Report to:

Abacus
MINING & EXPLORATION CORP.

Ajax Copper/Gold Project – Kamloops, British Columbia
Feasibility Study Technical Report

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WARDROP
A TETRA TECH COMPANY

Suite 800, 555 West Hastings Street, Vancouver, British Columbia V6B 1M1
Phone: 604-408-3788 Fax: 604-408-3722 E-mail: vancouver@wardrop.com

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AJAX COPPER/GOLD PROJECT – KAMLOOPS, BRITISH COLUMBIA FEASIBILITY STUDY TECHNICAL REPORT

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Prepared by: Hassan Ghaffari, P.Eng.
Andre De Ruijter, P.Eng.
Kenneth Lee, P.Eng.
Amir Karami, P.Eng.
Ali Farah, P.Eng.
Ting Lu, P.Eng.
Timothy O. Kuhl, R.M. SME
Ramon Mendoza Reyes, P.Eng.
Warren Newcomen, P.Eng.
Ken Brouwer, P.Eng.
Chris Lee, P.Eng.
Irwin Wislesky, P.Eng.

HG/jc

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NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Abacus Mining & Exploration Corp. (Abacus) by Wardrop, a Tetra Tech Company (Wardrop), AMEC Americas Limited (AMEC), Knight Piésold Ltd. (Knight Piésold), Golder Associates Ltd. (Golder) and BGC Engineering Inc. (BGC). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Wardrop's, AMEC's, Knight Piésold's, Golder's and BGC's 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 subject to the terms and conditions of Abacus' contract with Wardrop, AMEC, Knight Piésold, Golder and BGC. This contract permits Abacus to file this report as a Technical Report with Canadian Securities Regulatory Authorities pursuant to National Instrument 43-101, *Standards of Disclosure for Mineral Projects*. Except for the purposes legislated under provincial securities laws, any other use of this report by any third party is at that party's sole risk.

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GLOSSARY

UNITS OF MEASURE

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	Cdn$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t

Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a

Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

Abbreviations and Acronyms

Abacus Mining & Exploration Corp.	Abacus
Absolute Value of Relative Difference	AVRD
acid rock drainage	ARD
Afton Mines Ltd.	Afton Mines
Afton Operating Company	Afton OC
Application for an Environmental Assessment Certificate	Application
Application Information Requirements	AIR
AMEC Americas Limited	AMEC
atomic absorption spectrophotometer	AAS

atomic absorption	AA
Berens River Mines Ltd.	Berens
BGC Engineering Inc.	BGC
Bond Abrasion Index	Ai
Bond Ball Mall Work Index	BWi
British Columbia	BC
British Columbia Forestry Service	BCFS
British Columbia Water Quality Guideline	BCWQG
Bunchgrass	BG
Canadian Council of Ministers of the Environment	CCME
Canadian Institute of Mining, Metallurgy and Petroleum	CIM
Canadian Environmental Assessment Agency	CEAA
Canadian National Railway	CNR
Canadian Pacific Railway	CPR
capital cost estimate	CAPEX
carbon dioxide	CO_2
Certified Reference Materials	CRMs
chloride acid	HCl
closed circuit television	CCTV
Cominco Ltd.	Cominco
Consolidated Mining and Smelting Company of Canada Ltd.	CM&S
Construction Management	CM
Coquihalla Highway	Highway 5
copper equivalent	CuEq
copper	Cu
Cost of Risk	CR
Criteria Air Contaminants	CACs
Crusher Work Index	CWi
diamond drill holes	DDH
Distributed Control System	DCS
east waste dump	EWD
Environmental Assessment	EA
Environmental Assessment Office	EAO
Environmental Impact Assessment	EIA
Environmental Impact Statement	EIS
Environmental Management Plan	EMP
Environmental Management System	EMS
E&B Explorations Ltd.	E&B
Eco Tech Laboratories Ltd.	Eco Tech
Exploration data analysis	EDA
free carrier	FCA
free board marine	FOB
front-end loaders	FELs
G&T Metallurgical Services Ltd.	G&T
general and administrative	G&A
general arrangement	GA

Geostatistical Software Library	GSLIB
global positioning system	GPS
gold	Au
Golder Associates Ltd.	Golder
Granby Consolidated Mining, Smelting, and Power Company Ltd.	Granby
Health, Safety and Environmental	HS&E
heating, ventilation, and air conditioning	HVAC
High Pressure Grinding Rolls	HPGR
Impact Benefits Agreement	IBA
Induced polarization	IP
inductively coupled plasma	ICP
inductively coupled plasma-atomic emission spectroscopy	ICP-AES
inlet/outlet	I/O
in-pit crushing and conveying	IPCC
in-pit crushing, conveying and waste stacking	IPCC/S
Interior Douglas Fir	IDF
internal rate of return	IRR
inverse distance weighting to the fourth power	IDW^4
Iron Mask Hybrid	IMH
joint venture	JV
KGHM Ajax Mining Inc.	KAM
KGHM Polska Miedź S.A.	KGHM
Knight Piésold Ltd.	Knight Piésold
Lerchs-Grossman	LG
life-of-mine	LOM
Land and Resource Management Plan	LRMP
Liquid Limit	LL
Liquidity Index	LI
Major Projects Management Office	MPMO
Material Safety Data Sheet	MSDS
Measured & Indicated	M&I
Mintec MineSight®	MineSight®
mercury	Hg
methyl isobutyl carbinol	MIBC
mining cost adjustment formulas	MCAF
Ministry of Environment	MOE
Moisture Content	MC
motor control centres	MCCs
National Instrument 43-101	NI 43-101
National Topography System	NTS
nearest neighbour	NN
Neil S. Seldon & Associates Ltd.	NSA
net present value	NPV
net smelter return	NSR
New Gold Inc.	New Gold
Nicola Volcanics	NVV

nitric acid	HNO_3
non-acid generating	NAG
north waste dump	NWD
Object-Oriented Input Systems	OISs
operating cost estimate	OPEX
ordinary kriging	OK
piping and instrumentation diagrams	P&IDs
Plastic Limit	PL
Philips Enterprises, LLC	Philips
Plasticity Index	PI
Ponderosa Pine	PP
process flowsheet diagram	PFD
positive displacement	PD
potassium amyl xanthate	PAX
Preliminary Economic Assessment	PEA
probability-assisted constrained kriging	PACK
probable maximum precipitation	PMP
Project Execution Plan	the Plan
programmable controller	PC
programmable logic controller	PLC
qualified persons	QPs
quality assurance/quality control	QA/QC
quantile-quantile	QQ
reverse circulation	RC
rock quality designation	RQD
Rotating Biological Contactor	RBC
run-of-mine	ROM
semi-autogenous grinding	SAG
silver	Ag
Species at Risk Act	SARA
specific gravity	SG
Standard Reference Materials	SRMs
Sugarloaf Ranches Limited	Sugerloaf
Sugarloaf diorite	SLD
tailings storage facility	TSF
Teck Resources Limited	Teck
Thickened Tailings Plant	TTP
Trans-Canada Highway	Highway 1
Trans Mountain pipeline	TMPL
Transportation Association of Canada	TAC
three-dimensional	3D
three-dimensional block model	3DBM
Treatment and refining charges	TC/RC
Voice-over Internet Protocol	VoIP
Wardrop, A Tetra Tech Company	Wardrop
waste rock storage facility	WRF

water	H_2O
Work Breakdown Structure	WBS
Workplace Hazardous Materials Information Systems	WHMIS
x-ray fluorescence spectrometer	XRF

1.0 SUMMARY

1.1 INTRODUCTION

The Ajax Project, located south of downtown Kamloops, British Columbia (BC), will be a 60,000 t/d open pit operation. Ore will be processed in a conventional milling plant, and copper/gold (Cu/Au) concentrate will be transported to the Port of Vancouver for shipment to offshore smelters. The proposed Ajax Copper/Gold Project is currently 100% owned by KGHM Polska Miedź S.A. (KGHM) Ajax which is a joint venture (JV) company owned 51% by KGHM and 49% by Abacus Mining & Exploration Corp. (Abacus).

In May 2010, Abacus commissioned a team of engineering consultants to complete a feasibility study in accordance with National Instrument 43-101 (NI 43-101). All mines acts regulations with respect to health, safety and environmental considerations have been taken into account and incorporated into the feasibility designs and relevant cost estimates. In addition, the designs take into account the geological location of the Project. The following consultants were commissioned to complete the component reports for the purposes of the feasibility study:

- Wardrop, a Tetra Tech Company (Wardrop) – overall management, mineral processing, infrastructure, and financial analysis
- AMEC Americas Ltd. (AMEC) – geology, Mineral Resource estimate, mine design, and Mineral Reserve estimate
- Golder Associates Ltd. (Golder) – tailings handling, thickening and tailings area water management
- Knight Piésold Ltd. (Knight Piésold) – environmental studies, permitting, and social or community impact
- BGC Engineering Inc. (BGC) – pit slope designs, pit dewatering evaluations and site geotechnical evaluations excluding the tailings storage facility (TSF).

In addition, G&T Metallurgical Services Ltd. (G&T) conducted the metallurgical test work, and Krupp Polysius performed High Pressure Grinding Rolls (HPGR) pilot test work for the process design.

For the purposes of this study, all currencies are expressed in US dollars, unless otherwise specified.

1.2 PROJECT HISTORY

Exploration in the Ajax area began in the 1880s and continued intermittently until the 1980s. In the 1980s, Afton Operating Company (Afton OC) defined a mineral resource. Mining operations began in 1989 and were suspended in 1991 due to low metal prices. A second period of production began in 1994 and was again suspended in 1997. During the periods of production, it is estimated that 17 Mt were mined and 13 Mt were milled.

Abacus acquired holdings in the Ajax area in 2002 from Teck. Abacus has explored the Ajax property with diamond drillhole (DDH) methods from 2005 to 2010. In 2009 Wardrop completed a NI 43-101 compliant resource estimate and a positive preliminary economical assessment (PEA) for the Ajax area (Ghaffari et al, 2009). Abacus drill campaigns comprise over 86% of the drill data in the 2011 resource model database. Details of this drilling are provided in Section 10.0 in this report.

1.3 DEPOSIT GEOLOGY

Abacus prepared the regional and property geology information as shown in Section 7.0. In general three main rock units are combined from the identified 22 rock types in the Ajax area. These are composed of Iron Mask Hybrid, Sugarloaf Diorite, and Nicola Volcanics. Sugarloaf Diorite is characteristically a fine to coarse-grained, light to medium gray porphyritic diorite containing euhedral hornblende phenocrysts. The Iron Mask Hybrid is considered to be an assimilation of the Nicola Group into the intruding Pothook Diorite. The Iron Mask is coarse-grained and dioritic to gabbroic in composition. The Nicola Group consists of picrite and various fine-grained and pyroxene porphyritic mafic volcanic rocks.

The mineralization in the Ajax area is associated with structural corridors of highly fractured sections of Sugarloaf and Sugarloaf Hybrid phases of the Iron Mask Batholith. Chalcopyrite is the dominant copper mineral and occurs as veins, veinlets, fracture fillings, disseminations, and isolated blebs in the host rock. Concentrations of chalcopyrite rarely exceed 5%. Accessory sulphide minerals include pyrite, magnetite, molybdenite, and occasionally bornite.

1.4 GEOLOGY AND MINING

The results of the Mineral Resource estimate and Mineral Reserve estimate represent forward-looking information that are 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 information includes commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and risks regarding the cost estimates and completion schedule for the proposed Project infrastructure, in particular the need to obtain permits and governmental approvals.

1.4.1 DATA VERIFICATION

AMEC audited the Abacus drill database in 2008-2009, 2010, and 2011. Although minor errors were noted with the data, AMEC's conclusion was that the data was sufficient to support Mineral Resource and Mineral Reserve estimation. A block confidence restriction was placed on the use of legacy data in estimation of Measured and Indicated blocks.

1.4.2 MINERAL RESOURCE ESTIMATE

The Ajax West, Ajax East and Ajax East Extension areas were modelled. Assay data were composited to 12 m lengths.

Abacus provided AMEC with a 0.1% copper grade shell model for the West, East and East Extension areas. AMEC used the copper mineralization shell as an exploration data analysis (EDA) boundary and also the boundary for the copper and gold estimation. AMEC applied a probability-assisted constrained kriging (PACK) methodology to further define the high-grade copper and gold domains within the mineralized shell provided by Abacus. This controls the smearing of high-grade mineralization into low-grade areas.

The geological model was coded from the geological solids.

The Ajax density database contains 855 specific gravity (SG) determinations. The SG data were coded with the rock code (RCODE) from the MineSight® composite file.

The estimation methodology for the 2011 resource model update was inverse distance weighting to the fourth power (IDW^4). All estimations were completed by rock type. All rock boundaries were considered hard for estimation purposes. An outlier restriction was implemented on uncapped 12 m composites to address metal-at-risk. The model estimate was completed in three passes with expanding searches for each pass.

A classification of the Ajax model was developed based on the copper mineralization shell, grade continuity observed in cross-section and plan-section, and confidence limit calculations using a 60,000 t/d production schedule. To aid in the classification, the percentage of influence legacy drillholes had on the copper grade estimate was determined. The influence for the J-series drillholes (percussion drilling) was determined independently from other legacy drillholes: 87, 88, 89, and 90 series drillholes (core drilling). Where J series percussion holes contribute more than 60% of the weight used to make a copper grade estimate, the block is downgraded to Inferred. Blocks where legacy core holes contribute more than 50% of the weight used to make a copper grade estimate, Measured blocks are downgraded to Indicated.

The nominal drill spacing for Measured is 40 to 50 m for the Ajax West and East areas. Confidence limit calculations for the West and East areas suggest drill density for Measured Resource Classification of 35 to 50 m. The confidence limit calculation for the Indicated Resource Classification in the West and East areas suggest a drill density of approximately 50 to 75 m. Using a 0.100% Cu cut-off, 38% of the blocks are classified as Measured, 41% are classified as Indicated, and 21% are classified as Inferred.

A net smelter return (NSR) was calculated for each block using a NSR script in Gemcom software. To determine the reasonable expectations for economic extraction, a Lerchs-Grossmann (LG) pit optimization was completed using blocks classified as Measured, Indicated and Inferred. The LG parameters were based on a copper price of US$2.88/lb and a gold price of US$1,200/oz. The LG optimization was completed by Abacus personnel using Gemcom Whittle™ 4D (Version 4.1.3). Based on preliminary economic analyses, the net value of the resource shell exceeds capital cost estimates.

A copper equivalency (CuEq) grade for reporting of Mineral Resources was calculated using the following formula:

$$CuEq = \frac{(\%Cu)\,(CuRec)\,(22.0462)\,(\$lbCu) + (g/t/Au)\,(AuRec)\,(1/31.1035)\,(\$ozAu)}{(CuRec)\,(22.0462)\,(\$lbCu)}$$

Mineral Resources take into account geological, mining, processing and economic constraints, and have been confined within appropriate Lerchs–Grossmann (LG) pitshells, and therefore are classified in accordance with the 2010 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves.

The Qualified Person for the Mineral Resource estimate is Timothy O. Kuhl, SME Registered Member, an employee of AMEC.

Mineral Resources are reported using a copper price of US$2.88/lb and a gold price of US$1,200/oz, and have an effective date of May 26, 2011. Mineral Resources are summarized in Table 1.1 and reported at a Base Case CuEq grade of 0.20%.

Table 1.1 Ajax Mineral Resource Estimate, Effective Date May 26, 2011; Timothy O. Kuhl, R.M. SME

	Cut-off CuEq (%)	Tonnes (Mt)	CuEq (%)	Cu (%)	Au (g/t)	NSR (US$/t)	Contained Metal CuEq (Mlb)	Contained Metal Cu (Mlb)	Contained Metal Au (Koz)
Measured	0.20	255.8	0.42	0.31	0.19	15.71	2,389	1,734	1,555
Indicated	0.20	256.2	0.42	0.30	0.20	19.98	2,399	1,712	1,637
Measured + Indicated	0.20	512.0	0.42	0.31	0.19	17.85	4,788	3,446	3,193
Inferred	0.20	73.7	0.38	0.27	0.17	17.46	613	439	406

Note 1. Mineral Resources are contained within a conceptual Measured, Indicated and Inferred optimized pitshell using the following assumptions: maximum copper recovery of 91.17% and maximum gold recovery of 86.49% based on the following equations: CuRec = (-74.812 x (Cu%^2))+(85.727xCu%) +66.668 and AuRec = 92.586 x Au(g/t)^0.064; assumed throughput rate of 60,000 t/d; Whittle constraining shell slopes between pit slope angles ranging from 38º to 49º, waste and processed material mining costs of US$1.08/t, fill waste mining costs of US$0.89/t, total processing costs including reclamation of US$3.23/t, general and administrative costs of US$0.52/t, gold price of US$1,200/oz, and copper price of US$2.88/lb.

Note 2. Copper equivalency was calculated using the formula CuEq = ([(%Cu) x (CuRec) x (22.0462) x ($lbCu) + (g/t/Au) x (AuRec) x (1/31.1035) x ($ozAu)]) ÷ ((CuRec) x (22.0462) x ($lbCu))).

Note 3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

Note 4. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper pounds as imperial pounds.

Factors which may affect the geological models or the conceptual pitshells used to constrain the mineral resources, and therefore the Mineral Resource estimates include: the commodity price assumptions; the NSR value used to constrain the Mineral Resources is based on technical and economic parameters supplied by Abacus, should these assumptions change, then the pit constraining the Mineral Resources will also change; metallurgical recovery assumptions; pit slope angles used to constrain the estimates; and the SG values assumed for the rock types.

1.4.3 MINERAL RESERVES

Mineral Reserves were optimized for all Measured and Indicated blocks assuming a gold price of US$1,085/oz gold and a copper price of US$2.50/lb.

Pit optimization was performed on the mining block model using Gemcom Whittle™ software. AMEC obtained pit optimisation parameters from various sources: copper and gold prices were determined by Abacus and agreed by AMEC, the overall pit slopes were recommended by BGC, metal recoveries and processing costs were derived from work developed by Wardrop, while mining costs were derived from the 2009 PEA mining cost model.

The total NSR was calculated by adding the NSR attributable to copper to the NSR attributable to gold and then subtracting the freight costs, which include land freight, port charges, ocean freight and miscellaneous costs. The ore considered for processing in the optimization was based on a marginal cut-off value of US$4.53/t NSR.

Pitshell generation was based on a Base Case assumption that the Trans Mountain Pipeline, an oil pipeline operated by Kinder Morgan, which runs along the western limit of the proposed Ajax pit could be relocated away from the pit during the first years of operation. However, the pit optimization process and the operational designs was set-up in such a way that no excavation has been designed or scheduled on the ground below the pipeline location and its right of way.

The pit is unconstrained by considerations for infrastructure to the north, east and south. All the major infrastructure facilities planned for the Project: mineral processing facilities, stockpiles, waste dumps, offices, maintenance shops, fuel storage, tailings pond, water storage ponds, will be external to the current ultimate pit design and its area of influence.

Dilution was incorporated into the model using the following formula:

*Diluted Grade = (In-situ Grade * 75%) + ∑ (Dilution Factor * Grade of Neighbour Block)*

Geotechnical domains, design sectors, slope angles, and associated assumptions were provided by BGC, and modified for mining purposes by AMEC. Mine design has incorporated geotechnical and hydrogeological considerations.

The weighted average mining cost used was US$1.32/t and ranged from US$0.92/t to US$2.50/t for the different mining benches. The processing cost was US$3.38/t, and included an allowance of US$0.05/t of ore processed for closure costs, and a general and administrative (G&A) cost of US$0.51/t.

Mineral Reserves were modified from Mineral Resources by taking into account geologic, mining, processing, and economic parameters and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. The Qualified Person for the Mineral Reserve estimate is Ramon Mendoza Reyes, P.Eng., an AMEC employee. The estimate has an effective date of October 31, 2011 and is summarized in Table 1.2.

Table 1.2 Mineral Reserve Statement, Effective Date October 31, 2011, R. Mendoza Reyes, P.Eng.

Confidence	Cut-off	ROM	Average ROM Grades		Copper Equivalent	Contained Metal	
Category	**Grade (US$/t)**	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**CuEq (%)**	**Copper (Mlb)**	**Gold (Koz)**
Proven Mineral Reserve	4.53	279.5	0.27	0.17	0.38	1,680	1,520
Probable Mineral Reserve	4.53	223.5	0.26	0.17	0.37	1,280	1,230
Total Proven & Probable Mineral Reserves	**4.53**	**503.0**	**0.27**	**0.17**	**0.37**	**2,960**	**2,750**

Note 1. Mineral Reserves are estimated using a cut-off grade of US$4.53/t NSR, a copper price of US$2.50/lb, and a gold price of US$1,085/oz. The NSR is calculated by adding the NSR attributable to copper to the NSR attributable to gold and then subtracting the freight costs, which include land freight, port charges, ocean freight and miscellaneous costs. The attributable copper is calculated using the metallurgical recovery obtained by the formula: CuRec (%) = -74.812 * Cu(%)2 + 85.727 * Cu(%) + 66.668 with a maximum copper recovery of 91.17%. The attributable gold is calculated using metallurgical recovery obtained by the formula: AuRec (%) = 92.586 * $Au(g/t)^{0.0649}$ with a maximum gold recovery of 86.49%.

Note 2. Mineral Reserves are constrained within a pitshell, optimised using assumptions of a weighted average mining cost of US$1.32/t (ranging from US$0.92/t to US$2.50/t for the different mining benches); a processing cost of US$3.38/t (including US$0.51/t general and administrative costs, and US$0.05/t allocation for closure costs); and pit slope angles that vary from 40º to 49º.

Note 3. A 0.5% mining loss factor was applied to account for dilution; diluted grades are estimated at 1.7% lower than the in-situ grades.

Note 4. The life-of-mine (LOM), waste to ore strip ratio is 2.42. The assumed LOM throughput rate is 60 kt/d.

Note 5. Copper equivalency was calculated using the formula CuEq = ([(%Cu) x (CuRec) x (22.0462) x ($lbCu) + (g/t/Au) x (AuRec) x (1/31.1035) x ($ozAu)]) ÷ ((CuRec) x (22.0462) x ($lbCu))).

Note 6. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

Note 7. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces; contained copper pounds are Imperial pounds.

Factors which may affect the Mineral Reserve estimate include gold and copper price assumptions, effectiveness of surface and groundwater management, the assumption that granting of appropriate environmental and construction permits would be forthcoming from the relevant authorities, unrecognized structural complications in areas with relatively low drill hole density, changes to the proposed methodology for waste material, and changes to the pitshell shape if the material currently under the oil pipeline can be incorporated in pit design.

1.4.4 PROPOSED MINE PLAN

The proposed mine plan envisages a conventional open pit operation producing 60,000 t/d. The pit has been developed to have seven phases. The longest span of the pit will be approximately 2.6 km in an east-west direction and approximately 1.4 km in a north-south direction.

The mine plan is based on the extraction of 503 Mt of ore for processing during 23 years of operation at an overall stripping ratio of 2.4:1 waste to ore. Total material movement from the pit during the life of the mine is estimated at 1,701 Mt. The average head grade of process feed for the LOM is 0.267% Cu and 0.170 g/t Au equivalent to an NSR of 14.68 expressed in dollars per tonne.

An elevated cut-off strategy will be employed. This will result in four stockpiles, three of which are planned to be treated. The fourth stockpile represents potential Project upside and may be sent to process in times of higher commodity prices.

A conventional truck and shovel fleet will be used, in conjunction, from Year 7, with an in-pit crush and convey system. Mining will proceed on 12 m benches. Drill-and-blast will be required. Horizontal drains are proposed as the primary means to depressurize potential bench scale wedge and planar failures. In-pit water will be removed by way of ditches, pipes, sumps, pumps, and booster pumps.

Two waste rock storage facilities are planned. One will be constructed at the north side of the pit and the second will be located at the east side of the pit. A preliminary program to investigate the potential for acid drainage of the waste rocks is underway.

1.5 MINERAL PROCESSING AND RECOVERY METHODS

The process plant design for this project was developed from information provided by Wardrop and other consultants in the July 2009 report titled "NI 43-101 Preliminary Assessment Technical Report on the Ajax Copper/Gold Project" and supplementary metallurgical test work conducted by G&T from 2008 to 2011. The information available from the earlier work (1970 to 2002) was also reviewed and reinforced by supplementary data developed from 2002 to 2008. G&T conducted metallurgical test work to develop comminution and flotation studies to be used as the basis for mill process design. Samples from the diamond drill cores from exploration work have been used for analysis and extensive metallurgical tests. Section 13.0 presents the historical and the most recently completed test work review for the Ajax mineralization.

The Ajax concentrator has been designed to process a nominal 21,900,000 t/a, or 60,000 t/d, of copper-gold porphyry ore from an open pit operation. The concentrator has been designed to produce a marketable copper concentrate of 25% Cu containing approximately 18 g/t Au. The treatment plant will consist of stage-wise crushing and grinding, followed by a flotation process to recover and upgrade copper from the feed material. A gravity circuit will be included within the flotation circuit to enhance gold recovery. The flotation concentrate will be thickened and filtered and sent to the concentrate stockpile for subsequent shipping to smelters.

1.6 TAILINGS STORAGE FACILITY AND OTHER INFRASTRUCTURE

Golder Paste Technology Ltd., in collaboration with Golder Associates Ltd. completed the tailings storage facility (TSF). Section 18.4 shows the detailed information. A thickened, non-segregating tailings slurry discharge into a TSF was recommend by Golder for Ajax. Golder performed laboratory test work to provide information on the dewatering and rheological properties of the tailings. This information, combined with processing facility throughput and TSF location and footprint, completed the design basis of the Thickened Tailings Plant (TTP). The total required TSF volume is 389.6 Mm^3.

Based on the overall site geotechnical investigation report provided by BGC, Wardrop developed the site infrastructure, including access roads, site layout and structure, communication, auxiliary facilities and presented in Section 18.1, 18.5 and 18.7. In addition, Wardrop developed the water supply and power supply for the Ajax Project.

1.7 ENVIRONMENTAL AND PERMITTING

Knight Piésold initiated the environmental studies for the Ajax project in 2006, including ground and surface water quality and quantity, climatology, fish and fish habitat, wildlife, and vegetation studies. Discussions have been initiated with government regulatory agencies in order to develop appropriate avoidance and mitigation techniques. None of the environmental parameters identified to date are considered to have a material impact on the ability to extract the mineral resources or reserves. Section 20 outlines the environmental studies, and permitting application status.

Abacus submitted a Project Description to the BC Environmental Assessment Office (EAO) and the federal Canadian Environmental Assessment Agency (CEAA) in early 2011. The project description was accepted by EAO on February 25, 2011 and on March 16, 2011 by CEAA.

The Ajax Project received a Section 10 order from the EAO on February 25, 2011, stipulating that the Project must undergo an Environmental Assessment (EA). The scope and requirements for public consultation were outlined in a preliminary Section 11 in June 2011. An additional Section 11 order is expected in 2012, providing further direction regarding the scope, procedures, and methods for conducting the EA.

The CEAA commenced a comprehensive study on May 25, 2011 and posted a Notice of Commencement on the CEAA Registry on May 31, 2011. A project agreement was signed on August 17, 2011.

The Project Application Information Requirements (AIR) was provided to the EAO and CEAA on August 12, 2011 for distribution to the Technical Working Group. The Proponent Application/Environmental Impact Statement (EIS) is expected to be submitted in 2012.

1.8 Capital Cost and Operating Cost

The capital cost and operation cost estimates were provided by AMEC (mining section), Wardrop (processing and overall site infrastructure section) and Golder (tailings management facility section) in Section 21.0. The overall estimated capital cost is approximately US$795 million. The operating cost estimate will be US$1.32/t mined for mining and in-pit crushing and conveying, US$3.46/t milled for processing and tailings management facility and US$0.53/t G&A cost.

1.9 Economic Analysis

A pre-tax economic model has been developed by Wardrop from the estimated costs and the open pit production schedule. The base case has an internal rate of return (IRR) of 14.5% and a net present value (NPV) of US$416 million at an 8% discount rate for the 23-year LOM. The payback of the initial capital is within 7.8 years.

Wardrop conducted a logistics study to determine the options available and associated costs for transporting copper concentrates from the project site to a port facility for export and the results were used to the financial model preparation.

1.10 Interpretation and Conclusions

Abacus understands the requirements for infrastructure, workforce, and power, water and communication facilities to support future mining operations and is sufficient to support Mineral Resource and Mineral Reserve declaration.

Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation. The exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project. Additional exploration potential remains in the Project area.

Estimations of Mineral Resources and Mineral Reserves for the Project conform to industry best practices, and meet the requirements of CIM Definition Standards (2010).

Factors which may affect the Mineral Reserve estimate include gold and copper price assumptions, currency exchange rate, the assumption that granting of appropriate environmental and construction permits would be forthcoming from the relevant authorities, unrecognized geological structures in areas with relatively low drillhole density that could impact pit slope assumptions, changes to the assumptions regarding environmental characteristics of waste rock, and changes to the pitshell shape if the material currently under the oil pipeline can be incorporated in the pit design.

The Ajax concentrator has been designed with the objective to produce a saleable copper concentrate. The complete processing circuit design is based on various metallurgical test work programs conducted over the years and utilizing proven process equipment.

The feasibility level tailings management was designed from the pre-feasibility study incorporating new test work results on the dewatering and rheological properties of the tailings, and extensive experience on paste and thickened tailings plants and tailings storage facilities.

The initial capital cost for the Ajax Project is estimated to be approximately US$795 million. The base case of the pre-tax economic model has an internal rate of return (IRR) of 14.5% and a net present value (NPV) of US$416 million at an 8% discount rate for the 23-year LOM. The payback of the initial capital is within 7.8 years.

1.11 RECOMMENDATIONS

Based on the work carried out in this Feasibility study and the resulting economic evaluation, it is recommended that the Ajax Project proceed to the Detailed Engineering Design stage. Please refer to Section 26.0 Recommendations for more information. The proposed project execution plan is located in Section 24.0.

2.0 INTRODUCTION

In May 2010, Abacus commissioned a team of engineering consultants to complete a feasibility study in accordance with National Instrument 43-101 (NI 43-101). All mines acts regulations with respect to health, safety and environmental considerations have been taken into account and incorporated into the feasibility designs and relevant cost estimates. In addition, the designs take into account the geological location of the Project. The following consultants were commissioned to complete the component reports for the purposes of the feasibility study:

- Wardrop, a Tetra Tech Company (Wardrop) – overall management, mineral processing, infrastructure, and financial analysis
- AMEC Americas Ltd. (AMEC) – geology, mineral resource estimate and mine design
- Golder Associates Ltd. (Golder) – tailings handling, thickening and tailings area water management
- Knight Piésold Ltd. (Knight Piésold) – environmental studies, permitting, and social or community impact
- BGC Engineering Inc. (BGC) – pit slope designs, pit dewatering evaluations and site geotechnical evaluations excluding the tailings storage facility (TSF).

In addition, G&T Metallurgical Services Ltd. (G&T) conducted the metallurgical test work, and Krupp Polysius performed High Pressure Grinding Rolls (HPGR) pilot test work for the process design.

Engineering teams attended a kick-off meeting and site visit on June 23, 2010 for two days.

A summary of the Qualified Persons (QPs) responsible for each section of this report is detailed in Table 2.1.

Table 2.1 Summary of Qualified Persons

Report Section			Company	QP
1.0	Summary		All	Sign off by Section
2.0	Introduction		Wardrop	Hassan Ghaffari, P.Eng.
3.0	Reliance on Other Experts		Wardrop	Hassan Ghaffari, P.Eng.
4.0	Property Description and Location		Abacus	Hassan Ghaffari, P.Eng.
5.0	Accessibility, Climate, Local Resources, Infrastructure, and Physiography		Abacus	Hassan Ghaffari, P.Eng.
6.0	History		Abacus	Hassan Ghaffari, P.Eng.
7.0	Geological Setting and Mineralization		Abacus	Timothy O. Kuhl, R.M. SME
8.0	Deposit Types		Abacus	Timothy O. Kuhl, R.M. SME
9.0	Exploration		Abacus	Timothy O. Kuhl, R.M. SME
10.0	Drilling		Abacus	Timothy O. Kuhl, R.M. SME
11.0	Sample Preparation, Analyses, and Security		Abacus	Timothy O. Kuhl, R.M. SME
12.0	Data Verification		AMEC	Timothy O. Kuhl, R.M. SME
13.0	Mineral Processing and Metallurgical Testing		Wardrop	Andre De Ruijter, P.Eng.
14.0	Mineral Resource Estimates		AMEC	Timothy O. Kuhl, R.M. SME
15.0	Mineral Reserve Estimates		AMEC	Ramon Mendoza Reyes, P.Eng.
16.0	Mining Methods		AMEC	Ramon Mendoza Reyes, P.Eng.
17.0	Recovery Methods		Wardrop	Andre De Ruijter, P.Eng.
18.0	Project Infrastructure			
	18.1	Access and Site Roads	Wardrop	Hassan Ghaffari, P.Eng.
	18.2	Fresh Water Supply	Wardrop	Ali Farah, P.Eng.
	18.3	Geotechnical Conditions	BGC	Warren Newcomen, P.Eng.
	18.4	Tailings Storage Facility	Golder	Chris Lee, P.Eng./ Irwin Wislesky, P.Eng.
	18.5	Water Management	Wardrop	Ting Lu, P.Eng.
	18.6	Power Supply and Distribution	Wardrop	Hassan Ghaffari, P.Eng.
	18.7	Structural	Wardrop	Kenneth Lee, P.Eng.
	18.8	Communications	Wardrop	Hassan Ghaffari, P.Eng.
	18.9	Auxiliary Infrastructures	Wardrop	Hassan Ghaffari, P.Eng.
19.0	Market Studies and Contracts		Neil S Seldon & Associates	Hassan Ghaffari, P.Eng.
20.0	Environmental		Knight Piésold	Ken Brouwer, P.Eng.

table continues…

Report Section	Company	QP
21.0 Capital and Operating Costs		
21.1 Capital Cost Estimates		
21.1.1 Mining Capital Cost Estimate	AMEC	Ramon Mendoza Reyes, P.Eng.
21.1.2 Processing and Overall Site Infrastructure	Wardrop	Hassan Ghaffari, P.Eng./ Ting Lu, P.Eng.
21.1.3 Tailings Storage Facility Capital Cost Estimate	Golder	Chris Lee, P.Eng./ Irwin Wislesky, P.Eng.
21.2 Operating Cost Estimates		
21.2.1 Mining Operating Cost Estimate	AMEC	Ramon Mendoza Reyes, P.Eng.
21.2.2 Process Operating Cost Estimate and General & Administrative Costs	Wardrop	Andre De Ruijter, P.Eng.
21.2.3 TTP and TSF Operating Cost Estimate	Golder	Chris Lee, P.Eng./ Irwin Wislesky, P.Eng.
22.0 Economic Analysis	Wardrop	Amir Karami, P.Eng.
23.0 Adjacent Properties	Abacus	Hassan Ghaffari, P.Eng.
24.0 Other Relevant Data and Information	Wardrop	Hassan Ghaffari, P.Eng.
25.0 Interpretation and Conclusions	ALL	Sign off by Section
26.0 Recommendations	ALL	Sign off by Section
27.0 References	ALL	Sign off by Section

3.0 RELIANCE ON OTHER EXPERTS

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

Technical data provided by Abacus for use by Wardrop in this feasibility study is the result of work conducted, supervised, and/or verified by Abacus professional staff or their consultants.

Wardrop has not independently verified the legal status or title of the claims or exploration permits, and has not investigated the legality of any of the underlying agreement(s) that may exist concerning the property.

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

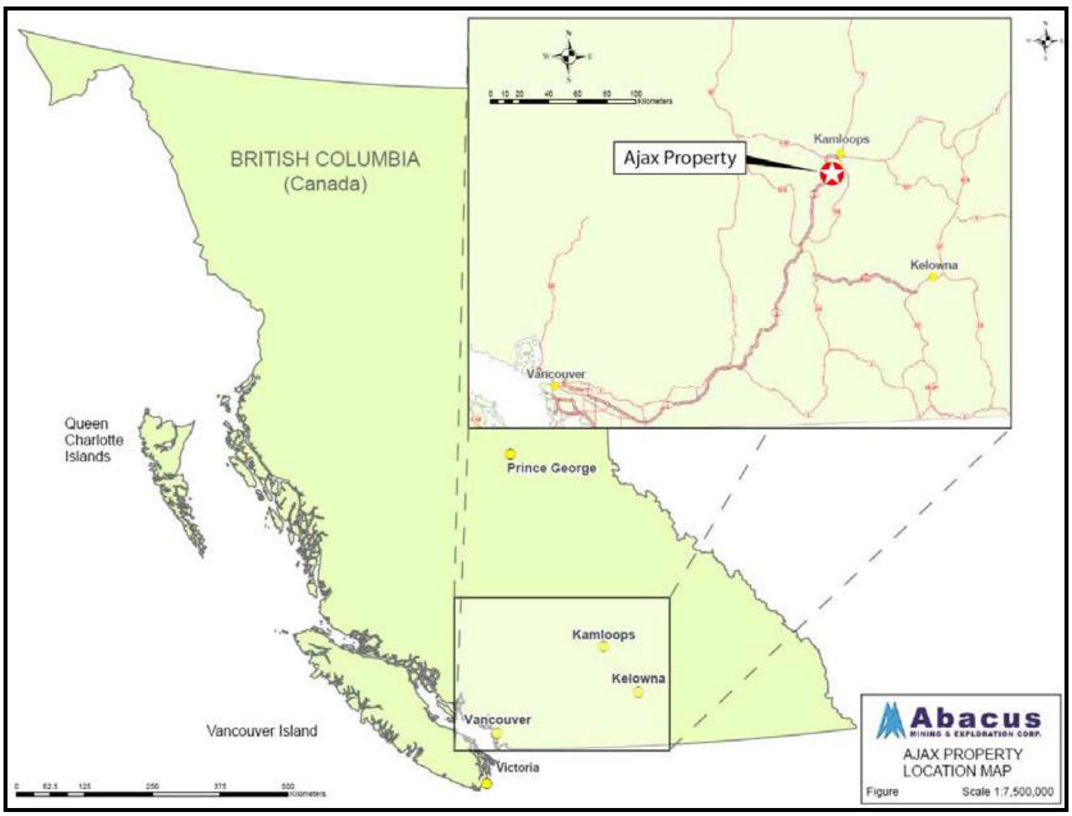
- Mr. D. Kinakin, P.Geo. of BGC, for matters relating to open pit designs in Section 15.2.
- Mr. T. Crozier, P.Eng. of BGC, for matters relating to pit hydrogeology and dewatering in Section 16.9.
- Mr. P. Quinn, P.Eng. of BGC, for matters relating to geotechnical investigations for foundation design in Section 18.3.
- Mr. R.C. Brodie, R.P.Bio., of Knight Piésold for matters relating to Environmental Studies, Permitting, and Social or Community Impact in Section 20.0.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Ajax property is located in the south-central interior of BC, south of the junction of the Trans-Canada Highway (Highway 1) and the Coquihalla Highway (Highway 5). The coordinates for the centre of the Ajax Project area are approximately 50°37' north latitude and 120°24' west longitude. The Ajax property is situated south of downtown Kamloops, BC (Figure 4.1) and is located on mineral titles reference map M09I068 (National Topography System (NTS) 92I/9) in the Kamloops mining division. Some components of the mine, including the north waste rock management facility, processing facility, truck shop, thickened tailings plant with emergency run-off pond and the tailings storage facility will be within Kamloops city limits. The north waste rock management facility, which is the closest proposed mine infrastructure to developed areas, is approximately 1.5 km away from the nearest housing developments.

Figure 4.1 Ajax Property Location Map



4.2 MINERAL TENURE

4.2.1 MINERAL RIGHTS

KGHM Ajax Mining Inc. (KAM) controls approximately 9,750.1374 ha (Figure 4.1 and Table 4.1) in the Ajax area. KAM has ownership of 58 mineral claims and 31 Crown Granted mineral claims. Of these claims, 48 of the mineral claims and 28 of the Crown Grants are contiguous with the Ajax area. The remaining claims are near Ajax but are not contiguous.

Figure 4.2 Ajax Property Claim Map

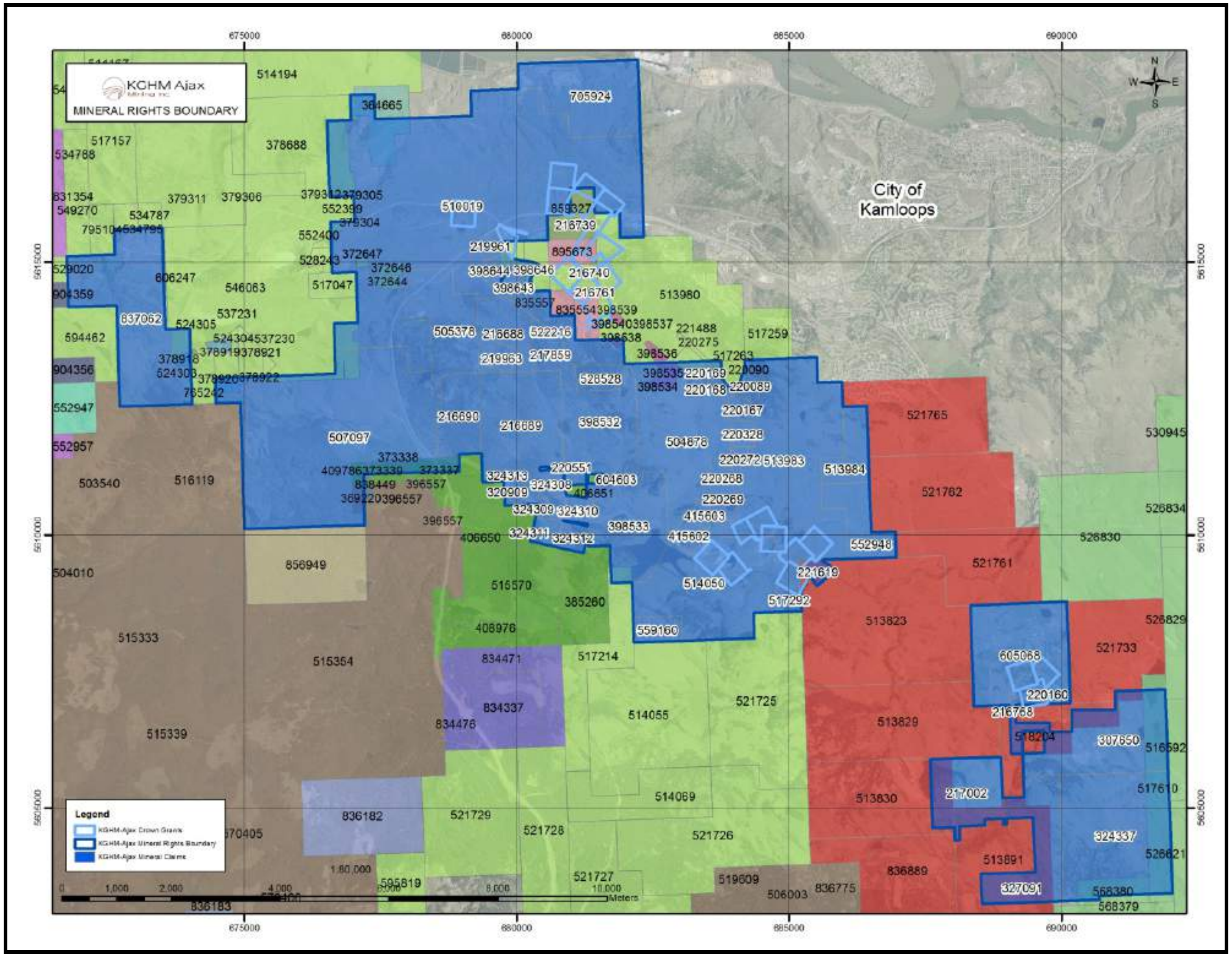


Table 4.1 Ajax Area Claims

Tenure Number	Tenure Type	Claim Name	Owner	Good to Date	Area (ha)
Contiguous KGHM Ajax Claims					
216688	Mineral	RAINBOW NE	KAM	2018/Oct/31	150
216689	Mineral	RAINBOW SE	KAM	2018/Oct/31	300
216690	Mineral	RAINBOW SW	KAM	2018/Oct/31	150
216740	Mineral	OR #14	KAM	2018/Oct/31	25
216745	Mineral	-	KAM	2018/Oct/31	25
216761	Mineral	DELTA 1061	KAM	2018/Oct/31	25
217859	Mineral	BILL	KAM	2018/Oct/31	225
219961	Mineral	-	KAM	2013/Mar/02	20.05
219963	Mineral	-	KAM	2013/Aug/10	20.89
220089	Mineral	PYTHON NO.15	KAM	2012/Sep/26	25
220167	Mineral	DOT NO.2	KAM	2012/Sep/26	25
220168	Mineral	DOT NO.3	KAM	2012/Sep/26	25
220169	Mineral	DOT NO.5	KAM	2012/Sep/26	25
220268	Mineral	JET NO.1	KAM	2012/Sep/26	25
220269	Mineral	JET NO.2	KAM	2018/Jun/01	25
220272	Mineral	JET NO.5	KAM	2012/Sep/26	25
220328	Mineral	JET NO. 11	KAM	2012/Sep/26	25
220333	Mineral	JET NO. 16 FR.	KAM	2012/Sep/26	25
220551	Mineral	X #16	KAM	2018/Oct/31	25
221619	Mineral	PLANE 19 FR.	KAM	2018/Jun/01	25
320909	Mineral	JAXD 8	KAM	2018/Oct/31	25
324308	Mineral	INK 1	KAM	2018/Oct/31	25
324309	Mineral	INK 2	KAM	2018/Oct/31	25
324310	Mineral	INK 3	KAM	2018/Oct/31	25
324311	Mineral	INK 4	KAM	2018/Oct/31	25
324312	Mineral	INK 5	KAM	2018/Oct/31	25
324313	Mineral	INK 6	KAM	2018/Oct/31	25
398532	Mineral	DCE 1	KAM	2018/Oct/31	300
398533	Mineral	DCE 2	KAM	2018/Oct/31	300
398643	Mineral	WIRE 1	KAM	2018/Oct/31	25
398644	Mineral	WIRE 2	KAM	2018/Oct/31	25
398645	Mineral	WIRE 3	KAM	2018/Oct/31	25
398646	Mineral	WIRE 4	KAM	2018/Oct/31	25
415602	Mineral	AJ 7	KAM	2018/Jun/01	25
415603	Mineral	AJ 8	KAM	2018/Jun/01	25
504878	Mineral	-	KAM	2018/Jun/01	573.974
505378	Mineral	-	KAM	2018/Oct/31	225.395
507097	Mineral	-	KAM	2018/Oct/31	1,004.384
510019	Mineral	-	KAM	2018/Oct/31	1,659.012

table continues…

Tenure Number	Tenure Type	Claim Name	Owner	Good to Date	Area (ha)
513983	Mineral	-	KAM	2018/Jun/01	635.528
513984	Mineral	AJ 9	KAM	2018/Jun/01	82.001
514050	Mineral	-	KAM	2018/Jun/01	451.2
517292	Mineral	AJAX	KAM	2018/Jun/01	20.511
522216	Mineral	DAVES DREAM	KAM	2018/Oct/31	122.947
528528	Mineral	522216 EXTRA	KAM	2018/Oct/31	40.988
552948	Mineral	AJ	KAM	2018/Jun/01	102.5259
559160	Mineral	NEW GOLD OPTION	KAM	2018/Jun/01	41.0241
604603	Mineral	AJ P EAST	KAM	2012/Sep/26	20.5013
Total					**7,120.931**
Contiguous KGHM Crown Grants					
4710	Crown Grant	AJAX	KAM	-	20.9
4712	Crown Grant	NEPTUNE	KAM	-	18
1496	Crown Grant	GRASS ROOTS	KAM	-	20.9
4716	Crown Grant	MONTE CARLO	KAM	-	18.3
4717	Crown Grant	SULTAN	KAM	-	18.9
2126	Crown Grant	WHEAL TAMAR	KAM	-	20.9
3015	Crown Grant	COPPER STAR FRACT.	KAM	-	10.6
3016	Crown Grant	FORLORN	KAM	-	16.7
878	Crown Grant	IRON MASK	KAM	-	15.5
879	Crown Grant	SUNRISE	KAM	-	19.8
880	Crown Grant	COPPER QUEEN	KAM	-	20.6
1036	Crown Grant	LUCKY STRIKE	KAM	-	5.8
1050	Crown Grant	EMEROY	KAM	-	17.5
1066	Crown Grant	ERIN	KAM	-	14
1067	Crown Grant	JUMBO	KAM	-	1.7
1068	Crown Grant	CIVIL EARNSCLIFFE	KAM	-	0.7
1301	Crown Grant	FRACTION	KAM	-	0.7
1311	Crown Grant	MAY FRAC.	KAM	-	10.5
4666	Crown Grant	SODIUM FRACTION	KAM	-	2.3
4667	Crown Grant	WINTY	KAM	-	20
5622	Crown Grant	CHAMPION NO.1	KAM	-	9.6
5623	Crown Grant	CHAMPION NO.2	KAM	-	19.1
5624	Crown Grant	L.S. NO.6	KAM	-	15.6
5625	Crown Grant	L.S. NO.7	KAM	-	17.4
5626	Crown Grant	L.S. NO.11	KAM	-	16.7
5627	Crown Grant	L.S. NO.10	KAM	-	20.9
5628	Crown Grant	L.S. NO.8	KAM	-	12.2
5629	Crown Grant	L.S. NO.9	KAM	-	14.9
Total					**400.7**
Total of all Contiguous Ajax Claims and Crown Grants					**7,521.631**

table continues…

Tenure Number	Tenure Type	Claim Name	Owner	Good to Date	Area (ha)
KGHM Claims Non-contiguous with Ajax					
216739	Mineral	OR #13	KAM	2012/Sep/26	25
220160	Mineral	ACE NO. 1	KAM	2012/Sep/26	25
837062	Mineral	AFTON WEST	KAM	2012/Nov/01	430.2645
216768	Mineral	WILDROSE 2	KAM	2012/Oct/31	25
217002	Mineral	SUNNY	KAM	2012/Oct/31	225
307650	Mineral	JOKER	KAM	2012/Oct/31	450
324337	Mineral	ACE	KAM	2012/Oct/31	500
327091	Mineral	ACE 2	KAM	2012/Oct/31	375
605068	Mineral	GRADEN	KAM	2012/May/28	328.2491
705924	Mineral	OR11	KAM	2018/Oct/31	245.6925
Total					**2,629.206**
KGHM Crown Grants Non-contiguous with Ajax					
1560	Crown Grant	BLACK BEAUTY	KAM	-	17.8
1561	Crown Grant	ADMIRAL DEWDNEY	KAM	-	7.9
1662	Crown Grant	CYCLONE	KAM	-	14.3
Total					**40.0**
Total of all Non-Contiguous Claims and Crown Grants with Ajax					**2,669.206**

4.2.2 PROPERTY AGREEMENTS AND ROYALTIES

FORMATION OF A JOINT VENTURE COMPANY

On June 28, 2010, Abacus incorporated a wholly-owned subsidiary KAM, the joint venture (JV) company. Abacus subsequently reduced its interest in KAM to 49% in conjunction with a cash contribution by KGHM for a 51% interest in KAM. The KAM JV agreement sets out the parameters for the development of the Ajax Project and the surrounding area, from feasibility study to production. KGHM is one of the world's largest copper and silver producers with annual production of over 500,000 t of copper and 1,100 t of silver.

The KGHM Investment Agreement included the following investment highlights:

- KGHM and Abacus completed a Cdn$4.5 million private placement involving the purchase of 15 million common shares (approximately 8.75% of the shares issued and outstanding following the private placement) of Abacus at a price of $0.30 per share.
- Abacus incorporated a wholly-owned subsidiary, KAM, and transferred its entire mineral property interests fair valued at US$35,549,020 to KAM in exchange for 4,900 common shares of KAM.

- KGHM acquired a 51% interest in KAM by investing US$37 million in cash in exchange for 5,100 common shares of KAM at the closing of the transaction on October 12, 2010. These funds are to be allocated to: (a) completion of the feasibility study and certain other obligations; and (b) acquisition of additional land areas and exploration of other targets of the Ajax property, in accordance with a jointly approved budget.
- Currently, Abacus is the operator of the Ajax Project. KAM will have the option of being the operator upon KGHM increasing its interest in KAM to 80%, pursuant to the terms of the Investment Agreement as described below.

Development option:

- Within 90 days of receipt of the feasibility study, KGHM will have the option to acquire an additional 29% interest, for a total 80% direct interest in KAM, for cash consideration of US$0.025/lb for the corresponding 29% of proven and probable copper equivalent reserves (as defined in the feasibility study) up to a maximum of US$35 million. This payment will be applied directly toward Abacus' proportionate share of the project's capital costs.
- KGHM will arrange the financing for its proportionate share of 80% of the project's capital costs and will offer to arrange the financing for the balance of Abacus's proportionate share of 20% of the project's capital costs on commercially reasonable terms, if Abacus wishes.

AFTON MINE ASSET ACQUISITION FROM TECK RESOURCES LTD.

On April 8, 2011 Abacus announced a transaction with Teck Resources Limited (Teck) whereby KAM acquired back-in rights which Teck held to certain Afton-Ajax area mineral claims as well as acquiring the tailings pond, mill, workshop, and office buildings of the former Afton Mine and the land upon which these assets are situated. A remaining final payment to Teck of $5 million is due on or before October 8, 2012 to complete this transaction.

As a result of this transaction Teck now owns 39,251,176 shares of Abacus, representing 19.9% of the issued and outstanding shares of Abacus.

Pursuant to the Asset Purchase Agreement, Teck has retained a 1.5% NSR royalty with respect to each of the Rainbow and Iron Mask properties that were purchased by Abacus from Teck and which comprise part of the Afton Ajax property. Each of the royalties can be purchased by Abacus for $3 million per royalty within two years from the date of commencement of commercial production on the respective properties.

The Asset Purchase Agreement also provides Teck with a right (but not an obligation) to participate in any future financings of Abacus to maintain Teck's proportionate shareholdings in Abacus.

In connection with the Asset Purchase Agreement and the acquisition by New Gold Inc. (New Gold) of certain other assets from Teck in the area of the former Afton Mine, Abacus, Teck, Afton, Sugarloaf Ranches Limited (Sugarloaf), a wholly-owned subsidiary of Teck, and New Gold entered into an agreement dated March 19, 2008, which sets forth certain agreements among the parties relating to the shared use of certain assets of the former Afton Mine site, including the use of and access to roads, rights of way, and use of the former Afton Mine water system.

Asset Swap Transaction with New Gold

On September 22, 2011 Abacus announced the completion of a transaction with New Gold whereby New Gold acquired from KAM the former Afton Mine buildings and the lot on which they are located. The buildings were not being used by KAM and are not required for future operations. In exchange, KAM has acquired from New Gold a net present interest (NPI) royalty which New Gold held on certain claims within the area of the Ajax pit, and KAM also obtained certain mining claims located north of the Ajax pit on which the Ajax north waste rock dump is proposed to be located.

Option to Purchase Agreement with Teck Regarding Sugarloaf Ranch

On July 13, 2009 Abacus announced the signing of Option to Purchase Agreements (Option Agreements) with subsidiaries of Teck, Afton OC and Surgarloaf (collectively "Teck") to acquire approximately 6,000 ac of land around the Ajax deposit, the exercise price of the options is $2,500/ac. An aggregate of $200,000, to be applied toward the purchase price, has been paid to secure this option agreement and to extend the closing date of the agreement to July 31, 2012. On June 29, 2010, Abacus assigned its rights to the Option Agreements to KAM.

It is a condition to the exercise of each option under the respective Option Agreements that both options be exercised contemporaneously, and that Abacus be in possession of a mine permit under the Mines Act (of BC) for the operation of a mine with respect to the Ajax property and associated mineral properties at the time of such exercises. If the options are exercised and Abacus decides to sell any part of the lands before a mine is brought into operation on the lands or the area that abuts the lands, then Teck has the option of buying back all of the lands for the purchase price.

Other Land and Tenure Related Negotiations

Abacus is negotiating Option Agreements and Right-of-Way agreements with a number of local landowners in order to acquire surface rights to certain lands on which mine infrastructure such as waste rock dumps and power lines might be located. In addition negotiations are ongoing with two companies holding mineral rights under the footprint of proposed Ajax Mine waste rock dumps and tailing dumps in order to reach agreement on appropriate condemnation drilling programs for these areas.

4.3 TAXES AND ASSESSMENT WORK REQUIREMENTS

There are no assessment requirements on Crown Granted mineral claims. Assessment requirements for mineral claims are Cdn$8.00/ha/a.

The yearly assessment for the KAM mineral claims is Cdn$78,001.

The yearly taxes on the Crown Grants controlled by KAM are Cdn$848.59.

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

5.1 ACCESSIBILITY

The Ajax property is accessed by the Afton Mine haul road, which crosses the Lac Le Jeune Road approximately 9 km southwards from the intersection of Lac Le Jeune Road and Copperhead Drive off of Highway 1, west of Kamloops.

5.2 CLIMATE

The Ajax area has a semi-arid climate. Average winter temperatures are commonly below 0°C, with lows falling below -20°C with light snowfall. Dry summers have moderate to hot temperatures ranging upwards of 30°C.

Exploration activities can operate year-round with appropriate equipment.

5.3 LOCAL RESOURCES

Local resources necessary for the exploration, development, and operation of the Ajax property are located in Kamloops. Kamloops has a resource-based economy and is a transportation hub for the Canadian National Railway (CNR) and Canadian Pacific Railway (CPR). Highway 1 services Kamloops and Highway 5 is situated west of the Ajax property. There is also an airport with daily scheduled flights to Vancouver, Calgary, Kelowna and Prince George.

Numerous service and supply companies which service resource industries are established in Kamloops, including several diamond drilling companies, light-to-heavy equipment contractors and a metallurgical testing laboratory.

Water used for exploration activities is commonly hauled by truck from the city of Kamloops.

5.4 INFRASTRUCTURE

Haul roads and road overpasses exist from previous mining operations on the property and portions of these could be upgraded and re-used if required.

5.5 PHYSIOGRAPHY

The Ajax area consists of rolling grasslands with timber at the higher elevations. Elevations range from 800 to 1,100 masl. Sugarloaf Hill is the prominent landform in the area and has an elevation of 1,130 m. The area has been glaciated and numerous drumlins are present.

At lower elevations, the vegetation is typically bunchgrass, sagebrush, and prickly pear cacti. Higher elevations commonly sustain growths of Lodgepole Pine, Douglas Fir, and Ponderosa Pine.

6.0 HISTORY

The exploration history in the Ajax area is summarized from information available on MINFILE (minfile.gov.bc.ca).

Exploration began in the area in the 1880s. Copper, gold, and iron mineralization was discovered at the Iron Mask Mine near Kamloops in 1896. Nearby properties were explored by underground methods in the following years, including the Wheal Tamar, Ajax, and Monte Carlo claims in the Ajax area (Figure 4.2, Table 4.1).

In the Ajax area, underground exploration began on the Wheal Tamar claim in 1898. Development work was completed on the Monte Carlo claim as early as 1905 and on the Ajax claim in 1906. The Monte Carlo claim included an 18 m shaft. Exploration is reported to have continued over the Wheal Tamar, Ajax, and Monte Carlo areas but became sporadic after 1914.

In 1916, Granby Consolidated Mining, Smelting, and Power Company Ltd. (Granby) completed diamond drilling on the Wheal Tamar group.

In 1928, the Consolidated Mining and Smelting Company of Canada Ltd. (CM&S) obtained options on claims in the Ajax area and completed surface drilling on the Ajax claims (10 DDH) and the Monte Carlo (3 DDH) claims. Sparse mineralization was reported.

In 1952, the property was optioned to Berens River Mines Ltd. (Berens). Berens completed four DDH between the Monte Carlo and Wheal Tamar claims but no mineralization was reported.

In 1954, CM&S and its successor, Cominco, entered into option agreements and explored the area until 1980. Exploration included electromagnetic (in 1954) and magnetometer (in 1967) geophysical surveys. Cominco completed 56 DDH (more than 7,500 m) in 1967 on the Ajax, Wheal Tamar, and Monte Carlo claims.

In 1973, Afton Mines Ltd. (Afton Mines) completed a reduced polarization survey and drilled 55 percussion drillholes totalling approximately 4,700 m on the Ajax, Wheal Tamar, and Monte Carlo claims.

In 1980, Cominco completed magnetometer and induced potential geophysical surveys, and drilled 190 percussion holes (14,347 m) in the Ajax area.

In 1986, Afton Mines, controlled by Teck, obtained an option to earn 70% interest in the Ajax properties from Cominco. In 1987, Afton Mines completed 77 DDH (11,582 m). In 1988, development work began on the Ajax West and East open pits and a haul road was constructed to the Afton mill (10 km northwest of the Ajax area).

Afton Mines commenced production at Ajax East and Ajax West in 1989. Production was suspended in 1991 due to low metal prices. A second period of production began in 1994 and was again suspended in 1997. During the periods of production, it is estimated 17 Mt were mined and 13 Mt were milled.

Abacus acquired the Afton property in 2002 from Teck. Abacus completed 62 km of three-dimensional (3D) induced polarization (IP) and magnetometer survey and diamond drilling on the Rainbow and Comet-Davenport areas during 2003 and 2004. In 2005, NI 43-101 compliant resource estimates were completed for the Comet-Davenport area (Darney et al, 2005a) and for the Rainbow area (Darney et al, 2005b).

Abacus has explored the Ajax property with DDH from 2005 to 2010. Details of this drilling are provided in Section 10.0 of this report. Wardrop completed a NI 43-101 compliant resource estimate and a positive PEA for the Ajax area in 2009 (Ghaffari et al, 2009).

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The regional geology of the Ajax area is dominated by the Upper Triassic Iron Mask batholith. The batholith is approximately 5 km wide, 20 km in length and trends northwest through the region (Figure 7.1).

The Iron Mask batholith intruded a sequence of Nicola Group flows and volcaniclasitc rocks of mafic and intermediate composition. Near the contact with the Iron Mask batholith, the Nicola Group rocks are commonly basalt to andesite flows and flow breccias. Stratigraphically above the Nicola Group is a series of serpentinized picrite basalts, which are present within the batholith and are apparently localized along major structural corridors.

