coalesce near the surface. The west limb is about 500 m thick and the east limb is about 300 m thick.
- There are several intrusive phases present, with the earliest phase being a fine grained diorite with 5% to 60% quartz-sulphide vein stockworks. This intrusive phase appears to contain the majority of metal. Later, coarser grained intrusive phases envelope and sometimes invade the earlier phase.

10.4.2 SULPHURETS DEPOSIT

No holes were drilled into the Sulphurets deposit in 2013 or 2015. In 2014, two holes were collared to the northwest of the Sulphurets deposit and were designed to test for an intrusive source associated with mineralization, but instead encountered intensely metamorphosed sedimentary and volcanic rocks with extensive fracture and vein controlled phyllic and potassic alteration. Long intervals of highly-anomalous gold concentrations were encountered in these drill holes that correspond to the dip projection of the Sulphurets deposit. For example, an interval from about 312 to 527 m in drill hole S-14-79 passed through the Raewyn copper-gold zone of the Sulphurets deposit confirming previously sampled gold and copper grades.

10.4.3 MITCHELL DEPOSIT

No holes were drilled into the Mitchell deposit in 2013 or 2014. Three core holes were completed in 2015 totaling about 5,000 m. These holes were designed to test the plunge projection of the Mitchell high-grade zone.

In order to drill test the deep projection of the central zone and maintain orientations as close as possible at right angles to the interpreted mineralization trend, the holes were started well above and outside of the 2012 Mitchell reserve volume. Directional drilling techniques were used down-the-hole to steer the holes to the target areas. The first two holes in the 2015 program confirmed continuity of mineralization in a panel above the MTF that hosts disseminated and veinlet chalcopyrite in magnetite skarn-style altered sediments and volcanics, which is believed to represent a distal component of the Mitchell porphyry system. Drill hole intersections up to 192 m wide graded 0.34% copper and 0.14 g/t gold.

Below the MTF, where the majority of the Mitchell Mineral Reserves and Resources are located, the holes encountered identical sections of altered intrusive rocks that are recognized as important host rocks for parts of the Mitchell deposit. The intrusion was found to be pervasively hydrothermally altered and containing abundant stockwork quartz veins. Alteration increased systematically down-hole, progressing through intense quartz-sericite-pyrite and into chlorite-magnetite-orthoclase alteration. The intervals encountered in holes M-15-130 and M-15-131 passed through several phases of the Mitchell intrusive system, some of which contain gold and copper grades above the Mitchell deposit average. Variable but mostly lower grades were encountered in a brecciated zone with abundant anhydrite, similar to the “bornite breccia” intersected several hundred meters higher in older drilling, but no bornite was intersected in the 2015 holes. The geometry of the breccia zone in the 2015 holes is consistent with the northwest dipping orientation of the recognized bornite breccia; however, copper and gold grades tend to be higher along the up-dip contact of the zone. This structure is interpreted as a late feature that controlled flow of hydrothermal fluids with an advanced argillic chemistry characteristic of the late stages of porphyry evolution. It is believed that bornite was confined mostly to shallower portions of the structure where cooler conditions favoured precipitation.

Hole M-15-131A intersected a distinct medium to coarse grained, sub-porphyritic monzodiorite from 1,376.8 to 1,655.0 m, with grades below the Mitchell average. This intrusion is interpreted to be a later phase, with primary potassium-feldspar phenocrysts, an alteration mineral assemblage dominated by secondary potassium-feldspar, magnetite, epidote and traces of actinolite, and a poor development of stockwork quartz veins and sulphides. This low-grade intrusion has been intersected in several other holes, but over much narrower widths suggesting the thickness in this hole reflects a local thickening or flexure, and does not reflect the true volume of displaced higher grade.

All three holes confirmed the presence of a roughly 50 m thick, banded, mylonitic shear zone that may offset the base of the Mitchell deposit (BSF), seen previously only in hole M-08-062. The zone dips to the northwest and appears to parallel the MTF.

10.4.4 IRON CAP DEPOSIT

Early drilling at Iron Cap was focused on near surface mineralization and relatively shallow drilling demonstrated the presence of a small epithermal vein system. Exploration by Seabridge beginning in 2010 showed that the epithermal system appears to be superimposed on the upper portion of a much larger gold-copper porphyry deposit. With prior success at intersecting deep mineralization at Kerr a decision was made to test for a deeper magmatic core beneath the currently recognized Iron Cap deposit.

During 2013 Seabridge drilled six core holes totaling about 5,800 m. These holes confirmed the 2012 Iron Cap grade models to a depth of about 200 m. Below that elevation, the 2013 drill holes intersected volcanic and intrusive rocks as well as chaotic breccia zones with variable intensity of veining and alteration. Mineralization was characterized by discrete intervals containing orthoclase and magnetite alteration, intense stockwork veining and concentrations of chalcopyrite with minor bornite, characteristic of a core zone. Evidence from the 2013 drilling program strongly suggested that the Iron Cap deposit sits above and is displaced to the south-southeast of a near magmatic high-grade core zone.

In 2014, ten core holes totaling about 10,429 m were completed at Iron Cap. The 2014 program was designed to test the continuity and extent of a potential northwest plunging core zone located below the Iron Cap deposit (Iron Cap Lower Zone).

Based on drill hole results after the 2014 drilling program, the Iron Cap Lower Zone is interpreted as a northwest plunging, northeast-southwest striking tabular body located immediately below the existing reserves. Drill holes testing the down plunge extent of the deposit intercepted higher copper and gold grades, which is consistent with a core zone.

The Lower Iron Cap Zone is characterized as a series of related, intermediate composition intrusions, each with extensive and intensive hydrothermal alteration including potassic, phyllic, and silicic alteration, all of which contain copper, gold and silver. Drill holes that targeted the southwestern strike projections of the target zone penetrated numerous intrusives, with variable grade distribution enhanced in the contact zones between the various intrusions. Holes drilled along the northern strike projection of the deposit encountered more consistent intrusive rock with much less grade variability, like hole IC-14-59 with 592.7 m of 1.14 g/t gold and 0.37% copper. Hydrothermal alteration in holes drilled along the northern portion of the deposit demonstrated continuous hydrothermal alteration over a vertical extent of over 1,000 m.

Drill hole IC-14-61 approached to within 1,000 m of the proposed trace of the MTT alignment, potentially making the Iron Cap Lower Zone an attractive early development option with lower capital and operating costs than other deposits at KSM which are further from key infrastructure.

10.5 QP COMMENTS REGARDING DRILLING AND SAMPLING FACTORS

Core recovery from the 2013 to 2015 KSM drilling campaigns was excellent, averaging approximately 97%. RQD and core recovery were affected in some of the pre-2012 drilling in the upper portions of the Kerr deposit due to a rubble zone created by the dissolution of anhydrite veinlets in various lithologic units. Core recovery in the 2013 to 2015 drilling at Kerr was excellent at depth within the modeled anhydrite veinlet population where no dissolution has occurred.

No material drilling, sampling, or recovery issues were encountered for other deposits within the KSM Property that were drill tested during the 2013 to 2015 campaigns.

In the opinion of the Qualified Person responsible for this section of this Technical Report, there are no drilling or sampling factors that could materially impact the accuracy and reliability of the assay results associated with the 2013 to 2015 KSM drilling. Furthermore, the Qualified Person responsible for this section of this Technical Report believes that the assays associated with the 2013 to 2015 drilling campaigns are suitable to be used to estimate Mineral Resources.

11.0 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 INTRODUCTION

This section contains information that was disclosed in previous Technical Reports (pre-2013 and 2012-2013 data), along with a discussion of methods, measures, and results associated with the 2014-2015 drilling data. The discussions are organized into pre-2012 data disclosed in the 2012 PFS (Tetra Tech 2012), 2012-2013 data disclosed in the March 2014 Technical Report (Lechner 2014), and an update of methods, procedures, and results associated with the 2014-2015 drilling campaigns.

11.2 PRE-2012 SAMPLE PREPARATION METHODS AND PROCEDURES

11.2.1 PRE-2012 STATEMENT ON SAMPLE PREPARATION PERSONNEL

Labourers contracted from Tahltan Native Development Corporation conducted all initial sample preparation (sawing and bagging) and were 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 PRE-2012 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 ft by 16 ft 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 inch 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 PRE-2012 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 hydrogen chloride, nitric acid, and water 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 11.1.

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 11.1 ICP Detection Limits – Pre-2012 Data

Element	Lower	Upper	Element	Lower	Upper
Ag	0.2 ppm	100.0 ppm	Mo	1 ppm	10,000 ppm
Al	0.01%	10.00%	Na	0.01%	10.00%
As	5 ppm	10,000 ppm	Ni	1 ppm	10,000 ppm
Ba	5 ppm	10,000 ppm	P	10 ppm	10,000 ppm
Bi	5 ppm	10,000 ppm	Pb	2 ppm	10,000 ppm
Ca	0.01%	10.00%	Sb	5 ppm	10,000 ppm
Cd	1 ppm	10,000 ppm	Sn	20 ppm	10,000 ppm
Co	1 ppm	10,000 ppm	Sr	1 ppm	10,000 ppm
Cr	1 ppm	10,000 ppm	Ti	0.01%	10.00%
Cu	1 ppm	10,000 ppm	U	10 ppm	10,000 ppm
Fe	0.01%	10.00%	V	1 ppm	10,000 ppm
La	10 ppm	10,000 ppm	Y	1 ppm	10,000 ppm
Mg	0.01%	10.00%	Zn	1 ppm	10,000 ppm
Mn	1 ppm	10,000 ppm			

11.2.4 PRE-2012 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, including 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 Gold Blank Performance

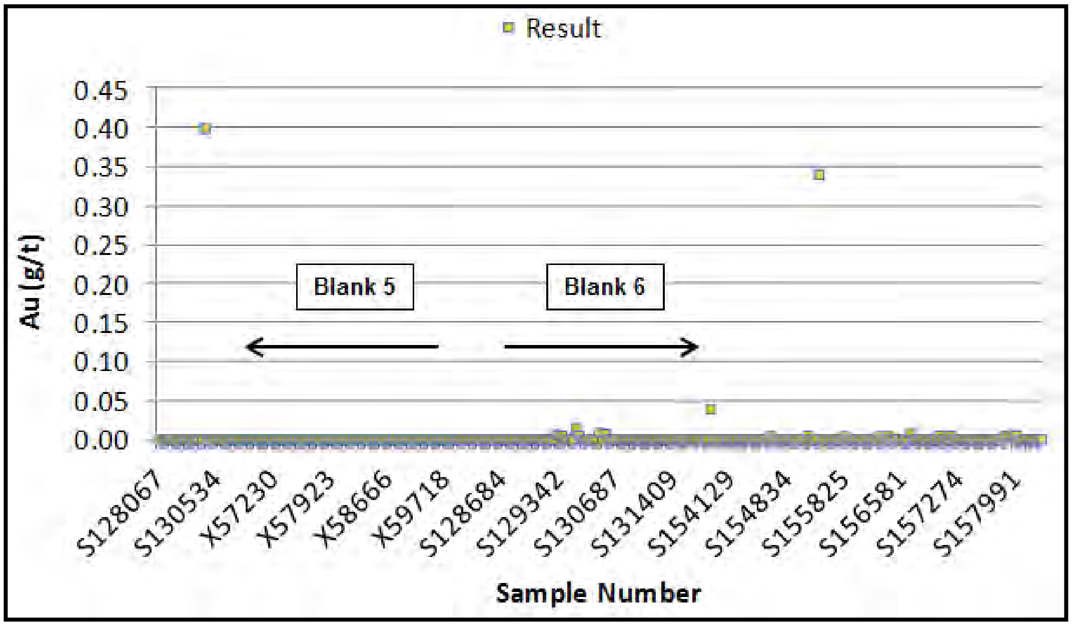
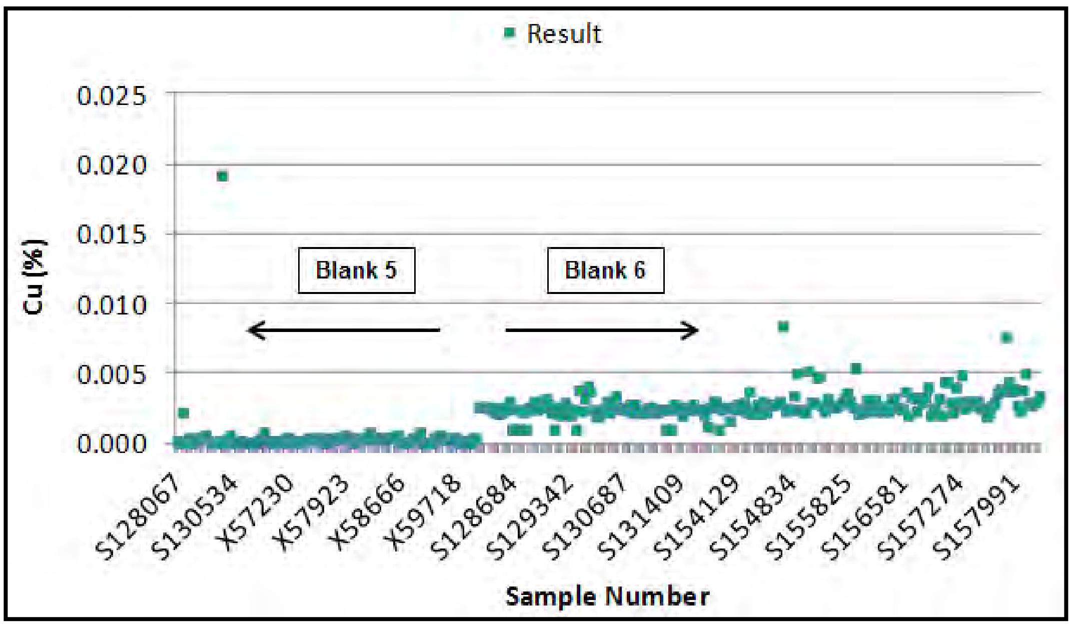


Figure 11.2 2011 Copper Blank Performance



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 Resource) out of Delta, BC. The CDN Resource 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 11.2 summarizes the SRMs that were used by Seabridge for their 2011 drilling campaign. Table 11.2 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 11.2 KSM Standard Reference Materials - Pre-2012 Data

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 Gold Standard CM-4 Performance

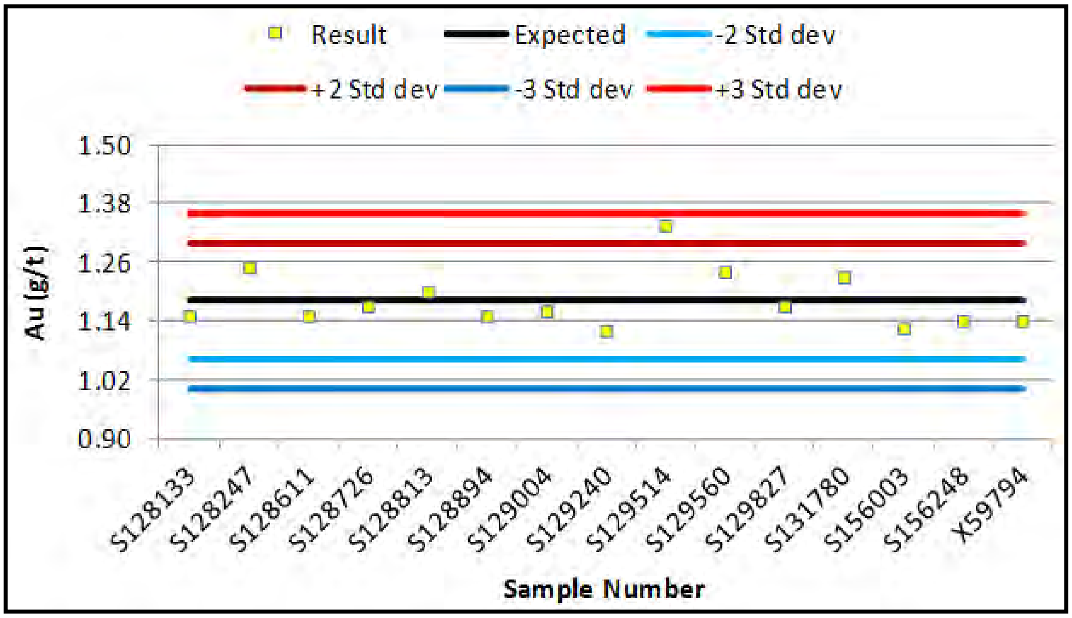


Figure 11.4 2011 Copper Standard CM-4 Performance

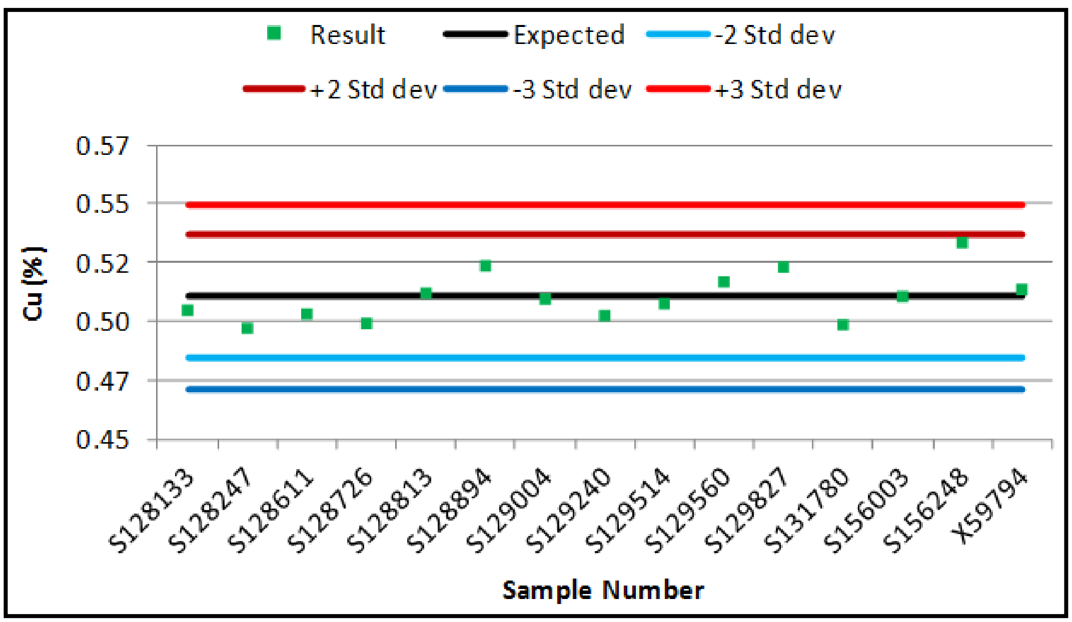


Figure 11.5 2011 Molybdenum Standard CM-4 Performance

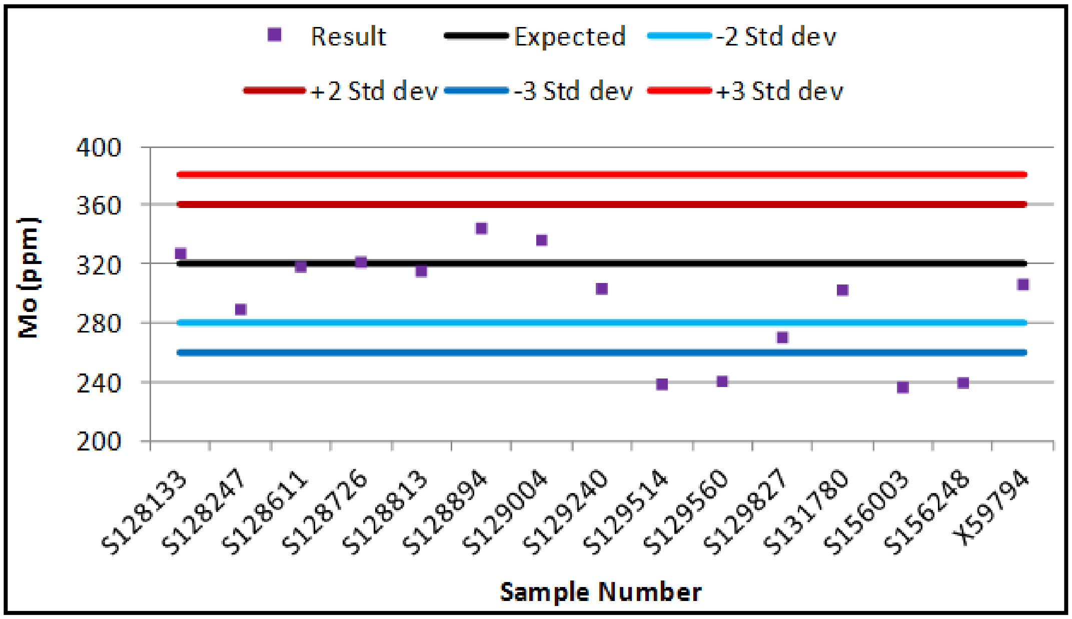


Figure 11.6 2011 Gold Standard CM-11A Performance

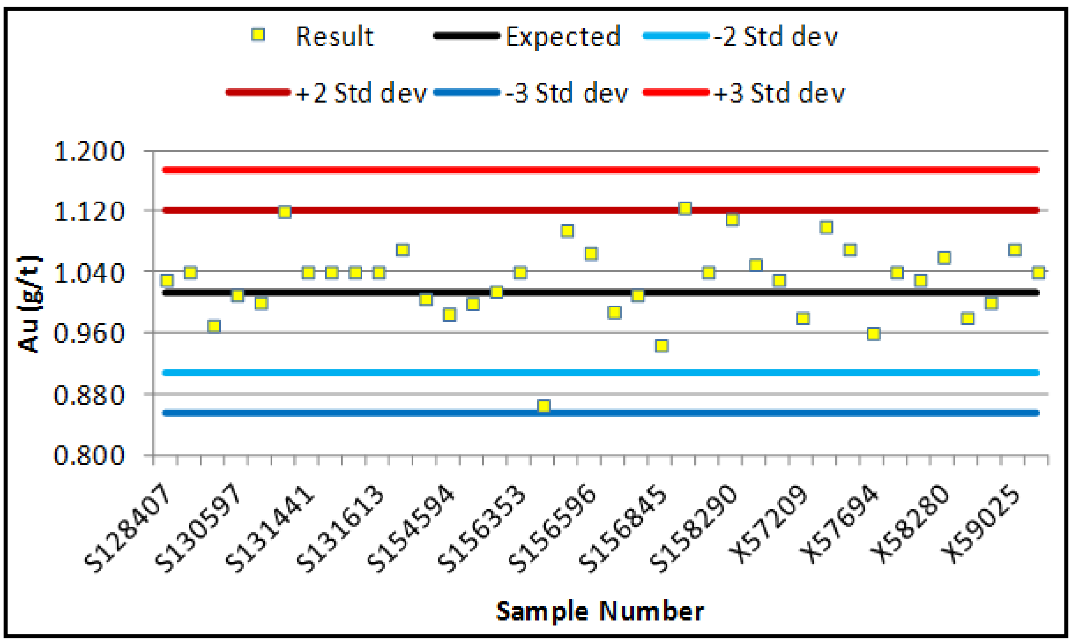


Figure 11.7 2011 Copper Standard CM-11A Performance

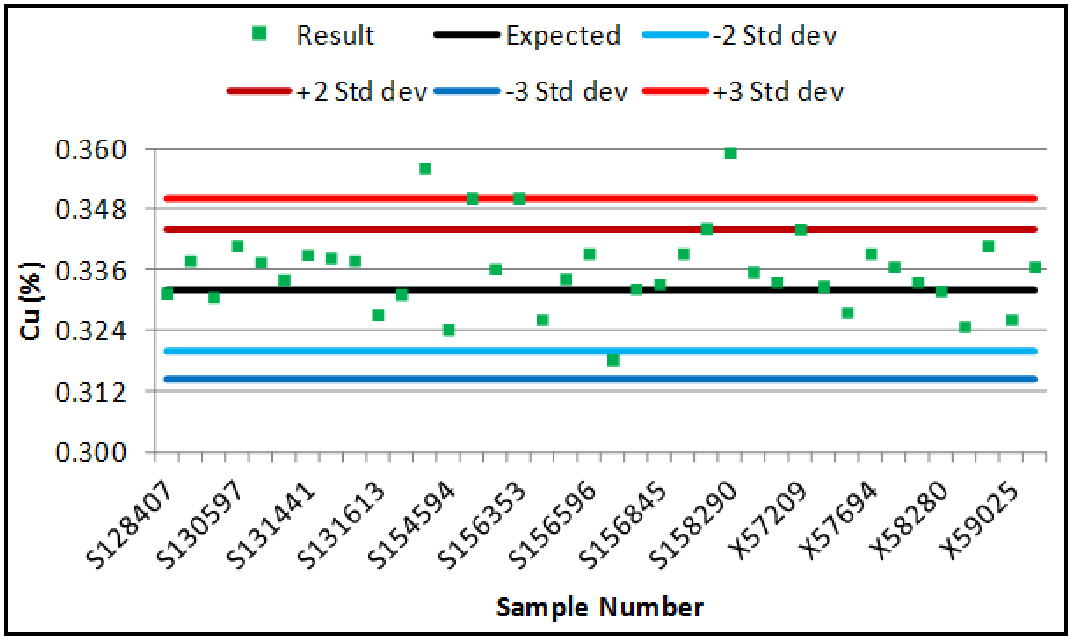


Figure 11.8 2011 Molybdenum Standard CM-11A Performance

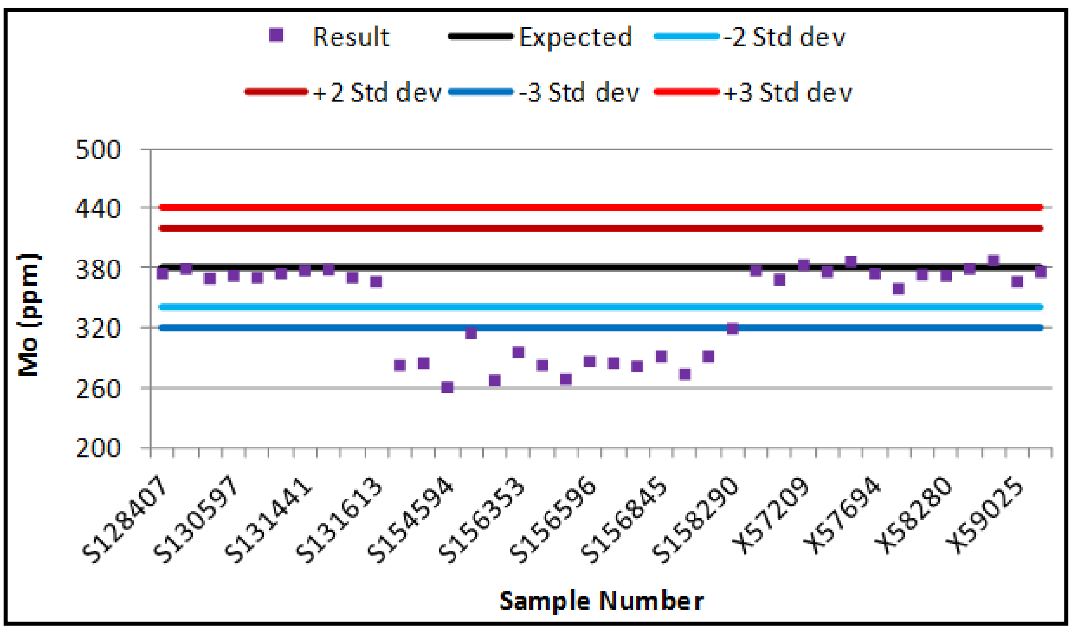


Figure 11.9 2011 Gold Standard CGS-19 Performance

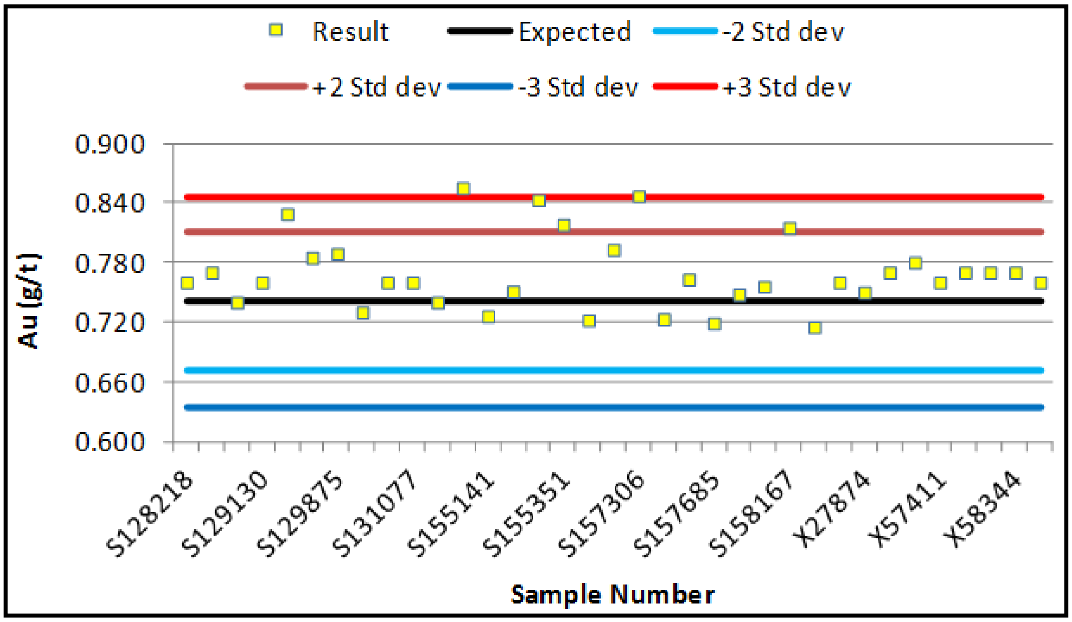


Figure 11.10 2011 Copper Standard CGS-19 Performance

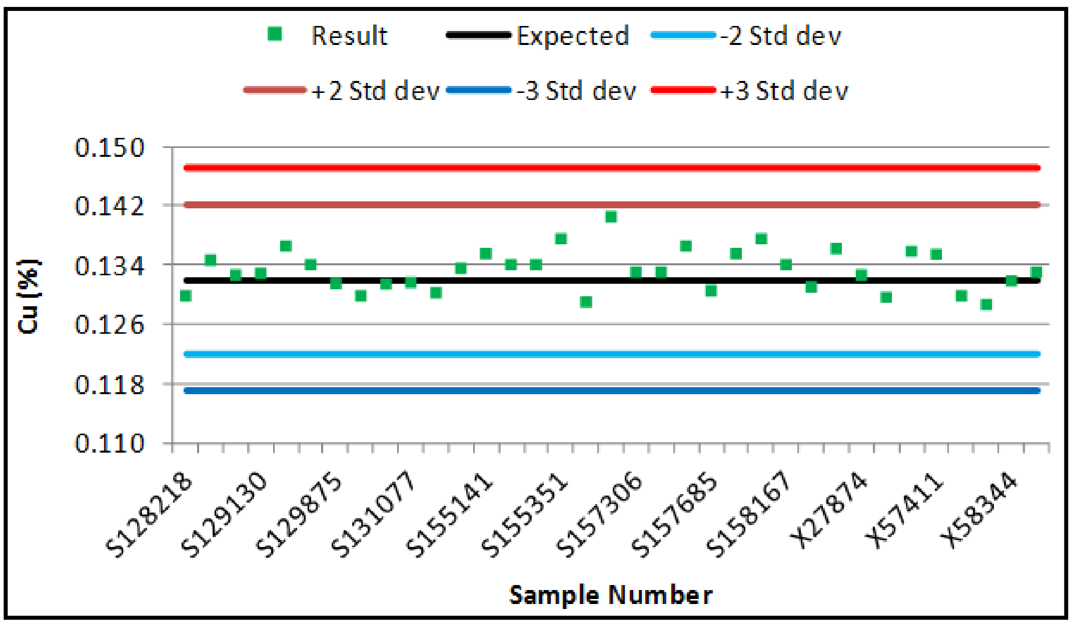


Figure 11.11 2011 Gold Standard CGS-22 Performance

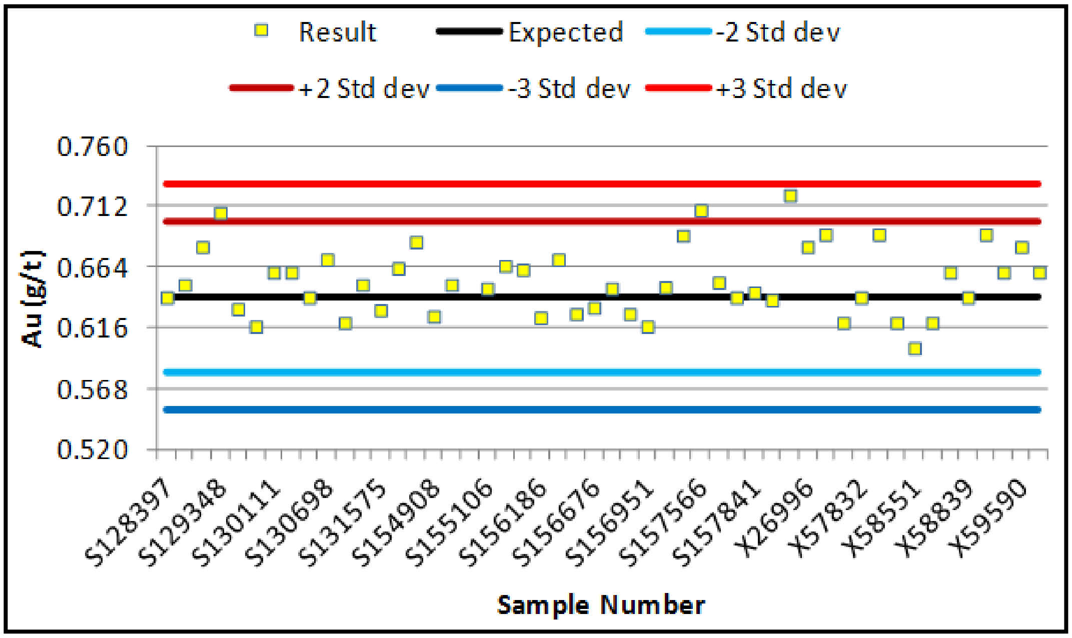


Figure 11.12 2011 Copper Standard CGS-22 Performance

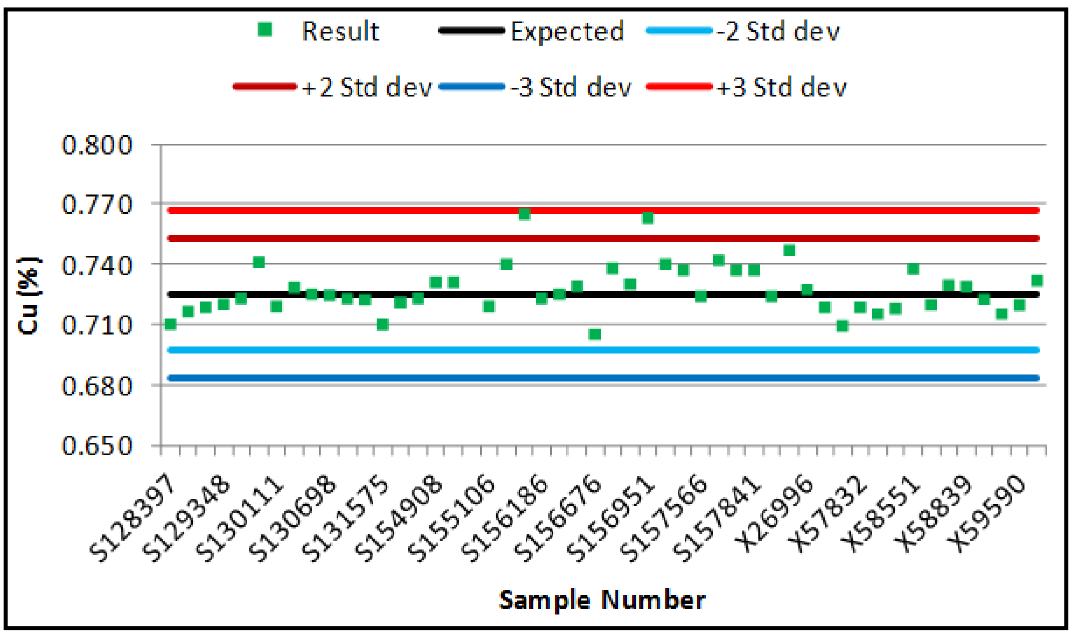


Figure 11.13 2011 Gold Standard CGS-27 Performance

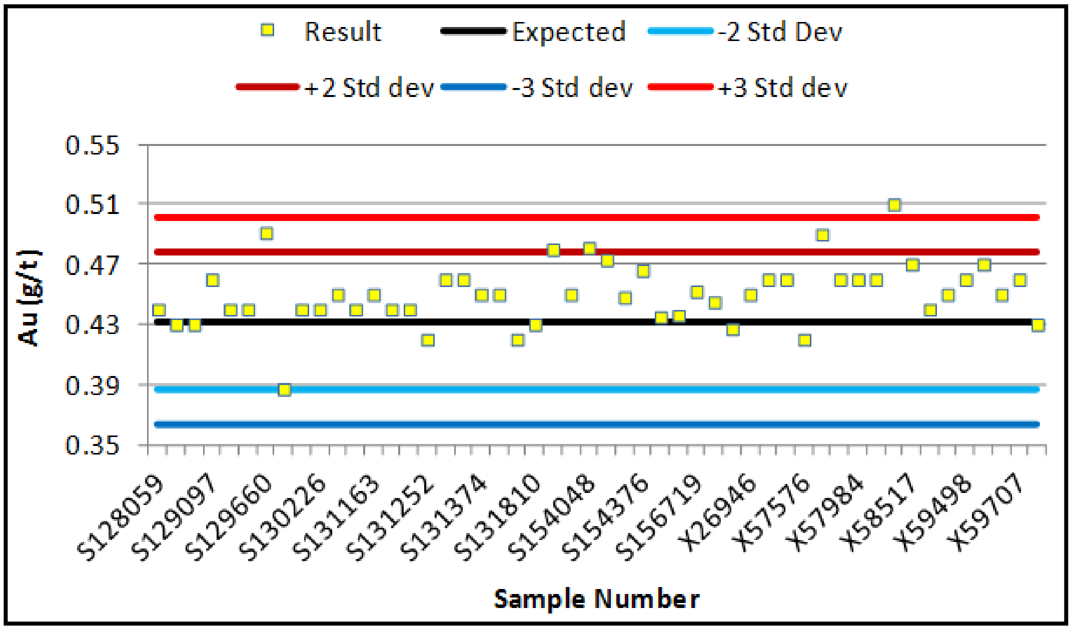


Figure 11.14 2011 Copper Standard CGS-27 Performance

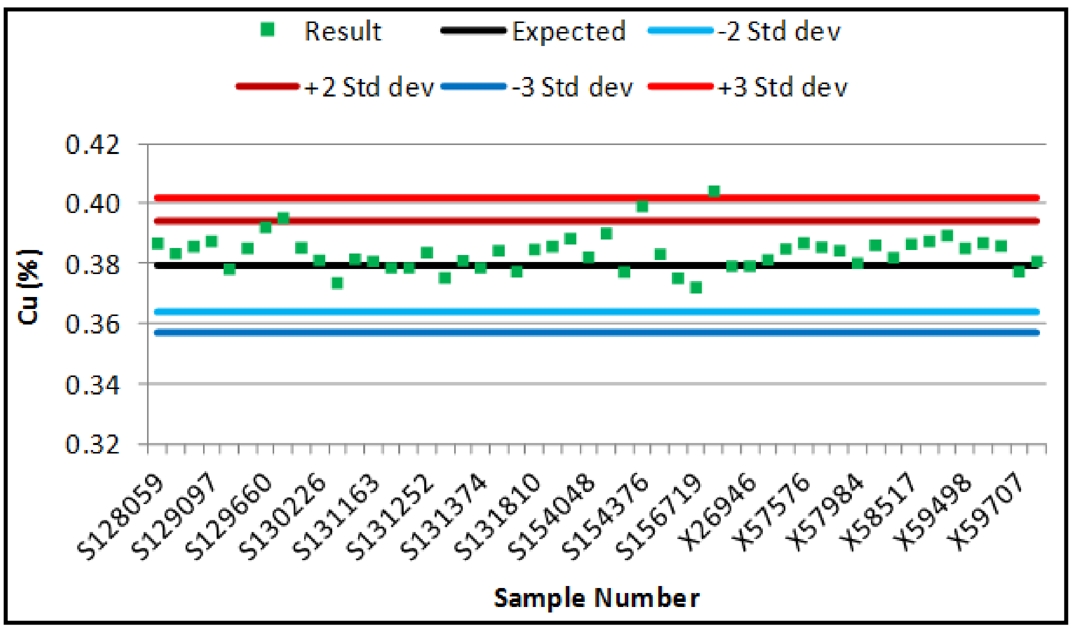


Figure 11.15 2011 Gold Standard SEA-CL2 Performance

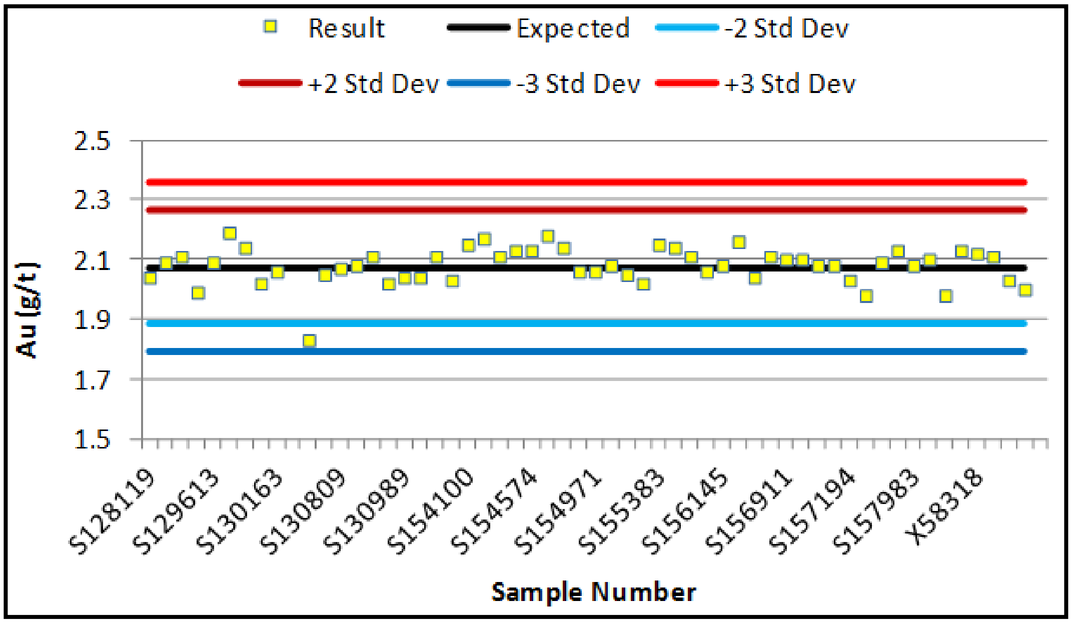


Figure 11.16 2011 Gold Standard SEA-KSM Performance

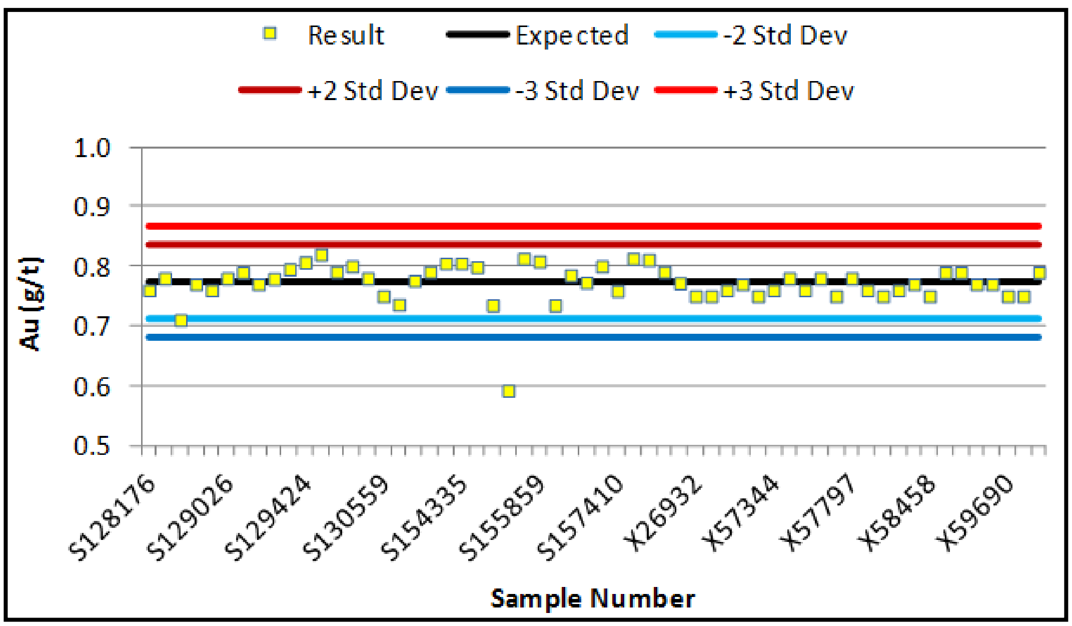


Figure 11.17 2011 Copper Standard SEA-KSM Performance

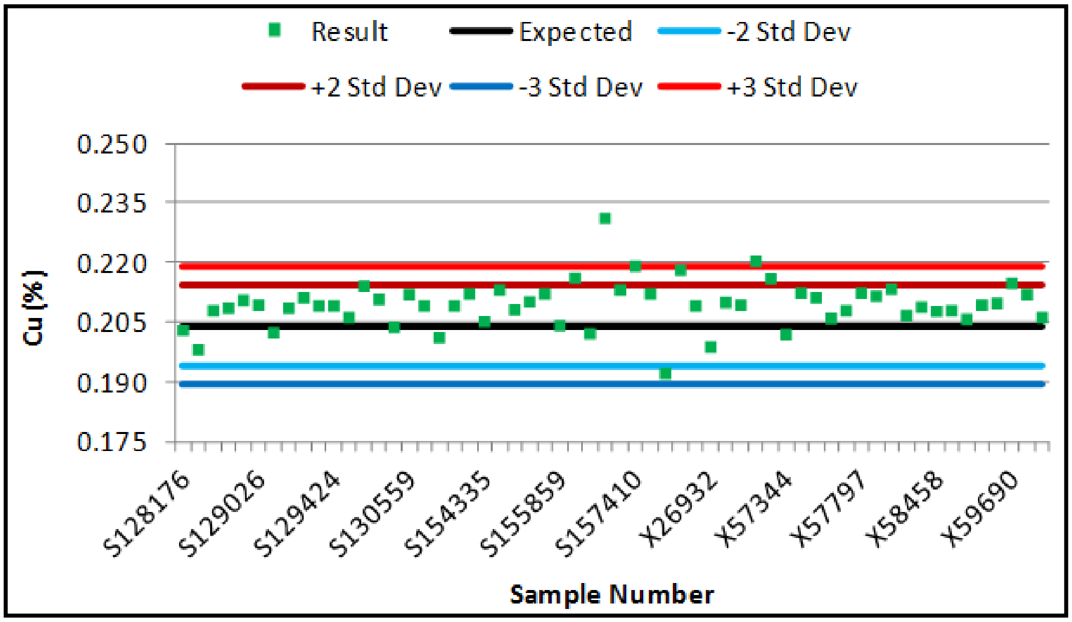
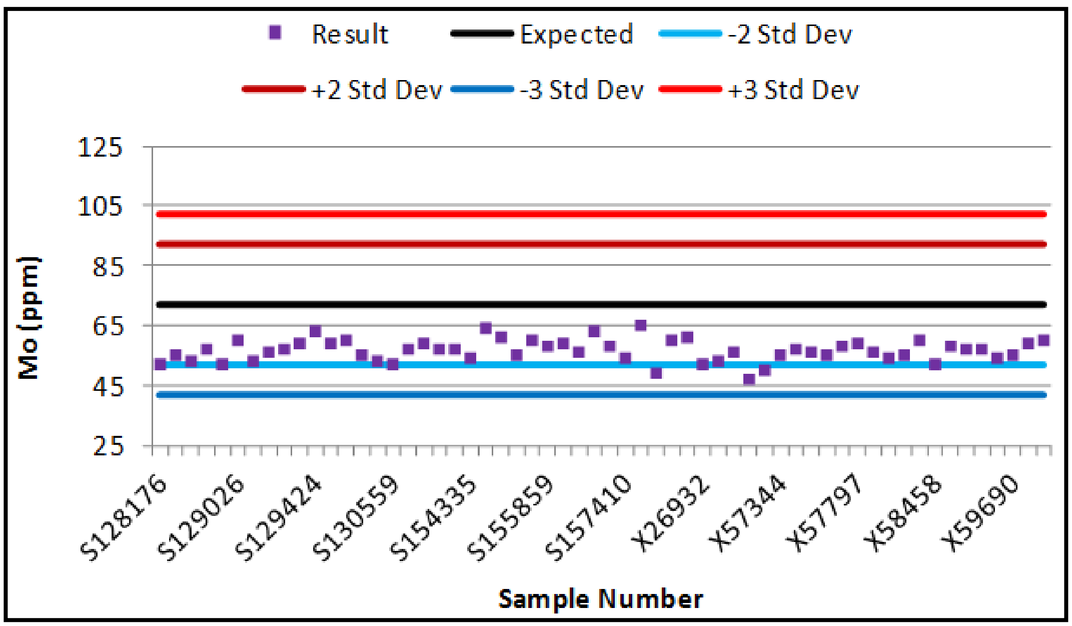


Figure 11.18 2011 Molybdenum Standard SEA-KSM Performance



In general, most of the SRM results track well within ±2 standard deviation units 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, quarter 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 11.3 summarizes the basic descriptive statistic for the "original" and "duplicate" quarter core samples for gold, silver, copper, and molybdenum.

Table 11.3 Summary of Pre-2012 Quarter Core Assay Results

Parameter	Au (g/t)		Cu (%)		Ag (g/t)		Mo (ppm)	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
Count	152	152	152	152	152	152	152	152
Minimum	0.001	0.001	0.0004	0.0003	0.0	0.0	0.1	0.1
Maximum	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 Quartile	0.111	0.098	0.0380	0.0394	0.6	0.5	4.0	4.0
3rd Quartile	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 Difference	5%		-1%		0%		1%	

Note: CV = coefficient of variation

As can be seen in Table 11.3, there is a relatively close comparison in the distribution of original and duplicate quarter core grades. RMI notes that the duplicate gold sample grades are about 5% higher than the original quarter core sample. The copper duplicate is about 1% lower than the original. The quarter 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 Quarter Core Gold QQ Plot

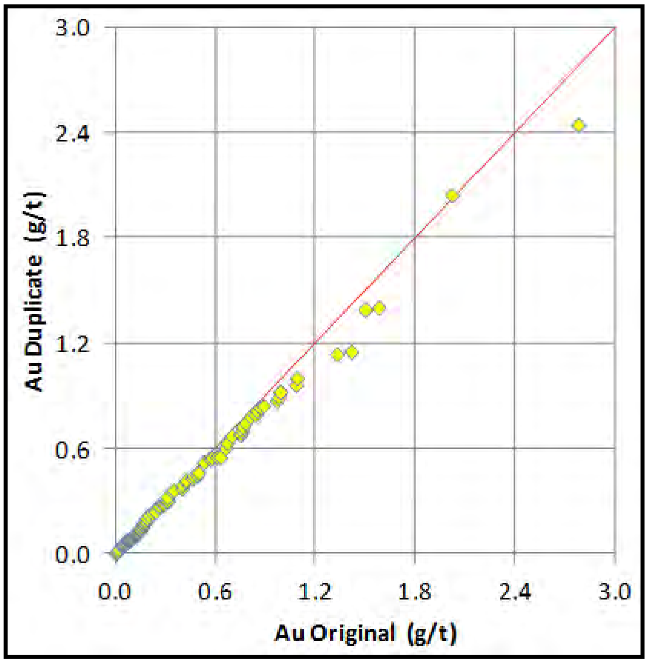


Figure 11.20 2011 Quarter Core Copper QQ Plot

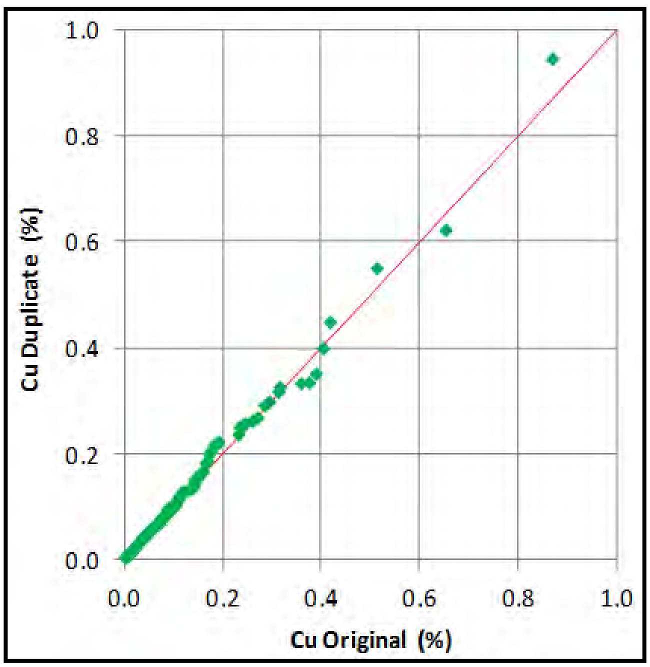


Figure 11.21 2011 Quarter Core Silver QQ Plot

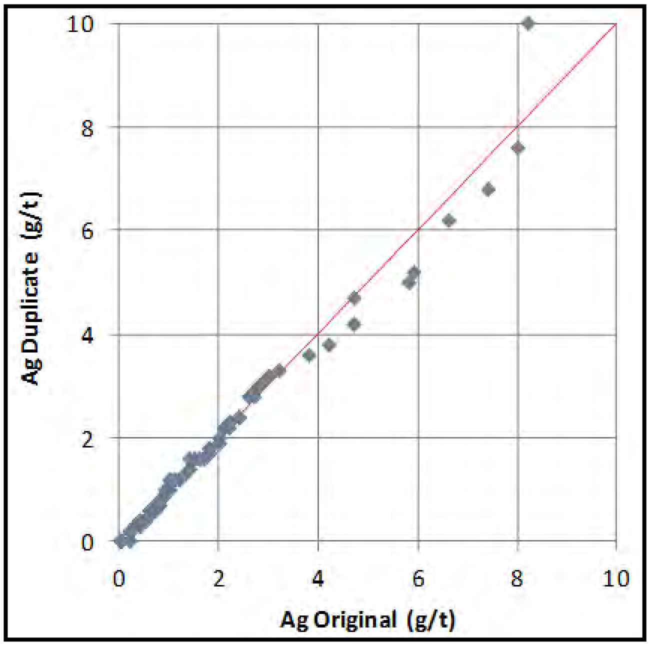
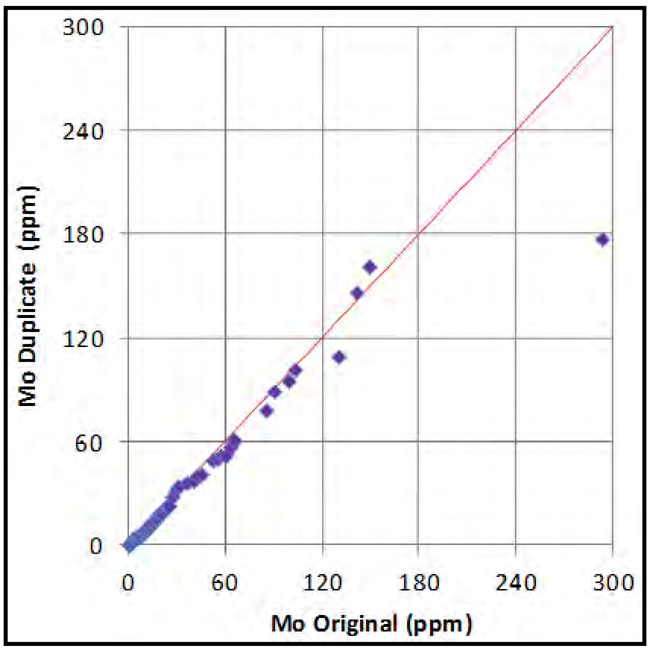


Figure 11.22 2011 Quarter Core Molybdenum QQ Plot



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 11.4 summarizes basic descriptive statistics comparing ALS Chemex (ALS in Table 11.4) and Eco Tech results by metal and analytical method. The data in Table 11.4 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 11.4 Summary of Pre-2012 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
Minimum	0.005	0.015	0.10	0.1	0.1	0.1
Maximum	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
1st Quartile	0.140	0.170	0.4	0.5	0.5	0.4
3rd Quartile	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 Difference	-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
Minimum	0.001	0.001	8	6	0.001	0.001
Maximum	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
1st Quartile	0.043	0.042	424	422	0.043	0.042
3rd Quartile	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 Difference	5%		-5%		0%	

Parameter	ALS Mo (%)	ALS ICP Mo (%)	ALS ICP Mo (ppm)	Eco Tech ICP Mo (ppm)
Count	596	597	597	597
Minimum	0.0005	0.0001	0.5	0.5
Maximum	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
1st Quartile	0.0010	0.0005	5.0	6.0
3rd Quartile	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 Difference	36%		-4%	

Figure 11.23 2011 Eco Tech vs. ALS Chemex Check Assays – Gold

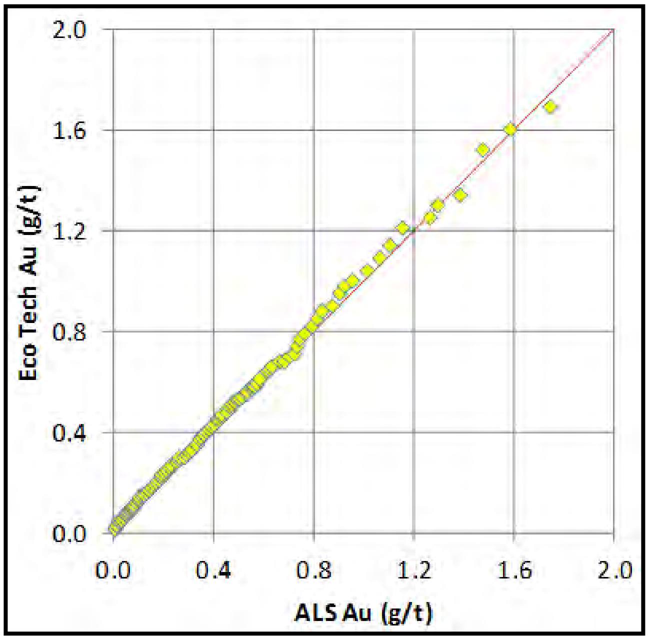


Figure 11.24 2011 Eco Tech vs. ALS Chemex Check Assays – Copper

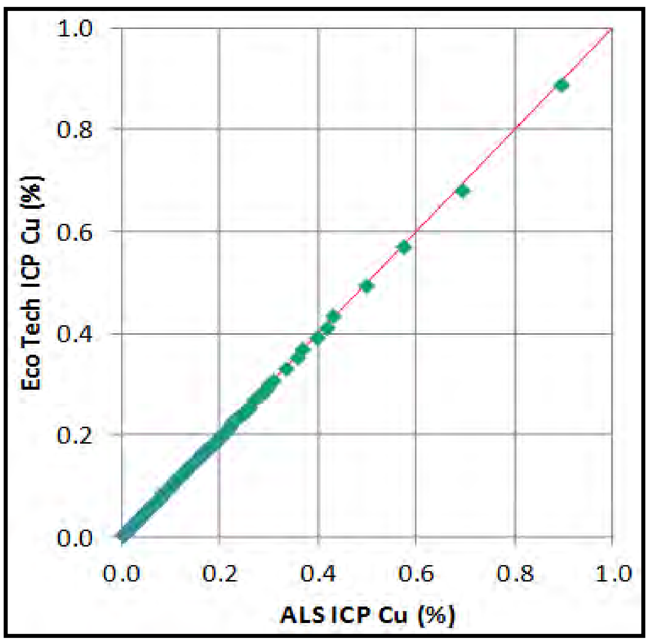


Figure 11.25 2011 Eco Tech vs. ALS Chemex Check Assays – Silver

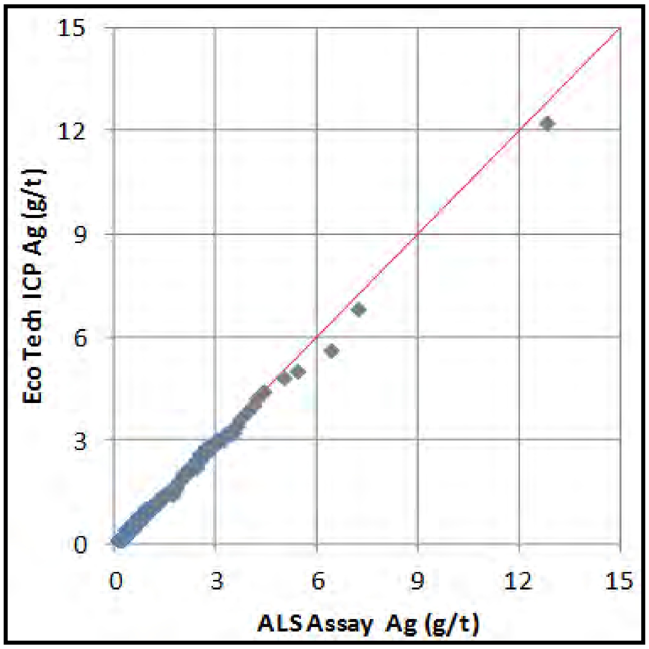
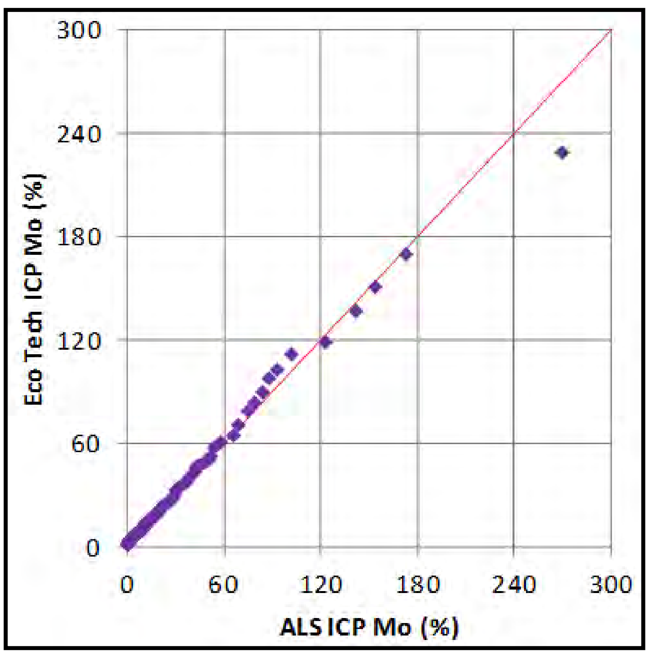


Figure 11.26 2011 Eco Tech vs. ALS Chemex Check Assays – Molybdenum



Both Eco Tech and ALS 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, atomic absorption spectrometry (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 PRE-2012 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 2012 SAMPLING PROGRAMS – QUALIFIED PERSON'S OPINION

In RMI's opinion, the sampling methods/approach, security, sample preparation, analytical procedures, and QA/QC protocols/results that were implemented for the 2012 PFS (Tetra Tech 2012) Resource were adequate, and the subsequent assays were suitable to be used to estimate Mineral Resources.

11.3 2012-2013 SAMPLE PREPARATION METHODS AND PROCEDURES

This section is taken from the March 2014 Technical Report (Lechner, 2014) and discusses the sampling methods, security, sample preparation, and quality control measures associated with the 2012-2013 Deep Kerr drilling programs.

11.3.1 2012-2013 STATEMENT ON SAMPLE PREPARATION PERSONNEL

Labourers contracted form Northern Labour Force Corporation conducted all initial sample preparation (sawing and bagging), and were trained by, and under the direct supervision of, geologists employed by Seabridge. Drill core and quality control samples were shipped to ALS Canada Ltd. (ALS) receiving facility located in Stewart BC. From there, the samples were shipped to ALS's prep facility located in Terrace, BC either by a commercial trucking firm or by ALS personnel. The prepped samples were then shipped by ALS to their assay laboratory located in North Vancouver, BC for analysis.

11.3.2 2012-2013 SAMPLE PREPARATION AND DISPATCH

Drill core, placed in wooden core boxes by the drilling contractor, was loaded into metal baskets and delivered twice daily by helicopter from each drill rig, to Seabridge's Sulphurets Creek camp facilities. After the core was delivered to the core logging tents, Seabridge geologists took an inventory of the core (a review of core condition, a check of run block depths, and a quick down-hole lithologic log prepared). After the drill core was completely logged—sample starting/ending points were laid out and a numbered sample tag stapled to the box at the beginning of the sample run—the core was photographed. Core boxes were then moved to the core cutting facilities located adjacent to the core logging tents. Two Weatherhaven tents containing four saws with 14 inch diamond impregnated blades designed for rock cutting were utilized for sawing the core longitudinally. 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 along with the sample number tag. The outside of the poly bags were marked with the sample number and then stapled shut. The remaining half of the drill core was carefully put back into the wooden core boxes as soon as cutting was complete.

The polythene sample bags were inventoried and then approximately three polythene sample bags containing HQ diameter core (or six bags with NQ core) were placed into large polyweave (rice) shipping bags. Approximately 30 rice bags were placed into wooden crates (“totes”) for shipment. The totes were flown by helicopter to a landing pad near km 54 on the Eskay Creek Mine Road located behind a locked gate. The totes were then placed into a locked steel sea-going container that can hold approximately eight totes. Once or twice a week, Granmac Services from Stewart, BC picked up the totes and delivered them to an ALS receiving facility located in Stewart, BC where the samples were logged into ALS's system. From Stewart, the samples were delivered by either ALS or Banstra Transportation Systems, Ltd. to an ALS preparation laboratory located in Terrace, BC.

11.3.3 2012-2013 Analytical Procedures

ALS served as Seabridge's primary assay laboratory. ALS is a leading provider of assaying and analytical testing services for mining and mineral exploration companies and has no association or affiliation with Seabridge. All of ALS's locations are International Organization for Standardization (ISO) 9001:2000 certified.

AcmeLabs served as a check assay laboratory for Seabridge. AcmeLabs is a leading geochemical and assaying facility and has no association or affiliation with Seabridge. In October 2011, AcmeLabs’ Vancouver, BC facility received ISO/International Electrotechnical Commission (IEC) 17025:2005 accreditations from the Standards Council of Canada.

At the ALS preparation facility located in Terrace, BC, samples were sorted and dried (if necessary), crushed through a jaw crusher and cone or roll crusher to 70% –2 mm using ALS protocol CRU-31. The crushed sample was then split using ALS protocol SPL-21 using a riffle splitter. A portion of the crushed sample was replaced into the polythene bag (coarse reject) and stored temporarily at the ALS facility. A portion of the crushed sample was then pulverized using ALS protocol PUL-31 using a ring and puck pulverizer, until approximately a 250 g sub-sample (pulp) was achieved with 85% passing 75 µm or better. The resulting pulp sample was placed in a numbered paper envelope and securely packed in cardboard boxes. These boxes were shipped by ALS to their assay facility located in North Vancouver, BC.

At the ALS analytical laboratory located in North Vancouver, a 30 g sample was split out from the pulp envelope and then fire assayed using ALS protocol Au-AA23 using appropriate fluxes. The resultant doré bead was parted and then digested with aqua regia followed by an AA finish using an Agilent AA 240 Series instrument. The lower limit of detection for gold is 0.005 g/t.

For other metals, a multi-element ICP-atomic emission spectroscopy (AES) analysis was completed using ALS protocol ME-ICP41. For this procedure, an approximately 0.5 g sample was digested with aqua regia in a graphite heating block. After cooling, the resulting solution was diluted to 12.5 mL with de-ionized water, mixed and analyzed by ICP-AES using an Agilent ICP 720/730-ES Series instrument. The analytical results were corrected for inter-element spectral interferences. ALS's ME-ICP41 lower and upper detection limits are summarized in Table 11.5 for the elements that Seabridge requested.

Table 11.5 ICP Detection Limits – 2012-2013

Element	Units	Lower	Upper	Element	Units	Lower	Upper
Ag	ppm	0.2	100	Mo	ppm	1	10,000
Al	%	0.01	25	Na	%	0.01	10
As	ppm	2	10,000	Ni	ppm	1	1,000
B	ppm	10	10,000	P	ppm	10	1,000
Ba	ppm	10	10,000	Pb	ppm	2	1,000
Be	ppm	0.5	1,000	S	%	0.01	10
Bi	ppm	2	10,000	Sb	ppm	2	1,000
Ca	%	0.01	25	Sc	ppm	1	1,000
Cd	ppm	0.5	1,000	Se	ppm	10	10,000
Co	ppm	1	10,000	Sn	ppm	10	10,000
Cr	ppm	1	10,000	Sr	ppm	1	1,000
Cu	ppm	1	10,000	Th	ppm	20	1,000
Fe	%	0.01	50	Ti	%	0.01	10
Ga	ppm	10	10,000	U	ppm	10	1,000
K	%	0.01	10	V	ppm	1	1,000
La	ppm	10	10,000	W	ppm	10	1,000
Mg	%	0.01	25	Zn	ppm	2	1,000
Mn	ppm	5	50,000				

During most of the 2012 and 2013 Deep Kerr drilling campaigns, ICP copper assays in excess of 5,000 ppm (0.50%) were re-assayed using ALS's “ore grade” protocol ME-OG46 (Cu-OG46). Additionally, in 2013 most of the samples that returned values greater than 2,500 ppm copper (0.25%) were also done by method ME-OG46. The sample pulp was digested in 75% hot aqua regia for 120 minutes. After cooling, the resulting solution was diluted with de-ionized water, mixed and then analyzed by ICP-AES or AAS.

Gold was routinely assayed by ALS using their Au-AA23 protocol. All over limit gold assays (i.e. greater than 10 ppm) where then Au-GRA21, which consisted of a conventional 30-g fire assay with gravimetric finish. A prepared sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents to produce a lead button. The lead button containing precious metals is cupelled to remove the lead. The remaining gold and silver bead is parted in dilute nitric acid, annealed and weighed as gold. Silver, if requested can them be determined by the difference in weights.

ALS assay results were distributed to key Seabridge personnel via signed paper certificates and also as digital .csv files. Likewise, check assay results were distributed to Seabridge personnel in both hard copy and digital formats.

11.3.4 2012-2013 Quality Control Measures

Seabridge implemented similar quality control procedures for their Deep Kerr drilling program that have been in use for their previous KSM drilling campaigns. A total of 14,351 assays were submitted to ALS during the 2012-2013 Deep Kerr drilling campaigns. The quality control measures that were used consisted of:

- the submission of barren or blank material at a frequency of one blank for about every 33 samples
- the submission of one SRM for about every 33 samples
- the collection and submission of duplicate quarter core samples at a frequency of one duplicate sample for about every 50 samples
- the submission of approximately 10% of the ALS pulps to AcmeLabs for check assay purposes.

Table 11.6 shows the type and number of control samples that were submitted to either ALS or AcmeLabs for the 2012-2013 Deep Kerr drilling campaign.

Table 11.6 Summary of Control Samples Submitted in 2012-2013

Type of Control Sample	Number Submitted	Submission Frequency
Blanks	435	1 in 33
SRMs	432	1 in 33
Duplicates	281	1 in 51
Check Assays	1,500	10%

11.3.5 2012-2013 Blank Sample Performance

Seabridge submitted two different barren or blank samples during their 2012-2013 Deep Kerr drilling programs. They included commercial landscaping materials consisting of crushed marble (Blank 5) and pea gravel (Blank 7). The blanks were submitted along with the sawn drill core at a frequency of one blank for every 33 samples.

Figure 11.27 through Figure 11.30 compare the analyzed blank values for copper, gold, silver, and molybdenum, respectively against various detection limit thresholds.

Figure 11.27 2012-2013 Blank Performance – Copper

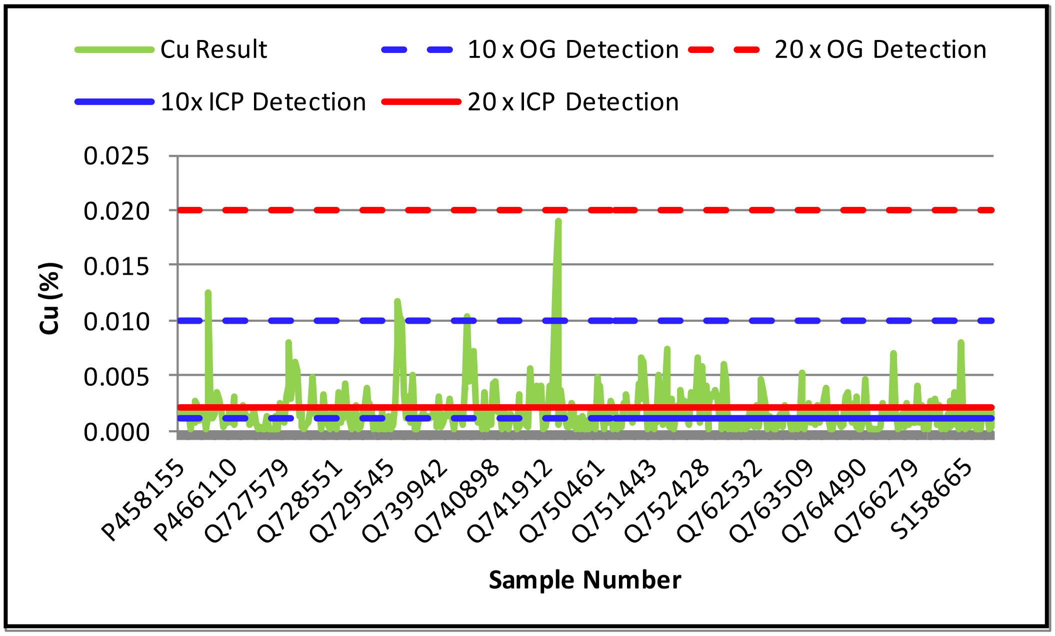


Figure 11.28 2012-2013 Blank Performance – Gold

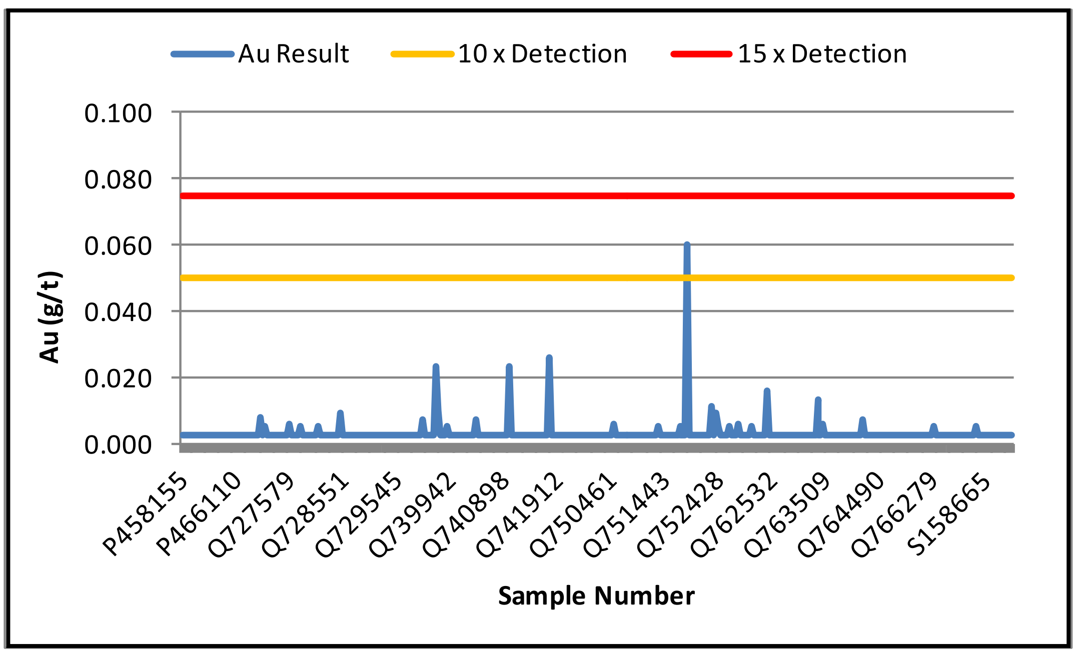


Figure 11.29 2012-2013 Blank Performance – Silver

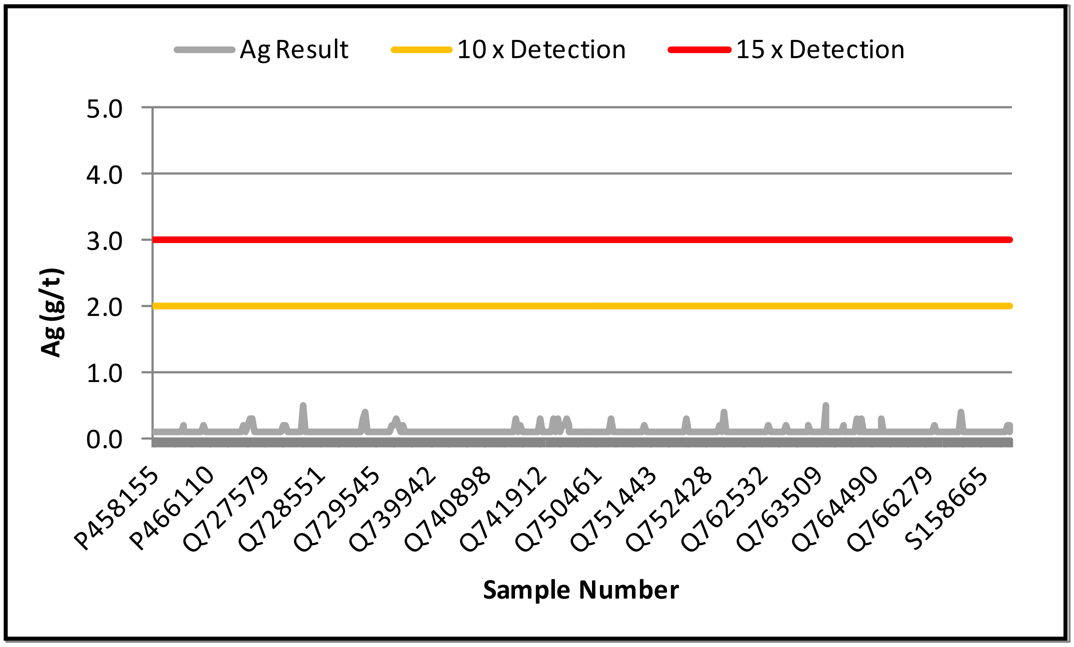
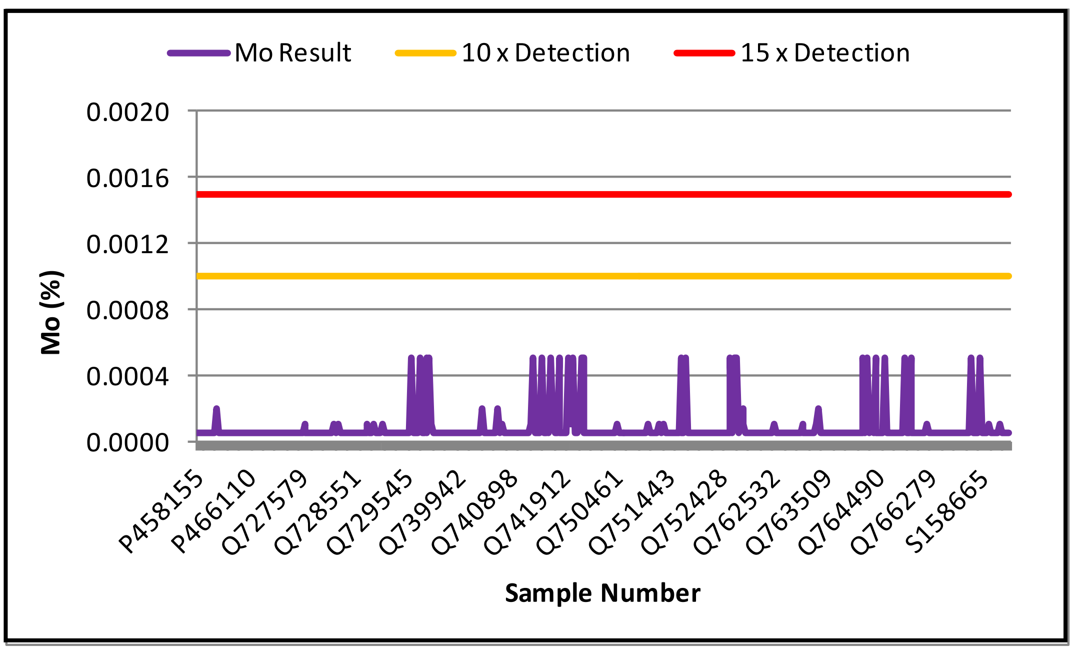


Figure 11.30 2012-2013 Blank Performance – Molybdenum



Ten and twenty times detection limit for copper are shown on the blank performance graph (Figure 11.27) for both ICP and ALS's "ore grade" assay protocols. The two blank materials used by Seabridge appear to be somewhat "dirty" with respect to copper as seen in Figure 11.27. Approximately 60% of the copper blanks returned values greater than 10 times the ALS ICP detection limit of 1 ppm for copper and about 29% came back 20 times detection (2 ppm). Only three blank samples returned values greater than 10 times the detection limit (0.01% or 100 ppm) and none yielded values in excess of 20 times the "ore grade" analytical method.

The apparent poor performance of the copper blank material may be a result of several factors including "dirty" blank material, potential contamination of crushing/pulverizing equipment, or contamination of the ICP apparatus. Based on these results, the QP for this report recommends that Seabridge obtain another source of barren or blank material. The performance of the blank material for other metals appears to be well within industry standards.

Most of the gold blanks came back below detection limits (0.005 g/t) although one blank came back slightly above 10 times detection. All of the silver blanks came back well below 10 times the detection limit. All of the molybdenum blanks came back well below 10 times the detection limit.

11.3.6 2012-2013 Standard Reference Sample Performance

A total of 432 SRMs were submitted by Seabridge for their 2012-2013 Deep Kerr drilling campaigns. All but two of these SRMs were purchased as certified standards from commercial vendors. Two SRMs were prepared from Seabridge exploration/development properties (SEA-CL2 from Courageous Lake and SEA-KSM from KSM). Those two SRMs were certified by Smee & Associates and CDN Resource. Table 11.7 summarizes the expected values and two standard deviations associated with the 432 SRMs that were submitted in 2012-2013.

In general, the ALS SRM results for molybdenum were lower than the expected value. This is attributed to the molybdenum not being completely liberated due to the aqua regia digestion method used in their ICP analytical method. Silver SRM results generally tended to be close to the expected value for all SRM's. Since molybdenum and silver values are generally low for the Deep Kerr deposit, graphs for those SRMs are not deemed to be material.

SRM performance graphs are shown for five SRMs (SEA-KSM, CM-25, CM-23, CM-14, and CM-17) for copper and gold only. These five standards represent 70% of the standards submitted and all have over 30 determinations. Figure 11.31 through Figure 11.40 show the performance for copper and gold for the five SRMs.

Table 11.7 Summary of SRM Samples Submitted in 2012-2013

Standard Name	Source of Material	Number Submitted	Expected Values					2 Standard Deviations				
			Au (g/t)	Cu (%)	Mo (%)	Ag (g/t)	Pb (%)	Au (g/t)	Cu (%)	Mo (%)	Ag (g/t)	Pb (%)
CGS-19	CDN Resource Laboratories Ltd.	1	0.740	0.132	-	-	-	0.070	0.010	-	-	-
CM-11A	CDN Resource Laboratories Ltd.	7	1.014	0.332	0.038	-	-	0.106	0.012	0.004	-	-
CM-14	CDN Resource Laboratories Ltd.	39	0.792	1.058	0.042	-	-	0.078	0.062	0.002	-	-
CM-16	CDN Resource Laboratories Ltd.	11	0.294	0.184	0.016	-	-	0.046	0.014	0.002	-	-
CM-17	CDN Resource Laboratories Ltd.	34	1.370	0.791	0.075	14.400	-	0.130	0.040	0.008	1.400	-
CM-19	CDN Resource Laboratories Ltd.	21	2.110	2.020	0.106	-	-	0.220	0.070	0.008	-	-
CM-23	CDN Resource Laboratories Ltd.	61	0.549	0.472	0.025	-	-	0.060	0.026	0.002	-	-
CM-25	CDN Resource Laboratories Ltd.	66	0.228	0.191	0.019	-	-	0.030	0.006	0.002	-	-
CM-27	CDN Resource Laboratories Ltd.	24	0.636	0.592	0.051	-	-	0.068	0.030	0.004	-	-
CM-29	CDN Resource Laboratories Ltd.	29	0.720	0.734	0.053	-	-	0.068	0.030	0.004	-	-
CM-30	CDN Resource Laboratories Ltd.	32	1.300	0.730	0.070	15.900	0.273	0.120	0.034	0.004	1.300	0.014
SEA-CL2	Smee & Associates Consulting Ltd.	2	2.073	-	-	-	-	0.188	-	-	-	-
SEA-KSM	CDN Resource Laboratories Ltd.	105	0.774	0.204	0.007	-	-	0.062	0.010	0.001	-	-

Figure 11.31 2012-2013 SEA-KSM SRM Performance – Copper

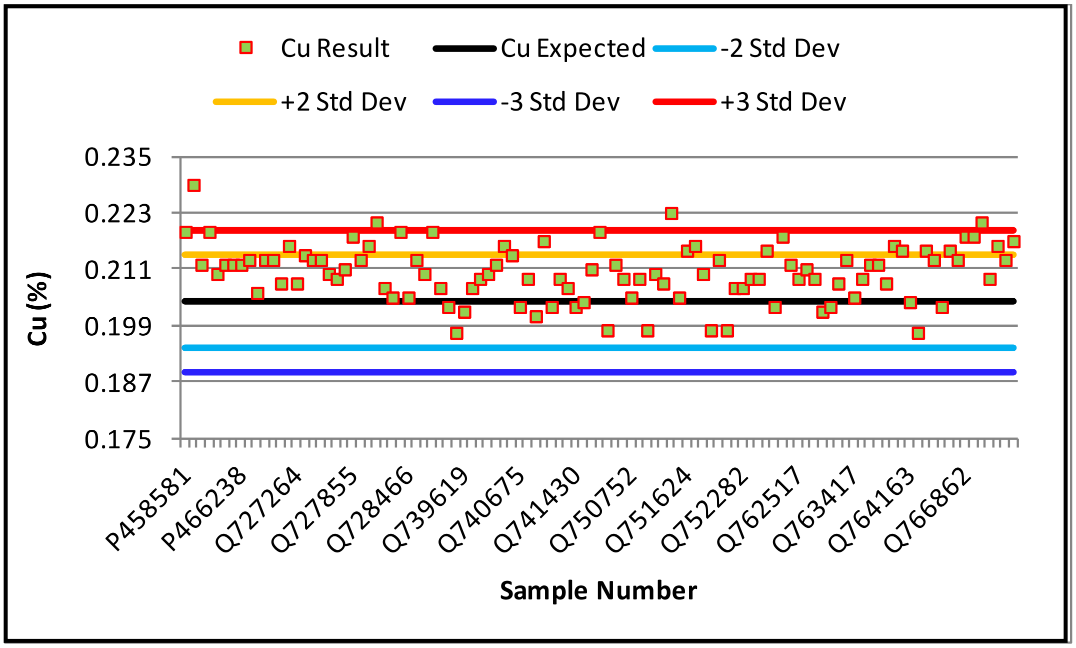


Figure 11.32 2012-2013 CM-25 SRM Performance – Copper

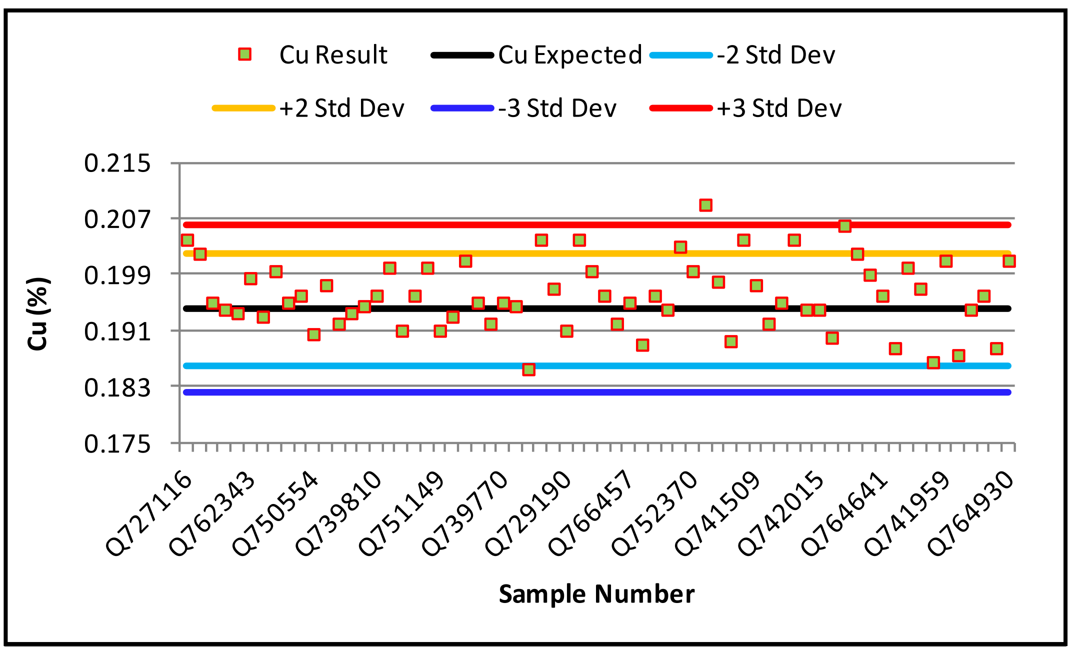


Figure 11.33 2012-2013 CM-23 SRM Performance – Copper

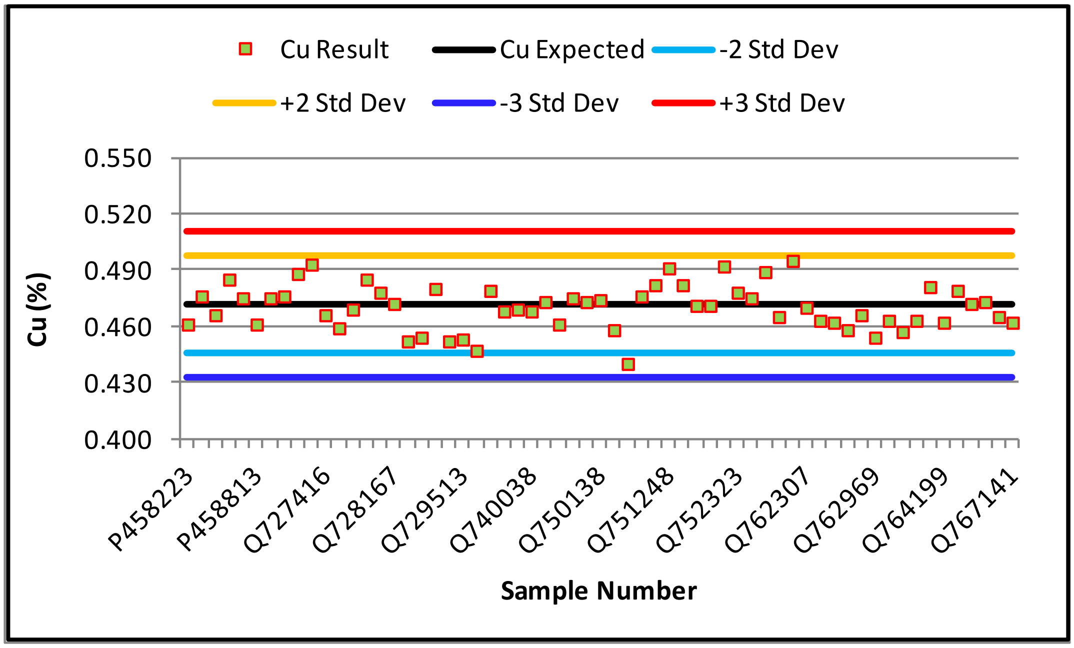


Figure 11.34 2012-2013 CM-14 SRM Performance – Copper

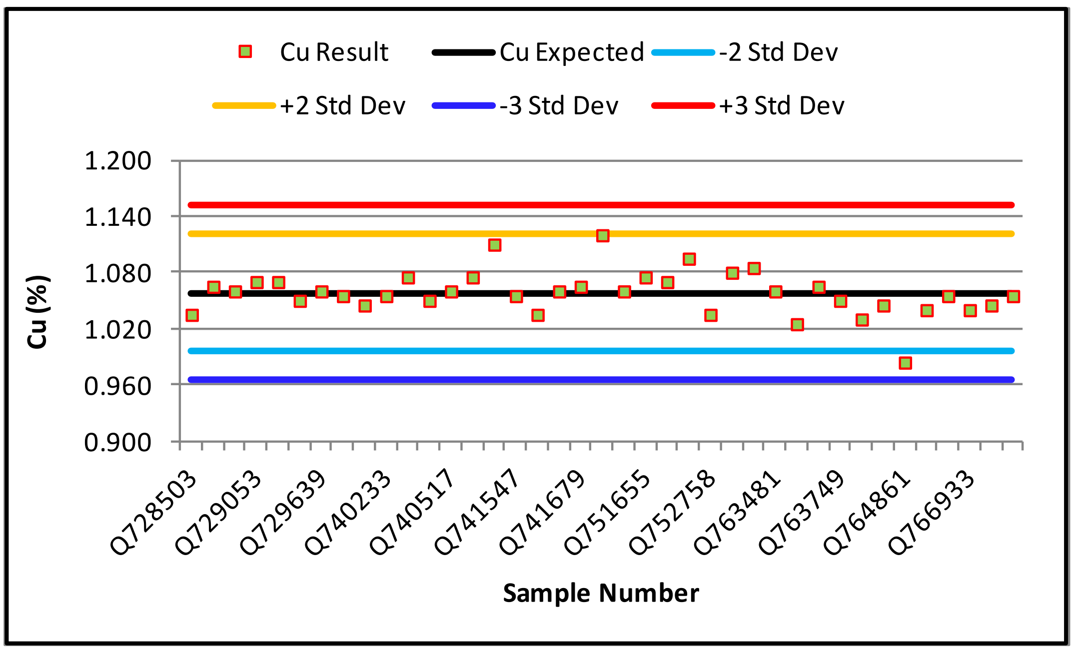


Figure 11.35 2012-2013 CM-17 SRM Performance – Copper

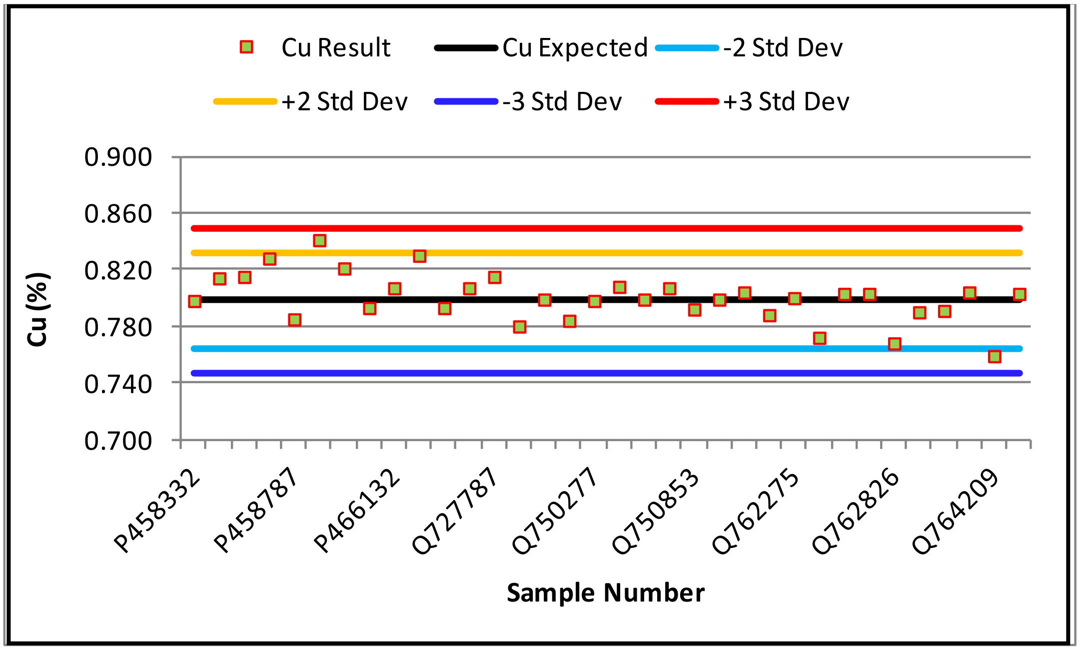


Figure 11.36 2012-2013 SEA-KSM SRM Performance – Gold

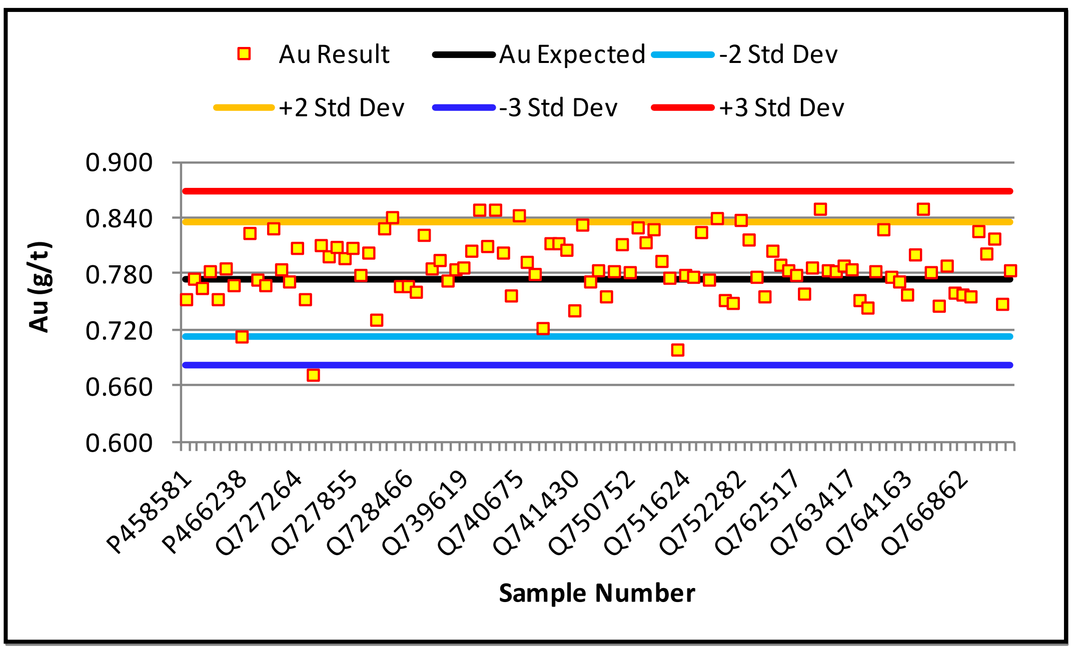


Figure 11.37 CM-25 SRM Performance – Gold

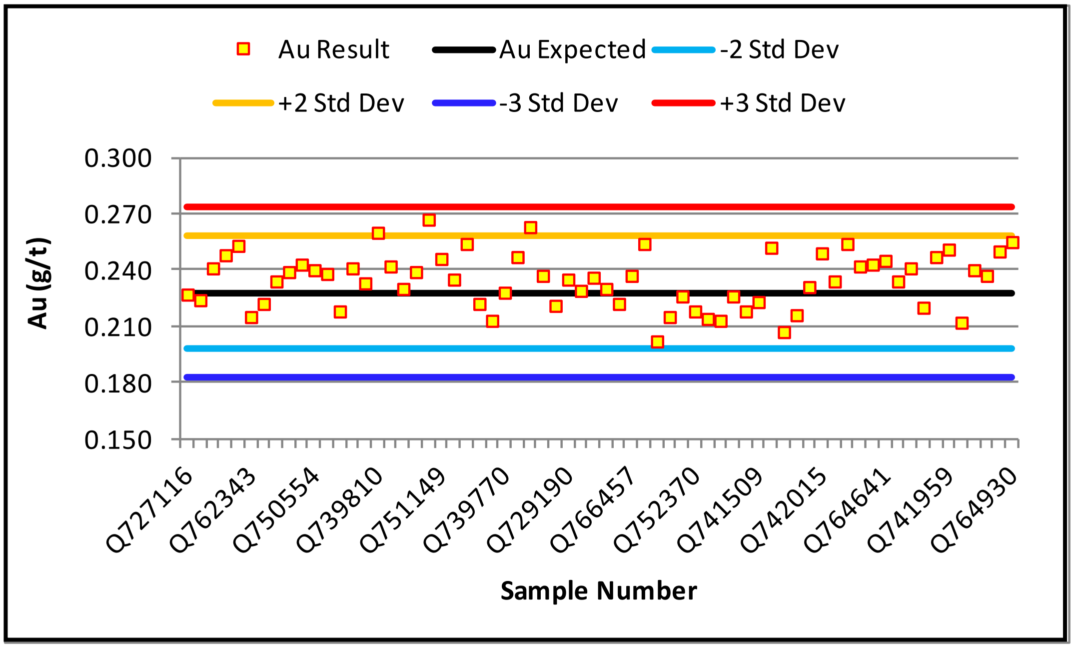


Figure 11.38 CM-23 SRM Performance – Gold

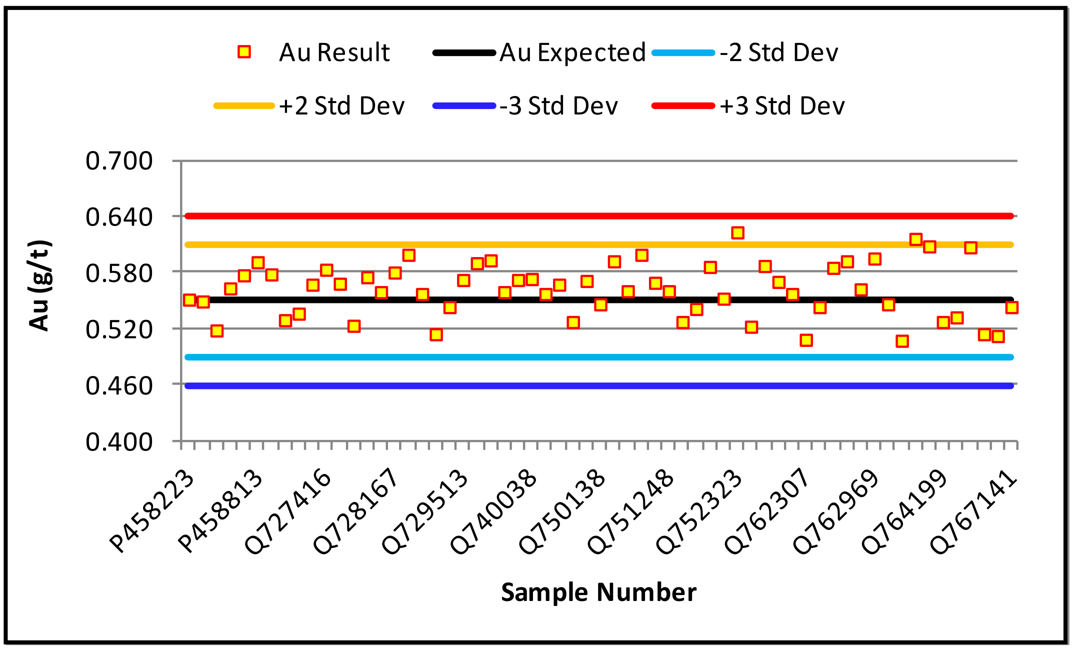


Figure 11.39 CM-14 SRM Performance – Gold

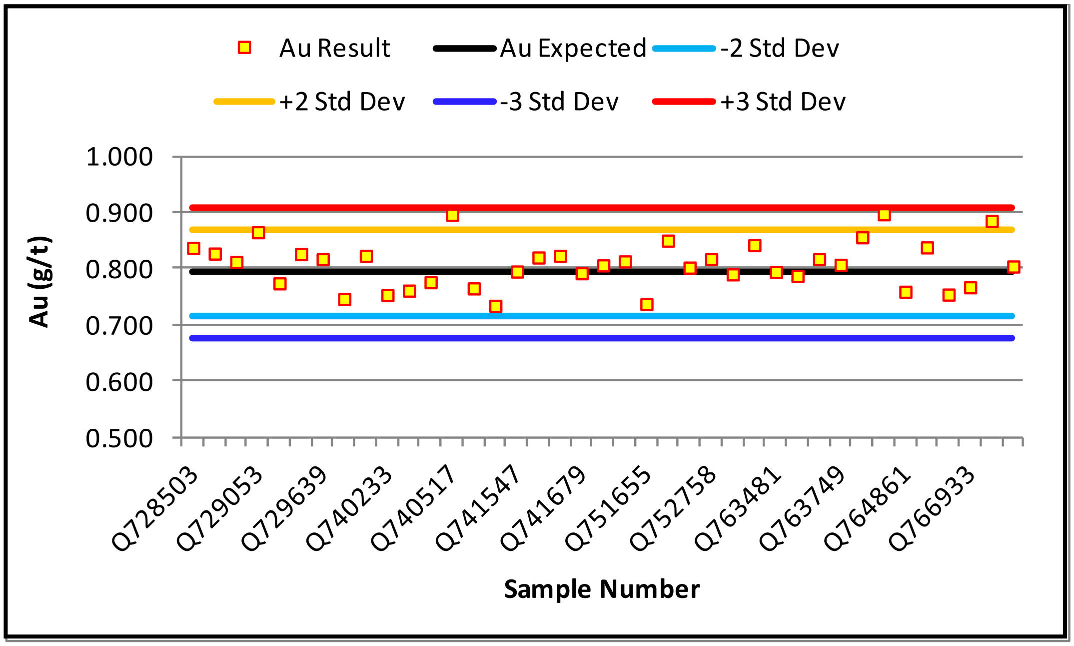
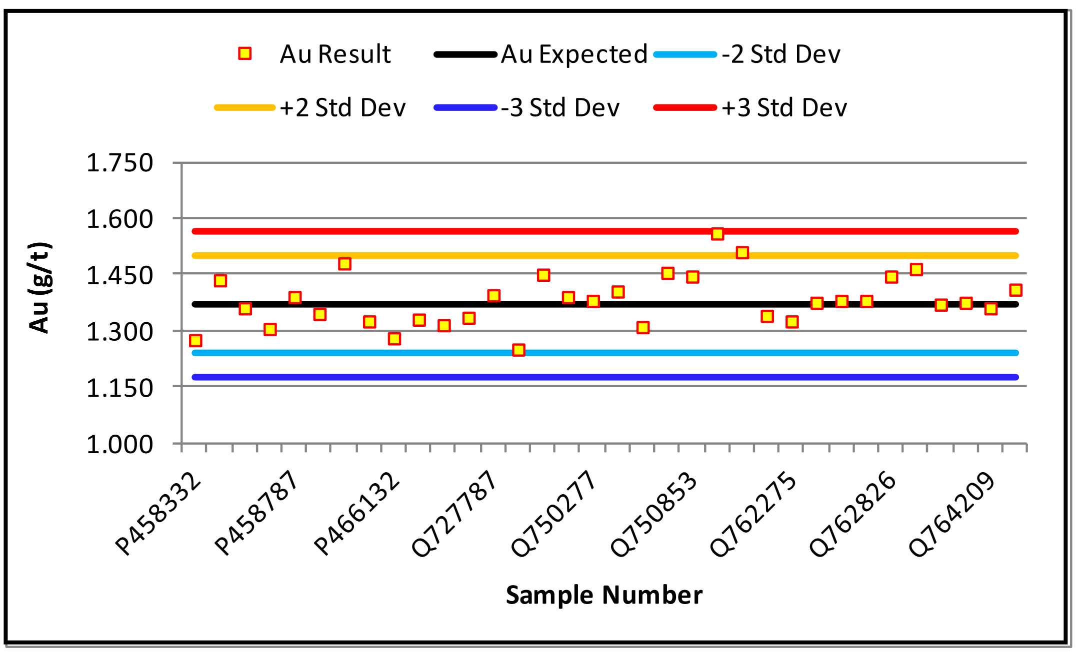


Figure 11.40 CM-17 SRM Performance – Gold



11.3.7 2012-2013 Duplicate Sample Performance

Seabridge collected a duplicate core sample at a frequency of about one duplicate for every 51 regular samples. The procedure consisted of taking one half of the sawn core and then sawing one of those halves into two pieces, each representing one quarter of the original HQ or NQ core. One sample was submitted as an "original" and the other as a "duplicate". Table 11.8 shows basic descriptive statistics for the original and duplicate core samples.

The mean grade of the original quarter core sample was lower than the duplicate quarter core for gold, copper, and silver. The correlation coefficient was above 90% for all but gold which was about 67%. QQ plots were generated for each the original-duplicate samples for each metal to examine the distribution of grades for each population. Figure 11.41 through Figure 11.44 show QQ plots for copper, gold, silver, and molybdenum, respectively.

Table 11.8 Summary of Duplicate Samples Submitted in 2012-2013

Parameter	Gold (g/t)		Copper (%)		Silver (g/t)		Molybdenum (%)	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
Count	281	281	281	281	281	281	281	281
Minimum	0.003	0.003	0.000	0.000	0.1	0.0	0.0001	0.0001
Maximum	6.430	6.270	2.390	2.520	21.6	17.2	0.0190	0.0210
Mean	0.217	0.221	0.266	0.273	1.4	1.5	0.0018	0.0017
Std Dev	0.456	0.467	0.342	0.340	2.0	2.2	0.0029	0.0027
CV	2.100	2.109	1.286	1.247	1.5	1.5	1.6337	1.5740
1st Quartile	0.033	0.033	0.017	0.017	0.4	0.4	0.0001	0.0001
Median	0.109	0.110	0.128	0.141	0.9	0.9	0.0006	0.0008
3rd Quartile	0.247	0.243	0.387	0.415	1.6	1.6	0.0020	0.0020
Correlation Coefficient	0.668		0.988		0.917		0.944	
Mean Grade Difference	-1.92%		-2.59%		-6.56%		5.70%	

Figure 11.41 2012-2013 Duplicate vs. Original Sample QQ Plot – Copper

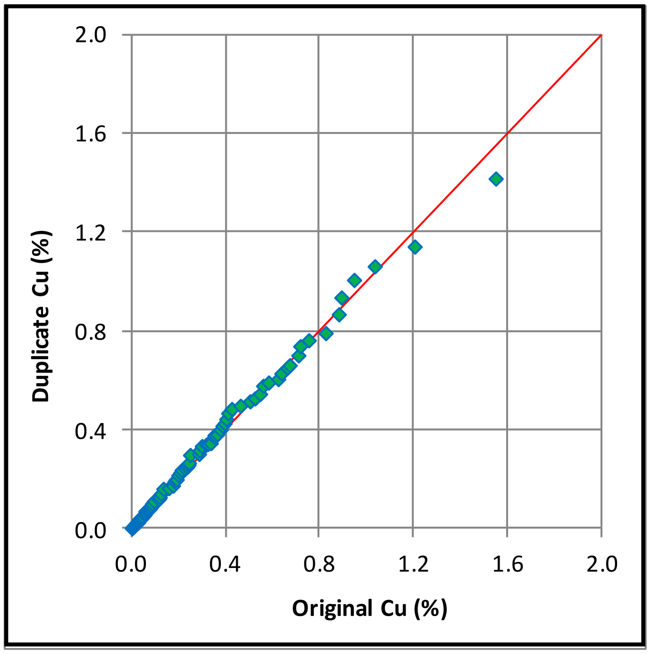


Figure 11.42 2012-2013 Duplicate vs. Original Sample QQ Plot – Gold

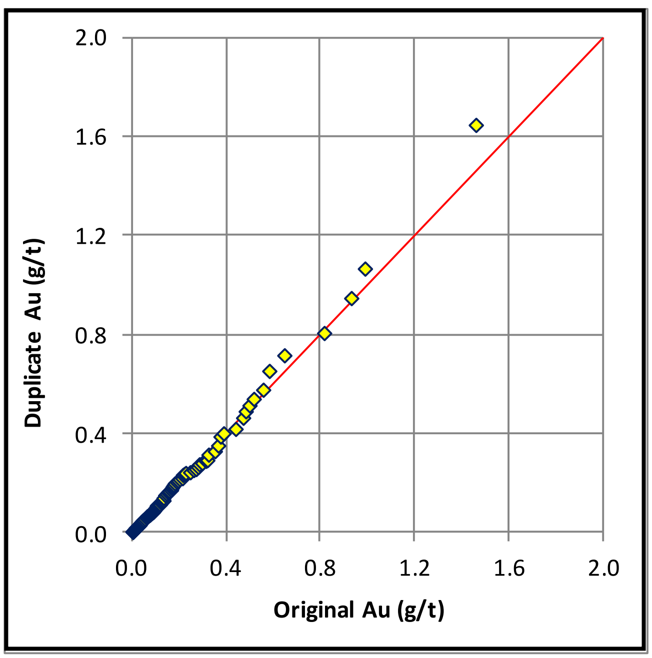


Figure 11.43 2012-2013 Duplicate vs. Original Sample QQ Plot – Silver

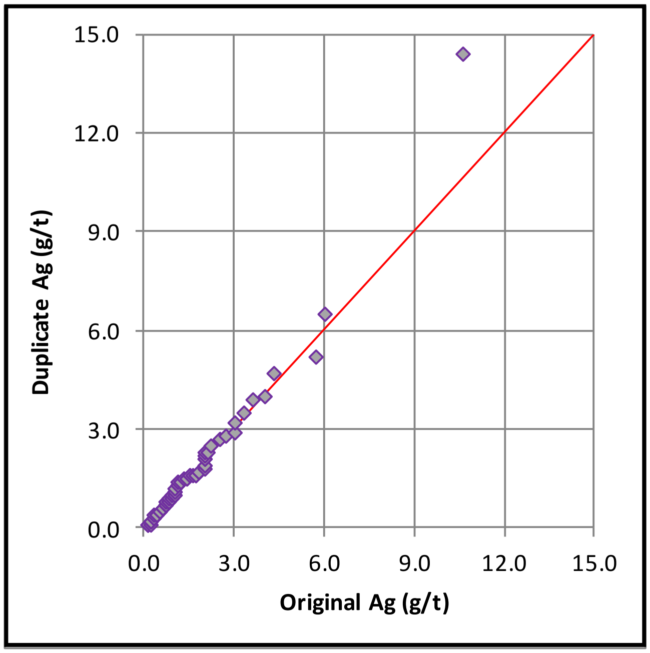
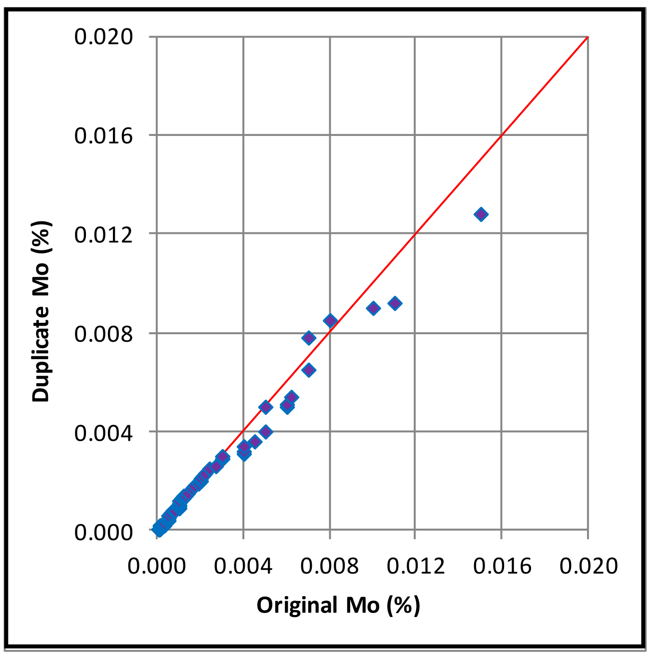


Figure 11.44 2012-2013 Duplicate vs. Original Sample QQ Plot – Molybdenum



11.3.8 2012-2013 Check Assay Results

As a part of their QA/QC program Seabridge randomly submitted 10% of the ALS pulps that were assayed for their 2012-2013 Deep Kerr drilling campaigns to AcmeLabs for check assay purposes. Table 11.9 compares basic descriptive statistics for copper, gold, silver, and molybdenum assays from the primary lab (ALS) and the check lab (AcmeLabs). Figure 11.45 to Figure 11.48 show the QQ plots for copper, gold, silver, and molybdenum, respectively.

Table 11.9 2012-2013 Check Assay Results - ALS vs. AcmeLabs

Parameter	Gold (g/t)		Copper (%)		Silver (g/t)		Molybdenum (ppm)	
	ALS	Acme	ALS	Acme	ALS	Acme	ALS	Acme
Count	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500
Minimum	0.003	0.003	0.000	0.000	0.10	0.15	0.5	0.5
Maximum	11.500	11.000	8.390	7.672	80.00	79.00	370.0	390.0
Mean	0.225	0.225	0.291	0.279	1.59	1.64	17.6	18.8
Q1	0.040	0.041	0.018	0.018	0.40	0.50	1.0	2.0
Median	0.129	0.128	0.144	0.139	0.90	1.00	7.0	7.0
Q3	0.266	0.269	0.430	0.416	1.60	1.80	20.0	22.0
Standard Deviation	0.45	0.44	0.42	0.41	4.04	4.02	29.00	30.29
CV	2.01	1.96	1.44	1.45	2.54	2.45	1.65	1.61
Correlation Coefficient	0.993		0.998		0.994		0.968	
Mean Grade Difference	0.02%		4.18%		-3.05%		-6.58%	

Figure 11.45 2012-2013 Check Assay QQ Plot - Cu

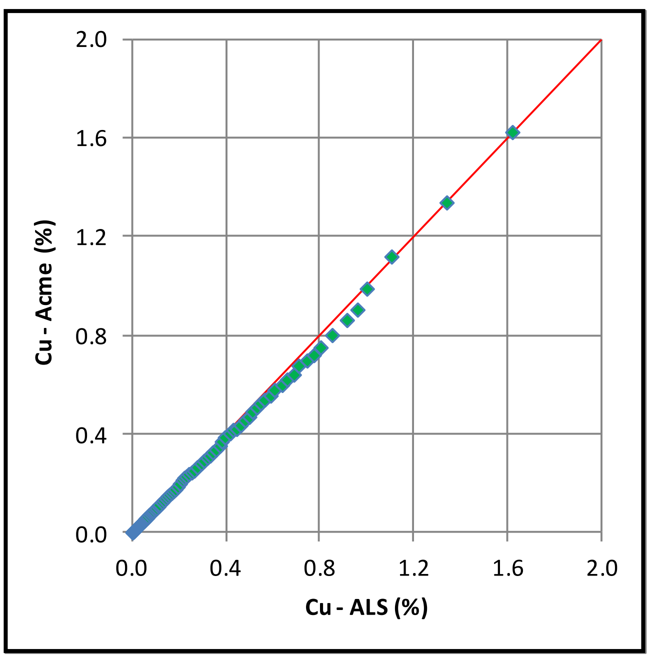


Figure 11.46 2012-2013 Check Assay QQ Plot - Au

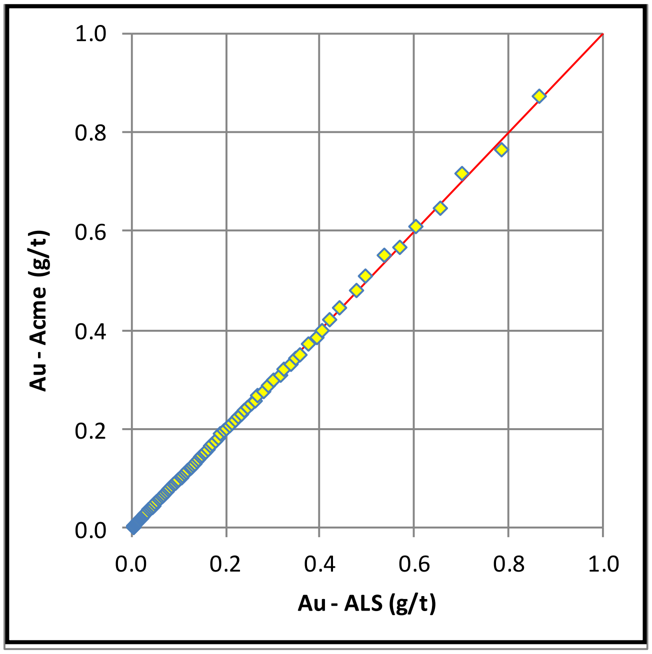


Figure 11.47 2012-2013 Check Assay QQ Plot - Ag

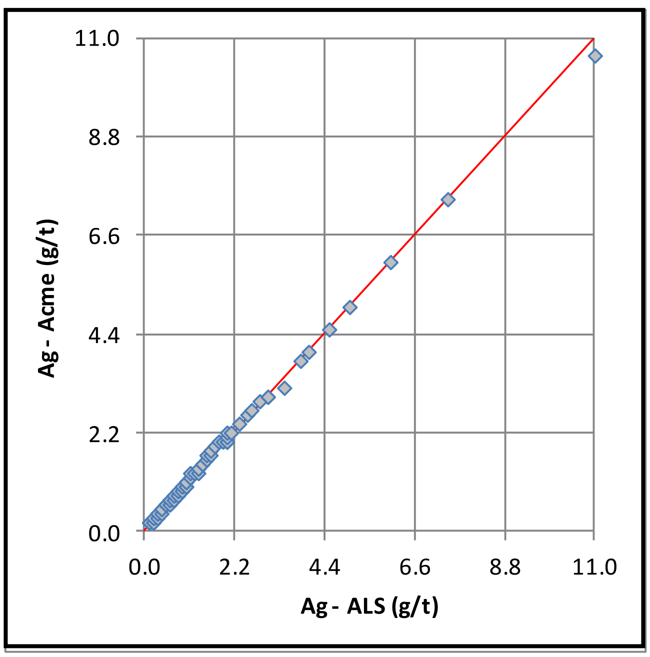
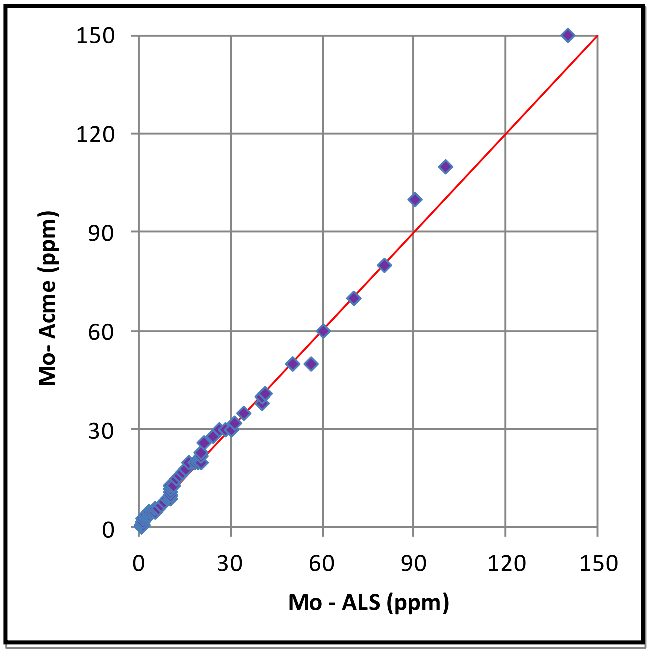


Figure 11.48 2012-2013 Check Assay QQ Plot - Mo



In general there is a very good correlation between ALS and AcmeLabs in assaying the same pulp. The average gold assays were less than 1% of one another while copper and silver showed more variance. Copper, as assayed by ALS, shows a slight high bias with respect to AcmeLabs results above a 0.5% cut-off. Most of the ALS copper assays were completed using ICP methods (ME-ICP41). Higher grade samples were re-assayed using the ALS "ore grade" protocol ME-OG46. Both methods used varying concentrations of aqua regia for digesting the sample. AcmeLabs also used 1:1:1 aqua regia for sample digestion for their ICP protocol (1D01). However, AcmeLabs used a four acid digestion for their "ore grade" copper protocol (8TD). The four acid digestion method should liberate more copper metal than aqua regia, so intuitively the AcmeLabs copper values should be higher than ALS, but that is not the case. Eight of the 13 commercial certified standard reference materials utilized by Seabridge had two certified copper values for the standard. One certified copper value was based on aqua regia digestion and the other certified value was based on a four acid digestion. The eight standards with two certified copper values were subjected to round robin assaying by various commercial labs. The expected copper values for two of the standards based on the different digestion methods yielded identical values. For two of the standards the four acid digestion gave a slightly higher expected value and four cases the aqua regia digestion method gave a slightly higher value. RMI recommends that Seabridge investigate these apparent differences and perhaps adjust their copper assaying methods for the 2014 drilling season.

11.3.9 2012-2013 Corrective Action

A total of 110 batches of samples were submitted to ALS over the course of the 2012 and 2013 drilling programs for Deep Kerr. Of these batches, initial results from the laboratory returned 33 batches where one or more standards or blanks were deemed to be out of compliance. For a standard, this meant that its result for either gold or copper was more than three standard deviations from its expected value.

If a single standard was deemed outside of limits within a batch containing more than two additional standards, where the remaining standards passed and there were no mineralized intercepts within that batch, the entire batch was deemed to have passed. This occurred in three batches. If a single standard or blank was deemed outside of limits and the batch contained mineralized intervals, the control sample plus three to five samples above and below the control sample in question were re-analyzed. In a case where two consecutive standards in a batch failed, then 10 samples above and below the two standards were re-run along with all of the samples between the standards.

When the results of the re-analysis were received from the lab the new values were compared to the original values and to the expected values for the standards. If the standard came back within tolerance and there was no bias noted in the surrounding samples, the laboratory was requested to reissue the certificate with the revised values for all samples. If there was any indication of bias in the results for the samples, then the entire sequence was re-analyzed and a new certificate issued. If the standard and associated samples returned similar values to the original run, then it was deemed that the standard was out of tolerance but consistent and the batch passed. In those cases the certificate that contained the lowest recorded value for the standard and the sample results that went along with that result were considered final for the database. This occurred five times out of the 110 batches.

There were several instances where blanks returned higher than expected values, especially for copper. In those cases the samples were sent back to the lab for re-analysis under similar conditions to the standards as described above, with the criteria being that the blank was reporting more than 20 times minimum detection limit. In most cases it was determined that there was minor residual cross contamination from the previous high-grade samples (less than 0.002%). While in a few cases there was found to be minor copper (less than 0.001%) in the blank material.

11.3.10 2012-2013 Qualified Person's Opinion

In the opinion of the QP responsible for this section of this Technical Report, sample security, sample preparation, analytical procedures, and QA/QC protocols/results associated with Seabridge's 2012-2013 Deep Kerr drilling campaigns were adequate and consistent with standard industry practices. RMI also believes that the assays are suitable to be used to estimate Mineral Resources.

11.4 2014-2015 Sample Preparation Methods and Procedures

This section summarizes topics regarding sample preparation methods and quality control measures employed for the 2014-2015 KSM drilling campaigns.

11.4.1 2014-2015 Statement on Sample Preparation Personnel

As in previous years, labourers contracted from Northern Labour Force Corporation conducted all initial sample preparation (sawing and bagging) and were trained by and under the direct supervision of geologists employed by Seabridge. Drill core and quality control samples were shipped to Seabridge's secured warehouse located in Stewart, BC. From there, the samples were shipped to the ALS preparation facility located in Terrace, BC, either by a commercial trucking firm or by ALS personnel. The prepped samples were then shipped by ALS to their assay laboratory located in North Vancouver, BC for analysis.

11.4.2 2014-2015 Sample Preparation and Dispatch

The same sample preparation methods discussed in Section 11.3.2 (2012-2013) apply to the 2014-2015 drilling campaigns. There were two minor exceptions to the dispatch protocols. Instead of securing individual rice bags with a tamper proof seal, the wooden tote boxes that were used to ship rice bags containing drill core samples were secured with a seal. ALS Chemex shutdown their sample receiving facility in Stewart, BC prior to the 2013 drilling campaign. After that date, Seabridge shipped samples to their secure warehouse located in Stewart where samples were shipped to Chemex's sample preparation facility located in Terrace, BC by either a commercial trucking company or ALS Chemex personnel.

11.4.3 2014-2015 Analytical Procedures

ALS Chemex and AcmeLabs remained Seabridge's primary and secondary labs for the 2014-2015 drilling campaigns. The information summarized in Section 11.3.3 for the 2012-2013 drilling data applies to the 2014-2015 campaigns.

11.4.4 2014-2015 Quality Control Measures

Seabridge maintained the same quality control procedures for their 2014-2015 KSM drilling campaigns that were described for the 2012-2013 campaigns (Section 11.3.4). A total of 20,644 assays were submitted to ALS during the 2014-2015 KSM drilling campaigns. The quality control measures that were used consisted of:

- the submission of barren or blank material at a frequency of one blank for about every 32 samples
- the submission of one SRM for about every 33 samples
- the collection and submission of duplicate quarter core samples at a frequency of one duplicate sample for about every 37 samples
- the submission of approximately 9% of the ALS pulps to AcmeLabs for check assay purposes.

Table 11.10 shows the type and number of control samples that were submitted to either ALS or AcmeLabs for the 2014-2015 KSM drilling campaigns.

Table 11.10 Summary of Control Samples Submitted in 2014-2015

Type of Control Sample	Number of Control Samples Submitted	Frequency (Control sample per regular sample)
Blank	638	1 in 32
Standard	634	1 in 33
Duplicate	552	1 in 37
Check Assay	2,104	1 in 10
Total Regular Samples	**20,644**	**n/a**

11.4.5 2014-2015 Blank Sample Performance

Seabridge submitted three different barren or blank samples during their 2014-2015 KSM drilling programs. They included commercial landscaping materials consisting of crushed marble (Blank 5), commercial landscape limestone (Blank 9), and crushed diorite from a quarry located near Stewart, BC (Blank 10). Sixty percent of the blanks submitted were from Blank 10, 35% from Blank 9, and 5% from Blank 5. The blanks were submitted along with the sawn drill core at a frequency of one blank for every 30 samples.

Figure 11.49 and Figure 11.50 show the analyzed blank values for gold and copper, respectively.

Figure 11.49 2014-2015 Blank Performance – Gold

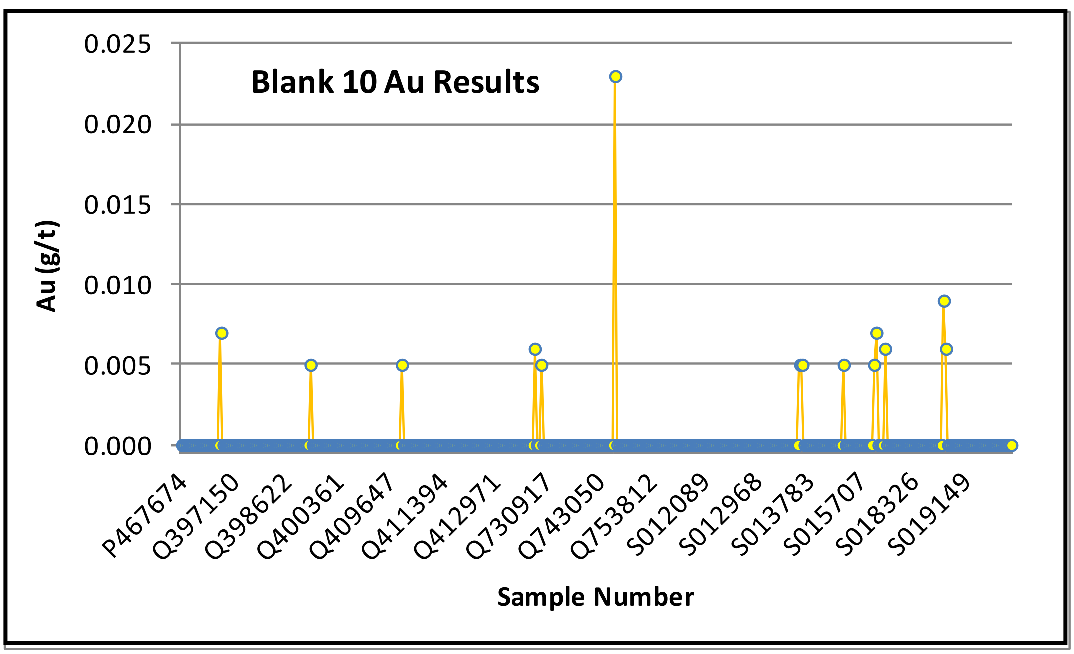
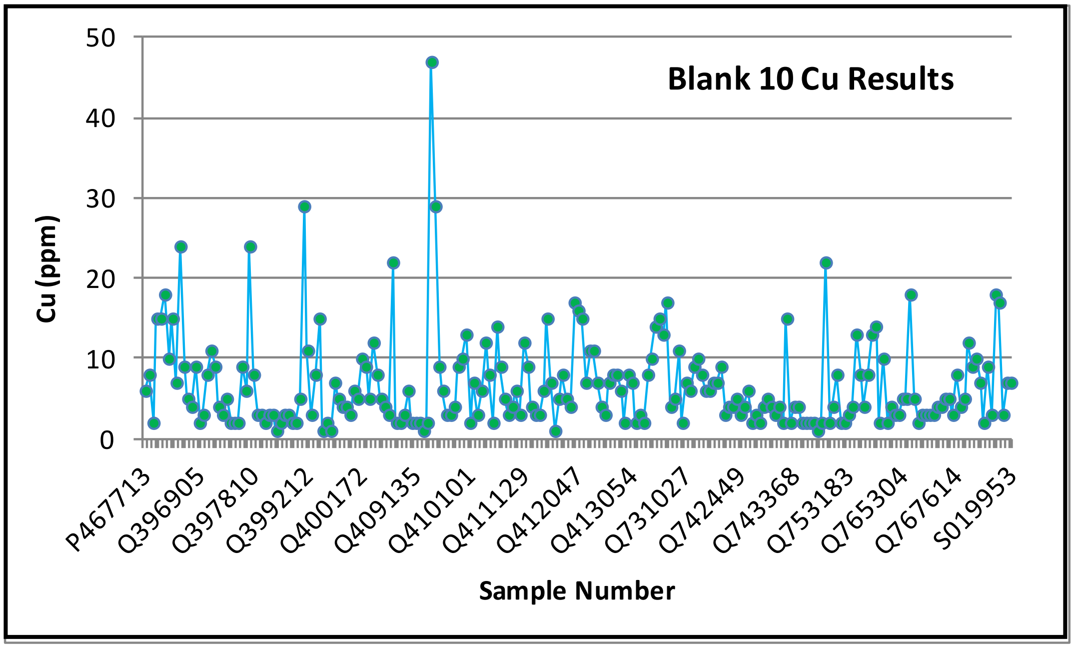


Figure 11.50 2014-2015 Blank Performance – Copper



11.4.6 2014-2015 STANDARD REFERENCE SAMPLE PERFORMANCE

A total of 634 standard reference materials were submitted by Seabridge for their 2014-2015 KSM drilling campaigns. All but one of these SRMs were purchased as certified standards from commercial vendors. One SRM was prepared from Seabridge samples collected from the Kerr-Sulphurets-Mitchell deposits (SEA-KSM) and was certified by Smee & Associates and CDN Resource. Table 11.11 summarizes the expected values and two standard deviations associated with the 634 SRMs that were submitted in 2014-2015.

SRM performance graphs are shown for five SRMs (SEA-KSM, CM-27, CM-25, CM-23, and CM-19) for copper and gold only. These five standards represent 70% of the standards that were submitted and all have over 50 determinations. Figure 11.51 through Figure 11.59 show the performance for copper and gold for the five SRMs mentioned above.

Table 11.11 Summary of Control Samples Submitted in 2014-2015

Standard Name	Source of Material	Number Submitted	Expected Values					2 Standard Deviations				
			Au (g/t)	Cu (%)	Mo (%)	Ag (g/t)	Pb (%)	Au (g/t)	Cu (%)	Mo (%)	Ag (g/t)	Pb (%)
CM-14	CDN Resource Laboratories Ltd.	6	0.792	1.058	0.042	-	-	0.078	0.062	0.002	-	-
CM-15	CDN Resource Laboratories Ltd.	3	1.253	1.280	0.054	-	-	0.118	0.090	0.004	-	-
CM-19	CDN Resource Laboratories Ltd.	60	2.110	2.020	0.106	-	-	0.220	0.070	0.008	-	-
CM-23	CDN Resource Laboratories Ltd.	59	0.549	0.472	0.025	-	-	0.060	0.026	0.002	-	-
CM-25	CDN Resource Laboratories Ltd.	61	0.228	0.191	0.019	-	-	0.030	0.006	0.002	-	-
CM-27	CDN Resource Laboratories Ltd.	60	0.636	0.592	0.051	-	-	0.068	0.030	0.004	-	-
CM-29	CDN Resource Laboratories Ltd.	43	0.720	0.734	0.053	-	-	0.068	0.030	0.004	-	-
CM-30	CDN Resource Laboratories Ltd.	51	1.300	0.730	0.070	15.900	0.273	0.120	0.034	0.004	1.300	0.014
CM-35	CDN Resource Laboratories Ltd.	49	0.324	0.248	0.029	2.700	-	0.032	0.012	0.002	0.400	-
CM-36	CDN Resource Laboratories Ltd.	37	3.160	0.227	-	2.000	-	0.034	0.012	-	0.200	-
SEA-KSM	CDN Resource Laboratories Ltd.	205	0.774	0.204	0.007	-	-	0.062	0.010	0.001	-	-

Figure 11.51 2014-2015 SEA-KSM SRM Performance – Copper

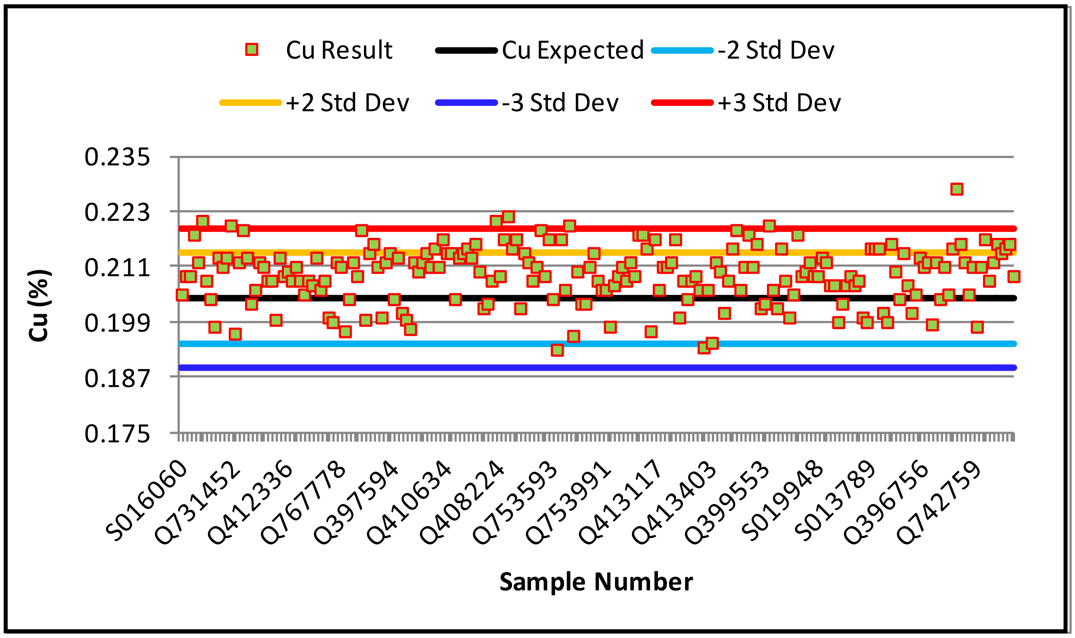


Figure 11.52 2014-2015 CM-27 SRM Performance – Copper

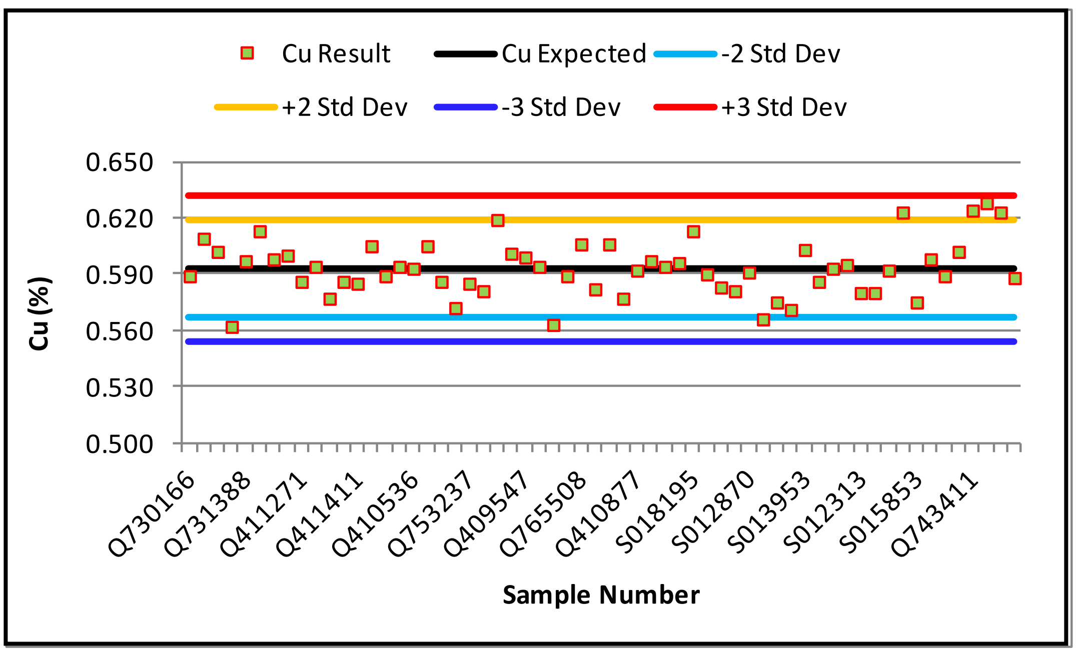


Figure 11.53 2014-2015 CM-25 SRM Performance – Copper

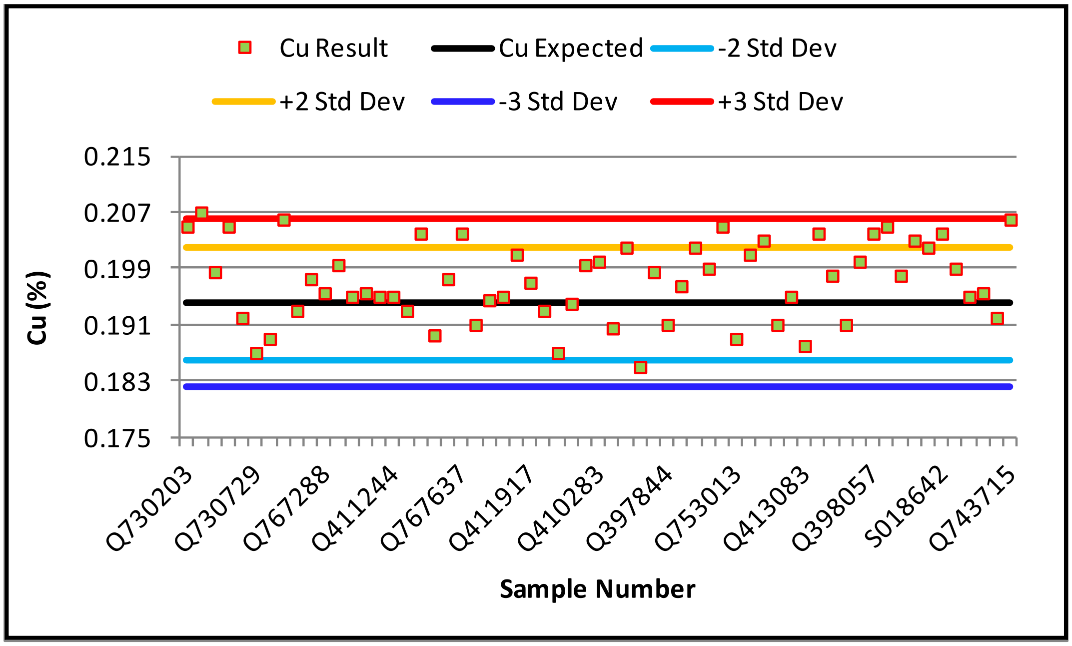


Figure 11.54 2014-2015 CM-23 SRM Performance – Copper

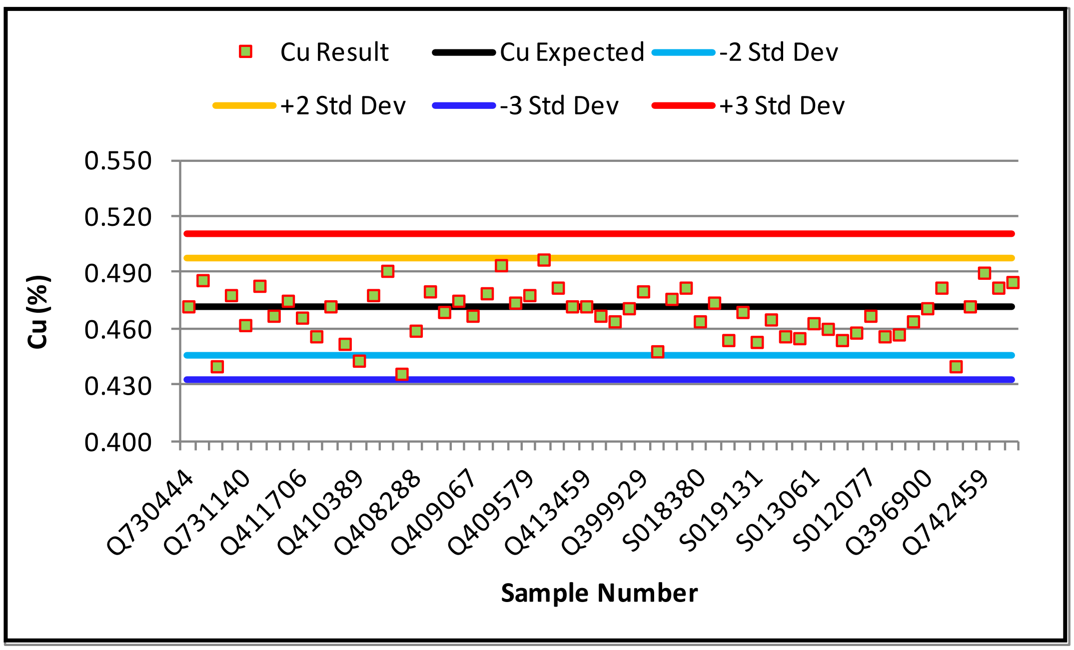


Figure 11.55 2014-2015 CM-19 SRM Performance – Copper

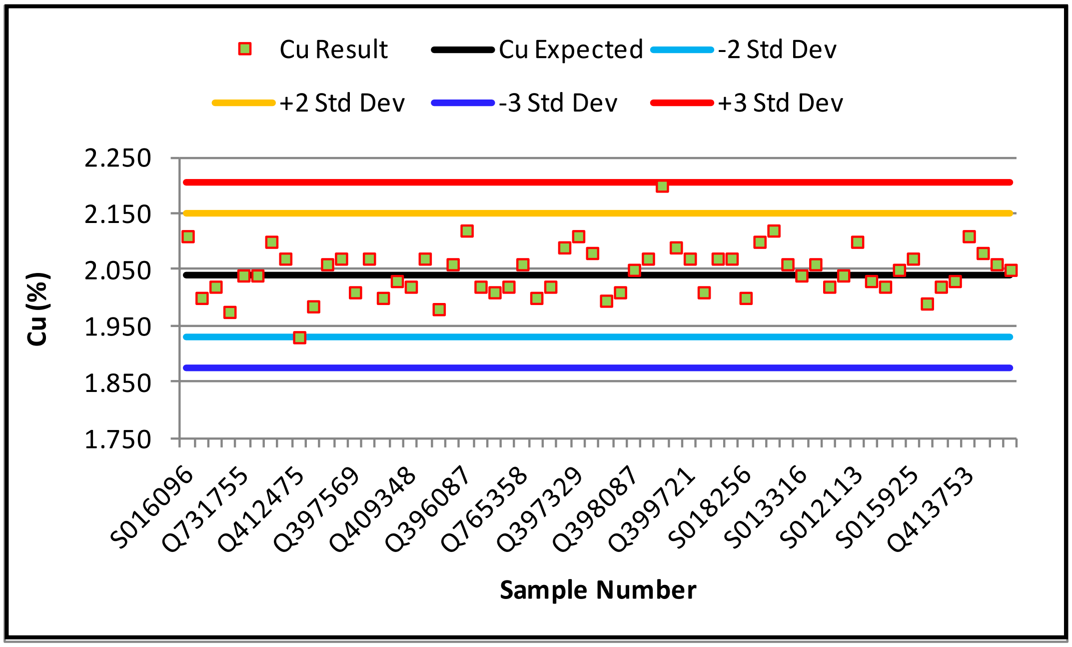


Figure 11.56 2014-2015 SEA-KSM SRM Performance – Gold

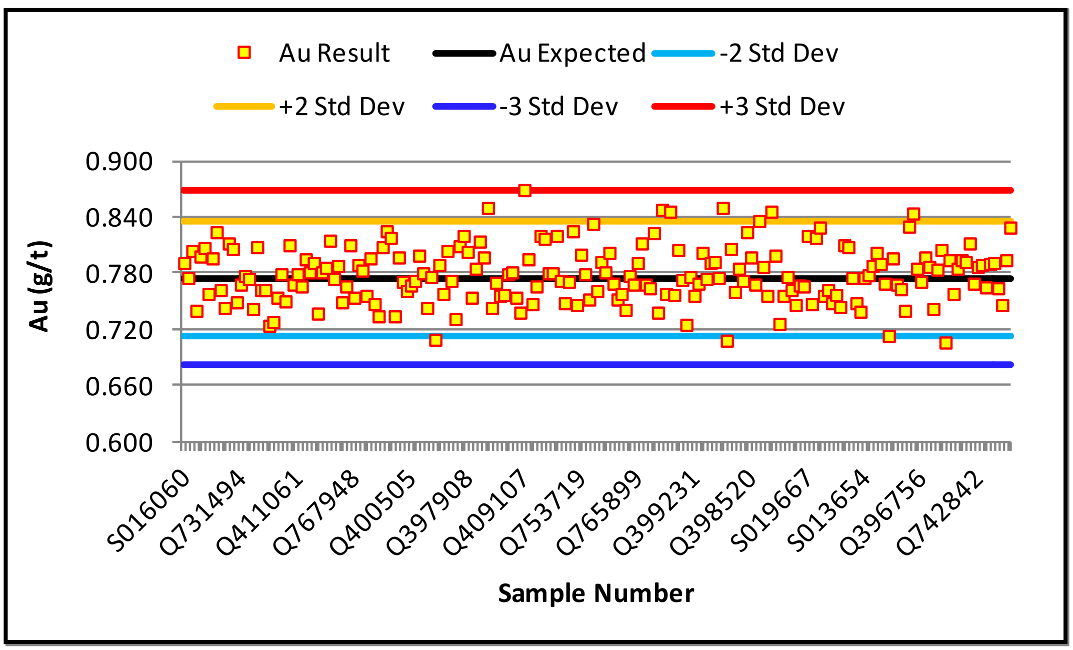


Figure 11.57 2014-2015 CM-27 SRM Performance – Gold

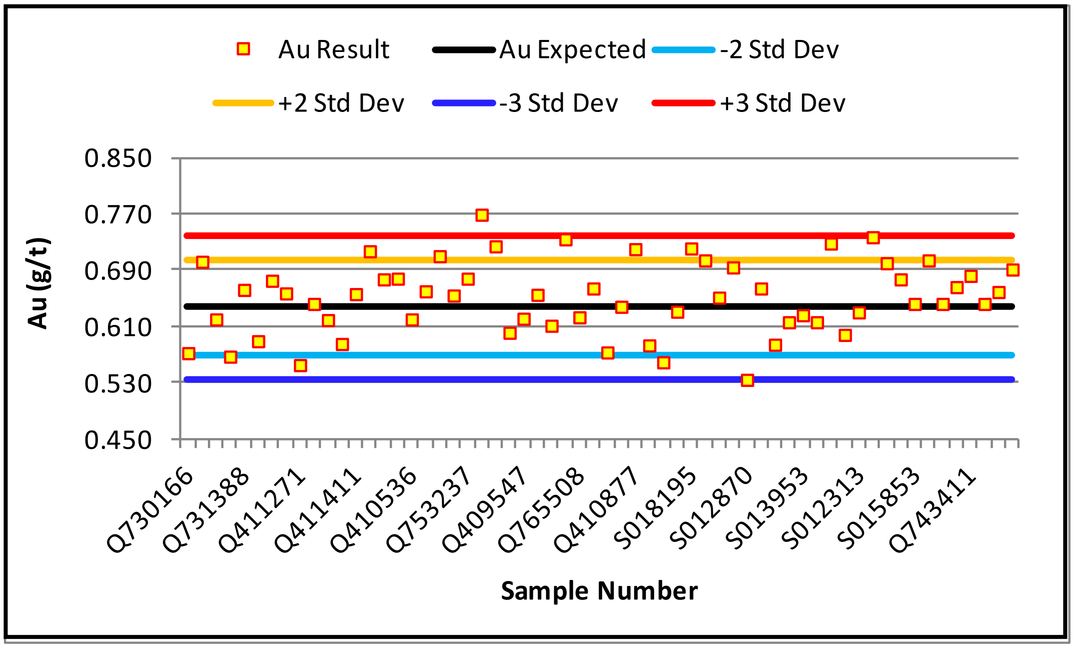


Figure 11.58 2014-2015 CM-25 SRM Performance – Gold

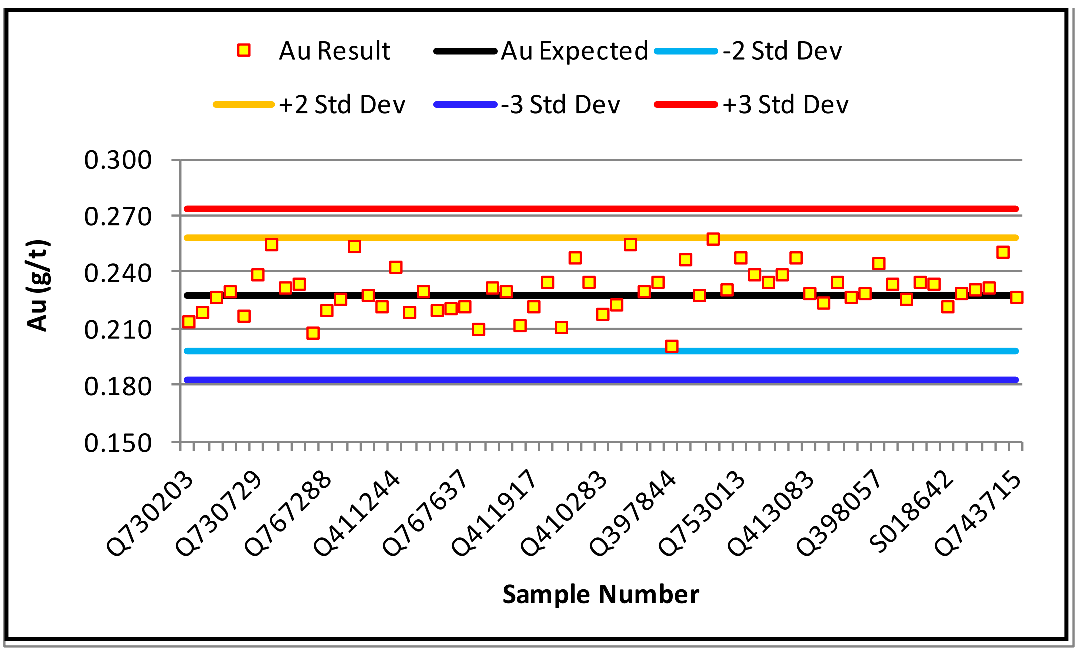


Figure 11.59 2014-2015 CM-23 SRM Performance – Gold

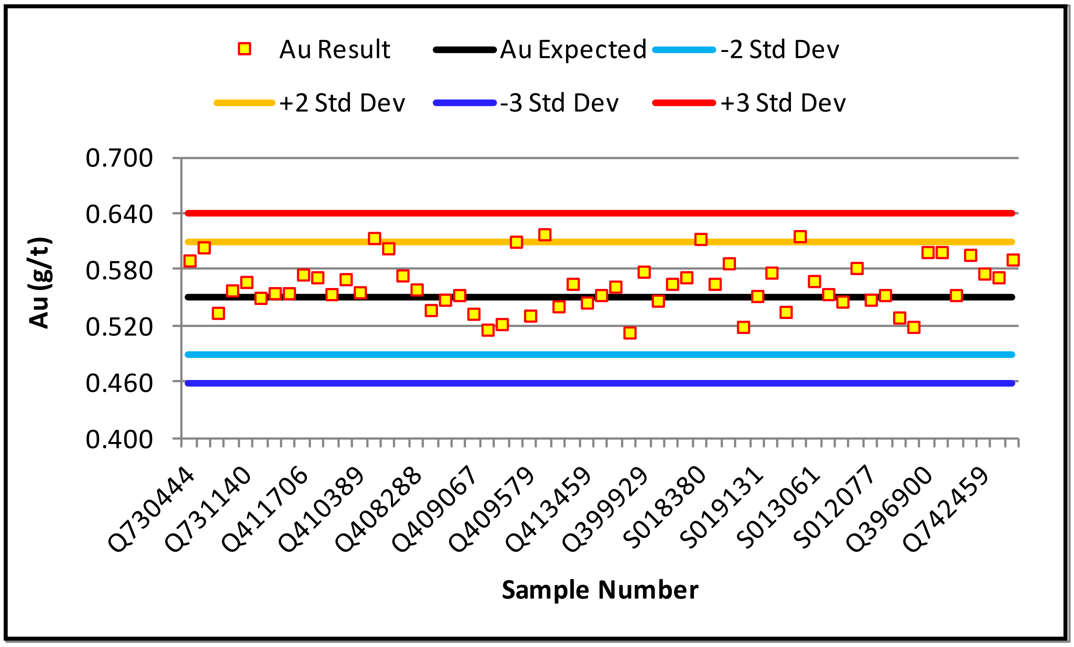
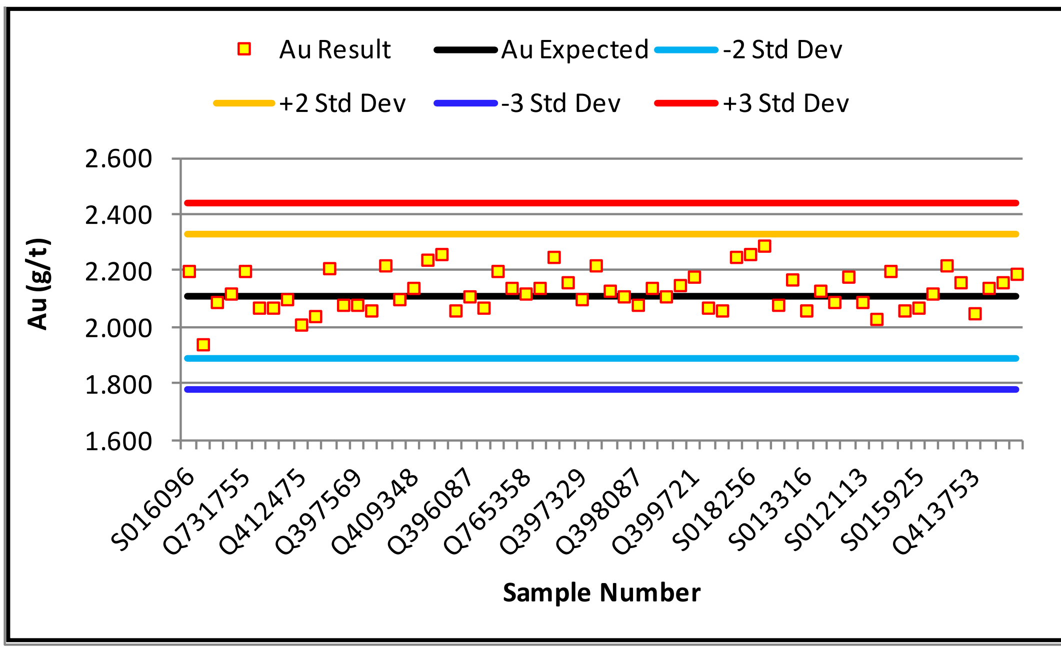


Figure 11.60 2014-2015 CM-19 SRM Performance – Gold



11.4.7 2014-2015 Duplicate Sample Performance

Seabridge collected a duplicate core sample at a frequency of about one duplicate for every 37 regular samples. The procedure consisted of taking one half of the sawn core and then sawing one of those halves into two pieces, each representing one quarter of the original HQ or NQ core. One sample was submitted as an "original" and the other as a "duplicate". Table 11.12 shows basic descriptive statistics for the original and duplicate core samples.

Table 11.12 Summary of Duplicate Samples Submitted in 2014-2015

Parameter	Gold (g/t)		Copper (%)		Silver (g/t)		Molybdenum (ppm)	
	Original	Duplicate	Original	Duplicate	Original	Duplicate	Original	Duplicate
Count	552	552	552	552	552	552	552	552
Minimum	0.003	0.003	0.000	0.000	0.1	0.1	0.5	0.5
Maximum	7.000	7.970	1.280	1.185	78.0	57.2	1400.0	1405.0
Mean	0.265	0.265	0.173	0.174	2.1	2.2	22.5	23.3
Std Dev	0.504	0.508	0.208	0.208	4.6	4.2	73.2	73.7
CV	1.905	1.918	1.201	1.199	2.2	1.9	3.3	3.2
1st Quartile	0.052	0.050	0.017	0.017	0.5	0.5	1.0	1.0
Median	0.133	0.140	0.092	0.091	1.0	1.1	6.0	7.0
3rd Quartile	0.309	0.298	0.238	0.242	2.0	2.2	19.3	20.0
Correlation Coefficient	0.907		0.986		0.972		0.966	
Mean Grade Difference	-0.05%		-0.19%		-0.69%		-3.64%	

The data in Table 11.12 show that mean grades of the original quarter core samples were slightly lower than the duplicate quarter but they are remarkably close for gold, copper, and silver. The correlation coefficient was above 90% for all metals. QQ plots were generated for each of the original-duplicate samples showed that there was a close comparison between the original and duplicate assay across the full grade ranges that were sampled.

11.4.8 2014-2015 Check Assay Results

As a part of their QA/QC program, Seabridge randomly submitted approximately 10% of the ALS pulps that were assayed for their 2014-2015 KSM drilling campaigns to AcmeLabs for check assay purposes. Table 11.13 compares basic descriptive statistics for copper, gold, silver, and molybdenum assays from the primary lab (ALS) and the check lab (AcmeLabs).

Table 11.13 2014-2015 Check Assay Results – ALS vs. AcmeLabs

Parameter	Gold (g/t)		Copper (%)		Silver (g/t)		Molybdenum (ppm)	
	ALS	Acme	ALS	Acme	ALS	Acme	ALS	Acme
Count	2,104	2,104	2,104	2,104	2,104	2,104	2,104	2,104
Minimum	0.003	0.003	0.000	0.000	0.1	0.1	0.5	0.5
Maximum	14.100	20.900	2.850	2.824	78.0	77.0	2320.0	2400.0
Mean	0.311	0.319	0.182	0.181	2.4	2.5	24.7	25.3
Std Dev	0.780	0.853	0.246	0.249	5.3	5.2	109.0	115.8
CV	2.508	2.673	1.356	1.377	2.2	2.1	4.4	4.6
1st Quartile	0.057	0.061	0.015	0.014	0.5	0.6	1.0	1.0
Median	0.144	0.149	0.095	0.092	1.0	1.2	5.0	5.0
3rd Quartile	0.319	0.323	0.248	0.244	2.3	2.7	20.0	20.0
Correlation Coefficient	0.983		0.999		0.989		0.996	
Mean Grade Difference	-2.6%		0.3%		-2.7%		-2.6%	

The data in Table 11.13 show that there is a very good correlation between the original sample assays (ALS Chemex) and the check assay (AcmeLabs). QQ plots show that there is a close comparison between the original and check assays across the full grade ranges that were sampled as illustrated by Figure 11.61 and Figure 11.62, which show the relationship between copper (Figure 11.61) and gold (Figure 11.62).

Figure 11.61 2014-2015 Check Assay QQ Plot – Copper

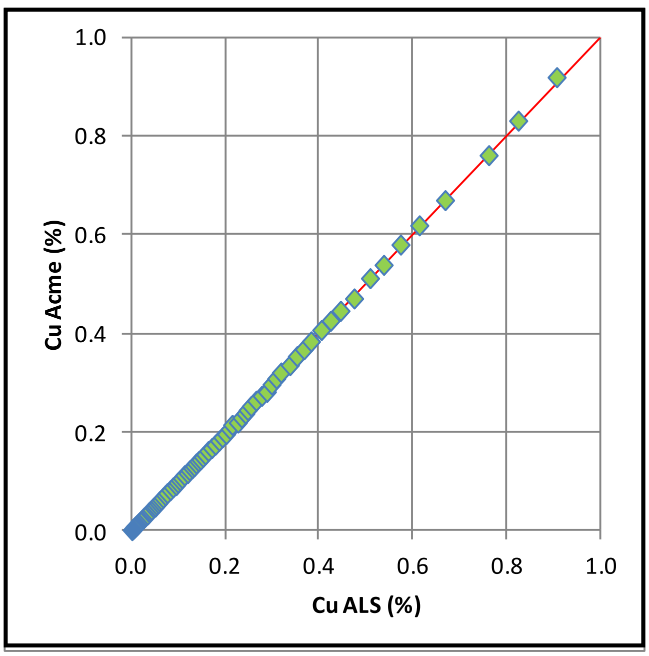
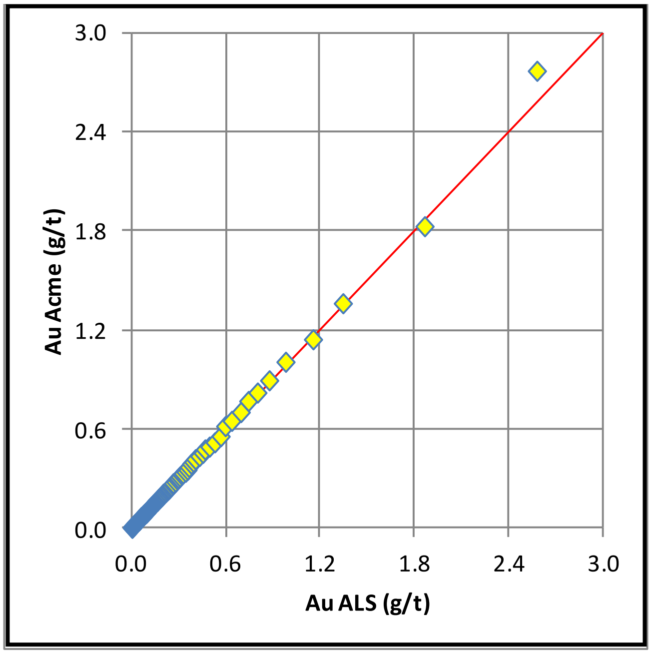


Figure 11.62 2014-2015 Check Assay QQ Plot – Gold



11.4.9 2014-2015 Corrective Action

A total of 264 batches of samples were submitted to ALS over the course of the 2014 and 2015 KSM drilling campaigns containing nearly 21,000 sawn core samples. Forty-three of the 264 batches contained one or more standards or blanks that were deemed to be out of compliance. Thirty-eight of the 47 control samples were resolved by re-assaying. Five of the nine unresolved control sample failures were not deemed to be material because the samples associated with the apparent failures represented non-mineralized intervals.

For a standard, a failure designation was assigned to the control sample if either the assayed gold or copper value was more than three standard deviations from its expected value. If a single standard was deemed outside of limits within a batch containing more than two additional standards, where the remaining standards passed and there were no mineralized intercepts within that batch, the entire batch was deemed to have passed. If a single standard or blank was deemed outside of limits, and the batch contained mineralized intervals the control sample plus three to five samples above and below, the control sample in question was re-analyzed. In a case where two consecutive standards in a batch failed, then 10 samples above and below the two standards were re-run along with all of the samples between the standards.

When the results of the re-analysis were received from the lab the new values were compared to the original values and to the expected values for the standards. If the standard came back within tolerance and there was no bias noted in the surrounding samples the laboratory was requested to reissue the certificate with the revised values for all samples. If there was any indication of bias in the results for the samples, then the entire sequence was re-analyzed and a new certificate issued. If the standard and associated samples returned similar values to the original run, then it was deemed that the standard was out of tolerance but consistent and the batch passed. In those cases, the certificate that contained the lowest recorded value for the standard, and the sample results that went along with that result, were considered final for the database

11.4.10 2014-2015 Qualified Person's Opinion

In the opinion of the QP responsible for this section of this Technical Report, sample security, sample preparation, analytical procedures, and QA/QC protocols/results associated with Seabridge's 2014-2015 KSM drilling campaigns were adequate and consistent with standard industry practices. The QP also believes that the assays are suitable to be used to estimate Mineral Resources.

12.0 DATA VERIFICATION

Previous NI 43-101 Technical Reports prepared by the QP responsible for this section discussed various data verification measures that were undertaken for the Kerr, Sulphurets, Mitchell, and Iron Cap mineralized zones. This section summarizes previous (pre-2012 and 2012-2013) and current (2014-2015) data reviews that were completed by the QP responsible for this section of this Technical Report. Most of the pre-2012 and 2012-2013 sections (12.1 and 12.2) were taken nearly verbatim from previously filed Technical Reports.

12.1 PRE-2012 DATA VERIFICATION

12.1.1 ASSAY VERIFICATION

Prior to the 2011 drilling campaign, the QP responsible for this section of this Technical Report personally compared assay certificates against the Seabridge drill hole assay database. Approximately 7,500 assay records from 50 drill holes representing about 15,000 meters of drilling data were reviewed and discussed in prior Technical Reports (Lechner 2007; Lechner 2008; Lechner 2009; and Lechner 2010). Minor errors associated with the 2008 campaign were discovered and corrected by Seabridge.

The QP 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. The QP 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 Pre-2012 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
Total	-	**1,013**	**1,981**	**1**	**1**	**2**	**1**

The QP notes that the errors which 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. Those errors were corrected in the drill hole database.

12.1.2 Drill Hole Logs

The QP responsible for this section completed numerous comparisons between drill hole core and paper copies of drill hole logs (lithology, alteration, mineralization, etc.) during site visits conducted in 2006 through 2011. Core recovery and RQD values were randomly checked by the QP during those site visits. It is the QP's opinion that the pre-2012 drill hole logging was completed in a professional manner and fairly represents the geology of the KSM deposits.

12.1.3 Down-hole Surveys

The QP responsible for this section completed numerous comparisons between the driller's handwritten down-hole survey records and the Seabridge electronic database. Between the 2006 and 2011 drilling seasons very few discrepancies were discovered. Those discrepancies were researched by Seabridge's geological staff and appropriate corrections completed.

12.1.4 Quality Assurance/Quality Control

The QP was provided with QA/QC data for the pre-2012 drilling programs. This data was analyzed with the results and conclusions presented in Section 11.0.

12.1.5 Topographic 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:

- the fact that no transform of the Z-coordinate was considered by the Canadian National Transformation software

- the inaccuracy of the initial GPS elevation
- the fact that many of the holes were surveyed immediately below the drill deck and not ground level or "stick-up" differences magnified by steep terrain.

12.2 2012-2013 Data Verification

Data from the 2012-2013 KSM drilling campaigns were verified by the QP responsible for this section and discussed in the March 2014 Technical Report (Lechner 2014) and summarized in Sections 12.2.1 through 12.2.4.

12.2.1 Assay Verification

The QP responsible for this Technical Report performed an audit of the 2012-2013 Deep Kerr drill hole database by comparing ALS certified assay results with values stored in Seabridge's electronic database. The QP manually checked gold and copper assays from five of Seabridge's 2012-2013 drill holes for verification. The data that were verified are summarized in Table 12.1 by drill hole. The data shown in Table 12.2 represent about 11% of the 2012-2013 Seabridge Deep Kerr assay data.

Table 12.2 2012-2013 Database Verification

Drill Hole	Number Checked	Meters Checked	Au Errors	Cu Errors	Ag Errors	Mo Errors
K-12-19	150	292.77	0	0	0	0
K-13-23A	420	820.80	0	0	0	0
K-13-28	565	1,075.80	0	0	0	0
K-13-28C	182	342.60	0	0	0	0
K-13-32A	250	489.60	0	0	0	0
Total	**1,567**	**3,021.57**	**0**	**0**	**0**	**0**

The QP notes that no errors were discovered with the four grade items that were checked. It is the QP's opinion that the 2012-2013 Deep Kerr electronic assay database is accurate and suitable to use for estimating Mineral Resources.

12.2.2 Drill Hole Logs

During the QP's September 2013 site visit portions of five diamond core holes were compared against Seabridge drill hole logs for holes K-13-23, K-13-23A, K-13-24, K-13-25, and K-13-29. The QP found that the logs reasonably described lithology, alteration, structural elements, and provided adequate descriptions of mineralized intervals.

The QP randomly selected fifteen core run intervals from three drill holes (K-12-23, K-13-25, and K-13-29). Recovered core was measured and compared against the geotechnical logs. No material differences were noted and those differences which were noted were attributed to the QP measuring sawn core while the original recovered core measurements was made from un-sawn core. It is the opinion of the QP that the

Seabridge core hole logs have been prepared in a professional manner and are suitable for subsequent geological interpretation.

12.2.3 DOWN-HOLE SURVEYS

The QP compared down-hole survey records (hand written field recording cards completed by the drill contractor) against the electronic database for three core holes (K-13-23, K-13-23A, and K-13-29). No errors were noted, although three DeviTool PeeWee survey records were recorded in the database with no paper backup record in the file. All of the down-hole survey records were checked by the QP for suspect entries by comparing adjacent records for abnormal azimuth and inclination deviations using a software routine written by RMI. Seabridge had flagged fifteen records as being suspicious and the QP's independent review likewise found them to be deviant. Those records were removed from the database.

Several minor discrepancies were noted regarding the drill hole collar and first down-hole survey azimuth for 11 drill holes. Those discrepancies were researched and corrected by Seabridge and the QP.

12.2.4 QUALITY ASSURANCE/QUALITY CONTROL

The QP was provided with QA/QC data for the 2012-2013 drilling programs. These data were analyzed with the results and conclusions presented in Section 11.0.

12.3 2014-2015 DATA VERIFICATION

The QP completed a review of various data associated with the 2014 and 2015 KSM drilling campaigns. The data verification procedures and results undertaken are summarized in Sections 12.3.1 through 12.3.4.

12.3.1 ASSAY VERIFICATION

The QP responsible for this section undertook a comprehensive review of the 2014 and 2015 drill hole assay database by completing a 100% check of the gold, copper, silver, and molybdenum assays that were used to estimate mineral resources. The QP obtained all of the ALS Chemex final assay reports that were provided to Seabridge (158 files). These .csv files were appended together and used to verify the assay records stored in Seabridge's electronic database. Forty-eight of the certificates represented reruns of previously assayed lots. The assay values from the reruns were noted to have successfully replaced values from the initial certificates. Table 12.3 summarizes key criteria associated with the QP's complete check of the 2014-2015 database.

Table 12.3 2014-2015 Database Verification

Year	No. of Holes	No. of Meters	Initial Certificates		Re-run Certificates		Total Assays Checked	
			No. of Certs.	No. of Assays	No. of Certs.	No. of Assays	No. of Certs.	No. of Assays
2014	26	25,563	85	11,142	25	3,309	110	14,451
2015	10	13,217	40	4,494	8	969	48	5,463
Total	**36**	**38,780**	**125**	**15,636**	**33**	**4,278**	**158**	**19,914**

The QP verified that nearly 20,000 gold, copper, silver, and molybdenum assays from the 2013-2014 Seabridge database matched certified assay records from their primary lab (ALS Chemex).

12.3.2 DRILL HOLE LOGS

During the QP's September 2015 site visit, select intervals from three core holes totaling about 1,100 m were compared against computerized drill logs (K-15-49, K-15-50, and M-15-131). The QP found that the electronic drill hole logs fairly represent the lithology, alteration, mineralization, and structural features observed in the core.

The QP performed core recovery checks for 25 intervals for each of the three core holes that were examined. No discrepancies were noted by the QP.

12.3.3 DOWN-HOLE SURVEYS

The QP made a 100% check of the 2014 and available 2015 down-hole survey records from the handwritten drillers logs and the electronic database. Several minor discrepancies were discussed with Seabridge's geological staff and the correct values were found to have been entered into the database.

12.3.4 QUALITY ASSURANCE/QUALITY CONTROL

The QP was provided with QA/QC data for the 2014-2015 drilling programs. This data was analyzed and the results and conclusions are presented in Section 11.0.

12.4 QP'S OPINION

Based on the QP's review of the various vintages of data that were used to estimate Mineral Resources for the KSM deposits, the drill hole assay, survey, and geologic data were found to be professionally collected and are thought to be adequate for estimating Mineral Resources.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 METALLURGICAL TEST WORK REVIEW

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

Several metallurgical test programs have been carried out to assess the metallurgical response of the mineral materials from the various deposits. The latest test programs were performed from 2007 through 2016, including an ongoing test program. The metallurgical testing programs, including historical testing programs, are listed in Table 13.1. The following sections summarize the testwork.

Table 13.1 Metallurgical Test Work Programs

Year	Program ID	Laboratory	Mineralogy	Flotation/ Cyanide Leach	Grindability	Others
2016	KM5063 (ongoing testwork)	ALS Metallurgy	√	√	√	
2016	SSW30216SD	Surface Science Western	√			
2015	KM4514	ALS Metallurgy	√	√	√	√
2015	KM4672	ALS Metallurgy	√	√	√	√
2014	KM4029	ALS Metallurgy	√	√	√	√
2013	KM3735	ALS Metallurgy	√	√		
2012	KM3174	G&T		√	√	
2012	KM3080	G&T		√		√
2011	KM3081	G&T				
2011	KM 2897	G&T		√		
2011	SSW47110	Surface Science Western	√			
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 Mineral Services		√	√	√
2009/2010	-	Köeppern – UBC			√	
2009	KM 2344	G&T	√	√	√	√

table continues...

Year	Program ID	Laboratory	Mineralogy	Flotation/ Cyanide Leach	Grindability	Others
2009	-	Pocock				√
2008	KM 2153	G&T	√	√	√	√
2008	-	Hazen			√	
2007	KM 1909	G&T	√	√	√	√
1991	-	Placer Dome Research Centre		√	√	√
1990	-	Placer Dome Research Centre	√	√	√	√
1989	-	Brenda Mines Ltd. Metallurgical Laboratory		√	√	
1989	-	Coastech Research Inc.		√		

Notes: The KM3174, KM3080, and KM3081 test work reports are available in Appendix D2 of the 2012 PFS (Tetra Tech 2012).
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

Bulk Sample	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, respectively.

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 sodium cyanide. Small amounts of sodium cyanide 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 2016

Since 2007, approximately twenty testing programs, including one ongoing testwork program, 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, although there are some variations among the samples from the different deposits. 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

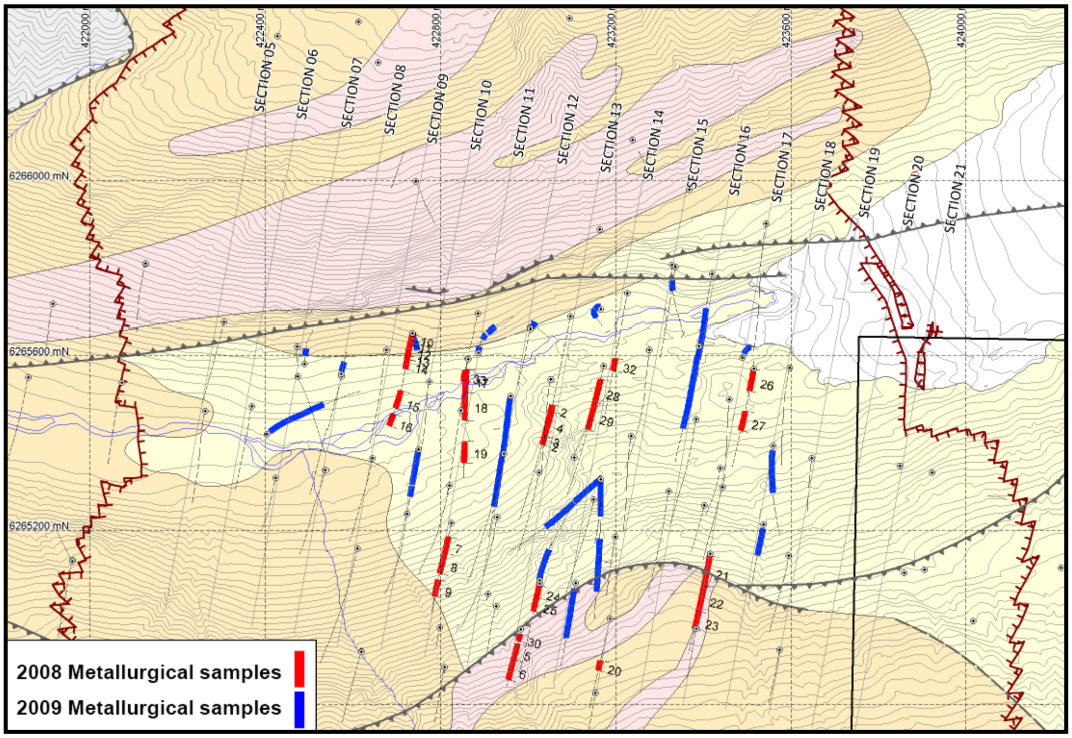


Table 13.9 Head Assay on Variability Test Samples – Mitchell, 2008 (G&T)

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

Note: *g/t for Au and Ag **from Sulphurets deposit
arsenic (As)

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)

Sample ID	Metal Content				
	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 was 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.

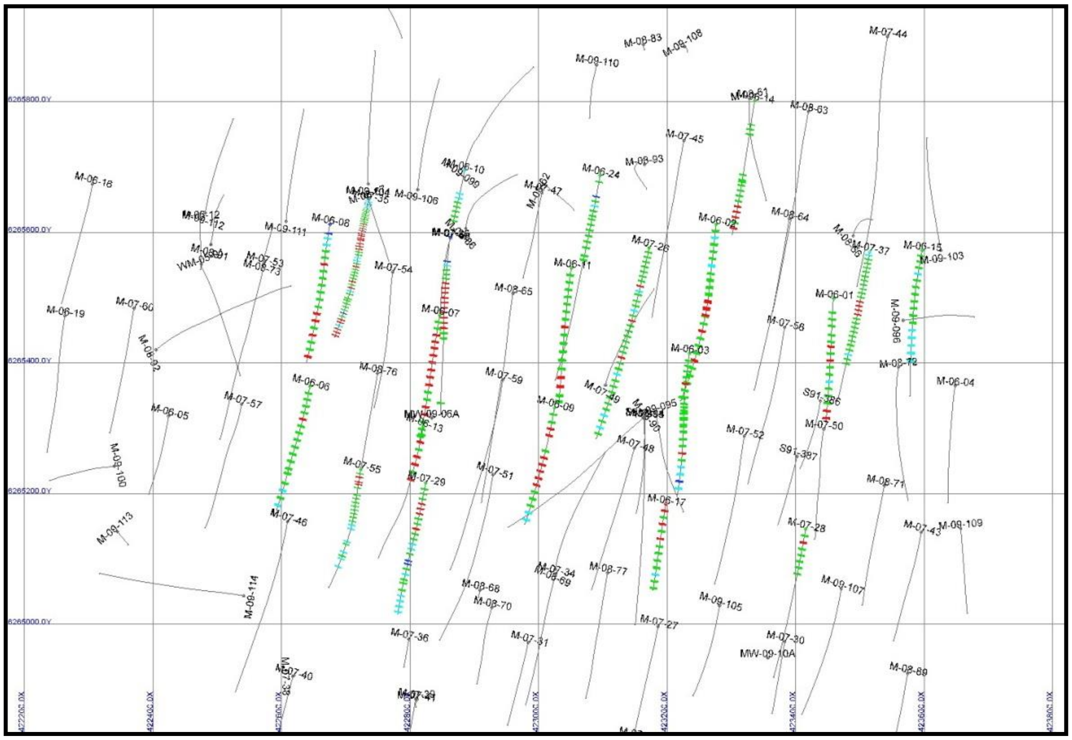
In 2010, three additional Mitchell Zone composites were generated using the drill core interval samples from the 2009/2010 drilling program. The sample details are shown below and in Table 13.12:

- 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



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.

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**

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

Note: iron (Fe); sulphur (S)

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 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 primary 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	-
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 higher CWi values.

Figure 13.4 Comparative Ball Mill Work Index – Variability Samples, 2008

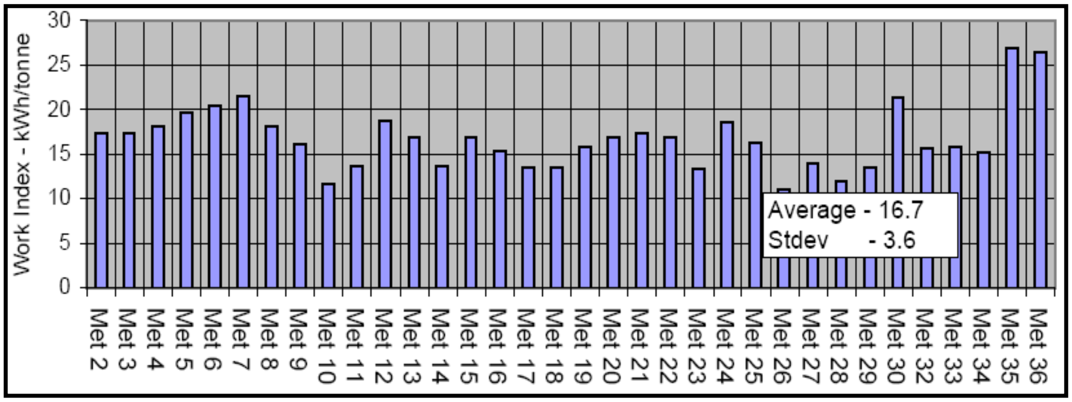
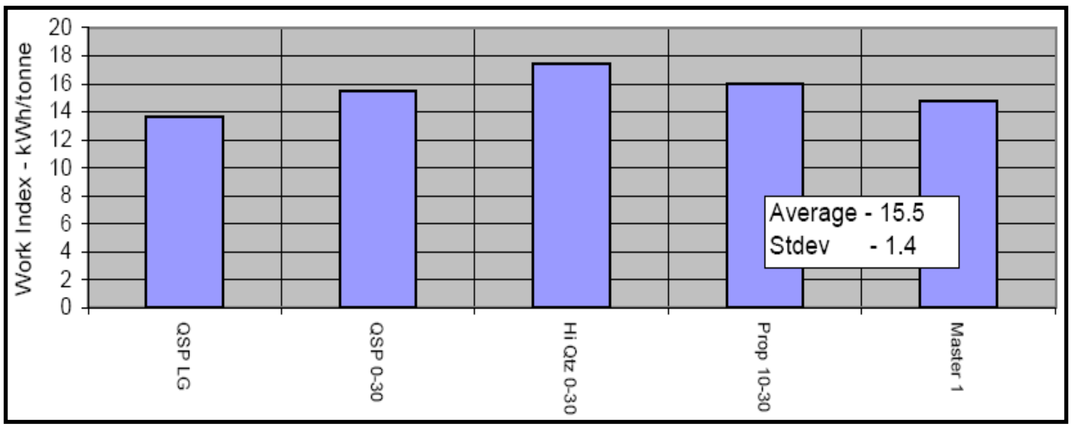


Figure 13.5 Comparative Ball Mill Work Index – Composite Samples (Mitchell, 2008)



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 were 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.

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.

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 (inches)		26	26	26	26	26	26

Note: *with phantom cyclones.
effective grinding length (EGL)

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 to those produced in 2008 (Table 13.1).

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/mm² 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

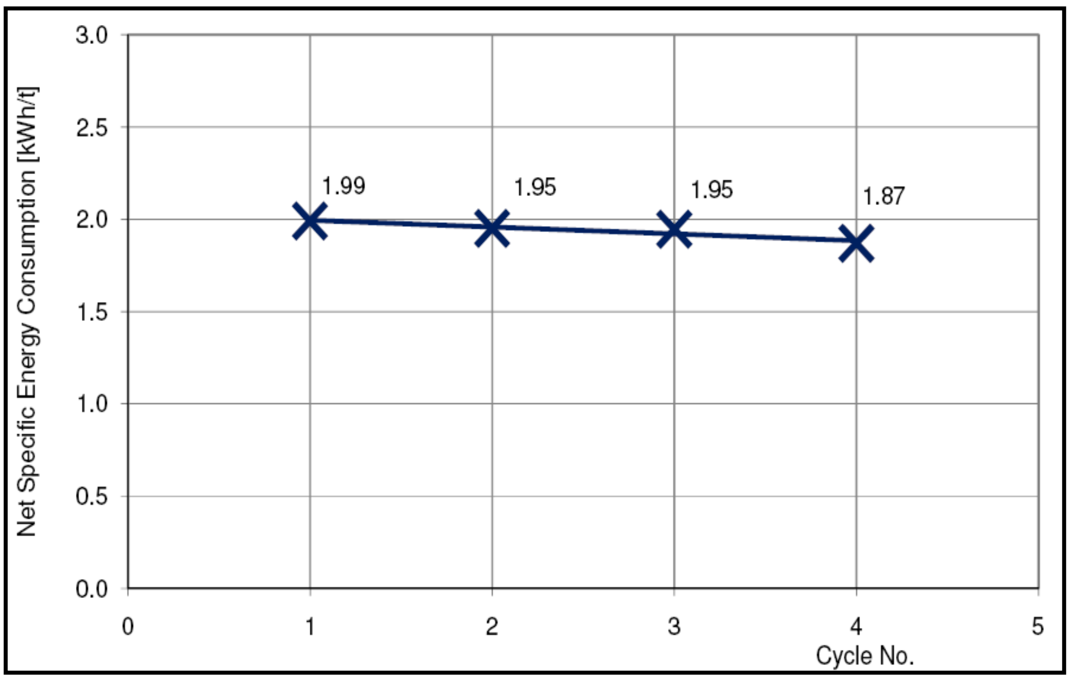
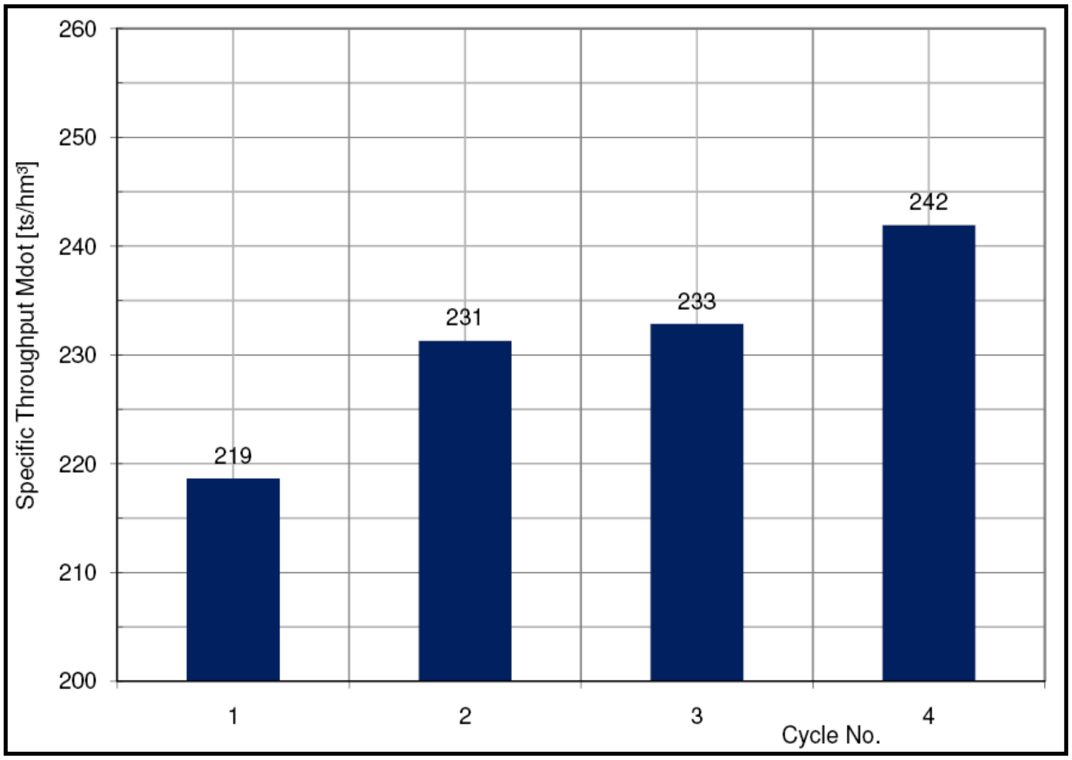


Figure 13.7 Specific Throughput (ts/hm³) vs. Cycle Number, 2010



The HPGR test work program showed that the Mitchell material is 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 13.21. 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 ft x 5.4 ft	23.5 ft x 40 ft	7.9 ft x 5.4 ft	23.5 ft x 40 ft
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	

Note: rotations per minute (RPM)

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

Many test programs have been conducted to develop the process flowsheet and to optimize the process conditions. 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)

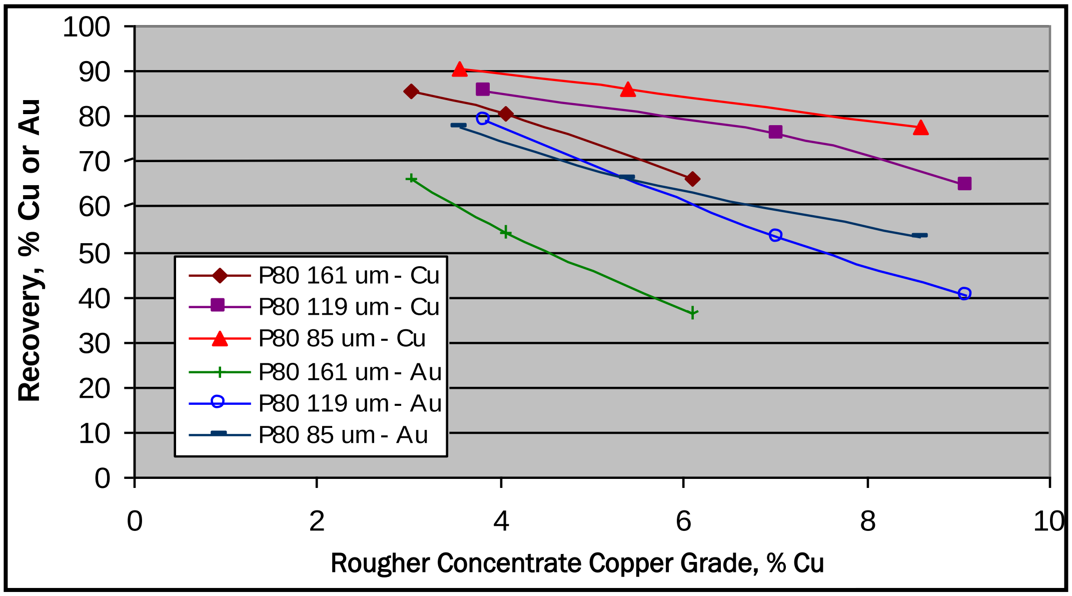
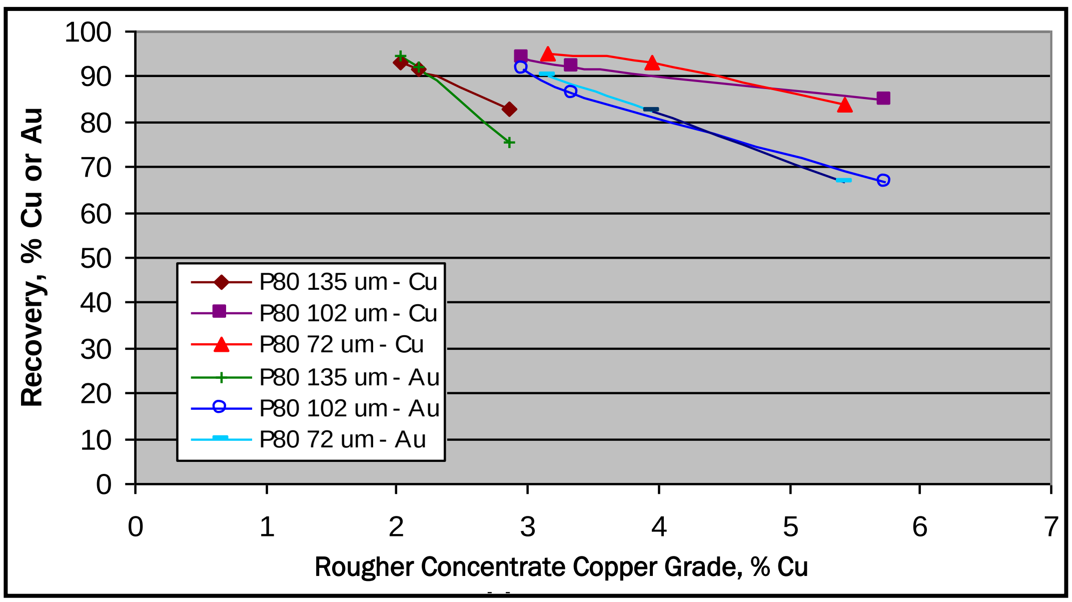


Figure 13.9 Metallurgical Performance vs. Primary Grind Size – Hi Qtz 0-30, 2008 (G&T)

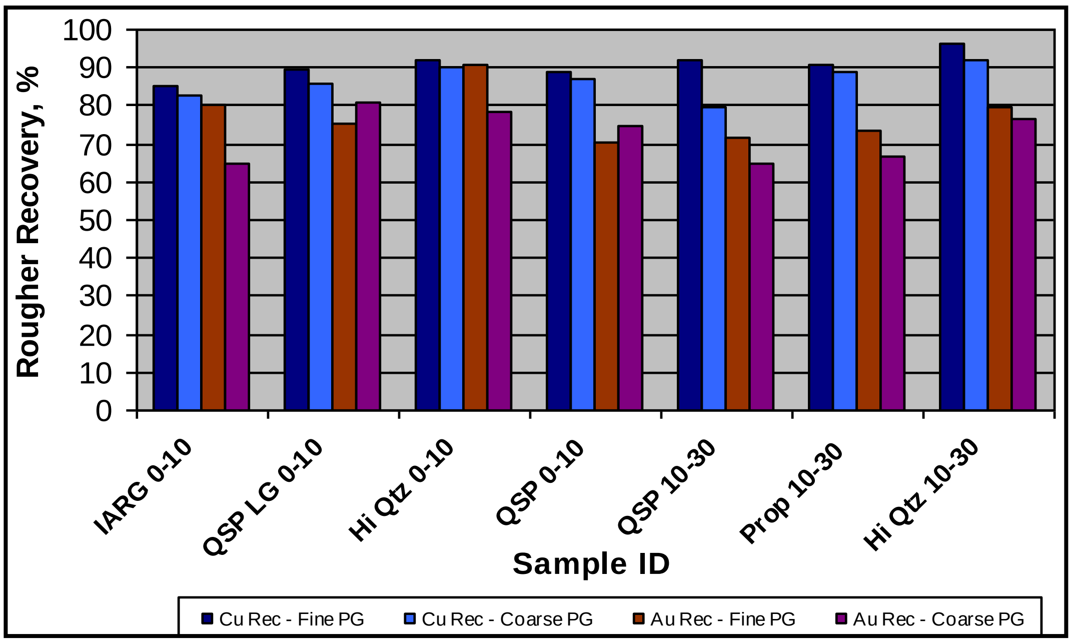


For QSP 0-30 composite, the copper recovery to a rougher concentrate, grading 4% copper, improved from 81 to 89% when the primary grind size was decreased from 80% passing 161 µm to 80% passing 85 µm. Gold recovery increased significantly with the increase in the grind fineness; however, there was no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

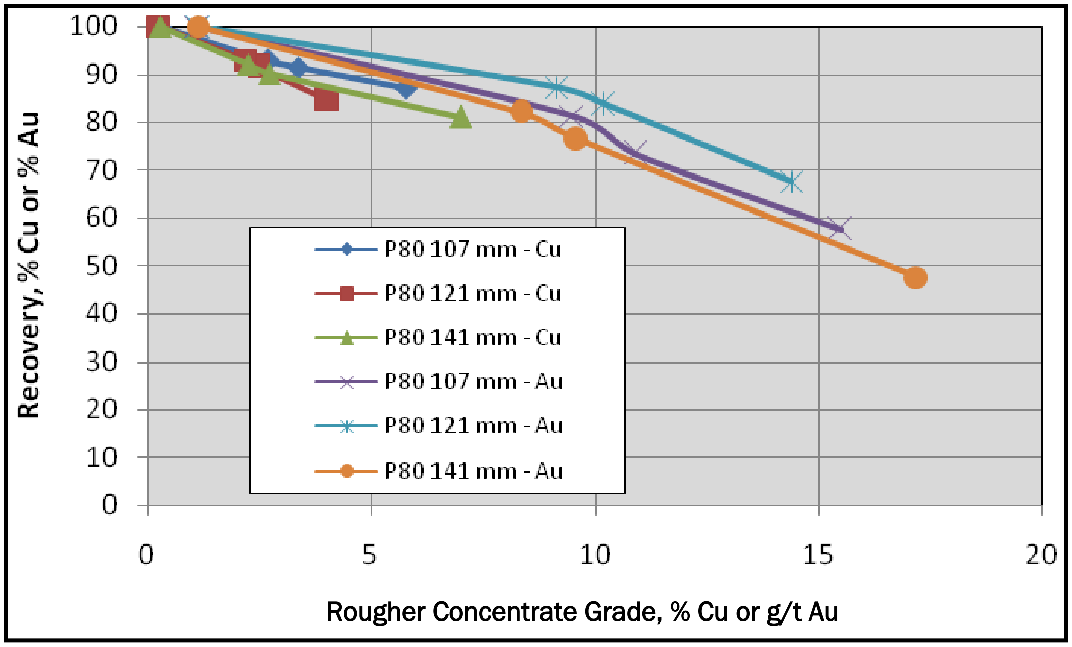


On average, the copper recovery reporting to copper rougher concentrate was 90.6% at the fine grind size, compared to 86.6% at the coarse grind size. The average gold recovery to the concentrate increased from 72.3 to 77.3%. However, QSP 0-10 and QSP LG 0-10 composites appeared to show different gold 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. Test results are provided in Figure 13.11. 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.

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% passing 18 µm prior to cleaner flotation.

It appeared that the copper recoveries reporting to the third cleaner concentrates in the open circuit tests increased with copper feed grade. As shown in Figure 13.12, G&T established the relationship between copper recovery and copper feed grade at a fixed concentrate grade of 25% copper. 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)

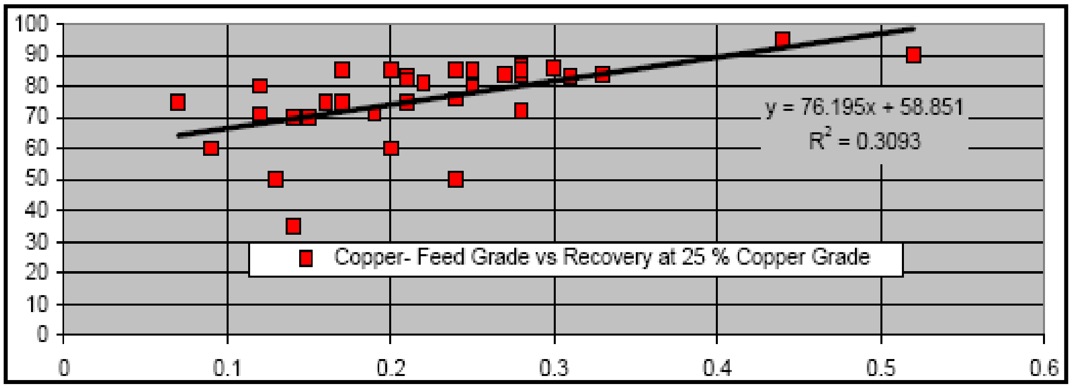
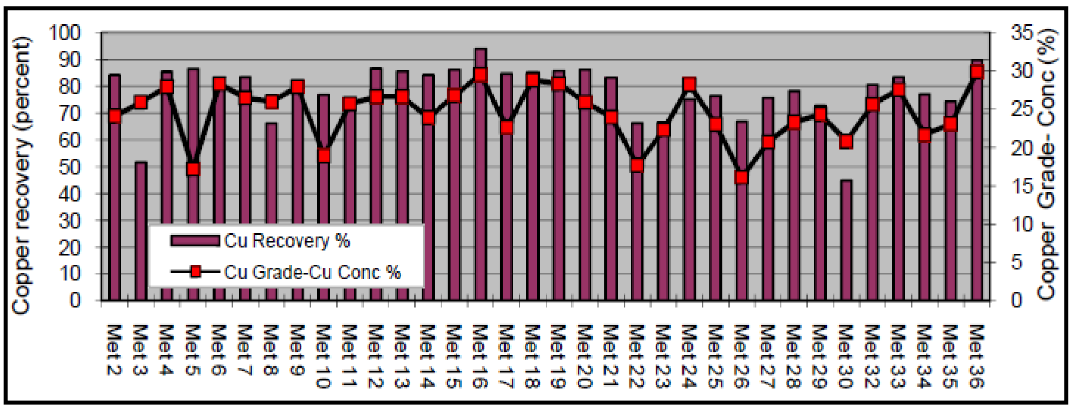
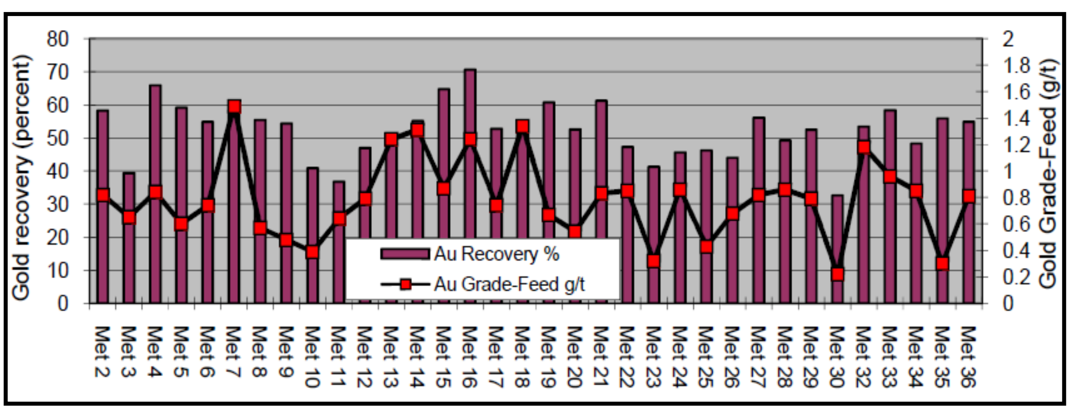


Figure 13.13 Copper Recovery & Concentrate Grade – Individual Samples, Mitchell, 2008 (G&T)



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.

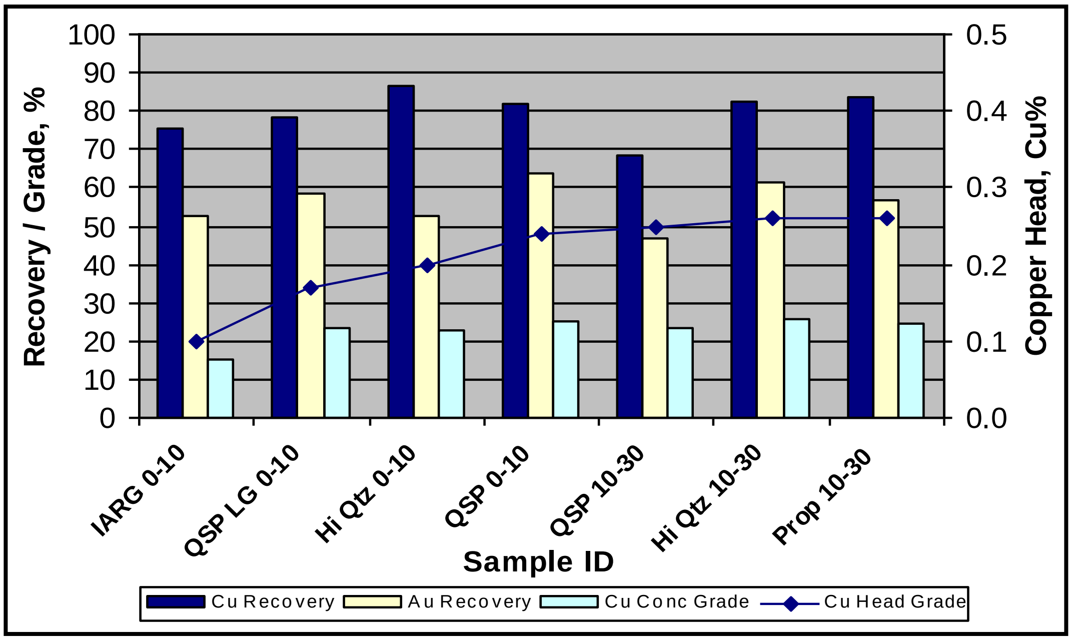
Figure 13.14 Gold Recovery & Feed Grade – Individual Samples, Mitchell, 2008 (G&T)



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.

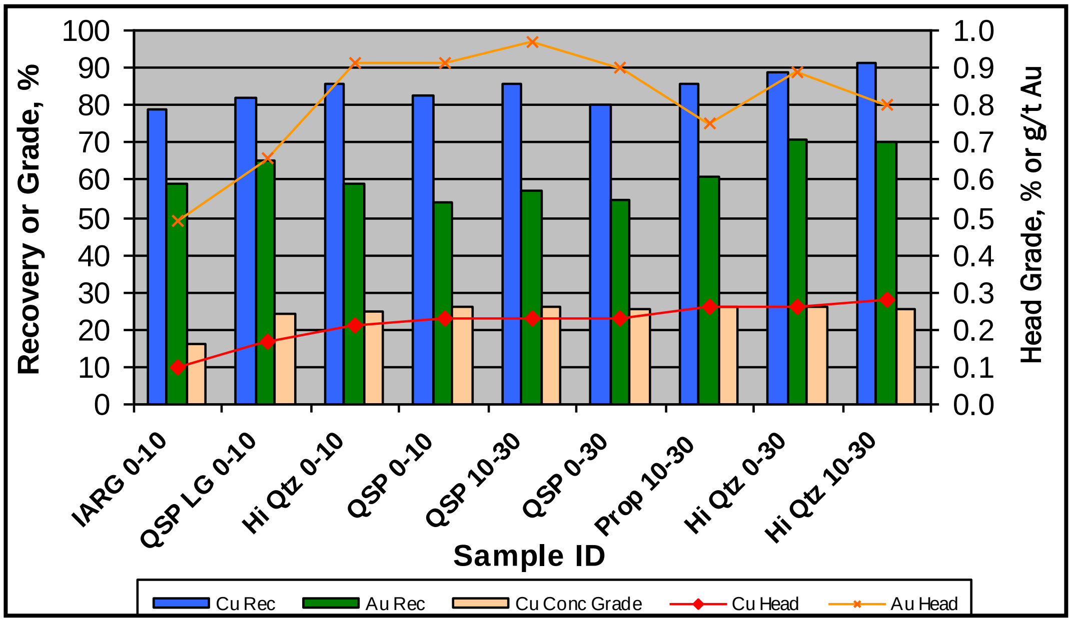
Figure 13.15 Metallurgical Performance – Composites, Mitchell, 2008 (G&T)



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)



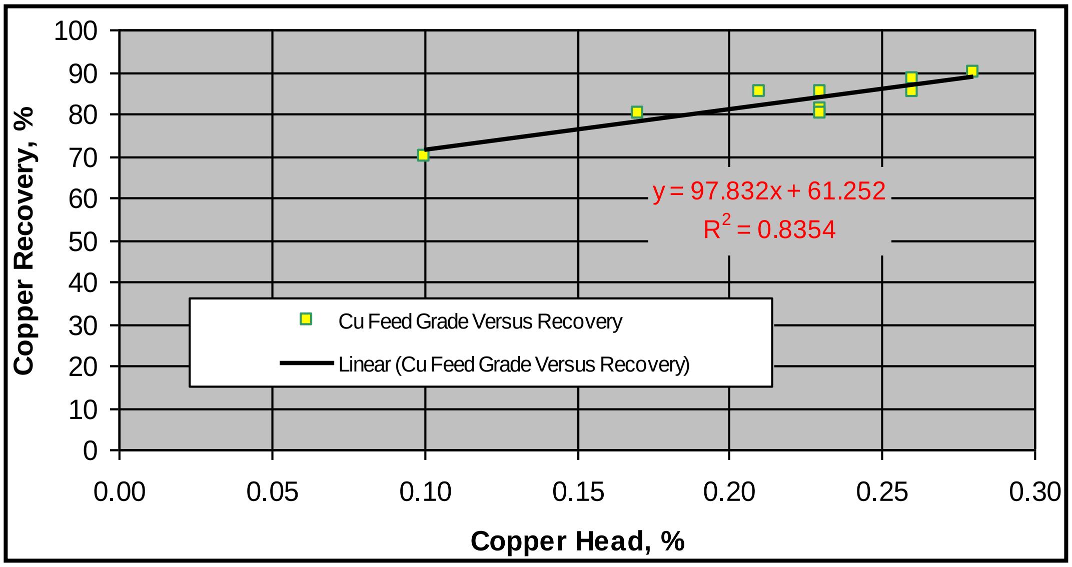
The second cleaner concentrate recovered between 79 to 91% of the copper and 54 to 71% of the gold from all the nine composites. On average, the metal recovery was 84.6% for copper and 61.2% for gold.

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% copper, excluding 16.2% copper from the IARG 0-10 composite and 24.0% copper from the QSP LG 0-10 composite.

After adjusting the copper recovery to reflect a concentrate grade of 25% copper, 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. The test work produced a similar relationship as shown in Figure 13.12, which are plotted using 2008 test results.

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 could be 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)

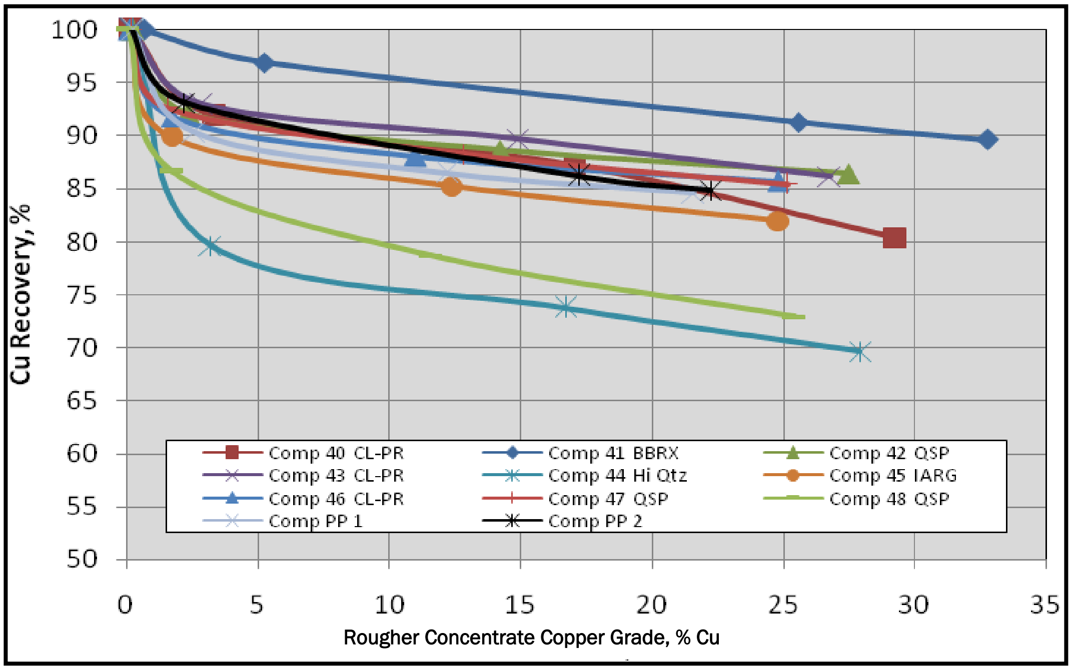
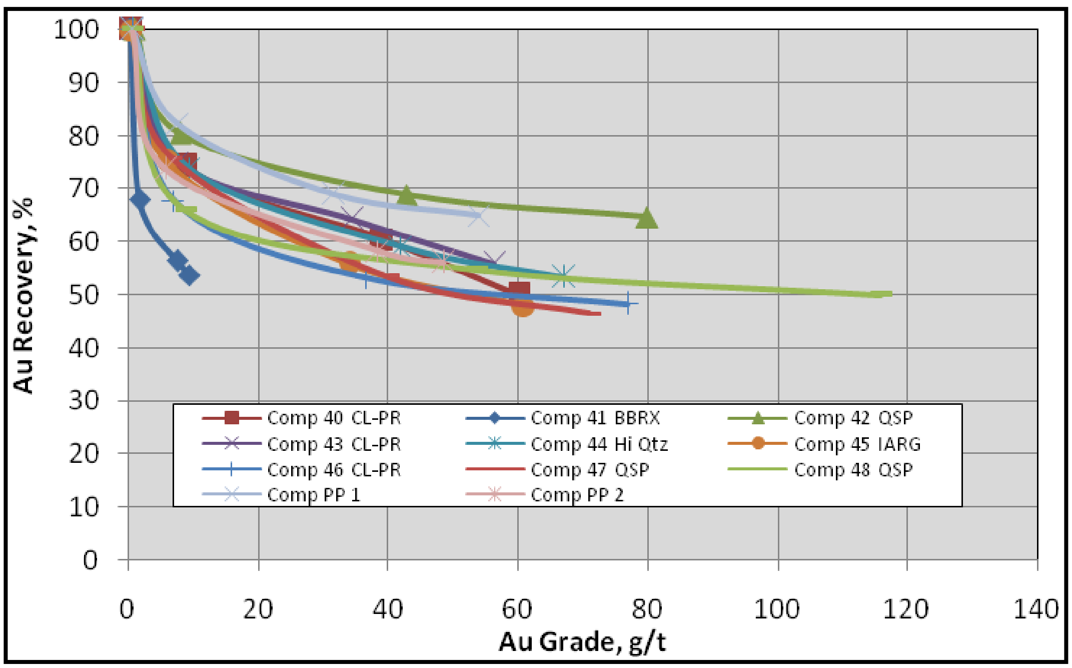


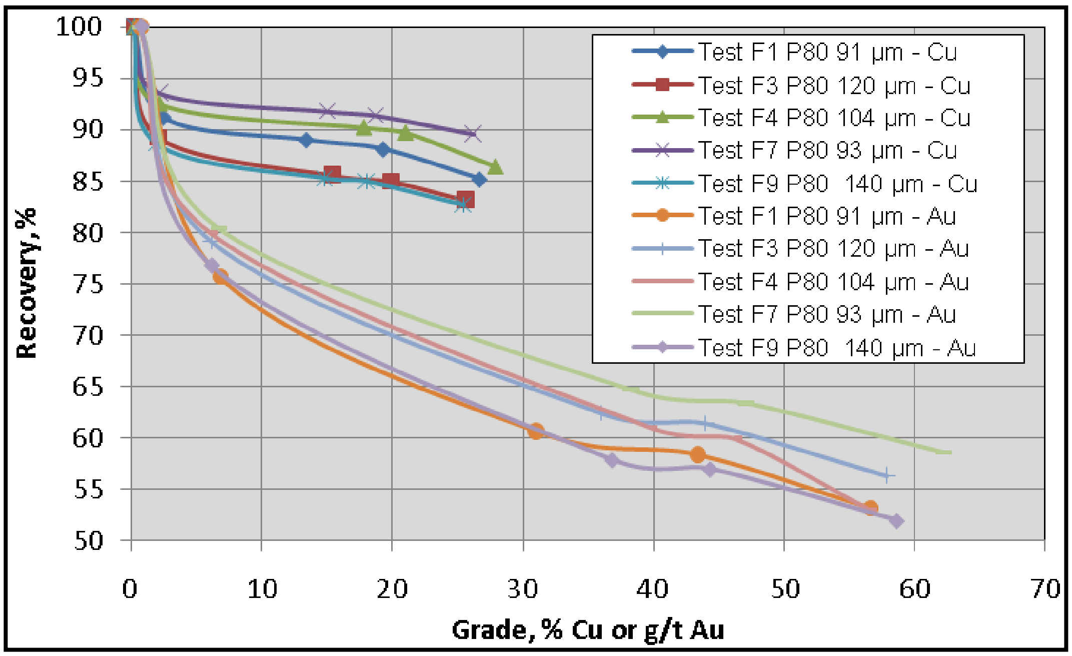
Figure 13.19 Gold Metallurgical Performance – Mitchell, 2009 (G&T)



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

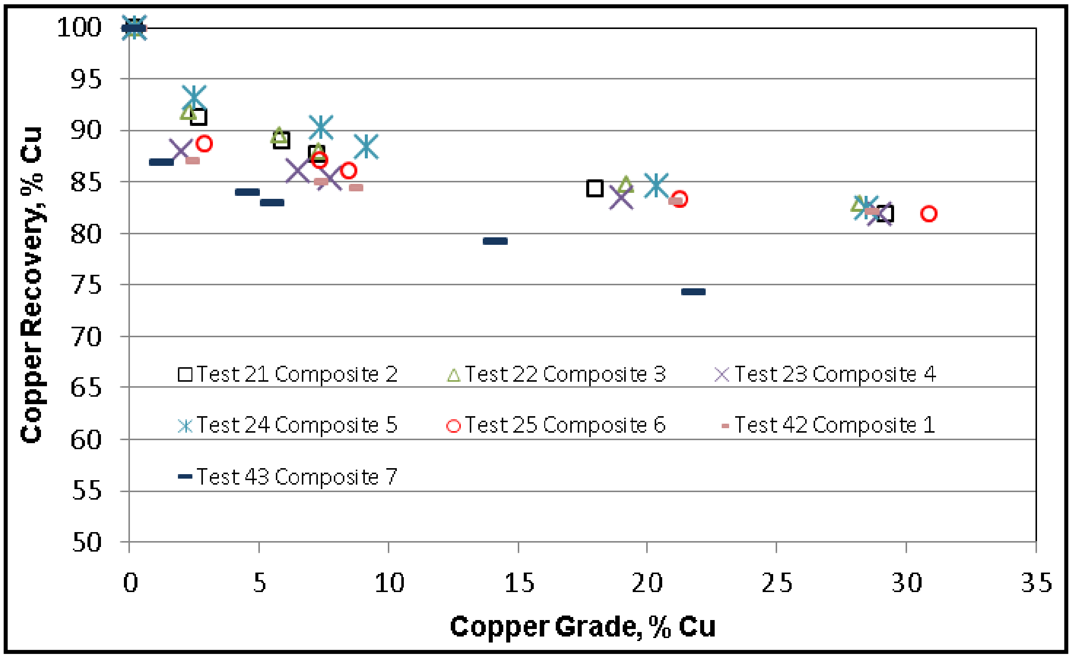


Figure 13.22 Gold Metallurgical Performance – Mitchell, 2011/2012 (G&T, KM 3080)

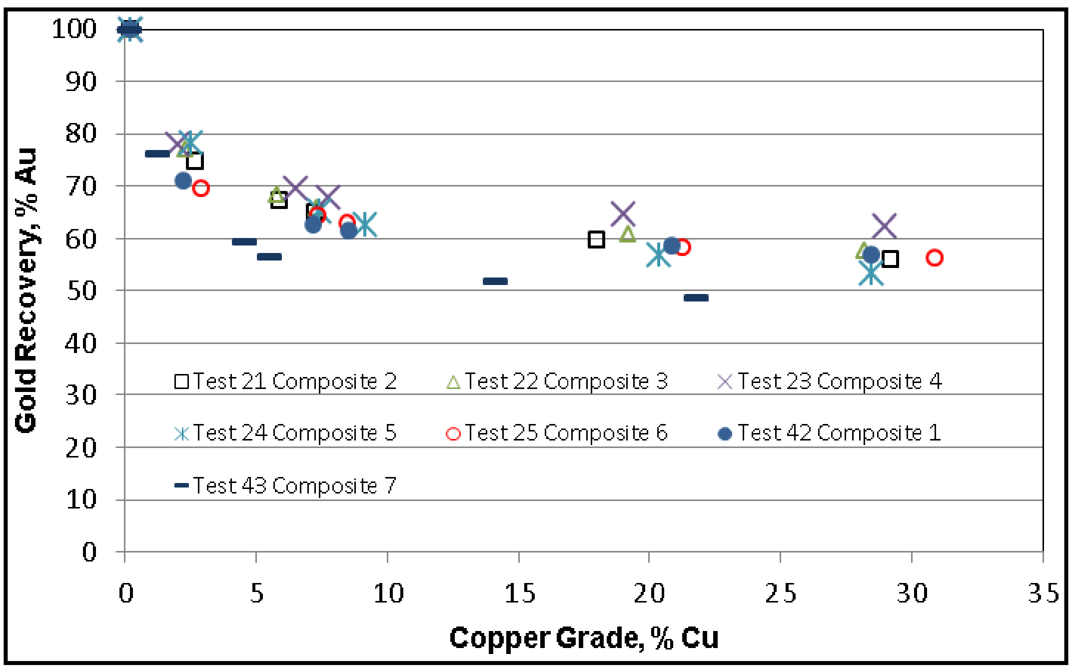


Figure 13.23 Copper Metallurgical Performance – Mitchell, 2011/2012 (G&T, KM 3174)

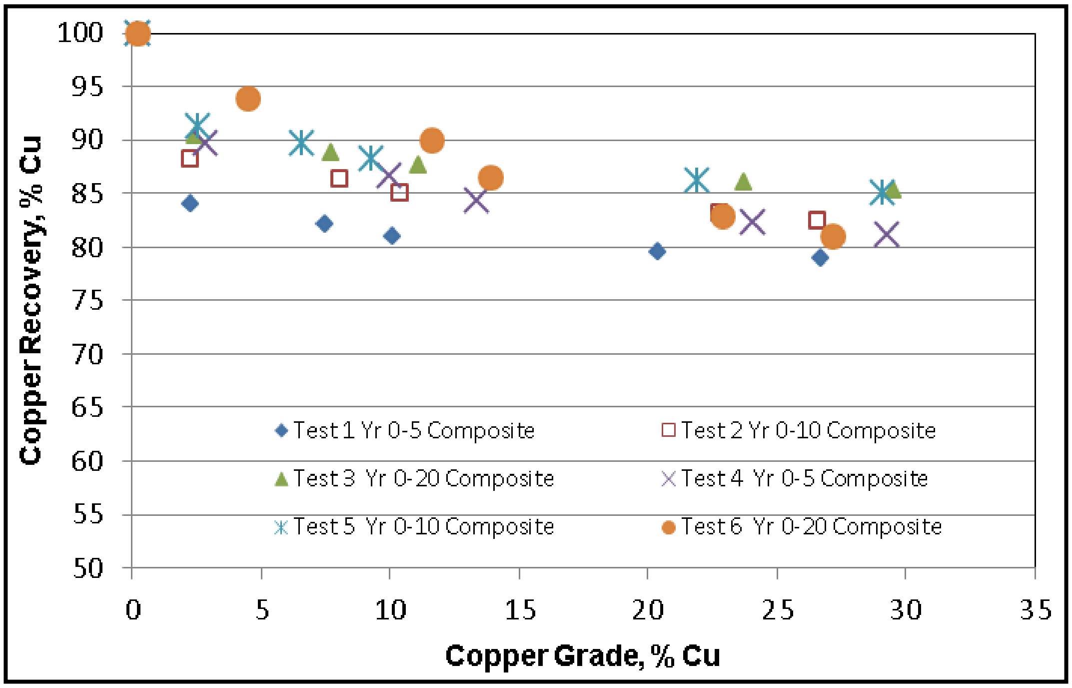
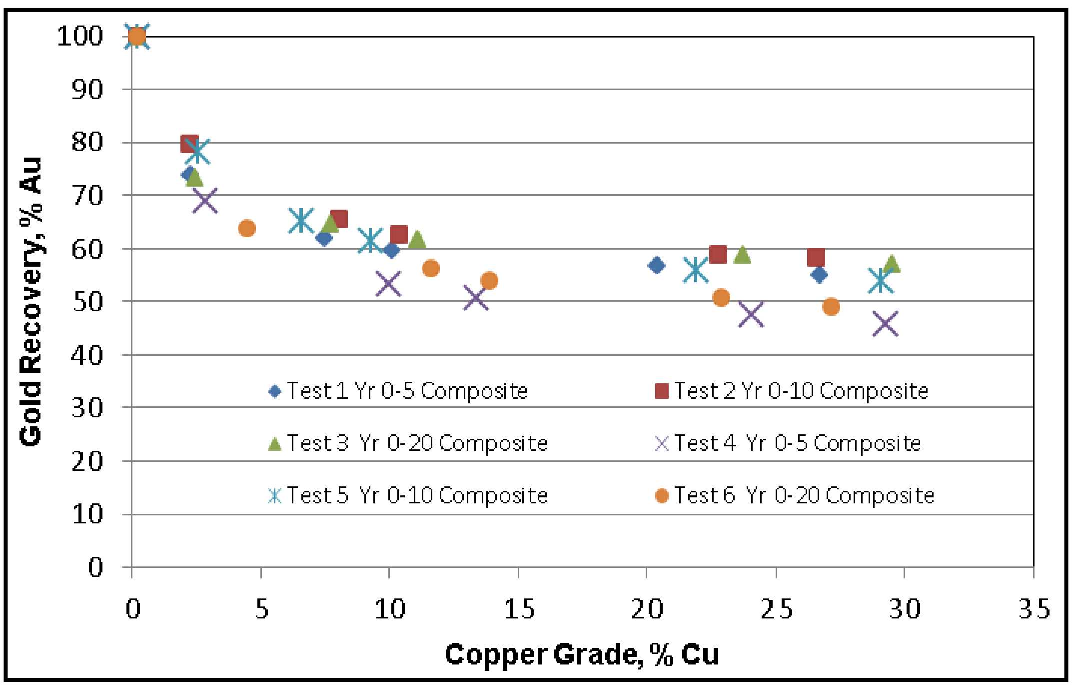


Figure 13.24 Gold Metallurgical Performance – Mitchell, 2011/2012 (G&T, KM 3174)



LOCKED CYCLE TESTS

Fifteen LCTs have been conducted on the various composite samples generated from the various testing programs since 2007, including the locked cycle test results achieved in 2015 on the Mitchell Year 0-5 composite which was constructed and tested in the previous program KM3080. 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 to 30% copper. On average, the final copper concentrate contained approximately 25.0% copper. The average recoveries to the concentrate were 84.7% for copper, 61% for gold, 50% for silver, and 56% for molybdenum. Approximately 26% of the gold and 28% 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 (P_{80} 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 (P_{80} 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
ALS 4514/30**	Mitchell Yr 0-5	Head	133/15/15	100.0	0.21	0.90	5	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.63	26.7	98.2	431	0.718	81.6	68.6	53.5	70.0
		Bulk Cleaner Tailing		8.91	0.20	1.68	10	0.010	8.7	16.6	17.6	13.9
		Au-Pyrite Concentrate		3.36	0.06	0.58	5	0.005	1.0	2.2	3.0	2.6
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

Note: *primary grind size/regrind size
**including a copper flotation on the pyrite flotation concentrate

Table 13.23 Locked Cycle Test Results – Blended Samples

Test Program	Comp	Product	Grind Size (P80 µm*)	Mass (%)	Grade Cu (%)	Grade Au (g/t)	Grade Ag (g/t)	Grade Mo (%)	Flotation Recovery (%) Cu	Flotation Recovery (%) Au	Flotation Recovery (%) Ag	Flotation Recovery (%) Mo
G&T 2535/19	Mitchell (PP Comp1)/ Kerr (52/53 Blend); 80%:20%	Head	127/20	100.0	0.31	0.70	3.5	-	100.0	100.0	100.0	-
		Cu/Mo Concentrate		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	141/22	100.0	0.22	0.67	2.8	0.007	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		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	135/15	100.0	0.24	0.79	-	0.004	100.0	100.0	-	100.0
		Cu/Mo Concentrate		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
ALS 4672/32**	Mitchell (Mitchell Yr 0-5)/Iron Cap (IC-2014-MC4)	Head	117/17/17	100.0	0.24	0.67	4	0.005	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.81	25.0	54.3	304	0.430	82.7	65.7	58.4	70.3
		Bulk Cleaner Tailing		6.73	0.20	1.89	13	0.006	5.7	19.1	20.8	8.8
		Au-Pyrite Concentrate		3.12	0.14	1.10	6	0.004	1.8	5.1	4.8	2.8
ALS 4514/31**	Mitchell (Mitchell Yr 0-5)/ Kerr (DK-2014-MC3)	Head	129/17/17	100.0	0.37	0.59	3	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.31	24.5	28.3	150	0.328	87.9	62.8	57.2	75.1
		Bulk Cleaner Tailing		10.6	0.13	1.07	5	0.004	3.6	19.1	15.4	7.4
		Au-Pyrite Concentrate		6.16	0.10	0.41	2	0.003	1.7	4.3	3.6	3.2

Note: *primary grind size/regrind size
**including a copper flotation on the pyrite flotation concentrate

PILOT PLANT TESTS

In the 2009 testing program, G&T carried out 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% copper 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 pilot plant results are summarized in Table 13.24.

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

Test	Grind Size (P_{80} 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**

Note: *primary grind size

In the 2010 testing program, G&T conducted two additional 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)

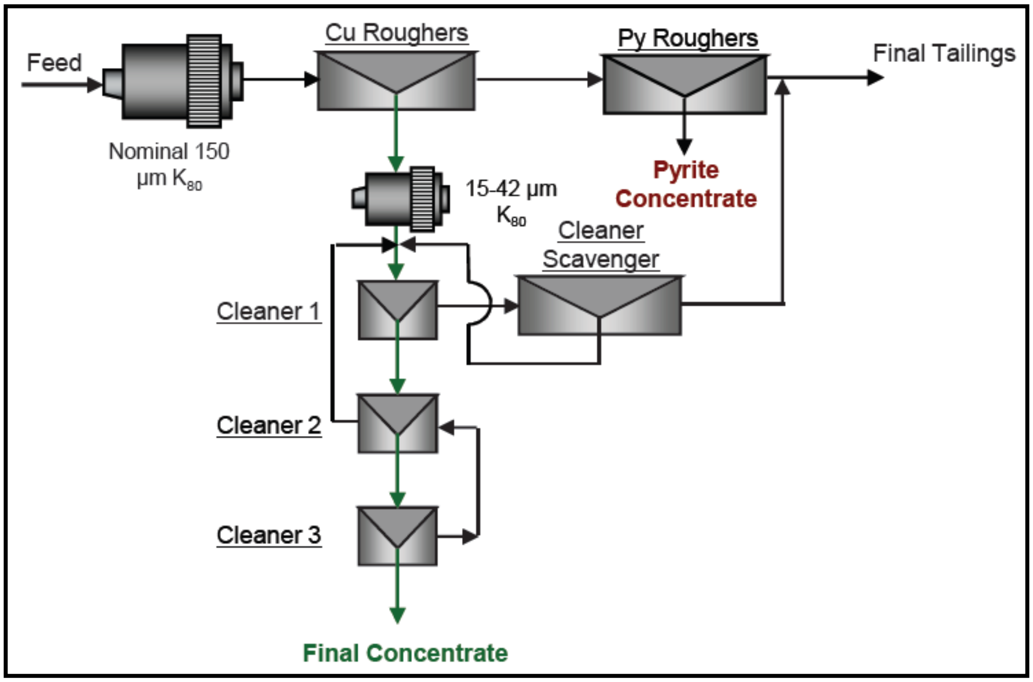


Table 13.25 Pilot Plant Test Results - Mitchell, 2010 (G&T)

Test	Grind Size (P_{80} 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**

Note: *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% copper. 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% copper. 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
Titanium 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% molybdenum. 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% molybdenum concentrate containing 1.8% copper.

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% molybdenum 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 sodium sulphide and PE 26 together with sodium cyanide or D910 as depressant are shown in Figure 13.27. It appears that molybdenum concentrate grade improved with adding sodium cyanide.

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

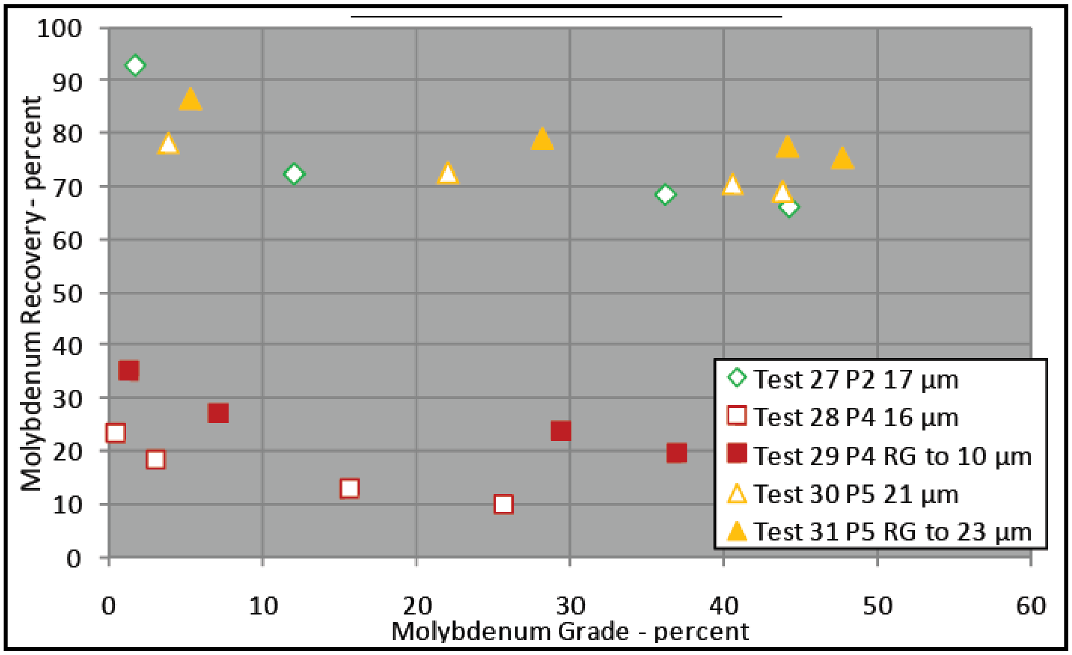
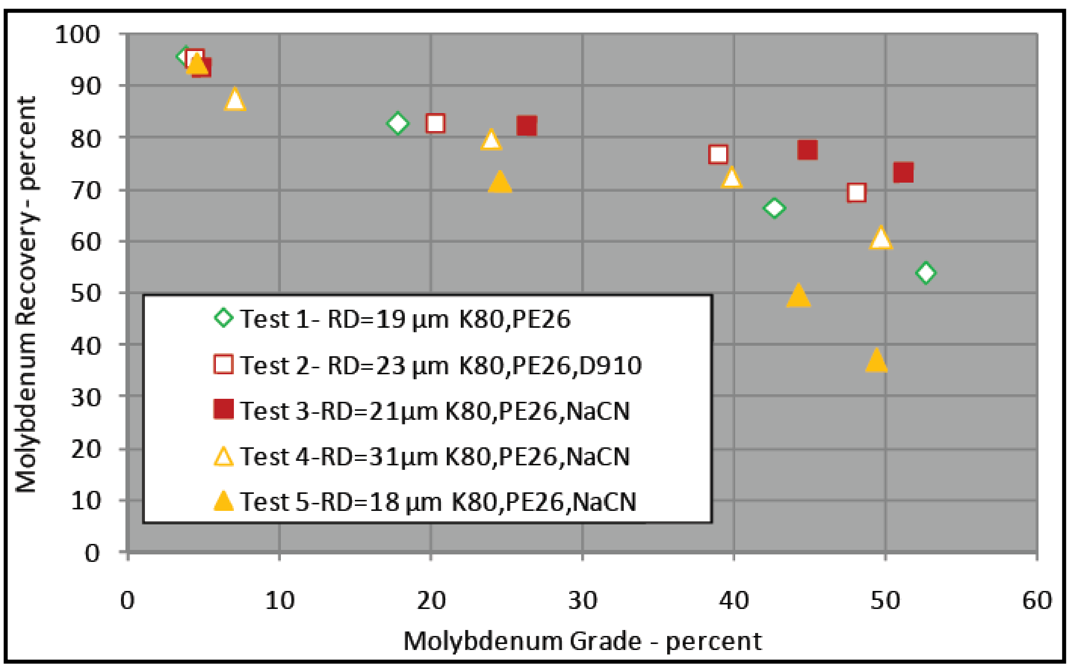


Figure 13.27 Cu-Mo Separation Open Circuit Flotation Tests, 2011



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 13.27.

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 Brenda-Leach test results indicated that the copper and lead contents of the molybdenum concentrate were reduced respectively from 2.06% to 0.26% copper and from 0.14% to 0.03% lead. 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**

Note: *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 gold and 9.6 g/t silver. 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. 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
4672	Mitchell/Iron Cap - Cl.Sc.Tls[4]	17	1.9	51.0	11.0	63.0
4672	Mitchell/Iron Cap – Py Conc[4]	17	1.2	50.0	5.0	71.3
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)
[4]50% Mitchell Year 0-5 + 50% IC-2014-MC4.

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 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 sodium cyanide 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 gold 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 gold 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

Note: direct cyanide leaching (DCN)

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

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

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

Most of the pan concentrates contained less than 50 g/t gold with a gold recovery of less than 17%, except for the MET 4 sample. Panning produced a 231 g/t gold concentrate and recovered 55% of the gold from the MET 4 sample. The results suggest that gravity concentration may not be applicable for most of the mineralization.

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. However, a poor match between the calculated gold and measured gold in the feeds was reported.

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

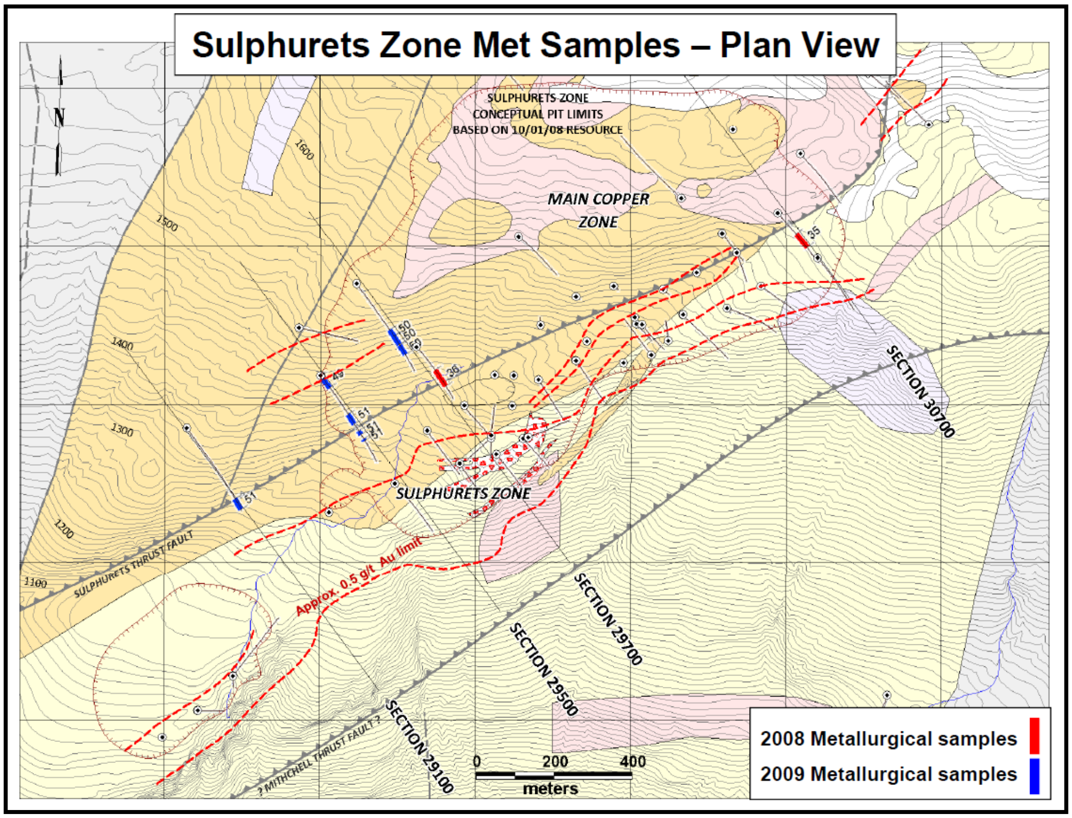


Table 13.34 Metal Contents of Composites – Sulphurets

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

Notes: 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)	62.0	69.0
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.

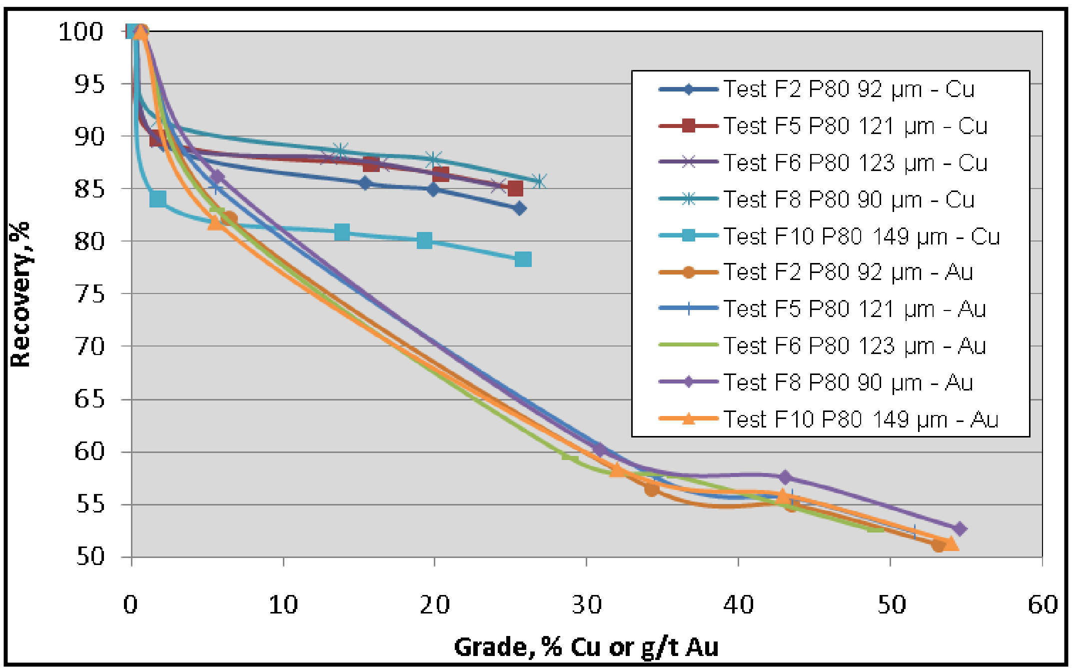
Table 13.37 Batch Flotation Tests –Sulphurets, 2009 (G&T)

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% copper grade versus 76% recovery at 28% copper grade. The test on Composite 9 produced a much lower copper recovery compared to the other tests. This could be the result of the 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.

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 13.39.

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**

Note: *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	-	-

KERR MINERALIZATION

There are two mineralization zones in the Kerr deposit. The lower Kerr Zone mineralization may be mined by underground block caving, while the upper Kerr Zone material is anticipated to be mined by open pit methods. Early testwork focused on the samples from the surface Kerr Zone. Since 2013, three test programs—KM3735, KM4514, and KM4029—have been completed to investigate the metallurgical performances of the mineralization from the lower Kerr, excluding an ongoing testwork program.

Kerr Mineralization- Upper Kerr Zone

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

Table 13.41 Metal Contents of Composites – Upper 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	1.0
Composite 11 – 2011/2012	QSP Quartz Crackle	0.68	0.29	0.001	2.0

Notes: 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 upper Kerr Zone 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 summarized in 13.1.1.

Table 13.42 Bond Ball Mill Work Index Test Results – Upper 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 upper Kerr samples to SAG mills. The test results revealed that the grindabilty of the upper 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 – Upper 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 upper 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 upper 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 upper Kerr samples was not as good as that achieved with the Mitchell and Sulphurets samples despite their lower copper head grades.

On average, the upper Kerr samples produced a 27.8% copper 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% upper Kerr) were very similar to those achieved with the Mitchell mineralization alone.

Table 13.44 Locked Cycle Test Results – Upper 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 – Upper 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 – Upper Kerr		**18**	**0.7**	**57.1**	**4.4**	**32.3**

On average, 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.

Kerr Mineralization- Lower Kerr Zone

As the various lower Kerr exploration annual drilling programs were completed, preliminary metallurgical testing was performed on available core samples using the basis 2012 PFS (Tetra Tech 2012) flow sheet parameters. Three test programs have been completed at ALS Metallurgy Kamloops (Test Programs: KM3735, KM4029-B and KM4514). The samples used were constructed from the various drill core intervals of the lower Kerr Zone from the different drill programs. The samples for these previous test programs were mainly generated from the areas adjoining to the proposed block caves within the deposits. A total of 24 samples have been tested, including 11 composite samples tested using LCT procedures. The head assay of the samples tested are shown in Table 13.46.

Table 13.46 Metal Contents of Composites – Lower Kerr, 2013-2015 (ALS)

Sample	Metal Content			
	Cu (%)	Au (g/t)	Mo (%)	Ag (g/t)
KM3735/DK-2012-03	0.92	0.32	0.004	5
KM4029/DK-2013-01*	0.53	0.34	0.008	2
KM4029/DK-2013-02*	0.41	0.27	0.006	1
KM4029/DK-2013-03*	0.47	0.89	0.008	4
KM4029/DK-2013-04*	0.79	0.53	0.005	2
KM4029/DK-2013-05*	1.75	1.04	0.004	3
KM4029/DK-2013-06*	1.41	0.63	0.005	4
KM4514/DK-2014-01	0.86	0.76	0.003	1
KM4514/DK-2014-02	0.55	0.54	0.003	1
KM4514/DK-2014-03	0.25	0.24	0.004	1
KM4514/DK-2014-04	0.44	0.37	0.005	<1
KM4514/DK-2014-05	0.42	0.31	0.004	<1
KM4514/DK-2014-06	0.68	0.45	0.002	1
KM4514/DK-2014-07	0.66	0.22	0.007	2
KM4514/DK-2014-08	0.46	0.21	0.009	2
KM4514/DK-2014-09	0.32	0.12	0.006	1
KM4514/DK-2014-10	0.51	0.20	0.006	<1
KM4514/DK-2014-11	0.62	0.22	0.007	1
KM4514/DK-2014-12	0.52	0.38	0.004	4
KM4514/DK-2014-MC1*	0.50	0.41	0.004	1
KM4514/DK-2014-MC2*	0.48	0.20	0.006	3
KM4514/DK-2014-MC3*	0.51	0.28	0.004	2
KM4514/DK-2014-MC4*	0.54	0.44	0.003	1
KM4514/DK-2014-MC5*	0.46	0.21	0.005	3

Note: *tested with LCT procedure

Increased drilling campaigns over the last couple of years will allow Seabridge to plan and complete a metallurgical testing program (KM5063) on samples representative of the planned lower Kerr underground mining blocks. This metallurgical work is currently underway and is expected to be advanced in 2017. A total of 22 samples, including five composites, are currently being tested at ALS Metallurgy Kamloops.

The preliminary testwork from these completed test programs showed that these widely spaced lower Kerr exploration samples responded well to the flotation flowsheet developed previously for the Project. The preliminary testwork from the ongoing testwork being conducted shows some variations in metallurgical response were noted. It appears that the samples from the proposed upper cave areas may have a similar metallurgical performance as the samples collected from the proposed Kerr open pit area, while the lower caves samples may behave more similarly to the samples of the lower Kerr Zone tested by the previous three test programs.

Flotation Tests

Three of the testing programs used open batch floatation test procedure to investigate the metallurgical responses of the lower Kerr samples to the flowsheet developed for the Mitchell and the other deposits, including some preliminary variability tests. The flow sheet and test conditions used for these tests are similar to those used for the other deposits or zones of the Project, including conventional flotation to produce a copper-gold-molybdenum bulk concentrate, with cyanide leach testing to further recover gold and silver from the cleaner flotation tailings and the gold-bearing pyrite concentrate. A copper scavenging flotation was added to float copper minerals from the gold bearing pyrite concentrate prior to cyanidation. The KM4029-B and KM4514 test programs also conducted locked-cycle flotation tests to further evaluate the metallurgical performance of the composite samples constructed. The LCT results are shown in Table 13.47.

The LCT results show that these lower Kerr mineralization responds well to the flow sheet proposed for the Project. Copper recovery ranged from 86 to 97% and gold recovery varied from 55 to 77%. The flotation concentrate grades were in a range of between 22 and 29% copper.

Table 13.47 Flotation Locked Cycle Test Results – Lower Kerr

Test	Sample	Product	Grind Size (P80 µm*)	Mass (%)	Grade Cu (%)	Grade Au (g/t)	Grade Ag (g/t)	Grade Mo (%)	Recovery (%) Cu	Recovery (%) Au	Recovery (%) Ag	Recovery (%) Mo
KM4029-22	DK2013-01	Head	118/15/15	100.0	0.59	0.36	1.9	0.007	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		2.2	25.3	11.5	36.5	0.243	92.7	69.2	41.0	78.1
		Bulk Cleaner Tailing		15.0	0.14	0.52	2.0	0.003	3.7	21.8	15.7	5.6
		Au-Pyrite Concentrate		7.4	0.08	0.23	1.0	0.005	1.0	4.8	3.9	5.0
KM4029-23	DK2013-02	Head	119/15/11	100.0	0.44	0.23	2.0	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.6	24.4	9.0	48.0	0.157	90.6	62.6	40.0	67.4
		Bulk Cleaner Tailing		16.5	0.09	0.29	2.0	0.002	3.4	20.3	16.9	8.7
		Au-Pyrite Concentrate		4.5	0.11	0.37	1.5	0.003	1.1	7.1	3.5	3.6
KM4029-18	DK2013-03	Head	121/14/10	100.0	0.50	0.94	4.0	0.007	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.9	23.0	32.9	136	0.215	89.7	68.2	65.3	59.4
		Bulk Cleaner Tailing		20.3	0.12	0.73	4.0	0.004	4.7	15.8	20.2	13.0
		Au-Pyrite Concentrate		9.8	0.04	0.45	2.5	0.013	0.9	4.7	6.0	18.0
KM4029-17	DK2013-04	Head	116/16/-	100.0	0.86	0.50	2.0	0.003	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		3.0	26.3	9.5	46.0	0.068	91.2	55.8	69.9	58.6
		Bulk Cleaner Tailing		20.4	0.10	0.81	1.0	0.003	2.4	33.0	10.5	17.4
		Au-Pyrite Concentrate		14.6	0.23	0.22	0.5	0.001	4.0	6.3	3.7	6.0
KM4029-16	DK2013-04	Head	116/15/11	100.0	0.82	0.50	2.4	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		2.9	26.3	10.6	48.0	0.071	93.6	61.5	58.3	54.8
		Bulk Cleaner Tailing		21.8	0.11	0.70	2.0	0.003	2.9	30.3	18.1	17.3
		Au-Pyrite Concentrate		12.8	0.04	0.18	2.0	0.003	0.6	4.5	10.6	11.4
KM4029-26*	DK2013-05	Head	127/23/10	100.0	1.83	0.93	3.3	0.002	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		6.18	28.7	11.2	43.5	0.010	96.6	74.7	80.5	26.4
		Bulk Cleaner Tailing		15.80	0.20	0.82	1.5	0.004	1.7	14.0	7.2	26.9
		Au-Pyrite Concentrate		4.00	0.09	0.34	1.0	0.008	0.2	1.5	1.2	13.9

table continues…

Test	Sample	Product	Grind Size (P80 µm*)	Mass (%)	Grade Cu (%)	Grade Au (g/t)	Grade Ag (g/t)	Grade Mo (%)	Recovery (%) Cu	Recovery (%) Au	Recovery (%) Ag	Recovery (%) Mo
KM4029-24*	DK2013-05	Head	127/30/11	100.0	1.81	0.93	3.1	0.002	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		7.7	22.8	9.4	33.8	0.006	96.6	77.1	82.6	29.6
		Bulk Cleaner Tailing		16.9	0.18	0.90	1.0	0.002	1.7	16.4	5.4	22.1
		Au-Pyrite Concentrate		6.1	0.07	0.15	0.5	0.007	0.2	1.0	1.0	24.4
KM4029-25	DK2013-06	Head	128/15/9	100.0	1.44	0.67	3.4	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		4.7	28.4	9.3	50.0	0.039	92.7	65.5	69.2	50.4
		Bulk Cleaner Tailing		16.4	0.18	0.78	3.0	0.003	2.1	19.1	14.5	14.7
		Au-Pyrite Concentrate		7.9	0.14	0.66	2.5	0.007	0.8	7.8	5.8	15.3
KM4514-23	DK-2014-MC3	Head	124/16/15	100.0	0.52	0.32	2	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		2.0	23.5	10.2	55	0.164	91.4	65.9	58.5	76.4
		Bulk Cleaner Tailing		18.2	0.11	0.43	2	0.002	4.0	24.8	19.0	9.3
		Au-Pyrite Concentrate		6.8	0.06	0.11	1	0.002	0.7	2.4	3.5	2.6
KM4514-24	DK-2014-MC1	Head	121/16/18	100.0	0.55	0.42	2	0.003	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		2.0	24.7	14.8	36	0.091	91.3	72.6	48.7	62.8
		Bulk Cleaner Tailing		16.5	0.16	0.50	2	0.003	4.8	19.6	21.8	15.1
		Au-Pyrite Concentrate		7.7	0.06	0.13	1	0.001	0.8	2.5	5.1	3.0
KM4514-25	DK-2014-MC2	Head	126/16/16	100.0	0.51	0.21	3	0.005	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.7	26.4	7.3	74	0.210	86.1	56.9	46.3	67.1
		Bulk Cleaner Tailing		15.8	0.22	0.44	5	0.005	7.0	32.6	26.9	16.8
		Au-Pyrite Concentrate		7.1	0.10	0.15	2	0.002	1.4	5.1	5.4	3.1
KM4514-26	DK-2014-MC4	Head	115/16/16	100.0	0.57	0.47	1	0.003	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.9	27.4	18.6	34	0.104	89.5	73.8	48.4	67.1
		Bulk Cleaner Tailing		16.8	0.22	0.57	1	0.003	6.6	20.5	18.8	16.4
		Au-Pyrite Concentrate		8.4	0.06	0.19	1	0.001	0.9	3.4	4.8	3.7

table continues…

Test	Sample	Product	Grind Size (P80 µm*)	Mass (%)	Grade Cu (%)	Grade Au (g/t)	Grade Ag (g/t)	Grade Mo (%)	Recovery (%) Cu	Recovery (%) Au	Recovery (%) Ag	Recovery (%) Mo
KM4514-27	DK-2014-MC5	Head	128/16/16	100.0	0.50	0.24	3	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.6	26.9	8.8	99	0.266	88.5	59.1	63.9	74.7
		Bulk Cleaner Tailing		16.5	0.17	0.44	3	0.004	5.6	29.5	19.4	11.7
		Au-Pyrite Concentrate		6.7	0.06	0.14	1	0.002	0.8	3.7	1.9	2.5
ALS 4514/31**	Mitchell (Mitchell Yr 0-5)/ Kerr (DK-2014-MC3)	Head	129/17/17	100.0	0.37	0.59	3	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.31	24.5	28.3	150	0.328	87.9	62.8	57.2	75.1
		Bulk Cleaner Tailing		10.6	0.13	1.07	5	0.004	3.6	19.1	15.4	7.4
		Au-Pyrite Concentrate		6.16	0.10	0.41	2	0.003	1.7	4.3	3.6	3.2

Note: *Repeat tests, the cleaner flotation for Test 26 was conducted on more diluted slurry (using a larger flotation cell

Leach Tests

ALS conducted the cyanidation tests on the first cleaner tailing and the gold-bearing pyrite concentrate produced from the LCTs. The leaching procedure used was similar to that used in the previous test programs. Test results are provided in Table 13.48

Table 13.48 Preliminary Cyanidation Test Results – Lower Kerr

Test	Sample	Extraction*	
		Gold (%)	Silver (%)
KM4029-29	Test 22,23 Bulk Cleaner Scavenger Tailings	72.6	86.2
KM4029-30	Test 22,23 Gold Bearing Pyrite Concentrate	77.2	90.8
KM4029-31	Test 16,18 Bulk Cleaner Scavenger Tailings	59.1	76.5
KM4029-32	Test 16,18 Gold Bearing Pyrite Concentrate	73.9	91.2
KM4029-33	Test 24,25 Bulk Cleaner Scavenger Tailings	57.5	73.1
KM4029-34	Test 24,25 Gold Bearing Pyrite Concentrate	72.2	n/a
KM4514-33	Bulk Cleaner Scavenger Tailings	64.2	82.6
KM4514-34	Gold Bearing Pyrite Concentrate	77.8	70.8
KM4514-35	Bulk Cleaner Scavenger Tailings	68.1	56.9
KM4514-36	Gold Bearing Pyrite Concentrate	72.9	78.9
KM4514-37	Bulk Cleaner Scavenger Tailings	66.3	78.2
KM4514-38	Gold Bearing Pyrite Concentrate	68.4	54.3
KM4514-39	Bulk Cleaner Scavenger Tailings	58.2	75.7
KM4514-40	Gold Bearing Pyrite Concentrate	63.7	72.1
KM4514-41	Bulk Cleaner Scavenger Tailings**	56.6	71.9
KM4514-42	Gold Bearing Pyrite Concentrate**	67.1	71.8

Note: *cyanide concentration: 1,000 ppm; pH: 11; carbon addition: 28 g/L;
** Mitchell (Mitchell Year 0-5)/ Kerr (DK-2014-MC3)

The gold extractions by cyanide leaching (CIL procedure) fluctuated from 56 to 73% for the bulk cleaner scavenger tailings and from 63 to 78% for the gold bearing pyrite concentrates.

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. In 2014 and 2015, further test work was conducted on the samples from the lower Iron Cap Zone. The assays of the head samples are provided in Table 13.49.

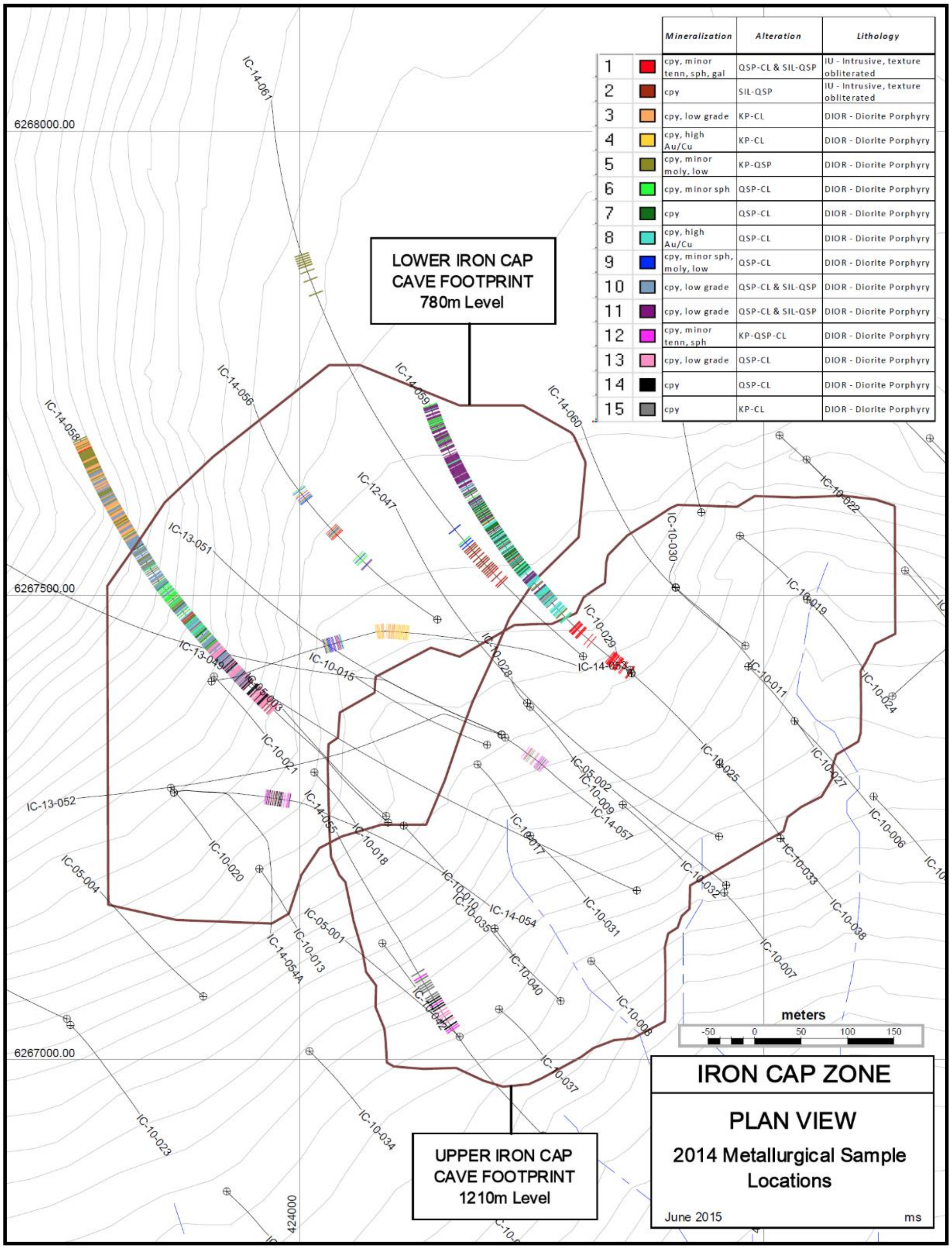
Table 13.49 Metal Contents of Composites – Iron Cap, 2010 (G&T)

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
IC-2013-01	0.24	-	-	0.59	0.008	3	2.4
IC-2013-02	0.25	-	-	0.49	0.007	4	3.3
IC-2013-03	0.22	-	-	0.28	0.003	4	2.2
IC-2014-01	0.47	-	-	0.28	0.007	10	3.7
IC-2014-02	0.38	-	-	0.11	0.003	4	4.0
IC-2014-03	0.17	-	-	0.47	0.002	2	2.6
IC-2014-04	0.30	-	-	1.08	0.001	3	3.2
IC-2014-05	0.31	-	-	0.39	0.007	1	1.7
IC-2014-06	0.25	-	-	0.33	0.004	4	2.4
IC-2014-07	0.44	-	-	1.27	0.002	4	3.7
IC-2014-08	0.33	-	-	1.72	0.002	4	3.3
IC-2014-09	0.41	-	-	0.16	0.007	5	2.3
IC-2014-10	0.18	-	-	0.34	0.002	1	1.3
IC-2014-11	0.16	-	-	0.34	0.002	2	2.1
IC-2014-12	0.25	-	-	0.47	0.002	9	3.6
IC-2014-13	0.23	-	-	0.26	0.004	2	1.6
IC-2014-14	0.36	-	-	0.47	0.005	3	2.1
IC-2014-15	0.19	-	-	0.66	0.003	2	2.5
IC-2014-MC1	0.34	0.008	0.020	0.63	0.003	5	3.1
IC-2014-MC2	0.26	0.005	0.018	0.46	0.004	4	2.5
IC-2014-MC3	0.16	0.003	0.010	0.35	0.002	2	1.8
IC-2014-MC4	0.30	0.005	0.018	0.52	0.004	4	2.8

The assay results indicate that arsenic and antimony contents range from 21 to 234 ppm for arsenic and from 1 to 201 ppm for antimony. Samples IC-2014-01 and IC-2014-12 show elevated arsenic content of 170 and 234 ppm and antimony content of 201 and 87 ppm.

The drill hole distribution and sample locations for these 2015 test samples are shown in Figure 13.30.

Figure 13.30 Sample Locations – 2015 Test Work



Mineralogy

In 2010 the mineral content, in each of the two master composites, was determined using the Bulk Mineral Analysis with Liberation (BMAL) function within Quantitative

Evaluation of Minerals by Scanning (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.

The 2015 mineralogical determination was conducted on Composite IC-2014-MC4. The estimated mineral contents are shown in Table 13.50. The sulphide minerals in the sample are approximately 6.3%. Most of the sulphide minerals are in the form of pyrite. Feldspars, micas, and quartz are the main gangue minerals.

Table 13.50 Mineral Content – Sample IC-2014-MC4 - Iron Cap, 2014 (ALS)

Minerals	Mineral Content (Wt %)
Chalcopyrite	0.9
Bornite	<0.1
Chalcocite/Covellite	<0.1
Tennantite/Enargite	<0.1
Pyrite	5.4
Iron Oxides	0.1
Quartz	18.3
Micas	25.6
Chlorite	1.8
Feldspars	44.7
Titanium Minerals	0.3
Carbonates	0.7
Kaolinite (clay)	0.2
Apatite	0.8
Calcium-sulphate	<0.1
Others	1.1

Notes: Iron oxides includes limonite, goethite, hematite and magnetite.
Micas include muscovite and biotite/phlogopite.
Feldspars includes K-feldspar, feldspar albite, alkali feldspar and plagioclase feldspar.
Titanium minerals includes rutile/anatase and minor amounts of sphene.
Carbonates includes calcite, ankerite and dolomite.
Others includes trace amounts of barite, sphalerite, and unresolved mineral species.

Mineralization Hardness

The 2010 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.

The IC-2014-MC4 master composite tested by the 2015 test work shows a slightly higher Bond ball mill work index of 16.5 kWh/t. The Ai was measured to be 0.099 g. The program also tested the grindability of the sample to SAG mill grinding using the SMC procedure. The results show that the SMC parameters are: A = 68.7, b = 0.54, and Axb = 37.1.

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 conducted in 2010 showed that the Iron Cap mineralization was not sensitive to the primary grind sizes ranging from 80% passing 120 µm to 170 µm. In 2015, rougher flotation tests were performed on Composite IC-2014-MC4 to investigate the effect of primary grind size on copper and gold recovery. The grind size ranged from 80% passing between 89 and 171 µm. It appears that copper and gold recoveries improved when the grind size got finer. The test results are depicted in Figure 13.31 and Figure 13.32.

Figure 13.31 Copper Recovery vs Rougher Mass Recovery and Grind Size - Iron Cap, 2015 (ALS)

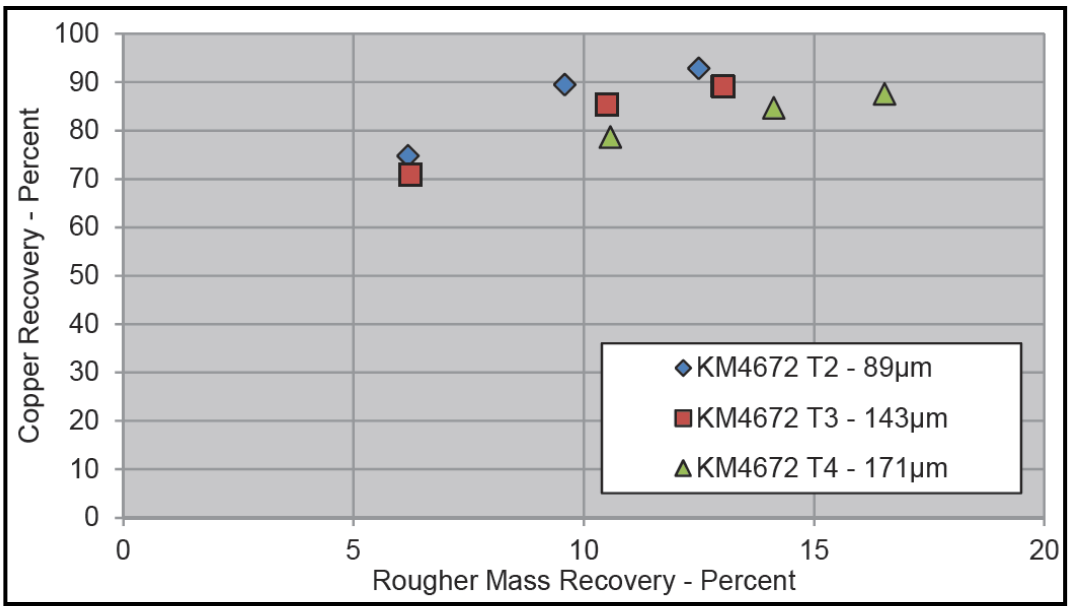
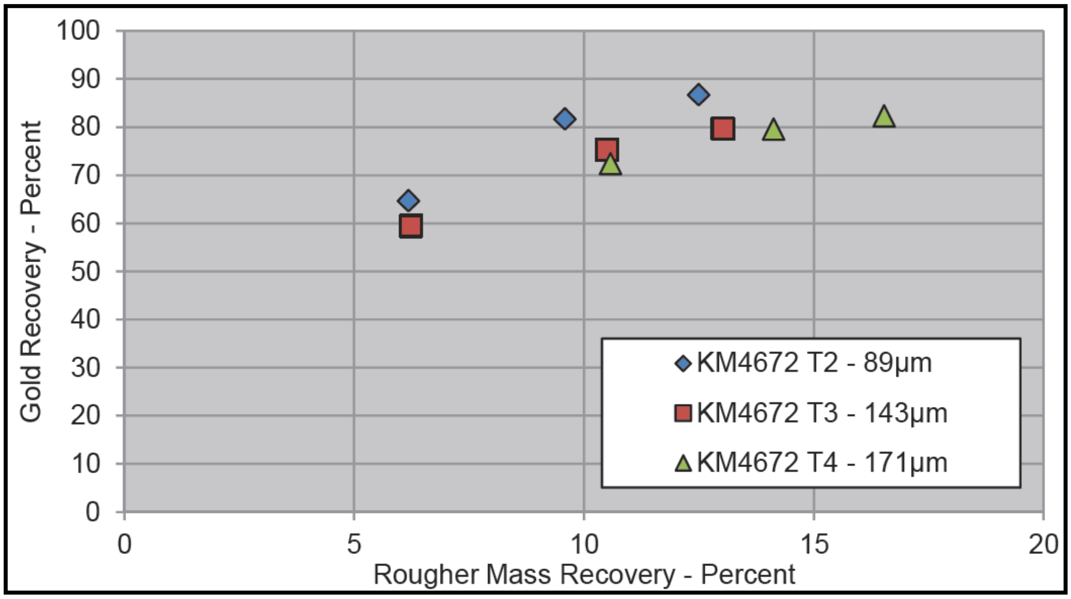


Figure 13.32 Gold Recovery vs Rougher Mass Recovery and Grind Size - Iron Cap, 2015 (ALS)



Cleaner flotation tests on four master composite samples showed a very similar copper metallurgical performance although there were some variations in gold performance. The test results are summarized in Figure 13.33 and Figure 13.34. Copper grades in the bulk concentrates produced from the four Iron Cap master composites averaged approximately 27.7%. Copper and gold recovery to the bulk cleaner concentrates averaged approximately 77% and 61%, respectively.

Figure 13.33 Copper Cleaner Flotation Performance - Iron Cap, 2015 (ALS)

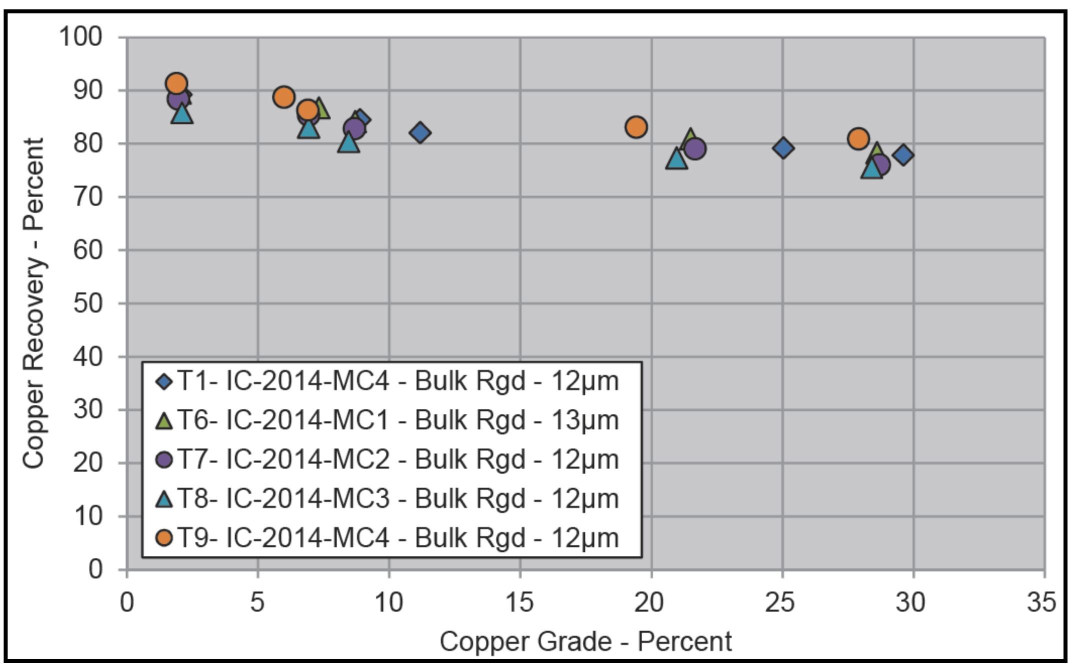
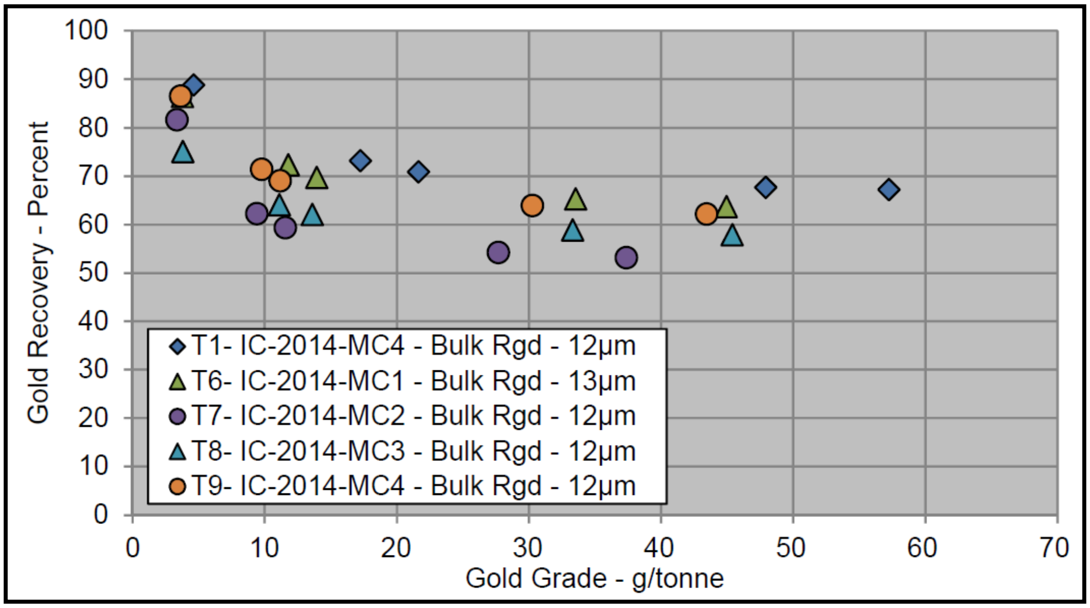


Figure 13.34 Gold Cleaner Flotation Performance - Iron Cap, 2015 (ALS)



As shown in Figure 13.35 and Figure 13.36, copper recoveries to the bulk concentrates for the fifteen Iron Cap variability samples ranged from 74 to 85% and averaged approximately 81%. The bulk concentrate grades ranged from approximately 25 to 30% copper, averaging approximately 27% copper. Variation in gold recovery was observed. On average, approximately 57% of the gold was recovered to the final cleaner concentrates.

Figure 13.35 Variability Test Results - Copper - Iron Cap, 2015 (ALS)

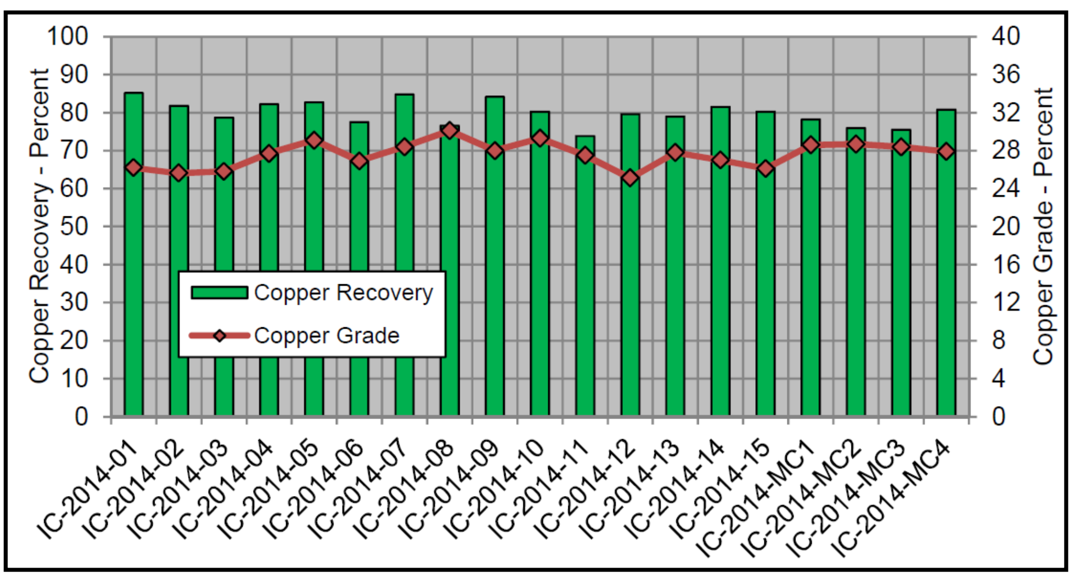
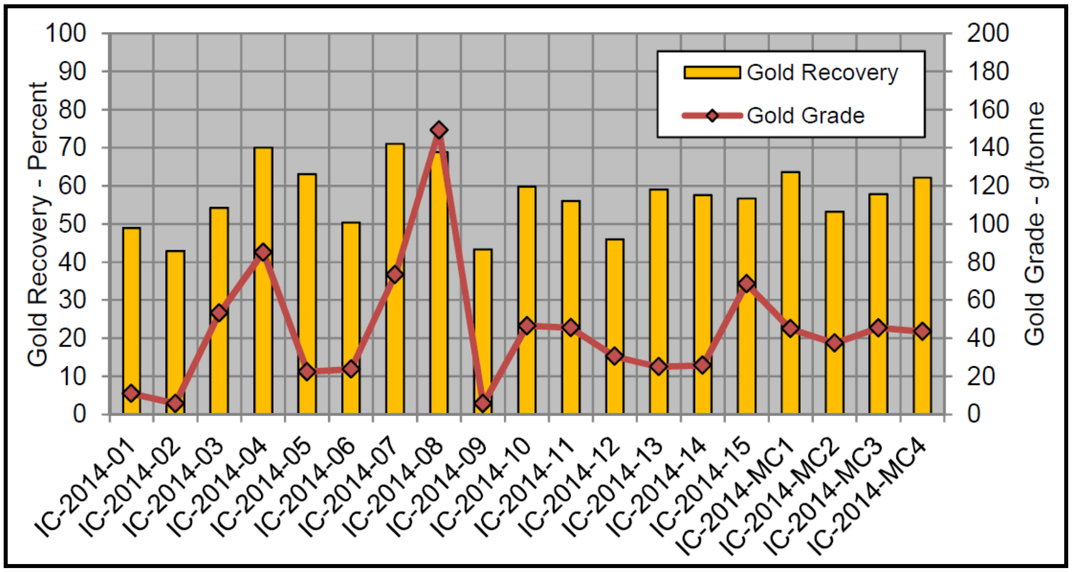


Figure 13.36 Variability Test Results - Gold - Iron Cap, 2015 (ALS)



The flotation LCT results are provided in Table 13.51. On average, the samples tested in 2010 produced a 25.7% copper 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 2014/2015 locked cycle flotation tests on the master composite samples produced fairly consistent results. Between 81 to 88% of the copper in the feed was recovered to a bulk concentrate grading between approximately 22 to 27% copper. On average, approximately 85.2% of the copper and 64.4% of the gold in the feed were recovered into the bulk concentrates. The gold reporting to the gold-bearing pyrite products (first cleaner tailing and gold-bearing pyrite concentrate) was approximately 29% of the gold. The averaged silver recovery to the flotation concentrate was 56.6%.

The results indicate that the copper recoveries from the Iron Cap samples were comparable to the Mitchell mineralization. It was concluded that the gold recoveries to the concentrates from the 2012 samples were lower than these achieved with the Mitchell mineralization; however, the 2014/2015 test work produced better gold recoveries to the flotation concentrates and the results are in line with the Mitchell mineralization. The averaged silver recovery to the flotation concentrate is slightly higher than the recovery achieved from the Mitchell mineralization. On average, approximately 61% of the molybdenum from the Iron Cap's samples reported to the final bulk concentrate.

As shown in Table 13.23, the Mitchell and Iron Cap blended samples did not show detrimental effects of the blending on the metallurgical responses.

Table 13.51 Locked Cycle Test Results – Iron Cap

Test Program	Composite	Product	Primary/ Regrinding Size (P80 µm)	Mass (%)	Grade Cu (%)	Grade Au (g/t)	Grade Ag (g/t)	Grade Mo (%)	Flotation Recovery (%) Cu	Flotation Recovery (%) Au	Flotation Recovery (%) Ag	Flotation Recovery (%) 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
ALS 4029/19	IC-2013-01	Head	117/16/7	100.0	0.25	0.56	3	0.006	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.8	26.7	50.4	179	0.481	83.4	71.4	48.4	69.0
		Bulk Cleaner Tailing		16.5	0.10	1.71	4	0.003	10.8	21.6	22.5	12.8
		Au-Pyrite Concentrate		5.8	0.06	0.08	1	0.004	1.3	0.8	2.9	4.3
ALS 4029/20	IC-2013-02	Head	119/17/11	100.0	0.26	0.51	4	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.0	22.9	36.4	273	0.238	85.4	68.4	62.8	64.4
		Bulk Cleaner Tailing		16.2	0.13	0.88	5	0.002	7.0	27.6	17.4	10.5
		Au-Pyrite Concentrate		6.4	0.06	0.08	1	0.002	1.5	1.0	1.5	3.6
ALS 4029/21	IC-2013-03	Head	130/15/11	100.0	0.23	0.24	4	0.003	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.9	23.4	14.8	258	0.205	87.5	53.4	56.3	56.4
		Bulk Cleaner Tailing		14.1	0.07	0.61	6	0.003	4.6	35.1	21.1	10.0
		Au-Pyrite Concentrate		5.4	0.04	0.15	2	0.005	1.0	3.4	2.7	8.8

table continues...

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
ALS 4514/30	IC-2014-MC1	Head	125/14/14*	100.0	0.33	0.69	4	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.1	25.4	41.5	230	0.238	87.1	67.8	60.1	70.3
		Bulk Cleaner Tailing		12.3	0.15	1.12	7	0.003	5.6	20.0	18.6	9.7
		Au-Pyrite Concentrate		5.2	0.09	0.29	2	0.003	1.4	2.1	2.4	4.1
ALS 4514/29	IC-2014-MC2	Head	127/14/16*	100.0	0.25	0.45	4	0.004	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.9	23.4	29.3	275	0.311	85.7	59.1	56.0	76.2
		Bulk Cleaner Tailing		11.4	0.14	1.28	9	0.003	6.4	32.2	22.9	9.2
		Au-Pyrite Concentrate		6.9	0.05	0.28	2	0.002	1.5	4.2	3.1	3.7
ALS 4514/31	IC-2014-MC3	Head	124/16/18*	100.0	0.16	0.32	2	0.002	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		0.6	22.6	37.6	139	0.187	81.4	67.9	47.4	62.9
		Bulk Cleaner Tailing		8.5	0.13	0.89	5	0.002	7.0	23.1	22.2	9.7
		Au-Pyrite Concentrate		4.5	0.07	0.27	2	0.001	2.0	3.7	5.2	2.6
ALS 4514/25	IC-2014-MC4	Head	124/14/15*	100.0	0.28	0.56	4	0.003	100.0	100.0	100.0	100.0
		Cu/Mo Concentrate		1.0	24.9	36.2	250	0.257	85.7	62.9	65.1	73.6
		Bulk Cleaner Tailing		14.3	0.15	1.03	6	0.003	7.6	26.3	22.9	12.0
		Au-Pyrite Concentrate		5.0	0.05	0.19	1	0.002	0.8	1.7	1.3	2.4

Note: *including a copper flotation on the pyrite flotation concentrate

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 13.52.

Table 13.52 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	-	-
ALS-4029	IC-2013-01/02/03 Cl.Sc.Tls	16	0.8	45.8	4.4	70.7
ALS-4029	IC-2013-01/02/03 Py Conc	10	0.2	54.7	1.6	87.4
ALS-4672	IC-2014-MC1 Cl.Sc.Tls	14	1.1	46.7	6	68.0
ALS-4672	IC-2014-MC2 Cl.Sc.Tls	14	1.2	29.2	7	69.7
ALS-4672	IC-2014-MC4 Cl.Sc.Tls	14	1.1	40.1	6	60.5
ALS-4672	IC-2014-MC1 Py Conc	14	0.4	50.6	3	57.2
ALS-4672	IC-2014-MC2 Py Conc	16	0.3	36.3	2	72.0
ALS-4672	IC-2014-MC4 Py Conc	15	0.1	74.9	5	73.2
Average – Iron Cap		**15**	**1.0**	**46.8**	**5.6**	**66.5**

On average, the gold extraction from both the gold-bearing products was approximately 47%. 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 67%, which is 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 bio-oxidation would be required to release this locked gold.

FLOTATION CONCENTRATE ASSAY

The multi-element assay data are provided in Table 13.53 to Table 13.55 for the concentrates from these 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 13.53 Multi-Element Assay – Mitchell Concentrate*

Element	Unit	Mitchell									
		2153/142 Master Comp	2344/73 Comp PP1	2535/18 Comp PP1	2535/20 Comp PP1	SGS/LCT1 Comp PP1	2670/18 Comp PP3	2670/ Pilot Plant Comp PP3	3081/82 Comp Y0-20	3081/93** Comp Y0-10	4514/30 Mitchell Y0-5
Cu	%	22.0	22.3	28	23.8	23.1	27.4	25.7	23.8	27.1	26.7
Au	g/t	64.7	55.7	77.8	62.0	53.7	70.5	65.5	44.2	58.5	98.2
Ag	g/t	257	-	260	248	-	275	304	223	427	431
Mo	%	-	0.23	0.12	0.12	0.41	0.62	0.33	0.24	0.33	0.72
S (T)	%	33.4	34.4	34.7	32.9	38.1	34.5	31.1	35.3	34.2	35.0
S (-2)	%	-	-	32.9	32.1	-	33.3	28.7	-	-	-
Fe	%	26.8	30.8	29.6	30.7	32.7	30.1	27.6	32.6	30.9	28.0
Sb	ppm	696	698	539	597	-	466	338	210	1,100	1,182
As	ppm	1,184	934	824	878	-	1174	821	690	2,044	2,080
Co	ppm	48	76	52	52	-	68	56	84	62	30
Cd	ppm	72	44	60	84	-	88	80	54	112	79
Bi	ppm	36	43	150	127	-	<10	<10	<20	<20	24
Hg	ppm	0.6	<1	<1	<1	-	1	<1	<1	<1	6.8
Ni	ppm	120	240	112	156	-	48	80	70	66	41
F	ppm	346	150	100	148	-	89	230	69	129	249
Cl	ppm	-	-	-	-	-	<0.01	<0.01	-	-	-
Se	ppm	72	102	82	70	-	73	70	59	75	-
P	ppm	230	215	146	189	-	55	492	52	81	<100
Pb	%	0.92	0.19	0.19	0.22	-	0.32	0.23	0.12	0.17	0.36
Zn	%	0.42	0.23	0.25	0.38	-	0.43	0.32	0.26	0.27	0.41
SiO_2	%	9.84	6.67	2.39	7.11	-	3.04	8.23	4.26	2.93	-
CaO	%	0.54	0.53	0.39	0.54	-	0.27	0.74	0.42	0.52	0.38

table continues...

Element	Unit	Mitchell									
		2153/142 Master Comp	2344/73 Comp PP1	2535/18 Comp PP1	2535/20 Comp PP1	SGS/LCT1 Comp PP1	2670/18 Comp PP3	2670/ Pilot Plant Comp PP3	3081/82 Comp Y0-20	3081/93** Comp Y0-10	4514/30 Mitchell Y0-5
Al_2O_3	%	3.31	1.76	0.62	1.37	-	0.57	1.83	0.98	0.85	1.11
MgO	%	0.48	0.36	0.18	0.34	-	0.15	0.47	0.16	0.13	0.17
MnO	%	0.02	0.03	0.011	0.026	-	0.015	0.035	0.012	0.009	0.010
Insol	%	-	8.46	4.02	8.87	-	3.23	10.3	-	-	-

Notes: *copper-gold/molybdenum concentrate before molybdenum separation
**test program and test ID

Table 13.54 Multi-element Assay – Sulphurets/Upper Kerr/Blend Concentrate*

Element	Unit	Sulphurets			Sulphurets/ Mitchell	Upper Kerr				Mitchell/ Upper Kerr
		2670/44 Comp	3174/8 Comp 8	3174/9 Comp 9	2670/62 Blend	2535/16 Comp 52	2535/17 Comp 53	3174/10 Comp 10	3174/11 Comp 11	2535/19** Blend
Cu	%	28.3	29.3	26.0	24.2	22.3	29.3	30.7	29.0	25.3
Au	g/t	41.8	31.4	63.7	52.0	4.05	5.58	7.2	5.1	40
Ag	g/t	82	34	130	178	33.5	31.8	49	77	168
Mo	%	0.70	0.227	0.170	0.66	0.013	0.017	0.023	0.038	0.056
S (T)	%	33.6	32.4	34.4	34.9	27.1	35.3	34.0	36.1	35.0
S (-2)	%	31.2	-	-	32.2	25.9	33.8	-	-	33.4
Fe	%	29.6	27.2	29.3	30.0	23.7	27.5	29.4	29.1	29.3
Sb	ppm	445	2,100	370	500	24	121	620	180	492
As	ppm	224	1,768	205	969	143	3,276	621	2793	1,369
Co	ppm	92	-	-	104	40	52	-	-	68
Cd	ppm	180	68	26	144	20	8	6	32	80
Bi	ppm	<10	-	-	<10	95	105	-	-	121
Hg	ppm	2	-	-	1	3.4	12	-	-	2.4
Ni	ppm	88	-	-	96	132	168	-	-	164
F	ppm	155	-	-	174	320	88	-	-	116
Cl	ppm	<0.01	-	-	<0.01	-	-	-	-	-
Se	ppm	118	-	-	89	140	109	-	-	76
P	ppm	92	-	-	113	1045	233	-	-	224
Pb	%	0.26	0.19	0.72	0.26	0.03	0.05	0.04	0.05	0.15
Zn	%	0.54	0.29	0.18	0.92	0.30	0.10	0.08	0.75	0.42
SiO_2	%	4.14	-	-	5.82	14.0	3.9	-	-	5.12
CaO	%	0.41	-	-	0.38	0.83	0.17	-	-	0.43

table continues…

Element	Unit	Sulphurets			Sulphurets/ Mitchell	Upper Kerr				Mitchell/ Upper Kerr
		2670/44 Comp	3174/8 Comp 8	3174/9 Comp 9	2670/62 Blend	2535/16 Comp 52	2535/17 Comp 53	3174/10 Comp 10	3174/11 Comp 11	2535/19** Blend
Al_2O_3	%	0.92	-	-	1.18	3.92	0.85	-	-	0.99
MgO	%	0.25	-	-	0.29	0.70	0.14	-	-	0.26
MnO	%	0.017	-	-	0.022	0.050	0.015	-	-	0.018
Insol	%	4.90	-	-	7.21	19.6	5.42	-	-	6.67

Notes: *copper-gold/molybdenum concentrate before molybdenum separation
**test program and test ID

Table 13.55 Multi-element Assay – Iron Cap/Blend Concentrate*

Element	Unit	Iron Cap									Mitchell/Iron Cap	
		2748/11 Comp 1	2748/12 Comp 2	4672/25 IC-2014-MC4	4672/29 IC-2014-MC2	4672/30 IC-2014-MC1	4672/31 IC-2014-MC3	4029/19 IC-2013-01	4029/20 IC-2013-02	4029/21 IC-2013-03	2535/19 Mitchell/Iron Cap Blend	4672/32** Mitchell/Iron Cap Blend
Cu	%	25.4	24.9	24.9	23.4	25.4	22.6	26.7	23.2	23.3	25.3	25.0
Au	g/t	146.8	10.9	36.2	29.3	41.5	37.6	52.3	36.9	14.7	40	54.3
Ag	g/t	774	255	235	286	243	143	176	281	253	168	332
Mo	%	0.18	0.12	0.247	0.306	0.245	0.197	0.48	0.23	0.2	0.056	0.454
S (T)	%	32.6	33.5	33.3	35.6	33.3	34.8	32.9	35.7	32.3	35.0	35.5
S (-2)	%	32.4	32.2	-	-	-	-	-	-	-	33.4	
Fe	%	26.5	27.8	27.5	27.7	26.6	28.9	26.4	27.9	25.4	29.3	30.8
Sb	ppm	4,379	2,876	3,200	2,893	3,272	274	530	2,100	4,500	492	2,116
As	ppm	3,067	1,107	1,890	1,768	2,089	383	740	2,600	3,500	1,369	1,970
Co	ppm	50	68	33	43	35	80	48	42	40	68	44
Cd	ppm	320	128	128	183	141	59	74	364	152	80	114.7
Bi	ppm	205	164	44	30	54	40	35	68	46	121	39.7
Hg	ppm	<1	2	13	6	13	3	2	10	25	2.4	7
Ni	ppm	50	88	22	34	34	28	50	50	48	164	42.4
F	ppm	162	494	310	439	378	297	-	-	-	116	265
Cl	ppm	<0.01	<0.01	-	-	-	-	-	-	-	-	
Se	ppm	180	108	230	-	-	-	239	202	187	76	-
P	ppm	143	135	100	100	100	100	141	136	173	224	<0.01
Pb	%	1.31	0.43	0.46	0.81	0.46	0.54	0.21	0.65	0.78	0.15	0.49
Zn	%	2.29	1.02	0.9	1.27	1.08	0.26	0.36	2.67	1.15	0.42	0.72
SiO_2	%	3.16	5.59	-	-	-	-	5.95	5.43	9.07	5.12	0.00
CaO	%	0.34	0.29	0.42	0.53	0.42	0.50	0.52	0.36	0.62	0.43	0.36

table continues…

Element	Unit	Iron Cap									Mitchell/Iron Cap	
		2748/11 Comp 1	2748/12 Comp 2	4672/25 IC-2014-MC4	4672/29 IC-2014-MC2	4672/30 IC-2014-MC1	4672/31 IC-2014-MC3	4029/19 IC-2013-01	4029/20 IC-2013-02	4029/21 IC-2013-03	2535/19 Mitchell/Iron Cap Blend	4672/32** Mitchell/Iron Cap Blend
Al_2O_3	%	0.85	1.28	1.53	2.32	2.10	1.97	1.89	1.42	2.53	0.99	1.28
MgO	%	0.12	0.18	0.18	0.30	0.18	0.27	0.27	0.10	0.28	0.26	0.17
MnO	%	0.01	0.02	0.02	0.02	0.01	0.02	0.03	0.01	0.03	0.02	0.01
Insol	%	5.15	7.66	-	-	-	-	-	-	-	6.67	-

Note: *copper-gold/molybdenum concentrate before molybdenum separation
**test program and test ID

Table 13.56 Multi-element Assay – Lower Kerr/Blend Concentrate*

Element	Unit	Lower Kerr											Mitchell/ Lower Kerr
		4029/16**	4029/18	4029/22	4029/23	4029/25	4029/26	4514/23	4514/24	4514/25	4514/26	4514/27	4514/31
		DK-2013-04	DK-2013-03	DK-2013-01	DK-2013-02	DK-2013-06	DK-2013-05	DK-2014-MC3	DK-2014-MC1	DK-2014-MC2	DK-2014-MC4	DK-2014-MC5	Mitchell/DK-2014-MC3
Cu	%	26.3	23.0	25.3	24.4	28.4	28.7	23.5	24.0	26.4	27.4	26.9	24.5
Au	g/t	10.6	32.9	11.5	9.0	9.3	11.2	10.2	14.8	7.3	18.6	8.80	28.3
Ag	g/t	48	136	36	48	50	44	54	42	74	34	99	150
Mo	%	0.071	0.215	0.243	0.157	0.039	0.010	0.164	0.091	0.210	0.104	0.266	0.328
S	%	36.5	32.9	35.2	34.5	33.3	38.4	36.7	36.1	35.7	35.2	35.7	34.6
Fe	%	30.0	27.7	29.5	29.0	28.0	32.2	30.5	28.6	28.4	30.9	29.8	29.6
Sb	%	0.082	0.075	0.10	0.035	0.017	0.19	648	237	1202	32	1964	814
As	%	0.23	0.21	0.23	0.12	0.055	0.17	1230	810	1774	<5	2875	1404
Co	ppm	50	52	54	56	48	44	48	53	38	37	37	43
Cd	ppm	24	30	14	16	20	10	13.7	8.7	24.7	4.4	35.5	47.3
Bi	ppm	<20	<20	<20	<20	<20	56	6.5	5.4	11.8	3.1	15.3	12.2
Hg	ppm	64	15	68	9	17	88	29	38	17	27	21	23
Ni	ppm	68	86	88	58	86	74	47.3	65.2	44.4	36.1	40.8	73.7
F	ppm	120	200	130	130	150	20	220	216	212	253	192	264
Cl	%	-	-	-	-	-	-	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Se	ppm	200	195	218	216	219	247	240	-	-	-	-	-
P	ppm	0.01	0.03	0.02	0.02	0.02	<0.01	0.01	0.01	0.01	<0.01	<0.01	<0.01
Pb	%	0.08	0.09	0.07	0.07	0.05	0.03	0.04	0.04	0.05	0.02	0.07	0.12
Zn	%	0.31	0.28	0.08	0.06	0.24	0.06	0.10	0.06	0.16	0.01	0.24	0.29
SiO2	%	3.40	8.00	4.13	6.27	5.43	2.76	-	-	-	-	-	-
CaO	%	0.38	0.71	0.53	0.63	0.39	0.11	0.48	0.45	0.60	0.35	0.43	0.41

table continues…

Element	Unit	Lower Kerr											Mitchell/ Lower Kerr
		4029/16** DK-2013-04	4029/18 DK-2013-03	4029/22 DK-2013-01	4029/23 DK-2013-02	4029/25 DK-2013-06	4029/26 DK-2013-05	4514/23 DK-2014-MC3	4514/24 DK-2014-MC1	4514/25 DK-2014-MC2	4514/26 DK-2014-MC4	4514/27 DK-2014-MC5	4514/31 Mitchell/DK-2014-MC3
Al_2O_3	%	1.11	2.38	1.51	2.06	1.42	0.21	1.02	1.85	1.45	1.38	1.02	1.80
MgO	%	0.12	0.30	0.18	0.41	0.12	0.03	0.27	0.38	0.30	0.25	0.22	0.33
MnO	%	0.01	0.04	0.01	0.03	0.01	<0.01	0.01	0.01	0.01	0.01	0.01	0.01

Note: *copper-gold/molybdenum concentrate before molybdenum separation.
**test program and test ID

However, arsenic, antimony, and mercury contents in some of the concentrates from the Iron Cap deposit and the Kerr samples may attract smelting penalties. Also the lead content of the concentrate from the Iron Cap Comp 1 may be higher than the penalty thresholds. Fluorine levels in some of the concentrates may be also higher than the penalty thresholds. It is anticipated that the Iron Cap and Kerr mill feeds will be processed together with the ore from the Mitchell deposit. Impurity contents in the copper concentrates produced from these blended mill feeds should be lower than the penalty thresholds set by most of the smelters. Further review with respect to smelting penalties should be conducted.

ANCILLARY TESTS

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

- leach residue cyanide destruction, including sulphur dioxide/air, Caro's acid (H_2SO_5), and hydrogen peroxide (H_2O_2)
- cyanide recovery from barren solutions, including the acidification, volatilization of hydrogen cyanide gas, and re-neutralization (AVR) process and the sulphidization, acidification, recycling, and thickening of precipitate (SART) process
- static and dynamic thickening tests for conventional thickener sizing and for high-rate thickener sizing for primary grinding product, first cleaner tailing together with 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-scale, 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 sodium cyanide consumed by the formation of thiocyanate was 1 kg/t feed, while 0.5 kg/t feed equivalent sodium cyanide was oxidized to cyanate. The amount of equivalent sodium cyanide 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 sodium cyanide.

The cyanide complexed with copper and the free cyanide should be recoverable by the AVR 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 13.57.

Table 13.57 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; CNS = thiocyanate

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 13.58.

Table 13.58 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 hydrosulphide (NaHS) dosage on recovering cyanide and copper from CN_{WAD} and copper cyanide complexes. The test results are as follows:

- At a sodium hydrosulphide 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 sodium hydrosulphide 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 re-neutralization 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 sulphur dioxide/air, Caro's acid, and hydrogen peroxide were tested for oxidation of cyanide and detoxification of the washed pulp. The objective of the test work was to produce treated effluent containing less than 2 mg/L CN_{WAD}. The results of the cyanide destruction test results are summarized in Table 13.59.

Table 13.59 Cyanide Destruction Test Results – 2009/2010 (SGS)

Test	Method	Oxidant Dosage Stoich (%)	Cumulative Retention Time (~h)	Composition (Solution Phase)			Cumulative Reagent Addition* (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**	9.0	2.8	1.7	-	37	-	21.9	-	-
H-1	H_2O_2	500	1.5**	10.1	12	11	-	-	-	-	6.5	-
SO_2/Air Partially Treated Pulp				10.0	10	10	-		-	-	-	-
C-2	Caro's Acid	500	1.5**	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	-

Notes: *copper added as $CuSO_4\ _5H_2O$; SO_2 added as $Na_2S_2O_5$
**reagent added in three 30-min stages

The results indicated that the residual CN_{WAD} in the washed pulp was reduced to less than 1 mg/L after the pulp was treated with 4 to 5 g equivalent sulphur dioxide and 0.14 g copper (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 sulphur dioxide/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 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 hydrogen peroxide.

Two-stage cyanide destruction involving sulphur dioxide/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 sulphur dioxide/air treated pulp was adjusted with sodium cyanide 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 hydrogen peroxide polishing treatment produced less than 2 mg/L residual CN_{WAD}. The hydrogen peroxide dosage was 10 times of the stoichiometric requirement and the copper addition was 0.011 g/L pulp.
- The solution phase (filtrate) of the sulphur dioxide/air partially treated pulp responded well to the hydrogen peroxide polishing treatment. The solution contained less than 1 mg/L residual CN_{WAD} after being treated with five times the stoichiometric hydrogen peroxide 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 13.60 and Table 13.61.

Table 13.60 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 Concentrate and 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 ($m^3/m^2/h$) 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 P80 170 um; simulating stage one primary grind size.

Table 13.61 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 13.62.

Table 13.62 Filtration Test Results – 2009 (Pocock)

Filtration Method	Bulk Cake Density (dry kg/m³)	Cake Thickness (mm)	Cake Moisture (%)	Filtration Rate (dry kg/m²h)	Dry Cake Weight (dry kg/m²)
Vacuum	1,785	15	19	265*	-
Pressure	2,511	51	8	-	117.8**

Notes: *includes scale up factors at vacuum of 67.7 kPa.
**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 deposit and the upper zone of the Kerr deposit.

The test results indicate that the samples from all the deposits are moderately hard to ball mill and SAG mill grinding, excluding the samples from the Sulphurets deposit showing much resistance to both ball mill and SAG mill grinding. The communition tests also show that the samples tested are amenable to particle size reduction by HPGR procedure.

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% copper with variable precious metal content and a molybdenum concentrate with 50% molybdenum. 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 13.63 to Table 13.66. The estimates are based on a primary grind size of 80% passing approximately 125 to 150 µm and a regrind size of 80% passing approximately 20 µm.

Table 13.63 Cu-Au Concentrate – Cu Grade

Cu Head (%)	Cu Grade (%)
>0.80	28
0.40-0.80	26
0.15-0.40	25
0.10-0.15	23
0.05-0.10	17
<0.05	5

Table 13.64 Cu-Au Concentrate – Metal Recovery Projections

Deposit	Description	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	= 0.0967 x (Cu Recovery, %) $^{1.4465}$
	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%
		0.49 - 0.8% Cu	= 90%
		0.10 - 0.49% Cu	= 90.786 x (Cu Head, %) $^{0.089}$
		0.05 - 0.10% Cu	= 30%
		<0.05% Cu	= 3%
	Gold Recovery	>2.0 g/t Au	= 80%
		0.75 - 2.0 g/t Au	= 72.5%
		0.05 - 0.75 g/t Au	= 78.128 x (Au Head, g/t) $^{0.3012}$
		<0.05 g/t Au	= 20
	Silver Recovery	>20 g/t Ag	= 83%
		10 - 20 g/t Ag	= 78%
		0.5 - 10 g/t Ag	= 39.945 x (Ag Head, g/t) $^{0.2602}$
		<0.5 g/t Ag	= 5%

Table 13.65 Au-Ag Doré – Metal Recovery Projections

Deposit	Head Grade	Recovery
Mitchell	*Gold*	
	>10 g/t Au	= (98 - (0.096 x (Cu Recovery, %) $^{1.446}$)) x 80% x 98%
	5 - 10 g/t Au	= (95 - (0.096 x (Cu Recovery, %) $^{1.446}$)) x 75% x 98%
	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%
	<0.1 g/t Au	= 0%
	Silver	
	>15 g/t Ag	= 88 - (1.427 x (Cu Recovery, %) - 70.11)
	8- 15 g/t Ag	= 86 - (1.427 x (Cu Recovery, %) - 70.11)
	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%
	<1 g/t Ag	= 0%
Sulphurets	*Gold*	
	>10 g/t Au	= (98 - (52.07 x ln(Cu Recovery, %) - 174.1)) x 70% x 98%
	5 - 10 g/t Au	= (95 - (52.07 x ln(Cu Recovery, %) - 174.1)) x 60% x 98%
	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%
	<0.1 g/t Au	= 0%
	Silver	
	>15 g/t Ag	= 52.7%
	8- 15 g/t Ag	= 50.7%
	1 - 8 g/t Ag	= (42.74 x (Ag Head, g/t) $^{0.336}$) - (1.065 x (Cu Recovery, %) - 44.80)
	<1 g/t Ag	= 0%
Kerr	*Gold*	
	>10 g/t Au	= (98 - (171.8 x ln(Cu Recovery, %) - 718)) x 75% x 98%
	5 - 10 g/t Au	= (95 - (171.8 x ln(Cu Recovery, %) - 718)) x 65% x 98%
	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%
	<0.1 g/t Au	= 0%
	Silver	
	>15 g/t Ag	= (88 - (132.48 x ln(Cu Recovery, %) - 542.9))/100; Cap at 88%
	8- 15 g/t Ag	= (86 - (132.48 x ln(Cu Recovery, %) - 542.9))/100; Cap at 86%
	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
	<1 g/t Ag	= 0%
Iron Cap	*Gold*	
	>2 g/t Au	= 8%
	0.05 - 2 g/t Au	= 10%
	<0.05 g/t Au	= 5%
	Silver	
	>20 g/t Ag	= 8%
	10 - 20 g/t Ag	= 11%
	0.5 - 10 g/t Ag	= 16%
	<0.5 g/t Ag	= 5%

Table 13.66 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 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, 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 QP for this report as per the requirements 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.

This section outlines the various methods that were used to estimate Mineral Resources for the Project, which currently consists of four known mineralized areas for which Mineral Resources have been estimated. Block model grades for two of those areas (Sulphurets and Mitchell) are based on grade models that were developed for the 2012 PFS (Tetra Tech 2012). Various statistical and modeling parameters associated with the Sulphurets and Mitchell deposits were taken from Tetra Tech (2012), although the Mineral Resource tabulations for those two areas have been updated to account for different metal prices and conceptual pit and block cave constraints that were used to better demonstrate "that there are reasonable prospects for eventual economic extraction".

Mineral Resources for the Kerr and Iron Cap deposits have been updated since the last public disclosures for those areas. Statistics, modeling parameters, and Mineral Resource tabulations for the Kerr and Iron Cap deposits are discussed in Sections 14.2 and 14.3.

14.1 SULPHURETS AND MITCHELL DEPOSITS

No material drilling has been completed within the Sulphurets or Mitchell Mineral Resource areas since Tetra Tech (2012) was filed. The following Sections (14.1.1 through 14.1.9) were taken nearly verbatim from Tetra Tech (2012). Descriptions for the Kerr and Iron Cap zones have been removed and are updated in Sections 14.2 and 14.3 of this Technical Report, respectively.

14.1.1 GOLD GRADE DISTRIBUTION – SULPHURETS AND MITCHELL

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 14.1.3 discusses grade capping. Various geologic wireframes were used to constrain the estimate of block grades for each zone. These geologic wireframes represent a combination of alteration, lithology, and gold grade (Sulphurets) and gold grade envelopes (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 and Table 14.2 for the Sulphurets and Mitchell deposits, respectively.

As shown in Table 14.1 and Table 14.2, the average gold grade increases in going from the Sulphurets to the Mitchell deposit Another important statistical parameter is that the CV is relatively low for the Sulphurets and Mitchell 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 14.1.3).

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 14.1.5.

14.1.2 Copper Grade Distribution – Sulphurets and Mitchell

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 14.3 and Table 14.4 for the Sulphurets and Mitchell deposits, respectively.

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 14.1.5).

Table 14.1 Distribution of Gold by AUZON – Sulphurets Zone

	Uncapped Au Statistics Above Cut-off								Capped Au Statistics Above Cut-off				
AUZON	Au Cut-off (g/t)	Total Metres (m)	Inc. (%)	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. (%)	Std Dev	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. (%)	Std Dev	CV
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
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
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 14.2 Distribution of Gold by AUZON – Mitchell

AUZON	Au Cut-off (g/t)	Uncapped Au Statistics Above Cut-off: Total Metres (m)	Inc. (%)	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. (%)	Std Dev	CV	Capped Au Statistics Above Cut-off: Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. (%)	Std Dev	CV
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
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 14.3 Distribution of Copper by CUZON – Sulphurets Zone

	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
CUZON	Cu Cut-off (g/t)	Total Metres (m)	Inc. (%)	Mean Cu (%)	Grd-Thk (%-m)	Inc. (%)	Std. Dev	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. (%)	Std. Dev	CV
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
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
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 14.4 Distribution of Copper by CUZON – Mitchell Zone

	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
CUZON	Cu Cut-off (g/t)	Total Metres (m)	Inc. (%)	Mean Cu (%)	Grd-Thk (%-m)	Inc. (%)	Std. Dev	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. (%)	Std. Dev	CV
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
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

14.1.3 ASSAY GRADE CAPPING – SULPHURETS AND MITCHELL

RMI used cumulative probability plots to identify high-grade outliers for both gold and copper assays. Figure 14.1 through Figure 14.4 show cumulative probability plots using the cumulative normal distribution function for gold and copper by mineral zone. Black circles in the cumulative probability plots approximate the capping limits that were established for each metal.

Figure 14.1 Sulphurets Zone Gold Assay Cumulative Probability Plot

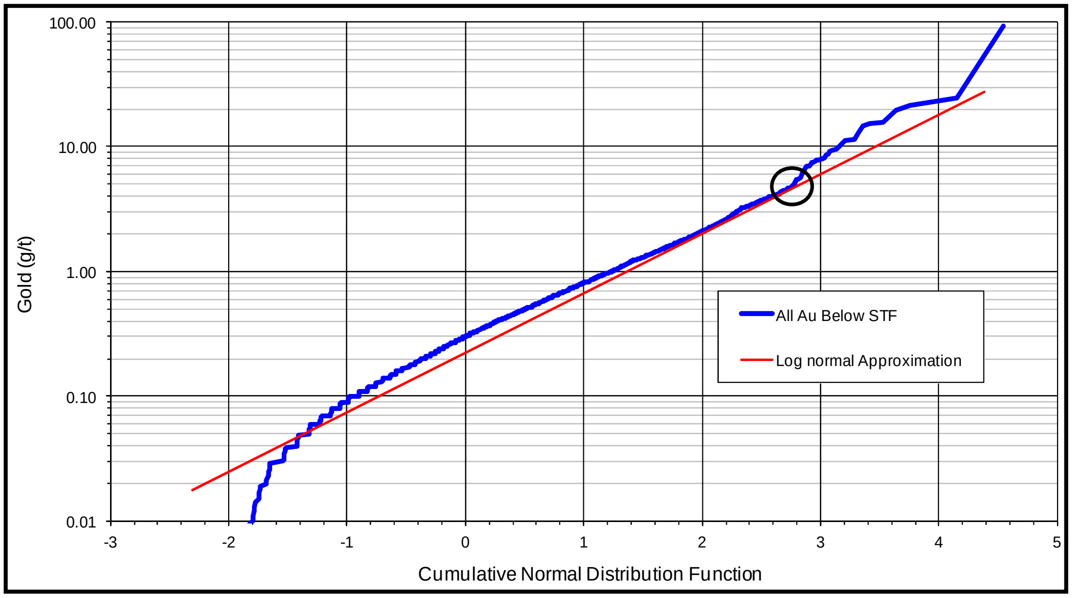


Figure 14.2 Mitchell Zone Gold Assay Cumulative Probability Plot

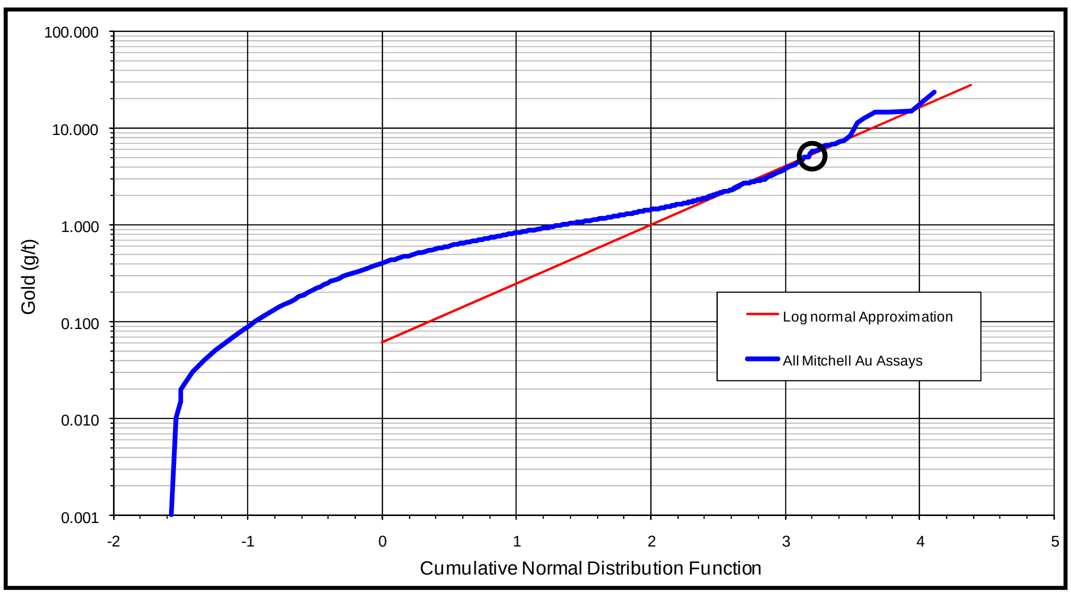


Figure 14.3 Sulphurets Zone Copper Assay Cumulative Probability Plot

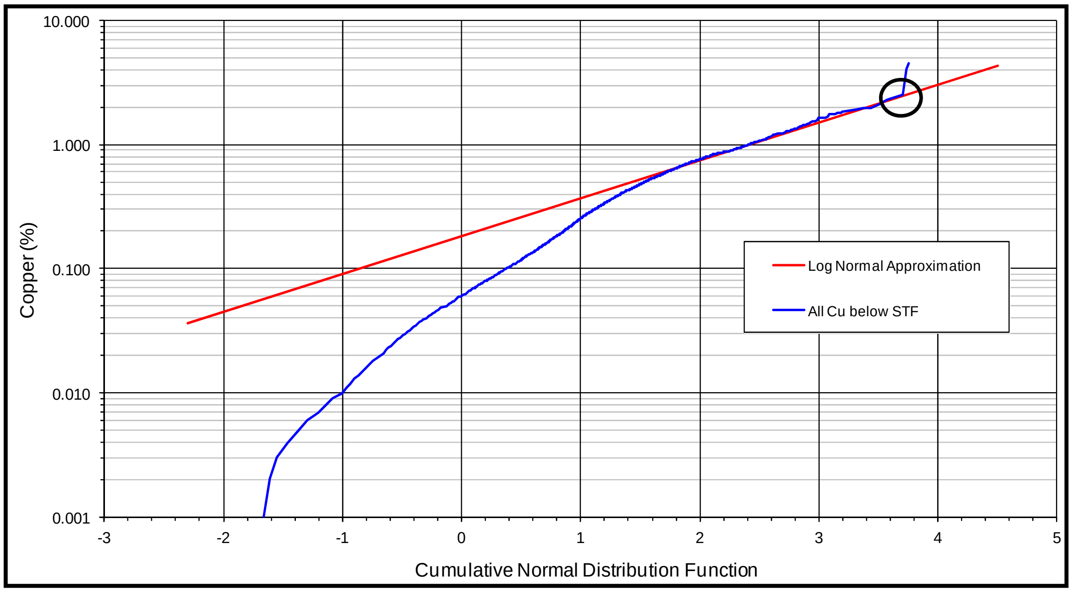
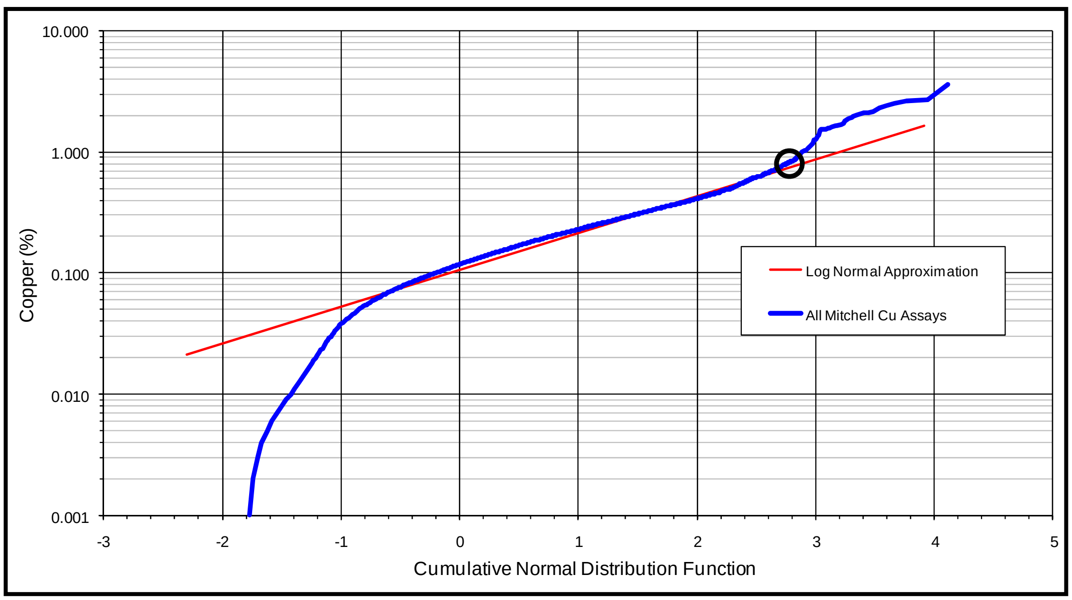


Figure 14.4 Mitchell Zone Copper Assay Cumulative Probability Plot



Based on the information shown in Figure 14.1 through Figure 14.4 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 14.5 through Table 14.7 summarize the capping limits that were established for gold, copper, and silver/molybdenum for the Sulphurets and Mitchell mineral zones.

Table 14.5 Sulphurets and Mitchell Gold Grade Capping Limits

Zone	Attribute	Cap Grade (g/t)
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

Table 14.6 Sulphurets and Mitchell Copper Grade Capping Limits

Zone	Attribute	Cap Grade (%)
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 14.7 Silver and Molybdenum Grade Capping Limits

Zone	Attribute	Ag (g/t)	Mo (ppm)
Sulphurets	Main Cu Hazelton	20	500
	Main Cu Monzonite	20	500
	All Others	30	1,250
Mitchell	All	180	1,200

14.1.4 DRILL HOLE COMPOSITES – SULPHURETS AND MITCHELL

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.

14.1.5 GEOLOGIC CONSTRAINTS - SULPHURETS AND MITCHELL

Various lithologic, alteration, structural domains, and metal grade envelopes were constructed for the Sulphurets and Mitchell 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, hybrid gold and copper envelopes were used to constrain the estimate of block grades for the Sulphurets and Mitchell deposits. Constraints used to estimate gold, silver, copper, and molybdenum are summarized in Table 14.8 for the Sulphurets and Mitchell deposits.

Table 14.8 Constraints Used to Estimate Block Grades – Sulphurets and Mitchell

Mineral Zone	Au	Ag	Cu	Mo
Sulphurets	AUZON	AUZON	CUZON	CUZON
Mitchell	AUZON	AUZON	CUZON	CUZON

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's 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 STF was used to define upper and lower plates. In the Mitchell Zone, the MTF was used to define upper and lower plates. Table 14.9 and Table 14.10 summarize definitions for AUZON and CUZON, respectively.

Table 14.9 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
29	Default Code

Table 14.10 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

14.1.6 VARIOGRAPHY – SULPHURETS AND MITCHELL

RMI generated a number of gold and copper correlograms and variograms using both drill hole assays and 15 m long drill hole composites. Figure 14.5 and Figure 14.6 show gold grade correlograms for the Sulphurets and Mitchell zones, respectively. Figure 14.7 and Figure 14.8 show copper grade correlograms for the Sulphuretsand Mitchell, zones, respectively. Figure 14.9 and Figure 14.10 show 0.5 g/t gold equivalent (AuEQ) correlograms for the Sulphurets and Mitchell zones, respectively.

Figure 14.5 Sulphurets Zone Gold Grade Correlogram

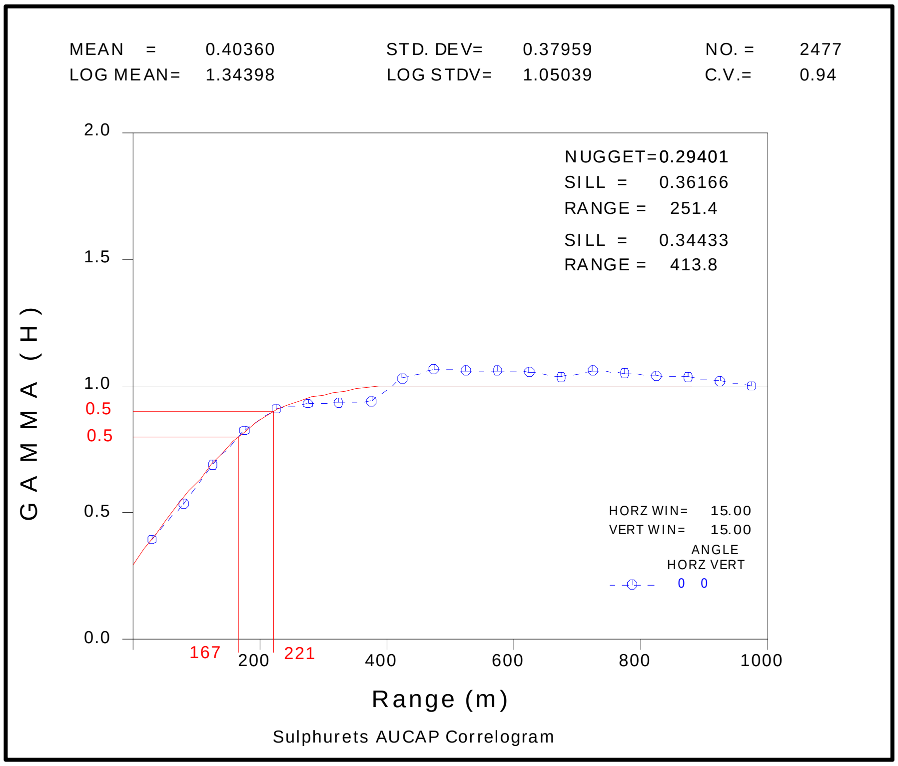


Figure 14.6 Mitchell Zone Gold Grade Correlogram

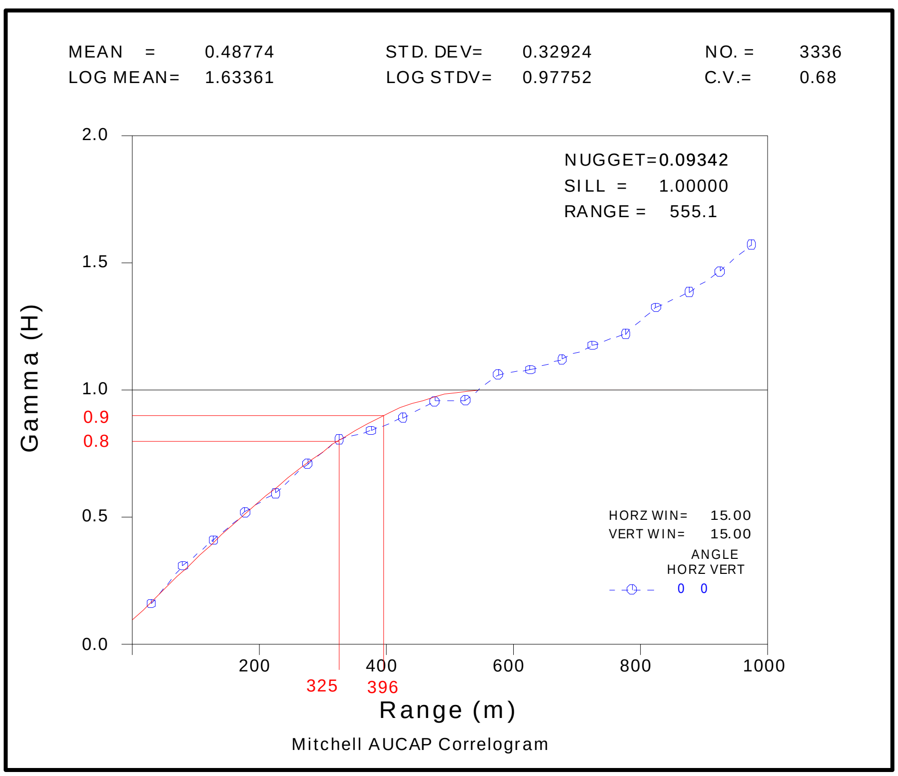


Figure 14.7 Sulphurets Zone Copper Grade Correlogram

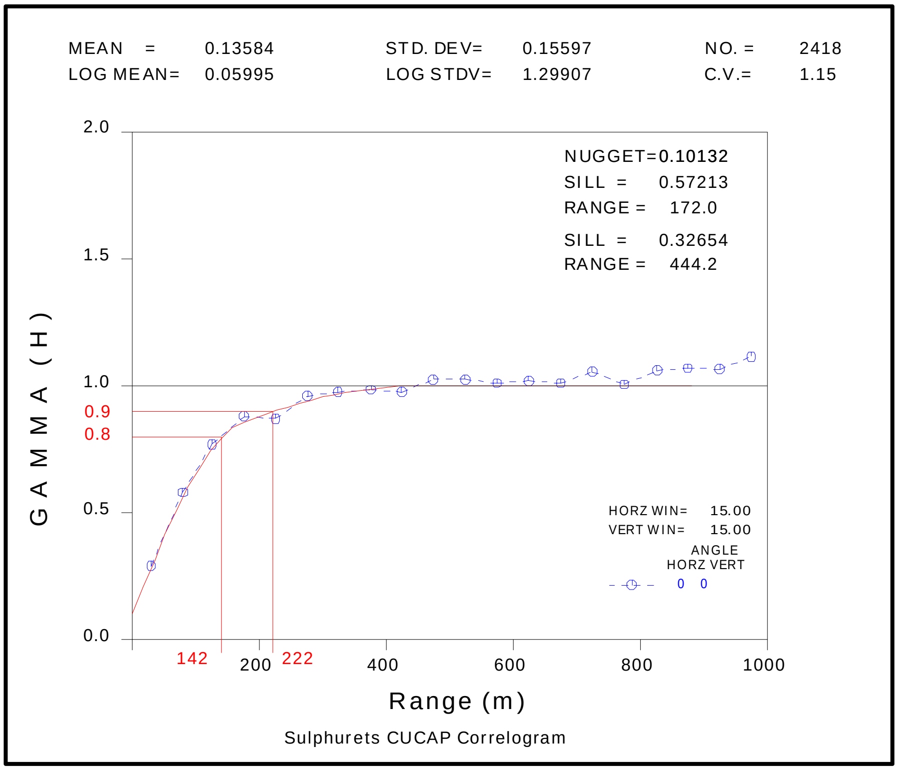


Figure 14.8 Mitchell Zone Copper Grade Correlogram

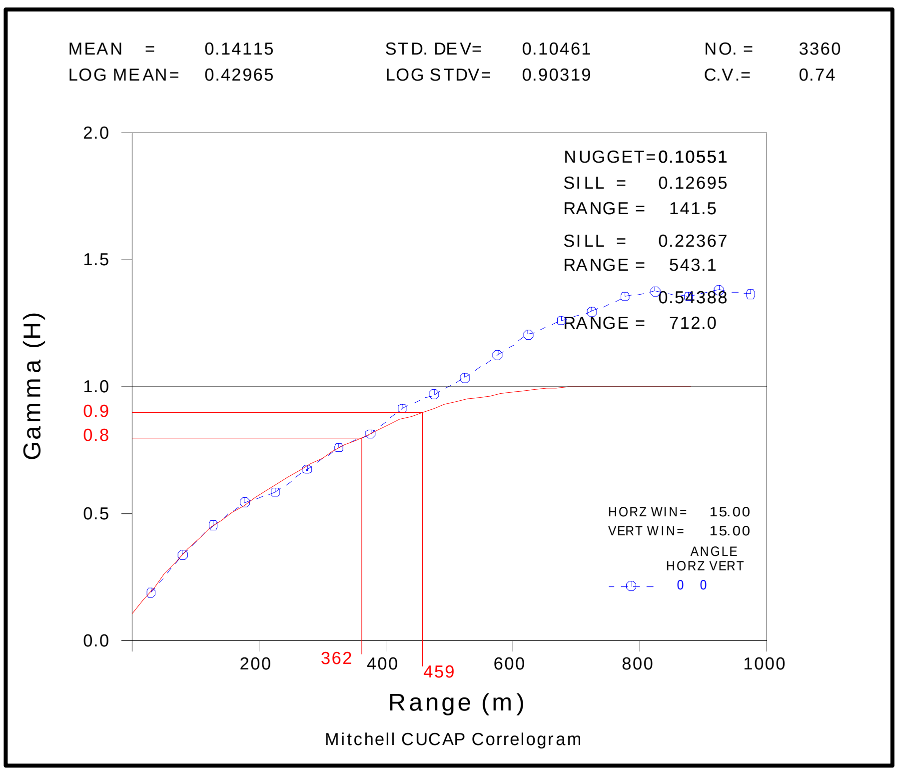


Figure 14.9 Sulphurets Zone 0.5 g/t AuEQ Correlogram

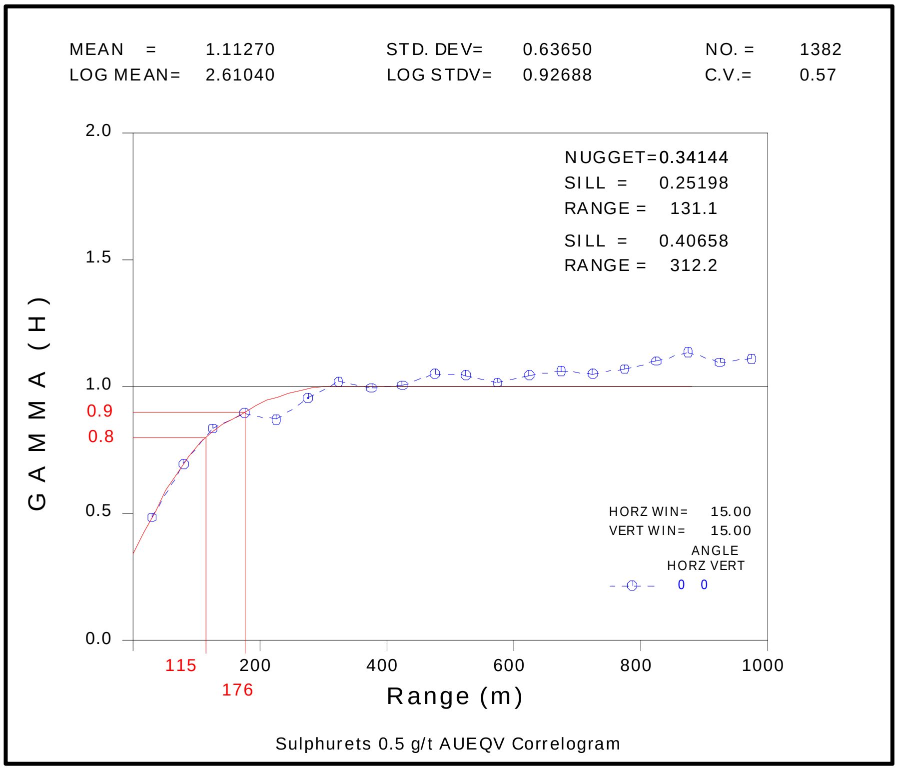
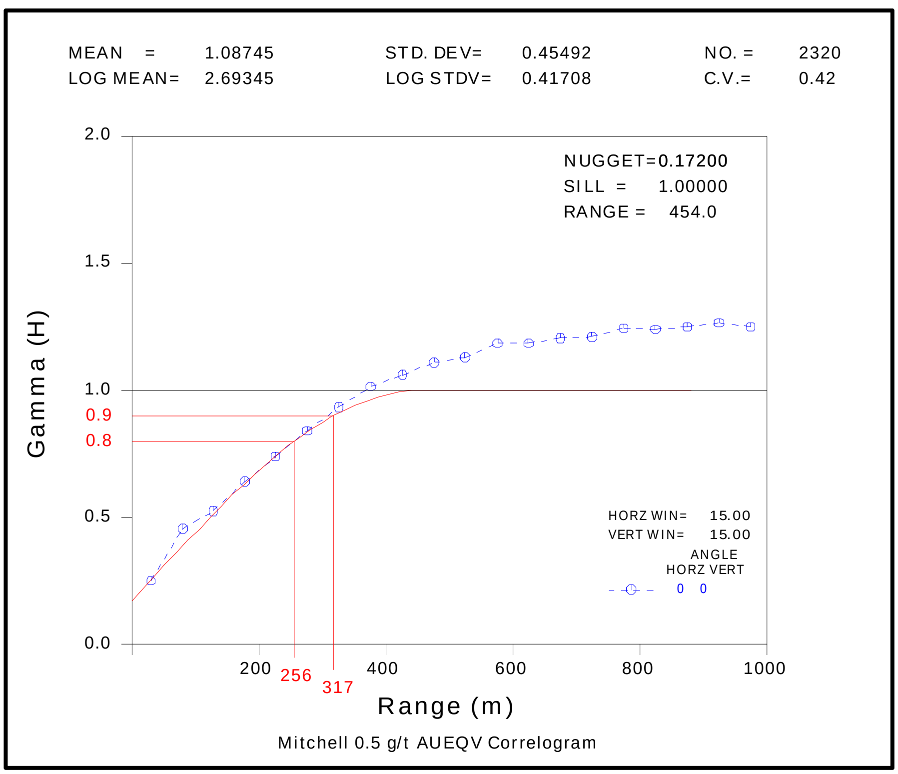


Figure 14.10 Mitchell Zone 0.5 g/t AuEQ Correlogram



The correlograms shown in Figure 14.5 through Figure 14.10 were modelled as either single structure spherical or nested spherical models. Total ranges for gold for each zone are as follows:

- Sulphurets: 414 m
- Mitchell: 555 m.

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

- Sulphurets: 167 m
- Mitchell: 325 m.

Total ranges for copper for each zone are as follows:

- Sulphurets: 444 m
- Mitchell: 712 m.

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

- Sulphurets: 142 m
- Mitchell: 362 m.

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

- Sulphurets: 312 m
- Mitchell: 454 m.

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

- Sulphurets: 115 m
- Mitchell: 256 m.

14.1.7 Grade Estimation Parameters – Sulphurets-Mitchell

RMI constructed a 3DBM using MineSight®, a widely recognized commercial mine engineering software package. Table 14.11 summarizes various block parameters for this non-rotated model which uses NAD83 UTM coordinates.

Table 14.11 KSM Block Model Dimensions

Parameter	NAD83 Coordinates		Block Size (m)	No. of Blocks	Areal Extent (m)
	Minimum	Maximum			
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 (IDW), and nearest neighbour (NN). RMI notes that the MineSight® block model described in Table 14.11 contains block grades for all four KSM deposits but the Kerr and Iron Cap grades in that model are obsolete. Block modeling methods for those two areas are discussed in Sections 14.2.5 and 14.3.5. Gold and copper Mineral 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 IDW estimation plans used strict block/composite matching.

Table 14.12 summarizes the parameters used to estimate block gold and silver grades for the Sulphurets Zone. Once a block was estimated, it was flagged so it would not be re-estimated in subsequent runs. The estimate of Sulphurets gold and silver block grades was constrained (controlled) by matching block and drill hole AUZON composite codes (Table 14.9) shows the definition of AUZON codes). The last two interpolation runs shown in Table 14.12 estimated block grades above the STF, while all of the prior runs estimated blocks below the STF. 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 14.12 Sulphurets Zone Gold 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 east-west axis
LRL = "Left-hand-right hand-left hand" rotation rule

Table 14.13 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 MTF was also used to limit or constrain the estimate of block grades. The field "FLTAR" (fault block) shown in Table 14.13 shows two codes where five means above the MTF and six means below the MTF. Similar to the

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 14.13 Mitchell Gold/Silver 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 14.14 summarizes the key estimation parameters that were used to estimate block copper and molybdenum grades using IDW 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 14.14. FLTAR codes 1 and 2 refer to blocks/drill holes below and above the STF, 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 14.14 Sulphurets Copper/Molybdenum 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

Table 14.15 summarizes the key estimation parameters that were used to estimate block copper grades using IDW 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 MTF, 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 14.16 summarizes the key estimation parameters that were used to estimate block molybdenum grades using IDW 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 14.15 Mitchell Copper 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 14.16 Mitchell Molybdenum Grade Estimation Parameters

Estimation Pass	IDW 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

14.1.8 GRADE MODEL VERIFICATION - SULPHURETS AND MITCHELL

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 14.11 and Figure 14.12 are northwest-southeast cross sections through the Sulphurets block model drawn at Cross Section 23. For reference, Cross Section 23 is shown in Figure 10.3, a drill hole plan map for the Sulphurets deposit. These figures show estimated block/composite gold grades (Figure 14.11) and block/composite copper grades (Figure 14.12). Figure 14.13 and Figure 14.14 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 14.15 and Figure 14.16 are northeast-southwest cross sections through the Mitchell block model drawn at Cross Section 11. For reference, Cross Section 11 is shown in Figure 10.4, a drill hole plan map for the Mitchell deposit. These figures show estimated block/composite gold grades (Figure 14.15) and block/composite copper grades (Figure 14.16, Figure 14.17, and Figure 14.18) are block model level maps drawn at the 660 m elevation through the Mitchell model showing estimated block/composite gold and copper grades, respectively.

The dashed black line shown on the block model cross sections and level plans shown in Figure 14.11 through Figure 14.18 represent a conceptual pit generated by RMI using gold and copper prices of US$1,300/oz and US$3.00/lb, respectively.

Figure 14.11 Sulphurets Zone Gold Block Model Cross Section 23

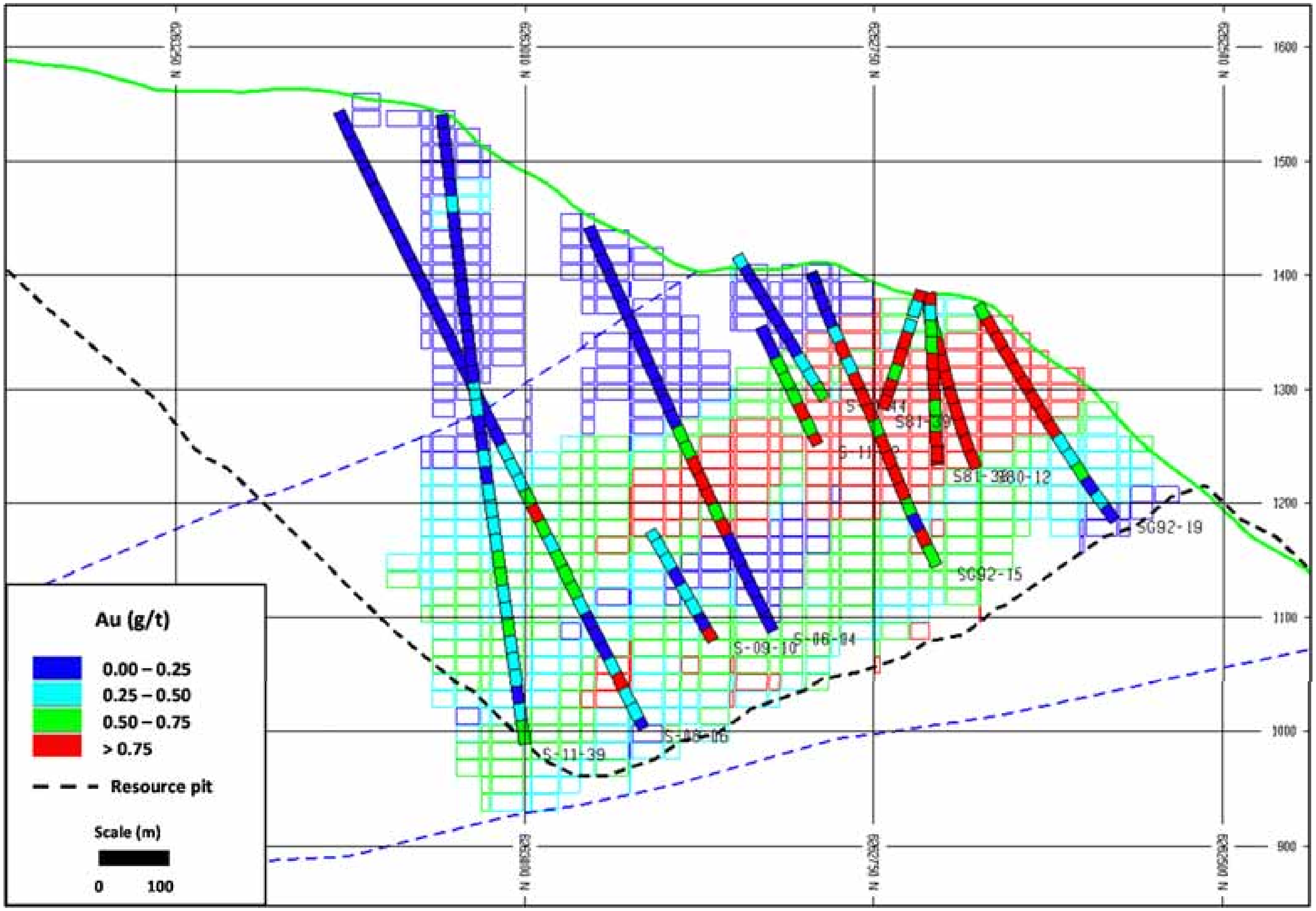


Figure 14.12 Sulphurets Zone Copper Block Model Cross Section 23

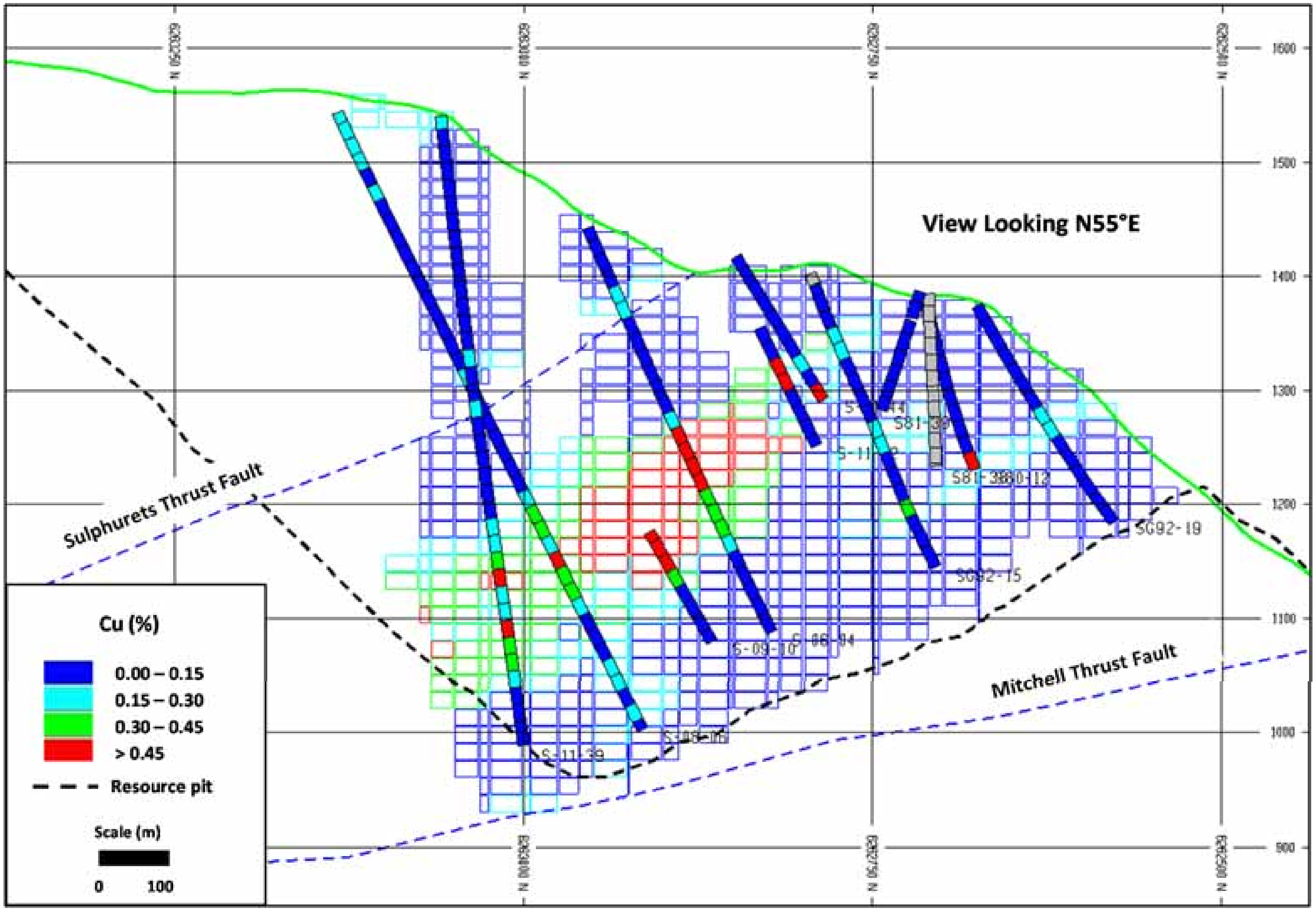


Figure 14.13 Sulphurets Zone Gold Block Model – 1,275 m Level

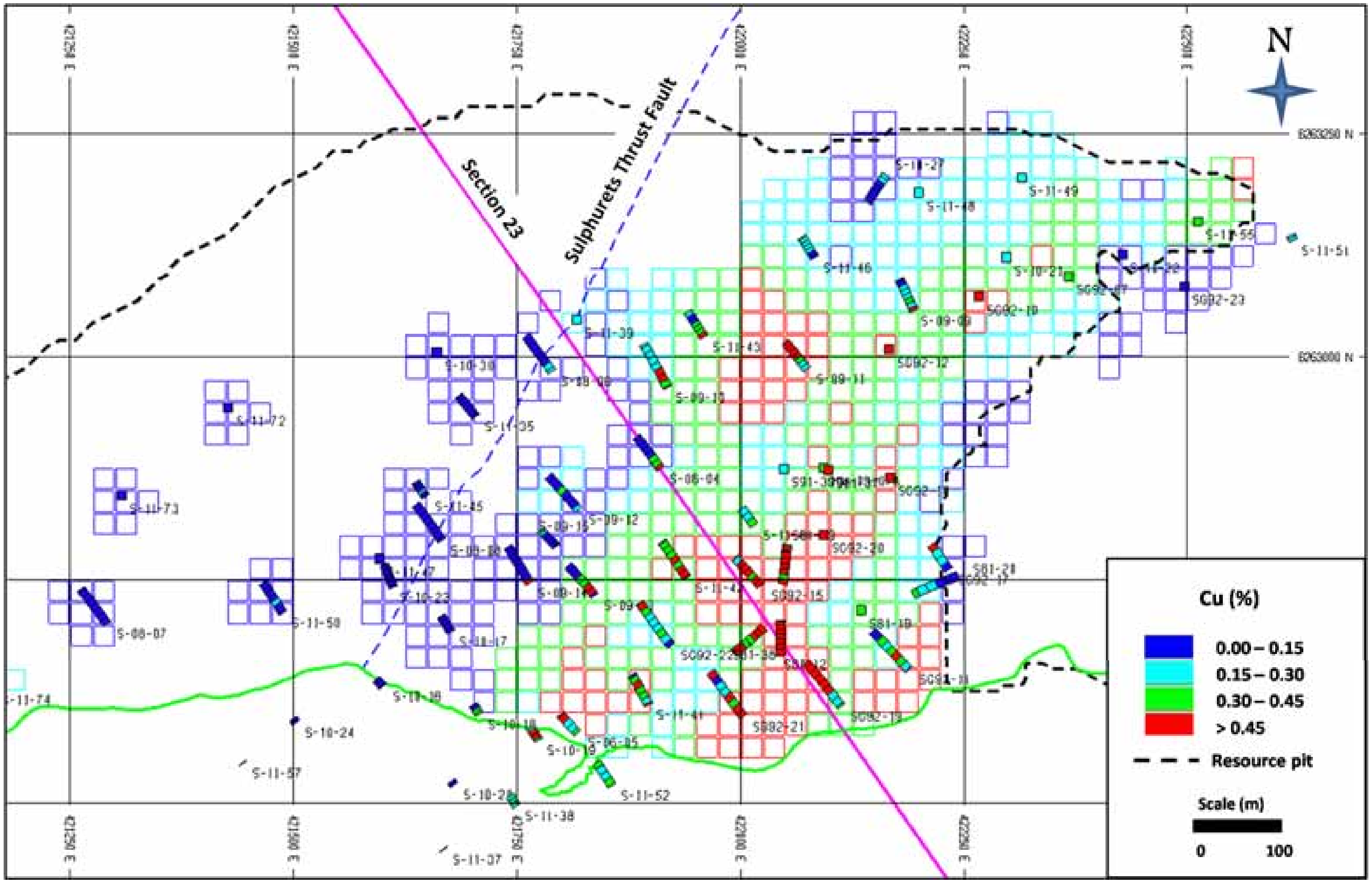


Figure 14.14 Sulphurets Zone Copper Block Model – 1,275 m Level

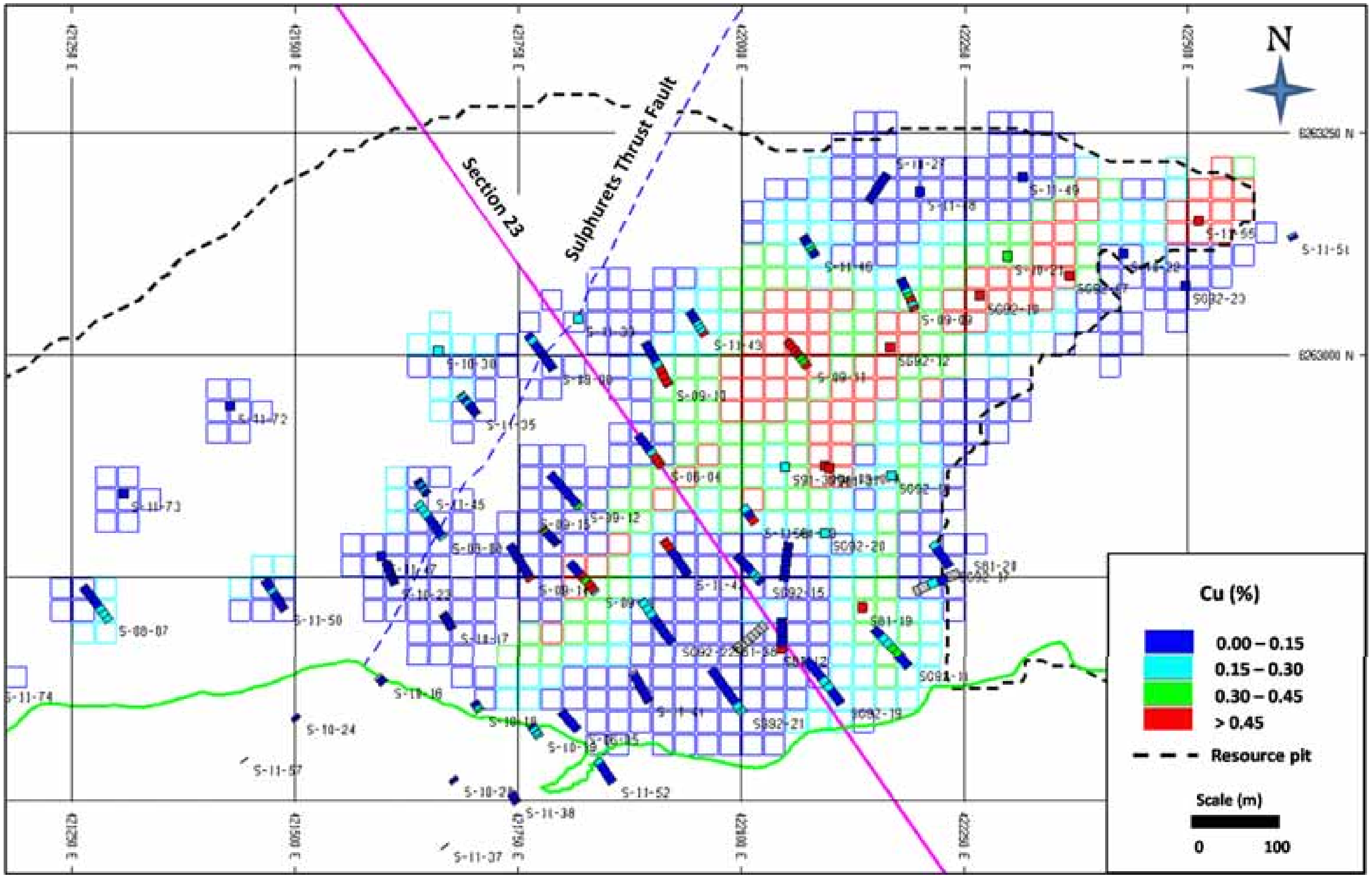


Figure 14.15 Mitchell Zone Gold Block Model Cross Section 11

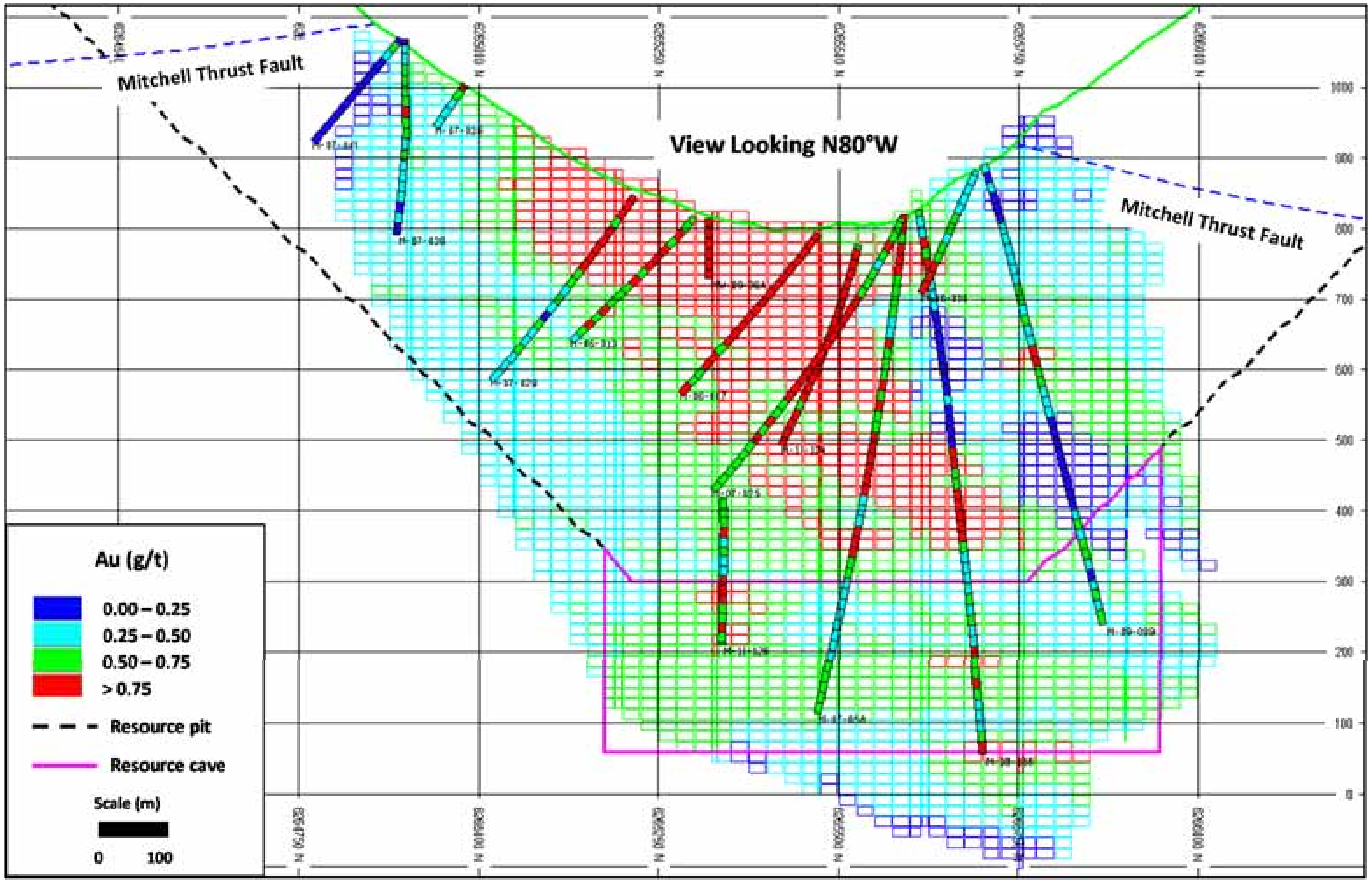


Figure 14.16 Mitchell Zone Copper Block Model Cross Section 11

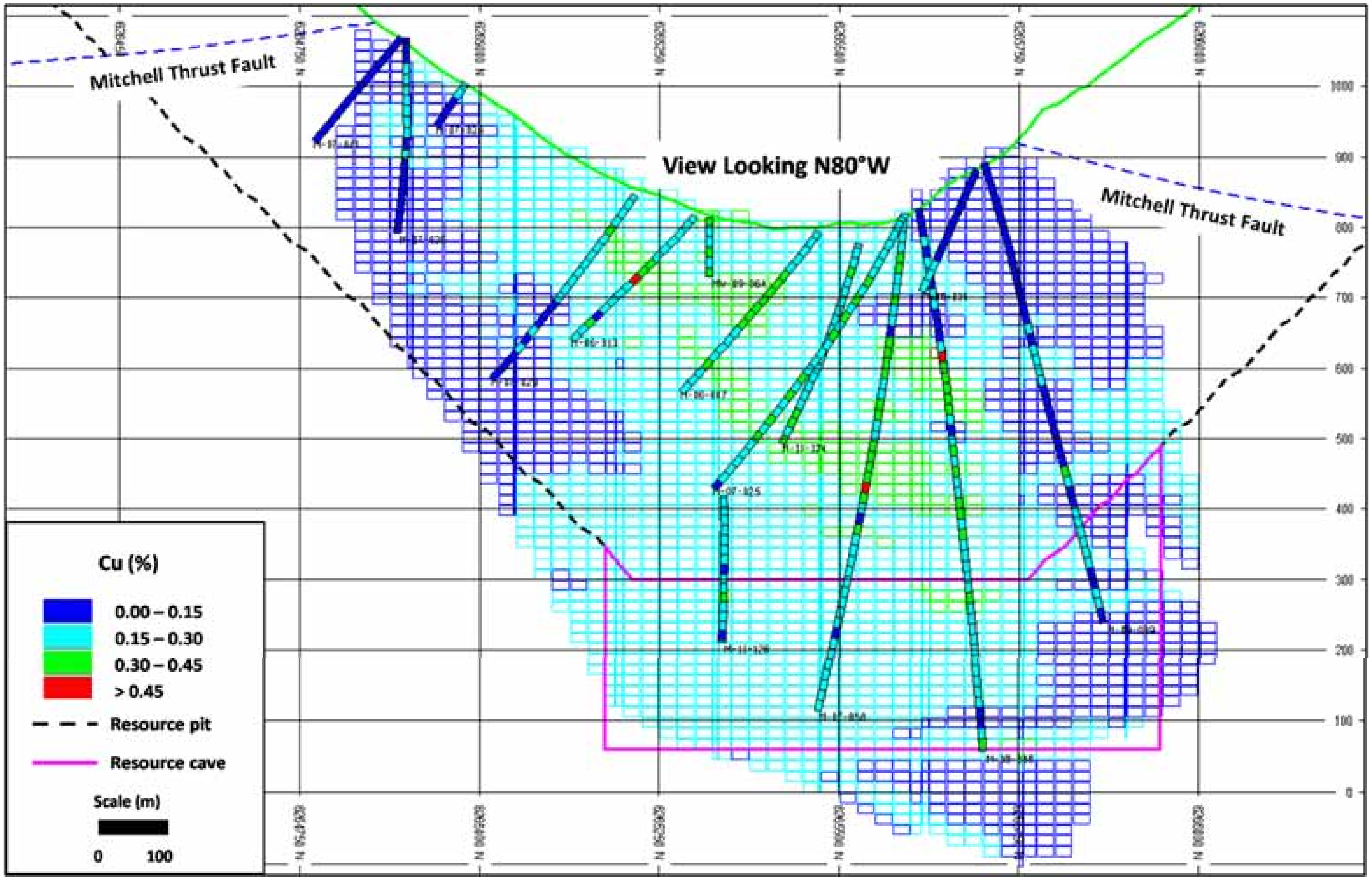


Figure 14.17 Mitchell Zone Gold Block Model – 660 m Level

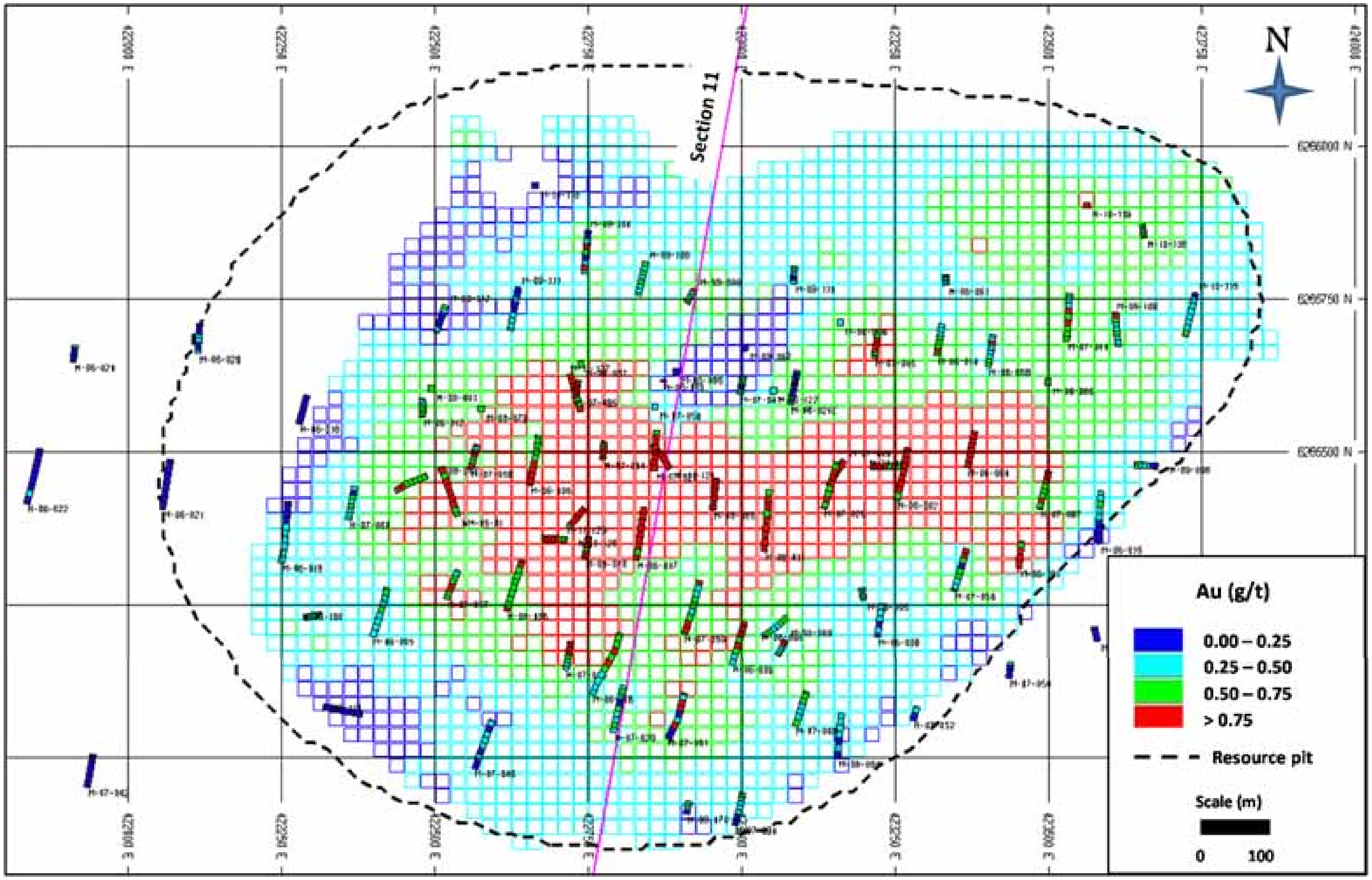
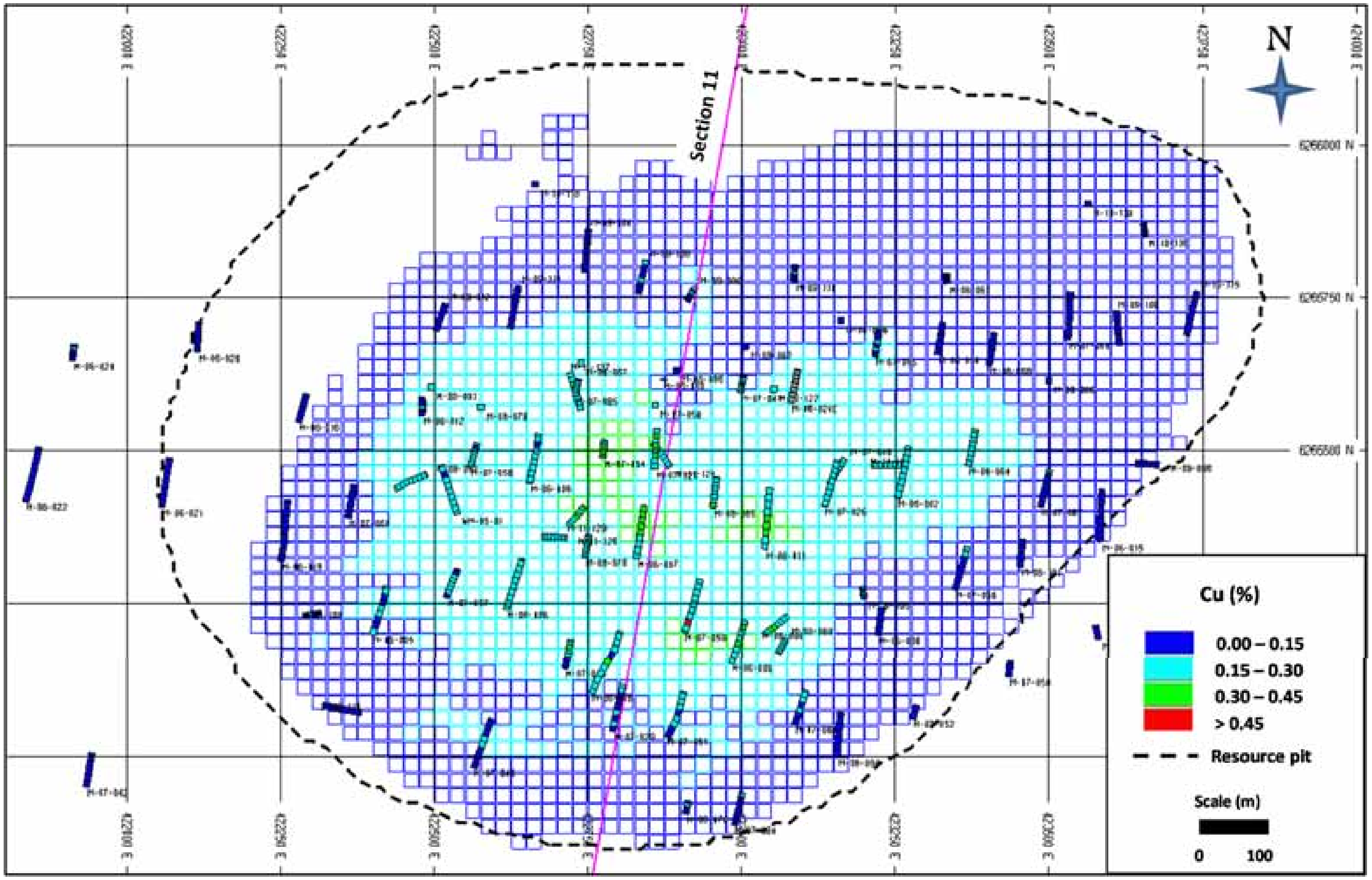


Figure 14.18 Mitchell Zone Copper Block Model – 660 m Level



RMI generated NN models for gold, copper, silver, and molybdenum in order to check for potential global biases in the estimated block grades. Table 14.17 compares mean NN and IDW grades at a zero cut-off grade for the Sulphurets and Mitchell zones by Mineral Resource category.

The results show that the IDW models compare very well with the NN 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.

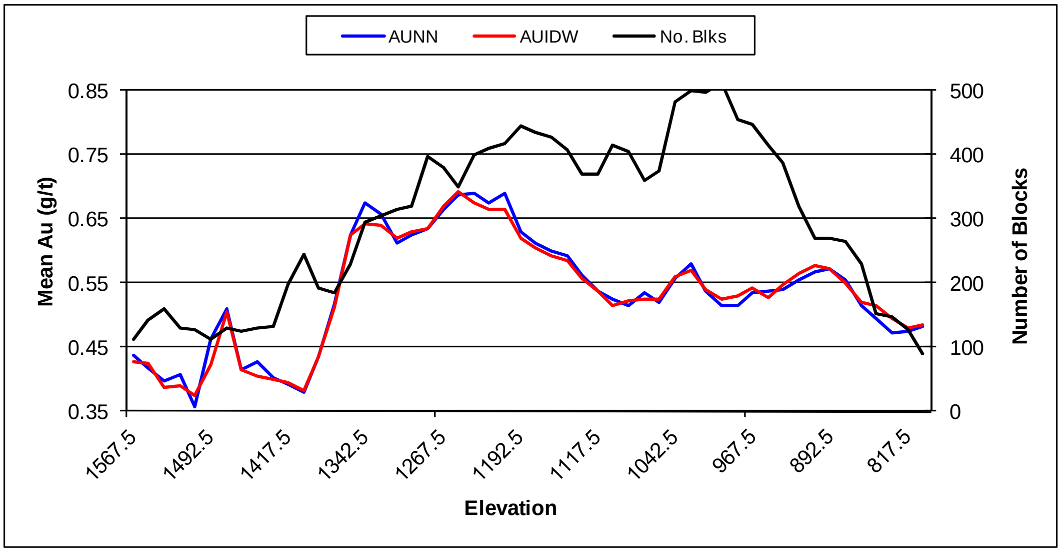
Possible local biases in the estimate of block grades were examined by preparing a set of "swath plots" for gold and copper. These plots compare mean estimated IDW gold and copper grades (AUIDW and CUIDW) with NN gold and copper (AUNN and CUNN) estimates by block model columns (eastings), rows (northings), and levels (elevation). Gold and copper swath plots by elevation are shown in Figure 14.19 and Figure 14.20 for the Sulphurets and Mitchell 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 14.19 and Figure 14.20 show a close comparison between the IDW and NN estimates. There do not appear to be any severe local biases in the estimate of gold and copper. Based on visual and statistical checks, it is the opinion of RMI that the Sulphurets and Mitchell grade models are globally unbiased and represent reasonable estimates of in situ block grades.

Table 14.17 Grade Model Bias Checks

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%

Figure 14.19 Sulphurets Zone Gold-Copper Swath Plots by Elevation



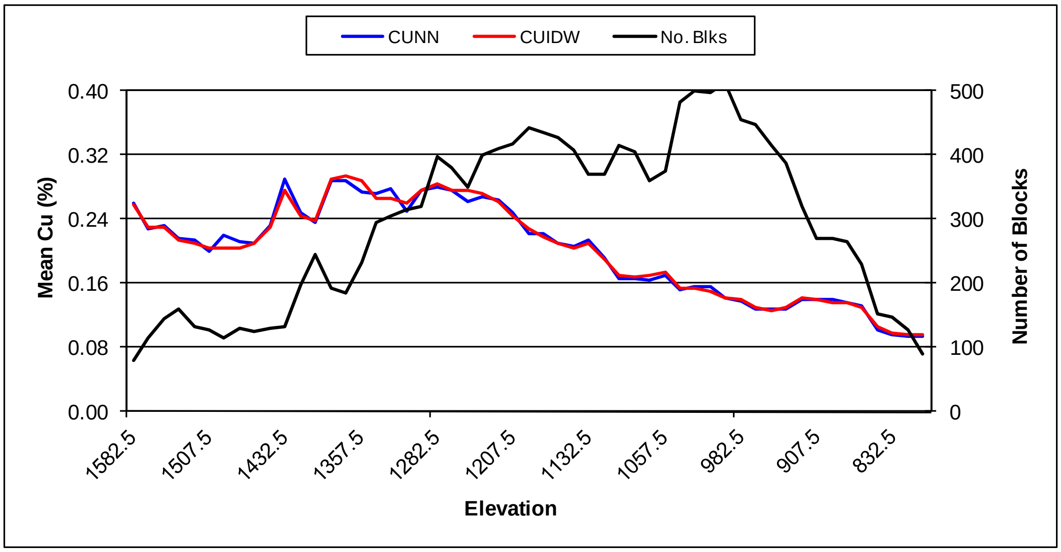
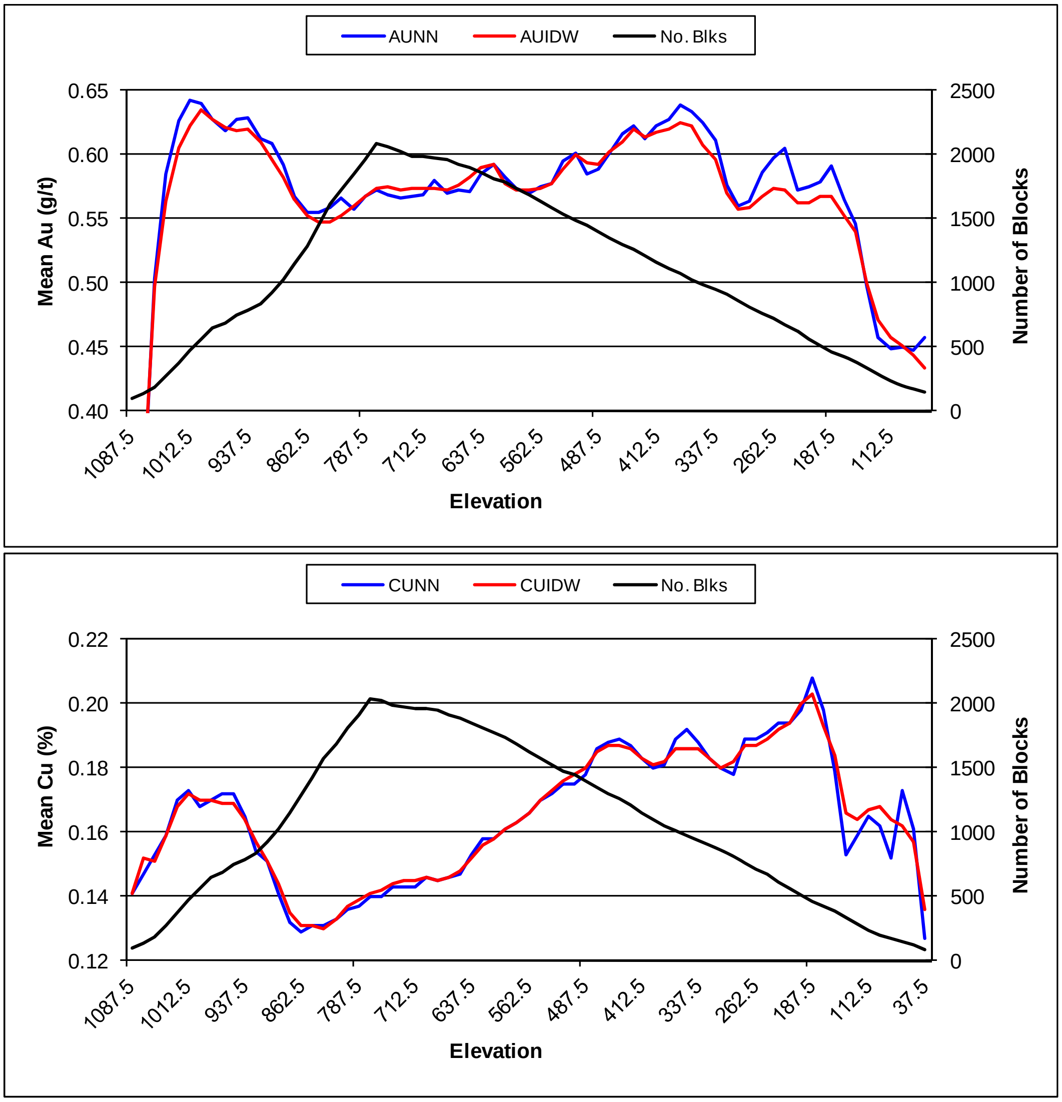


Figure 14.20 Mitchell Zone Gold-Copper Swath Plots by Elevation



14.1.9 RESOURCE CLASSIFICATION – SULPHURETS AND MITCHELL

RMI classified Sulphurets and Mitchell 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 the Sulphurets and Mitchell 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 14.18 summarizes the criteria that were used to establish Indicated Resources.

Table 14.18 Indicated Resource Criteria

Mineralized Zone	Block Location	Minimum No. Holes	Distance to Closest Composite (m)
Sulphurets	Inside mineralized continuity shape	≥2	≤75
Mitchell	Inside mineralized continuity shape and below MTF	≥2	≤125

Measured Mineral Resources (code = 1) were only assigned to the Mitchell Zone if:

- the blocks were located inside of the mineralized continuity shape
- 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 14.19 summarizes the criteria used to establish Inferred Resources.

Table 14.19 Inferred Resource Criteria

Mineralized Zone	Block Location	Minimum No. Holes	Distance to Closest Composite (m)
Sulphurets	Above STF	≥2	≤37.5
	Above STF	≥1	≤25
	Below STF, inside mineralized continuity shape	≥1	≤50
	Below STF, outside mineralized continuity shape	≥2	≤50
	Below STF, outside mineralized continuity shape	≥1	≤25
Mitchell	Above MTF, inside mineralized continuity shape	≥1	≤75
	Above MTF, outside mineralized continuity shape	≥1	≤50
	Below MTF, inside mineralized continuity shape	≥2	≤175
	Below MTF, outside mineralized continuity shape	≥2	≤75
	Below MTF, outside mineralized continuity shape	≥1	≤50

14.2 KERR DEPOSIT

Since the 2012 PFS (Tetra Tech 2012) was published, Seabridge has focused their exploration efforts at trying to locate potentially higher grade copper and gold mineralization located below the recognized near surface mineralization at the Kerr deposit. Drill campaigns in 2012 through 2015 have progressively provided more insight into the geometry and extent of mineralization at Kerr.

With substantially deeper drill hole data and an updated geologic interpretation, a new block model was constructed for the Kerr deposit. The remainder of Section 14.2 deals with the new Kerr model which replaces the 2012 PFS Kerr Mineral Resource.

14.2.1 METAL DISTRIBUTION – KERR

The distribution of gold grades based on raw uncomposited data is summarized at four different cut-off grades by the gold grade envelopes that were used in the block grade estimation process in Table 14.20. The data in Table 14.20 shows statistics for uncapped (left portion of table) and statistics for capped assays (right portion of table). Grade capping of high-grade outliers is discussed in Section 14.2.2. Copper assay statistics for both uncapped and capped data are summarized in Table 14.21 where the grades are broken out by copper grade envelopes that were used in the estimate of block grades.