Multiple phases are recognized in the Iron Mask Batholith:

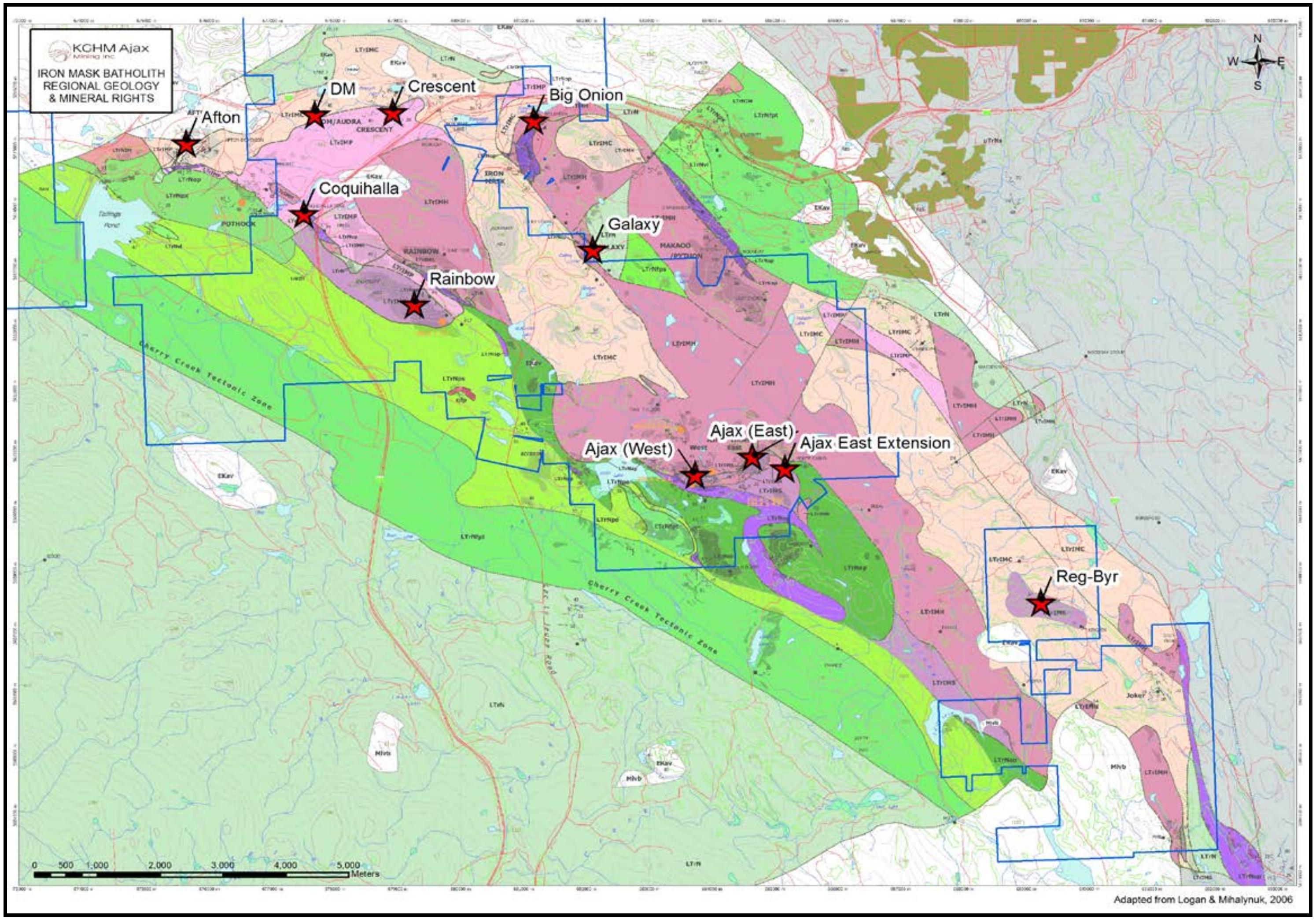
- The Pothook Diorite is the oldest phase in the batholith, these rocks consist of fine to coarse grained, generally equigranular pyroxene diorite to gabbro. Alteration is typically chlorite-epidote within variable potassium feldspar. Magnetite-apatite veins up to several metres wide occur within this unit. The Pothook diorite hosts the Pothook deposit and forms the inboard contact of KGHM's DM-Audra-Crescent deposit.
- The Iron Mask Hybrid is a very heterogeneous unit which represents the interaction between Pothook intrusives and Nicola Group volcanic rocks. Rock textures, alteration and mineralization are extremely variable within this unit. Portions of the Rainbow and the Ajax deposits are hosted within strongly albitized Hybrid diorite in contact with similarly altered and mineralized Sugarloaf diorites.
- The Cherry Creek Monzonite dominates the north and east margins of the batholith and forms a pluton northwest of the batholith. The Cherry Creek postdates the Pothook phase. Ubiquitous K-feldspar alteration generally gives the Cherry Creek a pinkish colour. KGHM's DM-Audra-Crescent deposit is hosted within Cherry Creek monzonite breccia along a fault contact with older unmineralized, Pothook diorite. Mineralization within the Cherry Creek rocks consists of blebby and disseminated chalcopyrite, bornite and pyrite.
- The Sugarloaf phase dominates the western margin of the batholith and also postdates the Pothook phase. The age relationship with Cherry Creek is

uncertain. The Sugarloaf phase is commonly a fine-grained porphyritic hornblende diorite. Albite alteration is common near zones of mineralization.

KGHM's Ajax and Rainbow deposits lie along a broad contact zone between older Iron Mask Hybrid diorite to the north and younger Sugarloaf diorite to the south. Sulphide mineralization at Ajax and Rainbow consists of disseminated, blebby and stringer pyrite-chalcopyrite associated with strongly albitized Sugarloaf diorite and Iron Mask Hybrid diorite.

The Kamloops Group, located north-east of the batholith, contains the youngest rocks in the region and consists dominantly of tuffaceous sandstone, siltstone, and shale with minor flows and agglomerates of basaltic and andesitic composition.

Figure 7.1 Regional Geologic Map



7.2 PROPERTY GEOLOGY

Numerous authors have reported on the geology of the Ajax area (Ross, 1993; Ross et al., 1995; Logan et al., 2007).

As many as 22 rock types have been recognized in the Ajax area, but these can generally be combined into three main rock units: Iron Mask Hybrid, Sugarloaf Diorite, and Nicola Volcanics (Figure 7.2).

Outcrops are generally abundant in the Ajax area. The contact between the Sugarloaf Diorite and the Iron Mask Hybrid strikes south-easterly through the West Ajax area and changes to a north-easterly strike through the East Ajax area. The Sugarloaf-Iron Mask contact is offset vertically by a south-easterly striking fault (Leemac Fault) at the north end of the East Ajax area and continues to the strike north-easterly in the in the Ajax Extension area to the east and north of the East Ajax area. The contact between the Sugarloaf Diorite and Nicola Group generally strikes south-easterly through the Ajax area.

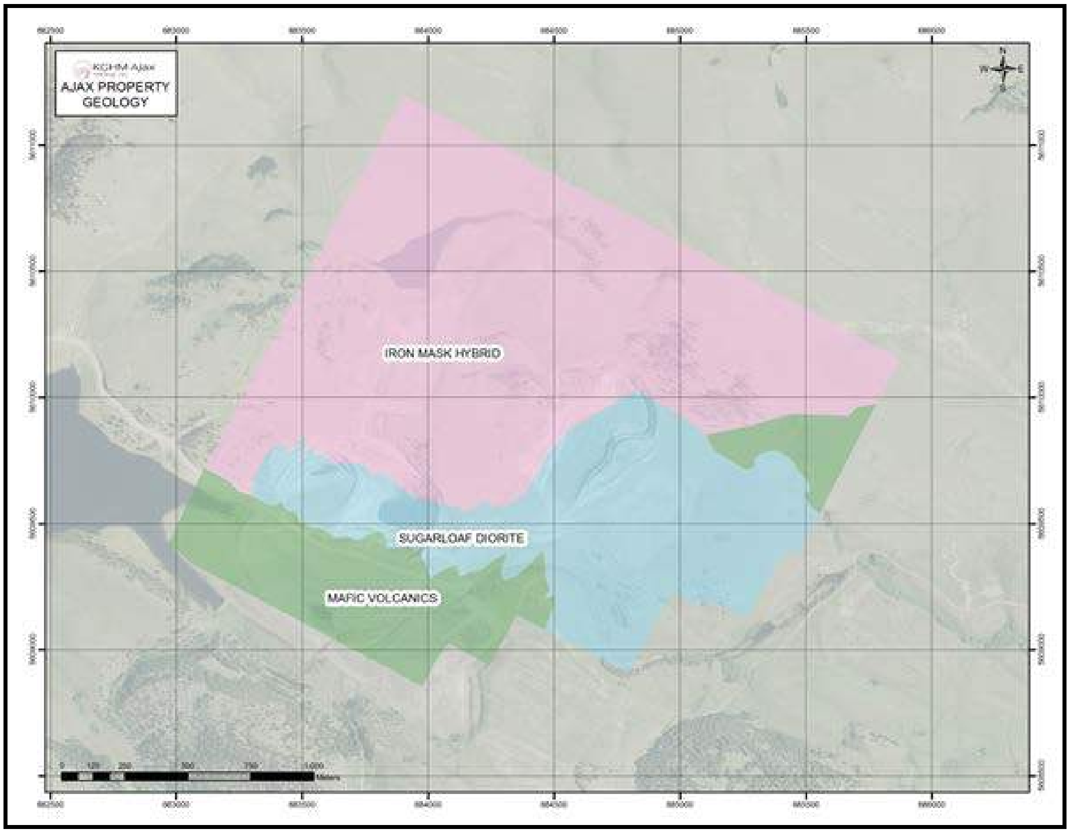
Sugarloaf Diorite is characteristically a fine to coarse-grained, light to medium gray porphyritic diorite containing euhedral hornblende phenocrysts. Unaltered Sugarloaf may contain up to 5% fine-grained magnetite. Locally, the Sugarloaf Diorite has assimilated rocks of the Nicola Group and is referred to as the Sugarloaf Hybrid. Albite and K-feldspar alteration is present in varying degrees. Strong albite alteration has commonly destroyed original textures locally. Sulphide mineralization is associated with albite alteration and consists predominantly of chalcopyrite and pyrite. Molybdenite, tetrahedrite, and bornite have been observed.

The Iron Mask Hybrid is considered to be an assimilation of the Nicola Group into the intruding Pothook Diorite. The Iron Mask is coarse-grained and dioritic to gabbroic in composition. Weak propylitic alteration is common with K-feldspar and albite alteration occurring locally. The Iron Mask Hybrid may contain up to 10% magnetite and locally chalcopyrite and pyrite are present.

The Nicola Group consists of picrite and various fine-grained and pyroxene porphyritic mafic volcanic rocks.

A variety of steeply dipping, unmineralized dykes up to 5 m wide intrude the main rock types. Dykes are composed of aplite, monzonite, latite, and fine-grained mafic rocks.

Figure 7.2 Geologic Map of the Ajax Property



7.3 MINERALIZATION

7.3.1 *IRON MASK BATHOLITH*

The Iron Mask Batholith is host to more than 20 known mineral deposits and occurrences. Table 7.1 shows a list of past production, historical and current resources for known deposits in the Iron Mask Batholith. Copper-gold mineralization associated with the Iron Mask batholith is classified as alkaline porphyry copper-gold deposits and is commonly associated with the Cherry Creek and Sugarloaf phases. Mineralization is generally localized along major fault zones and associated with albite and K-feldspar alteration.

Chalcopyrite is the dominant sulphide mineral. The presence of accessory sulphide minerals is highly variable and can include tetrahedrite and molybdenite. Secondary copper oxides (bornite and chalcocite) and native copper have been observed locally. Copper-gold mineralization in the Iron Mask Batholith shares a number of common geological characteristics:

- mineralization occurs on or near lithological contacts

- mineralization occurs in intrusive rocks or in volcanic rocks proximal to the intrusive contact
- the main hypogene copper mineralization is dominated by chalcopyrite in association with pyrite
- supergene mineralization is dominated by chalcopyrite, bornite, native copper and chalcocite/covellite
- deposits all have some fault bounded contacts due to late post mineralization faulting
- structures are dominated by several generations/orientations of steep faulting and a late generation of flat thrust faults
- all contain ultramafic volcanic (picrite) inclusions, often as fault bounded slices.

Table 7.1 Known Mineral Deposits in the Iron Mask Batholith

Deposit	Ownership	Class	Tonnes (M)	Cu (%)	Ag (g/t)	Cu Cut-off (%)	Mining Dates
Afton	New Gold	Past-production	22.1	0.91	0.67	-	1977-1987
New Afton	New Gold	2006 M&I Resource	68.7	1.02	0.77	0.7	2012-?
Pothook	New Gold	Past-production	2.4	0.35	0.77	-	1987-1988
Ajax East & West	Historic	Past-production	16.6	0.33	0.25	-	1989-1991 & 1994-1997
Ajax East & West	KGHM Ajax	2011 M&I In-pit Resource	658.7	0.26	0.17	0.10% CuEq	2016?
Crescent	KGHM Ajax	Past-production	1.4	0.44	0.18	-	1988-1989
Rainbow	KGHM Ajax	2005 Indicated/ Inferred Resource	31.6/ 1.1	0.41/ 0.29	0.09/ 0.07	0.25	-
DM-Audra-Crescent	KGHM Ajax	2005 Indicated/ Inferred Resource	16.2/ 9.4	0.35/ 0.32	0.19/ 0.15	0.25	-
Big Onion	C.N. Delorme	Historical Resource	3.3	0.71	0.44	-	-
Galaxy	Discover Corp. Ventures	Historical Resource	3.2	0.65	0.34	0.5	-

Note: M&I = Measured & Indicated

7.3.2 AJAX AREA

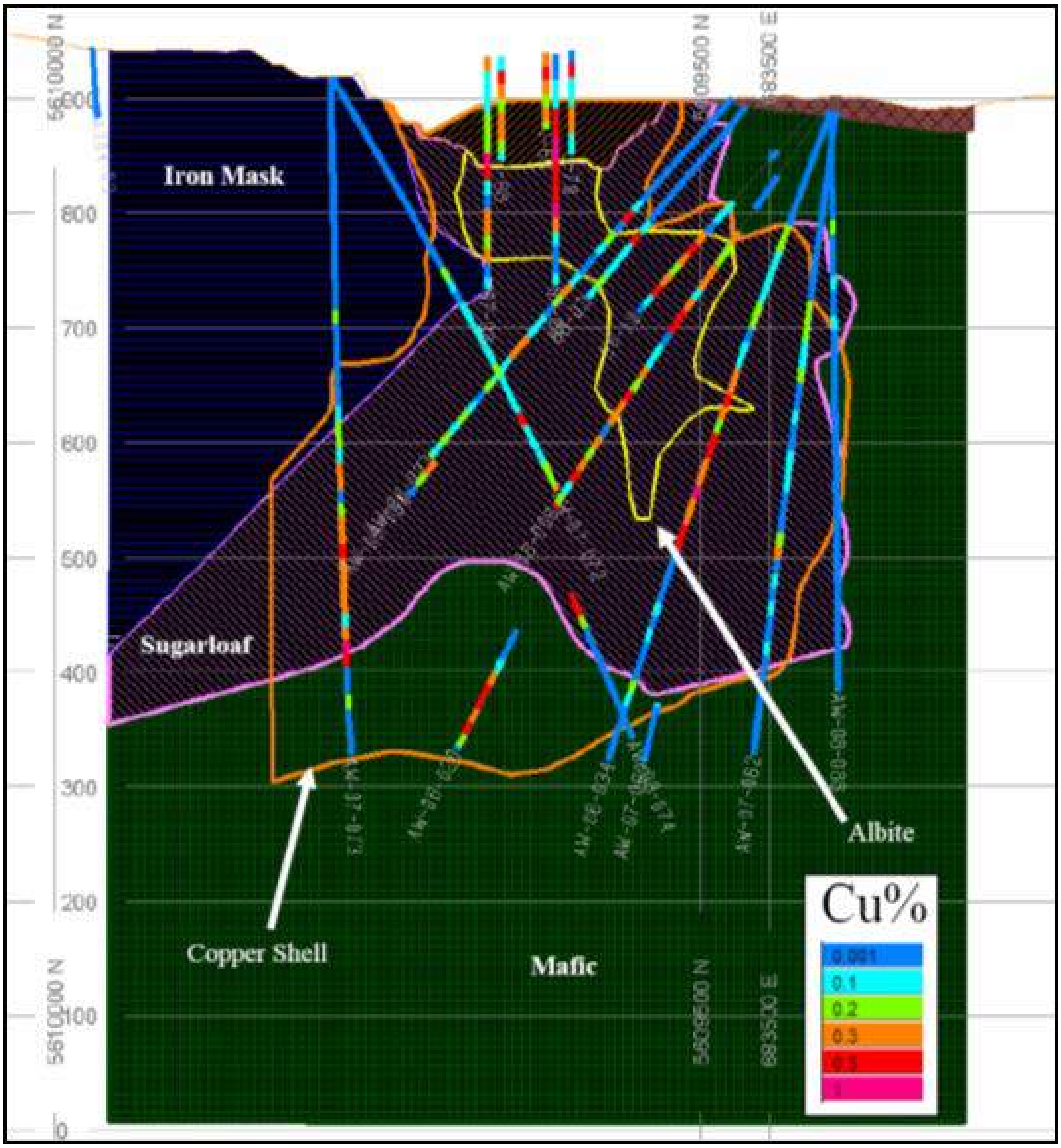
The mineralization in the Ajax area is associated with structural corridors of highly fractured sections of Sugarloaf and Sugarloaf Hybrid phases of the Iron Mask Batholith. Chalcopyrite is the dominant copper mineral and occurs as veins, veinlets, fracture fillings, disseminations, and isolated blebs in the host rock. Concentrations of chalcopyrite rarely exceed 5%. Accessory sulphide minerals include pyrite, magnetite, molybdenite, and occasionally bornite.

High-grade copper mineralization (more than 1.0% Cu) is confined to chalcopyrite vein systems. Copper grades decrease away from the chalcopyrite veins. High-grade mineralization can extend several metres from the vein structure. Low-grade copper mineralization (0.10 to 0.50% Cu) is generally associated with the Sugarloaf-Iron Mask contact. Mineralization extends to depths exceeding 400 m and has a strike length exceeding 2,000 m. Figure 7.3 shows a typical geological cross section with the geology and copper grades.

It is common for gold concentrations to be directly correlated with copper concentrations. Gold mineralization increases slightly in areas where strong albite alteration occurs. The albite alteration is in part controlled by fault and vein structures.

Minor palladium mineralization is associated with copper near the contacts of the Iron Mask Hybrid and Sugarloaf units.

Figure 7.3 Typical Geological Section



8.0 DEPOSIT TYPES

In the Afton-Ajax area, five alkalic copper-gold porphyry deposits have been mined by open pit methods by Afton OC since the 1970s. The Afton, Pothook, and Crescent deposits are approximately 9 km northwest of the Ajax area. The Ajax West and Ajax East mine areas are within the Ajax area. Copper-gold mineralization has also been identified at the Rainbow property, 6 km northwest of Ajax area.

Mineralization in the Iron Mask Batholith is typically associated with the Sugarloaf and Cherry Creek phases where they are in contact with the older Pothook and Iron Mask Hybrid.

Mineralization is commonly represented by chalcopyrite. Bornite, chalcocite, copper carbonates, and native copper are present locally in supergene zones in the Afton and Ajax areas. Gold mineralization is common and has a significant correlation with copper. Minor molybdenum and palladium mineralization has been identified in the Rainbow and Ajax West areas.

Alteration is variable and commonly consists of a broad propylitic assemblage of pyrite, chlorite, and epidote. Albite alteration is present in the Ajax area and is a strong control for mineralization.

Potassic alteration represented by plagioclase replaced by K-feldspar is present in all the deposits and is the dominant alteration in the Pothook and Cherry Creek phases.

Structural corridors, defined as zones of brittle deformation, are recognized as favourable zones for mineralization. Structural corridors have been interpreted from surface mapping and magnetic surveys and generally define the outer boundaries of the batholith.

9.0 EXPLORATION

Abacus initially focused exploration activities in the Ajax West area and subsequently expanded eastward into Ajax East. Exploration has consisted almost exclusively of diamond drilling completed during the years 2005 to 2010.

9.1 AJAX EXPLORATION WORK

9.1.1 DRILLING

Abacus exploration activity in the Ajax area has been limited to DDH drilling during the period November 2005 to December 2010. DDH are predominantly NQ (47.6 mm diameter). Some drillholes were completed with a BQ (36.4 mm diameter) tail when a reduction was required during drilling. Details of the Abacus drill programs are given in Section 10.0 of this report.

9.2 PREVIOUS OPERATORS EXPLORATION WORK

Previous operators explored the Ajax property using geophysics, geochemistry, and drilling. The exploration history of the Ajax property is included in Section 6.0. Exploration work carried out by previous operators is summarized in Table 9.1.

Table 9.1 Summary of Exploration Programs by Previous Operators

Year	Company	Activity
1916	Granby	DDH
1928	CM&S	DDH
1952	Berens	DDH
1954	CM&S	Geophysics
1955 to 1957	CM&S	DDH
1961	CM&S	DDH
1967	Cominco	Geophysics & DDH
1972/1973	Afton Mines	Geophysics & Percussion Drilling
1980	Cominco, E&B	Geophysics & Percussion Drilling
1981	Cominco, E&B	Percussion
1987	Afton OC	DDH
1988	Afton OC	DDH
1989	Afton OC	DDH
1990	Afton OC	DDH

Note: E&B = E&B Explorations Ltd.

9.3 PROSPECTS

Abacus controls other exploration prospects near the Ajax property. Abacus completed geophysical surveys and drilling on the Rainbow prospect during the 2002 to 2004 period, and on the DM-Audra-Crescent prospect in 2004, 2006 and 2007 (Figure 7.1). In 2005, resources were reported on these two areas (Darney, Friesen, and Giroux 2005a, 2005b). High level economic evaluations were completed for these areas and it was concluded that they do not currently meet reasonable prospects for economic extraction. However, further exploration or more favourable economics may make these prospects viable.

9.3.1 RAINBOW

The Rainbow area is located approximately 6 km northwest of the Ajax area. The Rainbow area is underlain by Pothook and Sugarloaf phases of the Iron Mask Batholith to the north and is separated from the Nicola Group to the south by the Leemac Fault zone. Abacus completed 53 drillholes (20,106 m) of NQ-size core during 2002, 2003, and 2004.

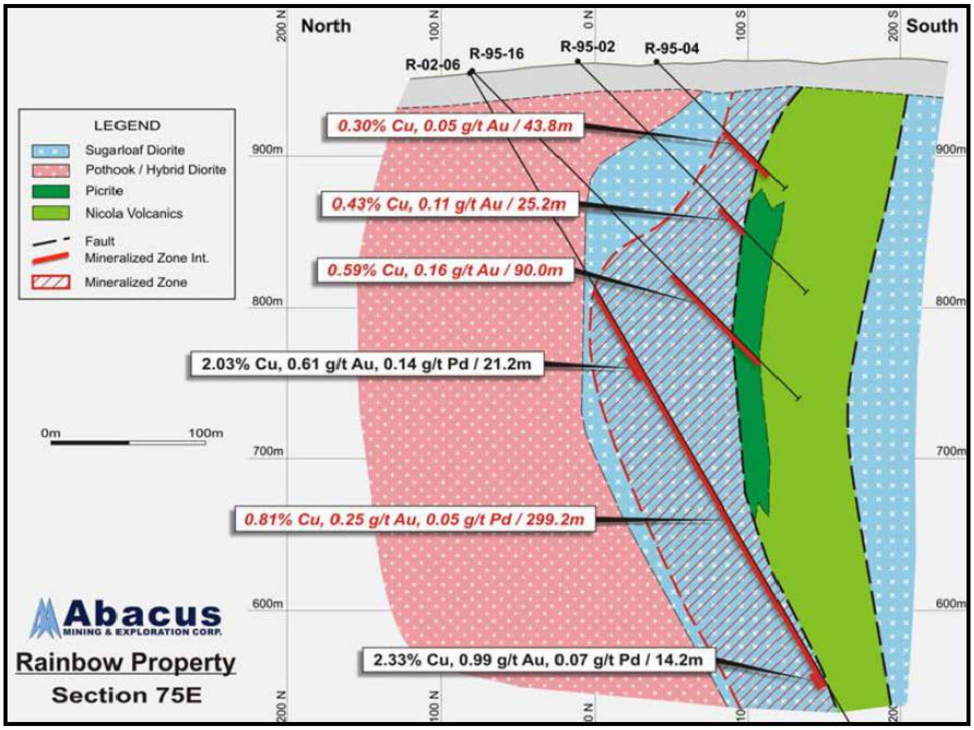
Mineralization in the Rainbow area is localized within and along the Leemac Fault zone (Figure 9.1). Three zones of mineralization have been identified: #2/22, #1 and #17 zones. Mineralization is controlled in highly fractured zones of the Sugarloaf phase. Chalcopyrite, with variable amounts of pyrite, magnetite, and molybdenite, is within veins, fracture fillings, and dissemination. Gold is also present and visible gold has been observed on instance in quartz-carbonate veinlets.

9.3.2 DM-AUDRA-CRESCENT

The DM-Audra-Crescent is located approximately 8.5 km northwest of the Ajax area. The DM-Audra-Crescent is underlain by Nicola Group Volcanics intruded with the Pothook and Cherry Creek phases of the Iron Mask Batholith. Abacus completed 87 drillholes (27,604 m) of NQ-size core during 2004, 2006, and 2007.

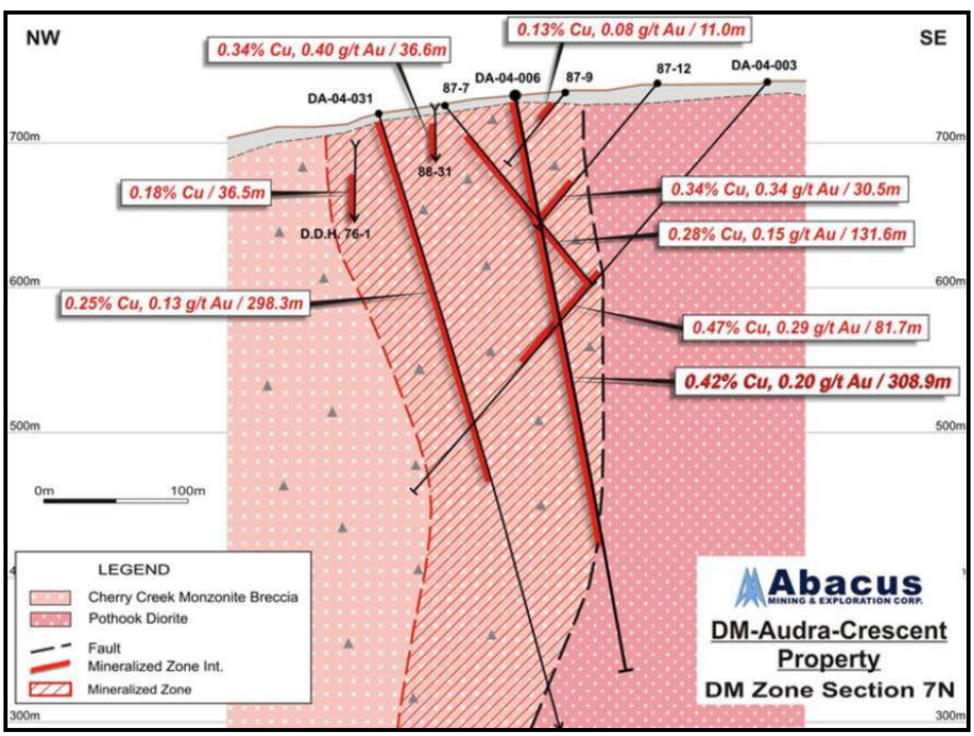
Mineralization in the DM-Audra-Crescent area is localized in potassic altered monzonite intrusive breccias that trend northeast (Figure 9.2). Mineralization consists of chalcopyrite, pyrite, and minor bornite within calcite-epidote-chlorite veins and disseminations. The DM-Audra zone has a strike length of approximately 800 m and varies in width from 20 to 200 m. Drilling has indicated the zone extends to a depth of at least 300 m.

Figure 9.1 Geological Cross Section for the Rainbow Prospect



Source: Abacus

Figure 9.2 Geological Cross Section for the DM-Audra Prospect



Source: Abacus

10.0 DRILLING

The Ajax area was initially drilled as early as 1916 and since that time, more than 750 drillholes have been completed in the area totalling 213,071 m (Table 10.1). Most early drilling, from 1916 to 1980, is poorly documented. Historic drilling on the Ajax property was concentrated in the areas of the open pit mines that were in production in the 1980s and 1990s. Recent drilling completed by Abacus has targeted extensions of mineralization along strike and to depth.

10.1 DRILL CAMPAIGNS

Documentation for the drilling campaigns completed in the Ajax area before 1980 is limited or not available. Because of the limited documentation, the pre-1980 drill data were not included in the 2011 resource model database.

Drilling campaigns completed during the period 1980 to 1990 have limited documentation generally consisting of geological logs, survey information, and assay certificates. Abacus has compiled this information into drill files located at the Ajax Project offices near Kamloops.

The resource database includes 564 drillholes totalling 184,411 m of drilling. Approximately 98% of the drill data included in the 2011 resource model database has been completed using DDH methods. The remaining data come from RC holes drilled by Cominco in 1981.

All Ajax drill campaigns are summarized in Table 10.1 and flagged to indicate whether they were used in this resource update. Figure 10.1 shows the spatial distribution of the drillholes used in the resource model database.

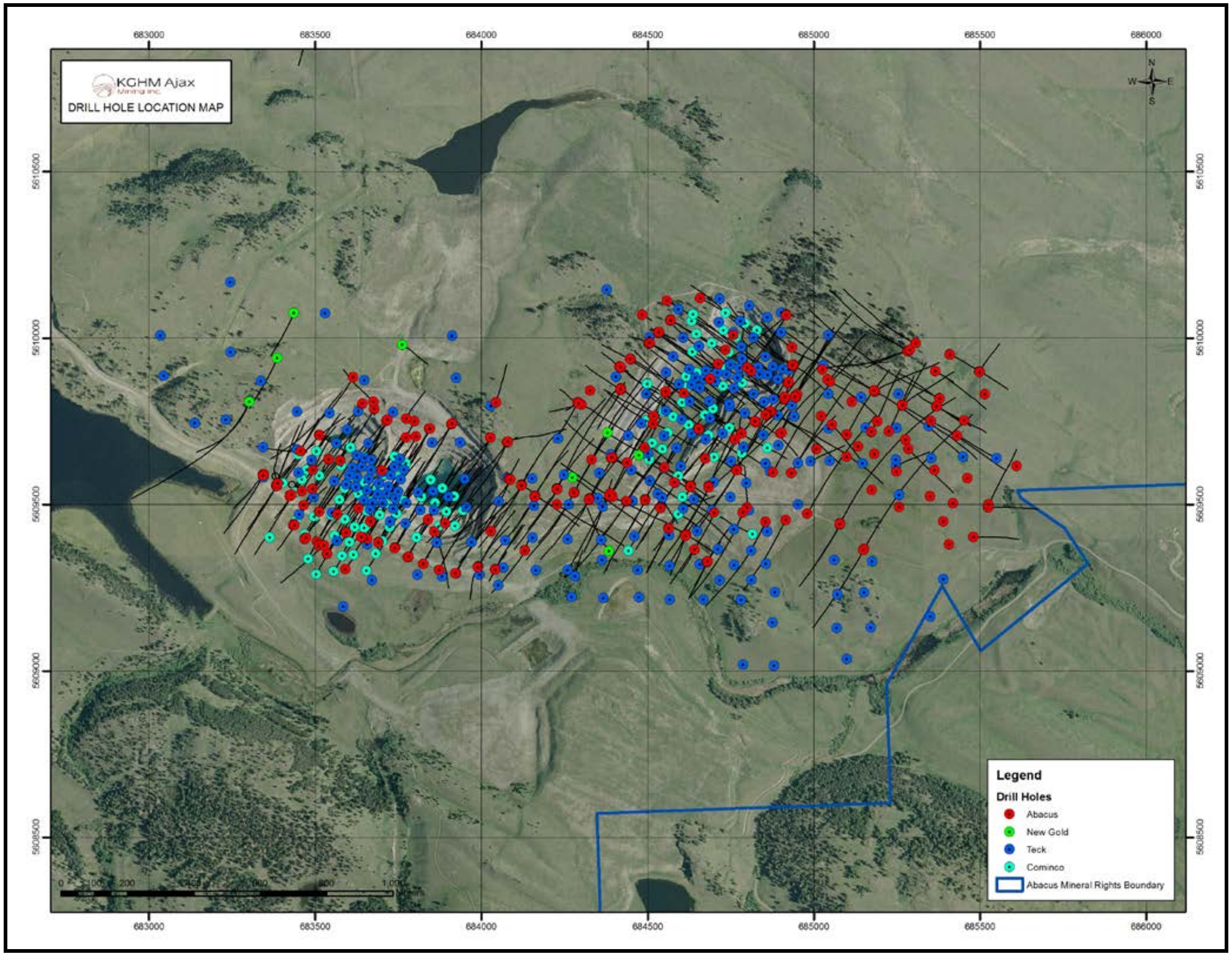
10.2 COMINCO CAMPAIGNS (1980 WT SERIES)

Documentation for Cominco WT series drillholes drilled in 1980 is limited to drill logs. These drillholes are reported to be vertical percussion drillholes that were less than 100 m in length. The drillhole collar locations were obtained from historic maps. The collar locations indicate WT drilling was completed on a grid spacing of 100 to 150 m. There is no documentation regarding drilling, sampling, or laboratory protocols. Generally, the WT drilling has been replaced with recent Abacus drilling and this campaign was not included in the 2011 resource model database.

Table 10.1 Summary of Drill Campaigns

Year	Company	Drill Type	Used for Resource Model	No. of Drillholes	Depth (m)
1928	CM&S	DDH	No	Unknown	621
1952	Berens River	DDH	No	Unknown	421
1955 to1957	CM&S	DDH	No	Unknown	4,633
1961	CM&S	DDH	No	Unknown	305
1967	Cominco	DDH	No	Unknown	1,271
1972/1973	Afton Mines	Percussion	No	Unknown	4,420
1970s	Cominco	DDH	No	7	1,045
1980	Cominco, E&B	Percussion	No	186	15,944
1981	Cominco, E&B	Percussion	Yes	53	4,387
1987	Afton	DDH	Yes	77	11,669
1988	Afton	DDH	Yes	15	2,518
1989	Afton	DDH	Yes	5	1,493
1990	Afton	DDH	Yes	13	3,507
2004	New Gold	DDH	Yes	6	2,016
2005	Abacus	DDH	Yes	5	2,714
2006	Abacus	DDH	Yes	50	25,811
2006	New Gold	DDH	Yes	4	2,620
2007	Abacus	DDH	Yes	88	41,104
2008	Abacus	DDH	Yes	113	48,025
2009	Abacus	DDH	Yes	21	7,599
2010	Abacus	DDH	Yes	114	30,948
Total				**757**	**213,071**

Figure 10.1 Ajax Drillhole Location Map



10.3 Cominco Campaigns (1981 J Series)

Documentation for Cominco J series drillholes is limited to drill logs. These were vertical percussion drillholes that were less than 100 m in length. The J series drilling was confined to the areas of what are now the Ajax East and West open pits. The drilling was completed on a grid system of 25 to 50 m. Significant portions of many of these drillholes were mined in the 1980s and 1990s during periods of open pit mine production.

10.4 Afton Campaigns (1987, 1988, 1989, 1990 Series)

Documentation for Afton drilling campaigns from 1987 to 1990 consists of geological logs and assay certificates. The drillholes had an average depth of 150 m. Typical depths ranged from 100 to 200 m. The deepest hole is 350 m. The drill core was NQ size (47.6 mm diameter) and is available at core farms near the Afton mill site. The drill core is stored in wooden core boxes and stacked in core racks. The core boxes and racks are showing advanced stages of weathering. There is no information regarding drilling and sampling protocols. The drilling procedures are not documented.

A twin sample investigation of this drilling was performed and it was concluded that the drilling could be included in the database for the 2011 resource model but limited to only supporting Indicated and Inferred Resources.

10.5 New Gold Campaigns (2004 and 2006)

New Gold completed 10 drillholes totalling 4,637 m in the Ajax area in 2004 and 2006. New Gold drilling was completed using NQ size core. Of the 10 drillholes, AX04-06 had only two samples near the collar and AX04-05 was not sampled at all; therefore, AX04-05 was not included in the resource model database.

10.6 Abacus Drilling Campaigns

Abacus drill campaigns comprise over 86% of the drill data in the 2011 resource model database. Documentation for Abacus drilling campaigns from 2005 to 2010 consists of geological logs, assay certificates, and survey reports. Most drilling was completed with NQ core but a few drillholes were completed with BQ core tails. The drill core is stored in wooden core boxes and stacked in covered core racks at the Ajax project office near Kamloops where the drillhole files are kept.

10.7 Abacus Drilling Procedures and Conditions

All Abacus drilling has been completed with DDH methods. Drilling commonly collected NQ size core (47.6 mm diameter) using a 10 ft (3.04 m) core barrel. Due to reduction while drilling, seven drillholes have BQ size (36.4 mm diameter) tails.

Drillhole collars are marked in the field with a wooden post placed in the drill hole. A tag is attached to the post identifying the drillhole. The collars are surveyed with a total station.

Core recovery is commonly excellent and exceeds 90%. The core recovery versus grade relationship was reviewed for both assays and composites and no grade trends related to percent core recovery were observed.

10.8 Geological Logging

The core logging facilities at the Abacus site are excellent and set up for all-season use. There is a reference core library and the drill core is photographed wet, prior to splitting.

In all future drill programs core logging will be direct-to-computer via the Gemcom™ GEMS core logger module which produces a Microsoft Access® database. For previous drill programs the core was geologically logged on paper forms by an Abacus geologist and the information from the paper logs was subsequently entered into the North Face Software Limited's "Lagger 3D Exploration" core logging program.

10.9 Geotechnical Logging

Geotechnical logging was conducted for each core interval and consists of rock quality designation (RQD) determination, core recovery, and other rock mass rating parameters including number of joints, weathering, rock hardness, and joint angle. The geotechnical information is recorded on a separate logging form and is entered into a computer spreadsheet.

10.10 Drill Collar Surveys

Collar locations from the Abacus 2005 to 2010 drill campaigns were surveyed using a Leica T1610 Total Station by Ward Garroway, an independent survey contractor of Kamloops, BC.

Survey methods for pre-Abacus drilling are not documented. Many collar locations have been mined out during mine operations in the 1980s and 1990s.

10.11 Down Hole Surveys

The majority of Ajax drilling completed by Abacus is oriented along drill sections with a 28° azimuth (True North) and is approximately perpendicular to the strike of the mineralization (Figure 10.1). A series of sections in the East Pit area are oriented at 118° azimuth to accommodate a change in strike of the mineralization in that area. The drillholes commonly have inclinations of -45° to -85°. Some holes are vertical.

During the process of diamond drilling, Abacus personnel completed down hole surveys using an Icefield MI3 Multi-Shot. In 2008, Abacus began using a Reflex Ez-Trac. An acid etch method was used to obtain the dip of the drillhole when the deviation survey tool was not functioning or not available.

DDH drilling completed by Afton was oriented along cross section lines at 28° and 118° azimuths (Figure 10.1). The drillholes have inclinations that vary from -45° to vertical. The majority of these drillholes are less than 200 m in depth. Down hole surveys were completed for 96 drillholes.

Percussion drillholes completed by Cominco (J series) are all vertical and do not exceed 100 m in depth. Down hole surveys were not completed due to the shallow depths drilled.

10.11.1 Magnetic Susceptibility Study

Disseminated magnetite, commonly observed in drill core and locally in veins, may exceed 5% in abundance. Both the Ice Field and Reflex survey tools used for deviation surveys use a magnetic instrument to obtain azimuth readings. Concerns regarding the reliability of the azimuth data due to the presence of magnetite observed in drill core resulted in a detailed review of the deviation survey data by AMEC. The magnetic susceptibility data were evaluated to determine the influence of magnetite on the deviation surveys.

Minimal correlation was observed between magnetic susceptibility readings, magnetite observed during geological logging, and deflections present in the deviation azimuth data. It was concluded that the deviation surveys are reasonable and suitable for use in the resource model to spatially locate mineralized intercepts.

10.11.2 Modelling Deviation Surveys

In 2008, all deviation surveys for Abacus drillholes were evaluated by AMEC using software that identifies abrupt changes in drill trajectory. The software flagged the depths at which azimuth and/or dip may be inconsistent with the trend of values above and below the survey depth or exceed a user-defined tolerance (degrees per unit length).

Anomalous changes in drill trajectory were observed in 40 drillholes and the anomalous data points were removed from the deviation survey.

Deviation surveys were modelled for 67 drillholes without deviation surveys. The modelled surveys were based on an average of deviation surveys from drillholes with similar azimuths and dips.

10.12 Drilling Results

The Ajax deposits have been sampled at drillhole spacing appropriate for a property at this level of development. The approximate drill spacing is 50 x 50 m in the areas of mineralization. The drill spacing increases at the margins of the defined mineralization. Mineralized intervals, averaging more than 0.10% Cu, commonly range between 100 to 300 m in length. High grade intercepts (greater than 1.00% Cu) are commonly less than 30 m in length and appear to be structurally controlled. Because of the irregular shape of the mineralized body, the orientation of the mineralization with respect to drill intercepts is unknown. The relationship between drill width and mineralization shape is illustrated in Figure 7.3.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sampling Method and Approach

11.1.1 *Introduction*

The following discussion applies to sampling methods in Abacus drilling programs which represent approximately 90% of the drilling carried out on the Ajax property to date. The information available regarding drilling and sampling protocols used by Cominco and Afton OC in their earlier programs is limited. Data from these early programs was statistically compared to Abacus drilling, and samples from the earlier programs were given lesser weight in the resource estimation process (see Section 11.1.5).

AMEC reviewed the Abacus sampling procedures and discussed them in their "NI 43-101 Technical Report on the Afton-Ajax E-W Deposit" dated October 31, 2008. The Abacus sampling facilities and procedures were also reviewed by Wardrop during a site visit on February 11 and 12, 2009, however no sampling took place on those days so Wardrop was not able to directly observe the sampling of core.

11.1.2 *Diamond Drilling Core Sampling*

Core is placed in wooden core boxes at the drill site with the core run footage marked on wood blocks, and the drillhole name and drill interval marked on the outside of the box. At least once per day, a geologist retrieves the full core boxes at the drill site and transports them to the core logging facility at the Ajax project office near Kamloops. The core is quick-logged for geology and areas of mineralization. An Abacus technician logs the core for core recovery and RQD.

In all future drill programs, core logging will be direct-to-computer via the Gemcom™ GEMS core logger module which produces a Microsoft Access® database. For previous drill programs the core was geologically logged on paper forms by an Abacus geologist and the information from the paper logs was subsequently entered into North Face Software Limited's "Lagger 3D Exploration" core logging program.

Sample intervals are noted on the geological log and recorded in a sample book with triplicate tags. The preferred sample interval is 3 m and this is also the maximum sample length. Samples honour geological contacts where appropriate. The minimum specified sample interval is 0.4 m.

An aluminum tag is engraved with the sample number and attached to the core box, along with a sample tag from the sample book, at the start of each sample interval.

The core is moved to the sample processing area for cutting. The core is sawn into halves using diamond-studded saw blades. One-half of the core is placed in clear plastic bags marked with the sample number, and the corresponding sample tag is placed in the bag. The remaining half-core is returned to the core box. The third sample tag remains in the sample book as a permanent record.

11.1.3 Sampling of Percussion Drillholes

There is no information regarding the sampling procedures for the J series drillholes. Sample intervals were 10 ft (3.04 m). There is no information available regarding sample sizes.

11.1.4 Twin Holes

No designated twin holes have been completed at the Ajax property.

11.1.5 Sample Pairs

Because of the lack of information for Afton OC and Cominco drilling, these drill campaigns were compared to Abacus drilling using paired samples. Sample pairs of Abacus-Afton OC and Abacus-Cominco composite samples were created using the Geostatistical Software Library (GSLIB) program *getpairs.exe*. A sample pair was created when a sample from each group was within a threshold distance of 25 m.

Abacus-Afton OC Pairs

A total of 471 Abacus-Afton OC composite pairs were examined. Histograms, probability plots, and quantile-quantile (QQ) plots were constructed to compare the Abacus and Afton OC copper values. Histograms indicate similar distributions and summary statistics. Mean grades are 0.227% Cu for Abacus samples and 0.234% Cu for Afton OC samples. The QQ plot indicates the two drilling campaigns have similar distributions.

Histograms, probability plots, and QQ plots were also constructed of gold values. Histograms and summary statistics are similar. The QQ plot indicates the Afton OC values have a low-bias in the grade ranges of 0.30 to 0.45 g/t Au. At grades greater than 0.45 g/t Au, the QQ plot suggests similar distributions but data is insufficient.

It was concluded that the copper and gold values are within reasonable tolerance. It was also concluded that the Afton OC drilling could be used for resource estimation with limitations. Model blocks with grade estimates primarily supported by Afton OC drilling should be limited to an Indicated classification.

ABACUS-COMINCO PAIRS

A total of 72 Abacus-Cominco pairs were identified. Only copper values are available for evaluation (gold was not analyzed). Histograms, probability plots and QQ plots were constructed for copper values. Histograms indicate similar distributions and summary statistics. Mean grades are 0.131% Cu for Abacus samples and 0.170% Cu for Cominco samples. The mean grades as well as the distribution observed in the QQ plot suggest the Cominco data has a high bias.

It was concluded that the Cominco data are acceptable for resource estimation with limitations. Model blocks with grade estimates primarily supported by Cominco drilling should be limited to an Inferred classification.

11.2 SAMPLE PREPARATION, ANALYSES AND SECURITY

Discussion of sample preparation, assay procedures, assay quality assurance/quality control (QA/QC) results and security is divided into sections by drill campaign, as defined in Section 10.0. The discussion below focuses on copper, gold, molybdenum, and silver assays. Palladium and platinum were analyzed for certain drill campaigns but are not considered relevant to the mineral resource at Ajax.

11.2.1 COMINCO DRILLING CAMPAIGNS

No information is available regarding sample preparation for drillholes completed by Cominco drill campaigns.

No tabulation of independent QA/QC data was provided for the Cominco drilling. The Cominco drill data was compared to Abacus drill data (detailed in Section 11.1.5).

11.2.2 AFTON DRILLING CAMPAIGNS

Samples were analyzed at the Afton mine laboratory. No information is available regarding sample preparation for drillholes completed by Afton drill campaigns.

No tabulation of independent QA/QC data was provided for the Afton drilling. The Afton drill data was compared to Abacus drill data (detailed in Section 11.1.5).

11.2.3 ABACUS DRILLING CAMPAIGNS

Abacus employed Eco Tech Laboratories Ltd. (Eco Tech) in Kamloops, BC for sample preparation and analysis.

The Eco Tech sample protocol has remained relatively unchanged throughout the Abacus drill campaigns. Minor changes were made starting with the 2010 drill campaign when Eco Tech purchased new inductively coupled plasma (ICP) equipment. Sample preparation is the same for both copper and gold analysis.

Samples with a minimum sample size of 250 g are catalogued and logged into the sample-tracking database. During the log-in process, samples are checked for spillage and general sample integrity and it is verified that samples match the sample shipment requisition provided by the clients. Samples are then assigned an Eco Tech number which is cross-referenced to the existing sample number. If necessary, the samples are transferred into a drying oven and dried.

Rock samples are crushed on a Terminator jaw crusher to -10 mesh ensuring that 70% passes through a Tyler 10 mesh screen.

Every 35 samples, a re-split is taken using a riffle splitter to be tested to ensure the homogeneity of the crushed material.

A 250 g sub sample of the crushed material is pulverized on a ring mill pulverizer ensuring that 95% passes through a -150 mesh screen. The sub sample is homogenized by rolling and bagged in a pre-numbered bag. A barren gravel blank is prepared before each job in the sample prep to be analyzed for trace contamination along with the processed samples.

Every ten samples, a repeat sample is taken to ensure proper weighing and digestion.

Gold assays are performed by fire assay on a 30 g sample size using appropriate fluxes. The flux used is pre-mixed and purchased from Anachemia Science. The flux contains Cookson Granular Litharge which is silver and gold free. The flux ratio is 66% litharge, 24% sodium carbonate, 2.7% borax and 7.3% silica (the charges may be adjusted based on the sample). Flux weight per fusion is 150 g. Purified silver nitrate or inquarts for the necessary silver addition is used for inquartation. The resultant dore bead is parted and then digested with nitric acid followed by hydrochloric acid solutions and then analyzed on an atomic absorption (AA) instrument (Perkin Elmer/Thermo S-Series AA instrument). Gold detection limit on AA is 0.03 to100 g/t. Any gold samples over 100 g/t will be run using a gravimetric analysis protocol. Internal lab standards and repeat/re-split samples (quality control components) accompany the samples on the data sheet for quality control assessment.

Copper assays are performed on a 0.5 g sample which undergoes an oxidizing digestion in 200 mL phosphoric flasks with final solution in aqua regia solution. The digested solutions are made to volume with Reverse Osmosis water and allowed to settle. An aliquot of the sample is analyzed on a Perkin Elmer/Thermo S Series AA instrument (detection limit 0.01% AA). Instrument calibration is done by verified synthetic standards purchased through SCP Science. Standards used narrowly bracket the absorbance value of the sample for maximum precision. Internal lab standards and repeat/re-split samples (quality control components), which have undergone the same digestion procedure as the samples, accompany the samples on the data sheet for quality control assessment.

Molybdenum is initially analyzed using an inductively coupled plasma-atomic emission spectroscopy (ICP-AES) method. This method requires dissolving a 0.5 g sample digested with a 3:1:2 chloride acid:nitric acid:water ($HCl:HNO_3:H_2O$) solution in a water bath at 95°C. The sample is then diluted to 10 mL with water. All solutions used during the digestion process contain indium, which acts as an internal standard for the ICP run. The sample is analyzed on an ICAP 6500 Radial ICP unit. Internal lab Certified Reference Material (CRM) is used to check the performance of the machine and to ensure that proper digestion occurred in the wet lab. Quality control samples are run along with the client samples to ensure no machine drift or instrumentation issues occurred during the run procedure. Repeat samples (every batch of 10 or less) and re-splits (every batch of 35 or less) are also run to ensure proper weighing and digestion occurred. Any check analyses are performed on a Thermo IRIS Intrepid II XSP ICP unit.

A further 31 elements (33 including copper and molybdenum) are analyzed by using the ICP-AES method above.

Results are collated and printed along with accompanying quality control data (repeats, re-splits, and standards). Results are emailed and mailed to Abacus. Any of the base metal elements (silver, lead, zinc) that are over limit (more than 1.0%) are immediately run as an assay as per the copper assay protocol above. Molybdenum is run as an assay if it exceeds 500 ppm during the ICP-AES method.

11.2.4 *Quality Assurance/Quality Control*

Beginning with the 2009 drilling program, Abacus updated the QA/QC procedures used. Abacus currently utilizes a program that consists of inserting CRMs and blanks into the sample stream at a rate of one CRM and one blank for every 20 samples tags, to maintain an insertion rate of 5% CRMs and 5% blanks. CRM samples are obtained from CDN Resource Laboratories Ltd. in Langley, BC and blank material is obtained from Eco Tech, which consists of crushed granite from Imasco Minerals Inc. in Creston, BC. As assays results are received, the results for CRMs and blanks are plotted on graphs and checked to make sure they fall within acceptable limits.

Following a drill program, Abacus submits pulp duplicates to a secondary lab at a rate of 5%; usually ALS Laboratory Group in North Vancouver, BC. These check pulps are selected at random and are re-labelled by an Abacus geologist before they are submitted.

In January 2011, Abacus started to submit 5% pulp duplicates to Eco Tech for re-analysis. These samples were selected at random and were re-labelled by an Abacus geologist so that they were blind to the lab. Pulp duplicates were submitted for all drill programs conducted in 2009 and 2010.

Going forward, Abacus will set up procedures to have check pulp and duplicate pulp assays done concurrently with drill programs.

11.2.5 SECURITY

The Ajax Project facility and core storage area is a barbed wire fenced area. Metal gates to the facility are locked when the area is not occupied. The facility contains permanent and temporary structures for offices, core logging, core sampling, lunchroom, core storage, and warehouse.

Core is collected at the drill site each day and transported to the Ajax Project facility by Abacus personnel. Core samples are delivered to Eco Tech on regular basis for processing. Core samples awaiting shipment to Eco Tech are stored in secured areas until they are shipped. Core is placed in permanent storage in covered core racks.

12.0 DATA VERIFICATION

The results of the Mineral Resource estimate represent forward-looking information that are 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 information includes commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and risks regarding the cost estimates and completion schedule for the proposed Project infrastructure, in particular the need to obtain permits and governmental approvals.

12.1 AMEC DATA VERIFICATION

AMEC completed audits of the Ajax drillhole database in 2008-2009, 2010, and 2011.

12.1.1 2008-2009 DRILL DATABASE REVIEW

This program consisted of manually comparing original log, collar and down-hole survey information against data contained in the database for four drillholes (AM-09-024, AM-09-034, AM-10-040 and AM-10-053). The data verified included collar survey, down-hole survey, geology tables, sample intervals, and sample numbers. The assay data were electronically verified for 100% of the new drilling information.

- The collar survey tables were compared to “jpg” format images of original collar survey printouts supplied by Abacus. The easting, northing and elevation values were examined. There were no serious errors noted; however a minor difference was noted for drillhole AM-10-040, attributable to the initial coordinates being used, rather than the final surveyed coordinates. AMEC recommended the final coordinate values be corrected in Abacus’s database.
- The down-hole survey tables were compared to jpg images of handwritten notes of Reflex survey results. The values for survey depth, azimuth, corrected azimuth and dip were examined. There were no errors noted either in the conversion of feet to meters or in the application of the overall correction factor. The survey results were also reviewed for kinks using AMEC’s routine “Kinkchk” program which identified six drillholes with deviation results of more than 5° per 30 m. Two of the drillholes had deviations of more than 7.5° per 45 m. AMEC recommended the down-hole

survey results for drillholes AE-06-008 and AW-06-021 be reviewed. For estimation purposes, all surveys were accepted.

- Some of the down-hole survey results contained in the database were obtained using acid-tube tests. None of the holes selected for audit contained results obtained from the acid-tube method.
- The assay tables were compared to digital files supplied directly to AMEC by Eco Tech Laboratory Ltd. of Kamloops, BC. All assay results for copper, gold and ICP values were audited. No discrepancies were noted.
- Lithology intervals and rock types contained in the database were compared to the original logs. No differences were noted. AMEC recommended that a field be added to the database to contain a code for the rock types in addition to the current descriptive field.

12.1.2 2010 Drill Database Review

The 2010 data audit consisted of review of drillholes AE-10-064 through AE-10-099, AM-10-054 through AM-10-086, AN-10-75 through AN-10-081, and AW-10-105 through AW-10-125. The data verified included collar survey, down-hole survey, geology tables, sample intervals, and sample numbers. The assay and ICP data was electronically verified for 100% of the new drilling information.

- The collar survey tables were compared to the collar survey certificate provided, and the easting, northing and elevation values were examined. There were no serious errors noted.
- The down-hole survey database was reviewed by comparing the survey notes for each hole recorded in log note books at the drill site. Of the 97 drillholes reviewed, four were not surveyed for various reasons, and only eight minor numerical errors were noted.
- Kinkchk was run to determine whether or not there were any possible unrealistic kinks detected in the surveys. Drillhole AM-10-058 was found to have a significant deviation. AMEC also ran Kinkchk on the entire Abacus drillhole database. The results of this test produced kinks in seven holes including AM-10-058 from the previous test, plus AE-06-008, AE-07-037, AN-08-020, AN-08-060, AW-06-021, and AW-06-041. For estimation purposes, all surveys were accepted.
- AMEC visually compared the lithology intervals and rock types contained in the database to the original logs for nine drillholes. No differences were noted. AMEC again recommended that a field be added to the database to contain a code for the rock types in addition to the current descriptive field.
- AMEC visually compared the sample intervals and sample numbers contained in the database to the original logs for the four holes selected for audit. No discrepancies were noted.

12.1.3 2011 QA/QC REVIEW

STANDARD REFERENCE MATERIALS

AMEC reviewed the results from the three Standard Reference Materials (SRMs) (CM5, CM6 and ME7) used by Abacus as part of their QA/QC protocol at Ajax. Abacus submitted a total of 543 SRMs and 9,976 samples for assay during the 2010 drill campaign, resulting in an SRM insertion rate of 5.4%. This is slightly above the AMEC recommended insertion rate of 5% for SRMs for each metal of economic importance. In AMEC's opinion the SRM results indicate the assays are of sufficient quality to support resource estimation.

- A total of 44 samples of SRM CM-5 were included for analysis, only two results fell outside the acceptable limits, one for copper, and one for gold.
- A total of 275 samples of SRM CM-6 were included for analysis. Two results for gold fell outside the acceptable limits.
- A total of 224 samples of SRM ME-7 were included for analysis. In general, the gold results fall above the stated best value. Four gold results fell slightly above outside of two standard deviations (however the standard deviations are too tight as a consequence of rounding the results to two significant places) from the best value. While the individual results from this standard appear to be somewhat variable, the average result is only 3.5% higher than the best value, which is an acceptable value, especially considering the low gold grade of this SRM. The copper results fall within the acceptable limits. AMEC recommended that the results for the four samples that were noted to exceed the acceptable limits established by AMEC be reviewed by Abacus and consideration given to resubmitting the samples in affected batches for re-analysis.
- AMEC did not note any significant biases indicated by the SRM results. All biases were below 5%.

DUPLICATES

Abacus supplied the following gold and copper results from pulp duplicate samples to AMEC:

- An evaluation of the pulp duplicate results indicated that the absolute value of the relative difference (AVRD) for gold and copper at the 90th percentile were 17% and 20% respectively. AMEC considered the results acceptable for gold, but precision for copper could be increased. AMEC recommended reviewing the sample preparation procedures to improve the precision achieved by the assay laboratory. Possible improvements include finer crushing of the sample prior to splitting, better blending and/or splitting of the sample, and splitting of a larger sub-sample for pulverization.

Blanks

Abacus submitted 490 blank pulps during the 2010 drill program.

At Ajax, AMEC considers that pass/fail limits should be set at two times the detection limit, being 0.06 g/t for gold, and 0.02% for copper. Results for sample 10E27490 appear to be mislabelled as a blank and are likely a regular sample. There appears to be occasional sample contamination for gold, but this is not considered to be material to the resource estimate.

Check Assays

At the end of the 2010 drill program, 5% of all Abacus samples were selected at random as check assay samples. A total of 490 pulps were sent for re-assay at ALS Chemex, for direct comparison with the results from Eco Tech.

The ALS Chemex vs. Eco Tech plots shows very good correlation for all paired data. Detection limits for gold and copper differed between the two labs. The detection limit for gold at Eco Tech is 0.03 g/t, compared to 0.01 g/t at ALS Chemex. The detection limit for copper at Eco Tech is 0.01%, compared to 0.001% at ALS Chemex, which may affect the correlation for gold and copper between the two laboratories due to differences in analytical precision.

On examination of the gold data, AMEC concluded that Eco Tech results are unbiased compared with the ALS Chemex results.

For copper, Eco Tech results are biased 2% lower than the ALS Chemex results. Relative biases between laboratories of ±5% are acceptable; therefore the ALS Chemex results are considered to confirm the Eco Tech results.

12.2 Comments on Section 12

The AMEC QPs consider that a reasonable level of verification has been completed during the audits, and that no material issues would have been left unidentified from the programs undertaken.

In the opinion of the AMEC QPs, the data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource and Mineral Reserve estimation.

The AMEC QPs applied a confidence restriction to legacy data as indicated in Section 14.13.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The Ajax Project will mine and process 60,000 t/d of porphyritic chalcopyrite-bearing hornblende diorite material.

In July 2009, Wardrop completed an historical review as part of a PEA-level study. Test work results from G&T, Report No. KM2228, dated October 2008, were used as the basis for the process design parameters in the PEA study. Since the PEA level report was issued, further metallurgical testing as confirmatory testing and variability testing was also completed. In November, 2010, G&T reported the test work results in Report No. KM2688. A memorandum was issued in January 2011 regarding the copper and gold feed grade and recovery relationships, based on the historical test results that included the most recent data available at the time. This memorandum, together with the G&T test reports, is provided in Appendix B.

13.2 HISTORICAL TEST PROGRAMS

The test work programs and reports listed in Table 13.1 were discussed in the PEA issued by Wardrop in 2009.

Table 13.1 Historical Test Work Programs and Reports

Document or Test Program	Author or Laboratory	Date
Mineral Resource Estimate for the Ajax West Deposit	Beacon Hill Consulting	May 2007
Metallurgical Testing on Samples from Ajax & DM-Audra Project, Report No. KM1929	G&T	May 2007
Summary Report on the 2007 and 2008 Abacus-New Gold Inc. Joint Venture Diamond Drill Program on the Ajax Property	Abacus	September 2008
Metallurgical Testing on Samples from Ajax & DM-Audra Project, Report No. KM2228	G&T	October 2008
NI 43-101 Technical Report on the Afton-Ajax E-W Deposit	AMEC	October 2008
Preliminary Metallurgical Data, G&T Excel Files – Test Program No. KM2350 (Molybdenum testing)	G&T	February 2009

This feasibility study builds on the results and premises of the previous findings, as well as the subsequent test work programs between 2009 and 2011. The latter documents are listed in Table 13.2.

Table 13.2 Subsequent Test Work Programs and Reports

Document or Test Program	Author or Laboratory	Date
Ajax Copper/Gold Project, Kamloops, British Columbia – Preliminary Assessment Technical Report. Doc. No.0854610100-REP-R0001-02	Wardrop	July 2009
Pre-Feasibility Metallurgical Testing, Abacus Mining Exploration Corp., Ajax Project. Report No. KM2435	G&T	October 2009
Ajax Copper/Gold Project, HPGR Energy Study, Doc. No.1054610100-REP-R0001-01	Wardrop	February 2010
High Pressure Grinding and Ball Mill Grindability Tests on Copper Gold Ore from the Ajax Project in British Columbia, Canada. Project No: 2337 4191	Krupp Polysius Research Centre	October 2010
Feasibility Metallurgical Testing, Ajax Project. Report No. KM2688	G&T	November 2010
Memorandum – Grade Recovery	L. Duncan to A. de Ruijter, Wardrop	January 25, 2011
Additional Flotation Testing, Ajax Feasibility Study, Abacus Mining & Exploration Corp. Report No. KM2905	G&T	February 2011

13.3 MINERALOGY

A number of mineralogical evaluations were performed on various samples from the Ajax deposit for the purposes of these previous studies and these evaluations led to the following findings.

The copper mineralization in the Ajax deposit is a porphyry style hypogene rock type characterized by the mineral chalcopyrite which is associated with the following minerals: variably occurring pyrite, locally enriched molybdenite and with tennanitite/ tetrahedrite, covellite, chalcocite, bornite, magnetite, and rare native copper also identified. The chalcopyrite mineralization occurs as stringers, fracture fillings, blebs and disseminations. Gold is present mainly in the range of 0.1 to 0.3 g/t Au. Gold was initially considered to be closely correlated with the copper mineralization, but an analysis of the subsequent metallurgical data has indicated that the apparent relationship is not statistically meaningful. The presence of rare visible gold strongly suggests that there may be a bimodal occurrence of the precious metal. Palladium has also been identified in the deposit and appears to be closely associated with the copper mineralization in localized areas. Palladium values do not exceed 0.5 g/t.

In the most recent metallurgical test program Report No. KM2688, G&T conducted mineralogical studies on the Year 1 to 5 composite sample. The investigation used

QEMSCAN® Particle Mineral Analysis as a means of determining the mineralogical content and fragmentation properties of the sample. The mineralogical breakdown is shown in Table 13.3.

Table 13.3 Mineral Composition and Copper Deportment Year 1 to 5 Composite: KM2688

Mineral Composition (%)				Copper Deportment (%)	
Sulphide Minerals	**Mass**	**Gangue Minerals**	**Mass**	**Copper Bearing Minerals**	**Mass**
Chalcopyrite	0.78	Iron Oxide	0.8	Chalcopyrite	82.9
Bornite	0.01	Quartz	2.1	Bornite	1.1
Chalcocite	0.02	Feldspars	48.6	Chalcocite	4.0
Covellite	0.08	Amphibole	18.2	Covellite	12.0
Tennantite	<0.01	Epidote/Chlorite	14.7	Tennantite	0.1
Pyrite	0.78	Micas	4.3		
		Carbonates	3.4		
		Pyroxene	2.0		
		Sphene	1.7		
		Other Gangue	2.6		
Total	**1.67**	**-**	**98.3**	**-**	**100**

The mineralogical analysis test work was performed on the variable samples by using QEMSCAN®. Both the Bulk Mineral Analysis function and Liberation function were employed in the analyses. For all 50 samples tested, the majority of the copper (at between 96 and 100%) was present as chalcopyrite. Minor amounts of bornite and chalcocite were observed in some of the samples. The main non-sulphide gangue minerals were present as the silicate minerals quartz, feldspars and amphibole-pyroxene, which constituted between 60 to 80% of the feed sample. The other main gangue component of the feed sample was the chlorite/epidote group of silicate minerals, constituting 10 to 15% of the feed mass. The main sulphide gangue mineral identified was pyrite. The mineralogical make-up of the variability samples is consistent with those of the Year 1 to 5 composite sample lending strength to the data.

Pyrite levels were deemed to be low, with the majority of the samples indicating pyrite to copper sulphide ratio of 1:1 or less. This result is significant because elevated pyrite levels can impact the ability to achieve acceptable copper concentrate grades. Elevated pyrite levels can also have a detrimental effect on gold recovery, depending on the deportment of the gold within the sample host.

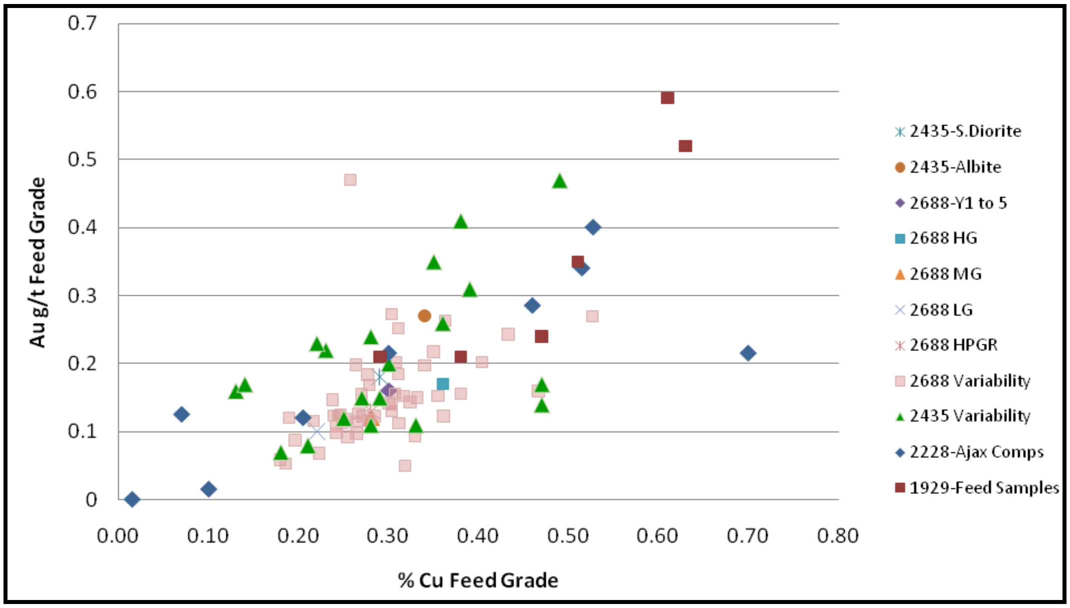
Krupp Polysius also conducted a mineralogical composition analysis of the two rock type samples in order to characterize process parameters for the HPGR technology, which has subsequently been incorporated into the process design. Compared with the results presented in Table 13.3, similar minerals, and abundance values, were obtained.