Table 14.20 Kerr Gold Assay Statistics

	Uncapped Au Statistics Above Cut-off								Capped Au Statistics Above Cut-off				
AUZON	Au Cut-off (g/t)	Total Meters (m)	Inc. (%)	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. (%)	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. (%)	Std. Dev.	CV
All Data	0.00	78,136	40	0.22	17,474	6.8	0.69	3.08	0.21	16,706	7.1	0.41	1.94
	0.10	47,263	27	0.34	16,283	17.0	0.86	2.50	0.33	15,515	17.8	0.50	1.52
	0.20	26,440	22	0.50	13,307	27.0	1.13	2.24	0.47	12,539	28.2	0.63	1.33
	0.40	9,402	12	0.91	8,588	49.1	1.82	1.99	0.83	7,820	46.8	0.96	1.15
<0.10 g/t	0.00	14,897	88	0.06	880	36.5	0.25	4.15	0.05	783	41.0	0.11	2.09
	0.10	1,758	7	0.32	559	15.4	0.66	2.06	0.26	462	17.4	0.22	0.83
	0.20	762	3	0.56	423	15.2	0.94	1.70	0.43	326	17.1	0.25	0.57
	0.40	278	2	1.04	289	32.9	1.44	1.38	0.69	192	24.6	0.22	0.32
Diorite	0.00	981	40	0.25	246	6.7	0.73	2.89	0.19	187	8.8	0.22	1.15
	0.10	589	31	0.39	230	17.3	0.91	2.33	0.29	171	22.8	0.24	0.81
	0.20	287	17	0.65	187	18.2	1.25	1.92	0.45	128	24.0	0.26	0.57
	0.40	122	12	1.17	142	57.8	1.80	1.54	0.68	83	44.4	0.23	0.33
PFMP Dyke	0.00	791	60	0.14	109	8.5	0.24	1.75	0.13	101	9.1	0.18	1.43
	0.10	318	16	0.31	99	17.3	0.30	0.97	0.29	92	18.6	0.20	0.68
	0.20	191	17	0.42	81	34.4	0.35	0.83	0.38	73	36.9	0.21	0.54
	0.40	56	7	0.77	43	39.7	0.49	0.63	0.64	36	35.4	0.21	0.33
0.10 g/t Shell	0.00	17,280	51	0.13	2,241	19.0	0.50	3.86	0.12	2,006	21.2	0.12	1.03
	0.10	8,445	37	0.21	1,815	39.1	0.71	3.28	0.19	1,579	43.7	0.14	0.73
	0.20	2,004	9	0.47	940	18.2	1.42	3.02	0.35	704	20.3	0.20	0.58
	0.40	448	3	1.19	532	23.8	2.88	2.42	0.66	297	14.8	0.22	0.33
0.20 g/t Shell	0.00	33,304	22	0.23	7,525	5.0	0.60	2.63	0.22	7,403	5.1	0.33	1.49
	0.10	26,135	36	0.27	7,149	23.2	0.66	2.43	0.27	7,026	23.6	0.36	1.34
	0.20	14,251	34	0.38	5,404	41.2	0.88	2.33	0.37	5,282	41.9	0.46	1.25
	0.40	2,828	8	0.81	2,302	30.6	1.92	2.36	0.77	2,180	29.4	0.93	1.21
0.40 g/t Shell	0.00	10,834	8	0.58	6,243	0.7	0.96	1.67	0.57	6,125	0.7	0.78	1.38
	0.10	9,979	10	0.62	6,202	2.5	0.99	1.59	0.61	6,084	2.6	0.80	1.31
	0.20	8,913	30	0.68	6,044	15.9	1.03	1.52	0.66	5,926	16.2	0.83	1.25
	0.40	5,644	52	0.90	5,053	80.9	1.25	1.39	0.87	4,935	80.6	0.98	1.12
4.00 g/t Shell	0.00	49	19	4.67	229	0.2	11.48	2.46	2.06	101	0.4	3.33	1.61
	0.10	40	15	5.75	229	0.5	12.50	2.18	2.53	101	1.1	3.53	1.40
	0.20	33	12	7.01	228	0.8	13.52	1.93	3.07	100	1.9	3.70	1.21
	0.40	27	54	8.45	226	98.5	14.51	1.72	3.66	98	96.6	3.84	1.05

Table 14.21 Kerr Copper Assay Statistics

	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
CUZON	Cu Cut-off (%)	Total Meters (m)	Inc. (%)	Mean Cu (%)	Grd-Thk (%-m)	Inc. (%)	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. (%)	Std. Dev.	CV
All Data	0.00	78,136	64	0.26	20,591	16.4	0.36	1.38	0.26	20,523	16.5	0.36	1.36
	0.25	27,743	17	0.62	17,208	23.8	0.41	0.66	0.62	17,140	23.9	0.39	0.64
	0.50	14,138	14	0.87	12,311	36.5	0.44	0.51	0.87	12,243	36.6	0.42	0.48
	1.00	3,282	4	1.46	4,793	23.3	0.57	0.39	1.44	4,725	23.0	0.51	0.36
<0.10%	0.00	23,224	99	0.03	621	83.8	0.10	3.61	0.03	602	86.5	0.06	2.16
	0.25	146	0	0.69	101	4.9	0.96	1.39	0.56	81	5.1	0.31	0.56
	0.50	58	0	1.20	70	4.2	1.36	1.13	0.87	51	4.3	0.27	0.31
	1.00	21	0	2.14	44	7.1	1.97	0.92	1.20	25	4.1	0.08	0.06
Diorite	0.00	981	94	0.08	75	68.4	0.11	1.39	0.08	75	68.4	0.11	1.39
	0.25	61	5	0.39	24	19.2	0.15	0.38	0.39	24	19.2	0.15	0.38
	0.50	15	2	0.61	9	12.4	0.09	0.14	0.61	9	12.4	0.09	0.14
	1.00	0	0	0.00	0	0.0	0.00	0.00	0.00	0	0.0	0.00	0.00
PFMP Dyke	0.00	791	75	0.17	136	15.3	0.29	1.70	0.17	131	15.8	0.26	1.57
	0.25	197	11	0.58	115	24.4	0.33	0.57	0.56	111	25.2	0.24	0.42
	0.50	111	12	0.74	82	43.5	0.37	0.49	0.70	78	45.0	0.22	0.32
	1.00	15	2	1.49	23	16.9	0.49	0.33	1.19	18	14.0	0.06	0.05
0.10% Shell	0.00	17,740	90	0.13	2,285	66.4	0.14	1.08	0.13	2,277	66.7	0.13	1.05
	0.25	1,774	8	0.43	767	19.8	0.23	0.54	0.43	759	19.8	0.21	0.49
	0.50	414	2	0.76	315	10.5	0.28	0.36	0.74	307	10.6	0.21	0.28
	1.00	58	0	1.30	75	3.3	0.33	0.25	1.16	67	2.9	0.10	0.08
0.25% Shell	0.00	15,529	46	0.28	4,302	22.3	0.19	0.69	0.28	4,278	22.4	0.18	0.65
	0.25	8,401	46	0.40	3,345	57.9	0.17	0.44	0.40	3,321	58.2	0.15	0.37
	0.50	1,245	7	0.69	856	16.8	0.28	0.41	0.67	832	16.9	0.18	0.27
	1.00	94	1	1.43	135	3.1	0.54	0.38	1.18	111	2.6	0.09	0.07
0.50% Shell	0.00	10,066	19	0.47	4,730	5.0	0.28	0.60	0.47	4,730	5.0	0.28	0.60
	0.25	8,195	37	0.55	4,492	30.4	0.25	0.45	0.55	4,492	30.4	0.25	0.45
	0.50	4,450	41	0.69	3,056	55.6	0.26	0.38	0.69	3,056	55.6	0.26	0.38
	1.00	297	3	1.43	425	9.0	0.50	0.35	1.43	424	9.0	0.49	0.35
0.75% Shell	0.00	5,481	11	0.67	3,697	1.5	0.32	0.48	0.67	3,697	1.5	0.32	0.48
	0.25	4,892	15	0.74	3,641	8.6	0.27	0.36	0.74	3,641	8.6	0.27	0.36
	0.50	4,073	64	0.82	3,322	71.0	0.23	0.28	0.82	3,322	71.0	0.23	0.28
	1.00	558	10	1.25	696	18.8	0.27	0.21	1.25	696	18.8	0.27	0.21
>1.00% Shell	0.00	4,324	6	1.10	4,745	0.5	0.64	0.58	1.09	4,733	0.5	0.62	0.57
	0.25	4,077	7	1.16	4,723	2.6	0.61	0.53	1.16	4,710	2.6	0.59	0.51
	0.50	3,771	35	1.22	4,601	25.4	0.59	0.48	1.22	4,588	25.5	0.57	0.46
	1.00	2,238	52	1.52	3,396	71.6	0.60	0.39	1.51	3,383	71.5	0.56	0.37

In general, silver grades tend to be higher in the more intensely mineralized zones following gold and copper. The average silver grade for the Kerr deposit is approximately 1.6 g/t with only one percent of the data above a 15 g/t cut-off.

Molybdenum grades at Kerr tend to be relatively low averaging approximately 25 ppm although most of the pre-2009 drill hole samples were not assayed for molybdenum.

14.2.2 HIGH-GRADE OUTLIERS – KERR

Cumulative probability plots were used to identify high-grade outliers for gold, copper, silver, and molybdenum using the original assay samples. The assays were examined with respect to logged lithology, alteration, and modeled grade envelopes. The assays were combined into low and high-grade domains for determining grade capping limits. Figure 14.21 is a cumulative probability plot that shows the distribution of raw gold assays representing low-grade gold domains (less than 0.2 g/t) including diorite and PFMP samples. Figure 14.22 is a cumulative probability plot that shows the distribution of gold grades within combined higher grade domains (greater than or equal to 0.2 g/t). Capping limits for the two domains are highlighted by the circle where the distribution of values deviates from an approximated log normal line.

Figure 14.21 Kerr Gold Cumulative Probability Plot – Low-grade Domains

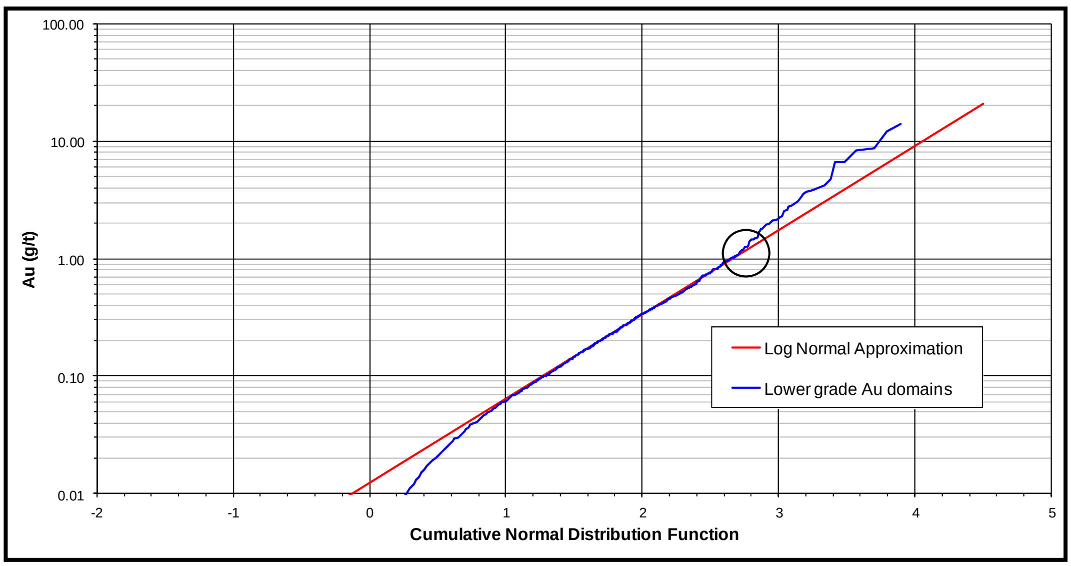


Figure 14.22 Kerr Gold Cumulative Probability Plot – High-grade Domains

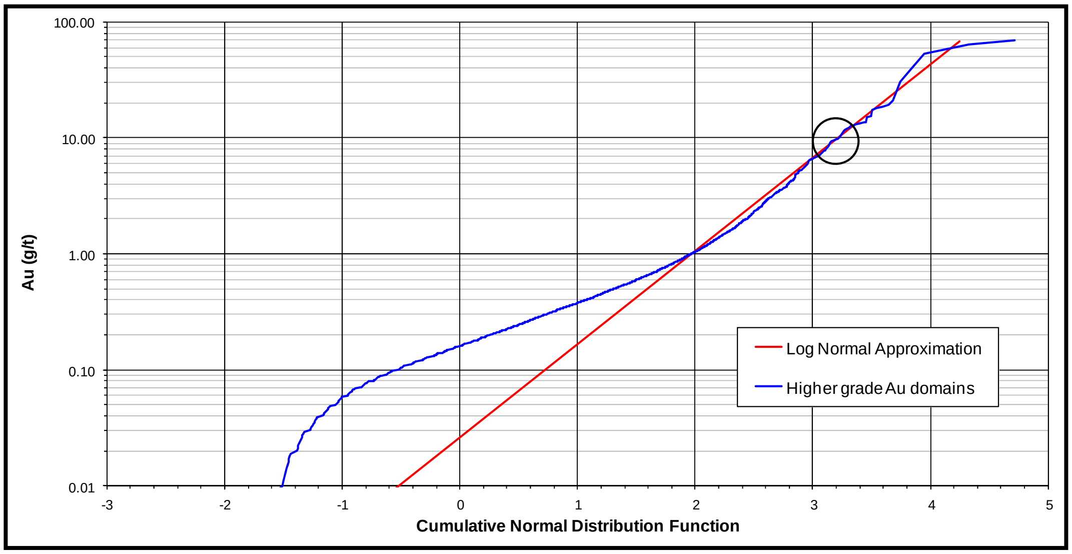


Figure 14.23 is a cumulative probability plot that shows the distribution of raw copper assays representing low-grade copper domains (less than 0.25%) including diorite and PFMP samples. Figure 14.24 is a cumulative probability plot that shows the distribution of copper assay grades within combined higher grade domains (greater than or equal to 0.25%).

Figure 14.23 Kerr Copper Cumulative Probability Plot – Low-grade Domains

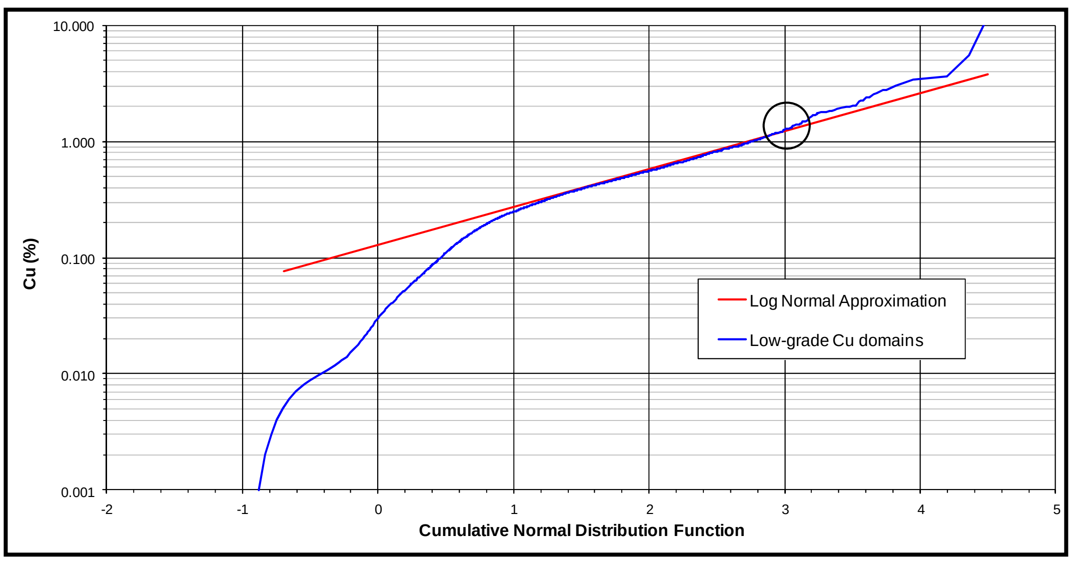
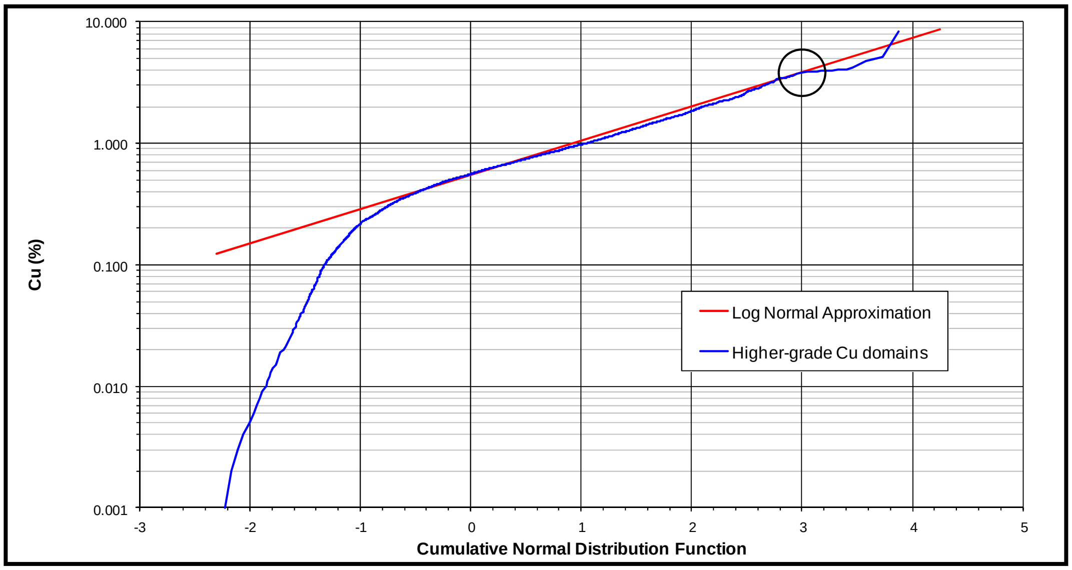


Figure 14.24 Kerr Copper Cumulative Probability Plot – High-grade Domains



Similar plots were constructed for silver and molybdenum. Table 14.22 summarizes the capping limits that were applied for Kerr gold, copper, silver, and molybdenum assays along with the number of assays that were capped.

Table 14.22 Kerr Grade Capping Limits

AUZON (Gold Grade Wireframes)	Gold Assays		Silver Assays	
	Cap Grade (g/t)	No. Capped	Cap Grade (g/t)	No. Capped
Low-grade Domains (<0.2 g/t)	1.0	132	30	62
High-grade Domains (≥0.2 g/t)	10.0	21	100	7
CUZON (Copper Grade Wireframes)	**Copper Assays**		**Molybdenum Assays**	
	Cap Grade (%)	No. Capped	Cap Grade (ppm)	No. Capped
Low-grade Domains (<0.25%)	1.25	60	400	16
High-grade Domains (≥0.25%)	4.00	7		

14.2.3 DRILL HOLE COMPOSITING – KERR

Drill hole assay data (both uncapped and capped intervals) 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. Two sets of composites were generated, one set used for estimating precious metal grades (gold and silver) and the other for estimating base metals (copper and molybdenum). Prior to creating the drill hole composites, the drill hole intervals were coded with the same grade wireframes that were used to constrain the estimate of block grades (i.e. gold or AUZON and copper or CUZON).

14.2.4 VARIOGRAPHY – KERR

A variety of grade and indicator variograms were generated for the Kerr deposit using MineSight® software. Figure 14.25 and Figure 14.26 are down-hole correlograms for gold and copper, respectively. The down-hole gold correlogram was modeled with a nugget effect of approximately 0.59 and a range of about 22 m. The down-hole copper correlogram shows a much lower nugget effect and appreciably longer range than the gold correlogram.

Figure 14.27 and Figure 14.28 are omni-directional correlograms for gold and copper, respectively. Nested spherical models were used in modeling the Kerr gold and copper correlograms shown in Figure 14.27 and Figure 14.28. Ranges of approximately 135 m and 475 m were modeled for the nested structures (Figure 14.27). The nested Kerr copper correlogram structures were modeled with ranges of approximately 160 m and 495 m reflecting the more robust nature of copper mineralization at the Kerr deposit.

Figure 14.25 Kerr Down-hole Gold Correlogram

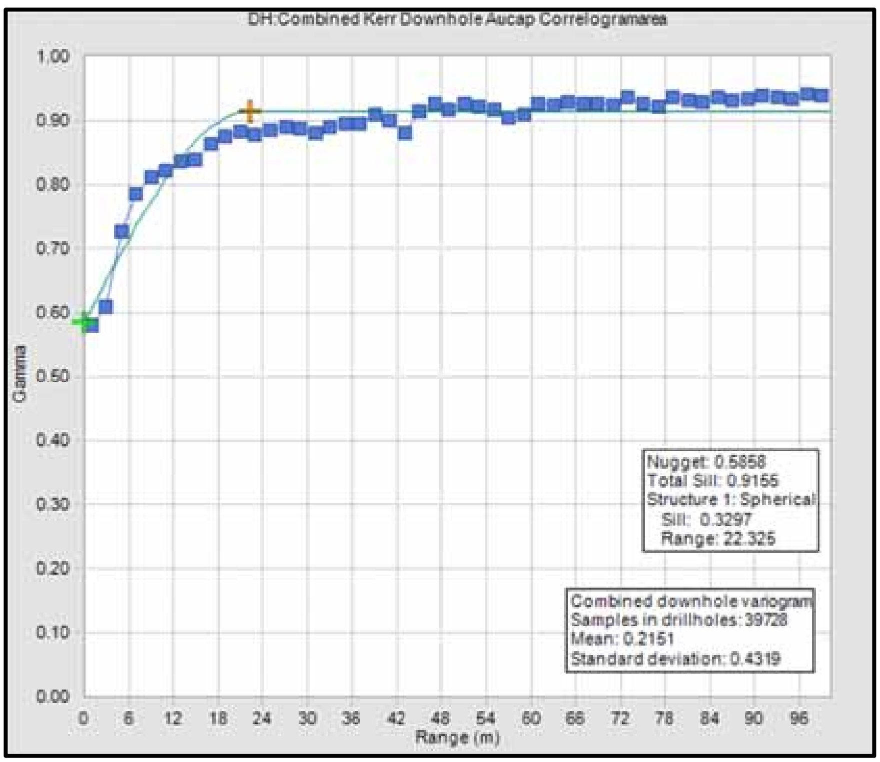


Figure 14.26 Kerr Down-hole Copper Correlogram

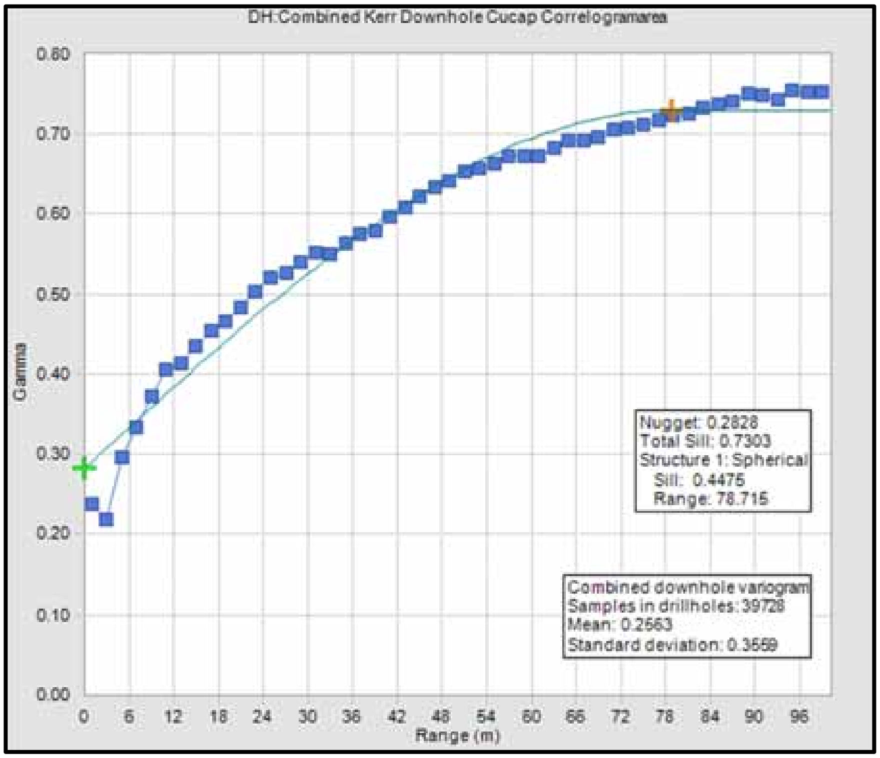


Figure 14.27 Kerr Omni-directional Gold Correlogram

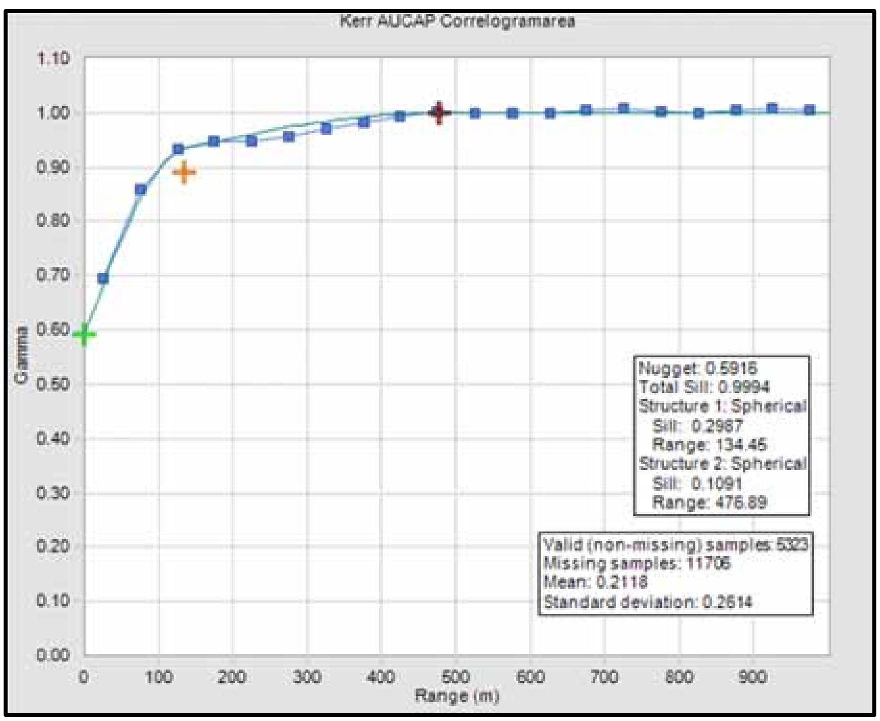
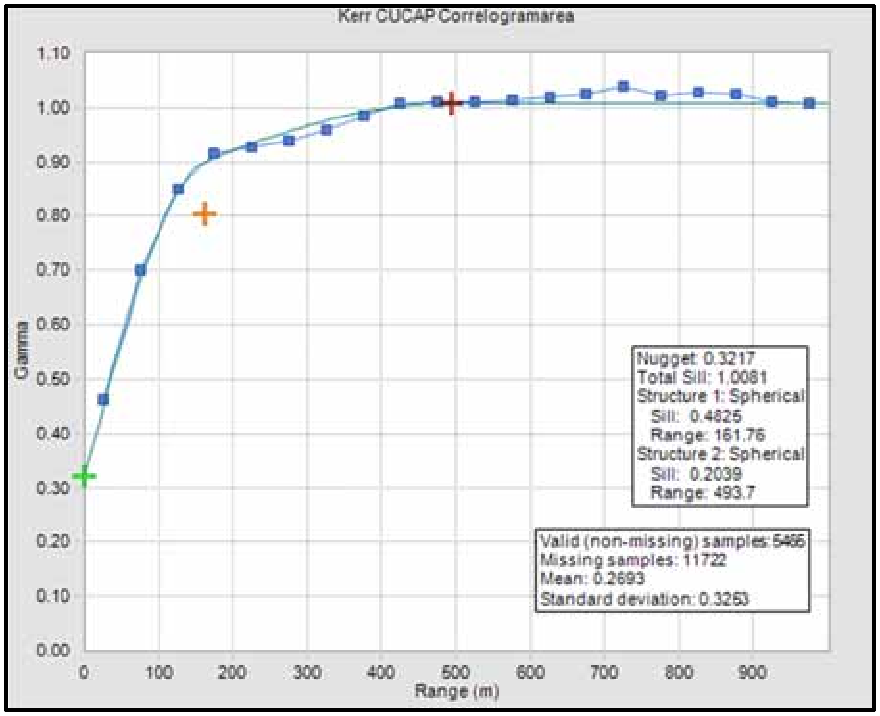


Figure 14.28 Kerr Omni-directional Copper Correlogram



14.2.5 GRADE ESTIMATION PARAMETERS – KERR

After the discovery of deeper mineralization located below the Kerr deposit, a new block model was established for estimating block grades over a larger vertical extent than was previously modeled. Table 14.23 summarizes block model dimensions and limits for the Kerr deposit.

Table 14.23 Kerr Block Model Limits

Parameter	NAD83 Coordinates		Block Size (m)	Number of Blocks	Areal Extent (m)
	Minimum	Maximum			
Easting	420,350	422,435	15	139	2,085
Northing	6,257,750	6,260,450	15	180	2,700
Elevation	-335	1,915	15	150	2,250

Block grades were estimated for the Kerr deposit using a two pass inverse distance cubed method. Instead of using traditional search ellipses for selecting eligible drill hole composites, a trend plane strategy was used. The trend plane method involves identifying the strike and dip of a "domain" and the allowable search distances along strike, down-dip, and perpendicular to the plane. The QP responsible for this section believes that this method of selecting samples minimizes grade smearing often associated with search ellipses and does a better job of ensuring that the distribution of the estimated block grades reflect the underlying structural controls associated with mineralization. The Kerr deposit was sub-divided into four structural domains based primarily on the orientation of the copper-gold mineralization and secondarily by the geometry of alteration and structure. The Kerr deposit consists of two "legs" of mineralization, an eastern steep (approximately 80°) leg and a shallower western leg (60 to 65°). In plan view, those two mineralized legs tend to have a slight northwesterly strike in the southern portion of the deposit which changes to a slight northeasterly strike towards the northern portion of the deposit. Table 14.24 summarizes the strike and dip values associated with the four structural domains that were used in the grade estimation plan.

Table 14.24 Kerr Block Model Domains

Domain	Strike Azimuth (°)	Dip Angle (°)
1	165	-62
2	190	-65
3	170	-80
4	182	-80

Gold and silver block grades were estimated using the same parameters and constraints. Gold grade envelopes and the aforementioned structural domains were the primary constraint used in the estimate of precious metal block grades. Table 14.25 summarizes the gold grade envelopes (AUZON) that were used to control the estimate of block gold and silver grades.

Table 14.25 Kerr Gold Grade Zones (AUZON)

AUZON Code	Description
1	<0.10 g/t Au
2	Diorite
3	PFMF dyke
10	0.10 to 0.20 g/t Au
20	0.20 to 0.40 g/t Au
40	0.40 to 4.00 g/t Au
50	>4.00 g/t Au

As previously mentioned, a two pass estimation strategy was used to estimate block precious metal grades which was constrained by four structural domains and seven gold grade zones. A total of 56 separate interpolation runs were used to estimate Kerr gold and silver block grades (2 * 4 * 7). The first estimation pass for each domain used search distances of 400 m by 400 m by 20 m (along strike, down-dip, and perpendicular to strike). A minimum of three composites were required to estimate block grades with a maximum of six composites and no more than two composites per drill hole allowed for both the first and second estimation passes. Blocks estimated by the first pass were flagged as estimated and not eligible to be estimated by the second pass. The along strike and down-dip ranges of 400 m were kept for the second estimation pass, while the distance perpendicular to strike was increased to 80 m. This strategy tended to generate a distribution of grades that appears to reflect the structural control that can be seen when examining core and drill hole cross sections.

In order to minimize potential boundary effects during the estimation process, the gold grade envelopes were treated as "soft" contacts. For example, blocks located in the 0.20 g/t gold domain could be estimated by composites from that grade domain and if available, composites from the next lower gold grade envelope (i.e. 0.10 to 0.20 g/t domain). This strategy was used for all of the gold grade zones.

The number of composites used to estimate each block, the number of drill holes used, the distance to the closest composite and the average distance of all composites used were stored during the estimation process.

The same estimation strategy that was used to estimate precious metal grades was used to estimate copper and molybdenum block grades (along with arsenic, selenium, sulphur, calcium, and iron). A two pass inverse distance cubed estimator using a trend plane sample selection strategy constrained by four structural domains and eight copper grade domains required 64 separate runs (2 * 4 * 8). Table 14.26 summarizes the eight copper zones that were used to constrain the estimate of base metals.

Table 14.26 Kerr Copper Grade Zones (CUZON)

CUZON Code	Description
1	<0.10% Cu
2	Diorite
3	PFMF dyke
10	0.10 to 0.25% Cu
25	0.25 to 0.50% Cu
50	0.50 to 0.75% Cu
75	0.75 to 1.00% Cu
100	>1.00% Cu

Similar to the soft boundaries used in the gold grade estimation plan, all but the highest grade copper domain (100) were treated as soft domains. For example, blocks in the 0.75 to 1.00% copper domain could be estimated by composites from that domain or if required, composites from the adjacent lower grade copper domain (0.50 to 0.75% copper domain). Blocks inside of the 1.0% copper domain were only estimated by composites from that domain. This decision was made based on observations of high-grade drill core that distinctly show a sharp contact with lower grade material.

The number of composites, drill holes, and distances to composite data were stored during the estimation process. These data were used in conjunction with other criteria in classifying the estimated blocks into various resource categories.

14.2.6 MODEL VALIDATION – KERR

Estimated block grades (gold, silver, copper, and molybdenum) were compared to drill hole composite grades in cross section and level plan views. Figure 14.29 and Figure 14.30 are east-west cross sections drawn through the Kerr block model at 6,259,800 north. 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 800 m elevation through the Kerr block model showing estimated block grades and drill hole composite for gold and copper, respectively. The constraining Mineral Resource pit is shown on the two block model cross sections as a dashed black line. The three conceptual block cave resource shapes are shown as a heavy purple line in the block model cross sections and level maps (Figure 14.29 through Figure 14.32).

Figure 14.29 Kerr Gold Model Section – 6,259,800 North

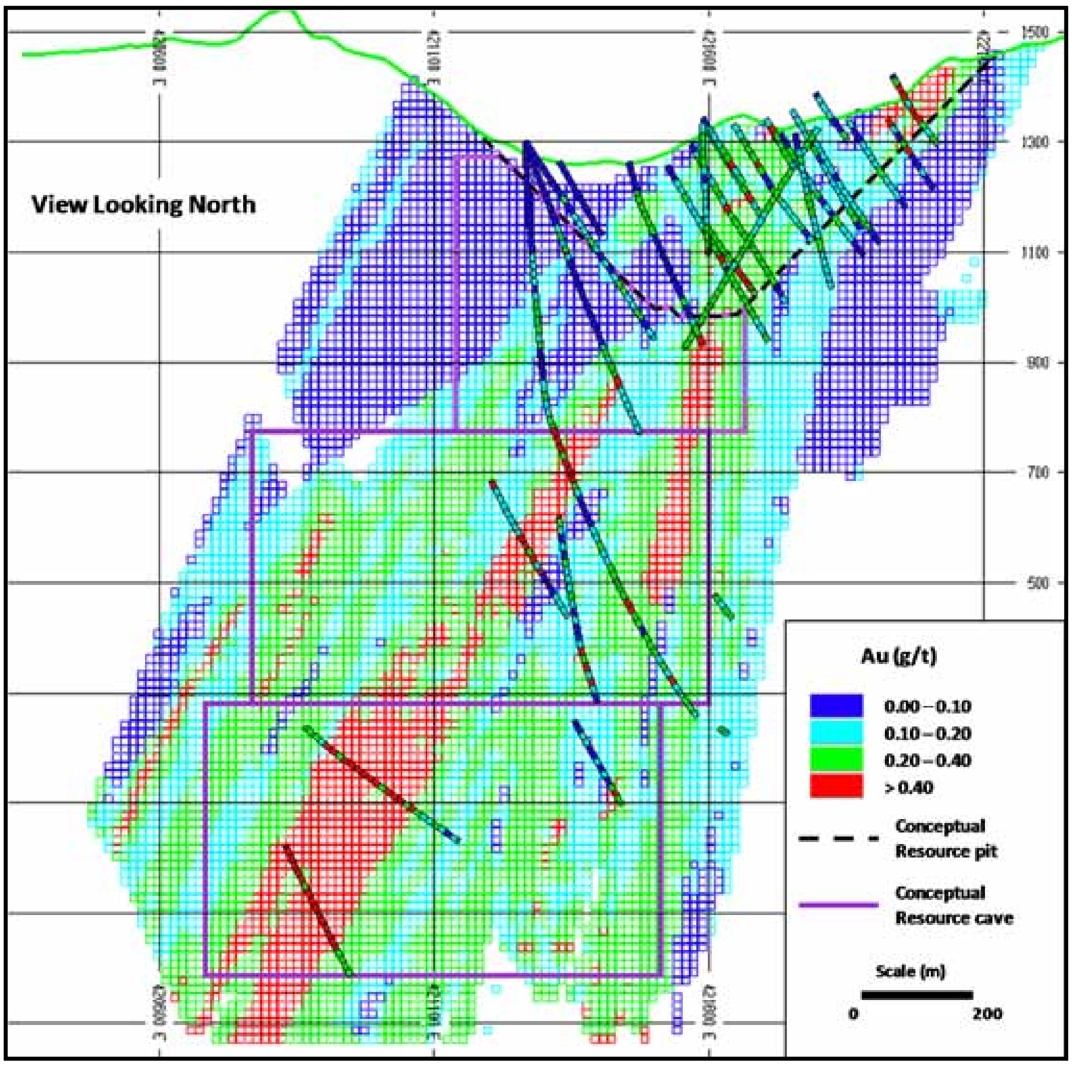


Figure 14.30 Kerr Copper Model Section – 6,259,800 North

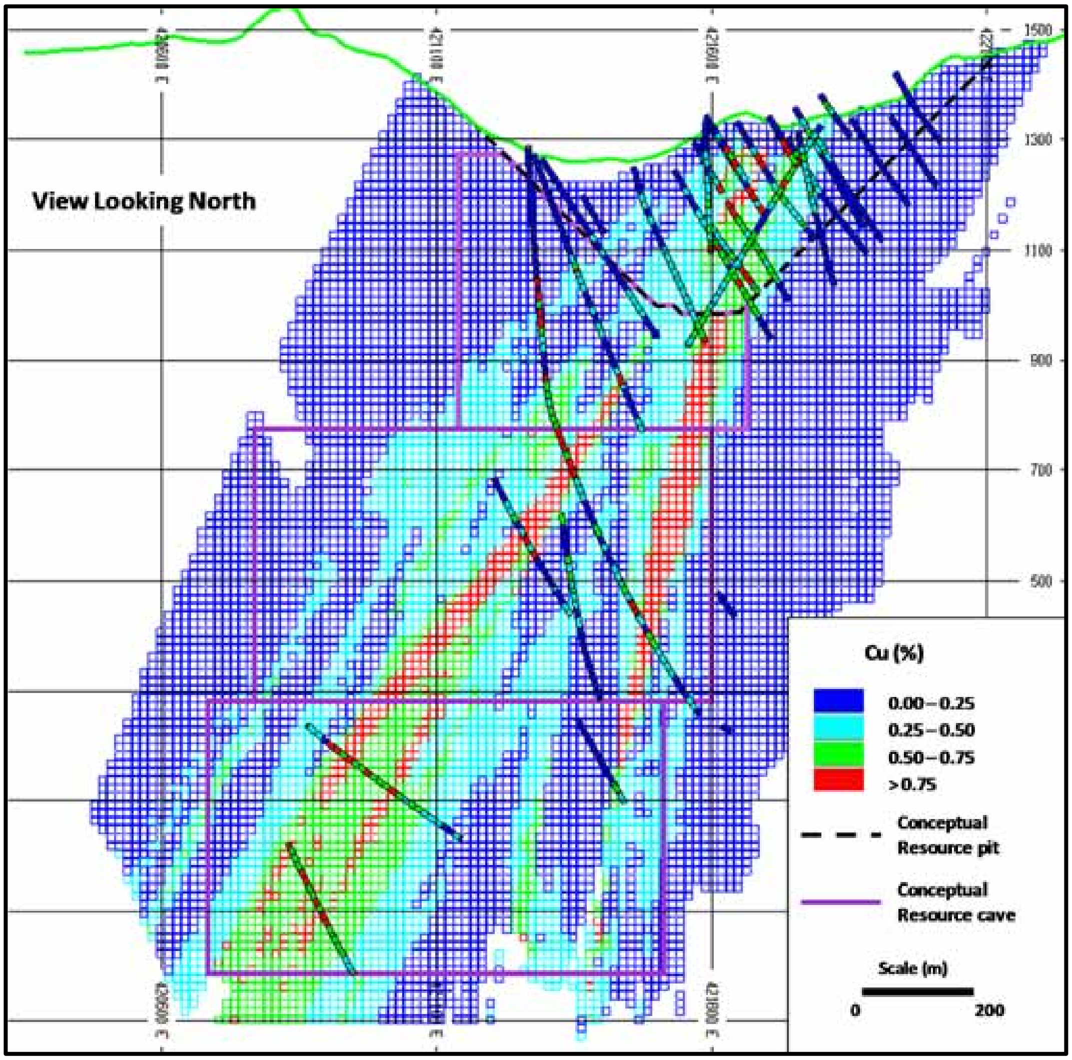


Figure 14.31 Kerr Gold Model – 800 m Level

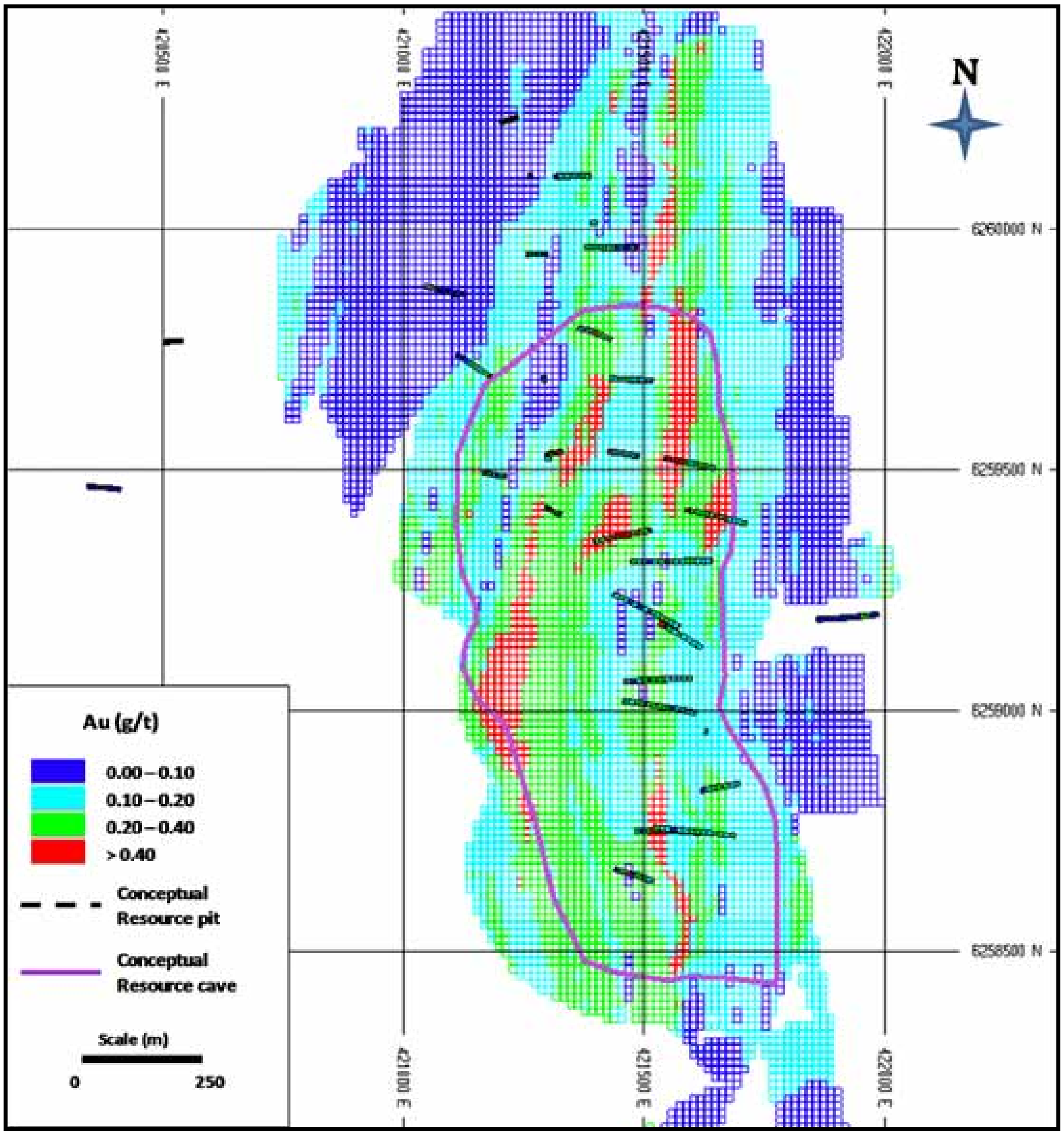
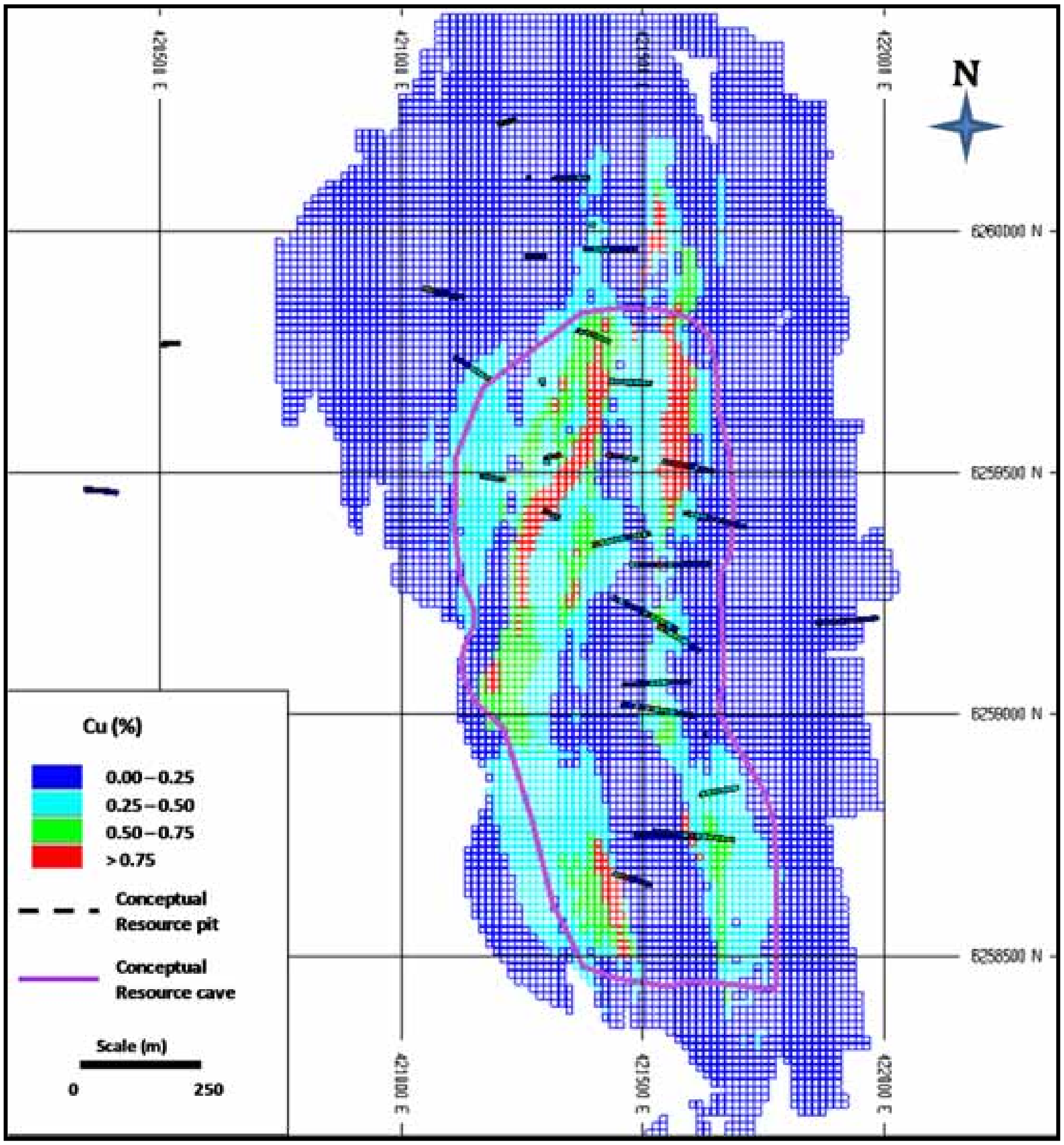


Figure 14.32 Kerr Copper Model – 800 m Level



The grade models were also validated by comparing the IDW block grades against NN models that were generated for gold, copper, silver, and molybdenum. Two NN models were constructed for each metal. Global NN models with no geologic constraints generated grades that were approximately 10% higher than the IDW grades. Conditional NN models that used the same geologic selection criteria as the IDW model compared more closely with the IDW models. The QP responsible for this section of this Technical Report believes that the projection of grades from the global (unrestricted) NN models tended to bias the mean grade and that the conditional NN model represents a more realistic bench mark.

Table 14.27 compares the IDW grades against the conditional NN models.

Table 14.27 Kerr Global Bias Checks)

Metal	Indicated			Inferred		
	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.1912	0.1931	-1.0%	0.2419	0.2436	-0.7%
Copper (%)	0.3195	0.3261	-2.0%	0.2683	0.2724	-1.5%
Silver (g/t)	1.2402	1.2441	-0.3%	1.6011	1.6080	-0.4%
Molybdenum (ppm)	4.4834	4.7408	-5.4%	18.2884	18.3391	-0.3%

The data in Table 14.27 show that most of the IDW grades are slightly lower than the NN grades. The lone exception is for Indicated molybdenum where the IDW grade is approximately 5% lower than the NN grade.

Grade swath plots were generated for rows (east-west), columns (north-south) and levels (elevations) through the block model comparing the IDW and conditional NN models at a zero cut-off grade. Figure 14.33 and Figure 14.34 show swath plots for gold and copper, respectively by elevation. These swath plots show Indicated and Inferred Resource grades.

Figure 14.33 Kerr Gold Model Swath Plot – Elevations

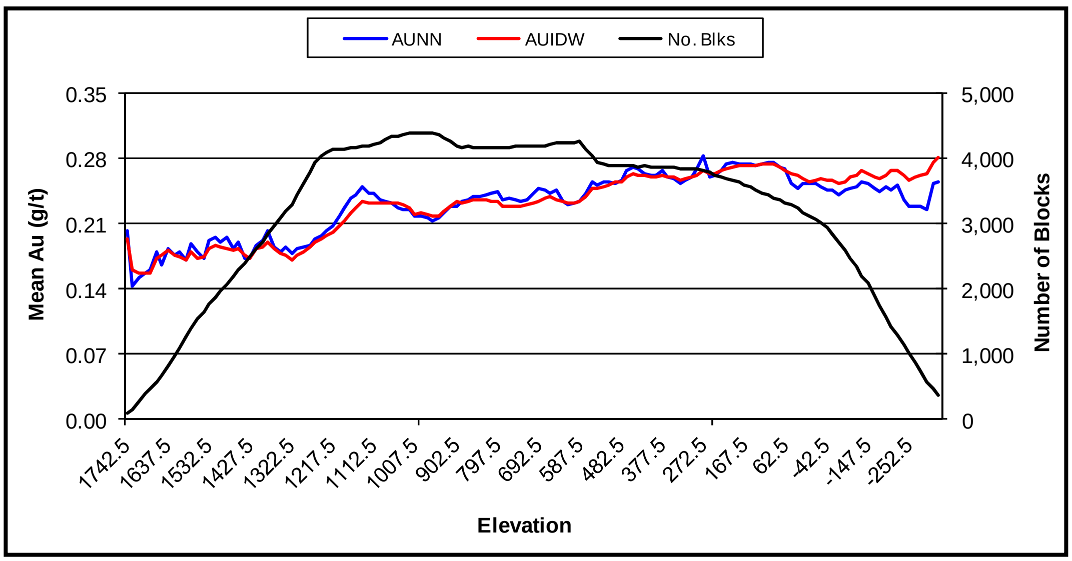
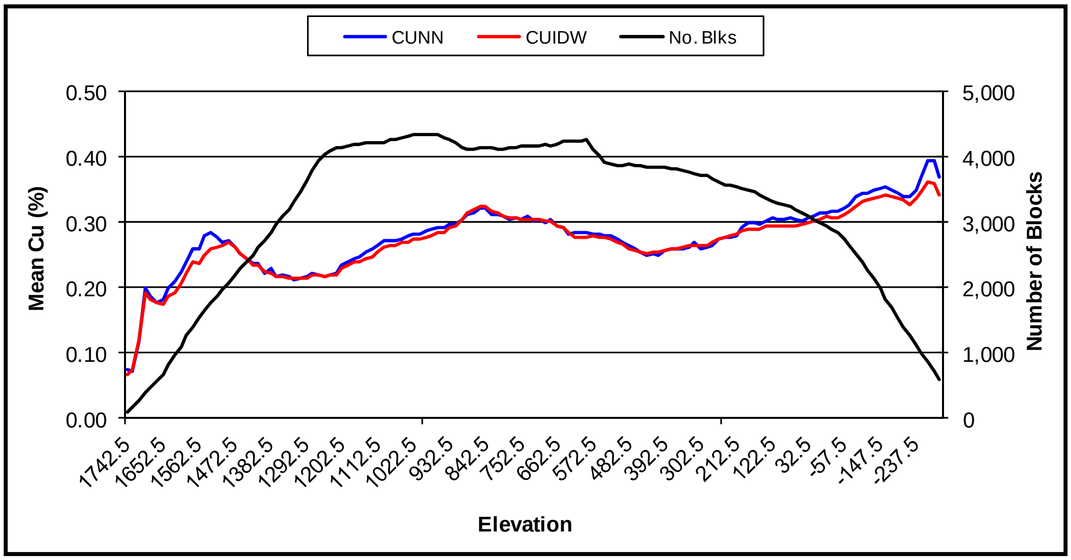


Figure 14.34 Kerr Copper Model Swath Plot – Elevations



In the opinion of the QP responsible for this section, the estimated Kerr block grades are reasonable and unbiased based on visual and statistical reviews.

14.2.7 RESOURCE CLASSIFICATION – KERR

Mineral Resources were assigned to the estimated blocks by constructing 3D solids for Indicated and Inferred Resources. These shapes were based on mineralized continuity as defined by exploration drill hole results. The Indicated Mineral Resource shape was locally extended 25 to 100 m deeper than the shape used for the 2012 PFS Mineral Resource based on drilling results obtained in 2013 through 2015 with an average drill hole spacing of approximately 50 m. The Inferred Mineral Resource 3D solid was significantly increased from the 2012 shape by virtue of deeper, relatively wide spaced drilling that defines two mineralized zones that coalesce near the upper portions of the Kerr deposit. The average drill hole spacing for Inferred blocks is approximately 125 m.

14.3 IRON CAP DEPOSIT

Since the 2012 PFS (Tetra Tech 2012) was published coupled with their success at exploring below the Kerr deposit, Seabridge conducted deeper drilling activities below the Iron Cap deposit in an effort to locate potentially higher grade copper and gold mineralization. A drill campaign in 2014 provided more insight into the geometry and extent of mineralization at Iron Cap.

With substantially more deeper drill hole data and an updated geologic interpretation, a new block model was constructed for the Iron Cap deposit. The remainder of Section 14.3 deals with the new Iron Cap model which replaces the 2012 PFS Iron Cap Mineral Resource.

14.3.1 METAL DISTRIBUTION – IRON CAP

The distribution of gold grades based on raw uncomposited data is summarized in Table 14.28 at four different cut-off grades by the gold grade envelopes that were used in the block grade estimation process. The data in Table 14.28 shows statistics for uncapped (left portion of table) and statistics for capped assays (right portion of table). Grade capping of high-grade outliers is discussed in Section 14.3.2.

The data shown in Table 14.28 show that the average uncapped gold grade for all Iron Cap drill hole samples is 0.39 g/t with about 21% of the samples above a 0.5 g/t cut-off. The CV after capping the high-grade outliers is approximately 1.0 for samples inside of the 0.25 and 0.50 g/t grade envelopes.

Copper assay statistics for both uncapped and capped data are summarized in Table 14.29 where the grades are broken out by copper grade envelopes that were used in the estimate of block grades.

The data shown in Table 14.29 show that the average uncapped copper grade for all Iron Cap drill hole samples is 0.21% with about 28% of the samples above a 0.25% cut-off. The CV even without capping high-grade outliers is significantly less than 1.0 for all samples.

Table 14.28 Iron Cap Gold Assay Statistics

AUZON	Uncapped Au Statistics Above Cut-off								Capped Au Statistics Above Cut-off				
	Au Cut-off (g/t)	Total Meters (m)	Inc. (%)	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. (%)	Std. Dev.	CV	Mean Au (g/t)	Grd-Thk (g/t-m)	Inc. (%)	Std. Dev.	CV
All Data	0.00	33,937	51	0.39	13,372	17.3	0.61	1.55	0.39	13,083	17.7	0.50	1.30
	0.25	16,765	28	0.66	11,059	25.2	0.78	1.18	0.64	10,770	25.8	0.61	0.95
	0.50	7,186	15	1.07	7,686	25.5	1.06	0.99	1.03	7,397	26.0	0.77	0.75
	1.00	2,202	6	1.94	4,280	32.0	1.59	0.82	1.81	3,991	30.5	1.01	0.56
<0.25 g/t	0.00	13,694	80	0.20	2,747	45.8	0.33	1.67	0.20	2,684	46.8	0.26	1.34
	0.25	2,803	15	0.53	1,490	25.0	0.63	1.18	0.51	1,427	25.6	0.45	0.88
	0.50	756	4	1.06	802	13.3	1.03	0.97	0.98	739	13.6	0.65	0.67
	1.00	203	1	2.15	437	15.9	1.51	0.70	1.84	374	13.9	0.72	0.39
0.25 g/t Shell	0.00	13,491	41	0.40	5,360	17.2	0.59	1.48	0.38	5,184	17.7	0.39	1.01
	0.25	7,986	40	0.56	4,440	35.2	0.72	1.30	0.53	4,264	36.4	0.44	0.83
	0.50	2,602	15	0.98	2,555	25.3	1.15	1.17	0.91	2,379	26.1	0.62	0.67
	1.00	584	4	2.06	1,201	22.4	2.08	1.01	1.76	1,025	19.8	0.84	0.48
0.50 g/t Shell	0.00	6,752	11	0.78	5,265	2.6	0.85	1.09	0.77	5,215	2.6	0.77	1.00
	0.25	5,976	32	0.86	5,129	15.2	0.88	1.02	0.85	5,078	15.3	0.79	0.93
	0.50	3,828	36	1.13	4,330	32.0	1.00	0.88	1.12	4,280	32.4	0.87	0.78
	1.00	1,416	21	1.87	2,643	50.2	1.34	0.72	1.83	2,592	49.7	1.11	0.60

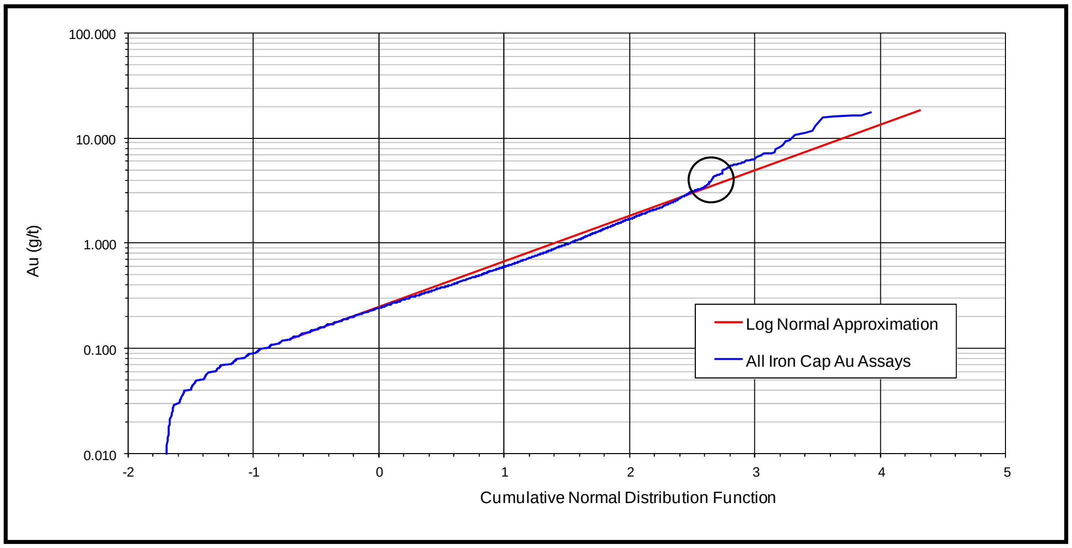
Table 14.29 Iron Cap Copper Assay Statistics

	Uncapped Cu Statistics Above Cut-off								Capped Cu Statistics Above Cut-off				
CUZON	Cu Cut-off (%)	Total Meters (m)	Inc. (%)	Mean Cu (%)	Grd-Thk (%-m)	Inc. (%)	Std. Dev.	CV	Mean Cu (%)	Grd-Thk (%-m)	Inc. (%)	Std. Dev.	CV
All Data	0.00	33,482	26	0.21	6,886	7.0	0.16	0.79	0.21	6,872	7.0	0.16	0.78
	0.10	24,721	46	0.26	6,402	37.7	0.16	0.61	0.26	6,388	37.7	0.15	0.60
	0.25	9,433	23	0.40	3,808	37.6	0.17	0.42	0.40	3,794	37.6	0.16	0.40
	0.50	1,814	5	0.67	1,222	17.7	0.20	0.29	0.67	1,208	17.6	0.17	0.25
<0.10%	0.00	11,270	48	0.13	1,459	18.0	0.12	0.92	0.13	1,448	18.1	0.11	0.85
	0.10	5,837	41	0.21	1,197	50.6	0.12	0.59	0.20	1,186	50.9	0.10	0.51
	0.25	1,248	10	0.37	459	23.8	0.17	0.47	0.36	448	24.0	0.12	0.32
	0.50	164	1	0.68	111	7.6	0.30	0.45	0.61	100	6.9	0.08	0.14
0.10% Shell	0.00	5,083	36	0.15	757	14.8	0.11	0.71	0.15	757	14.8	0.11	0.71
	0.10	3,246	51	0.20	645	55.5	0.10	0.51	0.20	645	55.5	0.10	0.51
	0.25	631	11	0.36	225	23.3	0.13	0.35	0.36	225	23.3	0.12	0.35
	0.50	76	2	0.63	48	6.4	0.14	0.22	0.63	48	6.4	0.14	0.22
0.20% Shell	0.00	5,265	14	0.20	1,071	4.9	0.11	0.56	0.20	1,070	4.9	0.11	0.56
	0.10	4,538	61	0.22	1,018	52.0	0.11	0.48	0.22	1,018	52.0	0.11	0.48
	0.25	1,309	23	0.35	461	36.4	0.12	0.33	0.35	461	36.4	0.11	0.33
	0.50	112	2	0.64	72	6.7	0.15	0.24	0.64	71	6.7	0.14	0.22
0.30% Shell	0.00	9,709	8	0.28	2,674	2.0	0.16	0.59	0.28	2,673	2.0	0.16	0.59
	0.10	8,979	45	0.29	2,620	29.6	0.16	0.54	0.29	2,620	29.6	0.16	0.54
	0.25	4,574	39	0.40	1,827	48.1	0.15	0.38	0.40	1,827	48.1	0.15	0.38
	0.50	819	8	0.66	541	20.2	0.17	0.25	0.66	541	20.2	0.17	0.25
0.40% Shell	0.00	2,156	2	0.43	924	0.3	0.23	0.53	0.43	922	0.3	0.22	0.52
	0.10	2,122	21	0.43	922	9.3	0.23	0.52	0.43	920	9.4	0.22	0.51
	0.25	1,671	48	0.50	835	41.8	0.21	0.42	0.50	833	41.9	0.20	0.41
	0.50	642	30	0.70	449	48.6	0.20	0.29	0.70	447	48.5	0.19	0.27

14.3.2 HIGH-GRADE OUTLIERS – IRON CAP

Cumulative probability plots were used to identify high-grade outliers for gold, copper, silver, and molybdenum based on the original assay samples. The assays were initially examined with respect to logged lithology and alteration types however the final analysis of high-grade outliers was completed using the grade wireframes that were used to interpolate block grades. Figure 14.35 is a cumulative probability plot that shows the distribution of raw gold assays for all Iron Cap assays.

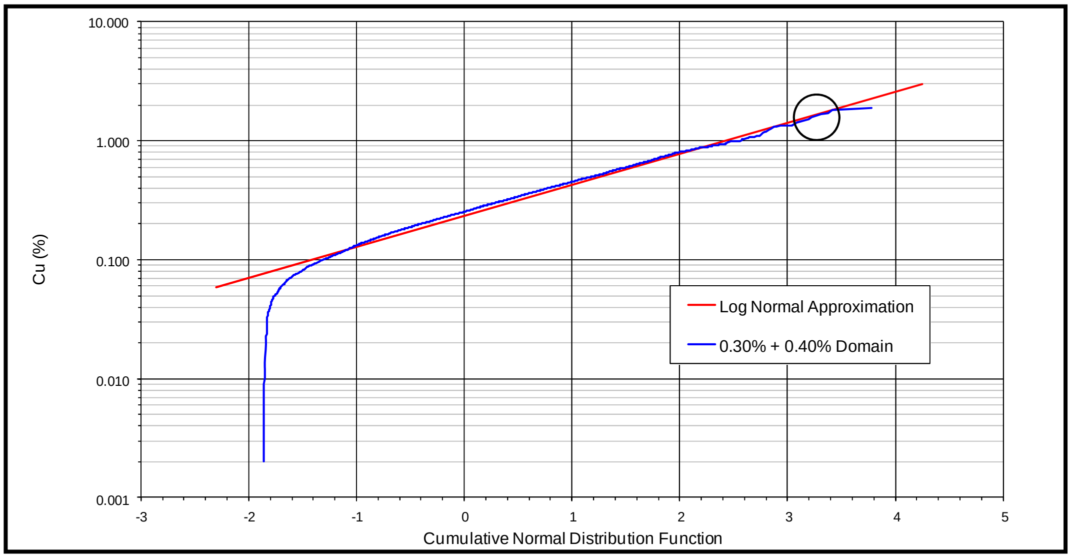
Figure 14.35 Iron Cap Gold Cumulative Probability Plot – All Samples



As can be seen in Figure 14.35, gold grades deviate from the approximated log normal line shown in red above 3 g/t. Outlier samples were capped at 3 g/t, 4 g/t, and 7 g/t for less than 0.25 g/t, 0.25 to 50 g/t, and greater than 0.50 g/t, respectively, based on cumulative probability plots.

Figure 14.36 is a cumulative probability plot that shows the distribution of copper grades within combined 0.30% and 0.40% wireframes.

Figure 14.36 Iron Cap Copper Cumulative Probability Plot – 0.3% and 0.4% Shells



Iron Cap assay capping limits for high-grade outliers are summarized in Table 14.30.

Table 14.30 Iron Cap Capping Limits

	Gold Assays	
AUZON (Gold Grade Wireframes)	Cap Grade (g/t)	No. Capped
Low-grade Domain (<0.25 g/t)	3.0	27
Medium grade Domain (0.25 to 0.50 g/t)	4.0	23
Higher-grade Domain (≥0.50 g/t)	7.0	15
	Copper Assays	
CUZON (Copper Grade Wireframes)	Cap Grade (%)	No. Capped
Low-grade Domain (<0.10%)	0.75	13
Medium-grade Domains (0.10 to 0.20%)	1.00	3
Higher-grade Domains (0.30 to 0.40%)	1.50	8
	Silver Assays	
AGZON (Silver Grade Wireframes)	Cap Grade (g/t)	No. Capped
Low-grade Domain (<3 g/t)	0.75	32
Higher-grade Domain (>3 g/t)	1.00	12
	Copper Assays	
MOZON (Molybdenum Grade Wireframes)	Cap Grade (ppm)	No. Capped
All Domains	2,500	4

14.3.3 Drill Hole Compositing – Iron Cap

Drill hole assay data (both uncapped and capped intervals) 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. Two sets of composites were generated, one set used for estimating precious metal grades (gold and silver) and the other for estimating base metals (copper and molybdenum). Prior to creating the drill hole composites, the drill hole intervals were coded with the same grade wireframes that were used to constrain the estimate of block grades.

14.3.4 Variography – Iron Cap

A variety of grade and indicator variograms were generated for the Iron Cap deposit using MineSight® software. Figure 14.37 and Figure 14.38 are down-hole correlograms for gold and copper, respectively. The down-hole gold correlogram was modeled with a nugget effect of about 0.30 and a range of about 24 m. The down-hole copper correlogram shows a slightly lower nugget effect and an appreciably longer down-hole range than the gold correlogram.

Figure 14.39 and Figure 14.40 are omni-directional correlograms for gold and copper, respectively based on 15 m drill hole composites. Nested spherical models were used in modeling the Iron Cap gold and copper correlograms shown in Figure 14.39 and Figure 14.40. Ranges of approximately 40 m, 125 m, and 165 m were modeled for the nested gold model structures (Figure 14.39). The nested Iron Cap copper correlogram structures were modeled with ranges of approximately 55 m, 135 m, and 215 m reflecting the more continuous nature of copper mineralization at the Iron Cap deposit (Figure 14.40).

Figure 14.37 Iron Cap Down-hole Gold Correlogram

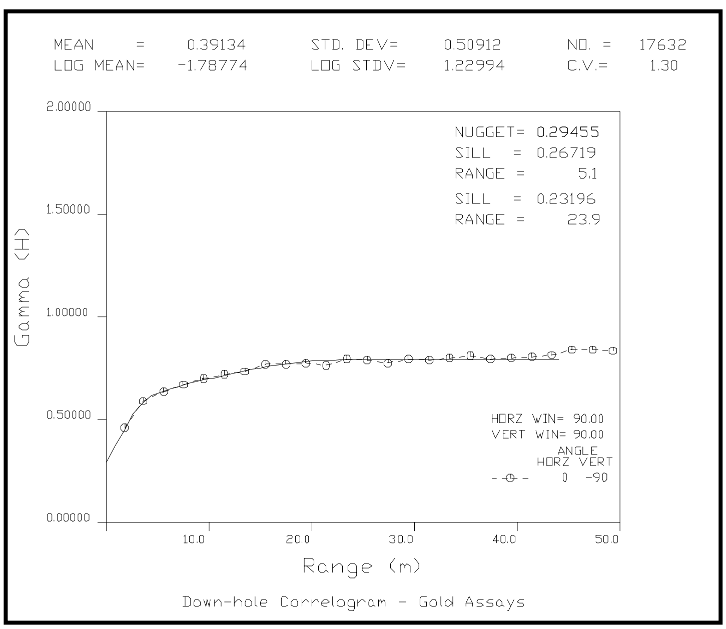


Figure 14.38 Iron Cap Down-hole Copper Correlogram

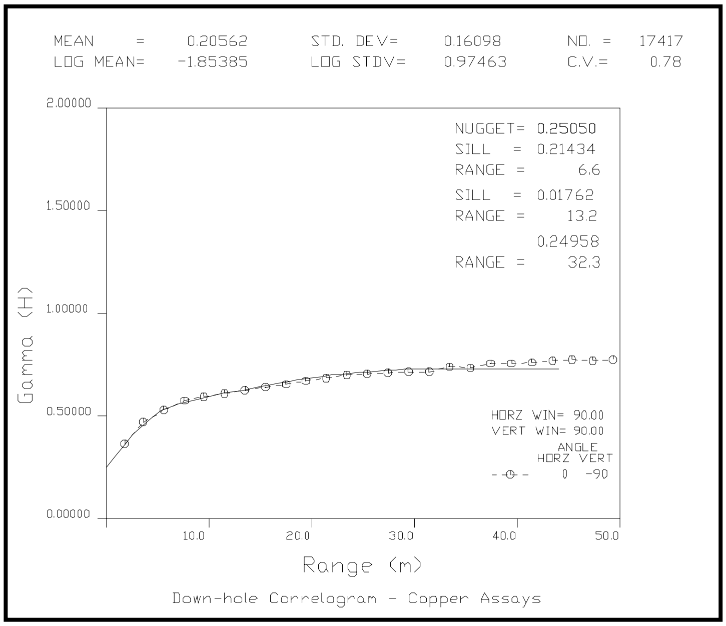


Figure 14.39 Iron Cap Omni-directional Gold Correlogram

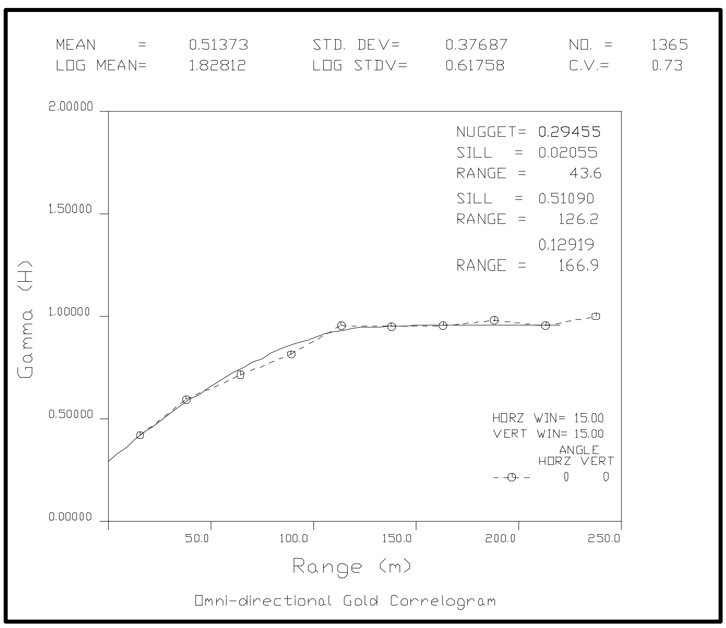
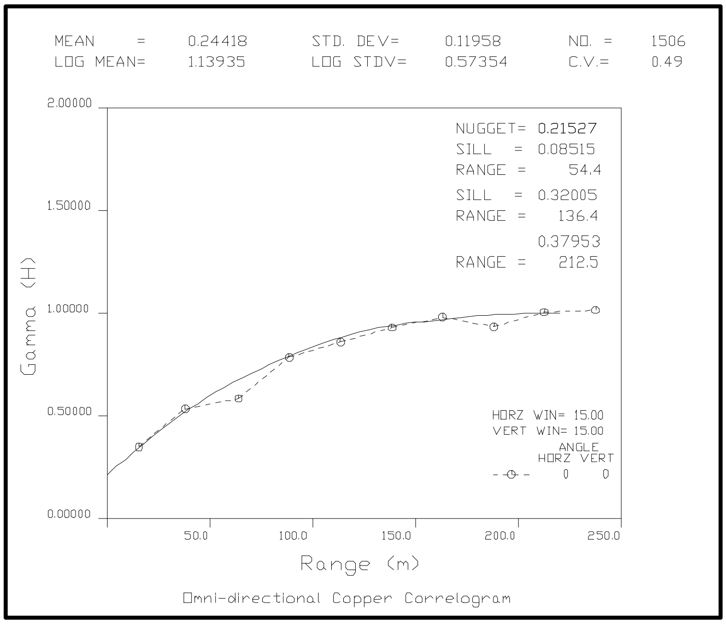


Figure 14.40 Iron Cap Omni-directional Copper Correlogram



14.3.5 GRADE ESTIMATION PARAMETERS – IRON CAP

Following their success at intersecting deep mineralization located below the Kerr deposit, Seabridge drilled a series of holes below the Iron Cap deposit in 2013 and 2014. A new block model was constructed in early 2015 for the Iron Cap deposit. Table 14.31 summarizes block model dimensions and limits for the Iron Cap deposit.

Table 14.31 Kerr Block Model Limits

Parameter	NAD83 Coordinates		Block Size (m)	Number of Blocks	Areal Extent (m)
	Minimum	Maximum			
Easting	423,000	425,070	15	138	2,070
Northing	6,266,400	6,268,380	15	132	1,980
Elevation	540	2,100	15	104	1,560

Block grades were estimated for the Iron Cap deposit using a three pass IDW method. Instead of using traditional search ellipses for selecting eligible drill hole composites, a trend plane strategy was used. The trend plane method involves identifying the strike and dip of a “domain” and providing for allowable search distances along the strike, down-dip, and perpendicular to the plane directions. The QP responsible for this section believes that this method of selecting samples minimizes grade smearing often associated with search ellipses and does a better job of ensuring that the distribution of the estimated block grades reflect the underlying structural controls associated with mineralization.

Block gold, copper, silver, and molybdenum grades were estimated using manually constructed grade envelope wireframes to constrain the estimate. The various wireframes used in the estimation plan are summarized in Table 14.32.

Table 14.32 Iron Cap Grade Envelopes

AUZON Code	Description
1	<0.25 g/t Au
5	<0.25 g/t Au – internal low-grade
25	0.25 to 0.50 g/t Au
50	>0.50 g/t Au
CUZON Code	**Description**
1	<0.10% Cu
5	<0.10% Cu – internal low-grade
10	0.10 to 0.20% Cu
20	0.20 to 0.30% Cu
30	0.30 to 0.40% Cu
40	>0.40% Cu
AGZON Code	**Description**
1	<3 g/t Ag
3	>3 g/t Ag
MOZON Code	**Description**
1	<25 ppm Mo
25	25 to 50 ppm Mo
50	>50 ppm Mo

Table 14.33 through Table 14.36 summarize the key parameters that were used to estimate gold, copper, silver, and molybdenum, respectively.

Table 14.33 Iron Cap Gold Estimation Parameters

Estimation Pass	AUZON Code	ID Power	Trend Plane Search Ranges (m)			Number of Composites Used			Max. Proj. Dist. 1 Comp (m)	Trend Plane Orientation		Eligble AUZON DH Comps
			Strike	Dip	Perp. Stk	Min	Max	Max/ Hole		Strike Azm (°)	Dip Angle (°)	
1	1	3	7.5	7.5	7.5	1	3	1	7.5	28	65	1
2	1	3	15	15	7.5	3	6	2	n/a	28	65	1
3	1	3	30	30	15	1	3	1	30	28	65	1
1	5	3	75	75	15	1	3	1	30	28	65	1 & 5
2	5	3	150	150	30	3	6	2	n/a	28	65	1 & 5
3	5	3	250	250	50	1	3	1	125	28	65	1 & 5
1	25	3	75	75	15	1	3	1	30	28	65	1, 5 & 25
2	25	3	150	150	30	3	6	2	n/a	28	65	1, 5 & 25
3	25	3	250	250	50	1	3	1	125	28	65	1, 5 & 25
1	50	3	75	75	15	1	3	1	30	28	65	25 & 50
2	50	3	150	150	30	3	6	2	n/a	28	65	25 & 50
3	50	3	250	250	50	1	3	1	125	28	65	25 & 50

Table 14.34 Iron Cap Copper Estimation Parameters

Estimation Pass	CUZON Code	ID Power	Trend Plane Search Ranges (m)			Number of Composites Used			Max. Proj. Dist. 1 Comp (m)	Trend Plane Orientation		Eligble CUZON DH Comps
			Strike	Dip	Perp. Stk	Min	Max	Max/ Hole		Strike Azm (°)	Dip Angle (°)	
1	1	2	7.5	7.5	7.5	1	3	1	7.5	28	65	1
2	1	2	15	15	7.5	3	8	2	n/a	28	65	1
3	1	2	30	30	15	1	3	1	30	28	65	1
1	5	2	75	75	15	1	3	1	30	28	65	1 & 5
2	5	2	150	150	30	3	8	2	n/a	28	65	1 & 5
3	5	2	250	250	50	1	3	1	125	28	65	1 & 5
1	10	2	75	75	15	1	3	1	30	28	65	1, 5 & 10
2	10	2	150	150	30	3	8	2	n/a	28	65	1, 5 & 10
3	10	2	250	250	50	1	3	1	125	28	65	1, 5 & 10
1	20	2	75	75	15	1	3	1	30	28	65	10 & 20
2	20	2	150	150	30	3	8	2	n/a	28	65	10 & 20
3	20	2	250	250	50	1	3	1	125	28	65	10 & 20
1	30	2	75	75	15	1	3	1	30	28	65	20 & 30
2	30	2	150	150	30	3	8	2	n/a	28	65	25 & 50
3	30	2	250	250	50	1	3	1	125	28	65	10 & 20
1	40	2	75	75	15	1	3	1	30	28	65	30 & 40
2	40	2	150	150	30	3	8	2	n/a	28	65	30 & 40
3	40	2	250	250	50	1	3	1	125	28	65	30 & 40

Table 14.35 Iron Cap Silver Estimation Parameters

Estimation Pass	AGZON Code	ID Power	Trend Plane Search Ranges (m)			Number of Composites Used			Max. Proj. Dist. 1 Comp (m)	Trend Plane Orientation		Eligble AGZON DH Comps
			Strike	Dip	Perp. Stk	Min	Max	Max/ Hole		Strike Azm (°)	Dip Angle (°)	
1	1	3	7.5	7.5	7.5	1	3	1	7.5	28	65	1
2	1	3	15	15	7.5	3	8	2	n/a	28	65	1
3	1	3	30	30	15	1	3	1	30	28	65	1
1	3	3	75	75	15	1	3	1	30	28	65	1 & 3
2	3	3	150	150	30	3	8	2	n/a	28	65	1 & 3
3	3	3	250	250	50	1	3	1	125	28	65	1 & 3

Table 14.36 Iron Cap Molybdenum Estimation Parameters

Estimation Pass	MOZON Codes	ID Power	Trend Plane Search Ranges (m)			Number of Composites Used			Max. Proj. Dist. 1 Comp (m)	Trend Plane Orientation		Eligble MOZON DH Comps
			Strike	Dip	Perp. Stk	Min	Max	Max/ Hole		Strike Azm (°)	Dip Angle (°)	
1	1	2	7.5	7.5	7.5	1	3	1	7.5	28	65	1
2	1	2	15	15	7.5	3	8	2	n/a	28	65	1
3	1	2	30	30	15	1	3	1	30	28	65	1
1	25	2	75	75	15	1	3	1	30	28	65	1 & 25
2	25	2	150	150	30	3	8	2	n/a	28	65	1 & 25
3	25	2	250	250	50	1	3	1	125	28	65	1 & 25
1	50	2	75	75	15	1	3	1	30	28	65	25 & 50
2	50	2	150	150	30	3	8	2	n/a	28	65	25 & 50
3	50	2	250	250	50	1	3	1	125	28	65	25 & 50

In order to minimize potential boundary effects during the estimation process, the grade envelopes were treated as "soft" contacts for all but the lowest grade domains. For example, blocks located in the 0.50 g/t gold domain could be estimated by composites from that grade domain and if available, composites from the next lower gold grade envelope (i.e. 0.25 g/t domain). This strategy was used for all four of the metals that were estimated for the Iron Cap deposit.

The number of composites used to estimate each block, the number of drill holes used, the distance to the closest composite and the average distance of all composites used were stored during the estimation process.

14.3.6 MODEL VALIDATION – IRON CAP

Estimated block grades (gold, silver, copper, and molybdenum) were compared to drill hole composite grades in cross section and level plan views. Figure 14.41 and Figure 14.42 are northwest-southeast cross sections drawn through the Iron Cap block model. These figures show estimated block/composite gold grades (Figure 14.41) and block/composite copper grades (Figure 14.42). Figure 14.43 and Figure 14.44 are block model level maps drawn at the 1,200 m elevation through the Iron Cap block model showing estimated block grades and drill hole composite for gold and copper, respectively. The constraining Mineral Resource block cave shapes are shown on the block model cross sections and level plans as thick purple lines.

Figure 14.41 Iron Cap Gold Model Cross Section

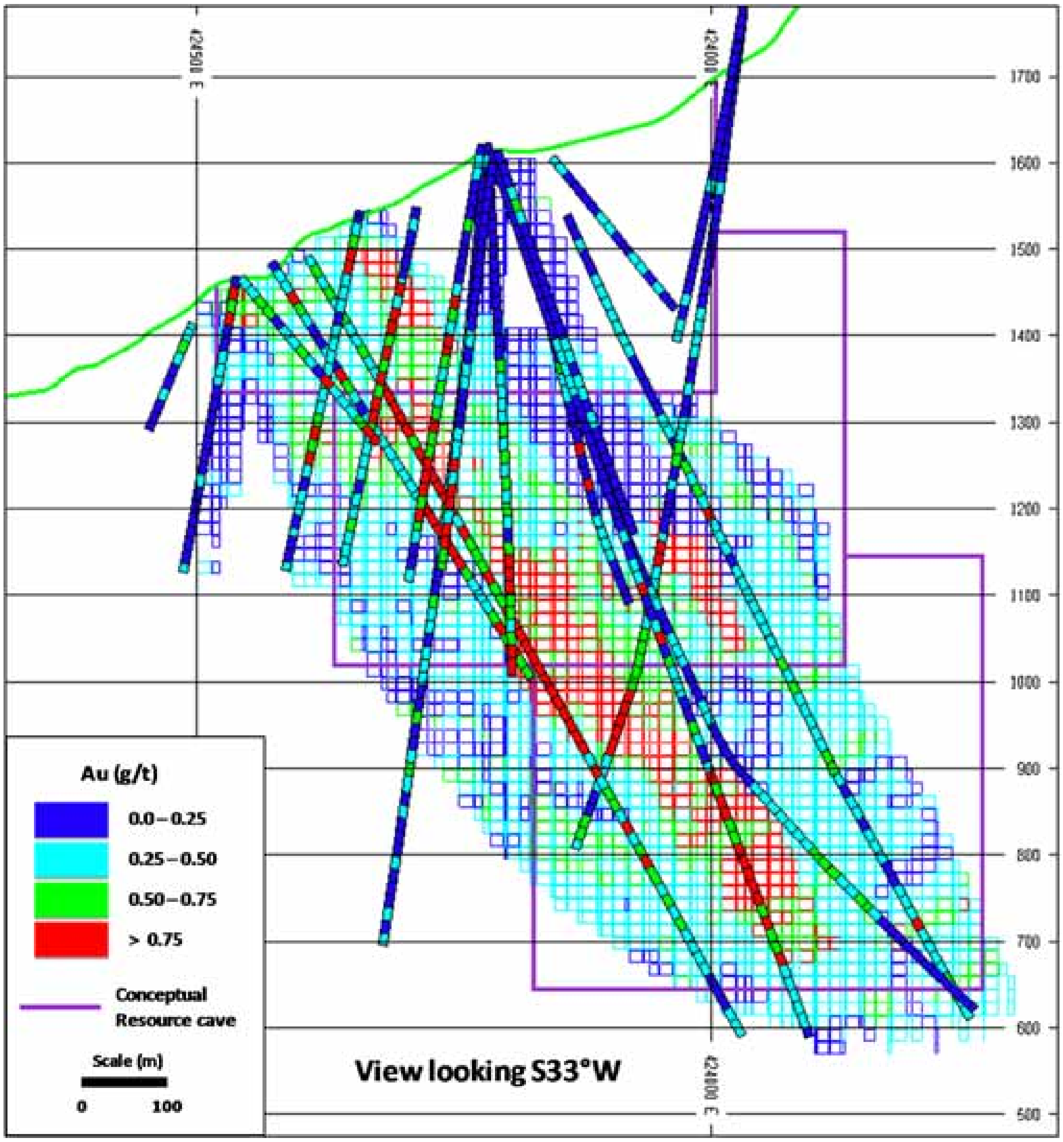


Figure 14.42 Iron Cap Copper Model Cross Section

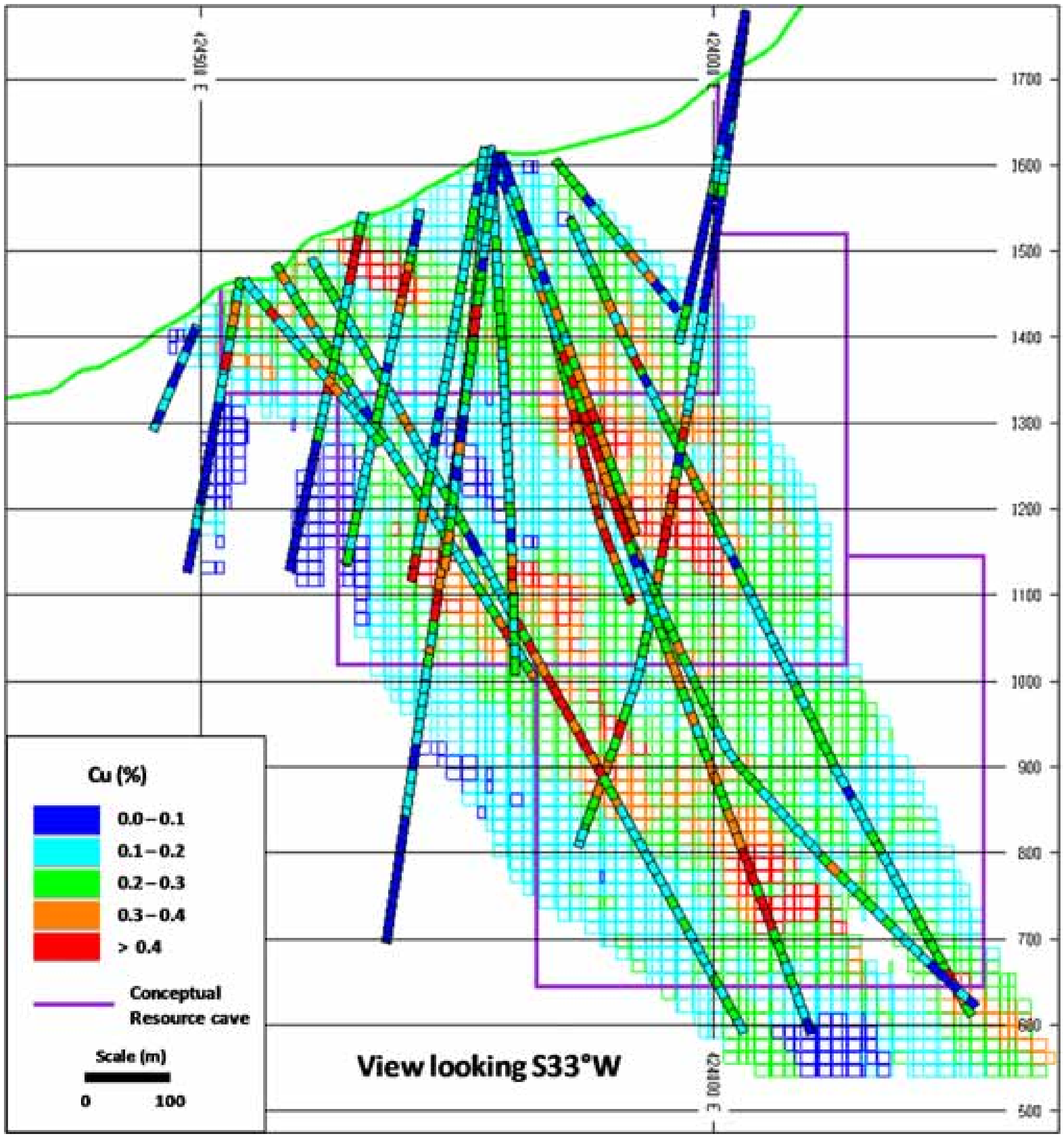


Figure 14.43 Iron Cap Gold Model – 1,200 m Level

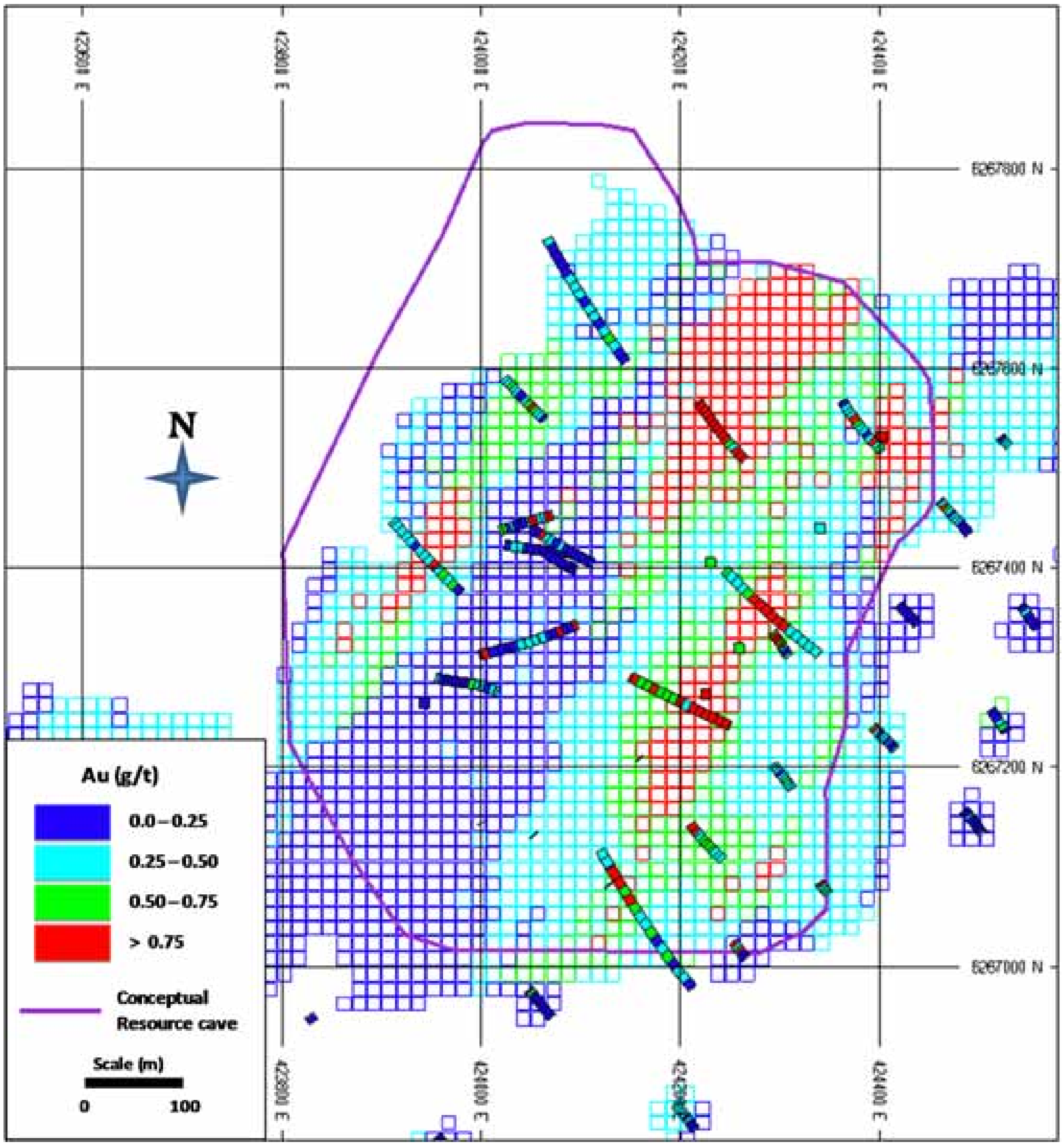
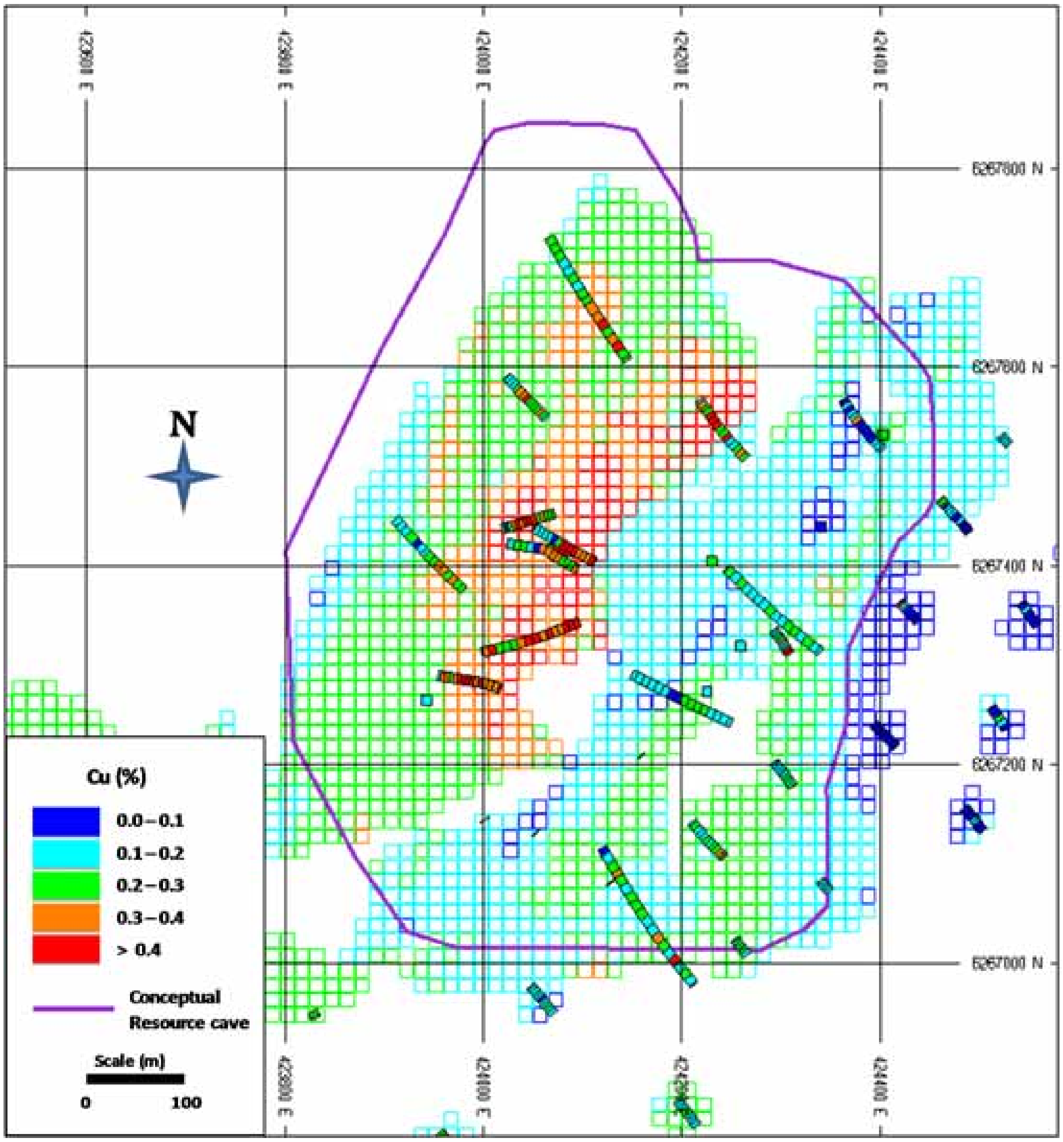


Figure 14.44 Iron Cap Copper Model – 1,200 m Level



The grade models were also validated by comparing the IDW block grades against NN models that were generated for gold, copper, silver, and molybdenum. Table 14.37 compares the IDW grades against the conditional NN models.

Table 14.37 Iron Cap Global Bias Grade Checks

	Indicated			Inferred		
Metal	IDW Grade	NN Grade	% Diff	IDW Grade	NN Grade	% Diff
Gold (g/t)	0.4704	0.4767	-1.3%	0.3850	0.3961	-2.8%
Copper (%)	0.2126	0.2176	-2.3%	0.2078	0.2110	-1.5%
Silver (g/t)	5.5613	5.5398	0.4%	4.6708	4.6250	1.0%
Molybdenum (ppm)	39.6292	41.0213	-3.4%	67.8675	68.9775	-1.6%

The data in Table 14.37 show that most of the IDW grades are slightly lower than the NN grades. The lone exception is for silver which shows that the IDW grade model is slightly higher than the NN model.

Grade swath plots were generated for rows (east-west), columns (north-south) and levels (elevations) through the block model comparing the IDW and NN models at a zero cut-off grade. Figure 14.45 and Figure 14.46 show swath plots for gold and copper, respectively by elevation. These swath plots show results from Indicated and Inferred Resource blocks.

Figure 14.45 Iron Cap Gold Model Swath Plot – Elevations

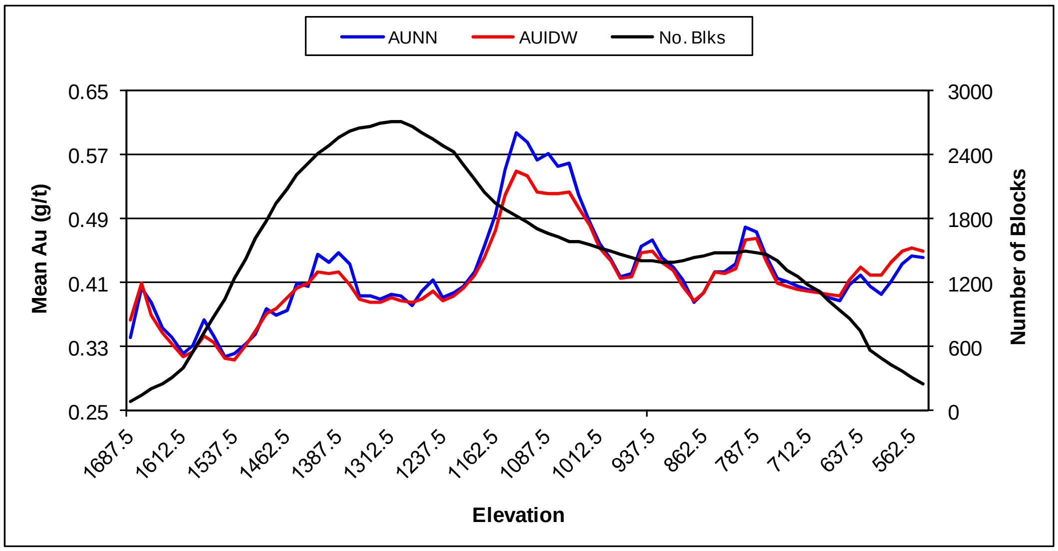
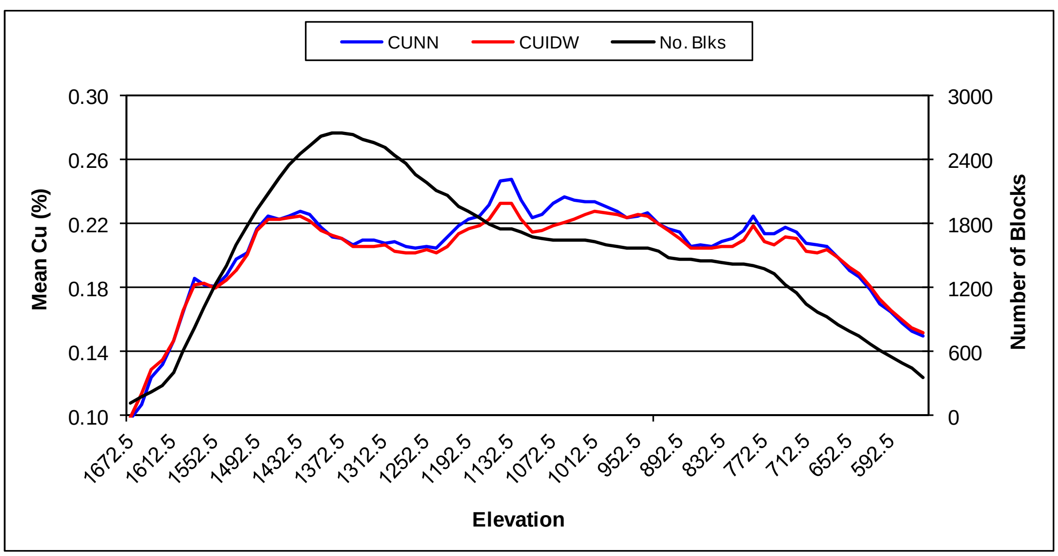


Figure 14.46 Iron Cap Copper Model Swath Plot – Elevations



The Iron Cap swath plots show several localized areas where the NN gold and copper grades are higher than the IDW grade. These areas represent volumes where a single composites used in the NN method overly influence the local grade. In the opinion of the QP responsible for this section, the estimated Iron Cap block grades are reasonable and unbiased based on visual and statistical reviews.

14.3.7 RESOURCE CLASSIFICATION – IRON CAP

Mineral Resources were assigned to the estimated blocks by constructing 3D solids for Indicated and Inferred Resources. These shapes were based on mineralized continuity as defined by exploration drill hole results. The 2012 Iron Cap Indicated Mineral Resource shape was locally deepened based on post-2012 drilling result. The Inferred Mineral Resource 3D solid was increased from the 2012 shape by virtue of deeper, relatively wide spaced drilling that confirmed the plunge of the recognized Iron Cap mineralized system. The average drill hole spacing for Indicated and Inferred blocks is approximately 45 m and 75 m, respectively.

14.4 SUMMARY OF MINERAL RESOURCES

A strategy for tabulating Mineral Resources for the Kerr, Sulphurets, Mitchell, and Iron Cap deposits was established by the QP responsible for this section after conferring with open pit and underground mining consultants that have examined potential mining methods that might be suitable for the various KSM deposits. Those engineers examined a number is “modifying factors” in their consideration of converting Mineral Resources to Mineral Reserves. (Canadian Institute of Mining, Metallugy and Petroleum [CIM] 2014).

The Kerr and Mitchell deposits are currently being planned to be mined by both open pit and underground methods. Because of that, the QP responsible for this section used a combination of conceptual put (LG) algorithms and underground draw point elevations that were provided by Golder (generated using GEOVIA’s PCBC™ Footprint Finder software) to define open pit and underground Mineral Resources for the Kerr and Mitchell deposits.

The Sulphurets Mineral Resources is defined soley by a conceptual pit, while the Iron Cap Mineral Resource is based on three conceptual block cave draw point elevations provided by Golder.

The Mineral Resources for all four deposits (Table 14.39) are based on applying similar mining constraints that were used to define Mineral Reserves (Section 15.0). Trade-off studies between open pit and underground mining (Kerr and Mitchell) optimized the location of the ultimate pit and block cave outlines for Mineral Reserve declaration. No such optimization was undertaken for defining open pit and underground Mineral Resources. However, a mining restriction surface was used to limit the depth of the conceptual Mineral Resource pits (Kerr and Mitchell) in order to leave a reasonable quantity of potential underground material for possible Mineral Resource declaration. Because a non-optimized boundary was used to limit the depth of the Mineral Resource pits in lieu of actual economic trade-off studies, the amount of underground Mineral

Resource (as defined by GEOVIA's PCBC™ Footprint Finder software) is less than the underground Mineral Reserve (Mitchell only).

The Mineral Resource pit is significantly larger than the Mineral Reserve pit, which left less material to be included in the underground Mineral Resource category. The QP responsible for this section of this Technical Report notes that the entire open pit and underground Mineral Reserve volumes fit wholly within the open pit and underground Mineral Resource volumes despite the apparent differences in quantities by mining methods.

Block revenue for determining the limits of the conceptual pit and block cave shapes was based on a NSR value that was calculated and stored in the block models. The calculated NSR value represents recoverable value in Canadian dollars less various off site transportation and smelting charges. Metal prices and other key criteria that were used in generating the conceptual pits and block caves are summarized in Table 14.38.

Table 14.38 Key Mineral Resource Parameters

Parameter	Units	Value
Au Price	US$/oz	1,300.00
Cu Price	US$/lb	3.00
Ag Price	US$/oz	20.00
Mo Price	US$/lb	9.70
Conceptual Pit Mining Cost	Cdn$/t	1.80
Conceptual Pit Processing + G&A Cost	Cdn$/t	9.00
Conceptual Pit Slope Angle	degrees	45
Conceptual Block Cave Mining Cost	Cdn$/t	6.00-7.00
Conceptual Block Cave Processing + G&A Cost	Cdn$/t	9.00
Conceptual Pit Cut-off Grade	Cdn$/t	9.00
Conceptual Block Cave Cut-off Grade	Cdn$/t	16.00

Metal recovery was based on metallurgical test work that has been completed by Tetra Tech on various KSM samples. Recovery is variable by mineralized area and specific grade ranges (recovery curves); for more detailed discussions regarding metal recovery refer to Sections 13.0 and 15.0 of this Technical Report. Mr. Tracey Meintjes, P.Eng., from MMTS wrote the MineSight® NSR calculation scripts that were used by the QP responsible for this section.

The QP responsible for this section of this Technical Report generated conceptual resource pits for the Kerr, Sulphurets, and Mitchell deposits using a Lerchs-Grossmann algorithm within the MSOP module within MineSight®. A mining restriction surface was used to limit deep conceptual pit mining for the Kerr and Michell deposits.

The conceptual block cave shapes that define resources for the Kerr, Michell, and Iron Cap deposits are based on work completed by Golder under the direction of Mr. Ross Hammett, P. Eng. Golder used GEOVIA's PCBC™ Footprint finder in generating optimized extraction levels from which 3D solids were created for tabulating Mineral Resources that show reasonable prospects for eventual economic extraction.

For the Kerr model, Golder established three conceptual resource block cave extraction levels at the 775 m, 280 m, and -215 m levels. A single conceptual resource block cave extraction level was established at the 60 m level for the Mitchell deposit. Three conceptual resource extraction levels were established for the Iron Cap model (1,335 m, 1,020 m, and 645 m levels). The extraction level polygons generated by Golder were extruded vertically approximately 500 m, representing a maximum height of draw. The upper Kerr conceptual block cave was clipped against RMI’s conceptual resource pit. The single conceptual Mitchell block cave was also clipped against RMI's resource pit.

The 2012 PFS block model was used for tabulating constrained Mineral Resources for the Sulphurets and Mitchell deposits (both conceptual pit and block cave quantities). The end-of-year 2014 and end-of-year 2015 models were used to tabulate constrained Mineral Resources (both conceptual pit and block cave) for the Iron Cap and Kerr deposits, respectively.

Table 14.39 summarizes Mineral Resources by resource category and further broken down by mineralized zone.

14.5 GENERAL DISCUSSION

The QP responsible for this section of this Technical Report is not aware of any specific environmental, permitting, legal, taxation, socio-economic, marketing, political or other relevant factors that could materially affect the Mineral Resource estimates that are the subject of this section.

Table 14.39 KSM Mineral Resources

Zone	Type of Constraint	NSR Cut-off (Cdn$/t)	Tonnes (000 t)	Au (g/t)	Au (000 oz)	Cu (%)	Cu (Mlb)	Ag (g/t)	Ag (000 oz)	Mo (ppm)	Mo (Mlb)
Measured Mineral Resources											
Mitchell	Conceptual LG Pit	9	698,800	0.63	14,154	0.17	2,618	3.1	69,647	59	91
	Conceptual Block Cave	16	51,300	0.59	973	0.20	226	4.7	7,752	41	5
	Total Mitchell Measured	n/a	750,100	0.63	15,127	0.17	2,844	3.2	77,399	58	96
Total KSM Measured	**n/a**	**n/a**	**750,100**	**0.63**	**15,127**	**0.17**	**2,844**	**3.2**	**77,399**	**58**	**96**
Indicated Mineral Resources											
Kerr	Conceptual LG Pit	9	355,000	0.22	2,511	0.41	3,208	1.1	12,555	4	3
	Conceptual Block Cave	16	24,400	0.24	188	0.48	258	2.0	1,569	14	1
	Total Kerr Indicated	n/a	379,400	0.22	2,699	0.41	3,466	1.2	14,124	5	4
Sulphurets	Conceptual LG Pit	9	381,600	0.58	7,116	0.21	1,766	0.8	9,815	48	40
Mitchell	Conceptual LG Pit	9	919,900	0.57	16,858	0.16	3,244	2.8	82,811	61	124
	Conceptual Block Cave	16	124,700	0.58	2,325	0.20	550	4.7	18,843	38	10
	Total Mitchell Indicated	n/a	1,044,600	0.57	19,183	0.16	3,794	3.0	101,654	58	134
Iron Cap	Conceptual Block Cave	16	346,800	0.51	5,686	0.23	1,758	4.5	50,174	14	11
Total KSM Indicated	**n/a**	**n/a**	**2,152,400**	**0.50**	**34,684**	**0.23**	**10,784**	**2.5**	**175,767**	**40**	**189**
Measured + Indicated Mineral Resources											
Kerr	Conceptual LG Pit	9	355,000	0.22	2,511	0.41	3,208	1.1	12,555	4	3
	Conceptual Block Cave	16	24,400	0.24	188	0.48	258	2.0	1,569	14	1
	Total Kerr M+I	n/a	379,400	0.22	2,699	0.41	3,466	1.2	14,124	5	4
Sulphurets	Conceptual LG Pit	9	381,600	0.58	7,116	0.21	1,766	0.8	9,815	48	40
Mitchell	Conceptual LG Pit	9	1,618,700	0.60	31,012	0.16	5,862	2.9	152,458	60	215
	Conceptual Block Cave	16	176,000	0.58	3,298	0.20	776	4.7	26,595	39	15
	Total Mitchell M+I	n/a	1,794,700	0.60	34,310	0.16	6,638	3.1	179,053	58	230
Iron Cap	Conceptual Block Cave	16	346,800	0.51	5,686	0.23	1,758	4.5	50,174	14	11
Total KSM Measured + Indicated	**n/a**	**n/a**	**2,902,500**	**0.54**	**49,811**	**0.21**	**13,628**	**2.7**	**253,166**	**44**	**285**
Inferred Mineral Resources											
Kerr	Conceptual LG Pit	9	80,200	0.27	696	0.21	371	1.1	2,836	6	1
	Conceptual Block Cave	16	1,609,000	0.31	16,036	0.43	15,249	1.8	93,115	25	89
	Total Kerr Inferred	n/a	1,689,200	0.31	16,732	0.42	15,620	1.8	95,951	24	90
Sulphurets	Conceptual LG Pit	9	182,300	0.46	2,696	0.14	563	1.3	7,619	28	11
Mitchell	Conceptual LG Pit	9	317,900	0.37	3,782	0.09	631	3.0	30,662	56	39
	Conceptual Block Cave	16	160,500	0.51	2,632	0.17	601	3.5	18,061	44	16
	Total Mitchell Inferred	n/a	478,400	0.38	6,414	0.10	1,232	3.0	48,723	55	55
Iron Cap	Conceptual Block Cave	16	369,300	0.42	4,987	0.22	1,791	2.2	26,121	21	17
Total KSM Inferred	**n/a**	**n/a**	**2,719,200**	**0.35**	**30,829**	**0.32**	**19,206**	**2.0**	**178,414**	**29**	**173**

Note: Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration. The Mineral Resources tabulated in Table 14.39 are inclusive of Mineral Reserves.

15.0 MINERAL RESERVE ESTIMATES

15.1 INTRODUCTION

Mineral Reserves are based on Measured and Indicated Resources, and use PFS-level engineering designs.

15.2 OPEN PIT RESERVE PARAMETERS

Open pit Mineral Reserves use whole block grades. Open pit mining loss and dilution assumptions are shown in Table 15.1 and Table 15.2. The derivation of loss and dilution assumptions are described in Section 16.0.

Table 15.1 Pit Mining Loss and Dilution

Pit	Total Loss (%)	Dilution (%)
Mitchell	2.2	0.8
Sulphurets	5.3	3.9
Kerr	4.5	3.2

The dilution tonnes are added as a percentage of ore tonnes from Table 15.1 at the average grade of mineralized material within the pits that is below cut-off as presented in Table 15.2.

Table 15.2 Grade of Dilution Material by Pit Area

	Mitchell	Kerr	Sulphurets
Au (g/t)	0.21	0.12	0.26
Cu (%)	0.04	0.08	0.04
Ag (g/t)	1.52	0.64	0.70
Mo (ppm)	52.00	2.50	16.00
NSR (Cdn$/t)	6.70	4.70	6.20

The open pit minimum NSR cut-off grade is based on an estimated process operating cost of Cdn$9.00/t. Process operating costs include plant processing (including crushing/conveying costs where applicable), G&A, surface service, tailing construction, and water treatment costs. The NSR grade used for mine planning is described in Section 16.0. A variable cut-off grade strategy has been used; a higher cut-off grade of Cdn$20.00/t is used until the end of Year 5 to maximize the NPV. During this time material between Cdn$9.00/t and Cdn$20.00/t is stockpiled and some of it is reclaimed through the mine life at the average grade of the stockpile. The premium cut-off grade in the early years of the mine schedule assists in minimizing the initial capital payback time. The cut-off grade by mine area is as follows:

- Mitchell Open Pit NSR Cut-off Grade – Cdn$9.00/t to Cdn$20.00/t
- Sulphurets Open Pit NSR Cut-off Grade – Cdn$9.00/t to Cdn$20.00/t
- Kerr Open Pit NSR Cut-off Grade – Cdn$9.00/t.

15.3 UNDERGROUND MINING RESERVE PARAMETERS

The underground Mineral Reserves have been determined using block grades from the Mineral Resource model with mining dilution and losses being determined as an integral part of the caving mining analysis undertaken using GEOVIA's PCBC™ software. The NSR grade used to determine value of the mineralized rock mucked from the drawpoints is described in Section 16.0. The site operating costs (mining and process) used in the analysis are presented in Table 15.4. The first part of the analysis determined the elevation of the production level and the shape of the production footprint at which the net value (NSR less site operating cost) of the mineralized rock to be mucked was a maximum.

The second part of the analysis determined a production and grade schedule based on mineralized rock mucked at the drawpoints that had a net positive value. If the net value during the mucking process is negative at any stage of the mining process, it is "shut-off". Both the first and second parts of the analysis incorporate rock that is mucked as diluted, and the shutting-off of uneconomic drawpoints results in losses of resources. Dilution includes Mineral Resources that have grade but are sub-economic (less than drawpoint shut-off). Inferred Mineral Resources and non-mineralized rock are assumed to have zero grade. Underground mining dilution estimates from the PCBC™ assessments for the Mitchell and Iron Cap deposits are shown in Table 15.3.

Table 15.3 Underground Mining Dilution

Deposit	Sub-economic Dilution (%)	Zero Grade Dilution (%)	Total (%)
Mitchell	2	13	15
Iron Cap	11	9	20

Drawpoint shut-offs (where the material being mucked becomes uneconomic) use the site operating cost (mining and process) shown in Table 15.4.

Table 15.4 Site Operating Cost – Drawpoint Shut-off

	Mitchell (Cdn$)	Iron Cap (Cdn$)
Undereground Mining	6.00	7.00
Process	9.00	9.00
Total	**15.00**	**16.00**

Process operating costs presented in Table 15.4 include plant processing (including crushing/conveying costs where applicable), G&A, surface service, tailing construction, and water treatment costs.

15.4 RESERVES

Proven and Probable Mineral Reserves are summarized in Table 15.5 and match the production plan described in Section 16.0.

Table 15.5 Proven and Probable Reserves

			Diluted Grades				Contained Metal			
		Ore (Mt)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)	Au (Moz)	Cu (Mlb)	Ag (Moz)	Mo (Mlb)
Proven	Mitchell Open Pit	460	0.68	0.17	3.1	59.2	10.1	1,767	45	60
	Kerr Open Pit	0	0.00	0.00	0.0	0.0	0.0	0	0	0
	Sulphurets Open Pit	0	0.00	0.00	0.0	0.0	0.0	0	0	0
	Mitchell Underground	0	0.00	0.00	0.0	0.0	0.0	0	0	0
	Iron Cap Underground	0	0.00	0.00	0.0	0.0	0.0	0	0	0
	Total Proven	**460**	**0.68**	**0.17**	**3.1**	**59.2**	**10.1**	**1,767**	**45**	**60**
Probable	Mitchell Open Pit	481	0.63	0.16	2.9	65.8	9.7	1,677	44	70
	Kerr Open Pit	276	0.22	0.43	1.0	3.4	2.0	2,586	9	2
	Sulphurets Open Pit	304	0.59	0.22	0.8	51.6	5.8	1,495	8	35
	Mitchell Underground	453	0.53	0.17	3.5	33.6	7.7	1,648	51	34
	Iron Cap Underground	224	0.49	0.20	3.6	13.0	3.5	983	26	6
	Total Probable	**1,738**	**0.51**	**0.22**	**2.5**	**38.2**	**28.7**	**8,388**	**138**	**147**
Proven + Probable	Mitchell Open Pit	941	0.65	0.17	3.0	62.6	19.8	3,444	89	130
	Kerr Open Pit	276	0.22	0.43	1.0	3.4	2.0	2,586	9	2
	Sulphurets Open Pit	304	0.59	0.22	0.8	51.6	5.8	1,495	8	35
	Mitchell Underground	453	0.53	0.17	3.5	33.6	7.7	1,648	51	34
	Iron Cap Underground	224	0.49	0.20	3.6	13.0	3.5	983	26	6
	Total Proven + Probable	**2,198**	**0.55**	**0.21**	**2.6**	**42.6**	**38.8**	**10,155**	**183**	**207**

Note: The Mineral Reserves shown in Table 15.5 are included in the Mineral Resources listed in this report. All Mineral Reserves stated above account for mining loss and dilution.

16.0 MINING METHODS

16.1 INTRODUCTION

A PFS-level production schedule, based on an annualized average 130,000 t/d mill feed rate, has been developed for the Project based on a combined open pit and underground mine plan. Pit phases at Mitchell, Kerr, and Sulphurets deposits are engineered based on the results of an updated economic pit limit analysis. Underground mining has been adopted at Iron Cap and below the Mitchell open pit to reduce the volume of waste generated from the mine.

16.1.1 PRODUCTION RATE CONSIDERATION

The 2012 PFS (Tetra Tech 2012) restricted the plant throughput rate at KSM to 130,000 t/d due to an anticipated power supply limitation. This power limitation has been lifted and a higher KSM mill throughput is therefore possible.

This 2016 PFS is still based on open pit and underground mine plans, to combine to an annual throughput of 130,000 t/d. The throughput that was assessed and approved during the recently completed EA review process. Alternative studies indicate the Project NPV may be improved by increasing the mill throughput above 130,000 t/d early in the mine life.

The entire open pit and underground mining operation results in a Project mine life of approximately 53 years.

16.2 OPEN PIT MINING OPERATIONS

16.2.1 INTRODUCTION

The open pit mine planning work for this study is based on previous work included in the 2012 PFS (Tetra Tech 2012) and updated with design criteria from the Application/EIS (Rescan 2013). The 3DBM discussed in Section 14.0 has also been used to update the ore tonnes and grades and waste rock characterization, with the most recent drill hole information.

In addition to the geological information used for the block model, other data used for mine planning include the base economic parameters (metal prices, etc.), 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.

16.2.2 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. Topography is described in Section 12.1.5.

16.2.3 Open Pit Production Rate Considerations

The ore production rate is maintained at 130,000 t/d up to Year 35. At the introduction of underground mining at Mitchell in Year 23, and continuing through Iron Cap ore production commencing in Year 32, the open pit production is adjusted so that the combined open pit and underground production matches the maximum mill throughput. The underground mine plans are described in detail in Section 16.3.

After Year 35 the underground production becomes the base production plan with reduced mill throughput to conform to the release of ore from the block caves. The open pit mine plan is further adjusted to provide a uniform feed tonnage at the reduced rates.

16.2.4 Open Pit Mine Planning 3D Block Model

The Mineral Resource models used in this study are based on the updated MineSight 3DBMs provided by RMI, as described in Section 14.0.

The block heights represent a suitable bench height for large scale mining shovels, and the block dimension are suitably sized for long-range planning.

Net Smelter Return

NSR (net of off-site concentrate treatment and smelter charges and including on-site mill recovery) is estimated for each block and is used as a cut-off item for break-even ore/waste selection, as well as for the grade bins used to optimize cash flow in the open pit production scheduling. It is also used for the underground mine planning as described in Section 16.3.

NSR is estimated using net smelter price (NSP) and process recovery as shown in the equation below. The NSP is based on base case metal prices; US dollar exchange rate; and off-site losses, transportation, smelting, and refining charges. The terms of a project smelter schedule will be negotiated during the course of the Project's development. The major smelter terms used to estimate NSP are specified in Table 16.1, not including minor terms for deductions/losses, payables, price participation, etc.

$$NSR = \frac{Cu}{100} \times \frac{RecCu}{100} \times NSPCu \times 2{,}204.6 + Au \times \frac{RecAu}{100} \times NSPAu + Ag \times \frac{RecAg}{100} \times NSPAg + \frac{Mo}{1 \times 10^6} \times \frac{RecMo}{100} \times NSPMo \times 2{,}204.6$$

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).

Table 16.1 Major Smelter Terms used in the NSR Calculation

	Amount	Unit
Copper Concentrate		
Smelting	75	US$/dmt
Au Refining	8,0	US$/oz
Ag Refining	0.5	US$/oz
Off-site Costs	236	Cdn$/wmt
Moly Concentrate		
Roasting	2.00	US$/lb
Other Off-site Costs	5,298	Cdn$/wmt
Gold Dore		
Au Refining + Transport	2.00	US$/oz

Copper to gold ratio in mill feed and estimated concentrate grades vary by KSM mining area. Off-site costs and NSPs are therefore different for each mining area. The metal prices and resultant NSPs used at this early stage of the study, are shown by pit area in Table 16.2 and Table 16.3.

Table 16.2 Metal Prices for Reserve NSR Calculation

	Metal Price (US$)
Cu	2.70/lb
Au	1200/oz
Ag	17.5/oz
Mo	9.70/lb
Exchange Rate (US$:Cdn$)	0.83

Table 16.3 Estimated NSP by Mining Area

	Cu NSP (Cdn$/lb)	Au NSP (Cdn$/g)	Ag NSP (Cdn$/g)	Mo NSP (Cdn$/lb)
Mitchell	2.82	41.6	0.551	6.5
Kerr	2.68	40.7	0.529	6.5
Iron Cap	2.82	41.6	0.551	6.5
Sulphurets	2.82	41.6	0.551	6.5

Metallurgical recoveries used for the NSR calculation are based on test work conducted by G&T and evaluated by Tetra Tech, and are described in Section 13.0.

MINING LOSS AND DILUTION

The Project is a large gold-copper porphyry deposit and the orebody occurs relatively continuously within the cut-off grade 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, blast hole assays will be used to determine the waste/ore boundaries for material designations on the pit bench for daily operations.

The interpolation of the metal grades to the 3DBM averages the drill hole composites to a single value in the block 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 cut-off grade 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 Mineral 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 an ore 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 2016 PFS, MMTS has estimated a mining loss and dilution factor that varies by pit area. Mining loss and dilution assumptions by pit area are provided in Section 15.0.

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 less than the cut-off NSR. The dilution grades are shown in Section 15.0.

16.2.5 PIT SLOPE DESIGN ANGLES

Overview

BGC has provided open pit slope design parameters for the three proposed open pits of the KSM Project: Kerr, Sulphurets, and Mitchell. The design parameters are based on geotechnical site investigations, available local and regional geological data, and well-established geotechnical design methods used to estimate the Project design pit slope angles.

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.

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 will rival some currently operating, very mature pits elsewhere in the Americas.

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 the 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 quality data was collected to characterize the geological units of the study area and support PFS-level 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, intersecting the north and south slope of the Mitchell pit. 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. The inter-ramp height limits and geotechnical berms provide: flexibility in the mine plan to mitigate potential slope instability; access for slope monitoring installations; and working space for in-pit wells, drains, and other water management infrastructure. 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 Mitchell open pit slope designs are outlined in Table 16.4.

Table 16.4 Mitchell 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
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.
inter-ramp height (IRH); inter-ramp angle (IRA); bench height (Bh); bench angle (Ba); bench width (Bw)

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 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 16.5 outlines the Sulphurets open pit slope designs.

Table 16.5 Sulphurets 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 (°)	
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.

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 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 16.6.

Table 16.6 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.

Slope Design Implementation

Achieving the proposed design criteria will require depressurization of the pit walls 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, a 3.5 km adit and drainage gallery will be required for the Mitchell pit north wall, and approximately 876 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 depressurization measures is estimated to be approximately 12,600 m^3/d throughout the life of the pit.

The average annual groundwater extraction rate for the Kerr pit depressurization measures is estimated to be approximately 1,300 m^3/d. 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 110 km of horizontal drains will be required to aid in depressurization of the pit slopes over the life of the pit.

The average annual groundwater extraction rate for the Sulphurets pit depressurization measures is estimated to be 1,100 m^3/d. Approximately 34 vertical wells with a total drilling length of 6,800 m will be required throughout the life of the pit. In addition, it is estimated that approximately 187 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 depressurization 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 FS-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 will meet or exceed the size and complexity of those systems currently in operation at other large open pits elsewhere. 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 Project area has been recently de-glaciated and large scale slope deformation features have been identified in the Mitchell and Sulphurets valleys.

Nine large landslides in the study area will require management during construction and operations. Conceptual management plans detailing monitoring and mitigation measures were prepared as part of this study (Appendix F10). Of particular note with respect to the open pits are: the Snowfield Landslide situated on the south slope of the Mitchell Valley and east of the Mitchell open pit, and the Kerr landslide situated on the south slope of the Sulphurets Valley and below the elevation of the proposed Kerr open pit.

The overall landslide management plan for the Project uses a risk based approach to determine the level of monitoring required for each landslide. The management plan for the Snowfield Landslide is comprehensive due to its proximity to the Mitchell open pit. The plan includes surface and subsurface deformation monitoring, surface water management, pumping wells, and a depressurization adit.

16.2.6 Economic Pit Limits, Pit Designs

Pit Optimization Method

The economic pit limit is selected after evaluating LG pit cases conducted with MineSight MSEP.

The LG assessment is carried out by generating sets of LG pit shells by varying revenue assumptions to test the deposit's geometric/topographic and pit slope sensitivity.

The ultimate pit limit is typically 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.

LG Pit Assumptions

Inputs to the updated LG pit limit assessment shown in Table 16.7 are based on the 2012 PFS (Tetra Tech 2012).

Table 16.7 LG Pit Limit Primary Assumptions

Assumption	Value
Mining Cost	Cdn$1.90/t
Process, G&A, Site Services, Water Treatment	Cdn$9.00/t
Process Recoveries	See Section 17.0
Pit Slope Angle	Variable See Section 16.2.5
Metal Prices	See Table 16.2

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. Figure 16.1 to Figure 16.3 summarize the revenue sensitivity cases for the Mitchell, Sulphurets, and Kerr pits, respectively.

Potential economic pit limits in Figure 16.1 to Figure 16.3 are shown where inflection points occur as 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 pit area the inflection points represent potential economic pit limits and are selected for each pit area.

The selected economic pit limits for Mitchell and Kerr are smaller than the potential pit limit.

Seabridge's objective to reduce waste mined has been achieved by selecting a pit limit with 40% less mill feed than the potential economic pit limit. Waste mined in the selected pit limit is 3.0 Bt less than the potential economic pit limit.

Figure 16.1 Mitchell Sensitivity of Ore Tonnes to Pit Size

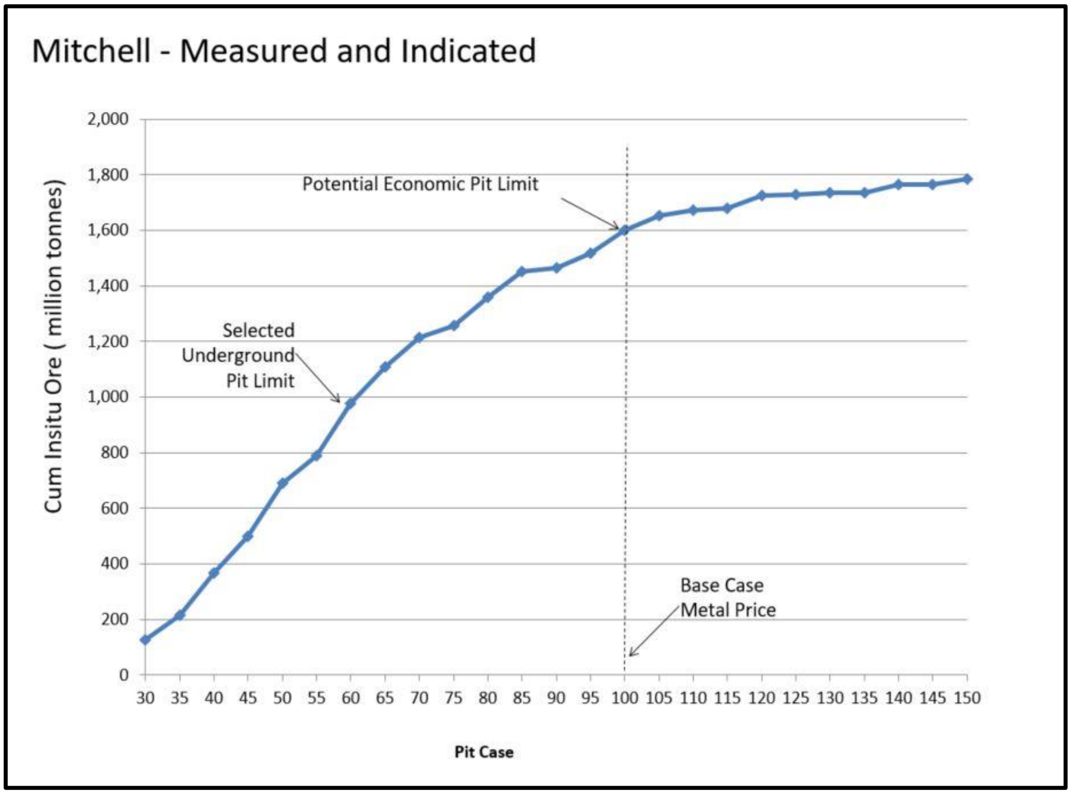


Figure 16.2 Sulphurets Sensitivity of Ore Tonnes to Pit Size

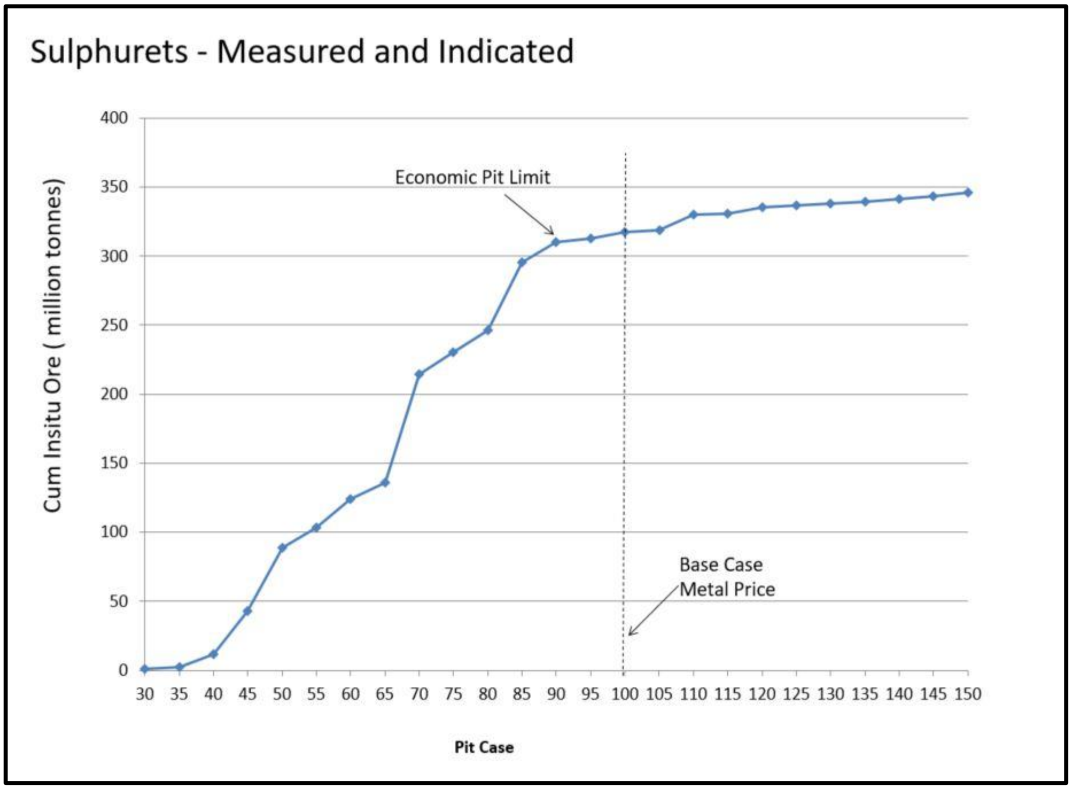
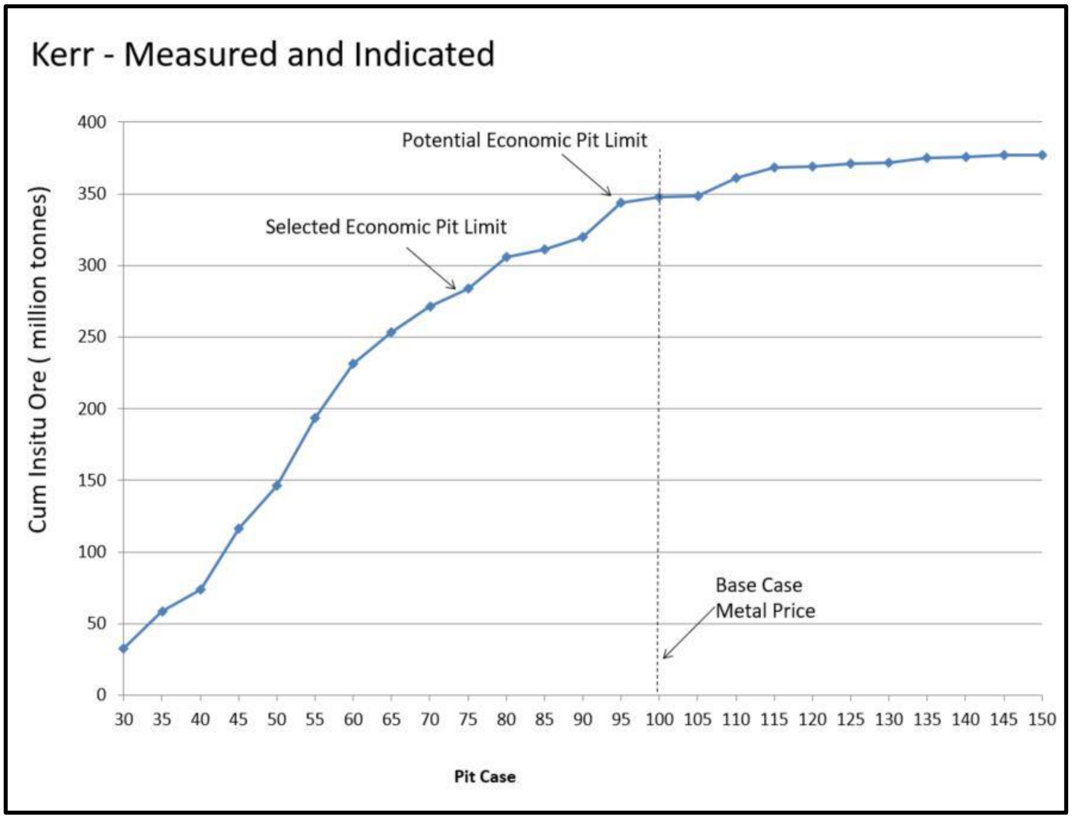


Figure 16.3 Kerr Sensitivity of Ore Tonnes to Pit Slope and Pit Size



The Kerr pit limit has been selected for a pit size similar to the one from the 2012 PSF (Tetra Tech 2012) to ensure that waste placement in the backfilled Sulphurets pit does not exceed the volumes as described in Rescan (2013).

The selected open pit limits are summarized below:

- Mitchell – open pit/underground: 60% Price Case
- Sulphurets – inflection pit case: 90 % Price Case
- Kerr – inflection pit case: 75 % Price Case

A plan view and north-south section views of the LG pits for the open pit mining areas are shown in Figure 16.4 through Figure 16.7.

Figure 16.4 Plan View of the KSM LG Pit Limits

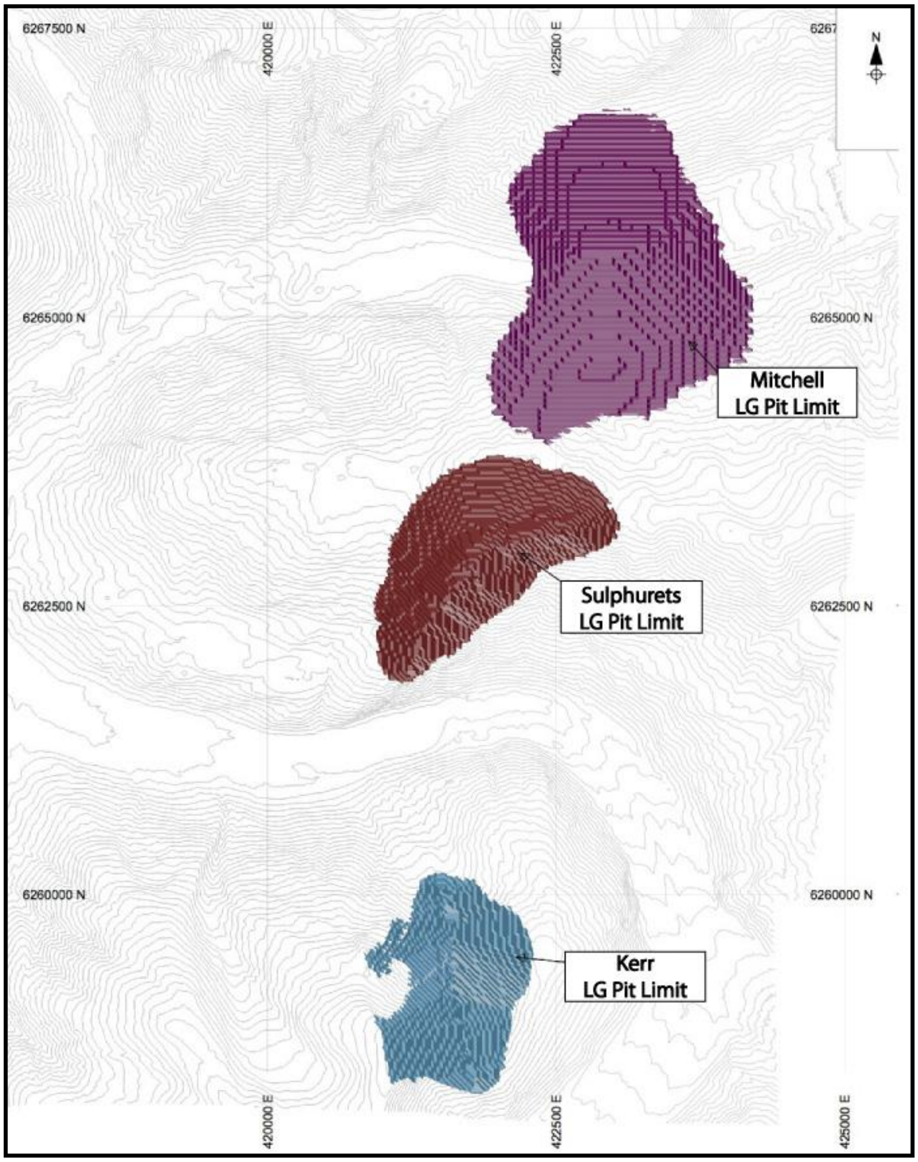


Figure 16.5 Mitchell Open/Underground Pit and Economic Pit Limit – North-South Section at East 422950, Viewed from the East

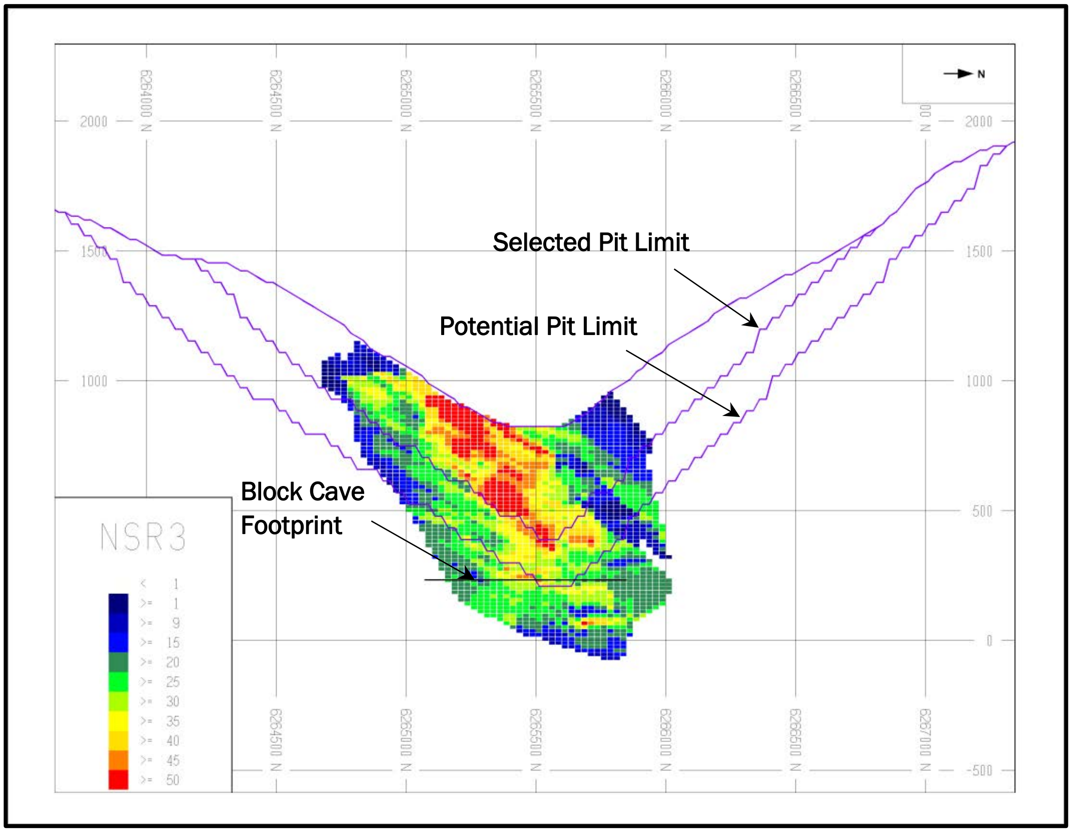


Figure 16.6 Sulphurets Economic Pit Limit – North-South Section at East 421725 Viewed from the East

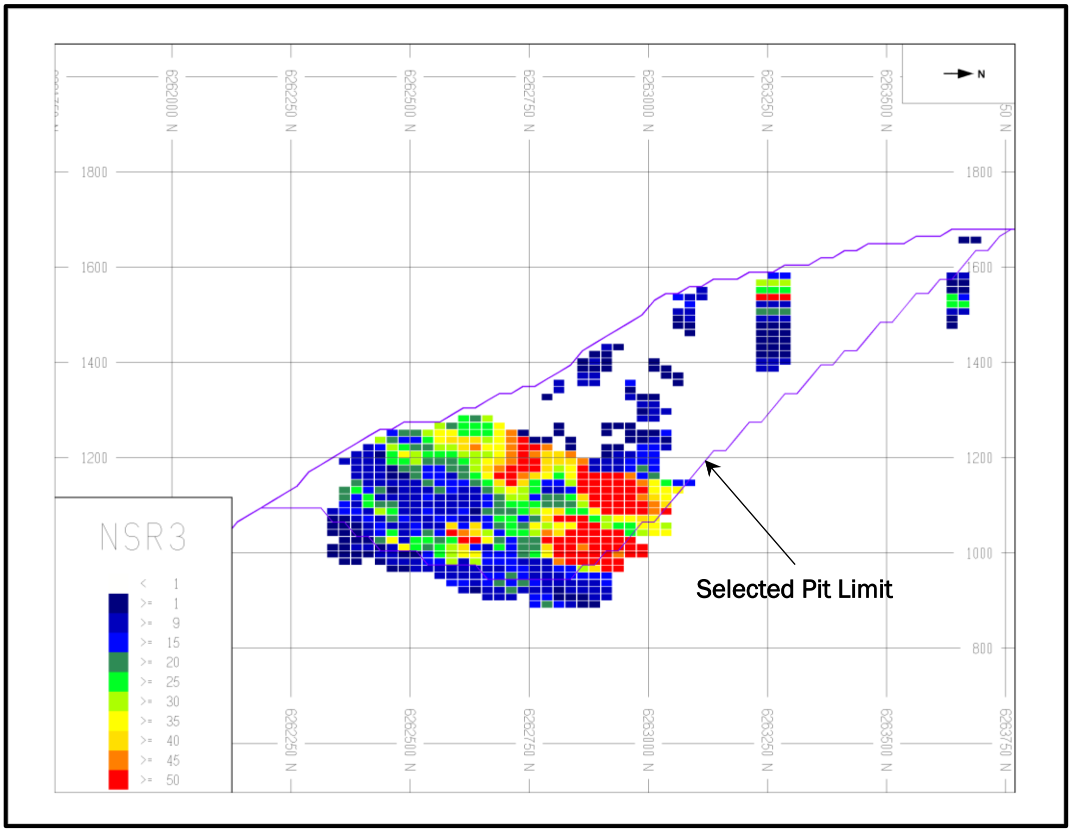
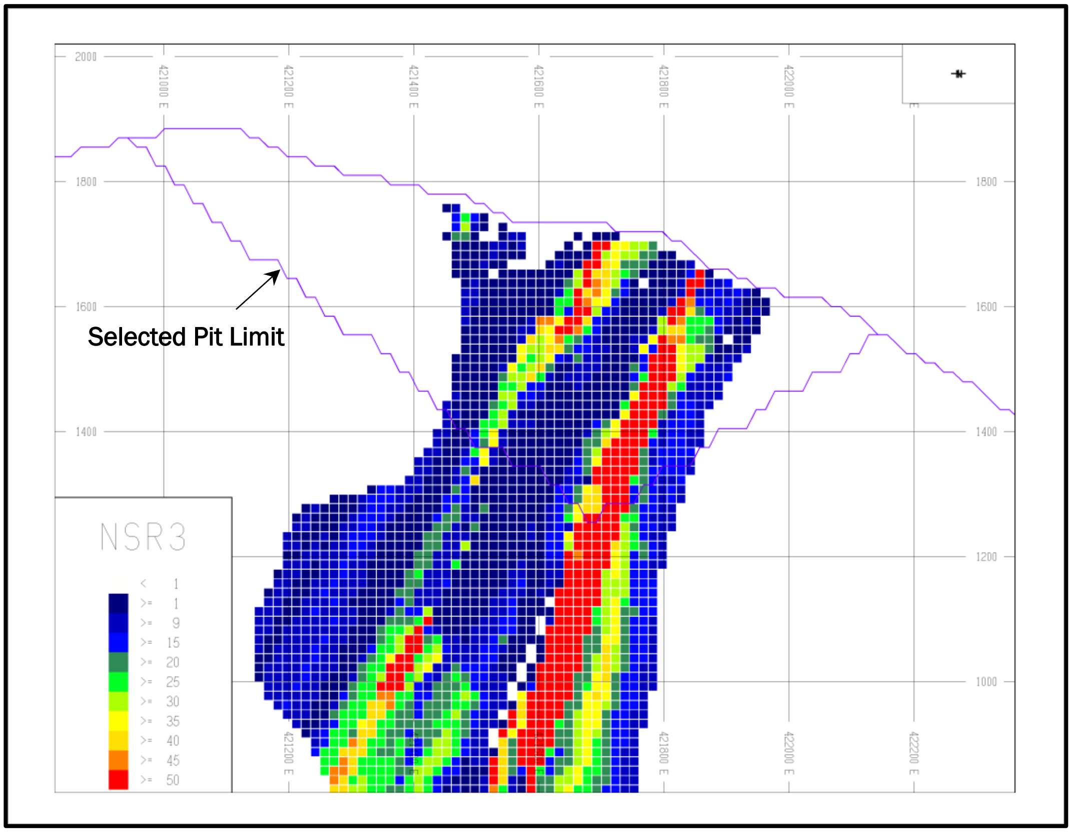


Figure 16.7 Kerr Economic Pit Limit – East-West Section at North 6258800, Viewed from the South



16.2.7 DETAILED PIT DESIGNS

PFS-level pit designs demonstrate the viability of accessing and open pit mining the economic resources at the KSM site. Pit designs use the selected LG pit limits as guides for estimated geotechnical parameters, suitable road widths, 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 high wall roads; there is adequate water drainage at the edge of the road between the crowned surface and lateral embankments, such as high walls or lateral impact berms.

Based on a 360-t truck, the haul road design basis is as follows:

- largest vehicle overall width: 9.8 m
- double lane high wall haul road allowance: 38.2 m
- double lane external haul road allowance: 47.2 m
- single lane high wall haul road allowance: 28.5 m
- single lane external haul road allowance: 37.4 m.

DESIGN STANDARDS

Detailed design parameters for pits and RSFs are provided by BGC and KCBL, respectively, according to their geotechnical testing and evaluations (Sections 16.2.5 and 18.1.6).

Minimum Mining Width

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 open pit 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 2016 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% due to traction concerns during snow season particularly with downhill hauls. 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, which in turn increases the overall strip ratio. 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 high wall 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 to be added are 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 are discussed above.

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.

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 for some requirements, 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. Some construction materials are sourced from quarries outside of the economic pit areas to ensure that construction rocks meet the required geochemical and mechanical properties.

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 digit signifies the pit phase number.
- A suffix of 'i' indicates that the reserve tonnage for the phase is incremental from the previous phase. If there is no 'i' specified, it is cumulative within the pit, 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.

Where possible, in order to balance the waste hauls and keep upper elevation waste going to upper elevation RSF platforms, the high wall waste is brought out of the pit using external side hill roads directly off the south benches.

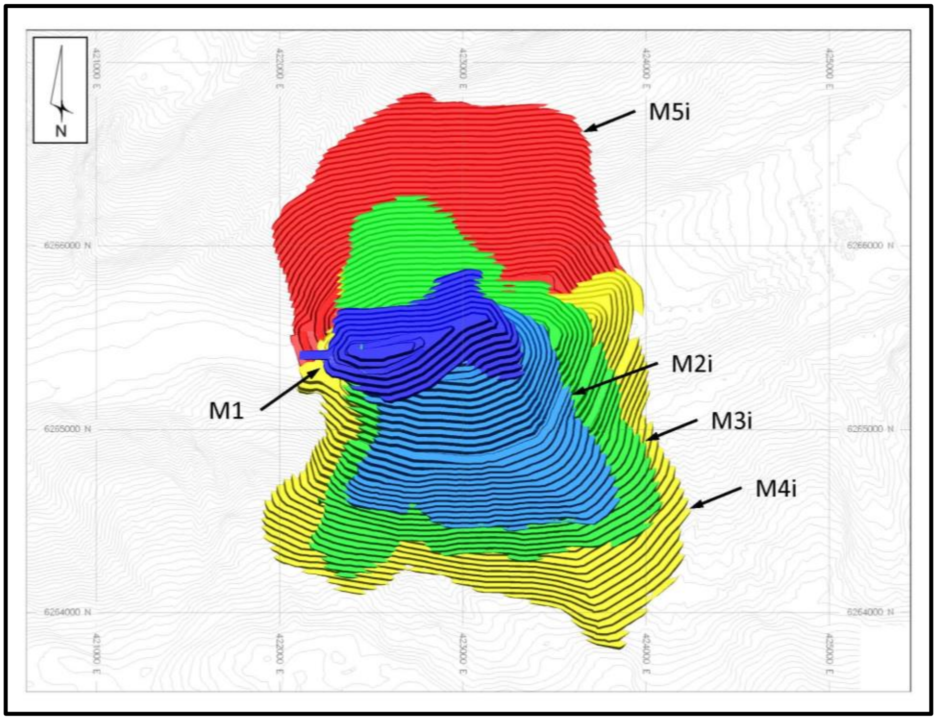
The Mitchell pit phases have been designed to mine vertically through the Snowfield Landslide on the southeast side of the pit and not undermine it.

Mitchell pit has five incremental phases. Pit phase M1 enables the mine to have sufficient exposed ore with a six month pre-strip period. Phases M2i and M3i mine south and north respectively to provide low strip ratio ore to the mill during pay-back period.

M4i and M5i are high strip ratio pit phases that mine to the ultimate pit limit in the south and then in the north.

A plan view of the Mitchell pit phases are shown in Figure 16.8.

Figure 16.8 Plan View of Mitchell Pit Phases



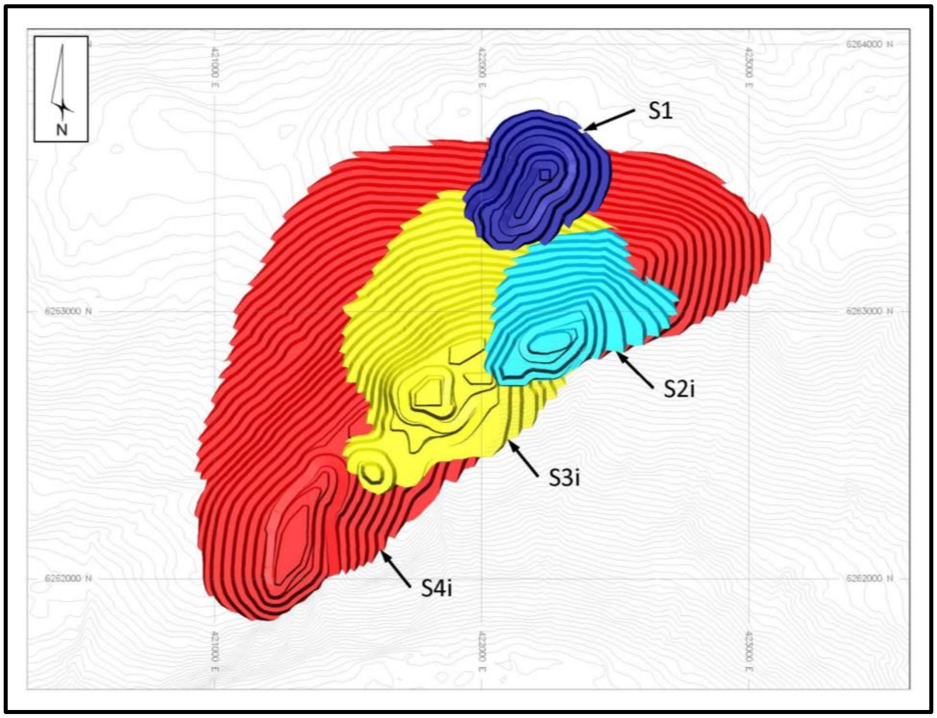
Sulphurets Pits

The mine plan for the Sulphurets area includes four mining phases, which are designed using the LG economic pit limit as the ultimate pit limit guide.

S1 is a quarry that provides non-potentially acid generating (NPAG) monzonite for construction of the WSD during the pre-production period. S2i and S3i are low strip ratio starter pits at Sulphurets. S4i is the final Sulphurets pushback.

A plan view of the Sulphurets pit phases are shown in Figure 16.9.

Figure 16.9 Plan View of Sulphurets Pit Phases

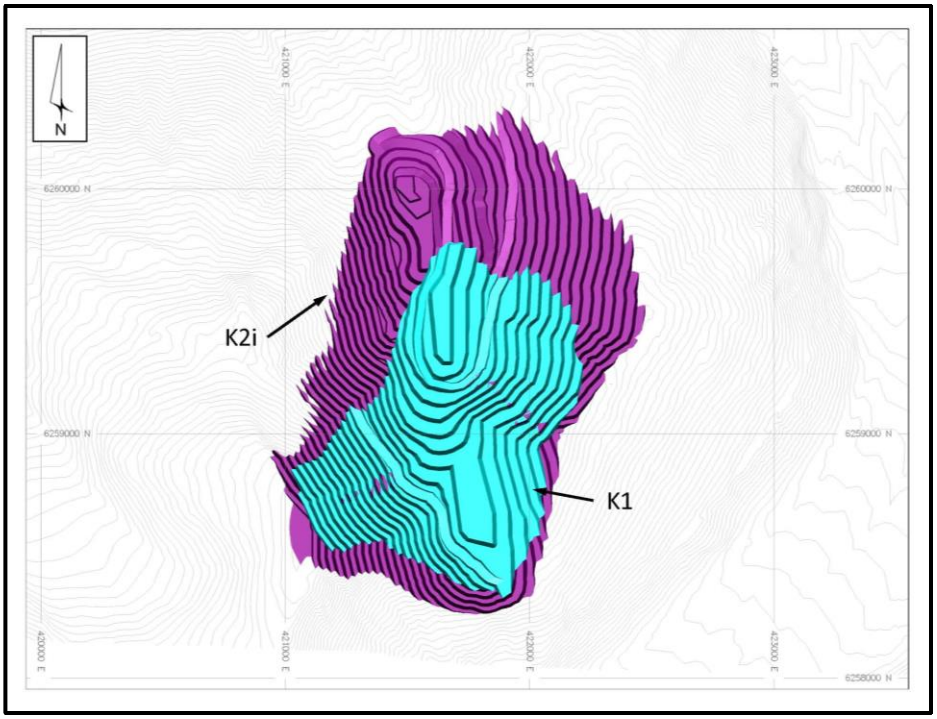


Kerr Pit

The Kerr deposit is mined with two pit phases: a starter pit K1 and an ultimate pit K2. All ore and waste is hauled to a primary crusher on the east side of the pit and conveyed to the Mitchell Valley using a rope conveyor, a tunnel conveyor (through the SMCT) to the OPC.

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 1,460 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 1,800 m where haul truck access can be established to the crusher. A plan view of the Kerr ultimate pit is shown in Figure 16.10.

Figure 16.10 Plan View of Kerr Pit Phases



16.2.8 OPEN PIT MINE PLAN

LOM OPEN PIT 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 open pit sequence is scheduled to optimize revenues and development costs and is then adjusted to include the block caves (see Section 16.3). The underground mining production schedule discussed in Section 16.3 is generated based on the development requirements for each mining area, 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 of the open pit and underground mining is inserted where it provides the best contribution to the Project economics. After inserting the underground ore production into the LOM sequence, the open pit ore targets are then adjusted to meet the mill capacity.

At start-up, all production comes from open pit sources, producing higher grades from lower cost areas (both operating and capital). In the later years of the schedule, the base ore production is from the underground, from Mitchell first and then Iron Cap is phased in. After Year 35 the mill throughput rate is reduced to match the switch to continuous underground production, and the open pits are mined to supplement the ore tonnes produced from the block caves to meet the mill requirements and to improve overall head grades. In the final years of the production schedule, the stockpile accumulated during the open pit operations is used to augment the underground production. The combined LOM schedule including open pit, underground, and stockpile reclaim, is presented in Section 16.4. The following describes the open pit sequencing to match the combined open pit underground mine plan.

In the open pit 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
- Operations begin in Year 1
- LOM operations: at full capacity Year 2 onward.

Details of the mine plan can be found in Appendix E1.

Open Pit Mine Load and Haul Fleet Selection

The mine load and haul fleet is selected prior to production scheduling. Previous studies and 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 360-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

In order to optimize the Project NPV, grade bins have been specified (based on NSR block values); the MS-SP optimizer develops a cut-off grade strategy to increase the Project NPV. This increases mill head grades and therefore revenues early in the production schedule. Seven grade bins have been used for the schedule optimization software to optimize the cut-off grade strategy. To achieve this in mine operations, it is planned to use an OCS based on blast hole assays, 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. Other pit/phase precedencies are determined by the timing of water diversions, bench access issues, and RSF phase sequencing. Early Sulphurets waste production is initially based on WSD construction requirements and later based on RSF rock drain requirements. Kerr pit is mined after Sulphurets pit is mined out so that Kerr waste can be backfilled to the Sulphurets pit.

Because of these complexities, each pit area is scheduled in MS-SP independently and then combined in a master LOM schedule

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. Key production schedule assumptions are shown in Table 16.8.

Table 16.8 Production Schedule Assumptions

Assumption	Value
Operating Days Per Year	355
Hours Per Day	21
Daily Mill Throughput	130,000 t/d
Haul Truck Speed Limit	50 km/h
Haul Truck Operator Efficiency	90%
Haul Truck Operating Efficiency	85%
Dump and manoeuvre time	1.5 min
Shovel Loading Time	35 s/pass
Shovel Spot and Wait Time	10 s
Shovel Operator Efficiency	84%
Shovel Operating Efficiency	85%

Allowance has been made for days where the cumulative effect of severe snow storms or poor visibility requires the mine to completely shut down.

Cut-off Grade Optimization

The pit phase designs and sequencing is typically from higher grades to lower, to mine the higher mill feed grade early in the schedule to and thereby increase the Project revenues early in the earlier years. This can be further enhanced by stockpiling low and mid-grade. 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 subsequently 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 cut-off grade strategy has been applied for the KSM production schedule to reduce the stockpiling, assist in haul fleet smoothing and maximizing NPV.

Rock Storage Facilities

Mined waste rock in the KSM mine plan is placed in RSFs in as close proximity to the mining areas as possible. Mitchell and Sulphurets waste is placed in Mitchell and McTagg RSFs. Kerr waste rock is backfilled into Sulphurets pit.

Further details on the RSF design are available in Section 18.0.

Construction Methods

Several different construction methods will be used for waste placement: top-down, bottom-up, and wraparounds. Top-down platform heights are restricted to approximately 300 m. Bottom-up lifts are 30 to 50 m high, or less if geotechnically required. Wraparounds are smaller top-down-type RSFs that are built onto the face of an existing RSF, creating a series of terraces 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

Design work for RSF foundation preparation will be performed as required at the feasibility-level design stage. Prior to mine development, soil will be salvaged from the footprint area where soil is suitable for reclamation purposes. Soils will generally not be salvaged on slopes steeper than 26° due to practical limitations on equipment access and operator safety. Soils salvaged from the RSF footprints will be stockpiled in the Ted Morris Valley.

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. Once the foundation is prepared, the basal drain is placed on top at the required lift height.

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 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 more efficient 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.

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. These are built for pit access and not for hauling. 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 high wall 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 RSFs for the Project will have overall slope angles of 26° to 30°. The final post closure configuration will be adapted in accordance with the closure plan as identified within Rescan (2013) and described in Section 20.7. 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.

Low-grade ROM Stockpile

Lower-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 not only increase the grade of the ore feed to the plant in the early years of the schedule and is also used to even out the waste mining requirements as required during periods of high pre-stripping for some of the pit phases. The low-grade ore stockpile is placed to the west of the Mitchell OPC. Provision has been made for an HDPE pipeline diversion around the surface of the stockpile, which can be moved as required.

OPEN PIT MINE PRE-PRODUCTION DETAIL

Pre-production Description

The open pit 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 in the mine area.

This section describes the development and pre-production activities that will be accomplished by the mine personnel and open pit mine fleet equipment, and are included as capitalized mining costs. 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 including water treatment
- 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 purpose of the collection ditch network is to collect and route water that comes into contact with the mining operation. This water is transported to the water storage dam, as necessary, where it will then be treated in the WTP prior to release to the environment. 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 18.2.7 (Mine Area Water Management).

Ore Haul Road Construction

A haul road is constructed from the first mining phase in the Mitchell and Sulpurets pits to the primary crusher during pre-production using mine waste rock.

Open Pit 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.

Open Pit Mine Infrastructure Construction

Site preparation is also included for:

- the mine equipment assembly 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 are 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.

Initial Pit Development

Once the pioneering roads are in place, the larger open pit mine equipment will have access to the working areas. Mining preproduction begins with the Sulphurets Quarry to provide construction rock for the construction of the WSD. Mining pre-production of the Mitchell and Sulphurets pits will only commence after the WSD, WTP and MDT are operational. 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 slope, or a shovel or loader will side cast over the bench crest to a lower bench elevation where the larger drill fleet and shovel-truck fleet can operate. The pioneering operations will create 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 will receive the highest priority.

Mining of quarry rock in the Sulphurets Quarry starts in Year -4 to produce construction rock for the WSD.

Waste pre-stripping pre-production at the Mitchell pit starts in Year -1 when the water storage and water treatment facilities are operational. Process start-up is scheduled for the beginning of Year 1.

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. Clearing and grubbing work for pre-stripping will take place in Year -2.

The site for mine equipment assembly 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 can be started in Year -4 and will be completed in Year 1, prior to blasting with emulsion (expected at the end of Year 1). Temporary explosives storage will be required for the pioneering pre-production stages of mine development.

The open pit 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 MTT tunnel is completed.

Before pre-production begins the large mining fleet will excavate colluvium from a borrow source in the Mitchell Valley to provide construction fill for the Mitchell OPC. During pre-production, Mitchell pit phase M1 is mined to 885 m and M2 is mined to 1290 m in the Mitchell Valley; Sulphurets pit phase S3 is mined to an elevation of 1485 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.

The mine layout at the end of pre-production is shown in Figure 16.11.

Open Pit Production

Year 1 to 20 – Open Pit Mining

The following is a summary of mining activity in Years 1 to 5:

- Mining in Year 1 to 5 focuses on delivering the grade required to payback initial capital.
- All Mitchell waste and Sulphurets waste material is placed in the Mitchell RSF.
- Sulphurets construction rock (NPAG monzonite) waste is hauled to be used for the basal and selenium drains beneath the Mitchell/McTagg RSF.
- Mitchell and Sulphurets ore is hauled directly to the Mitchell primary crushers. From Year 2 the Sulphurets ore is hauled to the Sulphurets crusher and crushed ore is then conveyed to the OPC through a tunnel.
- An ROM ore stockpile is built in the area to west of the Mitchell OPC.
- The Mitchell RSF is built in lifts at an overall slope of 2:1 with an access road in the final face.
- By Year 6 the M3 and S3 phases are mined out and mining begins on the higher strip ratio phases of Mitchell and Sulphurets.
- Stockpile material is reclaimed to supplement mill feed during periods where mining is limited by the large volume of waste pre-stripping or vertical advance rate.
- By Year 10 the Mitchell RSF is full and waste placement begins in the McTagg RSF.

- Mitchell and Sulphurets pits are mined out by the end of Year 20. All waste is placed on the McTagg RSF.

Year 20 to 30 – Open Pit production and Transition to Mitchell Underground Mining

The open pit mine production sequence is adjusted to meet plant feed requirements as the Mitchell block cave mine is brought online. As discussed in more detail in Section 16.3, the Mitchell block cave mine is ramped up to full production in this time interval.

Kerr open pit is mined out by Year 30. Waste from Kerr pit is conveyed to Sulphurets and backfilled into the mined out Sulphurets pit. Ore from Kerr is conveyed to the OPC.

Direct mining from the open pits is completed by the end of Year 30.

Year 30 to 53 – Stockpile Reclaim to Supplement Underground Mining

Open pit mining after Year 30 is limited to stockpile reclaim for supplementing Mitchell and Iron Cap block cave production (see Section 16.3).

Once the stockpile 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 open pit mine layout at LOM is shown in Figure 16.13.

Figure 16.11 End of Pre-production (Year -1)

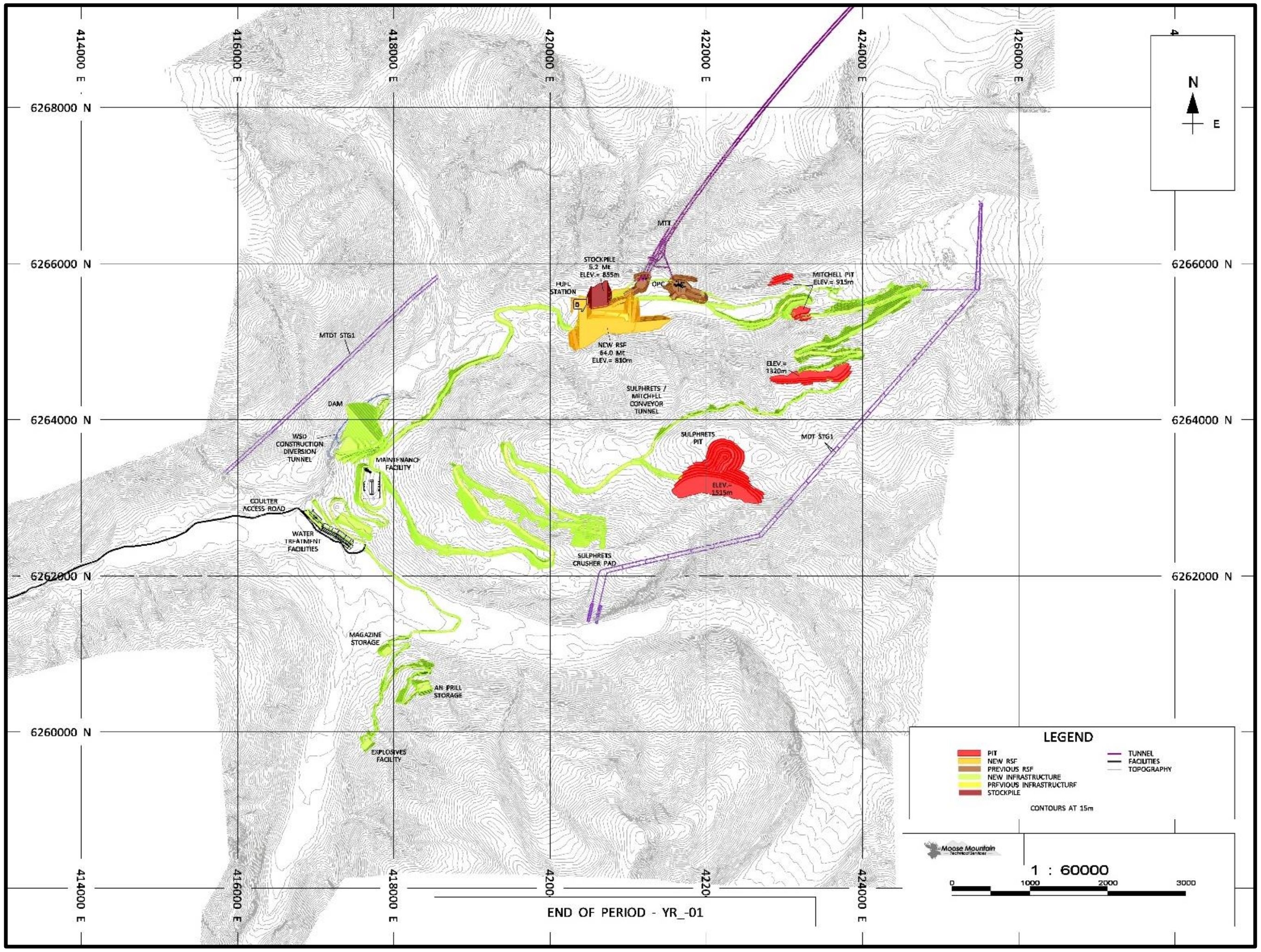


Figure 16.12 End of Year 5

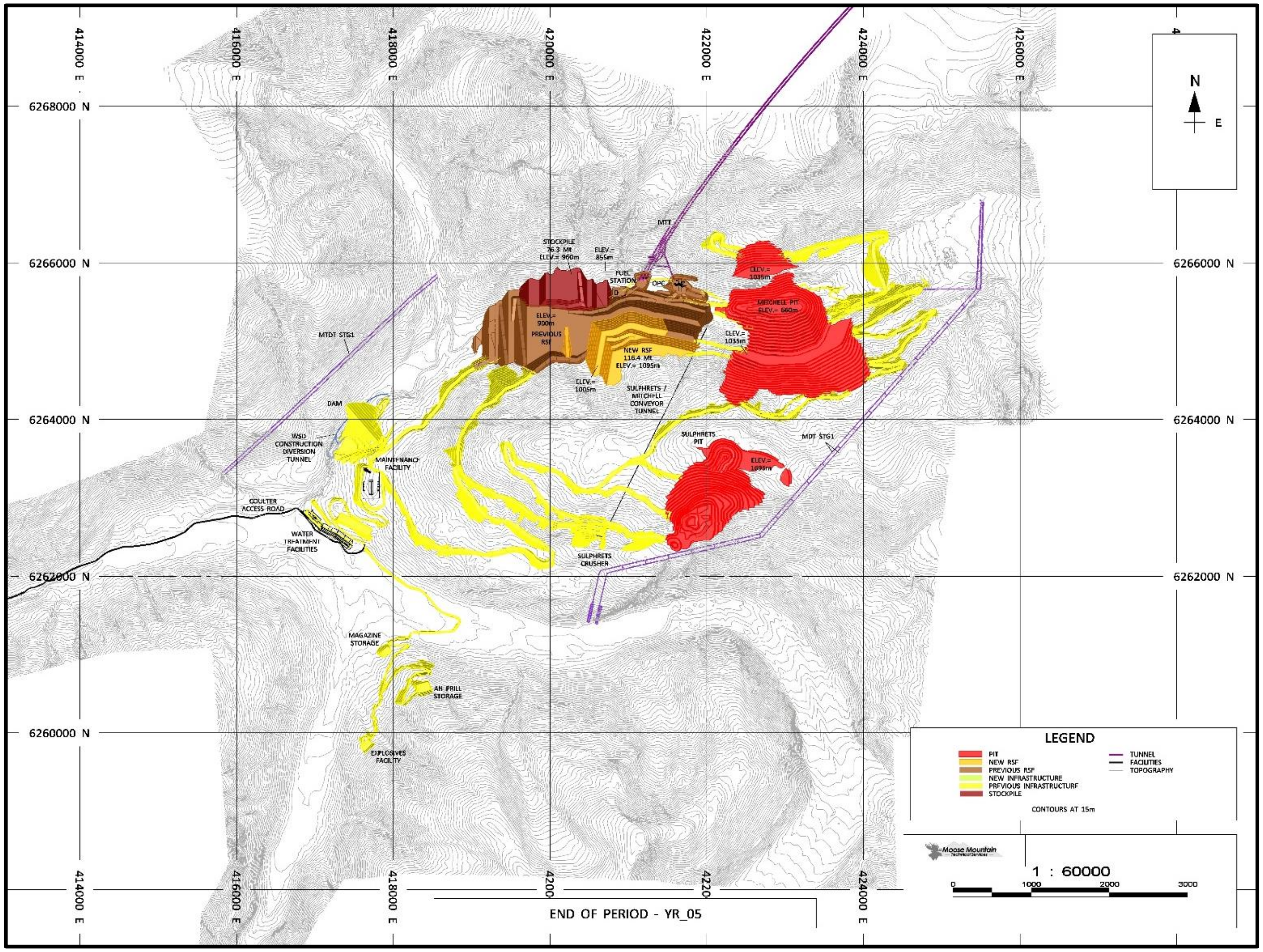
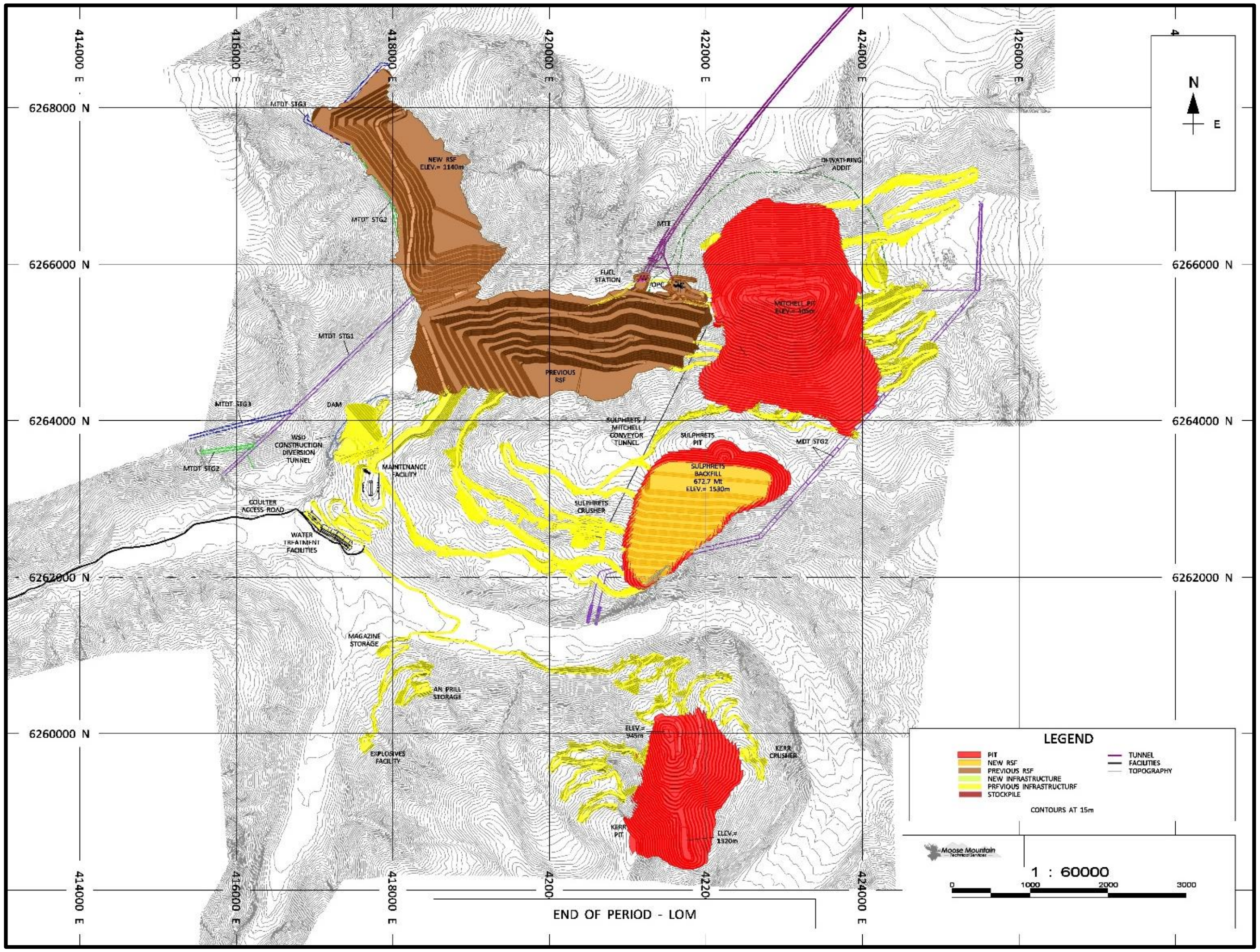


Figure 16.13 Open Pit Life of Mine



16.2.9 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 should be able to reduce costs by integrating some of the support services, staff and facilities. The following description is for the open pit operations. Underground operations details are provided in Section 16.3.

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 of a remote site, and will dilute the fixed overhead costs for the mine operations. Much of the general overhead for the mine operations can be minimized if the number of production fleet units and labour requirements are minimized.

Organization

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.

Direct Mining Activities – Open Pit

The direct mining area accounts for the drilling, blasting, loading, hauling, and pit maintenance activities in the mine.

Drilling

Areas will be prepared on the bench floor blast patterns in the in situ rock. Dozers will be used to establish initial benches for the upper portions of each pit phase. Drill ramps will be cut on original mountain side surfaces, 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.

Blasthole 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 grade interpolation 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 with a hole size of 6.5 inches (165 mm) will operate production benches 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. Similar large open pit projects in the KSM area use a powder factor of 0.32 kg/t for competent rock. A blasting study carried out by Orica suggests that a power factor of 0.35 kg/t is suitable in this area. Future Feasibility Study planning can investigate further mine to mill performance with respect to blasting.

Explosives

A contract explosives supplier will provide blasting materials and technology. Due to the remote nature of the operation, an explosives manufacturing plant will be built on site when emulsion is required. For this study, the owner provides a serviced site and all facilities to the explosives contractor who manufactures and delivers the prescribed explosives to the blast holes and supplies all blasting accessories.

It is anticipated that Production up to and including Year 1 will not require emulsion. After Year 1 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 adjacent to the mining areas.

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 BC MEMPR (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. The GPS guidance will be a necessity to be compatible with stakeless drilling.

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. This is a specific adaptation for open pit operations in mountainous and high snow fall areas. "Goat" trucks are similar to a logging skidder and are named because of their high manoeuvrability.

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.

Blasting Operations

The blasting crew will be comprised of mine employees and will be on day shift only. The blasting crew will coordinate drilling and blasting activities to ensure a minimum of two weeks of broken material inventory is maintained for each shovel. Blasting activities will need GPS control as blast patterns will not be staked. Blasters will require hand-held GPS to identify holes for pattern tie-in. A detonation system will be used that consists of electric cap initiation, detonating cord, surface delay connectors, non-electric single-delay caps, and boosters.

Blasting assumptions are summarized in Table 16.9. These parameters are typical for other mines in the Western Canada and will be re-evaluated in the future with a detailed blasting study, using site-specific rock strength parameters.

Table 16.9 Blasting Assumptions

Blasting Pattern – Ore and Waste	Specifications
Spacing	8.5 m
Burden	8.5 m
Hole Size	12¼ inches
	311 mm
Explosive In-Hole Density	1.25 g/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 by the OCS. A fleet management system will assist in optimizing deployment and utilization of the mine fleet

The design basis assumes a single model of each shovel type 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.

Bench widths are designed to ensure operating room is suitable for efficient double-sided loading of trucks at the shovels. 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. 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.

Hauling

Ore and waste will be hauled by 360-t off-highway haul trucks. Haul productivities have been estimated from pit centroids at each bench to designated dumping points for each time period.

Pit Maintenance

Pit maintenance services include haul road maintenance, open pit mine dewatering, transporting operating supplies, relocating equipment, and snow removal.

Haul road maintenance is paramount to low haulage costs; dozer and grader hours have been allocated 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 high wall 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. 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, personnel 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.

Open Pit General Mine Expense Area

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. Open pit 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.

Initial training and equipment operation will be provided by experienced operators as full time trainers. 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.

The geology department will include a senior geologist, pit geologists, and ore grade technicians. The geology department will also provide grade control support to mine operations, and will manage and execute the blast hole sampling and the short range grade models for operations planning and ore grade definition.

The geotechnical engineer will assume responsibility for all mine geotechnical issues including pit slope stability, RSF stability and hydro-geological studies. The geotechnical engineers will also have oversight for the whole property for any geo-hazard 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, and geo-hydrology expertise and third-party reviews in the open pit mine area.

16.2.10 Mine Closure and Reclamation

Details on mine closure and reclamation are available in Section 20.7

16.2.11 Open Pit Mine Equipment

Mining equipment descriptions in this section provide general specifications so that dimensions and capacities can be determined from vendor specification documents.

Major Equipment

The production requirements for the major mining equipment over the LOM are summarized in Table 16.10. The current production schedule requires a maximum haulage fleet of 60 trucks over the LOM.

Table 16.10 Major Equipment Requirements

	Pre-production	Year 5	Year 10	Year 20	Maximum
Drilling					
Primary Drill – 311 mm Electric Drill	0	4	5	5	5
Primary Drill – 311 mm Diesel Hydraulic Drill	2	3	3	1	3
High Wall Drill – 150 mm Diesel Hydraulic Drill	4	4	4	4	4
Loading					
Primary Shovel – 40 m^3 Diesel Hydraulic Shovel	1	2	3	3	3
Primary Shovel – 56 m^3 Electric Cable Shovel	0	3	3	3	3
Construction Shovel – 12 m^3	2	0	0	0	2
Hauling					
Haul Truck – 360 t	15	37	60	58	60
Construction Haul Truck – 90 t	14	0	0	0	14

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 16.11.

Table 16.11 Open Pit 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 16.2.9.

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 as identified in Appendix F8.

Open Pit Loading and Hauling Equipment

The shovel-truck fleet selected for KSM is the 56 m^3 dipper class of electric shovel, and the 360-t payload class of truck. A 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. Loading and hauling is discussed in Section 16.2.9.

Open Pit Dewatering Equipment

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 high wall
- water collection system.

Pit water will be collected and transported to the WSF.

OPEN PIT SUPPORT EQUIPMENT

The mine support equipment fleet requirements are summarized in Table 16.12. The fleet size in Year 5 and Year 10 is shown as representative of the LOM requirement.

Table 16.12 Mine Support Equipment Fleet

Fleet	Function	Year 5	Year 10
Hole Stemmer – 3 t	Blast Hole Stemmer	2	2
Track Dozer – 430 kW	Shovel Support	5	6
Rubber Tired Dozer – 350 kW	Pit Clean Up	2	3
Fuel/Lube Truck	Shovel and Drill Fuelling and Lube	2	3
Wheel Loader Multipurpose – 14 t	Pit Clean Up	2	3
Water Truck – 20,000 gal	Haul Roads Water Truck	2	2
Track Dozer – 430 kW	Dump Maintenance	3	3
Motor Grader – 400 kW	Road Grading	4	4
Tire Manipulator	Tire Changes	3	3

OPEN PIT ANCILLARY EQUIPMENT

The mine ancillary equipment fleet is listed in Table 16.13. The fleet sizes in Year 5 and Year 10 are shown as representative of the LOM requirement.

Table 16.13 Open Pit Ancillary Equipment Fleet

Fleet	Function	Year 5	Year 10
Track Dozer – 430 kW	Pit Support	2	2
Float Tractor/Trailer – 189 t	Float Tractor and Trailer	1	1
Hydraulic Excavator – 6 t	Utility Excavator	2	2
Sump Pump - 1,400 gal/min	Pit Sump Dewatering	6	6
Light Plant	Lighting Plant	6	8
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	5	5
Maintenance Truck – 1 t	Maintenance Truck	5	5
Fire Truck	Fire Truck	1	1
Screening & Crushing Plant - 12" max.	Road Crush and 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	2
Forklift – 30 t	Forklift	1	1
Forklift – 10 t	Forklift	2	2
Service Truck	Service Truck	5	5
Welding Truck	Welding Truck	4	4
Powerline Truck	Powerline Maintenance	2	2

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.

- Five 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.

OPEN PIT 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.

16.3 UNDERGROUND MINING OPERATIONS

The Mitchell and Iron Cap block caves are located within the Mitchell Valley and are accessed by the MTT. Figure 16.14 shows a plan view of the block cave area and the relationship between the block caves, the MTT, and the North Pit Wall Depressurization Tunnel. A section view shows the same elements in Figure 16.15.

Figure 16.14 Plan View of the Mitchell and Iron Cap Block Cave Mines

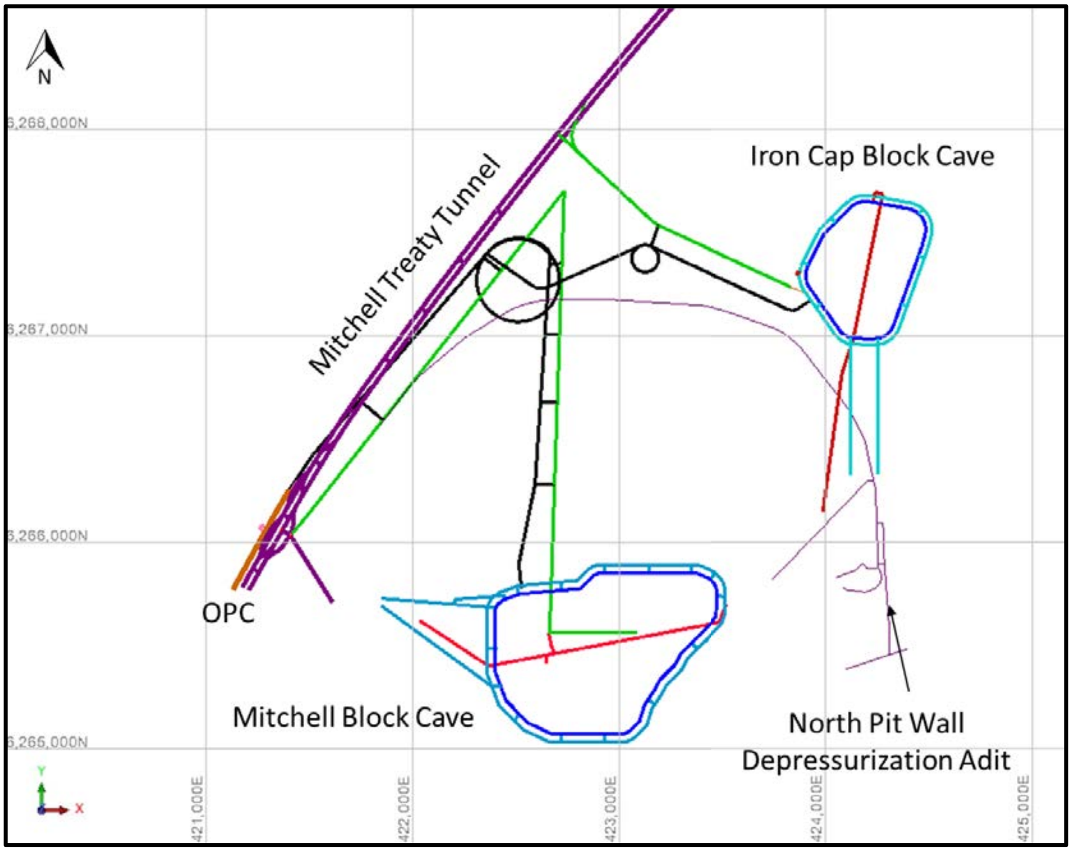
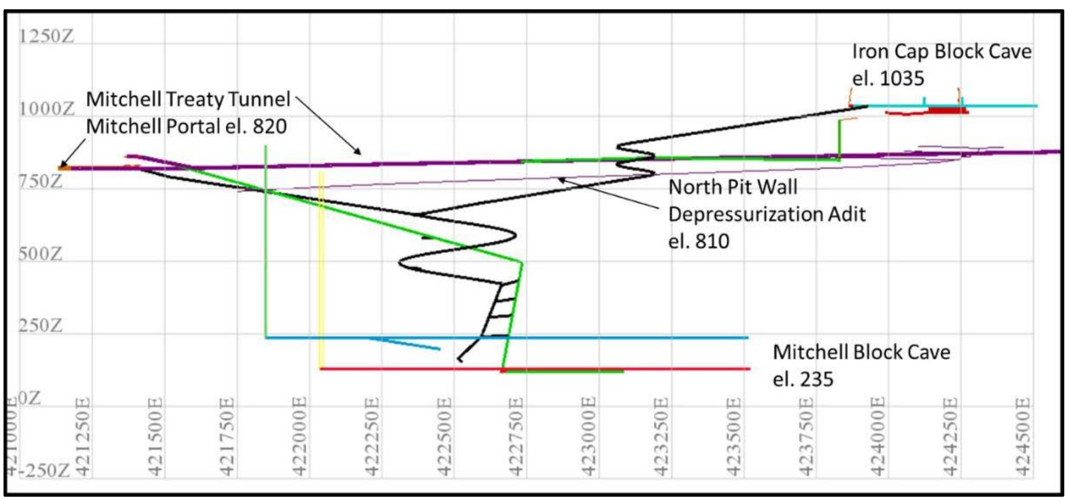


Figure 16.15 Section View of the Mitchell and Iron Cap Block Cave Mines (Looking North)



16.3.1 UNDERGROUND MINE DESIGN INPUTS

This subsection summarizes the block cave mine design inputs that are similar for both Mitchell and Iron Cap, and includes a geotechnical review and caving analysis.

The quality of the rock mass at the Mitchell deposit is rated as good. No major structural features are identified that might influence the caving mechanism and the progression of the cave in any significant manner.

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 for the Mitchell deposit, resulting in a corresponding lower predicted median in situ block size of 2.5 m^3, as compared to the Mitchell deposit. There are several gaps in data that are identified in the Iron Cap geotechnical and hydrogeological studies. These gaps will need to be addressed as part of future feasibility-level studies.

Caveability assessments for both the Iron Cap and Mitchell deposits have been completed using Laubscher's and Mathews' methods, which involve assessing caveability based on experience at other mining operations with similar rock quality. These methods indicate that the size (area) of a footprint required to initiate and propagate caving is between approximately 110 m and 220 m for both deposits. These dimensions are significantly smaller than the size of the deposit footprints that can potentially be mined economically by caving. This fact, together with the general large 3D shape of the deposits, suggests that both the Iron Cap and Mitchell deposits are amenable to block cave mining.

In situ stresses have been estimated at the Mitchell deposit using hydraulic fracturing tests. 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 will contribute to caving. More sophisticated numerical analyses to confirm and quantify stress-related impacts are recommended as part of future studies.

There have been no fracture propagation assessments applicable to preconditioning designs or in situ stress interpretations developed for 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 the Mitchell deposit is predicted to have block sizes greater than 6 m^3. At Iron Cap, block sizes are predicted to be 2.5 m^3. Without adopting some remediation measure, such large blocks will require significant secondary blasting, and a significant adverse impact on production and damage to the drawpoints that will require ongoing rehabilitation is likely. The cost estimates for the designs presented herein have considered remediation measures to accommodate large fragmentation.

The primary measure to accommodate the large fragmentation is to precondition the rock mass. The costs and scheduling to do this have been incorporated into this study for both cave mines. However, there are a number of uncertainties associated with preconditioning due to the limited number of caving mines where it has been applied and tested. The results from those mines employing preconditioning are encouraging, and there is sufficient experience in the industry to indicate that such fragmentation concerns do not represent a fatal flaw at either mine.

The uncertainty in the effectiveness of preconditioning to enhance fragmentation was addressed via production and cost risk mitigation measures. The average draw rate per column during steady state production is approximately half of maximum (165 mm/d), meaning there are roughly two drawpoints available for production for every one required to meet production targets. In addition, a fleet of mobile rock breakers and remote blockholers are included in the designs and costs to increase the time a drawpoint is producing, by decreasing the time it is blocked with oversize.

It is 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 and Mitchell, fragmentation of rock drawn down more than approximately 100 m is generally good. For this study, it is 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 will be required.

The expected coarse fragmentation at Mitchell and 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 17 m by 17 m or 18 m by 15 m, but it was considered prudent at this stage of study to adopt the slightly more conservative 15 m by 15 m spacing.

16.3.2 MITCHELL UNDERGROUND

The Mitchell deposit extends approximately 1,500 m east-west (along strike) and 400 m to 1,400 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 potential of mining the Mitchell deposit by a combination of open pit and underground methods was investigated previously (Golder 2011, Golder 2012) and these studies concluded that it is possible to mine the upper portions of the Mitchell deposit by open pit methods and the deeper portions by block caving.

MITCHELL UNDERGROUND MINERAL RESERVES

The Mineral Resource block model used for the study contains gold, copper, silver, and molybdenum grades, , as presented in Section 14.0, as well as the NSR values which are described in Section 16.2.4. The model contains Measured, Indicated, and Inferred grades, but the Inferred grades were set to zero and are not included in this PFS assessment. The Mineral Resources were constrained by the PFS pit and then evaluated using GEOVIA's PCBC™ software, to determine the Mineral Reserves for a block cave mine. Footprints at elevations of 180 m and 235 m produced the most value. Considering the footprint elevation of 235 m from the 2012 PFS (Tetra Tech 2012), and the similar geometries of the 2012 and 2016 footprints at this elevation, it was decided to maintain the footprint elevation at 235 m for this updated study. This resulted in 454 Mt of block cave Mineral Reserves, as shown in Table 16.14.

Table 16.14 Mitchell Block Cave Mineral Reserves ($15 NSR Shut-off)

Category	Tonnes (million)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)
Probable[1]	454	0.53	0.17	3.5	33.6

Notes: [1]Includes 10 Mt of mineralized dilution (the portion of Measured and Indicated material that is $0 < NSR < $16) and 59 Mt of non-mineralized dilution (material at zero NSR including the Inferred material set to zero grade).

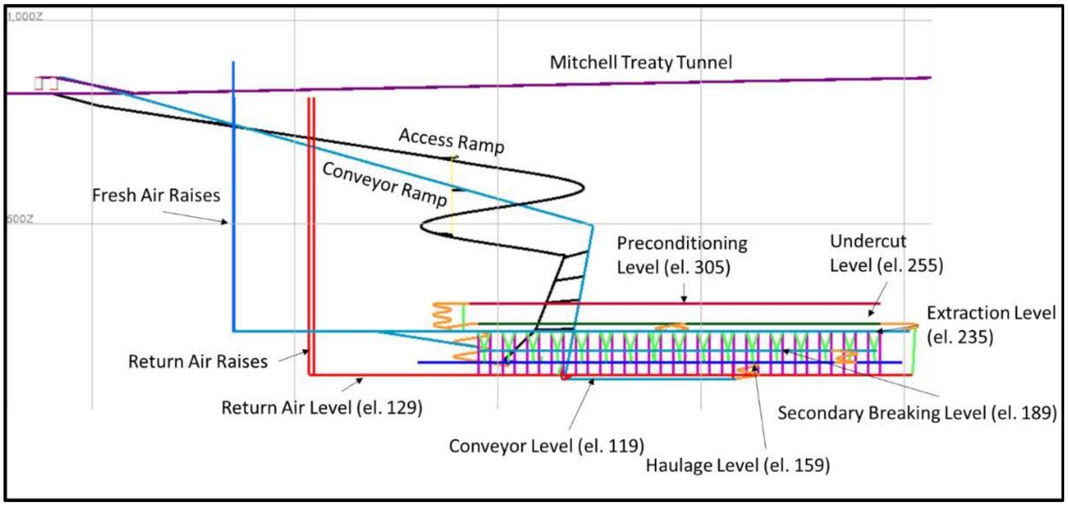
The Mineral Reserves contain dilution that include Mineral Resources that have grade, but are sub-economic (less than drawpoint shut-off), Inferred Mineral Resources that are set to zero grade, and non-mineralized material that is zero grade rock. Dilution estimates for the Mitchell block cave are 2% of sub-economic material (10 Mt) and 13% of zero grade dilution (59 Mt), for a total of 15% (69 Mt).

Mitchell Underground Mine Design

The underground mine design is based on modelling using GEOVIA's PCBC™ software and Footprint Finder module. Footprint Finder modelling indicates 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 established at 235 m. PCBC modelling indicates that the block cave could produce 20 Mt/a (55,000 t/d), requiring the development of 120 new drawpoints per year. The final mine design includes approximately 218 km of drifts and raises, including a 25% design allowance to account for the excavation of infrastructure such as service bays, fueling stations, wash bays, sumps, and electrical substations, and to account for over break.

The mine design comprises six main of levels: preconditioning, undercutting, extraction, secondary breakage, haulage, and conveying (Figure 16.16). In addition, the design includes a service ramp to surface to provide access for personnel, equipment and materials, and a conveyor ramp to the MTT ore bin for excavated material to be loaded on the MTT train to the mill. 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 will be strategically positioned to maintain access to the levels during caving and to meet ventilation requirements.

Figure 16.16 Section View of the Mitchell Block Cave Mine Design (Looking South)



There are 34 extraction drifts on the extraction level, and each drift is designed with three ore passes. This will reduce the average LHD haul distance to approximately 100 m and improve productivity. 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 centrally located gyratory crushers, where it will be crushed and conveyed to the surface.

In 2012, BGC evaluated the surface disturbance and ground deformation caused by block caving the Mitchell deposit (BGC 2012), and the analysis is still applicable to this study. It was found that the MTT and Mitchell OPC are outside the zone of disturbance resulting from caving mining.

MITCHELL UNDERGROUND EQUIPMENT AND MAJOR INFRASTRUCTURE

The proposed mobile diesel equipment is typical of that used in underground mines and will comprise units directly related to moving ore to the crushers (7 m^3 LHDs, secondary rockbreakers, and the train), development equipment (4.6 m^3 LHDs and 18 m^3 trucks), as well as ANFO loaders and ground support machines. In addition, service equipment is included for construction, supervision, engineering, and mine maintenance activities. At peak production, Mitchell will require a fleet of approximately 55 units 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 379 personnel in Year 30 (in Project years). Groundwater inflows are very small compared to surface runoff and will be readily handled by the proposed dewatering system.

The majority of the main ventilation infrastructure will be located on the extraction level. It will consist 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. An airflow of 860 m^3/s is required for the Mitchell mine to achieve a production rate of 55,000 t/d, based upon the diesel equipment utilized, air velocity considerations, and an allowance of 20% per level for items such air loss around regulators, poorly installed or damaged ducting, and ventilating any inactive headings in the active mining areas. Heating 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 25.1 MWh of electricity at peak operation. The main contributors to this are the crushers, conveyor belts, ventilation fans, and dewatering pumps.

The mine dewatering system will require an average of 3.9 MWh, with a maximum of 29 MWh during a peak storm event, which is greater than that required to operate the entire mine under normal conditions. The strategy will be to shut down or reduce operations in the underground mine, along with other site facilities, during flooding events when the high-powered pumps are required. This will allow power to be diverted from normal operations to power the pumps.

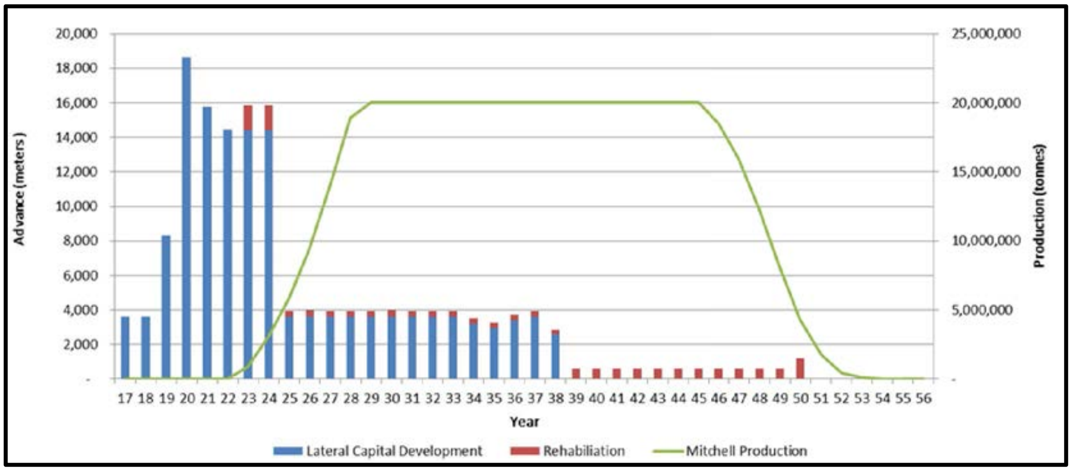
The hydrological characterization of the site indicates that a 200-year runoff event could lead to a maximum one day inflow of approximately 773,000 m^3 of water, even with the construction of diversion ditches beyond the crest of the pit. To accommodate this inflow, the dewatering plan includes significant pumping and storage capacity underground. Two, 6.0 km long, 7.5 m x 7.5 m dewatering tunnels have been designed to convey water from the mining area to beneath the water treatment plant where eight multi-stage centrifugal pumps will lift the water and transport it to the WSD. The system is designed to handle flows at variable combined rates up to 4 m^3/s.

MITCHELL UNDERGROUND SCHEDULES

The mine development schedule is separated into three phases; an initial pre-production phase, which develops the primary access ramp and conveyor drifts; a second ore production phase that creates sufficient openings to start and ramp-up production from the cave; and the final phase once the mine has reached steady-state production, and the development fleet is only required to create sufficient openings to maintain production. The average length of required annual development is approximately 4,000 m, with peak development occurring during the second phase, when approximately 15,000 m/a is required. The pre-production development phase for Mitchell block cave is six years.

The mine production schedule was developed using GEOVIA PCBC™ software. It is assumed that sloughing of peripheral waste rock will occur into the crater above the cave 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 will mix with mineralized material as dilution with zero grade, and the combined material will report to the drawpoint. 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 draw rate in the PCBC analysis. However, an average draw rate of only 108 mm/d is required to achieve production targets (the maximum draw rate modeled 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 six years, steady state production at 20 Mt/a for 14 years, and then ramp-down production for another 7 years. Figure 16.17 presents the lateral development, rehabilitation, and production schedules in Project years.

Figure 16.17 Mitchell Block Cave Mine Development and Production Schedules



16.3.3 IRON CAP UNDERGROUND

The Iron Cap deposit extends approximately 1,200 m east-west (along strike), 700 m north-south, and 800 m in the vertical direction. 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 an update to a PFS published in 2011. Similar to Mitchell, the location, dimensions, and dip of the mineralized material at Iron Cap indicates that it is a suitable candidate for block caving.

IRON CAP UNDERGROUND RESERVES

The Mineral Resource block model used for the study contains gold, copper, silver, and molybdenum grades as presented in Section 14.0, as well as NSR values described in Section 16.2.4. The model contains Measured, Indicated, and Inferred grades, but the Inferred grades were set to zero and are not included in this 2016 PFS. The Mineral Resources were evaluated using GEOVIA's PCBC™ software to determine the Mineral Reserves for a block cave mine. A footprint at an elevation of 1,035 m produced the most value and resulted in 224.6 Mt of block cave Mineral Reserves as shown in Table 16.15.

Table 16.15 Iron Cap Block Cave Reserves ($16 NSR Shut-off)

Category	Tonnes (million)	Au (g/t)	Cu (%)	Ag (g/t)	Mo (ppm)
Probable[1]	224.6	0.49	0.20	3.6	13.0

Notes: [1]Includes 25 Mt of mineralized dilution (the portion of Measured and Indicated material that is $0 < NSR < $16) and 20.2 Mt of non-mineralized dilution (material at zero NSR including the Inferred material set to zero grade).

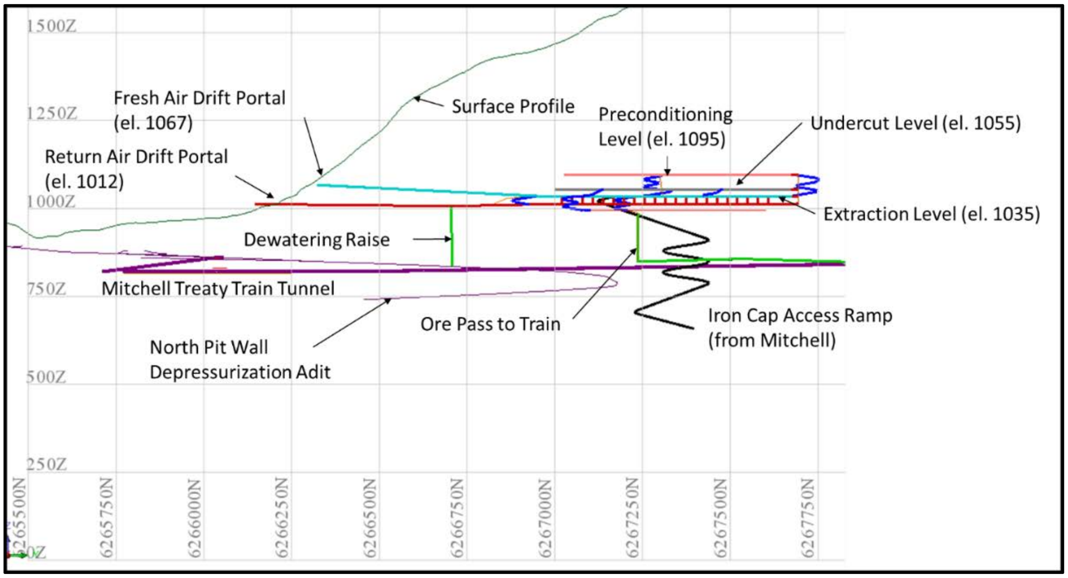
The Mineral Reserves contain dilution that includes Mineral Resources that have grade, but are sub-economic (less than drawpoint shut-off), Inferred Mineral Resources that are set to zero grade, and non-mineralized material that is zero grade rock. Dilution estimates for the Iron Cap block cave are 11% of sub-economic material (25 Mt) and 9% of zero grade dilution (20.2 Mt), for a total of 20% (45.2 Mt).

Iron Cap Underground Mine Design

The underground mine design is based on modelling using GEOVIA's PCBC™ software Footprint Finder module. Footprint Finder modelling indicates that the optimum footprint for the Iron Cap deposit is at an elevation of 1,035 m and is approximately 600 m wide in the north-south direction and 400 m wide in the east-west direction. PCBC modelling indicates that the block cave could produce 15 Mt/a (average of 40,000 t/d), requiring development of 120 new drawpoints per year. The mine design requires approximately 87 km of drifts and raises, including a 25% design allowance to account for the excavation of infrastructure such as service bays, wash bays, fueling station, refuge stations, sumps and electrical substations, and to account for over break.

The Iron Cap mine design includes four main levels: preconditioning, undercutting, extraction, and conveying (Figure 16.18). The design also includes a return air drift located between the conveying and extraction levels. 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.

Figure 16.18 Section Looking East of the Iron Cap block Cave Mine Design



Personnel, material, and supplies will access the Iron Cap mine through a drift driven from the Mitchell underground access ramp. Two fresh air portals and one exhaust portal are planned on the north slope of the Mitchell valley. These tunnels may act as an emergency egress. The fresh air tunnels will connect to surface and a perimeter drift will be constructed around the mine footprint to provide fresh air to the mine workings.

Excavated 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 third conveyor that will transport the production material to a surge bin located above the Iron Cap MTT train tunnel.

Iron Cap Underground Equipment and Major Infrastructure

The proposed mobile diesel equipment is typical of that used in underground mines and will comprise equipment related to moving ore to the crushers (7 m^3 LHDs and secondary rock breakers), 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, supervision, engineering and mine maintenance activities. At peak operation, Iron Cap will require a fleet of approximately 51 units 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 350 personnel in Year 38 (KSM production years).

The required airflow for the Iron Cap mine is 548 m^3/s based upon the total diesel equipment used on each mining level, including a 20% design allowance for items such air loss around regulators, poorly installed or ripped ducting, and ventilating unused headings in active sections of the mine. It is estimated that the Iron Cap mine will require 9.1 MWh of electricity at peak operation.

The hydrological characterization of the Iron Cap site indicates that a 200-year runoff event could lead to a maximum one-day inflow of approximately 292,000 m^3 of water. The underground water management system at Iron Cap is currently designed to handle 4 m3/s. This caters for the estimated groundwater inflow and ice melt. The surface inflows will report to the drawpoints and will be managed in a similar manner to the groundwater inflows. To provide for drainage, the underground drifts will be graded so that water will naturally drain towards the MTT. Any flood water will be directed through the return airway drift and into the Mitchell NPWDA by a series of raises connecting the two tunnels. Pumps are not required to dewater Iron Cap.

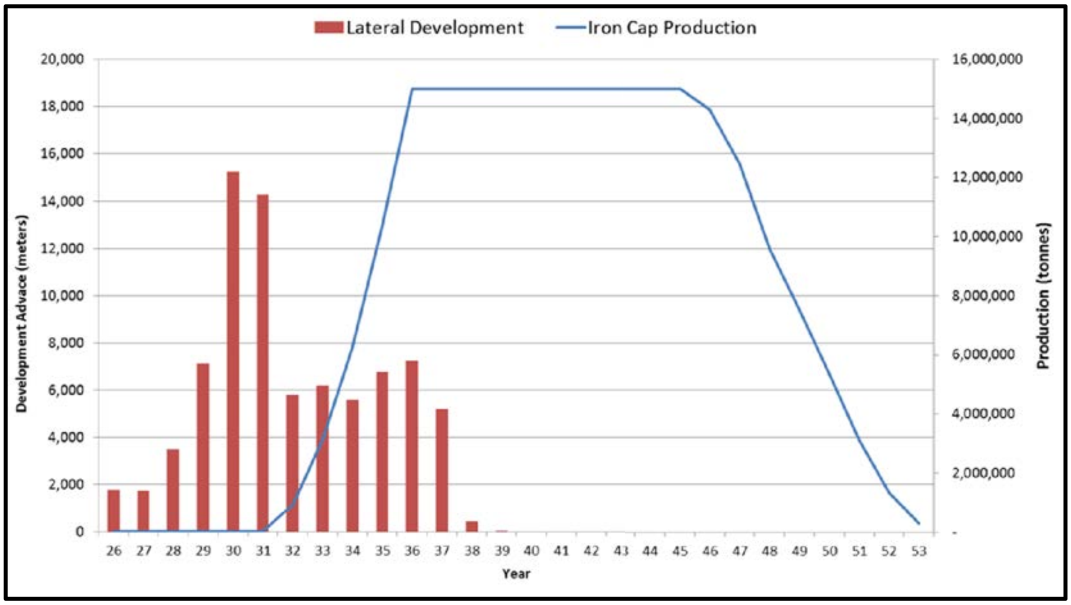
IRON CAP UNDERGROUND MINE SCHEDULE

The mine development schedule is separated into three phases: an initial pre-production phase, which develops the primary access ramp and conveyor drifts; a second ore production phase that creates sufficient openings to start and ramp-up production from the cave; and a final phase once the mine has reached steady-state production and the development fleet is only required to create sufficient openings to maintain full production. The length of development required during the peak development period is approximately 10,000 m/a. The pre-production development period for Iron Cap is six years.

The mine production schedule was developed using GEOVIA's PCBC™ software. It is assumed that sloughing of peripheral waste rock will occur into the crater and cover the upper surface of the material being drawn down. This was modeled in PCBC by assuming that an infinite supply of waste material is present on top of the mineralized material. As material is drawn from the drawpoints, the waste rock will mix with mineralized material as dilution with zero grade, and the combined material will report to the drawpoint.

The draw rates used in the PCBC modelling of Iron Cap are similar to those used at Mitchell, for similar reasons. During the early stages of mining, a draw rate of 200 mm/d was chosen as a maximum draw rate in the PCBC analysis. However, an average draw rate of only 110 mm/d is required to achieve production targets, so there are roughly twice as many drawpoints available as are required to meet production targets (the maximum draw rate modeled never exceeds 180 mm/d). Iron Cap is estimated to have a production ramp-up period of four years, steady state production at 15 Mt/a for 10 years, and then ramp-down production for another 9 years. Figure 16.19 presents the lateral development and production schedules in Project years.

Figure 16.19 Iron Cap Block Cave Mine Development and Production Schedules



16.4 MINE PRODUCTION SCHEDULE

The summarized production schedule results are shown in Table 16.16 and Figure 16.20, including both open pit and underground mining. The mine production plan starts in lower capital cost open pit areas using conventional large-scale equipment before transitioning into block cave underground bulk mining later in the mine life. Starting pits have been selected in higher grade lower strip ratio areas and cut-off grade strategy is used to enhance revenues for a minimum capital payback period. The cut-off strategy stockpiles lower grade open pit material early in the mine life. The Mitchell and Iron Cap underground block caves are brought into a development and production sequence to provide continuous mill feed during the LOM, while evening out the mine's sustaining capital requirements. The open pit sequencing is then adjusted to augment the underground ore production to meet the full mill throughput requirements. The Mitchell underground block cave starts ore production in Year 23 and ramps up to full production by Year 29. Iron Cap block cave ore production starts in Year 32 and reaches full production by Year 36. At this point, all ore is supplied by the Mitchell and Iron Cap underground block caves (a total of 96,000 t/d). As the Mitchell block cave production starts to ramp down (starting in Year 46), the mill feed is further reduced to 62,000 t/d. In the final years of production (Year 49 and onwards), the underground ore is supplemented with material stockpiled from the open pits to maintain 62,000 t/d of mill feed.

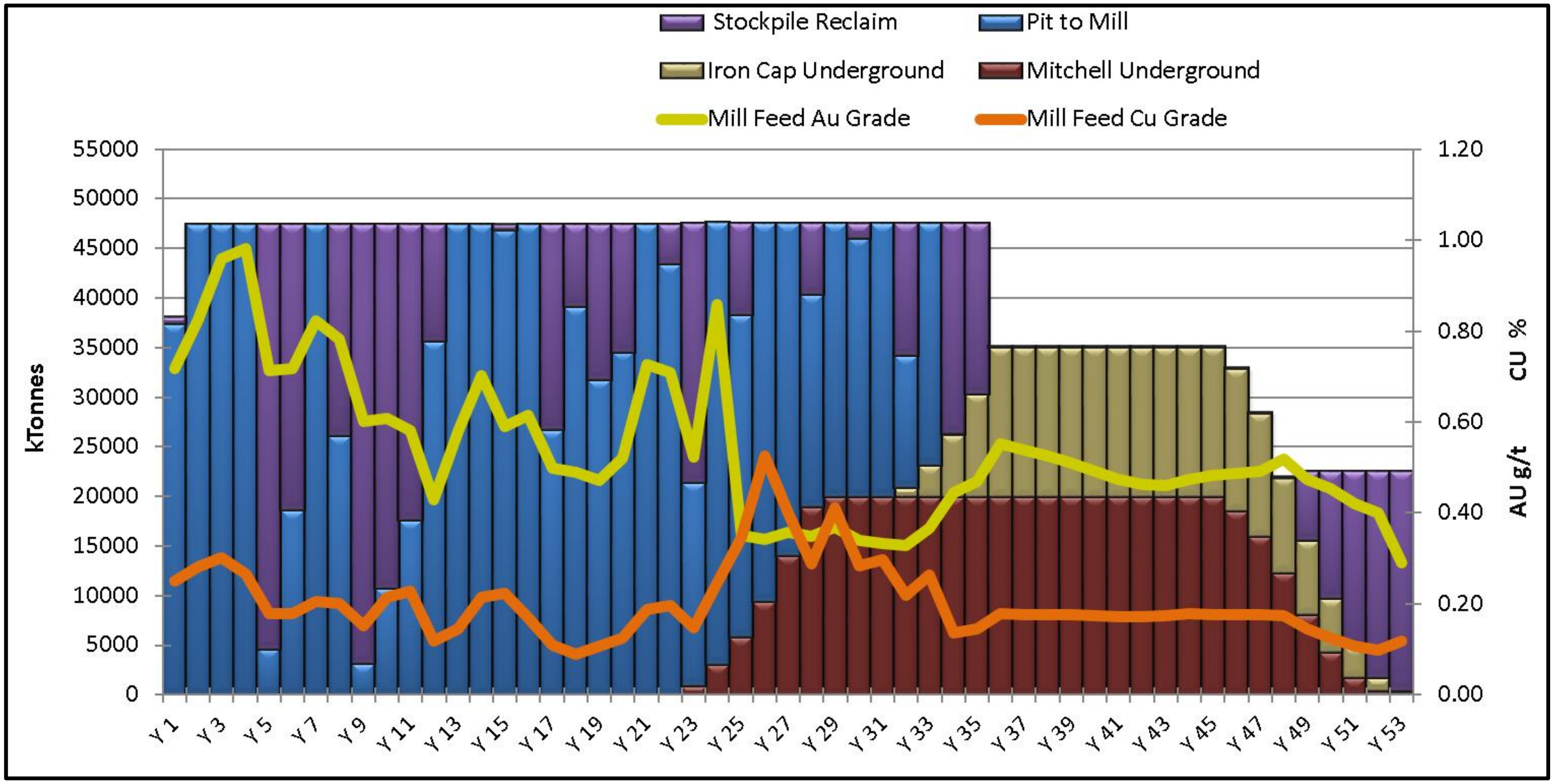
Details of the production schedule can be found in Appendix E1.

Table 16.16 Summarized Production Schedule – Open Pit and Underground

		Year																			LOM
		-4	-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	50 to 53	
Pit To Mill	**mt**	**-**	**-**	**-**	**-**	**37**	**47**	**47**	**47**	**5**	**19**	**47**	**26**	**3**	**11**	**375**	**335**	**-**	**-**	**-**	**1,066**
AU	g/t	-	-	-	-	0.715	0.824	0.959	0.981	0.992	0.780	0.823	0.926	0.494	0.626	0.586	0.484	-	-	-	0.604
CU	%	-	-	-	-	0.250	0.282	0.301	0.265	0.230	0.189	0.205	0.250	0.326	0.419	0.161	0.361	-	-	-	0.264
AG	g/t	-	-	-	-	2.51	2.05	2.30	4.54	3.62	2.23	3.39	3.31	0.64	0.97	2.13	2.42	-	-	-	2.32
MO	ppm	-	-	-	-	29.4	46.8	61.4	19.8	47.9	76.7	55.6	27.5	46.9	34.5	72.6	23.8	-	-	-	45.1
Pit To Stockpile	**mt**	**-**	**-**	**-**	**5.2**	**2.6**	**15.3**	**67.4**	**27.7**	**1.5**	**56.1**	**44.8**	**3.6**	**5.8**	**10.1**	**164.8**	**40.0**	**-**	**-**	**-**	**455**
Stockpile Reclaim	**mt**	**-**	**-**	**-**	**-**	**0.7**	**-**	**-**	**-**	**42.8**	**28.8**	**-**	**21.3**	**44.3**	**36.7**	**99.9**	**48.1**	**51.8**	**19.8**	**60.6**	**455**
AU	g/t	-	-	-	-	0.869	-	-	-	0.683	0.676	-	0.608	0.608	0.603	0.401	0.355	-	-	-	0.466
CU	%	-	-	-	-	0.191	-	-	-	0.174	0.173	-	0.140	0.140	0.154	0.123	0.116	-	-	-	0.129
AG	g/t	-	-	-	-	1.70	-	-	-	2.21	2.22	-	2.16	2.16	1.77	1.80	2.14	-	-	-	1.81
MO	ppm	-	-	-	-	48.4	-	-	-	56.7	57.5	-	84.8	84.8	72.9	66.1	62.0	-	-	-	60.4
Stockpile Inventory	**mt**	**-**	**-**	**-**	**5.2**	**7.1**	**22.5**	**89.8**	**117.6**	**76.3**	**103.6**	**148.4**	**130.6**	**92.1**	**65.5**	**130.4**	**122.3**	**80.4**	**60.6**	**- 0.0**	**-**
Mitchell Underground	**mt**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**92**	**200**	**159**	**2**	**453**
AU	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.594	0.508	0.523	0.510	0.531
CU	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.190	0.162	0.155	0.093	0.165
AG	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4.38	3.25	3.27	1.51	3.48
MO	ppm	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	16.5	27.5	50.8	62.8	33.7
Iron Cap Underground	**mt**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**96**	**124**	**5**	**224**
AU	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.564	0.434	0.530	0.492
CU	%	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	0.199	0.199	0.169	0.199
AG	g/t	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3.02	4.06	4.34	3.63
MO	ppm	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	9.9	15.6	7.7	13.0
Total Mill Feed	**mt**	**-**	**-**	**-**	**-**	**38.0**	**47.4**	**47.5**	**47.5**	**47.4**	**47.4**	**47.4**	**47.5**	**47.5**	**47.5**	**474.5**	**475.4**	**412.8**	**303.2**	**67.6**	**2,198**
AU	g/t	-	-	-	-	0.717	0.824	0.959	0.981	0.713	0.717	0.823	0.783	0.601	0.608	0.547	0.492	0.445	0.476	0.370	0.549
CU	%	-	-	-	-	0.249	0.282	0.301	0.265	0.180	0.179	0.205	0.200	0.152	0.214	0.153	0.303	0.197	0.169	0.108	0.209
AG	g/t	-	-	-	-	2.49	2.05	2.30	4.54	2.35	2.22	3.39	2.79	2.06	1.59	2.06	2.77	2.58	3.47	1.16	2.59
MO	ppm	-	-	-	-	29.8	46.8	61.4	19.8	55.8	65.1	55.6	53.3	82.3	64.2	71.2	26.2	23.8	35.9	24.6	42.6
Metal to the Mill																					
AU	million Oz	-	-	-	-	0.9	1.2	1.5	1.5	1.1	1.1	1.2	1.2	0.9	0.9	8.4	7.5	5.9	4.6	0.8	38.8
CU	million lb	-	-	-	-	209	295	316	278	188	188	215	210	160	225	1,602	3,183	1,793	1,132	161	10,155
AG	million Oz	-	-	-	-	3.0	3.1	3.5	6.9	3.6	3.4	5.2	4.3	3.2	2.4	31.4	42.4	34.3	33.8	2.5	183
MO	million lb	-	-	-	-	2.5	4.9	6.4	2.1	5.8	6.8	5.8	5.6	8.6	6.7	74.5	27.5	21.7	24.0	3.7	207
Total Waste Mined	**mt**	**10**	**10**	**9**	**61**	**141**	**136**	**40**	**84**	**129**	**130**	**111**	**140**	**193**	**219**	**1,008**	**510**	**72**	**-**	**-**	**3,003**
Strip Ratio (waste mined/ Plant Feed)	t/t	-	-	-	-	3.7	2.9	0.8	1.8	2.7	2.7	2.3	2.9	4.1	4.6	2.1	1.1	0.2	-	-	1.4

Note: [1]Waste mined in the production schedule in Figure 16.20 includes re-handled waste and waste mined from borrow pit sources for construction purposes.
[2]The mill feed specified in Table 16.16 only includes ore from the Proven and Probable open pit and underground Mineral Reserves and does not include any Inferred Mineral Resources.

Figure 16.20 KSM Mill Feed Production Schedule



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 planned mill life is approximately 53 years, excluding the production development stage and closure stage. The Mitchell deposit will be the dominant resource of mill feed for the process plant and will supply mill feed throughout the projected LOM. The ore from the Sulphurets deposit will be fed to the plant together with the ore from the Mitchell pit from Years 1 to 17, excluding Years 4, 5, 12 and 13, and with the ores from the other deposits during the last four years. The ore from the Kerr deposit, together with the ores from the other deposits, will be introduced to the plant during Years 24 to 34 and 53, while Iron Cap ore will be fed to the process plant from Year 32 to the end of mine life.

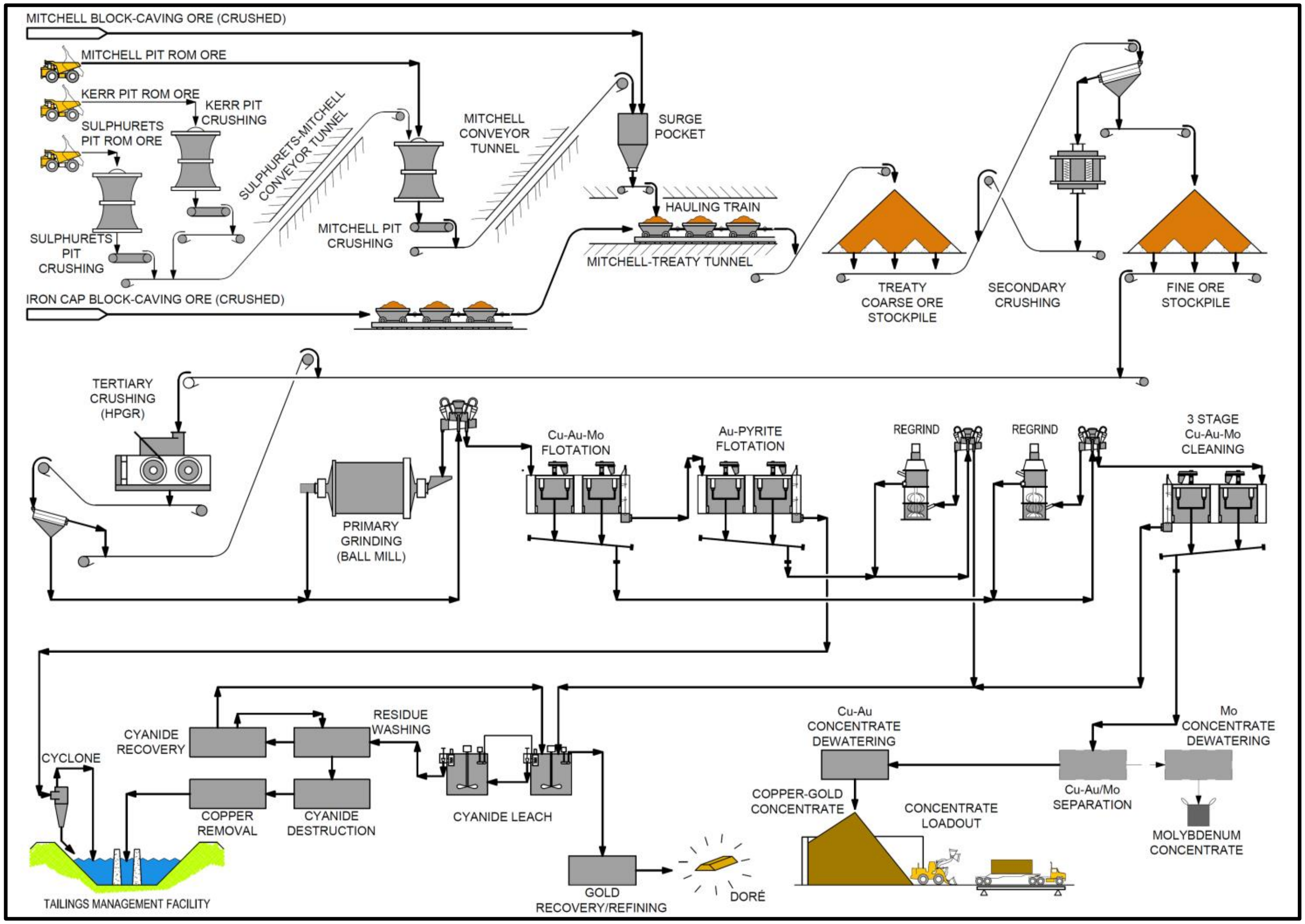
A combination of conventional flotation and cyanidation processes are proposed for the Project. The process plant will consist of three separate facilities:

- an ore primary crushing and handling facility at the Mitchell mine site
- an ore transportation system by trains 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/recovery, and concentrates dewatering.

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 Appendices C1 and C2, respectively.

Figure 17.1 Simplified Process Flowsheet



Source: Tetra Tech

The ores from the Sulphurets and Kerr deposits and the upper zone of the Mitchell deposit will be extracted by open pit mining, while the ores from the Iron Cap deposit and the lower Mitchell deposit will be mined by underground block caving. The primary crushing facilities located on the Mitchell mine site will reduce the ROM particle size to approximately 80% passing 150 mm by gyratory crushers. Ores from the Sulphurets and Kerr deposits will be crushed at their respective sites, excluding the Sulphurets ore produced in Year 1, which will be hauled to and crushed at the Mitchell site. The ores from the lower Mitchell Zone and the Iron Cap deposit will be mined by block caving and be crushed in the underground mine. The crushed materials will then be transported by conveyors, or by a combination of conveyors and trains, to the MTT and loaded onto the MTT ore transport trains. The crushing circuit at the Mitchell surface site will include:

- primary crushing by two 60 inch by 89 inch gyratory crushers
- crushed ore transport conveyors.

The crushed ore will be transported by a train transport 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 in closed circuit with screens
- tertiary crushing by HPGR in closed circuit with screens
- primary grinding by ball mills
- copper-gold/molybdenum bulk flotation with regrinding of bulk concentrate
- 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 by carbon elution and production of gold dore
- 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 CIL tailing, which will be stored in the lined tailing pond of the TMF.

The mill feed produced from the Mitchell crushing facility or from the block caving sites will be transported via the MTT train system to the coarse ore stockpile at the Treaty OPC site. The 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 five cone crushers (four in operation and one on standby) and then four HPGRs in closed circuit with vibrating screens.

The screen undersized material from the HPGR circuit 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 be fed into copper-gold/molybdenum rougher/scavenger flotation circuits, consisting of two operation two parallel circuits. The copper rougher flotation concentrates from the flotation circuits will be reground to a particle size of 80% passing approximately 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% copper. 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 approximately 9% moisture before being transported to 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 approximately 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 combination of a SART process, and an AVR process. The AVR process will recover the free cyanide from the solution by acidifying and stripping the solution and then absorping the stripped hydrogen cyanide gas by a sodium hydroxide solution to recover the cyanide for reuse.

The washed residues will be treated by an sulphur dioxide/air cyanide destruction process to destroy the residual weak acid dissociable (WAD) cyanide. The leach residues will then be further treated by carbon adsorption to remove dissolved copper. The copper will be stripped off from the activated carbon and precipitated by sodium sulphide as copper sulphide which will be blended into copper concentrates for sale.

The treated residues will then be transported by pipeline to the lined CIL pond of the TMF. The sulphide leach residues will be stored under water at all times to prevent the oxidation of sulphides.

The Treaty Process Plant layout and the primary grinding and flotation facility are depicted in Figure 17.2 and Figure 17.3

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
Average Daily Process Rate	t/d	130,000
Operating Year	d	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 Work Index - 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.

Figure 17.2 Treaty Process Plant Layout

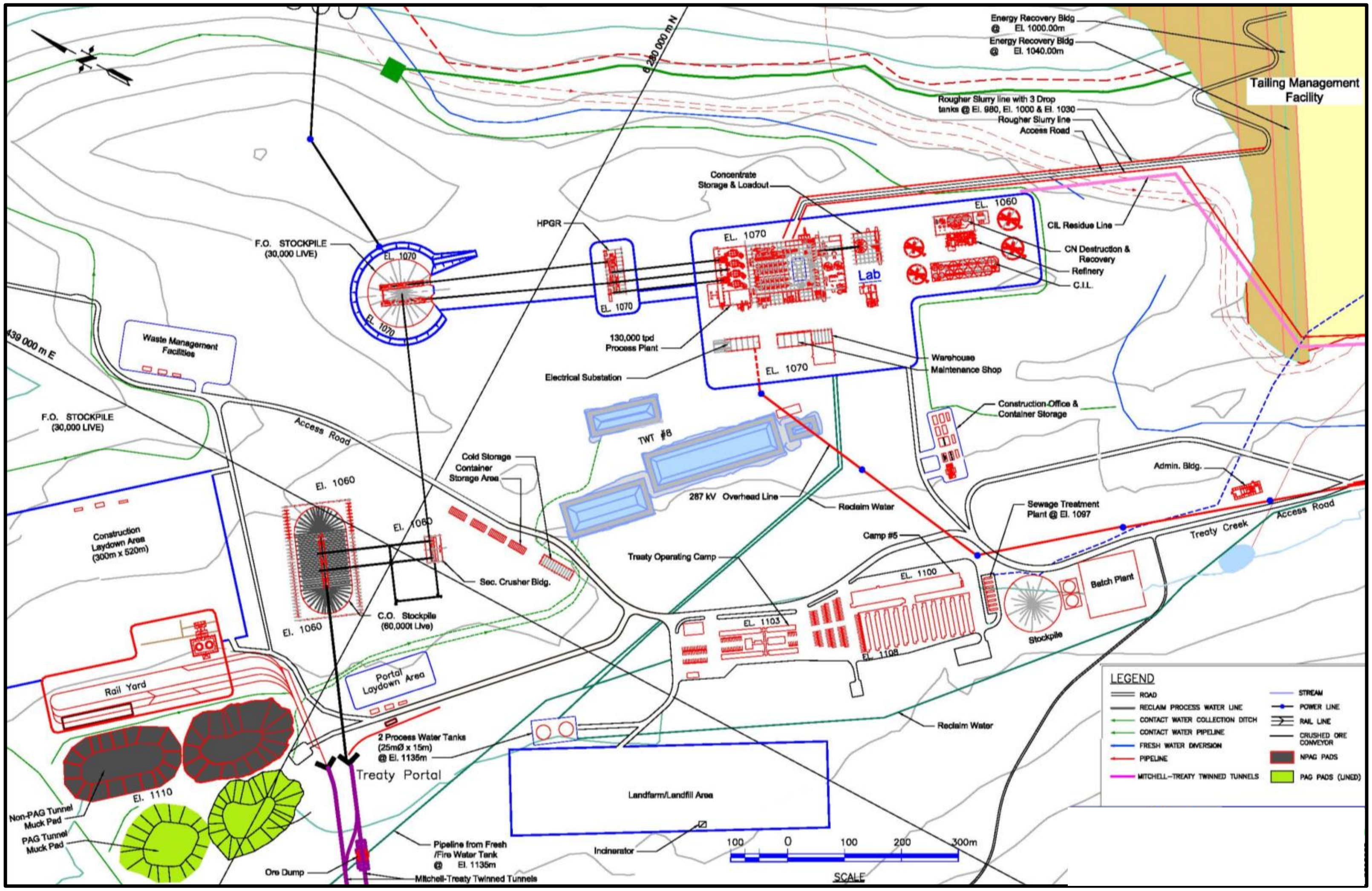
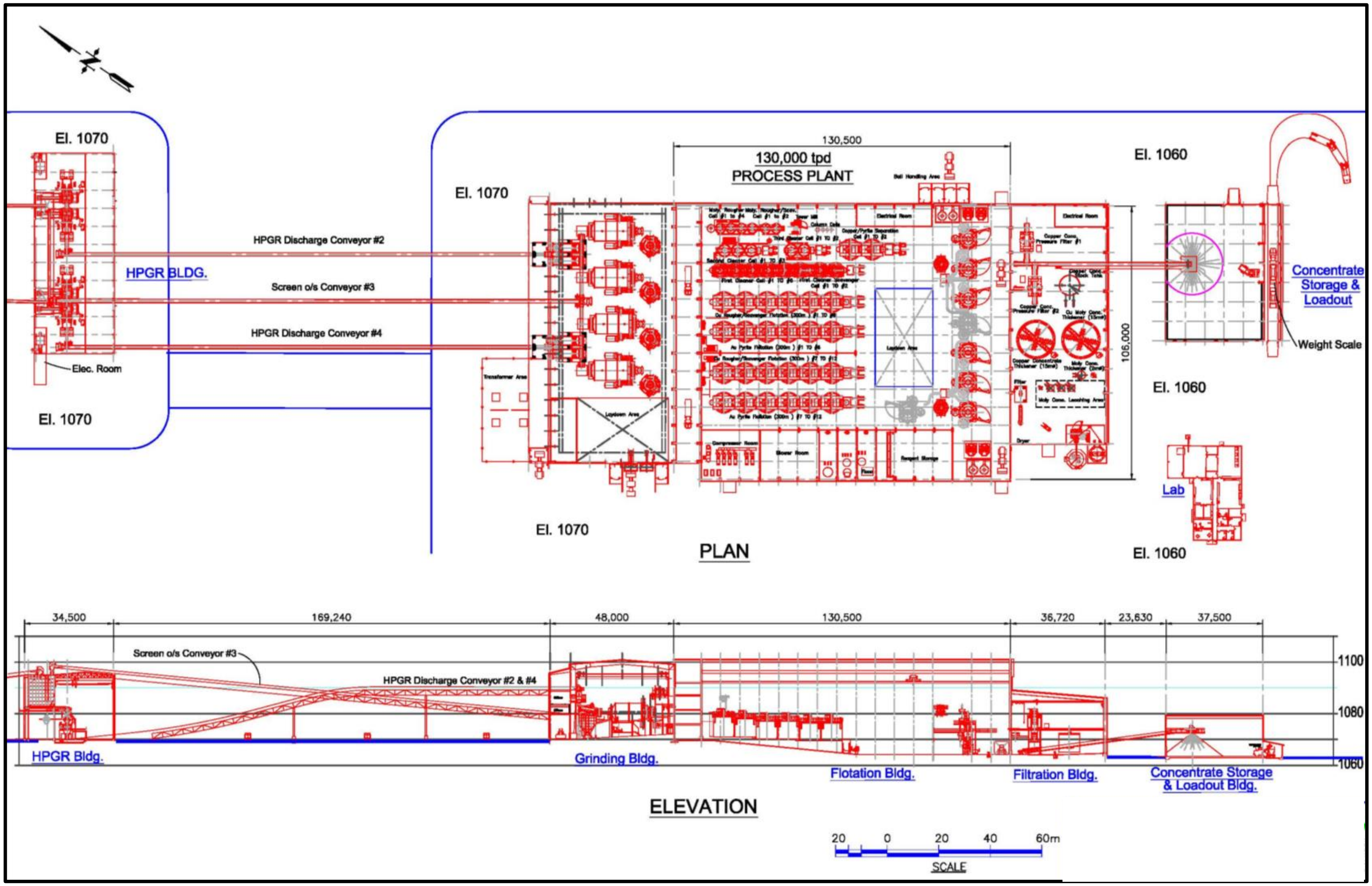


Figure 17.3 Treaty Process Plant Primary Grinding and Flotation Facility



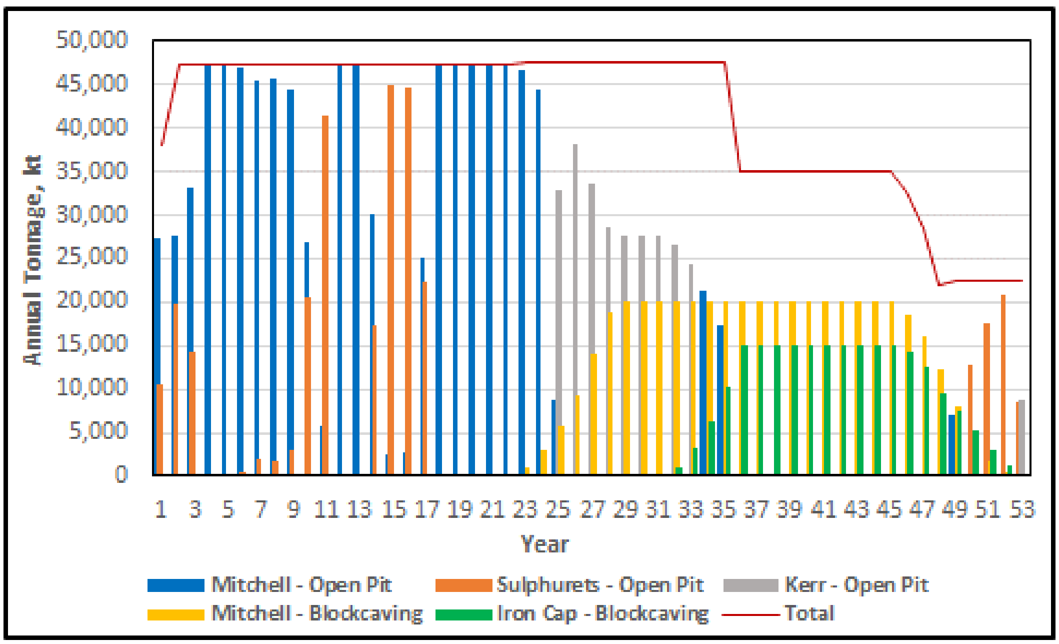
17.3 PROCESS PLANT DESCRIPTION

17.3.1 PRIMARY CRUSHING

There will be five primary crushing sites throughout the Project life at the Mitchell, Kerr, and Sulphurets sites, as well as in the underground block caving mines. The proposed mill feed rates from these deposits are shown in Figure 17.4.

At the Mitchell OPC site, primary crushing will mainly consist of two 60 inch by 89 inch gyratory crushers, two apron feeders, and one train load surge bin feed conveyor. The ROM material feeding to the gyratory crushers will be from the Mitchell pit and the first 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 450 m long train load surge bin feed conveyor, which will be located inside of the Mitchell surge bin feed conveyor tunnel. The surge bin is designed to have a live capacity of 30,000 t (there are two pockets, each 15,000 t). The ore from the Mitchell open pit will feed to the mill during Years 1 to 22, 34, 35, 48, and 49.

Figure 17.4 Proposed Mill Feed Rates from Open Pit and Blockcaving Operations



For the underground block cave operation, the ore from the lower Mitchell zone 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 30,000-t surge bin where the crushed ore will be blended with the materials from the Mitchell and Sulphurets pits and then loaded from the surge bin into the train cars and transported to the end of the MTT at the Treaty site. The ore will be fed to the mill during Years 19 to 49.

The Sulphurets ore will supplement the mill feed between Years 1 to 17, excluding Years 4, 5, 12 and 13, and with the ores from the other deposits during the last four years. The ROM ore from the Sulphurets pit will be trucked to the Mitchell site and crushed at the Mitchell crushing facility in Year 1. Starting from Year 2, the ore produced will be crushed by a 60 inch by 89 inch 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 stockpiled ore will then be reclaimed and trucked to the Mitchell crusher dumping pockets, where the ore will pass through the crushers and be sent to the 30,000-t surge bin together with the crushed Mitchell ore.

The ore from the Kerr deposit, together with the ores from the other deposits, will be introduced to the plant during Years 24 to 34 and 53, the ROM ore and waste rock from the Kerr pit will be crushed by two 60 inch by 89 inch 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 the 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. Similarly, the reclaimed ore will be trucked to the Mitchell crusher dumping pockets, where the ore will pass through the crushers and be sent to the 30,000-t surge bin together with the crushed Mitchell ore.

Waste rock from the Kerr mine will be conveyed from the Kerr to the Sulphurets pit via the rope conveyor. The waste rock will then be backfilled into the mined Sulphurets pit for storage.

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 a surge bin located at the end of the Iron Cap and MTT connection tunnel. The ore will supplement the mill feed from Year 32 to to the end of mine life.

17.3.2 Coarse Ore Transport From Mitchell Site to Treaty Site

The crushed ore will be reclaimed by two automatic train loading systems from the two coarse ore surge bins at the Mitchell site and transported to the plant site by a train transport system through the MTT. The ore from the Iron Cap site will be loaded into the train cars at the Iron Cap underground site from the Iron Cap surge bin and transported to the Treaty site via an Iron Cap and MTT connection tunnel and then through the MTT. Loading chutes under the ore surge bins will feed ores into awaiting trains that will transport the ores to an unloading station at the Treaty end of the MTT. Because the loading and unloading systems, including surge bins are located underground, the arrangement would mitigate potential freezing issues. The train cars will dump ore into a live underground unloading bin. Two apron feeders will unload the bin onto a conveyor to transport the ore to the top of the Treaty coarse ore stockpile. Loading chutes will be controlled remotely, and unloading chutes will operate autonomously. No onboard operators will be required within the tunnels during train system operation.

Each train will consist of one, 140-t electric locomotive and 16, 42-m^3 belly dump ore cars that will have the capacity to deliver 800 t/h from Mitchell to Treaty based on 90-minute cycle times. On average, eight trains will deliver approximately 130,000 t/d of ore to meet the process plant requirements. An additional four trains will be on standby to provide for mechanical availability or to handle an increase in plant feed of up to 10,000 t/h. The transport system is detailed in Section 18.3. Dust collecting systems will be installed at the loading and unloading points to collect fugitive dust.

17.3.3 Coarse Material Handling

The crushed ore from the trains will be continuously and automatically unloaded from the bottom discharge ore cars into the bin underneath, with each car taking an average of nine seconds to unload. The train will be driven via traction drives across the unloading station at a maximum speed of 2.5 km/h.

At the bottom of the surge bin, the ore will be reclaimed by two apron feeders and then onto a conveyor belt that will transport the ore to the surface and feed the Treaty coarse ore stockpile with a live capacity of 60,000 t at the Treaty OPC site. Apart from the conveyor tunnel, a vertical escape tunnel that joins up the unloading station and the surface will be constructed for any emergencies.

The ore will be reclaimed from the stockpile 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. The reclaim tunnel will be heated to prevent potential freezing during operation in winter.

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 for 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 cone crushers, each with an approximately 2.4 m diameter mantle and driven by a 750-kW motor or equivalent
- 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. Similarly, the stockpile reclaim tunnel will be heated to prevent potential freezing during winter operation.

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 milling circuits, which are made up of the following equipment:

- four 7.6 m diameter by 11.9 m long (25 ft by 39 ft) 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 a 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 two copper-gold-molybdenum bulk rougher flotation circuits. The overflow from two of the four hydrocyclone clusters from the primary grinding circuits will separately feed the two flotation trains, each consisting of six 300-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 940 t/d or 344,000 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 content, 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-m2 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 two parallel lines, each line with six, 300 m^3 pyrite rougher flotation cells.

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 approximately 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 to approximately 45% w/w 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 five 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 oil-free 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 flow through a back pressure control valve and then be cooled by exchanging heat with the barren solution prior to reporting 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 recycling.

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 and silver 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 after it is mixed with melting flux. A 125-kW induction furnace will be used for gold-silver 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 precipitates will be blended with the copper-gold concentrate for sale. 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. The precipitates will be blended with the copper-gold concentrate for sale.

Cyanide Destruction

The remaining cyanide in the washed leach residues from the second washing thickener will be decomposed by a sulphur dioxide (SO_2)/air oxidation cyanide destruction process. Sodium metabisulphite will be used as the sulphur dioxide 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 sulphur dioxide oxidation tanks, and a wet alkaline scrubbing system. Compressed air will be provided for the oxidation process. The treated residues will be sent to copper removal treatment circuit.

Copper Removal

A copper removal circuit is proposed to removal the dissolved copper from the treated residues if the copper level from the sulphur dioxide-air cyanide destruction circuit is higher than the requirement. Activated carbon will be added into the residue slurry after the slurry is treated by the cyanide destruction. The copper removal treatment will be carried in two stages in two reactors. The loaded carbon will be removed from the first stage of the copper removal reactor while the fresh carbon will be added into the section stage of the copper removal reactor. The copper loaded carbon will be stripped by acid washing and the copper in the washing solution will be precipitated by sodium sulphide. The precipitate produced will be blended with the copper-gold concentrate for sale. The treated residues will be sent to the lined CIL residue storage pond in the TMF.

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 . 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 a hydrogen peroxide (H_2O_2) water treatment plant to further to remove impurities before it is sent to the north or south tailing ponds.

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 by a reclaim water barge 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.

A separate barge equipped with reclaim water pumps will be installed in the flotation tailing storage pond to reclaim the water for the tailing classification operations (to provide dilution water for hydrocycloning) and for the excess water discharge via the Treaty Creek diffuser. Discharge will occur during a five month window beginning during spring runoff when the creek flows are highest. A floating skimmer will be installed. If required, flocculant will be added from the floating skimmer to improve the settlement of any suspended solids before the excess water is discharged.

17.3.13 REAGENTS HANDLING

The reagents used in the process will include:

- Flotation: PAX, 3418A, A208, fuel oil, MIBC, lime (CaO), NaHS, 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, NaOH, activated carbon
- Others: flocculant, antiscalant, H_2O_2.

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, H_2O_2 and antiscalant) will be added in the undiluted form to various process circuits via individual metering pumps.

All the solid type reagents (including PAX, NaHS, Na_2SiO_3 if required, NaOH, NaCN, $CuSO_4$, and MBS) will be mixed with fresh water to 10 to 25% solution strength in the respective mixing tank, and stored in separate holding tanks before being added to various addition points by metering pumps.

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 H_2O_2 WTP located at the plant site. The treated water will be sent to the north or south tailing ponds.

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 dedicated 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 three control rooms:

- Mitchell site primary crusher control room
- MTT train transport 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 in Year 2. The Sulphurets pit crushing control room will be relocated to the Kerr pit crushing plant in Year 20.

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.

An automated train dispatching system will be utilized to achieve a safe and efficient flow of trains through the system, with no on-board operators. The system employing full radio-based train spacing and speed supervision on the whole railway system will be supervised from a control room located in the train maintenance shop. The train control system will operate using a wireless communications system (Wi-Fi) that must be in place for the entire track. While wireless communications are the current state of the art technology for train control communications, it is recognized that more efficient and reliable communications may be developed in the future.

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 sulphur dioxide monitor/alarm system will monitor the cyanide destruction area as well.

17.4 YEARLY PRODUCTION PROJECTION

In general, the Project is designed to use conventional flowsheet and mature technologies for the process plant. The flowsheet proposed is relatively simple and mining will start with conventional open pit operation with an exposed ore body on the surface. The Project plans to use HPGRs for comminution circuit which is expected to have fewer potential rock hardness issues. Cerro Verde mine used a similar comminution and flotation plant initially for its approximately 120,000 t/d plant, that has since been sucessfully expanded to 360,000 t/d. They leveraged experience from the existing operation and ramped up to approximately 95% of the name plate rate in approximately 6 months. Also notable is that larger cone crushers and HPGRs greater than those which KSM will employ have been in operation.

It is estimated that the plant may take approximately twelve months to reach design capacity after the plant is wet commissioned.

According to the metallurgical projections described in Section 13.2 and the current mine 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.

As shown by the test results, it is anticipated that on average the impurity contents in the copper concentrates would be below the penalty limits as outlined for most of the smelters, although in short periods the impurity content may slightly exceed the penalty limits as outlined for some of the smelters. The projected copper concentrate quality is shown in Table 17.3.

In general, the molybdenum concentrate separated from copper and molybdenum bulk concentrate will be leached on site to remove copper, lead and other impurities. The anticipated molybdenum content is approximately 50%. The main impurities such as copper and lead are estimated to be lower than 0.2% and 0.3%, respectively.

Table 17.2 Projected Metallurgical Performance

	Unit	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27
Mill Feed																												
Tonnage, kt	ktonnes	38,050	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,450	47,550	47,626	47,550	47,550	47,550
Grade																												
Au	g/t	0.72	0.82	0.96	0.98	0.71	0.72	0.82	0.78	0.60	0.61	0.58	0.43	0.58	0.70	0.59	0.61	0.50	0.49	0.47	0.52	0.73	0.71	0.52	0.86	0.35	0.34	0.36
Cu	%	0.25	0.28	0.30	0.27	0.18	0.18	0.21	0.20	0.15	0.21	0.23	0.12	0.14	0.21	0.22	0.17	0.11	0.09	0.11	0.12	0.19	0.20	0.15	0.24	0.35	0.53	0.40
Ag	g/t	2.5	2.0	2.3	4.5	2.4	2.2	3.4	2.8	2.1	1.6	0.8	2.1	3.2	2.3	0.7	1.2	2.2	2.6	2.7	2.8	3.4	3.6	2.9	4.4	2.3	2.3	2.0
Mo	ppm	29.8	46.8	61.4	19.8	55.8	65.1	55.6	53.3	82.3	64.2	57.0	91.3	87.9	72.0	66.9	56.5	58.1	78.2	73.1	71.3	55.3	54.0	63.7	34.5	13.6	4.7	5.8
Metal Recovery																												
Copper-Gold Concentrate																												
Au	%	57.4	59.1	61.0	62.5	57.3	57.1	59.2	59.0	53.1	54.6	56.0	49.9	53.3	57.5	56.3	52.5	47.8	45.1	48.1	50.7	58.1	59.0	53.8	62.3	48.9	47.3	49.3
Cu	%	83.9	85.3	86.6	87.7	82.6	82.4	84.5	84.3	79.0	82.1	83.4	75.1	78.6	83.6	83.2	78.4	72.1	70.0	73.2	75.9	83.4	84.3	79.1	87.5	82.1	82.8	82.4
Ag	%	47.8	51.4	54.2	55.0	47.7	47.4	50.4	50.2	41.4	42.2	43.3	37.0	42.0	48.2	45.0	33.5	30.9	29.7	34.3	38.2	48.9	50.1	42.7	54.7	45.7	45.9	47.2
Dore																												
Au	%	17.7	16.4	16.0	16.1	18.6	18.7	17.7	17.6	20.7	18.4	15.9	21.9	20.5	17.3	15.5	17.5	21.3	25.4	23.4	21.9	18.1	17.4	20.0	15.9	22.7	23.7	22.5
Ag	%	16.0	9.8	6.5	16.1	9.3	8.6	14.7	10.7	13.8	10.9	3.0	17.8	20.9	13.0	1.3	16.9	26.2	28.9	25.6	22.1	15.4	15.8	18.4	15.7	14.1	15.0	14.9
Molybdenum Concentrate																												
Mo	%	23.5	31.4	40.7	-	35.0	34.9	34.3	34.8	34.6	32.5	30.1	35.0	35.0	35.0	35.0	35.0	32.1	35.0	35.0	35.0	35.0	35.0	34.8	24.3	27.1	-	-
Production																												
Copper Concentrate																												
Tonnage - Concentrate	ktonnes	312	456	496	441	281	280	330	321	245	344	363	184	233	340	353	254	188	174	161	195	298	316	241	407	523	799	605
Grade - Au	g/t	50.2	50.6	56.0	65.9	68.9	69.2	70.1	68.4	61.7	45.8	42.5	55.3	62.7	56.2	44.6	60.3	60.2	59.9	67.0	64.3	67.2	62.8	55.3	62.5	15.6	9.6	13.9
- Cu	%	25.5	25.0	25.0	25.0	25.0	25.0	25.0	25.0	23.3	24.3	24.9	23.0	23.0	25.0	25.0	24.8	20.1	17.0	23.0	23.0	25.0	25.0	23.1	25.0	25.8	25.9	25.8
- Ag	g/t	145	109	119	269	189	178	246	208	165	92	48	200	270	152	45	75	172	207	276	260	263	273	241	282	94	62	74
Tonnage - Dore																												
- Au	koz	155	206	234	241	202	204	222	211	189	171	141	143	181	185	139	164	162	189	168	174	200	188	159	209	121	124	123
- Ag	koz	488	307	227	1,116	332	291	759	457	435	263	39	567	1,005	446	14	308	884	1,127	1,064	942	795	872	806	1,059	485	516	453
Tonnage - Molybdenum Concentrate																												
Tonnage	tonnes	533	1,394	2,370	-	1,855	2,157	1,810	1,760	2,704	1,981	1,630	3,033	2,921	2,392	2,224	1,878	1,769	2,597	2,428	2,367	1,837	1,795	2,111	797	350	-	-
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	50.0	50.0	50.0	50.0	-	-	-	1.0	2.0

	Unit	Y28	Y29	Y30	Y31	Y32	Y33	Y34	Y35	Y36	Y37	Y38	Y39	Y40	Y41	Y42	Y43	Y44	Y45	Y46	Y47	Y48	Y49	Y50	Y51	Y52	Y53	Total
Mill Feed																												
Tonnage, kt	ktonnes	47,550	47,550	47,550	47,550	47,550	47,550	47,550	47,550	35,001	35,001	35,001	35,001	35,001	35,001	35,001	35,001	35,001	35,001	32,844	28,413	21,860	22,550	22,550	22,550	22,550	22,505	**2,198,559**
Grade																												
Au	g/t	0.35	0.37	0.34	0.33	0.33	0.37	0.44	0.47	0.55	0.54	0.52	0.51	0.49	0.47	0.46	0.46	0.47	0.48	0.49	0.49	0.52	0.47	0.45	0.42	0.40	0.29	**0.55**
Cu	%	0.29	0.41	0.28	0.30	0.22	0.26	0.14	0.15	0.18	0.18	0.18	0.18	0.17	0.17	0.17	0.17	0.18	0.18	0.18	0.18	0.17	0.15	0.13	0.11	0.10	0.12	**0.21**
Ag	g/t	2.2	2.8	2.0	1.8	1.9	1.7	2.5	2.6	2.9	3.0	3.2	3.4	3.7	3.9	4.1	4.2	4.1	3.6	3.2	3.1	2.8	2.6	1.8	1.4	1.1	1.1	**2.6**
Mo	ppm	9.1	11.6	10.1	10.0	10.3	9.8	46.4	41.0	18.8	21.8	23.9	26.5	30.3	34.6	37.9	38.8	38.7	37.0	32.7	30.3	31.4	48.6	27.4	24.0	22.8	27.1	**42.6**
Metal Recovery																												
Copper-Gold Concentrate																												
Au	%	51.2	50.8	50.2	49.9	49.8	51.1	52.0	54.7	61.0	60.4	59.8	59.1	58.3	57.5	57.1	56.9	56.9	56.9	56.8	56.8	56.9	53.1	48.9	44.8	41.9	32.0	**55.3**
Cu	%	81.8	82.5	81.5	81.5	80.7	81.2	76.0	77.6	81.7	81.7	81.8	81.7	81.6	81.4	81.5	81.5	81.7	81.6	81.6	81.5	81.2	77.5	72.0	66.6	62.8	64.0	**81.6**
Ag	%	46.4	45.0	45.3	44.9	43.7	43.9	35.0	37.6	47.5	48.0	49.2	50.2	50.8	51.5	52.7	52.9	51.3	49.0	48.6	49.4	48.9	41.5	43.2	37.6	30.1	18.7	**46.1**
Dore																												
Au	%	21.2	21.5	21.7	21.8	21.7	21.0	20.1	18.4	14.7	14.8	14.8	15.0	15.2	15.4	15.5	15.6	15.6	15.8	15.9	16.0	16.3	18.7	18.5	19.6	20.5	27.7	**18.4**
Ag	%	15.1	17.0	15.8	16.6	15.9	16.5	23.4	21.7	16.8	16.8	16.9	17.3	17.8	18.3	18.3	18.6	19.4	19.2	17.6	16.3	15.5	19.4	11.8	8.5	4.5	16.5	**16.6**
Molybdenum Concentrate																												
Mo	%	-	-	-	-	-	-	24.5	29.3	21.6	19.9	19.5	19.0	19.0	19.2	27.2	27.7	28.3	28.7	20.5	20.8	21.4	30.6	15.1	7.1	1.8	20.3	**30.1**
Production																												
Copper Concentrate																												
Tonnage - Concentrate	ktonnes	449	625	439	460	336	406	217	232	203	202	203	201	198	195	196	198	203	202	189	163	128	112	97	86	79	82	**15,244**
Grade - Au	g/t	18.9	14.3	18.4	17.2	23.1	22.0	50.7	52.7	57.9	56.4	54.2	52.3	50.6	48.8	46.9	46.2	46.4	47.6	48.1	48.8	50.3	50.7	51.6	49.4	47.9	25.4	**43.8**
- Cu	%	25.0	25.8	25.0	25.0	25.0	25.0	22.5	23.1	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	25.0	24.1	22.8	21.1	18.7	17.7	20.7	**24.6**
- Ag	g/t	108	95	97	85	115	88	196	198	236	248	271	301	328	359	386	396	362	303	271	267	235	219	178	135	91	54	**172**
Tonnage - Dore																												
- Au	koz	113	121	112	111	109	118	136	133	91	89	88	86	84	82	80	81	83	86	82	72	59	64	61	60	59	58	**7,126**
- Ag	koz	506	718	478	466	450	433	912	851	545	562	606	667	735	798	848	888	894	772	598	462	308	368	152	84	35	125	**30,321**
Tonnage - Molybdenum Concentrate																												
Tonnage	tonnes	-	-	-	-	-	-	1,080	1,140	285	304	325	352	404	465	721	752	765	744	440	358	293	670	186	77	19	248	**56,318**
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	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	50.0	**50.0**

Table 17.3 Projected Copper Concentrate Quality

Element	Unit	Content		
		Range	Years 1 to 10 Average	LOM Average
Cu	%	23 – 28	25	25
Au	g/t	28 – 109	61	43
Ag	g/t	123 – 505	174	190
Mo	%	0.03 – 0.21	0.15	0.10
S^T	%	28 – 41	34	34
S^{-2}	%	26 - 36	32	32
Fe	%	24 – 35	29	29
Sb	ppm	199 – 1,966	993	1,008
As	ppm	460 – 2,760	1,769	1,403
Co	ppm	42 – 97	47	61
Cd	ppm	33 - 172	88	92
Bi	ppm	15 – 156	27	48
Hg	ppm	1.2 – 10	2.8	3.6
Ni	ppm	49 – 233	76	91
F	ppm	75 - 399	241	193
Cl	%	0.01 – 0.02	0.01	0.01
Se	ppm	62 – 148	74	95
P	ppm	67 – 536	156	194
Pb	%	0.1 – 1.0	0.5	0.4
Zn	%	0.2 – 0.9	0.4	0.5
SiO_2	%	2.3 – 11	6.4	5.7
CaO	%	0.2 – 0.8	0.5	0.5
Al_2O_3	%	0.5 – 3.9	2.1	1.7
MgO	%	0.1 – 0.5	0.3	0.3
MnO	%	0.01 – 0.04	0.01	0.02
InSol	%	3.1 – 12	-	-

18.0 PROJECT INFRASTRUCTURE

18.1 SITE LAYOUT

There will be two separate areas of infrastructure associated with the Project: the Mine Site and the PTMA. The Mine Site is the center of mining activity and includes the primary crushing facilities (Mitchell OPC). Process facilities will be located at the Treaty OPC in the PTMA, approximately 23 km northeast of the Mine Site. Twinned tunnels (the MTT) will extend from the north side of the Mine Site into the upper reaches of the Treaty OPC. Along the MTT route there is a topographical low (valley) that is designated the Saddle Area, approximately 17 km from the Mitchell portal, where the MTT will be accessed via a construction adit.

18.1.1 MINE SITE LAYOUT

The Mine Site layout at the end of construction is shown in Figure 18.1. Individual facilities will be situated to take advantage of the natural topography in order to minimize impact on the environment. The MTT will be constructed to connect the Mine Site and the PTMA (see Section 18.4 for MTT details). Crushed ore for processing will be transported via the ore train system from the Mine Site to a stockpile at the Treaty OPC.

At the Mitchell pit, the crushed ore from two gyratory crushers will be conveyed to two ore surge bins located underground and adjacent to the MTT. The crushed ore will be loaded into ore train cars by two automatic train loading systems.

The Mine Site area will include additional infrastructure such as the initial staging, construction, and operations camps; truck/maintenance shop; explosive facilities; WSD; diversion tunnels; and power plants. Access and haul roads will be provided to all of these areas.

The WSF provides environmental containment of runoff water for the Mine Site. To achieve this, the WSF includes a rock fill-asphalt core WSD to collect contact water from the Mine Site for treatment at the HDS WTP.

The HDS WTP, Selenium WTP and the Energy Recovery Power Plant will be situated in the lower Mine Site area. The Energy Recovery Power Plant will use water pumped from the WSF to its crest that flows downhill from the WSF to the HDS WTP to generate electric power.

The HDS WTP will treat:

- open pit mine drainage from the Mitchell, Kerr, and Sulphurets pits
- water collected from pit slope vertical wells and horizontal drains
- MTT drainage
- surface drainage waters from the WSD and RSFs.

The Selenium WTP will collect and treat:

- seepage from the Mitchell and McTagg Waste Rock Piles
- Kerr waste pile placed within the Sulphurets pit
- water from with the WSF.

18.1.2 PROCESSING AND TAILING MANAGEMENT AREA LAYOUT

The PTMA includes process facilities, TMF, a construction and operations accommodation complex; administration, maintenance and support facilities; and related ancillary buildings. The Treaty OPC area is shown in Figure 18.2 and the TMF area is shown in Figure 18.3.

The Treaty OPC will be slightly terraced, and the site roads will be constructed from the Process Plant to the MTT Treaty portal exit elevation and to the TMF. All terracing quantities have been based on geotechnical information collected for the Treaty OPC.

The main process facilities at the Treaty OPC include:

- a COS with a live capacity of 60,000 t
- secondary crushing by cone crushers
- a fine ore stockpile with a live capacity of 30,000 t
- tertiary crushing by HPGR
- a primary grinding and flotation facility, including concentrate regrinding and dewatering
- a gold CIL cyanide leaching facility, including gold recovery from loaded carbon
- cyanide recovery and cyanide destruction of washed CIL residue prior to disposal of the residue in the lined pond within the TMF
- a concentrate storage and loadout facility
- tailing delivery system
- process related service facilities, including assay and metallurgical laboratory, water services, power supply.

The TMF is located in a valley comprising the upper catchments of North Treaty and South Teigen creeks, southeast of, and adjacent to the Treaty OPC. The TMF catchment will provide an initial start-up water volume for two months of mill water make-up stored behind the starter dams before mill start up. It will also provide a minimum water depth for floating of reclaim barges and achieve adequate water clarification for process purposes.

TMF structures at start up include three starter dams defining the North Flotation Tailing Cell and the CIL Residue Tailing Cell. Perimeter diversions are provided to dewater the area for construction and to reduce inflows during operation. By Year 25, the South Flotation Tailing Cell is required and this is formed by adding the Southeast Dam and the East Catchment Diversion Tunnel.

Two tailing energy recovery plants will be located at separate locations straddling the North Dam tailing line between the Process Plant and the TMF. Each energy recovery 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 PTMA. Process water supply will be reclaimed water from the TMF and fresh water will be provided 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 in conjunction with the Treaty OPC terracing.

Figure 18.1 Mine Site Layout after Initial Construction

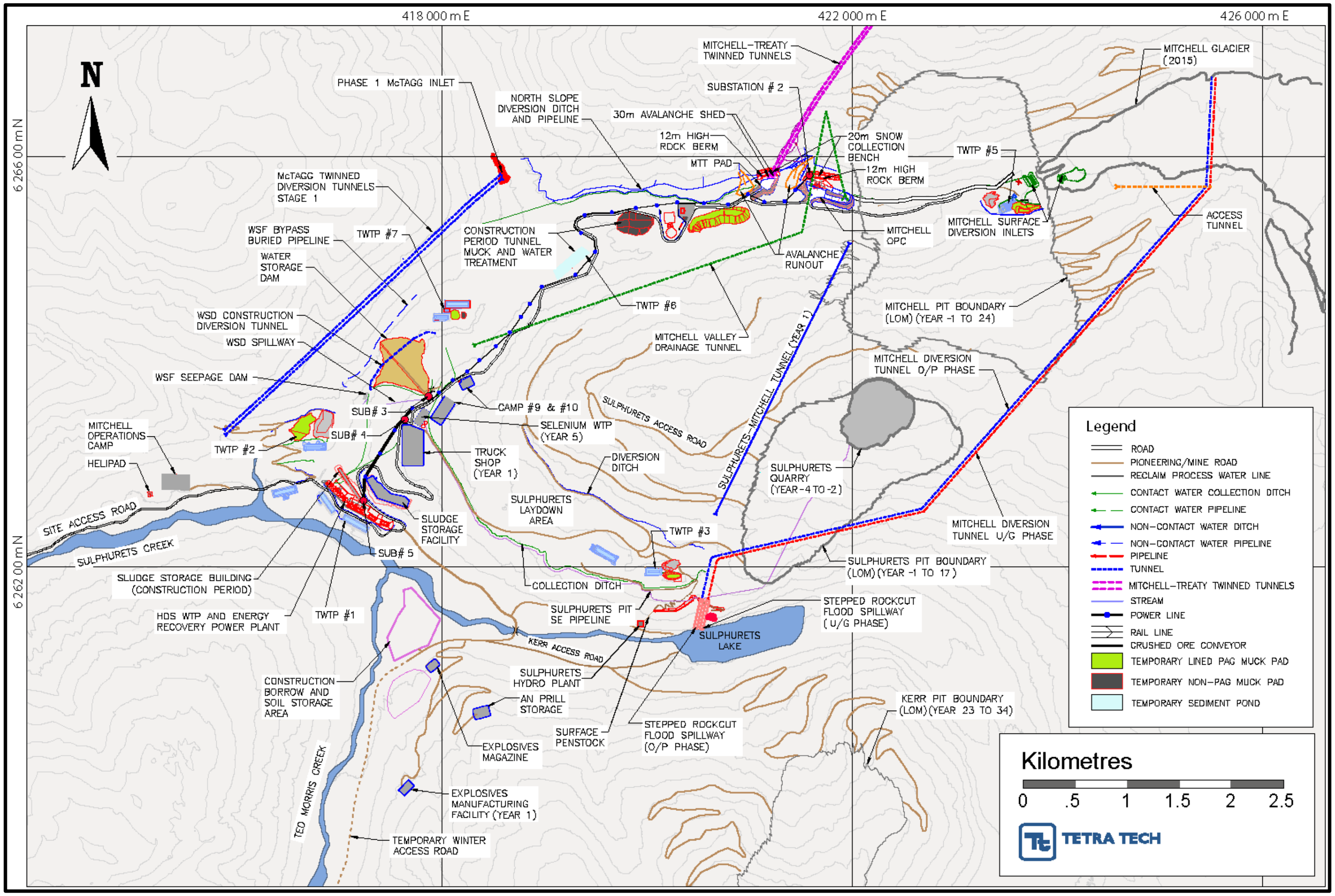


Figure 18.2 Treaty Ore Preparation Complex Layout

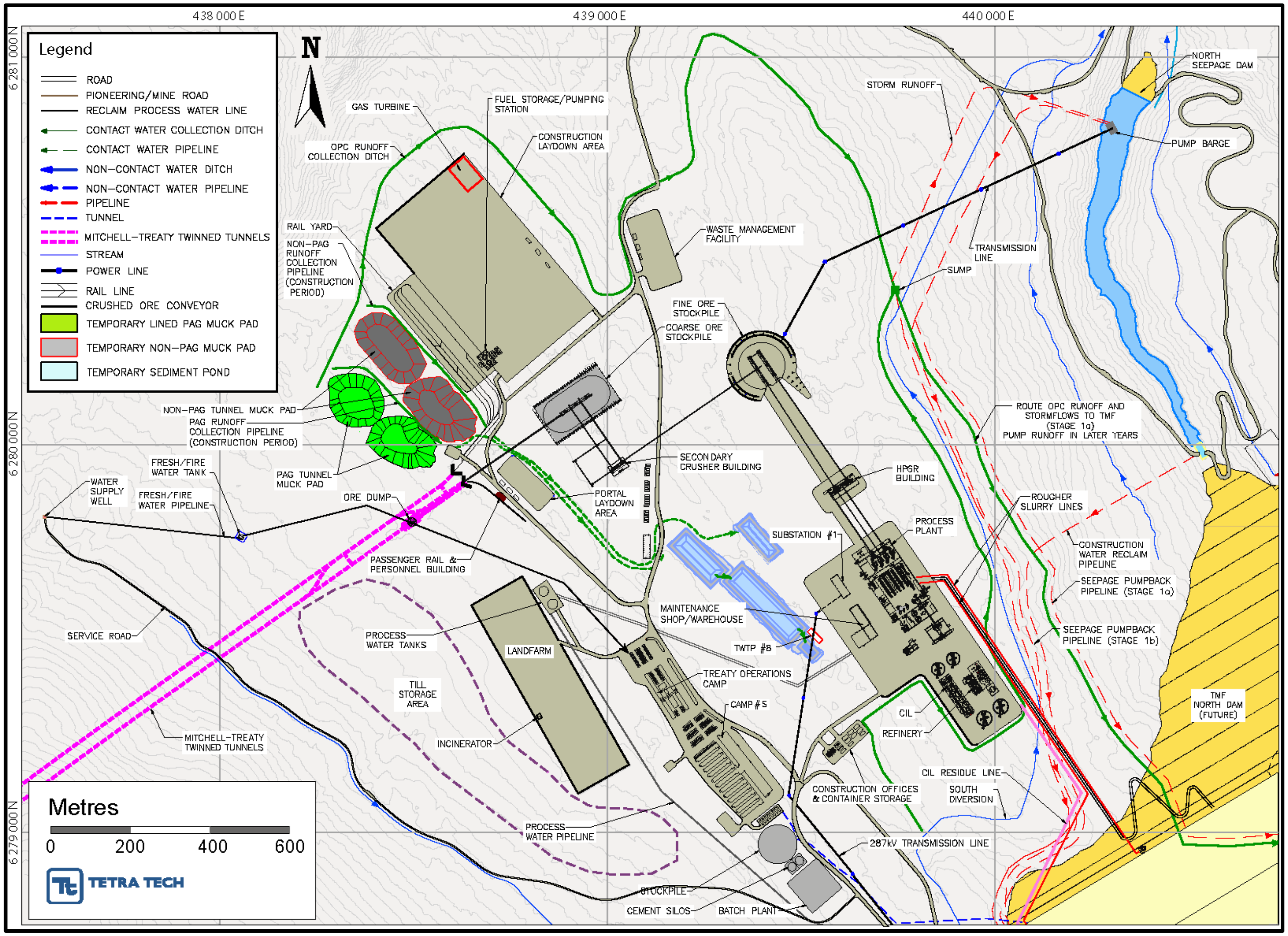
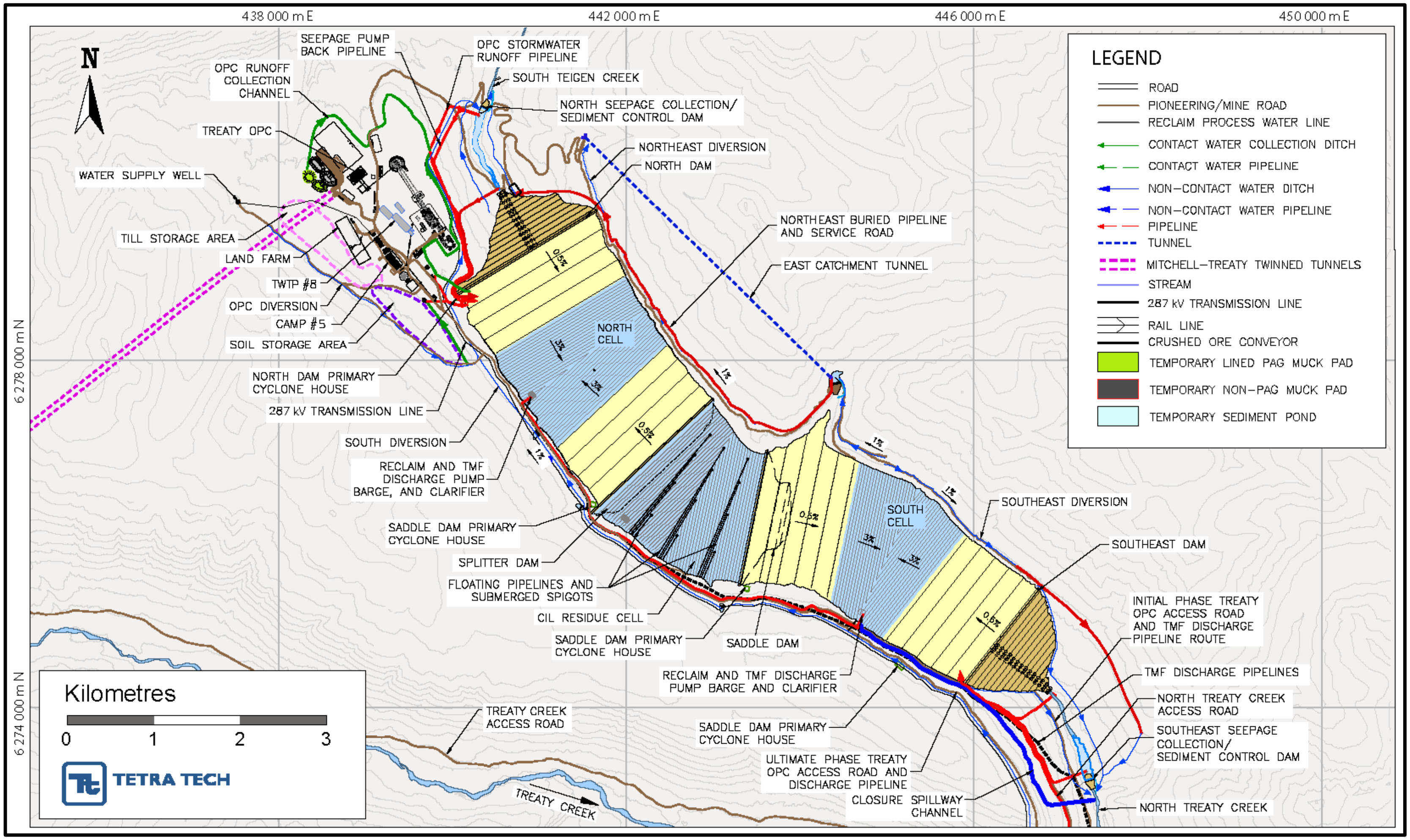


Figure 18.3 Ultimate TMF Layout



18.2 TAILING, MINE ROCK, AND WATER MANAGEMENT

18.2.1 INTRODUCTION

This section addresses 2016 updates to the geotechnical designs for tailing and mine rock management, as well as for site-wide water management. The 2016 design updates incorporate incremental improvements and changes to water management structures in the Mine Site and TMF areas, in response to commitments made during the EA process review.

Design changes to the TMF include:

- addition of a discharge pipeline system and a diffuser located in Treaty Creek to route operational-period discharges to Treaty Creek
- addition of the North Cell Closure Spillway
- addition of three energy dissipation dams to facilitate road crossings and to contain potential debris flows at significant stream crossings along the South Diversion Channel (this channel follows the TCAR to the Treaty OPC)
- relocation of the TMF seepage collection dams downstream to more effectively intercept seepage.

Design changes to Mine Site facilities include:

- relocation of MDT Inlets upstream to improve diverted water quality, and improved inlet design
- a shift of the WSF CDT outlet 200 m upstream to avoid an avalanche area, and design of a closure gate and permanent tunnel plug for the WSF CDT
- modification of discharge from the WSF to use pumped discharge in lieu of discharge pipes passing under the WSD
- expansion of the water treatment plant to a maximum capacity of 7.5 m^3/s treatment capacity once the HDS WTP is fully built
- improved water treatment discharge strategy designed to mimic natural flows within the Sulphurets drainage basin
- addition of a Selenium WTP
- placement of the Kerr waste rock in the Sulphurets Pit
- expanding the contact water management systems to handle a 200-year flood
- the inclusion of the MVDT
- design updates for the Sludge Storage Facility and Water Treatment Sludge Storage Building.

KCB re-assessed additional site climate and hydrology data recorded through 2015. These analyses determined similar values to those adopted for the 2012 PFS

(KCB 2012, KCB 2013a, Tetra Tech 2012) for an average year. As a result, the water management design basis remains unchanged from 2012.

Figure 18.1 and Figure 18.3 show the layouts for the updated Mine Site facilities and the updated TMF facilities, respectively.

Details of TMF, RSF, and SWM prefeasibility design updates for the Project are provided in the following 2016 KCB reports located in Appendix H:

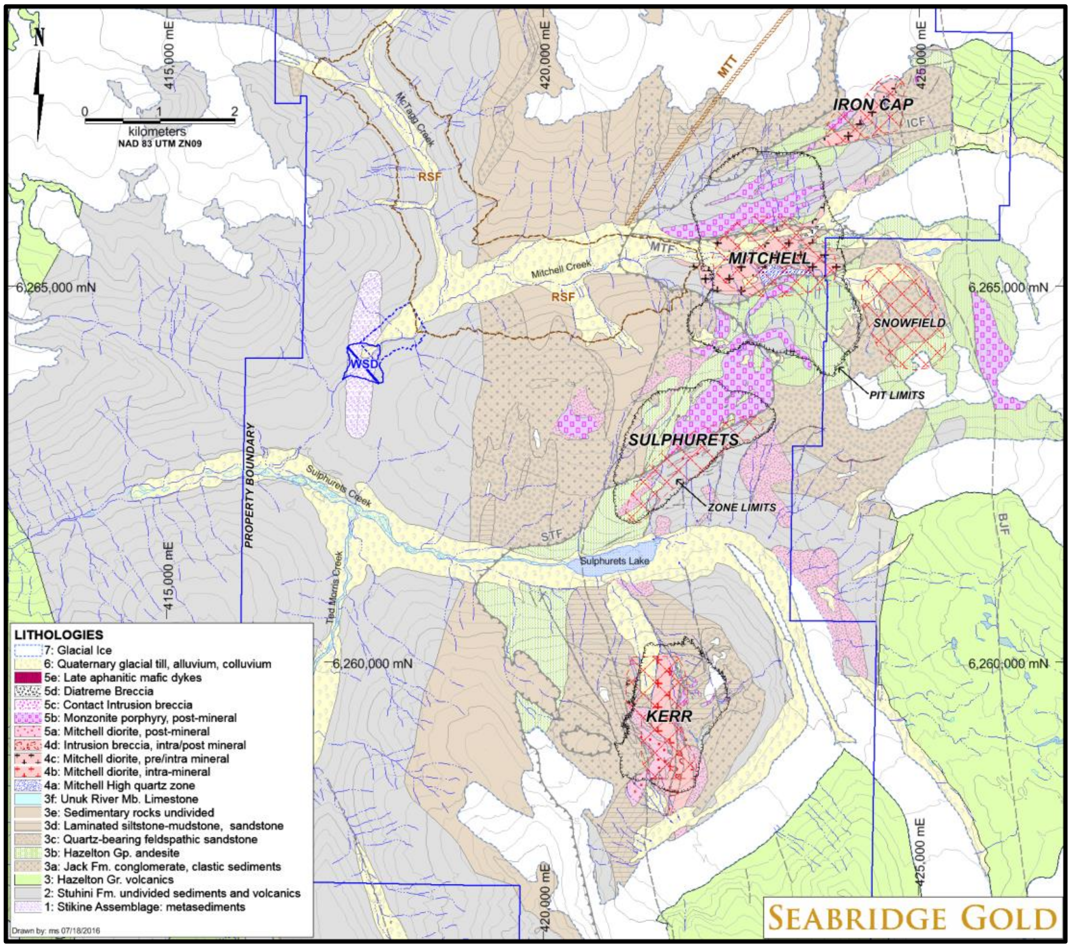
- Mine Area Water Management Addendum Report
- Tailing Management Facility Design Addendum Report
- Rock Storage Facility Addendum Report
- Tunnels and Temporary Water Treatment Addendum Report.

18.2.2 MINE SITE CHARACTERIZATION

No additional geotechnical site investigations have been completed for the WSD or RSF areas since the 2012 PFS (Tetra Tech 2012). The 2012 PFS provides a results summary of KCB Mine Site geotechnical site investigations completed up to 2012 (KCB 2009; 2010; 2011; and 2012).

In 2015, Seabridge completed a regional geological mapping update that resulted in a revised Mine Site geology map, presented in Figure 18.4. The mapping program provided additional information on the distribution of NPAG monzonite available within the Sulphurets pit area. Acid-resistant monzonite rock stripped from this area for development of Sulphurets pit will be used for upstream rock fill in the WSD. Local fill from the excavation of the WSD spillway and road and diversion cuts adjacent to the dam will be used for downstream rock fill.

Figure 18.4 Mine Site Mapped Geology



MINE SITE CLIMATE

Much of the Project site annual precipitation occurs as snowfall 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 region in the Sulphurets watershed (Mine Site) and the eastern region in the Treaty-Teigen watersheds (PTMA). The two regions are 23 km apart and have differing climates. The two areas are separated by the Johnstone Icefield (ranging from 1,800 to 2,200 m in elevation).

Significant orographic and rain shadow effects were recorded in the KSM area as part of the 2012 baseline study. In 2012, KCB and ERM performed extensive analysis of climate variations in the Project area for engineering design and EA purposes (Rescan 2013). Algorithms were developed based on the UBC watershed model to estimate effects of variation in precipitation with altitude, and to adjust glacier and snow melt rates in response to climatic variations.

In 2016, metrological and hydrological data collected since 2012 was reviewed (Appendix H). The result is that no significant trends have been observed in the additional data for average site parameters, as compared to the data prior to 2012. It is concluded that no adjustments are required in design assumptions and that the 2012 PFS climate assessment is still valid.

Mine Site Temperature

Weather data recorded at the Sulphurets weather station between 2007 and 2015 indicate the following:

- mean annual temperature is approximately 0°C
- mean monthly temperatures range from -13°C in December to 14°C in July
- temperature extremes range from -31°C to 30°C
- mean daily temperatures are above freezing from May to October and below freezing from October to May.

Canadian Metrological Service data indicates that frost penetration for the area is typically 1.5 m.

Mine Site Precipitation and Hydrology

The estimated mean annual precipitation is 1,652 mm at Sulphurets weather station (elevation of 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. 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, as well as the runoff distributions for the glacier catchments, is provided in Table 18.1.

Precipitation listed in Table 18.1 is representative of the Sulphurets weather station at 880 masl and is derived from site data between 2008 and 2011. Additional site data recorded since 2012 has not changed the values adopted for design. Assessment of tipping bucket rain gauge data indicates that precipitation increases at higher elevations within the Mine Site 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 Site and PTMA. These reports and present assessments of the additional climate and hydrological data obtained since 2012 (Appendix H).

Table 18.1 Climate Data for the Mine Site (Sulphurets Creek Climate Station)

Data Area	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**
Pond Evaporation (mm)[2]	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**

Notes: [1]Weather station at 880 masl elevation.
[2]Estimated pond evaporation based on pan evaporation data.

MITCHELL GLACIER RECESSION

Seabridge has been monitoring the recession of site glaciers by analyzing historical air photos, Light Detection and Ranging (LIDAR) data, ongoing global positioning system (GPS) and remote sensing measurements of the glacier extents. Glaciers in the Project area continue to recede; recession rates for the Mitchell Glacier toe area recession since 2008 have reached as much as 65 m/a and total recession has exceeded 2012 estimates. As a result, the Mitchell pit area is now ice free. The locations of the initial stage proposed surface contact water inlets in the toe area of Mitchell Glacier are now also free of ice.

18.2.3 TMF SITE CHARACTERIZATION

TMF SITE INVESTIGATIONS

No additional site investigations have been conducted at the TMF since the 2012 PFS (Tetra Tech 2012).

The 2012 PFS summarizes site conditions at the TMF used for the basis of the TMF designs. The 2012 PFS provides a results summary of TMF and Treaty OPC geotechnical site investigations completed up to 2012 (KCB 2008, 2009, 2010, 2011 and 2012b).

TAILING CHARACTERIZATION AND LABORATORY TESTING

No additional tailing testing has been conducted for TMF design purposes since 2012. The following section summarizes the test work completed to 2012 (KCB 2012c).

The Treaty Process Plant will produce two tailing streams: 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 using the CIL method and then processed for gold recovery. A two-stage cyanide destruction circuit is proposed, using the Inco sulphur dioxide process, followed by 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 submitted to KCB by G&T.

During 2011/2012, KCB completed an additional program of tailing geotechnical 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 this testing program show 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]This stream is referred to as "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: 80% passing 110 µm
- CIL residue tailing: 80% passing 20 µm
- cyclone sand: 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.

The flotation tailing is classified as 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 residue tailing is a high-sulphide concentration material and is classified as PAG. This material will be deposited under water in the CIL Residue Storage Cell in the centre of the TMF and kept saturated to mitigate against the onset of acid generation.

The 2011, the KCB laboratory testing program assessed 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

Additional climate and hydrological data collected since 2012 was reviewed and the conclusion is that no significant trends in average parameters are apparent. As a result, the climate and hydrology parameters used for design of the TMF in 2012 have been retained.

TMF Area Temperature

Weather data recorded at the TMF area from the Teigen Creek weather station between 2009 and 2011, with additional data now available through 2015, shows the following:

- mean annual temperature is approximately 0°C
- mean monthly temperatures range from -8°C from December to February, to 11°C in July
- temperature extremes range from -27°C to 29°C
- mean daily temperatures are above freezing from May to October and below freezing from October to May.

TMF Area Precipitation and Hydrology

Based on correlations to longer-term weather data at the Snowbank Road station and Eskay Creek Mine, as well as 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 18.2. Precipitation increases at higher elevations within the Teigen/Treaty valleys at a nominal rate of 5% per 100 m.

Table 18.2 Climate Data for the TMF (Teigen Creek Climate Station)[1]

Data Area	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Total
Precipitation (mm)	151	110	123	82	55	69	82	82	165	206	164	82	**1,371**
Pond Evaporation (mm)[2]	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**

Note: [1]Weather station at elevation 1,085 masl.
[2]Estimated pond evaporation based on pan evaporation data.

TMF AREA HYDROGEOLOGY

No additional hydrogeological testing or analysis has been conducted since the 2012 PFS (Tetra Tech 2012).

Details of the hydrogeology and groundwater modelling (i.e. FEFLOW® and Seep/W© models) of the TMF area and the dams are reported in the 2012 TMF Engineering Design report (KCB 2012c).

18.2.4 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 (KCB 2013a). The seismic hazard assessment was conducted in accordance with CDA (2007and 2013) recommendations for seismic hazard assessment. No additional analysis has been conducted since these assessments.

The PGAs listed in Table 18.3, derived from the seismic hazard assessment, are recommended for both the TMF and RSF sites. The assessment identified that a 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 from the assessment as listed in Table 18.4.

Table 18.3 Calculated Design PGAs for TMF and RSF Sites

Return Period (a)	PGA (*g*)
475	0.04
975	0.05
2,475	0.08
10,000	0.14

Note: *g* = gravitational constant

Table 18.4 10,000-year Return Period Spectral Accelerations for the TMF Site

Period (s)	Spectral Acceleration (5% Damped) (*g*)
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

18.2.5 Design Criteria

RSF and Mine Site Water Management Structure Design Criteria

Design criteria adopted for the 2016 PFS design of the Mine Site facilities are largely unchanged from the 2012 PFS (Tetra Tech 2012) and are summarized as follows:

- The LOM has been adjusted from 52.5 years to 53 years by revising the mine plan.
- Over a 53-year LOM, the production of 3 Bt of mine rock from three open pits will be stored in the Mitchell, McTagg and Sulphurets pit backfill RSFs.
- All RSFs will be placed bottom up.
- The criteria for dam safety and flood management in Table 18.5 are assessed for the WSD and the WSF Seepage Dam based on the 2007 Dam Safety Guidelines (CDA 2007). These are unchanged from the 2012 PFS (Tetra Tech 2012).

Table 18.5 WSD and WSF Seepage Dam Safety and Flood Management Criteria

Criteria	WSD		WSF Seepage Dam
CDA Consequence Category	Very High		Significant
Seismic	PGA of 0.14 *g* (10,000-year event)		PGA of 0.14 *g* (10,000-year event)
Diversions	200-year 24-h average daily flow		200-year 24-h average daily flow
Environmental Design Flood (EDF) and Storage	200-year return period 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 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 >1.5	FOS >1.5
	End of construction:	FOS >1.3	
	Rapid drawdown:	FOS >1.2	
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

Note: *WSD spillway designed to route PMF with diversions failed

The criteria shown in Table 18.6 have been used for the design of the RSFs, in conjunction with the guidelines developed by the BC Waste Dump Research Committee (1991-1995) and the Health, Safety, and Reclamation Code for Mines in BC.

Table 18.6 Geotechnical Stability Design Criteria for Key Areas of RSFs

Criteria	Region of the RSF		
	Above the WSF	Above the Mitchell Pit & Infrastructure	Areas of McTagg & Mitchell Valleys not above WSF or Infrastructure
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, short term and >1.4 long term 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 2016 PFS design of the TMF are largely unchanged since the 2012 PFS (Tetra Tech 2012) and are summarized as follows:

- Production schedule:
  - The LOM is 53 years with a maximum milling rate of 130,000 t/d of ore production, for a total of 2.2 Bt of tailing, which is the same total amount of tailing as in 2012. The average dry tailing density in the ultimate impoundment will be 1.5 t/m³. Although total ore milled will be 2.2 Bt, a TMF capacity of 2.30 Bt was selected to provide contingency storage.
- Tailing production:
  - Flotation tailing production will be 89.5% of ore production. Sufficient sulphide flotation will be achieved in the mill process to achieve NPAG behavior.
  - CIL residue tailing production will be approximately 10% of ore production. High-sulphide concentration results in PAG behavior. Design of the CIL Residue Storage Cell allows for storage of 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 5×10^{-6} m/s.

The criteria presented in Table 18.7 for tailing dam safety and flood management are assessed based on the 2007 Dam Safety Guidelines (CDA 2007) and are unchanged for 2016, except for the addition of discharge capacity for the critical duration PMF.

Table 18.7 TMF Dam Safety and Flood Management

Criteria	North Dam, Saddle Dam (Stage 1), Southeast Dam	Splitter Dam, Saddle Dam	Seepage Dams (North, Saddle, Southeast)
CDA Consequence Category	Extreme	Significant	Significant
Seismic	PGA of 0.14 g (1:10,000 year event)	PGA of 0.08 g (1:2,475 year event)	PGA of 0.08 g (1:2,475 year 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
Environmental Design Flood and Storage	Capacity to discharge Critical Duration PMF during operations via diffuser. Seasonal surplus flotation cell water will be discharged directly into Treaty Creek or treated	Seasonal surplus CIL Cell water will be reclaimed preferentially.	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 Routing	30-day PMF with 100-year snowmelt can be stored in the TMF without discharge, with diversions failed. Capacity to route PMF in Closure Spillway	30-day PMF with 100-year snowmelt can be stored in the TMF without discharge, with diversions failed. Capacity to route PMF in Closure Spillway	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 non-compacted tailing are liquefied	FOS >1.2 assuming all non-compacted tailing are liquefied	FOS >1.2

The TMF reclaim ponds are designed to:

- provide one month of mill start-up water
- provide a minimum water depth for floating a reclaim barge and achieving water clarification
- provide required water cover for the CIL residue tailing to prevent oxidation.

18.2.6 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. MMTS designed the RSF layouts, with geotechnical guidance on slope stability and geotechnical recommendations from KCB.

The RSFs will be built in progressive lifts (bottom-up construction) to initially confine toe areas and consolidate foundations to improve stability and reduce downslope risks.

To meet Project commitments during the Application/EIS (Rescan 2013), and since the 2012 PFS, a design has been advanced to collect seepage high in selenium from the Mitchell and McTagg RSFs. Stability analyses were re-run with the Selenium Seepage Collection System present and the RSFs meet the stability criteria. KCB provides details on the design of the Selenium Seepage Collection System in Appendix H.

A rendering of the ultimate mine site layout, including the McTagg and Mitchell RSFs is provided in Figure 16.13.

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 will be 26° (2H:1V) to facilitate final closure.

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 the Mine Site). 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 end-dumping coarse competent rock such that self-segregation forms permeable drain layers.

RSF Stability Analyses

RSF geotechnical stability during construction and closure was analyzed during 2012 and 2014, and again in 2016 with the addition of the Selenium Seepage Collection System. A wide range of potential RSF configurations were analyzed.

With the inclusion of the Selenium Seepage Collection System, all target FOS have still been met within the RSF.

RSF Safety and Monitoring

Mine staff will perform routine inspections of the RSF during construction, looking for indications that would provide advanced warning of a potential RSF failure:

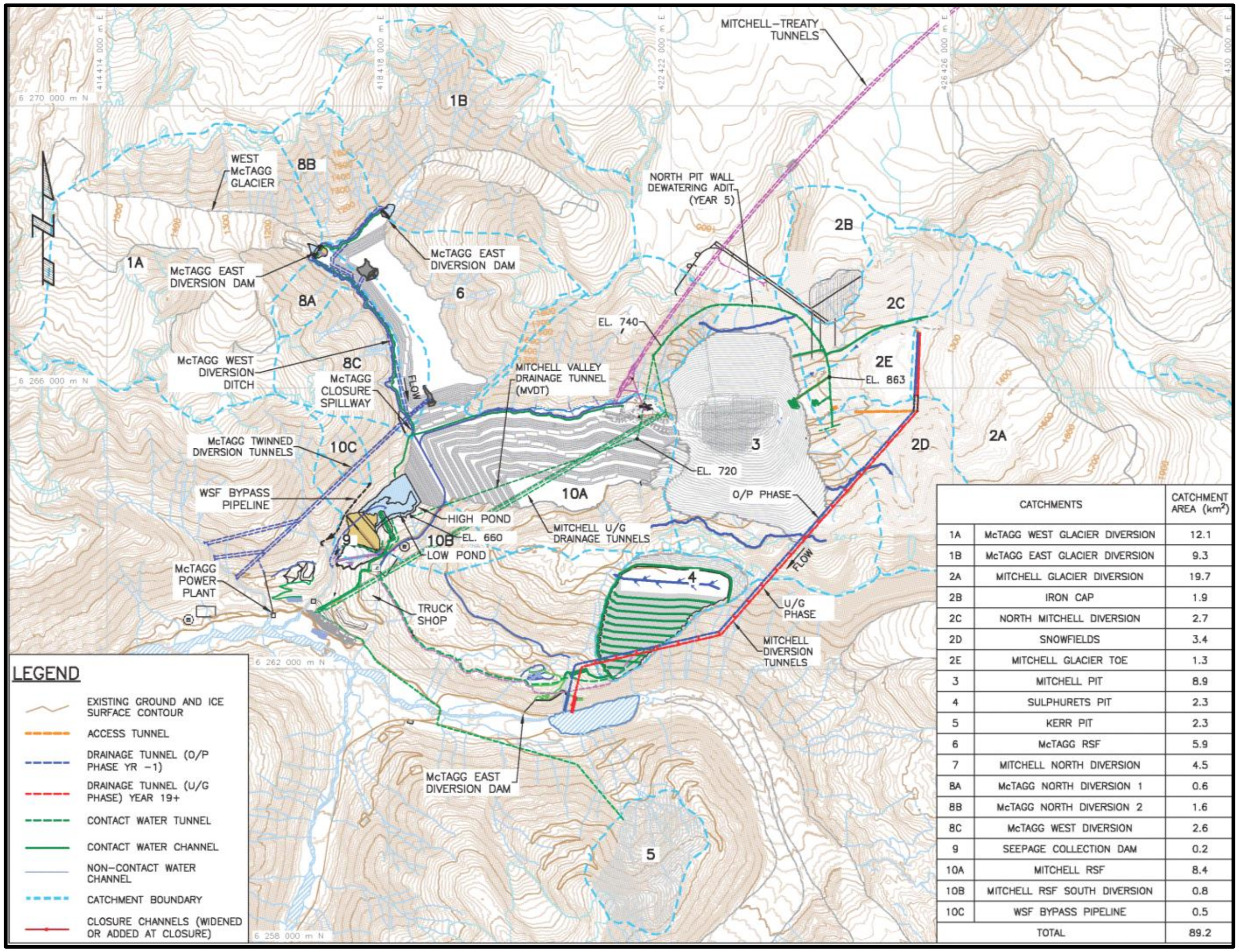
- readings of electric piezometers installed in foundation silt and clay layers
- readings of inclinometers installed in areas close to the Mitchell OPC facilities and the area facing the water storage pond
- monitoring of surface survey monuments installed during RSF construction; the use of automated radar or optical scanning instruments will be considered for areas with higher downslope hazards.

18.2.7 MINE SITE WATER MANAGEMENT

The overall SWM Plan has not substantially changed since the 2012 PFS (Tetra Tech 2012); however, several changes have been made as a result of the Application/EIS (Rescan 2013). These include moving the MDT Inlets upstream to areas of unaltered geology so as to improve diverted water quality, and the provision of the MVDT to route contact water around the Mitchell RSF.

Figure 18.5 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 with blue dashed lines.

Figure 18.5 Mine Site Ultimate Water Management Facilities



	CATCHMENTS	CATCHMENT AREA (km^2)
1A	McTAGG WEST GLACIER DIVERSION	12.1
1B	McTAGG EAST GLACIER DIVERSION	9.3
2A	MITCHELL GLACIER DIVERSION	19.7
2B	IRON CAP	1.9
2C	NORTH MITCHELL DIVERSION	2.7
2D	SNOWFIELDS	3.4
2E	MITCHELL GLACIER TOE	1.3
3	MITCHELL PIT	8.9
4	SULPHURETS PIT	2.3
5	KERR PIT	2.3
6	McTAGG RSF	5.9
7	MITCHELL NORTH DIVERSION	4.5
8A	McTAGG NORTH DIVERSION 1	0.6
8B	McTAGG NORTH DIVERSION 2	1.6
8C	McTAGG WEST DIVERSION	2.6
9	SEEPAGE COLLECTION DAM	0.2
10A	MITCHELL RSF	8.4
10B	MITCHELL RSF SOUTH DIVERSION	0.8
10C	WSF BYPASS PIPELINE	0.5
	TOTAL	89.2

After mining is complete, perimeter closure channels will be constructed at the top and margins of the RSFs, and the channels widened for closure along the toes of the RSFs. Operational phase channels are shown on Figure 18.5; the closure routings of fresh (non-contact) water aren in blue and the routing of contact water is in green.

Diversion Tunnels and Surface Diversions

The MDT Inlets have been shifted approximately 500 m upstream to improve water quality of the diverted water. As a result of the Mitchell Glacier Diversion Optimization study (Appendix H), the noncontact inlets of the Mitchell tunnels were simplified. The open pit phase non-contact water MDT was enlarged to allow a single tunnel to carry design diversion flows. Upon commencement of underground mining, the open pit phase tunnel is twinned with a second tunnel of the same dimensions to increase diversion capacity. There have been no changes to the design of the MTDTs or the Mine Site surface diversions since the 2012 PFS (Tetra Tech 2012).

Contact Water Collection Systems

There has been no change in the surface contact water collection systems since the 2012 PFS (Tetra Tech 2012).

To reduce water flow through the base of the Mitchell RSF and facilitate routing of contact water around the Mitchell RSF, the MVDT was added after the 2012 PFS (Tetra Tech 2012). This is a 5 km long, 5 m by 6 m tunnel that drains to the WSF. The tunnel connects to the NPWDA, which is added in Year 5, to accept pit wall drainage and local drainage of contact water from upstream of Mitchell pit and from the Snowfields area.

The 2016 PFS configuration for the Mitchell and McTagg RSFs includes a Selenium Seepage Collection System that is designed to collect up to 500 L/s of seepage and convey this flow to the Selenium WTP. Water from other sources will be treated when seepage collected from the RSFs is less than 500 L/s. The collection and treatment of seepage from these RSFs, and other high-selenium loading waters, will enable selective removal of selenium from flows with higher selenium concentrations, compared to lower concentrations within the WSF. The Selenium Seepage Collection System will be constructed in Year 5.

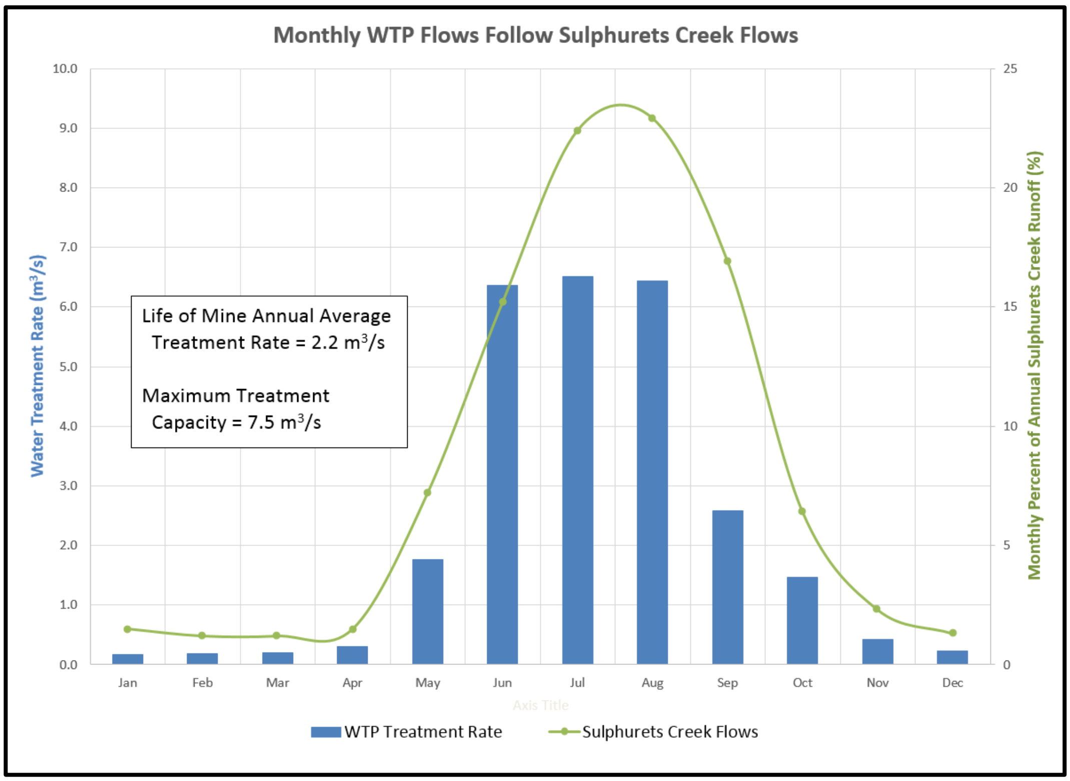
The Selenium Seepage Collection System is composed of a 60 mL LLDPE liner placed on a graded surface within the RSFs that drains to the western toe area at elevation 706 m. The liner will be keyed into natural ground to facilitate collection of seepage from the lateral areas of the RSF. Drainage of seepage from the RSF into the collection system is facilitated by a drain layer composed of competent drain rock stripped from the development of Sulphurets pit. The drain rock overlies a layer of moraine sands and gravels that protects the liner beneath the drain from damage. The slope of the drain directs seepage into a system of perforated pipes near the RSF toe, gathering flow for pumping to the Selenium WTP. The Selenium WTP will discharge to the WSF to allow further treatment for metals removal.

Water Storage Facility

Seepage from the Mitchell RSF, McTagg RSF, and Sulphurets Pit Backfill RSF requiring treatment for the removal of metals by the HDS process will be collected in the lower Mine Site by the WSD. The WSD is an asphalt core rock fill dam that will create the WSF pond, and will be large enough to handle seasonal freshet flows as well as volume accumulated from a 200-year wet year.

Figure 18.6 shows monthly average water treatment rates for flows from the WSF as blue bars plotted over the LOM. The installed ultimate HDS water treatment capacity of 7.5 m^3/s is greater than the annual average or monthly peak flows to allow treatment rate to vary seasonally with stream flow rates.

Figure 18.6 Mine Site Monthly Water Treatment Rate



The WSD design has been updated since the 2012 PFS (Tetra Tech 2012). Dam slopes were revised after the 2012 PFS based on a value engineering study (KCB 2012b). The internal zonation of the WSD was also updated (KCB 2012d). The revised slopes and zonation of the dam are shown in section view in Figure 18.7.

The WSD will be located in the lower Mitchell Creek area and founded on competent rock; unchanged from the 2012 PFS (Tetra Tech 2012). The WSD crest elevation is also unchanged from the ultimate dam height in the 2012 PFS (Tetra Tech 2012) and will be established at the full height of 716 masl (165 m height) before Year 1. An emergency spillway designed to route the IDF will be cut into rock on the southeast side of the dam. There will be appropriate freeboard for avalanche wave mitigation and flood routing.

Water in the WSF is predicted to be acidic, similar to existing seeps situated in upper Mine Site. An asphalt core will be included in the dam to control seepage. Asphalt is inert with respect to acidic water. To control seepage, the WSD and WSF Seepage Dam foundations will be grouted. Based on drilling results, the depth of the WSD grout curtain is designed to vary from 25 m at the west abutment to as deep as 150 m at the east abutment if required. Grout hole spacing will be 2.5 m.

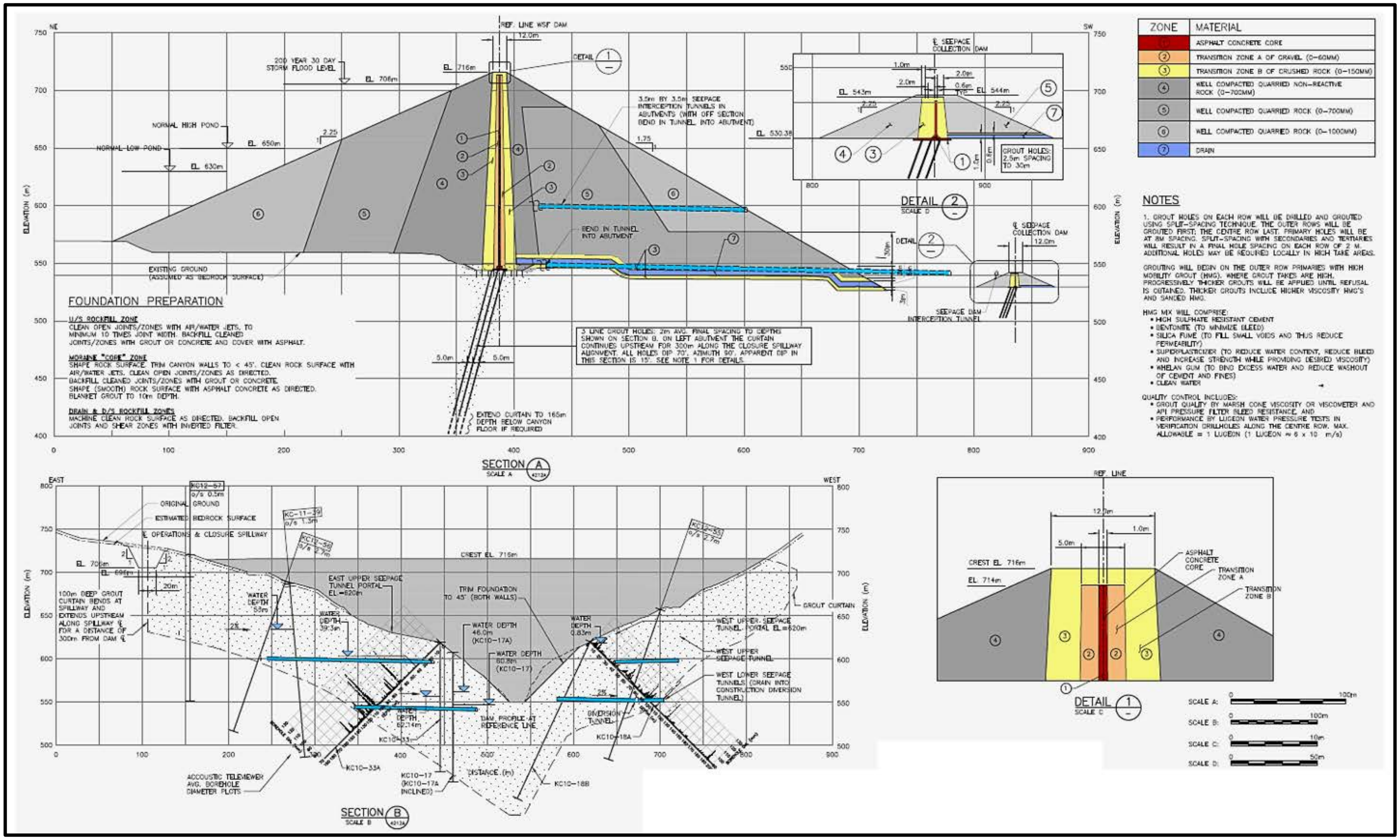
Fill for dam zones is specified such that critical zones of the dam (sections in contact with the core) and drain zones will be constructed with materials that have low potential to react with acidic water. This material will be sourced from stripping of Sulphurets pit.

The WSF discharge system has been modified to consist of submersible pumps mounted in inclined carrier pipes on the southeast bank of the WSF pond. These pumps will discharge to a head pond located above WSD crest elevation. A HDPE-lined steel penstock will lead from the outlet of the head pond to the Energy Recovery Power Plant and the HDS WTP situated below the WSD. The majority of the pumping energy will be recovered at the WTP.

During the Application/EIS (Rescan 2013) review process, additional mitigations were developed to minimize seepage from the WSD. These design enhancements are included in the 2016 PFS. Changes include six seepage interception tunnels that lower groundwater levels between the WSD and the WSF Seepage Dam to reduce the driving force on seepage. The seepage collection tunnels will also facilitate foundation grouting both during construction and for remedial grouting after completion of the WSD, if required.

An asphalt-core seepage collection dam will be located downstream of the WSF. The WSF Seepage Dam slopes were reviewed (KCB 2013a) and the revised dam section will incorporate 2.25H:1V upstream slope and 1.75H:1V downstream slope, with a low-permeability asphalt core and a grout curtain. Water collected in this dam will be sent to the WTP via an HDPE pipeline. During construction, this HDPE pipeline will be used to route runoff and sediment from the construction of the WSD and the WSD CDT to temporary water treatment facilities at the HDS WTP site.

Figure 18.7 Water Storage Dam Sections



The WSD CDT routes Mitchell Creek around the dam footprint during construction. In 2014, the diversion tunnel size was enlarged to 4.4 m by 5.0 m from the 4.3 m by 4.0 m size included in the 2012 PFS. As a result, the tunnel is now sized to pass flows from a 50-year storm event. In 2016, to reduce exposure to avalanche hazards, the downstream portal was moved upstream and the tunnel length was adjusted from 1,100 m to 900 m. Drawings of the shorter WSD CDT, and designs for an inlet cofferdam, tunnel closure gate, and tunnel plug are included in Appendix H. Upon completion of construction, the tunnel gate will be closed to allow construction of the permanent tunnel plug.

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 operating and will generate income and offset water treatment costs.

The later stages of the MTDTs include hydroelectric generation that comes into operation in Phase 2 (Year 10) with an installed capacity of 8.0 MW (Appendix I). In Phase 2 of the RSF layouts the tunnels are raised to have inlets above the expansion of the McTagg RSF; during Phase 3 (Year 15) the inlets are raised again once the RSF reaches its ultimate extent.

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.

18.2.8 Water Treatment

Temporary Mine Area Construction Period Water Treatment

For the 2016 PFS, design of the temporary water treatment facilities has been revised to enlarge the settling ponds to meet current guidelines for mine sediment control and settling ponds. The locations of ponds and tunnel muck pads have also been revised to meet adjustments in construction period tunnel portal activities as part of a tunneling schedule review.

During the construction period, six TWTPs for TSS and metal removal will be provided in the proposed mine area. Additional TWTPs will be located in the Saddle Area, and at the Treaty portal of the MTT. The TWTPs are intended to deal with drainage from existing mineralized zones, PAG cuts, tunnel portals, and runoff from PAG tunnel muck piles during the period before the permanent WTP is in operation.

For areas of the site where only TSS treatment is identified as required (such as soil borrow area and soil cuts), automated flocculent treatment systems and sediment ponds will operate to control TSS generated during the construction period. These treatment sites will be situated below earthworks and at sites identified as requiring only TSS control.

PAG tunnel muck will be stored on lined pads located at the temporary water treatment sites adjacent to the tunnel portals and the Mitchell OPC cuts. Where 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 HDS WTP are operational.

The WSF and WTP will be in operation during the six-month pre-production period to capture sediment and runoff from mine area stripping and from fill placement during Mitchell OPC and haul road construction.

A total of eight TWTPs 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 (Table 18.8).

Table 18.8 Temporary Water Treatment Plant Locations

TWTP	Location
TWTP #1	WSD/HDS WTP Area
TWTP #2	MTDT Outlet
TWTP #3	MDT Outlet
TWTP #4	Saddle MTT Portal Tunnel
TWTP #5	MDT Inlet
TWTP #6	MTT – Mitchell Portals
TWTP #7	WSD CDT Inlet
TWTP #8	Treaty MTT Portals

Note: TWTP #9 from the previous 2012 design was deleted as the MTT construction adit is not present in 2016 designs. TWTP #10 is not required for the 2016 designs as there are no TMF diversion tunnels required at start-up.

The TWTPs will include a grit pond, a reagent preparation system for lime and flocculent addition, settling pond, an air sparging pond when required, and a system for pH control. For additional details please see Appendix H8.

The purpose of the grit ponds is to remove particles larger than 0.1 mm from the water. It is expected that any particles larger than 0.1 mm in diameter will settle in the grit ponds. The grit ponds provide surge capacity with more than three hours of retention time during a 1-in-10-year, 24-hour rain event combined with an adit surge (both with and without sediment taken into consideration). The grit ponds will also have surge capacity for an additional inflow of 50 L/s for two hours to allow for flow increases upon crossing fractured zones.

Water decanted from the grit ponds will be pumped through the lime and flocculent preparation units. Hydrated lime will be added to increase the pH to between 8.5 and 9.5 to precipitate dissolved metals. The hydrated lime preparation system will feed through an inline mixer at a rate to be determined by the target pH requirement. The hydrated lime also functions as a coagulant altering the electrical charge on suspended particles to promote agglomeration and enhance flocculent performance. The hydrated lime will be stored in typical 20 kg bags on pallets. It will be continuously mixed with water to produce a 5% by weight solution of calcium hydroxide in the lime preparation system. The flow rate of the influent will be measured between the grit pond and the lime slurry injection point. The slurry will be prepared in a lime mixing holding tank. The lime slurry addition will be injected into the pipe in front of the inline mixer at a rate appropriate to meet set pH requirement.

A flocculent was chosen through bench scale testing of Mine Site sediment samples suspended in Mitchell Creek water. Based on these initial tests, the flocculent concentration required is approximately 0.1% by weight of BASF Magnafloc 351. The bulk flocculent will be added manually into a hopper where an automated dual tank flocculent preparation system will mix and dose the diluted solution into the water stream using calibrated metering pumps.

After lime and flocculent addition, water will flow into the settling pond to further reduce suspended solids. Particles will settle to the bottom as they move through the length of the pond. Most of the water treatment reaction related to the removal of metals will occur in the settling pond. This system will provide a hydraulic residence time of approximately 75 hours.

An air sparger system will be installed in a separate pond located below the settling pond to reduce ammonia concentrations in the water, when required, to around 10 mg/L. If required, the pH will be adjusted to between 10.5 and 11 in the air sparger pond with sodium hydroxide to volatilize the ammonia. A blower located adjacent to the container and close to the air sparger manifold system will be used to generate the air required.

Alkaline water is required for both dissolved metal precipitation and ammonia removal. However, the water must be neutralized before it can be discharged. After the air sparging pond, the water will be pumped through a neutralization treatment step located in a containerized unit. Sulphuric acid will be added through an inline mixer to adjust the pH of the water to between 6.5 and 8.5. The maximum required acid flow rate is less than 19 L/h (5.3 mL/s) at the nominal 50 L/s design flow rate.

High-density Sludge Water Treatment Plant

The HDS WTP is designed to treat water that comes in contact with areas of disturbance from mining operations and natural seeps in the area with a HDS lime. 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 HDS WTP will operate with a variable flow rate designed to match the natural hydrograph in Sulphurets Creek. Average annual variations in base case water treatment rates are shown in Figure 18.6; this includes additional treatment flows associated with treating natural contact water flows from upstream of the Mitchell pit.

Water will be collected in the WSF. Drainage from the Mitchell pit and Mitchell/McTagg RSFs will be directed by gravity to the WSF and contact water from the Sulphurets and Kerr pit areas will be routed to the WSF by gravity pipeline. The water from the WSF will be pumped over the WSD to the HDS WTP. The HDS WTP is designed with variable discharge rates in order to stage discharge to match the natural hydrograph, to ensure sufficient dilution capacity to minimize any effects on the receiving environment.

The HDS WTP maximum throughput capacity will be 7.5 m^3/s; however, the maximum rate is only anticipated to be required for a three-month period in the summer. Water pumped from the WSF will pass through hydroelectric generators installed at the Energy Recovery Power Plant, immediately upstream of the HDS WTP.

The HDS WTP installed generation capacity will be 9 MW and the two installed turbines will be capable of passing a flow of up to 7.5 m^3/s. To obtain this treatment capacity in a reliable and proven manner, initially five and ultimately seven circuits will be constructed and operated in parallel. The WTP design also considers very low treatment rates (0.10 to 0.25 m^3/s) in late fall, winter, and early spring.

The site selection for the HDS WTP is based on a +50-year mine life and post-closure treatment for 200 years. The HDS WTP will be located at an elevation of 520 m on a flat benched terrain above the flood plain near the confluence of Mitchell and Sulphurets creeks. An access road armored on the downstream side, located below the HDS WTP site, will perform as a levee for an additional level of flood protection from Sulphurets Creek. A complete HDS design and cost estimate was completed by SGS Canada Inc. in 2011 (Appendix H5). In 2013, additional design work was undertaken by SGS and Rescan to expand the components required to handle increased flow rates (Appendices H6 and H7).

The WTP conceptual design includes three large lime silos with slakers, 14 lime reactors for neutralization, three lime slurry stock tanks, three lime/sludge mix tanks, and ultimately seven conventional 60-m clarifiers sized for treatment rate of 0.8 to 1.1 m^3/s each. The winter sludge storage will be used during construction when sludge transport to the landfill or TMF is not available. The WTP discharge area will include a polishing pond for final pH adjustment and to provide additional solids settling capacity or retention, if required.

For discharge during low flows, one clarifier and a paired lime reactor will be adequate to achieve the discharge volume. The plant is designed so that under typical conditions, either individual paired lime reactors or a clarifier can be bypassed. The plant will be equipped with four, 100-plate press filters to produce sludge as a firm dry filter cake with a 50% moisture content with 25% solids clarifier under flow feed to the filter press. Approximately 360 t/d of sludge will be produced based on an average water treatment rate of 2 m^3/s. At a maximum throughput rate of 7.5 m^3/s, the total daily dry sludge load will be 1,360 t. During construction the sludge will be stored in the Sludge Storage Facility located near the WTP. In the winter the sludge will be stored in a shed that will be located immediately upslope of the WTP (Appendix H8).

During operation, the sludge will be transported year-round by truck to the Mitchell OPC and onto the ore trains within the MTT, where it will be transported with the ore to the stockpile located at the Treaty OPC, fed through the Treaty Process Plant, and deposited with the tailing in the TMF. At the maximum water treatment rate, the sludge will account for approximately 1% of the 130,000 t/d ore feed. On an annual average basis, the sludge will account for less than 0.3% of the ore feed.

The three principle reagents for the HDS WTP will be quick lime, dry flocculent, and sulphuric acid for pH adjustment to 7.5. Table 18.9 provides an estimate of annual reagent consumption based on an annual average of 71 Mm^3 of water treated. The predicted total annual volume of water will vary from 63 to 79 Mm^3. After closure, the predicted long-term volume of water for treatment will be 64 Mm^3/a.

Table 18.9 Annual Reagent Consumption for the HDS WTP

Reagent	Feed Rate	Average Annual Water Treated (Mm^3/a)	Total Annual Reagent Consumption
Quick lime to pH 10.5	0.83 kg/m^3	71	59,000 t
Magnafloc 10	3 g/m^3	71	215 t
H_2SO_4 to pH 7.5	11 mL/m^3 of 36.8N H_2SO_4	71	780,000 L

SELENIUM TREATMENT PLANT

BioteQ Environmental Technologies Inc. (BioteQ) demonstrated selenium removal of spiked Mitchell Creek feed water during a pilot-scale ion exchange water treatment study (Appendix H9; BioteQ 2015).

In previous studies, the Selen-IX™ Ion Exchange Circuit was optimized to select the best resin based on selenium selectivity, resin capacity, and regeneration characteristics (Appendix H10; BioteQ 2012). A preliminary design basis was developed for a 500 L/s Selenium WTP to be located adjacent to the WSF near the toe of the Mitchell/McTagg RSFs, to treat seepage from the Sulphurets Pit Backfill (Kerr waste rock), seepage from the RSFs, and water pumped from the WSF. The pilot study demonstrated reduction of selenium concentrations from 120 and 320 ppb feed water to less than 1 ppb (Appendix H11; BioteQ 2015).

Due to expected high iron and TSS concentrations in seepage water, a ferric circuit was designed as a standalone module serving as a pre-treatment step upstream of selenium removal. The primary goal of pre-treatment in the ferric circuit is to remove constituents that may interfere with ion exchange, included suspended solids, ferric iron, and selenite. Additionally, lime addition reduces sulphate concentrations.

The Selen-IX™ Ion Exchange Circuit is designed to selectively remove selenate from the feed water with a high efficiency in order to obtain the 1 ppb discharge limit, while concentrating the selenium into a small volume of brine solution that is directed to the eluate treatment circuit. Once the resin reaches a specified loading cycle duration, the selenate captured by the resin bed will be stripped from the resin using a sodium sulphate regenerant solution. The regenerant will be pumped from the recycled regenerant tank through the IX columns and into the spent regenerant tank for further downstream processing. The regenerated resin will then be available for further cycles of selenate loading after a brief wash cycle.

The eluate treatment circuit removes selenium from the spent regenerant (or eluate) solution produced by the ion exchange circuit with an electro-reduction process using iron and fixes the selenium into an iron-selenium solid that is easily separated from solution. The electro-reduction and precipitation of selenium out of solution takes places inside electrocells equipped with iron anodes. The solution discharged from the eluate treatment circuit is largely free of selenium and can be recycled back to the ion exchange circuit for re-use in resin regeneration.

As the Selenium WTP is only designed to remove selenium, effluent from the Selenium WTP will report to the WSF for further treatment at the HDS WTP, prior to discharge to the receiving environment.

18.2.9 Tailing Management Facility Design

The general layout of the TMF is shown in Figure 18.3. The TMF will be located in the same location as in 2012; within a cross-valley between Teigen and Treaty creeks. Three cells will be constructed: the North Cell and the South Cell will store flotation tailing, and the CIL Residue Storage Cell (fully lined with a geomembrane) will contain PAG CIL residue tailing. The layout, staging, and design of the main TMF dams has not changed for the 2016 PFS. The locations of the seepage collection dams have been moved downstream since the 2012 PFS (Tetra Tech) to facilitate seepage recovery. For the 2016 PFS, additional assessments were conducted on liner depressurization and underliner drainage systems. Designs have been advanced for extending the underdrains and associated dewatering pumping systems to mitigate liner uplift potential. These now extend to the ultimate CIL basin perimeter.

Tailing dam design has not changed for the 2016 PFS. The cyclone sand dams will be constructed on earthfill starter dams using the centerline construction method with compacted cyclone sand shells and low-permeability glacial till cores. The till in the cores of the North and Southeast dams will be amended with bentonite where necessary. The Saddle and Splitter dam cores incorporate geomembranes to limit seepage from the CIL residue tailing. The dams will be progressively raised over their operating life to an ultimate elevation of 1,068 m.

Process water in the North and South flotation tailing cells and CIL Residue Storage Cell will be reclaimed by floating pump barges and recycled to the plant. Non-contact runoff from surrounding valley slopes will be routed around the TMF. Diversion channels are sized to pass design flows and have large enough base widths for snow removal machinery. Buried pipe sections are used in active snow avalanche paths.

Figure 18.8 illustrates the staging of the TMF. The North Cell will be filled first; simultaneously, the CIL Residue Storage 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 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 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 water surplus 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. Additional site data collected since 2011 does not suggest that significant variations in these estimates would result from the additional data.

TAILING STAGING PLAN

Staging of the TMF has not changed in 2016. The layout of the TMF is shown on Figure 18.3; sequences of cell staging are shown in Figure 18.8. 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 Residue and South Cell when required during later stages of the Project.

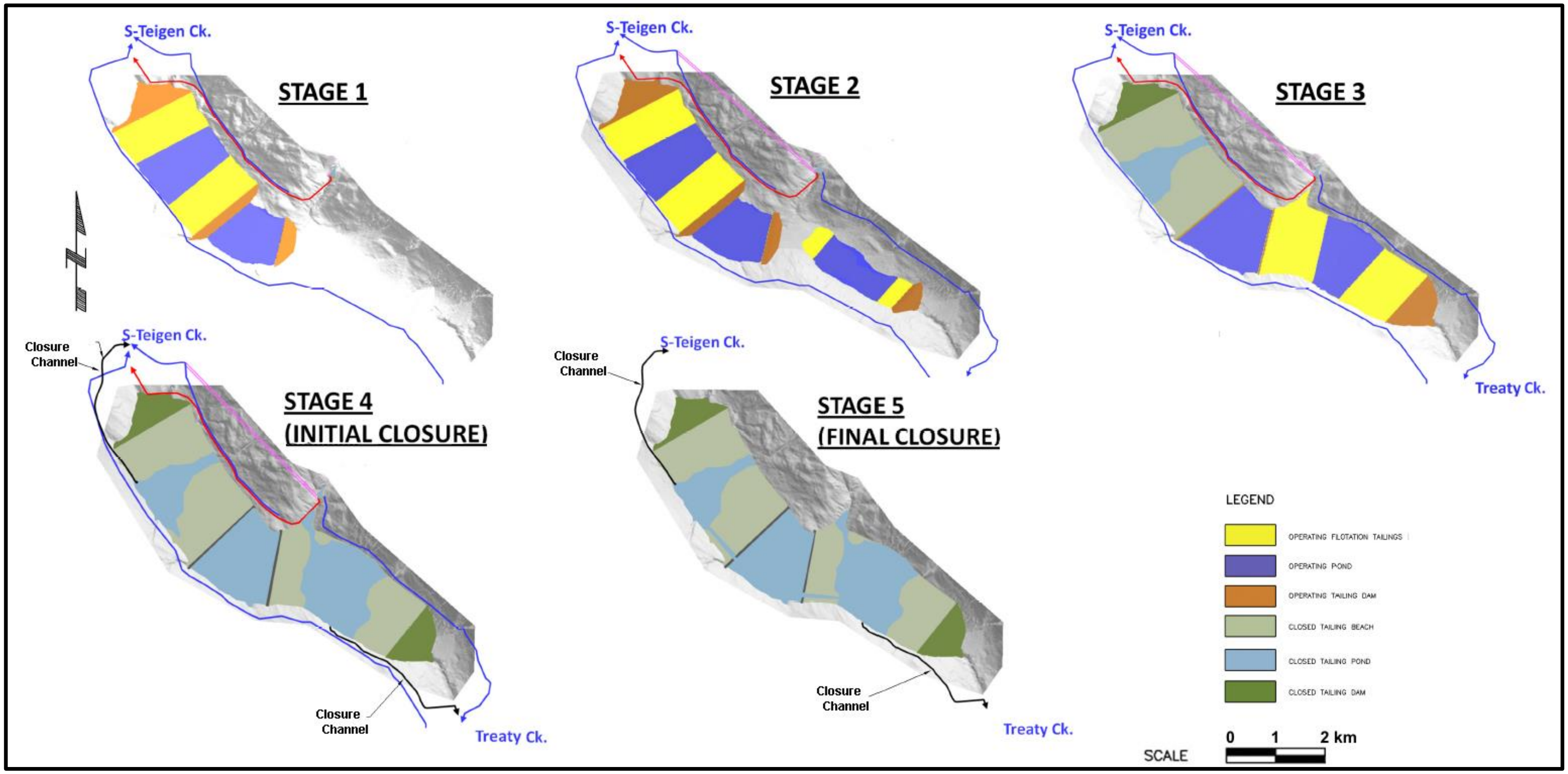
Tailing flows will be retained by four cyclone sand tailing dams: the North Dam, Splitter Dam, Saddle Dam, and Southeast Dam. During operation, elevations of annual dam crest raises will be set to provide 12 months of tailing storage and to store the PMF with 1 m of freeboard.

The North Cell will be constructed first and will store flotation tailing production for 25 years; at that point, this cell will 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. The South Cell and CIL Residue Cell will then be closed and reclaimed over a five-year period.

Based on the mill ramp-up schedule, and the assumed density ranges possible at start up, the starter dams can store between 18 and 24 months of tailing. The earth fill starter dams at the North, Splitter and Saddle Dam sites will be constructed to store a minimum of 8.4 Mm^3 of water for mill start-up. The design operating PMF ranges from 42 Mm^3 at start-up to 91 Mm^3 at the ultimate dam elevation. The dams will then be raised annually by cycloning tailing sand. Cyclone sand raises will continue to 1068 masl. The beach will be built up to separate the reclaim pond from the dams by at least 700 m, increasing to 1,200 m at the ultimate dam elevation. The separation between the tailing dam and pond created by the beach increases the margin of safety against overtopping of the tailing dam, and reduces seepage through the tailing dam and underlying foundation.

A starter dam, at elevation 930 m, will be completed by Year 25 to allow deposition to begin in the South Cell. Between Year 25 and closure, the Southeast Dam will be raised to its ultimate elevation. Final heights of starter dams will be re-evaluated during feasibility design based on initial tailing production ramp-up and seasonal timing of expected mill start-up.

Figure 18.8 TMF Staging Plan



Note: Raising of cyclone dams within each stage not shown on these diagrams.

TMF DAM STRUCTURES

Over an initial two-year construction period, three earth fill starter dams will be constructed at the North Cell and CIL Residue Cell (North, Splitter and Saddle) to provide start-up flotation and CIL residue tailing storage. These dams will be progressively raised over their operating life to an ultimate elevation of 1,068 masl, providing storage capacity of 2.3 Bt. A summary of the tailing dams is provided in Table 18.10.

TMF Starter Dams

Starter dams will be earthfill embankments, with shells of compacted random fill supporting the central glacial till cores. The glacial till cores will be keyed into the underlying foundations to cut off seepage through weathered near-surface soils and any pervious strata. A blanket drain is provided to lower the phreatic levels in the downstream shell. Riprap erosion protection will not be placed on the upstream slope due to the temporary exposure of the dam to the pond water. Figure 18.9, Figure 18.10 and Figure 18.11 show typical sections of the starter dams.

Main Tailing Dams

The North, Splitter, Saddle, and Southeast dams will be compacted cyclone sand dams with glacial till cores constructed by the centreline method. Dimensions of the dams are summarized in Table 18.10. Details of the North, Splitter, Saddle, and Southeast dam designs are shown in Figure 18.9, Figure 18.10, and Figure 18.11.

A system of finger drains will be installed at the base of the downstream shells of the North, Saddle and Southeast dams 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 will be placed on the downstream slopes for annual dam raises 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 coarse, cyclone underflow sand will be used for dam raise material.

In 2016, a review was conducted of cyclone sand supply and potential geotechnical enhancements to the tailing dams. It was concluded that sufficient cyclone sand was available for additional support zones and if required, toe berms to facilitate dam raising. The review concluded that requirements for additional cyclone sand placement be determined by stability assessments conducted within the next design phases.

An opportunity was also identified to lengthen the operating season for cyclone sand production by providing enclosed primary and secondary cyclones. This may significantly facilitate the provision of additional freeboard for the dams and would mitigate against potential sand shortfalls.

Table 18.10 Tailing Dam Summary

Dam	Starter Dam					Ultimate Dam				
	Crest Elevation (masl)	Maximum Height* (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	1,068	218	1,900	47.16	3.42
Splitter Dam	935	61	890	3.74	1.08	1,068	194	1,930	31.31	3.75
Saddle Dam	935	35	780	2.09	0.75	1,068	168	1,600	22.99	3.39
Southeast Dam	930	101	890	12.32	1.72	1,068	239	1,400	60.45	3.20
Totals	-	-	**3,240**	**21.73**	**4.50**	-	-	**6,830**	**161.91**	**13.77**

Note: *maximum height measured at dam crestline

Figure 18.9 North Tailing Dam

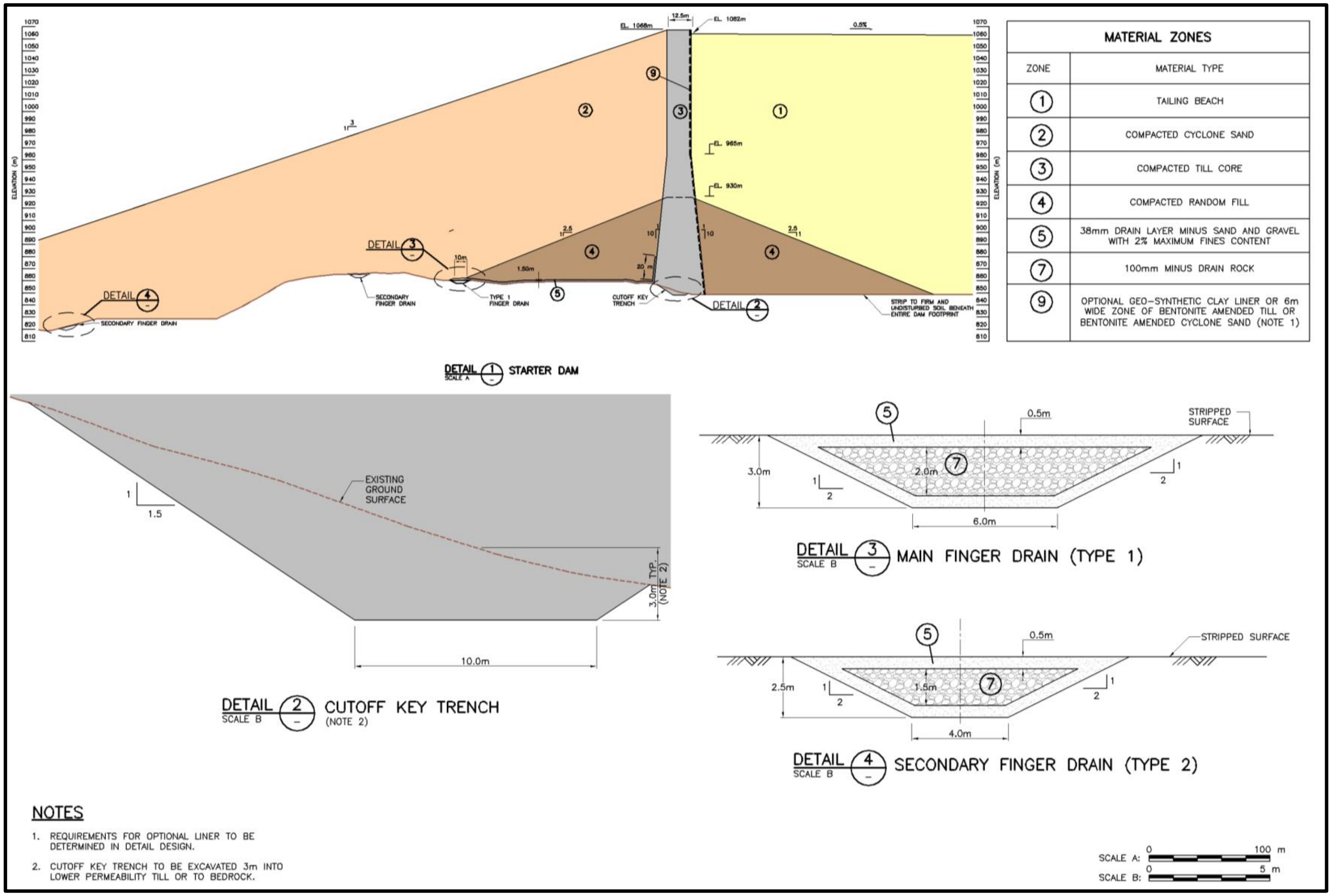


Figure 18.10 Saddle and Splitter Tailing Dams

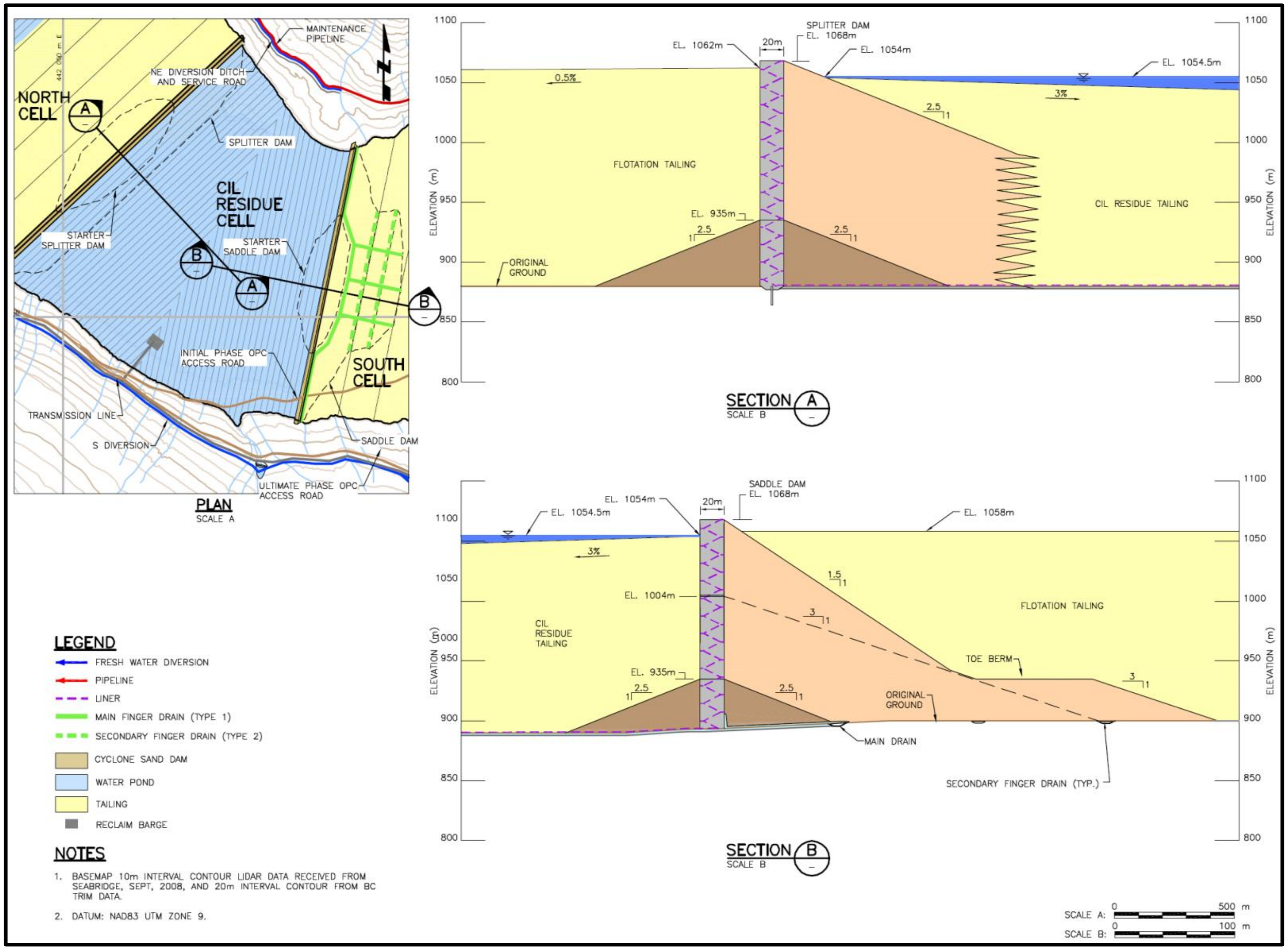
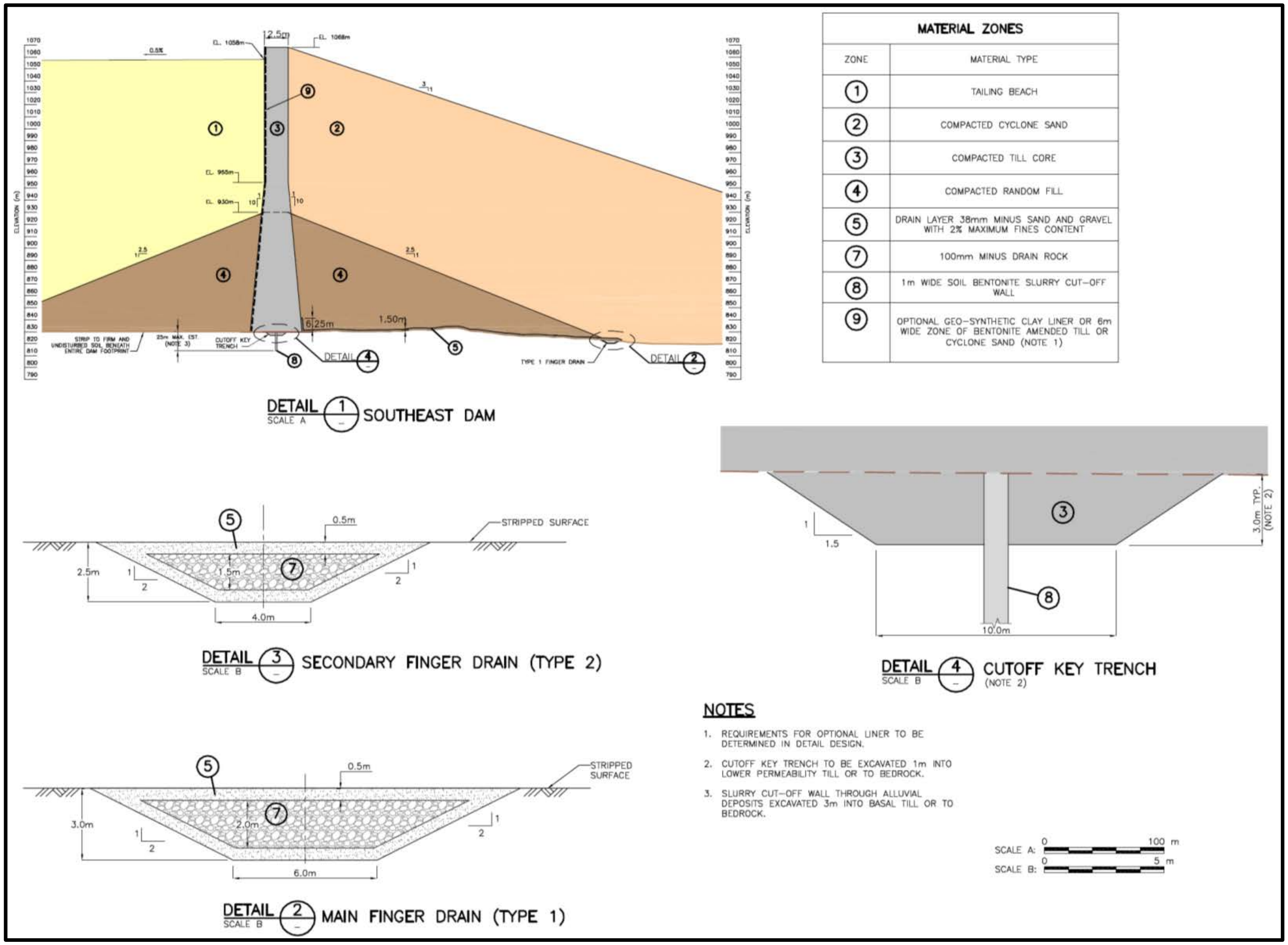


Figure 18.11 Southeast Tailing Dam



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 18.10 summarizes fill 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 (less than 2 km haul distance) that have been identified at each dam site. The cyclone sand TMF dams will be raised using a fleet of dedicated mine equipment.

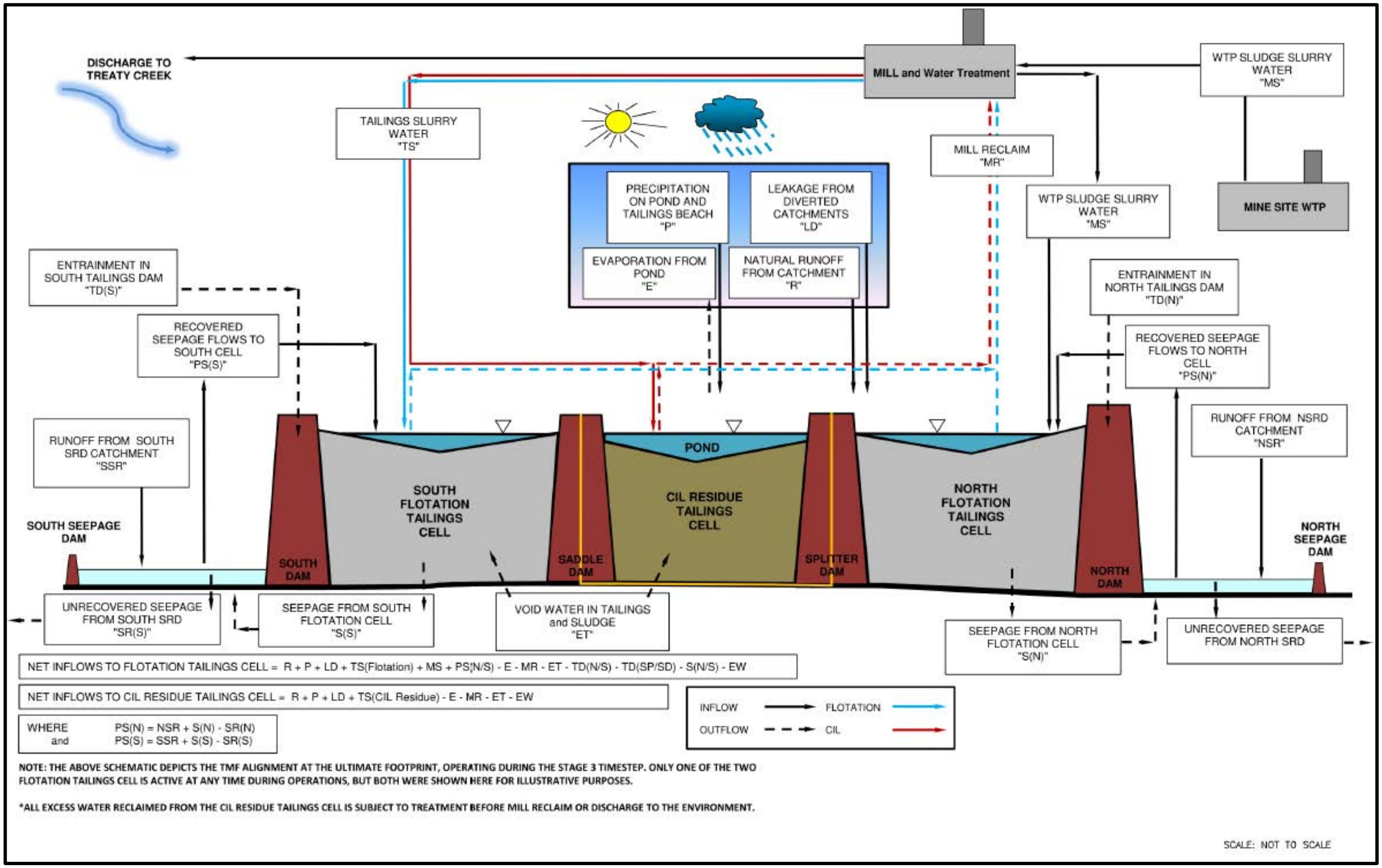
TMF Seepage Recovery Dams

The seepage recovery dams have been relocated approximately 1 km downstream from the 2012 locations. Seepage recovery dams will be constructed of compacted glacial 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; cut-off 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 18.12 shows the schematic water balance with water inputs and outputs from the impoundment.

Figure 18.12 Schematic TMF Water Cycle



TMF Diversion Channels

Design of the diversion channels has not changed for the 2016 PFS.

Two main diversion channels, the Northeast Diversion and the South Diversion, will be constructed around the TMF North Cell with additional diversions around the Treaty OPC 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 to the south end of the TMF valley to capture local flows.

Once in operation, the catchment area of the South Cell is diverted by the Southeast Diversion Channel, which routes non-contact flows to Treaty Creek around the east side of the South Cell. Diversion channels in the TMF area are designed to route 200-year peak flows. Diversion channels are shown on Figure 18.13.

To increase maintenance flows towards Teigen Creek, a diversion dam will be installed in the East Valley catchment. The East Catchment Diversion Dam diverts flows into a tunnel around a slide zone. The dam will initially 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. During the first stage of TMF operation, any higher flows from the East Valley will be passed over the East Diversion Dam spillway and into Treaty Creek tributary. As the South Cell is developed, flows from the East Catchment above 2 m^3/s will be routed north through the East Catchment Diversion Tunnel and into Teigen Creek.

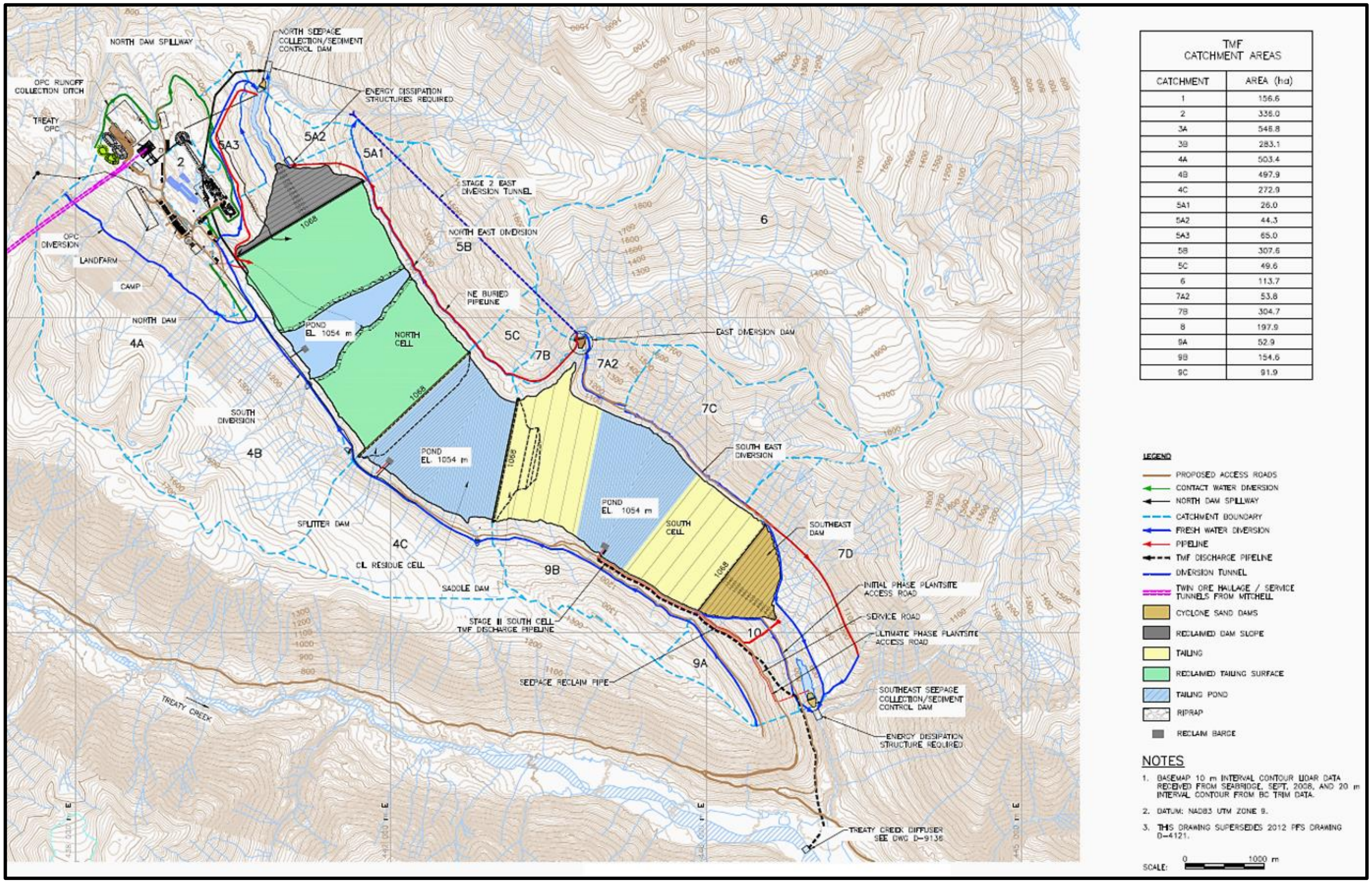
TMF AREA EXTREME FLOOD ROUTING AND STORAGE

The TMF cells are designed to be able to store extreme flood events without discharge. Specifically, the PMF can be stored, which is a flood resulting from a 30-day Probable Maximum Precipitation (PMP) storm, combined with a 100-year 30-day snow melt. The perimeter diversions are assumed to be inoperative during this extreme flood event. The PMF inflow volume is estimated to range from 42 Mm^3 at facility start-up to 91.0 Mm^3 at the ultimate stage.

For the 2016 PFS, staged TMF discharge pipelines were designed to route surplus water to diffusers buried in the channel of Treaty Creek. The capacity of the pipeline and diffuser system is designed to discharge the critical duration PMF.

During operations, water will be reclaimed from the ponds and routed back to the Treaty OPC, where it will be treated as part of the mineral separation process. Surplus water from the TMF will be discharged seasonally via the Treaty Creek Diffuser. Discharge will occur during an approximate period extending from May to mid-November, when the creek flows are highest.

Figure 18.13 Ultimate TMF with Catchments and Diversion Channels



TMF CATCHMENT AREAS

CATCHMENT	AREA (ha)
1	156.6
2	336.0
3A	546.8
3B	283.1
4A	503.4
4B	497.9
4C	272.9
5A1	26.0
5A2	44.3
5A3	65.0
5B	307.6
5C	49.6
6	113.7
7A2	53.8
7B	304.7
8	197.9
9A	52.9
9B	154.6
9C	91.9

18.3 TUNNELS

A number of tunnels will be excavated during both the pre-production period and operating period. These tunnels will be classified as either infrastructure tunnels or water tunnels as shown in Table 18.11 for tunnels constructed in pre-production and Table 18.12 for those constructed in the operational phase. Other tunnels associated specifically with block cave mining, are presented in Section 16.3.

The infrastructure tunnels provide for the transportation of ore, personnel, and supplies between the Mitchell, Sulphurets, and Kerr mining areas and the Treaty OPC. The principal infrastructure tunnel is the MTT, which transports all mined ore from the Mitchell OPC to the Treaty OPC, and personnel and freight between the PTMA and the Mine Site, via the train haulage system. Other infrastructure tunnels include load out, unloading, and freight sidings in the MTT, a spur off the MTT to Iron Cap, and an ore conveyor tunnel from the Sulphurets pit to the Mitchell OPC.

The water tunnels include the diversion tunnels as described in the SWM Plan and the slope drainage tunnels for the Mitchell high wall and the Snowfields landslide.

This section includes a description of the construction method, sequencing, and cost basis for the tunnels as applied to the Project schedule and capital estimate. The MTT design cross section has been modified to accommodate the change to train haulage from the previous ore conveyor system; however the alignment remains the same. The flow requirements for MDT, NPWDA, and MVDT have been modified as described in the Mitchell Glacier Diversion Optimization section of Appendix H1. Revisions to diversion tunnel designs are limited to cross sectional area, staging of tunnel twinned phases, tunnel slope and inlet configuration. Overall alignments as designed in the 2012 PFS (Tetra Tech 2012) remain.

Table 18.11 KSM Pre-Production Tunnels Summary

Tunnel	Description	Total Length (m)	Excavation Volume (m^3)
Infrastructure Tunnels			
MTT	Principal Alignment	49,406	1,266,770
	Treaty Ore Handling	514	27,174
	Mitchell Ore Handling	943	43,784
	Mitchell Freight and Fuel	576	28,898
MTT Subtotal		51,439	1,366,626
Water Tunnels			
MDT – Open Pit Phase	Diversion Tunnel Inlets	1,241	32,899
	Access Tunnels	870	20,402
	Main Diversion Tunnel	7,000	185,570
MDT Subtotal		9,111	238,871
MTDT	Stage I	8,020	253,038

table continues…

Tunnel	Description	Total Length (m)	Excavation Volume (m^3)
MVDT and Mitchell OPC Decline	-	5,424	182,787
CDT at WSD	Construction Diversion Tunnel	900	18,360
SCT at WSD	Seepage Collection Tunnels	1,780	19,402
Pre-production Tunnels Total	**-**	**76,674**	**2,079,084**

Note: Tunnel volumes are based on construction volumes and account for drill hole “look out” contractor tolerance and up to 200 mm of shotcrete as required.

Table 18.12 KSM Operational Phase Tunnels Summary

Tunnel	Description	Total Length (m)	Excavation Volume (m^3)
Infrastructure Tunnels			
Iron Cap Connection	-	972	35,983
SMCT	-	3,025	108,419
Water Tunnels			
MDT			
MDT – Underground Phase	Diversion Tunnel Inlets	1,095	29,028
	Access Tunnels	812	29,516
	Main Diversion Tunnel	7,000	185,570
MDT Subtotal		8,907	244,114
MTDT	Stage II	6,742	136,469
	Stage III	7,940	125,135
MTDT Subtotal		14,682	261,604
NPWDA and SSDA	Main NPWDA	3,000	100,260
	Access Tunnels	600	12,834
	Inlet Tunnels	589	26,855
	SSDA	1,104	22,345
NPWDA and SSDA Subtotal		5,293	162,294
East Catchment Tunnel	at TMF	4,000	67,733
Mitchell Block Cave Dewatering Tunnels	Underground Phase	12,000	675,000
Operation Phase Tunnels Total	**-**	**48,879**	**1,555,147**

Note: Tunnel volumes are based on construction volumes and account for drill hole “look out” contractor tolerance and up to 200 mm of shotcrete as required.

18.3.1 MITCHELL-TREATY TUNNELS

The MTT follows the same alignment as in the 2012 PFS design, but has been revised to accommodate the change from an ore conveyor and truck based transport system to a train based system. The tunnel cross-sections have been changed to match the selected train configurations and underground sidings and loading/unloading pockets have been added to the excavation designs. Excavations for the freight, personnel transport, and tunnel infrastructure has been substantially changed as well. Of particular note is the fuel pipeline between the PTMA and the Mine Site has been replaced with a rail based tanker system.

MTT DESIGN

Primary crushing of ore from the Kerr, Sulphurets, and Mitchell open pits will be done at the Mitchell OPC located at the Mine Site. The crushed ore will be transported through the MTT to the crushed ore stockpile located at the Treaty OPC, approximately 23 km to the east. Future underground ore from the Mitchell and Iron Cap block cave mines will connect with the Mitchell-Treaty ore transport system. The tunnels have been revised from the 2012 PFS to accommodate the change from the previous concept of an ore conveyor and truck delivery personnel and freight system, to a train-based transport system for ore, personnel and freight; however, the tunnel location and alignment have not changed. Under normal operations, the North Tunnel will be designated for westbound travel and the South Tunnel will be designated for eastbound travel (see Figure 18.16 in Section 18.4).

The MTT will comprise the following excavations:

- two, 22,715 m long tunnels
- 75, 30 m long cross-cuts, linking the two tunnels
  - 36 of these will be equipped with refuge stations
- nine track cross-overs to allow for flexibility and tunnel maintenance during operations
- Saddle Adit – a 265 m long decline at 15% complete with waste transfer chamber (for use during construction)
- 12 rectifier cut-outs, 6 in each tunnel
- train unloading station at the Treaty portal (see Figure 18.19 in Section 18.4)
- Treaty lower portal:
  - ore storage bin
  - conveyor tunnel to the COS
  - escape/ventilation raise to the MTT South Tunnel
- Mitchell portal:
  - two train loading sidings and two coarse ore bin excavations
  - conveyor tunnel from the primary crushers
  - shuttle conveyor tunnel between the coarse ore bins
  - escape/ventilation raise from the MTT South Tunnel up to the shuttle conveyor tunnel and an escape/ventilation tunnel from the MTT South Tunnel up to the conveyor tunnel; (the escape tunnel is required during excavation of the MTT in case of avalanches)
- Mitchell Freight Area

- fuel and freight handling areas located underground with accesses for mobile equipment from surface through the South Tunnel (see Figure 18.23 in Section 18.4)
- two cross-cuts from the fuel and freight handling areas to the Lower Mitchell Access Ramp, which will allow supplies to be transported directly to the future Mitchell block cave underground warehouses without having to go to surface.

MTT Tunnel Support and Advance Rates

From a regional review of the rock types that will be encountered on the MTT alignment it is anticipated that four general types of TSC will be required along the length of the MTT as described in Table 18.13. In the areas of the MTT that are identified as having PAG rock, corrosion-resistant rock bolts, and corrosion resistant mesh would be installed.

Table 18.13 Recommended Tunnel Support

Percentage of Tunnel (%)	Recommended Tunnel Support – MTT
29	2.4 m long rock bolts on 1.5 m spacing in roof, spot bolting on walls; welded wire mesh roof; may require shotcrete in localized areas of poor rock mass and across small shears
21	2.4 m long rock bolts on 1.2 m spacing in roof, 2 m spacing on walls; welded wire mesh roof; may require shotcrete in localized areas of poor rock mass and across small shears
42	50 mm fibre reinforced shotcrete; 2.4 m long rock bolts on 1.5 m spacing in roof, 2 m spacing on walls
8	100 mm to 200 mm fibre reinforced shotcrete, 3 m long bolts on 1 m spacing in roof and walls; may require reduced round length; may require pilling; may require grout cover; may require lattice girders

The activities required to install the four TSCs adds to the time required to take the round and therefore affects the daily advance rate of the tunnel. Table 18.14 uses the distribution of TSC from Table 18.13 along with associated advance rates as determined by the contractors. From the range in advance rates for each TSC, MMTS has chosen an appropriate rate to use in the scheduling for the MTT, as well as the other tunnels.

Table 18.14 Ground Support and Advance Rates

Ground Support (TSC)	Percentage of Tunnel (%)	Advance (m/d)
Class I	29	7.08
Class II	21	6.80
Class III	42	6.58
Class IV	8	5.08
Total/Average	100.0	6.65

MTT MINING METHOD

The tunnels will be constructed in accordance with BC *Mines Act* and Health, Safety and Reclamation Code for Mines in BC using conventional drill and blast techniques, and will follow the conditions contained within the License of Occupation for the MTT issued in September 2013 by the Government of BC.

WATER MANAGEMENT

Both MTT tunnels will be driven with a ditch in the floor and excavated in the corner that will contain a perforated pipe for the collection of tunnel water during the operations period; the pipe will be buried under the track ballast. The overall grade of the MTT is 1.2% with the Treaty portals being higher in elevation than the Mitchell portals, therefore in operation the water in the tunnels will flow by gravity back to the Mitchell portal where it will be routed to the water storage facility

During construction, tunnel water will be collected and treated as follows with each flow of water directed to temporary retention ponds associated with the temporary water treatment plant located near the portal:

- Mitchell (headings driven east): water will be collected in the perforated pipe and will flow by gravity back to the temporary treatment ponds at the Mitchell portals.
- Saddle (headings driven west): water will be collected at the face and pumped through a construction discharge line back to a sump located at the bottom of the Saddle Adit. From the sump, it will be pumped to temporary treatment ponds at the Saddle portal.
- Treaty (headings driven west): water will be collected at the face and pumped through a construction discharge line back to the temporary treatment ponds at the Treaty portals.

TUNNELING EXCAVATION CYCLE AND PRINCIPAL EQUIPMENT

The tunneling excavation cycle comprises the following unit operations for each round of advance. Each of the portal areas will have their own dedicated fleet of equipment as determined by the selected contractor.

Face Drilling

The faces will be drilled using three-boom electric-hydraulic jumbos drilling 5.8 m long blast holes and one 10 m long probe hole. The probe hole is to determine potential water inflow as the heading advances.

Loading and Blasting

A two-person crew will load explosives using a scissor truck. Explosives will comprise emulsion or ANFO depending upon water conditions of drill holes. An advance of 5.30 m is expected on average from the 5.80 m long holes drilled. It is anticipated that blasting will be carried out on both day and night shifts, at any time, under proper safety protocol so that crews don't have to wait until shift change to blast.

Face Mucking

Face mucking will use LHDs to the closest remuck bay to get the face cleared as quickly as possible.

Installing Tunnel Support

Once the face has been mucked, the tunnel support cycle will commence using electric-hydraulic bolting jumbos to install rock bolts, and welded wire mesh. Shotcrete will be placed using a shotcrete/fibercrete sprayer fed by a shotcrete transmixer. This activity will happen behind the jumbo while face drilling the next round. During the tunnel support cycle, a hole in the tunnel back will be drilled for future installation of hanging cable trays and catenaries for the permanent services in the tunnels.

Installing Services

Installation of services is not part of the regular tunneling cycle but takes place when the faces are suitably advanced.

Installation of services includes the following:

- advancing auxiliary ventilation fans
- hanging cables and ventilation ducting
- hanging air, water, and dewatering lines
- installing the construction and permanent dewatering lines.

Installing Track and Ballast

The track and ballast will be installed in both the North and South tunnels during the construction period.

Construction muck will be hauled out using the North Tunnel and an allowance has been included to install and later remove, a third rail to accommodate a different track gauge than the permanent gauge. Installing track at final gauge during construction will allow for early commissioning of the train system.

Muck Haulage to Surface

Muck from both the North and South tunnels, and the cross-cuts and track cross-overs will be placed in the remuck bays. Independent of the face drilling and advance, muck from the re-muck bay will be loaded into train cars and hauled to surface via the North Tunnel. After exiting the portal it will be placed into the rock dumps at the portals. The exception is the Saddle Adit where rail muck will be dumped into an underground chamber adjacent to the tracks and then loaded into low profile trucks and trimmed to the rock dump at the Saddle portal site

Temporary waste management facilities will be in place for all the tunnel muck exiting to surface from the multiple headings. The muck will be sampled for acid rock drainage (ARD) potential and stockpile as required. Muck will later be placed into a permanently designated storage location either at Mitchell (RSF) or Treaty (TMF). This includes remanding the construction muck at the portal through the completed Saddle to Treaty portion of the MTT.

As each heading of the MTT faces advances, the remuck bays and rails will be advanced. The re-muck bays are included in the excavation required for the cross-cuts.

Shift Schedule and Rotation

Tunneling personnel are scheduled to work two, 11-hour shifts, and two shifts per day with equal onsite/offsite crew rotation. Other tunnels projects in BC have been granted permits for between 10 and 11.5 hours per shift depending on local conditions and discussions with the regulators this Project will be considered on a similar basis. The 11 hour shifts used in this study are conservative.

MTT Mining Sequence

The MTT is on the critical path of the construction schedule and has therefore been broken into two segments to allow for concurrent development workplaces resulting in a shorter total tunnel construction period. This preliminary sequence will be accomplished using the Saddle, which is a transverse valley along the tunnel alignment, located approximately 6.1 km from the Treaty Portal of the MTT and 16.6 km from the Mitchell portal. An adit will be driven at a negative grade from the Saddle and access both the north and south MTT tunnels. It will allow the Mitchell-Saddle segment of the MTT to be driven from two headings from either end of the segment, with four independent crews at one time. Additionally, two crews will be advancing the Treaty-Saddle segments from the Treaty portals, one in the North Tunnel and one in the South Tunnel.

Crews from the shorter Saddle to Treaty segment of the MTT will be finished early and will be will then be used to develop the other excavations (e.g. ore bins, rectifier chambers, etc.) required for the complete MTT system design.

The North and South tunnels will advance together both from the Mitchell and Saddle headings. As the twin headings advance, cross-cuts will be developed every 300 m joining the two tunnels. The cross-cut closest to the face will be used for remucking, the next closest cross-cut will be used for the ventilation cross-over as discussed below and the cross-cut before that will be sealed and equipped with a refuge station.

Cross-Cuts and Track Cross-Overs

The track cross overs are included in the revised twin tunnel design to divide the MTT into three haulage sections to facilitate maintenance on the track, tunnels, and infrastructure while the ore transport is in continuous operation and to route trains into the loading and unloading sidings at each end. Air doors and fans are also provided to isolate sections while maintenance is in progress.

Cross-cuts are also designed between the tunnels, and with the cross-overs, provided access between the tunnels at 300m spacing along its length. This spacing will provide efficient work cycles for the dual heading construction method and facilitate a common muck haul and ventilation system during construction. During operations, the cross-cuts will be sealed with an air door to provide for independent airway and fitted with mandoors to meet the requirements for independent access for personnel.

Each of these excavations will have the same cross-section as the North and South haulage tunnels and will be excavated from the North and South tunnels. Some of the crossovers and crosscut excavation will be scheduled off the critical path for the tunnel crews, to compress the overall tunnel schedule.

MTT Infrastructure

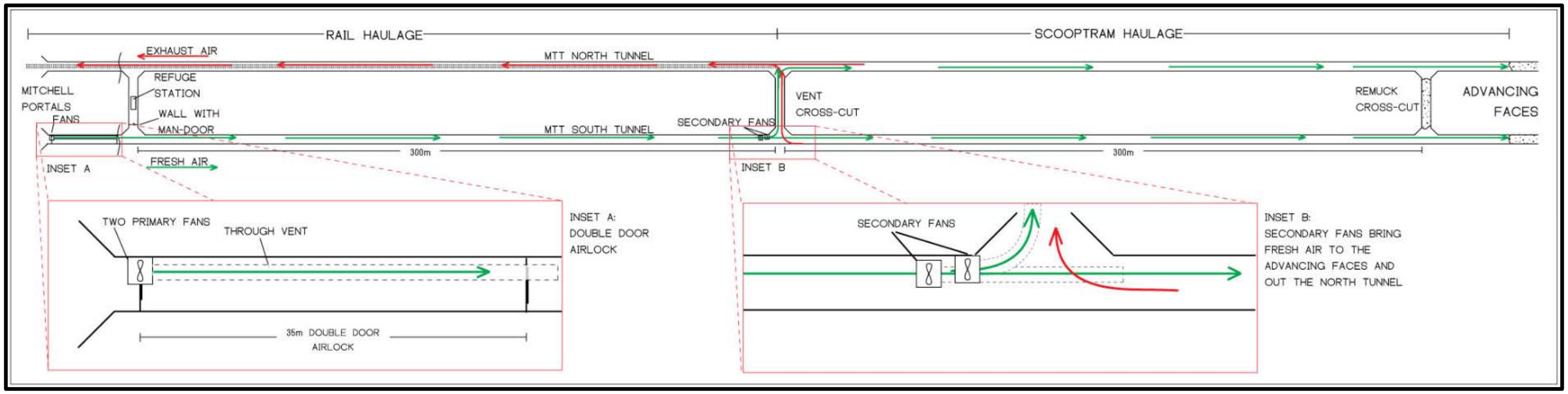
During tunnel construction, installation of the infrastructure required for the operation of the train haulage system will be installed where it doesn't disrupt the tunnel advance rate. Upon completion of tunneling, time is allocated to complete the fitting of the MTT for the operating systems and to commission the first trains. This will include installing parts of the electrical system required for the trains, the loading and unloading infrastructure, and the ventilation system required for the MTT suitable for the operating phase.

Ventilation During Construction

Since there are three sets of two headings advancing together, there will be three primary ventilation circuits. The primary circuit will be established by installing two fans in a bulkhead just inside the portal in the South Tunnel (and near the entrance to the Saddle adit), to provide fresh air under positive pressure through the South Tunnel, through the ventilation cross-cut with exhaust out the North Tunnel as shown in Figure 18.14.

The secondary circuits will be established to intercept fresh air from the primary circuits in order to ventilate the advancing faces. This will be done by two auxiliary fans with flexible vent ducting installed in the South Tunnel on the fresh air side of the active ventilation cross-cut and blowing air to the advancing headings in each of the South and North tunnels. The air from the South Tunnel will exhaust via the ventilation cross-cut where it will meet with the exhaust air from the North Tunnel. This exhaust air stream will then flow out the portal. As the tunnel faces advance, a new remuck cross-cut will be established and the previous remuck cross-cut will now act as the new ventilation cross-cut. The previous ventilation cross-cut will be sealed and equipped with the advancing refuge station. Figure 18.14 shows the primary and secondary ventilation system for the Saddle to Mitchell segment of the MTT.

Figure 18.14 MTT Ventilation Circuit



VENTILATION DURING OPERATIONS

During normal operations air will be moved through the tunnel by the piston effect of the trains. Thus ventilation in the North Tunnel will generally be from east to west and ventilation in the South Tunnel will generally be from west to east.

To allow for segments of the MTT to be isolated for maintenance, sets of ventilation doors with axial vane fans will be installed at the portals and at the track cross-overs. In this way fresh air will be supplied to the isolated sections of track where the crews will be working. Energizing or de-energizing of fans will be coordinated with train traffic so that they aren't working against each other or against closed vent doors.

Additionally, at the bottom of the escape ramp from the South Tunnel that leads to the conveyor tunnel at Mitchell, a fan will be placed in the vent doors to provide ventilating air in the ramp. This air will exhaust out the conveyor auxiliary ventilation systems are also designed for the lower Treaty portal at the unloading station and the Mitchell loading stations.

Both the freight and the fuel areas will be equipped with vent doors to control air flow for the entrance and exit for the freight trains and for mobile equipment from the underground freight sidings to surface.

An allowance for two future access tunnels from the Mitchell block cave access tunnel will be provided, these will be equipped with vent fans located above the doors, which will draw air out of the freight and fuel areas under a negative pressure, exhausting it into the Mitchell block cave access tunnel. This will need to be integrated with the ventilation design for the block caves.

A more detailed ventilation study is recommended at future levels of study.

TRANSPORT OF ORE, PERSONNEL AND FREIGHT

Train operations are discussed in Section 18.4.

Tunnel Maintenance

Tunnel maintenance will comprise the following:

- track maintenance – repair of track, ballast, and switches, including clearing of drainage pipes
- maintenance of ventilation fans and ventilation doors
- maintenance of catenary and communications systems
- tunnel support rehabilitation – planned and unplanned repair of tunnel walls and back.

The MTT has been designed with track cross-overs linking the North Tunnel to the South Tunnel over the 22,715 m tunnel length. The cross-overs will allow the tunnels to be separated into three approximately equal length segments such that trains can diverted from the isolated segment onto the corresponding piece of track in the opposite tunnel and maintenance can be carried out. The ore traffic control system will control the train traffic when a segment has been isolated and the other tunnel is handling train traffic in both directions. The ventilation for the isolated section is described on page 18-56.

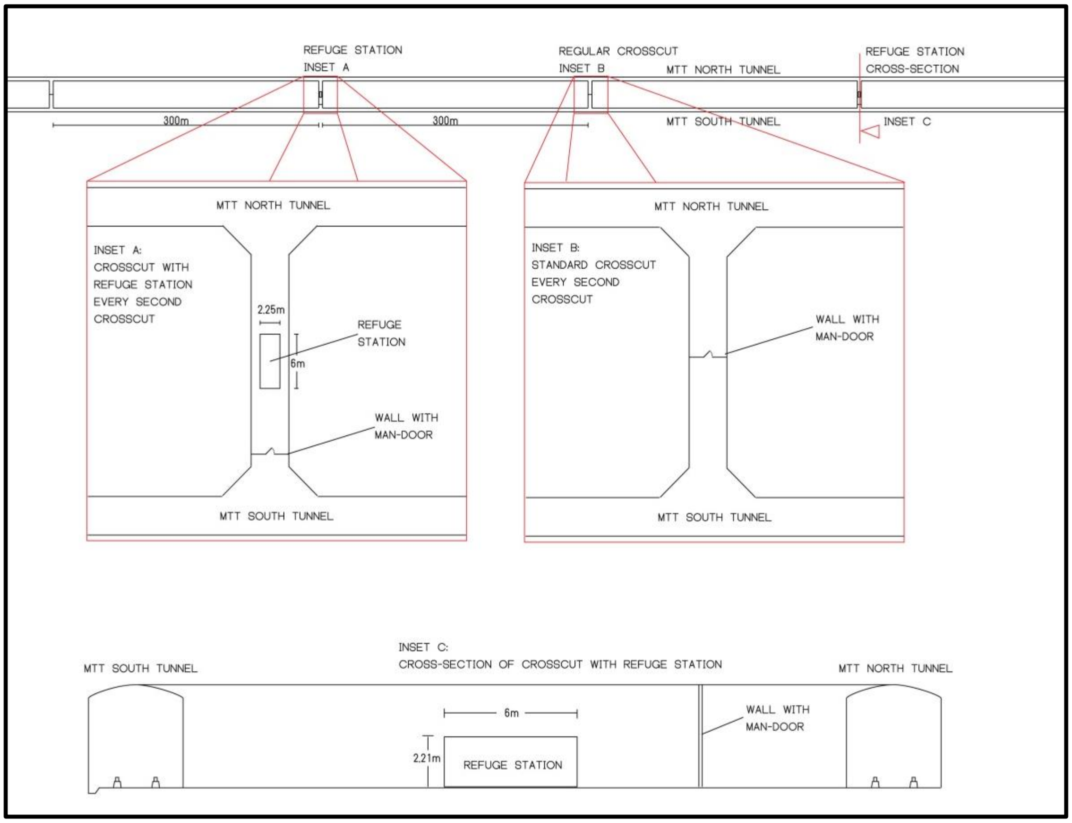
Tunnel Maintenance Personnel

Tunnel maintenance personnel will include electricians, instrumentation technicians, track maintenance and tunnel support personnel. With the amount of track and cable installed, and regular tunnel support maintenance, these maintenance crews will have to be dedicated solely to the MTT.

REFUGE STATIONS

At the end of the tunnel construction period, thirty-six, 12-person refuge stations will be set up in approximately half of the crosscuts connecting the North and South tunnels. These refuge stations have been designed primarily for the workers carrying out the various maintenance activities and will be located 600 m apart thus the furthest any personnel would be from the nearest refuge point would be 300 m. These refuge stations will be equipped with a self-contained supply of air and water and will be connected to the tunnel communication system. The final specification will be developed during further studies, to meet the requirements of the operation's Emergency Response Plan required by the BC Mines Regulations. Because there are twin tunnels with separate airways, crosscuts every 300 m, and vent doors that can control airflow in the event of a fire, the final placement of 36 refuge stations may be conservative. The general layout of the refuge stations is shown in Figure 18.15.

Figure 18.15 Plan and Section Showing Refuge Stations Locations



Transport of Personnel (Crew Change and regular operations)

The train control system will ensure there is no haulage of fuel or explosives when personnel are being transported in the tunnels. In the event of an emergency, the personnel cars will be equipped with personal protective equipment (PPE) kits including self-rescuers that will be located under the seats. This will be suitable for personnel to use to get to the closest rescue station or to exit into the other tunnel via the closest crosscut, through the man door in the brattice. In the event of a fire, the ventilation doors on each side of the segment containing the fire, will be closed by remote control and personnel will be required to leave the train and cross over into the opposite tunnel through one of the cross-cuts equipped with the man-doors. The ventilating fans in the MTT are not normally in operation but could be turned on for the tunnel in which personnel has crossed over to pending proper emergency protocol.

Fire Isolation during Operations

In the event of a fire in the fuel or freight areas during operations at the Mitchell marshalling area, these areas will be closed by remote operation of the four vent doors. For a fire in any other part of the MTT, the vent doors of the tunnel segment containing the fire, can also be closed remotely.

18.4 MINE TO MILL ORE TRANSPORT SYSTEM

At the Mitchell OPC, ore will be crushed and conveyed through a tunnel to two live underground ore bins within the MTT. Loading chutes under the ore bins will feed ore into awaiting trains that will transport the ore to an unloading station at the Treaty end of the MTT. The train cars will dump ore into a live underground unloading bin. Apron feeders will unload the bin onto a conveyor to transport the ore to the top of the Treaty COS.