13.4 FEED GRADE

A feed grade analysis characterization was conducted throughout the studies at various levels. The original analysis, conducted in 2007 and based on a limited number of feed samples, appeared to show an apparent correlation between the copper and the gold present in the deposit. An overall synopsis of the copper feed grade to gold feed grade relationship was considered to be required to prove or disprove this perceived correlation.

A graph of the copper feed grade versus gold feed grade for the various test composite samples (Figure 13.1), shows that there is very little correlation.

Figure 13.1 Feed Grade of Copper vs. Feed Grade of Gold



The copper feed grades generally vary between 0.13 and 0.63% Cu, and the gold feed grades generally vary between 0.05 g/t and 0.59 g/t Au. However, the individual results, as well as the results from the various complete test program campaigns, appear to indicate a visual trend between the copper and gold feed grades. Nevertheless, the variation in the results is considered to be too erratic for a meaningful statistical evaluation. A complete list of the head assay values used in the analysis is included in Appendix B.

The Year 1 to 5 composite sample, which was tested during the most recent test program and presented in Report No. KM2688, has a feed grade value of 0.29% Cu and 0.16 g/t Au. These feed values were used as the basis of design for the plant.

13.5 Grindability

13.5.1 *Bond Ball Work Index*

A Standard Bond Ball Mill Work Index (BWi) test was performed on a sample identified as "Ajax 9" in test program KM2228 (i.e. Metallurgical Testing on Samples from Ajax & DM-Audra Project, Report No. KM2228), with a reported result of 19.7 kWh/t. This result was reported in the appendix of the test report, but differs from the 19.8 kWh/t result provided in the body of the report. This discrepancy may be the result of rounding.

Comparative work indices for the additional test samples from the KM2228 program were also determined using results from the grind test determinations. The arithmetic average of the comparative work index for the nine Ajax samples tested was found to be 14.0 kWh/t. However, the Ajax 9 sample is of the type of rock which constitutes 83% of the potential ore, and had the much lighter actual grinding work index result of 19.7 kWh/t.

Additional grindability tests were conducted at the prefeasibility level. Test Report No. KM2435 involved the testing of an additional two samples, using the BWi methodology. Comparative testing results were calculated on an additional 20 samples, based on grind times. The Sugarloaf Diorite sample, which will account for 83% of the mineral deposit, gave a BWi result of 19.4 kWh/t. This result is consistent with the test result obtained in Report No. KM2228 on the Ajax 9 sample. The Albite sample obtained a result of 18.4 kWh/t. The actual BWi test results are reported in Table 13.4. The average comparative work index results for the 20 variability test samples completed in the test program reported in KM2435, was given as 18.2 kWh/t as shown in Table 13.5.

Test program KM2688 involved ten additional BWi tests to verify the 19.7 kWh/t test result used in the originally-proposed PEA study. The results of this test program are provided in Table 13.4, together with the original Ajax 9 sample results, as well as the results from test program KM2435.

Table 13.4 Summary of Measured Standard Bond Ball Mill Work Indices

Test Program	Sample ID	Measured BWi (kWh/t)
KM2228	Ajax 9	19.7
KM2435	S. Diorite	19.4
	Albite	18.4
KM2688	Met 2	17.6
	Met 3	17.8
	Met 10	17.1
	Met 15	18.7
	Met 17	17.8
	Met 20	21.1
	Met 22	20.9
	Met 23	20.9
	Met 26	18.5
	Met 46	19.4
Average	-	19.0
Maximum	-	21.1
Minimum	-	17.1

Comparative work indices were also calculated for all the variability samples tested in the Report No. KM2688, and this resulted in a calculated average value of 18.0 kWh/t (see Table 13.5).

A comparison of the test results of the two methods for obtaining grindability data is provided in Table 13.5.

Table 13.5 Comparison of Bond Work Index Results

	Comparative BWi KM2228 (kWh/t)	Comparative BWi KM2435 (kWh/t)	Comparative BWi KM2688 (kWh/t)	Measured BWi All Tests (kWh/t)
Average	14.0	18.2	18.0	19.0
Standard Deviation	2.8	2.8	3.0	1.4
Minimum	11.0	12.2	11.3	17.1
Maximum	19.7	23.8	26.1	21.1
Number of Tests	9	20	50	13

Based on these results, it is apparent that the original basis for using 19.7 kWh/t as the grindability index was justified. This value of 19.7 kWh/t has been used in the design criteria of this study.

The average comparative grindability work index results are consistently lower, and have a higher standard deviation, than the measured BWi values. Wardrop conducted a detailed investigation into the methodology that G&T used to calculate the comparative work index, and concluded that the methodology may have been in error, and/or was not applied consistently. The investigation also indicated that the methodology used to determine the comparability index was extremely dependant on the number of samples tested using the results from the BWi procedure.

BWi tests were also performed in Germany by Krupp Polysius, as part of the HPGR test program. Krupp Polysius tested a composite sample based on material of Year 1 to 5 slated for treatment. Two tests were performed, one on the HPGR feed sample (the as-received sample), and one on the HPGR discharge product sample.

The grindability work index result obtained for the HPGR feed sample was 19.3 kWh/t, which is similar to the BWi results used in previous studies. The HPGR discharge product sample gave a BWi result of 17.6 kWh/t, which is 9% lower than the result obtained for the feed sample. This difference in power required was attributed to micro-fractures that were developed during the HPGR crushing operation. At this time, the lower BWi number of 17.6 kWh/t will not be used in any of the mill sizing calculations since this outcome was not conformed. However, it offers as a potentially significant power savings.

13.5.2 Crusher Work Index

Crusher work index (CWi) tests were performed at Philips Enterprises, LLC (Philips), in Golden, Colorado. Philips considered the physical size of the diameter of the individual samples to be slightly too small for the nature of tests. Although the test work was completed, Philips recommended additional conformatory crusher work index testing using appropriately-sized sample material.

A total of four tests were completed on drill core samples, and the results are given in Table 13.6.

Table 13.6 Crusher Work Index Results

Test Number	Sample ID	CWi (kWh/t)
1	1007-51	7.68
2	1007-52	5.33
3	1007-53	8.28
4	1007-54	4.75

As a result of the high degree of variability and the small number of samples tested and results obtained, the conservative number of 7.68 kWh/t was selected for the crusher work index for design purposes.

13.5.3 *ABRASION INDEX*

G&T performed a Bond Abrasion Index (Ai) test on a sample of the Sugarloaf Diorite rock type during the KM2435 test program, with a reported result of 0.26 g.

13.5.4 *HPGR TESTING*

Two HPGR test programs were completed since the PEA study was conducted in 2009.

KÖPPERN MACHINERY, AUSTRALIA

The first HPGR test program was completed by Köppern Machinery in December 2009, at its testing facility at the University of British Columbia, Vancouver BC. This PEA-level work was conducted to determine the feasibility of using stage crushing with HPGR as the third stage, as an alternative to a conventional crushing and SAG milling circuit. The results were documented in the January 2010 report, "10546100-REP-001-01 Ajax HPGR Trade-off Study". The report concluded that the sample material supplied by Ajax for testing was amenable to the HPGR process.

KRUPP POLYSIUS, GERMANY

Krupp Polysius subsequently conducted a pilot plant test program with the objective to evaluate and confirm the conclusions reached by Köppern Machinery's. The results were reported in a detailed test report issued in October 2010. The most relevant test parameters required for equipment sizing were determined, including:

- specific throughput rate
- specific grinding force
- specific energy input versus specific grinding force.

The cut/transfer size and the circulating load of the product with the selected screen size aperture size was also evaluated and determined during the test program.

Two samples representing the two different rock types were tested:

1. the primary sample was made up of the Year 1 to 5 composite, predominantly Sugarloaf Diorite rock type
2. the secondary sample was a sample of the Albite rock type.

The test program included single-pass tests at three different pressure settings and three different moisture levels in areas in order to determine the optimum operating parameters for the test apparatus. The test program also incorporated locked cycle testing in order to simulate the product size distributions to be expected in an industrial sized HPGR circuit. The single-pass test program was used for comparison of the secondary rock type sample with the primary sample.

Krupp Polysius outlined the key parameters of the test work in the appendices of its report found in Appendix B. The results of the test work are summarized in Table 13.7.

Table 13.7 Summary of HPGR Test Findings

Description	Unit	Value
Specific Grinding Force	N/mm^2	3.5
Specific Energy Input	kWh/t	1.9
Specific Energy Consumption	kWh/t	2.98
Average Specific Throughput	t*s/m^3*h	260
Screen Aperture	mm	6
Feed Particle Size, P_{100}	mm	45
Product Particle Size, P_{80}	mm	3.25
Screen Oversize, Percentage of HPGR Discharge	%	50

The results indicated that the throughput was not found to be sensitive to moisture content of the feed material within the range tested, namely between 1 and 5% moisture content. It is therefore expected that there will be no significant influence on the throughput of the Ajax material, even though the moisture of the feed to the HPGR units may be as high as 7% as a result of the recirculation of the wet sizing screen oversize material as indicated in the design of the circuit.

Krupp Polysius performed their proprietary LABMill testing procedure in order to investigate the impact of HPGR treatment on the energy requirements in the subsequent ball milling stage. As mentioned previously, the ball mill energy requirements of the HPGR product has been found to be lower compared with a conventional crusher product as a result of the micro-fracturing of the particle caused by the HPGR treatment.

The LABMill tests compared the energy savings in the downstream ball milling process when using HPGR-processed material with that of a conventionally crushed sample. For the primary sample tested, the following apparent energy savings to obtain the indicated P_{80} grind sizes were obtained when using the HPGR:

- P_{80} = 200 µm, 14% energy savings
- P_{80} = 90 µm, 9% energy savings.

13.6 FLOTATION TESTS

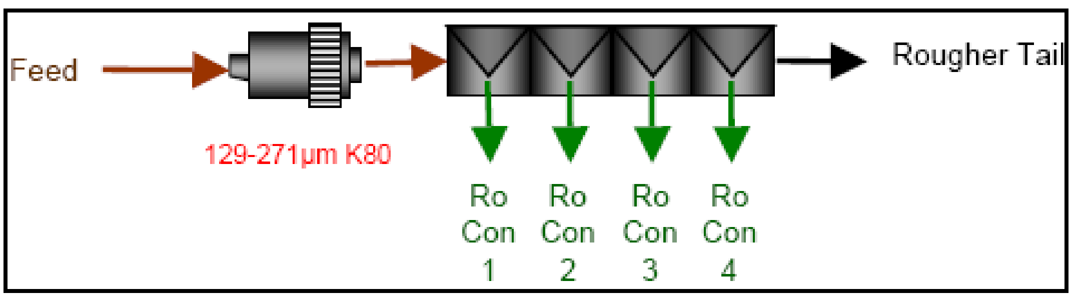
G&T conducted flotation test work from 2007 onwards, primarily to determine grade and recovery relationships for the copper and the gold for sample material having various feed grades.

A straightforward flotation methodology was developed during the test work phase for this material, which proved to be responsive to using a very basic flotation procedure with basic flotation reagents. The final reagent scheme used potassium amyl xanthate (PAX) as the collector, methyl isobutyl carbinol (MIBC) as the frother, and lime in the regrind and cleaner circuits to adjust the pH of the slurry.

Flotation tests at various grind sizes were conducted to establish the primary grind size for optimum rougher recovery. Although the results obtained suggested a trend towards improved recovery with finer grind size, this trend became less apparent at a P_{80} size of approximately 214 µm. For design purposes, the P_{80} size of 214 µm was selected based on liberation and recovery as giving the optimal results.

Relationships between rougher mass recovery and copper metal recovery were developed and compiled using kinetic rougher flotation tests that followed the flowsheet provided in Figure 13.2.

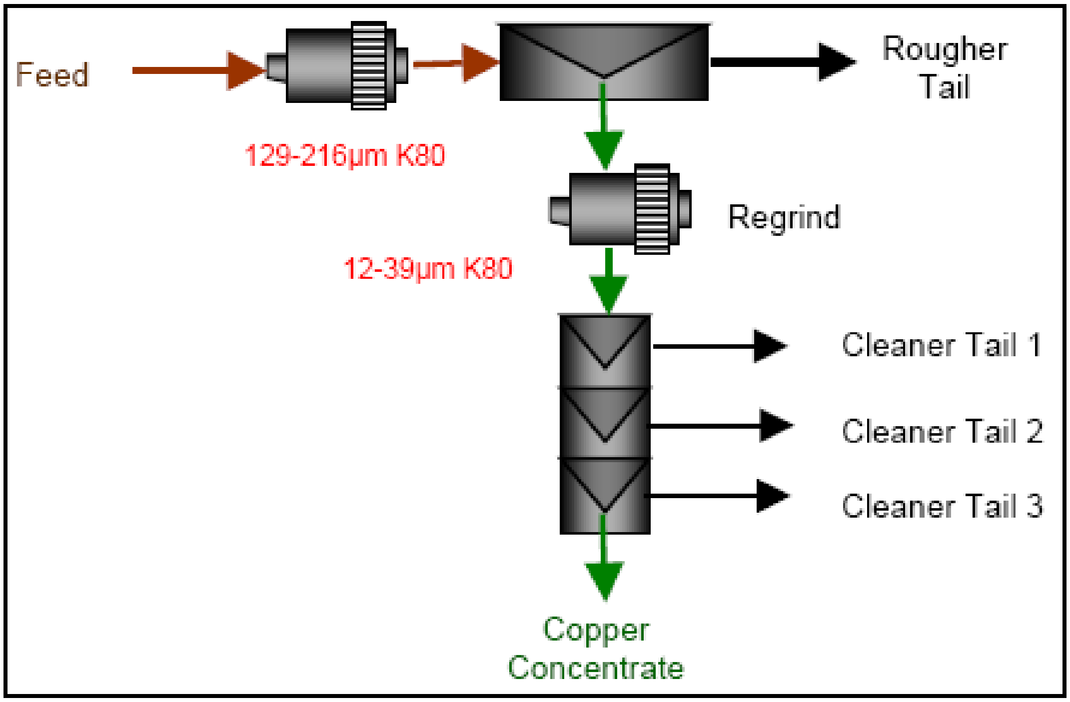
Figure 13.2 Rougher Flotation Flowsheet



Rougher concentrate masses of approximately 6 to 10% were selected as the target mass recovery, and then applied to the design of the flotation circuit.

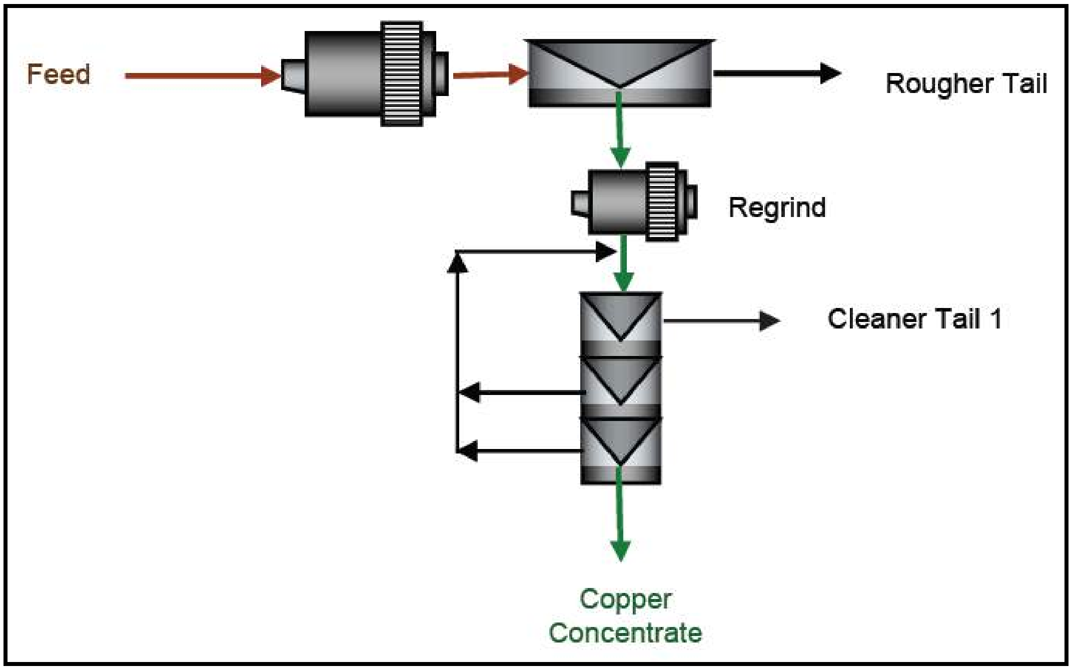
Cleaner flotation batch tests were conducted using the flowsheet provided in Figure 13.3, in order to define the reagent and regrind size for optimum cleaner circuit performance. This flowsheet was subsequently used for open cycle variability testing.

Figure 13.3 Batch Cleaner Flotation Flowsheet



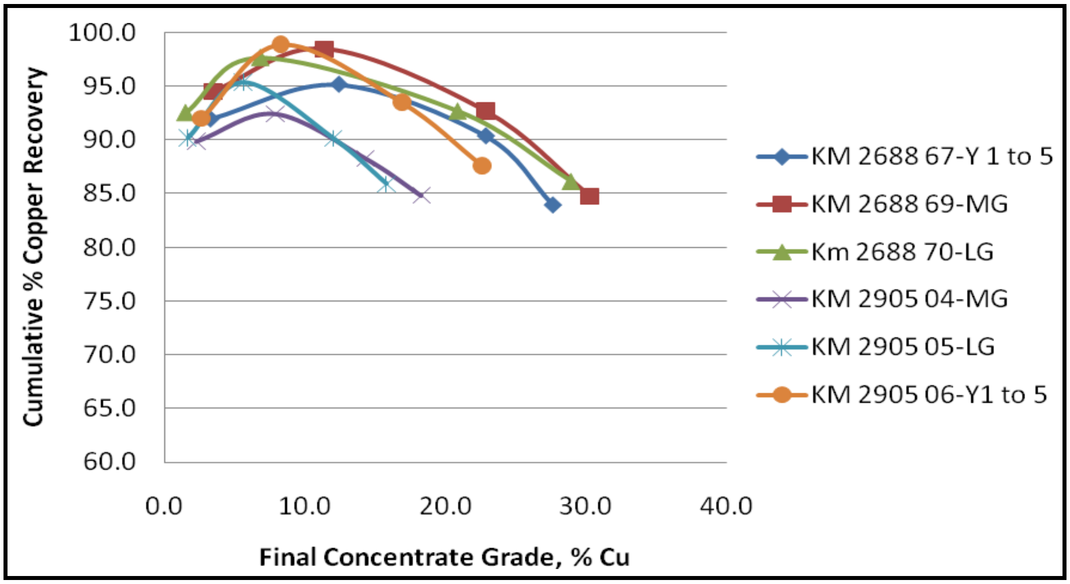
Using the open circuit flowsheet shown in Figure 13.3 as a basis, the locked cycle flowsheet provided in Figure 13.4 was developed, and then used throughout the test programs on various samples to determine the repeatability and robustness of the test procedure. Results from the locked cycle tests were compiled, analyzed and used as the basis for the design of the flotation circuit.

Figure 13.4 Locked Cycle Flotation Flowsheet



The regrind target size has changed slightly throughout the various test programs performed since 2007. The KM1929 test program on the Master Composite sample established an optimal regrind P_{80} size between 16 and 19 µm prior to the cleaner flotation stages. The subsequent test program, KM2435, targeted this regrind size, although at times finer regrinds were obtained during the actual testing. The test program conducted for this feasibility study, KM2688, had a target regrind P_{80} size of 18 µm. However, during the test program a P_{80} size of between 8 to 17 µm was obtained with a median of 12 µm. At that time, a request was made to review of the regrind size in the supplementary test program KM2905 conducted on three composite samples namely, the Low Grade, the Medium Grade and the Year 1 to 5 composite samples. The main objective of the KM2905 test program was to determine if there was a significant impact on metallurgical performance when the coarseness of regrind size was increased to a particle size P_{80} of about 20 µm. The test procedure included rougher kinetic flotation tests followed by locked cycle flotation tests. The locked cycle flotation test work showed that between 85 and 88% copper recovery was achieved to final concentrates which graded between 16 and 23% copper at a P_{80} regrind size of approximately 27 µm. The results obtained are shown in Figure 13.5.

Figure 13.5 Copper Recovery vs. Concentrate Grade - KM2688 and KM2905



As shown in Figure 13.5, the results from Tests KM2905-04 to KM2905-06, which used the coarser regrind size, are inferior to those obtained using the finer regrind sizes. On the basis of these results and the results from all the locked cycle tests to date, a regrind P_{80} size of 18 µm was selected for its optimal performance.

13.6.1 TEST PROGRAM KM2688, ROUGHER AND CLEANER KINETICS

A portion of the most recent test program, KM2688, was designed to determine the metallurgical performance of various composites samples compiled from the drill core samples. The composites were prepared to produce a range of feed grades, which fell into the following subsets (see Table 13.8).

Table 13.8 Composites for Test Program KM2688

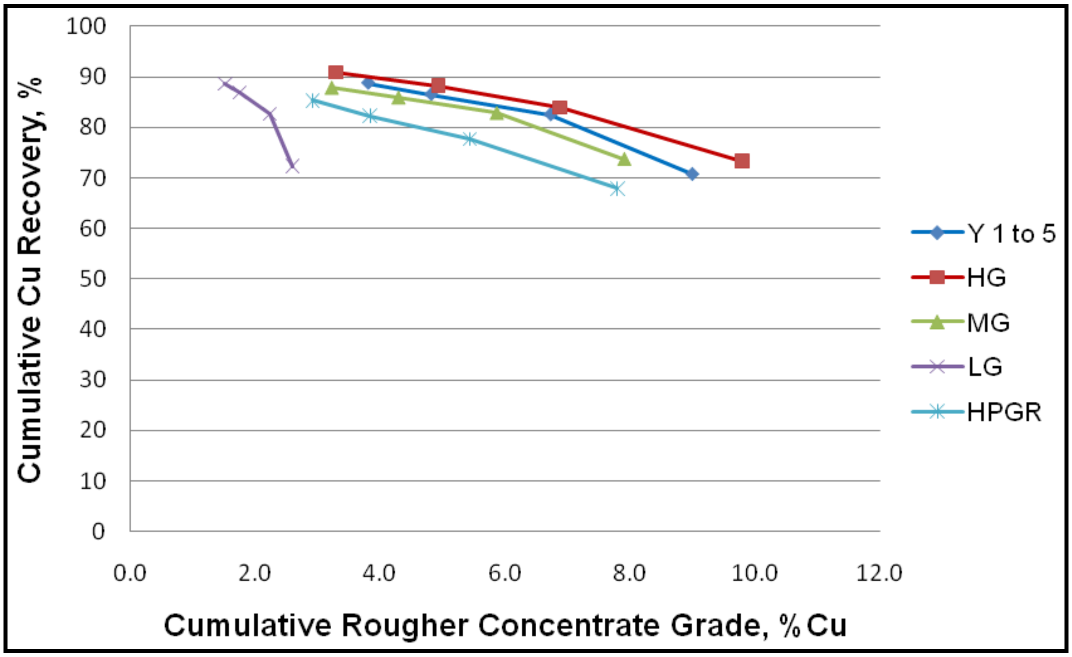
Composites	Cu (%)	Au (g/t)
Low Grade feed	0.24	0.08
Medium Grade Fee	0.27	0.13
High Grade feed	0.37	0.15
Year 1 to 5 feed	0.29	0.13
HPGR sample	0.27	0.13

As a separate exercise, the product from the HPGR test was used in the flotation to determine if there were any distinguishable recovery response differences. This HPGR sample had a feed grade of 0.27% Cu and 0.13 g/t Au, similar to the feed grades of both the Medium Grade and the Year 1 to 5 composite samples.

Open circuit rougher and cleaner flotation tests were conducted on each of the sample composites, followed by duplicate locked cycle tests on each of the composited samples.

Results of the rougher kinetic tests are shown in Figure 13.6 through Figure 13.8.

Figure 13.6 Copper Recovery vs. Concentrate Grade, Rougher Kinetic Testing – KM2688



The copper recovery versus rougher copper concentrate grade curves shown in Figure 13.6 indicate that there is little difference in the recovery grade curves, except in the Low Grade composite sample case.

To better illustrate the relationship between copper recovery and concentrate grade, the results were applied to a mass recovery versus copper recovery curve (Figure 13.7), and a copper recovery versus flotation time duration curve (Figure 13.8).

Figure 13.7 Copper Recovery vs. Mass Recovery, Rougher Kinetic Testing - KM2688

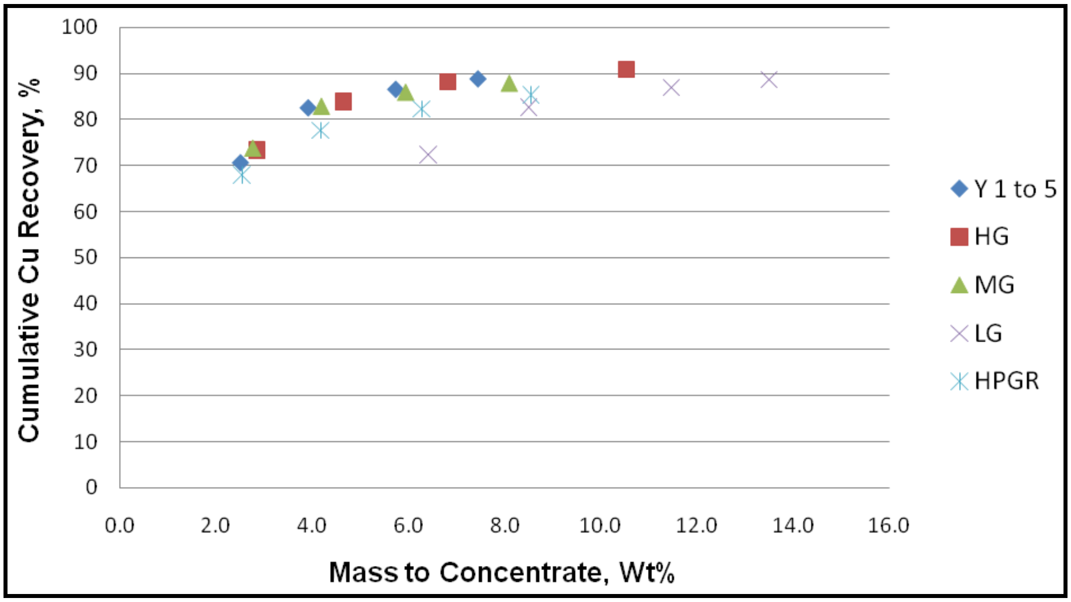
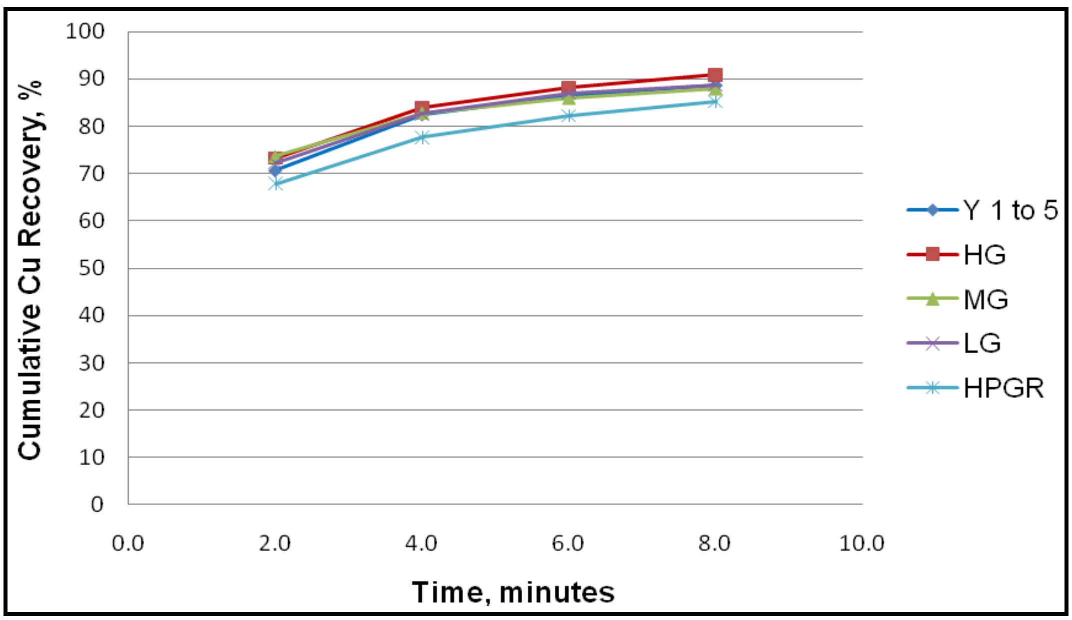


Figure 13.7 shows more clearly that the mass recovery relationship is comparable for all the samples tested.

Figure 13.8 Copper Recovery vs. Flotation Time, Rougher Kinetic Testing - KM2688



The copper recovery versus flotation time graph (Figure 13.8) shows that all the samples appear to float at the same rate, except for the HPGR sample, which appears to respond in a similar manner to the other samples but with a slightly inferior recovery response (Figure 13.6). Because the test program was limited, no absolute conclusions can be drawn from these results regarding the HPGR samples flotation behaviour.

A set of open circuit cleaner tests were also conducted on the same set of composite samples.

Figure 13.9 Copper Recovery vs. Concentrate Grade, Cleaner Kinetic Testing - KM2688

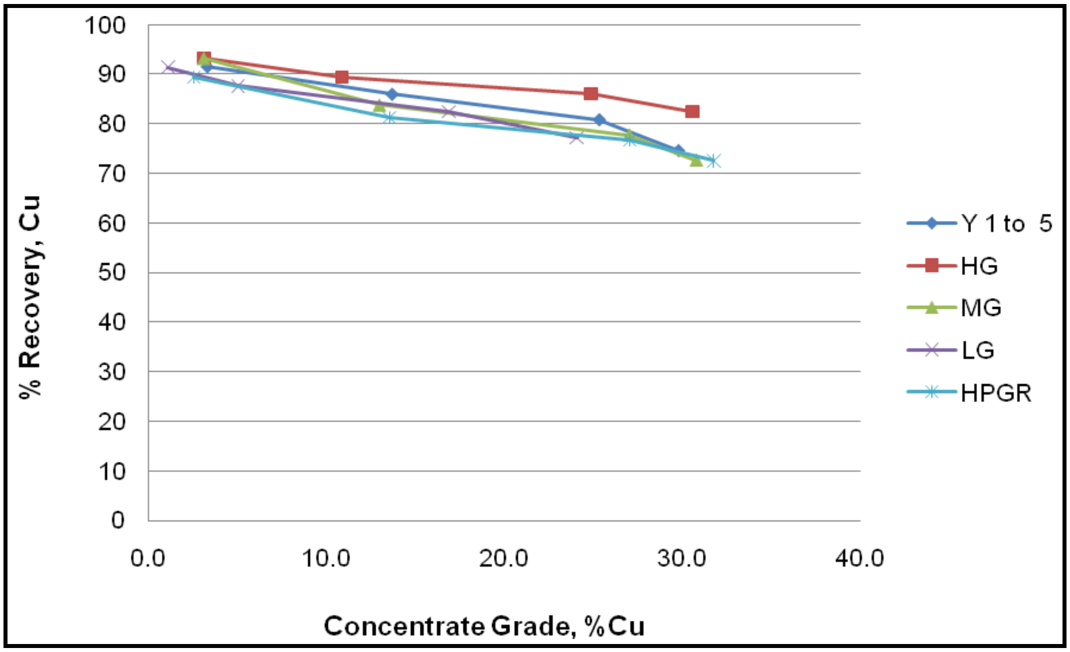


Figure 13.9 shows the combined results of all the grade recovery curves for the composites. Again, there is little discernable difference between any of the curves for the five samples tested. A slightly better recovery and grade response was obtained with the High Grade feed sample and could be attributed to a difference in mineralogy as a result of the higher feed grade.

Figure 13.10 Copper Recovery vs. Mass Recovery, Cleaner Kinetic Testing – KM2688

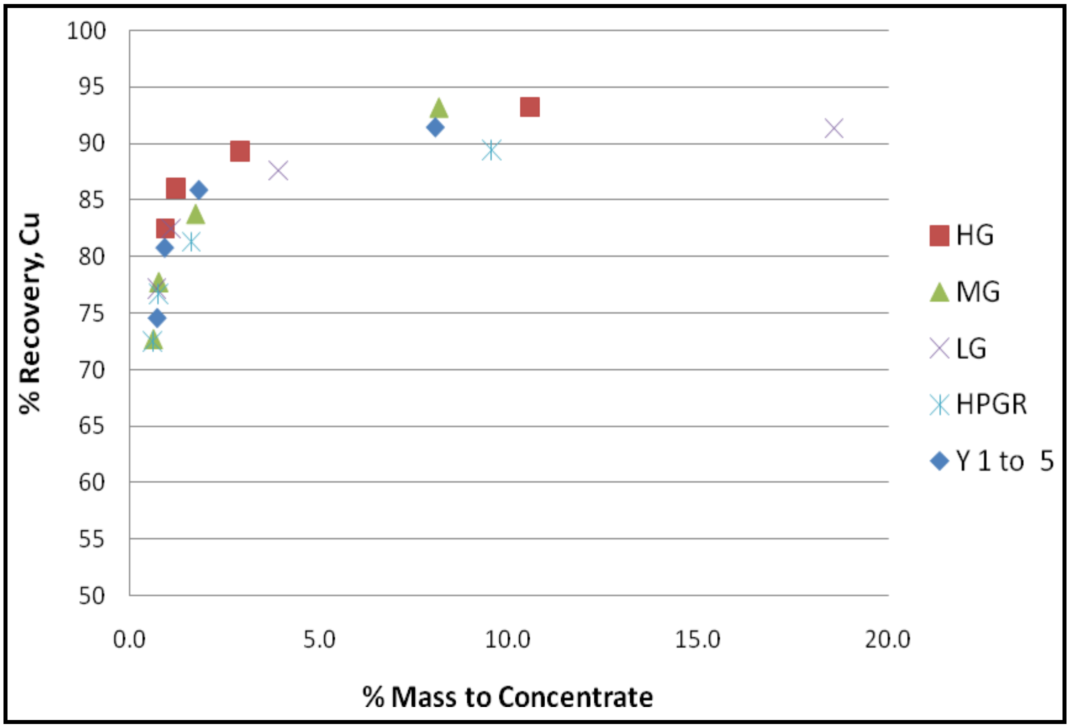


Figure 13.10 shows more clearly that the mass recovery relationship is comparable between all five of the samples tested, and consistent with the previous results obtained.

Based on all the preceding analysis of the results obtained, it is apparent that the overall metallurgical performance is consistent and robust, and very little variation in flotation response was noted between sample composites.

13.6.2 *Flotation Circuit – Basis for Design*

As mentioned in Section 13.1, a detailed memorandum was issued in January 2011 which updated the various copper and gold relationships developed in the PEA study. A summary of the relevant information from the memorandum is included in the following discussion.

The locked cycle test data listed in Table 13.8 were used to determine the various copper and gold relationships. The data spans all the test programs to date, starting at the PEA level study. Data from duplicate tests were omitted to reduce sample bias.

The complete test data details are available in the reports issued by G&T which are included in their appendices.

Table 13.9 Summary of Locked Cycle Tests used for Characterizing Copper and Gold Relationships

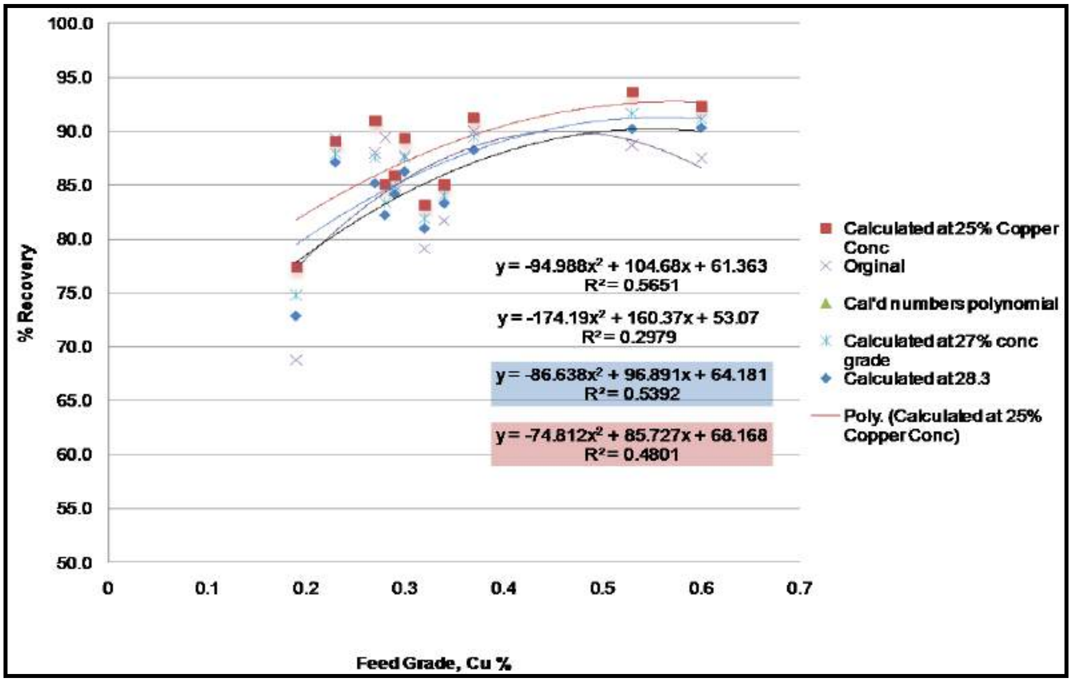
Test Number	Sample Description
PEA Level Study	
1923-25	High Grade Master Composite sample
2228-23	Low Grade sample
2228-25	High Grade sample
2228-26	Mid-grade sample
Pre-Feasibility Level Study	
2435-14	Sugarloaf Diorite sample
2435-16	Albite sample
Feasibility Level Study	
2688-72	Year 1 to 5 Composite sample
2688-73	High Grade sample
2688-74	Medium Grade sample
2688-75	Low Grade sample
2688-76	HPGR reject sample

COPPER RECOVERY ANALYSIS

An analysis of the flotation results was completed in order to produce a usable feed grade to recovery relationship for copper.

The data analysis used the results of the locked cycle test programs (provided in Table 13.8) as the basis to develop trend lines and generate equations. Figure 13.11 was generated from the data set created as copper Feed Grade versus % Recovery of copper at three particular copper concentrate grade values. The copper concentrate grades used to generate the points on the graph were 25, 27 and 28.3% Cu. These grades were selected to straddle the anticipated final product concentrate copper grade produced by the mine.

Figure 13.11 Feed Grade vs. %Recovery Locked Cycle Test Data



In the context of all the available information, and as an overall comparison, the actual uncorrected data points have been included and labelled "original" in the graph. The "original" data means the recovery obtained from a particular head grade, with the actual copper concentrate grade obtained during the testing.

Trend lines were taken from each data set to describe relationships of Copper Recovery to Copper Feed Grade, thereby generating Equation 1, Equation 2, and Equation 3, together with the corresponding correlation coefficients as indicated for in these equations. In the equations, *y* is equal to the copper recovery and *x* is equal to the copper concentrate grade.

For 25% copper concentrate grade:

Equation 1 $y = -74.812x2 + 85.727x + 68.168 \quad R^2=0.4801$

For 27% copper concentrate grade:

Equation 2 $y = -86.638x2 + 96.891x + 64.181 \quad R^2=0.5392$

For 28.3% copper concentrate grade:

Equation 3 $y = -94.988x2 + 104.68x + 61.363 \quad R^2=0.5651$

Polynomial rather than linear relationships were selected for a better fit to the data, particularly at the low end of the range. The generated equations are valid for copper feed grades ranging between 0.60 and 0.19% Cu, although there is some deviation at each end of the range. The correlation coefficients are generally not high, ranging between 48 and 57%.

Stable and reproducible laboratory test results obtained in a highly-controlled environment are considered the "best-case scenario". In contrast, typical in-plant operations generate operational recovery losses, particularly in the first two years of operations. This loss will decrease over time as production stabilizes. Accordingly, a 1.5% downward adjustment to the copper recovery was implemented for the equation. This adjustment was assumed to apply for the duration of the LOM.

The basis for process design uses 25% Cu concentrate grade, and the corresponding copper recovery has been calculated based on this concentrate product grade. After the adjustment is incorporated, the change to Equation 1 for a 25% Cu concentrate grade product then becomes:

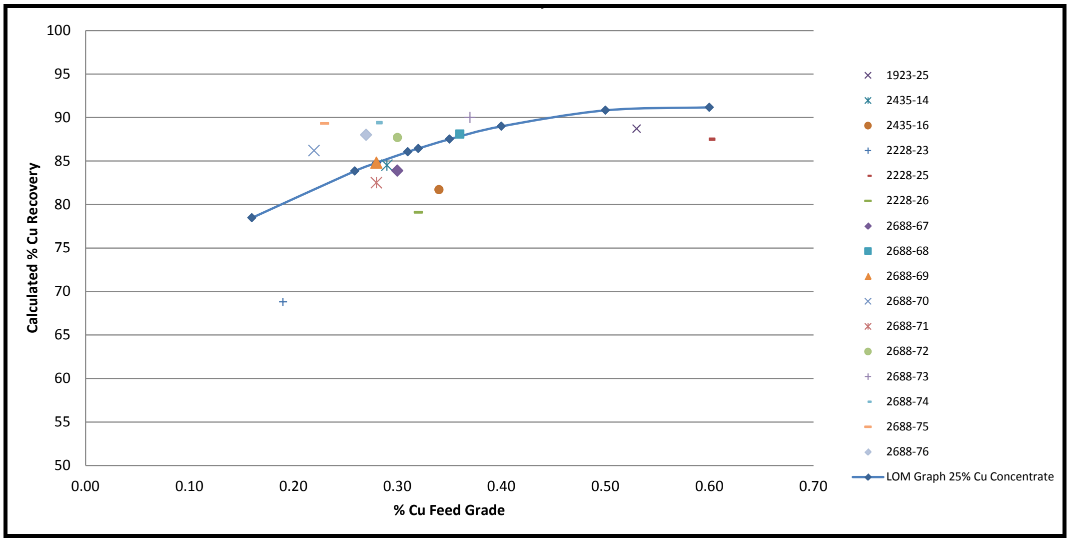
For 25% copper concentrate grade:

Equation 4 $$y = -74.812x2 + 85.727x + 66.668$$

Using Equation 4 and the LOM feed grade of 0.259% Cu, the calculated copper recovery for a 25% Cu concentrate grade is 83.9%.

Using Equation 4, a series of data points were generated and plotted to generate a copper feed grade versus recovery relationships (Figure 13.12). These results generated anticipated copper recovery and the actual copper recovery data obtained from the various locked cycle tests conducted to date. As indicated in Figure 13.12, the data varies considerably, primarily because the initial locked cycle tests were not conducted under optimum conditions, and because of sample make-up variability.

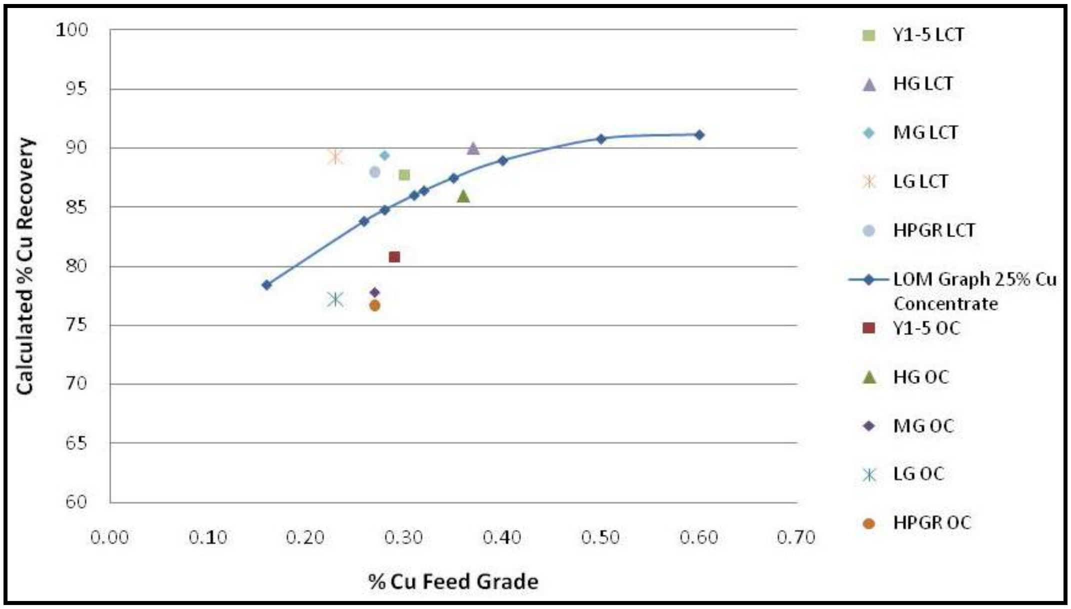
Figure 13.12 Comparison of Locked Cycle Test Data with Generated Grade/Recovery Relationship (Equation 4)



Open circuit tests were performed on the duplicate samples of material that were tested in the locked cycle tests. A comparison of the results show an approximate 2 to 10% increase in copper recovery of the locked cycle test results which was realized with the inclusion of the circulating load at the concentrate grade targeted. Better recovery improvements for the locked cycle tests were generally noted in the samples with a lower feed grade likely attributable to the need for the circulating load to be present in order to improve the final copper concentrate grade.

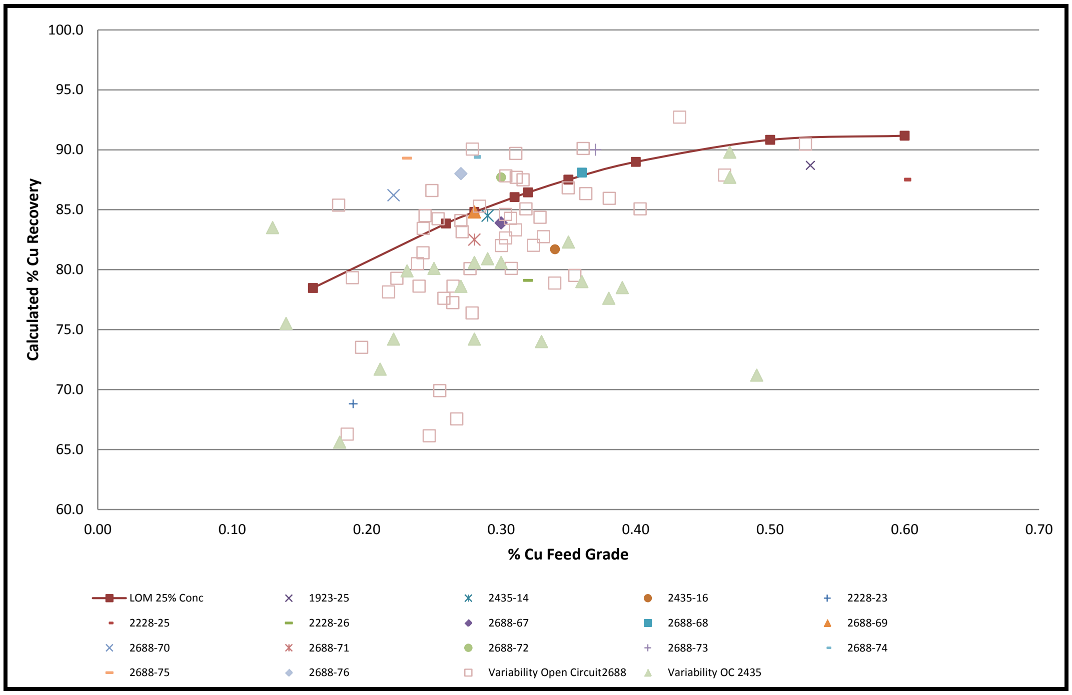
Figure 13.13 shows the comparison between the locked cycle test results and the open cycle test results for the same samples that were used in the development of Equation 4.

Figure 13.13 Comparison of Open Circuit Results with Locked Cycle Results



A comparison of the various test results (Figure 13.14) shows the differences between the relevant test data including the locked cycle tests, the open circuit variability tests, and the grade versus recovery relationship as developed from Equation 4.

Figure 13.14 Locked Cycle Tests, Variability Tests and Generated Grade/Recovery Relationship (Equation 4)



The grade versus recovery relationship therefore is seen to reasonably represent the anticipated behaviour of the copper, although variability will occur.

GOLD RECOVERY ANALYSIS

A comparison of the relationship of the gold-to-copper deportment was conducted to determine the gold recovery achievable into the copper concentrate,

Initially, an investigation into a relationship between the copper feed grade and the gold feed grade was considered for the locked cycle test results, as well as the variability data generated in the test programs. As discussed in Section 13.4, no statistically significant correlation exists between the gold and the copper feed grades.

To further develop the gold recovery analysis, the gold feed grades were compared with gold recovery for the open circuit variability tests; and an evaluation of the locked cycle test data followed. Copper recovery and copper grade were not taken into consideration in this evaluation.

The resulting open circuit variability tests indicated a poor correlation coefficient of only 18% (Figure 13.15).

Figure 13.15 Variability Test Results, Gold Recovery vs. Gold Feed Grade

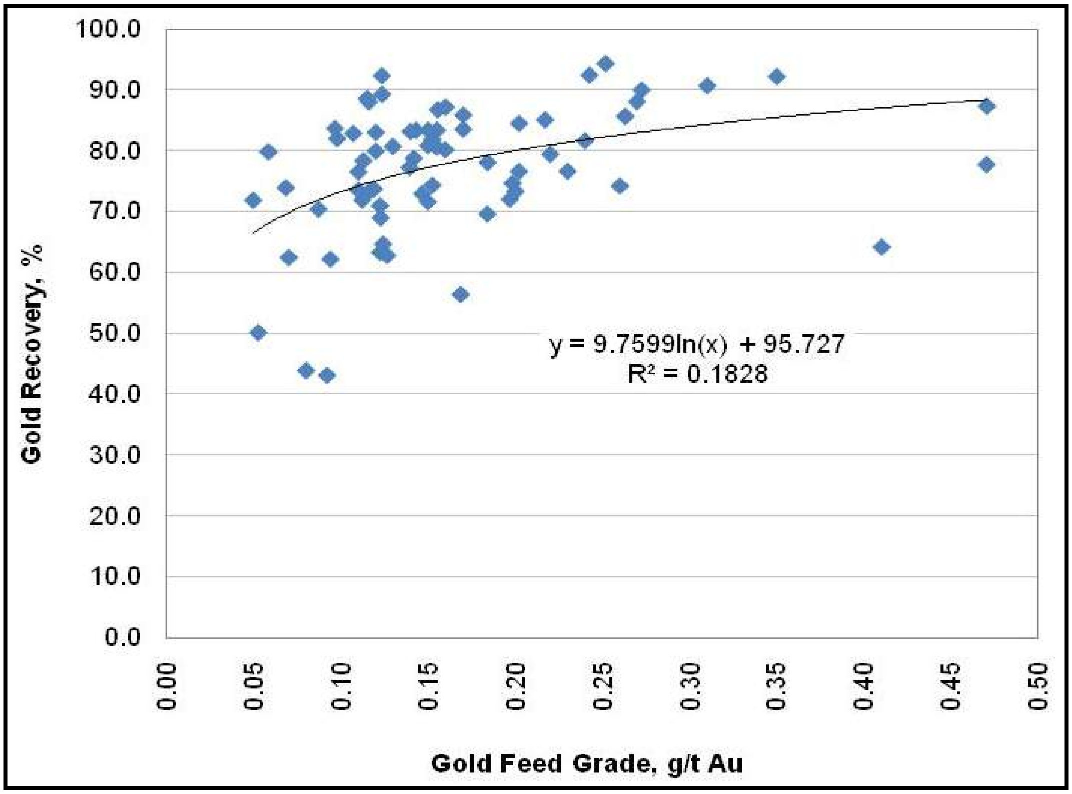
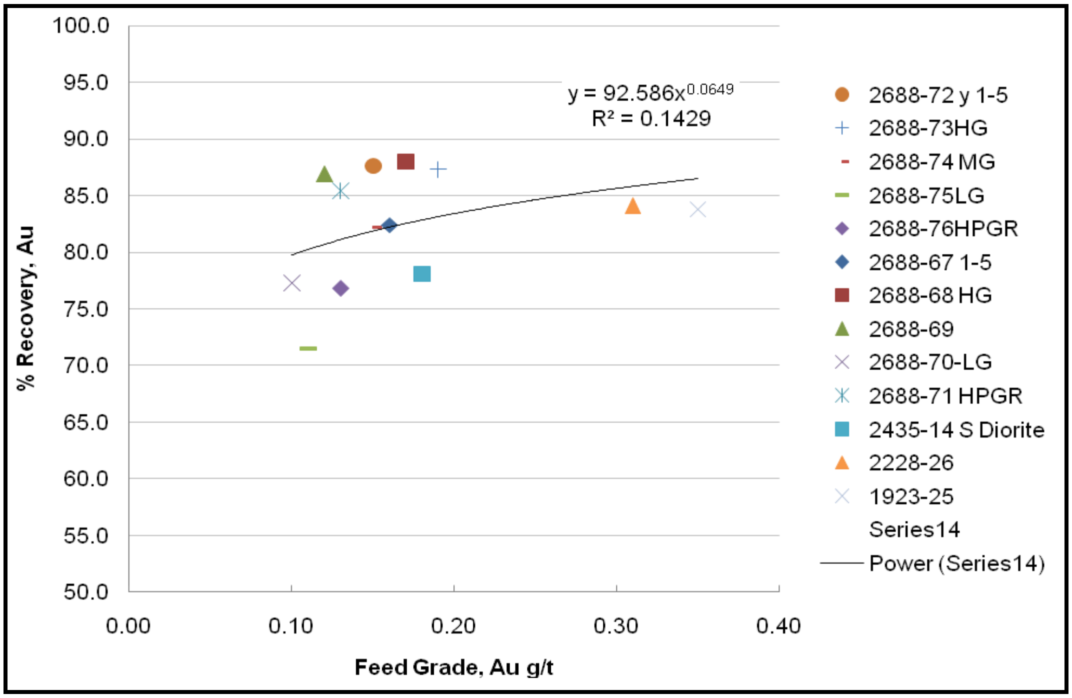


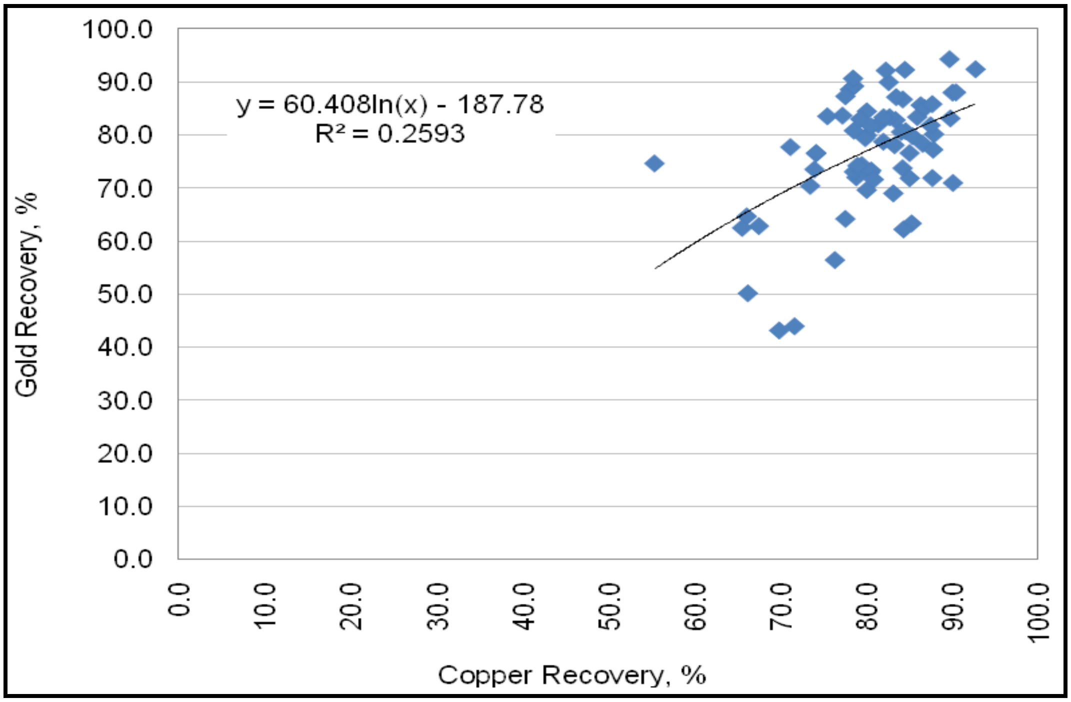
Figure 13.16 shows the relationship between the gold feed grade and the gold recovery to the final concentrate for specified locked cycle tests. The data plotted in Figure 13.16 indicates the best gold recovery obtained, and disregards the copper recovery and copper concentrate grade. Once again, a poor correlation coefficient of 14% is indicated.

Figure 13.16 Gold Recovery vs. Gold Feed Grade for Specific Locked Cycle Tests



Since the overall copper recovery obtained in the rougher circuit is the major contributor for the economics of the metallurgical process, it was necessary to determine if there is a correlation between copper recovery and gold recovery. A copper recovery versus gold recovery graph was created using the data generated in the variability testing as this was the largest and most comprehensive data set. The copper concentrate grade was not considered in this analysis. This also resulted in a poor correlation between the copper recovery and gold recovery as is shown in Figure 13.17.

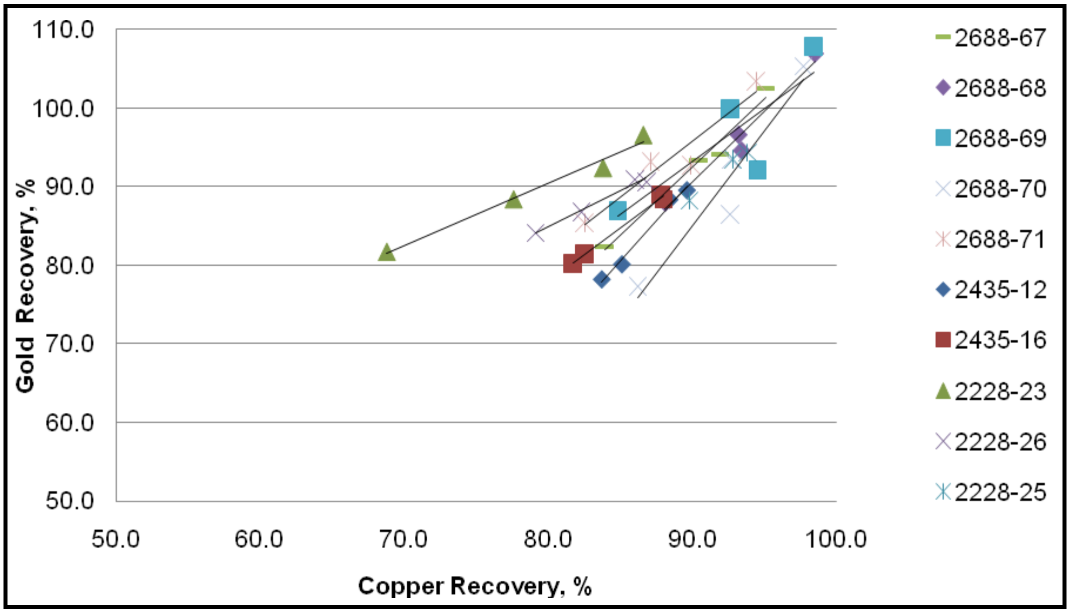
Figure 13.17 Copper Recovery vs. Gold Recovery Variability Tests: Test Programs KM2435 and KM2688



Narrowing the data to include only the locked cycle test data would also incorporate the recycle stream, which would more closely match anticipated full-scale production. Therefore, the locked cycle tests were subsequently used to correlate the gold and copper with the inclusion of the anticipated copper recovery.

Figure 13.18 shows the stage-wise recovery of gold and copper during flotation. Only specific locked cycle data was used because, in some instances, the calculated gold recovery data was not reliable.

Figure 13.18 Gold Recovery to the Final Concentrate vs. Copper Recovery to the Final Concentrate for Specific Locked Cycle Tests



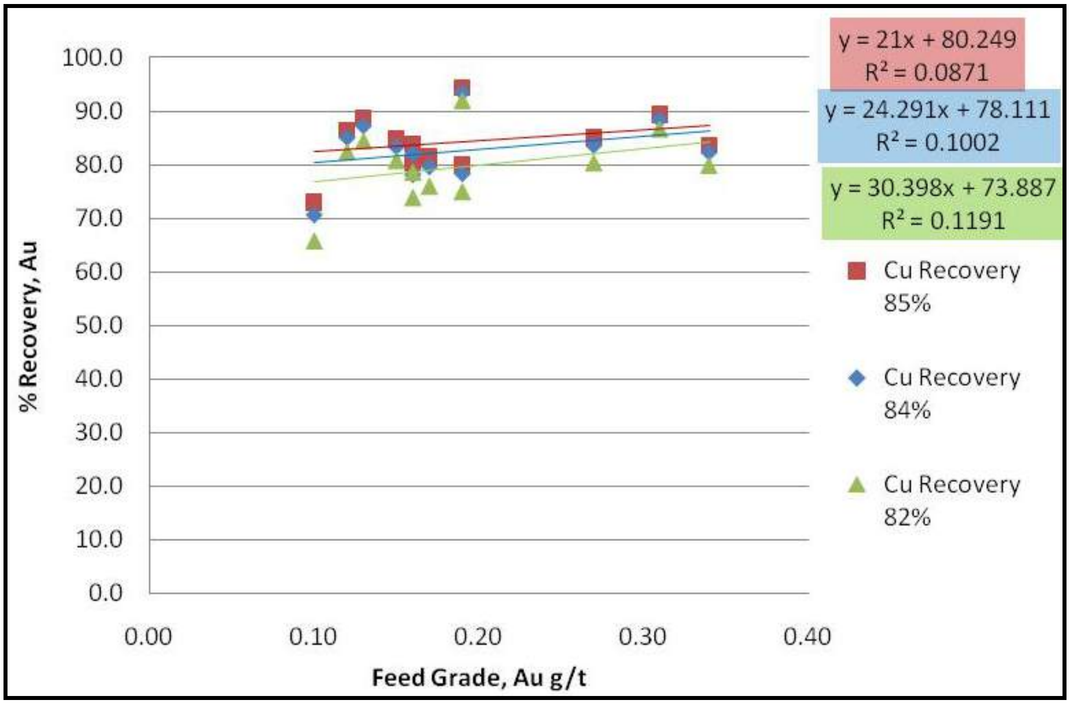
A gold recovery versus copper recovery relationship trend line and equation was developed for each of the locked cycle tests. For these trend line equations, a linear relationship was chosen within the relevant ranges of the data sets. In each case, the individual correlation coefficient values were found to be very good, although, as expected, the individual relationships vary greatly, depending mainly on the sample feed source (see Figure 13.18).

In order for a meaningful comparison of the results to be made, it was necessary to use the results as reported by G&T, and therefore the gold data has not been standardized to what would have been achieved at the 25% Cu concentrate grade level. The recovery at a specific copper recovery only was considered, but the copper feed grade and the concentrate grade of the copper was not considered.

The linear equations shown in Figure 13.18 were subsequently used to determine the gold recovery values for the respective copper recoveries of 85, 82, and 84% which are considered to be reasonable values based on the analysis of the results as developed.

The three sets of relationships obtained are shown in Figure 13.19.

Figure 13.19 Comparison of Gold Recovery to Gold Feed Grade at Varying Copper Recovery Values



By establishing that the copper recovery is 84% based on the recovery calculated using Equation 4, the anticipated gold recovery at a given feed grade was then calculated using the equation generated in Figure 13.19.