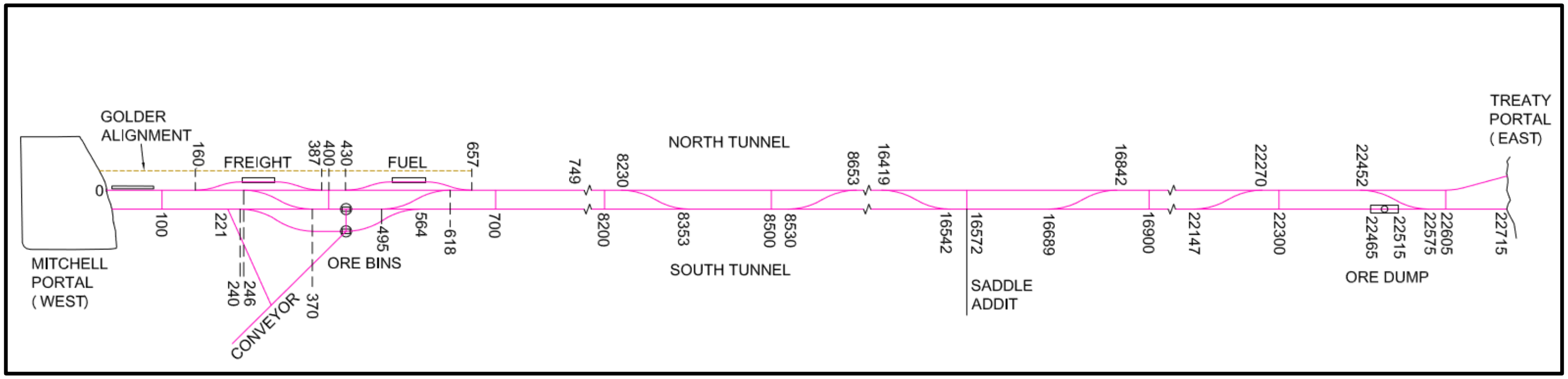
Each train will consist of one, 140 t electric locomotive and 16, 42 m^3 belly dump ore cars that have the capacity to deliver 800 t/h from Mitchell to Treaty based on 90-minute cycle times. On average, eight trains will deliver approximately 130,000 t/d of ore to meet the process plant requirements. An additional four trains will be on standby to provide for mechanical availability or to handle an increase in plant feed of up to 10,000 t/h, when required to meet the total mine to mill ore transport system requirements of approximately 130,000 t/d. The transport system capacity has been confirmed at a PFS level by using a high-level train routing simulation that incorporates static modelling of the rail system and average cycle times between Mitchell and Treaty for delivery of ore, materials, and personnel.

The trains will travel on a conventional ballasted track structure with timber ties and operate via an electrical overhead catenary system. Trains will be controlled by an automated train control system managed from a remote control room. Loading chutes will also be controlled remotely, and unloading chutes will operate autonomously. No onboard operators will be required within the tunnels during train system operation.

The train transport system is described in detail in the Nordic Minesteel Technologies Inc. (NMT) report (NMT 2016), which details specifications and dimensions for each component of the train transport system.

Figure 18.16 shows a plan view of the approximately 23 km long MTT dual track transport system. The south tunnel will be primarily utilized for loaded trains travelling from Mitchell to Treaty, and the north tunnel will be utilized for empty trains travelling from Treaty to Mitchell. The tunnels will run uphill from Mitchell to Treaty at a 1.2% incline, such that tunnel drainage will flow to the property's mine side. Cross-overs are planned at both end points of the tunnel, as well as in two intermediate points to split the route into three sections. One section of the tunnel can be isolated when maintenance is required, and one-way traffic can be implemented and safely controlled by the train automation system. A simulation has verified that these traffic flow restrictions will not compromise average daily train production requirements.

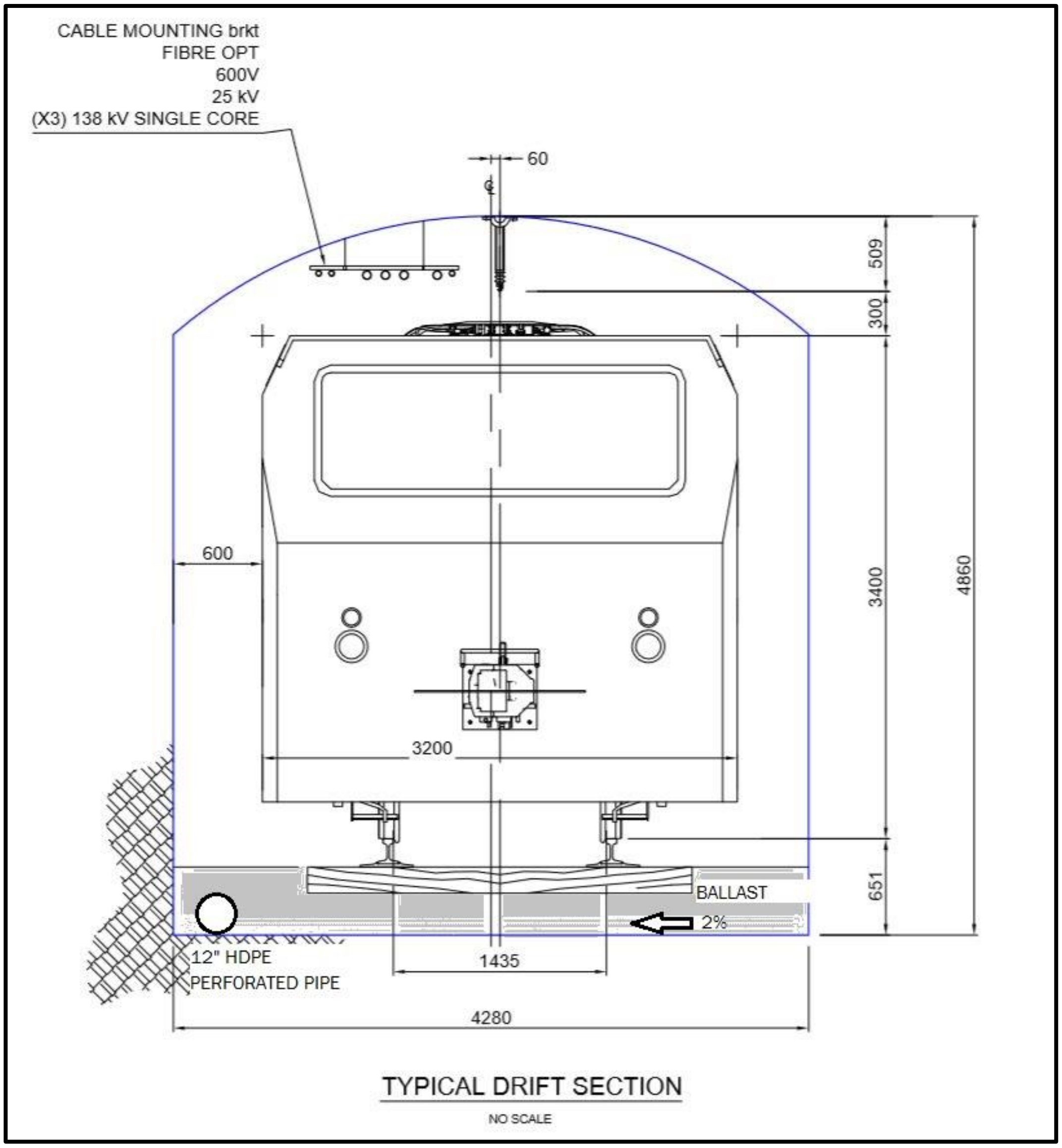
Figure 18.16 MTT Dual Track Plan View (Distances in Metres)



The numerical demarcations in Figure 18.16 show the distance, in metres, along the MTT tunnel of noted features from Mitchell to Treaty. The Golder Alignment refers to the planned access from a location near the Mitchell Portal to the underground block cave operations.

Figure 18.17 shows a typical cross-section through both of the MTT train tunnels. Standard gauge ballasted track, 1.435 m between inside edges, with base-plated timber sleepers, and fish-plated 56.5-kg/m rail will be utilized for the train running surface. The crushed stone ballast will be 250 mm thick, after tamping flat, with in-laid sleepers. The tunnel floor will be graded to one side with perforated HDPE piping installed in the ballast to carry water drainage.

Figure 18.17 MTT Train Transport Drift Section (Dimensions in Millimetres)



The 140 t locomotives are powered by an electrical overhead 25 kV alternating current (AC) catenary system that includes:

- two, 25 kV medium voltage substations
- three low-voltage substations equipped with a transformer 25 kV/0, 6 kV/208 V, and 600 V/208 V control panel
- 12 rectifiers, 6 in the north tunnel and 6 in the south tunnel
- a Siemens SCAT SR aluminum overhead conductor rail system.

Loaded trains will travel on an uphill slope, requiring a tractive force of approximately 275 kN. At a speed of 30 km/h, power consumption of 2,400 kW per train is expected and will require a current draw of 1,600 A from the catenary at an operational voltage of 1,500 V. To reduce the effects of voltage drop in feeding the cables, the catenary wire, and the rails, the system will be separated into six sections in each tunnel. Each 4.6 km section will be fed by three inputs from a rectifier station. The train automation system will manage the trains so that no one section becomes overloaded with too many trains travelling uphill loaded. Empty trains travelling downhill at 50 km/h will feed approximately 400 kW of energy back into the grid.

The locomotives, as well as the loading and unloading chutes, will carry their own fire suppression systems, and will not require a fixed system within the tunnel.

An automated train dispatching system will be utilized to achieve a safe and efficient flow of trains through the tunnels, with no on-board operators. The system will be supervised from a control room located in the train maintenance shop (Figure 18.22). The automated system technology employs full radio-based train spacing and speed supervision on the whole railway system, except inside service areas/workshops. Trains find their position themselves via positioning balises and sensors. By transmitting the positioning signal to a radio block centre, it is always possible to determine which position on the route the train has safely cleared. The following train can then be granted a movement authority up to the released position. This "moving block" system operates so that the track section reserved for a specific train and its route is automatically adapted to the train length, the traffic situation, and the train's current actual and allowed speed. This system allows the trains to move with as short headway as their brake distance allows.

The train control system operates on a wireless communications system that must be in place for the entire track. It is assumed that wireless infrastructure will be in place in the MTT. While wireless communications (Wi-Fi) are the current state of the art technology for train control communications, it is recognized that more efficient and reliable communications may be developed in the future.

In order to minimize tunneling costs across the approximate 23 km length, tunnel dimensions are designed to minimum operable dimensions. The locomotive height is minimized for the 140 t class, and the manufacturer will attempt to reduce it even further during the next stage of design. The Siemen SCAT SR overhead catenary bracketing, which allows for the train's overhead electrical system to take up the minimum amount of tunnel back space, will be used instead of standard tray bracketing. Future studies will examine the cost/benefit of utilizing a tubular track system, instead of the standard ballast system, to potentially reduce tunnel height even further. Minimum clearances of 600 mm and 480 mm will be left on either side of train. The 480 mm clearance adheres to Transport Canada's standard respecting railway clearance in tunnels, and the 600 mm clearance allows some extra room in case personnel or supplies need to be moved around a parked train. At no point will there be personnel or other vehicles in the tunnels while the trains are in motion.

As shown in Figure 18.16, two parallel underground spur lines coming off of the main tunnel will be used for train ore loading at the Mitchell end of the MTT. A 15,000 t capacity single train loading bin will be installed underground in each of these spurs, or loading zones. Although one loading zone is sufficient for overall system operations, a second zone has been added for contingency. Figure 18.18 shows a cross section of the ore bin and the train ore loading arrangement. The bins will be fed from the top with crushed ore via a conveyor from the primary crusher, and the trains will be loaded via an inline load chute below the bins. One train length of track has been allowed for beyond the loading chutes to accommodate reversing into the loading zones.

The inside width of the load chute extension (rock box) will be 3.0 m. The inside width of the throat opening will be 2.5 m. The maximum opening between the throat gates and the chute floor will be 1.2 m. The load chute is designed to handle both crushed and uncrushed ore and will not have any hang-up issues with the proposed particle size distribution from the primary crusher. The proposed chute will provide for safe maintenance access to the feed chamber, and the chute, without interrupting the rail operations. Personnel will be able to access the loading zones through the MTT, approximately 450 m from the Mitchell portal. A vertical escape way for personnel will be included between each loading chamber and the crusher conveyor tunnel above it.

An operator will load the train manually from the control room, which is conceptually shown in the train maintenance shop (Figure 18.22). The operator will be able to control, adjust, and follow the loading with complete emergency stop functions. Complete local control functions will also be available. The train controller will have a CCTV view of the loading process. It has been found in existing applications that the manual operation of loading ROM material is more effective than an automated system, due to issues associated with remote sensing the amount of ore in the wagon and the variable loading rates that are experienced. Future developments in technology may enable full automation.

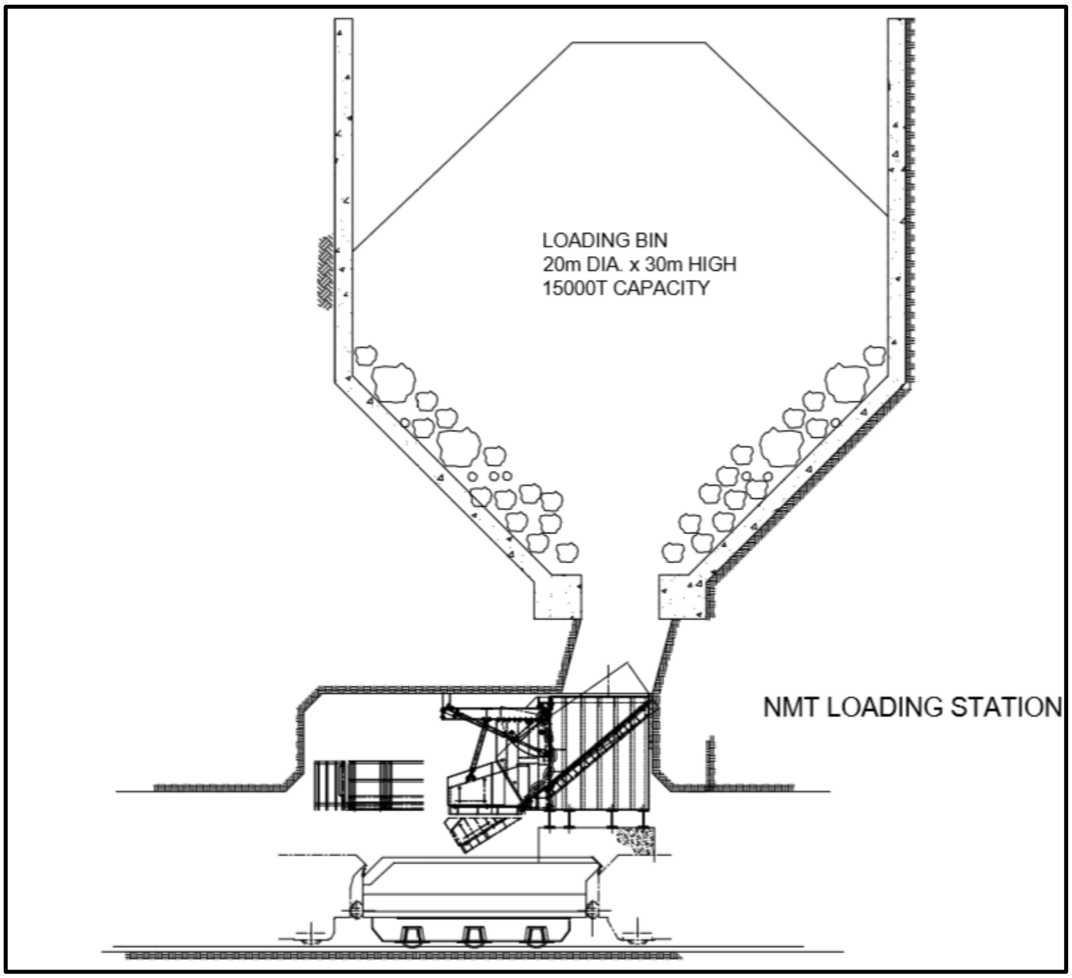
Trains will be continuously loaded without the need to stop the train between cars. Each 42 m^3 car is loaded in an average of 36 seconds. The mine ore cars adopted for this study are end hinged cars with an overlapping apron between the cars. This apron minimizes spillage between the cars during loading and train operation. Straight track will be utilized under the loading chutes to minimize derailment risks.

There are no additional provisions for the removal of tramp metal from the ore. The train control system proposed includes a profile detector to automatically stop over height trains prior to the start of the overhead catenary electrification wire; although the controlled loading of the trains with already crushed material means an overloaded situation should not occur. If a car is overloaded, the operator will reverse the train under the chute lip to rearrange the material between the cars; removal of material will not be required.

Approximate train weighing is currently included on the locomotives; however, the train control system can be specified to include automatic train weigh sensors after the loading chutes and the unloading station. The system can be used to manage over loading and under loading at chutes, together with identifying issues with particular ore cars.

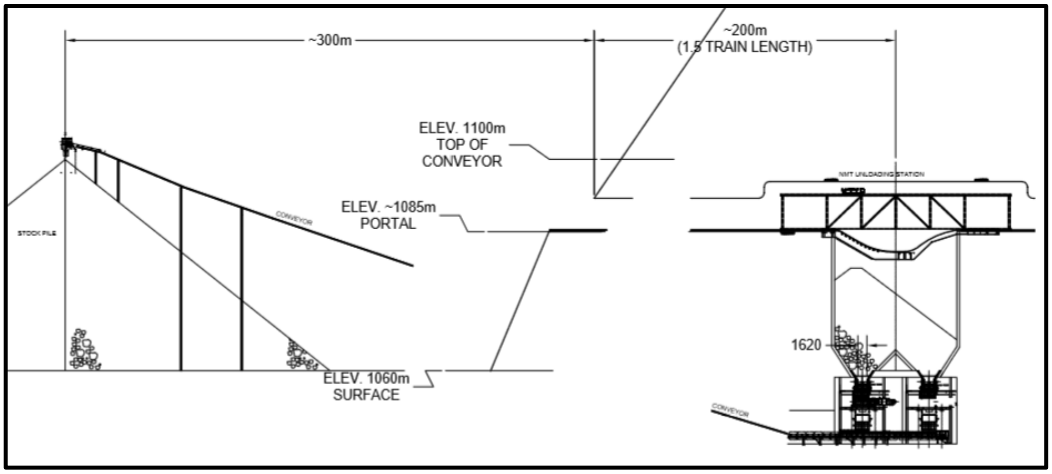
Embedded concrete track is specified at load chutes and the unload station, where some spillage is inevitable. This type of track will facilitate clean up by a track maintenance vehicle, which is included as part of the equipment fleet.

Figure 18.18 Ore Loading in MTT at Mitchell



As shown in the plan view of the overall system layout (Figure 18.16), a single underground train ore unloading zone is included at the Treaty end of the MTT. This has an ore bin capacity of 15,000 t. Figure 18.19 shows a cross section through the train unloading station and ore bin.

Figure 18.19 Ore Unloading in MTT at Treaty

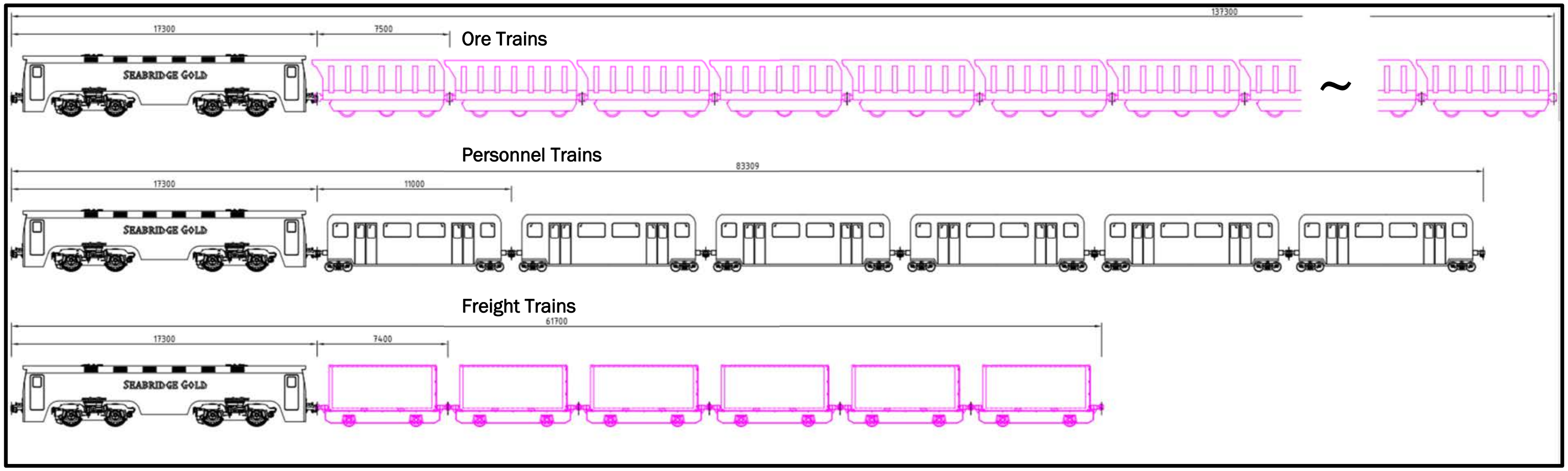


Trains must pass through the unloading station in one direction only, then cross over to the other track on the opposite side of the unloading station. Only a single train will operate in the unloading station at one time. The bottom discharge ore cars will continuously and automatically unload through the station to the bin underneath, with each car taking an average of nine seconds to unload. The train will be driven via traction drives across the unloading station at a maximum speed of 2.5 km/h. Two overhead travelling cranes will be installed for maintenance.

From the bottom of the ore bin, ore will discharge into two apron feeders and onto a conveyor belt that will transport the ore to the surface and feed to the Treaty COS. Maintenance personnel will access the conveyor tunnel through its portal on surface, as well as a vertical escape tunnel that joins up the unloading station and the MTT above.

Figure 18.20 shows the lengths of the typical train consists used for transporting ore, personnel, and freight. Personnel and freight transportation is described in more detail in Section 18.4.1.

Figure 18.20 Train Consists for Ore, Personnel, and Freight Trains (Dimensions in Millimetres)



18.4.1 MTT Freight and Personnel Transport

The train system will also be used for transportation of personnel and freight between the Treaty and Mitchell areas via the MTT. Freight and personnel transport will be scheduled on a daily basis, and the transport trains will be controlled by the automated train control system. Specially configured personnel and freight trains will transport personnel, freight, and fuel through the MTT, with marshalling and unloading areas at each end, separate from the ongoing ore transportation facilities. Personnel, freight, and fuel handling will only be scheduled during the day shift operations.

These transport trains will use the same 140 t locomotives as the ore trains, but will have specialty cars for personnel, freight, and fuel transport. Train consists are shown in Figure 18.20. Each train will be able to pull between six and 12 specialty cars per trip.

Each personnel car will carry up to 60 passengers and will be outfitted for underground operation. They will include pneumatic braking and independent parking brakes. The passenger wagon will be fitted with a heating, ventilation, and air conditioning (HVAC) system to ensure that personnel comfort is maintained at a suitable level in all operating conditions, and that no high-velocity air will impinge on personnel. The windscreens will be manufactured from safety glass. The cab side windows will be tinted to minimize light/heat transmission.

A 24 V direct current (DC) battery will be fitted to each personnel wagon and will supply the emergency load (lighting, ventilation, communications, and door controls) for a period of two hours for all train consists (with the battery charged to 80% of its full capacity), should there be a failure of the primary or auxiliary power supply. Fault detection will be provided on the battery and control circuits so that any fault will be indicated. Each personnel wagon will be outfitted with appropriate PPE and self-contained breathing apparatus (SCBA) for all passengers, in case of an emergency event that forces personnel to abandon the wagons within the MTT.

Freight cars with a 50 t capacity will be multi-functional and allow for transporting of interchangeable modules as required. They can be left as flatbeds and outfitted with freight directly, or they can be outfitted with sea-can trailers carrying freight, or they can be outfitted with tanks carrying liquid freight. The freight cars will include pneumatic braking and independent parking brakes.

The train control system will ensure that dangerous goods, such as fuel and explosives, are never present in the tunnel during personnel transport. The MTT will be gated on both ends, and personnel will be restricted from entering, except for maintenance. As described previously, segments of the system with maintenance personnel will be isolated and locked out without compromising total system operations.

Fuel and explosives delivery will be scheduled during available track time between trains. Simulation has shown that 21 operating hours per day are required to achieve the average daily ore production, with the remaining time in the day to be utilized for fuel and explosives delivery. Personnel and freight train trips will be sequenced with the ore train transport. Estimated freight, fuel, and personnel movement requirements through the

MTT will call for a daily average of five return train trips that have been included in the train production simulations. The estimated daily quantities are shown in Table 18.15.

The CCAR, when seasonally available, will be used to transport heavy volume or oversized items.

Table 18.15 Estimated Freight, Fuel, and Personnel Transport Daily Requirements

Transport	Unit	Amount
Lime	t/d	350
Ammonium Nitrate	t/d	175
Other Mine Operations Supplies	t/d	20
Camp Supplies	t/d	5
Fuel	L/d	400,000
Persons	persons/wk	360

Treaty Staging Areas

On-surface staging areas near the Treaty portal, shown in Figure 18.21, will be used to load personnel, freight, and fuel onto the specialty train cars. These staging areas will be road accessible and include a laydown area for all freight that is for transport through the MTT. Gantry cranes will load the flatbed train cars with freight payload. Empty fuel train cars will be loaded via fuel lines from the main Treaty fuel storage tank. Future studies will detail the safe marshalling procedure for the dangerous goods, including fuel and other combustibles, raw materials for the Mitchell explosives plant, and chemicals. There is sufficient space at the Treaty marshalling yard to accommodate isolation of various goods.

Freight and fuel marshalling out of the north tunnel will be separated from the personnel marshalling out of the south tunnel. The personnel loading area will include a structure to protect waiting passengers from the elements.

A train maintenance shop will be located in the freight marshalling area, as well as an extra track to park excess ore cars when not in operation. The workshop facilities are enclosed in a building on the south side of the pad, located outside the Treaty portal. The layout of this facility is shown in Figure 18.22. These facilities also include the control room for train system operations. Maintenance personnel for the train system are estimated as a standalone operation, and will work out of this shop. An opportunity exists to integrate this group under the umbrella of the overall Treaty OPC operation and maintenance crews.

Catenary for the 140 t electric locomotives ends at the portals, shown as purple arrows in Figure 18.21. Specialty personnel, freight, and fuel train cars will be shunted by 20 t battery locomotives into the tunnels, picked up by the 140 t electric locomotives, and transported from Treaty to Mitchell.

Figure 18.21 Treaty Personnel, Freight, and Fuel Staging and Marshalling

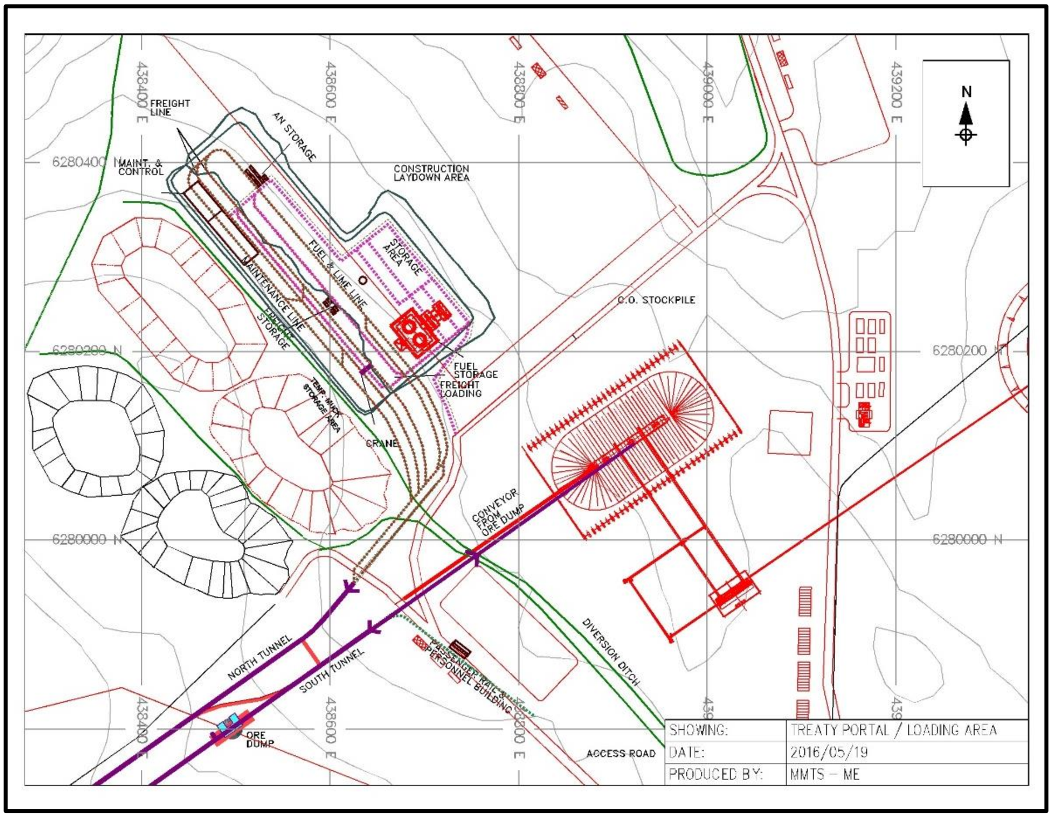
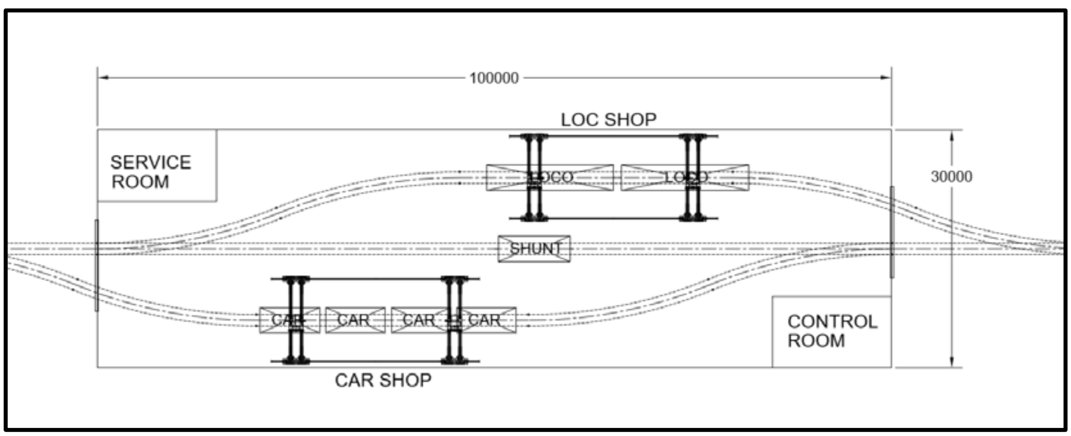


Figure 18.22 Treaty OPC Train Maintenance Shop



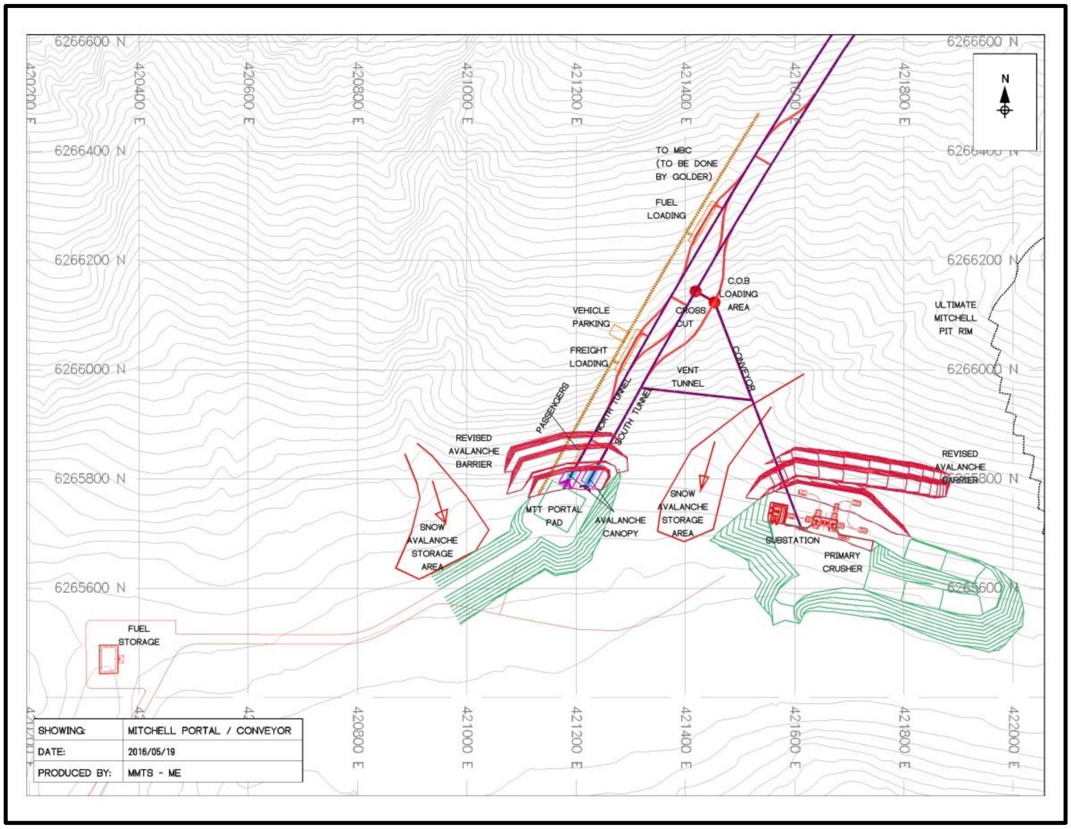
MITCHELL STAGING

Three separate, enclosed, underground staging areas near the Mitchell portal will be used to offload passengers, freight, and fuel, respectively (shown in Figure 18.23). The freight and fuel staging areas will be isolated with fire/explosion proof doors. Loaded fuel tanks and other hazardous freight will be shuttled out of the tunnel as soon as possible; there is no planned long-term underground storage for these materials. The transportation of fuel, including the shuttling out of the tunnel, will not occur during the transportation of other flammable/explosive freight or personnel.

Personnel will exit the Mitchell portal by bus or other light vehicle. In the event that the portals are inaccessible, personnel will be able to exit the MTT via the vent and conveyor tunnels towards the Mitchell OPC (as shown in Figure 18.23).

Freight and fuel staging areas will include gantry cranes to offload the train payloads onto awaiting flatbed tractor-trailer units. Freight will be driven out to its ultimate destination at the Mine Site. Fuel will be transported by tractor-trailer units to a nearby on-surface fuel depot, or directly to pit fueling stations, depending on the needs of the operations. Fuel train cars will be re-loaded with an empty fuel tank for return to Treaty.

Figure 18.23 Mitchell Personnel, Freight, and Fuel Staging and Marshalling



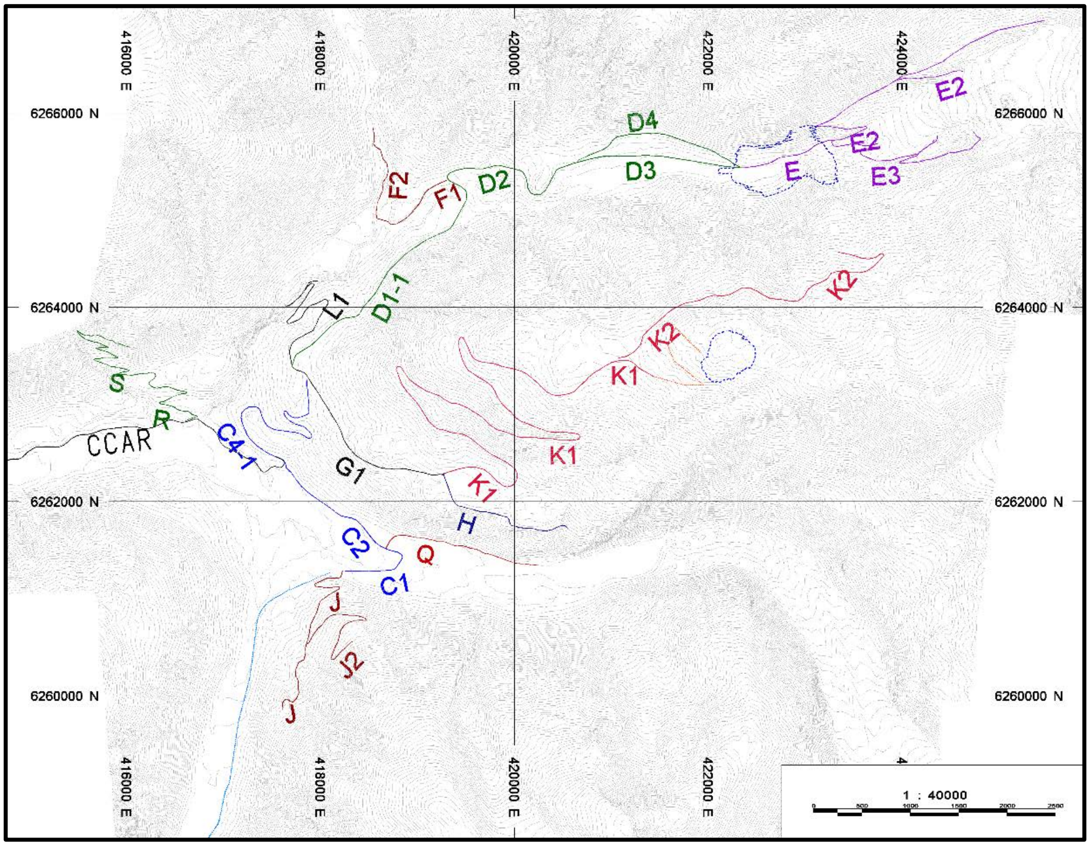
18.5 SITE ROADS

Currently, the KSM site can only be accessed by helicopter. Helicopter support will augment the road pioneering work and construction camps set up.

Avalanche protection will be constructed where appropriate so that work can be safely carried out at the tunnel portals. Rock storage landforms will be developed adjacent to the Mitchell pit.

The Mine Site roads are outlined in the Figure 18.24, with the starter pits outlines shown in blue. Road numbers and corresponding descriptions are shown in Table 18.16.

Figure 18.24 Mine Site Roads



BASIC ROAD DESIGN CRITERIA

This section outlines some of the basic design criteria for the Mine Site roads.

Maximum Grade

The maximum grade for haul roads is 8% and for access roads is 10 to 15%.

Road Width

Haul road widths are designed to comply with the following BC Mines Regulations:

- for dual lane traffic, a travel width of not less than three times the width of the widest haul vehicle used on the road
- for single lane traffic, a travel width of not less than two times the width of the widest haul vehicle used on the road
- a berm height of at least three-quarters the height of the largest tire on any vehicle hauling along the road, where a drop-off of greater than 3 m exists.

Ditches are included within the travel width allowance.

CUT-AND-FILL VOLUMES

Road cut-and-fill volumes are estimated using the Project topography surface. Final road widths are estimated based on main road use. The following road widths are designed:

- haul truck routes – 38.2 m
- major personnel transport route and equipment route – 20 m
- explosives access routes – 15 m
- lightly travelled roads (personnel and small pick-ups only) – 6 m
- pioneering roads – 3 m.

Roads are designed in balanced cut and fill where possible to reduce excess material requirements. The final road construction volumes are calculated for each road using the 3D shapes designed and the topography surface provided. The road volumes are shown in Table 18.16. Volumes are reported in bank cubic metres (bcm), which represents the in situ volume. A swell factor of 30% is used to convert bank cubic metres to placed fill volume (cubic metres).

Table 18.16 Mine Road Cut and Fill Estimates

Road	Description	Cut (bcm)	Fill (m^3)	Drill and Blast (%)
C1	Camp 2 to Bailey B1 (Sulphurets Creek)	4,470	44,275	0
C2	Bailey B1 to C3/C4 Junction	47,253	73,308	25
C4-1	Initial Maintenance Access	155,909	226,293	25
C4-2	Maintenance Access	0	331,632	75
D1-1	Maintenance Facility to Mitchell Valley	402,073	1,259,546	50
D2	Bailey B2 to Camp 4 (MTT Portal)	294,711	280,331	75
D3	Camp 4 to start of E	85,043	289,693	50
D4	Mitchell OPC Access	10,114	22,940	50
E	Mitchell Diversion	56,719	21,438	50
E2	MDT North Ditch Access	221,007	105,801	50
E3	MDT Sub-glacial Inlet Access	54,268	109,737	50
F1	Camp 4 to Bailey B4 (McTagg)	24,223	28,314	25
F2	Bailey B4 to North End MTDT	18,569	31,730	50
G1	Sulphurets Access Road (from Camp 9)	76,634	1,061,278	75
H	End of G1 to South End MDT Portal	40,698	50,955	25
J	Explosives Facility Access	101,312	116,136	50
J2	AN Prill Access	234,100	243,115	50
K	Sulphurets Pit Access	1,443,893	4,773,854	75
K2	Sulphurets Ridge alternative access to OPC	259,565	412,793	75
L1	Southeast WSF Access	196,213	140,999	75
Q	Upper Sulphurets Powerhouse Access Road	7,222	10,699	25
R	Camp 9 to South End MTDT	10,924	13,278	25
S	Camp 9 to South End MTDT Stage 2 and 3	46,552	32,901	75

PIONEERING ROAD VOLUMES

An initial 3 m wide pioneering road will be built along each road alignment using dozers and, where required, small excavators and drills. Pioneering road sizes are small relative to the accuracy of the topography; therefore, an allowance is made to account for the volumes and costs required.

FINAL ROADS

Roads are widened from the pioneering width to the final width after all critical mine areas have been accessed (WSD CDT, MTT portal, diversion tunnel inlets/outlets, etc.). Some roads will require fill to be placed with mine haul trucks to achieve final road width. The road between the Sulphurets quarry and the WSD is one such example.

18.6 PROCESS PLANT FACILITIES

The Mitchell OPC will include primary crushing and material handling systems, including conveying.

The MTT will extend from the north side of the Mine Site, approximately 23 km to the northeast, into the upper reaches of the Treaty OPC. The tunnels will transport ore, primarily crushed at the various crushing stations at the Mitchell OPC, by rail cars to the Treaty OPC. The process facilities at the Treaty OPC will include secondary and tertiary crushing, grinding, flotation, concentrate dewatering, concentrate loadout facility, cyanide leaching on gold bearing pyrite tailings, cyanide recovery and destruction, and flotation tailing/residue delivery facilities.

The main process equipment will be housed in structural steel buildings, complete with overhead cranes, electrical rooms, HVAC, and offices.

The process related facilities are detailed in Section 17.0.

18.7 ANCILLARY BUILDINGS

Ancillary building construction considered for the study will be pre-engineered, stick-built, or modular structures, as applicable. The HVAC for these buildings will be designed to industrial standards. The following ancillary buildings are included in the study:

- Treaty OPC:
  - fuel storage facility
  - fuel distribution station
  - administration building
  - assay and metallurgical laboratories
  - warehouse and maintenance building
  - concentrate storage/load-out building
  - cold storage/reagent storage building

- first-aid building
- ore train storage yard, maintenance shop and loading/unloading facilities
- permanent operations camp
- temporary construction camps
- potable water treatment plant
- sewage treatment plant
- incinerator
- substation and auxiliary power supply facilities
- construction laydown area
- pre-construction fuel storage

- EPCM and contractors' offices, concrete batch plant, construction camps, TWTPs, and numerous other construction related facilities
- Mitchell OPC and lower Mine Site areas:
  - truck shop including first aid facilities
  - HDS WTP with sludge storage facilities
  - Selenium WTP (operational by Year 5)
  - diesel fuel storage and dispensing
  - permanent operations camp
  - temporary construction camps
  - sewage treatment plants
  - incinerator
- off-site facility:
  - new concentrate storage and loadout at the Stewart port facility.

18.7.1 TREATY OPC

FUEL STORAGE AND DISTRIBUTION (PERMANENT AND CONSTRUCTION)

The main fuel storage tanks at the Treaty OPC are sized to store 2,500,000 L, which is enough for 6 days of fuel requirements. All fuel storage areas will be lined with containment berms and approved double-wall type tanks. Additional fuel stations will be located near the Mitchell OPC, truck shop, and at the Sulphurets and Kerr pits. Gasoline will also be similarly stored where required.

The majority of the fuel requirement is for mining activities at Mitchell; however, some fuel will be distributed to all Treaty OPC facilities via pipelines from the Treaty Fuel Storage Tank to the required facilities.

A pipeline from the Treaty Fuel Storage Tank to the train marshalling area will be installed, with hook-ups for fast fuel transfer directly to the ISO fuel tanks at the marshalling area. Wherever possible, the ISO fuel tanks will not be removed from the train cars on the Treaty end.

An additional allowance for the ability to fill ISO fuel tanks sitting on trailers next to the Treaty fuel storage tank is also recommended. This will allow for back-up mobile fuel transport from the Treaty storage tank to the trains, or to other Treaty OPC facilities. Cranes in the train marshalling area will load/unload the ISO fuel tanks between the tractor-trailers and the trains, if required.

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, Metallurgical, Acid-base Accounting, and Geotechnical Laboratory

The pre-engineered 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 as well as tunnel muck, site excavation and dam construction materials. The laboratory will be a 815 m^2 single-story structure.

First Aid Buildings

The first aid buildings will be pre-engineered structures, located at both the Mine Site and Treaty OPC, equipped with first aid facilities and emergency vehicles.

Concentrate Storage

The on-site concentrate storage facility will be a pre-engineered structure, approximately 2,000 m^2 in area. It will have a five-day storage capacity equating to approximately 4,600 t of concentrate. Concentrate will be loaded into trucks at the Treaty OPC and hauled to a concentrate storage and load out facility at the Stewart, BC port facility.

Cold Storage/Reagent Storage Building

The cold storage/reagent storage building will be located at the Treaty OPC and will be approximately 1,200 m^2 in area.

Permanent and Construction Camps

The Treaty construction and operating camps will service the PTMA and will provide accommodation for Treaty Process Plant personnel, camp services personnel, visitors, and other ancillary personnel during construction and operation. Both camps are located near the TCAR, approximately 0.5 km west of the Process Plant. To reduce redundancy and minimize footprint, the two camps will share some components such as water intake, liquid/sewage waste disposal systems, WTP, etc. The Treaty construction camp will be built in stages, initially in Year -5. In Year -4, the camp will be built to its maximum capacity to service the highest activity in Year -3 and Year -2. The Treaty operating camp is planned to be used for most of the construction period and the entire life of mine; however, after about 20 to 25 years of operation, the camp will be refurbished as components reach the normal end of their operating life.

The camp components will include: accommodation, office/recreation complex, kitchen/diner, parking, sanitary sewer, potable water treatment, waste water treatment and disposal field. A 500 kW diesel generator will provide electricity until the site is connected to the provincial electricity grid, after which time the generator will be retained as backup.

WAREHOUSE AND MAINTENANCE BUILDING

An 800 m^2 warehouse and maintenance pre-engineered building will be constructed at the Treaty OPC. It will be located adjacent to the cold storage facility.

Some warehousing facilities will also be constructed at the Mine Site.

18.7.2 MINE SITE

TRUCK SHOP

The Mine Site truck shop will be a pre-engineered building, approximately 9,500 m^2 in area. 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 Mine Site area, near construction camps Nos. 9 and 10.

The truck shop/mine dry will comprise eight maintenance bays, two light vehicle repair bays, a truck and lube bay, a truck wash bay, a welding and machine shop, an electrical and instrument shop, a 1,200 m^2 storage warehouse with an upper level mezzanine area, and a dry area including lockers, offices, restrooms, first aid, and emergency vehicle storage. Waste oil will be disposed of in the refuse incinerator with any remaining oil removed and disposed of at an approved facility.

ACID-BASE ACCOUNTING AND GEOTECHNICAL LABORATORY

The pre-engineered laboratory will be located in a separate building near the Truck Shop. It will be equipped to perform daily analysis of tunnel muck, site excavation and dam construction materials. The laboratory will be a 815 m^2 single-story structure.

PERMANENT AND CONSTRUCTION CAMPS

The Mitchell operating camp will be located between Sulphurets and Gingras creeks, just north of the CCAR. Like the Treaty operating camp, the Mitchell operating camp is planned to be used for most of the construction period and the entire life of mine; however, after about 20 to 25 years, it will be refurbished as components reach the end of their normal operating life. At the end of the mine life, the Mitchell operating camp will be the base of ongoing operating and maintenance activities for the HDS WTP and hydro-power facilities, with a reduced workforce.

The camp components will include a helipad, sleeping dorms, parking, fuel storage and loading area, recreation facility, sewage treatment, fire/fresh water tanks, generator, laundry, and kitchen/diner. Like the Treaty operating camp, a 500-kW diesel generator will provide power until the site is connected to the provincial electricity grid and will be retained for emergency backup.

During construction period there will be four construction camps erected in the Mine Site area: Ted Morris (Camp No. 2), Mitchell Initial (Camp No. 9), Mitchell North (Camp No. 4), and Mitchell Secondary (Camp No. 10).

Ted Morris Construction Camp No. 2 will be established first to support early works beginning in Year −6. Camp No. 2 will be deployed at the north end of the Frank Mackie Winter Access Road with helicopter support and become the headquarters for receiving equipment and materials arriving via the winter road from the Granduc Staging Area. Initial Mine Site road and CCAR construction will be based out of this camp.

The Mitchell Initial Camp No. 9 will be built in early Year −5 to support early construction activities such as mine internal roads, logging, site preparation and rough grading at the Mine Site. The Mitchell North Camp No. 4 will be built in the same construction year to support the MTT Mine Site portal, pads, ponds and TWTP No. 6 construction activities in the Mitchell OPC area. The Mitchell Secondary Camp No. 10 will be built in stages and reach its maximum size to service the highest activity from Years −4 to −2.

18.7.3 LANDFILLS

Two landfills are proposed to be permitted and developed, one for the Mine Site and one for the PTMA.

The Mine Site landfill, which will occupy approximately 6.5 ha, will be located within the Sulphurets laydown area. Any runoff or seepage will be collected and directed to the WSF and subsequently treated in the HDS WTP. A fresh water diversion will be constructed upslope of the proposed landfill site.

The PTMA landfill, occupying approximately 8.4 ha, will be located near the Treaty operating camp. Fresh water will be diverted around the site, and runoff from the landfill will be managed along with other contact water from the Treaty OPC. The landfill will also include an area for storage of contaminated snow from the Treaty OPC winter snow removal activities.

Each landfill will include a land farm. The land farms will accept contaminated soils from spill clean-ups and leaks, while the landfill will be used to dispose of non-inert, dry industrial, and forestry waste.

18.8 SEWAGE

Seabridge contracted McElhanney to design each construction and operation camp for the Project in accordance with the requirements of BC laws and regulations for industrial camps (e.g., Municipal Wastewater Regulation [MWR; BC Reg. 87/2012] of the *Environmental Management Act* [2003], Sewerage System Regulation [BC Reg. 326/2004], and *Sewerage System Standard Practice Manual* [BCOSSA 2007]). Each camp location was field checked, and the Project baseline data on soils, vegetation, groundwater quality, and water quality were used to design the wastewater treatment system. Each wastewater treatment system design is specific to ground conditions at the camp location and the water quality of the receiving environment.

The wastewater treatment system installed at the various camps will treat the anticipated maximum daily flow through a variety of processes to meet a secondary level of treatment as defined in the Municipal Wastewater Regulation (MWR) (BC Reg. 87/2012) of the *Environmental Management Act* (2003) for camps with occupancy greater than 100, or Type 3 effluent quality as defined in the *Public Health Act* (2008) sewerage guidelines for camps with occupancy less than 100. In general, the following treatment processes will be supplied for all construction and operating camps in accordance with BC laws and regulations (e.g., MWR [BC Reg. 87/2012], Sewerage System Regulation [BC Reg. 326/2004], and *Sewerage System Standard Practice Manual* [BCOSSA 2007]). Camps will include treated effluent storage and sludge digestion and dewatering, or a recirculating loop for nitrogen removal, except for camps with an occupancy of 100 or less. Additionally, the following will be submitted for each camp: Water System Application for Construction, Water System Operation and Maintenance Plan, Wastewater System Operation and Maintenance Plan, and a combined Emergency Response Plan for the Water and Wastewater Systems, in accordance with the appropriate regulations. The wastewater treatment system processes will include:

- screening
- flow equalization
- primary settling/primary air flotation (gravity separation)
- aeration tank (bio-chip reactor)
- secondary dissolved air flotation (gravity separation)
- effluent filtration
- disinfection via ultraviolet or chlorine
- treated effluent storage
- sludge digestion and dewatering
- recirculating loop in the sewage treatment plant to promote nitrogen removal.

18.9 COMMUNICATIONS SYSTEM

Permanent communications utilizing fibre-optic cables will be installed over the existing NTL, and the new 30 km spur transmission line to the Treaty OPC. Installation of fibre-optic cable on site will also allow for the installation of dedicated cellular service.

A fibre optic communication system will be installed in conjunction with the power distribution system in both the Treaty OPC and the Mine Site. A fibre-optic cable has been included in the MTT to provide communications between Treaty OPC and the Mine Site.

An ultra-high frequency (UHF) radio system will be used for mobile communications in both the PTMA and the Mine Site. Base stations and repeaters will be installed as necessary on ground and inside tunnels.

Treaty OPC wired telephone service will be provided by a Voice over Internet Protocol (VoIP) system. A local cell phone system is also planned, as is satellite television for the camps.

In addition, uninterruptible power supplies will be used to provide backup power to communication systems and critical control systems to facilitate orderly shutdown of process equipment and to back up computers and control systems.

18.10 Fresh and Potable Water Supply

Fresh and potable water for the Treaty OPC will be supplied from nearby wells to an elevated storage tank approximately 12 m in diameter and 9 m in height. Fresh water will be used primarily as:

- fire water for emergencies
- cooling water for mill motors, mill lubrication systems and reagent preparation
- the potable water supply.

By design, the fresh water tank will be full at all times and will provide at least two hours of fire water in an emergency.

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.

Fresh and potable water for the Mine Site will be supplied from nearby wells as well.

18.11 Power Supply And Primary Distribution

Power generation and transmission utilities in BC are regulated by the British Columbia Utilities Commission (BCUC), acting under the *Utilities Commission Act*. BC Hydro generates the majority of power in BC, although there are an increasing number of private, IPPs. BC Hydro owns and operates the major transmission and distribution system in BC and is the electric utility that would serve the Project via the newly constructed NTL.

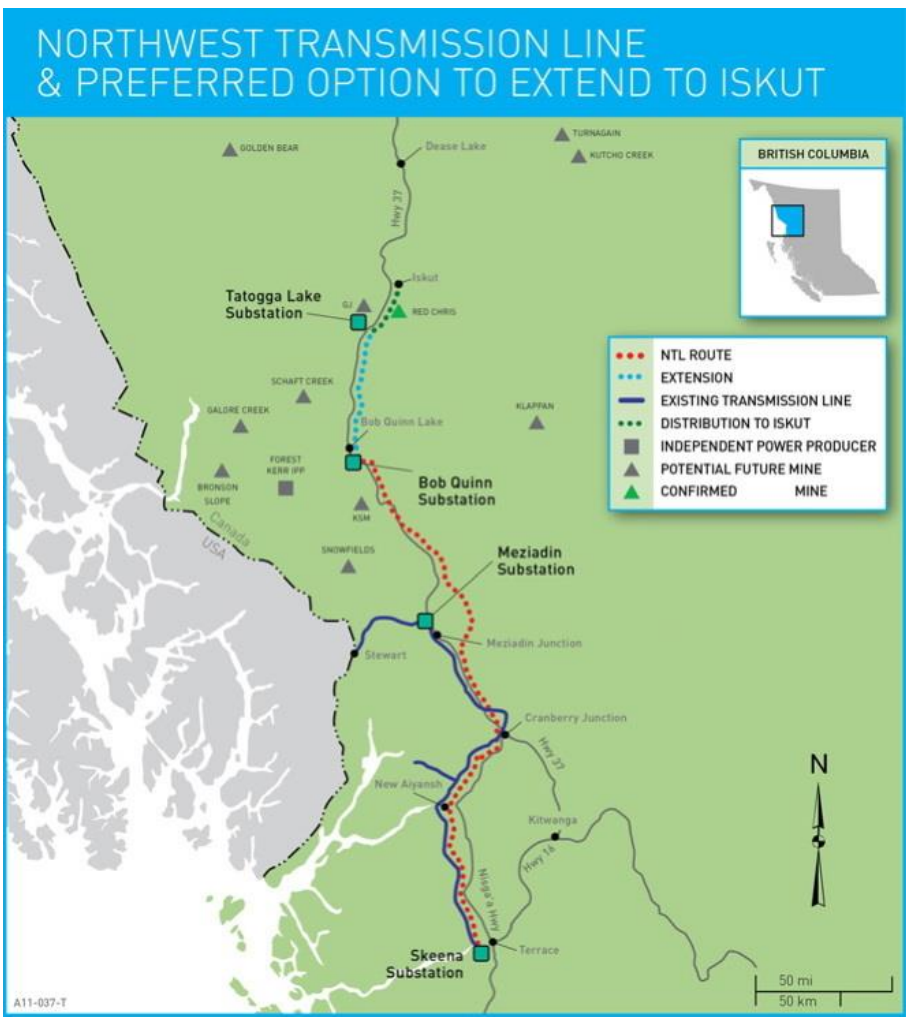
The interconnection capital cost for the Project will be as set out in a facilities agreement (the Facilities Agreement), an arrangement approved by the BCUC in January 1991, pursuant to Order G-4-91 that sets out the rights and obligations of BC Hydro and the customer for construction, ownership, and operation of the facilities necessary for electric service. The Facilities Agreement will be in accordance with the Project Facilities Study carried out by BC Hydro. A draft copy of the Facilities Study has been issued by BC Hydro.

18.11.1 Northwest Transmission Line

The 344 km long, 287 kV NTL runs from the Skeena substation near Terrace, BC, to a new substation near Bob Quinn Lake (Figure 18.25). This new transmission line was commissioned in the summer of 2014 and currently serves the AltaGas Forrest Kerr Hydroelectric Facility and the Red Chris Mine. A tap from this transmission line will service the Project.

Due to an overrun in the construction cost of the NTL, BC Hydro Tariff Supplement TS37, as approved by the BCUC, was put in place requiring NTL customers to share in the overrun cost. In accordance with TS37, based on a Project contract (peak) demand of 200 MVA, the required contribution will be just over Cdn$209 million. This amount is separate from system reinforcement and is a required cash contribution. Payment of the tariff is not due until the start of commercial production, and BC Hydro offers the option of spreading the payments out over five years, with an applicable finance charge. This required cost contribution is included in the Project sustaining capital costs.

Figure 18.25 NTL Project Map



Source: BC Hydro

18.11.2 Treaty Creek Switching Station

BC Hydro is responsible to deliver power from the transmission system to a customer at the point of delivery (POD). The customer is responsible to bring power from the POD to their site. For the Project, the POD will be the Treaty Creek Switching Station. The KSM site will take electrical service via a 30 km long, 287 kV line extension from the Treaty Creek Switching Station, to be located on the NTL adjacent to Highway 37, approximately 18 km south of Bell II. This installation will also be in the vicinity of the Treaty Creek Access Road junction with Highway 37. Metering will also be located at this point. The Treaty Creek Switching Station will form part of the BC Hydro system and will be constructed, owned, and operated by BC Hydro. BC Hydro has completed site selection and preliminary design.

Seabridge previously reimbursed BC Hydro for the cost of installing transmission line dead end structures when the NTL was constructed, as required to facilitate the connection of the of the proposed Treaty Creek Switching Station into the grid.

The power supply facility infrastructure will include BC Hydro System Reinforcement and the Basic Line Extension, which is the circuit breaker and metering at the POD. It is not the transmission line to the Project site. Construction of the Treaty Creek Switching Station, as per the BC Hydro Facilities Study, will require a direct cash payment from the Project to cover a large part of the cost of the installation that is classified as the Basic Line Extension. This cost is included in the KSM capital cost estimate. The remaining station cost is classified as System Reinforcement and is not a project capital cost. However, Tariff Supplement (TS) No. 6 (TS 6) Clause 5 (c) “Offset” requires that the customer provide bonding for up to seven years, such that BC Hydro is assured of receiving enough revenue from the Project to justify the capital expenditure. Security is required in the form specified in TS 6 Clause 13 and in the amount as per Clause 5(b). The foregoing bonding and charges are set out under the tariffs and are not negotiable. BC Hydro will return part of the security each year as the offset, based on power billing, reduces the required bonding. The amount of the bonding is not included under the direct project capital cost budget, but is otherwise accounted for in the Project economics. An important point to note is that as per the tariffs, the customer must pay the actual final cost of construction, not the amount estimated by BC Hydro in a Facilities Study.

BC Hydro is responsible for obtaining all approvals and permits for the Treaty Creek Switching Station. A formal environmental assessment of the Treaty Creek Switching Station under BC’s environmental assessment process for reviewing major projects is not required.

The Treaty Creek Switching Station does not require long delivery items such as high-voltage power transformers. The in-service date will depend on the time of year a power supply agreement is signed; however, the engineering and construction would not require more than three years and could be expedited.

18.11.3 TRANSMISSION LINE EXTENSION TO KSM

The voltage selection for the proposed 287 kV transmission line extension for the Project was based on the 287 kV transmission voltage selected for the NTL. A review of the technical requirements to serve a load larger than 150 MVA confirmed that stepping the voltage down to a lower level is not technically acceptable nor economic.

The Project will be responsible for the construction and operation of the transmission line extension, in accordance with the established BC Hydro tariff requirements. Line construction will utilize steel monopoles, such that the line can be generally run in the TCAR right-of-way, beside the road, thus largely eliminating the requirement for a separate access route.

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 TCAR, to following the South Diversion Cut-off Ditch, up to Substation No. 1 at the Treaty OPC. Steel monopoles are ideal for use where a transmission line is to be constructed next to a road and in areas of high snow fall. To protect against avalanche damage, several structures will be mounted on concrete piers, to raise the pole bases above the avalanche flow.

As may be required to meet Project schedules, the construction of the 287 kV line extension from Treaty Creek to Substation No. 1 at the Treaty OPC could logically commence in the second year of the Project construction schedule, after the access road is complete. Construction of the line could easily be completed in one summer and fall construction period, and Substation No. 1 could be completed by the end of Q1 of the following year.

The environmental assessment for the 30 km section of transmission line from Treaty Creek to Substation No. 1 was included in the Project environmental assessment process; therefore, approval is in place. Land tenure has been obtained for the transmission line right-of-way, from the Treaty Creek Switching Station to Substation No. 1 at the Treaty OPC.

The transmission line includes a fibre optic cable connection to the BC Hydro NTL fibre-optic cable system as required by the utility. This fibre connection will also carry the general communications to site for the permanent operations phase.

The transmission line land tenure has been obtained. No additional (specific) permits will be required other than the general mine permitting. The cost of right-a-way clearing is included in the road clearing budget. Further information can be found in the Treaty Creek Transmission Line Report and the Treaty Creek Transmission Line Clearing Criteria Report located in Appendix I1 and I2, respectively.

18.11.4 SYSTEM STUDIES

BC Hydro performs studies to determine the cost, method, and timing of transmission system customer interconnections. Seabridge first commissioned a BC Hydro System

Impact Study for the Project in 2009 to confirm the technical viability of the interconnection. Subsequently, several updates were commissioned to account for the construction of the NTL and several project changes, including changes to the Project access route from Highway 37. Seabridge also commissioned a Facilities Study, which is the next step in the interconnection process that defines interconnection costs. Due to Project changes, several draft editions have been issued. Currently, BC Hydro is making minor updates to the 2015 Facilities Study issue, based on technical changes resulting from their operational experience with the new NTL transmission line. Upon completion of this update, a Facilities Agreement and an Electricity Supply Agreement may be signed, which will finalize power supply to the mine.

System load flow studies have been performed by 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 power transformers have automatic tap changers and that there is automatic control of the process plant synchronous ball mill drive motor excitation systems for instantaneous voltage control, as requested by BC Hydro. Substation No. 1 also includes a ±20 MVA static var compensator, as identified by the BC Hydro System Impact Study as required, to ensure system transient stability.

The Project consultants 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 Main Substation No. 1 (25 kV bus bars)
  - momentary: apply American National Standards Institute (ANSI) factors (Appendix A of short-circuit study)
  - required interrupting: 23 kA minimum
- Mitchell Substation No. 2 (25 kV bus bars)
  - momentary: apply ANSI factors
  - required interrupting: 18 kA minimum.

Service from the Skeena Substation to the 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.

18.11.5 Electric Utility Requirements, Tariffs, And Cost of Electric Power

The electric service to the Project (including all terms and conditions such as rates and metering requirements, connection charges, and many aspects of the KSM connecting transmission line) will be in accordance with the latest edition of BC Hydro Electric Tariffs, in particular:

- Rate Schedule 1823 "Transmission Service – Stepped Rate" effective April 1, 2016
- Rate Schedule 1901 – Deferral Account Rate Rider
- BC Hydro Electric Tariff Supplement No. 5 (TS5) Agreement for Customers Taking Electricity under 1821 (1821 is now 1823) (TS5 is a template for the Electricity Supply Agreement with the format set as per the tariffs and is not subject to change)
- BC Hydro Electric Tariff Supplement No. 6 (TS6) Agreement for Transmission Service Customers (TS6 is a fill in the blanks template for the Facilities Agreement with the format set as per the tariffs and is not subject to change)
- BC Hydro Electric Tariff Supplement No. 37 (TS37) covering the required NTL "overrun" Contribution
- BC Hydro Electric Tariff Supplement No. 74 (TS74) Customer Baseline Load Determination Guidelines.

BC Hydro Rate Schedule 1823 is a two-tier schedule, nominally with 90% of the Customer Baseline Load charged at economical Tier 1 energy rates, and the last 10%, plus all power above the Customer Baseline Load, charged 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. BC Hydro, under their Power Smart program for demand side load control, offer incentives to transmission customers to reduce energy consumption, and for new customers incentives are given for energy-efficient plant design. Further information can be found in the Cost of Electric Power, 2016 Report located in Appendix I7.

In Appendix I7, the Project cost of electric power delivered to site is shown on a per kilowatt hour basis. The calculated cost is below regular rates due to a large reduction or elimination of costly Tier 2 energy in accordance with an efficient plant design as accepted by BC Hydro's "Power Smart" program. Thus, HPGR energy savings have an impact far greater than just the energy savings in the grinding area. A separate report to BC Hydro has confirmed that the use of HPGRs for the Project qualifies for these incentives.

TS6 currently requires potentially large non-refundable 500 kV transmission and generation system reinforcement charges for projects with a Contract Demand of over 150 MVA. This would apply to the Project and the required capital contribution would apply to the entire load, not just the load that exceeds 150 MVA. The Contract Demand (peak load) for the Project is currently estimated to be well above 150 MVA, even considering that energy conservation measures will be implemented. Seabridge currently has an application before BC Hydro for an increase in the Contract Demand from 150 MVA to 200 MVA, with the expectation that the generation reinforcement charges will be set at zero and 500 kV system reinforcement charges will only apply to the additional 50 MVA of Contract Demand, not the entire load, reflecting the state of the current system. However, as this is uncertain, and a determination will not be made until after publication of this study, a combustion turbine has been allotted for in the estimates for peaking purposes, as this is far more economical than the application of the currently applicable reinforcement charges (TS6).

The cost of power for the Project, delivered to the 25 kV bus bars of the Treaty OPC, has been estimated as Cdn$0.062/kWh, including applicable taxes and energy cost savings due to BC Hydro's Power Smart program. The Project power cost includes the transmission line losses from the metering point at the Treaty Creek Switching Station, plus Substations No. 1 and No. 2 transformer losses and peaking power cost.

The KSM power cost calculation takes into account reduced rates due to BC Hydro Demand Side Management (DSM) and associated Power Smart initiatives for energy conservation measures designed into new plants (such as using HPGR grinding in lieu of SAG milling). Such measures, as may be certified by BC Hydro, serve to reduce the standard 10% of energy under the two-tier 1823 Rate Schedule that would fall under the costlier Tier 2 category. If HPGR grinding and similar energy conservation measures were not to be implemented, there would not only be greater energy consumption, but the cost of electric power for the entire project would increase.

Each year on April 1 (the start of their fiscal year), BC Hydro sets new rates that are applied in accordance with the tariffs, subject to BCUC approval. Details of the electric power cost calculation are included in Appendix I7. This report also includes the maximum increases that BC Hydro will apply for in the next several years. Rate increases in the past several years have been significant in order to finance required general system upgrades; however, the maximum rate increases for the next several years are quite modest.

BC Hydro currently has a Rate Design Application (RDA) before the BCUC. Seabridge has attended several BC Hydro workshops regarding the RDA and proposed changes for Transmission Service customers, and has reviewed what has been submitted to the BCUC to date. BC Hydro has proposed no changes to Rate Schedule 1823. This is discussed in the RDA (Section 7), which is available on the BCUC website. Seabridge understands that any changes to the transmission tariffs would not be detrimental to the Project and that existing applications will be grandfathered in any event. Any BC Hydro proposals relative to TS6 have been deferred until a future application in 2017 (RDA Module 2) and are not part of the 2015 RDA.

18.11.6 TREATY PLANT MAIN SUBSTATION NO. 1

The KSM 287 kV step-down Substation No. 1 will be located at the Treaty OPC and will be constructed and owned by Seabridge in accordance with BC Hydro policy, which is also the most economical solution. This substation is a critical installation for the Project. The substation equipment has been sized based on the latest project load list. Redundant transformer capacity was included in the design. The substation will be a GIS switchgear) design, utilizing 138 kV and 287 kV gas insulated circuit breakers and bus bars, allowing a compact design contained in a building adjacent to the Treaty Process Plant. The circuit breakers will use point-on-wave switching as required by BC Hydro. Connections to transformers will use high-voltage solid dielectric cables.

The substation will include:

- three transformers, each of the three winding type oil filled 75/100/125 MVA, ONAN/ONAF1/ONAF2 step down power transformers, with automatic on-line tap changers
- six 287 kV GIS circuit breakers
- seven 138 kV GIS circuit breakers
- one 287 kV switched reactor for compensation of the incoming 287 kV line, to limit Ferranti effect over voltages
- two 138 kV switched reactors at the OPC end of the 24 km long, 138 kV cable to compensate for cable capacitance, thus controlling bus voltage
- 25 kV grounding transformers and resistors.

The three transformers included in Substation No.1 will be installed in a concrete vault. They will provide redundancy, allowing one transformer to be out of service. Shipping restrictions to the site were taken into account when sizing the transformers. The 138 kV tertiary windings will be connected to the 24 km long tunnel cables feeding the Mitchell (open pit mine) area Substation No. 2. Space has been allocated for a future fourth power transformer.

Substation No. 1 will also include a line-up of 25 kV metalclad or GIS switchgear, as required for power distribution around the Treaty OPC. The secondary distribution voltage for the Treaty OPC will be 25 kV.

Substation No. 1 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.

18.11.7 138 kV Cable

Substation No. 1 will be interconnected with Substation No. 2 by three, 138 kV, single-core, 300 mm^2, cross-linked polyethylene (XLPE) solid dielectric power cables suspended from the back (roof) in one of the MTT that will run between the two plant sites. The 300 mm^2 (600 kcmils) conductor size quoted is the minimum physical size that vendors typically manufacture at 138 kV (due to the electric field gradient at the conductor). This is ~~a~~ 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. Adequate (significant) space must be allowed in one of the train tunnels for these cables.

18.11.8 Mitchell Substation No. 2

The 138 to 69 kV - 25 kV Substation No. 2 is also critical infrastructure for the Project. As an alternative to a standard 138 kV air-insulated outdoor substation, Substation No. 2 is planned to be a GIS installation. This is a very compact design, requiring only a fraction of the space of a conventional air insulated high voltage substation and allows for the total installation to be included in a reinforced concrete building that provides a high degree of protection against geo-hazards such as avalanches. It also eliminates hazardous high-voltage overhead lines in the vicinity of the Mitchell OPC and requires much less plant area. The substation includes:

- two 138 - 69 - 25 kV, 55/73/90 MVA ONAN/ONAF1/ONAF2, oil filled power transformers with automatic on-line tap changers (two, 3 phase units are provided for redundant capacity, with space provided for a third unit to cater to future load growth)
- six 138 kV GIS circuit breakers and associated bus work
- two switched 138 kV reactors to compensate 138 kV cable capacitance
- six 69 kV GIS circuit breakers connecting to site 69 kV power distribution system
- grounding transformers and resistors
- a line-up of 25 kV metalclad or GIS switchgear for site local power distribution.

18.11.9 SITE POWER DISTRIBUTION

Site power distribution in the mine area from Substation No. 2 will be by 25 kV cables and overhead pole lines locally and by 69 kV overhead pole lines to feed large loads at more distant facilities where modular substations will step the 69 kV down to the local distribution voltage. The relatively long distances and high initial and future pumping loads require 69 kV distribution to transmit the power and limit voltage drop.

18.11.10 MINE POWER

Power to the Mitchell open pit itself will be provided by local 25 kV overhead 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 in the electrical project budget. 7.2 kV to 600 volt portable substations are also included for pit dewatering. Similar installations are included in sustain capital or the Sulphurets and Kerr open pits.

18.11.11 CONSTRUCTION AND STANDBY POWER

Modular diesel generator sets will be provided to supply construction power for tunnel driving, camps, temporary water treatment plants, plant construction sites, and other initial construction-related facilities. The capital and operating costs of these facilities plus local distribution including step-down transformers and overhead pole lines have been included in Project indirect costs (not in the power supply budget). Fuel and operating costs for construction power are also accounted for in the construction indirect costs. The power distribution costs for supply and installation of cable and electric panel boards within the various tunnels are included in tunnelling costs.

Any additional costs for moving equipment and fuel to site during the early stages of the Project, either by helicopter or by a glacier road, are included elsewhere in the capital cost estimate and are not in the construction power budget.

The construction generating stations are modular, complete with switchgear, and designed for 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. It is to be noted that fuel storage and use for tunnel and surface diesel powered construction equipment is not included in this budget.

Several of the construction gensets will be retained after initial construction is complete 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 very long KSM construction period will make construction genset rental uneconomic.

18.11.12 ENERGY RECOVERY AND SELF GENERATION

The Project presents several opportunities for energy recovery from process flows, as well as power generation from mini-hydro projects, taking advantage of water flows that must otherwise be diverted around the mining operations. 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 value of the generated power would either be at the BC Hydro rate schedule 1823 Tier 2 energy (set at BC Hydro’s marginal cost of generation) valued at Cdn$0.0892/kWh as of 2016 or would be sold back to the utility under the Standing Offer Program for small projects, with the mine acting as an IPP generator. The value of electricity in this region under the standing offer program would be just over Cdn$0.10/kWh. In either case the value of generation would be considerably higher than the 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.

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. 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 Project.

The generation projects included in this study are summarized in Table 18.17.

Table 18.17 Mini Hydro and Energy Recovery Power Generation

Project Name	Type	Installed Capacity (kW)	Net Annual Generation (kWh)	Machines
WTP*	Energy Recovery	9,000	23,773,000	Pelton Turbines
Tailings	Energy Recovery	1,194	7,794,060	Pumps as Turbines
Mitchell Diversion*	Mini Hydro	7,500	17,638,000	Pelton Turbine
McTagg Stage 2 Diversion (Years 10 to 15)	Mini Hydro	8,000	32,981,000	Pelton Turbine
McTagg Stage 3 Diversion* and **(Year 15 onwards)	Mini Hydro	12,000	45,242,000	Pelton Turbine

Notes: *Operation continues after mine closure.
**McTagg Stage 3 replaces Stage 2.

Further information is available concerning energy recovery and mini-hydro projects in Appendices I3, I4, I5, and I6.

18.12 Treaty OPC and Mine Site Secondary Electrical Power Distribution and Utilization

18.12.1 Mine and Plant Power Consumption

The total mine and process plant annual energy consumption is estimated to be 1,333 GWh based on the Tetra Tech (HPGR) load list for a 130,000 t/d operation valid on average for Years 1 to 5, based on the currently proposed blending or ore. This equates to an average annual load of 152 MW. With a load factor (LF) in the range of 0.85 as typical for a project such as KSM, the peak load (30-minute demand) is estimated as 179 MW. The plant running (normal every day) load is estimated to be 167 MW, again based on norms for this type of mine and milling operation.

The required utility supply will be reduced 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 almost zero. To prevent the Project demand from exceeding the 150 MW trigger point for generation reinforcement, the proposed peaking combustion turbine, located in at the Treaty OPC, will be operated. This has been included in the Project power capital and operating costs.

18.12.2 MAIN SUBSTATIONS

There will be two main substations. Main Substation No. 1, and Substation No.2, described in Sections 18.11.6 and 18.11.8, respectively.

18.12.3 POWER DISTRIBUTION – TREATY PLANT MAIN SUBSTATION NO. 1

The Treaty Substation No. 1 will contain three main 287-138-25 kV, 75/100/125 MVA transformers with automatic on-load tap changers. 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 breakers. There will be a total of five main and tie breakers and 35 feeder circuit breakers connected to distribution feeders. These feeders will distribute power throughout the Treaty OPC. The 25 kV system will be high-resistance grounded.

BALL MILLS

The Treaty process plant ball mills are major power consumers. Each of the four ball mills (rated 14,000 kW each) will be fed via dedicated 25 kV feeders and step-down transformers to 13.8 kV. The mills will each be equipped with two, 10,000 hp, low-speed "Quadratorque" fixed speed synchronous motors, directly driving mill dual pinions via air clutches, as has been used in the industry for many years.

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)
- four HPGR units, each complete with 2 by 4,000 hp dual drives with VFDs.

STEP-DOWN TO 600 V

Motors and other loads below 250 hp will be fed from one of several 600 V systems. Generally, these systems will consist of liquid insulated 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 OPC loads will be served by 25 kV overhead lines. Examples of remote loads include:

- fresh water pumping
- TMF return water and seepage pumps
- ancillary buildings.

18.12.4 MITCHELL SUBSTATION NO. 2

Mitchell Substation No. 2 will be fed by 138 kV cables running through the MTT from Treaty Substation No. 1. Substation No. 2 will have two, 138-69-25 kV, 55/73/90 MVA ONAN/ONAF1/ONAF2, oil-filled power transformers with automatic on-line tap changers, with space for a third unit. Thee transformers will feed a lineup of 25 kV metalclad or GIS switchgear. The 25 kV switchgear will be divided into two sections with main incoming breakers and a tie breaker. A total of 14 feeder breakers will be provided to feed the Mine Site and the MTT train system. The 25 kV system will be high-resistance grounded. Section 18.11 includes a detailed description of Main Substation No. 2.

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.

The mining electric shovels and drills will be served at 7.2 kV via portable 25 to 7.2 kV step-down substations fed from the perimeter pit pole line. The estimates include appropriate lengths of trailing cable and couplers. 7.2 kV to 600 V portable step-down substations and trailing cables are also included for pit dewatering.

A 69 kV GIS Circuit breaker and cable will feed an overhead pole line supplying the truck shop, WTP, explosives facility, and also connecting to the mini hydro and energy recovery power plants. There will be local substations stepping down from 69 kV to the local distribution voltage.

REMOTE LOADS

Remote loads will be served by the 69 kV sub-transmission overhead lines. Examples of remote loads include:

- truck shop
- permanent camp
- WSD pumping.
- HDS WTP and Selenium WTP
- explosives magazine.

18.12.5 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
- CCTV.

18.13 Permanent and Construction Access Roads

18.13.1 Background

Seabridge retained McElhanney to complete a study of permanent and construction road access options to the KSM zones and mine facilities. Tetra Tech 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 2012 in assessing the various options.

Current proposed permanent access roads include the existing 59 km long resource access route from Highway 37 to the former Eskay Creek Mine and camp facilities. The proposed 35 km long CCAR will commence near the southern limit of this existing road, and extend south then west to the proposed Mine Site.

The Treaty Creek Valley road network provides access to the Treaty OPC, the TMF, and the MTT Saddle Area. It will include a 30 km two-lane access route from Highway 37 to the Treaty OPC, TMF, and east portal of the MTT, and include portions of the TCAR and NTAR.

The TCAR will be constructed as a two-lane road from Highway 37 to approximately km 17.9. Beyond km 17.9, the TCAR will be constructed as a single-lane road, extending further up the Treaty Creek Valley to provide access to the MTT saddle area. The total road length from Highway 37 to the MTT saddle area will be approximately 33 km. Beyond there, consideration is also given to a potential MTT tunnel adit access road, extending approximately 3 km further west.

Roads known as the lower and upper NTAR will be built to the north, up the North Treaty/Teigen Creek Valley, to access the Treaty OPC. They will intersect with the TCAR at km 16.9 and km 17.9, respectively, and will be constructed at different times during the life of the mine. These will be two-lane roads and each will extend approximately 12 km to the north, accessing the Treaty OPC and the TMF.

Select sections of each road will parallel the drainage cut-off ditch in the North Treaty/South Teigen Creek Valley. Another single-lane (4 m wide) road, approximately 4 km in length, will provide additional access for construction and maintenance to the south end of the above mentioned drainage cut-off ditch.

McElhanney's full report relating to current access road options is contained in Appendix J.

18.13.2 OBJECTIVES AND METHODOLOGY

Utilizing LiDAR survey data acquired in summer 2008 and fall 2011, and the resulting digital elevation models developed, the preferred preliminary access routes identified by Seabridge and McElhanney were defined on the base mapping.

The preliminary road alignments were subsequently located in the field using GPS, and marked with survey flagging. 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 construction cost estimates were prepared. Engineering assessments were conducted in conjunction with available geotechnical and environmental studies of all proposed routes. The field reconnaissance and bridge site surveys confirmed the accuracy of the LiDAR data.

From 2009 through 2012, consultants BGC and Rescan (now ERM) conducted further geotechnical and environmental assessments, respectively, on the proposed, and altered routes. Where appropriate, McElhanney' QP (Mr. Bob Parolin) accompanied these consultants in the field to make joint determinations with respect to the most appropriate locations for specific sections of road. McElhanney worked with these consultants to optimize the road locations and designs.

All proposed final road locations were marked with survey flagging. Flagging was marked with survey crew and date information (black felt marker), and locations identified by real-time kinematic (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 soils, vegetation, potential borrow/waste areas, drainage culvert requirements, and other relevant information.

Stream crossing surveys were completed on "smaller" tributaries. Generally, this includes any stream with an estimated 100-year peak flow of 6.0 m^3/s or greater. Details for all such structures were completed to satisfy the requirements of the Ministry of Forests, Lands and Natural Resource Operations (MFLNRO) for the Special Use Permit (SUP) applications.

Preliminary stream crossing structure designs have been completed for all sites requiring bridges or major culverts.

Road designs have been completed taking into account other mine support infrastructure, and with consideration for construction and long term haul requirements.

18.13.3 ROUTE DESCRIPTIONS

The current proposed access roads include the:

- Eskay Creek Mine Route (existing – upgrade as required)
- CCAR
- TCAR
- lower NTAR (early mine life)
- upper NTAR – Phases 1 and 2 – (early and mid-mine life)
- Cut-off Ditch Access Road
- MTT Adit Access Road
- TMF Service Roads
- Frank Mackie Winter Access Road (design by Tetra Tech).

Currently proposed primary road locations are shown in Figure 18.1 and Figure 18.2.

The current updated route descriptions, including relocations of the road alignments, are provided in the following sections.

ESKAY CREEK MINE ACCESS ROUTE

Seabridge plans to use portions of the existing Eskay Creek Mine Access Road, linking Highway 37 to the proposed CCAR.

This road was constructed in the early 1990s to provide access to Barrick Gold's Eskay Creek Mine. The road commences at Highway 37, south of the Bob Quinn Forest Service Road, and follows the Iskut River Valley west for approximately 37 km to the crossing of Volcano Creek. The road was originally designed as a single-lane, 5 m wide gravel road, with a nominal design speed of 60 km/h. Substantial portions are built to a nominal 8 m wide (double lane standard), providing ample passing opportunities.

More recently, the road was used to service construction of the Forrest Kerr, Volcano Creek, and McLymont Creek hydro-electric developments by AltaGas. Construction of these developments is now complete.

South of the intersection at km 37.7, the 22 km-long Eskay Creek Spur Road was used solely for access to the Eskay Creek Mine and camp. 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. The road is passable, but upgrades to three of the bridges, and replacement of one of the bridges will be required.

From Highway 37 to approximately km 4.0 is the Bob Quinn Forest Service Road, and is owned and maintained by the MFLNRO. A Road Use Permit will be required for operations over this section of road.

Currently, AltaGas holds the SUP from the MFLNRO from km 4.0 to km 43.3. Barrick Gold holds the SUP over the remainder to the Eskay Mine Road (km 43.3 to km 59). Use of this road will be subject to shared access/maintenance agreements.

An overview evaluation of the road condition and its suitability for Seabridge's requirements was conducted; findings are summarized in McElhanney (2013).

Coulter Creek Access Road

The CCAR will be constructed as a single lane (6 m surface), radio-controlled road with pullouts for passing opportunities.

Heading southwest from near the end of the existing Eskay Creek Mine Access road (approximately 59 km off of Highway 37), this road will follow an existing mine access road for approximately three or more kilometres towards Tom MacKay Lake. It will then descend out of the alpine meadows, along the height of land between Coulter Creek and the Unuk River.

In this area the road traverses some difficult terrain. Much work was done in optimizing the design and realigning the road in an effort to avoid areas mapped by BGC (geotechnical consultants) as being potential geohazards, including some Class 4 Hazard terrain. Realignment efforts also sought to reduce earthwork volumes and minimize sustained steep grades. Maximum design grades along the road corridor are now 12%.

There are three bridges and two major culvert structures proposed between the start of the road and the Unuk River crossing at km 20.9. With the exception of the Coulter Creek crossing, ERM has determined that all other streams are non-fish bearing.

The Coulter Creek crossing at km 17.8 is located near the foot of a historic alluvial fan. Additional riprap protection and drainage relief has been noted on the design through this area, though risks of debris flood/flow have been assessed as low.

ERM identified and mapped a number of blue-listed ecosystems along the road as it descends along Coulter Creek to lower elevations. The road and stream crossings were located to avoid these sensitive ecosystems, and to reduce the impact on fisheries sensitive zones. The proposed three-span bridge crossing of the Unuk River 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*. The current clearance from Q100 flow design levels to the underside of the main bridge span is estimated at 2.1 m.

Beyond the Unuk River, the route traverses a short section of low-lying wet and swampy areas and includes a bridge crossing at approximately km 21.5. The road starts to climb at approximately km 21.7, then climbs steeply through a series of switchbacks from km 23 to km 25, into the Sulphurets Valley and canyon. 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 approximately 1 km, before following the steep north side of the Upper Sulphurets Creek Valley. Along this section, it traverses 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, 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.

Beyond approximately km 31.5, the Sulphurets Creek Valley widens considerably and the road location continues 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 aligned to accommodate the location of the HDS WTP on the bench lands. The alignment extends approximately 800 m to the southeast of the HDS WTP, avoiding potentially unstable ground to the south. It turns northeast, and climbs to km 35, where McElhanney's road design (and SUP boundary) 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 TCAR 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.

Meetings were held between McElhanney, Seabridge, and the provincial Ministry of Transportation and Infrastructure (MOTI) to discuss and establish a set of design criteria for the proposed intersection at the Highway 37/TCAR location.

McElhanney completed an assessment of the proposed intersection site based on current and projected traffic volumes, and relevant design criteria, and concluded that technically no significant intersection improvements were warranted. However, subsequent, and on-going dealings and negotiations with MOTI concluded with their refusal to grant necessary access permits without making some suggested intersection improvements to increase safety; considering the traffic volumes anticipated, the road geometry at site, and the potential for winter/icing conditions.

Additional detailed design drawings will be required to meet the Ministry's final requirements, prior to construction.

Initially the TCAR 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 was a previous bridge installation at this site. Access was gained to MFLNRO files defining the historical multi-span bridge installation at this site (removed in the 1990s). Noted previous issues with geotechnical stability of the west bank have been considered in development of the new bridge design, along with consideration for avoiding the original bridge pier piling groups.

This is a major river and will need to meet the requirements of the *Navigable Waters Protection Act*. Current clearance from Q100 flow design levels to the underside of the main span is estimated at 3.0 m.

The TCAR essentially follows an existing forestry access road for approximately 4 km. Significant upgrades will be required. Between approximately km 3.7 and km 5.2 the road crosses an alluvial fan. MFLNRO expressed concerns regarding the earlier road location and design, and potential effects of future flow/flood events. Subsequent correspondence through 2013 resulted in realignment of the road, and other design changes, to address those concerns.

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 approximately km 8.6, the road will start to climb. From here it will traverse moderately sloping side hill terrain to approximately km 17. A number of streams are crossed. Stream crossing structures have been designed to appropriately address fish passage and road geometry requirements, as applicable. A total of 10 bridges (at 9 sites) are designed between the Bell-Irving major crossing and the future intersection at km 17.9. Bridge lengths vary from 11 m to 24 m. A general arrangement design has been completed for one major culvert crossing at km 15.9.

The North Treaty Creek crossing at approximately 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. The primary consideration will be the application of active avalanche control measures along this route, similar to those described for the CCAR.

At approximately km 16.9, there will be an intersection with the lower NTAR. 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 OPC and TMF. This road is described later in this section.

The TCAR will continue as a double-lane road further west to a future intersection at approximately km 17.9. Heading west from there, the TCAR will transition into a single-lane road leading to the MTT saddle area. This route is described later in this section.

At the proposed km 17.9 intersection another double-lane road will be built in future. It will be known as the upper NTAR, and will be built at approximately the mid-mine life, once it becomes necessary to construct the southeast tailing dam. The South Cell and Southeast Tailings Dam will bury much of the lower NTAR, that 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 TCAR will transition into a single-lane road leading to the MTT Saddle Area. This single lane road will provide construction period access, and longer term maintenance access to the MTT. 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 CCAR.

Only one bridge is required along this section, at approximately km 20.5. Five major culvert crossings are required. Currently streams identified as being fish bearing have clear-span bridge structures. None are crossed with culvert or open-bottom arch type structures.

NORTH TREATY CREEK ACCESS ROADS

There are currently three permanent access road alignments proposed within the North Treaty/Teigen Creek valley. They shall be referred to as the lower NTAR, the upper NTAR, and the Cut-off Ditch Access Road. The upper NTAR is further split into Phase 1 (2.2 km) and Phase 2 (5.8 km).

The upper NTAR will leave the TCAR at approximately km 17.9. It will traverse approximately 12 km north from the TCAR to the Treaty OPC and TMF. Initially 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 southeast tailing dam elevation (nominally 1,070 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.

The road location and terrain dictates that significant portions of the upper NTAR will need to be built using full bench/end haul to waste construction. Construction of the south portions of this access road could be difficult and time consuming, and might delay critical early access for construction of the MTT at the Treaty OPC.

Earlier access can be obtained by constructing the lower NTAR. This will leave the TCAR at approximately km 16.9 and follow the lower valley. This route crosses a number of steep gullies and the terrain has some steep sections, but is generally flatter than the upper NTAR location.

The lower NTAR will be quicker to build. It will result in a slightly shorter haul distance between Highway 37 and the Treaty OPC and TMF, and with generally flatter grades. This road would be used for approximately the first half of the mine life, until such time as it is necessary to construct the southeast tailing dam. The lower NTAR would match to the upper NTAR approximately 8.1 km north of the TCAR main turnoff. The lower NTAR 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 southeast tailing dam.

Early in the mine construction it would still be necessary to build a section of the upper NTAR 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.

North Treaty/Teigen TMF Service Roads

During 2012 significant work was done to complete preliminary road designs for required TMF Service Roads. These provide access to the east side of the North Treaty and Teigen Creek valleys. A minor road to the camp water well was also designed. In total approximately 28.4 km of road were evaluated, including the following:

- South Teigen Road 11 (5.5 km)
- South Teigen Road 11a (3.15 km)
- South Teigen Road 12 (9.1 km)
- South Teigen Road 12a (0.3 km)
- Upper Dam Road 12b (1.2 km)
- South Teigen Road 15 (3.7 km)
- South Teigen Road 15a (2.7 km)
- Water Well Access Road (2.7 km).

All of these roads will be situated within the mine site boundaries. The majority provide access to the east side of the TMF and ancillary facilities. All are designed as single lane roads (maximum 6 m width), with pullouts, and most have low design speeds South Teigen Road 12 was designed to parallel the proposed uphill drainage channel requirements, to direct natural drainage off of the east slopes and away from the TMF.

Some input was provided with respect to geotechnical and environmental concerns for select segments of these roads. Some, but not all of the roads were field-truthed during the 2012 season. Additional future field work will be required to complete those assessments.

18.13.4 Road Design Requirements

The 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 Mine Road and CCAR will be maintained for the life of the mine to support the mine development, transport of oversize loads, and to provide alternate emergency access. However, these will only be used seasonally, and not used during winter months.

The CCAR will be a single-lane (6 m surface) radio-controlled road with turnouts and widenings to allow the largest vehicles and loads access to the mine site. The CCAR would have some sections with sustained maximum grades of 12%. Design speeds vary greatly, in large part controlled by the terrain.

The proposed TCAR to km 17.9, and the connecting upper and lower NTARs, will be required for permanent access to the Treaty OPC and TMF, and to the mine site via the

MTT tunnels. These will be two-lane roads (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 10% sustained grades (11% short pitch), and minimum 100+ m radius horizontal curves are utilized for the higher-traffic volumes anticipated on this route. Appropriate vertical profile crest and sag curve “K” values are applied. 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.

All bridges will be designed to BC Forest Service L100 loading (90,680 kg gross vehicle weight [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 TCAR, will be single-lane.

Major culverts have been designed to pass the estimated 100-year flood level (Q100) with no headwater.

18.13.5 DESIGN UPDATES (2012-2016)

As described elsewhere in this section, additional field work was conducted in 2012 along the proposed access routes by McElhanney, BGC and ERM. New information was incorporated to optimize the road and structure designs. Horizontal and vertical alignments were modified to best meet the environmental, geotechnical and archaeological concerns and requirements. During 2012 work was completed to locate potential borrow and waste sites, at appropriate locations to accommodate road grade construction requirements along the access corridors. Provision was also made to identify areas of potential gravel sources for road construction and surfacing materials.

Log landing locations were identified for decking of timber felled during right-of-way clearing operations. Log landings were located, and spaced, as appropriate for logging/skidding operations. Timber maturities/volumes etc. were considered in establishing proposed landing locations.

The proposed right-of-way (clearing) boundaries now defined on the design drawings are minimum 30 meters wide, expanded to include proposed borrow and waste areas, and log landings as described above. The approved SUP boundary limits extend a minimum of 37.5m either side of proposed road design centerline (total 75 m width), widened as required to incorporate additional areas as otherwise defined. The intention is that this will provide some flexibility in adjusting the design or construction methodology as may be required due to actual field conditions encountered, without requiring multiple amendment to the SUP during the construction period.

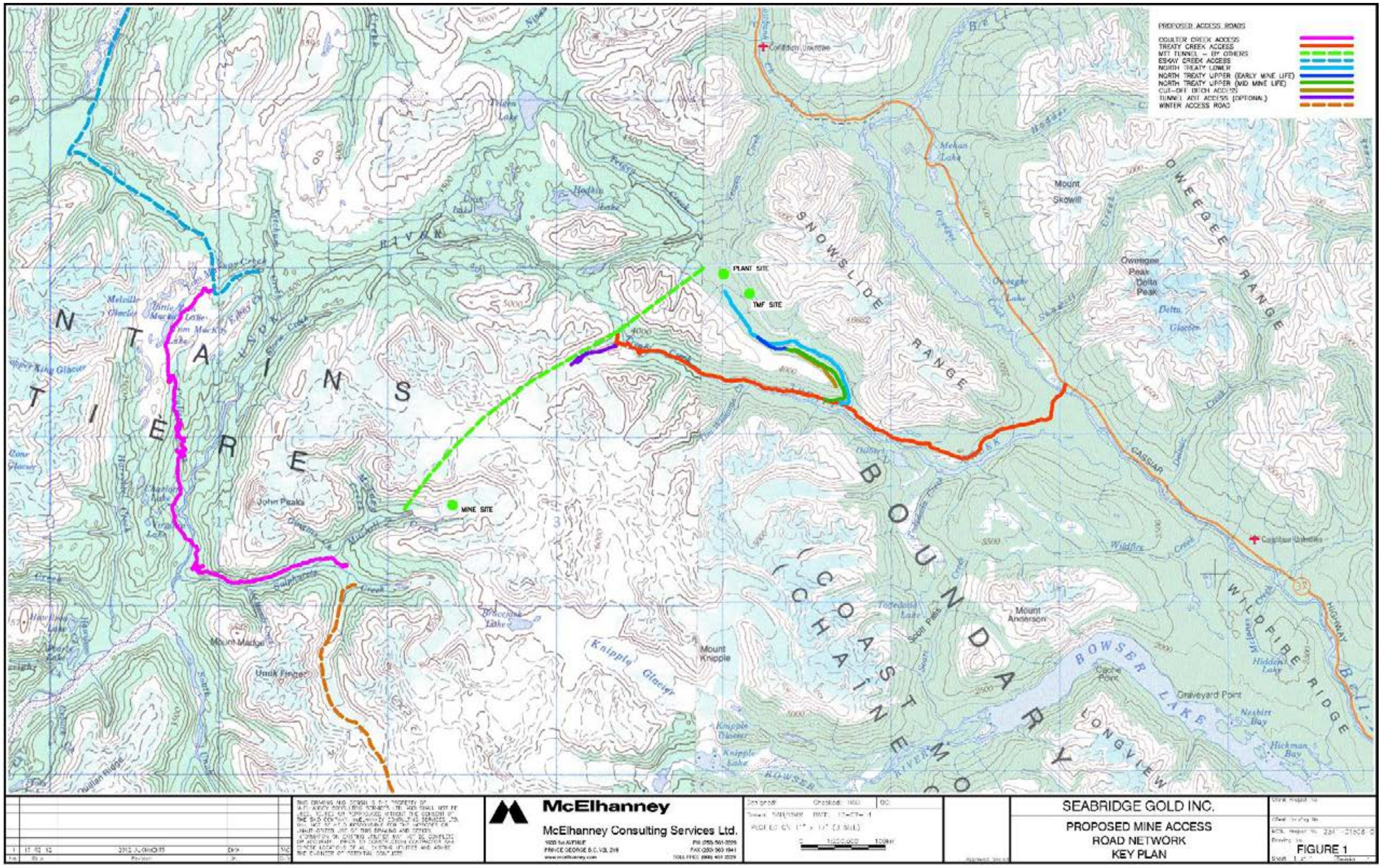
The potential for ML/ARD has been assessed for all access road right-of-ways. Additional assessments were conducted in late 2014 for km 0 to 7 of the CCAR which had been flagged by Government agencies as an area of special concern.

The SUPs for road construction associated with the Project were granted by the Provincial MFLNRO office on September 27, 2014. There is a requirement to provide a security, payable to MFLNRO, prior to commencing construction.

Access road surveys, designs and drawings were prepared in conformance with standards provided in the then most current version of the BC government Forest Service Engineering Manual (November 29, 2012). Detailed engineering of specific slope stability measures will be subject to review by the geo-technical engineer(s), immediately in advance of, and during construction activities.

Bridge and major culvert structure site plan surveys, designs and general arrangement drawings have been prepared in accordance with MFLNRO requirements and current industry standards. General arrangement design drawings have been signed and sealed independently by a professional engineer registered to practice in BC. Detailed structure design details will need to be completed in advance of construction.

Figure 18.26 Proposed Access Roads Network



18.14 PROPOSED WINTER ACCESS ROAD

A winter access road is proposed from a laydown area near the former Granduc Mine to the Project site. The approximate alignment for the proposed winter access road is depicted in Figure 18.27. An evaluation of the route was completed by Tetra Tech (Appendix J.).

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.

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 to 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

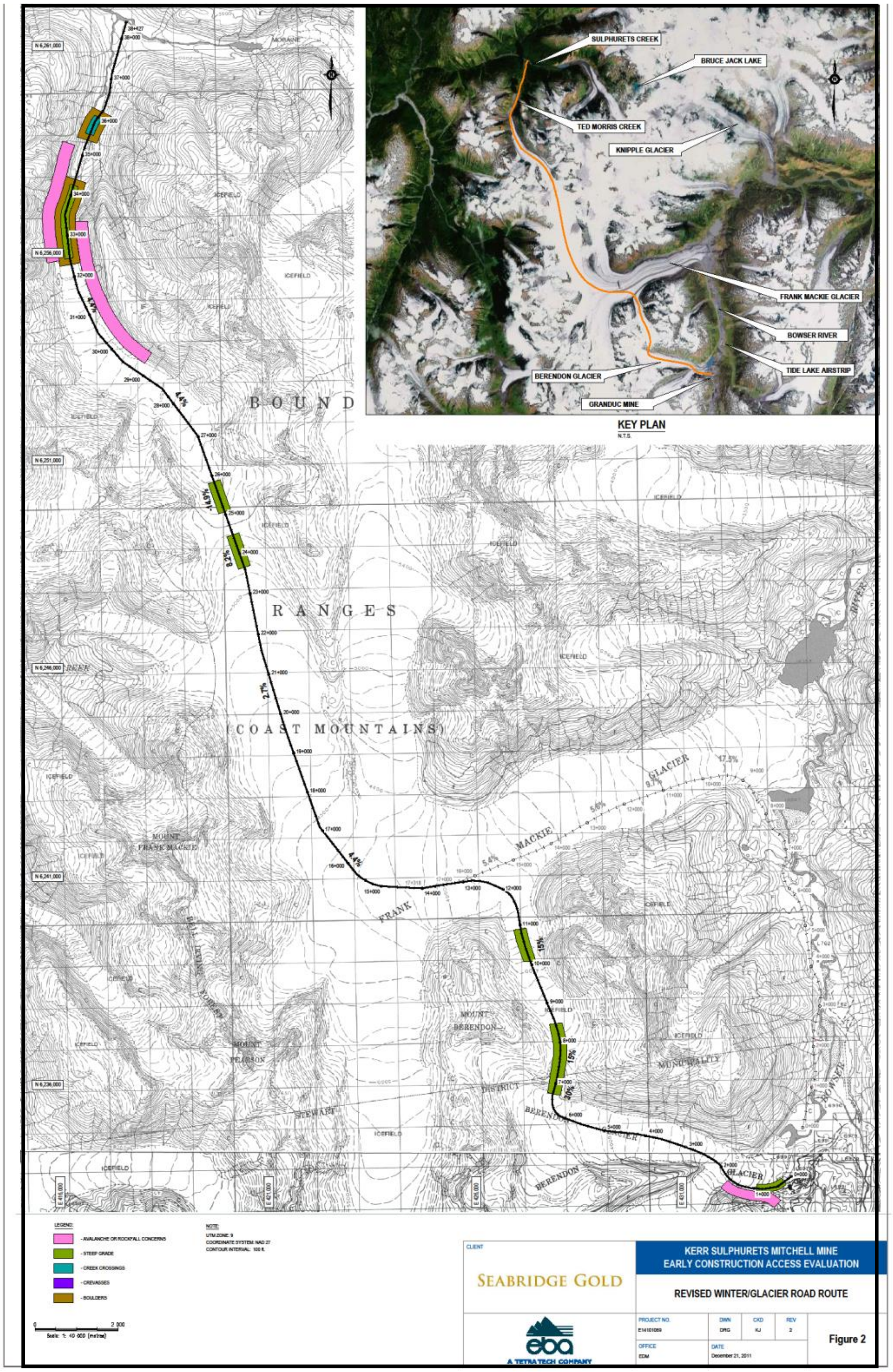


Figure 18.28 Looking South up the Frank Mackie Glacier*

Note: *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*

Note: *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.

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 road 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.

18.15 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 G3.

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 TCAR.
- Access to the KSM mine site involves extending the Eskay Creek Mine road to the south for a distance of about 35 km as the CCAR.
- 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 TCAR 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.

All bulk freight for the Project will be shipped to Stewart, BC by barge. There will be an initial marshalling/staging area set up at Stewart within the area provided by the port vendor. Prior to completion and opening of CCAR and TCAR, bulk freight will be transported by truck from Stewart marshalling/staging area to Granduc Staging area. Upon completion of the temporary winter access road construction, the bulk freight will be transported by tracked transporters to Mine Site.

Upon completion of the CCAR, TCAR and the marshalling/staging area at the Highway 37/TCAR intersection, bulk freight will be transported by trucks from Stewart marshalling/staging area to Highway 37 marshalling/staging area, prior to shipping material and equipment to the point of installation by the Project Team.

The KSM site is currently accessible by helicopter only. Helicopter support will be used to transport equipment, supplies, and personnel prior to completion of the access pioneering roads, and to support winter construction work ongoing as necessary on the Mine Site since CCAR is a seasonal road (closed winters) and MTT would not be operational until the end of the construction duration.

A proposed Winter Access Road will be constructed that leads to the KSM site, as detailed in Section 18.14. The Winter Access Road will be used to mobilize water treatment supplies and mobile equipment, as well as supplies for construction of access roads, construction of WSF and water diversions during the first season. The Winter Access Road will be used until the pioneering road of CCAR has been completed. It will also provide access for the construction of portions of the CCAR, near its east end and to the Mine Site.

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. Two full service ports exist at Stewart, each with roll-on/roll-off freight handling capacity and are either presently, or would by the time operations begin, be capable of concentrate storage and handling to ship loading.

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. Concentrates from other northern BC mines are currently shipped from this port. In addition, there is interest from other projects in the region for concentrate handling services at the terminal.

For the purposes of this study, Tetra Tech calculated that the copper concentrates will be shipped in bulk, and that the annual output for the initial 10 years will be approximately 350,000 t copper concentrate (dry tonne).

Molybdenum concentrate 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 1,155 t molybdenum (LOM).

Mr. Neil Seldon of NSA was relied on for matters relating to the smelting terms, refining terms, saleability, and sales terms for copper concentrate and molybdenum concentrate. See Section 19.0 for more detail on the marketing study related to concentrates.

18.16 PRELIMINARY CONSTRUCTION EXECUTION PLAN

The Construction Execution Plan describes how the Project could be constructed. It is a plan in the preliminary planning stage and briefly defines the construction elements required to successfully execute construction management for the Project.

18.16.1 INTRODUCTION

The Project is a complex project, requiring six years to construct, at a capital investment value of US$5 billion as described in Section 21.0. The construction scope for the Project is intended to meet the following key project-specific objectives:

- deliver an optimized, safe, and environmentally compliant project in accordance with the project systems and procedures
- perform project construction activities safely, striving for zero recordable accidents
- the Project is carried out in accordance with the Impact Benefit Agreements that are in place with the First and Treaty Nations
- ensure that regulations, license agreements, applicable specifications, and standards are met.
- complete Project construction within the agreed Project schedule, not exceeding the budget, and delivering the full scope as described in the Project authorization

18.16.2 EARLY WORKS PLAN

An Early Works Plan will be developed to ensure an efficient project start-up that is safe, controlled, and follows the Project objectives and guidelines. Certain key infrastructure (e.g., construction camps) and support services (e.g., catering) must be in place early in construction and functioning efficiently for a successful construction program.

The following project planning and field construction focus areas must be addressed in the Early Works Plan:

PLANNING

- permit review and renewal plan
- project procedures
- staffing, recruiting & labor relations plan, including commitments in accordance with the Impact Benefit Agreements that are in place with the First and Treaty Nations
- contracting strategy and plan, including commitments in accordance with the Impact Benefit Agreements that are in place with the First and Treaty Nations – vetted and approved by the Owner
- project access plan – Winter Access Road, pioneer roads, bridges, followed by completed permanent roads
- health, safety and security (HS&S) management plan and manual
- site and camp rules and regulations plan
- environmental and cultural sensitivity awareness training plan
- health and hygiene program – washrooms and lunchrooms supplied by the Owner ensuring a high standard throughout the construction program
- site safety and security orientation program
- geohazards and avalanche monitoring response plan
- logistics supply and materials management plan for early material requirements, including helicopter support
- employee transportation plan for early construction program; air and ground planning required
- environmental management plan to manage sediment control, waste, spills, fueling, etc. (includes condensed manual for front line supervision use)
- wildlife management plan for early construction activities
- community relations plan
- quality management system

- safety and emergency response plans including content related to medical facilities and medical attention, emergency medevac, etc. (includes condensed manual for front line supervision use)
- Final Level IV resource loaded construction schedule
- Final Project Execution Plan

FIELD CONSTRUCTION

- establish explosive supply storage and controls
- identification and proving project borrow pits
- sourcing road materials and aggregates; setting up crushing and screening facilities
- establish aggregate plant, aggregate wash plant, batch plant installation and supply of cement and aggregates
- install asphalt plant and supply of asphalt
- develop fuel supply and storage locations on site immediately upon achievement of road access (temporary double lined portable fuel storage tanks/fuel bladders within beamed and lined containment areas)
- build construction camps:
  - temporary accommodations only until construction camps established – exploration camp type
  - prefabricated construction accommodations to be installed as soon as the road access reach camp locations and site preparation is completed
- establish temporary construction power – standalone power supply systems (gensets) in containers with fuel systems. power distribution to begin on the ground then relocate to power poles when available
- build TWTP ponds and muck pads, and install TWTP's where tunneling on the Project is on the critical path (e.g., MTT)
- build pioneering roads along CCAR and TCAR alignments to establish road access to the Mine Site and Treaty OPC
- establish temporary construction facilities and services to support construction efforts while permanent infrastructure buildings and services are being built to support the site and staff.

18.16.3 PROJECT SCOPE

The Project scope outlined in this section, summarizes the main project items constructed as permanent facilities or activities required to support permanent constructions within and surrounding the Mine Site, Mitchell OPC, MTT and PTMA:

- Mine Site:
  - Frank Mackie Glacier Road (Winter Access Road)
  - upgrades to the existing Eskay Creek Mine Access Road
  - CCAR (35 km, permanent access road)
  - Mitchell OPC (primary crushing at a peak of 10,000 t/h)
  - WSF (crest built to elevation 716 masl) and ancillary facilities
  - HDS WTP with five large clarifiers and sludge storage to initially process up to 5.4 m^3/s (future expansion will add 2 clarifiers and ability to process up to 7.5 m^3/s)
  - six TWTPs and associated muck piles (where applicable) and treatment ponds
  - surface water diversion tunnels (MDT, 6.5 km; MTDT, 2.1 km; and MVDT, 3.5 km)
  - power distribution comprising construction gensets, overhead transmission lines, power distribution, and substation
  - logging, site clearing and grubbing (overburden, soils stockpiles & large woody debris for future reclamation purposes), rough/finish grading, structural excavations and fills, foundations, steel erection, architectural, mechanical, electrical and instrumentation works
  - Mine Site ancillary buildings and infrastructure such as camps (permanent and temporary), fuel storage yard, sludge storage building, diversion ditches and bypass pipelines, material handling, temporary truck assembly yard, energy recovery plants
- MTT:
  - two, 22.7 km long tunnels plus ancillary excavations
  - train system for ore transport at a peak of 10,000 t/h
  - two, 15,000 t ore bins and associated transfer conveyor from the Mitchell OPC
  - power distribution infrastructure
  - train maintenance building
  - train loading/unloading facilities
- PTMA and Saddle Area:
  - TCAR (33 km, permanent access road)
  - TMF (North Dam built to 930 masl, and Splitter and Saddle dams built to 935 masl, with a fully-lined and drained basin for placement of CIL tailing)
  - COS and transfer conveyors
  - Process Plant built for 130,000 t/d average throughput, including concentrate storage and load out
  - two TWTPs and associated muck piles and treatment ponds

- power distribution comprising construction gensets, overhead transmission lines, power distribution, and substations
- logging; site clearing and grubbing (overburden, soils stockpiles & large woody debris for future reclamation purposes); rough/finish grading; structural excavations and fills; foundations; steel erection; architectural, mechanical, electrical, and instrumentation works
- PTMA ancillary buildings and infrastructure such as camps (permanent and temporary), cold storage, fuel storage yard, diversion ditches and pipelines, and material handling

- Infrastructure (major, both on- and off-site):
  - concentrate storage and loading at the Port of Stewart
  - switching station at TCAR turn off from Highway 37(by BC Hydro)
  - 287 kV overhead power line from switching station to PTMA
  - fibre optics along power line right-of-way and tie-ins
  - Highway 37 marshalling yard and Highway 37 turnoff, including site security infrastructure.

The MTT tunnelling program and water treatment facilities will start at the Mine Site, Saddle Area, and Treaty OPC locations. The MTT tunneling program is on the critical path for Project construction and will require helicopter support for the first year of construction while pioneering roads are developed. Additional information on scope inclusions can be referenced in the Basis of Estimate Report (Appendix L1) and Basis of Schedule Report (Appendix G1).

18.16.4 PROJECT SCHEDULE

The 2016 PFS construction schedule was compiled in accordance with the AACE® International (AACE®) recommended scheduling guidelines (level of detail is at Level 2), with a Class 4 project definition. The Project construction schedule is estimated to be six years and has been designed to accommodate major seasonal and environmental constraints.

Critical path consists of pioneering roads along the alignments of the principal access arteries to the Project areas (CCAR and TCAR), MTT, and the train system that will connect the Mine Site with the PTMA. Prior to completion of the KSM site access pioneering roads, helicopter support will be utilized to support early construction activities at multiple road construction headings. The strategy is to establish site access pioneer roads as early as possible to reduce heli-support costs exclusively for Mitchell Valley construction activity during winter months. Upon completion of the access roads to full width, major equipment and materials can be transported to site via ground freight.

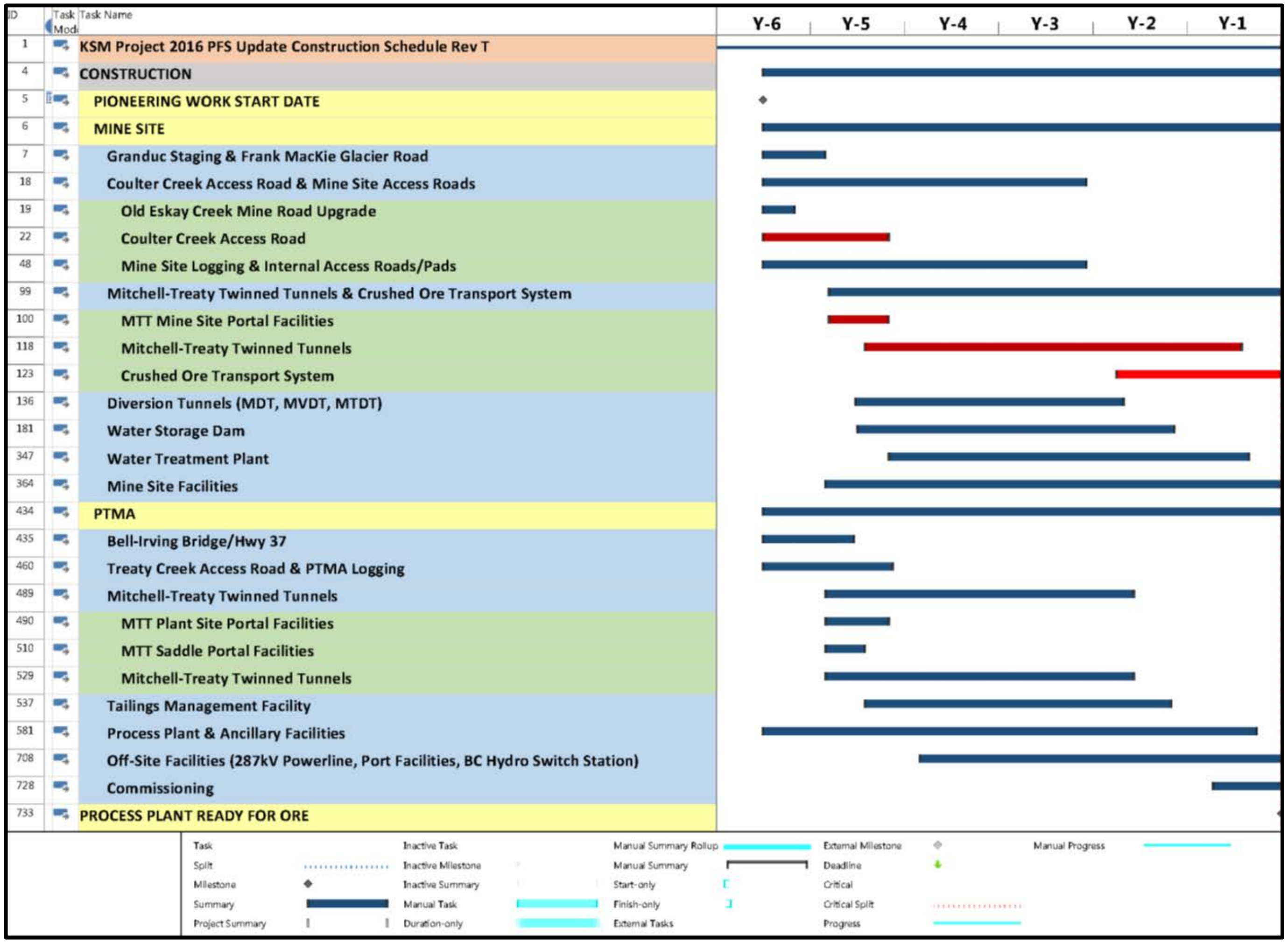
Major site infrastructure such as the WSF and the TMF could potentially be on the critical path should the MTT tunneling duration be shortened. Additional assumptions and details are listed in Appendix G1 for critical or potentially critical tasks such as pioneer roads and the MTT, as well as the WSD and TMF.

A preliminary Project schedule has been developed with a start date for the construction program assumed for mid-Year -6. Contractors to begin construction on the CCAR and TCAR construction would assume to mobilize for mid-Year -6.

Mine Site pioneering begins with the development of the site access roads to the major infrastructure pads such as HDS WTP area, WSF, Mitchell OPC, MTT portals, CCAR, batch plant, TWTPs, accommodation complexes and powder storage, initially from where the Frank Mackie Glacier Road ends. Early works material and equipment will mobilize on this Winter Access Road and the major equipment, general construction materials, and heavy earth moving equipment will mobilize via the CCAR. The Treaty OPC will utilize the TCAR to transport all material and equipment for PTMA construction.

The Project duration is estimated at 66 months. The high-level schedule is shown in Figure 18.30. The complete schedule is available in Appendix G1.

Figure 18.30 Construction Schedule Summary (Level 1)



18.16.5 Engineering and Procurement

Engineering and procurement activity will be managed by teams of professionals who will report up through the EPCM contractor's directorate. The Engineering Team will provide the required drawings, specifications, and documents to the Procurement Team in order to purchase all equipment and materials for the Project, and to allow field construction of the Project scope to the design intent. The EPCM contractor's scope will include process facility and infrastructure engineering, and for the Project this would include managing specialty contractors for major dam and tunnel designs. Mine designs will be developed and delivered by the Owner's Team.

The Procurement Team will receive the engineering documentation and obtain multiple quotations that meet engineering specifications, and provide a purchase recommendation to EPCM project director. After project director approval, the Procurement Team will purchase equipment and materials and arrange all logistics to deliver the items to the Project site ready for installation. The Procurement Team will also be responsible for establishing service contracts for engineering and field construction services.

Both the Engineering and Procurement teams will including commitments in accordance with the Impact Benefit Agreements that are in place with the First and Treaty Nations.

18.16.6 Construction Management

The Construction Management Team will be responsible for the management of all activities related to the construction management scope. The construction management scope includes all project activity in the mine, process, and infrastructure areas (on site and off site), except mining activity, environmental monitoring and reporting and community affairs, which will be the accountability of the Owner's Team.

Safety leadership will be the Construction Management Team's primary objective. After safety, the Construction Management team's next objective will be the administration and coordination of all contractors on site. This includes labor relations, monitoring and correcting when necessary, contractual compliance with safety performance, work quality, budget, schedule advance, completeness and timeliness of documentation submittal, environmental standards, and the Owner's Community Relations Plan. The Construction Management Team will oversee the installation of all materials and equipment according to engineering and manufacturers specifications, and build the Project facilities to satisfy the design intent and be fully operable. The Construction Management Team is also accountable for construction activity and the Project site until hand over to the Owner following dry commissioning.

18.16.7 Construction Supervision and Contractor Management

The objective of all site construction activities is the timely and cost-effective completion of the Project facilities in a safe manner to the design intent and required standards in accordance with schedule. Construction supervision staff, while ensuring that standards are maintained, will provide all oversight management to contractors in achieving this end.

The Contracts Management Group, which falls under the responsibilities of the site procurement manager, will use an integrated project data management system to track contractor invoicing, changes, and requests for information (RFIs). The EPCM contractor will develop a comprehensive set of project procedures, in conjunction with and approved by the Owner. These procedures will outline the requirements for the execution of the administrative activities.

The EPCM Contractor is responsible for the overall management of the construction site and will apply the following construction strategy and procedures:

- project organization, key names, and communication procedures
- reporting requirements including project systems, project meetings, minutes, and a communications matrix
- identification of the division of responsibilities among the Project stakeholders using a responsibility matrix format
- risk management procedures
- project data management, format, and distribution/filing requirements of project correspondence and documentation
- cost management and accounting procedures
- drawing and specification preparation including numbering, revision tracking, and transmittal procedures
- document control procedures
- equipment and materials procurement procedures
- project scheduling requirements, tools, formats, and frequency of delivery
- project accounting methods including cost reporting and forecasting systems
- construction contract procedures including bidding and awarding the work
- site administration procedures including camp administration rules
- site security
- field engineering
- safety procedures
- quality assurance expectations
- site and office personnel rules and regulations
- emergency site procedures and contact information
- construction temporary facilities (power, water, offices, and camp)
- site housekeeping and hazardous waste management
- mechanical completion expectations including lock-out procedures
- commissioning procedures

- project close-out and hand-over procedures
- other administrative matters and issues specific to the Project for use by the team.

18.16.8 CONTRACTING PACKAGING AND STRATEGY OVERVIEW

GENERAL

The preliminary construction strategy includes dividing the Project into contract packages including commitments in accordance with the Impact Benefit Agreements that are in place with the First and Treaty Nations. During the contractor expression of interest and pre-qualifications phase of the Project and during the advancement of detailed engineering, the contract packages will be combined to reduce the total number of contracts and form a final contracting strategy for the Project. A preliminary contract package approach and strategy are included in Appendix G1.

A portion of the contract packages listed in Appendix G1 have been combined logically, where work tasks have similar scope. The intent of combining work packages is to provide the various contractors with control of their work areas and reduce contractor-contractor interference in the field.

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 that fit with local contractors' technical ability, experience and financial capacity.

The contracting strategy document defines the type of package, contract, method of payment, engineering responsibility, purchasing of major equipment (mine and process), purchasing for minor and miscellaneous equipment, installation contractor and the management of each contract package.

When fully developed in the Project detailed design phase, the contract package approach will outline the scope of work for each package and the strategy for managing both the contractor- and Owner-supplied equipment, facilities, and services.

The contracting plan to be developed should contemplate the following methods of payment for contracts:

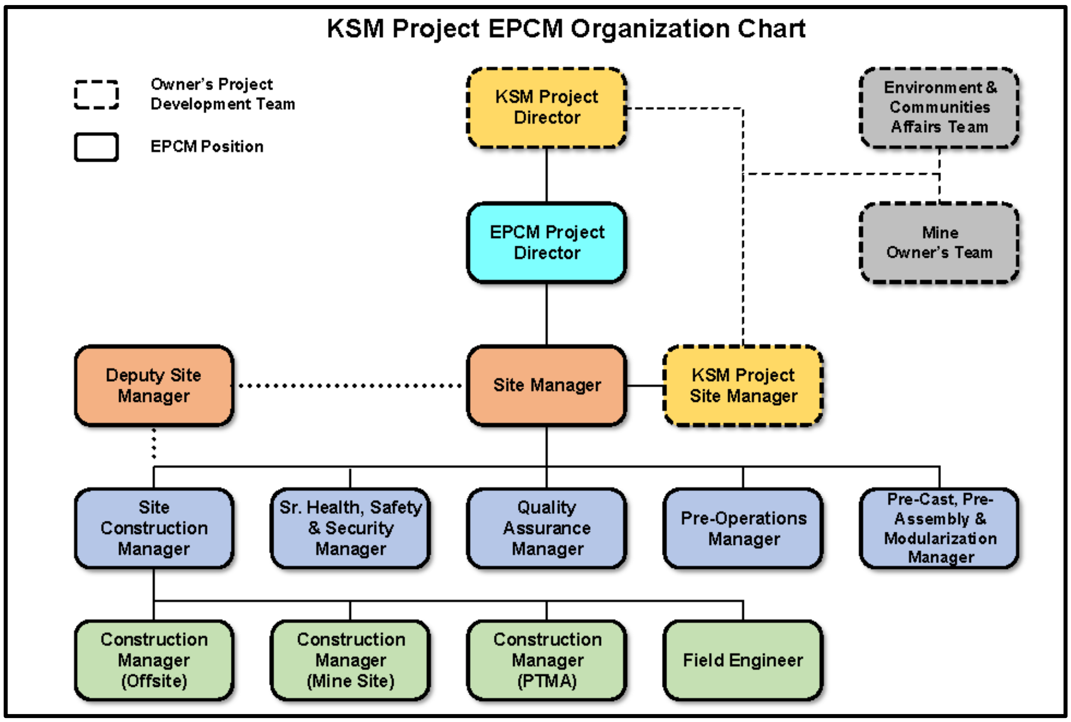
- lump sum – typically construction installation contracts
- lump sum and unit rates – typically earthworks and foundations
- time and material cost reimbursable – typically service contracts.

The type of contract requested at the tender will be dictated by how complete the engineering is. Where engineering on package scope is greater than 90% complete, it may be tendered as lump sump. Packages with a lower engineering definition may be tendered as unit rate or time and materials. This approach will provide the best possible pricing for the Project. It is intended to minimize the package risk and keep the Project on target for completion at or below approved budget.

18.16.9 SITE ORGANIZATION STRUCTURE

The EPCM site organization structure 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. A high level organizational chart is provided in Figure 18.31 and detailed organization charts are available in Appendix G1.

Figure 18.31 KSM Project EPCM Organizational Chart



The site organization and staffing plan has been designed by work type (e.g., engineering vs. project controls) with the geographical constraints of a large construction site incorporated. The staffing plan accommodates the following construction management aspects:

- site wide health and safety program
- staff rotational coverage
- office administration and project controls
- field engineering support (including quality assurance)
- commissioning
- close coordination with all stakeholders.

The organization chart provides for all site activities that are to be managed by the EPCM Team. Each of the two major construction sites will have a dedicated health and safety manager and multiple health and safety representatives to assist contractors with the daily issues and 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 locations where these orientations will be conducted and content for them will be established prior to construction starting.

The Mine Site and Treaty OPC will have a senior construction manager directly responsible for the management of cost control, scheduling, engineering, material coordination, QA support, and pre-commissioning planning. The construction manager will be directly supported by area managers and area leads.

Each construction and service contract will have a contracts administrator or specialist assigned as a single point of contact during the construction period.

The 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 has 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 each position.

Levels of Authority

The Delegation of Authority Guideline (DOAG) must be established by the Owner when authorization to proceed is granted. The DOAG will lay out the authority level throughout the hierarchy of the Project for both the Owner and EPCM teams. It will be implemented and managed by EPCM personnel.

The DOAG applies to all transactions executed on the Project, initiated by the Owner or EPCM teams. The managers with authority to execute transactions may only approve those within their area and level of authority.

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.

18.16.10 Construction Infrastructure

The construction infrastructure includes setup of all facilities required to support the construction effort in an efficient manner to achieve the Project goals and timelines. The establishment of construction infrastructure, for the most part, is set up during the early works scope and would be managed by the EPCM contractor. The operation of the construction facilities will be provided by a site services contractor.

The EPCM contractor would provide a site services superintendent to coordinate the Project requirements with contractors and service providers to ensure that there are no delays to the start of any construction work package.

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.

A detailed construction site infrastructure plot plan will be required to define the location of EPCM-, Owner-, and contractor-supplied facilities and services. Plot plans will be developed down to the detail level to address construction services such as power distribution, data management, telecommunications, emergency services, etc.

18.16.11 FIELD ENGINEERING

FIELD ENGINEERING SCOPE

The Field Engineering Group consists of a field engineering manager, field area engineers, site project document and data management, and project data system staff. The Field Engineering Group will be responsible to the project engineering manager on technical and procedural issues, and to the site construction manager regarding work flow. The group will work closely with the QA managers and technicians regarding QA and contractor QC programs, inspections, and reporting.

Project engineering will be managed 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, QA and contractor QC. 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
- volume verification for payment
- for record documentation
- planning and supervising all QA programs and non-destructive testing.

18.16.12 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 QC systems on site will be completed through the construction management team.

The discipline field engineer will establish and implement, in coordination with the QA manager, independent surveys, field inspections, and QC laboratory sampling and testing, as necessary, through the survey contractor and QC laboratory contractor.

18.16.13 HEALTH, SAFETY & SECURITY

The following is a brief overview of the HS&S management plan. Prior HS&S plans had been named health, safety, environment and community plans with scope related to environment and community management plans; however, they are excluded herein from HS&S scope since this will be managed by the EPCM contractor. Environment monitoring and reporting, and community affairs will be managed exclusively by the Owner's team. A detailed project-specific HS&S plan, HS&S manual, and standard operating procedures will be finalized during the early works construction period in Year −6.

HS&S concerns are of high importance in the engineering, design, construction, and commissioning of the Project. In order to achieve zero accidents, total commitment is required from all project personnel (Owner, EPCM Team, contractors, and vendors) to remove all conditions that could lead to injury or damage.

The Project will conform to applicable provincial, 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 director and the HS&S manager will lead in the development and implementation of the site-specific HS&S 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

Two medical facilities have been planned for the Project: one at the Mine Site, and the second at the Treaty OPC. Emergency helicopter support will be available 24 h/d, 7 d/wk, 365 d/a throughout construction. Emergency response teams will be developed during the first construction year and evolve to serve the needs of the Project as activity increases on site. Emergency medical technicians (EMTs) and nurses from the Owner's Team will be on site and available full time to address emergencies if they should arise starting in Year −5. Also, the Owner's team will include mine rescue teams comprised of staff working in mine operations.

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. A contingency plan would be prepared to account for all possible emergency events.

All site personnel will be notified of the first-aid/medical arrangements and the protocol for activating the emergency procedure during site safety orientation. 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 liberally 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, environmental and cultural orientation program prior to arrival at site. A site visitor orientation will be provided to all visitors and they will be required to be escorted by a safety-oriented worker at all times throughout their site visit.

In addition to the Project orientation, contractors will be required to provide job-specific safety orientation to all new employees prior to the start of work. This may involve verification of trade or craft certifications.

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 will be 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. Also, all equipment intended for work at site will be required to pass a designated inspection process prior to mobilization.

Safety Meetings

A weekly safety meeting will be required by every contractor (including the EPCM contractor and Owner's teams) on site or at the required frequency dictated by applicable regulation. It will provide an opportunity for all personnel to contribute timely information on safety items that relate to project activities. Weekly safety meetings will be conducted by contractor management and provide an important communication link between all their respective crews. Additionally, a safety committee meeting would be convened monthly or at the required frequency dictated by applicable regulation and comprised of a representative cross section of the work force and used as the forum for raising all issued related to worker safety.

Minutes of these meetings will be recorded on the weekly Safety Meeting Form. Contractors will be required to immediately address as many issues as possible. Issues from the weekly safety meetings that cannot be resolved immediately will be to be submitted to the Project safety department and transferred to the safety meeting action log for the superintendent's review and action.

Audits and Inspections

The Joint Health and Safety Committee (JHSC) comprised of Owner and EPCM leadership 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 HS&S manager for assignment of responsibility and completion date.

Emergency Response

Emergency response teams will be assembled from the site personnel. One team will be a dedicated group led by the HS&S Team for project-related scope. Another team comprised of qualified and trained miners from the Owner's Team will be available to provide emergency response during mining activity. 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 EPCM contractor. The contractor will develop a site-specific security plan in conjunction with the Owner.

The content of the site security plan should address the following topics:

- site access control and surveillance plans
- identification tag control
- traffic control and enforcement
- criminal activities (liaison with police authorities)

- material removal from site
- site visitor log and orientation.

18.16.14 ENVIRONMENTAL AND COMMUNITY AFFAIRS

Environmental and community affairs during construction will be managed exclusively by the Owner's team to maintain independency from the EPCM Team. Environmental knowledge and community relationships have been developed thru historical activity at the Project site and these relationships must continue to be managed appropriately in the context of regulatory permits granted and the societal expectations that have been expressed to the Owner's team. These activities will continue to be of paramount importance to the Owner well beyond the construction period, thus are best addressed by the mine owning entity that will have presence throughout the mine life. Environmental management and community relations plans are briefly described below and will be developed to a detailed level for implementation prior to construction as part of the Early Works plan described in Section 18.16.2.

The cultural awareness training program will identify and provide an overview of the various Aboriginal groups whom have an interest in the Project, focusing on their rights as it pertains to their traditional use of the natural resources of the area. Contractual obligations negotiated between the Owner and the various groups as components of Benefit Agreements will also be reviewed at a very high level.

ENVIRONMENTAL

Scope

A site-wide environmental management plan will be produced to guide the mitigation and management of environmental impacts arising from project activities. The purpose of the environmental management plan will be to:

- outline appropriate protection measures
- provide specific instructions for protecting the environment (e.g., spill prevention guidelines), including individual responsibilities, and minimizing environmental effects for achievement of zero incidents
- document environmental concerns and identify environmental management improvement areas, through a process of continuous improvement (report, implement, track, and verify)
- Report regulatory non-compliance

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 - Chance Find Procedures

Historical resources include archaeological, historical, and paleontological artifacts. 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 and environmental representative is to be notified
- the applicable government agency and Aboriginal group is to be notified
- construction activity in the area will not resume until the area has been investigated and cleared by the Owner's representative after discussions with the appropriate government agency and aboriginal group.

Community

The Owner will be responsible for community relations including providing updates to the local communities. The EPCM contractor will be responsible to make the Owner aware of any situation that is happening, or arising, that may affect directly or indirectly the communities that the Project impacts.

This may include, but is not limited to:

- 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
- harassment of Aboriginal employees
- harassment of wildlife
- illegal hunting and fishing activity.

18.16.15 Pre-commissioning/Commissioning

Overview

The commissioning period starts in any specific work area after all materials and equipment have been installed to design specification and the EPCM contractor certifies installation complete and hands the area over to the commissioning team. For the purposes of this PFS update, commissioning starts after equipment or material installation for a system or work area is complete and ends when ore starts to be processed to yield a revenue stream (i.e. the battery limit between Year -1 and Year 1). In this PFS the EPCM contractor's scope includes commissioning through dry commissioning when work areas are handed over to the Owner's commissioning team. The Owner's commissioning team executes wet and process commissioning with select operating staff and professionals that are separate in reporting line and accountability from the operations staff. When these phases of commissioning are complete, they will be handed over to Owner's operations staff who will operate the facilities through ramp up in Year 1 and on to normal operation. Support will be available from the EPCM contractor and other contractors to assist in situations or conditions that may be atypical of normal operation during wet and process commissioning.

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 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
- integration of 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.

Tagging systems will be defined for turnover of individual equipment, systems, buildings and the complete facility, based on systems descriptions and battery limits. A detailed deficiency list system will be developed to ensure the turnover to the operations team is complete.

COMMISSIONING PROGRAM

In general, the commissioning program is subdivided into three phases of mechanical completion of Project facilities.

Dry Commissioning

This first phase of mechanical completion is without ore or principal product also known as dry commissioning. 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.

Wet Commissioning

This second phase of mechanical completion proves that the facility can operate in a controllable and stable manner without significant load or feed (i.e. water tests only) also known as wet commissioning. It commences when:

- dry commissioning 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 wet commissioning activities are completed.

The Project will be considered complete and available to hand over to operations when the crushing and ore transport (train) system can deliver and operate according to design performance criteria. No hand over of mine facilities will occur since mining starts from day 1 with the mine Owner's team.

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.

18.17 OWNER'S IMPLEMENTATION PLAN

The Project will be constructed as outlined in Section 18.16 and in the time frames indicated in the Project schedule in Appendix G1. It is the Owner's responsibility to attain, and renew when necessary, all environmental and operating permits allowing site access road development, all Project construction, and all mine operations for the Project.

This Owner's implementation plan described herein attempts to provide a preliminary outline to the key responsibilities and actions the Owner's Team will take, including interaction with EPCM contractors during the construction stage and commitments in accordance with the Impact Benefit Agreements that are in place with the First and Treaty Nations.. It is not intended to be a comprehensive execution plan, rather to provide a basis for the Owner's cost structure represented in this PFS.

It is assumed the Project will be developed as a joint venture (JV) or consortium of two or more companies that will form a partnership to build and operate the KSM Mine. A JV organization would allow KSM's partners to reduce risk and spread capital expense. The "Owner" referenced in this section is synonymous with this JV organization. It is further assumed that the structure developed will assign decision making authority to the majority stakeholder to eliminate bureaucracy and streamline project development and mine production decisions. The KSM Mine will therefore have its own operating structure and reporting line through the JV partnership, maintaining its own profit and loss accountability to the JV partners. The Owner's organizational structure will have a KSM president with multiple reporting lines through a six-layer organization. Site based reporting lines to the president comprise projects, mine, and process with on-site administrative functional support as necessary to enable the Owner's Team's success. Additionally, off-site business and external relations functions would also report to the KSM president.

A central office in Vancouver is not anticipated. Instead, satellite offices will be located in Terrace, Smithers and Stewart, BC to facilitate support functions sufficiently close to the Project site to provide effective support, while building and maintaining business relationships with external stakeholders, including key Treaty and First Nations groups. Off-site locations for functions that support mine operations yield an overall lower cost structure because associated G&A staff housing, shift transportation and catering costs are mitigated.

The implementation plan described in this section highlights some key tasks required for execution by the Owner's Team over the course of construction. There are two initial critical tasks for the Owner's Team, starting with the identification and hiring of the KSM president, who will initially select a team, who in turn will do the same for their respective teams. This process is expected to be repeated throughout the course of construction until the entire enterprise organization has been built, while directing and supporting construction in various roles.

The second initial critical Owner's Team task is the engagement of an EPCM contractor early in the Project development schedule to drive the majority of the Project scope that resides outside of the Owner's direct responsibility. The type of contractual arrangement between the Owner and EPCM contractor has not been established, as it relies on the strategy that will be developed after forming the JV, and may be influenced heavily by the operating style of the JV partners.

The Owner will manage any early engineering work required to prepare design documents that support permit applications or renewals and compliance reports for permits issued by the Province of British Columbia and the Government of Canada. Site road access permits, construction camps approvals, and limited site development permits have been obtained. Additional permits will be required by the KSM Mine to ensure the completion of construction and the initiation of long term operations. The Owner will also manage any on-site drilling, hydrology, environmental monitoring, and geotechnical work, as well as the engineering work needed to prepare a final FS for the Project. The cost for these activities is excluded from this study. Infill drilling or Mineral Reserve conversion costs are also excluded, since a significant portion of the current reserve base is proven.

During Project construction, the Owner will be responsible for:

- mine development/construction including pre-stripping
- supervision of mine fleet assembly
- all environmental baseline monitoring, permitting and compliance
- operation of temporary water treatment and sewage treatment plants
- community and governmental relations
- Treaty and First Nation relations
- competitively bidding, adjudication and award of EPCM
- Owner's Team recruitment
- training of operating personnel for the Mitchell pre-mining phases
- medical support
- verification surveying for measurement and payment
- on boarding all G&A staff for both on-site and off-site positions in advance of wet commissioning to assist as part of the Commissioning Team and to develop process and procedures for each department/function to efficiently support operations
- all personnel required for ultimate mine and plant start-up and operations.

The Owner will recruit and train technical operations and administrative staff to work in the following locations:

- Treaty OPC Operations and Water Management – process plant and train operations, maintenance, TMF operations, security, and administrative personnel.
- Mine Site and Water Management – operations, maintenance, security and warehousing personnel.
- Smithers Office – proposed to be developed to service all of KSM's needs for external relations comprising governmental affairs, environmental management, permitting and compliance, public and community relations and communications, First Nations and Treaty Nation relations.
- Terrace Office – proposed as a business center where home office support will be based for these administrative functions: supply chain and logistics, human resources, IT, accounting functions, tax, business analysis, legal and audit. Health, safety and loss prevention may have occasional presence in this office, but will be primarily based on site to support ongoing operations as they develop; and,
- Stewart Port Site – management of deliveries and security for incoming construction equipment/materials and outgoing concentrate shipments.

A conceptual onboarding plan for specific G&A functions will be developed prior to turnover of constructed areas of the site from EPCM to the Owner's Team and is programmed to be well in advance of the turnover. This will allow sufficient time for the development of internal KSM Mine processes and procedures as a means to facilitate a smooth mine start up. The early onboarding plan is intended to cover gaps in service areas that may not have been detected in this early stage of design.

The time sequences for key Owner activities by year are outlined in Table 18.18. Note that this task list assumes that a FS has been completed or is running concurrently with the Project start and that either full or conditional/partial project funding will be granted by the Owner ahead of the Project start.

Table 18.18 Owner's Key Activities by Year

Year	Activities
(Year -6)	• complete PFS update by Q2 Year -6 • renew early works permits, as necessary, to support initial construction (e.g., roads) • recruit KSM president and project directors • recruit and onboard key department leads for process, human resources, business manager, and environment and community relations • initiate training programs required to meet First and Treaty Nation employment objectives • establish a business office in Terrace, BC • implement site environmental monitoring program during construction • recruit and install at site environmental monitors, TWTP and sewage plant operators, field coordinators to support early works construction mainly for road building and camp construction • competitively bid, adjudicate, and award EPCM services • in conjunction with the EPCM contractor, develop a detailed execution plan for the Project leveraging all previous engineering, project planning, and environmental permitting work. • establish project governance between Owner and EPCM teams • release early works contracts for road and bridge construction (critical path) • obtain permitting for use of Frank Mackie Glacier Road during winter • initiate fish habitat compensation construction • obtain required authorizations so the in-water bridge piers on the Bell-Irving and Unuk rivers can be installed during the first winter construction season
(Year -5)	• manage EPCM contractor focussing on detail engineering and long-lead procurement activities off site, and early works construction on site • augment site Owner's Team in these areas: environmental services, survey (measurement and payment), medical services • expand Smithers, BC office to accommodate larger office headcount • continue Mine Site development • establish road to Sulphurets quarry to support WSD construction

table continues...

Year	Activities
(Year -4)	• finalize concentrate smelting contract terms • finalize detailed engineering work for early phase construction activities, initiate detailed design for the remainder of project scope • augment site Owner's Team in the following areas: process operations, site administration, security, site project management, human resources and environmental services; process operations staff are expected initially to reside in the EPCM contractor's office to guide detailed design and process equipment selection • continue to manage EPCM contractor focussing on detail engineering and procurement activities off site and early works construction on site; procurement focussing on large equipment purchases required at site in Year -3 and beyond • continue mine site development • deliver Sulphurets rock to WSD for construction
(Year -3)	• complete final detailed design for all remaining project scope and finalize all equipment purchases. • initiate enterprise computer systems set up in off-site offices • continue to manage EPCM contractor whose focus is now shifted mainly to field activity • initiate business readiness planning leveraging Owner's Team resources, working collaboratively with the EPCM contractor • augment off-site human resources team • continue Mine Site development • continue delivery of Sulphurets rock to WSD for construction
(Year -2)	• continue to manage EPCM contractor whose focus is solely on field activity • establish satellite office and recruit port staff for facility at Stewart, BC • expand Terrace, BC office footprint to full size necessary to fully support mine and process operations on site • recruit all operations staff for HDS WTP to participate in wet commissioning of the facility following dry commissioning and handover by EPCM, initiate plant start up and operations • continue Mine Site development • continue delivery of Sulphurets rock to WSD for construction
(Year -1)	• recruit and on board all remaining positions for the process plant, metallurgical laboratory and TMF; plant operators are highest priority and TMF staff will be on boarded toward year end • complete recruitment for mine operations team • perform a significant amount of operator training in preparation for operations start up • oversight of OEM assembly of major mine fleet equipment • after completion of WSD, initiate ore mining in Mitchell open pit Phase I • delivery of first ore to Mitchell OPC • fully execute wet commissioning following dry commissioning and handover by EPCM of the process plant and all infrastructure • initiate ore transfer through the MTT controlled by process operations • introduce first ore to Treaty OPC and begin process plant ramp up

table continues...

Year	Activities
Operations Year 1	• complete recruitment and on boarding for remaining G&A and process operations staff during process plant ramp up • complete oversight of OEM assembly of major mine fleet equipment • produce first copper concentrate and doré at the Treaty OPC • initiate sand tailing production at the TMF and train employees on procedures for this long term dam construction effort • achieve commercial production • ramp up to full scale mining and primary crusher operations at the Mitchell OPC from Phase I mine development stages at Mitchell and Sulphurets • shipment of copper concentrate from the Port of Stewart

19.0 MARKET STUDIES AND CONTRACTS

Seabridge engaged NSA to provide an opinion report on marketing inputs for the 2016 PFS, excluding off-site transportation costs. The information and options in this section come from the opinion report located in Appendix B. All currency amounts used in this section are in US dollars, unless otherwise specified.

19.1 COPPER CONCENTRATE

19.1.1 MARKETABILITY

When considering the marketability of copper concentrates, quality and quantity are determining factors. There is considerable variation in the quality of concentrates and the requirements of various smelters do vary; such variation relates to the technical abilities of the smelter and its overall concentrate feed and blend.

Ideally, smelters prefer to blend their feed with approximately 30% copper and similar amounts of iron and sulphur. In the last several years, however, the grade of some of the major high-grade suppliers has been dropping. At the same time, many new suppliers tend to blend copper-gold concentrates with copper content in the low to mid 20% range. Consequently, the market has seen the blend for most smelters drop to a copper level of 27 to 28%. Apart from the level of copper, iron, and sulphur, other key elements in determining concentrate salability include the levels of gold and silver content, as well as any impurities.

Based on the impurity levels projected by Tetra Tech (using the test results completed to date – see Table 17.3), concentrates from the Project are relatively clean. Depending on the market situation at the time of contract negotiations, penalties will likely be minimal, if at all applicable. Some smelters, such as in Japan, South Korea, and Europe, are expected to have more interest in copper concentrates with high gold content.

19.1.2 SMELTING TERMS

COPPER CONCENTRATE SMELTING MARKET

Copper concentrates account for approximately four-fifths of total newly-mined copper production, with the balance of output coming from solvent extraction and electrowinning copper cathode and other copper-bearing by-products.

Concentrate supply started to increase over 2013 and is expected to continue to increase towards the end of this decade as a result of new projects and announced expansions of operational mines.

A significant portion of copper concentrate is processed by integrated smelters—captive plants that are vertically integrated with mines through ownership. However, an increasing annual volume of global copper concentrate is treated by custom smelters that generally are not integrated, although there is, in many cases, investment ownership in mines. Custom smelters have increased their overall smelting market share from 30% in 1980 to approximately 50% today.

Over the last two decades there has been a significant expansion of smelting and refining capacity, particularly in India and China. The Chinese smelting industry has increased imports, as limited domestic mine capacity could not meet demand. This trend has been a key determinant in world concentrate supply/demand balances.

The copper concentrate market has seen significant structural imbalances in the recent years between mine production and smelting capacities. Over the last couple of years there have been significant increases in smelter treatment charges and refining charges (TCs/RCs). The balance of supply and demand for concentrates is set by the whole of the concentrate output of the mining industry and by the availability of capacity across the smelting industry. The availability of custom concentrates, relative to smelting capacity, should, in theory, be the ultimate determinant of terms for custom treatment of concentrates.

19.1.3 COPPER CONCENTRATES CONTRACTS AND TERMS

The concentrate market is basically split into two types of contracts. First, there are long-term off-take contracts between mines and smelters that reflect, in general, the annual concentrate supply and demand balance. Second, there is spot or short-term business primarily between mines and traders and, on a much smaller scale, between mines and smelters. By its nature, such business is much more volatile and there is considerable variation in spot TCs/RCs, not only annually, but over each year.

CURRENT AND FUTURE TERMS

NSA suggests that annual benchmark numbers are beginning to reflect a move towards sustainable long-term numbers. NSA believes that the most likely scenario is that ultimately charges have to move up towards a level that is economic for the smelting industry over the long term. The benchmark numbers for the last several years are shown in Table 19.1

Table 19.1 Benchmark Smelting Terms

	2016	2015	2014	2013	2012
Copper Treatment Charges ($/dmt)	97.500	107.00	92.000	70.000	63.500
Copper Refining Charges ($/lb)	0.0975	0.1070	0.0920	0.0700	0.0635

For comparison, the recent spot market of April 2016 indicates sales into the Chinese market where the levels of TCs/RCs were between $90 and $95/dmt of concentrate and $0.090 and $0.095/lb of copper, respectively.

Over approximately 20 years (up to 2005), historical TCs/RCs averaged approximately \$77/dmt and \$0.077/lb (including approximately \$0.01 of participation for these purposes, split between the treatment charge and the copper refining charge) at an average price of \$0.93/lb of refined copper.

The general view today does not see price participation materializing in the near future; however, it should not be ignored. Historically, when price participation first became a factor in concentrate negotiations it was only applicable at a price level higher than the price existing at the time of negotiation.

NSA suggests that the annual benchmark terms realized over the last couple of years are likely to be a guide to future levels. With this in mind, and for purposes of this study, the assumption should be a copper treatment charge of \$100/dmt, with copper refining charges of \$0.10/lb of copper.

TCs/RCs are not the only terms that are used in valuing copper concentrates. Payments and deductions are a matter of negotiation and will vary with many factors, including supply and demand, and custom individual markets.

The following terms are an indication of "standard" long-term smelter charges, including suggested TC/RC terms. Delivery is on the basis of Cost, Insurance and Freight – Free Out (CIF-FO) smelter ports (the mine pays all costs up to delivery port and the buyer arranges and pays for cargo discharge).

PAYABLE METALS

Copper — Pay 96.5% with a minimum deduction of 1 unit (amount deducted has to equate to a minimum of 1% of the agreed concentrate copper assay).

Silver — If over 30 g/dmt pay 90%.

Gold — A scale is applicable with some variations of the following:

- less than 1 g/dmt, no payment
- 1 to 3 g/dmt, pay 90%
- 3 to 5 g/dmt, pay 93%
- 5 to 7 g/dmt, pay 95%
- 7 to 10 g/dmt, pay 96.5%
- 10 to 20 g/dmt, pay 97%
- over 20 g/dmt pay 97.5%
- over 30 g/dmt pay 97.75%.

Gold and silver payments may vary between smelter locations. In China, high gold in copper concentrates is not generally desired; relating more to internal pricing issues rather than technical concerns. Technically, the more modern smelting facilities are able to accept payment formulas similar to Japan and South Korea, but for many of the older smelters in North China, this is not the case. In Europe, with grades of over 40 g of gold content, payment of 97.75% with a minimum deduction of 1 g is likely to apply.

Refining Charges

Copper	$0.10/lb payable copper
Gold	$6.00 to $8.00/oz payable gold
Silver	$0.50/oz payable silver

Treatment Charges

Treatment Charge $100.00/dmt CIF-FO main smelter port.

Price Participation

Not applicable at present.

Penalties

Arsenic:	$2.50 to $3.00 per 0.1% over 0.1% up to 0.5% arsenic
Antimony:	$3.00 to $4.00 per 0.1% over 0.1% antimony
Lead:	$2.00 to $3.00 per 1% over 0.5% to 1.0% lead
Zinc:	$2.00 to $3.00 per 1% over 2% to 3% zinc
Mercury:	$2.00 per each 10 ppm over 10 ppm mercury
Bismuth:	$3.00 to $5.00 per 0.01% over 0.03 to 0.05% bismuth
Selenium:	$3.00 to $5.00 per 0.01% over 0.05% selenium
Tellurium:	$4.00 to $5.00 per 0.01% over 0.02% to 0.03% tellurium
Fluorine	$1.00 to $2.00 per 100 ppm over 300 ppm fluorine
Chlorine	$1.00 to $3.00 per 100 ppm over 300 ppm chlorine.

Furthermore, penalties may also vary from smelter to smelter. It should be noted that for the elements where a percentage range is used, this relates to ranges of penalty thresholds that are negotiated. The penalties noted in this section are generally in line with levels applicable over recent years, but there is a tendency towards higher levels.

Based on the anticipated impurity levels derived from the test results by Tetra Tech (as presented in Table 17.3), the concentrates from the Project are relatively clean, and depending on the market situation at the time of contract negotiations, penalties will likely be minimal if at all applicable. As most of the mill feeds will be the blended materials from different deposits and spatial locations, the blend should effectively mitigate penalty elements rising for the ore from some limit locations.

PAYMENT

A provisional 90% is paid three to 15 days after vessel arrival in Asia and India. Sales into Europe normally involve later payment, possibly more than 30 days after arrival. The 10% balance is paid when all facts are known.

OTHER OFF-SITE COSTS

Various indirect costs other than smelter charges include:

- Losses; assumed to be 0.1% or less, due to improvements in material handling.
- Insurance; marine insurance is assumed to be in the range of 0.1 to 0.15% of net invoice value of the concentrate.
- Supervision, assaying and umpire costs; the costs for third-party supervision and assaying are assumed to be approximately US$1/dmt.
- Marketing; the cost of marketing varies with concentrate tonnage, location, and number of smelters to be shipped. For the Project, the estimated marketing cost is in the range of US$5 to US$10/dmt.
- Concentrate transportation; the transportation costs for copper concentrate are based on the following assumptions by Tetra Tech:
  - trucking: US$38.06/wmt
  - port storage and handling: US$14.40/wmt
  - ocean transport to Asian port: US$26.00/wmt.

19.2 MOLYBDENITE CONCENTRATE

19.2.1 SMELTING CHARGE

Molybdenum concentrates of either primary production origin, or as a co-product, need to be further processed. This is initially to produce molybdenum oxide by roasting, or by use of autoclaves for upgrading. Quality is an important consideration and certain elements can be deleterious. As a rule of thumb, 50% molybdenum content is considered the minimum. Below that, buyers will begin to be a bit selective and charges will rise somewhat. One guideline, given for each 1% below 50%, would be an increase in charges of $0.05/lb of molybdenum.

Currently, the deduction for roasting is very quality dependent, with high copper content concentrates generally selling at a 10 to 15% discount. Assuming that the copper content of its molybdenum concentrates is reduced to 0.45% or less, the lower end of the range will apply to clean high-grade concentrates. In the recent years (2005 to 2009), discounts for high-copper molybdenum concentrates have reached 25%.

In summary, it is recommended to use a discount of 12% from the price, with a minimum of $1.00/lb and a maximum of $2.50/lb. This discount would be inclusive of all charges mine to market, such as delivery costs, irrespective of whether or not the concentrate is sold to a trader or a roaster directly.

On average, the molybdenum concentrate could contain approximately 1,000 to 2,000 ppm rhenium or higher. Given the high rhenium content, some roasters would recognize the rhenium content in the form of lower treatment charge or some payments. However, it is difficult to project the rhenium value at the current market and the level of study.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCI OR COMMUNITY IMPACT

20.1 LICENSING AND PERMITTING

The KSM Project is subject to the BC *Environmental Assessment Act* (BCEAA), the *Canadian Environmental Assessment Act* (CEAA), and Chapter 10 of the NFA.

As of June 2016, the Project has successfully gone through the provincial and federal processes, and the appropriate certificates/approvals have been obtained. Additionally, permits for early-stage construction activities have also been obtained. Seabridge is currently in the process of obtaining numerous provincial and federal permits to allow for the construction of parts of the Project, as well as to expand exploration activities. Details of the provincial, federal, and NFA processes and current statuses, as well as the current permitting status of the Project, are including in this section.

The Project underwent a harmonized EA process with the provincial and federal governments. Both governments conducted the EA cooperatively, in accordance with the principles of the Canada-BC Agreement on Environmental Assessment Cooperation (Cooperation Agreement 2004). The process included a working group comprised of federal and provincial officials, the NLG, Aboriginal groups, and local government agencies. Representatives of the US federal and Alaska state agencies were extensively involved in the EA process, as a matter of courtesy at the insistence of Seabridge, given that the mineral deposits are located on a tributary of the Unuk River, a transboundary river, 30 km upstream of the US/Canada border. Authorizations are not required from any US federal or state regulatory agency for the Project to procced into construction and operation.

20.1.1 PROVINCIAL PROCESS

Under the BC EA process and its regulations, certain categories of larger-scale projects must undergo an EA, and an EA Certificate must be obtained before the Project can proceed. The scope, procedures, and methods used for each assessment are tailored to the specific circumstances of a proposed project. The EA must assess a project's potential environmental, economic, social, heritage, and health effects.

Under the BCEAA Reviewable Projects Regulation, the proponent of a new mineral mine facility, with a production capacity of greater than 75,000 t/a of mineral ore, must obtain an EA Certificate. The Project will have an annual mill throughput of 43,800,000 t/a, which substantially exceeds this threshold.

Seabridge was accepted into the BC Environmental Review process in March 2008, following submission of a Project Description (Rescan 2013). In July 2013, Seabridge submitted an Application/EIS (Rescan 2013) under the BCEAA (2002) in accordance with the approved project Application Information Requirements (AIR) to the BC Environmental Assessment Office (BCEAO). The Application/EIS was approved and EA Certificate #M14-01 for the Project was issued on July 29, 2014.

The full Application/EIS can be found on the BCEAO web site here: http://a100.gov.bc.ca/appsdata/epic/html/deploy/epic_project_doc_list_322_r_app.html

The Project EA Certificate #M14-01, including Schedule A (Certified Project Description) and Schedule B (Table of Conditions) are provided in Appendix K1.

A plain English summary of the Application/EIS is provided in Appendix K4.

20.1.2 FEDERAL PROCESS

The CEAA (1992) was significantly amended in 2003 and 2010, and was entirely repealed and replaced by the CEAA (2012) on July 6, 2012. The CEAA (2012) dispensed with comprehensive studies, but also provided that comprehensive studies started under the relevant provisions of the CEAA (1992), as amended from time to time, should be completed under those provisions. Under the 2010 amendments, the lead federal agency for the comprehensive study for the Project is the Canadian Environmental Assessment (CEA) Agency, which assumed the role from several CEAA responsible authorities (those federal agencies that may issue approvals for the Project under the Law List Regulations). The CEAA (2012) provides for expedited timelines to conclude the EA process for those comprehensive studies that were underway before repeal of the CEAA (1992). For the Project, six months of federal government review time applied.

The Project became subject to the CEAA (1992) because it may require several statutory authorizations listed in the Law List Regulations. The Project was subject to a comprehensive study level of assessment under the CEAA (1992) because the proposed daily ore mill feed of 130,000 t/d exceeds two thresholds set out in the CEAA (1992) Comprehensive Study List Regulations; specifically, the 4,000 t/d threshold for metal mills, and the 600 t/d production threshold for gold mines. Certain dam structures proposed for the Project also exceed the 10,000,000 m^3/a threshold for water diversions.

The Project was deemed to require a “comprehensive study” in July 2009 and a “notice of commencement of an environment assessment” was submitted to Seabridge. The terms or the scope of assessment was developed and posted by CEAA for public comment in late May 2010. With the CEAA (2010) amendment, the terms of reference was subsequently re-posted for public comment by the CEA Agency in July 2010. The draft *KSM (Kerr-Sulphurets-Mitchell) Project Comprehensive Study Report* was subsequently issued by the CEA Agency in July 2014. The complete report is provided in Appendix K5.

The *KSM Project Comprehensive Study Report*, along with filed public comments from the NLG, other Aboriginal groups, and the public, were considered by the Minister of the

Environment when making her final EA decision. The Project received federal approval on December 19, 2014. The *Environmental Assessment Decision Statement* is provided in Appendix K6.

20.1.3 NISGA'A FINAL AGREEMENT

The NFA is a treaty concluded between Nisga'a Nation, the Government of Canada, and the Government of BC in 1999. The NFA came into effect in May 2000 under the federal *Constitution Act* and the BC *Nisga'a Final Agreement Act*, and sets out Nisga'a rights over approximately 27,000 km^2 of land in the Nass River system and surrounding drainages.

The NFA establishes three categories of lands with different specified Nisga'a interests - the Nisga'a Lands (approximately 2,000 km^2), the Nass Wildlife Area ([NWA], more than 16,000 km^2), and the Nass Area (approximately 27,000 km^2)—the latter incorporating the Nisga'a Lands and the NWA within it. The NFA affords title to Nisga'a Nation within the Nisga'a Lands and defines the rights of Nisga'a Nation to self-government and law making authority in this area. The NFA also specifies Nisga'a Nation rights to access and make use of natural resources in the NWA and the Nass Area.

Seabridge proposes to develop some components of the Project footprint within the Nass Area, including the Treaty OPC, the TMF, and the northern portion of the MTT. No Project components will physically occupy any portion of Nisga'a Lands or the NWA, both of which are located south of the potentially-affected portion of the Nass Area.

The NFA makes explicit provision for Nisga'a participation in federal or provincial EAs of projects sited anywhere within the outer Nass Area boundary. Seabridge was directed by the federal and provincial governments to ensure that it conducts its EA responsibilities for the Project in compliance with all relevant Nisga'a treaty rights, including those dealing with economic, social, cultural, and environmental interests. Chapter 10 of the NFA (Environmental Protection and Assessment), paragraphs 6 to 10, provide for meaningful Nisga'a participation in the EA through effective coordination, timely notice, provisioning of information and studies to Nisga'a Nation, and a clear focus on assessment of potential adverse project effects on residents of Nisga'a Lands, the Nisga'a Lands themselves, or more generally, on Nisga'a interests as set out in the NFA.

A federal approach was established in February 2011, following consultation with the NLG and the Province of BC, to clarify how the Government of Canada would meet Chapter 10, paragraph 8 requirements in the EA, including the assessment of effects under paragraphs 8(e) and 8(f), and the issuance of a Ministerial NFA Project Recommendation.

The Government of Canada worked collaboratively with the NLG and the Government of BC to facilitate the assessment of paragraphs 8(e) and 8(f) effects as part of the comprehensive study. Seabridge conducted an economic, social, and cultural impact assessment (ESCIA) on the well-being of Nisga'a citizens (i.e., paragraph 8(f) effects) based on a work plan that was required by the joint AIR. Effects defined under paragraph 8(e) were described in the Application/EIS as part of Seabridge's analysis of the Project's effects on environmental valued components (VCs).

The *KSM Project Comprehensive Study Report* (Appendix K5) examined both paragraphs 8(e) and 8(f) effects on Nisga'a citizens, lands, and interests, and provides the federal perspective on these effects. This information, along with comments received during the final public consultation and any agreements between Seabridge and the NLG concerning the effects of the Project, were used to inform the Minister of the Environment's NFA Project Recommendation. The Project received federal approval in December 2014. The Project Recommendation that was issued by the Minister under Chapter 10, Sections 8 and 9 of the NFA is provided in Appendix K7.

20.1.4 PROVINCIAL PERMITS

The Application/EIS was accompanied by applications for eligible provincial authorizations in accordance with the BCEAA (2002) Concurrent Approvals Regulation (BC Reg. 371/2002).

This set of initial permits is referred to as the "Batch 1 Permits" and included permits for the following mine components:

- KSM Project *Mines Act* and *Environmental Management* Act Permit Application for Limited Site Construction (May 2013)
- Special Use permits for the CCAR and TCAR
- KSM Construction Camps
- KSM Project Treaty Transmission Line
- MTT Permit Application

A complete list of the Batch 1 permits received during the concurrent permit review process is provided in Appendix K10.

In November 2015, Seabridge submitted an application to amend the existing *Mines Act* exploration permit MX-1-571 and an *Environmental Management Act* (2003) permit to facilitate the commencement of construction of the Deep Kerr Exploration Adit. The Deep Kerr Exploration Adit will be located near the existing temporary KSM exploration camp (Appendix K9).

As of June 2016, an estimated 108 provincial permits are still required to fully develop the Project. A draft list of these Batch 2 permits is provided in Appendix K10.

20.1.5 FEDERAL PERMITS

The Application/EIS included federal applications for the following:

- Fisheries Authorization application, including draft Compensation Plans
- MMER Schedule 2 Amendment Application
- *International Rivers Improvements Act* Licence Application
- Navigable Waters application.

The following sections provide additional details along with the current status of federal permits.

FISHERIES AUTHORIZATIONS AND MMER SCHEDULE 2 AMENDMENT APPLICATION

Legislation

Fish and fish habitat are protected under the *Fisheries Act* (1985), as well as other federal regulatory acts and principles. In 2012, the *Fisheries Act* was amended to legislate the federal government's direction to focus efforts on protecting the productivity of commercial, recreational, and Aboriginal fisheries; to institute enhanced compliance and protection tools that are more easily enforceable; to provide clarity, certainty, and consistency of regulatory requirements; and to enable/enhance partnerships with stakeholders.

The changes to the *Fisheries Act* include a prohibition against causing serious harm to fish that are part of or support a commercial, recreational, or Aboriginal fishery (*Section 35*), provisions for flow and passage (*Sections 20 and 21*), and a framework for regulatory decision-making (*Sections 6 and 6.1*).

On November 1, 2013, The Fisheries Protection Policy Statement (Fisheries and Oceans Canada [DFO] 2013a) was issued and replaced the earlier Policy for the Management of Fish Habitat (DFO 1986). Although the new policy statement does not include the "no net loss" (NNL) principle, as outlined in the earlier policy, application of this NNL principle provides some useful guidance when considering "serious harm to fish".

Any project or activity that causes a serious harm to fish that are part of, or support, a commercial, recreational, or Aboriginal fishery requires an authorization from DFO. Regulations have been developed to guide the application for this authorization: Applications for Authorization under Paragraph 35(2)(b) of the *Fisheries Act* Regulations. DFO has issued additional guidance in the "The Fisheries Protection Program Operational Approach".

The Project entered the EA process prior to the changes in the *Fisheries Act* and Fisheries Protection Policy Statement. As such, the original *Fisheries Act* and Policy for the Management of Fish Habitat have been used for planning and permitting.

KSM Project

Fish habitat will be impacted by the KSM Project in two ways:

- deposition of tailings into a fish-bearing watercourse, which requires an authorization under Section 36 of the *Fisheries Act* (1985)
- loss of fish habitat due to Project infrastructure (e.g., roads, transmission line, dams), which requires an authorization under Section 35 of the *Fisheries Act*

Fish habitat compensation plans related to the loss of fish habitat due to Project infrastructure and due to the depositions of tailings were completed as part of the Application/EIS. Compensation reports for both the harmful alteration, disruption, or destruction (HADD) of fish habitat and the MMER were developed with input DFO, the BC MOE, and key Aboriginal groups and stakeholders, as identified in the Section 11 Order and included MOE, Gitanyow First Nation and Wilp Wii'litsxw, Gitxsan First Nation, *wilp* Skii km Lax Ha, Tahltan Nation, Nisga'a Nation, and members of the public.

Update on MMER Schedule 2 Amendment Process

An additional round of public consultation sessions was held in September 2014, following the approval of the BC EA Process. The objective of this consultation, led by Environment Canada and DFO was to provide participants an opportunity to comment on proposed amendments to the MMER under the *Fisheries Act*. These amendments would add two water bodies (portions of the tributaries to the North Treaty and South Teigen Creeks) to Schedule 2 of the MMER, allowing Seabridge to use the water body for the disposal of mine tailings from the proposed KSM Mine. These meetings included information on the Project, fisheries compensation options, as well as multiple accounts analysis for the TMF.

The following public meetings occurred in 2014:

- September 8 – Iskut, BC
- September 8 – Dease Lake, BC
- September 9 – Telegraph Creek, BC
- September 10 – Hazelton, BC
- September 11 – Gitanyow, BC
- September 12 – New Aiyansh, BC
- September 13 – Terrace, BC
- September 17 – Gatineau, QC
- September 23 – Gitwinksihlkw, BC
- September 24 – Laxgalts'ap, BC
- September 25 – Gingolx, BC.

The Government of Canada approved the Project in December 2014. Recent correspondence with Environment Canada on the Schedule 2 process, dated January 5, 2016, stated that the regional impact analysis statement document was in progress and will likely be considered by the Treasury Board in 2016.

INTERNATIONAL RIVER IMPROVEMENTS ACT APPLICATION

Seabridge submitted an application for a license under the *International River Improvements Act* to Environment Canada on February 2015. The full application is provided in Appendix K11.

The application was prepared in accordance with Sections 6 and 7 of the International River Improvements Regulations (C.R.C., c982). The application is for improvements to the Unuk River, specifically within the Sulphurets Creek Watershed (which is a tributary to the Unuk River).

The application is pending approval by Environment Canada.

Navigable Waters Application

Exemptions to the *Navigation Protection Act* were submitted as components of the Application/EIS. Further to a letter received from Transport Canada dated August 1, 2014, Transport Canada subsequently determined that the waterways of the Project were not navigable waters as defined within the *Navigation Protection Act*, and as a result there was no requirement to obtain an exemption to the prohibition pursuant to Section 24 of the *Navigation Protection Act*. A copy of this letter is provided in Appendix K12.

20.2 Environmental Settings and Studies

The Project is a proposed gold, copper, silver, and molybdenum mine located in the coastal mountains of northwestern BC, approximately 950 km northwest of Vancouver, 65 km northwest of Stewart, and 35 km northeast of the BC-Alaska border.

The topography of the Project area varies from 240 masl at the proposed CCAR crossing of the Unuk River, to over 2,300 masl at the highest peak. A significant portion of the terrain that will host the mining activities occurs at treeline and in alpine terrain. Glaciers and ice fields dominate the terrain to the north, east, and south of the Project site. The glaciers have been receding in the last several decades.

The following sections summarize the environmental settings for valued components of the biophysical and socio-community aspects of the Project. Please refer to Rescan (2013) for additional details.

20.2.1 Biophysical Setting

Climate and Atmospheric Conditions

The meteorological conditions in the area are primarily influenced by the Pacific Ocean to the west and continental Arctic regions to the northeast. Hence, the Project is in a transition zone between wet coastal and dry/cold interior climate zones. The influence of the mountain ranges on the Pacific and continental air masses results in precipitation and air temperatures that are widely variable across the Project region.

Strong winds generally occur in all seasons at high elevations above the mountains, with winds generally coming from the northeast, southeast, and southwest quadrants in the winters and from the southwest quadrant in the summers. Winds at low elevations are funneled through valleys with a light to moderate down-valley flow of Arctic air from the northeast in the winter and a light up-valley flow of warm Pacific air from the southwest in the summer.

The regional hydro-climate reflects the interactions between incoming weather systems and local topography that produce a degree of spatial variability in snowfall and rainfall. When Pacific air streams confront the west-facing slopes of the Coast Mountains, the moisture-laden air is forced up the slopes. As the air cools and rises, it is less capable of holding moisture and releases it as rain or snowfall. The mountains also slow down cyclonic storms, which can lead to prolonged and sometimes heavy rainfalls. Over the mountain summit, the air descends and warms, which disperses the cloud and potential rain through evaporation. The result is a dramatic reduction of precipitation in the rain-shadow. Within BC, the series of mountain ranges that parallel the coast produce a decrease in precipitation with increasing distance from the ocean as storms pass over the successive ranges.

The climate in the local region is typical of temperate rainforest with average monthly air temperature ranging between -12 and 14.7°C. Within the four-year period of 2008 to 2011, the highest daily maximum temperature ranged between 25.3 and 30.2°C, and the lowest daily minimum temperature ranged between -22.1 and -31.1°C. Within the same period, annual precipitation ranged from 689 mm at the Teigen Creek station to 1,914 mm at the Eskay Creek station. The highest precipitation in the local region occurs in September and October. Subarctic conditions are present at high elevations (generally above 1,500 masl) where strong winds blowing in a westerly direction predominate in winter. At low elevations, winds are funneled through valleys—Arctic air from the northeast in the winter and warm Pacific air from the southwest in the summer.

The air quality in the area proposed for Project development and elsewhere in northwestern BC is predominantly unaffected by anthropogenic sources, reflecting the region's remoteness and the lack of, and localized nature of, sources of anthropogenic air emissions sources.

Geology and Geochemistry

Stikinia—a terrane of Triassic and Jurassic volcanic arcs, accreted onto Palaeozoic basement rocks of the western North American continental margin—forms the regional geological setting for the KSM Property. Late Cretaceous folding and thrust faulting of the main stratigraphic groups in the region generated Stikinia's current structural features. Thrust faulting is common, and some strata are tightly folded. Remnants of Quaternary Era basaltic volcanic eruptions occur throughout the region. Early Jurassic sub-volcanic intrusive complexes are common. Several complexes host hydrothermal systems rich in precious and base metals, including the copper-gold porphyry deposits at the Galore Creek, Red Chris, Kemess, Mount Milligan and KSM properties, as well as related polymetallic deposits, including those at the Premier, Eskay Creek, Snip, Bruceside, and Granduc properties.

Local geology is dominated by variably deformed oceanic island arc complexes. Late Jurassic and Cretaceous back-arc basins to the east of the KSM property contain thick accumulations of fine black clastic sedimentary rocks, all folded and faulted to differing degrees during late Cretaceous compressional tectonics. Unloading linked to glacier retreat in the Mitchell Creek and Treaty Creek valleys has resulted in the formation of exfoliation stress relief fractures parallel to the valley flanks. Dikes, sills and plutonic plugs were intruded into these strata in early Jurassic times. Copper-gold mineralization is typically best developed at the margins of these intrusions. Extrusive and intrusive activity has led to rock alteration of various types around the Project mineralized deposits.

The mineralized zones in the local area and more regionally, tend to be sulphide-rich. Where sulphide minerals such as pyrite are present, oxidation can create ARD, unless sufficient quantities of neutralizing minerals are available. In the event that acidic drainage is formed, low pH conditions can lead to higher rates of metal leaching (ML). Baseline surface water and groundwater quality in the vicinity of mineralized zones in the region exhibit relatively low pH and significant metal concentrations, reflecting the presence of sulphide minerals and the natural occurrence of ML/ARD processes.

Physiography

Today, the mountain topography is very rugged. Glaciers are common in high elevations. Most steep slopes are covered by bedrock and accumulations of rubbly colluvium. Gentler slopes have a thin mantle of morainal material (glacial till). Thick glacial deposits are generally restricted to the margins of major valley floors and adjacent lower slopes. Avalanches and slope failures are common features at high and intermediate elevations (above 1,500 masl).

Topography in the vicinity of the KSM Property ranges from a low elevation of 240 masl (at the proposed CCAR crossing of the Unuk River) to more than 2,300 masl at the highest peak. A large portion of the terrain is situated at, or above, the tree-line and in alpine areas. Glaciers and icefields dominate the terrain to the north, east, and south of the Project area. Glaciers in the area have been receding in the last several decades.

Geohazards

Locally and regionally, geohazards are linked primarily to landslides and snow avalanches. Landslide hazards are abundant throughout the region. They are attributed to several factors, including the presence of unstable surficial soils and weak bedrock, repeated geologically recent glaciations, resulting in over-steepened valley sidewalls, the loss of slope buttress support following glacial recession, abundance of veneers that are shallow to bedrock, and the high precipitation environment.

Thick glacial deposits are generally restricted to the margins of major valley floors and adjacent lower slopes. Much of the surficial cover in the Project area is unstable to potentially unstable, since all of the main valleys have been subject to glacial advance and retreat, and associated process such as erosion and deposition. Left behind are moderately steep upper slopes, steeper valley walls and gently sloping and wide valley floors.

Unstable lateral morainal till has been deposited on slopes at angles that exceed the angle of repose, resulting in rubbly colluvium accumulating along moderate steep slopes and valley bottoms. Post glacial processes have also contributed to terrain instability, as much of the recent deposits are loose and highly erodible. Periglacial processes are also in evidence, as several glaciers at the Project site are receding, leaving behind hanging valleys, over-steepened lateral moraines and glacio-fluvial outwash deposits in valley bottoms. The unloading of the valley walls following glacial retreat has led to pressure release cracks and associated local instability on over-steepened slopes resulting geohazards, such as rock fall, debris avalanches and slumping of surficial materials. These geohazard processes are endemic to the local area.

Snow avalanche hazards are abundant due to high elevation, substantial snow supply and generally steeper slope gradients, and tend to be associated with terrain that is open and steep. Since the region is located in a transition zone between maritime and continental climate zones, significant temperature and moisture fluctuations are experienced throughout an average winter. The avalanche season typically begins in early October at the higher elevations, and often extends until late June or early July. In valley bottoms, avalanches may be experienced from late October to late May.

Soil Development

Regional climate and geological history, in combination with local topography and vegetation, affect soil landscapes found in the local area. In high elevations solifluction, nivation, and cryoturbation disrupt, displace, and mix soil horizons, while the cold climate slows down mineral weathering and organic decomposition. Weathered volcanic rocks provide coarse-textured, acidic parent materials. As a result, soil development is often weak. The steep terrain results in unstable slopes where soil development is further hindered by mass movement of surficial materials.

Regosols (weakly developed, well-drained mineral soils in unconsolidated materials) and occasionally Cryosols (periodically frozen soils) occur in these areas. In lower elevations, soils are commonly subjected to seepage. Excess moisture and a high incidence of poorly drained soils are typical. Due to the steep terrain, most common parent materials consist of colluvial veneers. On lower slopes, soils often develop on morainal deposits. Dominant soils include Brunisols (well to imperfectly drained mineral soils with partial horizon development) and Ferro-Humic Podzols (characterized by low base saturation, low pH, high organic carbon, and a high concentration of iron and aluminum compounds).

Hydrology/Surface Water Quantity

Regional and local surface water quantity characteristics were determined from data collected from specially installed hydrometric stations, used in conjunction with a regional analysis prepared for long-term hydrometric data from Water Survey of Canada hydrometric stations. Analysis reveals a clear difference in hydrologic patterns between the Teigen Creek/Treaty Creek drainages and the Unuk River/Sulphurets Creek drainages. With the exception of the main stem of Treaty Creek, the amount of runoff from the Teigen Creek/Treaty Creek drainages is on average nearly 40% lower than from the Unuk River/Sulphurets Creek drainages. This difference reflects not only differences in local climate patterns between the two geographically distinct areas, but also the strong influence of glaciers on surface water volumes for the Sulphurets Creek drainages.

The monthly distribution of flow tends to be concentrated in the open water season (May to October), with less than 20% of the annual flow occurring from November to April at a majority of the regional stations. During the open water season, the distribution of flow depends on the timing of the freshet and the balance between the volumes of water released during the freshet and the volumes of water resulting from fall rains or glacial meltwater. Smaller regional watersheds with glaciers show a higher proportion of flow during July and August compared to the larger regional watersheds with a smaller glacier percentage. This pattern is also evident in the Project area, especially for the Sulphurets Creek watersheds.

Groundwater Quantity

Groundwater conditions correspond with the mountainous, wet environment that comprises the Mine Site and the PTMA. Groundwater gradients are high, driven by heavy rainfall and recharge at higher elevations in the mountains. Valley bottoms are discharge zones, with groundwater levels near or above (artesian) ground surface. Discharge zones also exist along valley walls in the Mine Site, where seeps of acidic water have been observed (with pH readings as low as 2.5). Groundwater levels tend to be deeper at high elevations (i.e., from 6 m to 33 m below surface) and show more seasonal variation (from 1 m to about 15 m), whereas groundwater levels in the valley bottoms are generally shallow and show less seasonal variation. Bedrock aquifers are confined (i.e., groundwater is at pressures greater than atmospheric pressure). Unconfined aquifers are limited to the glacial deposits in the valley bottoms.

In the Mitchell Valley, poor quality water at the toe of the glacier is thought to be affected by groundwater that has contacted mineralized rock (i.e., it has a discharge quality similar to that in the springs/seeps). Groundwater elevations in wells installed in overburden (comprised of glacial till) in the Mitchell Valley bottom are similar to the creek bed elevation, and show little annual variation (less than 1 m), suggesting a hydraulic connection between groundwater and surface water. Groundwater elevations in wells screened in bedrock are higher than wells screened in overburden, indicating upward hydraulic gradients.

Groundwater recharge is considered to be higher in the mountainous areas (reflecting the orographic effect). For modeling purposes, the recharge rate is estimated at 218 mm/a, or 13% of the mean annual precipitation of 1,650 mm where ground elevation is more than 1,300 masl, compared to 115 mm/a (or 7% of mean annual precipitation) where elevation is less than 400 masl. Beneath glaciers and snow pack, recharge is estimated at only 40 mm/a (2.4% of mean annual precipitation) because of frozen ground conditions.

Surface Water Quality

The hydrological regime is an important determinant of stream water quality in the Project region. Typical local streams experience a low-flow period between November and April, and higher flows between May and October associated with freshet, summer glacial melt, and fall heavy rain events. The hydrological regime affects water quality in two ways:

- increased flows during freshet, glacial melt, and heavy rainfall events dilutes concentrations of major ions and total dissolved solids

- increased sediment load and transport during high-flow periods leads to increased concentrations of TSS and particle-associated metals.

Streams near the mine site and PTMA have distinct surface water quality. ML due to naturally occurring ARD is associated with total and dissolved metal concentrations in Mitchell and Sulphurets creeks that are frequently higher than levels set in BC water quality guidelines for the protection of freshwater aquatic life. The high suspended sediment load, low concentrations of bioavailable nutrients and high concentrations of total and dissolved metals identified in Mitchell and Sulphurets creeks and the Unuk River are likely contributing factors to the poor productive capacity of mine site streams. The lower suspended sediment load, increased concentrations of bioavailable nutrients, and lower concentrations of total and dissolved metals identified in the Snowbank, Teigen, Treaty and Bell-Irving watersheds are likely contributing factors to the greater productive capacity of PTMA streams relative to the mine site.

GROUNDWATER QUALITY

Groundwater quality at the mine site is heavily influenced by the sulphide ore deposits. Groundwater is acidic near to, and within, the mineral deposits, with pH measurements as low as 2.5 in seeps along the valley walls of Mitchell Creek. Concentrations of certain metals are elevated in groundwater throughout the mine site, and are particularly high near and within the mineral deposits. Metals with elevated concentrations include iron, aluminum, copper, chromium, lead, manganese and zinc. Groundwater in the Mitchell Valley is not suitable for human consumption or the sustenance of fresh water aquatic life.

Dissolved metals concentrations are generally low in the PTMA. The water is fresh (low salinity) with neutral to slightly alkaline pH, ranging from 7.4 to 8.8.

FISHERIES

The baseline fish and aquatic habitat study area encompasses two major watersheds that include the Unuk and Bell-Irving rivers. The north and west areas of the Project are situated within the Unuk River watershed, which crosses into Alaska and discharges into Burroughs Bay and eventually the Pacific Ocean. There are eight assessed sub-watersheds within the Unuk River watershed, in addition to the main stem of the Unuk River. The eastern area of the Project is situated within the Bell-Irving River watershed, which discharges into the Nass River. There are eight assessed sub-watersheds within the Bell-Irving River watershed, in addition to the main stem of the Bell-Irving River. There is one assessed sub-watershed within the Bowser River watershed (Scott Creek), in addition to the main stem of the Bowser River.

There is a 200 m long cascade in Sulphurets Creek, approximately 500 m upstream of the confluence with the Unuk River. Dolly Varden are present in Sulphurets Creek below the cascade, but no fish species are present above the cascade, in areas around the mine site. No salmon species are present within Sulphurets Creek.

Dolly Varden is the only species present in North Treaty and South Teigen creeks within the footprint of the proposed TMF in the Bell-Irving watershed. Dolly Varden, bull trout, mountain whitefish, and rainbow trout are present in South Teigen Creek, downstream of a 2.5 m high falls and outside of the TMF footprint. Dolly Varden dominate the species composition (95%) downstream of the falls in the lower reach of South Teigen Creek. No salmon species have been observed in South Teigen, North Treaty, or Tumbling creeks.

AQUATIC HABITAT

Sediments in the area downstream of the mine site (in Mitchell Creek and Sulphurets Creek) are of poor quality. These sediments are often inhospitable, with low nutrient availability (total organic carbon, nitrogen, and phosphorus), relatively coarse sediment structure that limit the range of available habitat for benthic invertebrates, and metal concentrations that are frequently higher than sediment quality guidelines. Surveys of primary producer (periphyton) and benthic invertebrates in the creeks downstream of the mine site revealed low standing stocks (biomass and density) and low diversities (richness and Simpson's diversity) of the aquatic communities, which is consistent with both poor water quality and sediment quality.

Sediment quality in the PTMA is generally better than downstream of the mine site, but metal concentrations are often elevated above sediment quality guidelines. Some areas, particularly those downstream of the wetlands (e.g. South Teigen Creek), had relatively high organic carbon content and favorable particle size distributions that would provide a better range of suitable habitat to support more diverse benthic populations. There are some areas that support more abundant and diverse aquatic communities (e.g. Teigen Creek), while other areas have periphyton and benthic invertebrate communities that are less abundant and less diverse (e.g., Treaty Creek).

TERRESTRIAL ECOSYSTEMS-GENERAL REGIONAL CHARACTERIZATION

The Project is situated within the Skeena Mountains Ecoregion, the Boundary Ranges Ecoregion, and the Nass Ranges Ecoregion. Towards the coast, the Boundary Ranges consist of extensive ice fields, capping granitic intrusions remnant of the Coast Range Arc, and are dissected by several major river valleys, including the Nass River. Inland and east of the Boundary Ranges lies the Skeena Mountains Ecoregion, which consists of high rugged mountains and a moist, coast/interior transition climate, supporting many glaciers. The Nass Ranges Ecoregion, with a climate somewhat transitional between coastal and interior regimes, is a mountainous area situated west of the Kitimat Ranges (which are located south of the Project).

A wide range of topography and vegetation communities occur within the regional study area (RSA) and local study areas (LSAs) defined for the purposes of assessing terrestrial ecosystem effects. These include low-elevation wetland and shrub-dominated riparian and floodplain ecosystems, low- and intermediate-elevation forests, subalpine and alpine meadows, and sparsely- to non-vegetated rocky and glaciated terrain. Many of these ecosystems provide valuable habitat for wildlife, as well as economically important forest and non-timber forest resources.

Locally and regionally, six Biogeoclimatic Ecosystem Classification (BEC) units are present, four of which are forested units, with the other two being undifferentiated alpine-parkland units. These Biogeoclimatic Ecosystem Classification units include the Boreal

Altai Fescue Alpine unit, which is most widespread in the RSA, as well as the Coastal Mountain-heather Alpine, Coastal Western Hemlock, Engelmann Spruce–Subalpine Fir, Interior Cedar Hemlock and Mountain Hemlock units. Nearly half (46%) of the RSA consists of non- and sparsely-vegetated ecosystems, while 26% consists of forested ecosystems, and 21%, of shrub-dominated ecosystems (including avalanche ecosystems). Glaciers and permanent snow/ice comprise approximately 22% of the RSA. Of the forested area, 66% is mapped as mesic forest, followed by moist and wetter forests (12% and 11%, respectively).

Twelve ecosystems (six terrestrial and six wetland types) that have been blue-listed or red-listed by the BC Conservation Data Centre have been identified within the RSA and LSA. Of the 38 rare individual plant species that were observed within the LSA, most were found at high elevations in the Sulphurets Creek watershed. The 38 rare species include 27 lichens, nine vascular plants and two mosses.

WILDLIFE SPECIES

Mature forests, wetlands, alpine areas, and riparian forests provide high-value habitat to a diverse wildlife community. Common species or groups that occur in the RSA include ungulates (e.g., moose and mountain goat), omnivores/carnivores (e.g., grizzly bear, black bear and wolves), furbearers (e.g., fisher, marten and wolverine), hoary marmots, bats, birds (forest birds, raptors and waterfowl), and amphibians (e.g., Columbia spotted frog and western toad). Forest harvesting within the RSA has been minimal compared to many other areas in BC, due to the remoteness of the area and the relatively poor productivity of the forests, so that the wildlife habitats found in the majority of the wildlife RSA are essentially undisturbed.

Moose

Moose are common throughout BC's forested areas with an estimated population size of 170,000 animals. Habitat suitability modeling and winter aerial surveys identified moose habitat in the wildlife RSA. Winter habitat has been identified as critical for maintaining moose populations and habitat modeling focused on this season. The majority of good quality winter habitat for moose occurs along river valleys within the interior survey area on the eastern side of the RSA, including the Bell-Irving River, Treaty Creek, Snowbank Creek and Teigen Creek, and also surrounding Bowser Lake. A smaller amount of moose habitat occurs in the western, coastal-influenced part of the RSA, along the Unuk River.

Baseline aerial moose surveys in the winter of 2009 revealed that the density and number of moose (adjusted for sightability) was higher in the eastern interior area of the RSA, near the PTMA, Treaty Creek, Bell Irving River, and Bowser Lake (0.59 moose/km^2; 198 moose) than in the western coastal area, near the mine site and Unuk River (0.27 moose/km^2; 33 moose). A lower male to female ratio was observed in the interior area, which is indicative of harvest pressure on males where access to high-quality moose habitat is available from Highway 37 along the Bell-Irving River and along forestry roads near Bowser Lake. The regional moose population is currently vulnerable.

Mountain Goats

In 2000, the total number of mountain goats in BC was estimated at approximately 50,000 individuals, of which between 16,000 and 35,000 occur within the Skeena Region. The most suitable year-round goat habitat in the RSA occurs in the eastern RSA along the Snowslide Range, and in the western RSA around John Peaks to the west of the Mine Site. Within the LSA, suitable habitat was identified in the mine site and southeast of the TMF. Summer surveys in 2008 observed 230 goats in 62 groups in the RSA. Winter 2009 survey observed 178 goats in 69 groups in the RSA. Goats were observed near the mine site during both the winter and summer surveys. In the PTMA, goats were observed on the Snowslide Range (i.e., the mountain range between the PTMA and the Bell-Irving River). In addition, a potential mineral lick was identified in the valley between the Sulphurets and Kerr pits. An additional mineral lick was observed during baseline surveys for the Brucejack Mine on the Snowslide Range, which encountered slightly higher numbers of goats and groupings.

Grizzly and Black Bears

Grizzly bears are found throughout BC, from sea level and river-valleys to alpine regions. BC contains more than 50% of the Canadian population of grizzly bears, with an estimated 13,800 grizzlies in the province. Grizzly bears are considered a species of special concern by the Committee on the Status of Endangered Wildlife in Canada and are blue-listed in BC. Habitat suitability modeling revealed that overall, between 8% and 38% of habitat within the RSA was identified as Moderately High and High rated habitat for spring (27%), summer (38%), and fall. In addition, 5% of the LSA was identified as suitable denning habitat for grizzly bears, particularly in the PTMA. The area near the proposed TMF and TCAR has also been identified as a candidate grizzly bear Wildlife Habitat Area. Based on baseline studies in 2008 and 2009, the superpopulation (i.e. the total number of grizzly bears that used the RSA during the course of the studies) was estimated to include 31 females and 27 males, for a total of 58 bears.

Black bears are common and widespread in BC. The population estimate in 2001 was between 120,000 and 160,000 in the province, with highest densities along the coast, including within the wildlife RSA. During grizzly bear DNA baseline study, black bear hairs were collected incidentally. Black bears were detected throughout the RSA and LSA along all river drainages, particularly along the Unuk, Bell-Irving and Bowser rivers; and near Bowser Lake, and in the Treaty and Teigen creek valleys. In addition, black bears were the species most frequently observed incidentally in the LSA and RSA.

Furbearers

An evaluation of the BC Fur Harvest Database identified 14 furbearer species that were harvested in areas within and surrounding the RSA. The most commonly trapped species included American marten, American beaver, and red squirrel. Trapped species also include the provincially blue-listed fisher and the federally listed wolverine. American marten has historically been the most frequently harvested and most valuable component of the regional fur harvest. The majority of the forested habitat within the RSA was modeled as highly suitable winter habitat for marten. Within the RSA, continuous blocks of highly suitable habitat were distributed across low elevation within all major watersheds, particularly in mature forests along the Unuk River watershed. Over a quarter of the LSA was identified as highly suitable winter habitat for marten, including most of the forest habitat within the TMF and the low-elevation older forests along the Coulter Creek and Treaty Creek corridors. During wildlife baseline studies in 2008 and 2009, nine furbearer species or their sign were observed. The most frequently observed species and/or sign were black bears, red squirrel, and marten.

Small Mammals and Groundhogs

Small mammals are an important prey source for predatory birds and other mammals. Trapping surveys were conducted in the LSA in 2008 and 2009. Over the two-year baseline study, seven small mammal species were identified in the LSA, none of which are of conservation concern in BC. Species observed include Keen's mouse, Northern red-backed vole, meadow vole, meadow jumping mouse, Cinereus shrew, dusky shrew and Nearctic brown lemming. Productive habitats for small mammals were identified within low elevation riparian areas and adjacent coniferous forests.

Field studies of hoary marmots and Arctic ground squirrels conducted in 2008 and 2009 did not detect Arctic ground squirrels, but marmot colonies were distributed throughout the alpine in both the mine site and PTMA, with the highest densities observed in alpine areas (e.g., Snowslide Range) near the PTMA (average 0.62 colonies/km^2), surrounding the proposed TMF. The mine site is characterized by steep and rugged coastal mountain terrain, which is less suitable marmot habitat than occurs in the PTMA, which has larger areas of alpine meadow and gentler mountain topography.

Bats

Nine bat species potentially occur within the LSA, two of which were categorized as likely to occur—little brown myotis and Western long-eared myotis. The other seven species were categorized as possibly occurring—California myotis, Keen's long-eared myotis, northern long-eared myotis, long-legged myotis, Yuma myotis, silver-haired bat and big brown bat. Four of these nine species are of provincial or federal conservation concern—northern long-eared myotis, Keen's long-eared myotis, silver-haired bat and little brown myotis. Little brown myotis and western long-eared myotis were observed mainly within riparian habitat. The most important habitat features for bats are cave-based hibernacula, typically associated with karst (limestone) topography. The only area in the LSA with exposed limestone is located in McTagg Creek, extending south to Sulphurets Creek.

Birds

During 2008 and 2009 baseline studies, 93 bird species were detected—eight raptor species, 25 wetland bird species and 60 forest and alpine bird species. Raptors include hawks, falcons, owls and other birds of prey. Wetland birds include ducks, geese, shorebirds, and other bird families associated with water bodies. Forest and alpine birds include songbirds, hummingbirds, woodpeckers and game birds in terrestrial areas.

Eight raptor species were recorded in the RSA, including bald eagles, golden eagles, northern goshawks, ospreys, red-tailed hawks, merlins, rough-legged hawks and Swainson's hawk. In BC, the rough-legged hawk is blue-listed and the Swainson's hawk is red-listed. In addition, the northern goshawk *laingi* subspecies is red-listed in BC and designated as threatened under the Canada *Species at Risk Act* (SARA 2002). It is unknown if the northern goshawks observed during baseline surveys are of the *laingi* subspecies. Two raptor nests were observed, both in riparian areas.

Twenty-five species of wetland bird were identified during the 2008 and 2009 baseline surveys. Three species identified in the RSA are of regional or provincial conservation concern: harlequin duck (provincially ranked as vulnerable during the non-breeding season), surf scoter (which is blue-listed and provincially ranked as vulnerable during the breeding season), and trumpeter swan. Harlequin ducks were observed on the Bell-Irving River and along Teigen Creek during the spring. A group of seven surf scoters was observed on Treaty Creek during fall 2008, and trumpeter swans were detected along Treaty Creek and on Border Lake. Areas with high species diversity during the breeding period were identified in wetland complexes associated with the confluence of Teigen Creek and Bell-Irving River, and along Treaty and Todedada creeks. In contrast, the habitat associated with the mine site and its drainages does not appear to provide good breeding habitat for most wetland species.

Sixty forest and alpine bird species were observed in the RSA in 2008 and 2009. The greatest richness of species, highest numbers of individual birds and highest diversity of birds were recorded within the proposed TMF, along the CCAR corridor adjacent to the Unuk River, and near Bowser Lake. The olive-sided flycatcher, which is federally listed as threatened (Schedule 1), was observed within the RSA adjacent to Unuk Lake. Nine nests belonging to five different species were observed during field surveys. Seven nests were located in the mine site, and two near Teigen Creek. The five species with confirmed nests were yellow warblers, dark-eyed juncos, Swainson's thrush, American three-toed woodpecker and red-breasted sapsucker.

Amphibians

The western toad is a federally listed species of special concern that is protected under Schedule 1 of the SARA. In British Columbia it is considered secure but it is afforded protection under the *Wildlife Act*. During 2009, three western toad breeding sites were observed, all of which were located outside of the LSA in ponds at low elevation, in shallow open water, with an open canopy, and warm water temperatures. Two toad breeding sites were found on West Teigen Lake, and a third at low elevation on the lower reaches of Teigen Creek, near the confluence with the Bell-Irving River. Other breeding sites likely occur in the RSA, though no high-quality potential sites were identified within the Project footprint or LSA, although moderately suitable habitat is present.

Two additional amphibian species were observed within the RSA near Teigen and Treaty Creeks—Columbia spotted frogs and wood frogs. Neither of these two species is of conservation concern.

Species at Risk

Forty listed species either occur or could potentially occur within the RSA and LSA, based on species distribution maps. Five species are listed in Schedule 1 of *SARA* that are confirmed present or are likely to occur. Western toad and olive-sided flycatcher were observed during baseline surveys, and rusty blackbird and common nighthawk likely occur. The northern goshawk *laingi* subspecies occurs in coastal BC, mainly on islands. Although northern goshawks were observed during baseline surveys, it is unknown whether they were the *laingi* subspecies, or the *atricapillus* subspecies, which is not at risk. However, for the environmental assessment, northern goshawk *laingi* were considered to likely occur in the RSA or LSA (Rescan 2013).

20.2.2 ECONOMIC, SOCIAL, AND CULTURAL SETTING

GOVERNANCE

There are five levels of governance in the area of northwestern BC where the Project will be developed. Municipal, regional, provincial and federal bodies comprise the non-Aboriginal leadership, while Aboriginal communities have their own governing bodies.

The Project is situated in the Regional District of Kitimat-Stikine, and Electoral Area A of the Bulkley Nechako Regional District. Local communities include municipalities, Nisga'a villages, Indian reserves, and unincorporated settlements. Municipal governance only exists for the District of Stewart, the City of Terrace, the Village of Hazelton, the District of New Hazelton and the Town of Smithers. The remaining communities that are not administered by Aboriginal bodies (Dease Lake, South Hazelton, Bell II, Meziadin Junction and Bob Quinn Lake) are unincorporated, and are governed by the regional district in which they are situated.

For Aboriginal communities, the base level of governance is the Nation or Band, and they may be further represented by a multi-party council. Nisga'a communities include the villages of Gitlaxt'aamiks (New Aiyansh), Gitwinksihlkw (Canyon City), Laxgalts'ap (Greenville), and Gingolx (Kincolith). Populated Indian reserves include: Gitanyow 1, five Tahltan communities (Telegraph Creek 6 and 6A, Guhthe Tah 12, Dease Lake 9 and Iskut 6) and five Gitxsan Nation communities (Gitwangak, Gitsegukla, Gitanmaax, Glen Vowell, and Kispiox). The Skii km Lax Ha reside in the Hazelton area.

ECONOMIC SETTING

Economically, the Project region has been dependent upon timber and minerals for well over 100 years. The majority of non-Aboriginal communities in the region were initially established to serve natural resource activities such as the mine operation near Cassiar, Stewart, Smithers, and Bob Quinn Lake. To date, the region's economic and social diversity has been constrained by limited access and infrastructure, lengthy distances, remote and small communities which provide limited labour or services, and long winters. Investment within the region has fluctuated based on the strength of the forestry and mining industries, global commodity prices and the value of the Canadian dollar.

Forestry, fishing, and coal mining were the key economic drivers of northwestern BC through the 1950s to the 1980s. The BC government pursued a policy of industrial resource development that historically saw rapid community growth. Within the region, this helped establish local economies and an experienced, if modest, labour force focused on natural resource extraction. Typically, the region's communities tend not to be economically diverse, so they are highly sensitivity to resource demand fluctuations. With the downturn in the forestry industry over the past 10 years, the majority of sawmills and pulp mills in the region are currently closed. Closures of mines in the area, such as Eskay Creek Mine and Golden Bear, have also affected the resource sector. Nonetheless, today, the economies of local communities continue to be largely resource-based, and continue to focus on supporting these sectors in the region.

Transportation challenges throughout northwestern BC are primarily due to the mountainous topography, which restricts development of transportation networks primarily to valley bottoms. The existing transportation network, including road, rail, and port facilities, supports an economy focused on exporting its natural resource to southern and international markets. Highway 16 and highways 37 and 37A act as the primary transportation corridors. Highway 37 is the only road between Gitwangak (Kitwanga, at the junction with Highway 16) and the Yukon Territory. All highways in the region are paved, except for small sections of Highway 37 north of Iskut and Highway 51 to Telegraph Creek. Terrace and Smithers have major airports capable of handling jets, while Stewart, Bob Quinn, Dease Lake, Iskut, and Telegraph Creek have smaller airstrips. The CNR rail line connects the Port of Prince Rupert to the rest of North America via Prince George, running along the Highway 16 corridor through the communities of Terrace, Hazelton, New Hazelton and Smithers. Mobile cellular phone coverage is limited to the larger communities along Highway 16.

Overall, the economy in northwestern BC is gradually becoming more diversified. Newer industries that have become important to the region on recent years include energy production (including hydroelectric power generation) and tourism. In some communities, employment levels have increased in the public service, sales and service, tourism, transportation, and mineral exploration sectors. Employment sectors in local Aboriginal communities now include significant sales and service, mineral exploration, labour and government administration components. There are recent signs that the population decline may be reversing.

Today, the mining industry continues to provide an important source of employment in the region, supplying an estimated 30% of jobs for communities along Highway 37 in recent years.

SOCIAL SETTING

Recent economic changes have led to a general decline in the overall region's population over the past decade or more, largely due to the loss of jobs (e.g., mine closures), particularly among non-Aboriginal communities. This decline is especially evident in Stewart.

The considerable distances between communities exert a key influence on the social, economic and heritage environment of regional residents. It is common to travel two or more hours between communities. Isolation may also be exacerbated by weather-related road closures. The larger centres of Smithers and Terrace, located in the south of the region, provide much of the region's goods and services. Transportation and communication options are limited, and long travel distances are often required to reach service centres. The sense of isolation in northern BC is further accentuated by the location of BC's major urban centres in the extreme south of the province.

Services vary considerably, depending on the size of the community, with smaller communities providing limited services and accommodations. Smithers, Terrace, and to a lesser extent Stewart, provide a broad range of services and supplies, including accommodation and support for mining and forestry activities. The number of recreation, health, social and educational services available within communities has dropped in parallel with the population. Regional hospitals are located in Terrace and Smithers, and there are well-equipped health clinics in both Dease Lake and Stewart, although existing services are contingent on stable populations. Primary and secondary education facilities exist in many communities, while educational facilities within certain Aboriginal communities do not extend beyond elementary school. Northwest Community College and Northern Lights College also offer facilities and programs for regional residents.

Aboriginal Groups

Several Aboriginal groups may be potentially affected by the Project. The PTMA is situated within the Nass Area, as defined by the NFA, but falls outside the NWA, and also the Nisga'a Lands owned in fee simple by Nisga'a Nation under the terms of the NFA, which came into effect on May 11, 2000. The Tahltan First Nation (as represented by the Tahltan Central Council) asserts a claim over part of the Project footprint. Both the Gitanyow First Nation (notably *wilp* Wiiltsx-Txawokw) and the Gitxsan Nation (as identified by the Gitxsan Hereditary Chiefs Office), including *wilp* Skii km Lax Ha, which represented itself separately in the EA process, have identified potentially affected interests within the broader region, notably downstream of the PTMA. The Skii km Lax Ha are claiming an area covering the mine site and PTMA.

Aboriginal people have a significant physical, cultural and historical presence within the Project region. In 2006, approximately 32% of the Regional District of Kitimat-Stikine's population was reportedly Aboriginal. Furthermore, the populations of most of the region's smaller communities, notably those located along the north-south corridor of Highway 37 and the east-west corridor near Highway 16, are predominantly Aboriginal. The decline in the forestry and fishing industries since the 1980s has negatively impacted Aboriginal communities, as reflected by high unemployment rates. The current socio-economic setting of the region's Aboriginal communities is now in the process of evolving again due to opportunities provided by the mineral industry and tourism.

Land Use Setting

The Project is located in an area of northwestern BC known as the "Golden Triangle", due to its high mineral potential and the occurrence of several gold projects in the region. For the past century, land and resource uses in the region have been largely driven by forestry, mining and mineral exploration, and this is still true today. A limited amount of commercial and non-commercial recreation also occurs in the region, including hunting, trapping, fishing, heli-skiing, hiking and camping.

Within the Project region, exploration projects were historically focused in areas between the mountainous Knipple Glacier and Eskay Creek areas. Placer claims are present in several areas, including in Mitchell and Sulphurets creeks. Two mineral developments have been active within the region since the 1990s, including the Eskay Creek Mine which operated between 1994 and 2008, primarily extracting silver and gold, and the Brucejack (Sulphurets) Lake underground development project, which ended in 1993. Developments associated with the former Eskay Creek Mine include an access road connecting Highway 37 to the Eskay Creek area, a mill site, and other support facilities and roads.

Limited timber harvesting has been carried out within the region, with former operation limited to areas in the Nass Timber Supply Area (Nass TSA) along the Bell-Irving River and Highway 37. Timber harvesting contributed to the establishment of Meziadin Junction, with most of the harvesting activities occurring to the south of the Project area. Cut blocks within and immediately surrounding the region have been limited in scale and focused on pulpwood. Logs are transported to Stewart for shipping to overseas markets, or trucked to Terrace and Smithers.

Sections of the RSA are associated with the traditional hunting activities of local First Nations communities. Archaeological evidence suggests that pre-contact hunting activities have occurred in areas throughout the RSA. Sections along the Bell-Irving River have been used for traditional hunting and fishing, and cabins belonging to the Skii km La Ha are located within the RSA. Subsistence and resident hunting and fishing has continually occurred from the time of European contact in the region through to modern times. Resident hunting within the RSA has typically focused on moose within Wildlife Management Unit (WMU) 6-21, and on black bear and grizzly bear within WMU 6-16 and 6-17.

Trapping for fur-bearing animals has also historically influenced land use within the RSA, with both Aboriginal and non-Aboriginal trappers. Cabins associated with trapping activities are located along the Unuk and Bell-Irving River valleys, and are also used for hunting and fishing purposes. Registered traplines have records dating back to 1985, though the areas were potentially used before that time. Three traplines in the area are held by Aboriginal trappers. Areas near Treaty and Snowbank creeks have also been used for guide outfitting and angling operation.

Recreation, both commercial and private, such as guided mountaineering, guided river rafting and heli-skiing, has occurred in various areas within the RSA. Only a limited number of commercial operators have targeted the terrain within the RSA, due to its ruggedness and remoteness. Difficult access to these areas means that encounters with other individuals is infrequent, and the sense of isolation is an important part of the experience offered to clients. Areas near the Bell-Irving River (such as the Snowslide Range and Treaty Creek) see higher use because they are easier to access from Highway 37. Additionally, the Unuk River is used for commercial rafting adventures, and is accessible from the Eskay Creek Mine road or upstream from Alaska. Recreational activities, particularly fishing and heli-skiing, have contributed to the establishment of outdoor lodges, including Bell 2 and Spey/Boundary Lodge. Such activities, however, are seasonal and of short duration, so there are no formal recreational trails, roads or other infrastructure outside of the aforementioned lodges.

Land Use Planning Context

The Project area is subject to the provisions of two land use plans—the Cassiar Iskut-Stikine Land Resource Management Plan (LRMP) and the Nass South Sustainable Resource Management Plan (SRMP).

Land management within the Cassiar Iskut-Stikine LRMP includes objectives intended to preserve the physical, aesthetic, and cultural characteristics of the region. The LRMP created 14 protected areas for which resource conservation is emphasized, and three of which are located within or adjacent to the land use RSA. Notably, the LRMP acknowledges the mineral and energy resource potential within the Plan area. Under the LRMP, exploration and development of mineral deposits, as well as construction of access roads, are allowable activities, excepting within Protected Areas. One Resource Management Zone (RMZ), the Unuk River RMZ, overlaps the LSA, including portions of the Coulter Creek access road. The management goals for the Unuk River RMZ are focused on preserving grizzly bear habitat and maintaining visual quality of the terrain from the Unuk River, while allowing for adjacent logging and mineral development.

Land management goals within the Nass South SRMP were developed in partnership with NLG, the Gitanyow First Nation, stakeholders, and government agencies, with the goal of guiding development and conserving environmental and cultural resources within the southern portion of the Nass TSA. The Nass South SRMP provides guidance on permitted land uses, and addresses sustainable management issues for land, water and resources, while aiming to facilitate economic opportunities. Mineral resource activity, timber harvesting, commercial recreation and tourism, guide outfitting, hunting, fishing, trapping and cultural land uses are all allowable activities.

20.3 Water Management

Water is a key component in the mining process in that it is required for, and affected by, mining activity. Over the life of a mine, waterbodies can by temporarily diverted, created, or drawn upon to allow mining activity to occur. Additionally, water acts as a transport medium for potential contaminants to be introduced to the receiving environment. Therefore, water must be managed for a variety of reasons including compliance with operation permits, smooth and uninterrupted operation of the mine, and minimization of effects on water quality and quantity in the receiving environment.

20.3.1 Overview of Water Management

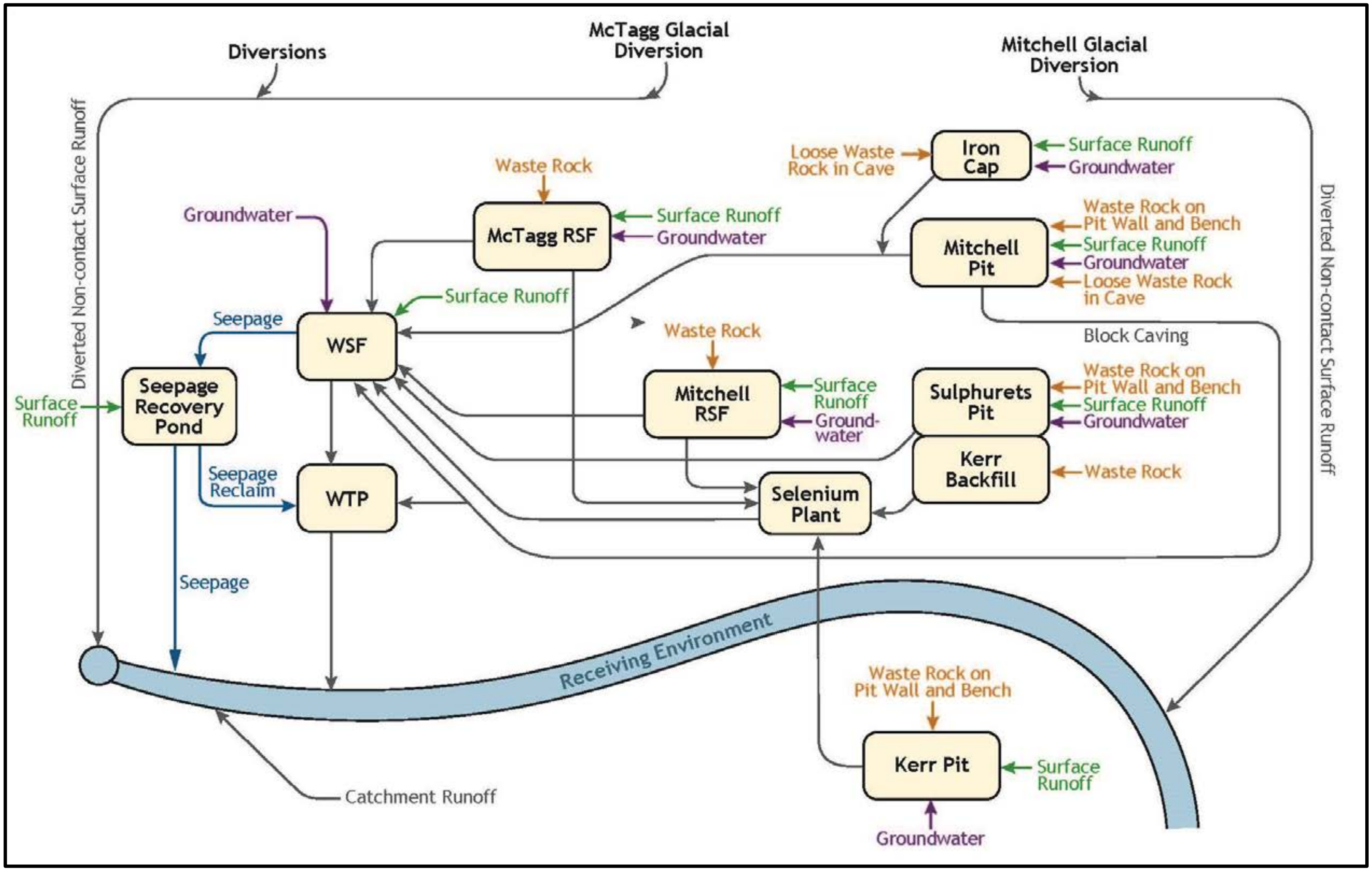
An extensive system of water management facilities will be constructed and maintained throughout the life of the Project to divert fresh (non-contact) water away from disturbed areas and to collect water that has contacted disturbed areas (contact water) for treatment before release into the environment. The mine site water management plans are laid out in five-year increments (shown in *Appendix 4-N* of the Application/EIS). Please refer to Section 18.2 of this PFS for details of the updated water management plans for the Project.

An overview of the water management plan for operations is included in Figure 20.1. The layout of the ultimate water management plan for the mine site is shown in Section 18.0, in Figure 18.5. These facilities will include:

- the WSF contained by a 165-m-high WSD, which will have a crest length of approximately 650 m (KCB 2012a)
- the HDS WTP to treat all contact water using a high density sludge water treatment process
- a Selenium WTP designed to treat 500 L of water from seepage collected from the base of the Mitchell-McTagg RSF, and/or Ker waste stored in the Sulphurets pit, or from other point sources such as the WSF
- the MDT and related inlet structures to divert clean water flows from the Mitchell Glacier and surrounding areas upstream of the proposed Mitchell pit and Mitchell block cave mine to the Sulphurets Creek drainage
- the MTDT and related dams and inlet structures to divert clean water flows from the McTagg Creek Valley away from the McTagg RSF and downstream mine facilities
- the Mitchell NPWDA to depressure the north wall of the Mitchell pit and to conduct surface contact water from the vicinity of the Mitchell Glacier around the Mitchell pit
- the MVDT to route water from the Mitchell NPWDA under the Mitchell Creek Valley to the WSF
- the Mitchell underground drainage tunnels to route water from the lower reaches of the Mitchell block cave mine to a point about 300 m below the HDS WTP where it will be pumped to surface
- secondary diversion ditches and pipelines implemented within the Mine Site during the operation phase to reduce contact water volumes and to direct open pit contact water and discharge from pit dewatering wells to the WSF.

These facilities are discussed in greater detail in the Project Description (Volume 4) in Rescan (2013).

Figure 20.1 KSM Project Mine Site Water Management at Operations



20.3.2 SUMMARY OF WATER MANAGEMENT PLAN

Seabridge is committed to a comprehensive Water Management Plan that applies to all mining activities undertaken during all phases of the Project. The main objective of the Water Management Plan is to regulate the movement of water in and around the area of the Project in order to ensure long-term environmental protection.

A separate, more detailed Environmental Effects Monitoring (EEM) Plan will also be developed in consultation with regulators and in compliance with the MMER (SOR/2002-222) and provincial discharge limits as a requirement of the permits and licenses under which the Project will operate.

OBJECTIVES AND TARGETS

The objectives of the Water Management Plan are to provide a basis for management of surface water on site including:

- diverting non-contact water around the Mine Site and PTMA
- protecting ecologically sensitive areas and resources and avoiding harmful impacts to aquatic life and wildlife habitat
- providing and retaining water for mine operation
- defining required environmental control structures
- collecting and treating contact water from the Mine Site to meet discharge requirements prior to release to the receiving environment.

The targets intended to optimize the Water Management Plan to achieve surface water objectives include:

- minimizing the production of contact water by implementation of best management practices and water diversion measures
- collecting and treating contact water where required in order to meet applicable water quality standards
- implementing and maintaining an on-site monitoring and control system to regulate surface water quantity and quality.

LEGISLATION AND STANDARDS

The Water Management Plan has been developed to satisfy guidelines and requirements specified in the following legislation:

- BC *Mines Act* (1996)
- *Fisheries Act* (1985)
- Canada *Water Act* (1985a)
- BC *Water Act* (1997n)
- BC *Water Protection Act* (1996o).

Actions to Avoid, Control, and Mitigate

A variety of diversion, collection, and treatment structures will be developed to manage surface water for the Mine Site and PTMA. Surface water management activities will consist of non-contact water diversion and contact water collection and treatment. By minimizing the amount of contact water that is produced on the Project site, surface water diversion reduces the volume of water that must be treated. Additionally, surface water diversion decreases the potential for erosion and sediment production by limiting the volume of water that enters a work area. The general water management measures that will be used during all phases of the Project can be found in detail in Volume 24 of Rescan (2013).

Mine Site

Water management structures and facilities will be constructed to separate contact water for treatment and to route non-contact water around the Mine Site to the environment. The main objective is to minimize the amount of non-contact water reporting to the WSF to reduce water treatment requirements.

Processing and Tailing Management Area

Water management in the PTMA is focused on the construction of diversion channels to control and divert water in the PTMA catchment area to either South Teigen Creek or North Treaty Creek.

Monitoring

Monitoring programs will enable Seabridge to measure the success of the management strategies and to identify where additional mitigation is necessary.

Several management plans and monitoring programs include components that will help ensure the long-term protection of the aquatic environment downstream of the Project. These management and monitoring programs include but are not limited to:

- Aquatic Effects Monitoring Program (AEMP)
- EEM Program
- Groundwater Management/Monitoring Plan
- Fish and Aquatic Habitat Management Plan
- Wetlands Management Plan
- Metal Leaching and Acid Rock Drainage Management Plan
- Tailing Management Facility Management and Monitoring Plan
- Water Storage Facility Management and Monitoring Plan
- Spill Prevention and Emergency Response Plan
- ML and ARD Management Plan
- Glacier Monitoring Plan

- Water Management Plan
- Construction Management Plan
- RSF Management and Monitoring Plan;

The AEMP includes water quantity, water quality, sediment quality, water toxicity, benthic invertebrate and fish assessments downstream of all mine infrastructure, including the WSF and TMF. The AEMP will monitor long-term effects (if any) of effluent decants or seepages to downstream areas.

20.4 WASTE MANAGEMENT

20.4.1 TAILING MANAGEMENT FACILITY MANAGEMENT AND MONITORING PLAN

Tailing produced after the mined ore has passed through a process of high pressure grinding, flotation, and leaching at the Treaty OPC will be transported by slurry pipeline to the TMF. Conventional flotation tailing will be stored in two cells, the North Cell and the South Cell. A separate tailing stream consisting of sulphide-rich CIL residue tailing, produced as a waste product in the cyanide leaching process, will be transported in a pipeline to a fully lined CIL Lined Pond located in the Centre Cell between the North and South cells, and will operate during the filling of the North and South cells. The TMF is designed to store 2.3 Bt of tailing produced over the 53-year mine life.

The TMF will ultimately consist of three storage cells retained by four compacted cyclone tailing dams, the North Dam, the Splitter Dam, the Saddle Dam, and the Southeast Dam. The tailing dams will be constructed to final heights of 218 m, 194 m, 168 m, and 239 m, respectively. Seepage from the tailing dams will be collected in seepage collection ponds constructed downstream of the tailing dams.

Requirements for the design, operation, and closure of TMFs on a mine site are legislated under the *Mines Act* (1996j) and are covered by sections of the Health, Safety and Reclamation Code (BC MEMPR 2008), including an updated Section 10 of the Code which was completed in 2016. In accordance with a Memorandum of Understanding between the BC Ministry of Energy, Mines and Petroleum Resources (now BC MEM and the Water Stewardship Division and Environmental Protection Division of the BC Ministry of Environment (now Ministry of Forests, Lands and Natural Resource Operations and Ministry of Environment, respectively), the *Mines Act* (1996j) regulations apply to tailing storage facilities unless a water licence or waste permit is required. It has been assumed that a waste permit may be required, but that a water licence will not be required for the TMF.

The Health, Safety and Reclamation Code (BC MEMPR 2008) applies to "major impoundments" and "major dams." A major impoundment is defined as an impoundment that has a maximum depth of material greater than 10 m at any point, or a maximum height of retaining dam or dike at any point that exceeds 15 m, or is a storage facility designed to contain more than 1 Mm^3 of fill, or any other impoundment or water management facility so declared by the Chief Inspector. A major dam is defined as a dam that is used to store and control water, slurry, or solids and that has a maximum height at any point that exceeds 15 m or that is between 10 and 15 m in height and has either a crest length that exceeds 500 m, a flood discharge rate that exceeds 2,000 m^3/s, or a reservoir capacity that exceeds 1 Mm^3, or any other dam so declared by the Chief Inspector (BC MEMPR 2008).

The tailing dams and associated seepage recovery dams proposed for the Project fall into the category of major dams.

LEGISLATION AND STANDARDS

BC's *Mines Act* (1996j) provides guidance and approvals for all activities on a mine site, from exploration through to development, production, closure, and reclamation. It also requires project proponents to obtain a permit approving the work system and reclamation program. To obtain this permit, a detailed Mine Development Plan and Reclamation Program has to be submitted to the Chief Inspector of BC MEM for approval.

Under the *Mines Act* (1996j), the Health, Safety and Reclamation Code (BC MEMPR 2008) requires proponents to provide:

- a mine plan showing the location of the TMF
- designs for major impoundments and dams developed in accordance with the criteria provided in the CDA Dam Safety Guidelines (2007)
- an operation, maintenance, and surveillance manual for major dams and impoundments prior to commissioning
- design for dam slopes that meet long-term stability criteria in accordance with the CDA (2007) Dam Safety Guidelines
- an Emergency Preparedness Plan for dams classified as High or Very High failure (or Extreme) consequence
- an annual dam safety inspection report
- the installation of a flood control spillway at closure
- a reclamation plan for tailing ponds and impoundment structures
- a dam stability monitoring program.

The Health, Safety and Reclamation Code (BC MEMPR 2008) requires proponents to provide plans for the prediction, and if necessary, prevention, mitigation, and management of ML/ARD. Details for ML/ARD management and monitoring are included in the Metal Leaching and Acid Rock Drainage Management Plan (*Volume 26, Section 26.14* of the Application/EIS).

The Health, Safety and Reclamation Code (BC MEMPR 2008) also requires proponents to provide plans for soil salvage and handling, erosion control, and soil contamination prevention. Details for soil management and monitoring are included in the Terrain, Surficial Geology and Soil Management and Monitoring Plan (*Volume 26, Section 26.13* of the Application/EIS).

The Environmental Management Plan for construction of the tailing dams, seepage collection dams, and water control structures is included in the Construction Management Plan (*Volume 26, Section 26.2* of the Application/EIS).

Plans for the control and diversion of water on the Mine Site, including the management of water around the TMF, are provided in the Water Management Plan (*Volume 26, Section 26.17* of the Application/EIS).

A set of detailed mine development and reclamation plans will be submitted at a later date as part of the Mine Plan and Reclamation Program Permit application.

Dam Stability Plan

Objective

The objective of the Dam Stability Plan is to construct and operate the tailing dams to provide safe and stable storage for processed tailing and CIL tailing residue and to construct and operate the seepage collection dams to provide safe and stable storage of seepage collected downstream of the tailing dams. The management and monitoring plan is intended to provide a basis for confirming design assumptions and for tracking performance during construction, operation, and after the mine closes.

Targets

Targets of the Dam Stability Plan include the following:

- prepare the foundation of dam sites;
- construct dams to meet short-term (during construction) and long-term stability criteria; and
- control surface and subsurface water.

Actions to Avoid, Control, and Mitigate

The tailing dams and the seepage collection dams have been classified based on the dam classification scheme presented in the CDA (2007) Dam Safety Guidelines. The classification scheme considers incremental losses associated with loss of life, environmental and cultural values, and infrastructure and economic values. The North dam, Saddle dam, and Southeast dam are classified as Extreme failure consequence. The Splitter dam is classified as High failure consequence. The seepage collection dams are classified as significant failure consequence.

Dam design flood and earthquake levels are selected based on the dam classification. The required minimum static factor of safety for dam design is 1.5 using peak frictional strength parameters and estimated operating pore pressures in the dam materials and underlying foundation soils. For earthquake conditions, the minimum required factor of safety is 1.0 based on pseudo static analysis, assuming 50% strength reduction in any uncompacted tailing deposits. Assuming full liquefaction of uncompacted tailing deposits, the minimum required post-earthquake factor of safety against sliding of the dam is 1.2.

Appropriate design criteria have been used for each of the dams (please refer to *Volume 26, Section 26.4.2* of the Application/EIS for details of the design criteria available in 2012. Please see Appendix H for the most up to date design criteria.

Details of dam construction, including material properties and construction specifications, are included in KCB (2012a) of the Application/EIS. Please see Appendix H for the most up to date construction details.

Landslides, debris flows, and frequent snow avalanches may occur in the East Catchment Valley and could affect dam stability, creek diversion structures, and mine personnel safety. Mitigation of slope hazards in the East Catchment Valley can be found in *Volume 26, Section 26.4.2* of the Application/EIS.

Please see *Volume 26, Section 26.4.2* of the Application/EIS for additional mitigation measures that will be implemented during the construction, operation, closure, and post-closure of the dam.

Monitoring

A monitoring program will be developed that will include requirements for inspection of dams and water control structures and procedures for instrumentation monitoring during the construction, operation, and closure phases. This information will be included in an operation, maintenance, and surveillance manual as required by Section 10.5.2 of the Health, Safety and Reclamation Code (BC MEMPR 2008). The manual will include principles for the safe operation of the TMF and will include details for monitoring, training, and inspections associated with dam safety (KCB 2012c).

The results of the monitoring program will be reviewed on a regular basis to measure the success of the management strategies, to compare the recorded data against design criteria, and to identify where design changes or additional mitigation may be necessary.

The tailing and seepage collection dams will be monitored during operation by piezometers to measure phreatic levels in the dam and foundation soils and settlement pins and to measure deformations of the structures. The monitoring instruments will be installed on representative sections of the dams. The final locations of the sections and instruments will be determined after the detailed design has been completed and prior to permitting.

A program of visual and instrumentation monitoring for the tailing dams and seepage collection dams may include:

- inspection of the dam crests and slopes for signs of cracking, slumping, settlement, seepage or piping;
- inspection of dam spillways for potential blockages due to snow avalanche, landslide, or rock fall debris;
- inspection of pumps and piping systems;

- inspection of surface water diversion channels and structures to identify sections where repairs may be required;
- inspection of diversion channels to identify damage or blockages attributable to snow avalanche or snow accumulation and to identify excessive erosion during spring runoff;
- installation of piezometers to measure pore water pressures in the dams and dam foundations;
- installation of survey monuments to measure dam fill settlement and surface displacement;
- monitoring of pond water levels;
- monitoring of tailing beach widths;
- monitoring of seepage downstream of the dams;
- monitoring of flows in the diversion channels; and
- monitoring of geohazards, in particular snow accumulation and avalanche potential.

A schedule for routine inspection and instrumentation monitoring will be developed at the time of mine permitting based on the mine construction and operation schedule. Threshold values (warning levels) for each instrument and response criteria will be established and included in an operational, maintenance and surveillance manual. A contingency measure for increasing dam stability could be to place additional fill to flatten the downstream slopes of the dams. Contingency measures for reducing net seepage losses from the tailing facility, if required, could be strategic deposition of tailing on the inside of the impoundment to reduce the seepage losses.

Geotechnical instrumentation installed at the tailing and seepage collection dams will be monitored during construction and operation. The instrument readings will be recorded twice per month during normal operating conditions. In addition to the above instruments, seepage from the dams will be monitored by weirs installed downstream of the dams.

Pond levels will be recorded monthly and used, in conjunction with pond filling curves, to plan the tailing discharge and the operation of the seepage collection ponds.

Flows in the diversion ditches will be recorded on a regular basis and will be used in assessment of the TMF water balance.

Inspection of the diversion channels will be carried out monthly. Clearing snow and debris may be necessary at critical channel sections up to four times each year.

Additional inspections of the dams and water control structures will be undertaken following extreme rainfall events, significant runoff events, or significant earthquake events.

The TMF will remain in operation after the end of mining operations until such time as the quality of the water stored in the impoundment reaches an acceptable level for discharge and reclamation is completed. The dam and associated facilities will require ongoing monitoring and maintenance to ensure dam safety and to meet regulatory requirements for dam safety.

In the event of temporary mine closure, visual inspection and maintenance of the dams, diversion channels, collection ditches, and spillways will be required.

Reporting

A review and evaluation of the visual and instrumentation monitoring data will be carried out regularly to identify the need for contingency measures to reduce seepage and/or to increase dam stability. The monitoring results and corrective actions will be included in the site documentation management system and reported to senior management and regulatory agencies as required.

An annual dam safety inspection report, describing the operation, maintenance, and surveillance of the tailing dams and water management facilities, will be prepared and submitted to BC MEMNG as required by Section 10.5.3 of the Health, Safety and Reclamation Code (BC MEMPR 2008).

Comprehensive dam safety reviews will be carried out periodically based on the consequence of dam failure for each individual dam. The frequency of review and reporting will be in accordance with the CDA (2007) *Dam Safety Guidelines*. The frequency of review is every 5 years for dams classified as Extreme consequence, every 7 years for dams classified as High consequence, and every 10 years for dams classified as Significant consequence.

The mine manager is ultimately responsible for mine operations. The engineering department, reporting to the Superintendent of Health and Safety, will be responsible to maintain the TMF monitoring records and generate reports at the required frequency.

20.4.2 TAILINGS ALTERNATIVE ASSESSMENT

A comprehensive Multiple Accounts Analysis (MAA) was conducted in order to evaluate numerous potential alternatives for the TMF. This analysis was carried out using the methods and processes outlined in the *Guidelines for the Assessment of Alternatives for Mine Waste Disposal* (the Guidelines) published by Environment Canada (2011) in order to meet the requirements of obtaining a Schedule 2 amendment under the MMER (SOR/2002-222). These regulations apply if a fish-bearing waterbody would be directly affected by a proposed TMF.

In order to address potential concerns, Seabridge was proactive in initiating a tailing management assessment two years prior to submitting the Application/EIS. Often a TMF assessment is conducted after the Application/EIS approval. The intent of this early action by Seabridge was to ensure that the siting of the TMF was done in a timely manner with appropriate consultation that allowed for the best environmental outcome.

The full MAA report is provided in Appendix K13 of this document. The following sections provide a summary of the methods and results of the alternatives assessment.

METHOD

The TMF alternatives assessment process involves seven steps to select a TMF site by a MAA process of systematic analysis and elimination. The main evaluative step in the MAA commences with the development of a multiple accounts ledger, which is an explicit list of all the potential adverse effects associated with each TMF alternative that generates a clear and measurable description of those effects. The seven steps of the MAA are outlined below:

- Step 1 – Identify Candidate Alternatives: identify preliminary TMF candidates near the Mine Site deemed feasible based on basic topographic, geologic, accessibility, precedence, and technical threshold criteria.
- Step 2 – Pre-screening Assessment: screen the number of potential TMF sites by applying a fatal flaw analysis to eliminate alternatives that are not feasible.
- Step 3 – Alternative Characterization: a non-evaluative characterization of the TMF alternatives not eliminated in the previous step.
- Step 4 – Multiple Accounts Ledger: systematically evaluate each TMF option based on the characterization parameters developed in Step 3 using a valuation system based on best professional judgment, and considering issues raised by Aboriginal groups; local, provincial, and federal government agencies; stakeholders; and Seabridge.
- Step 5 – Value-based Decision Process: conduct a final value-based evaluation to identify the preferred TMF candidate. This is done by scoring and weighting the indicators developed in Step 4 and applying a quantitative analysis to develop weighted merit ratings for each TMF candidate.
- Step 6 – Sensitivity Analysis: consider different value systems when weighting accounts, sub-accounts, and indicators.
- Step 7 – Document Process: transparently report on the TMF alternatives analysis process.

RESULTS OF THE KSM PROJECT TAILING MANAGEMENT FACILITY ALTERNATIVES ASSESSMENT

The results of the seven-step MAA process are summarized below (and presented in full in Appendix K13).

Step 1: Identify Candidate Alternatives

Seabridge conducted an initial screening of all potential tailing sites in a 50 by 50 km area surrounding the mine site. Threshold criteria were used by Seabridge to exclude sites in the 2,500 km^2 area that were not feasible due to basic topographic, accessibility/cost, and technological limitations.

The following threshold criteria were applied to determine reasonable potential TMF options for the Project.

- Exclusion based on topography – The region surrounding the Project is characterized by rugged terrain that limits the options for locating the TMF. Valley topography was a primary factor in excluding options. Many valleys are too steep or too small to qualify as appropriate options.
- Exclusion based on accessibility – Feasibility of the tailing disposal and storage sites varied based on relative accessibility associated with the mountainous topography rather than being primarily a function of distance from the mine site. High elevation regions and areas with limited or challenging access options were excluded as the technological challenges and/or cost of using these sites would be prohibitive.
- Exclusion based on technological limitations – Tailing disposal technologies considered as options included conventional impoundments, subaqueous or saturated storage, submarine storage, and in-pit tailing storage. Underground storage was not considered because underground mining will occur too late in the Project LOM to be of use for tailing storage.

Fourteen potential TMF candidate alternative sites were identified for MAA evaluation:

- Upper Treaty TMF
- West Teigen Lake TMF
- Bowser Lake TMF
- Segmented Bowser Lake TMF
- Knipple Lake TMF
- Ted Morris Creek Valley TMF
- McTagg Creek Valley TMF
- Sulphurets Creek Valley TMF
- In-pit Tailing Storage TMF
- Burroughs Bay Submarine Disposal TMF
- Scott Creek Valley TMF
- Combined Sulphurets Creek Valley and Ted Morris Creek Valley TMF
- Unuk Valley TMF
- Upper Treaty Creek Valley TMF.

Step 2: Pre-screening Assessment

The following pre-screening criteria were applied to each of the initial 14 TMF candidate alternatives:

- Do government policies recommend against specific deposition methods?

- Are geological foundations insufficient for safe construction and operation of containment dams?
- Do water management issues preclude safe operation of the TMF?
- Will the TMF result in negative life of Project economics?
- Does the proposed facility have insufficient capacity for the entire proposed mine life?
- Are engineering issues prohibitive given current technology?
- Do geological hazards preclude the safe operation of the TMF?

Of the above 14 candidate alternatives for potential TMFs, four potential tailing management alternatives—one individual site and three combinations of two sites—met all the TMF siting pre-screening criteria:

- Upper Treaty TMF (1)
- Scott Creek Valley TMF combined with West Teigen Lake TMF (11 and 2)
- Unuk Valley combined with West Teigen Lake TMF (13 and 2)
- Upper Treaty Creek Valley combined with West Teigen Lake TMF (14 and 2).

Step 3: Alternative Characterization

Step 3 expands the scope and detail of the characterization of each candidate alternative using project-specific criteria as recommended in the Guidelines (Environment Canada 2011). The criteria fall under the following four broad categories, referred to as "accounts":

- Environmental characterization – This account describes the local and regional environment surrounding each proposed TMF. Elements such as climate, geology, hydrology, hydrogeology, water quality, and potential effects on fish and wildlife are considered.
- Technical characterization – This account describes the engineered elements of each alternative such as storage capacity, dam size and volume, diversion channel size and capacity, dumping techniques, haul distances, seepage dam requirements, tailing discharge methods, pipeline grades and routes, closure design, discharge and/or water treatment infrastructure, and supporting infrastructure such as access roads.
- Project economic characterization – The account describes the life of Project economics. All aspects of the mine waste management plan are considered including investigation, design, construction (inclusive of borrow development and royalties where applicable), operation, closure, post-closure care and maintenance, water management, associated infrastructure (including transport and deposition systems), compensation payments, and land use or lease fees.
- Socio-economic characterization – This account describes how each proposed TMF may impact commercial land users and Aboriginal interests. Elements

considered include characterization and valuation of land uses, cultural significance, presence of archaeological sites, and employment and/or training opportunities.

Each account considers short- and long-term issues associated with construction through operation, mine closure, and, ultimately, post-closure maintenance and monitoring. Detailed characterization data and summary tables are provided in the full MAA report (Appendix K13).

Step 4: Multiple Accounts Ledger

As per section 2.5 of the Guidelines (Environment Canada 2011), a multiple accounts ledger identifies criteria from the alternative characterization that differentiate alternatives so that TMF options can be evaluated relative to one another. This ledger is derived from the characterization data. The multiple accounts ledger consists of two elements: sub-accounts (i.e., evaluation criteria) and indicators (i.e., measurement criteria).

To allow the accounts and sub-accounts to be measured and compared, the indicators must be measureable. As per the Guidelines, a six-point scale was used, because it provides sufficient range to differentiate without being overly detailed, and it is an even number scale that eliminates the tendency to select the "middle-of-the road" value. Qualitative (i.e., value-based) scales were developed (e.g., very high, high, low, etc.) when precise measurability was not possible, as per the Guidelines (Environment Canada 2011). Value scales were developed to have the following characteristics:

- *Reliable* – External reviewers must be able to rate an alternative according to the value scale and assign the same score.
- *Value relevant* – The value scale must be directly relevant to the indicator being scored.
- *Justifiable* – Any external reviewer should reach the conclusion that the value scale is reasonable and representative.

Each indicator has a scoring descriptor (table or textual), as described in the full report (Appendix K13).

Step 5: Value-based Decision Process

The value-based decision process involves the creation of scoring and weighting scales for all relevant criteria (account, sub-account and indicators), set previously in the multiple accounts ledger, and applying them to the four alternatives. Values weighting was done on a six-point scale with six being the most highly valued and one the least. This value-based ranking methodology was done in concordance with the Guidelines (Environment Canada 2011) in order to differentiate the benefit or loss associated with each site. The weighting scales are relative, not absolute, within a particular account or sub-account. As such, these weightings are only directly comparable within a particular account or sub-account and not among accounts or sub-accounts. Please see the full report (Appendix K13) for a full explanation of weighting methodology.

The base case weighting of each of the main accounts was that the environmental account rated the highest, the technical and socio-economic accounts given equal median weights and the economic account the lowest.

Using the weightings combined with the indicator scores derived from the characterization data, as described in Step 4, a qualitative score for each of the candidate TMF alternatives was calculated. Calculation methodology followed the Guidelines (Environment Canada 2011), and full calculation tables are provided in the full report (Appendix K13). The result of these calculations, i.e., the results of the MAA for the Project TMF alternatives assessment, is shown in Table 20.1.

Table 20.1 KSM Project Tailing Management Facility Multiple Accounts Analysis Results

	Upper Teigen/ Treaty	Scott Creek Valley-West Teigen Lake	Unuk Valley-West Teigen Lake	Upper Treaty Creek-West Teigen Lake
Base Case	4.5	2.5	2.4	2.2

The value-based MAA decision process result indicated that the Upper Treaty TMF is the most appropriate TMF alternative (i.e., resulting in the highest value from the MAA process). The remaining three sites are significantly less preferable, and roughly equivalent to each other.

Step 6: Sensitivity Analysis

A sensitivity analysis was performed for the Project TMF alternatives assessment according to the Guidelines (Environment Canada 2011). For the Project TMF MAA, the following sensitivity analyses were performed:

- all accounts were weighted equally
- only environmental and socio-economic accounts were considered
- only the environmental account was considered
- only the technical account was considered
- only the Project economics account was considered
- only the socio-economic account was considered
- all accounts and sub-accounts were weighted equally
- all accounts, sub-accounts, and indicators were weighted equally
- downstream fisheries and water quality indicators sub-accounts were weighted more significantly
- downstream fisheries sub-account was weighted more significantly.

The result of all the sensitivity analyses that were conducted was that the Upper Treaty TMF alternative consistently emerged as the preferred option. Full analytical results are presented in Rescan 2012 (Appendix K13).

Step 7: Document Results

The final document (Appendix K13) fully documents the TMF alternative assessment process undertaken by Seabridge for the Project, in conjunction with consultation with Nisga'a Nation and Aboriginal groups, as well as local, provincial and federal government agencies. The TMF alternatives analysis results were presented to the environmental assessment Working Group for the Project on September 15, 2011, and again on March 29 and 30, 2012, in Smithers, BC.

20.4.3 BEST AVAILABLE TAILINGS TECHNOLOGY ASSESSMENT

Seabridge has recently completed as assessment of tailing technologies, tailing facility locations, and management practices. The assessment was an update of the tailing alternative assessment that was completed as part of the Application/EIS and the Schedule 2 amendment process (see *Chapter 33 and Appendix 33-B* of the Application/EIS). The full details of the technological assessment can be found in KCB (2016).

This recent study was conducted in order to address the heightened awareness of tailing dam safety since the Mount Polley dam breach. An Independent Expert Engineering Investigation and Review Panel (the Panel) made recommendations regarding BAT and BAP as part of the review of the Mount Polley breach (MPC 2015). The recommendations of the Panel were considered and addressed in the Seabridge tailing technology report.

Thirty-one potential TMF locations were considered in the assessment, along with multiple technologies including filtered tailing. The result of the BAT study indicated that the Teigen-Treaty Cyclone Sand is the preferred TMF site and management strategy for tailing management at the Project.

Key findings of the 2016 BAT study include:

- the BAT has been selected for the site conditions
- the proposed design minimizes land disturbance, and potential impacts to water quality and fish
- the proposed design ensures both chemical and physical stability after mine closure
- the proposed design allows the site to regain its baseline characteristics soon after secession of mining activities
- economics was not a primary consideration in the evaluation of the best available tailings technology.

20.4.4 WASTE ROCK MANAGEMENT

The proposed management of waste rock is outlined in the *Rock Storage Facilities Management and Monitoring Plan,* which can be found in Volume 26 of the Application/EIS. The text below summarizes this management and monitoring plan.

Waste rock from open pit and underground mining operations that is not used for construction purposes will be consigned to the Mitchell RSF located in the Mitchell Creek Valley and to the McTagg RSF located in the McTagg Creek Valley. Waste rock from the Kerr pit will be backfilled into the mined-out Sulphurets pit. All waste rock placed in the RFSs is assumed to be potentially acid generating. The total amount of mine waste rock to be removed during open pit excavations at the Mitchell pit, Sulphurets pit, and Kerr pit is approximately 3 Bt over the LOM.

NPAG mine waste rock removed from the Sulphurets pit during pre-production will be used to construct the basal drain beneath the Mitchell RSF, and it will be used as rockfill material in the construction of the WSD. A rock drain will also be constructed under the McTagg RSF.

Requirements for design, operation, and closure of RSFs on a mine site are legislated under the *Mines Act* (1996j) and are covered by sections of the Health, Safety and Reclamation Code for Mines in British Columbia (the Health, Safety and Reclamation Code; BC MEMPR 2008). The Health, Safety and Reclamation Code (BC MEMPR 2008) applies to "major dumps." A dump is defined as an accumulation of rock fragments or other unconsolidated material formed by pushing or dropping loose material over a crest and allowing it to come to rest without further handling. A major dump is defined as a dump that has one or more of the following characteristics:

- contains a volume of material that exceeds 1 Mm^3
- has a dump height greater than 50 m
- has an area that is covered by a dump that exceeds 1 ha
- is founded upon natural or trimmed slopes that are sometimes steeper than 20° from a horizontal plane
- contains material dumped or placed in a water course having a potential peak flow greater than 1 m^3/s, once in every 200 years
- any other mine dump so declared by the Chief Inspector.

The RSFs proposed for the Project fall into the category of a major dump (BC MEMPR 2008).

Legislation and Standards

BC's *Mines Act* (1996j) provides guidance and approvals for all activities on a mine site, from exploration to development, production, closure, and reclamation. It also requires project proponents to obtain a permit approving the work system and reclamation program. To obtain this permit a detailed Mine Development Plan and Reclamation Program has to be submitted to the Chief Inspector of the BC MEM for approval.

Under the *Mines Act* (1996j), the Health, Safety and Reclamation Code (BC MEMPR 2008) requires proponents to provide:

- a mine plan showing the locations of waste disposal areas

- designs for major dumps developed in accordance with the Interim Guidelines of the British Columbia Mine Waste Rock Pile Research Committee
- designs for major dumps that allow for re-contouring and reclamation consistent with the end land use
- plans to operate and monitor major dumps in accordance with the Interim Guidelines of the British Columbia Mine Waste Rock Pile Research Committee
- an annual report on the performance of high risk dumps
- a design for major dump slopes that meet long-term stability criteria in accordance with the criteria provided in the Interim Guidelines of the British Columbia Mine Waste Rock Pile Research Committee
- a reclamation plan that ensures long-term stability and long-term erosion control;
- a waste dump stability monitoring program.

The Health, Safety and Reclamation Code (BC MEMPR 2008) requires proponents to provide plans for the prediction, and if necessary, prevention, mitigation, and management of ML/ARD. Details for ML/ARD management and monitoring are included in the Metal Leaching and Acid Rock Drainage Management Plan (*Section 26.14 of Volume 26* of the Application/EIS).

The Health, Safety and Reclamation Code (BC MEMPR 2008) also requires proponents to provide plans for soil salvage and handling, erosion control, and soil contamination prevention. Details for soil management and monitoring are included in the Terrain, Surficial Geology, and Soil Management and Monitoring Plan (*Section 26.13 of Volume 26* of the Application/EIS).

Plans for the control and diversion of water on the mine site, including of the management of water around the RSFs, are provided in the Water Management Plan (*Section 26.17 of Volume 26* of the Application/EIS).

A set of detailed mine development and reclamation plans will be submitted at a later date as part of the Mine Plan and Reclamation Program Permit application. The following sections outline the general provisions included in these documents in terms of RSF operation and monitoring.

Rock Storage Stability Plan

Objective

The objective of the Rock Storage Stability Plan is to construct and operate the RSFs to provide safe and stable storage of waste rock generated from open pit and underground mining operations. The management and monitoring plan is intended to provide a basis for confirming design assumptions and for tracking performance during construction and after the mine closes.

Targets

Targets of the Rock Storage Stability Plan include the following:

- prepare the foundations of rock storage sites
- construct RSFs to meet short-term (during construction) and long-term stability criteria
- control surface and subsurface water.

Actions to Avoid, Control, and Mitigate

Design Criteria

The RSFs have been classified based on the dump stability rating outlined in the *Mine Rock and Overburden Piles Investigation and Design Manual: Interim Guidelines* prepared for the British Columbia Mine Waste Rock Pile Research Committee (Piteau Associated 1991). The stability rating considers the probable failure hazard of the RSF based on key factors affecting stability including RSF size and geometry, foundation conditions, characteristics of the waste rock material, method of construction, climate and expected piezometric conditions, and rate of RSF construction and seismicity.

The dump stability ratings are summarized in A*ppendix 4-J* and the Rock Storage Facilities Design Report *Appendix 4-M* of the Application/EIS. The Mitchell RSF is classified as low-moderate failure hazard, the McTagg RSF is classified as low-moderate failure hazard, and the Sulphurets pit backfill is classified as low failure hazard (refer to *Volume 26* of the Application/EIS for further details).

Construction

A stable foundation is required under the RSFs to reduce the risk of RSF failure. Site preparation for the RSFs will include (details of RSF construction are included in *Appendix 4-J* of the Application/EIS):

- removal of merchantable timber
- stripping of organic, weak, and soft soils and/or of highly weathered rock where required
- proof-rolling where practical.

Operation

For the purposes of ensuring the safety of mine personnel working below areas of active dump construction or driving on access roads below rock storage areas, the following safety measures may be required:

- the construction of catch berms to impede rock rollout
- the establishment of exclusion zones at the toe of active dumps.

Seasonal accumulations of snow are expected in working areas of the Mine Site and in the water diversion channels, and will be removed periodically and prior to freshet. The snow will be considered to be contact water and will be disposed of in areas that drain to the WSF for treatment. Storage of snow and control of snow melt will include:

- Designated snow dumps located on the south facing slope of the McTagg RSF and on areas of the Mitchell RSF above the WSF where the snow will melt during the summer.
- Location of snow dumps in non-critical areas of the RSFs.

Geohazards including debris flows, debris slides, and frequent snow avalanches could affect the diversion structures. The storage capacity of the WSF is sufficient to handle additional volumes of water in the event that the diversion channels do not function; however, the following design and control measures have been considered:

- Diversion channels have been strategically located to avoid landslide and snow avalanche prone terrain wherever possible, and designed to minimize the risk of failure and to maximize channel efficiency; however, it is anticipated that ongoing maintenance of water diversion channels and structures will be required.
- Diversion channels will be constructed with a minimum 5 m base width to allow for snow removal by snow blowers or Caterpillar D-6 dozers. Snow and debris will be removed in a timely manner if the channels become blocked.

Periodic shotcrete reinforcement and rock bolting may be required along sections of rock-cut channels and spillways where discontinuities in the rock may lead to leakage or structural weakness.

Closure

For long-term stability and reclamation purposes, slopes of the Mitchell and McTagg RSFs will be re-contoured at closure. The modifications to the RSF areas will include:

- Re-contouring the Mitchell and McTagg RSF slopes below an elevation of 1,100 m (treeline) to an overall slope of 2H: 1V (27°) with a 50 cm soil cover and vegetated.
- Re-contouring the western face of the Mitchell RSF below an elevation of 840 m.
- Above the treeline, higher benches are proposed to be left un-vegetated to reflect existing talus slopes present in the Mitchell and McTagg Creek valleys.

Please refer to the Application/EIS (*Volume 26*) (Rescan 2013) for further details.

Post-closure

The post-closure phase includes complete reclamation of the RSFs and continued treatment of water collected in the WSF until the water quality meets acceptable standards for direct discharge to the environment. At that time, all facilities will be decommissioned and flows downstream of the Mine Site will be restored to pre-mine conditions.

Monitoring

A monitoring program will be developed to visually check the condition of RSF slopes and surface water control diversion channels during RSF construction and operation. The monitoring program will also include the installation of geotechnical instrumentation for the collection of data to confirm design assumptions, to warn of potential failure or deformation of RSF slopes, and to evaluate stability performance.

The results of the monitoring program will be used to measure the success of the management strategies and to identify where additional mitigation may be necessary. Monitoring will continue for a period of time after mine closure to confirm that reclamation objectives are being achieved and to identify repair or maintenance requirements. Inspection and monitoring may include:

- visual inspection of RSF platform, crest, and slopes to check for signs of cracking, settlement, or bulging
- visual inspection of the RSF toe area to check for signs of ground heave or seepage
- installation of piezometers during construction in the area of the Mitchell RSF foundation where lacustrine deposits are present to monitor pore pressure and phreatic levels
- installation of slope inclinometers in areas close to the Mitchell OPC during operation to monitor foundation deformation
- installation of surface survey monuments during RSF construction
- deployment of wireline extensometers as required
- inspection of surface water diversion ditches, channels, and pipelines to check the structural condition and to ensure that they are clear of obstruction.

A schedule for routine inspection and instrumentation monitoring will be developed at the time of mine permitting based on the mine construction and operation schedule. Instrumentation trigger levels and response criteria will also be established and included in an operation procedure for the RSFs.

Additional inspections of RSF slopes and surface water control structures will be undertaken following extreme rainfall events or significant earthquake events. Records will be kept to track RSF crest advance and loading rates, location of snow dumps, waste rock material quality, and any other information required to assess RSF performance.

Geohazards, such as debris flows, debris slides, and snow avalanches, in areas that could adversely affect worker safety and mine infrastructure, including surface water diversion structures, will be monitored and identified hazards will be controlled as necessary.

Monitoring will continue during periods of temporary mine closure.

Reporting

Regular inspection of active and inactive RSFs and review of pertinent data will be conducted by mine personnel. Inspection records will be maintained on site for review by the design engineer and by government mine inspectors. The information collected may be used for external reporting purposes.

The Health, Safety and Reclamation Code (BC MEMPR 2008) requires that an annual performance report be submitted for waste rock storage facilities that are classified as high risk. Information on the development and reclamation of the waste rock storage facilities may also be included in the annual reclamation and environmental monitoring report submitted to the BC government as required by the *Mines Act* (1996j) regulations under the Health, Safety and Reclamation Code (BC MEMPR 2008).

20.4.5 DOMESTIC AND INDUSTRIAL WASTE MANAGEMENT

The proposed management of domestic and industrial waste, including hazardous waste, is outlined in the *Domestic and Industrial Waste Management Plan,* which can be found in the Application/EIS (*Volume 26*) (Rescan 2013). The text below summarizes this management plan.

LEGISLATION AND STANDARDS

The minimum standards of acceptability of the Domestic and Industrial Waste Management Plan for the Project are to comply with federal and provincial legislation such as:

- *Environmental Management Act* (2003) and its regulations
- *Health Act* (1996f)
- *Forest and Range Practices Act* (2002c)
- *Fisheries Act* (1996c)
- *Water Act* (1996n)
- *Mines Act* (1996j)
- Health, Safety and Reclamation Code (BC MEMPR 2008)
- *Wildlife Act* (1996q)
- *Land Act* (1996h)
- *Environment and Land Use Act* (1996b)
- *Soil Conservation Act* (1996l)
- *Canada Transportation Act* (1996a)
- *Canadian Environmental Protection Act* (1999)
- *Hazardous Products Act* (1985d)

- Controlled Products Regulations (SOR/88-66)
- *Transportation of Dangerous Goods Act* (1992b).

OBJECTIVE

The primary purposes of the Domestic and Industrial Waste Management Plan (the Waste Management Plan) is to protect workers and the public, and to minimize any potential adverse effects to the environment, including fish and wildlife and their habitat, while ensuring compliance with regulatory requirements, permit and licence obligations, and the Proponent's Environmental Policy. The secondary purpose of the plan is to minimize the risk and cost associated with the recycling, storage, handling, disposal, and removal of waste from all aspects of the Project. A material is considered a waste when it can no longer be used for its original purpose.

The Waste Management Plan documents Seabridge's approach to waste management and outlines strategies that will be used to process the various waste streams to ensure maximum environmental protection. The Waste Management Plan will be reviewed regularly and revised as required to ensure continued best practices and compliance.

In order for the Waste Management Plan and associated procedures to work to their full extent, everyone on the site must be made aware of the plan and their corresponding responsibilities. All Project personnel, including contractors, need to be active participants.

TARGETS

The targets for the Waste Management Plan are to ensure that:

- all employees and contractors on site have at least overview training in Project waste management strategies, achieved through site orientation training
- every work area has a designated waste collection or disposal area
- every waste collection or disposal area has designated and secure areas or containers for disposal of specific waste types
- appropriate spill kits are available wherever there is a potential for a spill
- site workers are trained in spill prevention and spill response.

ACTIONS TO AVOID, CONTROL, AND MITIGATE

The Waste Management Plan focuses on the wise use of resources, which includes the four R's: reduce, reuse, recycle, and recover. Where possible, all of these methods will be exhausted before disposing of waste materials.

WASTE REDUCTION

Reducing the amount of material that is consumed is the most effective way of reducing the amount of waste that is generated. Consumption will be assessed by evaluating all procedures, processes, and consumed materials for possible reductions in raw material usage, as well as possible reductions in generated waste volumes. Examples of waste reduction measures that may be used include:

- product review, selection, and substitution; recyclable/reusable and non-hazardous materials will be used instead of non-recyclable/non-reusable and hazardous materials
- ordering chemicals or lube products in bulk/returnable containers
- keeping a workable minimum inventory to prevent expiration of products and resulting generation of waste
- decreasing the amount of solid waste by reducing the use of disposable items
- training personnel on waste minimization and reuse
- decreasing the amount of packaging on supplies by requesting that the suppliers provide less packaging materials on over-packaged products.

Material Reuse

Materials brought to the Project site should be used to the maximum extent possible, and where applicable, reused on the site. Examples of reusable materials include:

- scrap metal, conveyor belts, and wood
- chemical containers that can be returned to the supplier to be refilled
- waste oils, glycols, and solvents that can be reused for secondary jobs.

Recycling

A recycling program will be incorporated at the Project for successful management of waste streams. The program will recycle as many products as possible on site (e.g., salvageable lumber and scrap metal, paper, cardboard, and salvageable parts from vehicles).

Other recyclable materials will be shipped off site to the nearest recycling facility. Products that will be shipped off site include:

- used oil filters (oil removed, crushed, and recycled separately)
- lead-acid and alkaline batteries
- plastic petroleum pails
- used/damaged auto parts
- oil-based paints
- empty drums.

Recovery

Recovery is the final level of waste minimization and involves extracting usable material or energy as a by-product for other uses. Opportunities for recovery will be evaluated throughout the life of the Project.

Tracking and Enforcement

The management of domestic and industrial waste from the Project will be accomplished through the implementation and monitoring of domestic and industrial waste management procedures. Monitoring and enforcement of these procedures is fundamental to ensuring that the Waste Management Plan is functioning to its full extent for the life of the Project.

An imperative step in achieving compliance with the procedures is to ensure that all employees, contractors, and sub-contractors are aware of the plan and procedures and how the procedures apply to them. For example, all persons at the Project will be made aware of the recycling program and of how to direct waste to the correct waste stream through an orientation and training program on domestic and industrial waste procedures. Permanent employees will receive regular retraining and updates when new procedures or changes are introduced.

Waste produced by Project activities and personnel will be separated at the point of generation. Each department will be accountable for its workers, including Seabridge employees, contractors, and sub-contractors, to ensure that the waste is handled correctly as per the waste management procedures. Tracking waste streams is essential to ensure that each department is performing its waste management procedures responsibility.

Tracking

Off-site and on-site disposal will be documented by tracking waste type, volume, method of disposal, and location. Tracking of the waste streams by each department will show where changes can be made to further improve the waste management system over the life of the Project. Tracking will also alert management to areas or departments that may require new procedures or particular attention. A summary of the tracking will be reported annually by each department to the Environmental Superintendent, along with an analysis of the effectiveness of the existing systems and any proposed improvements.

Audit Program

An audit program is required to alert management when enforcement is necessary to ensure compliance with the plan and procedures. The mine manager will appoint a small audit team.

Waste Types and Sources

The Project will generate several forms of industrial and domestic waste throughout the life of the Project. This management plan discusses the solid waste, including hazardous waste, that may be generated, as well as their storage and final disposal methods.

WASTE TYPES GENERATED DURING CONSTRUCTION

Waste produced during construction will be generated at the road, tunnels, mine, and Treaty OPC construction camps, and by the construction of the mine and adjacent facilities, access corridors, Treaty Process Plant, and TMF.

The long duration of the construction phase, combined with the wide range of phase activities such as tunnelling, TMF construction, and pre-stripping, means that waste will be similar between the construction and operation phases. The primary difference will be the addition of waste related to production in the Treaty Process Plant during operation.

The types of waste that will be generated during the Project operation are listed in the Application/EIS (*Table 26.6-1 of Chapter 26*) (Rescan 2013).

Domestic and industrial waste produced during the operation phase will be controlled and monitored throughout the life of the Project.

WASTE COLLECTION/DISPOSAL FACILITIES

Construction

In order to meet the construction schedule of the access corridors, tunnels, the Treaty Process Plant, the TMF, WSF, WTP, and the Mine Site, 12 construction camps will be required. On completion of construction, the number of camps will be reduced to two operating camps

Each construction camp will have its own waste disposal facilities with clearly marked containers or areas for each type of waste. For example, the inert non-reactive solid waste (tires, conveyor belts, rebar, wood, etc.) will be stockpiled for later disposal in one of the Project landfills or in a licensed off-site landfill. Hazardous waste (petroleum waste, batteries, etc.) will be stored in appropriate sealed containers within a bermed area for transfer off site.

Waste Disposal Methods/Facilities for the Construction Camps

All construction camps will have incinerators to handle putrescible kitchen waste and portable sewage treatment plants (rotating biological contactors or other similar units) to handle both black and grey water waste. Sewage plant effluent will be discharged to a tile field more than 100 m from the high water mark. Sludge will be removed as required for efficient operation of the plants and will be hauled for disposal in a licensed landfill. Solid inert waste and hazardous waste will be stored until it can be transferred to appropriate disposal facilities.

Clearly labelled sealable containers will be provided at each camp and staging area for the different types of materials (e.g., hazardous waste and recyclable waste). Electric fencing will be used to deter bears from entering waste disposal facilities. Sealed containers from the camps will be hauled for disposal at the next most accessible disposal point. Ideally, inert non-reactive solid waste materials that cannot be recycled, reused, or burned in an incinerator will be stored until they can be disposed in the Project landfills once the landfills are established and road access is available. Where road access is available to public roads before the Project landfills are available, inert non-reactive waste from construction camps and staging areas may be hauled to licensed off-site landfills.

Operation

Central disposal facilities will be established at the Mitchell and Treaty OPCs where waste materials will be organized for coordinated disposal. These central disposal facilities will be created in the latter part of the construction phase and will be operational by the start of the operation phase. Waste materials from sites in the Unuk River drainage will be directed to the Mitchell OPC, and those from the Bell-Irving River drainage will be directed to the Treaty OPC.

The waste collection areas, landfills, and sewage effluent/sludge disposal systems will have waste containment and runoff control structures that will prevent the escape of untreated waste to the surface or ground water systems. Regular audits of these waste containment and runoff control structures will be conducted, and the records of these inspections will be kept for review upon the request of the site manager or an inspector. Regular inspection audits will be conducted on all the disposal systems as well to ensure that the waste is being handled correctly and filtered into the correct waste streams.

Waste Collection Areas

The waste collection areas will function as a storage area for waste until the waste is processed further or transferred off site to the appropriate approved recycling or disposal facilities or landfill. The waste collection areas will be properly designed to contain and prevent contamination of the environment. They will also be designed to adequately and safely store a sufficient quantity of waste over a prescribed time limit of one to three months. Where required, the waste collection areas will be covered and fenced to prevent attracting wildlife and to provide protection from weather. Additionally, hazardous waste disposal facilities will be adequately designed to contain spills.

The waste collection area will consist of three parts:

- **Recycle/reuse area:** This area will contain the items that can be recycled/reused on the site. Inert materials to be stored in this area include tires, scrap metals, and waste wood. These items will be placed in designated containers or areas within the recycle/reuse area of the waste collection area. This method will allow personnel to search the recycle/reuse area of the waste collection area for materials to reuse. Once these containers or areas become full, the contents will be either disposed of in a designated on-site facility or shipped off site for recycling at an approved facility.
- **Hazardous waste area:** The hazardous waste area will contain hazardous waste that is required to be shipped off-site. Hazardous waste, including used glycol, acids, solvents, laboratory chemical waste, oil that cannot be burned in incinerators, oily rags, absorbent pads, hydraulic fluid, and any other hazardous chemicals, will be stored in a bermed containment area. Hazardous waste will not be permitted to accumulate to excessive volumes, but will be shipped off site to avoid crowding.

- **Removal area:** This area will contain waste that will be disposed of in an on-site facility. Waste will include domestic and industrial waste that is not hazardous or recyclable.

The waste in the waste collection area will be segregated and stored using accepted management practices including the following:

- Fire prevention systems that are adequately designed for the materials being stored will be used.
- Spill kits, protective equipment, and other necessary equipment to clean and mitigate spills will be used.
- Only containers in good condition will be used to store items.
- Containers and liner materials will be compatible with the waste being disposed.
- Containers and drums will be labelled to identify the waste content and initial date of storage.
- Sufficient storage space will be left between containers to allow for safe access and handling of containers.
- Incompatible waste will not be stored in the same containers and will be stored at a safe distance from each other.

LANDFILLS

Two landfills will be established, one each in the Mitchell and Treaty OPCs. The landfills will be used to dispose of only solid inert, non-reactive waste such as used conveyor belts, empty dry latex paint cans, grinding balls, air filters, non-recyclable plastics, and incinerator ash. To deter wildlife attraction to the landfill, the landfill will be fenced and only solid inert waste that will not act as a wildlife attractant will be deposited there. The garbage will be periodically covered with not potentially acid generating waste rock or local till to prevent wind loss and to mitigate wildlife attraction.

Signs will be posted around the landfills to identify the disposal area, and the landfills will be audited regularly to ensure that they are only used for disposal of approved waste products. The audits will be recorded and evaluated for potential areas of improvement.

INCINERATORS

Incinerators will be used at all camps and at the Mitchell truck shop, whenever possible, for waste disposal. The Treaty operating camp will share an incinerator with the Treaty Process Plant and related facilities. The incinerators will be used to dispose of all waste that is a wildlife attractant, including food waste and food-related products. Food waste is a prime wildlife attractant and will therefore be incinerated in a timely manner, thus leaving no trace of attractants for wildlife. All kitchen, dining room, office, and accommodation waste will be incinerated to reduce the potential of attracting wildlife. All resultant incinerator ash will be placed in a landfill.

Incinerator areas will be fenced, further reducing wildlife interactions.

SEWAGE PLANT EFFLUENT/SLUDGE

The sewage waste disposal facilities will consist of rotating biological contactors or other similar units and tile fields appropriately sized for each camp or location. Sewage plants will be located at the Mitchell truck shop and near the primary crusher as well as at each camp. The effluent from the rotary bioreactors will be discharged to a tile field placed 100 m from the high water mark for further treatment. The sludge from the rotary bioreactors will be periodically pumped and disposed of in an appropriate manner depending on the final analysis of the sludge. During operation, the sludge may be deposited in the landfills or in the TMF.

PROCEDURES FOR MANAGING SPECIFIC WASTE

The waste collection areas provide a means to collect the waste streams and transfer them to their correct disposal areas. Specific areas will be designated for each type of waste and will be clearly labelled.

Please see the Application/EIS (*Volume 26*) (Rescan 2013) for typical waste types to be expected at the Project, their treatment strategies, and handling/disposal methods. Any updates on specific waste management since the Application/EIS can be found in Section 18.0 of this PFS.

HAZARDOUS WASTE

Hazardous waste will be produced in all Project phases. It includes materials such as waste oil, laboratory chemicals and solvents, lead-acid batteries, oil filters, and used oily rags and absorbent pads.

The Hazardous Waste Regulation (BC Reg. 63/88) under the *Environmental Management Act* (2003) defines “hazardous waste” (please see *Volume 26* of the Application/EIS for additional details. Any updates on hazardous waste management since the Application/EIS can be found in Section 18.0 of this PFS.

Hazardous waste requires special handling and training procedures. All employees, contractors, and sub-contractors who are handling hazardous waste for the Project will be provided with Workplace Hazardous Materials Information System training or will be required under contract to have that training, so they can identify hazardous waste and know how to handle it appropriately. Transportation of Dangerous Goods training will be provided, or required of employees, contractors, and sub-contractors who are receiving, off-loading, and storing potentially hazardous materials, or involved in the storage and shipment off-site of hazardous waste.

Hazardous waste will be transferred to an approved hazardous waste facility that will issue a certificate of destruction. Periodic audits of this facility to ensure proper handling and destruction of hazardous waste will be considered.

All hazardous materials and dangerous goods will be stored in clearly labelled containers or vessels and handled in accordance with regulations appropriate to their hazard characteristics.

Petroleum Waste Stream

Petroleum products will be used widely at the Project. The waste generated from the petroleum products will be used oil, diesel fuel, lubricants, gasoline, jet B, oily rags and absorbent pads, and solvents. These products have to be handled with caution because they can potentially adversely affect the environment. The handling, storage, and spill contingency for petroleum products are outlined in the Spill Prevention and Emergency Response Plan.

To properly manage the petroleum waste stream and make the individual waste streams easier to reuse, recycle or recover, the waste will be segregated into classes, as detailed in the Application/EIS (*Section 26.6 of Volume 26*) (Rescan 2013).

Laboratory Chemical Waste

The laboratory personnel will determine which laboratory chemical waste cannot be safely incinerated or disposed of in the landfill. This waste will be stored in appropriate containers and placed in the waste collection areas until they are shipped off site to a licensed disposal facility.

Biological Waste

The first aid areas will generate small amounts of hazardous waste in the form of needles, syringes, scalpel blades, and blood- and tissue-contaminated materials. This waste will be properly contained in biohazard containers in the first aid area under the supervision of the first aid staff. The blood- and tissue-contaminated materials will be incinerated and the other biohazardous waste will be shipped off site to an approved disposal facility.

Non-hazardous Waste

Non-hazardous waste will be produced in all phases of the Project. They include materials such as domestic garbage, food waste, paper materials, aluminum cans, glass, plastics, inert bulk waste, etc. (please refer to *Section 26.6, Volume 26* of the Application/EIS for additional details).

CLOSURE AND DECOMMISSIONING

Activities during the closure phase will be similar to the activities during the construction phase. A range of materials will become available for salvage, recycling, or disposal with the dismantling and removal of buildings, surface structures, fuel tanks, etc. The Reclamation and Closure Plan will cover the closure, reclamation, and decommissioning of the Project in detail.

WASTE MANAGEMENT DURING CLOSURE

Significant amounts of waste will be generated from the dismantling of buildings and process-related materials. The approach for waste management during closure will be to identify feasible salvage and recycling options.

Upon closure, the buildings, facilities, and process equipment will be dismantled and either disposed of at the site landfill (inert non-reactive materials only) or removed from the site for recycling and/or disposal. Any equipment or materials with market value will be removed for capital recovery.

Decommissioning of the Waste Management Facilities

Some of the waste management facilities at both the Mitchell and Treaty OPCs, such as the waste collection areas, incinerators, and the sewage plants, will be retained to support facilities remaining on the Project site for ongoing site maintenance and inspection purposes. The MTT, WTP, MDT, MTDT, hydroelectric facilities, and related accommodation and maintenance facilities will be retained. Other facilities will be dismantled and/or removed from the site. All scrap metal will be disposed of in a manner acceptable to an inspector. The remaining area after removal of structures will be reclaimed; for example, concrete foundations will be covered and re-vegetated.

The landfill will be covered with waste rock and reclaimed to ensure long-term stability and erosion control. All reclamation details and processes are described in detail in the Reclamation and Closure Plan.

20.5 Air Quality Management including Greenhouse Gases

The proposed management of air quality, including greenhouse gases, is outlined in the *Air Quality Management Plan and Greenhouse Gas Management Plan,* which can be found in the Application/EIS (*Volume 26*) (Rescan 2013). The following text summarizes these management plans.

20.5.1 Air Quality Management Plan

The Air Quality Management Plan consists of three sub-plans:

- Emissions Management Plan
- Fugitive Dust Emissions Management Plan
- Meteorology Monitoring Plan.

Emissions Management Plan

The Emissions Management Plan outlines:

- the legislation and standards relevant to emissions associated with the Project
- the primary emission mitigation methods that the Proponent will implement;
- the continual assessment and reporting of emissions that will take place throughout the Project life.

Legislation and Standards

The federal government has set National Ambient Air Quality Objectives (NAAQOs) and Canada-wide standards under the *Canadian Environmental Protection Act* (1999). Canada-wide standards are intended to be achievable targets that will reduce health and environmental risks within a specific timeframe, whereas NAAQOs identify benchmark levels of protection for people and the environment. In addition, BC has developed air quality objectives for a number of contaminants.

The applicable standards relating to the Project include:

- NAAQOs (Environment Canada 1999)
- Canada-wide standards (CCME 2000)
- BC Air Quality Objectives and Standards (BC MOE 2009a).

Objective

The Emissions Management Plan will establish measures to mitigate emissions from Project activities to meet air emission legislative requirements and to reduce the Project effects to reasonable levels. The objective of the Emissions Management Plan is to mitigate and monitor emissions from Project activities.

Targets

The Emissions Management Plan targets are to:

- avoid, control, and mitigate air pollution associated with Project operation
- establish a monitoring plan to collect on-site air quality data, results of which will be reviewed annually to determine if any trends are evident and if target criteria are being met.

Actions to Avoid, Control, and Mitigate

The main source of emissions during the construction phase of the Project will be from diesel exhaust. Diesel emissions include carbon monoxide, nitrogen oxides, sulphur dioxide, particulate matter, and residual unburned fuel vapours. Emissions will also be produced by the incineration of inorganic and organic wastes.

Activities that will produce emissions during the Project operation phase include blasting, operating diesel-powered mining equipment, and operating haul trucks for transporting waste rock and ore. Transportation of personnel and materials to and from the Mine Site will also produce emissions during operations. Emissions include sulphur dioxide, nitrogen oxides, carbon monoxide, and particulate matter.

In an effort to mitigate emissions during the various phases of the Project, Seabridge is currently implementing, or plans to implement, measures described in detail in the Application/EIS (*Section 26.11, Volume 26*) (Rescan 2013).

Monitoring

Stack testing of a selection of incinerators and process units will be conducted in compliance with permit requirements under the BC *Environmental Management Act* – Air Permit.

Air quality monitoring will be carried out to establish the emissions associated with site activities during operation. Monitoring stations will be located at various locations. Details of any adverse findings will be recorded and reported as required by regulatory authorities.

Reporting

The results of the emissions monitoring program will be reported on at the frequency specified in the Air Permit.

Adaption

The air quality monitoring data will be reviewed annually to determine if any trends are evident. The need for any corrective actions to on-site emission management or installation of additional control measures will be determined on a case-by-case basis. Indications of the need for corrective actions and additional control measures may include:

- monitoring data showing an increasing trend in emission concentrations
- issues raised by on-site staff, regulators, or local communities.

Discussions will be initiated to resolve any issues as soon as possible after the issue has been identified.

Components of the Emissions Management Plan may need to be revised over the life of the Project, based on regulatory changes and technological advances. Any modification made to the Emissions Management Plan will be communicated to regulatory authorities where applicable.

FUGITIVE DUST EMISSIONS MANAGEMENT PLAN

The Fugitive Dust Emissions Management Plan outlines:

- the legislations and standards relevant to dust emissions associated with the Project
- the main emission mitigation methods that the Proponent will implement
- the continual assessment and reporting of emissions that will take place throughout the Project life.

Legislation and Standards

The applicable standards include:

- NAAQOs (Environment Canada 1999)
- Canada-wide standards (CCME 2000)
- British Columbia Air Quality Objectives (BC MOE 2009a)
- The Pollution Control Objectives for the Mining, Smelting, and Related Industries of British Columbia (BC MOE 1979).

Objective

The main objective of the Fugitive Dust Emissions Management Plan is to ensure that the levels of fugitive dust generated by Project activities are at or lower than levels required to meet the Canada and BC ambient air quality and dustfall objectives. The Fugitive Dust Management Plan will also identify methods to reduce fugitive emissions from Project activities.

Targets

The Fugitive Dust Emissions Management Plan targets are to:

- ensure fugitive dust emissions generated by the Project do not cause a medium to long-term exceedance of the standards
- maintain a monitoring plan to collect on-site air quality data related to dust, results of which will be reviewed annually to determine if any trends are evident and if target criteria are being met.

Actions to Avoid, Control, and Mitigate

The main sources of fugitive dust emissions during the construction phase of the Project will be vehicles travelling on unpaved roads, construction of the access corridor, and other construction activities, including earthworks, topsoil removal, and blasting.

Activities that will produce fugitive dust emissions during the operational phase of the Project include vehicles travelling on unpaved roads, ore transfer, truck loading and unloading, crushing and blasting.

In an effort to mitigate dust emissions during the various phases of the Project, Seabridge is currently implementing, or plans to implement, measures described in detail in the Application/EIS (*Section 26.11, Volume 26*) (Rescan 2013).

Monitoring

Fugitive dust monitoring commenced in 2008, and consisted of dustfall monitoring of particulates, anions, cations, and total metals. Dustfall monitoring began at 5 locations and expanded to 10 locations as of 2012. Each station monitors dustfall over consecutive 30-day periods during the summer and early fall.

The locations of the dustfall stations during construction and operation are likely to be slightly different than the baseline locations due to shifting areas of activity on site as the Project progresses. The dustfall monitoring stations will be sited in accordance with ASTM Standard D1739-98 (ASTM 2010). The stations will be in open areas that are free of structures higher than 1 m within a 20 m radius of the collection container.

The dustfall monitoring will provide a 30-day average ground-level mass of deposited dust. These values will be compared to the relevant BC dustfall objectives stated in *The Pollution Control Objectives for the Mining, Smelting, and Related Industries of British Columbia* (BC MOE 1979). In addition, analysis of temporal trends will be undertaken to determine if there are any increasing trends in the measured concentrations with consideration to the time of year and meteorological conditions. Monitoring data will also be used to provide feedback to modify the dust management procedures incorporated at the site, if required. However, sampling does not occur in "real time" and there will be a delay between the events that lead to any elevated dust deposition and the receipt of monitoring results.

REPORTING

The results of the Fugitive Dust Monitoring Program will be reported in various annual reports and will be provided to senior management and regulatory agencies as required.

ADAPTION

The dustfall monitoring data will be reviewed annually to determine if any trends are evident. The need for any corrective actions to on-site dustfall management or additional control measures will be determined on a case-by-case basis. Indications of the need for corrective actions and additional control measures may include:

- monitoring data showing an increasing trend in deposited dust
- issues raised by on-site staff, regulators, or local communities.

Discussions will be initiated to resolve any issues as soon as possible after the issue has been identified.

It is possible that components of the Fugitive Dust Emissions Management Plan may need to be revised over the life of the Project, based on regulatory changes and technological advances. Any modification made to the Fugitive Dust Emissions Management Plan will be communicated to regulatory authorities where applicable.

METEOROLOGY MONITORING PLAN

Objectives

The objectives of the Meteorological Monitoring Plan are to measure and characterize the atmospheric conditions within the Project area to establish long-term data and review results annually.

On-site meteorological data are used for a variety of purposes for mining projects. For instance, wind speed and direction data were required for the Project to select sites for permanent camps and mineral processing facilities, in order to accommodate predominant wind patterns and mitigate the effects of fugitive dust and other emissions. Solar radiation, evaporation, and precipitation data are required in water balance calculations for water containment and treatment systems. Precipitation (as both snow and rain) data will facilitate monitoring and predicting potential hazards such as avalanches and landslides (see the Application/EIS, *Volume 26*) (Rescan 2013). Meteorological monitoring will also assist in understanding and better predicting the potential effects of climate change on the Project over its lifetime.

Monitoring

Since late 2007, a total of five automated meteorological stations have been installed and operated as part of the meteorology baseline monitoring program for the Project. There are four Environment Canada stations in the region of the Project that provide data for comparison to that collected by the Project meteorological stations.

Monitoring of meteorological data will continue through the Project life—including temperature, relative humidity, precipitation and snow depth, solar radiation, barometric pressure, and evaporation. The positioning of stations in the area of the Project may shift per regulatory reporting requirements and air quality, hydrological, climate change, safety, and other data needs over the life of the Project.

Reporting

The results of ongoing meteorological monitoring throughout the Project life will be reported in Annual Environmental Reports, and will be provided to senior management for the Project as required.

It is possible that components of the Meteorological Monitoring Plan may need to be revised over the life of the Project, based on the needs of the Project, regulatory changes, or technological advances. Any modification made to the Meteorological Monitoring Plan will be communicated to regulatory authorities, such as the BC MOE, where applicable. Also, modifications will be made so as to ensure that the continuity of data collection is not disrupted in a manner that would significantly alter its usefulness (such as the ability to provide information pertinent to health and safety, or to interpret air quality, climatic, or hydrological trends resulting from the Project, or that may cause effects to the Project).

20.5.2 Greenhouse Gas Management Plan

A description of the issue of climate change relating to the need to manage greenhouse gas (GHG) emissions for the Project is discussed in *Volume 6* of the Application/EIS, GHG emissions (Climate Change). The GHG management plan outlines:

- the legislative context for mitigating GHG emissions in BC
- the main GHG mitigation methods that the Proponent will use for the Project
- the assessment and reporting of Project GHG emissions that the Proponent will undertake.

Legislation and Best Practices Context

International agreements and North American national legislation with clear and enforceable GHG mitigation targets at the project level have yet to be determined. However, provincial and national development of such legislation is underway as described in the section below. Legislation, policy, and initiatives to address climate change adaptation are also being developed (CEA Agency 2003; IPCC 2007; BC MOE 2010c), but there is some regulatory uncertainty as to what legislation will apply during the Project life due to changes in political influences. In BC, carbon management and markets fall under both regulatory and voluntary domains, so organizations can implement carbon management strategies under several voluntary third-party programs that additionally promote best practices in the monitoring, reduction, and transparent reporting of GHG emission inventories.

Regulatory Context

The main pieces of legislation pertaining to carbon management for major projects in BC, including taxation and market mechanisms, as of 2012 can be found in the Application/EIS (*Volume 12*) (Rescan 2013). In the absence of regulations, many organizations seek to minimize GHG emissions voluntarily to meet corporate sustainability reporting goals, procure financing, address liability, or improve public relations.

Under the Copenhagen Accord in 2009, Canada signed on to reduce its total GHG emissions by 17% from 2005 levels by 2020, mirroring American targets. To meet this national GHG reduction target, Canada has also begun to implement regulations under the *Canadian Environmental Protection Act* (1999) and the Clean Air Regulatory Agenda for energy suppliers (starting with coal) and the transport sector (for heavy- and light-duty vehicle manufacturers). To demonstrate its reductions, Canada reports national GHG emissions annually to the United Nations Framework Convention on Climate Change.

Canada has set progressively aggressive fuel efficiency targets for manufacturers through national transport regulations—in line with those in the United States—which will help to provide transport sector GHG emissions reductions in future years, and consequently provide transport related GHG reductions for the Project from upstream sources. For instance, on November 27, 2012, the federal government announced new regulations for automobiles and light trucks manufactured between 2017 and 2025, which mandate improvements to engine fuel efficiency such that by 2025, vehicles in this category will consume 50% less fuel and will emit 50% less GHG emissions than similar 2008 models (Environment Canada 2012a). These proposed regulations will build on the Passenger Automobile and Light Truck Greenhouse Gas Emission Regulations (SOR/2010-201) for vehicles manufactured between 2011 and 2016, which mandates that 2016 models have about 25% lower GHG emissions compared to similar 2008 models. The proposed Heavy-duty Vehicle and Engine Greenhouse Gas Emission Regulation (SOR/2013-24) will mandate manufactured emission reductions for heavy-duty vehicles, commence in 2014, and will also help to lower transport related emissions of the Project compared to current estimates (Government of Canada 2012). For instance, heavy-duty vehicle models (i.e., large pick-up trucks, short/long-haul tractors, cement and garbage trucks, and buses) manufactured in 2018 will be required to reduce end-of-pipe GHG emissions up to 23% from those sold in 2010, and by 2020 overall national emissions from this vehicle class are projected to drop by 3 Mt/a (Environment Canada 2012b). These types of reductions are why the procurement of new vehicles is listed as a mitigation measure in the Application/EIS (*Section 26.12.1.5, Volume 26*) (Rescan 2013).

BC also has several provincial climate change regulations in place, often aligning targets and mechanisms with those in California. Through the BC Climate Action Plan (Government of BC 2008vb), the province has set more stringent targets—33% GHG emissions reductions by 2020, and 80% by 2050, compared to 2007 levels—than the national targets described above (Government of BC 2008b. BC currently also has a carbon tax, although the general *GHG Reduction (Cap and Trade) Act* (2008b) is currently slated to become the major legislative arm to regulate emissions in BC. The *GHG Reduction (Cap and Trade) Act* (2008a) also enabled the province to be the first Canadian province to join the regional (United States and Canada) Western Climate Initiative in 2007, but BC has not yet implemented regulations through the Western Climate Initiative and still has the option to opt out prior to its slated implementation in 2015.

The *GHG Reduction (Vehicle Emissions Standards) Act* (2008c) is also slated to roll out in BC in the next few years, putting initial caps on transport emissions, which will likely be raised incrementally in future years to be in line with target reductions in BC: a total of 33% by 2020 compared to 2007, and 80% below 2007 levels by 2050 (Government of BC 2008b). In conjunction with national transport regulations, this act will help reduce GHG emissions of contracted (Scope 3) haul truck emissions for the Project.

Regarding land-use change, in support of the Climate Action Plan, BC has enacted the *Zero Net Deforestation Act* (2010), targeting net zero deforestation for BC by December 31, 2015, starting with government reporting on deforestation in 2012. The objectives of the Act are to achieve net zero deforestation without "undermining economic development," and to use information and incentives to encourage voluntary action by industry to avoid and reduce deforestation and increase afforestation levels (BC MFML 2010).

Greenhouse Gas Emission Reporting and Reduction Requirements

In support of Canada's GHG mitigation targets, since 2010, facilities emitting over 50,000 t of carbon dioxide equivalent have been required to report emissions to Environment Canada for the *Greenhouse Gas Emissions Reporting Program* (Environment Canada 2010), under the jurisdiction of Section 46 of the *Canadian Environmental Protection Act* (1999). Data from the Reporting Program are used to supplement that from the annual Report on Energy Supply-Demand in Canada compiled by Statistics Canada in national inventory reports to the United National Framework Convention on Climate Change (Environment Canada 2012d).

In BC, since January 1, 2010, facilities emitting over 10,000 t of carbon dioxide equivalent must report to the BC Ministry of Environment, and those emitting over 25,000 t CO_2e must also have to have emissions verified by an independent and accredited third party under the BC Reporting Regulation (BC Reg. 272/2009) of the *Greenhouse Gas Reduction (Cap and Trade) Act* (2008b).

The above provincial and national reporting regulations only pertain to facility-level emissions, and so do not include land use change. If the Project facility-level GHG emissions surpass 50,000 t carbon dioxide equivalent/a, to satisfy federal and provincial reporting requirements, Project GHG emissions will need to be assessed, verified, and reported. Project GHG emissions will also be able to be reported through the online one-window reporting system, which was introduced in 2010 to harmonize the needs of federal and provincial reporting, prevent duplication, and reduce the reporting burden on industry (BC MOE 2011b).

There is no current cap on industrial GHG emissions mandating emission reductions for the Project; however, BC's carbon tax will also apply to purchases for the Project, and the *Greenhouse Gas Reduction (Cap and Trade) Act* (2008b), is designed to set the groundwork for a regulatory regime that was to be implemented through the tabled Emission Trading Regulation on January 1, 2012. The proposed Emission Trading Regulation is applicable to facility operations that emit over 25,000 t carbon dioxide equivalent/a from "emissions from general stationary combustion of fuel or waste with the production of useful energy" (BC CAS 2010), which would be applicable to the Project.

Implementation of the Emission Trading Regulation is designed to be concurrent with that of California's cap and trade system, as BC, California, Quebec, and other regional members have arrangements to be GHG emissions trading partners under a linked system arranged through the regional Western Climate Initiative. The California Air Resources Board (CARB) delayed the implementation of its own cap and trade system (under California's Global Warming Solutions Act, AB 32 [CARB 2006]) until 2013, which prompted the delay in BC as well.

California has now taken steps to initiate its cap and trade system. In September 2012, it officially launched the program, followed by the first auctioning of greenhouse gas allowances by the California Air Resources Board on November 14, 2012 (CARB 2012b), and its December 14, 2012 announcement of provisions for carbon offset projects (CARB 2012b). Quebec has also now become the first Canadian province to join California and the Western Climate Initiative in creating a regional carbon market by adopting regulations to join their two capped systems (MDDEFP 2012; Segun 2012). There is currently regulatory uncertainty as to whether BC will continue with its original plans under the *Greenhouse Gas Reduction (Cap and Trade) Act* (2008b) to join in a capped and regulated carbon market with California and Quebec or pursue other avenues of carbon management.

OBJECTIVE

The primary objective of the GHG Management Plan is to mitigate net GHGs emitted to the atmosphere by Project activities.

TARGETS

The GHG Management Plan targets are to:

- ensure that reporting is carried out to meet requirements
- identify and implement measures to progressively minimize GHG emissions and maximize fuel and energy efficiency
- maintain a plan to monitor on-site GHG emission data, the cumulative results of which will be reviewed periodically to determine if any trends are evident and if target criteria are being met

- maintain a plan to monitor GHG emissions associated with land use change GHG sources and sinks
- increase the carbon fixed as biomass by minimizing clearing and by maximizing vegetation reclamation where possible.

ACTIONS TO AVOID, CONTROL, AND MINIMIZE EFFECTS

Greenhouse Gas Mitigation Hierarchy: Introduction

There are several ways to mitigate the GHG emissions of a project. The GHG management hierarchy presents the ideal line of mitigation strategy for GHG emissions, starting with avoidance, then reduction, replacement, enhancement, and finally offsetting GHG emissions. Carbon offsetting for projects in this hierarchy only takes place after other reasonable mitigation measures have been implemented.

The actions at the base of the hierarchy are the most transformative and effective at reducing a company's GHG emissions profile. Avoidance, reduction, and replacement activities to mitigate GHG emissions involve reducing fuel use or energy consumption, and so are also typically cost saving as well. Enhancement includes actions that Seabridge has committed to regarding replanting activities, which will re-establish vegetation and natural carbon sequestration from the atmosphere via photosynthesis. Offsetting remaining GHG emissions that cannot otherwise be mitigated, can involve either purchasing offsets, or creating them via developing additional offset projects.

Description of Project Greenhouse Gas Emissions Sources

The major source of GHG emissions associated with the Project will be from facility-level emissions from the fuel/energy needs of the Project. This includes direct, on-site (Scope 1) sources, such as from diesel engines and blasting, and indirect sources such as imported electricity consumption (Scope 2) and activities by third parties such as on-site equipment operation and transport activities to off-site locations (Scope 3). The Mining Association of Canada (Stratos 2009), states that over 95% of the GHG emissions generated directly by the mining industry are a result of fossil fuel use. Therefore, controlling fuel use will result in the most significant GHG emissions reductions, as well as reduced expenses. In addition, decreasing the variability of energy use and improving operating and maintenance practices can reduce energy costs by 5 to 10% and in most cases do not require a capital expenditure.

There will also be net emissions associated with land use change GHG sources and sinks from activities such as clearing and burning of biomass on land (e.g., deforestation) to convert it for the Project, emitting GHGs, and restoration through replanting (e.g., reforestation) to convert land back to forested land, which will contribute to GHG sequestration over time.

Greenhouse Gas Mitigation Hierarchy: Implementation

In an effort to mitigate GHG emissions during the various phases of the Project, Seabridge has already implemented measures that will reduce GHG emissions at the design phase, and plans to implement a variety of other measures to be implemented throughout the Project life. Applying the mitigation measures outlined below will enable

Seabridge to work with its upstream and downstream partners toward reducing the atmospheric GHG emissions for the Project across its value chain. This partnership strategy will enable a cost-efficient method of GHG emissions mitigation for the Project as well as provide a means of reducing associated fuel/energy costs and other co-benefits.

Please refer to Volume 26 (Section 26.12) of the Application/EIS (Rescan 2013) for details on the mitigation measures that will be implemented during all Project phases.

Summary of Project Greenhouse Gas Mitigation Measures

The primary GHG mitigation measures for the Project can be summarized as:

- minimizing Project fuel use (e.g., by equipment, vehicles, and generators) through implementing fuel efficiency/conservation measures;
- minimizing Project energy use (e.g., by facility and electrical equipment) through implementing energy efficiency/conservation measures; and
- minimizing planned land use change clearing/burning and maximizing replanting/sequestration where possible.

There is the potential that the net GHG emissions for the Project could be mitigated significantly compared to those reported in the GHG assessment, depending on technological advances in fuel and energy efficiency measures over the life of the Project, as well as potential carbon offsetting schemes under a potentially regulated regime.

MONITORING AND REPORTING

Sources of GHG emissions (e.g., fuel and energy use) will be monitored, and resultant data will be used for GHG assessments and reporting for the Project as per provincial and federal guidelines and legislation as well as included in annual reports.

Seabridge will annually review GHG emissions associated with the Project, determine if any trends are evident, and will assess progress on targets. The need for any actions to correct or improve on-site GHG emission management will be determined on a case-by-case basis. Indications of the need for corrective actions and additional control measures may include:

- data showing an increasing trend in GHG emissions (with other factors of production held constant); and
- issues being raised by on-site staff, regulators, or local communities.

Discussions will be initiated to resolve issues, and identified solutions will be implemented and monitored for efficacy.

It is possible that components of the GHG Management Plan may need to be revised over the life of the Project, based on regulatory changes and technological advances.
Any modification made to the GHG Management Plan will be reported to regulatory authorities where applicable.

20.6 ENVIRONMENTAL MANAGEMENT SYSTEM

Seabridge has developed a conceptual Environmental Management System (EMS) and associated Environmental Management Plans for the Project. As stated in the KSM Project AIR document approved by the British Columbia Environmental Assessment Office (BC EAO; 2011), Environmental Management Plans are essential for the EMS for any major development.

An EMS is a requirement of a *Mines Act* Permit for mines in BC and is the high-level framework supporting each Environmental Management Plan. Environmental Management Plans are the specific and detailed goals, objectives, and procedures for the protection of worker health and safety, environmental monitoring, and operating procedures that show the regulatory agencies how legislation and regulations will be met at the Mine Site and the PTMA. Environmental Management Plans are managed collectively under the umbrella of the EMS.

Environmental Management Plans are to be applied during the planning, construction, operation, closure, and post-closure phases of the Project.

The EMS will identify the approach to the Project planning and to the development of the Project with respect to Seabridge's legislative and corporate environmental obligations. The fundamental component of the EMS is the Environmental Management Plans, which detail environmental protection measures to mitigate potential environmental effects. The Environmental Management Plan describe the environmental practices and procedures to be applied during the planning, construction, and operation phases of the Project.

It is necessary and prudent planning for projects to have an EMS in place to guide project performance from construction through to closure. An integrated system is required because there are inherent overlaps in activities, e.g., actions taken to protect workers' safety and health often protects the environment. For example, emergency response plans will address spills to the environment that can also have worker and public safety risks. Similarly, traffic safety and driver training programs can significantly reduce risks to workers and to the public, as well as spills to the environment. Appropriate training and other resources will be available to ensure that workers at the Project are properly equipped to perform their work. The Project operating company will develop an overall management system for the Project that includes:

- an organizational structure and reporting structure;
- the assignment of responsibilities;
- communication protocols;
- company policies;
- methods to evaluate environmental practices, procedure, and processes; and
- resource allocation.

The EMS will define Seabridge's environmental approach through all phases of the Project. The EMS is based on prevention, mitigation, and management of effects identified by the company Environment and Safety Superintendents with input from the Table of Conditions (Schedule B) from the EA Certificate (#M14-01) which was issued in July of 2014.

The purpose of the EMS is to organize and guide all activities during all phases of the Project to ensure orderly, safe, compliant, and environmentally and socially responsible operations at the mine. The EMS aims to coordinate human aspects of the Project to control or reduce the Project's effect on the environment (biophysical and human).

The EMS is the framework within which Environmental Management Plans will be developed, implemented, maintained, and updated. The following three-step process has been used for the development of Environmental Management Plans.

- **Step One:** High-level Framework: The high-level framework will commit to specific and detailed goals, objectives, and procedures for producing EMPs. Included in this step is a procedure to re-evaluate monitoring plans, methods, and objectives for adaptive management goals.
- **Step Two:** Production of the formal Environmental Management Plans: May be initiated after the issuance of the EA Certificate, but before construction begins. This step will follow the procedures and commitments developed in Step One.
- **Step Three:** Development of Standard Operating Procedures: Development of the Standard Operating Procedures will fulfill obligations as defined in Step Two.

This three-step process will be used to develop Environmental Management Plans for each phase of the Project.

The following Environmental Management Plans were prepared and submitted as part of the Application/EIS (Rescan 2013). Please refer to the Application/EIS (*Volume 26*) for details on each of these EMPs.

- Construction Management Plan
- Rock Storage Facility Management and Monitoring Plan
- Tailing Management Facility Management and Monitoring Plan
- Water Storage Facility Management and Monitoring Plan
- Domestic and Industrial Waste Management Plan
- Dangerous Goods and Hazardous Materials Management Plan
- Explosives Management Plan
- Emergency Response Plan
- Spill Prevention and Emergency Response Plan
- Air Quality Management Plan

- Greenhouse Gas Management Plan
- Terrain, Surficial Geology and Soil Management and Monitoring Plan
- Metal Leaching and Acid Rock Drainage Management Plan
- Groundwater Management Plan
- Glacier Monitoring Plan
- Water Management Plan
- Fish and Aquatic Habitat Management Plan
- Wetlands Management Plan
- Terrestrial Ecosystems Management and Monitoring Plan
- Wildlife Management Plan
- Noise Management Plan
- Heritage Management and Monitoring Plan
- Visual Quality Management Plan
- Traffic and Access Management Plan.

20.7 CLOSURE AND RECLAMATION

Closure and reclamation planning for the Project will contribute to the success of closure and reclamation during mining and at the end of mine life, which will reduce the need to restructure Project components, limit the amount of material re-handling, and reduce the environmental effects of the Project. Mine development and operation will incorporate techniques to minimize surficial disturbance and, where possible, progressively reclaim areas affected during construction and operation. Stabilizing and rehabilitating surfaces will reduce the potential for degradation of the resources due to extended exposure to climatic factors, reducing closure-related capital costs at the cessation of mining activities.

20.7.1 REGULATORY FRAMEWORK

BRITISH COLUMBIA MINES ACT (1996A) AND HEALTH, SAFETY AND RECLAMATION CODE (BC MEMPR 2008)

The BC *Mines Act* (1996a) and Health, Safety and Reclamation Code for Mines in British Columbia (BC MEMPR 2008) require mining operations to carry out a program of environmental protection and reclamation to ensure that, upon termination of mining, land, watercourses and cultural heritage resources will be returned to a safe and environmentally sound state and to an acceptable end land use. The *Mines Act* (1996a) and Health, Safety and Reclamation Code (BC MEMPR 2008) are administered by the Ministry of Energy, Mines and Natural Gas (MEMNG), now called the Ministry of Energy and Mines (MEM as of May 2013). The Chief Inspector of Mines has the ultimate legislative authority for all issues pertaining to the *Mines Act* and Health, Safety and Reclamation Code.

Proponents of mining projects are required to obtain a permit from the MEM prior to commencing any work on a mine site, in accordance with *Section 10* of the *Mines Act* (1996a). *Section 10* of the *Mines Act* requires that a permit application must include:

> *…a plan outlining the details of the proposed work and a program for the conservation of cultural heritage resources and for the protection and reclamation of the land, watercourses and cultural heritage resources affected by the mine, including the information, particulars and maps established by the regulations or the code (Section 10.1).*

As a condition of issuing a permit, the chief inspector may require a security for mine reclamation, and to provide for protection of, and mitigation of damage to, watercourses and cultural heritage resources affected by the mine (*Mines Act, Section 10.4*).

A financial security is required as a condition of all *Mines Act* permits (*Section 10.4 and 10.5*) for all, or part of, outstanding costs associated with mine reclamation and the protection of land, watercourses and cultural resources, including post-closure commitments. The security held under the *Mines Act* can also be used to cover the regulatory requirements of legislation, permits and approvals of other provincial agencies.

The objective of BCs reclamation security policy is to provide reasonable assurance that the provincial government will not have to contribute to the costs of reclamation and environmental protection if a mining company defaults on its obligations. In the case of a company default, the security should allow government to successfully manage the environmental issues at the mine site, complete any outstanding reclamation requirements, and continue to monitor and maintain the site for as long as is required (BC MEMPG 2009). In general, MEM reviews reclamation security at a mine site every five years, or whenever significant changes occur at the mine. The security can increase or decrease depending upon assessed liability at the time and financial factors such as real return bond yields.

FISHERIES ACT (1985)

Fish and fish habitat are protected under the *Fisheries Act* (1985), as well as other federal regulatory acts and principles. In 2012, the *Fisheries Act* was amended to establish into legislation the federal government's direction to focus efforts on protecting the productivity of commercial, recreational, and Aboriginal fisheries; to institute enhanced compliance and protection tools that are more easily enforceable; to provide clarity, certainty, and consistency of regulatory requirements; and to enable\enhanced partnerships with stakeholders. The changes to the *Fisheries Act* include a prohibition against causing serious harm to fish that are part of or support a commercial, recreational, or Aboriginal fishery (*Section 35*), provisions for flow and passage (*Sections 20 and 21*), and a framework for regulatory decision-making (*Sections 6 and 6.1*).

On November 1, 2013, The Fisheries Protection Policy Statement (DFO 2013a) was issued and replaced the earlier Policy for the Management of Fish Habitat (DFO 1986). Although the new policy statement does not include the NNL principle, as outlined in the earlier policy, application of this NNL principle provides some useful guidance when considering "serious harm to fish". Any project or activity that causes a serious harm to fish that are part of, or support, a commercial, recreational, or Aboriginal fishery requires an authorization from DFO. Regulations have been developed to guide the application for this authorization: Applications for Authorization under Paragraph 35(2)(b) of the *Fisheries Act* Regulations. DFO has issued additional guidance in the "The Fisheries Protection Program Operational Approach".

The Project entered the EA process prior to the changes in the *Fisheries Act* and Fisheries Protection Policy Statement. As such, the original *Fisheries Act* and Policy for the Management of Fish Habitat have been used for planning and permitting. The original *Fisheries Act* (1985) legislation and policies are discussed below because they have formed the basis for developing the Closure and Reclamation Plan.

The original *Fisheries Act* (1985) prohibits the harmful alteration, disruption, or destruction of fish habitat through physical, chemical, or biological means. The Policy for the Management of Fish Habitat (DFO 1986) puts forth the principle of NNL of productive capacity" of fish habitat. Under Section 35(2) of the *Fisheries Act* (1985), any project or activity that causes harmful alteration, disruption, or destruction requires an authorization from DFO. The *Fisheries Act* (1985) defines fish habitat as "spawning grounds, nursery, rearing, food supply, and migration areas on which fish depend directly or indirectly in order to carry out their life processes." Included in this definition are both fish-bearing and non-fish-bearing waterbodies, including the surrounding riparian area and fisheries sensitive zones.

The MMER (SOR/2002-222), enacted in 2002, were developed under Section 36 of the *Fisheries Act* (1985) to regulate the deposit of tailing, and other waste matter produced during mining operations, into natural fish bearing waters. These regulations, administered by Environment Canada, apply to both new and existing mines. If a developer proposes to use a natural fish-bearing waterbody for tailing management, a fish habitat compensation plan must be developed to ensure NNL of fish habitat results from the use of this waterbody.

Environmental Management Act (2003)

The BC *Environmental Management Act* (2003) prohibits the discharge of waste to the environment unless specifically authorized. While there are different types of authorizations under the Act, most mining operations require air emissions, solid refuse and effluent discharge permits (MOE 2013).

A permit authorizes the discharge of wastes from an industry, trade, business, operation or activity to the environment, and sets the terms and conditions under which the discharge may occur so that pollution is prevented. The terms and conditions include limiting the quantity and quality of waste contaminants, monitoring the discharge and the receiving environment, and reporting information to the Ministry. Permits are ongoing authorizations and may be amended, transferred to other dischargers, suspended or cancelled (BC MOE 2013).

Closed mines must obtain a Certificate of Compliance pursuant to *Section 53* of the *Environmental Management Act* (2003) and the Contaminated Sites Regulation (BC Reg. 375/96). The Contaminated Sites Regulation (BC Reg. 375/96) sets the applicable standards for soil, air, and groundwater that must be met to obtain a Certificate of Compliance.

Water Act (1996b)

In BC, ownership of water is vested in the Crown as stated in the BC *Water Act* (1996b). The Act is the principal law for managing the diversion and use of provincial water resources. Under the *Water Act*, approvals are required for making changes in and about a stream, to authorize constructions of works, the diversion and use of water and water withdrawals.

Riparian Management Area Guidebook (BC MOF and BC MOE 1995)

The Riparian Management Area Guidebook (BC MOF and BC MOE 1995) sets out activities on erodible soils in riparian areas. These guidelines are also consistent with the spirit of the *Mines Act* (1996a) and the Health, Safety and Reclamation Code for Mines in British Columbia (BC MEMPR 2008).

20.7.2 Closure and Reclamation Objectives

Part 10 of the Health, Safety and Reclamation Code (BC MEMPR 2008) focuses on reclamation and closure. *Section 10.7* identifies reclamation standards. *Section 10.7.4* (Land Use) indicates "The land surface shall be reclaimed to an end land use approved by the chief inspector that considers previous and potential uses." *Section 10.7.5* (Capability) indicates "Excluding lands that are not to be reclaimed, the average land capability to be achieved on the remaining lands shall not be less than the average that existed prior to mining, unless the land capability is not consistent with the approved end land use." *Section 10.7.6* (Long Term Stability) states "Land, watercourses and access roads shall be left in a manner that ensures long-term stability."

The conceptual closure and reclamation plan has three objectives that provide assurance to the Province that the site will be left in a condition that will limit any future liability to the people of BC:

- to provide stable landforms
- to re-establish productive land use
- to protect terrestrial and aquatic resources.

Provision of Stable Landforms

The design of the Project's permanent mine-related landforms, such as the open pits, the TMF, the WSD that will impound the WSF at the Mine Site, seepage containment dams, and the RSFs, has been undertaken to ensure long-term stability during mine operations, after mine closure, and after reclamation works are complete.

Stable landforms require a stable foundation. Field investigations were undertaken to enable pre-feasibility level design of the TMF, RSFs, the Treaty OPC, and the WSF. Geotechnical site investigations comprised geological mapping of tunnel routes, the WSD, and the TMF, as well as drilling of facility foundations. Sampling and geotechnical/

hydrogeological testing for characterization of foundations, borrow material, and tailing materials continued throughout 2012 (*Appendices 4-Q and 4-AB* of the Application/EIS). Investigations of the TMF and Mine Site hydrogeology were carried out. Seismic refraction geophysical surveys were conducted to determine the depth of bedrock at the proposed dam sites. Multi-spectral Analysis of Surface Waves was conducted to estimate the shear wave velocity in the alluvial deposits at the proposed dam sites (Tetra Tech 2012; *Appendix 4-C* of the Application/EIS). A Seismic Hazard Assessment was carried out (*Appendix 4-J* of the Application/EIS). Overburden was assessed at mapping sites and recovered from drill holes

Long-term stability of landforms will also be achieved through careful design. The design of RSFs has taken the BC Mines Waste Rock Pile Research Committee dump design guidelines (Piteau Associates 1991) into account. RSF geotechnical stability during construction and closure was analyzed using SLOPE/W© 2007 software (Tetra Tech 2012). The mine rock placement progression will build the RSFs in lifts (initially by bottom-up construction) to consolidate foundations and reduce downslope risks (KCB 2012). Where implemented, top-down designs in the RSFs is limited to 400 m maximum lifts, which are comparable to current lift heights implemented at other mines in BC, however the current design for interim placement of waste rock does not exceed 200 m.

The RSFs will be reclaimed. Therefore, final dump configurations are designed with maximum 105 m high terraces at “as dumped” angle of repose, with flat benches between terraces. The overall slope angle will be between 26° and 30° to allow for re-sloping to accommodate reclamation. The upper edge of the RSFs will be rounded off to shed water and to reduce the potential for erosion at the top of slope edges. Reclamation/re-vegetation of the RSFs will reduce the potential for infiltration of precipitation and surface erosion providing more stability.

The TMF will be constructed in three cells, known as the North Cell, the Centre Cell, and the South Cell as described in Chapter 18.2.

RE-ESTABLISHMENT OF PRODUCTIVE LAND USE

The pre-development land use and conditions form the basis for setting the end land use and capability objectives. The goal is to return the site to a use consistent with the current land uses. The current land use information has been obtained from the environmental and socio-economic baseline studies, which were initiated in 2008 and continued into 2012 (*Appendix 23-A* of the Application/EIS) (Rescan 2013).
These studies were undertaken in consultation with the KSM Project EA Working Group which includes provincial and federal government agencies, Nisga’a Nation, Tahltan Nation, Gitxsan Nation, Skii km Lax Ha, and the Gitanyow First Nation.

The end land use objective will be primarily to provide for wildlife habitat for the described wildlife species, including bears, mountain goats, and moose. It is proposed that the reclamation approaches will result in the development of complex ecosystems with time and will provide habitat for the species of animals and plants currently occurring around the proposed Project area. Reclamation will include the development of wetlands with their characteristic vegetation and use by wildlife such as moose, western toads, American martens, and fishers. With time, and with the development of complex ecosystems, suitable habitat for the other wildlife species found in the Project area will develop. Reclaimed areas could be used by guide outfitters and for traplines in the future, where safety concerns are not an issue. Lost fish habitat will be compensated through new habitat being created as part of the Fish Habitat Compensation Plan which is required under Canadian law.

PROTECTION OF TERRESTRIAL AND AQUATIC RESOURCES

The environmental management and monitoring systems that will be in place at the start of mine construction are designed to ensure the protection of terrestrial and aquatic resources. Consistent with best management and adaptive management principles, the Proponent will ensure that management and monitoring systems are updated, as required. Environmental Monitors will be assigned to the various phases of the Project to ensure compliance with applicable permits and the Proponent's environmental policies.

20.7.3 SOIL HANDLING PLAN

Site reclamation must be carried out in areas used for mining (Health, Safety and Reclamation Code for Mines in British Columbia [BC MEMPR 2008]). The general goal of reclamation is to restore, where possible, the equivalent land capability so that end land use objectives can be achieved. To this end, planning will include the conservation of soil materials suitable for reclamation purposes in areas disturbed by mining, and these areas will be re-vegetated, where feasible. The landforms resulting from the Project will also be designed, where possible, and reclaimed to accommodate the desired end land use objective. Reclamation efforts will be directed toward the development of appropriate and functional ecosystems. These efforts will be supported by appropriate soil material handling and re-vegetation strategies.

MATERIALS BALANCE

There are approximately 47 separate facilities that form part of the Project. They range in area from less than 5 ha, such as some of the construction camps, to 1,660 ha for the TMF. The total area that will be disturbed by the end of mine operation will be approximately 4,195 ha. Following the 53-year mine life, the majority of the Mine Site will be closed, decommissioned, and/or reclaimed. Some facilities, such as the pits, will not be reclaimed, but will be decommissioned. Other facilities will remain in operation during the post-closure phase, such as the WSF and the WTP, so they will neither be decommissioned nor reclaimed. The Process Plant and TMF will be closed and reclaimed. Soil will only be required for those facilities that will be reclaimed.

ROCK STORAGE FACILITIES

Reclamation of the RSFs will be focused on the slopes of the constructed RSFs. The surface area of the slopes of the RSFs will be approximately 4.4 Mm^2. The reclamation goal is to have a 50 cm thick soil cover on the slopes to provide sufficient water storage capacity, primarily to support a forested ecosystem. Therefore 2.2 Mm^3 of soil will be required to reclaim the RSFs.

The benches of the Kerr waste rock backfill located below 1,100 m in the Sulphurets pit, based on a 50-cm cover thickness, will require approximately 61,000 m^3 of soil for reclamation. It is expected that this material will be sourced from various areas, such as the borrow pit area for the WSD. This extra soil material will also be stored in the stockpiles located in the Ted Morris Valley.

Tailing Management Facility

Following closure of the TMF, a 50-cm cover of till will be spread on the TMF beaches. The beaches will have a 0.5% slope, so it is assumed that the till thickness will be roughly 50 cm over most of the surface, gradually thinning toward the edges. The beaches will occupy approximately 9,285,633 m^2 or 929 ha. The dam faces (the North and Southeast dams) will occupy approximately 1,363,364 m^2 or 135 ha. The dam crests, which will be used as rights-of-way for wildlife, will occupy approximately 65,881 m^2 or 6.6 ha. Approximately 1,071 ha of the TMF will be reclaimed, including the dam faces and beaches. Based on these plans, approximately 5,364,028 m^3 of soil or till is required for closure of the TMF.

20.7.4 Closure and Reclamation Planning

Introduction

This section describes the closure and reclamation approaches that will be used to close the various Project facilities and reclaim the Project area. It also includes a description of the design of new landforms created to meet the closure goals of minimizing long-term potential effects on the environment, and describes the plans for facilities that will not be closed, such as the WSF. Reclamation will be carried out in most areas. This will include placement of soils and planting of suitable plant species.

Mine Site

Mitchell Pit and Block Cave Mine

Closure

Open Pit

The Mitchell deposit will be mined as an open pit from Year -2 to Year 24 and as an underground block cave mine from Year 23 to Year 53. The open pit operations will be developed using conventional drill and blast methods to break the rock to a size suitable for loading and transportation by haul truck. The overall pit slope angles and bench configurations at the end of the mine life will vary based on wall orientation and stability (refer to Section 16.0 for details). The pit will have an ultimate wall height of approximately 1,230 m. Closure of the Mitchell pit includes backfilling with water to form a pit lake and placing large rocks on the benches to discourage wildlife access to the pit lake. The Mitchell pit cannot be backfilled with water until underground mining is completed.

Underground Mining

The Mitchell block cave mine will extend the depth of resource extraction an additional 180 m below the pit floor. Underground mining will be conducted using the block caving method as described in Section 16.3.

Closure of the Block Cave Mine

At closure, all mobile equipment and supplies will be removed and transported off-site. The major infrastructure, such as crushers, rock breakers, and conveyors will be left in the mine. Oils will be drained from motors, gears, and electrical equipment, and will be removed from the site. Explosives, chemicals, and lubricants will be removed from underground. All electrical cable and piping will be left in the mine. All surface ventilation fans will be removed. All openings to the surface will be sealed with engineered concrete plugs.

By the time block caving is completed, the area within the pit directly above the block cave footprint will have subsided into a block cave crater as described in Section 16.3.

Closure of the Mitchell Pit

The Mitchell pit will be partially flooded when underground mining is completed. Flooding of the pit will start after the Block Cave has been closed and will take five years. Approximately 320 Mm^3 of water will be required to flood the pit to an 810-m elevation. The water will be supplied from three sources. The water collected from under the Mitchell Glacier will provide approximately 88% of the water to flood the pit. Approximately 9% of the water used to flood the pit will be supplied from the Mitchell NPWDA. This adit will also collect contact water from the toe of the Mitchell Glacier. The remaining 3% of the water used to flood the pit will be supplied from precipitation and non-contact runoff. Precipitation is estimated at 2%, and surface runoff is estimated at 1%.

The Mitchell pit closure dam will be constructed on the west side of the Mitchell pit to allow for controlled discharge. The Mitchell pit closure dam will be largely constructed with mine rock from the Mitchell RSF and forms part of the RSF. It will have an acid-resistant low permeability core keyed into shallow bedrock near the pit rim. The dam will be located outside the zone of instability caused by subsidence of the block cave works. The crest of the closure dam will be constructed to an 870 m elevation. This will provide 60 m of freeboard above the normal pit lake. This freeboard will accommodate potential impact waves caused by landslides (e.g., a potential failure of the north or south pit wall slopes, initiated from above the pit lake) or avalanches that may enter the pit lake

Storm flows from events of less than 20 Mm^3 will be attenuated and retained in the pit lake and will slowly be bled off to the WSF via the Mitchell pit Lake base flow discharge pipe located on the north side of the dam at elevation 810 m. This water will flow to a lined channel (North Slope collection ditch for contact water) to the WSF. The Mitchell pit closure dam spillway will route storm flows from the pit lake associated with higher flow events to the North Slope collection ditch, which will be constructed along the north side of the Mitchell Creek Valley. The North Slope collection ditch will route extreme floods to the WSF.

As noted above, large rocks and debris will be placed on the exposed Mitchell pit benches to reduce the potential for wildlife access to the pit lake.

The Mitchell NPWDA will be 4.5 km long. It will no longer be required when the pit is flooded. However, the NPWDA promotes dewatering of the pit slope, which will add to slope stability. The tunnel will be left to drain. It will be plugged with a granular plug with a low enough permeability to back up a few metres of water in the tunnel to prevent acid rock drainage in the tunnel, but the plug will not be impermeable enough to hold back groundwater from the 500-m-high pit slope.

Ttwin 6 km long drainage tunnels (the Mitchell underground drainage tunnels) will be used to dewater the underground works during operation. These tunnels will be approximately 7.5 m high by 7.5 m wide. At closure, each of these tunnels will be sealed with an engineered concrete plug.

Sulphurets Pit

Closure

Once the Sulphurets pit has been mined out in Year 17, mining operations will shift to the Kerr pit. The Sulphurets pit will then be backfilled with the waste rock from the Kerr pit. The Kerr waste rock is predicted to have elevated selenium concentrations, so it will be placed in the Sulphurets pit to allow for the management of selenium.

The Sulphurets pit backfill will be constructed from the bottom up. Basal drain material will be placed in the bottom of the Sulphurets pit to provide drainage of the base of the pit and any water that has moved through the Kerr backfill. Kerr waste rock will then be placed in the pit in 50 m lifts with 22 m-wide benches. The outer edge of each bench will be lined with synthetic liner for a width of 100 m such that 78 m of the liner will be covered by each subsequent lift. This will provide an internal barrier to downward movement of water within the backfill (waste rock). The benches will have a 2% slope to enable water on the benches to drain out toward the edge of each bench.

A central collection channel will be located on each bench to capture precipitation and water that enters the inter-bench sloping area. The water collected in the central collection channel will be directed along the benches to a 10-m-wide rock-cut step spillway constructed along the Sulphurets pit wall from the top of the pit to the bottom. A channel will be built on the Sulphurets pit bench just above the Kerr backfill. This channel will drain water to the rock-cut stepway to prevent runoff from upper Sulphurets pit walls and benches from flowing onto the lined top of the waste rock. The water from the spillway will flow to a control weir. It will be directed to a 10-m-wide stepped spillway and to a pipeline to the Selenium Treatment Plant.

Precipitation occurring on the lined Kerr waste rock will be non-contact water. It will be directed to an existing stream adjacent to the top of the waste rock. Once the backfill is completed, more than 95% of the surface will be covered with a liner.

The upper Sulphurets haul roads will be decommissioned when mining of the Sulphurets pit is completed and when the pit has been backfilled. These roads have extensive areas of cut and fill. The cut and fill areas will be re-graded to provide stability to the road edges. All culverts will be removed to allow for the restoration of natural drainage. The road surfaces will be cross ditched to allow drainage across the surface, and they will be ripped to reduce the potential for surface erosion. Culverts will be removed off-site for reuse or recycling.

Similar treatment of the lower haul road will be carried out. However, this road will be required to monitor the Kerr waste rock backfill. Therefore, the road will be narrowed from 38 m to 15 m and ditched according to forest road standards.

Reclamation

The Sulphurets pit benches will be reclaimed with a 30-cm layer of protective gravel that will be placed on the liner surface, and with a 30 cm soil cover layer that will be placed over the top of the gravel layer. These benches will be re-vegetated with native grass seed mix. Pocket trees of subalpine fir and hybrid white spruce will also be placed in the inter-bench area.

Kerr Pit

Closure

The Kerr pit will be located on the south slope of the Sulphurets Valley, directly south of the Mitchell Valley. The pit will have steep sidewalls and benches. The inter-bench slope angle will vary and pit slopes will be approximately 600 m high of continuous slope in any area (refer to Section 16.0 for details). The ultimate pit will be approximately 2 km by 1 km. The pit will be separated from the Mitchell OPC by the Sulphurets Creek Valley and Sulphurets Ridge.

The ore and waste will be transported across the Sulphurets Creek Valley to the Sulphurets pit by the Kerr rope conveyor. At the end of mining, the Kerr rope conveyor will be dismantled and moved off-site, where parts will be reused or disposed of in an appropriate facility.

During operation, the Kerr pit will have a dewatering system designed to route contact water from the Kerr pit to treatment. The system will include a pipeline and inlet ponds that attenuate the peak and convey the 200-year 24-hour flood event. The pipeline will route the floodwater down through the South Sulphurets Creek Valley, across the Sulphurets Creek bridge, past the HDS WTP and Selenium WTP to the WSF. The pipeline will be a HDPE-lined steel pipe, buried where possible.

The Kerr pit external haul roads will be decommissioned. The cut and fill slopes will be re-graded for stability, where required. The culverts will be removed, and cross ditches will provide drainage. The surface will be ripped to reduce surface erosion. Any available stored topsoil material will be spread on the surface and re-vegetated. The access road will be retained to permit ongoing inspection of the pit.

Sulphurets-Mitchell Conveyor Tunnel and Associated Overland Conveyors

Closure

A 3-km long SMCT will be used to transfer crushed ore from the Kerr and Sulphurets pits and waste rock from the Sulphurets pit to the Mitchell OPC. The tunnel will be 5 m high and 6.5 m wide. A belt conveyor will be located inside the tunnel.

At closure, this tunnel will be dismantled and all mobile equipment and supplies will be removed from the tunnel. The non-salvageable electrical cables and conveyor will be left in the tunnel. The south portal, located adjacent to the Sulphurets pit, and a north portal, located west of Mitchell pit, will both be sealed with engineered concrete plugs.

The overland conveyors at the Sulphurets pit (conveyors 1 and 2 to the Sulphurets portal) will be dismantled. . These conveyors will be moved off-site where their parts will be reused or disposed of in an appropriate facility.

Iron Cap Block Cave Mine

Closure

<u>*Operation*</u>

The Iron Cap deposit will be mined by underground block caving from Year 32 to Year 53. The Iron Cap deposit is at an elevation of 1,210 masl. It is approximately 545 m long in the north-south direction, 570 m wide in the east-west direction, and has an average depth of 400 m. Mining activities will be conducted as described in Section 16.2

<u>*Reclamation*</u>

At closure, all mobile equipment will be removed from underground and taken off-site to be sold or recycled, or disposed of in an appropriate facility. Major equipment and infrastructure such as the crushers, rock breakers, conveyors (including belts), electrical cable, and piping will be left in the mine. Oils will be drained from the motors, and electrical gears will be removed from the site. Explosives, chemicals, and lubricants will be removed from underground. All surface ventilation fans will be removed, and all openings to the surface will be sealed with engineered concrete plugs, with the exception of the return air drifts.

At the completion of mining, the surface inflow water will continue to enter the crater and flow downward through the abandoned works. Drainage from the Iron Cap Underground Works will drain directly from the lower-level ventilation tunnels of the workings by raise bores into the Mitchell NPWDA. This water is considered contact water and will flow through the MVDT and into the WSF, pending treatment in the HDS WTP. The system is designed to accept the 200-year peak flow. The underground drifts will collapse following mining, creating a surface disturbance of approximately 96.4 ha.

Mitchell and McTagg Rock Storage Facilities

Closure

The Mitchell and McTagg RSFs represent two separate but adjacent RSFs that will be joined by a land bridge until the middle of the operation phase. The land bridge will allow waste rock to be hauled from the Mitchell RSF to the McTagg RSF. The land bridge will be removed at the end of the Sulphurets open pit operation.

Following the open pit operation, but before closure, two channels will be constructed at an 810-masl elevation along the north side of the Mitchell RSF. One will handle non-contact water (Mitchell North closure channel), including diverted water from the East McTagg closure channel, and the other will handle contact water, including the outflow from the Mitchell pit Lake. The fresh water (non-contact) diversion channel will be lined and located on the upslope side of the contact water channel. This will route north-slope runoff water around the WSF. The fresh water diversion will provide a source of water to wildlife. The contact water will flow to the WSF.

The Mitchell RSF will be constructed to a final elevation of 1,200 m, and the McTagg RSF to 1,020 m based on the planned mining activities (Section 16). The Mitchell RSF will be located on land that has a mean slope of 26º and a maximum slope of 62º. The McTagg RSF will be located on land that has a mean slope of 30º and a maximum slope of 57º.

A basal drain is constructed underneath portions of the Mitchell and McTagg RSFs.

The tops of the RSFs will be used to construct secure landfills to store sludge from the HDS WTP, for over 200 years. The sludge will be placed in 200-m-by-100-m cells, which will be 8 m deep. These cells will be continuously built up. A till/rock containment berm will be constructed around the edge of the landfills. A runoff collection channel will be located between the containment berm and the landfill, and this water will flow to the WSF. The landfill will be lined so that minimum seepage and precipitation will enter the RSF. The landfill will be constructed such that the surface will be covered with coarse rock.

The Mitchell RSF ore stockpile will be located along the northern edge of the Mitchell RSF (Figure 27.5-16 in Chapter 27 of the *Application/EIS*). It will occupy approximately 70 ha, will be constructed starting in Year -1, and will have been totally processed by Year 52. The maximum stockpile size is between 145-150M tonnes. There are two peak periods, one in Year 7 and the other in Year 16 of production.

A selenium seepage capture system has been designed for the Mitchell and McTagg RSFs. By Year 5 a seepage collection system incorporating a rock drain overlying a low permeability layer will be established starting at elevation 706 m at the toe of the RSFs above the spillway elevation of the WSF. Flows captured by the seepage collection system will vary seasonally. The system is designed to convey up to 500 L/s to a Selenium WTP located near the WSD. During months when RSF seepage captured is lower than the Selenium WTP Capacity of 500 L/s, flow to maximize treatment capacity will come from the WSD.

Closure

The Mitchell OPC is located west of the Mitchell pit. The Mitchell OPC will include facilities for rock crushing, coarse ore storage, and fuel storage, as well as an ore stockpile feed conveyor, an electrical substation, and roads. It will also be the location of the southern portals of the MTT and the Mitchell underground access and conveyor adits. The Mitchell OPC will occupy approximately 44 ha.

At the completion of mining, the Mitchell OPC will be decommissioned. Equipment will be removed from the site. The electrical substation will remain. All other structures will be dismantled and removed. Foundations will be broken up, and the concrete rubble will be buried on site. Any soils that are contaminated with fuel will be excavated and treated at a landfarm to remediate the soil.

Reclamation

The Mitchell OPC ground surface will be highly compacted as a result of ore processing activities. Therefore, the ground will be ripped in two directions to promote drainage. Once the surface has been ripped and covered with crushed rock, the area will be covered with up to 50 cm of topsoil. The reclaimed area will then be re-vegetated with native species

Mitchell Diversion Tunnels

Closure

The MDT will divert water from the Mitchell Glacier and the surrounding catchment during operations. This water will be discharged into bedrock stepped spillways and into Sulphurets Creek. This will prevent water from entering the Mitchell pit during open pit and underground mining. These diversion tunnels will also be used to generate hydro-electric power from the Upper Sulphurets Power Plant during operations. The diversion tunnels will continue to be used for hydro-electric power generation during closure, except when the water is being diverted to the Mitchell pit. The electricity generated will be used to operate the HDS WTP.

McTagg Diversion Tunnels

Closure

The MTDT will direct non-contact, glacial meltwater and runoff to Gingras Creek, which drains into Sulphurets Creek. The diverted flows will be used to generate hydro-electric power during operations. Power will continue to be produced during the closure phase. The McTagg Power Plant will be located east of the Gingras Creek bridge, and will be maintained indefinitely to generate electricity. The electricity will be used to operate the HDS WTP, or will be sold for use in the provincial electricity grid.

Water Storage Facility and Water Treatment Plant

The WSF will remain in service after mine closure to continue collecting contact water that requires treatment.

The HDS WTP is a high-density-sludge lime treatment facility. Sludge produced from the HDS WTP will be dewatered, producing a filter cake with a water content of approximately 40%. Approximately 165,000 m^3 of filter cake will be produced annually.

The HDS WTP and support infrastructure will remain in operation during the closure and post-closure phases. The plant will operate primarily in the spring, summer, and fall months, and minimally in the winter. The lime material will be transported to the site, and will be consumed during these warmer periods. At closure and post-closure, the filter cake (sludge) will be hauled by truck during the summer to the top of the RSFs and placed in an engineered landfill.

An ion exchange Selenium WTP located near the WSD will remain in service after mine closure.

Mitchell Truck Shop

Closure

At closure, all equipment and supplies will be removed and disposed of off-site. All oils and fuels will be removed from storage facilities and disposed of at an approved waste oil recycle facility. The electrical, lighting, and heating systems will be removed. Once the buildings have been emptied, they will be dismantled. Metal and any other material that can be recycled will be taken off-site for recycling. Demolition materials will be taken off-site and disposed of in a regulated facility. Flammables will be incinerated and some waste materials will be landfilled. Foundations of buildings will be broken up, and the concrete rubble will be buried on site where it will be used for road bed materials or placed on the RSFs before they are reclaimed. The ground surface will be sampled and analyzed to determine the degree of hydrocarbon contamination in high use areas. All contaminated soils will be excavated and treated in a landfarm facility.

Reclamation

The site will then be ripped to 30 cm in two directions to remove compaction and to allow for downward surface drainage. Soils previously salvaged from the area and stockpiled at the site will be spread on the surface. Care will be taken not to compact the soils. Any compacted areas will be lightly ripped. The area will be planted with native grasses, trees, and shrubs.

Mitchell-Treaty Twinned Tunnels

The MTT are approximately 23-km long, and will be used to provide a transportation route for ore, personnel, and supplies from the Mine Site to the PTMA. The system utilizes specialized electric trains with under computer control to manage the traffic and logistics in the twin tunnels. Details are provided in Section 18.3.

At closure and post-closure, the tunnels will be required to provide ongoing access to the Mine Site because the CCAR, which serves the Mine Site during operation, will be decommissioned. As the HDS WTP will continue to operate post-closure, lime will be required and will be transported from the PTMA through the tunnels to the Mine Site.

All supplies for monitoring, maintenance, and the operation of the HDS WTP will be transported through the tunnels. The train logistics system will be reconfigured and simplified to handle this much reduced traffic requirement.

Processing and Tailing Management Area

The PTMA will be located in the Upper Treaty and Teigen Creek areas. The processing area will include numerous structures related to ore processing as well as other facilities (Figure 27.5-19 in Chapter 27 of the *Application/EIS*) (Rescan 2013). The following is a description on how these facilities will be closed and the areas reclaimed. Water treatment for CIL water will be maintained on stand-by during closure until it is clear that it will not be required.

Treaty Process Plant, Carbon-in-Leach Plant, and Other Structures

Closure

There will be several structures at the PTMA that will be closed at the completion of mining. These include the Treaty Process Plant, the CIL Plant, the Treaty OPC waste management facilities, the Treaty OPC Batch Plant, the Crusher Building, and several other structures such as the warehouse and lab. All of these structures contain equipment that must be removed at closure. The Treaty Process Plant and CIL Plant include equipment such as:

- cone crushers;
- high pressure grinding rolls;
- ball mills;
- copper-gold/molybdenum bulk flotation cells;
- concentrate dewatering equipment; and
- leaching equipment.

At closure, any remaining processing chemicals will be moved off-site and disposed of in a designated facility. All oils and lubricants will be removed from equipment and moved off-site to a designated disposal facility. The equipment will be dismantled, and components will be taken off-site for reuse, recycling, or disposal in an on-site landfill. Electrical wiring and any other electrical components will be removed and taken off-site for disposal.

Once all of the equipment has been removed, the buildings will be dismantled and the materials will be moved off-site for recycling or disposal. Any contaminated soil will be collected and placed in the landfarm. Any materials that can be incinerated will be incinerated on-site. The concrete foundations of the various buildings will be broken up, buried, and used for road maintenance or as armouring in TMF reclamation. All equipment and debris will be removed from around the structures.

Reclamation

The footprint areas and the area surrounding the buildings will be deep-ripped to reduce compaction and to improve surface drainage. Approximately 30 cm of soil will be spread over the area as the surface material will be native soil that will provide additional rooting depth and soil water storage capacity for vegetation establishment and growth. Care will be taken not to compact the soil. The area will then be re-vegetated with native plants.

Coarse and Fine Ore Stockpiles

Closure

All of the ore in the coarse and fine stockpiles will be processed. The footprint areas will be cleaned and ripped to reduce compaction and to increase downward drainage.

Reclamation

The footprint areas will be covered with a lime mixture and topsoil, and then vegetated with native plants as described for the RSFs. The high traffic areas around the stockpiles will likely be compacted. These areas will also be ripped to reduce compaction and increase drainage. Once the sites are prepared, 30 cm of topsoil will be spread over the surface as the stockpiles will have been placed on native soils. The areas will then be re-vegetated with native plants.

Treaty Ore Preparation Complex Batch Plant Stockpile

Closure

The Treaty OPC may contain stockpiled materials left over from operations consisting of sands and gravels. Any remaining materials will be used for road maintenance or as rip-rap along the beach edges in the TMF. Once the materials have been removed, the site will be reclaimed.

Reclamation

The surface and surrounding high traffic areas will be ripped to 30 cm depth to reduce compaction and to improve drainage. Remnant coarse fragments will likely remain, so a 50 cm layer of soil will be spread on the surface over the stockpile area, grading to 30 cm over the high traffic area occurring on native soil. Care will be taken to not compact the soils. Once the site is prepared, the site will be re-vegetated with native plants.

Temporary Water Treatment 8 - Unlined Muck Pad

Closure

The muck pad that will be developed for the construction of the MTT process plant portals will be reclaimed once portal construction is completed in the construction phase. The coarse rock material from the muck pad is predicted to be non-potentially acid generating. The pad will be re-graded at closure.

Reclamation

The high traffic area around the muck pad will be ripped to reduce compaction and to improve soil drainage. Once the pad and surrounding area have been prepared, a 50-cm layer of soil will be spread on the pad footprint and will be graded to 30 cm on the adjacent area. The pad footprint will also be re-vegetated with native plants.

Laydown Areas

Closure

There will be two laydown areas in the PTMA: one large construction laydown area and a smaller MTT portal laydown area, which will be reclaimed during construction. A helipad will also be located on the large construction laydown area. These laydown areas will be covered with gravel to allow for trafficability while they are used. The helipad will also have a gravel pad.

Before reclamation, both areas will be inspected for hydrocarbon contamination because equipment and vehicles will be stored in these areas. In stained areas, the gravel will be pushed aside and stained soils will be excavated and placed in the landfarm. Hydrocarbon-covered gravel will also be placed in the landfarm.

Reclamation

The laydown areas will be ripped to 30 cm to reduce compaction and to improve surface drainage. They will then be covered with 50 cm of soil and re-vegetated with native plants.

Landfarm

Closure

The landfarm will be closed in the post-closure phase because it will be used during closure for treating any contaminated soils that were excavated during the construction, operation, and closure phases. When the landfarm is no longer required, the soils will be checked to assess if treatment has occurred. When no further treatment is required, the landfarm will be closed.

Reclamation

Once the land-farmed materials have been treated, the area will be covered with 30 cm of soil and will be re-vegetated with native plants.

Structures Required Post-closure

Some structures in the PTMA will be required for on-going monitoring of the Project post-closure. These include:

- the office complex
- the ambulance building
- substation 1

- the operating camp incinerator
- the administration building
- the operating camp (reduced in size from the operation phase).

As described above, the MTT will remain open because the HDS WTP will continue to operate post-closure, and reagents, including lime and personnel operating the HDS WTP, will be transported through the tunnels. A smaller camp will be required to accommodate personnel.

Tailing Management Facility

Closure

The TMF will be developed with three separate ponds as described in Section 18.2.

The South Cell will be bound by the Saddle and Southeast dams. It will operate between Year 25 and Year 53. Similar to the North dam, the Southeast dam will have a starter earth fill dam and an inner till core, and will be built up with cycloned, coarse rougher tailing.

Prior to closure, each of the cells will contain separate ponds. Beaches will be developed in the North and South cells as the tailing are deposited. The CIL tailing containing most of the sulphides will remain submerged. The sulphide tailing will be introduced subaqueously so that there will be no beaches constructed with CIL tailing.

The North Cell will be closed roughly five years after tailing deposition into it ceases. The timeline is based on the predicated time required for tailing water quality to improve sufficiently to satisfy water quality standards set for TMF discharges. For closure, the exposed beach will be expanded, decreasing the open water portion of the cell and reducing the potential for erosion of the dams. In order to minimize the need for dredging in constructing the beaches, strategic deposition of tailing will occur during late-stage North Cell operations. The majority of beach shaping will be done by moving the spigot points onto the beaches during final operation. Additional beach area will also be created by pumping down the pond level at closure. The water will be discharged to Treaty Creek. The objective is to increase the distance from the pond to the dam to reduce the potential for overtopping and erosion of the dam after closure, to reduce the flotation pond volume to the minimum possible at closure, and to provide additional wildlife habitat development.

The South Cell will be closed by means similar to those used to close the North Cell, with the beach surface increased to reduce potential effects on the dams and to provide additional wildlife habitat. The increased beach area will result in a reduced amount of open water in the cell.

The Centre Cell will be the last cell to be closed. It is predicted that the water quality in the cell will satisfy water quality discharge standards approximately five years following mine closure. The CIL tailing in the CIL Lined Pond will be covered with approximately 1 m of non-reactive rougher tailing. The submerged Centre Cell tailing will remain below water with a water cover of 5 m at closure. The beaches in the Centre Cell will be increased by strategic placing of the spigots. This will reduce the amount of open water in the pond.

Dredging of the channels within the flotation tailing may be required in areas where runoff into the pond has the potential to scour the water-covered tailing, causing turbulence and the suspension of sediment. Rip-rap will be used along these dredged areas to protect them from further scouring. As well, a strip of rip-rap approximately 1 m wide will be placed along the beaches at the water's edge to prevent erosion

A 20-m wide channel will be constructed along the southwest side of the TMF to allow the pond water to flow freely at an elevation of 1,054 masl. Closure pond levels may have to be adjusted, depending on the actual consolidated tailing elevations observed after deposition and settlement. The rock cut spillway channels will be at the same elevation so that, if required, water can be routed either to South Teigen or North Treaty creeks by adjusting the elevation of inlet weirs or control gates at the spillways. This system will allow for spillway maintenance by temporarily routing water to either spillway. Approximately 70% of flow will go to North Treaty Creek, and 30% will be used to maintain South Teigen Creek flows. These ratios are approximate, based on hydrological predictions and fisheries requirements.

The North Cell spillway will be constructed during operation, but it will not be used until final closure. Rock from spillway excavations will provide the rip-rap that will be placed on the beach edges. The South Cell spillway channel will be cut into rock on the west abutment of the Southeast dam. This will allow for the routing of floods from the South Cell beyond the Southeast seepage collection dam to a stepped rock-cut spillway discharging into North Treaty Creek.

During operation, the East Catchment Diversion dam will direct diverted water through the East Catchment diversion tunnel to South Teigen Creek. The diverted water will also be directed north in a buried pipeline (Northeast buried pipeline) to South Teigen Creek. These dams and diversions will be located on steep, sedimentary bedrock slopes with some of the lower portions of the diversion alignments vegetated with alder trees. At post-closure, the East Catchment Diversion tunnel portals will be sealed.

The Northeast diversion ditch will be breached at closure to allow water from the catchment basins to flow into the TMF.

The Southeast diversion ditch is located on steep, talus-covered slopes, with several large creeks crossing the diversion alignment. The slope of the catchment is typically 2H:1V to 3H:1V, with little underbrush or tall trees. Part of the Southeast diversion ditch will be an open channel. Two buried pipelines will occur in the downstream half. During operation, catchment water will be diverted north to South Teigen Creek. This diversion will be breached at post-closure, once the North Cell closure spillway becomes operational and allows flow from its catchment area into the TMF.

The North Cell and the Southeast seepage collection dams will remain in place post-closure. Water will be pumped back into the TMF pond until seepage collection pond water quality meets regulatory permit requirements. The seepage ponds downstream of the seepage collection dams will become wetlands.

Reclamation

Following operations, the TMF will be reclaimed to provide for wildlife and wetland habitat. The dams and beaches of the TMF will be reclaimed in stages, with the North Cell being reclaimed during operation, the South Cell during closure, and the Centre Cell during post-closure.

A 50-cm soil cover will be placed on the dam faces over the rip-rap so that the dam faces can be re-vegetated. The surface will be placed roughly on the dam face to reduce potential surface erosion. The dam faces will be planted with native grasses.

The crests of the North and Southeast dams will be 20 m wide. These will operate as wildlife corridors across the valley. Soil covers will be deposited to a depth of 60 cm, to support large trees for shelter and screening of wildlife without risk to the dam crests. These crest areas will be vegetated to a forest cover. Tree seedlings will include subalpine fir and hybrid white spruce, as well as grasses and shrubs, as described above.

Reclamation of the beaches will not be carried out until the beach surface is sufficiently drained to allow for the use of heavy equipment. A 50-cm cover of till will be spread on the TMF beaches. The beaches will have a 0.5% slope, so it is assumed that the till thickness will be roughly 50 cm over most of the surface, gradually thinning toward the edges.

ACCESS ROADS

Treaty Creek Access Road

During closure and post-closure, the TCAR will provide the only remaining road access to the Project site. All materials and personnel will be transported via the TCAR, which will extend to the Treaty OPC and the MTT portals, which will all remain open to allow for access to the Mine Site post-closure.

Coulter Creek Access Road

Closure

The CCAR will be decommissioned post-closure. The bridges will be dismantled, and materials that are combustible will be burned. Concrete will be broken and used as rip-rap along the creeks, if required, to reduce potential surface erosion that could occur during the dismantling of the bridges.

Culverts will be removed to restore natural drainage patterns. Cross-ditching will provide drainage across roads and will reduce the potential for surface erosion. The surface of the road and any compacted areas will be ripped, where required, to promote surface drainage and to reduce runoff and potential road bed failure.

Reclamation

Some soil may be salvaged when the road bed is being constructed. The stockpiled soils will be stored near the sites where they have been salvaged so that they can be spread when needed for reclamation. They will be vegetated with native seed mix and tree seedlings.

QUARRY AND BORROW SOURCES

Closure

The borrow areas will be cleared and grubbed. The quarry and borrow areas will be re-sloped and re-contoured to ensure escape routes for wildlife and to restore natural landscape.

Reclamation

Borrow sites located in granular material will be reclaimed. Stockpiled topsoil will be spread and the areas will be re-vegetated as described for the RSFs. Quarries developed in bedrock will not be reclaimed.

EXPLOSIVES

Closure

Explosives for the Project will be manufactured on site. The Explosives Manufacturing Facility, the explosives magazine, and the prilled ammonium (AN Prill) storage area will be located at separate sites. All three sites will be located in the Ted Morris Valley.
The manufacturing of explosives will be planned to minimize the accumulation of excess AN Prill and fuel at the end of mining.

All surfactant, oils and fuels, chemicals, and diesel storage tanks will be removed from the manufacturing facility and disposed of in a designated facility. All equipment will be removed. Wiring and transformers will be removed from the site and will be reused or recycled. The Explosives Manufacturing Facility will be dismantled, and any recyclable materials will be taken off-site. Materials that can be incinerated will be treated on site. Other materials will be disposed of in the on-site landfill, if suitable for the landfill. The concrete foundations will be broken up, and the concrete rubble will be buried on site, spread on the RSFs, or used where required. All contaminated soils will be taken to the landfarm facility. The gate and fence around the explosives manufacturing compound will be dismantled and sent off-site for recycling or reuse.

The building footprint and surrounding parking areas will be compacted as a result of the use of the site. In preparation for reclamation, the surface will be ripped in two directions to a depth of 30 cm to increase drainage.

The AN prill storage area will be closed. Any remaining prill will be used as fertilizer or will be disposed of off-site in a regulated facility. All electrical wire and transformers will be salvaged or will be removed from the site and reused or recycled. The building will be dismantled and its materials recycled, incinerated, or disposed of in the landfill, if suitable, as described above. Contaminated soils will be treated in the landfarm. The site and adjacent areas will be deep-ripped in two directions, as described above.

The explosives magazine will be removed at closure. Any remaining explosives will be detonated on site in a regulated manner to minimize impact to the environment. Similar to the other facilities, the building will be dismantled. The site will be ripped to reduce compaction.

Reclamation

The soils that will have been salvaged and stockpiled during construction of each of the facilities will be spread over the surface and vegetated with native plants as described for the RSFs. With time, the vegetation community will become more complex as native plants growing in the adjacent areas gradually move into the reclaimed areas.

Mini Hydro Plants and Energy Recovery

Several energy recovery and mini-hydro plants are included in the Project development plan. These plants generate electrical power by making use of facilities already included in the Project, resulting in significant net Project energy savings. All of the plants, similar to small independent power producer hydro-electric plants, will operate unattended and will be automatically controlled by PLC systems. The power will be fed into the local mine distribution transmission lines. The power plants will be used to provide electricity for the Project's closure requirements, or the electricity may be sold back to BC Hydro under the Standing Offer Program.

Water Treatment Plant Energy Recovery

Water pumped from the water storage pond to the HDS WTP will generate electric power. A small impulse Turgo-type turbine will be used. The output may be fed into the plant power distribution system at the HDS WTP. This facility will continue to operate after mine closure.

Mitchell Diversion Hydro – Upper Sulphurets Power Plant

The Upper Sulphurets Power Plant will make use of the normal (but not flood) stream flows that will be diverted around the mining operations by the MDTs to produce energy. The installation will consist of a Pelton turbine and will be very similar to independent power producer run-of-river hydro plants because it will make use of natural flows, unimpounded by water storage facilities or any other works other than those required to divert water around the mine. The equipment will be housed in a small powerhouse building near Sulphurets Creek. Power will be delivered to the electrical distribution system. This plant will continue to operate after mine closure.

McTagg Diversion Hydro – McTagg Power Plant

The McTagg Power Plant will generate energy in a manner similar to the Upper Sulphurets Power Plant. The McTagg Power Plant will be constructed in Year 10, once the diversion tunnel inlets of the MTDT are raised in Phase 2. It will consist of two Pelton turbines and will feed power into the plant distribution system at the HDS WTP. This facility will continue to operate after mine closure.

MUCK PADS AND SEDIMENT PONDS

Closure

There are a number of muck pads and sediment ponds located in the Project area. The treatment ponds will be cleaned out, sediment will be hauled for disposal to either TMF or within the RSFs and the pond excavations will be backfilled with NPAG rock to bring the surface to grade. The berms of the ponds will be pulled into the pond area to provide a level surface.

Reclamation

The muck pads will be covered with a layer of soil 30 to 50 cm deep, and will be re-vegetated with native plants. The surface of the muck pads will be rough, and some of the soil placed on these facilities will fall between coarse fragments, resulting in a variable thickness of soil. This will result in plant establishment that will be successful in some areas while other areas will remain un-vegetated. The soils will be vegetated with the native grass mix described for the RSFs.

Approximately 50 cm of soil will be spread over the surface of the backfilled sediment ponds. The areas will then be re-vegetated with species consistent with the adjacent vegetation. Care will be taken not to compact the soil cover. If it becomes compacted, it will be lightly ripped to provide for root establishment. These areas will be vegetated with the native grass seed mix.

CONSTRUCTION AND OPERATION CAMPS

Closure

There will be 2 operating camps (the Mitchell Operating Camp and the Treaty Operating Camp) and 10 construction camps. The camps will range in area from less than 1 ha to a maximum of 16.9 ha, with most camps occupying less than 5 ha.

There will be two operating camps, the Mitchell operating camp, which will accommodate 350 people, and the Treaty operating camp, which will accommodate 250 people. The Treaty operating camp will be reduced in size at closure, and will then be similar in size to the current Eskay Creek Mine operating camp. It will be used to support ongoing closure operation and monitoring.

The camps will generally include portable trailers, an incinerator, materials and equipment storage areas, a helicopter pad, a helicopter fuelling area, fuel storage, a septic field, water/sewage treatment, and diesel generators. The portables will be set up so that they can be dismantled and used at the different sites, as required.

For closure, the portables and all equipment and buildings will be removed. Fuels will be drained from the generators and tanks, and will be disposed of in a regulated disposal facility. The generators and fuel tanks will be removed from the site for reuse or recycling. The soils in the fuel storage areas will also be checked for hydrocarbon contamination. If hydrocarbon contamination is found, soils will be excavated and transported to the landfarm for remediation.

All construction and operation camp sites will be reclaimed to a slope compatible with the surrounding natural topography. The high traffic areas will be ripped in two directions to increase surface drainage and to allow for deeper root penetration.

Reclamation

Prior to construction, topsoil will have been salvaged from the camp site areas and stockpiled along the edges of the camps. At closure, this soil will be spread over the disturbed areas. These areas will then be re-vegetated with the native grasses, shrubs, and tree seedlings that were described for the RSFs. The tree seedlings will be selected based on the tree species occurring in the vicinity. It is predicted that with time, the vegetation community will become more complex as native plants growing along the edges of the camps sites will also naturally re-establish within the reclaimed areas.

Closure of the gravel helipads will entail topping the gravel with 50 cm of soil in order to provide sufficient moisture holding capacity for replanted vegetation and adequate rooting depth for tree seedlings. Approximately 20 cm of topsoil will be spread in areas that occur on native soils. These areas will also be re-vegetated with the native plants and tree seedlings described for the RSFs.

21.0 CAPITAL AND OPERATING COST ESTIMATES

21.1 INITIAL CAPITAL COSTS

An initial capital of US$5.005 billion is estimated for the Project, based on capital cost estimates developed by the following consultants:

- MMTS: open pit mining, mine roads, ore trains, and infrastructure and water diversion tunnels
- Kambert Civil Consulting Ltd. (KCC): costing of water management structures and the TMF, designed by KCB
- EBC Inc. (EBC): costing of the WSD, designed by KCB
- Tetra Tech: process plant and associated infrastructure, including plant site preparation, water treatment plant, construction camps
- Brazier: permanent power supply, fire detection, mini hydro plant, and energy recovery systems
- EBA: winter access road and review of KCC and EBC cost estimates
- ERM: environmental
- BGC: landslide management, avalanche management, and pit depressurisation
- McElhanney: main access roads (TCAR, CCAR)
- Seabridge: Owner's costs.

The capital cost estimate uses the structure summarized in Table 21.1.

Table 21.1 KSM Capital Cost Estimate Structure Summary

Area No.	Area Description	Cost (US$)
1	Direct Costs	3,311,102,000
2	Indirect Costs	862,237,000
3	Owner's Cost	160,233,000
4	Contingency	670,995,000

Notes: Costs have been rounded to the nearest thousands of dollars.

All currencies in this section are expressed in US dollars, unless otherwise stated. Costs have been converted using a fixed currency exchange rate of US$0.80 to Cdn$1.00. Metal prices are based on the three-year trailing average prices from July 31, 2016 back to August 1, 2013.

The expected accuracy range of the capital cost estimate is +25/-10%.

The costs stated in Table 21.1 include only initial capital, which is defined as all costs to build the facilities that mine, transport, and process ore to produce first concentrate and doré. Costs incurred during ramp-up of the mine and process plant in Year 1, through commercial production, are included in the operating costs in Section 21.3.

This estimate is prepared with a base date of Q2 2016. The estimate does not include any escalation past this date. Budget quotations were obtained for major equipment; vendors provided equipment prices, delivery lead times, spare allowances, and freight costs to a designated marshalling yard in northern BC, with some exceptions for delivery points to different BC locales. The quotations used in this estimate were obtained in Q1 and Q2 2016, and are budgetary and non-binding.

For non-major equipment (i.e. equipment less than US$100,000), costing is based on in-house data, quotes from previous projects, or maintained from the 2012 PFS (Tetra Tech 2012) estimate. No cost escalation from 2012 to 2016 is included. A comparison of 2016 quotes vs. 2012 quotes for the same vendor revealed no significant trend of escalation; therefore, no escalation was applied.

All equipment and material costs includes Incoterms FCA. Other costs such as spares, taxes, duties, freight, and packaging are covered separately in the estimate as indirect costs.

The estimate cost breakdown structure (CBS), which is based on the work breakdown structure (WBS) for the Project, is presented in Table 21.2.

Table 21.2 Capital Cost Summary

Major Area No.	Major Area Description	Cost (US$)
1 – Direct Costs		
1.1	Mine Site	1,218,098,000
1.2	Process	1,336,423,000
1.3	TMF	440,697,000
1.4	Environmental	14,592,000
1.5	On-site Infrastructure	22,851,000
1.6	Off-site Infrastructure	119,580,000
1.7	Permanent Electrical Power Supply and Energy Recovery	158,861,000
	Total Direct Costs	**3,311,102,000**
2 – Indirect Costs		
2.91	Construction Indirect Costs	449,092,000
2.92	Spares	34,314,000
2.93	Initial Fills	19,664,000
2.94	Freight and Logistics	99,015,000
2.95	Commissioning and Start-up	6,120,000
2.96	EPCM	230,957,000
2.97	Vendor's Assistance	23,075,000
	Total Indirect Costs	**862,237,000**
3 – Owner's Costs		
3.98	Owner's Costs	160,233,000
4 – Contingency		
4.99	Contingency	670,995,000
	2016 PFS Capital Cost Total	**5,004,566,000**

Notes: Costs have been rounded to the nearest thousands of dollars.

The detailed capital cost estimate breakdown is included in Appendix L1.

The parameters used in the estimate are shown in Table 21.3.

Table 21.3 Capital Cost Estimate Input Parameters

Item	Parameter
The following blended all-in construction labour rates are to be used:	
Earthworks	US$76.00
Civil Piping	US$76.00
Concrete Works	US$76.00
Steel	US$76.00
Architectural	US$76.00
Mechanical Equipment	US$76.00
Mechanical Bulks	US$76.00
Building Services	US$76.00
Electrical	US$76.00
Instrumentation	US$76.00
The following labour productivity factors shall be applied:	
For Work on Surface	1.30
For Work Inside Tunnels	1.20
Open Pit Mining	1.18
Concrete Unit Rate (Installed Cost) (includes aggregate/batch/wash/asphalt plant)	US$1,472/$m^3$
Steel Unit Rate (supply and installed cost)	US$4,774/t
In the event that spares are not quoted, the following percentages were used:	
Spares for Construction Calculated as a Percentage of Direct Capital	3%
Commissioning Spares	2%

Note: The base labour rate used is US$33.10.

21.1.1 EXCLUSIONS

The following items 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
- salvage value for assets only used during construction
- cost of financing (including interests incurred during construction)
- sales taxes (PST, GST and HST)

- royalties or permitting costs, except as expressly defined
- schedule acceleration costs
- working capital
- cost of this study and future feasibility study
- sunk costs.

21.1.2 LABOUR RATES

A standard labour rate has been applied to various areas of the Project. The standard construction labor rate used is US$76.00/h (Cdn$95.00/h) and is considered fully burdened. The base labour rate of US$33.10 (Cdn$41.38) was calculated from a combination of union rates published by Mine site for BC (2015), independent contractor quotes, Christian Labour Association of Canada, and recent construction projects in BC. The loaded labour rate build-up uses the following percentages:

- fringe benefits and burden 34%
- shift (day/night) premium 3%
- overtime premium 29%
- overhead, fees and profit 25%
- personal protective equipment, small tools and supplies 7%
- location premium 25%
- incentives on full shift attendance, safety, target achievement, etc. 5%

The detailed build-up for the construction labour rate is presented in the Basis of Estimate Report in Appendix L1.

21.1.3 CONCRETE AND STRUCTURAL STEEL

CONCRETE

The concrete composite unit rates are based on a quote by a supplier (including direct costs, indirect costs, overhead, and profit) applicable for the northern BC region.

The cost of standard concrete placed is shown in Table 21.4.

Table 21.4 Cost of Standard Concrete Placed

Item	Cost (Cdn$/m³)	Cost (US$/m³)
Concrete Price	380.00	304.00
Pumping Base Rate	3.00	2.40
Mobilization and Demobilization	7.40	5.92
Delivery	11.00	8.80
Winterization	7.60	6.08
Batched Concrete Subtotal	409.00	327.20
Rebar based on 110 kg/m³ @ $1.20/kg	130.00	104.00
Formwork	226.00	180.80
Concrete, Rebar and Formwork Subtotal	**765.00**	**612.00**
Rented Equipment	25.00	20.00
Labour to Place Concrete 8.5 h x 1.3 x $95	1,049.75	839.80
Total Concrete Placed	**1,839.75**	**1,471.80**

The total unit cost of concrete includes the cost of concrete (including wastage, pumping, mobilization/demobilization), delivery of concrete to formworks from the batch plant, the cost of formworks materials and installation and the cost of rebar materials and installation.

The unit cost of concrete excludes geotechnical site investigations, earthworks/excavation, backfill, compaction, heated ready-mix concrete, insulation on concrete forms, concrete admixtures/chemicals, water stops, concrete curing, cost of power, wash-off water, corrosion inhabitants, any special coatings, damp proofing, or paint.

STRUCUTRAL STEEL

Structural steel quantity material take-offs (MTOs) were derived from Tetra Tech historical data and adjustments were made, where necessary, to allow for any unique location requirements. Where historical data were not available, preliminary engineering was performed to provide basis for the quantity MTOs.

The pricing breakdown of structural steel is shown in Table 21.5.

Table 21.5 Cost of Structural Steel

Item	Cost (Cdn$)	Cost (US$)
Fabricate and Deliver	3,000	2,400
Erection 22 h x 1.3 x $95	2,717	2,174
Rented Equipment	250	200
Cost ($/t)	**5,967**	**4,774**

21.1.4 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 basis.
- Site work is continuous and is not constrained by the Owner.
- The work week is assumed to be 10 h/d with a rotation of 20-days-on/10-days-off for the construction phase of the Project. This is an atypical roster, yet purposefully selected to be compliant with daily maximum and weekly average maximum hours worked (including for time off) as per the BC *Mines Act.* This will cater for a significant proportion of construction staff who will travel long distances on their off time and to reduce fatigue risk of longer rotations (e.g. 28 days). The exception to this turnaround rotation for construction personnel are the WSD and tunnelling staff, which are based on 21-days-on and 7-days-off, and the WSD and tunnelling crew, which are based on 11 h/d with 28-days-on and 14-days-off.
- A productivity factor of 1.30 has been applied to the labour portion of the construction estimate to allow for the inefficiency of long work hours, climatic conditions, and the 20-day-on/10-day-off rotation. This is 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 is not allowed for in this estimate.
- Sales taxes (PST, GST and HST) are excluded from the estimate.

21.1.5 DIRECT COSTS

MINE SITE

The Mine Site capital costs are US$1.2 billion and are broken down in Table 21.6.

Table 21.6 Mine Site Capital Costs

Area No.	Area Description	Cost (US$)
1.1.01	Open Pit Mining	435,655,000
1.1.02	WSF	285,560,000
1.1.03	SWM	222,393,000
1.1.04	Water Treatment	182,614,000
1.1.05	Ancillary Buildings	4,070,000
1.1.06	Site Services and Utilities	1,959,000
1.1.07	Power Supply and Distribution	4,494,000
1.1.08	Camps	57,870,000
1.1.13	Geohazards	23,483,000
Mine Site Capital Costs Total		**1,218,098,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Open Pit Mining

Open pit capital costs are derived from a combination of supplier quotes and historical data collected by MMTS, valid for Q3 2016, and are shown in Table 21.7

Table 21.7 Open Pit Mining Capital Costs

Sub-area No.	Sub-area Description	Cost (US$)
1.1.01.01	On-site Roads and Initial Earthworks (Pioneering Works including Sulphurets Quarry)	72,134,000
1.1.01.02	Pre-production Operating Cost	199,780,000
1.1.01.04	Mining Equipment	155,643,000
1.1.01.05	Mining Pits Depressurization Wells	4,709,000
1.1.01.08	Electrical	3,389,000
Open Pit Mine Capital Cost Total		**435,655,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Open Pit Mining Capital Basis of Estimate

Mine mobile equipment capital costs are shown in Table 21.8 and are based on a unit cost for each piece in the fleet, multiplied by the number of units purchased during pre-production. Fleet sizes are described in greater detail in Section 16.1.

Pricing for all major units is based on budgetary quotes provided by vendors operating in the region, for equipment that is delivered, assembled, commissioned, and ready to work. Where possible, three quotes have been obtained. The capital cost of a new machine is recorded in the year it is scheduled to begin operation.

The equipment mine capital costs include delivery to the site, assembly, and an estimate of critical spares inventory, but do not include taxes or duties. Costs for freight, assembly, and spares are included as an indirect capital cost. All equipment and materials are purchased new.

Pioneering and open pit pre-production operating costs accrued before start-up are capitalized.

Pioneering activities completed prior to Year -3 will be performed with a mix of Owner and contractor equipment. Contractor equipment listed in Table 21.8 has a mobilization and demobilization capital cost of 10% of the purchase price of a machine, and a contractor margin/overhead of 25% added to operating and labour costs.

Table 21.8 Open Pit Mine Mobile and Engineering Equipment Capital Costs

Item	Cost (US$)
Drills/Shovels/Haulers	105,935,000
Contractor Equipment	4,272,000
Support Equipment	40,409,000
Engineering Equipment	5,027,000
Mobile and Engineering Equipment Capital Cost Total	**155,643,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Water Storage Facility

The WSF capital estimate is US$285.6 million and includes the items listed in Table 21.9. The main dam structure is the largest cost at US$122.2 million.

EBC used a bottom-up approach for costing and split the WSD into 15 sections using a WBS. The estimate EBC developed is a bid quality estimate and is based in part on their recent experience constructing the similar La Romaine asphalt core dam in northern Quebec. EBC incorporated the seasonal constraint of constructing a dam with temperature sensitive materials and a reasonable rate of rise to yield a multi-year construction duration that becomes a primary cost driver for the WSF. The WSF cost includes surface and subsurface (cofferdams and construction diversion tunnel) construction period water diversions, main and seepage dams and associated smaller structures, pumping and piping, and power infrastructure to deliver WSF collected water up over the dam to the HDS WTP.

Table 21.9 Water Storage Facility Capital Costs

Sub-area No.	Sub-area Description	Cost (US$)
1.1.02.02	Water Storage Construction Diversion Tunnel and Seepage Collection Tunnels	58,104,000
1.1.02.06	Cofferdam	423,000
1.1.02.08	Spillway	39,166,000
1.1.02.12	Main Dam	122,224,000
1.1.02.14	Seepage Dam	2,442,000
1.1.02.15	Fuel*	-
1.1.02.17	Pumping and Pipelines	49,452,000
1.1.02.18	Electrical (including portable substations)	13,749,000
Water Storage Facility Capital Cost Total		**285,560,000**

Note: *Included in earthworks unit rates
Costs have been rounded to the nearest thousands of dollars.

Surface Water Management

The surface water managment capital estimate is US$222.4 million and includes all the items that are required to collect and route surface water around the Mine Site. The different surface water management costs are listed in Table 21.10, with the majority of the costs related to the three principal water diversion tunnels (MDT, MVDT, and MTDT).

Table 21.10 Surface Water Management Capital Cost Estimate

Sub-area No.	Sub-area Description	Cost (US$)
1.1.03.22	Diversion Pipelines	7,771,000
1.1.03.23	Collection Ditches	6,281,000
1.1.03.31	MDT – Collection Galleries for Mitchell Glacier Diversion Tunnel	1,489,000
1.1.03.32	MDT (Open Pit Stage)	69,528,000
1.1.03.34	MDT (Underground Phase - Outlet Portals)	1,067,000
1.1.03.41	NPWDA and SSDA	991,000
1.1.03.50	MVDT and Mitchell OPC Decline Tunnel	60,269,000
1.1.03.61	MTDT (Stage 1)	74,996,000
Surface Water Management Capital Cost Total		**222,393,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Water Treatment

Water treatment at the Mine Site is divided into TWTPs and the permanent HDS WTP. The major cost for water treatment is the HDS WTP at US$159.7 million (Table 21.11). The Selenium WTP comes online in Year 5 of operations and is not presented as part of the initial capital costs.

Table 21.11 Water Treatment Capital Cost Estimate

Sub-area No.	Sub-area Description	Cost (US$)
1.1.04.01	TWTP – Temporary Pads For Tunnel Muck Storage	7,001,000
1.1.04.02	Temporary Water Treatment – Ponds and Piping	6,160,000
1.1.04.03	TWTP Mine Site – Process Equipment	9,746,000
1.1.04.04	HDS WTP – Permanent WTP	159,708,000
Water Treatment Capital Cost Total		**182,615,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Ancillary Buildings

The ancillary buildings at the Mine Site are estimated at US$4.1 million, with the temporary truck shop as the largest cost at US$2.5 million. The permanent truck shop and the explosives facilities are included as part of the sustaining capital costs.

Site Services and Utilities

A total of US$2.0 million is allocated to site services at the Mine Site, with the portable mine substation at the temporary truck shop as the largest contributor at US$1.4 million.

Power Supply and Distribution

A total of US$4.5 million is allocated to power supply and distribution, with the transmission lines from Substation No. 2 to the Mine Site as the largest contributor at US$2.1 million.

Camps

Total camp costs at the Mine Site are US$57.9 million and include Construction Camp Nos. 2 (Ted Morris), 4 (Mitchell North), 9 (Mitchell initial), 10 (Mitchell secondary), and the Mitchell Operations Camp. Each camp includes dormitories, kitchen and dining facilities, incinerator, recreation facilities, check-in and check-out areas, administrative offices, and first-aid facilities.

Details for camp costs can be found in the Basis of Estimate Report located in Appendix L1.

Geohazards

Geohazard costs include landslide management at US$1.8 million and avalanche management at US$21.7 million.

PROCESS

The battery limits for process in the CBS are the Mitchell OPC (primary crusher), the MTT (including Saddle), trains, and the Treaty OPC (crushing, grinding, flotation, concentrator and leaching, Treaty OPC buildings and equipment). TMF costs are separate from process costs. Process capital costs are estimated at US$1.336 billion and TMF capital costs are estimated at US$440.7 million.

Table 21.12 Process-Treaty OPC Capital Cost Estimate

Area No.	Area Description	Cost (US$)
1.2.01	Primary Crushing	52,244,000
1.2.02	Ore Delivery Tunnel	364,987,000
1.2.05	MTT Material Transportation	221,527,000
1.2.06	TWTP No. 4 Water Treatment – Saddle	4,367,000
1.2.07	TWTP No. 8 Water Treatment – Treaty	2,798,000
1.2.08	Treaty OPC – Coarse Ore Stockpile	82,841,000
1.2.09	Treaty OPC – Secondary Crushing	58,610,000
1.2.10	Treaty OPC – Fine Ore Stockpile	47,100,000
1.2.11	Treaty OPC – Tertiary Crushing - HPGR	70,486,000
1.2.12	Process Building	74,410,000
1.2.13	Primary Grinding	75,465,000
1.2.14	Copper Flotation	42,568,000
1.2.15	Pyrite Flotation	7,530,000
1.2.16	Pyrite Concentrate Regrinding	22,881,000
1.2.17	Cyanide Leaching	41,947,000
1.2.18	Gold/Silver Refinery	9,794,000
1.2.19	Copper Concentrate Handling	4,779,000
1.2.20	Molybdenum Floatation Circuit	3,805,000
1.2.21	Molybdenum Concentrate Handling (including Leaching)	4,158,000
1.2.22	Cyanide Recovery and Destruction	17,770,000
1.2.23	Reagent Area	5,390,000
1.2.24	Plant Control System	3,966,000
1.2.25	Site Services and Utilities	12,302,000
1.2.27	Treaty OPC - Temporary Laydown Area	584,000
1.2.30	Treaty OPC – Ancillary Buildings	25,984,000
1.2.31	Process Plant Utilities	7,217,000
1.2.32	Treaty OPC – Mobile Equipment	8,874,000
1.2.33	Power Supply and Distribution	5,868,000
1.2.34	Treaty OPC – Roads	1,149,000
1.2.35	Treaty Operations and Construction Camps	55,022,000
Process-Treaty OPC Capital Cost Total		**1,336,423,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Primary Crusher

The primary crusher includes crusher equipment, earthworks, and conveyors, including the tripper conveyor depositing ore to the loading bins at train loading station. The capital cost estimated for the primary crusher is US$52.2 million.

Ore Transport

Ore transport includes the tunneling cost for the MTT, as well as the train operations from loading bins at the Mine Site to the discharge/load out station at the Treaty OPC. The TWTPs required for tunneling are also included in this breakdown (Table 21.13).

Table 21.13 Ore Transport Capital Cost Estimate

Area No.	Area Description	Cost (US$)
1.2.02	Ore Delivery Tunnel	364,987,000
1.2.05	MTT Material Transportation	221,527,000
1.2.06	TWTP No. 4 Water Treatment – Saddle	4,367,000
1.2.07	TWTP No. 8 Water Treatment – Treaty	2,798,000
Ore Transport Capital Cost Total		**593,679,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Ore Delivery Tunnel

Multiple bid-quality construction cost estimates were prepared by tunneling contractors and benchmarked against data in northern Canada to establish appropriate unit excavation costs to be applied against variable geotechnical conditions along the tunnel alignments. Tunneling costs include for all labor, supervision, equipment, consumables to complete tunnel excavation, place track and ballast, and construct ancillary facilities. The estimated cost for the MTT is US$365.0 million. Greater detail on the development of tunneling costs is included in the Basis of Estimate Report in Appendix L1.

MTT Material Transportation

MTT material transportation comprises all components of the train system that will transport ore from the Mine Site to the Treaty OPC, consumables from the Treaty OPC to the Mine Site, and personnel in both directions. The train scope includes capital costs for all train rolling stock and train related infrastructure (i.e., track and ballast; load stations; electrical, including catenary, signaling and control system; unloading stations; feeders; maintenance shops; and control room).

Capital costs for the train systems for the Project are based on budgetary quotes provided by an equipment vendor operating in similar environments globally. The estimates were submitted in Q2 2016 with an assumed expiration date of Q3 2016.

The train system capital estimate, including power supply and fire detection, is US$221.5 million.

Coarse Ore Stockpile

The coarse ore stockpile includes the reclaim conveyors and snow covering for the stockpile and is estimated at US$82.8 million.

Secondary Crusher

Secondary crushing includes the crusher, conveyors, and screens and is estimated at US$58.6 million.

Fine Ore Stockpile

The fine ore stockpile includes the reclaim conveyors and snow cover structure and is estimated at US$47.1 million.

Tertiary Crusher

Tertiary crushing consists of two HPGR crushers and includes the earthworks, building enclosure, and all equipment and conveyors associated with tertiary crushing; this is estimated at US$70.5 million.

Flotation, Concentrator and Leaching

The process plant consists of the components listed in Table 21.14 and is estimated at US$314.5 million.

Table 21.14 Flotation, Concentration, and Leaching

Area No.	Area Description	Cost (US$)
1.2.12	Process Building	74,410,000
1.2.13	Primary Grinding	75,465,000
1.2.14	Copper Flotation	42,568,000
1.2.15	Pyrite Flotation	7,530,000
1.2.16	Pyrite Concentrate Regrinding	22,881,000
1.2.17	Cyanide Leaching	41,947,000
1.2.18	Gold/Silver Refinery	9,794,000
1.2.19	Copper Concentrate Handling	4,779,000
1.2.20	Molybdenum Flotation Circuit	3,805,000
1.2.21	Molybdenum Concentrate Handling (including leaching)	4,158,000
1.2.22	Cyanide Recovery and Destruction	17,770,000
1.2.23	Reagent Area	5,390,000
1.2.24	Plant Control System	3,966,000
Flotation, Concentration, and Leaching Capital Total		**314,463,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Treaty OPC Process Plant Mobile Equipment

The total cost for mobile equipment for the Treaty OPC is US$8.9 million and includes for various lifts (fork, scissor, etc.), light duty vehicles, cranage, and small miscellaneous equipment used by process operations and maintenance at both the Mine Site and Treaty OPC.

Treaty OPC Ancillary Buildings

Ancillary buildings, which include administration buildings and all related buildings associated with process, is estimated at US$25.9 million. Construction (Camp Nos. 5 and 6) and permanent operations camps are estimated at US$55.0 million. The total is estimated at US$80.9 million.

Site Services and Utilities

Site services and utilities are estimated at US$12.3 million, temporary laydown area at US$0.6 million, and process plant utilities, such as process water and plant site fuel unloading/pumping station (US$7.2 million), power supply and distribution (US$5.9 million), and plant site roads (US$1.1 million), totals US$27.1 million.

TAILINGS MANAGEMENT FACILITY

The TMF capital costs are US$440.7 million as show in Table 21.15.

The TMF estimate was developed by KCC within the AACE® Recommendation Practice No. 56R-08 guidelines. The estimate is based on a first principles approach to execute the work based on the design of the facility as detailed in Section 18.2, and typical equipment spreads for large earthwork construction, production factors for similar type construction accounting for workforce skill level, site conditions, climate, and accessibility. The tailing starter dam structures are the largest cost at US$187.5 million.

Table 21.15 TMF Capital Costs

Area No.	Area Description	Cost (US$)
1.3.01	General Costs and Consumables (e.g. fuel)	116,513,000
1.3.02	Water Management	44,702,000
1.3.03	Tailing Starter Dams	187,454,000
1.3.04	Seepage Dams	7,192,000
1.3.05	Discharge Pipeline and Diffuser	13,636,000
1.3.06	Geotechnical Laboratory	337,000
1.3.07	Tailing Disposal and Reclaim	68,257,000
1.3.08	Electrical	2,606,000
TMF Capital Cost Total		**440,697,000**

Note: Costs have been rounded to the nearest thousands of dollars.

ENVIRONMENTAL

The environmental capital costs are exclusively for fish compensation bonds estimated at US$14.6 million. Other environmental costs are captured under Owner's costs. Reclamation is excluded from this estimate as it is captured in the financial analyses.

ON-SITE INFRASTRUCTURE

On-site infrastructure includes overall site services and utilities, as well as other temporary services, and is estimated to cost US$22.9 million.

OFF-SITE INFRASTRUCTURE

Off-site infrastructure capital costs are show in Table 21.16 and are estimated at US$119.6 million. Off-site infrastructure includes the TCAR, CCAR, winter access road, Highway 37 marshalling yard, concentrate storage shed to be built at the Stewart, BC port, and off-site camps. Off-site camps comprise costs for camps at Granduc, along Highway 37, and those that support TCAR and CCAR construction (Camp Nos. 3, 7, 8, and 11). Off-site camps at Granduc and along Highway 37 assume more than double the daily operating costs to establish a reasonable capital cost for usage as compared to on-site camps to account for leasing or renting a camp from a third party.

Table 21.16 Off-site Infrastructure

Area No.	Area Description	Cost (US$)
1.6.01	Treaty Road Marshaling Yard at Camp 11	6,047,000
1.6.02	Permanent Access Roads	63,143,000
1.6.03	Temporary Winter Access Roads	9,095,000
1.6.05	Off-site Concentrate Storage (Stewart, BC)	24,308,000
1.6.06	Off-site Camps	16,987,000
Off-site Capital Cost Total		**119,580,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Permanent Electrical Power Supply and Energy Recovery

The permanent electrical power supply and energy recovery capital costs are show in Table 21.17. The permanent electrical power cost accounts for the cost of power supply to the Project from BC Hydro's NTL transmission line and the KSM primary site distribution and main substations. This includes a capital cost contribution covering part of the cost of BC Hydro's Treaty Creek Switching Station on the NTL, the cost of the 30 km long, 287 kV, KSM transmission line extension to the Project site, the cost of the main step-down substation at the Treaty OPC, the cost of the 24 km long, 138 kV power cables through the MTT, and the cost of the Mine Site step-down substation. The required capital cost contribution, required by the utility tariffs to cover a percentage of BC Hydro's NTL transmission line, are not due until the start of commercial production; thus, are included in sustaining capital.

The energy recovery capital cost includes installations as necessary to recover energy from otherwise required water diversions and normal process flows. This includes hydro generation from the MDT, hydro generation by the water flow from the WSD to the HDS WTP, and energy recovery pump turbines in the tailing line.

Table 21.17 Permanent Electrical Power Supply and Energy Recovery

Area No.	Area Description	Cost (US$)
1.7.01	Energy Recovery	24,215,000
1.7.02	Permanent Electrical Power	134,646,000
Permanent Electrical Power Supply and Energy Recovery Capital Cost Total		**158,861,000**

Note: Costs have been rounded to the nearest thousands of dollars.

21.1.6 Indirect Costs

Indirect costs for the Project are estimated at US$862.2 million, as shown in Table 21.18.

Table 21.18 Indirect Capital Costs

Area No.	Area Description	Cost (US$)
2.91	Construction Indirect Costs	449,092,000
2.92	Spares	34,314,000
2.93	Initial Fills	19,664,000
2.94	Freight and Logistics	99,015,000
2.95	Commissioning and Start-up	6,120,000
2.96	EPCM	230,957,000
2.97	Vendor's Assistance	23,075,000
Indirect Capital Costs Total		**862,237,000**

CONSTRUCTION INDIRECT COSTS

The construction indirect allowances cover the following items:

- temporary works
- temporary lighting
- temporary water supply
- temporary sewage such as portable toilets and sewage collection (not treatment)
- temporary facilities and structures including offices and storage
- temporary support systems and utilities
- temporary communication (main communication infrastructure included in direct costs)
- major construction equipment (standard construction equipment is included in direct costs)
- heavy duty craneage (PTMA site only)
- miscellaneous equipment rentals
- scaffoldings
- garbage collection
- contractors mobilization and demobilization, and mark-ups
- project final clean up.

The construction indirect costs for each area have been reviewed and classified as contractor or non-contractor based. Generally, a 5 to 10% construction percentage is used for a non-contractor based areas and 0% for contractor based areas.

Tetra Tech incorporated first principles calculations for indirect costs, apart from allowance based costs above, for the following items:

- winter and summer road maintenance
- bussing of workforce to site
- helicopter support
- construction camps included in the direct costs
- catering and housekeeping
- temporary power required during construction
- all personnel turn-around costs
- all personnel travel time to site for each shift

The main construction indirect exclusions are:

- temporary road costs are included in direct costs for open pit mining, WSD, TMF, and PTMA
- contractor fuel consumption costs during construction are included as direct costs
- avalanche control labour is included in direct costs
- small tools and PPE are included in the labour rate.

SPARES

Spares for construction are calculated as 3% of direct capital costs, and commissioning and start-up spares are calculated as 2% of the installed equipment and bulk material costs, with the exception of the following:

- open pit mining mobile equipment – capital spares only at 4.2% of equipment costs calculated by MMTS (no commissioning spares)
- plant mobile equipment – capital spares at 3% of equipment costs (no commissioning spares)
- permanent power supply and Energy Recovery Plant estimated amount of US$2.3 million.

INITIAL FILLS

Initial fills were estimated by Tetra Tech based on equipment sizing, process design criteria, and in-house experience. The initial operation required volumes were estimated based on unit reagent consumptions and quoted prices from suppliers and account for approximately the first 10 days of plant operation. Costs were also estimated for long-lead consumables (e.g., mill liners, crusher liners) based on the cost of the first replacement.

Freight and Logistics

Freight and logistics for bulk material and equipment was based on a study completed in 2012, with averages calculated from quotes received from shipping and transport vendors for a project in northern BC. Percentages used are as follows:

- generally – 8%
- mobile equipment – 5%
- cement and aggregates – 0%; included in the rates.

The use of 5% for mobile equipment freight correlates with 2016 vendor quotes, where freight costs range from as low to 1% to as high as 7%. This is due to less stringent packaging requirements (palletized and shrink wrapped, but not enclosed, nor significantly padded) when compared to more complex machinery.

A nominal allowance is allowed for air freight.

Commissioning and Start-up

Commissioning and start-up costs are calculated based on the number of engineers required on site, depending on the systems required to be commissioned, the estimated commissioning duration, and generally a rate of US$152/man-hour (Cdn$190/man-hour) reflecting current approximate market rates.

Engineering, Procurement, and Construction Management

Estimated percentages for EPCM costs for projects in BC, as stated in recent feasibility-level studies, range between 6 to 13.2%. These percentages were benchmarked from selected projects published on SEDAR (seven recent studies for mining projects in BC). Tetra Tech elected to apply the higher end of the range generally at 12%, combined for EPCM, to account for the scale and complexity of the Project. The estimated cost for EPCM is US$230.9 million.

Engineering and Procurement

Engineering and procurement costs are calculated on a percentage basis based on the relative effort anticipated to be necessary for completely designing and procuring all the necessary equipment, tools, and services to provide for effective field construction. Generally, a factor of 7% is anticipated to cover the engineering and procurement cost, although select areas received a lower percentage so as not to disproportionally distort the man-hour effort required if a uniform percentage across all areas were to be applied.

Construction Management

The conceptual contracting strategy for the Project is to have a construction management team overseeing several main contractors. It has been assumed in the estimate that no additional contractors mark-ups are needed except for labour. The labour rate is inclusive of contractors' overhead and profit.

The construction management allowance is generally assigned at 5% for this 2016 PFS, due to the size and magnitude of the Project, except for earthworks on tunnels, WSF, SWM, TWTP, HDS, TMF and access roads, all of which were calculated at 3%. Train and rail were calculated at 2% of direct cost as these costs were obtained from the estimating contractor who estimated construction management as part of their quotation.

VENDOR'S ASSISTANCE

Vendor assistance has been included for in the process, on-site infrastructure, and permanent electrical power supply areas where it is anticipated specialists may be required during commissioning and start-up. The cost estimated for vendor assistance is US$23.1 million.

21.1.7 OWNER'S COSTS

An estimate of US$160.2 million is included in the capital cost estimate for Owner's costs. This cost has been calculated by Seabridge from first principals based on personnel requirements and onboarding of Owner's staff for supporting both the latter part of the commissioning effort for the Project and onboarding to fill all operations and G&A departments. In addition to labour costs, Owner's costs include off-site office staffing, off-site office facilities, travel, off-site office general expenses, recruiting and training expenses, consulting, insurance, general field expenses, and mineral lease/claims costs. Environmental department Owner's costs include environmental monitoring programs, community relations costs, communication and public relations costs, wetland compensation, and permitting costs. Limited duty costs are also included for certain large capital equipment imported from outside North America.

21.1.8 CONTINGENCY

A contingency allowance is included to cover additional costs that 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. The estimated contingency cost is US$671.0 million and the detailed build-up for contingency is presented in Table 21.19. The contingency estimate was developed on a line-item basis to account for the specific design detail and information available for each area, rather than a single value applied to the sum of all direct, indirect, and Owner's costs. The values applied range from 5 to 25%.

Table 21.19 Contingency

Area No.	Area Description	Cost (US$)
4.99.01	Mine Site	184,289,000
4.99.02	Process	210,611,000
4.99.03	TMF	85,226,000
4.99.05	On-site Infrastructure	4,031,000
4.99.06	Off-site Infrastructure	18,117,000
4.99.07	Permanent Electrical Power Supply and Energy Recovery	21,397,000
4.99.91	Construction Indirects	67,354,000
4.99.92	Spares	3,080,000
4.99.94	Freight and Logistics	14,411,000
4.99.95	Commissioning and Start-up	913,000
4.99.96	EPCM	34,483,000
4.99.97	Vendor's Assistance	3,368,000
4.99.98	Owner's Costs	23,714,000
Contingency Total		**670,994,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Several elements of the estimate that represent significant costs are known to be more refined than others due to greater engineering detail, or due to recently received quotes (2016); therefore, a lower contingency was applied. Contingency was analyzed for all disciplines and the resultant contingency for this Project is 15.5%, measured as a percentage of the sum of direct, indirect, and Owner's costs.

21.2 SUSTAINING CAPITAL COSTS

The sustaining capital costs are all capital costs required from Year 1 of operations to sustain the mining operation for the LOM. The total sustaining costs of US$5.503 billion required for the LOM are presented in Table 21.20.

Table 21.20 Sustaining Capital Costs

Major Area No.	Major Area Description	Cost (US$)
1.1	Mine Site	3,932,699,000
1.2	Process	355,410,000
1.3	TMF	501,482,000
1.5	On-site Infrastructure	139,000
1.6	Off-site Infrastructure	10,739,000
1.7	Permanent Electrical Power Supply and Energy Recovery	195,722,000
2.91	Construction Indirects	57,424,000
2.92	Spares	33,545,000
2.94	Freight and Logistics	65,727,000
2.96	EPCM	22,349,000
2.97	Vendor's Assistance	34,757,000
4.99	Contingency	293,253,000
Sustaining Capital Cost Total		**5,503,246,000**

Note: Costs have been rounded to the nearest thousands of dollars.

21.2.1 MINE SITE

The sustaining capital for the Mine Site is US$3.9 billion presented in Table 21.21. This number covers the direct capital costs for the LOM and includes all open pit and underground mining operations, as well as the Selenium WTP and the geohazards direct capital cost.

Table 21.21 Mine Site Sustaining Capital Costs

Area No.	Area Description	Cost (US$)
1.1.01	Open Pit	820,579,000
1.1.03	SWM	261,760,000
1.1.04	Water Treatment	113,149,000
1.1.05	Ancillary Buildings	44,508,000
1.1.09	Mitchell Block Caving	1,848,211,000
1.1.10	Iron Cap Block Caving	779,461,000
1.1.11	Kerr Pit Power Supply	4,873,000
1.1.12	Sulphuret Pit Power Supply	1,999,000
1.1.13	Geohazards	58,159,000
Mine Site Sustaining Capital Cost Total		**3,932,699,000**

Note: Costs have been rounded to the nearest thousands of dollars.

21.2.2 OPEN PIT MINING

Sustaining capital is based on both fleet expansions and unit replacements over the LOM. Major fleet expansions are planned for Years 1, 6, and 10 as the mining rate and haul distances increase. Capital replacement costs for mobile equipment are calculated based on the expected life of the equipment, the cost of the unit, and the utilization for that equipment. Sustaining capital costs for the open pit are shown in Table 21.22.

Table 21.22 Open Pit Pit Sustaining Capital Costs

Fleet Capital Cost Item	Years 1 to 5 (US$ million)	Years 6 to 9 (US$ million)	Years 10 to 15 (US$ million	Years 16 to 53 (US$ million)	Total Cost (US$ million)
Drills/Shovels/Haulers	226,040,000	136,816,000	122,923,000	26,783,000	512,562,000
Contractor Equipment	4,272,000	-	-	-	4,272,000
Support Equipment	25,868,000	16,957,000	34,602,000	35,382,000	112,810,000
Pit Area Dewatering	30,729,000	38,521,000	57,034,000	64,650,000	190,934,000
Total	**286,910,000**	**192,295,000**	**214,559,000**	**126,815,000**	**820,579,000**

Note: Costs have been rounded to the nearest thousands of dollars.
Totals may not add up exactly due to rounding.

21.2.3 UNDERGROUND MINING (BLOCK CAVES)

The capital costs presented in this section are for both Iron Cap and Mitchell block cave mines, and are a combination of direct and indirect costs, and applied contingencies. The direct cost estimates have been obtained from supplier quotes for the major mobile equipment and capital infrastructure, such as conveyors, crushers, and ventilation infrastructure. Mine development, mobile equipment costs, and certain fixed and ventilation costs are the responsibility of Golder. The costs for the conveyors, secondary rock breakers (if applicable), crushers, related ancillary underground installations, and indirect costs on infrastructure are the responsibility of Tetra Tech. The underground electrical system cost estimate is the responsibility of Brazier.

For mine equipment, the quantity of spares was calculated based on the estimated life of the equipment and the life of the operation (typically a five-year replacement schedule, but this varies depending on the duty of the equipment). The unit cost of the mobile equipment was increased by 8% to account for freight. The indirect capital costs on mine infrastructure include freight, EPCM, vendor assistance, and spares. It is assumed that the KSM site procurement department will handle the procurement of the day-to-day supplies of the underground mines.

Contingency was applied to certain items and values range from zero (on most mine infrastructure), 15% on mobile equipment purchases, parts, and the electrical system, to a high of 30% for certain ground support items such as shotcrete and steel sets. Initial mine development contingency is 20%, but is not applied to post-production footprint development. The overall weighted average contingency on the capital cost of the underground mines is about 8% and is summarized in a separate line in the summary tables and WBS.

MITCHELL BLOCK CAVING

The Mitchell block cave mine capital cost estimate includes the purchase and installation of all equipment and the excavation of all the underground excavations. The total capital costs are US$2.0 billion as summarized in Table 21.23, which also includes a unit capital cost of US$4.50/t.

Table 21.23 Mitchell Block Caving Sustaining Capital Costs

Sub-area No.	Sub-area Description	Cost (US$)
1.1.09.01	Development and Preproduction	1,113,661,000
1.1.09.02	Infrastructure	404,571,000
1.1.09.03	Mobile Fleet	251,474,000
1.1.09.04	Ventilation and Services	39,985,000
1.1.09.07/10	Electrical	38,520,000
2.92.01.05; 2.94.01.03 2.96.01.03; 2.97.01.04	Indirect Costs (Spares, Freight, EPCM, Vendor Assistance)	46,220,000
4.99.01.02; 4.00.01.03	Contingency	150,000,000
Mitchell Block Cave Sustaining Capital Cost Total		**2,044,431,000**
Unit Capital Cost (US$/t)		**4.50**
Years		**37**

Note: Costs have been rounded to the nearest thousands of dollars.