The anticipated LOM gold feed grade is 0.165 g/t Au and using the generated equation from Figure 13.19, Equation 5 is established to be:

Equation 5 $\quad y = 24.291x + 78.111$

In this case:

- x is the gold feed grade
- y = 82.1% Au recovery
- R^2 = 0.10.

During the analysis of this sequence of data for this analysis, the copper concentrate grade was not taken into consideration. In addition, the correlation coefficient was very poor, at 10% for this equation, and the results obtained are based on an extremely complex methodology.

Based on these considerations, it was deemed appropriate to re-assess the relationship found in Figure 13.16 in order to simplify the analysis.

The locked cycle test data presented in Figure 13.16 generated a relationship with the equation:

Equation 6 y = 92.586x0.0649

Where:

- x is the gold feed grade
- y is the gold recovery
- R^2 is the correlated coefficient of 14%.

Using the anticipated LOM gold feed grade of 0.165 g/t Au in Equation 6, the resulting calculated gold recovery is 82.4%.

Because of the overall poor correlation between the gold and copper recoveries, and the gold feed grade and recovery in all the previous equations, the equation recommended for use with approximately the same level of accuracy is Equation 6.

In the case of gold, a penalty will not be included because this penalty has already been applied to the copper recovery.

As a general observation, it is apparent that for the determination of the anticipated concentrate grade of gold, there appears to be a 100 times upgrade from the feed grade of gold to the grade of gold in the final concentrate in the production of a concentrate grade of 25% Cu.

13.7 Quality of Concentrates

The copper concentrate from the locked cycle test for the Year 1 to 5 composite sample was analyzed for deleterious minor elements. Cobalt and nickel levels were elevated, at 796 and 256 ppm, respectively. The balance of the other minor elements analyzed were below typical smelter penalty limits.

Mercury (Hg) assays were performed on each of the concentrates produced in the 50 variability tests from test program KM2688. Five of the test samples produced concentrates that contained slightly elevated mercury levels in excess of the bulk of the concentrations measured. The median mercury assay value for the samples tested was 2 ppm Hg.

As a recommendation, the minor element data should be further reviewed by a concentrate marketing specialist to confirm that there will be no marketing issues with the concentrate produced.

The complete assay results of the copper concentrate are included in Appendix B.

14.0 MINERAL RESOURCE ESTIMATES

The results of the Mineral Reserve estimate represent forward-looking information that are 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 information includes the Mineral Resource estimate, commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and risks regarding the cost estimates and completion schedule for the proposed Project infrastructure, in particular the need to obtain permits and governmental approvals.

14.1 KEY ASSUMPTIONS/BASIS OF ESTIMATE

AMEC modelled the areas indicated in Figure 14.1. For the 2011 model update, the Monte Carlo area has been renamed to the Ajax East Extension (Figure 14.1).

The 2011 update of the geological model for Ajax West, Ajax East and Ajax East Extension areas consist of triangulated solids for Sugarloaf, Iron Mask and Mafic Volcanic Rocks.

The Albite domain from previous models was not included in the 2011 resource model update. Abacus geological staff observed logging inconsistencies in the identification and logging of Albite between the numerous drill campaigns and a re-logging program has been initiated. The albite domain was utilized to control higher-grade gold mineralization. For the 2011 resource model update, a high-grade gold shell was implemented to address higher-grade gold.

14.2 GEOLOGICAL MODELS

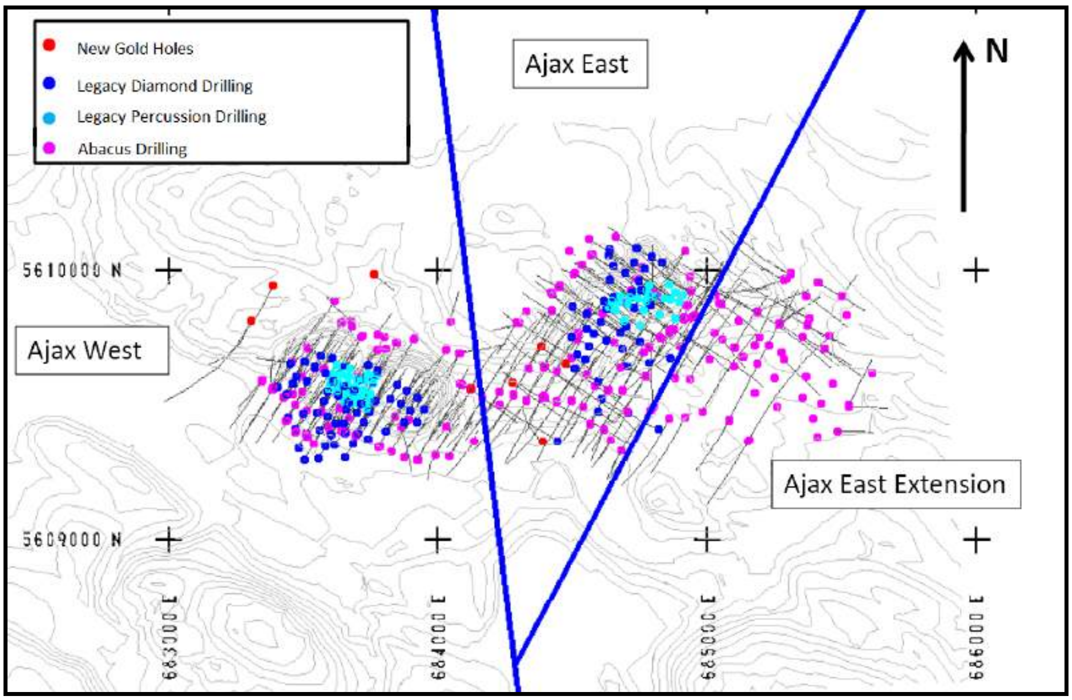
Abacus personnel provided a geological model of the Ajax area in the form of Surpac-triangulated solids and polygon strings in “DXF” format. AMEC imported the DXF files into MineSight® and validated the stability of the wireframe solids. MineSight® identified minor issues (duplicate triangles, crossing triangles and closure) with the wireframes. The MineSight® validation tool corrected the wireframes, and no changes were made to the geological model represented by the section polygons.

AMEC reviewed the geological model against drillhole data and concluded the geological interpretation was reasonable.

14.3 Mineralized Shell

Abacus provided a mineralized shell for the West, East and East Extension areas. The mineralized shell was constructed to identify a 0.10% Cu grade envelope. Generally the copper mineralization shell includes regions where copper mineralization exceeding 0.10% Cu can be expected, but also includes large drill intervals of low-grade mineralization (<0.10% Cu). In the West and East areas, the mineralization shell includes the Sugarloaf diorite and locally extends into the Iron Mask and Mafic Volcanic rock types. In the East Extension area, the mineralized shell is generally limited to the Sugarloaf Diorite.

Figure 14.1 Ajax Model Areas



It was determined in the early phases of the Project that there was a tendency to “smear” high-grade mineralization into low-grade mineralized areas and to dilute the high-grade with low-grade mineralization.

AMEC used the copper mineralization shell as an EDA boundary and also used the shell as the boundary for the copper and gold estimation.

AMEC applied a PACK methodology to further define the high-grade copper and gold domains within the mineralized shell provided by Abacus. This controls the smearing of high-grade mineralization into low-grade areas.

14.4 COMPOSITES

The Ajax assay data were composited to 12 m lengths using MineSight® procedures. Because mining is expected to be completed by bulk mining methods, no geologic boundaries were used in the compositing. The composite database contains 15,512 composites. The composite database contains 12,116 composites that are within the mineralized shell provided by Abacus.

The 12 m (length) composites were coded with lithology by back tagging from the 3DBM (field BKRT). Composites were also coded from the assay file (RCODE) and from the geological solids (LCODE). The three codes were inspected in cross section and plan section. For drillholes in areas that have been mined out or back filled, the BKRT field was coded from the geological solids. For regions where the geological solids could not code the 12 m composites, the assay table was used. Properly coding of the composites for lithology in the fill or mined-out area permitted these data to be used in the estimation process.

14.5 VARIOGRAPHY

14.5.1 COPPER INDICATOR VARIOGRAMS

The correlogram models used for the ordinary kriging (OK) interpolation of the copper PACK were determined using the autofit function in Sage2001®. The nugget effect for the variogram models were determined using down-hole variograms. Variograms were modelled for the Ajax West, Ajax East and Ajax East Extension areas. The general orientation of the copper indicator variogram model in the West Area is generally east-west, dipping steeply to the north. In the East Area, the variogram model is oriented to the northeast, dipping moderately to the west. In the East Extension, the copper indicator variogram model is generally oriented in a northerly direction and dips moderately to the west.

14.5.2 GOLD INDICATOR VARIOGRAMS

The general orientation of the gold indicator variogram model in the West Area is generally east–west with a near vertical dip. In the East Area, the gold indicator variogram model is oriented to the northeast, dipping steeply to the west. In the East Extension, the gold indicator variogram model is generally oriented in an east-west direction with a shallow dip to the north.

14.6 PACK Estimation/Interpolation Methods

14.6.1 *High-Grade Copper Domain*

Each 12 m composite was coded with an indicator using a 0.10% Cu threshold (< 0.10% Cu = 0 and ≥ 0.10% Cu = 1). Variograms (correlograms) were modelled on the high-grade indicators using Sage 2001®. The high-grade probability was then interpolated into each block using OK. A nearest neighbour (NN) model was also constructed to assist in the validation of the high-grade copper domain.

The high-grade probability was estimated using OK. The estimation was completed in one pass using the variogram criteria summarized in Table 14.1. Sample selection (search) criteria are summarized in Table 14.2.

A NN model of the copper indicator (0 or 1) was constructed to validate the volume of the high-grade copper domain. The volume of high-grade copper blocks in the NN model was compared to the volume of high-grade copper blocks at various probability thresholds. The NN model identified 48.2% of the Ajax area to be high-grade.

AMEC chose the probability threshold of 0.48 to define the high-grade copper shell which resulted in 48.3% of the estimated blocks. The probability values were visually inspected in section and plan.

The final high-grade copper domain was coded to the block model field HGCu (HGCu = 1 = high-grade; HGCu = 0 = low-grade).

14.6.2 *High-Grade Gold Domain*

In previous models, the high-grade copper domain, in association with the albite alteration domain were used to define the high-grade gold domain. The albite alteration domain was abandoned in the 2011 resource model update because Abacus geological staff determined the albite geological logging was inconsistent between drill campaigns from 2005 to 2010. For the 2011 resource model update, AMEC applied the PACK methodology to identify the high-grade gold domain.

Each 12 m composite was coded with an indicator using a 0.10 g/t Au threshold (< 0.10 g/t Au = 0 and ≥ 0.10 g/t Au = 1). Variograms were modelled on the high-grade indicators using Sage 2001. The high-grade gold probability was then interpolated into each block using OK. A NN model was also constructed to assist in the validation of the high-grade gold domain.

Table 14.1 Summary of Copper Indicator Variograms

	Area	West	East	East Extension
Nugget	C0	0.484	0.450	0.520
First Structure	C1	0.398	0.352	0.366
Range	X	87.6	20.3	71.1
	Y	79.2	39.8	107.1
	Z	55.0	36.2	50.3
Rotation About	Z – LH	51	-23	-73
	X – RH	-25	-92	116
	Y – LH	41	-2	67
Second Structure	C2	0.118	0.198	0.114
Range	X	373.3	178.0	900.0
	Y	684.5	90.9	310.9
	Z	107.5	460.5	600.0
Rotation About	Z – LH	-5	-45	-87
	X – RH	27	35	45
	Y – LH	-8	-21	155

Note: LH = Left-hand rule; RH = Right-hand rule.

Table 14.2 Search Strategy for PACK Interpolation

Area	Search (x,y,z)	Samples (max - min - max per DH)
Copper	300,300,300	3 - 6 - 2
Gold	300,300,300	3 - 6 - 2

The gold high-grade probability was estimated using OK. The estimation was completed in one pass using the variogram criteria summarized in Table 14.3. Search sample selection criteria were as summarized in Table 14.2.

A NN model of the high-grade gold indicator (0 or 1) was constructed to validate the volume of the high-grade copper domain.

The volume of high-grade gold blocks in the NN model (33.3%) was compared to the volume of high-grade gold blocks a various probability thresholds. The NN model identified 33.3% of the Ajax area to be high-grade gold. AMEC chose the probability threshold of 0.47 to define the high-grade gold shell for a volume of 33.2%. The probability values were visually inspected in section and plan.

The final high-grade gold domain was coded to the block model field HGAu (HGAu = 1 = high-grade; HGAu = 0 = low-grade).

Table 14.3 Summary of Gold Indicator Variograms

	Area	West	East	East Extension
Nugget	C0	0.450	0.527	0.421
First Structure	C1	0.263	0.299	0.391
Range	X	10.9	29.0	121.2
	Y	79.1	97.5	25.5
	Z	22.7	68.8	58.2
Rotation About	Z – LH	40	-29	18
	X – RH	-91	-73	58
	Y – LH	3	-59	-18
Second Structure	C2	0.287	0.174	0.188
Range	X	358.4	334.0	373.3
	Y	117.4	242.0	378.1
	Z	231.4	375.1	600.0
Rotation About	Z – LH	-15	-20	-9
	X – RH	27	-1	15
	Y – LH	29	-21	-29

14.7 Exploratory Data Analysis

EDA was completed using 12 m composites for the Ajax West and Ajax East and Ajax East Extension areas. Composites were identified as high-grade or low-grade by back-tagging from the 3D block model. The high-grade domain composites were inspected in cross-section and plan-section. EDA included pivot tables, histograms and box plots.

The probability plots generally display a slight curvilinear profile suggesting multiple populations; however the probability plots overall suggest a lognormal population.

Box plots of composites suggest the rock types can be combined for the estimation based on similar summary statistics. However, contact profiles suggest hard contacts should be used. Hard boundaries were used in the estimations.

14.8 Density Assignment

No additional SG modifications were made to the density model for the 2011 resource model update. The Ajax density database contains 855 SG determinations (Table 14.4). The SG data were coded with the rock code (RCODE) from the MineSight® composite file, and histograms were then constructed.

Table 14.4 Specific Gravity Determinations

Area	Rock Type	Code	No. of Determinations	Mean (SG)	Standard Deviation
Ajax West	Mafic Volcanic	3	91	2.875	0.009
	Sugarloaf	4	124	2.787	0.010
	Iron Mask	6	21	3.076	0.060
Ajax East & East Extension	Mafic Volcanic	3	47	2.823	0.012
	Sugarloaf	4	413	2.759	0.010
	Iron Mask	6	66	2.991	0.040
All	Fill[1]	10	none	2.000	2.000
	Overburden[2]	250	none	2.200	2.200

Note: 1. Fill determined using a 30% swell factor for all density values.
2. Overburden (gravels) from Jackson, K.C., 1970, Textbook of Lithology, McGraw-Hill, Inc.

14.9 MODEL CODING

The geological model was coded from the geological solids. The 3D block model was coded with the copper mineralization shell solid (CUSHL = 1) to identify blocks that would be estimated. Blocks within the mineralized shell (CUSHL=1) were coded as high-grade or low-grade domains.

14.10 INTERPOLATION

The estimation methodology for the 2011 resource model update was IDW^4. All estimations were completed by rock type. All rock boundaries were considered hard for estimation purposes. An outlier restriction was implemented on uncapped 12 m composites to address metal-at-risk.

The model estimate was completed in three passes with expanding searches for each pass. Table 14.5 summarizes the estimation plan for copper. Table 14.6 summarizes the estimation plan for gold.

14.11 GRADE CAPPING/OUTLIER RESTRICTIONS

AMEC addresses metal-at-risk using the following outlier restriction:

- In the high-grade copper zone, the outlier restriction for copper used a grade threshold of 1.25% Cu and a distance threshold of 15 m. Composite values that exceed 1.25% Cu were capped at 1.25% Cu beyond the distance threshold of 15 m.

Table 14.5 Copper Estimation Plan

	Variables	West LG	West HG	East LG	East HG	East Ext LG	East Ext HG
Pass 1	RX (m)	75	45	75	45	75	45
	RY (m)	75	45	75	45	75	45
	RZ (m)	75	45	75	45	75	45
	1st Rot (deg)	90	90	45	45	90	90
	2nd Rot (deg)	0	0	0	0	0	0
	3rd Rot (deg)	0	0	0	0	0	0
	Min of Comps	4	4	4	4	4	4
	Max of Comps	8	8	8	8	8	8
	# of DDHs	3	3	3	3	3	3
Pass 2	RX (m)	150	100	150	100	150	100
	RY (m)	150	100	150	100	150	100
	RZ (m)	150	100	150	100	150	100
	1st Rot (deg)	90	90	45	45	90	90
	2nd Rot (deg)	0	0	0	0	0	0
	3rd Rot (deg)	0	0	0	0	0	0
	Min of Comps	4	4	4	4	4	4
	Max of Comps	8	8	8	8	8	8
	# of DDHs	3	3	3	3	3	3
Pass 3	RX (m)	300	300	300	300	300	300
	RY (m)	300	300	300	300	300	300
	RZ (m)	300	300	300	300	300	300
	1st Rot (deg)	90	90	45	45	90	90
	2nd Rot (deg)	0	0	0	0	0	0
	3rd Rot (deg)	0	0	0	0	0	0
	Min of Comps	4	4	4	4	4	4
	Max of Comps	8	8	8	8	8	8
	# of DDHs	2	2	2	2	2	2
	IDW Power	4	4	4	4	4	4
Outliers	Cut-off	15	30	15	30	15	30
	Distance	0.1	1.5	0.1	1.5	0.1	1.5

Table 14.6 Gold Estimation Plan

	Variables	West LG	West HG	East LG	East HG	East Extension LG	East Extension HG
Pass 1	RX (m)	75	45	75	45	75	45
	RY (m)	75	45	75	45	75	45
	RZ (m)	75	45	75	45	75	45
	1st Rot (deg)	90	90	45	45	45	45
	2nd Rot (deg)	0	0	0	0	0	0
	3rd Rot (deg)	0	0	0	0	0	0
	Min of Comps	4	4	4	4	4	4
	Max of Comps	8	8	8	8	8	8
	# of DDHs	3	3	3	3	3	3
Pass 2	RX (m)	150	100	150	100	150	100
	RY (m)	150	100	150	100	150	100
	RZ (m)	150	100	150	100	150	100
	1st Rot (deg)	90	90	45	45	45	45
	2nd Rot (deg)	0	0	0	0	0	0
	3rd Rot (deg)	0	0	0	0	0	0
	Min of Comps	4	4	4	4	4	4
	Max of Comps	8	8	8	8	8	8
	# of DDHs	3	3	3	3	3	3
Pass 3	RX (m)	300	300	300	300	300	300
	RY (m)	300	300	300	300	300	300
	RZ (m)	300	300	300	300	300	300
	1st Rot (deg)	90	90	45	45	45	45
	2nd Rot (deg)	0	0	0	0	0	0
	3rd Rot (deg)	0	0	0	0	0	0
	Min of Comps	4	4	4	4	4	4
	Max of Comps	8	8	8	8	8	8
	# of DDHs	2	2	2	2	2	2
	IDW Power	4	4	4	4	4	4
Outliers	Cut-off	15	30	15	30	15	30
	Distance	0.1	1.5	0.1	1.5	0.1	1.5

- In the low-grade copper zone, the outlier restriction for copper used a grade threshold of 0.10% Cu and a distance threshold of 15 m. Composite values that exceeding 0.10% Cu were capped at 0.10% Cu beyond the distance threshold of 15 m.
- In the high-grade gold zone, the outlier restriction for gold used a grade threshold of 1.5 g/t Au and a distance threshold of 30 m. Composite values that exceed 1.5 g/t Au were capped at 1.5 g/t Au beyond the distance threshold of 30 m.

- In the low-grade gold zone, the outlier restriction for copper used a grade threshold of 0.10 g/t Au and a distance threshold of 30 m. Composite values that exceed 0.10 g/t Au were capped at 0.10 g/t Au beyond the distance threshold of 30 m.

The results indicate that 5.3% of copper metal was removed by the outlier restriction and 4.5% of the gold metal was removed (Table 14.7).

14.12 Block Model Validation

Model validation consisted of visual inspection of cross-sections and plan-sections. Box plots, contact profiles, swath plots, and Herco analysis were completed by area for high-grade blocks by rock type. A NN model was also completed for model validation. The NN model utilized the same search criteria as the IDW^4 estimate and was used for comparison of summary statistics in box plots, Herco and swath plots. Model validation was completed using blocks classified as Measured and Indicated.

Results of the validation included:

- AMEC checked the 2011 resource block model for global bias by comparing the average metal grades (at 0.00 cut-off) from the model (ID^4 grades) with means from NN estimates for Measured and Indicated blocks. Generally, all rock domains demonstrate minimal biases. For gold, the Iron Mask rock domain is a negative bias of 4%. This is within the acceptable tolerance and, there is also a small number of model blocks (4,900) compared to the Sugarloaf and Mafic domains.
- Inspections of cross sections and plan sections indicated comparable copper and gold composite grades and block grades. AMEC observed that the general grade continuity is less in the East Extension area for both copper and gold south of 5,609,700 N. This may reflect the mineralization being more confined to structural zones. AMEC recommends additional drilling in this area to increase the confidence of the Mineral Resource.

Table 14.7 Metal-at-Risk Adjustments

Area	No of Blocks	Uncapped Mean	Capped[1] Mean	Metal Removed
Copper	151,370	0.1871	0.1771	5.3%
Gold	151,370	0.1205	0.1151	4.5%

Note: As implemented outlier restriction was used beyond a specified radius of influence.

- It was observed in the validation process that mean grades of the IDW^4 estimate can exceed the NN estimate in some rock types for low-grade material. However, as the grades are well below a current economic cut off, it was determined that time spent improving the low-grade estimation was not currently justified.

- Herco analysis was completed for high-grade copper and high-grade gold in the West, East and East Extension areas using blocks with a resource classification of Measured and Indicated. The Herco plots for copper compare closely in the range of the proposed cut-off grade of 0.20% Cu. For gold, the Herco plots compare closely for the Ajax West and East Extension areas. The Ajax East area is slightly under smoothed.
- Swath plots were constructed by rock type using 12 m composites, IDW^4 estimated blocks, and the NN estimated blocks. Swaths were completed using blocks that have a resource classification of Measured and Indicated. Swath plots compare the grade estimates in swaths across the model. Swath intervals were 100 m in the easterly and northerly directions and 75 m in the vertical direction. Swaths show good agreement with the exception of areas where data are sparse.

14.13 Classification of Mineral Resources

A classification of the Ajax model was developed based on the copper mineralization shell, grade continuity observed in the cross-section and plan-section, and confidence limit calculations using a 60,000 t/d production schedule.

To aid in the classification, the percentage of influence legacy drillholes had on the copper grade estimate was determined. The influence for the J-series drillholes (percussion drilling) was determined independently from other legacy drillholes: 87, 88, 89, 90 series drillholes (core drilling). Adjustments were made to the Step 1 classification based on Step 2 parameters.

Classification was limited to blocks within the copper mineralization shell using the following criteria:

Step 1

1. Measured Blocks – Ajax West and East – three drillholes within 55 m with one drillhole within 39 m. Ajax East Extension, the classification for measured required three drillholes within 35 m.
2. Indicated Blocks – two drillholes within 75 m with one drillhole within 58 m.
3. Inferred Blocks – remaining blocks within copper shell that were not classified as Measured or Indicated.

Step 2

1. Visual inspection of the resource classification in Step 1. In the East Extension, south of the approximately 5609700 N, the mineralization demonstrates less continuity on cross section and plan. This region was inspected in detail using cross-section and elevation plans. Polygons were

constructed on bench levels, and the resource classification was down-graded to Inferred within the polygons.

Step 3 – Adjustment for Legacy Drillholes

1. Where J series percussion holes contribute more than 60% of the weight used to make a copper grade estimate, the block is down-graded to Inferred.
2. Blocks where Legacy Core Holes contribute more than 50% of the weight used to make a copper grade estimate, Measured blocks are down-graded to Indicated.

The nominal drill spacing for Measured is 40 to 50 m for the Ajax West and East areas. Confidence limit calculations for the West and East areas suggest drill density for Measured Resource Classification of 35 to 50 m. The confidence limit calculation for the Indicated Resource Classification in the West and East areas suggest a drill density of approximately 50 to 75 m.

In the East Extension, drill spacing ranges up to 120 m. Confidence limit calculations for Measured Resource Classification suggest a drill density of 25 to 40 m. Confidence limit calculations for Indicated Resource Classification in the East Extension suggest drill density of approximately 40 to 75 m. The observed continuity of mineralization in sections and plans in portions of the East Extension suggest that closer drill spacing is required south of coordinate 5609700.

Using a 0.100 %Cu cut-off, 38% of the blocks are classified as Measured, 41% are classified as Indicated, and 21% are classified as Inferred.

14.14 Reasonable Prospects of Economic Extraction

An NSR was calculated for each block using an NSR script in Gemcom Whittle™ software. The script was checked by Ramon Mendoza Reyes, AMEC's Principal Mine Engineer. The script calculated an NSR value for each block based on the criteria listed in Table 14.8.

To determine the reasonable expectations for economic extraction, a LG pit optimization was completed using blocks classified as Measured, Indicated and Inferred. The LG input parameters were checked by Ramon Mendoza Reyes, AMEC's Principal Mine Engineer.

The LG was completed by Abacus personnel using Gemcom Whittle™ 4D (Version 4.1.3). The LG parameters are shown in Table 14.9.

Based on preliminary economic analyses, the net value of the resource shell exceeds capital cost estimates.

14.15 COPPER EQUIVALENCY

A CuEq grade for reporting of Mineral Resources was calculated using the following formula:

$$CuEq = \frac{[(\%Cu)\ (CuRec)\ (22.0462)\ (\$lbCu) + (g/t/Au)\ (AuRec)\ (1/31.1035)\ (\$ozAu)]}{(CuRec)\ (22.0462)\ (\$lbCu)}$$

Where:

(Curec) (22.0462) ($lbCu) = in situ dollars / % of Cu

(Aurec/31.1035) ($ozAu) = in situ dollars / g of Au

%Cu = modelled copper grade

g/tAu = model gold grade

$%Cu = copper price

$oxAu = gold price

When:

F = Factor to give copper percent equivalent to 1 g/t Au

F = (Aurec/31.1035) (price/oz)/ [Curec(22.0462)(price/lb)]

And:

CuEQ = Cu + F(Au)

Table 14.8 NSR Parameters (Source: Abacus)

		Units[1]	Value
Recovery[2]	Copper	%	CuRec = (-74.812 x (Cu%^2))+(85.727xCu%) +66.668
	Gold	g/t	AuRec = 92.586 x Au(g/t)^0.0649
Concentrate	Moisture	%	8.5
	Grade	%	25
Freight	Land	$/wmt	27.90
	Port	$/wmt	27.00
	Ocean	$/wmt	50.00
	Misc	$/wmt	3.97
Smelter Terms	Participation	$/lb	None
	PP	%	None
	Conc Treat	$/dmt	46.50
	Conc Pay	%	97.00
	Conc Deduct	%	1.0
	Cu Refine	$/lb	0.0465
	Au Refine	$/tr oz	5.00
	Au Pay	%	97.0
	Au Deduct	g/t	1.0

Note: 1. Currency in US dollars.
2. Maximum copper recovery is 91.17%. Maximum gold recovery is 86.49%.

Table 14.9 Lerchs-Grossman Optimization Parameters (Source; Abacus)

	Item	Units[1]	Value
Mill Throughput	Tonnes per day	t/d	60,000
	Million tonnes per year	Mt/a	21.9
Mining Costs	Waste	$/t	1.08
	Fill Waste	$/t	0.89
	Processed Material	$/t	1.08
Process	Processing	$/t	3.18
	Reclamation	$/t	0.05
	Total	$/t	3.23
G&A	All	$/t	0.52
Price	Gold	$/troy oz	1200
	Copper	$/lb	2.88
Slopes	Overall	degrees	38 to 49[2]

Note: 1. Currency in US dollars.
2. Varies by geotechnical sector.

14.16 MINERAL RESOURCE STATEMENT

Mineral Resources take into account geologic, mining, processing and economic constraints, and have been confined within appropriate LG pitshells, and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The QP for the Mineral Resource estimate is Timothy O. Kuhl, SME Registered Member and an employee of AMEC.

Mineral Resources are reported using a copper price of US $2.88/lb and a gold price of US$1,200/oz, and have an effective date of May 26, 2011.

Mineral Resources are summarized in Table 14.10 and reported at a Base Case CuEq grade of 0.20%. Table 14.11 presents sensitivities of the Mineral Resource estimate to changes in the CuEq grade.

14.17 COMPARISON TO PREVIOUS ESTIMATE

The 2011 LG resource shell was compared to LG resource shells from 2010 and 2009.

The resource shells are determined with different input parameters (cost, prices and recovery functions), but general comments can be made. As expected, the resource tonnages have increased with the contribution of additional drilling and the use of higher commodity prices.

To determine the effects of the drilling, recovery functions, costs and commodity prices, AMEC recommends a "Due-To" table be constructed identify the variables that have most affected the Ajax Mineral Resource estimate.

The 2011 resource model update showed an 11% increase in Measured + Indicated tonnes over the 2010 resource model. This is a reflection of the additional infill drilling. Copper and gold grades are similar between the 2011 and 2010 models.

Table 14.10 Ajax Mineral Resource Estimate, Effective Date May 26, 2011; Timothy O. Kuhl, R.M. SME

	Cut-off CuEq (%)	Tonnes (Mt)	CuEq (%)	Cu (%)	Au (g/t)	NSR (US$/t)	Contained Metal		
							CuEq (Mlb)	Cu (Mlb)	Au (Koz)
Measured	0.20	255.8	0.42	0.31	0.19	15.71	2,389	1,734	1,555
Indicated	0.20	256.2	0.42	0.30	0.20	19.98	2,399	1,712	1,637
Measured + Indicated	0.20	512.0	0.42	0.31	0.19	17.85	4,788	3,446	3,193
Inferred	0.20	73.7	0.38	0.27	0.17	17.46	613	439	406

Table 14.11 2011 Resource Table at Various CuEq Cutoffs (Base Case is Shaded). Effective Date May 26, 2011; Timothy O. Kuhl, R.M. SME

	Cut-off CuEq (%)	Tonnes (Mt)	CuEq (%)	Cu (%)	Au (g/t)	NSR (US$/t)	Contained Metal		
							CuEq (lb)	Cu (lb)	Au (oz)
Measured	0.10	322.5	0.38	0.27	0.17	13.83	2,667,000	1,933,000	1,734,600
	0.20	**255.8**	**0.42**	**0.31**	**0.19**	**15.71**	**2,389,000**	**1,734,000**	**1,555,400**
	0.30	179.4	0.50	0.36	0.23	18.84	1,982,000	1,437,000	1,306,200
	0.40	114.2	0.59	0.43	0.27	22.53	1,485,000	1,073,000	996,600
	0.50	68.5	0.68	0.49	0.32	26.53	1,032,000	743,000	705,200
Indicated	0.10	336.2	0.36	0.26	0.17	16.70	2,665,000	1,897,000	1,818,100
	0.20	**256.2**	**0.42**	**0.30**	**0.20**	**19.98**	**2,399,000**	**1,712,000**	**1,637,400**
	0.30	173.3	0.51	0.36	0.25	24.36	1,946,000	1,375,000	1,369,300
	0.40	110.0	0.60	0.42	0.30	29.26	1,463,000	1,024,000	1,052,900
	0.50	68.5	0.70	0.49	0.35	34.14	1,055,000	735,000	764,500
Measured + Indicated	0.10	658.7	0.37	0.26	0.17	15.30	5,331,000	3,830,000	3,552,600
	0.20	**512.0**	**0.42**	**0.31**	**0.19**	**17.85**	**4,788,000**	**3,446,000**	**3,192,800**
	0.30	352.6	0.51	0.36	0.24	21.55	3,928,000	2,811,000	2,675,500
	0.40	224.2	0.60	0.42	0.28	25.83	2,948,000	2,098,000	2,049,500
	0.50	137.0	0.69	0.49	0.33	30.34	2,087,000	1,478,000	1,469,800

table continues…

	Cut-off CuEq (%)	Tonnes (Mt)	CuEq (%)	Cu (%)	Au (g/t)	NSR (US$/t)	Contained Metal		
							CuEq (lb)	Cu (lb)	Au (oz)
Inferred	0.10	115.7	0.30	0.21	0.13	13.39	753,000	538,000	499,200
	0.20	**73.7**	**0.38**	**0.27**	**0.17**	**17.46**	**613,000**	**439,000**	**405,700**
	0.30	39.6	0.49	0.35	0.23	23.31	429,000	306,000	291,600
	0.40	20.2	0.63	0.45	0.30	30.65	281,000	201,000	192,000
	0.50	12.2	0.76	0.53	0.36	37.02	203,000	144,000	141,900

Note 1. Mineral Resources are contained within a conceptual Measured, Indicated and Inferred optimized pitshell using the following assumptions: maximum copper recovery of 91.17% and maximum gold recovery of 86.49% based on the following equations: CuRec = (-74.812 x (Cu%^2))+(85.727xCu%) +66.668 and AuRec = 92.586 x Au(g/t)^0.064; assumed throughput rate of 60,000 t/d; Whittle constraining shell slopes between pit slope angles ranging from 38º to 49º, waste and processed material mining costs of US$1.08/t, fill waste mining costs of US$0.89/t, total processing costs including reclamation of US$3.23/t, general and administrative costs of US$0.52/t, gold price of US$1,200/oz, and copper price of US$2.88/lb.

Note 2. Copper equivalency was calculated using the formula CuEq = ([(%Cu) x (CuRec) x (22.0462) x ($lbCu) + (g/t/Au) x (AuRec) x (1/31.1035) x ($ozAu)]) ÷ ((CuRec) x (22.0462) x ($lbCu))).

Note 3. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

Note 4. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces, contained copper pounds as Imperial pounds.

14.18 Factors That May Affect the Mineral Resource Estimate

Factors which may affect the geological models or the conceptual pitshells used to constrain the mineral resources, and therefore the Mineral Resource estimates include:

- Commodity price assumptions.
- The NSR value used to constrain the Mineral Resources is based on technical and economic parameters supplied by Abacus. Should these assumptions change, then the pit constraining the Mineral Resources will also change.
- Metallurgical recovery assumptions.
- Pit slope angles used to constrain the estimates.
- SG values assumed for the rock types.

14.19 Comments on Section 14

The AMEC QPs are of the opinion that the Mineral Resources for the Project, which have been estimated using core drill data, have been performed to industry best practices, and conform to the requirements of CIM (2010).

Please refer to Appendix C for AMEC's full report.

15.0 MINERAL RESERVE ESTIMATES

15.1 KEY ASSUMPTIONS/BASIS OF ESTIMATE

15.1.1 CONSIDERATION OF POTENTIAL CONSTRAINTS ON PIT LIMITS

Pitshell generation was based on a Base Case assumption that the Trans Mountain Pipeline, an oil pipeline operated by Kinder Morgan, which runs along the western limit of the proposed Ajax pit could be relocated away from the pit during the first years of operation. This Base Case was referred to as Case B.

A Case A was also assessed, which considered that the pipeline would not be removed, and would remain in the current position.

Although Case B was selected for mine design purposes, the pit optimization process and the operational designs was set-up in such a way that no excavation has been designed or scheduled on the ground below the pipeline location and its right of way.

The pitshell generation was also constrained by the presence of Jacko Lake.

The pit is unconstrained by considerations for infrastructure to the north, east and south. All the major infrastructure facilities planned for the Project including mineral processing facilities, stockpiles, waste dumps, offices, maintenance shops, fuel storage, tailings pond, water storage ponds, will be external to the current ultimate pit design and its area of influence.

The current natural topographic surface at the Ajax site, based on a June 26, 2006 survey by Eagle Mapping Limited, was used.

15.2 CONVERSION FACTORS FROM MINERAL RESOURCES TO MINERAL RESERVES

15.2.1 CONSIDERATION OF SELECTIVE MINING UNIT

The following two NN block models were generated to study the impacts of block size and height:

1. 15 x 15 x 9 m
2. 20 x 20 x 15 m

These two block models were compared with grade-tonnage curves for copper (%) and gold (g/t). At a 0.26% Cu cut-off, a 2% loss of copper metal could be lost if 15 m benches were used compared to 9 m benches. The loss was more significant for gold, where at a 0.17 g/t Au cut-off, nearly 8% of metal could be lost when using 15 m benches. Abacus staff decided that in light of increasing gold prices, nuggety gold values could be better controlled with shallower 12 m benches.

Near the conclusion of the Phase One Resource Model (2008), planned process throughput was increased from 40,000 to 60,000 t/d. In order to achieve this mining rate, the block size was increased to 15 x 15 x 12 m in recognition of larger mining equipment needed. This was a compromise between potential metal loss and mining rate. All subsequent resource model revisions (2009, 2010 and 2011) used a 12 m block and bench height.

15.2.2 *Block Model*

The resource model was adjusted adding several new items to convert it into a mining model. Table 15.1 shows the list of the added items.

In order to verify the generation of the pitshells that define the final pit limit, as well as the mining phases of the deposit, the block model was coded using the defined slope zones, inter-ramp slopes angles, and required berm widths. These codes were added for Cases A and B, not moving and moving the pipeline respectively. Finally, the total economic value per block was calculated for use in the application of the LG algorithm in the pitshell generation process.

15.2.3 *Net Smelter Return Calculations for Marginal Cut-off Application*

The total NSR calculation involved the calculation of the NSR attributable to copper and the NSR attributable to gold. The parameters used to calculate the total NSR per tonne of ore are listed in Table 15.2.

The first step in calculating the NSR attributable to copper was to obtain the process recovery for copper in order to calculate the tonnes of dry concentrate produced. Then the payable pounds of copper were calculated using the dry concentrate tonnes, the payable fraction of copper and the copper concentrate grade. The next step would have been to adjust the copper price according to the price participation but in this case the price participation was set to zero. Finally the NSR attributable to copper was calculated for the Measured and Indicated Mineral Resource categories by subtracting the treatment and refining costs from the income produced by the saleable copper.

In the case of the NSR attributable to gold, after calculating the process recovery, the gold grade in concentrate was calculated based on the recovery, ore grade and the tonnes of copper concentrate. Then the payable fraction of gold was calculated in order to calculate the payable ounces of gold. The NSR attributable gold was finally calculated by subtracting the refining costs from the income produced by saleable gold.

The total NSR was calculated by adding the NSR attributable to copper to the NSR attributable to gold and then subtracting the freight costs, which include land freight, port charges, ocean freight and miscellaneous costs.

Table 15.1 Mining Model Data Items

Model Data Items	MineSight® Model Parameter Name
Topography – portion below topographic surface	TOPOG
Copper Assay Grade Diluted (%)	CUDIL
Gold Assay Grade Diluted (g/t)	AUDIL
Net Smelter Return for Copper ($/t)	NSR1
Net Smelter Return for Gold ($/t)	NSR2
Total Net Smelter Return ($/t)	NSR3
Block Economic Value ($)	BVAL
Slope Zone for Case A – Pipeline	SZONA
Slope Zone for Case B – No Pipeline	SZONB
Berm Width for Case A – Pipeline	BWIDA
Berm Width for Case B – No Pipeline	BWIDB
Interramp Slope Angle For Case A – Pipeline	SLOPA
Interramp Slope Angle For Case B – No Pipeline	SLOPB

Table 15.2 NSR Calculation Parameters

Parameter Name	Units	Parameter Value
Copper Price	$/lb	2.50
Gold Price	$/oz	1,085.00
Maximum Recovery for Copper[1]	%	91.17
Maximum Recovery for Gold[2]	%	86.49
Copper Concentrate Grade	%	25.00
Copper Concentrate Moisture	%	8.50
Payable Copper	%	97.00
Payable Gold	%	97.00
Copper Deduction	%	1.00
Gold Deduction	g/t concentrate	1.00
Treatment Cost	$/dmt concentrate	65.00
Copper Refining Charge	$/payable lb of copper	0.065
Gold Refining Charge	$/payable oz of gold	5.00
Land Freight	$/wmt concentrate	27.90
Port Charges	$/wmt concentrate	27.00

table continues…

Parameter Name	Units	Parameter Value
Ocean Freight	$/wmt concentrate	50.00
Miscellaneous Costs	$/wmt concentrate	3.97

Note: 1. CuRec% = -74.812 * Cu%2 + 85.727 * Cu% + 66.668.
2. AuRec% = 92.586 * $Au^{0.0649}$.

15.2.4 DILUTION AND MINING LOSSES

The effect of the expected internal dilution was analyzed and accounted for during the estimation of the grade components of the resource block model.

The in-situ grades from the resource model, together with the resource classification, were used for two-dimensional dilution estimation. Diluted grades were calculated based on neighbour blocks located in the same level, thus each block has four neighbour blocks to be analyzed: the north neighbour, the south neighbour, the east neighbour and the west neighbour.

If a neighbour block was classified as Measured or Indicated, then the analyzed block was diluted using the grade found in the neighbour block. If a neighbour block was classified as either Inferred, or not classified, then the analyzed block was diluted using zero grades.

The dilution factor considered for each of the neighbour blocks is 6.25%, representing a 1.0 m wide corridor on each side of the economic boundary. The formula applied can be represented as:

*Diluted Grade = (In-situ Grade * 75%) + ∑ (Dilution Factor * Grade of Neighbour Block)*

Measured and Indicated Resources within the final pit limits above a cut-off grade of US$4.53/t NSR have an average in-situ grade of 0.272% Cu; after running the dilution script, the average diluted grade was 0.267% Cu, representing a decrease of 1.7% in grade.

15.2.5 OPTIMIZATION CONSIDERATIONS

The Mineral Reserves at the Ajax deposit were delineated using a LG optimizing algorithm, which evaluates the profitability of each block in the block model based on its net value. The final pit limit determined by using the LG algorithm was smoothed to include practical mining constraints, such as minimum working areas, as well as ramps.

The net value used by the LG algorithm was based on the economic difference between the value per tonne of rock against the costs to mine and process each tonne of rock. The value was based on the estimated metal grades, the selected metal prices, dilution and ore loss factors, process recovery and smelter payments. The costs associated with each block of material included mining and processing.

Two optimizations were run. The first utilized the inter-ramp slope angle provided by BGC (refer to Section 15.2.6) and costs from the PEA study. The second incorporated the cost estimates obtained during the preliminary stages of feasibility examination and flattened the pit slopes as a result of mining considerations.

15.2.6 PIT SLOPES

BGC provided the open pit slope design and pit wall depressurization recommendations for the proposed Ajax open pit (refer to Appendix F).

The rock mass of the Ajax pit area has been divided into three geotechnical units, following the main geological units:

1. Nicola Group Volcanics: represents a single structural domain; found on the south and southwest limits of the proposed pit, at depth in the floor of the ultimate pit, and the upper east wall; typically moderately strong to strong, with weaker rock observed in the highly sepentinized zones; have a “poor” RQD and closely-spaced discontinuities.
2. Iron Mask Hybrid Diorite: represents two structural domains; located in the northeast, northwest and north-central areas of the proposed pit; strong to very strong; “good” RQD with closely- to moderately-spaced discontinuities.
3. Sugarloaf Diorite: represents four structural domains; located in the western, south-central, north-central and eastern portions of the proposed pit; strong to very strong; “fair” RQD with closely-spaced discontinuities.

The proposed open pit area has been divided into seven structural domains. Design discontinuity sets for each domain have been interpreted from the available rock mass fabric with considerations of set corresponding to the major fault fabrics of the study area and persistent structures as mapped from the photogrammetric or outcrop data. The design discontinuity sets are expected to be of adequate persistence to form multi-bench scale slope instabilities.

BGC established 40 design sectors based on geometric, geological, rock mass quality characteristics, and operational considerations. Based on these data, BGC derived pit slope design parameters.

Each material type of certain sector is coded with bench width and bench face angle based on the slope design parameters. Overall slope angles vary from 38º to 49º (Table 15.3).

AMEC coded the block model with variables containing these slope design parameters to be used for pit optimization and operational pit phase design.

Based on the results of the preliminary optimization and preliminary pit design, during the second pit optimization AMEC adjusted the overall slope angle taking in consideration the flattening effect caused by the incorporation of geotechnical berms, haulage ramps as well as ramps and platforms required for the IPCC infrastructure.

AMEC identified 15 design sectors to be used for pit design based on the domains and slope azimuth indicated by the optimized pitshell.

The mining block model was coded with overall pit slope angles based on flattened inter-ramp slope angle by two degrees for the walls on the north side of the pit and four degrees for the walls on the south side, and no flattening factor was applied for the walls on the west side of the pit.

15.2.7 *MINING COSTS*

The mining cost was based on first principle calculations for a conventional open pit mine using a truck and shovel fleet and the implementation of the IPCC method. Costs include direct operations and maintenance for drilling, blasting, loading and hauling. Other costs are general mine support for road, bench, and dump maintenance, dewatering, ore control, mining G&A costs and re-handling a nominal 5% of ROM ore. All costs are in US dollars.

The mining cost estimate also includes direct operation and maintenance for primary crushing and conveying from out-pit and in-pit locations.

Cost estimates from the preliminary pit optimization and design indicated an average mining cost of US$1.34/t. Details are shown in Table 15.4

An analysis of unitary mining cost by bench was carried out to assist on the definition of the reference mining cost and incremental cost by depth, ore and waste mining costs were assumed to be equal.

Three zones within the pit were defined and a specific mining cost adjustment formulas (MCAF) to represent increased haulage cost with pit depth for each zone:

1. Zone 1 (mined from bench elevation 832 m and above): Cost in $/t = (0.929 + 0.046i) where i is the number of 12 m benches below bench elevation 988 down to bench elevation 832.
2. Zone 2 (mined from bench elevation 820 to 688 m): An average cost of US$1.321/t for benches between bench elevations 820 and 688.
3. Zone 3 (mined from bench elevation 676 m and below): Cost in $/t = (0.916 + 0.066j) where j is the number of 12 m benches below 676 bench elevation.

Table 15.3 Pit Slope Design Details for the Base Case

Domain	Design Sector	Slope Azimuth		Catch Bench Geometry			Interberm Geometry		Overall Slope Geometry		Slope Design Control	Comments
		Start (°)	End (°)	Height Bh (m)	Angle Ba (°)	Width Bw (m)	Maximum Height Ibh (m)	Angle Iba (°)	Estimated Height Oh (m)	Angle Oa (°)		
NV	NV-055	050	060	24	65	19.7	144	38	216	40	Interberm (D1.F2)	-
NV	NV-068	060	075	24	65	16.4	-	41	144	44	Interberm (D1.F2)	-
NV	NV-088	075	100	24	65	11.6	144	46	264	47	Interberm (Rockmass stability)	-
NV	NV-135	100	170	24	65	18.6	144	39	264	41	Interberm (B1.B2; B1)	-
NV	NV-175	170	180	24	65	16.4	144	41	264	43	Interberm (B1.C1)	-
NV	NV-195	180	210	24	65	12.5	144	45	264	46	Interberm (B2.F2)	-
NV	NV-220	210	230	24	65	11.3	144	47	240	48	Interberm (Bench geometry)	-
NV	NV-245	230	260	24	65	11.6	-	46	144	49	Rockmass stability	F2.T is not likely to occur due to low to moderate set persistence.
SLDA	SLDA-028	020	035	24	65	11.6	-	46	144	49	Rockmass stability	-
SLDA	SLDA-050	035	065	24	65	11.6	-	46	144	49	Rockmass stability	-
SLDA	SLDA-090	065	115	24	65	12.5	-	45	144	48	Interberm (G3.T)	-
SLDA	SLDA-145	115	175	24	65	16.4	144	41	360	42	Interberm (B1.G2)	-
SLDA	SLDA-183	175	190	24	65	11.6	144	46	168	48	Interberm (B1.G2)	-
SLDA	SLDA-220	190	250	24	65	10.3	144	48	384	48	Interberm (Bench geometry)	E1.T is not likely to occur due to low density of set.
SLDA	SLDA-315	250	020	24	65	18.6	144	39	264	41	Interberm (D2+; D2.G3+; F2.T)	F2.T is not likely to occur due to moderate set persistence and dominantly shears.
SLDB	SLDB-128	090	165	24	65	10.7	144	48	384	48	Interberm (D2.T)	-
SLDB	SLDB-190	165	215	24	65	9.8	144	49	456	48	Interberm (Bench geometry)	-
SLDB	SLDB-248	215	280	24	65	20.9	-	37	144	40	Interberm (E1.T)	E1 is sub-parallel to regional fabric, but poorly documented in mapping dataset and weak in core data.
SLDC	SLDC-035	015	055	24	65	10.7	144	48	288	48	Interberm (A2.D2+)	-
SLDC	SLDC-073	055	090	24	65	18.6	144	39	288	40	Interberm (A1+)	-
SLDC	SLDC-105	090	120	24	65	18.6	144	39	288	40	Interberm (A1+; A1.B2+)	-
SLDC	SLDC-145	120	170	24	65	10.7	144	48	504	47	Interberm (A1.B2+)	-
SLDC	SLDC-180	170	190	24	65	10.7	144	48	312	48	Interberm (A1.B2)	-
SLDC	SLDC-348	320	015	24	65	16.4	144	41	168	44	Interberm (D2+; A2.D2+)	-
SLDD	SLDD-038	015	060	24	65	20.9	-	37	144	40	Interberm (A2.D4)	-
SLDD	SLDD-070	060	080	24	65	16.4	-	41	144	44	Interberm (A1.H1)	-
SLDD	SLDD-093	080	105	24	65	11.6	-	46	144	49	Rockmass stability	-
SLDD	SLDD-145	105	185	24	65	9.8	144	49	240	49	Interberm (Bench geometry)	H1.T is not likely to occur due to moderate set persistence.

table continues…

		Slope Azimuth		Catch Bench Geometry			Interberm Geometry		Overall Slope Geometry			
				Height	Angle	Width	Maximum Height	Angle	Estimated Height	Angle		
Domain	Design Sector	Start (°)	End (°)	Bh (m)	Ba (°)	Bw (m)	Ibh (m)	Iba (°)	Oh (m)	Oa (°)	Slope Design Control	Comments
SLDD	SLDD-218	185	250	24	65	12.5	144	45	192	47	Interberm (C1+)	-
SLDD	SLDD-258	250	265	24	65	9.8	144	49	288	49	Interberm (Bench geometry)	-
SLDD	SLDD-295	265	325	24	65	17.5	144	40	288	41	Interberm (D1+; D4.D1+; A1.D1+)	-
SLDD	SLDD-350	325	015	24	65	20.9	-	37	144	40	Interberm (A1.D1+)	-
IMHA	IMHA-000	325	035	24	65	18.6	144	39	432	40	Interberm (D2+; D2.E2+)	-
IMHA	IMHA-043	035	050	24	65	16.4	144	41	408	42	Interberm (A1; D2.B1+; A1.D2+)	-
IMHA	IMHA-298	270	325	24	65	9.8	144	49	456	48	Interberm (Bench geometry)	-
IMHB	IMHB-035	000	070	24	65	20.9	144	37	360	38	Interberm (A1+; A1.H1+)	-
IMHB	IMHB-085	070	100	24	65	9.8	144	49	336	49	Interberm (Bench geometry)	-
IMHB	IMHB-298	270	325	24	65	9.8	144	49	408	49	Interberm (Bench geometry)	-
IMHB	IMHB-330	325	335	24	65	10.7	144	48	408	48	Interberm (A1.H1)	-
IMHB	IMHB-348	335	000	24	65	10.7	144	48	456	47	Interberm (A1.H1+)	-

Table 15.4 Preliminary Mining Costs

	Unit	Ore
Drill	$/t	0.08
Blast	$/t	0.13
Load	$/t	0.12
Haul	$/t	0.60
Materials Handling	$/t	0.21
Support	$/t	0.10
Mining G&A	$/t	0.10
Total	**$/t**	**1.34**

Note: This unit cost estimate is only the basis of the LG optimization work, not the final mining cost.

After these incremental cost factors were applied, the weighted average mining cost resulted in US$1.32/t and ranged from US$0.92/t to US$2.50/t for the different mining benches. These factors were adopted in the second pit optimization process.

15.2.8 PROCESSING COSTS

The processing cost was based on first principle calculations for a 60.0 kt/d processing facility, estimated at US$3.38/t.

The G&A cost estimates for an open-pit mine supported by local employees without the need of a mining camp on-site were derived from the PEA and input from Abacus. A G&A cost of US$0.51/t of material processed was added to the processing cost in the pit optimization.

An allowance of US$0.05/t of ore processed was also included in the processing cost to account for closure costs.

15.2.9 PROCESS RECOVERY

The process recoveries are included in the NSR estimation.

15.2.10 SELLING COSTS

Treatment, refining, freight and selling costs are included in the NSR estimation.

15.2.11 METAL PRICES

The metal prices used for pit optimization were US$2.50/lb copper and US$1,085/oz gold. No other metals were considered in the revenue stream. These prices are included in the NSR estimation.

15.2.12 DISCOUNT RATE

No discount rate was used during pit optimization. Consequently, no bench discounting with depth was applied in the ultimate pitshell generation. However, for the selection of the mining phases and mine production schedule, a discount rate of 8% was used to calculate indicative NPV and other key performance indicators.

15.3 MINERAL RESERVES STATEMENT

Mineral Reserves have been modified from Mineral Resources by taking into account geologic, mining, processing, and economic parameters and therefore are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Qualified Person for the Mineral Reserve estimate is Ramon Mendoza Reyes, P.Eng., an AMEC employee.

Mineral Reserves are reported at a gold price of US$1,085/oz gold, a copper price of US$2.50/lb, and have an effective date of October 31, 2011. The copper equivalent grade used the following equation:

$$CuEq = \frac{(\%Cu)(CuRec)(22.0462)(\$lbCu) + (g/t/Au)(AuRec)(1/31.1035)(\$ozAu)}{(CuRec)(22.0462)(\$lbCu)}$$

Mineral Reserves comprising mineralized material classified in the Proven and Probable categories are expressed as run-of-mine (ROM) material above a marginal cut-off grade of US$4.53/t NSR and summarized in Table 15.5.

Table 15.5 Mineral Reserve Statement, Effective Date October 31, 2011, R. Mendoza Reyes, P.Eng.

Confidence	Cut-off	ROM	Average ROM Grades		Copper Equivalent	Contained Metal	
Category	**Grade ($/t)**	**Tonnes (Mt)**	**Cu (%)**	**Au (g/t)**	**CuEq (%)**	**Copper (Mlb)**	**Gold (Koz)**
Proven Mineral Reserve	4.53	279.5	0.27	0.17	0.38	1,680	1,520
Probable Mineral Reserve	4.53	223.5	0.26	0.17	0.37	1,280	1,230
Total Proven & Probable Mineral Reserves	**4.53**	**503.0**	**0.27**	**0.17**	**0.37**	**2,960**	**2,750**

Note 1. Mineral Reserves are estimated using a cut-off grade of US$4.53/t NSR, a copper price of US$2.50/lb, and a gold price of US$1,085/oz. The NSR is calculated by adding the NSR attributable to copper to the NSR attributable to gold and then subtracting the freight costs, which include land freight, port charges, ocean freight and miscellaneous costs. The attributable copper is calculated using the metallurgical recovery obtained by the

formula: CuRec (%) = -74.812 * Cu(%)2 + 85.727 * Cu(%) + 66.668 with a maximum copper recovery of 91.17%. The attributable gold is calculated using metallurgical recovery obtained by the formula: AuRec (%) = 92.586 * Au(g/t)$^{0.0649}$ with a maximum gold recovery of 86.49%.

Note 2. Mineral Reserves are constrained within a pitshell, optimised using assumptions of a weighted average mining cost of US$1.32/t (ranging from US$0.92/t to US$2.50/t for the different mining benches); a processing cost of US$3.38/t (including US$0.51/t general and administrative costs, and US$0.05/t allocation for closure costs); and pit slope angles that vary from 40º to 49º.

Note 3. A 0.5% mining loss factor was applied to account for dilution; diluted grades are estimated at 1.7% lower than the in-situ grades.

Note 4. The life of mine, waste to ore strip ratio is 2.42. The assumed life-of-mine throughput rate is 60 kt/d.

Note 5. The copper equivalency is calculated using the equation CuEq = [(%Cu) (CuRec) (22.0462) ($lbCu) + (g/t/Au) (AuRec) (1/31.1035) ($ozAu)] ÷ [(CuRec) (22.0462) ($lbCu)].

Note 6. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.

Note 7. Tonnage and grade measurements are in metric units. Contained gold ounces are reported as troy ounces; contained copper pounds are Imperial pounds.

15.4 Factors That May Affect the Mineral Reserve Estimate

Factors which may affect the Mineral Reserve estimates include:

- effectiveness of the dilution model
- gold and copper price assumptions
- LOM metallurgical recoveries
- geotechnical characteristics of the rock mass
- ability of the mining operation to meet the planned annual throughput rate assumptions for the process plant
- capital and operating cost estimates
- effectiveness of surface and ground water management
- likelihood of obtaining required permits and social licenses.

The AMEC QP is the opinion that these potential modifying factors have been adequately accounted for using the assumptions in this report, at this feasibility level of study, and therefore the Mineral Resources within the mine plan may be converted to Mineral Reserves using the appropriate confidence categories.

Factors which may affect the assumptions in this technical report in relation to Mineral Reserves include:

- commodity price

- unrecognized structural complications in areas with relatively low drillhole density could introduce unfavourable pit slope stability conditions
- ability to move the oil pipeline as the current pit design has not impacted on the present pipeline location
- granting of appropriate environmental and construction permits would be forthcoming from the relevant authorities
- effective surface and ground water management will be important to the safety and productivity of the mining operation.

15.5 Comment on Section 15

The AMEC QP is of the opinion that the Mineral Reserves for the Project, which have been estimated using core drill data, appropriately consider modifying factors, have been estimated using industry best practices, and conform to the requirements of CIM (2010).

Please refer to Appendix C for AMEC's full report.

16.0 MINING METHODS

16.1 THROUGHPUT CONSIDERATIONS

Most efforts to determine the process facility throughput were concentrated in the early stages of Project development.

Initial considerations in 2008 were for a 40,000 t/d (14.6 Mt/a) process facility. Subsequently, it became apparent that an increase in processing throughput to 60,000 t/d (21.9 Mt/a) resulted in a significant change to the NPV of more than 100%. The 60,000 t/d (21.9 Mt/a) throughput rate was used in the 2009 PEA.

During 2010, the schedule was revised from 60,000 t/d to 75,000 t/d and 90,000 t/d of material delivered to the process facility. Operating and capital costs were estimated for these higher throughputs and the project NPV was calculated. This indicated relatively small NPV increases of 9% and 6% for the 75,000 t/d and 90,000 t/d cases over the 60,000 t/d base case. This gain was based on significant estimated initial capital increases of $136 million and $200 million respectively.

A particular challenge for throughput rationalization studies during the feasibility study was the multiple impacts of increasing throughput. Throughput was not merely constrained by mill processing, but also potentially by in-pit crushing, conveying and waste stacking (IPCC/S), thickened tailings, and HPGR systems. Throughput rationalization therefore required multiple-discipline capital and operating cost estimates, as well as resource model reclassification confidence limit changes.

Due to indications from earlier work that higher throughput would have limited achievable NPV benefits; the preferred development course was to maintain the 60,000 t/d process facilities. As a consequence the throughput rate for the feasibility study remained at 60,000 t/d.

16.2 PIT DESIGN

Parameters considered in the pit design are discussed in Section 15.0.

16.3 PIT PHASES

During optimization five pitshells were selected to guide the pit design work; however, some modifications were made to improve production scheduling; as a result, two additional pit phases were designed.

Additional modification was required to join isolated mining areas and provide room for access ramps. The longest span of the pit is approximately 2.6 km in an east-west direction and approximately 1.4 km in a north-south direction. The highest crest elevation of the ultimate pit is 988 m and the pit bottom elevation is 412 m.

The resulting designed pit phases are shown in a plan view for the 868 m bench in Figure 16.1. Phase durations are as indicated in Figure 16.2. Phase 4 has been designed to accommodate backfilling late in the mine plan.

16.3.1 PIT DESIGN CONSIDERATIONS

The following were incorporated in the pit design:

• bench height, single-bench mining	12 m
• height between catch benches	24 m double bench
• final bench height	24 m double bench in final wall
• bench face angle	65º
• berm width	as per design sector
• road width	35 m
• road width at the bottom pit	25 m
• maximum grade uphill loaded	+10%
• maximum grade downhill loaded	-10%

16.4 PROPOSED MINING OPERATION

16.4.1 PIONEERING WORK AND PIT DEVELOPMENT

The terrain associated with the Ajax deposit does not require special preparation for mine development. Initial pioneering and pit development during the preproduction period will be accomplished with front-end-loaders (FELs), dozers, percussion drill, and rear end dump trucks. The objective is to remove the overburden, develop mine access roads suitable for large mining equipment, and “face-up” the initial pit into productive set-ups for the large shovel and mining equipment.

Organic material will be stripped from pit areas and waste dump foundations, as directed by environmental and geotechnical plans. This will be accomplished by pushing down the overburden material to centralized staging areas where the loader and truck fleet will load and haul the material away. Suitable organic material will be stockpiled for future use in reclamation; however, some will be placed into the waste dump storage. Drill-and-blast may be required to provide level working platforms.

Figure 16.1 Phase Designs in Plan at 868 m Elevation

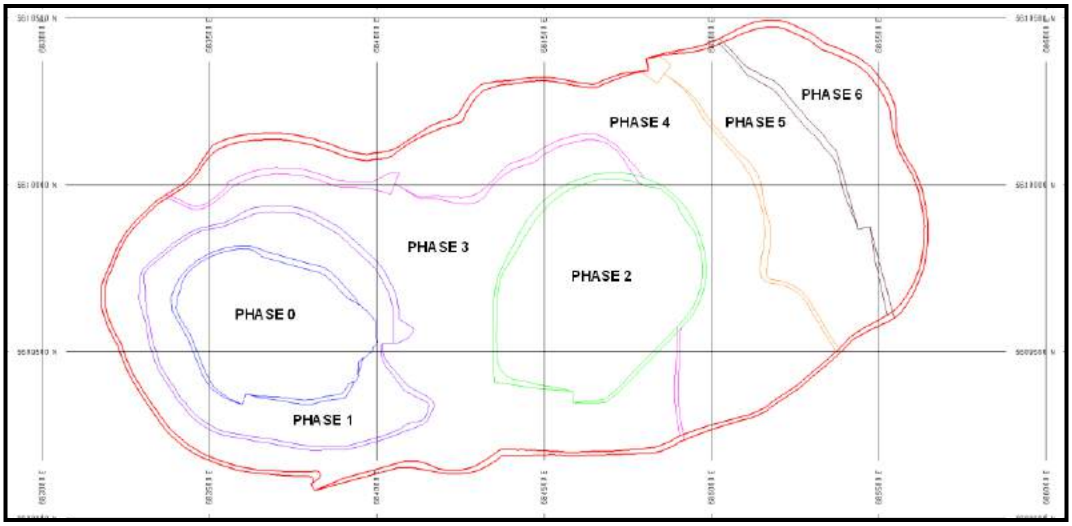
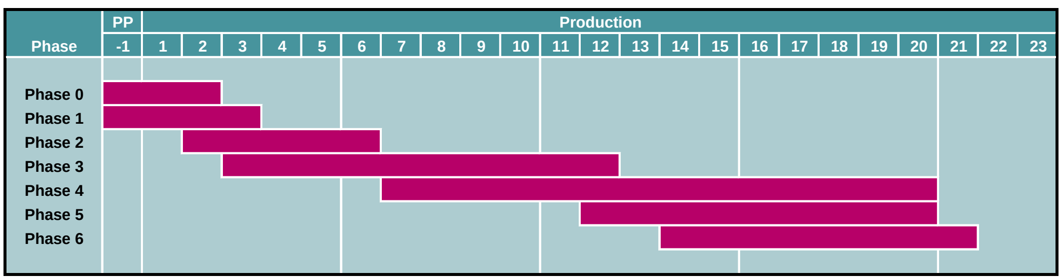


Figure 16.2 Proposed Pit Phase Scheduling



16.4.2 *Blast-hole Drilling*

Blast-hole drilling in predominantly waste areas will be performed with nominal 270 mm (10 5/8") diameter production drills.

Ore and waste zones will be mined on 12 m benches with nominal 13.5 m deep holes (12 m plus 1.5 m sub grade) drilled in a single pass. Planned blast-hole locations will be marked on cleaned benches by the survey team.

A wall control program consisting of pre-splitting and cushion blasts will be carried out along all ultimate walls including the intermediate pit phases. This wall control pattern will include a three-row trim blast and a pre-shear line. Two lines of the trim or cushion pattern will be drilled with the production drill rig. The last cushion blast line and the pre-shear holes will be drilled with a percussion drill.

16.4.3 BLASTING

To successfully achieve the feasibility open pit design, controlled blasting techniques will be required, including cushion (trim and buffer) and pre-split/pre-shear blasting. Dust control and vibration reduction for blasting has been taken into consideration. Blast damage to the slopes must be minimized to preserve strength along bedding planes defining the potential failure blocks.

Waste material will be blasted to produce a suitable particle size distribution for loading and transportation in 218 t class trucks. Mineralized material will be blasted to comply with fragmentation requirements and a specified particle distribution.

All blast patterns for production blasts will use a 10 5/8" (270 mm) diameter hole. This diameter blasthole was based on fragmentation results of two 15 hole test blast patterns completed at Ajax in February 2011 (Orica, 2011).

A blend of ammonium nitrate/fuel oil (ANFO) and emulsion explosives will be used during this process. A facility will be built by an explosives contractor following Canadian regulations to store the blasting materials and equipment safely. An explosives contractor will supply the mixed explosives products down the hole with the use of blend and pumping trucks. The tie-in and blasting function will be managed by the Owner.

16.4.4 LOADING

Primary loading of waste and ore on the full 12 m benches will be accomplished by electric-hydraulic shovels with a 42 m^3 bucket. Diesel FEL with a 40.5 m^3 bucket will be used as a back-up to the shovels and to work on less productive faces and muck pile clean-up.

Electric shovels and front-end-loaders will be assigned to operate in ore and waste, this will increase flexibility of the production plan, allowing operation of multiple mining faces. All shovels will be equipped with a global positioning system (GPS) to allow real-time updates of the digging face and relation to ore/waste contacts.

16.4.5 HAUL ROADS

Haul roads are required between the pit phases and the ore crusher, waste dumps, overburden stockpiles, construction areas, and truck shop. The roads have generally been laid out with a cut-and-fill balance inside the ultimate pit limit and as pure-fill roads outside the ultimate pit limits. The road design follows the criteria of 3.5 times the width of the widest haul equipment and a safety berm of 0.5 times the diameter of the tires, a small allowance for potential water runoff was included. The roads will be maintained by graders and wheel dozers.

16.4.6 HAULING

Large rear-end-dump haul trucks (218 t class) will be used for hauling both ore and waste to their destinations.

16.4.7 IN-PIT CRUSHING AND CONVEYING

IPCC is a system where the mined material is reduced in size by comminution to make it suitable to be transported by belt conveyors out of the pit. After the PEA, IPCC was identified as a potential cost reduction implementation for the Ajax Project when compared with a truck-haul option only. The mine plan was completed using conventional truck-haul with and without IPCC for comparison.

Two potential locations for the IPCC were assessed, on the north wall, and on the south wall; the south wall site was selected. Figure 16.3 shows the planned layout of the IPCC system.

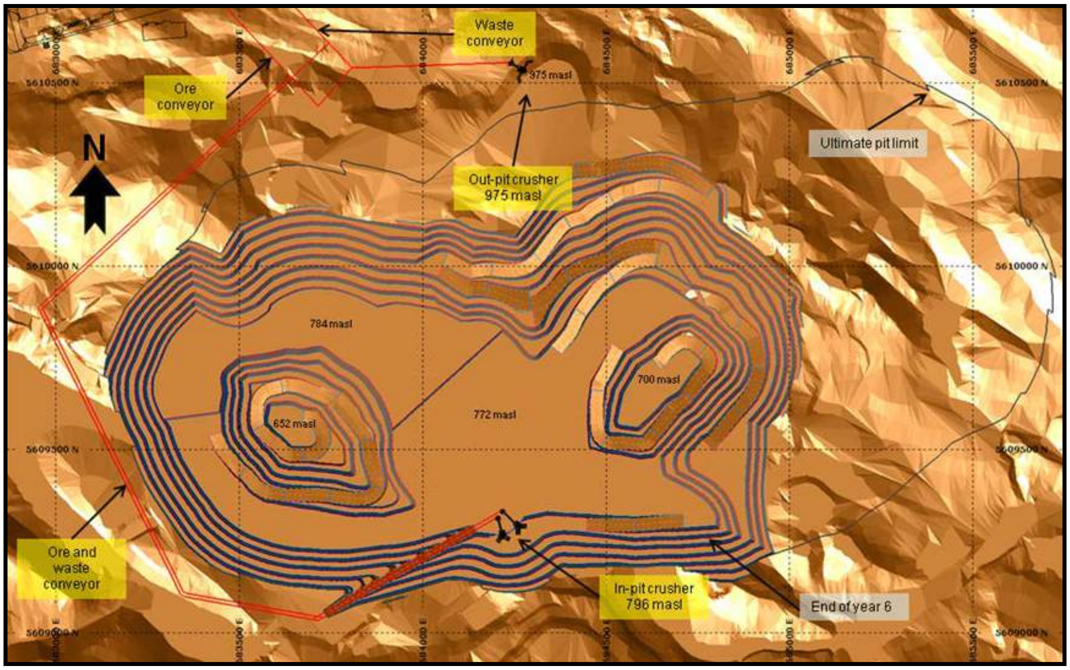
In general, the IPCC system will consist of three semi-mobile crushing plants and corresponding belt conveyors. The crushers will be mounted on concrete pontoons and can be relocated from time to time according to the mine development schedule. In the ore handling system, the ore will be crushed to the size which meets the process requirements. The crushed ore will be transferred to the coarse ore stockpile by the belt conveyors.

In the waste rock handling system, the rock will need to be crushed so that it can be handled by the down-stream belt conveyors and stacking system. Normally, the maximum lump size to be conveyed should be less one third of belt width or smaller to minimize the damage to the impact idler on the transfer chutes.

CRUSHER SYSTEM

The crusher installation will be divided into three phases. In Phase 1, a crusher will be constructed on the exit point of the ultimate pit design (out-pit crusher). This particular crusher will initially be used as an ore crusher and can be switched to be waste or ore crusher when the in-pit crushing system is in place. The crusher will remain on its location until the end of mine life.

Figure 16.3 IPCC Layout



During Phase 2, two in-pit crushers will be constructed at the 796 m elevation early in Year 7 and will be ready to be operated in the middle of Year 7. The two crushers will handle ore and waste material.

During Phase 3, the two in-pit crushers will be relocated from the 796 m elevation to the 700 m elevation. The relocation of both in-pit crushers will be done in Year 13 so that they can be used by the beginning of Year 14.

CONVEYORS

Conveyors will handle ore material from the ore crushers to the plant, and the waste material will be handled from the waste crusher to the North Dump equipped with a stacking system.

Conventional conveyors were selected for the IPCC system. The conveyors will be constructed on the south and west side of the ultimate pit limit. The conveyors will have to cross a shallow portion of the lake on the west side of the pit where a protection berm will be constructed to help prevent water flow into the pit. The overland conveyor on the west side will utilize this protection berm for its foundation.

EQUIPMENT SIZING

An overall availability of 70% was selected to size equipment. For the crusher design capacity, no design factor was considered. A design factor of 1.15 was considered for the belt conveyor and the stacking system design capacities to handle possible higher crushing capacity due to finer and softer ROM fed to the crusher.

CRUSHING PLANT

The crushing plant will consist of a truck dump pocket, gyratory crusher, surge pocket and belt feeder. One waste rock crushing plant will be located outside the pit and the two other crushing plants will be located at the same truck pad inside the pit (one for ore and one for waste). However, each crushing plant can be used to handle either ore or waste rock depending on the operational requirements. It should be noted that since the waste rock handling capacity will be about 34% higher than ore handling capacity, different main shafts will be required for ore and waste rock application.

WASTE ROCK STACKING SYSTEM

The waste rock stacking system will consist of shiftable conveyors, travelling tripper and spreader. A starter pile, which will be stacked by truck and located in the east side of the North Waste Rock Dump area, is proposed as the starting point for the stacking system. To reduce downtime due to equipment relocation, the height of the starter pile will be the same as the final dump height and stacking will be developed using downcast mode. The selected stacking mode will be carried out by radial advance stacking by shifting the conveyor around a pivot point.

16.5 PRODUCTION SCHEDULE

16.5.1 PROCESS CONSTRAINTS

Two main variables control the plant throughput:

- mill capacity – processing design circuit capability affected by plant utilization factor, which was considered at 92%.
- thickened tailings – maximum amount of tailings that can be processed with the design thickener capacity, also at 92% uptime.

Plant scheduling is based on optimizing copper production, making the best possible use of the mill capacity and using material from the mine and from ore stockpiles separated according to material value.

16.5.2 MINING CONSTRAINTS

Mining operational constraints include:

- Minimum operational widths of 60 m.

- Maximum vertical advance per phase per year: twelve 12 m benches.
- The process capacity is 21.9 Mt/a of ore. Waste production will be variable depending on the phases being mined.

16.5.3 PRODUCTION SCHEDULE

The mine production schedule was based on a requirement of providing an ore production schedule of 21.9 Mt/a. The projected mine life is approximately 23 years using seven pit phases.

The mine plan has higher-grade ore being mined during the first six years of production. To accomplish this, the ore below an NSR cut-off of US$9.20/t was assumed to be stockpiled in two different stockpiles during the first three years of production (Years 1 to 3). One stockpile would be for low-grade ore (US$4.53/t to US$5.88/t NSR) and the second would be for medium-grade ore (US$5.88 to US$9.20/t NSR). For the next three production years (Years 3 to 6), only the low-grade material would need to be stockpiled since the cut-off decreased to $5.88/t NSR. In Year 7 the cut-off decreases to US$4.53/t NSR so all of the ore can be directly fed to the process plant without stockpiling, and this continues until Year 18. During Year 18, the cut-off is increased to US$9.20/t NSR once again due to the availability of higher-grade ore.

The mine plan also incorporates a combined ore and waste mining rate that would not spike higher in individual years requiring drastic expansion of the mining equipment fleet requirements for short durations.

The LOM production plan is included as Table 16.1.

16.6 WASTE ROCK STORAGE CONSIDERATIONS

Two waste rock storage facilities (WRFs) are planned. One will be constructed at the north side of the pit, and will become the North Waste Dump (NWD). The second, the East Waste Dump (EWD) will be located at the east side of the pit (Figure 16.4).

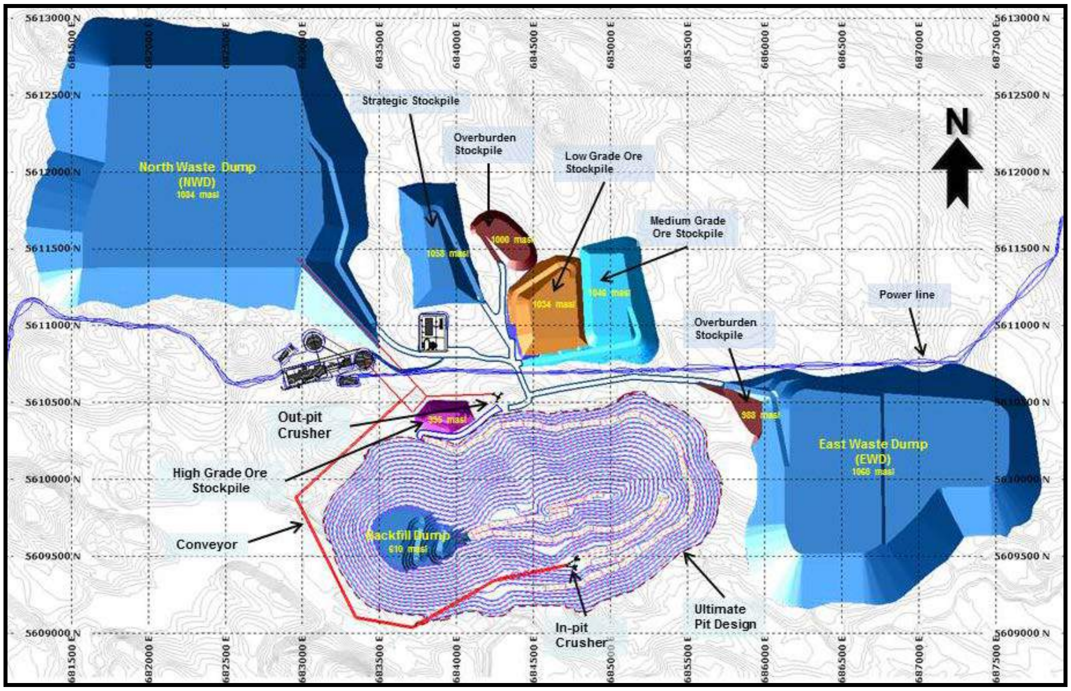
Table 16.1 Planned Production Schedule

			PP	Production																						
Description	**Units**	**Total**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**
Ore to Plant																										
Low Grade Stockpile	kt	22,770	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1,558	21,212
Medium Grade Stockpile	kt	36,464	0	0	0	0	0	0	0	0	0	1,028	2,052	2,440	898	0	0	0	137	0	384	1,930	66	7,185	20,343	0
High Grade Stockpile	kt	3,526	0	1,146	0	0	0	0	0	0	295	2,085	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 0	kt	19,253	0	18,148	1,105	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 1	kt	28,874	0	2,605	19,079	7,190	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 2	kt	53,770	0	0	1,715	14,710	19,628	14,519	3,197	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 3	kt	113,991	0	0	0	0	2,272	7,380	18,703	21,900	20,871	17,350	16,231	6,195	3,090	0	0	0	0	0	0	0	0	0	0	0
Phase 4	kt	129,525	0	0	0	0	0	0	0	0	733	1,437	3,618	13,265	16,516	14,176	13,224	16,601	14,141	12,504	13,146	8,077	2,087	0	0	0
Phase 5	kt	42,735	0	0	0	0	0	0	0	0	0	0	0	0	1,395	7,724	8,676	4,683	5,302	5,676	2,644	2,996	3,640	0	0	0
Phase 6	kt	52,100	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	616	2,319	3,719	5,727	8,897	16,108	14,714	0	0
Total	**kt**	**503,008**	**0**	**21,899**	**21,899**	**21,900**	**21,900**	**21,899**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,900**	**21,212**
Ore to Low Grade Stockpile																										
Phase 0	kt	1,017	164	823	30	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 1	kt	2,957	77	586	2,103	191	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 2	kt	7,704	0	0	1,138	2,975	2,862	600	129	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 3	kt	4,525	0	0	0	0	369	1,283	2,874	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 4	kt	1,526	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	891	412	224	0	0	0
Phase 5	kt	2,337	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	614	672	1,052	0	0	0
Phase 6	kt	2,702	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	183	352	1,267	900	0	0
Total	**kt**	**22,770**	**241**	**1,409**	**3,272**	**3,166**	**3,231**	**1,883**	**3,003**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1,688**	**1,436**	**2,542**	**900**	**0**	**0**
Ore to Medium Grade Stockpile																										
Phase 0	kt	2,836	281	2,441	114	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 1	kt	8,831	320	951	6,692	867	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 2	kt	9,247	0	0	2,604	6,643	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 4	kt	3,651	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	2,584	823	244	0	0	0
Phase 5	kt	4,879	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1,196	1,640	2,044	0	0	0
Phase 6	kt	7,021	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	412	1,438	3,580	1,591	0	0
Total	**kt**	**36,464**	**601**	**3,392**	**9,411**	**7,509**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**4,192**	**3,900**	**5,868**	**1,591**	**0**	**0**
Ore to High Grade Stockpile																										
Phase 0	kt	1,146	1,146	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 2	kt	2,380	0	0	2,380	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total	**kt**	**3,526**	**1,146**	**0**	**2,380**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**

table continues…

			PP	Production																						
Description	**Units**	**Total**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**19**	**20**	**21**	**22**	**23**
Material to Strategic Stockpile																										
Phase 0	kt	484	52	411	21	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 1	kt	1,593	23	314	1,150	105	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 2	kt	3,663	0	0	607	1,362	1,260	404	30	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 3	kt	5,535	0	0	0	0	134	597	1,550	1,652	707	513	312	53	16	0	0	0	0	0	0	0	0	0	0	0
Phase 4	kt	5,439	0	0	0	0	0	0	0	0	46	360	408	1,123	1,069	682	605	613	533	0	0	0	0	0	0	0
Phase 5	kt	1,857	0	0	0	0	0	0	0	0	0	0	0	0	92	515	784	186	280	0	0	0	0	0	0	0
Phase 6	kt	755	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	122	633	0	0	0	0	0	0	0
Total	kt	**19,325**	**75**	**725**	**1,778**	**1,468**	**1,394**	**1,001**	**1,580**	**1,652**	**753**	**874**	**720**	**1,176**	**1,176**	**1,197**	**1,389**	**921**	**1,446**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Waste																										
Phase 0	kt	25,145	14,549	10,458	138	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 1	kt	86,547	13,388	49,016	23,742	401	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 2	kt	62,890	0	0	31,279	18,056	9,261	4,033	260	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Phase 3	kt	305,177	0	0	0	41,401	58,114	65,084	67,157	48,114	17,408	6,145	1,420	240	95	0	0	0	0	0	0	0	0	0	0	0
Phase 4	kt	401,801	0	0	0	0	0	0	0	22,234	53,839	64,982	69,860	70,584	50,655	18,583	13,455	13,336	8,605	6,722	5,072	2,667	1,209	0	0	0
Phase 5	kt	145,768	0	0	0	0	0	0	0	0	0	0	0	0	20,074	52,221	37,057	9,001	8,171	8,999	3,563	3,027	3,655	0	0	0
Phase 6	kt	147,685	0	0	0	0	0	0	0	0	0	0	0	0	0	0	20,099	48,742	36,778	9,224	7,846	8,156	11,600	5,240	0	0
Total	kt	**1,175,013**	**27,937**	**59,474**	**55,160**	**59,858**	**67,375**	**69,117**	**67,417**	**70,348**	**71,247**	**71,127**	**71,280**	**70,824**	**70,824**	**70,803**	**70,611**	**71,078**	**53,554**	**24,945**	**16,481**	**13,849**	**16,464**	**5,240**	**0**	**0**
Totals																										
Ore to Plant	kt	503,008	0	21,899	21,899	21,900	21,900	21,899	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,900	21,212
Ore to Stockpile	kt	62,760	1,988	4,801	15,062	10,675	3,231	1,883	3,003	0	0	0	0	0	0	0	0	0	0	0	5,879	5,336	8,410	2,491	0	0
Waste	kt	1,194,338	28,012	60,199	56,938	61,326	68,769	70,118	68,997	72,000	72,000	72,000	72,000	72,000	72,000	72,000	72,000	72,000	55,000	24,945	16,481	13,849	16,464	5,240	0	0
Total Material	kt	**1,760,106**	**30,000**	**86,899**	**93,900**	**93,901**	**93,899**	**93,900**	**93,900**	**93,900**	**93,899**	**93,900**	**93,900**	**93,901**	**93,899**	**93,900**	**93,900**	**93,899**	**76,900**	**46,844**	**44,261**	**41,086**	**46,775**	**29,631**	**21,900**	**21,212**

Figure 16.4 Proposed WRF Layout



A 42% swell factor (loose density: 2.05 t/m^3) was applied to determine the amount of waste rock that could be stored in each WRF. Capacities are as indicated in Table 16.2.

The first phase of the NWD will be constructed by haul trucks in 12 m lifts. The haul trucks will build an initial pile at the NWD to the top of the dump which will be used for stacker and conveyor platform in Year 7. The grade of the conveyor ramp up to the peak of NWD will be 20%, and the width of the conveyor ramp is a planned 35 m. A stacker and conveyor system will be installed at the 1,084 m elevation once the initial pile is constructed. From the total capacity of 728 Mt on NWD, only 141 Mt (19%) will be built by haul trucks and the remaining capacity will be built utilizing the stacker system.

The EWD will be built by haul trucks in 12 m lifts except the first lift which will be built at the 988 m elevation.

The dumping by haul truck on NWD and EWD will be driven by hauling time which will be minimized on the early periods.

Table 16.2 Waste Rock Facility Storage Capacities

Waste Rock Storage Area	Volume Capacity (Mm^3)	Loose Density (t/m^3)	Top Elevation of the Design (m)	Tonnage Capacity (Mt)
North Waste Dump	355	2.05	1084	728
East Waste Dump	205	2.05	1060	420

During Years 20 and 21, 19.6 Mt of waste from Phase 6 will be directed to the west side of the pit (bottom of Phase 4) as backfill. Backfilling will be deposited in 12 m lifts with adequate catch berms to create an overall slope of 1V:2H. Tonnage capacities shown in Table 16.2 are smaller than the ones reported in BGC's WRSA design report due to the in-pit backfill considerations and changes in the waste dump geometries after BGC's design report was finalized.

16.6.1 *Acid Rock Drainage Considerations*

A preliminary program to investigate the potential for acid drainage of the waste rocks is underway. This program comprises the sampling of all waste rock units to perform static testing on them. Based on testing results, the samples will be classified according to their acid generation potential or their neutralization potential.

Although it is expected that rocks in the Ajax deposit present a low probability of acid generation, recommendations for future work include a more detailed sampling and testing program and the incorporation of kinetic tests to provide better understanding of the results of the static testing program.

In the case that the waste rock demonstrates acid generating potential, the waste disposal procedure needs to be revised. If disposal of waste requires different sequencing, layering, covering or any other procedure, it would likely impact the operating cost assumed in this technical report.

16.6.2 *Reclamation Considerations*

Concurrent reclamation activities for the WRFs will include the construction of erosion control structures to avoid material dispersion that could affect surrounding vegetation and habitat, block natural drainage lines and interfere with the operations of other land users in the area.

Additionally, at the end of WRF life, re-contouring of the slopes will be carried out to ensure the physical stability of the facility, taking into account applicable legislation. Topsoil will be spread over all surfaces at a thickness depending on the nature of the underlying waste rock. The surfaces will then be deep-ripped at an appropriate spacing. Finally, seeding and mulching will be undertaken.

16.7 Overburden Storage Considerations

A total of 7.5 Mt of in-pit overburden at the proposed Ajax pit will be stockpiled over LOM to meet the site reclamation requirements. The remainder is planned to be stored within the waste dump.

Concurrent reclamation of the waste dump will be undertaken during operations as sufficient area becomes available. Where overburden directly removed from the pit is unavailable, it will be reclaimed from the stockpiles.

A total of 4.5 Mt of the North Overburden Stockpile will be stockpiled close to the NWD and another 3 Mt in the East Overburden Stockpile will be stored close to the EWD. The overburden tonnages in the two stockpiles are considered sufficient to cover the reclamation requirements for the NWD and EWD. A location plan, which is an inset of Figure 16.4, is included as Figure 16.5.

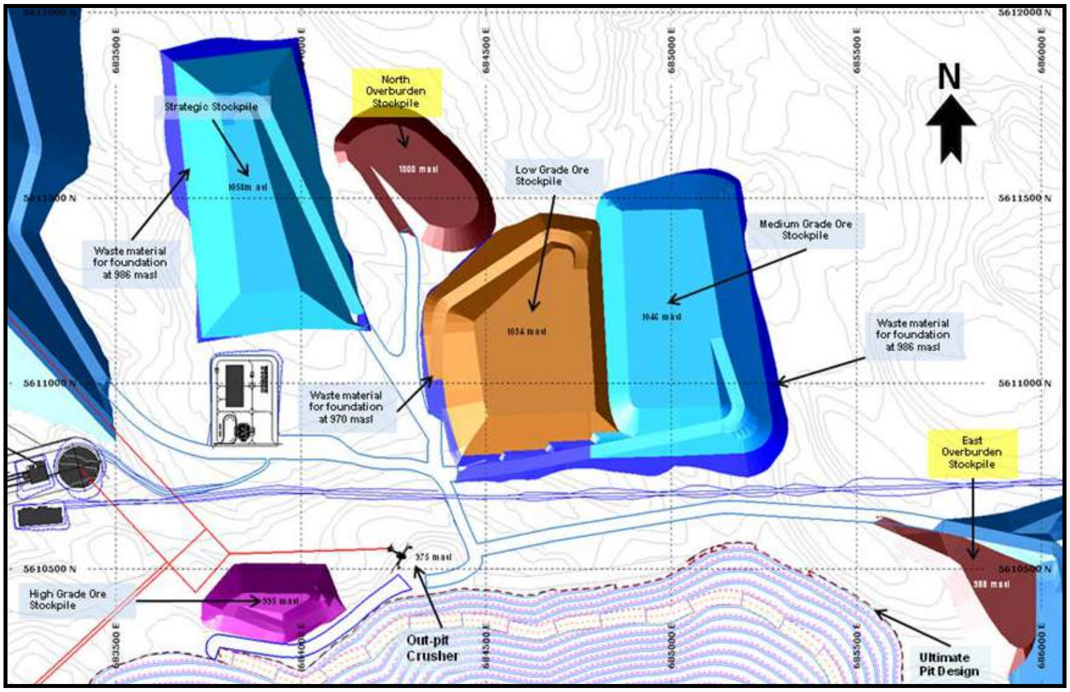
16.8 Ore Stockpiles

A total of 87 Mt of material will be required to be stored in four stockpiles to be located to the north of the pit (refer to Figure 16.5). The four stockpiles will consist of a high-grade ore stockpile, a medium-grade ore stockpile, a low-grade ore stockpile and a strategic stockpile.

The high grade stockpile will hold up to 2 Mt of ore which will be fed to the process plant in Years 1, 8, and 9 of the mine life. The medium-grade stockpile will hold 36 Mt of medium grade ore material. This stockpile will be built over a base of waste material at the 986 masl elevation and a ramp will be developed at a 10% grade to its crest at the 1,022 masl elevation.

The low-grade stockpile is designed to hold 23 Mt of low-grade ore material. This stockpile will be built over a base of waste material at the 970 masl elevation and a ramp will be developed at a 10% grade to its crest at the 1,034 masl elevation.

Figure 16.5 Proposed Stockpile Layout



Using the elevated cut-off NSR strategy adopted for the mine plan, some 19 Mt of low-margin mineralization will be mined during the Project life and stored in a "strategic stockpile". This material will be above the strategic cut-off grade of $4.53/t NSR. Although, if processed, this mineralization will theoretically generate a positive operating margin, it has been excluded from the Mineral Reserve statement since it is not scheduled for treatment; associated operating costs of re-handling this material have not been allowed for during feasibility evaluations. The mineralization in the strategic stockpile represents a potential upside to the Project if metal prices, re-handling costs and metallurgical recovery at the time result in a favourable economic outcome.

16.9 PIT WATER MANAGEMENT CONSIDERATIONS

16.9.1 DEWATERING HOLES

A contingency of 30 vertical holes has been allowed for the feasibility study, however no vertical dewatering holes are currently planned for long-term rock mass dewatering.

Horizontal drains are proposed as the primary means to depressurize potential bench scale wedge and planar failures, requiring that the slopes be depressurized to a distance of between 50 and 75 m behind the bench face.

To prevent water pressure from building up immediately behind pit walls, 140 mm diameter horizontal drain holes will be installed every 100 m, on every second double bench stack. These holes will be an average of 150 m long. The holes will be drilled into the bench at an upward inclined angle to promote drainage. Over the life of the project approximately 1,083 drain holes and 162,450 m of drilling will be required (BGC, 2011d Appendix F).

16.9.2 PIT WATER

Precipitation, seepage from walls, and horizontal drains will introduce water into the pit. Some of this water will be absorbed by the broken rock and hauled with the rock out of the pit. Ditches will be used to route the remaining water to sumps where the solids can settle out, and the water can be pumped to the central water pond (refer to Section 18.5). A large collection system of ditches, pipes, sumps, pumps, and booster pumps is needed to contain this water.

The mine drainage and dewatering system will perform the following tasks:

- maintain pit wall stability via horizontal depressurization holes
- drain water and prevent water pressures from building up behind the pit walls
- control surface water and runoff that enters the pit
- capture precipitation and drain it away from road running surfaces and active mining areas
- remove surface water that is collected in sumps.

The pit dewatering system is designed to handle a two-year return period rain storm. Rain events in excess of this will cause the lower areas of the pit to flood. During these rare events mining will be focused on the upper mining phases until the water is pumped out of the pit bottoms. Predicted groundwater inflow to the open pit will contribute an average of 6 L/s over the life-of-mine, with one year projected at nearly 24 L/s.

The water that flows into the pit bottom will be routed to small temporary sumps created as part of normal mining practices.

The sumps will allow for some settling of solids before lower-head submersible pumps pick up the collected water and discharge it into the gravity ditches or the permanent pump-stations. When the excavation proceeds, precipitation runoff and horizontal drain flow will be collected in ditches constructed along all haul roads and on selected benches of the pit excavation and lined as required.

Sumps and pumps at appropriate locations and elevations will remove the water.

16.10 Equipment Selection

Equipment was generally selected on minimizing total cost of ownership. Operating costs were estimated by first principles using vendor-supplied productivity estimates and hourly cost information.

The selection of representative machines was based on a preliminary technical and economic assessment of feasibility level estimates provided by suppliers and their dealers.

To determine the number of equipment units required for each major fleet, productivities were calculated based on estimated annual operating hours and mechanical availability. Annual operating hours varied by fleet due to associated availabilities. An operational factor of 83.3% (50 net operating minutes per gross operating hour) was applied to all equipment to account for time spent on non-primary production tasks.

Lifetime ranges of assumed availabilities were 93 to 83% for the drilling equipment, 91 to 85% for hauling equipment, and 89 to 85% for support and auxiliary equipment.

Equipment required over the life-of-mine is tabulated in Table 16.3.

16.10.1 Haulage Equipment

On average, 31 trucks are required from Year -1 to Year 5. The annual truck requirements from Year 6 to Year 15 is 43 units, in Year 16 the requirements start dropping until only three trucks are required in Years 22 and 23.

Table 16.3 LOM Equipment Requirements

Description	Units	Year -1 (Q1)	Year -1 (Q2)	Year -1 (Q3)	Year -1 (Q4)	Year 1 (Q1)	Year 1 (Q2)	Year 1 (Q3)	Year 1 (Q4)	Year 2 (Q1)	Year 2 (Q2)	Year 2 (Q3)	Year 2 (Q4)
Installed Units													
Shovels													
Shovel – 42 m^3 bucket	no.	-	1	1	1	2	2	2	2	2	2	2	2
Subtotal Shovels		**-**	**1**	**1**	**1**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Front End Loaders													
Front End Loader – 40.5 m^3 bucket	no.	-	1	1	1	1	1	1	1	1	1	1	1
Subtotal Front End Loaders		**-**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Haul Trucks													
Haul Truck – 218 t	no.	-	8	11	12	23	23	30	33	35	35	35	35
Subtotal Haul Trucks		**-**	**8**	**11**	**12**	**23**	**23**	**30**	**33**	**35**	**35**	**35**	**35**
Drills													
Rotary Drill – 270 mm	no.	-	2	2	2	3	3	3	3	3	3	3	3
Percussion Drill – 140 mm	no.	-	1	1	1	1	1	1	1	1	1	1	1
Subtotal Drills		**-**	**3**	**3**	**3**	**4**	**3**	**4**	**4**	**4**	**4**	**4**	**4**
Subtotal Primary Equipment		**-**	**13**	**16**	**17**	**30**	**30**	**37**	**40**	**42**	**42**	**42**	**42**
Support Equipment													
Dozer – 450 kW class	no.	-	2	2	2	3	3	3	3	3	3	3	3
Grader – 24' blade	no.	-	2	2	2	2	2	2	2	2	2	2	2
Water Truck – 30 kgal	no.	-	2	2	2	2	2	2	2	2	2	2	2
Wheel Dozer – 380 kW class	no.	-	1	1	1	1	1	1	1	1	1	1	1
Excavator - 4.5m^3 bucket	no.	-	1	1	1	1	1	1	1	1	1	1	1
Excavator – w/rock hammer	no.	-	1	1	1	1	1	1	1	1	1	1	1
Dozer – 450 kW class – stacking	no.	-	-	-	-	-	-	-	-	-	-	-	-
Pipe Layer – track mounted	no.	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal Support Equipment		**-**	**9**	**9**	**9**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**
Ancillary Equipment													
General Service Truck	no.	-	3	3	3	3	3	3	3	3	3	3	3
Boom Truck – 10 t, 9 m reach	no.	-	2	2	2	2	2	2	2	2	2	2	2
Mobile Crane – 160 t	no.	-	1	1	1	1	1	1	1	1	1	1	1
Cable Reeler – FEL mounted	no.	-	1	1	1	1	1	1	1	1	1	1	1
Mobile Cable Repair Station	no.	-	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant – 20 kW	no.	-	8	8	8	8	8	8	8	8	8	8	8
Fuel and Lube Truck – 30 kl + 6 tanks	no.	-	1	1	1	1	1	1	1	1	1	1	1
Pickup Trucks	no.	-	10	10	10	10	10	10	10	10	10	10	10
Articulated Truck – 40 t class w/ejector	no.	-	1	1	1	1	1	1	1	1	1	1	1
Subtotal Ancillary Equipment		**-**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**
Total		**-**	**50**	**53**	**54**	**68**	**68**	**75**	**78**	**80**	**80**	**80**	**80**

Description	Units	Year 3 (Q1)	Year 3 (Q2)	Year 3 (Q3)	Year 3 (Q4)	Year 4 (Q1)	Year 4 (Q2)	Year 4 (Q3)	Year 4 (Q4)	Year 5 (Q1)	Year 5 (Q2	Year 5 (Q3)	Year 5 (Q4)
Installed Units													
Shovels													
Shovel – 42 m^3 bucket	no.	2	2	2	2	2	2	2	2	2	2	2	2
Subtotal Shovels		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Front End Loaders													
Front End Loader – 40.5 m^3 bucket	no.	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal Front End Loaders		**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Haul Trucks													
Haul Truck – 218 t	no.	35	35	35	35	35	35	35	35	37	37	39	40
Subtotal Haul Trucks		**35**	**35**	**35**	**35**	**35**	**35**	**35**	**35**	**37**	**37**	**39**	**40**
Drills													
Rotary Drill – 270 mm	no.	3	3	3	3	3	3	3	3	3	3	3	3
Percussion Drill – 140 mm	no.	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal Drills		**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**
Subtotal Primary Equipment		**42**	**42**	**42**	**42**	**42**	**42**	**42**	**42**	**44**	**44**	**46**	**47**
Support Equipment													
Dozer – 450 kW class	no.	3	3	3	3	3	3	3	3	3	3	3	3
Grader – 24' blade	no.	2	2	2	2	2	2	2	2	2	2	2	2
Water Truck – 30 kgal	no.	2	2	2	2	2	2	2	2	2	2	2	2
Wheel Dozer – 380 kW class	no.	1	1	1	1	1	1	1	1	1	1	1	1
Excavator - 4.5m^3 bucket	no.	1	1	1	1	1	1	1	1	1	1	1	1
Excavator – w/rock hammer	no.	1	1	1	1	1	1	1	1	1	1	1	1
Dozer – 450 kW class – stacking	no.	-	-	-	-	-	-	-	-	-	-	-	-
Pipe Layer – track mounted	no.	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal Support Equipment		**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**	**10**
Ancillary Equipment													
General Service Truck	no.	3	3	3	3	3	3	3	3	3	3	3	3
Boom Truck – 10 t, 9 m reach	no.	2	2	2	2	2	2	2	2	2	2	2	2
Mobile Crane – 160 t	no.	1	1	1	1	1	1	1	1	1	1	1	1
Cable Reeler – FEL mounted	no.	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Cable Repair Station	no.	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant – 20 kW	no.	8	8	8	8	8	8	8	8	8	8	8	8
Fuel and Lube Truck – 30 kl + 6 tanks	no.	1	1	1	1	1	1	1	1	1	1	1	1
Pickup Trucks	no.	10	10	10	10	10	10	10	10	10	10	10	10
Articulated Truck – 40 t class w/ejector	no.	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal Ancillary Equipment		**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**
Total		**80**	**80**	**80**	**80**	**80**	**80**	**80**	**80**	**82**	**82**	**84**	**85**

Description	Units	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Installed Units													
Shovels													
Shovel – 42 m^3 bucket	no.	2	2	2	2	2	2	2	2	2	2	2	2
Subtotal Shovels		**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**	**2**
Front End Loaders													
Front End Loader – 40.5 m^3 bucket	no.	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal Front End Loaders		**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Haul Trucks													
Haul Truck – 218 t	no.	43	43	43	43	43	43	43	43	43	43	16	15
Subtotal Haul Trucks		**43**	**43**	**43**	**43**	**43**	**43**	**43**	**43**	**43**	**43**	**16**	**15**
Drills													
Rotary Drill – 270 mm	no.	3	3	3	3	3	3	3	3	3	3	3	2
Percussion Drill – 140 mm	no.	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal Drills		**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**3**
Subtotal Primary Equipment		**50**	**50**	**50**	**50**	**50**	**50**	**50**	**50**	**50**	**50**	**23**	**21**
Support Equipment													
Dozer – 450 kW class	no.	3	3	2	2	2	2	2	2	2	2	1	1
Grader – 24' blade	no.	2	2	2	2	2	2	2	2	2	2	1	1
Water Truck – 30 kgal	no.	2	2	2	2	2	2	2	2	2	2	1	1
Wheel Dozer – 380 kW class	no.	1	1	1	1	1	1	1	1	1	1	1	1
Excavator - 4.5m^3 bucket	no.	1	1	1	1	1	1	1	1	1	1	1	1
Excavator – w/rock hammer	no.	1	1	1	1	1	1	1	1	1	1	1	1
Dozer – 450 kW class – stacking	no.	-	-	1	1	1	1	1	1	1	1	1	1
Pipe Layer – track mounted	no.	-	-	1	1	1	1	1	1	1	1	1	1
Subtotal Support Equipment		**10**	**10**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**11**	**8**	**8**
Ancillary Equipment													
General Service Truck	no.	3	3	3	3	3	3	3	3	3	3	3	3
Boom Truck – 10 t, 9 m reach	no.	2	2	2	2	2	2	2	2	2	2	2	2
Mobile Crane – 160 t	no.	1	1	1	1	1	1	1	1	1	1	1	1
Cable Reeler – FEL mounted	no.	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Cable Repair Station	no.	1	1	1	1	1	1	1	1	1	1	1	1
Lighting Plant – 20 kW	no.	8	8	8	8	8	8	8	8	8	8	8	8
Fuel and Lube Truck – 30 kl + 6 tanks	no.	1	1	1	1	1	1	1	1	1	1	1	1
Pickup Trucks	no.	10	10	10	10	10	10	10	10	10	10	10	10
Articulated Truck – 40 t class w/ejector	no.	1	1	1	1	1	1	1	1	1	1	1	1
Subtotal Ancillary Equipment		**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**	**28**
Total		**88**	**88**	**89**	**89**	**89**	**89**	**89**	**89**	**89**	**89**	**59**	**57**

Description	Units	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23
Installed Units							
Shovels							
Shovel – 42 m^3 bucket	no.	2	2	2	1	1	1
Subtotal Shovels		**2**	**2**	**2**	**1**	**1**	**1**
Front End Loaders							
Front End Loader – 40.5 m^3 bucket	no.	1	1	1	1	1	1
Subtotal Front End Loaders		**1**	**1**	**1**	**1**	**1**	**1**
Haul Trucks							
Haul Truck – 218 t	no.	15	15	15	9	3	3
Subtotal Haul Trucks		**15**	**15**	**15**	**9**	**3**	**3**
Drills							
Rotary Drill – 270 mm	no.	2	2	2	1	-	-
Percussion Drill – 140 mm	no.	1	1	1	1	-	-
Subtotal Drills		**3**	**3**	**3**	**2**	**-**	**-**
Subtotal Primary Equipment		**21**	**21**	**21**	**13**	**4**	**4**
Support Equipment							
Dozer – 450 kW class	no.	1	1	1	1	1	1
Grader – 24' blade	no.	1	1	1	1	1	1
Water Truck – 30 kgal	no.	1	1	1	1	1	1
Wheel Dozer – 380 kW class	no.	1	1	1	1	-	-
Excavator - 4.5m^3 bucket	no.	1	1	1	1	-	-
Excavator – w/rock hammer	no.	1	1	1	1	-	-
Dozer – 450 kW class – stacking	no.	1	1	1	1	1	1
Pipe Layer – track mounted	no.	1	1	1	1	1	1
Subtotal Support Equipment		**8**	**8**	**8**	**8**	**5**	**5**
Ancillary Equipment							
General Service Truck	no.	3	3	3	3	3	3
Boom Truck – 10 t, 9 m reach	no.	2	2	2	2	2	2
Mobile Crane – 160 t	no.	1	1	1	1	1	1
Cable Reeler – FEL mounted	no.	1	1	1	1	1	1
Mobile Cable Repair Station	no.	1	1	1	1	1	1
Lighting Plant – 20 kW	no.	8	8	8	8	8	8
Fuel and Lube Truck – 30 kl + 6 tanks	no.	1	1	1	1	1	1
Pickup Trucks	no.	10	10	10	10	10	10
Articulated Truck – 40 t class w/ejector	no.	1	1	1	1	1	1
Subtotal Ancillary Equipment		**28**	**28**	**28**	**28**	**28**	**28**
Total		**57**	**57**	**57**	**49**	**37**	**37**

16.10.2 DRILL EQUIPMENT

Five different types of drilling will be used:

1. delineation drilling to provide samples for geological modelling
2. production blast pattern drilling to fragment the rock for mining
3. frost drilling to deal with previously blasted material that has become frozen
4. horizontal drain hole drilling to prevent water pressure from building up behind the pit walls
5. vertical dewatering wells.

The vertical dewatering well and horizontal drain drilling are specialized activities and will be performed by contractors.

16.10.3 SUPPORT AND AUXILIARY EQUIPMENT

The major tasks to be completed by the support equipment include the following:

- bench and road maintenance
- reclamation support
- stockpile construction
- general maintenance
- ditch preparation and maintenance
- waste stacking support
- shovel support/clean-up.

Equipment required will be three 450 kW class dozers, two 24’ blade graders, two 30,000 g water trucks, and one each of a 380 kW class wheeled dozer, 4.5 m^3 bucket excavator, an excavator with a rock hammer, a track-mounted pipe layer, and a 40 t class articulated truck with an ejector.

16.11 MAINTENANCE CONSIDERATIONS

Maintenance on the large drills and excavators will be performed in the field, whereas equipment that can be easily driven or towed will be serviced in the truckshop. The truckshop will be near the ore stockpiles, fuel tank farm, and haul road to provide easy access for haul truck maintenance and refuelling. The shop will be located outside the 500 m flyrock clearance limit beyond the ultimate pit.

The truck shop will be a pre-fabricated structure with five heavy vehicle repair bays, lubrication and tire facilities, two light vehicle repair bay, as well as a wash bay. Rails will be embedded in the floor of some of the repair bays to prevent wear and tear of the surface from tracked equipment. Cranes will be accessible in all bays.

The building will also accommodate a high-storage rack warehouse, dry changing facility, as well as offices, training room, conference room and a lunch room.

16.12 Mining Consumable Considerations

Quotes were obtained for all major consumables from local, regional, and international suppliers for diesel fuel, power, ammonium nitrate, emulsion, blasting accessories, drill bits, tire costs and tire life, lubrication materials and filters, maintenance parts and supplies, and other equipment consumables. These quotes were used to estimate the key amounts of consumables on an annual basis.

16.13 Ore Control

Mining operations will require the ability to accurately distinguish the different ore mining areas separated by cut-off and to accurately predict the actual waste to mineralized material contact. This will be particularly important during the first six years of production where higher grades will be fed to the processing plant.

The Ore Control group will be responsible for:

- sampling, and geologic mapping of blast holes
- merging assay data with blast-hole co-ordinates
- generating short-range-planning block models
- performing ore and waste delineation
- performing mine to mill reconciliations and quality control.

Ore Control Group staffing will include 24 h coverage by a geologist and sampling technician working at the mining face to assist mine operations with ore-waste decision-making. Ore control will rely heavily on digital methods such as high-precision GPS and virtual dig maps because the ore zones are not visually discernable by field personnel. Dark and snowy conditions will also add to the reliance on digital techniques.

16.14 Emissions Control

All equipment will be maintained in good working order so as to minimize carbon dioxide (CO_2) emissions. All main equipment is designed to meet regulatory standards including the diesel that will be consumed.

All main equipment is designed to meet occupational noise criteria. In addition, the truck bodies will be heated by cycling exhaust through the body structure, which will further dampen noise inside the cab. Noise emission from excavators and haul trucks are expected to comply with permissible levels under operating conditions.

Routine water spraying by two water trucks will suppress dust generated on roads, benches, and dump areas. Non-chloride dust suppressants may be applied in high traffic areas if necessary. During winter, graders will be used to blade snow over road surfaces as required to minimize dust.

All water captured in the mine area will be diverted to the contact water pond or tailings impoundment by a system of drains and pumps. Any contained sediments or residuals from blasting products will be captured in this system. All ex-pit mining areas will be designed to minimize surface erosion.

Used oil will be collected at the truck shop and temporarily stored on site. Some of the waste oil will be used as a constituent of the fuel component in the ANFO mix; the surplus oil will be temporarily stored prior to an authorized service provider shipping the oil to permanent disposal sites. All water collection facilities surrounding the shops and fuel islands will be routed to oil/water separators for oil removal.

Any oil spills on the mine site will be dealt with by excavating the contaminated material and moving it to a remedial land farm where the soil can be remediated under controlled conditions.

The mining department at Ajax will use GPS machine guidance and a fleet management system to guide and control the mining operation on a near real-time basis.

16.15 Workforce Considerations

Staff and labour requirements for mining vary over the mine life (Table 16.4). Abacus provided guidance on staffing levels for fixed personnel, while AMEC suggested adjustments as necessary. Some personnel have site-wide responsibilities.

Table 16.4 Life-of-Mine Mining Personnel Labour Requirements

Fixed Personnel	Max
Mine Operations Department – Fixed Personnel	**17**
Mine Superintendent, Mine General Foreman, Mine Shift Supervisor, Mine Shift Supervisors, Mine Trainer, Tool Crib Attendant, Warehouse Attendants, General Mine Labourers, Trainees and Pit Dewatering Labour	
Mine Maintenance Department – Fixed Personnel	**21**
Maintenance Superintendent, Mobile Maintenance General Foreman, Mobile Maintenance Supervisor, Maintenance Scheduler, Mine Maintenance Planning GF, Mine Senior Planner, Mine Junior Planner, Light Duty Mechanics, Lube Truck Drivers, Apprentices, HV Electrical	
Engineering & Geology Department – Fixed Personnel	**19**
Chief Mining Engineer, Senior Mining Engineer, Mining Engineers, Mine Technologists, Surveyors, Geotechnical Engineer, Chief Geologist, Senior Geologist, Geologists, Sampling Technicians	
Owner's Blasting Team	**9**
Blasters, Blaster's Helpers, Blasting Technicians	
Equipment Operators	**10**
Shovel - Type 1 Operators, FEL - Type 1 Operators	
Hauling	**145**
Haul Truck - Type 1 Operators	145
Drilling	**26**
Rotary Drill - Type 1 Operators, Rotary Drill - Type 2 Operators, Percussion Drill - Type 1 Operators	
Support	**33**
Dozer - Type 1 Operators, Grader - Type 1 Operators, Water Truck - Type 1 Operators, Wheel Dozer - Type 1 Operators, Excavator1 - Type 1 Operators, Excavator2 - Type 1 Operators, Dozer MAT - Type 1 Operators, Pipe Layer - Type 1 Operators	
Equipment Maintenance Crew Loading	**4**
Shovel and FEL	
Equipment Maintenance Crew Hauling	**80**
Haul Truck	
Equipment Maintenance Crew Drilling	**5**
Rotary Drill and Percussion Drill	3
Support	**9**
Dozer, Grader, Water Truck, Wheel Dozer, Excavator1, Excavator2, Dozer MAT, Pipe Layer	
Total Mine Operators & Maintenance **	**307**
Other Personnel (Headcount Only, Costed Separately)	
Blasting Contractors	6
Material Handling Personnel	84
Total Mining Manpower	**395**

Notes: *The Maintenance Superintendent will be a shared position between the processing plant and the mine, for costing purposes a factor of 0.5 has been applied to this position salary.

** It is assumed that Ancillary Equipment is operated and maintained by fixed personnel.

Hourly and supervision labour will generally work 12 h shifts, with two shifts per day on a four-days-on/four-days-off roster rotation. Hourly labour is scheduled for 2,190 h/a, with 1.5h deducted per shift for lunch, breaks, shift change, and other non-productive time, leaving 1,916 h/a available to perform work. In general, with the exception of the blasting crew, hourly and supervisory personnel will rotate between day and night shifts, while management and technical support staff will work the day-shift only. Where personnel are required to work day shift and night shift, year round, a payroll headcount of four is required per position (two shifts per day times two for rotation).

16.16 Mining Capital Costs

All mining capital costs were included in the Xeras financial model. Capital costs are based on the following:

- Capital costs for purchases and replacements of major mining equipment are based on budgetary quotations from equipment vendors.
- AMEC's internal database was used to estimate costs for certain fleets of ancillary equipment.
- AMEC's materials handling department estimated capital costs for the in-pit and out-pit crushers, conveyors, and waste dump stacking system.
- Abacus provided costs for engineering, office, and other equipment. This equipment includes the dispatch system, a pit slope monitoring system, and all mine planning, survey, and engineering equipment and software, as well as general office equipment.

Capital costs over the LOM for the mining function are summarized in Table 16.5.

16.17 Mining Operating Costs

Operating costs were derived by applying equipment productivities, equipment hourly operating costs, labour requirements, and wages and salaries to the proposed production forecast.

The mine operating cost estimate includes costs for equipment operators, staff, contractors, and operating and maintenance supplies for each year. Operating and maintenance supply costs are based on in house data and vendor quotations, and are exclusive of taxes. Consumables quantities (fuel, explosives, tires, blasting accessories, etc.) were determined from expected unit consumption rates (hours per tonne).

Table 16.5 Mine Capital Costs

Area	Cost (US$'000)
Pre-stripping	34,443
Drilling Equipment	9,969
Loading Equipment	81,338
Hauling Equipment	181,418
Support Equipment	29,953
Mine Maintenance Equipment	13,608
Dewatering	3,609
Crushing	81,600
Conveying	84,822
Stacking	54,910
Crushing – Sustaining	8,000
Stacking – Sustaining	1,200
Engineering Equipment	6,655
In Pit Power Supply Equipment	21,000
Total	**612,519**

Xeras financial modelling software was used to calculate, on a yearly basis, the expected requirements for equipment, consumables consumption, and direct labour by combining the mine schedule with unit consumption rates, costs, and various performance inputs. The resulting life-of-mine mining operating costs are summarized in Table 16.6.

Table 16.6 Mine Operating Costs

Operating Costs	Total (US$)	Mined ($/t)
Admin / Overhead	159,594,531	0.11
Loading	188,552,137	0.11
Hauling	1,055,978,642	0.62
Drilling	161,525,069	0.09
Explosives	227,353,080	0.13
Support	141,575,992	0.08
Ancillary	40,144,184	0.02
Dewatering Allowance	28,093,378	0.01
Material Handling	249,605,029	0.15
Total	**2,252,422,043**	**1.32**

Please refer to Appendix C for AMEC's full report.

17.0 RECOVERY METHODS

17.1 INTRODUCTION

The Ajax concentrator has been designed to process a nominal 21,900,000 t/a, or 60,000 t/d, of copper-gold porphyry ore from an open pit operation. The concentrator has been designed to produce a marketable copper concentrate of 25% Cu containing about 18 g/t Au.

17.2 SUMMARY

The unit processes selected were based on the results of metallurgical testing performed at G&T, along with resources set out by Abacus. The metallurgical processing procedures selected for the design will produce a saleable high grade copper concentrate containing by-product gold. Since the economic viability of the recovery of molybdenum has not been proven to date, the recovery process for molybdenum has been excluded from the process design. The inclusion of a molybdenum recovery section into the existing concentrator in the future would be a relatively straightforward undertaking.

The treatment plant will consist of stage-wise crushing and grinding, followed by a flotation process to recover and upgrade copper from the feed material. A gravity circuit will be included within the flotation circuit to enhance gold recovery. As shown in the simplified flowsheet (Figure 17.1), the flotation concentrate will be thickened and filtered and sent to the concentrate stockpile for subsequent shipping to smelters.

The final flotation tailings will be disposed of using thickened slurry deposition into a tailings pond. This will allow for greater control of water management. Process water will be recycled from the tailings thickener overflow and will be supplemented with process water recovered from the overflow of the concentrate thickener. Fresh water will be used for gland service, reagent preparation and gravity circuit fluidisation.

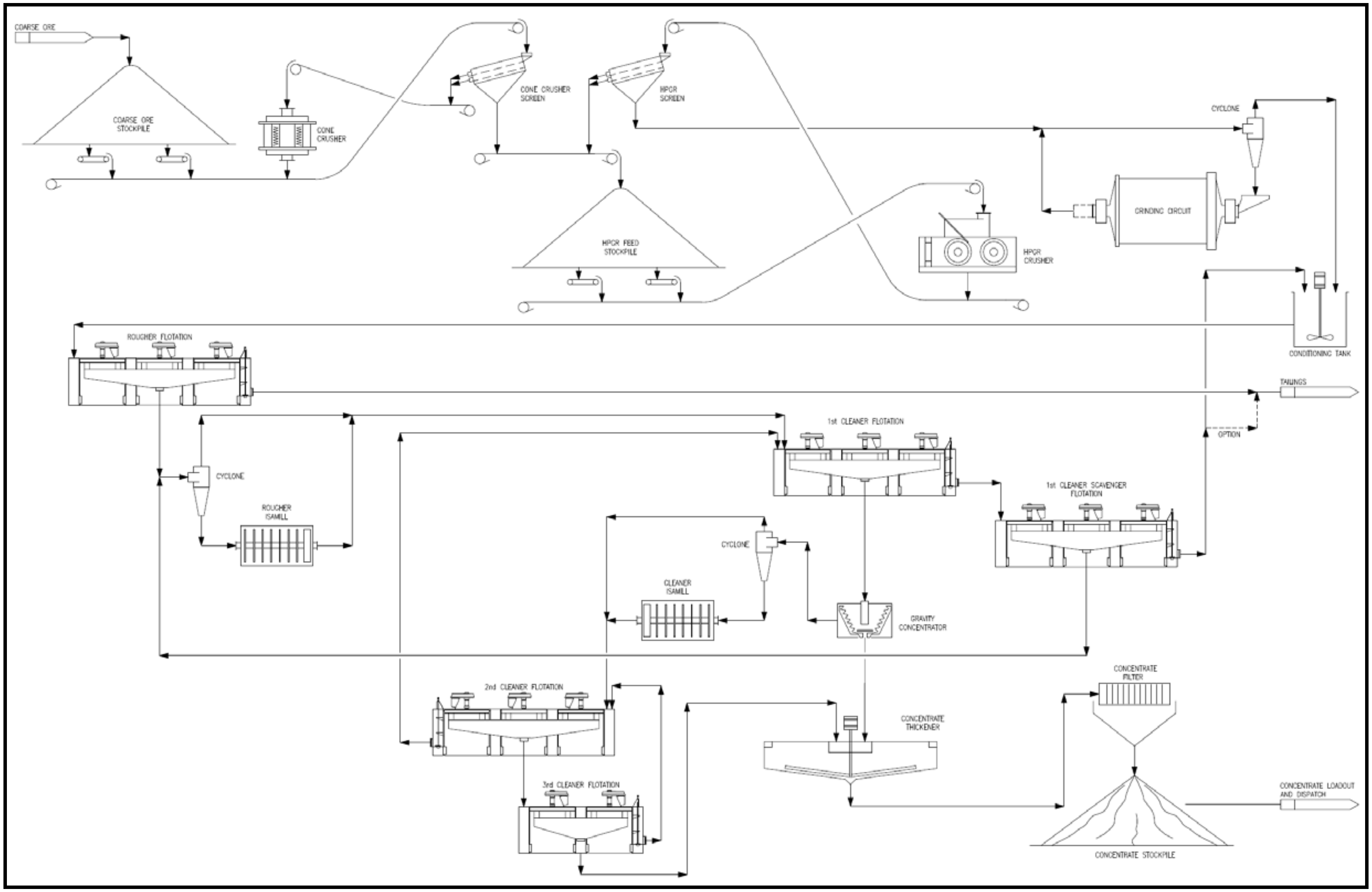
The process plant will consist of the following unit operations and facilities:

- ROM ore receiving and primary crushing (by AMEC)
- covered coarse ore stockpile
- coarse ore stockpile ore reclaim

- secondary crushing
- covered HPGR feed stockpile
- HPGR feed stockpile ore reclaim
- HPGR crushing circuit
- ball mill grinding circuit incorporating cyclones for classification
- copper rougher flotation
- copper concentrate regrinding stages
- copper cleaner flotation
- gravity gold circuit
- copper concentrate thickening, filtration, and dispatch
- tailings thickening and disposal to a tailings pond (by Golder).

The simplified flowsheet is shown in Figure 17.1. The detailed process flowsheets are located in Appendix D including the relevant process plant layout and the general arrangement drawings and the relevant process plant P&ID diagrams. Site conditions for the Project and the design criteria for mechanical and piping can be found in Appendix E.

Figure 17.1 Simplified Process Flowsheet



17.3 MAJOR DESIGN CRITERIA

The concentrator has been designed to process 60,000 t/d, equivalent to 21,900,000 t/a. The major criteria used in the design are outlined in Table 17.1. The complete design criteria are included in Appendix E. The process plant material balance and water balance are located in Appendix B.

Table 17.1 Major Design Criteria

Criteria	Unit	Value
Operating Year	d	365
Secondary Crushing Availability	%	75
HPGR Crushing Availability	%	92
Grinding and Flotation Availability	%	92
Cone Crushing Rate	t/h	2,875
HPGR Crushing Rate, per unit	t/h	2,717
Milling & Flotation Process Rate	t/h	2,717
HPGR Crusher Feed Size, 100% Passing	mm	45
HPGR Circulating Load	%	100
Ball Mill Feed Size, 80% Passing	µm	3,250
Ball Mill Product Size, 80% Passing	µm	214
Ball Mill Circulating Load	%	300
Bond Ball Mill Work Index	kWh/t	19.4
Bond Abrasion Index	g	0.238
Rougher Concentrate Regrind Size, 80% Passing	µm	60
Cleaner Concentrate Regrind Size, 80% Passing	µm	18

The design parameters are based on test work results obtained by G&T, particularly from the tests performed in 2008 using the results from Report No. KM 2228 and, more recently, Report No. KM 2688 issued in November 2010. Data from an HPGR test report conducted by Krupp Polysius Corporation and completed during 2010 was also incorporated.

The grinding mills were sized based on the Bond Work Index data for ball mills. The regrind mills were sized with assistance from the vendor.

The flotation cells were sized based on the optimum flotation times as determined during the laboratory test work. Typical flotation cell design parameters have been used in the design of the flotation circuit.

17.4 Plant Design

17.4.1 Operating Schedule and Availability

The crushing and processing plants will be designed to operate on the basis of two 12-hour shifts per day, for 365 d/a.

The cone crusher overall availability will be 75% and the HPGR crushing, ball mill grinding, and flotation circuit availabilities will be 92%. This will allow for a potential increase in crushing rate, and will allow sufficient downtime for the scheduled and unscheduled maintenance of the crushing and process plant equipment.

17.5 Process Plant Description

17.5.1 Primary Crushing (AMEC)

Primary crushing will be in-pit crushing. The mill feed will be conveyed to the covered coarse ore stockpile.

17.5.2 Coarse Ore Stockpile and Reclaim

The coarse ore stockpile is a production surge facility which will allow for a steady feed to the secondary crushing circuit.

The major equipment and facilities in this area includes:

- coarse ore stockpile, 46,600 t live capacity, covered
- reclaim apron feeders
- conveyor belts, metal detectors, self-cleaning magnets, and belt tear detectors
- belt scale
- dust collection system.

The in-pit crusher will have reduced the ROM material to 80% size of 150 mm. This material will be conveyed to the coarse ore stockpile.

The coarse ore stockpile will have a live capacity of 46,600 t. The material will be reclaimed from this stockpile by apron feeders at a nominal rate of 3,333 t/h. The apron feeders will feed a 1,828 mm wide conveyor which in turn will feed the cone crusher screen at the head of the secondary crushing circuit. The cone crusher feed conveyor belt will be equipped with a belt scale.

The coarse ore stockpile and reclaim area will be equipped with a dust collection system to control fugitive dust that will be generated during conveyor loading and the transportation of the feed material.

17.5.3 SECONDARY CRUSHING

The secondary circuit will consist of two cone crushers and will have a crushing circuit capacity of 2,875 t/h. The cone crushers will be operated in closed-circuit with sizing screens.

The major equipment and facilities in this area includes:

- cone crusher surge bin
- splitter chute
- two double-deck vibratory screens: each 3.7 m wide x 7.3 m long, 75/50 mm apertures (dry screening)
- two cone crushers, MP1000 equivalent each with 750 kW installed power
- belt feeders
- conveyor belts, metal detectors, self-cleaning magnets and belt tear detectors
- belt scale
- dust collection system.

Reclaimed material from the coarse ore stockpile will be conveyed to the secondary crushing facility. The circuit will contain a splitter chute which will dry feed two vibrating double-deck screens which will work in parallel. The screens will operate in closed circuit with two cone crushers. Screen oversize material will be directed by conveyor to the cone crusher surge bin. The cone crushers will be choke fed using belt feeders. The cone crusher product will return to the screen feed conveyor where it will combine with fresh reclaimed material prior to feeding the vibratory double deck screens. The fine screened product will be delivered to the HPGR feed stockpile by conveyor.

17.5.4 HPGR FEED STOCKPILE AND RECLAIM

The major equipment and facilities in this area includes:

- HPGR feed stockpile, 36,000 t live capacity, covered
- reclaim belt feeders
- conveyor belts, metal detectors, self-cleaning magnets and belt tear detectors

- dust collection system.

The HPGR feed stockpile will have a live capacity of 36,000 t. The feed material will be reclaimed from this stockpile by belt feeders at a nominal rate of 5,435 t/h. The belt feeders will reclaim the stockpile to feed a 1,828 mm wide conveyor which in turn feeds the HPGR feed splitter chute. The HPGR units will be the tertiary crushing circuit.

The tertiary crushing facility and the HPGR stockpile will be equipped with a dust collection system to control fugitive dust that will be generated during conveyor loading and the transportation of the crushed materials.

17.5.5 HPGR Crushing

Tertiary crushing will be done using two HPGR units to crush the material to a product size P_{80} of 3.25 mm prior to entering the grinding circuit.

The major equipment and facilities in this area includes:

- belt feeders
- two HPGR crushers: each 2,400 x 1,650 mm with two 2,600 kW motors
- HPGR screen surge bin
- two double-deck vibratory screens: 3.7 m wide x 8.54 m long, 15/6 mm apertures (wet screening).

There will be two HPGR crushers, each fed independently via a belt feeder from the HPGR feed splitter chute. The HPGR circuit will be in closed circuit with a vibrating double deck screen system. The HPGR product will be conveyed to the HPGR screen feed bin. The HPGR screens will be fed by belt feeders located under the HPGR screen feed bin. The screening of the HPGR product will be a wet process at a cut size of 6 mm. Screen oversize will be returned to the HPGR feed stockpile. Screen undersize will leave the crushing circuit as independent streams via a pipeline launder and report to the grinding circuit at a process flow rate of 2,717 t/h, or 1,359 t/h per HPGR line.

17.5.6 Grinding and Classification

The grinding circuit will consist of two parallel ball mill circuits. Grinding will be a single stage operation with the ball mills in closed circuit with the classifying cyclones. The grinding will be conducted as a wet process at a total nominal rate of 2,717 t/h, 1,359 t/h per grinding line. The grinding circuit will include:

- two ball mills, 7.93 m diameter x 12.8 m long (26 x 42 ft) each
- two cyclone feed pumpboxes

- two sets of cyclone feed slurry pumps
- two cyclone clusters
- two mass flow meters and nuclear density gauges
- two particle size analyzers
- two sampler systems.

There will be two ball mills in the grinding circuit and each ball mill will be operated independently in closed-circuit with a cyclone cluster. The HPGR product will flow by gravity to the individual grinding circuits using a pipeline launder system and will enter the grinding circuit via the cyclone feed pumpbox. The product from each ball mill will be discharged into its separate cyclone feed pumpbox combining with the respective HPGR screen discharge to constitute the cyclone feed. The slurry in each cyclone feed pumpbox will be pumped to a cyclone cluster for classification. The cut size for the cyclones will be a P_{80} of 214 µm, and the circulating load to the individual ball mill circuits will be 300% with the cyclone underflow returning to the ball mill as feed material.

The new feed to each ball mill circuit will be 1,359 t/h and the combined total of the two mills, 2,717 t/h, will constitute the feed rate to the copper flotation circuit. The ball mills will operate at a critical speed of 75%. Dilution water will be added to the grinding circuit as required.

The cyclone overflow from both classification circuits will be discharged into the respective copper flotation conditioning tanks ahead of the flotation process. The pulp density of the cyclone overflow slurry will be approximately 34% solids.