Iron Cap Block Caving

The Iron Cap block cave mine capital cost estimate includes the purchase and installation of all equipment and the excavation of all the associated underground workings. The total capital costs are estimated to be US$872 million as summarized in Table 21.24, which also includes a unit capital cost of US$3.87/t.

Table 21.24 Iron Cap Block Caving Sustaining Capital Costs

Sub-area No.	Item	Cost (US$)
1.1.10.01	Development and Preproduction	417,379,000
1.1.10.02	Infrastructure	152,297,000
1.1.10.03	Mobile Fleet	143,796,000
1.1.10.04	Ventilation and Services	33,397,000
1.1.10.07; 1.1.10.08	Electrical	32,593,000
2.92.01.06; 2.94.01.04 2.96.01.04; 2.97.01.05	Indirect Costs (Spares, Freight, EPCM, Vendor Assistance)	19,185,000
4.99.01.04; 4.99.01.05	Contingency	73,375,000
Total Capital Cost		**872,022,000**
Unit Capital Cost (US$/t)		**3.87**
Years		**28**

Note: Costs have been rounded to the nearest thousands of dollars.

21.2.4 Mine Site Water Treatment

The sustaining water treatment cost for the Mine Site is US$113.1 million as shown in Table 21.25.

Table 21.25 Mitchell Mine Site Water Treatment Sustaining Capital

Sub-area No.	Sub-area Description	Cost (US$)
1.1.04.05	HDS – Permanent WTP	27,483,000
1.1.04.07	Selenium – Permanent WTP	78,385,000
1.1.04.08	Kerr Water to WTP	7,280,000
Mine Site Water Treatment Sustaining Capital Cost Total		**113,148,000**

Note: Costs have been rounded to the nearest thousands of dollars.

The water treatment capacity at the HDS WTP will increase with the addition of two more clarifiers to cater for the increased water flow that is expected. It is important to note that all earthworks for this expansion are already accounted for in the initial capital for the HDS WTP.

When the HDS WTP is commissioned in Year -1, the initial maximum throughput capacity will be 5.35 m^3/s.

The required maximum throughput will increase as the mine operation continues. Water will be collected in the WSF. Drainage from the Mitchell pit and Mitchell/McTagg RSFs will be directed by gravity to the WSF and contact water from the Sulphurets and Kerr pit areas will be pumped to the WSF. The water from the WSF will be pumped to the HDS WTP in order to maintain safe water-level requirements in the WSF. The HDS WTP is designed with variable discharge rates in order to stage discharge to match the natural hydrograph, to ensure sufficient dilution capacity to minimize any effects on the receiving environment.

In Year 5, the plant capacity will increase from the 5.35 m^3/s initial capacity to the 7.5 m^3/s final capacity. Two additional circuits will be constructed and operated in parallel to the existing five circuits.

The HDS WTP installed generation capacity will be 9 MW and the two installed turbines will be capable of passing a flow of up to 7.5 m^3/s.

Selenium WTP

In Year 5, a 500 L/s Selenium WTP, located adjacent to the WSF near the toe of the Mitchell/McTagg RSFs, will be constructed and become operational to treat seepage from the Sulphurets pit backfill (Kerr waste rock), seepage from the RSFs, and water pumped from the WSF.

Due to expected high iron and TSS concentrations in seepage water, a ferric circuit has been designed as a standalone module serving as a pre-treatment step upstream of selenium removal.

21.2.5 Process

Process sustaining capital includes the Sulphurets pit primary crusher and ore delivery tunnel installed in Year 1, and the Kerr pit primary crusher and Kerr rope conveyor installed in Year 23. New primary crushers will be installed for the Mitchell underground mine in Year 22 and at the Iron Cap underground mine in Year 31.

During operation some process related facilities will be constructed after the mill is in operation. The crushing and related transport facilities at Sulphurets includes a 60 inch by 89 inch gyratory crusher station and a 3.0 km SMCT connecting the Mitchell and Sulphurets sites, to be constructed in Year 1. An overland conveyor will be installed inside of the tunnel for ore transportation.

The ore from the Kerr deposit, together with the ore from the other deposits, will be introduced to the Process Plant starting from Year 24. The ore and related waste rocks from the deposit will be crushed at the Kerr site by two, 60 inch by 89 inch gyratory crushers and conveyed to the Mitchell OPC through a 2,480 m cross-valley rope conveyor to the Sulphurets site. The ore will be further conveyed through the SMCT to the Sulphurets/Kerr coarse ore stockpile at the Mine Site. The waste rock from the rope conveyor will be backfilled into the Sulphurets pit.

The Iron Cap ore and the lower Mitchell ore will be mined by block caving and crushed on their own sites to 80% passing 150 mm or finer. The crushed ores will be conveyed to the train transport system for delivery to the coarse ore stockpile at the Treaty OPC.

The estimate also includes a replacement allowance for major process equipment.

The sustaining capital cost for process is US$355.4 million as presented in Table 21.26.

Table 21.26 Process Sustaining Capital

Area No.	Area Description	Cost (US$)
1.2.01	Primary Crushing	185,840,000
1.2.02	Ore Delivery Tunnel (SMCT)	40,969,000
1.2.04	Rope Conveyance	89,436,000
1.2.07	TWTP No. 8 Water Treatment – Treaty	40,000
1.2.11	Crushing – Replacement Allowance	10,400,000
1.2.13	Grinding – Replacement allowance	13,600,000
1.2.14	Flotation/Cynadation – Replacement Allowance	8,000,000
1.2.25	Site Services and Utilities – Replacement Allowance	85,000
1.2.32	Site Service - Mobile Equipment – Replacement Allowance	7,040,000
Processing Sustaining Capital Cost Total		**355,410,000**

Note: Costs have been rounded to the nearest thousands of dollars.

21.2.6 NORTHWEST TRANSMISSION LINE CONTRIBUTION

The 344 km long, 287 kV NTL runs from the Skeena substation near Terrace, BC, to a new substation near Bob Quinn Lake. This new transmission line was commissioned in the summer of 2014 and currently serves the AltaGas Forrest Kerr Hydroelectric Facility and the Red Chris Mine. A tap from this transmission line will service the Project.

Due to an overrun in the construction cost of the NTL, BC Hydro Tariff Supplement TS37, as approved by the BCUC, was put in place requiring NTL customers to share in the overrun cost. In accordance with TS37, based on a Project contract (peak) demand of 200 MVA, the required contribution will be just over US$167.7million (Cdn$209.6 million). This amount is separate from system reinforcement and is a required cash contribution. The tariff is not due until the start of commercial production, and BC Hydro offers the option of spreading the payments out over five years, with an applicable finance charge, that is applied in the 2016 PFS.

21.2.7 McTagg Diversion Tunnel Mini Hydro Generation Station

The MTDT mini hydro generation station installation is included in sustaining capital. The current tunnel designs show the initial available hydraulic head to be too low to allow the station to be effective if installed during initial construction. No power will be generated from the Phase 1 MTDT. A penstock tunnel and penstock will be constructed in Phase 2. To enhance hydro power generation during low-flow periods, weirs will be established in Gingras Creek just upstream of the Phase 2 and Phase 3 portals to capture base flow from Gingras Creek. The base flow will be routed to the penstock where it will be combined with the McTagg base flows to increase hydro power generation. Flows in excess of the capacity of the McTagg Power Plant will bypass the de-sanding works and exit into Gingras Creek via the portals. The base flow from the Phase 2 and Phase 3 diversions will pass through a 1 m steel penstock pipe set in a plug at the junction of the penstock tunnel with the diversion tunnel. This pipe will run out to the surface in the penstock tunnel, exiting at a portal on the slope above the McTagg Power Plant. A buried steel penstock will then run down the slope to the McTagg Power Plant. The MTDT mini hydo generation cost is estimated at US$28.0 million.

21.2.8 Tailing Management Facility

The sustaining capital for the TMF is US$501.4 million and is presented in Table 21.27.

Table 21.27 TMF Sustaining Capital

Area No.	Area Description	Cost (US$)
1.3.01	General Costs and Consumables	94,969,000
1.3.02	Water Management	47,002,000
1.3.03	Tailing Starter Dams	300,421,000
1.3.04	Seepage Dams	3,042,000
1.3.05	Discharge Pipeline and Diffuser	18,588,000
1.3.07	Tailing Disposal and Reclaim	37,460,000
TMF Sustaining Capital Cost Total		**501,482,000**

Note: Costs have been rounded to the nearest thousands of dollars.

Initial capital comprises initial TMF perimeter diversions; the North Dam, Splitter Dam, and Saddle Dam; and associated seepage collection dams that form the North and Centre cells. This includes basin preparation for the starter basins for the North Cell and Centre Cell, preparing and lining the starter basins, and provision of liner drainage.

Dam raising for the North and Centre cells is accounted for in both sustaining and operating expenses. Borrow, haul, and placement of till core and expansion of basin preparation, drains and liner are considered sustaining expenses that begin in Year 1. The processing and placement of cyclone sand is considered an annual operating expense for all stages through the LOM.

The South Cell includes additional sustaining capital items occurring between Year 23 and the LOM. These include development of additional perimeter diversions for the South Cell and the Southeast Starter Dam and associated seepage dam. The expansion of the Centre Cell continues as sustaining capital until the end of mine life and the cyclone sand raising continues as an operating expense.

Ancillary expenses such as seepage pumping and monitoring are included as operating expenses throughout the LOM.

21.2.9 OTHER SUSTAINING CAPITAL COSTS

Other sustaining capital costs of US$713.7 million are presented in Table 21.28. This total includes US$195.7 million for permanent power supply and for the energy recovery plants that will be constructed. An amount of US$293.3 million is included for contingency on sustaining capital for the LOM. Contingency has been calculated on a line-item basis against all new footprint development. Details for contingency development can be found in the Basis of Estimate Report located in Appendix L1.

Table 21.28 Other Sustaining Capital

Major Area No.	Major Area	Cost (US$)
1.5	On-site Infrastructure	139,000
1.6	Off-site Infrastructure	10,739,000
1.7	Permanent Electrical Power Supply and Energy Recovery	195,722,000
2.91	Construction Indirect Costs	57,424,000
2.92	Spares	33,545,000
2.94	Freight and Logistics	65,727,000
2.96	EPCM	22,349,000
2.97	Vendor's Assistance	34,757,000
4.99	Contingency	293,253,000
Other Sustaining Capital Cost Total		**713.655,000**

Note: Costs have been rounded to the nearest thousands of dollars.

21.3 OPERATING COSTS

The average operating cost for the Project is estimated at US$12.03/t milled at the nominal process rate of 130,000 t/d, or US$12.33/t for the LOM average (Table 21.29). The operating cost estimate does not include the energy recovery credit (approximately US$0.12/t milled LOM) from mini hydropower stations and the cost relegated to PST (approximately US$0.15/t milled LOM).

The mining operating costs are LOM average unit costs calculated by total LOM operating costs divided by LOM milled tonnages. The costs exclude mine pre-production costs.

The cost distribution for each area is shown in Figure 21.1.

All costs are expressed in US dollars, unless otherwise specified. The operating cost estimates in this section are based on budget prices obtained in Q1/Q2 2016 and/or from databases of the consulting firms involved in preparing the operating cost estimates.

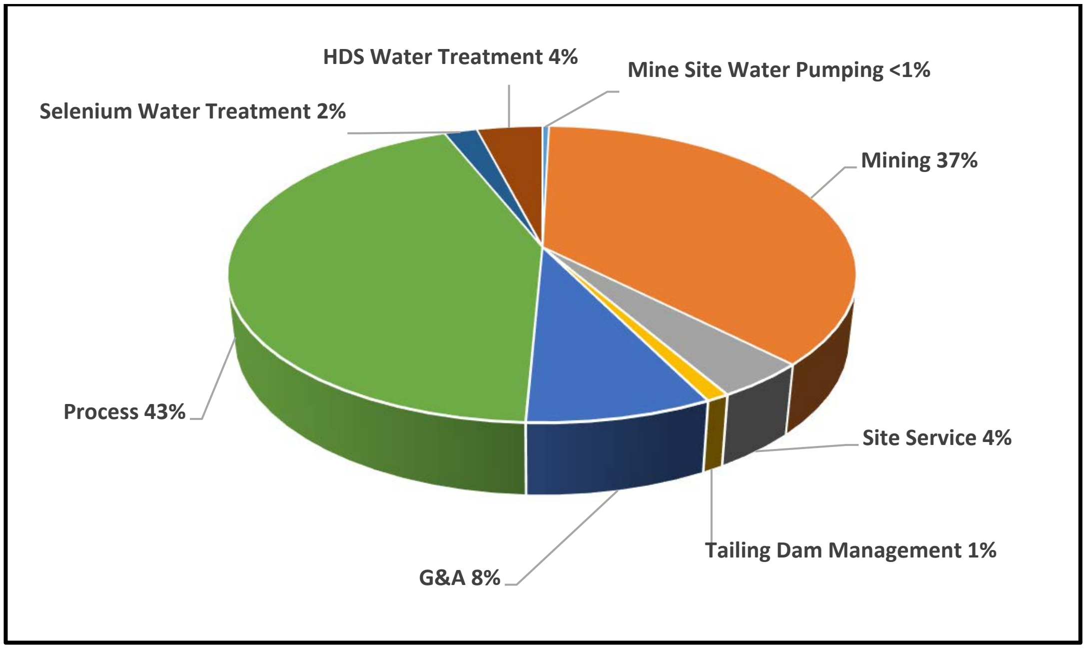
When required, certain costs in this report have been converted using a fixed currency exchange rate of Cdn$1.00 to US$0.80. The expected accuracy range of the operating cost estimate is +25/-10%.

Table 21.29 Operating Cost Summary

	At the Nominal Feed Rate of 130,000 t/d*		LOM Average (US$/t milled)
	(US$/a)	(US$/t milled)	
Mine			
Mining Costs – Mill Feed	190,223,000	4.59	4.59
Open Pit – Mill Feed	-	*4.40*	*4.40*
Block Caving – Mill Feed	-	*4.99*	*4.99*
Mill			
Process	251,066,000	5.29	5.34
G&A and Site Service			
G&A	43,272,000	0.91	1.03
Site Service	18,914,000	0.40	0.44
Tailing and SWM			
Tailing Dam Management	6,065,000	0.13	0.13
Selenium Water Treatment	9,469,000	0.20	0.21
HDS Water Treatment	22,033,000	0.46	0.53
Mine Site Water Pumping	2,453,000	0.05	0.06
Total Operating Cost	**543,495,000**	**12.03**	**12.33**

Notes: *The nominal feed rate estimate excludes mine operating costs and is based on a mill feed rate of 130,000 t/d; the costs do not reflect higher unit costs late in the mine life when the mill feed rates are lower.
Costs have been rounded to the nearest thousands of dollars.

Figure 21.1 Operating Cost Distribution



Power will be supplied by BC Hydro at an average cost of US$0.050/kWh at the plant 25 kV bus bars, based on the BC Hydro credits for energy conservation by using HPGR and similar, and the cost of "peaking" power to avoid a BC Hydro contract demand of over 150 MVA. 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 site services, water treatment plants, TMF seepage water pumping, and Mine Site water pumping are included in their area costs separately.

The estimated electrical power costs are based on the 2016 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 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 costlier Tier 2 power in the BC Hydro stepped-rate Schedule 1823. The 5% GST is not included in the power rates as it is an input tax credit.

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 financial analysis. Sustaining capital costs including all capital expenditures after process plant first production are excluded from the operating cost estimate.

21.3.1 OPEN PIT MINE OPERATING COSTS

Open pit mine operating costs, including operating and maintenance salaried staff and hourly labour, equipment major component and running repairs, fuel, power, and all other consumable goods, are derived from a combination of supplier quotes and historical data collected by MMTS. The quantities of consumables required are determined for each specific open pit mining activity from vendor input and in-house experience. Labour factors for operations and maintenance of the open pit mining equipment is also estimated based on vendor input and MMTS experience. These inputs are used to build up open pit mine operating costs from first principles.

Freight costs for all consumable goods and fuel are included in the estimate as part of the budgetary quotations.

Based on the open pit mine scheduled tonnes, the following primary open pit mine activity requirements are calculated from first principles:

- Production drill operating hours are based on hole size, pattern layout, bench height, material density, and penetration rate of the drill. High wall and pre-shear drilling is included. Geotechnical costs for high wall control blasting, horizontal drains, etc. are based on recommendations from BGC and from other study data collected by MMTS.
- The quantity of explosives is based on estimated rates for explosives provided by a previous vendor study on the Project, and an estimated pattern layout and an explosive density. An estimate for initiation systems and blasting accessories are provided on a per hole basis, and the labour costs for the blasting operations are included. The model assumes a 70/30 split between emulsion and ANFO. Input explosives costs are based on quotations by local vendors for site mixed product.
- Shovel and truck operating hours are based on the operating capacities of the equipment. The travel speed characteristics of the trucks and the haul road profiles are used to simulate representative truck cycle times including typical loading, dumping, and delay times.
- To support the open pit mine operations, a fleet of dozers, front-end loaders, graders, service and welding trucks, etc., is added. Operating hours for this support equipment is allocated based on estimated utilization of each unit.

Major component replacement for larger pieces of mobile equipment are calculated based on the expected life of the major component, the cost of the component, and the fleet size for that equipment. This puts large component repair costs into future years, giving a more representative LOM cash flow.

The cost of minor parts and running repairs are estimated as an hourly operating cost for the mining equipment.

Salaries for the supervisory and administrative job categories, and all hourly employee labour rates, are based on a project-specific labour survey conducted by Seabridge. Burdens are included in the salaries and labour rates.

Labour factors in man-hours/equipment operating hours are estimated for operations and maintenance labour for 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 an unallocated labour category.

The open pit mine hourly and salaried labour rates are summarized in Table 21.30 and listed in detail with manning levels in Appendix E.

Table 21.30 Open Pit Mine Hourly Labour Rates

Hourly Labour	Rate with Burdens (US$/h)
Operations and Maintenance	41.73
General Labourers	29.95
Salaried Labour	**Annual Salary with Burdens (US$)**
General Manager	154,080
Level 3S Staff (Superintendents/Senior Technical)	121,040
Level 4S Staff (Foremen, Engineers, Geologists)	103,620
Level 6S Staff (Technicians, Clerks, Administration)	61,544

GME is a category for open pit mine operations, mine maintenance, and technical services departmental overhead costs. It consists of costs for all salaried supervisory and technical staff, a consumable and rental allowance, crane rentals, and software and fleet management systems' licensing and maintenance. This category is a fixed cost, and does not vary by production or fleet size, with the exception of ramp-ups to full staffing.

LOM unit open pit mining operating costs are listed in Table 21.31 and Table 21.32. Complete open pit mine cost tables, including open pit mine capital and operating cost schedules, are available in Appendix E.

Table 21.31 Open Pit Mining Costs per Tonne Mill Feed

Area	LOM Open Pit Mining Cost (US$/t mill feed)*
Drilling	0.20
Blasting	0.58
Loading	0.53
Hauling	2.57
Pit Maintenance	0.32
Geotechnical	0.08
Unallocated Labour	0.01
General Mine Expense	0.12
Total Mining Cost	**4.40**

Note: *LOM open pit mining costs exclude capitalized pre-production operating costs.

Table 21.32 Open Pit Mining Costs per Tonne of Material Mined

Area	LOM Open Pit Mining Cost (US$/t material mined)*
Drilling	0.07
Blasting	0.20
Loading	0.18
Hauling	0.88
Pit Maintenance	0.11
Geotechnical	0.03
Unallocated Labour	0.00
General Mining Expense	0.04
Total Mining Cost	**1.51**

Note: *Material mined includes re-handled waste and borrow sources for construction material. LOM open pit mining costs exclude capitalized pre-production operating costs.

A graph of open pit mine unit operating cost is shown as dollar per tonne of 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 Open Pit Mining (US$/t Material Mined)

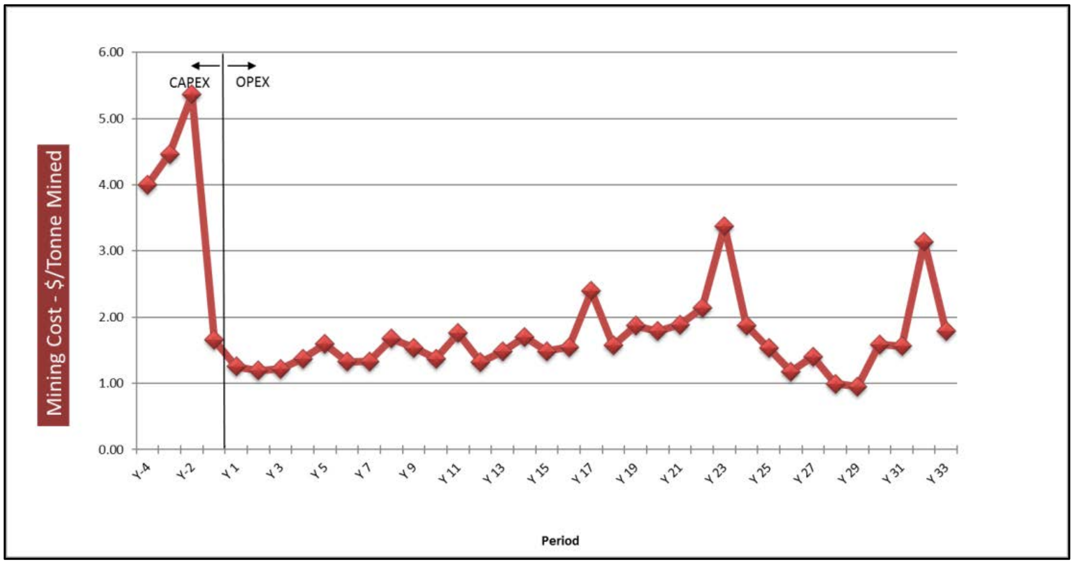
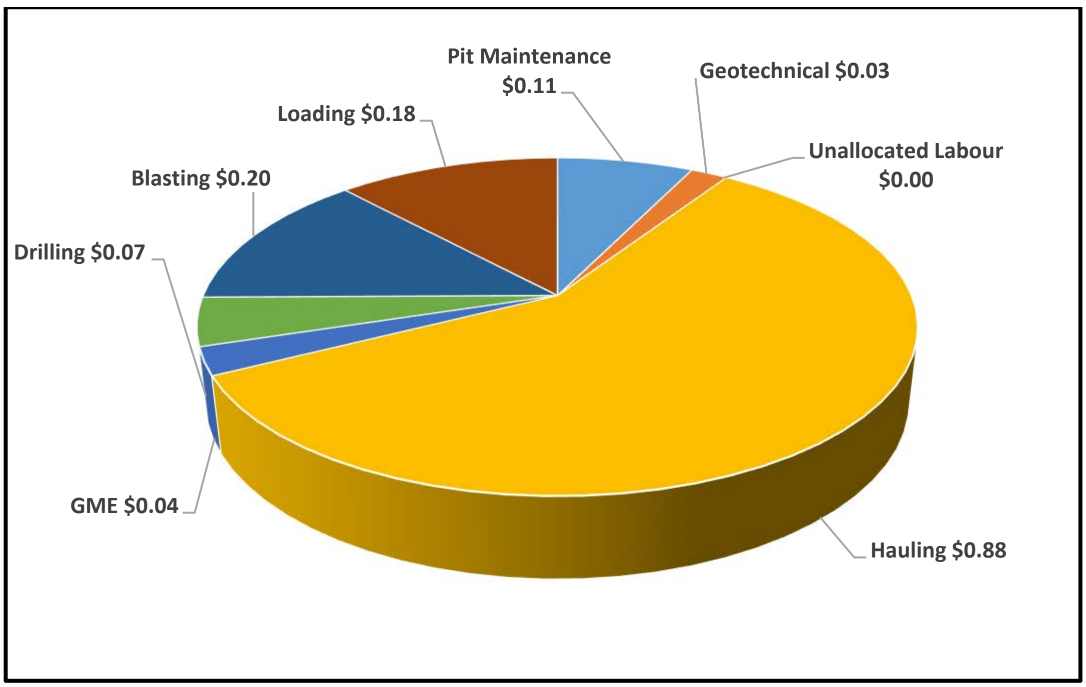


Figure 21.3 LOM Average Unit Operating Cost for Open Pit Mining (US$/t Material Mined)



21.3.2 UNDERGROUND MINING OPERATING COSTS

The underground mining operating cost estimates are based on first principles calculations and productivity modelling. The average daily production for each mine was used in the productivity model. The operating cost estimates include direct and indirect costs, which are estimated in a similar manner to direct and indirect capital costs. The indirect labour estimate was a combination of a factor based on the amount of mobile equipment operating for supervisor and maintenance staff, and experience at large, complex mines for technical staff. There are no contingencies added to the operating costs. Golder is responsible for the block cave operating cost estimates.

MITCHELL BLOCK CAVE OPERATING COSTS

The average Mitchell block cave mine operating cost is estimated to be approximately US$4.88/t, which includes the cost of equipment and labour required to move material from the drawpoint to the surface conveyor portal, and the fixed costs to operate the mine (Table 21.33). This includes operating the LHDs, secondary breakers, crushers and conveyors, and the labour required to plan and execute the mining plan.

Table 21.33 Summary of Mitchell Block Cave Mine Operating Cost by Activity

Activity	Cost (US$/t)	Percentage of Total (%)
Production Mucking	0.87	18
Production Locomotive	0.16	3
Crusher	0.20	4
Conveyors	0.90	18
Secondary Breaking	0.62	13
Drawpoint Rehabilitation	0.19	4
Mine Dewatering	0.10	2
Mine General Expenses	0.52	11
Indirect Labour*	0.66	14
Fixed Cost	0.66	13
Total	**4.88**	**100**

Note: *Direct labour is included within each activity cost.

IRON CAP BLOCK CAVE OPERATING COSTS

The average Iron Cap block cave mine operating cost is estimated to be US$5.22/t, which includes the cost of equipment and labour required to move material from the drawpoint to the MTT train tunnel and the fixed costs to run the mine (Table 21.34). This includes operating the LHDs, crushers, conveyors, mine services, and the labour required to plan and execute the mining plan. The dewatering cost is included in the fixed operating costs because the majority of the dewatering uses gravity and the cost per tonne is relatively low compared to Mitchell.

Table 21.34 Summary of Iron Cap Block Cave Mine Operating Cost by Activity

Activity	Cost (US$/t)	Percentage of Total (%)
Production LHD	1.12	21
Crusher	0.29	6
Conveyor	0.46	9
Secondary Breaking	0.73	14
Drawpoint Rehabilitation	0.28	5
Mine General Expenses	0.70	13
Indirect Labour*	0.94	18
Fixed Cost	0.70	13
Total	**5.22**	**100**

Notes: *Direct labour is included within each activity cost.

21.3.3 PROCESS OPERATING COSTS

SUMMARY

The LOM average annual process operating costs for the different mineralizations are estimated as:

- Mitchell and Iron Cap mineralization: US$249 million (US$5.25/t milled)
- Kerr mineralization: US$250 million (US$5.27/t milled)
- Sulphurets mineralization: US$273 million (US$5.75/t milled).

The process operating costs for these mineralizations are based on a process rate of 130,000 t/d and 94% plant availability. The estimated average operating cost for the Sulphurets ore is higher than the ores from the other deposits, due mainly to harder mineralization for the Sulphurets ore as compared to the other deposits. Due to the variations in operating costs for the different deposit ores, the average operating costs are 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 21.35, and include:

- personnel requirements, including supervision, operation and maintenance; salary/wage levels based on a project specific labour survey conducted by Seabridge, the payments include base salaries/labour rates and various burdens
- liner and grinding media consumption estimated from the Bond ball mill work index and abrasion index equations and quoted budget prices in Q1/Q2 2016 and/or Tetra Tech's database
- maintenance supplies based on approximately 6% of major equipment capital costs

- reagents based on test results and quoted budget prices in Q1/Q2 2016 and/or Tetra Tech's database
- other operation consumables including laboratory, filtering cloth, and service vehicles consumables
- power consumption for the process plant at the power unit cost of Cdn$0.062/kWh, or US$0.050/kWh (estimated by Brazier)
- no taxes or import duties are included in the estimate, unless specified.

Table 21.35 Summary of Process Operating Costs by Deposit

Area	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)
Human Power									
Operating Staff	39	3,921,000	0.083	39	3,921,000	0.083	39	3,921,000	0.083
Operating Labour	150	11,544,000	0.243	158	12,139,000	0.256	158	12,139,000	0.256
Maintenance	84	6,870,000	0.145	84	6,870,000	0.145	84	6,870,000	0.145
Subtotal Human Power	**273**	**22,335,000**	**0.471**	**281**	**22,930,000**	**0.483**	**281**	**22,930,000**	**0.483**
Major Consumables and Supplies									
Major Consumables									
Metal Consumables		55,134,000	1.162		68,233,000	1.438		53,891,000	1.136
Reagent Consumables		91,496,000	1.928		91,496,000	1.928		91,496,000	1.928
Supplies									
Maintenance Supplies		24,351,000	0.513		25,325,000	0.534		25,852,000	0.545
Operating Supplies		2,706,000	0.057		2,708,000	0.057		2,706,000	0.057
Subtotal Consumable and Supplies		**173,687,000**	**3.660**		**187,761,000**	**3.957**		**173,945,000**	**3.666**
Power Supply		52,868,000	1.114		62,153,000	1.310		53,165,000	1.120
Subtotal Power		**52,868,000**	**1.114**		**62,153,000**	**1.310**		**53,165,000**	**1.120**
Process Operating Cost Total	**273**	**248,890,000**	**5.245**	**281**	**272,844,000**	**5.750**	**281**	**250,039,000**	**5.269**

Note: Costs have been rounded to the nearest thousands of dollars.

PERSONNEL

The projected personnel requirements are between 273 and 281 persons, including:

- 39 staff for management and professional services
- between 150 and 158 operators, including personnel at laboratories for quality control, process optimization, and assaying
- 84 personnel for maintenance.

Salary/wage rates for management, technical support and operation are based on a project-specific labour survey conducted by Seabridge. The payments include base salaries/labour rates, holiday and vacation pay, government prescriptive benefits (e.g., Canadian Pension Plan, workman compensation insurance, etc.), discretionary employer sponsored benefits, and tool allowance costs.

The average total estimated personnel cost is approximately US$0.48/t milled. The detailed personnel description and costs are shown in Appendix L2 for each processing plant area.

OPERATING AND MAINTENANCE SUPPLIES

Major consumables and operating suppliers are estimated at US$3.66/t milled for Mitchell and Iron Cap mineralization, US$3.67/t milled for Kerr mineralization, and US$3.96/t milled for Sulphurets mineralization. The major consumables include metal and reagents consumables. The liner and grinding media consumption are estimated from the Bond abrasion index equation and the budget prices in Q1/Q2 2016 from the potential suppliers.

Reagent consumptions are estimated from laboratory test results and comparable operations. The reagent costs are estimated from Q1/Q2 2016 budget prices provided by potential suppliers.

Maintenance supplies are estimated at US$0.51/t milled for Mitchell and Iron Cap mineralization, US$0.55/t milled for Kerr mineralization, and US$0.53/t milled for Sulphurets mineralization. Maintenance supplies are estimated based on approximately 6% of major equipment capital costs.

OPERATING COSTS PER AREA OF OPERATION

Table 21.36 shows the operating cost of each processing area. Operating cost details for each processing area are further outlined in the following subsections and in Appendix L2.

Table 21.36 Operating Costs per Area of Operation by Deposit

Area	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)
Crushing, Grinding and Copper Flotation Plant	146	161,433,000	3.402	154	184,628,000	3.891	154	162,683,000	3.429
Tunnel Transport	44	11,496,000	0.242	44	11,496,000	0.242	44	11,496,000	0.242
Molybdenum Flotation Plant	8	5,881,000	0.124	8	5,884,000	0.124	8	5,880,000	0.124
Leach Plant	43	33,704,000	0.710	43	34,461,000	0.726	43	33,605,000	0.708
Cyanide Solution/Residue Handling	8	29,067,000	0.613	8	29,067,000	0.613	8	29,067,000	0.613
CIL Water Treatment	4	1,484,000	0.031	4	1,484,000	0.031	4	1,484,000	0.031
Tailing Pumping/Reclaim Water	20	5,824,000	0.123	20	5,824,000	0.123	20	5,824,000	0.123
Total	**273**	**248,890,000**	**5.245**	**281**	**272,844,000**	**5.750**	**281**	**250,039,000**	**5.269**

Note: Costs have been rounded to the nearest thousands of dollars.

Crushing, Grinding, Copper, and Pyrite Flotation

The operating costs for crushing, grinding, copper, and pyrite flotation is estimated to be approximately:

- US$3.40/t milled for Mitchell and Iron Cap mineralization
- US$3.43/t milled for Kerr mineralization
- US$3.89/t milled for Sulphurets mineralization.

The breakdown costs are shown in Table 21.37. The cost estimate includes personnel to operate the processing 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. The average operating cost for the Sulphurets ore is estimated be higher than the ores from the other deposits, mainly because of higher power and grinding media costs due to harder mineralization of the Sulphurets ore.

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:

- 846 GWh for Mitchell and Iron Cap ores
- 1,018 GWh for Sulphurets ore
- 854 GWh for Kerr ore.

Details of the estimate are shown Appendix L2.

Table 21.37 Crushing, Grinding, and Copper/Pyrite Flotation Operating Costs by Deposit

Area	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	26	2,535,000	0.053	26	2,535,000	0.053	26	2,535,000	0.053
Operating Labour	60	4,515,000	0.095	68	5,110,000	0.108	68	5,110,000	0.108
Maintenance	60	4,981,600	0.105	60	4,981,600	0.105	60	4,982,000	0.105
Subtotal Personnel	**146**	**12,032,000**	**0.254**	**154**	**12,627,000**	**0.266**	**154**	**12,627,000**	**0.266**
Supplies									
Major Consumables									
Metal Consumables	-	54,617,000	1.151	-	67,713,000	1.427	-	53,373,000	1.125
Reagent Consumables	-	34,213,000	0.721	-	34,213,000	0.721	-	34,213,000	0.721
Supplies									
Maintenance Supplies	-	17,093,000	0.360	-	18,067,000	0.381	-	18,594,000	0.392
Operating Supplies	-	1,498,400	0.032	-	1,498,000	0.032	-	1,498,000	0.032
Power Supply	-	41,981,000	0.885	-	50,509,000	1.064	-	42,377,000	0.893
Subtotal Supplies	**-**	**149,401,000**	**3.149**	**-**	**172,001,000**	**3.625**	**-**	**150,056,000**	**3.162**
Total	**146**	**161,433,000**	**3.402**	**154**	**184,629,000**	**3.891**	**154**	**162,683,000**	**3.429**

Note: Costs have been rounded to the nearest thousands of dollars.

Molybdenum Flotation

Table 21.38 shows that the estimated operating cost for molybdenum flotation is approximately US$0.12/t milled for the mineralization. Eight 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 3.2 GWh. Details of the costs are shown in Appendix L2.

Table 21.38 Molybdenum Flotation Operation Costs: Mitchell and Iron Cap*

Area	Personnel	Annual Cost (US$)	Unit Cost (US$/t milled)
Personnel			
Operating Labour	8	698,000	0.015
Subtotal Personnel	8	**698,000**	**0.015**
Supplies			
Major Consumables			
Metal Consumables	-	519,000	0.011
Reagent Consumables	-	3,310,000	0.070
Supplies			
Maintenance Supplies	-	296,000	0.006
Operating Supplies	-	20,000	0.000
Concentrate Leach	-	880,000	0.019
Power Supply	-	158,000	0.003
Subtotal Supplies	**-**	**5,183,000**	**0.109**
Total	**8**	**5,881,000**	**0.124**

Note: *The estimates are for Mitchell and Iron Cap mineralization; there is minor cost variation for the Kerr and Sulphurets, but insignificant to warrant reporting separately.Costs have been rounded to the nearest thousands of dollars.

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, including regrinding of gold bearing pyrite concentrate, is estimated to be US$0.71/t milled for all areas of mineralization, excluding the Sulphurets ore which is expected to be slightly higher at US$0.73/t milled. The personnel cost is estimated to be US$0.07/t milled (Table 21.39). The cost for major consumables is estimated at US$0.54/t milled. The power consumption for this circuit is estimated in the range of 58 to 73 GWh/a. A detailed cost estimate is shown in Appendix L2.

Table 21.39 Gold Leach and Recovery Circuit Operating Costs

	Area	Personnel	Annual Cost (US$)	Unit Cost (US$/t CIL)	Unit Cost (US$/t milled)
Mitchell/Iron Cap	**Personnel**				
	Operating Staff	7	728,000	0.132	0.015
	Operating Labour	20	1,539,000	0.280	0.033
	Maintenance	16	1,190,000	0.216	0.025
	Subtotal Personnel	**43**	**3,457,000**	**0.628**	**0.073**
	Supplies				
	Major Consumables				
	Major Consumables	-	25,500,000	4.630	0.537
	Supplies				
	Maintenance Supplies	-	1,074,000	0.195	0.022
	Operating Supplies	-	80,000	0.015	0.002
	Power Supply	-	3,593,000	0.652	0.076
	Subtotal Supplies	**-**	**30,247,000**	**5.492**	**0.637**
	Total	**43**	**33,704,000**	**6.120**	**0.710**
Sulphurets	**Personnel**				
	Operating Staff	7	728,000	0.132	0.015
	Operating Labour	20	1,539,000	0.280	0.033
	Maintenance	16	1,190,000	0.216	0.025
	Subtotal Personnel	**43**	**3,457,000**	**0.628**	**0.073**
	Supplies				
	Major Consumables				
	Major Consumables	-	25,500,000	4.630	0.537
	Supplies				
	Maintenance Supplies	-	1,074,000	0.195	0.022
	Operating Supplies	-	80,000	0.015	0.002
	Power Supply	-	4,350,000	0.789	0.092
	Subtotal Supplies	**-**	**31,004,000**	**5.629**	**0.653**
	Total	**43**	**34,461,000**	**6.257**	**0.726**
Kerr	**Personnel**				
	Operating Staff	7	728,000	0.132	0.015
	Operating Labour	20	1,539,000	0.280	0.033
	Maintenance	16	1,190,000	0.216	0.025
	Subtotal Personnel	**43**	**3,457,000**	**0.628**	**0.073**
	Supplies				
	Major Consumables				
	Major Consumables	-	25,500,000	4.630	0.537
	Supplies				
	Maintenance Supplies	-	1,074,000	0.195	0.022
	Operating Supplies	-	80,000	0.015	0.002
	Power Supply	-	3,494,000	0.634	0.074
	Subtotal Supplies	**-**	**30,148,000**	**5.474**	**0.635**
	Total	**43**	**33,605,000**	**6.102**	**0.708**

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$0.61/t milled for all the mineralization. This cost includes a labour cost of US$0.01/t milled and a total processing reagent supplies cost of US$0.58/t milled. The annual power consumption will be approximately 10.5 GWh. The estimates are shown in Table 21.40. A detailed cost estimate is shown in Appendix L2.

Table 21.40 Cyanide Recovery and Destruction Operating Costs

Area	Personnel	Annual Cost (US$)	Unit Cost (US$/t CIL)	Unit Cost (US$/t milled)
Personnel				
Operating Staff	8	595,000	0.108	0.0130
Subtotal Personnel	**8**	**595,000**	**0.108**	**0.0130**
Consumables and Supplies				
Reagent Consumables		27,535,000	4.999	0.5800
Maintenance Supplies		398,000	0.072	0.0090
Operating Supplies		16,000	0.003	0.0003
Power Supply		523,000	0.095	0.0110
Subtotal Supplies		**28,472,000**	**5.170**	**0.6000**
Total	**8**	**29,067,000**	**5.278**	**0.6130**

Note: Costs have been rounded to the nearest thousands of dollars.

Tunnel Transport

The MTT tunnel is the main connection between the Mitchell Mine Site and the Treaty OPC. The tunnel will be equipped with electrically powered autonomous trains to transport the crushed ores from the Mitchell Mine Site to Treaty OPC. The tunnel will be used to transport the workers and operation required materials, such as various consumables, spare parts and equipment between the two sites. At both sides of the tunnel portals, there will be a material handling facility to ensure that the ores, materials and personnel will be transported in a safe and efficient way. The estimated cost for the tunnel transports is shown in Table 21.41. The total unit cost is estimated to be US$0.24/t milled, including power supply, which is estimated at 66.9 GWh/a.

Table 21.41 Tunnel Transport Operating Costs

Area	Personnel	Annual Cost (US$)	Unit Cost (US$/t milled)
Personnel			
Operating Labour	44	3,664,000	0.077
Subtotal Personnel	**44**	**3,664,000**	**0.077**
Supplies			
Maintenance/Operating Supplies	-	4,513,000	0.095
Power Supply	-	3,319,000	0.070
Subtotal Supplies	**-**	**7,832,000**	**0.165**
Total	**44**	**11,496,000**	**0.242**

Note: Costs have been rounded to the nearest thousands of dollars.

Tailing and Reclaimed Water Operation

The cost estimates for tailing delivery to the TMF and water management, including reclamation and releasing, for all the mineralization are shown in Table 21.42, which details the unit costs for labour, maintenance supplies, operating suppliers, and power supply. The total cost is expected to be approximately US$0.12/t milled. The major cost contribution of tailing and reclaim water operations is power consumption for the operations at the TMF. The annual power requirement is estimated to be approximately 65.0 GWh, which accounts for US$0.07/t milled. A more detailed breakdown is shown in Appendix L2.

Table 21.42 Tailing Delivery and Reclaimed Water Operating Costs

Area	Personnel	Annual Cost (US$)	Unit Cost (US$/t milled)
Personnel			
Operating Labour	20	1,539,000	0.032
Subtotal Personnel	**20**	**1,539,000**	**0.032**
Supplies			
Maintenance Supplies	-	937,000	0.020
Operating Supplies	-	123,000	0.003
Power Supply	-	3,225,000	0.068
Subtotal Supplies	**-**	**4,285,000**	**0.091**
Total	**20**	**5,824,000**	**0.123**

Note: Costs have been rounded to the nearest thousands of dollars.

21.3.4 TMF DAM MANAGEMENT OPERATING COSTS

On average, the operating costs for tailing dam ongoing construction by cyclone sands, including seepage water pumping costs, are estimated to be approximately US$6.1 million/a, or US$0.13/t milled.

21.3.5 MINE SITE WATER MANAGEMENT COSTS

Overall SWM at the Mine Site includes HDS site water treatment, selenium removal treatment, and site water pumping. The average annual cost for Mine Site SWM is estimated to be approximately US$34.0 million/a, or US$0.72/t milled, at a mill feed rate of 130,000 t/d. The costs for the HDS WTP and Selenium WTP are detailed in Table 21.43 and Table 21.44, respectively.

The estimated average operating cost for the HDS WTP is approximately US$0.46/t milled at a mill feed rate of 130,000 t/d, or US$0.31/m^3 water, treated at an average flow rate of approximately 70 Mm^3/a. The maintenance manpower will come from the overall Mine Site maintenance team. The major cost for HDS water treatment is reagent consumption at US$0.33/t milled. Power consumption is estimated to be approximately 40.3 GWh/a.

Table 21.43 Water Treatment Plant Operating Costs

Area	Personnel	Annual Cost (US$)	Unit Cost (US$/t milled)*
Personnel			
Operating Labour	17	1,396,000	0.029
Subtotal Personnel	**17**	**1,396,000**	**0.029**
Supplies			
Reagent Consumables	-	15,577,000	0.328
Maintenance/Operating Supplies	-	3,062,000	0.065
Power Supply	-	1,998,000	0.042
Subtotal Supplies	**-**	**20,637,000**	**0.435**
Total	**17**	**22,033,000**	**0.464**

Note: *at a nominal mill process rate of 130,000 t/d
Costs have been rounded to the nearest thousands of dollars.

The estimated LOM average operating cost for the Selenium WTP is approximately US$0.20/t milled at a mill feed rate of 130,000 t/d, or US$1.02/m^3 water, treated at an average flowrate of approximately 9.3 Mm^3/a. The maintenance manpower will come from the overall site services maintenance team. The major cost for selenium water treatment is US$0.15/t milled for reagent consumption and maintenance. Power consumption is estimated to be approximately 11.8 GWh/a.

Table 21.44 Selenium Removal Water Treatment Plant Operating

Area	Personnel	Annual Cost (US$)	Unit Cost (US$/t milled)*
Personnel			
Operating Labour	17	1,550,000	0.033
Subtotal Personnel	**17**	**1,550,000**	**0.033**
Supplies			
Reagent Consumables	-	3,421,000	0.072
Maintenance/Operating Supplies	-	3,911,000	0.083
Power Supply	-	587,000	0.012
Subtotal Supplies	**-**	**7,919,000**	**0.167**
Total	**17**	**9,469,000**	**0.200**

Note: *at a nominal mill process rate of 130,000 t/d
Costs have been rounded to the nearest thousands of dollars.

21.3.6 GENERAL AND ADMINISTRATIVE

G&A costs are costs that do not relate directly to mining or processing operating costs. These costs include:

- Personnel: executive management, staffing in accounting, supply chain and logistics, human resources, external affairs functions, and other G&A departments
- expenses : including insurance, off site offices, administrative supplies, medical services, legal services, human resources related expenses, travelling, community and environmental programs, accommodation/camp costs, air/bus crew transportation, regional and property taxes, and external assay/testing.

G&A costs are estimated at approximately US$43.3 million/a, or US$0.91/t milled at a nominal mill feed rate of 130,000 t/d, including approximately US$0.28/t for personnel and US$0.63/t for general expenses. The major costs are accommodation and crew air transportation, estimated at about US$15.6 million/a. A summary of the G&A estimate for personnel and general expenses are shown in Table 21.45.

Table 21.45 G&A Operating Cost Estimate

Area	Personnel	Total Cost (US$/a)	Unit Cost (US$/t milled)
Personnel			
General Management and Service Manpower	143	13,275,000	0.280
Subtotal Personnel	**143**	**13,275,000**	**0.280**
General Expense			
Offices Expenses/General Administration	-	459,000	0.010
Insurances	-	1,368,000	0.029
External Assay/Testing excluding Water Sample Assays	-	400,000	0.008
Safety and Training	-	784,000	0.017
Medical Service/First Aid	-	160,000	0.003
Security Supplies	-	160,000	0.003
Legal Services - Allowance	-	80,000	0.002
Regulatory Compliance/Permits - Allowance	-	160,000	0.003
Consulting – Allowance	-	400,000	0.008
Small Vehicles/Mobile Equipment	-	446,000	0.009
Corporate Support Allocated to KSM	-	1,200,000	0.025
Recruitment	-	160,000	0.003
Communications	-	718,000	0.015
Travel	-	453,000	0.010
External and Government Affairs/ Community Support	-	340,000	0.007
Accounting Services including Auditing	-	160,000	0.003
System Management/Computer Services	-	734,000	0.015
Professional Associations	-	80,000	0.002
Accommodation/Camp Costs	-	11,400,000	0.240
Regional Taxes and Licenses Allowance including Mineral Leases	-	800,000	0.017
Environmental Expenses, including Water Sample Assays	-	3,680,000	0.078
Crew Air Transportation	-	4,210,000	0.089
Bus Transportation	-	982,000	0.021
Warehouse	-	160,000	0.003
Miscellaneous	-	320,000	0.007
Lease and Rental Off-site	-	184,000	0.004
Subtotal Expense	**-**	**29,998,000**	**0.631**
Total	**143**	**43,273,000**	**0.911**

Note: Costs have been rounded to the nearest thousands of dollars.

21.3.7 SITE SERVICES

Overall site service cost is estimated at US$0.40/t milled or approximately US$18.9 million/a. The estimate is based on requirements for this remote site in northern BC and on in-house experience. The estimate, as shown in Table 21.46, includes:

- Personnel: general site services human power
- site 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 21.46 Overall Site Service Cost Estimate

Expenditure Area	Personnel	Total Cost (US$/a)	Unit Cost (US$/t milled)
Personnel			
Site Service Manpower	112	8,700,000	0.183
Subtotal Personnel	**112**	**8,700,000**	**0.183**
Expenses			
Potable Water/Waste Management			
Potable Water	-	80,000	0.002
Domestic Waste	-	160,000	0.003
Hazardous Waste	-	240,000	0.005
Sewage	-	160,000	0.003
Building Maintenance			
Supplies Operating	-	240,000	0.005
Supplies Repairs	-	480,000	0.010
Tool Allowance	-	320,000	0.007
Services Purchased	-	400,000	0.009
Small Vehicles/Equipment	-	1,123,000	0.024
Supplies	-	320,000	0.007
Building Heating	-	1,218,000	0.026
Tunnel Ventilation	-	400,000	0.008
Road Maintenance	-	2,011,000	0.042
Power Line Maintenance	-	400,000	0.009
Power Supply	-	1,775,000	0.038
Avalanche Control	-	746,000	0.016
Off-site Operation Expenses	-	141,000	0.003
Subtotal Expenses	**-**	**10,214,000**	**0.217**
Total	**112**	**18,914,000**	**0.400**

Note: Costs have been rounded to the nearest thousands of dollars.

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

Tetra Tech prepared an economic evaluation of the 2016 PFS based on a pre-tax financial model. For the 53-year LOM and 2.198 billion tonne Mineral Reserve, the following pre-tax financial parameters were calculated using the base case metal prices:

- 10.4% IRR
- 6.0-year payback on US$5.005 billion initial capital
- US$3.263 billion NPV at a 5% discount rate.

Seabridge engaged Lilburn in Denver, Colorado to prepare the tax component of the model for the post-tax economic evaluation for this 2016 PFS with the inclusion of applicable income and mining taxes, and they engaged PwC in Toronto, Ontario to review this work. PwC is an Ontario limited liability partnership, which is a member firm of PricewaterhouseCoopers International Limited, each member firm of which is a separate legal entity.

The following post-tax financial results were calculated:

- 8.0% IRR
- 6.8-year payback on US$5.005 billion initial capital
- US$1.539 billion NPV at a 5% discount rate.

The base case results apply the following key inputs:

- gold – US$1,230/oz
- copper – US$2.75/lb
- silver – US$17.75/oz
- molybdenum – US$8.49/lb
- exchange rate – Cdn$1.00 to US$0.80.

Sensitivity analyses, along with multiple additional metal price scenarios, were developed to evaluate the Project economics.

The detailed financial model is provided in Appendix M.

22.2 PRE-TAX MODEL

Metal revenues projected in the Project cash flow models are based on the average metal values indicated in Table 22.1.

Table 22.1 Metal Production from the KSM Project

	Years 1 to 7	LOM
Total Tonnes to Mill ('000)	322,750	2,198,559
Annual Average Tonnes to Mill ('000)	46,107	41,484
Average Grades		
Gold (g/t)	0.82	0.55
Copper (%)	0.24	0.21
Silver (g/t)	2.8	2.6
Molybdenum (ppm)	48.3	42.6
Total Production		
Gold ('000 oz)	6,529	28,597
Copper ('000 lb)	1,434,560	8,270,423
Silver ('000 oz)	18,224	114,671
Molybdenum ('000 lb)	11,154	62,080
Average Annual Production		
Gold ('000 oz)	933	540
Copper ('000 lb)	204,937	156,052
Silver ('000 oz)	2,603	2,164
Molybdenum ('000 lb)	1,593	1,171

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.

Metal revenues, principally gold and copper, were calculated based on each scenario's prices. Operating cost for mining, processing, site services, G&A, tailing storage and handling and water treatment, energy recovery areas as well as off-site charges (smelting, refining, transportation, and royalties) were deducted from the revenues to derive annual operating cash flow.

Initial and sustaining capital costs as well as closure and reclamation costs have been incorporated on an annual basis over the mine life and deducted from the operating cash flow to determine the net cash flow before taxes. Initial capital expenditures include costs accumulated prior to first production of concentrate, including all pre-production mining costs. Sustaining capital includes expenditures for mining and processing additions, replacement of equipment, and TMF expansions.

Initial and sustaining capital costs applied in the economic analysis are US$5.005 billion and US$5.503 billion, respectively. LOM PST applicable to initial and sustaining capital is estimated to be US$134 million.

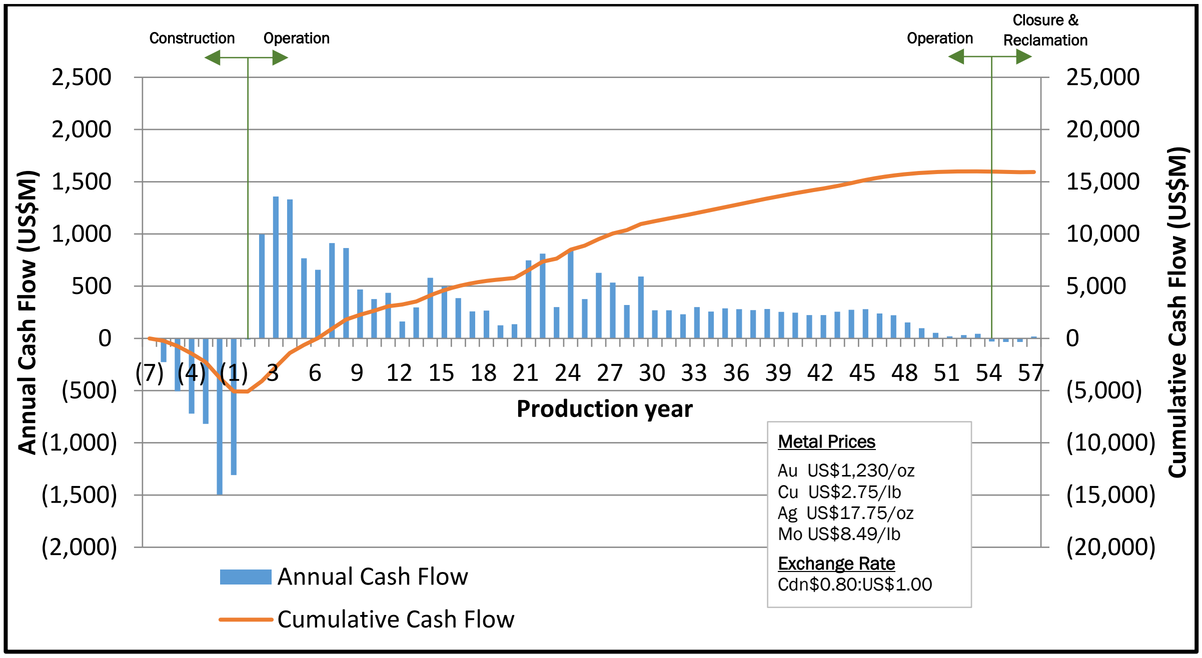
Financial evaluations account for physical reclamation costs at various times in the LOM, for the development of a fund to address water treatment costs post reclamation and for special use securities associated with permanent access roads.

Working capital is estimated at two months of receivables and one month of payables and varies from year to year. The working capital is recovered at the end of the mine life.

Pre-production construction period is estimated to be six years. NPV and IRR reported in this section are estimated at the start of this six-year period.

The pre-tax undiscounted annual cash flows are illustrated in Figure 22.1.

Figure 22.1 Pre-tax Undiscounted Annual and Cumulative Cash Flow



22.3 METAL PRICE SCENARIOS

The base case uses the three-year average metal prices as of July 2016 and a US$/Cdn$ exchange rate of 0.80. In addition to the base case, three metal price/exchange rate scenarios were also developed: the first uses the metal prices and exchange rate used in mine optimization and design (2016 Design Case); the second uses the spot metal prices and closing exchange rate on July 1, 2016 (Recent Spot Case); and the third uses higher metal prices to indicate upside potential (Alternate Case). The input parameters and pre-tax results of all scenarios are shown in Table 22.2.

Table 22.2 Summary of the Pre-tax Economic Evaluations

	Unit	Base Case	2016 Design Case	Recent Spot Case	Alternate Case
Gold	US$/oz	1,230.00	1,200.00	1,350.00	1,500.00
Copper	US$/lb	2.75	2.70	2.20	3.00
Silver	US$/oz	17.75	17.50	20.00	25.00
Molybdenum	US$/lb	8.49	9.70	7.00	10.00
Exchange Rate	US:Cdn	0.80	0.83	0.77	0.80
Undiscounted NCF	US$ million	15,933	13,727	16,101	26,319
NPV (at 3%)	US$ million	6,217	5,128	6,461	11,138
NPV (at 5%)	US$ million	3,263	2,510	3,507	6,541
NPV (at 8%)	US$ million	960	475	1,175	2,928
IRR	%	10.4	9.2	11.1	14.6
Payback	years	6.0	6.5	5.6	4.1
Cash Cost/oz Au	US$/oz	277	311	404	183
Total Cost/oz Au	US$/oz	673	720	787	580

Note: net cash flow (NCF)

22.4 SENSITIVITY ANALYSIS

Tetra Tech investigated the sensitivity of NPV, IRR and payback period to the key Project variables. Using the base case as a reference, each of key variables was changed between -30% and +30% in 10% increments while holding the other variables constant.

Sensitivity analyses were carried out on the following key variables:

- gold, copper, silver, and molybdenum metal prices
- exchange rate
- capital costs
- operating costs.

The analyses are presented graphically as financial outcomes in terms of pre-tax NPV, IRR, and payback period. The Project NPV is most sensitive to gold price and exchange rate, followed by operating costs, copper price and capital costs. The IRR is most sensitive to exchange rate, capital costs and gold price, followed by operating costs and copper price. The payback period is most sensitive to gold price and exchange rate, followed by capital costs, copper price and operating costs. Since majority of costs are in Canadian currency and the economic analysis is developed in American currency, a significant increase in the exchange rate by 30% will result in a significant increase in the costs when converted to American currency and this leads to sharp increase in the payback period. Also, when gold price decreases by 30%, the revenue side decreases significantly and this results in sharp increase in the payback period. Financial outcomes are relatively insensitive to silver and molybdenum prices. The NPV, IRR, and payback sensitivities are shown in Figure 22.2, Figure 22.3, and Figure 22.4.

Figure 22.2 Sensitivity Analysis of Pre-tax NPV at a 5% Discount Rate

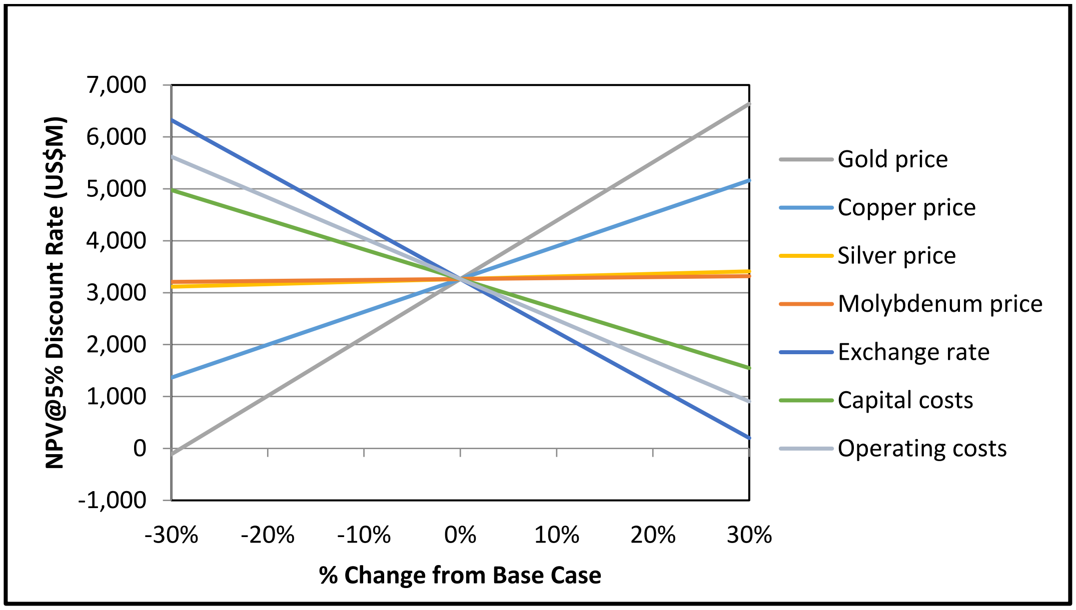


Figure 22.3 Sensitivity Analysis of Pre-tax IRR

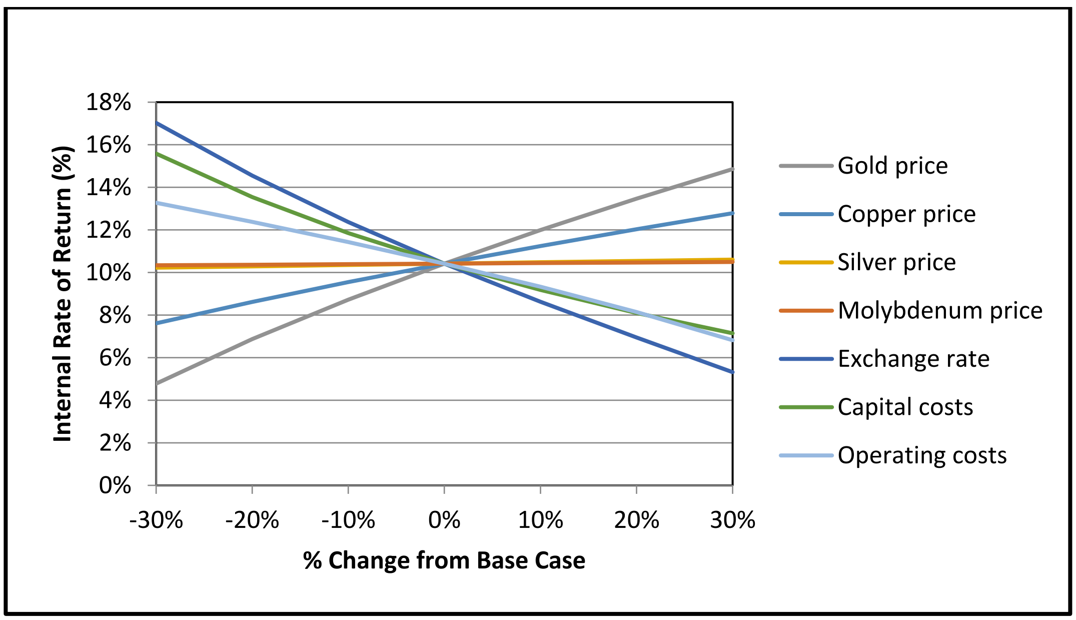
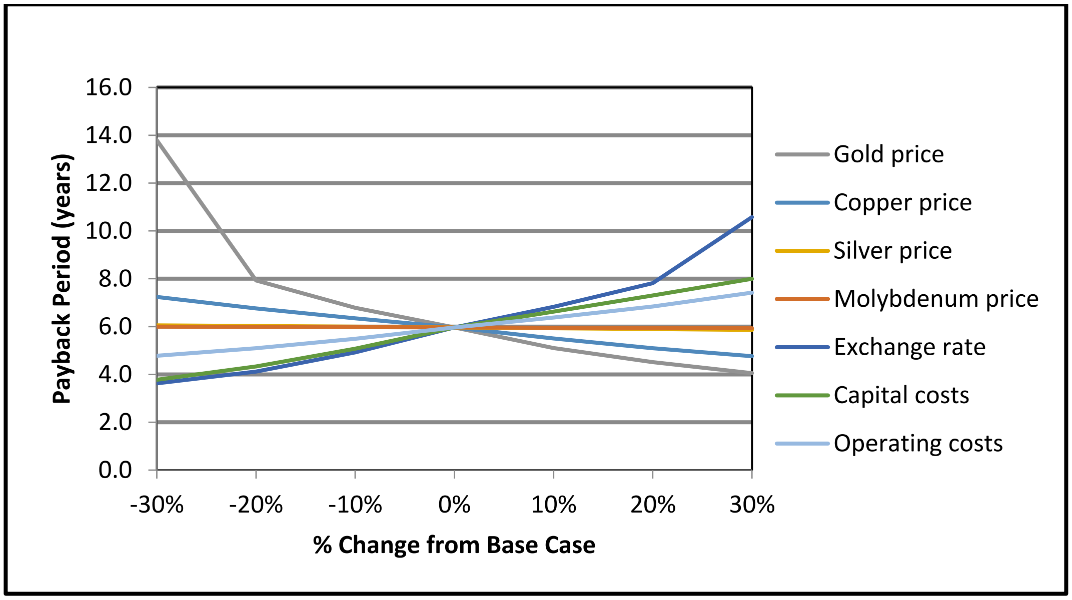


Figure 22.4 Sensitivity Analysis of Pre-tax Payback Period



22.5 POST-TAX FINANCIAL EVALUATIONS

Seabridge engaged Lilburn in Denver, Colorado to prepare the tax component of the model for the post-tax economic evaluation for this 2016 PFS with the inclusion of applicable income and mining taxes. Seabridge also engaged PwC in Toronto, Ontario to review the tax component of the model prepared by Lilburn.

The following general tax regime was recognized as applicable at the time of report writing:

22.5.1 CANADIAN FEDERAL AND BC PROVINCIAL INCOME TAX REGIME

The federal and BC provincial income taxes are calculated using the currently enacted corporate rates of 15% for federal and 11% for BC.

For both federal and provincial income tax purposes, capital expenditures are accumulated in tax pools that can be deducted against mine income at different prescribed rates, depending on the type of capital expenditures.

Historically, pre-production mining expenditures are accumulated in the Canadian Exploration Expense (CEE) pool. The CEE pool is generally amortized at 100%, to the extent of taxable income from the mine. A phase-out rule was enacted in 2013 to phase out the treatment of pre-production mine development expenses as CEE to Canadian Development Expense (CDE) from 2015 to 2017. Effective 2017, all the pre-production mine development expenses are treated as CDE.

In addition to pre-production mine development expenses (that are reclassified as CDE due to the phase-out rule), Canadian resource property acquisition costs and the costs of mine shafts, main haulage ways, and other underground workings are considered CDE and are accumulated in the CDE pool. The KSM Financial Model treats all such expenses as CDE.

Fixed assets acquired for the mine are accumulated in an undepreciated capital cost pool (Class 41) and are generally amortized at 25% on a declining balance basis. Certain fixed assets (acquired after March 20, 2013 and before 2021) may qualify to be accumulated in a Class 41.1 pool that can be amortized at an accelerated rate of up to 100%. However, as a substantive portion of the fixed assets are expected to be acquired post-2020 (after the phase out of the Class 41.1 pool), the KSM Financial Model assumes that the accelerated depreciation will not be available.

The Project is expected to incur costs related to the NTL as described in Section 21.2.6. These sustaining capital costs are expected to be incurred on a property that is not owned by the mine, the costs associated with the NTL should be treated as eligible capital expenditures for income tax purposes. Effective January 1, 2017, eligible capital expenditures are treated as a Class 14.1 asset with an amortization rate of 5%. The KSM Financial Model treats all the costs associated with the NTL overrun as a Class 14.1 asset.

22.5.2 BC Mineral Tax Regime

The BC Mineral Tax regime is a two tier tax regime, with a 2% tax and a 13% tax.

The 2% tax is assessed on "net current proceeds", which is defined as gross revenue from the mine less mine operating expenditures. Hedging income and losses, royalties and financing costs are excluded from operating expenditures. The 2% tax is accumulated in a Cumulative Tax Credit Account (CTCA) and is fully creditable against the 13% tax.

All capital expenditures, both mine development costs and fixed asset purchases, and mine operating expenditures are accumulated in the Cumulative Expenditures Account, which is amortized at 100% against the 13% tax.

The 13% tax is assessed on "net revenue", which is defined as gross revenue from the mine less any accumulated Cumulative Expenditures Account balance to the extent of the gross revenue from the mine for the year.

A "new mine allowance" is available in respect of new mine or capital costs incurred in connection with expansion of an existing mine commencing production with reasonable commercial quantities. Generally, this allowance provides that 133% of capital expenditures incurred prior to commencement of production may be used to offset net revenue for BC mining tax purposes. Under current legislation, the provision for the new mine allowance is scheduled to expire on January 1, 2020.

A notional interest of 125% of the prevailing federal bank rate is calculated annually on any unused Cumulative Expenditures Account and CTCA and is added to the respective balances.

BC Mineral Tax is deductible for federal and provincial income tax purposes.

22.5.3 TAXES AND POST-TAX FINANCIAL RESULTS

At the base-case long-term metal prices and exchange rate used for this study, total estimated taxes payable on KSM profits are $5.951 billion over the 53-year LOM. The total estimated taxes payable by the Project in all scenarios provided are shown in Table 22.3.

Table 22.3 Component of the Various Taxes for all Scenarios

	Unit	Base Case	2016 Design Case	Recent Spot Case	Alternate Case
Gold	US$/oz	1230.00	1200.00	1350.00	1500.00
Copper	US$/lb	2.75	2.70	2.20	3.00
Silver	US$/oz	17.75	17.50	20.00	25.00
Molybdenum	US$/lb	8.49	9.70	7.00	10.00
Exchange Rate	US:Cdn	0.80	0.83	0.77	0.80
Corporate Tax (Federal)	US$ million	2,229	1,953	2,242	3,562
Corporate Tax (Provincial)	US$ million	1,635	1,433	1,644	2,612
BC Mineral Tax	US$ million	2,087	1,804	2,106	3,424
Total Taxes	**US$ million**	**5,951**	**5,190**	**5,993**	**9,598**

Post-tax financial results are summarized in Table 22.4.

Table 22.4 Summary of Post-tax Financial Results

	Unit	Base Case	2016 Design Case	Recent Spot Case	Alternate Case
Gold	US$/oz	1230.00	1200.00	1350.00	1500.00
Copper	US$/lb	2.75	2.70	2.20	3.00
Silver	US$/oz	17.75	17.50	20.00	25.00
Molybdenum	US$/lb	8.49	9.70	7.00	10.00
Exchange Rate	US$:Cdn$	0.80	0.83	0.77	0.80
Undiscounted NCF	US$ million	9,983	8,537	10,109	16,721
NPV (at 3%)	US$ million	3,513	2,789	3,691	6,696
NPV (at 5%)	US$ million	1,539	1,028	1,718	3,663
NPV (at 8%)	US$ million	-2	-343	161	1,282
IRR	%	8.0	7.0	8.5	11.4
Payback	years	6.8	7.4	6.4	4.9

Table 22.5 summarizes the Project's annual cash flow for the pre-production period, Years 1 to 7, and the LOM, providing mine and mill production, revenue projections,

operating costs and capital costs, and undiscounted cash flows both before and after taxes.

Table 22.5 KSM Project Annual Cash Flow for Pre-production Period, Years 1 to 7 and LOM

		Pre-prod. Period	Production Periods							
	Unit	Years -6 to -1	Year 1	Year 2	Year 3	Years 4	Years 5	Year 6	Year 7	LOM
Mine and Mill Production										
Waste Mined	Mt	90	141	136	40	84	129	130	111	3,003
Mill Feed Processed	Mt	-	38	47	47	47	47	47	47	2,198
Grade										
Gold	g/t	-	0.72	0.82	0.96	0.98	0.71	0.72	0.82	0.55
Copper	%	-	0.25	0.28	0.30	0.27	0.18	0.18	0.21	0.21
Silver	g/t	-	2.5	2.0	2.3	4.5	2.4	2.2	3.4	2.6
Molybdenum	ppm	-	30	47	61	20	56	65	56	43
Metal Recovered										
Gold	Moz	-	0.7	0.9	1.1	1.2	0.8	0.8	1.0	28.6
Copper	Mlb	-	175.3	251.6	273.2	243.2	155.0	154.6	181.6	8,270.4
Silver	Moz	-	1.9	1.9	2.1	4.9	2.0	1.9	3.4	114.7
Molybdenum	Mlb	-	0.6	1.5	2.6	0.0	2.0	2.4	2.0	62.1
Gross Revenue										
Gold Revenues	US$ million	-	810	1,167	1,385	1,448	1,015	1,019	1,187	35,174
Copper	US$ million	-	482	692	751	669	426	425	500	22,744
Silver	US$ million	-	35	34	38	87	36	34	60	2,035
Molybdenum	US$ million	-	5	13	22	-	17	20	17	527
Gross Revenue	US$ million	-	1,332	1,906	2,196	2,204	1,495	1,498	1,763	60,480
Total On-site and Off-site Operating Costs	US$ million	-	662	774	739	745	692	753	775	33,216
Operating Cash Flow	US$ million	-	670	1,132	1,457	1,459	804	746	987	27,264
Total Capital Costs	US$ million	5,005	623	133	98	127	34	88	75	10,508
PST	US$ million	64	6	1	1	2	2	1	0	134
Reclamation- Water Treatment Fund and SUP Costs	US$ million	8	52	-	-	-	-	-	-	688
Pre-tax Undiscounted NCF	US$ million	(- 5,077)	(- 11)	998	1,358	1,331	767	657	912	15,933
Corporate Tax (Provincial)	US$ million	-	-	-	-	92	38	51	88	1,635
Corporate Tax (Federal)	US$ million	-	-	-	-	126	51	69	120	2,229
BC Mineral Tax	US$ million	-	14	24	31	31	17	17	22	2,087
Total Taxes	US$ million	-	14	24	31	249	105	137	230	5,951
Post-tax Undiscounted NCF	US$ million	(- 5,077)	(- 25)	974	1,327	1,082	662	521	682	9,983

22.6 ROYALTIES

No royalties are included in the KSM financial models. KSM is subject to a royalty of 1% of the NSR payable to Barrick Gold Corp., capped at US$3.6 million, with a predetermined buyout option. The full amount of the buyout option is paid in Year 1 in the financial model.

22.7 SMELTER TERMS

Based on Section 19.0, the following smelter terms for copper concentrate have been applied in the economic analysis:

- Copper: pay 96.5% with a minimum deduction of 1 unit (amount deducted has to equate to a minimum of 1% of the agreed concentrate copper assay). Refining charge is US$0.10/lb of payable copper.
- 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 US$7.00/accountable troy oz; lower gold contents are payable on a sliding scale to 90% payment at 1 g/dmt less a refining charge of US$7.00/accountable troy oz.
- Silver: pay 90% on the silver content in excess of 30 g/dmt less a refining charge of US$0.50/accountable troy oz.
- Treatment Charge: US$100.00/dmt of concentrate delivered.
- Penalty Charge: no penalty is applied according to concentrate assay data.
- Price participation: not applicable.

Gold and silver doré will generally include payment terms as follows:

- Gold: pay 99.8% of content less a refining charge of US$1.00/accountable oz. Doré transportation is assumed to be US$1.00/oz.
- Silver: pay 90.0% of content less a refining charge of US$1.00/accountable oz. Doré transportation is assumed to be US$ 1.00/oz.

Molybdenum concentrate contracts will generally include payment terms as follows:

- Molybdenum: pay 99% of content less a treatment charge of US$2.00/accountable lb.

22.8 CONCENTRATE AND DORÉ TRANSPORT LOGISTICS

For this 2016 PFS, an assumption is being made that the copper concentrates will be shipped in bulk and that the projected volume averages approximately 350,000 t/a (Years 1 to 10 average). A further assumption is being made that the customer base will be in Asia and that the concentrates will be delivered to Asian ports in handy size ocean vessels. Copper concentrate from the mine site will be transported by truck to a commercial port in Stewart B.C. Transportation costs for the copper concentrate are listed below:

- Trucking: US$38.06/wmt
- port storage and handling: US$14.40/wmt
- ocean transport to Asian port: US$26.00/wmt
- moisture content: 9%.

Gold and silver doré transportation cost is assumed to be US$1.00/oz.

For this 2016 PFS, an assumption is being made that the processed molybdenum will be loaded in one tonne bags for transport purposes and that the output is approximately 1,800 t/a. A further assumption is being made that the customer base will be in Asia and that the molybdenum will be delivered in standard ocean containers. 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: US$73.20/wmt
- port storage and handling: US$12.20/wmt
- ocean transport to Asian port: US$88.93/wmt
- moisture content: 5%.

22.8.1 CONCENTRATE TRANSPORT INSURANCE

Based on Section 19.0, an insurance rate of 0.125% was applied to the provisional invoice value of the concentrates and doré to cover land-based and ocean transport from the mine site to the smelter.

22.8.2 MARKETING AND OWNERS REPRESENTATION

Based on Section 19.0, a US$8.50/dmt charge was applied for marketing and 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.

22.8.3 CONCENTRATE LOSSES

Based on Section 19.0, for deliveries to Asia, an overall weight loss of 0.1% was applied in the economic analysis.

23.0 ADJACENT PROPERTIES

In 2010, Pretium purchased the Snowfield and Brucejack mineral resource properties from Silver Standard Resources, Inc. In February 2011, Pretium announced an updated estimate of Mineral Resources for their Snowfield project, which abuts against the east side 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 2011).

The QP responsible for this section of this Technical Report has not verified the Mineral Resources that were disclosed by Pretium for their Snowfield deposit. 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, or Iron Cap zones.

Pretium disclosed Mineral Resources for their Brucejack project in an updated NI 43-101 Technical Report dated December 19, 2013. The Brucejack deposit is located approximately 6 km east of Seabridge's Kerr deposit. Pretium disclosed Proven and Probable Mineral Reserves for the Brucejack project in a Feasibility Study and Technical Report that was dated June 19, 2014 (Tetra Tech 2014).

The QP responsible for this section of this Technical Report has not verified the publicly disclosed Mineral Resources and Mineral Reserves associated with the Brucejack project. Furthermore, the QP does not believe that the Brucejack mineralization is necessarily indicative of the mineralization associated with the various mineralized zones at the Property.

Table 23.1 Pretium Snowfield Mineral Resources Using a 0.30 g/t Cut-off

Resource Category	Tonnes (Mt)	Au (g/t)	Ag (g/t)	Cu (%)	Mo (ppm)	Re (ppm)	Au (000 oz)	Ag (000 oz)	Cu (Blb)	Mo (Mlb)	Re (Moz)
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	0.10	83.6	0.50	20,934	65,444	2.60	217.5	19.0
Measured + Indicated	1,370.1	0.59	1.72	0.10	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

Source: Pretium website (http://www.pretivm.com)

24.0 OTHER RELEVANT DATA AND INFORMATION

The PEA was undertaken to evaluate a different approach to developing the Project by emphasizing low-cost block cave mining and reducing the number and size of the open pits, which significantly reduces the surface disturbances in the re-designed Project. The PEA is a conceptual level of study based on the same Mineral Resource estimates used in the 2016 PFS, except the Inferred Mineral Resources are included in the PEA project design, and projected economics.

The PEA envisages a combined open pit/underground block cave mining operation that is planned to operate for 51 years. The proposed Process Plant for the PEA mine design will have an average process rate of 170,000 t/d. The Mitchell open pit and Deep Kerr underground mines will be the main source of mill feed, contributing approximately 83% of the total plant feed over the LOM, supplemented by the Sulphurets open pit along with the Mitchell and Iron Cap underground mine production.

The flotation plant would produce a gold/copper/silver concentrate for transport by truck to a nearby sea port at Stewart, BC for shipment to Pacific Rim smelters. Metallurgical testing indicates that KSM can produce a clean concentrate with an average copper grade of 25% with a high gold and silver content, making it readily saleable. Separate gold-silver doré would be produced at the KSM processing facility.

The PEA is preliminary in nature and includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the results of the PEA will be realized. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

24.16 MINING METHODS

24.16.1 OPEN PIT MINING METHOD

The PEA mining study was carried out with the aim of reducing the amount of waste rock produced in the open pits and the mill feed to draw more on the underground resources. In order to accomplish this goal, smaller pits were designed for the Mitchell and Sulphurets deposits as well as mining the Kerr deposit solely by underground mining methods. This approach substantially shrinks the Project’s footprint. The PEA study provides conceptual open pit designs that are consistent with the inputs used in the 2016 PFS open pit mine designs.

SUMMARY

Amec Foster Wheeler based the PEA mine plan on the Measured, Indicated, and Inferred Mineral Resources contained in a pit created with the LG algorithm. An elevated cut-off strategy was used to define the Mineral Resources suitable for processing using a minimum NSR cut-off of Cdn$16/t (see 24.16 for NSR calculation details). These Mineral Resources are shown in Table 24.1.

Table 24.1 Mineral Resources Included in the PEA Open Pit Mine Plan

Class/Pit	Tonnage (kt)	Au (g/t)	Cu (%)	Ag (g/t)
Measured				
Mitchell Pit	223,712	0.79	0.20	3.0
Sulphurets Pit	-	-	-	-
Total Measured	223,712	0.79	0.20	3.0
Indicated				
Mitchell Pit	194,575	0.75	0.19	2.8
Sulphurets Pit	91,771	0.70	0.29	0.6
Total Indicated	286,346	0.73	0.22	2.1
Measured + Indicated				
Mitchell Pit	418,287	0.77	0.19	2.9
Sulphurets Pit	91,771	0.70	0.29	0.6
Total Measured + Indicated	510,058	0.76	0.21	2.5
Inferred				
Mitchell Pit	11,618	0.47	0.20	5.2
Sulphurets Pit	11,052	0.59	0.25	0.8
Total Inferred	22,670	0.53	0.22	3.1

For the Mitchell pit, the Mineral Resource model was subjected to an optimization analysis using GEOVIA Whittle™ software to define the mining limits, involving a base case and 45 additional pit shells. The contribution of each incremental shell to NPV was calculated based on a base processing capacity of 47.5 Mt/a and a discount rate of 5%. In the case of the Sulphurets pit, a much smaller pit than the optimal pit limit was chosen in order to constrain the amount of waste rock produced during the LOM.

The Project is designed as a conventional truck-shovel operation. The pit design for the Mitchell area includes three nested phases to balance stripping requirements while satisfying the Process Plant requirements. In the case of the Sulphurets area the material is mined in one phase.

Amec Foster Wheeler designed a RSF with a total storage capacity of approximately 265 Mm^3, and a mill-feed stockpile with a capacity of 58 Mm^3, enough to satisfy the production schedule's maximum stockpiling capacity.

The production schedule results in a LOM of 8 years with stockpile reclaim extending into Year 14. The mine will require three years of pre-production before the start of the

Process Plant operations. Five, 363 t haul trucks will be required during Year -3 of pre-production, increasing to 19 by Year -1 of pre-production, and peaking at 39 in production Years 1 to 3.

PIT OPTIMIZATION

The pit shells that define the ultimate Mitchell pit limit, as well as the internal phases, were derived using the LG pit optimization algorithm. In the case of the Sulphurets pit, a much smaller pit than the optimal pit limit was chosen in order to constrain the amount of waste produced during the LOM. The optimization process takes into account the information stored in the geological block model, pit slope angles by geotechnical sector, commodity prices, mining and processing costs, process recoveries, and the sales cost for the metal produced. Table 24.2 summarizes the primary optimization inputs.

Table 24.2 Optimization Inputs

Parameter	Unit	Value
Metal Prices		
Gold	US$/oz	1200
Copper	US$/lb	2.70
Silver	US$/oz	17.50
Discount Rate	%	5
Slope Angles Mitchell Pit		
Variable by Domain	degrees	34-54
Slope Angles Sulphurets Pit		
Variable by Domain	degrees	34-50
Dilution	%	Accounted for in NSR Calculations
Operating Cost		
Mining	US$/t	1.90
Process	US$/t milled	9.00
Processing Rate	kt/d	130
Process Recovery		
Variable by Deposit/Commodity	%	57.9-89.7

Amec Foster Wheeler imported the Mineral Resource model, containing grades, block percentages, material density, slope sectors, and rock types, into the optimization software. The optimization run was carried out using Measured, Indicated, and Inferred Mineral Resources to define the optimal mining limits.

The optimization run included 46 pit shells defined according to different revenue factors, where a revenue factor of 1 is the GEOVIA Whittle™ base case. To select the optimal pit shell that defines the ultimate pit limit, Amec Foster Wheeler conducted a pit-by-pit analysis to evaluate the contribution of each incremental shell to NPV, assuming a Process Plant capacity of 47.5 Mt/a and a discount rate of 5% (Figure 24.1). The pit-by-pit analysis includes three cases named best, specified and worst. The best and worst cases are used to provide a bound for the analysis and they do not obey practical mining constraints. In contrast, the specified case is based in a series of pushbacks selected by the planner and reflects the expected cash flow produced by the mining sequence. Following this analysis, the best pit shell is usually smaller than the Whittle base case pit shell. It can be observed from Figure 24.1 that, for the specified case, the highest cash flow is produced by pit shell 15 which is US$4.4 billion higher than the Whittle base case. The selected pit shell for the Mitchell pit is shown in Figure 24.2.

Figure 24.1 Pit-by-pit Analysis Mitchell

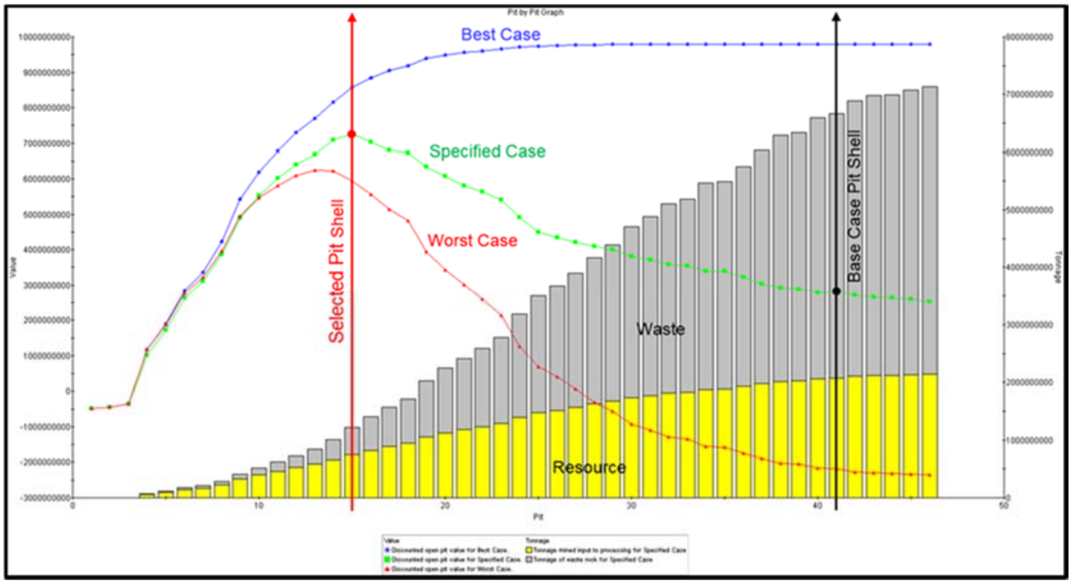
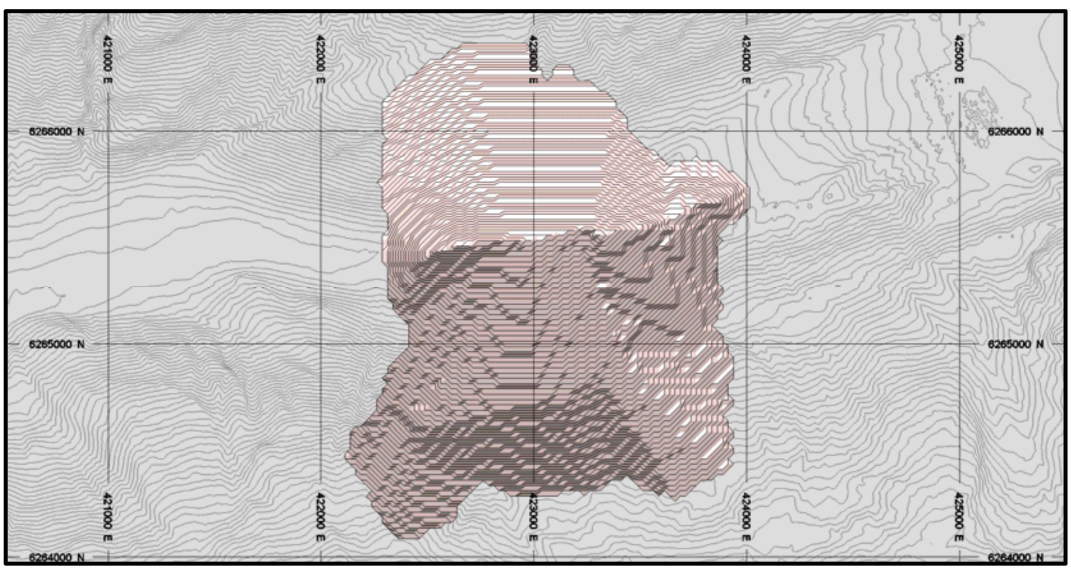


Figure 24.2 Selected Pit Shell Mitchell



MINE DESIGN

The Project is designed as a conventional truck-shovel operation with 363 t trucks and 56 m^3 and 40 m^3 shovels. For the Mitchell pit, the mine design includes three nested phases to balance stripping requirements while satisfying the Process Plant requirements, while Sulphurets is mined in one phase.

The design parameters include:

- ramp width of 38.20 m
- road grades of 8%
- bench heights of 15 m
- targeted mining width of 120 m
- berm interval of 30 m
- variable slope angles by sector
- minimum mining width of 50 m.

The Mitchell smoothed final pit design contains approximately 418.3 Mt of 0.77 g/t gold, 0.19% copper, and 2.9 g/t silver of Measured and Indicated Resources, as well as 11.6 Mt of 0.47 g/t gold, 0.20% copper, and 5.2 g/t silver of Inferred Mineral Resource and 414.8 Mt of waste, for a stripping ratio of 0.96:1.

The Sulphurets pit contains approximately 91.8 Mt of 0.70 g/t gold, 0.29% copper, and 0.6 g/t silver of Indicated Resources, plus 11.0 Mt of 0.59 g/t gold, 0.25% copper, and 0.8 g/t silver of Inferred Mineral Resource. These tonnages and grades were derived by following an elevated cut-off strategy in the production schedule. Figure 24.3 and Figure 24.4 show the ultimate pit designs for the Mitchell and Sulphurets pits, respectively.

Figure 24.3 Mitchell Pit Design

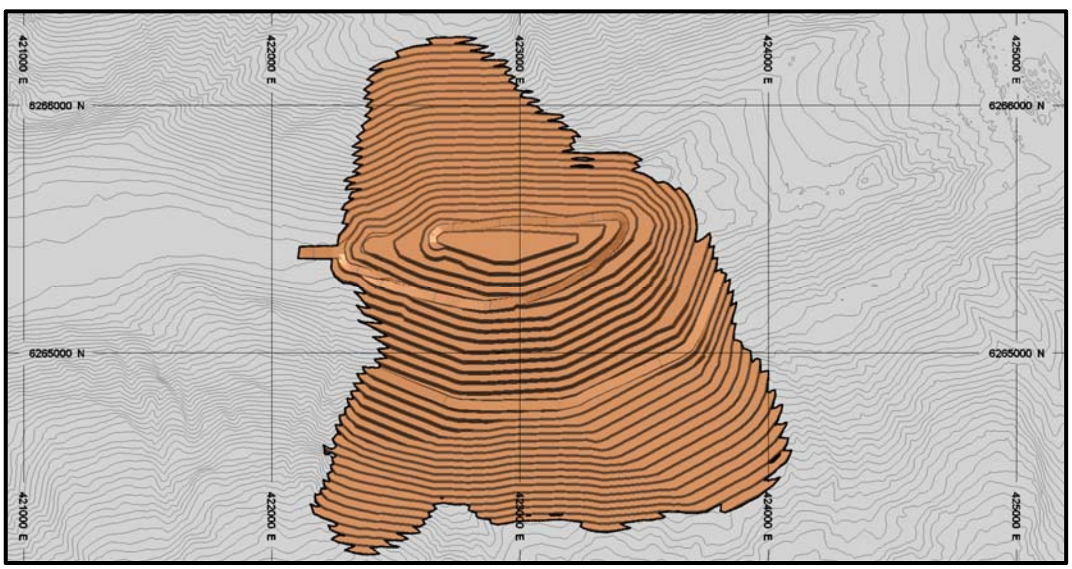
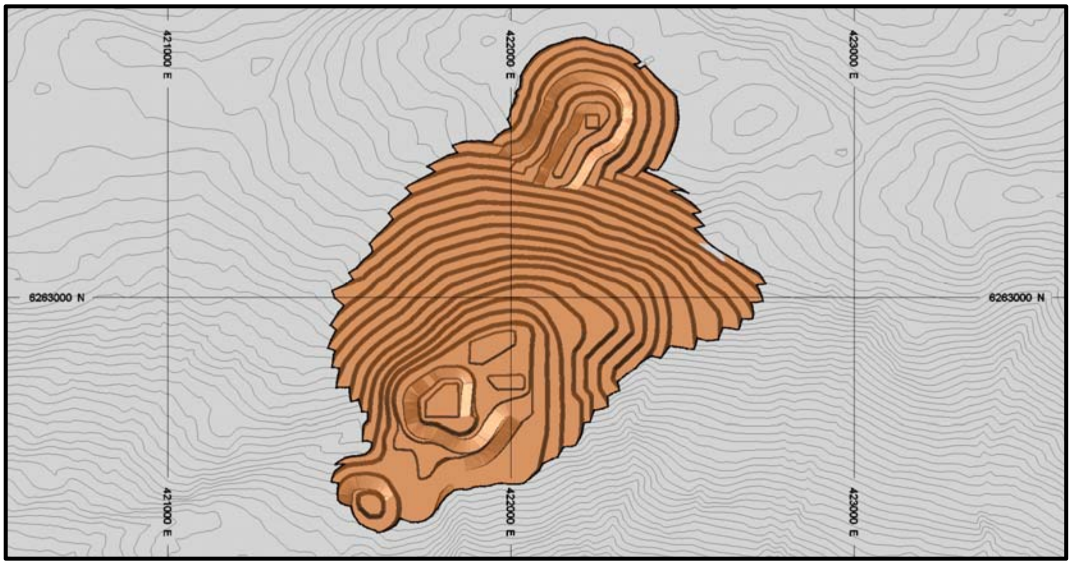


Figure 24.4 Sulphurets Pit Design



Waste Rock Facilities and Stockpile Designs

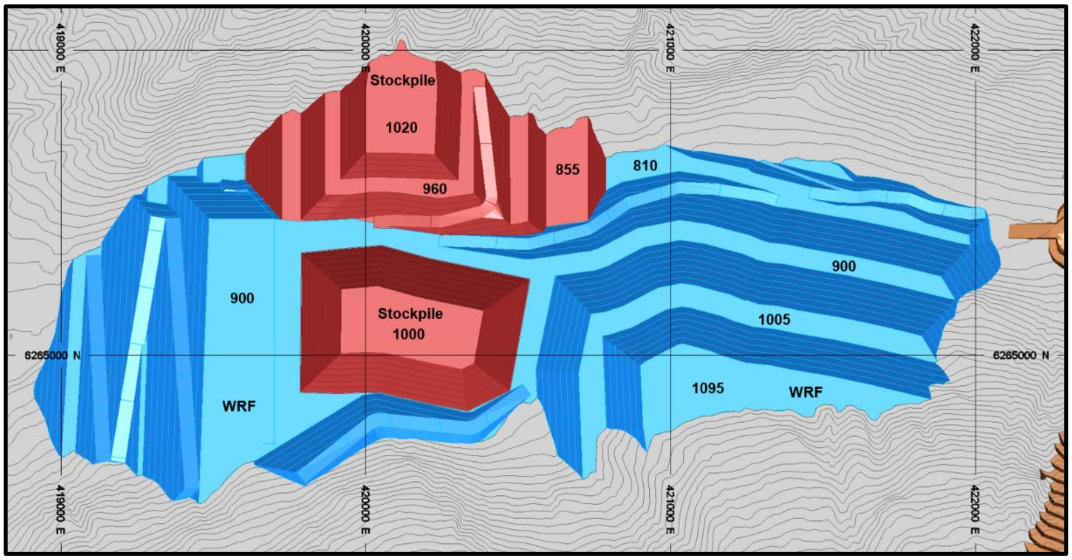
Mitchell and Sulphurets will use the same RSF and stockpiles facilities. Their design and construction should ensure physical and chemical stability during and after mining activities. To achieve this, the RFSs and stockpiles are designed to account for benching and geotechnical stability.

The RFS design criteria include:

- 30 m berms
- 2.5:1 overall slopes
- 15 m lifts
- 20% swell factor after compaction for estimating volumes.

The design was carried out to provide enough storage capacity for waste and low-grade stockpile. Figure 24.5 shows the RSF.

Figure 24.5 Waste Rock Facility and Stockpile Area Design



Production Schedule Open Pit

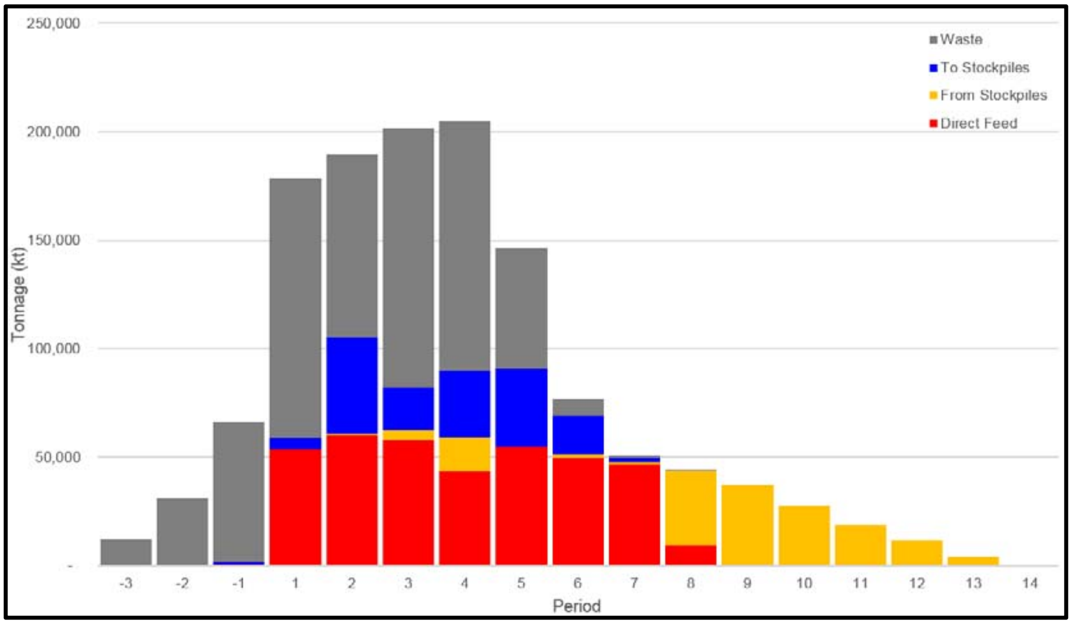
The production schedule for the Project includes the Process Plant ramp-up schedule to take into account the inefficiencies related to the start of operations. Full plant capacity is reached after two years of operation. In order to ensure a continuous feed to the mill, as well as to provide construction material for the associated facilities, the mine will require three years of pre-production.

The scheduling constraints set the maximum mining capacity at 200 Mt/a and the maximum number of benches mined per year at 12 per phase. To guide the schedule and to obtain the desired results, additional constraints included maximum stockpile capacity, and a reduction of mining capacity in later years during the LOM to balance the number of truck hours per period.

The schedule uses an elevated cut-off strategy feeding the resource with the highest grade first and sending lower grades to stockpiles for later processing. A minimum NSR cut-off of Cdn$15/t was used. This minimum cut-off was determined taking into account the stockpiling capacity and the availability of higher grade material from the Kerr underground mine. The schedule shows a LOM of eight years with stockpile reclaim extending into Year 14. The amount of re-handled mill feed is 157 Mt.

The average grades are 0.75 g/t gold, 0.21% copper, and 2.5 g/t silver. The LOM schedule is shown in Figure 24.6. The mill feed peaks in Year 3 and then starts steadily decreasing as higher-grade material from the Kerr underground operation becomes available.

Figure 24.6 Open Pit Production Schedule



WASTE MATERIAL HANDLING

Waste will be hauled to the RSF using 363 t trucks. The construction sequence starts at the bottom of the RSF by dumping the material in groups of three, 15 m lifts, leaving a 30 m berm every 45 m. The resulting overall slope angle of the dump face will be 2.5H:1V.

MINING EQUIPMENT

The KSM open pits are mined using a conventional owner-operated truck and shovel fleet with owner performed blasting. The supply and on-site manufacturing of blasting materials is contracted out. All infrastructure required for the blasting supply contractor is provided by KSM. The mine fleet is comprised of a combination of diesel powered equipment and electric drills and shovels. The fleet has a peak capacity to mine approximately 200 Mt/a operating on 15 m benches.

Equipment requirements are estimated annually. Equipment sizing and numbers are based on the mine plan, measured annual truck cycle times, benchmarking, and a 24 h/d, 7 d/wk work schedule. Peak major equipment numbers are shown in Table 24.3.

Table 24.3 Peak Equipment Fleet

Area/Type	Description	Number
Drilling		
Drill – Electric – 311 mm	Primary Drill	4
Drill – Diesel Hydraulic – 311 mm	Secondary Drill	3
Drill – Diesel Hydraulic – 175 mm	High Wall Drill	4
Blasting		
FEL Blast Hole Stemmer – 111 kW	Blast Hole Stemmer	2
Loading		
Hydraulic Shovel – 40 m^3	Loading Ore & Waste	3
Large Loader – 40 m^3	Loading Ore & Waste	2
Electric Shovel – 56 m^3	Loading Ore & Waste	2
Hydraulic Shovel – 12 m^3	Construction	1
Support:		
Dozer – 433 kW – Support	Shovel Support	6
Wheel Dozer – 372 kW – Support	Pit Clean Up	3
Fuel / Lube Truck – 4,000 gal – Support	Shovel Fueling & Lube	3
FEL – 373 kW – Support	Pit Clean Up	3
Haul Truck – 363 t	Hauling Ore/Waste	39
Haul Truck – 91 t	Construction	5
Water Truck – 20,000 gal – Support	Haul Roads Water Truck	2
Dozer – 634 kW – Support	Dump Maintenance	3
Grader – 397 kW – Support	Road Grading	4

Blasting

Amec Foster Wheeler relied on benchmarking to arrive at a 0.35 kg/t powder factor for both mill-feed and waste. That is, similar large open pit projects in the KSM area use a 0.32 kg/t powder factor for competent rock.

Mining activities are expected to encounter groundwater during production. Groundwater, coupled with precipitation, will require the use of a water resistant emulsion. For estimation purposes, it is assumed that a high-energy emulsion is used for all blasting.

Blasting activities will be performed by KSM utilizing explosives provided by an on-site manufacturer.

Drilling

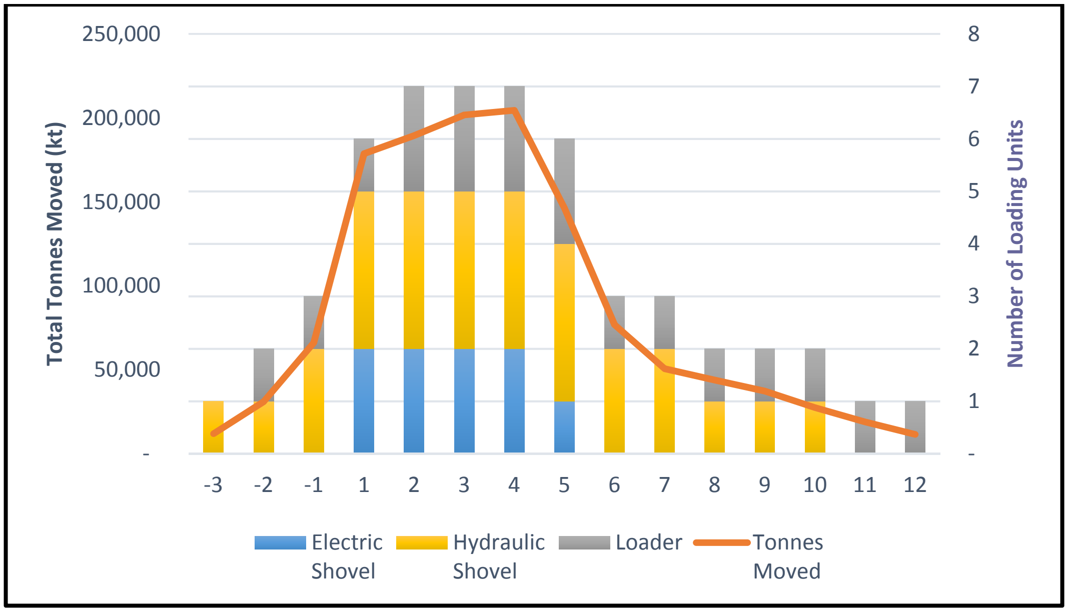
Electric rotary drills with a 311 mm hole size are the primary production drills. The electric drills are supported by three diesel drills also drilling a 311 mm hole size. Although more costly to operate, the diesel drills are selected because of their mobility, which provides additional flexibility to the mine operation. Penetration rates for both the electric and the diesel drills is assumed at 40 m/h instantaneous. Average tonnes drilled for sizing the production drilling fleet was estimated at 30 Mt/a per drill.

The drill fleet also includes four diesel powered percussion drills, drilling 150 mm holes, for buffer and trim drilling, for pioneer road drilling, and for initial bench drilling.

Loading

The primary production loading fleet is comprised of a combination of electric and hydraulic shovels and large loaders. At peak, the loading fleet includes two, 56 m³ electric shovels; three, 40 m³ hydraulic shovels; and two, 40 m³ front-end loaders (FELs). Figure 24.7 shows the primary loading fleet requirements by year to Year 12.

Figure 24.7 Primary Loading Fleet Requirements



For estimating numbers of loading units, the electric shovel, the hydraulic shovel, and the larger loader productivity rates were estimated at 40 Mt/a, 25 Mt/a, and 22 Mt/a, respectively.

Hauling

The haulage fleet is comprised of 363 t trucks. The number of 363 t trucks required are based on measured annual haul profiles. The haul profiles were measured from the pit centroids at each bench to a designated dumping point for each time period. Truck speed by segment was applied to the haul profiles to estimate cycle times, which were then adjusted to estimate total numbers of trucks required according to the operational factors.

Initial truck requirements are five during pre-production, Year -3. Truck requirements increase to 14, and then 19 during pre-production Period -2 and -1, respectively, before jumping to 39, the peak, in Periods 1 to 3. Following Period 3, truck requirements ramp down steadily over the remainder of the mine life. During the pre-production period, the primary production fleet is supplemented with a construction fleet of five, 91 t haul trucks for pioneering and construction work. Figure 24.8 and Figure 24.9 compare the total tonnes hauled to the average cycle times, inclusive of fixed times (load and dump), and the truck requirements, respectively.

Figure 24.8 Annual Total Tonnes Hauled and Average Cycle Time

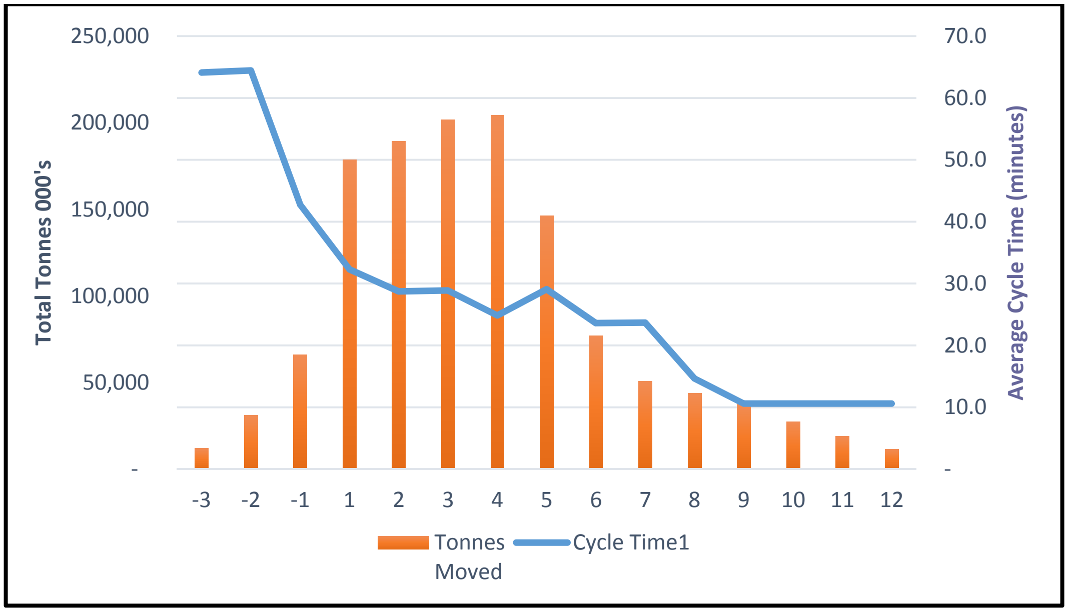
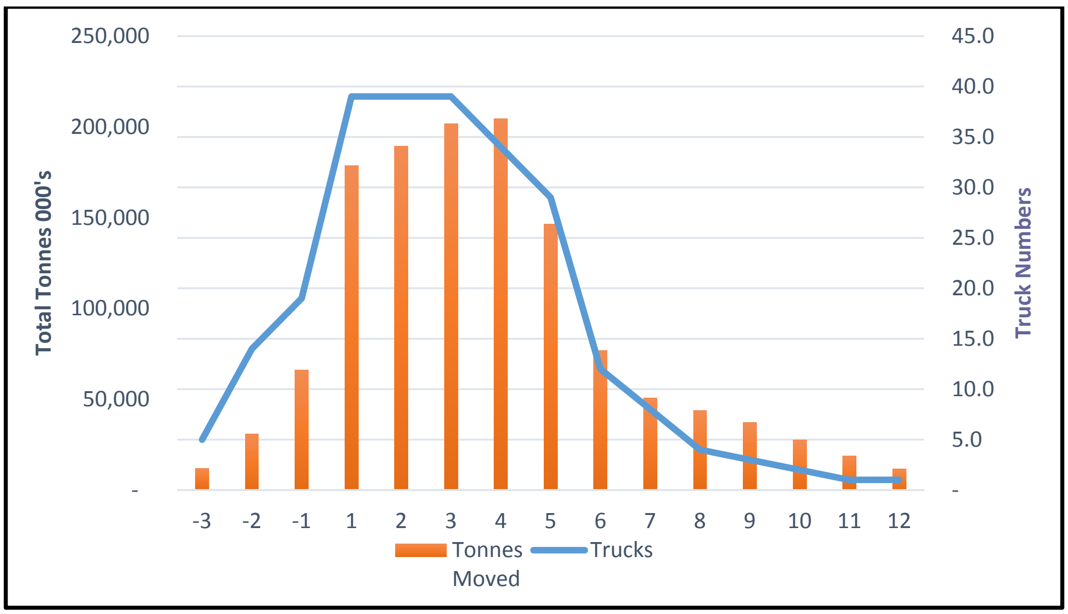


Figure 24.9 Annual Total Tonnes Hauled and Truck Numbers



Support

The peak support equipment numbers are listed in Table 24.4.

Table 24.4 Support Equipment

Support Equipment	Description	Peak Number
Dozer – 433 kW – Support	Shovel Support	6
Wheel Dozer – 372 kW – Support	Pit Clean Up	3
Fuel/Lube Truck – 4,000 gal – Support	Shovel Fueling & Lube	3
FEL – 373 kW – Support	Pit Clean Up	3
Water Truck – 20,000 gal – Support	Haul Roads Water Truck	2
Dozer – 634 kW – Support	Dump Maintenance	3
Grader – 397 kW – Support	Road Grading	4

Auxiliary

To support mine maintenance and mine operation activities, a fleet of auxiliary equipment is required. Based on a standard list of auxiliary equipment (Table 24.5), at peak the auxiliary fleet will include approximately 100 pieces of equipment.

Table 24.5 Auxiliary Equipment

Auxiliary Equipment List
Mine Maintenance
Forklifts
Cranes
Tire Handlers
Jacks
Light Vehicles
Service Trucks
Mine Operations
Lowboy and Trailer
Small Water Truck
Crew Bus
Light Vehicles
Light Plants
Crushing & Screening Plant
Snow Removal Equipment
Small dozer and loader
Mine Engineering
Light Vehicles
Survey Equipment
Mine Dewatering
Light Vehicles
Pumps
Excavators
Backhoe

Mine Dewatering

The open pit will require dewatering and depressurization. Infrastructure to support dewatering and depressurization will include horizontal drains, vertical wells, in-pit sumps, and a collection system. The mine department is directly responsible for operating the in-pit sumps. All other dewatering and depressurization activities are carried out and managed by others. To accomplish dewatering via in-pit sumps, the mine will utilize up to six, 1,400 gpm trailer mounted sump pumps.

Manpower

KSM mine operations will work 7 d/wk, 24 h/d with three crews rotating to fill the mine roster. Salaried staff will work multiple schedules depending on the job. Schedules will include day time only shifts and shifts that rotate with the mine crews.

Manpower requirements peak at approximately 490 employees in Year 4, the peak material movement year.

Primary Consumables

Primary consumables for mine operations include diesel fuel and explosives. The mine fleet's diesel requirements peak at 94.6 million liters in Year 3. Explosive consumption also peaks in Period 3 at 69.2 kt.

COMMENTS ON OPEN PIT MINING METHOD

The PEA mine plan is based on a subset of the Mineral Resource estimates and assumes open pit mining of the Mitchell and Sulphurets deposits will be followed by block caving operations at the Kerr, Mitchell, and Iron Cap deposits.

The open pit production schedule requires a three-year pre-production period for construction and pre-stripping, followed by a LOM of 8 years with stockpile reclaim extending into Year 14.

24.16.2 DEEP KERR MINING METHODS

GEOTECHNICAL CONSIDERATIONS

At the time of the PEA, dedicated geotechnical drilling or studies had not yet been conducted for the Deep Kerr resource area. Current geological exploration drilling indicates similar rock conditions to the Mitchell underground mine, and for the purposes of the PEA it is assumed the rock characteristics would be similar. Thus, the PEA geotechnical assumptions for Deep Kerr were kept in line with those established for Mitchell underground.

For Lift 3, pre-conditioning is assumed in order to manage the higher stresses anticipated at greater depths.

The empirical methods applied to Mitchell underground show that caving of the rock mass can be achieved with a minimum hydraulic radius of 28 m, which is an area of approximately 12,100 m^2 or around 50 drawpoints (this assumes a Mining Rock Mass Rating (MRMR) rating of 51). The ultimate hydraulic radius for each footprint in Deep

Kerr ranges from 66 to 142 m, is significantly greater than the minimum required hydraulic radius of 28 m, and supports both initial and sustained cave mining at Deep Kerr.

GEOVIA's PCBC™ Footprint Finder software was used to identify the elevation and shape of the highest value footprints. A maximum column height of 500 m was used to identify and locate these footprints, which is a common column height in modern caving mines. Freeport McMoran, Rio Tinto, Newcrest, and Codelco have experience in operating or planning for similar maximum column height. An exception was made for Lift 1, in which the footprint elevation and shape was fixed using a 500 m column with the height of draw extended to 750 m. The risks with this extension of column heights for Lift 1 to 750 m is considered acceptable since the materials in between 500 m and 750 m of column only account for less than 16% and 8% of the total tonnes for Lift 1 and overall Deep Kerr, respectively. In modern caving mines it is common to see this additional percentage as an overdraw in drawpoints. The risk of a production gap between Lift 1 and 2 is insignificant. In the event Lift 1 is unable to deliver the planned overdraw, Lift 2 will already be in production and can typically adjust its plan to compensate for shortfalls in Lift 1. Furthermore, mines such as the Cadia East Mine in Australia, which is currently in production based on columns up to 1,000 m, have developed technical and operational methods for accommodating these higher column heights.

A cave subsidence angle of 60° is assumed for Deep Kerr and is considered conservative based on comparison data from benchmarking studies from similar block cave mines in porphyry copper systems. No surface infrastructure has been planned within or directly below the caved zone or subsidence zones. LOM underground infrastructure (i.e., common conveyors, common vent drifts, common ramps, common vent shafts, etc.) are also not planned within or near the caved zone or subsidence zones.

CAVE MODELLING AND PRODUCTION SCENARIOS

Seabridge provided the Mineral Resource block model, which contains tonnages, grades and NSR values. NSR is defined in Section 16.1.4. Amec Foster Wheeler used this model to define the mineable shape for the block cave mining method using GEOVIA's PCBC™ software. This modelling software was used to identify the elevation and shape of the highest value footprints utilizing inputs based on the characteristics of the deposit, current industry standards, and also collaboration with other consultants responsible for the PEA-level engineering of two other block cave mines within the KSM Property. These inputs included maximum column heights, drawpoint spacing, draw rates, etc. Selection of the optimum footprint elevations for the various lifts was based on total discounted value of the materials. Selection of the optimum footprint elevations for the various lifts was based on total discounted value of the materials. Starting with the in situ block model, a diluted block model is calculated using a 50% Laubscher's dilution entry point. All calculations to determine the height of draw of block caving are undertaken using the diluted model.

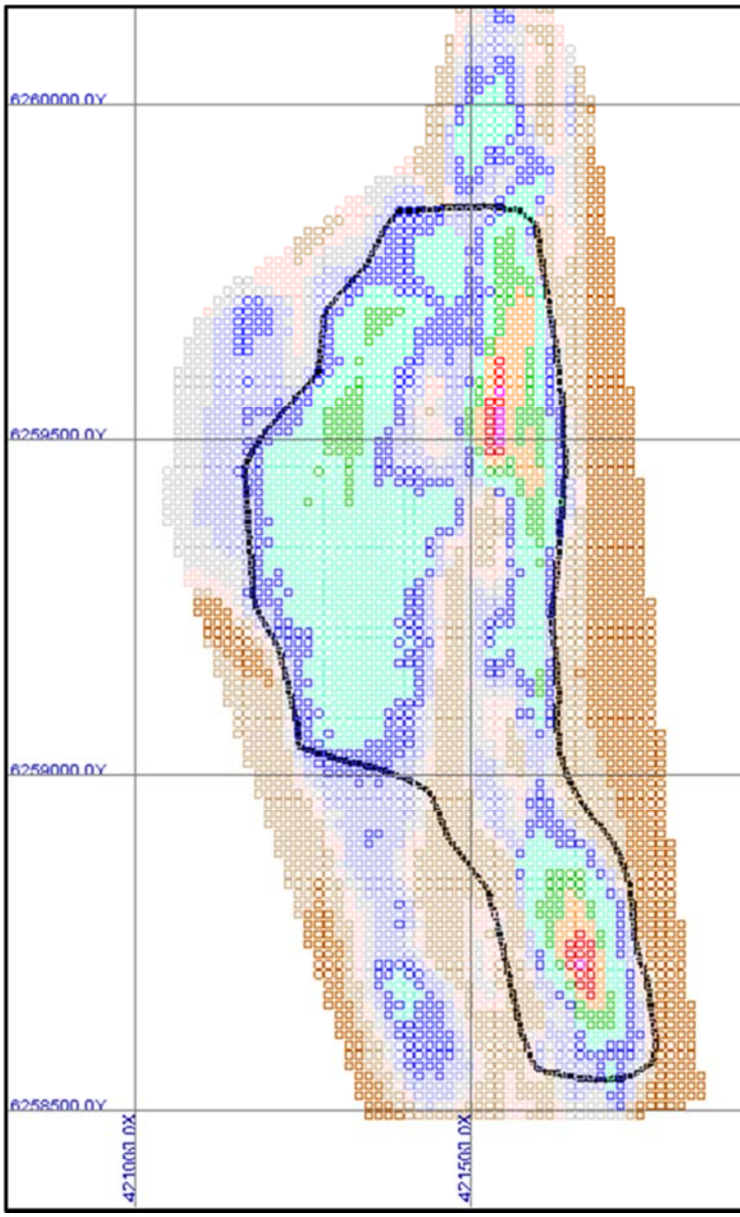
The ideal placement of Lifts 1 and 2 was determined to be at 625 m and 130 m, respectively, based on a column height of 500 m, thus requiring the levels remain 500 m apart. Shifting the 130 m level one step lower in elevation decreases mineralized material value by almost Cdn$1 billion, while the 625 m level progressively decreases in value as it is raised. Therefore, the placement of Lifts 1 and 2 is the optimum location that best balances the overall value of the material mined, and earlier timing of the higher-value material. Lift 3 has relatively less impact on the economics of the Project as this lift occurs much later in production. Therefore, the lift elevation selection was initially focused more on Lifts 1 and 2.

Instead of cut-off-grade, various production column shut-off values based on NSR were used to create various production scenarios that were compared to determine and optimize footprint shape. A series of GEOVIA PCBC™ runs were performed at varying NSR values from Cdn$20/t to Cdn$24/t, for various production rates from 60 to 80 kt/d. Higher shut-offs were tested but it was observed that at Cdn$26/t, total resources quickly diminished and footprint shapes became highly irregular. The results from these runs were used to generate hill-of-value tables which emphasized NPV and IRR. The shape of the deposit narrows with depth. With scenarios at an 80 kt/d rate and NSR shut-off greater than Cdn$20/tonne, it was observed that the lower elevation footprints could not sustain this rate either for a meaningful amount of time, or at all.

After Lifts 1 and 2 elevations were determined, the location of Lift 3 was evaluated in more detail. Ultimately, the optimum scenario that best balanced both NPV and IRR is identified to have three lifts at 625 m, 130 m, and -290 m elevation, with a drawpoint shut-off NSR value of Cdn$22/t, and a peak production rate of 70 kt/d.

Furthermore, this scenario is deemed to integrate well with the production from the adjacent KSM mines (i.e., Mitchell open pit, Mitchell underground, Iron Cap, Sulphurets, etc.) to support the various production scenarios considered for the PEA.

Figure 24.10 GEOVIA PCBC™ Footprint for Lift 1 at 625 m Elevation



Net Column Values (CAD$)		
>=	<	Colour
Low Bound	-2.0M	
-2.0M	-1.0M	
-1.0M	-0.5M	
-0.5M	0.0M	
0.0M	1.0M	
1.0M	2.5M	
2.5M	5.0M	
5.0M	7.5M	
7.5M	10.0M	
10.0M	12.5M	
12.5M	Hi Bound	

Figure 24.11 GEOVIA PCBC™ Footprint for Lift 2 at 130 m Elevation

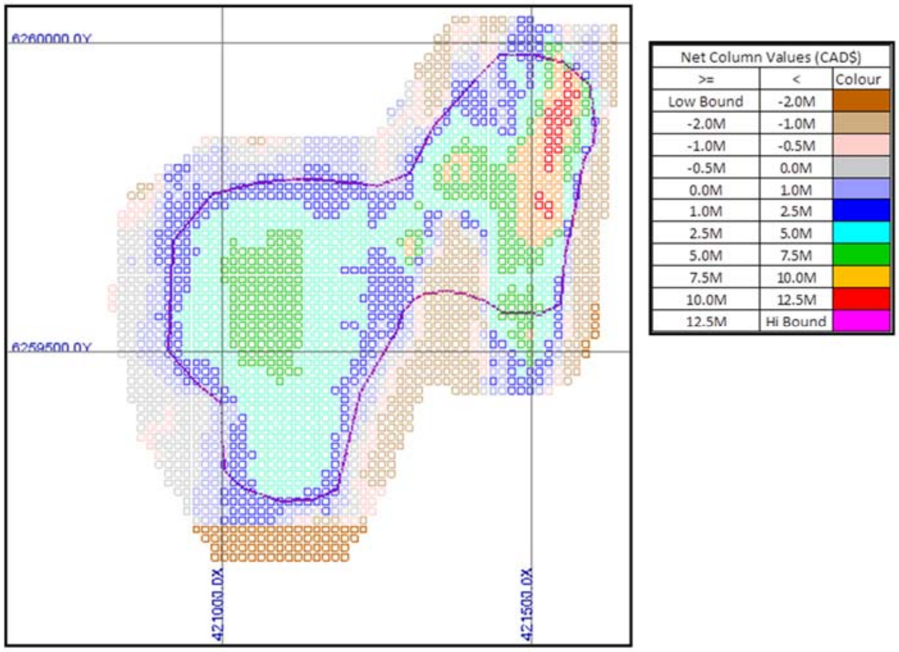
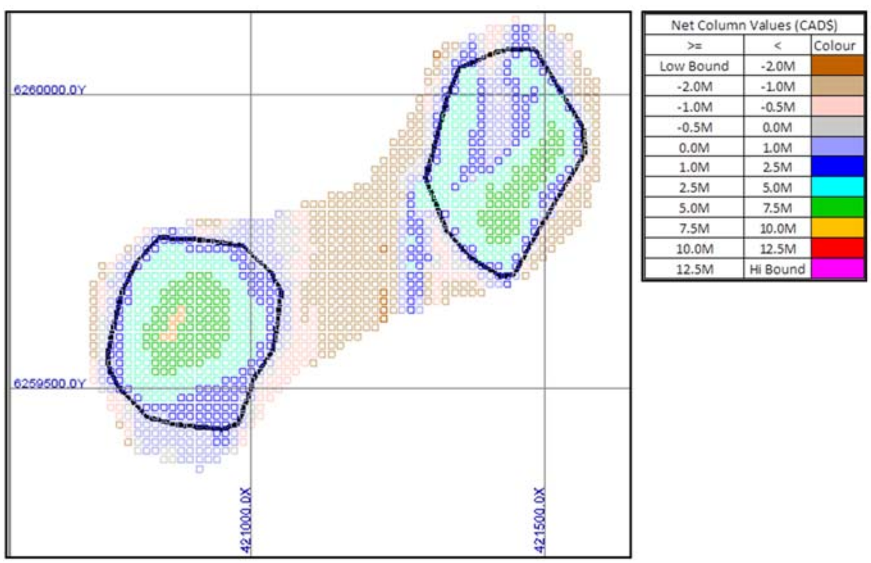


Figure 24.12 GEOVIA PCBC™ Footprint for Lift 3 at -290 m Elevation



DESIGN CRITERIA AND LAYOUT

The block cave mine design is based on a series of three lifts totalling 817,000 m². All lifts are accessed through a series of shared portals and vent adits along the northwestern base of Kerr Mountain, facing Sulphurets Creek. All lifts connect to a 7 km common underground incline conveyor system, which passes under the Sulphurets Creek and connects directly up to the MTT material handling system that supports all mines at the Project site. Mining on each lift assumes block caving with a post undercutting method, with LHD equipment for extraction, and a material handling system of mill feed passes, chutes, truck haulage, dual crushers, and conveyors.

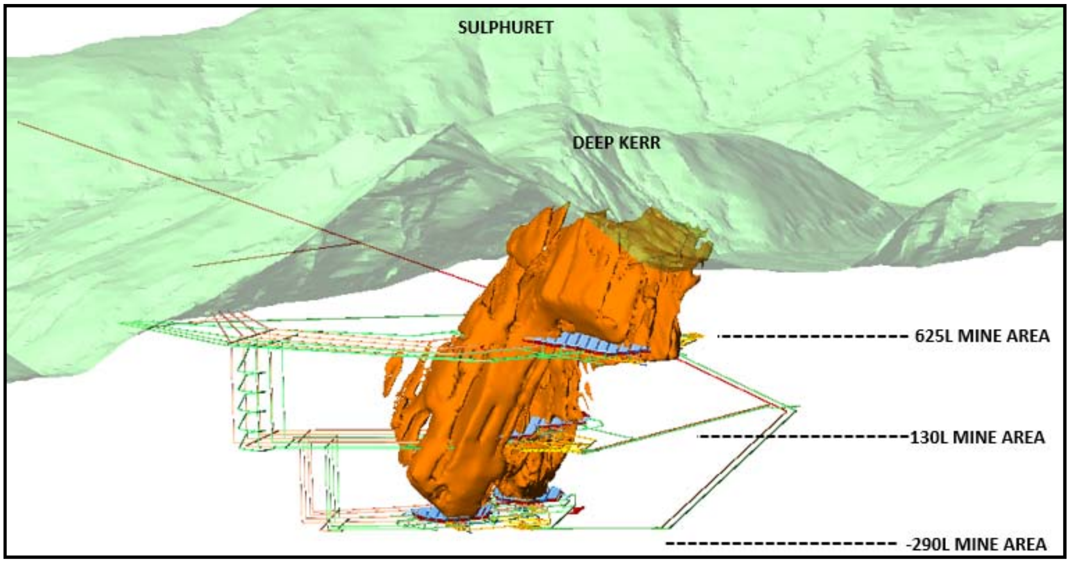
Lift 1 is the largest in tonnage and footprint and has an area of 422,000 m^2. The footprint shape is an irregular oval shape spanning 1,200 m in length, with varying widths up to 500 m, tapering at both ends. The extraction level elevation is 625 m, which is slightly higher than the portal elevation of 600 m and thus does not require major ramps or shafts for access or ventilation. Mine access is primarily through two of the intake portals, one of which directly connects to the footprint area as a main ventilation intake drive, and the second which eventually parallels the underground conveyor as a service drive towards the crusher area.

Lift 2 is the second largest in tonnage and footprint and has an area of 250,000 m^2. The footprint shape is an irregular shape spanning 780 m in length, with varying widths of 200 m up to 700 m, tapering at both ends. The extraction level elevation is 130 m, requires three decline ramps for access and mill feed conveyance, and seven bored ventilation shafts. The ramps and vent shafts are tied into the upper system constructed for Lift 1 in order to minimize cost and effort.

Lift 3 is the smallest in tonnage and footprint and consists of two separate slightly oval footprints with a total area of 145,000 m^2. The southwestern footprint has an area of 75,000 m^2 and the northeastern footprint has an area of 70,000 m^2. Both footprints span a length of 350 m, with varying widths up to 350 m. The extraction level elevation is -290 m, and similar to Lift 2, requires two decline ramps for access and mill feed conveyance, and eight bored ventilation shafts. The ramps and vent shafts are tied into the upper system constructed for Lift 2 in order to minimize cost and effort. Pre-conditioning is assumed for Lift 3 to manage the higher stresses anticipated at greater depths, and will be performed from the undercut (UC) level before UC longhole drilling commences on that level.

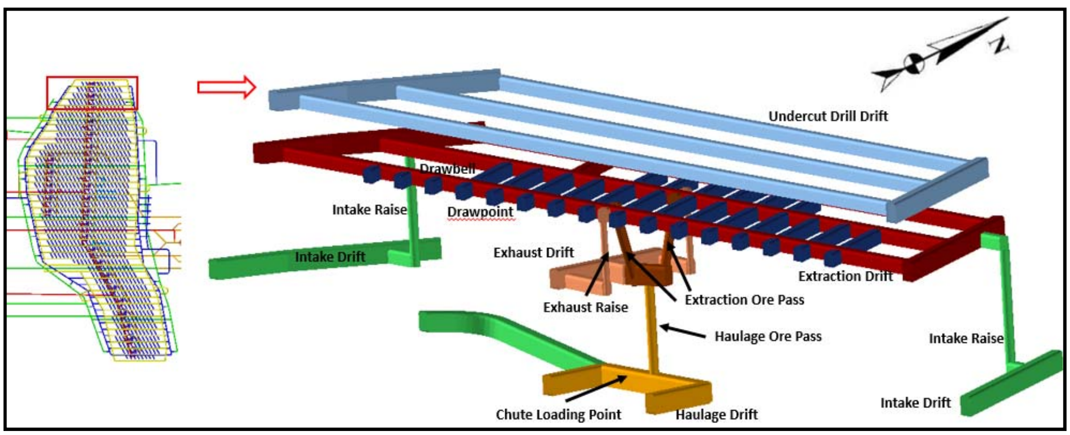
All preparation work at Deep Kerr will occur below the rock mass to be extracted for each respective lift.

Figure 24.13 Deep Kerr Mine General Layout with Topo



Note: Spacing between Lifts 1 and 2 is approximately 495 m. Spacing between Lifts 2 and 3 is approximately 420 m. The distance from the portals to the footprint of Lift 1 is approximately 2 km.

Figure 24.14 Typical Level Arrangement for Deep Kerr



Note: Light blue parallel undercut drifts and red parallel extraction panel drifts are spaced 30 m apart. Dark blue drawpoints are spaced 18 m apart.

VENTILATION

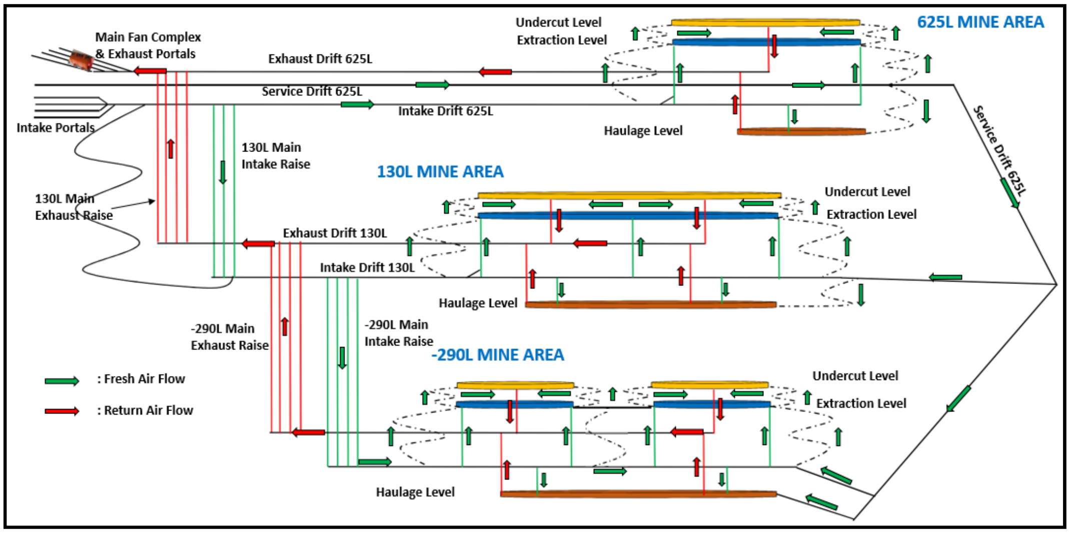
The magnitude of the required airflow is determined using available benchmark data. These data show that a modern, mechanized block cave mine requires approximately 0.024 m^3/s per tonne of mill feed per day of designed production. For the Deep Kerr concept, a rate of 0.024 m^3/s per tonne of mill feed per day is selected as being suitable for this, PEA stage. Thus, it is anticipated at the required airflow for this deposit, using a block cave mining method, will be on the order of 1,680 m^3/s. Given the Project stage and the nature of the deposit, no correction to elevation has been made.

Given the location of the Deep Kerr deposit, intake air heating is considered during the months with average surface temperatures below 3°C. Over the average heating season

(October through April/May), it is estimated that the equivalent of 6.1 to 7.9 million liters of propane (liquefied petroleum gas [LPG]) will be required.

Other elements of the mine ventilation system design are factored relative to the anticipated number of production panels and panel layouts to arrive at estimates for the number of intake and exhaust regulators, bulkhead, air doors, air locks, auxiliary fans, etc. A ventilation schematic is shown in Figure 24.15.

Figure 24.15 Ventilation Schematic



MINE DEWATERING

All drainage water for each lift is intended to gravity drain via a system of sumps and boreholes to a dewatering gallery level located at the lowest point for each lift. The only exception is the tail end of the conveyor system, which is designed with a designated sump and 30HP pump station which discharges horizontally to the dewatering gallery level.

The dewatering gallery contains two settling drifts each with a length of 100 m, with removable/replaceable timber weir walls every 20 m, intended to settle out fine sediment before being pumped by the main dewatering pumps. All water is assumed to be pumped to the portal, where a surface pump station transfers the water to the main Project water treatment dam and facility near the Mitchell valley area.

Although detailed hydrogeological drilling or studies have not yet been performed for Deep Kerr, KSM's guidance based on dewatering simulations for Mitchell UG indicate that a system sized to handle up to 65,000 gpm, i.e., 4 m^3 /s, would be required to accommodate a potential 1:200 year precipitation event. In addition to the 1:200 year event, there are more frequent seasonal events which are anticipated to exceed 1 m^3/s, i.e., 15,000 – 20, 000 gpm in the period from May until July.

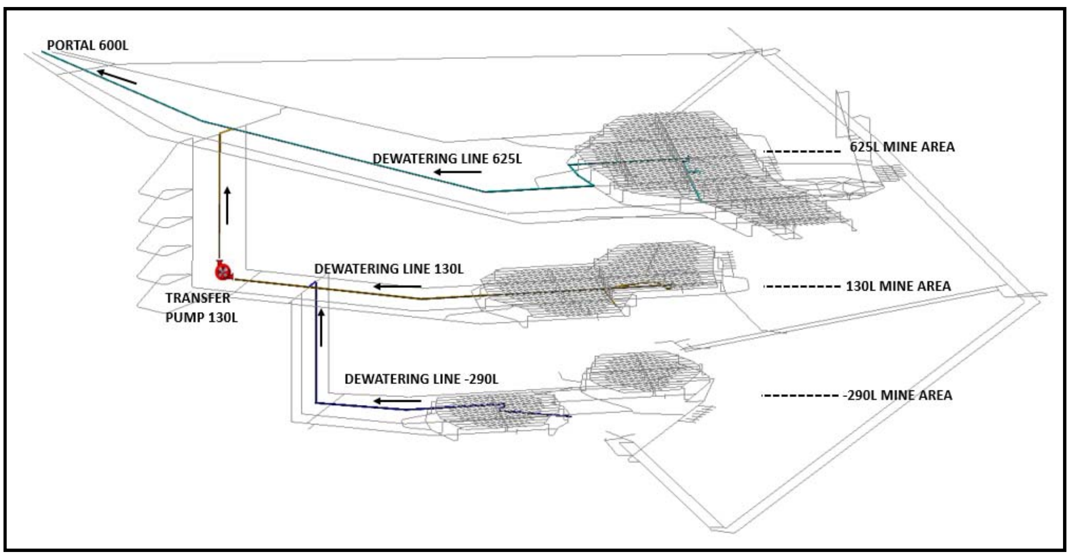
In addition to pumping, approximately 500,000 m^3 of additional water storage capacity to buffer the 1:200 year and other seasonal event is included in Lifts 2 and 3 on the dewatering level, via 10 km of 7.0 mW x 7.0 mH of additional drifting. This storage capacity is not required for Lift 1 since water in excess of the installed pumping capacity can be diverted to the intake adits which gravity drain out to the portals, and then pumped to the WTP.

For Lift 1 the installed power requirement for the main pump station in the dewatering gallery level is approximately 500 HP. This capacity is intended to accommodate approximately 2,000 gpm of mining process water drainage and minor groundwater inflow dewatering. During the larger seasonal dewatering events, provisions will be in place on the upper footprint levels to intercept and route a majority of water away from the footprint and down towards the portals in the ventilation adits via gravity.

For Lift 2 the installed power requirement for the main pump station in the dewatering gallery level is approximately 45,000 HP, which corresponds to a capacity of 4 m^3/s, or 65,000 gpm. The main pump station will pump from the dewatering level and discharge directly to the portals.

For Lift 3 the installed power requirement for the main pump station in the dewatering gallery level is also approximately 45,000 HP, which also corresponds to a capacity of 4 m^3/s, or 65,000 gpm. For Lift 3, an additional transfer pump station, equivalent in power and capacity to the main pump station, will be required at the Lift 2 elevation to transfer the water up to Lift 1 and out to the portals.

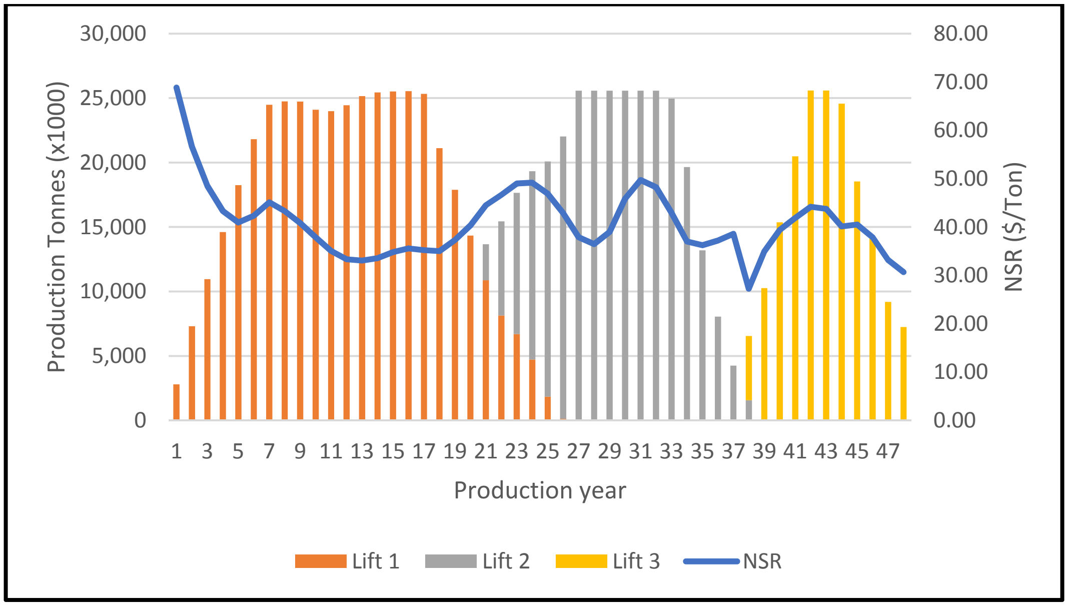
Figure 24.16 Mine Dewatering General Arrangement



Production Schedule

The proposed production schedule for Deep Kerr is shown in Figure 24.17. Note that the years shown apply to the underground operation only, not to the combined open pit and underground multi-mine plan.

Figure 24.17 Deep Kerr Mine Production Plan



Only classified Mineral Resources are included in the production plan. All the unclassified materials are treated as waste by setting the grades to zero. Table 24.6 shows the distribution of diluted mineral resources in the PEA mine plan based on their classifications.

Table 24.6 Deep Kerr Diluted Mineral Resources in PEA Mine Plan

	Measured	Indicated	Inferred
Tonnes (millions)*	-	24.4	931.5
Au (g/t)	-	0.26	0.31
Cu (%)	-	0.54	0.49
Ag (g/t)	-	1.1	1.7

Note: *Includes 90.4 Mt of mineralized dilution (the portion of Indicated and Inferred material that is below the Mineral Resource statement cut-off, \$0 < NSR < \$16), and 27.8 Mt of non-mineralized dilution (unclassified material set to zero grade).

BLASTING AND EXPLOSIVES

The PEA has assumed that there will be separate explosive magazines for each lift.

Explosives will be used underground for five main purposes:

- preparation development of mineralized areas
- draw bell construction
- undercutting
- development in barren material
- secondary blasting of boulders and hang ups.

The first four activities are ongoing operations up to until about four years prior to the end of underground production. The last activity will be an ongoing operation until production ends.

MINING EQUIPMENT

Primary and support equipment requirements for the proposed Deep Kerr panel cave operation are summarized in Table 24.7.

Table 24.7 Mobile Equipment Requirements, Deep Kerr

Equipment	No. of Units
Sandvik DD420-40C Jumbo	5.0
Sandvik LH514 LHD (7 m^3)	20.0
Sandvik LH517 LHD (8 m^3)	4.0
Sandvik TH550 55T Truck - Waste	12.0
Sandvik TH550 55T Truck - Production	22.0
Sandvik DS310 Rock Bolter	17.0
Atlas Copco Cabletec LC	5.0
Normet Utimec 1600 Transmixer	5.0
Normet Spraymec 6050W Shotcrete Sprayer	2.0
Normet Charmec 1605 ANFO Loader	4.0
Maclean BH2 Blockholer	6.0
Atlas Copco M6C Production Drill	7.0
Cat TH407 Telehandler	4.0
Kubota R520S Tractor/Backhoe	4.0
Normet Cassette Fuel/Lube Deck	4.0
Normet MF 540 Scissorlift Manlift	4.0
Toyota Personnel Transporter	13.0
Cat 140M Grader	4.0
Cat CS56 Vibratory Packer	1.0
Concrete Pump with Trailer - Meyco Altera	1.0
Marcotte M40 Boomtruck	1.0
Compressor Sullair 185	6.0
Mobile Crane - Grove RT540E (35T)	2.0
Ingersoll Rand 1550SE Electricians Service Vehicle	2.0
Ingersoll Rand 1550SE Mechanics Service Truck	2.0
Normet MF028 Sludge Truck	1.0
Normet Utimec MF500 Fire Truck	1.0
Normet MF350 Water Truck	1.0
Toyota Landcruiser Mine Ambulance	1.0
Toyota Landcruiser Mine Rescue Truck	1.0
Atlas Copco UV2 - Asphalt Sprayer	1.0

Manpower Considerations

The estimated workforce for Deep Kerr is approximately 2 to 250 persons per shift at the peak of preproduction construction period and also at peak production, based on the man-hour build up for the mine plan and cost estimate.

Comments on Deep Kerr Underground Mining

The PEA mine plan is based on a subset of the Mineral Resource estimates and assumes panel caving operations at the Deep Kerr deposit will be integrated with the production from the adjacent KSM mines (i.e., Mitchell open pit, Mitchell underground, Iron Cap, Sulphurets, etc.) to support the various production scenarios considered for the PEA.

The Deep Kerr panel caving mine will operate for 48 years, including 3 years of initial pre-production construction and development, and a 6-year ramp-up period to a full production rate of 70 kt/d.

24.16.3 Iron Cap Mining Methods

Geotechnical Considerations

For the 2016 PFS, the characterization of the Iron Cap rock mass focused on the rock in and around the extraction and undercut levels of the proposed block cave mine (1,035 m elevation) and on the mineralized rock above this that will be caved. Rock within 50 m of the ground surface is expected to be of poorer quality due to weathering. This rock will not have a significant impact on the caving response of the mineralized rock, and geotechnical information from this rock has not been included in the characterization of the rock mass that will be block caved.

Characterization of the rock was based on core photographs and data collected for exploration drill holes, detailed geotechnical data collected for drilling programs carried out by BGC in 2010 (BGC 2011), and an interpreted geological model provided to Golder by Seabridge.

There are three geotechnical holes in the Iron Cap deposit. No additional analysis of new exploration drill holes was completed for this PEA as the 2016 assessments are still considered valid for the new footprint that is now 175 m deeper.

The caveability assessments made using Laubscher's and Mathews' methods for the 2016 PFS indicated that the size of the footprint required to initiate and propagate caving is between approximately 100 m and 220 m. This minimum 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 generally large-size, continuous nature of the deposit, indicates that the Iron Cap deposit is amenable to cave mining.

The cave mining will draw down the mineralized rock, and a significant depression will develop on surface above the production footprint in the form of a crater. The crater typically develops on surface above and slightly laterally beyond the footprint of the production horizon of the cave mining. The top section of the crater is a relatively steep escarpment (60 to 70°) that is marginally stable but comprised of nominally in place dilated rock. Beneath this is failed broken rock that has progressively sloughed from the rim of the crater. This rock rills down to the bottom of the crater at about 40 degrees.

Mining Method Selection

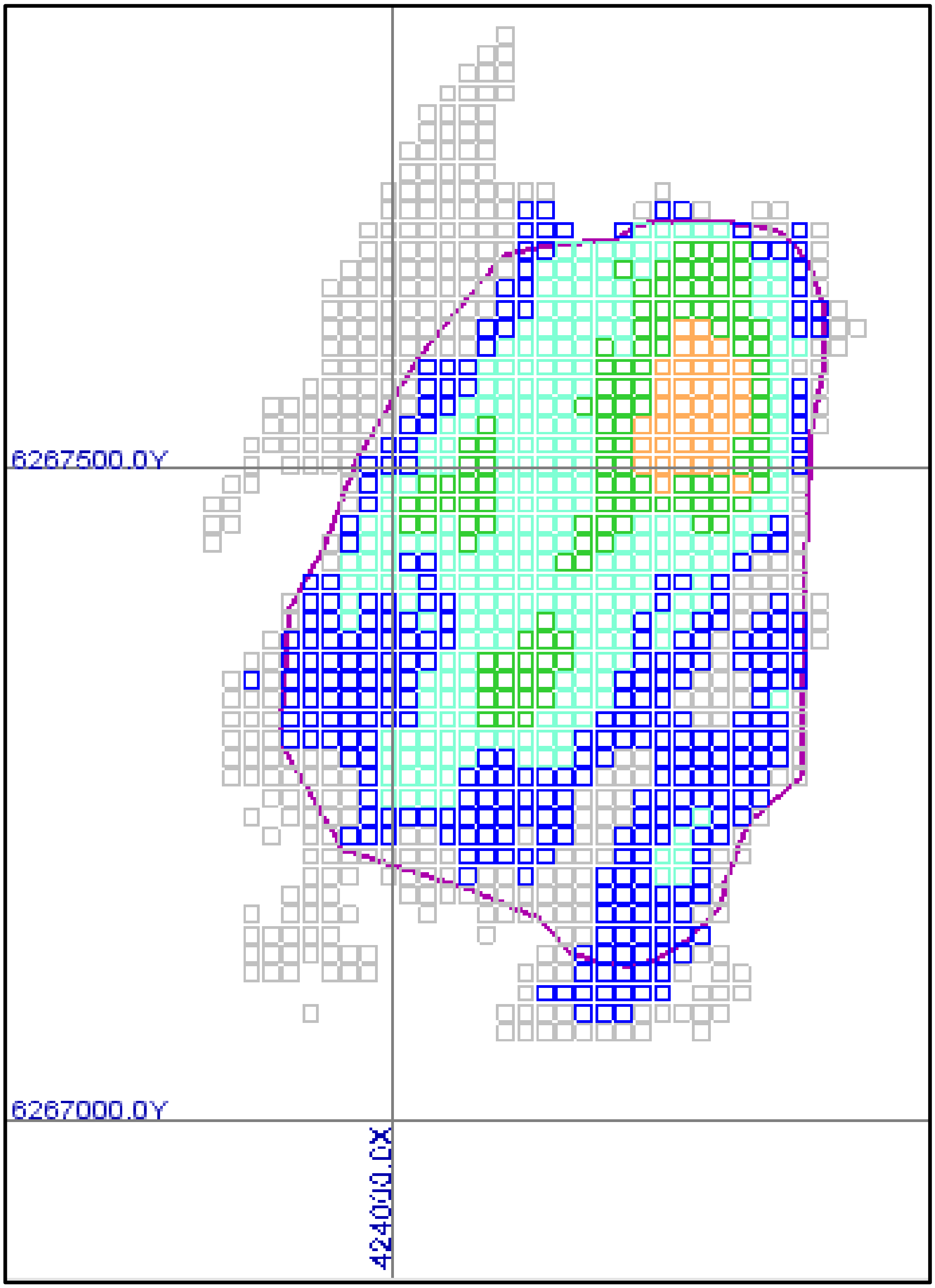
The most appropriate mining methods for this deposit are block caving, sub-level caving and sub-level stoping. Based on mining costs, deposit grade, geometry and depth, block caving was selected as the preferred mining method.

Cave Modelling and Production Scenarios

Seabridge provided the block model with updated NSR block values. Instead of cut-off-grade, various shut-off values based on NSR were used to determine and optimize footprint shape. A series of GEOVIA PCBC™ Footprint Finder runs were performed at varying NSR values from Cdn$20/t to Cdn$24/t, for a production rate of 40 kt/d. The results from these runs were used to generate hill-of-value tables which emphasized NPV and IRR. The footprint used for mine design was based on a NSR shut-off of Cdn$20/t for the footprint shape, and a NSR shut-off of Cdn$23/t for column height. A second lower cave lift was assessed but this was not of sufficient size to warrant further work.

A footprint at elevation at 900 m has the most value and the block cave design was based on this footprint. Figure 24.18 shows the GEOVIA PCBC™ footprint at the 900 m elevation.

Figure 24.18 GEOVIA PCBC™ Footprint for Lift at 900 m Elevation



Note: Figure prepared by Golder Associates Ltd., 2016. Grid system is based on UTM coordinates, with North being at the top of the image. Warmer colors indicate higher caving column NSR values, whereas cooler colors indicate lower caving column NSR values.

Dilution

Dilution for the block caving was derived using Laubscher´s dilution matrix, which is a common methodology employed by operating block caving mines. Starting with the in situ block model, a diluted block model is calculated using a 60% dilution entry point and 195 height of interaction zone. All calculations to determine the height of draw of block caving are undertaken using the diluted model.

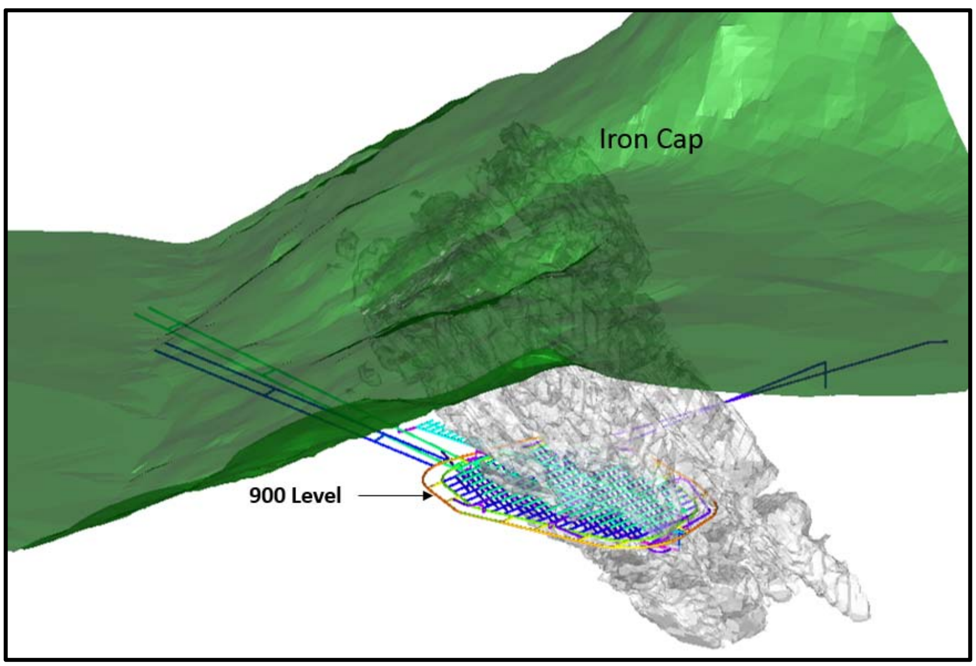
DESIGN CRITERIA AND LAYOUT

The block cave design for the Iron Cap footprint is at the 900 m elevation. The shape of footprint is irregular oval, with an area of 177,000 m^2, and a perimeter length of 1.6 km. The widest section of the footprint is 465 m.

Personnel, material, and supplies will access the underground through a drive off the Mitchell access ramp (this assumes that Mitchell will be mined first). 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.

The Iron Cap design has multiple drifts that can act as an emergency egress. The primary emergency egress will be the train tunnel or access tunnel, whichever is accessible. If both tunnels are inaccessible, it will be possible to exit the mine through one of the fresh air drifts. Figure 24.19 shows an isometric view of the Iron Cap mine layout looking towards the southwest.

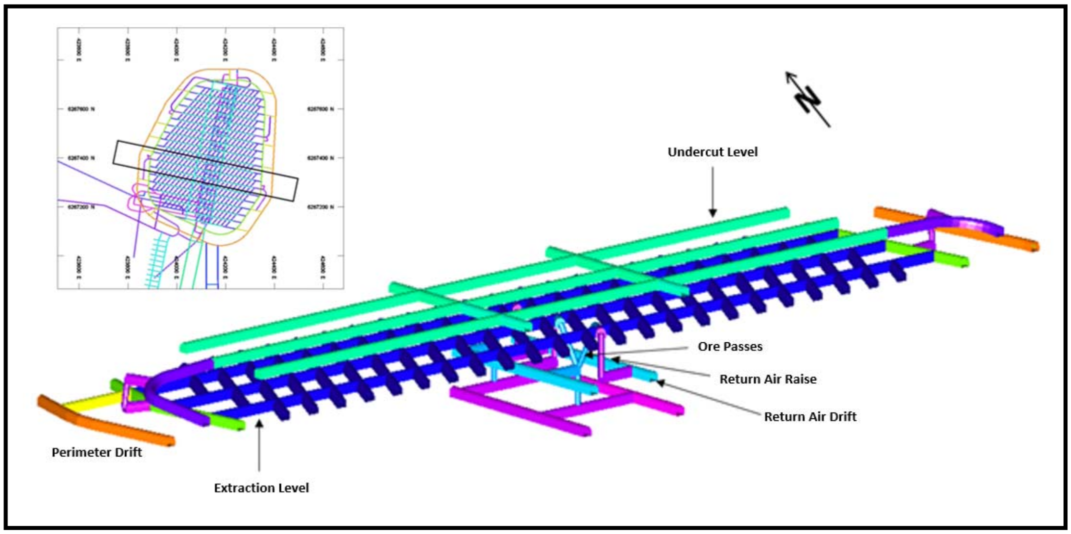
Figure 24.19 Iron Cap Mine General Layout with Topo and NSR above Cdn$20/t NSR Shell



Note: Figure prepared by Golder Associates Ltd., 2016.

Figure 24.20 shows the typical level arrangement for the Iron Cap mine.

Figure 24.20 Typical Level Arrangement for Iron Cap



Note: Figure prepared by Golder Associate Ltd., 2016.

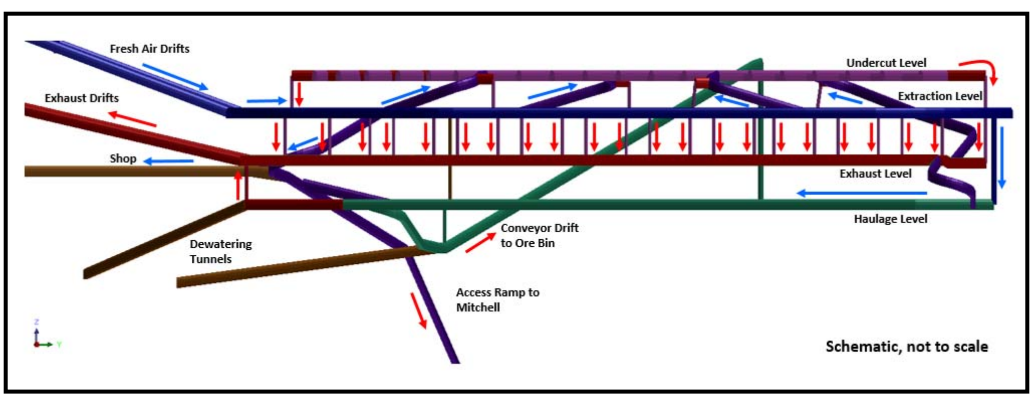
VENTILATION

The total airflow requirements were based upon air quantities of 0.063 m^3/s per kilowatt of diesel equipment, equipment utilization, and engine utilization. The total airflow requirement is estimated at 1,000 m^3/s, which is sufficient to appropriately dilute all noxious gases and particulate matter produced by the mining equipment and activities on each mining level. This equates to a rate of 0.025 $m^3/s/t/d$.

Heating of mine air in the winter months is included in the design and cost estimates and will be done by mine heaters located at each of the two main fan installations. The mine air heaters will heat 1,000 m^3/s to 3°C for five months per year (November through March). The propane requirement is 2.4 million liters per year.

Figure 24.21 is a schematic of the ventilation system showing major airflow directions.

Figure 24.21 Ventilation Schematic for Iron Cap Mine (Section Looking West)



Note: Figure prepared by Golder Associate Ltd., 2016.

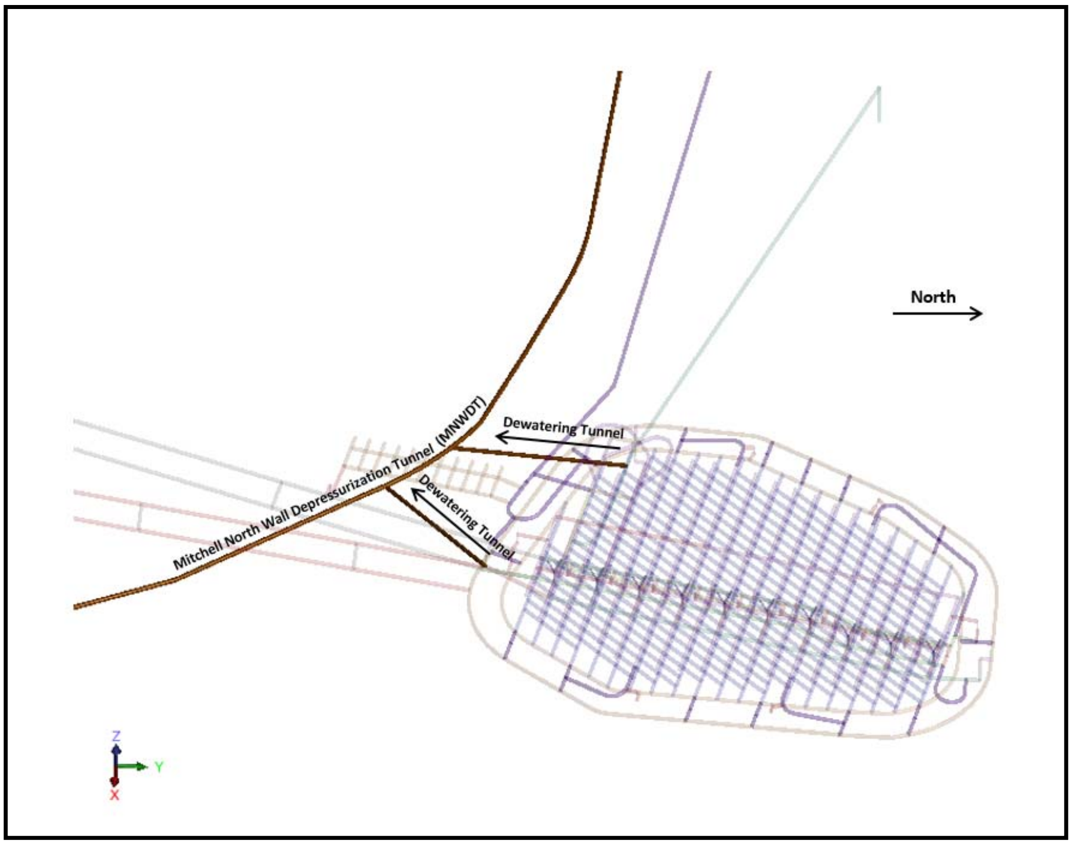
MINE DEWATERING

The mine water handling system is designed to handle the water that originates from the groundwater and surface inflows (including possible water from ice that caves into the crater when the ice field is undercut), and water that is introduced to the mine for operations.

The inflows to Iron Cap are conveyed by gravity away from the production levels to drifts that connect to the Mitchell North Slope Depressurization Tunnel located at the south end of the footprint. To provide for good drainage, the underground drifts have been graded so that water will run towards one of the two dewatering tunnels. Any flood water will also flow down these drifts where it will be collected in the existing “dirty” water infrastructure. Pumps are not required to dewater Iron Cap. The underground water management system at Iron Cap can handle 4 m^3/s (63,000 gal/min). This caters for both the groundwater and peak surface water inflows.

Figure 24.22 shows a schematic of the proposed dewatering strategy for the Iron Cap mine.

Figure 24.22 Mine Dewatering Arrangement for Iron Cap (Plan View)

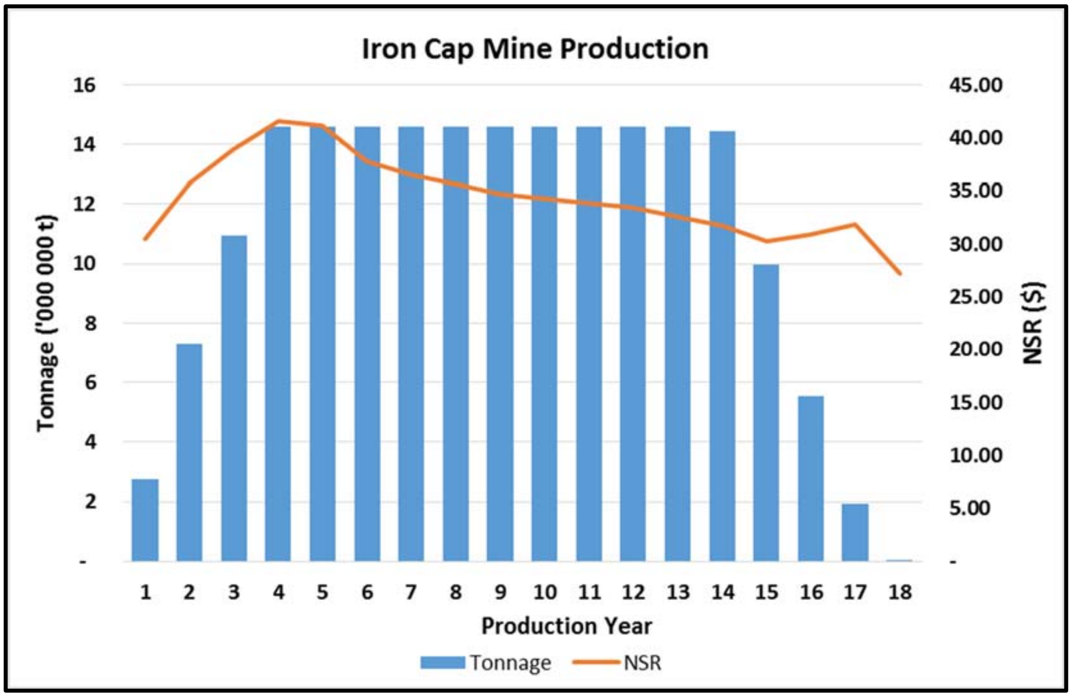


Note: Figure prepared by Golder Associate Ltd., 2016.

PRODUCTION SCHEDULE

The proposed production schedule for Iron Cap is shown in Figure 24.23. Note that the years shown apply to the Iron Cap underground operation only, not to the combined open pit and underground multi-mine plan of the Project.

Figure 24.23 Iron Cap Mine Production Plan



Note: Figure prepared by Golder Associates Ltd., 2016.

Only classified Mineral Resources are evaluated as having grades and NSR value in the production plan. All the unclassified materials are treated as waste by setting the grades to zero. Table 24.8 shows the distribution of diluted Mineral Resources in the PEA mine plan based on their classifications.

Table 24.8 Iron Cap Diluted Mineral Resources in PEA Mine Plan

	Measured	Indicated	Inferred*
Tonnes (millions)*	-	121.5	77.4
NSR (Cdn$/t)	-	38.08	31.17
Au (g/t)	-	0.64	0.46
Cu (%)	-	0.24	0.23
Ag (g/t)	-	4.1	3.5

Note: *Includes 15.7 Mt of mineralized dilution (the portion of Indicated and Inferred material that is below the Mineral Resource statement cut-off, $0 < NSR < $16) and 1.7 Mt of non-mineralized dilution (unclassified material set to zero grade).

Blasting and Explosives

The PEA has assumed that explosives will be stored in the Mitchell Valley to service both the Mitchell and Iron Cap mines. The explosives will be delivered by truck to the underground mine as required. Small underground explosive magazines are included for development mining and secondary blasting activities.

Explosives will be used underground for five main purposes:

- preparation development of mineralized areas
- draw bell construction
- undercutting
- development in barren material
- secondary blasting of boulders and hang ups.

Mining Equipment

Primary and support equipment requirements for the proposed Iron Cap caving operation are summarized in Table 24.9.

Table 24.9 Mobile Equipment Requirements for Iron Cap

Equipment	No. of Units
Development	
Two-boom Jumbo	3
LHD (8 m^3)	2
Haul Truck (55 t)	4
Rock Bolter	5
Shotcrete Sprayer (35,700 lb)	1
ANFO Charger	2
Transmixer (40,000 lb)	1
Raisebore	1
Scissor Lift and Boom Truck	4
Production	
Longhole Drill (ITH, 89-216 mm)	3
Emulsion Loader	1
LHD (7 m^3)	11
Haul Truck (55 t)	13
Block Holer	2
Scissor Lift and Boom Truck	2
Mobile Rockbreaker	4
Support	
Grader	2
Big Personnel Carrier	4

table continues…

Equipment	No. of Units
Small Personnel Carrier	6
Lube Truck	3
Contractor	
Two-boom Jumbo	1
Haul Truck (55 t)	1
LHD (8 m³)	1
Rock Bolter	1
ANFO Charger	1
Scissor Lift and Boom Truck	1

MANPOWER CONSIDERATIONS

The total estimated workforce for Iron Cap underground operation during peak construction is 263, while the total workforce required during the peak production is approximately 400 persons. The estimate is based on the development and production tonnage and equipment requirements.

COMMENTS ON IRON CAP MINING METHODS

The basis for the Iron Cap underground mine development is to support the mill feed requirement at the Project site from multiple mines. The development stage of the Iron Cap mine is approximately five years, which includes mine access, initial footprint development and construction of major mine infrastructure, such as underground crusher, material handling system, shops, dewatering system, primary ventilation fans, etc. Following the five year development stage, the initial ramp up period is three years to reach full production of 14.6 Mt/a (40 kt/d), with a total production operating life of 18 years.

24.16.4 MITCHELL MINING METHODS

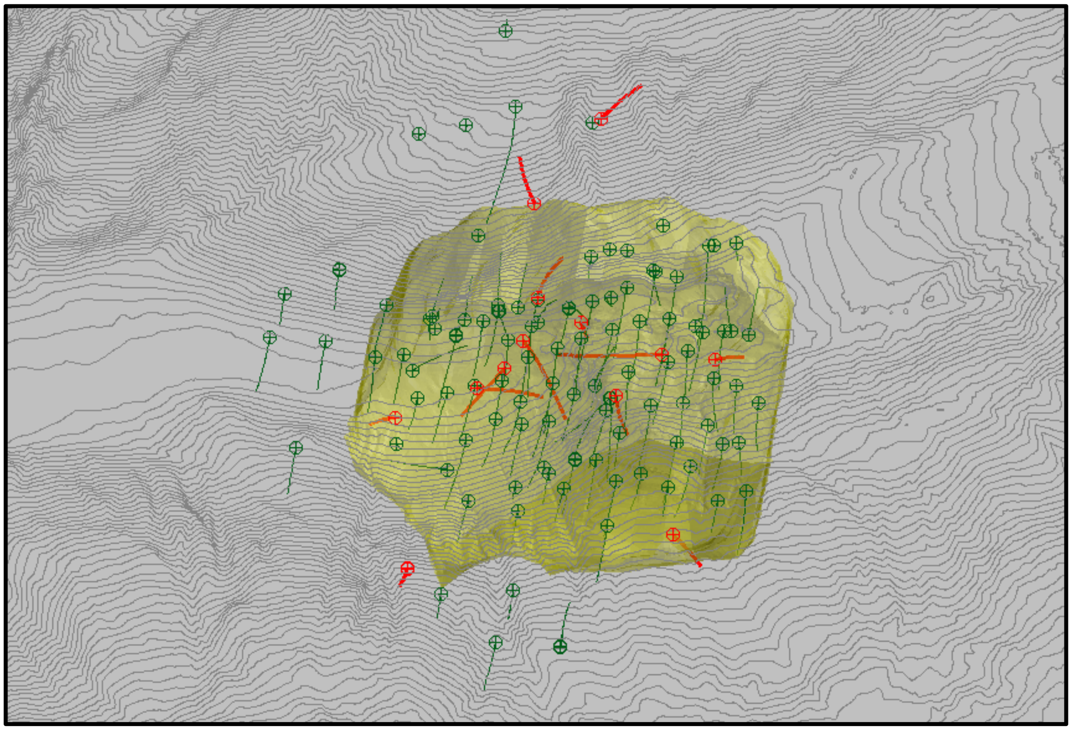
GEOTECHNICAL CONSIDERATIONS

For the 2016 PFS, the characterization of the rock mass focused on the rock in and around the extraction level of the proposed block cave mine and on the mineralized rock above this that will be caved (235 m elevation). A second area of interest involves the rock where the ramps, conveyor drifts, raises, and other mine infrastructure will be excavated to connect the production elevation to surface.

Characterization of the rock was based on geotechnical data collected for exploration drill holes, detailed geotechnical data collected from drilling programs carried out by BGC in 2009 (BGC 2010) and Golder in 2011 (Golder 2012a), outcrop mapping data (Golder 2012a), laboratory testing data (BGC 2010; Golder 2012a), and an interpreted geological model provided by Seabridge.

There are a total of 114 exploration holes and 14 geotechnical holes in the Mitchell deposit area. The borehole locations are shown in Figure 24.24. Geotechnical boreholes are shown in red. No additional analysis of new exploration drillholes was completed for this PEA as the 2012 assessments are still considered valid for the new footprint that is now 70 m deeper.

Figure 24.24 Mitchell Exploration and Geotechnical Borehole Locations



For the purpose of this study, host rock refers to the rock mass outside of the immediate area of mineralization. The host rock in which the mine infrastructure (e.g., raises, conveyor drifts, ramps, etc.) will be excavated has been assessed based on data collected from nearby drill holes.

The caveability assessments made using Laubscher's and Mathews' methods indicate that the size 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 three-dimensional shape of the deposit, suggest that block caving is a suitable mining method for the Mitchell deposit.

The caving mining will draw down the mineralized rock, and a significant depression will develop on surface above the production footprint in the form of a crater. The crater typically develops on surface above and slightly laterally beyond the footprint of the production horizon of the caving mining. The top section of the crater is a relatively steep escarpment (60 to 70°) that is marginally stable but comprised of nominally in place dilated rock. Beneath this is failed broken rock that has progressively sloughed from the rim of the crater. This rock rills down to the bottom of the crater at about 40°.

Mining Method Selection

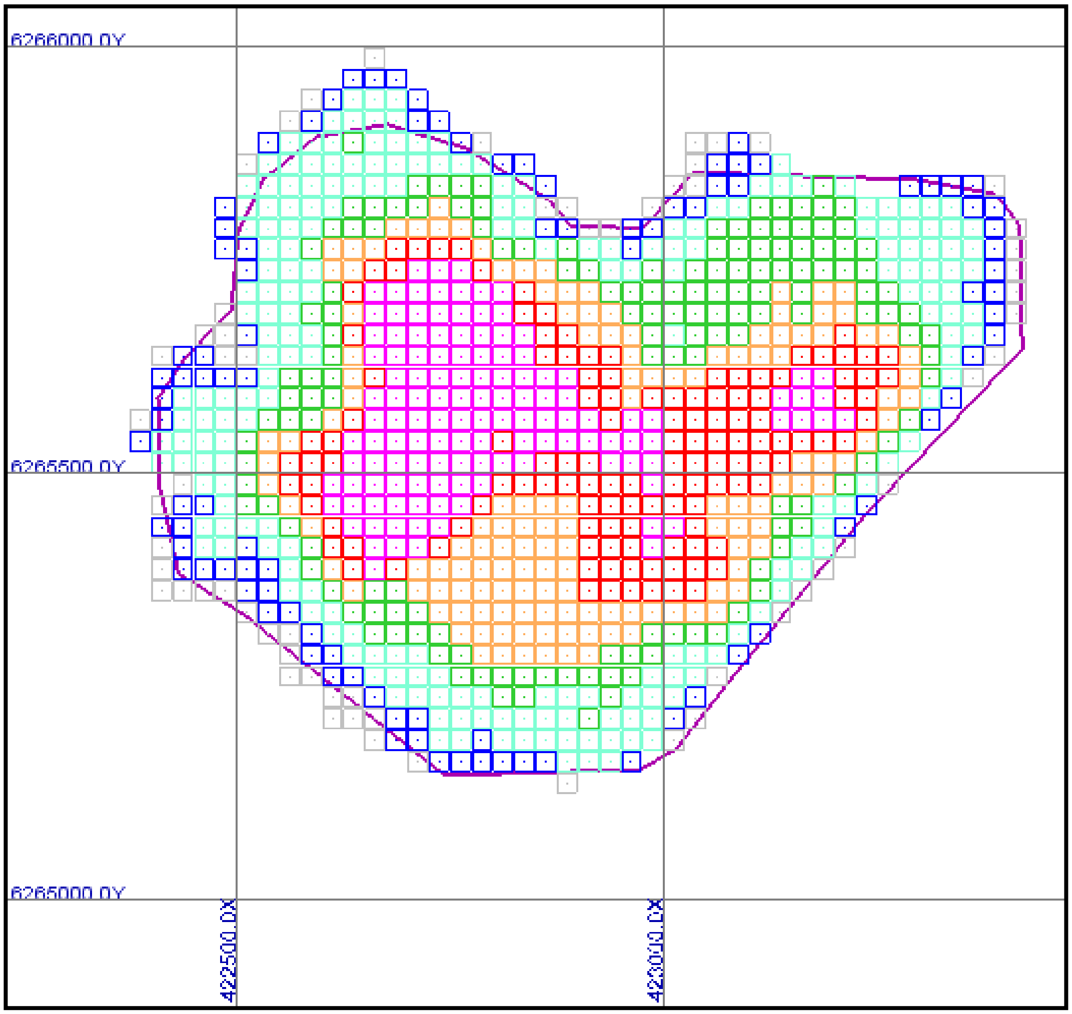
The most appropriate mining methods for this deposit are block caving, sub-level caving and sub-level stoping. Based on mining costs, deposit grade, geometry and depth, block caving was selected as the preferred mining method.

Cave Modelling and Production Scenarios

Seabridge provided the block model with updated NSR block values. Golder used the block model to run GEOVIA PCBC™ Footprint Finder to search for the optimum location and extents of the block cave. Instead of cut-off-grade, various shut-off values based on NSR were used to determine and optimize footprint shape. A series of GEOVIA PCBC™ Footprint Finder runs were performed at varying NSR values from Cdn$20/t to Cdn$26/t, for a production rate of 60 kt/d. The results from these runs were used to generate hill-of-value tables which emphasized NPV and IRR. The footprint used for mine design was based on a NSR shut-off of Cdn$20/t for the footprint shape, and a NSR shut-off of Cdn$20/t for column height. A second lower cave lift was assessed but was not of sufficient size to warrant further work.

A footprint at elevation at 165 m has the most value and the block cave design was based on this footprint. Figure 24.25 shows the GEOVIA PCBC™ Footprint at the 165 m elevation.

Figure 24.25 GEOVIA PCBC™ Footprint for Lift at 165 m Elevation



Note: Figure prepared by Golder Associates Ltd., 2016. Grid system is based on UTM coordinates, with north at the top of the image. Warmer colors indicate higher caving column NSR values, whereas cooler colors indicate lower caving column NSR values.

Dilution

Dilution for the block caving was derived using Laubscher´s dilution matrix, which is a common methodology employed by operating block caving mines. Starting with the in situ block model, a diluted block model is calculated using a 60% dilution entry point and 195 height of interaction zone. All calculations to determine the height of draw of block caving are undertaken using the diluted model.

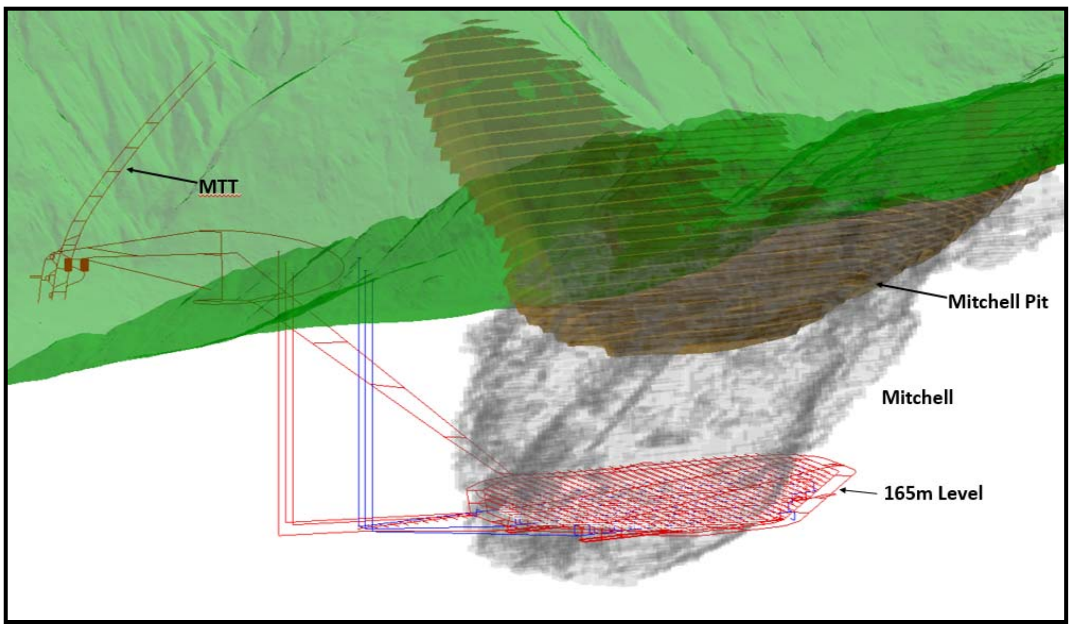
Design Criteria and Layout

The block cave design for Mitchell footprint is at the 165 m elevation. The shape of footprint is irregular oval, with an area of 532,000 m², and a perimeter length of 2.9 km. The widest section of the footprint is 800 m.

Personnel, material, and supplies will access the underground through a main access ramp which will be developed from a portal near the Mitchell OPC area at the 820 m elevation. The Mitchell underground is designed with two ramps to surface to provide auxiliary ventilation in the form of a raise or ventilation loop, to separate the conveyor from the main access, and to allow for emergency egress. Ventilation for the underground operation will be provided by two fresh air raises, and two exhaust air raises that extend from the extraction and haulage levels approximately 700 m to the surface. Twin dewatering tunnels that also provide temporary storage capacity extend 6.0 km from the production level to the southwest where they connect to a raise used to pump water to surface and then to the Water Storage Dam.

The Mitchell design has multiple raises and drifts that can act as an emergency egress. The primary emergency egress will be the access tunnel or conveyor tunnel, whichever is accessible. If both tunnels are inaccessible, it will be possible to exit the mine through one of the fresh air shafts. Figure 24.26 shows the general mine layout for Mitchell.

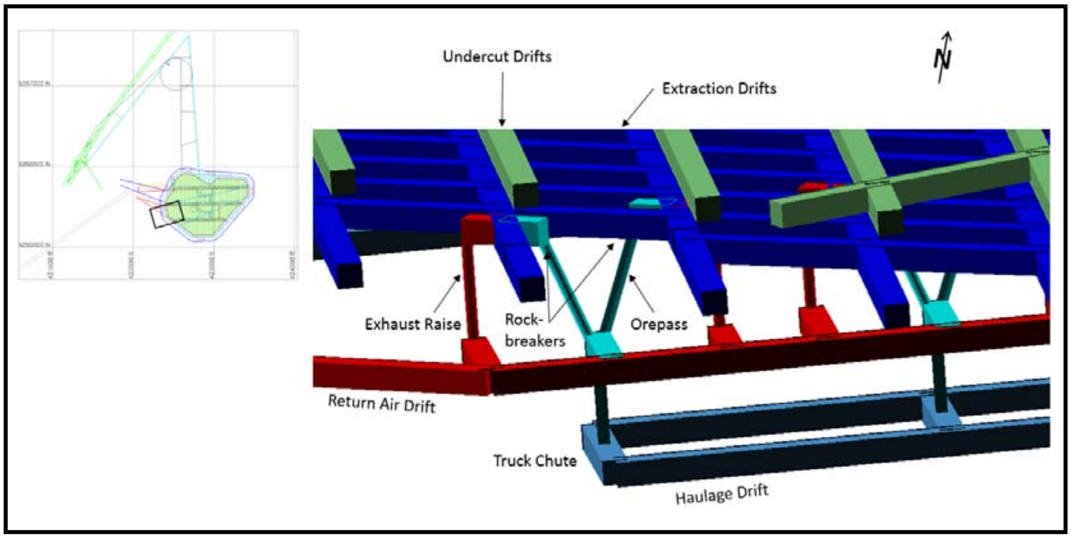
Figure 24.26 Mitchell Mine General Layout with Topo and NSR above Cdn$20/t NSR Shell



Note: Figure prepared by Golder Associates Ltd., 2016.

Figure 24.27 shows the typical level arrangement for the Mitchell mine.

Figure 24.27 Typical Level Arrangement for Mitchell



Note: Figure prepared by Golder Associate Ltd., 2016.

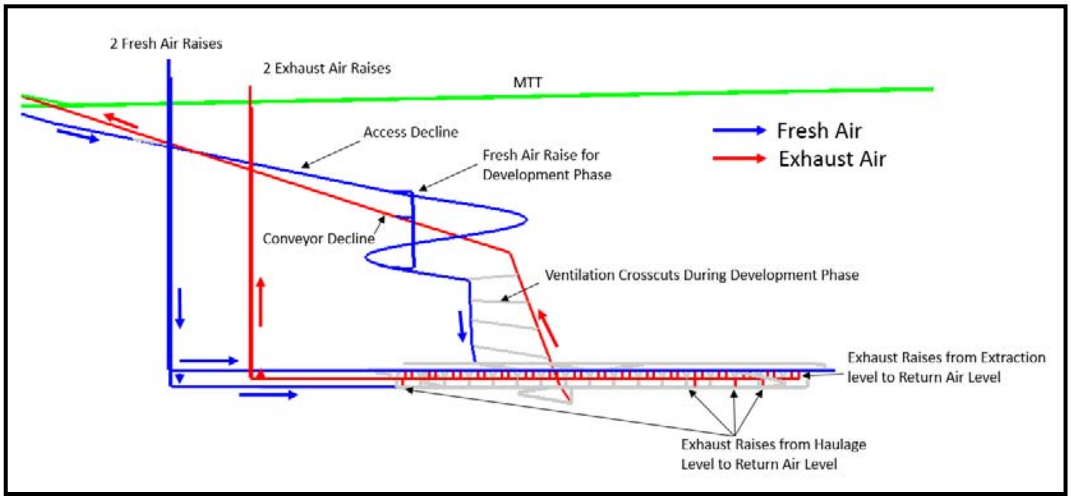
VENTILATION

The total airflow requirements were based upon air quantities of 0.063 m^3/s per kilowatt of diesel equipment, equipment utilization, and engine utilization. The total airflow requirement is estimated at 1,500 m^3/s, which is sufficient to appropriately dilute all noxious gases and particulate matter produced by the mining equipment and activities on each mining level. This equates to a rate of 0.026 m^3/s/t/d.

Heating of mine air in the winter months is included in the design and cost estimates and will be done by mine heaters located at each of the two main fan installations. The mine air heaters will have to heat 1,500 m^3/s to 3°C for five months per year (November through March). The propane requirement is 3.5 million liters per year.

Figure 24.28 is a schematic of the ventilation system showing major airflow directions.

Figure 24.28 Ventilation Schematic for Mitchell Mine (Section Looking North)



MINE DEWATERING

The mine water handling system is designed to handle the water that originates from the groundwater and surface inflows (including possible water from ice that caves into the crater when the ice field is undercut), and water that is introduced to the mine for operations.

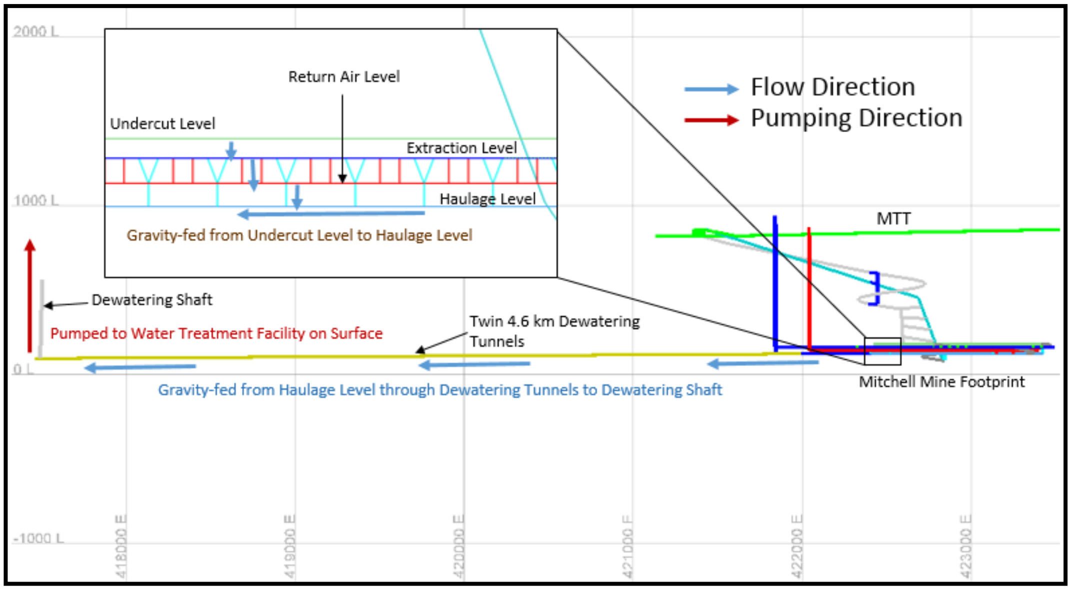
The area of the catchment around the Mitchell pit is approximately 9 km^2. Once the cave breaks through to the pit one to two years after the start of caving mining, the rainfall and snowmelt within this catchment that is not diverted will flow into the crater and percolate through the broken caved rock into the mine workings. Experience at other caving mines indicates that the broken caved rock will not retard the inflows to any significant degree. Under these circumstances, the short-term inflow rates and total inflow volumes for various event durations are expected to be significant and must be managed appropriately. The groundwater inflows by comparison are insignificantly small and have not been considered further in the underground mine water management plan.

In order to reduce surface inflows, diversion ditches will be constructed at strategic locations beyond the crest of the pit. This will include additional diversions and re-contouring required to re-direct water from entering the crater. The total diverted area is 3.28 km^2. The diversion ditches will be located outside areas that might become unstable during cave mining. For design purposes, the diversions were assigned efficiencies of 50 and 65%, depending on expected seasonal conditions to allow for adverse effects of ice, avalanches, snow blockages, etc.

The conceptual design for the underground management system consists of directing the water by gravity to two parallel drainage tunnels. These tunnels will convey water and serve as temporary water storage from the haulage level to the bottom of a shaft that extends up to the ground surface adjacent to the HDS WTP. The pumping system consists of two pumping stations, one underground and one on surface, and the associated pipelines that transfer the water to the WSF. The underground water management system at Mitchell can handle 4 m^3/s (63,000 gal/min). This caters for both the groundwater and peak surface water inflows.

Figure 24.29 shows a schematic of the proposed dewatering strategy for the Mitchell mine.

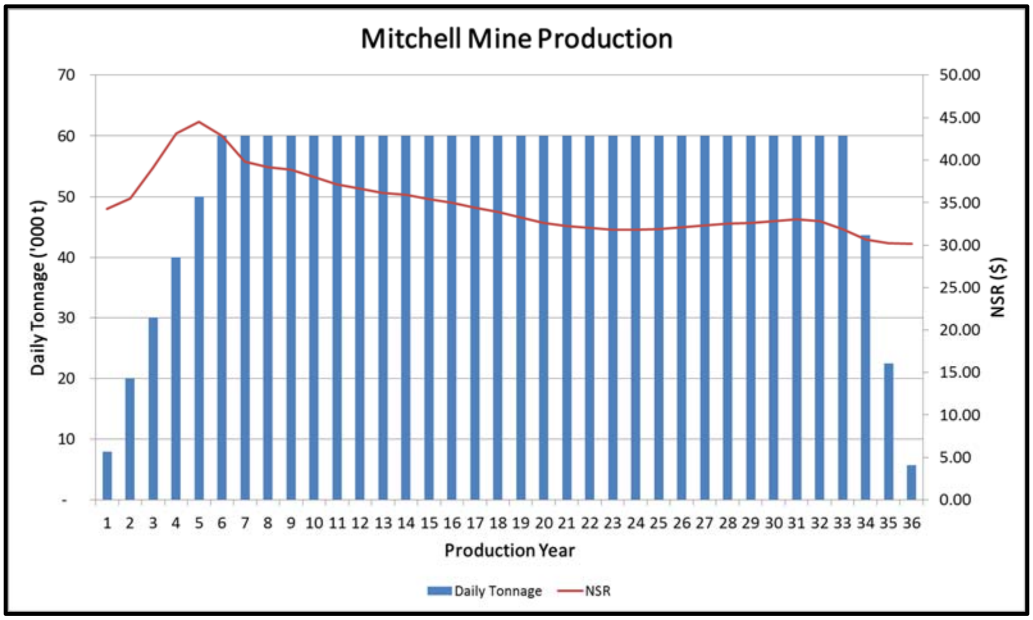
Figure 24.29 Mine Dewatering Arrangement for Mitchell (Long Section Looking North)



PRODUCTION SCHEDULE

The proposed production schedule for Mitchell is shown in Figure 24.30. Note that the years shown apply to the Mitchell underground operation only, not to the combined open pit and underground multi-mine plan of the Project.

Figure 24.30 Mitchell Mine Production Plan



Note: Figure prepared by Golder Associates Ltd., 2016.

Only classified Mineral Resources are evaluated as having grades and NSR value in the production plan. All the unclassified materials are treated as waste by setting the grades to zero. An NSR shut-off of $20/t was used to determine the economic footprint and column extractions. Table 24.10 shows the distribution of Mineral Resources in the PEA mine plan based on their classifications.

Table 24.10 Mitchell Diluted Mineral Resources in PEA Mine Plan

	Measured	Indicated	Inferred*
Tonnes (millions)*	244.9	361.0	87.5
NSR (Cdn$/t)	37.90	36.03	22.55
Au (g/t)	0.68	0.65	0.40
Cu (%)	0.21	0.20	0.13
Ag (g/t)	4.2	4.1	3.1

Notes: *Includes 16.3 Mt of mineralized dilution (the portion of Measured, Indicated, and Inferred material that is below the Mineral Resource cut-off, $0 < NSR < $16) and 18.3 Mt of non-mineralized dilution (unclassified material set to zero grade).

Blasting and Explosives

The PEA has assumed that explosives will be stored in the Mitchell Valley to service both the Mitchell and Iron Cap mines. The explosives will be delivered by truck to the underground mine as required. Small underground explosive magazines are included for development mining and secondary blasting activities.

Explosives will be used underground for five main purposes:

- preparation development of mineralized areas

- draw bell construction
- undercutting
- development in barren material
- secondary blasting of boulders and hang ups.