Provision will be made for the addition of lime to the cyclone feed pumpboxes for the adjustment of the pH of the slurry in the grinding circuit prior to the flotation process.

Grinding media will be added to the mills in order to maintain the grinding efficiency. Steel balls will be periodically added to each mill using a ball charging kibble.

17.5.7 FLOTATION AND REGRIND CIRCUITS

The milled pulp will be subjected to flotation to recover the targeted minerals into a high-grade copper concentrate containing gold. Tank style flotation cells will be used throughout the flotation circuit.

The copper flotation circuit will include the following equipment:

- two conditioning tanks, 6.7 m diameter x 7.3 m
- flotation reagent addition facilities
- rougher flotation tank cells, two lines of six cells, 300 m^3 each
- rougher concentrate regrind mill, Isamill, 3000 L

- cleaner concentrate regrind mill, Isamill, 1,000 L
- two classification cyclone clusters (one for each regrind stage)
- first cleaner flotation tank cells – 4 x 50 m^3 each
- first cleaner scavenger flotation tank cells – 5 x 50 m^3 each
- second cleaner flotation tank cells – 6 x 5 m^3 each
- third cleaner flotation tank cells – 4 x 5 m^3 each
- pumpboxes and standpipes
- slurry and concentrate pumps
- two particle-size analyzers, one for each regrind stage
- sampling system.

The cyclone overflows from the grinding circuit will feed the flotation circuit conditioning tanks by gravity flow from the ball mill grinding circuit cyclone clusters. The slurry will be monitored for P_{80} particle size, and flotation feed samples will be taken periodically for process control and metallurgical accounting.

The rougher flotation circuit will consist of two six-cell flotation trains which will operate in the following manner.

The cyclone overflow from each grinding circuit will discharge into a conditioning tank. Each conditioning tank will be equipped with an agitator and has been sized for a retention time, or conditioning period, of 3.5 min. The slurry will be conditioned in the conditioning tanks at the design feed rate of 1,347 t/h per line. The conditioning tanks will also receive the first cleaner scavenger tailings which will report to the conditioning tanks from the first cleaner scavenger tailings distribution box for reprocessing. Flotation reagents will be added to the conditioning tanks as defined through testing. The flotation reagents added will be the collector, potassium amyl xanthate (PAX) and the frother, methyl-isobutyl carbinol (MIBC). Provision will be made for the staged addition of the reagents in the rougher circuit as well as in the cleaner stage of the flotation circuit.

The conditioned slurry will overflow the conditioning tanks into the head end of the rougher flotation tank cell lines. Air injection will facilitate the flotation process. The copper minerals (mainly chalcopyrite) will be selectively floated into a rougher concentrate away from the other minerals and the gangue present in the slurry. The rougher concentrate will constitute approximately 7.2% mass of the plant feed. The rougher tailings will be sampled automatically prior to discharge into the final tailings pumpbox for process control and metallurgical accounting purposes. This stream will constitute the final tailings leaving the plant.

The rougher concentrate streams will be combined and form a single feed which will continue to the regrind and cleaner flotation sections of the flotation circuit for further processing.

To completely liberate the fine-sized grains of the copper minerals from the gangue constituents and to enhance upgrading of the copper concentrate, stage regrinding and cleaning will be incorporated in the cleaner flotation circuit. Two stages of regrinding and three stages of flotation cleaning have been chosen so that a final product of acceptable grade and recovery will be achieved.

The rougher concentrates will flow from the launders to the rougher Isamill cyclone feed pumpbox and will be pumped to the rougher regrind densification cyclone cluster. Also joining the rougher concentrates in the pumpbox will be the first cleaner scavenger concentrate. The rougher Isamill cyclones will be densification cyclones and will deliver the feed to the rougher Isamill at the required density of 50% solids.

The rougher regrind circuit cyclone will separate the finely ground flotation concentrate into a cyclone overflow product according to the design particle size P_{80} of 60 µm. The coarser, denser cyclone underflow will be the feed for the rougher regrind Isamill. The Isamill product will discharge the finely milled material into the rougher Isamill discharge pumpbox which will pump the finely ground product to the first cleaner feed pumpbox where it will be recombined with the rougher regrind densifying cyclone overflow prior to reporting to the head of the first cleaner flotation cells. The rougher regrind Isamill will have a design treatment rate of 179 t/h, and will have a circulating load of approximately 10% as specified by the vendor.

The rougher regrind circuit discharge will be combined with the second cleaner tailings to constitute the feed to the first cleaner flotation stage. This first cleaner stage will be directly followed by the first cleaner scavenger flotation stage. Tailings from the first cleaner scavenger flotation stage will report to the conditioning tank. Operationally, there will be the option of directing the first cleaner scavenger tailings to the final tailings pumpbox. The cleaner regrind circuit will have a similar design as the rougher regrind circuit although the design parameter in this circuit will be a particle size P_{80} of 18 µm. The cleaner regrind mill will have a design treatment rate of 29 t/h. Provision has been made to include a gravity concentration stage in the cleaner regrind circuit. The first cleaner concentrate will report to the gravity concentrator feed pumpbox and will be processed through the gravity concentrator. Tailings from the gravity concentration circuit will constitute the feed to cleaner regrind circuit.

The cleaner regrind circuit discharge will be combined with the third cleaner tailings as the feed to the second cleaner flotation stage. The second cleaner concentrate will report to the third cleaner flotation stage as flotation feed. The concentrate from the third cleaner flotation stage will be the final copper concentrate with a design copper grade of 25% copper. The copper concentrate will feed directly to the copper concentrate thickener for dewatering.

Provision will be made for the copper concentrate thickener overflow water to be re-used in the grinding and flotation circuit as process water providing this does not have a deleterious effect on the flotation of the copper and gold minerals.

17.5.8 GRAVITY CIRCUIT

The first cleaner concentrate will constitute the feed to the gravity circuit. This is an unusual application and it is based on recent changes in industry in the recovery of fine gravity gold. Typically, the gravity circuit would be installed in the cyclone underflow of the regrind circuit and would only treat a portion of the cyclone underflow. In this circuit configuration, the regrind mill will be operating in open circuit and therefore the treatment of the entire feed stream is recommended in order to enable gravity gold to be recovered. The gravity circuit feed material will be the first cleaner concentrate which will be pumped to the gravity concentrator. This gravity circuit consists of one stream, equipped with a trash screen and a centrifugal gravity concentrator. The gravity concentrator will remove particles of gold and high density gold-bearing sulphide mineral particles. The gravity tailings will be discharged into the cleaner Isamill cyclone feed pumpbox for densification ahead of the regrind mill circuit. The gravity concentrate will periodically be flushed into the gravity concentrate tank. This tank will act as a collector tank for the gravity concentrate as well as control the intermittent transfer of gravity concentrate as feed to the concentrate thickener which will be the final destination for the gravity concentrate.

The centrifugal concentrator will be equipped with an automated concentrate purging programmable logic controller (PLC) which will minimize operator intervention.

17.5.9 CONCENTRATE HANDLING

The cleaner flotation concentrate will be thickened, filtered, and stored prior to shipment to the smelter. The concentrate handling circuit will have the following equipment:

- concentrate thickener
- concentrate thickener overflow standpipe
- concentrate thickener underflow slurry pumps
- process water tank and pumps
- concentrate stock tank
- concentrate filter press feed pumps
- concentrate filter press
- filter press washing and filtrate handling equipment
- dewatered concentrate storage and dispatch facility.

The concentrate produced will be pumped from the final cleaner flotation stage to the concentrate thickener where it will be combined with the gravity concentrate in the thickener feed well. Flocculant will be added to the thickener feed to aid the settling process. The thickened concentrate will be pumped to the concentrate stock tank using thickener underflow slurry pumps. The underflow density will be 60% solids. The concentrate stock tank will be an agitated tank that will serve as the feed tank for the concentrate filter. The concentrate filter will be a filter press unit. Since filtration with a filter press unit will be a batch process, the concentrate stock tank will also act as a surge tank for the filtration operation. The filter press will dewater the concentrate to produce a final concentrate with a moisture content of about 10%. The filtrate will be returned to the concentrate thickener. The filter press solids will be discharged directly onto the concentrate stockpile. The dewatered concentrate will be stored in a designated storage facility and will periodically be loaded into trucks for dispatch off the property.

The thickener overflow solution from the concentrate thickener will be collected in the process water tank for recycling within the mill circuit. A portion of the overflow solution will be used within the filtration area as filter wash water.

17.5.10 Tailings Handling (Golder)

The flotation tailings will be the final plant tailings.

The final tailings will be pumped to a tailings pond facility for final deposition. The final plant tailings material will be initially thickened at the tailings storage facility to reduce the volume of slurry to be pumped to the tailing ponds, as well as to facilitate the recovery of process water required for re-use in the plant prior to final deposition. Water will be recovered as overflow from the tailings thickener facility for re-use in the plant as general process water.

17.5.11 Reagent Handling and Storage

The various chemical reagents will be added to the process slurry streams to facilitate the recovery of copper during the flotation process. The preparation of the various reagents will require:

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

Various chemical reagents will be added to the grinding and flotation circuit to modify the mineral particle surfaces and enhance the floatability of the valuable mineral particles into the copper-gold concentrate product. Fresh water will be used in the making up or the dilution of the various reagents that will be supplied in powder/

solids form, or which require dilution prior to the addition to the slurry. These solutions will be added to the addition points of the various flotation circuits and streams using metering pumps.

The PAX collector reagent will arrive at the plant in bulk bags and will be dumped into suitable sized hoppers for withdrawal of pre-determined quantities for mixing with water to the required solution strength of 10%. The reagent will be made up in a mixing tank, and then transferred to the holding tank, from where the solution will be pumped to the addition points in the circuit. The frother reagent, MIBC, will not be diluted and will be pumped directly from the bulk containers to the point of addition using metering pumps.

Flocculant will be prepared in the standard manner as a dilute solution with 0.30% solution strength. This will be further diluted in the thickener feed well.

Lime, as quick-lime, will be delivered in bulk and will be off-loaded pneumatically into a silo. The lime slurry will then be prepared in a lime slaking system as a 20% concentration slurry. This lime slurry will be pumped to the points of addition using a closed loop system. The valves will be controlled by pH monitors that will control the amount of lime added.

To ensure spill containment, the reagent preparation and storage facility will be located within a containment area designed to accommodate 110% of the content of the largest tank. In addition, each reagent will be prepared in its own bunded area in order to limit spillage and facilitate its return to its respective mixing tank. The storage tanks will be equipped with level indicators and instrumentation to ensure that spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, and Material Safety Data Sheet (MSDS) stations will be provided at the facility.

Each reagent line and addition point will be labelled in accordance with Workplace Hazardous Materials Information Systems (WHMIS) standards. All operational personnel will receive WHMIS training, along with additional training for the safe handling and use of the reagents.

17.5.12 *Assay and Metallurgical Laboratory*

The assay laboratory will be equipped with the necessary analytical instruments to provide all routine assays for the mine, the concentrator, and the environment departments. The most important of these instruments includes:

- fire assay equipment
- atomic absorption spectrophotometer (AAS)
- x-ray fluorescence spectrometer (XRF)
- Leco furnace.

The metallurgical laboratory will undertake all necessary test work to monitor metallurgical performance and, more importantly, to improve process flowsheet unit operations and efficiencies. The laboratory will be equipped with laboratory crushers, ball and stirred mills, particle size analysis sieves, flotation cells, filtering devices, balances, and pH meters.

17.5.13 Water Supply

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

Fresh Water Supply System

Fresh and potable water will be supplied to a fresh/fire water storage tank from the fresh water source identified as Kamloops Lake. Fresh water will be used primarily for the following:

- fire water for emergency use
- cooling water for mill motors and mill lubrication systems
- gland service for the slurry pumps
- reagent make-up
- gravity circuit fluidisation water
- potable water supply.

The fresh/fire water tank will be equipped with a standpipe which will ensure that the tank is always holding at least 40 m^3 of water, equivalent to a two hour supply of fire water.

The potable water from the fresh water source will be treated and stored in the potable water storage tank prior to delivery to various service points.

Process Water Supply System

Some process water generated in the flotation circuit as concentrate thickener overflow solution will be re-used in the process circuit via the process water tank. Reclaimed water will also be pumped from the tailings thickener overflow to the process water tank for distribution to the points of usage.

17.5.14 AIR SUPPLY

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

- low-pressure air for flotation cells will be provided by air blowers
- high-pressure air for the concentrate filter press operation will be provided by dedicated air compressors
- instrument air will come from the plant air compressors and will be dried and stored in a dedicated air receiver.

17.5.15 ON-LINE SAMPLE ANALYSIS

The plant will rely on the on-stream or in-stream analyzer for process control. The analyzer will analyze the various flotation streams in the circuit. A sufficient number of samples will be taken so that the circuit can be balanced by analytical resultant and calculation as required. Specific samples that will also be taken for metallurgical accounting purposes will be the flotation feed to the circuit, the final tailings, and the final concentrate sample; these samples will be collected on a shift-basis and will be assayed in the assay laboratory. On-stream particle size monitors will determine the P_{80} particle size of the primary cyclone overflow and the regrind circuit products.

17.6 PLANT PROCESS CONTROL

17.6.1 OVERVIEW

PLANT CONTROL

The type of plant control system used will be a Distributed Control System (DCS) that will provide equipment interlocking, process monitoring and control functions, supervisory control and an expert control system. The DCS will generate production reports and provide for data and malfunction analysis as well as a log of all process upsets. All process alarms and events will be also logged by the DCS.

Operator interface to the DCS will be via programmable computer (PC) based operator workstations located in the following control rooms of the following areas:

- primary crusher (AMEC)
- process plant including cone crushers, HPGR units, ball mills, flotation circuit and concentrate handling
- tailings area (Golder)

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

PLCs or others third party control systems supplied as part of mechanical packages will be interfaced to the plant control system via ethernet network interfaces.

Control strategies within the plant control system will be applied to control product particle size and to improve fresh ore feed tonnage through the crushing, grinding and flotation circuits. Mill solid concentration variable-ratio control, dilution water flow rate control and level control will be carried out to reach the control targets.

Further to the DCS system a future expert supervisory system may be considered to optimize process control.

Operator workstations will be capable of monitoring the entire plant site process operations, and will be capable of viewing alarms and controlling equipment within the plant. Supervisory workstations will be provided in the offices of the Mill Superintendent and the Mill Maintenance General Foreman.

Field instruments will be microprocessor-based “smart” type devices. Instruments will be grouped by process area, and wired to each respective area local field instrument junction boxes. Signal trunk cables will connect the field instrument junction boxes to DCS inlet/outlet (I/O) cabinets.

Intelligent-type motor control centres (MCCs) will be located in electrical rooms throughout the plant. A serial interface to the DCS will facilitate the MCC’s remote operation and monitoring.

17.6.2 Control Philosophy

Primary Crushing Control System (AMEC)

The control objective of the primary crushing area will be to provide a crushed product to the coarse ore stockpile. The stockpile will provide a buffer and a consistent delivery of ore to the process plant. The control philosophy for the equipment in this area will be documented by AMEC.

Process Plant Control System

Conveying operations onto the coarse ore stockpile will be monitored from a workstation located at the primary crusher. The primary crushing and process control will be documented by others. Each HPGR and cone crusher will be supplied complete with control systems, electrical systems, instrumentation, and operator stations. Control and monitoring of all crushing and conveying operations (including discharging onto the crushed ore stockpile) will be conducted from this location. Control and monitoring functions will include:

- plugged chute detection at all transfer points

- zero speed switches, side travel switches, emergency pull cords, and belt rip detection of all conveyors
- weightometers on selected conveyors to monitor feed rates and quantities
- equipment bearing temperatures and lubrication system status
- vendors' instrumentation packages.

An interface utilizing an industrial network protocol will be installed to tie the primary crusher control systems to the main plant control system. This interface will allow for collecting of data and general monitoring the primary crushing operation.

A central control room in the mill building will be provided with two Object-Oriented Input Systems (OISs). Control and monitoring of all processes in the mill building, secondary crushing, and remote ancillary areas will be conducted from this location.

The PC workstations will control and monitor the following:

- cone crushers (speed, bearing temperatures, lubrication systems, motors, screens, and feed rates)
- HPGR crushers (speed, bearing temperatures, lubrication systems, motors, screens, and feed rates)
- ball mills (mill speed, bearing temperatures, lubrication systems, clutches, motors, and feed rates)
- particle size monitors (for grinding optimization)
- pumpboxes, tanks, and bin levels
- variable speed pumps
- cyclone feed density controls
- flotation cells (level controls, reagent addition measurements, and airflow rates)
- regrind mills (power input, bearing temperatures, lubrication systems, motors, density monitors, level transmitters and feed rates)
- samplers and x-ray analyzer (for flotation optimization)
- thickeners (drives, slurry interface levels, underflow density, and flocculant addition)
- concentrate pressure filters and load out
- reagent handling and distribution systems
- air compressors
- air blowers
- fuel storage (via radio linked remote I/O)

- vendors' instrumentation packages.

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

An on-line dispersive x-ray fluorescence analyzer will be used to monitor the performance of the flotation process in various streams in order to optimize concentrate grade and metal recoveries.

A particle size-based computer control systems will be used to maintain the optimum grind sizes for the primary grinding and concentrate regrinding circuits. Particle-size analyzers will provide the main inputs to the control system.

TAILINGS FACILITY CONTROL SYSTEM (GOLDER)

The control objective of the tailings facility will be to dispose of thickened tailings and will include water storage, reclamation, and distribution back to the process plant. The control philosophy for the equipment in this area will be documented by Golder.

REMOTE MONITORING

Closed circuit television (CCTV) cameras will be installed at various locations throughout the plant, including the stockpile conveyor discharge point, the stockpile reclaim area, the cone crusher and HPGR crushing areas, the ball mill grinding area, the flotation area, and the concentrate handling building. The cameras will be monitored from the plant control rooms.

17.6.3 COMMUNICATIONS

OVERVIEW

Site wide communications will design will incorporate proven, reliable and state-of-the-art systems to ensure that personnel at the mine site have adequate voice, data and other communication channels available.

The communication systems will include a Voice-over Internet Protocol (VoIP) telephone system which utilizes the plant wide fibre optic network.

Hand-held mobile and base radios will be used by operating and maintenance personnel.

Internet connection and phone service external to the plant will be sourced via a local telephone company or internet service provider.

18.0 PROJECT INFRASTRUCTURE

The property associated with the Ajax Project has a number of previously-developed infrastructure items, services, and facilities, which are currently utilized for ongoing exploration. The facilities and infrastructure for the Project will include:

- access and site roads
- fresh water supply and sewage disposal
- TSF
- power supply and distribution
- structural
- communications
- auxiliary infrastructures.

18.1 ACCESS AND SITE ROADS

Wardrop completed the access and site roads design based on BGC's geotechnical assessment (refer to Appendix F). The geometric design meets Transportation Association of Canada (TAC) RCU60 or equivalent. The civil design criteria is provided in Appendix E.

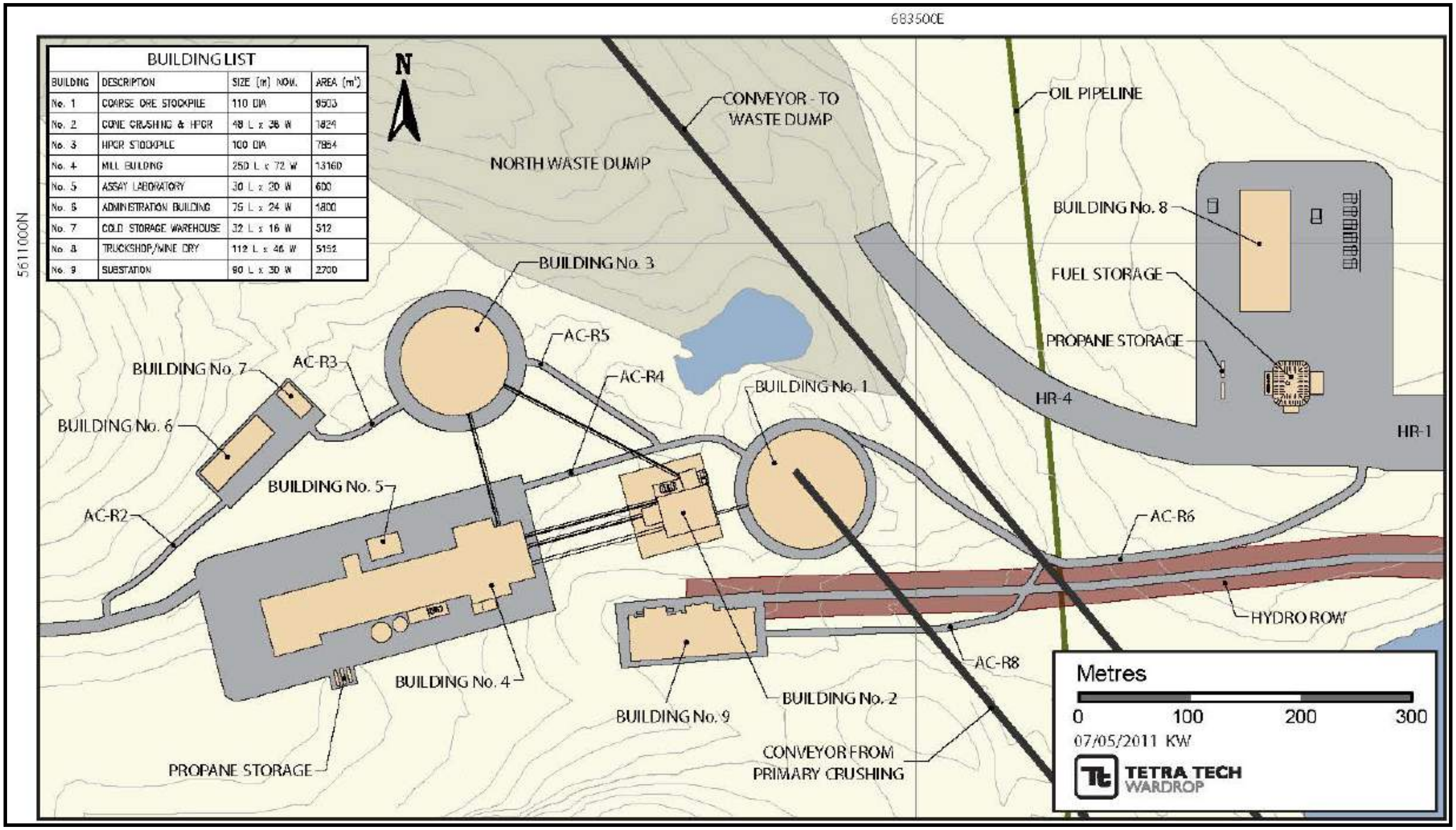
Figure 18.1 shows the proposed access roads AC-R1 to AC-R7 and haul roads HR 1 to HR 4. The road section layouts can be found in Appendix D.

The new all-weather 2.9 km long x 12.0 m wide access road, AC-R1, leads to Lac Le Jeune Road, which connects from Highway 1 via Copperhead Drive, west of Kamloops. The road AC-R1 will be used by concentrate haul trucks and service vehicles during the operations phase. The road is a private, single-lane road with passing bays. A remote-control gate located at the entrance to the access road will provide access to the mine site during mine operations.

Access roads AC-R2 to AC-R7 will connect the various on-site buildings. Haul roads HR 1 to HR 4 are designed for the traffic between the pit phases and the ore crusher, waste dumps, overburden stockpiles, construction areas, and truck shop.

Competent waste rock will be used for road base and capping material for both access and site roads. Road travelling surfaces, safety berms, and drainage channels will be regularly maintained.

Figure 18.1 General Arrangement Plan



18.2 FRESH WATER SUPPLY

Fresh water will be used for process commissioning and start-up, gland seal service for slurry pumps, reagent preparation, process water makeup, and fire water. Kamloops Lake will be the fresh water source for the Project.

Wardrop investigated several fresh water supply options to maximize the utilisation of the existing facilities. Wardrop presented its recommended option in the report "Ajax Copper/Gold Project, Kamloops, British Columbia – Fresh Water Supply Study Report, November 2010" (see Appendix G). A brief summary is presented in this section.

On April 8, 2010 Wardrop visited the existing water supply facilities, which had been used by the old Afton Mine. A hydraulic analysis using PIPE-FLO was subsequently conducted for the system at the designed water volume and pressure. The designed water flow rate is 2,000 m^3 (see Table 18.1). Approximately 1,688 m^3/h water was established as the design requirement of the Project; the remaining 312 m^3/h water will report to a nearby mine site owned by New Gold Inc.

Table 18.1 Water Flow Rates

Item	Description	Flow Rate (m^3/hr)	Safety Factor (%)	Design Flow Rate (m^3/hr)
1	Water to New Gold Tank	260	20	312
2	Water to Tailings Thickener	10	20	12
3	Water to Fresh Water Pond (plant site)	1397	20	1,676
4	**Total Water from Lake**	**1,667**	**20**	**2,000**

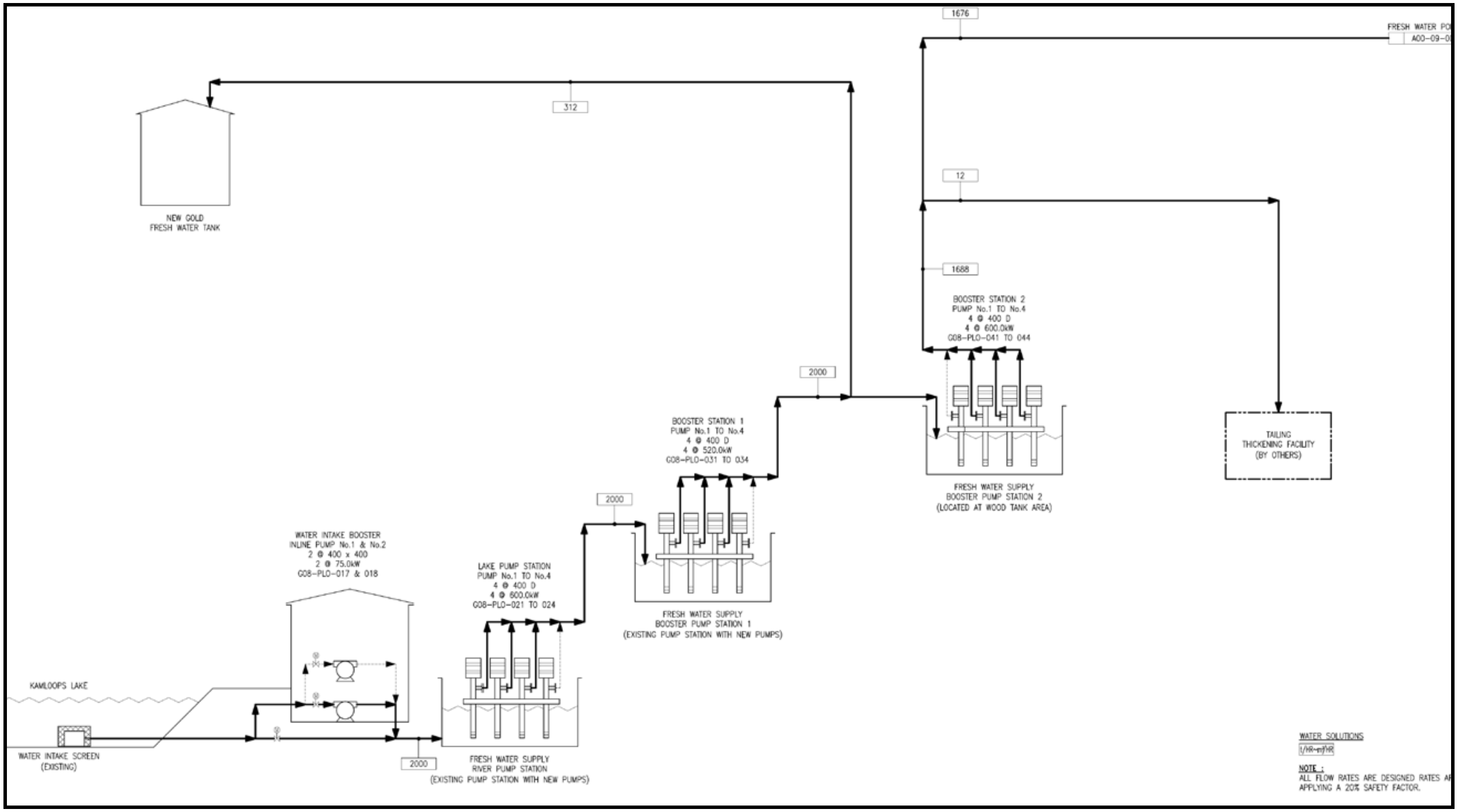
Wardrop recommends a multiple-stage pumping system water supply option with following items (Figure 18.2):

- a new water intake booster pump station
- the existing lake pump station (with new pumps added)
- the existing Booster Pump Station No. 1 (with new pumps added)
- booster Pump Station No. 2 to be installed.

Fresh water will be pumped through multiple pumping stages to a fresh/fire water tank at the proposed plant site. The relevant water distribution is described in Section 17.5.13.

Wardrop recommends a detailed inspection of the existing pipeline to assess the working conditions, pipe wall thickness, and pressure ratings. The current water supply system is based on a total flow rate of 2,000 m^3/h; however, this value might be reduced in the context of the reclaimed water amount estimated by Golder (Section 18.4). Accordingly, if the fresh water amount changes, Wardrop recommends an optimization of the water supply, including hydro analysis and pump quantities.

Figure 18.2 Fresh Water Supply Process Flowsheet



A potable water tank and hydro-chlorination system will be provided at the plant site.

The sewage treatment plant will be a pre-packaged Rotating Biological Contactor (RBC). The plant will be manufactured off-site and containerized for simple connection to the collection system on site. Once treated, the treated sewage effluent will be discharged into the environment in accordance with the requirements of the Environmental Impact Assessment (EIA).

18.3 GEOTECHNICAL CONDITIONS

BGC has completed geotechnical investigations for various components of mine site infrastructure for the proposed Ajax mine in Kamloops, BC (refer to Appendix F).

The proposed mine development will involve the open pit, two waste rock storage areas, tailing management facilities, process plant, crushers, explosives storage, and access road. BGC's geotechnical data report (refer to Appendix F) and foundation design report (refer to Appendix F) documents geotechnical engineering recommendations for foundations associated with the plant site and truck shop area, explosives storage area, ore crusher, and access road. Geotechnical investigations for the tailings management facilities are the responsibility of others.

The project area contains a variably thick till blanket, up to more than 35 m thick, over slightly to moderately weathered igneous bedrock. The till varies widely in grain size, density and thickness, but is typically relatively strong, comprising either stiff to very stiff silty glacial till or dense to very dense sandy or gravelly glacial till.

Heavy static or vibratory loads should be founded on bedrock, which is found at relatively shallow depths of up to about 7 m (average of 2.8 m) below existing ground surface in the area of the proposed plant site. Bedrock is also present close to existing ground surface at the location of the proposed ore crusher. Lightly to moderately loaded foundations may be placed on till or structural fill.

Access restrictions associated with the requirements for archaeological clearances limited the geotechnical investigation program. Additional investigation is required in specific areas, including: the truck shop area, where no subsurface data are available; the access road alignment, where typical conditions have been inferred from test holes in the general area, a possible borrow source for structural fill; and, the plant site area, where the presence of faults near proposed foundation grades requires further study.

18.4 TAILINGS STORAGE FACILITY

In 2010, Golder Paste Technology Ltd., in collaboration with Golder Associates Ltd. (collectively called "Golder"), carried out a tailings disposal alternatives study for Ajax which considered the disposal of slurry, high density tailings and paste in either the existing disposal area or in a new facility.

The Golder Prefeasibility Study (Project Ref# 09-1900-0096) recommended a thickened, non-segregating tailings slurry discharge into a new TSF.

The new location of the TSF, approximately 3.5 km west of the open pit in a north-south orientation between the Lac Le Jeune Road and Highway 5, was selected based on available land, proximity to the mill, ease of water management and meeting the tailings storage requirement. The TSF main water collection pond will be located in the Inks Lake area, to achieve a compact site footprint, and for its ability to support runoff from the deposition site, water diversions and seepage from the TSF, relatively close to the mill, isolation from Jacko Lake, and reasonable constructability. In addition, according to information provided by Abacus, it is also understood that Inks Lake has no fish habitat.

To produce the non-segregating tailings slurry, a TTP will be located at a high elevation, north of the TSF, to reduce pumping power and required energy consumption. The proposed location of the TTP and its location relative to the TSF will influence its design.

In June 2010, Abacus retained Golder to assist in the development of the current Prefeasibility Tailings Management design, including the new TSF, TSF water management, and TTP, to a feasibility level (refer to Appendix G).

The capacity of the TTP and TSF are dictated by the throughput of the processing facility. Abacus provided Golder with the tailings production rate and slurry density. Please refer to Table 18.2 below.

Table 18.2 TTP Feed Rates

Description	Value
Mine measured and indicated ore reserve (t)	580,000,000
Ore daily production - nominal (t/d)	60,000
LOM (a)	26.5
Mill tailings hourly production - design (t/hr)	2693.74
Mill tailings solids SG	2.76
Mill tailings slurry density (wt% solids)	34.31

Golder performed laboratory test work to provide information on the dewatering and rheological properties of the tailings. This information, combined with those previously mentioned, i.e. processing facility throughput and TSF location and footprint, completed the design basis of the TTP. Please refer to Table 18.3 below.

Table 18.3 TTP Design Criteria Summary

Description	Value
Thickener underflow (wt% solids)	70
Flocculant dosage (g/t)	30
Mill tailings pipeline length (m)	4,500
Discharge thickened tailings pipeline length (m)	3,800
Excess water pipeline length (m)	4,500
TTP elevation (masl)	930
Excess water discharge elevation at the processing facility (masl)	929.85
Max thickened tailings discharge elevation (masl)	1,022

The design of the TTP also includes a surrounding containment wall that will reduce the environmental impact in the event of a major plant failure and tailings spillage.

The TTP layout has been designed based on the installation and service of two freestanding high-density 39 m diameter thickeners, six positive displacement (PD) pumps and their ancillary equipment, including:

- flocculant and compressed air systems
- a crane
- four clean-up pumps
- four thickener underflow pumps
- two charging pumps to service the PD, gland, fresh and process water pumps
- one high pressure pump to provide flushing water to the thickened tailings discharge pipeline
- two excess water pumps to return thickener overflow and TSF reclaim water to the processing facility.

It also includes a fresh water and a process water tank, an agitated thickened tailings tank, and a tailings feed box that splits the mill tailings flow rate into two streams to feed both tailings thickeners.

The TTP will also have its own electrical room, control room, tool room and washroom. Adjacent to the TTP there will be two oil step down transformers and one emergency generator. A 25,000 V supply will be available. The plant medium voltage will be 4,160 V, 3-phase and 60 Hz, the plant low voltage will be 600 V, 3-phase and 60 Hz, and the control voltage will be 120 V, 1-phase and 60 Hz. The TTP is estimated to present a total connected load of 13,715 kW.

Also, remote from the TTP, there will be two pump stations to service the TSF water collection system and return the collected water to the TTP. Each pump station will house one step down transformer and two TSF Pond pumps.

In addition to this equipment, the processing facility will house three mill tailings pumps to transport the mill tailings to the TTP.

The TTP process flowsheet can be described as follows: mill tailings, diluted flocculant, and any thickener underflow returns, will be combined in the feed box which will feed two tailings thickeners by gravity.

- Both thickener underflow streams will be pumped to thickened tailings tank in order to homogenize the underflow from both thickeners and then pump it to the TSF by five PD pumps, which will be fed by the charging pumps.
- Thickener overflow water and TSF reclaim water will be piped to the process water tank for use as flocculant dilution, discharge pipeline flushing, plant clean-up, and the excess water will be pumped to the processing facility for its re-use in the milling process.

Based on the design criteria detailed above, and the assumption that the deposited thickened tailings have a void ratio of 0.7, the ultimate deposited thickened tailings dry density is 1.62 t/m^3 and the total required TSF volume is 389.6 Mm^3.

The main elements of the TSF will include thickened tailings stack, TSF water collection system, collection ditches, TSF underdrain, and a TSF pond emergency spillway. Water management of the TSF will include water collection ditches around the TSF for collection of seepage and runoff, and the TSF ponds.

During the prefeasibility study, one TSF pond was identified to collect water from the TSF and run-off from the natural ground catchment area. This single-water collection pond was located at the east side of the TSF. During the development of the feasibility study, a second water collection pond was introduced to the system to aid in the design of a zero surface water discharge to the environment.

The second water collection pond (the TSF West Pond), will collect water seepage from the northwest and southwest ditches, while the original water collection pond (the TSF East Pond), will collect water seepage from the north east and south east ditches.

During the operation phase of the Ajax Project, water will be withdrawn from the TSF pond as make-up water to operate the processing plant.

The crest elevations of both TSF pond dykes were established to provide sufficient storage capacity to safely contain the Probable Maximum Precipitation (PMP) event.

The TSF will be constructed in an upstream manner and thickened tailings will be discharged from spigot points positioned along the west side of the facility to form a gentle slope toward the east side of the facility, 2 to 3%, to minimize erosion from run-off while preventing ponding on the facility. Waste rock from the open pit development will be used for several purposes, for example, to construct an initial pipeline dyke to facilitate tailings discharge and to raise the tailings pile in stages to allow more tailings to be deposited within the TSF. The maximum elevation of the tailings would be at about 1,022 masl along the west side, which will be approximately 92 m above the proposed TTP elevation.

At closure, the reclamation of the tailings stack will consist of a closure dry cover, which will be placed over the entire tailings surface area. The proposed closure cover will minimize wind and water erosion, and reduce infiltration into the TSF, eventually reducing seepage from the toe of the TSF. The TSF Pond will be decommissioned once discharge requirements are met and a stable trend is attained. The TSF Pond dyke will then be breached to form a wetland, and the remaining TSF Pond dyke will be re-graded.

Knight Piésold carried out testing to confirm the acid generation potential of the ore and mine waste rock as part of the feasibility study. The results will be presented in a report by Knight Piésold. For the purpose of this study non acid-generating (NAG) tailings and mine waste rock were assumed.

Requests for quotations were prepared for the major process, mechanical and electrical equipment in order to obtain vendor budget quotes. Vendor quotes were received by Golder and bid analyses were produced for each of the packages.

Based on selected vendor quotes, Abacus' labour rates and Golder database and experience, capital, operating and sustainable capital costs estimates (+/-15%) were prepared for the TTP, TSF and water collection system. Summaries of the capital and operating cost estimates are provided in Table 18.4 and Table 18.5, respectively.

Table 18.4 TTP/TSF/Water Collection Capital Cost Estimate Summary (±15%)

Item Description	Material/ Equipment Total (US$)	Labour Total (US$)	Delivery Costs (US$)	Total (US$)
Mechanical Equipment	21,222,916	1,872,030	420,787	23,515,733
Electrical Equipment	6,949,676	688,024	54,975	7,692,674
Civil/Structural/Architectural	7,647,475	(included in material)	71,318	7,718,794
Non Equipment Direct Costs	1,820,922	(included in material)	20,801	1,880,823
External Pipeline	22,701,800	(included in material)	635,922	23,337,722
TSF	15,348,197	(included in material)	(included in material)	15,348,197
Total Direct	**-**	**-**	**-**	**79,493,944**
Indirect	-	-	-	9,250,600
Contingency (20%)	-	-	-	17,748,909
Total TTP	**-**	**-**	**-**	**106,493,452**

Table 18.5 TTP/TSF/Water Collection Annual Operating Cost Estimate (±15%)

Description	Cost (US$)
Average Plant Power Cost (Years 1 to 4)	1,054,100
Average Plant Power Cost (Years 5 to 9)	1,102,300
Average Plant Power Cost (Years 10 to 14)	1,169,500
Average Plant Power Cost (Years 15 to 19)	1,255,500
Average Plant Power Cost (Years 20 to 25)	1,357,400
Average Plant Power Cost (Year 26.5)	1,461,400
Plant Maintenance	1,288,700
Consumables (flocculant, lube, etc.)	2,861,700
System Manpower	567,100
TSF Operation	652,600
Years 1 to 4	
Total Average Annual Operating Cost	6,424,200
Average Operating Cost (per dry metric ton placed)	0.30
Years 5 to 9	
Total Average Annual Operating Cost	6,472,400
Average Operating Cost (per dry metric ton placed)	0.30
Years 10 to 14	
Total Average Annual Operating Cost	6,539,600
Average Operating Cost (per dry tonne placed)	0.30
Years 15 to 19	
Total Average Annual Operating Cost	6,625,600
Average Operating Cost (per dry metric ton placed)	0.31
Years 20 to 25	
Total Average Annual Operating Cost	6,727,500
Average Operating Cost (per dry metric ton placed)	0.31
Year 26.5 (max)	
Total Average Annual Operating Cost	6,831,500
Average Operating Cost (per dry metric ton placed)	0.31

A summary of the sustainable capital cost estimate is provided in Table 18.6 below:

Table 18.6 TTP/TSF/Water Collection Sustainable Capital Cost Estimate (±15%)

Description	Cost (US$)
Total Average Annual Cost (Years 1 to 4)	2,461,643
Total Average Annual Cost (Years 5 to 9)	3,273,098
Total Average Annual Cost (Years 10 to 14)	4,153,498
Total Average Annual Cost (Years 15 to 19)	4,783,354
Total Average Annual Cost (Years 20 to 25.2)	5,083,459

18.5 WATER MANAGEMENT

Wardrop summarized the overall storm water/fresh water supply systems and water balance based on information from the following consultants:

- Golder provided specialty services for the design of the TTP, TSF, and TSF water management.
- BGC provided open pit dewatering flow rate, water management in the North and East waste dumps.
- Knight Piésold provided hydrometeorological reports and data.
- Wardrop quantified the storm water for the plant site, truck shop, and explosives area and compiling the results of the other consultants" reports.

The details of the water management report can be found in Appendix G.

Water will be required for use as make-up water for the process and plant facilities, potable water, explosives mixing, fire protection, dust control, etc. This water can be from a variety on contact/recycled water sources.

Fresh (non-contact) water will be required for reagents preparation, gland sealing water, potable, process water make up and for firefighting purpose. Fresh water will be pumped from Thompson River and be stored in a fresh/fire water tank with a size of 14 m diameter x 14 m high. From this tank fresh water will be distributed to different applications. A gland sealing water tank of 6 m diameter x 6 m high has been designed. Recycled water (contact water) will be used to make up the water consumed in processing.

All water collected/recycled/used on the project will require containment or storage in man-made structures.

Contact water will be directed to and held in one of the following:

- West pond
- Central pond
- process water tank (19 m diameter x 19 m high)

Water, fresh and contact will be available in varying quantities from the following sources:

- Fresh water via a pipeline from Kamloops Lake pump station. A separate report by Wardrop will present fresh water supply details.
- Reclaim water from tailings thickener overflow pumped directly to the mill process water tank.

- Reclaim water from plant site concentrate thickener will be pumped directly to the mill process water tank.
- Reclaim water from the TSF pond including run-off and tailings seepage water (zero flow for the purposes of this report) from the TSF and associated areas will be pumped to tailings thickener plant process water tank and pumped back to processing plant; released as required.
- Pit groundwater inflow collection will be pumped to the central pond.
- Pit runoff will be pumped to the central pond.

Run-off redirected from other facilities to the appropriate storage area:

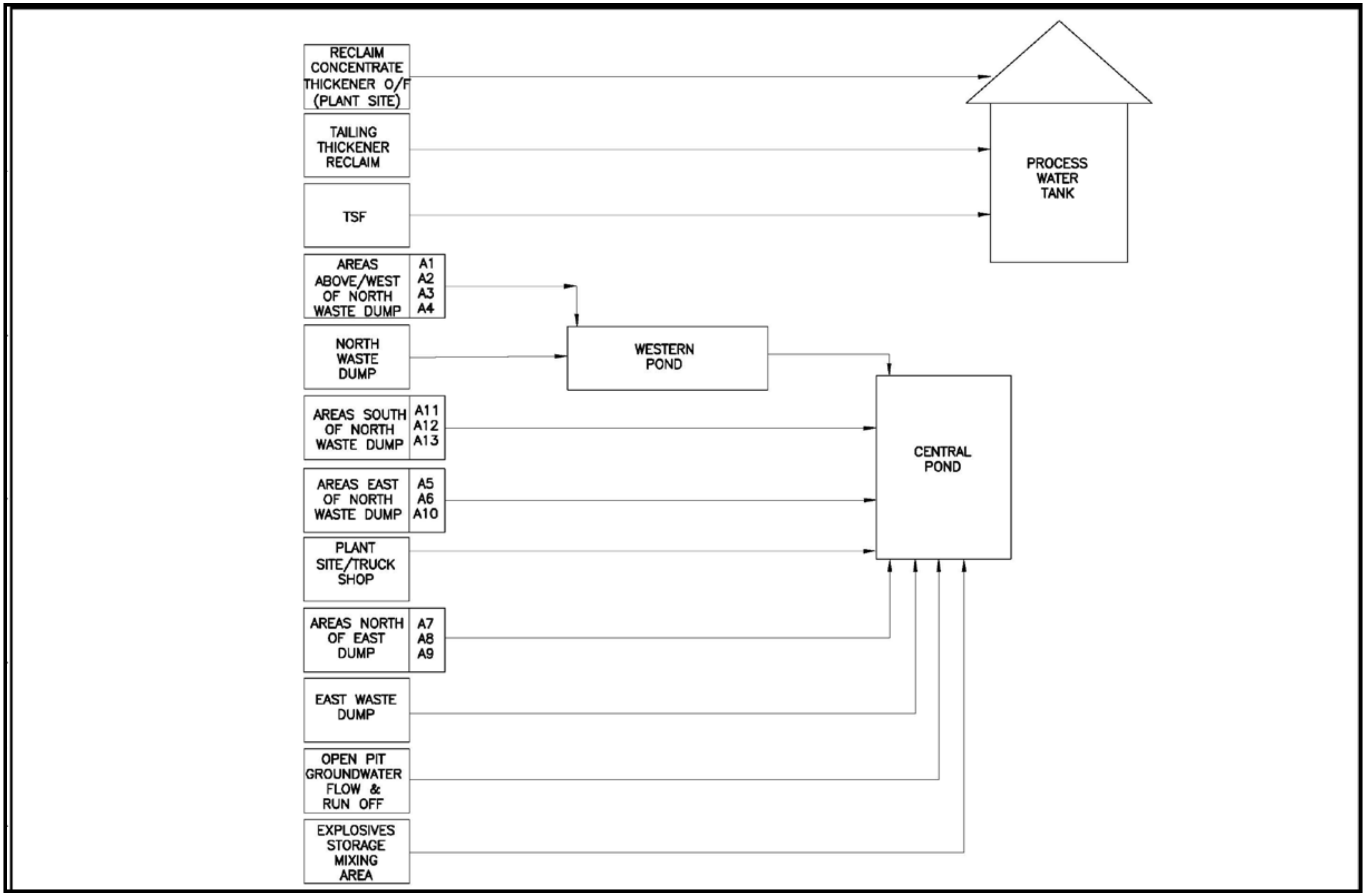
- waste rock dumps (to West and Central ponds)
- plant site (to Central pond)
- surrounding contributing areas (to West and Central ponds)
- explosives storage/mixing area (to Central pond).

In addition, evaporation from surface water storage areas has to be taken into account. Evaporation from the TSF has been included in the Golder TSF water balance/report.

Figure 18.3 shows the storm water catchment schematic. The following figures were obtained from the Knight Piésold report (2009):

- 100 year Dry Year 127 mm
- Average Year 223 mm
- 100 year Wet Year 319 mm

Figure 18.3 Stormwater Catchment Schematic



18.6 POWER SUPPLY AND DISTRIBUTION

18.6.1 230 KV UTILITY POWER SUPPLY AND 230 KV SITE SUBSTATION

The design criteria for the power supply and distribution can be found in Appendix E.

Power will be provided from a new tap point on existing BC Hydro 230 kV transmission line 2L265. This new tap point and approximately 9 km of 230 kV wooden pole structured overhead line will approach the site 230 kV step down substation from the east.

The fenced site 230 kV substation will contain an incoming structure and isolation switch, main circuit breaker, provision for utility metering, bus work to deliver 230 kV power to two pairs of step down transformers (four total), each complete with primary circuit breaker and isolating switches. One set of transformers and its associated secondary switchgear is arranged to provide 25 kV power via overhead lines to the pit, rock handling and tailings areas. The other set of transformers and its associated secondary switchgear is arranged to provide 13.8 kV power to the main processing plant and other local facilities. Provision is included for automatically switched capacitor banks to assist with site power factor correction at the 13.8 kV level within the substation yard, however, during normal operation, the majority of power factor correction will be provided by the four ball mill synchronous motors.

Three outdoor rated electrical rooms will house the 25 kV switchgear, the 13.8 kV switchgear and substation control and communications equipment.

Please refer to Appendix D for the electrical single-line diagrams.

18.6.2 SITE POWER DISTRIBUTION

25 KV

Overhead power lines will supply 25 kV power to in-pit, rock handling and tailings equipment and infrastructure.

13.8 KV

13.8 kV power will be delivered via underground cables to a switchgear line in the mill building for each of the two ball mill dual drive motors, and to power distribution step down transformers at the mill building and at the HPGR building.

4 kV

In the mill building, a secondary selective 4 kV motor control centre will deliver power to large mill process motors, and to large process-adjustable speed drives for various process and tailings pumps.

In the HPGR building, a single secondary selective 4 kV motor control centre will deliver power to the two sets of HPGR dual adjustable speed drives, and to two 600 V secondary step down transformers.

600 V

In the mill building, two pairs of secondary selective 600 V power distribution centres (four total) will deliver power to the mill and ancillary area 600 V motor control centres.

In the HPGR building, a single secondary selective 600 V power distribution centre will deliver power to the HPGR area mill motor control centres, and to coarse ore storage loads.

Please refer to Appendix D for the electrical single-line diagrams.

18.6.3 Emergency Power

A single 1,000 KW 600 V standby rated diesel generator set is provided at the mill building to provide standby power to mill building and HPGR building critical power loads identified as critical on the project equipment list.

18.6.4 Fresh Water System Power Distribution

Power for the fresh water supply pumping system will be provided via overhead line from the New Gold mine 13.8 kV system. The three pumping locations – an upgraded lake intake station, an upgraded booster Station No. 1, and a new booster Station No. 2 – will each contain three 4 kV pump motors (two to operate) and associated ancillary loads and control equipment.

At each station, motor soft starting equipment is provided for the pumps to mitigate flicker on the New Gold power system and minimize voltage drop on the overhead line.

18.7 STRUCTURAL

18.7.1 STRUCTURAL DESIGN BASIS

The structural design criteria outlining cost estimation parameters of structures and facilities for the Project are provided in Appendix E. Foundation preliminary designs are based on BGC's foundation report, available in Appendix F.

All exposed subgrades will be reviewed by a qualified geotechnical engineer prior to placement of concrete, to verify that the subgrade conditions are consistent with those assumed.

Conventional positive drainage away from structural foundations will be created both during construction and after completion.

During construction, excavated ground to the underside of footings should be cleaned and loose material removed prior to steel reinforcement and concrete placement.

Structural quantities for the following areas have been estimated based on the current layout drawings, preliminary structural engineering design and sketches:

- mill building
- stockpile cover and tunnels
- HPGR and secondary crushing building
- administration building
- shop and warehouse maintenance facilitycold storage warehouse
- conveying.

There was no allowance made for over-poured or wastage in the concrete quantities take-off.

All concrete is based on a 28-day compressive strength of 30 MPa.

Concrete unit rates include the cost for formwork, reinforcing steel, placement, and finishing of concrete.

Costing is based on ready-mix concrete.

Excavation quantities are taken to rough grade or underside of foundation.

18.7.2 Structural Descriptions

Mill Building

The mill building will be a conventional stick-built steel structure with insulated steel roof deck and insulated wall cladding. Structural drawings can be found in Appendix D.

Overhead cranes supported off the mill building include a 50-ton crane and two 25-ton cranes operating over the ball mill areas, a combined 50/25-ton crane over the flotation area and a 10-ton crane over the thickener area.

Major equipment will be supported on steel platforms complete with steel grating and handrails (e.g. cyclones, samplers, analyzer, screens, etc.).

These multi-level steel platforms provide services for ongoing operation and maintenance needs. Several means of egress and staircases are also provided.

The building will house modular prefabricated electrical rooms, control rooms, change rooms and offices.

Mat concrete foundations and piers were estimated for ball mills. Conventional spread footings were estimated for the mill building foundation, including perimeter grade beam, slab-on-grade and drainage sumps. Interior footings for various structures are also included.

Flotation cells were estimated to be supported on a concrete mat foundation, complete with raised pedestals to accommodate various cell elevations. Smaller cells will be supported on elevated steel platforms.

Reinforced concrete retaining walls are provided for the concentrate load out area. A truck scale pit is also provided.

Stockpile Cover and Tunnels

The coarse ore and HPGR ore stock pile covers are large dome structures supported on reinforced concrete ring footing. Slab-on-grade is not provided for the stock pile. Main stockpile reclaim tunnels are heavy reinforced concrete structures with elevated steel platforms designed to support apron feeders. These platforms include steel grating, handrails and staircases.

Each stockpile has an escape tunnel constructed from a corrugated steel culvert complete with an escape stair tower to the ground surface.

Structural drawings for stockpiles and reclaim tunnels can be found in Appendix D.

HPGR and Secondary Crushing Building

The HPGR and Secondary Crushing building will be a pre-engineered structure with insulated steel roof deck and insulated wall cladding panels. Structural drawings for the HPGR and Secondary Crushing building can be found in Appendix D.

A 120-ton and 25-ton overhead crane is included to be supported off the main building columns. Interior steel platforms on multiple levels are provided for ongoing operation and maintenance needs. Several means of egress and staircases are also provided.

Equipment will be supported on independent steel platforms, complete with steel grating and handrails.

The HPGR and cone crushers will be supported on heavy concrete mat foundation with reinforced concrete piers.

The HPGR and Secondary Crushing building will be supported on isolated spread footings complete with perimeter grade beams. Building slab-on-grade will be included. Interior footings to support various structures and equipment are also included.

The HPGR and secondary crushing surge bins will be independently-supported. A main stair tower providing access to the top level of the bin is also included.

Administration Building

The administration building will be a single-storey, pre-engineered building complete with insulated steel roof deck and steel wall cladding. It will house the administrative, engineering, and geology staff.

Shops and warehouse Maintenance Facility

The shops and warehouse maintenance facility is a pre-engineered building complete with insulated steel roof deck, insulated steel wall cladding and an overhead crane.

The floor space provides areas for maintenance shop activities, offices and facilities to support warehouse and maintenance personnel. The building will also include the mine dry area with lockers and showers.

Assay Laboratory

The assay laboratory will be a single-storey modular structure.

Cold Storage Warehouse

The cold storage warehouse will be a light, un-insulated sprung-type structure.

Conveying

Conveyor steel support frames, trusses, bents, take-up, and transfer towers are vendor supplied items. Foundations for conveyor bents,take-up and transfer towers are included in the estimate.

18.8 Communications

Proven, reliable and state-of-the-art communications systems will be installed to ensure that personnel at the mine site have adequate voice, data and other communication channels. The communication systems will include a voice over IP (VoIP) telephone system over a plant-wide fibre optic network. Operating and maintenance personnel will use hand-held, mobile and base radios.

Internet connectivity and external telephone service to the plant will be sourced via a local telephone company or internet service provider.

18.9 Auxiliary Infrastructures

Mine site auxiliary infrastructure items will include fuel storage tank, an administration building, crushing and concentrator buildings, and a truck shop and maintenance building. Due the Project's proximity to Kamloops, a construction camp will not be required. The design basis of the buildings are provided in Appendix E. Layout drawings for the auxiliary buildings can be found in Appendix D.

19.0 MARKET STUDIES AND CONTRACTS

19.1 INTRODUCTION AND SCOPE

Abacus has retained Neil S Seldon & Associates Ltd (NSA) to conduct a general overview of copper concentrate marketing, including projected smelting terms for the Ajax Project. This report, dated October 2011, is summarized in this section. The smelting terms were used as a basis by Wardrop for the financial model and sensitivity analysis. In the absence of letters of interest or letters of intent from potential smelters or buyers of the concentrate, the smelter terms provided by NSA are based on the assumption that the copper concentrate will be delivered to Asian smelters.

This section will also discuss the predicted quality and quantity of the copper concentrate that will be produced from the proposed plant.

19.2 COPPER CONCENTRATE MARKET (NSA REPORT 2011)

The concentrate market is basically split into two types of contracts between buyers and sellers. Firstly, there are long term off-take contracts between mines and smelters, which reflect generally speaking the annual concentrate supply demand balance. Treatment and refining charges (TC/RC) are for the most part agreed annually and such market terms are referred to as Annual Benchmarks.

For the record, a much smaller number of contracts are negotiated in the middle of each year and the resulting TC/RC are referred to as Mid-year Benchmarks. Some long term and mid-year TC/RC data from 2007 to 2011 are shown in Table 19.1.

Secondly there is spot or short term business between mines and primarily 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 TC/RC not only annually, but over each year. Table 19.2 lists the spot term TC/RC from 2007 to 2011. The spot terms in general represent the levels at which traders buy from mines as opposed to the sales by traders to smelters.

Table 19.1 Annual Contract Concentrates TC/RC (Benchmark Terms)

	2007 Annual	2007 Mid-year	2008 Annual	2008 Mid-Year	2009 Annual	2009 Mid-Year	2010 Annual	2010 Mid-Year	2011 Annual	2011 Mid-Year
TC										
US$/dmt	60	52	45	42.5	75	50	46.5	39	56.5	85
RC										
¢/lb	6	5.2	4.5	4.25	7.5	5.0	4.65	3.9	5.65	8.5

Table 19.2 Spot Concentrates TC/RC CIF Shanghai (Spot Term)

	2007 Start	2007 End	2008 Start	2008 End	2009 Start	2009 End	2010 Start	2010 End	2011 Start	2011 End
TC										
US$/dmt	50	45	45	75	80	20	5-10	70-80	70	N/A
RC										
¢/lb	5	4.5	4.5	7.5	8	2	0.5-0.1	7-8	7	N/A

With respect to copper prices, the consensus level for long term prices has gradually risen. Today the range of forecasts at the lower end is around US$2.25/lb and at the top end around US$3.00/lb.

The gold price over the same period in a move upwards primarily in response to economic concerns as a safe haven for wealth and at the same time the cost of producing gold have also risen substantially. Today the consensus long-term prices ranges from around US$900 at the low end to about US$1,200 at the top end.

19.3 COPPER CONCENTRATE

The predicted copper concentrate for this feasibility study will have the following quality and quantity characteristics (see Table 19.3).

Table 19.3 Concentrate Quality and Quantity Characteristics

Description	Unit	Value
Annual Average Copper Concentrate Production	t/a	197,435
Average Copper Grade in Copper Concentrate	%	25.0
Average Gold Grade in Copper Concentrate	g/t	15.6
LOM	y	23
Total Copper Produced, LOM	t	1,135,255
Total Gold Produced, LOM	oz	2,278,927

At a designed copper concentrate grade of 25% Cu, the copper recovery to the concentrate and the feed grade of copper is described in Equation 19.1. Section 13.6.2 in this feasibility study presents the detailed discussion.

$$y = -74.812x2 + 85.727x + 66.668 \qquad \text{Equation 19.1}$$

where: x is the predicted feed grade varied from 0.19% to 0.60% Cu

y is the predicted copper concentrate yield

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL STUDIES

Environmental studies were initiated for the Ajax Project in 2006, including ground and surface water quality and quantity, climatology, fish and fish habitat, wildlife, and vegetation studies. The following provides a summary of the environment surrounding the Ajax Project.

The Ajax Project is located within the Thompson Basin; a warm and very dry, low-elevation area of predominantly gentle slopes with elevations in the Project area ranging from approximately 800 to 1,130 masl. The Project area spans the Interior Douglas Fir (IDF), Bunchgrass (BG) and Ponderosa Pine (PP) biogeoclimatic zones and is characterized by rolling grasslands and timber at higher elevations (Figure 20.1). The IDF zone is primarily forested with Douglas Fir; it occurs in the southern portion of the Project area and on Sugarloaf Hill at higher elevations. The BG zone is characterized by hot, dry conditions and encompasses the central and northeast portions of the Project area. The PP zone occupies low elevation dry valley areas between the IDF and the BG zones within the Project area.

The Project area is located within the Peterson Creek and Cherry Creek watersheds (Figure 20.2). Peterson Creek drains into the South Thompson River near its confluence with the North Thompson River, to the east of Kamloops Lake. At Kamloops, the South Thompson River and the North Thompson River converge to form the Thompson River which flows west from Kamloops Lake, then southwest to its confluence with the Fraser River near Lytton, BC. Cherry Creek is a relatively large perennial drainage within the Thompson River watershed, flowing northwest from Greenstone Mountain to Kamloops Lake. It drains an area of approximately 56,000 km^2, carrying runoff from the Columbia and Monashee mountains. Additionally, alkaline lakes and small intermittent drainages are relatively common in the area.

Stream flow data for the Project area has been collected since April 2008 at five hydrology stations in and around the Project area. Continuous water level data (15 min intervals) recorded at these stations in combination with periodic discharge measurements will provide an estimate of water quantity for input to a site-wide water balance.