MINING EQUIPMENT

Primary and support equipment requirements for the proposed Mitchell caving operation are summarized in Table 24.11.

Table 24.11 Mobile Equipment Requirements, Mitchell

Equipment	No. of Units
Development	
Two-boom Jumbo	3
LHD (7 m^3)	3
Haul Truck (55 t)	7
Rock Bolter (33 to 45 mm)	9
Shotcrete Sprayer (35,700 lb)	1
ANFO Charger	2
Transmixer (40,000 lb)	1
Raisebore	1
Scissor Lift and Boom Truck	4
Production	
Longhole Drill (ITH, 89 to 216 mm)	5
Emulsion Loader	1
Haul Truck (55 t)	18
LHD (7 m^3)	24
Block Holer	2
Scissor Lift and Boom Truck	8
Mobile Rockbreaker (4 to 5.4 m^3)	4
Support	
Grader	2
Big Personnel Carrier	4
Small Personnel Carrier	6
Lube Truck	3
Contractor	
Two-boom Jumbo	1
Haul Truck (55 t)	1
LHD (7 m^3)	1
Contractor	
Rock Bolter (33 to 45 mm)	1
ANFO Charger	1
Scissor Lift and Boom Truck	1

MANPOWER CONSIDERATIONS

The total estimated workforce for Mitchell underground operation during peak construction is 351, while the total workforce required during the peak production is approximately 492 persons. The estimate is based on the development and production tonnages and equipment requirements.

COMMENTS ON MITCHELL MINING METHOD

The basis for the Mitchell underground mine development is to support the mill feed requirement at the Project site from multiple mines. The development stage of Mitchell deposit is approximately five years, which includes mine access, initial footprint development, and construction of major mine infrastructure, such as underground crusher, material handling system, shops, dewatering system, primary ventilation fans, etc. Following the development stage, initial ramp up period is five years to reach full production of 21.9 Mt/a (60 kt/d), with a total production mine life of 36 years.

24.17 RECOVERY METHODS

The proposed KSM plant will have an average process rate of 170,000 t/d. The Process Plant will receive mill feed from the Mitchell and Sulphurets open pits and the Deep Kerr, Mitchell and Iron Cap underground operations. The Mitchell open pit and Deep Kerr underground mines will be the main source of mill feed, contributing about 83% of the total plant feed over the LOM. Based on the available information, testwork performed on sample locations for the open pit production provides a reasonable indication of the mineralogical characteristics of the material examined in this PEA for both the surface and underground mine extraction scenarios. The process flowsheet considered in the 2016 PFS is also an appropriate basis for the PEA given the nature of the mill feed and the level of sampling performed through the deposit.

Several metallurgical test programs have been carried out to assess the recoverability of copper, gold and silver values using the projected flowsheet which is essentially identical to that utilized in the 2016 PFS by Tetra Tech. The results of the programs performed for the PFS indicate that the mineral samples from the Mitchel, Sulphurets, Iron Cap and Deep Kerr mineralization deposits are amenable to flotation followed by cyanide leaching process. With the use of the same flowsheet in the PEA as was used in the PFS, it was possible to rely upon the use of this testwork to support the PEA process.

Detailed characterization and metallurgical testwork on Mitchell, Sulphurets, Iron Cap, and Deep Kerr samples is presented in Section 13.0 of the 2016 PFS. This testwork examined comminution (crushing and grinding), concentration by flotation of copper and gold into a saleable concentrate, and the further recovery of gold from flotation tailings by leaching. Recovery equations from this testwork were produced by Tetra Tech, but reviewed and considered adequate for the purposes of supporting the PEA process design by Amec Foster Wheeler.

Comminution testwork results indicate that the samples from all the deposits are moderately hard for SAG and ball milling. Additional comminution tests showed that the samples tested were amenable to particle size reduction by the HPGR approach as preparation for ball mill grinding.

Flotation testwork (batch and locked cycle) indicate that the mineralization is amenable to concentration into a saleable copper-gold concentrate with no significant penalty elements. Following flotation, cyanidation tests showed that it was possible to recover gold and silver from the tailings to bullion. The metallurgical test results obtained from the various test programs were used to predict plant metallurgical performance parameters for copper, gold and silver. The metallurgical performance projections of the four KSM mineralization types are summarized in Section 13.2 of the 2016 PFS. In addition, work was performed to indicate the consumption of reagents and grinding media in comminution, flotation and cyanidation. Because of high cyanide consumptions, testwork on cyanide recovery was also performed and the results incorporated into the operating costs.

For the purpose of this PEA, the process circuit will incorporate three stage crushing, milling, conventional flotation and cyanidation processes for the recovery of copper, gold and silver. The overall process flow diagram developed for the PFS has been carried through to the PEA, except for the molybdenum recovery circuit, which has been eliminated. However, in order to process the higher throughput, equipment sizing for the PEA modified to larger but proven units available in the market, optimizing plant footprint and energy consumption for the tons processed. Redesign of the facilities was limited to optimizing the layout provided by the use of the larger equipment in the PEA relative to the 2016 PFS.

24.17.1 PROCESS PLANT

The Process Plant was designed appropriate to the testwork and will consist of five separate facilities for the handling and processing of mineralized material:

- a primary crushing and handling facility at the Mitchell open pit mine site
- a primary crushing and handling facility at the Sulphurets open pit mine site
- a primary crushing and handling facility at the Deep Kerr underground mine site
- a train transportation system through the MTT
- a main process plant facility at the Treaty OPC area, including coarse mill feed stockpiling, secondary/tertiary crushing, ball mill grinding, flotation, regrinding, concentrate dewatering, and cyanidation followed by treatment of tailings prior to deposition with the TMF.

24.17.2 FLOWSHEET DESCRIPTION

COMMINUTION

Primary crushing facilities will be located at each pit to reduce the ROM particle size to approximately 80% passing 150 mm using gyratory 60 inches x 113 inches gyratory crushers. Two units will be located at the Mitchell pit and one unit at Sulphurets pit.

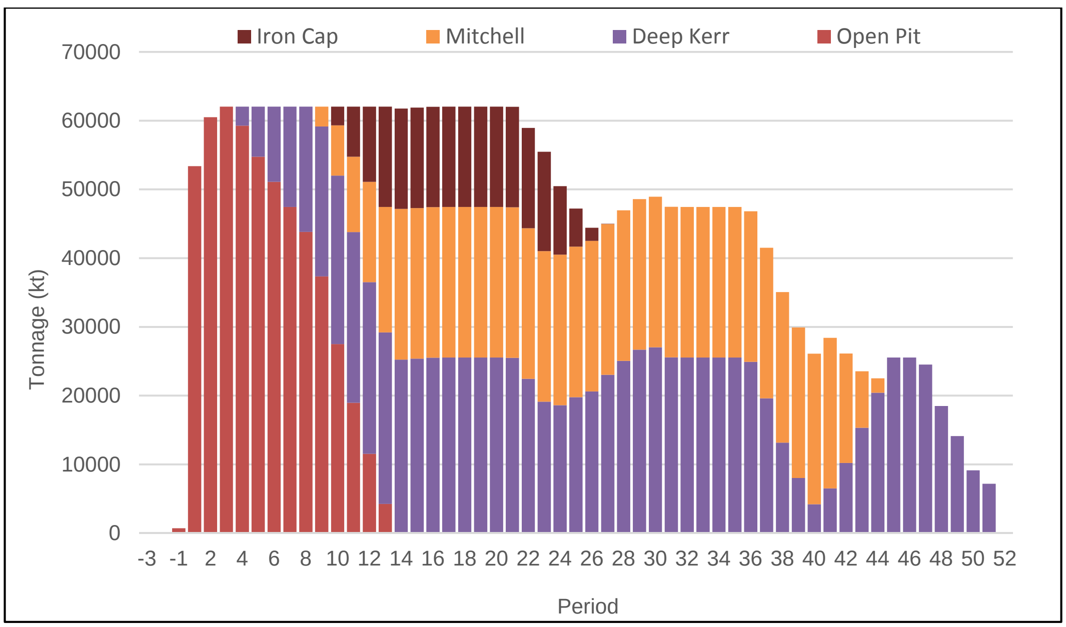
Crushed mineralized material will be conveyed and loaded onto the MTT train to the main plant site located at the Treaty OPC site, approximately 23 km northeast of the Mine Site.

Between years 4 and 51, underground mineralized material from Deep Kerr, Iron Cap and Mitchell will supplement open pit production and eventually replace it. At the Deep Kerr site, two, 60 inch by 89 inch gyratory crushers will reduce run-of-mine mill feed produced by block caving to 80% passing 165 mm. The crushed material will be conveyed on the Kerr-MTT conveyor to the MTT train for delivery to the coarse stockpile at the Treaty OPC. The Iron Cap and Mitchell mineralized material will also be mined by block caving and will be crushed by existing gyratories on site to 80% passing 150 mm or finer. The crushed material will also be conveyed to the MTT and loaded onto the MTT train to the coarse mill feed stockpile at the Treaty OPC.

The proportion of mine production being supplied through the various crushers is shown in Figure 24.31 indicating a blending effect on the mill feed being processed through the subsequent Process Plant (period shown is in years).

From the coarse mill feed stockpile, the reclaimed feed material will be conveyed to the secondary crushing facility and fed to two vibrating screens. The oversize from each screen oversize will feed a MP2500 secondary cone crusher or equivalent. The cone crusher product will be returned to the screen feed conveyor. 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.

Figure 24.31 Production Schedule – LOM



The crushed mill-feed material from secondary crushing will be reclaimed from the 60,000 t stockpile by reclaim apron feeders onto two HPGR feed conveyors. These conveyors will deliver the mill-feed material to two tertiary crusher HPGR feed surge bins.

The reclaimed mill-feed material will be further crushed by four HPGR crushers. Four belt feeders will withdraw the reclaimed mill-feed material from the two HPGR feed surge bins and feed each of the four HPGR crushers separately. Each HPGR crusher is in closed circuit with double deck vibrating screens with the HGPR discharge wet-screened at a cut size of 6 mm. The screen oversize will return via conveying to 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 nominal total process capacity of 7,535 t/h.

The grinding circuit will employ four conventional ball mills to grind the HPGR product to a particle size of 80% passing 125 to 150 µm. Each ball mill will be in closed-circuit with a cluster of cyclones. The cyclone underflow will gravity-flow to the ball mill feed chute, while the overflow will gravity flow to one of four copper-gold rougher flotation trains. The overall capacity of the grinding circuits is designed to have a nominal processing rate of 7,535 t/h.

FLOTATION

The products from the primary grinding circuits will feed the copper-gold rougher/scavenger flotation circuit. This circuit is composed of two parallel banks of four 600 m^3 flotation tank cells. The four cells of each bank will produce copper rougher flotation concentrates (at an overall mass pull of 6%) which will then be reground to a particle size of 80% passing 20 µm in two tower mills (each with an installed power of 3,356 kW). The reground copper-gold rougher concentrate will then be upgraded in a cleaner flotation circuit with three stages of copper cleaner flotation producing a copper-gold concentrate with a target grade of 25% copper. The first cleaning stage will consist of four 100 m^3 tank cells, the second stage of four, 50 m^3 tank cells and in the third stage three column cells will be used. First cleaner flotation tailing will be further floated in four 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 while the tailing will report to the gold leaching circuit. The tailing from the second and third cleaner flotation stages will be returned to the first cell in the prior cleaner flotation circuit.

The upgraded copper-gold concentrate will be thickened in a 21 m diameter high rate thickener. The thickener underflow will be directed to the three 160 m^2 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.

Located at the tail of each copper rougher scavenger bank is a further four cells which are used for pyrite flotation. These cells will produce a pyrite concentrate which is gold bearing. The final pyrite flotation tailing will be sent to center lined cell within the TMF for storage. The pyrite concentrate will be reground in two tower mills (each with an installed power of 3,356 kW) to a particle size of 80% passing 20 µm.

Gold Recovery from the Gold-Bearing Pyrite Products

The reground pyrite concentrate will report to the gold leach circuit. Both the reground gold-bearing pyrite concentrate and the first cleaner scavenger tailing will each be thickened in dedicated 40 m diameter thickeners to a solids density of 65%. The underflow of each thickener will be pumped to its own dedicated cyanide leaching line. Each line will consist of three pre-treatment tanks and eight conventional CIL tanks.

The loaded carbon leaving the CIL circuit will be transferred to the elution circuit for gold recovery followed by the reactivation of carbon in an electrically heated rotary kiln. The reactivated carbon will be circulated back to the CIL circuit. The tailings from the CIL trains will pass over a safety screen system prior to be processed for cyanide recovery.

The pregnant solution from the elution system will go through the electrowinning cell where a precious metal sludge will be produced. The sludge will be filtered, dried and smelted in an induction furnace to produce gold and silver doré.

Treatment of Leach Residues

The combined residue from the CIL circuits will be pumped to a single two-stage conventional counter current decantation (CCD) washing circuit. The CCD circuit will consist of two 45 m diameter high-rate thickeners in series. The thickener overflow from the first stage washing will be pumped to the cyanide recovery system. There, the barren leach solution containing copper cyanide complexes from cyanidation is treated with sulphuric acid and sodium sulphide to precipitate copper as copper sulphide, which is thickened and recovered. The acidification converts part of the cyanide to hydrogen cyanide gas allowing it to be recovered in a wet scrubber system as a high strength cyanide solution. The underflow (washed residues) of the second thickener will be sent to the cyanide destruction circuit prior to being pumped to the TMF.

Tails Management

The flotation tailing and the treated CIL residues will separately flow 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 reclaimed to the process plant 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 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 cyclone classification. The coarse fraction will be used to construct the tailing dam and the fines will directly report to the TMF. The second tailings pipeline will report directly to the TMF. 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.

REAGENTS HANDLING

All the reagents will be prepared in a dedicated reagent preparation and storage facility within a containment area. The liquid reagents will be added in the undiluted form via metering pumps. The solid reagents will be prepared into adequate strength solutions in dedicated mixing tanks and stored in holding tanks to be added to the processes via metering pumps.

WATER AND PRESSURIZED AIR 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.

Plant air service systems will supply blower air to flotation, moderate pressure air to leach, cyanide recovery and cyanide destruction, and high pressure air to filtration and general plant and instrumentation services.

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.

PROCESS CONTROL AND INSTRUMENTATION

The plant control system will consist of a DCS with PC-based OIS located in control rooms at the process facilities. 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 and sulphur dioxide monitoring/alarm systems will be installed at site where needed. In addition, CCTV support will be provided at various locations at the crushing and plant facilities to ensure comprehensive site monitoring.

PROCESS FLOW DIAGRAM

Figure 24.32 illustrates the overall process flow diagram. The major design consideration in the process plant equipment sizing and layout were the use of the largest equipment sizing available in order to minimize pumping and piping requirements, process building footprint and capital costs. Figure 24.33 illustrates the main process building layout.

Figure 24.32 Overall PEA Process Flow Diagram

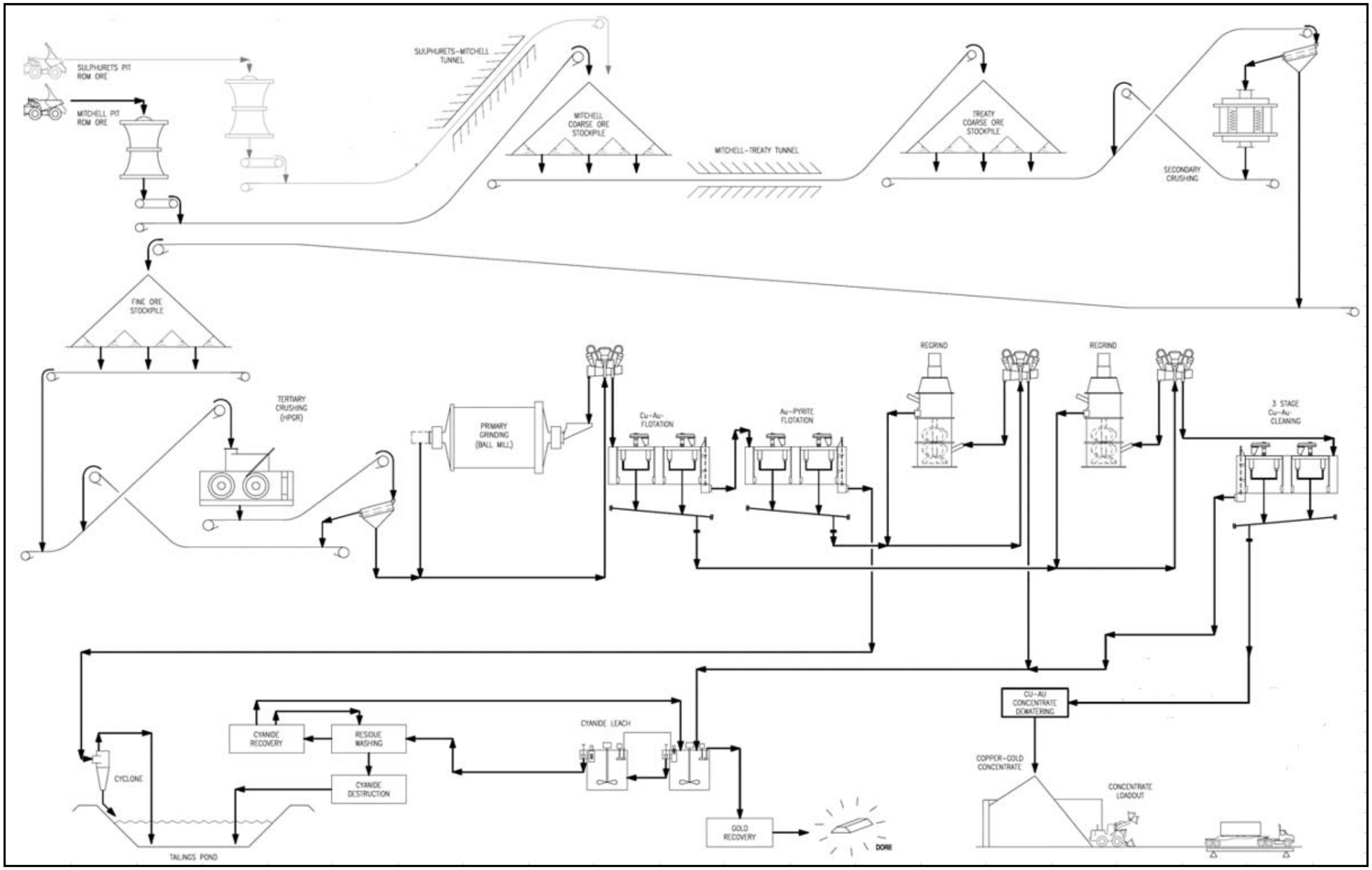
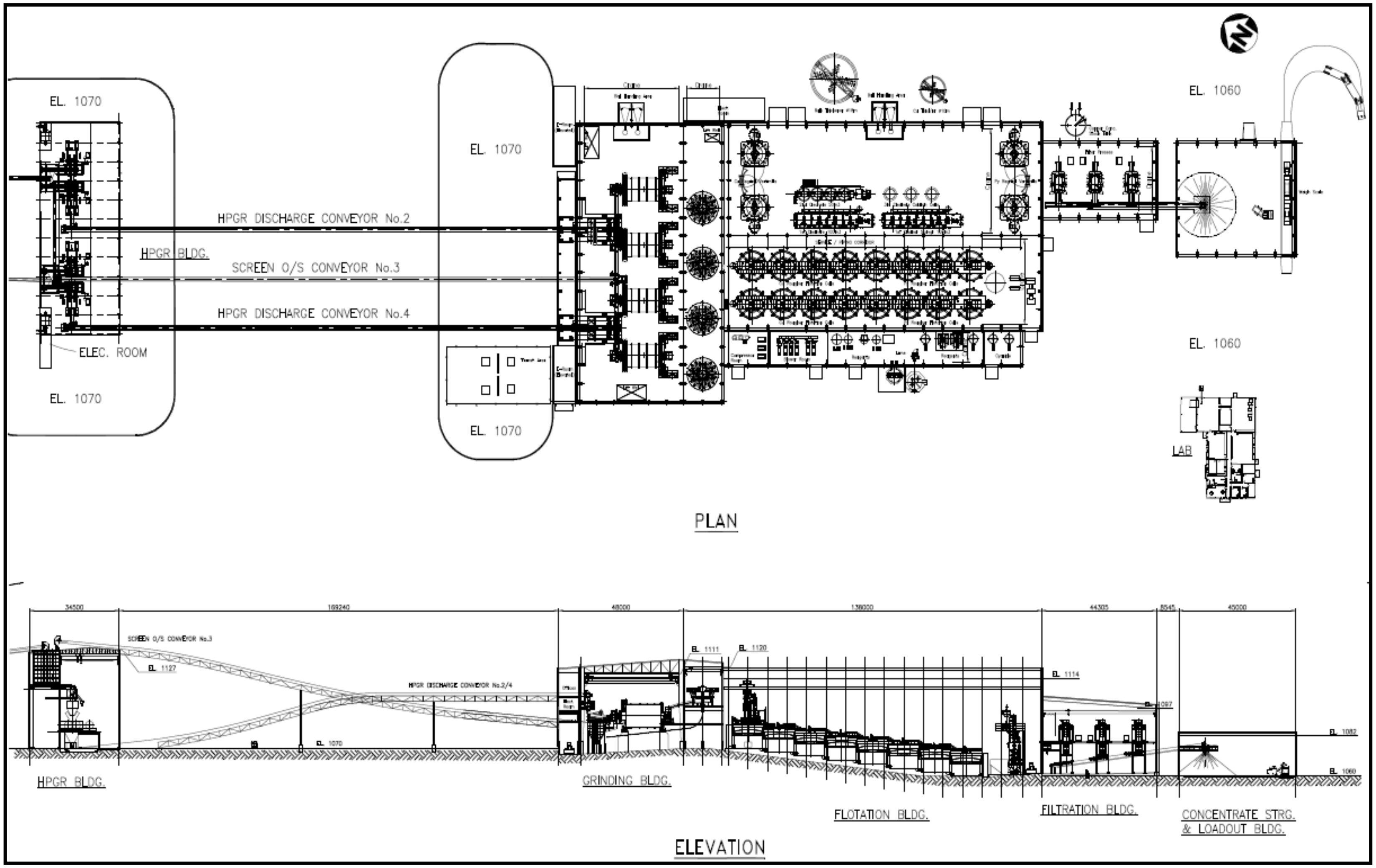


Figure 24.33 PEA Process Building Layout



24.17.3 MILL DESIGN CRITERIA

The process plant design criteria for the PEA flowsheet are based on the following sources of information and analysis:

- Section 13.0 of this report, which summarizes the metallurgical testwork analysis and recovery assessment by Tetra Tech
- crushing and grinding calculations by Amec Foster Wheeler.

Table 24.12 summarizes the main design criteria established for the PEA. The criteria are for a processing plant of 170,000 t/d capacity with a plant availability of 94%.

Table 24.12 Major PEA Process Design Criteria

Parameter	Units	Value
Plant Feed Rate		
Process Plant Availability	%	94
Annual Processing Rate	Mt/a, dry	62
Daily Processing Rate	t/d, dry	170,000
Hourly Processing Rate, Nominal	t/h, dry	7,535
Primary Grind Size, P_{80}	μm	125 – 150
Concentrate Regrind Size, P_{80}	μm	20
Leach Circuit	-	CIL
Elution Circuit	-	Pressure Zadra
Carbon Regeneration	-	Electric Kiln
Residue Management	-	CCD
Cyanide Recovery Method	-	AVR
Cyanide Destruction Method	-	SO_2 + Air
Head Grades and Recoveries		
Head Copper Grade, Average	%	0.32
Head Gold Grade, Average	g/t Au	0.52
Head Silver Grade, Average	g/t Ag	2.7
Recovery to Flotation Concentrate		
Copper	%	87.6
Gold	%	60
Silver	%	50.4
Recovery to Dore		
Gold	%	15.6
Silver	%	16.7

24.17.4 MAJOR EQUIPMENT SIZING CRITERIA

Metallurgical calculation output for comminution circuit equipment sizing and installed power is listed in Table 24.13 and Table 24.14.

Table 24.13 Comminution – Major Equipment Sizing

Equipment	Unit	Value
Primary Crushing – Gyratory Crusher (Open Pit)		
Number of Units	-	3
Installed Motor per Unit	kW	750
Recommended Gyratory Crusher Size	mm	1524 x 2870
Secondary Crushing – Cone Crusher		
Number of Units	-	2
Installed Motor per Unit	kW	1,865
Tertiary Crushing – HPGR		
Number of Crushers	-	4
Installed Power per Unit	kW	7,400
Recommended HPGR Size	mm Ø x mm W	2,600 x 1,750
Ball Mill		
Number of Ball Mills	-	4
Dimensions	m (ø x length EGL)	8.2 x 13.2
	ft. (ø x length EGL)	27 x 43.3"
Installed motor (per unit)	MW	19,000

Table 24.14 Copper-Gold Flotation – Major Equipment List

Equipment	Unit	Value
Rougher/Scavenger Flotation		
Number of Trains	-	2
Number of Cells/Train	-	4
Cell Volume	m^3	600
Installed Motor	kW	450
Bulk Concentrate Regrind		
Mill Type	-	Tower Mill
Number of Mills	-	2
Installed Power	kW	3356
1st Cleaner Flotation		
Cell Type	-	Tank Cell
Number of Cells	-	4
Cell Size	m^3	100
Installed Motor	kW	132
2nd Cleaner Flotation		
Number of Cells	-	4
Cell Volume	m^3	50
Installed Motor	kW	55
3rd Cleaner Flotation		
Cell Type	-	Column Cell
Number of Columns	-	3

table continues…

Equipment	Unit	Value
Column Size	-	-
Copper Concentrate Dewatering	-	-
Thickener Type	-	High Rate
Thickener Diameter	m	21
Filter Type	-	Vertical
Number of Filters	-	3
Filter Size	m^2	160

Table 24.15 Pyrite Flotation and Cyanidation – Major Equipment List

Equipment	Unit	Value
Pyrite Flotation		
Number of Trains	-	2
Number of Cells/Train	-	4
Cell Volume	m^3	600
Installed Motor	kW	450
Pyrite Concentrate Regrind		
Mill Type	-	Tower Mill
Number of Mills	-	2
Installed Power	kW	3356
Pre-Leach Thickeners		
Number of Thickeners	-	2
Thickener Size	m, diameter	40
Pre-Aeration Tanks		
Number of Trains	-	2
Number of Tanks/Train	-	3
Tank Volume	m^3	573
Cyanidation Tanks		
Number of Trains	-	2
Number of Tanks/Train	-	8
Tank Volume	m^3	2,474
Cyanide Recovery Circuit		
Number of Thickeners	-	2
Thickener size	m, diameter	40

24.17.5 INTERPRETATION AND CONCLUSIONS

The increased throughput envisioned in this PEA can be achieved using the largest proven units available in the market for each unit operation without significantly redesigning the facilities proposed in the 2016 PFS.

24.18 PROJECT INFRASTRUCTURE

24.18.1 ONSITE

Designs for the water and waste management structures within the PEA are based on the 2016 PFS level designs and provide conceptual level adjustments to include the PEA mine plan layouts, staging plan and capacities required for the PEA production schedule.

OPEN PIT ROCK STORAGE FACILITY

Rock storage requirements for the PEA are significantly lessened due to reductions in open pit phases of mining. The PFS mine plan includes 3.0 Bt of waste rock and the PEA mine plan only includes 0.6 Bt. As a result the PEA requires only the Mitchell RSF, with a smaller footprint than in the PFS. The geotechnical and water management design criteria, operating, and closure design concepts from the PFS are adopted for the Mitchell RSF as included in the PEA. A revised selenium drain configuration is provided in the PEA for the Mitchell RSF.

The smaller footprint of the Mitchell RSF may offer additional options for selenium capture. With the smaller footprint, the reduced interaction between the toe of the RSF and the WSF impoundment also increases the water storage volume available behind the WSD. These two potential design optimizations were not assessed within the PEA.

WATER MANAGEMENT

For the PEA water management and water treatment design criteria remain the same as in the 2016 PFS. The elimination of the McTagg RSF in the PEA mine plan results in a reduction of un-diverted catchment area above the WSD by 15%. In the PEA configuration all of the McTagg Valley catchments are diverted by the Stage 1 McTagg tunnel and only Stage 1 is required which reduces sustaining capital. As inflows scale with catchment area the required HDS WTP capacity and estimated capital cost were reduced by the same factor. For the PEA level of design the WSD crest elevation was kept at the same elevation. WSD storage could potentially be reduced, reducing the WSD crest elevation, although that optimization was not considered for the PEA.

TAILINGS MANAGEMENT

TMF configurations for the PEA adopt the 2016 PFS TMF design, adding modifications to provide a conceptual level PEA TMF design. The higher peak PEA throughput of an average annual 170,000 t/d during early operations requires 10 m higher starter dams for the North, Splitter and Saddle dams which increases initial capital. The higher throughput results in a TMF Stage 1 duration for North Cell operation of 19 years instead of 25 years. Due to reduced throughput during the underground only Stage 2 of the TMF, which requires construction of the Southeast Dam and operation of the South Cell, there is no change required in starter dam height for the Southeast Dam. Final overall crest elevations for all dams and the total design capacity of 2.3 Bt are similar to the PFS design and within the contingency storage provided for the PFS. The only other change to the TMF required for the PEA configuration is an enlargement of the CIL Residue tailing cell. In the PFS design an allowance of 13% of storage is provided for CIL tailing. CIL storage for the PEA was increased to 20% due to the increased sulphide content of Deep Kerr mineralization. The lined CIL basin is enlarged to accommodate the increased CIL storage by a relocation of the Saddle Dam centerline to the south.

MTT Rail System

MTT rail system will transport crushed mill feed, freight, fuel and personnel between the Mitchell Valley and Treaty Valley

Crushed mill feed will be conveyed to one of two 15,000 t capacity underground bins. Loading chutes under the bin will feed mineralized material into train wagons for transport to Treaty where the wagons will bottom dump the crushed mill feed into a 15,000 t capacity ore bin. Apron feeders will then reclaim the mill feed to a belt conveyor which will report to the 60,000 t live capacity COS.

Ten mill feed trains will be needed to deliver an average of 170,000 t/d of mill feed to the process plant. Each mill feed train will be made-up of a 140 t electric locomotive and 16 off 42 m^3 bottom-dump mill feed wagons.

Specially configured personnel and freight trains will transport freight, fuel, and personnel through the MTT (Table 24.16). Staging areas at each end of the MTT for marshalling and loading/unloading trains at each end will separate these activities from those for mill feed transport. Personnel, freight, and fuel handling will be scheduled during the day shift operations. A maintenance shop and siding for rolling stock will be located in the Treaty staging area.

Train operations, with the exception of train loading, will be controlled by an automated controls and scheduling system. The main control room will be located in the Treaty staging area. Locomotives will be unmanned and not require engine drivers.

Table 24.16 Freight, Fuel, and Personnel Volume

Transport	Unit	Amount
Freight	t/d	550
Fuel	L/d	300,000
Persons	persons/wk	560

To meet the daily production requirements and provide sufficient surge capacity during periods of peak demand, the rolling stock fleet will be as follows:

- 12 off 140 t locomotives
- 192 off 42 m^3 bottom-discharge mill feed wagons
- 12 off 50 t multi-purpose freight wagons
- 6 off ISO fuel transport tanks wagons
- 6 off ISO lime slurry transport wagons

- 6 off 60-seat personnel carriages
- 2 off battery shunting wagons
- 1 off rail maintenance vehicle
- 6 off heavy-duty drop frame tri-axle rubber tired ISO-tank chassis.

The MTT rail system will have a maximum capacity of 210,000 t/d of mill feed.

Facilities, Buildings, and Services

Facilities, buildings and services in the Mitchell Valley and Treaty Valley will include the following:

- site access roads
- fuel receiving, storage, and dispensing
- sewage treatment
- medical buildings and ambulance stations
- administration offices
- consulting engineers' and contractors' offices and storage/laydown areas
- warehouses, cold storage, and maintenance buildings
- tuck maintenance and wash bay
- batch plants
- accommodation facilities including receptions, lunch rooms and cafeterias, mine dry and wash cars, recreation facilities, kitchen, and parking areas
- potable water treatment
- potable, process, fire and fresh water systems
- container storage
- laydown and equipment and materials storage areas
- landfill
- waste management storage and handling and Incinerators
- security fencing and gates
- helipads
- explosives magazines and AN prill storage areas
- explosives manufacturing plant
- communication systems
- auxiliary truck and support vehicles
- electrical power supply and distribution.

Main Substation No. 1 will supply electrical power to plant and equipment in the Treaty Valley and to Mitchell Substation No. 2. High, medium or low voltage systems will distribute power to local loads. Over the LOM the peak electrical load is estimated to total approximately 240 MW.

24.18.2 OFF-SITE INFRASTRUCTURE

ELECTRICAL POWER SUPPLY

Electrical power will be supplied from the 287 kV NTL transmission line that runs from the Skeena Substation near Terrace, BC, to the substation near Bob Quinn Lake. KSM will connect to the NTL transmission at the Treaty Creek Switching Station located adjacent to Highway 37, approximately 18 km south of Bell II. A new 30 km long, 287 kV tap line will connect this substation to the Main Treaty Substation No. 1.

ACCESS ROADS

The access roads to the Project site will be as follows:

- Eskay Creek Access Road
- CCAR
- TCAR
- Lower NTAR
- Upper NTAR
- Cut-off Ditch Access Road
- MTT Tunnel Adit Access Road
- Frank Mackie Winter Access Road.

LOGISTICS

Inbound equipment and materials will be transported either by barge to Stewart, BC, or by rail to Terrace, BC, where these loads will be consolidated at local marshalling yards or staging areas for onward transport by truck to KSM.

Copper concentrate will be transported by truck to a deep water port in Stewart, BC, where it will be held in storage until loaded onto oceangoing vessels.

24.18.3 DEEP KERR PROJECT INFRASTRUCTURE

ROAD AND LOGISTICS

Nearly all personnel and materials accessing to the Deep Kerr Mine will come via the MTT tunnel system initiating in the Treaty Valley area. After exiting the MTT system in the Mitchell Valley area, further connections via a series of roads developed for the overall Project site will lead to the Deep Kerr surface complex area just outside of the Deep Kerr portals.

All Deep Kerr mines are accessed through a series of shared portals and vent adits along the northwestern base of Kerr Mountain, facing Sulphurets Creek. Mine access is primarily through two of the intake portals, one of which directly connects to the footprint area as a main ventilation intake drive, and the second which eventually parallels the underground conveyor as a service drive towards the crusher area. These two primary access ways to surface will be extended via ramps to the lower subsequent lifts.

Underground Crushers and Conveyors

Each lift will have two, 60 inch x 89 inch gyratory crushers on the haulage level, which is sufficient to support the peak production rate of 70 kt/d. Each crusher will have three dumping points for rear dump articulated underground haul trucks. The crushers will discharge to an incline conveyor system which connects to a 7 km common conveyor system that transports mineralized material underground, under the Sulphurets Creek, and up to the approximately 20 km MTT system which is shared by the entire Project mine sites.

For Lift 1 only, a temporary jaw crusher facility is utilized for the early production years to advance production initiation. In later years it can be used to support ongoing mine development efforts.

Underground conveyors assume belts of 60-inch width moving at speeds ranging from 4.0 to 5.5 m/s. Conveyor modules are assumed to be suspended from drift backs with chains. Conveyor inclinations do not exceed 15%.

Underground Mine Service Water System

Mine service water is supplied from the surface complex near the Deep Kerr portal area and piped underground through a backbone to the main areas for each lift.

From this backbone, smaller pipelines will branch off to support mine development headings, longhole drilling, shop activities, etc.

Underground Electrical Reticulation System

Power to the mine will be sourced from a 115 kV substation located in the surface complex near the Deep Kerr portal area. From here power will be distributed via two separate 13.8 kV feeders for each lift to create a ring main style system .

Waste Storage Facilities

About 5.5 Mt of waste rock will be generated during the initial mine development and ramp-up, and brought to surface. The PEA mine plan assumes the NPAG portion of this material will be used in construction of the pad for the surface complex located outside of the portal areas (e.g. camps, shops, offices, warehouses, etc.), and the other material will be hauled to the Mitchell RSF.

Waste rock generated after first production is assumed to be either diluted into the production mill feed or hauled to the surface as needed.

Water Management

The underground mine dewatering system (refer to Section 24.16.2 Mine Dewatering) is discharged to the surface complex established outside of the Deep Kerr portal areas. From here a water pumping system will transfer the water via pipeline to the main water treatment dam and facility near the Mitchell valley area which will be established to support all water treatment for the Project site.

Surface Infrastructure

The PEA design assumes the support for the Deep Kerr operations will be integrated with the overall Project site. This will include major equipment maintenance and fuel provision.

Surface infrastructure required to support operations will include:

- equipment shop for major repairs
- explosives magazine
- warehouse
- water pump stations
- general mine offices
- control room and training center
- first-aid station and mine rescue room
- construction and operations camp
- mining contractor facilities
- change house
- instrumentation shop.

Camps and Accommodation

The camp for Deep Kerr is assumed to be adjacent to the portal area and within the Deep Kerr surface complex. The camp assumes approximately a 500 person capacity during construction, which supports the day and night shift peak estimates of 250 people for each shift for the initial preproduction period. As construction demand decreases, parts of the camp will be reassigned to operations personnel and operations offices. During operations, it is expected that the Deep Kerr camp will still accommodate about 200 persons for each shift to support full production.

Underground Infrastructure

Underground facilities at Deep Kerr would include on each lift:

- extraction level equipment shop
- haulage level equipment shop

- warehouse
- fuel bay
- explosives magazine
- concrete / shotcrete slickline and loading system (for Lift 2 & Lift 3 only)
- sampling room
- refuges
- lunch rooms
- offices, map and meeting room, training room
- first-aid station
- comfort stations
- service water tanks.

Fire water tanks and fire suppression systems, primarily for crushers, conveyor transfers and shops.

Comments on Deep Kerr Infrastructure Section

Infrastructure requirements have been assessed at the PEA level to support underground mining activities.

24.18.4 Iron Cap Project Infrastructure

Road and Logistics

Personnel, material, and supplies will access the Iron Cap mine through a ramp driven from the Mitchell access ramp (assuming that the Mitchell block cave mine is developed first).

Multiple drifts have been designed that can act as an emergency egress. The primary emergency egress will be the train tunnel or access tunnel, whichever is accessible. If both tunnels are inaccessible, it will be possible to exit the mine through one of the fresh air drifts.

Underground Crushers and Conveyors

The Iron Cap deposit will be able to generate 40,000 t/d of mill feed. The mill feed will be hauled by LHD directly from the drawpoints to a mill feed pass near the center of the footprint and on every extraction drift. To maintain trafficability, a 0.3 m thick reinforced concrete sill will be installed on the extraction level. A rock breaker will be located above each mill-feed pass and two passes will feed one chute on the haulage level below. Fifty-five tonne haul trucks will load from the chutes and haul the mill feed to a gyratory crusher. The crusher will have two dumping points for rear dump articulated underground haul trucks.

The crushed material will be transported by a 730 m long, 1.22 m wide conveyor belt which inclines at 12% to the top of a 65 m high, 2,000 t surge bin located above the Iron Cap-MTT train tunnel. From there it is fed into the main MTT train and transported to the Treaty OPC.

Underground Mine Service Water System

The mine will require 30 m^3/h of process water for the bolters and the development and production drills. In addition, it is estimated that 50 m^3/h of water will be required by the conveyor fire suppression systems. This water will be supplied through the main access ramp from the Mitchell ramp which is fed from the main Mitchell portal area by pumps. The water will be delivered to the working face through steel pipes.

Underground Electrical Reticulation System

The Iron Cap mine will require approximately 5 MWh of electricity at peak operation. The main contributors to this demand are the crushers, ventilation fans and conveyor belts.

The main power will be supplied to the underground from the Mitchell substation through a 25 kV cable hung from the back of the access ramp and MTT train tunnel to create a ring main style system. Each of the main levels will have a 25 kV line which will be stepped down to the required voltage by skid mounted dry-type transformers. Equipment that draws larger loads (e.g., ventilation fans, conveyors and crushers) will be equipped with a permanent transformer.

Waste Storage Facilities

About 2.2 Mt of waste rock will be generated during the initial mine development and ramp up and hauled to surface by truck. Waste rock generated after first mill-feed production will be trucked to one of the crushers and diluted into the mill-feed stream. The majority of the mine development at Iron Cap will be in mineralized rock above NSR cut-off. The quantity below the NSR cut-off is sufficiently small relative to the production of the overall KSM site that, if mixed with the production mill feed, it will have minimal impact on the overall mill-feed grade. Separating the waste from the mill feed stream is not practical for the proposed mine since it would require duplication of material handling infrastructure and significant additional cost.

Water Management

The underground mine dewatering system (refer to 24.16 Mine Dewatering) is discharged into the Mitchell North Pit Wall Depressurization Tunnel via graded drifts that allow for water to drain by gravity. From there, it will flow to the main Mitchell Water Storage Facility. Pumps are not required to dewater Iron Cap.

Surface Infrastructure

The PEA design assumes the support for the Iron Cap operations will be integrated with the overall Project site and in particular, the nearby Mitchell block cave mine. This will include major equipment maintenance and fuel provision.

Surface infrastructure required to support operations will include:

- equipment shop for major repairs
- explosives magazine
- warehouse
- water pump stations
- general mine offices
- control room and training center
- first aid station and mine rescue room
- construction and operations camp
- mining contractor facilities
- change house
- instrumentation shop.

CAMPS AND ACCOMMODATION

The camp for Iron Cap is assumed to be adjacent to the Mitchell portal area and within the Mitchell OPC.

UNDERGROUND INFRASTRUCTURE

Underground facilities at Iron Cap would include:

- equipment shop
- warehouse
- fuel bay
- explosives magazines
- sampling room
- refuge stations (fixed and portable)
- lunch rooms
- offices, map and meeting room, training room
- first-aid station
- water tanks

Fire water tanks and fire suppression systems, primarily for crushers, conveyor transfers and shops.

COMMENTS ON IRON CAP INFRASTRUCTURE

Infrastructure requirements have been assessed at the PEA level to support underground mining activities.

24.18.5 MITCHELL PROJECT INFRASTRUCTURE

ROAD AND LOGISTICS

Personnel, material, and supplies will access the underground through a main access ramp which will be developed from a portal near the Mitchell OPC area at the 820 m elevation.

The Mitchell design has multiple raises and drifts that can act as an emergency egress. The primary emergency egress will be the access tunnel or conveyor tunnel, whichever is accessible. If both tunnels are inaccessible, it will be possible to exit the mine through one of the fresh air shafts; however, the fan in the shaft will have to be shut off because, during normal operations, the wind speed will be too high for personnel entry.

UNDERGROUND CRUSHERS AND CONVEYORS

It is estimated that the Mitchell deposit will be able to generate 60,000 t/d of mill feed. The mineralized material will be hauled from the drawpoint to one of three mill feed passes in the same extraction drift. The mill feed pass will be 4 m in diameter and equipped with a 1 m by 1 m grizzly. These mill feed passes are equipped with a chute that will feed haulage trucks on the haulage level. Fifty-five tonne haul trucks will load from the orepass chutes and haul the mill-feed to one of two gyratory crushers, where it will be crushed.

The crushed material will be transported to surface by a 4.3 km long, 1.22 m wide conveyor belt which inclines at 20% to the top of a 30 m high, 1,000 t surge bin located above the MTT train tunnel. From there it will be transported to the Treaty OPC.

UNDERGROUND MINE SERVICE WATER SYSTEM

It is estimated that the mine will require 30 m^3/h of process water for the bolters and the development and production drills. In addition, it is estimated that 114 m^3/h of water will be required by the conveyor fire suppression systems. This water will be supplied through the main access decline by gravity. The water will be delivered to the working face through steel pipes.

UNDERGROUND ELECTRICAL RETICULATION SYSTEM

It is estimated that the Mitchell mine will require approximately 20 MWh of electricity at peak operation. The main contributors to this total are the crushers, conveyor belts, ventilation fans and mine dewatering pumps.

The mine dewatering system requires an average of 4 MWh with a maximum of 30 MWh during a peak storm event, which is greater than the power requirements of the mine under normal conditions. The strategy during a peak storm event will be to shut down or reduce operations in the underground mine along with other site facilities when the high-powered pumps are required. This will allow power to be diverted from normal operations to power the pumps.

The main power will be supplied to the underground from the Mitchell substation through a 25 kV cable hung from the back of the access ramp and conveyor tunnel to create a ring main style system. Each of the main levels will have a 25 kV line which will be stepped down to the required voltage by skid mounted dry-type transformers. Equipment that draws larger loads (e.g., ventilation fans, conveyors and crushers) will be equipped with a permanent transformer.

WASTE STORAGE FACILITIES

About 4.3 Mt of waste rock will be generated during the initial mine development and ramp up and hauled to surface by truck. Waste rock generated after first mill-feed production will be trucked to one of the crushers and diluted into the mill-feed stream. The majority of the mine development at Mitchell will be in mineralized rock above NSR cut-off. The quantity below the NSR cut-off is small enough relative to the production of the overall KSM site that, if mixed with the production material, will have minimal impact on the overall mill-feed grade. Separating the waste from the mill feed stream is not practical for the proposed mine since it would require duplication of material handling infrastructure and significant additional cost.

WATER MANAGEMENT

The underground mine dewatering system (refer to Section24.16.4 Mine Dewatering) allows for water to be pumped to the HDS WTP and then to the WSF.

SURFACE INFRASTRUCTURE

The PEA design assumes the support for the Mitchell operations will be integrated with the overall Project site. This will include major equipment maintenance and fuel provision.

Surface infrastructure required to support operations will include:

- equipment shop for major repairs
- explosives magazine
- warehouse
- water pump stations
- general mine offices
- control room and training center
- first-aid station and mine rescue room
- construction and operations camp
- mining contractor facilities
- change house
- instrumentation shop.

CAMPS AND ACCOMMODATION

The camp for Mitchell is assumed to be adjacent to the Mitchell portal area and within the Mitchell OPC.

UNDERGROUND INFRASTRUCTURE

Underground facilities at Mitchell would include:

- equipment shop
- warehouse
- fuel bay
- explosives magazines
- sampling room
- refuge stations (fixed and portable)
- lunch rooms
- offices, map and meeting room, training room
- first-aid station
- water tanks.

Fire water tanks and fire suppression systems, primarily for crushers, conveyor transfers and shops.

COMMENTS ON MITCHELL PROJECT INFRASTRUCTURE

Infrastructure requirements have been assessed at the PEA level to support underground mining activities

24.19 MARKET STUDIES AND CONTRACTS

Seabridge engaged NSA to provide an opinion report on marketing inputs for the 2016 PFS and PEA. The information and options in this section come from NSA (2016). All currency amounts used in this section are in US dollars, unless otherwise specified.

24.19.1 COPPER CONCENTRATE

MARKETABILITY

When considering the marketability of copper concentrates, quality and quantity are determining factors. There is considerable variation in the quality of concentrates and the requirements of various smelters do vary; such variation relates to the technical abilities of the smelter and its overall concentrate feed and blend.

Ideally, smelters prefer to blend their feed with approximately 30% copper and similar amounts of iron and sulphur. In the last several years, however, the grade of some of the major high-grade suppliers has been dropping. At the same time, many new suppliers tend to blend copper-gold concentrates with copper content in the low to mid 20% range. Consequently, the market has seen the blend for most smelters drop to a copper level of 27 to 28%. Apart from the level of copper, iron, and sulphur, other key elements in determining concentrate salability include the levels of gold and silver content, as well as any impurities.

Based on the impurity levels projected for the Mitchell and Iron Cap deposits, concentrates from the Project are relatively clean. Depending on the market situation at the time of contract negotiations, penalties will likely be minimal, if at all applicable. Some smelters, such as in Japan, South Korea, and Europe, are expected to have more interest in copper concentrates with high gold content. Preliminary minor element assays of the final bulk concentrates for Kerr indicate concentrations of arsenic and antimony may be near or above typical smelter penalty limits. For the Deep Kerr samples, mercury in the bulk concentrate may also be above smelter penalty limits. This may be mitigated by blending and reviewed in more advanced studies.

SMELTING TERMS

Copper Concentrate Smelting Market

Copper concentrates account for approximately four-fifths of total newly-mined copper production, with the balance of output coming from solvent extraction and electrowinning copper cathode and other copper-bearing by-products.

Concentrate supply started to increase over 2013 and is expected to continue to increase towards the end of this decade as a result of new projects and announced expansions of operational mines.

A significant portion of copper concentrate is processed by integrated smelters—captive plants that are vertically integrated with mines through ownership. However, an increasing annual volume of global copper concentrate is treated by custom smelters that generally are not integrated, although there is, in many cases, investment ownership in mines. Custom smelters have increased their overall smelting market share from 30% in 1980 to approximately 50% today.

Over the last two decades there has been a significant expansion of smelting and refining capacity, particularly in India and China. The Chinese smelting industry has increased imports, as limited domestic mine capacity could not meet demand. This trend has been a key determinant in world concentrate supply/demand balances.

The copper concentrate market has seen significant structural imbalances in the recent years between mine production and smelting capacities. Over the last couple of years there have been significant increases in smelter TCs/RCs. The balance of supply and demand for concentrates is set by the whole of the concentrate output of the mining industry and by the availability of capacity across the smelting industry. The availability of custom concentrates, relative to smelting capacity, should, in theory, be the ultimate determinant of terms for custom treatment of concentrates.

COPPER CONCENTRATES CONTRACTS AND TERMS

The concentrate market is basically split into two types of contracts. First, there are long-term off-take contracts between mines and smelters that reflect, in general, the annual concentrate supply and demand balance. Second, there is spot or short-term business primarily between mines and traders and, on a much smaller scale, between mines and smelters. By its nature, such business is much more volatile and there is considerable variation in spot TCs/RCs, not only annually, but over each year.

Current and Future Terms

NSA suggests that annual benchmark numbers are beginning to reflect a move towards sustainable long-term numbers. NSA believes that the most likely scenario is that ultimately charges have to move up towards a level that is economic for the smelting industry over the long term. The benchmark numbers for the last several years are shown in Table 24.17

Table 24.17 Benchmark Smelting Terms

	2016	2015	2014	2013	2012
Copper Treatment Charges ($/dmt)	97.50	107.00	92.00	70.00	63.50
Copper Refining Charges ($/lb)	0.0975	0.1070	0.0920	0.0700	0.0635

For comparison, the recent spot market of April 2016 indicates sales into the Chinese market where the levels of TCs/RCs were between $90 and $95/dmt of concentrate and $0.090 and $0.095/lb of copper, respectively.

Over approximately 20 years (up to 2005), historical TCs/RCs averaged approximately $77/dmt and $0.077/lb (including approximately $0.01 of participation for these purposes, split between the treatment charge and the copper refining charge) at an average price of $0.93/lb of refined copper.

The general view today does not see price participation materializing in the near future; however, it should not be ignored. Historically, when price participation first became a factor in concentrate negotiations it was only applicable at a price level higher than the price existing at the time of negotiation.

NSA suggests that the annual benchmark terms realized over the last couple of years are likely to be a guide to future levels. With this in mind, and for purposes of this study, the assumption should be a copper treatment charge of $100/dmt, with copper refining charges of $0.10/lb of copper.

TCs/RCs are not the only terms that are used in valuing copper concentrates. Payments and deductions are a matter of negotiation and will vary with many factors, including supply and demand, and custom individual markets.

The following terms are an indication of “standard” long-term smelter charges, including suggested TC/RC terms. Delivery is on the basis of CIF-FO smelter ports (the mine pays all costs up to delivery port and the buyer arranges and pays for cargo discharge).

Payable Metals

Copper Pay 96.5% with a minimum deduction of 1 unit (amount deducted has to equate to a minimum of 1% of the agreed concentrate copper assay).

Silver If over 30 g/dmt pay 90%.

Gold A scale is applicable with some variations of the following:

- less than 1 g/dmt, no payment
- 1 to 3 g/dmt, pay 90%
- 3 to 5 g/dmt, pay 93%
- 5 to 7 g/dmt, pay 95%
- 7 to 10 g/dmt, pay 96.5%
- 10 to 20 g/dmt, pay 97%
- over 20 g/dmt pay 97.5%
- over 30 g/dmt pay 97.75%.

Gold and silver payments may vary between smelter locations. In China, high gold in copper concentrates is not generally desired; relating more to internal pricing issues rather than technical concerns. Technically, the more modern smelting facilities are able to accept payment formulas similar to Japan and South Korea, but for many of the older smelters in North China, this is not the case. In Europe, with grades of over 40 g of gold content, payment of 97.75% with a minimum deduction of 1 g is likely to apply.

Refining Charges

Copper $0.10/lb payable copper

Gold $6.00 to $8.00/oz payable gold

Silver $0.50/oz payable silver

Treatment Charges

Treatment Charge $100.00/dmt CIF-FO main smelter port.

Price Participation

Not applicable at present.

Penalties

Arsenic: $2.50 to $3.00 per 0.1% over 0.1% up to 0.5% arsenic

Antimony: $3.00 to $4.00 per 0.1% over 0.1% antimony

Lead: $2.00 to $3.00 per 1% over 0.5% to 1.0% lead

Zinc:	\$2.00 to \$3.00 per 1% over 2% to 3% zinc
Mercury:	\$2.00 per each 10 ppm over 10 ppm mercury
Bismuth:	\$3.00 to \$5.00 per 0.01% over 0.03 to 0.05% bismuth
Selenium:	\$3.00 to \$5.00 per 0.01% over 0.05% selenium
Tellurium:	\$4.00 to \$5.00 per 0.01% over 0.02% to 0.03% tellurium
Fluorine	\$1.00 to \$2.00 per 100 ppm over 300 ppm fluorine
Chlorine	\$1.00 to \$3.00 per 100 ppm over 300 ppm chlorine.

Furthermore, penalties may also vary from smelter to smelter. It should be noted that for the elements where a percentage range is used, this relates to ranges of penalty thresholds that are negotiated. The penalties noted in this section are generally in line with levels applicable over recent years, but there is a tendency towards higher levels.

Based on the anticipated impurity levels derived from the test results by Tetra Tech (as presented in section 24.17), the concentrates from the Project are relatively clean, and depending on the market situation at the time of contract negotiations, penalties will likely be minimal if at all applicable. As most of the mill feeds will be the blended materials from different deposits and spatial locations, the blend should effectively mitigate penalty elements rising for the mill feed from some limit locations.

Payment

A provisional 90% is paid three to 15 days after vessel arrival in Asia and India. Sales into Europe normally involve later payment, possibly more than 30 days after arrival. The 10% balance is paid when all facts are known.

Other Off-site Costs

Various indirect costs other than smelter charges include:

- losses
- insurance
- supervision, assaying and umpire costs
- marketing
- ocean freight.

24.20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The Project successfully underwent a joint federal provincial harmonized environmental assessment. Both governments conducted the EA cooperatively, in accordance with the principles of the Canada-BC Agreement on Environmental Assessment Cooperation

(Cooperation Agreement 2004). In July 2013, Seabridge submitted the Application/EIS (Rescan 2013) under the BCEAA (2002) in accordance with the approved project AIR to the BCEAO. The Application/EIS was approved and EA Certificate #M14-01 for the Project was issued on July 29, 2014. Federally, the Project was subject to a comprehensive study level of assessment under the CEAA (1992) because the proposed daily mill feed of 130,000 t/d exceeds two thresholds set out in the CEAA (1992) Comprehensive Study List Regulations; specifically, the 4,000 t/d threshold for metal mills, and the 600 t/d production threshold for gold mines. Certain dam structures proposed for the Project also exceed the 10,000,000 m^3/a threshold for water diversions.

The Project was deemed to require a "comprehensive study" in July 2009 and a "notice of commencement of an environment assessment" was submitted to Seabridge. The terms or the scope of assessment was developed and posted by CEAA for public comment in late May 2010. With the CEAA (2010) amendment, the terms of reference was subsequently re-posted for public comment by the CEA Agency in July 2010. The draft KSM Project Comprehensive Study Report was subsequently issued by the CEA Agency in July 2014. The Project Comprehensive Study Report, along with filed public comments from the NLG, other Aboriginal groups, and the public, were considered by the Minister of the Environment when making her final EA decision. The Project received federal approval on December 19, 2014. Further details on the environmental/permitting regime may be found in Section 20.0 of the PFS.

The PEA which incorporates the inclusion of Deep Kerr, involves the same environmental issues as the PFS project which successfully received its environmental approvals in 2014 and is outlined in the PFS. The PEA project, is situated in exactly the same geographical area, has the same environmental characteristics and will involve similar disturbances required to develop infrastructure and the mine as was previously assessed and approved.

Based on this comparison, and upon completion and evaluation of additional technical studies required to support the inclusion of Deep Kerr material into the mine plan and to identify and examine the net environmental benefit of proposed project changes, it is anticipated that the PEA would be approved to operate by the appropriate regulatory authorities. This regulatory approval would be forthcoming only after the appropriate conversations and information sharing has occurred with the Nisga'a Nation and First Nations whom have an interest in the Project.

Potential project changes resulting in an overall net environmental benefit associated with the Project include the following items:

- a reduction in the number and size of proposed open pit mines
- an increased reliance of underground block cave mines with a subsequent smaller surface disturbance
- a reduction of the proposed tonnage of waste rock from 3 Bt to 0.6 Bt
- a potential reduction in the volume of water requiring treatment from the WSF

- a reduction in the overall footprint of the Project.

Additional details on the environmental setting and studies relevant to this PEA may be found in Section 20.0 of the 2016 PFS.

The closure plan costs outlined in the PFS (Section 20.7) have been updated to reflect the proposed mine plan changes outlined in the PEA. As a result of these changes, the overall estimated closure costs including water treatment costs, have been reduced resulting from smaller surface disturbance footprints associated with underground block cave mines and a smaller RSF requiring long-term care and maintenance.

Seabridge has developed long term respectful relationships with the Nisga'a Nation and the four other First Nations groups whom are potentially influenced by the Project, over the past eight years of project development and especially during the recently completed environmental assessment process for the Project (Section 20.0). These relationships, including the requirements contained within the Benefits Agreement signed with the Nisga'a Nation in June 2014 and the Sustainability Agreement negotiated with the Gitanyow Wilps, also signed in June 2014 respectively, would remain in good standing and the agreements would continue to be adhered to by the Project operating company as the proposed mine plan outlined in the PEA is implemented.

24.21 CAPITAL AND OPERATING COSTS

The capital cost estimate is the product of engineering developed during the 2016 PFS and the conceptual level analyses done for the PEA. The unit rates used for the estimates (labour, power, fuel and other consumables) are the same for both studies. Amec Foster Wheeler and Golder used well established internal benchmarks to develop, check capital and operating costs to verify reasonableness of cost levels. These benchmarks are internal to each company.

The capital cost estimates were produced by the consulting firms named in the Table 24.18 with the area of responsibility for each firm identified.

Table 24.18 Capital Cost Estimate Responsibilities by Firm

Capital Cost Area	Firm Responsible
Mine site (open pit infrastructure and capitalised operating costs), process plant, on-site infrastructure, off-site infrastructure, permanent electrical power supply and energy recovery, Underground mine development; underground mining equipment; dewatering; ventilation and services; underground mine infrastructure; underground material handling (Deep Kerr block cave), construction indirects, spares, initial fills, freight and logistics, commissioning and start-up, EPCM, vendor's assistance, owner's cost (except for Mitchell and Iron Cap block cave)	Amec Foster Wheeler
TMF, water management and water treatment	Quantities by KCB with costing by Amec Foster Wheeler

table continues...

Capital Cost Area	Firm Responsible
Underground mine development; underground mining equipment; dewatering; ventilation and services; underground mine infrastructure; underground material handling (Mitchell and Iron Cap block cave)	Golder

24.21.1 CAPITAL COST ESTIMATES

This PEA level estimate includes:

- direct field costs of executing the Project including mining, construction, installation and commissioning of all structures, utilities, materials, and equipment
- indirect costs associated with design, mining, construction and commissioning
- provisions for contingency

See Table 24.19 for summary of the PEA capital cost estimate, summarized by activity.

Table 24.19 PEA Capital Cost Estimate Summary in US$M

Area	Initial	Sustaining	Total
Direct Costs			
Mine Site	1,272	6,827	8,100
Process	1,447	164	1,611
TMF	509	539	1,047
Environmental[2]	15		15
On-site Infrastructure[3]	23	0	23
Off-site Infrastructure	120	11	131
Permanent Electrical Power Supply and Energy Recovery	167	196	364
Total Direct Costs	3,553	7,737	11,290
Indirect Costs			
Construction Indirects	462	688	1,150
Spares	35	19	54
Initial Fills	20	0	20
Freight & Logistics	64	216	281
Commissioning and Start up	6	19	26
EPCM	245	64	309
Vendor's Assistance	15	26	41
Total Indirect Costs	848	1,033	1,880
Owner's Costs	161	0	161
Contingency	927	1,248	2,175
Total Cost	**5,489**	**10,018**	**15,507**

Notes: [1]Sums may not add due to rounding, PST not included in table please see section 24.22 for details
[2]All costs associated with closure cost are included in separate estimate presented below
[3]Most of sustaining on-site infrastructure is included in mine site category

BASIS OF ESTIMATE

This estimate falls under the AACE® Class 5 Estimate classification and its accuracy is expected to be within -30% to +50% of final project cost.

Initial capital cost is defined as all costs associated with development of the operation until first ore in Year 1. It includes Mine Site, processing facility, TMF, environmental infrastructure, on-site/off site infrastructure and power supply.

INDIRECT COSTS AND CONTINGENCY

Allowances were made for indirect costs based on first principle calculations, in-house data, or project parameters.

Contingency has been applied based on a deterministic approach, based on historical studies. The amount of contingency applied differs depending on the cost area of the estimate. Most of the new areas designed for the PEA are using 30% of the direct and indirect costs. Areas where the design is the same as the PFS, a lower contingency factor was applied. The overall contingency factor for the initial capital is estimated at 24% (calculated as a percentage of direct and indirect cost minus capitalised operating costs and environmental bonding).

OWNER (CORPORATE) CAPITAL COSTS

Given the similar nature of the first years of the Project life, the Owner's cost are assumed to be the same as in the PFS.

SUSTAINING CAPITAL

Sustaining capital costs are capital costs incurred after the Project has commenced operations. Two types of capital cost are included in the sustaining capital.

Type one: items required either to replace worn-out or exhausted assets or to support planned growth of the mine that does not increase production capacity. Projects that improve operational efficiency, safety, or decrease costs are usually considered sustaining capital.

Type two: development of new deposits that will maintain the throughput of the operation. Is included in sustaining capital, the development of Deep Kerr, Mitchell and Iron Cap block cave operations. This explains why mine site sustaining capital is higher than initial.

Closure costs are estimated at US$540 million. This number is derived from the PFS estimate and adjusted to reflect smaller RSF size. They include bond reclamation for disturbance, TMF north cell reclamation, sinking fund payments and SUPs.

24.21.2 OPERATING COST ESTIMATES

The operating cost estimate is the product of engineering developed during the 2016 PFS and the conceptual level analyses done for the PEA. The unit rates used for the estimates (labour, power, fuel and other consumables) are the same for both studies.

The PEA operating cost estimates were produced by consulting firms named in Table 24.20 with the area of responsibility for each firm identified.

Table 24.20 Operating Cost Estimate Responsibilities by Firm

Operating Cost Area	Firm Responsible
Open pit mining, underground mining (Deep Kerr), processing, G&A, site services, provincial sales taxes	Amec Foster Wheeler
Tailing storage/handling, water management/treatment, energy recovery	Quantities by KCB, Costing by Amec Foster Wheeler
Underground mining (Mitchell and Iron Cap)	Golder

BASIS OF ESTIMATE

Operating costs detailed in the PEA were derived from a variety of sources including, but not limited to, benchmarking analysis, derivation from first principles, and factoring from costs in the 2016 PFS where possible.

The operating cost estimate is considered to have a level of accuracy of –25% to +35%. Overall assumptions for operating costs:

- Costs are presented in 2016 US dollars, unless stated otherwise. When required, certain costs in this report have been converted using a fixed currency exchange rate of Cdn$1.00 to US$0.80.
- The costs per tonne of material treated (US$/t) provided in this report are the average costs over the LOM.

A summary of the average operating costs onsite for the PEA is estimated at US$11.61/t details are presented in Table 24.21.

Table 24.21 Average Onsite Operating Costs for PEA

Operating Costs On Site	Cost (US$/dmt milled)	Cost (LOM US$ million)
Mining Cost	4.47	10,648
Process Cost	5.19	12,361
G&A	0.86	2,044
Site services	0.41	965
Tailings Storage/Handling	0.12	285
Water Management/Treatment	0.56	1,342
Energy Recovery	(0.10)	(243)
Provincial Sale Tax	0.10	233
Operating Costs On Site/dmt milled	11.61	27,636

The operating costs are LOM average unit costs calculated by total LOM operating costs divided by LOM milled tonnages. The costs exclude mine pre-production costs.

Open Pit Mine Operating Costs

Open pit operating cost were estimated using an Excel® based cost model utilizing first principle build up for the haulage costs and benchmarking from a similar study for all other cost areas. The most significant components of the operating costs are fuel and labour. The average cost for the LOM open pit mining is 2.60 US$ per tonne milled; open pit mining represents 22% of the mill feed source.

Underground Mine Operating Costs

The major components of the underground mining cost are labour and mobile equipment maintenance. Operating costs include the following indirect costs:

- mine management, supervision, Technical Team, Owner's Team
- electrical power
- propane for heating
- camp and catering
- rotational travel
- mine rehab allowance
- fixed plant maintenance
- PPE and workshop allowance.

Underground mining costs were estimated for each block cave operation using similar methodology as used for the capital cost estimate. The LOM cost per tonne milled vary between $5.68 (Iron Cap), $5.15 (Mitchell) and $4.78 (Deep Kerr). The LOM underground mining cost per tonne is estimated at $5.01: underground mining represents 78% of the mill feed source.

Process Operating Costs

The most significant process costs are reagents, steel consumables, and power. Minor cost include labour, maintenance parts and assaying.

Reagent costs were based on testwork and are comparable to the 2016 PFS or in some cases slightly higher. Steel consumables and power costs are derived from the 2016 PFS but with verification to ensure that these are appropriate for the PEA. Labour costs are based on a manpower list to reflect the PEA plant design while utilizing the same unit labour costs as the 2016 PFS. Assaying costs use the same unit rates as the 2016 PFS.

The costs are based on a 170,000 t/d basis. As annual tonnage decreases, the annual cost for reagents, steel consumables and power decreases in step. However for the minor cost components, the annual costs decrease more conservatively to reflect the requirement for minimum manpower in plant operations and other constraints.

Process costs are also specific to each deposit, responding to differences in reagent, power and steel consumption. The annual cost typically reflects the proportional influence of each source of mill feed in the financial model. These typical costs reflect both concentration paths – flotation and cyanidation for each of the mineral deposits.

Tunnel and material handling cost are also included in the process operating cost, they represent US$0.24/dmt of the overall process cost. The LOM average process cost is US$5.19/dmt.

GENERAL AND ADMINISTRATIVE OPERATING COSTS

G&A costs used in the PEA were derived through factoring of costs from other similar operations considering the relevant economies of scale. G&A cost is averaging US$0.86/t milled over the LOM.

INFRASTRUCTURE OPERATING COSTS

Infrastructure operating costs are divided into four categories: site services, tailings storage and handling, water management, and energy recovery. Provincial sales taxes are covered in Section 24.22.

Site services cost are estimated at US$0.41/dmt. They were derived from the previous and current studies and include the following elements:

- personnel – general site services human power
- site 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.

Tailings storage and handling cost are estimated at US$0.12/t milled. They were derived from the previous and current studies.

Water management costs are estimated at US$0.56/dmt and were derived from previous and current studies and adjusted to reflect the PEA requirements. They include the following operating costs:

- mine site water management (pumping)
- HDS WTP
- selenium water treatment

Water treatment plant operating costs were lowered by 15% from PFS costs to reflect expected smaller quantities of water to be treated in the PEA scenario.

24.21.3 COMMENTS ON SECTION 21

The capital costs estimate for this PEA corresponds to a Type 5 study and have an accuracy of -30% to +50%. The operating cost estimate is considered to have a level of accuracy of –25% to +35%.

The Project initial capital cost estimate is US$5.5 billion before PST.

On-site operating costs over the LOM are estimated at US$27.6 billion, and averages US$11.61/dmt milled over the LOM.

24.22 ECONOMIC ANALYSIS

The results of the economic analysis in the PEA represents forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in PEA section of this Report include, but are not limited to, timing and amount of future cash flows from mining operations, forecast production rates and amounts of copper, gold, and silver produced from the KSM mining operation, estimation of the Mineral Resources and the realization of the Mineral Resource estimates within the PEA mine plans, the time required to develop the Project based on the PEA mine design, statements with respect to future price of copper, gold and silver, currency exchange rate between the US dollars and Canadian dollars, assumptions regarding mine dilution and losses, the expected grade of the material delivered to the mill, metallurgical recovery rates, initial capital and sustaining capital costs, as well as mine closure costs and reclamation, timing and conditions of permits required to initiate mine construction, maintain mining activities, and mine closure, and assumptions regarding geotechnical and hydrogeological factors.

The reader is cautioned that the actual mine results of mining operations may vary from what is forecast. Risks to forward-looking information include, but are not limited to, unexpected variations in grade or geological continuity, as well as geotechnical and hydrogeological assumptions that are used in the mine designs. There could be seismic or water management events during the construction, operations, closure, and post-closure periods, that could affect predicted mine production, timing of the production, costs of future production, capital expenditures, future operating costs, permitting time lines, potential delays in the issuance of permits, or changes to existing permits, as well as requirements for additional capital. The plant, equipment or metallurgical or mining processes may fail to operate as anticipated. There may be changes to government regulation of mining operations, environmental issues, permitting requirements, and social risks, or unrecognized environmental, permitting and social risks, closure costs and closure requirements, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

A portion of the Mineral Resources in the mine plans, production schedules, and cash flows include Inferred Mineral Resources, that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA will be realized.

Due to the conceptual nature of the PEA, none of the Mineral Resources in the PEA have been converted to Mineral Reserves and therefore do not have demonstrated economic viability.

24.22.1 METHODOLOGY USED

The Project has been evaluated using a discounted cash flow (DCF) analysis. Cash inflows consist of annual revenue projections for the mine. Cash outflows such as capital, including the six years of pre-production costs, operating costs, taxes, and royalties are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are taken to occur at the end of each period.

To reflect the time value of money, annual net cash flow (NCF) projections are discounted back to the Project valuation date using several discount rates. The discount rate appropriate to a specific project depends on many factors, including the type of commodity; and the level of Project risks, such as market risk, technical risk and political risk. The discounted, present values of the cash flows are summed to arrive at the Project's NPV.

In addition to NPV, IRR and payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. Payback is calculated as the time require to achieve positive cumulative cash flow for the Project.

24.22.2 FINANCIAL MODEL PARAMETERS

BASIS OF ANALYSIS

The financial analysis was based on the Mineral Resources presented in Section 14, the mine and process plan and assumptions detailed in Sections 24.16 and 24.17, the projected infrastructure requirements outlined in Section 24.18, the doré and concentrate marketing assumptions in Section 24.19, the permitting, social and environmental regime discussions in Section 24.20, and the capital and operating cost estimates detailed in Section 24.21.

METAL PRICING

Base Case economic evaluation was undertaken incorporating historical three-year trailing averages for metal prices as of July 31, 2016. Two alternate cases were constructed: (i) a Recent Spot Case incorporating recent spot prices for gold, copper, silver and the US$/Cdn$ exchange rate; and (ii) an Alternate Case that incorporates higher metal prices to demonstrate the Project's sensitivity to rising prices. Metal prices of each scenario are presented in Table 24.22.

Table 24.22 Metal Price Assumptions in PEA

Metal Prices:	Base Case	Recent Spot	Alternate
Gold (US$/oz)	1,230	1,350	1,500
Copper (US$/lb)	2.75	2.2	3
Silver (US$/oz)	17.75	20	25
US$/Cdn$ Exchange Rate	0.8	0.77	0.8

TRANSPORT COSTS

Doré

Doré transport and insurance costs are expected to average US$1.25 per ounce of doré produced.

Copper Concentrate

The transport costs for concentrate including trucking to port, port cost, ocean freight and representation, is expected to be about US$86.19 per wet metric tonne.

WORKING CAPITAL

Working capital cash outflow and inflows are included in the financial model. The calculations are based on the assumptions that accounts payable will be paid within 30 days and accounts receivable within 60 days. The impact of the working capital on NPV 5% is approximately US$148 million.

ROYALTIES

The only royalty included in the financial model is a royalty of 1% of the NSR payable to Barrick Gold Corp., capped at US$3.6 million, with a predetermined buyout option. The full amount of the buyout option is paid in Year 1 in the financial model.

TAXES

Canadian Federal and BC Provincial Income Tax Regime

The federal and BC provincial corporate income taxes are calculated using the currently enacted rates of 15% for federal and 11% for BC. For both federal and provincial income tax purposes, capital expenditures are accumulated in tax pools that can be deducted against mine income at different prescribed rates, depending on the type of capital expenditures.

Historically, pre-production mining expenditures are accumulated in the Canadian Exploration Expense (CEE) pool. The CEE pool is generally amortized at 100%, to the extent of taxable income from the mine. A phase-out rule was enacted in 2013 to phase out the treatment of preproduction mine development expenses as CEE to Canadian Development Expense (CDE) from 2015 to 2017. Effective 2017, all the pre-production mine development expenses are treated as CDE.

In addition to pre-production mine development expenses (that are reclassified as CDE due to the phase-out rule), Canadian resource property acquisition costs and the costs of mine shafts, main haulage ways, and other underground workings are considered CDE and are accumulated in the CDE pool. The KSM Financial Model treats all such expenses as CDE.

Fixed assets acquired for the mine are accumulated in an undepreciated capital cost pool (Class 41) and are generally amortized at 25% on a declining balance basis. Certain fixed assets (acquired after March 20, 2013 and before 2021) may qualify to be accumulated in a Class 41.1 pool that can be amortized at an accelerated rate of up to 100%. However, as a substantive portion of the fixed assets are expected to be acquired post-2020 (after the phase out of the Class 41.1 pool), the KSM Financial Model assumes that the accelerated depreciation will not be available.

The Project is expected to incur costs related to the NTL as described in Section 21. As the capital costs are expected to be incurred on a property that is not owned by the mine, the costs associated with the NTL are treated as eligible capital expenditures for income tax purposes in the KSM Financial Model. Effective January 1, 2017, eligible capital expenditures are treated as a Class 14.1 asset with an amortization rate of 5% per annum. The KSM Financial Model treats all the costs associated with the NTL as a Class 14.1 asset.

BC Mineral Tax Regime

The BC Mineral Tax regime is a two tier tax regime, with a 2% tax and a 13% tax.

The 2% tax is assessed on "net current proceeds", which is defined as gross revenue from the mine less mine operating expenditures. Hedging income and losses, royalties and financing costs are excluded from operating expenditures. The 2% tax is accumulated in a Cumulative Tax Credit Account (CTCA) and is fully creditable against the 13% tax.

All capital expenditures, both mine development costs and fixed asset purchases, and mine operating expenditures are accumulated in the Cumulative Expenditures Account, which is amortized at 100% against the 13% tax.

The 13% tax is assessed on "net revenue", which is defined as gross revenue from the mine less any accumulated Cumulative Expenditures Account balance to the extent of the gross revenue from the mine for the year.

A "new mine allowance" is available in respect of capital costs incurred in respect of new mines. Generally, this allowance provides that 133% of capital expenditures incurred prior to commencement of production are included in the Cumulative Expenditures Account. Under current legislation, the provision for the new mine allowance is scheduled to expire on January 1, 2020.

A notional interest of 125% of the prevailing federal bank rate is calculated annually on any unused Cumulative Expenditures Account and CTCA and is added to the respective balances.

BC Mineral Tax is deductible for federal and provincial income tax purposes.

Provincial Sales Tax

PST was applied to capital and operating cost. They were calculated as part of the estimating process for the capital component. They were estimated on $/t basis for the operating cost. Mining cost were increased by US$0.046/t and process costs by US$0.043/t to account for PST.

FINANCING

The model does not include any costs associated with financing.

INFLATION

There is no adjustment for inflation in the financial model; all cash flows are based on 2016 dollars.

24.22.3 FINANCIAL RESULTS

Table 24.23 summarizes the financial results. The after-tax NPV at a 5% discount rate over the estimated mine life is US$ 3.366 billion. The after-tax IRR is 10.0%. After-tax payback of the initial capital investment is estimated to occur in 6.4 years after the start of production. Table 24.24 shows the cash flow broken out on an annualized basis. Please note that the years presented in the table are for illustrative purposes only and do not necessarily represent the start dates or actual production that would occur in the specified years.

The average life of mine operating cost per ounce of gold recovered is US$ -179 over the life of mine. Operating statistics are included as Table 24.25.

Table 24.23 PEA Financial Analysis Summary

	Base Case	Recent Spot	Alternate
Pre-tax Return			
Cumulative CF (US$ million)	26,345	24,107	38,742
3% NPV (US$ million)	10,850	10,103	16,963
5% NPV (US$ million)	**6,105**	**5,752**	**10,234**
8% NPV (US$ million)	2,400	2,314	4,904
IRR (%)	12.7	12.9	16.9
Payback (Years)	5.6	5.3	3.9
After-tax Returns			
BC Mining Tax (US$ million)	3,388	3,223	5,006
BC Income Tax (US$ million)	2,636	2,515	3,819
Federal Income Tax (US$ million)	3,594	3,430	5,207
Cumulative CF (US$ million)	16,727	15,283	24,710
3% NPV (US$ million)	6,515	6,051	10,456
5% NPV (US$ million)	**3,366**	**3,162**	**6,036**
8% NPV (US$ million)	900	871	2,529
IRR (%)	10.0	10.1	13.4
Payback (Years)	6.4	6.1	4.7

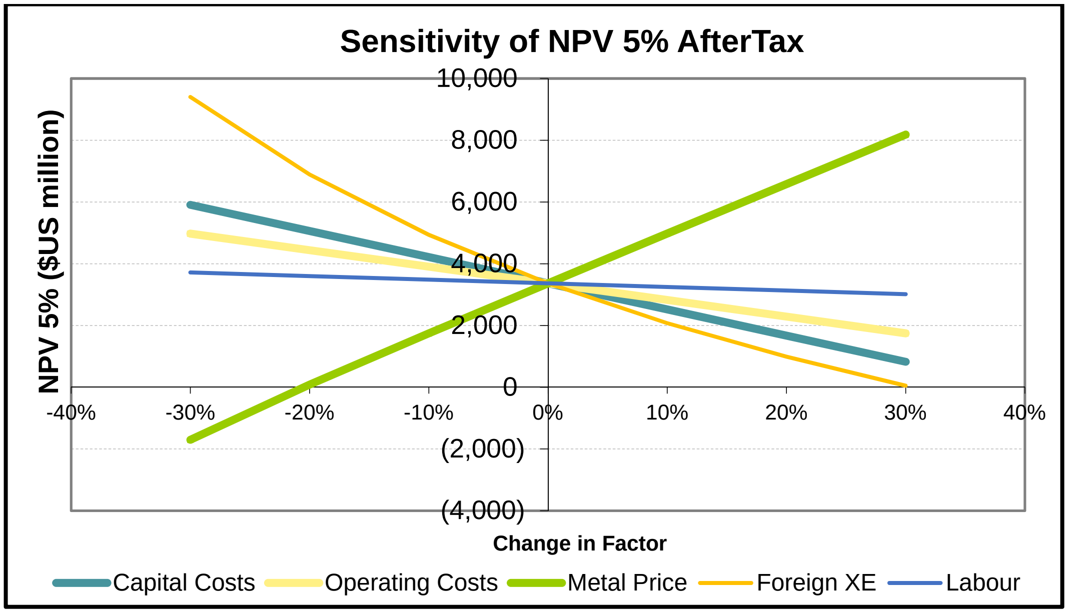
See Figure 24.34 for additional sensitivities to metal prices.

24.22.4 Sensitivity Analysis

Sensitivity analysis was performed on the base case NPV after taxes that examines sensitivity to metal prices, operating costs, capital costs Cdn$/US$ foreign exchange and labour costs.

Sensitivities are shown in Figure 24.34.

Figure 24.34 PEA Sensitivity Analysis



The Project is most sensitive to changes in metal prices and foreign exchange, less sensitive to changes in capital costs, and least sensitive to operating cost and labour costs changes.

Mill feed grade is not presented in the sensitivity graphs as the impact of changes in grade is similar to the impact of changes in metal price.

Table 24.24 PEA Cashflow on an Annualized Basis

CASHFLOW MODEL																														
Project Time line																														
Year (December 31st)			**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**	**2027**	**2028**	**2029**	**2030**	**2031**	**2032**	**2033**	**2034**	**2035**	**2036**	**2037**	**2038**	**2039**	**2040**	**2041**	**2042**	**2043**	**2044**
Project time			**-6**	**-5**	**-4**	**-3**	**-2**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**
	UNITS	**LOM**																												
Metal Prices (Base Case)																														
Gold	US$/oz	1,230.00	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230
Silver	US$/oz	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75
Copper	US$/lbs	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
CAD/USD	CAD/USD	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
Mill Feed Mined																														
Mill Feed From Mine	kmt	2,223,405						726	53,370	59,902	57,996	45,949	61,932	60,704	61,006	27,490	24,704	34,544	43,077	50,508	57,802	61,651	61,890	62,019	62,046	62,050	62,050	62,049	62,011	58,955
Stockpile Reclaim	kmt	157,535								592	4,054	16,101	118	1,346	1,044	34,560	37,346	27,506	18,973	11,542	4,248	106								
Waste Mined																														
Waste Mined	kmt	610,793				11,952	31,046	64,159	120,017	84,152	119,925	114,631	55,754	8,062	1,013	82														
Mill Feed																														
Mill Feed	kmt	2,380,940						726	53,370	60,494	62,050	62,050	62,050	62,050	62,050	62,050	62,050	62,050	62,050	62,050	62,050	61,757	61,890	62,019	62,046	62,050	62,050	62,049	62,011	58,955
Au	g/t	0.522						0.67288	0.70579	0.90229	0.88769	0.85550	0.65200	0.69221	0.73529	0.58086	0.44396	0.43837	0.42409	0.47345	0.52601	0.52635	0.48864	0.48636	0.48564	0.47782	0.46809	0.45879	0.45136	0.46150
Cu	%	0.32						0.23	0.20	0.27	0.27	0.31	0.25	0.25	0.30	0.29	0.28	0.31	0.32	0.33	0.33	0.32	0.31	0.30	0.30	0.31	0.31	0.31	0.30	0.30
Ag	g/t	2.74						7.77	3.08	2.57	2.78	2.49	1.91	3.03	2.95	1.81	1.89	2.28	2.61	2.85	3.18	3.37	3.12	2.97	2.95	2.96	3.02	3.18	3.40	3.76
REVENUES AND DEDUCTIONS																														
Metal Revenues																														
Copper Concentrate	000 US$	67,319,500						20,961	1,393,075	2,112,965	2,144,233	2,186,511	1,619,404	1,735,569	2,036,390	1,682,830	1,442,479	1,566,682	1,598,170	1,728,925	1,856,333	1,844,194	1,721,868	1,696,188	1,691,177	1,687,722	1,679,400	1,656,274	1,633,469	1,570,205
Dore	000 US$	8,199,093						3,944	275,458	353,685	354,519	328,500	302,985	333,771	305,841	219,045	173,482	171,276	165,517	177,882	190,971	189,725	174,622	173,942	176,255	176,123	175,127	174,496	174,993	174,049
Total	**000 US$**	**75,518,593**						**24,904**	**1,668,533**	**2,466,649**	**2,498,752**	**2,515,011**	**1,922,388**	**2,069,340**	**2,342,232**	**1,901,875**	**1,615,961**	**1,737,958**	**1,763,688**	**1,906,806**	**2,047,304**	**2,033,919**	**1,896,490**	**1,870,130**	**1,867,433**	**1,863,845**	**1,854,526**	**1,830,770**	**1,808,462**	**1,744,255**
Treatment Charges																														
Treatment Copper Concentrate	000 US$	2,646,372						571	36,614	56,219	58,242	67,015	51,683	53,685	65,557	61,514	59,832	67,153	69,105	70,925	71,472	69,887	66,343	65,072	65,254	66,357	67,104	66,538	65,722	62,522
Total Treatment Charges	**000 US$**	**2,646,372**						**571**	**36,614**	**56,219**	**58,242**	**67,015**	**51,683**	**53,685**	**65,557**	**61,514**	**59,832**	**67,153**	**69,105**	**70,925**	**71,472**	**69,887**	**66,343**	**65,072**	**65,254**	**66,357**	**67,104**	**66,538**	**65,722**	**62,522**
Transport and Representation																														
Copper Concentrate	000 US$	2,506,521						541	34,680	53,248	55,164	63,473	48,952	50,848	62,093	58,263	56,670	63,604	65,453	67,176	67,695	66,193	62,837	61,633	61,805	62,850	63,557	63,022	62,249	59,218
Dore	000 US$	41,416						59	980	617	810	944	633	1,285	932	531	712	978	1,205	1,268	1,411	1,468	1,275	1,198	1,196	1,187	1,201	1,247	1,317	1,374
Total Transport and Representation	**000 US$**	**2,547,937**						**601**	**35,660**	**53,865**	**55,974**	**64,417**	**49,585**	**52,133**	**63,025**	**58,794**	**57,382**	**64,582**	**66,659**	**68,444**	**69,106**	**67,661**	**64,113**	**62,831**	**63,001**	**64,038**	**64,758**	**64,269**	**63,566**	**60,592**
Insurance Costs																														
Copper Concentrate	000 US$	80,841						25	1,696	2,571	2,607	2,649	1,960	2,102	2,464	2,027	1,728	1,874	1,911	2,072	2,231	2,218	2,069	2,039	2,032	2,027	2,015	1,987	1,960	1,885
Dore	000 US$	10,249						5	344	442	443	411	379	417	382	274	217	214	207	222	239	237	218	217	220	220	219	218	219	218
Total Insurance Costs	**000 US$**	**91,090**						**30**	**2,040**	**3,013**	**3,051**	**3,060**	**2,338**	**2,520**	**2,846**	**2,300**	**1,945**	**2,089**	**2,118**	**2,295**	**2,470**	**2,455**	**2,288**	**2,256**	**2,253**	**2,247**	**2,234**	**2,205**	**2,178**	**2,102**
NSR																														
Copper Concentrate	000 US$	62,085,765						19,823	1,320,085	2,000,927	2,028,220	2,053,374	1,516,809	1,628,934	1,906,277	1,561,027	1,324,248	1,434,051	1,461,700	1,588,751	1,714,934	1,705,896	1,590,618	1,567,444	1,562,086	1,556,488	1,546,723	1,524,726	1,503,538	1,446,581
Dore	000 US$	8,147,428						3,879	274,134	352,626	353,266	327,145	301,973	332,069	304,527	218,240	172,553	170,084	164,105	176,391	189,321	188,020	173,129	172,527	174,839	174,716	173,707	173,031	173,458	172,457
Total	**000 US$**	**70,233,193**						**23,702**	**1,594,219**	**2,353,553**	**2,381,486**	**2,380,519**	**1,818,781**	**1,961,003**	**2,210,804**	**1,779,266**	**1,496,801**	**1,604,135**	**1,625,805**	**1,765,142**	**1,904,255**	**1,893,916**	**1,763,746**	**1,739,971**	**1,736,925**	**1,731,203**	**1,720,430**	**1,697,758**	**1,676,996**	**1,619,038**
OPERATING COSTS ONSITE																														
Mining	000 US$	10,779,053							250,126	247,477	263,898	269,469	234,411	148,587	130,997	115,862	151,775	211,901	241,850	268,419	287,609	299,270	299,168	302,773	304,537	303,062	302,443	301,791	303,026	292,121
Process	000 US$	11,784,690							259,016	292,263	300,730	306,129	301,055	299,424	300,306	305,135	306,358	305,885	304,971	304,135	303,282	301,524	302,174	302,787	302,914	302,933	302,935	302,932	302,749	288,187
G&A	000 US$	2,147,292							40,353	43,358	47,050	44,411	46,560	41,403	42,948	40,556	42,477	45,062	45,029	45,097	46,446	47,759	45,845	46,364	47,726	48,607	49,179	49,682	50,682	49,318
Oversite Services	000 US$	965,246							18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926
Tunnel Transport & Material Handling	000 US$	576,663							12,930	14,656	15,033	15,033	15,033	15,033	15,033	15,033	15,033	15,033	15,033	15,033	15,033	14,962	14,994	15,026	15,032	15,033	15,033	15,033	15,024	14,283
TMF Seepage Water Pumping	000 US$	45,682							896	896	896	896	896	896	896	896	896	896	896	896	896	896	896	896	896	896	896	896	896	896
HDS Water Treatment Plant	000 US$	857,307							16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810
Selenium Water Treatment Plant	000 US$	378,712											8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058
Minesite Water Management - Pumping	000 US$	106,216							2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083
TMF Dam Raising Sands	000 US$	239,034							4,960	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185	6,185
Energy Recovery	000 US$	(242,563)							(2,829)	(2,829)	(2,829)	(2,829)	(2,829)	(2,829)	(2,829)	(2,829)	(2,829)	(5,135)	(5,135)	(5,135)	(5,135)	(4,739)	(4,739)	(4,739)	(4,739)	(4,739)	(4,739)	(5,271)	(5,271)	(5,271)
Total onsite operating cost	**000 US$**	**27,637,331**							**603,272**	**639,826**	**668,782**	**677,114**	**647,188**	**554,575**	**539,413**	**526,716**	**565,772**	**625,703**	**654,705**	**680,507**	**700,192**	**711,733**	**710,399**	**715,168**	**718,428**	**717,854**	**717,808**	**717,125**	**719,169**	**691,596**
OPERATING COSTS OFF SITE																														
Other Royalty	000 US$	3,600							3,600																					
Total off site operating cost	**000 US$**	**3,600**							**3,600**																					
OPERATING PROFIT																														
Operating Profit	**000 US$**	**42,592,262**						**23,702**	**987,348**	**1,713,727**	**1,712,704**	**1,703,406**	**1,171,593**	**1,406,428**	**1,671,391**	**1,252,550**	**931,029**	**978,431**	**971,100**	**1,084,635**	**1,204,063**	**1,182,183**	**1,053,347**	**1,024,803**	**1,018,497**	**1,013,349**	**1,002,622**	**980,633**	**957,827**	**927,442**
BC Mining Tax	000 US$	3,388,213						474	19,819	34,275	34,254	34,068	23,432	28,129	33,428	41,792	54,887	69,028	90,020	113,947	132,025	131,062	118,247	119,333	112,884	89,824	92,427	95,555	96,005	94,852
BC Income Tax	000 US$	2,635,813						2,555			24,701	91,103	47,611	83,843	117,512	74,725	39,448	45,891	47,990	64,501	81,483	83,907	75,361	76,220	78,621	78,308	74,455	71,191	69,204	67,082
Federal Income Tax	000 US$	3,594,290						3,484			33,683	124,231	64,924	114,332	160,244	101,898	53,793	62,578	65,442	87,956	111,114	114,418	102,765	103,936	107,211	106,783	101,530	97,078	94,369	91,476
Taxes	**000 US$**	**9,618,316**						**6,513**	**19,819**	**34,275**	**92,639**	**249,402**	**135,967**	**226,304**	**311,183**	**218,416**	**148,129**	**177,497**	**203,452**	**266,404**	**324,622**	**329,388**	**296,372**	**299,489**	**298,716**	**274,915**	**268,412**	**263,824**	**259,578**	**253,410**
CAPITAL COSTS																														
Initial Capital	000 US$	5,489,356	211,858	525,994	698,854	963,104	1,750,583	1,338,962																						
Sustaining Capital	000 US$	10,017,564							725,159	268,065	335,802	302,601	422,718	325,697	488,351	402,852	502,322	441,738	275,079	205,461	186,080	171,790	141,922	105,492	148,239	318,279	287,921	242,460	216,529	195,288
Provincial Sales Tax (PST)	000 US$	200,519	2,740	6,802	9,037	12,454	22,637	17,314	9,377	3,466	4,342	3,913	5,466	4,212	6,315	5,209	6,496	5,712	3,557	2,657	2,406	2,221	1,835	1,364	1,917	4,116	3,723	3,135	2,800	2,525
Closure Costs	000 US$	539,957	3,960					3,622	27,600									11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069
Total capital costs	**000 US$**	**16,247,397**	**218,557**	**532,796**	**707,891**	**975,558**	**1,773,220**	**1,359,898**	**762,136**	**271,531**	**340,144**	**306,514**	**428,184**	**329,908**	**494,666**	**408,061**	**508,818**	**458,519**	**289,705**	**219,187**	**199,555**	**185,080**	**154,826**	**117,925**	**161,225**	**333,464**	**302,714**	**256,665**	**230,398**	**208,883**
Working Capital																														
Change in working capital	**000 US$**	**0**						**(3,896)**	**(208,583)**	**(121,818)**	**(2,212)**	**844**	**89,881**	**(30,991)**	**(42,309)**	**69,894**	**49,643**	**(12,718)**	**(1,179)**	**(20,784)**	**(21,250)**	**2,648**	**21,288**	**4,300**	**769**	**893**	**1,767**	**3,671**	**3,581**	**7,261**
VALUATION INDICATORS																														
Pre Tax																														
Pre-Tax Cash Flow	000 US$	26,344,865	(218,557)	(532,796)	(707,891)	(975,558)	(1,773,220)	(1,340,092)	16,628	1,320,378	1,370,348	1,397,735	833,290	1,045,529	1,134,416	914,383	471,854	507,194	680,217	844,664	983,258	999,751	919,809	911,178	858,041	680,779	701,676	727,639	731,010	725,821
Cumulative cashflow	000 US$		(218,557)	(751,353)	(1,459,245)	(2,434,803)	(4,208,023)	(5,548,115)	(5,531,487)	(4,211,108)	(2,840,760)	(1,443,025)	(609,735)	435,794	1,570,209	2,484,593	2,956,447	3,463,641	4,143,858	4,988,522	5,971,780	6,971,531	7,891,340	8,802,518	9,660,559	10,341,338	11,043,014	11,770,653	12,501,663	13,227,484
NPV 5%	000 US$	6,104,938																												
Payback period	Years	5.6																												
IRR before tax	%	12.7%																												
After Tax																														
After Tax Cash Flow	000 US$	16,726,549	(218,557)	(532,796)	(707,891)	(975,558)	(1,773,220)	(1,346,605)	(3,191)	1,286,104	1,277,710	1,148,333	697,323	819,225	823,232	695,968	323,725	329,698	476,764	578,260	658,636	670,363	623,436	611,689	559,325	405,864	433,263	463,815	471,432	472,411
Cumulative cashflow	000 US$		(218,557)	(751,353)	(1,459,245)	(2,434,803)	(4,208,023)	(5,554,628)	(5,557,819)	(4,271,715)	(2,994,005)	(1,845,673)	(1,148,350)	(329,125)	494,107	1,190,075	1,513,801	1,843,498	2,320,263	2,898,523	3,557,159	4,227,523	4,850,959	5,462,648	6,021,973	6,427,837	6,861,100	7,324,915	7,796,346	8,268,757
NPV 5%	000 US$	3,366,176																												
Payback period	Years	6.4																												
IRR after tax	%	10.0%																												

2045	2046	2047	2048	2049	2050	2051	2052	2053	2054	2055	2056	2057	2058	2059	2060
23	**24**	**25**	**26**	**27**	**28**	**29**	**30**	**31**	**32**	**33**	**34**	**35**	**36**	**37**	**38**
1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230
17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75
2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
55,474	50,483	47,205	44,423	45,015	46,961	48,590	48,927	47,470	47,450	47,450	47,450	47,450	46,830	41,519	35,059
55,474	50,483	47,205	44,423	45,015	46,961	48,590	48,927	47,470	47,450	47,450	47,450	47,450	46,830	41,519	35,059
0.46967	0.48383	0.49457	0.49552	0.48280	0.45844	0.43567	0.42219	0.42890	0.44677	0.47331	0.47863	0.46774	0.46423	0.47412	0.51267
0.31	0.32	0.34	0.37	0.39	0.39	0.37	0.33	0.31	0.32	0.36	0.39	0.39	0.35	0.29	0.26
3.95	3.43	3.05	3.06	3.03	2.77	2.51	2.30	2.37	2.56	2.74	2.77	2.75	2.71	2.59	2.62
1,490,409	1,409,923	1,377,965	1,354,920	1,397,189	1,423,694	1,397,199	1,291,569	1,207,646	1,255,232	1,394,417	1,479,844	1,454,971	1,328,257	1,053,646	859,409
169,945	161,634	156,573	151,625	151,594	150,649	148,296	145,208	143,942	149,300	156,753	158,443	155,975	153,842	142,384	133,256
1,660,353	**1,571,557**	**1,534,537**	**1,506,544**	**1,548,783**	**1,574,343**	**1,545,495**	**1,436,778**	**1,351,588**	**1,404,531**	**1,551,170**	**1,638,287**	**1,610,946**	**1,482,098**	**1,196,030**	**992,665**
59,128	57,278	57,304	58,056	61,577	64,432	63,599	57,172	51,676	53,255	60,524	66,114	65,418	57,379	41,983	31,389
59,128	**57,278**	**57,304**	**58,056**	**61,577**	**64,432**	**63,599**	**57,172**	**51,676**	**53,255**	**60,524**	**66,114**	**65,418**	**57,379**	**41,983**	**31,389**
56,003	54,251	54,276	54,988	58,323	61,027	60,238	54,151	48,945	50,440	57,326	62,620	61,961	54,347	39,764	29,730
1,358	1,108	954	910	911	879	833	775	773	814	856	853	834	800	674	569
57,361	**55,360**	**55,230**	**55,898**	**59,234**	**61,906**	**61,071**	**54,926**	**49,718**	**51,254**	**58,181**	**63,473**	**62,795**	**55,146**	**40,438**	**30,300**
1,789	1,691	1,651	1,621	1,670	1,699	1,667	1,543	1,445	1,502	1,667	1,767	1,737	1,589	1,265	1,035
212	202	196	190	189	188	185	182	180	187	196	198	195	192	178	167
2,002	**1,893**	**1,847**	**1,811**	**1,859**	**1,887**	**1,852**	**1,725**	**1,625**	**1,689**	**1,863**	**1,965**	**1,932**	**1,781**	**1,443**	**1,202**
1,373,489	1,296,703	1,264,733	1,240,255	1,275,619	1,296,536	1,271,696	1,178,703	1,105,580	1,150,034	1,274,900	1,349,342	1,325,854	1,214,943	970,634	797,255
168,374	160,323	155,424	150,525	150,494	149,582	147,278	144,252	142,989	148,300	155,702	157,392	154,946	152,850	141,532	132,520
1,541,864	**1,457,026**	**1,420,157**	**1,390,780**	**1,426,113**	**1,446,118**	**1,418,974**	**1,322,955**	**1,248,569**	**1,298,334**	**1,430,602**	**1,506,735**	**1,480,800**	**1,367,792**	**1,112,167**	**929,774**
275,149	261,859	254,019	240,850	227,109	227,979	237,074	239,626	231,104	232,242	230,898	231,784	231,387	228,796	207,904	184,334
271,631	248,972	234,478	222,135	225,427	234,722	242,482	244,090	237,144	237,051	237,051	237,051	237,051	234,095	208,784	177,979
51,224	48,494	47,539	48,916	43,216	43,129	42,854	42,757	42,166	41,149	41,069	41,581	41,582	40,894	39,532	38,005
18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926
13,440	12,231	11,436	10,762	10,906	11,377	11,772	11,854	11,501	11,496	11,496	11,496	11,496	11,346	10,059	8,494
896	896	896	896	896	896	896	896	896	896	896	896	896	896	896	896
16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810
8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058
2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083
6,185	6,185	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401
(5,271)	(5,271)	(5,271)	(5,666)	(5,666)	(5,666)	(5,666)	(5,666)	(5,666)	(5,666)	(5,666)	(5,666)	(5,666)	(5,666)	(5,666)	(5,271)
659,131	**619,243**	**592,375**	**567,170**	**551,165**	**561,714**	**578,689**	**582,834**	**566,422**	**566,445**	**565,021**	**566,419**	**566,023**	**559,636**	**510,786**	**453,714**
882,733	**837,783**	**827,782**	**823,610**	**874,948**	**884,403**	**840,285**	**740,121**	**682,147**	**731,889**	**865,580**	**940,315**	**914,778**	**808,156**	**601,381**	**476,060**
64,196	80,432	82,126	80,158	93,787	92,058	97,127	84,789	78,286	87,418	103,752	112,517	107,936	96,752	67,130	49,248
62,422	55,519	55,970	56,949	63,050	65,654	62,847	55,874	52,392	58,961	73,404	81,566	79,661	69,824	50,538	38,440
85,121	75,708	76,323	77,658	85,977	89,528	85,700	76,191	71,444	80,401	100,096	111,226	108,629	95,214	68,916	52,418
211,739	**211,659**	**214,419**	**214,766**	**242,814**	**247,239**	**245,673**	**216,854**	**202,122**	**226,780**	**277,253**	**305,309**	**296,226**	**261,790**	**186,584**	**140,106**
383,951	216,278	193,538	204,365	151,554	174,018	91,968	86,773	78,929	58,687	66,624	73,844	83,424	63,094	83,912	95,990
4,965	2,797	2,503	2,643	1,960	2,250	1,189	1,122	1,021	759	862	955	1,079	816	1,085	1,241
11,069	11,069	11,069	11,069	11,069	11,069	11,069	31,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069
399,985	**230,144**	**207,110**	**218,077**	**164,582**	**187,337**	**104,226**	**118,964**	**91,018**	**70,515**	**78,554**	**85,868**	**95,572**	**74,979**	**96,066**	**108,300**
10,018	**10,667**	**3,852**	**2,757**	**(7,124)**	**(2,421)**	**5,857**	**16,125**	**10,879**	**(8,179)**	**(21,860)**	**(12,400)**	**4,231**	**18,052**	**38,005**	**25,292**
492,766	618,307	624,525	608,291	703,242	694,645	741,916	637,282	602,007	653,195	765,166	842,047	823,436	751,229	543,321	393,051
13,720,250	14,338,557	14,963,081	15,571,372	16,274,614	16,969,259	17,711,175	18,348,458	18,950,465	19,603,660	20,368,826	21,210,874	22,034,310	22,785,539	23,328,859	23,721,910
281,027	406,648	410,105	393,524	460,428	447,405	496,243	420,428	399,885	426,416	487,913	536,738	527,210	489,439	356,736	252,945
8,549,784	8,956,432	9,366,537	9,760,062	10,220,490	10,667,895	11,164,139	11,584,567	11,984,452	12,410,868	12,898,781	13,435,519	13,962,729	14,452,168	14,808,904	15,061,850

2061	2062	2063	2064	2065	2066	2067	2068	2069	2070	2071	2072	2073	2074	2075	2076
39	**40**	**41**	**42**	**43**	**44**	**45**	**46**	**47**	**48**	**49**	**50**	**51**	**52**	**53**	**54**
1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230	1,230
17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75	17.75
2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75	2.75
0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80
29,917	26,113	28,407	26,139	23,539	22,531	25,550	25,550	24,526	18,493	14,120	9,141	7,194			
29,917	26,113	28,407	26,139	23,539	22,531	25,550	25,550	24,526	18,493	14,120	9,141	7,194			
0.55584	0.59309	0.54405	0.49235	0.43123	0.36811	0.35226	0.35526	0.32784	0.32827	0.30166	0.26312	0.24847			
0.24	0.21	0.19	0.25	0.34	0.45	0.51	0.50	0.46	0.47	0.44	0.39	0.36			
2.69	2.73	2.45	2.17	2.17	2.14	2.12	1.99	1.72	1.57	1.37	1.15	0.90			
720,498	608,541	598,749	598,282	635,082	715,513	893,938	882,046	770,567	589,287	419,208	236,263	170,216			
126,564	121,597	123,875	102,239	73,207	50,489	50,611	50,727	44,688	33,519	23,431	13,199	9,320			
847,062	**730,137**	**722,625**	**700,521**	**708,289**	**766,002**	**944,549**	**932,773**	**815,255**	**622,806**	**442,639**	**249,462**	**179,537**			
24,121	18,478	18,306	22,362	28,300	36,013	46,917	46,013	40,235	30,981	22,261	12,625	9,091			
24,121	**18,478**	**18,306**	**22,362**	**28,300**	**36,013**	**46,917**	**46,013**	**40,235**	**30,981**	**22,261**	**12,625**	**9,091**			
22,846	17,501	17,338	21,180	26,805	34,109	44,438	43,581	38,109	29,344	21,084	11,958	8,611			
496	447	450	368	322	297	334	316	264	184	124	68	18			
23,342	**17,948**	**17,789**	**21,548**	**27,127**	**34,407**	**44,772**	**43,898**	**38,372**	**29,528**	**21,208**	**12,026**	**8,629**			
870	738	726	720	758	849	1,059	1,045	913	698	496	280	201			
158	152	155	128	92	63	63	63	56	42	29	16	12			
1,029	**890**	**880**	**848**	**850**	**912**	**1,122**	**1,108**	**969**	**740**	**525**	**296**	**213**			
672,661	571,824	562,380	554,020	579,218	644,541	801,524	791,406	691,310	528,265	375,367	211,400	152,312			
125,910	120,998	123,270	101,743	72,793	50,129	50,214	50,348	44,369	33,293	23,278	13,115	9,291			
798,571	**692,822**	**685,650**	**655,763**	**652,011**	**694,670**	**851,738**	**841,754**	**735,679**	**561,557**	**398,645**	**224,515**	**161,603**			
166,112	151,053	164,219	148,980	126,900	116,363	121,323	122,772	115,646	91,358	75,364	57,246	49,028			
147,721	129,575	140,517	131,231	120,824	117,604	132,512	132,512	127,640	98,921	78,108	54,410	45,147			
38,958	40,145	36,069	36,644	32,991	33,483	34,891	34,903	34,605	32,907	31,687	30,268	29,728			
18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926	18,926			
7,248	6,327	6,882	6,333	5,703	5,459	6,190	6,190	5,942	4,480	3,421	2,215	1,743			
896	896	896	896	896	896	896	896	896	896	896	896	896			
16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810	16,810			
8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058	8,058			
2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083	2,083			
3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401	3,401			
(5,271)	(5,271)	(5,271)	(5,271)	(5,271)	(5,271)	(5,271)	(5,271)	(5,271)	(5,271)	(5,271)	(5,271)				
404,942	**372,002**	**392,590**	**368,090**	**331,320**	**317,811**	**339,819**	**341,279**	**328,736**	**272,569**	**233,482**	**189,041**	**175,820**			
393,629	**320,820**	**293,060**	**287,673**	**320,691**	**376,859**	**511,919**	**500,475**	**406,943**	**288,989**	**165,162**	**35,474**	**(14,216)**			
24,054	6,416	14,236	11,913	31,823	41,933	57,960	60,741	47,511	33,589	17,681	2,879				
29,461	19,140	13,586	12,477	15,804	23,675	38,739	39,223	31,977	21,862	11,084					
40,174	26,100	18,526	17,014	21,551	32,285	52,826	53,486	43,605	29,812	15,114					
93,689	**51,656**	**46,347**	**41,403**	**69,178**	**97,892**	**149,525**	**153,450**	**123,093**	**85,263**	**43,879**	**2,879**				
205,935	270,463	178,724	193,533	74,929	53,608	65,231	32,811	40,947	30,220	28,782	13,158	11,480	949		
2,663	3,497	2,311	2,503	969	693	844	424	529	391	372	170	148	12		
11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	11,069	(15,252)	15,400	15,400	15,400
219,667	**285,029**	**192,104**	**207,104**	**86,966**	**65,370**	**77,144**	**44,305**	**52,546**	**41,680**	**40,223**	**24,397**	**(3,623)**	**16,361**	**15,400**	**15,400**
17,559	**14,676**	**2,871**	**2,899**	**(2,405)**	**(8,123)**	**(24,011)**	**1,761**	**16,406**	**24,006**	**23,568**	**24,971**	**9,255**	**12,114**		
191,521	50,467	103,827	83,468	231,319	303,367	410,765	457,931	370,803	271,315	148,507	36,048	(1,338)	(4,247)	(15,400)	(15,400)
23,913,432	23,963,899	24,067,726	24,151,194	24,382,513	24,685,880	25,096,645	25,554,576	25,925,380	26,196,695	26,345,202	26,381,250	26,379,912	26,375,665	26,360,265	26,344,865
97,833	(1,189)	57,480	42,065	162,141	205,474	261,240	304,481	247,711	186,052	104,628	33,169	(1,338)	(4,247)	(15,400)	(15,400)
15,159,682	15,158,493	15,215,973	15,258,038	15,420,179	15,625,653	15,886,893	16,191,374	16,439,085	16,625,137	16,729,765	16,762,934	16,761,596	16,757,349	16,741,949	16,726,549

Table 24.25 Average Cost per Ounce of Gold Recovered

Ares	Cost (US$/oz)
Operating Costs On Site	
Mining Cost	356.64
Process Cost	389.91
G&A	71.05
Others	96.82
Operating Costs On Site/oz Au Recovered	914.41
Operating Costs Off site	
Total Treatment Charges	87.56
Total Refining Charges	54.57
Total Transport and Representation	84.30
Total Insurance Costs	3.01
Royalties	0.12
Operating Costs Off Site/oz Au Recovered	317.61
Credit	
Silver	(82.80)
Copper	(1,328.42)
Total Credit	(1,411.22)
Operating Cost/oz Recovered Net of By-products	(179.21)

25.0 INTERPRETATIONS AND CONCLUSIONS

25.1 INTRODUCTION

This NI 43-101 Technical Report presents a summary of the results of two separate studies for two separate mine development options for the Project. The first is at a PFS level, which is an update of the 2012 PFS (Tetra Tech 2012). The other is at a PEA level, which evaluates a different approach to the Project by emphasizing low-cost block cave mining and reducing the number and size of the open pits, which significantly reduces the surface disturbances in the re-designed project. The PEA assesses the potential impacts of incorporating higher grade Inferred Mineral Resources delineated at Deep Kerr and Iron Cap Lower Zone into the mine design, and increasing the annual average maximum mill throughput from 130,000 t/d envisioned in the 2016 PFS to 170,000 t/d in the PEA.

The results of the 2016 PFS update remain valid and represent a viable option for developing the Project, with the PEA assessing an alternative development option at a conceptual level.

The 2012 PFS was used as the basis for submitting the Application/EIS (Rescan 2013). The Project received its EA approvals from both the provincial and federal governments in July and December of 2014, respectively, including numerous provincial permits covering the first years of construction. Those reviews and subsequent decisions concluded that the Project would not result in significant adverse effects to the environment, identifying KSM as a responsible project.

The 2016 PFS is based on Mineral Reserves that stem from a combination of Mineral Resources that have been updated since the 2012 PFS (Kerr and Iron Cap deposits) and Mineral Resources that remain the same as those used in the 2012 PFS (Sulphurets and Mitchell). Appreciable drilling was completed at both the Kerr and Iron Cap deposits since the 2012 PFS was completed that necessitated updating their Mineral Resources.

The 2016 PFS update maintains the Project scope as a large-tonnage open pit and underground block cave mining operation, at a nominal rate of 130,000 t/d of ore fed to a flotation mill capable of producing a copper/gold/silver concentrate for transport by truck to the nearby deep-water sea port at Stewart, BC. A gold-silver doré, and a separate molybdenum concentrate, will also be produced at the processing facility.

25.2 2016 KSM Mineral Reserves and Resources

25.2.1 Mineral Resources

Significant drilling and other exploration activities have been conducted in the KSM district since the 1960s. A number of major mining companies completed various exploration programs prior to Seabridge's entry into the district in 2000. Since then, Seabridge has drilled the Kerr, Deep Kerr, Sulphurets, Mitchell, Iron Cap, and Lower Iron Cap deposits, testing both the limits and geometry of the extensively altered and mineralized systems that appear to be centered on hypabyssal, early-Jurassic intrusions that are located adjacent to regional thrust faults. Seabridge's geological staff developed geological models for each of the mineralized systems. Those geological models were used to create grade models that were used to tabulate Mineral Resources for the KSM Project. The KSM Mineral Resources are constrained by conceptual open pit shapes for material that could be potentially mined from surface, and conceptual block cave shapes for material that could be potentially mined using underground methods. All of the KSM Mineral Reserves are contained within the Mineral Resources described in Section 14.4.

25.2.2 Mineral Reserves

The 2016 KSM PFS update is based on the updated Mineral Resource model described in Section 14.0, which includes the updated geology and assay information within the revised Mineral Resource model, as well as updated metallurgy and cost information developed in this PFS. Mineral Reserves are based on NSR values that were calculated in the block model using US$1,200.00/oz of gold, US$2.70/lb of copper, US$17.50/oz of silver, and US$9.70/lb of molybdenum, varying process recoveries for the different mining areas, and applicable off-site charges. The NSR values were used as a dynamic cut-off grade for defining ore and waste in the open pit, with a minimum of Cdn$9.00/t. Mining loss and dilution parameters for the open pits and underground mine plans are applied as described in Section 15.0. The estimated Proven and Probable Mineral Reserves as of July 31, 2016 are 38.8 Moz of gold and 10.2 Blb of copper (2.2 Bt at an average grade of 0.55 g/t gold and 0.21% copper per tonne) are slightly above the 2012 KSM PFS estimates.

The methods used in this estimate comply with CIM standards, with reasonable engineering practices for a PFS-level study, and economic estimates based on the technical and economic parameters stated in this report. As such, the Mineral Reserves stated in this report are subject to future metal prices and cost inputs, as well as the results of the future studies stated in Section 25.10 (Project Risks) and Section 26.0 (Recommendations) of this report. This Mineral Reserve is reliable to a PFS level of estimate.

25.3 Open Pit Mining

The open pit mine plan establishes the economic mining limits of the Mitchell, Sulphurets, and Kerr Mineral Resource areas using large-tonnage mining methods capable of providing mill feed at a nominal rate of 130,000 t/d when the open pit is supplying all ore. The LOM mine plan accommodates the local adverse conditions comprising snow, cold, remoteness, and steep terrain. Waste and water management designs are incorporated into the mine plan, as specified in the current site plans, as reviewed and approved in the Application/EIS review process completed in 2014 (Rescan 2013). The chosen mine equipment is well known and suitable for the expected operating conditions, and the productivity assumptions are reasonable and achievable. The resultant unit mining costs are comparable when benchmarked against other similar operating mines when considering the site operating conditions. Given the stated design parameters and assumptions, the open pit mine plan would achieve the forecast production schedule and the annual levels and costs within the expected range of accuracy of the PFS estimate.

25.4 UNDERGROUND MINING

The underground block cave mining plans for the Mitchell deposit, beneath the previously mined Mitchell open pit, and for the Iron Cap deposit were developed from assessments of the economic shut-off at the drawpoints using GEOVIA's PCBC™ software. These assessments incorporate dilution associated with sub-economic and waste rock that mixes with the mineralized rock above shut-off value as it is drawn down and mucked at the drawpoint. The GEOVIA PCBC™ assessments were based on industry standard approaches of developing production and grade schedules, and incorporated benchmarked production controls for ramp-up rates, maximum draw rates, and drawpoint construction rates. The size of the block cave footprints on the production levels at Mitchell and Iron Cap are much larger than the minimum estimated area required to initiate and propagate the cave to surface, and so there are no concerns about these deposits caving.

The drawpoint layout and spacing were based on estimates of fragmentation and cave mining experience in maintaining favorable interaction between adjacent draw columns that control premature reporting of waste dilution at the drawpoints. The chosen mucking equipment, the haulage distances from drawpoints to ore passes, the handing of oversize muck by secondary blasting, and the design of the ore passes and grizzly system allow for production rates of 55,000 t/d at Mitchell and 40,000 t/d at Iron Cap. The production ramp-up periods for these operations are six years and four years, respectively, which are within the ranges that have been achieved at other block cave operations. Production from Mitchell, and subsequently from Iron Cap, allow the total KSM Project production to be maintained at 130,000 t/d from Year 23 to 35 as production from the open pits decrease, and then dominate mill feed for the final third of the LOM.

Based on the various factors discussed above, the block cave mine plan is expected to achieve the forecasted production schedule and the annual levels and costs within the expected range of accuracy of the 2016 KSM PFS update estimate.

25.5 MTT TRANSPORTATION SYSTEM

Transportation of ore, freight, and personnel through the MTT will be achievable with a two tunnel automated train transport system, at an average rate of 130,000 t/d and a peak capacity of 10,000 t/h. The required deliveries of freight, including bulk transport of fuel and lime, mine site consumables, as well as personnel movement for periodic crew changes and daily requirements between the Mitchell and Treaty project areas are also achievable using the tunnel and rail infrastructure with appropriate traffic control management systems.

The system is scalable and flexible, as it has the ability to add or remove train sets to meet higher or lower throughput requirements, and components can be taken out of operation for maintenance without compromising total system operation.

This scale of underground tunnel transport via trains has proven effective in other mines globally, utilizing equipment and infrastructure specified within this PFS. The planned train system operations can be reasonably accomplished at this PFS's estimated productivities and costs.

25.6 TUNNELLING

The conventional drill and blast methodology for excavating infrastructure and water tunnels is a valid basis for the scheduling and costing of the long tunnels required in this PFS. Using a twinned tunnel for the MTT provides advantages in construction with three sets of advancing twin headings, enabling the use of one tunnel at each heading as a fresh airway, and the other tunnel as a return airway, which has a significant impact on ventilation and advance rates for long tunnels. Other infrastructure and water tunnels would be excavated with a single advancing heading from each portal, a well proven method in the mining industry.

Advance rates for the MTT have been determined by two contractors using detailed cycle time calculations for the varying ground conditions. The contractor-developed advance rates and costs have been adapted for the other tunnel excavations. The contractor's estimates have also been benchmarked, indicating the estimates are within the accuracy of a PFS. The License of Occupation for the MTT route was issued by the BC Government in September 2014.

25.7 METALLURGY AND PROCESS

Several wide-ranging metallurgical test programs were conducted between 2007 and 2016 to assess the metallurgical responses of the mineral samples from the KSM deposits, especially the 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 Plant is designed based on the flowsheet developed from the testwork results. The proposed flotation process is projected to produce a copper-gold concentrate containing approximately 25% copper. Copper and gold flotation recoveries will vary with changes in head grade and mineralogy. The average copper and gold recoveries to the concentrate are projected to be 81.6% and 55.3%, respectively, for the LOM mill feed containing 0.55 g/t gold and 0.21% copper. As projected from the testwork, the cyanidation circuit will increase the overall gold recovery to a range of 60 to 79%, depending on gold and copper head grades. Silver recovery from the flotation and leaching circuits is expected to be 62.7% 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 flowsheet proposed for the Project is conventional and has been widely used in processing porphyry copper-gold ores. The equipment type and sizing selected for the Project are common in other mining projects.

In general, the copper and molybdenum concentrates produced are anticipated to be acceptable by most of the copper and molybdenum smelters. On average, the impurity contents in the copper and molybdenum concentrates should be lower than the penalty thresholds set by most of the smelters.

25.8 DAM STRUCTURES

25.8.1 TAILING MANAGEMENT FACILITY

The TMF will be a conventional storage facility with three main cells, using a starter dam construction with ongoing centerline dam raises using cyclone sand. The TMF layout and sequencing allows for staged construction of the facility, which optimizes both start-up and operating costs, while managing geotechnical and environmental risks. Using a three-cell system provides advantages for storage of CIL tailing, and allows progressive, staged closure of the facility to mitigate environmental and closure risks. The cell system reduces TMF discharge of excess water and also allows for progressive development of water management facilities.

A BAT (KCB 2016) assessment, completed at the same time as the 2016 KSM PFS update, shows that the current TMF design is the most appropriate to meet environmental, operability, and geotechnical criteria. Dry stack options were either not the most appropriate or did not meet these criteria.

Key conclusions include:

- TMF starter dams safely store 18 to 24 months of tailing under the range of start-up assumptions assessed as established in design criteria presented within the TMF Design Addendum Report (Appendix H4). This will allow the Project flexibility during start up, with a minimum of one winter season and potentially two winter seasons accommodated.
- TMF dam designs (from starter dams through closure) are stable under static and pseudo-static conditions, as designed, and these designs meet all applicable regulatory criteria.

25.8.2 WATER STORAGE FACILITY

The WSF provides environmental containment of runoff water for the Mine Site. The facility includes a rock fill-asphalt core WSD to collect contact water from the Mine Site for treatment at the HDS WTP. The facility is capable of storing inflows from extreme events and is designed to minimize seepage. The WSD will be built to full height before start up to provide 50 Mm^3 of storage for a 200-year wet year runoff from the Mine Site catchments.

Key conclusions include:

- The rock fill-asphalt core WSD design is confirmed to be the preferred structure type to meet the Project's environmental, durability, and geotechnical design criteria. A value engineering study (KCB 2012) showed that this design has both low-seepage rates, as well as constructability and cost advantages over other types of dam structures analyzed for this location and purpose.
- A dam construction contractor provided improved reliability estimates of structure cost and construction schedule duration for completion of the WSD.

25.9 PFS STUDY ECONOMICS

Tetra Tech prepared an economic evaluation for the KSM 2016 PFS update based on a pre-tax financial model. The tax component of the model was prepared and reviewed by other consultants (please see Section 22.0 for further details). Based on this tax analysis, Tetra Tech prepared the post-tax economic evaluation of the KSM Project.

For the 53-year LOM and 2.198 Bt Mineral Reserve, Table 25.1 summarizes the results of the base case as well as three additional cases.

Based on the economic results of Table 25.1, it can be concluded that:

- The Project has a short payback period compared to the long mine life.
- The Project has low cash and total costs per ounce of gold produced net of by-product credits.
- Lower revenue caused by metals prices in this 2016 KSM PFS update, as compared to the 2012 KSM PFS, are partially offset by a lower exchange rate.
- The Project costs include capital cost commitments resulting from the EA review process and thus represent a realistic cost basis moving forward as compared to those projects which have yet to complete an environmental review.
- Realistic closure costs including long water treatment costs are included in the KSM economic evaluation.
- The KSM Project, based on study results herein, is considered an economic project, and thus merits additional study in the next design phase.

Table 25.1 Summary of Major Pre- and Post-tax Results by Metal Price Scenario

	Unit	Base Case	2016 Design Case	Recent Spot Case	Alternate Case
Metal Price					
Gold	US$/oz	1,230.00	1,200.00	1,350.00	1,500.00
Copper	US$/lb	2.75	2.70	2.20	3.00
Silver	US$/oz	17.75	17.50	20.00	25.00
Molybdenum	US$/lb	8.49	9.70	7.00	10.00
Exchange Rate	US:Cdn	0.80	0.83	0.77	0.80
Pre-tax Results					
NPV (at 0%)	US$ million	15,933	13,727	16,101	26,319
NPV (at 3%)	US$ million	6,217	5,128	6,461	11,138
NPV (at 5%)	US$ million	3,263	2,510	3,507	6,541
NPV (at 8%)	US$ million	960	475	1,175	2,928
IRR	%	10.4	9.2	11.1	14.6
Payback	years	6.0	6.5	5.6	4.1
Cash Cost/oz Au	US$/oz	277	311	404	183
Total Cost/oz Au	US$/oz	673	720	787	580
Post-tax Results					
NPV (at 0%)	US$ million	9,983	8,537	10,109	16,721
NPV (at 3%)	US$ million	3,513	2,789	3,691	6,696
NPV (at 5%)	US$ million	1,539	1,028	1,718	3,663
NPV (at 8%)	US$ million	-2	-343	161	1,282
IRR	%	8.0	7.0	8.5	11.4
Payback	years	6.8	7.4	6.4	4.9

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.

25.10 PRELIMINARY ECONOMIC ASSESSMENT CONCLUSIONS

The PEA mining study took a different approach to the 2016 PFS. The PEA mine plan was carried out with the aim of reducing the amount of waste rock produced in the open pits with the mill feed drawing more on the underground resources. The PEA envisages a combined open pit/underground block cave mining operation that is planned to operate for 51 years. The mine production plan starts in lower-cost open pit areas using conventional large scale equipment before transitioning into block cave underground bulk mining later in the mine life. Starter pits have been selected in higher-grade areas and cut-off grade strategy optimizes revenues to minimize the payback duration. Smaller pits were designed for the Mitchell and Sulphurets deposits as well as mining the Kerr deposit solely by underground mining methods. This approach substantially shrinks the Project's footprint.

The Mitchell open pit and Deep Kerr underground mines will be the main source of mill feed, contributing approximately 83% of the total plant feed over the LOM, supplemented by the Sulphurets open pit and Iron Cap underground mine production.

Over the entire 51-year mine life, mineralized material would be fed to a copper and gold extraction mill. The proposed plant for the PEA mine design will have a nameplate process rate of 170,000 t/d. The major design consideration in the Process Plant equipment sizing and layout for the PEA was the use of the largest equipment sizing available in order to minimize pumping and piping requirements, process building footprint, and capital costs. Re-design of the facilities was limited to optimizing the layout provided by the use of the larger equipment in the PEA relative to the 2016 PFS. Estimated unit operating costs are 6% lower than the 2016 PFS, primarily due to reduction in process and G&A cost associated with higher throughput.

The PEA offers a viable option for development of the Project and reduces a number of project risks. By including Deep Kerr, annual average maximum throughput of 130,000 t/d envisioned in the 2016 PFS could be increased to 170,000 t/d in the PEA without significant redesign of facilities. Increased throughput increases the metal production, reduces payback periods, and improves estimated projected IRRs and NPVs. The PEA mine plans in total would reduce the amount of waste rock by 81% (approximately 2.4 Bt) compared to the PFS, substantially shrinking the Project's foot print and its environmental impact, and reducing water treatment costs.

25.11 PREFEASIBILITY STUDY PROJECT RISKS

There are risks that could affect the economic viability of the KSM Project. Many of these risks are based on the current extents of sufficiently detailed information and engineering in specific project areas. These risks can be further managed as more drilling, sampling, testing, design, and engineering are completed in the next study stage.

However, several risks have been reduced through activities completed by Seabridge and its team during the period from 2012 to 2016. Specifically, the permitting risk has been addressed substantially with the receipt of environmental approvals granted by both the federal and provincial governments in 2014, and the granting of the early-stage construction permits for the Project, including the permit covering the MTT route. It can be effectively stated that Seabridge earned "the social license" for the KSM Project through the successful completion of the environmental review process with the support of the nearby communities through the submission of letters of support and Aboriginal groups support. Aboriginal support is evidenced by the signed "Benefits Agreement between the Nisga'a Nation and Seabridge" (negotiated in 2014) and by the signed "Sustainability Agreement" between Seabridge and the Gitanynow Wilps, also negotiated in 2014.

The most significant unknowns associated with the Project are related to the extent of available geotechnical data for the tunnels. As several tunnels are on the critical path for construction, potential delays in construction could occur if unforeseen rock mass conditions or groundwater inflows are encountered that cause durations in excess of those anticipated in the preliminary construction schedule.

These risks are common to most mining projects, many of which can be mitigated with adequate engineering, planning, and pro-active management. Some external risks, such as metal prices, exchange rates, and government legislation are beyond the control of the developer and operator and are difficult to anticipate and mitigate, although in some instances measures for risk reduction have already been included in conservative design such as in selection of economic mining limits. Risk reduction measures are also included in tunnel and access road design, and through the inclusion of early scheduling for items potentially subject to risk of delay. The means to address risk for a project the size of KSM moving forward is to establish a formal risk management program during advanced study phases that continues through development and into mine operation. The KSM Project Team will systematically review risks and opportunities during project development and construction, and take appropriate action to minimize the impact on overall costs and scheduling.

25.11.1 OPEN PIT

The Mitchell pit is a very large open pit with world-class slope heights. The Sulphurets and Kerr pits have slope heights typical of other open pit metal mines in BC and other parts of the world. The main mining risks for the open pits relate to slope stability, safety, and production delays. Throughout the planning, design, and engineering of the open pits, the KSM Project Team has identified hazards and developed mitigation plans to reduce and manage risks related to the open pit slopes. Previous studies included a comprehensive risk analysis of the open pit mine design and operating issues. Geotechnical and hydrogeological drilling and test work, geohazard studies, and weather and snow studies have been completed. The results of these have been incorporated into the mine designs for this current PFS update.

The following issues have been considered in the current work:

- Slope deformation or rock falls from unstable slopes requiring restricted access to the pit, slope rehabilitation, and lost or reduced mine production. These risks are mitigated via geotechnical design and assessment prior to mining, a comprehensive slope monitoring and management plan during mining, the addition of extra-wide benches at regular 150 m intervals in the pit slope configurations, and standard operating procedures that include good wall control practices and mine operations planning.
- Snow avalanches at the pit crests resulting in restricted access to the pit. These risks are mitigated via an avalanche management plan coordinated with mine operations.
- Visibility or weather shutdowns resulting in reduced productivity in the open pits. These risks are mitigated via snow handling crews and equipment, planned lost days and the use of ore stockpiles.
- Slope depressurization to reduce high pore water pressures in the pit slopes that might result in pit slope instability. These risks are mitigated by an extensive slope depressurization plan that includes vertical wells, horizontal drains, and dewatering adits as a multi-layer system to achieve the design depressurization targets.
- Surface water reporting to the pits due to failure or inundation of water management structures. Excessive surface water into the pits may result in

localized slope failures, reduced mine productivity or increased pumping costs. These risks have been mitigated by adjusting the mine plan to limit the exposure of haul ramps to potential erosion from surface water sources and adequate geotechnical design of the water management infrastructures.

The risks associated with the open pits of the KSM Project are common for many open pit mining projects with large open pits mined in areas of mountainous terrain with high precipitation. The mining and geotechnical teams have drawn on experience from operations in these conditions and applied it to the designs in this study to mitigate the risk through design, and operating practices, and procedures integrated into the plan.

25.11.2 TUNNELS

The KSM Project involves approximately 75 km of tunneling at start up. This includes both infrastructure and water diversion tunnels to accommodate the challenges of building a mine in mountainous terrain and to reduce project risks. In later stages of the Project, the total constructed tunnel length rises to over 100 km. Tunnels provide more direct routes for ore transport from the mining areas to the Treaty OPC and divert surface water around mining areas and facilities. Tunnels were assessed as having lower operational risks than alternative surface routes. Surface routes are more susceptible to conflicts with other surface activities, and also present risks from climate and geohazards. It is notable that the permit covering the MTT route was secured from the BC Government in September 2014.

The PFS level of design for the tunnels in this study is based on reasonable assumptions made from preliminary-level investigations that include geological mapping of tunnel routes, geophysical surveys, drilling, and hydrogeological/geotechnical sampling and testing. Poor ground conditions caused by unforeseen faults, areas of weaker than anticipated rock, and/or higher than assumed differential stresses are issues that could cause major disruptions to tunnel construction and operation. If not accounted for properly in design, tunnel collapse or poor performance issues causing business interruptions could result, thus tunnel design represents a significant risk for the Project and warrants additional investigation in subsequent design phases. There are many successful tunnel projects, including comparable tunnel projects in the KSM area, such as the nearby Granduc Tunnel, that have addressed similar risks successfully. Higher speed methods for long axis tunnelling are becoming more prevalent in mining and civil applications and tunnelling technologies are developing rapidly. However, the cost of mitigation versus the impact of schedule delays must be considered in a risk analysis. Mitigations already incorporated include: the use of twinned tunnels to provide opportunities to advance alternate headings in problem areas, drilling and geophysical testing from surface, the use of probe and pilot drilling in areas of difficult ground conditions, and obtaining the now in place permitting allowing early tunnelling starts.

The results of the risk analysis will include cost-benefit analyses of additional investigations, assessment of risk sensitivities for alternate tunnelling methods, review of available changes in designs or routings, and in some cases alternative overland systems so that costs and risks of the alternatives can be considered in any decisions.

25.11.3 CONSTRUCTION CRITICAL PATH

The construction critical path runs through early establishment of pioneering roads along the TCAR and CCAR alignments, to establish access to the MTT portals, and then four years of tunnel excavation to complete the twinned 23 km tunnels. Pioneering road construction will be performed from the middle of Year −6 through Q2 Year −5 and includes multiple headings supported by helicopters to install culverts, bridges, and construction of the pioneering road that will expand to full width before the start of the second construction winter in Year −5.

While considered feasible, the schedule is considered aggressive, and additional measures to mitigate potential impacts completion of these roads may have on the critical path is recommended for action in the next design phase. A notable mitigation to the construction critical path is the fact that road construction permits have been granted to Seabridge, allowing road construction to begin earlier than anticipated in this 2016 KSM PFS update.

25.11.4 METALLURGICAL PERFORMANCE AND PROCESS

Lower than expected metal recoveries and copper grade of the concentrate produced is the principal process risk, particularly through the Project payback period, due to significant deviations in metallurgical performance. Unexpected inferior metallurgical performance would affect the Project economics.

Additional testwork is required to better characterize metallurgical response of mineralization from the different ore sources. Further process condition and flowsheet optimization would improve process design to accommodate potential variations in metallurgical performance. Current recoveries and economic returns are estimated based on the process design, mainly developed from the metallurgical test results of the Mitchell samples. Samples from the Kerr, Sulphurets, and Iron Cap deposits were also subjected to metallurgical tests and responded similarly to the Mitchell mineralization.

The PFS proposes to use energy efficient HPGR as a part of the comminution process. The test results from the Mitchell and Sulphurets samples show that this mineralization is amenable to HPGR treatment, although the Sulphurets samples are more resistant to HPGR treatment as compared to the Mitchell samples. The HPGR circuit performance is more sensitive to mill feed moisture and clay contents than other portions of the comminution circuit. Lower than expected performance from the HPGR comminution circuit due to higher than expected moisture and/or clay content can result in a reduction of mill production that could have an impact on the Project economics.

25.11.5 BLOCK CAVES

A conservative approach was adopted for the design of the critical aspects of the Mitchell and Iron Cap block caves that might impact meeting the production and mill feed schedules. Examples of this include the rate at which the construction of the drawbells is accomplished, the production ramp-up curve and maximum rate of draw of individual drawpoints, the number of drawpoints available for production at any one time, and the travel distance for the LHD vehicles. Design assumptions for all of these are within demonstrated industry experience and are considered achievable. The development advance rate during the critical pre-production period, and subsequently in the production period, have been demonstrated to be industry achievable, and with the technology advances that are currently being developed, may be demonstrated to be conservative in the future. As further conservatism, technological advances with regard to automated and battery powered equipment have not been incorporated in the design at this stage. The adoption of these more advanced technologies may lead to operational efficiencies and reduced costs in the future as they are studied further and adopted into the design.

In developing the production and grade schedules using GEOVIA PCBC™ software, it has been assumed that an infinite supply of zero grade waste rock is available above the columns being drawn. This and other factors controlling the estimates of ore dilution using GEOVIA PCBC™ have been chosen conservatively.

There are recognized uncertainties in predicting the fragmentation of the caved rock at the drawpoints, and the relatively low-fracture intensity of the rock mass at Mitchell, and to a lesser extent Iron Cap, led to the estimated fragmentation being assessed in some detail. Other block caving mines such as Palabora Mine have demonstrated that with careful planning and the availability of equipment to deal with oversize rock, drawpoint production rates comparable to those proposed at Mitchell and Iron Cap are achievable. Furthermore, if the ore pass and grizzly systems do not achieve the planned production rate, and the material transported to the passes is coarser than expected, mitigation measures can be introduced such as employing additional mobile breakage equipment and redesigning the undercut blasting to enhance the fragmentation during the early stages of the column draw.

As an additional measure to mitigate against possible adverse fragmentation issues at the drawpoints, the designs for Mitchell and Iron Cap incorporate pre-conditioning of the rock mass by hydraulic fracturing, which was not applied at Palabora Mine. This is expected to enhance the fragmentation of the material reporting to individual drawpoints. It also mitigates against possible concerns regarding adverse impacts from stress-induced rock bursts at Mitchell, although the proposed depth of mining and the geotechnical characteristics of the rock mass do not indicate that this will likely pose a major issue. The stress conditions at Iron Cap are estimated to be relatively benign because the deposit is located above the valley floor in a relatively stress-free zone.

The geotechnical characterization of the Mitchell deposit is based on a number of well-positioned geotechnical boreholes that have been geotechnically logged in detail and tested hydrogeologically. Some geotechnical holes have been drilled in the general vicinity of the proposed access development and these holes provide adequate indications of the quality of the rock mass to minimize major uncertainties. Further mitigation of additional uncertainties will be achieved by undertaking further geotechnical drilling in the future.

The geotechnical characterization of the Iron Cap deposit has been based on several holes that were drilled for geotechnical purposes, but reliance was also placed on comparisons of the geological, lithological, and alteration characteristics between the

Mitchell and Iron Cap deposits, and developing geotechnical correlations between the two. The correlations were shown to be quite favorable, but further confirmation will be required by undertaking additional geotechnical drilling in the future.

The spacing and layout of the drawpoints of 15 m by 15 m was conservatively selected to achieve favorable interaction between draw columns, minimize early entry of waste dilution material, and to maintain good draw control. Future studies may indicate that this conservatism is more than is required, and an increased drawpoint spacing and layout can possibly be adopted with associated enhanced project economics.

Industry standard approaches have been adopted to support the drawpoints and to maintain their stability. If future analyses of abutment stresses and the interaction of the progressive advance of the undercut on the underlying drawpoints indicate more adverse stress conditions than currently estimated, this can be mitigated by installing higher capacity and more resilient support. It can also be mitigated by adopting a stress-shadowing advance undercut approach, instead of the currently proposed concurrent undercut approach.

The maximum height of draw (HOD) assumed in the design, which controls the maximum tonnage that is drawn at an individual drawpoint, has been limited to 500 m. Based on favorable experience at a number of block cave mines operating in good quality rock, the trend in the industry under these circumstances is to plan to increase the maximum HOD to beyond 500 m. However, the more conservative 500 m limit has been adopted for the Mitchell and Iron Cap designs. There is still the potential requirement for ongoing rehabilitation of drawpoints and mucking drives as a result of changing stress conditions and the adverse impact of secondary blasting. An estimate of the cost of this rehabilitation has been made and this has been included in the block cave operating costs.

The extent of surface disturbance from caving and the associated formation of the crater and more peripheral surface cracking have been estimated conservatively for the relatively good geotechnical quality of the rock mass at Mitchell and Iron Cap. Estimates have also been made of the influence of this disturbance on the stability of the walls of the Mitchell open pit and adjacent valley walls. Based on the analyses to date, all permanent infrastructure is beyond the potential impact zone. Major infrastructure that might be critically impacted has been conservatively located well beyond the potential impact zone. If further assessments and future monitoring indicate a potential impact of some of the less critical infrastructure, there is sufficient space on surface for this to be re-located at modest additional cost. The Iron Cap block cave design does not include infrastructure that might be influenced by the extent and nature of the surface disturbance, other than the surface water runoff that might enter the cave that is discussed further in this section.

The estimate of the inflow of surface runoff from rainfall and snowmelt within the catchment area formed by natural slopes, the open pit (in the case of Mitchell), and the crater and adjacent surface cracking, has been based on a 1-in-200 year event. A conservative approach has been adopted in estimating the runoff coefficient, water retention, and impeding factors which control the intensity of this runoff water reaching the production levels underground. The inflow quantities and rates reaching the underground workings are potentially very large for a 1-in-200 year event.

A water management system has been developed for Mitchell, which does not rely on any temporary storage of water in any of the mine openings otherwise required for the operation of the mine. Instead, dedicated water diversion tunnels with a large dedicated storage capacity have been included in the design, with a dedicated shaft and pumping system to extract the water to surface. If experience in the future indicates that the storage capacity is too small to prevent flooding of the operational part of the mine, the pumping rate storage volume can readily be increased or more water diversion tunnels can be added for extra storage. Also, if there is concern about the capacity and functionality of the partial diversion system beyond the rim of the pit that helps control some of the inflow water, or if there is concern that an even larger event than the 1-in-200 year event might occur, consideration can be given to diverting water underground to temporarily flood areas that are benign and do not house critical underground infrastructure, and/or to installing bulkhead doors that can be closed under extreme conditions to limit flood damage to critical underground infrastructure.

The elevation of the production level at Iron Cap is higher than the Mitchell valley floor. This provides for a much simpler water management system than at Mitchell. This is based on gravity flow of water from the entry points underground to a short shaft that connects to the North Slope Depressurization Tunnel. The water then gravity drains from this tunnel to the WSF for treatment.

The development of voids beneath the back of the active cave, and the associated concerns about hazardous conditions developing and air blasts occurring, is mitigated by maintaining good knowledge about the cave profile as cave mining progresses. This knowledge is obtained by ongoing monitoring of the geometry of the caved material using techniques such as microseismic monitoring and seismic tomography, which are important mitigation measures to maintain necessary draw control.

Experience at other block cave mines, where there is more potential for mud rushes to occur because of the increased presence of fines (naturally occurring or generated by the attrition of caved rock as it is drawn down) than those at Mitchell or Iron Cap, has shown that such concerns can be mitigated. The measures that can be adopted include monitoring of the flow of water from individual drawpoints, temporary closure of certain areas that are deemed vulnerable until water flows at drawpoints decrease sufficiently, and the adoption of remote mucking until conditions are deemed to be acceptable to return to full entry.

25.12 PRELIMINARY ECONOMIC ASSESSMENT RISKS

25.12.1 UNDERGROUND MINING RISKS

Greenfield block cave projects have certain risks that are characteristic of the mining method. The risks are fixed because the effectiveness of the design cannot be determined until after large portions of the mine have been constructed and put into service. The risk factors include:

- cave performance such as cavability, propagation, fragmentation
- geotechnical issues resulting from mining induced stresses such as pillar dimensions, production area design, and ground support designs
- these risk factors are often magnified when employing a workforce that does not have experience in underground mining.

These risks are reduced when opening a new area in an existing mine because the new project can take advantage of experience gained in the existing operation and will be able to utilize experience labor, management, and technical staff.

25.12.2 Marketing Risks

A small number of the minor element assays of the final bulk concentrates from Deep Kerr indicate that concentrations of arsenic, antimony and mercury may be near or above typical smelter penalty limits. The concentrate minor element data should be reviewed by a concentrate marketing specialist to identify potential concentrate marketing issues. Additional testwork will identify the extent of the risk and whether it can be managed through blending in the mine plan.

26.0 RECOMMENDATIONS

26.1 INTRODUCTION

This section outlines areas to investigate for improvements to the KSM Project. A high-level budgetary estimate for the completion of each recommended item is provided in US dollars.

26.2 PRE-FEASIBILITY STUDY RECOMMENDATIONS

The QPS for this 2016 PFS agree that the project should advance to the next logical phase of study along its development toward becoming an operating mine.

26.2.1 MINING

OPEN PIT AND RESERVES

During more advanced studies, the following optimization studies are recommended:

- Conduct a detailed blasthole drill study in conjunction with drill manufacturers to determine site-specific penetration rates for the rock types from each mining area. The 2016 PFS is based on typical values.
- Conduct a detailed haulage simulation study to refine the truck/shovel production performance in the mining schedule. Simulations should also investigate opportunities to reduce mining costs (e.g., combine pit and dump design strategies to lower fuel consumption from increased downhill loaded hauls). This optimization can have significant efficiency and cost impacts in mountain mines.
- Analyze alternative equipment power solutions in conjunction with the manufacturers, such as liquefied natural gas (LNG) and electric trolley assist.
- Evaluate the basal and selenium drain design(s) to reduce the volume requirements, which could lead to reduced mining costs and greater flexibility in the timing of waste rock placement.
- Obtain budgetary quotes from experienced mine contractors for the capital cost estimates of constructing the initial bench access roads and mine area haul roads.
- Obtain budgetary quotes for the operating costs estimates for an optional leased truck fleet in the early years of the mine schedule (pre-production and initial years of production).

The total estimated cost for these optimization studies is between US$50,000 to US$75,000.

The open pit mine planning designs will need to be reworked to a feasibility-level of detail. The total estimated cost for this open pit mine planning is between US$200,000 to US$250,000.

Also, close-spaced drilling and a higher resolution Mineral Resource model will be required for the material in the first several years of open pit operations (Mitchell and Sulphurets) to test for grade and metallurgical continuity. The estimated cost for the extra drilling, assaying, Mineral Resource modeling, and metallurgical testing, is between US$3 million and US$4 million.

UNDERGROUND

The following optimization studies are recommended for underground block cave mining of the Mitchell deposit, beneath the Mitchell open pit, and of the Iron Cap deposit.

Mitchell

- Undertake a detailed study of the optimum pit profile to transition from open pit to underground mining.
- Review available geotechnical logging information within the mineralized rock beneath the pit and if deemed beneficial, geotechnically log additional available core to upgrade assessments of fragmentation, drawpoint hang ups, oversize, and drawpoint layout and spacing.
- Evaluate the impact of fragmentation, oversize, and hang ups on productivity.
- Undertake a detailed cost-benefit assessment regarding fragmentation and mitigating rock stress impacts, and evaluate the associated impact on schedules of incorporating the pre-conditioning as presently designed into the overall design and mine plan to confirm it provides a net positive benefit.
- Undertake a study of the potential to increase the drawpoint layout and spacing from the currently proposed 15 m by 15 m and maintain favorable interaction between draw cones without experiencing adverse dilution and premature reporting of sub-economic rock to the drawpoints.
- Undertake additional laboratory rock strength tests on core to characterize the rock mass strength of the various lithologies and alteration types and intensities.
- Undertake computer analyses of the stress conditions controlling the extent to which stress-induced fracturing enhances primary fragmentation in the back of the cave, and the potential to experience associated rock bursting as a result of this stress-induced fracturing.
- Undertake computer analyses of the abutment and cave front stress concentrations. Assess whether an advanced undercut approach, rather than the presently proposed concurrent undercut approach, needs to be adopted to

control the stress concentrations on the drawpoints and maintain adequate drawpoint stability.

- Undertake additional studies of the progressive formation of the crater and resulting destabilizing influence of this on the overlying pit and natural valley slopes. Conduct more detailed assessments of the overall surface disturbance and the possible influence of this on the surface infrastructure adjacent to the block cave operation.
- Develop and undertake a geotechnical drilling and logging program to characterize the rock that the peripheral infrastructure will be excavated in, including access ramps, ventilation shafts, water conveyance and storage tunnels, crusher stations, underground excavations to support production activities, etc.
- Undertake a detailed review of the surface water runoff inflows and the underground water management system. Conduct optimization studies to meet conveyance and temporary storage requirements to minimize flooding of the mine and pump the water from the mine in the most cost effective manner.
- Undertake detailed GEOVIA PCBC™ scheduling studies to optimize caving front and drawpoint sequencing and draw control requirements.
- Conduct mine productivity simulations to confirm operability and identify potential restraints and restrictions in achieving the proposed production schedules.
- Develop feasibility-level mine designs and designs of underground infrastructure including crushers, conveyor systems, shops, and ventilation.
- Assess opportunities to incorporate mine automation and battery powered equipment into the design to reduce ventilation demand and operating costs.

Iron Cap

Similar studies to those recommended for Mitchell are required for Iron Cap as well, although in a number of cases the results of the assessments for Mitchell can be applied to Iron Cap on a comparative basis, with the geotechnical characteristics of the Mitchell and Iron Cap rock masses being somewhat similar. Studies are not required of a transition from open pit to underground mining, and the underground water management system at Iron Cap does not require temporary storage and pumping.

The current production schedule includes production from Iron Cap starting in Year 32, and there are no transition restraints with other production sources in bringing Iron Cap into production. Based on this, the level of study required for Iron Cap is less than for Mitchell to undertake relevant economic assessments for a feasibility-level study.

Notwithstanding this, the most significant gap in advancing Iron Cap to a feasibility level is that no geotechnical drilling and logging has been undertaken specifically directed to characterizing the response of the rock mass to block cave mining. In the current study, this has been done primarily by comparison and inference with the Mitchell deposit, and limited geotechnical drilling for pit stability assessments in the mineralized rock. It is recommended that a limited program of geotechnical drilling appropriate for block cave mining assessments be undertaken at Iron Cap to advance the studies to a feasibility level.

This feasibility-level underground work for both Mitchell and Iron Cap, including a limited geotechnical drilling program at Iron Cap, is projected to cost between US$2.5 million to US$3.0 million.

PIT SLOPES

Additional field work, including geotechnical drilling, hydrogeological testing, and laboratory testing are recommended by BGC for the proposed Mitchell and Sulphurets pits at the next stage of study. This work is not only for the open pit operations, but also to examine the effect of the future Mitchell block cave on the excavated pit walls and surrounding infrastructure. It is expected that additional work will be completed to further refine each deposit's geological interpretation by Seabridge, including the location of faults and alteration zones. Future work for the open pit slope design and hydrogeological evaluations will focus on increasing the confidence level of the slope designs and slope depressurization plans. Specific tasks to be completed include:

- Long-term pumping tests of the rock mass at each of the proposed open pits using 6 to 10 inch diameter wells to provide bulk estimates of rock mass hydraulic conductivity and provide data for the optimization of the slope depressurization plan.
- Numerical stress/deformation modelling of the proposed Mitchell pit, specifically the north and south slopes.
- Refinement of the design and timing for the proposed Mitchell NPWDA.
- Assessment of pit slope stability to infrastructure closely sited near the pit crest.
- A risk assessment for potential water into the pit over the Mitchell pit east wall from flows exceeding the capacity of the MDT. 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 NPWDA into the Water Management Plan.

It is estimated that the next stage of combined geotechnical and hydrogeology work for the KSM Project will require a budget of approximately US$6 million for engineering fees, geotechnical drilling, and hydrogeological drilling and testing.

26.2.2 ROCK STORAGE FACILITIES

KCB recommends that a geotechnical site investigation program be carried out for further delineation and characterization of foundation soils in Mitchell Valley, within the footprint of the Mitchell RSF, with a focus on the area of a known deposit of weak soils near the MTT Mitchell portal. The characterization would use Standard Penetration (SPT) testing and undisturbed sampling for collection of samples for laboratory strength testing. This program will include:

- Sonic geotechnical drillholes in the area adjacent to the MTT portal and in the RSF toe above the WSF pond area. The holes should be advanced up to 60 m in overburden, with Shelby tube samples and SPT tests. Mobilization of a sonic rig for this program can be combined with sonic drilling in the TMF area for borrow assessment.
- Test pits to assess surface soil conditions (for early-stage investigations these will require helicopter supported equipment mobilization).
- Geotechnical testing program including consolidation, triaxial, direct shear, and index testing.

The cost of this drilling and subsequent geotechnical laboratory program is estimated at US$650,000.

A stability assessment and design review is required to assess the findings of the site investigations and to review suitability of existing designs to mitigate the presence of this known weak soil layer. This is estimated to cost US$80,000.

26.2.3 WATER

WATER STORAGE FACILITY AND MINE SITE WATER MANAGEMENT

KCB recommends additional hydrogeological and geotechnical site investigations in the WSD footprint area including:

- Hydrogeological/geological data gap analysis and review of additional water level and time series piezometer data to interrogate interaction of geological model assumptions with hydrogeological model parameters.
- Additional geological mapping and geophysical surveying of the WSD footprint area.
- Additional hydrogeological site investigations consisting of drilling six, 150 m deep holes in the WSD area, with packer testing, televiewer downhole imaging, and installation of multi-level piezometers equipped with data loggers. Based on the results, assess the requirements for drilling of a large diameter well and if required, conduct a larger scale pumping test, with associated monitoring holes.
- Geotechnical testing included for WSD area hydrogeology drill holes to characterize soils in the WSD footprint, including SPT sampling and geotechnical laboratory testing program. Complete a test pit program in WSD footprint area to assess soil conditions.
- Installation of thermistors chains within new or existing drill holes in the WSD footprint area to obtain annual rock mass temperature variations with depth to assess effect of temperature on grouting performance.
- Acid-base accounting (ABA) and geotechnical laboratory testing of local rock fill sources for the WSD.

- Geotechnical drilling and ice radar in the areas of the Mitchell Glacier diversion inlets.

The cost of these site investigation programs is estimated at US$2.5 million, including an allowance for drilling, helicopter support, and contractor costs, and excluding large-diameter well drilling and testing.

Several of the IGRB recommendations for design updates to the WSD and Mine Site water management were addressed within design updates included in the technical reports developed for the 2016 KSM PFS update (Appendix H). The following design studies are recommended to address the remaining IGRB recommendations:

- Optimization of WSD grouting gallery design, potential inclusion of initial concrete valley plug structure at base of WSD and potential for WSD gravity discharge structures.
- Assessment of local rock fill properties to examine potential for reduction of WSD fill haul costs.
- Assessment of options for capture of selenium seepage from RSFs and from other sources to develop an optimum strategy for control of selenium.
- Update of WSD and Mine Site water balance to incorporate revised hydrological and meteorological parameters including updated estimates of response to extreme events.
- Update of hydrogeological seepage models for WSD based on updated site investigations and site parameters.
- Advancement of WSD, diversion tunnel, spillway and Mine Site water management designs to a feasibility level of detail.

The cost of these design updates is estimated at US$2.0 million.

WATER BALANCE

Design parameters such as the 50-year, 100-year, and 200-year storm and flood events; the 100-year dry year and 200-year wet year have been used in Project design for the sizing of diversion ditches and tunnels and for evaluation of water quality in receiving waters. These parameters have to date been developed from several years of on-site precipitation and streamflow gauging, the nearby longer period Eskay Creek station, as well as use of Global Climatic Models (GCMs). While these analyses provide suitable design parameters for the PFS, more comprehensive analyses incorporating long-term records in the wider region surrounding the sites are required for advanced Project design phases. In general terms, the longer-term (20 to 30 plus years) meteorological and runoff data from regional stations provide more reliable statistical distributions that may then be extrapolated to the various areas of the site by correlating with on-site stations, some of which have now been monitored for 8 years.

It is important for the Project design evolution that the meteorological and runoff database is expanded and the design parameters refined by conducting a regional hydrologic evaluation incorporating precipitation and stream flow monitoring data from a larger area surrounding the mine site. Typically, the steps in such an analysis include:

- Selection of long term meteorological and runoff stations.
- Single station frequency analyses for the parameters of interest (such as precipitation of durations ranging from hours to a year, annual maximum flood peaks, etc.).
- Regional analyses to determine frequency curves applicable to the various areas such as the Mine Site, the TMF area, etc.

Following completion of this metrological and runoff data base review, evaluation is recommended to be undertaken to update the site-specific water balance that will be used as the basis in future Project design. The cost recommended to complete this work is estimated at US$400,000.

TEMPORARY WATER TREATMENT PLANTS

A total of eight TWTPs 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. The TWTPs will include a grit pond, lime and flocculent preparation units, settling ponds as described in Section 18.2.8.

Changing environmental regulations regarding the capacity of these treatment systems to handle larger upset flow conditions have resulted in the requirement for substantial increases in the sizes of the grit and settling ponds. Given these circumstances, and the adherence to the overall philosophy of minimizing environmental impacts where possible, it is recommended that the over strategy associated with the use of TWTPs be re-evaluated to focus on the following:

- Are there alternative treatment systems that could result in a net decrease in the surface disturbance associated with temporary water treatment systems, while still achieving the discharge standards, required by the environmental regulations?
- Could alternative approaches to project execution yield dual purpose facilities—construction use, then different use during operations—that optimize the approach to site water management systems, such as early construction of the TMF cells to initially be used to store grit and act as a settling pond?

A budget of approximately US$250,000 will be required to complete this task.

GEOCHEMISTRY DATABASE

The geochemistry database for the 2012 KSM PFS was based on typical numbers of samples, representative of geochemical properties of the ore, waste, and non-deposit rock in the Project area. Annual updating of the geological models for the Project have occurred since 2012, as reported in the 2016 KSM PFS update, and re-visiting of the ABA geochemical block model is appropriate. An independent, geochemical review is recommended to test model assumptions, particularly because it has been recommended that exploration assay results be used as surrogates for more typical ABA results, increasing the potential number of samples from approximately 2,200 to over 50,000 that could be used to develop the model. The intent of the surrogacy is to improve the spatial representation of the sampling program and increase regulator confidence in the ABA block model, as well as ensuring the validity of the existing model as the Project moves into the next phase of study. A cost of US$100,000 is estimated for completion of this study.

26.2.4 TMF Area

KCB recommends additional hydrogeological and geotechnical site investigations in the TMF footprint area consisting of:

- Hydrogeological data gap analysis and review of additional water level and time series piezometer data. Data gap analysis to assess requirements for pumping tests or other investigations.
- Data gap review of geotechnical (foundation) and geochemical (characterization of borrow and diversion excavations) data requirements.
- Drilling and hydrogeological testing (packer, multilevel piezometer and data logger installations) at the North, Saddle and Southeast seepage dams (three drill holes, each hole of 70 m depth).
- Seep mapping and overburden characterization, additional overburden permeability testing in the CIL Residue Cell area to further inform the next design stages for determination of drain requirements beneath the Saddle Dam and CIL Cell liner.
- Borrow and foundation geotechnical investigations consisting of six additional sonic/SPT/core drill holes and test pits in the TMF dam footprint and borrow areas with geotechnical laboratory test program. Test pitting for borrow investigations may require helicopter support for equipment mobilization.

The cost of these site investigation and laboratory testing programs is estimated at US$2.5 million including an allowance for drilling and helicopter support.

The following design updates are recommended to address remaining IGRB TMF area recommendations:

- Update of TMF area hydrogeological and seepage models and review of required seepage mitigations based on updated site investigations.
- Updates to drainage designs for TMF dam and CIL liner footprint areas based on revised hydrogeological data and models with liner ballast assessment.
- Review of required TMF dam cross sectional designs required to meet seasonal storage, freeboard related stability, starter dams and constructability requirements relative to cycloning design optimizations, borrow availability and feasibility level design requirements.

- Review and update of volume/storage requirements for each cell, based on revised CIL Residue cell volumes.
- Review of CIL Residue Cell subaqueous deposition system, and liner and drain placement plans for constructability/operability and cost optimizations.
- Update to TMF area water balance with design updates for discharge pipeline and diffuser system.
- Advancement of TMF area water management designs to FS-level.

The cost of these design updates is estimated at US$1.5 million.

26.2.5 TUNNELS

GEOTECHNICAL

KCB recommends additional geological and geotechnical investigations at the portal locations, where these have not been investigated already, and also where feasible along the Project tunnel alignments. The recommended investigations consist of:

- Data gap analysis for tunnel design parameters including geological, geotechnical and geochemical aspects.
- Additional mapping and rock sampling to better characterize properties of lithological units at the portals and along the alignments for geotechnical and geochemical assessments.
- More detailed mapping of portal areas and targeted geophysical investigations to assess locations of potential structures (e.g., faults, contacts or water bearing structures along tunnel alignments).
- Drilling at select locations along the permitted MTT alignment (for example, undrilled opportunities for readily accessible drilling to MTT elevation along the alignment include 200 m deep, 500 m deep, and 750 m deep locations). Drilling will provide an opportunity to obtain geotechnical and geochemical samples for laboratory testing, to assess in situ stress regimes, and to perform televiewer and packer testing
- Drilling for diversion tunnels and project infrastructure tunnels will involve shallower holes, with focus on portal and inlet area rock mass characteristics and sampling of lithological units.
- Geotechnical and geochemical laboratory testing on samples obtained from drilling and mapping programs.

The programs will inform selection of the optimum tunnelling method, allow refinement of project tunnelling risk, better identify diversion tunnel lining requirements, portal locations and portal development designs and assist with determination of appropriate contingencies. The cost of these programs is estimated to range between US$2.0 million and US$3.0 million, including for drilling costs and helicopter support.

TUNNEL DESIGN

An expert-level risk assessment of the critical tunnels is recommended following the tunnel geotechnical data gap analysis to identify aspects of tunnels with high consequence to the construction schedule. This should also be done for water diversion tunnels with high impact to future facilities. Some of the assessments can be deferred until after start up. If the end results are design alternatives that cannot mitigate high-consequence risk, then significant data must be collected regarding ground conditions and geologic structure. The risk analysis, and any subsequent data collection and studies, should be done prior to a FS so that the Project can be re-evaluated before significant FS work starts. The analysis should include re-evaluation of opportunities and risks with sensitivities of other tunneling methods to unforeseen conditions. Methods reviewed should include high speed drill and blast, use of a single tunnel with an internal dividing wall, or mechanical systems such as a Tunnel Boring Machine or a Continuous Miner.

The results of the opportunity and risk analysis will indicate where, or if the following are needed:

- Geophysical studies to identify major structures and geology to estimate possible ground conditions.
- Drilling of areas along the tunnel alignment to confirm structures and geology indicated in the geophysical survey.
- A detailed risk study may also inform if alternate mitigations such as tunnel lining or other measures to improve tunnel stability and capacity can be considered instead of the redundant water diversion tunnels included in the water management designs at certain times during mine life and beyond.
- Ongoing analysis of tunneling method and machines is needed as this is a continuously advancing technology.

The above geophysical surveys and drilling costs are already recommended in Section 26.7.

Following the completion of the above risk analysis, possible data acquisition programs, and assessment of resultant alternative design concepts, advanced design updates and budgetary quotes for costing of mitigation options may be needed at an estimated cost of US$250,000.

26.2.6 METALLURGICAL TESTING AND PROCESS ENGINEERING

Further metallurgical test work to optimize process conditions and to establish design-related parameters for the next stage of study is recommended. 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 all deposits, especially from the Sulphurets, Upper Kerr, and Iron Cap zones. The variability tests should be a part of the geo-metallurgical testing program that is recommended for the Project to better understand metallurgical responses of the mineralization in these resources. The test program will provide inputs for geological modelling development and mine plan update for better mill feed control. The cost of the test work, including the initial stage of the geo-metallurgical testing, is estimated at approximately US$2 million.

- Further study should be conducted to optimize the proposed cyanide recovery and destruction methods (US$200,000).
- A metallurgical laboratory test program should be performed on ore composites representing each year of the initial seven years of open pit mine production from Mitchell and Sulphurets (US$200,000) according to the updated mining plan.
- Further study into economical water treatment methods is recommended for water from the CIL pond (US$200,000).

All the costs for the testing to verify metallurgical performance of the samples and collect process design-related parameters do not include the sample collection and shipping costs.

26.2.7 MTT Transport System

The KSM Project should advance to more advanced levels of study. Along with updating the level of detail in the train system designs, the following optimization studies are recommended as part of the next level of study:

- Optimize the size and configuration of the train consists using dynamic train movement modelling.
- Investigate alternative rail systems such as tracks, ballast and ties, and pre-manufactured systems etc. to optimize supply and construction cost and schedule impacts.
- Evaluate the construction cost and construction schedule of the MTT with respect to tunnel dimensions and alignment, in conjunction with the above optimization study of the train consists.
- Update the analysis of freight, fuel, lime and personnel transport requirements through the MTT to support the mine area operations, especially if the other project design changes affect these components.
- Define and coordinate labour requirements between train, Process Plant, and Mitchell OPC.
- Engage several train system suppliers to optimize the train fleet prior to FS-level budgetary quotes. Due to the specific requirements of the system, it may be necessary to pay for these studies.

Train system designs should be updated during more advanced studies. The train system engineering portion to FS-level detail is estimated to cost between US$250,000 to US$350,000.

26.2.8 KSM Exploration

The QP responsible for Mineral Resources recommends that Seabridge continue exploring around and beneath the currently recognized mineralized systems (Deep Kerr, Sulphurets, Mitchell, and Iron Cap). These programs should follow past successes in the district by re-examining all geologic controls to develop drill targets that focus on potentially higher grade zones, which may be greatly controlled by a combination of intrusive phases and structural preparation. A program consisting of 40,000 m estimated to cost US$13 million should be considered by Seabridge over the next several drilling seasons.

26.3 Recommendations from the Preliminary Economic Assessment

Amec Foster Wheeler recommends the PEA development option for the KSM project be advanced to the next stage of evaluation. The following recommendations would support more advanced studies.

26.3.1 Deep Kerr Exploration Adit

It is recommended that an exploration adit be developed from near the Sulphurets Valley floor at the base of Kerr Mountain towards the south, to access areas of the deposit that are difficult to drill from surface. The PEA-level analysis described in Section 24.0 indicates underground mining as the preferred mining method for the Kerr deposit. In order to upgrade the Mineral Resources from Inferred to a higher-quality definition, close proximity to the deposit is required because targeted drilling from the surface is logistically difficult.

On September 26, 2016 Seabridge received the necessary permits from the BC Government to develop the exploration adit.

Development of the exploration adit will consist of surface facilities, including water treatment and a lined waste rock pile that will be covered at closure. Additionally, approximately 2 km of drifting and hydrogeological testing will be carried out. The total cost will be in the range of US$70 million.

26.3.2 Deep Kerr Exploration and Mineral Resource Drilling

The QP responsible for Mineral Resources recommends that Seabridge conduct an infill drilling program with the goal of upgrading as much Inferred Mineral Resources that were used in the 2016 PEA. The majority of the effort should focus on the Deep Kerr deposit. It is estimated that approximately 60,000 m of drilling may provide sufficient confidence to allow for the upgrade of the Mineral Resources. The estimated cost for this infill drilling campaign is US$20 million.

26.3.3 DEEP KERR GEOTECHNICAL STUDIES

Amec Foster Wheeler recommends that the exploration drilling program on Deep Kerr should be augmented with geotechnical activities, which will provide a better understanding of the rock structure characteristics important for the mine design. These activities include: additional data collection from exposures in the planned Deep Kerr exploration adit; in conjunction with the proposed exploration drilling program on Deep Kerr, core logging geotechnical surveys, Acoustic Televiewer surveys (ATV), and oriented core data logging conducted. Following the data collection from the drilling program, geotechnical assessment and modeling will then need to be integrated with the mine design efforts. The expected costs of this geotechnical program is US$500,000 (additional to the cost of developing the exploration adit and exploration drilling).

26.3.4 DEEP KERR METALLURGICAL TESTWORK

Amec Foster Wheeler recommends that Seabridge perform further testwork for the Deep Kerr mineralization, investigating the response of various zones to both flotation and cyanidation. This will require composite sampling to assess:

- Specific energy, pressing force, and abrasion work to confirm suitability of the HPGR approach to comminution. This testwork is to be done through the use of HPGR testing, piston load testing and pilot plant work on samples.
- Optimization of primary and regrind milling will be necessary. Bond rod and ball mill work index tests together with regrind jar test work will be necessary for the sizing of ball and regrind mills.
- Flotation optimization which will include rougher flotation optimization, cleaning work and confirmation of performance through rougher and cleaner batch flotation work followed by locked cycle flotation testwork.
- Cyanidation amenability including direct cyanidation kinetics, reagent consumption optimization, and retention time optimization. This testwork will involve both direct cyanidation and CIL batch tests.
- Cyanide recovery through the SART approach will be tested through a combination of bench scale batch and bench scale continuous work.

This work would confirm the applicably of the process flowsheet used in the 2016 PFS for the Deep Kerr mineralization considered in the PEA.

Testing can be performed using drill core from past and future work. It is expected that the comminution testwork would involve approximately 120 point samples requiring approximately 1.5 t of material, 3 small scale continuous HPGR tests requiring another 1 t of material, and approximately 4 t of material for the pilot plant level HPGR work. The smaller samples would be produced through diamond drill hole drilling while the large quantities would be produced through bulk underground sampling. Subsequently, comminution samples would undergo flotation and cyanidation testwork, both batch and locked cycle. This would require approximately 200 tests of various forms. Comminution testwork would cost approximately US$1 million, while the concentration work would cost a further US$1 million, taking into account also assaying and mineralogical work. It is expected that the new sampling by diamond drill hole work would be approximately US$0.5 million if performed from underground working. It is also expected that the bulk sample for HPGR work could be produced for another US$0.5 million if planned as part of the overall underground drifting work planned.

This testing would confirm comminution, flotation and cyanidation viability. With confirmation of the process parameters, the testing of domain composites and point samples would address geometallurgical variability to a PFS level. The overall cost of the program using both available and new sample would be approximately US$3 million to provide information to the PFS level for Deep Kerr. This cost may be lower if drilling is also performed in conjunction with other requirements but it may be also higher if metallurgical issues are indicated by new drilling.

27.0 REFERENCES

1985 Fisheries Act, RSC. C. F-14. (1996a).

ALS Canada, Ltd., ALS Metallurgy Kamloops (2013). Preliminary Metallurgical Testing – Kerr Deep and Camp Zone KSM Project – (KM 3735). May 9, 2013.

ALS Canada, Ltd., ALS Metallurgy Kamloops (2014). Preliminary Metallurgical Testing – Deep Kerr and Iron Cap Zones – KSM Project - (KM 4029, Part A). May 16, 2014.

ALS Canada, Ltd., ALS Metallurgy Kamloops (2014). Preliminary Metallurgical Testing – Deep Kerr and Iron Cap Zones – KSM Project – (KM 4029, Part A). May 27, 2014.

ALS Canada, Ltd., ALS Metallurgy Kamloops (2015). Preliminary Metallurgical Testing – Deep Kerr – KSM Project – (KM 4514). August 05, 2015.

ALS Canada, Ltd., ALS Metallurgy Kamloops (2015). Preliminary Metallurgical Testing – Iron Cap – KSM Project – (KM 4672). August 04, 2015.

BC MEMPG (2009). Guide to Processing a Mine Project Application under the British Columbia Mines Act. British Columbia Ministry of Energy, Mines and Petroleum Resources, January 2009. http://www.coalwatch.ca/sites/default/files/Guide-to-Processing-A-Mine-Project-Application-Under-The-British-Columbia-Mines-Act.pdf.

BC MEMPR (2008). Health, Safety and Reclamation Code for Mines in British Columbia. British Columbia Ministry of Energy, Mines, and Petroleum Resources. http://www.empr.gov.bc.ca/Mining/HealthandSafety/Documents/HSRC2008.pdf (accessed November 2010).

BC MOE (2013). The Effluent Permitting Process under the Environmental Management Act: An Overview for Mine Project Applicants. Ministry of Environment, April 2103. http://www.env.gov.bc.ca/epd/industrial/mining/pdf/effluent_permitting_guidance_doc_mining_proponents_apr2013.pdf

BC MOF and BC MOE (1995). Riparian Management Area Guidebook. Government of British Columbia: Victoria, BC.

BGC (2012). Preliminary Assessment of Open Pit Slope Instability Due to the Mitchell Block Cave. December 24, 2012.

Brenda Mines Ltd. (1989). Preliminary Metallurgical Testwork on “106” Low Grade Sample. Peachland, BC. May 1989.

Canadian Institute if Mining, Metallurgy and Petroleum (CIM) (2014). CIM Definition Standards for Mineral Resources and Mineral Reserves. May 10, 2014.

CDA (2007). Dam Safety Guidelines. http://www.imis100ca1.ca/cda/CDA/Publications_Pages/Dam_Safety_Guidelines.aspx (accessed October 2012).

Contaminated Sites Regulation, BC Reg. 375/96.

DFO (1986). Policy for the Management of Fish Habitat. DFO/4486. Fish Habitat Management Branch, Department of Fisheries and Oceans Canada: n.p.

Environmental Management Act, SBC. C. 53.

Febbo, et al. (2015). Geology of the Mitchell Au-Cu-Ag-Mo porphyry deposit, northwestern British Columbia, Canada. British Columbia Ministry of Energy and Mines, British Columbia Geologic Survey Paper 2015-1, 28 p.

G&T (2007). Preliminary Assessment on Mitchell Zone Samples (KM 1909). June 2007.

G&T (2008). Pre-Feasibility Metallurgical Testing Mitchell Zone – Kerr Sulphurets (KM 2153). September 2008.

G&T (2009). Metallurgical and Pilot Plant Testing on Samples from the Kerr-Sulphurets-Mitchell (KSM) Project (KM 2344). December 2009.

G&T (2010). Ancillary Testing Kerr Sulphurets-Mitchell-KSM Project (KM 2755). December 2010.

G&T (2010). Bench Scale and Pilot Plant Testing Kerr-Sulphurets-Mitchell Project (KM 2670), August 2010.

G&T (2010). Miscellaneous Metallurgical Testing on Samples From the KSM Project (KM 2535). March 2010.

G&T (2011). Data Reports (KM 2897). February – March 2011.

G&T (2011). Flotation and Cyanidation Testing Snowfields Samples, Composites and Blends – (KM 3081). November 28, 2011.

G&T (2011). Preliminary Metallurgical Testing on Samples from the Iron Cap Zone – KSM Project (KM 2748). January 2011.

G&T (2012). Flotation and CIL Cyanidation Testing – Kerr-Sulphurets-Mitchell Project – (KM 3080). January 10, 2012.

G&T (2012). Ore Hardness and Flotation Testing – Kerr-Sulphurets-Mitchell (KSM) Project – (KM 3174). January 6, 2012.

Golder (2011). Block Cave Mine Study. Report Number REP 0516_11. Submitted May17, 2011.

Golder (2012).Pre-Feasibility Block Cave Mine Design – Mitchell Deposit. Prepared for Seabridge Gold Inc. Submitted May 2012.

Hazen (2008), Comminution Testing (Project # 10724). February 2008.

IGRB (2016). Review of Water Dam, Water Management, and Tailings Storage Systems, KSM Project, British Columbia, Canada. Rev C.1. April 2016.

KCB (2009). 2008 KSM Site Investigation Report.

KCB (2010). 2009 KSM Site Investigation Report.

KCB (2011). 2010 KSM Site Investigation Report.

KCB (2012a). 2012 Site Investigation Report for the Mine Area.

KCB (2012b). 2012 TMF Site Investigations.

KCB (2012c). Engineering Design Update of Tailing Management Facility.

KCB (2012d). KSM Water Storage Dam - Value Engineering Study Report.

KCB (2013a). 2012 Geotechnical Design of Rock Storage Facilities and Design of Associated Water Management Facilities (Technical Report in Support of 2012 PFS), January 29, 2013.

KCB (2013b). KSM Project. Rock Storage Facilities Design Report. July 8, 2013.

KCB (2014). KSM Project Mitchell/McTagg RSF Water Collection for Selenium Treatment. March 4, 2014.

KCB (2016). Best Available Technology (BAT) Study for Tailing Management at the KSM Project. June 22, 2016.

Köeppern Machinery Australia Pty Ltd. (2010). High Pressure Comminution Test Work on Processing of Mitchell Zone Ore. February 2010.

Lechner, M.J. (2007). Mitchell Creek Technical Report, Northern British Columbia, NI 43-101 Technical Report prepared for Seabridge Gold.

Lechner, M.J. (2008a). Kerr-Sulphurets Technical Report, Northern British Columbia, NI 43-101 Technical Report prepared for Seabridge Gold.

Lechner, M.J. (2008b). Updated Mitchell Creek Technical Report, Northern British Columbia, NI 43-101 Technical Report prepared for Seabridge Gold.

Lechner, M.J. (2009). Updated KSM Mineral Resources, NI 43-101 Technical Report prepared for Seabridge Gold.

Lechner, M.J. (2010). January 2010 Updated KSM Mineral Resources, NI 43-101 Technical Report prepared for Seabridge.

Lechner, M.J. (2011). March 2011 Updated KSM Mineral Resources, NI 43-101 Technical Report prepared for Seabridge.

Lechner, M.J. (2014). Initial Deep Kerr Resource Estimate, British Columbia, Canada, NI 43-101 Technical Report.

Metal Mine Effluent Regulations, SOR/2002-222.

Mines Act, RSBC. C. 293. (1996b).

Nordic Minesteel Technologies, 2016, KSM – MTT Automated Rail Ore Transportation System, May 2016.

Piteau Associates (1991). Mine Rock and Overburden Piles Investigation and Design Manual: Interim Guidelines. Prepared for the British Columbia Mines Waste Rock Pile Research Committee by Piteau Associates Engineering Ltd. http://www.empr.gov.bc.ca/Mining/Permitting-Reclamation/Geotech/Documents/MinedRock+OverburdenPiles/MinedRockOverburdenPile_Investigation+DesignManual.pdf (accessed January 2013).

Placer Dome Inc. (1990). Metallurgical Research Centre, Kerr Project Report No.1. October 1990.

Placer Dome Inc. (1991). Metallurgical Research Centre, Kerr Project Report No.2. April 1991.

Pocock (2009). Flocculant Screening, Gravity Sedimentation, Pulp Rheology, Vacuum Filtration and Pressure Filtration Studies. December 2009.

Pretium (2011). Snowfield Resource Increase, press release dated February 23, 2011.

Rescan (2013). Application for an Environmental Assessment Certificate/Environmental Impact Statement for the KSM Project. May 2013.

SGS Minerals Services (2010). An Investigation into the Grindability and Flotation Characteristics of Two Samples From the KSM Deposit. February 2010.

SGS Minerals Services (2010). An Investigation into the Recovery of Cyanide and Detoxification of Leach Tailing From Cyanidation of KSM Project Samples. February 2010.

Surface Science Western, the University Of Western Ontario (2010). Sub-microscopic Gold determination of Cyanide Leach Feeds. December 21, 2010.

Surface Science Western, the University Of Western Ontario (2016). Sub-microscopic Gold Determination of Leach Residues. August 30, 2016.

Tetra Tech (2012). 2012 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study. Document No. 1252880100-REP-R0001-05. Prepared for Seabridge Gold Inc. June 22, 2012.

Tetra Tech (2014). Feasibility Study and Technical Report Update on the Brucejack Project, Stewart, BC. Document No. 1491990100-REP-R0001-01. June 19, 2014.

Wardrop (2008). Kerr-Sulphurets-Mitchell Preliminary Economic Assessment 2008, report written for Seabridge and later filed as NI 43-101 Technical Report, December 19, 2008.

Wardrop (2011). Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update 2011, report written for Seabridge and later filed as NI 43-101 Technical Report, June 15, 2011.

Water Act, RSBC. C. 483. (2003).

CERTIFICATE OF QUALIFIED PERSON

I, R.W. Parolin, of Prince George, British Columbia, do hereby certify:

- I am a Senior Project Engineer with McElhanney Consulting Services Ltd. with a business address at 12-556 North Nechako Road, Prince George, British Columbia, V2K 1A1.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (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 35 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 Sections 1.17.1, and 18.13 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 sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 1st day of November, 2016 at Prince George, British Columbia.

"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, Derek Kinakin, M.Sc., P.Geo., P.G., of Kamloops, British Columbia, do hereby certify:

- I am a Senior Engineering Geologist with BGC Engineering Inc. with a business address at 234 St. Paul Street, Kamloops, British Columbia, V2C 6G4.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (the "Technical Report").
- I am a graduate of Simon Frasier University (B.Sc. (Hons), 2002; M.Sc., 2005).
- I am a member in good standing of Association of Professional Engineers and Geoscientists of British Columbia (#32720).
- My relevant experience includes 14 years of rock mechanics research, slope stability assessments, and slope designs for open pit mines in Canada, US, and Africa.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was August 19th to 21st, 2013.
- I am responsible for Sections 1.16, 16.2.5, and 26.2.1 (pit slopes) of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had no prior experience with the Property that is the subject of this Technical Report.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 1st day of November, 2016 at Kamloops, British Columbia.

"Original document signed and sealed by
Derek Kinakin, M.Sc., P.Geo., P.G."

Derek Kinakin, M.Sc., P.Geo., P.G.
Senior Engineering Geologist
BGC Engineering Inc.

CERTIFICATE OF QUALIFIED PERSON

I, J. Graham Parkinson, P.Geo., of Vancouver, British Columbia, do hereby certify:

- I am a Geoscientist with Klohn Crippen Berger Ltd. with a business address at #500-2955 Virtual Way, Vancouver, British Columbia, V7M 4X6.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (the "Technical Report").
- I am a graduate of the University of British Columbia, (Bachelor's 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).
- My relevant experience includes 20 years with Klohn Crippen and Klohn Crippen Berger engaged in the evaluation and development of mine waste and water management 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 of 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 June 6 to 8, 2016, as well as during 2007, 2008, 2009, 2012 and 2014.
- I am responsible for Section 1.14, 18.2, 24.18.1 (water management and tailings management), 25.8, 26.2.2, 26.2.3 (water storage dam and mine site water management), 26.2.4, 26.2.5 (geotechnical), and 27.0 (only references from sections for which I am responsible) 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, the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011 and the KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study" dated June 22, 2012 .
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 31st day of October, 2016 at Vancouver, British Columbia.

"Original document signed and sealed by J. Graham Parkinson, P.Geo."

J. Graham Parkinson, P.Geo.
Senior Geoscientist
Klohn Crippen Berger Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Hassan Ghaffari, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Director of Metallurgy with Tetra Tech WEI Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, BC, V6C 1N5.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#30408).
- My relevant experience includes 25 years of experience in mining and plant operation, project studies, management, and engineering.
- I am a "Qualified Person" for the purposes of National Instrument 43-101 (the "Instrument").
- I conducted a personal inspection of the Property on September 20, 2014.
- I am responsible for Sections 1.1, 1.2, 1.4, 1.10, 1.17.3, 1.18, 1.19, 1.21, 1.23.1, 2.1, 2.2.1, 2.3, 2.4 2.5, 3.1, 18.1, 18.5, 18.7, 18.8, 18.9, 18.10, 18.15, 18.16, 18.17, 21.1 and 21.2 (all other capital costs except for open pit mining, underground mining, permanent electrical power supply and distribution, energy recovery, NTL contribution and MTDT mini hydro generation station costs), 22.0, 25.1, 25.9, 25.11.3, 26.1, and 27.0 (only references from sections for which I am responsible) 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 "KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study" dated June 22, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 3rd day of November, 2016 at Vancouver, British Columbia

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

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

CERTIFICATE OF QUALIFIED PERSON

I, Jianhui (John) Huang, Ph.D., P.Eng., of Coquitlam, British Columbia, do hereby certify:

- I am a Senior Metallurgist with Tetra Tech WEI Inc. with a business address at Suite 1000, 10th Floor, 885 Dunsmuir Street, Vancouver, British Columbia, V6C 1N5.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (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 30 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 1.12, 1.13, 1.20, 3.1, 13.0, 17.0, 18.6, 19.0, 21.3 (excluding open pit and underground mining cost), 24.19, 25.7, 25.11.4, 26.2.6, and 27.0 (only references from sections for which I am responsible) 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, the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011 and "2012 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study" dated June 22, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 3rd day of November, 2016 at Vancouver, British Columbia.

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

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

CERTIFICATE OF QUALIFIED PERSON

I, James H. Gray, P.Eng., of Calgary, Alberta, do hereby certify:

- I am a Mining Engineer with Moose Mountain Technical Services with a business address at #210 1510 2nd Street North, Cranbrook, British Columbia, V1C 3L2.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (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), Engineers Geoscientists New Brunswick (#L5018), and the Association of Professional Engineers, Geologists, and Geophysicists of Alberta (#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 September 25th, 2008; September 10th, 2009; and April 13th, 2010.
- I am responsible for Sections 1.11 (open pit mining), 1.17.4, 1.17.5, 15.1, 15.2, 15.4 (open pit mining), 16.1, 16.2 (except 16.2.5), 16.4 (open pit mining), 18.3, 18.4, 21.1 (open pit mining costs), 21.2 (open pit mining costs), 21.2.3 (open pit mining costs) 25.2.2, 25.3, 25.5, 25.6, 25.11.1, 25.11.2, 26.2.1 (open pit and reserves), 26.2.5 (tunnel design), 26.2.7, and 27.0 (only references from sections for which I am responsible) of the Technical Report.
- I am independent of Seabridge Gold Inc. as defined by Section 1.5 of the Instrument.
- I have had previous 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, the "Kerr-Sulphurets-Mitchell (KSM) Prefeasibility Study Update" dated June 15, 2011, and the "2012 KSM (Kerr-Sulphuretes-Mitchell) Prefeasibility Study" dated June 22, 2012 and amended November 11, 2014.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 3rd day of November, 2016 at Vancouver, British Columbia.

"Original document signed and sealed bu
James H. Gray, P.Eng."

James H. Gray, P.Eng.
Principal Mining Engineer
Moose Mountain Technical Services

CERTIFICATE OF QUALIFIED PERSON

I, Kevin Jones, P.Eng., of St. Albert, Alberta, do hereby certify:

- I am the Vice President – Arctic Development with Tetra Tech with a business address at 14940, 123 Ave NW, Edmonton, Alberta, T5V 1B4.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (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 British Columbia (#L38044), Association of Professional Engineers and Geoscientists of Alberta (#M34713) and the Northwest Territories (#L341).
- My relevant experience includes over 32 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 two mining projects in Nunavut (over 100 km each) 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 1.17.2 and 18.14 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, and the "KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study" dated June 22, 2012.
- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 1st day of November, 2016 at Edmonton, Alberta.

"Original document signed and sealed by Kevin Jones, P.Eng."

Kevin Jones, P.Eng.
Vice President – Arctic Development
Tetra Tech

CERTIFICATE OF QUALIFIED PERSON

I, Michael J. Lechner, P.Geo., RPG, CPG, 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 “2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment”, with an effective date of October 6, 2016 (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 American Institute of Professional Geologists (#10690), a Registered Member of the Society of Mining, Metallurgy and Exploration (#4124987) and professional geologist with the Association of Engineers and Geoscientists of British Columbia (#155344).
- My relevant experience includes over 36 years of work as an exploration geologist, mine geologist, engineering superintendent, and resource estimator for a variety of precious metal and base 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 September 11th to September 14th, 2015.
- I am responsible for Sections 1.5, 1.6, 1.7, 1.8, 1.9, 3.1, 4.0, 5.0, 6.0, 7.0, 8.0, 9.0, 10.0, 11.0 12.0, 14.0, 23.0, 25.2.1, 26.2.8, 26.3.2, and 27.0 (only references from sections for which I am responsible) 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
  - "NI 43-101 Technical Report of Initial Deep Kerr Resource, British Columbia, Canada", dated March 31, 2014.

- I have read the Instrument and the sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 3rd day of November, 2016 at Stites, Idaho.

"Original document signed and sealed by
Michael J. Lechner, P.Geo., RPG, CPG"

Michael J. Lechner, P.Geo., RPG, CPG
President
Resource Modeling Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Neil Brazier, P.Eng., of Richmond, British Columbia, 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 "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (the "Technical Report").
- I am a graduate of the University of Saskatchewan (B.Sc. Electrical Engineering, 1969).
- 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 1 to 4, 2013.
- I am responsible for Sections 1.17.6, 18.11, 18.12, and portions of Section 21.1 and 21.2 related to permanent electrical power supply and distribution, energy recovery, NTL contribution, and MTDT mini hydro generation station costs 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 sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 3rd day of November, 2016 at Vancouver, British Columbia.

"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, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am an Environmental Engineer with ERM Consultants Canada Ltd. with a business address at 1500-1111 West Hastings St., Vancouver, British Columbia, V6E 2J3.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (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 (#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 1.15, 20.0, 24.20, 26.2.3 (water balance, temporary water treatment plants, and geochemistry database), and 27.0 (only references from sections for which I am responsible) 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 sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 3rd day of November, 2016 at Vancouver, British Columbia.

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

Pierre Pelletier, P.Eng.
Division Managing Director, Canada
ERM Consultants Canada Ltd.

CERTIFICATE OF QUALIFIED PERSON

I, Ross David Hammett, Ph.D., P.Eng., of Burnaby, British Columbia, 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, British Columbia, Canada, V5C 6C6.
- This certificate applies to the technical report entitled "2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment", with an effective date of October 6, 2016 (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 (# 11020).
- I have 40 years of experience in mining and civil engineering. I have provided 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 1.11 (underground mining), 15.1, 15.3, 15.4 (underground mining), 16.1, 16.3, 16.4 (underground mining), 21.1 (costs pertaining to underground mining), 21.2 (costs pertaining to underground mining), 21.3 (costs pertaining to underground mining), 24.16.3, 24.16.4, 24.18.4, 24.18.5, 25.4, 25.11.5, 25.12.1, 26.2.1 (underground), 26.3.1 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 sections of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the section of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 3rd day of November, 2016 at Burnaby, British Columbia.

"Original document signed and sealed by
Ross D. Hammett, Ph.D., P.Eng."

Ross D. Hammett Ph.D, P.Eng
Senior Engineer & Principal
Golder Associates Ltd.

CERTIFICATE OF QUALIFIED PERSON

- I, Simon Allard, am employed as a Principal Consultant and Study Manager with Amec Foster Wheeler Americas Limited with a business address at 400 – 111 Dunsmuir Street, Vancouver, BC, V6B 5W3, Canada.
- This certificate applies to the technical report entitled “2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment”, with an effective date of October 6, 2016 (the “Technical Report”).
- I am a graduate of Université Laval with a Baccalaureat cooperatif en génie des mines et de la minéralurgie Degree in 2004.
- I am a member in good standing with the Association of Professional Engineers and Geoscientists of British Columbia, Canada.
- I have practiced my profession for 12 years in the mining industry. My relevant experience includes cash flow modelling, risk evaluation, financial analysis, marketing studies, mine planning, and mining study supervision.
- I am a “Qualified Person” for those sections of the Technical Report that I am responsible, for the purposes of National Instrument 43-101 (NI 43-101).
- I have not completed a personal inspection of the Property.
- I am responsible for Section 1.3, 1.4, 1.22, 1.23.2, 2.2, 2.2.2, 2.3, 2.4, 3.2, 24.0, 24.16.1, 24.18, 24.18.1, 24.18.1(Open Pit Rock Storage Facility, MTT Rail System, Facilities-Buildings and Services), 24.18.2, 24.21.0, 24.21.1, 24.21.1(Basis of the Estimate, Indirect and Contingency, Owners Capital Costs, Sustaining Capital), 24.21.2, 24.21.2(Baisis of the estimate, Open Pit Mine Operating Costs, Infrastructure Operating Costs), 24.21.3, 24.22, 25.1, 26.3, 27 of the Technical Report.
- I am independent of Seabridge Gold Inc. as independence is defined by Section 1.5 of NI 43-101.
- I have worked on a review of the 2012 PFS study in 2014.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3th day of November, 2016.

“signed and sealed”

Simon Allard, P.Eng.

Amec Foster Wheeler
111 Dunsmuir St, Suite 400,
Vancouver, BC,
Canada, V6B 5W3
Tel: +1 604-664-4315

CERTIFICATE OF QUALIFIED PERSON

- I, Ignacy (Tony) Lipiec, am employed as Director, Process Engineering with Amec Foster Wheeler Americas Limited with a business address at 400 – 111 Dunsmuir Street, Vancouver, BC, V6B 5W3, Canada.
- This certificate applies to the technical report entitled “2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment”, with an effective date of October 6, 2016 (the “Technical Report”).
- I am a graduate of the University of British Columbia with a B.A.Sc. degree in Mining & Mineral Process Engineering, in 1985.
- I am a member in good standing with the Association of Professional Engineers and Geoscientists of British Columbia, Canada.
- I have practiced my profession for 31 years in the mining industry. My relevant experience includes metallurgical design and process engineering for base metal and precious metal projects in North and South America. My project work and work experience includes Escondida, Pebble, Red Chris, Mount Milligan, Huckleberry, Bell Copper, Equity Silver and Mount Polley.
- I am a “Qualified Person” for those sections of the Technical Report that I am responsible, for the purposes of National Instrument 43-101 (NI 43-101).
- I have not completed a personal inspection of the Property.
- I am responsible for Section 1.3, 1.22, 1.23.2, 3.2, 24.17, 24.21.0, 24.21.1, 24.21.1(Basis of Estimate, Sustaining Capital), 24.21.2, 24.21.2(Basis of the Estimate, Process Operating Costs, G&A Operating Costs), 24.21.3, 25.1, 25.12.2, 26.3.4, 27 of the Technical Report.
- I am independent of Seabridge Gold Inc. as independence is defined by Section 1.5 of NI 43-101.
- I was involved in a review of the metallurgical test program on the KSM Project in 2014 and made recommendations on additional testwork.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3th day of November, 2016.

“signed and sealed”

Ignacy (Tony) Lipiec, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

- I, Mark Ramirez, am employed as a Principal Mining Engineer with Amec Foster Wheeler E&C Services Inc. with a business address at Suite 2-1000, 2000 South Colorado Blvd, Denver, CO, 80222, USA.
- This certificate applies to the technical report entitled “2016 KSM (Kerr-Sulphurets-Mitchell) Prefeasibility Study Update and Preliminary Economic Assessment”, with an effective date of October 6, 2016 (the “Technical Report”).
- I am a graduate of the Colorado School of Mines with a Bachelor of Science Degree in Mining Engineering, in 1997.
- I am in good standing as a Registered Member of the Society for Mining, Metallurgy, and Exploration.
- I have practiced my profession for 20 years in the mining industry. My relevant experience includes a decade as a mining engineer at Freeport’s Henderson and Grasberg underground mines, consulting studies on underground studies for the expansion at Bingham Canyon in Utah, and the Maturi mine in Minnesota, and the Fruta del Norte project in Ecuador. I provided in-depth senior consulting services for Codelco’s El Teniente New Mine Level and Chuquicamata Underground caving mine projects in Chile. I worked full time on the value engineering, feasibility study and detailed engineering for the 95,000 tpd block caving development at Oyu Tolgoi.
- I am a “Qualified Person” for those sections of the Technical Report that I am responsible, for the purposes of National Instrument 43-101 (NI 43-101).
- I have not completed a personal inspection of the Property.
- I am responsible for Section 1.3, 1.22, 1.23.2, 3.2, 24.16.2, 24.18, 24.18.3, 24.21.0, 24.21.1, 24.21.1(Basis of the Estimate, Sustaining Capital), 24.21.2, 24.21.2(Basis of the Estimate, Underground Mine Operating Costs), 24.21.3, 25.1, 25.12.1, 26.3.3, 27 of the Technical Report.
- I am independent of Seabridge Gold Inc. as independence is defined by Section 1.5 of NI 43-101.
- I was briefly involved with some scoping level mining exercises in 2014 for Deep Kerr.
- I have read NI 43-101 and the sections of the Technical Report that I am responsible for have been prepared in compliance with NI 43-101.
- As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the sections of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 3th day of November, 2016.

“signed and sealed”

Mark Ramirez, RM SME