Figure 20.1 Biogeoclimatic Zones

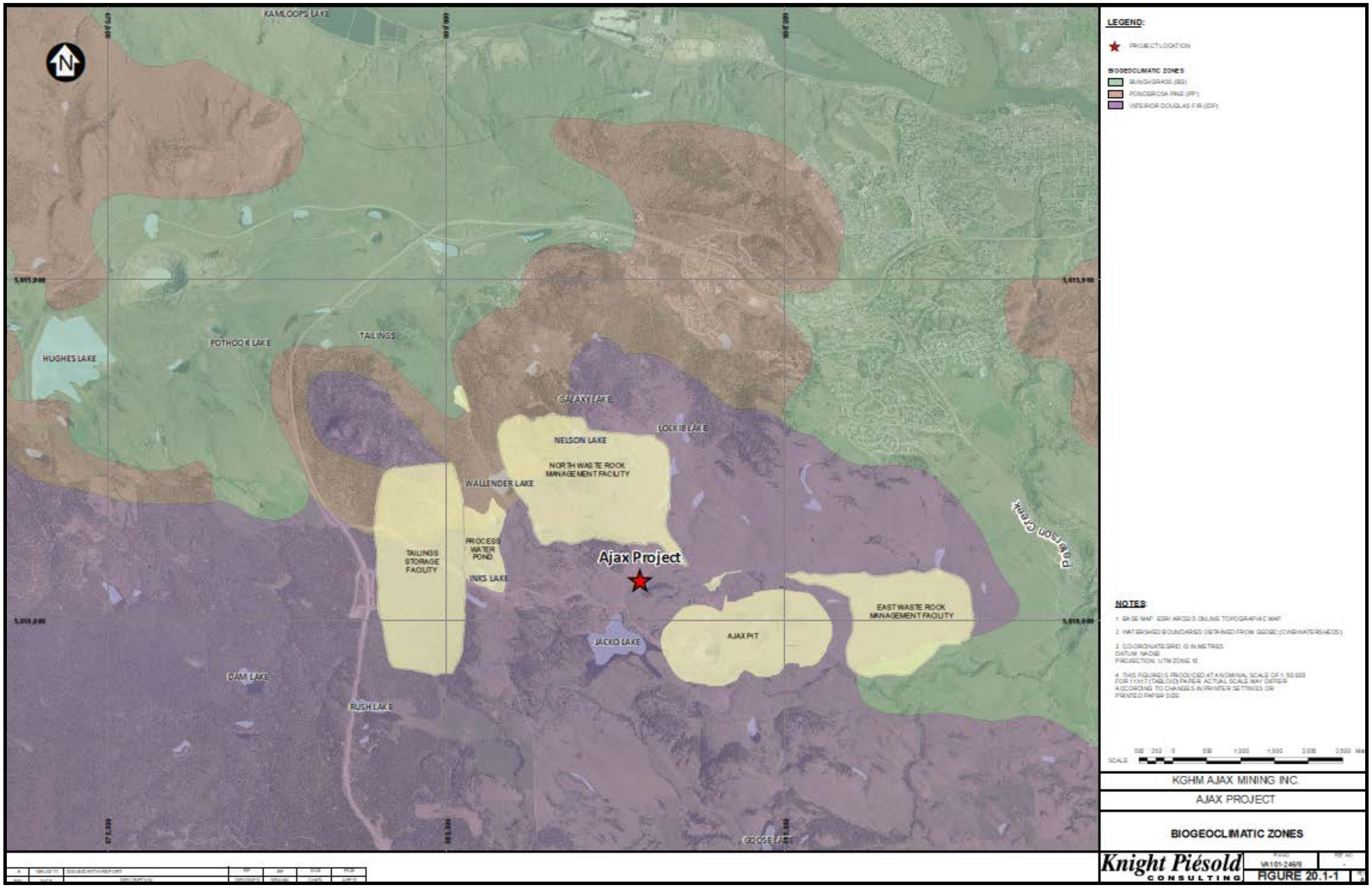
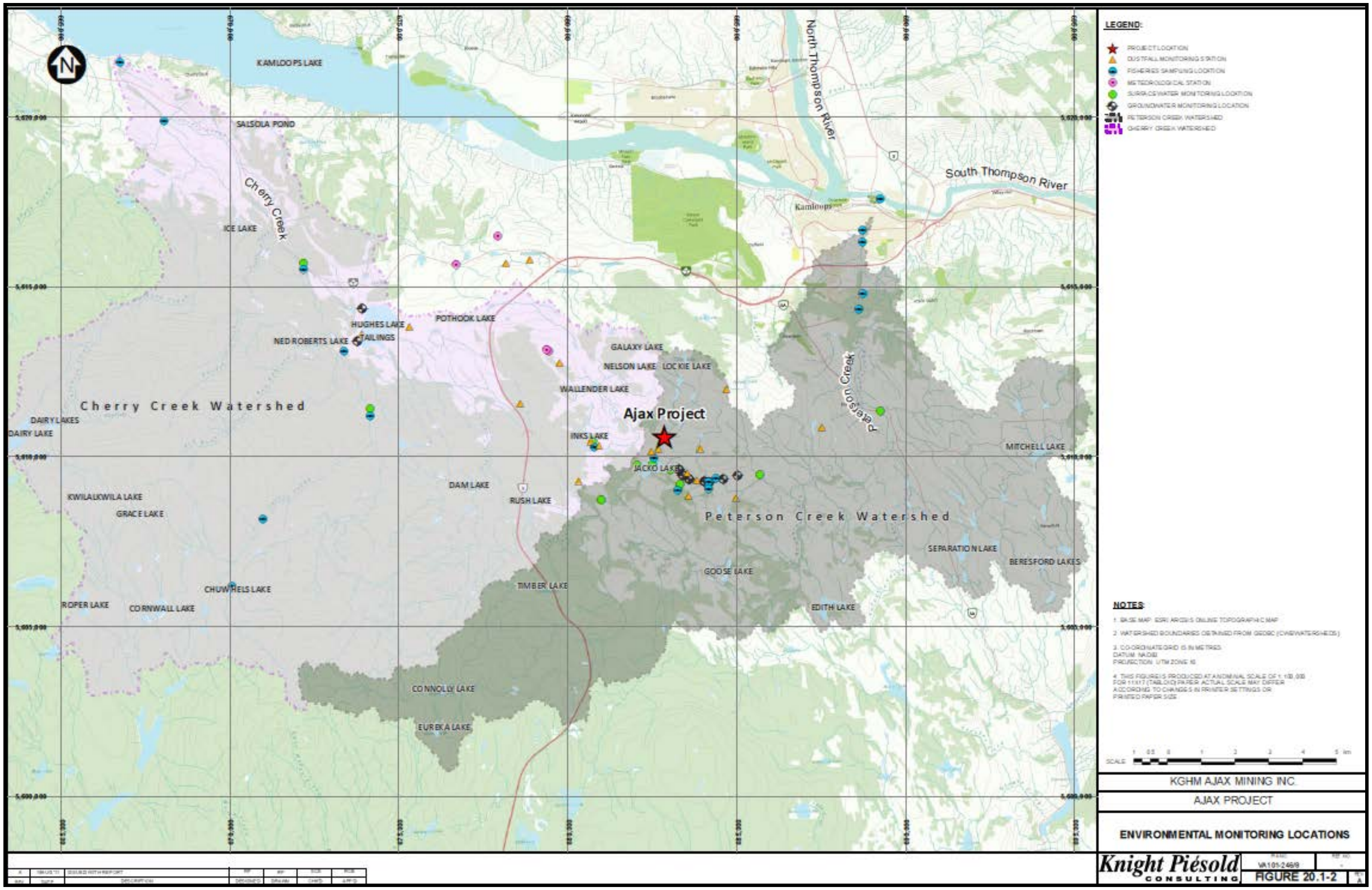


Figure 20.2 Environmental Monitoring Locations



Automated weather stations installed in the Project area and meteorology stations operated by the Ministry of Environment (MOE) and British Columbia Forestry Service (BCFS) have been used to characterize the local and regional climate. The climate of the Kamloops area is semi-arid and characterized by generally cool, dry winters with occasional cold snaps, and hot, dry summers. The long-term average annual temperature is 7.9°C, with average monthly temperatures ranging from -3.0°C in January to 19.9°C in July. Based on long-term precipitation data from the Afton BCFS station and the Kamloops Afton Mines MOE station, mean annual precipitation for the Project area is estimated to be 223 mm, with 75% falling as rain (166 mm) and 25% falling as snow (57 mm). Mean annual lake evaporation is estimated to be 604 mm.

Wind speed and direction in the Project area varies throughout the year. Short-term data collected from the Ajax Meteorological Station (2010 to 2011) indicates that winds (measured at 10 m above the ground) in the Project area are predominantly from the southeast. Site data from the Golder Meteorology Station from November 2006 to September 2008 also indicates a predominant southeast wind, with average and maximum wind speeds of 6.3 m/s and 45.1 m/s, respectively (measured at 2 m above the ground). Longer-term data from the Afton Mines MOE Meteorology Station (March 1989 to December 2008), located northwest of the Project area, indicates a dominant southwest direction, with an average wind speed of 9.5 m/s and maximum wind speed of 70.3 m/s.

Surface water quality monitoring sites were established by Knight Piésold in April 2007 throughout the Project area to characterize existing water quality conditions in the Peterson Creek and Cherry Creek watersheds. Water bodies associated with historical mining activity at the Afton Mine were also monitored, including pit lakes, and the tailings and seepage ponds.

Static geochemical testing was carried out as part of the initial environmental baseline study. These results indicate that the waste rock and ore are not acid generating (Golder, 2010). Comprehensive kinetic and heavy metals leach potential testing will need to be carried out to evaluate the characteristics of effluents from the waste management facilities (ie. TSF and waste rock management facilities). The preliminary project design has been based on the assumption of non-acid generating (NAG) tailings (Golder, 2010). The design of the tailings storage and waste rock management facilities has included recovery of seepage and runoff from the facilities for use in the mill process during operations. The post closure water management plan assumes diversion of all impacted waters into the post closure pit lake.

Baseline surface water quality samples collected in Peterson Creek and Cherry Creek had occasional dissolved metals concentrations exceeding levels set by the Canadian Council of Ministers of the Environment (CCME) and the British Columbia Water Quality Guideline (BCWQG) limits for the protection of aquatic life; however, the majority of samples were within acceptable levels for dissolved metals, nutrients and major ions. Sulphate occasionally measured above the BCWQG limit in Peterson, Cherry and Alkali Creeks and tended to be highly concentrated in historical mining-related water bodies, as well as in smaller natural water bodies including Inks Lake and Polygon Pond. Likewise, nitrogen-based nutrient concentrations frequently exceeded guideline limits in Inks Lake, Polygon Pond, and the tailings and seepage ponds. Jacko Lake water quality samples were consistently below guideline limits with the exception of dissolved selenium, which occasionally exceeded the CCME limit. Appendix H (Tables 20.1-1 through 20.1-10) contains the sample results to date for the freshwater sites around the Project area.

Groundwater monitoring wells were established at eight sites around the proposed pit in 2007. An additional 11 sites were established in 2011, primarily to characterize the area around the TSF and waste rock management facilities. Quarterly sampling has been conducted in the wells established in 2007 and sampling has been initiated in the wells established in 2011. Results will be used in the development of a watershed model using an analytical approach; the model will be calibrated with monthly stream-flow measurements. Results to date for the wells installed in 2007 are provided in Appendix H (Tables 20.1-11 through 20.1-18).

An air quality monitoring program has been initiated to characterize ambient air quality conditions and to develop an understanding of potential impacts of the Project on air quality. The Project is expected to generate Criteria Air Contaminants (CACs) (particulate matter and atmospheric emissions) from surface disturbance and fossil fuel combustion during construction, operations, decommissioning, and reclamation of the Project. Fugitive dust is expected to be the major emission relevant to air quality and the primary concern for nearby residents. Dust fall monitoring stations have been sited to collect representative data for the Project, taking into consideration existing disturbance and potential sensitive receptors in the area.

Fish presence in the Project area is noted in Jacko Lake, Peterson Creek, and Cherry Creek. Jacko Lake is stocked with rainbow trout and used locally as a sport fishing venue. Bridal Veil Falls, on Peterson Creek near the Kamloops city limits prevents upstream passage of fish.

Provincial records indicate that bull trout, rainbow trout, Chinook salmon, sockeye salmon, turbot, and species of whitefish, lamprey, carp, sucker and sculpin are present in the North and South Thompson River systems, Kamloops Lake and the Thompson River.

Site-specific fish and fish habitat assessments were initiated by Knight Piésold in 2007 in the Peterson Creek and Cherry Creek watersheds to document fish habitat utilization, abundance, and distribution; to quantify and assess available fish habitat; and to document barriers to fish migration. Coho salmon fry were captured at the confluence of Peterson Creek and the South Thompson River in July and August 2008. In September 2008, coho salmon and rainbow trout were captured at this site. Downstream movement of rainbow trout from Jacko Lake is possible during high flow events, when the lake water level exceeds the height of the spillway, located at the end of the southeast arm of the lake. Rainbow trout have been captured in Peterson Creek Park upstream of Bridal Veil Falls, located approximately 2 km upstream from the mouth. Rainbow trout have been captured in Cherry Creek during sampling by Knight Piésold and in previous surveys. Fisheries and aquatic surveys are ongoing.

The Project is located within the Kamloops Forest District and the Thompson-Nicola Wildlife Management Unit. There are no Wildlife Habitat Areas (as legislated by the *Forest and Range Practices Act*) or parks present within the Project area. Critical deer and moose winter range has been mapped by the Kamloops Land and Resource Management Plan (LRMP) outside of the Project footprint.

Wildlife surveys conducted during the 2007 and 2008 field seasons included breeding bird surveys, woodpecker surveys, bat detector surveys, waterfowl surveys, small mammal trapping, amphibian surveys, snake hibernacula surveys and owl call playback surveys. Notable results included detection of a number of listed wildlife species in the Project area, including badger (Provincially Red-listed, Species at Risk Act (SARA) Schedule 1 Endangered), Great Blue Heron (Provincially Blue-listed), Swainson's Hawk (Provincially Red-listed), Peregrine Falcon (Provincially Red-listed, SARA Schedule 1 Threatened), Barn Swallow (Provincially Blue-listed), Great Basin Spadefoot (Blue-listed and SARA Schedule 1 Threatened) and Western Toad (SARA Schedule 1 Special Concern).

Discussions have been initiated with government regulatory agencies in order to develop appropriate avoidance and mitigation techniques. None of the environmental parameters identified to date are considered to have a material impact on the ability to extract the mineral resources or reserves.

20.2 WASTE AND TAILING DISPOSAL, SITE MONITORING, AND WATER MANAGEMENT

20.2.1 *TAILINGS, WASTE AND WATER MANAGEMENT*

The principal design objectives for the design and management of the TSF will be to ensure protection of the regional groundwater and surface waters during operations and post closure, and to achieve effective reclamation at mine closure.

A water management plan will be required to demonstrate appropriate control all surface water within the mine area. Goals of the plan include preservation of water quantity and quality downstream of the Project, optimization of water use, maximization of water re-use, minimizing mixing of clean and mine-contact water, managing seepage, utilizing water diversion, and eliminating uncontrolled releases.

Key aspects of water management include the following:

- Ajax pit water and run-off from the waste rock management facilities will be transferred to the site process water ponds during pre-production and throughout the mine life.

- Tailings bleed water and runoff from external slopes of the TSF will be diverted through ditches to the TSF Run-off Pond. Water from this pond will be pumped to the Thickened Tailings Plant (TTP) or to the processing facility as needed.
- The site is in a water deficit. This deficit can be made up through fresh water supply from Kamloops Lake via an upgraded system.
- During operations, all affected site water will be captured in the site process water ponds or seepage collection ponds for containment and use in the processing plant. All non-contact water will be diverted around the site.
- The post closure water management plan assumes that mine affected site water will be routed into the open pit. The filling rate and water quality of the pit lake, as well as any long-term water treatment requirements will need to be submitted to the regulatory agencies for approval.

20.2.2 *MONITORING*

Environmental and operational monitoring and management plans will be developed as part of the Environmental Management System (EMS) to ensure that measures and controls are in place to minimise the potential for environmental degradation during all phases of Project development, including post closure. An Environmental Management Plan (EMP) will be an integral part of the EMS, and will provide clearly defined actions and procedures to ensure that human and environmental health and safety is accounted for through all phases of the Project.

20.3 PERMITTING

20.3.1 *ENVIRONMENTAL ASSESSMENT REVIEW PROCESS*

Major mining projects in BC are subject to environmental assessment as part of the legislated certification and permitting process. Depending on the scope of a proposed project, it may also be subject to federal or harmonized provincial/federal review. A typical EA is generally completed within a two to three year period. During the pre-application phase, the proponent submits a Project Description to the BC EAO and the federal CEAA. The EAO and the CEAA assess whether the project will require an assessment based on any legislative triggers, and the formal scope of the review. The federal Major Projects Management Office (MPMO) develops a Project Agreement among federal regulatory bodies that specifies how and within what timelines the Project will be reviewed.

The environmental review process achieves the following:

- opportunities for all stakeholders and First Nations to identify potential issues and provide input

- an understanding of the environmental, social, economic, heritage, and health effects of the Project
- opportunity to identify ways to prevent or minimize negative effects from the Project or Project-related activities
- an opportunity for input from all interested parties during the EA process.

Detailed environmental and socioeconomic baseline studies continue following submission of the Project Description, and typically require a two-year period to complete. During this time, a draft Application Information Requirements (AIR) is prepared for both the provincial Application for an Environmental Assessment Certificate (Application) and the federal EIS. Submission of the draft AIR is followed by regulatory meetings, at least one public open house, and ongoing First Nations consultation. The final AIR is issued, incorporating relevant comments received during the review period.

Following completion of the baseline studies and based on feasibility studies, an impact assessment is prepared for the Application/EIS. Following submission of the draft Application/EIS to the EAO and federal agencies, a 30-day review period for completion and concordance with the AIR is initiated. The final Application/EIS is then submitted and a 180-day review period begins. The EAO and the federal responsible authorities complete their respective assessment reports within the 180-day period, based on detailed review of the Application/EIS and any comments received from First Nations or the public. Provincial ministers complete their review and sign off on the project environmental assessment certificate within 45 days of receipt of the assessment report. Under the concurrent approval regulation, provincial bodies have 60 days to issue provincial permits. Federal responsible authorities then issue their Notice of Decision, and KGHM Ajax Mining Inc. (KAM) and consultants work with federal regulators to move forward on any required federal permits and authorizations.

The federal *Fisheries Act* prohibits the harmful alteration, disruption, or destruction of fish habitat without specific authorization. Fish habitat compensation will be required to balance the loss of fish habitat in Peterson Creek as a result of pit development and construction of the EWRM Facility. Additionally, fish habitat compensation will likely be required to balance the loss of fish habitat on the tip of the northeast arm of Jacko Lake resulting from pit development. Compensation measures will be explored and adopted to achieve a "No Net Loss Policy".

20.3.2 *Provincial Permits, Authorizations, and Licenses*

Provincial authorizations and permits that may be required for the proposed project include the following:

- BC Ministry of Forests, Lands, and Natural Resource Operations
  - License to cut

  - *Land Act* Authorizations

- Ministry of Environment
  - Water license for new sediment control ponds/detention ponds
  - Section 9 Water Act Approvals
  - Water license for surface water withdrawal/water use, if required
  - Permits under the Environmental Management Act for effluent (e.g., sediment control measures, TSF and waste rock management facilities), air emissions (crushers), and refuse.

- Ministry of Energy and Mines
  - *Mines Act* permit approving the Mine Plan and Reclamation Program (including establishment of reclamation bonding requirements).

- Interior Health Authority
  - Operation permits (drinking water, sewage disposal, etc.).

Specific permitting requirements will be further refined during the review process and through discussions with provincial agencies.

20.3.3 FEDERAL PERMITS, AUTHORIZATIONS, AND LICENSES

Federal authorizations and permits that may be required for the proposed project include the following:

- Fisheries and Oceans Canada
  - Section 35(2) Authorization under the *Fisheries Act* for the harmful alteration, disruption or destruction of fish habitat in Jacko Lake and Peterson Creek.
- Environment Canada (and Canadian Wildlife Services)
- *SARA*
- *Migratory Birds Convention Act*
- Metal Mining Effluent Regulations (MMER).

20.3.4 PERMITTING STATUS

The Proponent submitted a Project Description to the EAO and CEAA in early 2011. The Project Description was accepted by the EAO and posted on the EAO Project Information Centre website on February 25, 2011. The Project received a Section 10 order under the BC *Environmental Assessment Act* on the same date, stipulating that the Project must undergo an EA.

An initial working group meeting was held in Kamloops on April 27, 2011. Representatives from the following agencies and organizations were present at the meeting:

- City of Kamloops
- Thompson Nicola Regional District
- Tk'emlups Indian Band
- Ministry of Energy and Mines
- Ministry of Forests, Lands and Natural Resource Operations
- Ministry of Environment
- Department of Fisheries and Oceans
- Health Canada
- Transport Canada
- Environment Canada.

A preliminary Section 11 order was received by the Proponent in June 2011, setting out the scope of the EA and outlining the requirements for public consultation. An additional Section 11 order is expected in 2012, providing further direction regarding the scope, procedures, and methods for conducting the EA.

The CEAA distributed the Project Description and Coordination Letter to federal departments and agencies on February 16, 2011, and accepted it on March 16, 2011. Following this, the CEAA commenced a comprehensive study on May 25, 2011 and posted a Notice of Commencement on the CEAA Registry on May 31, 2011. A project agreement was signed on August 17, 2011.

The Project AIR was provided to the EAO and CEAA on August 12, 2011 for distribution to the Technical Working Group.

The Proponent Application/EIS is expected to be submitted in 2012.

20.4 Social or Community Requirements

Public comment in relation to the Project must be sought, addressed, and documented through public open houses, meetings and presentations, and through the provincial EAO Project Information Centre (e-PIC) and federal CEAA registry.

The Ajax Project is located in central BC, south of the City of Kamloops, BC, which has a population of approximately 85,000 people. Kamloops is surrounded by the smaller communities of Cherry Creek, Pritchard, Savona, Scotch Creek, Adams Lake, Chase, Paul Lake and Pinantan, among others.

The Project lies within the traditional territory of the Secwepemc Nation. Within the Secwepemc Nation, the Tk'emlúps te Secwepemc (Kamloops Indian Band) and the Skeetchestn Indian Band are the First Nations in closest proximity to the Project. Both bands are members of the Shuswap Nation Tribal Council.

The Lower Nicola Indian Band, a member of the Nicola Tribal Association and the Ashcroft Indian Band, a member of the Nlaka'pamux Nation Tribal Council are also in the vicinity of the Project. Each of these bands is part of the Nlaka'pamux Nation.

Community and First Nations consultation has been initiated by KAM and will be ongoing throughout the pre-application and review phases of the EA.

The Project has garnered significant interest in the local media, given its proximity to the City of Kamloops. Careful management of public concerns is being conducted by the owner in order to maintain local support.

20.5 Mine Closure

A comprehensive closure plan and cost estimate has not yet been developed. It is estimated that reclamation activities will be conducted concurrently during mine operations to the extent practicable and that final reclamation activities after mine closure will proceed over a period of two years. The reclamation and monitoring phase will follow for an additional three or more years.

20.5.1 Closure of Tailings Storage Facility

General considerations that are likely to be incorporated in the closure plan for the TSF include the following:

- removal and restoration of disturbed areas including structure footprints, access roads, conveyance structures, pipelines, etc.
- stabilization, shaping, contouring, capping and re-vegetation of disturbed surfaces
- post closure water management
- monitoring activities to confirm the design assumptions adopted for closure.

20.5.2 Waste Rock Management Facilities Closure

The current plan regarding the waste rock management facilities will be to leave the waste rock in place and construct an engineered cover complete with growth medium and vegetation. In order to accomplish this, suitable growth medium will be identified, salvaged, and stored during construction and operation for use during reclamation and closure.

Post closure water management will be implemented as appropriate to ensure protection of down-gradient water resources.

20.5.3 Ajax Pit

Once mining has ceased, site drainage will be altered to allow mine impacted run-off to be routed into the open pit. Post closure water quality in the open pit will be modelled based to approximate pit lake water quality and to determine any water treatment requirements prior to releasing any overflow to the receiving environment. Water quality sampling will be conducted as the pit fills to verify the accuracy of the pit lake water quality model and any define water treatment requirements to meet established discharge criteria. Passive and active treatment strategies can be considered as potential treatment options, if required. An earthen safety berm will be constructed around the open pit to prevent accidental entry into the decommissioned pit area.

20.5.4 Processing Plant

All of the buildings and structures identified in the mine description above will be dismantled and/or demolished and then removed from the mine site. Salvageable material will be re-used, recycled, or transformed into other useful forms.

All materials removed from the site will be disposed of in accordance with applicable legislation and regulations. Any contaminated material (e.g., petroleum hydrocarbons or heavy metals) will also be stored, handled, and disposed of in accordance with applicable legislation and regulations. Once the buildings and structures have been removed, the areas will be shaped, covered with growth medium, and vegetated with appropriate plant species.

20.5.5 Access Road

At this stage in the Project, it is not clear whether access to the area will be required following closure, but in the event it is no longer required, closure will include the following:

- reclamation of access roads
- removal of bridges, culverts, and other watercourse crossing structures
- restoration of affected stream banks and riparian areas
- re-vegetation of affected areas with appropriate plant species.

20.5.6 Reclamation Bonding

The BC *Mines Act* legislates requirements for closure and reclamation bonding. Section 10 of the *Mines Act* dictates that, as a condition of receiving a permit to commence mining, the owner may be required to give a security deposit for mine reclamation and protection of watercourses and cultural heritage resources in the amount and form specified by the Chief Inspector of Mines. The reclamation bond amount is typically based on a reclamation plan and associated cost estimate approved by the Ministry of Energy and Mines.

Additionally, the owner must deposit an annual security so that, together with the security deposit mentioned above, sufficient funds will be available to complete all permit conditions.

Security amounts can be adjusted annually to accommodate mine plan changes, and concomitant reclamation activities.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATES

The capital cost estimates for the Ajax Project were prepared by the following consultants:

- Wardrop: mineral processing and overall site infrastructures
- AMEC: mine capital costs including mobile equipment, rock storage areas, and in-pit crushing and conveyance related costs
- Golder: TTP and TSF.

Wardrop established the capital cost estimate using a hierarchical Work Breakdown Structure (WBS). The accuracy range of the estimate is ±15%. All currency is expressed US dollars unless otherwise stated. Wardrop used a foreign currency exchange rate of Cdn$1.00 to US$0.92 where applicable.

An initial capital of US$795 million is estimated for the project. Table 21.1 summarizes the capital cost estimate and the detailed breakdown can be found in Appendix I.

Table 21.1 Capital Cost Summary

		Cost (US$ '000)
Direct Works		
A	Overall Site (Wardrop)	46,280
B	Open Pit Mining (AMEC)	109,886
C	IPCC (AMEC)	42,464
D	Crushing and Stockpiles (Wardrop)	89,614
E	Grinding and Flotation (Wardrop)	144,815
F1	TSF (Golder)	79,494
F6	Water Treatment (Wardrop)	2,407
G	Site Services and Utilities (Wardrop)	12,897
J	Ancillary Buildings (Wardrop)	18,639
K	Fresh Water Pipeline (Wardrop)	14,805
L	Plant Mobile Fleet (Wardrop)	4,437
Direct Works Sub-total		**565,740**

table continues…

		Cost (US$ '000)
Indirect Works		
X	Project Indirects	107,176
Y	Owner's Costs	34,500
Z	Contingencies	87,570
Indirect Works Sub-total		**229,246**
Total		**794,987**

EXCLUSIONS

The following items are excluded from the above capital cost summary:

- Site security, construction power/services, insurance, bonding, permits, legal costs, and taxes/duties have been excluded from the cost estimate (Golder).
- Any project sunk costs, including this study, working or deferred capital, financing costs, refundable taxes and duties, currency fluctuations, lost time due to severe weather conditions/force majeure, fire proofing, additional costs for accelerated or decelerated deliveries of equipment, materials, and services resultant from a change in project schedule, warehouse inventories other than those supplied in initial fills, Owner's costs unless provided by Owner, escalation post (Q4 July 2011), and community relations (Wardrop).

21.1.1 MINING CAPITAL COST ESTIMATE

The results of the mining capital cost estimate represent forward-looking information that are 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 information includes the Mineral Resource estimate, the Mineral Reserve estimate, commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and risks regarding the cost estimates and completion schedule for the proposed Project infrastructure, in particular the need to obtain permits and governmental approvals.

AMEC prepared the capital cost estimate and indirect costs for mining-related activities. AMEC did not check for accuracy nor verify costs and quantities for other parties' cost estimates. Warehouse inventory was excluded from the capital cost estimate.

Equipment estimates was based on vendor quotes, because the material handling design will not be installed for nine years and therefore not designed to sufficient detail to prepare material take-offs, and the estimate was based on factors or allowances.

Normal and accepted allowances were included in the estimate by discipline, as appropriate. Conceptual quantities were prepared where drawing information was not available. Metric units were assumed throughout the estimate.

There is no allowances for taxes, duties, or escalation in the capital cost estimate. No contingency was estimated in the mining costs.

Capital costs are based on the following:

- Capital costs for purchases and replacements of major mining equipment are based on budgetary quotations from equipment vendors.
- Sustaining capital costs are included in the mining capital costs.
- AMEC's internal database was used to estimate costs for certain fleets of ancillary equipment.
- AMEC's materials handling department estimated capital costs for the in-pit and out-pit crushers, conveyors, and waste dump stacking system.
- Abacus provided costs for engineering, office, and other equipment. This equipment includes the dispatch system, a pit slope monitoring system, and all mine planning, survey, and engineering equipment and software, as well as general office equipment.

Capital costs over the life of mine for the mining function are summarized in Table 21.2.

Table 21.2 Mine Capital Costs

Area	Cost (US$ '000)
Pre-stripping	34,443
Drilling Equipment	9,969
Loading Equipment	81,338
Hauling Equipment	181,418
Support Equipment	29,953
Mine Maintenance Equipment	13,608
Dewatering	3,609
Crushing	81,600
Conveying	84,822
Stacking	54,910
Crushing – Sustaining	8,000
Stacking – Sustaining	1,200
Engineering Equipment	6,655

table continues…

Area	Cost (US$ '000)
In-Pit Power Supply Equipment	21,000
Total	**612,519**

21.1.2 *Processing and Overall Site Infrastructure (Wardrop)*

Wardrop's detailed capital cost estimate is provided in Appendix I. The major considerations in the capital cost estimate include:

- Major mechanical and electrical equipment cost estimates are prepared from the bids of the qualified vendors. All equipment and material costs is included as free carrier (FCA) or free board marine (FOB) manufacturer plant and exclusive of spare parts, taxes, duties, freight, and packaging. These costs, if appropriate, are covered in the indirect section of the estimate.
- Where appropriate, material quantities were developed from general arrangement drawings, process design criteria, process flow diagrams, and equipment lists. Electrical, platework, instrumentation, piping, and heating, ventilation and air conditioning (HVAC) are based on the detailed process flowsheet diagram (PFD), piping and instrumentation diagrams (P&IDs) and general arrangements (GA).
- A blended labour rate of $78.20/h was used throughout the estimate, with the exception of input received from external consultants. The labour rate was developed based on Construction Labour Relations BC (Millwrights Local 2736), escalated to 2011.

A productivity factor of 1.15 was applied to the labour portion of the estimate to allow for the inefficiency of long work hours, climatic conditions and due to the three-week–in-one-week-out rotation.

- The truck shop and warehouse complex building (including mine dry), administration building, and cold warehouse are based on engineered steel framed structure complete with roofing, cladding, doors and architectural finishes. The gatehouse, assay laboratory are modular buildings based on modular buildings.
- Project indirect costs including construction, commissioning and start-up, spare parts, and freight and logistics are calculated on a percentage basis based on Wardrop and other external consultant's work experience. Wardrop has also provided additional allowance for a spare set of HPGR tires and rollers and liner spares for Mills and IsaMills. Allowances for initial fills are provided for grinding media, reagents and fuel, engineering and procurement. Construction management allowance is calculated on a percentage basis based on Wardrop in-house experience. Vendor

assistance allowance is calculated based on number of engineers required on site, estimated duration and average manhour rate of $150 per manhour.

- An allowance of $34,500,000 has been supplied by the Owner for Owner's costs which include Land and Asset Purchase, and Cost of Risk (CR) Mitigation and NSR Payments. The estimated contingencies are allowances for undefined items of work which is incurred within the defined scope of work covered by the estimate. Each discipline was allocated different contingency factors due to the varied risk level. The average contingency for Ajax project is 15% of the total direct costs.

21.1.3 *Tailings Storage Facility Capital Cost Estimate (Golder)*

The TTP and TSF capital costs are presented in full in Appendix I and summarized below:

- The total process mechanical equipment costs were obtained from vendor quotations based on equipment specifications prepared by Golder.
- Installation labour rates were based on information provided by Wardrop on behalf of Abacus and it was indicated as Cdn$85.00/h (US$78.20/h) and major equipment.
- Installation manhours were provided by Golder based on their expertise. The cost of earthworks, footings and foundations and steelwork were determined using material take-offs from preliminary design drawings.
- The costs for ventilation and air conditioning, electrical, instrumentation, actuated valves, PLC and programming, piping, construction equipment and miscellaneous equipment were estimated based on comparisons to similar previous projects and Golder expertise.
- Pipeline costs were estimated based on the cost for supply and installation from suppliers and from previous projects.
- Tanks costs for installation and materials were provided by Golder based on its expertise as the vendors declined biding.
- HVAC, PLC, electrical, construction equipment and miscellaneous equipment are estimated based on comparisons to similar previous projects and Golder expertise.
- Commissioning spare parts were calculated as a percentage of the total major mechanical equipment costs.

The capital cost summary of the TTP and TSF are provided in Table 21.3 below.

Table 21.3 Capital Cost Summary of the TTP and TSF

Item Description	Material/ Equipment Total (US$)	Labour Total (US$)	Delivery Costs (US$)	Total (US$)
Mechanical Equipment	21,222,916	1,872,030	420,787	23,515,733
Electrical Equipment	6,949,676	688,024	54,975	7,692,674
Civil/Structural/ Architectural	7,647,475	(included in material)	71,318	7,718,794
Non Equipment Direct Costs	1,820,922	(included in material)	20,801	1,880,823
External Pipeline	22,701,800	(included in material)	635,922	23,337,722
TSF	15,348,197	(included in material)	(included in material)	15,348,197
Total Direct	**-**	**-**	**-**	**79,493,944**

21.2 Operating Cost Estimates

The operating cost estimates for the Ajax Project were prepared by the following consultants:

- Wardrop: mineral processing and overall site infrastructures
- AMEC: mine capital costs including mobile equipment, rock storage areas, and in-pit crushing and conveyance related costs
- Golder: TTP and TSF.

The accuracy range of the estimate is ±15%. All currency is expressed US dollars unless otherwise stated. Wardrop used a foreign currency exchange rate of Cdn$1.00 to US$0.92 where applicable.

Table 21.4 summarizes the operating cost estimate and the detailed breakdown can be found in Appendix C, Appendix G and Appendix I.

Table 21.4 Operating Cost Summary

Description	Operating Cost (US$/t Milled)
Mining& In-Pit Crushing (AMEC)	4.48*
Processing (Wardrop)	3.15
G&A	0.53
Tailings (Golder)	0.31
Operating Cost Total	**8.47**

Note: *The cost of US$4.48/t milled is equivalent to US$1.32/t mined

21.2.1 *Mining Operating Cost Estimate (AMEC)*

The results of the mining operating cost estimate represent forward-looking information that are 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 information includes the Mineral Resource estimate, the Mineral Reserve estimate, commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, uncertainties and risks regarding the estimated capital and operating costs, uncertainties and risks regarding the cost estimates and completion schedule for the proposed Project infrastructure, in particular the need to obtain permits and governmental approvals.

Operating cost estimates have been assembled by area and component, based on estimated staffing levels, consumables, and expenditures; according to the mine plan. Operating costs have been prepared in Q4 2011 US dollars with no allowances for escalation, sales tax, import duties, or contingency.

Basis of Estimate

Salary and wage rates were based on information provided by Abacus. The work schedules assume that production will operate 24 h/d, 7 d/wk, 365 d/a. All hourly (non-exempt) personnel assigned to positions that form part of continuous operations will work a four-days-in/four-days-out rotation. The operations and mining personnel will also work a four-days-in/four-days-out rotation.

Mining Operating Costs

Operating costs were derived by applying equipment productivities, equipment hourly operating costs, labour requirements, and wages and salaries to the proposed production forecast.

The mine operating cost estimate includes costs for equipment operators, staff, contractors, and operating and maintenance supplies for each year. Operating and maintenance supply costs are based on in house data and vendor quotations, and are exclusive of taxes. Consumables quantities (fuel, explosives, tires, blasting accessories, etc.) were determined from expected unit consumption rates (per hour, per tonne).

XERAS financial modelling software was used to calculate, on a yearly basis, the expected requirements for equipment, consumables consumption, and direct labour by combining the mine schedule with unit consumption rates, costs, and various performance inputs. The resulting LOM mining operating costs are summarized below in Table 21.5.

Table 21.5 Mine Operating Costs

Operating Costs	Total	Mined ($/t)
Administration/Overhead	159,594,531	0.11
Loading	188,552,137	0.11
Hauling	1,055,978,642	0.62
Drilling	161,525,069	0.09
Explosives	227,353,080	0.13
Support	141,575,992	0.08
Ancillary	40,144,184	0.02
Dewatering Allowance	28,093,378	0.01
Material Handling	249,605,029	0.15
Total	**2,252,422,043**	**1.32**

21.2.2 Process Operating Cost Estimate and General & Administrative Costs (Wardrop)

The process operating costs for the Ajax concentrator includes grinding, flotation, thickening and filtration to produce a copper concentrate. G&A costs have also been estimated and are included in the operating cost estimate.

Operating Cost Summary

Table 21.6 gives the overall estimated cost summary for the processing facility and the G&A costs, and is based on 60,000 t/d with a mill availability of 92% and 365 operating days per year.

The annual operating costs for the process facilities excluding primary crushing and tailings handling sections is estimated to be $68.9 million, or $3.15/t of ore treated at the processing rate of 60,000 t/d. An incremental increase in G&A operating cost is estimated to be $11.6 million, or $0.53/t of ore treated.

Table 21.6 Operating Cost Summary

Description	Personnel	Annual Cost (US$)	Unit Cost (US$/t Treated)
Operations Staff	5	707,940	0.03
Operations Labour	45	3,454,956	0.16
Maintenance Labour	42	3,555,582	0.16
Laboratory & Quality Control	17	1,201,807	0.06
Sub-total	**109**	**8,920,285**	**0.40**
Operating Supplies	-	34,454,150	1.57
Maintenance Supplies	-	9,108,000	0.42
Power Supply	-	16,450,021	0.75
Sub-total	**-**	**60,012,171**	**2.74**
Total Process Operating Costs	**109**	**68,932,456**	**3.15**
G&A Staff	46	3,899,880	0.18
G&A Expenses	-	7,680,000	0.35
G&A Power Cost	-	49,227	0.002
Total G&A Costs	**46**	**11,629,107**	**0.53**

The annual operating cost estimate includes the following:

- staffing and maintenance manpower complements and base salaries, including a burden of 35%, salary information is based on staffing complements as supplied by Ajax, similar project salary cost and “Western Mines” benchmark salary rates
- power consumption, based on the estimated power drawn
- reagent consumption values
- cost of reagents has been based on recent prices received from reagent suppliers
- an estimated maintenance costs which was based on the capital cost estimate.

MANPOWER

Table 21.7 gives the estimated operating and maintenance manpower requirements for the process plant and Table 21.8 gives the G&A complement.

Table 21.7 Process Plant Manpower Requirements

Description	Manpower	Loaded Annual Salary (US$)	Annual Cost Payroll (US$)	Unit Cost (US$/t Milled Ore)
Average Benefit Rate / Burden	-	35%	-	-
Staff				
Mill Superintendent (Plant Manager)	1	204,930	204,930	0.009
Chief Metallurgist	1	186,300	186,300	0.009
General Foreman	1	124,200	124,200	0.0006
Chief Assayer (Chief Chemist)	1	124,200	124,200	0.0006
Mill Clerk	1	68,310	68,310	0.003
Sub-Total 1	**5**	**-**	**707,940**	**0.032**
Operations				
Mill Foreman (Shift Supervisor)	4	111,780	447,120	0.020
Senior Metallurgists	2	124,200	248,400	0.011
Junior Metallurgists	2	99,360	198,720	0.009
Secondary Crusher Operators	8	70,883	567,061	0.026
Control Room Operators	4	77,111	308,446	0.014
Grinding Operators	4	70,883	283,531	0.013
Flotation Operators	4	77,111	308,446	0.014
De-Watering Operators	4	70,883	283,531	0.013
Concentrate Load-out	4	68,000	271,998	0.012
Plant Trainer	1	93,150	93,150	0.004
Labourer/Trainee	8	55,569	444,554	0.025
Sub-Total 2	**45**	**-**	**3,454,956**	**0.158**
Laboratory				
Mill Technicians	2	67,322	134,645	0.006
Senior Chemist	1	124,200	124,200	0.006
Assayer	6	70,883	425,296	0.019
Sample Bucker	8	64,708	517,667	0.024
Sub-Total 3	**17**	**-**	**1,201,807**	**0.055**
Plant Maintenance				
Mill Maintenance Foremen	2	108,501	217,002	0.010
Mill Maintenance Planner	2	86,413	172,827	0.008
Mill Maintenance Clerk	2	68,310	136,620	0.006
Millwrights-Journeyman	10	86,795	867,946	0.040
Welders-Journeyman	6	86,795	520,767	0.024
Apprentices (level 2)	6	70,883	425,296	0.019
Electrician	8	86,795	694,356	0.032
Instrument Technicians	6	86,795	520,767	0.024
Sub-Total 4	**42**	**-**	**3,555,582**	**0.162**
Total Manpower	**109**	**-**	**8,920,285**	**0.407**

Table 21.8 G&A Requirements

Description	Manpower	Annual Cost/Employee (US$)	Annual Cost Payroll (US$)	Unit Cost (US$/t Milled Ore)
G&A Manpower				
General Administration				
General Manager	1	254,610	254,610	-
Secretary	1	86,940	86,940	0.004
Administration Manager	1	99,360	99,360	-
HSEC Manager	1	124,200	124,200	0.006
IT Services Manager	1	86,940	86,940	0.004
Accounting				
Chief Accountant	1	111,780	111,780	0.005
Payroll Clerk	1	74,520	74,520	0.003
Accounting Clerk	1	74,520	74,520	0.003
Purchasing/Warehouse				
Chief Purchaser	1	111,780	111,780	0.005
Assistant Purchaser	1	86,940	86,940	0.004
Warehouse Shipper & Receiver	4	74,520	298,080	0.014
Janitorial	4	65,826	263,304	0.012
Gate Security	6	65,826	394,956	0.018
HR and Training				
Human Resources Manager	1	111,780	111,780	0.005
Human Resources Clerk	1	74,520	74,520	0.003
Health and Safety				
Health and Safety Coordinator	1	99,360	99,360	0.005
Safety and Training Officer	1	90,666	90,666	0.004
First Aid	4	78,246	312,984	0.014
Environmental				
Environmental & Community Relations Manager	1	111,780	111,780	0.005
Environmental Technician	2	86,940	86,940	0.008
Site Services				
Site Services General Foreman	1	111,780	111,780	0.005
Site Services Foreman	1	86,940	86,940	0.004
Site Yard Labourers	4	62,100	248,400	0.011
Site Electrician	2	74,520	149,040	0.007
Site Mechanic	2	86,940	173,880	0.008
Site Carpenter	1	86,940	86,940	0.004
Total G&A Manpower	**46**	**-**	**3,899,880**	**0.178**

Notes: Assumptions Used in the Manpower Estimate.

The operating costs have been determined using the operating plant complement required to run and maintain the plant facilities. These numbers do not include the staffing numbers required for operations and maintenance of the primary crusher facilities nor the tailings facility.

The manpower complement has been estimated on the basis of having two crews under equal rotation time and two 12 h shifts per day. This equates to 2,190 h/a for shift workers, and 2,080 h/a for salaried staff.

POWER AND SUPPLIES

Table 21.9 through to Table 21.13 show the operating cost details relating to power and supplies, as well as the G&A expenses.

Table 21.9 Power Supply Required for Process

Plant Power	56,924 kW running			
Administration Buildings	238 kW running			
Supplies	**kWh/year**	**Unit Cost (US$/kWh)**	**Total Cost (US$/year)**	**Unit Cost (US$/t Ore)**
Plant Power	470,538,348	0.035	16,450,021	0.75
Total Plant Power Supply	**470,538,348**	**0.035**	16,450,021	0.75
G& A Power Supply	**1,408,109**	**0.035**	**49,227**	**0.002**

Table 21.10 Maintenance Supplies Allowance

Area	Total Cost (US$/year)	Unit Cost (US$/t Ore)
Cone Crusher and Related Equipment	920,000	0.0420
HPGR Crusher	920,000	0.0420
Grinding Area	2,300,000	0.1050
Flotation Area	2,760,000	0.1260
Concentrate Dewatering	920,000	0.0420
Reagents	230,000	0.0105
Assaying	138,000	0.0063
Miscellaneous Mill Supplies	460,000	0.0210
Misc. Building Complex Supplies	460,000	0.0210
Total Maintenance Supplies	**9,108,000**	**0.416**

Table 21.11 Plant Operating Supplies

Supplies	Consumption (kg/t Ore)	Unit Cost (US$/kg)	Total Cost (US$/year)	Unit Cost (US$/t Ore)
Cone Crusher Liners	0.009	2.79	575,797	0.026
HPGR Rolls	2 sets/year	250,000 per set	500,000	0.023
Screen Panels	10 sets/year	9,000 per set	92,000	0.004
Sub-total	**-**	**-**	**1,167,797**	**0.053**
Ball Mill Balls	0.950	0.86	17,800,758	0.813
Ball Mill Liners	0.081	2.63	4,667,103	0.213
Isamill Grinding Media, Rougher	0.005	4.60	509,019	0.023
Isamill Grinding Media, Cleaner	0.002	4.60	173,313	0.008
Isamill Spare Grinding Discs	9 sets/y	12,880/set	115,920	0.005
Isamill Spare Liners	3 sets/y	123,300/set	369,840	0.017
Sub-total	**-**	**-**	**23,635,953**	**1.079**
PAX	0.032	2.81	1,966,445	0.090
MIBC	0.045	2.76	2,719,980	0.124
Lime	0.579	0.23	2,916,423	0.133
Flocculant	0.005	4.42	483,552	0.022
Sub-total	**-**		**7,910,609**	**0.36**
Filter Cloth (Concentrate)	allowance	-	184,000	0.008
Dewatering Supplies	allowance	-	184,000	0.008
Sub-total	**-**	**-**	**368,000**	**0.0.017**
Laboratory Supplies	allowance	-	460,000	0.021
Sub-total	**-**	**-**	**460,000**	**0.021**
Mill Light Vehicle Operation	allowance	-	276,000	0.013
Miscellaneous	allowance	-	460,000	0.021
Sub-total	**-**	**-**	**736,000**	**0.034**
Total Operating Supplies	**-**	**-**	**34,454,150**	**1.573**

Table 21.12 G&A Expenses

	Total Cost (US$/year)	Unit Cost (US$/t Ore)
Corporate Office Expenses	250,000	0.011
Office Supplies	100,000	0.005
Professional Associations	50,000	0.002
Consultants	150,000	0.007
Insurance	1,200,000	0.055
Legal Services	500,000	0.023
Regulatory Compliance/Audit	250,000	0.011
Travel & Expenses	80,000	0.004
Communications: Tel Fax Internet	250,000	0.011

table continues…

	Total Cost (US$/year)	Unit Cost (US$/t Ore)
Computer and IT Services and Supplies	200,000	0.009
Community Public Relations & Donations	500,000	0.023
Recruitment	250,000	0.011
Training	600,000	0.027
Safety & Training Supplies	500,000	0.023
Safety Incentives	250,000	0.011
Medical Service/First Aid	250,000	0.011
Security Supplies	200,000	0.009
Environmental Supplies	400,000	0.018
Purchasing and Logistics	150,000	0.007
External Assays/Testings	250,000	0.011
Janitorial	150,000	0.007
Powerline Maintenance	100,000	0.005
Road Maintenance	300,000	0.014
Crew Transportation	500,000	0.023
Miscellaneous	250,000	0.011
Total G&A Expenses	**7,680,000**	**0.351**

Table 21.13 First Fills – Capital Cost

Supplies	Source	Total Cost (US$)
Reagents	Estimate	846,151
Liners and Grinding Media	Estimate	3,084,612

Table 21.9 gives the annual power requirements for the process sections based on the estimated power usage. The unit cost of power was given to be $0.035/kWh and equates to $0.75/t of ore processed excluding the power required at the primary crusher and at the tailings and reclaim areas.

Note that the power requirements for the administration buildings are listed separately in Table 21.9. The annual maintenance supplies requirement for the plant has been estimated for each of the sections, and these costs are shown in Table 21.10. The plant cost of maintenance supplies has been calculated to be $9.1 million, or $0.42/t of ore treated.

The annual estimated plant operating supplies requirements are provided in Table 21.11. The operating supply costs have also been estimated for the different sections of the process plant. The costs of the grinding media and reagents have been included. The total cost of operating supplies for the main plant has been determined to be $34.5 million, or $1.57/t of ore processed.

Table 21.12 details the annual G&A expenses and does not include any contingency. The unit G&A expenses operating cost has been calculated to be $0.35/t of ore processed.

The process section "First Fills" requirement is provided in Table 21.13, although this is generally considered to be a capital cost item.

ASSUMPTIONS USED IN POWER AND SUPPLIES REQUIREMENTS

The estimated power requirements for the process sections are given as a total value in Table 21.9. The power value records the anticipated annual operating power usage and not the installed power for the equipment. The complete equipment list and electrical load are located in Appendix J. The unit electrical cost of $0.035/kWh was provided by Ajax.

The costs of the plant operating supplies costs include the reagent costs for the usage as given in the process design criteria. The grinding media costs were based on the calculated consumptions values, and vendor estimates.

The maintenance supplies costs are estimated values and are reflected as an allowance as based on a percentage of the capital cost estimate. Similarly, the G&A expenses have been based on estimated allowance values.

First Fills costs are estimates only based on the yearly consumption values used in the operating cost estimate. The reagent allotment is an allowance for one month supply. Due to the closer proximity to steel supply vendors, the steel allowance for the first fill is a one month allotment.

21.2.3 *TTP AND TSF OPERATING COST ESTIMATE (GOLDER)*

Golder provided the operating cost estimates for TTP and TSF facilities (see Table 21.14). The operating cost of TTP and TSF were based on the following unit costs for a full production year.

- Power cost is based on operating equipment power draw using Abacus' unit rate.
- Plant power consumption increases as the pumping power demand increases and it is shown as an average of the following periods:
  - Year 1 to 4
  - Year 5 to 9
  - Year 10 to 14
  - Year 15 to 19
  - Year 20 to 25
  - Year 26.5 (maximum power consumption).

- Flocculant cost estimated from vendor pricing.
- Equipment maintenance costs based on 5% of process mechanical costs.
- Labour rates based on Wardrop's input.
- Tailings production quantity is based on nominal annual production.

Table 21.14 Annual Operating Cost Estimate

Description	Cost (US$/year)
Average Plant Power Cost Years 1 to 4	1,054,100
Average Plant Power Cost Years 5 to 9	1,102,300
Average Plant Power Cost Years 10 to 14	1,169,500
Average Plant Power Cost Years 15 to 19	1,255,500
Average Plant Power Cost Years 20 to 25	1,357,400
Average Plant Power Cost Year 26.5	1,461,400
Plant Maintenance	1,288,700
Consumables (flocculant, lube, etc.)	2,861,700
System Manpower	567,100
TSF Operation	652,600
Years 1 to 4	
Total Average Annual Operating Cost	6,424,200
Average Operating Cost (per dry tonne placed)	0.30
Years 5 to 9	
Total Average Annual Operating Cost	6,472,400
Average Operating Cost (per dry tonne placed)	0.30
Years 10 to 14	
Total Average Annual Operating Cost	6,539,600
Average Operating Cost (per dry tonne placed)	0.30
Years 15 to 19	
Total Average Annual Operating Cost	6,625,600
Average Operating Cost (per dry tonne placed)	0.31
Years 20 to 25	
Total Average Annual Operating Cost	6,727,500
Average Operating Cost (per dry tonne placed)	0.31
Years 26.5 (max.)	
Total Average Annual Operating Cost	6,831,500
Average Operating Cost (per dry tonne placed)	0.31

22.0 ECONOMIC ANALYSIS

22.1 INTRODUCTION

Wardrop prepared an economic evaluation of the Ajax Project based on a pre-tax financial model. Pit optimization and financial analyses have been conducted based on metal prices agreed upon as of June 12, 2009. The metal prices used in the financial analysis are as follows:

- copper – US$2.75/lb
- gold – US$1,085.00/oz
- exchange rate (Forex) – $0.92:$1.00 (US$:Cdn$).

Based on the metal prices above, pit optimization has generated a 23-year LOM and 503 Mt mineable ore. The pre-tax financial results were as follows:

- 14.5% IRR
- 7.8 year payback on US$795 million capital
- US$416 million NPV at 8.0% discount rate.

Wardrop also conducted a sensitivity analysis to evaluate the sensitivity of the project merit measures (NPV and IRR) to the main inputs in metal prices. The sensitivity analysis is found in Section 22.2.4.

22.2 PRE-TAX MODEL

22.2.1 MINE/METAL PRODUCTION IN FINANCIAL MODEL

The LOM and average ore tonnages, head grades and metal production for the metal prices are indicated in Table 22.1.

Table 22.1 Metal Production from the Ajax Project

Description	Value
Total Tonnes to Mill (‘000)	503,012
Average Annual Tonnes to Mill (‘000)	21,870
Mine Life (Years)	23
Average Head Grade	
Copper (%)	0.27
Gold (g/t)	0.17
Total Production	
Copper (‘000 lb)	2,502,783
Gold (‘000 oz)	2,279
Average Annual Production	
Copper (‘000 lb)	108,817
Gold (‘000 oz)	99

22.2.2 Financial Evaluations – NPV and IRR

Production figures from AMEC have been incorporated into the 100% equity pre-tax financial model to develop annual recovered metal production from the relationships of tonnage milled, head grades, and recoveries.

Copper payable values and additional metal credits were calculated based on metal prices. NSR was calculated each year by subtracting the applicable treatment, smelting, refining and penalties (if applicable) charges from the payable metal value. At-mine revenues are then estimated by subtracting transportation, insurance and representation costs.

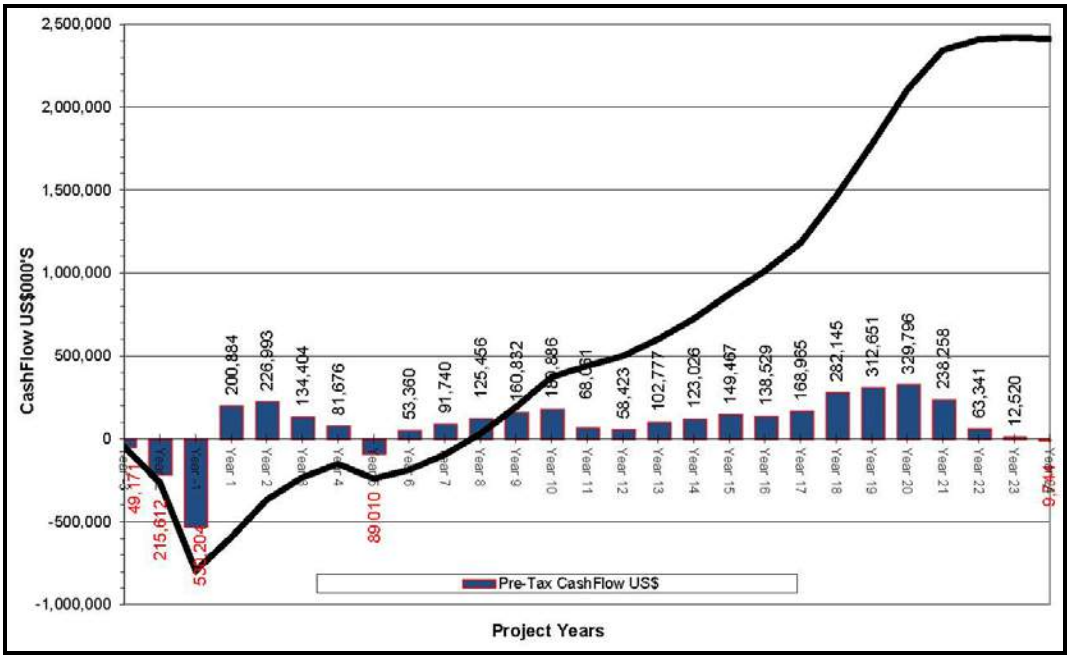
Unit operating costs for mining, processing, power, fuel, and G&A were applied to annual mined/milled tonnages to determine the overall operating cost which was deducted from the revenues to derive annual operating cash flow. Initial and sustaining capital costs as well as working capital have been incorporated on a year-by-year basis over the mine life and deducted from the Net Revenue to determine the Net Cash Flow before taxes. Initial capital expenditures include costs accumulated prior to first production of concentrate; sustaining capital includes expenditures for mining and milling additions, replacement of equipment, and waste management.

Working capital is estimated at one month of the Year 1 operating cost. The working capital is recovered at the end of the mine life.

No salvage value was considered. Mine closure and reclamation cost of US$25.15 million is considered and incurred at the end of mine life.

The undiscounted annual net cash flows and cumulative net cash flows are illustrated in Figure 22.1.

Figure 22.1 Undiscounted Annual and Cumulative Cash Flow



22.2.3 SUMMARY OF FINANCIAL RESULTS

The pre-tax financial model was established on a 100% equity basis, excluding debt financing and loan interest charges. The financial outcome has been tabulated for NPV, IRR, and payback of capital (Table 22.2). A discount rate of 8% was applied.

Table 22.2 Summary of Pre-Tax Financial Results by Metal Price Scenario

Scenario	Value
Total Payable Metal Value ('000 US$)	9,052,307
Treatment, Smelting, Refining, and Penalties ('000 US$)	491,596
Total NSR ('000 US$)	8,560,711
Concentrate Transport, Insurance, Representation ('000 US$)	523,633
At-mine Revenue ('000 US$)	8,037,077
Royalties ('000 US$)	0
Operating Costs ('000 US$)	4,225,001
Operating Cash Flow ('000 US$)	3,812,076
Capital Expenditure ('000 US$)	1,401,624
Net Cash Flow ('000 US$)	2,410,452

table continues…

Scenario	Value
DCF NPV ('000 US$) at:	
0.00%	2,410,452
5.00%	843,230
8.00%	416,125
12.00%	108,196
Payback (years)	7.8
IRR (%)	14.5

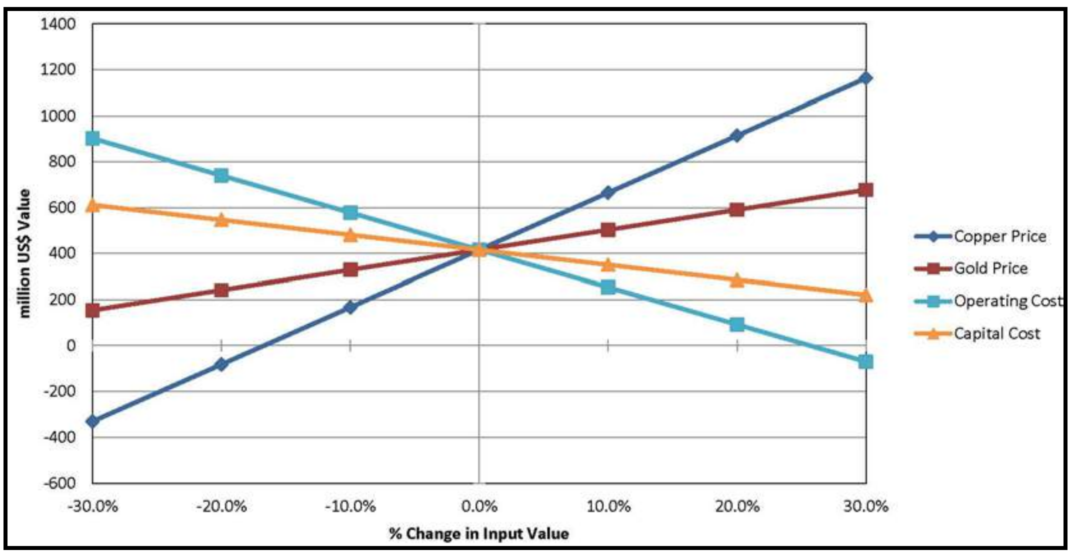
22.2.4 *SENSITIVITY ANALYSIS*

Sensitivity analysis was carried out on the following parameters:

- copper price
- gold price
- initial capital expenditure (CAPEX)
- mine site operating costs (OPEX)

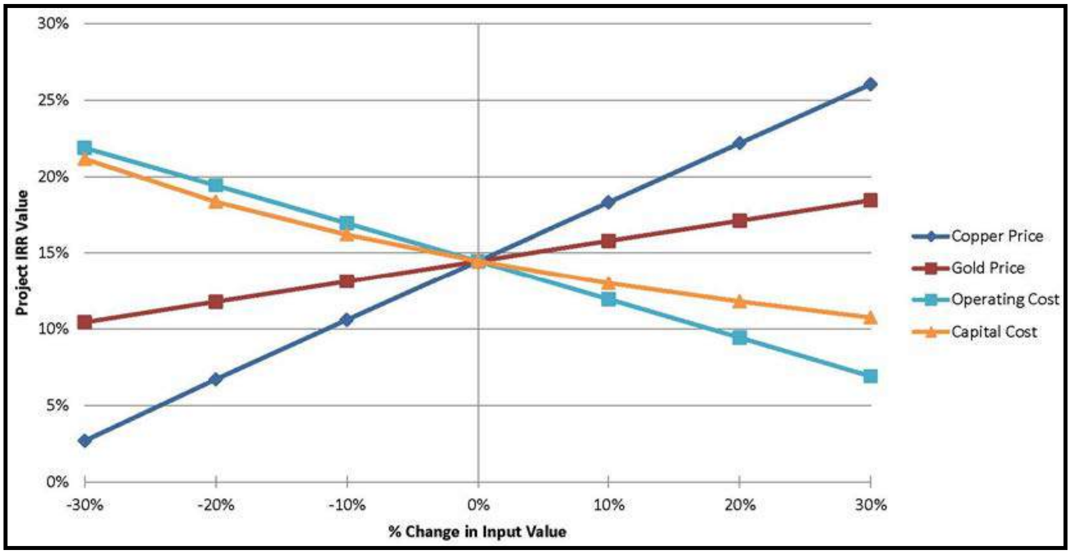
The analysis is presented graphically in Figure 22.2 and Figure 22.3 as financial outcomes in terms of NPV and IRR. The Project NPV (8% discount) is most sensitive to copper price, and is inversely most sensitive to operating cost.

Figure 22.2 NPV Sensitivity Analysis



Similarly, the project IRR is most sensitive to copper price and inversely to operating cost.

Figure 22.3 IRR Sensitivity Analysis



22.2.5 *Royalties*

No royalties were calculated for this analysis.

22.3 Smelter Terms

Copper concentrate contracts will generally include payment terms as follows:

- **copper** – pay 96.5% of copper content less a refining charge of US$0.068/accountable pound
- **gold** – pay 97.5% on the gold content less a refining charge of US$5/accountable troy ounce
- **Treatment Charge** – US$68/dmt of concentrate delivered.
- **Penalty Charge** – Penalty charges were applied as listed below:
  - Moisture $3.00 per 1% if greater than 11%
  - Arsenic $3.00 per 0.1% if greater than 0.1%
  - Antimony $3.00 per 0.1% if greater than 0.1%
  - Bismuth $3.00 per 0.01% if greater than 0.03%
  - Zinc $2.00 per 1% if greater than 2%
  - Aluminum $3.00 per 1% if greater than 3%
  - Selenium $4.00 per 0.01% if greater than 0.05%
  - Mercury $10.00 per 50 ppm if greater than 100 ppm
  - Combined Ni/Co $0.30 per 100 ppm if greater than 500 ppm.

22.4 Concentrate Transport Logistics

Concentrate from the mine site will be transported by truck to the Port of Vancouver. Wardrop conducted a detailed logistic report which can be found in Appendix K. Transportation charges have been based on average concentrate tonnage of 197,436 t/a and are as follows:

- rail transport – US$27.90/wmt
- stevedoring (port storage handling) – US$33.00/wmt, reduced to US$21.00/wmt from Year 6 to end of LOM
- ocean transport to Asian port – US$50.00/wmt
- moisture content – 8.5%.

22.4.1 *Owners Representation*

For a 10,000 wmt shipment lot, a charge of US$24,700 would be applied for services provided by the Owner's representative. Duties would include attendance during vessel unloading at the smelter port, supervising the taking of samples for assaying, and determining moisture content.

22.4.2 *Concentrate Losses*

Concentrate losses are estimated at US$1.50/dmt during shipment from the mine to smelter. For deliveries to Asia, the concentration losses will include losses due to the following handlings:

- loading truck at mine
- off-loading truck at port
- reloading vessels at port
- off-loading at port storage shed
- loading vessel
- off-loading vessel into truck transport to smelter
- off-loading truck into smelter storage bins.

23.0 ADJACENT PROPERTIES

The New Afton copper-gold project, being developed by New Gold is located approximately 10 km northwest of the Ajax property. Mineralization at New Afton is hosted in the Cherry Creek phase of the Iron Mask Batholith. The mineralization is controlled by east-northeast striking steeply dipping faults that are constrained within a west northwest striking structural corridor. The New Afton deposit is under development and will be mined using underground block cave method. Full production is scheduled to start in June 2012. As of December 31, 2010 New Gold reported a Proven and Probable Mineral Reserve for the New Afton project of 47 Mt grading 0.95% Cu, 0.69 g/t Au, 2.03 g/t Ag.

Information about the New Afton project was compiled from reports filed on SEDAR by New Gold. This information has not been verified. It is stated here for reference only and is not necessarily indicative of the mineralization to be found on the Ajax property.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION

24.1.1 PROJECT EXECUTION PLAN

PURPOSE

This Project Execution Plan (the Plan) for the feasibility study defines the relationship between engineering, procurement, and construction activities. The Plan is designed to:

- provide effective safety, environmental monitoring and quality methods
- effectively manage mine development
- reduce project scheduling risks
- limit or avoid capital cost overruns
- consider constructability issues during design
- establish communications between various project groups
- take advantage of cost and timeline reductions
- work with the vendors and contractors to provide quality products, services and costs
- provide expected deliverables in a timely and efficient manner
- reduce long-lead item delivery times
- limit escalation
- take advantage of bulk material pricing
- develop an effective cost-reporting and control system
- develop a cost-forecasting system
- establish meaningful project procedures.

IMPLEMENTATION STRATEGY

The Owner will design and direct pre-production mine development activities, and appoint a small project team to oversee project execution.

The initial steps of the implementation strategy will include:

- The selection of an engineering, procurement and construction management (EPCM) company to undertake the design of the material handling, process plant, TSF, and infrastructure facilities and to manage the full project construction activity.
- Ordering long-lead equipment items.

The selected EPCM contractors will have proven engineering, procurement and project delivery skills. Project risks will be managed using the EPCM scheduling, cost reporting and control, and purchasing and contracting strategy. The construction contracting strategy may be based on either:

- the award of multiple contracts commensurate with when the various engineering components will be delivered
- a general contractor that will supply the majority of labour and equipment and make use of specialty sub-contractors as needed.

Separate construction contracts will be awarded for the construction of the TSF, access roads and transmission line work. The decision as to which method will be used for the remainder of the work will depend on:

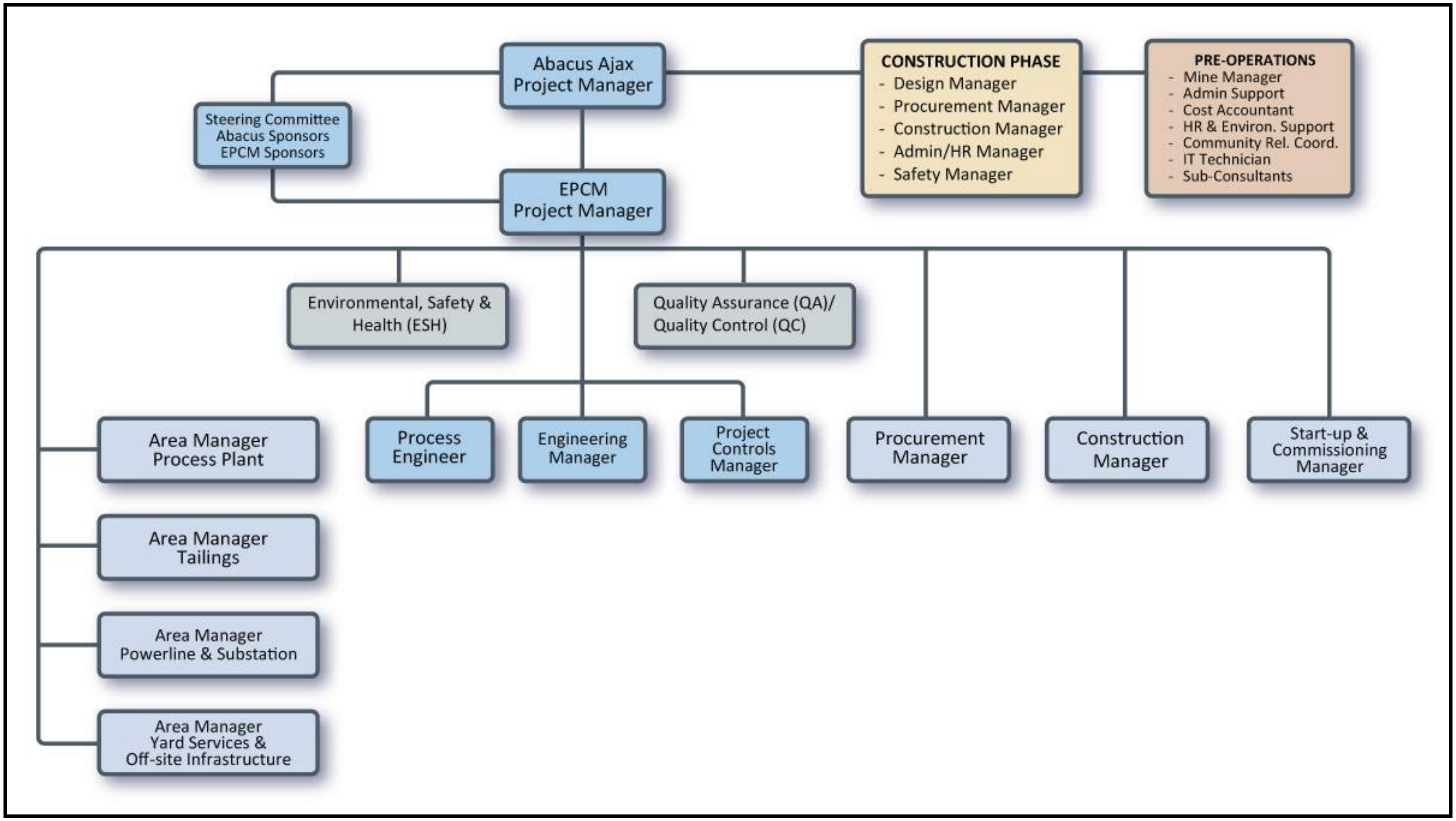
- the degree of engineering work completed at the time fieldwork is scheduled
- the availability and quality of contracted resources
- cost advantages of selecting one method over another
- the ability to define as much of the construction work as possible as lump sum
- the timing of the work over the construction period
- project cash flow
- permits and approvals.

PROJECT ORGANIZATION AND MANAGEMENT

Organization Overview

Working under the administration of the Owner's project team, the EPCM contractors will provide their own management teams. The management team will be led by a Project Manager, who will be responsible for the execution of the project's established schedule, capital cost, safety, environmental, and quality targets. The organizational chart is shown in Figure 24.1.

Figure 24.1 Project Management Organization Chart



Owner's Responsibilities

The Owner will retain responsibility for the following activities:

- mine planning and early-mine operations
- communication with local communities and the media
- accounting and invoice payment
- all permits or licenses required for construction, operation, and environmental compliance
- operation staffing, operator training, commissioning, and start-up.

EPCM Contractor's Responsibilities

The EPCM teams will establish a project management system that will contribute to the overall cost reporting and scheduling for the Project, as well as establish a comprehensive baseline plan to monitor project progress and productivities. This plan will help project teams quickly identify and correct early departures from the plan.

Project Procedures Manual

A comprehensive Project Procedures Manual will outline the procedures and requirements for the execution of the project administrative activities, as well as Owner and EPCM contractor rights, authorities and obligations. The overall Project Procedures Manual will be prepared by the EPCM contractor and then issued to the Owner for review and comment prior to formal issue. The procedures will include:

- project organization, key names, and communication procedures
- identification of the division of responsibilities between the project stakeholders using a responsibility matrix format
- reporting requirements, including project systems, project meetings, minutes and a communications matrix
- project data management, format and distribution/filing requirements of project correspondence and documentation
- drawing and specification preparation, including numbering, levels of issue and transmittal procedures
- site procedures, including safety, environmental, emergency numbers, camp administration rules, security and quality requirements
- equipment and materials procurement and construction contract procedures
- scheduling requirements, formats and issue times
- cost reporting and forecasting

- commissioning – partial until the commissioning team has been established by the Owner
- site and office personnel rules and regulations.

24.1.2 ENGINEERING

ENGINEERING STRATEGY

Primary out-sourced engineering work will include the following categories:

- water, waste and tailing design
- process facilities, site development and infrastructure, including the main access road, tailings thickener, tailings dam and the transmission line
- mine design.

DETAIL ENGINEERING

Detail engineering tasks will include the following contents:

- Additional geotechnical investigations, as required, to support detailed design. Complete the engineering calculations associated with the construction of new facilities.
- Produce construction drawings based on the results of site investigations, including surveys, inspections, and commensurate with the details set out in the feasibility study to as great an extent as possible, specifically including:
  - Process and Utility Flow Diagrams (PFDs) will be updated from Feasibility level and finalized to include major equipment vendor information. PFDs will also be rebalanced for all existing and new process equipment.
  - The Equipment List will be updated to include specific information from the firm-price bids for each selected vendor.
  - Piping & Instrumentation Diagrams (P&IDs) will be updated to include major equipment vendor information.
  - The Preliminary Instrument List will be updated based on the P&IDs.
  - The General Arrangement Drawings for new facilities will be advanced to incorporate certified vendor information as received. Secondary equipment, including main pipe headers, ducting, A/C equipment, etc., will be incorporated into the layouts.
  - Preliminary primary and secondary distribution Electrical Single Line Diagrams (SLDs) will be updated or prepared for the Project.
  - Electrical Load Analysis and Load Flow Reports will be completed.

- Produce specifications associated with the construction of new facilities, specifically including:
  - Specifications will be developed to obtain competitive and firm price bids for major equipment that requires a long lead for delivery, or which needs to be purchased to avoid high future escalation. All manufacturing hold points will be included where the engineer may decide that inspection is required. Bids will be technically and commercially evaluated by the EPCM contractors and recommendations made to the Owner for purchasing.
  - Develop equipment specifications for the remainder of the non-long lead equipment together with the commercial and technical analysis and recommended for purchase to the Owner. All manufacturing hold points are to be included in the specifications.
  - Provide material requisitions for the purchase of bulk materials based on material take-offs for items such as electrical cables, cable trays, hi-bay lighting, and piping.
  - Provide the construction technical specifications to be included in the construction contract bid packages, as well as pertinent drawings, vendor information when available and quantities for the Form of Tender. Also provide all technical reports that will be referred to by the contractors when bidding the construction work for items such as the geotechnical analysis, studies, and special information from vendors for installation of equipment and special requirements for handling.
- Once the capital cost estimate (CAPEX) has been approved, it will be "converted" to a control budget, updated with recent cost information including the values of the released purchase orders.
- Purchasing procedures and standard documentation will be developed. Purchase orders for long delivery equipment will be issued by the EPCM contractor on behalf of the Owner.
- The project overall schedule will be updated.
- The engineering contractor will provide licensing and permitting assistance to the Owner as required.
- A construction contract boiler plate will be developed for the Owner's approval.

24.1.3 PROCUREMENT AND CONTRACTS

PROCUREMENTS AND EXPEDITING

The EPCM contractor's Purchasing Group will provide capital equipment procurement, vendor drawing expediting and, when required, equipment inspection. The procurement department will package the technical and commercial documentation and manage the bidding cycle for equipment and materials to be supplied by the Owner to the contractors. Standard procurement terms and conditions approved for the project will be utilized for all equipment and materials purchase orders. Suppliers will be selected based on locations, quality, price, delivery and support service.

The Construction Management (CM) group will organize bulk materials purchases, assemble contract tendering documents, establish qualified bid lists, issue tenders, analyze and recommend suitably qualified contractors to the Owner, and prepare executed contracts for issue.

A field procurement manager will support ongoing construction needs for miscellaneous materials and services to be provided by the Owner as well as provide expediting services. They will also be responsible for the receipt, storage and disbursement of purchased materials and equipment at the job site.

A plan for expediting equipment purchase orders based on the project schedule and equipment list will be prepared. Expediting will be coordinated by the construction group. Third-party resources may be used to inspect equipment in various parts of the world during manufacture. The extent with which purchase orders will be expedited will be based on the order's complexity, manufacturing cycle time and schedule criticality. For purchase orders requiring the highest level of expedition, a resident expediter will be placed at the supplier's facility. Expediting reports will be entered into the material control reporting system after each contact with suppliers.

Logistics

The EPCM team will direct logistics and freight for incoming equipment and materials, to be transported by truck through Vancouver where possible. A single-point freight forwarding company will coordinate with manufacturing facilities, establish shipping points and dates, forward the shipments to the most convenient ports, and complete trans-shipments to the project site.

An experienced freight forwarder with representation in BC will function under the direction of the EPCM contractor, who will issue timely shipment and materials status reports.

24.1.4 Construction

Construction Management

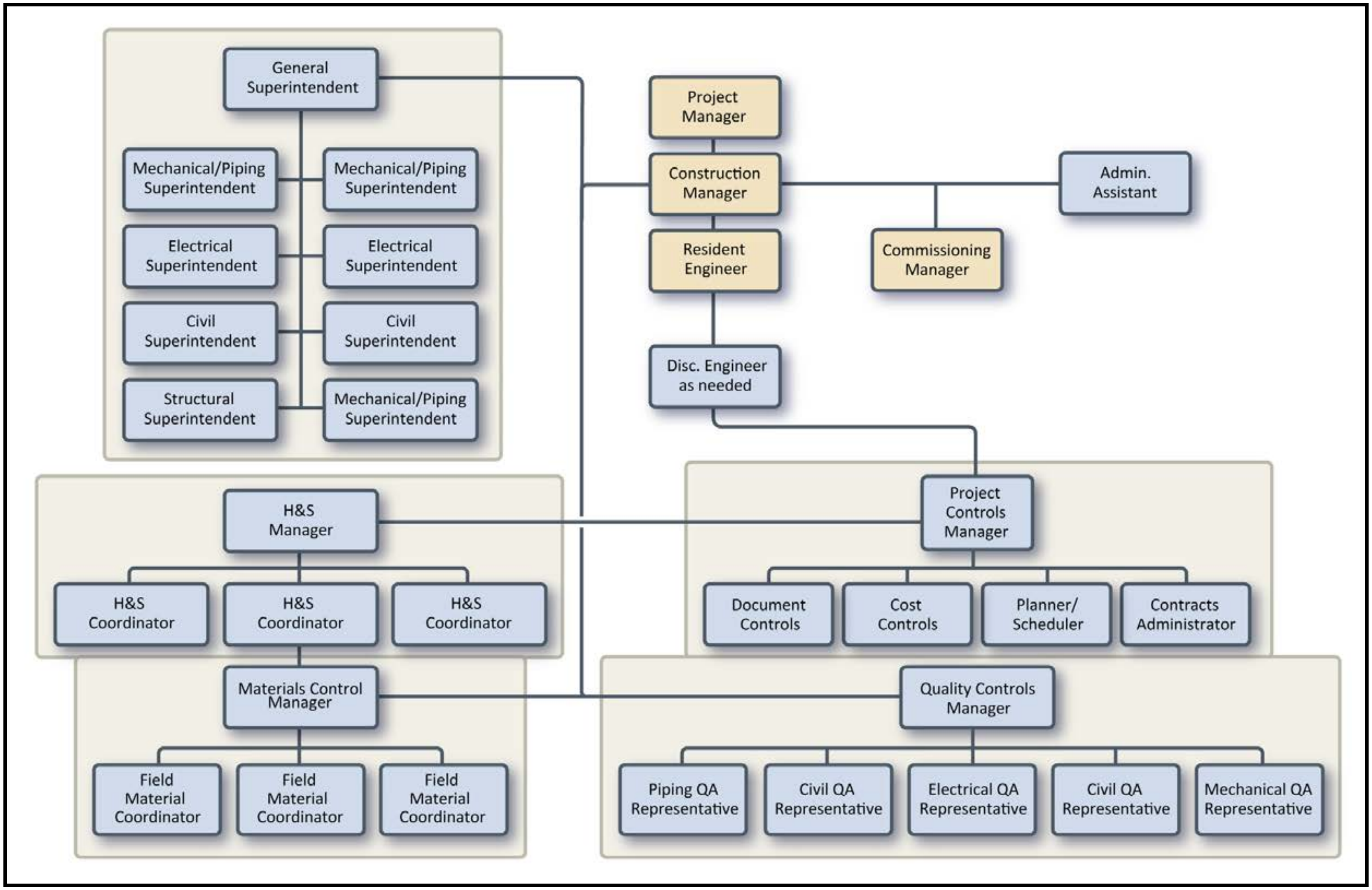
The CM group will be responsible for the management of all field operations. Reporting to the Owner, the Construction Manager will plan, organize and manage construction quality, safety, budget and schedule objectives.

Construction of the Project will be performed by contractors under the direction of the CM team, reporting to the Owner's representative. The CM key objectives are to:

- Conduct Environmental, Health and Safety policy training and enforcement for all site and contractor staff. Site hazard management tools and programs will be employed to achieve the no harm/zero accident objective.
- Apply contracting and construction infrastructure strategies to support the project execution requirements.
- Develop and implement a construction-sensitive and cost-effective master project schedule.
- Establish a project cost control system to ensure effective cost reporting, monitoring and forecasting as well as schedule reporting and control. A cost trending programme will be instigated whereby the EPCM contractor will be responsible for evaluating costs on an ongoing basis for comparison to budget and forecasting for the cost report on monthly basis.
- Establish a field contract administration system to effectively manage, control and coordinate the work performed by the contractors.
- Apply an effective field constructability program, as a continuation of the constructability reviews performed in the design office.
- To develop a detailed field logistics and material control plan to maintain the necessary flow and control of material and equipment to support construction operations.
- Meet the schedule for handover of the constructed plant to the commissioning team.

The Construction Organization Chart (Figure 24.2) shows the CM team organization plan for the site.

Figure 24.2 Construction Management Organization Chart



FIELD ENGINEERING

Surveying

The CM survey crew will verify the accuracy of the existing control system before construction begins. Contractors will use only applicable control data for the project. Additional monuments will be set as needed. The Construction Manager will verify surveys prior to construction. Contractors will supervise day-to-day field surveying, and the CM team will provide spot checks.

Quality Control/Quality Assurance

Contractors will establish and observe their own Quality Control program in accordance with the construction technical specifications and the applicable codes and standards. The CM Field Engineering Team will employ independent CSA-qualified Quality Assurance specialists to ensure quality control.

CONSTRUCTION CONTRACTING

The contracting strategy will be designed to make maximum use of the local labour force, create a responsible and sustainable relationship with the nearby communities and provide a mix of senior management and specialists to support the safety, quality, schedule and cost objectives of the project. In addition, contracts will be designed to combine timing, scope, battery limits and contract value into manageable packages.

Construction contracts will designed based on geography and/or discipline. Contract packages will be designed to take advantage of a staggered work schedule; some contracts, such as transmission line, truck fuel station and truck shop construction, may be issued before the scope of the overall project is fully defined.

Table 24.1 outlines the major construction contract scope and pricing strategies for individually-issued contracts. Additional contracts and options will be made available during the execution phase.

Table 24.1 Summary of Contracts

Contract Description	Type	Comments
Plant Site Preparation & Tailing Construction	Unit Price	Includes tailing extension work during operating years
230 kV transmission line – design/build	Lump Sum	BC Hydro
Fresh Water Supply	Lump Sum	None
Truck Fuel Station – design/build	Lump Sum	None
Truckshop and Warehouse – design/build	Lump Sum	Pre-engineered
Administration Complex – design/build	Lump Sum	Modular
Concrete Supply	Unit Price	None
Concrete Installation	Unit Price	None
Structural Steel Installation	Unit Price	None
Cladding & Roofing	Unit Price	None
Mechanical & Piping	Fixed Price	None
Insulation	Fixed Price	None
Flotation Cells – design/build	Fixed Price	Technology vendor
Electrical & Instrumentation	Fixed Price	None
Process Control	Fixed Price	None
Architectural Finishes	Fixed Price	None

Approved contract *pro formas* will be utilized for construction and service contracts. Construction contractors will be responsible for:

- all construction labour
- all construction equipment
- worker transportation
- site offices and temporary services
- site management
- contractor surveying
- quality control
- contract scheduling
- safety
- environmental safeguarding
- security for their tools and equipment
- supplying permanent materials as required by contract.

The Owner will provide contractors and construction management staff with:

- on-site, project-wide first aid services
- project-wide security
- locations for offices and equipment/material lay down
- local electrical panels and temporary generator sets
- sources for water
- diesel fuel, including storage, for construction equipment
- sources for all concrete and structural aggregates
- permanent bulk materials and all capital equipment
- audits of quality assurance and control
- vendor-representative assistance.

MATERIALS MANAGEMENT

Warehousing

The Site Materials Management group will receive, inspect, and log all incoming materials, assign storage locations, and maintain a database all materials received and dispensed to the contractors. Ongoing reconciliation with the procurement system, including reconciliation to the freight consolidation point, will confirm the receipt of materials and payment of suppliers. An allowance for the lease or purchase of warehousing equipment has been included in the construction budget.

Site Housekeeping and Hazardous Waste Management

Procedures for waste management and spill response will be implemented for the construction period. These procedures will be established in the Project Procedures manual, and will and outline compliance, auditing and reporting requirements. Procedures regarding ongoing cleanup and rubbish removal, as well as safe handling, storage and disposal of batteries, fuels, oil and hazardous materials, will be established and observed for the duration of the construction phase. Waste will be recycled to the extent feasible. Ongoing dust suppression and rain water management programs will also be established and followed for the duration of the construction phase. Specific procedures and storage areas will be designated for the demolition waste prior to recycling or removal from the plant.

CONSTRUCTION EQUIPMENT

Individual contractors will be responsible for the equipment required to meet their contract obligations. All equipment must comply with Mine Safety Branch requirements; the Owner's CM team will perform regular spot checks. Large cranes may be supplied by a single company, managed by the CM team.

CONSTRUCTION CAMPS

No construction camp is required. All contractors including transmission people and Owner's operating personnel (including mining pre-stripping operators) will reside in Kamloops.

TEMPORARY FACILITIES AND CONSTRUCTION SITE INFRASTRUCTURE

Communication

The Owner's systems manager will determine the appropriate telecommunications technologies for the project. Requirements include voice and data link technologies adequate to support growth construction and plant operation growth.

The communications framework for management offices will be installed early in the construction period. The system will be supplemented with additional telephones in common areas and individual room internet connections.

Construction Power

Approximately 1 MW of operating power capacity and one spare for camp utilities (excluding heat) and limited construction facilities will be required during the 2013 and 2014 construction period. At the end of 2014, permanent power will supply all mine equipment and peak construction power loads for the remaining duration of the construction phase.

First-Aid and Site Security

The Owner will provide a fully-equipped first-aid facility, ambulance and fire engine for project-wide use. The facility will normally be staffed 12 h/d, with on-call services ensuring continuous coverage. Contractors will be expected to provide basic first-aid stations at the site.

The Owner will supply a 24-hour staffed site security program during the initial field mobilization in 2013. Access to the site will be controlled at the principal road entrance and will be limited to personnel who have attended induction training, as well as approved visitors.

EARLY CONSTRUCTION ACTIVITIES

Receipt of the construction permits in March 2013 will launch initial construction activity that will focus on:

- construction management team complex set-up
- road improvement
- tailing thickening and tailings dam area clearing and grubbing
- fresh water supply
- explosives storage
- earthworks for the crusher, mill site, truck shop and administration office construction
- supply of materials for administration office erection
- transmission line.

TAILING THICKENER AND TAILINGS STORAGE FACILITY CONSTRUCTION

Development of the TSF development will be in stages, beginning with a starter dam in 2013, utilizing local borrow sources. The dam extensions and development in 2016 and 2017 will use a combination of borrow sources and mine-stripping materials. Construction will continue during the operating years, using a combination of borrow sources and mined overburden and waste rock.

The Mine Operations Group will support TSF construction; to begin in 2013 with mine area stripping and the construction of preliminary mine haul roads to TSF embankments.

When full-scale mine stripping begins, materials will be hauled by the mining fleet to the TSF embankments; the contractor will be responsible for the placement and compaction of the core materials, and the construction and management of the sedimentation control structures.

The engineer's quality specifications will govern TSF construction. The TSF components that have "no-build" restrictions during the winter and during the spring melt are:

- topsoil stripping
- dam footprint foundation preparation
- foundation drains
- embankment drains
- cut-off trench excavation and backfill

- core, filter and transition zone materials
- basin area glacial till liner.

HEALTH, SAFETY AND ENVIRONMENTAL

A fully-integrated workplace safety and environmental responsibility program will be implemented to help achieve a "zero-harm" goal. The development of effective Health, Safety and Environmental (HS&E) practices will require a high level of communication, motivation and involvement, including alignment with site contractors on topics such as safety training, occupational health and hygiene, hazard and risk awareness, safe systems of work and job safety analysis. Tools will be implemented for performance tracking and accountability, including procedures for incident management.

All contractors will be required to create individual safety programs and safety plans that meet the expectations of the CM team. These Safety Management Manuals will be issued at the specific contractor kick-off meeting and will be reviewed by the CM team and any updates, corrections and inclusions will be addressed by the contractor prior to mobilization.

The Project was designed using sound environmental protective approaches that impact decisions such as the location of mine, mill, and related structures upslope of the TSF.

All design and engineering stages incorporate criteria for responsible methods of process flows, effluent, and waste products, in order to meet established capture and containment guidelines. The design also incorporates clean plant design standards, including operational safety and maintenance access requirements.

A Hazard and Operability Analysis (HAZOP) will be conducted by the project design team during each plant area's detailed design stage. This systematic team approach will identify hazards during the design phase that require attention. Environmental protection will be incorporated in both the design of the main plant processes, as well transportation, storage and material disposal.

Safety tagging and lockout procedures will be established in order to identify the status of equipment and systems, and to identify equipment "ownership" as the commissioning process moves forward.

24.1.5 *PRE-PRODUCTION MINE DEVELOPMENT*

The delivery of mine support equipment is expected in March 2013; by late 2014, mining operations will include topsoil stripping, clearing and grubbing of the mine area, and mine haul roads construction.

The main transformer will be operational in late 2014, coinciding with the delivery of the first mine shovel and mine trucks. Mine pre-stripping will commence in March 2014, delivering overburden and waste rock to construct the TSF Dam. Ore will be exposed for delivery to the mill in Q2 2015.

24.1.6 PROJECT CONTROLS

COST

The project WBS defines the elements of project scope. The first step in the project implementation process will be to confirm the project WBS and distribute the control estimate.

The definitive estimate produced at the conclusion of basic engineering will become the control estimate for project execution. Budgets will then be cast for the approved scope, which will become the baseline control document against which the Project will be measured.

Cost management allows the EPCM team to quickly evaluate and minimize any potential negative effects of capital control budget deviations. A project cost management system will provide comprehensive cost reporting, forecasting and trending.

The Project team will implement a coordinated program to review the project Estimate at Completion (EAC), together with engineering, procurement and the construction management team. This detailed EAC review will be performed at regular intervals for each procurement and construction package to reflect the latest available trends. These include changes in quantities, unit prices, site conditions, delays, interferences and other factors that may influence cost.

A cash flow will be established once the contracting strategy and construction schedules are finalized and the estimate is recast into procurement packages. Cash flow will be generated to support the client in estimating anticipated cash requirements for the Project. As the project advances, actual expenditures will be recorded and compared to the original cash flow estimate. Deviations from the scope of the control estimate will be evaluated with respect to cost and schedule impact against the funds approved for the Project.

PLANNING

The planning process identifies critical sequences that must be managed to execute the Project successfully. The project master schedule highlights project milestones and critical sequences. Executive level reports provide an ongoing overview of project status. Detailed schedules track the planned and actual progress throughout the duration of the Project.

Scheduled activities are detailed to the engineering work packages level in the engineering work plan. Procurement activities are determined by the equipment list, matched to the procurement packaging strategy, tracking not vendor fabrication/delivery requirements as well as engineering information exchange and drawing reviews.

The construction contracting strategy provides the engineering schedule linked to construction contract scope, engineering deliverables and procured equipment and services. Construction, commissioning and start-up schedules are developed using input from engineering, construction and operations groups. Construction contract milestone dates are determined for each tender package and detailed contractor schedules are closely monitored to the master schedule.

Critical sequences and tie-ins are micro-scheduled as needed in the construction contractor's work plan.

Progress reporting is based on earned or weighted value progress measurement for work in place, trend analysis and other techniques as required to ensure precise reporting of project status.

Document Control

Effective document management is crucial to the successful implementation of the Project. A collaborative document control system will be implemented that provides status and version control for all issued documents. The system will be capable of publishing documents, text, drawings, photographs or 3D models to the internet. Documents will be linked to the equipment database to ease the delivery of an accessible control system to the operations group at the completion of the design and construction phases.

Risk Management Strategy

The project will include risk reviews of safety, environmental, technical, business and commercial, and project delivery issues. These reviews will identify and assess relevant risks and determine mitigation strategies.

A formal risk management program will commence during the feasibility study phase and will continue through commissioning. The Project team will periodically review risks and opportunities and take appropriate action to minimize the impact on overall costs and scheduling.

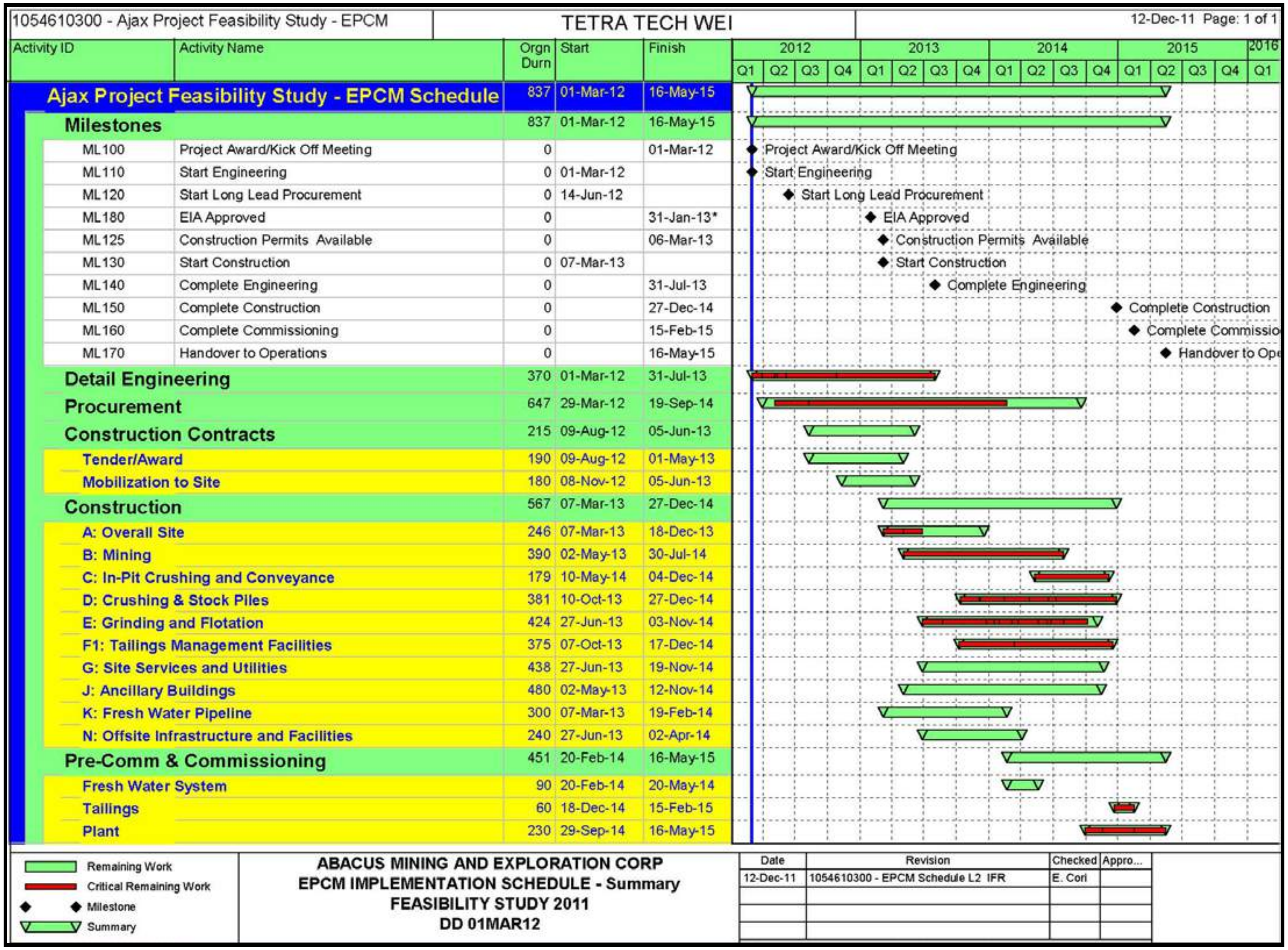
24.1.7 PROJECT SCHEDULE

GENERAL

The first scheduled project implementation tasks are long delivery equipment (grinding mills, shovels, trucks, stacking system and primary crusher) ordering, process facilities basic engineering commencement, and facilities construction. The overall project execution duration, from commencement to plant commissioning, is expected to be 38 mo. The duration from commencement of field construction to mechanical completion (substantially) is 22 mo (including access roads and early earthworks). The overall duration assumes that required financing will be in place to allow all phases of the Project to proceed at their projected start times, and that all permits will be in place for the work to proceed as planned and without stoppages.

The Project development schedule is summarized in Figure 24.3. The full schedule is provided in Appendix L.

Figure 24.3 Preliminary Construction Schedule

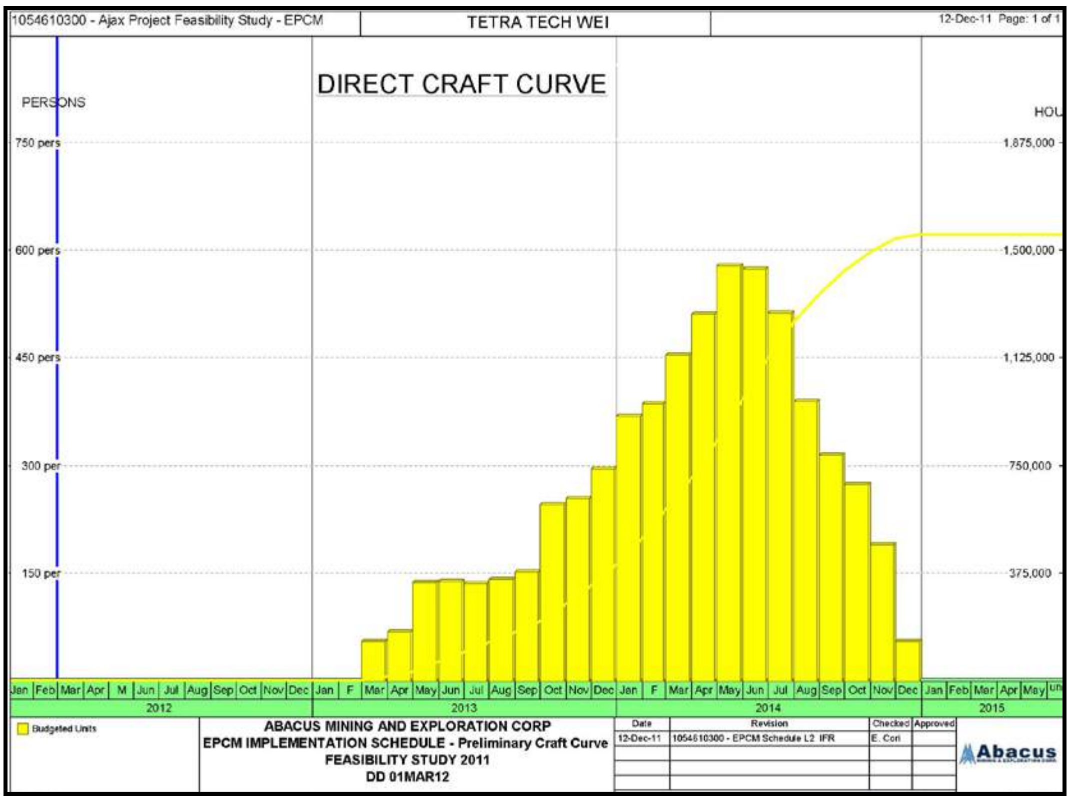


CONSTRUCTION LABOUR

The schedule is based on a 60 h work week. Crew rotations are assumed to be three weeks onsite and one week offsite.

It is anticipated that there are about one million direct construction man-hours, excluding mine pre-development and the main transmission line set up. A peak about 580 persons is also expected during the construction phase of the Project, again excluding mine pre-development and the main transmission line set up. The distribution of labour is shown Figure 24.4.

Figure 24.4 Preliminary Construction Manpower

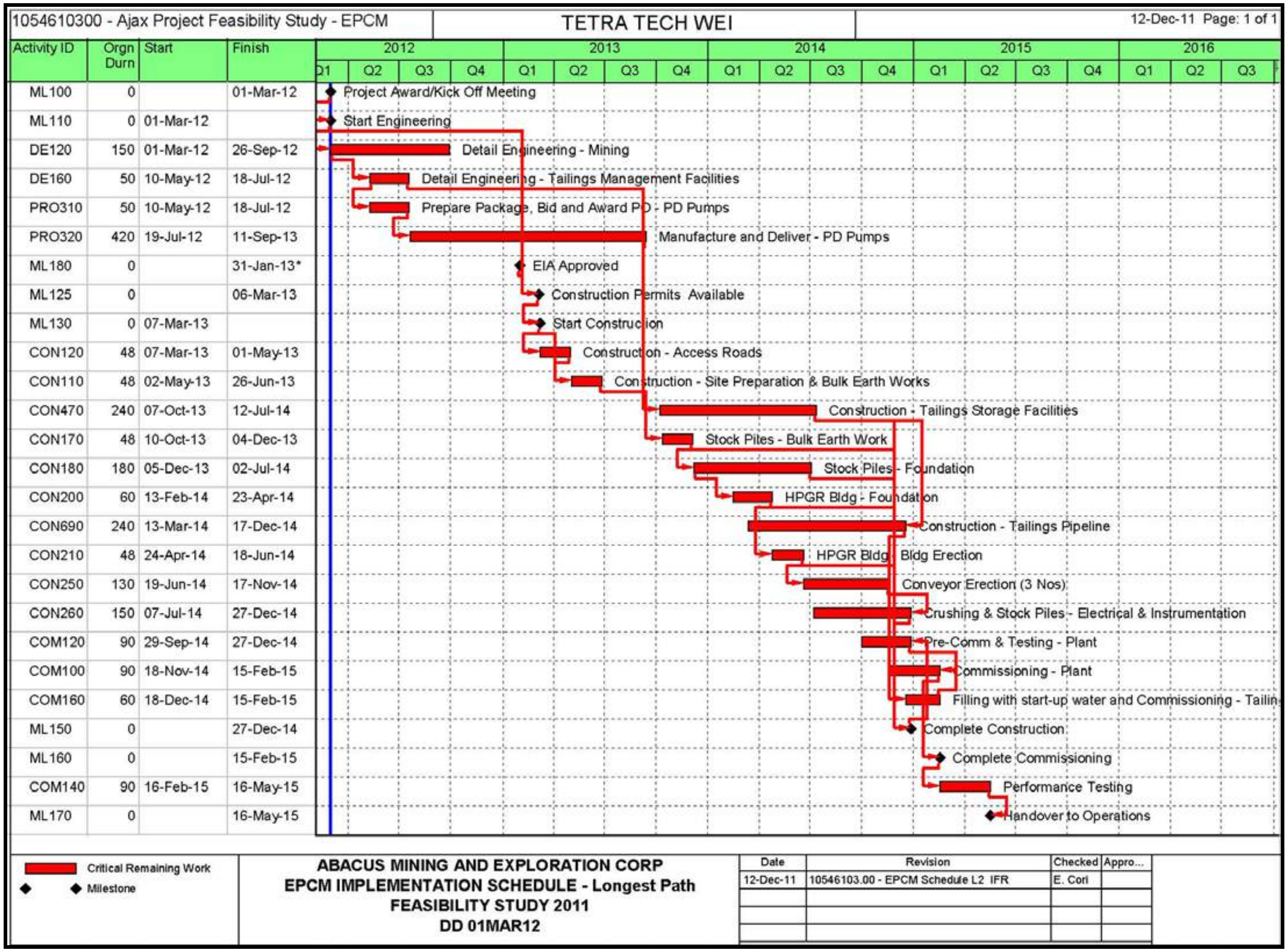


MILESTONES AND SCHEDULE STRATEGY

The schedule reflects a traditional approach to project execution. Field construction will being only after engineering tasks are adequately advanced, in order to accommodate major equipment long lead delivery times. Engineering will continue early in the Project, and many of the construction work packages can be issued on a fixed-price basis, thus reducing cost risk.

The project schedule requires interim project financing shortly after the completion of the feasibility study. This interim financing will accommodate long delivery items and power transformer preparedness. The anticipated longest path of the project is shown in Figure 24.5.

Figure 24.5 Anticipated Longest Path



CRITICAL ACTIVITIES

The availability of early interim financing is critical to complete operational commissioning in mid February 2015.

After the completion of the feasibility study, the following tasks should be completed as soon as possible:

- the execution of the EPCM contract will ensure that basic engineering commences, allowing a level of design adequate to order long delivery items
- transmission line and fresh water system design
- TSF design
- long delivery capital equipment ordering
- permits must be in place to allow full construction to proceed in March 2013
- Operators must be trained and ready to take over the operation of the facility upon project turnover.

24.1.8 MECHANICAL COMPLETION AND COMMISSIONING

MECHANICAL COMPLETION

Mechanical Completion indicates that systems are ready for safe dry-testing. It also indicates that final work inspections can be completed, contract deficiency lists generated and resolved, and documentation for contract closeouts can be processed. Mechanical Completion poses the greatest safety risk as construction concludes and systems and equipment are tested. An added emphasis will be placed on communication, safety procedures, and the coordination of construction and commissioning personnel.

Each process system and ancillary facility will be checked for compliance with drawings and specifications, vendor data, and lubrication charts. Mechanical equipment will be checked for proper installation, alignment, and rotation. Conveyors will be run without load to verify belt alignment. Tanks and piping will be water/air tested. Electrical equipment and circuits will be checked for proper installation. Instrumentation circuits will be checked and instruments will be zero-calibrated. When all installations have been verified, each system will be run under no-load conditions. Permanent records will be maintained for each piece of equipment.

The project will phase the mechanical completion sequence to level the resource requirements and stage the transition from construction to operation. Critical site infrastructure facilities will be completed first, followed by critical process utilities commissioning, and then the main process equipment.

Mechanical Completion and Commissioning Interface

With the exception of mining activities, the EPCM contractor will have responsibility over the facilities until Mechanical Completion. Then, system-by-system, the plant will be transferred to the Owners' operations team for commissioning. The final Mechanical Completion of the grinding area will be 22 mo after the commencement of construction.

Detailed responsibility matrices describing the mechanical completion and commissioning phase will be documented in the Project Procedures manual. The Start-up and Commissioning Procedures manual will address administrative procedures, pre-operational testing procedures, commissioning and ramp-up procedures, and performance test procedures.

After Mechanical Completion, the EPCM contractor and the construction contractors will provide assistance during the commissioning and start-up phase as requested.

Plant Operation and Performance Testing

The plant operations group will be responsible for plant operation and performance testing, including facility operation at various design sizes and capacities. The group may request support from equipment suppliers and process design engineers to ensure optimum equipment performance.

The Owner's operational personnel will perform performance tests, with assistance from suppliers. Selected members of the project commissioning team may be asked to assist as required.

24.2 Social and Community Interaction

KAM is working with the Tk'emlups and Skeetchestn Bands of the Secwepemc Nation to develop an Impacts Benefits Agreement (IBA). This agreement is signed between mining companies and First Nation communities in Canada in order to establish formal relationships to reduce the predicted impact of a mine as well as secure economic benefits for affected communities.

25.0 INTERPRETATION AND CONCLUSIONS

25.1. MINE DEVELOPMENT (AMEC)

The AMEC QPs consider that requirements for infrastructure, workforce, and power, water and communication facilities to support future mining operations are well understood by Abacus, and the understanding is sufficient to support Mineral Resource and Mineral Reserve declaration. There is sufficient area within the Ajax ground holdings to support development of an open pit. Additional ground that hosts plant site infrastructure is held through an agreement completed in 2005 with Afton OC. The Afton OC ground also hosts an existing tailings facility and shop facilities. It is a reasonable expectation that appropriate surface rights to support Project development and operations can be obtained, and therefore declaration of Mineral Resources and Mineral Reserves can be supported. It is expected that any future mining operations will be able to be conducted year-round.

In the opinion of the AMEC QPs knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation. The exploration programs completed to date are appropriate to the style of the deposits and prospects within the Project. Additional exploration potential remains in the Project area.

From the AMEC review of the drill programs, the AMEC QPs concluded that documentation for the drilling campaigns completed in the Ajax area before 1980 is limited or not available. Because of the limited documentation, the pre-1980 drill campaigns were not included in the 2011 resource model database. No factors were identified with the data collection from the drill programs used that could affect Mineral Resource or Mineral Reserve estimation. The Ajax deposits have been sampled at drill-hole spacing appropriate for a property at this level of development. Because of the irregular shape of the mineralized body, the orientation of the mineralization with respect to drill intercepts is unknown.

In the opinion of the AMEC QPs, sample collection, preparation, analytical and QA/QC data from the Abacus drilling programs were appropriate and meet industry standards. AMEC concludes the Abacus analytical data can be used to support Mineral Resource and Mineral Reserve estimation without limitation. Because of the lack of information for Afton OC and Cominco drilling, AMEC compared these drill campaigns to Abacus drilling using paired samples. AMEC concluded that model blocks with grade estimates primarily supported by Afton OC drilling should be limited to an Indicated classification. Model blocks with grade estimates primarily supported by Cominco drilling were limited to an Inferred classification.

The AMEC QPs consider that a reasonable level of verification has been completed during the audits undertaken in 2008-2009, 2010 and 2011, and that no material issues would have been left unidentified from the audit programs undertaken. The data verification programs undertaken on the data collected from the Project adequately support the geological interpretations, the analytical and database quality, and therefore support the use of the data in Mineral Resource estimation. However, AMEC applied a confidence limitation during Mineral Resource block classification on legacy data support.

In the opinion of the AMEC QPs, estimations of Mineral Resources for the Project conform to industry best practices, and meet the requirements of CIM (2010). An LG shell has been used to constrain the estimate. Measured and Indicated Mineral Resources total 512 Mt grading 0.42 CuEq, 0.31% Cu and 0.19 g/t Au, at a 0.2% CuEq cut-off grade. In addition there are 73.7 Mt in the Inferred category, grading 0.38% CuEq, 0.27% Cu and 0.17 g/t Au at a 0.2% CuEq cut-off grade.

Factors which may affect the geological models or the conceptual pitshells used to constrain the mineral resources, and therefore the Mineral Resource estimates include: the commodity price assumptions; the NSR value used to constrain the Mineral Resources is based on technical and economic parameters supplied by Abacus, should these assumptions change, then the pit constraining the Mineral Resources will also change; metallurgical recovery assumptions; pit slope angles used to constrain the estimates; and the SG values assumed for the rock types.

In the opinion of the AMEC QPs, estimations of Mineral Reserves for the Project conform to industry best practices, and meet the requirements of CIM (2010). An LG shell has been used to constrain the estimate, and appropriate modifying factors have been applied to convert Measured and Indicated Mineral Resources to Mineral Reserves. The Proved and Probable Mineral Reserves total 503 Mt grading 0.37% CuEq, 0.27% Cu, and 0.17 g/t Au at a US$4.53 NSR cut-off.

Factors which may affect the Mineral Reserve estimate include gold and copper price assumptions, effectiveness of surface and groundwater management, the assumption that granting of appropriate environmental and construction permits would be forthcoming from the relevant authorities, unrecognized structural complications in areas with relatively low drillhole density, changes to the proposed methodology for waste material, and changes to the pitshell shape if the material currently under the oil pipeline can be incorporated in the pit design.

The proposed mine plan has been developed at a 60,000 t/d rate. The average head grade of process feed for the life-of-mine is 0.267% Cu and 0.170 g/t Au equivalent to an NSR of 14.68 expressed in dollars per tonne. Mining will be accomplished using a conventional truck and shovel operation, with in-pit crushing and conveying. The proposed mine has an expected mine life of 23 years, and an overall stripping ratio of 2.4:1 waste to ore.

25.2. PROCESSING PLANT DESIGN (WARDROP)

The Ajax concentrator has been designed to produce a saleable copper concentrate. The complete processing circuit has been designed according to metallurgical test work conducted over the years and proven equipment.

25.3. TAILINGS STORAGE FACILITY DESIGN (GOLDER)

Golder developed the feasibility level tailings management design from the pre-feasibility study incorporating with new test work results on the dewatering and rheological properties of the tailings. The capacity of the tailings management facility is dictated by the throughput of the processing plant, i.e. 60,000 t/d nominal concentrator solids feed rate, which is equivalent to 2,694 t/h tailings solids rate, at 92% processing availability.

Golder proposed the thickened, non-segregating tailings slurry discharge into a TSF for the Ajax operations, mainly consisting of a TTP, a tailings storage system, and two TSF water collection ponds.

The flotation tailings pumped from concentrator will be combined with flocculants, and any thickener underflow before feeding the two tailings thickeners. The thickener underflow streams will be pumped to the common thickened tailings tank for homogenize purpose, and then pumped to the TSF by five PD pumps. Thickener overflow water and TSF reclaim water will be piped to the process water tank for applications in TTP and milling process.

The total required TSF volume is 389.6 Mm^3, based on two assumptions: (1) the deposited thickened tailings void ratio is 0.7, and (2) the ultimate deposited thickened tailings dry density is 1.62 t/m^3. The TSF will be constructed in an upstream manner and thickened tailings will be discharged from spigot points. Waste rock from the open pit development will be used to construct an initial pipeline dyke and to raise the tailings pile in stages.

At closure, the reclamation of the tailings stack will consist of a closure dry cover placed over the entire tailings surface area. The TSF Pond will be decommissioned once discharge requirements are met and a stable trend is attained. The TSF Pond dyke will then be breached to form a wetland, and the remaining TSF Pond dyke will be re-graded.

For the purpose of this study non-acid generating tailings and mine waste rock were assumed.

26.0 RECOMMENDATIONS

26.1 MINING (AMEC)

The AMEC QPs recommend the following work items to support upgrades in Mineral Resource estimate confidence categories. This work can be completed as part of a single phase.

- All core recovery data to be captured into the database (estimated cost of US$2,000).
- The final coordinate values to be corrected in Abacus's database for AM-10-040 and the down-hole survey results for drillholes AE-06-008 and AW-06-021 be reviewed (estimated cost of US$200).
- Abacus does not submit coarse reject duplicates as part of their QA/QC program. Abacus should submit coarse reject duplicates to allow evaluation of the sample preparation performance at a frequency of one duplicate pair per 20 samples (estimated to add approximately 1% to the cost of future drill programs).
- Abacus should review the sample preparation procedures to improve the precision achieved by the assay laboratory. Possible improvements include finer crushing of the sample prior to splitting, better blending and/or splitting of the sample, and splitting of a larger sub-sample for pulverization. The AVRD charts indicate the results are acceptable for gold, but precision for copper could be increased (estimated cost of US$2,000).
- Abacus should re-log core to better define the albite alterations that are associated with higher-grade gold mineralization. The albite alteration can be added to the resource model (estimated cost US$10,000).
- AMEC recommends additional drilling in the East Extension area to increase the confidence of the mineral resource estimates in that area. This should include approximately 20 drillholes (16,000 m), at an approximate drill metre cost of US$75/m (estimated cost of US$120,000).
- Abacus should construct a "Due-To" table to identify the variables that have most affected the Ajax resource estimate and determine the effects of the drilling, recovery functions, costs and commodity prices (estimated cost of US$5,000).

The following work is recommended by the AMEC QPs in support of mine planning and Mineral Reserve estimates. This work should be performed concurrently with the geological and mineral resource program above.

- Before proceeding to development of this project, complete kinematic testing of the waste rock to finalize acid rock drainage (ARD) characterization. In the case that a substantial portion of the waste rock demonstrates acid generating potential, waste disposal procedure needs to be revised to mitigate potential acid generation as well as to minimize the potential for water treatment. If disposal of waste material requires different sequencing, or intensive layering, covering or any other procedure, it would likely impact the operating cost and closure costs assumed for support of Mineral Reserves (estimated cost of US$1 million to US$1.5 million).
- The pit design is based on the assumption that the Trans Mountain pipeline is relocated away from the pit during the first years of operation. Although the pit has been designed and scheduled in such a way that no excavation is considered on the ground below the pipeline and its right of way, an agreement with Kinder Morgan and a detailed relocation plan of this pipeline is needed at the early stages of the project (estimated cost of US$18 million).

26.2 Processing (Wardrop)

26.2.1 HPGR Circuit

The HPGR circuit requires a review of the available data with regard to the wet screening circuit and the return of the wet screen oversize either to the stockpile or directly back to the HPGR to join the new feed material.

26.2.2 Confirmation of Concentrate Quality

As a recommendation, the minor element assay data of the copper concentrate should be further reviewed by a concentrate marketing specialist to confirm that there will be no marketing issues with the occasional higher mercury content as has been indicated by the variability concentrate assays. Of these data, five of the 50 test concentrates contained mercury levels in excess of 10 ppm Hg.

26.2.3 Froth Cameras

The potential use of froth cameras to assist with process control should be considered.

26.2.4 Adding Molybdenum Recovery System

The proposed metallurgical processing design will produce a saleable high grade copper concentrate containing by-product gold. As the economic viability of the recovery of molybdenum has not been proven to date, the molybdenum recovery process has been excluded. In the future the inclusion of a molybdenum recovery section into the existing concentrator would be a relatively straightforward undertaking.

26.3 Overall Site Infrastructure (BGC and Wardrop)

26.3.1 Fresh Water Supply

Wardrop recommends a detailed inspection of the existing pipeline to assess the working conditions, pipe wall thickness, and pressure ratings.

26.3.2 Geotechnical Conditions

Access restrictions associated with the requirements for archaeological clearances limited the geotechnical investigation program. Additional investigation is required in specific areas, including: the truck shop area, where no subsurface data are available; the access road alignment, where typical conditions have been inferred from test holes in the general area, a possible borrow source for structural fill; and, the plant site area, where the presence of faults near proposed foundation grades requires further study and the waste rock piles where the fine grained soils were encountered in the foundations that require further delineation and testing.

26.4 Tailings Management Facility (Golder)

The feasibility level study has been developed from laboratory test data and extensive Golder experience in the design of paste and thickened tailings plants and tailings storage facilities. Golder considers that the review of this feasibility study during the detailed engineering should also include, and not be limited to the following items:

- perform laboratory testing with representative sample to confirm data obtained during the feasibility lab testing campaign
- perform an on-site flow loop pumping test to determine more accurately the friction losses in the thickened tailings discharge pipeline and to confirm the ability of centrifugal pumps to pump the Ajax non-segregating thickened tailings
- review of the emergency backup power design basis

- review of the thickened tailings discharge pipeline location to facilitate the TSF berm raising
- develop a more detailed analysis of evaporation to determine the monthly distribution for lake evaporation
- installation of an evaporation tank at the project area to measure pan evaporation on site
- additional boreholes should be drilled at both TSF Ponds dyke and TTP areas for a better understanding of the geological, geotechnical and hydrogeological conditions at the areas
- further testing will be required to confirm the selected geotechnical parameters used for slope stability analyses, particularly those selected for the thickened tailings
- seepage and liquefaction analyses should be carried out for the TSF
- a detailed survey of the Lac Le Jeune Road section adjacent to the east dyke of the TSF Pond is required to better define the dyke requirements.

26.5 ENVIRONMENTAL (KNIGHT PIÉSOLD)

Knight Piésold considers that additional studies will be required during detailed design and to support project permitting. These additional studies will include, but not be limited to, the following.

- geochemical characterization of tailings and waste rock including kinetic testwork to establish effluent water quality
- development of a site-wide water balance for modelling of surface and groundwater conditions during operations and post-closure
- closure plan and cost estimate for reclamation bonding.

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APPENDIX A

Certificates of Qualified Persons

ALI FARAH, P.ENG.

I, Ali Farah, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Lead Mechanical Engineer with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the "Technical Report").
- I am a graduate of Shiraz University, (BS, Mechanical Engineering, 1984). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #144443. My relevant experience includes 20 years of experience with hydraulic calculations, equipment design/selection and design of pumping systems. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was April 8, 2010 for two days.
- I am responsible for Sections 18.2 and 26.3.1 of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. 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 parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the part of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Vancouver, British Columbia

"Original document signed and sealed by Ali Farah, P.Eng."

Ali Farah, P.Eng.
Lead Mechanical Engineer
Tetra Tech WEI Inc.

AMIR KARAMI, P.ENG.

I, Amir Karami, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Rock Mechanics Engineer with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the "Technical Report").
- I am a graduate of McGill University (M.Eng., 1995) and the University of Alberta, (Ph.D., 2002). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #34646. I have over 10 years experience on rock mechanics engineering and design for both open pit and underground mines. I have conducted economic studies, financial analyses and due diligence on a number of mining projects. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not completed a personal inspection of the Property.
- I am responsible for Sections 1.9 and 22.0 of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. 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 parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Vancouver, British Columbia

"Original document signed and sealed by Amir Karami, P.Eng."

Amir Karami, P.Eng.
Senior Geotechnical Engineer
Tetra Tech WEI Inc.

ANDRE DE RUIJTER, P.ENG.

I, Marinus Andre De Ruijter, P.Eng., of Delta, British Columbia, do hereby certify:

- I am a Senior Metallurgical Engineer with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the "Technical Report").
- I am a graduate of the University of Witwatersrand, Johannesburg, South Africa (B.Sc. Physics, Mathematics, 1970; B.Eng., 1973; M.Eng., 1979). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #31031. My relevant experience with respect to this project includes copper and base metal sulphide mineral flotation projects, and 25 years experience in the gold industry encompassing laboratory research and development, operational gold plant responsibility, and process plant design. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was February 11, 2009.
- I am responsible for Sections 1.5, 1.8, 1.10, 13.0, 17.0, 21.2.2, 25.2 and 26.2 of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. 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 parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Vancouver, British Columbia

"Original document signed and sealed by Andre De Ruijter, P.Eng."

Marinus Andre De Ruijter, P.Eng.
Senior Metallurgical Engineer
Tetra Tech WEI Inc.

CHRIS LEE, P.ENG.

I, Chris Lee, P.Eng., of Sudbury, Ontario, do hereby certify:

- I am a Managing Principal – Paste Engineering and Design with Golder Associates Ltd. with a business address at 1010 Lorne Street, Sudbury, Ontario, P3C 4R9.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the "Technical Report").
- I am a graduate of Queens University (B.Sc. Mechanical Engineering, 1993 and B.Sc. Mining Engineering, 1994). I am a member in good standing of the Association of Professional Engineers and Geoscientists of Ontario, License #90458993. My relevant experience is 13 years of consulting and detailed design work in the field of thickened tailings and paste backfill. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I did not personally inspect the Property.
- I am responsible for Sections 1.6, 1.8, 1.10, 18.4, 21.1.3, 21.2.3, 25.3, 26.4 and 27.0 (Tailings Management Facilities section only) of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. 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 parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Sudbury, Ontario

"Original document signed and sealed by Chris Lee, P.Eng."

Chris Lee, P.Eng.
Managing Principal – Paste
Engineering and Design
Golder Associates Ltd.

HASSAN GHAFFARI, P.ENG.

I, Hassan Ghaffari, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Manager of Metallurgy with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the "Technical Report").
- I am a graduate of the University of Tehran (M.A.Sc., Mining Engineering, 1990) and the University of British Columbia (M.A.Sc., Mineral Process Engineering, 2004). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #30408. My relevant experience with respect to mineral process engineering includes 22 years of experience in mining and plant operation, project studies, management, and engineering. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was April 8, 2010 for two days and June 23, 2010 for one day.
- I am responsible for Sections 1.1, 1.8, 1.11, 2.0, 3.0, 4.0, 5.0, 6.0, 18.1, 18.6, 18.8, 18.9, 19.0, 21.1.2, 23.0, 24.0 and 27.0 (Mineral Processing and Overall Site Infrastructures section only) of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. 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 parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Vancouver, British Columbia

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

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

Irwin Wislesky, P.Eng.

I, Irwin Wislesky, P.Eng., of Mississauga, Ontario, do hereby certify:

- I am a Senior Geotechnical Engineer with Golder Associates Ltd. with a business address at 102, 2535 – 3rd Avenue, S.E., Calgary, Alberta, T2A 7W5.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the “Technical Report”).
- I am a graduate of the University of Toronto, (BASc, 1978). I am a member in good standing of the Association of Professional Engineers and Geoscientists of Ontario, License #50625508. My relevant experience includes 30 years of experience working on the design of tailings containment facilities as a professional engineer. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was June 24, 2010 for one day.
- I am responsible for Sections 1.6, 1.8, 1.10, 18.4, 21.1.3, 21.2.3, 25.3, 26.4 and 27.0 (Tailings Management Facilities section only) of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. 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 parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Calgary, Alberta

“Original document signed and sealed by Irwin Wislesky, P.Eng.”

Irwin Wislesky, P.Eng.
Senior Geotechnical Engineer
Golder Associates Ltd.

KEN BROUWER, P.ENG.

I, Ken Brouwer, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Managing Director with Knight Piésold Ltd. with a business address at Suite 1400 – 750 West Pender Street, Vancouver, British Columbia, V6C 2T8.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the “Technical Report”).
- I am a graduate of the University of British Columbia, (BApSc., 1982). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #15117. My relevant experience is includes environmental baseline studies, water management planning, environmental monitoring, mine permitting and mine closure. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was in the summer of 2008 for one day.
- I am responsible for Sections 1.7, 20.0, 26.5 and 27.0 (Environmental section only) of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. as defined by Section 1.5 of the Instrument.
- I have previously conducted engineering and environmental studies on the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Vancouver, British Columbia

“Original document signed and sealed by Ken Brouwer, P.Eng.”

Ken Brouwer, P.Eng.
Managing Director
Knight Piésold Ltd.

KENNETH LEE, P.ENG.

I, Kenneth Lee, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Senior Structural Engineer with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the “Technical Report”).
- I am a graduate of the University of British Columbia, (M.Sc. Structural Engineering, 1984). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #16349. My relevant experience includes 23 years of structural design, engineering studies, upgrades and inspection of mining process related facitilies within North America and abroad. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was April 8, 2010 for two days.
- I am responsible for Section 18.7 of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. 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 parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Vancouver, British Columbia

“Original document signed and sealed by Kenneth Lee, P.Eng.”

Kenneth Lee, P.Eng.
Senior Structural Engineer
Tetra Tech WEI Inc.

RAMON MENDOZA REYES, P.ENG.

I, Ramon Mendoza Reyes, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Principal Mining Engineer with AMEC Americas Limited with a business address at 111 Dunsmuir Street, Suite 400, Vancouver, British Columbia, V6B 5W3.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the "Technical Report").
- I am a graduate of National Autonomous University of Mexico, (Mining Engineering, 1989) and the Colorado School of Mines (M.Sc. Degree in Mining & Earth Systems Engineering, 2003). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia. My relevant experience is 22 years and I have previously been involved with mine designs, mine planning and mine operations for precious and base metal, disseminated sulphide and industrial mineral projects in North America and South America. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was Feburary 17 to 19, 2011.
- I am responsible for Sections 1.4, 1.8, 1.10, 15.0, 16.0, 21.1.1, 21.2.1, 25.1, 26.1 and 27.0 (Mining section only) of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. as defined by Section 1.5 of the Instrument.
- I have been involved in the Project in 2011 in the role of preparation of Mineral Reserve estimates and mine design that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Vancouver, British Columbia

"Original document signed and sealed by Ramon Mendoza Reyes, P.Eng."

Ramon Mendoza Reyes, P.Eng.
Principal Mining Engineer
AMEC Americas Limited

TIMOTHY O. KUHL, R.M. SME

I, Timothy O. Kuhl, R.M. SME, of Sparks, Nevada, do hereby certify:

- I am a Principal Geologist with AMEC E&C Services Inc. with a business address at 780 Vista Boulevard, Sparks, Nevada, 89434
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the "Technical Report").
- I am a graduate of South Dakota School of Mines and Technology, (B.Sc., Geological Engineering, 1975 and M.Sc., Geology, 1982). I am a registered member of the Society for Mining, Metallurgy and Exploration (SME; RM SME). I have practiced my profession continuously since 1982. I have been directly involved in estimation of mineral resources for various mining and exploration projects for commodities including gold, copper, platinum, palladium and nickel. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was June 28 to July 5, 2008.
- I am responsible for Sections 1.2, 1.3, 1.4, 1.8, 1.10, 7.0, 8.0, 9.0, 10.0, 11.0, 12.0, 14.0, 25.1, 26.1 and 27.0 (Geology and Mining sections only) of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. as defined by Section 1.5 of the Instrument.
- I have been involved with the Ajax Project as the Qualified Person responsible for Mineral Resource estimation since 2008, when I was co-author of the technical report entitled, "Kuhl, T., and Kozak, A., 2008: Technical Report on the Afton-Ajax E-W Deposit, Kamloops, British Columbia" effective date October 31, 2008".
- I have read the Instrument and the parts of the Technical Report that I am responsible for has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Sparks, Nevada

"Original document signed by
Timothy O. Kuhl, R.M. SME"

Timothy O. Kuhl, R.M. SME
Principal Geologist
AMEC E&C Services Inc.

TING LU, P.ENG.

I, Ting Lu, P.Eng., of Vancouver, British Columbia, do hereby certify:

- I am a Mineral Processing Engineer with Tetra Tech WEI Inc. with a business address at #800 – 555 West Hastings Street, Vancouver, British Columbia, V6B 1M1.
- This certificate applies to the technical report entitled Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the “Technical Report”).
- I am a graduate of Queen’s University (M.A.Sc., 2006) 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, License #32897. My relevant experience includes 15 years of experience in the mineral processing industry. I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was June 23, 2010 for two days.
- I am responsible for Sections 1.8, 18.5 and 21.1.2 of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Vancouver, British Columbia

“Original document signed and sealed by Ting Lu, P.Eng.”

Ting Lu, P.Eng.
Mineral Processing Engineer
Tetra Tech WEI Inc.

Warren Newcomen, P.Eng.

I, Warren Newcomen, P.Eng., of Kamloops, British Columbia, do hereby certify:

- I am a Senior Geological Engineer 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 Ajax Copper/Gold Project – Kamloops, British Columbia Feasibility Study Technical Report, dated January 6, 2012 (the "Technical Report").
- I am a graduate of University of British Columbia, (B.A.Sc., 1985) and the University of California at Berkeley (M.S., 1990). I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia, License #16123. My relevant experience is with Cortez Hills Project, Nevada; Donlin Creek Project, Alaska; Galore Creek Project, British Columbia; New Afton Project, British Columbia; KSM Project, British Columbia; Spanish Mountain Gold Project, British Columbia; Eagle Gold Project, Yukon. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was March 3, 2011 for one day.
- I am responsible for Sections 10.9, 15.2.6, 16.2, 16.3.1, 16.6, 16.9.1, 18.3 and 26.3.2 of the Technical Report.
- I am independent of Abacus Mining & Exploration Corp. as defined by Section 1.5 of the Instrument.
- I was previously involved in the Preliminary Economic Assessment of the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2012 at Kamloops, British Columbia

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

Warren Newcomen, P.Eng.
Senior Geological Engineer
BGC Engineering Inc.